Techniques in Underground Mining

Selections From Underground Mining Methods Handbook

Edited by Richard E. Gertsch and Richard L. Bullock



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Foreword

In 1982, SME published the *Underground Mining Methods Handbook*—a 1,754-page comprehensive review of underground mining. It contained one of the best collections of information on underground mining of any book published since the *Mining Engineers Handbook* by Robert Peele. However, the original is out of print, and there is a lack of printed information currently available on this topic. Thus, a condensed version of the *Underground Mining Methods Handbook* was produced to aid in the training of mining engineers and practicing professionals.

In producing the original handbook, Dr. William Hustrulid, an editorial advisory board, and the SME-AIME Book Publishing Committee recruited 165 authors and section editors from academia and the mining industry. Their goal was to put together a collection of knowledge that was comprehensive and unique. They succeeded, and the *Underground Mining Methods Handbook* served the industry well. And although it was published over 15 years ago, most of the information is still meaningful and applicable in the education of new mining engineers today.

Therefore, this new version, *Techniques in Underground Mining–Selections from the Underground Mining Methods Handbook*, is published by SME as a teaching tool and source book.

Richard Bullock

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Preface

Considered a shrinking industry in the late 1980s and early 1990s, underground mining is now experiencing a revival. Unfortunately, there is a void of material that adequately addresses the important mining methods and operational techniques needed by both mining engineering students and industry professionals. As a temporary solution, the original *Underground Mining Methods Handbook* (published in 1982 and now out of print) has been condensed in this new version, *Techniques in Underground Mining–Selections From the Underground Mining Methods Handbook* .

The discriminating reader will question why the editors included or excluded certain sections from the original. To assist the editors, guidelines were established to produce an inexpensive volume that would include descriptions of underground methods for mining students.

The guidelines followed by the editors were:

- include only those sections that address and describe underground mining methods;
- limit the mining methods to noncoal mining methods;
- reduce the text to one-third of the original text; and
- make no revisions to sections or chapters (although deletions within a chapter were allowed if it improved the chapter).

The editors excluded sections and chapters that covered equipment, coal mining, rock fragmentation, rock mechanics and ground control methods, ventilation, economic evaluation, financial consideration, and exploration. These subjects are covered in other available texts.

One mining method–the use of timber support– extensively covered in the original book had to be deleted in order to meet the page limit guideline. Although timbering is mines is still important, particularly in drifting, there is a definite trend away from mining methods requiring timber as the main support. Thus, this method is not included in this version.

Although many chapters on mining equipment in the original handbook are still applicable, many others are obsolete. The change from pneumatic to hydraulic drills is a good example. Since updated information is available from other sources, mining equipment is not included.

Chapters on economics and cost considerations were a compromise. The sections that originally stood alone were eliminated. However, the cost analysis that accompany the mining methods are included, even though the costs are 15 years out of date. The methods used to calculate costs are worth preserving and the actual updated costs could easily be substituted into that methodology.

The introductory section was also a compromise. The editors decided to leave in enough material to introduce the subject and to provide adequate discussions of engineering matters unique to underground mining, although somewhat shared with surface mining, including sampling, reserve analysis, and planning.

Reprinting a condensed version of the *Underground Mining Methods Handbook* is only a temporary solution to the problem of providing an updated text on underground mining methods. A completely new version must soon be written. This version should include updates to all the mining method changes, particularly those changed in vertical crater retreat, undercut and fill with high-density cemented or paste backfill, in-situ or solution mining, and hardrock longwall mining methods. It seems unlikely that any new work will encompass the sweeping range of topics covered in the 1982 original. Thus, any new handbook will likely cover several volumes.

Richard Gertsch

Richard Bullock

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SECTION 1

Overview

CHAPTER 1

Ore Reserve Calculation

Leigh A. Readdy, David S. Bolin, and Graham A. Mathieson

INTRODUCTION

The estimation of ore reserves is a process that begins with the earliest exploration stages on a property and continues throughout any subsequent evaluation and exploitation of the deposit. During exploration and preliminary evaluation, the results of these reserve estimates constitute the basic data for prefeasibility studies and economic analysis. The decision to continue exploration and development or to abandon a prospect is often based upon these studies.

During the active life of a mine, reserve computations are continuously revised to assist in development planning, cost and efficiency analyses, quality control, and improvement of extraction and processing methods. Accurate reserve estimates are also required when financing a project, purchasing or selling a property, and for accounting purposes such as depletion and tax calculation.

It is important to remember that the reliability of ore reserve estimates varies progressively through time as more and more information becomes available. The lowest order of reliability of estimation of reserves exists at the time of discovery. The maximum level of certainty concerning the ore reserves within a deposit is reached when the deposit is completely mined out. Between these two extremes are variable levels of certainty as to the tonnage and grade of the resource. This is particularly true of that portion of the mineral resource that constitutes the "minable reserve," as this portion is dependent upon economic as well as geological and technological factors.

In the following discussion, several of the factors affecting ore reserve computation and some of the commonly used methods of calculation are presented. The first part of the discussion is confined to classical methods of hand calculation utilizing level maps and sections. The second part of the discussion presents an overview of somewhat more sophisticated methods of geostatistics

that have been developed within recent years. Geostatistical methods frequently provide the best ore reserve estimates for many deposits. However, a digital computer is required for such geostatistically derived estimates due to the number and complexity of the computations involved. For many properties the classical nongeostatistical methods are adequate, particularly in the early exploration stages. In fact, it is the author's contention that even at those properties where the geostatistical approach is employed for reserve estimation, the geological and engineering staffs should prepare an estimate by traditional means. This provides an internal audit for the reserves and at the same time requires a continuing close appraisal of geologic problems that influence the presently known geostatistically estimated reserves and production at the mine.

CLASSIFICATION OF ORE RESERVES

The definition and classification of ore reserves have varied over the period of development of the modern mining industry. "Ore" is generally understood to be any naturally occurring, in-place, mineral aggregate containing one or more valuable constituents that may be recovered at a profit under existing economic conditions. This definition ignores special situations, such as wartime production, or those cases when an otherwise unprofitable deposit may be exploited for political or social reasons.

Ore reserves are classified with respect to the confidence level of the estimate. Traditionally, ore reserves have been classified as proven (measured), probable (indicated), possible, and inferred. Historically, proven ore has been regarded as that which is "blocked out" (i.e., measured, sampled, and assayed on four sides); probable ore as blocked on three sides; possible ore as blocked on two sides; and inferred ore as ore-grade material that is known on only one side.

The U.S. Bureau of Mines (USBM) has introduced the following ore reserve classification:

Measured Ore

Measured ore is ore for which tonnage is computed from dimensions revealed in outcrops, trenches, workings, and drill holes, and for which the grade is computed from the results of detailed sampling. The sites for inspection, sampling, and measurement are so closely spaced and the geologic character is so well-defined that the size, shape, and mineral content are well established. The computed tonnage and grade are judged to be accurate within stated limits, and no such limit is judged to differ from the computed tonnage or grade by more than 20%.

Indicated Ore

Indicated ore is ore for which tonnage and grade are computed partly from specific measurements, samples, or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for

inspection, sampling, and measurement are too widely or otherwise inappropriately spaced to outline the ore completely or to establish its grade throughout.

Inferred Ore

Inferred ore is ore for which quantitative estimates are based largely on a broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. These estimates are based on an assumed continuity or repetition for which there is geologic evidence; this evidence may include comparison with deposits of similar type. Mineral bodies that are completely concealed may be included if there is specific geologic evidence of their presence. Specific estimates of inferred ore usually include a statement of the special limits within which the inferred ore may occur.

The traditional classification system has been particularly applicable to tabular vein or bedded-type deposits. This results from normal mine practice for such deposits, which is to develop the deposit by a system of rectangular blocks, the boundaries of which are defined by drifts and raises. Prior to extraction, the vein is exposed on all sides and may be sampled in detail. In more recent years, the underground mining of large deposits of indefinite shape, such as porphyry copper deposits, has led to an increasing reliance on diamond drill-hole data to define ore continuity. Prior to mining these deposits, however, it is still common practice to do a limited amount of underground development. This underground development serves to obtain sample information, including large bulk samples for metallurgical testing, as well as to provide additional detailed geological and engineering data for ore reserve estimation and verification.

By the time a deposit is ready for development, there usually exist two ore reserve estimates: a geologic reserve or total resource estimate, and a mining ore reserve. The geologic reserve is an estimate including all known mineralization above a certain grade within the deposit. However, the geologic reserve figure may not be associated with a specific mining cutoff grade. The mining reserve constitutes that portion of the geologic reserve that can be mined and processed at a profit. The mining reserve is always less than or equal to the geologic reserve estimate because a variable proportion of the orebody must be left unmined for a variety of reasons. These reasons include the need for pillars for ground support, metallurgical problems, width of mineralization, or other economic and engineering factors.

ORE RESERVE PARAMETERS

An ore reserve estimate contains two important parameters: the amount of ore and the average grade or value of that ore. In metal mines, the amount of ore is usually expressed in either metric tons (1000 kg) or short tons (2000 lb). Grades are normally expressed as a percentage for base metal ores, whereas precious metals may be reported as troy ounces per ton, pennyweights per ton, or grams/metric ton. The reporting system depends upon the geographic location of the deposit and/or company policy. It is also fairly common in old

reports to find gold and silver values stated in dollars. This practice makes the utilization of these data difficult unless the conversion prices are given in the report. Another doubtful practice, which is common in some countries, is to state reserves simply as contained metal, that is, pounds of uranium oxide or fine tons of tin. Although these figures are sometimes useful for certain economic analyses, they provide little useful engineering information and constitute a resource statement rather than an ore reserve estimate.

The calculation of the tonnage and grade of a deposit requires the collection and documentation of a considerable amount of data. These data include accurate assay information, plans and sections, details of ore controls, the tonnage factor, applicable cutoff grade to be used, potential mineral recovery, and engineering details such as minimum mining width and anticipated dilution. These items are discussed in the following sections.

Grade Determination

The average grade of an ore deposit or of a specific block within a deposit is based on assays of samples collected within the block or deposit. Sample collection, preparation, and analysis are often the most critical operations in evaluating the reserves for a mineral property. Sampling theory and practice constitute a complex subject in their own right and only some of the more important points are touched upon in the following summary. (For further information, the following references should be consulted: Forrester, 1946; McKinstry, 1948; Gy, 1968; Cummins and Given, 1973; Koch and Link, 1970, 1974; David, 1977; Barnes, 1980.)

Cutoff Grade. Associated with the definition of ore grade is the concept of cutoff grade. The cutoff grade is the minimum ore grade that can be mined at a profit under economic conditions existing at a particular point in time. The cutoff grade can vary with time due to changes in such factors as commodity prices, operating costs, and taxes. The cutoff grade used for any reserve calculation should always be stated.

Sampling. Sampling of an ore deposit is a process of approximation. The objective is to arrive at an average sample value that most closely represents the true average value for the body in question. The importance of attention to detail during the sampling program becomes apparent when it is realized that in the case of a very well sampled block of ore from a vein-type deposit, the actual sample volume may represent only about 0.25% of the block. In other cases, such as the sampling of a porphyry copper deposit by diamond drilling, the sample volume may constitute only about 0.004% of the orebody. To obtain the most accurate grade estimation, it is imperative that the sampling crews and procedures be carefully monitored by members of the geological and engineering staff.

Channel Sampling. The classical method of sampling an ore deposit consists of cutting a relatively precise channel of constant depth and width across the exposed width of the vein. These samples may be cut with a hammer and chisel or an air hammer, and the chips collected on a canvas sheet spread on the floor of the working area. The samples are collected across the full width

of the vein, or at some uniform fixed length in wide or indefinite zones. In complex veins, any identifiable subdivisions should be sampled separately. Samples are collected in a consistent manner throughout the deposit. These may be taken at right angles to the contacts and dip of the vein, or if this is impractical, measured sample vein widths can be corrected to true widths by simple trigonometry. However, in regular dipping bodies the sample can be collected as a horizontal channel across the zones because vertical height of projection times horizontal width is equal to true width times true length. It is important for the samples to be collected in a consistent manner and for any local variation or change in the sampling practice to be recorded. If possible, samples should be collected at regular intervals along the working. Samples can be taken from either the face, rib, or back of the working.

The working area to be sampled should be washed down or at least brushed clean before sampling begins. This is done to reduce the potential for contamination of the sample by muck and loose fragments on the face being sampled. The sample area should be chipped clean and rough projections should be removed.

The location of the sample is then marked on the face, rib, or back using spray paint or chalk. If the sample channel is to be broken into several parts representing distinctly different portions of the vein or mineralization, these should also be marked on the face. A drawing and, if possible, a photograph of the area being sampled provide additional documentation. This is particularly important if the sample is collected from a surface that will be mined or covered by support materials. Such records are of considerable value when it becomes necessary to reinterpret the correlation of mineralization if the sample site has been removed from access by other mining activity.

Following marking of the sample site, a clean canvas tarp is spread beneath the sample area. The sample is then chipped out along the marked line, and the fragments and fine material are collected on the tarp. When the sample cutting has been completed, the material is homogenized by "rolling" the sample back and forth on the tarp. The sample is then split, if necessary, and put into plastic or cloth bags. Sample numbers and tags are then prepared and placed inside the bag.

Channel sampling probably provides the most accurate sample, but the process is laborious, time-consuming, and expensive. In many cases satisfactory results may be obtained by "chip channel" sampling. In this method, the samples are laid out as for channel sampling, but instead of cutting out a channel, a band about 0.3 m (1 ft) wide is chipped with a geology pick, or hammer and chisel, across the width of the vein. An effort should be made to keep a relatively constant sample volume proportional to the widths of the vein. Care must be taken to collect approximately the same size chips across the zone being sampled. Chip points should also be as regularly spaced as possible.

In some cases it will be necessary to reduce the volume of the sample collected prior to sending it out for assay. It is common practice to make the first reduction at the mine, usually by the traditional method of "cone and quarter." However, the average fragment size of a sample should be reduced

TABLE 1.1 Minimum permissible sample weight for a given particle size

Diameter of Largest Pieces		Very Low Grade or Very Uniform Ore, lb*	Medium Ores, lb*	Rich or Spot Ores, lb*
In.*	Mesh			
4		4,800	35,556	
2		1,200	8,889	51,200
1		300	2,222	12,800
0.5		75	556	3,200
0.25		19	139	800
0.131	6	5.15	38.1	220
0.065	10	1.29	9.5	55
0.0328	20	0.322	2.37	13.76
0.0164	35	0.081	0.59	3.44
0.0082	65	0.020	0.15	0.86
0.0041	150	0.005	0.038	0.21

*Metric equivalents: in. × 25.4 = mm; lb × 0.4535924 = kg. After Cummins and Given, 1973.

prior to any splitting operation. Several empirical methods for determining minimum fragment size with respect to type of mineralization are reported in the literature (Gy, 1968; Cummins and Given, 1973). Table 1.1 illustrates the data from one of these methods (Cummins and Given, 1973).

Diamond Drill Sampling. As a result of improvement in technology and equipment, diamond drilling has become increasingly important as a sampling method. During exploration, the core is usually brought in from the drill, measured, weighed, and either split by a mechanical splitter or sawn lengthwise with a diamond saw. Half of the core is sent out for assay and the remaining half logged by a geologist and stored for reference. During later stages of exploration and development, it has been the practice at some properties to ship the entire core for assay, keeping only representative specimens or color photographs for reference.

Samples are usually collected at constant intervals down the length of the core, although samples may be taken at shorter intervals through highly mineralized areas or veins. Often, parts of the core known to be barren, or without visible evidence of mineralization, are not split or assayed.

In any drilling program, there likely will be areas where drilling is difficult and core recovery is poor. In these zones, it is common practice to collect samples of the drilling fluid, or sludge. McKinstry (1948) discusses various methods of integrating core and sludge analyses.

Miscellaneous Sampling Techniques. Besides the sampling methods mentioned previously, there are various other techniques employed in specific situations. These methods include such procedures as random "grab" samples, such as sampling broken muck, and large bulk samples, usually

collected in lots of a half ton or larger. These large samples are taken for metallurgical testing or to improve sample accuracy in deposits such as epithermal gold deposits, where ore mineral distribution is particularly erratic.

Rotary or churn drill cuttings are utilized for sampling many types of industrial mineral deposits as well as disseminated-type metal ores. This drilling method is rapid and of relatively low cost. The chips and fine material produced may be collected over uniform intervals, split if necessary, and bagged for analysis. The sampling of uranium and thorium deposits by radiometric logging is a specialized technique and will not be discussed in this presentation.

Assaying. Assaying may be done by a commercial laboratory or by an in-house company lab. In any case, a certain percentage of the samples, usually a minimum of 10%, should be assigned a new sample number and resubmitted for a repeat analysis to provide a check on the analytical precision of the laboratory. It is also recommended practice to send a percentage of the samples to a different laboratory for accuracy comparison. Should there be any doubt as to the accuracy of the particular laboratory used, a few standard samples, including a blank, should be submitted for analysis.

Statistical Analysis of Sample Data. When the assays have been received from the laboratory and the validity of the results has been satisfactorily established, it is often useful to make some simple statistical analyses of the data. Classical statistical techniques are based on two assumptions: that the samples are random and that the data have a normal distribution. The problem of obtaining random samples is somewhat difficult to analyze. Samples collected from an ore deposit are seldom statistically independent of one another. This factor is used to advantage in the calculation of geostatistical ore reserves, a subject discussed in detail in a later section.

Several statistical techniques are known to determine if sample data are random. One of the easiest to calculate is the sample volume-variance relationship (Hazen, 1967). If samples are available in several different volumes—for example, from drill holes of various diameters—and if the sample data are random, the following relationship should be true:

$$S_1V_1 = S_2V_2 \ldots S_nV_n$$

where S is sample variance for one discrete sample volume and V is sample volume.

In many types of deposits, the fact that sample data are not random is not of serious concern. The results of the common statistical calculations may still be utilized, keeping in mind the bias introduced by the lack of statistical independence among the samples.

A normal sample distribution is seldom fulfilled with geological samples, although the logarithms of the data often show a normal distribution. To examine the type of statistical distribution for a data set, a simple histogram of the sample data may be prepared as shown on Fig. 1.1. Figure 1.1 also illustrates some of the sample distributions common to mineral deposit data.

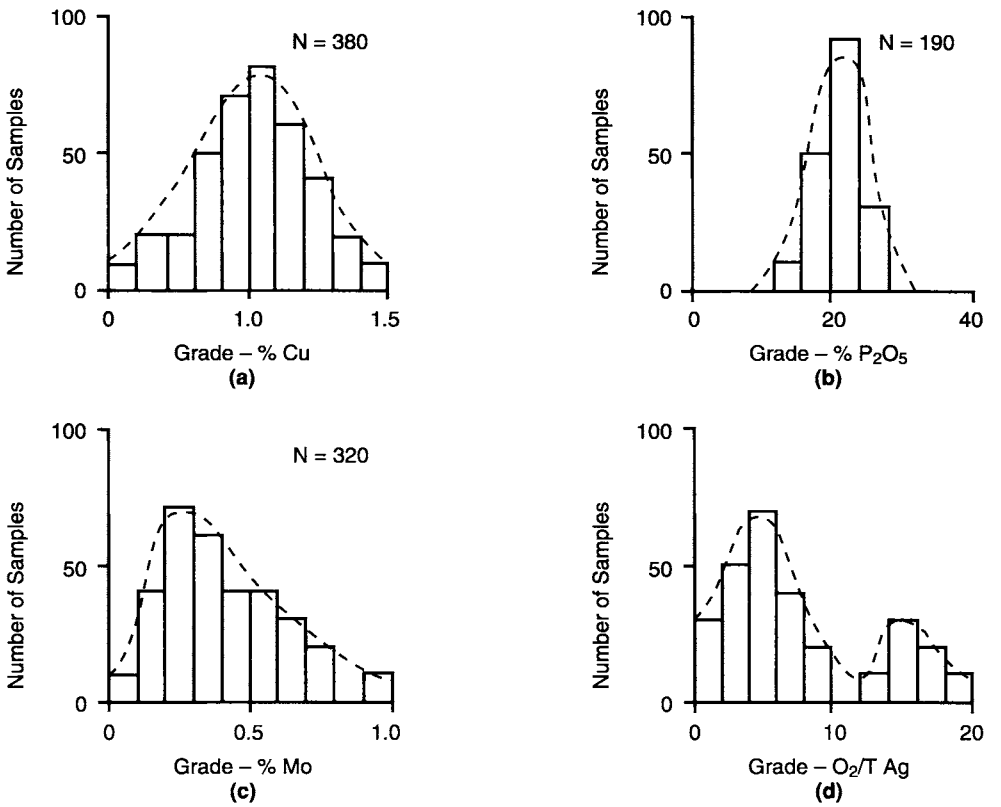


FIGURE 1.1 Typical sample distribution (a) Normal distribution, moderate variability, typical of some stratiform and massive sulfide deposits (b) Normal distribution, low variability, found in certain industrial mineral, iron, and manganese deposits (c) Lognormal distribution, common in many molybdenum, tin, tungsten, and precious metal deposits (d) Bimodal distribution, which may be produced by sampling two distinct ore types, or sampling across a zonation boundary in the mineralization

If an examination of the sample frequency distribution as shown on a histogram indicates an approximate normal distribution, then the normal statistical parameters may be calculated for the data set. The mean (average) variance and standard deviation calculated for the sample data should provide a reasonable approximation of these parameters for the deposit or portion of the deposit sampled.

In those cases where examination of the sample distribution indicates a lognormal distribution, then the log parameters, geometric mean, and standard deviation should be calculated. The geometric mean will differ from the arithmetic mean, and in many types of deposits—particularly epithermal precious metal deposits and hydrothermal tin deposits as well as others—the geometric mean provides a more realistic approximation of true average grade for the deposit.

The normal and lognormal distributions are two types commonly encountered in mineral deposit sampling. Various other distributions are known but are beyond the scope of this discussion. For further information, see Hazen (1967) and Koch and Link (1970–1971).

If it can be assumed that the standard deviation of the samples provides a reasonable approximation of the standard deviation of the entire deposit population, then this figure may be used to establish a precision on the grade estimate. The standard deviation is used to establish the standard error of the mean, $S = s/\sqrt{N}$, where s is the sample standard deviation and N is the number of samples. To establish the confidence interval, the standard error of the mean (S) is multiplied by the statistical factor t, usually referred to as Student's t, a function based on desired confidence limits and the number of samples. The appropriate values of t for the desired confidence level may be found in any handbook of statistical tables. At the 95% confidence level and greater than 50 samples, the value for t is approximately 2.0. Thus, if a particular set of 60 samples has a mean of 8.5% lead with a standard deviation of 1.2, then the confidence interval for the mean would be:

$$CI = 8.5 \pm \frac{1.2}{\sqrt{60}} \times 2$$

$$= 8.5 \pm 0.31$$

$$CI = 8.19 < 8.5 < 8.81$$

There is only one chance in 20 (5% probability) that the true grade lies outside the range of 8.19% to 8.81% Pb.

Alternatively, the formula for the computation of confidence limits can be rearranged such that:

$$N = \left(\frac{2 \times S}{\frac{CI}{2}} \right)^2$$

and the formula can then be used to approximate the number of samples that are used to establish a required precision for the estimate of the mean. For example, in the USBM definition of proven ore, the grade determination should be within 20% of the estimated value. If the calculation of a set of sample data indicates an average grade of 1.0% copper, then to be considered as proven, the grade must be between the confidence limits of 0.8% and 1.20% copper. If the standard deviation of preliminary samples is 1.5, then:

$$N = \left(\frac{2 \times 1.5}{\frac{0.4}{2}} \right)^2 = 225$$

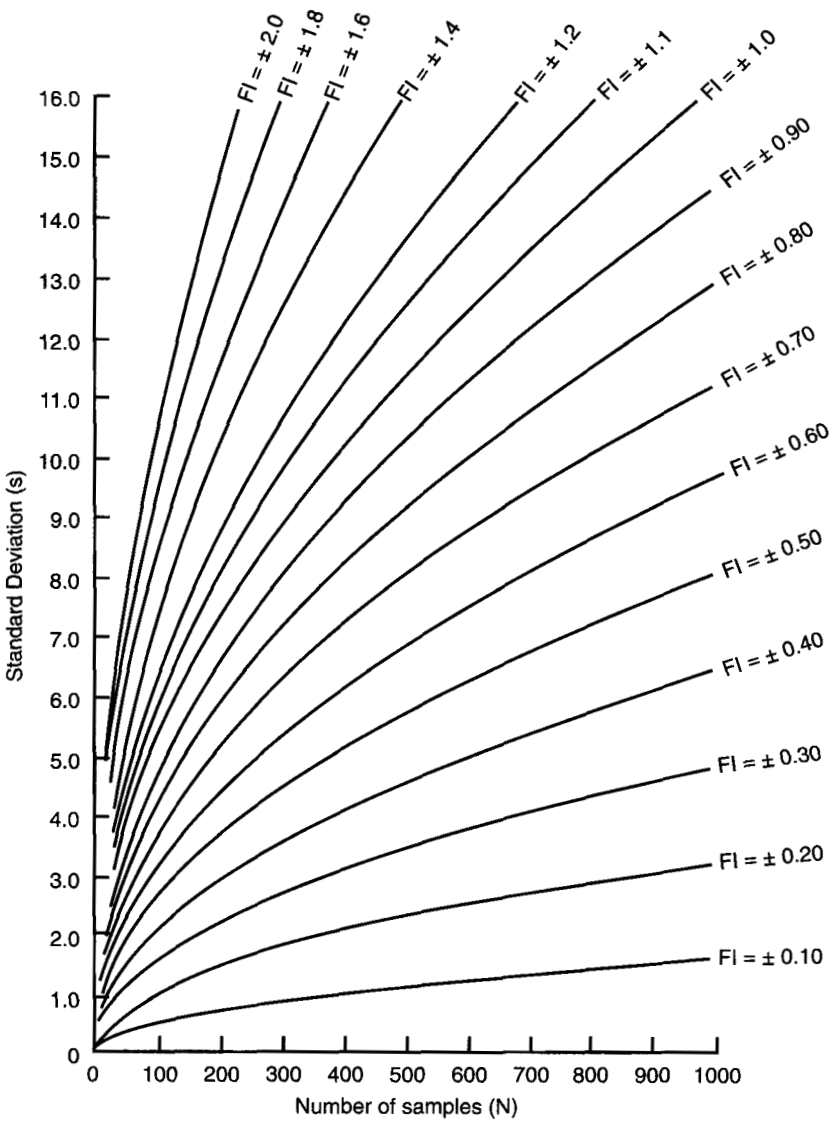


FIGURE 1.2 Relationship among number of samples, standard deviation, and confidence interval (designated here as FI) at the 95% confidence level (after Hazen, 1961)

If the preliminary sample consisted of 60 assays and produced a standard deviation of 1.5, then approximately 225 samples will be required before the grade of the deposit may be considered as meeting proven ore reserve standards. In actual practice, the standard deviation, as well as the error of the mean, will decrease as the sample number increases. Therefore, the confidence intervals should be recalculated as each group of samples comes in until the required precision is obtained. Peters (1978) illustrates an example of this same technique applied to a diamond drill program. Figure 1.2, reproduced from Hazen (1961), is a graph of the previous equation for the 95% confidence level, and may be used for rapid approximations. For example,

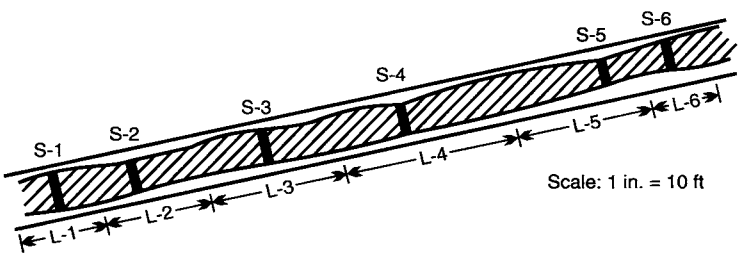


FIGURE 1.3 Sample plan of a silver-lead vein, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m

using Fig. 1.2 and the previous data, for which the standard deviation is 1.5 and confidence limits of ±0.2 are required, the expected number of samples is approximately 200, the same as for the original calculation.

Values will be slightly different as the curves of Fig. 1.2 are calculated for exact values of t rather than the approximation of 2.0.

Many other examples and applications of statistical techniques to ore sampling are available in the literature such as Koch and Link (1970, 1974); Peters (1978); Hazen (1961); Canadian Institute of Mining and Metallurgy (1968); Davis (1973); and Barnes (1980). Sinclair (1976) gives some interesting examples of application of cumulative probability graphs to sample analysis.

Sample Weighting

It is often necessary to compute a value for a composite sample, developing a weighted average for unequal sample widths or lengths. The method of such weighted sample calculation is illustrated in Fig. 1.3, which shows a series of samples collected from a vein exposure in a raise in a hypothetical lead-silver mine. Note that the samples are collected at right angles to the dip of the bed and, in this case, the vein is assumed to extend at right angles to the plane of the page. If for some reason it is impossible to collect samples across the true width of the vein, the measured width should be corrected to the true width by a simple trigonometric calculation, as the true width will be needed for tonnage calculations. Normally, samples are collected at regular intervals, with the interval dependent on the nature of the mineralization. Erratic mineralization, such as is common in epithermal gold-silver deposits, requires sampling at much closer spacing than more regular deposits. Here, samples have been shown at irregular spacing to illustrate the principle of weighting for area of influence. For purposes of calculation, see Table 1.2.

Discounting of Irregular High-Grade Assay Sections. Discontinuous or irregularly occurring high-grade assays are frequently encountered in certain types of orebodies. These may occur within a zone of slightly higher grade values or be encountered as a single apparently spurious value surrounded by the normal range of values. Examples of this phenomenon are commonly found in gold and silver deposits of all types, in vein-type uranium, tin, and tungsten, as well as other deposits. Proper treatment of such anomalous high values is of particular importance in the high unit value commodities. Improper or imprudent use of anomalously higher grade assay samples for the high unit value

TABLE 1.2 Calculation of a weighted average

Sample No.	Area of Influence, L, ft*	Width, W, ft	W × L, ft	% Pb	Oz* Ag/t	Pb × W × L	Ag × W × L
S-1	6.0	3.0	18.0	6.4	11.3	115.20	203.40
S-2	7.5	2.5	18.75	7.6	14.7	142.50	275.63
S-3	10.0	3.0	30.00	5.6	8.6	168.00	258.00
S-4	12.5	2.3	28.75	8.8	12.9	253.00	370.88
S-5	10.0	2.0	20.0	8.2	13.7	164.00	274.00
S-6	5.0	2.6	13.0	6.7	10.8	87.10	140.40
Total	51.00	15.40	128.50			929.80	1522.31

$$\text{Average grade Pb} = \frac{929.80}{128.50} = 7.2\% \text{ Pb}$$

$$\text{Average grade Ag} = \frac{1522.31}{128.5} = 11.9 \text{ oz Ag per st*}$$

$$\text{Average thickness} = \frac{128.50}{51.0} = 2.52 \text{ ft}$$

*Metric equivalents: ft × 0.3048 = m; oz × 0.02834952 = kg; st × 0.9071847 = t.

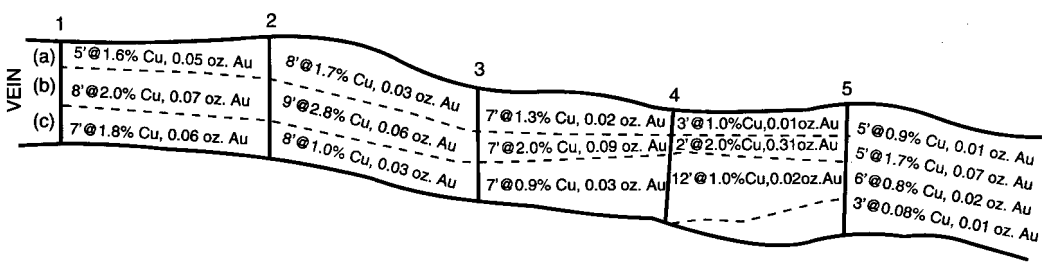


FIGURE 1.4 Plan view of a gold vein, metric equivalent: ft × 0.3048 = m

commodities could have a seriously misleading impact on the reserve estimate and value of the deposit. As a general rule, if there is no correlatable zone of higher values, then the abnormally high assay should be reduced to the value of the highest adjacent assay. If during mining there is a greater value present than that anticipated, all well and good. If, however, as mining progresses, the grade of mineral encountered is considerably lower than has been indicated using nondiscounted anomalous high values in the ore reserve process, this could result in serious operating losses. A method of adjusting a series of samples for erratic high grade intervals is shown in Fig. 1.4.

Planimeter Reading	Difference
1 _____	
	_____
2 _____	
	_____
3 _____	
	_____
_____	_____
Average _____	_____

FIGURE 1.5 Suggested format for recording and checking planimeter readings

In the next case, five crosscuts or horizontal drill holes cross a well-defined vein structure with three identifiable portions of the vein. This example could just as well be a vertical section through a tabular ore zone.

The 0.6-m (2-ft) sample of 8.6 g (0.31 oz) gold per ton in cut four is considerably higher than the samples surrounding it in cuts three and five or observed in samples in cuts one and two. To err on the side of caution, it should be discounted to either the value of the highest adjacent correlatable sample (i.e., 0.09 oz per ton in cut three) or to the average of the adjacent correlatable samples in cuts three and five [(0.09 + 0.07 oz)/2 = 0.08 oz per ton]. Whichever of these two procedures is used should be used consistently during the entire reserve calculation procedure.

Tonnage Determination

The calculation of tonnage for an ore deposit requires that the volume of the mineralized zone and the tonnage conversion factor be known.

Volume Calculation. The volume of the mineralized zone is calculated by measuring the area of the mineralization and multiplying the area by the corresponding thickness of material above cutoff grade. The area may be estimated by breaking the area into small, regular geometric shapes and calculating the total area by geometry. For very irregular or curved areas, the area is most easily determined by planimetering.

Measuring areas by planimetering is an important part of many ore reserve estimations. Unfortunately, all too often insufficient care or attention to detail is given to this aspect of reserve estimation. There are several problem areas in planimetering. One of the major problems is that for smaller areas, small measurement errors can become of sufficient magnitude to seriously affect the area measurement and thus the reserve calculation. Another frequently encountered problem is an erroneous reading of the planimeter. Both these problems are essentially eliminated by the planimetering of all areas at least three times. A format that has proven exceptionally useful for recording and checking the accuracy (or reproducibility) of planimetering is shown in Fig. 1.5.

TABLE 1.3 Examples of planimeter readings

	Acceptable Set of Three Readings		Acceptable After More Than Three Readings	
	Readings	Difference	Readings	Difference
1.	00197		00208	
		00199		00199
2.	00396		00407	
		00197		00200
3.	00593		00607	
				00197
			00804	
	Average:		Average Discounting 1st Reading:	
	$\frac{593}{3} = 197.6\ (198)$		$\frac{804 - 208}{3} = 198.6\ (199)$	

This format provides for continued cross-checking of the results and develops a record of the relative reliability of the readings. All readings are made consecutively without rezeroing the planimeter. This removes the initial random starting or missed zero error. Particular attention is given to zeroing the instrument at the start of planimetering of any one area. The area is then planimetered and the first reading recorded. The area is then replanimetered without rezeroing the instrument and the cumulative reading is recorded, as is the difference between this and the previous reading. This difference should be nearly identical to the first reading. The area is replanimetered a third time and the cumulative reading and difference recorded. The differences and the first reading should agree within at least 3%. If this criterion is not met, then yet another reading is taken and recorded along with the difference, and the previous single reading that is in obvious error is discarded. With reasonable care differences between readings of 1% or less are possible for larger areas.

Once the readings are acceptable in reproducibility, then the average of the readings is computed. This should approximate the first reading and the difference values. This average is then used to calculate the area. Two examples of reading sequences are shown in Table 1.3. In the series on the right, reading No. 1 appears to be in error by comparison with differences. Discounting this initial reading, we have three readings totaling 596 units, and the average of 596/3 = 198.6 should be used.

Tonnage Factor Calculation. The tonnage factor provides the mechanism for the conversion from volume of ore to weight of ore. In the English system, the tonnage factor is normally expressed as cubic feet per ton of ore. In the metric system, the tonnage factor is the specific gravity of the ore. The tonnage factor is dependent upon the specific gravity of the ore, and the specific gravity is a function of the mineral composition of the ore. Probably the most accurate

TABLE 1.4 Specific gravity of common rocks and minerals

	Specific Gravity		Specific Gravity
Rocks		**Minerals *(continued)***	
Andesite	2.4–2.8	Chromite	4.5
Basalt	2.7–3.2	Copper	8.8
Diabase	2.8–3.1	Covellite	4.6
Dolomite	2.7–2.8	Cuprite	6.0
Gabbro	2.9–3.1	Feldspar	2.6–2.8
Granite	2.6–2.7	Fluorite	3.1
Gravel (dry)	1.6–2.0	Galena	7.6
Limestone	2.7–2.8	Gold	17.5
Rhyolite	2.2–2.7	Graphite	2.2
Sandstone	2.0–3.2	Gypsum	2.3
Schist	2.6–3.0	Hematite	5.2
Shale	1.6–3.0	Molybdenite	4.8
		Muscovite	2.9
Minerals		Pentiandite	4.8
Anglesite	6.3	Platinum	19.0
Anhydrite	2.9	Pyrite	5.0
Argentite	7.3	Pyroxene	3.3
Arsenopyrite	6.0	Pyrrhotite	4.7
Barite	4.5	Quartz	2.7
Bauxite	4.5	Scheelite	6.0
Bornite	4.9	Sericite	2.6
Calcite	2.7	Silver	10.6
Cassiterite	7.0	Smithsonite	4.4
Cerussite	6.5	Sphalerite	4.1
Chalcedony	2.6	Stibnite	4.6
Chalcocite	5.7	Sulfur	2.1
Chalcopyrite	4.3	Uraninite	9.4

method of determining specific gravity of an ore is to calculate an average specific gravity using specific gravities of individual minerals (Table 1.4), provided the relative percentages of ore minerals present are accurately known. For example, if a massive sulfide ore is 10% galena, 35% sphalerite, and 55% pyrite, the specific gravity would be:

$$7.6 \times 0.10 = 0.76$$
$$4.1 \times 0.35 = 1.44$$
$$5.0 \times 0.55 = \underline{2.75}$$
$$4.95 = \text{sp gr of ore}$$

The specific gravity of an ore may also be computed by weighing a core or specimen of the ore in air, then weighing the same sample suspended in water. The specific gravity is calculated by the following formula:

$$\text{Sp gr} = \frac{Wa}{Wa - Ww}$$

where Wa = weight in air and Ww = weight in water.

If the ore volume has been computed in cubic meters, the volume multiplied by the specific gravity is the tonnage in metric tons directly. If working in the English system, the tonnage factor is calculated as follows:

$$\text{Sp gr} \times 62.5 \text{ (lb per cu ft water)} = \text{lb per cu ft ore}$$

$$\text{Tonnage factor} = \frac{2000 \text{ lb per ton}}{\text{lb per cu ft ore}} = \text{cu ft per ton ore}$$

For example, if a porphyry copper ore has a specific gravity of 2.8, then:

$$2.8 \times 62.5 \text{ (lb per cu ft of water)} = 175 \text{ lb per cu ft ore}$$

$$\text{Tonnage factor} = \frac{2000}{175} = 11.43 \text{ cu ft per ton ore}$$

For purposes of ore reserve estimation, a single or even a few samples of core or ore specimens would not be suitable. Specific gravity determinations would be made of both the mineralization and gangue from many drill hole and other samples.

A third method of calculation, which is only occasionally employed, but is preferred by the authors, is to carefully measure the volume of an excavation and weigh all the material extracted. The tonnage factor is simply the volume of the excavation in cubic feet divided by the weight of the material recovered in tons. In reality, this bulk density procedure provides the most accurate tonnage factor possible for bulk material sampled. Other procedures do not allow for differences related to void spaces such as fractures.

Problems are encountered with use of a constant tonnage factor where the deposit changes character in different portions of the body. In such cases, separate tonnage factors for each portion of the body must be calculated. For massive sulfide deposits and replacement deposits with simple mineralogy it is often possible to prepare a nomograph relating tonnage factor to assay data. The tonnage factor used for ore is keyed to changes in the ore content and grade.

Engineering Considerations

Before proceeding with an explanation of various methods of reserve computation, a brief discussion of pertinent engineering factors is in order.

Geological Considerations. In many instances, particularly in the exploration stage of a project, it is common practice to project ore extensions based on geologic inference. These projections should never be extended across geological discontinuities such as faults, contacts, unconformities, fold axes, etc. until positive ore correlation data are available on both sides of the discontinuity. Preliminary drilling and other sampling will give an indication of the nature of ore boundaries, whether they are sharply defined or gradational. Lateral or vertical mineralogical zonation, development of discrete ore shoots, and other potential problems will become apparent as exploration work progresses.

Mining and Metallurgical Recovery. Once the deposit is reasonably well defined as to its limits, shape, and character, consideration can be given to selection of an appropriate mining method, and then an estimate can be made concerning percentage extraction from the deposit. The portion of mineralization above the cutoff value that can actually be exploited constitutes the *minable ore reserve*.

Metallurgical recovery may be a critical factor in the economic analysis of a property. However, it usually is used only in calculation of a cutoff grade during reserve computation for a deposit. Any increase or decrease in metallurgical recovery has the same effect as an increase or decrease in grade of ore mined.

Dilution. Dilution is the unavoidable extraction of barren or below cutoff grade material along with the ore. In vein deposits the most common source of dilution is blasting overbreak in the walls of the deposit. Dilution may be handled in various ways. In veins that have at least one gradational or "assay" wall, it is common to cut samples over the normal mining width and use an average grade and tonnage factor for the entire interval. In the case of narrow veins with sharp boundaries, when wall rock must be taken with the vein, the dilution may be calculated as follows:

Ore block:	30.48 × 15.25 m (100 × 50 ft)		
Average width:	0.6 m (2.0 ft)		
Average grade:	10.0% Pb		
Tonnage factor:	9.0 cu ft per ton ore		
	12.0 cu ft per ton wall rock		
Minimum mining width:	0.9 m (3 ft)		
Ore tons:	100 × 50 × 2.0 ÷ 9	=	1111 tons
Waste tons:	100 × 50 × 1.0 ÷ 12	=	417 tons
	Total tons mined	=	1528 tons
Grade:	1111 × 10.0%	=	11,110
	417 × 0.0%	=	00,000
			11,110

$$\text{Diluted grade} = \frac{11{,}110}{1528} = 7.27\%\ \text{Pb}$$

In some instances, it is possible for dilution to drop a block of ore grade material below cutoff grade. In such instances, the block economically ceases to be ore and must be left until the cutoff grade is lowered or greater selectivity in mining can be made. One common practice with thin veins is to "double shoot" the vein—the vein is drilled, blasted, and removed, then the wall rock is blasted for fill. However, in such cases the mineral value in the ore mined and milled must cover the cost of blasting the waste material.

As mentioned previously, there are various methods of accounting for dilution in mining. One technique that is not permissible should perhaps be mentioned. Upgrading the average ore grade of a tin deposit when the vein width exceeds the minimum mining width is not acceptable engineering practice.

Cutoff Grade. As stated before, the cutoff grade is the minimum grade that can be mined at a profit. As economic conditions change, the cutoff grade may increase or decrease. It is common practice to compute the ore reserves of a mine for various cutoff grades and plot the results as a series of grade-tonnage curves. These curves should be updated regularly to aid in mine planning.

METHODS OF CALCULATION

In this section, several of the common traditional reserve calculation methods are explained and illustrated by simple examples. The calculation methods discussed are by mining block, by polygons, by triangles, and by section. Many of the other calculation methods presented in the literature are only somewhat sophisticated variations of these methods. No one technique is universally applicable to all deposits. Methods such as mining blocks and sections work well in steeply dipping veins and tabular deposits, whereas polygonal methods have found wide application to disseminated and flat-lying bedded deposits. The method selected for any particular deposit depends upon the geological and engineering elements unique to each deposit, and usually the ore reserves will be calculated several different ways as an internal audit.

Simple computer programs may be written to handle any of the following calculation methods, but all that is really required are the basic data, maps, sections, a hand calculator or slide rule, and considerable patience. More sophisticated computer-oriented methods such as interpolation using inverse distance weighting functions are available but not discussed in this presentation. A review of geostatistical methods is provided as an appendix to this chapter.

Calculation by Mining Block

Figure 1.6 shows the method of estimating the tonnage and grade of an ore block in a vein-type mine. Samples have been assumed to be cut at regular intervals. The vein in this area is assumed to exceed the minimum mining width of 0.91 m (3 ft). The method of calculation is simply an extension of the method of sample weighting illustrated in Fig. 1.3.

Figure 1.6 represents a block of ore between two levels 30.5 m (100 ft) apart, and coordinates 00E and 250E. The upper figure in the block is the average

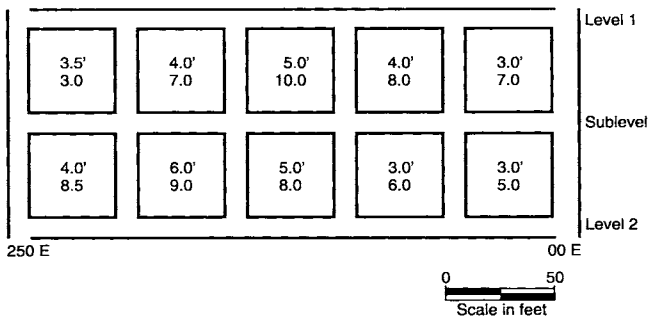


FIGURE 1.6 Longitudinal projection of an ore block, metric equivalent: ft × 0.3048 = m

TABLE 1.5 Calculation of ore blocks

Length		Width		Thickness	Tonnage Factor		Tons*		Grade		Tons-Grade
50	×	50	×	3.5	9	=	972	×	5	=	4,861
50	×	50	×	4.0	9	=	1,111	×	7	=	7,777
50	×	50	×	5.0	9	=	1,388	×	10	=	13,889
50	×	50	×	4.0	9	=	1,111	×	8	=	8,888
50	×	50	×	3.0	9	=	833	×	7	=	5,833
50	×	50	×	3.0	9	=	833	×	5	=	4,166
50	×	50	×	3.0	9	=	833	×	6	=	5,000
50	×	50	×	5	9	=	1,388	×	8	=	11,111
50	×	50	×	6	9	=	1,666	×	9	=	15,000
50	×	50	×	4	9	=	1,111	×	8.5	=	9,444
							11,246				85,972

Total tons = 11,246

Average grade = $\frac{85{,}972}{11{,}246}$ = 7.64 oz* Ag/ton

*Metric equivalents: oz × 0.02834952 = kg; st × 0.9071847 = t.

thickness and the lower figure is the average grade in ounces of silver per ton. The tonnage factor is 9 cu ft per ton. The average grade and tonnage for the block are computed in Table 1.5.

Calculation by Polygons

The method of calculation by polygons is often used with drill-hole data. Polygons may be constructed on plans, cross sections, or longitudinal sections. The polygons, once constructed and ranked as to class of ore, are planimetered to determine the area of mineralization. The thickness of above cutoff grade mineralization is applied to the entire polygon to establish the volume

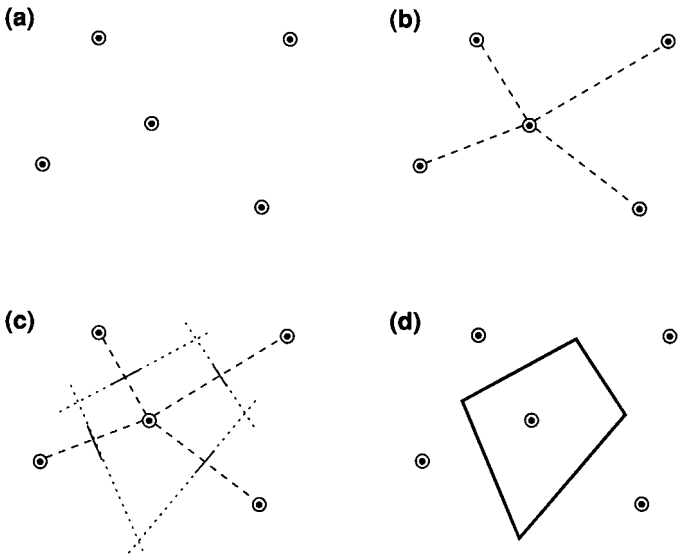


FIGURE 1.7 Construction of polygons (a) Drill-hole plan (b) Connecting lines between drill holes (c) Construction of perpendicular bisectors of connecting lines (d) Construction of final polygon

estimate. In this method, the average grade of mineralization encountered by the sample point within the polygon is considered to accurately represent the grade of the entire volume of material within the polygon.

The construction of polygons is quite simple. The method assumes that the area of influence of any sample point extends halfway to the adjacent sample points. The procedure for construction of polygons is illustrated in Fig. 1.7, which shows five drill holes (a), connecting lines for the drill holes (b), the construction of perpendicular bisectors of the lines between adjacent drill holes (c), and the final polygon (d).

The polygon method makes the basic assumption that the area of influence of a drill hole extends halfway to the next adjacent hole. An alternative viewpoint is that of a circular area of influence for drill-hole intercepts. The concept of circular area of influence about a mineralized intercept can be used to assign the relative classification to the polygon blocks. One important aspect of the circular area of influence for drill holes, shown in Fig. 1.8, is completely covering the area contained within a square grid of holes by the circular area of influence of the holes requiring a drill-hole spacing of (r) $(\sqrt{2})$, where (r) is the radius of influence. This means that although a drill-hole spacing of 60.96 m (200 ft) in a porphyry copper deposit would more than reasonably define proven ore grade mineralization, the radius of the circle of influence for that spacing is in reality (200) $(\sqrt{2})$ = 86.2 m (282.8 ft).

Once the area of influence of the drill holes corresponding to proven and probable ore has been determined, it can be used to check and rank the polygons. A rapid, easy, and acceptable method of ranking polygons is to draw a set of circles that represent the radius of influence for proven, probable, and possible ore reserve categories on a stable base material and then overlay the

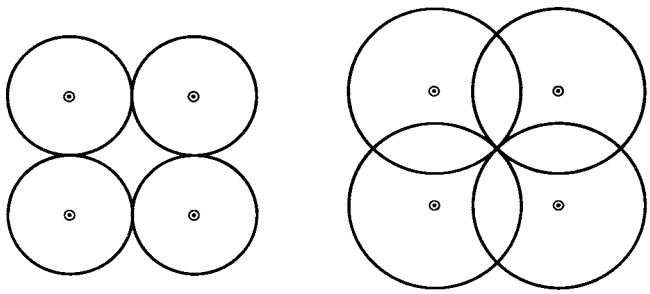


FIGURE 1.8 Polygon classification by circular area of influence. Left, incorrect method; right, correct method.

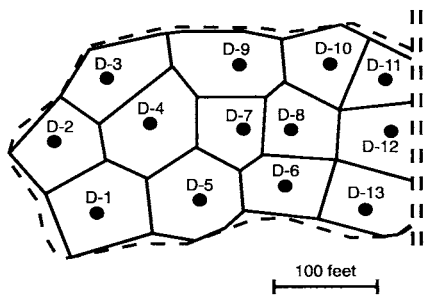


FIGURE 1.9 Diamond drill-hole plan of the Bonanza copper deposit, metric equivalent: ft × 0.3048 = m

finished polygon map over this on a light table. Each drill-hole polygon is superimposed on the center of the "proven" area of influence circle matching the center points. If the polygon falls completely within the "proven" range circle, the polygon is marked as belonging to the proven category. Subsequent checks are made of the polygons not meeting the criteria to be classified as proven blocks using the other area of influence circles until all the polygons are ranked. For record and bookkeeping purposes, polygons are conveniently referenced to the drill-hole number and the section or level being evaluated (e.g., polygon DDH-8-16, level 2080). Figure 1.9 illustrates the computation of an ore block by the polygon method.

The method of calculation by polygons is often used with drill sample data. The method makes the assumption that the area of influence of each drill hole extends half the distance to each adjacent drill hole, with appropriate modifications for known geologic factors such as faults, contacts, or mineralization limits. The areas of the polygons may be measured by planimeter or calculated geometrically by breaking up each polygon into a series of triangles. The average grade and thickness of each drill hole may be determined as shown in Table 1.6 for drill hole D-1. In this example, the mineralization is assumed to be copper, and the cutoff grade is 0.40% Cu.

The average grade and thickness is determined for each drill hole, and the reserves are calculated as shown on Table 1.7. Each polygon is labeled for the contained drill hole.

TABLE 1.6 Assay data for drill hole D-1

Interval, ft*	Thickness, ft	Grade, % Cu	Grade × Thickness	
0–100	100	0.31	0.00	(Below cutoff)
100–110	10	0.47	4.70	
110–122	12	0.73	8.75	
122–130	8	0.96	7.68	
130–150	20	1.04	20.80	
150–200	50	0.82	41.00	
200–220	20	0.54	10.80	
220–250	30	0.42	12.60	
250–270	20	0.35	0.00	(Below cutoff)
	150		106.33	Thickness and grade-thickness above cutoff

$$\text{Average grade} = \frac{106.33}{150.00} = 0.71\%\ \text{Cu}$$

Thickness = 150 ft

*Metric equivalent: ft × 0.3048 = m.

TABLE 1.7 Ore reserves for Bonanza copper deposit

Polygon	Area, A, sq ft	Thickness, T, ft	A × T, cu ft	Tonnage Factor/TF, cu ft*/ton	(A × T)/TF, tons ore	Grade, % Cu	Ton × Grade, ton %
D-1	5,320	150	798,000	12.5	63,840	0.71	45,326
D-2	5,300	135	715,500	12.5	57,240	0.66	37,778
D-3	4,400	180	792,000	12.5	63,360	0.82	51,955
D-4	5,520	175	966,000	12.5	77,280	0.75	57,960
D-5	6,800	155	105,400	12.5	84,320	1.00	84,320
D-6	4,960	180	892,800	12.5	71,424	0.97	69,281
D-7	4,520	250	1,130,000	12.5	90,400	1.21	109,384
D-8	4,640	240	1,113,600	12.5	89,088	1.36	121,159
D-9	5,840	150	876,000	12.5	70,080	0.93	65,174
D-10	4,840	135	653,400	12.5	52,272	0.87	45,476
D-11	3,760	120	451,200	12.5	36,096	0.81	29,237
D-12	4,270	165	637,200	12.5	50,976	0.75	38,232
D-13	4,800	135	648,800	12.5	51,840	0.68	35,251
					858,216		790,553

Tons ore 858,216

Average grade 0.92% Cu

*Metric equivalents: ft × 0.3048 = m; sq ft × 0.09290304 = m^2; cu ft × 0.02831685 = m^3; st × 0.9071847 = t.

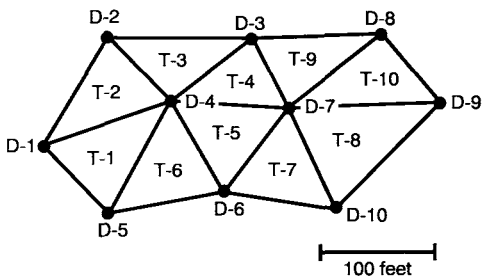


FIGURE 1.10 Diamond drill-hole plan of the Ojala copper deposit, metric equivalent: ft × 0.3048 = m

Computation by Triangles

Another method of computing reserves is a modification of the polygon method. In this method a series of triangles is constructed with the drill holes at the apices. This method has the advantage in that the three points are considered in the calculation of the thickness and grade parameters for each triangular reserve block. The construction and calculation of ore reserves by use of triangles are shown in Fig. 1.10.

The method of calculation by triangles is a modification of the polygonal method in which the drill area is divided into triangles by connecting adjacent drill holes with construction lines. This method has the advantage that the areas are easily calculated by geometry or by coordinates by use of a computer or programmable calculator. The thickness above cutoff grade and average grade are calculated for each drill hole, as illustrated in the previous example. For the purposes of this problem, each drill hole is presumed to have the average grades and thicknesses shown in Table 1.8.

In like manner, the tonnage and grade for each triangle can be computed, and Table 1.9 can be constructed.

Calculation by Section

The basis of this method is to calculate a block of ore that is bounded by regularly spaced cross sections (see Tables 1.10 and 1.11). The following equation illustrates the detailed calculation of a typical block of ore by the cross section method. The ore outline of each bounding section is divided into areas of influence based on the drill-hole or other sample data. The areas of influence are then either planimetered or calculated geometrically. The individual areas are totaled for each section and the volume calculated by the average and area formula:

$$V = \frac{A_1 + 2A_2 + 2A_3 + \ldots + 2A_{n-1} + A_n}{2} \times L$$

where A_n is area of section n and L is a constant section spacing, or when using only two adjacent sections:

$$V = \frac{(A_1) + (A_2)}{2} \times L$$

TABLE 1.8 Assay data for the Ojala copper deposit

Drill Hole No.	Thickness, ft*	Average Grade, % Cu
D-1	50	0.93
D-2	75	0.77
D-3	60	0.82
D-4	100	1.05
D-5	75	0.72
D-6	60	0.49
D-7	105	1.63
D-8	80	0.91
D-9	70	0.86
D-10	75	0.74

Given these data, the tonnage and grade calculation for Triangle T-1 would be as follows.

Area = 4400 sq ft* (by geometry)

Average Grade-Thickness for Triangle T-1

Drill Hole	Thickness, ft	Average Grade, % Cu	Grade × Thickness, ft %
D-1	50	0.93	46.50
D-4	100	1.05	105.00
D-5	75	0.72	54.00
	225		205.50

$$\text{Average grade} = \frac{205.50}{225.00} = 0.91\%\ \text{Cu}$$

$$\text{Tonnage} = \text{area} \times \text{average thickness} \times \text{tonnage factor}$$

$$= 440 \times \frac{225}{3} \times \frac{1}{12.5} = 26{,}400\ \text{st}^*$$

*Metric equivalents: ft × 0.3048 = m; sq ft × 0.09290304 = m^2; st × 0.9071847 = t.

The volume is then converted to tons by application of the appropriate tonnage factor.

Figure 1.11 shows two cross sections spaced 30.48 m (100 ft) apart. These sections show a tabular dipping vein sampled by a surface trench, two drill holes per section, and one crosscut per section. The vein is assumed to be copper ore with a tonnage factor of 9.5 cu ft per ton.

TABLE 1.9 Ore reserves for the Ojala copper deposit

Triangle	Drill Holes	Tons* Ore	Average Grade	Tons × Grade
T-1	D-1, D-4, D-5	26,400	0.91	24,024
T-2	D-1, D-2, D-4	26,400	0.94	24,816
T-3	D-2, D-3, D-4	22,500	0.91	20,475
T-4	D-3, D-4, D-7	22,260	1.23	27,380
T-5	D-4, D-6, D-7	18,550	1.15	21,332
T-6	D-4, D-5, D-6	27,260	0.79	21,535
T-7	D-6, D-7, D-10	26,240	1.07	28,076
T-8	D-7, D-9, D-10	40,500	1.15	46,575
T-9	D-3, D-7, D-8	24,418	1.20	29,301
T-10	D-7, D-8, D-9	28,917		34,411
		263,445		277,927

Tonnage = 263,445 st

$$\text{Average grade} = \frac{277{,}927}{263{,}445} = 1.05\%\ \text{Cu}$$

*Metric equivalent: st × 0.9071847 = t.

TABLE 1.10 Assay data for Section 100N, Big Rat copper vein

Sample No.	Area of Influence, sq ft*	Grade, % Cu	% Cu × sq ft*
T-1	A = 510	0.80	408
DDH-1	B = 1000	2.55	2550
DDH-2	C = 1040	1.66	1726
C-1N	D = 710	1.70	1207
	3260		5891

Total Area = 3260 sq ft*

$$\text{Average grade} = \frac{5891}{3260} = 1.81\%\ \text{Cu}$$

*Metric equivalents: sq ft × 0.09290304 = m^2; cu ft × 0.02831685 = m^3.

TABLE 1.11 Assay data for Section 200N, Big Rat copper vein

Sample No.	Area of Influence, sq ft*	Grade, % Cu	% Cu × sq ft*
T-2	A' = 848	0.92	780
DDH-4	B' = 1792	2.32	4157
DDH-3	C' = 1280	1.59	2035
C-2N	D' = 976	1.63	1591
	4896		8563

Total area = 4896 sq ft*

$$\text{Average grade} = \frac{8563}{4896} = 1.75\%\ \text{Cu}$$

*Metric equivalents: sq ft × 0.09290304 = m^2; cu ft × 0.02831685 = m^3.

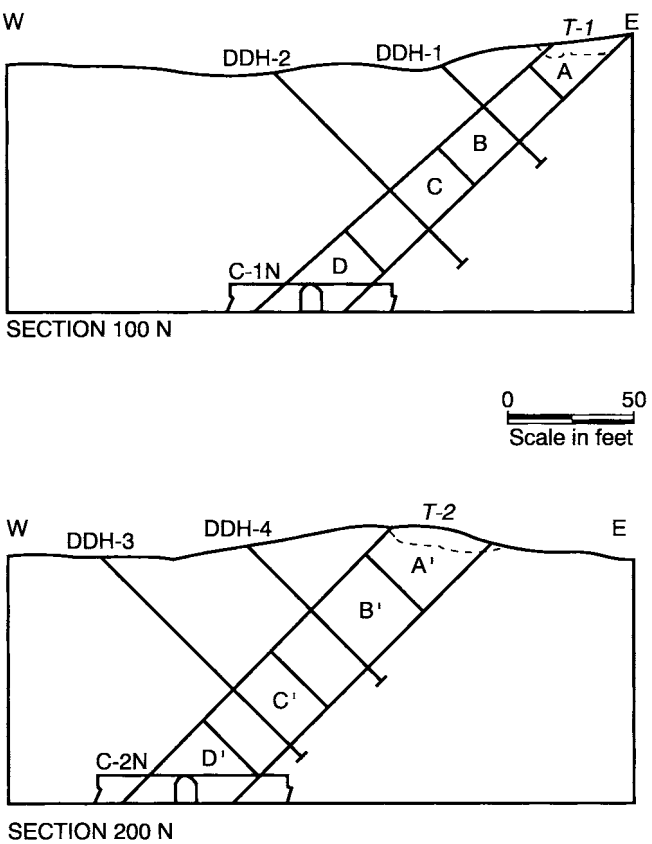


FIGURE 1.11 Cross sections of the Big Rat copper vein, metric equivalent: ft × 0.3048 = m

Block Average Grade:

Section	Area, sq ft	Average Grade, % Cu	% Cu × ft²
100N	3260	1.81	5901
200N	4896	1.75	8563
	8156		14,463

$$\text{Average block grade} = \frac{14{,}463}{8156} = 1.77\%\ \text{Cu}$$

Volume of Ore Block:

$$v = \frac{\text{area section 100N} + \text{area section 200N} \times \text{section spacing}}{2}$$

$$v = \frac{3260 + 4896 \times 100}{2} = 407{,}800\ \text{cu ft}$$

$$\text{Tonnage} = \frac{407{,}800}{9.5} = 42{,}926\ \text{st}$$

The geologic reserve of this deposit between 100N and 200N is 38,934 t (42,926 st) with an average grade of 1.77% Cu. Similarly, the reserve calculations can be extended north and south to cover the entire minable strike length of the vein by adjacent pairs of sections.

Documentation

It is extremely important, no matter what ore reserve estimation methodology is used, to carefully document the method used and the limits applied. If the method is documented, anyone using the reserve figures will have a fuller understanding of the confidence limits inherent in the stated reserves. Also, in case of any change of mine personnel, it will be possible for future calculations to be consistent with previous reserve data. Adherence to this policy also allows for future critical review of the applicability of the methodology to the specific deposit. Should a problem be encountered with the application of the reserve estimation method to the deposit, it may be possible to correct the problem without completely recalculating the entire reserve inventory.

Writing up the reserve estimation methodology also forces the staff to fully identify potential problem areas and to define methods of handling such problems in a consistent manner. The documentation should be prepared in the form of a manual keyed to the specific property by examples. Adequate space should be provided for notes to be added by the staff as problems or questions arise or ideas for improving the reserve calculation method are encountered. Such an ore reserve preparation manual provides each new staff member with a set of uniform procedures and guidelines to be used on the specific property.

The cutoff limits used—those related to both mining method and grade—should be specified in the reserve statements. Examples of such qualifying statements include:

proven 3,795,000 tons averaging 2.15% Cu

probable 5,600,000 tons averaging 2.69% Cu

possible 2,901,000 tons averaging 2.80% Cu

based on a minimum mining width of 3.04 m (10 ft) of material above a cutoff grade of 0.80% Cu.

Note (1) all reserves in the "possible" category are located below the 3500 level, (2) all reserve figures are reported to the nearest 1000 st, and (3) additional inferred reserves exist at depth below current mining limits. These reserves presently are poorly defined and hence are not reported.

REVIEW OF GEOSTATISTICAL METHODOLOGY

Introduction

Geostatistics as an ore reserve estimation methodology emerged in France in the early 1960s from the work of Georges Matheron and was based on original studies by Danie Krige in South Africa. The techniques are not merely an amalgamation of the geological sciences with probability theory and classical statistics, but rather an entirely new methodology.

It is not possible within this discussion to give a comprehensive treatment of the theory and application of geostatistics. The objectives here are merely to develop an overview of this methodology, to elaborate on some of its virtues and problems, and to provide some preliminary understanding of the basic techniques and nomenclature.

The primary purpose of any natural resource estimation method is to reliably estimate the overall ore reserves and the distribution of in situ and recoverable tonnages and grades throughout the deposit. Conventional methods (i.e., area of influence or polygonal, other geometrical, distance weighting, etc.) may provide a good total or global estimate of an orebody's reserves. However, a good geostatistical reserve study with careful attention to geologic controls on mineralization will provide not only a good total reserve estimate, but also a more reliable block-by-block reserve inventory with an indication of relative confidence in the block grades estimated.

Obviously, geostatistical methods, like any others, cannot increase the quantity of basic sample information available, nor can they improve the quality or accuracy of the basic assays. However, geostatistics properly applied does derive from the raw data the best possible estimates of orebody parameters. This is particularly critical with marginal grade deposits, in which a relatively minor change in anticipated mill head grade can have quite dramatic effects on the project's profitability. Geostatistical techniques should be regarded as a

comprehensive suite of ore reserve estimation tools, which, if they are appropriately understood and used, should generally lead to few surprises when the mine comes into production.

Classical statistics and tests have been used in ore reserve evaluation for many years. These methods do assume, however, that samples taken from an unknown population are randomly selected and are independent of each other. In the context of an orebody, this implies that the position from which any sample was taken is not important. Theoretically, using classical statistics, taking samples on opposite sides of an orebody would be just as good as taking them a short distance apart. Sample assays taken from holes drilled in close proximity to one another, within an orebody, obviously should not be random or independent. Closely spaced samples should demonstrate some correlation or, in other words, reflect some degree of continuity in the mineralization. If this is not the case, there is either no continuous orebody or the samples have been taken over an excessively large spacing.

Despite the limitations of classical statistics in ore reserve estimation, much can be gained from studies of sample distributions in terms of providing estimates of the overall or global orebody parameters (i.e., total ore tonnage, grade, and metal content). Serious mistakes, however, can occur if this theory is applied on a small scale. Even in the global case, one must exercise considerable caution in making predictions from sample grade distributions of recoverable ore tonnages and grades above a particular cutoff grade. This is necessary primarily because drilling pattern irregularities are very common, and hence the grade distribution may be biased (high-grade areas are typically drilled out more thoroughly than low-grade areas). Secondly, the variance or dispersion of basic sample assay values is much greater than the variance of large blocks, which can be selectively mined as ore or waste.

Unlike classical statistical approaches, geostatistics recognizes that samples in an ore deposit should be spatially correlated with one another, and that nearby samples will probably not be independent. The techniques are based on Matheron's idea of "regionalized" variables or, in other words, variables that are associated with both a volume (called a "support" in geostatistics) and a position in space. Thus, geostatistical methods utilize an understanding of the interrelationship of assay data within various geologic ore control regimes of a deposit. Geostatistics thus represents a major advance in ore reserve estimation technology.

The relationship between sample variance and size (decreasing variance with increasing size of the support) is utilized in developing estimates of recoverable ore reserves given a certain degree of mining selectivity and given a cutoff grade. The amount of spatial correlation or continuity is determined by the primary geostatistical tool—the variogram.

Variograms, which represent the mineralization's characteristics, are a prerequisite to any geostatistical ore reserve estimation. If the sampling density is too low for any underlying correlation to be detected, if the orebody is extremely homogeneous, or if poor sample collection, preparation, and assaying procedures were used, then no structure or continuity will be visible in the

variogram. In this case, geostatistical methods become those of classical statistics, where samples can be assumed independent and the best estimate of any block of ground is the average grade of all samples within the entire deposit. Under these circumstances, reliably predicting which part of the deposit is of ore grade and which is waste is not possible with any acceptable level of confidence.

Geostatistics theory utilizes somewhat foreign terminology and relatively advanced statistical concepts involving Lagrange multipliers, triple integration, etc. In addition, early publications in this field were in French, and even those in English were not readily comprehensible to the average geologist or mining engineer. Because of these factors, geostatistics has sometimes been regarded skeptically as a method suited only to computers manned by systems analysts, who probably have little real appreciation for the geology or mining implications of the orebody they are studying.

Although the "black box" computer approach to geostatistics has been historically true at some operations, it should not, and need not, exist. Certainly geostatistics does entail a relatively large amount of computation, involves somewhat "heavy" mathematical and statistical terminology, and generally requires the use of a computer. This, however, does not mean that the work need be done in isolation from the geologic staff. In fact, geologic cross sections, bench plans, and most important, the acquired understanding of the orebody by the geologic staff in terms of the lithologic, structural, or other controls on the mineralization, is of paramount importance in any geostatistical study.

A geostatistical ore reserve study will generally entail the following steps:

- Study of the potential geologic controls on the grade of mineralization, and any zoning of the deposit through geologic interpretation, with the aid of statistical analysis correlating grade with host rock type, alteration type, fracture density, etc.
- Computation of variograms within each geologic zone, interpretation of any differences in the continuity of mineralization in different directions (anisotropy), and selection of suitable variogram models for use in kriging.
- Division of the orebody into three-dimensional matrices of blocks, panels, or cells, and estimation of the in situ grade and the estimation error for each block from the surrounding sample values using kriging (in situ or geologic reserves). Properly selected limits on block estimation errors allow each block to be classified subsequently into the standard "measured," "indicated," and "inferred" categories.
- Study of the distribution of sample values, selection of suitable models, and estimation of the tonnage proportion of each in situ block and its average grade that can be recovered above a given cutoff grade with a given degree of mining selectivity (recoverable reserves at a particular cutoff grade).
- Printing recoverable grade distribution plans on a convenient scale by each level or bench for mine planning purposes.

The Variogram

Computation of variograms within like geologic zones is the first step in any geostatistical ore reserve study. Variograms are used in all subsequent phases, including kriging. Even if kriging is not undertaken, a variogram study is of invaluable assistance in quantitatively defining the traditional concept of "area of influence" of a sample within a particular deposit. Moreover, it can be used in determining the optimum drill-hole spacing to define the reserves at a particular level of confidence.

A variogram, if properly determined within similar geological zones, numerically describes how an orebody parameter (i.e., grade, thickness, accumulation, bedrock elevation, etc.) is spatially correlated. It expresses the similarity or dissimilarity of parameter values as it changes with distance.

The mathematical formulation of a variogram function is as follows:

$$\gamma(h) = \frac{1}{2n} \sum_{t=1}^{n} [Z(x_i) - Z(x_i + h)]^2$$

where $Z(x_i)$ is the value of the regionalized variable (e.g., grade) at point x_i, $Z(x_i + h)$ is the grade at another point at a distance (h) from the point, x_i, and n is the number of sample pairs.

The sample pairs are each oriented in the same direction [hence the vector notation $\gamma(h)$], are each separated by the same distance (h) in meters, and are of equivolume (i.e., constant variogram support).

In practice, the squared differences between all pairs of sample grades at a distance (h) apart are averaged. The process is repeated for different values of the distance (h), and the half averaged squared differences in grade, called $g(h)$, are then plotted against the corresponding distance (h). The resulting graph is often known as an experimental variogram, because it is based only on samples, and is aimed at representing the true underlying variogram of the deposit.

Often there is evidence of a relationship between mean grade and variance, that is, high variances are associated with high-grade material, known as the "proportional effect" in geostatistics. If this is found to be the case, a so-called "relative" variogram should be computed. This involves dividing each $\gamma(h)$ value by the square of the mean of those samples used to calculate that value. Alternatively, variograms can be computed on the logarithms of grade.

Commonly, values of $\gamma(h)$ increase steadily with increasing distance and reach a limiting or plateau level, as indicated in Fig. 1.12. The variograms in Fig. 1.12 demonstrate that nearby samples have similar grades and hence low variances, whereas others farther apart are quite different. This is an expression of the continuity in mineralization. An experimental variogram permits the interpretation of several characteristics of the mineralization as follows:

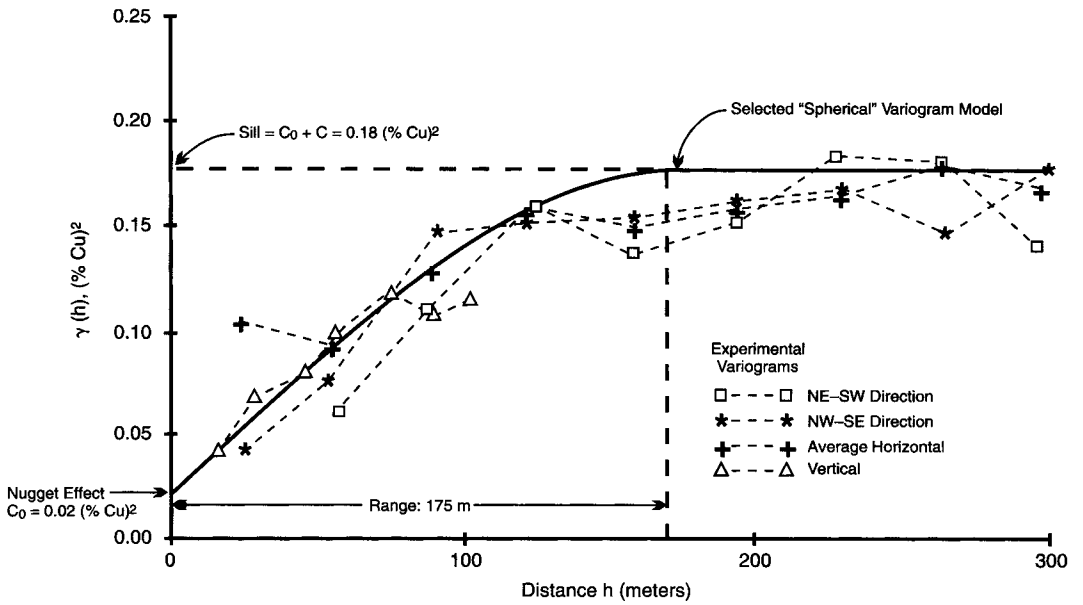


FIGURE 1.12 Variograms for the secondary enrichment zone of a porphyry copper deposit based on a 15-m bench composite support

Range. The distance at which the variogram levels off at its plateau value. This reflects the classical geologic concept of an area of influence. Beyond this distance of separation, sample pairs no longer correlate with one another and become independent.

Sill ($C + C_0$). The value at which the variogram function plateaus. For all practical purposes, the sill is equal to the variance of all samples used to compute the variogram. The difference between this value and the value of the variogram at distance (h) can be thought of as a level of certainty that exists when a sample assay is extrapolated ("extended," in geostatistics terms) over this distance. In other words, as the range is approached, the estimation variance based on extending sample grades increases to a maximum level of this sill.

Nugget Effect (C_0). The variogram value $\gamma(h)$ at distance $h = 0.0$ m. It expresses the local homogeneity (or lack thereof) of the deposit. High nugget effect values relative to the sill can indicate that either the mineralization is poorly disseminated (i.e., tends to be concentrated in pockets or lenses), that the zone on which the variogram was computed was severely disjointed (e.g., major postmineralization structural discontinuities that exist in the deposit have been ignored), or that poor sample preparation and assaying procedures were used. The latter item is called a "human" nugget effect.

Directional Anisotropy. Simply denotes whether the mineralization has greater continuity in a particular direction compared to another. The characteristic is analyzed by comparing the respective ranges of experimental variograms computed along different directions. Fig. 1.12 illustrates that for the

deposit studied the mineralization was essentially isotropic, because the experimental variograms in alternate directions were very similar.

In practice, because drilling grids are rarely uniform, variograms are computed with a latitude on distance (i.e., half average squared differences between grades of sample pairs at a distance $h \pm \Delta h$ apart) and a spread or "window" on direction (i.e., all pairs falling within a direction of $\phi + \Delta\phi°$). Sample pairs are thus grouped into classes of Δh. This class size, the "window" angle on direction, and the sample or support size are very important parameters to choose correctly. They are often varied until a variogram showing the best structure is obtained.

To achieve reliable results, variograms should be computed on geologically similar zones of mineralization, or zones for which the mineralization had the same apparent genesis and for which there is no internal postmineralization faulting. If structurally continuous zones can be isolated, variograms can be computed independently for each, and then the individual results accumulated to produce an average variogram representative of the total mineralization. If major discontinuities are ignored, irregular distance distortions could occur that might lead to erratic or "noisy" variograms.

Poor variograms will also occur if there is only a limited amount of sample data, if the data are from a widely spaced drilling program, or, finally, if the support chosen is too small. Obtaining good representative variograms involves a considerable amount of computation, and interpretation by experienced geologists and geostatisticians. However, these efforts are amply rewarded through an enriched understanding of the deposit under study. An appreciation of the continuity in mineralization early in an exploration program is made possible by drilling a series of closely spaced "crosses" of holes, which can then be used to compute variograms. Such studies can subsequently be used to assist in planning an appropriate drilling grid that will permit ore reserve estimation to a certain required level of precision.

Ore Reserve Estimation Using Kriging

Kriging, the geostatistical grade interpolation method, calculates the grade of a block or panel as a linear combination of the grades of the nearest samples. The coefficients of such a linear combination are obtained indirectly from the variogram. The method will yield the best estimates possible for in situ block grades, particularly where the sampling grid is very irregular and where the continuity in the mineralization is very different in alternate directions. Unlike other methods, kriging also gives a confidence level on each block estimate and on the overall reserves.

Given a series of spatially distributed drill holes or other samples, any ore reserve estimation method must obtain estimates for the average grade of blocks or panels between these drill holes. This is accomplished by some form of interpolation or extension technique, typically on a level-by-level or section-by-section basis. Because these are sample assays and not block assays, some error will be made in this estimation process. The "best" method will be one that keeps the errors as low as possible. Expressed in another way,

if Z and Z^* are the true and estimated block grades, the variance of differences (Z–Z^*) for all blocks must be minimized.

Kriging is simply a linear estimation method that develops optimal weights to be applied to each sample in the vicinity of the block being estimated. It uses both the position of the samples with respect to the block and the continuity of mineralization in different directions as portrayed by selected variogram models. The kriging estimator has the following general form:

$$Z^* = \sum_{i=1}^{n} \lambda_i x_i = \lambda_1 x_1 + \lambda_2 x_2 + \lambda_3 x_3 + \ldots + \lambda_n x_n$$

where Z^* is the estimated in situ block grade, x_i is the sample grade in the vicinity of the block, λ_i is the weighting coefficient assigned to each respective x_i, and n is the selected number of nearest neighbor samples that will be used to estimate the block grade.

Even though the true block grade, Z, is unknown, the estimation variance [i.e., variance of all ($Z - Z^*$) errors] for such a linear combination of sample grades can be expressed as a function only of the variogram and of the weights λ_i. Therefore, having obtained a variogram(s) for a deposit, it is then possible to evaluate the estimation variance—and hence the confidence limits—that could be expected for any drilling grid and for any linear estimation method of determining block grades. This feature of analyzing the precision to be expected in estimating blocks of a given size and orientation using samples in a particular pattern around the block is unique to geostatistics. Classical statistical methods estimate confidence limits for a particular drillhole spacing based on the standard error of the mean, an approach that is valid only if samples are independent of one another, such as in an initial widely spaced exploration grid.

Returning to the equation of the linear estimator, it should be noted that an interpolation method involving inverse distance weighting is also of this form where the λ_i coefficients become:

$$\lambda_i = \frac{1/d_i^e}{\sum_{i=1}^{n} 1/d_i^e}$$

where d is the distance of the particular sample to the center of the block, and e is the selected distance weighting exponent.

Kriging is not directly linked with distance as are distance weighting methods. Sample-to-block distances are calculated, but only as a means to determine the corresponding sample-to-sample and sample-to-block covariances from the variogram.

Kriging utilizes the three-dimensional locations of samples, that is, their distances and directions from the block. Irregular drilling patterns and/or highly anisotropic orebodies are much more reliably studied using geostatistics as compared with other methods that involve only distance weighting.

Kriging involves the optimal selection of the weights λ_i in such a manner that the estimation variance is minimized and such that $\Sigma\lambda_i = 1$. In minimizing this estimation error or variance, kriging results in a series of simultaneous equations. The equations can be solved for each weighting factor, λ_i, given the position of the sample, the size of the block to be estimated, and a model of the variogram representative of the mineralization being studied. The kriging process is quite mechanical, although it may be time-consuming. The calculated λ_i are then multiplied by respective sample grades to give the in situ block grade estimate. The estimation error for each block is, of course, also given. These estimation errors will be higher in regions of low drilling density and lower, as one would expect, where the deposit has been extensively drilled with closer spaced holes.

Linear kriging is the most common of the geostatistical estimating techniques. Nonlinear estimators such as lognormal and disjunctive kriging are available for more advanced studies, particularly relating to the determination of recoverable ore reserves. Disjunctive kriging is substantially more complicated and more demanding of computer resources than linear kriging. In most applications, the use of linear or lognormal kriging to estimate large in situ reserve blocks, followed by an assumption that selective mining units are distributed normally or lognormally within the large blocks, will lead to very good estimates of recoverable reserves. Needless to say, this latter step is vital, because we are trying to estimate the ore tonnage within economic excavation limits and the grade that will be mined, which will be reported to the mill.

Comparison With Other Methods

Understanding geostatistical ore reserve methods is a far more arduous task than understanding the simpler technique involving area of influence type grade assignment used by the polygonal or other geometric methods. Geostatistics, like any of the more complex smoothing methods that utilize many samples in the vicinity of the block rather than only one, can be economically achieved only with a computer. Auditing of ore reserve calculations then becomes more difficult, as does one's ability to sensibly compare estimated block grades with the original sample values, because the former have a much reduced variability.

Given these problems and others, no matter what is said about the benefits of geostatistical techniques, there will always be doubts as to whether the added sophistication and cost is worthwhile, particularly in the absence of actual mining experience. During mining, of course, estimated block grades can be compared with reality. Such studies have been done at many operations, and the results offer substantial encouragement to the proponents of geostatistics. With such comparisons, however, additional doubts arise as to the representability of blasthole assays themselves and of their validity as estimators of "true" block grade. Moreover, not all operations can afford the luxury of trial

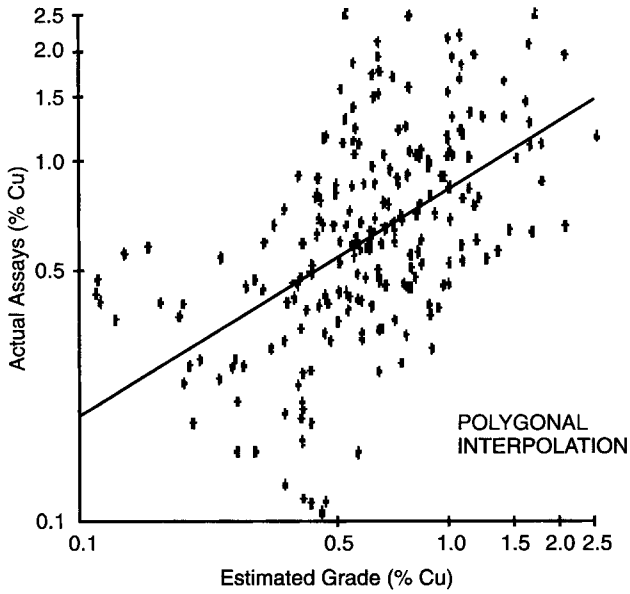
mining to confirm their grade prediction as part of a predevelopment feasibility study. As an alternative, the study described here provides a method of comparing various interpolation methods in the absence of actual mined grades in a quantitative way, rather than in terms of subjective comparisons.

This study was undertaken for the same porphyry copper deposit for which the variograms of Fig. 1.12 were calculated. It involved the comparison of point estimated grades with actual point samples. Drill-hole sample assays were first composited as bench averages weighted by sample length. The composite values were considered as so-called "actual values." Some 200 drill-hole composites were successively removed from the total data set, and the same point in space was estimated from surrounding point assay values using a variety of methods: nearest sample grade assignment (polygonal approximation), inverse distance square interpolation using both horizontal and ellipsoidal searches, and finally linear kriging.

The removed holes were then "redrilled," as it were, and the estimated composite values were compared with true assays through the use of linear regression. The estimation method yielding a regression line that had a slope closest to 0.78 rad (45°) and passed closest to the origin with minimum variability or scatter about the best fit line would be favored as the preferred methodology.

For each estimation method, the differences between the true assay values and estimated grades were also computed and plotted in the form of a histogram of errors. Once again, the best method would be the one that had a mean error of zero and the least variance on this error curve. Figures 1.13 through 1.16 give the results of these studies for each of the methods examined.

Figure 1.13 shows, even for this large, relatively uniform isotropic porphyry copper deposit, that the polygonal method usually did not estimate the true composite grade correctly. In general, the ellipsoidal search inverse distance square approach (Fig. 1.15) gave much better results than the horizontal case (Fig. 1.14). This was expected from the variogram study, which indicated that the mineralization was almost as continuous vertically as it was horizontally. Thus, samples directly above and below a block should have just as much "right" to be used in predicting the block grade as those at the same distance on the same level. Figures 1.15 and 1.16 illustrate that there was a further marginal improvement in adopting linear kriging over the ellipsoidal inverse distance square method. Prior knowledge of the variograms to define appropriate ellipsoidal search dimensions significantly enhances the performance of such methods, as is indicated here. Modern computer algorithms permit the user to vary both the search volume dimensions and the distance weighting exponent independently in each of three orthogonal directions to simulate anisotropic continuity in the mineralization. Under these circumstances, very good results can often be obtained, sometimes at less cost, using this less expensive, ellipsoidal inverse distance weighting approach rather than kriging.



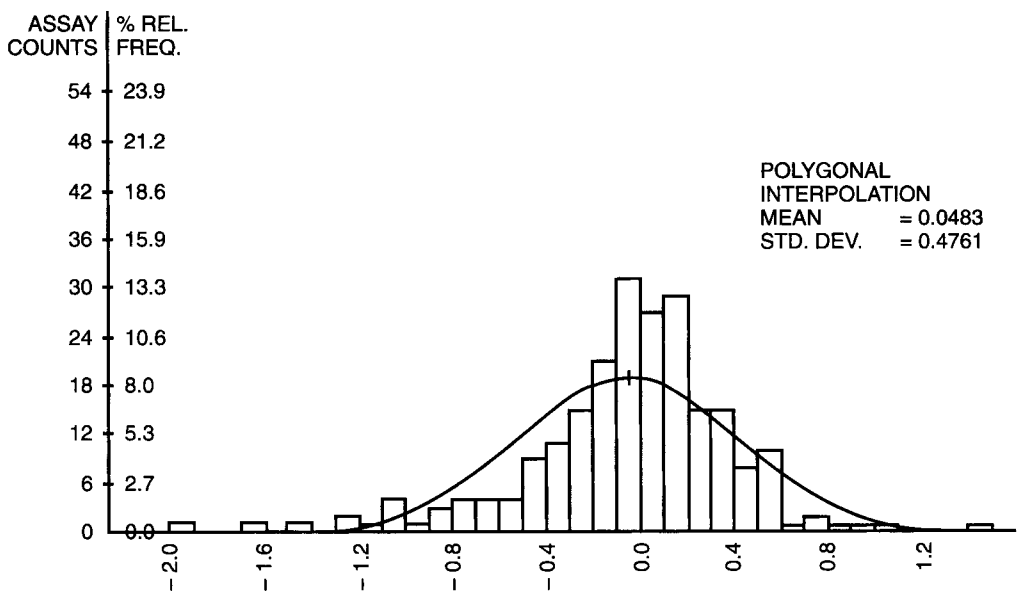


FIGURE 1.13 Scatter diagram and error histogram for the polygonal interpolation method

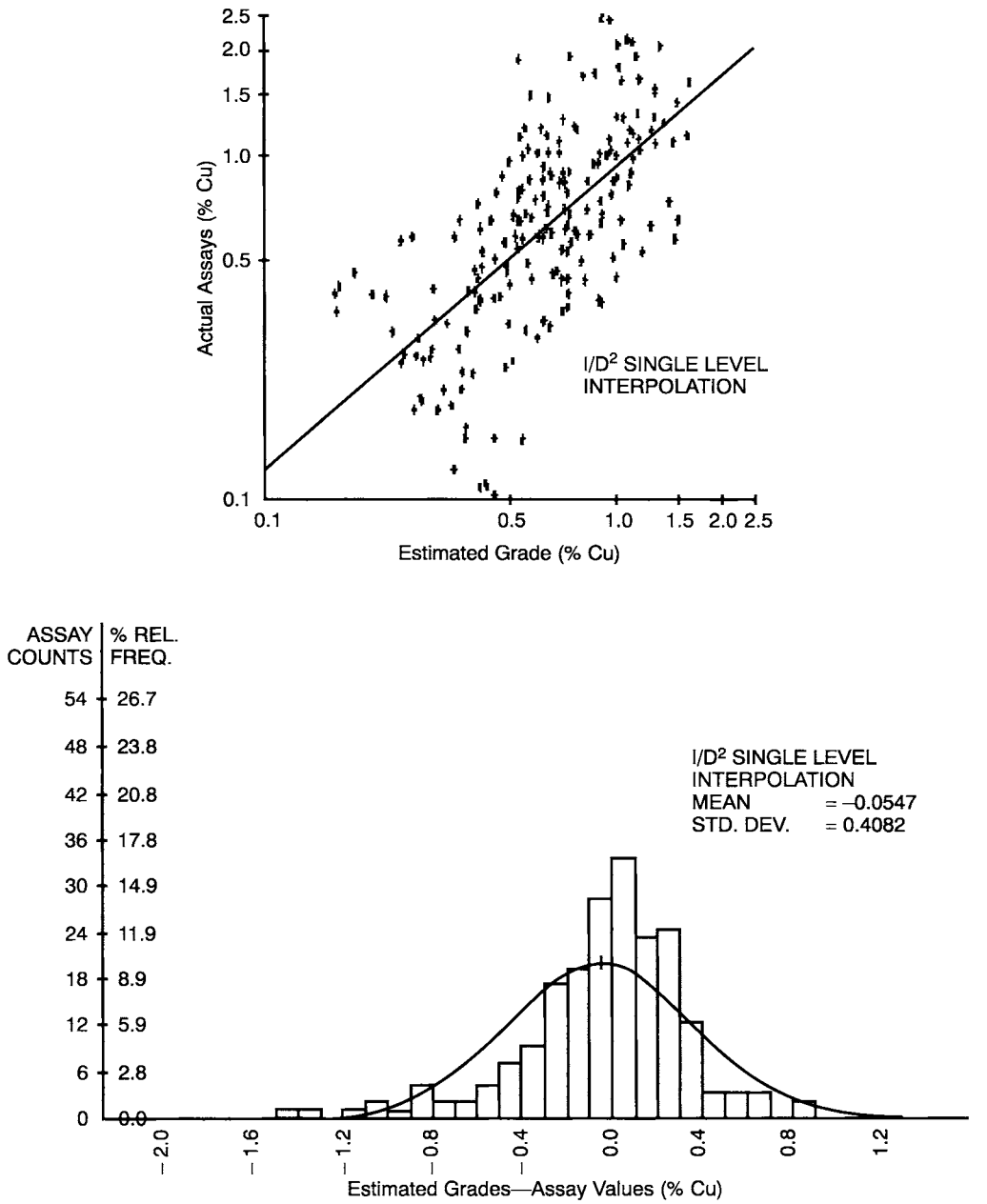


FIGURE 1.14 Scatter diagram and error histogram for the inverse distance square (single level) interpolation method

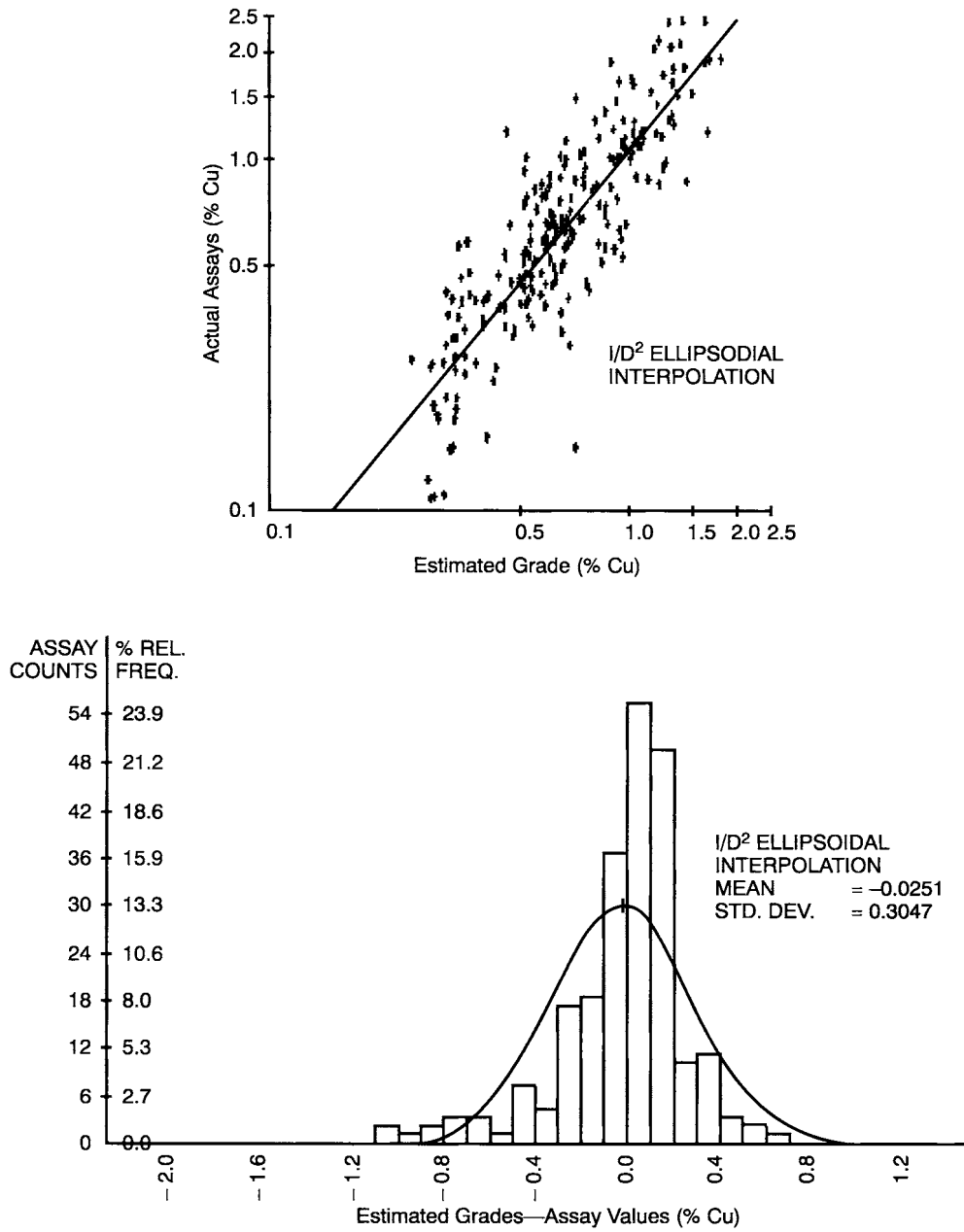
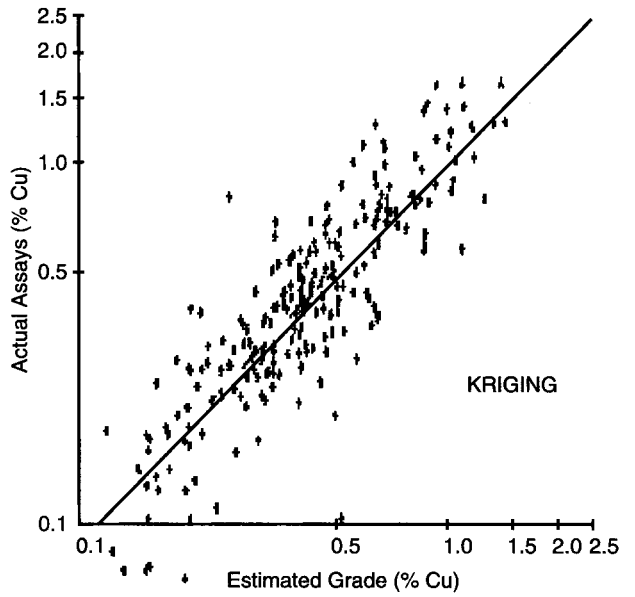


FIGURE 1.15 Scatter diagram and error histogram for the inverse distance square ellipsoidal interpolation method



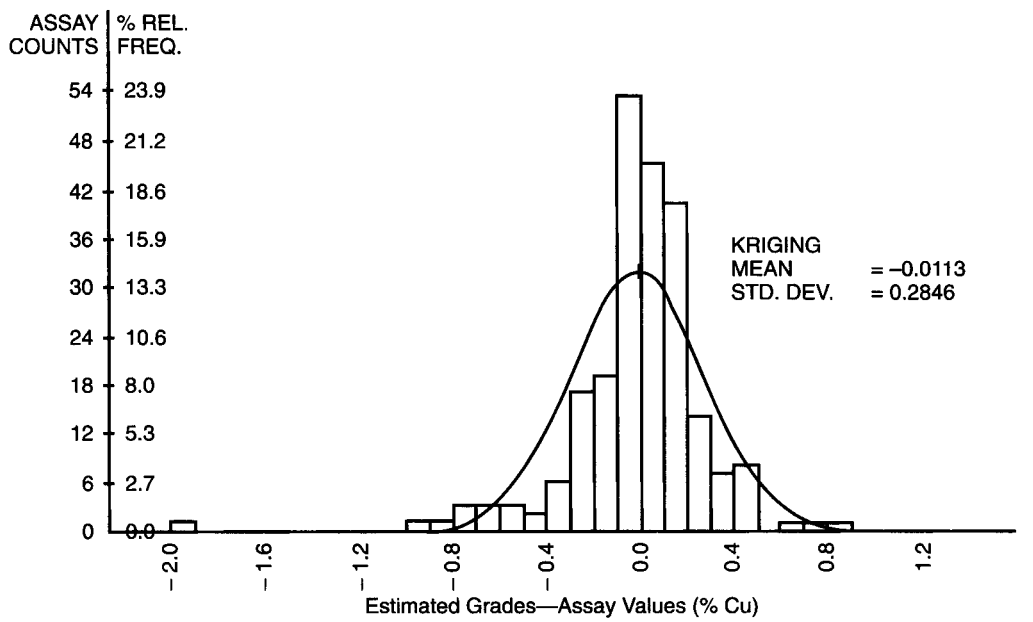


FIGURE 1.16 Scatter diagram and error histogram for the kriging method

Summary Remarks

Geostatistics provides a comprehensive suite of tools that must be tempered by an experienced ore reserve geologist/engineer to obtain reliable estimates—not only for the total ore reserve parameters, but also for the distribution of recoverable grade throughout the deposit for mine planning and scheduling purposes. Specifically, the geostatistical ore reserve methodology, if applied wisely, can yield quantitative information regarding the underlying characteristics of the mineralization, which are often heavily masked by detail. Historically, the economic geologist has had to subjectively define these ore controls and characteristics of mineralization based on experience with similar deposits. Having undertaken this interactive step of geological and variogram modeling of an orebody, one is then in a position to compute block grade estimates through kriging to obtain an estimate of the probable error associated with each block grade, and to compute recoverable reserves based on smaller selective mining units above a certain cutoff grade.

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CHAPTER 2

Choosing an Underground Mining Method

Hans Hamrin

INTRODUCTION

Once an orebody has been probed and outlined, and sufficient information has been collected to warrant further analysis, the important process of selecting the most appropriate method or methods for mining can begin. At this stage, the selection is preliminary only, serving as the basis for a project layout and feasibility study. Later, it may be found necessary to revise the details, but the basic principles for the ore extraction should remain a part of the final layout.

With respect to the basic principles employed, relatively few mining methods are used today. For the purposes of this presentation, the principles are classified as follows:

1. For naturally supported stopes, the available methods are room-and-pillar mining and sublevel stoping.
2. For artificially supported stopes, the available methods are shrinkage stoping, cut-and-fill mining, square-set mining, and longwall mining.
3. For caving methods, the available techniques are sublevel caving and block caving.

Due to the uniqueness of each ore deposit, the variations on each of these methods are nearly limitless. It is impossible to include even the major variations in this chapter; the goal of this chapter is to present the principles as a guide to the mining engineer in adapting the general methods to specific needs. The selection process described herein is intended to supply the techniques by which the candidate methods available for a given orebody can be reduced to one or two feasible approaches. The feasible approaches then can be evaluated in detail, and the particular modifications can be investigated.

In describing costs and economic factors, all values herein are expressed in terms of 1978 U.S. dollars.

BASIC UNDERGROUND MINING METHODS

Before describing the selection of the method or methods applicable to the mining of a particular orebody, the following paragraphs briefly summarize the characteristics of the major mining methods. In part, this material has been adapted from the "Brief Guide to Mining Methods," published by Atlas Copco MCT AB (publication 15073a).

Definitions of Terms

To better understand the material presented herein, some of the more common mining terms are defined in the following paragraphs; Fig. 2.1 further clarifies some of the terms.

adit – An adit is a horizontal or nearly horizontal entrance to a mine.

back – The back is the roof or overhead surface of an underground excavation.

chute – A chute is a loading arrangement that utilizes gravity flow to move material from a higher level to a lower level.

cone – A cone is a funnel-shaped excavation located at the top of a raise, and it is used to collect rock from the area above.

crosscut – A crosscut is a horizontal or nearly horizontal underground opening that is driven to intersect an orebody.

dip – The dip is the angle at which an ore deposit is inclined from the horizontal.

drawpoint – A drawpoint is a place where ore can be loaded and removed. A drawpoint is located beneath the stoping area, and gravity flow is used to transfer the ore to the loading place.

drift – A drift is a horizontal or nearly horizontal underground opening.

finger raise – A finger raise is used for transferring ore. The usual arrangement is as a system of several raises that branch together to the same delivery point.

footwall – The footwall is the wall or rock under the ore deposit (compare dip).

grizzly – A grizzly is an arrangement that prevents oversize rock from entering an ore transfer system. A grizzly usually consists of a steel grating for coarse screening or scalping.

hanging wall – The hanging wall is the wall or rock above an ore deposit (compare dip).

level – A level is a system of horizontal underground workings that are connected to the shaft. A level forms the basis for excavation of the ore above or below.

manway – A manway is an underground opening that is intended for personnel access and communication.

ore – An ore is a mineral deposit that can be worked at a profit under existing economic conditions.

orepass – An orepass is a vertical or inclined underground opening through which the ore is transferred.

prospect – A prospect is a mineral deposit for which the economic value has not yet been proven.

raise – A raise is an underground opening that is driven upward from one level to a higher level or to the surface; it may be either vertical or inclined (compare winze).

ramp – A ramp is an inclined underground opening that connects levels or production areas; ramps are inclined to allow the passage of motorized vehicles. Ramps usually are driven downward.

shaft – A shaft is a vertical or inclined underground opening through which a mine is worked.

slot – A slot is a vertical or inclined ore section that is excavated to open up for further stoping.

stope – A stope is an underground excavation that is made by removing ore from the surrounding rock.

strike – The strike is the main horizontal course or direction of the mineral deposit.

sublevel – A sublevel is a system of horizontal underground workings; normally, sublevels are used only within stoping areas where they are required for the ore production.

wall rock – The wall rock is the wall in which the ore deposit is enclosed.

waste – The waste is the barren rock or the rock of too low a grade to be mined economically.

winze – A winze is a vertical or inclined underground opening that is driven downward from one level to another level or from the surface to a level (compare raise).

Stopes with Natural Support

Naturally supported stopes may be mined either by room-and-pillar mining or by sublevel stoping.

Room-and-Pillar Mining. In the method known as room-and-pillar mining, the orebody is excavated as completely as possible, leaving parts of the ore as pillars to support the hanging wall. The dimensions of the stopes and pillars

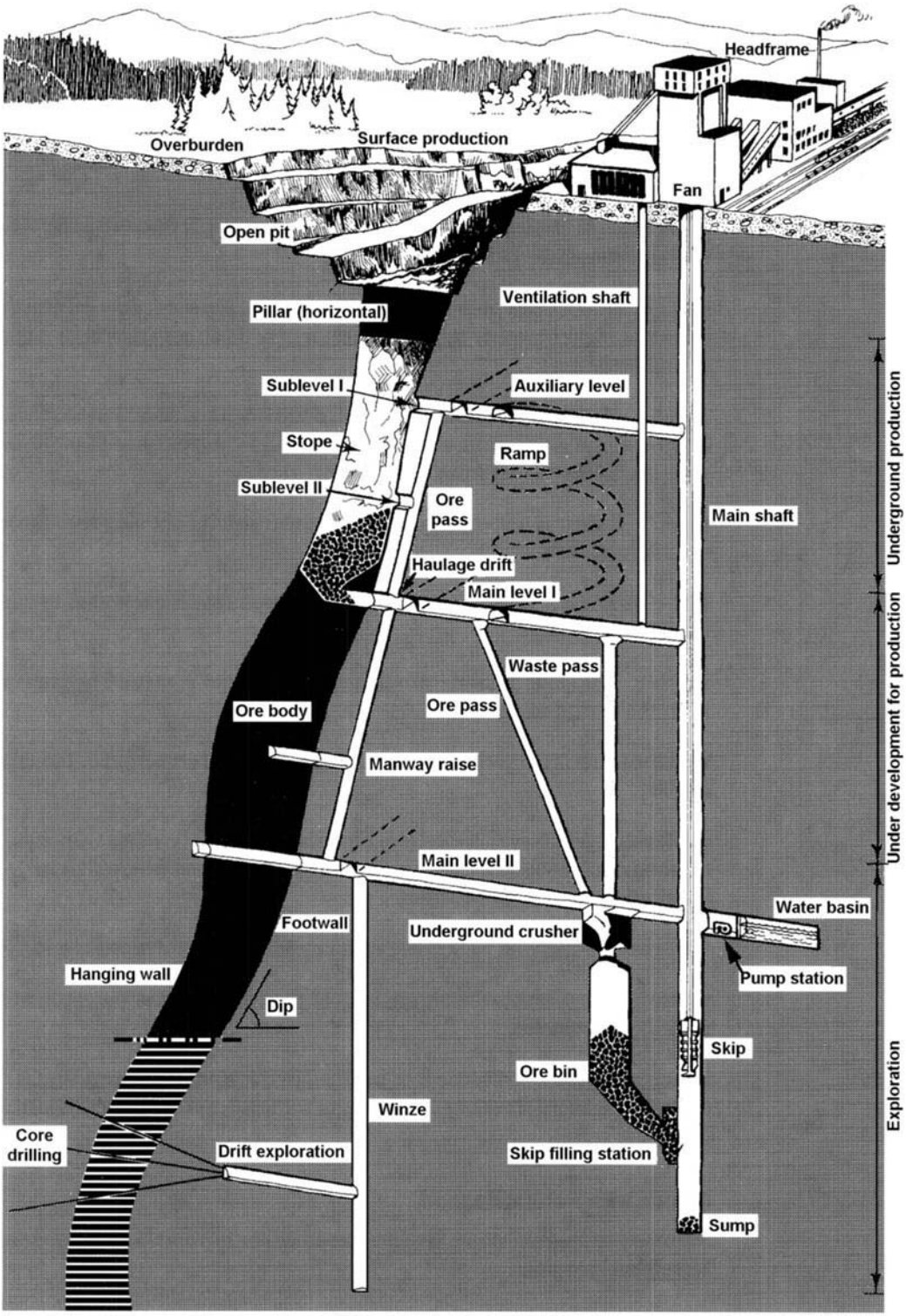


FIGURE 2.1 Sample layout of an underground mine, identifying various mining operations and terms

depend upon factors such as the stability of the hanging wall, the stability of the ore, the thickness of the deposit, and the rock pressure. Generally, the objective is to extract the ore as completely as possible without jeopardizing working conditions or personnel safety. Typically, the pillars are arranged in a regular pattern, and they can be circular, square, or shaped as longitudinal walls that separate the stopes.

Although some of the ore left in the pillars can be extracted by "robbing" as a final operation in the mine, the ore in the pillars usually is regarded as non-recoverable.

Applications. The normal applications of room-and-pillar mining include orebodies with a horizontal or flat dip, normally having an inclination that does not exceed 0.52 rad (30°). To use room-and-pillar mining, the hanging wall and the ore must be relatively competent.

In this respect, the stability of the ore and the hanging wall is a flexible concept. Increasing the number of pillars and reducing the stope width can compensate for poor ground conditions. However, ore recovery is sacrificed, since a larger portion of the orebody is left to support the back. Another method of increasing the stability of the hanging wall utilizes roof bolting, and this method is used extensively in room-and-pillar mining.

Room-and-pillar mining is the only feasible method of mining flat deposits of limited thickness. To a great extent, the method is used for mining bedded deposits of sedimentary origin, including copper-mineralized shales, coal, and many industrial minerals such as limestone, salt, and potash.

Three different systems of room-and-pillar mining may be used. The first and most common system is applicable to horizontal or nearly horizontal deposits. It also is applicable to inclined deposits of greater thickness, with the floor arranged for a moderate slope that allows the use of mobile equipment.

The second system is applicable to orebodies that are inclined from 0.35 to 0.52 rad (20° to 30°). The stoping proceeds upward along the dip. The steep gradient of the stope precludes the use of mobile equipment with this system.

The third system adapts horizontal stoping to an inclined orebody. A special layout of the stopes and a special sequence of ore extraction result in working areas that have moderately sloping floors and that allow mobile equipment to be used in the inclined orebody.

Horizontal Mining. Figure 2.2 illustrates the general approach to room-and-pillar mining of flat or nearly horizontal ore deposits.

Development. In a flat or nearly horizontal ore deposit, minimum development is required prior to mining. Roadways are required for transporting the broken ore and for access between working areas. Often, the required development can be combined with the actual mining operation, and certain mined-out stopes can be utilized as transport routes.

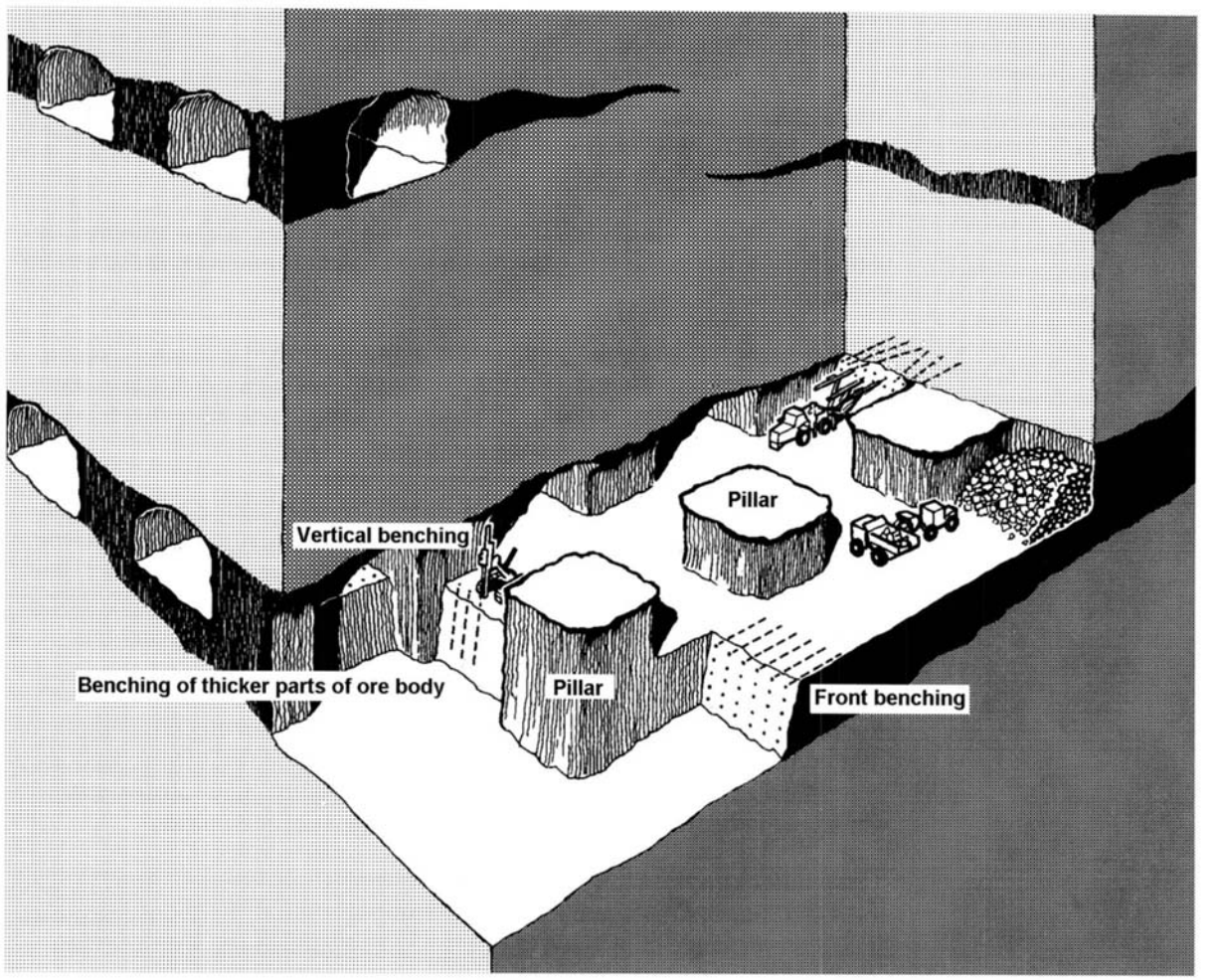


FIGURE 2.2 Room-and-pillar mining of a flat orebody

Production. In horizontal orebodies, the drilling and blasting operation can be compared to drifting and slashing, where the room width and height correspond to the drift dimensions. The available space and access to the rooms determine the mechanized equipment that can be used. Generally, both space and access are ample, allowing the use of large drill jumbos for high productivity.

In thick orebodies, the total thickness often is not excavated in a single step. Normally, extraction starts with the top section; during this phase, roof control (e.g., roof bolting) is accomplished. The remainder of the ore is recovered by benching in one or more steps. Crawler rigs can be used for drilling vertical or inclined holes; the bench also can be drilled from the front with standard drill jumbos.

Ore Handling. The broken ore is loaded directly at the working faces, usually by means of diesel-powered loaders. Various transport systems may be used, depending upon the space available and the distance to be traversed. With sufficient height in the stopes, standard dump trucks of suitable size provide economical ore transportation. For low back heights, special low-profile ore carriers are available. For even thinner orebodies, load-haul-dump (LHD) units may be used.

Because the ore has a large horizontal extent with production at a number of working faces spread over a large area, mining equipment with high mobility is required (drilling, loading, and haulage machinery). On the other hand, the mining layout allows access roads to be arranged easily in the mined-out stopes for rapid movement from one location to another.

Comments. Room-and-pillar mining of a flat or nearly horizontal orebody favors the use of mechanized equipment. Usually, the mine layout is schematic, several production areas can be arranged, and communication and access are simple. These factors allow high utilization of both personnel and machinery in a very efficient mining process.

Inclined Mining. Figure 2.3 illustrates the general approach to room-and-pillar mining of inclined ore deposits.

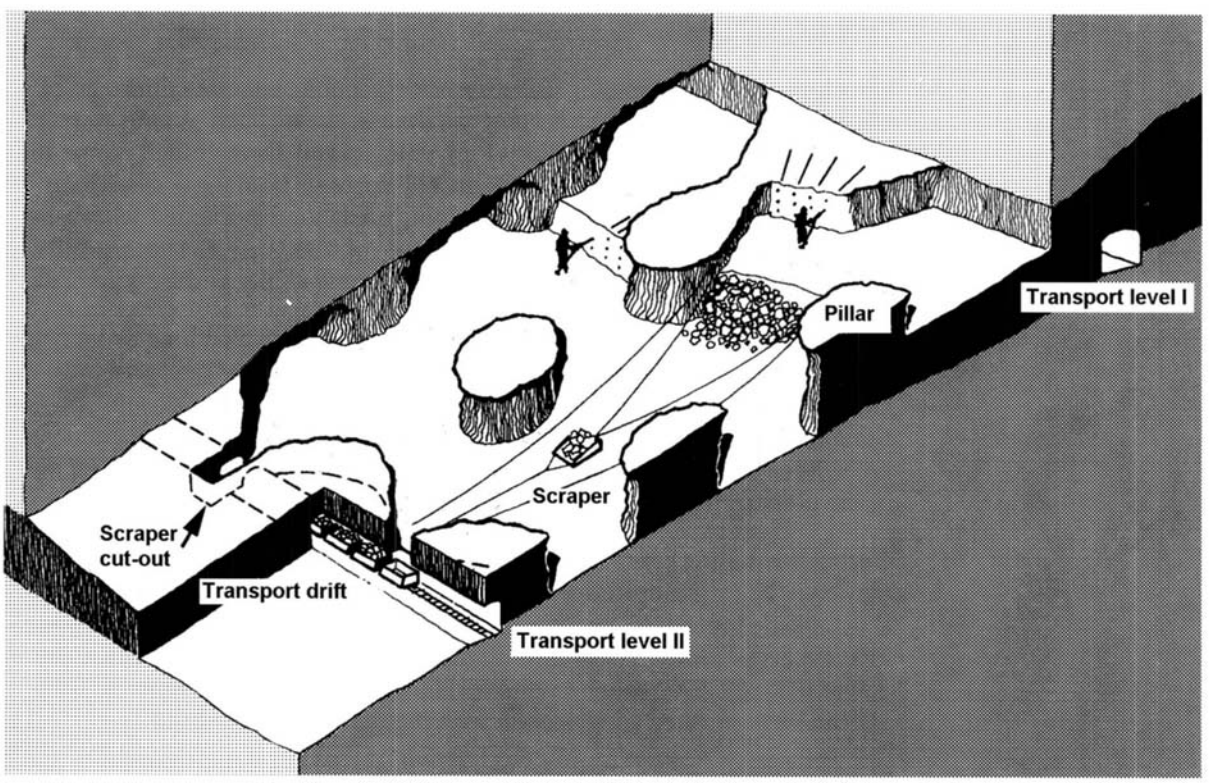


FIGURE 2.3 Room-and-pillar mining of an inclined orebody

Development. Developing an inclined orebody usually requires the development of several horizontal levels at specified vertical intervals. On each level, a haulage drift is prepared, following the footwall of the orebody. These haulage drifts serve both for access to the production areas and for transporting the broken ore to the hoisting shaft.

Production. Mining starts from one of the horizontal levels and proceeds upward, along the dip, until the next level is encountered. Normally, drilling is performed with air-leg drills, and the stope is advanced with holes arranged in a slashing pattern. The inclined and uneven surface precludes the use of mechanized equipment, requiring considerably more manual labor and physical effort than many other mining methods.

Ore Handling. Because the floor of the stope slopes, slushers are the predominant system used to bring the broken ore down to a point where it can be transferred into mine cars for haulage to the shaft.

Comments. The unfavorable dip results in a labor-demanding operation, with little or no possibility for mechanization. Although various attempts have been made to develop systems by which the efficiency can be improved, most have been unsuccessful. However, step mining is one variant that has been used successfully, replacing the conventional method of inclined room-and-pillar mining.

Step Mining. Figure 2.4 illustrates the general approach to step mining in an inclined ore deposit.

Development. Step mining utilizes trackless mobile equipment instead of the rail-bound haulage vehicles that are used in the inclined mining system. The trackless vehicles are capable of traveling up and down graded ramps and floors, so they are not restricted to almost horizontal haulage levels. Therefore, inclined drifts for access to the mining areas can be oriented to traverse the orebody. If the orebody has a high degree of regularity, the access drifts can be assigned directions that match the climbing ability of the equipment.

Production. Ore extraction starts with horizontal drifts that branch out from the access drifts. These stope drifts follow the ore boundary (hanging wall) as closely as possible. Successive sequences start similar drifts or slashing operations adjacent and parallel to the preceding drifts, at levels immediately below the last level. This procedure is repeated regularly, with the extraction of the orebody advancing downward, one step at a time.

During the mining operation, part of the ore must be left as pillars, and this may complicate the procedure to some extent. However, ore extraction is accomplished in stopes having reasonably level floors, interconnected by the inclined access drifts. These conditions allow the use of mobile mechanized equipment and a subsequent reduction in manual labor requirements. Drilling can be performed conveniently with regular drifting jumbos.

Ore Handling. The ore handling method for step mining is the same as that for horizontal room-and-pillar mining; the ore is picked up at the face and is carried to the shaft or other transfer point. The available space and headroom

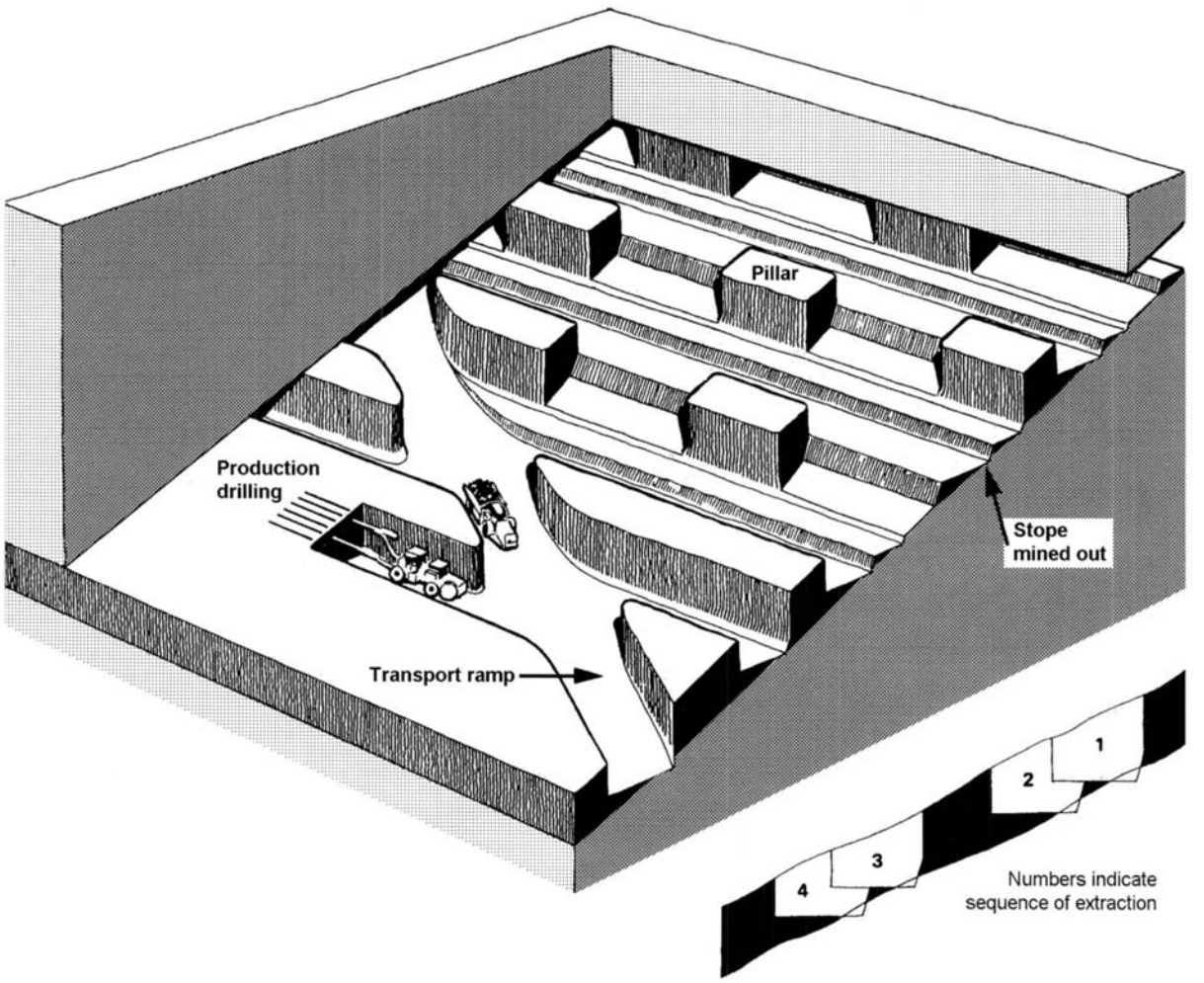


FIGURE 2.4 Room-and-pillar mining, using step mining on an inclined orebody

determine the equipment used, and the equipment may range from trucks to LHD vehicles.

Comments. The concept of step mining is a comparatively recent development in the industry. Although its applicability to inclined orebodies cannot be generalized completely, the approach has significant potential as an efficient modern approach to mining orebodies where the dip normally would be considered an inconvenience.

Sublevel Stoping (Blasthole Stoping). Figure 2.5 illustrates the general approach to sublevel stoping, showing ring drilling as the primary means of breaking the ore. Sublevel stoping is a mining method in which ore is extracted, and the stope is left empty. In this case, the stope is often very large, with the largest dimension being in the vertical direction. The sublevel stoping method applies only to vertical or steeply dipping orebodies.

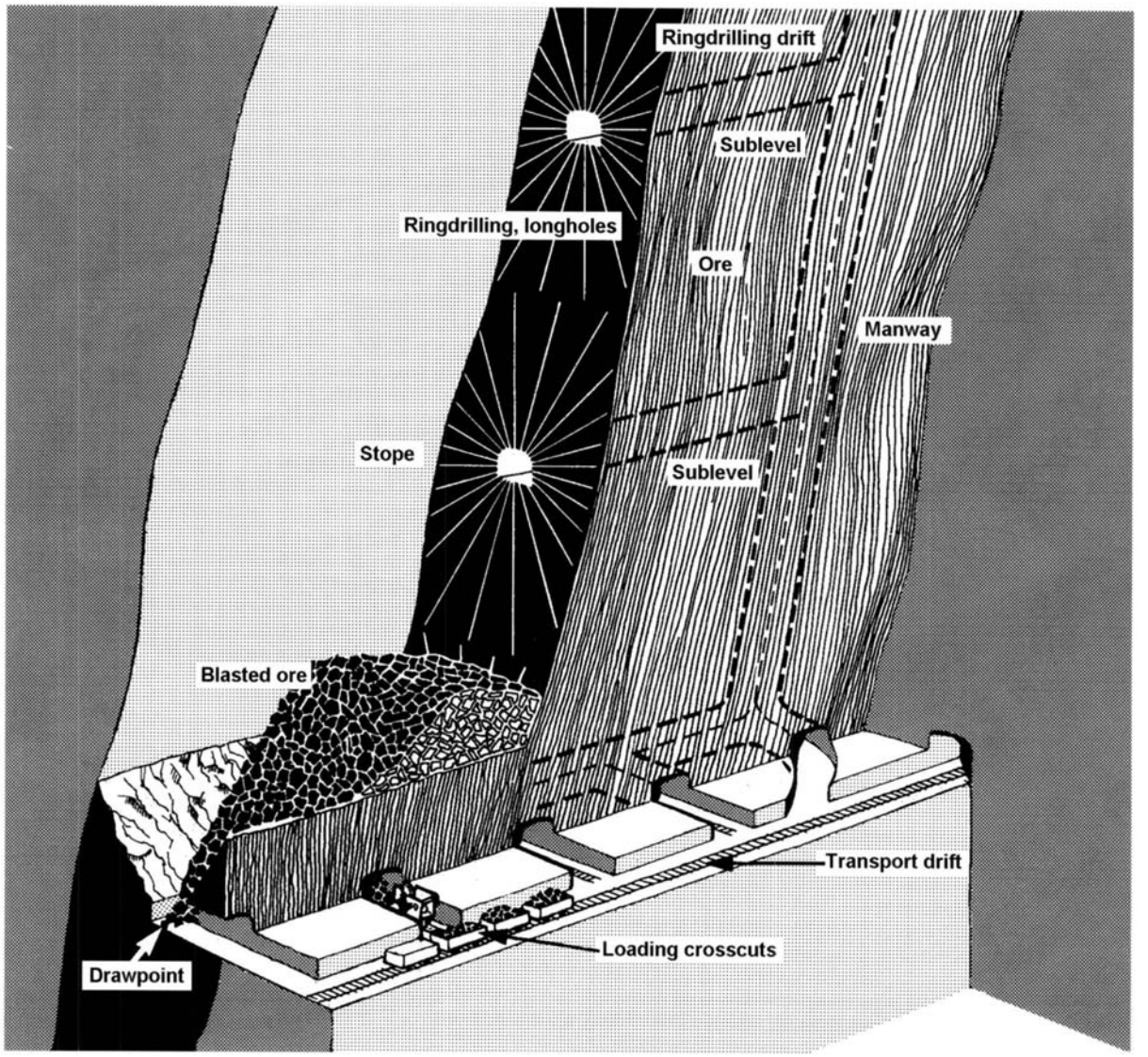


FIGURE 2.5 Sublevel stoping with ring drilling as the primary means of breaking ore

To prevent the stope walls from collapsing, large orebodies normally are divided into two or more separate stopes. Between the stopes, part of the ore is not mined; instead, it is left in place to serve as the support for the hanging wall. These pillars can be designed as both vertical and horizontal separations, sometimes being of substantial thickness.

In some cases, the pillars can be recovered, either partially or fully. Normally, the pillar recovery takes place during the final stage of the mining operation, when the eventual collapse of the surrounding rock no longer poses a hazard to mining personnel and equipment and would no longer affect normal mining activities.

In sublevel stoping, the mining is accomplished from levels at predetermined vertical intervals. Sublevels may be developed between the main levels, and the ore is attacked from both the main levels and the sublevels. The ore is drilled and blasted from drifts on the sublevels. The ore, broken in large vertical slices, falls to the bottom of the empty stope, where it can be recovered for transport out of the mine.

Applications. Normally, sublevel stoping is used in orebodies having the following characteristics:

1. The orebody must have a steep dip; that is, the inclination of the footwall must exceed the angle of repose.
2. The surrounding rock must provide a strong hanging wall and a strong footwall.
3. The orebody must be competent.
4. The boundaries of the orebody must be regular.

Development Comprehensive development is needed for sublevel stoping, including the following:

1. A haulage drift must be developed at the main level below the bottom of the stope.
2. Raises must be developed to provide access to sublevels and subsequent development of those sublevels.
3. Drilling drifts must be driven through the ore on the sublevels.
4. An undercut must be made at the bottom of the stope.
5. A loading drawpoint system must be developed to allow the ore at the bottom of the stope to be recovered safely.
6. A slot raise must be developed at the end of the stope; subsequently, the slot raise is enlarged to a full slot to open up the area for blasting.

Several variations of the undercut drawpoint development are possible. To avoid the complicated and time-consuming preparation of raises, the procedure can be simplified by locating a drilling drift on the drawpoint level, as shown in Fig. 2.5.

Production Today, production drilling in sublevel stoping operations is accomplished exclusively by means of longhole drilling, using extension drill steels to achieve the appropriate hole depth.

When ring drilling is used, the entire cross section of the stope is drilled with holes that radiate from the drilling drifts. The drilling pattern is matched to the shape of the orebody and the location of the drift. The pattern may consist of holes arranged as a part of a circle or as a complete circle. In narrow orebodies, parallel holes or holes in a fan-shaped pattern may be feasible and may be preferable to a ring pattern.

Two principal drilling systems are used. The traditional and most common system entails the use of special long-hole rock drills, equipped with extension drill steels in 1.2 to 1.8 m (4 to 6 ft) sections. The bit diameter is approximately 51 mm (2 in.). The hole length varies with the hole pattern, but

normally does not exceed 24 m (80 ft). Longer holes are possible, but they may cause problems as a result of deflection from the intended direction.

Recently, the application of down-the-hole hammers has become common in sublevel stoping. These hammers provide a hole size between 102 and 178 mm (4 and 7 in.); the larger holes accommodate a much greater burden with wide spacing between the holes. Consequently, the drilling requirements are reduced substantially. The larger holes are reported to provide improved fragmentation of the ore, and, to date, no adverse side effects have been experienced as a result of using heavy explosive charges in the large holes. This method offers another advantage in that the down-the-hole technique minimizes hole deflection, allowing the hole length to be increased to 45 to 60 m (150 to 200 ft). The longer hole lengths reduce the development effort by allowing an increase in the vertical spacing between the sublevels. Figure 2.6 illustrates a generalized sublevel stoping method that employs the down-the-hole drilling technique.

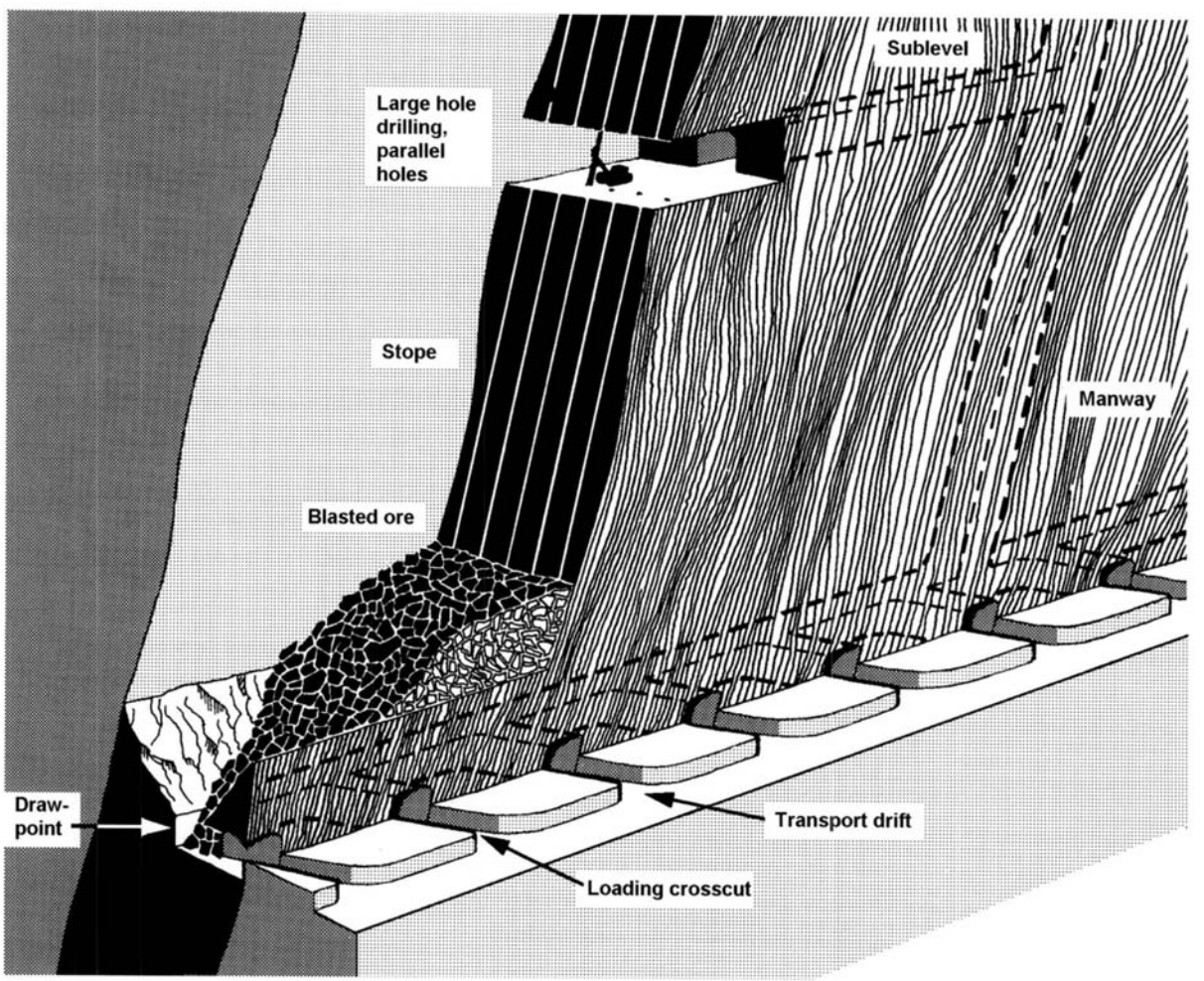


FIGURE 2.6 Sublevel stoping with large-hole drilling and blasting

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In sublevel stoping, the drilling often can be done considerably in advance of the actual mining operation. Large sections of the ore can be drilled, left in place, and blasted later as required. Because the drilling can be accomplished as a separate operation, with a large area drilled from a single drift, this method favors the use of special mechanized equipment that provides controlled and accurate hole alignment.

Ore Handling For further handling of the ore, the lower part of the stope is designed and developed to match one of the following systems:

1. The ore may be loaded through chutes and into mine cars. Boulders can be frequent in sublevel stoping, and they can make this system inconvenient. Because blasting (secondary breaking) in the chutes is quite complicated, the presence of boulders in the ore lowers the overall production capacity.
2. Slushers may be used to draw the ore from the drawpoints and into the mine cars. This system provides good access to oversize rock and makes handling of boulders much simpler.
3. Overhead loaders may be used for drawpoint loading, placing the ore into the mine cars. This approach is practiced in many large mining operations, whereas LHD loaders are used with a trackless mining layout.

Comments With the development of extension steels, special long-hole rock drills, and large-hole blasting techniques, sublevel stoping has grown in popularity throughout the mining industry. The complicated and comprehensive development is a drawback, but efficient production operations are the compensation. Drilling, blasting, and loading are operations that can be performed practically independent of each other, offering the potential for high utilization of mechanized equipment, high output with few units, and minimal personnel. High productivity can be obtained from a concentrated area within the mine.

Sublevel stoping requires a straightforward layout of the stopes and ore boundaries. Inside a stope, everything qualifies as ore, and there is no chance of recovering small mineralizations in the wall rock. Key factors for a successful application of the sublevel stoping method include a thorough knowledge of the geology, a knowledge of the ore boundaries, and careful control of the hole alignment in the long-hole pattern.

Methods With Filling

The mining methods that use filling techniques include shrinkage stoping, cut-and-fill mining, square-set mining, and longwall mining.

Shrinkage Stoping. Figure 2.7 illustrates the general approach to shrinkage stoping. In this method, the ore is excavated in horizontal slices, starting at the bottom of the stope and advancing upward. Part of the broken ore is left in the mined-out stope, serving as a working platform while mining the ore above and supporting the stope walls.

The blasted ore increases its occupied volume by about 70%. Therefore, 30 to 40% of the ore must be drawn off continuously to keep a suitable working distance between the back and the top of the broken ore. When the stoping has

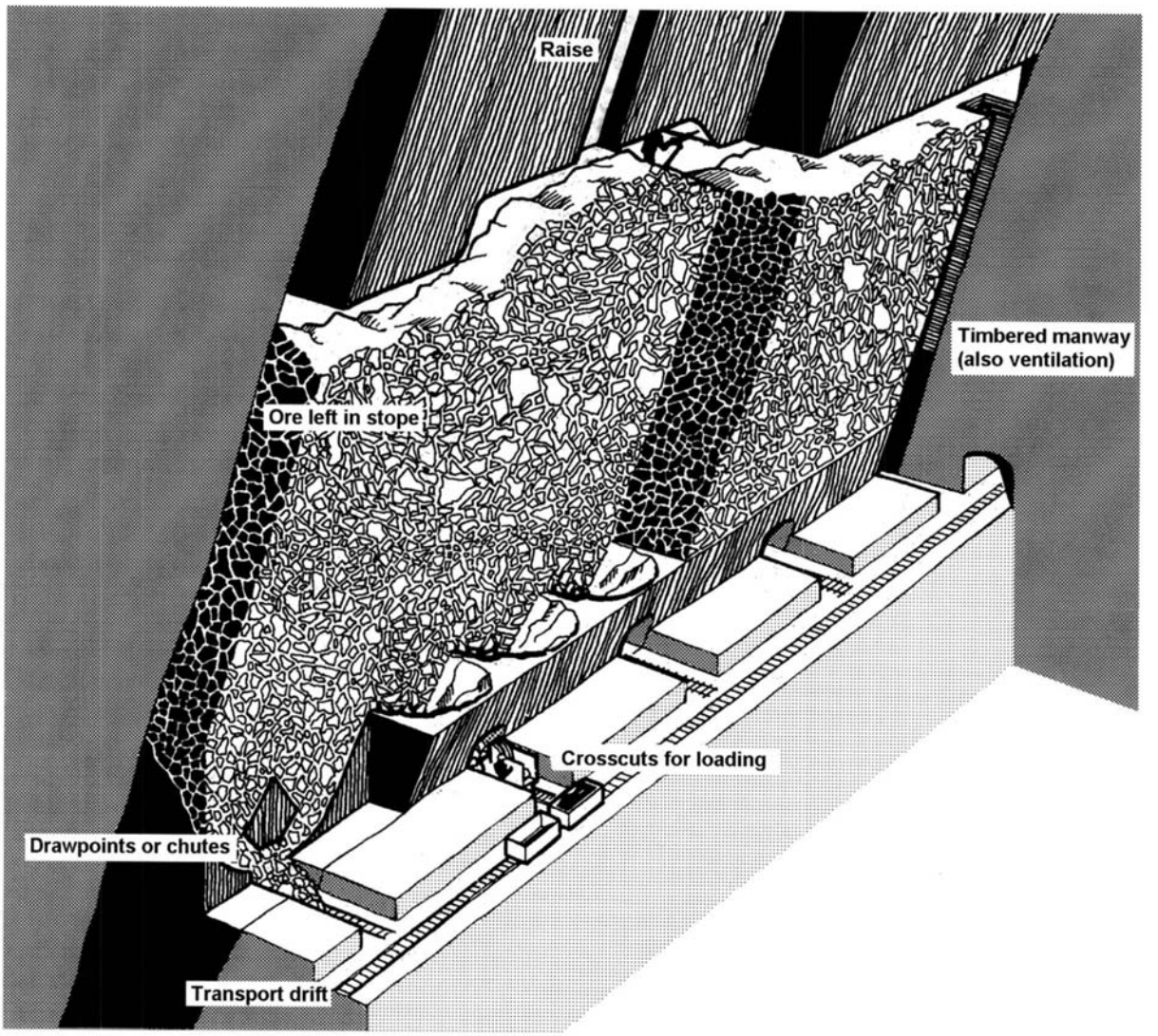


FIGURE 2.7 Shrinkage stoping in a large vertical orebody

advanced to the upper limit of the planned stope, drilling and blasting are discontinued, and the remaining 60 to 70% of the ore can be recovered.

Small orebodies can be mined as a complete shrinkage stope, but large orebodies usually are divided into separate stopes with intermediate pillars that stabilize the hanging wall. Generally, the pillars are recovered when the regular mining operations have been completed.

Applications Shrinkage stoping can be used in orebodies having the following characteristics:

1. The orebody must have a steep dip that exceeds the angle of repose.
2. The orebody must be firm and relatively competent.

3. The hanging wall and footwall must be stable.
4. The orebody must have regular boundaries.
5. The ore must not be affected by storage in the stope. Certain sulfide ores oxidize and decompose when exposed to air, making them unsuitable candidates for shrinkage stoping.

Development Shrinkage stoping requires the following development effort:

1. A haulage drift must be developed along the bottom of the stope.
2. Crosscuts must be made into the ore under the stope.
3. Finger raises and cones must be developed from the crosscuts to the undercut.
4. The stope must be undercut, taking a complete bottom slice at a distance of 5 to 10 m (15 to 30 ft) above the haulage drift.
5. A raise must be provided, passing from the haulage drift, through the undercut, and to the main level above. This raise provides both access and ventilation to the stope.

The development of the bottom section can be simplified in the same manner as the bottom level for sublevel stoping; the finger raises are eliminated, and the crosscut is used for the drawpoint loading.

Production Drilling and blasting are performed as overhead operations. The rough surface of the pile of ore in the stope prevents the use of mechanized equipment, so the standard practice utilizes air-leg rock drills and stopers. Occasionally, some horizontal long-hole drilling is practiced.

Ore Handling In shrinkage stoping, the conventional method of handling the ore entails direct loading into rail cars from chutes at the bottom of the finger raises. Today, loaders are more effective when they are used in a drawpoint loading system.

Comments When labor-saving machines were little used in underground mining, shrinkage stoping was a common and important mining method. The ore could be dropped directly into cars from the chutes, eliminating the hand loading requirements of other methods.

Today, this is of little importance, and the shrinkage stoping method has been largely replaced by other approaches. Its drawbacks include the fact that it is a labor-intensive method, with difficult and comparatively dangerous working conditions and with limited production capacity. Furthermore, the bulk of the ore remains tied up in the mine for a long period of time. Under the same conditions, sublevel stoping, sublevel caving, and cut-and-fill mining usually can be practiced with a considerable economic advantage over shrinkage stoping.

However, shrinkage stoping has not been totally eliminated from the mining industry. It is still practiced where a small-scale operation is undertaken with a minimum investment in costly machinery.

Cut-and-Fill Mining. As illustrated in Fig. 2.8, cut-and-fill mining excavates the ore in horizontal slices, starting at the bottom of the stope and advancing

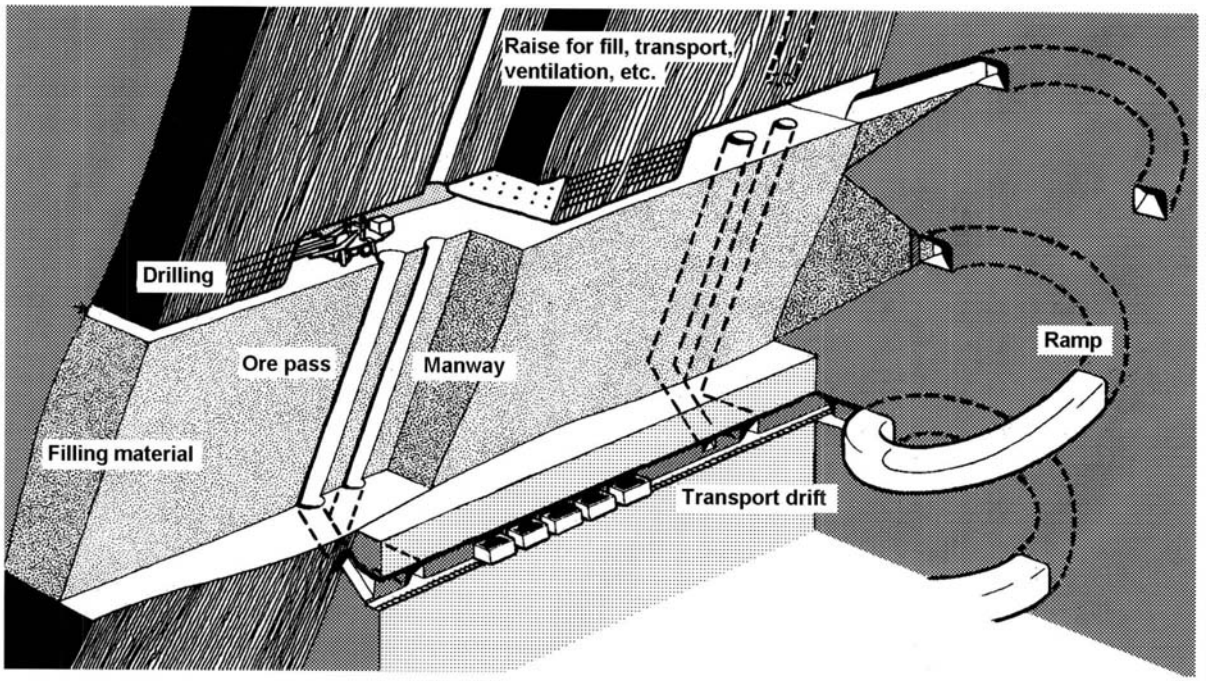


FIGURE 2.8 Cut-and-fill mining in a large vertical orebody

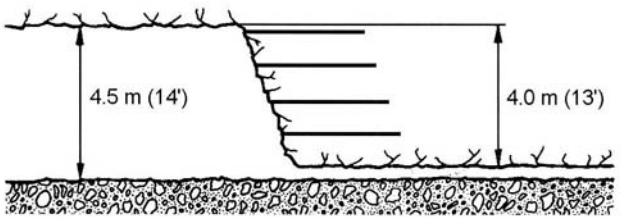
upward. The broken ore is loaded and completely removed from the stope. When a full slice has been excavated, the vacated volume is filled with waste material that supports the walls and provides a working platform while the next ore slice is mined.

The fill material can consist of waste rock, such as that from development work in the mine; the waste rock is distributed mechanically over the stoping area. However, in modern cut-and-fill mining, the common practice is to use hydraulic filling methods. The filling material consists of fine-grained tailings from the mill, mixed with water and transported into the mine for distribution through pipelines. When the water is drained off, a competent fill with a smooth surface is produced. In some cases, the material is mixed with cement to provide a harder and more durable surface with improved support characteristics.

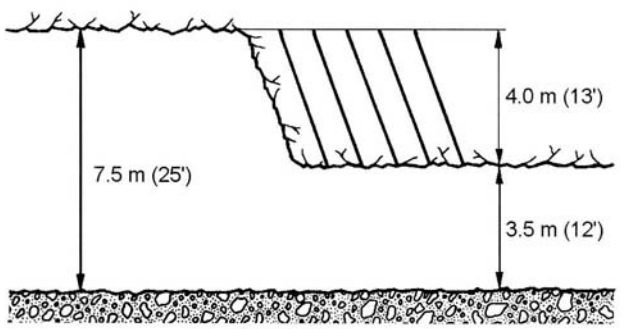
Applications Cut-and-fill mining can be applied in steeply dipping orebodies having reasonably firm ore. Compared with sublevel stoping and shrinkage stoping used with similar orebodies, the cut-and-fill method offers the advantage of selectivity. The method can be adapted to irregular and discontinuous orebodies, extracting the high-grade ore and leaving the low-grade material behind in the fill.

Development The development required for cut-and-fill mining consists of the following:

1. A haulage drift must be developed along the orebody at the main level.
2. Short raises and manways to an undercut must be provided.



Horizontal Drilling



Vertical Drilling

FIGURE 2.9 Comparison of cut-and-fill drilling systems

3. The stope area must be undercut, usually 5 to 10 m (15 to 30 ft) above the haulage drift.
4. Raises for ventilation and fill transport must be provided to the level above.

Production As shown in Fig. 2.9, two different systems may be used for drilling and blasting. The most common method is to drill vertical or inclined holes upward into the back, and then bring down a slice of ore in a manner that resembles inverted benching. Large sections of the roof can be drilled without interruption, and large rounds can be blasted. Simple lightweight wagon drills are available for the roof drilling.

One drawback to this method is that the headroom in the stope is increased to about 7 m (23 ft) when the ore is removed. Blasting creates a ragged back that is difficult to control and is potentially dangerous to the miners, unless the surface is trimmed by smooth-blasting prior to mucking.

As an alternative to overhead drilling, horizontal holes may be drilled, and the ore may be blasted with a breasting technique. The stope is filled as completely as possible with hydraulic fill, and only a narrow slot remains between the

previous back and the surface of the fill. The drilling can be accomplished with regular drifting jumbos. The size of the round is limited to what can be drilled from the face, and it is much smaller than the round resulting from vertical holes. With modern mobile drilling and loading equipment, the round has little influence on the stoping efficiency.

Face drilling offers several advantages over vertical drilling. First, the holes are horizontal, and the back can be blasted to leave a more even surface. After the ore has been removed, the headroom is only on the order of 4 m (14 ft) making the back easy to control. Face drilling also provides selectivity, because the low-grade material can be left in place to an extent. It also is possible to adjust the outline of the stope to extract mineralizations found in the walls.

Ore Handling Some version of the LHD technique commonly is applied to the handling and transport of the ore. The surface of the hydraulic fill is smooth and suitable for rubber-tired equipment. The travel distance is limited because transportation usually consists of bringing the ore to an orepass within the stope.

Comments Cut-and-fill mining has a broad range of applications, as a direct result of selective mining, good recovery, and practicability under rather weak and unpredictable rock conditions. Hydraulic fill has improved the economical and technical aspects of this method, and cut-and-fill mining often has replaced other methods.

With modern mobile equipment, complete and efficient mechanization can be achieved. Where mobile equipment is used, inclines or ramps commonly are provided for access to the stopes. The equipment can be moved to other stopes during interruptions for filling, and good equipment utilization can be achieved.

It is characteristic of cut-and-fill mining that the ore production from the stopes is discontinuous. This is attributed to the interruptions in the production cycles as the fill is distributed. However, hydraulic filling reduces the interruptions to comparatively short periods.

Square-set Mining. The traditional method of square-set mining is of limited importance today. Its main advantage is that it may be applied to almost any orebody shapes and ground conditions. However, it is extremely labor intensive, and it requires a steady supply of timber.

The method is based on a timber-support system, where precut modular pieces are assembled in a regular square pattern throughout the excavated area. The space between the timbers normally is filled with waste material, leaving only certain sections that serve as drifts, manways, and orepasses. Extraction is an entirely manual operation, and handling the timbers and erecting the supports are laborious operations.

Square-set mining is suitable for small orebodies of high-grade material, where good recovery is important and where the grade justifies the high operating costs.

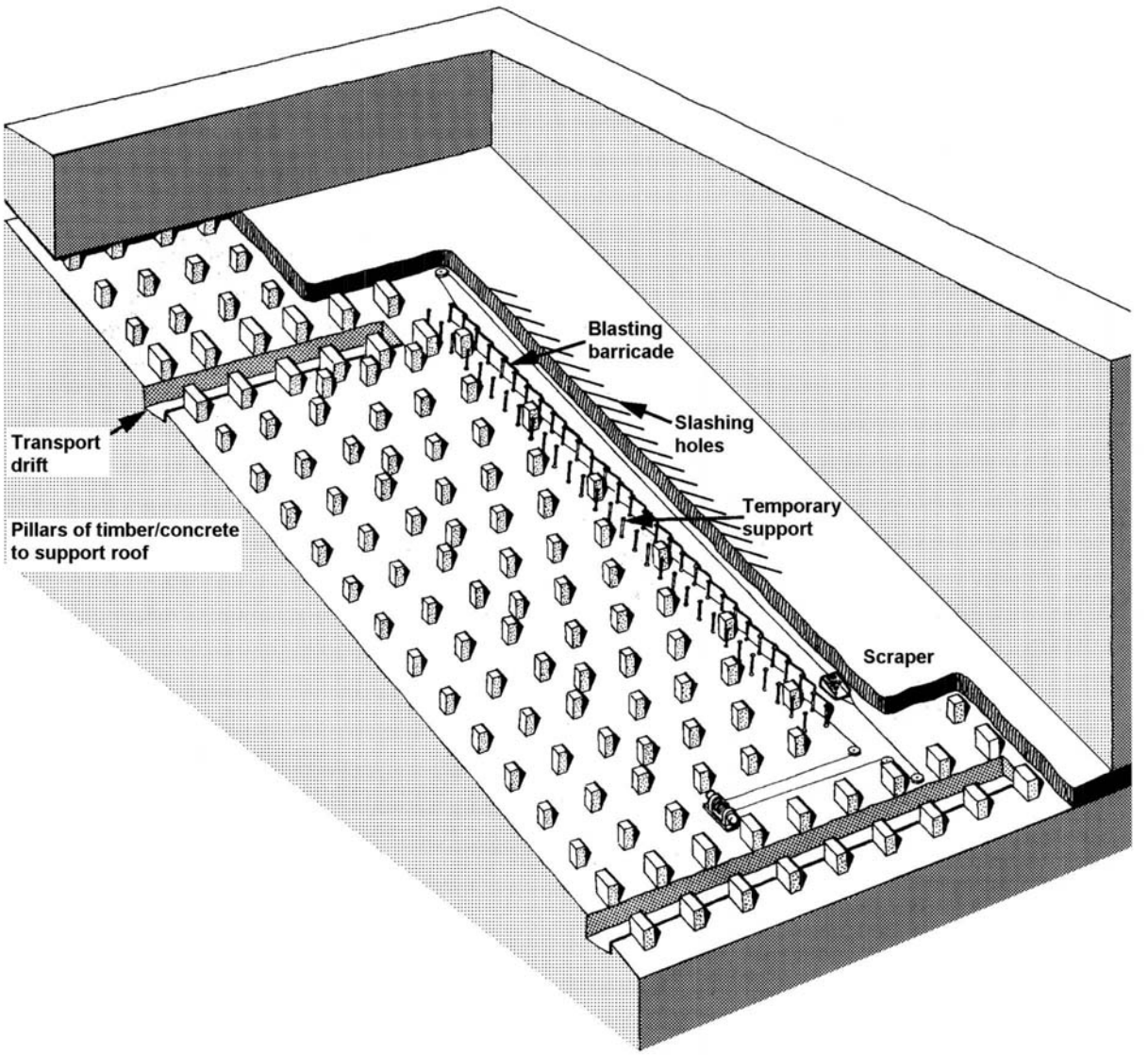


FIGURE 2.10 Longwall mining in a hard-rock environment

Longwall Mining. Figure 2.10 illustrates the general approach to longwall mining in hard rock, and Fig. 2.11 illustrates longwall mining in coal. The ore is excavated in slices, usually along a straight working face. The excavated area close to the face is supported to provide space for drilling and ore removal. At some distance from the face, the roof may be allowed to cave.

Applications Longwall mining is applicable only to thin-bedded deposits of uniform thickness, normally occurring as a flat seam of large horizontal extent. The method can be used for hard or soft rock conditions, because the working

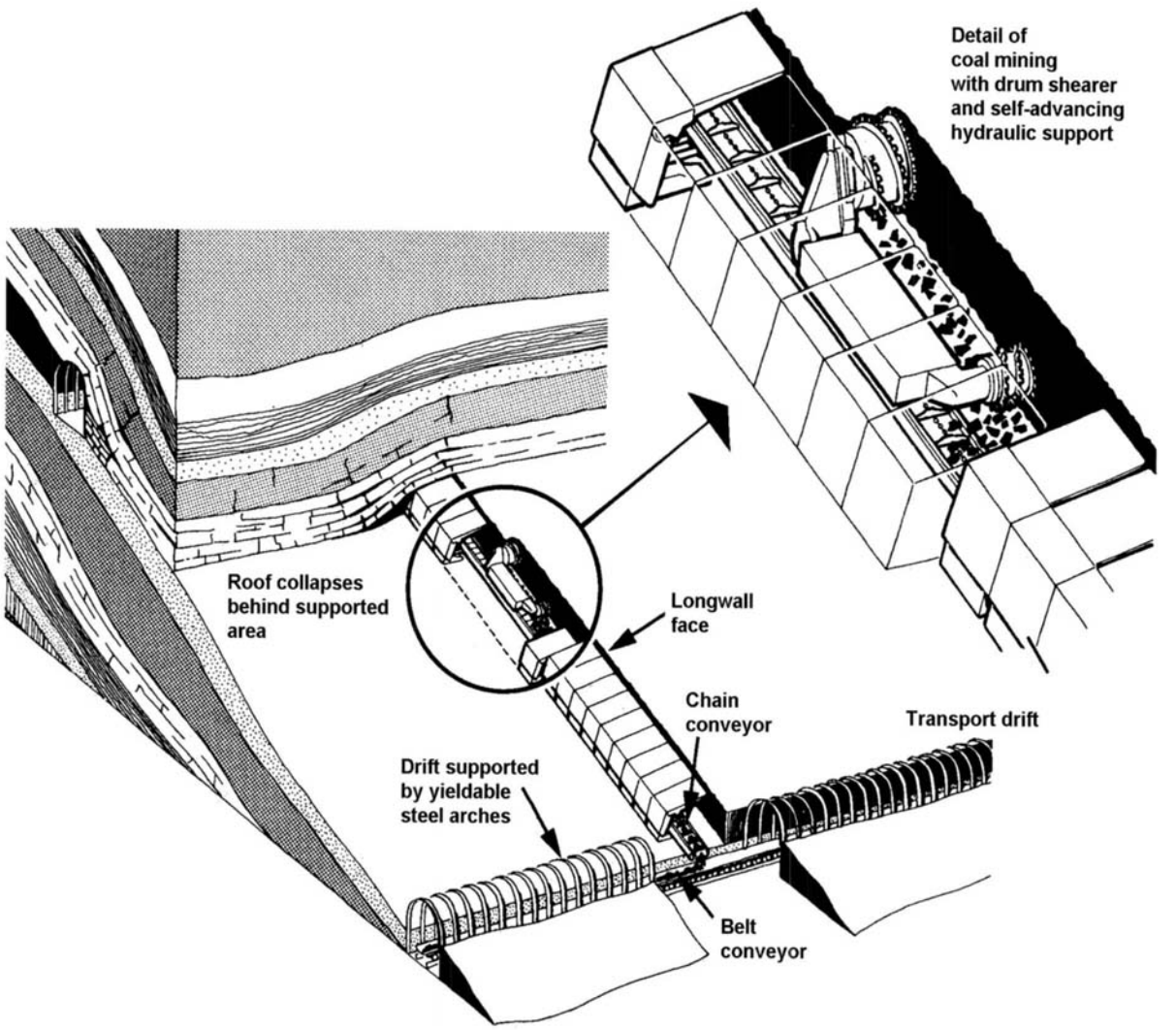


FIGURE 2.11 Longwall mining of a coal deposit

area can be supported well. The South African gold mines are examples of longwall mining in hard rock, whereas soft-rock longwall mining occurs in many coal mines.

Development Longwall mining development mainly consists of providing a system of haulage drifts for handling the extracted material and for access to the production areas. The mining is performed along a face or front that connects two haulage drifts.

With a regular seam or with mineralization continuing over a large area, haulage drifts can be arranged in a pattern that provides longwall faces or panels of the same length.

Production In hard rock, the ore is drilled and blasted with side-slashing holes. Because the headroom is limited, it usually is necessary to use machines that are operated manually.

In contrast, longwall mining of coal and other soft materials is a process that can be mechanized almost completely. The soft material does not require drilling and blasting, but can be cut or broken mechanically with plows, drum shearers, and other machines that are specially designed for this purpose.

Ore Handling Under hard-rock conditions, scraping is the normal means of bringing the ore from the face to the haulage drift for further transport. Low headroom and inclined stopes preclude the use of other machinery. The handling of soft material is more convenient and efficient; the special production machines usually are integrated with material handling systems that bring the broken material out from the longwall face. Chain conveyors are quite common for material handling in soft-rock mining.

Caving Methods

All caving methods are based on fracturing the mineralized rock and the surrounding waste rock under more or less controlled conditions. The fractured material fills the voids left by ore extraction, creating a caved area on the surface over the orebody. Complete and continuous caving is essential, because large empty spaces underground can collapse suddenly with severe aftereffects on the mining operation.

Sublevel Caving. Figure 2.12 illustrates the general approach to sublevel caving. The orebody is divided into sublevels with fairly close vertical spacing, normally 8 to 15 m (25 to 50 ft) apart. Each sublevel is developed with a regular network of drifts that penetrate the complete ore section. In wide orebodies, the drifts are laid out as crosscuts from a footwall drift; in narrow deposits, the drifts are parallel along the strike.

From the sublevel drifts, the ore section immediately above is drilled with long holes in a fan-shaped pattern. The drilling is done ahead of the blasting, and several sublevels can be drilled completely before blasting and production are started.

Blasting on one level starts at the hanging wall or the far end of the orebody, retreating along the footwall. Ore extraction follows along an approximately even front, allowing several drifts to be operated on the same sublevel. The sublevels below and above also are included in the production sequence.

The blasting of one fan breaks the ore, causing it to cave into the drift. From the drift, the ore is loaded and transported to orepasses. The waste rock in the hanging wall continuously breaks and fills the void as the ore is removed. In the drift, this is realized as a gradually increasing waste dilution of the loaded material. At some point, the dilution reaches such a proportion that the loading is stopped and the next fan is blasted. Some of the ore remains in the caved area and cannot be recovered. The dilution of the ore with waste can vary between 10% and 35%, with the ore losses varying between 10% and 20%.

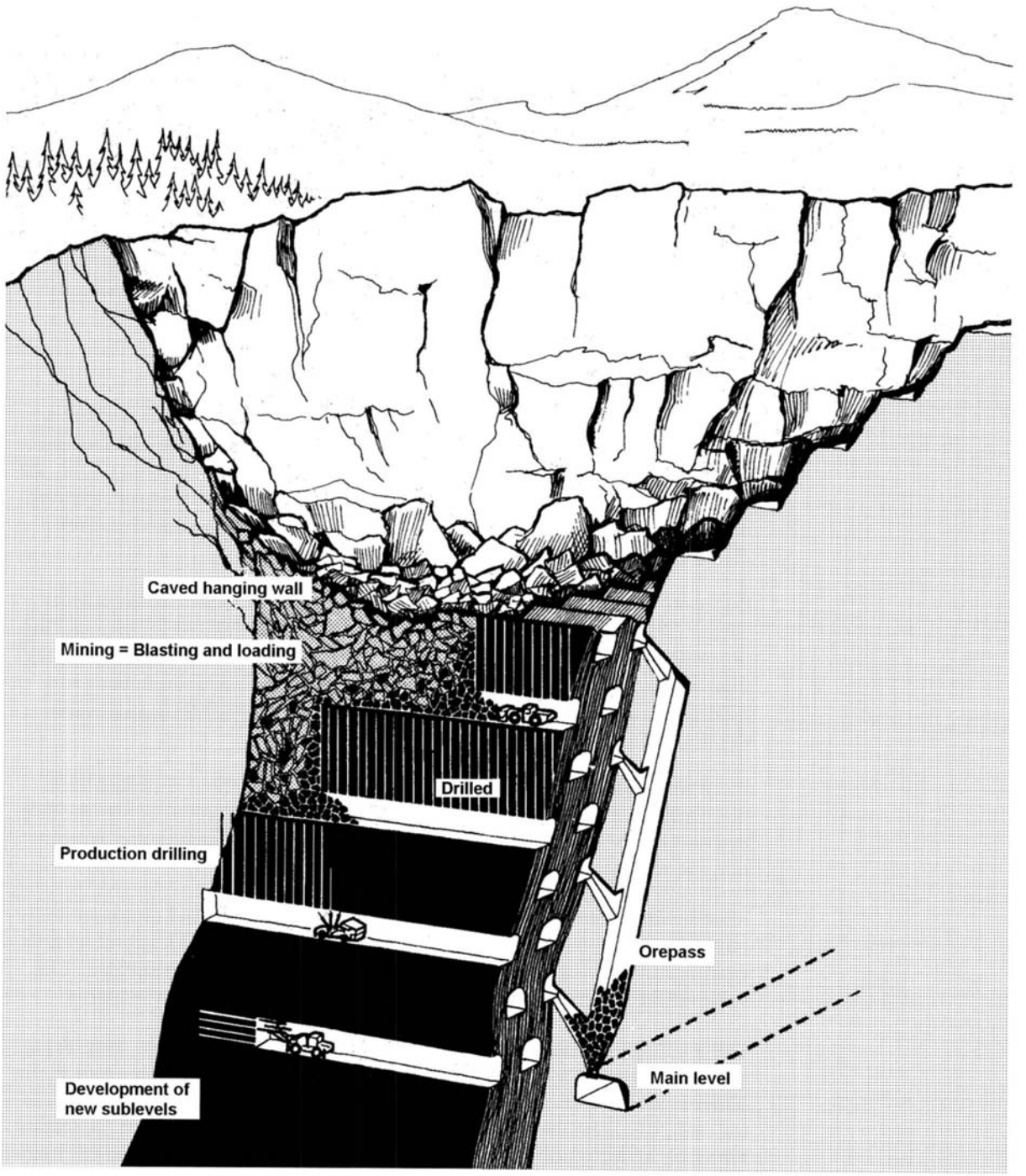


FIGURE 2.12 Sublevel caving in a large and steeply dipping orebody

Applications Sublevel caving is used in steeply dipping orebodies and other deposits having large vertical dimensions. A minimum requirement for the stability of the ore is that the sublevel drifts be largely self-supporting, requiring only occasional reinforcement. The hanging wall must follow the ore extraction in a continuous cave, and surface conditions must allow subsidence.

Dilution with waste material and ore losses are factors that influence the practical application of sublevel caving techniques. The method is preferred for ores from which the waste rock and the paying mineral are separated easily (e.g., by a simple magnetic process). However, this situation is different where the ore boundaries are determined by a vague cutoff grade, and where the surrounding rock is mineralized.

Development The major part of the development effort consists of drifting to prepare the sublevels. This is a comprehensive operation, and as much as 20% of the ore can be extracted during the development. In addition to the sublevel drifts, orepasses and inclined drifts are required to connect the sublevels with the main levels.

The amount of development work in sublevel caving is substantial. However, the drifting is a repetitive and regular operation that can be organized and performed efficiently with modern equipment.

Production Drilling is performed in a fan-shaped pattern that radiates upward from the sublevel drift. Specially designed fan-drilling rigs are used, making longhole drilling a highly efficient mechanized operation.

Ore Handling In the production area, ore handling consists of loading the ore in the sublevel drift, transporting it, and dumping it into an orepass. These conditions favor the use of LHD equipment. Often, the sublevels are designed for transportation distances matched to a particular size of loader. High efficiency is achieved in the loading and transport phases, because a loader can be kept in continuous operation by moving from one sublevel drift to another.

Comments Sublevel caving is systematic and utilizes regular layout and working procedures. Development, production drilling, and loading are carried out on separate levels, and each is a continuous operation independent of the others. A number of working faces always are available for the operations. These factors make sublevel caving a method in which mechanization can be introduced throughout, thus maintaining an efficient mining process. The mining operation can be compared to an industrial process.

Waste dilution and ore losses are drawbacks that cannot always be accepted. To minimize these factors, an extensive research program has been undertaken, involving both model and full-scale tests. The experience gained from these investigations, combined with a follow-up of practical results, now has developed sublevel caving into a method with high technical standards.

Block Caving. Figure 2.13 illustrates the general approach to block caving. The ore fractures and breaks by itself as a result of internal stresses and forces. A minimum of drilling and blasting is needed in the production operation. The

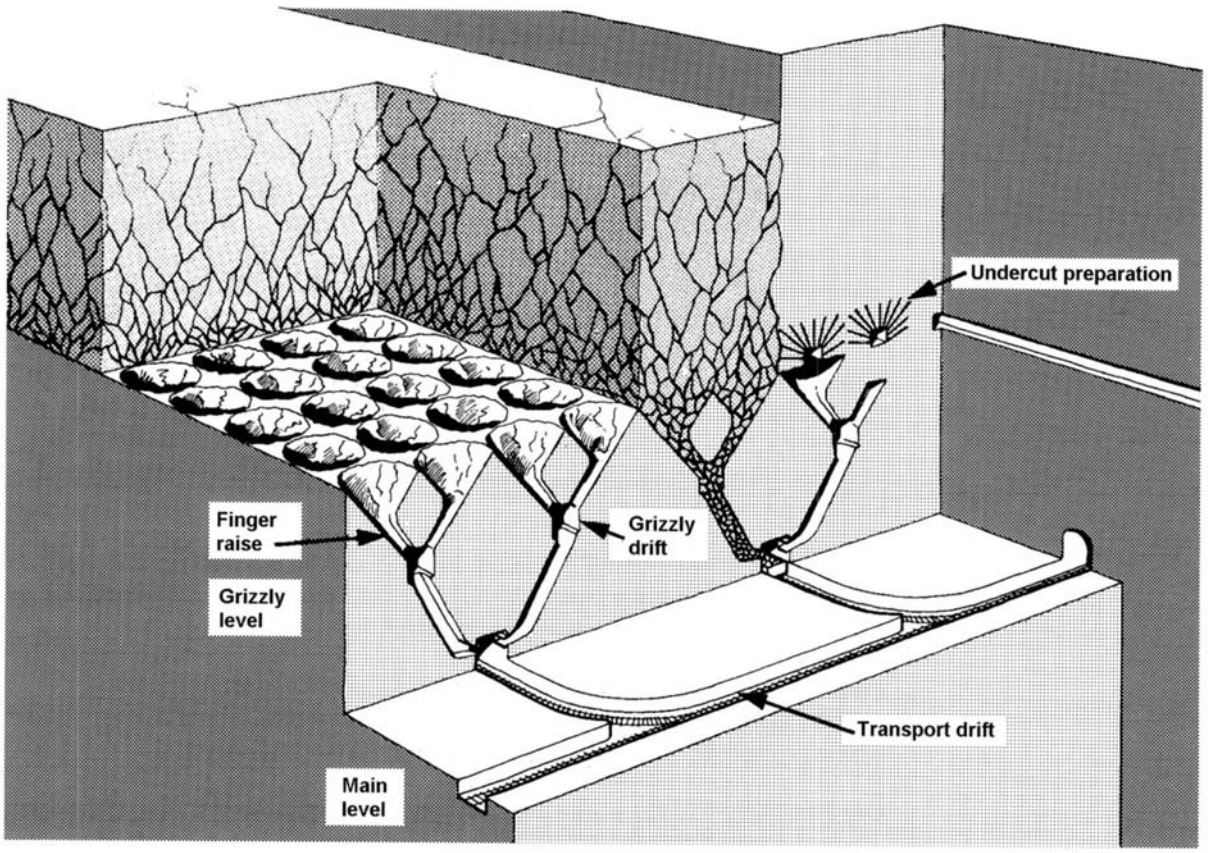


FIGURE 2.13 Block caving in a massive orebody, showing the conventional mining layout

mining plan is divided into large sections or blocks, usually with a square horizontal cross section and an area of more than 1000 m^2 (11,000 sq ft). Each block is undercut completely by a horizontal slot that is excavated in the lower part of the block. Gravity forces on the order of millions of tons act on the rock masses, and successive fracturing occurs, affecting the entire block. As the rock pressure increases at the bottom of the block, the ore is crushed to a fragmentation that allows removal at drawpoints or through other arrangements.

Applications Block caving is used for large and massive orebodies having the following characteristics:

1. The orebody must exhibit a steep dip. In a massive orebody, there must be a large vertical extension.
2. After undercutting, the rock must be capable of caving and breaking into suitable fragments.
3. The surface conditions must allow subsidence of the caved area.

These rather specific conditions limit the application of the block-caving method. Common applications are found in the mining of iron ore and the mining of low-grade disseminated ores (e.g., copper or molybdenum).

Development The development efforts for block caving normally include the following:

1. A pattern of loading and haulage drifts are developed underneath each block.
2. Orepasses or finger raises are provided up to a grizzly level.
3. A grizzly level is developed where fragmentation is controlled and secondary blasting is performed.
4. Another set of finger raises are widened into cones at the undercut level.
5. An undercut is developed underneath the block.

By arranging the two sets of finger raises similar to the branches of a tree, the ore from a larger area on the undercut level is funneled together to one point on the haulage level.

The layout described is for a conventional mine plan, where the ore is loaded into mine cars through chutes. In a mine using a trackless layout, drawpoint loading is common, and development is simplified by omitting the grizzly level and the second set of finger raises.

All excavations underneath the block are subject to high rock pressure. Therefore, the cross sections of the drifts and other excavations are kept as small as possible. Heavy reinforcement with concrete often is necessary. Overall, the development stage is a time-consuming and complicated process, usually requiring several years to bring a block to production.

Production Once the undercut is completed, the ore should start falling down the finger raises, continuing to do so as the material is removed on the haulage level. Theoretically, no production drilling is required. In actual practice, it sometimes is necessary to start or assist the fracturing by drilling and blasting longholes in a widely spaced pattern. Secondary drilling of oversize rock is a frequent operation in block caving.

Ore Handling The conventional block-caving system, with a grizzly level and finger raises, utilizes gravity forces to deliver the material into rail cars. However, the close control of fragmentation that is required for loading through chutes can make the grizzly treatment a labor-intensive operation, as well as a bottleneck in the production cycle. With the layout shown in Fig. 2.14 for trackless mining and drawpoint loading, the handling of oversize material is simplified greatly, and development work is reduced substantially.

To obtain the best ore recovery from a block, it is important that the upper caved surface be kept even. Therefore, the volume of ore drawn from different positions within the block must be controlled and must follow a planned schedule.

Comments Under favorable conditions, block caving is an economical mining method. Its drawbacks are the comprehensive development required and the time lag before production reaches full capacity. Also, there are certain hazards, because caving and fragmentation are events that cannot be predicted or

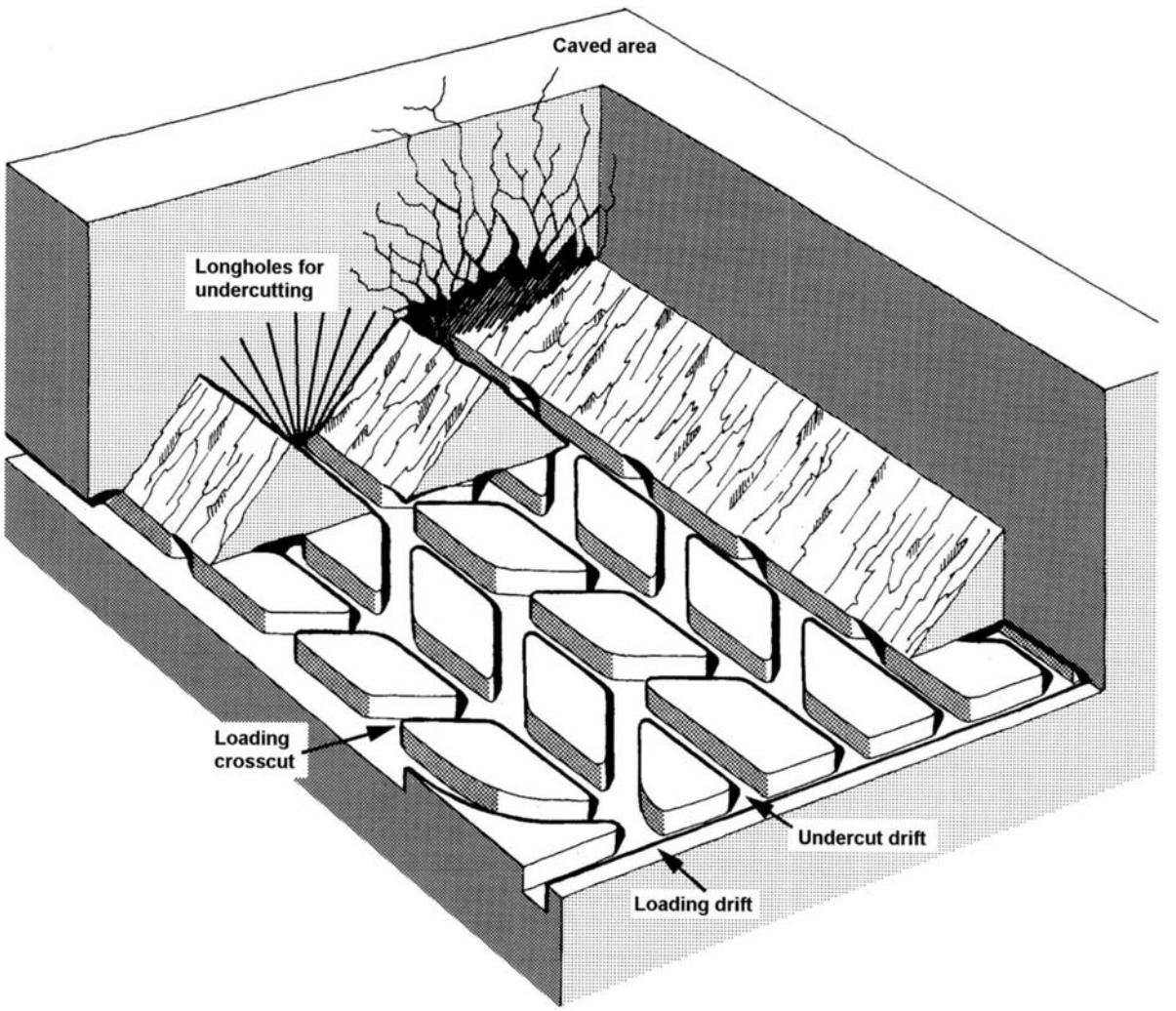


FIGURE 2.14 Block caving in a massive orebody, showing the layout for trackless mining

controlled fully. Unexpected hangups and large boulders can cause serious problems that are difficult to tackle in material that is already fractured.

GUIDELINES FOR SELECTING A MINING METHOD

The basic objective in selecting a method to mine a particular ore deposit is to design an ore extraction system that is the most suitable under the actual circumstances. This can be interpreted as aiming for maximum profit from the operation, but it is a decision based on both technical and nontechnical factors. For example, high productivity, complete extraction of the ore, and safe working conditions all are factors that must be considered in the selection of a mining method.

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General Considerations

In some cases, the conditions around an ore deposit may be quite distinctive, and they may dictate one particular method or the immediate exclusion of other methods. Under such circumstances, the selection of the method is limited to adapting and refining the general method to the particular orebody. In other cases, the conditions may favor the application of several methods, which then must be compared and evaluated.

In either case, the planning and evaluation of the mining method, together with a preliminary selection of equipment, is a task requiring careful study and consideration. Once the plans are set and development begins, it is extremely difficult and costly to change to an alternative method. In most cases, only minor alterations can be made, sacrificing work already accomplished that cannot be properly utilized under the modified plan.

The emphasis on careful skilled engineering is underlined by the time factor. A project started today will require several years to reach production, and it is expected to produce ore for many years after that. Although the basic principles of a mining method can be expected to remain the same, rapid developments are being made in the machinery and its utilization, making the ore extraction process more efficient. Up-to-date knowledge of the latest developments in mining techniques and a feeling for future trends are necessary to design a successful project.

It is advisable to incorporate features that allow flexibility and growth in the mining system. Looking at mines that were developed only a few decades ago, difficulties now are found with the limitations of shaft dimensions, ventilation systems, etc. Many mines no longer can accommodate the increasing dimensions of new mining machines and the voluminous exhaust gases emitted by those machines.

The process of selecting a mining method begins with a compilation and evaluation of the facts already known about the ore deposit. The available information can vary within wide limits, from observations made through core drilling of a recently explored mineralization to the extension of an orebody that is well known from previous underground mining.

Under all circumstances, effective evaluation of the mining methods depends upon the information available. Rarely is it possible to do more than a preliminary study from core drilling observations and other surface investigations. Information from actual underground workings can suffice for a final development plan, so a combination of surface investigation and detailed studies of underground conditions is necessary to avoid mistakes in the early stages of mine development.

Figure 2.15 illustrates surface drilling complemented by underground drifting and drilling. In this case, drifts, shafts, and other underground workings are engineered for use in the future production stage of the mine.

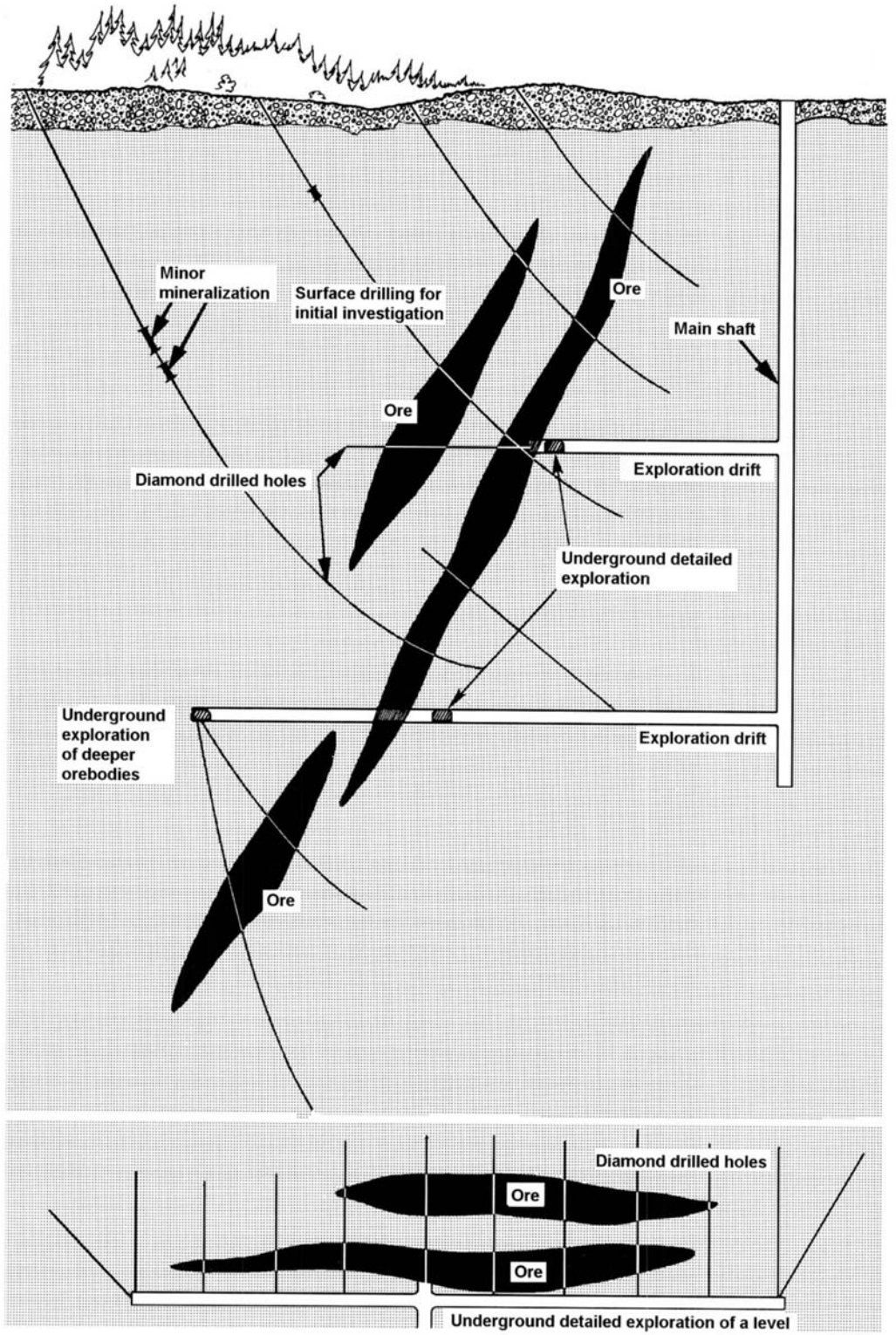


FIGURE 2.15 Exploration of orebodies using surface and underground techniques

TABLE 2.1 Relationships between orebody dips and mining methods

Dip	Mining Method	Comments
Flat	Room-and-pillar mining	Competent horizontal orebody
Flat	Longwall mining	Thin seam-type orebody
Medium	Room-and-pillar mining	Competent orebody
Medium	Inclined room-and-pillar mining	Slope precludes mechanization
Medium	Step room-and-pillar mining	Stepping allows mechanization
Medium	Longwall mining	Thin seam-type orebody
Medium	Cut-and-fill mining	Firm orebody; selectivity; mechanization
Medium	Square-set mining	High-grade ore; labor-intensive
Steep	Sublevel stoping	Competent ore with regular boundaries
Steep	Shrinkage stoping	Competent ore, regular boundaries, delayed ore recovery
Steep	Cut-and-fill mining	Firm orebody; selectivity; mechanization
Steep	Sublevel caving	Large orebody; extensive development effort
Steep	Block caving	Massive orebody; extensive development effort
Steep	Longwall mining	Thin seam-type orebody
Steep	Square-set mining	High-grade ore; labor intensive

Rarely is the occurrence of minerals an isolated or unique phenomenon in a particular area. New orebodies are most likely to be found in the immediate proximity of existing mines or in the same mineral-bearing region. Valuable information can be obtained from studies of mines already in operation, with seemingly comparable geology and orebodies.

Geological Conditions

Most of the factors that physically influence the choice of a mining method are included in the concept of geology, that is, the ore situation in the rock and the behavior that can be expected of the ore and the surrounding rock.

Dip. The dip of the orebody is a factor influencing the mining method. Normally, the dip is classified as either steep or flat, with a rather undefined medium range between the two extremes. Steep dips range from the angle controlling gravity flow, about 0.87 rad (50°), to vertical. Flat dips are more difficult to define, because they are connected with equipment capabilities; they normally range between horizontal and an inclination of 0.35 rad (20°). The medium dips are difficult to fit into the description of mining methods but range from 0.35 to 0.87 rad (20° to 50°). Because mine workings can be oriented at an angle to the dip, manageable grades can be achieved for mobile equipment even in medium dips. Analysis calls for a case-by-case examination of orebodies with respect to their dips. Table 2.1 defines the relationship between the dip of the orebody and the mining methods that may be applicable.

TABLE 2.2 Rock classifications

Rock Types	Compressive Strength			Comments
	kPa	bars	psi	
Quartzite, basalt, and diabase	over 220,000	over 2200	over 32,000	Very high strength
Most igneous rocks, stronger metamorphic rocks, hard limestone, and dolomite	140,000–220,000	1400–2200	20,000–32,000	High strength
Shales, limestone, sandstone, schistose rock	100,000–140,000	1000–1400	14,500–20,000	Medium strength
Friable sandstone, mudstone, and coal measures	40,000–100,000	400–1000	6000–14,500	Low strength

Rock Strength. The characterization of rock as weak or strong is a very subjective matter; what might be considered strong rock in a coal mine could be viewed quite differently in a hard-rock mining environment.

It is difficult to formulate a quantitative measure of rock strength that can be applied to a mining system. However, in the preliminary stages of a mine investigation, some conclusions can be reached from core samples, particularly if supportive data are available from nearby mining operations.

Core samples do give a general geological picture of the type of rock that can be expected in the hanging wall, the footwall, and the ore itself. A hanging wall consisting of solid Precambrian rock probably will allow mining with open stopes.

Another approach is to measure the uniaxial compressive strength of the rock. Different types of rock having varying compressive strengths are listed in Table 2.2.

Another observation that can be made from the core samples is the percentage of the core length that is recovered. For a strong rock, the recovery should be approximately 90% of the core length. In another system, recording the lengths of individual pieces of the core may prove meaningful.

Regardless of the system used, it is extremely difficult to determine the actual mining conditions from observations of the core samples, unless there is a clear reference from a nearby mining operation.

Span Limits. In any mining system, the rock strength determines the limits for the sizes of the spans that can be excavated, either without support or with a particular method of support. If the span is too large, working conditions become unsafe, and caving may occur. To counteract potential mishaps, the size of the underground openings must be decreased. Reliable predictions can be made only as a result of actual underground investigation.

TABLE 2.3 Example of grade and ore relations

Case	Cutoff Grade	Average Grade	Ore Reserves, mt
A	4.0%	5.5%	500,000
B	1.5%	2.5%	5,000,000
C	0.5%	0.9%	50,000,000

However, the results of geological mapping and exploration drilling, compiled with other information, usually suffice to decide between open stoping or a filling method. The rock strength is one of the parameters used for the detailed layout of the particular method chosen.

Ore Reserves and Grades

Only the mineralization that can be exploited commercially to yield a profit for the mining enterprise is classified as ore. Anything else is just rock, even though minor metal contents can be given a certain value for complete economic evaluation.

Calculating the ore reserves assumes that mining costs are known, at least approximately, during the evaluation of borehole loggings and other observations. However, for a new prospect, the operational costs for mining can be left as a fairly rough estimation; the primary question is whether the potential tonnage and grade of the ore can justify the investment necessary to undertake a mining operation. This calls for a financial analysis adapted to the particular project; this is a subject beyond the scope of this presentation.

Normally, the boundaries of a mineralization are not distinct. Borehole logging may show areas with occasional high metal values, surrounded by scattered sections with lower grades and by barren rock. To define and map an orebody, it is necessary to establish a cutoff grade that represents the lowest grade (or combination of grades) at which the mineralized rock qualifies as ore. By applying different cutoff grades to the outlines of the orebodies, varying tonnages and average grades can be identified. Under typical conditions, gradually lowering the cutoff grade causes the estimated ore reserves to increase from a narrow high-grade vein to a massive low-grade orebody. The examples of Table 2.3 illustrate this principle.

If Table 2.3 related to a copper mineralization with steeply dipping characteristics, the three alternatives would represent orebodies related to both the mining method and the productivity of the planned mining operation.

Outlining a small high-grade orebody indicates a mining operation of very modest size, where the investment in plant operations and equipment is minimized. Square-set mining, combined with labor-intensive vein mining, would be adequate in Case A.

In Case B, the ore reserves are sufficient to justify a mine of more normal size. Cut-and-fill mining or sublevel stoping appear to be appropriate for the situation, but further careful study is required to select the most feasible system.

Case C indicates the feasibility of large-scale mining, with high productivity and low operational costs. Block caving would be indicated in this case.

Widely differing plans can be created for one particular mineralization. The relationships between production capacity, ore grade, and available reserves are factors that must be included in the selection of a mining method.

Ore Evaluations

One method of determining the value of the ore is to multiply the metal content by the unit price for that mineral. For example, 0.9 t (1 st) of rock with a 2% grade of copper contains 20 kg (44 lb) of copper. Multiply the copper content by a price of $1.54/kg ($0.70 per lb) to determine that the metal would be worth approximately $31.

However, putting such a price tag on a ton of rock is a misconception; the unit price is that of metallic copper on the metal trade market. There is a long and costly procedure before the chalcopyrite becomes a marketable product, all of which are costs that have to be borne by revenue from metal sales.

From a mining viewpoint, the value is more appropriately defined after all costs for processes following the mining have been deducted from the market value. In this way, a value can be calculated for the crude ore delivered to the surface plant. This value should suffice to cover the operating cost in the mine, while giving an acceptable return on the investment.

In comparing mining methods, the point of delivery can be brought one step closer to the solid rock. An underground mine requires certain facilities and general services, no matter what mining method is applied. These include hoisting, water drainage, general haulage, etc. When the costs for these common services are deducted, the value can be calculated for ore delivered from the particular mining operation, providing an improved basis for the comparison of the mining methods.

To illustrate this reasoning, a practical example has been designed. The cost figures in the calculations are simulated, and they are not intended to be true references.

With a unit price of $1.54/kg ($0.70 per lb), the sale price is $1540/t ($1400 per st). A mine normally delivers its product to a smelter, upgraded to a concentrate of about 25% copper. The smelter treats and refines the concentrate, incurring certain costs and minor losses due to imperfections in the process. The smelter services can be applied as a cost charged to the concentrate. Assuming a smelter charge of $85/t ($77/st), the value of the concentrate delivered to the smelter becomes:

$$(0.25 \times \$1540) - \$85 = \$300/t$$

$$(0.25 \times \$1400) - \$77 = \$273 \text{ per st}$$

Costs for transporting the concentrate from the mill to the smelter must be deducted. These costs are dependent upon the means of transportation and

the distance, and they must be calculated for each individual case. Assuming a cost of $20/t ($18 per st), the concentrate is worth $280/t ($255 per st) at the mill or dressing plant.

With a theoretical grade of 2.0% copper in the solid ore underground, a 10% waste admixture is unavoidable, regardless of the mining method used. Therefore, the actual grade of the run-of-mine ore is:

$$0.02/1.10 = 0.0182, \text{ or } 1.82\% \text{ copper}$$

In the mill, the ore is concentrated to 25.0% copper, and a minor amount of the metal is lost in the tailings. Assuming that the tailings contain 0.1% copper, the production of the 25.0% concentrate requires an ore-to-concentrate ratio of:

$$0.25/(0.0182 - 0.001) = 14.5$$

This means that 14.5 t of ore are needed to produce 1.0 t of concentrate (or the same ratio in short tons). If the milling costs are $5.00/t ($4.54 per st), the value of the crude ore at the surface becomes:

$$(\$250/14.5) - \$5.00 = \$19.31 - \$5.00 = \$14.31/t$$

$$(\$255/14.5) - \$4.54 = \$17.59 - \$4.54 = \$13.05 \text{ per st}$$

Deducting a cost of $2.50/t ($2.27 per st) for the general mine installation and services results in a final value of $11.81/t ($10.78 per st) for the ore in the stope. This value must cover the direct mining costs, leaving a balance that supports the investment in the mining venture.

A similar calculation must be made for each individual case, accounting for the type of mineral mined, the grade of the concentrate, etc. There may be variations and other costs to include, but the principle remains the same. When accuracy is required, it is necessary to obtain professional information concerning the probable result of the milling process, the grade of the concentrate, and the degree of recovery.

In the example presented, only copper was considered as a source of revenue. However, many ores contain more than one recoverable metal, as well as traces of precious metals (e.g., gold, silver, etc.). Even in small quantities, these increase the value of the concentrate and can be credited to the ore value.

As a more general method of calculating the value of an ore at the mine surface, the following formula is applicable:

$$V = \{[(N \cdot c)/100] - S - F\} \cdot [(m - t)/c] - M$$

where V is the value of the crude ore at the surface in dollars per ton, N is the market price of the metal in dollars per ton of metal, S is the smelter charge or cost in dollars per ton of concentrate, F is the freight charge in dollars per ton of concentrate, M is the milling cost in dollars per ton of ore, c is the grade of

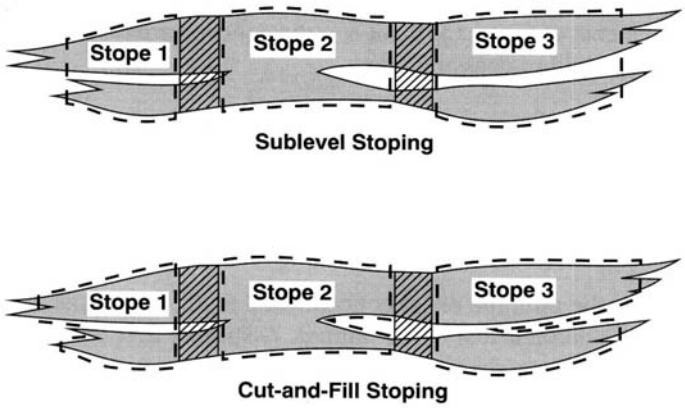


FIGURE 2.16 Stope-outline variations for different mining methods

the concentrate in percentage of metal, *m* is the grade of the crude ore in percentage of metal, and *t* is the grade of the mill tailings in percentage of metal.

Mining Costs and Ore Values

In selecting a mining method, the anticipated cost of mining exerts a major influence. However, there are considerations other than simply finding the least costly procedure of excavating the rock. The characteristics and advantages of different mining methods also must be considered. For example, a method known to require more labor than another may allow selective mining, thus producing ore of a higher grade and yielding a more valuable product.

This reasoning can be supported by an example where sublevel stoping is compared to cut-and-fill mining. For sublevel stoping, the stopes must be designed with regular boundaries, and the ore includes any low-grade and waste material within those boundaries. The cut-and-fill system allows the outlines to be adjusted to irregularities in the shape of the orebody, and it makes it possible to avoid large low-grade sections. Figure 2.16 illustrates the comparative techniques.

If a theoretical grade of 2.2% copper has been calculated from the core drilling logs, and a 10% waste admixture is included for cut-and-fill mining, the expected average grade in the material from a cut-and-fill stope becomes 2.0% copper.

The sublevel stoping layout incorporates an estimated 15% of extra material assumed to have a grade of 0.5% copper. So, the average grade for this method becomes:

$[(0.022 \times 0.85) + (0.005 \times 0.15)]/(1.00 + 0.10) = 0.0177$, or 1.77% copper

If there is 25.0% copper in the concentrate and 0.10% copper in the tailings, cut-and-fill mining requires 13.2 mt of ore to produce 1.0 mt of concentrate (or the same ratio for short tons). For sublevel stoping, the ratio becomes

15.0 mt of crude ore to produce 1.0 mt of concentrate. These ratios are calculated as follows:

Cut-and-Fill: $0.25/(0.020 - 0.001) = 13.16$

Sublevel Stoping: $0.25/(0.0177 - 0.001) = 14.97$

Furthermore, using a value of $280/t ($255 per st) of concentrate, the value of the ore produced by the two methods compares as follows:

Cut-and-Fill: $280/13.2 = $21.21/t

$255/13.2 = $19.32 per st

Sublevel Stoping: $280/15.0 = $18.67/t

$255/15.0 = $17.00 per st

The difference is $2.54/t ($2.32 per st), in favor of the cut-and-fill mining method. This difference is of a magnitude that may compensate for the higher mining costs incurred with the cut-and-fill method. Although this example is designed for educational purposes, the background exists in reality; a mining method that brings a product of higher quality out of the ground can compete with other more effective methods that yield a product of lower quality.

Productivity and Mechanization

Productivity in mining has become synonymous with mechanization, replacing manual labor with powerful machines. Over the last few decades, a tremendous development has taken place, rationalizing underground mining methods with the introduction of new machinery of increasing sizes and capacities.

Mining methods and underground working have adjusted to accommodate the new equipment. The application of mining methods has shifted toward increasing mechanization where conditions are favorable for the use of heavy machines.

Machine Considerations. Mechanization means that the majority of the underground work is accomplished by machines. In principle, the capacity of a machine is related to its size, so it is advantageous to select the largest units possible. However, there are limitations to the choice. Underground openings are not of unrestricted size, and operating within the available space limits the physical dimensions of a machine.

Another factor is the capacity that can be utilized effectively. Often, underground workings are at different vertical elevations or are otherwise separated from each other, leaving no practical way of transferring a machine from one location to another quickly. The peak capacity of a machine is of no importance when it cannot be utilized effectively with a minimum of nonproductive time.

Mine Considerations. Productive mechanization is related closely to achieving high utilization, that is, operating machines with as few and as short interruptions as possible. This is best achieved by using a mining method with several working locations, within easy reach of each other, combined with mobile equipment that can shuttle from one location to another. Alternatively, a large volume of work may be concentrated into a few locations, allowing the machinery to be less mobile because little movement is required.

Room-and-pillar mining is a typical example of a method allowing complete mechanization. Equipment can travel on nearly level roadways that are arranged for the best possible access to the working locations. Sublevel caving is equally favorable, with a large number of working faces on the same level. With sublevel caving, there is the additional advantage of being able to drill, blast, and muck without sequencing these operations in a perfect cycle.

Block caving may be considered favorable, although the production process is not as mechanically controlled as in other methods; boulders and hangups can disturb the regular flow of ore. Sublevel stoping, in turn, is an example of a method where production operations are concentrated into a few locations (e.g., the ring drilling and the mucking).

In terms of mechanization, room-and-pillar and sublevel caving have a slight advantage over block caving and sublevel stoping. The development efforts for room-and-pillar mining and sublevel caving are integrated with the production process. Block caving and sublevel stoping both require extensive development according to special plans and schedules before any production actually begins.

Efficiency Considerations. A common basis for comparing the efficiency of mining operations calculates the ton-per-worker shift ratio. This is the output from the mine per working shift, divided by the number of underground workers; it also includes the labor not directly involved in ore production. Because operating conditions vary widely, even when comparing mines using the same method, the ton-per-worker shift ratio must be considered as a general characteristic of the mining method, and it must not be weighted too heavily.

Table 2.4 lists some representative ton-per-worker shift ratios for the different mining methods. The high ends of the ranges indicate mining operations where conditions are entirely favorable and equipment capacities are fully utilized. The lower ends of the ranges indicate more normal operating conditions and degrees of equipment utilization. Absolute lower limits cannot be specified, and they would be of no interest in the context of this presentation.

Mining Equipment and Productivity

Mining techniques have become totally dependent upon machines of various types. Selecting a machine for a given type of work and matching its capacity to the required output have become two of the most important tasks in mine planning and evaluation. This section briefly reviews common machines, their applications in mining methods, and their potential productivity.

TABLE 2.4 Productivity of mining methods

	Tons-per-Worker Shift Ratio	
Method	Normal	High
Room-and-pillar	30–50	50–70
Sublevel caving	20–40	40–50
Block caving	15–40	40–50
Sublevel stoping	15–30	30–40
Cut-and-fill mining	10–20	30–40
Shrinkage stoping	5–10	10–15
Square-set mining	1–3	—

The development of new machines is a continuous process, and some of the values presented herein may change. Updated figures can be obtained by consulting the manufacturer of the equipment in question.

Drilling Equipment. The fundamental operations in hard-rock mining are drilling, blasting, and removing the broken rock from the stoping area. The drilling operation is intended to produce a hole in the rock, into which the explosive is deposited. To assure successful blasting results, the hole must be in a specific position in relation to the rock surface. The variations in the systems by which holes are drilled are comparatively few, as shown in Figs. 2.17 and 2.18.

In these figures, typical drilling units have been matched to the type of drilling. The figures contain information on expected performance and normal productivity in terms of rock volume blasted per meter drilled. Drilling performance is stated in meters per hour in Fig. 2.17, whereas Fig. 2.18 presents drilling performance in terms of meters per shift.

Loading and Transport Equipment. Although the drilling system is related to a specific mining method, loading and transport are not. The selection of equipment should entail an integration and optimization of both loading and transport operations, mostly related to the transport distance and the required capacity.

Figure 2.19 illustrates common loading and transport systems used in modern mining. It includes graphs from which an estimate of the capacity of each system can be obtained. The capacity is related to the bucket size of the loader, and it is always noted in solid cubic meters (cubic yards) of rock. This allows an easy conversion to calculate the ton-per-hour capacity for material of a different density.

The LHD technique is an integrated loading and transport operation. Its capacity always must be related to the transport distance. For crosscut loading and for loading into trucks, transportation is handled by separate carriers,

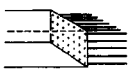
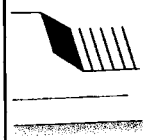
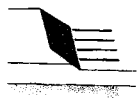
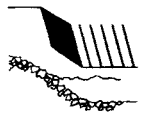
Mining Method	Room-and-Pillar Mining		Cut-and-Fill Stoping		Shrinkage Stoping	
Drilling and Blasting Technique	 **Drifting and slashing**  **Frontal benching**	 **Vertical or downward benching**	 **Roof-drilling or overhand stoping**	 **Frontal stoping, breasting**	 **Overhand stoping**	 **Frontal stoping, breasting**
Applicable Drilling Equipment	 **Mechanized Drifting Jumbo**	 **Mechanized Airtrack drill**	 **Mechanized Light Wagon drill**	 **Mechanized Drifting Jumbo**	 **Hand-held Stoper drill**	 **Hand-held Airleg drill**
Drilling Data						
Hole diameter, mm	38 – 48	64 – 76	33 – 38	38 – 48	29 – 33	29 – 33
Hole depth, m	3.0 – 5.5	as required	3.0 – 4.0	3.0 – 4.0	2.0 – 2.5	2.0 – 3.5
Drilling Equipment Performance						
With pneumatic rock drill m/h	60 – 75	15 – 25	20 – 40	60 – 70	8 –12	10 –15
With hydraulic rock drill m/h	90 –110	(25 – 35)	na	90 –110	na	na
Drilling—blasting						
Production, m^3 rock per drilled meter	1.5 – 2.0	3.0 – 4.0	0.9 –1.2	1.0 –1.4	0.7 – 0.9	0.7 – 0.9

FIGURE 2.17 Common drilling methods, equipment, and performance

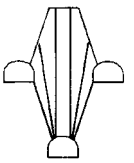
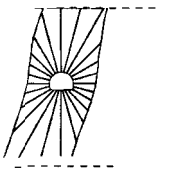
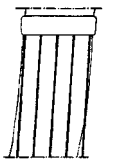
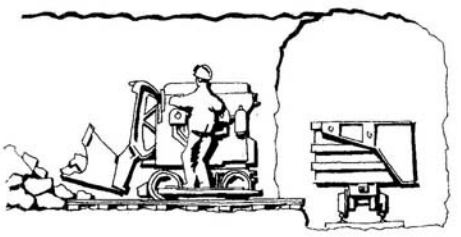
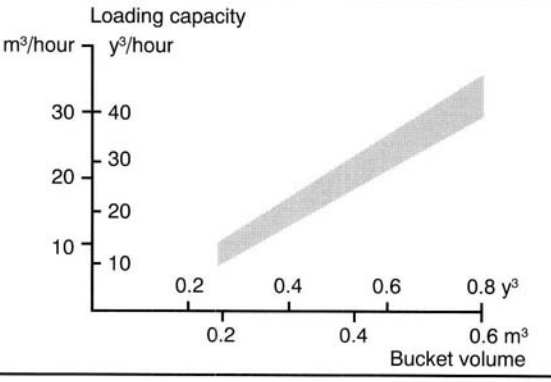
Mining Method	Sublevel Caving	Sublevel Stoping		Sublevel Stoping	
Drilling and Blasting Technique	 Fan drilling	 Ring drilling		 Parallel drilling	
Applicable Drilling Equipment	 Mechanized Fan drill	 Bar-and-arm rigging	 Mechanized Ring drill rig	 Mechanized Airtrack with DTH hammer and high pressure	
Drilling Data					
Hole diameter, mm	48 – 51 (64)	48 – 51	48 – 51 (64)	105 –115	152 –165
Maximum hole length, m	12 –15	15 – 20	15 – 25	50 – 60	50 – 60
Drilling Equipment Performance					
With pneumatic rock drill m/shift	200 – 240	50 – 60	100 –120	50	50
With hydraulic rock drill m/shift	240 – 300	na	120 –180	na	na
Drilling—blasting					
Production, m^3 rock per drilled meter	1.8 – 2.3	1.5 – 2.5	1.5 – 2.5	8 –10	14 –18

FIGURE 2.18 Common longhole drilling methods, equipment, and performance

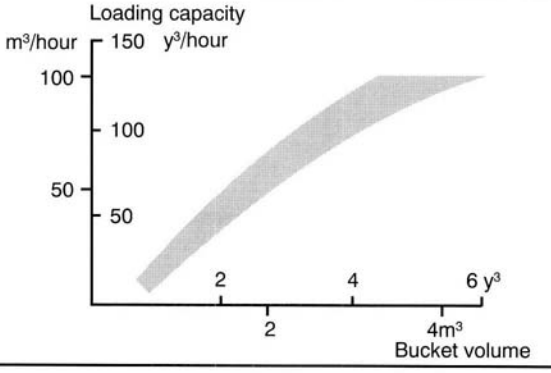


Crosscut Loading in Mine Cars



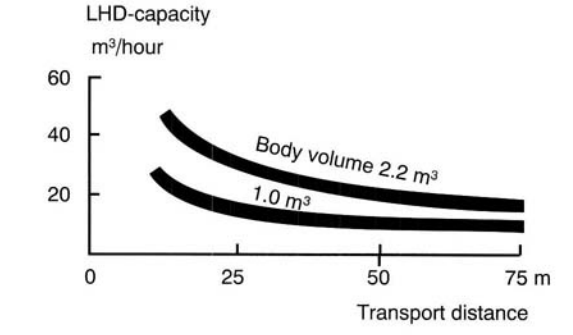


Front Loader and Transport Vehicles





Pneumatic Autoloader





Diesel-driven LHD Loader

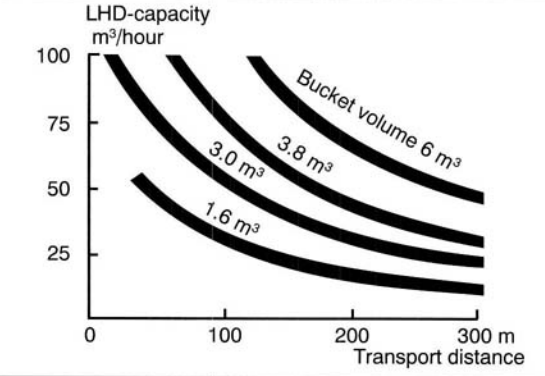


FIGURE 2.19 Common loading methods and performance

which must be matched to the size of the loader. The capacity is stated in cubic meters (cubic yards) per hour of loading capacity, and it is valid for a system in which the loader is supplied continuously by ore carriers. The capacity includes normal delays for exchanges; if the loader has to wait for a train to return from the dump, the waiting time must be included in calculating the shift capacity.

The stated capacities are intended as a guide for estimating purposes. Loading and transportation are operations for which the productivity can be calculated quite accurately when the conditions (e.g., type and size of equipment), transport distance, and velocity are known.

COMMENTARY

In the preceding sections, the author has tried to present conventional underground mining as clearly as possible, including a general discussion of the factors that influence the selection of a mining method. Naturally, there are additional considerations that cannot be included within the scope of this text.

Some readers may miss the inclusion of more detailed figures in the text. However, variations in ore deposits are so great and the state of mining technology is so dynamic that being too specific could mislead the reader. Every orebody is unique. The successful application of a mining method requires more than textbook knowledge; it also requires practical reasoning with a creative mind that is open to new impressions. The application of a mining method is a distinct challenge to any mining engineer.

The author wishes to express his gratitude to Atlas Copco MCT AB for permission to use the material included herein and for assistance in preparing the illustrations.

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CHAPTER 3

General Mine Planning

Richard L. Bullock

INTRODUCTION

Vince Lombardi once said, "Practice doesn't make perfect, perfect practice makes perfect." When it comes to building a mine that will operate at the optimum level for the set of geologic conditions from which it was developed, Lombardi's remark might be paraphrased to describe the problem: planning won't guarantee the best possible mine operation unless it is the best possible mine planning. Any sacrifice in the best possible mine planning introduces the risk that the end results may not reach the optimum mine operation desired. This chapter addresses many of the factors to be considered in the initial phase of mine planning. These factors have the determining influence on the mining method, the size of the operation, the size of the mine openings, the mine productivity, the mine cost, and, eventually, the economic parameters used to determine whether the mineral reserve even should be developed.

A little-known fact, even within the metal-mining community, is that room-and-pillar mining accounts for most of the underground mining in the United States. According to a 1973 study on noncoal mining (Dravo Corp., 1974), more than 76% of the producing mines [of more than 1089 t/d (1200 stpd) capacity] produced approximately 70,000,000 t (77,000,000 st), or 60%, of the nation's underground tonnage of material by room-and-pillar mining. That same year, 96.8% of the nation's underground coal mines produced 262,950,000 t (289,911,000 st) of coal extracted from room-and-pillar mines (National Coal Association, 1976). Thus, nearly 333,000,000 t (367,000,000 st) of U.S. raw material is produced from mines using some form of the room-and-pillar mining system.

Because approximately 90% of all mining in the United States is done by some variation of room-and-pillar mining, it is appropriate to give special emphasis to the effects of the various elements of mine planning on room-and-pillar mining. The relationship of these elements to other mining methods will become apparent as the elements are described in later sections in this chapter.

TECHNICAL INFORMATION NEEDED FOR PRELIMINARY MINE PLANNING

Assuming that the reserve to be mined has been delineated with diamond-drill holes, the items listed in the following paragraphs need to be established with respect to mine planning for the mineralized material. Although it would be nice to have all of the following information before beginning every project or mine design and development, much of this information remains very speculative at the start of a project. One can work with only the information available. Only the items preceded by an asterisk (*) are considered critical information needed prior to the initial development. There is a limit as to how much it is prudent to spend and how much time should be taken to determine the information while still at the stage of surface exploration drilling but having already established that the mineralization definitely is minable.

General knowledge of similar rock types or structures in existing mining districts always is helpful. In developing the first mine in a new district, there is far more risk of making costly errors than in the other mines that may follow.

Geologic and Mineralogic Information

The geologic and mineralogic information needed includes the following:

1. The size (length, width, and thickness) of the areas to be mined within the overall area to be considered, including multiple areas, zones, or seams.
2. The dip or plunge of each mineralized zone, area, or seam, noting the maximum depth to be mined.
3. The continuity or discontinuity within each of the mineralized zones.
4. Any swelling or narrowing of each mineralized zone.
5. The sharpness between the grades of mineralized zones within the material considered economically minable.
6. The sharpness between the ore and waste cutoff, including (a) whether this cutoff can be determined by observation or must be determined by assay or some special tool; (b) whether this cutoff also serves as a natural parting resulting in little or no dilution, or whether the break between ore and waste must be induced entirely by the mining method; and (c) whether the mineralized zone beyond (above or below) the existing cutoff represents submarginal economic value that may become economical at a later time.
7. *The distribution of various valuable minerals making up each of the minable areas.
8. The distribution of the various deleterious minerals that may be harmful in processing the valuable mineral.
9. Whether the identified valuable minerals are interlocked with other fine-grained mineral or waste material.
10. The presence of alteration zones in both the mineralized and the waste zones.

Structural Information (Physical and Chemical)

The needed structural information includes the following:

1. *The depth of cover.
2. A detailed description of the cover including: the type of cover; * the structural features in relation to the mineralized zone; * the structural features in relation to the proposed mine development; and * the presence of and information about water, gas, or oil that may be encountered.
3. The structure of the host rock (back, floor, hanging wall, footwall, etc.), including: * the type of rock; * the approximate strength or range of strengths; * any noted weakening structures; * any noted zones of inherent high stress; noted zones of alteration; the porosity and permeability; * the presence of any swelling-clay or shale interbedding; the rock quality designation (RQD) throughout the various zones in and around all of the mineralized area to be mined out; the temperature of the zones proposed for mining; and the acid generating nature of the host rock.
4. The structure of the mineralized material, including all of the factors in item 3 plus: * the tendency of the mineral to change character after being broken (e.g., oxidizing, degenerating to all fines, recompacting into a solid mass, becoming fluid, etc.); * the siliceous content of the ore; the fibrous content of the ore; and the acid-generating nature of the ore.

Economic Information

The needed economic information includes:

1. *The tons of the mineral reserve at various grades in all of the mining zones, areas, or seams. This is listed according to proven, probable, and inferred ores.
2. *The details on the land ownership and/or leaseholdings, including royalties to be paid or collected.
3. *The availability of water and its ownership on or near the property.
4. *The details of the surface ownership and surface structures that might be affected by subsidence of the surface.
5. *The location of the mining area in relation to: * any existing roads, railroads, or navigable rivers; * power; the community infrastructure; and * available commercial supplies.
6. The local, regional, and national political situations that have been observed with regard to the deposit.

PLANNING RELATED TO MINING SIZE

Sizing the Production of a Mine

There is a considerable amount of available literature on the selection of a production rate to yield the greatest value to the owners (Carlisle, 1955; Lama, 1964; Tessaro, 1960). Basic to all modern mine evaluations and design

concepts is the desire to optimize the net present value or to operate the property in such a way that the maximum internal rate of return is generated from the discounted cash flows. Anyone involved in the planning of a new operation must be thoroughly familiar with these concepts. Equally important is the fact that any entrepreneur planning a mining operation solely from the financial aspects of optimization and not familiar with today's problems of maintaining high levels of concentrated production at a low operating cost per ton over a prolonged period is likely to experience unexpected disappointments in some years with low (or no) returns.

Other aspects of the problem of optimizing mine production relate to the potential effect of net present value. Viewed from the purely financial side, producing the product from the mineral deposit at the maximum rate yields the greatest return. This is due to the fixed cost involved in mining, as well as the present-value concepts of any investment. Still, there are ". . . practical limitations to the maximum intensity of production, arising out of many other considerations to which weight must be given" (Hoover, 1909). There can be many factors limiting mine size, some of which are listed here.

1. Market conditions and the price of the product.
2. The grade of the mineral and the corresponding reserve tonnage.
3. The effect of the time required before the property can start producing.
4. The attitude and policies of the local and national government and the degree of stability of existing policies, taxes, and laws that affect mining.
5. The availability of a source of energy and its cost.
6. The availability of usable water and a method of bringing in supplies and taking out production.
7. The physical properties of the rock and minerals to be developed and mined.
8. The amount of development required to achieve the desired production related to the shape of the mineral reserve.
9. The size and availability of the work force that must be obtained, trained, and maintained.

Market Condition and Price

Normally, the individual working on mine design is not the person doing the market analysis. To digress very far from the subject of mine planning to the subject of market effects on the size of mine production is not justified in this section—the problems do not relate to any particular type of mining but rather to the product produced. However, there are a couple of points concerning the elastic nature of markets that do relate to mine design. One of the advantages of the room-and-pillar mining method is that it is extremely flexible in operation, and the mine can react to market needs faster than with other mining systems.

Most room-and-pillar mines have multiple faces for each unit or section to work. Where there is considerable irregular lateral extent to the orebody, this could involve as many as 15 to 20 faces, or as few as one or two very large

faces in each section. For short periods of time in each stope unit, it usually is possible to work only the best (or poorer) faces, depending upon the market. This usually has a drastic effect on the grade within a few days. Similarly, spare equipment can be put into reserve stopes to increase production if the remaining materials flow can take the added capacity. However, if these practices are carried on too long or too often, mine development also has to be accelerated. If maintained, old stopes can be reactivated quickly to mine lower-grade minerals that become minable because of economic cycles.

Even in room-and-pillar mining, changes in the rate of mining (momentum) cannot be assumed to be free. It often takes several months with an increased labor force to regain a production level that seemed easy to maintain before a cutback. If spare equipment is used, maintenance probably will convert to a "breakdown" overtime schedule compared to the previous preventive maintenance schedule on shift, at least until permanent additional equipment can be obtained. Nevertheless, the necessary changes can be made.

Another item often overlooked by individuals not accustomed to planning noncoal room-and-pillar mining is the situation in which mineral values are gradational. There is the option of mining through the better areas of the mineral reserve and maintaining a grade of ore that satisfies the economic objectives at that time. At a later time, when the mining economics may have changed, the lower-grade areas left as remnant ore reserves can be mined while slabbing or removing high-grade pillars. Thus, even in the latter years of the mining operation, some of the "sweetener" is left to blend with the lower grade. Although not unique to room-and-pillar mining, this technique certainly is easier to accomplish in a room-and-pillar operation than in other more complex mining systems.

Grade and Reserve Tonnage Affecting Mine Size

No magical or statistical formulae can be applied to the parameters of grade (or value) of the mineral plus the tonnage available for mining to determine the ideal rate of mining, the proper mine life, or the best ultimate recovery of mineral values. For many reasons, such a determination is a complex problem.

First, the mine life and the rate at which the product is produced must yield the desired rate of return on the investment. So, the mineral reserve must satisfy at least these conditions. However, unless mining a totally homogeneous mass, it may make a considerable economic difference as to which portion is mined first and which is mined last. Furthermore, no ore reserve has an absolute fixed grade-to-tonnage relationship; trade-offs always must be considered. In most mineral deposits, lowering the mining grade (or quality) cutoff means there will be more tons available to mine. Even in bedded deposits such as coal, the ability or willingness to mine a lower seam height may mean that more tons eventually can be produced from the reserve. In such cases, the cost per unit of value of the product generally increases.

In considering the economic model of a new property, all of the variables of grade and tonnage, with the related mining costs, must be tried at various

levels of mine production that, in the engineer's judgment, are reasonable for that particular mineral reserve. At this point in the analysis, the external restraints of production are ignored to develop an array of data that illustrate the return from various rates of production at various grades corresponding to particular tonnages of the reserve. At a later time, probability factors can be applied as the model is expanded to include other items.

Timing Affecting Mine Production

For any given ore reserve, the development required before production start-up generally is related to the size of the production. For example, a very large production may require a larger shaft or multiple hoisting shafts, more and larger development drifts, simultaneously opening more minable reserves, and a greater lead time for planning and engineering all aspects of the mine and plant. In combination, all of these factors could amount to a considerable difference in the development time of a property. In turn, this would have two indirect economic effects:

1. The capital would be invested over a longer period of time before a positive cash flow is achieved.
2. The inflation rate-to-time relationship can pressure the costs upward by as much as 10% to 20% per year, thereby eliminating the benefits of the "economy of scale" of the large-size projects.

To aid the engineer in making rough approximations of the time parameters related to the size and depth of the mine shafts, Table 3.1 lists the average times for excavating, lining, and equipping vertical concrete circular shafts and slopes with a 0.26-rad (15°) slope for mines varying in size from 4535 to 18,140 t/d (5000 to 20,000 stpd).

The lateral development on each level of a room-and-pillar mine opens up new working places, and the mine-development rate can be accelerated each time a turnoff is passed, provided that there is enough equipment and hoisting capacity available. That is in contrast to mines that have a very limited number of development faces per level but more levels.

The timing of a cost often is more important than the amount of the cost. Timing is an item that must be studied in a sensitivity analysis of the financial model for the mine being planned. In this respect, any development that can be postponed until after a positive cash flow is achieved without increasing other mine costs certainly should be postponed. The flexibility of room-and-pillar mining often allows that to be done.

Government Attitudes, Policies, and Taxes Affecting Mine Size

Government attitudes, policies, and taxes generally affect all mineral-extraction systems and should be considered as they relate to the mining method and the mine size.

Assume that a mine is being developed in a foreign country and that the political scene currently is stable but is impossible to predict beyond five to eight years. In such a case, it would be desirable to keep the maximum amount of

TABLE 3.1 Construction schedule comparison (in weeks) between production shafts and slopes

	Shaft, 6.1-m (20-ft) diam			Slope, -0.26 rad (-15°), 22 m^2 (240 sq ft)		
Depth, m (ft)	Construction	Equipment	Total	Construction	Equipment	Total
4536 t/d (5000 stpd)						
305 (1000)	43.7	20.3	64.0	46.2	5.8	52.0
610 (2000)	65.0	25.0	90.0	85.0	9.0	94.0
914 (3000	85.0	28.0	113.0	124.1	16.9	141.0
9072 t/d (10,000 stpd)						
305 (1000)	43.7	26.3	70.0	46.2	6.8	53.0
610 (2000)	65.0	31.0	96.0	85.0	11.0	96.0
914 (3000)	85.0	34.0	119.0	124.1	19.9	144.0
13,608 t/d (15,000 stpd)						
305 (1000)	43.7	30.3	74.0	46.2	7.8	54.0
610 (2000)	65.0	35.0	100.0	85.0	13.0	98.0
914 (3000)	85.0	39.0	124.1	124.1	23.9	148.0
18,144 t/d (20,000 stpd)						
305 (1000)	43.7	34.3	78.0	46.2	8.8	55.0
610 (2000)	65.0	39.0	104.0	85.0	15.0	100.0
914 (3000)	85.0	46.0	131.0	124.1	26.9	151.0

Source: Dravo Corp. (1974).

development within the mineral zones, avoiding development in waste rock as much as possible. That would maximize the return in the short period of political stability. Such a practice usually is possible in a room-and-pillar mine, but it may result in a less efficient mine during later years. Also, it might be desirable to use a method that mines the better ore at an accelerated rate to get an early payback on the investment. If the investment remains secure at a later date, the lower-grade margins of the reserve then might be exploited.

As a result of the flexibility of room-and-pillar mining, such a practice does not jeopardize the recovery of the mineral remaining in the mine; block caving or longwall mining might jeopardize such recovery. Similar situations might arise as a result of a country's tax or royalty policies, sometimes established so as to favor mine development and provide good benefits during the early years of production; in latter years the policies change. That would have the same effect as the preceding case; again, the flexibility of the mining rate and system must be considered.

PLANNING RELATED TO PHYSICAL PROPERTIES

The physical nature of the extracted mass and the mass left behind are very important in planning many of the characteristics of the operating mine. Four aspects of room-and-pillar mining are particularly sensitive to rock properties:

1. The competency of the rock mass determines the unsupported open dimensions unless specified by government regulations. It also determines whether additional support is needed.
2. When small openings are required, they have a great effect on the productivity, especially in harder materials for which drill-and-blast cycles must be used.
3. The hardness, toughness, and abrasiveness of the material determines the type and class of equipment that can extract the material efficiently.
4. If the mineral contains or has entrapped toxic or explosive gases, the mining operation will be controlled by special provisions in the government regulations.

Preplanning from Geologic Data

Using geologic and rock-property information obtained during preliminary investigations, isopach maps should be constructed to show the horizons to be mined and those that are to be left as the roof and floor. Such maps show variances in the seam or vein thickness and identify geologic structures such as channels, washouts (wants), and deltas. Where differential compaction is indicated, associated fractures in areas of transition should be examined. Areas where structural changes occur might be the most favored mineral traps, but they usually are areas of potentially weakened structures. Where possible, locating major haulage drifts or main entries in such areas should be avoided; if intersections are planned in these areas, they should be reinforced, as soon as they are opened, to an extent greater than that ordinarily necessary elsewhere in the opening.

Extra reinforcing (or decreasing the extraction ratio) also may be necessary in a metal mine where the ore becomes much higher in grade than is normal, therefore having less strength. Where the pillars already have been formed prior to discovering the structural weakness, it probably will be necessary to reinforce the pillars with fully grouted reinforcing bars or friction stabilizers. It is advisable to map all joint and fracture information obtained from diamond-drill holes and from mine development, attempting to correlate structural features with any roof falls that might occur.

Room and Support Planning Related to Productivity

In relation to the stress imposed upon the material, the rock-mass strength of the material left behind in the mine has a definite relationship to the opening size and, therefore, to the mine productivity. This relationship is felt more in noncoal, nongassy room-and-pillar mines than in mines classified as gassy. The gassy mines arc subject to regulations of the Coal Mine Health and Safety Act that designate the maximum span of the open room; other types of mines are permitted to use good engineering practices to establish optimum but safe room widths. Specific design criteria related to rock characteristics can be found in other sections of this handbook.

The more competent the rock mass, the larger the openings can be for any given thickness and depth of a mineral reserve. Generally, a larger room allows

TABLE 3.2 Relationships between room size and productivity

Type of Mine	Face Area, m^2 (sq ft)		Extraction, %	Production Rate, t/d (stpd)		Productivity t (st) per worker shift	
Limestone	156	(1680)	71	2700–4500	(3000–5000)	94.4	(104.1)
Limestone	178	(1920)	75	900–2700	(1000–3000)	66.7	(73.5)
Limestone	139	(1500)	66	2700–4500	(3000–5000)	53.3	(58.8)
Limestone	47	(510)	55	2700–4500	(3000–5000)	30.5	(33.6)
Metal ore	49	(528)	80	4500–6400	(5000–7000)	55.2	(60.9)
Metal ore	45	(480)	78	4500–6400	(5000–7000)	29.5	(32.5)
Metal ore	47	(504)	84	2700–4500	(3000–5000)	21.3	(23.5)
Metal ore	31	(339)	85	10,000+	(12,000+)	14.4	(15.9)

greater maximum production per day from that face and greater productivity from the work force. However, because other factors affect the overall mine productivity, this generalization is not infallible. Table 3.2 lists some productivity relationships, indicating a degree of correlation between the face area and the corresponding percentage of extraction and resulting productivity.

All of the mines listed in Table 3.2, except the last one, were breaking and loading rock of a similar nature. There are reasons why the limestone mines can open rooms that usually are much larger than the metal mines, but that is of little importance here. The important indication is that the mine engineer should seek to open rooms as large as possible to achieve maximum productivity without causing problems with grade control, safety, or rock stability.

Hardness, Toughness, and Abrasiveness of Extracted Material

The hardness, toughness, and abrasiveness of the material determine whether the material is extracted by some form of cutting action (raking, plowing, or scraping), by drilling and blasting, or by a combination of both methods.

Technological advances in hard-metal cutting surfaces, steel strengths, and available thrust forces allow increasingly harder materials to be extracted by continuous mining machines. The economics of continuous cutting or fracturing as compared to drilling and blasting gradually are being changed for some of the semihard or tough materials. However, for continuous mining to be competitive with modern high-speed drills and relatively inexpensive explosives, it appears that the rock strengths must be less than 68,900 to 103,400 kPa (10,000 to 15,000 psi).

At times, reasons other than simple economics favor the use of one mining system over another. A tunneling machine may be advantageous in protecting the remaining rock where blasting might be prohibited. Drilling and blasting may be necessary to avoid pulverizing a shale parting between two coal seams. A continuous boring tool may be desirable for totally extracting an orebody without personnel having to enter the stoping area.

Material Containing Gas

Whether the mine is classified as gassy drastically affects the relationship of noncoal hard-rock mine design, efficiency, productivity, and overall profitability. A gassy mine is more regulated by the Federal Mine Health and Safety Act. It would be difficult, if not totally uneconomical, to operate any hard-rock mine (regardless of the material being mined) that must drive multiple entries where only single entries are needed and that has to follow all other current special interpretations of a mine with a gassy classification as they apply to coal.

Rock Strata Conditions—Effects of Unknown Factors

Any unexpected and abnormal condition creating a stability problem in a stope or pillar or causing excessive inflow of water or gas undoubtedly causes the mine to produce less than its design capacity, at least temporarily. Although these problems may be solved, encountering them unexpectedly may delay reaching full capacity until the problems are corrected. If the ground is much worse than expected, the back and pillars may have to be reinforced more than planned, requiring time and resulting in a temporary loss of production. The size of the rooms may have to be reduced, again decreasing the productivity of that unit. Maintaining planned production may require the addition of more units and an acceleration of the development effort.

Because the development headings of room-and-pillar mines usually are well advanced beyond the operating faces, it is the development drifts or entries that normally receive most of the water or gas inflows. Sufficient pumps must be provided to handle the water, or sufficient ventilation must be provided to dilute or remove the gas. If the mine has reached the limit or capacity of either pumping or ventilation systems, the development advance then must be slowed to accommodate the problem.

To avoid problems of this type, it is important to obtain as much information as possible before attempting to develop a mine in an unfamiliar area.

PLANNING THE ROOM-AND-PILLAR MINE

When describing room-and-pillar mining, there is some confusion as to the meaning of "production development." That confusion results from different interpretations in hard-rock mines and mines for softer materials such as coal, trona, salt, and potash.

A development in a hard-rock mine is any initial opening driven to provide an access way anywhere in the mine. It may not necessarily be in the ore. Such access ways usually are driven as single openings and provide space for roads, ventilation, power, air and water lines, drainage, storage areas, shops, dump stations, offices, etc. They also expose the ore so that production mining can begin by opening rooms around pillars beginning at the point of exposure. As used in these mines, the term "development" has nothing to do with whether the opening also is starting the rooms and pillars. It does not provide

any indication of whether the pillars will be recovered while advancing or retreating, and it does not indicate whether the mine is advancing a production face in virgin ground that may or may not have other slices taken either above or below it. The term implies only that there is an opening for some mine service or where production might begin.

In contrast, basic coal-mining techniques use the term "development" to refer to that phase of the mining cycle during which multiple entries are advanced through a virgin seam, forming pillars by breaking crosscuts or breakthroughs between the entries. When the development has reached the extent of the advance, the coal-extraction process removes all or part of the pillar on the retreat. The only area of obvious similarity between the two meanings is where the development opens more minable mineral.

Some confusion also exists between the two mining applications when describing the initial entrances into the room-and-pillar mine. With coal-mining techniques, "drift entries" are driven nearly level into the side of a hill, and "slopes" are driven downgrade at an angle too low for the material to move by gravity. In hard-rock mines, the initial entrance driven nearly level into the side of a hill is known as an "adit"; if driven downgrade, it is known as a "decline" or "ramp." If an opening is driven upgrade at an angle too low for the rock to move by gravity, it is known as an "incline" or "ramp." In any mine, an opening driven upward at an angle steep enough for the rock to move by gravity is known as a "raise." A vertical opening sunk into the mine from the surface is known as a "shaft."

In metal mines, underground openings parallel to the general trend of the main structure are known as "drifts," but when driven at a right angle or a high angle to these structures or drifts, the openings are known as "crosscuts." In coal mines, the breakthroughs between entries forming the pillars sometimes are known as "crosscuts."

The two sets of terminology developed as a result of little past association between proponents of the two broad categories of room-and-pillar mining. However, this has changed, and many companies now operate both types of mines, and many of the equipment manufacturers now sell and service both groups. Furthermore, the same government research and enforcement agencies work with both types of mines.

The use of the same government agencies for both types of mines is one clue to the underlying problem of trying to administer a common interpretation of laws that have been developed by people understanding only one mining system and not really understanding the differences or the purpose for which the mine opening might be driven in the other system. For example, a single development in a limestone or metal mine may serve any of a number of purposes, none of which is the same as or has any resemblance to the same conditions that require service in a coal mine. Still, some people attempt to apply the same rationale to these extreme conditions, forcing an unjustified hardship onto all of those involved.

TABLE 3.3 Various types of mine developments classified according to their purpose

Obtain Information	Production Access Way	Service Access Way
Examples of Functions		
Quality of mineral	Expose areas of minable material	Normal movement of people
Quantity of mineral	Open access ways for production	Emergency movement of people
Continuity of mineral	Movement:	Normal movement of supplies
Physical and structural properties of mineral or rock surrounding it	Horizontally on strike	Movement of air intake
	Horizontally against strike	Movement of air discharge
Stress condition of mineral or rock surrounding it	Vertically up	Water drainage or discharge
	Vertically down	Water storage and pumps
Mineralogical-metallurgical nature of mineral		Cleanout for other openings
		Rooms for supplies, repairs, and offices
Physical Development Variables		
Inclination	Inclination	Inclination
Size	Size	Size
Depth or distance	Depth or distance	Depth or distance
Arrangement when completed	Arrangement when completed	Arrangement when completed
Method of development	Method of development	Method of development
Examples of Development Names		
Exploration adit	Main haulage drift	Person and material decline
Prospect decline	Ore shaft	Ventilation shaft
Exploration shaft	Belt slope	Main sump
Prospect drift entry	Orepass	Shaft cleanout raise
Experimental entry intersection	Production roadway	Ventilation "submain"
Exploration winze	"Main" belt-line entry	Main pump station
	Skip-loading station	Supply house and shop

It is extremely difficult to describe each and every type of mine development because of the terminology confusion and because openings can be driven in any direction, in all sizes, and by many different methods. Table 3.3 is provided in an attempt to show that there are three basic reasons for which all developments are driven. For each development, decisions must be made pertaining to each of the five physical variables. In the final analysis, each development must serve the purpose for which it was intended and usually named.

Development Driven for Mine-Planning Information

During the very early stages of evaluation and planning, a great deal of information is needed to make the correct decisions concerning the property. Therefore, the initial opening might be driven to test a particular aspect of the mineral reserve that is considered critical for successful exploitation.

Most of the technical information listed in the preceding paragraphs can be obtained from surface diamond-drill holes, particularly where the mineralized zone is fairly shallow, has continuity, and is flat. Unfortunately, it is common that the true economic potential of many mineral reserves cannot be determined until after an exploration development is driven. From the development opening, underground prospecting can further delineate the materials present, a bulk sample can be taken for metallurgical testing, and/or the true mining conditions can be revealed. This procedure involves considerable

financial exposure, and, unfortunately, the delineated reserve may turn out not to be economically suitable. Nevertheless, there may be little choice about driving a development opening if the potential of the reserve is to be proven and the risk to larger downstream capital expenditures is to be minimized.

In contrast to deep-shaft development for a test mine, it often may be possible and more economical to drive the development opening as an adit or slope.

Regardless of the reason for driving a preliminary development, three basic principles should be observed:

1. The development should be planned to obtain as much of the critical information as possible without driving the opening in a location that will interfere with later production development. In room-and-pillar mining, it usually is possible to locate at least a portion of the exploration development where it can be utilized in the developed mine, although it later may be necessary to enlarge the openings to accommodate the production equipment.
2. The preliminary development should be held to the minimum needed to obtain the necessary information, and the development should be completed as quickly as possible. Most exploration drifts for metal mines are only large enough to accommodate small loading and hauling units [from 2.4 × 3.0 m to 3.0 × 4.3 m (8 × 10 ft to 10 × 14 ft)]. If geologic information is needed, it often is possible to locate the developments in close proximity to several of the questionable structures, without actually drifting to each one. With modern high-speed diamond drills having a range of 213 to 305 m (700 to 1000 ft), these structures usually can be cored, assayed, and mapped at minimum cost.
3. Once the preliminary development has been done to satisfy a specific need, the development should be used to maximize all other physical and structural information to a maximum extent that will not delay the project.

There are no rules governing the size of the shaft or entry for exploration development. The size depends upon many factors, including the probability of mine development to the production stage. If the probability favors follow-up with full mine development immediately after exploration development, and if the shaft depth is not excessive [e.g., 244 to 366 m (800 to 1200 ft], it would be logical to sink a shaft sized for later use in production development. This is particularly true of operations normally using small shafts for production.

Using a small shaft of 3.7 to 4.0 m (12 to 13 ft) diam is not uncommon in coal mining in the United States, as shown in Fig. 3.1. In a survey of 83 conventional shafts, 31% were 12.4 m^2 (133 sq ft) [4.0 m (13 ft) diam] or less, and 49% were 16.4 m^2 (177 sq ft) [4.6 m (15 ft) diam] or less (Dravo Corp., 1974). Conventional sinking of a 3.7- to 4.0-m (12- to 13-ft) diam shaft is quite cost competitive with blind drilling a 1.8-m (6-ft) shaft that is lined with steel.

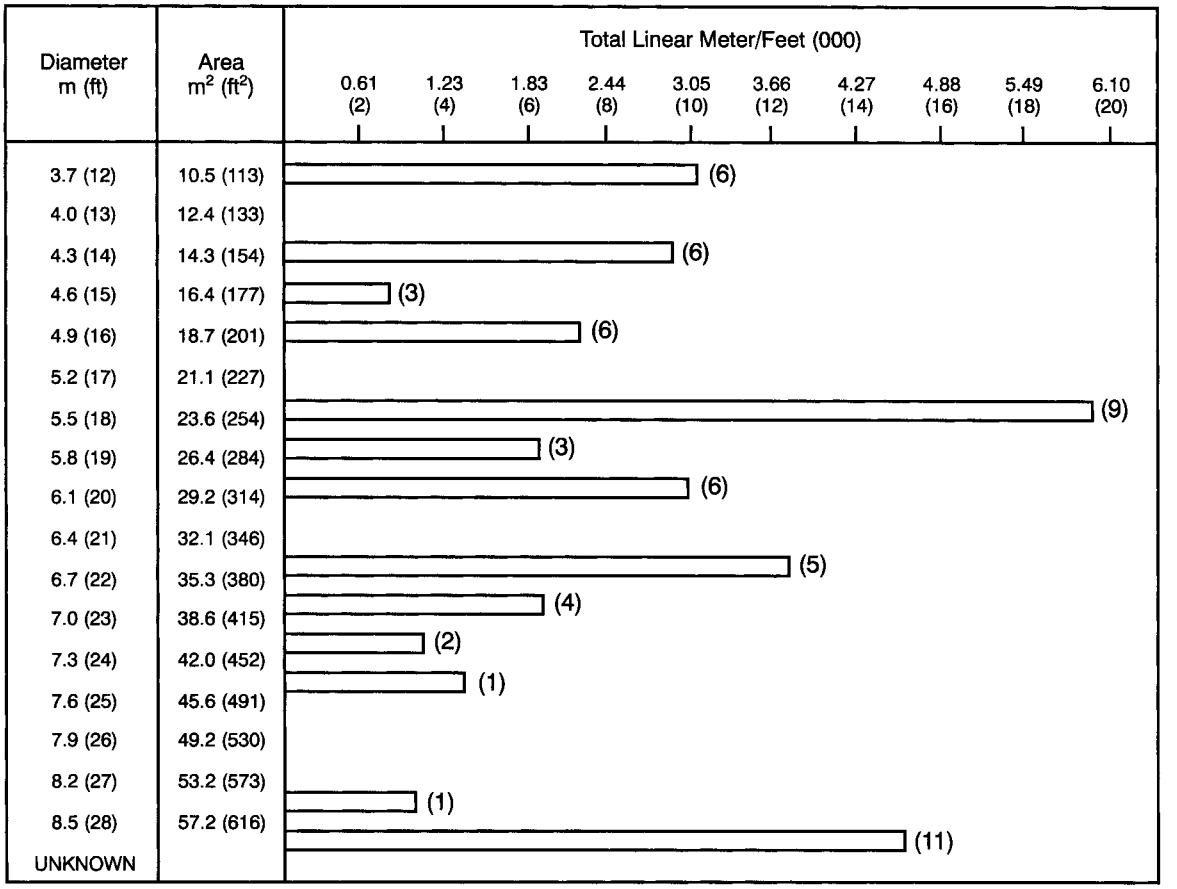


FIGURE 3.1 Size distribution of conventional circular shafts (adapted from Dravo Corp., 1974)

The Elmwood mine of New Jersey Zinc Co. is a good example of this practice. A 3.7-m (12-ft) diam conventional shaft was sunk for exploration. After the continuity and quality of the ore was proven by drifting and underground prospecting, the shaft was used for further development of the mine and eventually was used as the production shaft.

In contrast, some companies choose smaller drilled shafts to speed shaft sinking and minimize early expenses. Such shafts usually are completed considerably faster than conventional shafts but not as fast as raise boring (where an opening already exists, the hole is reamed upward, and the cuttings fall back into the mine). Because this section is directed toward initial mine openings, cuttings removal must be upward through the hole and out the collar; various methods use air or mud with either direct or reverse circulation. Of the 83 shafts surveyed, amounting to 16,587 linear m (54,420 linear ft) of boring, 58% were blind drilled. Although bored shafts stand very well without support, 58% of the surveyed shafts contained either a steel or a concrete lining.

Recently, shotcreting bored shafts has become acceptable and probably results in substantial cost savings.

Of the 83 shafts, 73% were drilled 1.8-m (6-ft) diam or smaller. Of all the blind-drilled shafts, 55% were bored 305 m (1000 ft or less), and 39% were over 305 m (1000 ft) but less than 550 m (1800 ft). Only one shaft was over 610 m (2000 ft). In the survey of conventional shafts developed between 1963 and 1973, 41 of the mines were room-and-pillar mines, and 88% of these were less than 550 m (1800 ft) deep. These data suggest that many of the mines were within reach of blind-bored shafts if other conditions such as rock hardness and abrasiveness were amenable and a small diameter shaft could have been utilized for development.

Developing the Production Access Way and System

The basic considerations in planning and developing the production access way and system are related directly to the design parameters of the development openings. These are the function in the production flow, the inclination of the opening, the size of the opening, and the arrangement and support of the opening.

The Production/Development Function. Each element of the production-flow process must be considered, and each must be planned to be compatible with the elements of flow on both sides of it. In a room-and-pillar mine, a typical sequence of these elements in the development openings for material flow from the production face might be: a preliminary load-haul-dump drift, an orepass, the main haulage drift, the main ore pocket, an orepass to the crusher-skip loading station, and a shaft for hoisting to the surface.

Although other configurations and production flow sequences are used, the preceding sequence is typical of noncoal room-and-pillar mining in the United States.

In the following paragraphs detailing inclination, size, and arrangement, the production developments from the surface are considered first, followed by descriptions of the internal room-and-pillar mine developments.

Inclination of Developments from the Surface. As described earlier, the inclination determines the name of the development. The inclination is determined by a combination of the relative elevations of the surface and the mineralized zone, and the economics of driving the development and operating the production system through the development.

Drift Entry (Adit). An adit, or drift entry, is the most economical approach when the minable material is above the floor elevation of a nearby valley. Literally thousands of Appalachian room-and-pillar coal mines have been opened this way to mine coal seams exposed on the hillsides; these are known as "punch mines." When coal was mined in the west to power steam locomotives, this same type of mine was known as a "wagon mine." Most of the early western mines used this type of development, as did most of the limestone and dolomite mines throughout the Missouri-Mississippi river basin. The

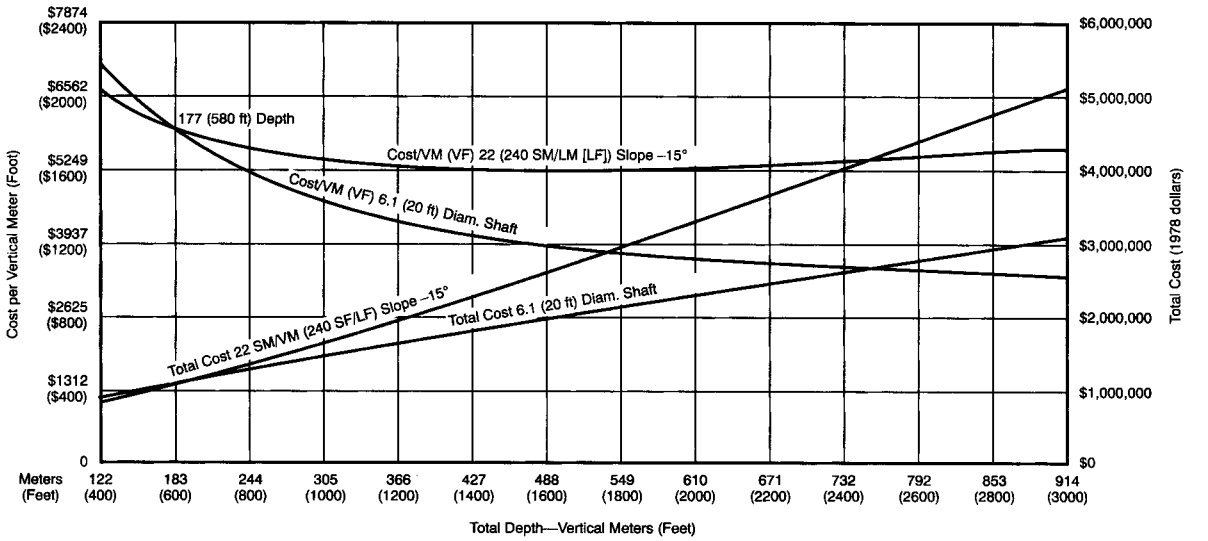


FIGURE 3.2 Comparison of cost per unit of vertical distance versus total depth (excluding equipment)

exposed ore could be opened through horizontal adits into the sides of the mountain bluffs. When the minable material goes to a depth lower than the surrounding terrain, either a slope (decline) or a shaft must be developed.

Slopes (Declines) vs. Shafts. There is no general rule governing whether a shaft or a slope is the proper type of entry to be developed; each has advantages and disadvantages. Slopes usually are limited to relatively shallow mines because, for a given vertical depth, they require approximately four times the linear distance required for a shaft. Thus, to reach a depth of only 460 m (1500 ft) a normal slope would be approximately 1840 m (6000 ft) long. Although the cost per unit distance for a decline may be somewhat lower for a slope than for a vertical shaft, the overall cost usually is higher and the development time usually is longer as a result of the greater length required. Figure 3.2 compares costs per vertical unit of distance with total depth, and Table 3.1 compares development times.

Most new shafts being sunk today are concrete lined, offering excellent ground stability. Most slopes have a rectangular shape, with the widest span for the floor and roof. In incompetent ground, slopes can be very difficult to develop and very difficult to maintain over the life of the property. In ground that must be frozen prior to excavation, a slope requires freezing a much larger volume of material than does a shaft, and it undoubtedly would cost far more for a slope than for a shaft. However, slopes do offer some advantages:

1. Where conveyor-belt haulage brings the material to a central point, it is logical to continue haulage to the surface by means of a belt conveyor up a slope. Where slopes and conveyor systems are used to move the material to the surface, the mine generally is geared for a consistent high rate of production.

2. Slopes provide easier access for mobile equipment entering or leaving the mine; in many cases, the equipment can be transported or driven into the mine intact. With a conventional shaft, large trackless equipment may have to be disassembled completely and then reassembled underground; that is particularly difficult when the unit is a large truck, and the bed must be split and rewelded. A similar advantage is found when using rail haulage in a slope for transportation of supplies–the material can remain in its original container for transport from the surface.
3. Where minor ground movements are a problem, a production slope offers considerably less difficulty in maintenance than does a vertical production shaft.

Despite the advantages of slopes, the disadvantages of cost and development time appear to have the greater influence on noncoal mines. Although most noncoal mines using room-and pillar mining are reasonably shallow, vertical shafts continue to be used for their development. In a 10-year period around 1970, only four noncoal mines had been developed by declines, and three of those were room-and-pillar mines (Dravo Corp., 1974). The depths varied from 105 to 590 m (345 to 1940 ft) with grades varying from 0.17 to 0.35 rad (10° to 20°) and averaging 0.26 rad (14.7°). For that time period, this represents only 9% of the total linear footage of noncoal mine development from the surface.

Four exceptions did not fit into the survey mentioned above: the White Pine copper mine of Copper Range Co., the Maysville mine of Dravo Corp., the Lost Creek mine of New Jersey Zinc Co., and the Gordonsville mine of New Jersey Zinc Co. Of these, the first three use conveyor belts to transport the material from the operating areas to the surface. The economics of using declines become obvious only where conveyor belts are used throughout the operation or where the mine is very shallow and rubber-tired vehicles can be driven from the working area to the surface. Many small mines, close to the surface, also are opened up with declines; the mines found in the Illinois-Wisconsin zinc district are typical of this type of entry.

Size of the Opening. The size of the opening should be tailored to accommodate the specified purpose. For production shafts, the shaft cross section is influenced (but not dictated) by the skip capacity, and the skip capacity is determined by the variables of mineral density, hoist capacity, hoisting speed, skip loading time, skip dumping time, and allowable hoisting time. Table 3.4 lists the sizes of a number of shafts and slopes for several room-and-pillar mining operations producing different products. The slopes ranged from 3.7 × 4.3 m to 3.7 × 6.1 m (12 × 14 ft to 12 × 20 ft).

Arrangement of Facilities in the Production Opening. The production shaft or slope must accommodate the conveyances for production material and may, at times, also contain some of the services such as an emergency man cage and/or ladderway. Whether such services are located in the production opening is dependent upon the number of other openings from the mine to the surface. Usually, the service facilities are contained inside pipes that may be embedded in the concrete shaft lining (Lama, 1964). Drawings of general

TABLE 3.4 Mine development from surface of typical room-and-pillar mines by product

Mine Product	Conventional Shaft: Production Shaft: Size: Average, m (ft)	Conventional Shaft: Production Shaft: Size: Range, m (ft)	Conventional Shaft: Production Shaft: Depth: Average, m (ft)	Conventional Shaft: Total Reported: No.	Conventional Shaft: Total Reported: m (ft)	Conventional Shaft: % of Dev. Rep. by Prod.	Bored Shafts: Size: Average, m (ft)	Bored Shafts: Size: Range, m (ft)	Bored Shafts: Depth: Average, m (ft)	Bored Shafts: Total Reported: No.	Bored Shafts: Total Reported: m (ft)	Bored Shafts: % of Dev. Rep. by Prod.	Slopes: Total Reported: No.	Slopes: Total Reported: m (ft)	Slopes: % of Dev. Rep. by Prod.	All Development from Surface Reported
Copper	—	—	—	1	503 (1,650)	33.0	2.7 (8.9)	1.8–3.7 (6–12)	511 (1,675)	2	1,021 (3,350)	67.0	—	—	—	1,524 (5,000)
Fluorspar	N.A.	N.A.	219 (720)	1	219 (720)	100.0	—	—	—	0	—	0	—	—	—	219 (720)
Gilsonite	—	—	—	—	—	0	1.8 (5.9)	1.5–3.4 (5–11)	450 (1,475)	2	899 (2,950)	100.0	—	—	—	899 (2,950)
Gypsum	—	—	—	—	—	0	2.4 (8.0)	2.4 (8.0)	107 (350)	1	107 (350)	20.8	1	405 (1,330)	(79.2)	512 (1,680)
Lead	4.7 (15.3)	3.8–6.1 (12.5–20)	—	11	3,984 (13,070)	65.0	2.1 (6.8)	1.5–2.7 (5–9)	326 (1,070)	5	1,631 (5,350)	26.6	1	518 (1,700)	8.4	6,133 (20,120)
Limestone	—	—	—	—	—	0	1.9 (6.2)	1.9 (6.2)	195 (640)	1	195 (640)	100.0	—	—	—	195 (640)
Potash	5.6 (18.5)	4.6–6.7 (15–22)	564 (1,850)	5	2,030 (6,660)	55.7	2.3 (7.5)	1.8–3.0 (6–10)	538 (1,766)	3	1,615 (5,300)	44.3	—	—	—	3,645 (11,960)
Salt	5.0 (16.5)	4.9–5.5 (16–18)	469 (1,538)	6	3,042 (9,980)	76.4	2.6 (8.4)	2.3–3.4 (7.5–11)	235 (772)	4	942 (3,090)	23.6	—	—	—	3,984 (13,070)
Trona	6.0 (19.6)	3.7–7.3 (12–24)	451 (1,480)	9	3,975 (13.040)	100.0	—	—	—	—	—	—	—	—	. —	3,975 (13,040)
Uranium	4.1 (13.5)	3.5–4.3 (11.5–14)	376 (1,234)	11	4,612 (15,130)	39.3	1.9 (6.3)	1.5–5.0 (5–16.5)	255 (836)	28	7,135 (23,410)	59.2	—	—	—	11,747 (38.540)
Zinc	5.9 (19.3)	5.5–7.0 (18–23)	604 (1,980)	9	4,395 (14.420)	60.1	1.8 (5.9)	1.5–2.7 (5–9)	244 (799)	7	1,704 (5,590)	23.3	2	1,219 (4,000)	16.6	7,318 (24,010)
				53	22.759 (74,670)	56.9				53	15,249 (50,030)	38.0	4	2,143 (7,030)	5.3	40,151 (131,730)

Source: Modified from Dravo Corp. (1974).

shaft arrangements are contained in other literature (Dravo Corp., 1974; Gerity, 1975).

Most new conventional shafts are lined with 0.3 to 0.6 m (1 to 2 ft) of concrete. However, there are cases in which only part of the shaft is lined, such as Asarco's Young mine, in which only the 107-m (350-ft) portion passing through incompetent rock is lined. Timbering for support of shafts generally is limited to deepening older rectangular shafts or to small mine openings not using mechanized shaft sinking. Hoisting in a shaft is accomplished either with two containers (skips) that balance each other or with one large skip that is balanced by a counterweight. Figure 3.3 illustrates the various hoisting arrangements that are or have been in use.

Most slopes (declines) utilize roof bolting and/or shotcrete with a concrete portal. The three arrangements for production slopes are: (1) one open compartment, (2) two side-by-side compartments, or (3) two over/under compartments.

The decision to use a conventional hoisting system or a Koepe hoisting system is not limited to room-and-pillar mines. However, because room-and-pillar mines normally are fairly shallow and most of the hoisting material is collected on one level, conventional hoisting systems are prevalent in room-and-pillar mines in the United States.

Drifts, Entries, and Crosscuts for Production. Although the following information primarily addresses noncoal room-and-pillar mines, many of the concepts apply to basic mine-development planning for other types of mines.

The configuration of a room-and-pillar mine generally follows the shape of the mineralized orebody to be mined. Therefore, the basic shape of the mine should start by determining the path of the main haulageways (entries), branching from these to reach the lateral extent of the reserve. Wherever possible, a few basic principles should be followed in planning the production developments.

The central gathering point for hoisting should be as near to the center of the minable mass as practical. That keeps the haulage, personnel, and material movement distances to a minimum. Although this may mean that a large shaft or slope pillar is left in the center of the mineral reserve of a flat-lying deposit, it does make the mining operation more efficient. At the end of the mining project, most of the pillars probably can be extracted.

Normally, room-and-pillar mines without ground-stability problems should be developed as symmetrically as possible to allow the maximum number of working areas to be opened with a minimum of development time. In trackless mines, it is advisable to cut the main haulage opening as much as 50% larger than the minimum size needed to operate the existing equipment. Large main haulageways improve productivity, ventilation, and road conditions. They also allow productivity to be maintained as the haulage distance to the shaft increases by permitting efficient operation of larger equipment.

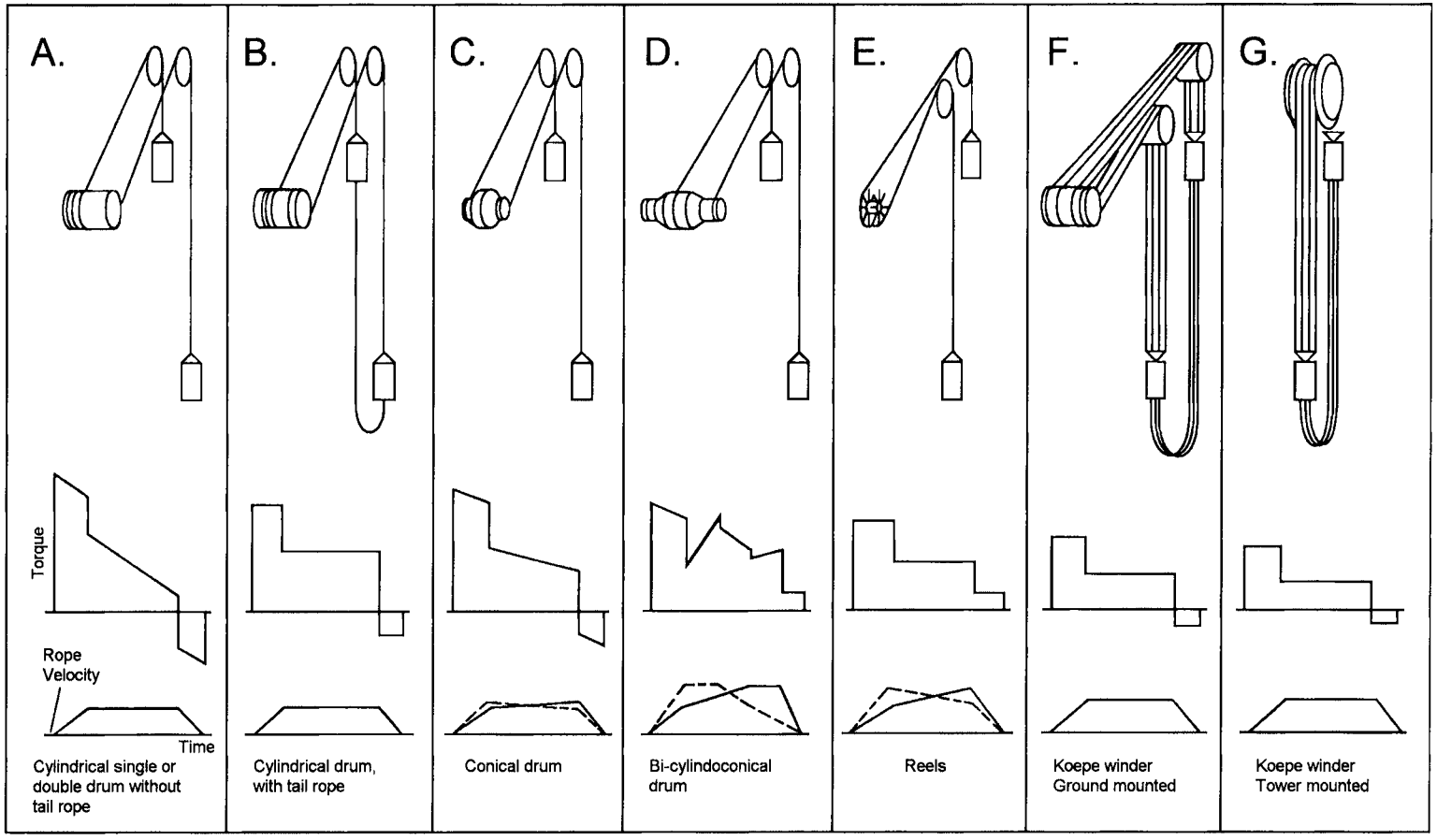


FIGURE 3.3 Types of hoists, with torque and rope velocity comparisons (adapted from Brucker, 1975)

Haulage roads and railroads should be kept as straight as possible, with steep grades other than conveyor slopes being kept as short as possible. The grade limits depend upon the type of haulage equipment being used. For main haulageways, railroad grades should be less than 3% and it is advisable to keep the grade less than 8% for rubber-tired diesel equipment. However, for very short distances where the equipment is allowed to gain mass momentum, 15% to 18% grades can be negotiated quite efficiently. If a 20% decline is to be driven for a conveyor system or service slope, rubber-tired diesel-powered loaders are the best equipment for loading and hauling the material from the development.

The radius of a curve that can be negotiated by trackless equipment is somewhat dependent upon the size of the equipment in relation to the size of the drift and the turning radius of the equipment. Normally, a radius of 15 m (50 ft) is handled easily by any trackless mining equipment; unless it is articulated, a road grader usually requires the greatest turning radius. For rail-mounted haulage, a radius of 24 m (80 ft) is adequate for stope development, but a limited-service main line requires a minimum radius of 38 m (125 ft). However, a radius of 60 to 150 m (200 to 500 ft) would be appropriate for main-line haulage, depending upon the speed of the train.

Preceding sections have outlined the need for operating flexibility in relation to the size, shape, and thickness of the mineral reserve. Other aspects of the mine also require flexibility in the plan. For example, it often is advantageous to incorporate a lower level of uniform grade below the main ore horizon to create a ventilation circuit, providing mine drainage that removes water from the operating headings and roadways to make ore movement more efficient. Figures 3.4 and 3.5 illustrate the Fletcher mine of St. Joe Minerals Corp., incorporating multiple-level planning with all of the planning features described. This mine has been extremely efficient, and it is expected to remain efficient as it grows for another 20 years. The haulage drifts in this mine are driven 5.5 × 9.8 m (18 × 32 ft), and the mine uses very large equipment such as 36-t (40-st) or larger trucks.

In contrast, conditions in eastern Tennessee do not warrant such large drift openings in trackless room-and-pillar developments. For example, at Asarco's Young mine, the openings are driven 4.3 × 5.2 m (14 × 17 ft). In ore, the openings are slabbed to widths as great as 12.2 to 18.3 m (14 to 17 ft). At the Elmwood property of New Jersey Zinc Co. in central Tennessee, the original developments were only 2.4 × 4.3 m (9 × 14 ft). In eastern Tennessee, the same company's development drifts were 4.0 × 4.0 m (13 × 13 ft), although they sometimes had arched backs. At the nearby Davis mine of US Steel, the developments are driven 3.0 × 4.3 m (10 × 14 ft). These examples show the considerable differences in the practices of various companies within the same district.

It is important for the mine planner to remember that the main haulage drifts will be used for all of the ore produced in the mine. Therefore, these drifts should be planned carefully for optimum production. If the ore can be moved quickly to an efficient main transportation system, it will not matter nearly as much if the stope developments are small or crooked.

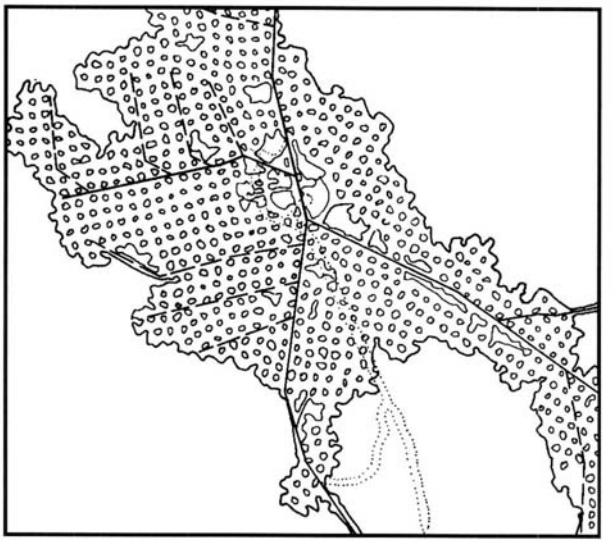


FIGURE 3.4 Mine development at Fletcher mine, showing symmetric development to allow maximum number of stopes to be placed into operation in minimum time. Shaft is located as close to center of orebody mass as possible (adapted from Bullock, 1973).

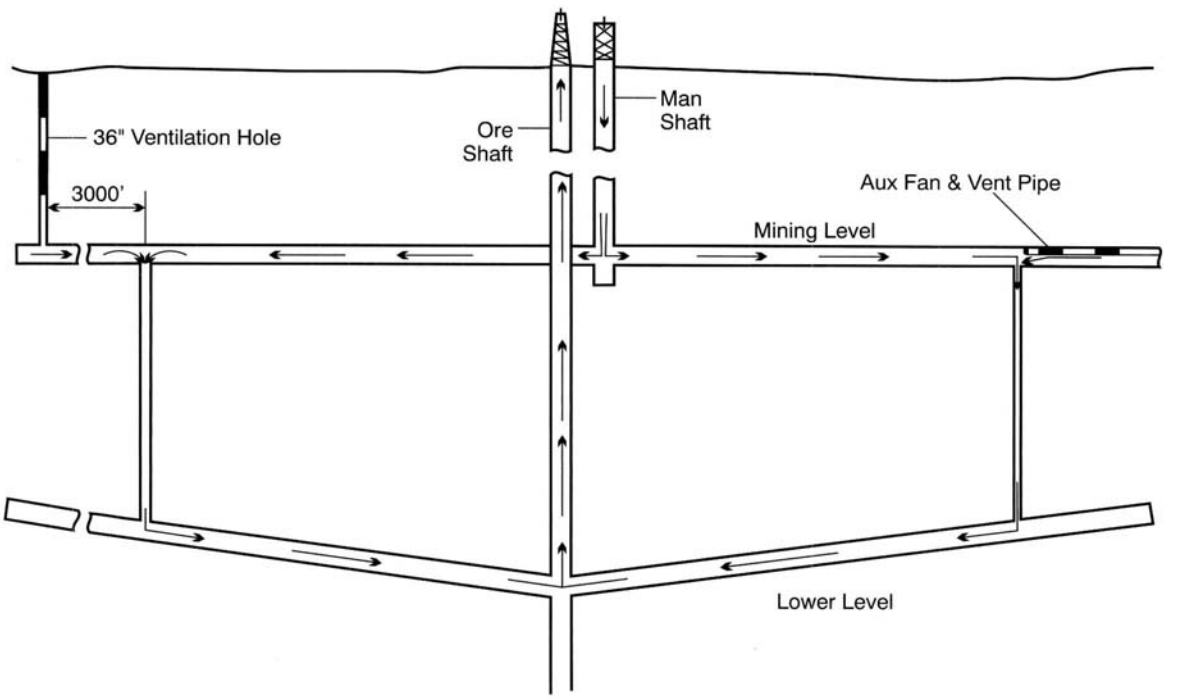


FIGURE 3.5 Typical ventilation system used in St. Joe Mineral Corp.'s trackless mines (adapted from Bullock, 1973)

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Raises and Ramps as Production Developments. The use of raises and ramps as production arteries also must be planned. If the ore is to be transferred vertically to the main transportation artery, two different approaches may be used: either the ore is dumped into an orepass where it moves by gravity and can be reloaded at a lower level, or the ore is kept in the same conveyance for transport down or up a decline or ramp to the desired location.

In modern rail or conveyor-haulage mines, material from the stopes usually is dropped through an orepass to reach a lower gathering point. Years ago, in the Old Lead Belt of Missouri, mining major sloping areas commonly involved moving the ore long distances by rail while changing elevations both upward and downward by several hundred feet. When these mines were closed, they contained more than 483 km (300 miles) of connecting track, and the average haulage distance was 11.3 to 12.9 km (7 to 8 miles) to a central dump point.

To move ore up a grade that is too steep for self-propelled track or trackless haulage, the ore either can be hoisted up an incline in individual rail cars, or it can be conveyed up an incline for subsequent transfer into rail cars or onto another conveyor. For the past 60 years, the Eagle mine of New Jersey Zinc has used the underground rail-incline system. The Carlsbad operation of IMC Corp. (Johnston, 1963) and the Young mine of Asarco (Li, 1976) are two examples of the conveyor-to-rail system used to extend and add flexibility to the underground rail system (now closed). The haulage system of the Young mine typifies the flexibility needed for room-and-pillar mining:

> *Rail haulage and gallery conveyors transport ore from stope loading pockets to the main shaft. . . . At present, two rail networks haul ore from 1.1 to 1.8 km (0.67 to 1.125 miles) radial distance from the shaft. . . . In the northwest quadrant, ore is now hauled from the stope areas by a 610-m long, 1.2-m (2000-ft long, 48-in.) conveyor, at a +15% grade. . . . The eventual length will be 1219 m (4000 ft) of conveyor. . . . In the southwest quadrant, 1097 m (3600 ft) of conveyor, at 10% is used. . . . In addition, from this southwest conveyor, a 701-m (2300-ft) conveyor will bring ore from the southeast direction up an 11% grade.*

In modern mines using trackless haulage throughout, the practices are mixed—some operators use steep ramps, whereas others use orepasses and a reloading system. The advantages of not having to reload the material include no time lost for dumping and reloading, no extra mine development for an orepass, no extra cost for installing a reloading system, elimination of the high-risk chute operator's position for reloading, and maintenance of less equipment.

Factors favoring the use of an orepass include no time lost traveling an inefficient curving ramp, no time lost negotiating steep grades, less danger to vehicle operators because haulage is on moderate grades, lower maintenance and operating costs for haulage vehicles operating on moderate grades, and the availability of automatic or semiautomatic chutes for loading trucks or belts. High-productivity feeders can load large trucks within 1 or 2 min, or front-end loaders can reload themselves quickly; a chute operator would not be required.

The decision as to which method should be used must be based upon the tonnage eventually taken out of the area in relation to the criteria for efficient

haulage. Today's trend is to maintain very high productivity; if the haulage ramp is at all steep, most room-and-pillar mine operators favor orepasses and reloading if the tonnage justifies the installation. If ramps are used, it has been found that ramps with 3.14-rad (180°) turns and long straight sections are more efficient than continuous spirals.

When planning raises for production, several critical factors must be considered: the size of material to pass through the raise and whether it will flow by gravity, the effects of passing ore on the size and stability of the opening, and the least-cost method of developing the raise in relation to its length. The same design problems become even more important in mining systems that rely heavily upon orepasses, chutes, or drawpoints.

Ore-Storage Pocket. Some surge capacity must be provided between the normal mine production gathered from various parts of the mine and the conveyance that transports the material to the surface. The need arises because the material usually has many production paths to reach the shaft, slope, or adit, but only one path out of the mine. Therefore, the single-path conveyance is vulnerable to downtime for maintenance and repair, but such downtime cannot be allowed to disturb the rest of the mining cycle. Similarly, the multipath production can operate intermittently (two shifts per day), while the material can flow from the mine continuously.

Production in room-and-pillar mines normally originates from one or two levels, and the material does not have to be dropped a great distance to be stored for hoisting. Thus, the problem is simplified in one sense, but it also can be a problem if there is insufficient room between the bottom level of the mine and the skip-loading facility. An example of a good arrangement is illustrated in Fig. 3.6, where the distance between the bottom level and the top of the skip is about 58 m (190 ft). This allows ample room for a 4535-t (5000-st) ore-storage pocket. In the mines using this layout, there actually are two such pockets side by side, with room to develop duplicate pockets between the top and bottom levels of the mines for potential storage of 18,141 t (20,000 st).

The correct size of the surge pocket depends upon what it is intended to accomplish. For example, if management decided to try to hoist 20 shifts per week but stoped two shifts per day in a mine producing 4535 t/d (5000 stpd) between midnight Friday and 7:00 a.m. Monday, the minimum pocket size would be found from:

$$C = [S_m/S_h] \times S_{md} \times T_{ms}$$

where C is the capacity needed in short tons or metric tons, S_m is the number of shifts per week the mine operates, S_h is the number of shifts per week the hoist operates, S_{md} is the number of shifts the mine is down, and T_{ms} is the number of metric tons or short tons produced per mine shift.

For this example, the capacity would be calculated as:

$$C = [10/20] \times 6 \times 2500 = 7500 \text{ st}$$

$$C = [10/20] \times 6 \times 1267 = 3801 \text{ t}$$

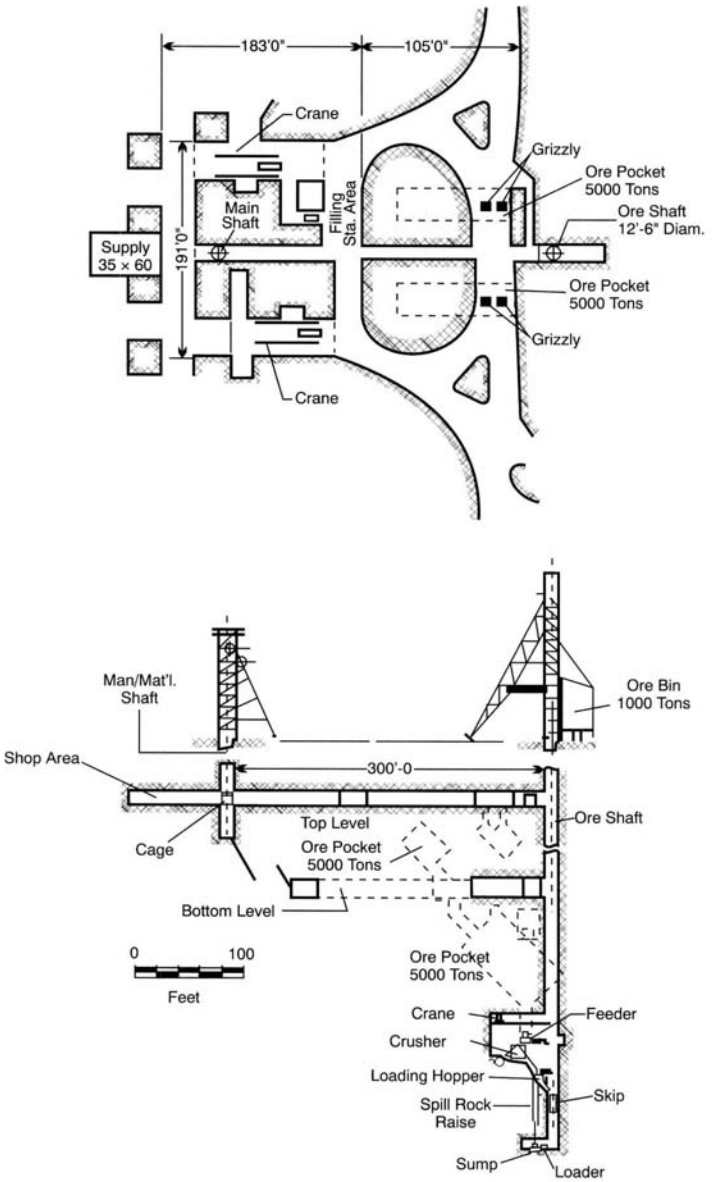


FIGURE 3.6 Plan view and cross section of typical shafts and shop area of a room-and-pillar noncoal mine (adapted from Casteel, 1973)

In this example, the storage was calculated to be 150% greater than the daily mine capacity. Although this is typical for some room-and-pillar operations, it is much larger than the requirement for many others. Table 3.5 summarizes hoisting practices of room-and-pillar mines, listing 11 hoisting rates that range from 227 to 1089 t/h (250 to 1200 stph) and storage pockets that range from 227 to 13,608 t (250 to 15,000 st). Assuming that hoisting is done 21 hr per day, the storage capacity of these 11 operations varies from 3% to 191% of the hoisting capacity. There is no general agreement on the optimum size of

TABLE 3.5 Hoisting practices at room-and-pillar mines

Mine No.	Hoist Type	Shaft Depth, m (ft)	Shaft Size, m (ft)	Shaft Purpose	Hoisting Capacity, t/hr (stph)	Skip Size t (st)
1	Dbl cyl drum	704 (2310)	3.0 × 4.9 (10 × 16)	Production	295 (325)	10.2 (11.2)
	N.A.	704 (2310)	3.0 × 3.7 (10 × 12)	Service	—	—
2	Dbl cyl drum	617 (2025)	4.9 (16 diam)	Production	363 (400)	14.1 (15.5)
	Friction		3.7 (12 diam)	Personnel	—	—
3	Dbl cyl drum	396 (1300)	5.4 × 5.4 (18 × 18)	Production & personnel	327 (360)	9.1 (10)
	N.A.	396 (1300)	1.5, 2.4 & 3.7 (5, 8 & 12 diam)	Vent & escapeway	—	—
4	Dbl cyl drum	347 (1134)	2.7 × 8.5 (9 × 28 Oval)	Production	771 (850)	15.0 (13.1 PL) 16.5 (14.4 PL)
5	Dbl cyl drum	320 (1050)	4.6 (15 diam)	Production	454 (500)	11.3 (12.5)
	Dbl cyl drum	274 (900)	6.7 (22 diam)	Personnel & supplies	—	—
6	Dbl cyl drum	311 (1020)	4.3 (14 diam)	Production	318 (350)	6.8 (7.5)
	Dbl cyl drum	280 (920)	4.3 (14 diam)	Personnel & supplies	—	—
7	Dbl cyl drum	283 (927)	6.1 (20 diam)	Production	363–431 (400–475)	7.3–9.1 (8–10)
	N.A.	259 (850)	4.9 (16 diam)	Personnel & supplies	—	—
8	N.A.	366 (1200)	4.6 (15 diam)	Personnel & supplies		
9	Friction	1021 (3350)	4.9 (16 diam)	Production	544/1089 (600/1200)	21.8 (24)
	Friction	991 (3250)	4.9 (16 diam)	Personnel & supplies	—	—
10	Friction	372 (1220)	4.9 (19 diam)	Production, personnel & supplies	N.A.	18.1 (14.5 PL) 20 (16 Payld)
11	Friction	454 (1490)	6.1 (20 diam)	Production	454 (500)	16.3 (18)
	N.A.	454 (1490)	5.1 (20 diam)	Personnel & supplies	—	—
12	Sgl cyl drum	408 (1340)	3.8 (12.5 diam)	Production	227 (250)	7.7 (8.5)
	N.A.	305 (10000)	3.8 (12.5 diam)	Personnel & supplies	—	—

Source: Modified from Dravco, Corp., 1974.

Skip Type	No. of Skips	Cage Details, m (ft)	Rope Speed, m/min (fpm)	Hoist, kW (hp)	Trip Time, (min)	Storage & Shaft t (st)	Hoist Operation
Bottom dump	2		N.A.	N.A.	2.1	362 (400)	Fully auto
—		2.4 x 2.0 (94" x 80")	N.A.	N.A.	N.A.	—	Fully auto
Bottom dump alum. & steel	2		411 (1350)	1864 (2500)	1.5–1.7	10.8–13,605 (12–15,000)	Part (convert'g)
—		1.2 x 1.2 4' x 8' (2–deck) (20 workers)	137 (450)	37 (50)	N.A.		Fully auto
Bottom dump	2	1.8 x 1.8 (6' x 6')	N.A.	N.A.	N.A.	7256 (8000)	(Fully auto prod) Nonauto (workers)
—	—	1.4 (4.5' diam. torpedo)	N.A.	N.A.	N.A.	—	Nonauto
Bottom dump	2	N.A.	N.A.	N.A.	N.A.	7256 (8000)	N.A.
Bottom dump	2		N.A.	N.A.	N.A.	907 (1000)	Fully auto
—		2-deck (28 workers)	N.A.	N.A.	N.A.	—	N.A.
Bottom dump	2	—	N.A.	N.A.	1.2	227 (250)	Fully auto
—	—	1.8 x 3.7 (6' x 12')	N.A.	N.A.	N.A.	—	Nonauto
Bottom dump	2	—	N.A.	N.A.	1.2	4989 (5500)	Fully auto
—	—	1.8 x 4.3 6'x 14' (25 workers)	N.A.	N.A.	N.A.	—	Nonauto
—	—	N.A. (40 workers)	—	—	N.A.	—	N.A.
Side dump	2	—	549 (1800)	2610 (3500)	N.A.	13605 (15000)	Fully auto
—	—	2.1 x 3.0 (7' x 10') (2 deck) (20 workers)	396 (1300)	597 (800)	N.A.	—	Semiauto
Bottom dump	1	2.4 x 2.7 (8' x 9') (37 workers) (12 ton)	411 (1350)	746 (1000)	N.A.	3628 (4000)	Fully auto (ore) Semiauto (workers)
Bot. arc gate	2		381 (1250)	1044 (1400)	4.3	1361 (1500)	Fully auto
	—	2.4 x 4.3 (8' x 14')	N.A.	N.A.	N.A.	—	N.A.
Bottom dump	2		N.A.	N.A.	N.A.	9070 (10000)	Fully auto
	—	N.A.	N.A.	N.A.	N.A.	—	Semiauto

an ore pocket; both the nature of the operations and the management priorities differ. However, the two major considerations are: (1) whether there is storage at the discharge end of the hoisting operation before the ore proceeds to the next step in the flow, and (2) the size of this storage and whether interruptions before the next step are critical.

Another reason for using ore-storage pockets before hoisting is to keep various products or waste materials segregated for separate hoisting. The products may be different minerals, or they may be the same mineral but owned by different people. The ownership might insist that the material be kept separate until after crushing, weighing, and sampling so that royalties on the value can be paid correctly. Such practices are quite common in mines where government ownership of the mineral is involved.

Planning the Service Developments

Any mine opening is a service development if it is not developed specifically to gain information about the mineral reserve and mining conditions, to obtain a bulk sample, or to handle the production system. Service developments are subject to the same basic considerations as are the production openings, i.e., their functions, inclinations, sizes, and arrangements. The name of the development is a combination of the function and the inclination, as are the names of production developments. Thus, service developments include ventilation raises, personnel and materials service shafts, sump cleanout drifts, emergency escape shafts, etc. A service development may serve a combination of functions or change its function over the life of the property. Basically, the design rationale for service developments follows that of production developments, modified by special considerations associated with the service function.

Ventilation Openings. The current trend in ventilation raise and shaft development is for the opening to be bored or upreamed, rather than drilled or blasted. Such holes can be completed by an outside contractor or by the mine crew that may own or lease a boring machine.

Of the total development from the surface into noncoal mines, 22.4% of the developments over a period from 1963 to 1973 were bored shafts (Dravo Corp., 1974). Of these, approximately 25% were bored by the mine owners. Almost all of the 22.4% (75 of 82) are thought to be for noncoal mines using some form of open-stope room-and-pillar mining. Most of the bored shafts were either 1.5 or 1.8 m (5 to 6 ft) diam; these may be too small to meet today's requirements. The trend should be to enlarge the opening size to reduce ventilation power consumption and to move the needed amount of air at minimum cost. That is a problem of optimization that has to be solved for each individual user. In horizontal developments or slopes, ventilation tubing or ducting is used in most metal mines, and brattice cloth or canvas is used in coal mines.

Shop and Storehouse Developments. The efficiency of a modern mining system depends heavily upon the productivity and the availability of the equipment used to extract the material. Compared to other mining methods, little gravity movement of the ore is used in room-and-pillar mines. Therefore,

most of the energy to move the material must be provided. Because the material may be very heavy and/or abrasive, and because the environment of an underground mine imposes adverse operating conditions, underground mining equipment requires a great deal of maintenance and repair. One of the most serious and most prevalent errors made in designing a mine is the failure to provide adequate space and equipment for necessary maintenance and repair work. The amount of service, if any, provided underground depends upon several factors:

1. The degree of difficulty in moving equipment into or out of the mine is a major consideration. If the property is a limestone mine with adit entrances and a good shop on the surface, it probably would not be advantageous to duplicate the facilities and personnel underground.
2. Underground shops should provide a safe and good working environment. In mines that are gassy or carry a gassy classification, building underground shop facilities may not be practical. However, there are the same advantages for gassy and nongassy mines; therefore, some room-and-pillar trona mines have very good, large, and well-equipped underground shops even though the mines are classified as gassy.
3. If the active working area will be abandoned totally after a fairly short life moving to another such area nearby it would not pay to invest in an extensive underground shop area. Mines such as the punch mines in Appalachia are typical examples of this situation.

Most other room-and-pillar mines should develop good underground shop facilities. In the New Lead Belt of Missouri, where the mines are developed with shafts 152 to 396 m (500 to 1300 ft) deep, all of the mines rely upon large mining equipment and have well-developed underground shop areas. The total area usually utilized as a shop area can range from 465 to 1394 m^2 (5000 to 15,000 sq ft). The important criteria for shop design are to provide:

1. One or two large bridge cranes over the motor pits.
2. Easy access from several different directions for access to and around the cranes when they are in service for extended periods of time.
3. A motor pit and service area for scheduled lubrications performed as part of the preventive maintenance program.
4. A separate area for welding operations.
5. A separate area and equipment for tire mounting and repair.
6. A separate area for recharging batteries.
7. Various work areas equipped with steel worktables or benches.
8. Close proximity to the main supply house.
9. An area for washing and steam cleaning the equipment.
10. An office for the shop foreman, records, manuals, catalogs, drawings, and a possible drafting table. The main pit areas should be visible from the office windows.

The size of the underground supply room also depends upon several factors. Some mines have no supply room, whereas others have very large rooms providing as much as 1115 m^2 (12,000 sq ft). Factors influencing the inventory policy include the frequency and ease with which the mine receives supplies, whether a supply system is installed at the collar or portal of the mine, and the dependability of the parts suppliers for the equipment being used.

Sump Area and Pump Station. In mines below surface drainage, areas must be provided to store the water before it is pumped to the surface and discharged. There is no general rule of thumb to determine the capacity for water storage. The water flow into mines in the United States has varied from 0 to 2208 L/s (0 to 35,000 gpm).

Adequate pumping capacity is the only permanent solution to removing water from a mine. However, fluctuations in the water inflow and/or the time periods when some or all of the pumps are inoperable must be handled with an adequate sump capacity. Sumps also are needed in wet trackless room-and-pillar mines; the rubber-tired vehicles traveling on the roadways create fine material that eventually collects in the water ditches if the roadways have drips or continuous streams of water on them. Although this situation should be corrected to keep water off the roadways, it invariably occurs or reoccurs. A place must be provided for the fines to settle out of the water so that they do not damage the clear-water high-head pump impellers. That sometimes can be done effectively in a small catch basin that is cleaned out frequently. However, if the water flows are substantial, the main sump receives the bulk of the fine material. As a result, provisions must be made to clean the sump with conventional or special equipment, divert the water into another sump while one sump is being cleaned, provide a place to put the fines ("soup") removed from the sump, and devise a method of transporting the fines there.

One of the best methods of cleaning a sump is to use conventional front-end loaders to remove the material. However, a ramp must be provided down to the sump-level floor. If deep-well impeller sections are suspended from a pump station down into the sump, considerable care must be taken to avoid bumping the impellers with the loader. Other systems using slurry pumps, scrapers, or diverting the material into skips have been tried, but there still is no good way to clean a sump.

Diverting the water flow into another sump while cleaning one sump also can be a problem unless at least twice the normally required pumping capacity is provided. Half of the pumps have to handle all of the inflowing water while one sump is being pumped down and cleaned. Mines that make large amounts of water also produce large amounts of fine material that fill the sumps much faster than anticipated. The sump of a new mine in the development stage cannot be cleaned too soon; if the mine is not producing much water at a particular stage of development, the sump still should be kept clean. The next shift may bring on more water than can be handled with half the pumps, and once that point is reached there will be no way to get to the muck in the bottom of the sump for cleaning.

Two methods often are used to divide the sumps for cleaning. One is to locate a concrete wall between the two areas and to provide a means of closing off one side and making the water flow to the other side. With this method, the wall should be anchored properly on the top and bottom with reinforcing steel into the rock. This provides a safety factor in preventing the wall from collapsing into the side being cleaned. The other system is to develop two physically separate sumps. Both sumps must be provided with pumps, a method of stopping the water inflow and diverting to the other sump, and a means of access for cleaning.

Some mines locate the pump station below the sump chambers, using horizontal centrifugal pumps instead of deep-well turbine pumps. These mines avoid the installation of a vacuum priming system because the overflow feeds the main pump (Schwandt, 1970). Although this system provides easier access to the sumps and makes cleaning easier, the risk of losing the pump station by flooding is increased.

If the power for the pumps originates from a source not maintained by the mine (i.e., public power with or without "tie lines"), the mine may be without power for extended periods of time. Therefore, every protective measure should be provided. Using deep-well pumps on one of the upper or middle levels helps protect the facility for the maximum length of time. If the pumps are in a room or chamber with the power lines coming in and the water pipes going out, it may be impossible to access the pumps for replacement without a small overhead crane on a rail above the pumps. If deep-well pumps are used, sufficient headroom is needed to remove the multistage impellers.

Other Service Developments. A variety of other rooms or drifts also must be developed. A lunch area or room is nearly essential around an underground shop. Office space should be provided for the privacy of the underground supervisor and the security of records and equipment.

Ramps driven strictly for access are important. Normally, these can be driven steeper than haulage ramps because the equipment using them would not be loaded and time to negotiate them would not be as critical. However, the ramp must not be so steep as to prevent unloaded equipment from having good traction. Equipment manufacturers can advise the mine planner on this point. A small area at the bottom of the shaft may have to be provided for the equipment to clean the shaft bottom. In some cases, equipment such as an overshot loader will have to remain on the ramp, so there must be sufficient maneuvering room between the shaft bottom and a bucket that lifts the loaded spill rock out of the bottom of the shaft. Ideally, but seldom, there is a drift intersecting the bottom of the shaft and a ramp leading back up to the bottom level of the mine. In such cases, a steel deflection door can be pushed into place for shaft cleaning in the access drift while hoisting is in progress. Shaft cleanout also may be accomplished by remote-controlled loading equipment on the shaft wall or on the steel below the skip-loading facility but above the shaft bottom.

PLANNING THE ORGANIZATION AND REQUIRED EQUIPMENT

The amount of equipment or personnel for the needs of all mines cannot be specified in general terms. However, personnel and equipment productivity ranges can serve as guidelines for planning.

Work Force and Production Design

It is necessary to consider several factors concerning the work force to operate the mine. Many questions needing investigation will be difficult to answer, but they have profound effects on the financial success of any mining project and eventually must be faced.

1. Is the supply of labor adequate to sustain the production level dictated by other economic factors? If not, can the needed labor be brought in, and at what cost?
2. What is the past history of labor relations in the area? Are the workers accustomed to a five-day work schedule and, if so, how will they react to a staggered six- or seven-day schedule?
3. Are the local people trained in similar production operations, or must everyone be trained before production can achieve full capacity?
4. Can people with maintenance skills be attracted to the property, or will the maintenance crew have to be built up through an apprenticeship program?

Apprenticeship programs are very slow processes. Accordingly, some state laws restrict the number of people who can be trained each year in such programs. That one item could cause a mine designed and equipped for a very large daily production to fall far short of its goals.

It is important that the "people problems" be investigated at the same time that the property is being evaluated and designed. That provides adequate time for specialized training, minimizes unexpected costs, and prevents basing economic projections on policies that, if implemented, could destroy employee morale or community confidence. The productivity and profitability difference between an operation with good morale and good labor relations and an operation with poor morale and poor labor relations (with many work stoppages) can make the difference between profit or loss, expansion or curtailment, and go or stop. Of all the items involved in mine design, this one is the most neglected and can be the most disastrous.

One of the advantages of modern room-and-pillar mining systems is that every task can be mechanized to some degree, provided that it is economically sound to do so. That minimizes the operating labor force and makes manning the operation easier. The high-capacity equipment for modern room-and-pillar operations is reasonably simple to learn and to operate. Anyone having operated any heavy machinery in construction work, the military, or even on a farm has little trouble adapting to loading and hauling equipment in a room-and-pillar operation. If there is a nucleus to start from, there usually is help available from the equipment manufacturer in training maintenance people. Table 3.6 lists the production figures for various types of

TABLE 3.6 Productivity of various types of room-and-pillar mines (1971 data)

Mine Products	Type of Rock	Location	No. of Mines in Sample	Range of Production, t/d (stpd) × 1000	Range of Productivity, t (st) per worker shift	Average Productivity t (st) per worker shift
Lead/Zinc	Dolomite	New Lead Belt, Missouri	8	1.6–4.5 (1.8–5.0)	19.2–59.0 (21.2–65.0)	32.6 (35.9)
Zinc	Cherty Limestone	E. Tennessee	5	1.4–3.1 (1.5–3.4)	12.5–26.6 (13.8–29.3)	17.8 (19.6)
Copper	Shale & Sandstone	N. Michigan	1	6.2 (6.8)	—	14.6 (16.1)
Limestone	Limestone	Various	45	1.1–8.7 (1.2–9.6)	31.8–136.1 (35.0–150.0)	74.5 (82.1)
Phosphate	Phosphate	Utah	1	2.2 (2.4)	—	52.2 (57.5)
Potash	Shale & Potash	N. Mexico	6	3.6–8.3 (4.0–9.2)	22.0–75.0 (24.2–82.7)	42.8 (47.2)
Salt	Salt	Various	10	1.6–10.8 (1.8–11.9)	14.3–63.6 (15.8–70.1)	36.9 (40.7)
Sandstone	—	Pennsylvania	1	2.4 (2.7)	—	73.7 (81.2)
Trona	Salt	Wyoming	3	3.2–5.6 (3.5–6.2)	27.9–63.1 (30.8–69.6)	41.1 (45.3)
Coal	Coal	Various	All	—	—	10.8 (12.0)
Uranium	Sandstone	Various	All	—	—	5.1 (5.6)

Sources: Dravo Corp. (1974), Runes (1974), and Klemenic (1977).

room-and-pillar mines; it was compiled from several sources and updated with recent information.

Table 3.7 approaches the subject in a somewhat different manner. The table was prepared using 15 noncoal mines, with the personnel requirements divided into production, maintenance, and support categories. That shows the effects of the type and thickness of the mined material on the productivity of the mine. For example, Table 3-7 lists a case in which the material is fairly thick [2.4 m (8 ft)] and is soft enough for continuous miners. The operation requires only 39 people to produce 6530 t/day (7200 stpd). The table also lists a case in which the material is abrasive or hard and where rock-mechanics problems exist. That requires 36 times the number of people to produce approximately three times the tonnage of the first case. Table 3.8 regroups the 15 mines of Table 3.7 to emphasize the type and thickness of the material mined.

TABLE 3.7 Detailed labor classification of selected room-and-pillar mines*

Total Height Mined m (ft)	Daily Production,† t (st)	Type of Entry‖	Production			Maintenance			Support§			Total Mine#		
			Number‡		t (st) per worker shift	Number‡		t (st) per worker shift	Number‡		t (st) per worker shift	Number‡		t (st) per worker shift‖
Limestone														
11.7	1533				102.1			218.9			1451.2			66.7
(32)	(1690)	A	15	(14)	(112.6)	7	(6)	(241.4)	1	(0)	(1600.0)	23	(20)	(73.5)
14.6	4345				108.6			724.1						94.4
(40)	(4790)	A	40	(36)	(119.7)	6	(5)	(798.3)	—	—	—	48	(41)	(104.1)
20.0	3628				84.4			226.8			403.1			53.3
(55)	(4000)	D	43	(39)	(93.0)	16	(15)	(250.0)	9	(6)	(444.4)	68	(60)	(58.8)
10.2	4535				60.4			82.4			241.9			30.4
(28)	(5000)	V	60	(50)	(66.7)	44	(37)	(90.9)	15	(14)	(266.7)	119	(101)	(33.6)
Evaporite														
6.2	5079				69.6			95.9			153.9			31.9
(17)	(5600)		73	(67)	(76.7)	53	(44)	(105.7)	33	(16)	(169.7)	159	(127)	(35.2)
21.9	5805	V			111.7			252.4			1451.2			73.4
(60)	(6400)		52	(47)	(123.1)	23	(22)	(278.3)	4	(0)	(1600.0)	79	(69)	(81.0)
3.8	10,884				84.4			149.1			494.8			48.6
(10.5)	(12,000)	V	129	(121)	(93.0)	73	(66)	(164.4)	22	(0)	(545.5)	224	(187)	(53.6)
2.0	7256	V			86.3			136.9			157.7			39.6
(5.5)	(8000)		84	(63)	(95.2)	53	(45)	(150.9)	46	(28)	(173.9)	183	(136)	(43.7)
3.6	5986	V			52.1			103.2			285.1			30.8
(10.0)	(6600)		115	(100)	(57.4)	58	(54)	(113.8)	21	(9)	(314.3)	194	(163)	(34.0)
2.9	6458				96.4			170.0			461.3			54.2
(8)	(7120)	V	67	(50)	(106.3)	38	(34)	(187.4)	14	(10)	(508.6)	119	(94)	(59.8)
2.9	6530	V			408.2			466.4			725.6			167.4
(8)	(7200)	V	16	(12)	(450.0)	14	(12)	(514.3)	9	(0)	(800.0)	39	(24)	(184.6) M
Metal Ore														
2.6–5.5	20,408				31.6			76.7			40.4			14.4
(7–15)	(22,500)	D	647	(566)	(34.8)	266	(243)	(84.6)	506	(333)	(44.5)	1419	(1142)	(15.9)
4.4–36.5	3628				51.1			71.1			77.2			21.4
(12–100)	(4000)	V	71	(63)	(56.3)	51	(41)	(78.4)	47	(29)	(85.1)	169	(133)	(23.7)
3.3–21.9	4716				62.9			142.9			90.7			29.4
(9–60)	(5200)	V	75	(67)	(69.3)	33	(28)	(157.6)	52	(27)	(100.0)	160	(122)	(32.5)
5.8	4807				96.1			282.8			240.4			55.2
(16)	(5300)	V	50	(46)	(106.0)	17	(15)	(311.8)	20	(8)	(265.0)	87	(69)	(60.9)

Source: Modified from Dravo Corp. (1974).

*Personnel data from records examined during period of mine visits.

†Based on nominal average tonnage produced per day during same period of mine visits.

‡First number indicates both hourly and salaried people; number in parentheses shows hourly people only.

§Support includes: service labor, staff, safety, engineering, hoist personnel, skip and cage personnel, top personnel, and office personnel chargeable to underground mine operation. (Where neccessary, office staff has been prorated in direct ratio to the percent of personnel working in the mine and plant.)

#Number of people charged to underground operations.

‖A—adit; D—decline; V—vertical shaft; M—used continuous miners.

TABLE 3.8 Detailed labor classification grouped by product and thickness of material mines (from Table 3.7)

					Manpower Distribution										
					Production		Maintenance		Support		Totals				
Product	No. of Mines	Ground Height*	Mining System†	Type of Entry	Hourly, %	Salary, %	Hourly, %	Salary, %	Hourly, %	Salary, %	Hourly, %	Salary, %	Average Daily Production, mt (st)	Personnel Total Numbers Hourly	Salary	Total
Evaporite	1	Thin	CM–SS	V	30.8	10.2	30.8	5.1	—	23.1	61.5	38.5	6,530 (7,200)	24	15	39
Evaporite	4	Thin	D&B–SS	V	46.5	8.4	27.6	3.2	6.5	7.8	80.6	19.4	7,646 (8,430)	145	35	180
Evaporite	2	Thick	D&B–SS	V	47.9	4.6	27.7	4.2	6.7	8.9	82.3	17.7	5,442 (6,000)	98	21	119
Limestone	4	Thick	D&B–MS	A&D	54.3	7.4	24.6	3.9	7.8	2.0	86.7	13.3	3,283 (3,620)	56	8	64
Metal Ore	1	Thin	D&B–SS	D	39.9	5.7	17.1	1.6	23.5	12.2	80.5	19.5	20,408 (22,500)	1142	227	1419
Metal Ore	3	Thick	D&B–MS	V	42.3	4.8	20.2	4.1	15.4	13.2	77.9	22.1	4,384 (4,833)	108	31	139
All (Less High/ Low Value)	13	—	—	—	46.8	6.8	25.3	3.7	9.0	8.4	81.0	18.9	5,212 (5,746)	102	23	125
Average<—II	15	—	—	—	43.4	6.3	21.6	2.8	15.6	10.3	80.6 (6,960)	19.4	6,313	166	40	206

*Thin—under 3.4 m (11 ft); thick—over 3.4 m (11 ft).

†CM—continuous mining; D&B—drill and blast; SS—single-slice or one-pass mining; MS—multiple-slice or several passes of mining; V—vertical shaft; A—adit; D—decline.

Equipping Room-and-Pillar Mines

Detailed information concerning mine equipment is contained in other sections of this handbook. In this section, comments about mine equipment apply particularly to room-and-pillar mining and, to a lesser extent, to some other types of mining. Remaining competitive in underground mining in the United States requires constantly increasing productivity. The best way to increase productivity is to utilize equipment designed for that purpose, that is, to mechanize wherever a safe and reliable piece of equipment is available to magnify the physical and mental efforts of an individual. Following that philosophy for room-and-pillar mining, the following paragraphs provide recommendations that influence the selection of equipment.

Field-Tested Equipment. The selected equipment should be produced by manufacturers that field test their equipment for long periods of time before marketing it to the mining industry. Too many manufacturers build a prototype machine and install it in a customer's mine on the contingency that they will stand behind it and make it work properly. Eventually, after both user and manufacturer redesigns, rebuilds, reinforces, and retrofits, a workable machine finally is obtained. However, the cost in lost production is imposed on the mine operator, not on the manufacturer. The manufacturer then can proceed to sell the "field-tested" retrofitted model to the entire industry, including competitive mines. In the case of an LHD unit introduced several years ago, there were 53 design changes between the prototype and the final production model. In another case, a prototype drill jumbo was modified so extensively over a two-year period that nearly every auxiliary component was retrofitted in the user's operation; even after two years, the jumbo still was not performing according to the specifications at the time of purchase. Even though manufacturers do need the help and cooperation of the mining industry to develop equipment, the manufacturer must pursue longer periods of testing in the industrial environment, rather than selling the units to the industry and then pursing continuous research on the prototype models.

Equipment Versatility. The equipment selected for room-and-pillar mining should be as versatile as possible. For example, in one mining operation (Bullock, 1973), the same high-performance rotary percussion drilling machines were used for drilling the bluff or brow headings and were mounted on standard drill jumbos for drilling holes for burn-cut rounds and slabbing rounds in the breast headings. Because the drills on these jumbos penetrate extremely fast, they also were used to drill the holes for roof bolts and, in some cases, to drill holes for reinforcing pillars. The same front-end loader was used to load trucks in one stope and to perform as an LHD unit in another stope. By switching working platforms, the same forklift tractors served as explosive-charging vehicles and as utility service units for hanging air, water, and power lines. They also served as standard forklifts for handling mine supplies. That philosophy results in the following advantages:

1. There is less equipment to purchase and maintain.
2. Less training is required for operators and maintenance personnel. At the same time, all personnel have a better chance of becoming more efficient at their jobs.

3. Having fewer types of machinery means having less inventory to obtain and maintain.

Some of the possible disadvantages of this philosophy include:

1. A more efficient machine may be available to do the job being done by the versatile machine. Therefore, if a great amount of that type of work is to be done, it may be advisable to use the specialized machine.
2. The mine may become too dependent upon a single source of equipment supply.

As an example of the first disadvantage, the mine previously cited was drilling a great number of high bluffs, some brows, and breakthrough pillars. As a result, the mine eventually switched over to air-track drills because those machines were more efficient in the particular applications.

Equipment Acceptance. The equipment selected should have a very broad acceptance and be in common use throughout both the mining and construction industries. Since mines impose a headroom restriction not encountered on the surface, that is not always possible. However, where headroom is not a problem, selecting a standard piece of equipment means that the components will have endured the rigorous use of those industries. Furthermore, parts for such equipment normally are off-the-shelf items in the distributor's warehouses across the country.

Application Flexibility. The selected equipment should be flexible in application. That is, the equipment should be able to accelerate and move rapidly, have good balance and control at high speeds, be very maneuverable, and have plenty of reserve power for severe applications. Both trucks and loaders should have ample power to climb all grades in the mine and to accelerate quickly to top speed on long straight hauls. Table 3.9 lists the minimum mobile equipment necessary to produce approximately 4535 t/d (5000 stpd) from a trackless room-and-pillar metal-ore mine.

Table 3.10 lists the minimum mobile equipment necessary to equip either a conventional or continuous coal-mining section. Where roof bolting is the "bottleneck" in the operation, as it usually is, the current trend is to use either two-boom roof bolters or two complete single-boom roof bolters. Where the seam height permits very high productivity from the loading machine or continuous miner, more than two shuttle cars may be required.

PLANNING PERSONNEL AND SUPPLY HANDLING

During the early stages of mine planning, consideration should be given to the methods and problems involved in transporting personnel and materials into the mine and to the working face. Because every operating room of a room-and-pillar mine has an access roadway that can be negotiated by mobile equipment, personnel and material transportation is considerably easier in these mines than in other types of mines that are on steeply dipping veins.

TABLE 3.9 Minimum amount of mobile equipment needed for 4536 t/d (5000 stpd) trackless metal mine (assuming 80% availability)

No. of Units	Type of Unit Needed to Mine Multiple-Slice Room-and-Pillar System
7*	Two-boom drill jumbo; each machine capable of drilling 1.5 m/min (5 fpm) (used for production and development).
6†	4.6 to 6.1-m^3 (6 to 8-cu yd) front-end loaders that can double as LHD units.
7	Utility units, truck or tractor mounted with extendable and interchangeable work platform. Used for charging the drill rounds and utility work.
2	Standard size (Cat-112E) road graders.
1	D-6 size bulldozer.
1	Maintenance truck.
2	Hydraulic impact roof scalers on multicrane with 12 m (40 ft) reach.
1	Aerial giraffe with 21 m (70 ft) reach.
3	Small foremans' vehicles.
1	Tractor-mounted diamond drill and percussion drill for prospecting.

Source: Modified from Bullock (1974).

*Some Jumbos operate only one shift.

†Assumes 83% availability and 90% efficiency of operator with 6 to 7 min cycle time per load. As cycle time increased, two 36-t (40-st) trucks were added to this mine.

TABLE 3.10 Minimum amount of mobile equipment needed per coal-mine section

No. of Units	Type of Unit Needed
Conventional Section	
1	Cutting machine
1	Coal drill
1	Loading machine
2	Shuttle cars
1	Roof bolter
1	Small rock duster
1	Utility vehicle
1	Feeder breaker
1	Personnel vehicle
Continuous Section	
1	Continuous miner
2	Shuttle cars
1	Roof bolter
1	Small rock duster
1	Utility vehicle
1	Feeder breaker

TABLE 3.11 Personnel and supply handling times and associated delays in two room-and-pillar mines

	Evaporite Mine	Lead-Zinc Mine
Daily projection, t (st)	8,084 (8,911)	6,077 (6,700)
Underground work force	190	225
Average production time, hr per shift	5.82	6.37
Tons produced per production hour on three shifts per day	463 (510)	318 (351)
Travel and on/off equipment time, hr per worker-shift	.81	.53
Average travel time converted to potential production, t/d (stpd)	1,118 (1,233)	505 (557)
Other personnel delays, hr per worker-shift	.88	.60
Personnel delays converted to potential production, t (st)	1,218 (1,343)	579 (631)
Supply related worker-hours per month	530	395
Supply related time as percentage of underground worker-hours	1.6	1.0

Source: Modified from Theodore Barry and Associates (1975).

Two case studies illustrate the magnitude of the problems involved (Theodore Barry and Associates, 1975). One study was in an evaporite mine utilizing basic coal-mining methods. The other was a New Lead Belt lead/zinc mine. Table 3.11 lists data associated with the average time required to transport personnel to the working area, the average delay time lost due to other reasons, and the average production time remaining in a 7.5-hr shift. The tonnage equivalents represented by the time delays also are listed, showing that reducing some of the delays would increase the daily production. Table 3.11 also lists the time required to deliver supplies each month and the relation to the total number of underground worker-hours.

The following conclusions concerning excessive personnel and material transportation time came from that report (Theodore Barry and Associates, 1975):

1. Typically, equipment transport speeds are not up to the level of performance currently achievable through existing technology.
2. Nonmechanized means of travel such as walking and climbing are significantly slower than mechanized transport. Over long distances, such travel results in lost mining time. Walking and climbing have been identified as major problem areas with respect to lost-time accidents in personnel movement. Underground environments typically make walking or climbing slow, tedious, and hazardous.
3. General mine conditions such as poor track and rough or crooked roadways adversely affect the operational efficiency of personnel transport systems.
4. In some cases, equipment design impedes efficient loading and unloading of personnel, thereby increasing the transportation time.

The same report lists the following recommendations as ways to improve personnel movement:

1. To optimize personnel transport, each mine should identify specific groups of personnel according to job functions, listing their ultimate destinations, intermediate transport requirements, etc.
2. The system component dependence should be handled by personnel-scheduling techniques. The horizontal system should accept the personnel unit on a predetermined schedule.
3. An engineering study of all personnel-movement elements should be conducted to establish present and future requirements for a balanced transport system.
4. Crews should be transported to a point as close as possible to the actual work, and they should be picked up on a predetermined schedule at the same point for outbound transport.
5. Supervisory staff should use mechanized transport wherever possible.
6. Daily production reporting procedures should be utilized, rather than end-of-shift recording procedures.

An incentive or supervision system must be incorporated to assure that the time gained at the face is utilized productively. It cannot be assumed that putting a miner at his working place a given number of minutes early automatically will increase his daily output—the miner must be shown some personal benefit in actually utilizing those extra minutes.

Concerning the improvement of supply hauling techniques, the report stated (Theodore Barry and Associates, 1975):

> *Additional unitization of supplies and the increased use of mechanical material-handling equipment is needed at all mines [studied]. . . . Adequate space must be made available for loading/unloading transfer operations, as well as in-mine storage.*

Although coal mines were not included in the study, personal observations indicated that the supply and personnel transportation problems are more difficult in coal mines than in noncoal mines. The practice in most coal mines in the United States is to develop all entries only to the heights of the seams. That limits the movement of anything in the mine other than that transported on conveyor belts. The European practice of developing full-height entries offers many advantages, especially when the mine has been developed extensively.

The problem of low-entry transport becomes quite critical when the mine has a poor bottom on which to operate. A combination of water inflows, undulating scams, and clay bottoms might be a sufficient reason to develop at least one entry to full height, with subsequent installation of a mechanized personnel and material transport system. As an example, the Greenwich Colliery of Allentown, PA, is a coal mine in which a 107-mm (42-in.) seam is mined and a slice of the bottom rock is taken to gain a full-height supply entry (Trevorrow, 1975). There are satisfactory methods of handling the extra rock, although it is not accomplished as easily or as cheaply as handling coal (Murry, 1972).

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CHAPTER 4

Sampling, Dilution, and Recovery

C. Alan Tapp

INTRODUCTION

Mine sampling determines the practicality of any mining operation. Improper sampling can result in an incorrect appraisal of present production and future potential. Therefore, the mine department in charge of ore-reserve calculations and mine sampling should be overseen by competent and experienced professionals with technical backgrounds qualifying them to produce accurate results.

Sampling is a process by which portions of an orebody are collected and analyzed to estimate the average mineral content of the entire orebody. It is incorrect to assume that a large number of samples eliminates any errors in the sampling method. To obtain unbiased samples, proper sample location with respect to rock type and mineralization is just as important. The sampling procedure must yield correct results for the type of mineral deposit, and careful consideration should be given to whether the sampling technique has been developed to an extent sufficient to eliminate as much human error and bias as possible. Only after the ore has been mined and milled is the sampling accuracy known.

Sampling also provides information about the bulk composition of the ore for mineralogical and metallurgical tests that determine the economic ore-waste boundaries and the geologic trends for exploration. Actual mining plans can be developed from this information to maximize profits. Accurate sampling is critical, and thus must be approached in a scientific manner.

SAMPLING TECHNIQUES

Sampling practices and techniques are as varied as the mines in which they are used. The method(s) chosen must be tailored to suit the company and mining needs. For instance, tabular uranium deposits, vein gold deposits,

and porphyry copper deposits pose special problems in conducting unbiased sampling. The mine geologist or engineer in charge must develop a sampling method, test it in a sample area, and then critically evaluate the results. If the results from the test area are accurate within the economic limits established by the company, they then may be adopted for general use in the mine.

Four routine sampling methods are suitable for specific sampling objectives in the daily mine routine: (1) channel sampling, (2) chip sampling, (3) grab sampling, and (4) bulk sampling. The final sampling results depend upon how the four methods are combined to accurately determine the grade of the orebody. When used in conjunction with each other during different stages of mine development, the channel, chip, grab, and bulk sampling methods provide an in-house check or a comparison by which mining methods and sampling procedures can be evaluated. However, the most valid check is based upon the daily mill production.

Channel Sampling

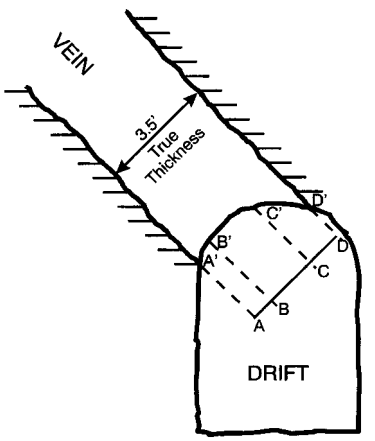
Channel samples consist of cuttings collected from a groove cut into the rock about 102 mm (4 in.) wide and 19 mm (0.75 in.) deep. Various tools ranging from a 1.8-kg (4-lb) hammer and moil to a pneumatic chisel can be used to cut the sample. Accessibility and rock hardness determine the applicable sampling tools.

Before attempting to take a sample, the rock surface must be cleaned thoroughly; the method of cleaning depends upon the amount of mine dust accumulated on the surface or the degree of alteration of the rock surface. Typical cleaning methods employ a wire brush, water, or chipping a fresh surface.

Next, the sample outline is marked on the prepared surface, taking care to choose appropriate sample locations. After determining the proper sample outline and length, the sample can be chiseled out, catching the rock fragments on a canvas tarpaulin on the floor, in a powder box, in a canvas bag to avoid contamination, or by some other suitable means to capture all of the sample. Vertical veins present a special problem because the drift's back usually is arched, not square. For example, Fig. 4.1 shows that if sample intervals are measured from A' to D', the length of the sample interval is greater than the true vein width, resulting in an incorrect calculation of the ore reserves. The correct method would collect three samples at A'-B', B'-C', C'-D'using either actual measurements in the mine represented by A-B, B-C, and C-D or trigonometric calculations to determine the true ore thickness. Then, each sample can be weighted by grade and true thickness for the vein. Channel lengths usually are a maximum of 1.5 m (5 ft), and it is good practice to divide longer samples into smaller intervals according to structures, changes in rock types, or differences in rock hardnesses. The influence of those features on the mineralization then can be determined.

Chip Sampling

Chip sampling is a variation of channel sampling used when the rock is too hard to channel sample economically or when little variation in the mineral content indicates that this sampling method will yield results similar to those of channel



CORRECT METHOD

Sample Interval	Thickness, ft.	Assay, oz. Au/ton	Product, Th × Assay
A–B	0.8'	0.09	0.07
B–C	1.5'	0.50	0.75
C–D	1.2'	0.13	0.16
	3.5'		0.98

$$\text{Average Grade} = \frac{\Sigma(n \cdot \bar{x})}{\Sigma n}$$

$$\text{Average Grade} = \frac{0.98}{3.5'}$$

Average Grade = 0.28 oz./ton over a true thickness of 3.5 feet.

INCORRECT METHOD

Sample Interval	Thickness, ft.	Assay, oz. Au/ton	Product, Th × Assay
A'–B'	1.0'	0.09	0.09
B'–C'	1.5'	0.50	0.75
C'–D'	1.7'	0.13	0.22
	4.2'		1.06

$$\text{Average Grade} = \frac{\Sigma(n \cdot \bar{x})}{\Sigma n}$$

$$\text{Average Grade} = \frac{1.06}{4.2'}$$

Average Grade = 0.25 oz./ton over an incorrect thickness of 4.2 feet.

FIGURE 4.1 Calculation of a true sample thickness for mine sampling

sampling. Rather than cutting a channel in the rock, small chips are flaked off at regular intervals over the entire face or area being sampled. Care must be taken to assure that the sample is representative of variations in the rock hardness and type. This method is fast and useful in preliminary evaluations, but it should not be used for quantitative ore-reserve calculations.

Grab Sampling

Grab sampling is a fast method for double checking either channel or chip-sampling procedures, and, in some instances, mine production can be estimated from carefully taken grab samples. Grab sampling takes equal amounts of material at selected intervals over a mine dump, a muck pile, or from an ore car to estimate its mineral content. Generally, this method is not considered reliable. Many independent variables can affect this type of sampling process. Thus, if the ore occurs in the softer fraction and a proportional amount of the resulting fines are not sampled, the results are erroneous. The sample may also consist more of one rock color, rather than having the correct proportions

of each. If each sampler is consciously aware of every variable that can affect grab sampling, the reliability of the method increases.

Bulk Sampling

Bulk sampling is used to evaluate ore zones with irregularly distributed mineralization or to determine composite ore characteristics. The amount collected depends upon the nature of the mineralization and the size of the area being tested. A bulk sample can range from tens of kilograms (pounds) to several tons. For example, if a mine dump is to be evaluated economically, it is necessary to take several bulk samples at predetermined locations, or several small samples can be combined into one bulk sample. Then, weighted averages of the assays, based upon the portion of the mine dump they represent, are combined to obtain a representative value for the dump. As another example, if the accuracy of channel samples from a vertical stope is questionable, the area can be bulk sampled before mining. In this case, several holes drilled into the back are shot, with the sample collected on a canvas tarpaulin. A sample like this could amount to several hundred kilograms (pounds). However, care must be taken to avoid "salting" or otherwise influencing the results through poor sampling techniques.

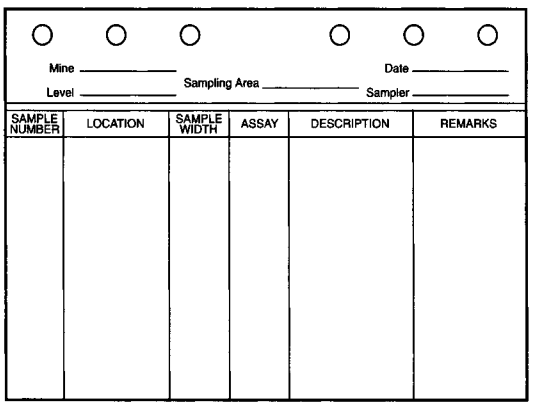
MINE SAMPLING

In the day-to-day operations of a mine, the importance of accurate sampling sometimes can become obscured by the routine nature of the job. However, this daily information is necessary for the mining department to develop accurate mine-production plans. Such daily sampling provides information for four phases of mining: guiding the development headings, providing data for ore-reserve calculations, indicating production grades for every stope, and allowing mineralogical and metallurgical evaluation of the orebody.

Development Headings

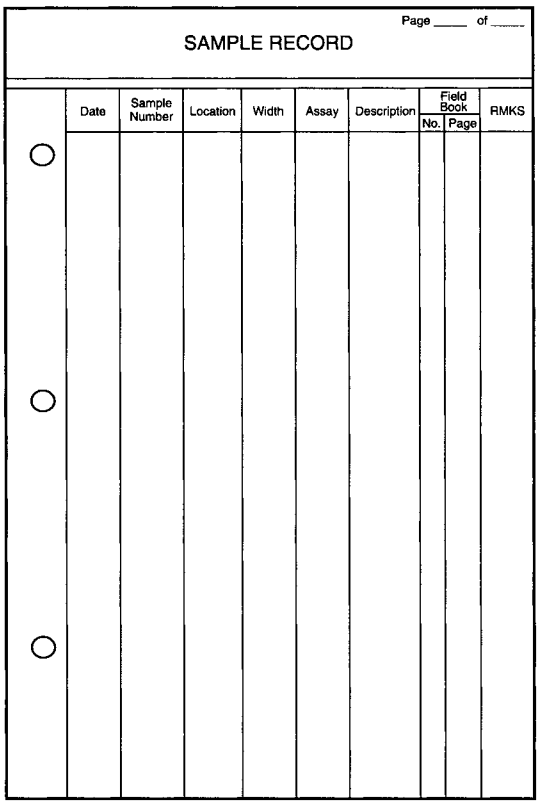
All development headings are sampled on a daily basis to determine the ore content in the mining face. This determines the assay boundaries and helps in the projection of mill head grades. Customarily, channel or chip samples are taken at the face, provided the face has been cleaned from the previous round; the ore characteristics will determine the best type of sampling method. If the face is not clean, grab samples may be taken from the muck pile; although grab samples may not be as accurate as channel samples, they can be used with caution.

When the development headings are sampled, all pertinent information is recorded on worksheets immediately. The location of the sample should be referenced against some permanent point. The thickness of the sample intervals, rock type, sample type, sampling party, date, and sample number all should be recorded. As a general rule, daily worksheets are kept in looseleaf notebooks so completed sheets can be left in the office. If the sample book is lost, only one day's records are lost, rather than the records for an entire month or year. An example of such a daily sample sheet is shown in Fig. 4.2.


Mine _____ Sampling Area _____ Date _____
Level _____ Sampler _____

SAMPLE NUMBER	LOCATION	SAMPLE WIDTH	ASSAY	DESCRIPTION	REMARKS

FIGURE 4.2 Example of a daily sample sheet


Page ____ of ____

SAMPLE RECORD

Date	Sample Number	Location	Width	Assay	Description	Field Book No.	Field Book Page	RMKS

FIGURE 4.3 Form to record daily sampling for permanent mine records

Once in the office, the worksheet data are then transferred to the permanent mine records. Figure 4.3 shows a sample sheet for these records. The permanent sample records are no more than a summation of the daily sampling, prepared in concise and easily recognizable form. In most cases, that means the creation of a tabular form for the files. Accurate records are essential for maintaining the mine's head grade; they are the key mine expansion and development of potential exploration targets.

Assay maps and cross sections of the appropriate scale for the area also are updated with the new sample information. They are the most usable form for evaluating daily sampling information. The maps and cross sections present a visual picture of the limits and orientation of the ore. As shown in Figs. 4.4 and 4.5, it is customary to have two sets of mine maps showing both small-scale and large-scale features. Small-scale mine maps at 25 mm = 61 m (1 in. = 200 ft) are useful in showing mine-development plans, gross geological features, property boundaries, and general ore trends; a generalized overview of the mine property is shown on these small-scale maps. The large-scale maps at 25 mm = 15 m (1 in. = 50 ft) are used to plot the daily sampling results and mining plans. These large-scale maps permit more accurate

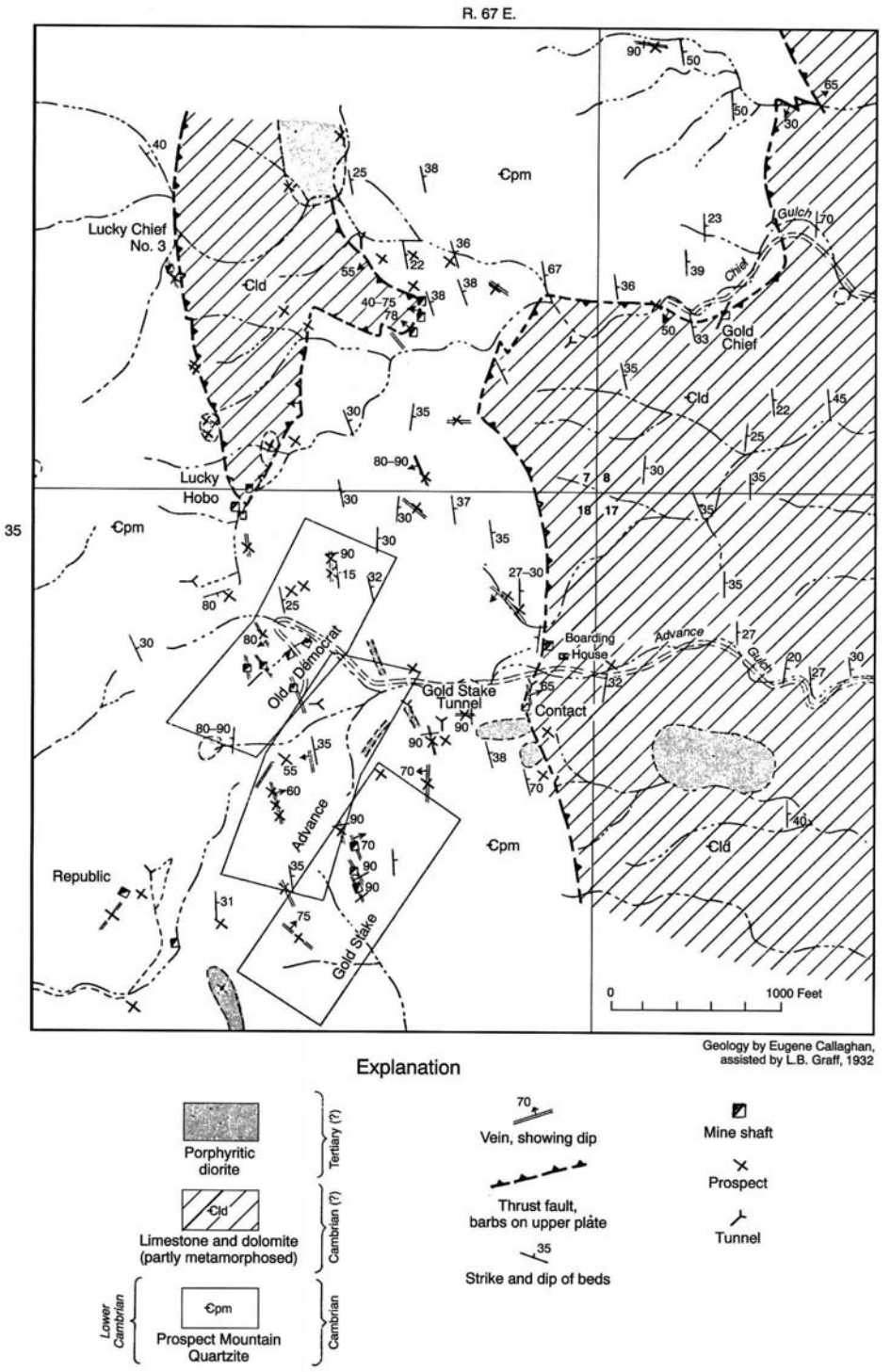


FIGURE 4.4 Small-scale geologic map of the Chief district, metric equivalent: ft × 0.3048 = m

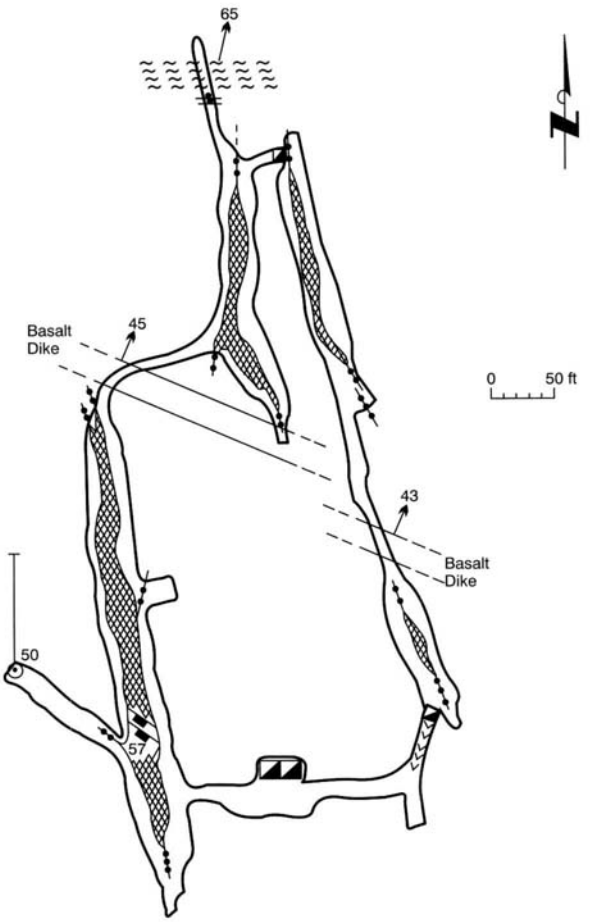


FIGURE 4.5 Example of a large-scale mine map, metric equivalent: ft × 0.3048 = m

determination of daily mine face advancement. They also provide excellent working maps for the mine geologist. Features not large enough to be plotted on a small-scale map, but important to the interpretation of the deposit, are plotted on the large-scale maps.

Cross sections also are made at two scales for similar reasons. Figure 4.6 shows both a small-scale longitudinal section and a plan-view map of a hypothetical mine.

Ore-Reserve Calculations

Ore-reserve data are parts of the permanent mine record and should be updated on a periodic basis to include newly developed reserves and to reflect any depletion of reserves from stoping areas. For most mines, once the ore has been blocked out, the perimeter of each pillar is sampled on equally spaced centers. Large-scale assay maps are produced from these sample data. When the mine maps are in the same plane as the orebody, pillar areas can be planimetered to estimate tonnages accurately. Weighted averages for all of

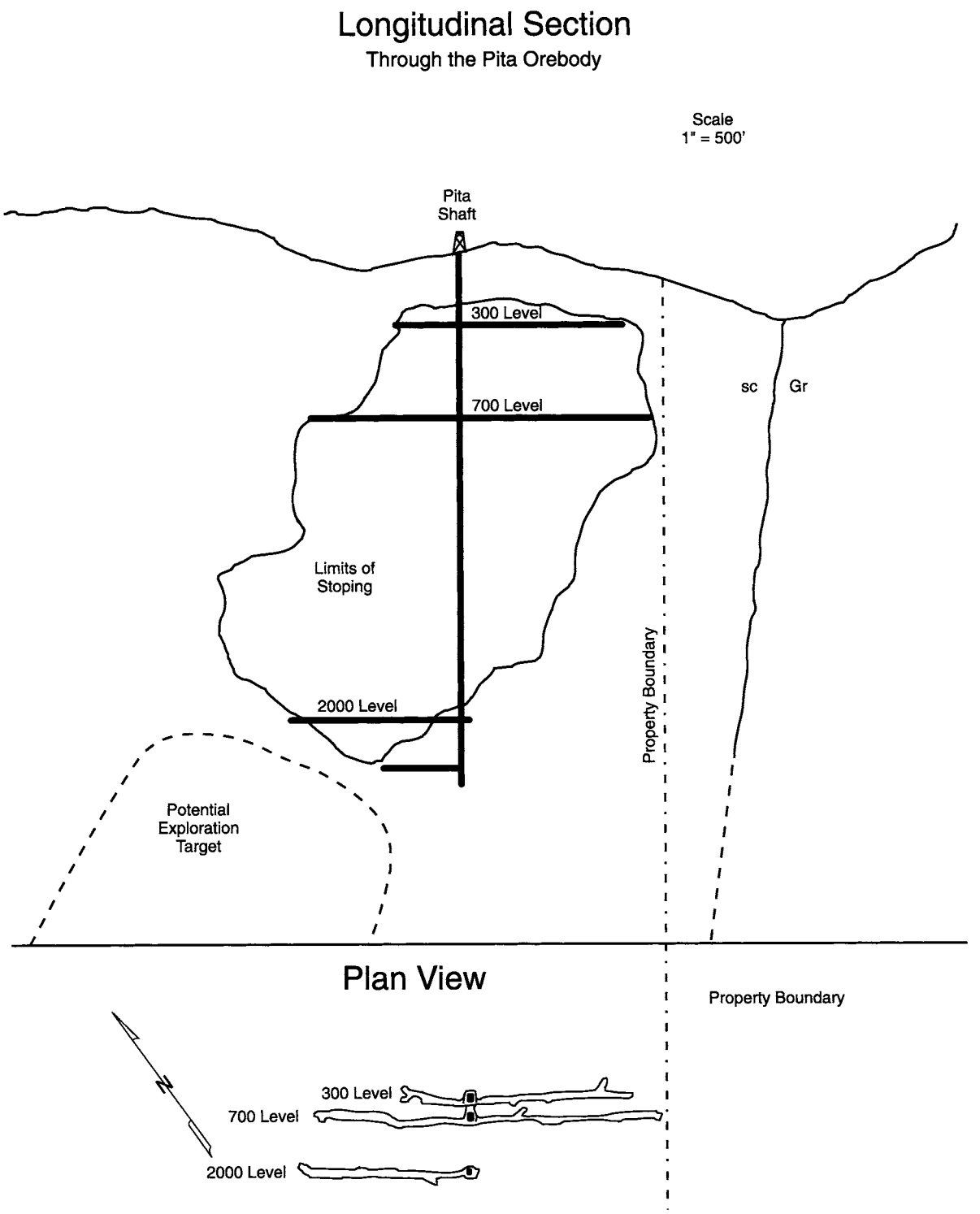
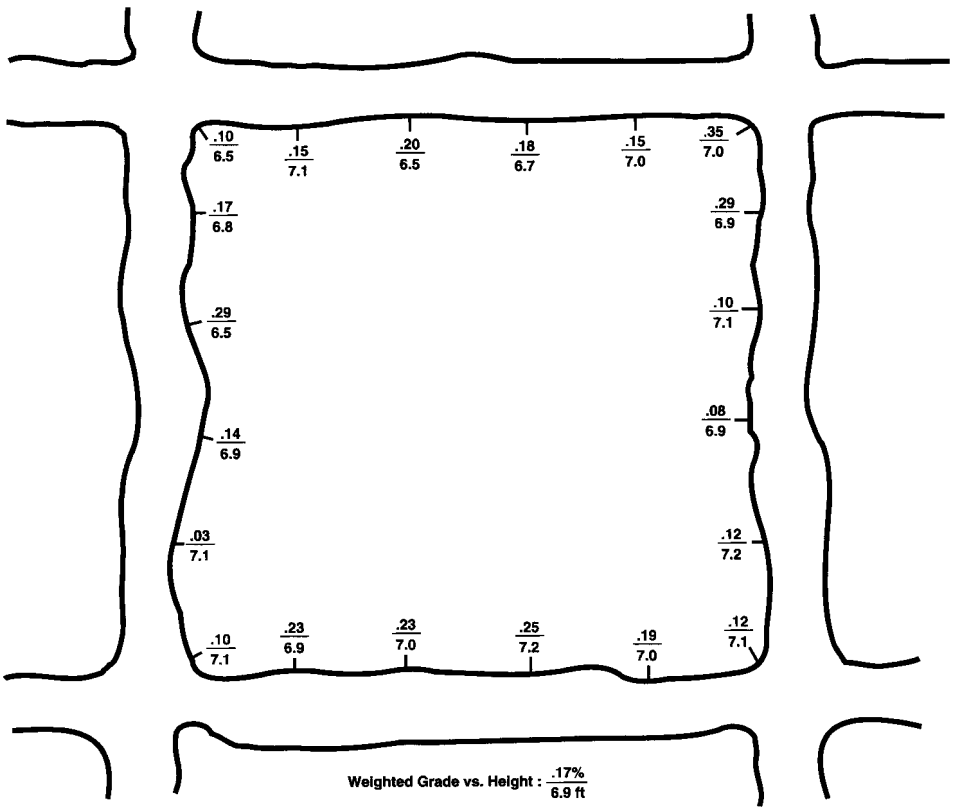


FIGURE 4.6 **A small-scale longitudinal section with accompanying plan view, metric equivalent: ft × 0.3048 = m**



$$\frac{\text{Weighted Grade}}{\text{Weighted Height for Pillar}} : \frac{.17\%\ U_3O_8}{6.9\ \text{ft}}$$

$$\text{Area} : 2600\ \text{ft}^2$$

$$\text{Tons} : \frac{(\text{Area} \times \text{Thickness})}{\text{Tonnage Factor}} = \frac{(2600)(6.9)}{12.5}$$

$$= 1435 \text{ tons in place}$$

$$\text{Pounds} : (\text{Tons})(\text{Grade})(20) = (1435)(.17)(20)$$

$$= 4879 \text{ pounds in place}$$

Tons and pounds in place can then be added to ore reserve figures with the appropriate extraction and dilution rates, to yield mining ore reserves.

FIGURE 4.7 Ore reserve calculations from a large-scale mine map

the sample points are calculated to produce ore grades for the pillar, as shown in Fig. 4.7. Because ore minerals usually are not distributed homogeneously throughout an orebody, it may be necessary to spot check the sample locations, a task easily accomplished by bulk sampling the points where channel samples are in doubt.

Production Grades

Day-to-day sampling procedures provide sample control for daily production. Mine-head grade, as opposed to mill-head grade, is watched closely to assure that the mill receives the proper grade of ore. Mine-head grade is monitored by taking grab samples from every car, train, or shipment of ore. The sample assays for a working day then are computed, with their respective tonnages, for a weighted average grade that should be an approximate mine-head grade.

Internal Checks

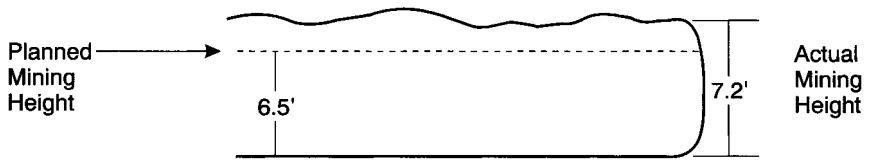
One duty of the sampling department is to run internal checks on the in-house sampling accuracy. The simplest method is to compare daily production sampling with a calculated composite grade of all other mine samples (weighted appropriately for their representative proportions), finally comparing these against the mill's production. The sampling program should provide enough varied forms of sample data, including ore reserves, so that different combinations of daily sampling data can be used to determine the accuracy of the program. There never can be enough samples; the information that they provide is essential to meet the production quotas.

DILUTION AND RECOVERY

In a mining context, dilution is the extracted tonnage of material below the economic cutoff grade for the mine. The extracted material may contain the mineral being mined but in insufficient quantities to be recovered economically. Dilution must be kept within the economic limits of the mining operation. Under existing economic conditions, maximum mining efficiency can be defined as 0% dilution at 100% extraction of the mineral being mined.

There are two stages in the development of a mine when mine dilution occurs—(1) preproduction dilution estimates based upon the mining method, and (2) dilution due to overbreak during the mining process. Once a mineralized trend has been delimited sufficiently to be designated as an orebody, a mining method is chosen to produce the maximum amount of ore with minimum dilution. Mining methods such as block caving, sublevel stoping, or room-and-pillar mining have characteristic dilution and recovery rates that arc dependent on factors such as the mine depth, rock competency, ore type, and methods of internal mine support. Self-supported openings are more selective and yield lower dilution rates than block-caving methods. A high degree of control over what rock is broken as ore in self-supported openings develops dilution rates from 5% to 15% and recovery rates ranging from 60% to 85%. Caving systems rely almost entirely on the inherent ability of the rock to fracture. Features such as dikes, joint patterns, and heterogeneity control the caving and can add unwanted dilution. Excessive dilution during caving may result in an early termination of mining, in turn reducing the overall mine recovery. Dilution in excess of 15% usually is expected in block caving, with extraction rates of 85% to 100%. Experience based upon known mines is the best indicator for predicting dilution rates for different caving methods.

During the mining process, internal rock characteristics and the efficiency of the chosen mining method have a direct effect on dilution. Highly fractured or



$$\text{Mining Dilution, \%} = \frac{\text{Actual} - \text{Planned}}{\text{Planned}} \times 100$$

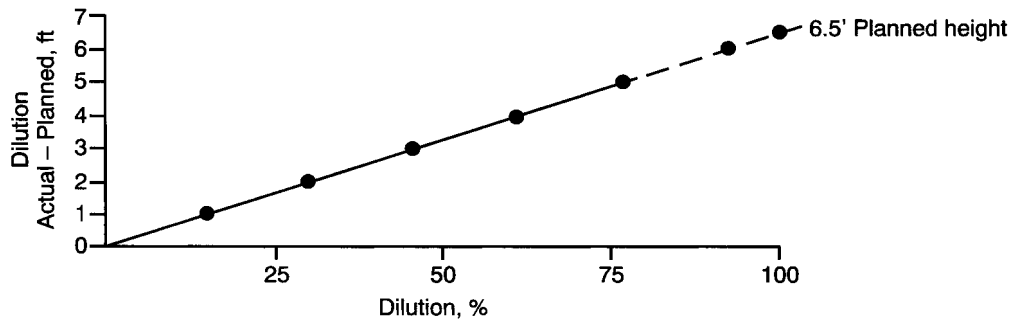


FIGURE 4.8 Calculation of mining dilution

incompetent back and ribs do not support open ground, adding waste to the mined ore. Unexpected ground problems, marginal-grade mineralized zones around the orebody, and crushed pillars all add unwanted dilution. Mining costs also increase when such areas must be supported by excessive timbering or other support techniques. Controlling and reducing dilution by increasing mining costs is profitable as long as the increased mining cost is equal to or less than the economic effects of reduced dilution (Ingler, 1975).

Dilution during mining is calculated by actual measurement and is a quantitative means for reporting the effectiveness of the mining operation. Figure 4.8 illustrates a standard method for calculating mining dilution.

SAMPLING PROCEDURES AT THE HOMESTAKE MINE*

The Homestake gold mine is located in the Black Hills, in Lawrence County, at Lead, S.D.

General Geology

During the Precambrian era, an accumulation of mudstone was deposited and later highly refolded and transformed by regional metamorphism. Extreme compressional effects resulted in a plastic-type flow of the rocks, producing a

*This section was contributed by Steven Mitchell.

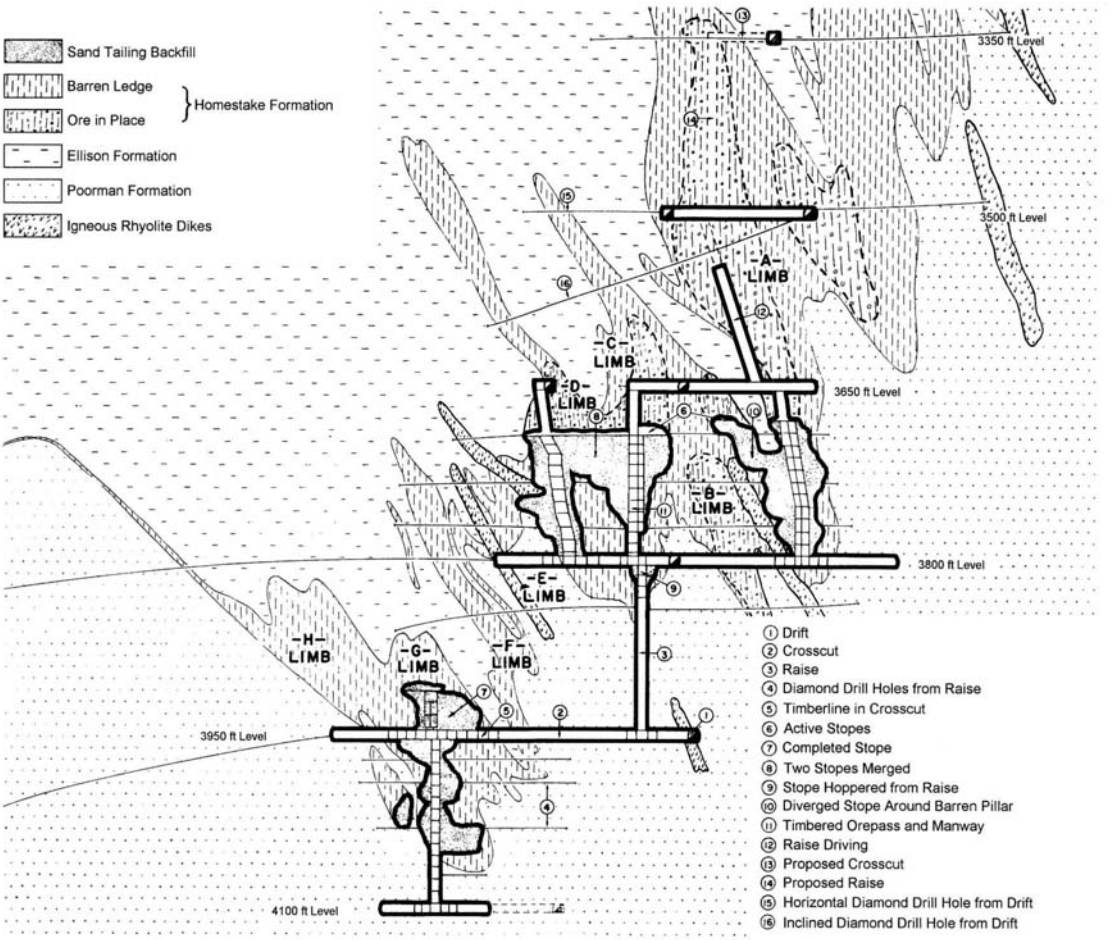


FIGURE 4.9 Cross section of a nine-ledge structure

series of anticlines and synclines, locally known as "ledges." Figure 4.9 illustrates a cross section of the ledge structure. After much erosion, sediments of the Paleozoic and Mesozoic eras accumulated, forming the following stratigraphic sequence.

Tertiary, Quaternary, and Recent Gravel Deposits

Unconformity

Tertiary Intrusive Rocks

Cambrian Deadwood Formation

[90 to 150 m (300 to 500 ft)]

Unconformity

Precambrian Intrusive Rocks

Precambrian Sedimentary Rocks

Grizzly Formation [910 m (3000 ft)]

Flag Rock Formation [1520 m (5000 ft)]

Unconformity

Northwestern Formation [1220 m (4000 ft)]

Ellison Formation [910 to 1520 m (3000 to 5000 ft)]

Homestake Formation [60 to 90 m (200 to 300 ft)]

Poorman Formation [610+ m (2000+ ft)]

Generally, all of the gold ore found within the mine is located exclusively within the Homestake Formation at a depth of 60 to 90 m (200 to 300 ft). However, not all of the Homestake Formation is mineralized, and the values that are found often are distributed very erratically. Sideroplesite within the Homestake Formation generally is fine grained, except in the ore zones where the grain size increases with the degree of mineralization. Banding due to varying amounts of graphite is common in certain areas of the orebody. In addition to sideroplesite and graphite, the Homestake Formation contains quartz, biotite, chlorite, pyrite, chalcopyrite, pyrrhotite, arsenopyrite, and ankerite. The gold content averages approximately 5 g/t (0.2 oz per st).

Mine Sampling

The various methods of sampling at the Homestake mine can be summarized as: primary sampling with diamond-drill holes and dry drills; secondary sampling with test holes, drill samples, and pick samples; and check sampling with grab samples and car samples.

The applicability of these sampling methods is governed by factors such as the adaptability to the area being sampled, the reliability of the samples that can be recovered, the associated operating costs, and the duration of any one method.

Primary Sampling. Diamond drilling and dry-drill sampling probably are the most reliable sampling methods currently used at the Homestake mine. However, the cost and expertise required to obtain representative samples makes these methods impractical for implementation at the production stage. Consequently, diamond drilling and dry drilling are development-stage sampling methods used exclusively by the geologists in estimating tonnages and ore grades for each stope block throughout the mine.

Diamond Drilling. Diamond drills are pneumatically powered and utilize water as the drilling medium. An "AQ" wireline overshot apparatus is used on all drills to remove the inner barrel and rock core, without physically pulling the rods. Unfortunately, such core drilling is a relatively slow and expensive process. Although distances of up to 58 m (190 ft) have been reported, progress of 12 m (40 ft) per 8.5-hr shift per drill crew is considered to be average. The total operating costs for AQ wireline drilling are approximated at $18.04/m [$5.50 per ft (1977 U.S. $)].

Dry Drilling Usually the LeRoi Dry Ductor drill (LLV) has been used as a primary sampling device. The recovery of dry-drill cuttings is about 90%. Because the cuttings are drawn back through the drill rods, a special 25-mm

(1.0-in.) drill rod is required in conjunction with a 41-mm (1.625-in.) Vacumatic® bit. Dry-drilling sample data are used by the geologists to supplement the diamond-drill data in estimating the ore reserves.

Secondary Sampling. Test holes and drill samples both involve collecting the wet drill cuttings or sludge from percussion jackleg drills. These two sampling methods are used extensively within the active stoping blocks that have been outlined by previous diamond-drill and dry-drill sampling procedures.

Test Holes. To obtain a representative sample at the mining face, test holes are drilled across the "grain" of the ore-bearing Homestake Formation. To begin the process, the collar of the test hole is drilled approximately 254 mm (10 in.) into the wall. Then, a drain hole is drilled upward at the flattest possible angle to intersect the bottom of the test hole. A piece of hose or pipe, about 0.6 m (2 ft) long and having a beveled end, then is inserted tightly into the drain hole, and a canvas sack measuring 356 × 254 mm (14 × 10 in.) is attached by a rope onto the end of the drainpipe to collect the wet cuttings.

Using a minimum amount of drilling water, the test hole then is drilled to a depth of 1.5 m (5 ft). The sample sack is removed from the pipe, and the cuttings within the sack are allowed to settle before the water is decanted off. Finally, the test hole is extended to a depth of 4.6 m (15 ft), with a separate sample collected every 1.5 m (5 ft). The hole is flushed thoroughly between each sample, and care must be taken to assure that the correct sample tag is placed within the corresponding sample sack.

Back samples are taken from a type of test hole used extensively within each open cut-and-fill (OCF) stope in the mine. These samples are collected exactly as described in the preceding paragraphs, taking them from sections located on 6.1-m (20-ft) spacings along the entire back of the OCF stope; Fig. 4.10 illustrates the spacings. The angle and length of these holes usually are 0.52 rad (30°) and 4.6 m (15 ft), respectively. Ideally, each test hole samples the back to a height of 2.3 m (7.5 ft). However, an average blast in an OCF stope breaks a 3.0-m (10-ft) horizontal slice of rock out of the back so about 33% of the rock broken as ore is beyond the limits of the sampled volume. In most stopes, the sample density, as shown in Fig. 4.11, is approximately 66 t (73 st) per ore sample.

At the ends of each row or section of sample holes, additional 4.6-m (15-ft) test holes often are drilled into the footwall and hanging wall. These holes are drilled approximately 50% of the way between the floor and the end back-sample test hole in that row. In addition to the back samples and the wall samples, each stope usually contains at least 10 to 15 other test holes. Those holes generally are used to check back-sampled areas that are of questionable ore grade or to test for possible ore extensions within the areas of the stope walls.

Drill Samples Drill samples are similar to those taken from test holes, except that the cuttings from an entire hole are collected, rather than from separate and distinct intervals within the hole. If a drift round is to be sampled by a drill hole, the cuttings normally are allowed to run onto the floor of

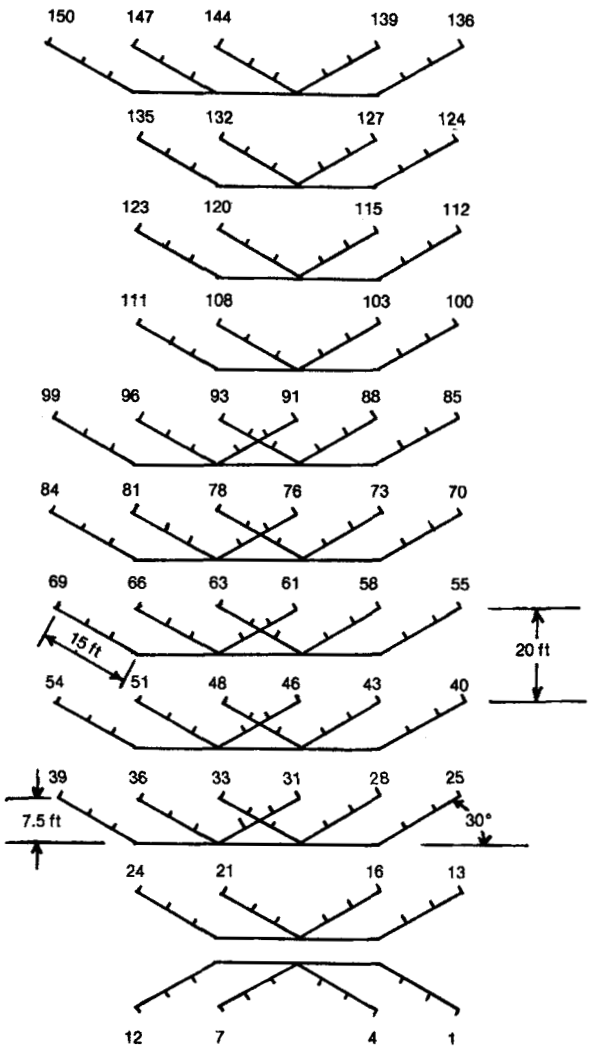


FIGURE 4.10 Typical map showing test-hole pattern in an open cut-and-fill (OCF) stope, metric equivalent: ft × 0.3048 = m

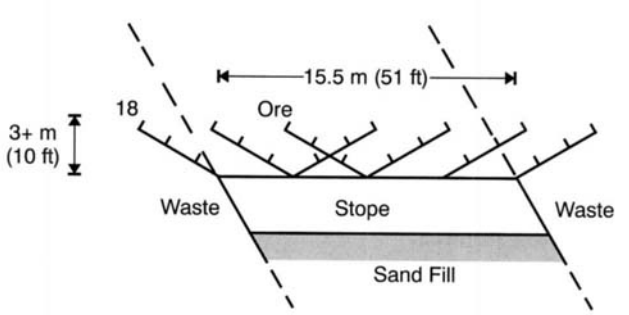


FIGURE 4.11 Typical cross section of an open cut-and-fill (OCF) stope showing sample density (51 × 10 × 20 ft = 10,200 cu ft; 10,200 cu ft ÷ 10 cu ft/st = 1020 st; 1020 st/14 samples = 73 st/sample)

the drift until the round is completed. The cuttings then are loosened with a pick and placed into a sample sack, along with an identifying tag. Often, an extension hole is drilled to sample the rock ahead of the round that presently is being drilled. In that case, the cuttings are collected in the same manner as for a test hole.

Pick Samples. If properly taken, the pick sample is reliable, and it is used in conjunction with test holes and drill samples. Pick samples are used at the faces of drift rounds, slab rounds, and raise rounds. A typical area to be sampled usually is about 0.3 m (1.0 ft) wide and up to 2.4 m (8.0 ft) long. Small pieces of rock are chipped from within this area, taking care to remove equal amounts of rock across the grain and across the entire face of the drift or raise round to assure a reliable and representative sample. A pick sample does not relate to the size or extent of the body of rock behind the face.

Check Sampling. Check sampling serves only to indicate rock grade that has been mined previously and is now lying in the broken state. A sufficient portion of the rock is gathered at random to constitute a representative sample. Such a sample is a valuable tool for checking the reliability of primary and secondary sampling and to reveal sources of undesirable dilution.

Grab Samples. Grab samples test piles of rock that have been mined but are of questionable gold content. Normally, a grab sample is collected by having two or three people randomly select rocks from several locations on the pile. A sample sack then constitutes one grab sample from that stope. If large rocks are predominant throughout the broken pile, pieces of these rocks are broken off and placed into the sample sacks.

Car Samples. Car samples usually are taken by the chute puller or train operator. A car sample should consist of at least 75% of a sack of rock, removed in equal amounts from each car of a 10-car train. Like grab samples, car samples are used to substantiate the reliability and accuracy of the stope sampling methods. If the car samples are taken properly and consistently from all producing areas, they allow daily ascertainment of the average run-of-mine grade.

Ore Reserve Estimates

Tonnage and Grade. In estimating tonnage and grade, the first step is to implement a diamond-drilling program on the mine levels situated within a particular stope block. Customarily, the mine levels are 46 m (150 ft) apart vertically, and diamond-drill stations on individual levels are located on stope reference lines 29 to 31 m (95 to 102 ft) apart. As each horizontal hole is drilled through the orebody, the core is collected, logged, and fire assayed; the resulting data are recorded on geologic level-plan maps. To supplement these data, dry-drill samples also are obtained between successive diamond-drilling stations.

In a similar fashion, data derived from angle-hole drilling and raise drilling are plotted on geologic cross sections that conform to vertical projections of the stope reference lines. Usually, the plan maps and cross sections are the only means by which the geologist can estimate the tonnage and grade within a particular stope block.

Once the sampling data have been plotted on maps and cross sections, a stoping outline can be established in accordance with the overall mine cutoff grade (determined by prevailing economic conditions). The techniques used for estimating tonnages are variations of the polygon method of calculating ore reserves. For example, an ideal situation would be one in which the stope outline conforms to a cross-sectional triangle. With such a configuration, the tonnage can be estimated using a "wedge" volume calculation.

To estimate the grade for this same stope block, a variable dilution factor would be incorporated into the calculation to compensate for hanging wall dilution, as well as for dilution from the hydraulically backfilled sand floor used in OCF stoping. The averages of the assays from the level plans and cross sections then would be weighted in relation to the size of the planimetered area in that plan or cross section. Finally, as with any estimate, the tonnage and grade would be upgraded constantly as test-hole samples became available from actual stope production.

Production Forecasting. Ultimately, the ore estimates from each stope block on every level are added together to arrive at a total figure for the ore reserves. Every month thereafter the ore estimates for each potentially productive stope are reviewed and a forecast is made to predict the upcoming month's ore production and grade for the entire mine. Similar forecasts are made for mine production six months and one year in advance. Then, check samples from every ore and waste transfer system—from the mine to the mill—can be compared to the calculations, allowing correlation studies for the reliability of the primary and secondary sampling methods and for the accuracy of the ore-estimating technique.

Conclusions

The sampling procedures used at the Homestake mine have proven to be a satisfactory method of estimating ore reserves. However, the system has many pitfalls, requiring constant and rigid supervision of all phases to avoid faulty data that would cause serious errors in the ore estimates and grade control.

SAMPLING PRACTICES AT THE HENDERSON MINE*

The Henderson molybdenite (MoS_2) deposit is located 1100 m (3600 ft) under the 3750-m (12,311-ft) summit of Red Mountain in Colorado. Red Mountain is located in the Daley mining district at the extreme western edge of the Colorado mineral belt and is 13 km (8 miles) west of the town of Empire and about 69 km (43 miles) west of Denver.

General Geology

The geology at Red Mountain consists of a classic subvolcanic rhyolite porphyry sequence emplaced during the mid-Tertiary period in multiphase Precambrian Silver Plume granite of the Front Range. Figure 4.12 illustrates a generalized geologic map of the area.

*This section contributed by Bruce R. Stanley.

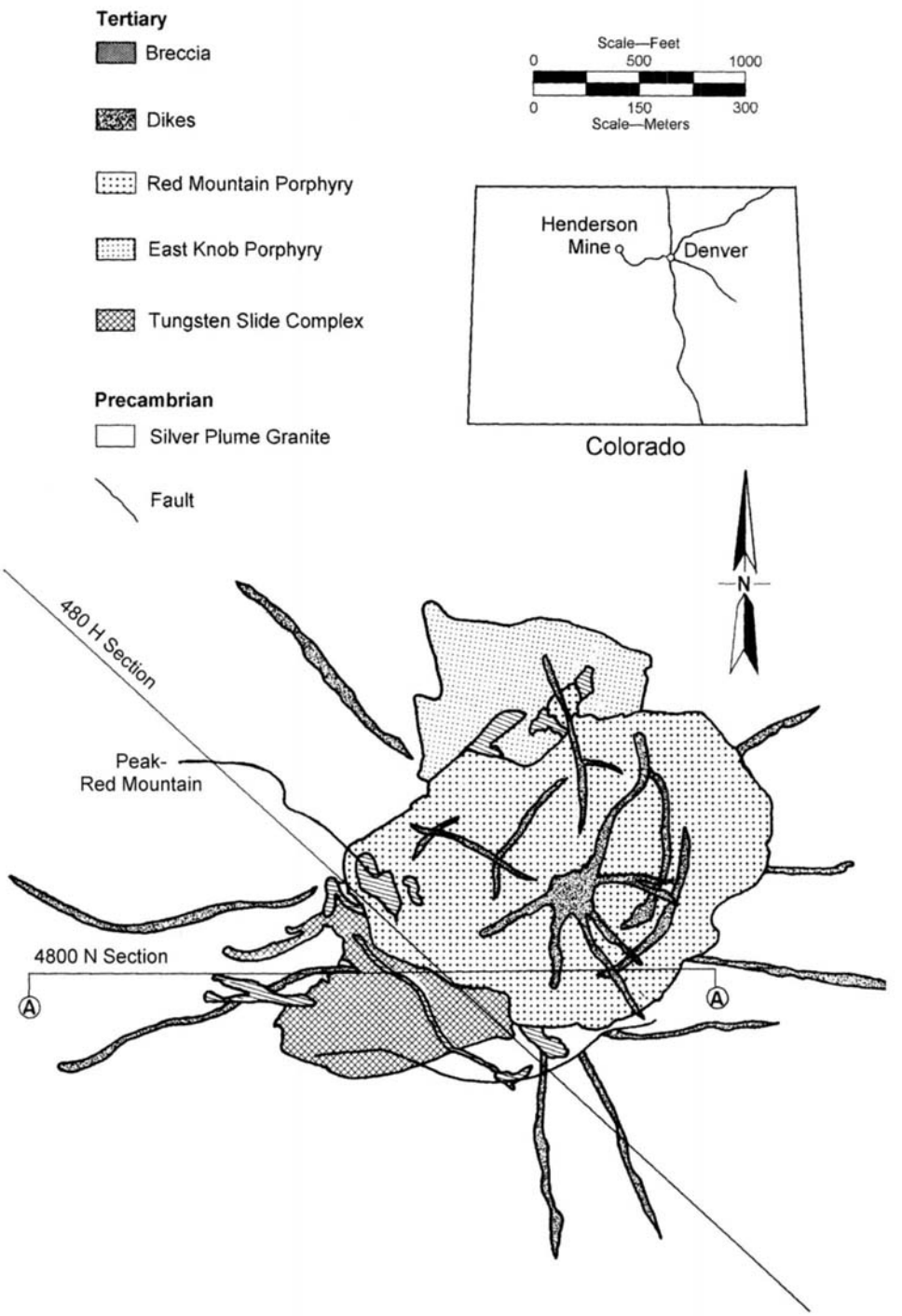


FIGURE 4.12 Generalized geologic map of Red Mountain showing Section 480H of Fig. 4.13 (modified from Epis and Weimer, 1976)

Emplacement of the porphyries probably was localized by the intersection of major Precambrian faults reactivated in the Tertiary period. Igneous pulses, some never quite reaching the surface, cooled and crystallized into a pipe-shaped complex that consists of at least two breccia dikes, several rhyolitic dike systems, and six plugs. The earliest plug and the lower related stock complex probably were responsible for the molybdenite mineralization. Stockwork quartz-molybdenite mineralization occurs both in the extensively fractured granite and porphyry breccia near the top of Red Mountain (Urad orebody) and in the more coarsely crystalline Urad-Primos porphyry Henderson-granite stock complex deep beneath Red Mountain (Henderson orebody).

The Henderson orebody, 820 m (2700 ft) directly below the base of the Urad deposits, has the shape of an inverted teacup, elongated in a northeast direction. The orebody is asymmetric and bilobate in section, and the total extent of the known mineralization measures 305 m (1000 ft) thick and 610 × 915 m (2000 × 3000 ft) in plan.

Increasing in intensity from the outer fringes toward the center of the orebody, the main alteration types enveloping the deposit include the porphyllitic, argillic, quartz-sericite-pyrite, and potassic zones. Ten alteration zones have been identified at the Henderson mine, as shown in Fig. 4.13.

The orebody is sampled by underground diamond drill holes, using an east-west fan pattern originating on the 8035 level. These fans are spaced roughly 61 m (200 ft) apart in the north-south direction and are designed to intercept the 0.2–0.3 zone interface at a spacing of about 61 m (200 ft). The cores obtained are logged and assayed every 3 m (10 ft), and the data are plotted on cross sections. The geology department then contours the grade zones on the cross sections according to simple averaging techniques and personal expertise.

Mine Sampling

Although every mine would like to have an absolute value for the grade being produced in each mine location, general sampling theories are not applicable in all situations because mineralogical structures vary greatly. The two best situations that could be encountered are a wide vein deposit and a homogeneous porphyry orebody. For each, fairly simple sampling methods can be utilized to give reasonably good results.

Sampling in a porphyry is not a precise science. Chip and channel sampling can be difficult in many locations, leaving grab sampling the only logical alternative. Such grab sampling consists of taking pieces of rock at random from a desired location. Because a single rock cannot be expected to represent the location's grade, a 4.5- to 6.8-kg (10- to 15-lb) composite is compiled as a sample from randomly chosen pieces taken from all parts of the rock pile.

When the rock is shot or caved, it loses the structure that might have been definable when it was in place. A mathematical-trend grade then must be established, rather than a structural-grade trend. Many people believe taking larger samples automatically provides better values. Because it is impractical

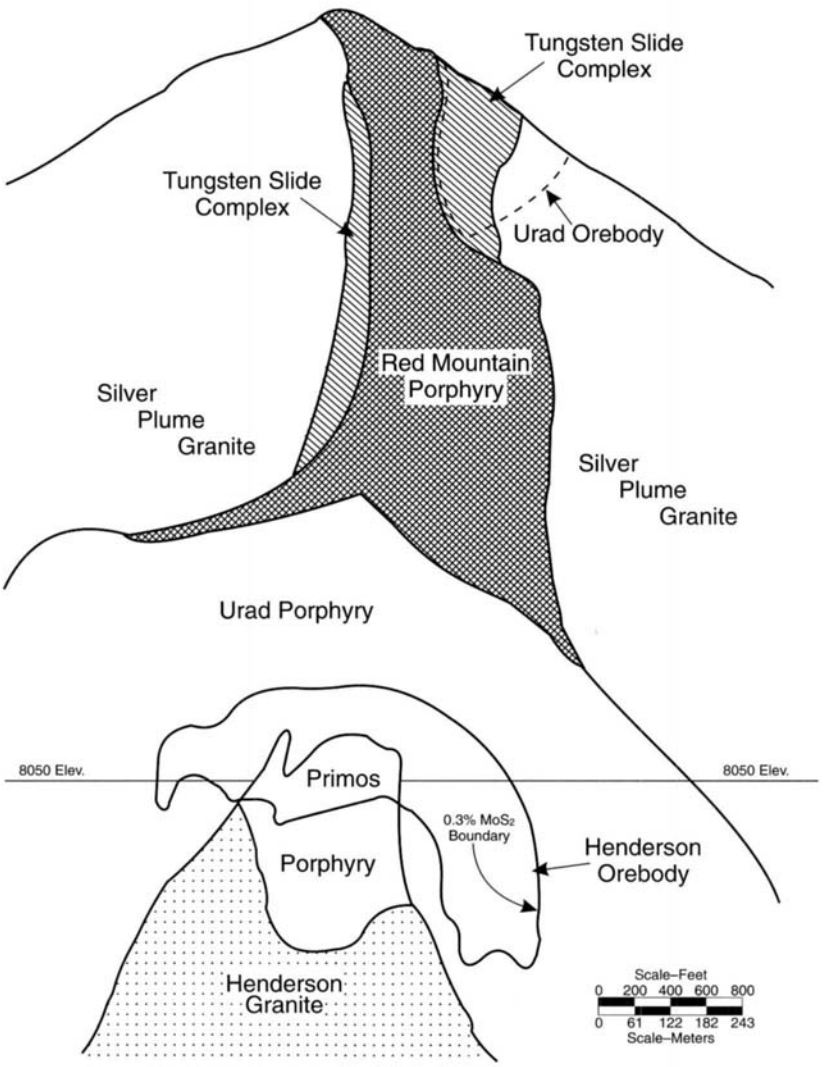


FIGURE 4.13 Generalized geologic cross section of Red Mountain, Section 480H (modified from Epis and Weimer, 1976)

to take 1.8-t (2.0-st) samples, a consistent sampling schedule is best when used with a consistent sample size of 4.5 to 6.8 kg (10 to 15 lb); small individual samples produce erratic results in that type of orebody. Typically, a 1.0-g (0.03-oz) grain of 6.4-mm (0.25-in.) molybdenite contributes 0.19% to the assay of a 0.5-kg (1.0-lb) sample of 6.4-mm (1.0-in.) material.

In development and production sampling, a series of actual assays is analyzed for a "best-fit" trend, using an interactive forecasting system of computer programs. A number of methods were tested on the data, and an exponential smoothing method of forecasting was found to provide the best indication of grade trends. Using the last eight assays for any one working area, it is possible to predict the grade trend by weighting the most recent assay heaviest and

each recessive assay less heavily according to the exponential formula. At this property, the method has been found to be accurate within 5% of the true values obtained.

These theories are applied to general daily sampling practices; some further specifics are necessary in the categories of daily development sampling, daily production sampling, and ore-reserve sampling.

Daily Development Sampling. Many levels throughout the mine contribute to the development tonnage, but only three major levels are considered for development grades (various other levels are lumped into one of the three categories). Development samples are taken in an area currently being worked, so the trend grade obtained represents an entire area, rather than any one development heading. Although seemingly an inaccurate approach, the development tonnage is such a minor proportion compared to the production tonnage that inconsistencies make little or no difference to the overall results.

Daily Production Sampling. Production tonnage is considered to be only the tonnage extracted from the cave (a massive caving system is used at the Henderson mine). Therefore, the production samples are taken only at drawpoints, assuring accurate records of the grade at each location. Newly caved areas or areas still well within the ore column (with respect to grade) are sampled at the drawpoint at a rate of one 6.8-kg (15-lb) sample per 360 t (400 st) drawn. New grade trends are calculated by computer after each new sample assay is received, with the least recent assay eliminated from the historical list. As the drawpoints begin to exhibit decreasing grades, the frequency of sampling is increased, with a drawpoint being sampled each day as it approaches the cutoff grade. The sampling is accomplished by the operators of the load-haul-dump (LHD) vehicles—a procedure that may serve to be inconsistent but also may compensate for biases introduced by each individual's sampling habits.

Ore-Reserve Sampling. Sampling for ore reserves is a completely separate system from the daily development and production samplings. Diamond drilling is used to obtain samples for determining the ore zones and geological characteristics of the deposit. Currently, all such drilling is done from underground, and the cores are split for assaying and logging purposes. Drillings are conducted in fan patterns laid out to intercept a theoretical 0.2% to 0.1% interface at approximately 61-m (200-ft) intervals. The fans are lined up from east to west, with the rows of fans spaced on centers about 61 m (200 ft) apart from north to south. The drill cores are sampled in 3-m (10-ft) lengths, and the assays are entered into a computer file for easy retrieval of the associated geologic and assay information.

Conclusions

The subjects of ore reserves, grade calculations, and tonnage calculations are direct applications of geostatistics beyond the scope of this section. The calculations for percentage recovery and percentage dilution cannot be explained adequately at this time. The figures used at the Henderson mine have been

taken directly from the Climax and Urad mines and from the experience derived at those mines.

SAMPLING PROCEDURES AT RIO ALGOM'S LISBON MINE

The Lisbon mine, in the Big Indian district, is located at the northern end of the Lisbon Valley, San Juan County, Utah, and is approximately 56 km (35 miles) southeast of the town of Moab. The Big Indian mining district forms a belt approximately 25 km (15 miles) long and 0.8 km (0.5 mile) wide on the southwestern limb of a breached salt anticline.

General Geology

The Lisbon mine is on the downthrown side of the Lisbon fault at a depth of 790 m (2600 ft) along a northwesterly mineralized trend. Mineralization occurs in the basal 9 m (30 ft) of the Moss Back member of the Chinle Formation (Triassic age). A generalized stratigraphic section is shown in Fig. 4.14.

Generally, ore thicknesses in the district average 1.8 m (6 ft) and usually are close to the Triassic unconformity. The deposits are tabular, with the ore being concordant to the bedding. The Moss Back is fluvial-lacustrine fine-grained sandstone, conglomeritic sandstone, and interbedded mudstones. Most units are carbonaceous to varying degrees. Sandstones are the important host rock but, where mudstone lenses are interbedded with mineralized sandstones, they usually are mineralized.

The principal ore mineral is uraninite, occurring interstitially, partially replacing sand grains and mudstone clasts and, to varying degrees, also replacing carbonaceous material. A district average places the grade of ore at 0.39% uranium oxide (U_3O_8) (Wood, 1967).

Mine Sampling

Various accepted sampling procedures have been employed at the Lisbon mine. As a result, the standard sampling practice historically used in the district with slight modifications is being used. Workable results have been produced from a program consisting of development sampling, production sampling, ore reserve sampling, and check sampling.

Development Sampling. Due to the mine's depth and nature of mineralization, close-spaced surface drilling to outline the orebody was impractical. A rather wide-spaced program outlined the major mineralized ore trend. Mining economics and the tabular form of the deposit also limit the underground exploration drilling of the ore reserves.

A workable alternative to close-space surface drilling is to use existing surface drill holes as a guide for mining, while keeping close control of individual mining faces on a daily basis to delimit the assay wall boundaries. Development sampling provides data enabling the mining department to balance the development ore with the stoping ore for optimum mill feed. Three times a day, every working face in the mine is scanned radiometrically. Ore and waste

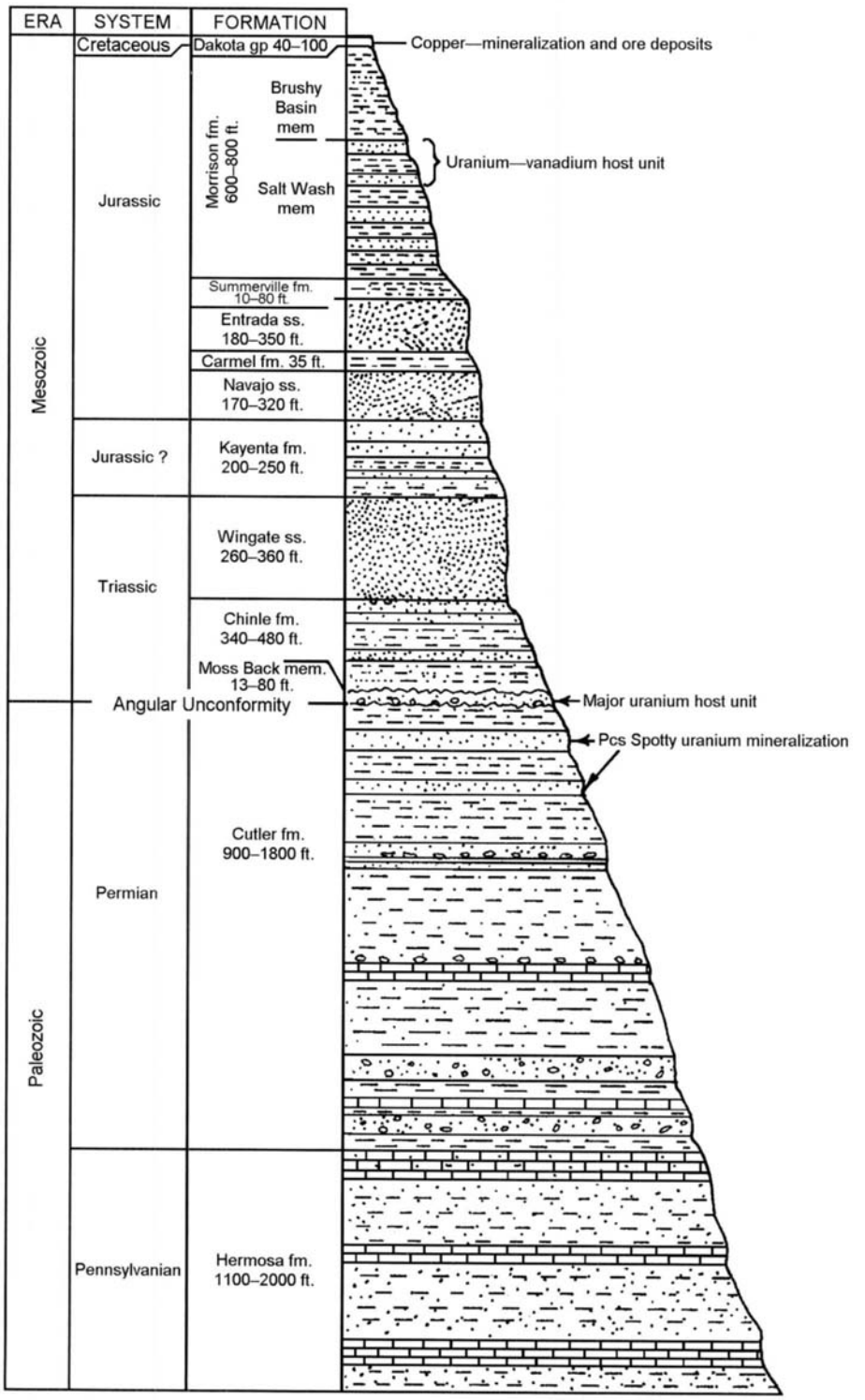


FIGURE 4.14 Generalized stratigraphic section in the Lisbon Valley area (Wood, 1967)

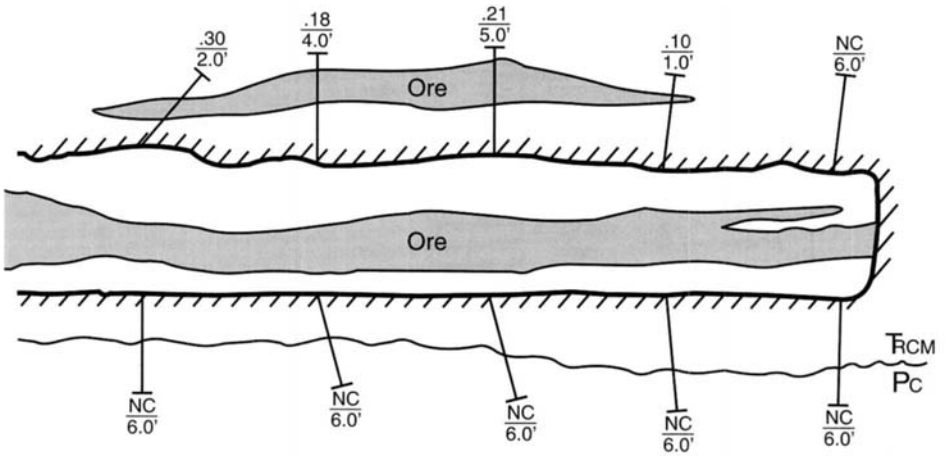


FIGURE 4.15 Example of test holing to locate small irregular ore pods above or below the main ore zone

boundaries are marked directly on the working face. Samplers then can control mining heights for excessive overbreak or, where the ore thickness exceeds the face height, they can be adjusted accordingly.

When a face has not been cleaned from the previous round, the muck pile is probed for its average grade. Close sampling of this type enables better control over the mining dilution and provides grade control at the mine face by separating ore from waste before haulage.

The voluminous samples act as an averaging tool for erratic ore or any nugget effects. Because uranium mineralization is rather spotty, placing a high degree of confidence in a few samples is misleading. Daily averages of 500 to 1000 samples, along with check samples, theoretically provide a means of good data averaging for the erratic nature of the mineralization. Internal sampling errors, problems with the sampling technique, or equipment failures also become easier to detect.

As a matter of practice, chip samples at the face and grab samples of muck piles are taken on a regular basis. Because radiometric sampling is of questionable accuracy, other sampling methods are used as internal checks. Any malfunctioning or miscalibrated equipment then can be corrected. Furthermore, rock samples can be assayed for any disequilibrium inherent in uranium deposits.

In addition to scanning faces and probing muck piles, test holes are drilled at regular intervals in the development headings. The test holes generally are drilled on 6.1-m (20-ft) centers, at an angle of ±1.57 rad (±90°) and between depths of 1.8 and 6.1 m (6 and 20 ft). As many as three separate ore bands have been mined; on a regular basis, test holing defined this irregular mineralization above or below the main ore zone, as shown in Fig. 4.15.

Production Sampling. Production sampling simply consists of car sampling. Because a reliable daily production grade is needed for the various stopes, each ore car is probed radiometrically several times before being skipped to

the surface. These car probes then are averaged by the day and computed to arrive at a daily estimate of grade and tonnage.

Although this technique is only as accurate as the individual doing the sampling, the results over an extended period usually correlate well with the ore reserve grades and mill heads. Periodic checks are performed to determine the accuracy of the probing, taking grab samples in equal proportion from each car in a train. Assays from the laboratory then are compared to radiometric probing results to detect any irregularities.

Ore-Reserve Sampling. Once the pillars have been blocked out for mining, their grades and tonnages must be calculated for the mine's ore reserves. The method is similar to development sampling, with each pillar radiometrically scanned on 3-m (10-ft) centers around its perimeter. A weighted average of height versus grade then is computed for each sample point. In turn, each weighted sample is weighted against all sampled points for the pillar. The result is the weighted height and grade for the pillar (refer to Fig. 4.7). From these values, it is easy to arrive at the extractable tons and kilograms (pounds) per pillar. Periodically, the mining grades from the pillars do not compare with the ore reserves exactly; the usual causes are ore variations within the pillars and, to a lesser extent, uncontrollable dilution.

Check sampling of the ore reserves is accomplished best by comparison with the production rates. Once the stoping operations begin, if excess dilution is held to a minimum as outlined by pillar scanning, production grades for the pillar life should correlate with the ore reserves. Excess dilution is easy to detect from the increased tonnage and resultant decrease in grade.

Check Sampling. Although check samples have been described in the preceding paragraphs, their importance warrants a further comment. All phases of the sampling program are checked independently, examining the efficiency of each sampling area, development sampling, production sampling, and ore-reserve sampling. Thus, any improvements or corrections can be made without affecting the entire program. If all phases check with the daily sampling procedures, the entire program can be evaluated on the weighted average of each, enabling all aspects of the sampling program to be checked, adding to the validity of the results.

Recovery and Dilution

Initially, recovery estimates are an engineering problem. Factors such as the mining method, rock types, mine depth, and past performance are all considerations when performing the initial calculations of the mine production. Weighing all of these factors, the Lisbon mine was set up at 75% extraction and 15% dilution. By using a theoretical example of a drill intercept with 0.9 m (3.1 ft) at 0.29% U_3O_8, Table 4.1 lists the extraction and dilution applied to hypothetical ore reserves from surface drilling.

During the mining operations, recovery is calculated from proven ore reserves by subtracting the actual production tonnages. They are daily recovery guides and are used to modify mining operations on a daily basis. An ore reserve

TABLE 4.1 In-place ore reserve calculations

Hole Number	Area, m^2 (sq ft)	True Thickness, m (ft)	Grade, % U_3O_8	Tonnage, t (st)	Yield, kg (lb) U_3O_8
52	26,268	1.3	0.21	86,184	180,987
	(282,743)	(4.2)		(95,002)	(399,007)
53	11,675	1.7	0.18	51,071	91,930
	(125,664)	(5.6)		(56,297)	(202,670)
54	11,675	0.9	0.29	28,272	81,989
	(125,664)	(3.1)		(31,165)	(180,755)
55	8,602	2.0	0.15	43,677	65,516
	(92,589)	(6.5)		(48,146)	(144,439)
Totals:	58,219	1.4	0.20	209,206	420,422
	(626,660)	(4.6)		(230,610)	(926,871)
Adjust to 2.0-m (6.5-ft) mining height:		2.0	0.14	295,618	420,422
		(6.5)		(325,863)	(926,871)
Mine Extraction at 75%:		2.0	0.14	221,713	315,316
		(6.5)		(244,397)	(695,153)
Dilution at 15%:		2.0	0.12	254,970	315,316
		(6.5)		(281,056)	(695,153)
Totals		2.0	0.12	254,970	315,316
		(6.5)		(281,056)	(695,153)

reconciliation is calculated every six months; partially mined pillars are planimetered and the ore reserves are revised accordingly. Any newly developed areas then are added to the reserves. As the mining operations are modified and become more efficient, an improvement over the original recovery estimate is developed.

At the Lisbon mine, dilution is reported to management in terms of mining dilution (refer to Fig. 4.8). In effect, this is a measurement of the mining efficiency.

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SECTION 2

Methods Requiring Minimum Support

CHAPTER 5

Room-and-Pillar Method of Open-Stope Mining

Richard L. Bullock

OPEN STOPING

An open stope is an underground cavity from which the initial ore has been mined. Caving of the opening is prevented (at least temporarily) by support from the unmined ore or waste left in the stope, in the form of pillars, and the stope walls (also called ribs or abutments). In addition to this primary support system of open stoping, some secondary support may also be required using rockbolts, reinforcing rods, split pipes, or shotcrete to stabilize the rock surface immediately adjacent to the opening. The secondary reinforcement procedure does not preclude the method classified as open stoping.

There are many forms of open-stope mining used to extract the initial material from a mine. Having once established that the mineral and waste rock are competent enough to use an open-stoping method, and assuming that the reserve is not classified as gassy, the form the method will take is primarily determined by the dip and thickness of the reserve. How these two factors affect the selection of the open-stope mining is discussed in a later chapter. At this point it will suffice to say that the classification of the open-stope mining system that follows is based on whether dry, broken material flows by gravity or whether it must be moved by nongravity methods for which energy must be supplied to move the material.

Room-and-Pillar Mining

Room-and-pillar mining is an open-stoping method by which mining progresses in a nearly horizontal or low-angle direction by opening multiple stopes or rooms, leaving solid material to act as pillars to support the vertical load. Because the direction of excavation (angle of dip) is below that which would cause the dry material to flow by gravity to a drawpoint or gathering point, the material must be loaded in the room where it was extracted and transported to a point where it will flow, either by gravity or mechanical

means, to a central gathering point to be taken out of the mine. This is an important aspect of room-and-pillar mining that differentiates the system from other open-stope mining methods, which rely heavily upon gravity to transport ore from where it was broken to a lower elevation, usually through a drawpoint. There are many variations of the method, which go by a number of names in local districts: breast stoping, breast-and-bench stoping, board-and-pillar, stall-and-pillar, and panel-and-pillar are all basically open-stope room-and-pillar mining.

In some instances detailed stope planning is almost nonexistent; that is, the operator simply follows the visual pay values, leaves pillars only where necessary, and tries to locate them in the zones of lower value. This method of mining is as old as the beginning of underground mining itself, dating back thousands of years. Early in the history of mining in this country, the term "gophering" was used to describe this method. The term is appropriate, for it brings to mind the exact results of this type of system—a random and irregular room-and-pillar mine.

In other instances where the mineral values are consistent both in physical dimensions and quality, the mine layout can be planned to the last detail, resulting in a uniform room-and-pillar mine. Coal, trona, gilsonite, potash, oil shale, salt, limestone, and sandstone mines can usually follow such a system. Today, most metal mines using a room-and-pillar operation try to mine as regular a pattern as possible but deviation in height, width, thickness, dip, and grade of the ore results in comparable deviation in the mine plan.

Variations of the Room-and-Pillar System

It is necessary to briefly describe some of the many variations of the room-and-pillar system of mining, enabling the reader to fully explore the concepts and become familiar with the terminology used before going on to the details of mine design.

Full-Face Slicing. If in the process of opening the rooms the total vertical extent of the mineral values of the particular seam or strata are extracted from the advance of one vertical face; the term used to describe this is full-face slicing. This face is also known as the "breast."

There is no mineral of economic value intentionally left either in the floor or the roof (back) to be mined later. To be able to extract the full-face height in one pass, the mining equipment must obviously be designed to reach as high as the back. In an Appalachian coal mine, this may be all of 660.4 mm (26 in.); for a future oil shale property it might be 15.24 m (50 ft). Normally, however, in a majority of mines where the mining face gets over 6.09 to 6.7 m (20 to 22 ft), the tendency is to divide the face into more than one pass. Over this height, it becomes difficult to properly see and remove loose rock from the back with a hand "mining bar." Where the process of taking down loose rock has become mechanized, higher full-face mining can be safely practiced. Most eastern and midwestern coal seams and western uranium, trona, and potash seams in the United States are easily reached in a single face; many limestone, lead, and

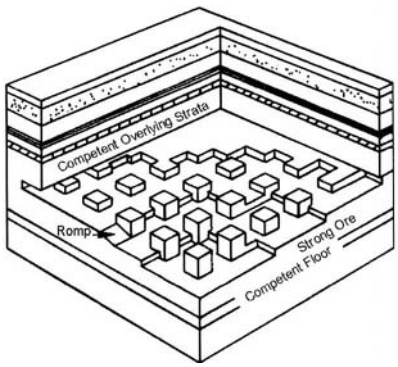


FIGURE 5.1 Open-stope room with regular pillars (Chas. D. Martens & Assoc., 1972)

zinc mines must resort, at least in part, to "multiple-slicing" to remove all the minerals of value.

Multiple Slicing. This method is also known as multiple-pass mining. In many cases it is not practical to carry the full vertical height of the mining horizon as a full face. The face is divided into parts known as the breast, bench, and/or brow. Ideally, the operator, if the vertical extent of the mineralized zone is known, will drill and blast the first pass at the top of the zone, thereby creating the breast stope at the elevation where the permanent "back" (roof) will be. This allows easy access to remove any "loose" slabs of rock from the back while the rock is within easy reach and to secure the back with reinforcement bolts or pins if necessary. The process of scaling loose rock from either the roof, pillars, brow, bench, or breast is in some districts referred to as "mining loose."

Years ago, breast-and-bench stoping was practiced in a somewhat different fashion than it is today. The benches were very narrow [0.91 to 1.52 m (3 to 5 ft)], which allowed most of the fly rock from the breast to be blasted onto the floor of the stope, leaving very little rock lying on the benches.

Figure 5.1 (Chas. D. Martens & Assoc., 1972) illustrates a more typical stope for today's trackless mining, where again the breast "heading" is carried at the top of the mineral zone but the benches are wide and, in effect, can serve as simultaneous levels of mining activity. The floor of each bench may reflect the bottom of the mineralized zone of a dipping orebody, or the lower benches may be advanced through the upper floors, reflecting a thick, flat orebody. Working such an orebody in horizontal slices in a downward direction is known as "underhand stoping."

Roof scaling and reinforcement (rock bolting) would normally take place from the breast heading as it advances. Where the breast height does not reach the top of the mineralized zone, an inverted bench can be carried in the roof (Fig. 5.2; Casteel, 1973). Thus, "taking down back" is a form of overhand stoping whereby horizontal slices are removed in an upward direction. In benching, the point where the upper floor meets the vertical bench is called the "bluffline" or "bluff"; in taking down back, where the roof of the previous slice meets the vertical face of the inverted bench, it is called a "brow line" or

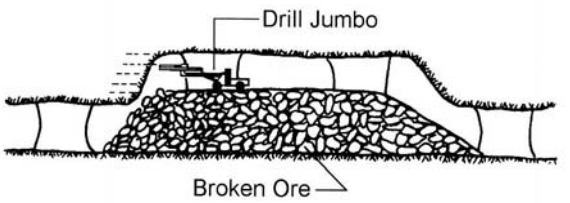


FIGURE 5.2 Cross section at bottom of stope showing initial cut at bottom of ore with remaining ore being removed by back-slabbing (Casteel, 1973)

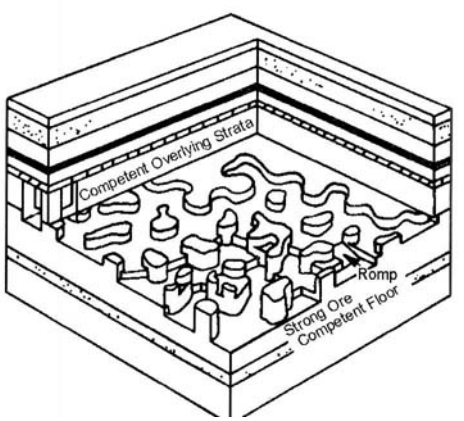


FIGURE 5.3 Open-stope rooms with random pillar supports (Chas. D. Martens & Assoc., 1972)

"brow." Many times in irregularly shaped orebodies, it becomes necessary to leave the broken rock on the stope floor to provide a means of gaining access to the height needed to drill and blast successive slices of the back. The broken "back rock" can provide an excellent surge capacity in times of slack mine production once the top of the ore zone has been reached and the back made secure. But until that time is reached, it also represents inaccessible ore except for any excess ore that must be removed to make room for more broken ore.

The mine in Fig. 5.3 (Chas. D. Martens & Assoc., 1972) has been worked in the same fashion illustrated in Fig. 5.1. There is some degree of uniformity in the size and spacing of pillars, but the uniformity often yields to the necessity of maintaining ore grade control. Consequently, the overall shape of the mineral zone dictates the irregular shape of the existing mine. The large "bars" were left either because of low-grade ore or because they were needed for ground stability.

Other Concepts of Room-and-Pillar Mines. There are several concepts suggested in room-and-pillar mining that call for leaving long "bar pillars" on a regular basis for various reasons. This is common practice in softer and weaker materials such as coal, potash, or shale, but it has not been widely adopted in the United States for use in hard rock mines. Such a system, designed for an entirely different reason, is shown in Fig. 5.4. This scheme

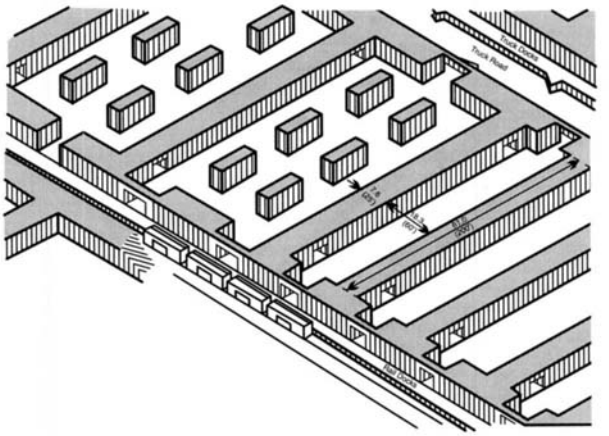


FIGURE 5.4 A special arrangement of isolated panels containing rooms and pillars developed for utilization of industrial space (Christiansen and Scott, 1975)

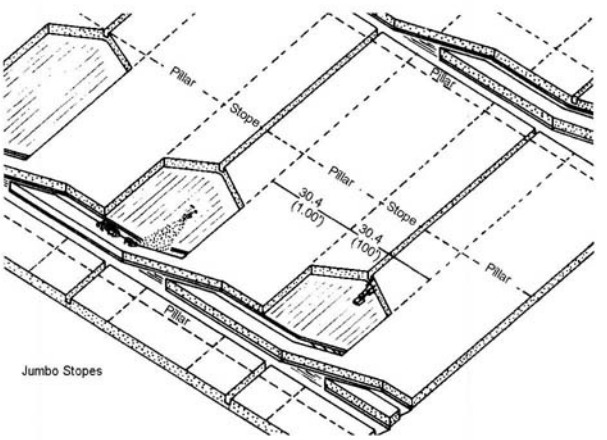


FIGURE 5.5 Stope-and-pillar mining used in the early mining at Elliott Lake

might be needed where there is a need for the underground space to be utilized in an industrial development (Christiansen and Scott, 1975). This method of mining creates a series of parallel rooms almost completely separated from each other by continuous "rib pillars." Where large areas are mined for industrial stone, and it is unlikely that one occupant could utilize the entire area, this method has the advantage of creating rooms without the necessity of constructing partitions after mining.

Still another mining plan, shown in Fig. 5.5 and known as "stope-and-pillar," was used at Elliot Lake (Hedley and Grant, 1972). Here, the panels were laid out in a series of 30.48-m (100-ft) stopes and 30.48-m (100-ft) pillars. These pillars were left by mining the stopes out 15.24 m (50 ft) on each side of a predevelopment inclined drift.

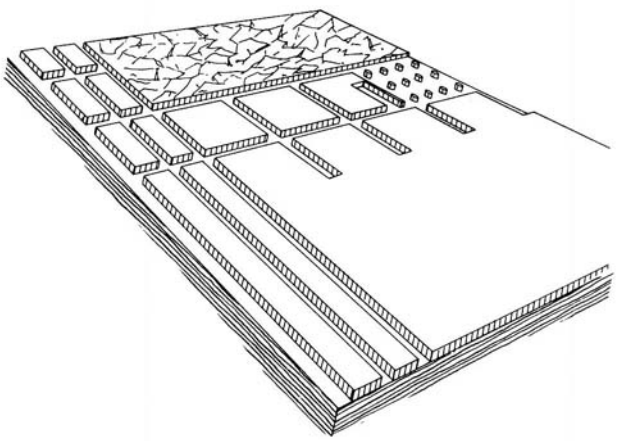


FIGURE 5.6 Room-and-pillar mining typical of underground coal mining a 1.82-m (6-ft) seam, 182.88 to 243.84 m (600 to 800 ft) deep (Paulick, 1963)

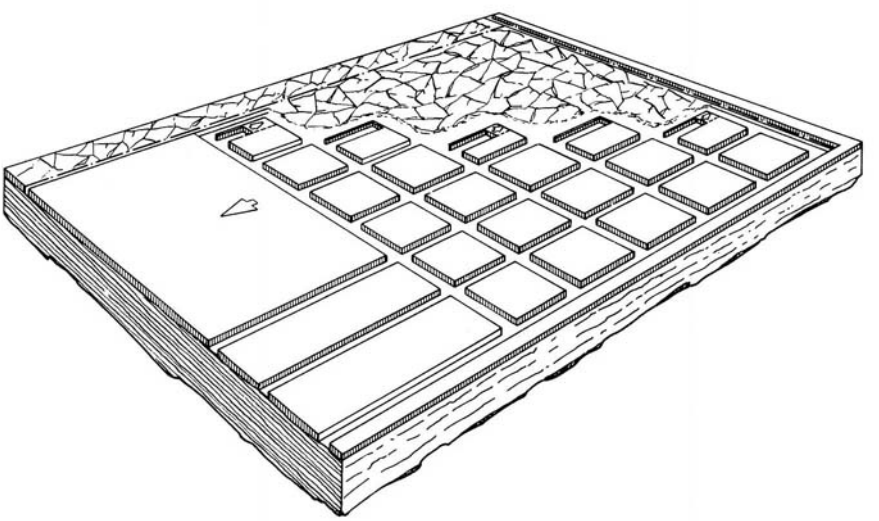


FIGURE 5.7 Room-and-pillar mining typical of underground coal mining a 1.52- to 1.82-m (5- to 6-ft) seam, 91.44 to 182.88 m (300 to 600 ft) deep (Paulick, 1963)

Finally, we come to the most common of all room-and-pillar mining systems, and that which is basic to most coal mining in the United States. There are many different development plans used throughout the country, differing primarily in pillar size and shape, entry size, shape, and number of parallel entries that must be developed simultaneously through the mine. Figures 5.6 and 5.7 are typical (Paulick, 1963). They show not only different schemes of seam development, but also a difference in the method used for mining the pillars while retreating from the mining area.

Structural Character of Deposits Minable by Room-and-Pillar

If most mining engineers were asked to describe orebodies applicable to room-and-pillar mining, their first response would probably be to describe them as any orebody that is large, flat, and in competent rock. Certainly, these are some of the ideal conditions that make room-and-pillar mining relatively efficient; however, they are by no means the limiting criteria from which room-and-pillar mining should be selected.

The discussion of the physical properties of ore and waste given subsequently is intended to make clear to the reader why room-and-pillar mining has such a broad application. Any criteria considered in determining the type of mining method used must, in the final analysis, include all of the following factors: safety; optimal mineral recovery; a mining environment consistent with current social, political, and legal requirements; and an operation that returns the highest economic gain.

Strength of Rock Mass. The earth's composition consists of a number of materials, each having its own physical properties. These materials are arranged in the earth's crust in various forms: solid or broken mass, homogeneous or nonhomogeneous, rock mass subjected to tectonic stress, or rock free from tectonic stress, etc. However, as rock is normally encountered in mining, it is in equilibrium with the other forces of nature. The mine engineer contemplating the design of an open-stope system must assess the general magnitude of these forces, as well as the strength of the mass of material that will be exposed by the opening. Accordingly, the engineer will design the extraction opening in such a way as to preserve the structure of the opening long enough to successfully complete the extraction. This, by our original description of an open stope, as well as mine practice and mining regulations, may not include roof-and-pillar reinforcement. Also, it may not include yielding rock structures that are gradually failing at a carefully monitored rate. In fact, when one looks at the broad spectrum of materials that are commonly mined by room-and-pillar methods, from all types of "host" rocks ranging in strength from 275.7 to 344.7 MPa (40,000 to 50,000 psi) down to 27.5 to 34.4 MPa (4000 to 5000 psi), and at mining depths varying from around 15 m (50 ft) down to more than 914 m (3000 ft), one begins to realize that with today's methods of reinforcing the rock mass—or by using the design tools of rock mechanics in developing such systems as the pressure arch panel and pillar systems, or the use of a yielding structure or mechanical systems, which at a predetermined rate eventually fail—it seems that nearly any rock that will sustain a development opening without immediate major support could probably be mined by some form of room-and-pillar mining. This is not to say that it would always be wise to do so—indeed, it becomes a matter of sufficient total recovery of the mineral as well as selecting the method that will yield the greatest financial gain. Yet, there is sufficient evidence of the use of room-and-pillar mining methods in fairly weak materials that leads one to conclude that the rock need not be entirely competent. Examples of this are seen in many of the deep salt mines, the sandstone mining of New Mexico's and Wyoming's uranium districts, some of the brecciated riff zone structures of the New Lead Belt in Missouri, the mining at depth of the Nonesuch shale

at White Pine, and the potash beds of Saskatchewan. Later in this section, methods of reinforcing rock and details of room-and-pillar layouts as well as the percent of allowable extraction of the reserve, will be covered.

Effect of Size, Shape, and Thickness of Mass. According to an industry study (Dravo Crop., 1974), 15 of the 23 largest noncoal mines in the United States use room-and-pillar mining. Obviously, it is adaptable to very large mineral reserves. One has the means of opening many production areas by merely extending the lateral extent of the mine without the necessity of deepening the shaft and rebuilding the ore handling and hoisting systems in the process.

Many of the large mines use room-and-pillar methods, and a large percentage of very small mines also use them. In fact, although the size of the mass to be mined does affect the size of the operating mine, it has no effect on whether it is advisable to use room-and-pillar mining. Many small room-and-pillar zinc mines have been operated in the Illinois-Wisconsin area, as well as in the old Tri-State district of Missouri, Oklahoma, and Kansas. Yet, the same basic system is operating at White Pine, where as much as 20,407.5 t/d (22,500 stpd) have been produced.

If the mineral to be mined is dipping steeply enough that the material will flow by gravity, and at the same time is very thick or very thin (narrow), the reserve is not one that should be mined by room-and-pillar methods. Other than these limitations related to the shape of the deposit and where single entries can be used, room-and-pillar mining is flexible enough in both the horizontal and vertical directions to follow the outlines of all other mineral reserves. To the extent that the extracted material can be loaded and moved from where it is broken by mechanical equipment and where competent pillars can be formed (or reinforced), the shape of the mineral body to be mined will not preclude the selection of room-and-pillar mining. There are a few examples that illustrate this flexibility. To mine the zinc reserves of eastern Tennessee, the room-and-pillar mining system must be flexible enough to follow the narrow ore stringers formed by fracture filling, which often lead to collapsed dome structures. To mine these massive structures, the stoping has to expand horizontally and vertically in order to extract the full height and width of the mineralized zone. In a similar mining situation in portions of the Bonne Terre mine of the Old Lead Belt of Missouri, vertical pillars as high as 60.96 m (200 ft) were left after many mining slices were removed (Wykoff, 1950). At the Iron Mountain mine near Pilot Knob, Missouri, an orebody the shape of the periphery of an inverted bowl was successfully mined by room-and-pillar methods (Pettit et al., 1957).

As to the thickness of the mineralized zone and how this affects the selection of the room-and-pillar mining method, it has already been pointed out that extremely thick zones can be mined, but in most cases pillar deterioration becomes the critical problem. In the thicker mining zones either pillar diameter must be increased or pillars must be reinforced—or both. This depends on many factors within the mining scheme, the inherent fractures in the rock, and the strength-to-load ratio of the pillar design.

Deteriorating pillar problems will certainly be encountered in trying to mine some of the thick coal seams of the western United States by conventional room-and-pillar methods. Still, thick seam extraction, leaving tall pillars, can be mined successfully; but it becomes a matter of permissible extraction ratio and/or pillar reinforcement. There are reported examples of successfully mined dipping beds of up to 0.52 rad (30°) and up to 91.44 m (300 ft) thick by a room-and-pillar system. Horizontal rooms in both the strike and dip directions are driven, forming superimposed and matched pillars at varying elevations (Dravo Corp., 1974).

At the other extreme, the very thin seams offer special problems to any conventional mining system; however, mining seams as low as 711.2 mm (28 in.) are common in room-and-pillar coal mines throughout Appalachia. Not long ago, seams as low as 304.8 mm (12 in.) were mined in Holland by utilizing scrapers to move the coal to the haulage entry. But by today's standards—working conditions and high labor costs—there is definitely an economic limit as to how low one can successfully carry on room-and-pillar mining.

Dip of Mass to Be Mined. The dip of the mineralized mass to be mined very strongly affects whether the mining method would be classified as a room-and-pillar system. The so-called "cascade mining" used at Mufulira (Anon., 1966), the panel-and-pillar method used between sublevels of the Soumont mine in Normandy (Anon., 1975a), the pillar-and-chambering system used at the Konrad mine in Germany (Anon., 1975b), and the net hole benching method drilled from Alimak climbers at the Denison mine in Canada (Anon., 1974), are all examples of open stoping of steeply dipping orebodies that result in a combination of open rooms or chambers and pillars. However, the cascade mining at Mufulira also removes the pillar almost immediately in a second cycle of mining, which then allows the hanging wall to cave as the mining retreats along the strike. If the dip is not steep enough for the broken rock to move by gravity, mechanical methods must be employed to move the ore from the room where it was extracted. This results in men and/or machinery reentering the stopes after blasting and remaining there at least until the material is removed. This aspect of room-and-pillar mining differentiates it from other open-stope methods. Because of this, safety precautions must be taken in the pillar design, ventilation requirements, the roof control plan, and the necessity of roof-and-pillar scaling. In other open-stope methods the material can flow out of the stoping area by gravity, and personnel and equipment do not have to be exposed to the open stopes.

Advantages and Disadvantages of Room-and-Pillar Mining

Before proceeding to the details of mine planning, it is important to understand the advantages and disadvantages of room-and-pillar open-stope mining. Ideally, the mine engineer will plan an operation that will maximize and/or assure the continuation of the system's advantages while minimizing or eliminating the known disadvantages.

Major Advantages. Inherent in the mining plan of a room-and-pillar system is a high degree of flexibility (adapting to rapid changes), which can be utilized if necessary. However, until such a change is indicated, the system also has the advantage that many aspects of the mining cycle are repetitious. Thus, the system lends itself to mechanization, which can be easily learned and steadily improved.

The system can be applied as a very selective mining system, leaving waste in pillars, or, if the seam thins, by lowering and raising the back or floor, depending on the amount of dilution allowed. Yet the same system can become a bulk mining system, taking everything at a given horizon and thickness and leaving only the remnants of uniform pillars.

The system can be applied to multiple levels simultaneously, in a manner that the operation of one mining level will not affect the structural conditions of the other mining level.

Where the mineral reserve is large and applicable to room-and-pillar mining, it is usually very easy to develop a rather large number of operating production areas. Furthermore, where the development effort is in the same horizon as the mineral to be mined, some mineral production will result from the development. Likewise, if the mine has nongassy classification, the amount of development that will open sufficient working areas can be held to a minimum prior to production.

Equipment replacement in the stoping area is easily accomplished as more efficient or dependable equipment becomes available. Also, by comparison to other stoping methods, equipment maintenance is easier because usually all of the equipment is mobile enough to leave the stoping area and travel to a maintenance area. In some cases, because the materials moving problem is very similar to the same problems encountered in surface operations, the exact equipment can be utilized—minimizing the need for designing and building special underground mining equipment at a higher cost. Equipment can be moved easily from stope to stope, thereby maintaining maximum equipment utilization and output. This is in contrast to mining methods for which equipment is isolated on one level of mining and cannot be moved easily.

The physiological conditions for the noncoal room-and-pillar miners are usually better than for most of the other noncoal mining systems because good ventilation is relatively easy to install and maintain and, with an abundance of equipment in the mine, the places where the equipment operates are normally well lighted. Also, nearly all jobs, even roof sealing and boulder breaking, can be mechanized if it is economical to do so, and this leaves very little strenuous work for the miner.

Major Disadvantages. Among the disadvantages of room-and-pillar mining is maintenance of the roof of the active areas over prolonged periods of time. Because so much roof is exposed as the mine is developed, maintenance can be a problem of safety as well as of cost.

Some "free running" or swelling ground simply will not stand open for any length of time. Therefore, attempting to mine very incompetent rock by open-stope methods is at best extremely costly, and at the worst a complete failure.

Because of the increased load on the support pillars with depth, the normal practice is to decrease the extraction ratio by increasing the size of pillars or decreasing the room size, or both, resulting in less return to the operation in both product and income. There are various methods of overcoming or minimizing this disadvantage, however.

The operation of an efficient room-and-pillar mine today requires a very large capital expenditure outlay for equipment. It seems to be more "capital intensive" for operating equipment than for the other mining methods, because nearly every job is mechanized. However, a factor that helps to reduce the capital expenditure is the versatility of underground equipment. Also, the higher capital cost is normally compensated for by lower operating cost per ton of mineral produced.

Because in some mines the rooms become very high, the back can become difficult to maintain. Special equipment, which has some form of a high lift, must be used to inspect and maintain such areas. The higher the room, the greater will be the force of the impact of a piece of falling rock and the greater will be the damage to anything that it hits. This point is an obvious fact of physics, but is often forgotten by those who promote a false sense of security by using or requiring canopies in "high back" stopes. Keeping the back secure is the correct solution to prevent injury and damage.

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CHAPTER 6

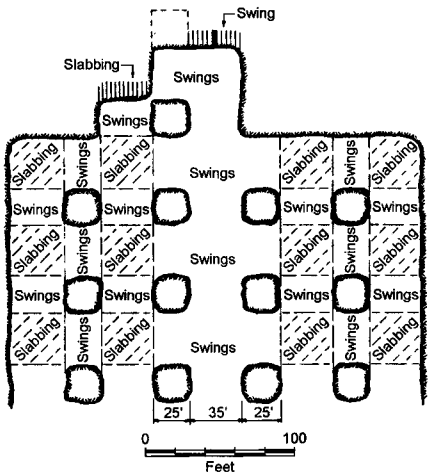
Production Methods of Noncoal Room-and-Pillar Mining

Richard L. Bullock

FULL-FACE SLICING BY DRILLING AND BLASTING

When the entire mineralized thickness is taken in one pass of mining, it is known as full-face slicing. In this method, most ore extraction takes place while advancing through the area rather than while retreating from the area. There is no mineral of economic value intentionally left either in the floor or the back. Theoretically, there is no limit as to how high the face could be in a single pass. But there are practical limitations to equipment size, pillar stability, and control of local loose slabs on the face, roof, and pillars that dictate current practices. The range of face height of the 15 typical room-and-pillar mines covered in the Dravo report (Dravo Corp., 1974) was 1.7 m (5.5 ft) to 9.8 m (32.1 ft), the average being 5.2 m (17.1 ft). In the Gaspe copper mine, Murduchville, Que., the drill jumbos were constructed with an extendable tower for full-face mining up to 15.2 m (49.9 ft) if necessary. However, a more common practice at that mine is to take the ore in multiple slices (Hall, 1959).

In many metal mines, the normal practice when starting stoping is to drive a single development drift into the ore zone a distance that will allow at least four or five rooms to be opened on each side of the initial drift. Because this opening will probably serve as the main haulage drift, it should be kept as straight and as level as possible. As to the other criteria for the stope developments, it might be well at this point for the mine planner to review the remarks presented earlier on drifts, entries, and crosscuts for production. After the initial drift is driven, it should be slabbed to the full room width if it is not that wide already. Next, the future pillars should be marked on the ribs and the rooms driven between them. If the ore extends beyond the length of the initial drift and is to be mined in the normal sequence of mining, care should be taken in a random room-and-pillar system to see that the extension of the initial roadway remains straight. If the shift leader or the miners make the decision on pillar "spotting," they probably will not realize the importance of maintaining a uniform pillar line next to the roadway, or of keeping the line straight. Invariably, a pillar will end up in the middle of the future drift


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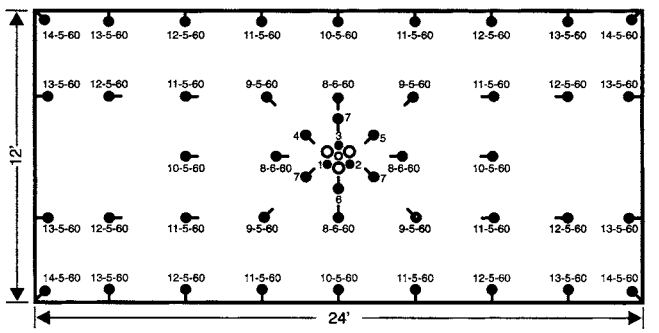


FIGURE 6.2 Burn-cut drift or stope pattern used in dolomite. It can be easily extended in height and width to any size desired (after Bullock, 1975).

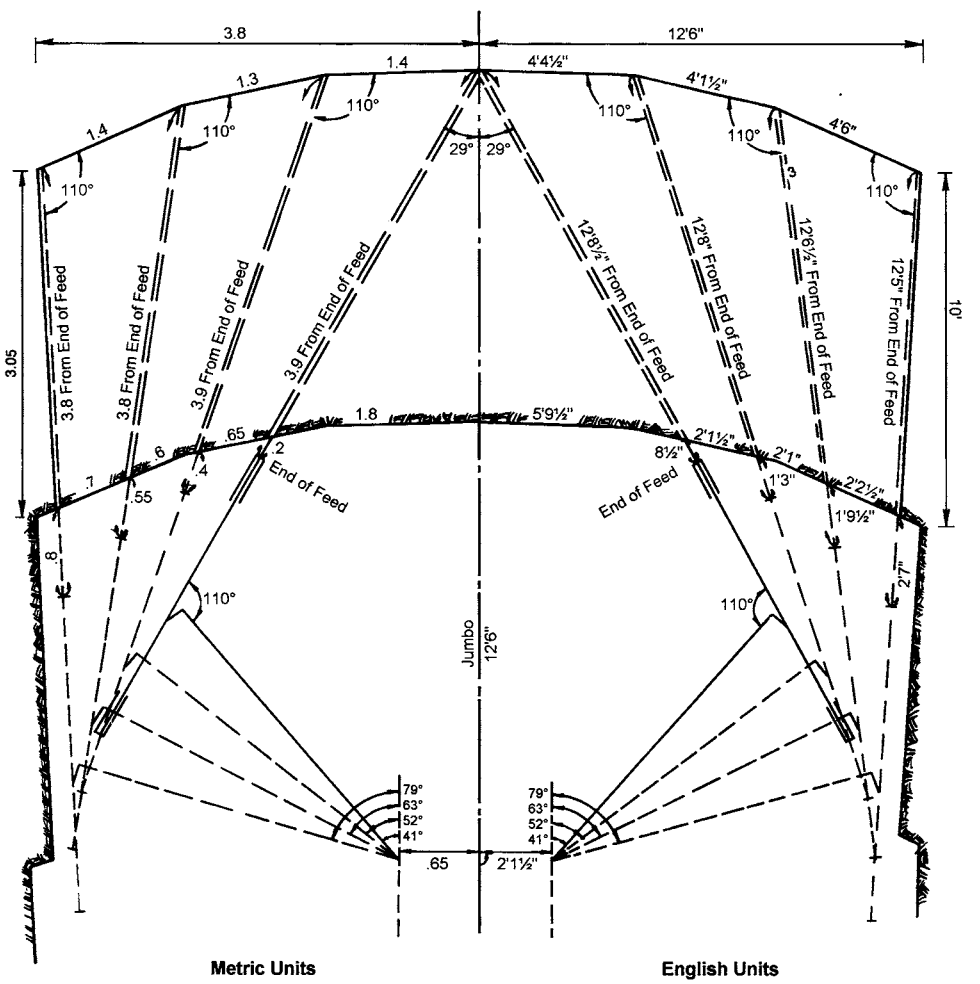


FIGURE 6.3 Typical "vee"-cut round drilled in dolomite (Casteel, 1952)

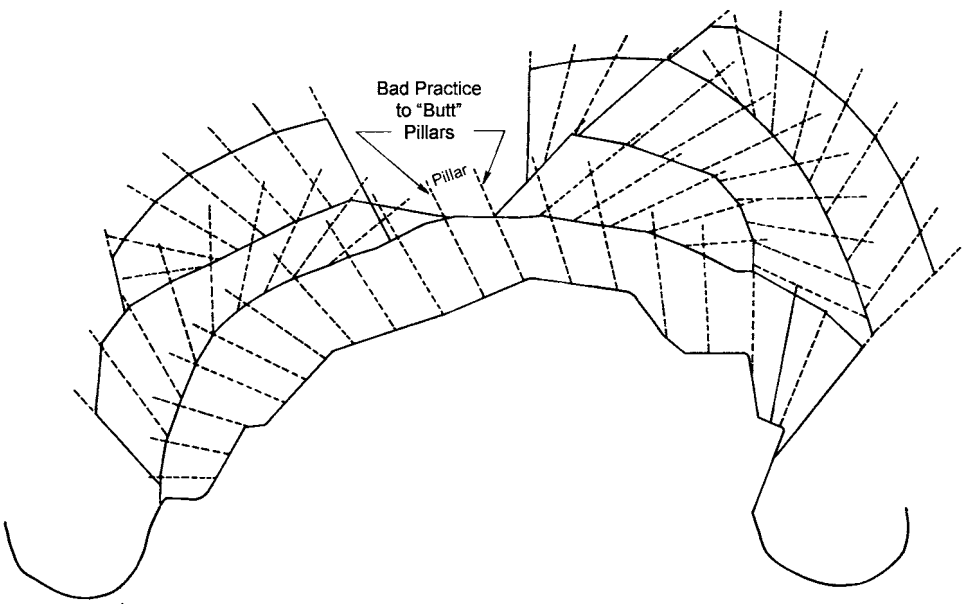


FIGURE 6.4 A slabbing method known as "side swing" or "fan drilling" used in dolomite. It is possible to break a considerable amount of rock, but with a lot of wasted drilling.

second free-face as blasting progresses across the face. Figure 6.4 illustrates an actual surveyed example of the results of this type of drilling. It is much more difficult to teach new drillers this system of drilling compared to either burn- or vee-cut rounds. Even experienced drillers leave much of the hole in the rock that does not break (bootleg), as can be seen in Fig. 6.4. Furthermore, such a practice may be extremely difficult on one side of the pillars when the drill hole bootlegs are left in the pillars. This practice, known as "butting" a pillar, should never be allowed on any type of drilling.

Selecting a Drilling and Blasting Face Pattern

Trying to determine which type of drill pattern to use when there are so many to try is not an easy task. Every miner who has ever drilled and blasted rock will have their own particular ideas as to what must be drilled and blasted to be efficient. The best place to start in order to make this determination for a particular rock, in a particular mining situation, is in a good mining library. So much has been published on drilling patterns in all types of rock and mining situations that it is foolish to try to invent a new round. Select what looks to be the most successful in other similar mines and go from there. The following points are given as starting points in the selection of drilling and blasting patterns for room-and-pillar mining.

The pattern should be simple to learn, easy to drill, and applicable to all of the faces to be drilled in the mine with mechanized, multiple-boom drill jumbos.

There should be a minimum of measuring and marking required by the drillers in order to duplicate the round successfully every time. When miners drill

the pattern in an operating situation day in and day out, there will be drilling errors. The pattern should be overdesigned enough to account for a normal amount of these errors.

The drill round should break consistently every time it is drilled correctly. It should also break at least 90% of the drilled depth of the holes when drilled by the miners on an everyday basis. When drilled and blasted in a research or experimental situation, it should break at least 95% of the drilled depth in repeated back-to-back swings.

The burden and firing sequence should be such that all rock is well fragmented.

In spite of the capability of rubber-tired front-end loaders to scrape up flyrock effectively, the ideal face pattern breaks the rock but does not scatter it all over the stope. This minimizes loading time, equipment maintenance, flyrock damage to equipment, and allows utilities to be kept closer to the active face.

Slabbing should be used only where there is a true free face nearly parallel to the holes for their full length. Table 6.1 shows the 15 mines of the Dravo report (Dravo Corp., 1974) and illustrates what is typically found throughout the United States. Notice that the evaporite mines all have material that is soft enough to be undercut. This undercutting provides for a second free face and, therefore, all of the mineral is broken by slabbing.

DRILLING AND BLASTING TO REMOVE MINERAL FROM THE FLOOR

There are two common methods of drilling and blasting benches—with horizontal holes, as shown in Fig. 6.5, and with vertical holes, as shown in Fig. 6.6. The practice applied with vertical holes is similar to that for drilling a small quarry face on the surface. The equipment usually used for the horizontal hole is the same multiple-boom, rubber-tired jumbo that is used for face drilling. For vertical drilling it is usually better to go to a crawler track-mounted rig with a vertical mast. The data given in Table 6.2 show several examples of both types of drilling. Notice that the No. 3 mine is actually drilling 1.22 rad (+70°); that is, it is not a bluff but a 3.4-m (11.2-ft) thick brow. Note also that the No. 6 mine is drilling and slabbing presplit holes around the pillars. This is an excellent practice and one that should be followed where pillars are going to be very high or have height-to-width ratios much greater than one. The bench pattern that this particular mine drills is shown in Fig. 6.7.

The advantages of drilling benches with vertical holes include: (1) larger tonnages per blast, enabling loaders, trucks, and jumbos to be scheduled more effectively, which results in lower operating, drilling, and blasting costs per ton broken (by 44% according to Grenia and Shannon, 1973); (2) pillar strength to be maintained by better presplitting practices if necessary; (3) lower capital cost for drilling equipment; and (4) the possibility of the next group of holes in the bench being drilled before all rock from the previous blast is loaded, so presplit drilling and blasting can be done months in advance.

TABLE 6.1 Face drilling practices at room-and-pillar mines (after Dravo Corp., 1974)

		Design Practice												
Material Mined	Face Dimensions m (ft)	Drill Type	Type of Cut	Holes/ Round	Hole Diam., mm (in.)	Hole Depth, m (ft)	Depth/ Round Pulled, m (ft)	Penetration, m/min (fpm)	Meters (Feet) Drilled/Unit Shift	Meters (Feet)/ Round	No. Stopes/ Unit Shift	Tons (Short Tons) Broken/ Round	Tons (Short Tons)/ Foot Drilled	Max Footage/ 8-hr. Shift at 0.8 efficiency (384 min)	Actual % Drilled Footage Achieved*
Limestone	9.8 x 18.3 (32 x 60)	Percussion	Vee	82	51 (2)	3.7 (12)	3.4 (11.0)	1.3 (4.4)	409 (1342)	341 (1118)	1.2	1580 (1742)	1.41 (1.55)	2060 (6760)	19.9
Limestone	8.5 x 18.3 (28 x 60)	Percussion	Vee	70	48 (1⅞)	3.7 (12)	3.4 (11.0)	0.6 (2.0)	305 (1000)	299 (980)	1.0	1382 (1524)	1.41 (1.55)	469 (1540)	65.1
Limestone	7.6 x 18.3 (25 x 60)	Rotary-Percussion	Vee	140	57 (2¼)	4.9 (16)	4.3 (14.0)	4.3 (14) New	408 (1340)	610 (2000)	0.67	1569 (1730)	0 78 (0.86)	1640 (5380)	24.9
Limestone	5.2 x 9.1 (17 x 30)	N.A.	Vee	54	44 (1¾)	3.7 (12)	3.0 (10.9)	0.5 (1.5)	198 (648)	198 (648)	1.0	381 (420)	0.59 (0.65)	701 (2300)	28.1
Salt	5.2 x 15.2 (17 x 50)	Rotary	152 mm (6 in. Undercut)	50	44 (1¾)	3.0–3.4 (10–11)	2.7–3.0 (9–10)	6.1 (20.0)	754 (2475)	168 (550)	4–5	524 (578)	0.95 (1.05)	4682 (15,360)	16.1
Salt	9.1 x 24.4 (30 x 80)	Rotary	152 mm (6 in. Undercut)	108	48 (1⅞)	3.7 (12)	2.9 (9.5)	1.8 (5.76)	592 (1944)	395 (1296)	1.5	1306 (1440)	1.00 (1.11)	1347 (4420)	43.9
Salt	3.2 x 19.8 (10.5 x 65)	Rotary	127 mm (5 in. Undercut)	52	41 (1⅝)	3.0–3.7 (10–12)	3.2 (10.5)	2.4 (80)	1219 (4000)	190 (624)	6.4	419 (482)	0.67 (0.74)	1871 (6140)	65.1
Evaporite	1.7 x 9.8 (5.5 x 32)	Rotary	152 mm (6 in. Undercut	15	48 (1⅞)	3.4–4.0 (11–13)	3.0–3.7 (10–12)	3.7 (12.0)	402–475 (1320 or 1560)	50.2–59.4 (165 or 195)	8.0	102–122 (112 or 135)	0.83 (0.69)	1405 (4610)	33.8
Evaporite	2.1 x 8.5 (7 x 28)	Rotary	Undercut	21	48 (1⅞)	2.7 (9)	2.3–2.4 (7.5–8)	2.5 (8.3)	346 (1134)	57.6 (189)	6.0	86 (95)	0.45 (0.50)	972 (3190)	35.6
Evaporite	3.0 x 8.5 (10 x 28)	Rotary	Undercut	21	41 (1⅝)	2.7 (9)	2.4 (8.0)	1.3 (4.2)	N.A.	57.6 (189)	6–8	136 (150)	0.74 (0.82)	N.A.	N.A.
Evaporite	2.4 x 7.6 (8 x 25)	Rotary	140 mm (5.5 in. Topcut)	16	44 (1¾)	2.9 (9.5)	2.9 (9.5)	3.8 (12.6)	324 (1064)	46.3 (152)	7.0	118 (130)	0.77 (0.85)	2950 (9680)	11.0
Metallic ore	2.6 x 9.8 (8.5 x 32) & 4.4 x 8.5 (14.5 x 28)	N.A.	Vee	36-42	41 (1⅝)	3.0–3.7 (10–12)	3.0 (10.0)	0.9 (3.0)	244 (800)	122 (400)	2.0	216–322 (238 or 355)	0.54 (0.60)	701 (2300)	34.8
Metallic ore	4.3 x 11.0 (14 x 36)	Percussion Burn	3–76 mm (3–3 in. diam)	68	38 (1½)	3.4 (11)	3 2 (10.5)	1.5 (5.0)	N.A.	228 (748)	N.A.	363 (400)	0.48 (0.53)	585 (1920)	N.A.
Metallic ore	4.6 x 9.8 (15 x 32)	Rotary-Percussion Burn	3–76 mm (3–3 in. diam)	80–100	38 (1½)	3.7 (12)	3.4 (11.0)	1.4 (4.5)	366 (1200)	329 (1080)	1.0†	435 (480)	0.41 (0.45)	1052 (3450)†	35.0†
Metallic ore	4.9 x 10.0 (16 x 33)	Rotary-Percussion Burn	3–78 mm (3–3 in. diam)	60	38 (1½)	3.5 (11.5)	3.2 (10.5)	1.1–1.5 (3.5–5.0)	244 (800)	210 (690)	1+†	426 (470)	0.62 (0.68)	995 (3264)†	25.0†

N.A. = not available.

*Actual % drilled = $\frac{\text{Feet drilled/unit shift}}{\text{Max footage 8-hr shift}} \times 100$.

†The drillers also charge the round at 0.8 efficiency.

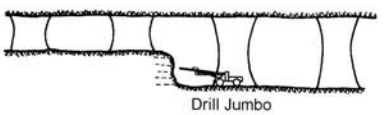


FIGURE 6.5 Cross section of stope showing initial cut at top of ore, with succeeding benches taken by horizontal drilling (Casteel, 1973)

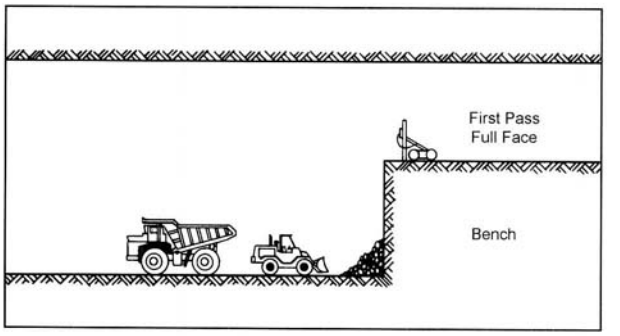


FIGURE 6.6 Cross section through room of typical full face and bench mining operation with the bench being blasted with vertical drilling (after Dravo Corp., 1974)

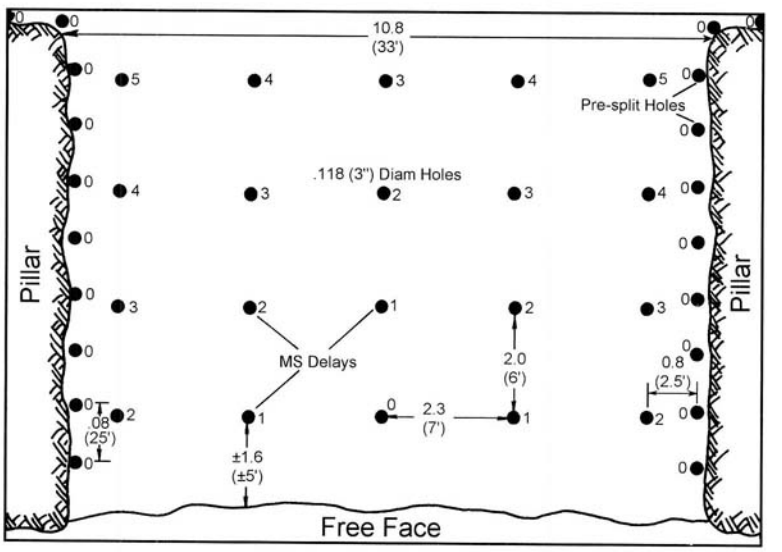


FIGURE 6.7 Standard longhole bench round at Amax Lead Co., Missouri (Grenia and Shannon, 1973)

The advantages of drilling benches with horizontal holes include better control of the mined grade of material because the mineralized thickness is divided into more benches and the horizontal advance of the face is usually less per blast. These factors allow closer examination of mineralization before the next bench is drilled; holes are drilled horizontally, so the explosive force from the holes is primarily directed across the bedding planes of sedimentary rock, making the force more effective in causing failure of these beams (layers) than if it were parallel. This should result in the following: (1) a lower powder factor occurs, all other factors being equal (see mine No. 7, Table 6.2); (2) where both face drilling and bench drilling are done in close proximity, the same equipment can be utilized for both, minimizing parts

TABLE 6.2 Bench and brow drilling and blasting practices typical of room-and-pillar mines

Mine No.	No. 1	No. 2	No. 3	No. 4	No. 5	No. 6	No. 7
Mineral:	Limestone			Salt	Metal Ore in Dolomite		
Drill:							
Type	R.P	R.P	Percussion	Rotary	Percussion	Percussion	R.P
Mounting	Crawler	Crawler	Rub.	N.A.	Rub.	Crawler	Rub.
Steel, mm (in.)	38 (1½ in.)	38 (1½ in.)-Rd	32 (1¼ in.) Hex	N.A. D.D. A-Rod	N.A. N.A.	N.A. 1700 Ser.	32 (1¼)-Rd
Bench height and width, m (ft)	3.7 × 18.3 (12 × 60)	9.1 × 18.3 (30 × 60)	3.4 × 9.1 (11 × 30)	9.1 × 24.4 (30 × 80)	Varies × 11.0 (Varies × 36)	5.4–7.6 × 0.1 (18–25 × 33)	3.7–5.4 × 9.8 (12–18 × 32)
Holes: Size, mm (in.)	76 (3.0)	70 (2.75)	44 (1.75)	70 (2.75)	N.A. N.A.	76 (3.0)	38 (1.5)
Inclination, °	–90	–80	+70	–90	0	–90	+10, –5
Spacing, m (ft)	1.8 × 1.8 (6 × 6)	1.5 × 1.5 (5 × 5)	2.3 × 2.3 (7.5 × 4.5)	4.0 × 1.1 (13 × 7)	1.2 × 1.2 (4 × 4)	2.1 × 1.8 (7 × 6)	1.4 × 1.2 – 1.8 (4.5 × 4.6)
No./Row	11	13	5	7	N.A.	5	7
Additional at rib	N.A.	2	N.A.	2	N.A.	6*	0
Blasted/round	150–200	15	75	18	N.A.	40*	14–21
Depth drilled	12–13	35	12	33	Varies	N.A.	11.5
Drilled/unit shift, m (ft)	293 (960)	76 (250)	146 (480)	272 (891)	N.A. N.A.	110 (360)	229 (750)
Explosive:							
Type	ANFO	ANFO	ANFO	ANFO	N.A.	ANFO-Water gel	ANFO-Semi gel
Amt. used, kg (lb)	1987–2599 (4381–5729)	616 (1357)	346 (762)	620 (1366)	N.A. N.A.	N.A. N.A.	73–109 (160–240)
Tons produced Short tons produced	4202–5496 (4633–6060)	671 (740)	1669 (1840)	1941 (2140)	N.A. N.A.	1292–1796 (1425–1980)	363–544 (400–600)
Kg/t Lb per st	0.47 (0.95)	0.92 (1.83)	0.21 (0.41)	0.32 (0.63)	N.A. N.A.	N.A. N.A.	0.20 (0.40)

*Includes presplit around pillar.

R.P = Rotary-Percussion; N.A. = not available.

Adapted from: Dravo Corp., 1974; Grenid and Shannon, 1973.

inventory, training, and number of pieces of capital equipment, and maximizing the possible working places for a drill crew; (3) the floor can be kept more uniform, keeping loading and hauling more efficient; and (4) there is no need to clean up flyrock or be concerned with back break at the top of the bench.

In deciding which system to use–horizontal or vertical holes–all of the various factors should be considered as to the effect of improving overall efficiency and cost. In either case, one of the most important things in meeting this objective is to try to achieve good, consistent fragmentation that can be loaded easily.

DRILLING AND BLASTING TO REMOVE MINERAL FROM THE BACK

Because it is normally considered an advantage to take the first ore slice at the top of the zone so that the back need be scaled and reinforced only once, it may not be obvious why one would ever leave ore in the back that would have to be removed after the bottom has been cut away beneath it. Actually there are many reasons, a few of which include: (1) the ore may not have been known to be there until the first slice was removed and the back prospected; (2) it may have been known to be there but the top of the ore zone undulated upward in that area and, knowing that it would turn back down at a later point, a normal face height was maintained; (3) the low-grade mineral left in the back can become ore in later years when the economics of the commodity or mine change (this type of remnant orebody constituted a major portion of the mines of the Old Lead Belt, Missouri, which lasted 106 years); and (4) in order to maximize cash flow or to increase the production of a particular mineral at a particular point in time, a high-grade zone may have to be taken, leaving a lower grade zone immediately adjacent to it, even though it is ore. Such mining practices, followed to satisfy a particular company or corporate goal, should not be done without giving full consideration to the trade-off of increasing the operating cost at a later date when the mine will be mining a lower grade ore. This can be a very difficult policy decision to make. Usually there is justification for having a flexible mineral cutoff grade, depending upon the mining situation.

As to the method of drilling and blasting ore in the back after an initial slice has been taken, there are at least five methods used:

1. By using a jumbo that has considerable reach (possibly with boom extensions) and depending upon the thickness of the first slice, it may be possible to drill horizontal holes from the floor of the first pass, reach up [6.1 to 7.6 m (20.0 to 24.9 ft) to 15.2 m (49.9 ft) at Gaspe], and remove all of the ore.
2. Similarly, by using an extendable boom containing a basket (and work platform) and by using personnel in the basket with hand-held machines, extremely high back can be reached. Normally, however, this system is inefficient, and such practice is usually done only when a small amount of rock must be taken down from a high place for safety reasons. Anyone interested in how extremely high stopes (in the Old Lead Belt mines) could be reached before mobile "giraffes" and multicranes were used in the

mines should read of the milling techniques referred to as "trapeze mining" (McClelland, 1982; Wycoff, 1950). Although the system is inefficient by today's standards of productivity, it was a unique system in that it could be applied in any height of ground, go any distance, and could follow any undulation of the ore. Even though it appeared extremely hazardous, it was, in fact, quite safe where properly applied.

3. The jumbo can operate on the initial floor, but instead of raising the booms very high and drilling horizontal holes, the feeds are tilted up at an angle of 0.77 to 1.22 rad (45° to 70°) and the jumbo retreats away from the brow being drilled and blasted. The difficulty with this system is that the back might tend to break irregularly and could be very difficult to hold if all of it did not break evenly to the same layer.
4. The jumbo can drill from the top of a rock pile left from previous blasting and can advance over the same area as the initial slice; it can continue to go up still higher in multiple slices, or it can even work beyond the original stope outline in order to follow an irregular virgin orebody at that elevation. Because broken rock requires more space than solid rock, excess rock may have to be loaded out of the rock pile. In this regard it is similar to working on broken ore in a shrinkage stope. In fact, in the east Tennessee zinc district, the practice of mining the collapsed dome structure by this method is often referred to as "shrinkage stoping." A D-4 or D-6 dozer may have to be kept on the rock pile to doze excess rock off the pile and/or to build roads up to the new level of operation.
5. Where the ore zone to be mined is extremely high and large in lateral extent, it may become more efficient to use a system of longhole blasting. This is done by driving inclined drifts up from the main level to the top of the ore, removing a full-face slice at the top of the ore, using the initial slice at the bottom opening, then using longhole stoping to blast the two levels together This has many of the same advantages and disadvantages as drilling the high benches. The mining system in that stope would become more like the sublevel, open-stope mining system mentioned again later in this chapter and described in detail in another chapter of this section.

FULL-FACE MINING BY CONTINUOUS MECHANICAL MINING SYSTEMS

In full-face mining, most of the ore extraction takes place while retreating out of the area rather than while advancing through the area.

The use of various types of continuous miners to extract ore is mainly limited to the softer materials. Noncoal mines that typically, or at least occasionally, use continuous miners are potash, salt, gilsonite, trona, and, in some instances, uranium and iron ore. The breaking action of the various types of continuous miners may be described as milling, scraping, abrading, cutting, or sometimes impacting. In any case, it is mechanical breaking as opposed to chemical breaking with explosives. As the technology of high-strength and wear-resistant metal has gradually improved, these mechanical systems have been applied to rocks that are stronger and tougher than ever before.

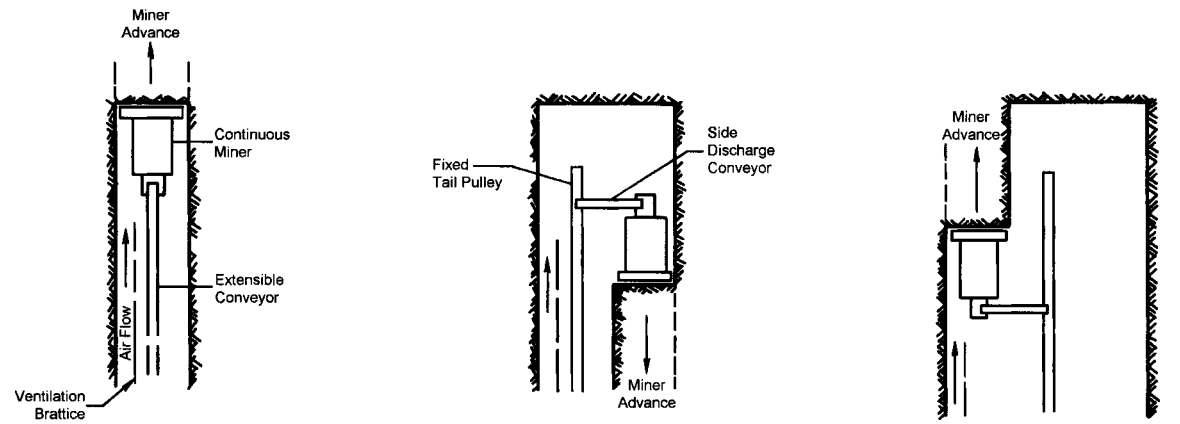


FIGURE 6.8 Plan view of typical mechanized mining operation using continuous miner and full conveyor haulage. (a) First mining pass with 7.6-m (25-ft) wide cut for full room length, extensible conveyor and brattice installed as miner advances, and power and control cables following behind right side of miner. (b) Second mining pass with 6.4-m (21-ft) wide cut in opposite direction; extensible conveyor remains in place, the ore is discharged through a special side conveyor behind the miner, and the brattice is removed as miner retreats. (c) Final mining pass with 6.4-m (21-ft) wide cut, with similar ore handling to the second mining pass. The brattice is reinstalled as miner advances, and completed room width is 22 m (67 ft).

Continuous miners have either a large single rotating drum across the front, multiple rotating cutter faces at the front, a rotating ball-shaped head with cutters on the end of a boom, or, for very hard material, an impact breaker on the end of a boom. Where continuous miners are used, the material is usually loaded into shuttle cars that transport the material to a conveyor belt feeder. Where the material is easily cut by the continuous miners, the shuttle cars can serve as a "bottleneck" to the system. In such cases, it is best to install a continuous haulage system to truly match the miner's capacity. Such is the case at Hudson Bay's potash mine in Saskatchewan (Benson, 1970). Here, panels measuring 1219 x 1219 m (4000 x 4000 ft) are developed on both sides by two panel entries. The potash is cut 2.4 m (8 ft) high on 58.8-m (193-ft) centers—the full 1219 m (4000 ft)—by making three passes, 7.3, 6.4, and 6.4 m (24, 21, and 21 ft) wide, respectively. As can be seen in Fig. 6.8, the conveyor is established on the first pass and left in position for the next two passes. The stope width of 20.4 m (67 ft) is cut, leaving 38.1-m (125-ft) wide bars, 1219 m (4000 ft) in length for 35% recovery. The mine is approximately 914 m (3000 ft) deep, which accounts for the low recovery. But the productivity of operating units and the mine as a whole is extremely high (see Table 6.3, no. 11).

A fairly common method in noncoal mines is to use the rotating head cutters on the end of a boom (Alpine and Doasco types) for cutting fairly small development drifts in uranium mines. These machines are crawler mounted, and after the material is excavated, it is gathered by a continuous-flight conveyor, which discharges to a second belt for final discharge. Production capacity of 54.4 t/hr (60 stph) has been claimed for this type of loader in the situation described. One manufacturer of this type of equipment claims that rock up to 124.1 MPa (18,000 psi) can be mined without excessive maintenance or

TABLE 6.3 Ore haulage practice at room-and-pillar mines (Dravo Corp., 1974) (metric units)

Mine No.	Material Mined	Haulage Equipment	Equipment Description	Unit Size or Capacity	No. of Units		Avail., %	No. Assigned/ Loading Unit	Dump Point	Grade & Roadbed	Haul Distance (1 Way) Metric	Avg. Unit Cycle Time	Avg. Hauled/ Unit Shift, t	Total Trips/ Shift*
					Total	In-Use								
1	*Limestone*	*Off-highway truck*	*Diesel*	21 t	3	2	100	2	Crusher & portal	Flat, crushed rock	488–518	9 min	1794	76–86 (2)
2	Limestone	Off-highway truck	24- & 45-t diesel	27–32 t	2 2	1 2	N.A. N.A.	3	Crusher & portal	Flat, asphalt	1524–2012	12 min	3628/ 3 trucks	100–120 (3)
3	Limestone	Off-highway truck	Diesel	18 t	4	2–3	N.A.	2–3	Underground crusher	10% ramp	1615	15 min	543–566	50 (2)
		Conveyor	0.8 × 847 m Cable suspended	317 t/hr	1	1	N.A.	—	Surface bin	–2 in. rock +17% ramp	847	—	1270 (est)	—
4	Limestone	Load-haul-dump	Diesel-electric	19 m^3 (18–36 t)	3	2	N.A.	—	Vibr feeder-crusher	Flat, bedrock	762	7 min	907	65+
		Conveyor	0.8 m	363 t/hr	1	1	N.A.	—	Shaft pocket	Flat	1317	300 fpm	1814	—
5	Salt	Off-highway truck	18-t diesel	17 t	17	6	95	3	Underground crusher	Flat, salt	305	12 min	499	180 (6)
		Conveyor	1.1 m chain, hung, Floor mounted	726 t/hr	1	1	N.A.	—	12–15 t Shaft store bin	Chain, hung Flat, floor mtd	1676	—	3020	17 hr/day
6	Salt	Off-highway truck	32 t diesel	27 t	4	2–3	N.A.	2–3	Underground crusher	Flat, pckd salt	457	13 min	816	90 (3)
7	Salt	Shuttle cars	Diesel	23 t	13	12	82	1–3	Fdr-bkr to cnvyr	Flat, salt	183	15–20 min	481 t/hr	256 (12)
		Conveyor	Cable suspended	363 t/hr	N.A.	N.A.	N.A.	—	Rail cars	Flat	366–610	—	363 t/hr	—
		Locomotives	14 t	505.9 t cars	14	14	98	N.A.	Rotary dump & skip (shaft)	Track	3048	N.A.	304 t/train	18
8	Evaporite	Shuttle cars	N.A.	N.A.	11	11	N.A.	2	Feeder-breaker	Flat	122–454	N.A.	N.A.	N.A.
		Conveyor	1.1 & 1.2 m Cable suspended	166 m/min	3	3	N.A.	—	N.A.	Flat	2682–7315	—	N.A.	—
9	Evaporite	Shuttle cars	N.A.	8.6 or 10 t	16	6	75	2/section	Feeder-breaker	Flat, salt	46	N.A.	334	219 (6)
		Conveyor	Rigid frame Rope belt	366 m/min 166 t/hr	N.A	N.A.	N.A.	—	N.A.	Flat	1609–3219 km.	—	N.A.	—

Mine No.	Material Mined	Haulage Equipment	Equipment Description	Unit Size or Capacity	No. of Units		Avail., %	No. Assigned/ Loading Unit	Dump Point	Grade & Roadbed	Haul Distance (1 Way) Metric	Avg. Unit Cycle Time	Avg. Hauled/ Unit Shift t	Total Trips/ Shift
					Total	In-Use								
10	Evaporite	Shuttle cars	Diesel	13–15 t	12	6	N.A.	3	Feeder-breakers	Flat, salt	204	4–5 min	544	240 (6)
		Conveyor	Roof suspended	0.9 m panel belt 1.1 m main belt	N.A.	N.A.	99	—	Shaft storage bin	Flat	914 m panel 610 m main	Panel: 550 fpm Main: 700 fpm	N.A. N.A.	— —
11	Evaporite	Conveyor: extensible	Cable suspended 1.1 × 1524 m	1088 t/hr —	2	1	N.A.	—	Panel conveyor	Flat, floor mtd	1219 max.	500 fpm	6530	—
		Conveyor: panel	1.1 m Roof suspended	—	2	1	N.A.	—	Main conveyor	Flat	N.A.	550 fpm	6530	—
		Conveyor: main	1.2 × 5896 m	—	1	1	N.A.	—	Storage bin	Flat	1981	N.A.	6530	
12	Metallic ore	Load-haul-dump	Diesel	4 & 6 m^3	N.A.	N.A	N.A.	N.A.	Vibr belt fdrs	Crushed rock	152–305	N.A.	N.A.	N.A.
		Low-profile-truck	Diesel	15–18 t	N.A.	N.A.	N.A.	N.A.	Vibr belt fdrs	Crushed rock	152–305	N.A	N.A.	N.A.
		Conveyor	Cable suspended	1.1 × 610 m panels	N.A.	N.A.	N.A.	N.A.	Surface storage (Main belt)	11 in. ramp (Main belt)	610 m/sectn	300 fpm	N.A.	N.A.
13	Metallic ore	Load-haul-dump	Diesel	5 m^3	3	2	N.A.	—	Coarse ore bin to crusher	Flat, cr rock	1524	15 min	(Minor)	N.A.
		Low-profile-truck	Diesel	25 & 34 t	2, 2	1, 2	N.A.	2–3	Coarse ore bin to crusher	Flat, cr rock	1524	N.A.	1814	N.A.
		Conveyor	Cable suspended	544 t/hr max	1	1	N.A.	—	Crusher to ore bins	Flat	183	400 fpm	1814	N.A.
14	Metallic ore	Load-haul-dump	Diesel	5.0 or 6 m^3	9	6	N.A.	—	Orepasses	Flat, cr rock	244–305	N.A.	408–454	N.A. (6)
		Locomotives	18.1 t	8–10 Bottom dump 14 t Top Cars	1	1	N.A.	—	Crusher	Flat, track	610	30 min	1814	13.3 (1)
15	Metallic ore	Front-end loaders	Diesel	5.0 m^3, 9-t, 87 t/hr	6	4.5	83	—	Orepass	Flat, cr rock	274	4.68 min	512	255 (4.5)

*Number of units per shift shown in parentheses ( ).

N.A. = Not available.

downtime (Bullock, 1976). In another reported noncoal application of continuous miners, the Jeffrey double-drum 120-HR Heliminer has been successfully used to extract an average of 136 t (150 st) per worker-shift for the face worker at the Arbed French Div. iron mines (Leandri, 1972).

Several advantages of mechanical cutting are the elimination of the delays associated with the drill-blast-ventilate cycle, increased safety because no explosives are handled, the increase in the rock competency of the opening resulting from the rock being cut rather than blasted, and better selectivity if it is required to separate ore from waste at the face. The principal disadvantage of a mechanical mining system is that its success is limited to the softer, less abrasive materials, and when it is used in harder, more abrasive materials, the maintenance costs go up and the productivity goes down. It has another disadvantage from the safety point of view, which is the creation of excessive dust. Furthermore, continuous miners do not have the flexibility or mobility of most trackless equipment. From a specific energy point of view, all continuous mining systems rate rather inefficiently when compared to drilling small holes and blasting. Yet this is not a fair comparison because the degree of comminution in continuous mechanical mining is apt to be considerably greater than that of blasting the rock. Weighing all of the advantages versus the disadvantages, in cases where the rock can be extracted fairly easily with continuous mechanical miners, it pays to take at least the first full-face cut with them.

MOVING EXTRACTED MATERIAL FROM ROOM TO SURFACE

Conveyance Systems

Loading and moving the broken ore out of the rooms and to the main ore storage at the hoisting shaft or the conveyor-equipped stope is normally the most costly phase of noncoal room-and-pillar mining (Casteel, 1973; Dravo Corp., 1974). Therefore, it pays to give this part of the mining system a large amount of engineering study to determine which of the many combinations of materials-handling equipment will result in the desired production objectives and at the same time be the least costly. In most room-and-pillar mines using conventional mining techniques (drill and blast), rubber-tired front-end loaders do most of the loading of the face rock. However, in the softer materials, gathering-arm loaders are still used to good advantage. Semistationary crawler-mounted shovels are used where there is no problem with headroom, the broken rock piles are large enough so that the shovel can stay in one area for long periods of time, and the flexibility of rubber-tired loaders is not needed. This type of shovel is common in Europe and is used successfully as the hydraulic-powered, short-boom excavator (Geyer, 1973).

Where semistationary shovels do the loading, some sort of vehicle must haul the rock either to the next part of the materials-handling system, located near the stoping area, or all the way to the shaft. The haulage vehicles used in most trackless room-and-pillar mines today are either diesel-powered trucks or diesel or electric shuttle cars. As implied, these vehicles may move the material only to the next part of the system—a transfer station. This might

be a feeder-breaker tied to a conveyor system or an orepass connected to either a conveyor, a rail haulage system, or another truck haulage system. From this final haulage system the ore can then be moved to the hoisting shaft or stope.

The concept of a self-loading hauler had its beginning with a unit known as the "gismo," developed in the room-and-pillar mines of American Zinc Co. in 1953. The concept was acquired by Sandford-Day, and by 1958 the first rubber-tired Transloader had been built from a diesel-powered Wagner mixer mobile power unit, and underground materials-moving technology took a giant step forward. After the Joy Manufacturing Co. acquired the Sandford-Day Transloader rights, the loaders spread worldwide. The perfection of the Transloader system between 1963 and 1969 was most apparent in the southeast Missouri room-and-pillar mines, where operators on incentive bonus systems could produce 454 to 544 t (500 to 600 st) per loader shift hauling 610 to 914 m (2000 to 3000 ft) (Bullock, 1975).

At the same time other concepts of front-end loading and hauling that were very adaptable to room-and-pillar mining were being developed by Wagner, Eimco, and Caterpillar. Wagner, for example, introduced the concept that its front-end loader could either load a truck or load-haul-dump (LHD), and in some cases, to balance operating conditions with equipment, the loader would load a truck, pick up its own load, and follow it to the dump point. This system, known as load-and-follow (LAF), is discussed in more detail later.

At this point it becomes necessary to deal in more specific terms as to what type of system or combination of systems will optimize the loading and hauling cost and productivity of a room-and-pillar mine. The systems chart shown in Fig. 6.9 illustrates, to a degree, how complex the optimization can become. Tables 6.3 and 6.4 illustrate loading and hauling equipment of the 15 examples of room-and-pillar mines that were also illustrated in the drilling and blasting section.

Trackless Rubber-Tired Loading and Hauling Equipment

In a noncoal world mining survey conducted on trackless haulage equipment (Johnston, 1975) utilizing LHD equipment, it was determined that 75% of all material was being handled in some phase of the material-moving process by LHD equipment. From this survey one might conclude that for the initial movement, LHD is always the best way to move the material. Still others with surface mining experience may feel that all rock should be loaded with a front-end loader into the haulage trucks. The fact is that for underground room-and-pillar mining, neither statement is always true. For very short, horizontal haul distances, LHD is usually best, and because many of the orebodies around the world are very limited in horizontal distance, they are logically very adaptable to development by short haulage drifts and vertical orepasses. As to when to use loaders with trucks, one cannot always rely on past experience for guidance in a new mining situation. All factors of loading and hauling cost must be considered and include the extra mine development required for some of the systems.

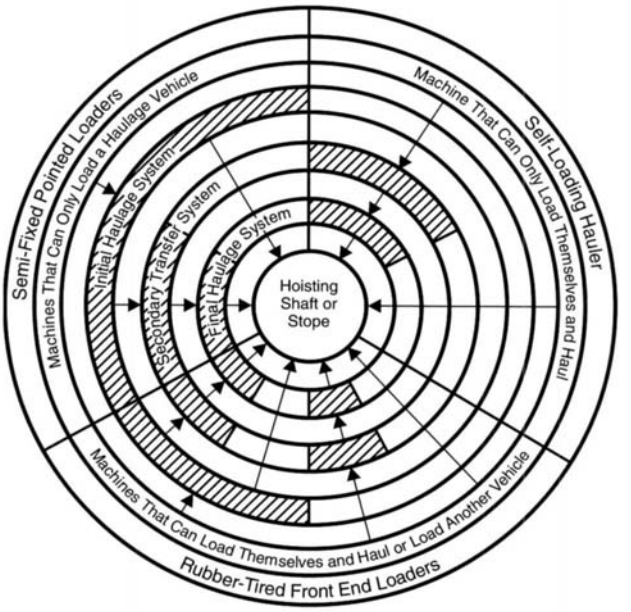


FIGURE 6.9 Materials-moving systems chart illustrates the combination of methods that can be used to transport the material from the room to the hoisting facility

Optimizing Trackless Loading and Hauling Equipment. The problem of optimization (Bullock, 1975) is caused by the very features of the trackless equipment for which it was selected in the first place. That is, the versatility, the flexibility, and the tremendous power of trackless mining and construction equipment not only create a multitude of possibilities for equipment selection as to size and brand, but also application for each specific job that must be done. There has always been some degree of flexibility in trackless mining, and this is the primary reason why it developed. However, with fixed-point loading of some type of mechanical shovel into a given type of conveyance, such as a truck or shuttle car, flexibility was extremely limited, and so were the decisions as to how the equipment was to be utilized. Then came the development of the load-haul-dump concept with equipment that loaded itself and hauled to the dump point. Here again, the equipment is extremely flexible in use but not so versatile in application. If you use only equipment that will load itself, there are only two practical ways to move the ore: either LHD all the way to the shaft, or dump it at an intermediate orepass and pick it up below with another LHD and haul it to the shaft. Therefore, it was not a problem of deciding the most efficient application of the equipment—although there was, of course, a question of how far an LHD could be used and still achieve a minimum cost. The concept of LHD has been expanded to include equipment that can either front-end-load or load-haul-dump.

Utilizing this equipment in conjunction with trucks for hauling part of the distance and/or an orepass and a vibrating feeder, the possible practical methods of moving the ore from the face to the shaft have been expanded to at least four basic methods plus four additional combinations. Even after the

TABLE 6.4 Ore loading practices at room-and-pillar mines (Dravo Corp., 1974)

Mine No.	Material Mined	Loading Equipment	Loader Description	No. Units Used		Avail., %	Unit Loading Capacity	Avg Loaded/ Unit Shift	Faces Served/ Unit Shift	Clean-Up Operations & Remarks
				Total	Prod.					
1	Limestone	Power shovel	1.9 m^3 (2.5-cu yd diesel)	1	1	100*	275 t/hr (299 stph) (Est)	(2000 st)	1 +	4.6 m^3 (6-cu yd FEL) (10% Prod)
2	Limestone	Front-end loader	7.6 m^3 (10-cu yd diesel)	2	1	100*	544 t/hr (600 stph)	(2000 st)	3	Own
3	Limestone	Front-end loader	5.0 m^3 (6.5-cu yd diesel)	2	1–2	N.A.	181 t/hr (200 stph)	907–1134 t (1000–1250 st)	0.5	Own
4	Limestone	Load-haul-dump†	19.1 m^3 (25-cu yd diesel-electric)	3	2	N.A.	136 t/hr (150 stph)†	907 t (1000 st)	3	Dozers
5	Salt	Gathering arm	Electric	7	2	50–60	272 t/hr (300 stph)	1088-1451 t (1200–1600 st)	3–4	FEL & dozer (10% Prod)
6	Salt	Front-end loader	5.0 m^3 (6.5-cu yd diesel)	2	1	N.A.	317 t/hr (350 stph) (Est)	2177-2268 t (2400–2500 st)	1–2	Small FEL
7	Salt	Gathering arm	Electric	9	4	92	188 t/hr (207 stph) (Act)	1361 t (1500 st)	3–4	1.1 m^3 (1.5-cu yd LHD)
8	Evaporite	Gathering arm	Electric	10	5	100	13.6 t/min (15 stpm)	907 t (1000 st)	8	0.8 m^3 (1-cu yd LHD)
9	Evaporite	Gathering arm	Electric	12	4–5	75	N.A.	*648 or* 508 t (714 or 560 st)	4	Dozers
10	Evaporite	Gathering arm	Electric	7	2	N.A.	13.6 t/min 15 stpm	1633 t 1800 st	13	N.A.
11	Metallic ore	Load-haul-dump‡	3.8 & 6.1 m^3 (5 & 8-cu yd diesel)	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.	Own
12	Metallic ore	Front-end loader	3.4 m^3 (4.5-cu yd diesel)	2	1	N.A.	N.A.	1814 t (2000 st)	5	Own
13	Metallic ore	LHD-bottom dump	4.2 & 5.4 m^3 (5.5 & 7-cu yd diesel)	9	6	68–72	N.A.	408 t (450 st)	N.A.	Own
14	Metallic ore	Front-end loader†	5.0 m^3 (6.5-cu yd diesel)	6	4.5	83	N.A.	N.A.	N.A.	Own

*One shift per day.

†Includes hauling.

‡Loads trucks and hauls ore.

N.A. = Not available.

equipment is selected and is being used, the mining conditions change each shift by an ever-changing length of haul and the variable condition encountered by following the ore horizon. Furthermore, with the high productivity possible with today's equipment, many different faces may have to be loaded within the same day with the same equipment, each having a different set of parameters. Yet for any given mine load-haul condition—that is, distance, grade, road conditions, operator practices, etc.—there is only one least-cost method.

When the equipment must be selected prior to mining and one must consider which of the various rubber-tired trackless methods should be used to yield the least cost, and all likely brands and sizes of loaders and trucks have to be considered, the problem becomes incomprehensible for easy analysis and monumental for a straightforward engineering hand calculation. A computer simulation method of accurately determining the optimum method of materials moving for the basic systems is the logical approach. This approach, combined with an abundance of field data, possibly from a similar mine, should guide the mine engineer in using the right trackless equipment in the correct manner.

Basic Methods to Move Materials With Rubber-Tired Trackless Equipment

Load-Haul-Dump I (LHD-I). The rubber-tired loading vehicle fills its dipper (or hopper) and travels to the hoisting shaft or slope where it dumps its load.

Load-Haul-Dump, Haul-From-Chute (LHD-HFC). In this case, the loader as an LHD unit carries the load to a transfer raise, which is equipped with a mechanism to reload the material into a truck, which then hauls the material to the hoisting shaft or slope. It may require more than one truck or loader to balance the production of the other.

Front-End-Load (FEL). The loader fills the bed of one or more trucks in the room where the rock is broken, and the trucks haul the material to the shaft while the loader remains in the stope. The number of trucks that most nearly balances the loader capacity or the desired production should be used. As is often the case in metal mines, the stopes being operated at a particular time are at varying haulage distances and grades from the shaft. But even though a front-end loader has been balanced with the correct number of trucks to eliminate delays in one stope, usually when the group moves to clean up the rock in the next stope, it is unlikely that the system will still be balanced. Where the rooms are very large and the rock is sufficient enough to last over a long period of time, this problem is minimal. But in low-seam mining or small rooms, it is critical.

Front-End-Load/Load-And-Follow (FEL-LAF). A loader fills the bed of one truck, fills its own dipper, and follows the truck to the dumping point. Like LHD, there is no mismatching of equipment if they can travel at about the same speed.

Other variations of these four basic methods are:

Load At Midpoint (LAM). One or more front-end loaders load their dippers and travel out of the stope to load one or more trucks at a midpoint, determined by balancing cycle times for the combination. The trucks return to the shaft while the loaders return to the stope. Often, this is a case of necessity where there is a long development from which the rock is being loaded and there is insufficient room for the loader to maneuver to load the trucks.

Load-Haul-Dump II (LHD-II). Where there is insufficient tonnage to justify a vibrating feeder-truck application on a lower level and a large difference in elevation that would cause too steep a grade to haul efficiently or safely, the rock is moved by LHD to the transfer raise and dumped. Later, the same LHD unit (or another) reloads itself at the lower level and hauls to the shaft.

Front-End Load II (FEL-II). To concentrate production in a large stope, two front-end loaders work simultaneously from both sides to load the trucks in the rooms. This system often is used in surface coal pits.

Front-End-Load/Load-And-Follow Plus Haul-From-Chute (LAF-HFC). Where the haulage has progressed a great distance from the transfer raise but the production level must still be maintained from that point, this combination can be used to continue to balance the desired level of production.

The formulas found in the appendix to this chapter have been developed for only the first four basic methods and are in reference to the simulation tables described later.

Simulation of Rubber-Tired Trackless Equipment. The simulation techniques that have been developed by some researchers are particularly applicable to the problem presented earlier (Morgan and Peterson, 1968; Thieme, 1968; Zambas and Yegulalp, 1973; Bullock, 1975). If one must do all of the simulation in an academic environment and rely totally on the stochastic approach to solve the problem, the research work of others might be a better place to begin (Paden, 1960; Gibbs et al., 1967; Mutmansky, 1972). Other work on the analysis of time study data from shorthaul LHD equipment might also be of help (Wilson and Blackwell, 1974; Gignac, 1977). The work of Bullock (1975) should be of particular interest because it addresses this problem directly. A brief review of this source follows.

A computer program simulating the equipment's haulage capabilities over various mine conditions was adapted from the travel time and earth-moving production program of Caterpillar Tractor Co. Extensive time studies of all the unsimulated events led to structuring the program in the deterministic mode, because it was felt to be more acceptable to the operating management who would eventually use the techniques developed. As a starting point, the computer simulation program calculated only the time of a segment of a course. All segments added together, both loaded and empty, represent the "haul" and "return" time for the equipment. Add to this the unsimulated "fixed time" of that particular system to get the "total cycle" time. The hourly ownership and operating cost were correctly determined for seven different loaders,

three different trucks, and an orepass equipped with a vibrating feeder to load trucks. Study in the mines revealed that the equipment had an 83% availability and operator efficiency was about 90%. During the productive hours, all production units were on incentive systems. Thus for a 6-hr workday, each hour could be expected to produce 44.82 min of simulated production from the personnel-equipment combination.

With this information, an actual LHD-mine condition was simulated for a two-week period (20 shifts), using all the courses actually traversed by all of the loaders in the mine. The actual recorded tonnage produced was within 10% of the simulated tonnage predicted for the mine. Furthermore, the validation was continued for a full calendar month to project the cost prior to the accounting department producing the cost sheets. The actual LHD cost for that mine for that month was within 1% of the simulated cost, illustrating the accuracy to management of the work done up to that point. Similar validations were performed on the trucks moving material with comparable accuracy. Array tables were then generated for all the segment conditions and equipment cited. In this study, haulage segments of 76 m (250 ft) up to 9659 m (31,690 ft) were simulated for various grades from −9 to +17%. The information on the loaders simulated the equipment operating as LHD units and the trucks as hauling material from the vibrating feeder chutes (HFC). Mathematical relationships then had to be developed between the numbers in the tables based on the logic of interactions between pieces of equipment as they moved material from the stopes to the ore pockets by different methods. These relationships were incorporated into the formulas given in Appendix A. The array tables were both printed and stored (by data processing) and revealed haulage and return cycle times, tons per hour produced, and cost per ton produced. By using these tables and the various formulas that describe the production and cost developed by the four basic methods of moving materials, the production capabilities as well as the related cost could be determined for all combinations of the equipment simulated for any existing operating condition or future condition that might be planned for the mines.

Both the formulas developed and the computer programming took the two approaches just mentioned–(1) planning production for existing conditions, knowing what equipment was available to optimize "production planning"; and (2) planning future mining situations, given a desired production to be met and reasonable guidelines for selecting equipment to be purchased. In the latter case, if the optimizing cost leads to the selection of fractional pieces of equipment that might be possible to utilize when considering the expansion of an existing mine, the mode of the approach is termed "situation planning." If, on the other hand, the fractional equipment cannot be utilized, the cost is adjusted according to the idle time that will result from not having the equipment utilized to its full capacity. This is termed "mine planning." If one is using "production planning," the user specifies the type of equipment available, and the distance and grade for each of the courses that would correspond to the four methods. The simulation program then generates each system production in tons per hour, cost in $/ton and cycles per hour. It then lists (explicit enumeration) by least cost per ton the equipment combination for each of the methods. If one is using either the situation or mine-planning approach, the amount of production required, the course description of the

TABLE 6.5 Summary of simulation validations

Method Validated	Range of Distance, meters (feet)	Range of Grade, %	Total Tons (Short Tons) Moved	Total Hours Simulated	Composite Error of Production by that Method	Range of Course Errors, %
1st LDH	457–762 (1500 to 2500)	–1.9 to 3.9	34,175 (37,680)	428	+1.3	–10.1 to 18.9
LAF	658–1448 (2160 to 4750)	–1.0 to 8.0	7435 (8197)	54	+1.1	–12.4 to 14.8
FEL	914–1448 (3000 to 4750)	–2.0 to –1.0	2279 (2513)	18	–10.4	–16.1 to –2.0
2nd LDH	549–914 (1800 to 3000)	–2.0 to –3.8	86,493 (95,362)	1085	+1.2	—
HFC	914 (3000)	–1	3198 (3526)	36	+7.7	—

four methods, and which of the combinations of equipment should be considered must all be specified. The program then computes the amount of equipment required for the types specified and prints them as an explicit enumeration in ascending order of least cost for all four methods. Any of these situations could also be considered by a mine operator using the production and cost tables, the formulas, and a hand calculator. For more complex situations and for those having access to and knowledge of the computer, the programs would do the work. Before the project proceeded to any predictive effort, data from the array tables for the equipment available and the formulas were validated in many actual mining situations. The results of those validations are given in Table 6.5. Where the hours or tons were large, the simulation was very accurate.

Table 6.6 shows the results when two sizes of loaders and two sizes of trucks were specified to produce 154 t/hr (170 stph) in the mine-planning mode.[1] Similar programs were run for all the equipment and for 24 projected mine-planning situations for three mines. Table 6.7 shows three examples of how the data were assembled for management. The simulated projections shown in Table 6.8 show all 24 cases. From this table, it is very easy to see which equipment should be purchased and which operating methods give the least cost.

The results from this study revealed that there were far greater cost differences between the method of moving the ore than there were between brands of equipment. LAF was the least-cost method in 18 out of the 24 situations studied. The LHD method never appeared as the least-cost method even at distances as low as 457 m (1500 ft). For the HFC method to become the optimum, an LHD distance (to the chute) of only 152 m (500 ft) had to be specified. The FEL became optimum at distances between 1478 and 2743 m (4750 and 9000 ft), where the tonnage specified was 154 t/hr (170 stph). However,

1. The average tonnage produced by a jumbo working in a 4.6 to 5.2 m (15 to 17 ft) high stope on two shifts in that particular mine.

TABLE 6.6 An example of the ore hauling optimization, using the mine planning approach (the cost figures are fictitious, but all other figures are real). Required production = 154 t/hr (170 stph).

	Method 1 Distance = 914 m (3000 ft) Grade = -1				Method 2 Distance = 457 m (1500 ft) Grade = 0				Method 3 Distance = 914 m (3000 ft) Grade = -1			
No.	Costs Index	Loader	Quan	LProd*	LCost Index	LTime	Truck	Quan	TProd*	TCost	TTime	Method†
1	0.35	988	1.00	350.43 (386.38)	0.90	3.480	621	1.00	188.58 (207.92)	0.15	9.950	4
2	0.45	988	1.00	350.43 (386.38)	0.95	4.640	631	1.00	251.36 (277.13)	0.20	11.110	4
3	0.50	980	1.00	210.27 (231.83)	0.20	5.800	621	2.00	188.58 (207.92)	0.30	12.270	3
4	0.55	988	1.00	350.43 (386.38)	0.25	3.480	621	2.00	188.58 (207.92)	0.30	9.950	3
5	0.65	988	2.00	97.88 (107.92)	0.50	4.150	621	1.00	188.58 (207.92)	0.12	6.470	2
6	0.66	980	1.00	210.27 (231.83)	0.20	7.733	631	2.00	251.36 (277.13)	0.45	14.203	3
7	0.70	988	1.00	350.43 (386.38)	0.25	4.640	631	2.00	251.36 (277.13)	0.45	11.110	3
8	0.73	988	2.00	97.88 (107.92)	0.50	4.150	631	1.00	251.36 (277.13)	0.20	6.470	2
9	0.75	988	3.00	62.97 (69.43)	0.75	6.680	0	0.0	0.0	0.0	0.9	1
10	0.80	980	3.00	62.49 (68.90)	0.65	3.900	621	1.00	188.58 (207.92)	0.12	6.470	2
11	0.85	980	3.00	62.49 (68.90)	0.65	3.900	631	1.00	251.36 (277.13)	0.20	6.470	2
12	0.90	980	4.00	39.51 (43.56)	0.90	6.170	0	0.0	0.0	0.0	0.0	1

*All production shown in tons and short tons in ( ).
†Method: 1 LHD 2 LHD-HFC 3 FEL 4 LAF

where only 77 t/hr (85 stph) were specified, LAF was optimum at distances ranging from 1372 to 1829 m (4500 to 6000 ft). Concerning the two sizes of trucks, at that particular point in time, the smaller truck's depreciated cost per ton hauled was considerably less than the large truck's. Thus the smaller truck usually turned out to be the optimum size. Later, the manufacturer repriced the small truck in relation to the larger one, and reversed this situation (but not until after the mines had purchased several of them).

To assist mine management in general mine planning, using the conditions of 77 t/hr (85 stph) and 154 t/hr (170 stph) for orebodies 152 m (500 ft) and 457 m (1500 ft) wide, repectively, sets of equipment and operating planning charts were generated. Each set contained a chart showing the least cost combination of equipment specified for all course conditions. The next four charts showed the least-cost method for using particular combinations of equipment for all conditions. Figures 6.10a–e are examples of the 40 charts that were developed for management's use.

TABLE 6.7 Examples of cost analysis, generated by the simulation program production required per hour-77t or 85 st (the numbers under cost/st are relative indexes)

Origin of Rock (Stope, Area or Chute)	Dump Point	Haul Dist. m (ft)	Avg Grade, %	Index of Cost/st for Equipment Considered: Loader A	Loader B	Loader C	Loader D	Loader E	Loader F	Truck	Method of Moving Ore
No. 17 Stope	Shaft	1829 (6000)	–3	1.468	1.209	1.107	1.756	1.238	1.102	None	LHD
	Chute	838 (2750)	–4	0.929	0.810	0.758	0.846	0.808	0.754	621	LHD-HFC
	Shaft	991 (3250)	–1	0.948	0.829	0.777	0.865	0.827	0.773	631	
	Shaft	1829 (6000)	–3	0.705	0.741	0.773	0.780	0.808	—	621	FEL
				0.789	0.804	0.832	0.841	0.863	—	631	
	Shaft	1829 (6000)	–3	0.566	0.532	0.515	0.727	0.589	—	621	FEL-LAF
				0.599	0.540	0.518	0.720	0.583	—	631	
No. 29 Stope	Shaft	457 (1500)	–2	0.534	0.409	0.374	0.422	0.400	0.370	None	LHD
											DUMMY
				—	—	—	—	—	—	621	LHD-HFC
				—	—	—	—	—	—	631	INFO
	Shaft	457 (1500)	–2	0.595	0.631	0.664	0.670	0.698	—	621	FEL
				0.661	0.675	0.703	0.712	0.734	—	631	
	Shaft	457 (1500)	–2	0.334	0257	0.239	0.262	0.247	—	621	FEL-LAF
				0.380	0.280	0.258	0.283	0.260	—	631	
No. 25 Stope	Shaft	1447 (4750)	–3	1.209	0.991	0.906	1.387	1.009	0.903	None	LHD
	Chute	457 (1500)	–1	0.661	0.521	0.490	0.537	0.522	0.486	621	LHD-HFC
	Shaft	914 (3000)	–1	0.679	0.539	0.508	0.555	0.540	0.504	631	
	Shaft	1447 (4750)	–3	0.675	0.711	0.744	0.750	0.779	—	621	FEL
				0.753	0.768	0.795	0.805	0.826	—	631	
	Shaft	1447 (4750)	–3	0.502	0.458	0.440	0.599	0.496	—	621	FEL-LAF
				0.538	0.469	0.446	0.599	0.494	—	631	

* In these cases the cost per ton for the two lines must be added together for the total cost for LHD-HFC.

It must be emphasized that all of the specific information presented here and in the appendix to this chapter reflects the operating conditions, the operator experience and attitude, the management philosophy, and policies that exist in one specific group of room-and-pillar trackless mines. One must be very careful in trying to apply the results shown here to any other set of conditions. The results may not be the same. What is shown here that can be applied in any room-and-pillar mine are the methods that need to be considered and an analytical approach of optimizing the loading and hauling cost using only trackless equipment.

PILLAR MINING IN NONCOAL ROOM-AND-PILLAR SYSTEMS

The extraction of the initial passes of mining for noncoal room-and-pillar properties ranges between 30% and 85% of the ore reserve, but it is usually possible to improve the recovery even more by some system of pillar recovery. There are many things to be considered, however, before pillar recovery starts. Most important is how will it affect the overall mine stability and the

TABLE 6.8 Optimum method and equipment study for future mine haulage for three open stope room-and-pillar mines optimum method and equipment

From Stope to Shaft			Optimum Method and Equipment								
			Best Combination				Worst Combination				
Distance					Truck (Size)				Truck (Size)		Relative Cost Index Between Best and Worst Case, $/t
m	(ft)	Grade, % (Approx)	Method	Loader Brand	t	st	Method	Loader Brand	t	st	
1829	6000	–3	LAF	C	27	30	LHD	D	—	—	1.24
457	1500	–2	LAF	C	27	30	FEL	E	36	40	0.50
1448	4750	–3	LAF	C	27	30	LHD	D	—	—	0.95
533	1750	–4	LAF	C	27	30	FEL	E	27	30	0.44
1829	6000	–2	LAF	C	27	30	LHD	D	—	—	1.05
1524	5000	–2	FEL	B	27	30	LHD	D	—	—	0.81
1067	3500	–2	LAF	B	36	40	LHD	A	—	—	0.80
683	2250	–1	LAF	C	27	30	FEL	E	36	40	0.48
1143	3750	–2	LAF	C	27	30	FEL	E	36	40	0.41
1372	4500	–1	LAF	C	27	30	LHD	A	—	—	0.76
1219	4000	–2	LAF	C	27	30	LHD	A	—	—	0.68
1295	4250	–2	LAF	C	36	40	LHD	D	—	—	0.68
1829	6000	–1	LAF	C	27	30	LHD	A	—	—	1.00
2743	9000	–1	FEL	B	27	30	LHD	F	—	—	0.83
1448	4750	–2	LAF	C	36	40	LHD	D	—	—	0.78
1829	6000	–2	FEL	B	27	30	LHD	E	—	—	0.69
914	3000	–4	LAF	C	27	30	LHD	A	—	—	0.50
991	3250	–2	LAF	C	27	30	LHD	A	—	—	0.56
610	2000*	+3	LAF	C	27	30	LHD	A	—	—	0.36
838	2750	+3	LAF	C	27	30	LHD	A	—	—	0.47
914	3000*	+2	HFC	C	27	30	LHD	A	—	—	0.51
1143	3750	+2	LAF	C	27	40	LHD	A	—	—	0.66
1219	4000*	+1	HFC	C	27	30	LHD	A	—	—	0.73
1448	4750	+1	FEL	B	27	30	LHD	B	—	—	0.50

*Only in these three cases was the LHD distance to the chute specified at 152 m (500 ft), signifying a long narrow orebody. All other distance to the chute was 381 m (1250 ft) or greater.

safety of those working in all parts of the mine? How will it affect the strata above the mine opening if the mine is allowed to cave? Will it allow inflows of water or gas into the mine? Will caving eventually reach the surface and, if so, what social or legal repercussions will this have? Will it affect the surface water table if it has been "perched" during the life of the property? If these and other questions are answered and it is determined that the pillars can and should be partially or fully extracted, the questions of "how to" and "how much" must be solved for each mining area. One fact is common to all properties: before actually beginning any pillar extraction, if the mine up to this point has not started a "rock mechanics" program, then this is the first step. Rock mechanics instruments as described in the literature (Reed and Mann,

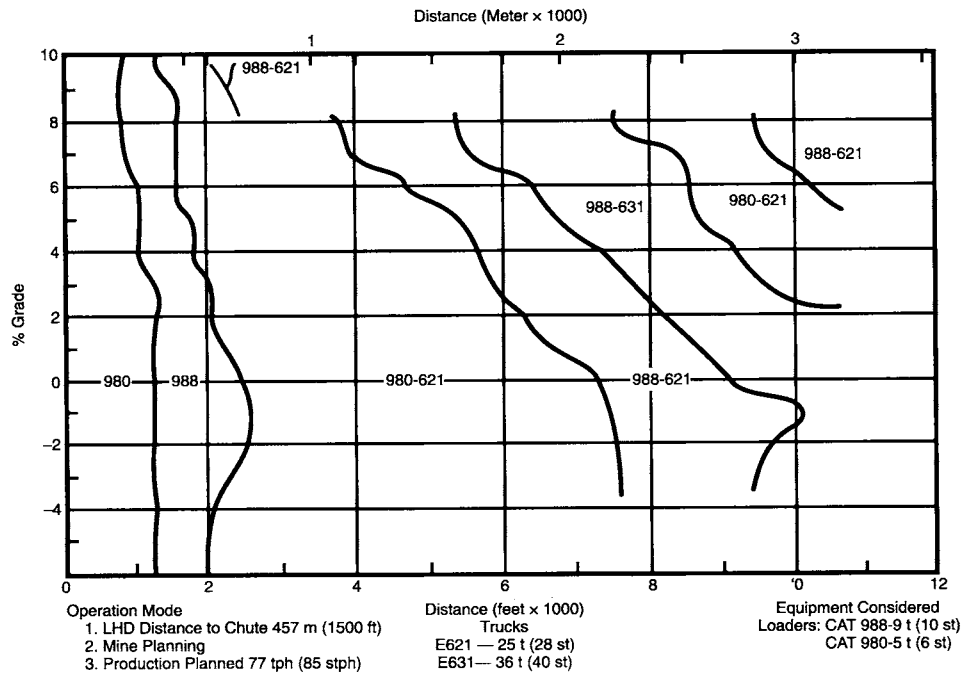


FIGURE 6.10a Equipment and operations planning charts from the materials-moving simulation program (Bullock, 1975)

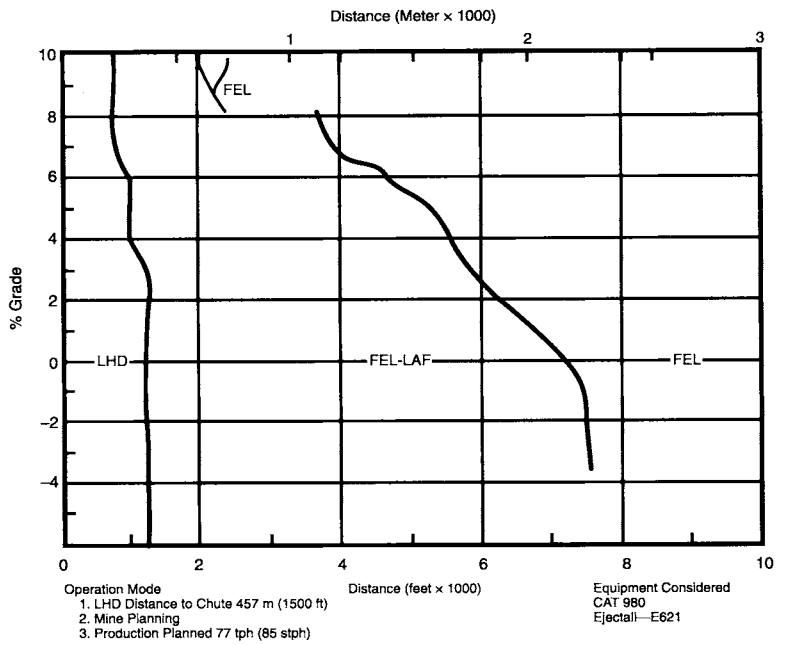


FIGURE 6.10b Equipment and operations planning charts (Bullock, 1975)

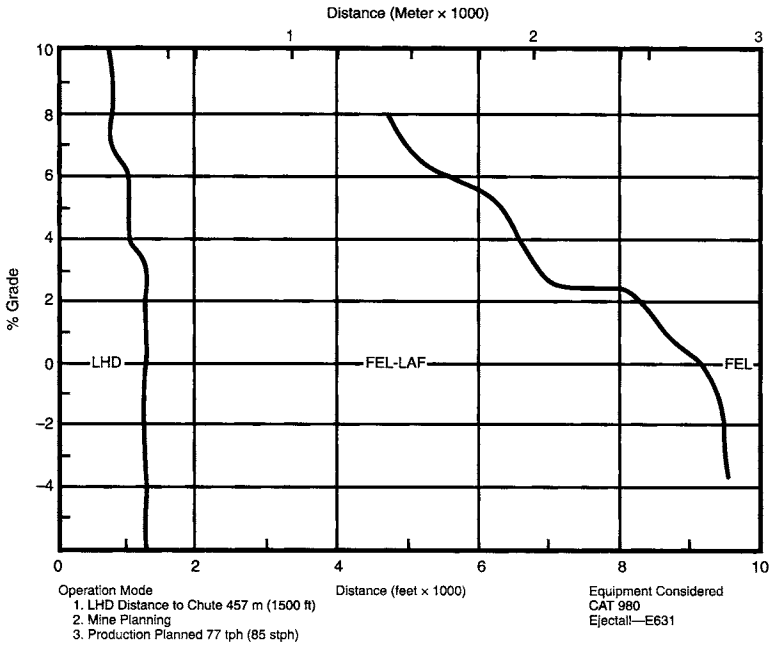


FIGURE 6.10c Equipment and operations planning charts (Bullock, 1975)

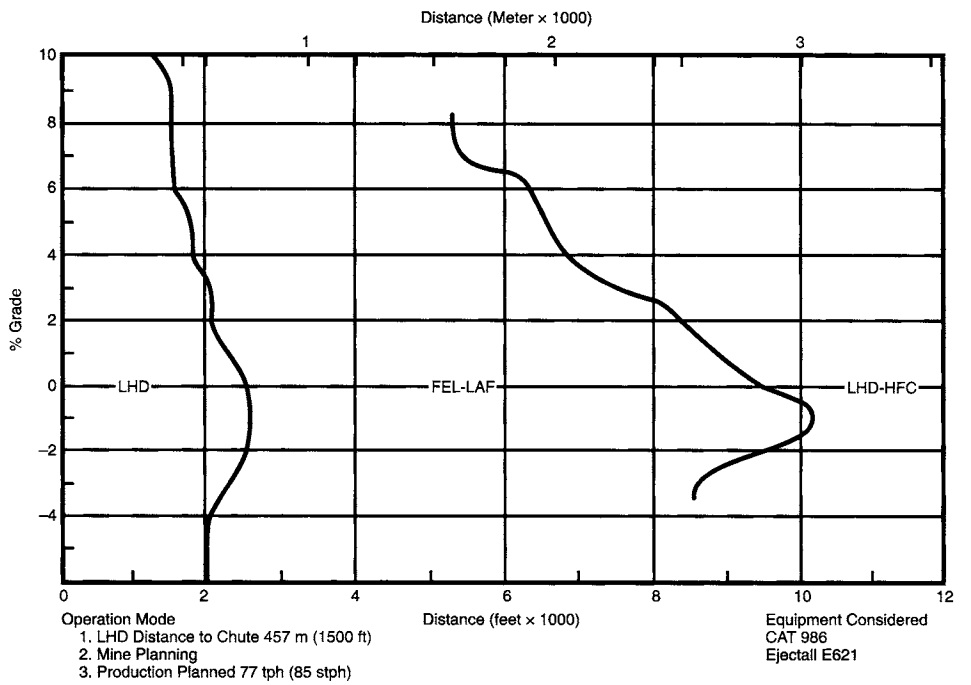


FIGURE 6.10d Equipment and operations planning charts (Bullock, 1975)

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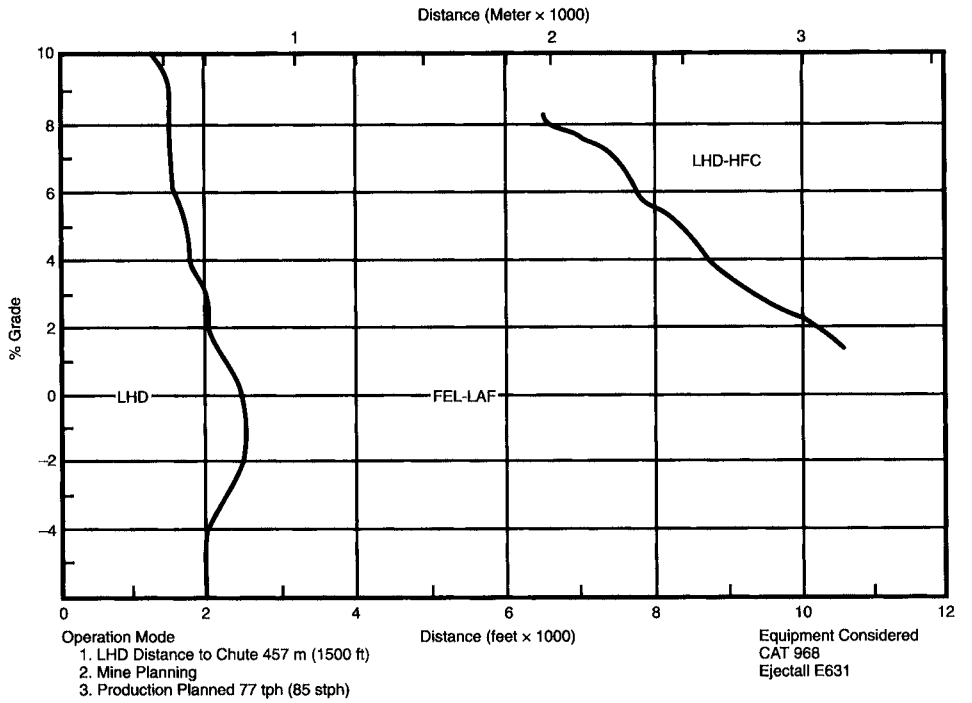


FIGURE 6.10e Equipment and operations planning charts (Bullock, 1975)

1961; Reed, 1959; Budivari and Potts, 1970) can be simple, but must be carefully installed and monitored. If installed before problems begin, they can be the tools that help keep the operator out of serious trouble or help determine when stability has returned after it has been lost.

The major variations of pillar recovery that must be determined are whether partial extraction or full extraction should be attempted and whether the operator should depend upon yielding pillars to partially support the back, or whether caving should be induced or contained to relieve the load on the remaining pillars. These are problems for both those experienced in rock mechanics and for those with a great deal of mining experience in the mining district where the pillars are to be taken.

CASE STUDIES

Extracting Potash Pillars in the Carlsbad Area

The initial mining that began in 1931 in some of the Carsbad area mines recovered 60 to 70% of the mineral. By 1949, very little pillar recovery had been tried, and the primary concern was the large amount of water believed to be in the upper strata (Haworth, 1949) and the possibility of mine flooding. However, by 1953, US Borax had been attempting some pillaring. The first trial was in an area that contained enough clay minerals in the upper

strata that it was felt the layers would "flex readily . . . but not break." The experiment was successful in that 90% extraction was achieved in an area of 244 × 457 m (800 × 1500 ft), with the subsidence reaching the surface 305 m (1000 ft) above within 35 days with no major inflows of water. However, the success from that point on was erratic due to a very weak back. By 1965, two-thirds of the corebody had been "final mined" with only 83% recovery. Roof falls, even in narrow drifts, were common. With some initial mining yet to be done, it was determined that it was better to advance the full distance within a panel just wide enough to have sufficient faces for production, and after the boundary was reached, immediately retreat back with final mining of the pillars.

This system (much like modern coal mining) worked better than the previous method. At the same time, a third system was established, which was not a pillar mining system at all, but because of adequate instrumentation and observation during the experimental pillar removal test periods, a satisfactory solution to the final mining problems was revealed. In this system, only the development entries were driven to the limits of the orebody. Retreat mining was started in which the maximum extraction was done at one time with "bored miners." The initial experimental area that was 599 × 762 m (1800 × 2500 ft), 2.1–2.3 m (7–7½ ft) high was extracted 90 to 94%. US Borax concluded that "it [had] not been possible to date to get the back over the 'final mined' area to cave to the extent that the pressure on the working retreat [pillar] line [was] materially reduced." But with the borer mining system, they ceased "attempting to introduce [a] cave . . . but [depended] on a minimum of temporary support, approximately 8%, which allowed the strata to yield at such a rate that [the] back immediately above the mining area will not be fractured."

Extracting Pillars in the Grant Uranium District

The pillars of ore left after the initial mining of a large orebody usually run from 30 to 70% in the Grant uranium district (Gay, 1963), where the depth, at that time of reference, usually ran from 244 to 427 m (800 to 1400 ft). However, at one property where mining was only 55 to 84 m (180 to 274 ft) below the surface, considerable difficulty resulted in trying to remove pillars on the first attempt. At the Marquez mine in particular, only 40 to 50% was extracted on the initial development. The ore averaged 2.4 to 2.7 m (8 to 9 ft) thick but occasionally ran 7.3 m (24 ft). The two methods of mining the pillars are shown in Fig. 6.11. Room does not permit the full details of this case study, but for any who are about to embark on a pillar-recovery program for the first time, it is recommended for study. The first experience resulted in the loss of a fairly small area and some surface subsidence.

The summary of the problem states: "The reasons for surface subsidence at the Marquez mine appear to be caving of the mined-out area and the unextracted area settling into mudstone. The first had little effect. Both surface and underground control points show 1.2 to 1.4 m (4 to 4½ ft) of subsidence over unextracted [pillar] ground."

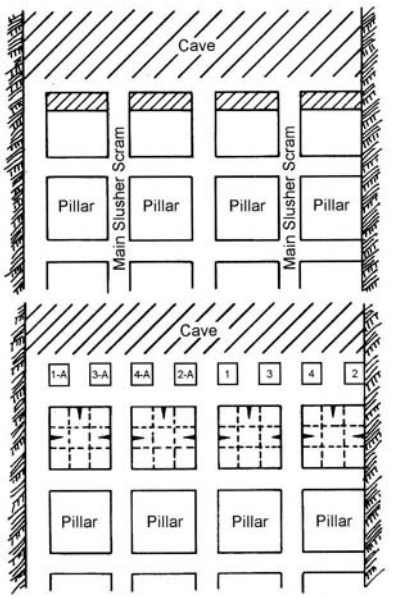


FIGURE 6.11 Two systems of mining pillars used at the Marquez mine (Gay, 1963)

"The conditions leading to pillar subsidence may be outlined as follows:

1. Natural Setting

    a. Incompetent mudstone underlying comparatively competent ore zone.

    b. Natural water course from surface to this same impervious mudstone. . . . Water is not considered as a cause, but it facilitated the movement, once started. . . .

2. Imposed Conditions due to Mining

    a. Pattern and pillar size . . . [were] adequate to maintain rigid floors in the other mine areas but [they] definitely [were] not adequate for the particular area under consideration.

    b. . . . Swinging the stope line abruptly into the south fringe probably started the settling in that locality, as it left the two sides of the corner unsupported. The competent Dakota sandstone did not fail immediately, causing the cantilever action to the east and northeast.

    c. The north and south limits of the west area were completely fringed with crosscuts. The pillars adjacent to them were comparatively narrow. Evidently the stress set up roughly parallel to and above them was enough to cause the failure started by curving the retreat line. This formed a 'floating peninsula.' The Dakota soon

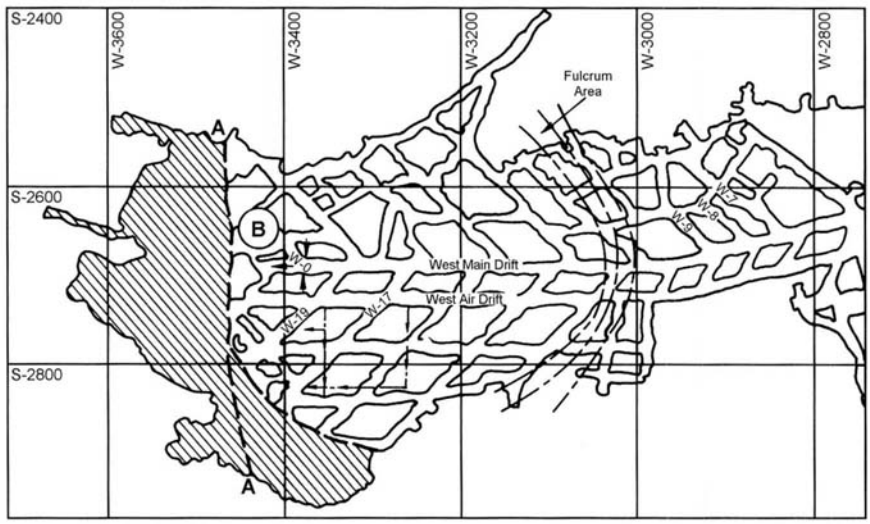


FIGURE 6.12 West area Marquez mine, extracted area is crosshatched and caveline is represented by dark broken lines (Gay, 1963)

> snapped at the fulcrum [(Fig. 6.12)], forming a partially independent island of the orebody. The fractures at the fulcrum run roughly north and south, curving back to the northwest and southwest. This fracture zone is nearly 12.2 m (40 ft) wide, east to west (West-11 to West-10). . . .

"As a sequel to this report, it is interesting to note that [later] a method of fill and mining proved adequate to complete the retreat of the affected area, and this was realized in August 1962. . . . Heaving action stopped completely at about W-7 where competent sandstone tongued in between the ore horizon and the mudstone. . . . Extraction of the remaining reserves of this [mining] area was about 85%."

Pillar Extraction in Old Lead Belt of Missouri

For a period of at least 35 years, some pillar mining was extensively done in the mines, which were originally opened 75 years before. There was no consistent method in use because there were no two mining areas that were similar enough to be "pillared" in exactly the same way. There were several basic approaches to the problem, which are briefly mentioned here.

Very Tall Pillar Removal. Usually where the pillars were over 15 to 18 m (50 to 60 ft) high [but at times often as high as 91 m (300 ft)], the top 2.4 to 3.0 m (8 to 10 ft) were removed and the back bolted as a first step. Sometimes the pillar was drilled and blasted with longholes and sometimes it was drilled and blasted in steps, as if in benching. The ingenuity of the miners in using "trapeze hanging scaffold," as it was called, or in getting themselves and their tools in a position to drill and blast, is an amazing story in itself. On a few rare

occasions, the bottom was shot away first with the expectation that the pillar would break clean of the top and fall. Invariably, however, a remnant was left hanging from the back. In at least one case after the top was removed, the bottom was shot away and it was toppled like a large tree. The very large blocks were then drilled with hand-held drills and blasted.

Normal Pillar Removal. Where pillars were only 3.7 to 15.2 m (12 to 50 ft) high, they could be removed by drilling and blasting in one shot. As before, the top was sometimes removed before shooting the rest of the pillar for pillars over approximately 7.6 m (25 ft) high. The trapeze system was originally used to accomplish this, but gradually gave way to more mechanized systems. Now, with mobile aerial platforms capable of reaching 33.5 m (110 ft), this would not be a problem. Where the overall width of the stoping area was fairly narrow (three or four pillars wide), sometimes all pillars could be removed if the back consisted of thick competent beds of dolomite. In other areas, if all of the pillars were to be removed, the back had to be uniformly bolted prior to pillar removal even in a fairly narrow stope (Reed, 1959).

Pillar Removal with Replacement of Concrete Pillars. There was one early attempt to replace high-grade ore pillars with concrete in the early 1940s (Chellson, 1941). This system failed because the concrete pillars could be loaded only up to 14.5 t (16 st) before the ore pillars were removed. Thus the back had to converge to load the concrete pillars, and when this occurred, the thinly bedded fractured back "unraveled" and came down very soon after a few of the pillars were removed. In the late 1950s, the system of prestress, preloaded "flat jack" concrete pillars was developed. They were installed to reinforce back support in a few areas where pillar slabbing had started problems that threatened to spread beyond the immediate stoping areas. In some cases, caving encroached on permanent installations that needed protection (Reed and Mann, 1960). Although it was not economically possible to replace ore pillars with this type of concrete pillar, it was now at least technically possible. For the first time the mine engineer was able to design and install a pillar that was capable of taking the load without letting the back converge to load the pillar. In fact, in some instances slight roof divergence was measured near such pillars. Later the same system was used to build very high concrete pillars, partially using precast concrete "donut" rings that could be stacked. However, none of these systems could be considered as an economically feasible system to mine the ore pillars.

Pillar Slabbing. During the last 35 years of Old Lead Belt mining, the Southeast Missouri Division of St. Joe Mineral Corp. (St. Joe) tried many different ways of "pillaring" in order to determine the best way to increase the recovery beyond the 85% achieved on the initial mining. At the conclusion of most of the tests similar to those described earlier, it was felt that pillar slabbing—which left a small support that controlled the back convergence—was the most cost-effective method of pillar recovery. This method brought total recovery of the economic ore reserve to approximately 95% in most areas.

No doubt, in many cases the yielding pillars allowed a pressure arch to form between the abutments of the walls and/or the many uneconomical bars of waste that had to be left in their system of random room-and-pillar mining.

Therefore, the convergence that actually took place in most instances represented only the load that was placed on the pillars until the pressure arch formed. The principal exceptions to pillar slabbing being the best way to recover pillar in the Old Lead Belt mines were when the stope was fairly narrow and had a competent back, or where the pillars were so tall that if they were slabbed to a thin diameter, they would fall out. In such cases, the better mineralized pillars were taken completely, with the poorer ones being left intact. Usually 50 to 60% of the pillars could be removed in this manner. Pillar drilling evolved from hand-held machines, through column-mounted drills, and finally to the various jumbos adapted for that purpose. The only criteria for these jumbos was that they be maneuverable, could be taken through fairly small drifts, and have as much reach as possible.

ROOM-AND-PILLAR SYSTEMS EVOLVING TO OTHER SYSTEMS

It may be apparent to the reader by now that in order to follow the many variations in ore horizons and shapes, or in order to maximize recovery of the valuable mineral, sometimes the mining system that starts out as basic room-and-pillar open stope mining ends up more like some of the other mining systems. A few examples are cited here.

Room-and-Pillar to Modified Sublevel Longhole Stoping

Where the ore is very thick, it may be more economical to mine two separate levels by room-and-pillar methods, with a level at the top and one at the bottom of the ore, and the ore between blasted with longholes. The drilling and blasting portion of the system is similar to sublevel open-stope mining. In fact, some mines, because of varying thicknesses and dips of the ore, are continually switching back and forth between these two systems. Two examples are the Denison mines at Elliot Lake (Anon., 1974) and St. Joe at Balmat (*E&MJ* Staff, 1976).

Room-and-Pillar to Modified Shrinkage System

In the Ambrosia Lake uranium district, stoping areas are well prospected as the room-and-pillar system is initially formed. Extraction on the initial pass is only about 30% and the ore is scraped and dropped through orepasses, spaced at 46-m (150-ft) intervals to the haulage level below. If it is determined that the ore is 6.1 to 15.2 m (20 to 50 ft) thick, then a slot is opened at one end of the ore from top to bottom, and the ore is longhole blasted from a drilling drift (sublevel) through the center of the orebody. If it is determined that the ore is greater than 15.2 m (50 ft) thick, scram drifts and finger raises are then developed and the ore is broken with flat longholes, drilled from a raise with the ore going to what is termed a "modified shrinkage stope" (Horne, 1963).

Room-and-Pillar Mining to Modified Cut-and-Fill Methods

The Leadwood mine of the Old Lead Belt was only about 132.8 m (425 ft) deep, but the back was thin-bedded dolomite, interbedded with shale and glauconite, and was also badly fractured and leached. The "roof bolt" was originally developed there in the early 1930s as a means of tying these layers

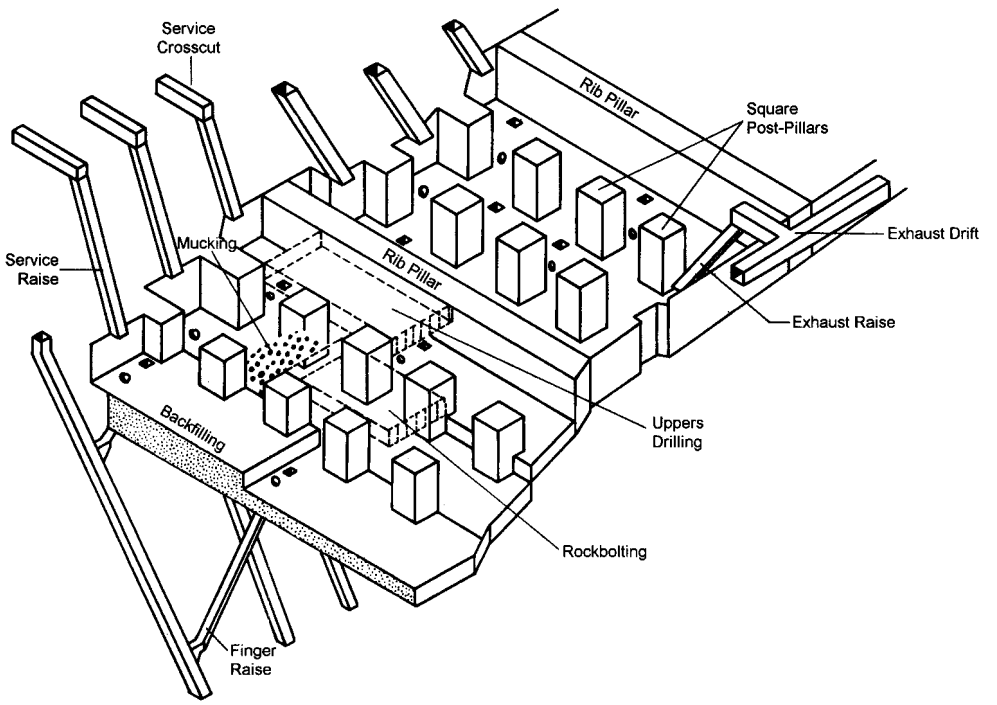


FIGURE 6.13 Layout of post-pillar cut-and-fill stoping (after Cleland and Singh, 1973)

together to act as a beam (Weigel, 1943; Casteel, 1964). Even though the pillars were equally as bad, as a means of economic survival, pillar removal and slabbing took place over a period of 25 years. Occasionally, local cave-ins would occur after an area had been "pillared." But because these cave-ins were beneath uninhabited St. Joe–owned land, they were of no real concern. However, when slabbed pillars between two of these smaller cave-ins began to fail and a third and fourth cave-in occurred in more critical areas, there was a considerable amount of concern. The initial extraction of some of this area involved multiple-pass mining, and room heights were mostly 6.1 to 12.2 m (20 to 40 ft). Final mining of the area resulted in ore extraction of approximately 95%. To stop the caving in the third and fourth areas, over a million tons of tailings were put into the mines, filling the rooms nearly to the back. The results were successful in controlling the converging failing ground.

Compare this case to the "post pillar" mining system used by Falconbridge Nickel Mines (Cleland and Singh, 1973), and the Elliot Lake uranium mines' method (Hedley and Grant, 1972), which is a cut-and-fill system of mining. Figure 6.13 shows the layout of the mining method. A block of ore is developed by driving openings at right angles to each other to form a pattern of square pillars and rooms (i.e., stopes). Mining then progresses upward by drilling and blasting the roof of the rooms. Backfill is poured around the pillars after each cut to provide a working floor for the next cut and prevent spalling of the pillars. This results in very slender pillars extending vertically between the footwall and hanging wall. The initial layout of this mining method was 6.1-m (20-ft) square "post" pillars with rooms 13.4 × 9.1 m

(44 × 30 ft), which gave an extraction of 87%. Because this was a new mining method, rib pillars were left between sets of stopes, and as the method proved itself, the rib pillars were converted to post pillars. The advantages of this method are that a large proportion of the orebody is mined using a less expensive primary method, which is extensively mechanized, and recovery of rib and crown pillars is eliminated. The 13% of the ore left in the post pillars is being abandoned, although techniques may be developed to recover part of these pillars (Hedley and Grant, 1975).

Even though the two cases described here were developed entirely differently and the extraction throughout the systems was different, the end result is the same. The back is being supported by relatively small pillars, which would be yielding if they were not held together with backfill.

APPENDIX: DERIVATION OF PRODUCTION AND COST FORMULAS

Production Planning Formulas[1]

LHD. Because the tables of cost and production[2] were set up for LHD, this information can be read directly from the tables and applied to any number of loaders:

$$P_{r\text{-}1} = L_c \times L_n$$

$$S_{\$\text{-}1} = L_{\$}$$

LHD-HFC. The loader production and cost can again be obtained directly from the tables. Let P_L be the loader production. Then

$$P_L = L_c \times L_n$$

The production and cost tables were set up for the trucks moving and dumping ore, but a time and cost factor must be added for loading with a chute. The chutes that are used by St. Joe load at the rate of 20.8 t/min (23 st per min). This figure was developed by time study of the No. 28 mine chute, loading a 24.4-t (27-st) truck [rated capacity, 25.3 t (28 st)]. Actually, the loading time average was 1.17 ± 0.17 min for 30 loads. Considering the 83% availability and 90% efficiency, there are 44.82 working min in each hour. Therefore, using P_T as the truck's production moving rock from the chute, it became:

$$P_T = T_n T_{bc} \left[\frac{44.82}{T_t + \frac{T_{bc}}{23}} \right]$$

1. For letter designation, see nomenclature at end of appendix.
2. In the original source (Bullock, 1975), cost and production tables were supplied.

$$= \frac{1030\ T_{bc}T_n}{23\ T_t + T_{bc}}$$

P_{r-2} is the smaller of P_T or P_L

In all probability, a mismatch will exist between the two types of equipment in the system, which will result in lost time due either to loaders waiting for dumping room or trucks waiting for ore to haul. The cost for the system must be adjusted to reflect this loss of time and production by increasing cost in the ratio of potential production to system production. Use of this ratio maintains the cost per hour, as determined by ownership and operating cost, at a constant level. The system cost must also include the contribution made by the chute to the overall production cost. Considering these factors, the system cost becomes:

$$S_\$ = L_s\left[\frac{L_c \times L_n}{P_{r-2}}\right] + \frac{1030\ T_\$ T_{bc}T_n}{P_{r-2}(23T_t + T_{bc})} + \left(\frac{1.90}{P_{r-2}}\right)$$

FEL. This system assumed the use of only one loader, and one or more trucks. From time study information, the average time to load one dipper into a truck was 1.16 min (including time for laying out boulders, scraping up, etc.). Therefore, the time to load one truck is:

$$L_{tr} = \left(\frac{T_{bc}}{D_c}\right) \times 1.16$$

By combining this time with the truck's haul time, taken directly from the table, the truck cycle time becomes:

$$T_{ct} = L_{tr} + T_t$$

The truck production, as indicated in the table, will be reduced according to the table cycle time. On this basis, the system production will be:

$$P_{r-3} = \frac{(T_n \times T_c)}{\left[\frac{T_t + 1.16(T_{bc}/D_c)}{T_t}\right]}$$

This rate may be any rate up to the loader's zero distance capacity, as given in the tables, L_{co}. Therefore:

$$P_{r-3} \leq L_{co}$$

The zero distance loading cost given in the tables assumes full production and must be adjusted according to the ratio of zero distance loading capacity to system production. Likewise, the truck cost must reflect an increase in the ratio of truck production to system production. These adjustments account for any mismatch of equipment that may exist, and maintain a constant cost per hour for each piece of equipment. In addition to these adjustments, $0.04 per

ton has been added to the system's cost to account for increased truck abuse, increased ventilation requirements, and truck driver bonus. Considering these factors, the system cost becomes:

$$S_{\$-3} = L_{\$o}\left[\frac{L_{co}}{P_{r-3}}\right] + T_{\$}\left[\frac{T_c \times T_n}{P_{r-3}}\right] + 0.04$$

FEL-LAF. This system assumes only one loader and one truck working together. The time to load one truck is the same as for the FEL system:

$$L_{tr} = 1.16 \times \frac{T_{bc}}{D_c}$$

The truck cycle time and the loader cycle time can be taken directly from the tables. The system time is the time to load the truck plus the larger of the loader cycle time or the truck cycle time:

$$S_t = L_{tr} + L_t \text{ or } L_{tr} + T_t$$

Considering the 0.83 availability and 0.90 efficiency, there are 44.82 working min in each hour. Therefore, the system production is:

$$P_{r-4} = \left[\frac{44.82}{1.16(T_{bc}/D_c) + (L_t \text{ or } T_t)}\right](T_{bc} + D_c)$$

The truck costs must be weighed according to the ratio of potential production, and then both costs must be balanced according to the cost capacity of the truck and the cost capacity of the loader. On this basis, the system cost is:

$$S_{\$-4} = \frac{D_c L_{\$} + T_{bc}\left[\frac{T_c T_s}{P_{r-4}}\right] + L_{\$o}}{T_{bc} + D_c} + 0.04$$

As in FEL, the factor of \$0.04/ton has been added to compensate for taking the truck into the stope.

Mine and Situation Planning Formula

These equations are intended for use to select equipment on the basis of its production potential and a required tonnage over a planned distance and grade of haulage road. Situation planning gives the equipment required in fractional parts and its exact cost (noted by the letter *S* with the equation number). Mine planning rounds up to the next whole piece of equipment and adjusts cost according to the mismatch that results from idle time (noted by the letter *M* with the equation number.)

LHD. Take the loader capacity directly from the production table at the specified distance and grade.

$$L_n = P_r/L_c$$

For situation planning, read the loader cost directly from the cost table.

$$L_\$ = \text{direct from table}$$

For mine planning, modify the loader cost in the ratio of potential production to required tonnage:

$$L_{\$a} = L_\$ \frac{L_c L_n}{P_r}$$

LHD-HFC. Take the loader capacity directly from the production tables at the specified distances and grades. The truck capacity must be adjusted to account for the chute loading time of 20.8 t/min (23 st per min).

$$L_n = \frac{P_r}{L_c}$$

$$T_n = \frac{P_r}{T_{bc}\left\{\dfrac{44.82}{T_t + \dfrac{T_{bc}}{23}}\right\}}$$

$$T_n = \frac{P_r(23T_t + T_{bc})}{1030T_{bc}}$$

For situation planning, read the loader and truck costs directly from the cost tables. Include \$1.90 per hr for the ownership and operation of the chute and feeder:

$$S_\$ = L_\$ + T_\$ + \frac{1.90}{P_r}$$

For mine planning, modify the loader and truck costs in the ratio of their potential productions to the required tonnage:

$$S_{\$a} = L_\$\left[\frac{L_c L_n}{P_r}\right] + \frac{T_\$ T_{bc} T_n\left\{\dfrac{44.82}{T_t + \dfrac{T_{bc}}{23}}\right\}}{P_r} + \frac{1.90}{P_r}$$

$$S_{\$a} = L_\$\left[\frac{L_c L_n}{P_r}\right] + \frac{1030 T_\$ T_{bc} T_n}{P_r(23T_r + T_{bc})} + \frac{1.90}{P_r}$$

FEL. This method is limited to one loader working with as many trucks as required. The time to load one truck (L_{tr}) is:

$$L_{tr} = \frac{T_{bc}}{D_c} \times 1.16$$

Find the truck cycle time in the production table at the specified distance and grade. Based on 0.83 availability and 0.90 efficiency, there are 44.82 working min per hour. The number of required trucks becomes:

$$T_n = \frac{P_r}{\left[\frac{44.82}{1.16(T_{bc}/D_c) + T_t} \times T_{bc}\right]}$$

The maximum number of trucks one loader can keep busy is:

$$T_{max} = \frac{1.16(T_{bc}/D_c) + T_t}{1.16(T_{bc}/D_c)}$$

and

$$T_n \leq T_{max}$$

The zero distance loading cost given in the tables assumes full production and must be adjusted according to the ratio of zero distance loading capacity to required tonnage. Likewise, the truck cost must be increased in the ratio of truck production to required tonnage. To compensate for taking the truck into the stope, \$0.04 per ton is added to obtain the system cost. This cost calculation is the same for situation and mine planning.

$$S_\$ = L_{\$o}\left[\frac{L_{co}}{P_r}\right] + T_\$\left[\frac{T_c T_n}{P_r}\right] + 0.04$$

FEL-LAF. This system assumes only one loader and one truck working together. The time to load one truck is the same as the FEL method:

$$L_{tr} = 1.16 \times \frac{T_{bc}}{D_c}$$

The truck cycle time and the loader cycle time can be taken directly from the tables. The system time is the larger of the loader cycle time or the truck cycle time added to the time to load the truck:

$$S_t = 1.16(T_{bc}/D_c) + (\text{the larger of } T_t \text{ or } L_t)$$

With 44.82 working min per hour, the system production is:

$$S_c = \frac{44.82}{S_t}(T_{bc} + D_c)$$

For the system to be acceptable:

$$S_c \geq P_r$$

The truck cost must be weighted according to the ratio of potential production to required production. Both loader and truck costs must be balanced according to the cost capacity of the loader and the cost capacity of the truck. On this basis, the system cost for situation planning:

$$S_\$ = \frac{D_c L_\$ + T_{bc}\left[\frac{T_\$ T_c}{S_c} + L_{\$o}\right]}{T_{bc} + D_c} + 0.04$$

For mine planning, this cost must be modified in the ratio of system production to required production to compensate for the mismatch between the capability of the equipment and the requirements of the applications:

$$S_\$ = S_c \frac{\left\{\frac{D_c L_\$ + T_{bc}\left[\frac{T_\$ T_c}{S_c} + L_{\$o}\right]}{T_{bc} + D_c}\right\}}{P_r} + 0.04$$

Examples of the Use of Production and Cost Formulas

A situation exists where the mine captain wishes to use a Caterpillar 988 loader and an E621 truck to move ore from a stope 800 m (2625 ft), at a +3.5% grade. If he wished to put in an orepass and feeder chute, the LHD to the chute will average 1500 at +4% grade and the haul from the chute to the shaft will be 1500 at +1%. What method would optimize production?

Variables needed to be identified (L_{c_1}, L_{c_2}, L_t, and T_t values are from data listed in the generated production tables):

L_{c_1} = 62.13 tph (assumed 10-ton capacity)

L_{c_2} = 86.78 tph

L_n = 1

L_t = 7.53 min

T_n = 1

T_{bc} = 30 tons (assumed)

T_t = 7.05

T_c = 189.9

Solution. Production Planning

LHD:

$$P_{r\text{-}1} = L_{c_1} \times L_n = 62.13 \times 1 = 62.13 \text{ tph}$$

LHD-HFC:

$$P_{r\text{-}1} = \frac{1030 T_{bc} T_n}{23 T_t + T_{bc}}$$

$$= \frac{1030 \times 30 \times 1}{23 \times 7.09 + 30} = 160.1 \text{ tph/truck}$$

$$P_{L_2} = L_{c_2} \times L_N$$

$$= 86.78 \times 1 = 86.78$$

$$P_{r\text{-}2} = 86.78 \text{ tph/loader}$$

FEL:

$$P_{r\text{-}3} = \frac{T_n \times T_c}{T_t + 1.16(T_{bc}/D_c)}$$

$$= \frac{1 \times 189.9}{\dfrac{7.09 + 1.16(30/10)}{7.09}} = 127.4 \text{ tph}$$

LAF:

$$P_{r\text{-}4} = \frac{4482}{1.16(T_{bc}/D_c) + (L_T \text{ or } L_L)}(T_{bc} + D_c)$$

$$= \frac{44.82}{1.16(30/10) + 7.53}(30 + 10)$$

$$= 4.07 \times 40 = 162.83 \text{ tp}$$

Therefore, the maximum production would come from the LAF combinations. If he wanted to determine the least cost method it would be found in the same manner, only looking up $L_{\$}$, $L_{\$o}$, and $T_{\$}$. It is interesting to note that LAF would produce even greater tph than two 988s LHD-HFC, with the truck capacity of 160.1 stph being the limiting factor.

Nomenclature

PT = Production tables in original work

CT = Cost tables in original work

P_r = Production rate or required tonnage (tons/hour)

L_c = Loader capacity to LHD (tons/hour), PT

L_{co} = Loader capacity to FEL (tons/hour), PT

L_t = Loader cycle time (minutes), PT (including loading)

L_n = Number of loaders

$L_\$$ = Loader cost to LHD ($/ton), CT

$L_{\$o}$ = Loader cost to LHD–number of units rounded up ($/ton)

D_c = Loader dipper capacity (tons)

T_c = Truck hauling capacity (tons/hour), PT

N_n = Number of trucks

$T_\$$ = Truck hauling cost ($/ton), CT

T_{bc} = Truck bed capacity (tons)

T_t = Truck cycle time (minutes), PT (haul and dump only)

$S_\$$ = System cost ($/ton)

$S_{\$a}$ = System cost–number of units rounded up ($/ton)

S_c = System production capacity (tons/hour)

Source of information: production table (PT); cost table (CT)

1.90 = This is a correction factor used to account for the cost of operating a vibrating feeder-chute.

0.04 = This is a correction factor used to account for additional maintenance cost caused by loading ore into a truck in the stoping area, rather than by a chute.

Any additional subscripts of 1, 2, 3, or 4 imply that this formula applies only to that method.

LHD–Load-haul-dump method (subscript 1 in formulas)

HFC–Haul-from-chute method (subscript 2 in formulas)

FEL–Front-end-load method (subscript 3 in formulas)

LAF–Load-and-follow method (subscript 4 in formulas)

±–Throughout this paper, this is used to denote the standard deviation from the mean of all of the data taken.

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CHAPTER 7

Open Stope Mining at the Magmont Mine, Bixby, Missouri

G.D. Bates

INTRODUCTION

The Magmont mine is a joint venture of Cominco American Inc. (operator) and Dresser Minerals, Inc. The mine-mill operation is located approximately 160 km (100 miles) southwest of St. Louis, Missouri, in what is commonly referred to as the "Viburnum Trend."

The Magmont mine is designed for a production rate of 3810.2 t/d (4200 stpd) on a five-day week, three-shifts-per-day basis. Initial production began in 1968. The mine is open stope, room-and-pillar, and essentially horizontal along the trend of the orebody.

Briefly, the main geological structure can be described as a brecciated graben bounded by reverse faults. The orebody in cross section is shaped like a bell curve with some lateral extension at the lower part. Presently outlined ore is 609.6 to 762 m (2000 to 2500 ft) in width and 2133.6 m (7000 ft) in length. The ore varies in thickness from 4.87 m (16 ft) on the fringes to an average of 27 m (90 ft) in the high ore areas bounded by the reverse faults. Lead is the primary metal with zinc and copper secondary.

MINE DESIGN

The basic design of open stope, room-and-pillar mines has been described by several writers and need not be repeated here (Anon., 1970; Bullock, 1973; Casteel, 1972; Christiansen and Scott, 1970; and Lane, 1964). This discussion covers the mining sequence as applied to the particular problems at the Magmont mine, the use of equipment, and deployment of the work force.

In the upper portion of the Magmont orebody is a layer locally called the False Davis shale. This layer lies below the true Davis shale, is normally interbedded with dolomite, is of varying thickness, and if mineralized, is included in the top pass of the mining sequence.

In thick ore areas this layer will be 2.13 to 2.43 m (7 to 8 ft) in thickness and will occur in the upper portion of the pillars. Due to its incompetency, the presence of this False Davis layer is of primary concern in mine planning and operation.

Mining areas are divided into three basic groups by ore thickness. First are areas of ore up to 6.09 m (20 ft) in thickness. These areas are below the False Davis shale and are mined single pass with drill jumbo.

Second are those areas up to 13.71 to 15.24 m (45 to 50 ft) in height. The first 4.87-m (16-ft) pass is taken at the top of the ore and the back and pillars secured. Benching the lower portion(s) in 4.57- to 4.87-m (15- to 16-ft) passes is then done with either a drill jumbo drilling horizontally or a crawler drill drilling vertically. Normally these areas are below the False Davis shale.

These areas may also be mined by back slashing, or overhand benching, where the first 4.87-m (16-ft) pass is taken at the base of the ore and successive 4.87-m (16-ft) passes are taken upward. A minimum amount of back slashing is done at Magmont because it requires repetition of roof control on each pass, and roof control is the single largest stoping cost at Magmont. Ore left to provide a working platform oxidizes and is coated by oil spills thus reducing metallurgical recoveries.

The third mining area is more than 15.24 m (50 ft) in height up to a maximum of 40.23 m (132 ft) and encompasses the False Davis shale. These areas are mined by first driving +15% inclines to the top of the orebody. The top pass is mined and the back is bolted and roof mats installed as a matter of standard practice to minimize roof problems as mining progresses downward. Once the back and pillars on the top pass are secured, benching begins on successive passes with either the drill jumbo or crawler drill. Pillars on all successive passes below the top pass are secured as necessary. While benching progresses below the top pass, the pass at the base of the orebody is mined, leaving a sill of 4.57 to 7.62 m (15 to 25 ft) in thickness to be removed with the crawler drill in a retreating manner.

Rooms are mined on a 1.57 rad (90°) grid pattern to insure alignment of pillars where multiple passes are taken. Pillars are designed on a 17.98-m (59-ft) spacing with rooms up to 10.66 m (35 ft) in width. Heading widths are wide enough for the mobile equipment to turn without additional allowance for curves. The result is a flexible layout, which provides a maximum number of headings available for high extraction rates and grade control.

PRODUCTION

Incentive Bonus

Incentive bonuses play an important part in the mine production at Magmont. Production crews are trained to perform only one of the mining functions of drilling, blasting, mucking, or roof bolting. This specialization, or functional-

TABLE 7.1 Productivities per worker-shift

Production drilling	700	st*
Charging	1300	st
Mucking	600	st
Rock bolting	33	bolts
Overall mine crew	55	st

*Metric equivalent: st × 0.9071847 = t.

ization, is augmented by development to open all possible stoping areas as early as possible in the life of the mine. This insures that each crew will have enough headings to perform its specialty.

The incentive bonuses increase exponentially as output increases. The lucrative incentive bonus coupled with the specialization of the production crews and proper mine development have combined to give the high productivities shown in Table 7.1.

Development crews perform all mining functions in their area. An incentive bonus is paid on a per foot basis. Crews on different shifts working the same heading share equally in the bonus proportional to their contract hours worked.

Drilling

The daily production of 3809.4 t (4200 st) is achieved with two drill jumbos and two crawler drills. Jumbo rounds are nominally 4.87 m (16 ft) in height and 8.53 to 10.66 m (28 to 35 ft) in width. Normal advance is 3.04 m (10 ft) per round, breaking 408 to 453.5 t (450 to 500 st). All rounds are drilled with 38.1-mm (1½-in.) diam bits, 3.5 m (11½ ft) deep using a standard burn cut (Bullock, 1961).

Two-person crews use three-boom drill jumbos for both production and development drilling. The center boom has a pantograph feature for maintaining parallel holes for the burn cut. The two-person, three-boom concept has several advantages over both a two-person, two-boom or a one-person, two-boom concept. Duties can be shared on the center boom, so that all three drills are fully utilized; many minor maintenance problems can be handled by two people more efficiently; moving these types of jumbos and their hoses and cables almost necessitates two people; and there is sufficient time to keep the working place clean and safe.

Jumbo driller production is maximized by minimizing moving time. A minimum of 12 headings is maintained for each drill jumbo crew. The drill crews will drill three or four rounds, approximately one shift's drilling, before moving to another stope.

Crawler drill rounds are drilled 5.48 m (18 ft) deep using a 1.82-m (6-ft) sectional drill steel with 76.2-mm (3-in.) diam bits on a 1.52 × 1.52-m (5 by 5-ft) pattern. Benches will break to a nominal depth of 4.51 m (15 ft). Any number

of rows may be drilled before blasting, but normal blasts consist of six rows or less. Because the top horizontal layers tend to rip backward into the next bench to be drilled, entire intersections are drilled and blasted in one stage.

Although the crawler drill is capable of drilling much deeper holes, the 5.48-m (18-ft) round was selected for several reasons. The higher the bench, the greater the chance of exposing unforeseen problems in pillars that would be difficult to correct because most other equipment is designed for maintaining pillars in 4.57- to 4.87-m (15- to 16-ft) passes; shorter holes tend to deviate less, thereby reducing the chance of overdrilling into the pillars; the driller can easily clean the shorter holes of drill cutting by blowing them with air; the 5.48-m (18-ft) holes can be effectively dewatered with a diaphragm pump and a suction hose for blasting with ammonium nitrate–fuel oil (ANFO); there is greater flexibility in converting a jumbo bench to a crawler drill bench or vice versa or connecting areas mined from different directions with different drilling equipment; and any lateral extensions of ore into the ribs will be exposed at the proper height for jumbo drilling.

The decision to use drill jumbos or crawler drills in benching depends in part on the location of the False Davis shale in the pillars. If the shale is close to the pass being removed, the drill jumbo will be used because horizontal drilling produces smaller diameter drill holes for blasting, thereby lessening the chance of concussion damage to the pillars.

Crawler drills are used wherever possible because of their high productivity, lower maintenance costs versus a drill jumbo, and lower capital costs versus a drill jumbo. Also, the crawler drill can continue drilling on the top of the bench while mucking and scaling are progressing below the bench; it is closer to the back and therefore closer for visual inspection of potential problems than a drill jumbo on the same level as the bottom of the bench.

The main disadvantage of the crawler drill is its low mobility. The Magmont mine is developed to minimize this drawback by blocking out large areas where the crawler drill can work for several weeks or even months at a time.

Blasting

All charging and blasting is performed by two two-person crews with one crew on each of the two production shifts. The blasting crews use a 2.26-t (2 ½-st) truck with a scissor-type lift platform. This truck has an ANFO pressure placer mounted on the platform and carries all supplies for one shift of charging.

Because the sulfide ores oxidize readily, resulting in lower metallurgical recoveries, only the daily production requirement is blasted each day. Although drillers are allowed to drill as many headings as possible in advance of blasting, the grade can be controlled by assigning the rounds to be blasted. Blasting is done at the end of each production shift.

All headings and benches are charged with ANFO where conditions are dry enough. Semigelatin dynamite is used in wet headings and benches, and water

gel in wet crawler drill rounds. Present usage is 80% ANFO, 16% dynamite, and 4% water gel.

Mucking and Hoisting

Mucking is performed by one mucking crew on each shift. One front-end loader loads three trucks on day shift and one front-end loader loads two trucks on evening shift. Each shift of muckers works as a crew. Each person on each crew shares equally in the incentive bonus proportional to the number of contract hours worked.

All mucking is done with 4.06-m (4½-yd) front-end loaders, and haulage is by 31.74-t (35-st) trucks for a distance of 975 m (3200 ft) average and up to 1828.8 m (6000 ft) maximum. This combination of loaders and trucks provides a very versatile and flexible mucking system. At these distances there is minimum waiting time for the trucks to be loaded.

Roads are well maintained using a road grader. Waste rock from development headings is crushed underground with a 457.2 × 609.5-mm (18 × 24-in.) portable crusher to provide road material. Some water wetting of the roads is required during the dry winter months to reduce road dust.

Ore is crushed underground in a 1066.8 × 1219.2-mm (42 × 48-in.) jaw crusher to −152.4 mm (−6 in.) before being hoisted to the surface in a 14.5-t (16-st) skip.

Ground Control

The two rockbolting crews use 41.28-mm by 1.52-m (1⅝-in. by 5-ft) friction rock stabilizers on approximately 1.52-m (5-ft) centers. In the high ore areas, roof mats are also installed. These crews use a three-boom jumbo equipped with a personnel basket mounted only on the center boom.

All pillars are individually mapped for the presence of shale layers, faults, or fractures. The pillars are then reinforced across these planes of weakness. No. 6 [19.05-mm (¾-in.)] concrete reinforcing bars 3.04 m (10 ft) in length are grouted into 38.1-mm (1½-in.) diam drill holes with a mixture of cement to fly ash in a 2:1 ratio. In extreme cases, larger diameter reinforcing bars up to 6.09 m (20 ft) in length may be used.

Rock mechanics instrumentation at the Magmont mine consists primarily of wire extensometer stations to measure roof and floor convergence or divergence.

EQUIPMENT

The philosophy of the Magmont management is to purchase production equipment that has been well tested and proven and to refrain from purchasing specialized pieces of equipment. The following is a list of equipment needed to maintain present production rates.

4 3-boom jumbos for drifting, benching, and development

2 2-boom jumbos for roof bolting

2 crawler drills for benching

4 4.11-m (4½-yd) front-end loaders for production, development, and utility work

5 31.7-t (35-st) trucks for haulage

2 mechanical scaling machines

1 13.71-m (45-ft) scaling tower

1 25.9-m (85-ft) scaling tower

2 1.37-m (1½-yd) front-end loaders for utility work

1 road grader

1 grouting machine

1 shotcreting machine

2 forklifts for moving supplies

4 supervisor's tractors

1 service truck with 6.09-m (20-ft) boom tower

1 surveyor's truck with 9.14-m (30-ft) boom tower

1 portable crusher for road material

An underground crusher-conveyor system has been installed in lieu of expanding the truck haulage system. The conveyor system is 1158 m (3800 ft) long, cable-supported, suspended from the roof, and equipped with a centrally located 762 × 1066.8-mm (30 × 42-in.) jaw crusher. The crusher-conveyor system has reduced the average haul distance to 792.48 m (2600 ft) from 1920.24 m (6300 ft).

VENTILATION

Ventilation requirements are satisfied by downcasting through the main materials ore shaft and upcasting through either or both of two 1828.8-mm by 335.28-m (72-in. by 1100-ft) bored vent shafts equipped with 2133.6-mm (84-in.) axivane reversible fans.

Approximately 165 m^3/s (350,000 cfm) of fresh air is required to ventilate the mine during the hot summer months. This large volume of air is necessary to maintain the velocity of the airflow through the mine high enough to prevent the hot humid air from producing fog as it cools. In winter months 118 m^3/s (250,000 cfm) is sufficient to ventilate the mine. Either or both vent fans can be reversed to prevent icing of the shaft in extremely cold periods.

Secondary ventilation is accomplished using 1016-mm (40-in.) axivane fans and vent tubing or ducting.

Ventilation doors are used to control the air flow within the various mine areas. The doors are activated automatically by vehicles passing through "electric eye" mechanisms as they approach the doors without halting.

MINE DRAINAGE

Mine pumping is at the rate of 0.1 m^3/s (1600 gpm) with a pollution control program for treating mine drainage. A hydrophobic, oil-absorbing material is used to collect oil floating on the water surface. A flocculent is used to settle out suspended particles to which oil may be attached. The flocculent is successful to the point that the silt removed from the sumps is not suitable for the mill flotation process because of the oil content.

The mine drainage and treatment system consists of two sumps in parallel. The alternate sump system permits diversion of all water to one sump so that the silt in the other can air dry and be removed to a mined-out part of the mine.

Treated mine water is pumped to the surface in one stage by a bank of eight pumps having a combined capacity of 0.37 m^3/s (6000 gpm). Mine drainage combines with other plant discharges and goes through the tailings pond and two settling ponds before being discharged into the local surface drainage system.

PERSONNEL

Training programs for the locally hired work force together with an attractive wage scale have resulted in a stable work force with low turnover. The following is a list of the required work force needed to maintain production:

Production miners	23
Development miners	10
Scalers	12
Service workers	16
Hoist attendants	3
Skiptenders	3
Crusher operators	3
Supply attendants	2
Shaft extra workers	3
Grader operators	2
Total	77

The mine department staff consists of 1 mine superintendent, 8 mine supervisors, and 5 engineering personnel.

In addition to these personnel in the mine department, there is a total of 3 supervisors and 21 hourly personnel in the maintenance department assigned to the three underground shifts. The underground shop is well equipped to handle all major maintenance needs.

SAFETY

Training for new employees begins with safety indoctrination. Safety consciousness is continually reinforced by a vigorous safety program supported by all levels of management. The result is that the Magmont operation has enjoyed an excellent safety record.

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CHAPTER 8

Sublevel Stoping

C. David Mann

INTRODUCTION

The sublevel stoping mining method is usually applied to a relatively steeply dipping, competent orebody, surrounded by competent wall rock. Ore is produced by drilling and blasting longholes, which can range from 50 mm (2 in.) to 200 mm ($7^7/_8$-in.) diam, with lengths up to 90 m (300 ft). Longholes can be inclined in any direction, but the ring or pattern usually forms a plane, and the holes are blasted as a unit. Recently developed mobile drilling and loading machinery, as well as new explosives products, blasting techniques, and cemented sand and rock fill have made sublevel stoping a highly efficient and versatile mining method.

In the design of a sublevel stoping production system, it should be kept in mind that production rates from conventional sublevel stopes vary widely throughout the life of the stope. Production is at a low rate early on, coming only from the drawpoints near the slot, but increases as new drawpoints are reached by the stope face. As the stope nears completion, again, fewer drawpoints are productive. Enough drawpoints must be available at any time to provide required production. Drawpoint availability should be compared to equipment availability; plan for more drawpoints than are needed at any one time. Accurate, realistic scheduling is essential to smooth production rates. Also, initial recovery of ore in a stope/pillar block is normally from 35% to 50% in sublevel stoping. Planning of pillar recovery, representing the majority of ore tonnage in a production block, must be done during early mine planning. Because much of the development already done for primary stoping (access for drilling, drawpoints, and haulageways), can be used for pillar recovery, early production from pillars is highly desirable. The following description of components of the system is an attempt to highlight some of the most important features and requirements of mechanized sublevel stoping methods. Similar comments would apply to the use of older equipment (column-and-arm drill setups, slushers, etc.) in similar methods. As in any

good mining system, maximum economic recovery of the resource in the ground is the primary consideration.

STOPE DESIGN CHARACTERISTICS

Length and Width

The following are some of the factors that affect sublevel open stope length and width dimensions: orebody geometry, principal stress directions, competence of stope back, optimum drill pattern, and drilling drift layout.

In new mines, initial stope layout design may occur before the orebody is actually intersected by mine workings. Stope dimensioning is a critical decision, and assistance from as many knowledgeable people as possible at this stage is essential. Operators with past experience in similar orebodies, rock mechanics experts, and others with mine design experience should participate at this stage of stope planning.

Height

The following are some of the factors that must be considered in determining stope height: competence of stope pillar and stope/fill walls; slenderness ratio of adjacent pillars; orebody dip; orebody thickness; hole depth capability of the drilling machine; fragmentation characteristics of the ore; and level intervals in existing mines.

In competent ground, drill-hole length and accuracy are the most important determinants of stoping height. Frequently entire drilling sublevels can be eliminated because of the depth capability of sophisticated drilling equipment, resulting in significant development cost savings.

Drawpoint Location and Design

Some of the most important considerations of a good drawpoint system are optimum spacing of drawpoints, within the constraints of stope dimensions, for uniform drawdown and maximum recovery; excavations designed for stability for the life of the ore block to be drawn (primary stope ore as well as subsequent pillar ore); floor or roadway design including type of surface, reinforcing, grade for water runoff; orientation with respect to the main haulageway, for optimum loader maneuverability and ground stability at the intersection; and length, to allow articulated front-end loaders to work in a straight configuration.

Careful drawpoint design and construction are keys to successful production. Extra care in development, such as smooth wall blasting, rockbolts or grouted rebar, wire mesh, and shotcrete usually will ensure long drawpoint life. Human exposure during production loading is of longer duration than during development or production drilling, and consequently preparation of drawpoints is easily justified, particularly when pillar ore can be drawn through the same drawpoints. Secondary blasting of boulders can weaken drawpoints, also justifying good ground control techniques. A smooth drawpoint floor of

poured, reinforced concrete, on a grade of +3% or +4% toward the ore pile facilitates water flow out of the drawpoint, and ease of loader bucket penetration into the muck pile.

Slot Raising, Slotting

A slot or other space for rock expansion is necessary in conventional sublevel stoping where vertical rings or rows of holes are blasted. The slot can be started at a slot raise driven by conventional raising methods, raise boring, drop raising (predrilling and blasting a raise from the top, using small diameter—less than 200-mm ($7^7/_8$-in.)—holes for relief), or crater blasting (similar to drop raising, but without relief holes). The slot usually extends from the extraction level to the back of the stope. It is normally expanded to full stope width by longhole slashing and should be 4 to 5 m (13 to 16 ft) wide.

The vertical crater retreat (VCR) method eliminates the need for a full-stope-height slot; a slot to provide relief for the undercut only is needed, and can be excavated from the undercut level if desired. No slot is necessary in the case described in the next section where the undercut is eliminated.

Undercutting

Proper undercutting is essential to the success of conventional sublevel stope production because of its effects on production blasting and loading efficiency. Undercutting normally should not be carried out more than a few rings in advance of production blasting, mainly to ensure undercut back stability.

Mobile fan drill jumbos are available for undercutting, capable of drilling uphole fans or rings to 25-m (80-ft) depths, at the proper inclination off the vertical plane for safety and optimum blasting efficiency.

In the VCR method, undercutting is optional but preferred. Especially in a narrow stope, the undercut can be eliminated and the stope blasting by VCR can be started at a central drift connecting the drawpoints to the main haulageway.

Production Drilling

Factors to be considered in the design of production drilling patterns, equipment, and drilling locations include drillability of the ore, considering hardness, abrasiveness, fractures, vugs, and specific gravity; expected fragmentation characteristics; type and accuracy required of the drilling machinery; orientation and spacing of drill holes; hole size; and hole length.

Innovation in production drilling equipment has improved the efficiency and expanded the applications of sublevel stoping more than any other factor. Since the early 1970s the column-and-arm longhole drilling method, commonly using 50- to 75-mm (2- to 3-in.) diam holes, has largely been replaced by mechanized longhole drilling. Standard percussion longhole drills are still being used on mobile carriers, but the most efficient methods employ high-pressure pneumatic down-the-hole drills, and electric-hydraulic rotary and

rotary-percussion equipment, capable of drilling a range of hole sizes up to 200 mm (8 in.). Holes can commonly be drilled to depths of 90 m (300 ft) with less than 2% deviation with down-the-hole equipment. Best stope drilling and blasting efficiency result when a pattern of vertical, parallel holes can be designed.

An excellent starting point for drilling pattern design is to select a powder factor, in kilograms (pounds) of explosive per ton (short ton) of ore, and apply it to the block of ore to be produced, using various combinations of hole size, spacing, and explosive density. Experience in similar ore types or mining conditions is important in estimating initial power factor.

Production Blasting

The type of explosive selected is determined by all of the following factors: powder factor, kilograms (pounds) of explosive per ton (short ton) of ore, required by design; explosive density; explosive detonation velocity; orientation and inclination of blastholes; and accessibility to the hole collar for loading explosives (charging).

Initiation of the blast can be made by electric or nonelectric blasting caps and regular or millisecond delays.

Nonelectric caps would be used where danger of premature detonation from stray currents or static charges exists. Delay types have a considerable effect on blast vibrations and ore fragmentation.

The maximum size of a production blast is affected by potential for damage to adjacent mine workings or other structures, maximum allowable weight of explosive per delay, and amount of void space in the stope.

In sublevel stoping, explosive types can be conventional dynamite, slurry, water gel, ammonium nitrate–fuel oil (ANFO) mixtures, or combinations of each. A variety of initiation techniques can be applied. The VCR method requires very specialized blasting techniques. Explosives technology is advancing rapidly, and many specialized products are now available. Technical discussions with explosives specialists should be held prior to final sublevel stoping method design.

The following chapters of this section demonstrate the versatility of sublevel stoping methods, illustrated by their use in a variety of orebodies throughout the world.

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CHAPTER 9

Considerations for Sublevel Stoping

Brian W. Lawrence

INTRODUCTION

Sublevel stoping generally is a large-scale open stoping method. It sometimes is referred to as *longhole* or *blasthole stoping*. This method usually is applied to strong orebodies that require minimal support and are surrounded by strong country rock. The orebody should be fairly regular in shape and well defined. The dip of the footwall normally should be sufficient to allow broken ore to gravitate freely, although the method has been adapted successfully to mine some of the flatter orebodies. Sublevel stoping is not dependent on the width of the orebody. Widths of less than 6 m (20 ft), however, make the utilization of longhole drilling techniques more difficult.

The main criteria for sublevel open stoping are competent ore and stable host rock, regular ore boundaries, and a footwall dip that exceeds the angle of repose of broken ore.

Basically, the method entails providing access to the orebody at various subintervals between the main haulage levels in order to drill and blast the intervening ore. Stope drilling is carried out from drilling drifts on the sublevels, and the ore is blasted in slices toward an open face, which generally is vertical on the down holes and may be inclined toward the open face for the up holes. The blasted ore gravitates to the bottom of the stope and is collected through drawpoints. Figure 9.1 illustrates a typical sublevel stoping mine.

The method requires extensive orebody development with relatively high capital expenditures. However, much of the development is in ore, and production costs are comparatively low. Productivity rates are in the 13.6 to 27.2 t (15 to 30 st)/worker-shift range. The drilling, blasting, and loading operations are performed independently, and equipment utilization is high. Large outputs can be obtained with few units and limited personnel. Dilution with

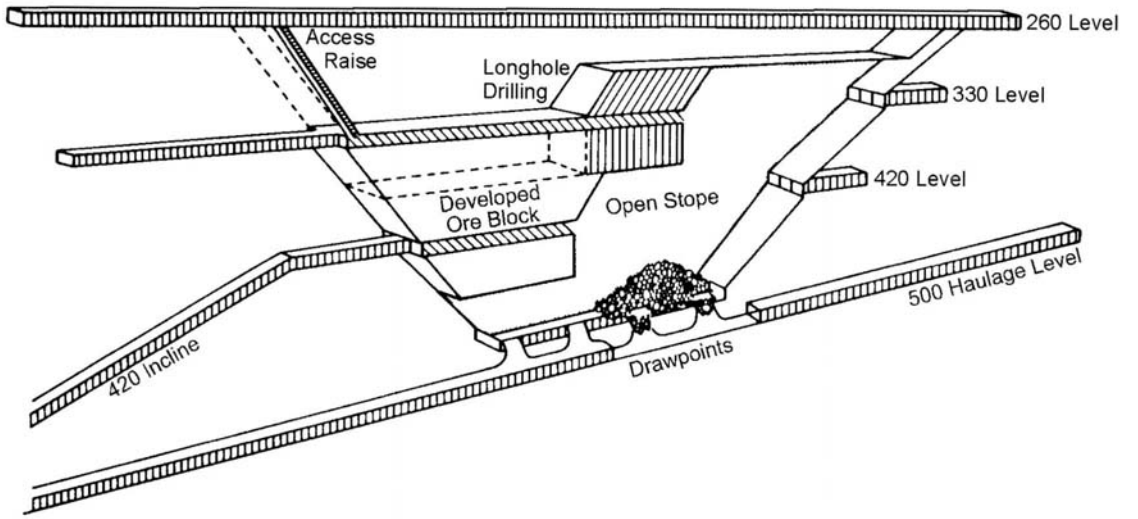


FIGURE 9.1 General view of an open stoping mine (Moore and Taylor, 1973)

waste rock may occur if ore boundaries are irregular or if caving occurs, but 100% of the ore within the stope usually is recovered. Pillar recovery sometimes is a problem. It is at this point that most of the waste dilution occurs.

In 1973, the sublevel open stoping method was used by more than 21% of the metallic ore mines in the United States [9 out of 42 mines producing 1088+ t/day (1200+ stpd)] and provided more than 12% of the total metallic ore production from these mines.

DEVELOPMENT

Access to a sublevel open stope mine is gained by a stope or shaft that is normally sunk in the footwall of the ore zone, away from any possible effects from blasting or other production operations in the stoping process. The main level interval is selected and usually varies from 45.7 to 121.9 m (150 to 400 ft), depending on the vertical extent of the orebody, the number of drawpoints required to maintain output, and the eventual stope height.

The main haulage drifts are located at the bottom of the stope horizon, either directly in ore or in the footwall, with access crosscuts at intervals. Raises are driven up through the ore to connect to the level above and provide access and ventilation to the sublevel drilling drifts, which are driven horizontally in the ore for the length of the stope.

Depending on the drawpoint system to be used, the stope is either undercut by a raise-and-cone system, from a scram or slusher drift, or simply provided with a drilling drift from which access is made at intervals to the main haulage level.

A raise driven either at the end or in the middle of the stope (if stope access is from both ends) provides a start for opening up a vertical slot across the stope to establish a free or open face for longhole drilling and blasting.

Sublevel stoping has proved to be a relatively safe and economical system.

PRODUCTION TECHNOLOGY

Production mining in the sublevel system is achieved almost exclusively by longhole drilling, with ring or fan drilling from the sublevels being the most common method. Where the orebody is narrower, the sublevel drifts may be slashed out to the full ore width, and thus provide a base for drilling parallel holes. The length of the holes depends on the shape of the orebody and the predetermined sublevel spacing. Longholes do not normally exceed 25 m (80 ft) because hole deviation and control become major problems beyond this length. Hole burden and spacing usually depend on the degree of fragmentation required, and can range from 1 m (3 ft) for narrow slot drilling to 2 to 3 m (5 to 10 ft) for regular production drilling. The fragmentation required depends on the size and type of ore handling equipment to be used. Scraping from slusher drifts into a chute or mine car requires good fragmentation, to avoid chute hang-ups and secondary blasting. Drawpoint systems using load-haul-dump (LHD) equipment can handle large pieces, especially if underground crushing facilities are available to crush the ore to a more manageable size for hoisting to surface.

The average cost-per-ton for the different methods of mining varies appreciably. When the many parameters that can influence an individual mine's optimum productivity or performance are considered, it becomes apparent that "average" cost figures are, at best, a very broad gage statistic.

Figure 9.2 shows the distribution of mining costs at one trackless sublevel stope mine.

SUBLEVEL STOPING AT MINE A

General

A 447.36-kW (600-hp) double-drum hoist operates 5.4-t (6-st) skips in a 609-m (2000-ft) shaft, and produces 1632.6 t/day (1800 stpd) of iron ore for a 5-day week. The mine employs 10 salaried and up to 107 hourly rated personnel, and has an overall productivity rate of 14.5 t (16 st) per worker-shift. Total underground productivity, including supervisory personnel, averages 20.8 t (23 st) per worker-shift. The mine produces for 10 shifts per week, with six stopes in production and two under development.

Haulage

Two methods are used to haul ore to the shaft. Most of the ore is drawn from chutes and then hauled by trains for an average distance of 640 m (2100 ft) to a dump point. At this point, the ore is transferred to a belt conveyor that moves it another 290 m (950 ft). A minor amount of ore is pulled from chutes directly to collecting belts that feed the main 290-m (950-ft) belt.

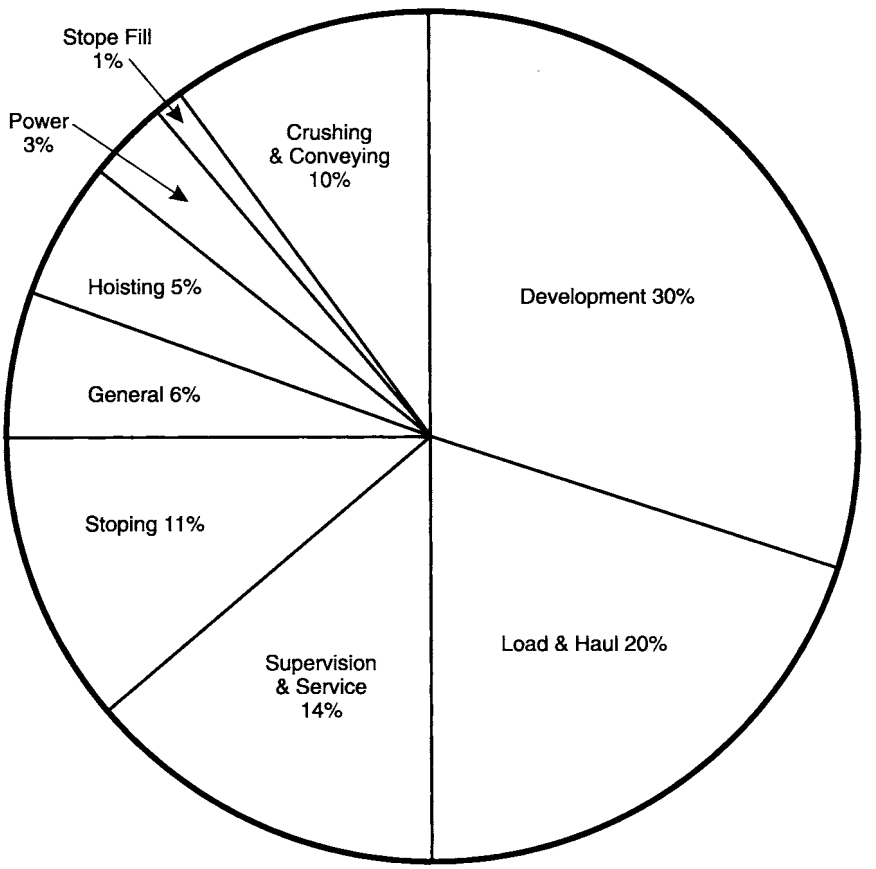


FIGURE 9.2 Mining cost distributon for one trackless sublevel open stope mine

Trains consist of five 3.11-m^3 (110-cu ft) Granby-type cars pulled by a 9.07-t (10-st) locomotive. Cars are emptied by running them over a camel back at the dump point. The bin at this point has the capacity for one trainload of ore.

Ore is loaded onto the 914-mm (36-in.) main conveyor belt by a hydraulically operated feeder. The system's secondary gathering-feeding belts are 760 mm (30 in.) wide and range from 15 to 76 m (50 to 250 ft) in length. The conveyor system is operated by one person located at the shaft, who also is responsible for keeping both the car tally and the skip tally.

Two trains per shift, two shifts per day, handle this mine's production, except for the minor volume of ore transported by the collecting conveyor belts.

Stope Development

The stopes are developed on 9.7-m (32-ft) floor-to-floor sublevel intervals, and have subdrifts that are 1.2 × 1.8 m (4 × 6 ft) in cross section and driven through the middle of the stope area from the manway raise, on the hanging wall, to the slot raise on the footwall of the ore.

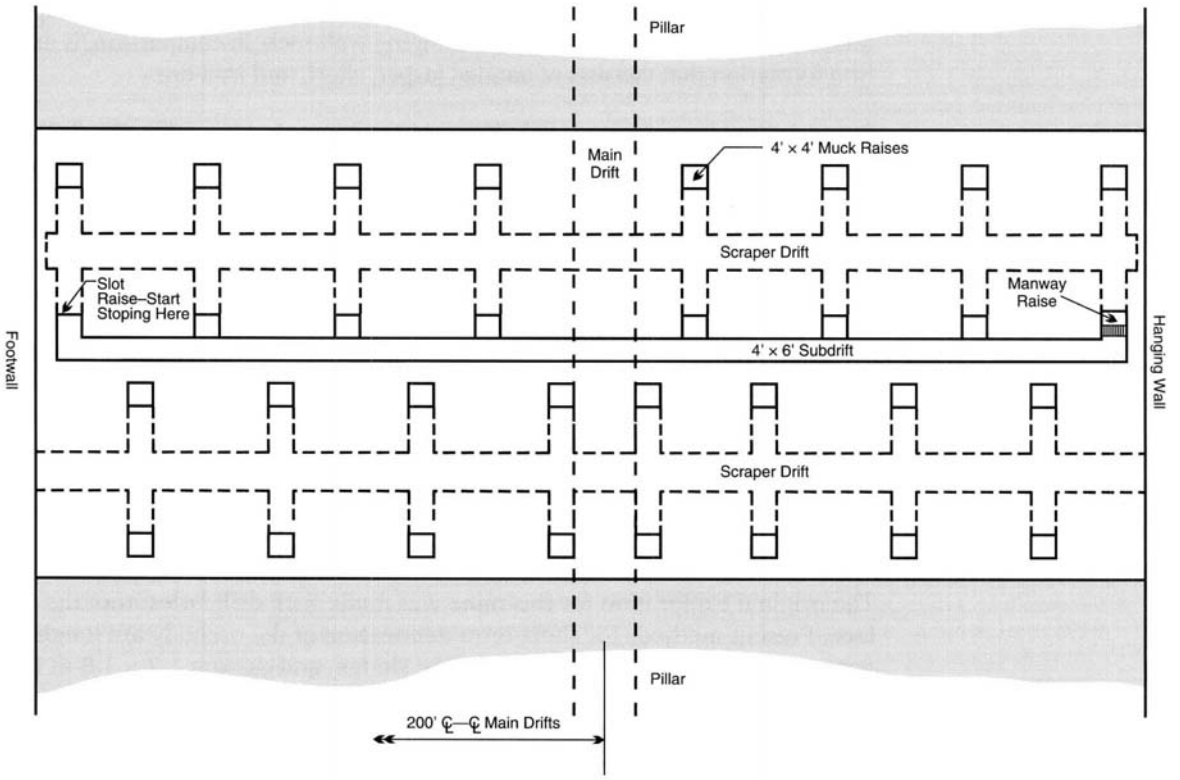


FIGURE 9.3 Mill-hole level, mine A, metric equivalent: ft × 0.3048 = m

All raising is done by conventional methods. Raise cross sections are 1.2 × 1.2 m (4 × 4 ft), and most raises are driven in 7.9-m (26-ft) lifts. Raises are offset on each sublevel. Stopers are used to drill off 1.2-m (4-ft) rounds.

Scraper or slusher crosscuts with 2.1 × 2.7-m (7 × 9-ft) cross sections are established below the first sublevel (two per stope), about 15 m (50 ft) above the main haulageway, which runs at right angles along the strike of the ore. Mill-hole raises are driven from each side of the scraper crosscuts to the elevation of the first sublevel.

Stope Mining

Mining begins in the middle of the stope on the first sublevel above the scraper crosscut, and at the last mill raise adjacent to the footwall contact (see Fig. 9.3). Mill raises are belled, and slash rounds are taken around the mills to develop an undercut. After an adequate area has been opened on the first sublevel, uppers are taken around the slot raise toward the sublevel above. The miners then move up to the next sublevel and start slashing around the slot raise. After sufficient room has been opened on the second sublevel, stope holes (down holes) are blasted to accommodate broken muck. Uppers then are taken toward the next sublevel, and the process is repeated. Retreat usually is taken from the footwall toward the hanging wall. The footwall consists of graphitic slate,

which will cave when exposed. (In addition, sulfur in this slate will ignite spontaneously when exposed to air.) The hanging wall rock, in comparison, is an iron formation that consists of banded jasper, chert, and iron ore.

Stope drilling is performed with 76-mm (3-in.) bore jackleg drills. For drilling stope holes, the drill is removed from the leg. Hole size is 51 mm (2 in.), and 1.8-m (6-ft) rounds are used in slashing, with 4.6- to 5.5-m (15- to 18-ft) holes used for stope holes and uppers. The miners select the pattern for blasting. The transverse stopes are 24 m (80 ft) wide (along the strike), about 61 m (200 ft) long (depending on width of orebody), and 61 m (200 ft) high. Ventilation is accomplished by diverting part of the main airflow through the stope.

Productivity (including slushing, development, and stope mining) was 35.6 t (40.3 st) per worker-shift in 1971. It increased to 41.5 t (45.6 st) per worker-shift in 1972. Productivity for stope mining only was 53.6 t (59 st) per worker-shift.

Grade Control

The original exploration for this mine was made with drill holes from the surface. Present methods for short-term delineation of the orebody are longhole percussion drilling, with sampling of the sludge, and driving 1.2 × 1.8-m (4 × 6-ft) drifts to the contacts. Subdrifts are channel sampled to establish stope grade. Grab samples are taken from each workplace every day, and the dumptor operator, on the surface, takes a grab sample for mine grade.

There is a distinct color variation between the ore and waste in this orebody. The contact between the red ore and black slate is clearly marked. The iron formation in the hanging wall also is a reddish color, but this waste shows distinct chert bands.

SUBLEVEL STOPING AT MINE B

General

Using a 447.36-kW (600-hp) friction-type fully automatic hoist, Mine B hoists some 8163 t/day (9000 stpd) from a depth of 370 m (1200 ft) in 10.9-t (12-st) skips. Operating 16 shifts per week, the mine produces 1.64 million t (1.8 million st) of iron ore per year. There are 8 producing stopes and 19 under development.

This mine employs 23 salaried and 206 hourly rated people underground, with a productivity of 35.7 t (39.3 st) per worker-shift.

The original discovery of deep ore at this site was made by an aeromagnetic survey. There had been old workings at this mine, dating from about 1860, but the old mine had been abandoned after the surface ore was depleted. The anomaly located by the aerial survey was diamond drilled on a 61-m (200-ft) square grid, which was considered adequate to define the orebody. At the time of this exploration, equipment to survey drill holes in magnetic ore did not exist. Subsequently, the mine operators learned that the test holes had

deviated considerably, producing inaccurate definition and affecting the initial mine planning. The grid drilling pattern had been inadequate for accurate delineation of the orebody's very irregular configuration.

A continuing program of underground diamond drilling is being used to define areas for near-term development. Much of this work is done to locate contacts that were not defined in the surface drilling phase.

Haulage

This trackless mine uses 12 3.82-m^3 (5-cu yd) and two 7.64-m^3 (10-cu yd) capacity LHD units to haul stope production from drawpoints spaced on 15-m (50-ft) centers on the main haulage levels to a transfer orepass system that feeds a 1066 × 1219-mm (42 × 48-in.) jaw crusher on the fourth level. Tramming distances range from 91.4 to 243.8 m (300 to 800 ft) and will increase as the mining advances. The roadbeds are paved with 19-mm (¾-in.) crushed rock, and are maintained with a grader.

A tramming crew of 20 people handles the daily production. Availability of the LHD units is about 75%.

Haulage drifts are connected by ramps and are driven 5.5 m (18 ft) wide by 4.2 m (14 ft) high, using three machine jumbos to drill 3-m (10-ft) rounds.

Stope Development

Sublevels consisting of rooms 4.2 m (14 ft) high by 8.2 m (27 ft) wide, which are driven off the mine's ramp system on 20-m (65-ft) floor-to-floor intervals, are established (see Fig. 9.4). Development jumbos with three booms are used to advance the rooms. It requires 67 holes to pull a 3-m (10-ft) round, and rounds are drilled 3.6 m (12 ft) in length.

Ammonium nitrate–fuel oil is used to blast rooms, and powder consumption is 111.5 kg/m (75 lb per ft) of advance. Blasting is performed with electric blasting caps. The jumbo is operated by a two-person crew, and advance is 1.3 m (4.5 ft) per worker-shift. Blasted rooms are mucked with LHD machines.

Stopes in this mine are 30 m (100 ft) wide and are bounded by 9-m (30-ft) nonrecoverable pillars. Three rooms, separated by two 2.7-m (9-ft) temporary pillars, make up one stope area. This type of mining allows for many headings, which are necessary for efficient utilization of trackless mining equipment. Another advantage is that this method requires little development work to prepare a stope.

Slot raises 1.8 × 1.8 m (6 × 6 ft) in size are driven from sublevel to sublevel to provide a breaking point for starting the stope. These raises are driven by drilling 76-mm (3-in.) down holes around a 152-mm (6-in.) burn hole. The burn is blasted through in 2.4-m (8-ft) lifts from the bottom up, and the raise is slashed out to full size, using 76-mm (3-in.) holes. Most pilot raises are 15 m (50 ft) long. Raises are drilled out with a 114-mm (4.5-in.) drifter mounted on an air-track carrier. One raise can be blasted through in about a week.

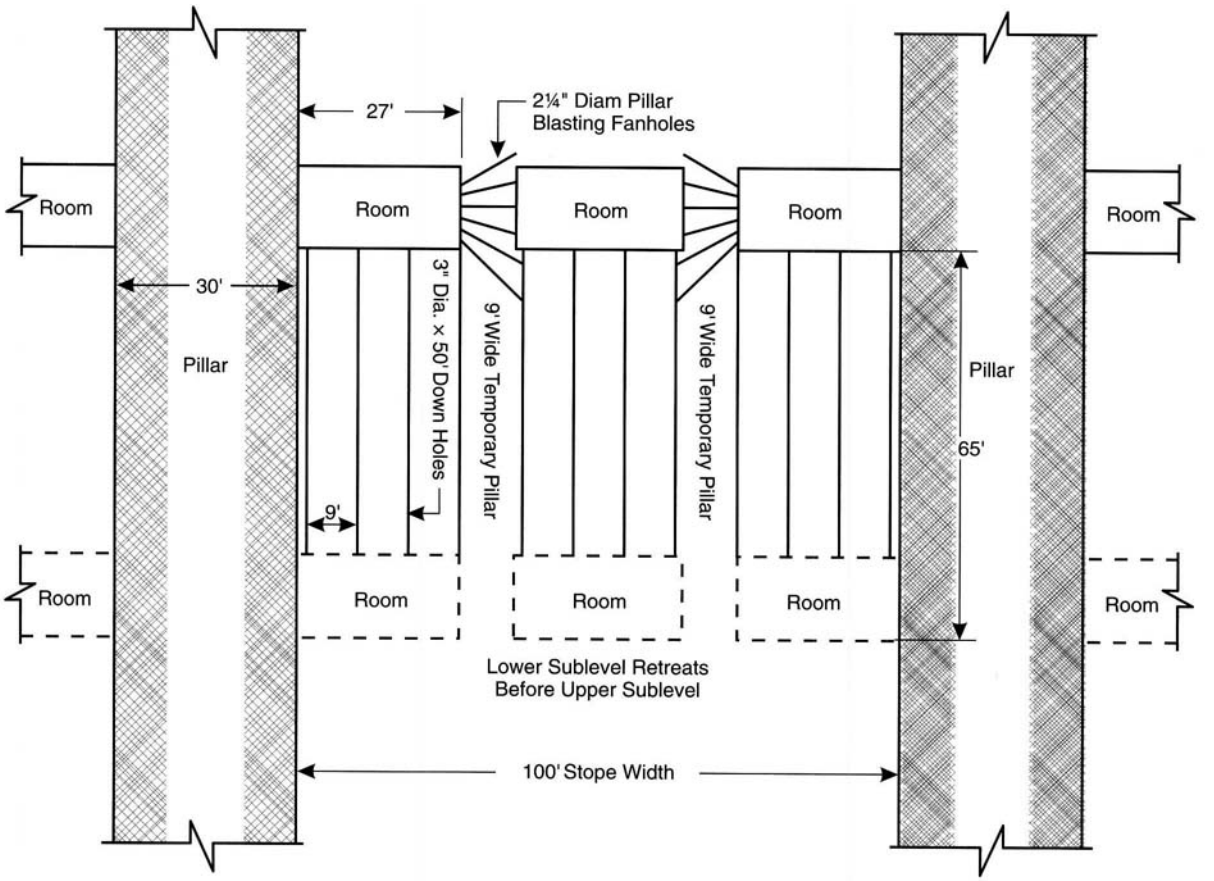


FIGURE 9.4 Stope cross section, mine B, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m

Stope Mining

After rooms are driven to the back of the stope, and pilot raises are driven, mining begins on the retreat toward the ramp system. Four 76-mm (3-in.) downholes are drilled in each room on 2.7-m (9-ft) centers and with 2.4 m (8 ft) of burden, using an air-track drill. In this mine's extremely hard ground, productivity is 27.7 m (91 ft) per worker-shift, with one person operating the drill.

The 2.7-m (9-ft) pillars are wrecked with 57-mm (2.25-in.) fan holes (see Fig. 9.4). Fan drilling productivity is 30.4 m (100 ft) per worker-shift. Holes are charged with ammonium nitrate slurry in the downholes and ANFO in some of the fan holes. Blasts are timed with electric blasting caps connected to detonating cord. The powder factor in stope blasts is 0.23 kg (0.5 lb) per ton of ore. It is necessary to limit the amount of powder per delay and per blast to control vibration and concussion effects in the hard rock. Explosives are limited to about 453.2 kg (1000 lb) per delay and from 4535.9 to 6803.8 kg (10,000 to 15,000 lb) per blast.

Production blasts are made between shifts when there are no personnel in the mine. There have been complaints from local residents about surface vibration during blasting operations. Monitoring by the U.S. Bureau of Mines has shown that these vibrations are not harmful.

Primary ventilation air is supplied through the mine's 4.5-m (15-ft) diam vent shaft at the rate of 165.18 m^3/s (350,000 cfm) at 156 mm (6 in.) water gage (WG). A 186.4-kW (250-hp) electric motor drives the main fan. Axial vane fans are used to distribute secondary air in the working areas. Primary air is heated in winter. In addition, all diesel-powered equipment used in this mine is equipped with either catalytic or water-bath exhaust scrubbers.

The mine requires virtually no ground support, but there are some problems with rock bursts, and scaling is a continual problem. A crew of 33 scalers is employed to control loose rock. Hand scaling, a time-consuming process, has proven to be the only practical way to solve this problem.

Grade Control

Grade control is accomplished by drilling holes up to 3 m (10 ft) long with production drills. These holes then are probed with a magnetic susceptimeter to determine the magnetic iron. Cuttings from percussion drill holes are run through the Davis tube, and the sludge also is analyzed for silicon oxide (SiO_2) content, a factor in determining grindability of the ore. The correlation coefficient between SiO_2 and grindability is 0.96. Cutoff grade is based on grinding economics and the percent of magnetic ore.

SUBLEVEL STOPING AT MINE C

General

Hoisting is accomplished by two 1677.8-kW (2250-hp) friction hoists and 17.2-t (19-st) skips, which operate in a 731.5-m (2400-ft) shaft and produce 9614.2 t/day (10,600 stpd) of iron ore. Operating three shifts per day, six days per week, this mine produces some 2.7 million t (3 million st) of ore annually. In addition, 45,350 t/a (50,000 stpy) of waste rock are hoisted through another shaft.

Underground personnel total 400, including salaried employees, and total underground productivity is in the 24 t (26.5 st) per worker-shift range.

The original target for drilling at this mine was outlined by a U.S. Geological Survey (USGS) magnetometer survey. The mining company then drilled a 579.1-m (1900-ft) hole within the anomaly area and hit iron ore. The orebody was further outlined by drilling 19 holes from the surface, 12 of which hit ore. The next exploration step was to drive an exploration drift from a shaft, at a depth of 510.5 m (1675 ft) below the surface. From this level, diamond-drill holes were drilled on 60.9-m (200-ft) centers. This drilling was the basis for the estimate of ore reserves. (No short-range drilling is being done at this mine to develop reserves.)

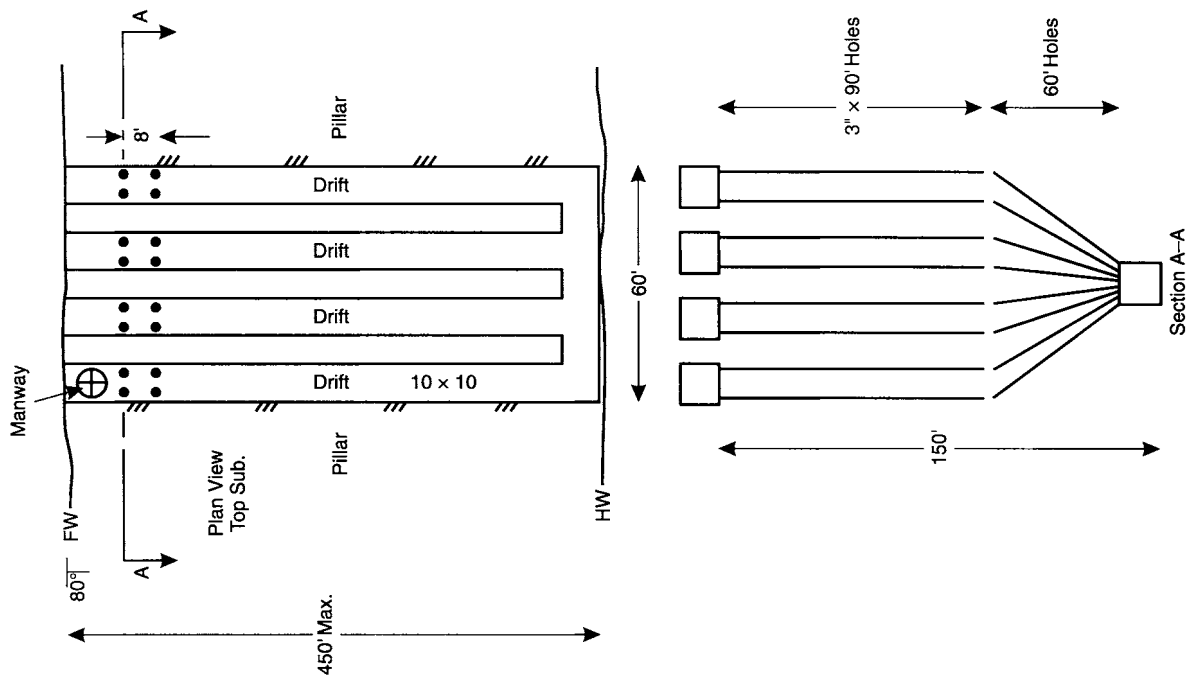


FIGURE 9.5 Open stope plan and section, mine C

Ore was mined initially by open-stoping methods because of the proximity of several known aquifers, which, if breached by a caving method, would create high water inflows. However, subsequent studies revealed that this was not the case, and the mine is now changing over to a sublevel caving method to take advantage of the higher degree of mechanization possible with this method. Late in 1972, about 40% of the production was from four sublevel caving stopes, with the balance obtained from four sublevel open stopes, pillar recovery, and development. At this time, the development of eight caving stopes, four sublevel open stopes, plus five pillar recoveries, was providing 13% of the ore tonnage. (See Fig. 9.5.)

Haulage

The mine's original method used 74.5-kW (100-hp) slushers, pulling 1371-mm (54-in.) scrapers, which slushed the ore into six-car trains of 7.07-m^3 (250-cu ft), 18.1 t (20 st) Grangesberg cars. The loaded trains were pulled to the shaft by 9.9-t (11-st) trolley locomotives, on a 914-mm (36-in.) gage track of 31.7-kg (70-lb) rails. When the train was pulled through the car dump, the Grangesberg cars discharged from the bottom. With a 304.8- to 365.7-m (1000- to 1200-ft) haul, train productivity was 12 or 13 trips per train per shift.

Ore from the car dump went into transfer raises, which, in turn, emptied into slusher trenches at the shaft. The ore was slushed into loading pockets with 111.8-kW (150-hp) slushers pulling 1828-mm (72-in.) scrapers.

In 1972, only about 4% of the ore produced was being handled by slushers in slusher drifts. This method is being phased out in favor of LHD machines

loading in drawpoints. Trains still are used in slusher drift loading and in some lateral transfers, but they also are being phased out in the deeper levels of the mine.

The new method uses 6.1-m^3 (8-cu yd) capacity LHD machines for loading and hauling. Because of the ore's higher density, however, these units are equipped with 4.9-m^3 (6.5-cu yd) buckets.

On the bottom level, the LHDs haul and dump directly into a 1066-mm (42-in.) gyratory crusher. Ore also is drawn into this crusher from transfer raises that lead to the sublevel caving, sublevel open stoping, and pillar recovery operations on the upper levels. After crushing, the ore travels up an inclined 1066 mm by 533.4-m (42 in. by 1750-ft) conveyor to two 4535-t (5000-st) storage bins. It then moves from these bins to loading pockets at the shaft. On the bottom level, with a 304.8-m (1000-ft) haul, LHD productivity is 181.4 to 226.7 t (200 to 250 st) per machine shift.

Stope Development

Main haulage drifts have been established outside the ore at vertical intervals of 45.7 m (150 ft). Transverse open stopes 18.2 m (60 ft) wide, up to 137.1 m (450 ft), long, and 45.7 m (150 ft) high are developed with pillar widths varying from 18.2 to 21.3 m (60 to 70 ft), depending on ground conditions. A 3.6 × 3.6-m (12 × 12-ft) undercut crossing is driven from the main haulage across the centerline of the stope to provide a drill base at the bottom of the block to be mined, and is connected to the loading crosscut by drawpoints. At the level above, there are four 3 × 3-m (10 × 10-ft) crosscuts that run across the orebody and parallel to the lower crosscut. A raise is driven, usually on the footwall end of the stope, to provide a slot from which to start mining.

Stope Mining

The slot raise is first opened up for the full width of the stope. The bottom sublevel then provides a base for drilling 76-mm (3-in.) up holes in a fan pattern to a vertical height of 18.2 m (60 ft). Toes of the holes are spaced at 2.5 m (8.5 ft), and ring burden is 1.2 m (4 ft). Downholes from the upper sublevel are drilled in the same vertical plane as the up holes. Eight holes, two per heading, are drilled about 28.3 m (93 ft) long. Percussion drills with a 114-mm (4.5-in.) bore are used. Productivity amounts to 163.2 to 181.4 t (180 to 200 st) drilled per drill shift, at a rate of 15 t/m (5 st per ft) of hole drilled.

Holes are loaded with ANFO or cartridged slurry, depending on the amount of water present, and are initiated by detonating cord.

Pillars are recovered by longholes reamed to 127 mm (5 in.) and blasted into adjacent mined-out stopes.

The mine planning engineer is responsible for scheduling the drilling, blasting, and mucking of all open stopes.

Grade Control

One diamond drill is being used for grade control only. Mined grade is within 0.1% of estimated grade.

On a daily basis, the mine planning engineer decides which drawpoints need a heavy, normal, or slight draw to keep the pull uniform, and to prevent loss of ore. Draw instructions are posted daily in the foreman's office. Crew foremen report the actual draw for each workplace at the end of each shift.

OTHER MINES

Sublevel open stoping is used worldwide as a mining method. The following are examples of interest. The reader is referred to the bibliography for references that provide more complete descriptions.

Pyhasalmi Mine, Finland

Although presently mined by sublevel open stoping, the Pyhasalmi mine (Anon., 1972a) will be converted to sublevel caving and shrinkage methods at depth. In spite of rockbolting on the stope walls, dilution in the sublevel stopes is 15%.

The sublevel interval is 21.3 m (70 ft), with blastholes drilled up and down in a ring pattern. These 51-mm (2-in.) diam holes vary in length from 14.9 to 29.8 m (49 to 98 ft). Access is by means of a 4.5 × 3.6-m (15 × 12-ft) ramp driven on a 14% grade. The ramp floor is paved with a 76-mm (3-in.) asphalt-concrete surface over a macadam bed.

Broken ore is transported by 3.8-m^3 (5-cu yd) LHDs or a converted front-end loader. Productivity for the LHDs, with an average tram of 110 m (361 ft), is 131.5 t/hr (146 stph); the front-end loader, working on a 79.8-m (262-ft) average haul, produces 139.6 t/hr (154 stph).

Dannemora Mine, Sweden

The low-grade orebody of Dannemora mine (Anon., 1969) is developed by driving parallel drilling drifts along both contacts. Half fans then are drilled toward the center of the stope. The broken ore falls into cone-shaped raises and is drawn into slusher drifts.

Sublevel interval averages about 19.8 m (65 ft), and drilling drifts have 9-m^2 (97-sq ft) cross sections to accommodate longhole drills (see Fig. 9.6). Fans are drilled with 1.5-m (4.9-ft) burden and 3.0-m (9.8-ft) maximum toe spacing. Rings consist of 10 to 14 holes ranging from 20 to 26.5 m (66 to 87 ft) in length. Most of the drilling is done with a rig that mounts two 114-mm (4.5-in.) bore drifters with independent rotation. Productivity with these machines is 100 to 120 m (328 to 394 ft) per worker-shift. See Fig. 9.7.

The rings are loaded by a two-person crew, which charges from 300 to 400 kg (660 to 860 lb) of high explosives per shift. Loading is performed with pneumatic loaders. Usually, 3 to 15 rings are shot at one time, producing from

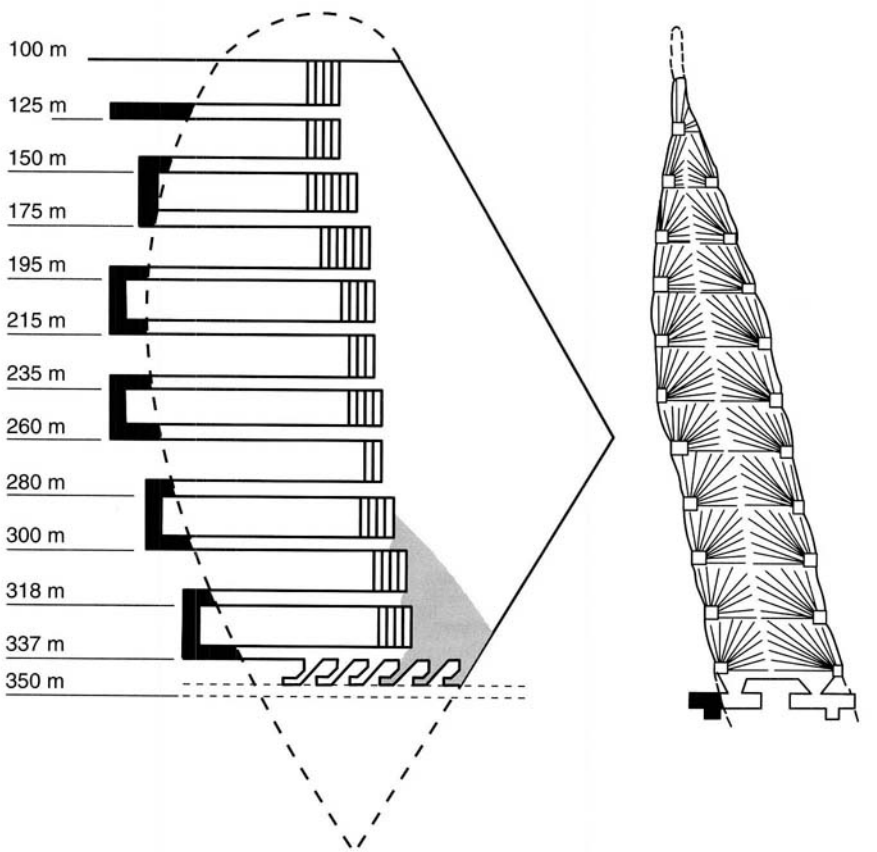


FIGURE 9.6 Longitudinal and cross sections, Dannemora mine

99,770 to 149,655 t (110,000 to 165,000 st) per blast. Powder factor ranges from 0.09 to 0.11 kg (0.20 to 0.25 lb) of explosives per ton of ore.

Slushing is performed with 55.9-kW (75-hp) electric-powered units that pull 1600-mm (63-in.) scrapers. Slushing productivity ranges from 299.3 to 399 t (330 to 440 st) per worker-shift.

Tramming is accomplished with trains consisting of nine 3.6-m^3 (127-cu ft) cars and a 10-kW battery-powered locomotive. The slusher operator, using radio control, spots the loco. These trains are taken to a siding where they are connected to large (20 kW) locos that take them to the crusher.

This mine is installing a new system that will use ten 5-m^3 (177-cu ft) cars and two 15-kW locos, one at each end of the train. This new system is being designed to operate automatically between certain points.

Bellefonte Mine, United States

The orebody of Bellefonte mine (Gustafson and Pinto, 1964) is a high-calcium limestone, averaging 21.3 m (70 ft) in thickness, and the main haulage level is 210.3 m (690 ft) below the surface. Development is along the strike, and rings

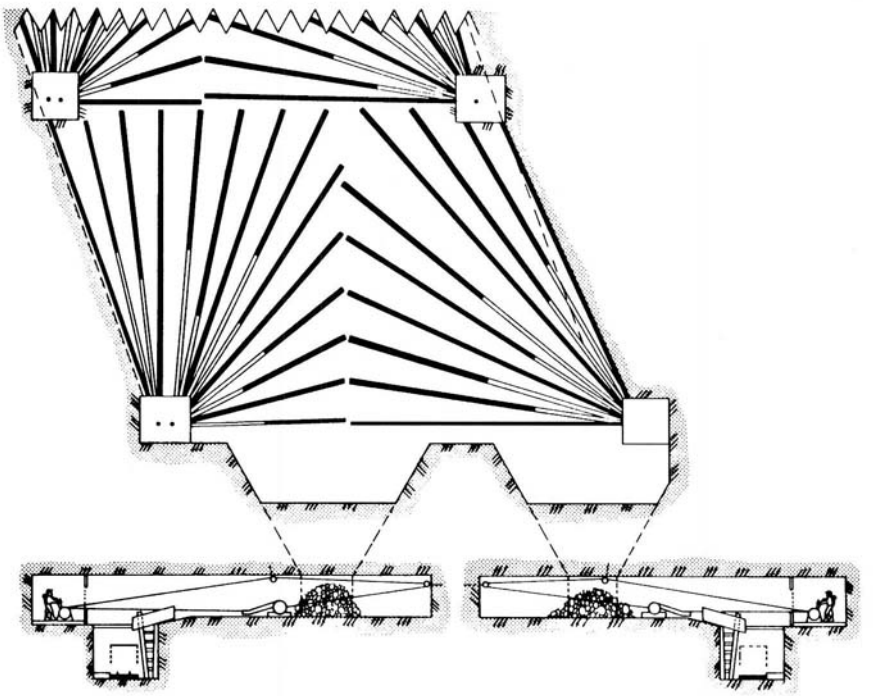


FIGURE 9.7 Slusher drift cross section, Dannemora mine

are drilled downward from the horizontal. Sublevel intervals range from 42.6 to 51.8 m (140 to 170 ft). Stoping retreats along the strike, with rings being drilled in a plane normal to the strike. See Fig. 9.8.

An interesting feature at this mine is that the broken ore is loaded from drawpoints by means of 1.5-m (5-ft) pan feeders (Fig. 9.9). These feeders deliver the ore to trains in the main haulageway, which runs along the strike of the orebody. The previous method of loading from the drawpoints utilized gathering-arm loaders, which were moved from one drawpoint to another.

Use of pan feeders in the drawpoints has reduced operating costs appreciably: labor, 22%; maintenance, 55%; explosives for secondary blasting, 30%; power, 83%; and lubrication, 86%. The net reduction in overall loading costs was 40%.

Geco Mine, Canada

Sublevel open stopes at Geco mine (McLeod and Schwartz, 1970) are backfilled with quarried rock from the surface. The backfill follows the top of the ore down as the stope is worked out. In addition, the tops of the stopes are supported by a system of cable bolts that forms an arch of rock. It was found

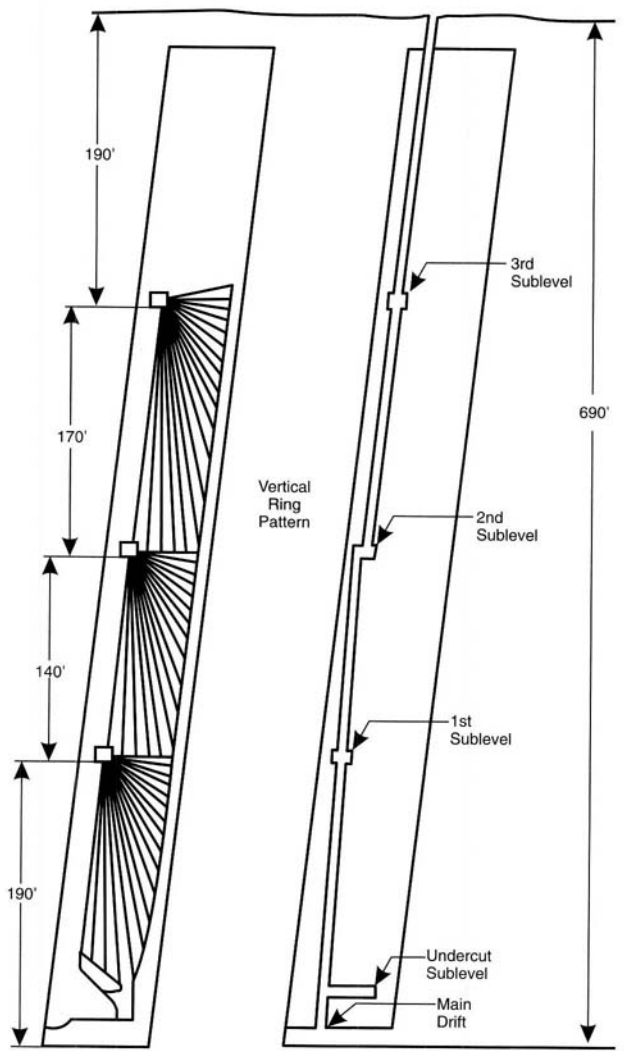


FIGURE 9.8 Cross sections, Bellefonte mine

that the rock fill could be stabilized by introducing a slurry of mill tailings and cement. The slurry filled the interstitial voids and formed an artificial conglomerate that would stand while pillars were being mined.

Sullivan Mine, Canada

The Sullivan mine (Sullivan Mine Staff, 1970) obtains 95% of its ore production from pillar recovery. Four basic methods have been developed to recover pillars at Sullivan: slot-and-shell, multistage slot, core-and-shell, and overcut with separate slot-and-shell stages.

In the core-and-shell method, the core of a pillar is mined out, then the shell is blasted, with draw continuing until fill material and hanging wall waste

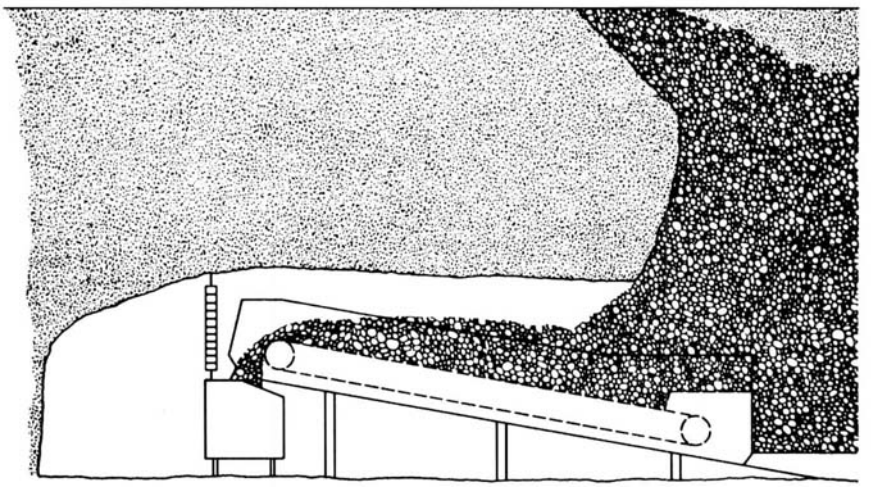


FIGURE 9.9 Feeder in drawpoint, Bellefonte mine

appear in the slusher drifts. This method is used in areas of low stress to extract 15.2- to 30.4-m (50- to 100-ft) pillars.

The multistage slot method is used in higher pillars as well as those under greater stress. It is basically the same as the preceding method, except that the pillar is mined in lifts of about 18.2 m (60 ft), retreating from the hanging wall toward the footwall. Temporary crown pillars are left between these 18.2-m (60-ft) lifts to reinforce the pillar. Drilling is performed either with a diamond drill using a 33-mm ($1\frac{5}{16}$-in.) plug bit or with a 102-mm (4-in.) bore percussion drill. The broken ore falls into slusher drifts and is gathered by slushers, varying from 22.3 to 93.2 kW (30 to 125 hp) in size.

Slushers of 22.3-kW (30-hp) capacity are used to scrape ore from small pillars. Those in the 37.2-, 44.7-, and 55.9-kW (50-, 60-, and 75-hp) range are used in large pillars for scrapes up to 60.9 m (200 ft). For distances over 60.9 m (200 ft), 93.2-kW (125-hp) slushers are used.

If the dip of the hanging wall is 0.65 rad (37°) or steeper, the stope will be filled by gravity. When the dip is less than 0.65 rad (37°), the stope is filled by a slusher or by a pneumatic stower that was developed at Sullivan. According to tests, a cost saving of about 33% can be realized when the custom-made stower is used instead of the slusher to backfill a stope (Reynolds, 1972).

Burlington Mine, United States

The main level interval of the Burlington mine (Fletcher and Evans, 1973) is 39.6 m (130 ft), and sublevel interval is 6.0 m (20 ft), back-to-floor. The ore is mined in an open slot in the back of the stope, with retreat beginning on the top sublevel. During the initial stages, the ore falls into the slot, which provides the loader operator with a solid cover. As the stope is mined back on the

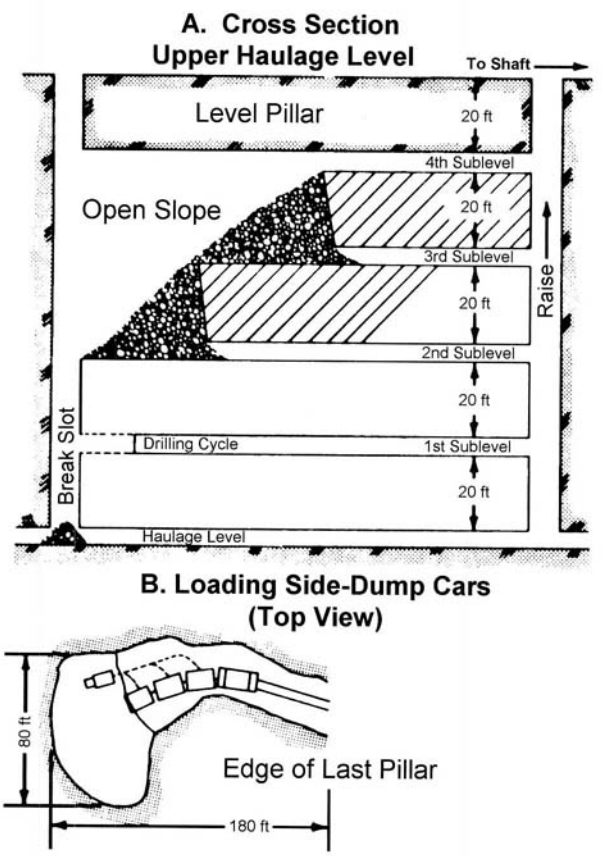


FIGURE 9.10 Initial phase of sublevel stoping, Burlington mine

bottom sublevel, the loader is operating under a back that is 39.6 m (130 ft) high. To keep the operator in a safe location, the loader uses a remote-control system.

The loader is an air-operated overshot loader on rubber tires. The operator controls it from a safe, remote position by control lines encased in an umbilical cord. The loader trams to the muck pile, loads, and returns to the cover of the pillar. It then either dumps directly into tram cars or onto the track, if no cars are available. Ore dumped directly on the track then is loaded into the cars by a rail-mounted overshot loader. See Figs. 9.10 and 9.11.

Madeleine Mine, Canada

The main level interval of the Madeleine mine (Gaumond and Parfitt, 1971) is 45.7 m (150 ft), with one intermediate sublevel at 22.8 m (75 ft). Stopes are 18.2 m (60 ft) wide, 45.7 m (150 ft) high, and spaced on 30.4-m (100-ft) centers. A temporary 12.1-m (40-ft) wide rib pillar is left, and a sill pillar also is left on a temporary basis.

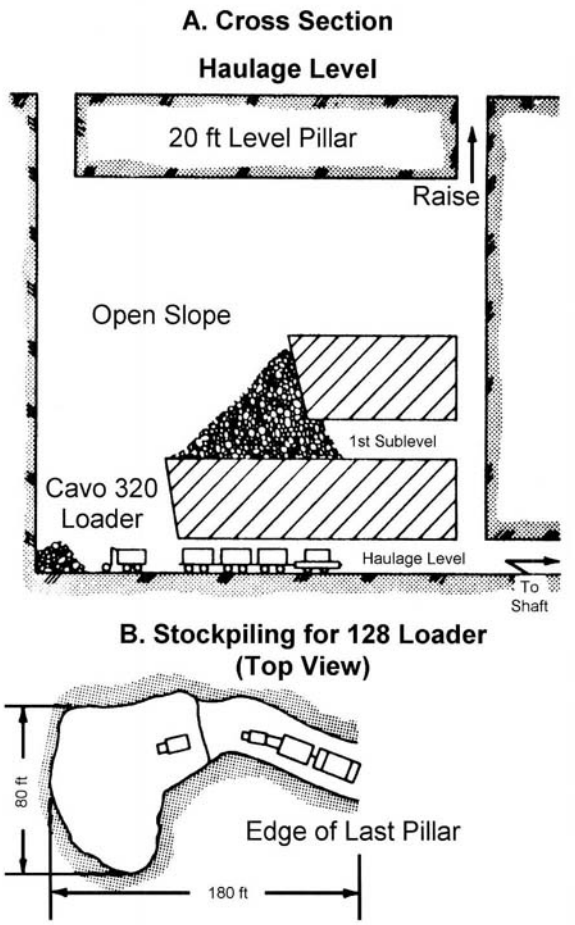


FIGURE 9.11 Final phase of sublevel stoping, Burlington mine

Stoping begins in a slot at one end of the stope. Fans inclined 1.4 rad (80°) toward the open face are drilled up from the main level, and vertical fans are drilled downward from the top level, as shown in Fig.9.12. Rib pillars are removed by drilling rings from the intermediate sublevel, and sill pillars are dropped into the open stope below. The overall direction of retreat is from the top down and from one end of the orebody.

Floor and rib pillars at this mine are recovered during the stope mining process. This method is similar to that used at Stråssa. The orebody is mined in a vertical retreat, leaving no permanent pillars.

LHD units of 3.0-m^3 (4-cu yd) capacity handle production at this mine. Production haulage ranges from 30.4 to 182.8 m (100 to 600 ft) and averages 83.8 m (275 ft). Average productivity is 544.2 t (600 st) per shift. Floors of haulage drifts are finished with a 152-mm (6-in.) concrete surface that is reinforced with wire mesh.

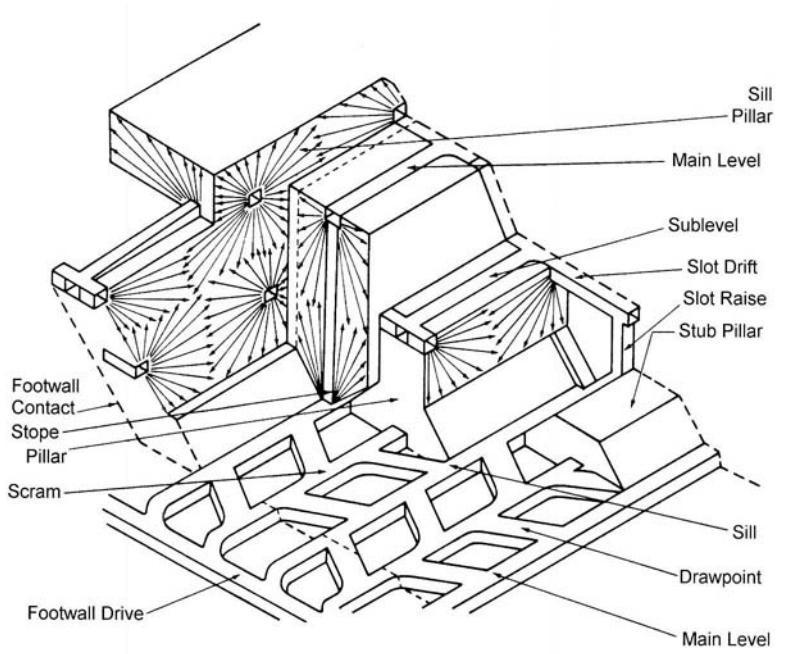


FIGURE 9.12 Stoping method at Madeleine mine

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CHAPTER 10

Excavation Design and Mining Methods in the 1100 Orebody, Mount Isa Mine, Australia

B. Hornsby and B.J.K. Sullivan

INTRODUCTION

Mount Isa Mines Ltd. operates a copper and silver-lead-zinc property at Mount Isa in northwest Queensland, Australia.

Mount Isa is located in the tropics at latitude 20°43 'S and longitude 139°30 'E (Fig. 10.1). It is on the upper watershed of the Leichhardt River and at an elevation of 365 m (1200 ft) above sea level. The average rainfall is 360 mm/y (14 in. per year), the greater part of which falls in January to March (the hot summer months). The mean summer temperature for these months is 30°C (86°F) (dry bulb) with a maximum temperature of 43°C (109°F); the mean wet bulb temperature is 25°C (77°F) with a maximum of 28°C (82°F).

Mount Isa is located geologically toward the western edge of the Precambrian Shield of northwestern Queensland. This complex forms a north-tilting dissected peneplain, which extends in the north-south direction for 640 km (400 miles), with widths varying from 32 km (20 miles) to 190 km (120 miles). The combination of differential erosion, regional strike, and steep dips has produced a corrugated topography of low north-south oriented hills and valleys.

The field was first discovered in 1923 as a silver-lead-zinc field. Mount Isa Mines Ltd. was formed the following year and subsequently became the sole operating company on the field. Lead and copper ore have been mined and treated separately and simultaneously at Mount Isa since 1953.

Intensive exploration commenced in 1954 and an active exploration program provided the justification for a series of production expansion programs in both silver-lead-zinc and copper (see Table 10.1).

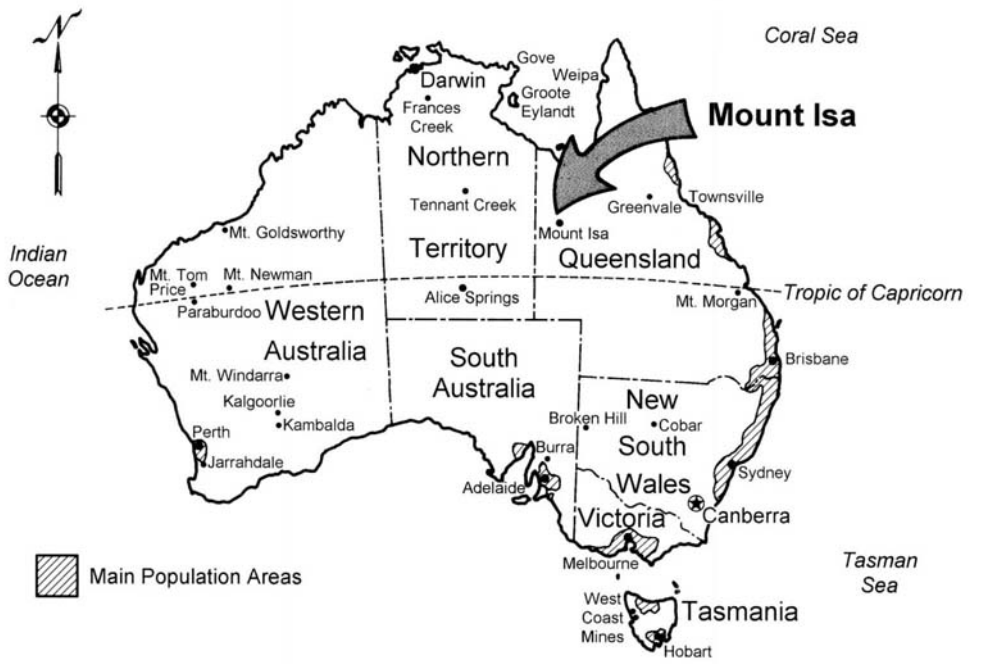


FIGURE 10.1 Location plan

TABLE 10.1 Production expansion in silver-lead-zinc and copper

Years	Total t (st) mined	Lead ore, t (st)	Grade, % Pb	Copper ore, t (st)	Grade, % Cu	Crude lead, t (st)	Blister copper, t (st)
1950–51	556,173	556,173	7.3	—	—	32,959	—
	(612,903)	(612,903)		—		(36,321)	
1955–56	1,334,674	639,661	7.7	695,013	3.7	35,960	24,017
	(1,470,811)	(704,906)		(765,904)		(39,628)	(26,467)
1960–61	2,287,821	800,236	7.9	2,037,585	3.6	52,447	46,858
	(2,521,179)	(881,860)		(2,245,419)		(57,797)	(51,638)
1965–66	3,612,800	1,057,412	6.8	2,555,388	3.2	60,575	74,656
	(3,981,306)	(1,165,268)		(2,816,038)		(66,754)	(82,271)
1970–71	5,933,883	2,179,848	7.2	3,754,035	3.1	142,040	100,338
	(6,539,139)	(2,402,192)		(4,136,947)		(156,528)	(110,572)
1974–75	7,417,639	2,484,007	6.3	4,873,573	3.2	131,700	160,250
	(8,174,238)	(2,737,376)		(5,370,677)		(145,133)	(176,596)

The most recent and largest production expansion program at Mount Isa commenced when a decision was reached in December 1969 calling for an increase in copper production from 100,000 to 150,000 t/y (110,200 to 165,300 st per year) of blister copper. A major factor that influenced this decision was the substantial addition to the ore reserves in the so-called 1100 orebody, which is a

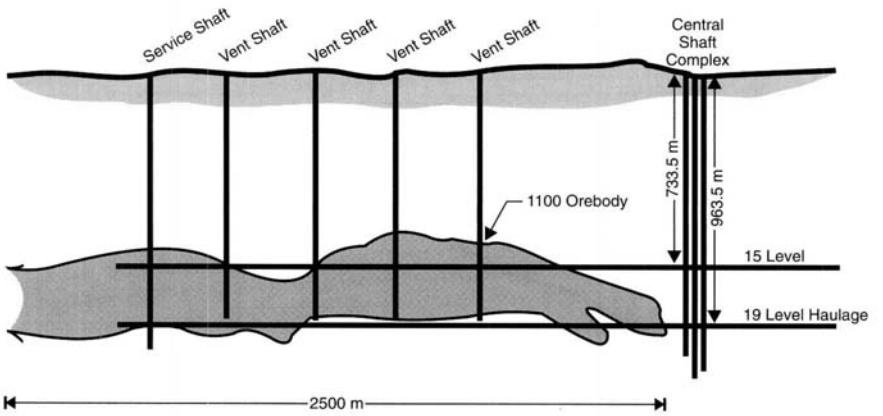


FIGURE 10.2 Longitudinal section through 1100 orebody

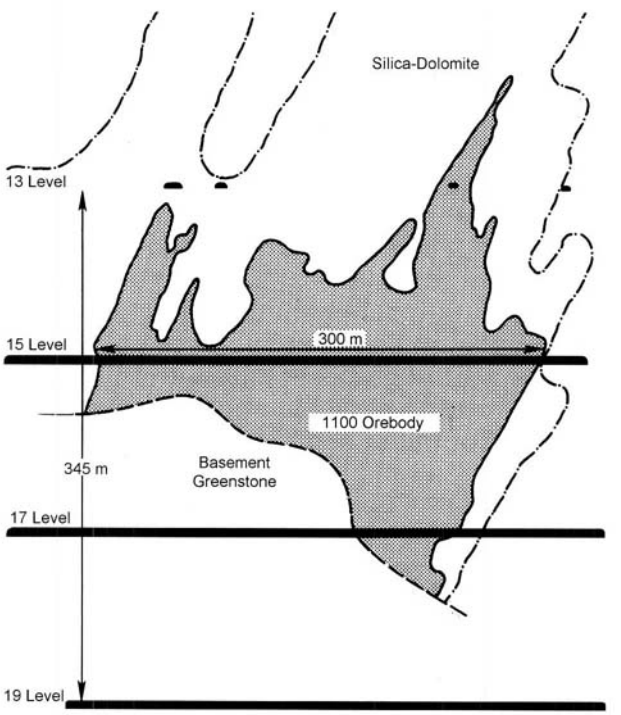


FIGURE 10.3 Typical cross section through 1100 orebody

massive copper sulfide orebody with a known strike length of approximately 2500 m (8202 ft), a height that is variable up to 300 m (984 ft), and a width of generally 300 m (984 ft) up to a maximum of 370 m (1214 ft) (see Figs. 10.2 and 10.3).

Host to the copper mineralization is a complex and variable rock generally called "silica dolomite."

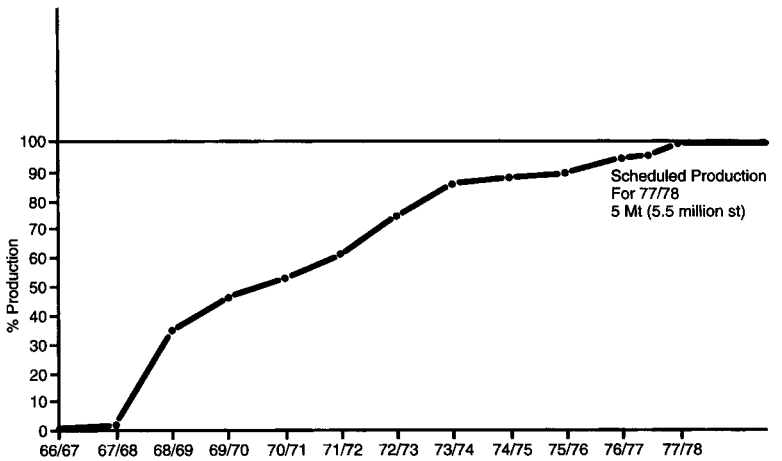


FIGURE 10.4 Production from the 1100 orebody from its inception to 1977, shown as a percentage of total Mount Isa mine copper production

The orebody strikes 6.02 to 6.1 rad (345° to 350°), dips west between 0.78 and 1.13 rad (45° and 65°), and is truncated at depth by fault-implaced basement rocks. It first occurs some 600 m (2000 ft) below the surface and bottoms at about 1000 m (3300 ft) below the surface. The unconfined compressive strength of the silica dolomite ore varies from 150 to 250 MPa (21,800 to 36,400 psi), whereas the basement rocks, carbonaceous mylonite, slaty shale, buck quartz, and greenstone (a chloritic schist) are considerably weaker, varying in strength from 20 to 80 MPa (2900 to 11,600 psi). The distribution of mineralization within the orebody is variable, but can be considered as generally high grade at or just above the basement contact and diminishing toward the upper reaches of the orebody. The current ore reserves of the orebody are about 100 Mt (110 million st) at 3.0% copper.

Production first commenced from the 1100 orebody in April 1967. The rate of production and its importance to Mount Isa operations both increased rapidly from 1968 to 1973, and it is currently Mount Isa's sole producing copper orebody (Fig. 10.4). Through 1978, in excess of 35 Mt (39 million st) of ore were mined from the 1100 orebody, and production is planned to continue at a rate of approximately 5 Mt/a (5.5 million st per year). This chapter reviews the factors that have influenced excavation design and the selection of mining methods.

OPERATIONS PRIOR TO 1970

Initial Selection of a Mining Method for the 1100 Orebody

The 1100 orebody was first discovered in the mid 1950s. By the early 1960s, it was realized that the orebody was very wide, relatively rich, and of considerable strike length. It was also accepted that the ore was not particularly suited to self-caving because of its generally very competent nature, and that it was cut off at depth by an extremely weak barren mass of rock locally called greenstone, which is a chloritic schist.

Mining methods used underground at Mount Isa up to and including the early 1960s consisted, in the main, of varieties of open stoping with transverse, rib, and floor pillars; 60 to 70% of the ore was recovered from the open stopes and most pillar recovery was to be left to a later date. Some pillar recovery was achieved by mass blasting rib pillars into open stopes where hanging wall conditions indicated this was possible, following which the remaining voids were filled with a variety of materials. Sublevel caving and/or small cement-filled open stopes were two methods of recovery subsequently used to recover some of the remaining pillars. It was planned to introduce sublevel caving and a mechanized cut-and-fill method to cater to specific conditions that existed in particular orebodies, but these conditions did not exist in the 1100 orebody, and both methods were untried and still to be perfected at Mount Isa.

Against this background, the choice of large open stopes followed by mass pillar blasts as the mining method in the 1100 orebody was logical because the ore competency suggested that open stoping would operate favorably, the method was well proven at Mount Isa, and open stoping on a large scale would be economical with development, which is an expensive item in mining. Also, development in such an area could be made more economical because of the massive orebody, which would allow trackless mechanized mining with multiple development headings. Primary stoping and mass pillar blasting were expected to yield between 500 and 650 t/m (170 and 220 st/ft) developed. With block caving not considered tenable because of the competency of the ore, and sublevel caving giving between 130 and 250 t/m (44 and 84 st/ft) developed, this open stoping figure was very attractive.

In addition, indications were that high productivity could be obtained using load-haul-dump units in good ground. Large numbers of stopes could be prepared in a centralized area, assuring a high production potential. Pillars remaining after mass blast recovery could be mined by sublevel caving; and the overall method was expected to give high metal recovery with minimum dilution.

Major problems envisaged with this method were mining near the greenstone contact, high rock pressures due to the depth of the orebody, and the regular occurrence of significant geological structures such as faults.

INITIAL DESIGN AND PRODUCTION FROM THE 1100 OREBODY

To keep abreast of almost continual expansion of copper production through the 1960s, additional copper sources were required as quickly as possible from the 1100 orebody. A start was made when two large open stopes, R29 and S32, were designed toward the northern end and in the upper part of the orebody.

Both stopes were designed for extraction using low-level longitudinal scrams (entries) with 93.2 kw (125 hp) electric-driven slushers fitted with 1.8-m (6-ft) folding scoops and loading directly into 1.07-m gage (3.5 ft), 7-m^3 (9-cu yd) Granby cars on 15 level. Initial extraction was designed to be on 15 level [about

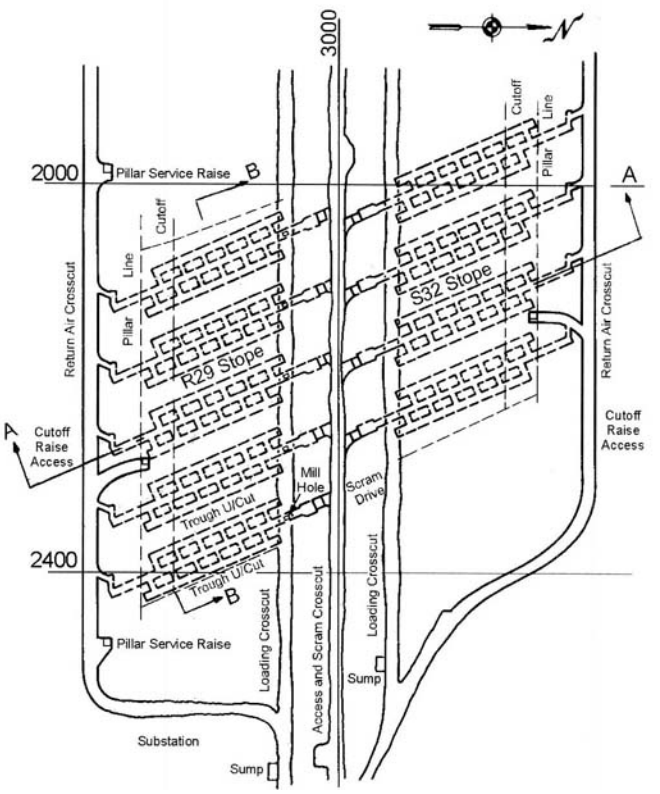


FIGURE 10.5 Plan of low-level scrams of 1100 orebody at 15 level

730 m (2400 ft) below the surface] although the orebody/basement contact was, at that northing, around 17 level [850 m (2800 ft) from the surface]. This was because 15 level development was well advanced whereas 17 level was only just beginning and it would mean a delay of several years in the start of production if 17 level was the initial production level. The top of both stopes was at about 13 level [some 600 m (2000 ft) beneath the surface]. Basic stope layout is as shown in Figs. 10.5 and 10.6.

The first stope designed was R29, about 100 m (300 ft) wide (east-west) and 40 m (120 ft) long (north-south). The width was a function of the desirability of having a large stope, whereas the length was chosen to optimize the productivity of scram production. This latter had a significant bearing on the overall orebody layout because it fixed the extraction crosscut positions at 160-m (520-ft) centers. When load-haul-dump unit extraction was introduced, north-south basic dimensions were not altered because they suited the 140-m (450-ft) desired maximum one-way production tramming distance.

Following the design and commencement of R29 and S32 with scram extraction, the next stopes were designed for extraction using 3.8-m^3 (5-cu yd) load-haul-dump units from up-hole bench leading stopes. A typical layout is shown in Fig. 10.7.

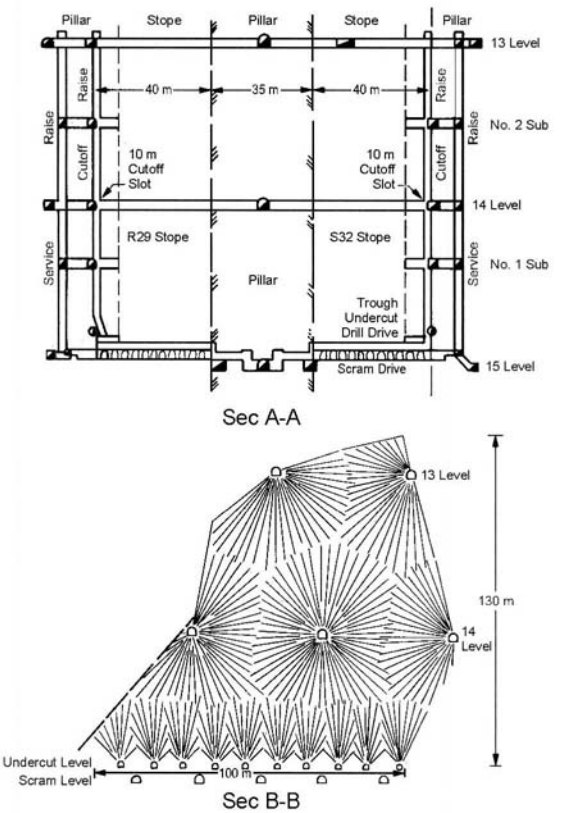


FIGURE 10.6 Diagrammatic sections of sublevel open stoping, 1100 orebody above 15 level. Top represents longitudinal section R29 and S32 stopes, looking west. Bottom represents cross section looking north; typical ring design for the R29 stope.

The three main reasons for this design were (1) to take advantage of the greater productivity offered by use of LHD units compared to electric slushers; (2) to minimize the time required to bring a stope into production and thus receive a quick return from the high-grade ore just above and below 15 level (stopes such as R29 and S32 required a minimum of two years' work on development and construction before they could be brought into production, whereas a leading stope could be brought into production at a rate of 1000 t/d (1100 stpd) six months after development of the stope commenced); and (3) to improve the overall safety of the operation by eliminating the need for slusher operators to enter scram controls to bomb them down.

Leading stope extraction crosscuts (3.7 × 3.7 m) (12 × 12 ft) were developed from a footwall drive using diesel-powered Gardner-Denver rubber-tired three-boom drill jumbos and Wagner 3.8-m^3 (5-cu yd) load-haul-dump units on the level where extraction was to commence. A cutoff drive was then created at right angles to these crosscuts and a cutoff raise established to the top of the ore block. A slot was then formed by drilling parallel up holes and firing them into the raise, allowing production to start immediately. The height of the slot that could be formed was limited by drilling accuracy to about 24 m (78 ft). Fans of blastholes were drilled from the extraction crosscuts and fired

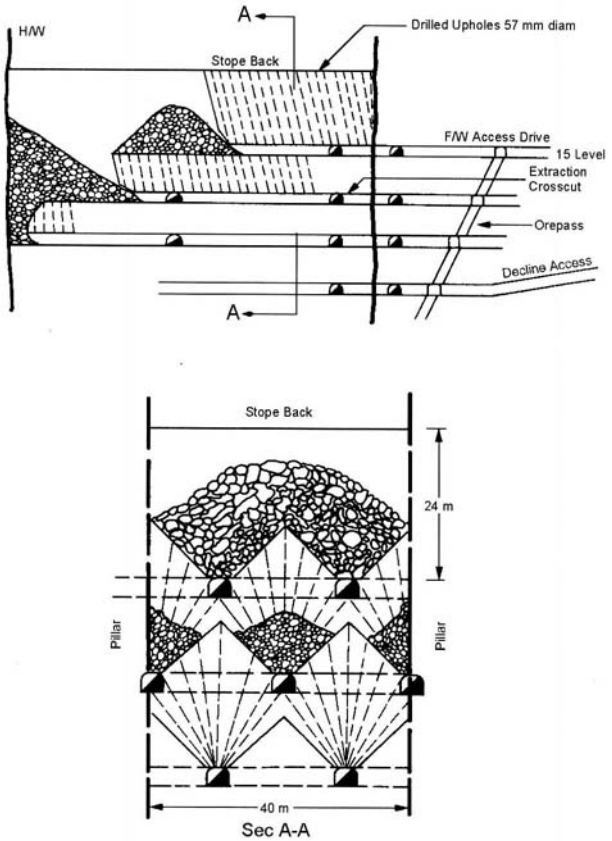


FIGURE 10.7 Diagrammatic sections of leading stopes, 1100 orebody starting 15 level. Top represents cross section looking north; bottom, longitudinal section looking west.

separately toward the cutoff slot. As each fan was fired, that ore that could be reached without going beyond the brow was mucked out before the next fan was fired. Production mucking was by means of 3.8-m^3 (5-cu yd) load-haul-dump units, which initially had to tram to 15 level and side load into 7-m^3 (9-cu yd) Granby cars. Once 17 level was developed, ore was tipped via ore-passes to the 17 level haulage crosscuts.

While production commenced on the first extraction level, a footwall decline was developed to allow lower extraction sublevels to be developed. Broken ore left on the upper sublevels beyond the reach of the load-haul-dump unit was recovered on the lower sublevels as production progressed. The final sub-level was equipped with drawpoints to recover the remainder of the broken ore. This method also permitted extension of the stope upward or downward with conventional ring drilling and extraction via the drawpoints on the final sublevel, as shown in Fig.10.8.

Leading stoping had a very definite cash-flow advantage and overall profitability advantage over low-level scram open stoping; it also fulfilled the more immediate production and operating requirements of achieving metal tonnage earlier than was possible with low-level scram stopes (see Fig. 10.9).

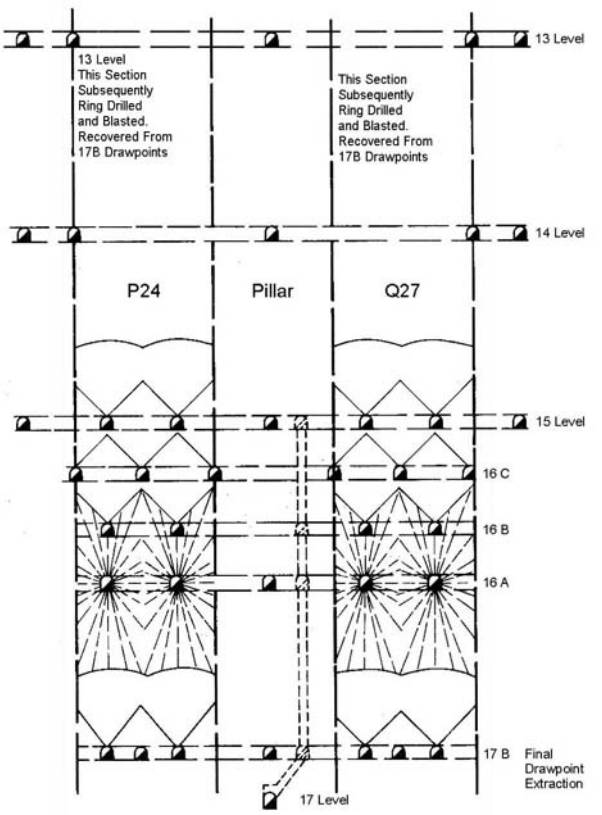


FIGURE 10.8 Longitudinal section looking west showing first two leading stopes and typical ring drilling following leading stope extraction

Blastholes of 57-mm (2.2-in.) diam were standard for this early production with hole length limited to 27 m (90 ft) maximum in ring drilling to restrict deviation to an acceptable degree. Ring burden and spacing of 1.8 × 2.4 m (6 × 8 ft) gave a primary explosive factor of 0.26 kg/t (0.5 lb per st) using blow-loaded ANFO.

Planning at the End of 1969

Following the completion of the design of the R29 and S32 stopes, by 1967, the overall layout for the 1100 orebody was well advanced. It comprised a network of stopes (those in the 27 and 24 blocks being designed initially as leading stopes) and pillars similar in plan dimensions to those of the R29-S32 block. Design concepts had been established between the planning and rock mechanics groups. Stopes were wide from east to west, and stopes and transverse pillars were of similar length, about 40 m (Fig. 10.10).

A major longitudinal pillar was placed about midspan to help buttress the transverse pillars and support the wide expanse of back. The main extraction horizon north of 2000N was to be immediately above 17 level. It was assumed that primary stoping would be open stoping with subsequent mass blasting of

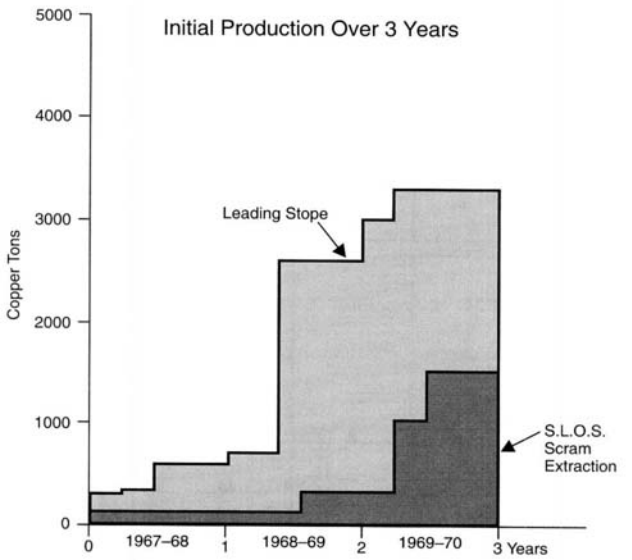


FIGURE 10.9 Tons metal contained versus time (estimated production per period)

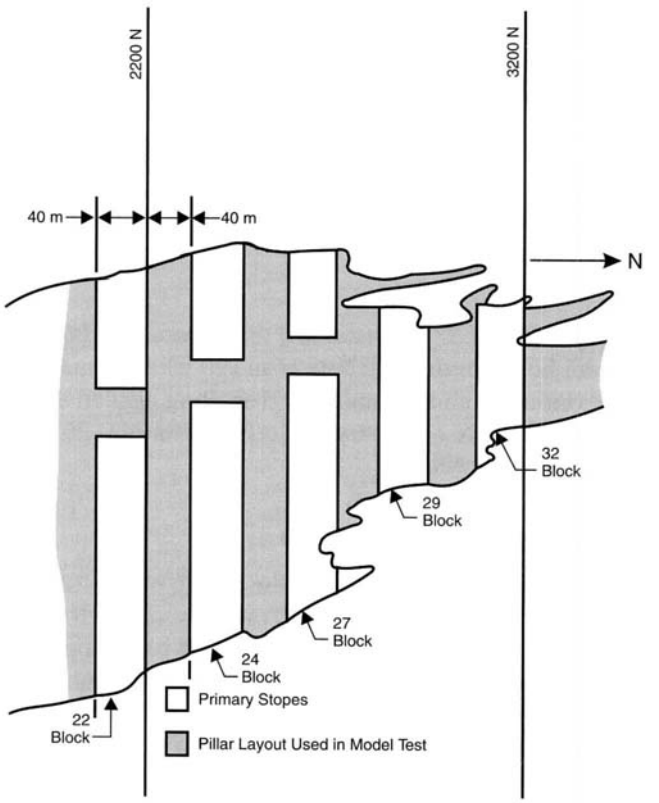


FIGURE 10.10 Plan view of the 1100 orebody showing initial layout of stopes and pillars

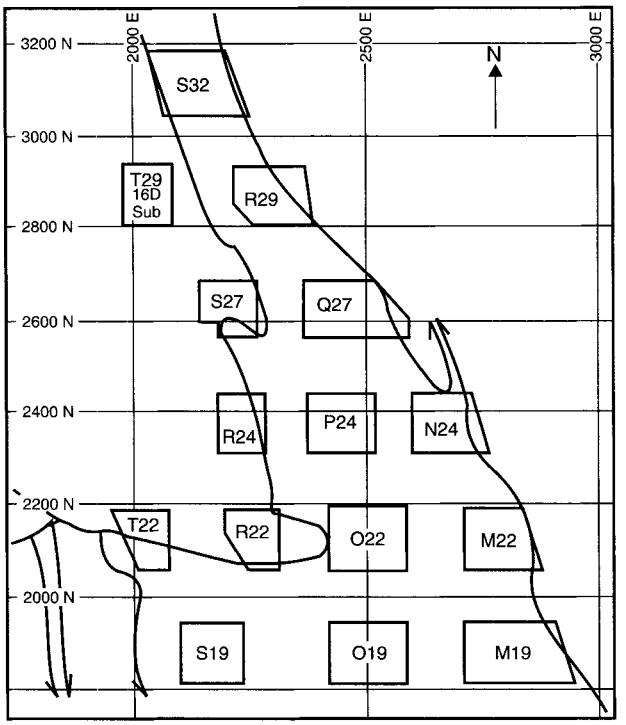


FIGURE 10.11 Plan view of system of open stopes with rib and transverse pillars adopted at the end of 1969

rib pillars, after which the stopes would be filled, leaving final pillars that may not be recoverable.

The layout was critically reviewed in early 1968 following a structural weakness exposed in R29 stope, which caused serious drilling and blasting problems that affected production. It was decided to limit the east-west extent of R29 stope and reduce the design east-west extent in the 27 and 24 blocks to 60 m (200 ft). A material model test of the whole 1100 orebody area, with only transverse and main longitudinal pillars remaining (see Fig. 10.10), was completed by the Australian Coal Industries Research Laboratory in November 1968 and this showed that, although the overall area showed little significant damage, major spalling of the walls of the transverse pillars in the 27 and 24 blocks could occur.

By the end of 1969, a system of open stopes and pillars in a roughly square pattern was adopted (Fig. 10.11). Methods of pillar recovery were still to be resolved, but the then-current thinking was to extract primary open stopes, fire rib pillars or sections of transverse pillars into open stopes and extract, fill all stopes with coarse fill from the surface, and sublevel cave the remaining transverse and longitudinal pillar–this to take place long after primary pillar extraction is completed.

Although sublevel caving was the proposed method for recovering secondary pillars (which contained over 50% of the ore), this was still largely unplanned,

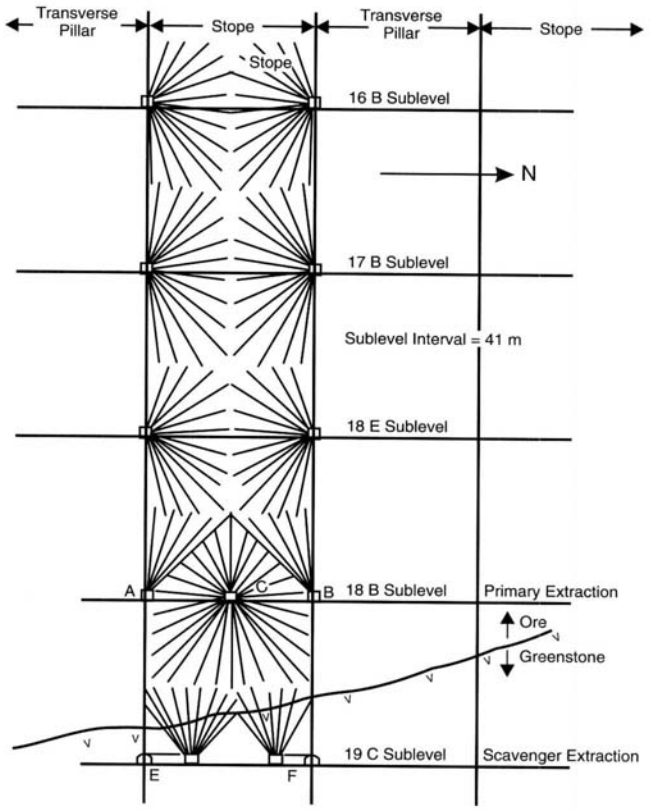


FIGURE 10.12 Long section looking west of a typical open stope with two-stage extraction. Initial extraction on 18B sublevel, retreating drawpoints A and B, leaving center heading C for firing out the scavenger; scavenger extraction on 19C sublevel from fixed drawpoints E and F.

and other ideas such as cut-and-fill stopes and cement-filled open stopes were considered. It was accepted that some cemented fill would probably be required in the primary stopes to assist or improve pillar recovery.

OPERATIONS 1970–1975

Primary Stoping for Basic Production

Throughout 1970–1975, the basic design and extraction of sublevel open stopes changed little, and almost all of the production obtained from the 1100 orebody, except for production from development, was from high-rise, sublevel open stopes. The practice of leading stoping had served its purpose in providing earlier production than would otherwise have been possible, and thus was phased out as drawpoints were developed according to schedule on the final design extraction horizons.

For these high-rise sublevel open stopes (Figs. 10.12, 10.13, and 10.14) major sublevels were established in the orebody at vertical intervals of 41 m (130 ft), which facilitated production blasthole drilling using bar and arm rigs

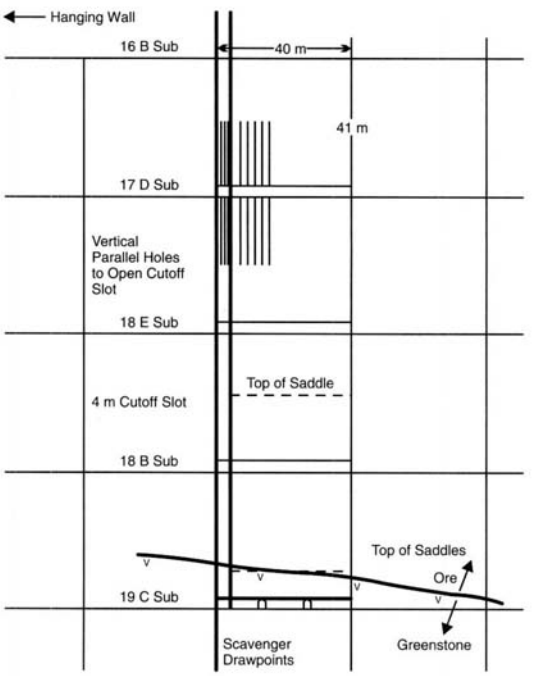


FIGURE 10.13 Typical cross section of a sublevel open stope looking north

and PR123 machines. Stopes were laid out on a fairly regular pattern, as indicated in Fig. 10.11. Stope limits were set by the cutoff grade at the top of the orebody, a maximum east-west length of 60 m (196.8 ft) [this was exceptional, with 40 to 50 m (130 to 160 ft) being more typical initially; subsequent variations are detailed in the following section on pillars], and the bottom major sublevel that was in ore within the east-west limits of the stope. A 1.8-m (6-ft) diam raise-bored cutoff raise was used to open up a 4-m (13-ft) north-south slot and full-face ring firings were then used to extract the stope. (The practice of full-face ring firing sometimes developed into overcutting of the stope to reduce problems with rings falling off due to adverse geological structures.)

The extraction horizon was located in ore because of the problems of extracting high tonnage from drawpoints developed in the weaker basement rocks. Positioning the primary extraction drawpoints in ore meant that the majority of the tonnage contained in the stope, in most cases, was extracted from good retreating drawpoints located in competent ore.

When primary extraction of the stope was completed, the stope was scavenged from a sublevel developed in the greenstone basement beneath the orebody. Scavenger drawpoints were developed as fixed drawpoints in order to maximize recovery from the stope.

These fixed drawpoints required extensive support because of the relatively incompetent nature of the basement rock types in which they are mined. A few stopes contained more than 1 Mt. (1.1 million st), but 750,000 t/stope

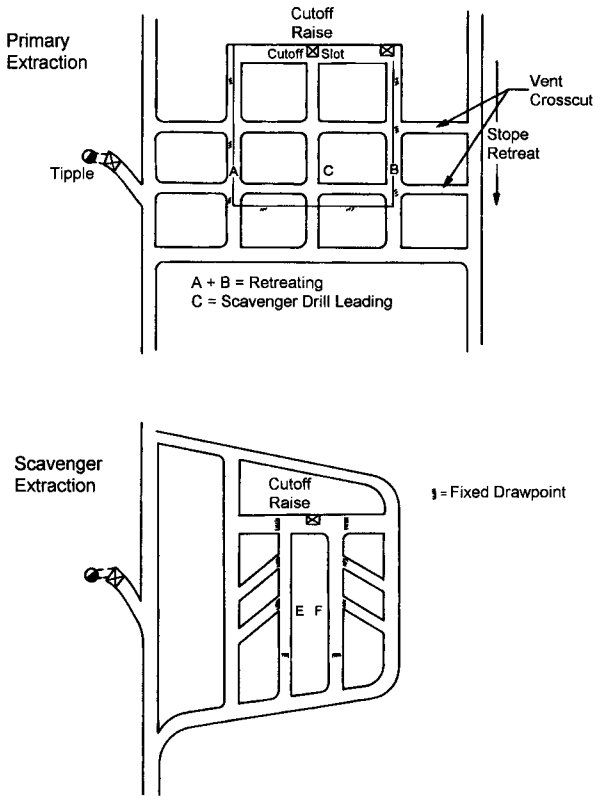


FIGURE 10.14 Plan view of typical primary and scavenger drawpoint layouts for open stopes

(826,500 st per stope) was an average stope size, of which more than 500,000 t (550,000 st) were extracted through the primary retreating stope drawpoints, the remainder being extracted through the scavenger drawpoints.

Pillar Recovery

Several problems deferred planned pillar recovery in the 1100 orebody, with the result that by the end of 1975, although 20 primary sublevel open stopes had been fully extracted to meet the mine's production commitment, pillar recovery had not started.

Pillar blasting into open stopes was the planned recovery method in late 1969 for some of the pillars. Further work by the Australian Coal Industries Laboratory in 1970 and 1971 and doubts about the stability of large openings—failures were occurring in the R29 stope at that time with rings falling off—caused this method to be abandoned for the 1100 orebody before it was attempted.

Sublevel caving was the other recovery method that had been considered, although no detailed design work had been completed, and it was envisaged as being the final stage of extraction to recover whatever was left after primary stoping and initial pillar recovery.

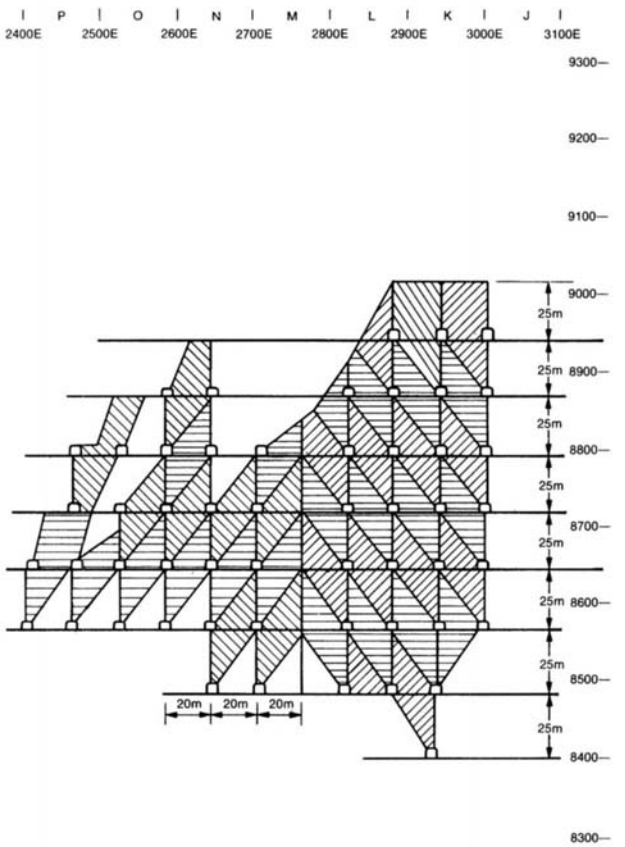


FIGURE 10.15 Cross section showing how panel stoping was envisaged. Starting at the basement contact from a central cutoff slot, panel stopes were to be mined upward, retreating progressively to hanging wall and footwall of the orebody.

A proposal for a shrink horadiam method—basically a series of shrink stopes against primary stopes containing cemented fill—was rejected mainly on the grounds of safety because personnel would have to work in raises with the high possibility of brow failure resulting from continuous undercutting.

A revision of this horadiam method was included in a proposal for panel stoping (Fig. 10.15). This involved a series of small lozenge-shaped open stopes, three sides of which were to be formed by cemented fill. This proposal was studied in detail, but the stope size was severely limited by the ability of cemented fill to stand freely when exposed by the pillar stope, and it appeared the maximum tonnage per stope was approximately 50,000 t (550,000 st). This presented almost insurmountable problems in scheduling the production, fill, cure, production cycle in a sufficient number of such small stopes to meet tonnage targets. Consequently, this method of pillar recovery was not favored and was shelved until larger fill exposures proved obtainable.

Having abandoned the original planning for pillar recovery and then considered the aforementioned alternatives, Mount Isa's Mining Research Department recommended a further proposal. This involved adoption of a layout of

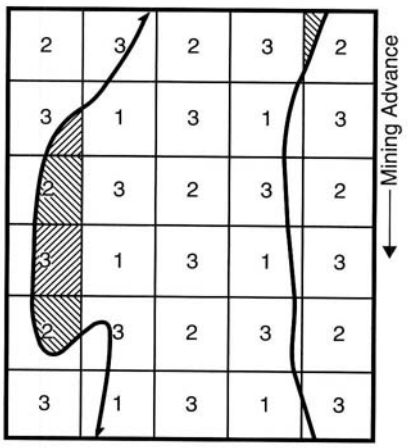


FIGURE 10.16 Rigid "egg crate" 40-m (130-ft) square layout shown for random ore limits. 1 = primary sublevel open stope, filled with cemented rock fill, surrounded by four rock walls; 2 = secondary stope, filled with cemented rock fill, mined against rock walls with cemented rock fill at the four corners; 3 = tertiary stope, mined against cemented rock fill on three or four sides, rock filled. Hatched areas can be taken as extensions on primary or tertiary stopes.

pillar lines to a rigid 40-m (130-ft) square "egg crate" pattern and extraction of the orebody using a high-rise open stoping technique with three phases—primary stopes, and secondary and tertiary pillars (see Fig.10.16). No detailed design work was done for individual secondary or tertiary pillars, but an integral part of the general design was the use of cemented rock fill in primary and secondary pillars. The reasons for this were to increase the back support capabilities of fill pillars, to enable larger exposure of the fill to be made, and to enable access development to be mined through filled stopes.

Mass blasting using slurry explosive formed an integral part of the proposed pillar recovery method.

This proposal was accepted in the main, and it was decided at the end of 1972 that primary stopes would be filled with a cemented rock fill (in effect, a weak concrete) as soon as possible after completion of extraction. Also, the first two transverse pillars formed in the orebody (3000 and 2800) would be mined by a controlled draw sublevel caving method beneath introduced rock fill. [The "egg crate" primary, secondary, tertiary approach was not well suited here because of the size (and hence excessive fill exposure) of the adjacent primary stopes, R29 and S32.] High-rise methods would be used to recover other pillars to the south within the "egg crate" layout.

The most significant aspect of the adoption of this proposal was to incorporate rock in the fill system. Work began immediately on the specification and selection of the suitable rock type, and subsequently the Kennedy siltstone open cut, some 2500 m to the north of the orebody, was designed and commissioned to provide rock fill, and a major underground fill horizon was established to distribute it (see Fig. 10.17). However, this did not happen overnight, and the lost time resulted in further delay in the pillar recovery program. Intensive production from the primary open stopes was necessary to

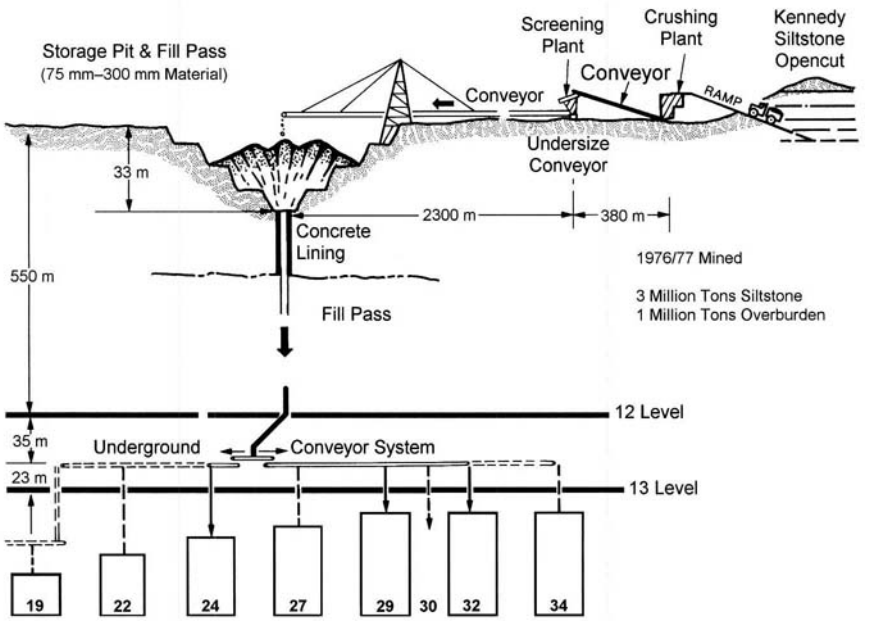


FIGURE 10.17 Diagram illustrating graded rock-fill system

meet production targets, and the temptation to increase stope dimensions beyond the 40 × 40-m (130 × 130-ft) "egg crate" layout was therefore always present.

Work on the detailed planning of secondary and tertiary stopes had commenced assuming the use of mass blasting into an expansion slot. This slot was to be rotated through 1.57 rad (90°) on alternate sublevels in the tertiary stopes to limit fill exposure to 41 × 13 m (135 × 43 ft).

Other concepts were also introduced, including that final stoping retreat should be from north to south and from hanging wall to footwall, expansion slot sizes should be kept as small as possible, and regional support should become a prime consideration in overall planning and, as such, monitoring would become a key to planned recovery of the 1100 orebody.

A monitoring program was introduced, and planning continued using these concepts.

M54-L56 Mass Blast

Concurrent with decisions taken to adopt the "egg crate" system, plans were introduced to open up expansion room and mass blast the remaining ore in the M54-L56 block [approximate strike length 120 m (400 ft)] to the north of M545 stope after this stope was filled with cemented rock fill.

This approach was adopted mainly because the hanging wall of the orebody in this area was extremely incompetent and it was felt that the spans associated with sublevel open stoping would not be stable. Graded rock fill would

be introduced into the void as drawdown of the broken ore took place to maintain hanging wall stability and restrict the exposure of the cemented rock fill wall of the M545 stope to ensure its stability.

A single large mass firing incorporated approximately 1.3 Mt (1.4 million st) and this was followed by two comparatively small firings to complete the firing of the block. Approximately 46% expansion volume was allowed for the large mass firing, reduced to just over 30% in the later two blasts.

Adopting this approach was considered advantageous because (1) the method would reduce the need to reenter drill headings that would have become cracked or inaccessible as progressive firings continued; (2) it would enable more efficient firings (i.e., it would not be necessary to clean off drill headings prior to each single or multiple ring firing, which so often occurs in open stopes); (3) it would result in improved overall fragmentation; and (4) it would provide experience in mass blasting and draw control, which would assist in the design of future pillar recovery by mass blasting.

Production commenced from this operation in 1975 and continued throughout 1976 and the early part of 1977. At completion, an estimated 2.2 Mt (2.4 million st) at 3.0% copper was pulled from an anticipated fired 2.5 Mt (2.8 million st) at 3.3% copper.

OPERATIONS 1976–1977

Reappraisal of the Situation

The steady change in the mining philosophy for the 1100 orebody, which took place up to and including the first six months of 1975, was followed by a more radical reappraisal during 1976 and 1977, prompted by a number of factors.

Serious Pillar Cracking Around Stopes. Cracking was detected both from strategically placed borehole extensometers and visual observations made by using a television camera specially designed and mounted to operate in a 100-mm (4-in.) diam hole drilled for inspection purposes.

Doubts about the Viability of Tertiary Stopes. The Mine Planning Department subsequently voiced its doubts about the practicality and viability of tertiary stoping, especially in regard to mining and accessing through cemented rock fill. In early 1975, serious doubts about its feasibility were again raised following analysis by the boundary element method, which is a more sophisticated method of analyzing stress distribution in and around the orebody.

Need to Contain Large Cost Escalations. Large cost escalations (on the order of 15–20% per annum) emphasized the need to trim costs and defer major capital expenditure while maintaining production at optimum levels. To do this, more emphasis had to be placed on lower cost mining techniques that could still attain metal targets and be technically feasible and maintain safety standards.

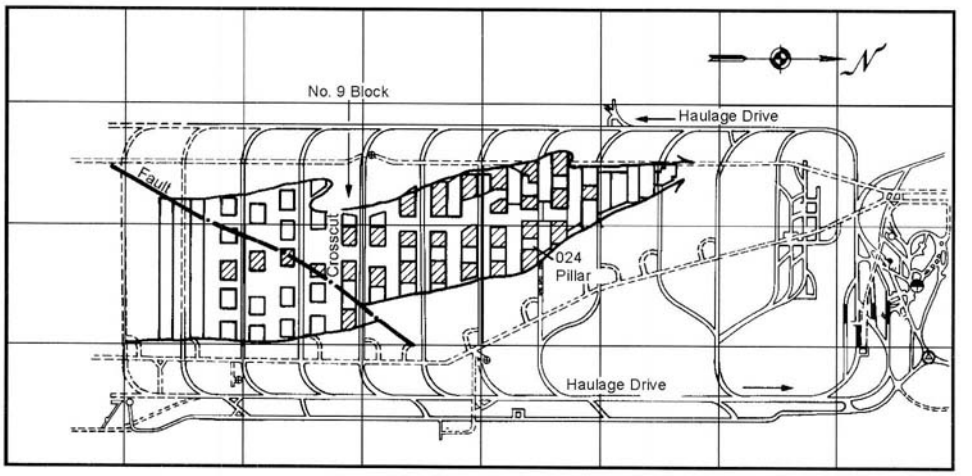


FIGURE 10.18 Plan of 19 level showing existing stoping blocks in 1100 orebody

Inconsistency of the Orebody as Stoping Continued Farther South. South of 9 block in the orebody, the nature of the orebody changes from massive to lenticular and faulted. Designing stopes to the rigid "egg crate" pattern in this part of the orebody would have resulted in very small, low-grade primary stopes and much larger, higher-grade secondary and tertiary pillars.

Decisions Taken as a Result of the Reappraisal

The reappraisal resulted in three significant decisions. First, primary stope design south of 9 block would not be restricted to the rigid "egg crate" layout. The north-south pillar layout as set out would be adhered to, because 19 level haulage, which is the major copper haulage horizon, was already installed. East-west limits of primary stopes would be selected to optimize ore recovery from the primary stopes (Fig. 10.18) with due regard to stope layout to the north and south of the particular block. Using the "egg crate," primary, secondary, tertiary system, 50% of the total orebody tonnage was tied up in the tertiary pillars, with between 25% and 33⅓% (maximum) extracted in the primary stopes. By designing stopes to suit the lenticular nature of the orebody, a greater percentage of the ore will be recovered from the primary stopes than would otherwise have been recovered by strict adherence to the "egg create" layout.

Second, a rib pillar would be mined as soon as possible to provide an early test of high rise stoping in rib pillars. Design work for the extraction of the P522 rib pillar toward the north end of the orebody commenced immediately. Figures 10.19 and 10.20 show the basic situation of the P522 pillar. The cemented rock fill in the adjacent stopes was not considered to be of high quality because of the way in which it had been placed. The plan was to provide an adequate expansion box in the pillar, mass fire the remainder of the pillar into the expansion box, and then introduce rock fill to maintain stability of the cemented rock-filled stope walls as a controlled drawdown of the

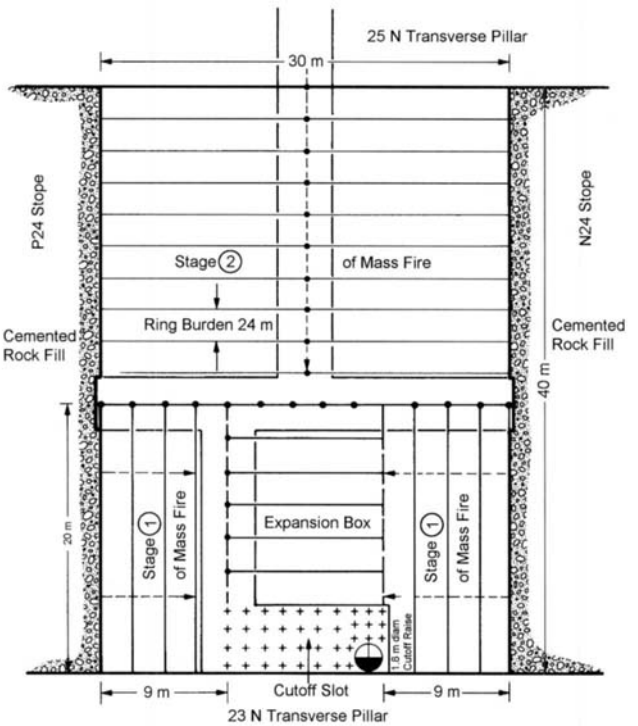


FIGURE 10.19 Typical P522 sublevel layout. Rings drilled from positions marked. Cutoff slot fired out first, then expansion box. Finally, mass fire, stages 1 and 2.

broken ore. The starting point for the design was to design a stable expansion box. Several alternative designs were considered with regard to pillar geometry, structural features occurring within the pillar, and the stress field existing prior to and during the opening of the box. A figure of 25% expansion room was agreed for P522. Figure 10.19 shows a typical sublevel layout with expansion box, as well as the basic ring design and firing sequence that were used in the pillar mass fire.

For a controlled draw to be effective, drawpoint layout is critical. A 13 × 13-m (43 × 43-ft) spacing was used because this was considered the maximum allowable horizontal separation of brow positions. Figure 10.21 shows how this was achieved in practice, necessitating mining through the fill in the adjacent stopes. In order to guard against drawpoint failure (loss of one drawpoint will curtail all production from the source until that drawpoint is rehabilitated if a controlled bin draw is to be maintained), extensive construction work involving concrete and steel universal beams was undertaken to support the drawpoints at the brow positions.

Regrettably, this early test of high-rise stoping using a slot and mass fire approach was inconclusive. The mass fire partially misfired, and, because of safety considerations, it was not possible to refire the misfired ore. The pillar was abandoned with only 240,000 t (264,000 st) of ore extracted.

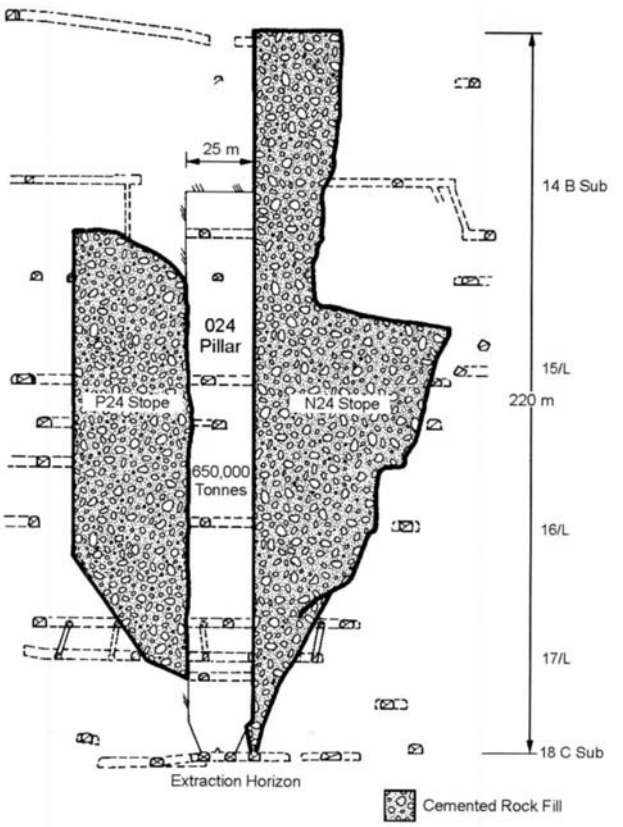


FIGURE 10.20 Typical P522 cross section, looking north

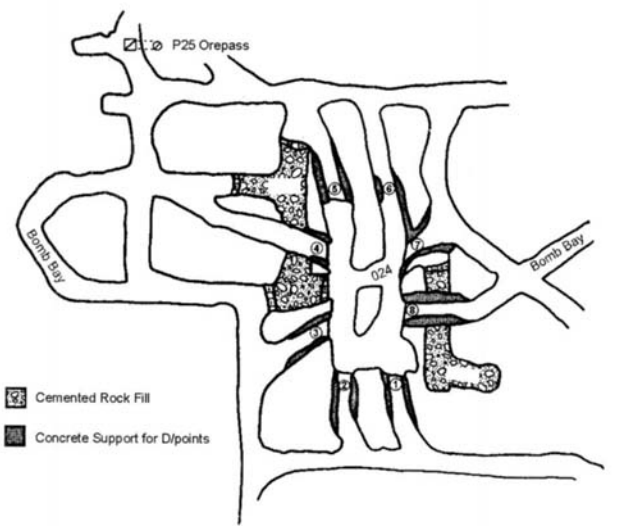


FIGURE 10.21 024 extraction horizon, showing drawpoints 1 to 8. Drawpoint nos. 4 to 8 had to be mined through P24 and N24 filled stopes, respectively.

Preparation has continued for the extraction of other rib pillars in the orebody using the basic slot and mass fire approach used in the design of the P522 pillar. It is still intended to introduce rock fill, when necessary, to maintain the stability of the cement rock-filled stope walls adjacent to each particular pillar as drawdown of the mass-fired ore takes place.

The above decisions enabled short-term planning and production to proceed. Another decision resulted in a task force being set up to determine and evaluate extraction strategies with reference to rock mechanics considerations, safety and environmental standards, production and operating requirements, and metal recovery and costs.

Rock mechanics considerations had the greatest impact on this work, and the main concerns were to ensure that overall stability was maintained at all times, ore remaining to be mined was left in the best minable condition, major installations were preserved until no longer required, and safe working conditions existed in production and access areas.

As safety and environmental standards were difficult to quantify, a qualitative approach was used. Degree and frequency of exposure of personnel to hazard was considered the best indicator of the comparative safety of a strategy, and estimates of likely ventilation conditions, and noise and dust levels were used to assess environmental conditions as a strategy progressed.

Production and operating requirements were ranked high, medium, or low in evaluating various extraction strategies. Achieving metal targets with optimum safety was the top requirement, and other requirements such as good fragmentation, five-day production week, minimum dilution, among others were considered within the overall range from high to low ranking.

Factors affecting metal recovery, reliability of ore reserves, metal distribution in the orebody, confidence level for recovery in different situations, and extraction or dilution factors for various mining methods were considered. A computer program was required to handle all the necessary information relating to costs.

SUBSEQUENT OPERATIONS

The guidelines that evolved from the extensive review of 1976–77 continue to be used as the basis for forward planning. Factors that could have a major effect on excavation design and mining methods include future developments in rock mechanics, large diameter blastholes, fragmentation and draw control, and developments in fill technology.

Rock Mechanics

Monitoring of the orebody is continuing and will continue in order to attempt to gain a more complete understanding of the behavior of the rock. Particular emphasis is being placed on those pillars that are known to be highly stressed, and information gathered will be used to determine how best to destress and mine them.

New developments in analysis techniques will be used wherever they may be applicable.

Significant information gathered from monitoring or evidence gained from fresh analysis could well influence any future change in design and/or mining methods used in the 1100 orebody.

Large Diameter Blasthole Drilling

Two open stopes were designed for large diameter blastholes to investigate the potential of this type of open-cut technology underground. Using a Robbins 11MD rotary drill to drill 200-mm (8-in.) diam blastholes up to 64 m (210 ft) in depth, two stopes in the southern part of the orebody, one containing 360,000 t (400,000 st) and the other containing 850,000 t (940,000 st)—were designed, developed, and successfully drilled and blasted. Cost per ton of ore broken in these stopes compared favorably to that of conventional 57-mm (2¼-in.) diam blasthole stopes, and fragmentation was generally excellent. Some problems were encountered with big muck in the stopes from falloff when the big hole rings were undercut, but these were not considered to be serious.

The major concern in undertaking this trial was the potential for increased damage, caused by vibration from the firings, to the rib and transverse pillars adjacent to the stopes and the possibility that this could jeopardize subsequent recovery of these pillars. All firings were therefore monitored from the surface for vibration levels. The levels measured were greater than those measured from conventional 57-mm (2¼-in) blasthole firings, although not significantly greater. Damage to the adjacent pillars, assessed visually by inspection of development in the pillars, did not appear to be either more or less significant than the damage sustained in conventional 57-mm (2¼-in.) blasthole firings.

The major advantage of this successful development is that it will permit the elimination of one in every three sublevels by enabling the spacing between sublevels to be increased from 41 to 62 m (130 to 200 ft), and possibly more. This is because the accuracy achieved when drilling longholes is much greater by rotary drills than is possible with conventional percussion drills. However, before planning can proceed with confidence to increase the sublevel interval—and therefore effect major cost savings in a development—it is necessary to demonstrate that this technology can also be successfully applied to pillar recovery.

The other main advantage of this development in drilling and blasting practice is the increased productivity it offers. As a one-person operation, drilling averages 15 m (50-ft) per person-shift, equivalent to approximately 1500 t (1650 st) per person-shift, compared to about 900 t (990 st) per person-shift by an operator on a percussion machine drilling 57-mm (2¼-in.) blastholes and averaging 70-m (230 ft) per person-shift.

In-the-hole hammer drilling, using an Atlas Copeo ROC 306 rig to drill 165-mm (6½-in.) blastholes, has also been tested in the 1100 orebody. Results

to date have been far less encouraging than those achieved by the 11MD, mainly because of very poor bit performance in the highly abrasive silica dolomite ore. The abrasiveness of the ore tends to decrease toward the upper reaches of the orebody, so there may be increased scope for this type of drilling in the upper part of the orebody.

Hydraulic Ring Drill

Tests have also been carried out with an Atlas Copeo 1038 hydraulic ring drill, drilling 63-mm (2½-in.), 70-mm (2¾-in.), and 89-mm (3½-in.) diam blastholes. Performances with the smaller diameter bits have been encouraging, but problems were experienced with the 89-mm (3½-in.) drilling. These problems included excessive deviation when the rod string was unstabilized, and unfavorable economics when using stabilizer couplings to overcome the deviation problem.

Although the potential exists for improving productivity with hydraulic ring drilling, it has not yet influenced stope design or mining method.

Fragmentation and Draw Control

The success of any mining operation is highly dependent on fragmentation. It has a direct effect on the safety of the operation, production rates (hang-ups in brows, orepasses, chutes, and tipples, and wear and tear on load-haul-dump units), metal recovery (draw control, dilution), and direct operating costs (secondary breaking, mucking, hauling, crushing, and hoisting costs).

A positive approach is being adopted to improve fragmentation and maintain it at an optimum level, because this has a direct effect not only on the overall productivity of the system, but specifically on the success of draw control. Close supervision and accurate accounting are essential if good draw control is to be effected. However, these factors will count for very little if fragmentation is unsatisfactory, particularly where drawdown is beneath sized rock fill, because the rock fill will rapidly percolate through the ore, causing excessive dilution in the drawpoints and eventual poor metal recovery.

Every effort will be made to ensure that fragmentation is satisfactory and draw control is effective. Good fragmentation is vital for successful draw control, and this places the emphasis back onto good drilling and blasting design and practice.

Developments in Fill Technology

As the gap between the total void created in the 1100 orebody and the total void filled decreases (see Fig. 10.22), the opportunity to improve the develop fill technology increases through actual experience gained allied to research work.

In planning to extract pillars by slot and mass fire methods in between cemented rock-filled stopes, the ability of the cemented rock-filled walls to stand unsupported with increasing exposure and stress will have a major

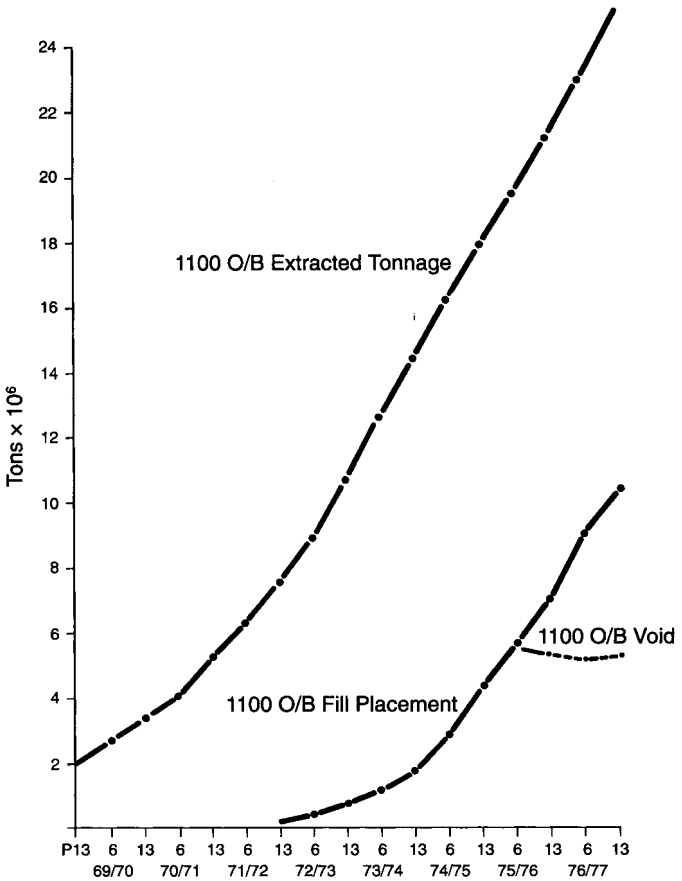


FIGURE 10.22 Tons extracted from 1100 orebody; tons of fill placed in 1100 orebody; calculated void tons remaining to be filled, English equivalent: t × 1.102 = st

impact on extraction strategies. Experience to date has suggested that initial predictions of the capability of cemented rock-filled walls to stand unsupported were conservative. Precisely how conservative, and exactly what realistic criteria can be followed, are still to be fully determined, but the potential to extract blocks of ore between filled stopes with greater cemented rock-filled exposures would have a significant impact on excavation design.

It is unrealistic to expect the cemented rock-filled stope walls to stand unsupported over unlimited heights. The present planned intention to support these cemented rock-filled walls with introduced rock fill places very great importance on the success of a controlled drawdown of the broken ore beneath the rock fill and the maintenance of a relatively undisturbed ore/rock-fill interface. If this is not achieved and dilution becomes excessive, thus reducing metal recovery, the overall approach will have to be revised.

CONCLUSIONS

The extraction of the 1100 orebody has reached its most challenging and interesting stage to date. Extraction strategies, and therefore mining methods and excavation design, continue to be evaluated with reference to

the considerations as outlined in this chapter. Other specific considerations likely to influence excavation design and mining methods are continuing developments of drilling and blasting technology; increasing experience and understanding of cemented rock-fill exposures in filled stopes; and the success of controlled drawdown of mass-fired ore beneath introduced rock fill.

With Mount Isa Mines Limited's firm commitment to the continuing use and further development of large diameter blasthole technology underground, in excess of 50% of its current copper production could be provided from stopes and pillars extracted using large-diameter blastholes. The success of this development should lead to a substantial reduction in basic orebody development and improved productivity underground.

The overall position in the 1100 orebody is extremely complex. With so many considerations to be taken into account, in what is a constantly changing and evolving situation, evaluation of extraction strategies will have to be both ongoing and flexible.

ACKNOWLEDGMENTS

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CHAPTER 11

Strássa Mine

K.-A. Björkstedt

INTRODUCTION

Strássa lies in the central part of Bergslagen, a traditional mining district, on the eastern side of the Storå Valley at an elevation of about 200 m (650 ft) above sea level. A railway siding runs between the mine and the Storå railway station from which there are railway connections to the shipping port and iron and steel works in Oxelösund, about 224 km (140 miles) away. The distance to the provincial capital Örebro is about 60 km (37 miles). The climate typical for this part of central Sweden is illustrated by the diagram of monthly precipitation and temperatures for the years 1968–1975 (Fig. 11.1).

HISTORY

There is no certain information as to when the Strássa mine was first worked, but it is known from surviving accounts of mine inspectors that there were smelting works in operation in nearby villages in the twelfth century. An example is the Gusselhytta ore smelting works, 10 km (6 miles) south of Strássa, which dates from this period. Around the year 1540 there were two smelting works in Strássa, the Upper Kårberg and Lower Kårberg works. Ore for these smelters was probably taken from Strássa and from the adjacent Blanka mine. In the year 1624, Strássa was mentioned by the painter Jöns Nils Krook in an account of the iron mines in the Linde mining district (Linde Bergslags Järngruvor). Several mines were listed in the area, the deepest being about 30 m (100 ft). An impressive power installation is mentioned in 1639, including a piston system of lashed poles for transmitting power from the Storå River to the Strássa fields. Its length was 2670 m (8750 ft). Common ground comprising about 20.2 km^2 (5000 acres) of forest was allocated in 1689 for the furtherance of mining operations.

Until the beginning of the twentieth century only the rich central parts of the orebody were mined, and these yielded, after handpicking, lump ore suitable for smelter feed. An example of the ore grades from these early times is an

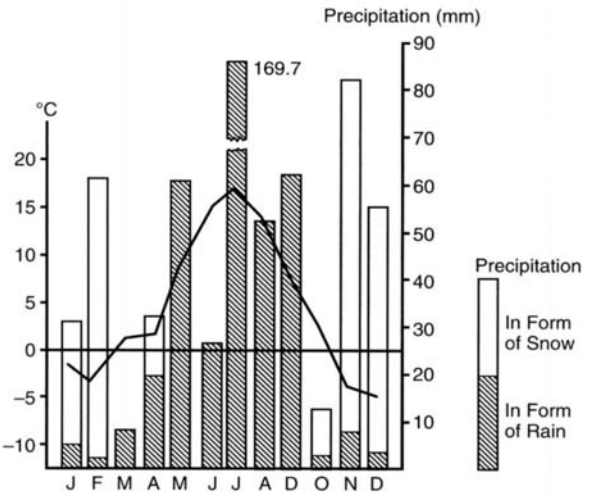


FIGURE 11.1 Diagram of monthly precipitation and temperatures for the years 1968–1975, English equivalent: mm × 0.039 = in.

analysis of ore from the "Big Mine" (Storgruvan) from the year 1873: 48.5% Fe, 0.008% P, and 0.06% S. This same year a total of about 18,000 t (19,800 st) was extracted from the Strássa mine.

OWNERSHIP

The mine was owned and run until 1874 by a mining association made up of 119 so-called "bergsmän," who were homesteaders often engaged in agriculture and timber-cutting as well. In that year the Strossa Grufvebolag (Mining Co.) was founded. In 1906 it was converted into a joint stock company, the Strossa Gruveaktiebolag. This was acquired in 1907 by Metallurgiska AB for the implementation of Gustav Gröndal's beneficiating and briquetting methods, for which the Strássa ore was well suited. The same year saw the completion of a new ore dressing plant with an annual production of 46,000 t (50,700 st) of ore concentrate. In 1911 the mine passed to new hands, and in 1913 it was purchased by an Austrian company. Extensive new installations were made, and in 1915 a new dressing and briquetting plant was completed with twice the capacity of the old one. In 1917 the Strássa mine was acquired by Gränges. Because of unfavorable business trends and technical difficulties, mining operations were brought to a close in 1923.

Pumping kept the mine free of water until 1933, but it was completely filled 10 years later. Up to 1950 the surface buildings and installations remained intact, but the large dressing and briquetting plant burned to the ground in that year. Only the machine shop remains from this earlier period of operation, now housing parts of the Mineral Processing Laboratory.

The decision to take up mining operations again was made in 1955, and construction work began the following year. Of the old installation, only the "southern shaft" could be used for some development drifting after it had

TABLE 11.1 Development of the mine's production

	Tons (st)		
1858–1867	20,224	(22,287)	
1868–1877	69,157	(76,211)	Lump ore with about 45% Fe
1878–1887	39,803	(43,863)	
1888–1897	53,251	(58,683)	
1898–1907	101,644	(112,012)	
1908–1917	1,441,268	(1,588,277)	Concentrating plant 1898 Production of concentrate and briquettes
1918–1923 (6 years)	1,161,526	(1,280,002)	
1859–1968	10,277,637	(11,325,955)	Production of concentrate and pellets
1969–1978	15,027,450	(16,560,249)	

been completed with a new headframe. Otherwise, all the buildings and installations required for the operations had to be rebuilt. New installations ready by 1960 were office and personnel facilities, a new shaft and headframe, a sorting and concentrating plant, a macadam plant, settling basins, pump stations, and a railway and yard with transport equipment. The installation was completed with two plants for pelletizing, the first in 1963 and the second in 1966.

Table 11.1 shows the development of the mine's production during some periods of its history.

Two smaller mines in the district have also been operated from Strássa: Blanka, which is nearby, and Mangruvan, which is 15 km (9 miles) to the northwest of Strássa. Blanka is connected with Strássa by drifts on the 225-m (725-ft) level. No mining has taken place in Blanka since 1969, and Mangruvan has been closed since 1972.

GEOLOGY OF THE STRÅSSA REGION

The iron ores of the Strássa region belong to the leptite series, which constitutes the main part of the area. To the south and east, the leptite series is delimited by a synorogenic gneissose granite. Late orogenic granite and pegmatite intersperse the leptite series and the gneissose granite. In the Strássa area the leptite series consists of metarhyolite, metadacite, mica schist, fine-grained mica-poor gneiss (leptite), gneiss with cordierite, sillimanite, and almandine, as well as dolomite and calcite marble, associated with skarn rock. The marble belongs to a zone that can be followed from the district of Nora to about 1 km (0.62 mile) southwest of Strássa (see Fig. 11.2).

The iron ores in Strássa are quartz-banded ores with transitions to varied skarn ores. In the Blanka mine, which is situated 2 km (1.2 miles) to the south of Strássa, the ores correspond to those in Strássa. This mine has been worked since the sixteenth century, despite the fact that the ore area is only a fraction of that of the Strássa field. In the district of Håkansboda, 1.5 km (1 mile)

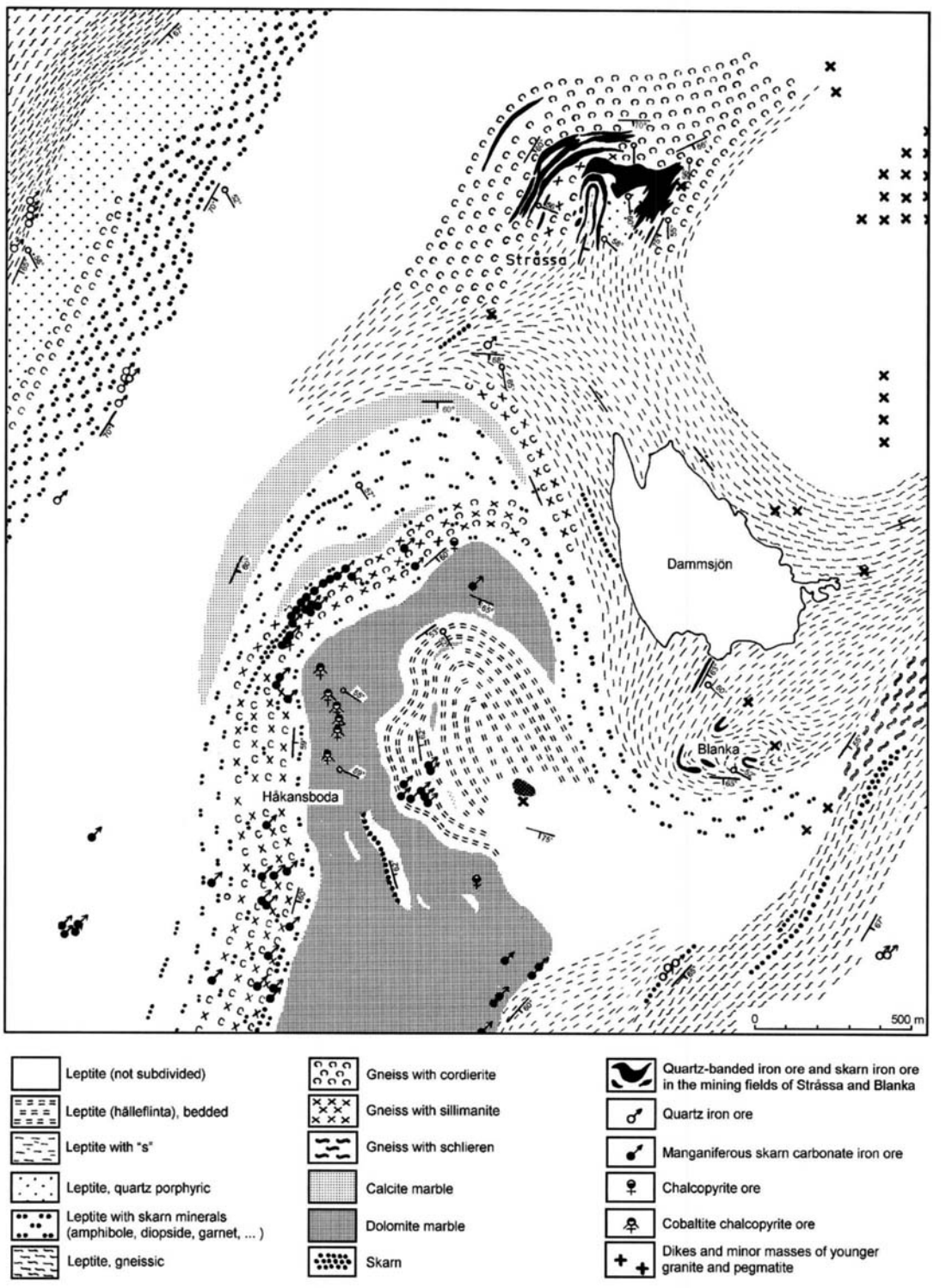


FIGURE 11.2 Strássa-Håkansboda-Blanka region

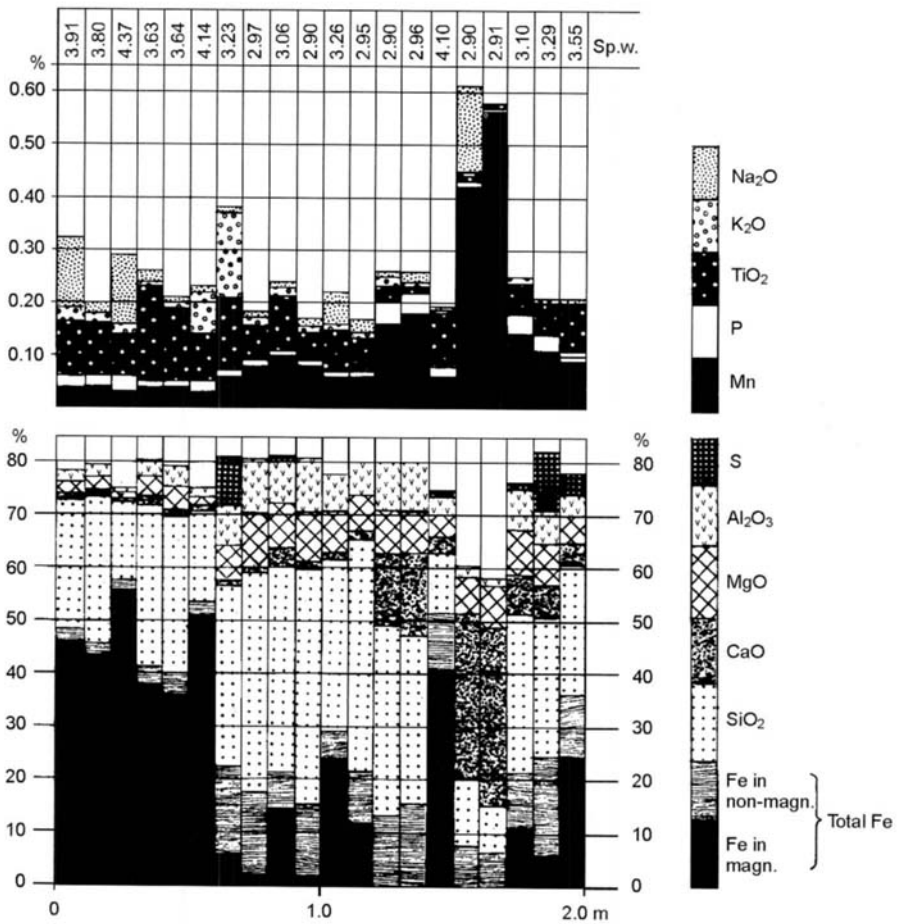


FIGURE 11.3 Analysis of 10-cm (4-in.)-long sections in a 2-m (6.6-ft) drill hole on the 195-m (640-ft) level. The iron content over the entire length is 29.5% Fe, and the mean specific weight is 3.4.

southeast of Strássa, manganiferous skarn iron ores occur accompanied by the marble. A chalcopyrite-cobaltite ore has been mined earlier in the dolomitic marble of Håkansboda.

Magnetite and hematite are the main minerals in the Strássa ore. The magnetite ore is usually homogeneous and bedded and is only occasionally quartz banded. Hematite is rare and occurs only as martite. Gangues consist of quartz, feldspar, phyllosilicates, amphibole, and occasionally cordierite and sillimanite.

Some hematite ores, which are comparatively homogeneous in appearance, are rich in quartz and poor in skarn silicates. The larger mass of the hematite ores, however, is stratified and of heterogeneous mineralogical composition (see Fig. 11.3). The reason for the mineralogical heterogeneity is the varying content of the different skarn silicates (amphibole, diopside, epidote, sometimes garnet) and the phyllosilicates (chlorite, biotite, muscovite). Magnetite usually occurs in the hematite ore as a metamorphic reduction product.

TABLE 11.2 Chemical analysis of crude ore and final products, percentages

	Crude Ore	Coarse Concentrates	Pellets
Fe_3O_4	26.5	5.0	0.5
Fe_2O_3	13.5	83.5	94.4
MnO	0.13	0.08	0.20
CaO	1.8	0.90	0.20
MgO	3.0	0.70	0.35
Al_2O_3	5.1	0.95	0.40
SiO_2	47.0	8.0	3.8
TiO_2	0.17	0.20	0.03
V_2O_5	<0.005	<0.005	<0.005
P_2O_5	0.045	0.025	0.007
S	0.025	0.10	0.001
Na_2O	0.45	0.04	0.03
K_2O	1.35	0.06	0.05
Cu	0.005	0.015	0.003
Total	**99.08**	**99.58**	**100.01**
Fe	28.5	62.0	66.5
Mn	0.10	0.06	0.15
P	0.020	0.015	0.003
V	<0.005	<0.005	<0.005

Quartz banding is more usual than in the magnetite ores. However, the quartz-banded iron ore types constitute less than 15% of the total ore volume of the deposit.

In some places stratiform barite bodies with magnetite impregnations have been found, as well as local intercalations of chalcopyrite, pyrite, and fluorite. These latter minerals occur even in fractures and fault zones, where bornite also may be formed.

The average iron content throughout the whole area is about 35%. The chemical composition of the ore is roughly stated in Tables 11.2 and 11.3. The analysis for these tables includes about 20% waste rock from gangues and wall-rock workings. The wall rock consists of cordierite gneiss, mica schist, rhyolitic metatuffite, and gneiss leptite. Gneisses with sillimanite and almandine are less frequent as are marbles.

The ore of the Strássa field can be divided into several stratigraphic units. They are separated by intercalations of cordierite and sillimanite gneisses, metarhyolites, and leptites. From the hanging wall to the foot, the units are Kronort-Kilort, Diamanten unit; Östergruve-Nygruve unit; Storgruve unit; and Konsum ores.

These units consist of different ore types. The Östergruve-Nygruve unit as well as the Storgruve unit shows the complexity of ore combinations previously mentioned The Kronort-Kilort-Diamanten unit mainly consists of quartz-banded ores, often rather rich in skarn minerals. In the footwall of the Storgruve unit the Konsum ores consist either of schistose hematite or homogeneous magnetite.

The ore and the other supracrustal basement within the area have been subject to at least two main folding phases, an earlier one with rather flat axes and a later one with steep axes. In the area between Riddarhyttan and southwest of Nora, these resulted in the formation of a synclinorium. The Stråssa ores are located in a syncline belonging to this synclinorium, which is overturned to the west.

The tectonic pattern in Stråssa is fairly complicated and sometimes confusing. The lens-shaped Storgruve ore forms a slightly synform structure whose axis plunges to the SE [0.69 to 1.22 rad (40 to 70°)]. The Nygruve-Östergruve unit in the upper parts of the mine shows an open fold, which becomes an isoclinal synform below. The axis plunges to SW-S [0.87 to 1.22 rad (50 to 70°)]. The axial plane, which is folded in the SW, is accompanied by schistosity above all in the NE. Here the axial plane schistosity likewise has influenced a slumping breccia ore. B-B' structures are common. Large-scaled crossfolds exist on the lower levels of the Nygruve-Östergruve ore and Kronort-Kilort ores. Kornort-Kilort ores lie in an overfolded isoclinal synform. Down to the 135-m (440-ft) level they consist of two layers, below which a single orebody is formed. Figure 11.4 shows the tectonic style and the distribution of the ore units on some levels.

MINING AND EXPLOITATION FROM 1959 TO 1980

When operations resumed in 1959, a decision had to be made as to a suitable mining method, and numerous factors had to be weighed. There remained from past operations several open pits, irregular underground benches, and to a lesser extent, shrinkage stopes. The deepest level reached was 135 m (440 ft). These conditions had to be considered when deciding the future mining method. Furthermore, the Stråssa ore has good strength properties, which make it possible to drive large drifts and open large rooms. The ore has a low iron content, approximately 35% in place. This meant that the mining method had to be inexpensive and efficient. The country rock inclusions and especially the pegmatite gangue required that the admixture of rock from the hanging wall had to be kept under control. Careful mapping and supervision of the pegmatite dikes are necessary so that the raise and cutoff openings can be planned in the correct way.

The quartz content of the ore is high, and this requires a mining method for which effective ventilation can be guaranteed.

After considering these factors, the mine management reached the conclusion that sublevel stoping would be a suitable choice. This method has been practiced ever since, although modifications have successively been made as experience has been gained. The method employed will be described in greater detail.

TABLE 11.3 Analysis of ores by type, percentages

	Homogenous Magnetite Ore	Homogenous Hematite Ore	Quartz-Banded Magnetite Ore	Quartz-Banded Hematite Ore	Hematite Magnetite Ore with Intercalations of Skarnoide and Leptite
SiO_2	35.8	38.7	33.0	33.7	41.6
TiO_2	0.10	<0.05	<0.05	<0.05	0.12
Al_2O_2	4.3	1.8	0.7	1.4	4.6
Fe_2O_3	36.5	49.7	45.3	53.0	28.7
FeO	15.6	5.52	19.7	3.97	14.9
MnO	0.13	<0.05	0.18	<0.05	0.45
CaO	2.2	0.5	0.4	4.0	3.1
MgO	2.5	2.4	1.0	2.9	2.5
Na_2O	0.20	0.25	0.13	0.09	0.04
K_2O	0.50	0.25	<0.10	0.10	0.50
P_2O_5	0.042	0.030	0.010	0.018	0.025
S	0.006	0.004	0.008	0.010	0.050
BaO	0.04	0.02	<0.01	0.06	0.05
V_2O_3	0.005	0.003	0.002	0.003	0.009
B_2O_3	<0.001	<0.001	<0.001	<0.001	<0.001
Ignition loss 1000° corr	2.4	1.0	0.5	0.8	3.8
SO_3					
SrO					
Total	**100.3**	**100.2**	**100.9**	**100.1**	**100.4**
Fe tot	37.6	39.0	47.0	40.2	31.6
Cr*	0.010	0.015	0.009	0.015	0.010
Ni	<0.0010	<0.0010	<0.0010	<0.0010	<0.0010
Co	0.0038	<0.0010	0.0021	<0.0010	<0.0010
Mo	<0.001	0.001	<0.001	0.002	0.001
Cu	<0.001	0.002	0.003	0.002	0.095
Pb	<0.001	<0.001	<0.001	<0.001	≤0.001
Sn	<0.001	<0.001	<0.001	<0.001	<0.001
Ga	<0.0011	<0.0010	<0.0010	<0.0010	0.0019

*Presumably defiled on grinding by Cr-steel.

Analyzed by Sveriges Geologiska Undersökning, Stockholm.

	Magnetite-Bematite Ore with Intercalations of Skarnoide and Leptite	Porphyroblastic Magnetite Ore	Magnetite Ore in Diopside-Aktinolite Skarnoide	Magnetite-Hematite Ore in Amphibole Skarnoide	Magnetite-Baryte Ore
SiO_2	36.2	34.7	31.5	36.2	7.7
TiO_2	0.15	0.06	0.12	0.06	0.08
Al_2O_2	4.2	2.4	4.5	2.0	1.0
Fe_2O_3	48.3	43.8	35.3	46.0	19.7
FeO	1.75	13.6	15.3	9.86	8.22
MnO	0.08	0.14	0.12	0.06	0.33
CaO	2.6	0.9	3.3	1.4	1.8
MgO	4.0	3.0	5.5	3.4	2.5
Na_2O	0.13	0.37	0.10	0.21	0.09
K_2O	1.1	0.20	0.75	0.45	0.09
P_2O_5	0.045	0.031	0.062	0.025	0.013
S	0.003	0.006	0.055	0.003	See SO_3
BaO	0.03	<0.01	0.03	0.04	33.7
V_2O_3	0.004	0.005	0.004	0.003	<0.002
B_2O_3	<0.001	<0.001	0.020	<0.001	<0.001
Ignition loss 1000° corr	1.5	1.8	4.0	1.0	2.9
SO_3					20.3
SrO	<0.01	<0.01	<0.01	<0.01	0.30
Total	**100.1**	**101.0**	**100.6**	**100.7**	**98.7**
Fe tot	35.1	41.2	36.6	39.9	20.3
Cr*	0.015	0.008	0.010	0.014	0.001
Ni	<0.0010	<0.0010	<0.0010	<0.0010	<0.0010
Co	<0.0010	0.0034	0.0064	0.0038	0.0022
Mo	0.001	0.001	0.001	0.001	0.003
Cu	0.002	0.002	0.003	0.001	0.003
Pb	<0.001	<0.001	<0.001	<0.001	<0.001
Sn	<0.001	<0.001	<0.001	<0.001	<0.001
Ga	<0.0010	0.0010	0.0013	<0.0010	<0.0010

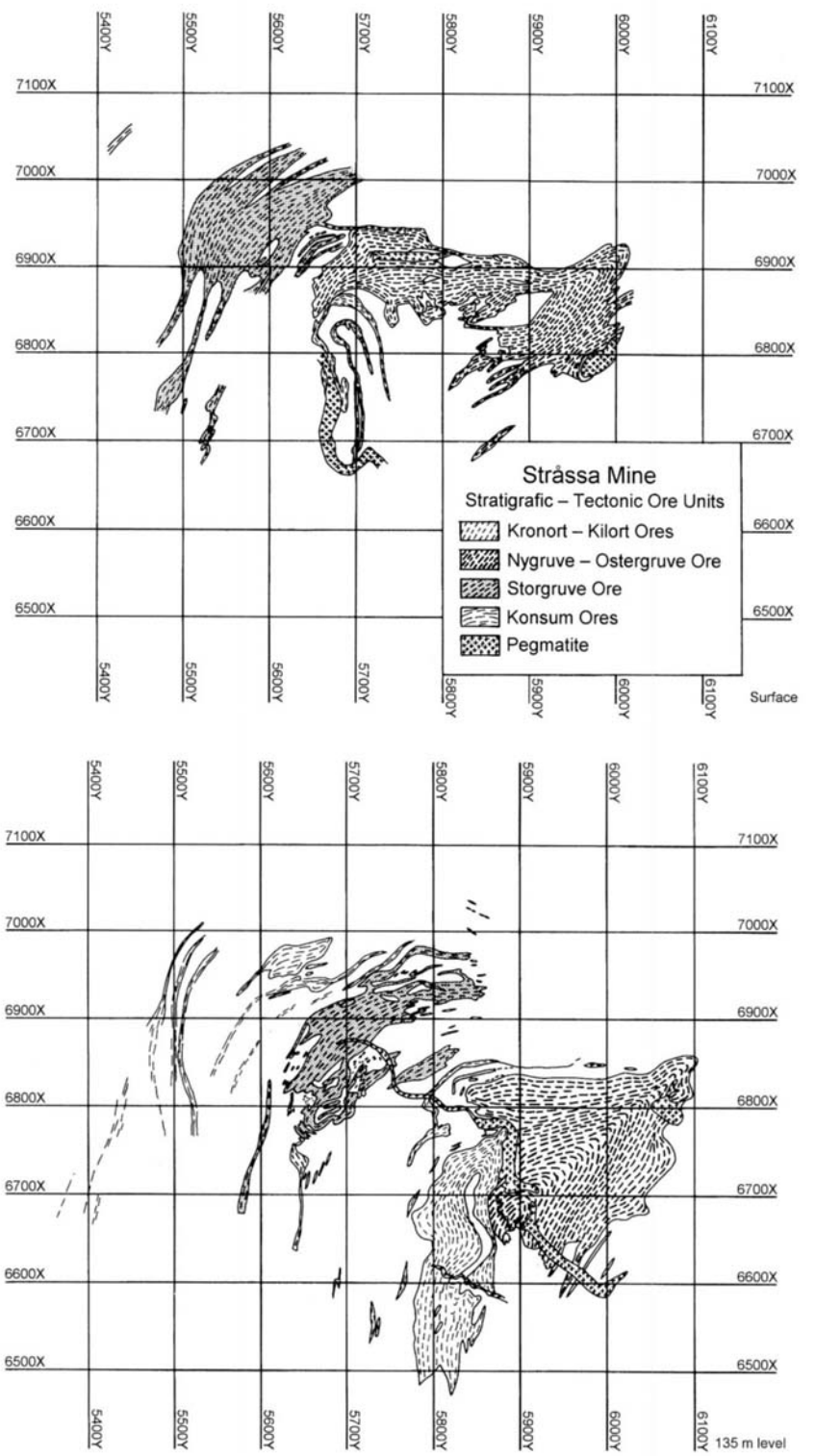
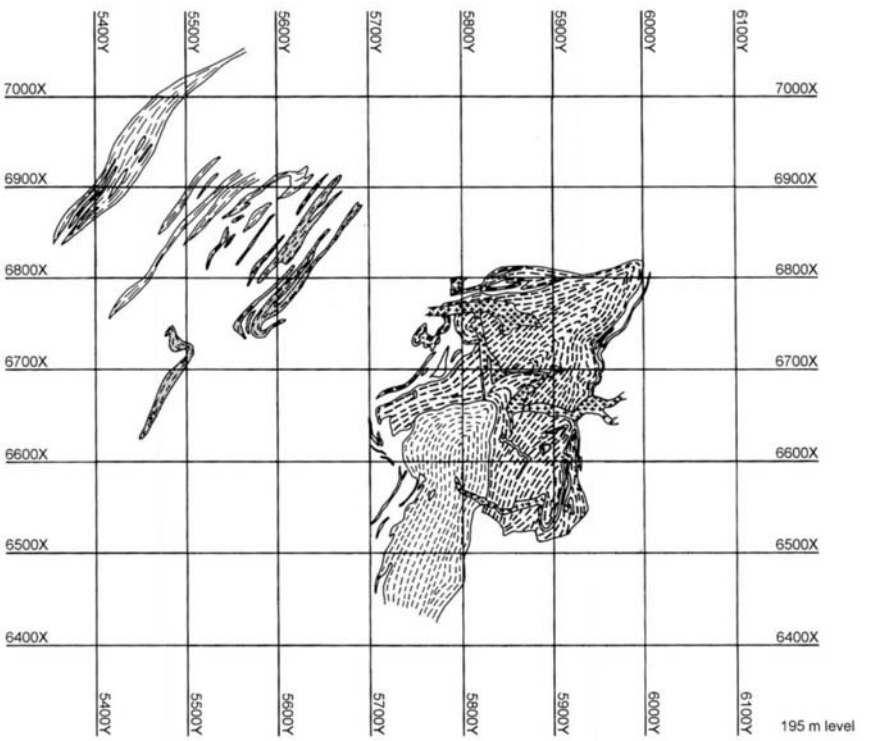
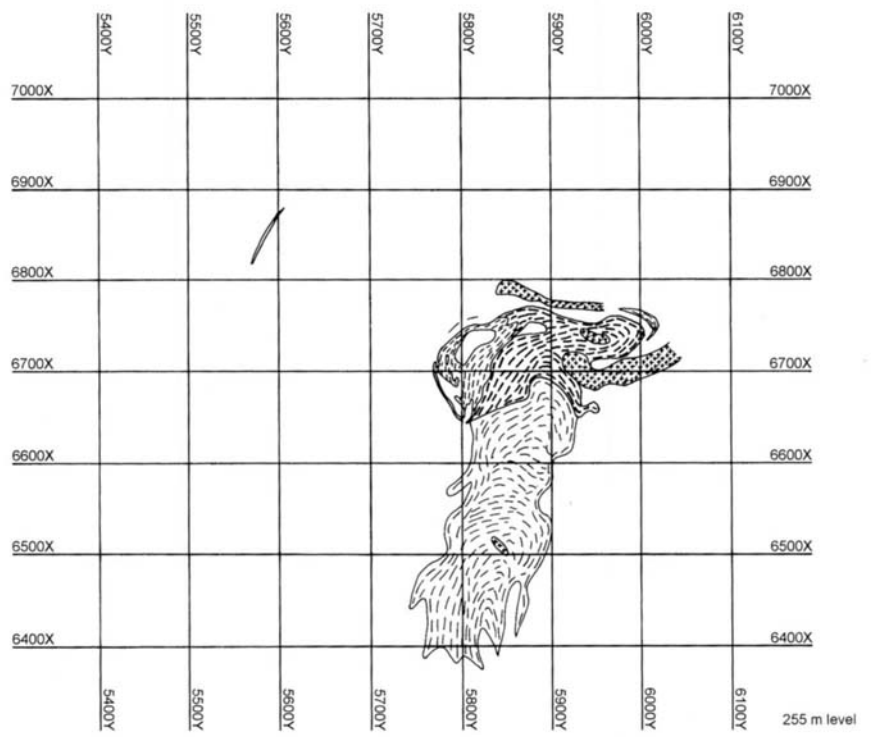


FIGURE 11.4 Strássa mine, showing the tectonic style and the distribution of ore units on some levels


5400Y
5500Y
5600Y
5700Y
5800Y
5900Y
6000Y
6100Y
7000X
6900X
6800X
6700X
6600X
6500X
6400X
195 m level


5400Y
5500Y
5600Y
5700Y
5800Y
5900Y
6000Y
6100Y
7000X
6900X
6800X
6700X
6600X
6500X
6400X
255 m level

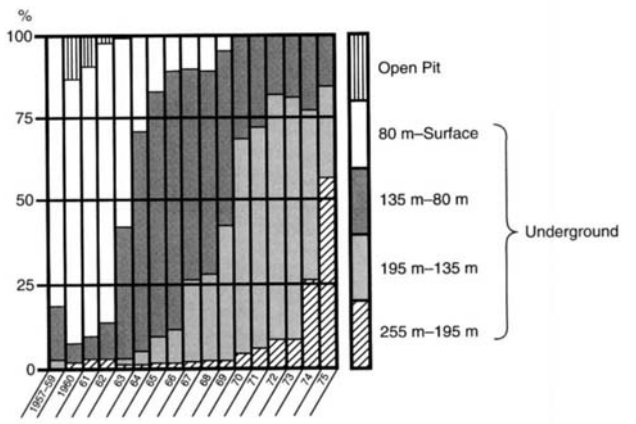


FIGURE 11.5 Distribution of mining activity by level

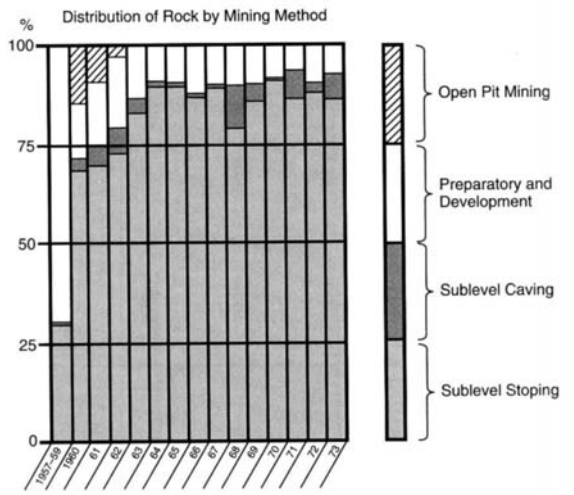
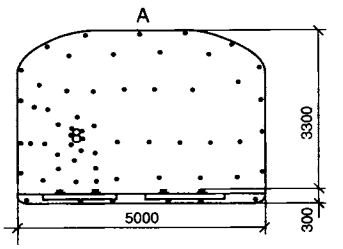


FIGURE 11.6 Effect of distribution of rock on mining method

When mining operations began in 1959, the upper sublevel from the 80-m (260-ft) level to the open pit was subsequently worked with sublevel stoping and to a certain extent by open-pit mining. The distribution of mining operations with regard to the different methods and different sublevels is shown in Figs. 11.5 and 11.6.

MINING METHODS, 135-M TO 195-M LEVEL

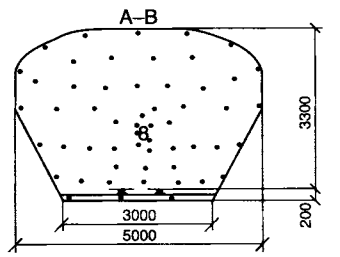
The slice of ore between the 135- and 195-m (440- and 640-ft) levels is divided into transversal stopes 22 m (72 ft) in width with an intermediary pillar 12 m (40 ft) wide. During the first phase, a 10-m (33-ft)-thick roof is left over the sublevel stope. Development for this mining method proceeds as follows; the drift shapes and sizes referred to can be found in Fig. 11.7.



Haulage drift on main level

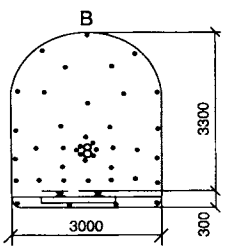
Theoretical cross section 16.9 m^2
Advance 2.7
Qty drill holes: 60 + 2 large holes, Ø 31 mm
and, Ø 76 mm

Drill meters per meter of drift: 69
Kg dynamite per meter of drift: approx. 24
Drilling with rig ZAX 202 or 203 and rock drill BBC 34



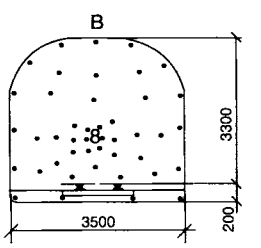
Drilling drift on main and intermediate levels

Theoretical cross section 14.2 m^2
Qty drill holes: 54 + 2
Drill meters per meter of drift: 54
Kg dynamite per meter of drift: approx. 20
Drilling with rig ZAX 202 or 203



Loading drift on main levels

Theoretical cross section 9.7 m^2
Qty drill holes: 41 + 2
Drill meters per meter of drift: 49
Kg dynamite per meter of drift: approx. 16
Drilling with pusher leg BMT 50 and rock drill BBD 45 or BBD 51



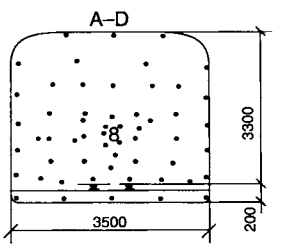


FIGURE 11.7 Drift sizes and shapes, English equivalent: m × 3.2808 = ft

On the 195-m (640-ft) level, a footwall drift (cross section A) is driven along the strike in the country rock. From the footwall drift, drilling drifts (cross section A-B) and haulage drifts (cross section B) are driven across the orebody at center-to-center distances of 17 m (55 ft). The drilling and haulage drifts alternate. Joining these drifts are loading drifts (cross section B) at center-to-center distances of 16 m (52 ft). They are scissored on each side to permit loading from two directions. The haulage drifts are driven in the hanging wall with cross section B to make it possible to load trains of nine cars from the loading drift nearest the hanging wall. Similarly, the connecting drifts between the footwall and the drilling drift have the B cross section and single track.

An intermediate level was developed at 170 m (550 ft) intended for drilling the sublevel stopes and for drilling the pillars between the stopes.

Directly above the drifts on the 195-m (640-ft) level, a drilling drift (cross section A-B) is driven, and above the haulage drifts a communication drift is driven (cross section C). At right angles to the latter, a drilling drift (cross section A) is driven over the width of the pillar every 34 m (112 ft) to permit the pillar to be drilled up. The drilling drifts in the pillar are driven up to the stope so that they can also serve as control drifts when blasting the stopes.

On the 143-m (470-ft) level is an intermediate level for drilling up the roofs left over the sublevel stopes. The communication drifts (cross section C) are driven directly above the corresponding drifts in the pillar on the 170-m (550-ft) level. Every 34 m (112 ft), directly above the drilling drifts in the pillars, drilling sites are prepared for drilling up the roof [cross section A, approximately 5 m (16 ft) long].

The cutoff opening is at the hanging wall and is 4 m (13 ft) wide. A raise is first driven along the center line of the projected stope. A Jora raise lift is used for this operation. To improve safety at the cutoff opening, loading drifts are opened on each side of the cutoff slot on the 195-m (640-ft) level.

Mining of the stope can now be started with drilling and blasting toward the raise with fan-patterned holes from the 195-m (640-ft) level and parallel holes from the 170-m (550-ft) intermediate level. The procedure is illustrated in Fig. 11.8.

After the first cutoff round, parallel fans are drilled on both the 195- and 170-m (640- and 550-ft) levels. A great deal of care is taken in lining up and drilling these fan patterns; hole alignment is to one decimal point and the length of the holes is indicated in both number of rods and in meters. Figure 11.9 shows an example of a drilling pattern for fan drilling.

Drilling is performed with Atlas Copco Simba 25 rigs and with two BBC 120 rock drills. The rig is operated by one person. The burden is 1.6 m (5.2 ft). Normally three to four rows are fired at the same time, which brings down a quantity of about 18,000 t/round (19,800 st pr round). The mining operations move backward toward the footwall, leaving as large a roof area as possible. Blasting in the stopes on the 170-m (550-ft) intermediate level is a

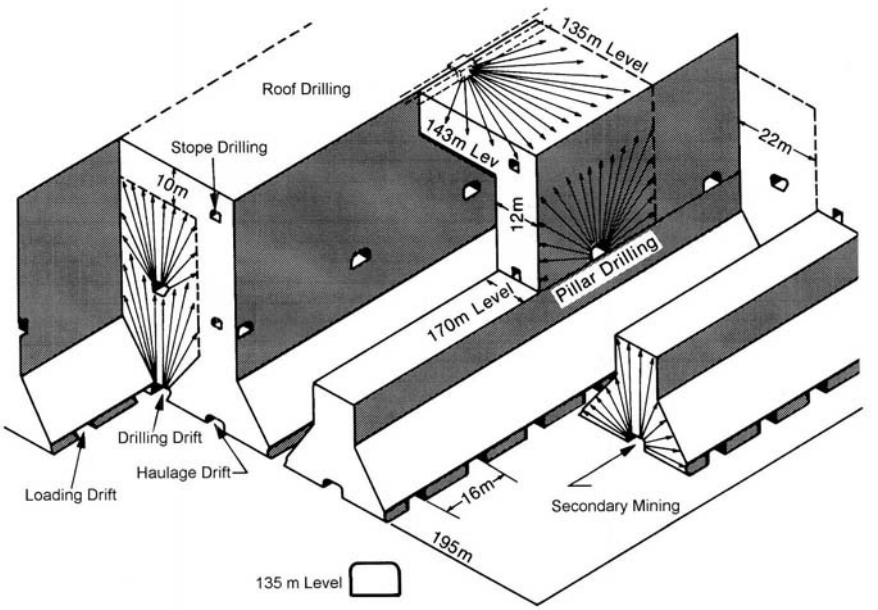


FIGURE 11.8 Procedure for mining of the stope with drilling and blasting toward the raise with fan-patterned holes from the 195-m level and the 170-m intermediate level, English equivalent: m × 3.2808 = ft

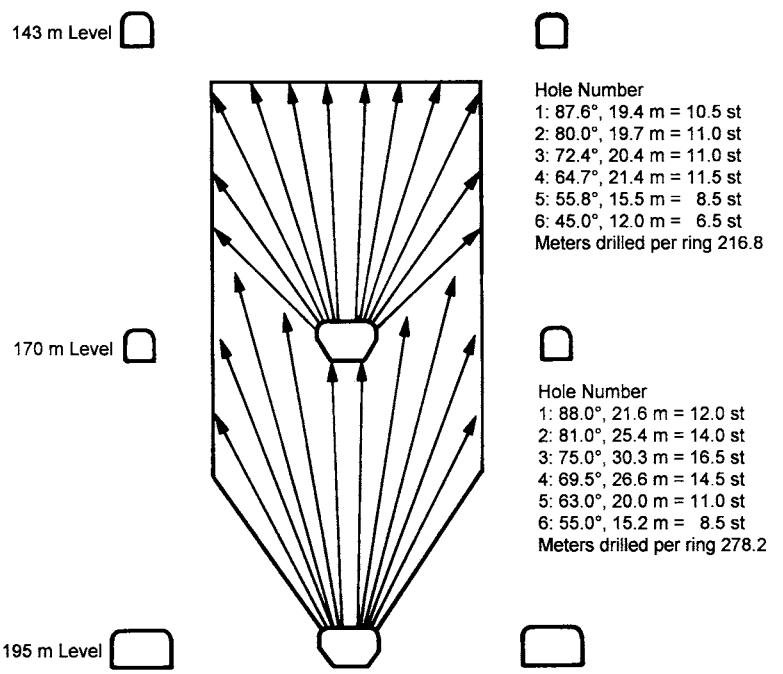


FIGURE 11.9 An example of a drilling pattern for fan drilling, English equivalent: m × 3.2808 = ft

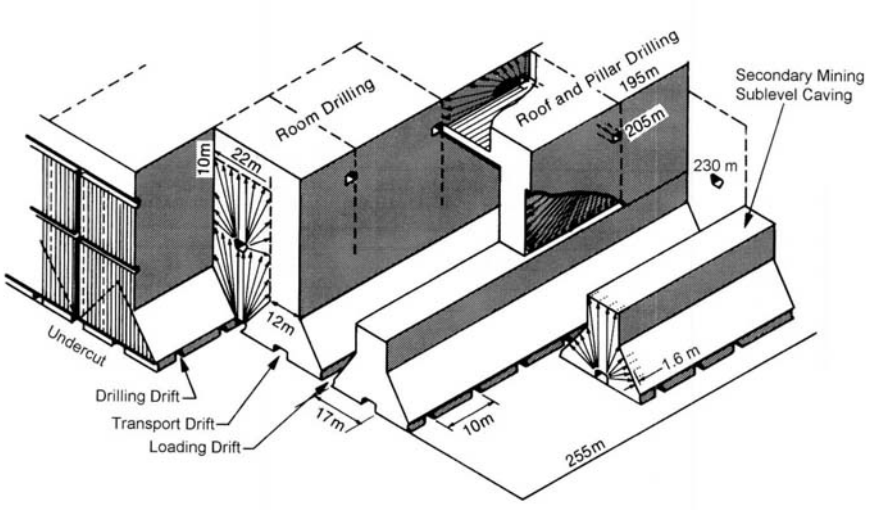


FIGURE 11.10 Procedure for the mining method for the 195- to 225-m level, English equivalent: m × 3.2808 = ft

couple of rows ahead of operation, so that the drilling drifts toward the stope are always covered by rock.

Mining goes on in adjacent stopes at the same time, but operations are somewhat staggered. When mining is completed in two adjacent stopes over a certain distance, the intermediate pillar and the roof above the last-mined stope are blasted.

The communication drifts for this roof-blasting are situated over the next pillar, which means that it is not necessary to work on a free-standing pillar on the 143-m (470-ft) level.

Roof drilling is performed with a special drilling rig, designed and manufactured by the mechanical staff of the mine. The drilling pattern consists of fans inclined from −0.26 to 1.04 rad (−15 to +60°).

Finally, the remaining pillars over the haulage drifts on the 195-m (640-ft) level are mined with sublevel caving (see Fig. 11.8).

MINING METHODS, 195-M TO 255-M LEVEL

The procedure follows in the way previously described with the following important modifications (see also Fig. 11.10). The distance between the loading drifts is reduced from 16 to 10 m (53 to 33 ft), which makes possible more efficient extraction of the ore. Cutoff drilling is done with parallel blastholes, which are left uncharged along the bottom perimeter of the stope. Drilling in the stope is then with fan patterns, as described earlier. The drilling up of roof and pillars differs here from the previous system in that drilling is now carried out from four drifts along the strike on the level of the roof, 205 m (670 ft),

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and the roof holes are now drilled in vertical parallel fans and along the longitudinal axis of the stope. In the pillars, downward directed fans are drilled. Residual pillars are mined in the normal way with sublevel caving.

To drill the fans on the 230 m (750 ft)-intermediate level, where trackless equipment is employed, the drilling rigs are mounted on chassis with rubber tires and skids. The entire rig can in addition be swiveled on the chassis for accurate alignment of the plane of the fans.

Special equipment has been designed for drilling the roof and pillar holes. Two COP 130 rock drills with feeds are mounted on a wheeled chassis and can be separately advanced in relation to each other. The rig can drill from +1.57 to −1.57 rad (+90 to −90°). Settings can be adjusted in the horizontal plane as well so that holes can be aligned parallel to the longitudinal axis of the stope and pillars.

However, the problem with this method is that when blasting the roof and pillars of the first section, the caved rock lying above the stope will mix with the ore (see Fig. 11.11). To avoid this, the method has been changed to that shown in Fig. 11.12. After blasting and emptying the first section, a cross pillar with a width of 8 m (26 ft) is saved. A new slot is made in the next section, and the mining goes on as described previously. In this way, it is possible to extract the ore without any dilution. The method was put into practice in mid-August 1978, resulting in a higher percentage of iron (see Fig. 11.13). Because of the higher density and better fragmentation, the productivity has increased from 8 to 11 t (8.8 to 12.1 st) per staff-hour to 10 to 13 t (11.0 to 14.3 st) per staff-hour (see Table 11.4). Simultaneously, the consumption of explosives for secondary blasting has considerably decreased (see Table 11.5).

TECHNICAL EQUIPMENT

The technical execution of drifting is illustrated in Fig. 11.7. It should be noted that ZAX 202 and 203 are operated by one person. A typical working cycle for a drift with cross sections A and A-B is as follows [with setup time and downtime included; drilling with ZAX-202 and mucking with LM 250 4 m^3 (5.2-cu yd) Granby cars; 3.2-m (10-ft) drill steels; and an advance of approximately 2.7 m (8.8 ft)]:

	Cross Section A of drift			Cross Section A–B of drift		
	min/m	(min/ft)	%	min/m	(min/ft)	%
Drilling	94.4	(28.8)	29.6	88.5	(27.0)	31.7
Charging, blasting, and ventilation	65.0	(19.8)	20.4	58.0	(17.7)	20.7
Loading	133.9	(40.8)	42.0	111.8	(34.1)	40.0
Hauling	25.8	(7.9)	8.0	21.3	(6.5)	7.6
Total	318.8	(97.2)	100.0	279.6	(85.2)	100.0
m of drift/8-hr shift		1.5			1.72	

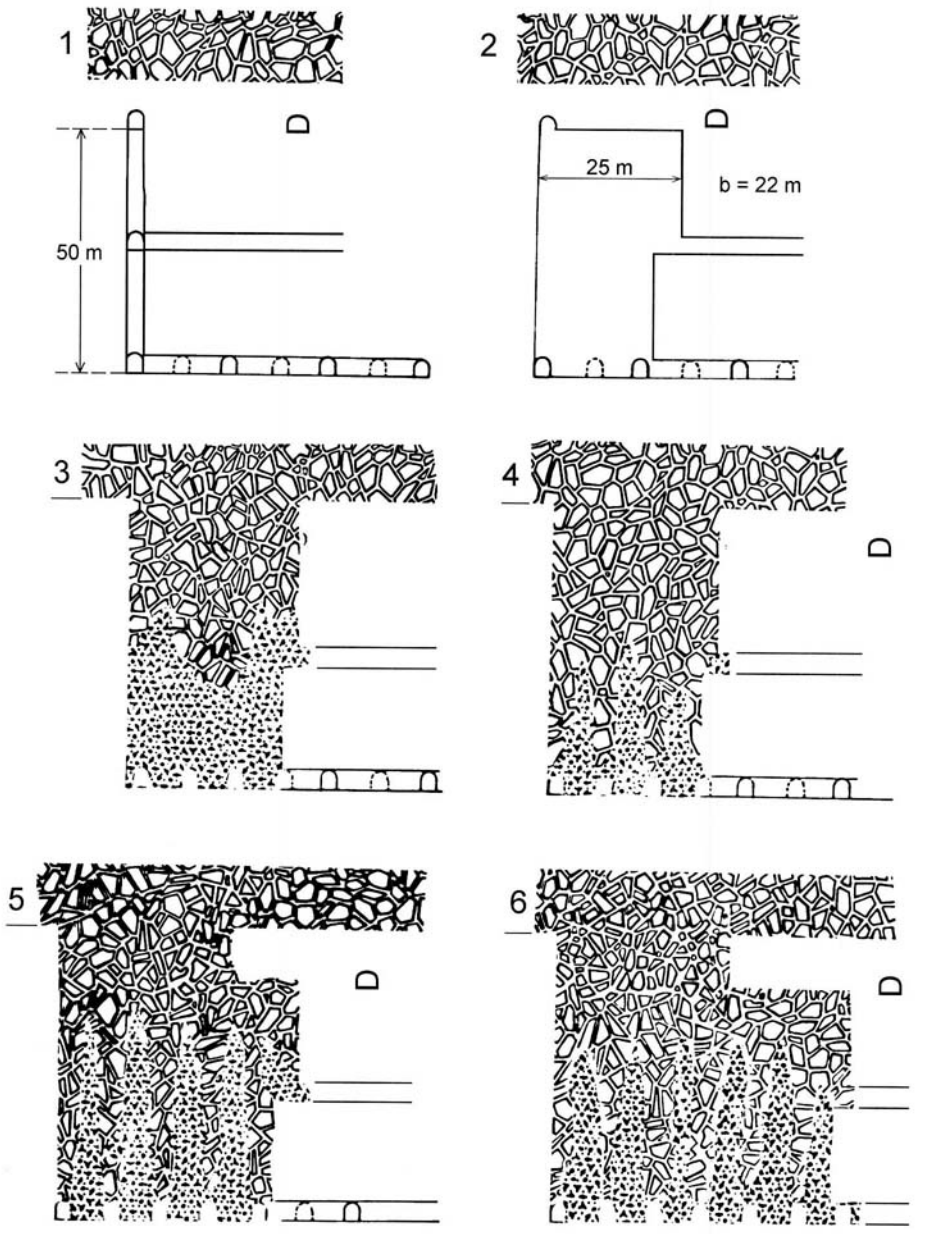


FIGURE 11.11 Original method

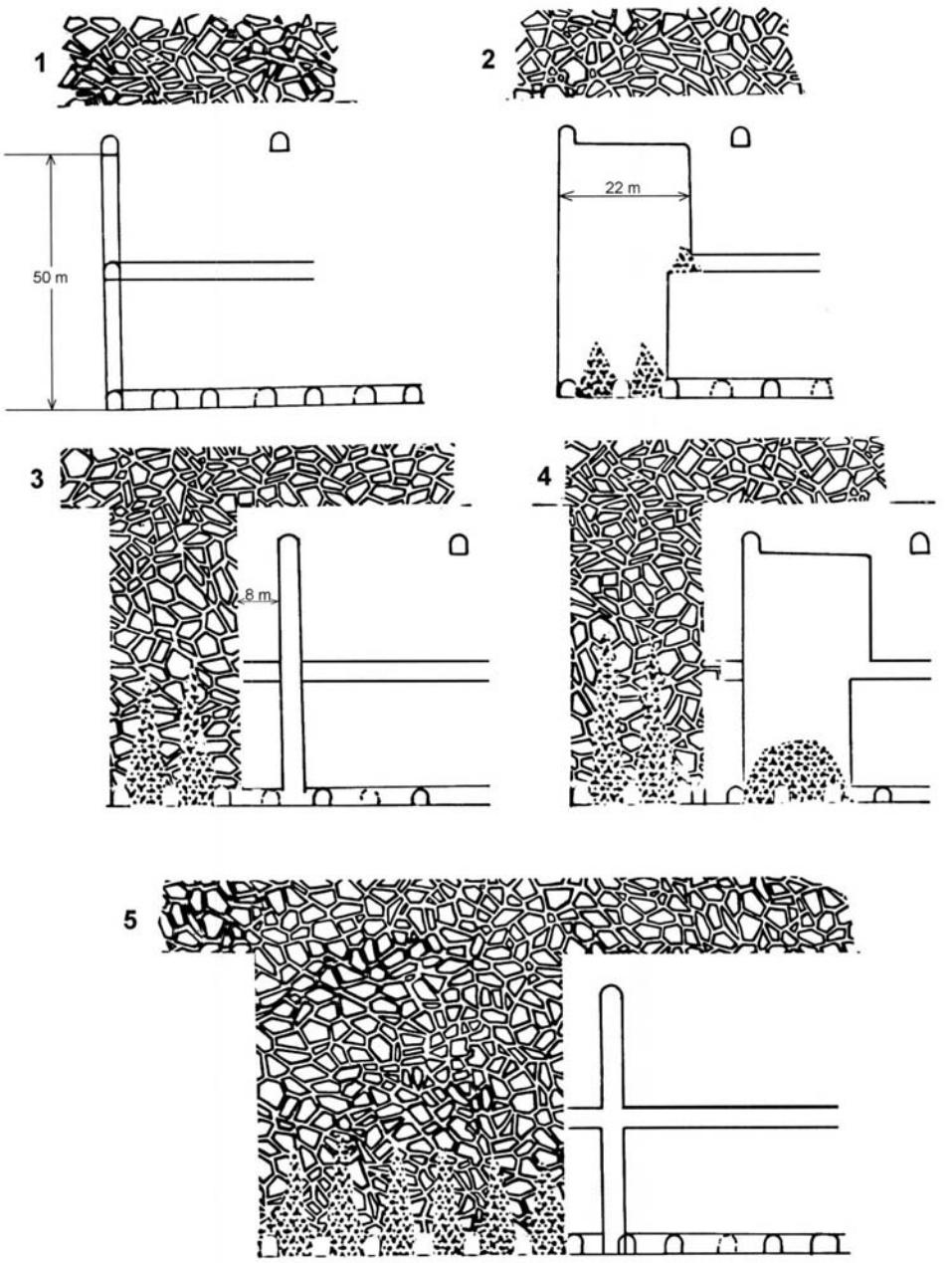


FIGURE 11.12 Cross-pillar method

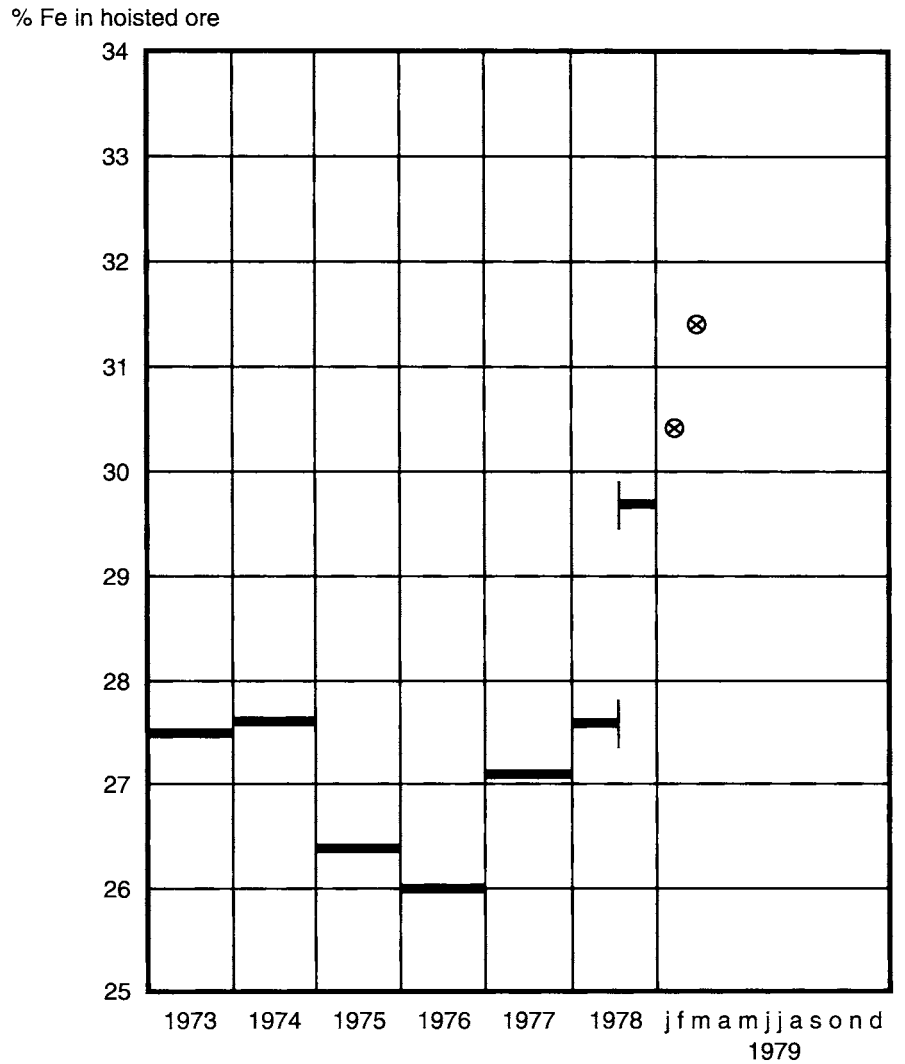


FIGURE 11.13 Percent iron in hoisted ore following cross-pillar method

Ammonium nitrate–fuel oil (ANFO) is now used to a great extent in drifting. Charging is carried out with a Jet-Anol unit. Dynalit, however, is used in the "toe holes" and in many cases dynalit column charges are used in the roof holes. In 1975 ANFO was employed in 76.3% of the drifting operations.

About 10% of the drifts have been driven with cross section A, 19% with cross section A-B, 17.5% with cross section B, 31% with cross section E, and 23% with cross section A-D.

The average service life for drill steels used in drifting in 1978 was as follows: 4-m (13-ft) drill steel with 35-mm ($1\frac{3}{8}$-in.) bit, 274 drill m (900 drill ft); cut hole steels with pilot rod, 58 drill m (190 drill ft); and cut hole steels with ring bit 76 mm (3 in.), 700 drill m (2300 drill ft).

TABLE 11.4 Productivity underground [t (st) per person-hour]

1973		10.3	(11.4)
1974		9.1	(10.0)
1975		10.2	(11.2)
1976		11.1	(12.2)
1977		10.7	(11.8)
1978	January	7.5	(8.3)
	February	10.9	(12.0)
	March	11.0	(12.1)
	April	10.2	(11.2)
	May	8.3	(9.1)
	June	9.9	(10.9)
	August	9.4	(10.4)
	September	10.6	(11.7)
	October	12.0	(13.2)
	November	12.6	(13.9)
	December	13.5	(14.9)
1979	January	11.3	(12.5)

TABLE 11.5 Secondary blasting, explosive consumption, loaded rock [kg per 1000 tons (lb per 1000 st)]

1975		41.5	(83.0)
1976		41.2	(82.4)
1977		28.0	(56.0)
1978	January	27.3	(54.6)
	February	23.8	(47.6)
	March	22.0	(44.0)
	April	37.0	(74.0)
	May	33.6	(67.2)
	June	28.3	(56.6)
	August	35.7	(71.4)
	September	20.5	(41.0)
	October	16.6	(33.2)
	November	15.3	(30.6)
	December	17.3	(34.6)
1979	January	15.0	(30.0)

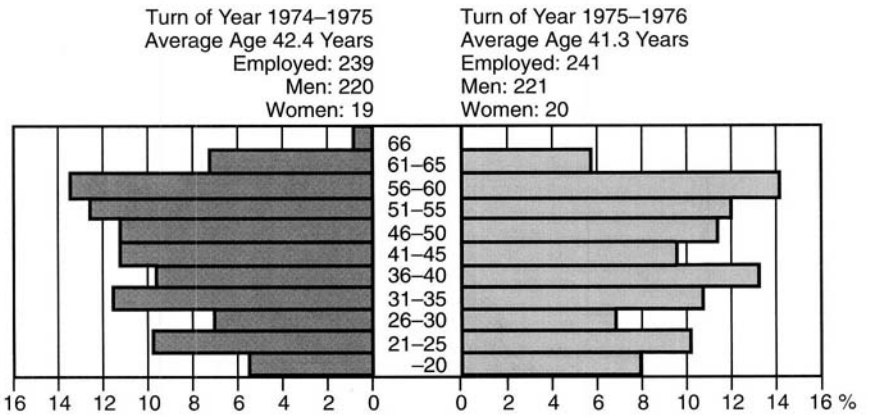


FIGURE 11.14 Age distribution of all employees, 1974–1975 and 1975–1976

The share of drifting work and allied activities in the total number of work hours performed is indicated in Fig. 11.14. By way of example, a breakdown of drifting costs [in 1975 Swedish krona (Skr)] is given in Table 11.6 for a drift with cross section A.

Permanent track laying is carried out by a special crew. The track gage is 750 mm (29.25 in.), and the rails are in sizes 24.8 kg/m (16.7 lb per ft) for haulage and 43 kg/m (30 lb per ft) at loading places. Especially on the main levels, the track system is relatively extensive and includes four switches at every system of crosscuts (see Fig. 11.15). The switch and switch elements are standardized for a 20-m (66-ft) curve radius and with switch tongues 2.5 or 1.2 m (8 or 4 ft) in length.

TABLE 11.6 Breakdown of 1975 costs* for drifting (cross section A)

	Skr/m of Drift	Totals	%
Wages			
Drilling, charging, blasting	85.87		
Loading, hauling	92.32		
Laying track	33.40	211.59	26.3
Material			
Explosives (Prillit 78:20, dynamite 18:44, fuses 50:02)	146.66		
Drill steel	59.71		
Rails, ties, ballast	138.00	344.37	42.7
Repairs and Maintenance			
Drill rig	92.84		
Loader (LM250)	19.01		
Locomotive, cars	15.21	127.06	15.8
Compressed Air (0.076 kr/kWh)			
Drill rig	50.50		
Loader (LM250)	72.13	122.63	15.2
Total		805.65	100.0

*In early 1980, 1 Skr = U.S. $0.24, English equivalent: m x 3.2808 = ft.

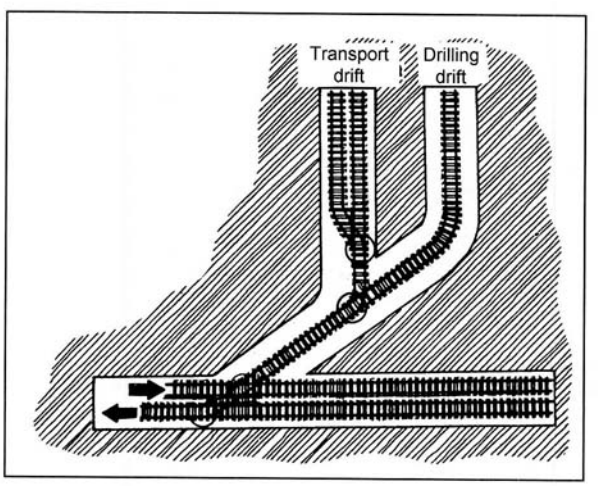


FIGURE 11.15 Diagrammatic view showing track system in transport drift and drilling drift

The rounds in the drifts are mucked out with an LM 250 loader into 4-m^3 (5.2-cu yd) Granby cars. The intermediate level at 230 m (750 ft) has been driven tracklessly by two BUT 10 hydraulic booms on a T4G loader chassis. The rig is equipped with COP 90 rock drills and 25-mm (1-in.) drill steels in 4-m (13-ft) lengths. The rock was loaded with a Cavo 511 unit.

With this equipment, the following performance figures were noted in 1972: drift = 0.33 m/h (1 ft per hour); loading = 20.6 t/h (22.7 stph). These

TABLE 11.7 Drilling material costs* (1978 figures)

	Life Length Costs	
	m†	Skr/Drill Meter
Drill bits	377	0.84
Extension rods	241	1.62
Coupling sleeves	100	1.17
Shank adapters	524	0.62
Total		4.25

*In early 1980, 1 Skr = U.S. $0.24.

†English equivalent: m × 3.2808 = ft.

calculations are based on 844.7 m (2770 ft) of completed drift. About 42.9% of the total time was employed in loading and hauling. The majority of the drifts were with cross section A-B.

RAISE DRIVING

As noted earlier, a raise is driven at the cutoff opening for each stope up to the intermediate level at 230 m (750 ft). From the intermediate level up to the roof level at 250 m (820 ft), only one raise is driven and the entire cutoff is fired toward it. The raises are driven with the Jora raise lift with advances of about 2.2 m (7 ft). The rock drill is a BBD 46 with telescopic feed and drill steel in lengths of 1.2 and 2.4 m (4 and 8 ft). The dimensions of the raises are 2 × 2 m (6.5 × 6.5 ft). The following figures can be drawn from several years of statistics: output = 1.54 m (5 ft) per gross shift; dynamite = 13.4 kg/m (9 lb per ft) raise.

An Alimak raise climber is normally used in other raise driving. In 1972, a ventilation raise was driven to the surface in this way. It was about 166 m (550 ft) long with dimensions of 2.5 × 2.5 m (8 × 8 ft). Driving figures were: drift = 1.64 m (5.4 ft) per gross shift; dynamite = 16.9 kg/m (11.4 lb per ft) raise.

LONGHOLE DRILLING

An Atlas Copco Simba 25 track-bound mechanized longhole drilling rig is employed, operated by one person. The rig is equipped with two heavy rock drills of type BBC 120. BBC 120 is being replaced in stages with COP 130. Special warm-air devices have been installed to eliminate water mist during drilling. With three such rigs normally in operation, the average number of drill meters per year is 200,000 (650,000 ft).

Drill steel equipment consists of Sandvik Coromant 31.7-mm (1 ¼-in.) hexagonal extension rods with tandem threads and with 48- and 51-mm (1 ⅞- and 2-in.) four-point bits. Average life lengths and costs for equipment in 1978 are shown in Table 11.7.

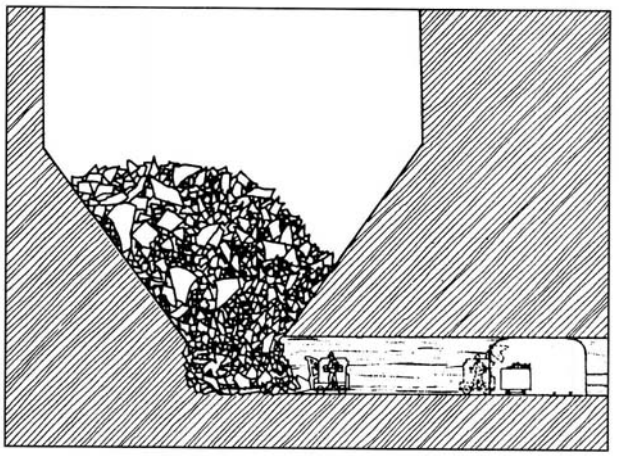


FIGURE 11.16 Vertical section showing loading of ore from stopes

The average output for the Simba 25 rig is about 150 m (500 ft) per 8-hr shift. The effective drilling time, including setup time, is about 90% of the total working time; time for repairs is about 7%, and miscellaneous time is 3%.

The fans are charged with Nitro-Nobel's automatic charger. Output with this equipment is about 700 kg (1500 lb) per shift with a crew of two.

For initiating large blasts, the Tri Electronics AB firing system TS 100 is used. With this system, the effectiveness of the blasting has increased and the number of misfires has been reduced to a minimum.

PRODUCTION LOADING

Loading is effected with track-bound Atlas Copco LM 250 units into trains made up of nine Granby cars, each with a capacity of 4 m^3 (5.2 cu yd), which corresponds to about 6.8 to 7.2 t (7.5 to 7.9 st) per car of ore. The train is pulled by two accumulator locomotives, one on each end. The locomotives are joint-controlled, rated at 10 kW (13.4 hp) each with a working voltage of 70 V. The general layout of the loading points is shown in Figs. 11.16 and 11.17.

At several of the loading points, a cable has been installed in the track so that the locomotive can be remote controlled from the loading drift to move the cars ahead.

Trials had also been made with the remote control of loaders. This can come to have great importance for the elimination of the accident risks of falling rock and the like. The results of these trials had not been completely positive as far as output and general functioning were concerned, but the tests were continued.

The loader's work consists primarily of loading, moving the cars forward, haulage to the crushing station, and arch control and secondary blasting.

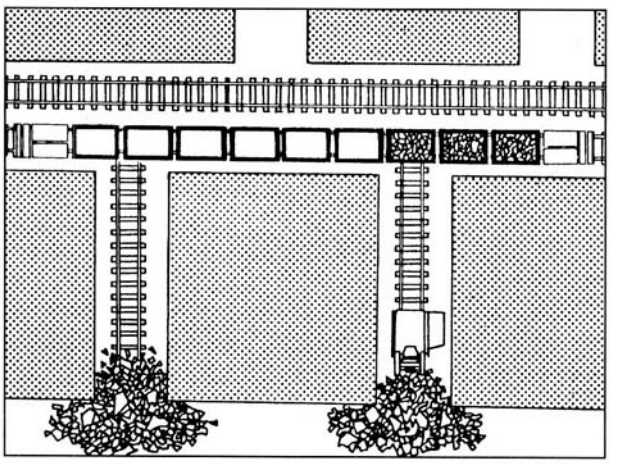


FIGURE 11.17 Plan view showing loading operation

In 1975 this last item and miscellaneous work accounted for 12.0% of the total loading time. Output for the same period was 64 t/h (70.5 stph) including all the phases of operations previously mentioned. In terms of shifts, the output is about 450 t (500 st). The loaders can call on the help of a mining service crew, two people per shift who assist with the more difficult arch control, tearing up and adjusting tracks, local scaling, etc. The consumption of explosives in mining operations in 1975 was about 0.246 kg/t (0.5 lb per st) of explosive including secondary blasting. About 17.1% of the dynamite was consumed in secondary blasting and arch control.

Of the explosives used in mining, 95.6% were nitroglycerin-based explosives (Mitex, Dynamex, Gurit) and 4.4% AN prills.

HAULAGE

The loader also looks after the transportation of the rock to the crushing station. The accumulator locomotives, two for each set of cars, have a lead accumulator or a Nife accumulator. A battery usually lasts one shift. There are recharging stations on each main level.

After dumping into the crusher, the set of cars is moved by radio control. The loader controls both the dumping cylinder for the cars and the advance of the set of cars from a platform above the crusher, where the entire process can easily be supervised. In this connection, the control of switches with the aid of pneumatic cylinders can be mentioned. On the wall of the drift about 10 m (33 ft) before each switch is an air control that works the cylinder and moves the switch. The signal system in the drifts consists of lamps that are regulated by the locomotive drivers. By setting a green line for their track, they block off the crosscut drifts with a red light.

The Granby cars, capacity 4 m^3 (5.2 cu yd), are now all equipped with a rubber lining along the bottom of the bin. Previously, loading with the LM 250 units proved very hard on the cars and occasioned frequent repairs to the bins.

During 1975, 56.5% of the rock was drawn from the sublevel between 255 and 195 m (840 and 640 ft), 28.0% from 195 to 135 m (640 to 440 ft), and 15.5% from 135 to 80 m (440 to 260 ft) (see Fig. 11.5).

CRUSHING

On the main levels where development and production are currently in progress, coarse ore crushers have been installed. Such crushing stations are found on the 135-, 195-, and 255-m (440-, 640-, and 840-ft) levels.

The crushers are of the Svedala jaw crusher type TRG 150120 R with an intake opening of 1500 × 1200 mm (58½ × 46¾ in.) and can crush up to 250 mm (9¾ in.). The capacity is about 350 t/h (380 stph). The coarse crushers are all connected to a common ore chute, which discharges just above the 255-m (840-ft) level. Here two rotary crushers, Symons Standard 2 m (7 ft) have been installed, each with a capacity of 400 t/h (440 stph). The intake opening is 340 mm (13¼ in.) and the smallest crushed piece is 32 mm (1¼ in.).

Maintenance costs per ton (st) in 1977 Swedish krona* were:

	Lining	Other
Jaw crushers	0.05 (0.06)	0.31 (0.34)
Rotary crushers	0.259 (0.285)	0.558 (0.615)

*In early 1980 the Swedish krona was equivalent to about U.S. $0.24.

The crushing station is supervised by TV monitors from the skipping station on the 285-m (935-ft) level. From the ore pocket under the crusher, the rock is fed by conveyor to the measuring bin where the skips are filled automatically. Transportation to the surface is also automatic. The layout of the shaft is given in Fig. 11.18.

The conveyor belt to the measuring bin is equipped with a scrap iron detector, which automatically stops the belt when refuse is detected in the ore. Data for the hoisting equipment in the shaft are summarized in Table 11.8. Note that the personnel skip is also automatic.

The headframe, which is 48 m (157 ft) high, houses a water tank for the industry and community as well as the ore bin and hoisting machinery.

PUMPING STATION

On the 255-m (840-ft) level there are a sump and a pumping station. The water inflow to the Strássa mine is quite modest; the annual average is about 2000 L/min (1800 qt/min). The installed pump plant is four centrifugal pumps of the

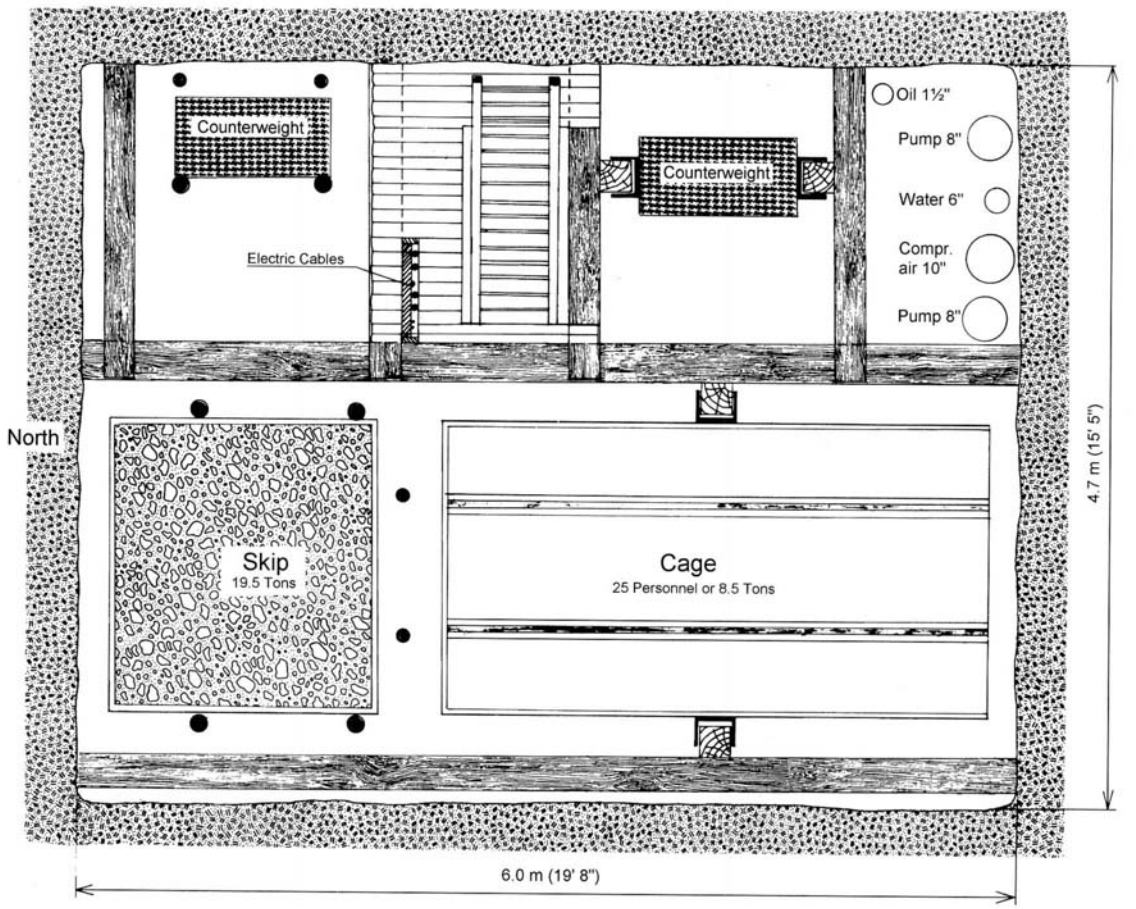


FIGURE 11.18 Layout of the shaft

TABLE 11.8 Data for the hoisting equipment in the shaft

	Ore Hoist	Personnel Hoist
Type	Koepe	Koepe
Skip and cage	Bottom-emptying, 19.5 t (21.5 st)	25 people
Motor	1500 kW (2010 hp) asynchronic	400 kW (536 hp) Ward-Leonard
Speed	9 m/s (29.5 fps)	3 m/s (10 fps)
Cables	4 × 32 mm (1¼ in.) Cables 4 + 2, 32 mm (1¼ in.) counterweight 24 t (26 st)	4 × 32 mm (1¼ in.) Wooden guides
Guides	Buffer lines 2 × 32 mm (1½ in.)	
Bottom cables	2 × 52 mm (2 in.), 266-ply	4 × 32 mm (1¼ in.), 114-ply

type NLGA IV, each with a capacity of 1500 L/min (1350 qt/min). With this overcapacity it is possible by automatic pumping during the night shift to exploit inexpensive electric power.

COMPRESSED AIR

The compressor center is above ground adjacent to the mechanical and electrical workshop, and has a capacity of 360 m^3/min (470 cu yd/min) of free air. By way of an aftercooler, each compressor is connected to an 8-m^3 (10.5-cu yd) air tank, from which compressed air is distributed in a 203-mm (8-in.) main line to the points of consumption above and underground.

VENTILATION

The high quartz content of the ore and accompanying silicosis risk make it essential to maintain good ventilation in the mine. The basic principle is to bring in prewarmed fresh air through the main shaft and older shaft, used during the earlier history of the mine (Fig. 11.19). Fans in the bulkheads by the shaft distribute the fresh air to the main level, where it is forced past the working places out into the caved rock. At points where a great deal of dust is formed, e.g., the crushing stations, there are special suction devices for the foul air, which is forced out through fan tubes to a special dust-evacuation raise.

By the rotary crushers on the 255-m (840-ft) level, an acoustic dust filter (INOMA) has been installed. The filter area is 192 m^2 (2067 sq ft). Crushers and screens have also been encapsulated. The filter now separates dust with a specific surface of 4320 cm^2/g and a SiO_2 content of 56.2%, of which approximately 66.5% is free SiO_2.

To reduce the formation of dust, especially during loading, water sprays have been installed over the caving muck at all work sites.

The air is continuously sampled at 20 places in the mine once a month. This sampling is made with the MSA Monitaire sampler. The underground personnel are X-rayed every second year.

The effective ventilation has also curtailed the radon and radon daughters contents of the air. Several series of tests have been made of the mine air on different levels and at different points in the mine. The radon daughter values normally lie between 2 to 8 pCi/L. Sporadically, because of ventilation problems, values about 20 pCi/L have been measured.

It should be noted that the hygienic limit value is 30 pCi/L. The uranium and thorium contents of the country rock are also low; analyses have shown that the average contents in the country rock are about 13 ppm of thorium and 6 ppm of uranium. No radioactive mineralization has been noted in the Strássa mine, although a uraninite-filled fissure was discovered in the Blanka mine in the early 1960s. Practically all of this uranium deposit, some hundred kilograms (220 pounds), has been mined, and now probably figures prominently in mineral collections around the country.

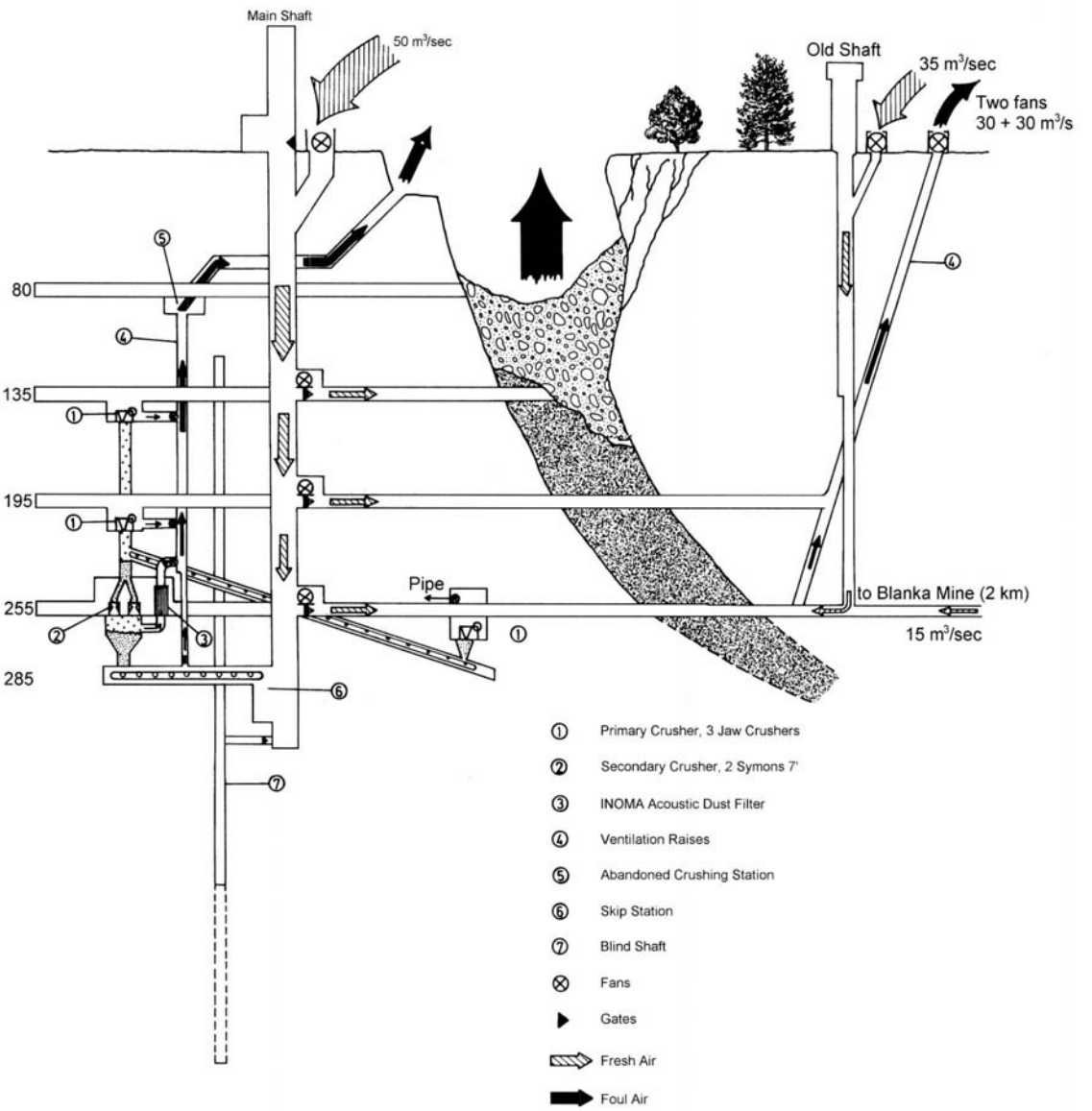


FIGURE 11.19 Ventilation system at Strássa mine

EXPLORATION

A continuous program of exploration is practiced, consisting primarily of diamond core drilling. The mine's own personnel drill with an Atlas Copco Diamec 250. Drill holes to a depth of 200 to 300 m (650 to 1000 ft) are drilled with this equipment. For longer holes, contractors are engaged.

About 2000 m (6600 ft) are drilled annually with the mine's own equipment. The output, inclusive of all hours, is 1.24 m/h (4 ft per hour). In terms of percent, the hours are broken down as follows: transportation 5.1%; setting up, 20%; core drilling, 73.2%; and miscellaneous, 1.7%.

Costs are distributed as follows: diamond drill bits, 12.1%; reaming shells, 4.2%; drill rods, 2.2%; repairs and material and working costs, 4.4%; assembly, materials, and transport, 4.4%; wages and drillers, 53.1%; and capital costs (five-year depreciation), 19.6%.

On occasion, magnetometric measurements are made in boreholes. The necessary chemical analyses are made in the Mineral Processing Laboratory in Strássa. Normally, exploration costs amount to about Skr 0.20/t, and about 70% of this figure is attributable to diamond core drilling.

ORE TREATMENT PLANTS

The crude ore from the mine has an iron content of around 28 to 30%, and the ore minerals are magnetite and hematite in a ratio of roughly 3 to 2. Secondary quantities of iron silicates are also present. The ore does, however, vary greatly in mineral composition.

PRINCIPLES OF SORTING AND DRESSING

On the basis of the mining method and the mineral composition, the following basic principles have shaped the ore treatment: two-stage crushing underground; common hoisting of ore and waste; separation of coarse waste with magnetic and wet mechanical methods in a sorting plant; and milling of the sorted crude ore and dressing with magnetic and wet mechanical methods in an ore dressing plant.

With this treatment, the crude ore is divided into four products: coarse waste, hematite concentrate, magnetite concentrate, and milled waste.

POWER SUPPLY

Strássa receives its electrical power from Grängesberg in a 50-kV line to an outdoor transformer station. Consumption in 1978 was as follows:

	kWh	%
Mine		
Hoisting	2,348,000	
Crushing, ventilation, etc.	4,921,000	
Compressed air	6,176,000	
Total, mine	13,446,000	20.6
Sorting, dressing plants	34,256,000	52.5
Pelletizing plant	16,231,000	24.9
Mineral Processing Laboratory, workshops, offices	1,364,000	2.1
Total	65,297,000	

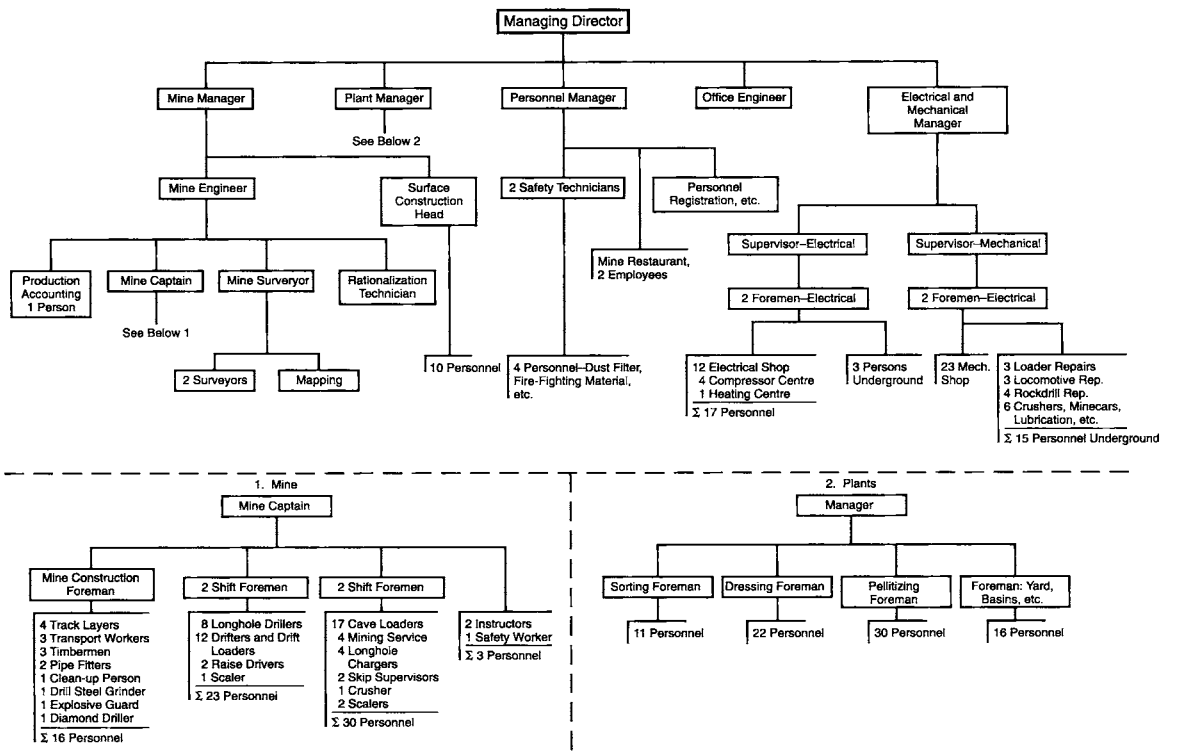


FIGURE 11.20 Organization chart for Stråssa mine, 1974. (The chart illustrates functions, not the relative levels of positions.)

PERSONNEL ORGANIZATION

The organization chart for Stråssa mine in 1974 is shown in Fig. 11.20. The total number of employees at the mine and plants at the beginning of 1979 was as follows:

Office employees and foremen	41*
Mine personnel	55
Sorting, dressing, and pelletizing plant	72
Electricians and mechanics	42
Above-ground construction	7
Security, store, etc.	14
Total	231

*Includes five part-time personnel.

As a result of continuing rationalization within the Stråssa unit and efforts toward coordination within the Gränges Mines group, there has been an interesting development in the personnel sector. With increases in overall efficiency, it represents a very satisfactory trend. This is illustrated in Table 11.9, which refers to the complete Stråssa unit.

TABLE 11.9 Developments in the personnel sector

Year	Office Employees, Foremen	Mine and Plant Personnel	Total	Tons (st) per Person per 8 hr Underground
1966	66	293	359	59.2 (65.2)
1967	59	271	330	64.0 (70.5)
1968	54	265	319	63.2 (69.6)
1969	50	258	308	69.6 (76.7)
1970	49	248	297	76.0 (83.8)
1971	47	249	296	80.8 (88.2)
1972	45	235	280	79.2 (87.3)
1973	44	232	276	82.4 (90.8)
1974	46	235	281	76.8 (84.6)
1975	49	241	290	81.6 (89.9)
1978	41	195	236	83.2 (91.7)

During 1978 the turnover for operating personnel was 10.8%. The corresponding figure for salaried employees was 2%. Statistics for absences due to illness were 13.4% for operating personnel and 4% for salaried personnel.

The accident rate at the Strássa industrial site can be illustrated by the following figures for 1978, which can be considered fairly representative.

	Number of Accidents	Accidents per 1 Million Working Hours
Above ground	5	25.9
Underground	5	54.1
Total	10	80

Most of the accidents were minor. To encourage greater attentiveness and protection consciousness in the workplace, all operating personnel participate in a bonus system based on accident-free days. Between 7000 and 8000 Skr are awarded to personnel annually on this basis.

Figure 11.21 gives a breakdown of ages for all employees above and underground for the years 1974–75 and 1975–76.

PAYMENT SYSTEM

The mine runs for the most part on the basis of two shifts a day between 6:00 a.m. and 10:10 p.m. About 30% of the personnel, mainly mine construction personnel whose working time is from 7:00 a.m. to 2:58 p.m., work the day shift underground.

Over the years, the system of payment has changed from a rather large volume of piece work to hourly wages with a merit bonus. The volume of

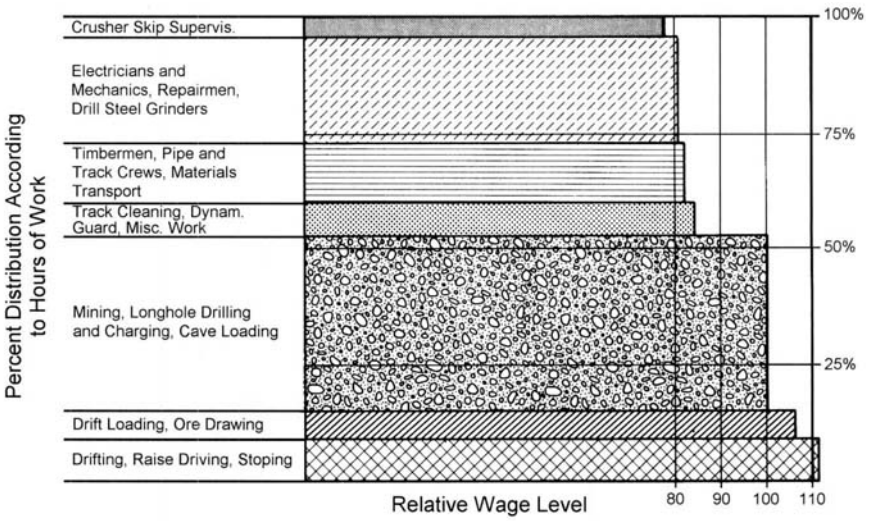


FIGURE 11.21 Percent distribution of various work functions and relative wage levels

TABLE 11.10 Production and relative production costs, Skr,* 1971–1975 and 1978

		Relative Costs per Ton Ore			Total Pelletizing		Production Ton† × 1000	
Year	Tons† Skipped × 1000	Common Costs	Mine	Sorting and Dressing Plants	Costs	Relative Costs per Ton†	Coarse Concentrates	Pellets
1971	1.717	16.3	47.8	35.9	100.0	100.0	211	455
1972	1.642	23.2	52.5	38.1	113.8	101.6	148	440
1973	1.676	23.6	57.6	37.1	118.3	110.5	160	472
1974	1.520	31.6	78.2	49.1	158.9	163.0	152	422
1975	1.534	37.4	96.7	62.5	196.7	193.4	132	449
1978	0.964	72.3	134.2	104.5	311.0	247.2	50	438

*In early 1980, 1 Skr = U.S. $0.24.

†English equivalent: t × 1.102 = st.

piece work is currently about 25% of all worker-hours underground. For Stråssa, both above and underground, the corresponding figure is 8%. The majority of underground work is loading caved muck, drifting, and longhole drilling.

For the relative pay rate for different jobs underground, see Fig. 11.14. The height of the column represents the volume of work in 1975. Table 11.10 gives a summary of production and relative production costs for the five-year period 1971–1975 and for 1978. Finally the percentage cost (1975) breakdown for the underground mine is shown in Fig. 11.22.

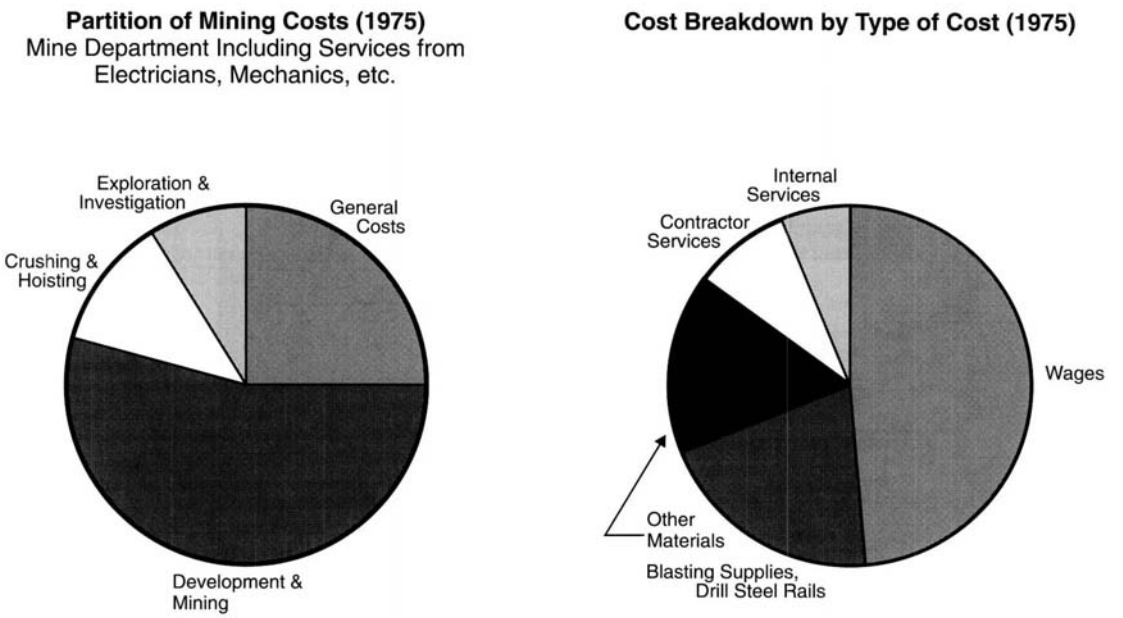


FIGURE 11.22 Percentage cost breakdown for the underground mine

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CHAPTER 12

Sublevel Stoping at the EB Massive Reefs at Loraine Gold Mines, Ltd.

R.D.H. Bebb

GEOLOGICAL NOTES

The area under discussion is situated approximately 150 m (500 ft) north of Loraine No. 3 shaft at ±1500 m (±5000 ft) below surface. The block being mined extends from about halfway between 50 and 52 levels to just below 48 level (see Fig. 12.1).

The EB reefs form the lower portion of the EB zone of the Elsburg series, and in the 81 line area the various conglomerate bands show the best development. Up to 18 separate reef bands are developed, and the distance from the bottom conglomerate band to the top is ±26 m (±85 ft). The thickness of the quartzite that forms the middling between the various reef bands varies from 0.1 to 2 m (4 in. to 6.6 ft). The strike is roughly north-south.

The reef bands are payable individually. However, owing to the steepness of the reefs and their close proximity, conventional mining would prove very difficult.

The nonpayable reef bands all carry varying amounts of gold, and the whole area is a massive payable reef body approximately 26 m (85 ft) wide and 60 m (200 ft) high at 300 to 400 m (1000 to 1300 ft) on strike.

The major structure of the area consists of a north-south trending syncline. The western flank of the syncline has a steep easterly dip, in places approaching the vertical. This flank cuts out against the overlying rocks of the EA zone. The latter forms a nonconformity with the underlying EB zone. The eastern flank of the syncline dips to the west at an average dip of 0.17 rad (10°).

The 0.1 rad (6°) north-plunging syncline was subjected to severe faulting. The most important are the Nos. 4 and 10 faults, both of which have a roughly north-south strike. The No. 4 fault dips to the west at an angle of 1.04 rad (60°) with a downthrow of approximately 150 m (500 ft) to the west. The dip

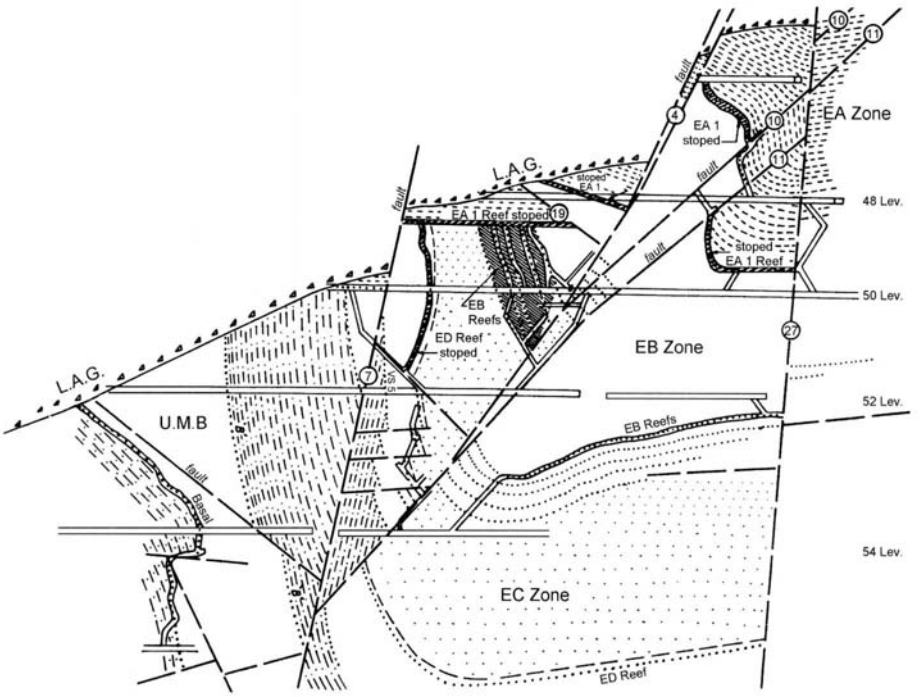


FIGURE 12.1 Geological section in the 81 line area looking north

of the No. 10 fault is 0.69 to 0.81 rad (40° to 50°) west, with the western block having moved 150 m (500 ft) updip relative to the eastern block.

The EB reefs are being mined by the sublevel stoping method down from the suboutcrop position to the cutoff against the No. 4 fault. On the downthrow side of the No. 10 fault, which is just above the 54 level, two reef bands are payable and will be mined conventionally.

ROCK MECHANICS

From the section in Fig. 12.1, it can be seen that the area under consideration is surrounded by stoped-out areas. This will naturally change the stress distribution extensively. The only way to calculate the stress field for such a geometry is to use the finite element method. It was established that this method worked for the present geometry and so the method was applied to predict the stress behavior of the final geometry.

From the results, it appeared that the effect of the old stopes was to destress the area.

There are, however, some areas—such as the 50 and 48 north footwall haulages—that will require continuous attention. In these two footwall haulages, deterioration can take place due to tensile stress, which acts at right angles to the bedding planes. Because of this, frequent observation is maintained.

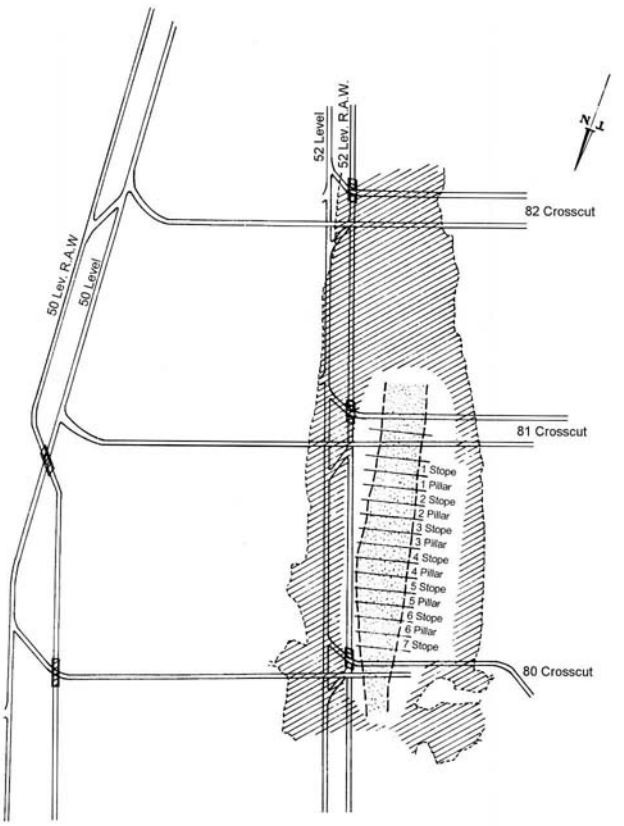


FIGURE 12.2 Locality plan

It is anticipated that the hanging wall of the old EA1 stope above (see Fig. 12.2) will collapse, so the mining method is being kept flexible as a contingency measure.

STOPING CONSIDERATIONS

The middling between the various payable reef bands varies between 1 and 6 m (3.3 and 19.7 ft), the bands themselves being about 1.35 m (4.4 ft) thick.

An important consideration was the occurrence between the various payable reef bands of several minor conglomerate developments with less gold content.

If only five reef bands were mined by core stoping, a certain gold content would have remained in situ. It was also considered that it may not have been possible to do a full extraction of the five main bands due to the difficult mining operation.

Extensive horizontal diamond drilling had been done on the No. 3 sublevel to test the limits and payability of the deposit. The values obtained were subsequently proved accurate by the stope development.

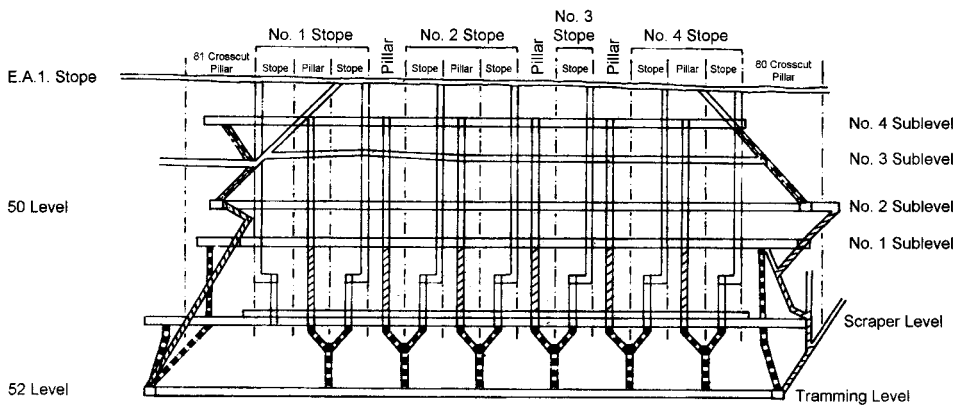


FIGURE 12.3 A section showing the development of the stoping layout

The following values were established: average values of individual bands ±25 g/t (0.8 oz/st) at ±135 cm (4 ft), and average values of massive deposit ±15 g/t (0.5 oz/st) at ±2500 cm (82 ft).

After taking into account cost and ventilation considerations, it was decided to sublevel stope the deposit.

DEVELOPMENT

A total of 5068 m (16,627 ft) had been developed by the end of the 1970s.

When planning the development, it was obvious to all concerned that without previous experience with this type of mining, it would be difficult to visualize the final picture. It was decided to build a wooden model indicating the planned development. The model proved to be a great help. As development progressed, completed portions were marked in red, and the values obtained in the development were also recorded on the model. Development was planned between the 80 and 81 crosscut lines, using the 52 and 50 haulages as bases from which to work.

Crosscut pillars were left on both crosscut lines with an approximate width of 25 m (82 ft). This was necessary to protect the crosscuts on the 50 level, which were used for tramming purposes. The orebody was divided into 10-m (32.8-ft) stopes and 10-m (32.8-ft) pillars, starting with the No. 1 stope next to the 81 crosscut pillar (see Fig. 12.3).

Four sublevels were planned at 10-m (32.8 ft) vertical intervals with a scraper level at the cutout of the orebody.

Development of the Crosscut Pillars

As it was necessary to connect all sublevels to an orepass system and traveling ways, old reef raises on both crosscut lines eliminated most of the development necessary (see Fig. 12.4). The scraper and No. 1 sublevels were planned

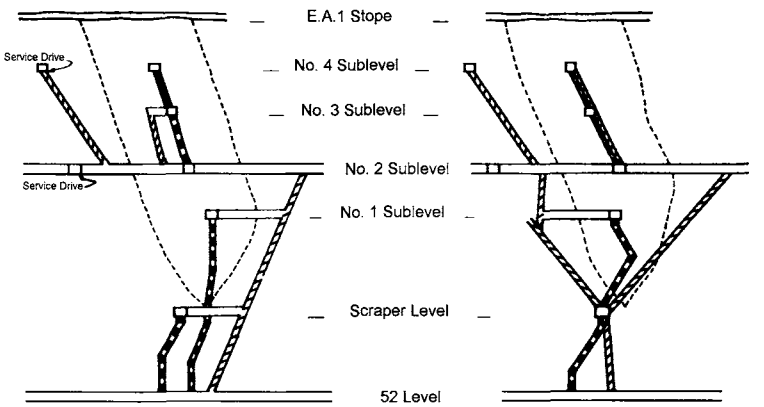


FIGURE 12.4 Development of the crosscut pillars, showing a section of the 81 crosscut (left) and the 80 crosscut (right)

from the 52 level. A boxhole was necessary from the 52 level to the scraper level on both sides. These boxholes would later be used as the main orepasses for stoping.

From the 50 level, the Nos. 2, 3, and 4 sublevels were planned. The No. 2 sublevel was planned on the same elevation as the 50 level. An exploration drive 10 m (32.8 ft) above the 50 level was used as the No. 3 sublevel. The No. 4 sublevel needed a boxhole only on the 81 side.

Cleaning of the sublevels was done by scraping.

Sublevel Development

The sublevel drives were planned to be in the approximate center of the orebody. Development was started from both sides to speed up operations. The scraper level was planned to be 4 m (13.1 ft) away from the cutout of the orebody on the footwall side. Drawpoints were planned for the center of every stope and pillar. Development sizes were 2.8 m (9.2 ft) wide and 2.4 m (7.9 ft) high to ensure easy access for longhole drilling machines.

Stope Development

Stope development consists of slot drives and slot raises. Slot drives were planned on every sublevel from the hanging wall to the footwall of the orebody. A slot raise on the footwall side connecting all sublevels was developed, forming a breaking face for the slot.

Pillar Development

Because it was possible to start stoping operations on both sides of a pillar (see Fig. 12.5), it was necessary to predevelop all pillars. All sublevels were connected to traveling ways to give access to stopes once the slot stoping had

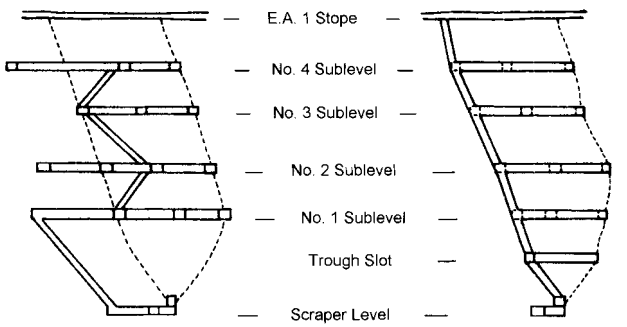


FIGURE 12.5 Pillar development. Left, pillar section; right, slot section.

been done. To speed up operations, two service drives were developed on the 50 level and No. 4 sublevel elevations. These drives were connected to all the pillars.

Drilling Drives

After a visit to the Prieska copper mine, it was decided to develop a drilling drive on the footwall and hanging wall side of the orebody on every sublevel. This ensures more accurate drilling and breaking.

Ore Recovery

Ore recovery was based on the trough stoping method (see Fig. 12.6). As the cutout of the reefs formed a natural trough at 0.87 rad (50°), this seemed to be the most effective method to use. To stope this portion of ground, a slot raise and slot drive were developed out of the scraper level. A drilling drive on the cutout point was necessary to drill and stope the portions of ground on both sides of the slot.

To ensure a steady flow of rock into the scraper level, development was planned to ensure a natural angle of repose.

After stoping the first trough, it was found that the scraper could not cope with the big rocks, which tended to hold up operations. It was decided to develop a tramming drive on the 52 level with boxholes to all drawpoints in the scraper level. Boxholes with Y-legs were developed to serve two drawpoints at the same time.

Drilling and Stoping

Drilling consists of slot and ring drilling using the Seco S36 (Figs. 12.7 and 12.8).

Layouts of all drilling patterns are done and copies are sent in duplicate to the miner in charge of drilling operations.

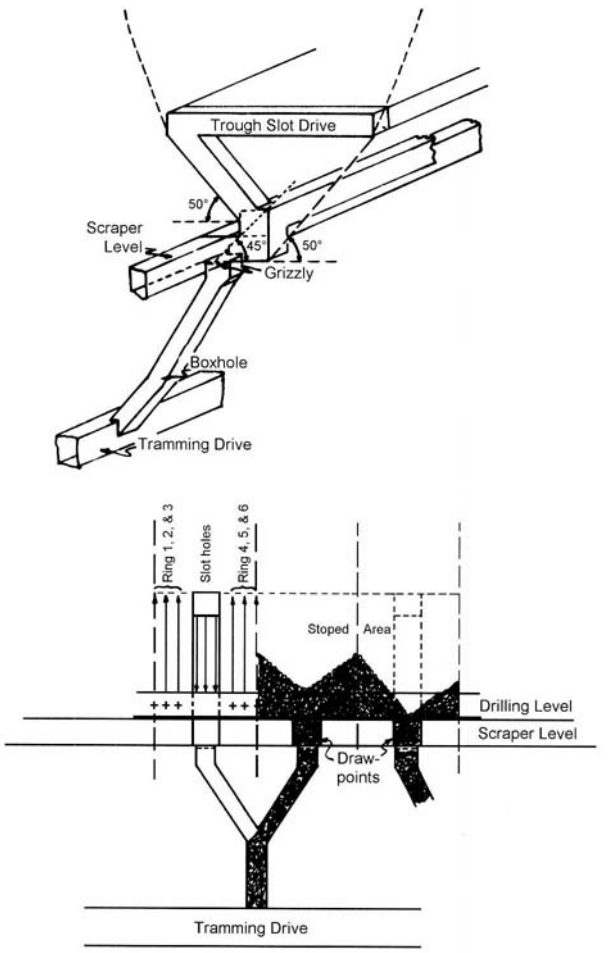


FIGURE 12.6 Trough layout with the boxhole system, top; section, bottom. 1° × 0.01745329 = 1 rad.

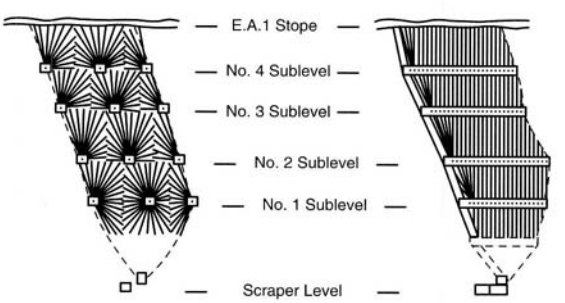


FIGURE 12.7 Stope drilling. Left, ring; right, slot.

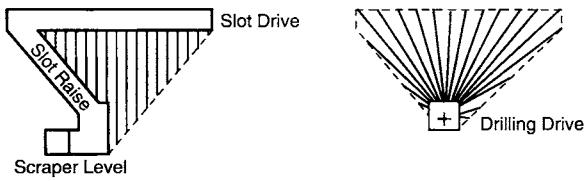


FIGURE 12.8 Trough drilling. Left, slot; right, ring.

Slot drilling consists of vertical up and down holes at a ±1-m (±3.3 ft) burden. Three slot rings are drilled on every sublevel to ensure that no hang-ups occur.

Ring drilling is done out of the drilling drives on every sublevel, using a 2-m (6.6-ft) burden between rings.

After consultation with an African explosives expert on blasting, stoping started in the No. 1 stope trough. After blasting the slot rings in the trough, forming a clean breaking face, the rings on both sides were blasted. It was found that the 2-m (6.6-ft) burden used between rings tended to break big rocks. The burden for the trough rings was therefore reduced to 1.2 m (3.9 ft).

Blasting of the No. 1 stope slot now commenced. This was done by starting on the No. 4 sublevel, blasting only a few holes at a time. This method was repeated on all sublevels, and precautions were taken not to undercut any part of the slot.

After the slot was stoped out, sublevel stoping proper commenced.

The old EA1 reef stope at the top of the orebody is standing up well and at this stage the future looks comparatively controllable.

Drilling Equipment and Accessories

type of machine –S36 reversible rotation drifter, power fed mounted, remotely controlled.

size of piston –114 mm (4½ in.).

rigs – For ring drilling the machine is mounted on an H-frame vertical rig; for slot drilling the machine is mounted on a Seco mechanical skid base rig.

drill steel –Shank adapters of the lugged type 38 mm (1½ in.) diam series 23.

couplings –Length 160 mm (6¼ in.), diam (ext.) 46 mm (1¾ in.), series 23.

extension rods –25 mm (1 in.) Hex. 1 m (3.3 ft) long series 23.

bits –Female threaded series 23, sizes 57 mm (2¼ in.), 54 mm (2⅛ in.), and 51 mm (2 in.), bit discard size 46 mm (1¾ in.), sharpenings per discard 12, and bit grade 57 mm (2¼ in.) = 438, 54, and 51 mm (2 in.) = 442.

lubricants –Caltex Neptune heavy rock drill 012.

grease for couplings –Caltex Texclad 2 grease.

Explosives and Accessories

Ammon gelignite 60%, 45 × 560 mm (1¾ × 21¾ in.), is used. All holes are lined with Cordtex with a core load of 10 g (0.35 oz) of pentaerythritol tetranitrate (PETN). Holes are initiated by short-period delay detonators (millisecond) connected in series, in parallel to the centralized blasting system. All holes are well tamped.

VENTILATION

Development

Development was ventilated by normal forced ventilation. The No. 3 sublevel was developed first to establish through ventilation. The No. 3 level was then utilized as an airway from which all other ends were developed on the forced system using 410-mm (16-in.) columns.

Stoping and Drilling

The sublevel stoping is being ventilated by 40 m^3/s (85,000 cfm). The total volume is being circulated from 80 to 81 line through the sublevels, the drilling drives, and the scraper drive. Temporary brattices are installed where no work is taking place, which allows the volume to be increased at the face. The handling of the broken rock has caused no dust problems because it is wet from the drilling water.

REVENUE

The massive body of the 80 to 82 crosscut contains 840,000 tons (930,000 st) of ore, from which as estimated 840,000 × ⅔ = 560,000 tons (930,000 × ⅔ = 620,000 st) will be mined. The average value of the ore is 15 g/t (0.5 oz/st). Therefore, the number of tons to mill at samplers grade is 560,000 at 15 g/t, or 620,000 st at 0.5 oz/st.

Mine Call Factor 88%

Tons to mill at mill grade is 560,000 (620,000 st) at 13.2 g/t (0.44 oz/st); the residue is 0.4 g/t (0.01 oz/st); the recovery grade is 12.8 g/t (0.43 oz/st); and gold recovered from 560,000 t (620,000) at 12.8 g/t (0.43 oz/st) equals 7168 kg (15,770 lb). The revenue at an estimated gold price of R2000/kg is R14,336,000.

COSTS

The capital expenditure is R60,000.

Working Costs

For development of 10,100 m (33,000 ft), European labor is R11.5/m and Bantu labor is R17.4/m. Explosives and accessories are R7/m. Water, air, electricity, and other stores are R21/m. This gives a total cost of R56.9/m, so 10,100 m (33,000 ft) at R56.9/m will cost R574,690. Including the capital expenditure, the cost will be R634,690.

Stoping

To maintain 13,000 t/month (14,326 st per month), one stoper, $\frac{1}{2}$ N/S cleaner, $\frac{1}{2}$ shift boss, and $\frac{1}{8}$ M/O will cost R17,000. Three team leaders, 27 machine/spanners, and 24 others will cost R22,000. Drilling costs, machine maintenance, and steel bits at R4/m; and average m/ring being 1000, four rings at 13,000 t (14,000 st) equaling 4000 × R4, or total R16,000. Explosives and accessories (285 cases × R6.94 per case); 5200 m (17,000 ft) of cordtex (R8.3/100 m); and blasting wire, blasting cable, electrical starters, etc. (R3 per blast) total R2400. The cost of secondary breaking and cleaning 13,000 t (14,000 st) at 10 c/t is R1300. Piping, scraper rope, cables, ventilation, etc. (13,000 t (14,000 st) at 16 c/t) total R2100. Therefore stoping costs for 13,000 t (14,000 st) are R25,700, or R1.97/t. The cost to stope 560,000 t (617,000 st) will therefore be R1,103,200.

Other Costs

Underground transport for 560,000 t (617,000 st) at 63 c/t is R352,800; hoisting 560,000 t (617,000 st) at 85 c/t is R476,000; ventilation for 560,000 t (617,000 st) at 50 c/t is R280,000; water control for 560,000 t (617,000 st) at 29 c/t is R162,400; sundry mining charges for 560,000 t (617,000 st) at 68 c/t is R380,800; valuation for 560,000 t (617,000 st) at 21 c/t is R117,600; general mine expenses at 560,000 t (617,000 st) at 137 c/t are R767,000; and reduction for 560,000 t (617,000 st) at 65 c/t is R364,000. The total is thus R4,654,000.

Applying a cost escalation of 15% over a period accounts for an extra R5,352,000. Hence profit will be R14,336,000–R5,352,000 = R8,984,000, or rounded off, R9,000,000.

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CHAPTER 13

Large Blasthole Stoping at Rødsand Gruber, Norway

Tore M. Larsen

INTRODUCTION

The Rødsand Gruber mining company is a subsidiary of the Norwegian industrial corporation Elkem AS, and its mine is situated in the northwestern part of southern Norway.

The ore from this mine is a relatively low grade, magnetite ore with a small amount of vanadium. The orebodies are very irregular in shape, resulting in a high amount of waste rock intrusion in the stopes. The annual underground production in the mines is approximately 1 million t (1,100,000 st).

Because of varying dimensions of the orebodies, both transverse and longitudinal stopes are employed. The inclination of an orebody is usually 1.04 to 1.59 rad (60 to 90°), and the horizontal width varies up to 100 m (330 ft).

Sublevel stoping has been the traditional mining method in this mine. Until a few years ago, the ore was primarily blasted using horizontal blasthole fans drilled from Alimak raises. Recently, however, there has been an increasing demand for a less expensive mining method. Alternate systems were considered, and finally a method based upon large blasthole stoping was found to be the most suitable.

GENERAL LAYOUT

The general layout of a stope is shown in Fig. 13.1. As with all the figures in this chapter, only the principles of the method are emphasized, and the details may not be valid for any particular site in the mine.

In transverse stopes, the horizontal width of the ore determines the length of the stopes. The average length is approximately 50 m (164 ft), the width of the stopes is 20 m (65.6 ft), and there is a 10-m (32.8-ft) wide vertical pillar

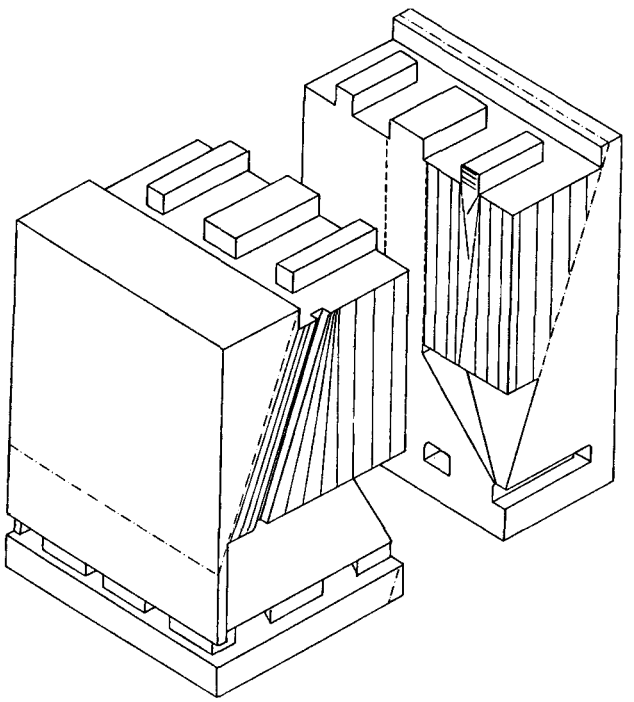


FIGURE 13.1 General layout of large blasthole stoping at Rødsand Gruber. Inclination: approximately 1.31 rad (75°). Horizontal width: approximately 45 m (148 ft). Width of stope: 20 m (65.6 ft). Width of pillar: 10 m (32.8 ft). Length of holes: maximum 40 m (130 ft). Distance between levels: 60 m (200 ft).

between. The distance between the main levels is 60 m (200 ft). Usually a horizontal (sill) pillar 10-m (32.8-ft) thick is left below the mucking level.

Each stope is first undercut along the entire stope length. This undercut is made with 50.8-mm (2-in.) diam drill holes in a fan pattern. This is done by a trackless Atlas Copco Promec jumbo equipped with two Cop 130 pneumatic rock drills. The drill drift is 4 × 4 m (13.1 × 13.1 ft), and the crater is 20 m (65.6 ft) high. This leaves a 36-m (120-ft) stope height to mine with large blastholes.

To achieve a fair portion of vertical drilling with large blastholes, there are two parallel drill drifts at the top of each stope. These drifts are 4 m (13.1 ft) high and 9 m (29.5 ft) wide, separated by a 4-m (13.1-ft) wide pillar in the middle of the stope.

To ease the drilling of perimeter holes, the drifts extend 1 m (3.3 ft) into the vertical pillar on each side of the stope. The pillar will thus be only 8 m wide on the drill drift level, but 10 m (32.8 ft) wide down the rest of the stope. The drifts on top of the stope are linked together with crosscuts at the footwall and the hanging wall.

The slot is made close to the hanging wall, using an Alimak raise for the opening. The mining retreats from the hanging wall toward the footwall.

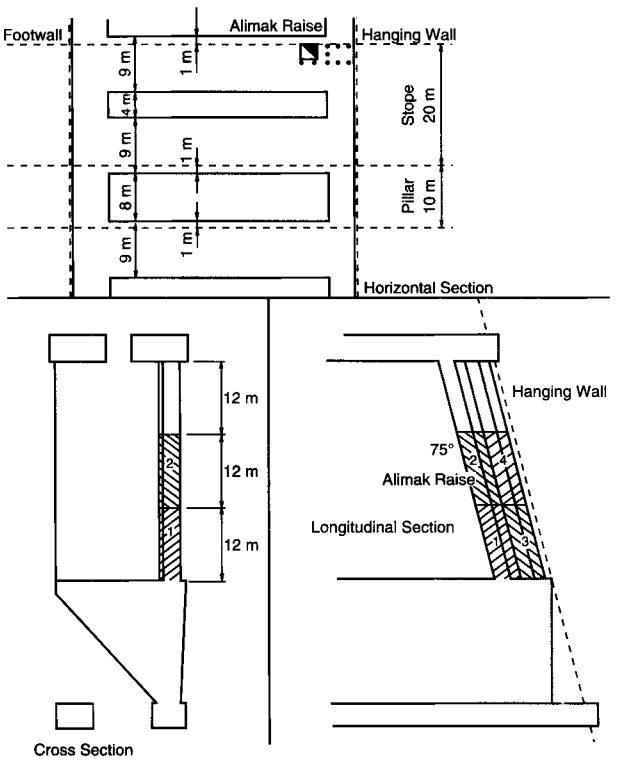


FIGURE 13.2 First step in the opening of the slot. 1° × 0.01745329 = 1 rad.

The mine is situated close to a Norwegian fjord, and all mining activities take place several hundred meters below sea level. Some stopes are actually being worked under the sea bottom. Because of some instability, all pillars are left in the mine and no attempts are made to recover them. The stopes are also backfilled with a mixture of uncemented waste rock, moraine, gravel, sand, and tailings. A stope must be backfilled before an adjacent one can be mined.

OPENING OF SLOTS

From the mucking level, a 1.3-rad (75°) inclined Alimak raise is driven close to the hanging wall. The raise is 2.5 × 2.5 m (8.2 × 8.2 ft) and is used as an opening both for the undercut trough and the slot.

The first step in slot opening, where the slot is opened toward the hanging wall in four separate blasts, is shown in Fig. 13.2.

In the second step, the slot is made vertical in two blasts, each 12 m (39.4 ft) high (see Fig. 13.3).

Finally, the slot is opened in the whole width of the stope in four blasts, as shown in Fig. 13.4.

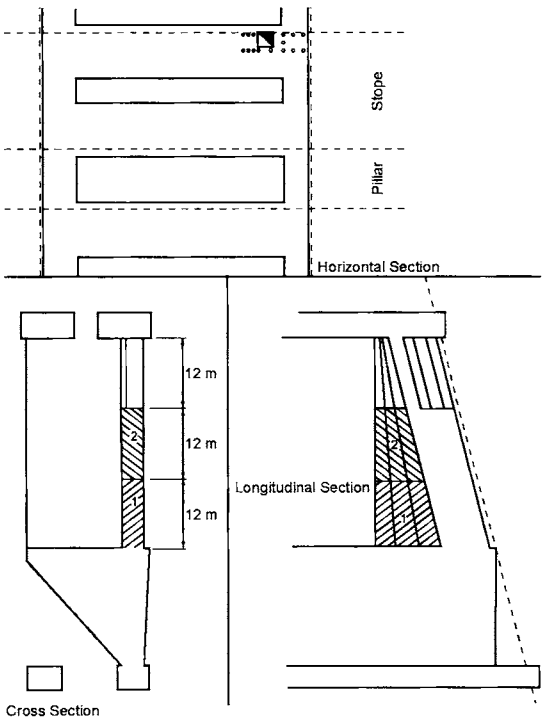


FIGURE 13.3 Second step in the opening of the slot

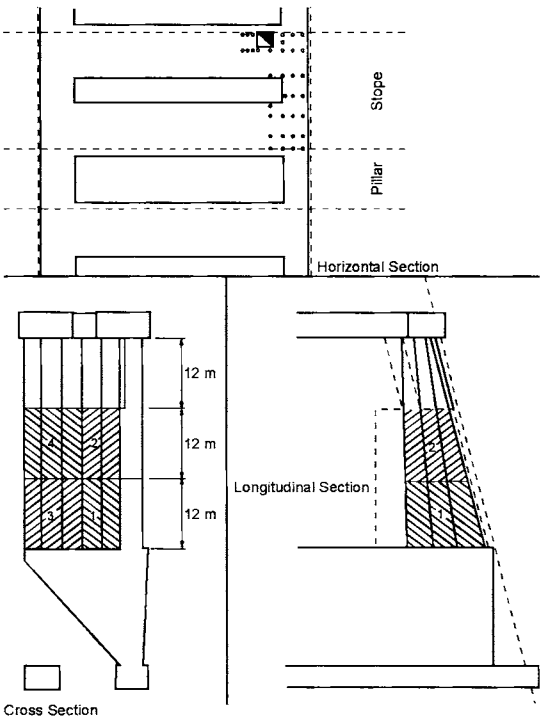


FIGURE 13.4 Third step in the opening of the slot

The upper 12 m (39.4 ft) of the slot will not be blasted until later, when the large blasthole stoping has retreated somewhat.

DRILLING LARGE BLASTHOLES

The general pattern of holes is shown in Figs. 13.5 and 13.6; the spacing is 3 m (9.8 ft), the burden is 4 m (13.1 ft), and the perimeter holes have a spacing of 2 m (6.6 ft). As shown in Fig. 13.6, the inner perimeter holes are not blasted until the following blast.

At the present time, the holes are 101 mm (4 in.) diam and are drilled with down-the-hole equipment. The drill jumbo used is a crawler-mounted Atlas Copco Roc 306 with pneumatic chain feed and rotation. The drill hammer is a Cop 4 operating at 12 kPa/cm^2 pressure from a booster compressor. To minimize the deviation, the rig is equipped with a Dit 70 accuracy drilling instrument.

The drill rods have a length of 1.52 m (5 ft), and the thread wear is negligible. The 101-mm (4-in.) button bits have an average life of 300 m (1000 ft).

To achieve sufficient air flushing at the bottom of the holes, an additional flushing adaptor has been installed between the hammer and the rods.

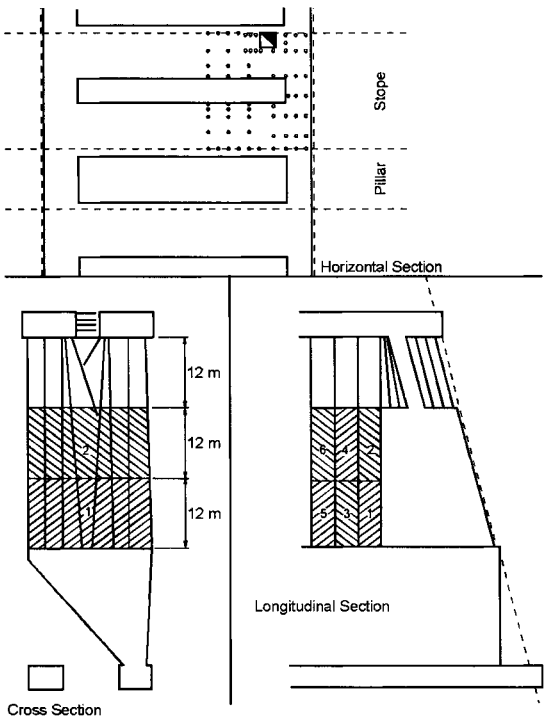


FIGURE 13.5 Large blasthole stoping

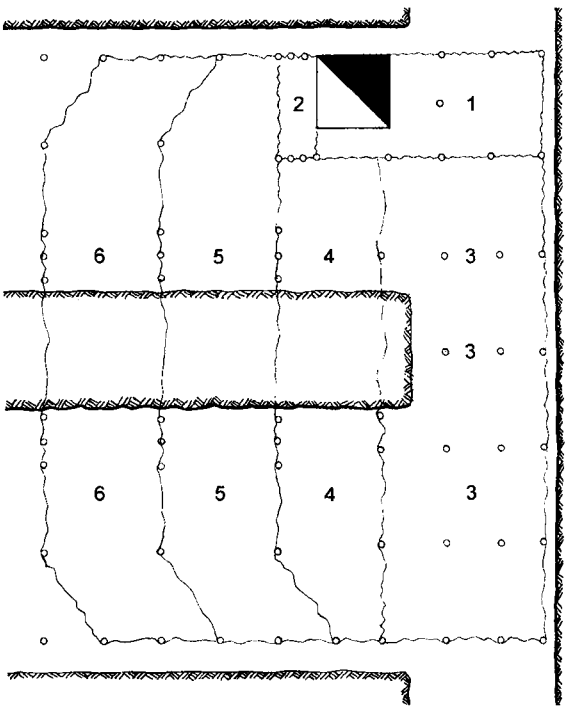


FIGURE 13.6 Drill pattern for the slot and first stoping round

Both dry and wet drilling have been tried. Water mixed with the compressed air was unsuccessful, due to removal of drill lubricants, so relatively large amounts of water are now poured directly into the hole adjacent to the drill rods.

The deviation is about 1% of the hole's length, and the normal drill rate is 30 m (100 ft) per shift, with the rig being operated by one person.

CHARGING THE HOLES

Most holes are drilled through to the undercut crater. These holes are dry, and are therefore charged with ammonium nitrate–fuel oil (ANFO) poured directly into the holes. The holes that are not drilled through usually hold some water and are therefore charged with slurry explosives [90 × 900 mm (35 in.)].

A 101-mm (4-in.) borehole has a volume of 8.1 dm^3/m (15 cu in. per ft). With a charging density of 1.05 kg ANFO/dm^3 (0.04 lb ANFO/$in.^3$), there will be 8.5 kg explosives per meter (5.6 lb per ft) of borehole. To reduce damage from blast vibrations, only a 12-m (39.4-ft) height of the stope is blasted in each round, with not more than one hole per delay. A section of a stope will therefore sometimes appear as shown in Fig. 13.7.

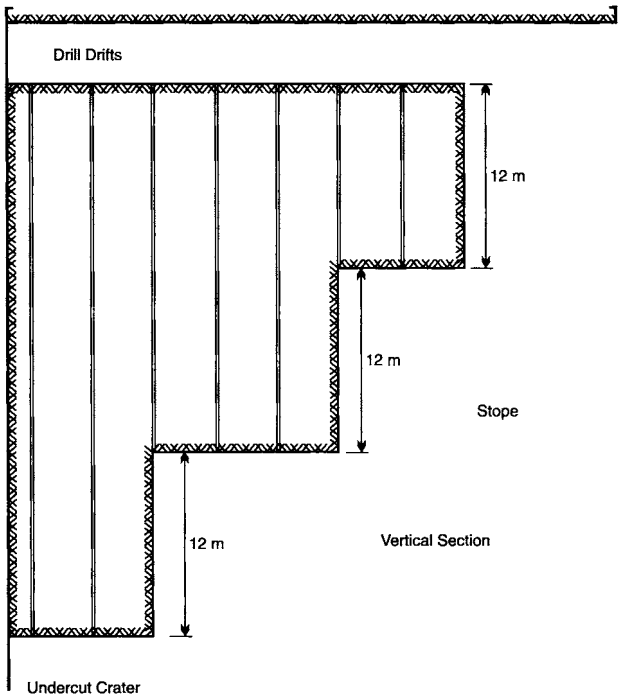


FIGURE 13.7 Mining front

The rounds are initiated by electric millisecond-delay blasting caps. There are two caps in each hole; one in the bottom and one in the middle of a 12-m (39.4-ft) high column of explosives. Dynamite [50 × 600 mm (23½ in.)] is used as a primer.

The perimeter holes have only half the burden compared to the other holes. They are charged with smaller diameter explosives, and with an ANFO-sawdust mixture.

MUCKING

The fragmentation with this mining system is satisfactory; in fact, it is better than with the previous system used.

The general layout of a mucking level is shown in Fig. 13.8. The drawpoints are driven as crosscuts into the crater-drill drift. Caterpillar 966s or Wagner ST-8s load and haul the ore to orepasses, where it goes to the crusher located at a lower level. The average hauling distance is 100 m (330 ft).

Rocks requiring secondary blasting are hauled to another drawpoint for later treatment.

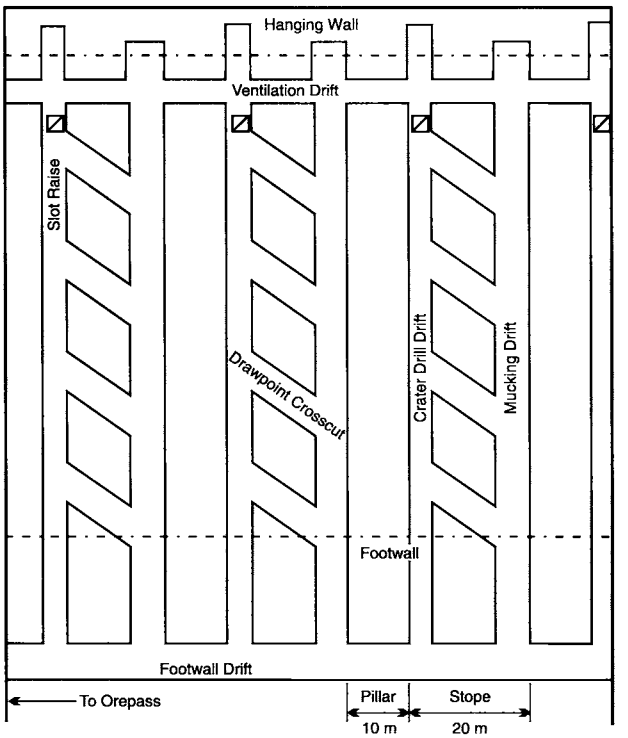


FIGURE 13.8 Mucking level for transverse mining

EVOLUTION OF THE METHOD

When the horizontal width of the orebody is less than 25 m (82 ft), the ore will be mined by longitudinal stopes. The width of the stopes will be determined by the ore, and the length of the stopes usually is about 50 m (164 ft). To achieve minimum fan drilling, the holes are drilled parallel to the inclination of the orebody.

The 10-m (32.8-ft) horizontal pillar located beneath the mucking level will be dropped in the future if stability conditions are favorable. The drilling of large blastholes will then be done from the mucking level downward, making the drill holes 14 m (46 ft) longer. The overlying ore in the crater area will be blasted with small boreholes [50 mm (2 in.) diam] from the mucking level.

A large portion of the ore is in the crater (undercut) itself. When this is mined by upward fan drilling, the costs for this ore are relatively high. Thought has been given to drilling the crater with large blastholes from above, while only presplitting the crater footwall by drilling from below (see Fig. 13.9).

The large blasthole blasting causes some cracking in the pillars between the drawpoints. It is probable that the crater blasting with large blastholes will do even more damage to the drawpoints. The crater-drill drift must therefore be raised approximately 2 m (6.6 ft) higher than the mucking drift. Together

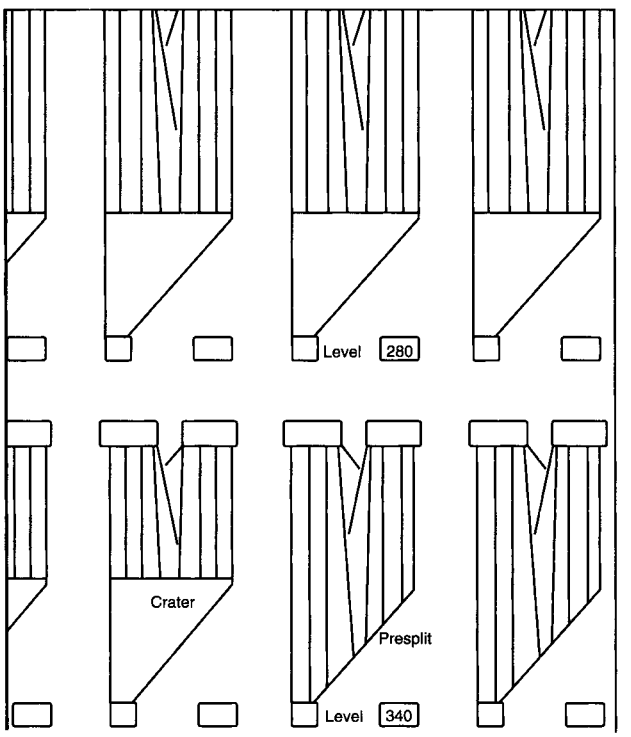


FIGURE 13.9 Crosscut of mining above level 340

with reduced charges in the lower sections of the blastholes, this adaption may secure the stability of the drawpoints. This method, however, has not as yet been tried.

Further evolution and modifications to the mining method will be done as experience is gathered. So far, the method has been shown to increase the output capacity and to decrease costs.

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CHAPTER 14

Vertical Crater Retreat: An Important Mining Method

L.C. Lang

INTRODUCTION

The introduction of 165-mm ($6\frac{1}{2}$-in.) holes to underground mining operations has made possible the application of Canadian Industries Ltd.'s (CIL) vertical crater retreat (VCR) mining method. This unique and revolutionary new application of spherical charge technology (see the appendix to this chapter), when applied to primary stopes and pillar recovery, eliminates raise boring, slot cutting, and dilution of ore by backfill; greatly improves fragmentation; reduces labor and time requirements; eliminates uphole drilling and blasting; and minimizes or completely eliminates damages by blasts to the walls and retreating backs of the stope or pillar.

If vertical large diameter holes are drilled on a designed pattern from a cut over a stope or pillar to bottom in the back of the undercut, and spherical charges of explosives are placed within these holes at the calculated optimum distance from the back of the undercut and detonated, a vertical thickness of ore will be blasted downward into the previously mined area.

Repeating this loading and blasting procedure, mining of the stope or pillar retreats in the form of horizontal slices in a vertical upward direction until the top sill is blasted and the mining of a stope or pillar is completed.

The VCR method is also applicable to drop raises and has the potential to replace all other raising methods under most circumstances.

PILLAR RECOVERY

Levack Mine

Inco Metals Co., Ontario Div., provided the first opportunity for the method in pillar recovery. The Levack mine's high-grade ore pillar No. 4800 on the 975-m (3200-ft) level was used for the production-scale experiment (Figs. 14.1 and

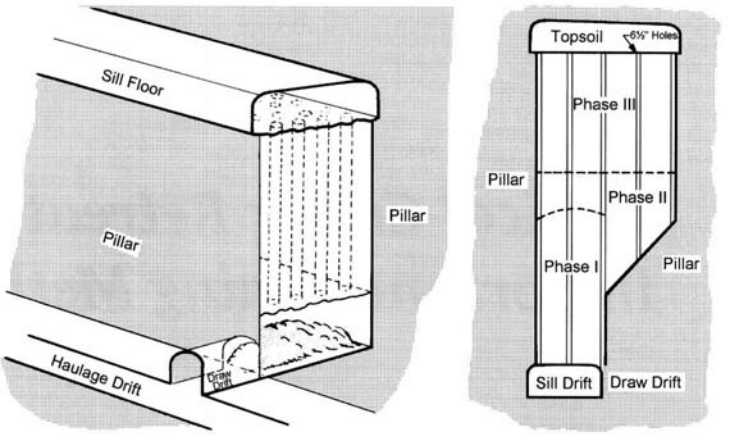


FIGURE 14.1 Spherical charges placed in large diameter holes will blast downward a horizontal slice of ore into the previously mined area. This process can be repeated in a series of load and blast procedures until the entire stope or pillar is mined. Metric equivalent: in. × 25.4 = mm.

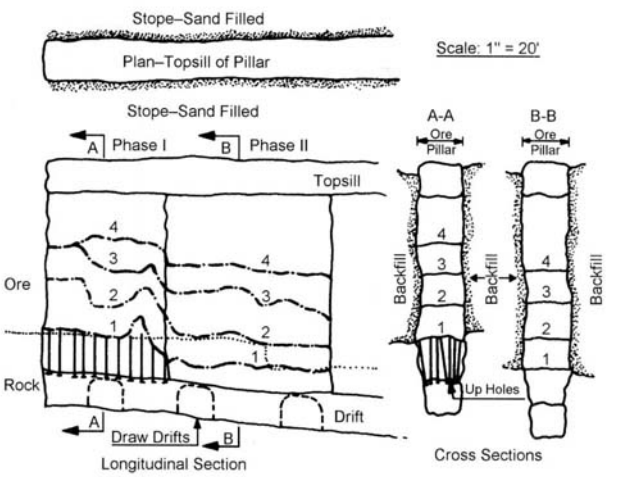


FIGURE 14.2 Vertical crater retreat in a high-grade pillar at Inco's Levack mine. Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m.

14.2). The pillar was about 49 m (160 ft) long, 6 m (20 ft) wide, and 20 to 26 m (65 to 85 ft) high. The mined area on both sides of the pillar was backfilled with a 1:30 cement:sand mixture.

The pillar was removed in two phases. In phase 1, the standard up-hole method was used to blast down the 18-m (60-ft) long section of waste from the bottom of the ore into the undercut.

From the pillar's top sill, 165-mm ($6\frac{1}{2}$-in.) holes were drilled downward into the pillar, and by measuring the depth of the holes, the results of the up-hole

blast were determined and roof line 1 was established. The bottom of each hole was plugged, then filled with sand to place the center of gravity of each spherical charge (loaded from the top sill) at a predetermined optimum distance from the horizontal free surface. The charges were then detonated.

After detonation, both draw drifts were full of extremely well broken material. The depth of each hole was measured again, and the plot of these depths resulted in roof line 2. The same blasting procedure was repeated and the resultant new back elevation was marked by roof line 3.

The poor results of the initial uphold blast at one location (notice the peak in area 1) appeared to influence the subsequent new backs. A third blast successfully evened the back, and resulted in roof line 4. An unblasted slab averaging 6.3 m (20.9 ft) thick remained below the pillar's top elevation as the final sill.

In all three spherical charge blasts, fragmentation of the blasted material was extremely fine. The backfill was fully exposed on both sides of the now-blasted pillar. The backfill remained undamaged and the ore was not diluted by sand. The remnants of all the 165-mm ($6\frac{1}{2}$-in.) holes remained clean and undamaged, and the holes had well-defined bottoms that could easily be measured and plugged. Each blast took down a 3.9- to 4.2-m (13- to 14-ft) thick horizontal slab of ore. Productivity was three times greater than that of the previously practiced cut-and-fill method.

Because this was the first such experiment, blasting the remaining 6-m-(20-ft)-thick final slab was the subject of some deliberation. If the described method was repeated, it could have resulted in a 1.8-m (6-ft) thick sill on which it was unsafe to work. It was therefore decided to blast the whole sill using two spherical charges properly placed in each hole, but with the application of delays in the vertical direction. In this way, the lower initial charges would provide a free face for the higher second charges.

The fourth blast was also a success, with one exception: a layer of ore 0.6 m (2 ft) thick remained along the backfill on one side of the pillar, and the ends of rockbolts protruded into the cavity that was created. This was the result of more cautious loading of the side holes than was necessary. The layer was knocked down without any difficulties by the application of a few small ammonium nitrate–fuel oil (ANFO) charges on the surface, and the backfill did not suffer any damage.

Before proceeding with phase 2, the void created by blasting and drawing off the ore in phase I was backfilled. Timber bulkheads were built on the top sill and in the No. 4800 drift to hold the sand.

The first blast in phase 2 took down an average 2.9-m (9.6-ft)-thick slab of waste immediately above the drift. In contrast to phase 1, the initial undercut was done by caving with spherical charges. The carefully considered loading and delay pattern resulted in excellent fragmentation and caused no damage to the brows of the draw drifts or to the sand-filled area of phase 1.

The second blast advanced an average of 3.6 m (12 ft) upward, and the third blast averaged 5 m (16.6 ft). The fourth blast removed an average of 3.8 m (12.6 ft), and a top sill averaging 9.8 m (32.2 ft) thick remained in place.

For safety reasons and for further experimentation, the thick slab was blasted down in one blast. Two charges, placed at optimum vertical distances in each hole, were vertically delayed by CIL's "TOE-DET" system.

Blasting of the last sill was a complete success. The sand walls remained undamaged by the blasts, and fragmentation was excellent.

The lesser advance in the initial undercut is explained by the lack of backfill on both sides of the blasted waste. The same advance should be expected in primary stoping because of the lack of free surface on both sides.

If drawing of the blasted ore proceeds quickly to give sufficient void for swell and drop for secondary fragmentation, two lifts independently delayed may be dropped in a single blast in a pillar for greater efficiency. Drawing of the ore should be well synchronized with blasting, and the void should be backfilled as soon as possible.

Pillar recovery by the VCR method has become an accepted mining method by Inco Metals Co., Ontario Div., and a substantial number of pillars have been mined out successfully at the Levack mine.

Strathcona Mine

At Strathcona mine of Falconbridge Nickel Mines Ltd., Onaping, Ont., the first pillar has been successfully recovered by the VCR method. Pillar 25-D2-D4 was 61 m (200 ft) high and divided into two parts of equal height by a sublevel. The bottom 15 m (50 ft) of the lower segment was developed into a cone for the drawpoints by using small diameter fanning holes and conventional blasting. The average width of the pillar was 6.7 m (22 ft), and only 30 m (100 ft) of its length was taken at this time. On both sides and on top the pillar was surrounded by backfill. Every 2.7 m (9 ft), a 0.3-m (1-ft) layer of 1:8 cement:sand was poured while the remainder was a 1:30 ratio.

First the lower and then the upper section of the pillar was mined by the VCR method with great success. Further pillar recovery is planned at Strathcona mine by this method.

PRIMARY STOPING

Levack West Mine

Following the successful extraction of pillars by this technology, it was considered more than appropriate to extend this concept to primary stopes. With this objective, Inco Metals Co., Ontario Div., laid out a stope between the 950 and 1080 levels at Levack West mine. The analysis of this VCR stope suggested that it was the safest method of mining known, and was competitive with any other forms of stoping in terms of cost and production.

What has been clearly proven over the duration of extraction is that the VCR method provides exceptionally fine muck, sound pillar walls, and drawpoints. In areas such as haulageways and drawpoints, no wall or back deterioration was experienced. Sound pillar walls were very much in evidence, as indicated by 165-mm ($6\frac{1}{2}$-in.) borehole remnants along all of the remaining walls.

It was observed that in-the-hole drilling has to be set up accurately to obtain the designed pattern in the VCR method. Excess stretching of drill patterns causes substandard results. Although no actual study was performed on drilling accuracy, it was observed that initial alignment is vitally important.

The extraction of the experimental stope was followed by others at Levack West with the same good results.

Birchtree Mine

The Manitoba Div. of Inco Metals Co. has initiated the use of the VCR mining method at the Birchtree mine 6.4 km (4 miles) south of the city of Thompson. The orebody at Birchtree is irregular both in strike and dip and has been mined by longitudinal cut-and-fill and sublevel blasthole methods. Peridotite bodies associated with the ore adversely affect dilution during mining. The VCR method provides the advantage of maintaining the stope full of broken ore during blasting, followed by rapid draw and filling.

The 83 stope on the 1300 level was selected for the experiment (Fig. 14.3). The dimensions of the mining block were 38 m (125 ft) long, 33.5 m (110 ft) high, with widths of 3 to 9 m (10 to 30 ft) at the top sill and 7.6 to 15 m (25 to 50 ft) at the undercut. Holes of 152-m (6-in.) diam were drilled, with the perimeter holes 1.5 m (5 ft) inside the footwall and hanging wall contacts. It was necessary to form a 0.78 rad (45°) ledge on the hanging wall to enable ore flow into the undercut. This part of the operation required eight blasts. Following the initial phase of the blasting, the average vertical advance was 3 m (10 ft) per blast.

The fragmentation of the blasted ore was satisfactory. Some large pieces of ore appeared in the drawpoints due to overbreak from the hanging wall and footwall contacts.

Centennial Mine

Hudson Bay Mining and Smelting Co.'s Centennial mine at Flin Flon, Man., has provided an opportunity for preliminary testwork in the sill of the 161 vertical pillar on the 165-m level. The study, conducted by CIL, indicated the feasibility of the VCR method despite the very high strength of this low-grade orebody. The experimental stope presently in operation is about 43 m (140 ft) high, and has an average width of 6 m (20 ft). Because the orebody dips about 1.3 rad (75°), all holes had to be inclined. While the drilling of the recommended pattern had been rather difficult due to the inclination of the holes, the stope has been retreating satisfactorily. The fragmentation is good, and no damage to the footwalls and hanging walls has occurred. Upon the extraction of the total quantity of ore available in this stope, the final results will enable the mine to evaluate the economic advantages or disadvantages of this method under these rather difficult conditions.

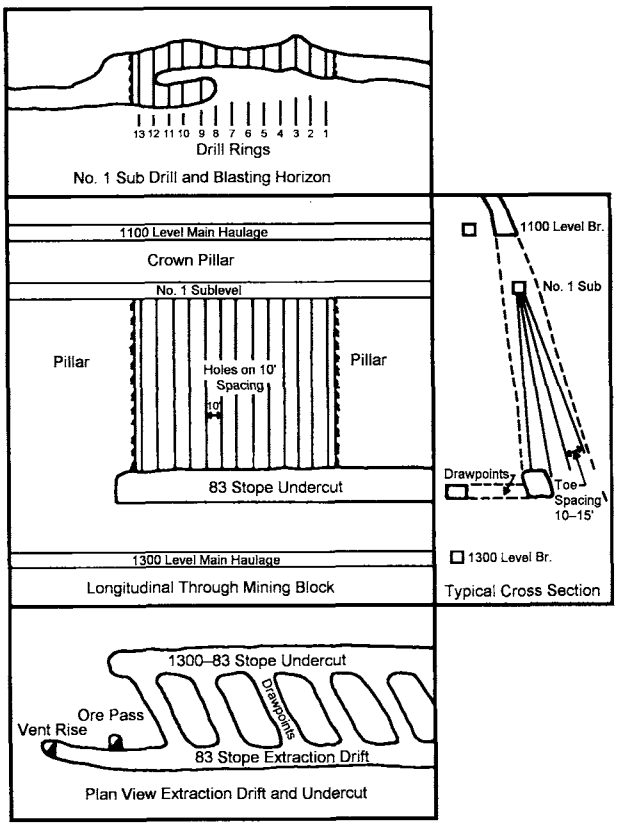


FIGURE 14.3 Birchtree mine showing the 83 stope on the 1300 level, using the vertical crater retreat method. Due to the irregular shape of the Birchtree orebody, boreholes had to be drilled at different depths and angles. Scale: 1 in. = 50 ft, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m.

Rubiales Mine

Rubiales mine of Exploración Minera España S. A. (EXMINESA) in northwestern Spain began to produce ore from the first stope in the upper ore blocks in mid-April 1977. It was estimated that at least 80% of the production would be extracted from open stopes averaging 30 m (100 ft) in width and 70 m (230 ft) in height.

C10-512 is the first stope of a series opened on the top sill to gain access to the stope block. Inco's big hole blasthole method was considered for the effective mining of these stopes, and a Robbins 61R raise borer reamed a 1.8-m (6-ft) diam raise to start the slot. An Atlas Copco in-the-hole (ITH) drill was to drill the 165-mm ($6\frac{1}{2}$-in.) holes that would be blasted into the slot.

The Rubiales ores occur in the drag-folded limb of an anticline. Silicified limestone and argillite rock layers alternate in a complex ladder structure caused by the drag folding. Rock failures of different sizes were evident from the backs of development works, but a formidable failure in the experimental stope has caused considerable concern regarding cut-and-fill and bench-type mining methods.

The extraction of ore from stope C10-512 by VCR began in mid-April 1977, with design and supervision of loading and blasting operations provided by CIL.

The stope measures about 15 × 25 m (49 × 82 ft) and 70 m (230 ft) in height. A pillar was left in the center of the top sill to provide further support to the back, already secured by resin-anchored roof bolts. Several lines of 165-mm ($6\frac{1}{2}$-in.) diam vertical holes were drilled with an Atlas Copco ITH drill to a depth of about 55 m (180 ft), five holes in each line, thus establishing the recommended drilling pattern.

The first step was the blasting of the cone-shaped undercut. This was done by conventional methods from the undercut with fanning small diameter holes up to about 15 m (49 ft). The resultant new back was accessible for viewing from a sublevel when the VCR operation commenced.

In six rounds of consecutive blasts the stope retreated about 15 m (49 ft) upward. The fragmentation was fine and ideal for loading, and free of larger chunks requiring secondary blasting. A statistical 32.8 t/m (11 st per ft) of borehole were produced at an explosives consumption of 0.34 kg/t (0.68 lb. per st). The walls and the back remained stable. Further advance in this part of the stope was temporarily suspended to allow time for drawing the blasted ore. During this period, a constant check of hole depth indicated no slough from the back, which remained stable.

About 70,000 t (77,000 st) of ore was successfully extracted from this stope. The third VCR stope C10-517 has had similar success. The good results obtained have confirmed all predictions and expectations, and there is well-founded reason to believe that the VCR method is successful at Rubiales.

Pamour Porcupine Mine

Pamour Porcupine mine was the first gold mine to adopt the VCR method. When the small open-pit reached the most economical bottom, a decline was driven from the pit bottom, and the orebody was undercut. The 36-m (118-ft) crown pillar was extracted by the VCR method. The concept, applicable to any open-pit mine, is shown in Fig. 14.4.

Abminco N.L.

At Ardlethan, N.S.W., Australia, a small offspring of the main tin orebody was found about 40 m (131 ft) below the surface just outside the open-pit mine. An adit was driven from the pit bottom, and the Carpathia orebody was undercut (see Fig. 14.5). All holes were drilled from the surface, and the accuracy of these 80- to 90-m (262- to 295-ft) holes was of utmost importance. The cross section of the holes, the two VCR stopes, and the rib pillar are shown in Fig. 14.6, and the plan view of the mining operation is shown in Fig. 14.7. The ore was mined out successfully without diluting its low-grade but precious tin content.

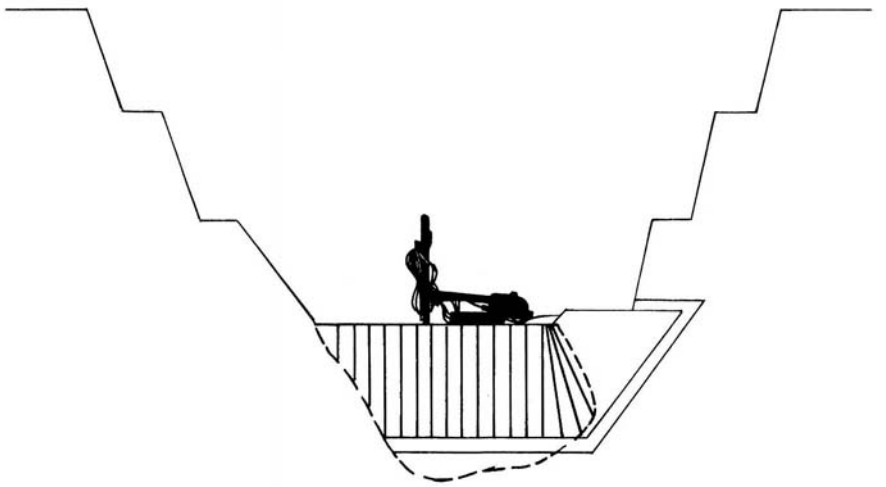


FIGURE 14.4 A crown pillar of ore left at the bottom of an open-pit mine is extracted by the VCR method

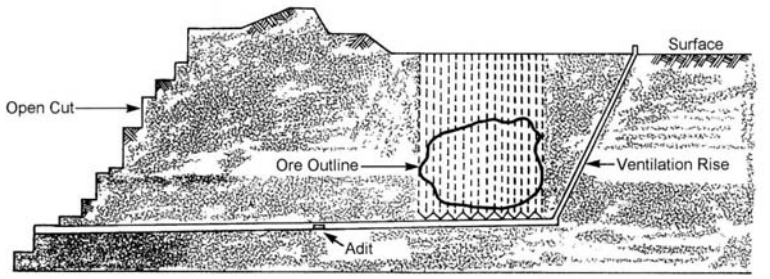


FIGURE 14.5 The Carpathia orebody is drilled from the surface for the VCR method

DROP RAISING

The principles described previously also apply in drop raising. The VCR method in drop raising is quite different from the Boliden experiment and is more productive.

Two 36.5 m (120 ft) long, 3 × 3-m (10 × 10-ft) drop raises were completed for Agnew Lake mine, Ont. Although drilled off to apply conventional drop raising techniques, these raises were advanced most of the way using spherical charge technology.

Depending on hole configuration and good blasting practices, 3-m (10-ft) advances per blast are possible. With adequate ventilation, two 3-m (10-ft) advances per shift, or at least three per day, are quite feasible. In conventional raising, setup time takes up a large portion of the total required time. Scaling,

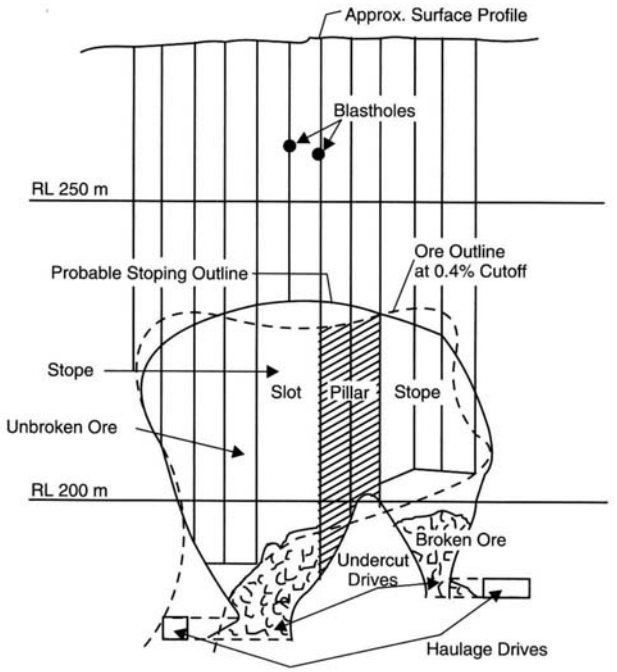


FIGURE 14.6 Cross section of the Carpathia orebody shows the development work and the sequence of the VCR mining

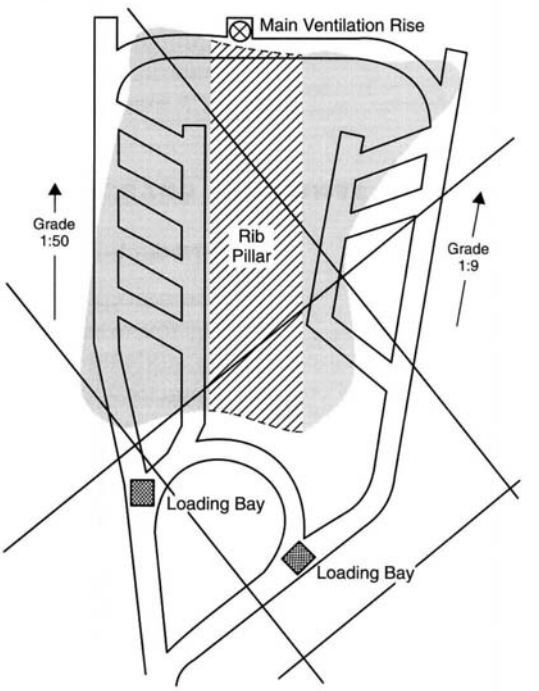


FIGURE 14.7 Plan view of the VCR operation at the Carpathia operation

always a necessity, absorbs time as well. Drop raising eliminates these two practices, and in doing so it provides greater advance and a safer working environment.

Post-blast air shock must be vented through drawpoints to eliminate collar eruptions. It is therefore essential to continually muck drawpoints to keep them open. All the walls of these raises were rather even and free of any damage.

It is appropriate to point out that it is better to design larger drop raises where the VCR method is employed. The larger the surface blasting area, the better the advance. Experience indicates that a 3 × 3-m (10 × 10-ft) area represents a practical minimum.

SUMMARY

The breakage mechanism of a spherical charge greatly differs from that of the cylindrical charge used in underground mining. The advantages of the spherical charge could not be utilized until Inco Metals Co., Ontario Div., introduced large-diameter holes to its operation.

Production-scale experiments were carried out with spherical charges in stope-and-pillar mining with success. Then the new blasting method developed into a new mining method called vertical crater retreat, and is gaining acceptance by the industry.

Its application in pillar recovery and in primary stopes has resulted in the elimination of raise boring and slot cutting. Dilution of the ore by backfill material is reduced. Furthermore, fragmentation is greatly improved. The VCR method shows great potential in drop raises.

APPENDIX: SPHERICAL CHARGE BASIS OF VCR METHOD

Introduction

The concept of cratering and its development may be attributed to C.W. Livingston, whose lifetime of long and devoted work resulted in an excellent tool for studying and understanding the blasting phenomenon, and finally finding a practical application for it in underground mining (Livingston, 1973).

A crater blast is a blast in which a spherical charge is detonated beneath a surface that extends laterally in all directions beyond the point where the surrounding material would be affected by the blast.

In analyzing crater blasts, it has been found that there is a definite relation between the energy of the explosive and the volume of material that is affected by the blast, and this relationship is significantly affected by the placement of the charge.

Livingston determined that a strain-energy relation exists, expressed by an empirical equation:

$$N = EW^{1/3}$$

where N is the critical distance at which breakage of the surface above the spherical charge does not exceed a specified limit, E is the strain energy factor, a constant for one given explosive-rock combination, and W is the weight of the explosive charge.

The same equation may be written in the form of:

$$d_b = \Delta EW^{1/3}$$

where d_b is the distance from the surface to the center of gravity of the charge (i.e., depth of burial), and Δ equals d_b/N, which is a dimensionless number expressing the ratio of any depth of burial compared to the critical distance. When d_b is such that the maximum volume of rock is broken to an excellent fragment size, this burial is called optimum distance (d_0).

Cratering Experiments

An actual cratering experiment consists of a number of cratering-type shots in which the type of explosive, its charge weight, and the rock are constant, and only the depth of burial varies.

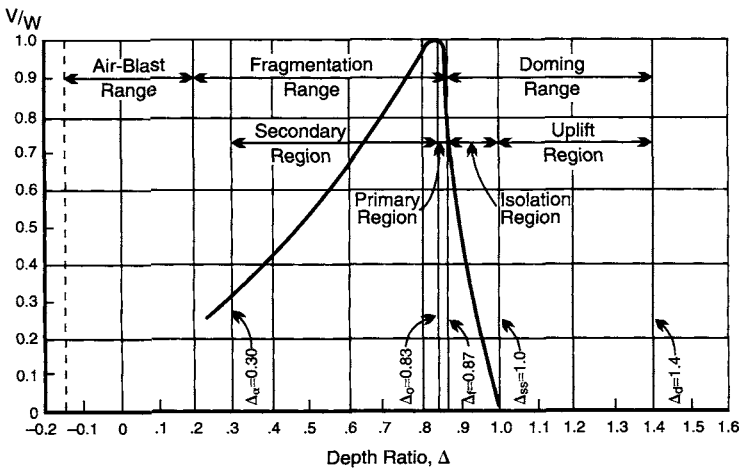


FIGURE 14.8 Cratering test in which depth ratios are plotted against energy levels. This idealized curve of Livingston (1973) also shows the important transition limits from the isolation range to the other extreme, which is the air-blast range.

Critical distance is obtained by observation. All craters are excavated, and their volume determined. Fig. 14.8 shows the plot of data obtained from a cratering test in which the depth ratios ($\Delta = d_b/N$) are plotted against energy levels (volume/charge weight: V/W). This idealized curve of Livingston (1973) also shows the important transition limits from the isolation range to the other extreme, which is the air-blast range.

Once having this curve established, by taking the appropriate value from the corresponding part of the curve and applying the generally accepted system of cube root scaling, production-scale blasts can be designed for spherical charges to satisfy any demand from simply isolating the rock from its surroundings to the effective breaking of the rock, or even for casting with explosives.

Effect of Charge Shape

Livingston (1973) recognized the importance of the shape of the charge, and in his breakage process equation

$$V/W = E^3 ABC$$

assigned C to stress distribution number expressing the effect of charge shape; A is the energy utilization number related to loss or inefficient use of the available energy, and B is the materials behavior index, related to changes under conditions of the loading or unloading process.

The effect of C was demonstrated by detonating two equal charges of the same explosive but of different shape in the same type of rock. The spherical charge broke a much greater volume of material than the cylindrical charge, even though the only difference was the shape of the two charges. Table 14.1 shows the comparison between spherical and cylindrical charges.

TABLE 14.1 Comparison of spherical and cylindrical charges

	Spherical Charge	Cylindrical Charge
Type of explosive	C2	C2
Charge weight	4.5 kg (10 lb)	4.5 kg (10 lb)
Hole diameter	114 mm (4½ in.)	67 mm (2⅝ in.)
Diameter-to-length ratio	1:2.7	1:15
Volume of crater	4.4 m^3 (155 cu ft)	1.1 m^3 (38.6 cu ft)
Crater radius	1.7 m (5.7 ft)	1.5 m (4.8 ft)

Livingston's method was an empirical one. Ash (1974) has offered a more scientific explanation of the different breakage mechanisms of spherical and cylindrical charges.

The explosive's contribution in blasting is to provide pressure. The forces generated by pressure acting over a borehole's surface area accomplish the necessary work to cause stress conditions within the surrounding mass for fracture and displacement.

The explosion produces two distinct and separate pressures. The first is the detonation pressure developed as the detonation front passes through the explosive charge. The explosive's detonation velocity directly affects the magnitude of this pressure. The value of the detonation pressure is approximately proportional to the explosive's density and its detonation velocity squared. This pressure is applied at only a very short period of time against the surrounding mass at a given section of charge length. For a cylindrical explosive charge in a typical blasthole, the detonation pressure would have only a minimal influence on the rock surrounding the borehole, and the detonation pressure would have its greatest effect only at the charge and opposite the point of initiation. One can conclude that the detonation pressure in a cylindrical borehole is not very effective to fragmentation.

Borehole Pressure

The second pressure that quickly follows the first is the borehole pressure produced by the high-temperature gases formed by the chemical reaction. The entire surface area of the borehole where the explosive is contained will be exposed to a sustained loading condition. It would be, therefore, the borehole pressure that dominates in the process of breaking the rock.

Dynamic loading by borehole pressure in a cylindrical hole is predominantly directed laterally, or radially outward from the borehole axis, with little or no force being directed toward the charge ends.

The breakage mechanism of a spherical charge is quite different. The forces produced by a spherical charge are directed radially outward from the center in a uniform spherically diverging action in all planes passing through the center. It follows that the entire surface area of the cavity confining the spherical charge receives all the detonation pressure, and the borehole pressure.

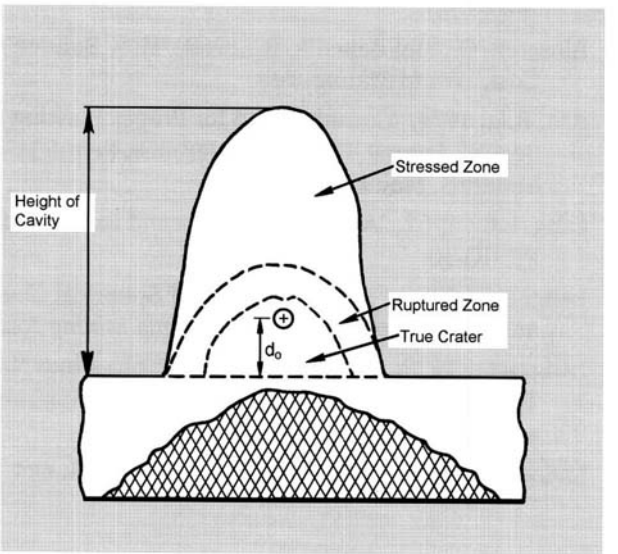


FIGURE 14.9 Depending on rock properties and structural geology, the total height of the cavity may exceed the optimum distance of the spherical charge from the back many times

Livingston (1973) used true spherical charges, but it has been found that as long as the deviation from the true spherical charge (diameter = length) is not greater than 1:6 diameter-to-length of charge ratio, the breakage mechanism and the results are practically the same as that of a true spherical charge.

Spherical charges had no practical application in production-type blasting in the past. When cratering is used for research purposes, the charges are fired in an upward direction, thus cratering toward a horizontal free face.

But if we detonate spherical charges toward a free face such as the back of a mine opening or ceiling of any underground excavation, an entirely new concept of cratering is found, forming the basis of a new blasting technology, which has been developing into a new underground mining method.

In this application, the crater is formed in a downward direction. Results are not affected by the adverse effects of gravity and friction. To the contrary, gravity enlarges the crater dimensions by removing the entire rupture zone.

Once the equilibrium of the mass is disturbed by the excavation of an underground opening, a stressed zone of elliptical shape is formed above this opening. Depending on the stability of the rock, the material within the stress zone caves in sooner or later unless proper support is provided by rockbolts, timbering, etc. When a crater is blasted in the back of an underground opening, this stressed zone is enlarged, and the material within suffers further damages and caves in following the detonation. Depending on rock properties and structural geology, the total height of this cavity may exceed the optimum distance of the spherical charge from the back many times (see Fig. 14.9).

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CHAPTER 15

Crater Blasting Method Applied to Pillar Recovery at Falconbridge Nickel Mines Ltd.

C.J. Monahan

INTRODUCTION

With the introduction of large-diameter hole drilling to underground mining, a need for the development and utilization of new blasting technology has arisen. This new technology has resulted in more efficient blasting and in easier and cheaper mining. The application of spherical charges to the crater blasting method is an important part of this technology. Spherical charges, or their geometric equivalent, make for highly efficient use of explosives in the cratering application. Their geometrical configuration, normally a length-to-diameter ratio of ≤6:1, limits their charge weight size. For example, in a 165-mm (6½-in.) hole, a spherical charge weighs approximately 34 kg (75 lb).

These dimensional and weight restrictions require careful engineering design and control in the production application.

Strathcona mine, a large-tonnage mechanized cut-and-fill and blasthole stoping operation located on the north rim of the Sudbury Basin, has successfully employed the crater blasting method in production blasting. As a result, this method, using spherical charges, has made possible the development and implementation of a vertical retreat blasthole mining method, which is being used for rib pillar recovery.

The crater method used at Strathcona essentially involves blasting off horizontal slices of ore into an undercut while retreating vertically to the pillar overcut elevation. Loading and priming of the holes is achieved from the overcut, and the blasted muck is recovered by load-haul-dump (LHD) units through a drawpoint system. Valuable experience on the behavior and performance of explosives in large-diameter holes has been gained in both the operating and engineering fields from the application of this blasting method. It has been used in drop raising and is currently being evaluated for use in primary stoping operations. Also, serious consideration is being given to its use for the recovery of post pillars in mined-out and backfilled cut-and-fill stopes.

The method has significant advantages over conventional methods of pillar recovery from economic, ground control, and safety points of view. These include cost benefits due to a minimum stoping development, less ground support, reduced labor requirements, and faster mining rate. Research on the blasting mechanism indicated the method should result in less damage to fill walls than would be normally expected with conventional longhole or large-diameter-hole cylindrical blasting (benching). This is primarily due to the limited charge weight per delay because of the size dimensions of a spherical as compared with a cylindrical charge and also because the large-diameter holes experience less deviation than standard-sized longholes. Experience has shown that wall damage has been minimized with the use of this method. The safety aspects of the working environment are also improved because there is no requirement for personnel to go under exposed backs after blasting, which is a significant improvement over the cut-and-fill method.

CRATERING THEORY

General

The term "cratering," in blasting, is applied to the formation of a surface cavity in a material as the result of detonating an explosive charge in that material. This blasting concept was initially used as a tool in the evaluation of explosives performance; however, it has been utilized more recently on surface and to a lesser extent in underground blasting operations.

Explosive charges used in the crater method are normally spherical or the geometric equivalent. Research into the application of this breakage mechanism to rock indicates that spherical charges or their equivalent produce optimum results. In blasting practice, spherical charges have been defined as having a length-to-diameter (L:D) ratio of 4:1 or less, and up to, but not exceeding L:D = 6:1. Thus, for holes of 165 mm ($6\frac{1}{2}$ in.) diam, a charge 165 mm ($6\frac{1}{2}$ in.) in diameter and 990 mm (39 in.) in length would constitute a spherical charge.

Crater Testing

Crater testing for underground application is normally accomplished by drilling smaller than production-sized horizontal holes into drift walls, placing relatively small charges of explosives in the holes at various depths of burial, firing the charges, and measuring the results. The results are scaled to develop charge weight size and depth of burial parameters for production hole sizes. In theory, this procedure is valid and is applicable for surface blasting; however, in underground applications where the craters are inverted, there is some question as to whether extrapolation of the test results is appropriate. In addition, craters are inverted, with resultant gravity stresses, and testing is often done in a waste rock drift where the properties differ from those in ore rock. There is also some question as to whether a 51- or 76-mm (2- or 3-in.) diam charge behaves similarly to a 165-mm ($6\frac{1}{2}$-in.) charge in a confined borehole.

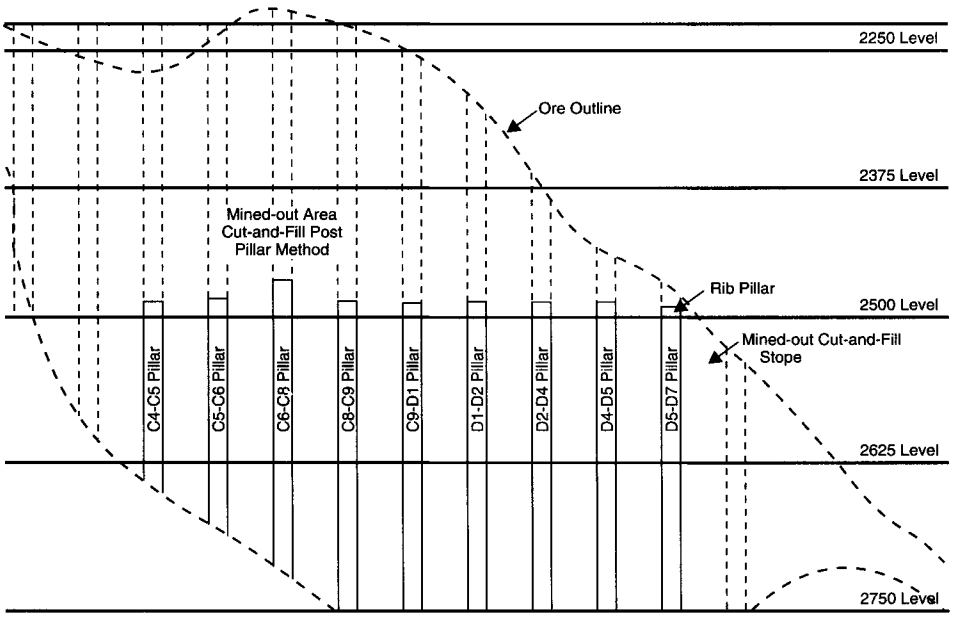


FIGURE 15.1 Cross-sectional view at the rib pillars

Crater testing at Strathcona involved drilling 76-mm (3-in.) diam holes at various depths ranging from 1 to 3 m (3 to 10 ft). The test holes were drilled in the wall of a predominately waste drift as well as in a stope wall. Charges were placed at various depths of burial, were confined, fired, and the results measured. Although the results were somewhat erratic, possibly because of the aforementioned reasons, they did provide guidelines for basic design for production application.

MINING METHOD APPLICATION FOR PILLAR RECOVERY

Following initial crater testing, assessment, and technical evaluation, it was decided to use this blasting technique in a production application, and to incorporate crater blasting in the development of a vertical retreat type of blasthole method.

A series of rib pillars located in the Deep Zone stoping area, between the 2500 and 2750 levels, appeared to lend themselves best to the proposed stoping method (see Fig. 15.1). These pillars are nominally 6.7 m (22 ft) wide, up to 107 to 122 m (350 to 400 ft) in length, about 61 m (200 ft) high, and were designed to limit the stoping span and to provide support to the walls during primary cut-and-fill operations in the ore block. At this point adjacent mining had progressed beyond the top of the pillars. There was no evidence that these pillars are under any significant induced mining stresses.

Adjacent mined-out cut-and-fill stopes approximately 15 m (50 ft) wide were mined out with uppers and the open flat-back breasting method. These stopes were backfilled with a consolidated cement-to-tailings ratio of 1:32 with

254- to 305-mm (10- to 12-in.) thick floors consisting of 1:8 every 2.7 to 3.6 m (9 to 12 ft), depending on the height of the cut taken at the time.

In the initial pillar recovery program, prior to the crater blasting method, the tight cut-and-fill method was employed quite successfully. Some square-set sections were required in areas where the pillar had broken up near the top, but generally the ground was controllable with rockbolts and screen, and there was little fill dilution as a result of mining alongside the mined-out primary stopes.

In order to improve productivity, a conventional sublevel stoping method with longhole ring drilling was introduced as a method of recovering the ribs, and proved quite successful. Some fill dilution was experienced; however, this was limited to a maximum of 1.2 to 2 m (4 to 7 ft) sloughing off in local areas, due largely to blasthole deviation. Some operational problems such as caved longholes due to broken ground and redrilling due to hole deviation were encountered with this method.

In advance of the crater blasting method, an ITH machine was acquired to drill large-diameter 165-mm ($6\frac{1}{2}$-in.) holes, and production drilling began in the initial stope from which an assessment and evaluation of the mining method would be made. The proposed method would involve blasting horizontal slices of ore into an undercut. Muck would be recovered by a drawpoint system, the open stope would be backfilled with consolidated fill, and mining would progress to an adjacent panel.

STOPE DEVELOPMENT, DRILLING, AND BLASTING

Development

In the recovery of the rib pillars described earlier, one of the obvious advantages of crater blasting (vertical retreat blasthole method) over other methods is the reduction in the amount of stoping development required. Development of the stope involves excavation of an overcut, an undercut, drawpoint cones, and access drifts. A slot raise, either for a breaking face or for ventilation purposes, has not been required in this application.

Cone development, in the first panel of one of the pillars in the initial experimental program, involved using a modified tight cut-and-fill rill method (see Fig. 15.2). This method was used in order to maximize recovery of the ore in the proposed cone area and to minimize the amount of development required. Waste development muck was used to form the rill, and the method was considered successful. Also, a crater cut method was used to develop the slot in the cone prior to slashing with conventional longholes. This involved drop raising with crater charges for approximately 12.2 m (40 ft). No major problems were encountered.

Excavation of the overcut generally involves leaving a narrow rib or shell pillar 0.9 to 1.2 m (3 to 4 ft) thick along each wall to provide vertical support for the crown pillar above the overcut. Ground support is limited to the installation of 2-m (7-ft) resin-grouted rebars in the back and walls of the overcut

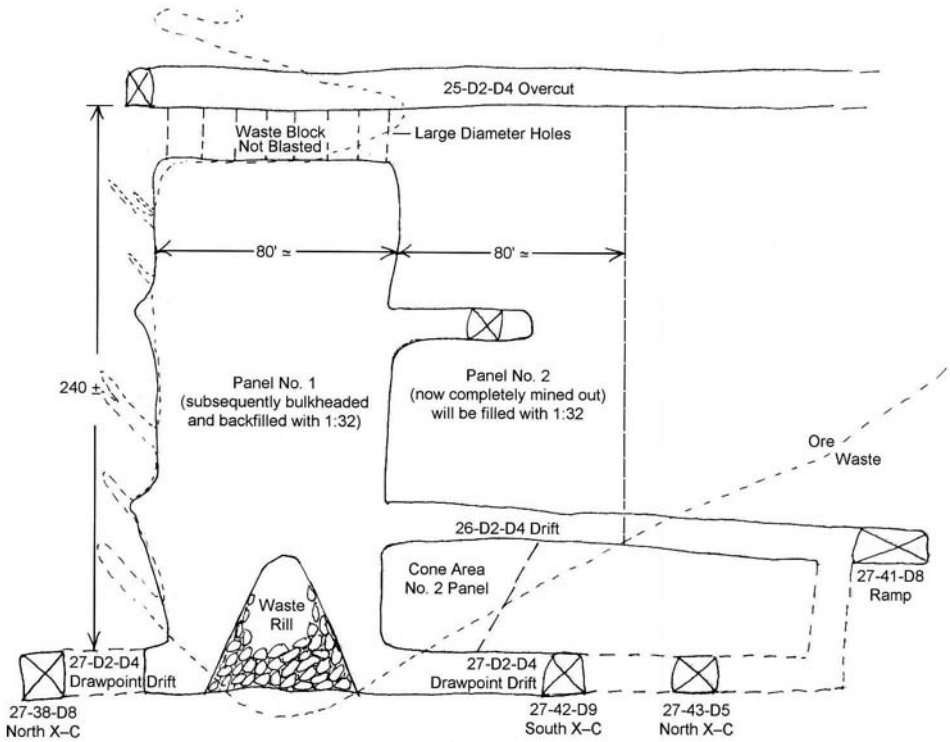


FIGURE 15.2 Longitudinal section through 27-D2-D4 pillar showing the initial two panels and waste-rill cone for panel No. 1, metric equivalent: ft × 0.3048 = m

and in the brow and walls of the drawpoint area. An expanded metal or heavy-duty wire-mesh screen is also placed on the back of the overcut. Timber sets or posts are stood up in the overcut as a precautionary support for the immediate back area when it is felt that they are necessary. Mechanical rockbolts, and a screen if required, are installed in the back of the undercut.

The pillars mined to date have been divided into panels 24 m (80 ft) in length. The primary reason for this dimension restriction is commitment to previous drawpoint locations development. Future plans call for panels 30 m (100 ft) or greater in length.

Drilling

Drilling is done with an ITH drill set up in the overcut. Holes are 165 mm ($6\frac{1}{2}$ in.) diam, are vertical, and average 56 m (185 ft) in length. All holes are drilled to break through into the undercut. From the undercut it is possible to observe the amount of deviation and, if necessary, to redrill, although deviation has not been a major problem. No extensive analysis of deviation has been done in this stoping block; however, from general experience in Sudbury operations, 1 to 2% deviation with careful setup is possible with this hole length and inclination.

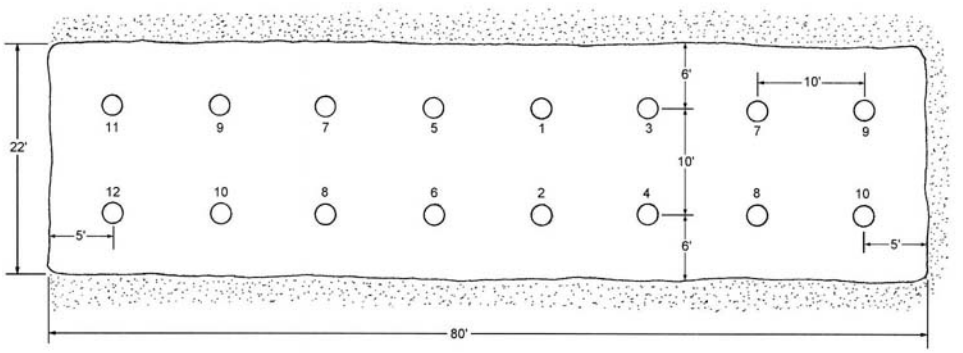


FIGURE 15.3 Plan of the rib pillar panel showing the blasthole pattern and order of firing, metric equivalent: ft × 0.3048 = m

The blasthole pattern (Fig. 15.3) is nominally 3 × 3 m (10 × 10 ft), but because of lack of maneuvering room for the drill because of timber sets or irregular walls, patterns of 1.8 m (6 ft) and 2.7 m (9 ft) of spacing and 3 m (10 ft) of burden, involving two lines of holes, have also been used successfully. Initially in the trial period, a three-hole pattern was drilled, which was essentially a two-hole 3 × 3-m (10 × 10-ft) pattern with a third hole staggered between lines and in the center of the stope. This extra hole was found to be unnecessary for the required degree of fragmentation.

Blasting

This phase of the mining operation is carried out from the overcut. Horizontal slices of ore are blasted into the undercut using it as a free face, and breaking is by a combination of cratering and slashing. The loading operation involves careful measuring of the lengths of holes, setting of the plugs, charging the holes, priming, and firing. Experience has shown that tight technical and operating control of all aspects of the operation is required for its success. This is due mainly to critical measurements concerning the positioning of the charge—proper charge weight, proper delaying sequence, and initiation system—all of which are intrinsic to the crater blasting technique.

Experimentation with depths of burial from approximately 0.9 to 1.2 m (3 to 4 ft) to 2.4 m (8 ft) resulted in approximately 1.8-m (6 ft) being considered the best "working" depth of burial for this site. A depth of burial of 1.8 m (6 ft) normally results in a vertical break of 3 to 3.6 m (10 to 12 ft). This overbreak above the toe of the charge is largely attributed to the fractured nature of the rock in the pillar.

In keeping with the definition of a spherical charge, the charge is limited to 990 to 1016 mm (39 to 40 in.), which results in an L:D ratio of 6:1 for a 165-mm (6½-in.) diam hole. Packaged slurry (water gel) is used as the explosive, and approximately 34 kg (75 lb) are required for this length. All holes in one horizontal lift of the pillar are loaded and fired on one blast. Holes are delayed using either LP or SP millisecond delay blasting caps. Cutoffs have not been a problem.

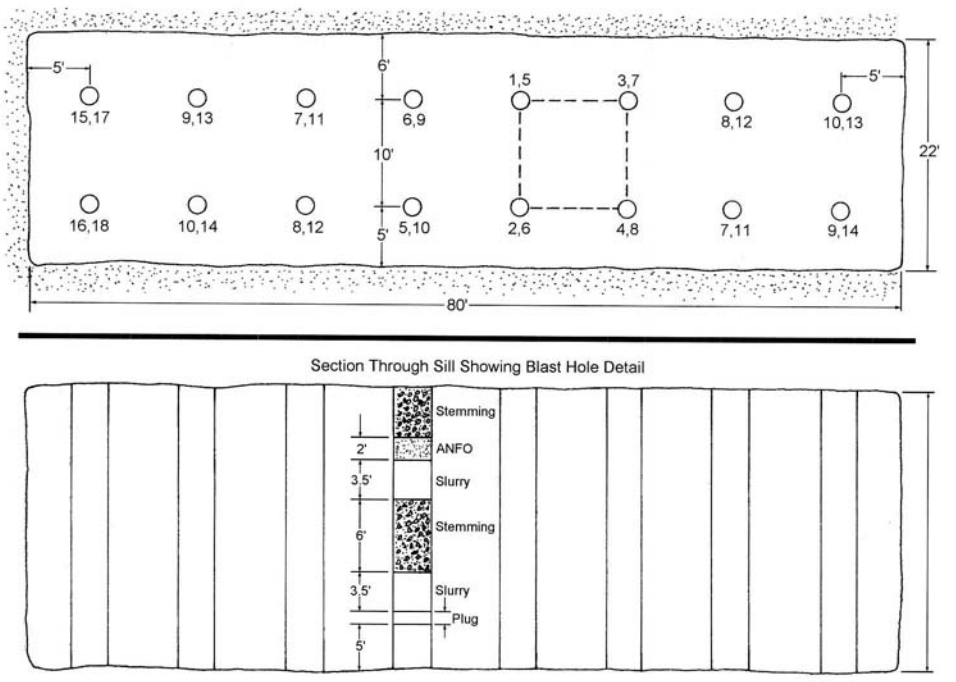


FIGURE 15.4 Plan showing the overcut sill floor with blastholes and firing order and a section through the sill showing blasthole detail, metric equivalent: ft × 0.3048 = m

Blasting of the horizontal slices continues up to the sill pillar under the overcut. Sill pillar thicknesses have ranged from 7.6 to 10.6 m (25 to 35 ft), and experience shows that it is preferable to blast to a 7.6-m (25-ft) thickness because this makes for less complicated sill recovery. The sill pillar is recovered in a one-stage blasting operation using stacked or decked spherical charges. The technique basically involves bringing a slot up through the pillar by cratering and subsequently breaking the remainder of the pillar into it by a combination of cratering and slashing using spherical charges (see Fig. 15.4).

The general steps followed in hole loading in both the lower section of the stope as well as in the sill pillar (see Fig. 15.5) are as follows: a wooden plug is located approximately 1.2 m (4 ft) from the free face of the undercut by lowering it down the hole with a 6.3- or 9.5-mm ($\frac{1}{4}$- or $\frac{3}{8}$-in.) polypropylene rope, and care must be taken to ensure this is done through careful measurement; one 11-kg (25-lb) bag of slurry is cut in half and both pieces are dropped down the borehole, forming a tight seal with the plug; a 0.45-kg (1-lb) high-strength cast-type primer is lowered into the hole on a detonating cord (Scuf-flex) downline; two additional 11-kg (25-lb) bags of slurry are dropped into the hole, which is remeasured to locate the exact position of the top of the charge; a high-strength blasting cap is attached to the detonating cord at the collar of the hole and is in turn attached to a trunkline laid out in the overcut; and the holes are stemmed with water immediately prior to firing the blast (a water hose readily available in the stope makes this a convenient as well as an

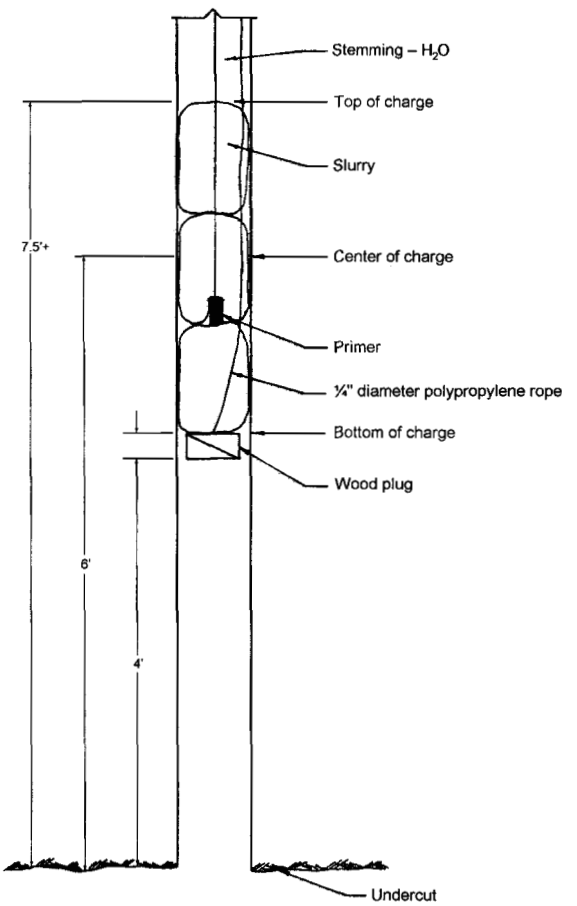


FIGURE 15.5 Section through the blasthole showing loading detail, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m

effective form of stemming, and little water is lost through cracks in the rock). Approximately 1.8 to 2.5 m (6 to 8 ft) of stemming is used. This type of stemming has been found to be sufficiently effective in this operation. More recently, crushed gravel, normally 6.4 to 9.6 mm ($\frac{1}{4}$ to $\frac{3}{8}$ in.) angular, has been found to be more effective for charge confinement.

RESULTS OF APPLICATION IN RIB PILLAR RECOVERY

To date, the crater blasting method has been employed in the recovery operations of two pillars, and two panels have successfully been mined in each pillar. Plans have been made for the recovery of the remaining seven pillars in this mining zone by this method. Significant improvements in the specific areas of cost benefits, improved ground control, and improved safety over previous methods of pillar recovery have been realized.

OTHER APPLICATIONS OF THE CRATER BLASTING METHOD

Other applications of the crater blasting method currently in use at Falconbridge include drop raising with a crater cut and use of this method in a vertical retreat (shrinkage) mining method in primary stoping. Additionally, consideration has been given to its utilization in the recovery of post pillars at the Strathcona operation.

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CHAPTER 16

Cost Calculations for Sublevel Longhole Stoping

Gordon M. Pugh and David G. Rasmussen

INTRODUCTION

Sublevel longhole stoping is a large tonnage stoping method that may be used when there is a fairly competent straight vein and the walls will stand with little support. It is a method that lends itself to a high degree of mechanization. Two variations of the method are illustrated in Figs. 16.1 and 16.2.

Figure 16.1 illustrates development typical for a 3-m (10-ft) wide vein. Initial development is by a lower haulageway, which may be either by track or trackless.

A two-compartment raise is driven in the footwall of the vein. At about 12 m (40 ft) above the main heading, a footwall scram is driven and pockets through the vein are cut on 7.6-m (25-ft) centers. Each of the sublevel blocks selected in this example is 45.7 m (150 ft) long by 12 m (40 ft) high.

Stoping starts at the slot raises at stope endlines and retreats to the timbered raise. This retreat sequence starts on the lowest sublevel, and the opening cut is blasted into the slot raise. This retreating system is used in order to provide a good safe back over the miners working in any given sublevel. A crawler-mounted down-the-hole or conventional pneumatic drill puts the holes down from one sublevel to the next on any desired drill pattern. Gravity flow brings the broken ore down to the scram level. Figure 16.1 shows a large slusher mounted at the timbered raise and slushing from the pockets back to the raise chute.

Development of the ore block in Fig. 16.2 is typical for a 5.5-m (18-ft) wide vein. Initial development is by a trackless heading driven in the footwall of the vein. Pockets are then put into and through the vein on 7.6-m (25-ft) centers. For ease in using trackless rubber-tired mucking equipment, these pockets are crosscut to the vein at approximately 1.04 rad (60°) rather than perpendicular. A two-compartment timbered raise is driven down from one level to the

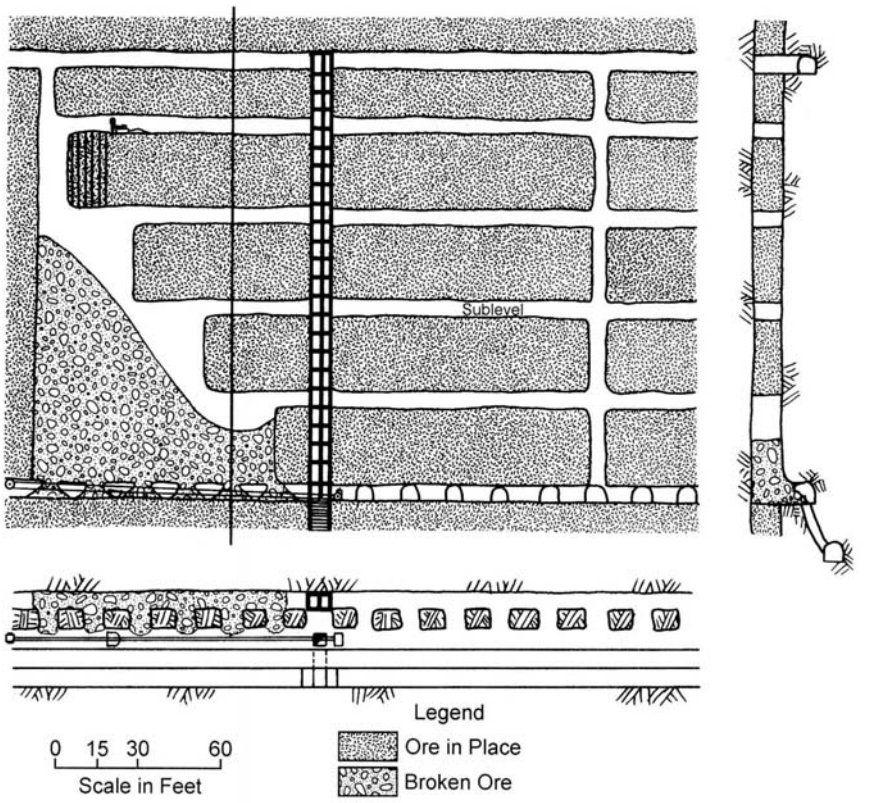


FIGURE 16.1 A narrow vein sublevel longhole stope with a scram drift developed above haulage level. The sublevels are on 12-m (40-ft) centers. Blastholes are drilled with a small drill jumbo. Metric equivalent: ft × 0.3048 = m.

next. Endline slot raises are again put through from level to level. Each of the sublevel blocks selected in this example is 40 m (130 ft) long by 33.5 m (110 ft) high.

Stoping starts at the slot raises and retreats back to the timbered raise. Longholes are drilled from one sublevel down to the next by starting between the lowest pair of sublevels and then working up the ore block. Mucking out of the pocket drawpoints is accomplished by rubber-tired load-haul-dump (LHD) units, which then dump into a centrally located orepass.

COSTS AND PRODUCTIVITIES

The costs derived were calculated, based on knowledge of both the productivity and cycle time elements, for each of the mining methods being considered. Costs for equipment, fuel, tires, and other expendables are 1976 figures, and equipment hourly operating costs for each piece of equipment are included at the end of the entire cost section. The labor cost used has been fixed at $10.50 per hr, based on average rates at underground mines in the United States and including a 35% mark-up for fringe benefits and 25% for an incentive payment system.

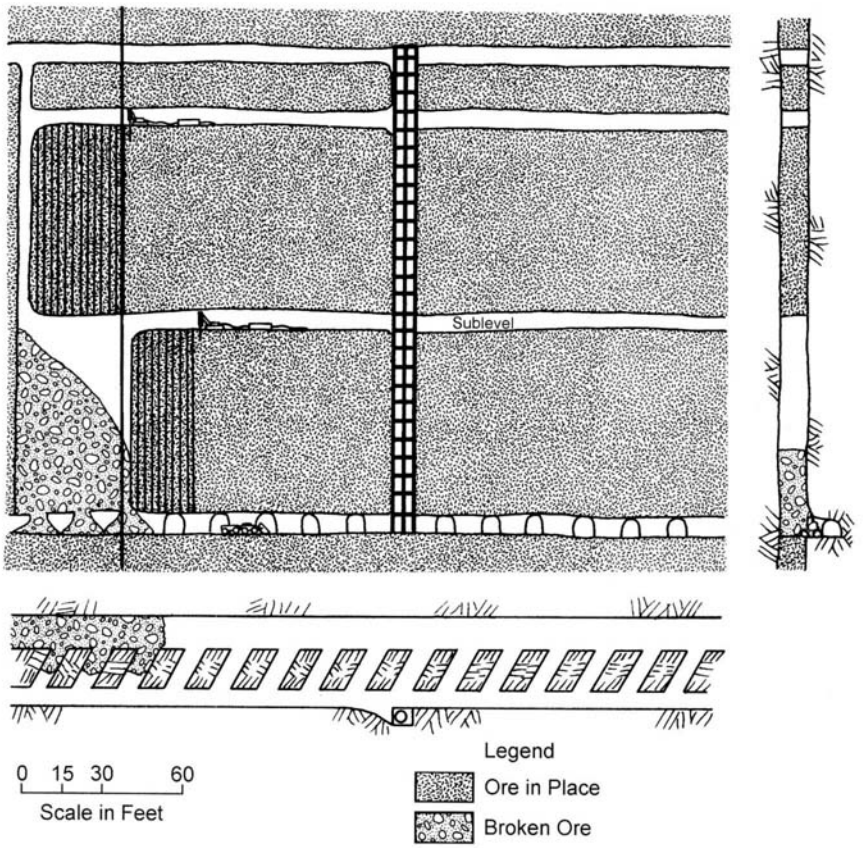


FIGURE 16.2 A mechanized sublevel longhole stope with a scram drift at haulage level. The sublevels are on 30-m (100-ft) centers. Blastholes are drilled with a down-the-hole drill. Metric equivalent: ft × 0.3048 = m.

The total mining cost, based on the foregoing assumptions, consists of two parts: development costs and stoping costs. Development costs are determined by first creating a development plan of drifting, raising, and chutes or mill holes. Several specific development plans are considered for each stoping method. Prior to this, appropriate unit costs were applied to the development plan. Total costs calculated from dollar per foot figures were then expensed over the ore block directly associated with the development. The development costs do not include shafts, slopes, or other major support systems, but only the costs incurred for the development of the specific tonnage block.

Stoping costs were calculated, using derived unit operation costs, for drilling, blasting, mucking, stope preparation, and ground support installation when required. Selection of proper unit operations was made for specific stoping cases and their related costs compiled to give a stoping cost.

It should be reemphasized that costs given include labor, direct operating costs of equipment, and expendables. These costs have been allocated to the

TABLE 16.1 Development costs for 3 x 3-m (10 x 10-ft) trackless heading, 1976 U.S. dollars

Conditions*	Trackless and Off the Vein, No Timber	
Round drilled with 2 jacklegs and 1 3/8-in. bits		
Drill 6.5 ft and advance face 6 ft		
2-person crew		
48.6 st broken per blast		
Cycle times		
Drill and blast	3.5 hr (approximately)	
Muck removal, 2 cu yd rubber-tired	2.5 hr	
Ground support†	1.0 hr	
Miscellaneous: 6 ft of 4-in. pipe, 2-in. pipe, and vent bag	1.0 hr	
Travel and lunch	2.0 hr	
Total	10.0 hr	
Costs	**Labor, $**	**Supplies, $**
Drill and blast, 7 miner-hours	73.50	96.78
Muck removal, 5 miner-hours	52.50	
Loader maintenance @ $0.30 per st		14.58
Ground support, 2 miner-hours	21.00	
6 rockbolts @ $2.64		15.84
2 mats @ $2.70		5.40
Miscellaneous, 2 miner-hours	21.00	
6 ft of 2-in. pipe @ $1.87 per ft		11.22
6 ft of 4-in. pipe @ $3.98 per ft		23.88
6 ft of vent tube @ $5.50 per ft		33.00
Small tools and miscellaneous supplies		
Travel and lunch, 4 miner-hours	42.00	
	210.00	200.70
Total = $410.70, or $68.45 per ft		

*Metric equivalents: in. x 25.4 = 1 mm; ft x 0.3048 = m; cu yd x 0.7645549 = m^3; st x 0.9071847 = t.

†One rockbolt per foot of advance is assumed.

assigned tonnage for the example stoping block. Equipment capitalization and depreciation have not been included.

The costs for stope development are given in Tables 16.1 through 16.6. Productivities and cycle times for various cases are given in Tables 16.7 and 16.8, indicating a stoping cost range of $3.10 to $6.47 per st, and productivities from 34.5 to 89.9 t (38.1 to 99.2 st) per miner-shift. For detailed cost calculations, see G.M. Pugh and D.G. Rasmussen (1977), "Stope Mechanization—Vein Mining," U.S. Bureau of Mines HO 26024, August, Dravo Corp., especially Section 3.1, Chapter 4; Section 3.2, Chapters 10 and 11; Section 3.3, Chapter 5; and Section 3.4, Chapter 7.

TABLE 16.2 Development conditions and costs for 3 × 3-m (10 × 10-ft) crosscut to vein, 1976 dollars

Conditions*	10 × 10-ft Trackless Short Crosscut to Vein	
Round drilled with 2 jacklegs ($1\frac{3}{8}$-in. hole diam)		
Drill 6.5 ft advance 6 ft		
2-person crew		
48.6 st broken per blast		
Mucking with Cavo-type mucker		
Cycle times		
Drill and blast	3.5 hr	
Muck	3.0 hr	
Ground support†	1.0 hr	
Travel and lunch	1.5 hr	
	9.0 crew hr	
Costs	**Labor, $**	**Supplies, $**
Drill and blast, 7 miner-hours	73.50	96.78
Muck removal		
Labor, 6 miner-hours at $10.50 per hr	63.00	
Mucker maintenance, 48.6 st at $0.41 per st		19.93
Ground support		
Labor, 2 miner-hours at $10.50 per hr	21.00	
Bits, steel, and drill, 36 ft at $0.15 per ft		5.40
6 rockbolts at $2.64		15.84
2 mats at $2.70		5.40
Travel and lunch, 3 miner hours at $10.50 per hr	31.50	
Miscellaneous supplies, 6% labor		11.34
	189.00	154.69
Total cost per 6 ft advance = $343.69		
Cost per ft advance = $ 57.28		

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; cu yd × 0.7645549 = m^3; st × 0.9071847 = t.

†One rockbolt per foot of advance is assumed.

TABLE 16.3 Conditions and costs for sublevel development of a 5.5-m (18-ft) wide vein, 1973 dollars

Conditions*	18 × 10-ft Trackless Sublevel Development	
2-person crew, drill with jumbo 7.6 ft advance		
Muck with LHD		
Ground support rockbolts, 3 bolts per ft advance		
Short tons broken per blast	110.81	
Cycle times		
Drill and blast	3.5 hr	
Muck	3.0 hr	
Ground support	3.0 hr	
Miscellaneous travel and lunch	2.0 hr	
Cost	**Labor, $**	**Supplies, $**
Drill and blast, 7 miner-hours at $10.50 per hr	73.50	
Muck		
Labor, 3 miner-hours at $10.50 per hr	31.50	
Maintenance and supplies, 3 hr at $17.60 per hr		52.80
Ground support		
Labor, 6 miner-hours at $10.50 per hr	63.00	
Bits, steel, and maintenance, 126 ft at $0.15 per ft		18.90
21 rockbolts at $2.64 per bolt		55.54
3 mats at $2.70 per mat		8.10
Miscellaneous travel and lunch,		
4 miner-hours at $10.50 per hr	42.00	
Small tools, etc., 6% labor		12.60
	210.00	147.94
Total cost per 7.6 ft advance = $357.94		
Cost per ft advance = $ 47.10		

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 16.4 Summary of drilling and blasting costs for stoping, 1976 dollars

Method Description*	Tonnage per Round	Blasting Cost, $ per st*	Drilling Cost, $ per st*	Total Cost, $ per st*
2-in. diam down holes,				
40-ft long, jumbo	388.8	0.46	0.83	1.29
4-in. down-the-hole drill holes,				
100-ft long, jumbo	2332.8	0.28	0.69	0.97

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 16.5 Summary of mucking costs, 1975 dollars

Mucking Machine Designation*	Labor Operator Cost $/hr	Machine Operating Cost $/hr	Machine Ownership Cost $/hr	Production Rate st/hr*	Labor Operating Cost $/st*	Machine Operating Cost $/st*	Machine Ownership Cost $/st*	Total $/st*
LHD								
1 cu yd–300 ft haul	10.50	5.62	4.83	17.5	0.60	0.32	0.28	1.20
2 cu yd–600 ft haul	10.50	8.90	5.29	28.0	0.38	0.32	0.19	0.89
5 cu yd–600 ft haul	10.50	31.32	8.21	65.0	0.16	0.48	0.13	0.77
Cavo 511								
150 ft haul	10.50	6.70	5.03	22.1	0.48	0.30	0.23	1.01
Slusher								
150 ft	10.50	1.05	0.25	10.5	1.00	0.10	0.02	1.12

*Metric equivalents: ft x 0.3048 ≈ m; st x 0.9071847 = t.

TABLE 16.6 Sublevel longhole development cost summary, 1976 dollars

	10-ft Wide Vein, Development in Footwall with a Crosscut to Vein, Slushing into Orepass* (Fig. 16.1)	18-ft Wide Vein, Development in Footwall with Crosscut to Vein. LHD Equipment and Big Hole Drill Used* (Fig. 16.2)
Haulage level development	8 x 9 ft bald drift $98.25 per ft x 300 = $29,475	10 x 14 ft bald drift (trackless) $71.36 per ft x 420 = $29,971
Scram drift development	10 x 10 x 300 ft $57.28 per ft x 300 ft = $17,184	
Crosscut to vein	10 x 10 ft bald $57.28 per ft x 10 ft x 12 each = $6874	10 x 10 ft bald drift (trackless) $68.45 per ft x 25 ft x 15 each = $25,669
Undercut vein	10 ft wide x 10 ft high x 300 ft $71.36 per ft x 300 = $21,408	18 ft wide x 10 ft high x 420 ft $73.75 per ft x 420 = $30,975
Slot raises (Alimak)	7 x 7 x 220 ft $82.47 per ft x 220 ft x 2 each = $36,286	7 x 7 x 250 ft $82.47 per ft x 250 ft x 2 each = $41,235
Manway-haulageway raise	13 x 7 ft (2 compartment) $117.25 per ft x 220 = $25,795	13 x 7 ft (2 compartment) $117.25 per ft x 250 ft = $29,313
Sublevel development	10 ft wide x 10 ft high x 300 ft $57.28 per ft x 300 ft x 4 each = $68,736	18 ft wide x 10 ft high x 400 ft $73.75 per ft x 420 ft x 2 each = $61,950
Orepass	5 ft diam x 220 ft $55.10 per ft x 200 = $12,122	5 ft diam x 250 ft $55.10 per ft x 250 = $13,775
Chute front	1 each at $3200 = $3200 Excavation $7228	1 each at $3200 = $3200 Excavation $7228
Total cost	$228,308	$243,316
Tons of ore developed	10 ft wide x 230 ft high x 300 ft long $690{,}000 \times \frac{162}{2000} = 55{,}890$	18 ft wide x 260 ft high x 430 ft long $2{,}012{,}400 \times \frac{162}{2000} = 163{,}004$
Development cost per st ore	$4.08	$1.49

*Metric equivalents: ft x 0.3048 ≈ m; st x 0.9071847 = t.

TABLE 16.7 Stoping cycle times/productivities (miner shifts), 1976 dollars

	A	B	C	D
Tonnage per round (st*)	388.8	388.8	2332.8	2332.8
Drill and blast time (hr)	4.5	4.5	18.0	18.0
Muck time (hr)	5.7	3.4	10.4	5.5
Total miner-shifts†	10.2	7.9	28.4	23.5
Productivity (st per miner-shift)	38.1	42.9	82.1	99.2

Example of productivity calculation (case C):

$$\frac{\text{2332.8 st per round}}{\text{28.4 miner-shifts per round}} = \text{82.1 st per miner-s hift}$$

*Metric equivalent: st × 0.9071847 = t.

†Stope crew size: 3 miners working 6.5 hr per 8-hr shift.

TABLE 16.8 Sublevel longhole stoping—costs and productivities, 1976 dollars

Unit Operation*	Unit Cost, $/st of Ore*	Vein Width 10 ft* Small Diam Longhole Stopes with Scram Drift Developed above Haulage, Good Ground Conditions		Vein Width 18 ft* Large Diam Down-the-Hole Longhole Stope with Mechanized Loading, Good Ground Conditions		Qualifications
Development		A	B	C	D	8-hr shift, 6.5 hr effective time at face.
Development						
Case I 10-ft vein	4.08	4.08	4.08	—	—	Labor rate $10.50 per hr includes bonus and fringe.
Case II 18-ft vein	1.49	—	—	1.49	1.49	
Drill and blast						Unit operation costs include labor.
2-in. diam downholes, 40 ft long, 10-ft vein 388.8 st, 2-in. diam	1.29	1.29	1.29	—	—	Equipment costs include maintenance, parts and labor, tires and fuel.
4-in. diam down-the-hole, 100 ft long, 18-ft vein 2332.8 st, 4-in. diam	0.97	—	—	0.97	0.97	Equipment depreciation is not included.
Stoping						All materials are based on 1976 prices.
Mucking						Machine ownership costs are excluded.
Slusher—25 hp	1.10	1.10	—	—	—	
LHD 1 cu yd—diesel	0.92	—	0.92	—	—	
LHD 2 cu yd—diesel	0.70	—	—	0.70	—	
LHD 5 cu yrd—diesel	0.64	—	—	—	0.64	
Total Cost per st		$6.47	$6.29	$3.16	$3.10	
Stope Crew Size		3.0	3.0	3.0	3.0	
Productivity st per miner-shift		38.1	49.2	82.1	99.2	

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; cu yd × 0.7645549 = m^3; st × 0.9071847 = t; hp × 745.6999 = W.

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CHAPTER 17

Cost Estimation for Sublevel Stoping—A Case Study

A.J. Richardson

INTRODUCTION

Before the development of the underground stoping and mining costs can be considered, certain facts about the orebody, the proposed mine, markets, etc., must be known or determined.

In the case to be studied, the zinc-lead mineralization occurred with a narrow vertically dipping structure of undetermined length and vertical extent.

Exploration has revealed 6 million t (6.5 million st) of proven reserves. A further 744,000 t (820,000 st) of indicated reserves is considered capable of being expanded by a factor of approximately four after more detailed drilling.

After studying the market conditions and completing a very preliminary feasibility study, it was decided that production would be 662,000 t/year (730,000 stpy) [or 1800 t/day (2000 stpd)] of ore. First-year production would be at the rate of 1360 t/day (1500 stpd).

The main design criteria for the selection of the mining methods are minimizing surface subsidence, maximum recovery of the orebody, maximum degree of grade control, maximum productivity, and safe working conditions. Two basic extraction systems are considered capable of meeting these requirements: mechanized cut-and-fill stoping and sublevel longhole stoping with filling.

The primary development system of the mine has been designed to give maximum flexibility in stoping systems and layout and to permit changes if considered necessary as a consequence of actual production experience.

Access to the mine is by a circular concrete-lined vertical shaft, 5 m (16 ft) diam, sunk to a depth of 420 m (1380 ft). Two exploration levels have been driven within the ore zone at depths of 50 m (165 ft) and 380 m (1246 ft)

below the surface outcrop. The development had the objective of sampling the mineralization and producing detailed information on the outline of the orebody and the distribution and controls of zinc and lead values.

In an attempt to satisfy the basic design criteria for the mine, it was decided that production would be best achieved by a combination of 40% sublevel longhole stoping and 60% cut-and-fill mining.

Costs of exploration and capital development of permanent underground facilities are normally written off over the life of a mine.

Production expenditures, on the other hand, are of a temporary nature and are normally charged as and when incurred as an operating expense. Reasonably accurate predictions of mine production costs can be built up from engineering design and estimates of individual mine activities for ultimate inclusion in the comprehensive data required for financial decision making. The simulated operations can be costed on a detailed basis in the form of a monthly operating budget.

The budget format can be generalized or detailed, depending upon the scope of the project. However, experience suggests that a fairly detailed format has the advantage of assuring that all significant cost items are included. For underground costing it is suggested that the budget structure include five major cost centers (i.e., development, diamond drilling, ore extraction, hoisting and transportation, and general mine expense). These, in turn, are detailed under numerous subheadings.

The mechanism for compiling an operating budget is illustrated. Because of its relative simplicity, ore extraction under sublevel longhole stoping has been chosen for illustration. All other activities, simple or complex, can be estimated in similar fashion.

BLOCK AND STOPE DEVELOPMENT

Longhole blocks, used where advantageous, will be up to 76 m (250 ft) in height, depending upon the vertical continuity of the mineralization, and approximately 91 m (300 ft) long. Drawpoints will be at 11-m (36-ft) intervals and serviced by loading crosscuts driven from a footwall drift parallel to and close to the ore zone.

Pillars between the stopes will be 15 m (50 ft) wide. Stopes will be drilled off with vertical rings of blastholes drilled from sublevels approximately 18 m (60 ft) apart vertically. This drilling will be done by percussion drilling machines [89 mm ($3\frac{1}{2}$ in.)] mounted on a trackless drilling rig. Load-haul-dump (LHD) equipment will be used to move broken ore from the drawpoints to the orepass connecting to rail haulage systems. On completion, longhole stopes will be backfilled to prevent caving and to facilitate later pillar removal.

From a planned stope layout, a forecast of production and development is made in Table 17.1.

TABLE 17.1 Block tonnage and stope development

	Quantity*	
	Ore	Waste
Total ore block	375,000 st	
2 stopes	310,000 st	
1 pillar	65,000 st	
Access crosscuts, 4 at 100 ft		400 ft
Drill sublevel drifts, 6 at 300 ft	1800 ft	
Stope raises, 3 at 250 ft	750 ft	
Undercut sublevel drifts, 2 at 300 ft	600 ft	
Loadout crosscuts at 35-ft intervals	550 ft	100 ft
	3700 ft	500 ft
Total development footage	4200 ft	
Tons per foot of development	89.29 st	

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 × t.

STOPING

The mine was expected to produce 1360 t/day (1500 stpd) in the first year, increasing to 1814 t/day (2000 stpd) thereafter. Because fragmentation of ore can have considerable influence on the rate of production, the design of drill and blast layouts was made knowing that the optimum muck size would be a primary objective. Basic data are developed as shown in Table 17.2, which indicates that longhole loading would be 2865 m/month (9400 ft per month).

PRODUCTIVITIES AND CREW SIZE

Once a basic forecast has been developed, it becomes possible to determine productivities and crew sizes. Such determination requires experience and good judgment to estimate performance of suppliers and other operations in a specific case (see Table 17.3).

Stope development equipment will consist of a drill jumbo, load-haul-dump (LHD) units, roof bolting rig, and service truck for personnel and supplies. The crew will be composed of a driller, a LHD unit operator, and a utility person. Using the manufacturer's claim for drill jumbo performance and a LHD unit rating of 90.7 t/hr (100 stph) on a short haul, development advance 6 m (20 ft) per 3-person shift for a productivity of 2 m (6.6 ft) per miner-shift.

This calculation assumes 100% reliability and availability of crew and equipment. It does not allow for variable ground conditions, equipment failures, faulty blasts, and other miscellaneous problems, which will plague normal advance. Average productivity is affected by all these events, and it would be reasonable to assume 50% of the original estimate. Therefore, a figure of 1 m (3.3 ft) per miner-shift for stope development is used in figuring productivity

TABLE 17.2 Basic forecast

Powder factor	0.65 lb ANFO per st of ore, or 0.33 kg/t of ore
Loading factor for 2-in. diam holes	1.4 lb per ft, or 2 kg/m
St broken per ft of hole	$\frac{1.4}{0.65} \cong 2$, or 6 t/m
Longhole drilling	$\frac{375{,}000}{2} = 187{,}500$ ft, or 57,000 m
Primary explosive usage	375,000 × 0.65 = 244,000 lb, or 111,000 kg
Monthly forecast production rate	$\frac{1500 \times 365 \times 0.40}{12} = 18{,}250$ stpm, or 16,500 t/month
Block life	$\frac{375{,}000}{18{,}250} \cong 20$ months
Longhole stope development	$\frac{3800}{20} = 190$ ft per month, or 58 m/month
Longhole drilling	$\frac{187{,}500}{20} = 9400$ ft per month, or 2865 m/month
Longhole loading ≅	9400 ft per month, or 2865 m/month

in Table 17.3. To achieve 58 m (190 ft) of development advance per month thus will require 58 miner-shifts per month.

Longhole drilling productivity can show a wide range of performance, taking into account mechanical failures, inexperience, survey error, etc. A productivity of 61 m (200 ft) per miner-shift, including moves, would be acceptable on a sustained basis. In a normal month for which 2865 m (9400 ft) of drilling was previously estimated, the calculated miner-shifts required for drilling would be 47 miner-shifts per month.

Priming, loading, and blasting at a productivity of 190 m (625 ft) per miner-shift would require 15 miner-shifts per month.

Ore production, or loadout, should yield a high productivity because blasting was designed for good fragmentation, and mucking designs were optimized with multiple drawpoints and a short tram. Initial estimates of productivity are based on manufacturers' claims [i.e., 90.7 t/hr (100 stph) for 6 hr of an 8-hr shift].

Therefore the miner-shifts required per month to move normal ore tonnage of 16,500 t (18,250 st) would be 31.

OPERATING COSTS

Estimation of supply costs, which can include equipment rentals, involves such straightforward calculation as explosive costs where powder factor, footage to be loaded, and other item costs can easily be determined. Without

TABLE 17.3 Productivities

Development

Crew size—3

Advance per shift—20 ft, or 6 m

Theoretical productivity— $\frac{20}{3}$ = 6.6 ft per miner-shift, or 2 m per miner-shift

Practical productivity = 3.3 ft per miner-shift, or 1 m per miner-shift

Miner-shifts required per month— $\frac{190}{3.3}$ = 58

Longhole Drilling

Productivity—200 ft per miner-shift, or 61 m per miner-shift

Miner-shifts required per month— $\frac{9400}{200}$ = 47

Blasting

Productivity—625 ft per miner-shift, or 190 m per miner-shift

Miner-shifts required per month— $\frac{9400}{625}$ = 15

Loadout

Productivity—100 stph for 6 hr, or 90.7 t/hr for 6 hr

Miner-shifts required per month— $\frac{18,250}{600}$ = 31

past experience, it is difficult to estimate equipment parts cost. Dealers and other operations can help. Often it is expressed in terms of a machine rental. For this case, overall supply cost involved in ore extraction from longhole stopes is estimated to be $0.44 per t ($0.40 per st).

Determination of maintenance costs involves a complex study of maintenance schedules for the various pieces of equipment. It entails considerable guesswork because there are so many factors involved, the effects of which are unknown. Equipment and operator performance are unknown. The effect of mine conditions on tires and other parts is unknown. Discussion with other operators who have similar conditions to contend with is one of the more reliable means of estimating maintenance costs. For the case under discussion this has been estimated to be $0.275 per t ($0.25 per st).

It is assumed that there is an initial union labor agreement that specifies the base rate of pay for mine job classifications. There may also be an incentive bonus or contract system for underground workers. Payroll loading to cover vacations, pensions, medical plans, unemployment insurance, and worker's compensation is calculated and may run anywhere from 15 to 40%; in this case, it is 25%.

The underground operating labor rate is calculated in Table 17.4.

TABLE 17.4 Loaded labor rate, 1976 dollars

Base rate	$ 8.00 per hr
Contract at 50%	$ 4.00 per hr
Subtotal	$12.00 per hr
Loading at 25%	$ 3.00 per hr
Total	$15.00 per hr
Labor cost per shift	15 x 8 = $120.00

With all basic cost data estimated, it should be possible to determine a total budget cost for longhole open stope ore extraction.

Drilling, Blasting, and Loadout	
Drilling, miner-shifts per month	47
Blasting, miner-shifts per month	15
Loadout, miner-shifts per month	31
Miscellaneous, scaling, bolting, etc.	8
Total operating labor	101 miner-shifts
Normal operating labor cost per month, 101 miner-shifts x $120/miner-shift	$12,120
Equipment and Operating Supplies	
Normal supply cost per month, 16,500 t x $0.441 per t	7,280
Maintenance	
Normal repair labor cost per month, 16,500 t x 0.275 per t	4,540
Total	$23,940

Therefore, the cost per metric ton produced by longhole is $1.45 ($1.31 per st).

CONCLUSION

Estimated operating dollars for other cost centers such as primary development, diamond drilling, hoisting and transportation, concentration, general mine expense, and local general overhead can be determined similarly. The whole can then be put together to form a complete operating budget (see Tables 17.5 to 17.10). This then contains the forecast activity of the operation, the productivities that are expected for each activity, and the operating dollars needed to accomplish the forecast activity. With this information (return on capital, etc.) a profit and loss statement can be prepared. If it is favorable, approval is sought to commence development.

TABLE 17.5 Budget data code

Date September 1976	Compiled by P. F. Saxton	Account Class	Code Name Production Drilling (Longhole)
Date	Accepted by	Standard Period Month	Cost Center 5.225
Date	Approved by	Standard Unit ft*	Normal Budget $7760
Productivity—200 ft per miner-shift			
Tons per foot hole—2 st*			
Tons per month—18,250 st			
Feet per month—9400 ft			
Standard Data			**$/Month**
Operating Labor			
Staff:			
General roll: 47 miner-shifts at $120/miner-shift			$5640
(Rate = $/hr × hr/shift × loading = $/miner-shift)			
Pay $/hr = 8.00			
Bonus $/hr = 4.00			
Loading 25%			
Operating Supplies		**¢/ft**	
Steel		3.1	
Bits		8.0	
Machine maintenance		(see below)	
Tools and supplies		2.0	
Tanner gas		0.4	
Jumbo maintenance		(see below)	
Freight		2.0	
Miscellaneous		2.0	
		17.5 × 9400	1650
Repair Labor and Supplies			
Labor: included in general maintenance			
Supplies: jumbo maintenance 3.0¢/ft* drill maintenance 2.0¢/ft*		5.0 ¢/ft × 9400	470
Normal Budget Allowance			$7760

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 17.6 Budget data code

Date September 1976	Compiled by P. F. Saxton	Account Class	Code Name Production Blasting (Longhole)
Date	Accepted by	Standard Period Month	Cost Center 6.225
Date	Approved by	Standard Unit st*	Normal Budget $9745

Powder Factor—0.65 lb per st*		
Tons ore per month—18,250 st		
Ft of hole per month—9400 ft*		
Standard Data		**$/Month**
Operating Labor		
Staff:		
General roll:		
Loading and primary	15 miner-shifts at $90/miner-shift	$1350
Blasting	2 miner-shifts at $90/miner-shift	180
Base pay = $7.00/hr		
Bonus pay = $2.00/hr		
Loading 25%		
Operating Supplies		
ANFO at $0.20/lb, need 0.65 × 18,250 × $0.20 =		2375
Bottom primer at $0.50 unit, $\frac{9400}{60} \times 0.50 =$		80
Top primer at $0.50 unit, $\frac{9400}{60} \times 0.50 =$		80
Primacord at $0.50 per ft, 9400 × 1.20 × 0.05 =		565
Other accessories at $0.03/st, . . 0.03 × 18,250 =		550
Repair Labor and Supplies		
Loading machine maintenance at $0.10/st: 18,250 × 0.10 =		1825
Hoses, etc., at $0.05/st: 18,250 × 0.05 =		915
Transportation at $0.10/st: 18,250 × 0.10 =		8125
		$16,045

*Metric equivalents: ft × 0.8048 = m; st × 0.9071847 = t.

TABLE 17.7 Budget data code

Date September 1976	Compiled by	Account Class	Code Name Transportation
Date	Accepted by	Standard Period Month	Cost Center 6.225
Date	Approved by	Standard Unit st*	Normal Budget $

1 person on LHD—1 person on train

LHDs

1 ST-5A loads 625 st/shift

18,250 stpm

30.4 days/month

600 stpd to be moved

LHD shifts/day = $\frac{600}{625}$ = 1 shift/day

LHD shifts/month = 30

Trains

Train moves = 8 × 12 = 96 st/trip

Train makes about 20 trips/shift

Total st moved/shift = 96 × 20 = 1,920

Train shifts/day = $\frac{600}{1920}$ = 0.31

Train shifts/month = 9.5

Standard Data **$/Month**

Operating Labor

Staff:

General roll:

LHD 30 shifts at $120/shift

Train 9 shifts at $120/shift

($8.00/hr base, $4.00/hr bonus, 25% loading)

Operating Supplies

Power:

LHD (fuel) 30 shifts at $3.50/shift

10 gal/hr at 7 hr/shift at $.50/gal*

Trains (electricity)

Lubrication:

LHD

Trains

Maintenance Labor

Incorporated in shops

Maintenance Supplies

LHD:	Trains:
Tires	Car repair
Parts	Parts

Hydraulic fluid

Normal Budget Allowance

*Metric equivalents: gal × 3.785412 = L; st × 0.9071847 = t.

TABLE 17.8 Budget data code

Date September 1976	Compiled by	Account Class	Code Name Summary—Longhole Stoping	
Date	Accepted by	Standard Period Month	Cost Center	
Date	Approved by	Standard Unit	Normal Budget $23,940	
Drilling, Blasting, and Loadout				
Drilling, miner-shifts per month			47	
Blasting, miner-shifts per month			15	
Loadout, miner-shifts per month			31	
Miscellaneous, scaling, bolting, etc.			8	
Total operating labor			101 miner-shifts	
Normal operating labor cost per month, 101 × 120 =				$12,120
Equipment and Operating Supplies				
Normal supply cost per month, 18,250 × 0.40 =				7,300
Maintenance				
Normal repair labor cost per month, 18,250 × 0.25 =				4,520
Total				$23,940

Cost per st* — $\frac{23{,}940}{18{,}250}$ = $1.31

Metric equivalent: st × 0.9071847 = t.

TABLE 17.9 Operating budget detail

		$/Month 1000s	1500 stpd 45,625 stpm $/st*	2000 stpd 60,833 stpm $/st*
Development				
20.01	Track development	$ 2.7		
20.02	Scooptram primary development	4.7		
20.03	Borehole raising	8.7		
20.04	Conventional raising	4.8		
20.05	Subdrifting	9.8		
20.06	Cut-and-fill stope development	4.7		
	Total development	$35.4	$0.77	$0.58
Diamond Drilling				
21.01	Corehole diamond drilling	$ 8.6	$0.19	$0.14
Ore Extraction				
30.01	Longhole stoping	$23.9		
30.02	Cut-and-fill stoping	31.9		
	Total ore extraction	$55.8	$1.20	$0.92
Hoisting and Transportation				
40.01	Hoisting and shaft maintenance	$ 8.3		
40.02	Transportation	8.8		
		$17.1	$0.38	$0.28
General Mine Expense				
50.01	Air and waterlines	$ 5.0		
50.02	Pumping and drainage	2.1		
50.03	Ventilation	3.1		
50.04	Engineering	6.2		
50.05	Geology	7.4		
50.06	Surface transportation	0.83		
50.07	Mine lamps	0.83		
50.08	Mine supervision	10.1		
50.09	Bits and rods	7.5		
50.10	Hoses and drill repair	3.5		
50.11	Hand tools	1.3		
50.12	Rockbolt and timber	5.0		
50.13	Mucking machines	0.22		
50.14	Underground lighting and telephone	0.42		
50.15	Mobile equipment	53.2		
50.16	Power and compressed air	17.2		
		$123.9	$270	$2.04
Crushing				
59.10		$ 2.3	$0.05	$0.037
Local Overhead				
11.01	Supervision	$ 5.1		
11.02	Personnel	0.33		
11.03	Finance	3.5		
11.04	Purchasing and warehousing	2.6		
11.07	Communication	1.7		
11.08	Employee transportation	3.3		
11.09	Insurance, fire protection, taxes	4.8		
11.11	Mine overhead	1.7		
13.01	Camp buildings	1.7		
13.02	Camp services	3.3		
13.03	Yards and roads	3.3		
13.04	Company residences	2.5		
13.05	Surface reclamation	0.08		
13.08	Dry facilities	1.33		
13.09	Safety and first aid	1.6		
13.10	Surface maintenance	5.0		
	Total local overhead	$41.8	$0.92	$0.69

Metric equivalent: st × 0.9071847 = t.

TABLE 17.10 Operating budget summary

Summary	$/Month 1000s	1500 stpd 45,625 stpm $/st*	2000 stpd 60,833 stpm $/st*
Development	$ 35.4	$0.77	$0.58
Diamond drilling	8.6	0.19	0.14
Ore extraction	55.8	1.2	0.92
Hoisting and transportation	17.2	0.38	0.28
General mine expense	123.8	2.7	2.04
Crushing	2.3	0.05	0.037
Total mine	$243	$5.29	$3.997
Local overhead	$ 41.8	$0.92	$0.69

*Metric equivalent = st × 0.9071847 = t.

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SECTION 3

Stopes Requiring Additional Support Other Than Pillars

CHAPTER 18

Shrinkage Stoping: An Introduction

William Lyman

GENERAL DESCRIPTION

Shrinkage, or shrinkage stoping, refers to any mining method in which broken ore is temporarily retained in the stope to provide a working platform and/or to offer temporary support to the stope walls during active mining. Because ore "swells" when broken, it is necessary to shrink the muck pile a corresponding amount by drawing some of the broken ore out as the stope is advanced–hence the name. Broken ore retained during stoping is drawn out after the stope has reached its limits. The stope may be left empty or may be filled with waste contemporaneous with, or subsequent to, the final draw. Traditionally the method implies conventional overhand stoping methods with miners working between the muck pile and the stope back, in a space that advances updip with mining and is maintained by balancing "swell" with "shrink." The shrinkage classification is also applicable to so-called "semi-shrinkage" methods in open pillar-supported stopes where broken ore is temporarily retained as a working platform but offers no wall support, as well as to various blasthole shrinkage methods that utilize broken ore temporarily retained in the stope for wall support, but which do not require miners to work from muck pile in the stope.

The method is generally applied to steeply dipping veins of strong ore between strong walls.

APPLICATION

Geometry

The geometry of a shrinkable vein is described in terms of dip, width, and regularity along dip. Overall strike and dip dimensions and irregularities along the strike generally impose no restrictions on the method.

Dip is ideally 1.2 to 1.5 rad (70 to 90°). As dip falls below 1.2 rad (70°), the shrinkage draw begins to strongly favor the hanging wall side, thus leaving a poor working platform for conventional overhand work. This is particularly true in relatively wide stopes. The support afforded to the hanging wall also diminishes with decreasing dip, reaching nil as the dip approaches the repose angle of broken ore. Dips below 0.78 to 0.87 rad (45 to 50°) are not generally shrinkable except by open stope "semishrinkage" methods.

Minimum mining width is fixed by working space requirements in the stope—generally about 1 m. Shrinkage in narrower veins requires that waste rock from one or both walls be broken with the ore and the attendant dilution accepted to achieve the minimum width. Narrow stopes are less suitable, encouraging hang-ups and bridging of broken ore, with the attendant problems of erratic draw and incomplete recovery of broken ore. Maximum practical width may be 3 m or less to more than 30 m, depending upon the competency of the ore and its ability to stand unsupported across the stope back. This is a vital safety consideration in conventional overhand stopes, but is much less of a factor in blasthole shrinkage methods. Very wide veins and massive orebodies have been mined by transverse vertical shrinkage panels separated by transverse vertical pillars, which are either abandoned or recovered later by other methods.

Regularity along the dip is a prerequisite of shrinkage as there must be no serious obstruction to the flow of broken ore downward through the stope to the sill level. Gentle rolls along the dip are acceptable if the local footwall dip everywhere exceeds 0.78 to 0.87 rad (45 to 50°). Off-dip hanging wall and/or footwall splits can generally be mined selectively from a conventional shrink stope as they are encountered without adversely affecting subsequent continuation of shrinkage mining updip on the main vein. Vertical offsets or major rolls along the dip that cannot be "smoothed over" generally require that a sublevel be established with new draw control development. Blasthole shrinkage methods are much less flexible (and thus less selective) in their ability to accommodate any of these irregularities.

Ground Conditions

The wall rock must be strong enough to stand with the minimal support afforded by the dynamic mass of broken ore in the stope. During active mining, local sloughing from the walls is restrained, but the broken ore affords little, if any, useful resistance to closure of the stope walls. Such squeezing, if present, may bind up the stope and cause the loss of much ore. Pillars left between and/or within stopes are effective in preventing closure but reduce overall recovery. Walls may be reinforced by bolting after each stope cut in conventional shrinkage but not in blasthole shrinkage.

Ore in place must be strong enough to stand with no natural support across the stope width, although temporary artificial support or reinforcement may be used locally in conventional stopes. Some spalling or sloughing is permissible in blasthole shrinkage because personnel are never present in the stope. Physical and/or mineralogical characteristics of the broken ore may impose restrictions on stope design and/or operational planning, and may even

preclude the use of shrinkage altogether. Examples include: ores that, when broken, are cohesive or tend to pack or cement together under the influence of ground water, wall pressure, and/or chemical reaction. Such conditions precipitate erratic draw during mining and often result in difficult and/or incomplete final draw; pyritic ores, which oxidize very rapidly in the stopes and may generate heat, imposing a fire hazard by spontaneous combustion; sulfide ores, which oxidize sufficiently in the stopes to adversely affect mill recovery by flotation; and ores (especially those containing uranium minerals), which exude radon gas and thereby impose ventilation constraints on stope design. In most cases these problems can be minimized by limiting the size of stopes, by minimizing the duration of mining activity in each stope, and by promptly drawing each stope empty following completion of mining.

Advantages and Disadvantages

The relative advantages and disadvantages must be considered in comparing shrinkage to alternative methods applicable to steeply dipping tabular orebodies. When considering shrinkage, these alternatives are usually reduced to some form of sublevel open stoping or cut-and-fill stoping.

Shrinkage is unique in that at least two-thirds of the total production from a given stope must lag in time behind the mining. This production holdback effect can, depending on size and number of stopes, represent a significant tie-up of capital, the cost of which must be charged against the method. This peculiarity may, however, be turned to good advantage by allowing significant "broken reserves" to accumulate in completed stopes. This affords the opportunity, by selective drawing of the stopes, to insure uniform production tonnage and grade, independent of any fluctuations in stoping activity, or to quickly increase or decrease production to take advantage of transportation, milling, or market conditions without a corresponding sudden change in stoping activity. In any case, the production holdback effect, which is most objectionable early in the life of a property, is offset to some degree because shrinkage generally requires a smaller initial capital outlay and less development work than other vein mining systems.

Compared to sublevel open stoping, conventional shrinkage is a higher-cost method, requires more competent ore for a given stope width, is more skilled-labor intensive, is less easily mechanized, and has a poorer safety record. However, it will accommodate significantly weaker wall rock with less dilution, requires a smaller capital outlay and less development work, is generally more selective, and is applicable to veins or masses having a greater degree of irregularity (albeit small). Crater and sublevel shrinkage, or variations thereof, are relatively new but show promise of achieving the sublevel advantages of low cost, easy mechanization, safety, and applicability to relatively weaker ores while retaining most of the shrinkage advantages. These gains are at the expense of whatever selectivity and ability to handle irregularities conventional shrinkage has.

Compared to cut-and-fill stoping, conventional shrinkage is less expensive, generally requires less development work, and does not require the often elaborate systems for contemporaneous filling. Cut-and-fill stoping can,

however, accommodate weaker wall rock and slightly weaker ore, has greater potential for mechanization, is more selective and flexible in its ability to accommodate irregularities in the orebody or walls, and generally gives better recovery with less dilution.

DETAILED DESCRIPTION

Development Layout

Haulage levels are driven at vertical intervals of 30 m (100 ft) or less to more than 200 m (650 ft). Generally flat-dipping, narrow, or irregular veins require close level spacing to minimize problems of erratic draw. Alternatively, such veins, particularly if strike length (and thus haulage distances) are great, may be developed with one or more sublevels between the main haulage levels, each with an independent draw system development and the orepasses needed to transfer ore to the main haulage levels. In any case, maximum practical stope height is fixed by draw control considerations and will seldom exceed 75 to 100 m (250 to 325 ft).

Draw systems provide the means of shrinking the muck pile as the stope advances and of recovering broken ore when stoping is completed. Ore is typically drawn from the stope sill level through chutes, slusher pockets, or drawpoints spaced at 5- to 15-m (16- to 49-ft) centers along the sill. Close spacing requires more development work but minimizes the tonnage of ore stranded on the sill after the final draw and allows a more uniform muck surface to be maintained in the stope. Wide spacing is acceptable if the sill (and thus ore stranded on the sill) will be recovered as part of lower-level stoping, or if the sill can be safely cleaned after the final draw, for example, by the use of remote-controlled load-haul-dumps (LHDs). An uneven surface of broken ore in the stope is typical even with relatively close draw spacing. This is acceptable in blasthole shrinkage methods, but conventional overhand stopes may require some amount of hand mucking, slushing, or temporary drill staging between stope cuts. This work in the stope is to be traded off against development costs at the sill level in choosing a draw spacing. Numerous draw system layouts and variations are possible. The following are representative examples.

Example A: A haulage drift is driven on the vein. One or two stope cuts are taken out of the drift back and mucked clean to track level. Timber or lagged steel and chutes (either timber or steel) are installed at the design spacing and stoping begins above the timber. No ore pillars are left and no waste development is required. However, material, installation, and maintenance costs are high; haulage is restricted while chutes are being pulled; chute problems prevail (hang-ups, chute blasting, poor safety record, etc.); and the system is relatively skilled-labor intensive. Application of this method is to narrow veins (to drift-width or less) in small rail-developed mines.

Example B: A haulage drift is driven on the vein. Short raises are driven on the dip at the design spacing and flared to intersect along the strike, thus forming the stope sill. A chute is installed at the collar of each raise. The discussion under example A applies in this case, except that installation and

maintenance costs are reduced by leaving some ore in the back pillars. Pillar recovery destroys the haulage.

Example C: A haulage drift is driven on the vein favoring the hanging wall. Short footwall crosscuts are driven at 0.78 to 15 rad (45 to 90°) to the vein at the design spacing. A raise is driven from the end of each crosscut inclining back over the haulage and flaring along strike to connect with adjoining raises to form the stope sill. Draw is accomplished with mucking machines or LHDs. Chute installation, operation, and maintenance problems are traded for corresponding problems with muckers, and development work is increased, typically including some waste development depending on vein width. Haulage restriction during draw remains a problem and pillar recovery destroys the haulage.

Example D: An exploratory drift, which will become the stope sill, is driven on the vein and, lagging behind it, a parallel haulage drift is driven in the waste, preferably in the footwall, leaving a 3- to 10-m (10- to 33-ft) waste pillar between drifts. Connecting crosscuts are driven at 0.78 to 1.5 rad (45 to 90°) to the drifts at the design spacing to serve as mucker or LHD drawpoints. No pillars are left in the ore, and the haulage remains intact after stoping is complete; however, considerable waste development is required and haulage restriction remains a problem. If the vein is at least three drift widths across, both drifts may be in ore, one serving as the stope sill, the other as the haulage. Waste development is eliminated but recovery of the haulage drift pillars destroys the level.

Example E: A haulage drift and a smaller slusher or LHD scram drift are driven on the vein, the scram updip from the haulage leaving a 3- to 10-m (10- to 33-ft) pillar of ore above the haulage. Drawpoints are developed and the stope sill is established from the scram as described in example C. The scram-level slusher or LHD is used to move stope muck from the drawpoints to a central grizzly over a raise pocket. The raise pocket may be fitted with a loading chute over the main haulage or may discharge to a mucker or LHD drawpoint in the footwall of the haulage drift. Stope activity including draw can occur independent of activity in the haulage, but development requirements are extensive and significant tonnages are tied up in pillars that can be recovered only by destroying the haulage. Parallel double scrams with crosscut drawpoints, as described in example D, will reduce pillar losses but will increase waste development if the vein is less than three times the width of a scram drift.

Example F: A haulage drift is driven parallel to the vein in the footwall. Drawpoint development follows example E. No ore pillars are left and the haulage is unobstructed by stoping activities and remains intact after stoping is complete; however, the development burden is relatively heavy.

Stope Access and Ventilation

Stope access and ventilation must be provided for conventional overhand shrinkage, although not for blasthole shrinkage. A variety of vertical development configurations and combinations thereof are possible. The following are representative examples.

Example G: After each stope cut, timber stulls are placed at 1- to 2-m (3- to 6½-ft) vertical intervals from the footwall to hanging wall 2 to 3 m (6½ to 10 ft) short of one end of the stope. The stulls are laced or lagged vertically to form an open compartment bounded on three sides by rock. The "raise" is advanced updip with the stope. If the vein is wider than a convenient stull length, the raise can be formed by carrying a 1- to 2-m (3- to 6½-ft) wide slot in ore at the end of the stope or, if dilution is acceptable, in waste in the footwall. No prior vertical development is required, and the stope may be advanced "blind." If such a raise is to be the primary access, equipment such as ladders, landings, pipe, vent bag, materials slide, etc. must be installed piecemeal as the stope is advanced. Flyrock must be excluded from the raise by button boards or a hatch door secured by the miners as they leave the stope.

Example H: A four-sided timber crib located midway between draw holes is carried updip with the stope and is pinned to the footwall to stabilize it against muck movements during draw. Comments under example G apply here as well.

Example I: A raise is driven or bored level-to-level (or spanning several levels) on the vein, and "doghole" drifts are driven in both directions from the raise on the strike at regular vertical intervals [typically 5 to 10 m (16½ to 33 ft)]. The raise is fully equipped with a manway, utilities, material conveyance, etc. as part of the normal development. Each raise serves two stopes. Entry to the stopes is through the dogholes in ascending order as the stope is advanced, each successive doghole being tapped by a short raise driven from the stope back, just prior to the preceding doghole being choked off with stope muck. Two vertical rows of ore pillars remain after the stopes are complete. These can generally be drilled from the manway and recovered in a single blast after the manway has been stripped.

Example J: A raise is driven or bored level-to-level (or spanning several levels) a short distance into the footwall, and "doghole" crosscuts are driven from the raise to the vein at regular vertical intervals. The discussion under example I applies here except that no ore pillars remain and the manway remains intact after stoping—this to be balanced against the waste development required.

Example K: A raise is driven or bored level-to-level (or spanning several levels) in the ore midway along the strike between the manway raises, as in examples I or J, and is left bald with the possible exception of a chain escape ladder, air and water services, etc., which might be suspended from the top level. The raise remains open above the active stoping horizon for ventilation purposes but is destroyed as the stope advances. Services in the raise are shortened and recovered as the stope advances.

Example L: A raise is driven or bored level-to-level (or spanning several levels) in the ore or in the footwall midway along the strike between manway raises, as in examples I and J. If in ore, stoping is stopped just short of the raise and ventilation breakthroughs are made from the stope at vertical intervals as needed. If in waste, stoping is continuous along the strike and the ventilation breakthroughs will be short crosscuts producing some dilution. The

raise remains open during and after stoping and is utilized for exhaust from the stope and from the drawpoint level, as in examples D through F. Raise pillars, if in ore, are lost.

Stoping

Overhand Shrinkage. Overhand shrinkage stopes are usually advanced by maintaining a nominally flat back. Slightly inclined, wavy, or stepped-back configurations may be advantageous, depending on the location and spacing of draw holes, vein width, stope access, and the ventilation scheme.

Drilling up holes or "uppers" with stopers is generally preferable to breasting, particularly in narrow and steeply dipping veins, as the entire stope cut can usually be drilled out in one continuous pass, thus minimizing time lost in frequent drill setup and teardown. This is accomplished by beginning each drill-out at the opposite end of the stope from the service raise (or in the middle of the stope if services are available at either end) and retreating toward the services. Explosives loading, shrinkage draw, blasting, and barring down are then accomplished in that order over retreating portions of the strike length on each shift without requiring teardown of drills or service lines, and generally without even interrupting the drilling except during blasting. Breasting with feedleg drills may give equal or better productivity if the stope is wide enough to allow fairly sizable rounds; however, fragmentation may be poorer because the first row of holes breaks to an unrestricted free face, generating coarse muck from the vicinity of the toes of these holes. Breasting is generally considered safer in that drillers are not working directly under ground subject to loosening by the drills. Possibilities for mechanized drilling are limited by the uneven muck surface typical in shrink stopes, which precludes utilization of most available mechanized drill carriers. Air-propelled tracked carriers may be suitable, depending on stope width and dip.

Ground support for stope walls, beyond that temporarily afforded by the broken ore, is possible by local or pattern bolting and/or by pillars left in the stope. Preferably, these pillars are small, teardrop shaped, and situated along the strike in vertical rows midway between draw holes to minimize any adverse effect on draw and recovery. Temporary support of the ore across the stope back is possible by installation of bolts and/or wire mesh and/or shotcrete after each cut; or by prepinning of the back by full-length grouting of rebar or cable. Such materials must break up sufficiently in the muck pile so as not to hinder draw, and they may create serious problems in milling.

Shrinkage draw is a required element of each stope cut, and should be accomplished after blastholes in the subsequent cut are drilled and loaded but before those holes are shot. This sequence is preferable because the muck pile following a blast will generally be a better working surface for subsequent drilling than the muck pile following shrinkage draw, particularly if the drill pattern and blast timing are designed to move the broken ore toward the subsidence cones that will exist following draw. This is especially true in relatively flat-dipping stopes where draw favors the hanging wall, and in stopes having a large draw spacing. It should be noted that the subsidence zones over the drawpoints may, especially in narrow stopes, be defined as

walls, which are considerably steeper than the angle of repose of broken ore owing to such factors as compaction, cementing, wall squeezing, and two-dimensional confinement of the broken ore. A very real hazard exists with bridging or hang-up of broken ore in the stope. If such a condition develops undetected during shrinkage draw, miners may be "swallowed" in the subsidence when the muck finally does move as, for example, under the influence of drilling water and vibration. It is important, therefore, that the shrinkage draw be observed from the stope, and advisable that stope miners be the only personnel authorized to draw muck from this stope. Occasional bridging may be broken with water, but persistent erratic draw generally requires that the stope be abandoned and a new draw system development be installed if stoping is to be continued updip.

Blasthole Shrinkage. Blasthole shrinkage methods do not utilize conventional overhand techniques to drill and break the ore. Miners, therefore, do not work in the stopes, and the broken ore is retained only for temporary wall support. Historically, these methods have included the Latouche or Beatsen method, in which patterns of nearly horizontal longholes are drilled and loaded in an updip retreating sequence from raises driven in ore; and the Alaska Juneau method, in which ore is broken by coyote blasting from small drifts driven for that purpose perhaps 10 m (33 ft) above the back of the stope. Experience favors either crater shrinkage (sometimes called crater retreat), sublevel shrinkage, or variations of these methods.

Crater shrinkage stopes are set up with a draw system and a stope sill development similar to overhand stopes but with no vertical development above the stope sill. Downward drilling of carefully aligned parallel holes is done with in-the-hole or rotary drills from a stope-width drift on the level above. The level interval is limited to perhaps 40 to 80 m (130 to 260 ft) by drilling accuracy. Holes are drilled to break through in the sill level. When drilling is complete along the length of a stope block, plugs are installed in the bottoms of the holes and each is loaded from above with a short column of stemming, a "spherical" charge of explosive ($L/D \leq 6$), and another short column of stemming in that sequence. The charges are detonated from the level above in a single blast to complete the first stope cut. Fragmentation is by cratering, hence the name. The plugging, loading, and blasting sequence is repeated numerous times in the same holes with all operations including hole plugging conducted from the drilling level until the stope is complete. The stope is drawn to maintain the muck pile several meters (feet) below the stope back, the shrinkage operation being relatively independent of loading and blasting activity. The elevations of the stope back and muck pile are monitored by measurements taken through the blastholes after each cut. Vein regularity and provision for accurately aligning and drilling the holes are obvious and essential prerequisites of the method.

Blasthole diameter and drill pattern are important design parameters. In narrow veins with good wall parting, a single row of holes down the center of the vein may be sufficient. In wider veins, multiple rows of holes may be necessary or desirable, particularly if wall parting is poor. In any case, a cylindrical charge ($L/D = 6$) properly located with respect to the stope face may conservatively be expected to generate a crater having a maximum diameter of about

20 times the blasthole diameter. Twenty hole diameters is thus a good starting figure for hole spacing and for maximum vein width for a single row of holes. Conservative design would suggest drilling oversize holes, starting with a charge *L/D* of 6, and adjusting that ratio downward as far as practical to achieve a balance between fragmentation and explosives economy.

An alternative method involving virtually identical level development and very similar blasthole drilling and top loading schemes, but not dependent on cratering for fragmentation, is achieved by driving a central raise or wall-to-wall slot in ore, loading the holes with columnar charges of perhaps 3 to 10 m (10 to 33 ft) in length rather than "spherical" charges, and timing them to break by slabbing to the central opening. The necessity for prior vertical development is avoided if the central "cut" or break opening is advanced at least one cut ahead of the stope by crater retreat or top loaded "drop raise" techniques.

Sublevel shrinkage combines shrinkage principles with the development and production schemes used in sublevel caving, and only a general review will be given here. Each stoping block is developed by a series of sublevels spaced at vertical intervals of 10 to 20 m (33 to 66 ft). Stoping progresses from top to bottom. On each sublevel, a break raise or slot is established between that sublevel and the one above at the strike limit (in the case of longitudinal stoping), or at the footwall (in the case of transverse stoping). Production drilling is accomplished with mechanized carriers drilling 40- to 80-mm (1½- to 3⅛-in.) up holes beginning at the break slot and retreating down the sublevel, with the plane of each row or fan of holes striking normal to the sublevel and inclining forward (i.e., toward the slot) at 1.04 to 1.3 rad (60 to 80°). One or two rows are blasted at a time, the first row breaking to the slot, and subsequent rows breaking against stope muck from the previous shot. After each shot, a calculated swell volume of broken ore is removed from the single "drawpoint" at the retreating face of the sublevel to provide a measure of relief for the next shot. When stoping is complete on each sublevel in the block, the broken ore remaining in the stope is recovered either by establishing parallel-drift drawpoint development at the bottom sublevel, or by remote-controlled LHDs operating on the bottom sublevel.

FINAL DRAW

The final draw of the ±60% of broken ore remaining in the stope is usually begun immediately following completion of mining in the stope (see "Advantages and Disadvantages" for exceptions). It is generally prudent to hedge against possible draw problems by constructing a catwalk or rock ledge on the footwall 2 m (6½ ft) below the stope back from which stranded ore can be bombed and/or washed to the stope sill. Water at a pressure of 5 to 10 kg/cm^2 (71 to 142 psi) delivered through a small fire-fighting monitor having a 20- to 30-mm (¾- to 1⅛ in.) nozzle is very effective in prompting ore movement and in securing maximum recovery of broken ore. The draw may be accomplished by working all drawpoints together, or by starting at one end of the stope and pulling the drawpoints dry in sequence down the length of the stope. The former method is more likely to lead to bridging problems in

narrow stopes, but is preferable if dilution from hanging wall slough is a problem, and essential if waste fill is to be introduced concurrently with the draw. The latter method, particularly if aided by water, leads to rifling of the ore toward the active drawpoint, a factor that can be used to minimize slushing and/or tramming distances because a high percentage of the ore is recovered from the first drawpoint to be pulled. In either case, as noted in the discussion of shrinkage draw, the walls of the cone (i.e., a two-dimensional wedge) of depression that develops over a drawpoint being pulled will generally, in the absence of much water and/or bombing, be much steeper than the normal angle of repose of broken ore. Empty stopes may be filled with development waste or mill tailings or, barring safety and subsidence constraints, may be left to cave. Coarse waste fill introduced concurrently with the final draw has been used with some success as a ground control measure at the expense of some dilution and loss of recovery.

PRODUCTIVITY AND COSTS

Productivity and costs are variable in shrinkage mining, perhaps more so than in any other major underground mining method. Conventional overhand shrinkage applied to hard ore in narrow irregular veins may result in productivities of 10 t (11 st) or fewer per stope miner-shift with direct costs in the $10 to $15/t ($9 to $13.50/st) range. Under favorable conditions where wide regular veins or masses are minable by blasthole shrinkage methods, productivity may approach 200 t (220 st) per stope miner-shift with direct costs in the range of $1.00 to $1.50/t ($0.90 to $1.35/st). In conventional narrow-vein stopes, where opportunities for mechanization are limited, productivity and costs are closely tied to the skill and industry of miners in the stopes, and are very sensitive to stope dimensions and ore hardness. Under such conditions, piecework labor incentives (i.e., "gypo" contracts) are common and, with careful administration, are generally effective in maximizing labor output. Where geometry and ground conditions permit relatively larger and more mechanized stopes, and particularly where blasthole shrinkage methods are applicable, labor specialization is the rule; productivity and costs are more closely tied to machine performance and availability. Under such conditions, careful selection of equipment, comprehensive technically oriented training and supervision of operators and maintenance personnel, and careful detail planning and scheduling are of paramount importance.

In view of the constraints on the applicability of shrinkage, the wide variety of possible stope configurations, and the noted variability of costs, it is prudent for each potential application to estimate costs by analyzing individually the various unit operations involved in the plan. Where applicability is questionable or limited or estimated costs marginal, full-scale stope testing should be implemented prior to making all-out development or equipment commitments.

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CHAPTER 19

Shrinkage Stoping at the Crean Hill Mine

K.J. Henderson

INTRODUCTION

Shrinkage mining at Inco Ltd.'s Crean Hill mine can best be defined as a horizontal breasting method utilizing the broken ore for the mining floor and to provide wall support, with draw from a series of boxholes or drawpoints at the base of the stoping block.

In choosing to shrink a given block of ore, the initial investigation centers on the ore configuration and grade, the expected ground conditions in the stoping area, the expected profit compared to alternative mining methods, the future production requirements, development lead time, the availability and type of equipment in the area, and the availability of experienced miners. These factors are weighed against other mining methods, and shrinkage mining is chosen when the advantages outweigh the disadvantages. These will be discussed in more detail later.

At Inco Ltd.'s Crean Hill mine, shrinkage mining makes up 17% of total mine production and is decreasing due to the advent of advanced blasthole techniques, notably in-the-hole drilling. Shrinkage is an adaptable mining method that is currently employed in narrow irregular ore zones, as undercuts for in-the-hole blasthole stopes, in orebodies that do not extend from level to level, and in areas where quick production is required with a minimum of capital outlay or minimum development. Two major types are classified on the basis of type of draw system employed. These are the utilization of load-haul-dump (LHD) equipment drawing from drawpoints, and the slusher and train method drawing from boxholes into a slusher trench and loading into a train. A third method, being phased out, is drawing from chutes directly into a train. In all three methods the actual stoping operation remains the same.

In the initial planning of a mining block, the orebody is first located using an exploration diamond drilling program. This is essentially done at a 130-m (400-ft) spacing and it gives an indication of the amount of ore, grade of ore,

the configuration of the orebody, as well as other geological features, such as ground conditions, that will affect future mining. This drilling is done from an exploration drift 165 m (500 ft) long in the hanging wall and covers a vertical extent of 330 to 670 m (1000 to 2000 ft). However, these data are not accurate enough for detailed planning. When the decision has been made to mine, a preliminary development drill drift is driven in such a position that it can serve the dual purposes of allowing the orebody to be drilled off in detail, or of later serving as a haulage or access drift. Diamond drilling from this drift is usually on a 15-m (50-ft) grid spacing and gives adequate information for detailed planning.

From the drill data, a decision is made on the mining type. For a shrinkage operation, the stoping block is ideally 49 m (150 ft) in length and a maximum of 13 m (40 ft) in width. Lengths of 65 m (200 ft) are considered a practical maximum because of the length of time the back of the stope remains open during the mining of a cut. A minimum length of 13 m (40 ft) is used because the amount of time spent in opening a new cut increases in comparison with the total stoping time. The maximum width of 13 m (40 ft) is used because experience with the ground indicates that generally stope backs tend to be unstable over greater widths. However, widths of 33 m (100 ft) have been mined. A minimum width of 5 m (15 ft) is used to allow an effective draw, but in unusual cases, a width of 3 m (10 ft) has been mined successfully. The dip of the ore should not be less than 1.04 rad (60°). Angles flatter than this will not pull properly, resulting in inefficient mining because of ore hang-ups on the footwall side of the stope and the need for large stagings on the hanging wall.

In considering shrinkage, mining ground conditions play an important role. Strong shearing, cross faulting, or dikes are avoided. These usually cause dangerous mining conditions, and an alternative safer mining method that will give adequate support to these adverse ground conditions is employed.

Production requirements are taken into consideration when planning a shrinkage mining program. In itself, shrinkage mining is capable of producing ore in a relatively short time with little capital development. If quick production is required, this method can produce ore rapidly because all that is required is that the draw system be completed and the miners with jacklegs and stopers produce a limited amount of ore. The obvious disadvantage is that not all the ore broken can be pulled from the stope immediately. However, a mining horizon with several shrinkage stopes in different stages of production can adequately maintain scheduled tonnage requirements, the cycle being stopes under development, stopes actively mining, and stopes completed and pulling.

When the decision has been made to use shrinkage mining in a given block, the haulage and draw system is driven. The proximity of the orepass system dictates whether the system is to be an LHD operation or a slusher trench operation. If the orepass system is in close proximity to the stoping block, LHD equipment is employed because of its ability to operate efficiently over short haulage distances. A long tram to the orepass usually dictates a slusher trench operation with haulage by train. Internal orepass systems are used effectively with LHD equipment with a load-out and retramming taking place on a lower horizon. In addition, stoping blocks in older areas of the mine are developed

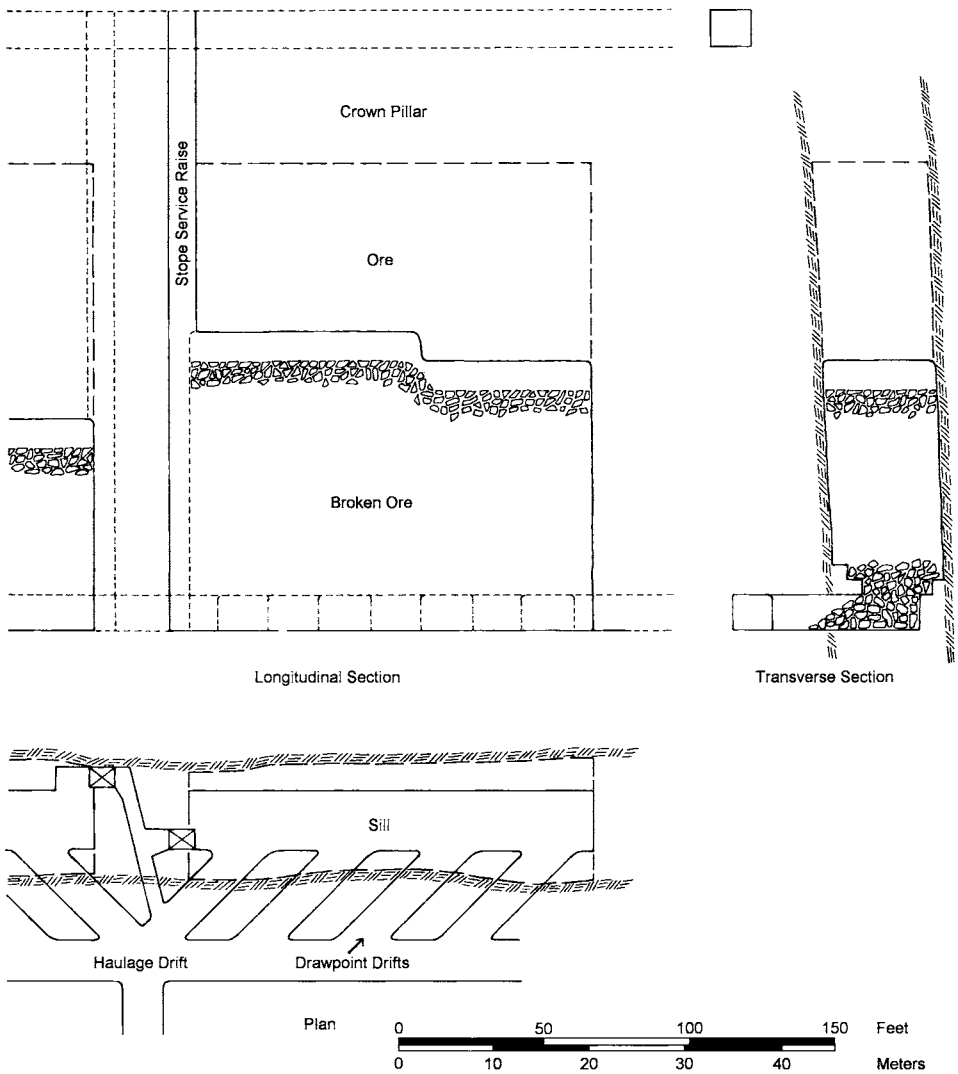


FIGURE 19.1 The load-haul-dump (LHD) operation

track because of the capital cost involved in converting an existing level to trackless mining.

DRIFT DEVELOPMENT USING LHD EQUIPMENT

The haulage drift for a draw system using LHD equipment is driven parallel to the orebody in the footwall at a sufficient distance to allow an 11.5-m (35-ft) drawpoint to be driven into the ore (Fig. 19.1). The actual position of the drift depends entirely upon the ore configuration, and the drift size depends on the equipment to be used. Drift sizes of 4.6 × 4.1 m (14 × 12½ ft) for an ST-4 and 3.8 × 3.6 m (11½ × 11 ft) for an ST-2 meet company and government clearance standards and allow sufficient space to accommodate a ventilation pipe.

The drawpoint sizes are smaller, 3.9 × 2.9 m (12 × 9 ft) for an ST-4 and 3.8 × 2.9 m (11½ × 9 ft) for an ST-2. This is because they do not have to carry a ventilation pipe. A crosscut is driven off the haulage drift as soon as possible into the rib pillar to serve as an access for the stope raise. This allows the raise crew to start driving the stope raise in conjunction with drift development.

The haulage drift is then continued, driving the first drawpoint with the drift. This drawpoint is driven on the edge of the pillar and completely through the ore because it connects the chute side of the stope raise to facilitate the raise mucking and also serves as a starting point for the stope sill. The remaining drawpoints are driven as they are reached by the haulage drift. These drawpoints are at 11.5-m (35-ft) intervals (center to center) and are driven the layout length of 11.5 m (35 ft)

Upon completion of the haulage drift and drawpoints, the stope sill is started from the No. 1 drawpoint. This silling operation is at a drift elevation height of 3 m (9 ft) and is done by the same crew that drove the drift. The initial sill width is limited to 6.5 m (20 ft) or to the ore contacts if the ore width is less than 6.5 m (20 ft). The sill breaks into each boxhole as it is reached. When the sill is complete, stoping operations begin.

DEVELOPMENT FOR SLUSHER TRENCH OPERATION

The haulage drift is driven at right angles to the stope through the orebody long enough to provide adequate tail room for the train to load the stope muck. A gangway is excavated at the slusher trench location and the stope raise is then driven from the gangway through to the next level. This provides ventilation for the remaining development and stoping operations.

The slusher trench is driven 2.3 × 2.3 m (7 × 7 ft) down the center of the orebody at an elevation of 2.3 m (7 ft) above the drift base of rail. The boxhole access drifts are driven at right angles to the slusher trench at 9.8-m (30-ft) intervals center to center. These are 2.3 × 2.3 × 3.3 m (7 × 7 × 10 ft). The muck from this development is slushed into cars through the gangway. Upon completion of the slusher trench, the boxholes are belled. The first bell is in the form of a semicircle, with the side adjacent to the slusher trench not belled (Fig. 19.2). This bell reaches a height of 2.3 m (7 ft) above the back of the slusher trench. The second bell is then taken, widening out to a full circle, which extends the boxhole over the top of the slusher trench. When all boxholes are belled, the stope sill is started from the manway raise with the ore breaking down into the boxholes. The sill advances from the manway, opening the stope to full size. Upon completion of the sill, the development crew is transferred to a new development block and a stoping crew takes over.

RAISE DEVELOPMENT

Stope raises for shrinkage mining consist of two types: a center raise with access drifts and a stope raise on the pillar wall. Both raises are located in the rib pillar, or at the end of the stope if the ore is to be mined with a single

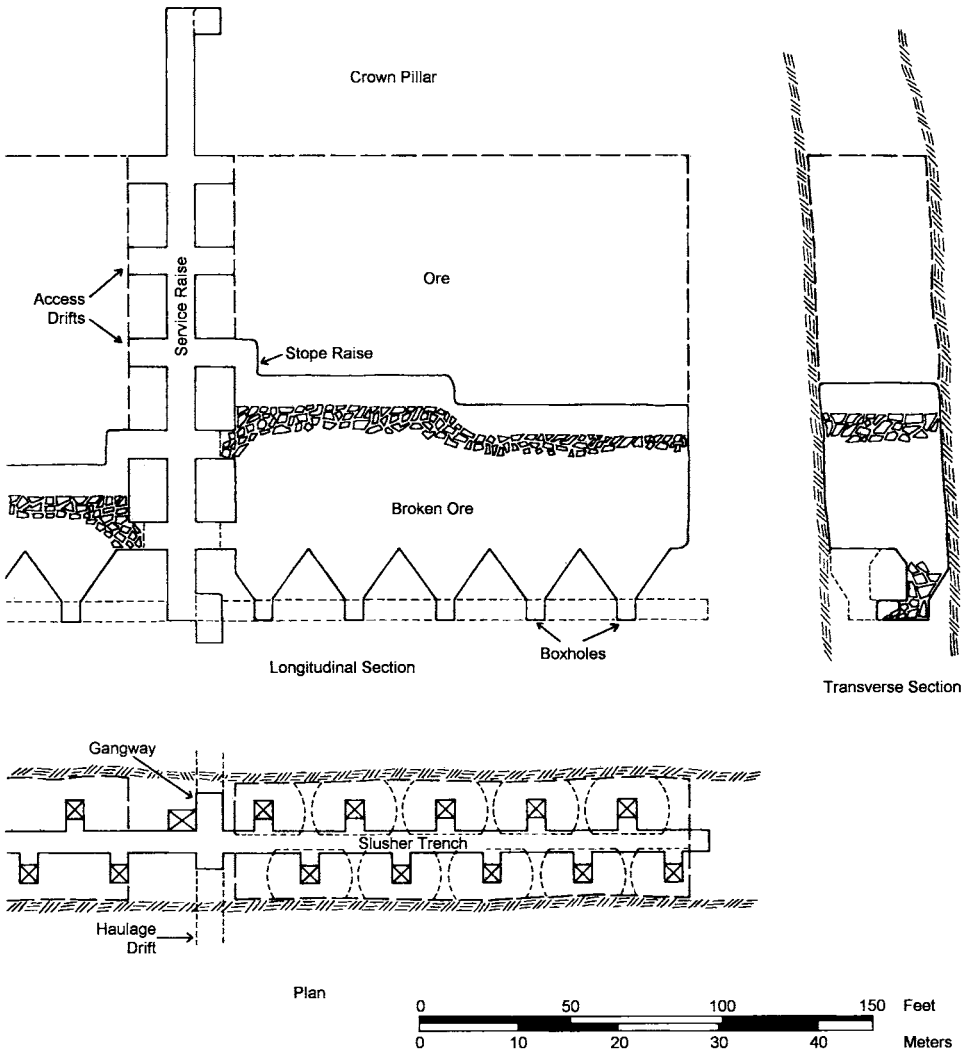


FIGURE 19.2 The slusher trench operation

stope. Raises are driven cribbed 2.3 × 3 m (7 × 9 ft), lined with a 51-mm (2-in.) plank, and use a two-person raise crew and a nipper who does miscellaneous work on the level when not involved with the raising.

The single center raise (Fig. 19.1) is the more traditional method, with the raise located in the center of the pillar and short access drifts 2.3 × 3 m (7 × 9 ft) driven through the pillar to the future stope position at 9.8-m (30-ft) vertical intervals. This has the obvious advantage of driving only one raise to service two stopes.

The stope raise on the pillar wall (Fig. 19.2) is the type more commonly used at the mine. This raise is driven on the pillar wall 2.3 m (7 ft) inside the future stope with the chute side adjacent to the stope. This has several advantages: it

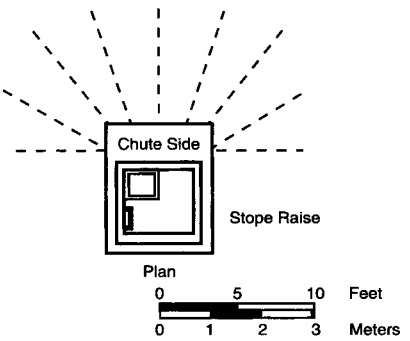


FIGURE 19.3 The drilling pattern used in opening a new cut

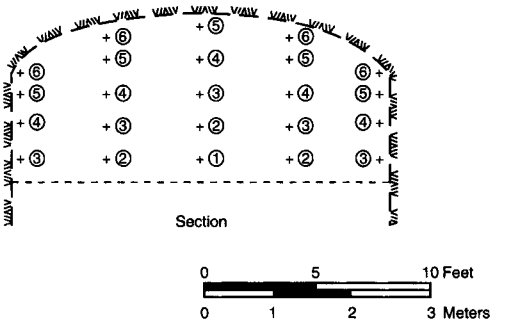


FIGURE 19.4 The mining face drilling pattern used during the blasting sequence. + indicates drill hole, ○ indicates blasting sequence.

eliminates the need to drive the 9.8-m (30-ft) internal stope raises during the stoping operation, which in effect amounts to one crib raise and two bull horn and raw raises to mine two stopes, and therefore the stope crews need not be experienced raise miners; it provides a safer work area because personnel do not have to work under the open stope raise; and it provides for easier nipping of materials into the stope because all supplies can be nipped directly to the stope mining level and do not have to be handled up an internal raise. It must, however, be bulkheaded on the chute side at 9.8-m (30-ft) intervals to provide head protection for a safe entrance to the stope. This entrance is directly under the chute side of the raise.

STOPE MINING

The mining of a shrinkage stope is essentially the same for both draw systems and for both raise types. Upon completion of the stope sill, a stoping crew takes over the stope. This frees the development crews for work on new stoping blocks. Stoping crews consist of two people, a stope leader and a driller, working two shifts per day. This crew drills off blasts at one round per shift, and in addition completes the required roof bolting and screening.

The stoping operation starts with the opening of a new cut. With a stope raise on the pillar wall, an air-powered chain saw is used to cut through the raise cribbing for a distance of 3.3 m (10 ft) above the back of the sill cut. It is important that a head cover be placed directly above this opening in the raise. If a center raise is used, a stope raise must be driven from the sill cut to connect with the access drift 9.8 m (30 ft) above, with a bull horn and staging installed. The new cut is then drilled off using a fan pattern (Fig. 19.3), 2.6 m (8 ft) in depth. This procedure is continued driving toward the stope walls, establishing the mining breast, bolting, and scaling as required. The stope is then advanced using 3.9-m (12-ft) steel. The drilling pattern (Fig. 19.4) is at 1.3-m (4-ft) intervals horizontally and 490 mm (18 in.) vertically, with an arch to back width of 0.33 to 3.3 m (1 to 10 ft). With the exception of very wide stopes, a breast is drilled and blasted in one shift (8 hr).

The equipment used in the stope consists of three jackleg air drills (one as a spare) and two stopers. Air and water are supplied from the raise through a 25.4-mm (1-in.) bull hose for water and a 51-mm (2-in.) bull hose for air. It was found to be impractical to carry steel air and waterlines in the stope. During normal blasting, air and waterlines are coiled 16 m (50 ft) from the face.

Before approaching the face at the start of the shift, the back is scaled and bolted. The bolting is done on a 1.3 × 1.3-m (4 × 4-ft) pattern and a screen walkway is provided down the center of the stope and carried to the face. All personnel entering the stope are aware that the only safe travelway to the face is under the screen, thus reducing the back area that must be maintained for safety purposes and reducing the labor to scale from footwall to hanging wall. The screen is the lightweight (No. 9 welded wire construction) variety, which breaks up easily in the muck when succeeding cuts are taken. In addition, the hanging wall is also bolted on a 1.3 × 1.3-m (4 × 4-ft) pattern to prevent unnecessary slough when pulling the stope.

Blasting of the face is done using ammonium nitrate–fuel oil (ANFO) explosive with pneumatic loaders and anodets. The anodets are ignited with tape fuse and B line. This is used rather than electric firing because it was found to be very difficult to maintain electric lines in the manways. The delay sequence is shown in Fig. 19.4.

Mining control is established by the engineering department, which provides lines and grades in each new cut. This is necessary because there is a tendency for the back of the stope to climb as it advances. A geologist visits the stope every second or third breast to establish ore contacts and keep the amount of wall-rock dilution to a minimum. If the ore widens, slash is drilled; if the ore narrows, the perimeter is reduced. This is a further indication of the flexibility of the mining method for irregular ore.

The procedure is repeated on successive cuts until the stope is completed to the crown pillar elevation.

PULLING

The pulling of broken ore in the shrink stope is controlled by the stope leader and the mine foreman. During the mining of the first four benches, the stope crew is out of the stope when pulling is in progress. During the remainder of the mining, the crew remains in the stope. However, close contact is maintained by the stoping crew and the motor crew. The pulling is monitored by a stope pulling sheet on which the stope crew indicates which boxholes or drawpoints are to be pulled and how many cars or buckets are to be removed. This is signed by the stope leader and mine foreman and given to the pulling crew.

During the mining operation, pulling is done ahead of the mining face and enough muck is pulled to free the face for the next blast. This usually amounts to 30% of the ore blasted. If a hang-up occurs during the pulling, the stope crew is removed from the stope and pulling continues until the hang-up

comes down. Mining is then resumed. Upon completion of the stope, pulling is done at a scheduled rate to meet the mine's production requirements.

If hanging wall slough is expected, the stope will be pulled at an equal rate from each boxhole. This keeps dilution to a minimum because the rock will remain on top of the ore and can be left in the stope when pulling is complete.

PRODUCTIVITY

Shrinkage mining at Crean Hill has produced 1723 t/d (1900 stpd), amounting to 50% of the mine's total production. This was accomplished by strict cycling of stoping blocks. Stopes were developed at a rate fast enough to replace ore being pulled. Efficiencies for development crews are 0.36 m (1.1 ft) per miner-shift average for all developing blocks including drifting, raising, and boxhole widening. Stoping crews drilling and primary blasting are producing 42.1 t (46.5 st) per miner-shift. Individual stopes have produced as much as 96.2 t (105 st) per miner-shift over extended periods of time. Pulling operations in slusher trenches are 68 t (75 st) per miner-shift. This includes slushing, tramming, and secondary blasting. In LHD operations, the pulling rate varies with the equipment size and the haulage distance, but in general, an ST-2 produces 181.4 t (200 st) per miner-shift and an ST-4 produces 290.2 t (320 st) per miner-shift, hauling directly to an orepass a maximum distance of 165 m (500 ft).

In general, where shrinkage mining has been employed at the mine, it has proven to be very effective in meeting production requirements in a safe and efficient manner.

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CHAPTER 20

Shrinkage Stoping at the Idarado Mine

Compiled by William A. Hustrulid*

INTRODUCTION

The Idarado mine lies high in the San Juan Mountains on the divide between the Uncompahgre and San Miguel Rivers and consists of a consolidation of a number of old and prominent mining properties through which course some famous and very productive veins. Among the better known are the Smuggler, Tomboy, Montana-Argentine, Black Bear, Liberty Bell, Virginius, Flat, Barstow, and Japan, as well as many others. Most of these veins have been extensively mined over the past century and the Montana-Argentine and Black Bear veins have been the backbone of the Idarado mine; therefore, the description of the veins is limited to them.

Access to the mine is through either the Treasury tunnel, whose portal is below Red Mountain Pass on US Highway 550 at an elevation of 3244 m (10,643 ft), or the Mill Level tunnel entrance 3.2 km (2 miles) east of Telluride, Colo., at an elevation of 2761 m (9060 ft). The Treasury tunnel intersects the Black Bear vein 2643 m (8670 ft) from the portal, and the Mill Level tunnel intersects the Argentine vein 2179 m (7150 ft) from the portal. Mining is by shrinkage stoping from slusher sublevels. The size of the stope blocks varies somewhat, but the standard size is 67 to 76 m (220 to 250 ft) long and 61 to 76 m (200 to 250 ft) high. The mine ranks either first or second in Colorado in yearly production of gold, silver, copper, lead, and zinc.

It is 9.6 km (6 miles) from the Red Mountain plant to the Pandora plant, via interconnecting drifts and raises. There are engineering offices at both plants as a matter of convenience. The Red Mountain plant includes the company general offices, warehouse, carpenter and machine shops, and mine change room. The Pandora plant consists of the mill and assay office, machine shops,

*This chapter was compiled by William Hustrulid from material supplied by Peter N. Loncar, Mine Manager, Foote Mineral Co.

and mine change room. The flotation mill has a capacity of 1632 t/d (1800 stpd), making a bullion product and separate concentrates of lead, copper, and zinc.

HISTORY OF THE MINE

The Montana–Argentine vein was first extensively worked by the Tomboy Gold Mines Co., Ltd., a British concern. This company mined the stoped areas above the Ophir level between 1910 and the late 1920s, and most of the stoped areas above the 2100 level between 1900 and the late 1920s. Gold was the principal ore metal mined.

The area between the Revenue and Ophir levels was mined chiefly by the Revenue Mines Co. between 1900 and 1910. The ore was worked from the Revenue tunnel, which portals in Canyon Creek. Gold and silver were the chief metals recovered.

The stopes between the 1700 and Revenue levels, as well as some higher stopes, were mined by Telluride Mines Inc. during the 1940s. The Mill Level tunnel was driven by that company from 1945 to 1948. Lead and zinc then became economically more important than the precious metals. In 1953, Idarado purchased Telluride Mines, which merged with the parent company in 1956.

The Black Bear vein was first extensively worked by the Black Bear Mining Co. in the 1900s and by the Colorado Superior Mining Co. from about 1914 until snowslides at the mine camp [altitude 3750 m (12,300 ft)] terminated the company's operations in 1924. Leasers operated at intervals until 1934. The Treasury tunnel, formerly the Hammond tunnel, had been started before 1900 and reached the 1646-m (5400-ft) mark early in the 1900s, at which time activity lagged until the late 1930s. In the early 1940s, Idarado extended the Treasury tunnel from its heading at 1646 m (5400 ft) to the Black Bear vein and established a raise connection with the 600 level, the lowest level in the old mine.

Since completion of initial work in the mid-1940s, systematic development of the mine, both in the driving of new headings and the utilization of older openings, has resulted in an extensive network of workings.

GEOLOGY

Introduction

The oldest rocks of the area are the Precambrian metamorphic rocks, the Uncompahgre Formation, which are massive quartzites, some phyllites and slates, and dolomitic or limestone beds. West-dipping Paleozoic and Mesozoic strata lie on the Uncompahgre Formation. These units are separated by a major angular uncomformity from overlying, essentially horizontal, Tertiary formations that include the basal Telluride conglomerate and several thousand feet of overlying volcanic rock. Intrusive rocks, mostly Tertiary in age, are common in the area, and are in the form of dikes, stocks, and sills. Dikes are very closely associated with some of the ore veins (Mayer).

Description of Veins

The Black Bear and Argentine veins range from 0.6 to 7.6 m (2 to 25 ft) in width, but in most places are 1.5 to 2 m (5 to 7 ft) wide. They vary in character from a well-defined tabular structure between sharp "frozen" walls, or gouge seams, to an irregular zone of quartz and quartz-sulfide stringers. Many gouge seams within the veins make the veins blocky and loose.

Common gangue minerals in the veins include quartz, pyrite, rhodonite, chlorite, sericite, clay minerals, epidote, calcite, adularia, rhodocrosite, fluorite, and specularite. Quartz constitutes 60 to 70% of the veins and varies widely in character, ranging in color from clear through white, gray, and green, to amethyst. Chlorite, sericite, the clay minerals, and fine-grained quartz are common alteration products of the vein walls and of wall rock fragments in the veins. Epidote is an alteration of the dike or, less commonly, of tuff-breccia. Calcite, adularia, fluorite, and rhodocrosite, which are widespread but not abundant, form vein fillings with banded or crustified late-stage quartz. Rhodonite is more abundant and occurs either replacing country rock or as a vein filling. Pyrite is moderately abundant, occurring as disseminated 2.5-mm (0.1-in.) cubes in the vein walls and as scattered granular masses or crystals in the veins. Specularite occurs in small quantities in scattered areas.

The major ore minerals are sphalerite, galena, chalcopyrite, and native gold. The sulfides, which occur in grains 1.5 to 12.7 mm (0.06 to 0.5 in.) diam, may show crystal faces, but euhedral crystals are limited to tiny crystals in vugs. The sulfides occur in separate bands or streaks in the veins, although in juxtaposition with bands or intergrowths of the other sulfides. This is particularly true of chalcopyrite. Intergrowths of sulfides usually involve any two sulfides, less commonly all three. The overall ratio of Zn:Pb:Cu for the veins is about 3:2:½.

Native gold occurs as thin plates around quartz crystals or sulfide grains, and rarely as wire or in crystals. The gold is usually finely disseminated and not readily visible to the unaided eye. The gold has a wide range of fineness and ranges in color from pale yellow to almost a copper red.

The vein filling varies considerably in detail. One 4.5-m (15-ft) section of the vein can show radical changes with adjacent sections as to structure, character, and amount of gangue and ore minerals present, and the texture of ore. Every conceivable combination of mineralogy, structure, and ore texture described in this report can be found in the veins.

Alteration of wall rock, or of wall-rock septa within the veins, is common but variable. In places the walls are of relatively fresh rock, but elsewhere they may show extensive "bleaching" for distances of 1 to 3 m (3 to 10 ft) from the vein. Alteration nearest the vein is to quartz-sericite, with local argillization. Outward from the vein the alteration is propylitic, with development of chlorite and related minerals in the groundmass of the dike or in the matrix or groundmass of included fragments in tuff-breccias. In the dikes, replacement of feldspar phenocrysts by chlorite or rhodonite has been observed. Minor epidotization occurs throughout the veins, but it is notably intense along the south Argentine 2000 level.

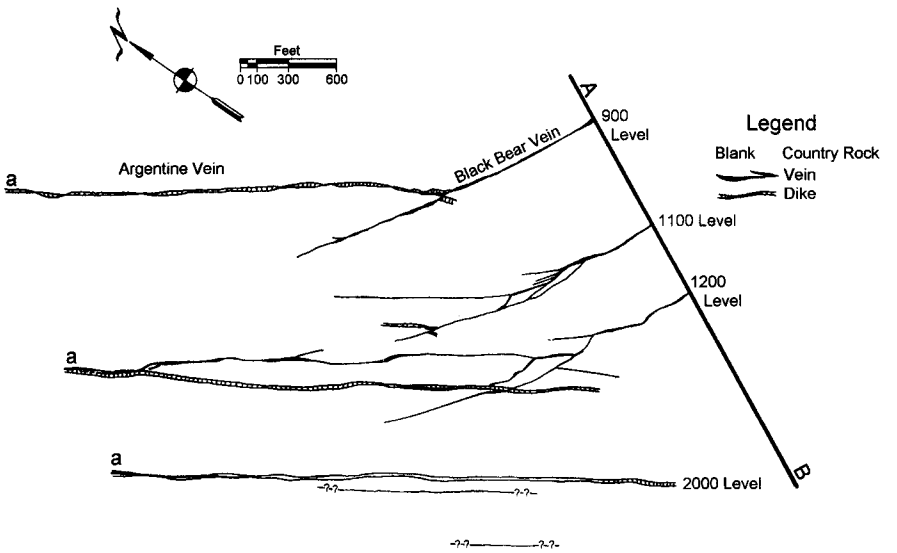


FIGURE 20.1 Geology of the Black Bear–Argentine vein junction. Note that to avoid overlap, the levels have been shifted laterally four times their normal separation along line A-B. This results in distortion of downdip projection on the Argentine. Points labeled "a" on the Argentine lie downdip of one another. Restoration of the levels to their proper positions will bring the point into alignment. The vein and dike widths shown are approximate only. Metric equivalent: ft × 0.3048 = m.

Structure of the Montana–Argentine Vein

The southern half of the Montana-Argentine vein strikes N 0.61 rad (35°) W and the northern half N 0.17 rad (10°) W. The overall dip is about 1.22 rad (70°) west but in detail may depart markedly from this attitude. The vein lies along a fault zone that closely follows a premineralization andesite dike. The dike is 1 to 6 m (3 to 20 ft) wide and in many areas is complex, pinching out, forming two or more segments enclosing large horses of country rock, or fingering irregularly into the country rock.

The vein that has been traced for 4877 m (16,000 ft) is terminated at the north by the northwest-striking Terrible vein and at the south by the Black Bear vein. The dike persists farther south but is offset horizontally 3 to 4.5 m (10 to 15 ft) to the west in the hanging wall of the Black Bear vein. However, mineralization is limited to sporadic gouge slips or narrow quartz stringers on the walls of the dike.

Structure of the Black Bear Vein

In general, the Black Bear vein strikes N 0.78 rad (45°) W in its east half and N 1.09 rad (63°) W in its west half (Fig. 20.1) and dips about 1.04 rad (60°). However, there are marked local variations from this attitude. Only the west half has been explored. Work on the southeast segment is limited to the last few hundred feet of the east 900 and 1200 levels (Fig. 20.1).

The limits of the vein have been defined only on the west, and only approximately there. The west margin is marked by the gradual fadeout of the controlling fault and a decline in mineralization along it. The margin rakes southeast, for vein structure is present at the west face of the 900 and 1200 levels but only a wide zone of curved sheeting is present below these points on the 2000 level.

Geologic Factors in Mining

Geologic factors affecting systematic mining fall into three categories: ore distribution and control, dislocation by faulting, and conditions leading to excessive dilution in the shrinkage stopes.

The effect of offset by the intersecting veins is self-evident. For this reason the veins cause less trouble than the longitudinal faults. The faults cut across the vein at a low angle, and so the miners tend to regard them as the true vein wall with which the faults actually or nearly coincide for a short distance. If followed, however, faults will lead the stope away from the vein into the country rock. Systematic probing of smooth vein walls eliminates this hazard.

In the Black Bear vein, dilution results from caving of walls that are strongly altered or are cut by strong fractures paralleling the pay streak of the vein. Such areas are not abundant.

Dilution in the Montana–Argentine vein is a considerable problem. Some of the causes are:

1. In areas where the vein shows lode character, the pay streak may be confined to a part of the lode, the remainder consisting of barren quartz. The pay streak can swing from one wall to the other or lie in the center of the lode. In such areas it is necessary to determine if the pay streak can be broken from the rest of the vein or, if not, whether the vein will average ore grade for the width to which it will be necessary to mine.
2. The existence of gouge slips along the dike walls, and of fracturing within the dike and country rock parallel the dike and vein, have been described. These features, coupled with the tendency of the pay streak to switch from wall to wall and center of the dike, make the Montana-Argentine very susceptible to dilution. As the pay streak switches from one position to another, the stope will undercut the intervening blocky ground. Pillaring is commonly required—first, to prevent excessive dilution; second, to prevent large slabs from breaking away and blocking the stope draw; and third, as a safety measure for crews working in the stopes.

   In some areas, the vein and dike flatten together, and a set of fractures parallel to the steeper course persists upward. Heavy blocky ground invariably results.

   A certain amount of dilution cannot be avoided because of slivers of dike rock 0.3 to 1 m (1 to 3 ft) thick occurring between the vein wall and a gouge zone within the dike or along the dike wall. Such slivers cannot be

held in mining, and sampling must include them or an erroneous impression of the grade of minable ore will result.

3. As previously noted, two of the potential vein positions may simultaneously carry notable mineralization. Under these conditions it is necessary to decide whether the metal content of both is sufficient to carry the intervening dike waste, and, if not, which position should be followed. In the majority of cases the width of dike waste [1.5 m (5 ft) or more] was sufficient to preclude mining it and both vein positions. Of the two veins, one was generally narrow [0.7 to 1.2 m (2½ to 4 ft)] and of good grade, whereas the other would be 1.2 to 1.8 m (4 to 6 ft) wide and little better than minimum grade.

   Choice of which vein position to follow when both cannot be mined depends on several factors: the preferential position (if any) of the pay streak with respect to the dike in that area; whether the pay streak consists chiefly of stage 2 quartz carrying abundant sulfides (most favorable), stage 1 heavy sulfide mineralization, or stage 2 quartz with little sulfide (least favorable); and the fact that the vein position that will pinch out updip is more often associated with a very strong gouge slip.

Recent Mining Geology

Most of the ore mined from Idarado properties came from veins in the San Juan tuff, and a minor amount came from the latite and andesite members of the Silverton series. In 1976 through 1978, the major portion of ore mined came from replacement ore in the Telluride conglomerate and was found adjacent to veins containing base metals, which served as conduits for fluids that formed the replacement ores.

The replacement ores occur in three forms: lateral replacement, which is a thin shell adjacent to the feeder vein; swells, which extend farther from the vein; and manto type, where the width of the swell greatly exceeds the thickness. Replacement deposits along dike-filled vein structures may occur at several horizons, whereas along veins without dikes, ores are confined to the basal beds of the Telluride conglomerate (Mayer).

MINING

Introduction

The Montana–Argentine–Black Bear veins are being mined over a strike length of 5486 m (18,000 ft) and a dip of 915 m (3000 ft). Dip of the veins varies between 0.78 and 1.57 rad (45 and 90°), with most workings averaging 1.22 to 1.48 rad (70 to 85°). Vein width generally is 1.5 to 2 m (5 to 7 ft), and is occasionally as narrow as 0.6 m (2 ft) and as wide as 7.6 m (25 ft). Replacement ore in the Telluride conglomerate has been mined to widths of 18 m (60 ft) with no alteration of the basic stoping plan.

Pre-Idarado production from vein deposits included in the Idarado property (compiled from USBM records) was:

Tons (st)	Au oz	Ag oz	Pb %	Cu %	Zn %
12,121.026 (13,363.866)	0.26	2.69	0.85	—	—

The production and average grade of the Idarado mine from 1946 to 1976 was:

Tons (st)	Au oz	Ag oz	Pb %	Cu %	Zn %
9,528,332.4 (10,505,328)	0.07	1.95	2.33	0.72	3.62

Development

Levels are driven on the vein at 122- to 152-m (400- to 500-ft) intervals. Footwall drifts have been driven, but with poor results due to variability of the strike and insufficient geological information from diamond drill holes. Raises are driven in the footwall of the vein with an Alimak raise climber, leaving a 3-m (10-ft) pillar of country rock between the raise and the vein. Dogholes are driven from the hanging wall of the raise through the vein on 9-m (30-ft) centers as the raise is being driven. In 1970, two 1.2-m (4-ft) bored raises were tried but a tendency of the pilot hole to steepen in dip made the resulting raises difficult to use. Later, however, using improved techniques, numerous 1.2- and 1.5-m (4- and 5-ft) raises were bored with excellent results. A number of these were up to 260 m (850 ft) in length. Bored raises were used in lieu of cribbed manways. A 1.2- or 1.5-m (4- or 5-ft) raise would be bored at the end of the 76-m (250-ft) block to the level above. In this bored raise, air and waterlines were installed along with steel chain ladders. Sections of pipe and ladders were removed as the stope progressed upward. The raise also served as an emergency exit, means of ventilation, and as a cut to which to blast.

Elimination of the cribbed manway saved much time and labor in the handling and placing of crib timber.

The only time a cribbed manway would now be carried would be in a blind area where no upper level exists or where the boring machine could not be taken.

Figure 20.2 shows the generalized mine layout, and Fig. 20.3 shows details of the vein service raise and single scram with hogged pockets.

Stope Preparation and Mining

Stope preparation begins with cutting a chute and pocket cutout and constructing a slusher platform and grizzlies at the elevation of the scram sill—about 9 m (30 ft) above the sill of the haulage drift. Scrams then are driven on the vein 61 to 76 m (200 to 250 ft) in either direction. A scram is connected to another scram that has been driven toward it from the adjacent raise. At the breakthrough, a second chute and slusher platform are constructed to lessen

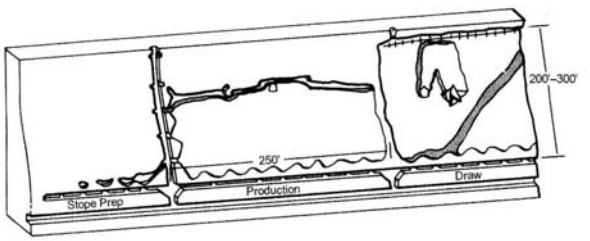


FIGURE 20.2 General layout of the shrinkage stopes at the Idarado mine, at the 122-m (400-ft) level interval, metric equivalent: ft × 0.3048 = m

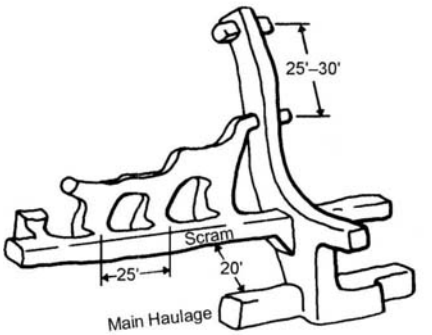


FIGURE 20.3 Vein service raise and single scram with hogged pockets, metric equivalent: ft × 0.3048 = m

the slushing distance. Pockets then are cut in the footwall on 7.6-m (25-ft) centers, the first round being 1.8 m (6 ft) high and 3.6 m (12 ft) wide, and the second 2.4 and 2.1 m (8 and 7 ft). Three rounds then are blasted up from the back of the pocket to the vein, each successively wider. Care must be taken to air the pocket out after each hog round has been blasted. The fourth or fifth hog round should be on the vein and wide enough to break into the next pocket. The back then is blasted down to fill the pocket, and, after the pockets are all hogged together, a leveling-up cut is taken out of the back. A finger raise is driven from the end of the scram up into the stope and a bearing set for the cribbed raise is constructed. At the front end of the stope, a finger raise is driven up into the first doghole of the service raise, and stope preparation is complete.

Stoping begins by taking a V-cut out of the middle of the stope. This provides a brow to which successive cuts are broken. Twenty to 25 holes are blasted at a time, but the last three rows of holes in each blast are not shot to provide holes through the shattered brow for the next blast. Where a bored raise is used to replace the cribbed raise, it can be used as a cut to start mining instead of a V-cut. If a pillar must be left in the stope, it is placed between pockets, mined two cuts high, and hogged over. Before each blast, the pocket below should be slushed to lower the broken ore in the stope. Consequently, the crew on the next shift will have free access to the entire stope and can begin barring down and working immediately. Stopes at Idarado are mined until the draw becomes difficult to control. This frequently is about 76 m (250 ft), at which point finger raises are driven up to gob waste from the next scram and provide ventilation. A catwalk is installed to facilitate draw of the broken muck. Some stopes have been mined the full distance between levels, but this is exceptional.

When the stope is finished, draw begins immediately by slushing the first pocket empty. If the ore rills well, a large percentage can be drawn from the first few pockets. However, if the stope was driven in unstable ground, all the stope pockets are pulled at an even rate to minimize dilution from crumbling

TABLE 20.1 Development cost per foot

	For 4782 ft of Drift*	%	For 1126 ft of Raise†	%
Labor	$16.92	20	$30.74	20
Explosives	6.19	7	8.86	6
Timber	1.18	1	5.80	4
Indirect	38.87	45	69.58	43
Overhead	23.21	27	41.94	27
Total	$86.37	100	$156.92	100

*Metric equivalent: ft × 0.3048 = m.

†Includes 1100 ft. of doghole.

TABLE 20.2 Operating charges per ton milled (for 403,500 st*)

Labor	$5.93	51%
Supplies	3.22	28%
Electric power	0.50	4%
Miscellaneous	1.99	17%
Total	$11.64	100%

*Metric equivalent: st × 0.9071847 = t.

walls. Washing and bombing from the catwalk above generally is necessary to free hang-ups and remove the muck from pillars and flat dips in the footwall. Large muck and slabs are broken in the scram as encountered without hindering other operations in the mine.

When the draw is complete, scram pillars are removed and the stope frequently is used as a waste gob hole. Many of the stopes between the 1200 and 2400 levels in the mine have been filled in this manner, although no attempt is made to fill particular ground.

Productivity and Mining Costs

The development and operating cost breakdowns for 1969 are given in Tables 20.1 and 20.2. However, the percentages were about the same in 1976 (the last year of major production). Typical incentive payments at that time are given in Table 20.3.

Some miscellaneous stoping data are as follows: 39 t (43 st)/miner-shift in stope; 11 holes per miner-shift; 0.36 t (0.4 st) broken per 0.3 m (1 ft) of hole; and 0.56 kg (1.25 lb) of explosive per ton of rock broken.

Overall productivity was 8.59 t (9.47 st) milled per underground miner-shift and 4.14 t (4.56 st) milled per total miner-shifts.

TABLE 20.3 Typical incentive payments

Drifts, per ft*	$14.00
Raises and dogholes, per ft	10.00
Scram, per ft	9.00
Hogging, per cube	0.15
Stoping, per ton* (depending on width and tonnage)	0.69–1.60

*Metric equivalent: st × 0.9071847 = t; ft × 0.3048 = m.

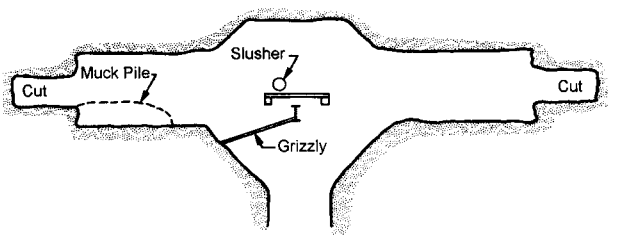


FIGURE 20.4 Longitudinal section of a typical beginning of stope preparation

APPENDIX: STANDARD IDARADO PRACTICE FOR STOPE PREPARATION AND STOPING

This appendix has been included both to describe the practices at the Idarado mine and to present a model format that might be useful to other mines in preparing such instructions.

STOPE PREPARATION

Introduction

All underground mining methods require preparation of blocks of ore for mining before actual stoping operations can begin. Following drifting and raising, stope preparation at Idarado provides the necessary openings for controlled movement of the broken ore from the stope area to a haulage level for transport to the mill.

Efficient mining requires that stope preparation work be completed properly and in time to meet the needs of the stoping crews. The following, which includes potential safety hazards, is meant as a guide to safe and efficient preparation of ore blocks for stoping.

Procedure

The actual stope preparation work begins after the chute cutout and slusher installation work has been completed. The working place at this time is shown in Fig. 20.4.

Driving Scrams. The bazooka is hooked up to the air supply and used for ventilation. When the working place is first entered, the air mover is turned on; the water hose is hooked up; the muck pile is wetted down; and the ribs,

face, and back are washed, so possible missed holes can be seen. Any missed holes are blasted as soon as possible. Water is run on the muck pile while the back, ribs, and face are thoroughly barred down.

A 0.6-m (2-ft) rockbolt, with a washer and expansion shell, is inserted in the pinhole and tightened. The tail sheave and cable are hung on it, taking care to lift properly. The water hose is removed from the muck pile and hooked up to the bazooka so the air-water spray keeps dust down while slushing.

Slushing out is done using sheave blocks to hold up and align the pull-back cable. The slusher and line starter are shut off when finished, which will prevent accidental starting of the slusher due to blast, or when men are at the face. The brake drum is freed and enough slack is pulled off for the next round.

The bazooka is shut off, and the air and water hoses shut off and dragged to the face. The tail sheave and eye bolt are removed from the area near the face, and the face again checked for missing holes. Any loose rock is barred down from the back, ribs, and face.

A third round is drilled out, making sure one or more back holes, 2.7 m (9 ft), are drilled for a pinhole. The scram is to be on the ore, as straight as possible, and wide enough to allow easy passage of the slusher bucket. The scram is kept on the hanging wall of the vein to decrease the waste rock excavation required later when hogging pockets.

The drilling equipment is removed from the blast area, and loading equipment and supplies are carried to the face on the return trip.

The round is loaded following prescribed safety rules for handling explosives and keeping the round shunted. The bottom 0.9 m (3 ft) of the 2.7-m (9-ft) pinhole is not loaded. A stemming or hold primer is used at the proper depth to avoid springing the pinhole.

All equipment is removed from the blast area, the slusher guarded against the blast, and the bazooka hooked up away from the blast. At the end of the shift, the round is wired into the blasting system, blasting is done, and the bazooka is turned on.

Slushing, drilling, and blasting cycles are repeated until the scram is completed. Pinholes are drilled in the rib for air, water, and ventilation lines during the drilling cycle. The ventilation line is kept within 15 m (50 ft) of the face. When time allows or the round in the face cannot be drilled, the first pocket rounds are drilled out but not blasted. Pockets are to be on 7.6-m (25-ft) centers as marked by supervision.

A slusher safety screen and a safety hand line are installed around the grizzly opening as soon as face blasting can no longer damage them.

Hazards to watch for include gas (if there is too much, the working place should be left after turning on the air to air it out); loose rock in back, ribs, and face; missed holes; tripping hazards in the scram and on slusher platforms; broken wires on the slusher cable; dust; ungrounded electrical

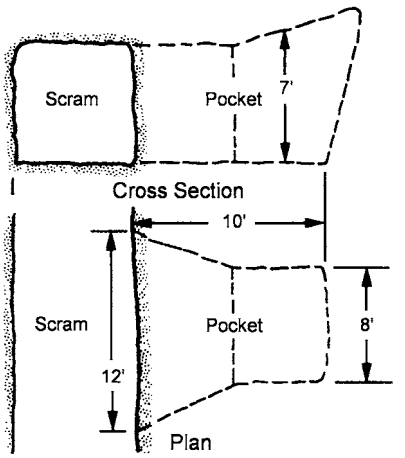


FIGURE 20.5 A typical stope preparation pocket, metric equivalent: ft × 0.3048 = m

equipment; lifting and carrying equipment improperly or with poor footing; and loose air hose connections.

General Procedures. General procedures include (1) keeping the slusher, slusher motor, and slusher rollers free of muck; (2) not pulling the bucket onto the grizzly or into the rollers, and using the bucket of muck to push previously slushed muck onto the grizzly; (3) keeping at least three wraps of cable on the drum to avoid breaking set screws; (4) having the proper size Allen wrench for set screws; (5) adding the maximum amount of cable when needed to keep a minimum number of knots; (6) putting the pull cable on the drum next to the motor; (7) using the proper knot to splice the cable and for attachment to the bucket; (8) keeping and using the cable cutter for cutting cable; (9) disconnecting the lead wire from the blasting line and rolling it up to avoid tripping over it; (10) disconnecting and shunting the blasting line before entering the working place; (11) warning the trammer of excess water in the scram or chute to avoid accidents from spills; (12) picking up bits and ordering powder and primers before going underground; (13) filling the oiler at the beginning of the shift and as often as needed; (14) blowing the air hose, flushing the water hose, washing off connections, and pouring a little oil into the air hose before hooking it up to the machine; (15) checking the screen in the gooseneck to make sure it is in good order and cleaning it before hooking the water hose up; and (16) sending a broken machine to the drill shop as soon as possible.

Cutting Pockets. Pockets are to be cut as shown in Fig. 20.5. Centers will be marked by supervision. Procedures for ventilating, barring down, and wetting muck piles are the same as for driving the scram.

When the scram is too narrow to drill straight into the rib, a V-cut round is drilled first.

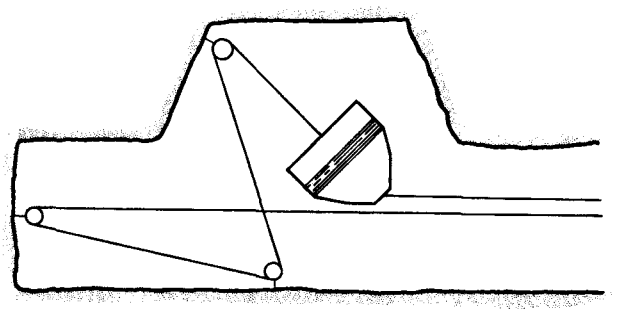


FIGURE 20.6 A plan of the method for slushing the first pocket if necessary

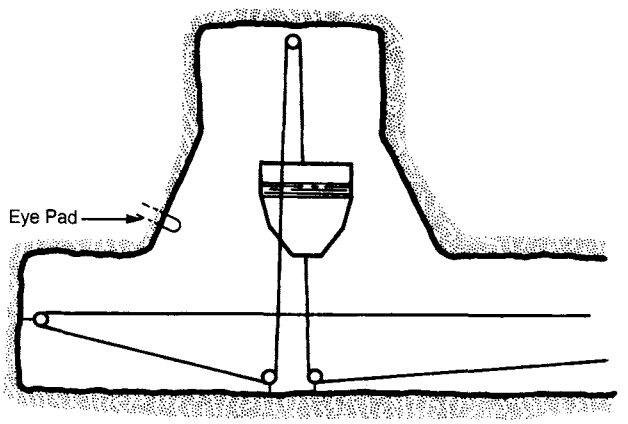


FIGURE 20.7 A plan of the method of double slushing the second round out of the pocket

To slush out a pocket, pinholes are drilled and the cable arrangement used as shown in Fig. 20.6. If a V-cut is used for the round, there will be very little muck left in the pocket.

On the second round in a pocket, the back holes are looked up enough to gain headroom for a stoper (Fig. 20.5). The pocket is slushed out by double slushing, as shown in Fig. 20.7.

Eye pads for sheaves at each pocket are installed as shown in Fig. 20.7.

Hazards to watch for are same as for driving the scram.

Hogging the Pockets. The method of hogging the pockets is shown in Figs. 20.8 to 20.11. The first round is drilled and blasted up, 1.5 × 2.4 m (5 × 8 ft). The procedures for ventilating the workplace, barring down, and wetting the muck pile are the same as for driving the scram and the pockets.

The second round is drilled and blasted, drilling from the muck pile. The pocket is slushed open and the muck wetted down, and work is done in another pocket until the first one is aired out.

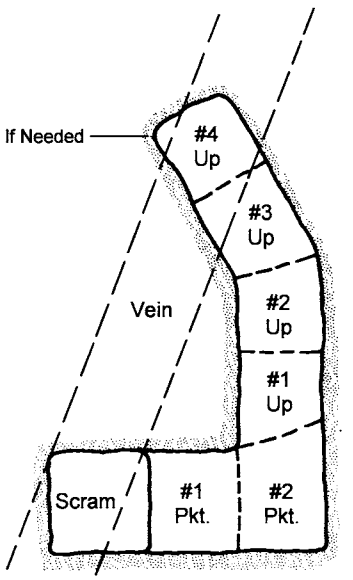


FIGURE 20.8 Cross section of hogging to catch the vein

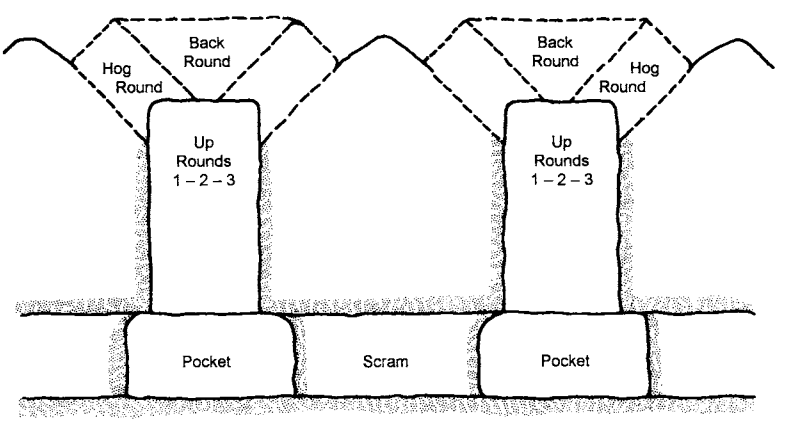


FIGURE 20.9 Longitudinal section showing hog and back rounds in the vein

The third round is staged up, drilled, and blasted, looking at it toward the vein. This is repeated until all of the vein is exposed, remembering all staging must conform with safe staging practices. All the pockets must be brought to this stage of completion to avoid mixing ore and waste.

The back and hog rounds are staged up and drilled out both ways, blasting only the hog rounds. All hog rounds are inspected for missing holes (blasting if necessary) before drilling the hog rounds; then the hog rounds are repeated, keeping in the vein until the pockets are connected. Finally, the pocket is blasted back down to fill the pocket.

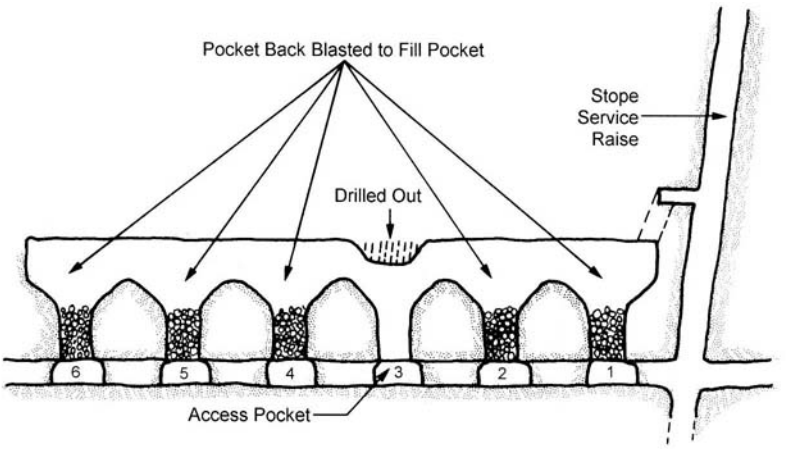


FIGURE 20.10 Longitudinal section showing stope preparation for the third pocket left open for access for finger driving

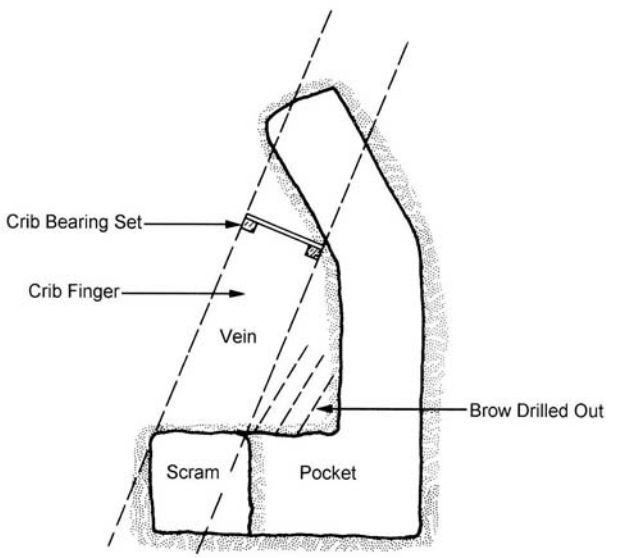


FIGURE 20.11 Cross section showing crib finger and bearing set, plus pocket brow drill out

Before blasting the back down in the No. 3 pocket, a finger is driven from the No. 1 pocket to the raise doghole. This allows equipment to be stored in the No. 2 pocket, with access through the No. 3 pocket while driving the finger (see Fig. 20.10).

A finger raise from the scram is driven to connect with the end pocket for a cribbed manway. This may be done at the same time the pockets are being taken up. Then the crib bearer set and ring of cribbing (see Fig. 20.11), as well

as the air and waterlines across the scram and up the cribbed manway, are installed.

A level up cut across the stope and drill pocket brows to be blasted in the draw cycle (see Fig. 20.11) is made.

Hazards to watch for include falls from staging; gas (if it is too gassy, stay out of the pocket until it is ventilated); loose rock; tripping in the pockets, scram, and on the slusher platform; lifting and carrying equipment improperly or with poor footing; missed holes; plugging off the entrance to the pocket while in the pocket barring down; and drilling setups on unstable muck in partly filled pockets.

General Hints. Every trip should pay, eliminating unnecessary trips; the equipment should be organized; tail sheaves, cables, water hoses, and scaling bars should be dragged to the muck pile on the first trip; and the brake drum should be freed when slushed out so the cable can be pulled from the drum.

Scram Hints. The slusher bucket and a ladder or steel crossways are used in the scram to hold the feed leg when drilling lifters; the lifters are timed to throw the muck pile away from the face; two back holes, 2.7 m (9 ft) deep, are drilled, loading only the front 1.8 m (6 ft) and avoiding drilling separate pinholes if possible; pockets are used for equipment storage; pins are used in the scram rib to hold hoses, steel, and machines up to avoid carrying them back and forth to the slusher platform; and some steel wedges should be available in case the pinhole does not hold the anchor bolt.

Pocket Hints. A chinaman shot is used to kick out the muck from the first up round and the cut is shot by itself when possible to be sure it pulls.

STOPING

Introduction

Stoping is the excavation of a block of ore from underground and is designed to be the fastest and most efficient method of mining in a particular orebody. At Idarado, shrinkage stoping is used.

In order to maintain the high production needed to operate at a profit, each miner must use his time and knowledge in the best possible manner. The following procedure for stoping is designed to help a six-man crew on two shifts use its time and effort as a team to break the largest amount of ore possible.

The actual stoping begins after the stope preparation is finished, including one level up cut and a finger raise connection to the raise doghole, as shown in Fig. 20.12.

Before Entering the Stope

Before the shift begins, the shifters' note should be read to see what the opposite shift did, what problems they had and what equipment will be needed for that day, such as water needles, drill steel, etc.

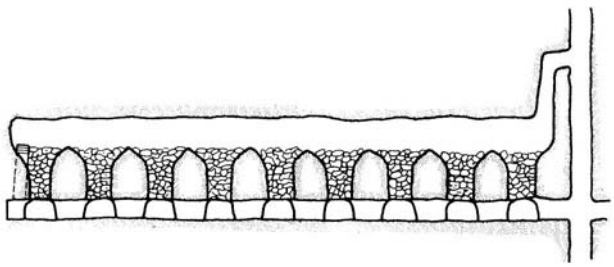


FIGURE 20.12 Longitudinal section showing start of the stope

The powder needed should be ordered from the shifter, and needed equipment picked up at the warehouse.

Upon arrival at the stope service raise, arrangements should be made with the nipper to send down powder, steel, cribbing, or whatever else is needed to eliminate extra trips up and down the raise as well as wasted time.

Entering the Stope

All three miners should enter the stope and completely bar down the back and walls before anything else is started, beginning the barring down in the area where the back is free of loose material, and progressing across the blasted area. The back should be barred down first and then the walls, while standing clear and uphill from rocks being barred down. The rest of the stope should be checked for loose rock and barred down where needed. If a large slab cannot be barred down, it should be blasted down with a small blast. The working area should be well barred down before moving equipment or stringing hose. If it is not barred down, the area is not only unsafe, but equipment and hoses have to be moved out and back in again, tripling the time needed to start work. Freshly blasted muck piles must be wetted down.

Setting Up to Drill

In a well-organized stope, the muck pile will normally be drawn down enough to allow room for drilling but not down enough to require staging to reach the back. Approximately 2 m (7 ft) is about right.

When hooking the machine up, the following procedures should be observed: the air hose should be blown to make sure it is free of obstruction, taking care that dirt is not blown in the face; the water hose should be washed out to be sure it is free of obstruction, and the machine fittings washed off; the oiler hose should be oiled prior to connection to the machine, which gives lubrication to the machine and prevents excessive wear; the air hose should be securely connected to the machine to prevent loss of lubrication and the connection breaking loose, causing injuries; and the oiler should be filled and adjusted for proper machine lubrication.

If the muck pile is too close to the back, a hole deep enough to allow headroom for the drill will have to be dug out.

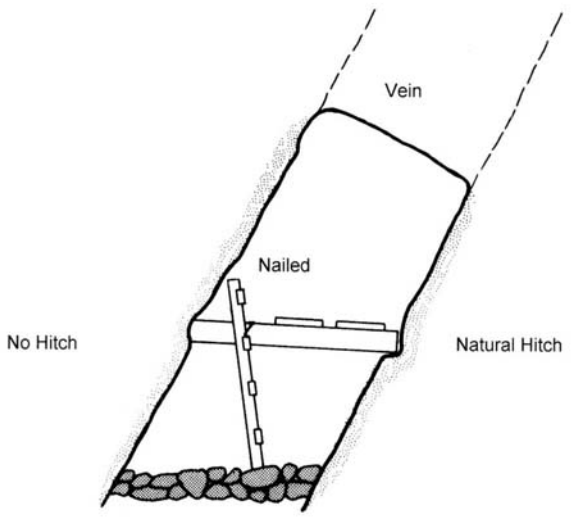


FIGURE 20.13 Staging with ladder

Staging up will have to be done if the muck pile is drawn down too far for drilling. Proper staging proceeds as follows: barring down the area to be staged; measuring down from the back the amount needed for headroom; selecting the natural hitch in walls if possible, and if none is available, picking or moiling at hitch or using a steel hitch or a ladder as shown in Fig. 20.13; measuring the shortest distance between hitches for the bottom of the sprag and cutting the sprag on a bevel so the top is slightly longer than required to assure good wedging action; placing the sprag in hitches and wedging it tight (if additional wedges are needed, they should be placed on the end with the best hitch); checking the sprag several times with an axe to make sure it is secured and, if not, tightening it; placing staging boards on the sprag where needed and nailing them to prevent sliding or tipping [two should be used except when staging is high, when staging should be three boards wide, doubled, and a safety staging installed 1.8 m (6 ft) below the one being used]; and after a few holes have been collared or drilled, inspecting the sprags to make sure they are still tight. See Fig. 20.14 for a complete staging.

Drilling

An average of 25 holes per machine shift is required in each stope to maintain production. However, the holes must be drilled correctly or the ground will not break and the holes will be wasted. The following items must be followed to get the most out of each hole.

Proper Spacing and Burden. This comes with experience and varies with the ground. If there is too little ground on the holes, the rock will break very finely but some of the holes will not be needed. This results in wasting time and effort in drilling too many holes. If there is too much ground on the holes, the rock will not break and will result in more work to come back and drill additional relievers. The way to determine the right amount of ground is to

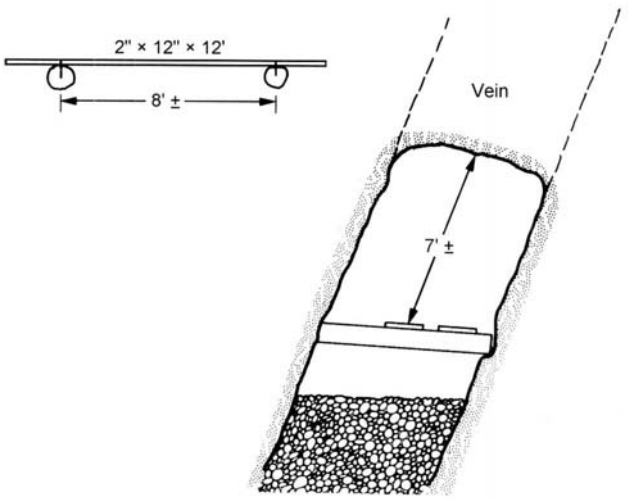


FIGURE 20.14 A complete staging, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m

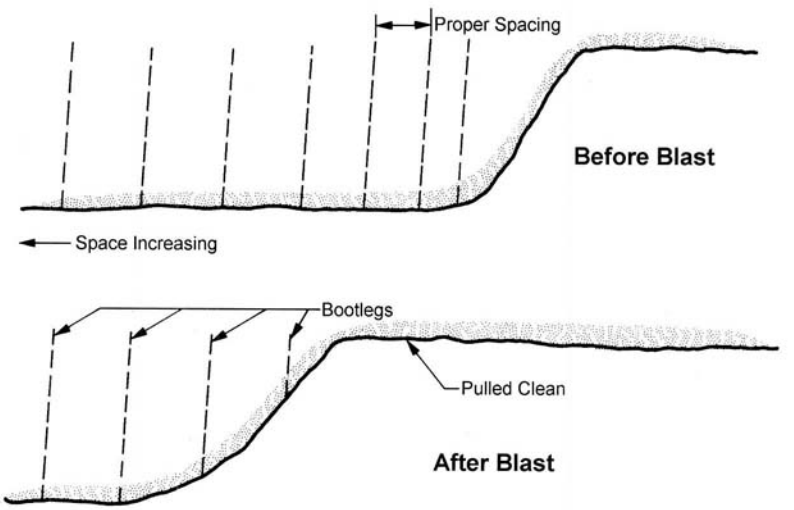


FIGURE 20.15 Section showing the method of determining burden

start with a little less than should be on the holes and then start increasing it. Repeat this until some bootleg appears and then shorten a few inches. See Fig. 20.15.

Drill Holes Parallel. To drill all the holes parallel is very important, as it maintains the proper spacing and burden along the entire hole, not just at the collar. This will require that the stinger be moved for each hole to prevent fanning the holes. A loading stick—not a drill steel—in the hole just drilled will help align the next hole parallel. See Fig. 20.16.

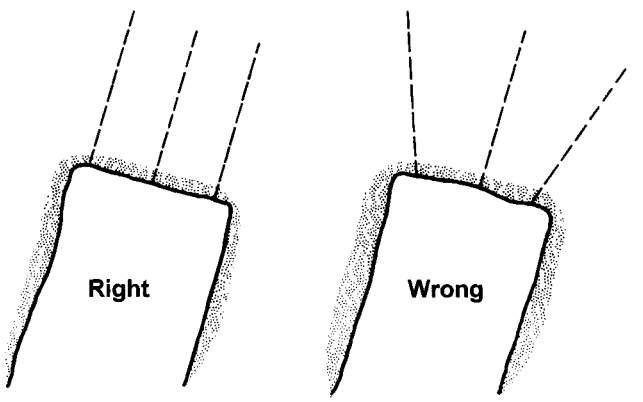


FIGURE 20.16 Section showing hole alignment

Preserving the Hanging Wall. In the dipping veins at Idarado, it is important to keep the holes out of the hanging wall. A broken hanging wall not only breaks waste, but starts the hanging wall slabbing. Then a pillar will have to be left for support, which causes extra work, loses the ore left in the pillar, and makes the stope harder to draw.

The following items are to be remembered and followed when drilling to prevent injuries. Frequent barring down should be done because drilling causes the rock to loosen up in the back and the hanging wall and footwall; a drill steel should never be left in a hole–it may fall out and cause a broken foot; the drill should be watched when changing steel in a hole–it could hang up, come out of the drill, and fall out of the hole, possibly breaking a hand or a foot. The steel should be rotated a little when pulling it out of a hole to keep it from sticking; a person should always stand so that if the machine should shank, balance is not lost, thus avoiding falls or back injuries (if the machine gets out of control, it should be let go); drilling should always be off a stinger block; when drilling in or through a mud slip or gouge zone, the machine should be rotated full speed, but not given too much feed. Too much feed will just poke the steel into the mud and plug or stick it. The same is true when drilling through a vug hole. Also the chuck should be watched. If the steel gets plugged, water will blow out of the chuck around the steel. When this happens, the steel should be blown quickly before it plugs tight, and the feed should be reduced until through the mud seam. If a steel is stuck, it should be removed using a twister. The twister should not be worn, and should be pulled on while twisting the steel back and forth. Sometimes the twister may be put close to the back and a bar used to pry down on the twister. Another method is to drill a hole alongside the stuck steel to the depth of the bit and then free the stuck steel.

Making a Cut

To make a cut across the stope, both machines are set up in the middle of the stope. One person drills a V-cut as shown in Fig. 20.17, and both miners drill back from the cut, as also shown in Fig. 20.17. A typical drill pattern is shown in Fig. 20.18.

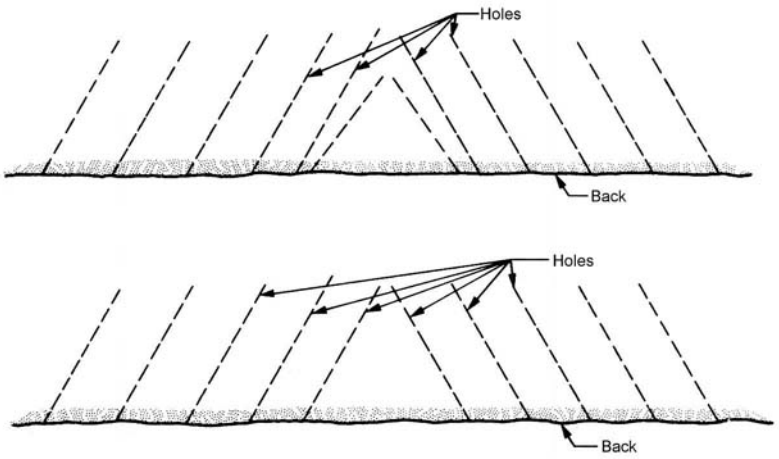


FIGURE 20.17 Section showing V-cut holes. Top shows V-cut for difficult ground and bottom shows V-cut for easy pulling ground.

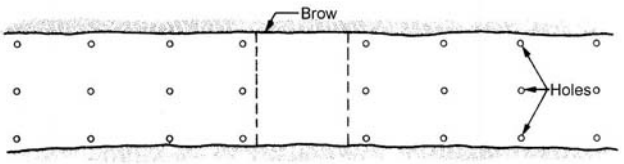


FIGURE 20.18 Plan showing a typical drilling pattern

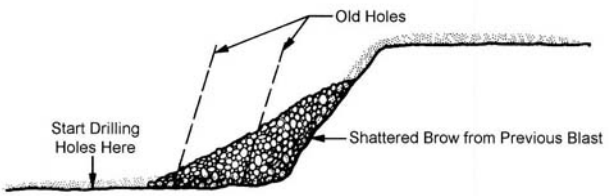


FIGURE 20.19 Section showing the proper brow for drilling

The following items are to be remembered: the V-cut is drilled so the holes do not cross; if they do, the cut will pull only to the point where they cross. Sometimes a baby V will be required to pull the cut; the cut should pull. A good brow is very important to shoot to. The cut should be shot by itself if possible to make sure it pulls, and to allow redrilling and reshooting it if it does not, before anything is shot to it; two or three rows of holes are always left unblasted to avoid drilling in a shattered brow (see Fig. 20.19); while retreating from the center of the stope, extra hose should be unhooked and coiled; the stope should be kept higher in the middle than on both ends, which makes each row of holes bottom a little lower than the previous row and insures a good face to which to shoot

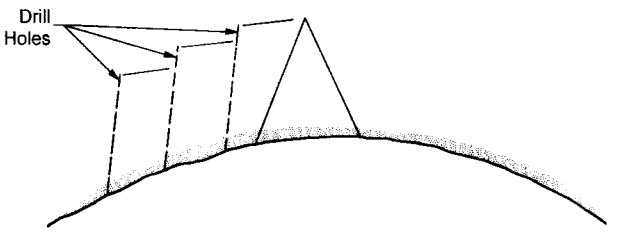


FIGURE 20.20 Section showing the stope arch for better drilling

(see Fig. 20.20); and equipment should always be moved back from the blast area to keep from covering it up and breaking it with the blast. Also, equipment should be kept away from areas that are being drawn to avoid sucking the equipment down into the muck.

Leaving a Pillar

To leave a pillar when marked by supervision, it should be drilled up to its edge while retreating from the center of the stope; then a drop is made behind the pillar, a new V-cut started, and drilling done to the other edge of the pillar. This is repeated for one more cut, laying over slightly into the pillar to shorten the hog round distance, as shown in Fig. 20.21.

Then the hog rounds and the back are drilled out as shown in Fig. 20.22, but only the hog rounds are shot.

The hog rounds are repeated if necessary to connect over the pillar, always making sure there are no missed holes that could be drilled into. Then the drilled-out back is shot down to fill in the area around the pillar. The opening over the pillars is made large enough for good access and to allow a basket stretcher to pass easily. See Fig. 20.23.

Driving a Finger

The driving of a finger raise for access from the next raise doghole is delayed until the stope is one or two cuts above the old doghole. Then a finger round and the back over the No. 1 pocket is drilled out. Only the finger round is shot. Drilling and blasting finger rounds are repeated until connection with the doghole is made. Care is taken on the last finger round connecting to the doghole. If possible, a hole is drilled through into the doghole to determine how the round should be drilled so muck or concussion will not blow out the timber or pipe in the manway. Holes are not drilled above the bottom of the doghole, as this will blow muck into the raise. The finger round should break into the bottom of the doghole.

If the doghole is 0.9 m (3 ft). deep or less, the cut for the finger round should be drilled as close to the stope side of the finger round as possible.

The round should be delayed as much as possible when loading it to minimize the number of holes going at same time. This reduces concussion and the possibility of blowing out the timber or pipe.

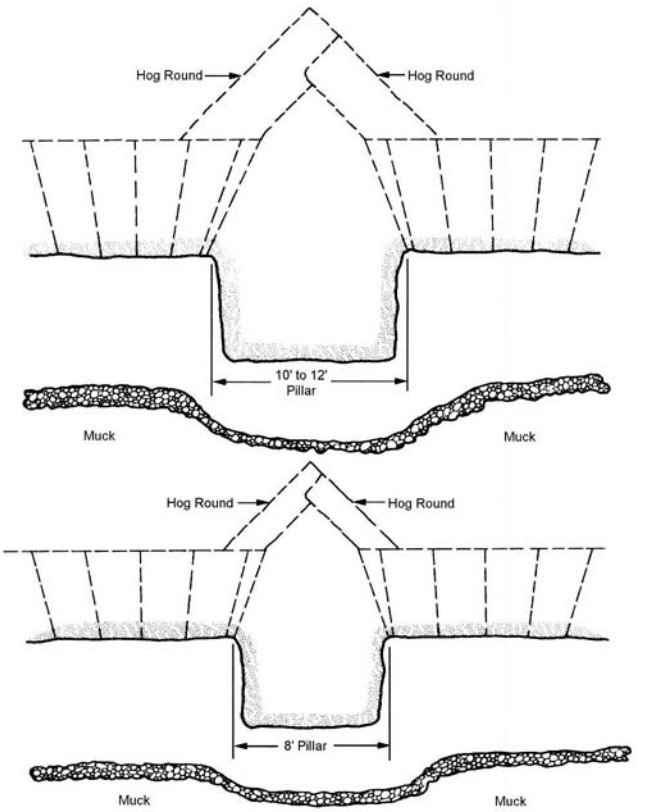


FIGURE 20.21 Section showing the method of leaving pillars, metric equivalent: ft × 0.3048 = m

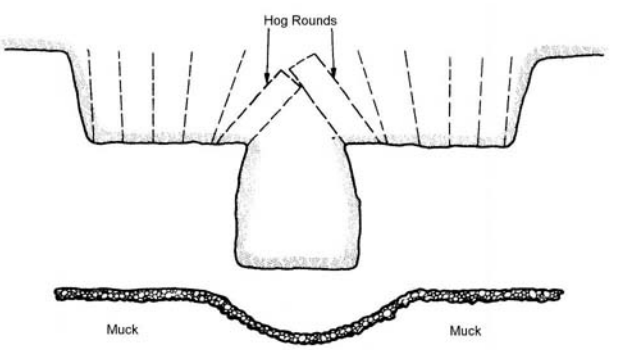


FIGURE 20.22 Section showing a blasting hog round over a pillar

Then the drilled-out back is shot down to fill in the No. 1 pocket. The finger and connection are made large enough to allow passage of the basket stretcher. The finger is driven straight up the vein to connect with the bottom of the doghole and is not broken into the raise. Sprags and ladders are installed for safe access into the stope. See Figs. 20.24 and 20.25.

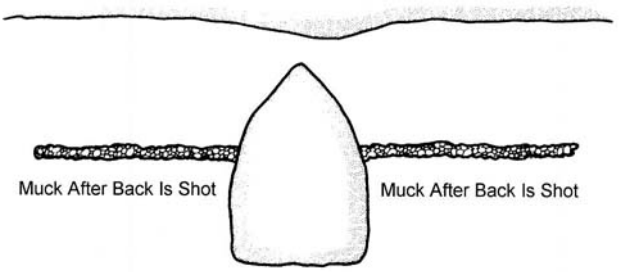


FIGURE 20.23 Section showing a pillar hogged over

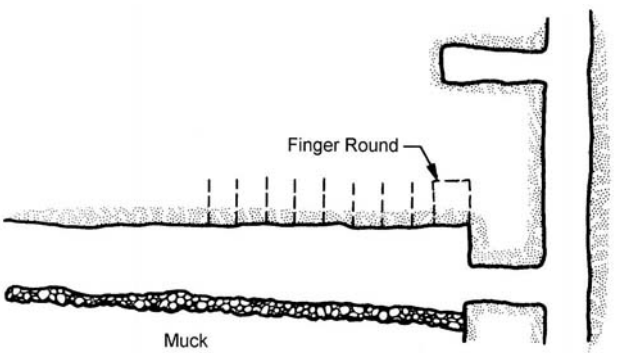


FIGURE 20.24 Section showing the start of a stope finger

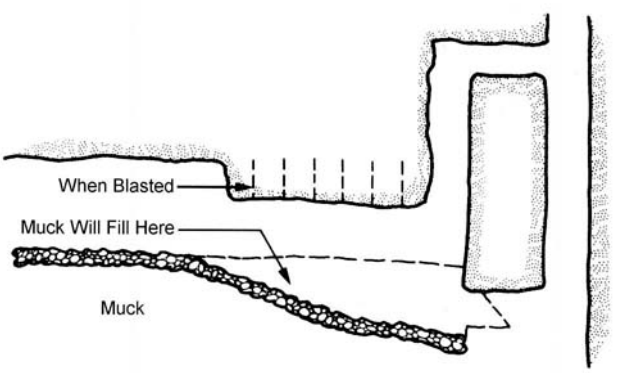


FIGURE 20.25 Section showing a finger breakthrough

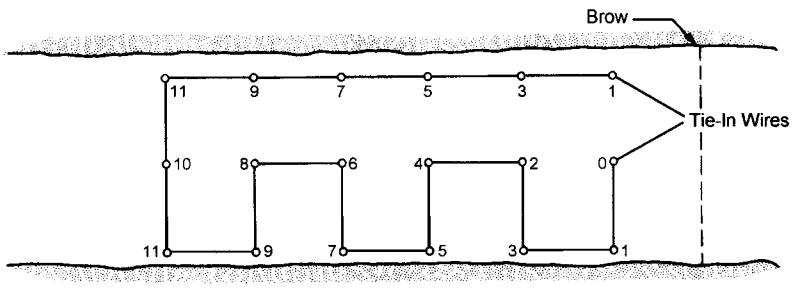


FIGURE 20.26 Plan showing a typical timing pattern

The Third Driller

The third driller has a very important job. It is the responsibility of this worker to keep the two drillers free to drill. Therefore, this driller must be flexible and able to decide what to do first to keep the stope running smoothly. The following is a list of the responsibilities of the third driller.

Slushing. The third driller must draw the stope enough to allow drilling from the muck pile without digging for room or staging up to reach the back, even though it is easier to stage up than dig down. It takes experience to know how much to pull and out of which pocket. This worker should draw the muck after a section has been loaded but before blasting. This allows the section to be blasted without getting muck-bound, and allows the drawn area to be blasted again, making sure it has dropped completely. The third driller should always warn the crew where slushing is taking place and make sure the muck is drawing to avoid sucking the crew down in the muck.

Loading. The third driller should arrange to have powder, prills, and primers sent down early so the crew can help carry them to where they are needed. This avoids having partners walk in empty-handed and having to make extra trips. When a section has been drilled out to where the machine is more than 3 m (10 ft) away, this worker can start loading.

The holes should be timed as shown in Fig. 20.26. First, a cap is inserted completely into a stick of dynamite. With the cap pointing to the bottom of the hole, a half hitch is made around the stick of dynamite. The stick of dynamite is inserted into the hole and tamped firmly with a wooden loading stick (a metal loading stick is never used). The procedure is repeated until all the holes are primed. The wires are left shunted.

Next, the prill pot is hooked up, the air turned on, and the pot blown to clean it out. The air is shut off, the pot bled, and then opened and charged with prill. Each sack of prill will load approximately 11 holes. This figure is used to calculate the amount of prill needed. Air is turned back on. The loading hose is inserted into the hole about 0.3 m (1 ft). The ball valve is opened and the hole is allowed to fill up with prill, which should take only 1 sec or so. The ball valve is shut and the procedure repeated on remaining holes. After the holes have been loaded, the air is shut off, the pot bled and washed out, and all the equipment is moved away from the blast area.

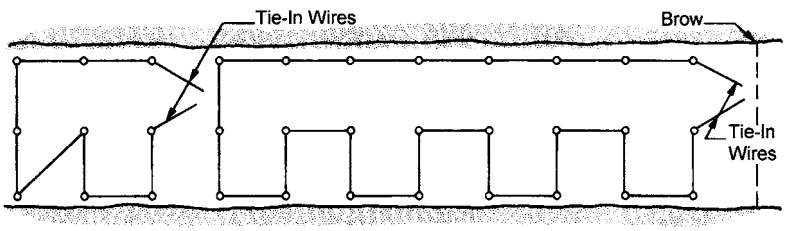


FIGURE 20.27 Plan showing a timing pattern for over 25 holes

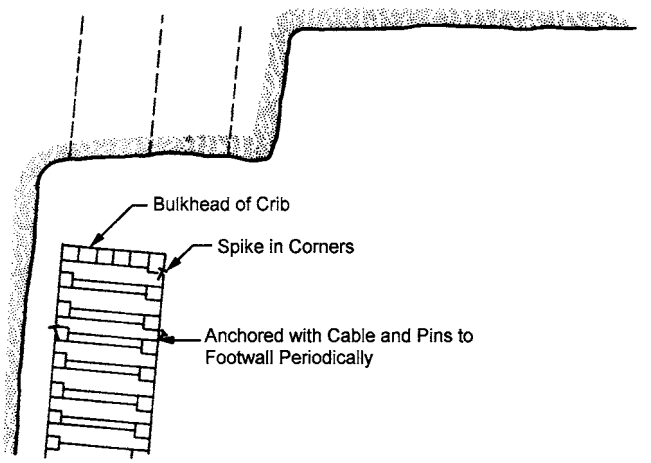


FIGURE 20.28 Section showing the crib manway. Note: One ring raises the crib 0.3 m (1 ft).

Then all the holes are wired up in series, as shown in Fig. 20.26, making sure that all the holes are wired in and not in a separate circuit. The tie-in wires are left shunted until blast time. If there are more than 25 holes to shoot, two series will have to be made, as shown in Fig. 20.27.

When everything is ready to blast, the lead wire from the tie-in wires is strung to the blasting line. The circuit is checked with a "bug" and, if it checks, is wired into the blasting line. If it does not check, all connections are inspected. If no bad connections are found, all individual holes are checked until the problem is found and corrected. At the end of the shift, after making sure everyone is out of the working place, the blasting box is hooked up and blasting takes place. The blasting box is unhooked and the blasting line shunted out before leaving.

Raising the Crib. When cribbing or pipe is to be raised, it should be sent down early so the partners can help carry it in. To raise the crib, the back is drilled out over it, and the bulkhead and nail crib removed as shown in Fig. 20.28. Pinholes are drilled every 3 m (10 ft) or so, and cable is used to anchor the crib, as shown in Fig. 20.28. The crib is bulkheaded over and then the back is loaded and blasted over the crib.

To raise the pipe, the air and water down in the scram are shut off. The driller on that end helps to remove the manifolds, install the pipe, and put the manifolds back on. Victaulic couplings should be ordered with the pipe.

Making Setups. All setups would be made as described earlier in staging.

Drilling. In order to keep the drill running as much as possible, the third driller may drill for one partner, and then the other, as they each take a lunch break. This allows both machines to keep running through lunchtime and should give about 10 to 12 more holes per shift than if everyone took a lunch break at the same time.

Leaving the Stope

Before leaving at the end of the shift, a check is made to see that all equipment is clear of any blast. The bazooka or air mover is turned on if needed, and a check is made to see what the opposite shift will need. The shift boss is told how many holes were drilled, how much steel and bits were lost, how many holes were blasted and where, and what equipment the next shift will need. It is also good practice to leave a note for the opposite shift to tell them what was done, what problems were encountered, and what to watch out for in the stope. This note may be left in a partner's locker or some other prearranged place.

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CHAPTER 21

Cost Calculations for Mechanized Shrinkage Stoping

Gordon M. Pugh and David G. Rasmussen

INTRODUCTION

Shrinkage stope mining can be employed in steeply dipping veins where both the wall and the vein material are sufficiently strong to stand with only minimal ground support. Shrinkage stope mining takes advantage of the swell in volume produced in breaking in-place ore. This expanded volume of ore is allowed to occupy part of the mined-out vein, and in so doing provides ground support and a working surface for the miners. Only enough of the ore is drawn after each stoping cut to allow working room. Not until the entire mining block has been broken is the full amount of broken ore drawn from the stope.

DEVELOPMENT PRACTICES FOR SHRINKAGE STOPING

The vein is developed into stoping blocks, each of which is 76 to 91 m (250 to 300 ft) in length and 61 m (200 ft) between levels. Level development, described later, is either on the vein or parallel to it with crosscuts periodically driven over to the vein. There are usually two exits from each stope. The first entry is a raise at one end driven either conventionally by raise climber, or by a raised borehole. Practice is that this raise, put in from level to level, is either a 2.1 × 2.1-m (7 × 7-ft) Alimak raise or a 1.2- to 1.8-m (4- to 6-ft) diam borehole. Either raise system will require provisions for an ore chute carried with the stope to the haulage level or the scram drift. At 9- to 12-m (30- to 40-ft) vertical intervals, dogholes are developed from the raise so that when stoping breaks back into these dogholes, timber or steel sets and necessary service facilities can be installed. The second entry to the stope may be either a similar raise at the stope endline or, more economically, a cribbed raise carried with the stope.

Development of the ore block is achieved by either drifting on the vein or by paralleling the vein with a footwall lateral and then crosscutting over to the

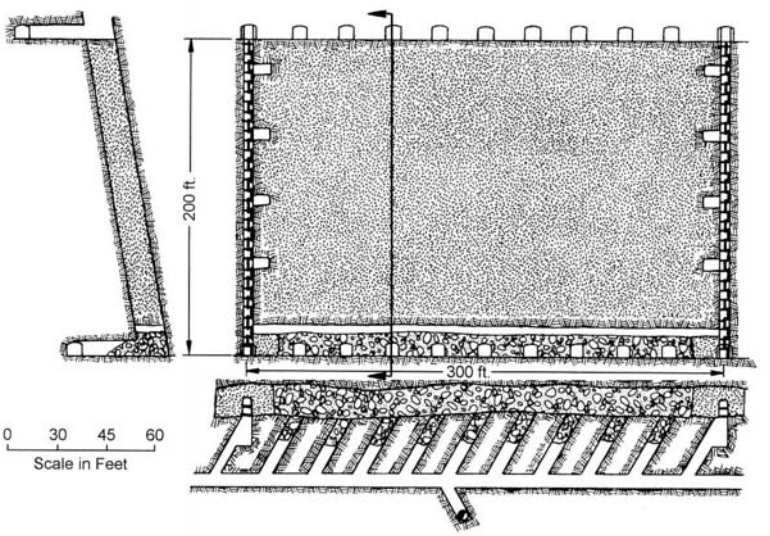


FIGURE 21.1 Mechanized shrinkage stoping with drawpoints developed on the haulage level by undercutting, metric equivalent: ft × 0.3048 = m

vein at specified intervals. In the following discussion, three basic cases are illustrated in Figs. 21.1 to 21.3.

In Fig. 21.1, development is in the footwall by a 3 × 3-m (10 × 10-ft) trackless lateral. A heading is also driven on the ore and taken either full width or driven a standard size and slabbed to the vein width. Approximately 9-m (30-ft) long crosscuts on 7.6-m (25-ft) centers are driven from the lateral to the vein. For ease in muck handling, they are usually put in on a slant rather than at right angles. A single orepass, located off the lateral and in the center of one stope block, can serve not only that stope, but also one on either side, up to a tramming distance of not more than 137 m (450 ft). Raises are put at the stope endlines on 91-m (300-ft) centers. The back of the vein is shot down and just enough broken material is drawn off to permit access for men to work off the broken ore.

Development as shown in Fig. 21.2 progresses by driving a 3 × 3-m (10 × 10-ft) trackless heading on the footwall side of the vein. The ore block is delineated by raises on 91-m (300-ft) centers. Crosscuts are again driven on 7.6-m (25-ft) centers over to the vein. Approximately four rounds are then taken up to the vein, and each round is successively belled out so that all the crosscuts (pockets, in this case) connect. In this development, as opposed to that shown in Fig. 21.1 where there were no belled raises, a portion of the vein material must be left in place as drawpoint pillars during the mining phase. As soon as breaking at the top of the stope has been completed, and the ore has been drawn down, these pillars may be recovered. The development steps are summarized in Table 21.1.

Development as shown in Fig. 21.3 differs from the previous two cases in that either the track or trackless heading is driven on the vein. As in the other

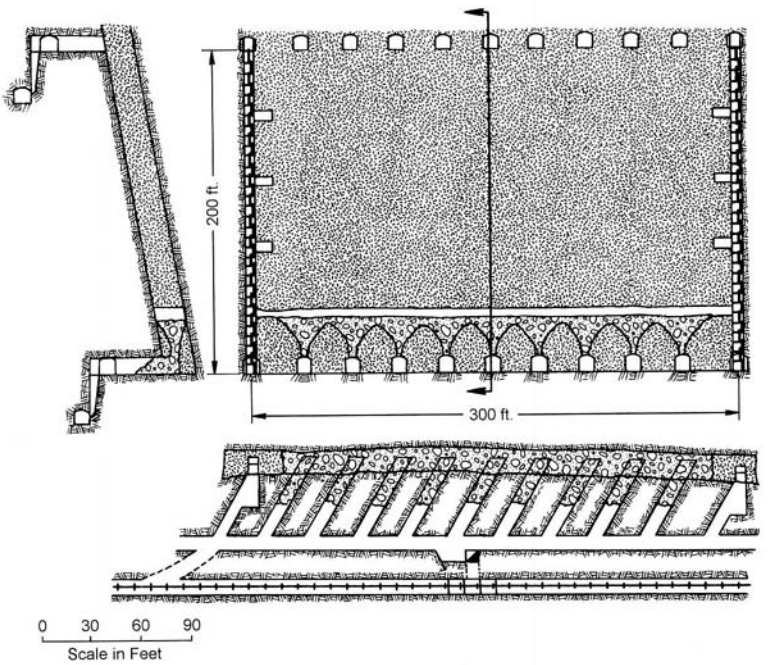


FIGURE 21.2 Mechanized shrinkage stoping with an intermediate scram drift and belled drawpoints, metric equivalent: ft × 0.3048 = m

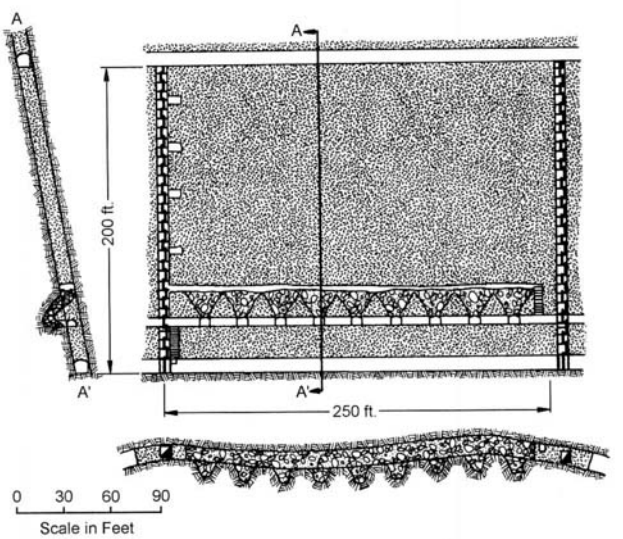


FIGURE 21.3 Mechanized shrinkage stoping with an intermediate scram drift and belled drawpoints. This differs from both Figs. 21.1 and 21.2 in that either the track or trackless heading is driven into the vein. Metric equivalent: ft × 0.3048 = m.

TABLE 21.1 Steps in the development of an ore block for shrinkage stoping

1.	Drift on ore in bottom of ore block.
2.	Drift on ore in top of ore block.
3.	Raise up for Alimak cutout at scram elevation.
4.	Drive Alimak raise through to the next level directly above.
5.	On 30-ft* intervals along the raise, put in 10-ft dogholes (2 rounds each).
6.	Drive 250 ft of 6 × 6-ft scram drift.
7.	Drive pockets on 25-ft centers into the footwall at a distance of 20 ft. They are 6 × 6 ft.
8.	Hogging of the pockets to form the belled raises and undercut (enlargement).

Note: Refer to Figs. 12.1 and 21.2.

*Metric equivalent: ft × 0.3048 = m.

cases, primary stope access is established by using either an Alimak raise or a borehole at the stope endline. This raise will serve a 76-m (250-ft) long stope in both directions along the strike. Above a sill pillar at the bottom of the ore block, an intermediate scram is developed in the ore. The scram is driven using an 18.6- or 22.3-kW (25- or 30-hp) slusher. If the vein is wider than 1.8 m (6 ft), the heading is on the footwall side and the ore on the hanging wall is then slabbed off. Pockets about 1.8 × 1.8 m (6 × 6 ft) are then driven horizontally on 7.6-m (25-ft) centers into the footwall for a distance of about 6 m (20 ft). As in Fig. 21.2, four rounds are taken up and belled out so they will all connect. These upward rounds also must angle back to the hanging wall because the pockets have started in waste. Above the scram, at 9- to 12-m (30- to 40-ft) intervals, 3-m (10-ft) doghole headings are driven off the access raise. Ground is excavated, and a lined chute is installed through the sill pillar immediately adjacent to the raise. A slusher located at the raise will pull development and stope rock along the slusher lane and into the chute. The stope is then ready for mining as soon as a secondary access, in the form of a cribbed raise, is driven off the scram to connect with the excavation above the last pocket.

STOPING

Regardless of the block development method, the same general mining procedure will apply. Breaking is done by two miners using either stopers or jacklegs, depending on whether back stoping or breasting is done.

Shrinkage stoping offers two distinct advantages over many other mining methods. First, the slushing or mucking of broken ore is comparatively free of cycle time limitations and is really an independent function, the only requirement being that there is sufficient broken rock to handle. Second, because of the spaced dogholes from the access raise and the resultant pillars, there are not the traditional problems created by blasting each successive first cut off the raise timber. The independence of the rock handling procedure makes back stoping more attractive than breasting for shrink stopes. There is seldom any fixed size of round required. Essentially, production tonnage is controlled by the muck requirements in the pockets and working room requirements in

TABLE 21.2 Summary of costs for development operations

	Cost per ft*
7-ft intermediate drift (12 hr cycle, 6 ft per round)	$71.52
7-ft Alimak raise (8-hr cycle, 6.65 ft per round)	82.47
8 × 9-ft track drift	98.25
10 × 8-ft timbered track drift	139.88
13 × 7-ft conventional raise	117.25
5-ft diam bored raise	55.10
Initial stope cut over drift	66.05

*Metric equivalent: ft × 0.3048 = m.

the stope. Muck or broken rock drawdown is extremely important because the broken rock forms the working floor, and any extreme variations in the amount of pull would not only be difficult for the movement of personnel and supplies but would also create dangerous conditions.

As stoping progresses upward, a new doghole will be broken into off the access raise. As soon as this breakthrough permits entry, the doghole immediately below is allowed to become muck-filled and inaccessible.

COSTS AND PRODUCTIVITIES

The costs were calculated, based on knowledge of both the productivity and cycle time elements, for each of the mining methods being considered (see Tables 21.2 to 21.15). Costs for equipment, fuel, tires, and other expendables are 1976 figures, and hourly operating costs for each piece of equipment are also given in these tables. The labor cost used has been fixed at $10.50 per hr, based on average rates at underground mines in the United States in the late 1970s, including 35% markup for fringe benefits and 25% for an incentive payment system. For detailed cost calculations, see Pugh and Rasmussen (1977), Section 2.2, Chapter 13; Section 3.2, Chapters 10 and 11; Section 3.3, Chapter 5; and Section 3.4, Chapter 7.

The total mining cost, based on the foregoing assumptions, consists of development costs and stoping costs. Development costs are determined by first creating a development plan of drifting, raising, and chutes or mill holes. Several specific development plans are considered for each stoping method. Prior to this, appropriate unit costs were applied to the development plan. Total costs calculated from dollar-per-foot figures were then expensed over the ore block directly associated with the development. The development costs do not include shafts, slopes, or other major support systems, but only the costs incurred for the development of the specific tonnage block.

Stoping costs were calculated using derived unit operation costs, for drilling, blasting, mucking, stope preparation, and ground support installation when required. Selections of proper unit operations were made for specific stoping cases and their related costs compiled to give a stoping cost.

TABLE 21.3 Development conditions and costs for 7 × 7-ft crosscuts from scram drift to drawpoints

Conditions*	
Rounds drilled with 1 jackleg, 6 ft advanced per round, 23.8 st per round	
There will be 3 rounds per pocket	
2-person crew	
Muck removed with slusher (6 stph double slush)	
1 rockbolt per ft of advance	
Cycle times	
Drill and blast	4.0 hr (approximately)
Muck removal	4.0 hr
Ground support	1.0 hr
Miscellaneous	0.5 hr
Travel and lunch for 1½ shifts	2.5 hr
Total	12.0 hr

Costs*		Labor, $	Supplies, $
Drill and blast	8 miner-hours	84.00	56.57
Muck removal	8 miner-hours	84.00	
	Slusher at $0.30 per st		7.14
	Tram at $0.30 per st		7.14
Ground Supports	2 miner-hours	21.00	
	Bits and steel at $0.15 per ft		5.40
	6 rockbolts at $2.64 each		15.84
	2 mats at $2.70 each		5.40
Miscellaneous	1 miner-hour	10.50	
	6-ft 2-in. pipe at $1.87 per ft		11.22
	6-ft 4-in. pipe at $3.98 per ft		23.88
	6-ft vent bag at $5.50		33.00
Travel and lunch	5 miner-hours	52.50	
Small tools and supplies at 6% labor			15.12
Subtotal		252.00	180.71
Total	$432.71 per round, or $72.12 per ft		

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 21.4 Conditions, cycle times, and costs for the sublevel cutouts off a standard 7 × 7-ft Alimak raise

Ore Block Height Calculation*			
200 ft = level interval (track to track)			
–10 ft = 10 ft high track drift			
–20 ft = crown pillar = 20 ft			
170 ft = ore block height for stoping			
Assume 4 stubbed-in sublevels, each only 10 ft long, with a 7 × 7-ft cross section			
Conditions*			
Round drilled with jackleg, using $1\frac{3}{8}$-in. bit, 6 ft advanced per round (23.8 st)			
2-person crew			
One half of each round will fall directly down, and the other half will be hand scraped away			
No rockbolting or timber required			
Cycle			
Drill and blast	4.5 hr (approximately)		
Muck removal	3.5 hr		
Ground support	no time		
Miscellaneous	1.5 hr		
Travel and lunch (for $1\frac{1}{2}$ shifts)	2.5 hr		
Total	12.0 hr		
Costs*		**Labor, $**	**Supplies, $**
Drill and blast	9 miner-hours	94.50	56.57
Muck removal	7 miner-hours	73.50	
	Mucker at $0.15 per st		3.57
	Tram at $0.30 per st		7.14
Miscellaneous	3 miner-hours	31.50	
Travel and lunch	5 miner-hours	52.50	
Subtotal		252.00	67.28
Total	$319.28, or $53.21 per ft		

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 21.5 Conditions and costs for a 5 × 5-ft crib raise with manway (10-ft vein)

Conditions*

Assume that each 5-ft round pulls 6 ft. It is drilled with 1 stoper, using $1\frac{3}{8}$-in. bits, and yields 12.1 st per round

Cycle times

Drill and blast	3.0 hr
Muck removal at 12 stph	1.0 hr
Ground support 4 piece crib, 4 ft by 6 × 6 in.	1.5 hr
Miscellaneous	1.0 hr
Travel and lunch	1.5 hr
Total	8.0 hr

Cost per Round*		Labor, $	Supplies, $
Drill and blast	6 miner-hours	63.00	56.57
Muck removal	2 miner-hours	21.00	
	Slushing at $0.15 per st		1.82
	Tramming at $0.30 per st		3.63
Ground support	3 miner-hours	31.50	
	4 sides × 12 pieces per side = 48 pieces of timber. Each piece = 4 ft by 6 × 6 in. = 12 bd ft, 48 × 12 at $0.28 per bd ft		161.28
Miscellaneous	2 miner-hours	21.00	
	6 ft of 4-in. pipe at $3.98 per ft		23.88
	6 ft of 2-in. pipe at $1.87 per ft		11.22
	6 ft of vent tube at $5.50 per ft		33.00
	Small tools and miscellaneous supplies at 6% of labor		10.08
Travel and lunch	3 miner-hours	31.50	
Subtotal		168.00	301.48
Total	$469.48, or $78.25 per ft		

Cost per st of ore

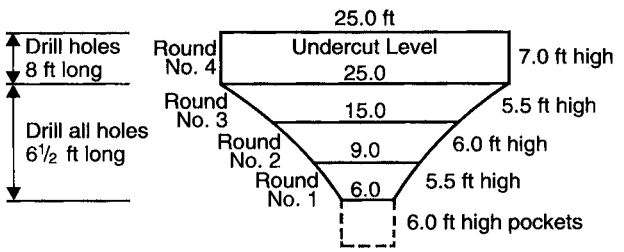
$$\$ \text{ per st raise construction} = \frac{\text{total cost per raise round}}{\text{associated tonnage per raise round}}$$

$$= \frac{\$469.48}{1100 \text{ st}}$$

$$= \$0.43 \text{ per st}$$

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; bd ft × 0.002359737 = m^3; st × 0.9071847 = t.

TABLE 21.6 Assumed geometry for the hogging of pockets in figs. 21.1 and 21.2 (9 identical pockets)



Volume Calculations During Hogging

	10-ft Width	18-ft Width
Cut No. 1	$5.5 \times \frac{(9.0 + 6.0)}{2} \times 10 = 413$	$5.5 \times \frac{(9.0 + 6.0)}{2} \times 18 = 743$
Cut No. 2	$6.0 \times \frac{(15.0 + 9.0)}{2} \times 10 = 720$	$6.0 \times \frac{(15.0 + 9.0)}{2} \times 18 = 1296$
Cut No. 3	$5.5 \times \frac{(25.0 + 15.0)}{2} \times 10 = 1100$	$5.5 \times \frac{(25.0 + 15.0)}{2} \times 18 = 1980$
Cut No. 4	$7.0 \times 25.0 \times 10 = 1750$	$7.0 \times 25.0 \times 18 = 3150$
	3983 cu ft	7169 cu ft

*Metric equivalents: ft × 0.3048 = m; cu ft × 0.02831685 = m^3.

TABLE 21.7 Conditions assumed for the development of belled drawpoints

At each of 9 pockets there will be 4 rounds of hogging

Drill and Blast* Round	For 10-ft Wide Vein			For 18-ft Wide Vein		
	Short Tons	Cycle Time	Cost	Short Tons	Cycle Time	Cost
1	33.6	2.0 hr	$33.29	60.4	3.1 hr	$58.26
2	58.1	1.9	55.49	104.6	3.0	97.10
3	89.5	2.8	88.78	161.1	4.5	155.36
4	142.6	3.8	125.41	256.6	6.4	219.47
Totals	323.8 st	10.5 hr	$302.97	582.7 st	17.0 hr	$530.19

Muck Removal—This is all done by double slush. Approximately one-fourth of the broken material is slushed. The balance of the broken ore is used as a work platform.

Ground Support—Using the equivalent of 1 rockbolt per ft of 7 × 7 ft of intermediate drift, or 49 cu ft, or 4 st and 1 mat every 3 rockbolts.

Miscellaneous—There will be no 2-in., 4-in., or vent pipe. Any such usage will be accounted for under Small Tools and Supplies.

Travel and Lunch—Based on 1½ hr per shift.

Small Tools and Supplies—6% of labor.

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; cu ft × 0.02831685 = m^3; st × 0.9071847 = t.

TABLE 21.8 Summary of hogging drill and blast quantities

	10-ft Wide Vein*	18-ft Wide Vein*
	4 rows of holes from footwall to hanging wall	7 rows of holes from footwall to hanging wall
Round 1	Drill 6.5 ft Resultant height = 5.5 ft % pulled = 85% Volume = 413 cu ft No. of holes = 12	Drill 6.5 ft Resultant height- 5.5 ft % pulled = 85% Volume = 743 cu ft No. of holes = 21
Round 2	Drill 6.5 ft Resultant height = 6 ft % pulled = 92% Volume = 720 cu ft No. of holes = 20	Drill 6.5 ft Resultant height = 6 ft % pulled = 92.3% Volume = 1296 cu ft No. of holes = 35
Round 3	Drill 6.5 ft Resultant height = 5.5 ft % pulled = 85% Volume = 1100 cu ft No. of holes = 32	Drill 6.5 ft Resultant height = 5.5 ft % pulled = 85% Volume = 1980 cu ft No. of holes = 56
Round 4	Drill 8 ft Resultant height = 7 ft % pulled = 88% Volume = 1750 cu ft No. of holes = 40	Drill 8 ft Resultant height = 7 ft % pulled = 88% Volume = 3150 cu ft No. of holes = 70

*Metric equivalents: ft × 0.3048 = m; cu ft × 0.02831685 = m^3.

TABLE 21.9 Summary of drilling and blasting supply costs for hogging a 10-ft wide vein

1st Round*			3rd Round*		
Average width of heading = 7.5 ft			Average width of heading = 20 ft		
Height of heading = 10 ft			Height of heading = 10 ft		
No. of holes = 12			No. of holes = 32		
No. of drills = 1			No. of drills = 2		
St per round = 33.6			St per round = 89.5		
Round advance = 5.5 ft			Round advance = 5.5 ft		
Hole length drilled = 6.5 ft			Hole length drilled = 6.5 ft		
Drilling time = 1.3 hr			Drilling time = 1.73 hr		
Blast time = 42 min			Blast time = 62 min		
Drilling costs = 12 × 6.5 × $0.15 per ft	=	$11.70	Drilling cost = 32 × 6.5 × $0.15	=	$31.20
Blasting costs	=	21.59	Blasting cost	=	57.58
Total	=	$33.29	Total	=	$88.78
2nd Round*			**4th Round***		
Average width of heading = 12 ft			Average width of heading = 25 ft		
Height of heading = 10 ft			Height of heading = 10 ft		
No. of holes = 20			No. of holes = 40		
No. of drills = 2			No. of drills = 2		
St per round = 58.1			St per round = 142.6		
Round advance = 6 ft			Round advance = 7 ft		
Hole length drilled = 6.5 ft			Hole length drilled = 8 ft		
Drilling time = 1.1 hr			Drilling time = 2.67 hr		
Blast time = 50 min			Blast time = 70 min		
Drilling cost = 20 × 6.5 × $0.15	=	$19.50	Drilling cost = 40 × 8 × $0.15	=	$48.00
Blasting cost	=	35.99	Blasting cost	=	77.41
Total	=	$55.49	Total	=	$125.41

Note: Assume drilling costs of $0.15 per ft.

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 21.10 Total costs for the development of belled drawpoints, 10-ft wide vein

1st Round (33.6 st Broken)*	
Drill and blast	2.0 hr
Muck removal† $33.6 \times \frac{1}{4} = 8.4$ st at 6 stph double slush	1.5 hr
Ground support‡ $\frac{33.6}{4} = 8$ rockbolts and 3 mats	2.0 hr
Miscellaneous	1.0 hr
Travel and lunch	1.5 hr
Total	8.0 hr

Costs		Labor, $	Supplies, $
Drill and blast	4 miner-hours	42.00	33.29
Muck removal	3 miner-hours	31.50	
	Double slush at $0.30 per st × 8.4 st		2.52
	Tram at $0.30 per st × 8.4 st		2.52
Ground support	4 miner-hours	42.00	
	Bits, steel at $0.15 per ft		7.20
	8 rockbolts at $2.64 each		21.12
	3 mats at $2.70 each		8.10
Miscellaneous	2 miner-hours	21.00	
Travel and lunch	3 miner-hours	31.50	
Subtotal		168.00	74.75
Total	$242.75 per round, or $7.22 per st		

2nd Round (58.1 st Broken)*	
Drill and blast	2.0 hr
Muck removal† $58.1 \times \frac{1}{4} = 14.5$ st	3.0 hr
Ground support‡ $\frac{58.1}{4} = 15$ rockbolts and 5 mats	3.0 hr
Miscellaneous	1.5 hr
Travel and lunch (for $1\frac{1}{2}$ shifts)	2.5 hr
Total	12.0 hr

Costs		Labor, $	Supplies, $
Drill and blast	4 miner-hours	42.00	55.49
Muck removal	6 miner-hours	63.00	
	Double slush at $0.30 per st × 14.5 st		4.35
	Tram at $0.30 per st × 14.5 st		4.35
Ground support	6 miner-hours	63.00	
	Bits, steel at $0.15 per ft		13.50
	15 rockbolts at $2.64 each		39.60
	5 mats at $2.70 each		13.50
Miscellaneous	3 miner-hours	31.50	
Travel and lunch	5 miner-hours	52.50	
Subtotal		252.00	130.79
Total	$382.79 per round, or $6.59 per st		

(continues)

TABLE 21.10 Total costs for the development of belled drawpoints, 10-ft wide vein *(continued)*

3rd Round (89.5 st Broken)*	
Drill and blast	3.0 hr
Muck removal† $89.5 \times 1/4 = 22.4$ st	4.0 hr
Ground support‡ $\frac{89.5}{4} = 22$ rockbolts and 7 mats	4.0 hr
Miscellaneous	2.0 hr
Travel and lunch (for 2 shifts)	3.0 hr
Total	16.0 hr

Costs		Labor, $	Supplies, $
Drill and blast	6 miner-hours	63.00	88.78
Muck removal	8 miner-hours	84.00	
	Double slush at $0.30 per st × 22.4 st		6.72
	Tram at $0.30 per st × 22.4 st		6.72
Ground support	8 miner-hours	84.00	
	Bits and steel at $0.15 per ft		19.80
	22 rockbolts at $2.64 each		58.08
	5 mats at $2.70 each		13.50
Miscellaneous	4 miner-hours	42.00	
Travel and lunch	6 miner-hours	63.00	
Subtotal		336.00	193.60
Total	$529.60 per round, or $5.92 per st		

4th Round (142.6 st Broken)*	
Drill and blast	4.0 hr
Muck removal† $142.6 \times 1/4 = 35.7$ st	6.0 hr
Ground support‡ $\frac{142.6}{4} = 36$ rockbolts and 12 mats	6.0 hr
Miscellaneous	4.0 hr
Travel and lunch (for 2½ shifts)	4.0 hr
Total	24.0 hr

Costs		Labor, $	Supplies, $
Drill and blast	8 miner-hours	84.00	125.41
Muck removal	12 miner-hours	126.00	
	Double slush at $0.30 per st × 35.7 st		10.71
	Tram at $0.30 per st × 35.7 st		10.71
Ground support	12 miner-hours	126.00	
	Bits and steel at $0.15 per ft		32.40
	36 rockbolts at $2.64 each		95.04
	12 mats at $2.70 each		32.40
Miscellaneous	8 miner-hours	84.00	
Travel and lunch	8 miner-hours	84.00	
Subtotal		504.00	306.67
Total	$810.67 per round, or $5.68 per st		

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

†Assumes only one-fourth of material requires slushing.

‡Assumes 1 rockbolt needed for every 4 st of material removed and 1 mat for every 3 rockbolts.

TABLE 21.11 Summary for hogging 10-ft* wide vein

	Short Tons*	Cycle Time, hr	Cost, $
1st round	33.6	8	242.75 per round
2nd round	58.1	12	382.79
3rd round	89.5	16	529.60
4th round	142.6	24	810.67
Total	323.8	60	1965.81 per drawpoint development
		(elapsed time 7½ days)	
9 pockets × 7.5 days = 68 days			
Assuming 22 working days per month, 68/22 = 3.1 months			

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 21.12 Summary of drilling and blasting costs for stoping*

Method Description†	Short Tons† Broken per Round	Blasting Cost, $ per st	Drilling Cost, $ per st	Total $ per St
Breasting—10-ft vein (jackleg)	51.8	0.74	0.83	1.58
Breasting—18-ft vein (jackleg)	93.2	0.61	0.77	1.38
Back stope—10-ft vein (stoper)	35.2	0.91	0.60	1.51
Back stope—18-ft vein (stoper)	63.4	0.71	0.53	1.24

*These costs take into account prorated lunch and travel times. Costs are based on an 8-hr shift.

†Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 21.13 Drawpoint mucking using LHD units

LHD Capacity = 2 cu yd*	
Haul distance = 600 ft (average)	
Hourly operating cost (including labor)	= $19.40
Calculated production rate round trip time = 3.9 min production rate = 37 stph 75% efficiency	= 28 stph
Mucking cost per st = $\frac{\$ \text{ per hr}}{\text{stph}} = \frac{\$19.40}{28}$	= $0.69 per st
LHD Capacity = 5 cu yd*	
Haul distance = 600 ft (average)	
Hourly operating cost (including labor)	= $41.71
Calculated production rate round trip time = 3.9 min production rate = 87 stph 75% efficiency	= 65 stph
Mucking cost per st = $\frac{\$ \text{ per hr}}{\text{stph}} = \frac{\$41.71}{65}$	= $0.64 per st

*Metric equivalents: ft × 0.3048 = m; cu yd × 0.7645549 = m^3; st × 0.9071847 = t.

TABLE 21.14 Summary of mucking costs during stoping

Mucking Machine Designation*	Labor Operator Cost, $/hr	Machine Operating Cost, $/hr	Machine Ownership Cost, $/hr	Production Rate, stph*	Labor Operator Cost, $/st	Machine Operating Cost, $/st	Machine Ownership Cost, $/st	Total $/st
LHD								
1 cu yd—300-ft haul	10.50	5.62	4.83	17.5	0.60	0.32	0.28	1.20
2 cu yd—600-ft haul	10.50	8.90	5.29	28.0	0.38	0.32	0.19	0.89
5 cu yd—600-ft haul	10.50	31.32	8.21	65.0	0.16	0.48	0.13	0.77
Cavo 511—500 ft haul	10.50	6.70	5.03	22.1	0.48	0.30	0.23	1.01
Slusher 150 ft	10.50	1.05	0.25	10.5	1.00	0.10	0.02	1.12

*Metric equivalents: ft × 0.3048 = m; cu yd × 0.7645549 = m^3; st × 0.9071847 = t.

TABLE 21.15 Ground support for stoping (10-ft wide vein)

Assumptions*

Ground support based on 1 rockbolt per 6 st of ore and 1 mat for every 3 rockbolts
10 ft example round = 104 st

Ground Support Installation*	Labor, $	Supplies, $
Drilling 17 holes × 6 ft per hole = 102 ft 102 ft at $0.15 per ft		15.30
17 rockbolts at $2.64 each		44.88
6 mats at $2.70 each		16.20
Labor 7.4 miner-hours (including travel)	77.54	
Miscellaneous small tools and supplies calculated at 6% of labor		4.65
Subtotal	77.54	81.03
Total	$158.57, or $1.52 per st	

*Metric equivalents: ft × 0.8048 = m; st × 0.9071847 = t.

It should be reemphasized that costs given include labor, direct operating costs of equipment, and expendables. These costs have been allocated to the assigned tonnage for the example stoping block. Equipment capitalization and depreciation have not been included.

As indicated in Table 21.16, development shown in either Fig. 21.1 or Fig. 21.2 is cheaper than the rather elaborate scram and pocket development shown in Fig. 21.3.

Productivities and cycle times for various cases are given in Table 21.17. Several examples have been selected in Table 21.18 to give a comparative range

TABLE 21.16 Shrinkage stoping development cost summary

Example*	18-ft Wide Vein Mechanized Shrink Stope Trackless Development of Haulage with Crosscuts to Vein and Undercutting Stoping Blocks*	18-ft Wide Vein Mechanized Shrink Stope Rail Haulage with Ramp to Scram Drift and Development of Stope with Belled Drawpoints*	10-ft Wide Vein Conventional Shrink Stope with Scram Drift Development in Ore Block and Belled Drawpoints*
Haulage level development	10 × 10-ft bald trackless $71.36 per ft × 300 ft = $21,408	9 × 8-ft bald track Drift $98.25 per ft × 300 ft = $29,475	10 × 10-ft timbered Drift $139.88 per ft × 250 ft = $34,970
Scram drift development		10 × 10-ft bald sublevel $68.45 per ft × 300 ft = $20,535	7 × 7-ft bald intermediate $71.52 per ft × 246 ft = $17,594
Crosscuts to vein	10 × 10-ft bald, 10 each × 30 ft $68.45 per ft × 300 ft = $20,535	10 × 10-ft bald, 10 each × 30 ft $68.45 per ft × 300 ft = $20,535	7 × 7 ft bald, 9 each × 15 ft $72.12 per ft × 135 ft = $9,736
Undercut vein	10 × 18-ft bald sublevel $73.75 per ft × 300 ft = $22,125		
Drawpoint development		L.S. 9 each, $3,109 × 9 = $27,981	L.S. 9 each, $1,966 × 9 = $17,964
Raise development	7 × 7 ft Alimak $82.47 per ft × 191 ft = $15,752	7 × 7 ft Alimak $82.47 per ft × 191 ft = $15,752	7 × 7 ft Alimak $82.47 per ft × 191 ft = $15,752
Orepass and transfer chute development	5-ft diam orepass (shared) $55.10 per ft × 200 ft × 1/3 = $3,673	7 × 7-ft raise to sublevel (shared) $82.47 per ft × 30 ft × 1/3 = $825	7 × 7-ft raise to sublevel $82.47 per ft × 30 ft = $2,474
Chute fronts		L.S. (shared), $3,200 × 1/3 = $1,067	L.S., $3,200 × 1 each = $3,200
Cutout from manways for stope access	10 ft × 10 each $53.21 per ft × 100 ft = $5,321	10 ft × 8 each $53.21 per ft × 80 ft = $4,257	10 ft × 10 each $53.21 per ft × 100 ft = $5,321
Cribbed manway and ramp to sublevel		10 × 10-ft ramp (shared) $71.36 per ft × 150 ft × 1/3 = $3,568	6 × 6 ft cribbed $78.25 per ft × 30 ft = $2,348
Total cost development	$88,814	$123,995	$109,089
Total ore developed	$300 \times 200 \times 18 \text{ ft} \times \frac{162}{2000} = \$87{,}480$	$200 \times 300 \times 18 \text{ ft} \times \frac{162}{2000} = \$87{,}480$	$200 \times 250 \times 10 \text{ ft} \times \frac{162}{2000} = \$40{,}500$
Total cost per st of ore developed	= $1.02	= $1.42	= $2.69

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 21.17 Stope mechanization, vein mining shrinkage stoping, cycle times/productivities

	10-ft* Wide Vein Shrink Stopes with Belled Drawpoints, Good Ground Conditions		18-ft* Wide Vein Mechanized Shrink Stopes with Vein Undercut, Good Ground Conditions		18-ft* Wide Vein Mechanized Shrink Stopes with Belled Drawpoints, Good Ground Conditions	
	A	B	C	D	E	F
Tonnage per round	51.8	35.2	93.2	63.4	93.2	63.4
Drill and blast time†	2.0	1.2	3.0	1.6	3.0	1.6
Mucking time	2.5	0.8	1.7	1.8	0.7	1.1
Ground support installation time	3.7	3.7	6.1	6.1	6.1	6.1
Travel/lunch	0.1	0.1	0.4	0.4	0.2	0.2
Total crew hours	8.3	5.8	11.2	9.9	10.0	9.0
Crew size	3	3	3	3	3	3
Productivity (st* per miner-shift)	16.6	16.2	22.2	17.1	24.8	18.8
Example of productivity calculation (example E):	$93.2 \text{ st per round} \times \frac{\text{8-hr shift}}{\text{10-hr elapsed time cycle}} \times \frac{1}{\text{3 miners in crew}} = 24.8 \text{ st per miner-shift}$					

*Metric equivalents: ft × 0.3043 = m; st × 0.9071847 = t.

†Drill and blast times were taken from the computer output for each specific round.

of stoping unit operation costs for various ground conditions and vein widths. Direct stoping costs for shrinkage range from $5.00 to $8.00/t ($4.57 to $7.25 per st) of ore. Productivity ranges from 14.7 to 22.5 t (16.2 to 24.8 st) per miner-shift.

REFERENCE

Pugh, G.M., and Rasmussen, D.G., 1977, "Stope Mechanization–Vein Mining," U.S. Bureau of Mines HO 26024, August, Dravo Corp.

TABLE 21.18 Shrinkage stopes—costs and productivities*

	Unit Operations	Unit Cost $/st of Ore	Vein Width 10 ft* — Narrow Vein Shrink Stopes with Belled Drawpoints, Good Ground Conditions A	B	Vein Width 18 ft — Mechanized Shrink Stopes with Vein Undercut, Good Ground Conditions C	D	Mechanized Shrink Stopes with Belled Drawpoints, Good Ground Conditions E	F
Development	Development							
	18-ft wide vein (Fig. 21.1)	$1.02	—	—	1.02	1.02	—	—
	18-ft wide vein (Fig. 21.2)	$1.42	—	—	—	—	1.42	1.42
	10-ft wide vein (Fig. 21.3)	$2.69	2.69	2.69	—	—	—	—
	Drill and Blast							
	Breasting—10-ft vein, 51.8 st jackleg	$1.58	—	1.58	—	—	—	—
	Breasting—18-ft vein, 93.2 st jackleg	$1.38	—	—	1.38	—	1.38	—
	Back stope—10-ft vein, 35.2 st stoper	$1.51	.1.51	—	—	—	—	—
	Back stope—18-ft vein, 63.4 st stoper	$1.24	—	—	—	1.24	—	1.24
	Mucking							
	Slusher—25 hp	$1.10	1.10	—	—	—	—	—
	LHD—1 cu yd diesel	$0.92	—	—	—	0.92	—	—
	LHD—2 cu yd diesel	$0.70	—	—	0.70	—	—	0.70
	LHD—5 cu yd diesel	$0.64	—	—	—	—	0.64	—
	Cavo 511 pneumatic	$0.78	—	0.78	—	—	—	—
	Ground Support							
	Rockbolts—10-ft wide vein	$1.52	1.52	1.52	—	—	—	—
	Rockbolts—18-ft wide vein	$1.47	—	—	1.47	1.47	1.47	1.47
Stoping	Stope preparation							
	Cribbed manway—10-ft vein width	$0.43	0.43	0.43	—	—	—	—
	Total Cost per Short Ton		$7.25	$7.00	$4.57	$4.65	$4.91	$4.83
	Stope Crew Size		3	3	3	3	3	3
	Productivity (st per miner-shift)		16.6	16.2	22.2	17.1	24.8	18.8

*Metric equivalents: ft × 0.3048 = m; cu yd × 0.7645549 = m^3; hp × 7456999 = W; st × 0.9071847 = t.

Qualifications

1. 8-hr shift, 6.5 hr effective time at face.
2. Labor rate $10.50/hr, includes bonus and fringe.
3. Unit operation costs include labor.
4. Equipment costs include maintenance, parts and labor, tires, and fuel. Equipment depreciation is not included.
5. All materials are based on 1976 prices.
6. Machine ownership costs excluded.

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CHAPTER 22

Cut-and-Fill Stoping: An Introduction

Joel K. Waterland

GENERAL DESCRIPTION

Open cut-and-fill stoping for many years was probably the most widely used mining method in underground metal mines. Then for a time this method was largely supplanted by the blasthole stope. It again became popular as many mines reached depths or grades where methods requiring large open voids to remain open for extended periods of time became unsuccessful, often as a result of excessive dilution.

The open cut-and-fill method is very flexible and is readily adaptable to almost any orebody. The standard application requires that a slice of ore usually 2.4 to 3 m (8 to 10 ft) thick be removed from the back of the stope, and as the ore is taken down, the back is dressed and rockbolted. After the back is secured, the broken rock is removed through rock passes to the level below. When the rock has been removed, the rock passes are extended upward the height of the ore removed, the stope is backfilled, and another cycle is mined.

This method is best employed in plunging orebodies with considerable vertical extent, ore areas that require selective mining, ore areas where weak wall conditions exist, and orebodies that have an ore value that will carry this relatively expensive mining method. Blasthole stoping, shrinkage stoping, and other mining methods that do not employ rock passes in a stope are not efficient in plunging or flatly dipping orebodies because the footwall makes ore removal quite difficult. Because mining is accomplished by taking down slices of the back, only small areas of the wall rock are exposed at any one time, and these only for short periods. Due to limited back height, areas of uneconomic rock may be left in place, or they may be mined and the material gobbed in the stope. Because the miners in the stope must work under freshly blasted areas, the amount of ground control is usually great. The volume of rock that is broken during one section of mining is relatively small and the amount of nonproductive work required is high. This results in limited productivity for

the stope, and, because of the nonproductive work that must be done on a regular basis, the production from the stope may be quite cyclical.

SUITABLE OREBODIES

The open cut-and-fill method may be adapted to almost any type of orebody with a relatively high vertical extent. The orebody must be accessible at both top and bottom as well as at regular intervals throughout its vertical extent.

Although adaptable to most orebodies, the method is probably best employed where the ore has poor continuity and where most types of bulk mining would produce excessive dilution. In areas of poor ore continuity, the capability of continuous and extensive sampling during the mining of each cycle makes this method very desirable. This capability also minimizes the amount of evaluation sampling that must be done before mining is started. Because of the extractive system used, the size and shape of the stope may be as readily changed as the sampling mandates. Probably the only ore characteristic demanded is that the ore has strength enough to be supported through the use of rockbolts or cable bolts during the mining and backfilling cycles. Good planning, systematic sampling, and careful supervision will produce a product with less dilution than any other open stoping method.

PLANNING

Evaluation Planning

Once it has been decided that the open cut-and-fill method would be the most efficient for mining a particular orebody, the next considerations would probably be the availability of an economical backfill material and the selection of an efficient transport system for this material. Although hydraulically transported mill tailings are the most widely used product, this is not always practical due to mill location or the quality of the tailings. In such cases, backfilling may be used. The type of backfill and the type of equipment used will determine if a floor or capping on the backfill is required to minimize dilution during ore removal.

The early selection of rock removal equipment is important because equipment usually determines the amount of development work required to bring a stope into production and the size of the openings needed. The size and continuity of the orebody will usually determine the type of loading equipment. The use of slushers or load-haul-dump (LHD) equipment captive in the stopes will minimize the amount of development. If the ore continuity is such that a ramp system for extraction can be used, the cost of development will be increased but the flexibility of continuous mining will minimize the cyclical nature of the production.

The height of the mining section usually is determined by the strength of the wall rock and the amount of back bolting required. Once this has been decided, the appropriate drilling equipment can be chosen. The number and sizes of the rock passes employed depend upon the type and size of the extractive equipment and the type of backfill that is to be used. Because the miners

must enter and leave the stope each shift, the level interval is usually maintained at approximately 45 m (150 ft). Access from the level above into the stope must be maintained at all times. The employees perform all the work in the stope, and adequate ventilation must be provided.

Stope Planning

Due to the flexibility of the method and the variability of the ore zones, layout is usually done on a stope basis. In areas where continuity is a problem, the size of the stope is usually determined by the boundary of the ore (with all of the ore within that boundary being removed). In areas of good continuity where ramps are to be used, the length of the stope may be determined by the length of time each of the cycles (preparation, backfill, mining, and ore extraction) requires. The ramp work is then laid out so that access to the various parts of the stope is available to the equipment needed for the various cycles. When rock must be transported, the steepness of the ramp may be up to 0.26 rad (15°). When only entry of equipment is needed, the ramp may have a grade of 0.26 to 0.35 rad (15 to 20°).

The drilling equipment is selected primarily by the height of the ore to be removed in each section and the type of extraction equipment to be used. If the back height is between 2.4 and 4.5 m (8 and 15 ft) usually jackleg drills or very simple jumbo drills are employed. In stopes with a definite plunge, it is usual to have both types of drill available. In ore areas where ramps are used, more sophisticated jumbo drills may be employed. If holes of greater length are required in the back, standard longhole equipment is generally used.

The location of the rock pass system is controlled primarily by the access on the level under the stope and through the evaluation of sampling results. Most rock pass systems are designed to accommodate direction changes should the stope size change. These rock passes may be lined with timber, circular steel, or poured concrete. Regardless of the material used, the wear factor of each type should be assessed according to the estimated tonnage that will be drawn through the opening.

DEVELOPMENT

Production Level

The development required to bring an open cut-and-fill stope into production is minimal if a slusher or captive LHD equipment is used. This usually requires an access entry for each of the drawpoints on the level below the stope. Access raises are usually driven through between the two levels, but in ore areas where this is not possible, crib manways are carried through the backfill. Some means of loading the broken ore from the rock pass into the level haulage system is necessary. In most cases this is done by a chute at the bottom of the rock pass.

Where the ore continuity and ground conditions are such that continuous mining may be used, the ramps are driven in either the footwall or hanging wall of the stope block. The ramps are driven so that the extraction raises may

be installed at the most economical intervals and also so that short access ramps (incline or decline) may be driven into the stope at regular intervals as the mining progresses.

In both cases an adequate haulage system must be developed on the level below the stope so that the ore may be efficiently transported to the shaft.

Access Level

In most cases a level must be developed above the stope to give access to personnel, equipment, and supplies into the stope. In the case of captive equipment in the stope, a level interval of approximately 45 m (150 ft) is desirable. Where ramps are used, the level interval may be increased. The access level also serves as a part of the ventilation circuit.

Access Raise

Because of the entry of personnel into the stope and the need for supplies, some type of access into the stope is necessary at all times. If possible the access raise should be carried from the production level to the access level, and be of adequate size to efficiently pass the equipment in and out of the stope. It must also be large enough to maintain an adequate manway and space for air, water, and fill lines, as well as a power line if electrical power is needed. This raise should also serve as part of the ventilation circuit. When using the ramp system, access raises may not be used, but care should be taken in the original planning to ensure that adequate ventilation and backfill capacity can be maintained without them.

If crib raises are to be carried up through the backfill, then the problems of ventilation and backfilling should be carefully studied before mining starts.

DRILLING

Breast mining and the drilling of "uppers" are the two most common practices in mining by the cut-and-fill method. The breast-mined stope is practical where poor wall conditions exist. The stope is backfilled to within 0.9 or 1.2 m (3 or 4 ft) of the back, and mining proceeds by drilling horizontal holes into the face and breaking the ore down into this void. The broken ore is then removed from this area and the back is rockbolted. LHD or overshot loaders are best suited for rock extraction using this method, as the miners are not exposed to uncontrolled ground and a slusher does not remove ore from the face as efficiently. The section of ore removed is usually 3 to 3.7 m (10 to 12 ft) in height. A more continuous flow of ore is achieved using breast mining than "uppers" drilling.

The "uppers" holes are usually drilled using a simple stope jumbo or a jackleg drill after the stope has been backfilled to within 2.4 to 3 m (8 to 10 ft) of the back. A "V" or toe cut is drilled and blasted across the stope, usually at one end. The remaining holes are drilled into the back at an angle of approximately 1.04 rad (60°). These holes [normally 3 to 4.5 m (10 to 15 ft) in length] are drilled in rows across the stope with the burden and spacing of the

holes determined by the ground to be broken. The complete back of the stope is drilled in this manner; if any ore is left on the periphery of the stope from the last section mined, it is slabbed. Any "uppers" holes that must be drilled over the area where the slabbing was done are drilled with a jackleg. The back is loaded and blasted in relatively large blasts, and the back is barred and bolted using the broken ore as a work platform. A slusher works well with this method because in many cases, for efficient ground control, broken ore must be removed from the top of the pile.

A system of drilling long [up to 15 m (50 ft)] "uppers" holes into the back has proven successful in long narrow orebodies with strong wall rock. The extraction is made with small electric LHD or overshot loading units, which can be remotely controlled. With this system, lost time because of ground control is minimized. The drilling in this method is done with conventional longhole drilling equipment and sectional steel.

VENTILATION

As in all underground mining methods, a supply of clean air must be introduced into the working area. This air is supplied by small 3.7 to 11 kW (5 to 15 hp) auxiliary fans that force the air through vent lines into the stope via the access raise. The air then exhausts through the access raise and into the exhaust system. Where electric- or pneumatic-powered equipment is used, the required air volume is relatively small. If diesel equipment is used, the quantity of air is mandated by state or federal regulations and is calculated by the amount of horsepower used.

ADVANTAGES AND DISADVANTAGES

The advantages of the open cut-and-fill method are (1) the ore block is continuously and extensively sampled as it is being mined; (2) development is minimal before mining is started; (3) selective mining can be used to reduce dilution; (4) its versatility makes it possible to efficiently mine plunging orebodies; (5) the openings are small, minimizing wall dilution; (6) a change to another method can be made readily; (7) equipment investment is relatively small; (8) and large-scale ground movement is kept to a minimum.

The disadvantages are (1) ore production is cyclical; (2) the method is labor-intensive and requires skilled miners; (3) it is not as suited to mechanization as other methods, so there is lower productivity; (4) the personnel must work under freshly blasted ground, which creates a safety problem; (5) the degree of ground control required is high; (6) proper ventilation is difficult and expensive; and (7) a high-grade orebody is required, so costs are higher.

DATA SUMMARY

A summary of data from some mines using the open cut-and-fill method is presented in Table 22.1.

TABLE 22.1 Summary of the open cut-and-fill method

Mine	Tons Yearly Production*	Stope Access	Type Drilling	Height Panel*	Drilling Equipment	Loading Equipment*	Efficiency Tons/ Miner-shift	Sill Haulage
Mount Isa Mine, Australia	845,000	Footwall raise	Uppers Breast	12 ft 12 ft	2-boom jumbo jackleg	5 yd LHD 5 yd LHD	130 130	Rail Rail
New Broken Hill Consol. Ltd., Australia	1,838,130	Raise	Uppers	12 ft	Crawler-mounted drifters	5 yd LHD	7	Rail
Hecla Mining Co. Star Mine	282,000	Crib raises	Uppers Breast	5–10 ft	Jacklegs Stopers	Slusher	20	Rail
Lucky Friday Mine	173,000	Raise (steel)	Uppers Breast	8 ft	Jacklegs Stopers	Slusher Cavo 310	13	Rail
Cyprus Mines Corp. Bruce Mine	90,000	Ramp raise	Breast Breast	7–12 ft	2-boom jumbo jackleg	2 yd LHD Slusher	13.2 7.8	LHD LHD
Homestake Mining Co. Homestake Mine	1,700,000	Raise	Uppers	10 ft	Stopers jumbo jackleg	Slusher	15.74	Rail
Dome Mines	708,000	Raise	Uppers Breast	10–50 ft	3-boom jumbo jackleg Wagon drill	Slusher LHD CAVO Loader		Rail

*Metric equivalents: ft × 0.3048 = m; yd × 0.9144 = m; st × 0.9071847 = t.

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CHAPTER 23

Principles of Stope Planning and Layout for Ground Control

P.A. MacMillian and B.A. Ferguson

INTRODUCTION

Jack Spalding in *Deep Mining*, Chapter 3, states:

> *In deep mining, to start stoping an orebody without a definite plan of operations covering the whole extraction from beginning to end is to invite serious trouble. Any stoping without plan is bound to leave, toward the end, a number of pillars, remnants, or promontories, which, as they are reduced in size, are liable to fail and cause a general collapse. The object of stope planning is so to mine an orebody that at no stage in the operation is a remnant left. Definition: "When a block of ore is stoped in such a way that eventually a small piece is left entirely surrounded by stoping, that piece is termed an island remnant." The design of and adherence to a plan of stoping is of greater importance in preventing rock bursts than the type of ground control adopted.*

This, of course, is just an example of the old axiom that an ounce of prevention is worth a pound of cure. And in this context, mining does not have to be very deep before poor ground conditions, made worse by lack of good planning, force a mining company to resort to more expensive methods of ore recovery. At all times when rock pressures become excessive, stoping plans must be devised to avoid the creation of pillars or promontories.

This is most readily recognized when mining deep narrow ore veins by open methods. It would be a little more general to say that stoping systems should seek to avoid or minimize localized concentrations of rock stress. Orebodies, and particularly those at Falconbridge nickel mines, can reach large widths, and leaving pillars for extraction at a later time cannot be avoided. However, good planning can reduce the effect of factors that create bad ground conditions. The necessary ingredients of planning are time and information: time to formulate and revise ideas; information, gained in exploration and early development, to provide the fullest knowledge of the orebody. In this way, plans become meaningful. It must always be kept in mind that the long-term

results of good planning often require decisions not compatible with reaching earliest full production.

PLANNING FOR DEEP MINING

At Falconbridge, orebodies being found and prepared for production lie at depths in excess of 610 m (2000 ft) below surface. Underground exploration is being directed toward favorable locations below 914.4- and 1219.2-m (3000- and 4000-ft) depths, at most mines. The Onaping Deep orebody has been traced to a depth of 1508.8 m (4950 ft), and at Falconbridge mines the orebody is under development below the 5000 level. Planning for more efficient ground control has therefore become an important consideration.

HYDRAULIC BACKFILLING AND CUT-AND-FILL STOPING

The advent of hydraulic backfilling has led to the increasing use of flat-back cut-and-fill stoping methods until it has probably become the commonest method of mining deep ore deposits in Canada. These methods account for almost all the production from Falconbridge mines at this time. The advantages claimed are (1) safety—a minimum of open ground; (2) very little supplementary ground support (except rockbolted backs and walls); (3) greater selectivity and flexibility in extraction (horizontal extension of the vein, high-grade ore stringers going out into the walls); and (4) high productivity and increased mechanization in larger ore widths.

There is some difference of opinion as to whether hydraulic classified tailing backfill provides any great resistance to initial closure of the stope walls. The evidence available suggests that it does not. Addition of portland cement to the backfill just prior to placement has produced a material having the properties of a weak concrete. It is confidently expected that this material will assist greatly in the efficient mining of pillars and generally in giving increased ground support to oppose closure.

PRINCIPLES OF STOPE PLANNING BY LONGITUDINAL FLAT-BACK CUT-AND-FILL

Again quoting from Spalding:

> *The practice of controlling output by stopping and starting stopes is bad—stopes once started should proceed without interruption, and control must therefore be obtained by altering the rate of stoping. It is therefore necessary that faces should normally advance at a medium rate which can be boosted, if required, to give temporarily a greater output. Modern high productivity stoping methods, however, generally aim to achieve and maintain the maximum output possible from a stope. Control of tonnage and grade, therefore, becomes a more complex problem requiring very careful scheduling and organization. With the basic method of flat-back overhand stoping using hydraulic fill, it is not usually possible to mine in such a way as to avoid creating pillars.*

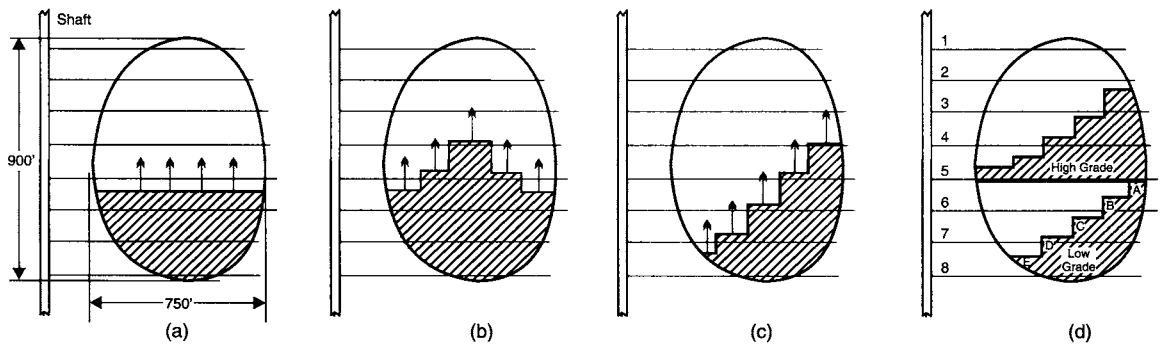


FIGURE 23.1 Stope planning for longitudinal flat-back cut-and-fill. The ore zone is 274 m (900 ft) high, 228 m (750 ft) on strike, by 4.5 m (15 ft) wide, representing 907,000 t (1,000,000 st). (a) Advance one long single face through all levels from the bottom to the top in five stopes, each 45 m (150 ft) long. (b) Hanging wall failure would be reduced by staggering individual stope faces. (c) A sequence of retreating toward the shaft reduces travelway maintenance but takes longer to achieve full production. (d) Mining from several horizons creates problems of sill pillars.

Consider an orebody as illustrated in Fig. 23.1. It is 274.3 m (900 ft) high by 228.6 m (750 ft) long on strike and averages 4.6 m (15 ft) in ore width. It represents approximately 907,000 t (1,000,000 st).

The orebody could be mined by advancing one long single face through all the levels from bottom to top. A total face length of 228.6 m (750 ft) would accommodate five 45.7-m (150-ft) stopes (see Fig. 23.1a). Each of these stopes would produce approximately 1814 t/m (2000 stpm), and the orebody would require a period of 10 years for complete extraction. A possible improvement to this scheme might be to stagger the individual stope faces (see Fig. 23.1b). Problems of hanging wall failure can be reduced by this arrangement.

From the point of view of reducing travelway maintenance, it is good practice to employ a stoping system of retreat. Stopes farthest from the shaft are mined first so that on completion, haulage drifts below can be abandoned (see Fig. 23.1c). However, this alternative, compared with that in Fig. 23.1b will take longer to reach full production. That is, at 1.5 years per level advance, an extra three years are required to reach full production of 9070 t/m (10,000 stpm). This basic scheme of advancing one single long face or series of staggered faces may be unacceptable for two reasons: the total rate of production may be too low, and if a large variation in grade between various parts of the orebody exists, it may cause too great a fluctuation in grade over the life of the orebody, or too low an initial grade.

These circumstances have occurred in the planning of cut-and-fill stoping at the Strathcona mine. The only solution to this problem is to commence mining from more than one elevation.

This solution, however, creates the further problem in that a sill pillar is created immediately below the second horizon (see Fig. 23.1d). If this pillar is of relatively high grade (it would be if the second mining horizon were created

because of the second reason given previously), then extraction of the pillar will be economic. At some point before reaching level 5 in Fig. 23.1d, cut-and-fill mining will become unsuitable because of crushed ground, and the remaining sill pillar will have to be removed by a modified mining method such as square-set stoping or undercut-and-fill.

The question now arises as to the best timing for mining this pillar. There appear to be two alternatives: complete the mining of stopes A, B, C, D, and E and extract the pillars above them as convenient; and complete the extraction of each individual section of the pillar before the adjacent stope reaches the pillar elevation.

The second alternative may be the only practical method if ground closure is severe (see Fig. 23.1d). For this system of operation, the shorter the stope length the better. The size of the pillar would be a function of several factors. Obviously, under heavy rock stress conditions, the height of the pillar is extremely important. A height of twice the stope width is often used. The most common method of sill pillar extraction is by square-set retreat, mining a vertical face the full height of the pillar. For proper scheduling, it may be necessary to retard the rate of mining in stope B during the period of extraction of pillar A. The necessity of maintaining sequence and the low rate of extraction possible from pillar recovery constitutes a limiting factor on pillar size.

GENERAL RULES FOR LAYING OUT A BLOCK OF LONGITUDINAL STOPES

1. Establish as few pillars as possible. Commence mining on alternate levels or even farther apart, if possible, to eliminate sill pillars. Limiting factors include maximum production rate consistent with safety, irregularity in the orebody, depth of shaft available for establishing the mining horizons, and/or a financial or time limitation. Attempt to extract 100% of the orebody safely at lowest overall cost.
2. Take advantage of barren or submarginal areas of the orebody for establishing pillars that may not be reclaimed. In dipping orebodies, the rib pillar affords more support than a sill pillar of equal volume. The long axis of the pillar should be parallel to the principal stress direction, which in many cases may be the downdip direction. (This may also be true of room-and-pillar mining in flat-lying orebodies.)
3. Attempt to maintain a sequence of retreat to avoid high-level maintenance under bad ground conditions and high mining service costs.
4. Simulate position of stope faces throughout the period of extraction of the orebody. It may be wise to start a mining sequence at the lowest possible elevation.
5. Schedule the extraction of pillars as early as possible to avoid excessive deterioration through crushing, and to avoid deferring all high-cost pillar or remnant mining to the final operation.
6. Correlate geological structures with stope and overall mine geometry.

DEVELOPMENT OF THE STOPING BLOCK

It is necessary to diamond drill the ore formation completely and establish the scope and position of the whole orebody. Unforeseen complications in mining offshoots or irregularities to the main orebody, which could destroy the schedule, are thereby avoided. Diamond drilling is also able to provide information on the magnitude and nature of inherent rock stresses. This may have a considerable influence on the mining method adopted and the direction and position of stope development (i.e., access crosscuts, service raises, etc.).

The establishment of a footwall haulage drift parallel to and some distance from the ore has become standard at Falconbridge for the flatter dipping orebodies. Ore is loaded from chutes, on the footwall drift, which are fed by boxholes or ore transfer raises connected to stope mill holes. The advantage of this haulage position is that it is remote from the orebody and remains in more stable ground throughout the life of the orebody and is much cheaper to maintain. It also provides a base for the detailed diamond drilling referred to earlier. If there is a second orebody to be mined farther along strike, the footwall haulage is a better means of access than through a timbered drift in a mined-out area or a drift in a disturbed hanging wall. In other words, there is less interference of stope preparation and sill mining to production.

THE MINING LAYOUT OF WIDER OREBODIES

When the orebody reaches widths in excess of 9.1 m (30 ft) from hanging wall to footwall, stoping by simple longitudinal cut-and-fill methods becomes difficult and often impossible. The determining factor will generally be the ability of the ore back to support itself over the width. This must be accomplished by the ore acting as a beam. This "beam action" can be assisted by patterned rockbolting of the backs, and by forming an "archy" shape to the back during mining. The arch, although assisting back support, does add to the difficulties of mining in very wide ore as the distance between center back and the fill elevation becomes excessive. Further, if the junction between the ore and the hanging wall is characterized by slips and faults, a dangerous condition exists, especially where the ore is flat dipping. The orebody must then be divided into transverse stopes and intervening rib pillars, as shown in Fig. 23.2.

During the mining of the transverse stopes, the rib pillars serve mainly to provide abutments for the stope back and give some overall support of the hanging wall. The rib pillars shown, say 6.1 m (20 ft) wide, provide partial support or resistance to closure between hanging wall and footwall, and become crushed before their extraction can take place. It must be pointed out that the crushed pillars are confined with fill and therefore support from them is better than many engineers realize.

Sill pillar extraction can be difficult if ore width and stope and pillar width are both large. Pillar support in some types of ground must yield to minimize the danger of stress buildup and violent failure. Extraction in this case would be by square-set stoping, mining the back in horizontal cuts reducing pillar height to, say, three sets 5.5 m (18 ft), and then retreating from hanging wall to footwall in a way similar to the method described in pillar recovery for

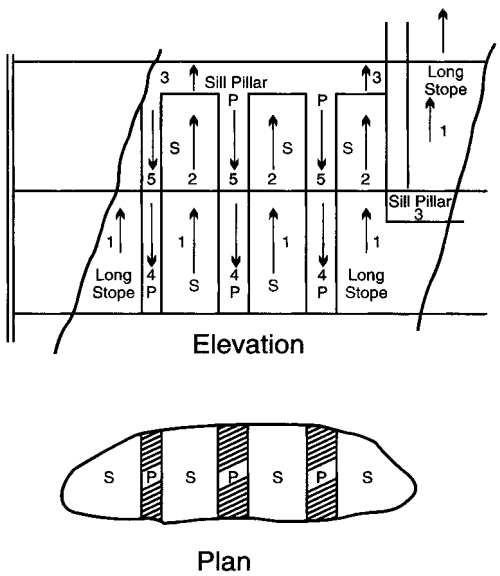


FIGURE 23.2 Stope and pillar layout at Strathcona mine for a cut-and-fill mining block between the 1700 and 2700 levels. The numbers represent the sequence of mining.

longitudinal stopes. Undercut-and-fill stoping can also be used for narrow pillar widths. In many cases preparation costs for undercut-and-fill are prohibitive for extracting shallow sills.

Rib pillar extraction is by undercut-and-fill working downward in the sequence shown in Fig. 23.2. This method is most efficient when the pillar width is less than 6.1 m (20 ft).

Transverse stopes and pillars have been used successfully for mining the Hardy mine orebody lying between the surface and the 1000 level. Even so, hydraulic sand fill was not able to prevent hanging wall subsidence of the order of 0.6 m (2 ft). This closure affected the hanging wall rocks and mine workings in the hanging wall for a distance of several hundred feet from the ore.

For this method to be used successfully at depth, or to allow mining of a parallel orebody in the hanging wall, closure of the hanging wall must be greatly reduced. The answer must therefore lie in the properties of the fill material used. Hydraulic tailings fill loses volume after complete drainage or drying out takes place. Ideally, the fill material should be such that it can minimize closure and take up the full load previously carried by the extracted ore with very little closure.

The addition of portland cement to tailings backfill is rapidly becoming standard practice at Falconbridge mines. In proportions down to 30:1 it produces a weak grade of concrete having some compressive strength.

In the cut-and-fill mining block at Strathcona mine, situated between the 1700 level and the 2700 level, a stope and pillar layout similar to Fig. 23.2 (although over a much greater strike length) has been chosen. Sill pillar

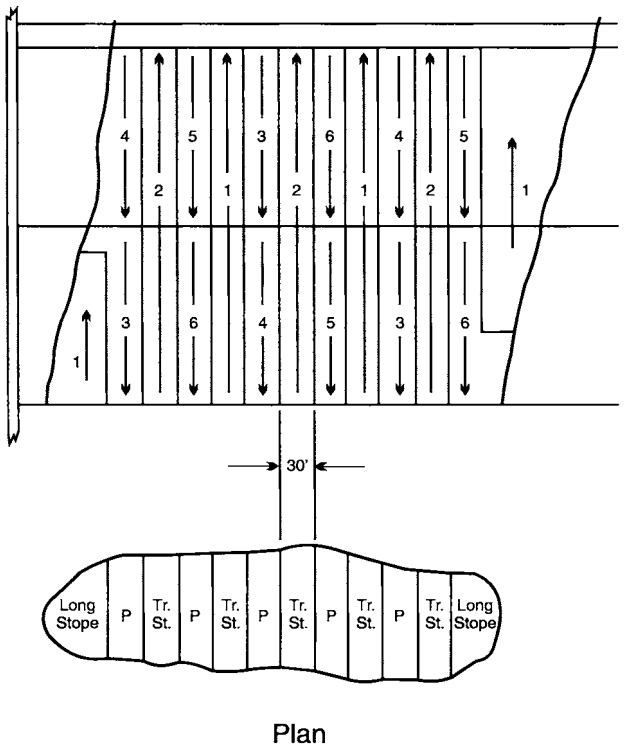


FIGURE 23.3 Mining sequence using alternate stopes and pillars, each 9 m (30 ft) wide, in ore 24 m (80 ft) wide

intervals are from three to five levels, 114.3 to 190.5 m (375 to 625 ft). Cement tailings backfill will be used exclusively, and as a result ground conditions are expected to be better during mining of this orebody.

If the orebody is situated in highly stressed rocks, this support system may be insufficient to prevent excessive closure. A rock mechanics study of diamond drill cores taken from that part of the Onaping Deep orebody lying between the 4250 level and the 4950 level reveals that the orebody is situated in competent but highly stressed granite. The ore is of such width that transverse cut-and-fill stoping must be used in some sections.

What may be required for the Onaping Deep is to provide support by both pillars and consolidated fill. In the last example (Fig. 23.2), the rib pillars served as abutments for the stope arch and provided limited hanging wall support (function of pillar size and percent coverage of hanging wall on strike and dip). In this case they give full support of stress from the walls. The pillar is wider than the width of the orebody. In Fig. 23.3, it is assumed that fill provides no support, the ore is shown approximately 24.4 m (80 ft) wide, and therefore pillar width must be in excess of this. If the orebody were to be divided along strike into a series of alternating stopes and pillars each 9.1 m (30 ft) wide, and stoping commenced and completed in alternate stopes (Fig. 23.3), this condition would be initially met. The strength resisting

closure would then be provided by 75% of the original rock, plus consolidated fill in the completed stopes. The intervening stopes are now mined and resistance to closure is now reduced to 50% of the original rock plus consolidated fill in the two intervening stopes. This resistance to closure may be more than sufficient to prevent excessive crushing of the pillars and to allow sequential extraction of these pillars by undercut-and-fill. It is important that strict sequencing be maintained throughout the whole extraction process in order that the total stress load be distributed evenly over the orebody. This will determine the number of operating stopes at any one time. Unfortunately this approach may create bursting pillars. Over 50% of the orebody may be subject to high-cost secondary and tertiary mining. Primary stoping costs are also higher initially due to the cement cost for consolidation up to first pillar extraction.

ROLE OF ROCK MECHANICS IN STOPE PLANNING

To date, stope and pillar dimensions are determined arbitrarily, based on those factors of mining efficiency already discussed and on past experience. There is no reason why these criteria cannot eventually be supplemented by the calculated results of rock strength and geological structure studies.

The magnitude of the stress perpendicular to the dip and strike of the orebody that induces closure can be measured quantitatively, and the compressive strengths of the ore, rock, and consolidated fill can also be measured. An appropriate theory for calculation of pillar area will be developed and incorporated into the stoping layout. Advance knowledge of rock stress conditions, geological structure, and previous experience of behavioral aspects of known rock types under stress can play a major part in designing a suitable mining method or in modifying methods presently being used. In the Onaping mine, the main orebody lies in competent granite on the footwall side of the granite-norite contact with which all ore occurrences are associated. However, in the upper section of the orebody (between the 2300 level and 3050 level), the ore is in contact at the hanging wall with incompetent quartz-diorites. These rocks are characterized by slips and faults running roughly parallel to the ore. Deep exploration drilling has confirmed that these conditions persist to the 4250 horizon. The ore is relatively narrow [3 m (10 ft)] and short [76.2 m (250 ft)]. Rock mechanics studies reveal high stress concentrations in these rocks, and findings to date will influence the development layout and mining method adopted for this orebody.

Geological structure probably yields the most important and useful parameters for mine design, such as the orientation of stopes, pillars, and large underground openings. Weakness planes in the orebody itself in many cases lead operators, in retrospect, to realize that another mining direction might have made the stopes, support pillars, surrounding waste confinements, and artificial support media more effective and produced safer mining conditions. Definition of weakness planes in many cases helps the mine operator to modify his mining direction or procedure for safer conditions.

It becomes apparent that planning for effective ground control generally results in a less flexible sequence of extraction, closely scheduled and controlled. Limitations are placed on the initial levels of production, which are

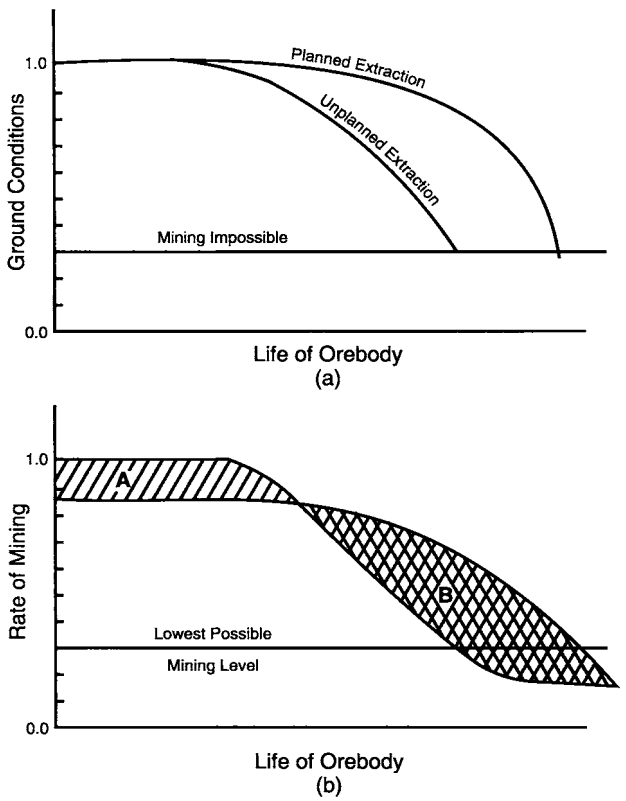


FIGURE 23.4 Ground conditions and rate of mining versus the life of the orebody. In (a), with planned extractio ground conditions remain good for a longer period and never deteriorate to make mining extremely difficult and hazardous. In (b), area B is greater than area A. Total extraction is greater with a method based on planned extraction at 85% of the possible rate.

sometimes difficult to justify when set against immediate ore requirements. The second effect of this type of planning is really an extension of the first. The concentration of operating stopes at any one horizon may be reduced. This means that, for the same initial total production, the amount of preproduction lateral development required from the shaft is correspondingly increased, because the number of operating levels must be increased.

When mining depth becomes extreme, the measures outlined in this chapter become vital to successful mining of the orebody. At shallower depths the question becomes an economic one and a little more difficult to define. What may be recognized as a superior mining method involving a lower total direct cost of extraction may not be the most economical method if it results in a lower rate of extraction, higher investment costs in development, and less flexibility. Of course, it is not always easy to predict how seriously ground conditions will deteriorate if a certain course of action is followed, and the natural tendency is to be optimistic. Bitter experience usually comes too late for corrective measures to be effective. Figure 23.4 may serve to illustrate the problem graphically.

In conclusion, realistic appraisal should always be made and ample margin be allowed for poor or deteriorating ground conditions. Mine design should have flexibility for alternative methods to accommodate change. Most important, initial mining plans must cover mining schedules for the complete extraction of the orebody under consideration.

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CHAPTER 24

Cut-and-Fill at Mount Isa Mines Ltd.

B. Hornsby and Staff, Mount Isa Mines

At the Mount Isa mines, Mount Isa, Queensland, Australia, sublevel open-stope mining is used for copper ore, whereas both sublevel open-stope and cut-and-fill methods are used for the silver-lead-zinc ores. Production in 1979–80 was 4,730,000 t (5,200,000 st) copper ore and 2,590,000 t (2,850,000 st) of lead ore. The deepest mine level currently is 960 m below the surface.

GENERAL OREBODY REQUIREMENTS AND LIMITATIONS

Size, Shape, and Dip

The mine is located in the valley of the Leichhardt River in Lower Proterozoic sediments of the Mount Isa group. The regional strike of the sediments is north-south, and they have a persistent westerly dip of about 1.1 rad (65°). In the mine area, the Mount Isa group comprises a sequence of alternating bands of dolomitic shales and siltstones some 3.2 km (2 miles) in width. It is bounded on the east by greenstones and quartzose sediments of the Eastern Creek volcanics, and on the west by the Mount Isa fault, west of which occur schists and other metamorphic rocks.

The orebodies as currently known at Mount Isa are restricted to the Urquhart shale, a formation of thinly bedded pyritic, dolomitic, and volcanic shales some 1070 m (3500 ft) thick. Toward the top of the Mount Isa group sediments, copper and silver-lead-zinc orebodies occur as contiguous but discrete entities and are mined and treated separately (see Figs. 24.1 and 24.2). Argentiferous galena, sphalerite, pyrite, and minor associated sulfides occur in distinct concordant bands throughout the Urquhart shales. Wherever these bands are grouped together in sufficient density, they constitute silver-lead-zinc orebodies with widths varying up to 36.6 m (120 ft). Other mineralized bands are common in intervening beds. Individual orebodies persist for many hundreds of meters concordant with the strike and dip of the enclosing shales.

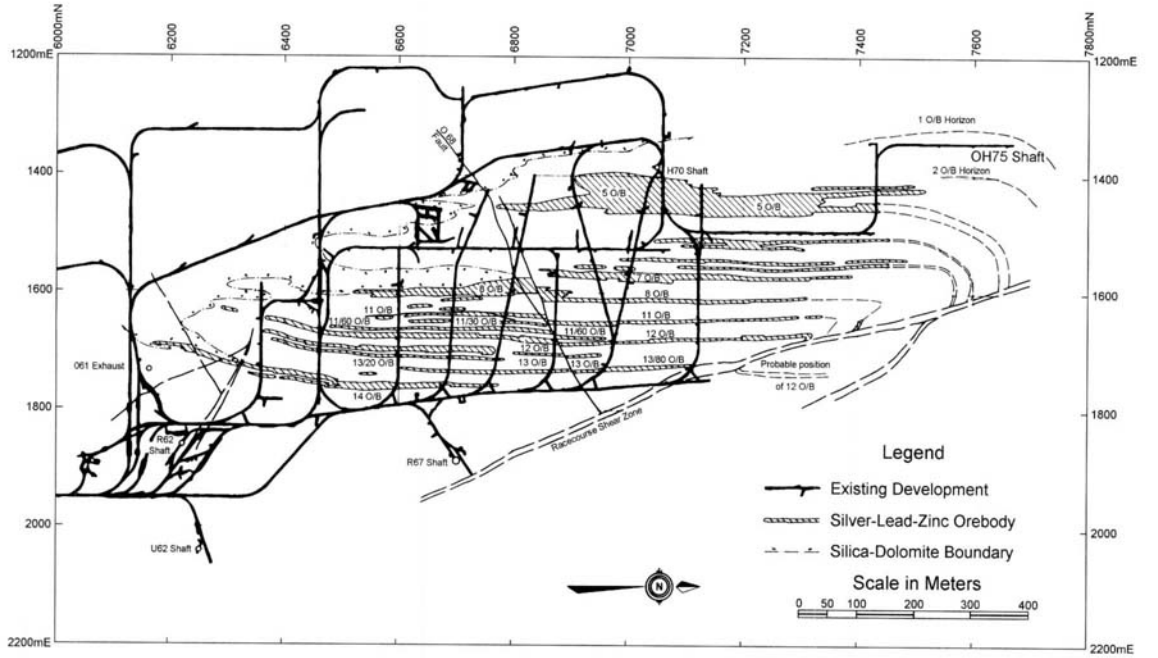


FIGURE 24.1 Plan view of 15 level

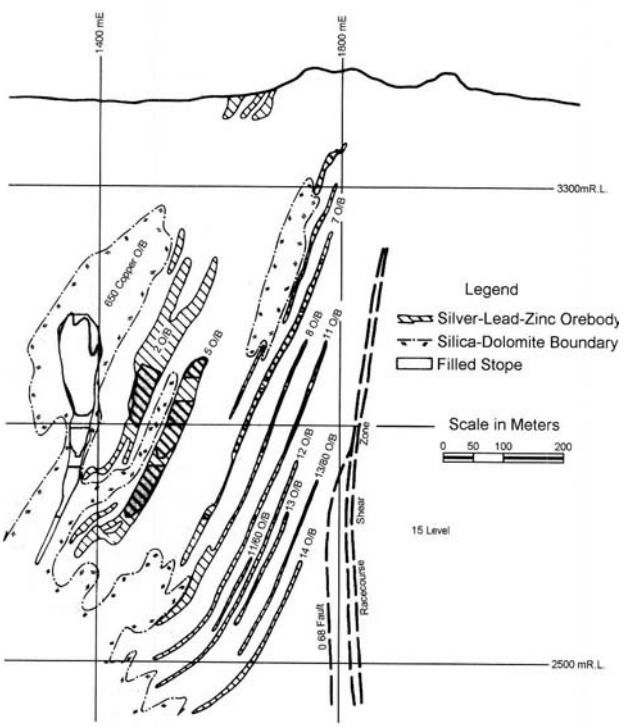


FIGURE 24.2 Cross section of copper and silver-lead-zinc orebodies (looking north)

Intersections of these beds with fold axes provide local zones of enrichment, but otherwise the mineral content of the orebodies varies only slightly throughout the extent of the dip and strike.

Chalcopyrite is the only important primary copper mineral. It occurs in association with pyrite and pyrrhotite as disseminated flocks and as vein fillings throughout irregularly shaped zones of folded, brecciated, and recrystallized Urquhart shale. These zones, which are locally termed "silica-dolomite" bodies, are broadly transgressive to the shale bedding and confine all significant copper mineralization to the virtual exclusion of silver-lead-zinc mineralization. In some areas, the downward percolation of acidic water through faults and shears has caused extensive leaching of the coarse-grained carbonates to depths in excess of 610 m (2000 ft), but most of the silica-dolomite remains unleached and the ground is competent.

The majority of lead ore being mined is north of the main shaft pillar (R62, U62) area. The major lead-producing areas for many years have been the Black Star orebodies (2 and 5 orebodies) and the Racecourse orebodies (6 to 14 orebodies), each area contributing about half the total silver-lead-zinc ore.

Lead ore production originates from 5/110 to 14 orebodies between 11 and 13 levels (mechanized cut-and-fill, or MICAF); 2 orebody open stopes between 10 and 12 levels; 5, 6, 7, and 8 orebody open stopes between 13 and 15 levels; and 5/110 to 14 orebodies (modified MICAF) between 13 and 15 levels.

Ground Condition, Ore Value, and Production Requirements

The ground is fairly competent, but requires extensive rockbolting for secondary support.

The ore values are 6.5 zinc by weight, 6.4 lead by weight, and 150 g/t (4.5 g per st) silver.

Production from cut-and-fill was 1,210,000 t (1,330,000 st) in 1979–80.

REASONS FOR ADOPTING THE CUT-AND-FILL METHOD

Mechanized cut-and-fill is a method that has been used for nearly all mining of the long, narrow, near-parallel and closely spaced Racecourse lead orebodies below 9 level. Among the reasons for adopting the MICAF method for the silver-lead-zinc orebodies are its suitability for selective extraction of ore, ability to cope with poor wall rock conditions, high production rate, and continuity of production. Cut-and-fill is used in the footwall orebodies, whose hanging walls are, in general, not as competent as the orebodies in the hanging wall of the sequence, and whose widths are less. The method is restricted to a maximum horizontal mining width of 11 m for ground control and a minimum width of 3 m because of the size of equipment used.

In the wider orebodies, sublevel open stoping is used. Development and stope preparation time may be longer than with cut-and-fill, but, with effective activity scheduling, production continuity can be maintained.

DESCRIPTION OF THE MINING METHOD

Two forms of cut-and-fill mining are practiced at Mount Isa. The older conventional MICAF relies on only vertical access. This method is gradually being phased out. The more modern modified MICAF involves the use of footwall declines, permitting ready access to all the orebodies by mobile equipment.

Both methods are described here, but very little conventional MICAF is now done. In both, uphole and horizontal hole drilling techniques are employed. Because of the ground conditions, horizontal holes (flat-backing) are used most.

In flat-backing, the height of exposed hanging wall can be limited to a safe amount; miners are working under safe, rockbolted backs; and any ground problems in the face or hanging wall can be dealt with more safely and effectively than with uphole blasting.

Development

Stope Division. Where possible, stopes are divided into two equal sections on a tonnage basis. One of these sections is north and the other south of the balance point. However, in several instances there is a third smaller section (a separate orebody), which proves more economical to mine with the two larger sections than by itself. This procedure increases production efficiency, as it can provide ore in the nonproducing periods of the major sections. Where possible, the orebodies are grouped to give a production of 12,000 to 18,000 t (13,228 to 19,842 st) per unit stope. This balances the total mining cycle and reduces the overall number of orepasses required.

Stope Access. Access to Mount Isa cut-and-fill stopes is through raises located right on the footwall of each orebody and inclined at approximately the same dip as the orebody (Fig. 24.3).

Conventional raising (mostly for short raises), an Alimak raise climber, and a Robbins 61R raise borer are used for developing access raises.

Where necessary, wall support is provided by rockbolting and meshing the raises. Main service raises are located close to the stope balance point and are 3 × 2.4 m (10 × 8 ft) in cross section. They are of sufficient size that a diesel load-haul-dump (LHD) unit (Wagner ST5A) can, by being broken down into three sections, be lowered into, or removed from, the stope. The raise is equipped with rail skids and a skip operated by an Atlas Copco MHK 82 or 61 air winch. This is used for lowering stores and equipment from the access level to the stope and has an operating capacity of 1 t (1.1 st) net weight. Larger capacity winches and ropes are rigged when required. A ladderway, air, water, fill, and fuel lines, as well as telephone, welding, and firing cables are installed in the raise. The top of the serviceway is decked with a steel stage with trap doors. Storage facilities with rail access are provided for operating stores.

Secondary accesses, generally 1.8-m (6-ft)-diam bored raises, are located midway between these service raises and the ends of the stopes to provide

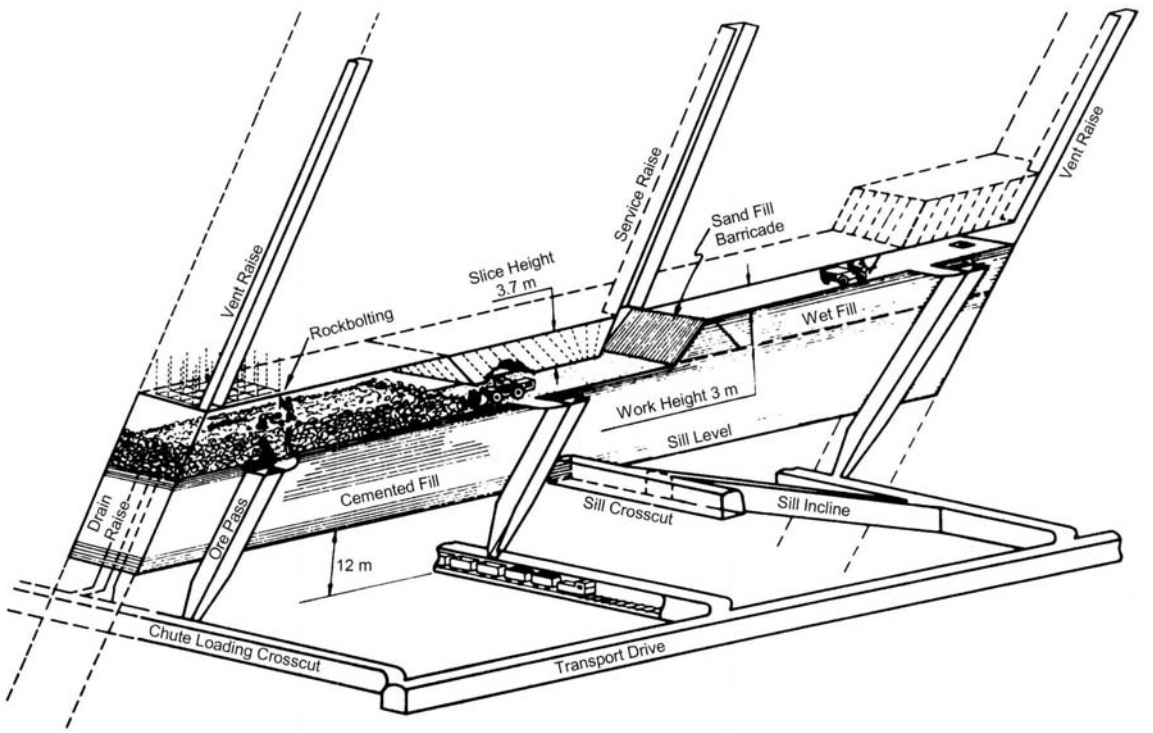


FIGURE 24.3 Diagrammatic layout of MICAF uphole stope

more flexible entry. They are equipped with ladders and pipe skids to allow mining and rockbolting to be started from them.

Permanent fuel stores are built on the main level above each stope and fuel lines run down the raises to provide easy refueling facilities for the diesel units in the stopes. Fuel stores contain a measuring drum to restrict the amount of fuel fed to the unit and to avoid diesel spillages in the stopes.

Stope Preparation. *Sill Development*—Each 2-level lift is started 12 m (39 ft) above the haulage level, which simplifies haulage layout and eliminates orebody intersections with existing development. It also provides some storage in the orepasses at the start of production.

Initial Stoping. Mining is started by taking vertical lifts above the strike drives. These vary in height, depending on the stoping method. During the first three vertical slices, stripping is done horizontally along the footwall so that the slices eventually cover the full width of the orebody or reach a maximum width of 11 m (36 ft). The fill in these initial lifts can contain 10% portland cement to provide a solid fill at the base of the stope and so permit subsequent floor pillar recovery from below with minimum dilution from sand fill.

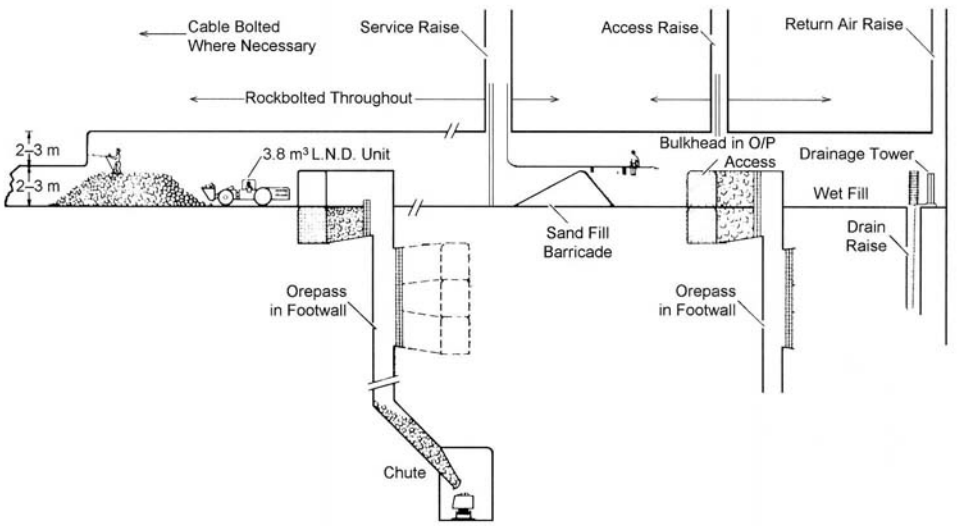


FIGURE 24.4 Diagrammatic layout of MICAF flat-back stope

Stope Cycle Detail

Drilling. *Uphole Drilling*—The uphole drilling method was used in most orebodies in the 11 to 9 level section. In the 13 to 11 level lift, nine orebodies were upholed initially; however, the method is no longer used.

In the MICAF methods, blastholes are drilled by DH123 longhole percussion drill machines mounted on air-driven, two-boom Gardner-Denver jumbos. With two booms mounted on a carrier, two rows of holes can be drilled across the stope from each carrier position. Holes are drilled in the bedding plane, and from the changeover point are inclined toward the end of the orebodies at 1.1 rad (65°). Changeover rings are then necessary from 1.1 rad (65°) north through 1.1 rad (65°) south (Fig. 24.3). All holes are drilled with 48- to 51-mm ($1\frac{7}{8}$- to 2-inch) bits on 1.8-m (10.5 ft) 1400 series extension rods. Rows of holes are drilled 1.4 m (4.5 ft) apart along strike with a spacing of 1.5 m (5 ft). An easer hole is drilled between the rings on the footwall, and the hanging-wall holes are kept about 0.5 m (1.5 ft) from the stope walls to prevent overbreak. Hole depths of 4.5 m (15 ft) give a slice height of 3.6 m (12 ft).

Flat-backing—Flat-backing is currently used in all stopes in the 13-11 level MICAF (Fig. 24.4). This is because ground conditions do not permit the use of up holes, and close control of the hanging wall is essential. Hand-held Holman Silver 3 machines drill 3-m (10-ft) holes with four rows usually being used for a vertical lift of 2.5 to 3 m (8 to 10 ft). The orebody dips at about 1.1 rad (65°) and the back is carried normal to the footwall and hanging wall. This means that the back on the hanging-wall side is much higher than it is on the footwall. A 3-m (10-ft) round is drilled and blasted each shift with the drilling being done off the top of the pile from the previous round.

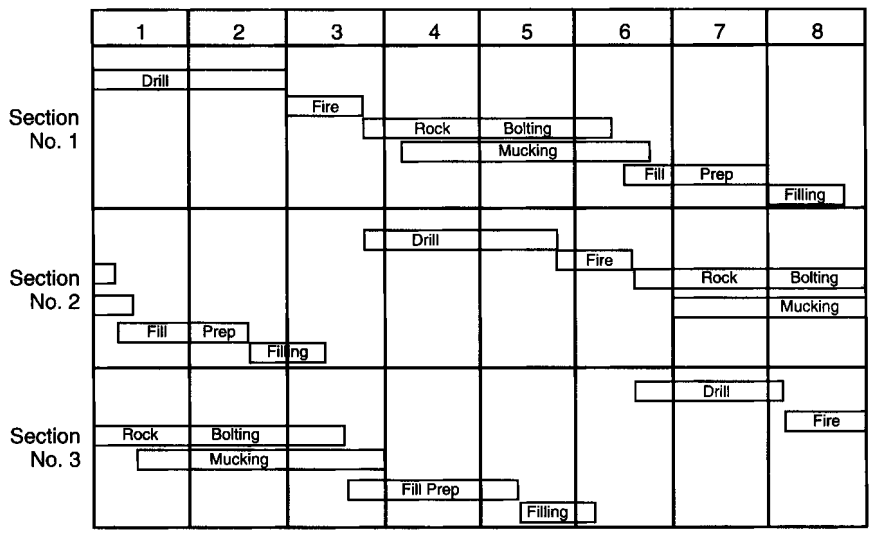


FIGURE 24.5 Typical uphole stope

Blasting. *Uphole Blasting*—Holes are loaded with ammonium nitrate–fuel oil (ANFO) and primed by anodets. Electric initiators are blasted from remote firing boxes to fire each set of holes. The return air raises are used to form a slot, and firing then proceeds back to the central changeover rings. Generally, six rings (about 50 holes) are fired before mucking of the swell is necessary. This allows sufficient expansion room for subsequent firings and leaves enough space on top of the muck pile for rockbolting.

Flatback Blasting—Exactex is used in the back holes to reduce overbreak and provide better back conditions; ANFO is used in all other holes. Electric delay detonators are used, again ignited from remote firing boxes.

Removal of Broken Ore. Mucking is by Wagner ST5A LHD units, which are serviced regularly in the stopes by diesel fitters. An ST2B is used in one stope of a narrow orebody. With good preventive maintenance and a favorable orebody layout (three sections, Fig. 24.5), up to 28,000 t (30,850 st) per four-week period can be mined. A good average production rate is 16,000 t (17,600 st) per period.

Orepasses for Extraction. Ore is transported on the 13 level. Orepasses from the 13 level are placed 6 m (20 ft) to the east (footwall) of each stope and are mined in two ways: (1) raised using the Alimak raise climber or the Robbins 61R raise borer, or (2) extended by mining into them after each fill lift and by stripping out the roof over the orepass. The first method is preferred and is used wherever possible, as a stronger and better running orepass results. Also, the pillar between stope and orepass is more regular, and greater strength is obtained.

Orepass accesses are driven on alternate sides of the orepasses every three or four lifts in order to prevent a plane of weakness forming between the orepass and stope.

Stope tramming distances are reduced by the provision of orepasses at 90- to 120-m (295- to 395-ft) intervals along strike, but two of them are situated close to the balance point (usually the main service raise) to reduce delays in the cycle.

Backfilling. After mucking of any section is completed, that section is prepared for filling. Fill is run in until there is about 3 m (10 ft) of headroom on the footwall, leaving sufficient clearance for the jumbos to start drilling. Orepass accesses are mined during the drilling phase, using either the jumbos or hand-held machines. Ideally, this phase of drilling and firing should be completed before mucking is finished in the other half of the stope in order to provide a continuous supply of ore (Fig. 24.4).

Fill Preparation and Filling—Water must be drained away from the hydraulic fill as soon as possible after placement so that the fill pipes can be quickly extended during fill, and production on the next lift can be started with minimum delay.

Drain towers are built on brick foundations at the ends of the orebodies and pipes are run from them to drain lines in drain raises kept open in the fill. The towers are made of wire mesh with a 150-mm (5⅞-in.) aperture, tightly wound, wrapped with four laps of hessian (strong coarsely woven cloth), and are 4 m (13 ft) high. The drain raises are kept open by bolting cribbed timber together to form a pigsty-type structure, which is wrapped with chicken wire, hessian, and polyethylene. The fill water then can escape only through the towers, where the drain lines direct it to sumps on the haulage level (Fig. 24.4).

Before filling, all orepass accesses and crosscuts are closed off with 1- or 0.5-m (3- or 1.5-ft) concrete brick bulkheads. If an orepass access is to be recut from the same direction, the main bulkhead is placed at the start of the access with another bulkhead at the edge of the pass. This latter bulkhead is so positioned that it will not be damaged at a later date by falling ore, nor is it so far from the pass that it unduly enlarges the pass. After filling, the access is then stripped by dropping waste or refuse rock in between the bulkheads. A fill wall is built with the ST5A at the turnover point, using dry fill. Wet fill placement starts at this wall with the fill lines being extended along strike as the new fill builds up to the required height.

Normal Fill—Deslimed mill tailings are pumped to the stopes in a 69% solids (by weight) suspension. Interlevel flow is mostly through 100-mm (4-in.)-diam boreholes with 100 and 150 nominal bore (NB) [100 and 150 mm (4 and 6 in.)] flanged pipe used for level distribution. All permanent fill lines are rubber-lined to reduce wear. Fill lines within the orebodies are extended, using 100 NB [100 mm (4 in.)] victualic pipes. These are used pipes no longer suitable for air or water recirculation.

Cemented Fill—In certain circumstances, a fill with greater self-supporting properties is required, and 10% portland cement is added to the normal lift. This is sometimes used in the first three or four lifts from the sill to reduce fill dilution during subsequent crown pillar recovery.

Ore Haulage

Air-operated chutes on each orepass control the flow of ore into 7-m^3 (9-cu yd) Granby mine cars on the haulage level. Trains of 12 cars are pulled by 20-t (22-st) Greenbat electric locomotives to the main tipple and orepass system. The ore is put through primary and secondary crushers located on the 20 and 20 secondary levels, respectively, and is hoisted to the surface up the RB2 shaft. This orepass system and the RB2 for lead ore hoisting through development down to 19 level are used.

Ground Support

Before mucking starts, rockbolting is carried out from the top of the broken ore, using 16-mm (5/8-in.) high-tension expansion shell bolts; 1.5- to 2.3-m (5- to 7.5-ft) bolts are installed in the back of a 1.2 × 1.2-m (4 × 4-ft) spacing. The hanging wall is supported by 2.3- or 3-m (7.5- or 10-ft) bolts. If the hanging wall is friable, metal strips tie the bolts together for added support, and sometimes 150 × 150-mm (5 7/8 × 5 7/8-in.) wire mesh is used for better back control. Once rockbolting is advanced 15 m (50 ft) or so beyond the changeover rings, production with the LHD units can start.

Conventional Mount Isa cut-and-fill operations will cease for a short period about 12 m (39 ft) from the 11 level, while 18-m (60-ft) untensioned cable bolts are installed updip into the crown pillar. The cables being used are anchored in 50-mm (2-in.) holes and consist of seven strands, each 7-mm (1/4-in.) diam, of high tensile steel. An alternative type consists of two strands of steel rope of 12-mm (1/2-in.) diam. The load capacity of each cable bolt is 50 t (55 st). A portland cement/water grout with an expansion agent is pumped into the hole, and it bonds the cable bolt to the rock. The cable bolting spacing is a nominal 2.4 × 2.4-m diamond pattern. As MICAF proceeds through the cable bolts, scatter protection with the normal 16-mm (5/8-in.) high-tensile rockbolts will be used. It is expected that cut-and-fill will continue slightly above 11 level, thereby extracting the bulk of the crown pillars. A diagram of crown pillar recovery in the 8, 9, and 10 orebodies on 9 level is shown in Fig. 24.6. This method was used at completion of the 11 to 9 level lift.

Ventilation

Return air raises are mined at the north and south extremities of each stope and are equipped with ladders, as well as air and water lines. They can be used as escape routes and for stope access if the normal cycle is delayed. They extend from the sill to the top access level in each block; that is, the return airway for the 13 to 11 level block is on 11 level. On this level the raises are connected by horizontal airways to a major exhaust shaft. Thus, fresh air is drawn down the central service raises and splits north and south in the stope.

Airflow of 12 to 14 m^3 (15.7 to 18.3 cu yd) per second ventilates each end of an orebody, primarily to dilute exhaust fumes. If necessary, circuit fans are installed at the tops of the return air raises to maintain this airflow.

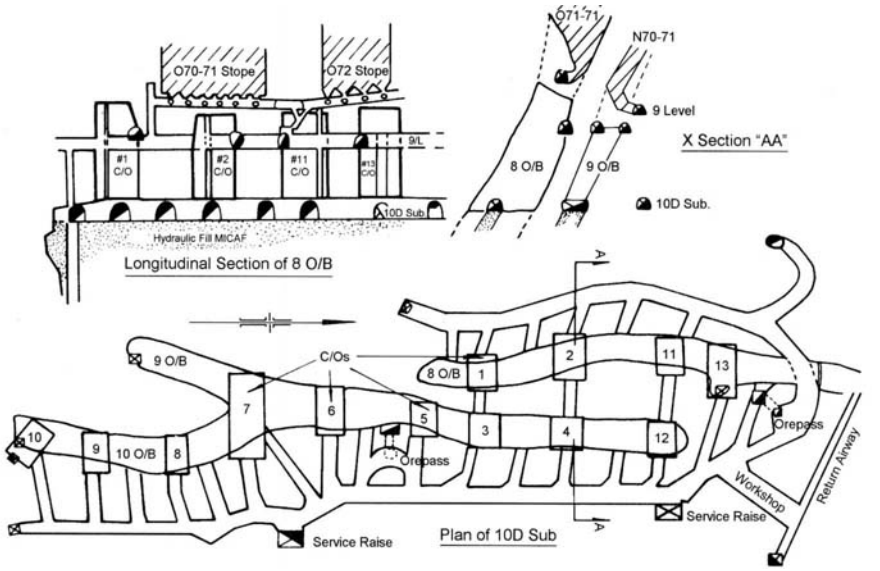


FIGURE 24.6 Crown pillar recovery of 8, 9, and 10 orebodies at 9 level

PRODUCTION UNITS

Labor

Three mining crews and a diesel operator are employed in each large stope, making the method comparatively labor-intensive. One crew is deployed at the central service raise, and the other two start at a secondary access raise, working north and south. Working off the broken ore, the hanging wall and back are barred down after each firing, and both are rockbolted to within 1 m (3.3 ft) of the face before the next cut is taken.

Productivity

Productivity in 1979–80 was 30 t (33 st) per miner-shift, with 35,000 t (38,500 st) produced per four-week (20-day) period from three stoping units. Mucking was at the rate of 330 t (364 st) per miner-shift. In the flat-backing operation, 41 m^3 (54 cu yd) were broken per miner-shift. In a miner-shift, 25 rockbolts [5.5 t (6 st)] were installed. Explosives used were 0.2 kg/t (0.4 lb per st).

MODIFIED MICAF

The major difference in the cut-and-fill mining method below 13 level has been the provision of two footwall declines from which crosscuts are driven through the orebodies. A third crosscut is located 300 m (980 ft) north of the northern decline. Whereas in conventional MICAF the stope backs are advanced updip roughly in a line at right angles to the dip, in modified MICAF the orebodies are advanced updip together because their connections must be maintained for vehicle access. This change from the previous approach, which was introduced

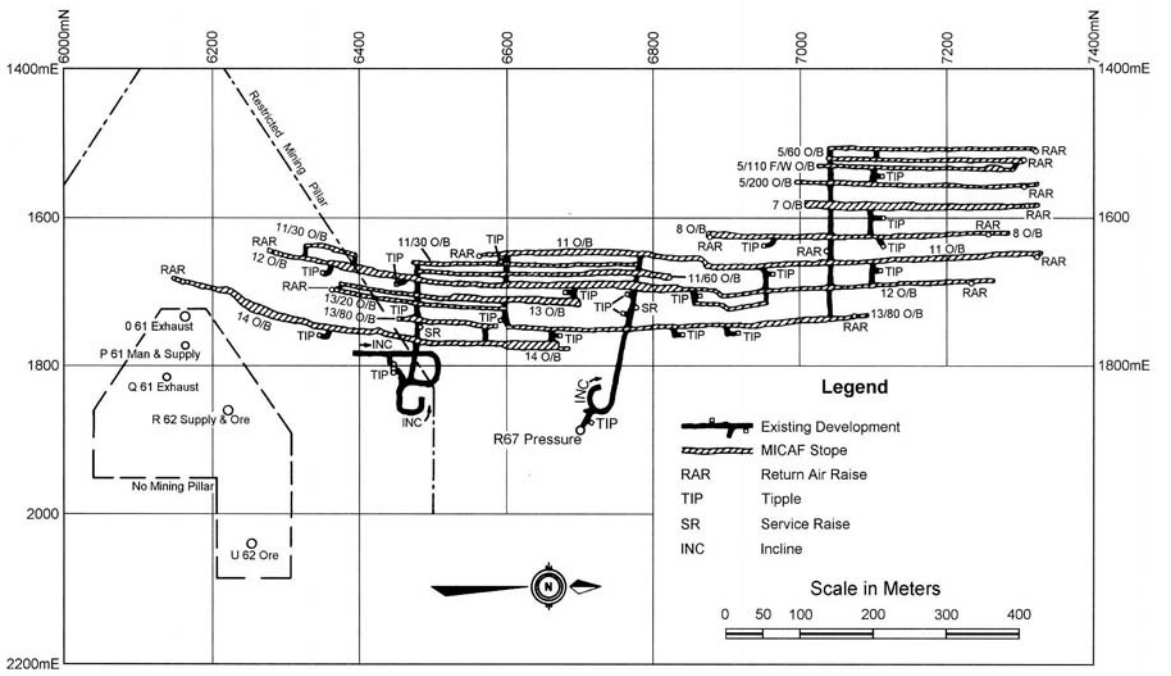


FIGURE 24.7 Plan view of modified MICAF after four lifts

on rock mechanics considerations, has necessitated increased ground support, especially cable bolting. Another change was the extension of stoping some 250 m (820 ft) north of the major exhaust shaft. See Fig. 24.7.

DEVELOPMENT

Stope Division

Because the stopes are not single entities, there is not the same requirement for careful activity scheduling and they are not split into two equal sections. The orebodies are roughly divided into the northern, central, and southern blocks for production purposes.

The provision of vehicular access has created much more flexibility, and crews can be moved within the block quite easily. Mining can be carried out in the northern block, for example, while all the southern orebodies are being filled.

Stope Access

The ramps are mined 3.7 m (12 ft) high by 4.3 m (14 ft) wide and at an average grade of 1:8. They link 14 level and stopes that are 15 m (50 ft) above 15B sublevel, and the southern one is extended to 15 level. Before mining goes above 14 level, the inclines will be extended to 13 level.

On 14 level, a major diesel workshop, fueling station, tire bay, and stores-handling area have been provided. All the diesel-powered units are maintained here. Forklifts transport the mining supplies (e.g., rockbolts, drill steels, bricks) to the mining faces.

Two crosscuts are developed from the ramps, and the third is 300 m (1000 ft) north and mined from the footwall orebody. As each lift is taken, the backs of the crosscuts are stripped in a manner similar to that used in the stopes. The waste material is left as fill in adjacent empty stopes.

Raise-bored exhaust ventilation raises (1.8 m diam) are provided at the extremities of the orebodies. Some services (compressed air, water, fill) are located in other raise-bored raises in the center of the mining block.

Stope Preparation

The development of the sill is done in a method similar to that described earlier.

Initial Stoping

In this case, the stopes are mined full width, and no cemented fill is placed. Because of increasing ground stresses with depth, it is expected that a crown pillar will have to be left on 15 level, so there will not be the need to mine right up under the old stopes. Much more ground support, including 18-m (60-ft) long cable bolts, is installed.

STOPE CYCLE DETAIL

Methods very similar to conventional MICAF are used. Because of the need to advance the stopes at the same rate, survey control is more rigorous. Similar techniques are used in drilling and blasting.

Removal of Broken Ore

Wagner ST5A and Wagner ST5E are the chief load-haul-dump (LHD) units in use, an average production rate being 20,000 t (22,000 st) per period.

Orepass for Extraction

Ore is transported on 15 level. Orepasses are located at suitable intervals near the access crosscuts and are extended each lift. Solid brick and concrete bulkheads 1 m (3 ft) thick are built at the edge of the pass to ensure their permanence. Haulage crosscuts are provided approximately 100 m (330 ft) apart and have three chutes per crosscut on average.

Backfilling

The filling methods are developments of those used in conventional MICAF. Because of the layout, the filling in one part of a stope is more related to the filling requirements in an adjacent orebody than to the breaking of ore in another part of the same orebody. In general, longer runs of fill can be made because the length of stope to be filled is greater.

Loading Ore on Sill and Haulage

The equipment used on 15 level is similar to that on 13 level, and from the tipple the ore enters the same ore stream.

Ground Support

Ground support techniques are similar, but the requirement for their use is greater. Hanging wall and footwall deterioration in some orebodies is more severe than expected. Use is made of mechanical point anchor bolts in the back, hanging, and footwalls, together with mesh and metal strips. Where necessary, cable bolts are installed. In areas of major weakness, concrete brick supports may be constructed.

Ventilation

Return air raises are mined at the north and south of the orebodies, and circuit fans installed on them, while the fresh air comes down the declines from 14 level and down some raise-bored service raises. Care is taken that when the round is fired, the broken ore does not choke off the flow of air under the face. Mining is usually done toward the extremities of the orebodies and hence the exhaust raise.

PRODUCTION UNITS

Labor

There is not the same disposition of labor with modified MICAF as with conventional MICAF. Mining crews are not allocated to a particular stope nor is there a diesel unit operator per stope who loads the ore and transports stores. Here there are eight crews of six miners (two for each three shifts) and six LHD drivers. The miners drill, blast, scale, and rockbolt. Servicing the crews is performed by a group of 18 nippers who use mobile service vehicles. Cable bolting is done by a specialist crew.

Productivity

Productivity is improved over conventional MICAF at 40 t (44 st) per miner-shift, with 56,000 t (61,700 st) being produced per 20 days in 1979–80.

Mucking performance is 580 t (640 st) per miner-shift, flat-backing 47 m^3 (61 cu yd) per miner-shift, and rockbolting 23 bolts per miner-shift [6.3 t (69 st) per bolt]. The explosives consumption is 0.18 kg/t (0.36 lb per st).

MINING PLANS

The two major areas for development are the control of ground conditions and mechanization. Much effort is being directed to rock mechanics investigations, such as structural studies, ground support techniques, and mathematical analyses of stress distributions.

The only area where mechanization has not been satisfactorily achieved is rockbolting, although units that show promise are available on the market.

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CHAPTER 25

The Zinc Corporation, Ltd., and New Broken Hill Consolidated, Ltd.

R.E. Le Messurier

GENERAL DESCRIPTION

The Zinc Corp. (ZC) and New Broken Hill Consolidated (NBHC) mines are located at Broken Hill in the western part of New South Wales, Australia, 930 km (580 miles) west-northwest of Sydney.

Mining Methods

Where pillars must be left between stopes, extraction of a block of ore between two or more levels is carried out in stages:

1. Stopes are mined overhand by horizontal cut-and-fill stoping or by longhole stoping. The various methods of horizontal cut-and-fill stoping used are designated as: (a) loader cut-and-fill stoping with horizontal blastholes and diesel load-haul-dump (LHD) units, (b) open stoping with horizontal blastholes and compressed air or electric scrapers, or with diesel or electric LHD units, or (c) square-set timber stoping, where ground condition precludes open stoping. The backs are supported with timber square sets. Pattern rockbolting of the backs may be used as an intermediate stage between open stoping and square-set stoping.
2. Vertical pillars are mined by undercut-and-fill stoping or by vertical crater retreat stoping.
3. Where a horizontal pillar has been formed below the level above the primary stope, this level pillar, usually 6.1 to 8.2 m (20 to 27 ft) in depth, may be mined by overhand square-set stoping or by undercut-and-fill.

Longhole stoping is used at the ZC and NBHC mines to mine parts of the zinc-rich A and B lodes. Drilling is carried out from drill drives and, in some cases, sills. Hole sizes used are 165 mm (6½ in.), 110 mm (4½ in.), and 65 mm (2½ in.) nominal diam with hole lengths being limited to 50, 30, and 20 m (165, 100, and 65 ft), respectively. Ammonium nitrate–fuel oil (ANFO) and water gel explosives are used. Broken ore is loaded from drawpoints at

TABLE 25.1 1979 production by mining method

	ZC			NBHC		
	t	(st)	% of total	t	(st)	% of total
Cut-and-fill open stopes	171,426	(188,911)	19.02	151,230	(166,655)	12.91
Cut-and-fill timber stopes	223,581	(246,386)	24.81	102,167	(112,588)	8.72
Undercut-and-fill stopes	63,001	(69,427)	7.00	20,251	(22,317)	1.73
Loader cut-and-fill stopes	104,268	(114,903)	11.57	130,190	(143,469)	11.12
Longhole stopes	283,164	(312,047)	31.43	664,016	(731,746)	56.69
Development	54,129	(59,650)	6.01	99,726	(109,898)	8.51
Other	1,490	(1,642)	0.16	3,683	(4,059)	0.32
Total	901,059	(992,966)	100.00	1,171,263	(1,290,732)	100.00

the bottom of the stope, using rubber-tired diesel-driven LHD units to deliver ore to an orepass.

Annual production at ZC and NBHC according to mining method is given in Table 25.1.

At Zinc Corp., mining is carried out at depths varying between 200 and 832 m (660 and 2730 ft). At NBHC, it ranges from 528 to 915 m (1730 to 3000 ft) below the surface. The average travel time to the stope is 30 to 45 min.

GENERAL OREBODY REQUIREMENTS

Size, Shape, and Dip

There are six separate lodes that carry minable ore in the mines of Zinc Corp. and New Broken Hill Consolidated. The lowest of these and the main orebodies mined at Zinc Corp. are the lead lodes, consisting of a siliceous fluoritic orebody (No. 3 lens) and overlying it a calcite rhodonitic orebody (No. 2 lens). Each possesses characteristic ratios of lead, silver, and zinc, although the grade within each lens is variable, in places being as high as 30% lead. There are also low-grade portions, which consist chiefly of poorly mineralized rhodonitic material.

No. 3 lens terminates within the Zinc Corp. leases, whereas No. 2 lens continues into the NBHC leases where it provides a significant proportion of production from the mine.

Overlying the No. 2 lens are the various zinc-rich orebodies. In ascending sequence these are lower No. 1 lens, upper No. 1 lens, A lode, and B lode. A typical assay for the small No. 1 lens group orebodies is 8% lead, 50 g/t silver, and 20% zinc.

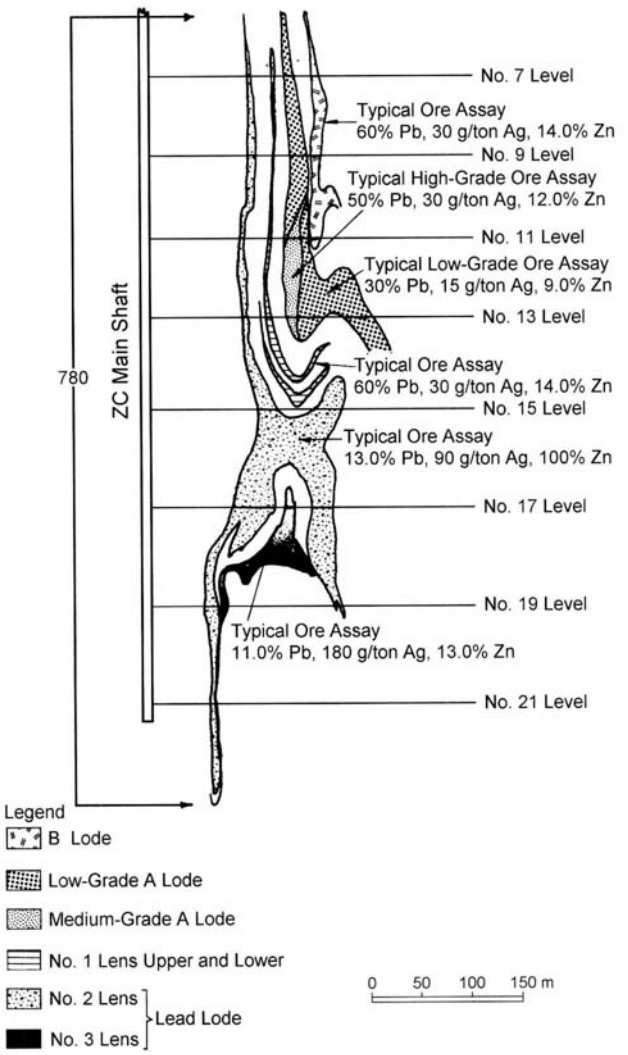


FIGURE 25.1 The Zinc Corp., Ltd., geological cross section No. 30 through the main shaft looking south 0.31 rad (18°) east, English equivalent: m × 3.2808 = ft

The A lode is a large low-grade orebody in which the higher-grade sections assay 4% lead, 30 g/t silver, and 10% zinc; larger quantities of lower-grade material also occur. B lode, the main zinc lode, typically assays 5% lead, 30 g/t silver, and 17% zinc. B lode increases in size as it extends south, and in NBHC it is a major orebody, contributing about 52% of the production.

All the lodes pitch south, the average pitch being 0.52 rad (30°) in the Zinc Corp. leases and rather flatter than this in the NBHC leases (see Figs. 25.1 and 25.2).

The ground is all competent rock.

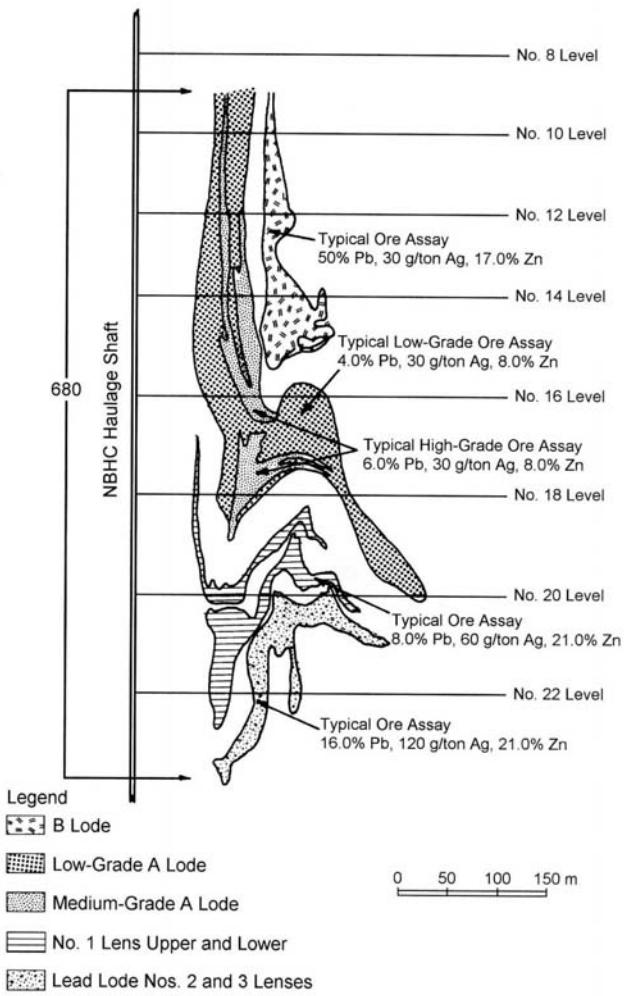


FIGURE 25.2 New Broken Hill Consolidated, geological cross section No. 62 looking south 0.31 rad (18°) east, English equivalent: m × 3.2808 = ft

Production requirements at Zinc Corp. are 900,000 t (990,000 st), and at NBHC they are 1,200,000 t (1,320,000 st).

Reasons for Adopting Open Cut-and-Fill

The high grade of the orebodies, especially the lead lode orebody and the consequent heavy nature of this ore, required a flexible approach in regard to economic limits of stope width and a mining method whereby local ground support could be quickly provided. Also, cut-and-fill stoping had the potential for following high-grade stringers of the ore into the stope walls. The New South Wales Inspectors of Mines normally require stope backs to be within 3.7 m (12 ft) of sand fill. This may be extended to 4.9 m (16 ft) in some special circumstances.

The slusher extraction methods evolved from hand-shoveling operations. In the loader cut-and-fill stopes, rubber-tired extraction is coupled with horizontal drilling.

DETAILED DESCRIPTION

Development

The size, blasting round specifications, explosives used, method of mucking, and ground support necessary for the various openings (drives, crosscuts, conventional raises, and cage and longhole raises) are given in Table 25.2. Figures 25.3 and 25.4 show development operations at No. 17 and 20 levels.

Stoping Cycle Detail

Sampling. Chip samples are taken from the stope backs and pillar walls. Other sampling is by visual observation and by assaying diamond drill core.

Drill and Broken Rock Removal. Loader cut-and-fill stoping is used to mine areas of the B lode at NBHC and of lead lode at both ZC and NBHC. In these stopes holes are drilled horizontally, 3.6 to 4.8 m (11¾ to 15¾ ft) long. Stope backs are 4.9 m (16 ft) above the sand fill and either two-boom jumbos or crawler-mounted drifters are used for drilling. The broken ore is loaded and transported to the chute by diesel or electric LHD units.

Rockbolting of stope bucks is carried out from platforms mounted on Furakawa tractors or Wagner personnel transporters, or while standing in the buckets of LHD units.

Blasting. Ammonium nitrate–fuel oil (ANFO) is fired electrically, using ½-sec and millisecond delay detonators and A.N. Gelignite 60 primers.

Rockway. *Gangway Timbering*—In sill stopes, timbered single- or double-deck gangways are constructed to provide access for servicing the stopes above. In drives outside the orebodies, timber or steel "clap-me-down" sets may be used, or, where appropriate, cyclone wire mesh held by 25-mm (1 in.)-diam rockbolts may be used to support weak ground.

Chutes—The standard chutes on the older levels were built of close-cribbed 250 × 200-mm (9¾ × 7⅞-in.) hardwood joggles, forming 1.1-m (3.6-ft) square openings for the passage of ore from stopes to levels.

Chutes have been installed having a cross-sectional area of 3.1 m^2 (33.4 sq ft) up to 24.4 m (80 ft) above rail level, above which the area is reduced to 1.5 m^2 (16.1 sq ft). The larger-sized chutes increased storage capacity and reduced ore hangups. More than 24.4 m (80 ft) above the level either timber or circular steel chutes have been used, with timber being preferred for new installations.

On No. 15 and 21 levels at NBHC and No. 19 level at Zinc Corp., the chutes measure 1.8 × 1.8 m (6 ft × 6 ft) ID and are constructed of timber only or of alternating timber and steel members, depending on the tonnage of ore expected to pass through the chute. The chutes are spaced between 12.8 and

TABLE 25.2 Development operations at Zinc Corp. and New Broken Hill Consolidated

Opening	Size*	Round*	Explosives†	Method of Mucking	Ground Support
Drives and crosscuts	0.61 m gage trackwork; 2.7 m high by 2.4 m wide (32 holes in face including 2 of 57 mm diam)	2.4 m ends, burn cut; each end usually blasted in 1 or 2 firings	Mainly ANFO, fired electrically with ½-sec delay detonators and AN Gelignite 60 primers	Mechanical loading into Granby-type cars with Eimco 25, Eimco 24B, Eimco 21, or Atlas Copco LM 70 loaders	Timber or steel sets or rockbolts where necessary
	0.91 m gage trackwork; 3.5 m high by 3.2 m wide (38–44 holes including 2 of 57 mm diam)	2.4 m ends, burn cut; each end usually completed in 3 shifts			
	3.7 m high by 4.0 m wide (37 holes including 1 of 102 mm diam)	3.7 or 4.0 m ends, burn cut; each end usually completed in 1 to 2 shifts			
				As above. Also, as appropriate, Atlas Copco CAVO 310 or diesel-powered LHD units (1.5- or 3.8-m^3 capacity)	
Raises (conventional)†	1.8 × 1.8 m (28 holes in face)	2.4 m ends; 5- to 9-hole burn cut	AN Gelignite 60, fired electrically with ½-sec delay detonators	Rock rills on to the level, loaded into trucks by Eimco loaders	Rockbolts where necessary
Cage raises	1.8 m diam (25 holes in face)	2.4 m ends; easers to 100 mm diam cage raise borehole; drilled off cage suspended from level above through 100-mm borehole	As for conventional raising	As for conventional raising	Rockbolts where necessary
Raise-bored raises	1.2- or 1.5-m diam Robbins 32R raise borer. 1.8- or 2.1-m diam Robbins 63R raise borer	—	—	Atlas Copco CAVO 310 or diesel-powered LHD units (1.5- or 3.8-m^3 capacity)	Rockbolts where necessary
Longhole raises	1.8 × 1.8 m. Under 18 m long—17 holes 65-mm diam and 2 holes 108-mm diam. Over 18 m long—8 holes 108-mm diam and 1 hole 159-mm diam	Cut advances per round restricted to 3.3 m. Strip holes can be blasted over full length in one firing.	65-mm blastholes loaded with Isanol 50 primed with 32-mm AN Gelignite 60 fired with millisecond-delay detonators. 108-mm blastholes loaded with Isanol 50 primed with 140-g cast pentolite-type primer. Fired with nonelectric millisecond-delay detonators in the blasthole. A 10 g/m detonating cord trunkline is initiated by electric millisecond-delay detonators.	Atlas Copco CAVO 310 or diesel-powered LHD units (1.5- or 3.8-m^3 capacity)	Rockbolts where necessary

*English equivalents: m × 3.2808 = ft; mm × 0.039 = in.

†AN Gelignite 60 being replaced with cartridged water-gel explosives.

‡Under certain conditions, close timbering to form two compartments is required by the Mines Inspection Act.

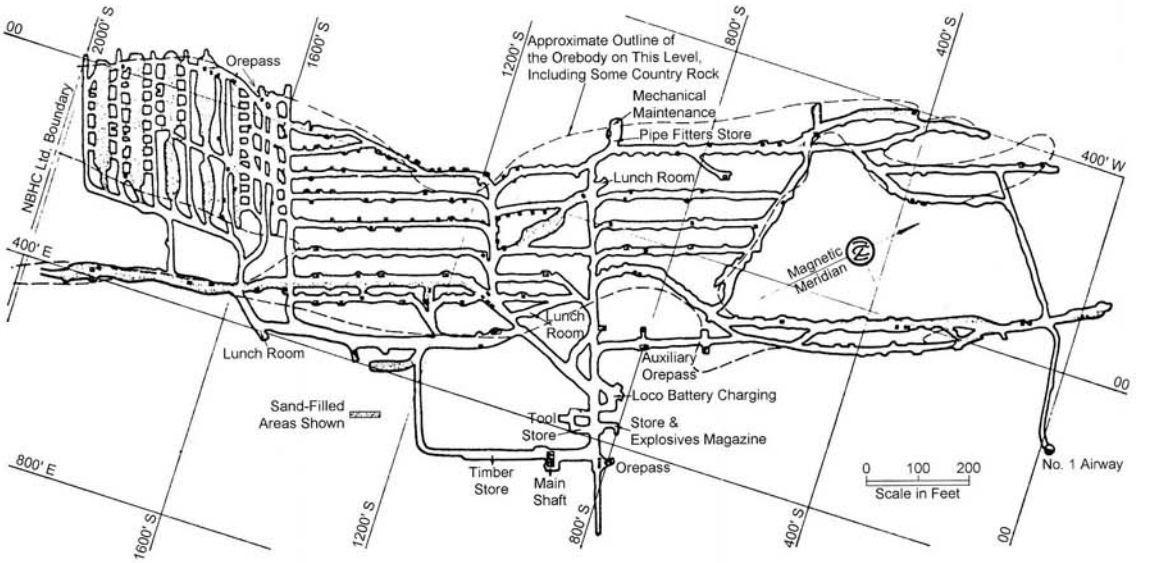


FIGURE 25.3 Plan of No. 17 level, Zinc Corp., 743.7 m (2440 ft) below the surface, metric equivalent: ft × 0.3048 = m

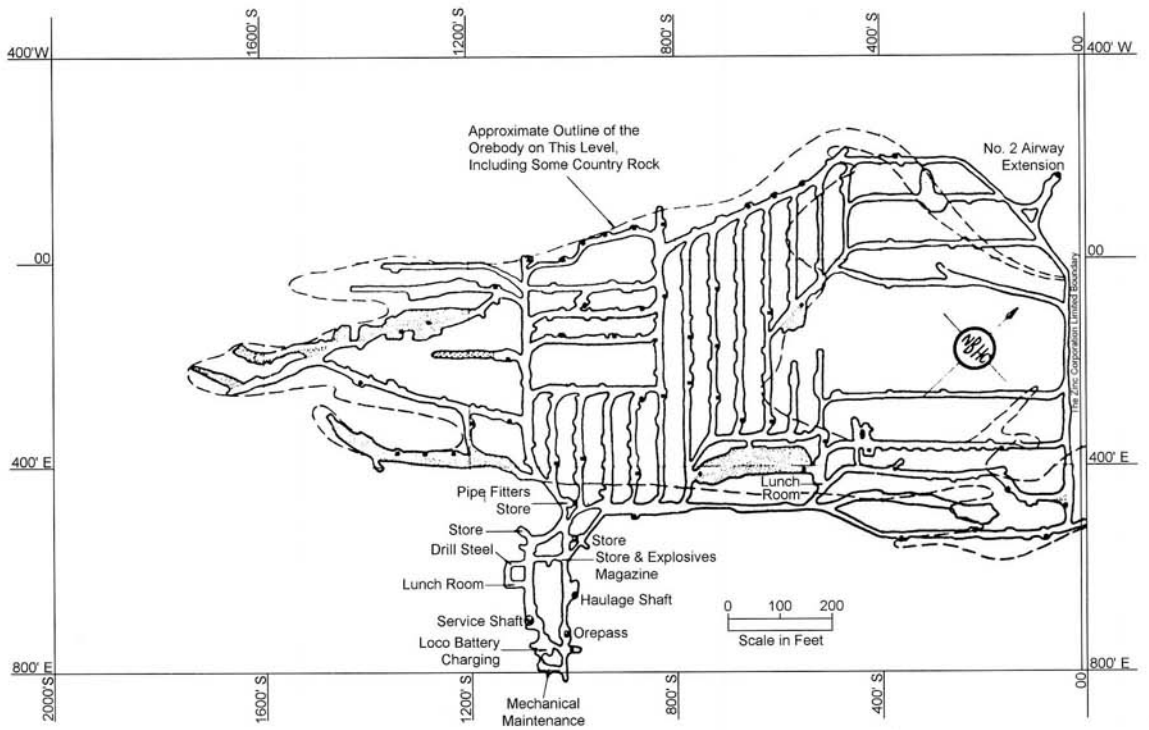


FIGURE 25.4 Plan of No. 20 level, NBHC, 877.44 m (2920 ft) below the surface, metric equivalent: ft × 0.3048 = m

TABLE 25.3 Typical grading of sand for filling (1979)

Size (microns)	+300	−300 +210	−210 +150	−150 +150	−105 +75	−75 +53	−53
% wt	21	18	24	16	11	5	5

Metric equivalents: micron × .000039 = in.

27.5 m (42 and 90 ft) apart, with new chutes tending to be installed at the greater spacing.

On the levels, chute fronts are fitted with undercut air-operated arc doors 1.37 m (4½ ft) wide. On the No. 15 and 21 levels at NBHC and No. 19 level at Zinc Corp. the doors are 1.82 m (6 ft) wide.

Backfilling. The sand used for stope filling is partly deslimed mill residues. The deslimed and dewatered residues are stored in two silos on the surface and are drawn into the mine through a vertical sand pass as required. A typical grading of sand for filling is given in Table 25.3.

Two mixing stations, one 24.4 m (80 ft) below the surface and one at No. 12 level of Zinc Corp., handle filling requirements for both mines. At the stations, the sand is mixed with water to produce a pulp of 70% solids, which is piped by gravity to the stopes to be filled. The mixing is done in steel mixing cones lined with fused basalt to resist abrasion by the sand. Water consumption in each mixing cone varies between 61.4 and 68 kL/hr (16,200 and 17,950 gph).

The main vertical pipelines are 125-mm (4⅞-in.) bore pipe, rubber-lined to 110 mm (4¼ in.) ID, and the branch lines on the levels are 100-mm (4-in.)-diam, 5.4-mm (¼-in.) victaulic pipe in 3.28-m (10¾-ft) lengths. The pulp is distributed in the stopes through 75-mm (3-in.)-diam abrasion-resistant rubber hoses or 100-mm (4-in.)-diam plastic pipe. The path of the pulp flow is controlled by steel disks inserted in the appropriate pipe joints or by cast-iron plug valves at major change points. The filling rate is from 103 to 115 m^3 (135 to 150 cu yd) of dry sand placed per hour.

Stopes are prepared for sand filling by wrapping the ladderways, chutes, etc., with 340 g/m^2 (1.1 oz per sq ft) burlap or with plastic sheeting. Sections of stopes that are required to be filled are surrounded with a barricade of wire netting supported on boards and covered with burlap. Square-set stope scraper gangways are prepared by nailing boards to the legs and lining with burlap.

The water from the pulp drains or decants into the chutes, ladderways, etc., finally leaving a compact fill. This water gravitates to level drains and sumps and flows into the mine drainage system.

Approximately a quarter of the sand fill placed at NBHC has cement added to it to consolidate the sand. At Zinc Corp, primarily the fills in No. 19 level stopes are so treated. Cement addition is controlled from the subsurface mixing

station. The cement is drawn from 204-t (225-st) cement silos and slurried in mixing tanks in the proportion desired for the type of fill being placed. The cement slurry is then piped to the sand mixing cones in the subsurface mixing station and mixed with the sand. Most of the fills placed have a sand-to-cement ratio of about 30:1 by mass. A 300-mm (11¾-in.)-thick layer of sand-to-cement of 10:1 by mass is used to provide a surface sufficiently hard to withstand blasting and scraping. In loader cut-and-fill stopes, a 300-mm (11¾-in.)-thick layer of plain sand is placed on top of the cement-consolidated fill so that any filling material picked up by the diesel loaders will not contaminate the milling circuit with cement.

Haulage System. *Trackwork*—Except on No. 15 and 21 levels and in the No. 3 airway district, NBHC uses 0.61 m (2 ft) gage track laid with 15 or 22 kg/m (10 or 15 lb per ft) rails. The maximum grade is 1 in 150. The standard curve radius has been 11 m (36 ft), but 0.61-m (2-ft) gage development, where practicable, has 15.2-m (50-ft) radius curves. On No. 15 and 21 levels, 0.91-m (3-ft) gage is used with 31 kg/m (20 lb per ft) rails. The grade is 1 in 200 maximum. A minimum curve radius of 15.2 m (50 ft) is used with 45.6-m (150-ft) radius where practicable.

Equipment—Where the trackwork is 0.61 m (2 ft) gage, 3.1 and 1.9 m^3 (4 and 2.5 cu yd) Granby-type cars are used. Where 0.91-m (3-ft) gage trackwork is used, 7.1 and 4.3 m^3 (9.3 and 5.6 cu yd) Granby-type cars are used, as well as a train of 11.5 m^3 (15 cu yd) Haggslund shuttle cars. For level cleanup, 1.1 and 1.9-m^3 (1.4 and 2.5 cu yd) Granby-type cars are used. For concrete handling, 0.9-m^3 (1.2 cu yd) side-tipping cars and 3.2 m^3 (4.2 cu yd) ballast cars are used.

For the 0.61-m (2-ft) gage trackwork, a 4-t (4.4-st) standard AX Mancha locomotive with Exide 45-cell NC-19-2 lead-acid batteries and an 8.15-t (9-st) Gemco with 45-cell Exide Kathanode NC-19-2 lead-acid batteries are used. For the 0.91 m (3 ft) gage, three locomotives are used: 9.45-t (10.4-st) Gemco with 55-cell Exide ND-19-2 lead-acid batteries, 20.04-t (22-st) articulated Gemco with two 55-cell Exide ND-19-2 batteries in series, and 4-t (4.4-st) standard AX Mancha with Exide 45-cell NC-19-2 lead-acid batteries.

Ground Support. Areas with a weak back are supported by hardwood timber beams 200 × 150-mm (8 × 6-in.) or 200 × 125-mm (8 × 5-in.) in cross section and a length of 1.8 m (6 ft), or by random rockbolts, or regularly spaced [1.2 × 1.2-m (4 × 4-ft) pattern] rockbolts of 2.4-m (8-ft) length. Slabby waste rock walls are secured by rockbolts.

Extensive cable bolting is done. Bolts of 20 m (65.5 ft) length are delivered to the mine individually coiled, and bolts of other lengths are made up on-site, as required, of strand. Strand consists of seven high-tensile steel wires of 5.08-mm (3/16-in.) diam laid up with right-hand lay. The programs have involved 20-m (65½-ft) bolts installed on 2 × 2-m (6½ × 6½-ft) pattern in 64-mm (2½-in.) diam holes bored with 1600 series percussion equipment (see Fig. 25.5).

The bolt consists of two 15.2-mm (⅝-in.) diam high tensile steel strands, a breather tube, and spacers located at regular intervals along the length of the

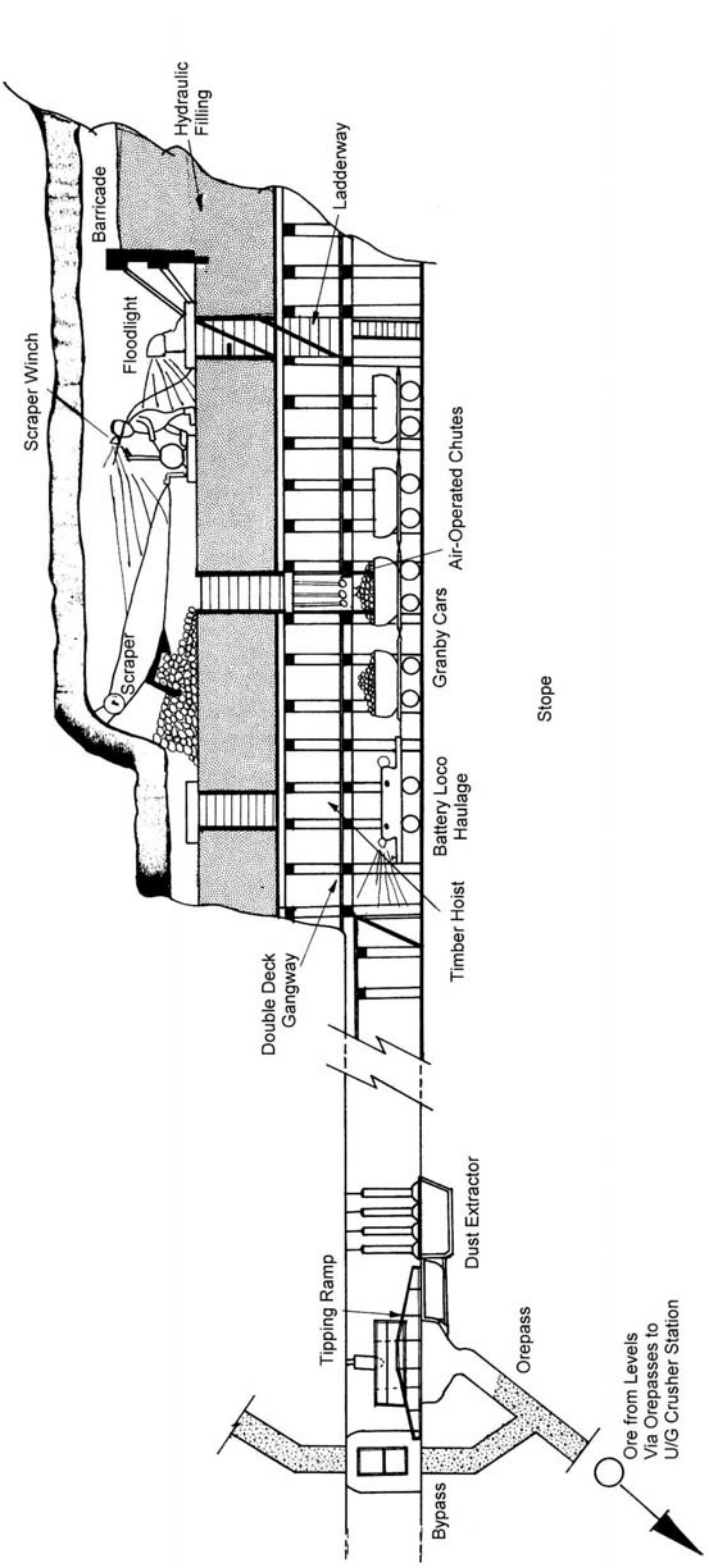


FIGURE 25.5 Diagram of ore flow at NBHC from cut-and-fill stope

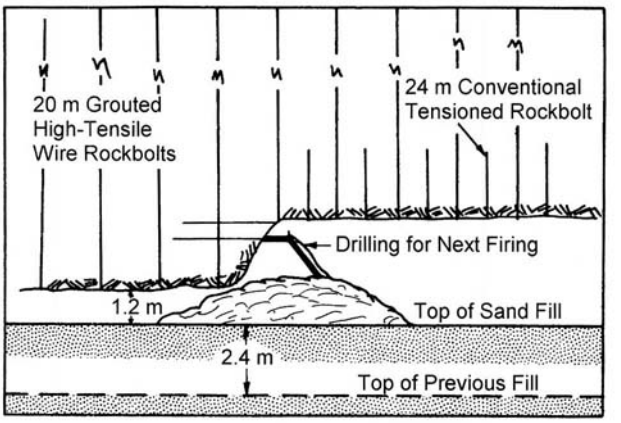


FIGURE 25.6 Schematic of ground support with conventional tensioned rockbolts and the grouted high tensile wire rockbolts, English equivalent: m × 3.2808 = ft

bolt. Bolts are not tensioned before they are cement-grouted in place. In up holes the bolt is retained in place prior to grouting by an anchor formed by bending back one wire of one strand at the end of the bolt that is to be inserted into the hole. A feature of the bolts is their high strength-to-weight ratio, enabling them to be installed easily by three miners working on the consolidated hydraulic fill under 3-m (10-ft) backs.

Bolt grouting is done with a high early-strength cement combined with fluidizing and expanding agents. A two-stage Mono pump, which develops a maximum pressure of approximately 900 kPa (130 psi), is used to pump the grout into the hole through a 19-mm (¾-in.) diam polyethylene pipe. The hole is sealed and grouted from the collar up. The air bleed is achieved through a 12-mm (½-in.)-diam polyethylene pipe extending to the top of the bolt. Flow of grout through this pipe indicates a completed hole. Sieving of the cement and close control over the water-cement ratio reduces blockages.

In addition to the tensile strength of the bolt of 552 kN (approximately 50 t), the technique relies upon the strength of the grout-steel bond. Although laboratory tests suggest a bond strength of about 500 kN should be sufficient to utilize the full strength of the steel bond, it is necessary to design support on a much more conservative basis to allow for the progressive debonding that occurs with time, when high loadings are developed.

A 20-m (65½-ft) bolt is expected to provide reinforcement for approximately 15 m (50 ft) above the back of the stope. The cost of this reinforcement is 1.7 times that of conventional 2.4-m (8-ft) pattern bolting.

The ground support systems are shown in Fig. 25.6.

Ventilation. *General Scheme*—An exhaust system is used to draw air through both mines. Figure 25.7 shows a generalized layout of the airflow pattern.

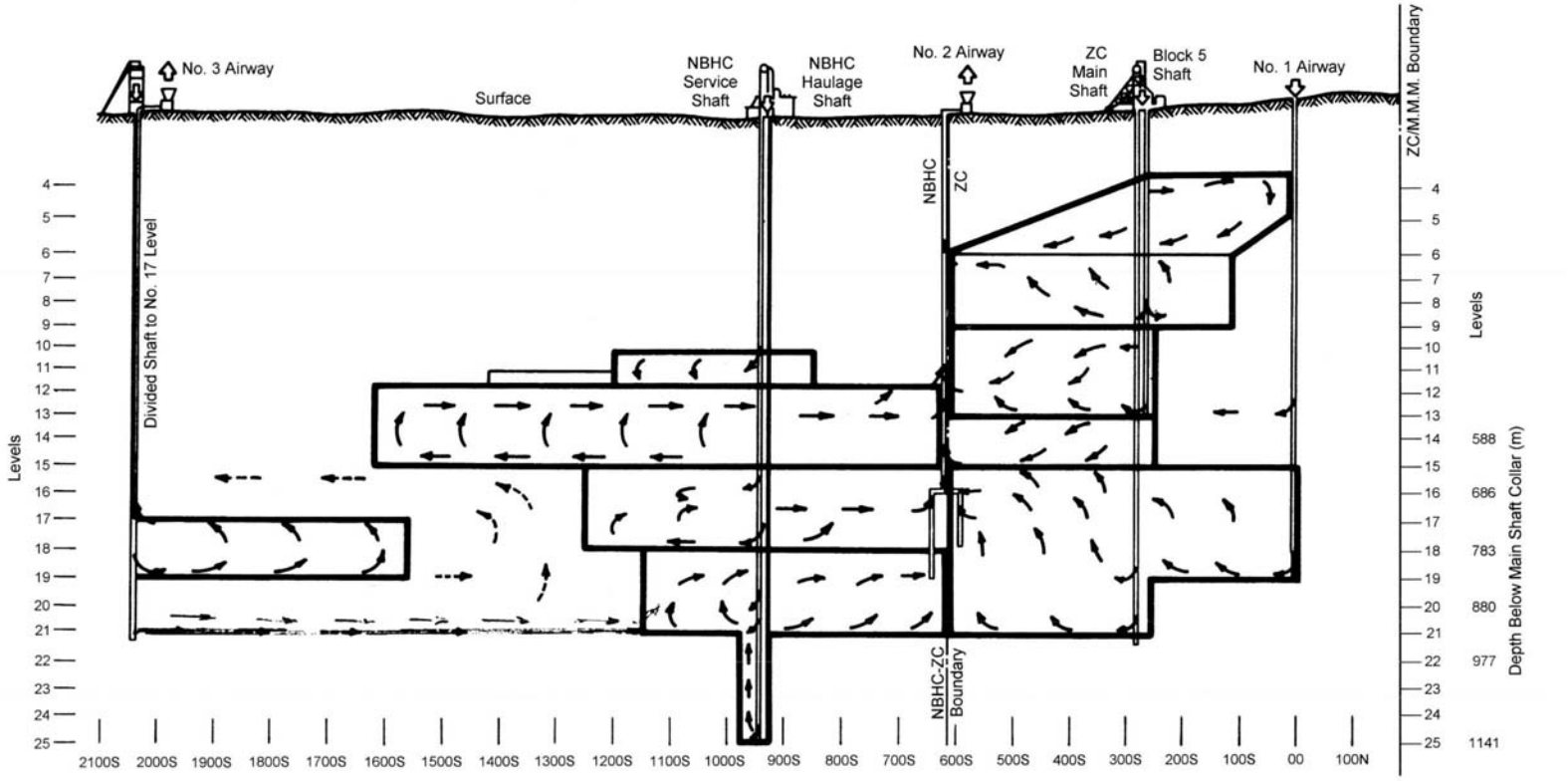


FIGURE 25.7 Ventilation system in the Zinc Corp. and New Broken Hill Consolidated mines, showing the existing major airflow patterns. Broken lines indicate proposed patterns and facilities as of 1979. English equivalent: m × 3.2808 = ft.

The main fans are on the surface at the top of No. 2 upcast shaft located near the boundary between the two mines and at the top of the upcast segment of No. 3 airway shaft at the southern end of the NBHC mine. To provide an exhaust connection for the deeper workings of the NBHC mine, an unlined shaft (the No. 2 airway eastern extension shaft) is located 80 m (262 ft) east of the bottom of the No. 2 upcast airway shaft on the No. 16 level and extends the return air system down to the No. 22 level.

Air enters the mines through their respective downcast shafts, namely the main shaft, block 5 shaft, and No. 1 airway for the Zinc Corp. mine, and the haulage shaft, service shaft, and No. 3 airway for NBHC mine.

No. 3 airway shaft is divided into an upcast and a downcast segment and ventilates the southern portion of NBHC, separated from No. 2 airway district by doors on those levels that join the service shaft and No. 3 airway. On No. 21 level, air downcast in No. 3 airway is exhausted through No. 2 airway after augmenting intake from the service shaft.

Ascending ventilation is the general practice. Airflow to the various levels is regulated from the downcast shafts at the bottoms of the working areas. Some regulation is also required at the entrances to the upcast shaft. The distribution of flow to the working places between the levels is achieved by partially or completely sealing the winze collar openings above them.

Equipment—The exhaust system is operated at No. 2 airway by three radial flow fans in parallel. The fans are Richardson 3460CY type with 4394-mm (170-in.)-diam impellers. Each has a maximum airflow capacity of 840 m^3/s (1.8×10^6 cfm).

The No. 2 airway extension booster fan, situated in the horizontal connection between the bottom of the No. 2 airway and the top of the extension, draws up to 260 m^3/s (550,900 cfm) through the workings by means of a 3660-mm (143-in.)-diam single-stage Richardson axial-flow fan operating in series with the No. 2 airway fans.

Another underground booster fan on No. 13 level at New Broken Hill Consolidated, ventilating the B lode stopes, operates in parallel with the extension booster fan. An auxiliary booster fan with a capacity of 25 m^3/s (53,000 cfm) is also used for the bottom levels of NBHC mine.

Auxiliary ventilation for individual working places is usually provided by electrically operated axial-flow fans of different sizes with capacities in the range of 2.4 to 7 m^3/s (5000 to 14,800 cfm). Ventilation ducting may be of galvanized iron, plastic-coated terylene (flex), or plastic with spiral wire support (spiroflex) of 400-, 560- or 760-mm (15½-, 22-, or 30-in.) diam.

Short development headings are served by 400-mm (15½-in.)-diam compressed-air-operated axial-flow fans delivering up to 2.4 m^3/s (5000 cfm). In long development drives and for special applications, electrically operated axial-flow fans of either 5- or 7-m^3/s (10,600- or 14,800-cfm) minimum capacity are used with 760-mm (30-in.)-diam ducting.

TABLE 25.4 Production per contract stoping miner-shift (1979)

	ZC t (st)		NBHC t (st)	
Cut-and-fill open stopes	29.18	(32.10)	36.88	(40.57)
Cut-and-fill timber stopes	13.76	(15.14)	13.46	(14.81)
Undercut-and-fill stopes	20.01	(22.01)	22.98	(25.28)
Loader cut-and-fill stopes	52.98	(58.28)	56.51	(62.16)
Longhole stopes	86.56	(95.22)	74.17	(81.59)
Total	27.21	(29.93)	44.93	(49.42)

Conventional and cage rises are ventilated by compressed-air-operated fans with 200-mm (8-in.)-diam steel piping.

PRODUCTION DATA

Labor

There are 1166 underground employees paid daily and 987 surface employees paid daily. Staff numbered 672 in December 1979.

Incentive System

All miners are on a contract system based on tonnage produced.

Stope Capacity

Open stopes are budgeted at 2000 t (2200 st) per four-week period. This is based on two shifts per day, five days per week.

Productivity

Production per total worker-shift surface and underground, including all auxiliary services and overtime shifts, at the Zinc Corp. and NBHC mines is 3.54 t (3.9 st). Underground production per the total worker-shift underground (including No. 3 airway) is 6.69 t (7.4 st) at Zinc Corp. and 7.34 t (8.1 st) at NBHC. For both mines combined it is 7.04 t (7.8 st). Tonnage production per contract stoping miner shift is given in Table 25.4.

All costs are confidential and therefore are not available. However, some production statistics and costs for Zinc Corp. are given in Table 25.5 and for New Broken Hill Consolidated in Table 25.6.

TABLE 25.5 1979 production statistics and costs, The Zinc Corp., Ltd.

Average number of workers employed, including staff, but excluding outside contractors	1563	
Wages and salaries paid (excluding lead bonus)	$17,390,076	
Lead bonus paid	$8,301,913	
Ore mined, t (st)	901,059	(992,967)
Assay value of crude ore*	8.3, 68.0, 9.2	
Lead concentrate produced, t (st)	91,782	(101,144)
Assay value*	76.4, 587.0, 3.3	
Zinc concentrate produced, t (st)	146,086	(160,987)
Assay value*	0.96, 24.0, 51.8	
Metal recoveries:		
Lead in lead concentrate	94.3%	
Silver in lead concentrate	87.7%	
Zinc in zinc concentrate	91.4%	
Residue, t (st)	660,530	(727,904)
Assay value*	0.43, 6.1, 0.62	
Total ore mined to end of 1977, t (st)	34,150,384	(37,633,723)
Present ore reserves	6,100,000	(6,722,200)
Development:		
Driving and crosscutting, m (ft)	1714.8	(5626)
Rising, m (ft)	593.1	(1946)
Diamond drilling, m (ft)	3339.0	(10,955)
Mining timber used, m^3 (cu ft)	6210.0	(219,145)
Explosives used, kg (lb)	297,729	(656,373)
Water used, ML ($\times 10^6$ gal)	355.17	(93.8)
Power consumed:		
Electricity per hour, kW (hp)	39,819,432	(53,358,038)
Compressed air, m^3 (cu yd)	95,829,801	(125,249,540)

*% lead, g/t silver, % zinc.

TABLE 25.6 1979 production statistics and costs, New Broken Hill Consolidated Ltd.

Average number of workers employed, including staff but excluding outside contractors	1238	
Wages and salaries paid (excluding lead bonus)	$14,458,556	
Lead bonus paid	$6,582,444	
Ore mined, t (st)	1,171,263	(1,290,732)
Assay value of crude ore*	6.9, 56.0, 12.1	
Lead concentrate produced, t (st)	100,632	(110,896)
Assay value*	74.7, 572.0, 3.5	
Zinc concentrate produced, t (st)	248,551	(273,903)
Assay value*	0.86, 17.0, 52.7	
Metal recoveries:		
Lead in lead concentrate	93.6%	
Silver in lead concentrate	87.3%	
Zinc in zinc concentrate	92.4%	
Residue, t (st)	820,499	(904,190)
Assay value*	0.37, 5.1, 0.88	
Total ore mined to end of 1977, t (st)	23,951,619	(26,394,684)
Present ore reserves, t (st)	7,000,000	(7,714,000)
Development:		
Driving and crosscutting, m (ft)	5939.1	(19,485)
Rising, m (ft)	1016.3	(3334)
Diamond drilling, m (ft)	19,763	(64,838)
Mining timber used, m^3 (cu ft)	4128	(145,673)
Explosives used, kg (lb)	479,960	(1,058,120)
Water used, ML ($\times 10^6$ gal)	513.26	(135.5)
Power consumed:		
Electricity per hour, kW (hp)	51,180,178	(68,581,438)
Compressed air, m^3 (cu yd)	85,195,227	(111,350,120)

*% lead, g/t silver, % zinc.

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CHAPTER 26

Cut-and-Fill Stoping at Star Mine

Gordon Miner

GENERAL DESCRIPTION

At the Star mine, Hecla Mining Co., located in Burke, Shoshone County, Idaho, the mining method is horizontal slicing (timbered or untimbered) cut-and-fill using hydraulic sand backfill. Back stoping as well as breast stoping is used; see Fig. 26.1. The annual production from all cut-and-fill mining is 259,816 t (286,400 st). The average depth ranges from 366 to 2408 m (1200 to 7900 ft). A majority of the tonnage comes from an average depth of 2240 m (7350 ft). Average travel time to the stope is 1 hr.

GENERAL OREBODY REQUIREMENTS AND LIMITATIONS

Size and Ground Conditions

The orebody is 610 to 1219 m (2000 to 4000 ft) along the strike and 2408 m (7900 ft) on the dip, with an average vein width of 2.6 m (8.5 ft). The dip is essentially vertical.

The orebody is heavily sheared and broken on the main fault structures; it is more competent on the wall veins but everywhere subject to rock bursting at depth. The hanging and footwalls are strongly fractured on the main fault structures, again more competent but still fractured on the wall veins. All host rock is Revett quartzite.

Value and Production Requirements

The ore value is $56 per ton ($51 per st). Production requirements are 1089 t/d (1200 stpd), five days per week.

Why Cut-and-Fill?

The slusher extraction method is the most satisfactory and practical for narrow vertical veins with sheared walls. The method must be quite selective to

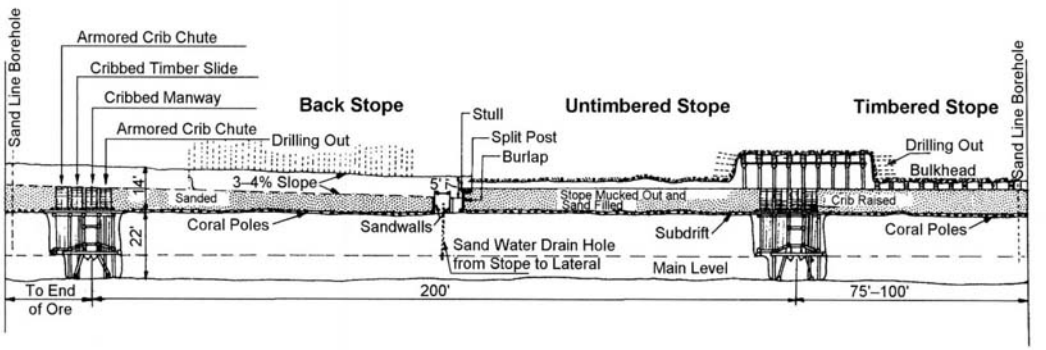


FIGURE 26.1 Typical stoping methods at Star mine, Hecla Mining Co., metric equivalent: ft × 0.3048 = m

be commercial. The veins are too narrow for rubber-tired extraction and the grade is not high enough to make resuing practical.

DETAILED DESCRIPTION

Development

Access to the main shaft is by the 3.2-km (2-mile) main haulage tunnel. The level intervals in the vertical timbered shafts are 61 m (200 ft). Vein development is by crosscut and laterals located 30.5 m (100 ft) into the vein wall. Vein crosscuts are on ±61 m (200 ft) centers. Access to the vein is by raises carried up with the stope. The raises are equipped with four wood-cribbed chutes (two orepasses reinforced with angle irons, one manway, and one timber slide, Figs. 26.2 to 26.6). The initial cut is located 6.7 m (22 ft) above the main level (Fig. 26.1).

Stoping Cycle Detail

Sampling. Grab samples are taken from the mine cars. Channel sampling of the faces and backs is done when necessary.

Drilling. For stope drilling, Ingersoll-Rand 300M percussion drills are used, with Ingersoll-Rand 300M and Gardner-Denver RB83M stopers used for back stoping. Breasting down is done with Gardner-Denver S83FM jacklegs. The back stoping rounds are 0.9 to 1.8 m (3 to 6 ft) and the breast stoping rounds 1.5 to 3 m (5 to 10 ft).

Blasting. For stope blasting DuPont Tovex 100 watergel with fuses and igniter cord is used primarily. Approximately 60% of the blasting is done with ammonium nitrate–fuel oil (ANFO). For development headings and shaft sinking, DuPont Drivex watergel with Ensign Bickford Nonel Primadet primers is used.

Mucking. Broken ore is removed entirely by slusher in the stopes. These are mostly 3.7–7.5-kW (5–10-hp) Ingersoll-Rand air-powered or 7.5–37.3-kW (10–50-hp) Ingersoll-Rand electric-powered. In the horizontal development

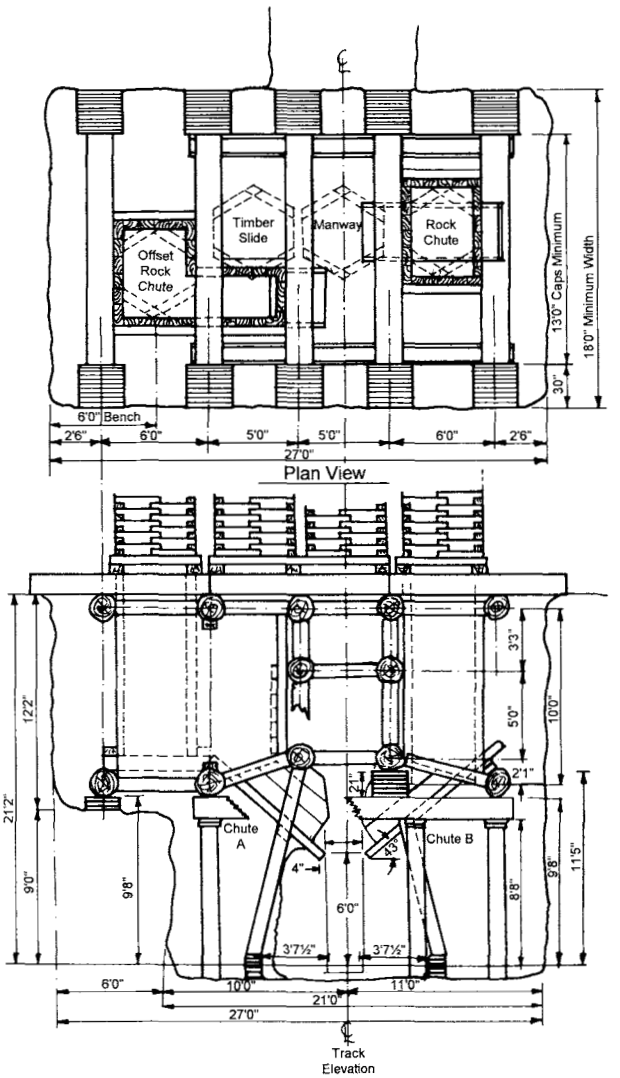


FIGURE 26.2 Timber and rock dimensions, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m

headings Eimco 12B and 21 and Atlas Copco LM-56 overshot mucking machines and a 0.76 m^3 (1 cu yd) hydroelectric Wagner MST-1 LHD (load-haul-dump) unit are used.

Rockway for Extraction. The cribbed chutes (Fig. 26.7) are nearly vertical, hexagonal in shape, and are made of spaced 152 × 203-mm (6 × 8-in.) timber, reinforced with varying weights and sizes of angle iron, depending on wear requirements. Rectangular cribbed chutes are used where ground pressure is not a problem. Cribbing is added in 2.4- or 3-m (8- or 10-ft) extensions according to the floor height (whether timbered or untimbered). See Figs. 26.8 and 26.9.

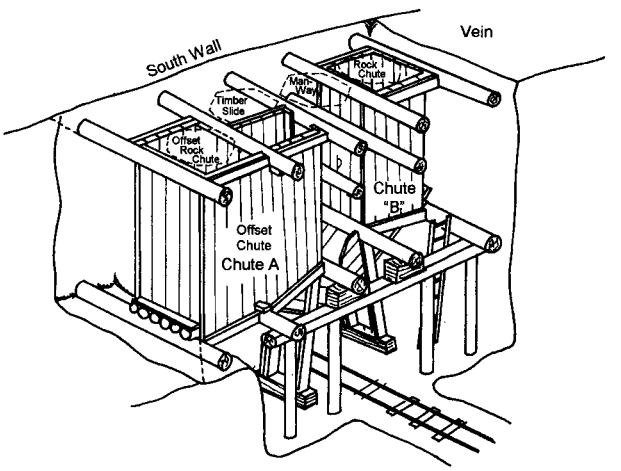


FIGURE 26.3 General vein-access raise layout

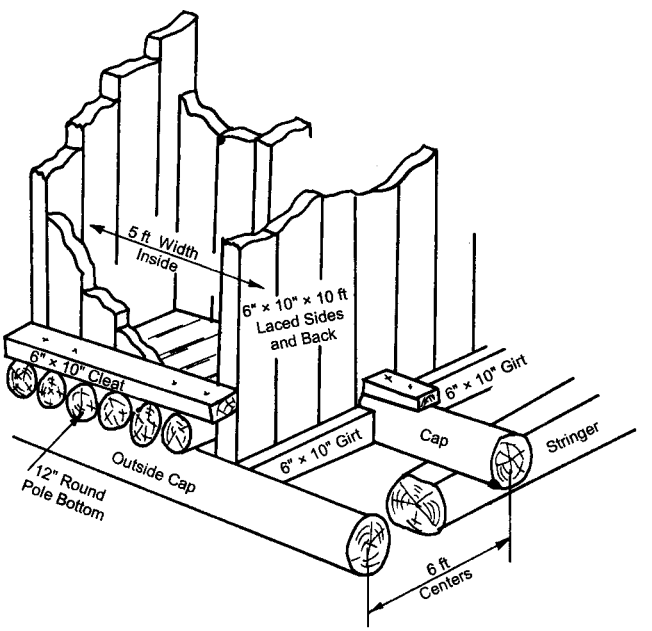


FIGURE 26.4 Offset chute bottom detail for chute A, metric equivalents: In. × 25.4 = mm; ft × 0.3048 = m

Backfilling. Backfilling is done with hydraulic sand consisting of classified mill tailings at 65% solids. The fill contains approximately 20% 325-mesh fines. The tailings are pumped 3353 m (11,000 ft) at 50–55% solids 22.7 dry t/h (25 dry stph) with a duplex piston pump. Cycloning increases the tailings density to 65% solids. Fill storage is in a 998-dry-t (1100-dry-st) agitated storage

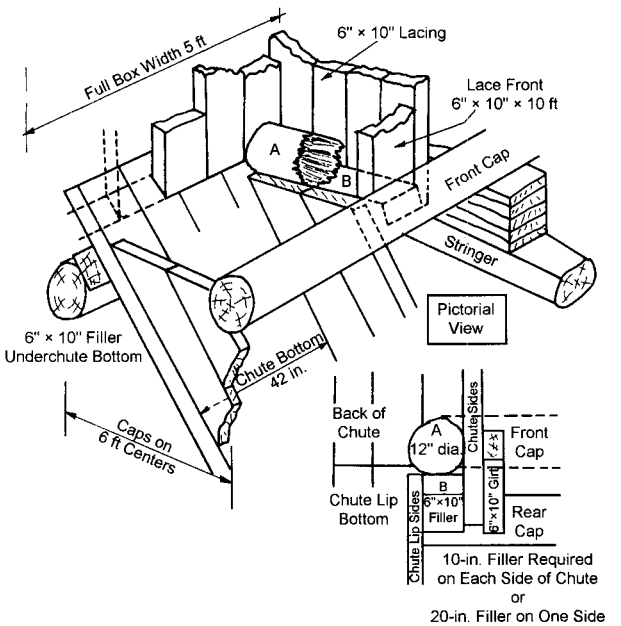


FIGURE 26.5 Front section view of chute bottom detail, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m

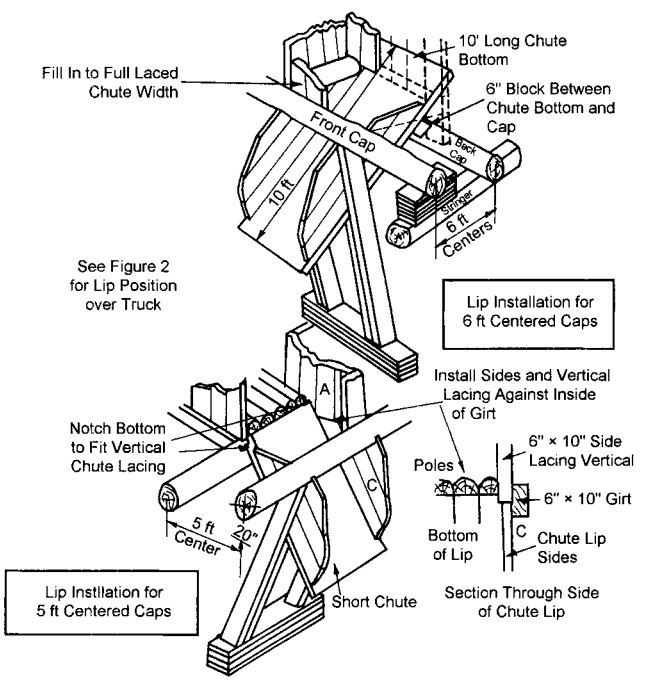


FIGURE 26.6 Chute lip detail, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m

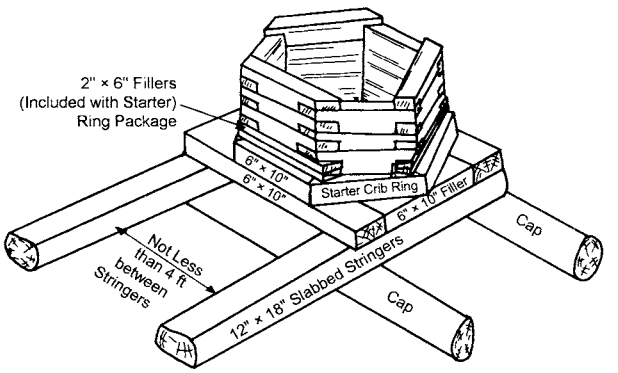


FIGURE 26.7 Crib bearing detail, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m

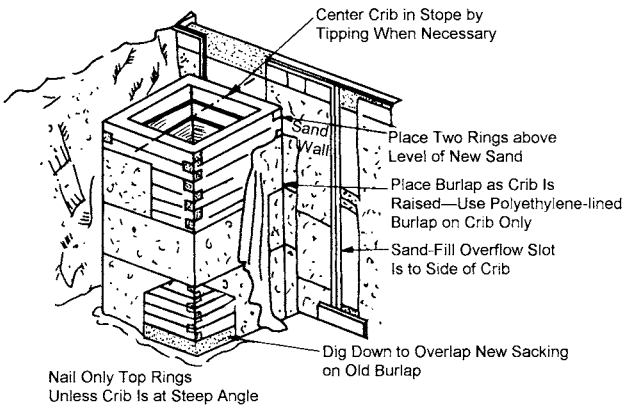


FIGURE 26.8 Square crib construction

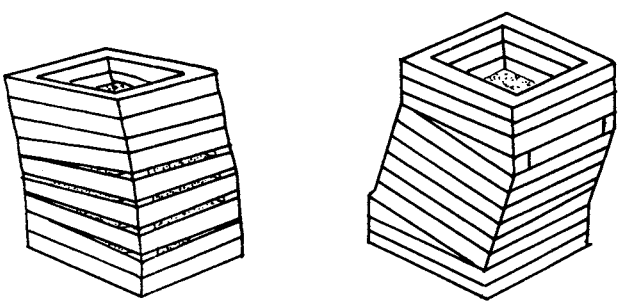


FIGURE 26.9 Standard method for tipping chute. Left depicts situation using 25.4-mm (1-in.) wedges, and right depicts the situation when using wedge segments is avoided.

chamber. Pouring is via 63.5 mm (2½ in.) ID vertical rubber-lined schedule 40 sand line in the shaft at 86.2 dry t/h (95 dry stph). Ceramic Venturi-shaped restriction devices are installed every 103.6 m (340 ft) in the sand line to reduce velocity and wear. Decanting of water from the stopes is by a pipeline carried up in the fill from a borehole leading to the lateral. All sand water is fed to a central vertical settler where accumulated slimes are pumped back to the stopes that are ready to be filled. Flocculation is also used to clarify the stope overflow. In timbered stopes, lagging is used on the floor to maintain separation of broken ore and backfill. In unlimbered stopes no floor lagging is used.

Loading and Haulage. To load the ore, lever-operated arc gates on timbered chute lips located in the sill are used. For ore hauling, Westinghouse or Greenburg 5.4-t (6-ton) battery locomotives pull three to six 1.7-m^3 (60-cu-ft) side-dump Granby cars on 609.6-mm (24-in.) gage, 40-lb rail.

Ground Support. Timber stull and post sets or rockbolts with plates, wire mesh, or steelplate mats, are used. Shotcrete is also used in large openings such as hoist rooms.

Stope Ventilation. Air from the main mine ventilation system is directed into the stopes through the use of 7.5- and 14.9-kW (10- and 20-hp) axial flow fans and 559-mm (22-in.) steel vent pipe and vent tubing. In general, the majority of the stoping is ventilated in conjunction with the various mine air-conditioning systems. A typical level has a 208.7-t (230-st) central refrigeration unit circulating 31.5 L/s (500 gpm) of 7°C (45°F) chilled water to 18.1- or 36.3-t (20- to 40-st) coil units. The coil units [36.3 t (20 st)] are equipped with a 283.2-m^3/s (600,000-cfm) axial flow fan feeding cooled air from the main ventilation circuit directly to the stope working face via 559- and 533-mm (22- and 21-in.) vent lines buried in the sand fill. Cooled air is delivered to the stope face at ±24°C (±75°F). Exhaust air returns to the circuit via the raise. Ground temperatures on the lower levels are from 47 to 48°C (113 to 118°F). Stope temperatures are maintained in the 24 to 31°C (75 to 85°F) range.

Production Statistics

Manpower. There are 83 people engaged in underground production and 28 in underground maintenance. More than 100 are engaged in miscellaneous operations, serving as hoist operators, in haulage, as nippers, cagers, etc.

Incentive System. Development incentive is based on rate/meter (rate per foot) advance. The stoping incentive is based on the volume of ore broken and slushed. There are also rates of payment for different types of ground support and other miscellaneous operations.

Stoping Capacity and Productivity. The stope capacity varies with the width and length of the stope. There are approximately 24 operating stopes for an average production of 21,319 t/month (23,500 st per month). Productivity is 18.4 t/miner-shift (20 st per miner-shift).

TABLE 26.1 Capital expenditures at Star Mine, late 1970s

	$, thousands
Initial Investments*	
Underground and miscellaneous, surface	3150
Surface plant	2556
Concentrator (estimated)	750
Primary—capital (historic cost)	6500
Working	Negligible
Supporting (if applicable)	Included
Total capital investment	6500
Designed production rate [t/a (stpy) ore for which foregoing investment was made]	254 (280)
Major Capital Expenditures (mining operation only)	
Installations	
No. 4 production shaft (1969) 1646 m (5400 ft)	2750
No. 4 hoist (1969) 3581 kW (2500 hp)	1250
No. 4 shaft pumping system 1463 m (4800 ft)	750
Vertical silt settler 6.7 × 19.8 m (22 × 65 ft)	75
Refrigeration unit, 208.7-t (230-st)	60
Gardner-Denver Minibore jumbo	56
Waste disposal conveyor system	152
Moving, reinstalling existing 59.5 m^3 (2100 cu ft) compressor	31
Additional Expenditures (direct costs, 1979 average)	
Shafts, 4.73 m (1553 ft)	640.0
Level development, 5651.6 m (18,542 ft)	2301.8
Stope development for 1,299,084 t (1,431,995 st)	1518.8
Additional equipment pumping system, 1974–1975	760
Approximate annual replacement capital	Expensed as purchased

*Excludes additional and replacement capital expenditures.

Stope Cycling Problems. The major cycling problems are with sand-fill scheduling, rock bursts, bad ground, plugged chutes, manpower scheduling due to vacations, etc.

COSTS

Capital expenditures in the late 1970s are given in Table 26.1. Operating cost data for the same period are given in Tables 26.2 to 26.4.

PLANNING

Full-scale test stopes are used for student miner training.

TABLE 26.2 Operating cost data, Star Mine, 1975 average

Costs by Process*	1975 Average	
	Cost per ton of ore, $	% of total
Development	3.07	9.4 (direct)
Stoping, preparation and sand fill		
Drilling	Direct 0.77	2.4 (direct)
Blasting	Direct 0.80	2.4 (direct)
Loading	none	none
Total stoping and preparation	10.96	33.5 (direct)
Hauling		
Levels 1300-ft average	1.42	4.3 (indirect)
Main tunnel, 11,000 ft	0.76	2.3 (indirect)
Crushing (primary underground only)	none	—
Hoisting, 4100–5700 ft	1.33	4.1 (indirect)
Support, rockbolts, timber	2.08	(Included in 9.4 direct cost)
Ventilation, 80,000 cfm	0.92	2.8 (indirect)
Pumping, 900 gpm	1.30	4.0 (indirect)
Mine general	1.41	4.3 (indirect)
Other (repair $1.65, supervision $1.63, other allocated $4.67, materials handling $2.11, power $1.56)	11.52	35.2 (indirect)
Total indirect	18.66	56.9 (indirect)
Total operating cost (mine direct and indirect)	32.69	100.0

*Costs include general plant cost or overhead. Costs are without depreciation. Metric equivalents: st × 0.907 = t; ft × 0.0348 = m; cfm × 0.4720 = L/s; gpm × 0.06308 = L/s.

TABLE 26.3 Unit costs, direct and indirect, Star Mine, 1975 average

Development	Size*	Cost per ft, $
Rectangular 4-compartment timbered shaft	9.5 × 23.5 ft	1232.00
Rectangular 2-compartment timbered shaft raise	9.5 × 12 ft	330.00
Rectangular 2-compartment timbered shaft on 4-ft borehole	9.5 × 12 ft	407.00
Drifts, crosscuts, laterals	11 × 10 ft	320.00
Raise boring	5 ft diam	163.00
Diamond drilling	AX	15.80
Longholing	3½ in.	14.53

*Metric equivalents: ft × 0.3048 = m; in. × 25.4 = mm.

TABLE 26.4 Operating costs by requirement, Star Mine, 1975 average

	Direct Mining Cost per Ton of Ore*	Costs, % of Total
Supervision	1.53	4.7
Operating labor	15.65	47.9
Operating materials	9.15	28.0
Maintenance labor	1.98	6.0
Maintenance materials	2.33	7.1
Other, direct	2.06	6.3
Total operating cost	32.70	100.0

*Metric equivalent: st × 0.9071847 = t.

Consideration is being given to remote-controlled hoisting and jogging capability for the ore loading and pump warning systems.

Among future mechanization possibilities is the use of an impact hammer mining machine coordinated with timber placement and muck removal. This system would replace the drilling and blasting cycle currently used.

There is need for further research and development efforts to increase productivity and to lower unit costs.

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CHAPTER 27

Cut-and-Fill Stoping at Lucky Friday Mine

Staff, Lucky Friday Mine

GENERAL DESCRIPTION

The Lucky Friday mine is located in Mullan, Shoshone County, Idaho. The mining method is horizontal slice cut-and-fill (with timbering for support or untimbered) using hydraulic sand. Cemented sand fill at a 1:17 ratio is used in sill pours and at a 1:7 ratio for the final 152 mm (6 in.) capping in successive stope floor pours.

Annual production with 100% cut-and-fill mining is 167,829 t (185,000 st).

The average depth of mining is 1296 m (4250 ft), and the average travel time to the stope is 45 min.

GENERAL OREBODY REQUIREMENTS AND LIMITATIONS

Size

The orebody is 396 to 488 m (1300 to 1600 ft) long on strike, 1421 m (4660 ft) on dip, with an average width of 2.3 m (7.5 ft). Dip varies from vertical to 1.2 rad (70°).

Ground Conditions, Ore Value, and Production Requirements

The orebody contains quite strong argentiferous galena and quartz. The hanging wall and footwall are strongly sheared and fractured.

Ore value is $211/t ($192/st).

The production requirements are 657 t/d (725 stpd), five days per week.

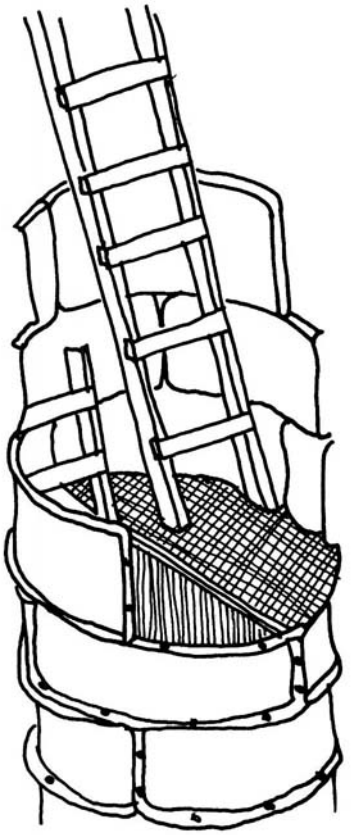


FIGURE 27.1 Commercial shearing reverse liner plate for chutes and manways, 1.2 m (4 ft) diameter

Reason for Adopting Cut-and-Fill

Slusher extraction is the most practical and satisfactory method for narrow, crooked, steeply dipping veins that require selective mining. Where possible, in the wider stopes, the Atlas Copco Cavo 310 air-powered load-haul-dump (LHD) unit is used. Resuing is not practiced.

DETAILED DESCRIPTION

Development

Main access is through a 122-m (400-ft) tunnel to the top of the main rectangular timbered shaft. Level intervals are 64 m (210 ft). Level development is by laterals located 30.5 m (100 ft) in the wall of the vein with crosscuts through the vein. Raises on the vein lead to a subdrift located 6.7 m (22 ft) above the crosscut rail (Fig. 27.1). Access raises are 1.5 m (5 ft) in diam and are lined with 1.2 m (4 ft) diam by 6.4-mm ($\frac{1}{4}$-in.) pressed steel segments. These raises are bored from the level above approximately 58 m (190 ft) to the initial raiseup off the crosscut. For details of the development, see Figs. 27.1 to 27.5.

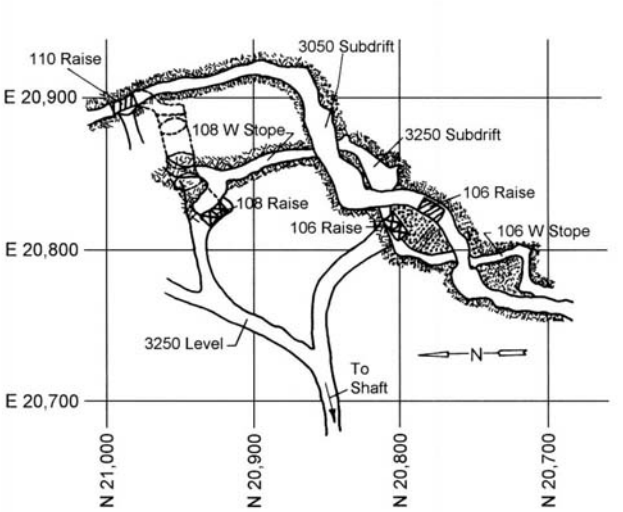


FIGURE 27.2 Typical vein plan, 3050 and 3250 levels, Lucky Friday mine

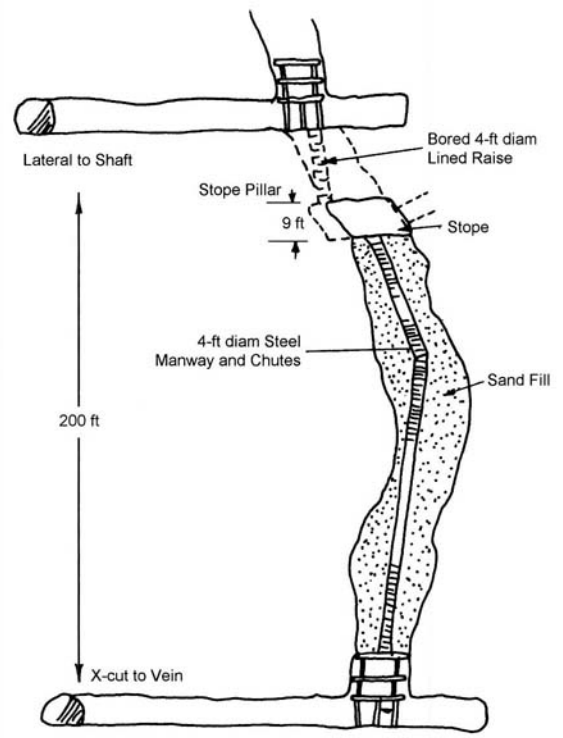


FIGURE 27.3 Typical section through Lucky Friday mine, metric equivalent: ft × 0.3048 = m

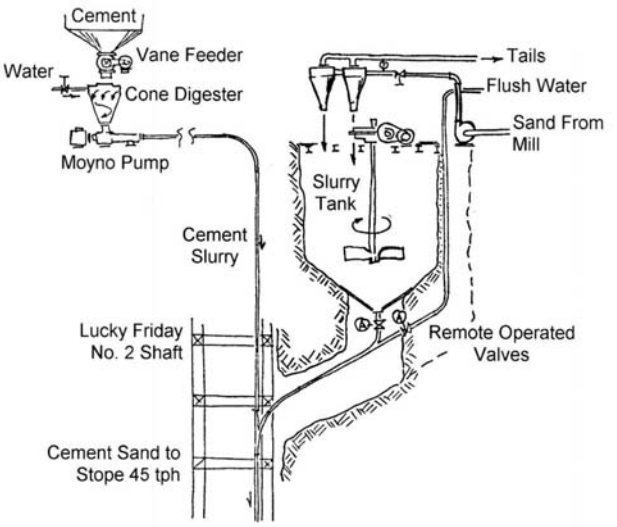


FIGURE 27.4 Lucky Friday mine sand-cement fill system

Stoping Cycle Detail

Sampling. Grab samples are taken from the ore cars. Channel sampling of the faces and backs is done as required.

Drilling. Drilling is done with Gardner-Denver percussion drills. Ingersoll-Rand and Gardner-Denver stopers are used for back stoping, and Gardner-Denver jacklegs for breasting down. The back stoping rounds are 0.9 to 1.8 m (3 to 6 ft). The breast stoping rounds are 1.5 to 4 m (5 to 13 ft).

Blasting. One hundred percent of the stoping is done with Aqua-gel slurry. Ninety percent of all blasting is done with nonelectric primers and primacord, ten percent is with safety fuse and caps. The development headings are blasted with either Aqua-gel slurry dynamite or ammonium nitrate–fuel oil (ANFO). All development blasting is initiated with nonelectric primers and primacord. Primacord initiation is with safety fuse and cap.

Mucking. Broken ore is removed primarily with Joy 14.9 kW (20 hp) electric double-drum slushers. Occasionally in the widest stopes, the Atlas Copco Cavo 310 LHD is used. In the horizontal development headings Eimco and Atlas Copco overshot track-mounted mucking machines and Einco 901 Scoop-trams are used.

Rockway for Extraction. For construction details, see Figs. 27.4 and 27.5. The 1.2-m (4-ft) diam by 6.4-mm (¼-in.) thick pressed steel plates in quadrants 406 mm (16 in.) high are raised with the stope and encased in the sand fill. The rockways are extended by adding 406-mm (16-in.) high bolted rings in sufficient numbers to accommodate a 2.4-m (8-ft) high fill.

Backfilling. The material for filling is hydraulic sand backfill consisting of classified mill tailings at 65–68% solids. The fill contains approximately

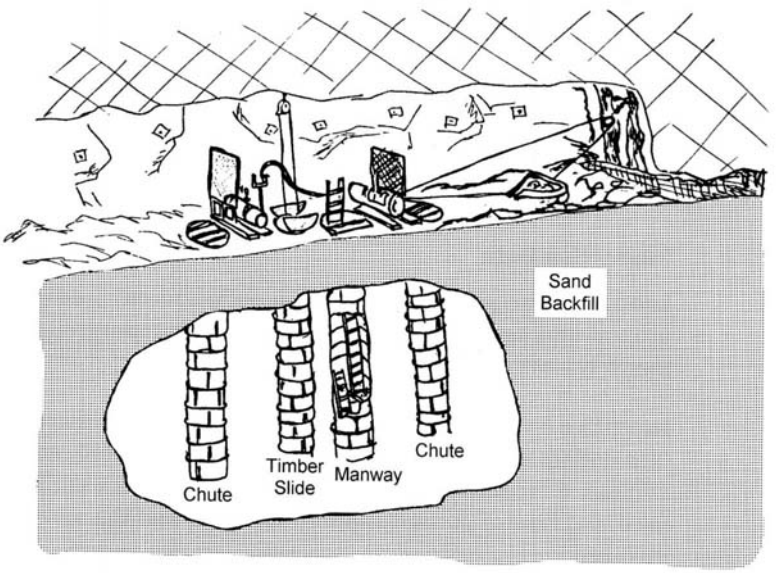


FIGURE 27.5 Typical Lucky Friday stope

20% −325 mesh fines. The tailings are pumped 274 m (900 ft) from the mill at 27% solids to a 109-dry-t (120-dry-st) agitated storage tank at the head of the main shaft. Cyclone classification over the tank increases the density to 65–68% solids.

Backfill is poured at 40.8 t/h (45 stph) as required through a 50.8-mm (2-in.) ID rubber-lined vertical sand line in the main shaft. Remote control is accomplished by a two-wire telemetering system with the transmitter located at the end of the sand line in the stope. The system is operated by one person per shift during sand-pouring operations. When cemented sand is required, dry cement from a 160-barrel storage bin on the surface is mixed into a slurry and introduced into the sand line at a point below the sand storage tank. The cement addition is controlled manually on orders communicated over the sand-fill telephone system.

A 152.4-mm (6-in.) thick cemented sand-fill capping with a sand-to-cement ratio of 1:7 separates the broken ore from the backfill.

Loading and Hauling. The ore is loaded into cars via air-operated upcut chute gates on timbered chute lips located on the sill.

Three to six 1.7-m^3 (60-cu ft) Granby-type side dump cars are pulled by 5.4-t (6-ton) battery locomotives on 609.6 mm (24 in.) gage 18-kg (40-lb) rail. Also used are 1.4-t (1.5 st) Mancha locomotives and 2.8-m^3 (100-cu ft) bottom dump cars.

Ground Support. For ground support, timber stull and post sets, stulls only, or rockbolts with plates, mats, or cyclone fencing are used. For shaft support, grouted or epoxied bolts up to 7.3 m (24 ft) long are used along with

rectangular framed timber. Shotcrete is used in support of large openings such as shaft stations or hoist rooms.

Stope Ventilation. For ventilation, 2830–5664 L/s (6000–12,000 cfm) auxiliary axial flow fans draw from the mine ventilation circuit, blowing to the stope face via a 533.4-mm (21-in.) diam culvert encased in the sand fill near the raise area. Exhaust is usually through the bored raise to the level above.

Production Data

Labor. There are 95 employees in underground production; 38 in underground maintenance; and 120 who serve as underground hoisters, sanders, motor operators, nippers, cagers, etc.

Incentive System. The system is based on productivity. For development, it is in dollars per meter (foot) of advance. The stoping incentive is based on the volume of broken and slushed rock. The rates vary with the mining method used, the type of ground support required, and the miscellaneous work involved.

Stope Capacity. The capacity varies with the width and length of stopes available. There are approximately 20 operating stopes for the 13,608 t (15,000 st) mined on average per month.

Productivity. In stoping operations, the productivity is 12.1 t (13.3 st) per miner-shift.

Stope Cycling Problems. The major problems are the sand-fill scheduling, rock bursting, ground support, plugged or hung chutes, and worker scheduling.

CAPITAL EXPENDITURES AND OPERATING COSTS

The capital investment is summarized in Table 27.1. Operating costs data are given in Tables 27.2 to 27.4, in which costs by process, by mine unit, and by requirement are summarized, based on 1979 figures.

PLANNING AND IMPLEMENTATION

Full-scale test stopes are used for student miner training.

Among the remote-control systems are a hoisting and jogging capability whereby the cager can control the hoist from individual skip packets. The sand-fill, pump-warning, and rock-burst-monitoring systems are remote-controlled.

In a mechanized operation, an impact hammer mining machine would be incorporated with a timber placement and muck removal system. This would replace the drilling and blasting cycle in stoping.

TABLE 27.1 Capital Investment, Lucky Friday mine

Initial investments*	
Primary, capital	$2,500,000
Working	Negligible
Supporting	Included
Total capital investment	$2,500,000
Designed production rate (stpy ore for which investment was made)†	173,000
Major capital expenditure (mining operation only)	
Concentrator	$600,000
1250 hp main hoist (1960)†	$540,000
Surface plant	$240,000
Additional expenditures (direct costs)	
Shafts, 51 ft†	$294,000
Level development, 2380 ft†	$225,600
Stope development for 173,245 st†	$204,200
Additional equipment	Expensed as purchased

*Excludes additional and replacement capital.

†Metric equivalents: st × 0.9071847 = t; hp × 0.746 = kW; ft × 0.3048 = m.

TABLE 27.2 Operating costs by process, Lucky Friday mine

Process*	Cost per ton of ore,* 1979 avg., $	% of total
Development	5.87	10.0 (direct)
Stoping, stope preparation, sand fill		
Drilling	1.12	2.8 (direct)
Blasting	1.09	2.2 (direct)
Loading	—	—
Total stoping	23.23	36.8 (direct)
Hauling, +800 ft	2.34	4.0 (indirect)
Crushing (primary underground only)	—	—
Hoisting, 2000–4000 ft	3.54	7.1 (indirect)
Support, timber, rockbolts	3.18	6.3 (incl. indirect cost)
Ventilation, 130,000 cfm	1.21	2.4 (indirect)
Pumping, 275 gpm	0.83	1.7 (indirect)
Mine general	2.98	5.9 (indirect)
Other: compressed air 0.40; repair 1.20; supervision 2.61; other allocated 3.92; materials handling 1.20; power 0.83 (sand fill 1.61 included in direct)	11.77	23.5
Total indirect	29.40	50.3
Total operating	58.36	

*Metric equivalents: ft × 0.3048 = m; cfm × 0.472 = L/s; gpm × 0.06308 = L/s.

TABLE 27.3 Lucky Friday mine, unit costs direct and indirect, 1979 average

Development*	Size*	Cost per ft*
Rectangular compartmented timbered shaft	83 ft	$3275
Drifts, crosscuts laterals	1746 ft	488
Raise boring, 356 ft	430	293
Diamond drilling	AX	21

*Metric equivalent: ft × 0.3048 = m.

TABLE 27.4 Operating costs by requirements (1979), Lucky Friday mine

	Cost per ton of ore, $	Direct mining costs, % of total
Supervision	3.04	5.2
Operating labor	29.99	51.4
Operating materials	17.16	29.4
Maintenance labor	2.45	4.2
Maintenance materials	2.04	3.5
Other	3.68	6.3
Total	58.36	

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CHAPTER 28

Cut-and-Fill at the Bruce Mine

Ronald T. Johnson, Keith E. Dyas, and John Nelson

GENERAL DESCRIPTION

The Bruce mine of Cyprus Mines Corp. is located in Bagdad, Arizona. The mining method used is open cut-and-fill. Of the annual production of 81,647 t (90,000 st), approximately 83% is taken from load-haul-dump (LHD) stopes and the balance from slusher stopes. All ore is produced from the area between the 1250 level and the 2300 level. The average travel time from the shaft pocket to the stope is approximately 5 min.

GENERAL OREBODY REQUIREMENTS AND LIMITATIONS

Size, Shape, and Dip

The Bruce orebody occurs in quartz-sericite schist with Dick rhyolite on the footwall and andesite on the hanging wall. Diabase dikes are found in the hanging wall; there is also a dike coming off the footwall and crosscutting the orebody. All of the rock types are of the Precambrian Yavapai series and have been subjected to regional metamorphism. A composite of the orebody is given in Fig. 28.1.

The deposit is of massive sulfides occurring as a steeply dipping replacement body. On the upper levels the ore is veinlike with widths from 0.6 to 4.6 m (2 to 15 ft), dipping at 1.4 to 1.5 rad (80° to 85°). On the lower levels the ore is dipping from 1 to 1.2 rad (60° to 70°) with widths from 3 to 16.8 m (10 to 55 ft). The strike length varies between 107 to 183 m (350 to 600 ft).

The rhyolite footwall generally has a knife-edge contact with the massive sulfides. The exceptions to this are the upper levels, where there is a 1.5 to 3 m (5 to 10 ft) band of silicified sericite schist between the sulfides and the rhyolite.

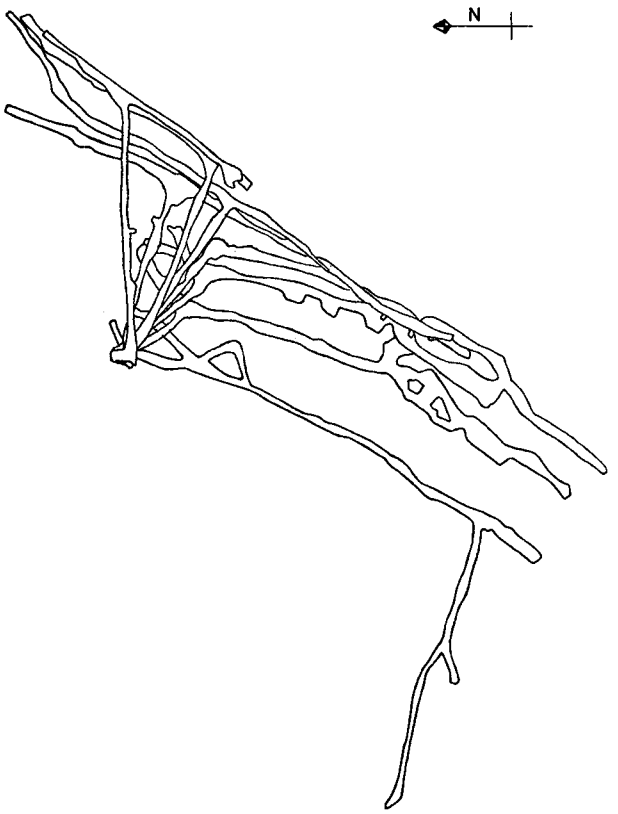


FIGURE 28.1 Composite of the Bruce orebody, 1100 to 2150 level; metric equivalents: in. × 25.4 = mm, ft × 0.3048 = m

In the southern part of the orebody the hanging wall is tuffaceous andesite and andesite. In this area the contact is generally sharp and easy to follow. However, to the north there is a large chlorite schist zone that crosscuts the bedding and comes in contact with the massive sulfides. This is apparently due to hydrothermal alteration of the andesite. The chlorite schist is highly mineralized with chalcopyrite and pyrite and quite often forms economic pockets of ore.

In the massive sulfides the chief ore minerals are sphalerite and chalcopyrite. Pyrite is the predominant sulfide, with considerable pyrrhotite throughout. Bright arsenopyrite ouhedrons in fine-grain massive sulfides are quite common. Occasionally small amounts of galena are seen, usually near the foot or hanging-wall contacts. On rare occasions tennanite is associated with massive arsenopyrite. Minor amounts of quartz, calcite, and unreplaced remnants of sericite schist occur, but essentially pyrite is the gangue in which the ore minerals occur.

The ore values are in excess of 3.5% copper and 12.5% zinc with some silver and rare gold as by-products.

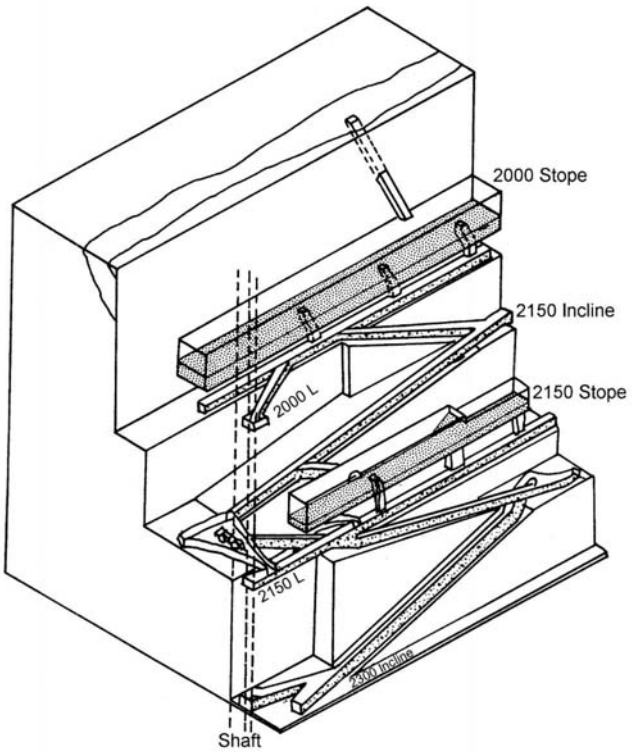


FIGURE 28.2 Diagram of the Bruce mine 2000 to 2300 level incline (not to scale)

Ground Conditions

The massive sulfides are generally self-supporting. One exception is in the 1850 stope where the orebody is 9 to 11 m (30 to 55 ft) wide and 152 m (500 ft) long. There are flat to shallow dipping slips and seams in the ore, creating extremely blocky ground. For support, old 25.4-mm (1-in.) hoist ropes were installed, tensioned to 27 t (30 st), and then cement grouted over the entire length in longholes [14 to 15 m (40 to 50 ft) in length] drilled on 3-m (10-ft) centers from the level above. This has tied the formation together very successfully and virtually eliminated the blocky ground condition. Both the hanging wall and footwall are quite shaley in some areas.

Reasons for Adopting Trackless Open Cut-and-Fill Methods

First, any method other than open cut-and-fill would have caused too much dilution.

The use of rubber-tired mining equipment in the production stopes requires a footwall ramp. The inclines in ore will be mined out, so this ramp in the footwall will provide access to and from the stopes (Fig. 28.2). This incline is very expensive, but necessary to convert existing stopes to LHD mining.

The final cost of ore mined by the LHD machines has not been determined. As of 1972, tons per miner-shift in the 2150 stope—the only one to complete a full cut—had increased from 7.58 (8.36 st) to 12.83 (14.14 st). Even though the first long section of footwall incline was charged against the stope, and labor charges rose $0.63 per hr, the costs show a savings of $0.55 per ton. Besides the increase in tons per miner-shift, a savings is being realized by being able to drop timbered chutes and manways. In the past, three manways and four or five chutes had to be maintained per stope, but with the LHD system, only two manways are required, one at each end of the stope.

The LHD machine eliminates the need for a slusher, which requires pin holes (extra drilling), sheave blocks, and a great deal of cable. This in turn eliminates pulling chutes on the level, that is, motor, car, track, and chutes.

Using the loader in the stopes allows the use of a self-propelled rubber-tired two-boom Alimak drill jumbo. This unit is equipped with two high-speed Tamrock drifters that drill 3-m (10-ft) holes [34.9 mm ($1\frac{3}{8}$ in.) diam] in the ore in 45 to 60 sec. Drilling with this jumbo allows longer and faster rounds. Larger machines can be handled with ease and there is less machine maintenance due to the fact that the machine is never being thrown in the muck. The stuck steel problem is ended with power retraction.

The ramp system also allows easy access to the stopes. In the past, everything had to be lowered on ropes or cables to the stopes. Then, once equipment and materials were in the stope, they had to be carried on employees' backs. Each man had to climb ladders to gain access to the stope, which was no easy chore.

The Bruce mine has three Eimco 912 LHDs and one Alimak drill jumbo. Without footwall inclines, when a loader or jumbo breaks down, the heading is down. When the ramps are connected, it is then possible to replace the broken piece of equipment.

DETAILED DESCRIPTION

Development

A diagram of the shaft, showing a month's ore development, is given in Fig. 28.3.

Drifts and Crosscuts. Main haulage drifts and crosscuts are driven 1.8 × 2.4 m (6 × 8 ft), with Gardner-Denver 58 and 63 jacklegs. Mucking is done with Eimco 12B loaders. Throughout the mine, 15.88-kg (35-lb) rail is used at a gage of 609.6-mm (24-in.). A 50.8-mm (2-in.) air line, a 25.4-mm (1-in.) water line, and a 110-v blasting line are installed at the time the drifts and crosscuts are driven.

For ventilation, a portable fan is set up and 304.8-mm (12-in.) diam Nyprene tubing is carried as the drift advances.

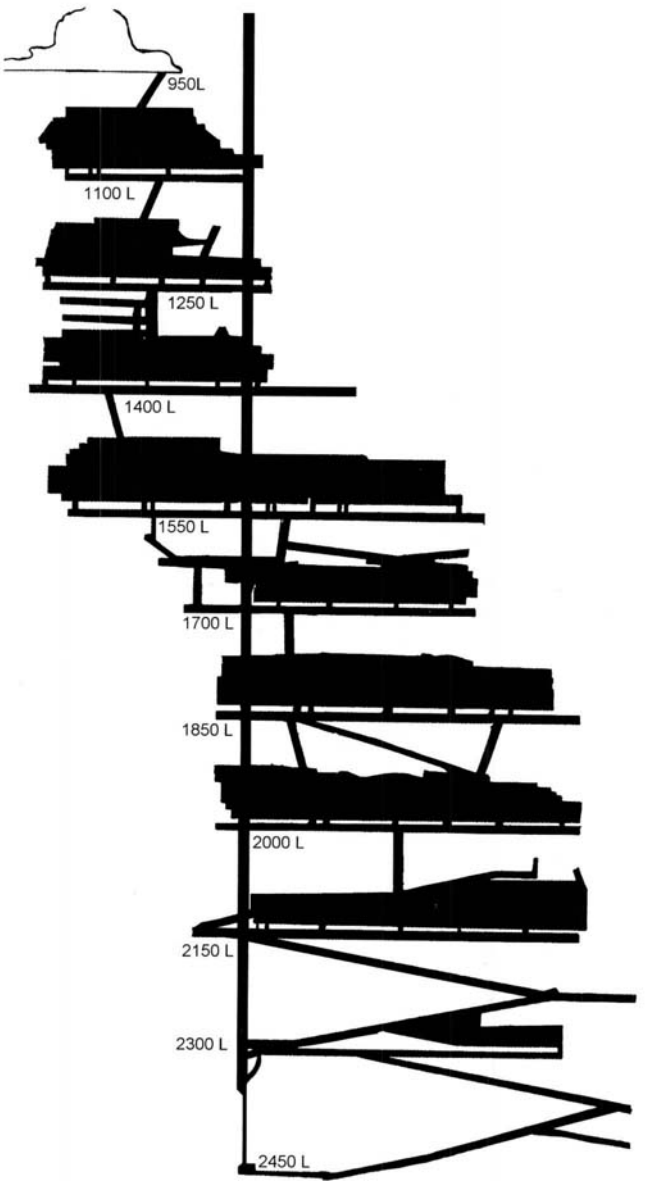


FIGURE 28.3 Bruce shaft showing ore development for a month

Raising. Initially 1.2 × 1.5-m (4 × 5-ft) raw raises are driven from one level to the next. Only pins, stulls, and ladders are put in. Using Gardner-Denver RB-53 stopers, 1.5 or 2.1-m (5 or 7-ft) rounds are drilled. A round consists of about 25 holes and is loaded and blasted with 28.6 × 203.2-mm ($1\frac{1}{8}$ × 8-in.) 40% Amogel stick powder and electric delay detonators.

After the raise has been holed through to the next level, it is enlarged to 1.5 × 3 m (5 × 10 ft) from the top down with jackhammers. A manway with

offset ladders, timber slide, and a sand-fill line is then installed. The completed raise, besides providing service to the stope, serves as a second exit and becomes part of the ventilation system.

Stoping Preparation. After the drift is run the entire length of the orebody and the ventilation raise is driven up to the next level, stope preparation can begin. Approximately 4.6 m (15 ft) from the ventilation raise, a parallel raise is driven 7.3 m (24 ft) high. A 2.7-m (9-ft)-high sublevel is then connected between them. This establishes a 4.9-m (16-ft) drift pillar and a temporary orepass. A two-drum slusher is set up, and the sublevel is driven the entire length of the orebody. Because the massive sulfides are quite competent and require no support, the sublevel is driven the full width from footwall to hanging wall at the same time. While driving the sublevel, a raise is connected to the main drift every 27.4 m (90 ft) to reduce the slushing interval. These are later enlarged for one-, two-, or three-compartment crib chutes that are framed out of 203.2 × 203.2-mm (8 × 8-in.) Douglas fir timber. The inside dimensions of the orepass compartments are 1.2 × 1.2 m (4 × 4 ft).

At both ends of the orebody, crib manways 1.1 × 1.1 m ($3\frac{1}{2} \times 3\frac{1}{2}$ ft) ID and timbered out of Douglas fir are carried up with the stope. These provide access to either end of the stope and are part of the ventilation and second exit system.

After the sublevel is silled out, another 2.7-m (9-ft) horizontal cut is removed, leaving an opening 5.5 m (18 ft) high for the entire length and width of the orebody.

The timbered chutes and manways are then installed. Extreme care must be taken to obtain a tight cement seal around these chutes at the start. Once sealed, the crib is then raised to within 0.9 or 1.2 m (3 or 4 ft) from the back of the stope. Two layers of 283.5-g (10-oz) burlap are wrapped around and stapled to the crib. At this point the stope is ready for the initial sand-fill pour (Fig. 28.4).

The strike length at the 2150 level of the ore was estimated at 152.4 m (500 ft); the normal level intervals in the Bruce mine are 45.7 m (150 ft). With these figures as a reference, an incline was driven on a −15% grade from the shaft end (north end) of the orebody to the south end. At the south end, a switchback was cut and the incline driven back to the north. When the correct level interval was reached, a crosscut was driven to a point directly under the shaft. From this point, a small [1.2 × 1.2-m (4 × 4-ft)] raise was driven to connect the shaft. The raise was then slabbed from the top down and the shaft sets hung. While this shaft work was being done, a 30.5-m (100-ft) incline was driven to a point directly under the shaft 6.1 m (20 ft), below the 2300 station. A raise was then holed through, and this provided skip loading room and a sump that could be mucked out with the LHD machines.

With this proven ore and the completion of the shaft and ramps, mine personnel were ready to start thinking of developing a stope. In such development, the first thing that was done was to take a cut the full width of the ore. This first cut defined the footwall and hanging wall contacts, as well as their competence and the competence of the ore.

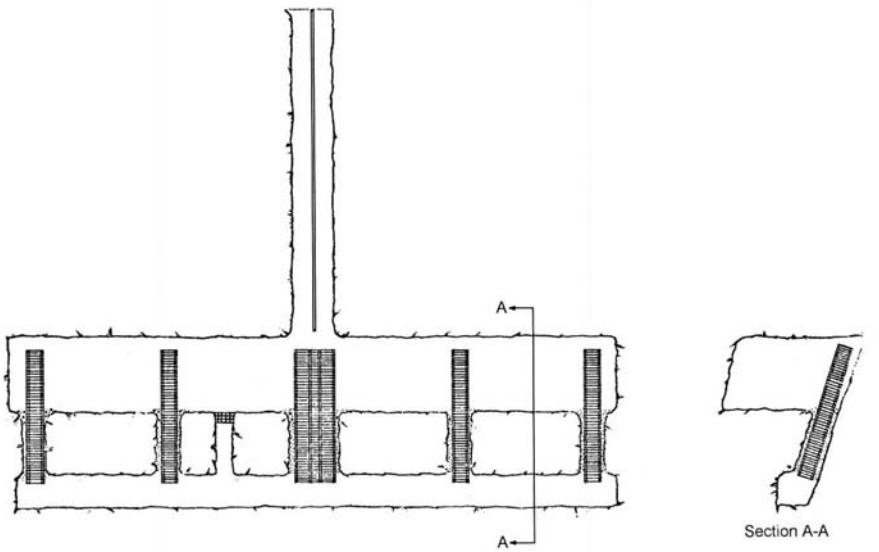


FIGURE 28.4 Diagram of a typical stope ready for sand fill

A 2.1 × 2.4-m (7 × 8-ft) level was then driven under the stope, leaving a 3.6-m (12-ft) pillar. Several drill holes were then drilled from the level to the bottom of the 2300 stope. These holes will provide drainage when the stope is in the sand-fill cycle. A raise was connected from the back of the stope to this level. This raise provides a second exit, ventilation, and additional drainage.

With this done, a section of ramp in the footwall must be driven. Done in this sequence, the waste created by this section of ramp can be used as fill in the first cut. The ramp has to be completed now because the existing ramp will be mined out.

This method of development has proven itself very advantageous to the mine because it eliminated much of the waste that would have to be hoisted. It also cuts down on the time that the shaft would be tied up by conventional shaft-sinking methods.

In exploration, this method of sinking inclines is very advantageous because the ore mined pays for the incline. It eliminates the need for a long crosscut and diamond drilling that is very expensive; these methods are not always precise and sometimes will miss the target or give a false indication of the thickness or grade of the ore. In the larger inclines you know what ore you have.

Stoping Cycle Detail

All stoping is by breasting down, using Gardner-Denver 58 and 63 jacklegs, rounds 1.8-m (6-ft) in length are drilled, using integral chisel bit 22.2-mm ($\frac{7}{8}$-in.) and hex. tungsten carbide drill steel. Drill holes are loaded with ammonium nitrate–fuel oil (ANFO) prills, using one stick of 25.4 × 203.2-mm

(1 × 8-in.) 60% Amogel powder per hole. Electric detonating caps are used throughout the mine. Better fragmentation is obtained in the stopes by using millisecond delays. Boulders are handled by drilling short plugs and blasting with stick powder, or if not too large by simply mudcapping them.

The usual cycle is to start in the center of the stope and establish a 2.1- to 3.6-m (7- to 12-ft) brow. This height is determined according to the firmness of the foot and hanging walls in order to minimize dilution. The cut is then mined to one end of the stope, all muck cleaned out, and equipment moved back to the center. While mining starts toward the other end, a gob fence is built, cribbing is raised and burlapped, and sand fill is started. In this manner the stopes, mainly the larger ones, can be maintained in almost continuous production.

A two-person crew is used in each stope. They handle all phases including drilling and blasting, slushing, pulling chutes, and tramming ore to the shaft pockets. On the upper levels where the ore is narrow, mining is on a one-shift-per-day basis. In the larger stopes, mining is carried out three shifts per day, seven days a week.

Ore Transportation. Two-drum slushers are used in all stopes at the present time. Where the vein is narrow, 7.5-kW (10-hp) double-drum air and electric Joy FF-211 slushers, pulling a 0.9-m (36-in.) wide rake, are used. In the wide stopes 18.7-kW (25-hp) Joy A2F-211 and B2F-211 electric double-drum slushers, pulling a 1.2-m (48-in.) Pacific rake, are used to scrape the muck into the chutes.

During crosscutting, drifting, and stope preparation, the muck is removed with Eimco 12B loaders, dumping into 1.1-m^3 (40-cu ft) Card rocker-type dump cars. Cars are trammed to the shaft pocket by a 1.4-t (1.5-st) battery-operated Mancha locomotive. After the cribbed chutes are installed, the Eimco 12B loaders are eliminated except for occasional cleanup work.

An Atlas Copco Cavo 310E load-haul-dump was tested in one of the large stopes for a four-month period to determine if tons per miner-shift could be increased over that obtained with the electric 18.7-kW (25-hp) slushers. Although the unit performed well, it did not meet Bruce mine requirements because:

1. Tons per miner-shift were not increased.
2. With the drilling method (breasting down) used in the mine, a slusher still had to be used. After a round is blasted, approximately one-third of the muck must be scraped off to make room to drill the next face. Alternative drilling methods were considered; however, they were not deemed desirable.
3. Because our sandfill contains no cement, the operators had a tendency, usually unavoidable, to dig too far into the sand, thereby diluting the ore.

Bruce mine operators have started using three-drum slushers, Ingersoll-Rand 25.4-kW (34-hp) air-operated, in the larger stopes. These units are more versatile, because the sheave block does not have to be manually changed for each setup. Tons per miner-shift should be substantially increased.

Ore is removed from the stope with 1.5-m^3 (2-yd) Eimco 912 LHDs and trammed directly to the shaft via the incline ramps.

Sampling Methods and Ore Control. Main drifts are channel sampled every 1.5 m (5 ft). Records are kept of the grade of ore removed from each cut in the individual stopes. These cuts are surveyed every two weeks. This, along with diamond drill holes and percussion holes, is used for estimating ore reserves. To be conservative, 0.28 m^3 (10 cu ft) per ton of ore in place is the factor used.

Each skip hoisted is sampled by the skip tender. A daily comparison is made between mine samples and the automatic head sampler at the mill. Tonnages are checked by the mill's weightometer. This, along with some selective mining, enables mine personnel to somewhat control the mill feed.

Backfilling Materials. Located adjacent to the Bruce shaft are two 136-t (150-ton)-capacity sand storage tanks and a 113,562-L (30,000-gal) water tank. The sand tanks can be filled in two ways:

1. Mill tailings are piped to a 101.6-mm (4-in.) cyclone by the sand tanks for desliming. Out of 181 t (200 tons) in 24 hr, approximately 91 t (100 tons) of sand are obtained, with the balance going to the tailings pond.
2. The Bruce sand plant is located below the berm of the old tailings pond. The tailings are picked up with a Payloader and conveyed to the repulper. In the repulper, 26.5 L/s (420 gpm) are required to handle 63.5 t/hr (70 stph) of sand. This is then pumped, using a 152.4-mm (6-in.) 29.8-kW (40-hp) Morris pump, to a 381-mm (15-in.) cyclone above the sand tanks. From the cyclone, 44.5 t/hr (49 stph) of sand and 5.3 L/s (84 gpm) of water go into the tanks and the remainder flows by gravity to a settling pond.

All overflow water is recycled back into the circuit. With both methods, mine personnel try to obtain sand that is 80% +200 mesh. The rest of the sand fill operation, including placement in the stope, is by gravity flow. A flow sheet of the operation is given in Fig. 28.5.

Underground Backfill. When a stope is ready to be filled, one worker on the surface will inject water and air into the base of one sand tank to break the cone and start the flow. A mixture of approximately 70% solids is sent underground through a 63.5-mm ($2\frac{1}{2}$-in.) rubber-lined schedule 40 pipeline.

At the shaft station on the level above the stope to be filled, the sand line is disconnected and a 1.5-m (5-ft) flexible rubber section is installed between the shaft and the drift sand lines.

Rubber-lined pipe is also used throughout the level and down the service raise to the stope. In the stope 50.8-mm (2-in.) plastic lines were first used to direct flow to the desired area. Currently, 76.2-mm (3-in.) pipe is used.

As previously mentioned, all openings around the timbered chutes and manways must be well sealed and the timber burlapped. From past experience, it

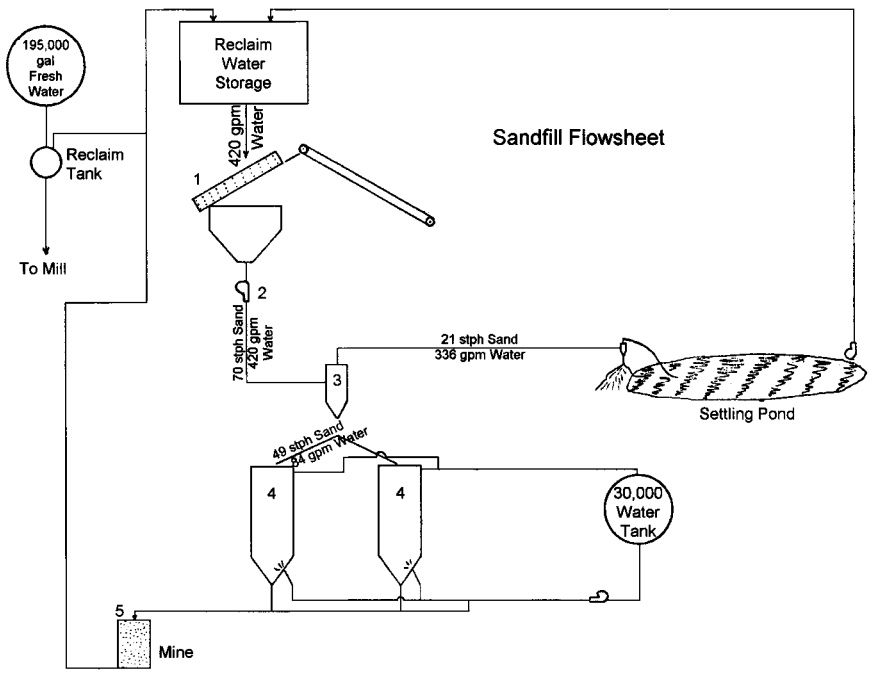


FIGURE 28.5 Flowsheet of the sand-fill operation; metric equivalents: gal × 3.785412 = L, stph × 0.9071847 = t/hr

has been found that an opening as small as 12.7 mm (½ in.) can drain tons of sand in a very short period of time.

Water generally percolates through the sand and drains off sufficiently so that mining can be resumed within a few days.

In the original plans, cement was going to be added to the tailings to form a hard floor in the stopes. However, due to the oxidation of old tailings, the desired hardness could not be obtained. American Cement Corp. did extensive test work on both old (oxidized) and fresh (nonoxidized) tailings. The test work showed that with a 1:15 ratio of cement and unoxidized tailings, a long period of time would still be required for hardening. After much study, it was decided to forgo the addition of cement because (1) a large enough supply of fresh tailings was not available, and (2) the economics of losing a small amount of ore in the sand, or dilution from it, seemed to outweigh the expense involved with the large amounts of cement that would be required.

Ventilation. Fresh air is downcast from the collar of the Bruce shaft to the 2150 level. Air doors on each level keep the airflow in the right direction. On the 950 level, which connects the Old Dick workings and shaft, a Joy 44.8-kW (60-hp) Axivane fan is mounted just beyond the raise that goes down to the next level. With the fan at the #6 blade setting, it pulls 20,060 L/s (42,500 cfm) from the inner workings of the mine and exhausts through the old Dick mine and shaft.

In the LHD stopes, 11.2- and 14.9-kW (15- and 20-hp) Joy incline blowers are used with 15.8-m (24-in.) vent tubing.

PRODUCTION UNITS

Labor

There are 56 people in production, including 5 supervisors, 4 hoisters, 3 skip tenders, 1 nipper, 1 carpenter, 1 service crew (2 people), 1 utility person, 26 miners, 9 muckers, and 4 jumbo operators. There are 14 people in maintenance.

Incentive System

All development and stope mining is on an incentive basis. Contract prices vary according to the size and type of opening, ground conditions, etc. Each working place is measured every two weeks by the engineering department. Bonuses are calculated by the mine superintendent and accounting department. Daily wages are always guaranteed.

Productivity

In 1975, the productivity in the slusher stopes was 7.1 t (7.81 st) per miner-shift and 12 t (13.22 st) per miner-shift in the LHD stopes.

COSTS

Capital expenditures (1975 figures) were: $817,000 for shaft and pockets, $394,000 for crosscuts and drifts, $93,000 for raises, $304,000 for inclines, and $1,003,000 for mine equipment.

Operating costs (1975 figures) per ton of ore by function were: stoping (drill, blast, and load), $12.71; haulage, $0.63; hoisting, $2.21; backfill, $2.21; ventilation, $0.33; pumping, $1.34; mine general, $0.97; mine safety, $0.65; and other, $3.64. Total operating costs per ton of ore were $24.69.

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CHAPTER 29

Cut-and-Fill at Homestake Mining Co.

Joel K. Waterland and Staff, Homestake Mining Co.

GENERAL DESCRIPTION

Homestake Mining Co.'s operations are at Lead, SD. Production from the various methods used are: 1,155,753 t/a (1,274,000 stpy) by open cut-and-fill; 68,039 t/a (75,000 stpy) by square set cut-and-fill; 144,242 t/a (159,000 stpy) by shrinkage stoping; 107,048 t/a (118,000 stpy) by blasthole stoping; and 67,132 t/a (74,000 stpy) in development work, for a total of 1,542,214 t/a (1,700,000 stpy). Mining takes place from the 616-m (2000-ft) level to the 2073-m (6800-ft) level. The average travel time to the stope is 45 min.

GENERAL OREBODY REQUIREMENTS AND LIMITATIONS

Size, Shape, and Dip of Orebody

Six major ledges varying from 914 m (3000 ft) long by 228.6 m (750 ft) wide to 305 m (1000 ft) long by 122 m (400 ft) wide are mined. Within these ledges are irregular-shaped areas of varying sizes. These ledges dip at approximately 1.2 rad (70°) to the east, and plunge to the south from 0.26 to 1.57 rad (15° to 90°).

Ground Conditions

The ore is cummingtonite (chlorite schist), which is very competent except for major bedding planes and fracture patterns. The footwall is a slate, but because of the dip presents very few problems. The hanging wall is also a slate, which because of the dip and bedding is very weak and requires considerable rockbolting. No problem has been encountered with rock bursts to a depth of 2438 m (8000 ft).

Ore Value

The ore values vary somewhat from ledge to ledge, but the average ore delivered to the mill is approximately 6.2 g (0.2 oz) of gold per ton. A small amount of silver is the only by-product of this operation. Annual production of gold is approximately 10,575,183 g (340,000 troy oz).

Production Requirements

The mine has a mill capacity of approximately 1,723,651 t (1,900,000 st), but because of several factors (labor, depth, etc.) this mill capacity has not been attainable.

Reasons for Adopting Open Cut-and-Fill

Open cut-and-fill methods are used primarily for grade control and ground control. This method permits selective mining of the ore so the waste can be left unmined in the stope.

DESCRIPTION OF MINING METHODS

Development

Above and below each orebody on each level, 2.3 × 2.3-m (7½ × 7½-ft) crosscuts are driven. A 1.5-m (5-ft) diam raise is then bored in ore connecting each of the levels.

Stope Cycle Detail

Stope mining is started from this bored raise at the lowest extremity of the ore, and progresses up plunge and up dip in 3-m (10-ft) thick horizontal slices to the upper extremity of the ore. If the orebody is longer (north-south) than 61 m (200 ft), the stope is mined vertically between preestablished stope lines.

Sampling. The gold occurs as very small particles of free gold finely disseminated throughout the rock. In order to determine the areas of economic value, each of these areas must be sampled extensively because visual examination is impossible. The initial sampling is done by drilling diamond drill holes on 30.5-m (100-ft) centers from the longitudinal drifts on each level in each ledge. The diamond drill holes are plotted on mine maps, and the areas that could contain gold (Homestake Formation) are assayed in 1.5-m (5-ft) increments. When rock of ore grade is found, secondary angle holes are drilled above and below the horizontal hole. These, in turn, are also plotted on mine maps after the ledge areas have been assayed. This diamond drilling is done before the crosscuts are driven.

After the first 3-m (10-ft) section of the stope has been mined, the ore removed, and backfill placed, the stope is sampled by drilling lines of 4.5-m (15-ft) holes across the back. These lines are oriented east-west and spaced 6 m (20 ft) apart in a north-south direction. The point of one hole is positioned directly over the collar of the next hole. Sludge samples are collected from the holes in 1.5-m (5-ft) increments; these are then assayed and the values plotted on maps to

guide the supervisors in mining the ore. The mining also requires additional sampling because no ore can be mined unless it has been sampled. Grab samples are taken from the pile in the stope and from loaded cars on the sill if there is any reason to believe the material is not ore grade.

Drilling. After the sampling has been completed, the complete back of the stope is drilled, using stope jumbos. These machines are mounted on two wheels with the third leg of the tripod being formed by the end of the feed shell. The feed shell is preset on either a 0.96 or 1 rad (55° or 60°) stope. This ensures that all holes are drilled parallel and to the same height above the floor. A 76.2-mm (3-in.) bore feed-shell mounted rock drill is used to drill the 31.75-mm (1¼-in.) blastholes, using 22.2-mm (⅞-in.) and 19.05-mm (¾-in.) hexagonal drill steel. The 19.05-mm (¾-in.) steel is used on the longer [4.5-m (15-ft)] holes to allow for springing the steel into the hole in the short headroom. The drilling pattern normally consists of rows of holes drilled across the stope (east and west) and spaced on 1.01- or 1.07-m (40- or 42-in.) centers (spread). The next row of holes is spaced 0.8 m (30 in.) (burden) from the preceding row. These rows are drilled until the ore area in the back of the stope is completely drilled. Drilling is usually done with four jumbos in the stope, and each miner operates two drills. This gives production of up to 304.8 m (1000 ft) of drilling per miner-shift.

When the drilling in the back is completed, the periphery of the stope must be mined by slabbing. This occurs in almost every stope because of the plunge and dip of the orebody. This slabbing is done with 76.2-mm (3-in.) jackleg drills drilling the same 31.75-mm (1¼-in.) holes. When limits of the ore have been attained around the periphery of the stope, the rows of upper holes in the back are extended out over the slabbed area using the jackleg machines. A break (slot) is drilled at one end of the stope into the back. This break is done by drilling several rows of 4.5-m (15-ft) holes across the stope and up into the back. The first row of holes is almost flat, and each subsequent row is at a steeper angle until the last row is parallel to the previously drilled upper back holes.

Blasting. The slabbing is normally blasted on the same shift that it was drilled. Ammonium nitrate prills are used as the blasting agent and are fired with small tubes of plastic explosive that fit over the millisecond electric blasting cap. The removal of the broken rock is generally done on one shift to allow the other shift to continue the mining.

After the slabbing is completed and all the upper back holes drilled, the break at the end of the stope is blasted. If the holes do not break completely, they are reblasted until the break is completely removed.

Large areas of the stope are then loaded. The size of the blast in the back is determined by the size of the stope, condition of the back, position of the rockway, etc. These blasts usually contain 200 or 300 holes, but may be as large as 1000 holes. The holes are primed with electric caps, both millisecond and regular delay, with the small tubes of plastic explosives (detaprimes) as primers. When loading is completed, the complete round is fired using 1000-cap condenser discharge blasting machines.

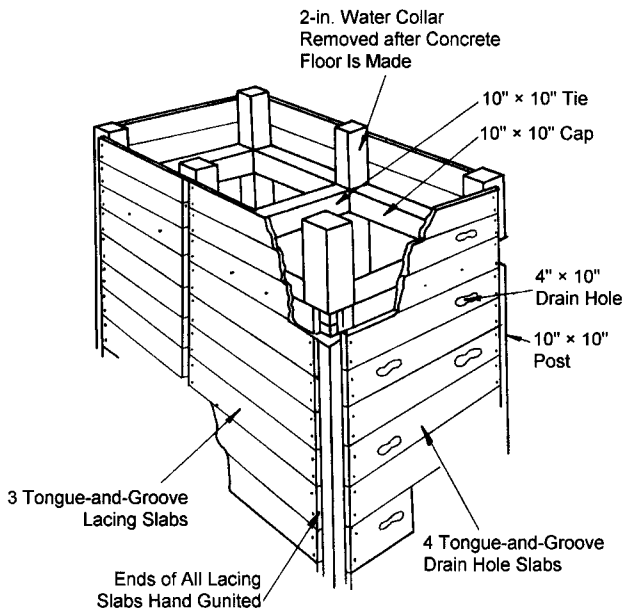


FIGURE 29.1 Standard two-compartment chimney using 0.17 rad (10°) vertical posts; one compartment is the rockway and one is the manway. Metric equivalent: In. × 25.4 = mm.

Removal of Broken Ore. When all the holes have been blasted in the stope, the back and walls are barred and dressed down, and any rockbolting that is needed is done. The broken ore is then removed with 18.6- and 22.4-kW (25- and 30-hp) triple-drum slushers and 12.2-m (48-in.) scraper buckets. In most cases, the grizzly bars are not installed at the top of the chimney (rockway) until the pile is broken through and sloped away from the hole. The 101.6-mm (4-in.) bars are then placed across the hole, leaving a 457-mm (18-in.) opening at each side and a 559-mm (22-in.) opening in the center. Usually one member of the crew is in the stope doing the slushing and the other member of the crew is on the sill below pulling the ore from the chute. As the ore is removed and the walls of the stope are exposed, subsequent rockbolting is done to stabilize the walls. Slushing is usually done by slushing down the complete pile rather than trenching the pile. If boulders are encountered that require secondary blasting, they are pulled aside and all blasted at one time. Production from the stopes during the slushing or clean-down cycle generally averages about 136 or 145 t (150 or 160 st) per shift.

Any waste material that must be mined in order to maintain the stope is usually gobbed in the stope and used as backfill.

Rockway or Chimney. The chimney (Fig. 29.1) usually consists of two sets of square-set timber (six posts). The posts are set on 1.8-m (6-ft) centers and can be leaned in any direction down to an angle of 0.96 rad (55°). One set is used as a rockway and one set is used as a manway. When the posts are stood, the outside of the chimney is laced with 76.2-mm (3-in.) tongue-and-groove slabs. The sixth side (generally in the manway set) is laced with 101.6-mm

(4-in.) tongue-and-groove slabs. Each of these 101.6-mm (4-in.) slabs has a 101.6 × 254-mm (4 × 10-in.) hole. These holes are used to decant water from the backfill. All seams on the sides of the chimney and around the connection to the chimney below are hand-gunited.

After the backfill has been made, the rockway is lined with 254 × 254-mm (10 × 10-in.) wooden blocks with the crosscut face of the block facing into the rockway. The section between the rockway and the manway is backed on the manway side with 50.8-mm (2-in.) slabs. Subsequent swelling of the wood locks the blocks in place.

Backfilling. When all the ore has been removed, waste mined, and the chimney stood, the stope is filled with mill tailings from which the slimes have been removed. It consists of the following particles:

1.0%	+80 mesh
8.3%	+100 mesh
17.4%	+150 mesh
22.4%	+200 mesh
45.9%	−200 mesh

This sand fraction is 60% of the material that is milled. The sand is sluiced from the 625-t (700-st) leaching vats and placed in a 4536-t (5000-st) holding dam. This material is then reslurried and sent into the mine through 152.4- and 127-mm (6- and 5-in.) rubber-lined pipes at approximately 60% solids. The sand is run through each of the two pipes at approximately 181 dry t/h (200 dry stph) of sand. When the sand reaches the stope, it is allowed to run into the void; because the sand settles from the slurry very rapidly, the clear water is drained from the stope through the previously installed drain holes. When the sand reaches the level of the drain hole, a drain board is nailed over the hole with a piece of burlap between the board and the lacing slab. This forces the water up and out of the next higher drain hole. When the sand reaches a point 152.4-mm (6 in.) below the top of the chimney, the sand is stopped. A 0.6-m (2-ft)-high water collar is then constructed around the top of the chimney using 50.8-mm (2-in.) slabs for lacing, and 152.4 mm (6 in.) of sand and cement slurry is poured into the stope. The cement is introduced into the sand at the surface. All water from the sand and cement mixture is held in the stope until the cement has settled. The water is then drained and the water collar removed. The complete floor of the stope is then raked to prevent slipping on the cement.

Loading Ore on the Sill

When the ore in the stope is slushed into the chimney it falls down the rockway to a chute at the bottom. This chute (Fig. 29.2) is a prefabricated wooden chute 1.07 m (42 in.) wide that is lined with steel plate. Four pieces of used anchor chain are suspended from the upper side of the chute throat and allowed to hang into the bottom of the chute. These chains retard the flow of rock as it passes under them.

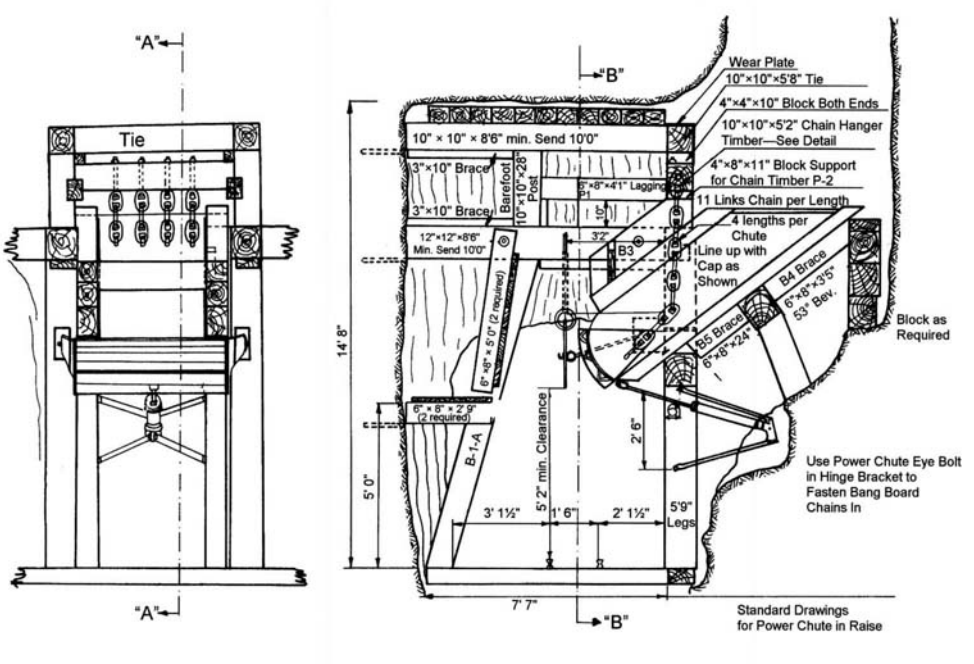


FIGURE 29.2 Standard drawings for the power chute in the raise. Left is front view of section B-B and right is front view of section A-A. Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; ° × 0.01745329 = rad.

At the lip of the chute is an arc undercut steel gate that can be raised and lowered by means of a pneumatic cylinder. The controls for the cylinder are installed in the set alongside the chute bench. A steel bang board is hung in front of the chute and over the car to deflect rock into the car.

Hauling Ore

By raising and lowering the gate the rock flow into 1.78-m^3 (63-cu ft) Granby-type cars is controlled. The cars are moved in front of the chute by a remote-controlled air tugger. When the train of cars (usually ten) is loaded, the cars are hauled to the ore dump over 457.2-mm (18-in.) gage track with 5.4- or 7.3-t (6- or 8-st) battery locomotives.

Ground Support

The ore in the mine is of a very competent nature, but the fracturing pattern requires that some rockbolting be done. Bolting that is done is usually on a selective basis, but in some stopes a 1.5 × 1.5-m (5 × 5-ft) pattern is followed. For bolting, 1.5- and 2.4-m (5- and 8-ft) expansion shell high tensile bolts are used. All stope hanging walls are bolted as the rock is removed. If the rock conditions are extremely poor, the ore is removed by square-set methods.

Stope Ventilation

Because of the depth and the complexity of the underground workings, 424,800 L/s (900,000 cfm) of fresh air are required in the mine. This

TABLE 29.1 Labor for the Homestake mine

Total mine labor	
Mine (including geology and safety)	1054
Metallurgical	109
Service departments (maintenance)	336
Purchasing and supply	15
Medical	36
Administrative and accounting	41
Total	1591
Underground labor	
Miners (stope and development)	506
Motor operators	90
Service	229
Shaft (including maintenance, operation, crushing, etc.)	67
Sand fill	31
Diamond drill	31
Salary (supervision, clerical, safety, and geology)	100
Total	1054

ventilation system consists of two separate exhaust air systems with both of the operation shafts and one air shaft being downcast or intake air. Each working place is equipped with a 7.5- or 11.2-kW (10- or 15-hp) fan that picks up the primary ventilation air and delivers it into the working place through 406.4- or 508-mm (16- or 20-in.) vent bags. Each working place must have a minimum of 2360 L/s (5000 cfm) of air. In working areas where heat is a problem [27.8°C (82°F)], refrigeration spot coolers of 27.2- and 54.4-t (30- and 60-st) capacity are used. The use of fans and regular wetting of the pile normally provide suitable ventilation. Blasting is always done at the end of shift to allow removal of the smoke and gas before the next shift.

PRODUCTION UNITS

Labor

Labor figures for the total mine and for the underground operations alone are given in Table 29.1. Underground maintenance for mechanical and electrical equipment is done by the respective departments and is separate from the mine department. Other maintenance as well as all underground construction is done by the mine department.

Incentive System

Approximately 90% of the ore production and development work is under an incentive plan by which the crews share equally in any incentive earnings. The incentive system usually has a sliding scale of four prices that are paid for each unit of production [cubic meter (cubic feet) of ore mined and loaded on sill, meters (feet) of drift, etc.]. After the costs of explosives and any hourly

TABLE 29.2 Production and productivity in the Homestake mine

Production		
Tons ore produced	1,497,256 t	(1,650,442 st)
Tons waste produced	160,790 t	(177,241 st)
Tons sand backfilled	707,561 t	(779,952 st)
Drift driven	7,684 m	(25,211 ft)
Raise bored	2,900 m	(9,515 ft)
Diamond drilling	22,523 m	(73,893 ft)
Productivity per miner-shift		
Square set cut-and-fill	10.34 t	(11.40 st)
Shrinkage stoping	13.05 t	(14.38 st)
Open cut-and-fill	14.28 t	(15.74 st)
Blasthole	37.48 t	(41.31 st)
Average	14.38 t	(15.85 st)
All mine, including mine salaried	6.11 t	(6.73 st)
Homestake mine, including all salaried	4.18 t	(4.61 st)

wages paid are subtracted, the miners then are paid the incentive pay according to the hours worked in any contract working place. Incentive wages normally represent an increase of 75% above the average miner's hourly wage.

Backfilling, sampling, haulage, and mine maintenance are generally done at a basic hourly wage.

Stope Capacity

The active stopes, which normally number between 100 to 110, will ordinarily produce approximately 1270 t/month (1400 st per month). However, because of the cyclical nature of open cut-and-fill mining, the variation in stope size from stope to stope, and in a given stope from lift to lift, it is very difficult to predict or program what the production will be. Production can be increased only by increasing the number of stopes being actively mined.

Production and Productivity

Total production and productivity per miner-shift at the Homestake mine are given in Table 29.2.

Equipment

Equipment used in the mine is listed in Table 29.3.

Stope Cycling

Figure 29.3 is a graph that illustrates the time spent in each phase of the open cut-and-fill mining operation.

TABLE 29.3 Equipment used at Homestake mine

Description	No.
Jackhammers, 76.2-mm (3-in.) bore	482
Feed legs, 1.63-m (6-ft) telescopic	395
Power shovels	73
Slushers, 18.6 and 22.4 kW (25 and 30 hp)	116
Slusher buckets, 12.2 m (48 in.)	85
Locomotives	
7.3-t (8-st), battery	4
5.9-t (6.5-st), battery	16
5.4-t (6-st), battery	36
5.4-t (6-st), trolley	3
1.8-t (2-st), battery	2
0.9-t (1-st), battery	2
Granby-type cars, 1.78 m^3 (63 cu ft)	692
Rocker cars, 0.91 m^3 (32 cu ft)	560

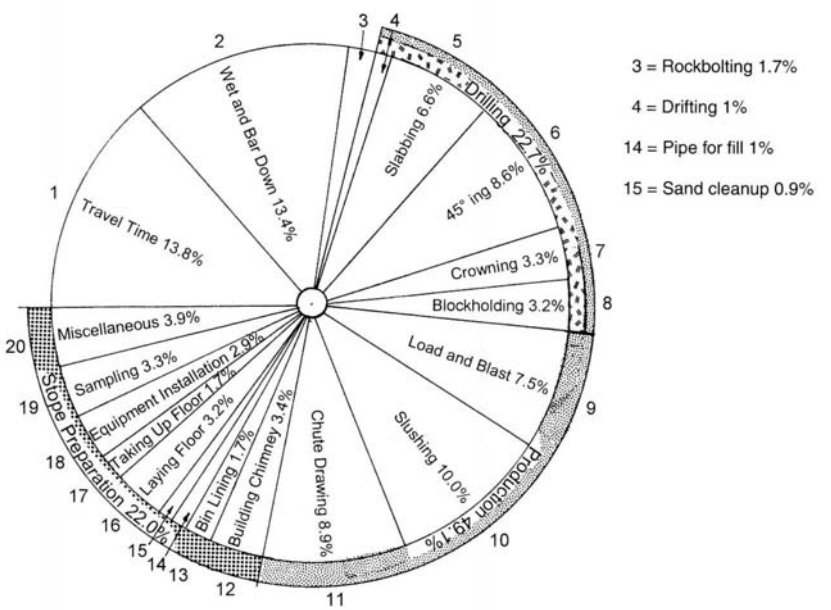


FIGURE 29.3 Time cycle for operations in Homestake's open cut-and-fill stope

TABLE 29.4 Direct operating costs at Homestake mine, late 1970s

	Costs per Ton of Ore, $
Development	2.04
Stoping, loading, backfilling	5.86
Haulage	1.76
Other mine operating	2.22
Mine general	2.78
Hoisting and shafts	1.42
Crushing	0.32
Surface tramming	0.12
Treatment	1.73
Administrative (including taxes, employee benefits)	6.14
Total	24.39

TABLE 29.5 Direct costs for open cut-and-fill at Homestake mine, late 1970s

	Costs per Ton of Ore, $
Labor	4.65
Supplies	0.18
Other services (maintenance department charges)	0.03
Timber	0.51
Explosives	0.33
Total	5.70

OPERATING COSTS

The Homestake mine is an old mine, more than 100 years old. The total capital expenditures over this period are undeterminable.

The direct operating costs (late 1970s figures) are given in Table 29.4. The direct costs (late 1970s figures) for the open cut-and-fill methods are given in Table 29.5.

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CHAPTER 30

Cost Calculations for Highly Mechanized Cut-and-Fill Stoping

Gordon M. Pugh and David G. Rasmussen

INTRODUCTION

The two unit operations in cut-and-fill stoping that would benefit most from increased mechanization are drilling blastholes and moving broken ore to the orepasses. Drilling in current practice is done with hand-held jackleg and stoper drills. Broken ore is commonly moved from the face to the orepass by a small slusher. This type of equipment can be easily moved through the raises.

Hand-held drills have a slow drilling rate and require one operator for each machine. Drilling can be done much faster by small jumbo-mounted machines because two or more drills can be operated by one miner.

Load-haul-dump (LHD) units move broken ore from the face to one or more orepasses at a much faster rate with less loss and dilution than slushers. These units are mobile and may be used at more than one place during a shift.

The problem that must be overcome in applying this higher degree of mechanization is the restricted means of access inherent in current practices. Cut-and-fill methods are usually applied to mining veins that are narrow, irregular in dip and strike, or where ground conditions require the support of sand fill. The mechanized equipment needed to accomplish high production cannot be raised or lowered through raises without disassembly.

A jumbo that will give the desired performance would be mounted on a rubber-tired carriage with a diesel engine for propulsion. Twin hydraulic booms run by a small air motor would carry pneumatic drills. Overall dimensions of the unit would be about 8 m (26 ft) in length, 1.6 m (5 ½ ft) in width, and 1.6 m (5½ ft) in height.

An LHD of the desired capacity would be rubber-tired and either diesel or electric-powered. The overall dimensions of a 2.3-m (3-cu yd) unit will be

about 8 m (26 ft) in length, 1.8 m (6 ft) in width, and 1.5 m (5 ft) in height. It will weigh about 12,700 kg (28,000 lb).

It can readily be seen that conventional raises cannot handle this size of equipment. The most desirable means of access enables equipment to be driven into the stoping areas. Ramps can enter the stope by being driven upward in the vein walls or by being developed in the vein on top of the fill as the stope advances. This ramp system not only permits the use of rubber-tired equipment in multiple working places but also enables its removal for repairs without component breakdown.

In the proposed method, as in conventional cut-and-fill stoping, sand fill will replace the mined-out ore. By constructing an inclined timber floor at one end of the stope, a ramp can be maintained on top of the fill and advanced as the stope is advanced cut by cut (Fig. 30.1). The ramp is therefore developed during the stoping process and is driven as a tunnel only when rising over and around the sill and crown pillars.

A stoping section consists of three blocks of different geometric shapes. The first and last blocks to be mined will be triangular in the plane of the vein, and the blocks between will have the shape of parallelograms (Fig. 30.1).

The first block will develop the initial ramp by breast stoping and filling to the ramp floor (Fig. 30.2). The ramp will need to be completed through to the next level before stoping of the subsequent block can begin. The subsequent blocks are mined by back stoping with the next ramp being developed at the other end of that block (Fig. 30.3). The length of each subsequent block can be varied without changing the slope of the ramp. As the new ramp is developed, the previous ramp is filled along with the stope cut. Access up and down is maintained through the stope. The final block is again breast stoped because no new ramp will be developed at the end line for jumbo egress.

Boreholes provide ventilation between levels as well as utility access. They can also provide a secondary escapeway for personnel during periods of development. These boreholes would be located about 3 to 4.5 m (10 to 15 ft) in the footwall and spaced about 183 m (600 ft) apart along the vein.

The orepasses are spaced about 61 m (200 ft) apart in the vein. This spacing allows not only a short tram for the LHD unit but also provides surge capacity for the ore between haulage trains. These passes are raised from the haulage levels. As development progresses, the raises are driven through the sill pillar, and then carried in the fill as stoping progresses. Ventilation and utilities can be maintained through an orepass when it is not needed for ore. The end line orepasses are intended for this purpose as well as for personnel emergency egress.

Overall dimensions and criteria assumed to illustrate the ramp method of access in a highly mechanized cut-and-fill stope are haulage level spacing of 61 m (200 ft); ramp gradient of 17%; average dip of approximately 1.57 rad (90°); initial stope length of 358 m (1177 ft); subsequent stope length of 179 m (588 ft); average stoping width of 3 m (10 ft); sill pillar thickness of

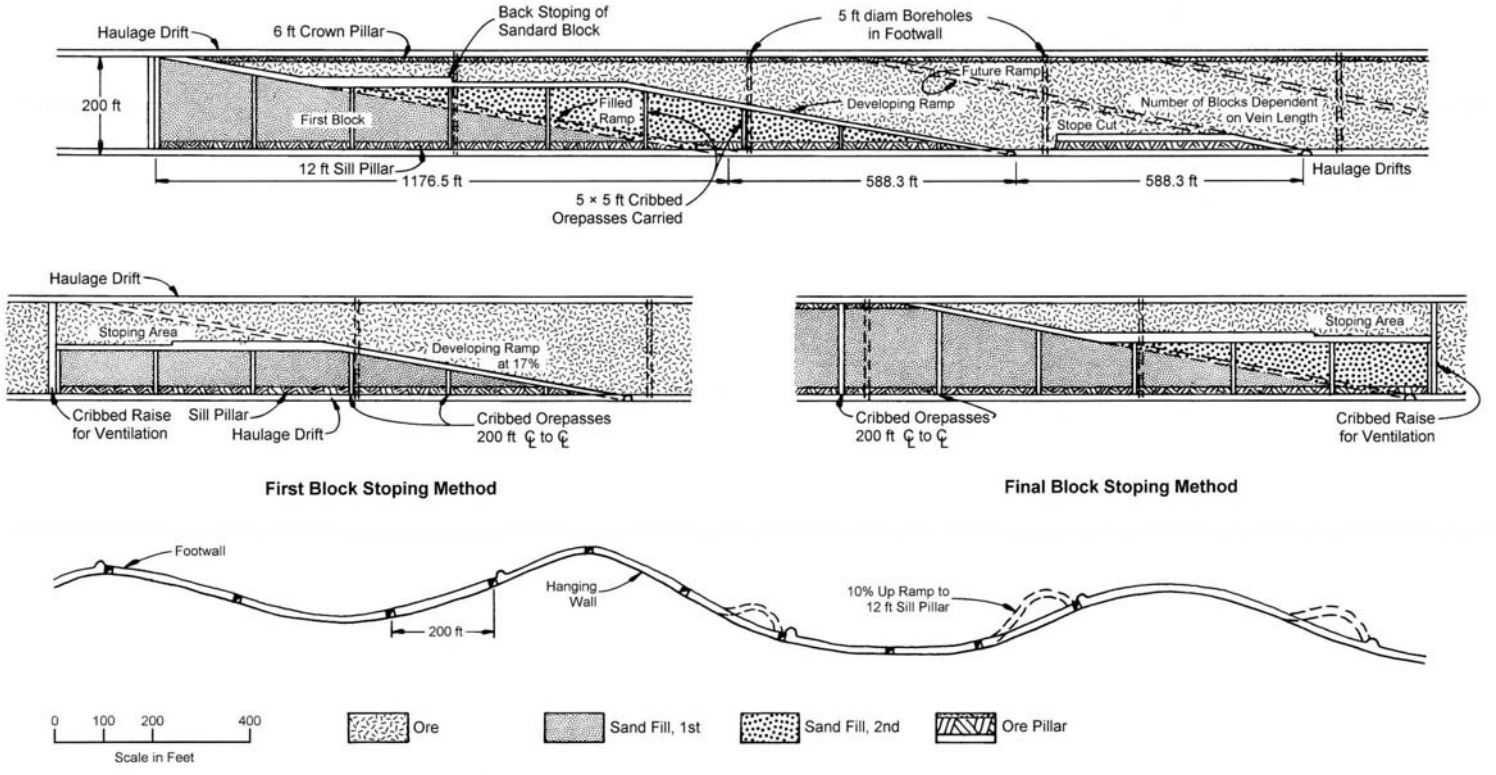


FIGURE 30.1 Highly mechanized cut-and-fill stoping, metric equivalent: ft × 0.0348 = m

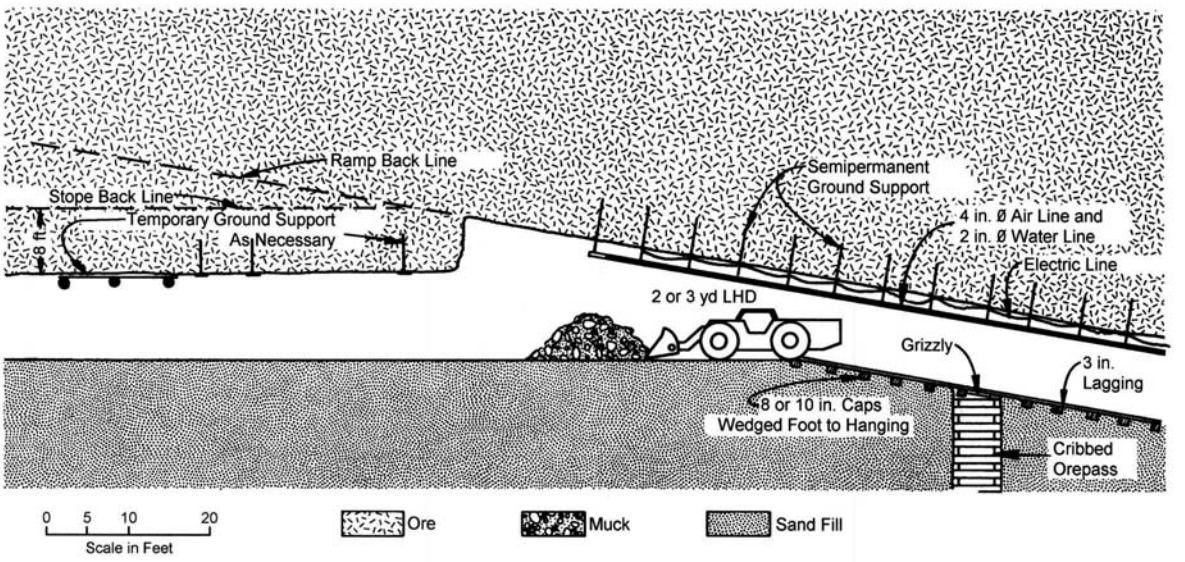


FIGURE 30.2 First block unit operation, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; yd × 0.9144 = m

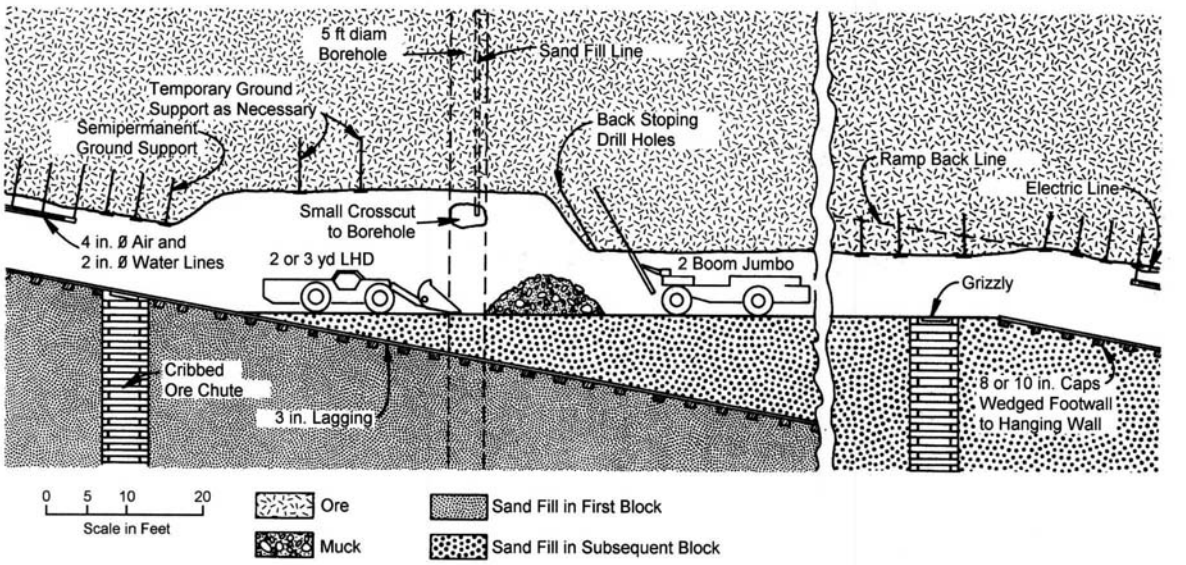


FIGURE 30.3 Back stoping operations, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; yd × 0.9144 = m

3.6 m (12 ft); crown pillar thickness of 1.8 m (6 ft); bored raises of 1.5-m (5-ft) diam; and cribbed orepasses of 1.2 × 1.2 m (4 × 4 ft) through the sill pillar and 1.2 m (4 ft) square inside the cribbed portion.

DEVELOPMENT

The ramp access system lends itself well to conducting the several phases of development concurrently.

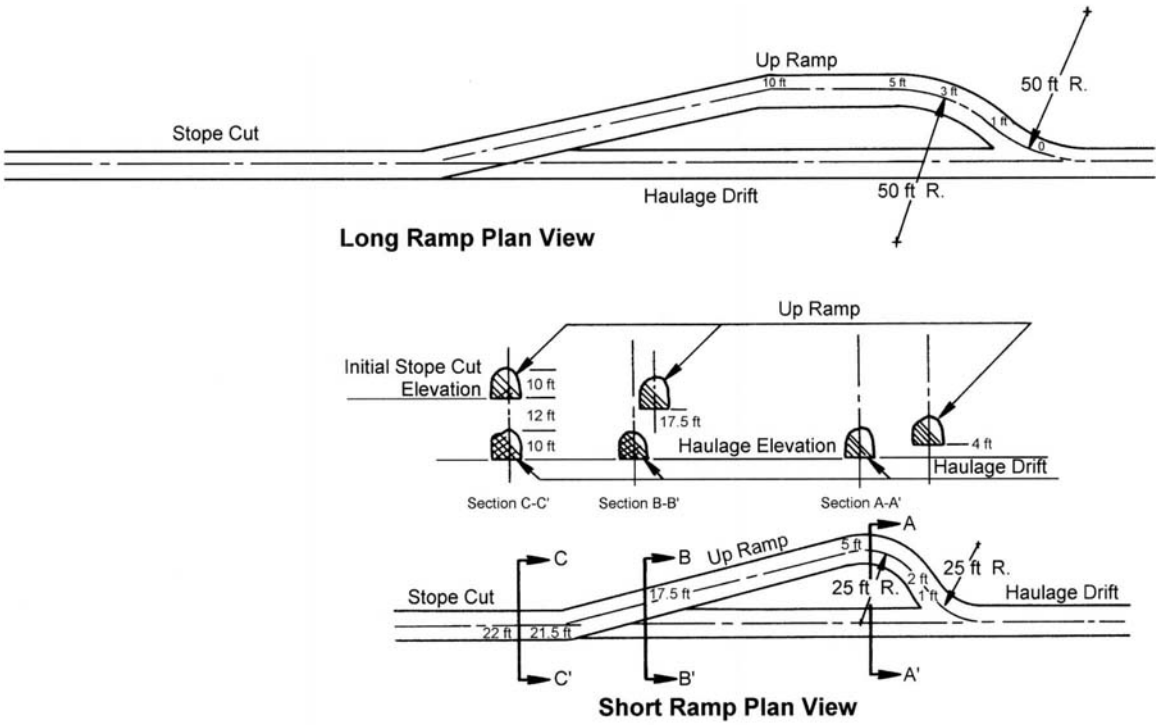


FIGURE 30.4 Typical up ramp turnouts, metric equivalent: ft × 0.3048 = m

Haulage Drifts and Raises

The first development step will be to drive the 3 × 3-m (10 × 10-ft) haulage drifts above and below the stope blocks. The boreholes can be made from short footwall crosscuts while the drifts progress. The orepasses can also be driven from the lower drift through the planned sill pillar.

Sill Pillar Up Ramps

The up ramps can be started as their locations are passed by the haulage drifts. They will crosscut into the footwall at angles from 2.09 to 2.35 rad (120 to 135°) with the haulage drift direction. The ramp will curve back toward the vein at an average gradient of about 0.2 rad (10%), then steepen to 0.3 rad (17%) on the straight segment. Fig. 30.4 shows two plans; one ramp is 57 m (188 ft) in length and the other 88 m (290 ft). The difference in length is due to the curve lengths and gradients. Actual design lengths could fall between these distances depending on ground conditions or other factors. In this chapter, the longer length was used for all calculations.

Initial Cuts

The initial stope cut is advanced in the vein above the sill pillar, and because it involves drift-type breaking and mucking, the cut is considered to be development. The orepass raises will be broken into as the cut is advanced to provide short tramming distances. A grizzly will be installed at the raise collar. The raise at the end line will be used for ventilation access.

STOPING

One of three stoping systems will be used, depending on the individual block's position within the vein length. The ramp is developed on the fill, through to the upper drift by the first stope. The subsequent stope operations then backfill the former ramps and develop new ones along their endlines. The final stope is then worked upward by filling the last ramp without developing a new one at its endline.

First Block

Stoping in the first block progresses by enlarging the area above the projected ramp line to form a 2.4-m (8-ft) high face, then by breasting down 2.4-m (8-ft) deep rounds with a two-boom jumbo. Calculations indicate two rounds should be made for each mining shift. LHD units will move the broken ore to the orepasses. They can be either diesel or electric and have a capacity between 1.5 and 2.6 m^3 (2 and 3½ cu yd).

When a cut is completed to the endline, the equipment and utility lines are removed, and the cribbed orepasses are built up to the next floor level. The ramp gradient is projected up the rib walls and holes drilled to support the timber caps on 1.2-m (4-ft) centers. The caps are wedged between the walls and cradled in "bull horns" inserted in the drill holes. A deck of 76-mm (3-in.) thick planks is then nailed to the caps. Burlap covers the underside of this ramp chinked into any openings. Fill can then be run into the stope for its full length with care to be taken to see that it fills tightly under the ramp decking (Fig. 30.2). A ramp 14.6 m (48 ft) in length is constructed for each 2.4 m (8 ft) of fill height.

The remaining stope cuts will proceed in a similar manner until the crown pillar is reached. The ramp is then turned into the footwall again and driven to the elevation of the upper haulage drift. At the sill elevation this tunnel intersects the drift. The last stope cut can then be filled tight to the crown pillar.

First sand fill is essential to the success of this method. Sand that is as free of slimes as possible should be used for rapid draining and quick setup. To ensure a hard traveling surface, it may be necessary to add portland cement to the upper few inches of fill.

Subsequent Blocks

While the first stope is still producing, the up ramp and the initial stoping cut should be advanced in the next block. However, stoping cannot begin until the up ramp in the preceding stope is completed through to the upper level.

The first ramp must be filled to the floor level of the initial cut to provide through access. Back stoping can then begin at either end of this stope. The two-boom jumbo will drill up holes in a pattern of four holes wide for 9 to 12 m (30 to 40 ft) along the vein in each shift to break a 2.4-m (8-ft) cut. The broken ore will be moved to the nearest orepasses with an LHD. Drilling and mucking can therefore continue simultaneously without being isolated by a muck pile, because access can be maintained by way of either ramp.

Fill will be added in much the same manner as it was in the first stope. The portion of the up ramp for the preceding stope below the stope level must be filled along with the stope. As the stope progresses upward, the utility lines can be removed from the ramp to be filled and added to extend the lines up the near ramp.

Final Block

The final block will not develop a new ramp as did the former blocks. Because there will be no new ramp, breasting rounds will probably again be used to avoid trapping the jumbo behind a muck pile. Filling of this block will be simplified as no new ramp deck will need to be constructed.

APPLICATION MODELS

This general concept can be used in many situations and orebody dimensions. To be able to estimate costs, productivity, and other comparable measurements, however, certain criteria were assumed to provide a model. These were as follows: daily production of 725.7 t (800 st); 240 operating days per year; a vertical shaft 1067 m (3500 ft) deep; an average vein thickness of 3 m (10 ft); an average selling value of ore of $40.00 per ton; an initial mine development time of two years; a total mine life of 17 years; costs and rates as derived in the earlier portion of this study; all blastholes drilled with a two-boom jumbo; the broken ore is mucked and trammed by a 2.6-m^3 (3-cu yd) LHD; rockbolts are used for ground support; stope cuts are 2.4 m (8 ft) high; stope fills are made after each cut; all development headings 3 × 3 m (10 × 10 ft) are untimbered but rockbolted; rail haulage for all ore from the stopes is adequate to maintain continuous production; the raise boreholes are done by a contractor and are to be drilled in the footwall at 183-m (600-ft) intervals; the maximum ramp gradient is 0.3 rad (17%); the cribbed orepasses are carried in fill at 63-m (208-ft) intervals; haulage levels are 61 m (200 ft) apart for calculation purposes, but this concept could enable greater spacing; and all ore excluding the pillars is taken. Costs for the drill-and-blast simulation model are shown in Appendix B of this chapter.

COSTS AND PRODUCTIVITIES

For detailed cost calculations, see Appendix A of this chapter, and Pugh and Rasmussen (1977), Section 2.2, Chapter 13; Section 3.1, Chapter 4; Section 3.2, Chapter 10; Section 3.3, Chapter 6; and Section 3.4, Chapter 7. All figures are based on values from the mid to late 1970s.

Table 30.1 gives the cost summaries and productions from each of the three stoping block types. Table 30.2 gives breakdowns of both the development costs and the stoping costs for the first block type. This block will yield about 83,440 t (92,000 st). Table 30.3 shows similar information for the subsequent block types for production of about 78,000 t (86,000 st) per block. The final block cost breakdown is shown in Table 30.4 for production of about 62,130 t (68,500 st). The first block is estimated to be the most expensive to mine because it involves breast stoping and high development costs. The subsequent

TABLE 30.1 Summary of mining costs

	Cost, $	Minable Tonnage*†	Cost Per Ton,†$
First block	639,343	92,069	6.95
Subsequent block	464,186	86,262	5.38
Final block	308,375	68,501	4.50
Total	1,411,904	246,832	5.61

*Excluding sill and crown pillars to give a recovery of 89.4% of reserves in place.

†Metric equivalent: st × 0.9071847 = t.

‡Operating costs only. Does not include amortization of capital, etc.

TABLE 30.2 Summary of first block mining costs

Mining Operation	Tonnage Produced*	Cost, $	Cost Per Ton, $	Cost Per Ton Prorated Over Block Tonnage, $
Haulage drift	10,308	125,603	12.19	1.36
Ventilation boreholes	436 cu yd waste	33,060		0.36
Drive ramp through sill plate	1074 cu yd waste	23,548		0.26
Drive ramp through crown pillar	1074 cu yd waste	20,694		0.22
Initial stoping cut	8748	74,214	8.48	0.81
Drive orepasses	146	4,729	32.84	0.05
Install chutes		16,000		0.17
Total development†	19,202	297,848	15.51	3.23
			Cost Per Ton Stope Ore, $	
Breast stoping	72,827	195,176	2.68	
Build timber ramp		21,865	0.30	
Timber operations		19,980	0.27	
Backfill		91,034	1.25	
Miscellaneous		13,440	0.18	
Total stoping	72,827	341,495	4.68	
Total production	92,029	639,343		6.95

*Metric equivalents: cu yd × 0.7645549 = m^3; st × 0.9071847 = t.

†Except where noted all development is done in the vein.

blocks are less expensive to stope because back stoping can be used, although the development costs are still high because of the smaller tonnage developed. The final block will be the least expensive to mine because the applicable development costs are very much lower than the other two types.

The costs derived were calculated, based on knowledge of both the productivity and cycle time elements, for each of the mining methods being considered. Costs for equipment, fuel, tires, and other expendables are 1976 figures, and equipment hourly operating costs for each piece of equipment are included at the end of the entire cost section. The labor cost used has been fixed at

TABLE 30.3 Summary of subsequent block mining costs

Mining Operation	Tonnage Produced, st*	Cost, $	Cost per Short Ton, $	Cost per Short Ton Prorated Over Block Tonnage, $
Haulage drift	4,860	60,678	12.49	0.70
Orepass raises	73	2,227	30.50	0.03
Bored vent raise	Waste	11,020	—	0.13
Ramp and initial stope cut	4,172	57,445	13.77	0.67
Orepass chutes	—	9,600	—	0.11
Final ramp	Waste	20,694	—	0.24
Total development†	9,105	161,664	56.76	1.88
Drill and blast (back stoping)	77,157	79,472	1.03	
Mucking	—	27,005	0.35	
Temporary ground support	—	49,380	0.64	
Build ramp	—	20,176	0.26	
Crib raises	—	16,603	0.22	
Backfill	—	96,446	1.25	
Miscellaneous	—	13,440	0.17	
Total stoping	77,157	302,522	3.92	5.38
Total production	86,262	464,186		

*Metric equivalent: st × 0.9071847 = t.

†Except where noted all development is done in the vein.

TABLE 30.4 Summary of final block mining costs

Mining Operation	Tonnage Produced, st*	Total Cost, $	Cost per Short Ton, $	Cost per Short Ton Prorated over Block Tonnage, $
10 × 10-ft* bulkhead	68,493	308,586	39.75	0.01
Orepass	16	636	39.75	0.01
Orepass chute	—	3,200	—	0.05
10 × 10-ft* bulkhead	—	208	—	0.01
Initial stope cut	243	2,141	8.81	0.03
Total development	259	6,185	23.88	0.11
Breast stoping	68,234	182,867	—	2.67
Backfilling	—	85,292	—	1.25
Cribbed raises	—	20,592	—	0.30
Miscellaneous (engineering, salvage, etc.)	—	13,650	—	0.20
Total stoping		302,401	—	4.42
Total production	68,493	308,586		4.53

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

$10.50 per hr, based on average rates at underground mines in the United States and including 35% markup for fringe benefits and 25% for an incentive payment system.

The total mining cost, based on these assumptions, consists of two parts: development costs and stoping costs. Development costs are determined by first creating a development plan of drifting, raising, and chutes or mill holes. Several specific development plans are considered for each stoping method. Prior to this, appropriate unit costs were applied to the development plan. Total costs calculated from dollar-per-foot figures were then expensed over the ore block directly associated with the development. The development costs do not include shafts, stopes, or other major support systems, but only the costs incurred for the development of the specific tonnage block.

Stoping costs were calculated, using derived unit operation costs, for drilling, blasting, mucking, stope preparation, and ground support installation when required. Selection of proper unit operations was made for specific stoping cases and their related costs compiled to give a stoping cost.

It should be reemphasized that costs given include labor, direct operating costs of equipment, and expendables. These costs have been allocated to the assigned tonnage for the example stoping block. Equipment capitalization and depreciation have not been included.

The stope productivity measured in tons per miner-shift will depend on the blasting technique employed. The first and the final blocks require breast stoping, and for this method 30.2 t (33.3 st) per miner-shift was estimated to be the productivity. The intermediate or subsequent blocks are back stoped to produce about 45 t (50 st) per miner shift. Table 30.5 gives a derivation of these estimates.

This ramp method of cut-and-fill stoping will be best suited for very long ore shoots in veins or continuous beds. Any number of the subsequent blocks may be added to extend the stoping area. The lengths of these intermediate stopes may be extended beyond the 179 m (588 ft) shown in this model.

Short ore shoots can also be stoped using this ramp system by three different approaches: the vertical distance between the haulage levels can be reduced, which may be practical for operating mines with established close-level spacing, but the development costs would be somewhat increased on each ton of ore produced; the ramp gradient could be steepened (most of the available rubber-tired equipment can travel grades efficiently up to 20%); or the ramps can be made to switchback to gain vertical distance to whatever haulage level spacing is desired.

An example for this variation on the ramp system as shown in Fig. 30.5 has a strike length of about 244 m (800 ft) and a level separation of 165 m (540 ft). The ramp switchbacks turn into the footwall at right angles to make level stations. To make the reversed continuation, the up ramp would be started in the footwall parallel with the vein, then turned into the vein. A sturdy pillar will be left between the ramps. The ramp would be developed through to the

TABLE 30.5 Summary of productivity estimates for highly mechanized cut-and-fill stoping

Calculations of the number of personnel required to produce 800 stpd*

Backstoping

200 st per shift per stope is achievable on a continuous basis

Stope crew = 1 jumbo operator
1 miner
1 LHD operator
3 per shift

Assume 2 stopes each being worked on 2 shifts per day to give 800 stpd
12 miners per day + 4 utility crew (stope preparation, salvage, piping, etc.)
Total = 16 miners per day

$$\text{Backstoping productivity} = \frac{800 \text{ st}}{16 \text{ miners}} = 50 \text{ st per miner-shift}$$

Breast Stoping

Assume 2 miners per stope per shift will produce 100 st (2 rounds per shift) so that 4 stopes worked on 2 shifts per day basis produce 800 st.

Stope crew = 1 jumbo operator
1 LHD operator
= 16 miners per day + 8 utility crew
= 24 miners per day

$$\text{Breast stoping productivity} = \frac{800 \text{ st}}{24 \text{ miners}} = 33.3 \text{ st per miner shift}$$

*Metric equivalent: st × 0.9071847 = t.

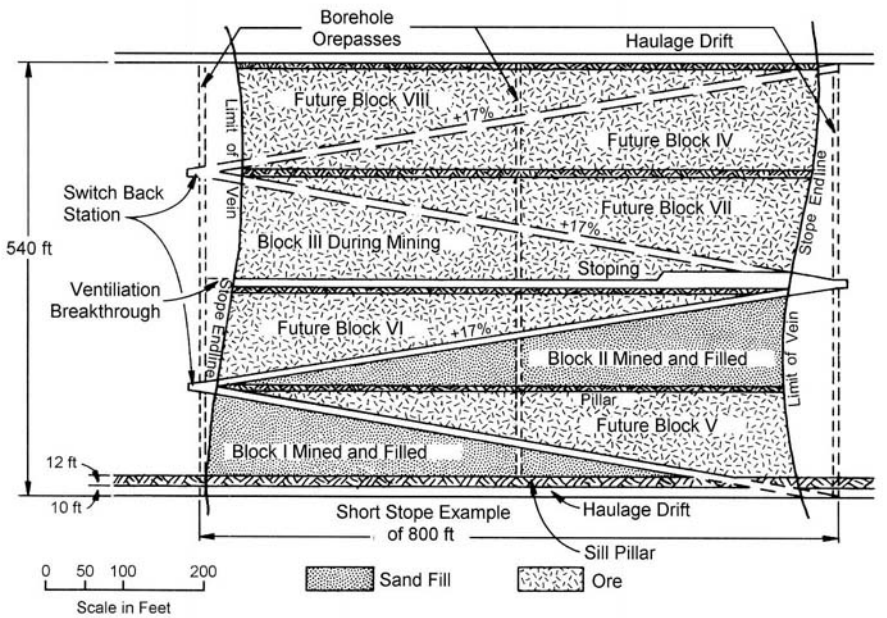


FIGURE 30.5 Variation of mechanized cut-and-fill for a short vein strike length, metric equivalent: ft × 0.3048 = m

upper level in the same fashion as previously described under "First Block." Alternating blocks will be taken, leaving the blocks above the ramps in place until the upper level has been reached. The remaining blocks will be removed in an upward retreating sequence similar to the method described previously under "Final Block." The down ramps will be filled as mining progresses upward. Because one ramp will be above the other, orepasses cannot be carried in the fill. Bored or driven raises can be located in the footwall near the switchback stations to serve as orepasses. Other convenient locations for these orepasses could be intersected by short crosscuts.

Irregular endlines and raking ore shoots can be worked by locating the switchback stations to conform with the endlines. The orepasses would be the limiting factor.

MINE SYSTEM CONSIDERATION

Any innovation or change in a stoping system will affect the total mine system. There will be improvements to certain facets but there are also some limitations involved with any stoping system.

Improvements

This highly mechanized ramp system will show significant improvements over conventional practices in productivity, operating costs, and safety.

Productivity. This will be increased in the following operations: mucking of broken ore, drilling blastholes in stopes, handling stoping supplies, installing utility lines, ventilating stopes, and hauling ore from stopes.

Operating Costs. These will be lowered due to the following factors: fewer miner-shifts per ton will not only lower operating costs, but will require fewer support employees; material handling costs will be reduced; sand fill pours will be larger and therefore more efficient; haulage costs from stopes will be less; and rubber-tired equipment can be moved from one stope to another for greater utilization.

Safety. Safer working conditions with less manual labor will result from the following: jumbo drill operators will be back from the face under supported ground or possibly under a canopy; danger from slusher cables will be removed; material handling will be with LHD units or underground trucks in place of handling by workers; personnel access will be by the ramp instead of the raises; and ventilation through the stope will be superior because all dust and gases should exhaust away from the working area.

Limitations

Several factors may limit the application of this cut-and-fill stoping system: the long stope lengths must be capable of standing open without extensive ground support; stope fills will require more time and larger capacity equipment than the smaller pours of the conventional method; capital investment in equipment will be 50% higher than for the conventional method (see

Table 30.6); the method cannot be as selective in mining discontinuous veins as can timbered methods; and highly faulted or irregular veins would be difficult to follow.

COMPARISONS BETWEEN PROPOSED AND EXISTING CUT-AND-FILL METHODS

A full evaluation of a new mining method can be made by comparing it with a contemporary method applied to the same conditions. The typical application model as described previously was used for evaluation of both the new and the old methods. This model is similar to many narrow-vein mining operations now or formerly in production.

Comparisons were made in the subjects of development, stoping, equipment, and labor requirements, and an economic evaluation analysis known as ECONEVAL was made by computer for each case.

Development

The mine development periods, to gain full stope production, are of the same duration. It is assumed that the first two years of development are in waste. Both systems start stope development by driving three 3 × 3-m (10 × 10-ft) haulage drifts in ore.

Stoping

Stoping in the conventional system will begin at 45 t/d (50 stpd) about the fifth month after the start of the third year (months begin at the start of the third year). Production will then increase by 45 t/d (50 stpd) per month until the 24th month when 20 stopes will be available to give continuous production of 725 t/d (800 stpd). Twenty-five percent of the stopes are assumed to be out of production for fill or development at all times (Fig. 30.6). Development ore could bring total mine production up to 725 t/d (800 stpd) as early as the eighth month, but this production would not be maintained.

Stoping in the highly mechanized cut-and-fill method would begin about the eighth month after the start of the third year and continue at 363 t/d (400 stpd) to the 26th month, when it would increase to the full 725 t/d (800 stpd). Four stopes in full production will actually produce sufficiently more than 725 t/d (800 stpd) so as to make up for any downtime for sand fills. Development ore, again, could bring total mine production up to 725 t/d (800 stpd) in the eighth or ninth month. Ore from the haulage drifts would be intermittent after the 12th month (see Fig. 30.7).

The conventional stoping system could come into production earlier than the new ramp method; however, considerably more miners would be required. These miners would need to be highly skilled to accomplish the development work required.

In the highly mechanized system, the miners that drove the three haulage drifts could phase into the stope development work and then into the actual

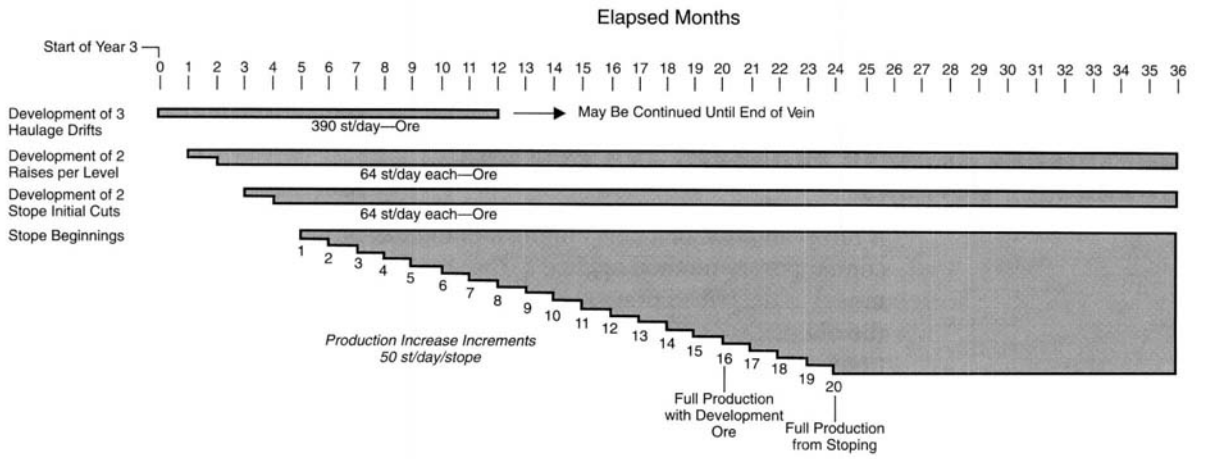


FIGURE 30.6 Start-up schedule of conventional cut-and-fill stoping, metric equivalent: st × 0.9071847 = t

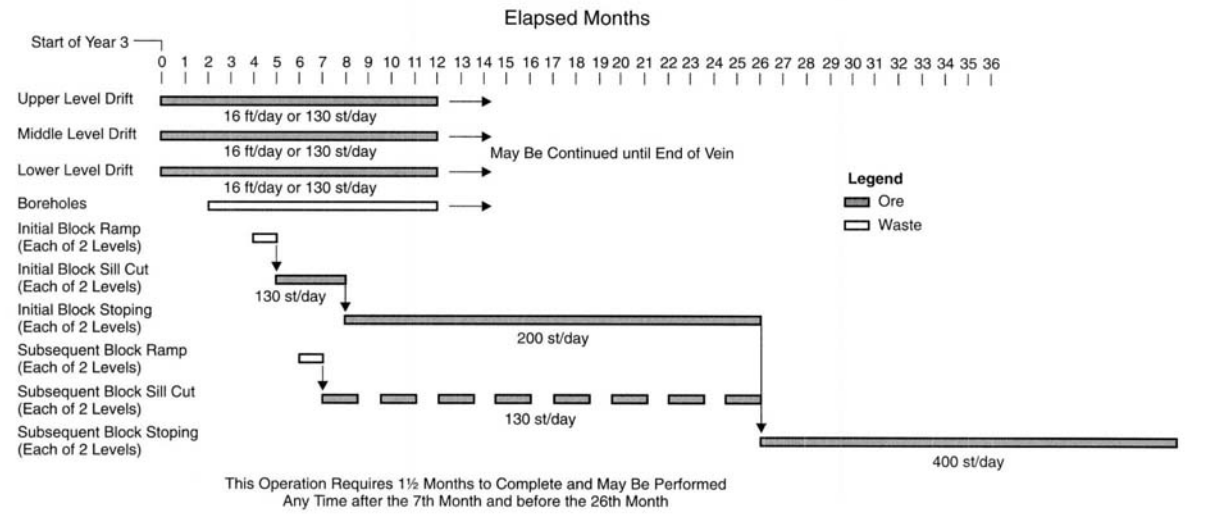


FIGURE 30.7 Start-up schedule of highly mechanized cut-and-fill stoping, metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t

stoping. The net effect would be reduced personnel requirements. Table 30.6 is a comparison of development costs and shows the ramp method to be 28.6% higher than conventional.

EQUIPMENT AND PERSONNEL REQUIREMENTS

Major equipment prices are shown in Table 30.7 to give a comparison. This equipment would cost approximately $593,000 over three years for the conventional method. The equipment for the highly mechanized method would cost approximately $900,000 over a two-year period. To start up, the highly mechanized system would therefore cost approximately 50% more than the

TABLE 30.6 Vein mining development cost comparison for stope mechanization

Conventional Method			Ramp Application			
10 × 10-ft* bald drift at $71.36 per ft	=	$21,468	First Block			
7 × 7-ft bald I-drift at $71.52 per ft	=	$20,169	Short tons* ore developed	=	92,029 st	$297,848
4-compartment raise	=	$7,902				
Chute fronts, 2 at $3,200	=	$6,400	Subsequent Block			
7 × 7-ft bald raise, 191 ft at $82.45 per ft	=	$15,752	Short tons ore developed	=	86,262 st	$161,664
Total development cost	=	$71,691	Final Block			
Short tons ore developed in block: $300 \times 200 \times 10 \text{ ft} \times \frac{162}{2000}$	=	48,600 st	Short tons ore developed	=	68,493 st	$6,185
			Total development cost			$465,697
Cost per short ton ore developed	=	$1.48	Cost per short ton ore developed			$1.89

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 30.7 Comparison of equipment requirements

Conventional Method				Highly Mechanized Method			
No.	Type	Unit Cost, $	Total Cost, $	No.	Type	Unit Cost, $	Total Cost, $
2nd Year				**2nd Year**			
3	Mucking machines	33,000	99,000	3	Jackleg drills	2,800	8,400
8	Jackleg drills	2,800	22,400	3	Stoper drills	2,000	6,000
8	Stoper drills	2,000	16,000	3	LHD diesels	87,240	174,480
Total			137,400	Total			188,880
3rd Year				**3rd Year**			
10	Jackleg drills	2,800	28,000	3	LHD diesels	87,240	261,720
10	Stoper drills	2,000	20,000	4	2-boom jumbos	105,520	422,080
8	Slushers	13,500	108,000	3	Jackleg drills	2,800	8,400
8	Scrapers	3,200	25,600	3	Stoper drills	2,000	6,000
8	Tugger hoists	2,000	16,000	4	Timber saws	900	3,600
Total			197,600	Total			701,800
4th Year				Two-year total			890,680
7	Jackleg drills	2,800	19,600				
7	Stoper drills	2,000	14,000				
12	Slushers	13,500	162,000				
12	Scrapers	3,200	38,400				
12	Tugger hoists	2,000	24,000				
Total			258,000				
Three-year total			593,000				

TABLE 30.8 Comparison labor requirements for salaried personnel

	Conventional Method			Highly Mechanized Method	
Job Classification	Annual Salary, $	No. Employed	Combined Cost, $	No. Employed	Combined Cost, $
Mine superintendent	30,000	1	30,000	1	30,000
General mine supervisor	25,000	1	25,000	1	25,000
Maintenance superintendent	25,000	1	25,000	1	25,000
Shift supervisor	20,000	8*	160,000	4†	80,000
Maintenance	20,000	3	60,000	3	60,000
Sand fill boss	20,000	1	20,000	1	20,000
Senior mining engineer	20,000	1	20,000	1	20,000
Senior geologist	19,000	1	19,000	1	19,000
Safety engineer	18,000	1	18,000	1	18,000
Ventilation technician	16,000	1	16,000	1	16,000
Survey crew	15,000	3	45,000	3	45,000
Samplers	15,000	3	45,000	2	30,000
Bonus clerk	12,000	1	12,000	—	—
Draftsman	12,000	1	12,000	1	12,000
Secretary	12,000	1	12,000	1‡	12,000
Mine clerk	12,000	1	12,000	—‡	—
Total Salaried		29	531,000	22	412,000
+ 35% burden			716,850		556,200

*Based on 2 shifts per day, 4 shift supervisors per shift.

†Based on 2 shifts per day, 2 shift supervisors per shift.

‡Clerk and secretary combined.

conventional method. This is to be expected because expensive, but more efficient, equipment would be used in place of cheaper but less efficient equipment.

Labor requirements for both the conventional and highly mechanized methods are estimated and compared in Tables 30.8 through 30.10. Table 30.8 shows 29 salaried employees for the conventional and 22 for the highly mechanized method. Fewer shift supervisors will be required to oversee stoping, as well as fewer support personnel. Table 30.9 shows 136 hourly employees for the conventional and 85 for the highly mechanized method. The greatest reduction in labor is in the stope mining. Total personnel are compared in Table 30.10. Labor cost per ton is $18.01 for the conventional method and $11.82 for the highly mechanized method, for a reduction of 34.4%. Productivity is 4.4 t (4.85 st) per miner-shift for the conventional and 6.78 t (7.48 st) for the highly mechanized method. Tons per total miner-shift will therefore be increased by 54.2% if the highly mechanized method is used.

TABLE 30.9 Comparison of labor requirements for hourly personnel

Job Classification	Conventional* Method	Highly Mechanized† Method
Electrician	4	4
Welder	3	3
Warehouse worker	3	2
Lamp operator	2	2
Janitor	1	1
Laborer	4	4
Total surface	17	16
Surface shaft worker	4	4
Hoister	4	4
Underground shaft worker	6	6
Grizzly tender	2	2
Motor operator	8	8
Track maintenance	3	3
Utility workers	4	4
Laborers	4	4
Total underground service	35	35
Haulage drifts	8	8
Raises	8	2
Total development miners	16	10
Stope miners	64	18
Fill crew	4	6
Total miners	68	24
Total hourly	136	85
Total cost at $10.50 per hr‡ × 1920 hr per year	$2,741,760	$1,713,600

*Based on 2 shifts per day, 2 operating levels, 8 stopes per level, and 2 miners per stope crew.

†Based on 2 shifts per day, 3 stoping areas, and 3 miners per stoping area.

‡Includes burden.

TABLE 30.10 Comparison of labor requirements for total personnel

	Conventional Method		Highly Mechanized Method	
	No.	Annual Cost, $	No.	Annual Cost, $
Salaried	29	716,850	22	556,200
Hourly	136	2,741,760	85	1,713,600
Total	165	3,458,610	107	2,269,800
Miner-shifts per year		39,600		25,680
Annual production		192,000 st*		192,000 st
Labor cost per short ton		18.01		11.82
Short tons per miner-shift		4.85		7.48

*Metric equivalent: st × 0.9071847 = t.

ECONOMIC EVALUATION ANALYSIS

The comparison of the two cut-and-fill methods was carried further by applying the costs and productivities to the computerized evaluation program, ECONEVAL. This program was developed by the Office of Coal Research to provide a reliable standardized estimate of the rate of return on investment. It is this figure that will be of paramount interest to any investor or loan institution.

Input data used in the analysis were specified in the application model described earlier in this chapter and are based on figures from the mid to late 1970s. Other costs were assumed to be similar to costs obtained from working mines. These assumed costs are equal for both methods. Typical interest, tax, depletion, and depreciation rates were applied.

Estimates of operating costs for each year were needed as input data. These costs are developed in Table 30.11. It is shown that the average operating cost for the conventional method is $19.63 per ton, whereas for the highly mechanized method it is $13.72 per ton. The operating cost for the mechanized method will then be 70% of that for the conventional method. This savings of 30% in operating cost is very significant when one realizes that it comes totally from the increase in stoping efficiency. The other unit operations and costs are held constant.

The commodity value of $40.00 per ton of ore produced was first applied to the ECONEVAL program. The results were that the conventional method of stoping would not only yield no profit but would never return the equity in investment. Because there can be no negative value for a rate of return, the program was rerun for the two methods using a commodity value of $50.00 per ton. For conventional cut-and-fill stoping, the equity rate of return is 6.31%, the total investment rate of return is 6.31%, and the equity payback is 11.04 years. For highly mechanized cut-and-fill stoping, the equity rate of return is 18.48%, the total investment rate of return is 18.48 %, and the equity payback period is 6.26 years.

It can readily be seen then that the rate of return on the investment in the highly mechanized mine is nearly three times the rate of return for the conventional mine. The equity would also be paid back in just over one half the time.

The computer program also produces a sensitivity analysis of the rate of return by the variation of the commodity value (Table 30.12). These values are varied from 20% above the $50.00 value to 20% below. Figure 30.8 illustrates the results of this analysis graphically with the varied commodity values noted at the corresponding method that the break-even commodity value is about $45.00. If we assume that a rate of return of no less than 10% would make the mining venture attractive, then for the conventional method, the value of the commodity would need to rise to about $54.00 per ton. The highly mechanized method, however, appears to be profitable from values of $40.00 on up. Drops in the commodity value could be much more than 20% before a loss would occur while using the highly mechanized method. The

TABLE 30.11 Operating costs per year for conventional cut-and-fill versus mechanized cut-and-fill

	Conventional Cut-and-Fill					Mechanized Cut-and-Fill			
	Year 1, $	Year 2, $	Year 3, $	Year 4, $	Years 5–17, $	Year 1, $	Year 2, $	Year 3, $	Years 4–17, $
Stoping	0	0	1,621,139	1,921,640	2,215,680	0	0	877,282	1,079,040
			(11.54)	(11.54)	(11.54)			(5.62)	(5.62)
Underground haulage	0	0	140,480	166,520	192,000	0	0	156,100	192,000
			(1.00)	(1.00)	(1.00)			(1.00)	(1.00)
Hoisting	0	0	84,288	99,912	115,200	0	0	93,660	115,200
			(0.60)	(0.60)	(0.60)			(0.60)	(0.60)
Pumping	0	0	112,384	133,216	153,600	0	0	124,680	153,600
			(0.80)	(0.80)	(0.80)			(0.80)	(0.80)
Ventilation	0	0	4,214	4,996	5,760	0	0	4,683	5,760
			(0.03)	(0.03)	(0.03)			(0.03)	(0.03)
Miscellaneous surface equipment	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000	12,000
			(0.09)	(0.09)	(0.06)			(0.08)	(0.06)
Ore-waste handling	0	0	280,960	333,040	384,000	0	0	312,000	384,000
			(2.00)	(2.00)	(2.00)			(2.00)	(2.00)
Sand fill operation	0	0	175,600	208,150	240,000	0	0	196,125	240,000
			(1.25)	(1.25)	(1.25)			(1.25)	(1.25)
Surface lagoon	0	0	84,288	99,912	115,200	0	0	93,660	115,200
			(0.60)	(0.60)	(0.60)			(0.60)	(0.60)
Surface ore haulage	0	0	210,720	249,780	288,000	0	0	243,150	288,000
			(1.50)	(1.50)	(1.50)			(1.50)	(1.50)
Miscellaneous operating expense	20,000	20,000	35,000	35,000	50,000	20,000	20,000	35,000	50,000
			(0.25)	(0.21)	(0.26)			(0.22)	(0.26)
Total annual cost			2,761,073	3,264,166	3,771,440			2,138,740	2,634,800
Short tons mined	0	0	140,480	166,520	192,000			156,100	192,000
			16.95 per st	19.60 per st	19.64 per st			13.71 per st	13.72 per st
Average				19.63 per st				13.72 per st	

Note: Figures in parentheses represent costs per short ton in dollars. Metric equivalent: st × 0.9071847 = t.

TABLE 30.12 Sensitivity analysis to variation of commodity value

Commodity Value, $ per st*	Percent Variance	Conventional Method Equity Rate of Return	Highly Mechanized Method Equity Rate of Return
$40.00	-20.00	0.0	9.76
	-19.00	0.0	10.33
	-18.00	0.0	10.85
	-17.00	0.0	11.34
	-16.00	0.0	11.83
	-15.00	0.0	12.31
	-14.00	0.0	12.73
	-13.00	0.0	13.14
	-12.00	0.0	13.56
	-11.00	0.0	13.99
	-10.00	0.0	14.45
	-9.00	0.45	14.90
	-8.00	1.16	15.35
	-7.00	1.86	15.79
	-6.00	2.55	16.19
	-5.00	3.21	16.58
	-4.00	3.86	16.96
	-3.00	4.51	17.34
	-2.00	5.13	17.72
	-1.00	5.76	18.10
$50.00	0.00	6.31	18.48
	1.00	6.83	18.89
	2.00	7.35	19.30
	3.00	7.86	19.71
	4.00	8.37	20.11
	5.00	8.87	20.52
	6.00	9.38	20.91
	7.00	9.87	21.32
	8.00	10.36	21.70
	9.00	10.85	22.05
	10.00	11.31	22.40
	11.00	11.73	22.75
	12.00	12.15	23.09
	13.00	12.58	23.44
	14.00	12.98	23.78
	15.00	13.38	24.11
	16.00	13.73	24.46
	17.00	14.08	24.80
	18.00	14.44	25.13
	19.00	14.83	25.47
$60.00	20.00	15.22	25.81

*Metric equivalent: st × 0.9071847 = t.

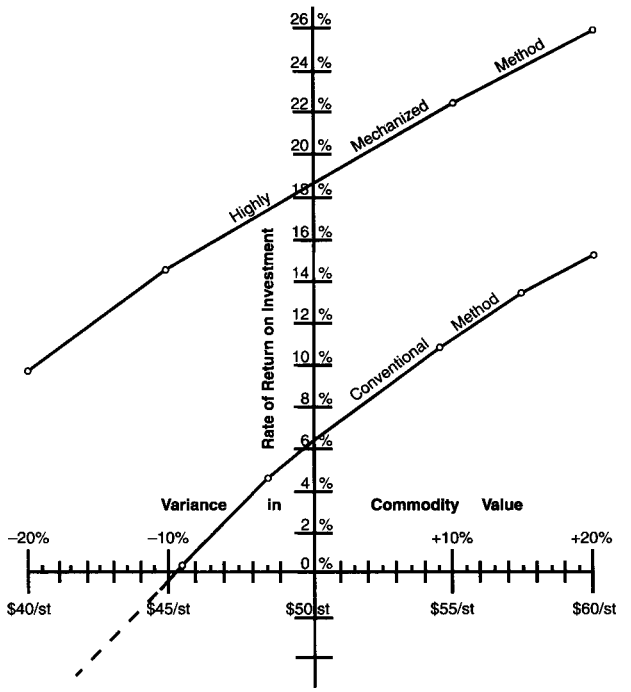


FIGURE 30.8 Graph of sensitivity analysis, through the ECONEVAL economic evaluation, metric equivalent: st × 0.9071847 = t

element of risk to investment for this method is therefore much lower than it is for the conventional stoping method.

The highly mechanized cut-and-fill stoping method described in this volume is a practical approach to increasing production and decreasing costs. The key to its success is the use of the efficient mobile equipment suggested. To use less productive equipment even for this unique stoping system would fail to produce the results derived in this study.

REFERENCE

Pugh, G.M., and Rasmussen, D.G., 1977, "Stope Mechanization–Vein Mining," U.S. Bureau of Mines HO 26024, August, Dravo Crop.

APPENDIX A – COSTS

TABLE 30.13 Costs to drive haulage drift and ventilation raises

Drive Haulage Drift		
Cost of trackless 10 × 10-ft* drift at 7.6 ft advance per round	=	$71.36 per ft
Track installation		
labor cost = 1 worker-hr per ft installed	=	$10.50 per ft
supply costs	=	$ 19.27 per ft
		$101.13 per ft
1242 ft of haulage drift at $101.13 per ft = $125,603		
tonnage mined = 10,308 st*		
cost per short ton of development ore = $12.19		
cost per short ton prorated over block tonnage = $1.36		
Ventilation Raises		
ventilation raise cost = $55.10 per ft		
3 raises per block × 200 ft × $55.10 per ft = $33,060		
cost per short ton prorated over block tonnage = $0.36		

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 30.14 Costs to drive ramp and initial stoping cut

Ramp Cost		
40 ft* of sill and 290 ft of ramp × $71.36 per ft	=	$23,548.80
Initial Stoping Cut		
1040 ft × $71.36 per ft	=	$74,214.40
Total		$97,763.20
Tonnage produced = 8748 st*		
Development tonnage cost = $11.18 per st		
Cost per short ton prorated over block tonnage = $1.06		

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 30.15 Estimated conditions and costs for driving a 5 × 5 ft raise for 14-ft height

Conditions	
Drill and blast two 8-ft* rounds to break 14 ft total	
Assume 35 holes per round maximum	
Assume additional brow support of 4 mats and 15 rockbolts	
Assume 2 workers for 3 shifts for this work	
Powder factor of 4.8 lb per cu yd*	
Caps and primers $1.21 per hole	
Costs	
Drilling and blasting costs	$336.00
drill and blast 2 workers, 2 shifts at $10.50 per hr	72.60
blasting supplies, 60 caps at $1.21 each	10.66
ANFO, 4.8 lb × 7.4 cu yd × $0.15 × 2	168.00
support brow, 2 workers for 1 shift at $10.50 per hr	10.80
supplies, 4 mats at $2.70 each and 15 bolts at $2.64 each	39.60
stoper, bits and steel, 650 ft at $0.15 per ft	97.50
For 14 ft of raise, drilling and blasting costs $52.51 per ft	$735.16
Mucking costs	
7.4 cu yd × 2 = 14.8 cu yd × 1.2 swell factor = 17.76; use 18 loose cu yd	
with a 2-cu yd bucket at 90% fill factor = 10 scooptram loads	
with a 10-min cycle time and 50 min hour = 2 hr mucking	
2 hr at $10.50 per hr	21.50
2 hr at $16.01 per hr	32.02
	$53.02
Total Costs	
Drilling and blasting	$735.16
Mucking	53.02
	$788.18
Short tons of ore produced = 24	
Cost per short ton of raise ore = $32.84	
Raise cost = 6 × $788.18 each = $4,729.08, or $0.05 per st of stope block tonnage	
Chutes	
5 x $3,200 = $16,000 = $0.17 per st	
Total raise cost prorated over stope block tonnage = $0.22 per st	

*Metric equivalents: ft × 0.3048 = m; lb × 0.4535924 = kg; cu yd × 0.7645549 = m^3.

TABLE 30.16 Breast stoping costs

Drilling and Blasting

Using computerized simulated drilling program it was found that one 8-ft* breast round could be taken by 2 miners in 32.4 + 42 min = 75 min, and would cost $1.31 per st*

Temporary support

8 bolts at $2.64 each	$21.12
drilling material and supply costs, 50 ft at $0.26 per ft	13.00
	$34.12, or $0.64 per st

drilling bolt holes takes	32 min
installing bolts assume	28 min
	60 min

Mucking

Mucking use 3½-cu yd* LHD

3.5 cu yd × 90% fill factor × 1.2 swell factor × 75% availability = 1.97 cu yd per cycle, or 4.41 st per cycle

Haul cycle

0.5 min load
3.0 min haul (maximum 200 ft, 150 fpm* average speed)
0.5 min dump
4.0 min total haul cycle at 50 min per hr = 12.5 cycles = 55.13 stph capacity

Assume 1 hr mucking time

Operating cost of 3½-cu yd LHD = $23.03 per hr = $0.43 per st

Totals

Total time to take round = 75 + 60 + 60 = 195 min, or 3.25 hr
2 miners could take 2 rounds per 6½-hr shift
Cost of labor for travel and lunch = 2 miners × 1½ hr × $10.50 per hr = $31.50
= $ 0.30 per st

Total cost per short ton = $2.68

*Metric equivalents: ft × 0.3048 = m; fpm × 0.00508 = m/s; cu yd × 0.7645549 = m; st × 0.9071847 = t.

TABLE 30.17 Costs to build timber ramp

Timber Requirements		
12 10 × 10-in.* timbers required on 4-ft* centers per lift		
12 10 × 10 in. by 10 ft	=	1000 bd ft*
48 10-ft × 3-in. × 12-in. plank	=	1400 bd ft
		2400 bd ft
Assume $150 per 1000 bd ft. therefore timber cost	=	$366.00
Add 10% for spikes, wedges, etc.		36.60
Total timber		$402.60
Other Supplies		
Use bull horns to facilitate installation		
Drill four 2-ft holes per timber = 96 ft at $0.15 per ft	=	$ 14.40
24 bull horns at $3.00 each	=	72.00
1 roll brattice at $60.00 per roll	=	60.00
Total supplies		$146.40
Time Analysis		
2 hr drill		
2 hr hang 10 × 10s		
2½ hr lay deck		
1½ hr lunch and travel, i.e., 4 workers, 1 shift at 10.50 per hr	=	$336.00
Ownership and labor for the scooptram at $14.19 per hr for 8 hr	=	$113.52
Totals		
Total cost = $998.52		
Cost per ft of ramp = $998.52/48 ft = $20.80 per ft		
Total ramp length = $\sqrt{162^2 + 990^2} + 48 = 1003.7 + 48 = 1051.17$		
Total cost of ramp floor = 1051.17 × $20.80 = $21,865 = $0.30 per st*		

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m, bd ft × 0.002359737 = m^3; st × 0.9071847 = t.

TABLE 30.18 Costs to raise 5 × 5-ft cribbed orepasses

Labor Costs
- Consider 2 workers
  - remove grizzly rails — 0.25 hr
  - build 8-ft crib — 0.50 hr
  - replace rails — 0.25 hr
  - 1.00 hr
- 6 cribs at 2 worker-hr each = 2 worker-shifts
- Labor cost = 16 hr at $10.50 per hr = $168.00

Material Costs
- 6 × 6-in.* cribbing by 5 ft* long × 4 sides = 120 bd ft* per ft of height
- 6 cribs × 8 ft high = 48 ft of crib or 5760 bd ft

Use $200 per 1000 bd ft to allow for cutting	=	$1,152
Add 10% for nails, etc.	=	115
Use 3 rolls of brattice cloth at $60.00 each	=	180
		$1,447

Supplies Transport
- Use LHD to deliver materials to job site
- 1 trip per crib + 30 min per trip = 3 hr. Allow 4 hr for contingencies
- 4 hr × $28.10 (ownership and operating costs of LHD) = $112.40

Total Cost
- $168.00 + $1,447.00 + $112.40 = $1,727.40

Cost Summary
- Cost per foot = $36.00
- 555 ft total of cribbed raise
- Cost per short ton prorated over stoped tonnage = $0.27
- Cost of fill has been calculated previously at $1.25 per st*

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; bd ft × 0.002359737 = m^3; st × 0.9071847 = t.

TABLE 30-19 Subsequent block development costs

Haulage Drift	
600 ft* × $101.13 per ft = $60,678	
Produce 4860 st* development ore	$12.49 per st
Haulage drift cost amortized over block tonnage	$ 0.70 per st
Orepass Raises	
3 × $53.02 per ft × 14 ft = $2,226.84	
Produce 73 st development ore	$30.50 per st
Orepass raises amortized over block tonnage	$ 0.03 per st
Vent Raise	
200 ft × $55.10 per ft = $11,020	
Amortized over block tonnage	$ 0.13 per st
Ramp and Initial Stope Cut	
(290-ft ramp and 515 ft initial stoping cut) × $71.36 per ft = $57,444.80	
Produce 4274 st development ore	$13.44 per st
Amortized over block tonnage	$ 0.67 per st
Orepass Chutes	
3 at $3,200 each = $9,600	$ 0.11 per st
Final Ramp	
290 ft × $71.36 per ft = $20,694	
Amortized over block tonnage	$ 0.24 per st

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 30.20 Subsequent block backstoping costs

Drilling and Blasting*

A computer drill and blast simulator program was used. Data obtained from a rock drill manufacturer for machine energy output was used as input for the program and a 2-boom jumbo was considered. The printout shows that 442 ft per shift can be drilled and blasted at a cost of $265.55 per blast or $1.03 per st.

Mucking*

Use a 3.5-cu yd LHD. Use the same time cycle as in first block, which produces 32.75 cu yd per 60 min hour = 77.22 stph. Because 200 st per shift is required, muck can be removed in 2.59-50 min hour at a machine availability of 86%. Assume 3 shift-hr are used to muck out 200 st, 3.5-cu yd LHD at $23.03 operating cost for 3 hr ($60.09) = $0.35 per st.

Temporary Support

As in first block = $0.64 per st

Cost to Build Ramp

$20.80 per ft × 970 ft = $20,176 = $0.26 per st

Cribbed Raises

4612 ft × $36.00 per ft = $16,603.00 = $0.22 per st

Fill*

Use the $1.25 per st figure

*Metric equivalents: ft × 0.3048 = m, cu yd × 0.7648549 = m^3; st × 0.9071847 = t.

TABLE 30.21 Subsequent block engineering and miscellaneous costs

Assume the same engineering and miscellaneous costs as in first block: $13,440 or $0.16 per st*

*Metric equivalent: st × 0.9071847 = t.

TABLE 30.22 Miscellaneous costs

Assume

1	engineering shift per lift
1½	shifts pipe salvage, etc.
4	shifts material supplies, etc.
6.5	worker-shifts at $10.50 per hr = $546.00 per lift

With 20 lifts per stope = $10,920

72,827 st* of stoped ore = $0.15 per st

Ramp around Crown Pillar

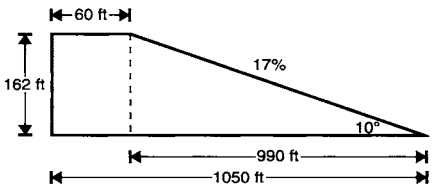
235 ft* at $71.36 per ft = $16,769

72,827 st = $0.23 per st

*Metric equivalents: ft × 0.3048 = m; st × 0.9071847 = t.

TABLE 30.23 Volume of ore in first block

Assume 10-ft thick vein and 12.35 cu ft per short ton.*



Stope Tonnage

Volume of □ = 97,200 cu ft

Volume of Δ = 801,900 cu ft

at 162 lb per cu ft = 72,827 st stoping

Development Tonnage

Haulage drift = $1242 \times 10 \times 10 \text{ ft} \times \frac{162}{2000}$ = 10,308 st

Sill drift = $1080 \times 10 \times 10 \text{ ft} \times \frac{162}{2000}$ = 8,748 st

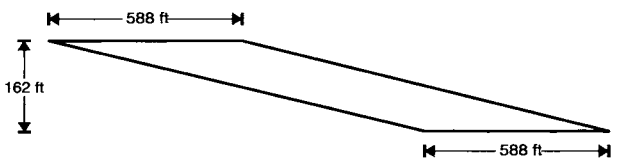
Raises = $5 \times 5 \times 12 \text{ ft} \times 6 \times \frac{162}{2000}$ = 146 st

Development = 19,202 st

Total block minable tonnage = 92,029 st

*Metric equivalents: ft × 0.3048 = m; cu ft × 0.02831685 = m^3; lb × 0.4535924 = kg; st × 0.9071847 = t.

TABLE 30.24 Minable tonnage in subsequent block



Stope Tonnage*

Volume of stoped block 588 × 162 × 10 ft = 952,560 cu ft, or 77,157 st

Development Tonnage

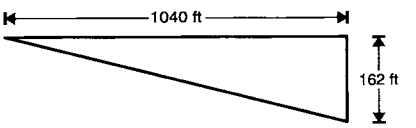
Haulage drift	600 × 10 × 10 ft ÷ 12.35 cu ft per st	=	4,858 st
Initial stope cut	515 × 10 × 10 ft ÷ 12.35 cu ft per st	=	4,170 st
Orepass raises	3 × 12 × 5 × 5 ft ÷ 12.35 cu ft per st	=	73 st
			9,101 st

Total minable tonnage in block = 86,208 st

*Metric equivalents: ft × 0.3048 = m; cu ft × 0.02831685 = m^3; st × 0.9071847 = t.

TABLE 30.25 Volume of ore in final block

Stope tonnage*	
Volume of ore in block	$= \frac{1040 \times 162}{2} \times 10$
	= 842,400 cu ft
Stoping	= 68,243 st
Development tonnage	
Volume of development ore*	
initial stope	= 30 x 10 x 10 ft = 3000 cu ft = 243 st
orepass through sill pillar	= 4 x 4 x 12 ft
	= 192 cu ft = 16 st
Development	= 259 st
Total tonnage	= 68,502 st



*Metric equivalents: ft x 0.3048 = m; cu ft x 0.02831685 = m^3; st x 0.9071847 = t.

TABLE 30.26 Final block development

Total length of cribbed raise = 572 ft*
One ore chute
One 5 x 5-ft orepass
One 10 x 10-ft bulkhead at top of ramp in waste
Vent raise haulage drift plus ramp are existing from previous block
Waste development is nil

*Metric equivalent: ft x 0.8048 = m.

APPENDIX B – DRILL AND BLAST SIMULATION MODEL

This appendix shows a representative run for breasting using jacklegs to develop drilling and blasting times and costs for a particular run. *

Performance Summary	
Tons broken per blast	20.34 tons
Yards broken per blast	13.95 yd^3
Footage advanced per blast	5.98 ft
Drilling rate	12.00 ipm
Resultant powder factor	2.44 lb per ton

Cost Summary (In dollars)		
Underground Blasting	Percussive Drilling	Totals
1.42	1.24	2.67 per ton
2.07	1.81	3.89 per cu yd
28.94	25.32	54.26 per blast
4.84	4.23	9.07 per ft of advance
	0.32	0.32 per ft drilled

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; cu ft × 0.0283168 = m^3; cu yd × 0.7645549 = m^3; lb × 0.4535924 = kg; st × 0.9071847 = t.

Working Variables (Input + Calculated)	
Rock Descriptors	
1. Coefficient of rock strength	1.25
2. Weight of solid rock, lb per cu ft	162.00
Drilling Equipment Description	
1. Type of drill (1.0 = jackleg, 2.0 = drifter)	1.00
2. Actual bit diameter, in.	1.38
3. Amount of mechanization (0.3 = column mounted, 0.6 = jumbo)	0.70
Blast Round Design	
1. Average hole depth, ft	6.50
2. Number of holes per blast	12.00
3. Percent pull	0.92
4. Excavation type (1.0 = drift, 2.0 = stope)	2.00
5. Width of heading, ft	0.0
6. Height of heading, ft	0.0
7. Angle of holes to face, rad	1.57
8. Number of drill holes per row	4.00
9. Number of rows blasted per blast	3.00
10. Hole spacing on each row, ft	1.75
11. Burden, ft	2.00
12. Percent powder that is ANFO	0.95
13. Weight of explosive, lb per cu ft	61.25
14. Burden factor, k	0.0
Mine Design	
1. Number of drills drilling per face	1.00
2. Number of drillers working per face	1.00
3. Number of driller helpers working per face	0.0
4. Average number of headings	1.00
5. Percent of supervisor's time allocated to each face	0.0
6. Number of powder loaders	1.00
7. Actual, or measured drilling rate over an 8-hr shift, ipm	12.00
Cost Variables	
1. Labor rate—driller, incl. fringe, $ per hr	10.50
2. Labor rate—driller's helper, incl. fringe, $ per hr	10.50
3. Cost of bit, $ per bit	7.50
4. Jackleg drill cost complete, $	0.00
5. Jumbo cost complete, $	0.00
6. Drifter drill cost, $	0.00
7. Supervisor's wage rate, incl. fringe, $ per hr	0.00
8. Labor rate—blaster, incl. fringe, $ per hr	10.50
9. ANFO cost, $ per lb	0.00
10. Cost of primers and caps per hole, $	1.25

Output Variables	
Penetration Rate Outputs	
1. Theoretical bit penetration, ipm	0.0
2. "Rule of thumb" check of theoretical bit penetration, ipm	0.0
3. Penetration rate further modified for the effect of bit diameter	0.0
4. Dummy variables (for "rule of thumb" checks of theoretical bit penetration)	0.0
5. Drilling time per blast round, in hrs.	1.30
6. Average penetration rate (overall drilling rate), in ipm	12.00
7. Apparent specific energy, in lb per cu in.	0.0
Drilling Cost Outputs	
1. Drilling labor cost per blast, in $	13.65
2. "Direct" drilling labor cost per foot drilled, in $ per ft	0.17
3. Drilling labor per shift per face, in $ per ft	13.65
4. "Direct" labor cost per foot of advance $ per ft	2.28
5. "Direct" labor cost per ton broken, $ per ton	0.67
Related Drilling Cost Outputs	
1. Bit cost per foot drilled, in $ per ft	0.03
2. "Direct" drilling power (i.e., compressed air) cost per foot drilled, $ per ft	0.02
3. Steel (wear) cost per foot drilled, $ per ft	0.06
4. "Direct" drill (or jumbo) write-off cost per foot drilled, $ per ft	0.0
5. "Direct" drill maintenance cost per foot drilled, $ per ft	0.04
6. "Direct" drilling supervision per foot drilled, $ per ft	0.0
Blasting Results and Costs	
1. Resultant powder factor, in lb of explosive per tons actually broken	2.44
2. Resultant per cubic yard powder factor, in lb of explosive per cu yd of material broken	3.56
3. Blasting labor (i.e., powdering the holes) cost per blast (including loading, priming, and wiring time), in $ per blast	7.35
4. Explosives cost per blast (includes ANFO, dynamite, primers, caps, and line), in $ per blast	21.59
5. Blasting supervision cost per blast, in $ per blast	0.0
6. Blasting equipment (i.e., pneumatic loaders, etc.) write off cost per blast, $ per blast	0.0
7. Blasting labor (i.e., powdering) cost per minute, in $ per min	0.17
8. Number of blasters leading (i.e., powdering) the face	1.00
9. Powdering time, in min	42.00
Conversion Factors Used in Model	
1. Total footage drilled per blast, in feet	78.00
2. Tonnage broken per blast, tons	20.
3. Designed volume of rock to be broken per blast, cu ft (i.e., before % pull)	273.00
4. Actual (loose) volume of rock pulled (i.e., actually broken) per blast, in cu yd per blast)	13.
5. Footage drilled per shift per face, in ft per shift per heading	78.00
6. Machine (drill or jumbo) cost, in $	0.
7. (Average) footage drilled per hour of machine in use in fph	60.00
8. Corrected (or perpendicular) depth of round	6.50
9. Actual (or resultant) depth of drift advance per blast, in ft per blast	5.08

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; cu ft × 0.0283168 = m^3; cu yd × 0.7645549 = m^3; lb × 0.4535924 = kg; st × 0.9071847 = t.

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CHAPTER 31

Undercut-and-Fill Mining: An Introduction

J.W. Murray

GENERAL DESCRIPTION

The undercut-and-fill method of mining was developed by Inco Ltd. in the Sudbury district of Ontario, Canada, to deal with abnormal ground conditions encountered in pillar recovery. Although it is now sometimes utilized as a primary stoping method, the main application continues to be in pillar recovery.

In any mining method in which fill is employed, pillar systems tend to fail gradually as stoping progresses. As a result, most pillars are completely destressed and broken by the time pillar recovery commences. Control of the back in such ground becomes a major problem and is contingent on local conditions as they exist in the workplace. In most mining methods, operations such as drilling, blasting, slushing, and timbering can be readily assessed and the mining cycle predicted. However, in the mining of pillars with overhand methods, the time and effort required to maintain a safe back varies to a quite unpredictable extent. For years the square-set method was the most effective means available for pillar recovery. However, in areas where bad ground was encountered, square setting was a costly and sometimes hazardous method. As the tonnage mined from pillars increased, it was apparent that a safer and less expensive approach was required. Past experience indicated that the best way to deal with broken ground was to mine from the top down. Top slicing systems were examined but were rejected due to the major subsidence that resulted. However, ideas generated from studying top slicing finally developed into the present method of undercut-and-fill. This method, which has almost completely replaced square-set mining, exhibits reduced costs, has improved safety and efficiency, eliminates the factor of bad ground overhead, and allows a closer prediction of the actual mining cycle. The result is more accurate scheduling of pillar production. This factor is of increasing importance as the tonnage from pillar recovery increases in relation to total mine production.

Undercut-and-fill is a method of extracting a block of ore by mining successive cuts, working from the top down. After a cut of ore is completely mined out, laminated timber stringers are constructed along the sides for the full length of the cut. Round logs covered by a wire screen are laid across the stringers to form a mat. The cut is then tightly filled with hydraulically placed cemented sand fill. Mining is then resumed on the cut immediately below the mat. As the cut advances, the timber stringers are supported by round timber posts seated on the solid bottom of the cut. Drilling, blasting, muck removal, and timbering operations are repeated until all the ore in the cut has been removed. At this time, another log mat is laid and the cut is filled. This sequence is continued until the block has been mined from level to level.

The basic principles of undercut-and-fill can be adapted to a variety of mining situations. It can be applied transversely or longitudinally in orebodies of varying width that dip from vertical to flat lying. The method can be applied using conventional equipment or it can be mechanized through the introduction of small diesel or air-powered units for muck removal. Undercut-and-fill is a selective method, ore recovery is high, and dilution can be controlled to an acceptable level. However, it is a high-cost, labor-intensive method and as such its use is restricted to areas where the less costly methods of ground support are not successful or where bulk mining methods are not adaptable. One basic requirement for undercut-and-fill is an adequate source of hydraulically placed cemented sand fill and a dependable sand plant and distribution system.

The success of the undercut method is to a large degree dependent on the competence of the fill placed in each of the successive cuts. In theory, the weight of the fill on each cut is supported in part by the cemented sand itself as it binds to the rough sides of the mined-out areas next to the pillar walls, and in part by the timber support. As mining progresses beneath the mat, a downward pressure is exerted within the cemented fill and a voussoir arch develops above the mat, across the full width of the cut. This supports the major portion of the load. The log mat and posts have a double function in that they carry the remaining weight not supported by the voussoir arch and in addition prevent any pronounced settling of the fill mass. Excessive settling breaks the bond between the cemented fill and the pillar walls, eliminating the arch effect and transferring the entire load to the mat and posts. The theory of a voussoir arch forming within the cemented sand fill mass is substantiated in practice by the fact that pillars have been mined from level to level with no more significant weight on the bottom cuts than was experienced on the top cuts. In effect, as mining progresses downward, there is no additional transfer of weight from previous cuts to the cut being mined.

APPLICATION OF UNDERCUT-AND-FILL

Because of their size, the Sudbury orebodies are, for the most part, mined transversely and, to a considerable extent, by filling methods. Transverse stoping blocks are designed with 9-m (30-ft) crown pillars established below each level and with stopes separated by vertical rib pillars. In past years, floor pillars were left above the level as well, but present practice favors the silling

of primary stopes at the base of the rail. Rib pillars are designed to meet the ground control requirements of the primary stoping methods. Local conditions and past experience generally dictate their size.

Historically, since the square-set method was to be used for recovery, rib pillars were designed to accommodate standard 1.7-m (5½-ft) sets. A typical rib pillar is four sets wide [6.7 m (22 ft)] when located between two square-set or cut-and-fill stopes having a width of 8.4 m (27.5 ft). When the undercut-and-fill method was developed, a mat width of two sets was adopted. This divided each cut of the common four-set pillar into two 3.35-m (11-ft) slices. Due to sand sloughing along the stope walls as the pillar is being mined, 3.65-m (12-ft) logs are required on the mat to effectively cover the slice. As a result, a four-set pillar is removed by mining two 3.65-m (12-ft) wide slices on each cut. This 3.65-m (12-ft) width can be modified to fit odd size pillars. Experience has shown, however, that mining widths under a mat should not be less than 3 m (10 ft) because of loss in efficiency, or more than 4.2 m (14 ft) because of possible mat support problems.

A variety of conditions is presently encountered when crown and rib pillar recovery is undertaken, depending upon when and how the primary stopes were extracted and filled at the different mines throughout the district. These conditions, as they exist at a particular mine, dictate where the pillar will be silled, the amount of development required to establish an orepass, the width and number of slices in each cut, and whether the crown will be mined with the rib or mined separately as a mechanized block. The advent of hydraulically placed sand, followed by the technique of adding cement to consolidate the fill, has changed some of the basic mining practices over the past few years. As a result present-day undercut-and-fill mining takes place under highly variable pillar conditions. The pillars may be located next to square-set stopes filled with waste rock, with top silling done under a double 51-mm (2-in.) plank sill floor. On the other hand, the pillars may be adjacent to shrinkage stopes backfilled with well-consolidated cemented sand fill. Here the pillars top sill under a solid back.

Crown and floor pillars are normally recovered in conjunction with the rib pillars. The procedure is to mine the rib, plus half of the crown over each of the primary stopes on either side of the rib. This provides an initial mining width of approximately 15 m (50 ft), which is mined in four 3.65-m (12-ft) slices. The four slices are mined to the top of the primary stopes, at which point the two outside slices are dropped and the two center slices continue down through the 6.7-m (22-ft) rib. In areas where a floor pillar exists, the two outside slices can be reestablished beneath the sill floor of the primary stope and the four slices mined to the level below.

The usual practice when mining a rib pillar is to mine and fill one 3.65-m (12-ft) slice, then mine and fill the second slice to complete the cut. In the past, both slices have sometimes been mined full width as one slice under two mats. Experience has shown, however, that where more than one mat is exposed, support problems can develop. As a result this practice has been largely curtailed.

At one mine where 5.5-m (18-ft) wide pillars were established between stopes filled with unconsolidated alluvial sand, the width of the undercut slice was modified to adapt to two particular situations. In one case, where a gob fence was installed on both sides of the pillar and is retaining the dry fill, the 5.5 m (18 ft) of ore is mined full width. The mat is constructed with three stringers instead of the regular two, and a third line of posts is carried down the center of the cut. In the second situation, the 5.5-m (18-ft) pillars were located between shrinkage stopes filled with alluvial sand but without a gob fence. As the sand is not consolidated with cement, runs of fill could occur if the pillar broke through into the stope. To minimize this possibility while at the same time recovering the maximum tonnage of ore, a 4.3-m (14-ft) wide slice is mined down the center of the pillar, leaving a 0.6-m (2-ft) shell of ore on either side to retain the fill.

One application of load-haul-dump (LHD) equipment to undercut-and-fill mining is in the recovery of longitudinal crown pillars up to 152 m (500 ft) in length as a single mining block. Rather than mining small sections of the crown in conjunction with the rib pillars, as described earlier, the crown is left intact and the ribs taken separately. In this case, the ribs are top silled below the crown and are completed before mining of the crown commences. A typical crown recovery layout consists of a haulage drift, called a slot, driven longitudinally along the footwall of the crown with transverse slices running from the slot to the hanging wall. The transverse slices are driven at 1.22 rad (70°) to the slot to facilitate turning in and out of the slices with the LHD unit. Orepasses are established by driving short boxholes from the top of existing orepasses in the primary stopes, through the crown to the top sill elevation. Where timber sets are required for top silling, wide sets with 3.35-m (11-ft) long caps are used to provide sufficient room to operate the LHD machine. All other aspects of undercut-and-fill mining are the same, whether mechanized (LHD) or conventional (slushers) haulage equipment is used.

Depending on the location of the top sill and the thickness of the crown, one cut of square sets and two or more cuts of undercut-and-fill can be mined. Mining across a series of primary stopes in this manner requires that the stopes be tightly backfilled. If direct access to the stope is not possible for checking and filling, then test holes are drilled from the second cut above the stopes, and if voids are found, sand is introduced until the voids are filled. A layout of this type provides continuous mining efficiently. Multiple faces are available, and once the footwall slot is far enough advanced, a sequence of mining, fill preparation, and sand pouring can be initiated to ensure that one or more headings are always breaking ore.

As well as being the principal means of pillar extraction, undercut-and-fill is now the primary stoping method in blocks of ore that formerly would have been mined by square sets. In transverse blocks, a longitudinal slot with a centrally located orepass is established across the full width of the block. The location of the slot is usually on the footwall side.

Broken muck from the individual slices is scraped back to the slot and then cross-slushed to the orepass. It is evident from this arrangement that the slot is open for a longer period of time than are the transverse slices adjacent to it,

and therefore additional mat logs and additional posts under the stringers may be required to maintain the slot until the cut is completed. Blocks consisting of up to six 3.65-m (12-ft) slices per cut have been mined in this manner with air-operated slushers. The availability of multiple headings produces tonnage on a more consistent basis than is possible with a single-face working place. Delays due to the fill cycle (typical of most fill mining methods) are minimized. They do occur, however, when the slot is being filled, prior to the new cut being opened.

As a further step to increased productivity, blocks as described earlier but up to eight slices wide [30 m (100 ft)] have been mechanized by replacing air slushers with diesel-powered LHD units of 1 m^3 (1.3 cu yd) bucket capacity. The use of LHD equipment permits the mining of larger blocks and a greater flexibility in the location of slots and orepasses than is possible when mining is restricted to the capabilities of slusher hoists.

DETAILED DESCRIPTION

Development

In the mines where Inco is using undercut-and-fill for pillar recovery, the existing drifts and crosscuts, which were initially driven to develop the levels for primary stoping, are utilized. One of the first requirements when preparing mining layouts is to establish the best location for an orepass with as little additional development as possible. As control chutes are established at existing gangways on the level below, there is a limited choice as to the location of the pass at the bottom end. The location at the top sill elevation is determined to a large extent by the following considerations: (1) orepasses should be kept as near vertical as possible and should not be inclined at less than 1.13 rad (65°) from the horizontal for satisfactory pulling characteristics when handling muck; (2) the slushing distance of either side of the orepass should not exceed 38 m (125 ft) if the length of the pillar from footwall to hanging wall is quite long; (3) the orepass should be located in ore to avoid rock development and the mining of rock on each successive cut, but if the competence of the pillar ore is such that excessive sloughing can be expected and there is doubt whether the orepass would last the life of the pillar if driven in ore, then the pass is located in footwall rock; and (4) geological mapping obtained during mining of the primary stopes provides more accurate information on ore outlines and grade than was available when primary stopes were designed, so certain low-grade tonnages within the pillar may be dropped as uneconomic, and this will affect the final location of the pass.

In crown and rib pillar recovery, only one pass is required and it is established along the centerline of the pillar. In the crown area, where four 3.65-m (12-ft) slices are normally mined, the broken muck requires cross slushing from the outside slices to the pass, while in the ribs a wing is blasted out on the side of the pass and no cross-slushing is necessary.

The primary stoping blocks (more than four slices in width) are similar to the crowns in that only one pass is required and, being centrally located in the

slot, cross-slushing is necessary to move the muck to the pass. In mechanized blocks, however, it is mandatory to carry two orepasses, one located at the end of the slot and the second located at the center of the mining block. This arrangement is necessary for the purpose of opening up a new cut. The procedure, described later, is quite different with an LHD unit than with slusher hoists.

At Inco, the common method for driving orepasses is to bore a 1.5-m (5-ft) diam raise using a Robbins 61R raise borer. The preliminary development for setting up the borer consists of driving a short [2.4 × 2.4 m (8 × 8 ft)] access from either the footwall or hanging wall drifts to the location of the pass. A station, 3.65 m (12 ft) wide by 6.7 m (22 ft) long by 2.4 m (8 ft) high, is excavated, with a 1.8 × 1.8-m (6 × 6-ft) raise driven at the inclination of the borehole to a height of 6 m (20 ft) above the base of the rail. A concrete pad is poured for the raise borer base. On the level below, a bit station, 2.1 m (7 ft) wide by 3 m (10 ft) high, is driven at gangway elevation to provide access for attaching the reaming head. The pilot hole is oriented to break through at a point 4.5 m (15 ft) from the centerline of track. After the hole has been reamed, a head cover is installed (at the bottom of the hole) and a control chute constructed. The chute is fitted with an underslung gate activated by an air cylinder controlled through a three-way valve installed on the gangway.

When a raise borer is not available, a 2.1 × 2.7-m (7 × 9-ft) cribbed raise is driven with the cribbing supported by pins every 9 m (30 ft). After the raise is completed, the control chute is constructed and then the raise is stripped by blasting out the individual sections of pinned cribbing.

Top Silling

When the orepass and control chute are completed, top silling begins on the initial cut of the pillar. The general practice is to top sill at the base of the rail, keeping the back height at gangway elevation. If for some reason the pillar is to be silled at a horizon below the base of the rail, then the 1.5-m (5-ft) borehole is widened from the top down and a manway timbered to the sill elevation.

In ground where conditions permit, the first cut is mined using bolts and screen for back support. However, in poor ground or when mining under the sill floor of a primary stope, the conventional square-set silling method is used. Wide sets with long girts are used wherever possible, but standard 1.6-m (5½-ft) sets may be necessary under a weak sill floor or in areas of particularly heavy ground. The wide sets improve efficiency during mining and are more convenient to work under when building the mat. In mining under the sill floor, care is taken not to break through the old floor, particularly under stopes containing dry fill. If no sill floor exists in the primary stope, a 3-m (10-ft) crown is left between the stope and the top sill. Where primary stoping was initiated after the introduction of undercut-and-fill mining, the sill floor in many stopes consists of laminated stringers and log mats. In this case the initial cut in the pillar is taken as a regular undercut with the stringers posted in the standard manner.

Constructing the Mat

When mining is completed on the top sill, the muck is cleaned out and a mat consisting of two laminated stringers covered by logs and a screen is laid over the entire floor of the cut. In a 3.65-m (12-ft) wide slice, the two longitudinal mat stringers are laid 2.7 m (9 ft) apart over the length of the pillar. Stringers are constructed by laying two pieces of 127 × 203 mm by 3.65-m (5 × 8 in. by 12-ft) timbers side by side with the end of one at the midpoint of the other. This procedure is followed for the length of the pillar and results in a continuous stringer, 203 mm (8 in.) deep and 254 mm (10 in.) wide, with a joint located every 1.8 m (6 ft). The stringers are held together at the joints by banding (two bands on each side of the joint) the timbers with 25.4-mm (1-in.) wide steel strapping. The stringers are blocked underneath as they are installed to keep them level and on a flat grade.

At both ends and at 6-m (20-ft) intervals in between, the stringers are tied with 12.7-mm (½-in.) cable secured to rockbolts at the footwall and hanging wall or to the stringers of the mat above. The cables at the extreme ends prevent the ends of the stringers from collapsing when the last blast is made. The cables along the length of the stringers provide temporary support should any posts get knocked out during the mining cycle.

Prior to placing the mat, 3.65-m (12-ft)-round, 203- to 254-mm (8- to 10-in.)-diam logs are laid between the stringers [2 logs for each 1.8-m (6-ft) section or one for each joint]. During the mining cycle, these logs are recovered from under the mat and used to post up under the stringers.

To build the mat, 3.65-m (12-ft)-round, 152- to 203-mm (6- to 8-in.)-diam logs are placed across the stringers at 0.6-m (2-ft) centers. Wire screen [101 × 101 mm (4 × 4 in.) by No. 9] is laid down on top of the logs and extended 0.9 m (3 ft) up the walls of the pillar. Due to overbreak, the 3.65-m (12-ft) mat logs do not in all cases reach to the walls and the screen along the sides helps to prevent loss of fill from that area during mining of the cut below.

Filling Preparations

Concurrent with construction of the log mat, the orepass grizzly is removed and the pillar prepared for the filling operations. There are different techniques for sealing around the orepass and manway; one that has proven effective is described in the following section.

The floor of the cut around the top of the orepass is blown clear and the borehole is covered with a bulkhead of 127 × 203-mm (5 × 8-in.) timbers. The bulkhead is covered with Fabrene (a woven plastic-coated synthetic fabric), which is extended beyond the sides of the bulkhead and sealed tight to the floor using a thick mortar of cement and water. A 3.65 × 3.65-m (12 × 12-ft) manway is established over the orepass by installing three transverse stringers spaced 1.8 m (6 ft) apart. These manway stringers rest on top of the longitudinal mat stringers and extend from wall to wall across the pillar. No mat logs are laid within the manway area, and landings are constructed by laying a

floor of 76 × 76-mm (3 × 3-in.) pieces across the transverse stringers. Manways contain a ladderway and a nipping compartment for hoisting timber and supplies, as well as air and waterlines and auxiliary ventilation tubing. The large size manway also provides sufficient room for storing the slusher hoist, drills, and miscellaneous equipment during sand filling.

On each cut, the manway is enclosed by a fill fence, which is built by standing round log posts, 1.8 m (6 ft) apart, from the transverse stringers to the overhead mat. On all four sides, 127 × 203-mm (5 × 8-in.) timbers are nailed to the posts at a 762-mm (30-in.) spacing and covered with screen and Fabrene. The Fabrene is stapled to the overhead mat and to the timbers around the manway. The bottom end is extended to the floor of the pillar, but not sealed to the floor, as is the case with the Fabrene covering the orepass bulkhead. On either the footwall or hanging wall side of the fill fence, a small entranceway, called a doghouse, is constructed by placing two inclined 127 × 203-mm (5 × 8-in.) timbers from the top of the mat to a point 1.5 m (5 ft) up the fence. The doghouse is enclosed with screen and Fabrene and is not filled with sand during the pour. It is available for access to the pillar when the next cut is being filled, and allows the pour crews entry to repair any leaks that may develop as well as to monitor the progress of the pour.

Filling

The filling of each cut is essentially a backfill operation. An obvious requirement for the success of undercut-and-fill mining is that the fill be placed as tightly as possible to the mat of the cut above. To accomplish this, pour points are located 9 m (30 ft) from the manway and every 9 m (30 ft) thereafter for the length of the pillar. These pour points are carried down through the fill from cut to cut. They are established on the top sill by installing a separate 101-mm (4-in.) victualic thin-walled pipe above the mat from the manway to each of the pour point locations. At each location the pipe is brought below the mat, turned 3.14 rad (180°) upward using two 1.57 rad (90°) elbows, and extended vertically to within 305 mm (12 in.) of the back. As mining takes place on each cut, the 1.57 rad (90°) elbows are exposed. To utilize the line for filling, it is disconnected at the elbows, extended below the mat of the cut being filled, and again turned upward and extended to the mat above. To prevent fill being washed from the overhead mat, a 51-mm (2-in.) plank is nailed to the logs directly above the pipes to deflect the sand. At the manway each of the 101-mm (4-in.) lines is connected to the main sand line through a separate valve. By opening the proper valve, the pour crew can direct the sand to the various pour points within the pillar. This piping arrangement is normally reestablished on every fifth cut because beyond that point the distance between the valves and the cut being poured prohibits the sand fill crew from properly controlling the pour. A new setup may be required more often if pipes wear out or become plugged with sand.

Depending upon the mine, sand is delivered to the pillars at rates varying from 145 to 180 t/h (160 to 200 stph) at pulp densities of 65 to 70% solids. When pouring 15:1 cemented fill at a rate of 145 t/h (160 stph) and a density of 65%, the inflow of water is 0.02 m^3/s (370 gpm). To drain the water, two lengths of 101-mm (4-in.) diam perforated plastic drainage pipe wrapped in

two layers of Fabrene are installed from the ends of the pillar to the manway. The pipes are hung along the posts, and on each side of the pillar, and are inclined from the overhead mat at the footwall and hanging wall ends to the floor of the cut at the manway. The pipes are extended through the fill fence and drain off relatively clear water as the fill is being poured. By avoiding a buildup of water in the pillar, more sand can be poured at any one time before it is necessary to stop and allow the water to drain.

In the manway, a 152 × 152-mm (6 × 6-in.) decant box is installed through the orepass bulkhead. As water drains into the manway, it is decanted into the box and down the orepass. Before any sand is poured, the pass is pulled empty and a catch pit is built around the chute on the gangway. The catch pit retains the drainage water for a short period of time, allowing slimes to settle and clear water to overflow into the drift below. It also retains any spillage of sand that may occur should a leak develop around the orepass seal or in the fill fence.

The initial pour is made from the pour points next to the orepass, and the sand is allowed to flow into the manway under the fill fence. It covers the orepass bulkhead and is allowed to build up around the fill fence until the bottom edge of the Fabrene is well covered. At that time, the pour is stopped temporarily and the water is drained, resulting in a sand-tight seal around the bottom of the manway. On continuing the pour, the procedure is to bring the level of the sand 0.9 to 1.2 m (3 to 4 ft) above the stringers in the manway area to stabilize the fence. The pour is then completed by filling the footwall and hanging wall ends first and working back toward the manway.

The mines using undercut-and-fill are supplied with sand from different sources with the result that the plants produce cemented fills of varying compressive strengths. At each mine, local experience over the years has indicated which ratio of sand to cement provides the best results when pouring undercut-and-fill pillars. As a result, one mine may pour 15:1 cemented sand for the entire cut, whereas another may pour 20:1. A third property may pour 0.9 m (3 ft) of 10:1 above the mat, then 1.5 m (5 ft) of 30:1, and finish off the pour with regular sand.

Opening a New Cut

One method used to open up new cuts is to drill downholes around the 1.5-m (5-ft) raise borer hole and bench down 3.65 m (12 ft) to the floor of the next cut. The downholes are drilled prior to pouring sand fill. To keep the 31.7-mm (1¼-in.) drill holes from plugging with sand, the collars are reamed with a 51-mm (2-in.) bit, and 31.7-mm (1¼-in.) ID vinyl antistatic tubing is inserted into each hole and sealed at the collar with cement mortar. Sufficient tubing is left protruding from the hole to ensure that no sand enters the hole when the orepass plug is being poured. A number of blasts are made around the pass until an area approximately 4.5 m (15 ft) in length is opened up across the width of the pillar. A good percentage of the broken muck from the benching sequence enters the pass when blasted; the remainder is hand mucked or moved to the pass with a blowpipe.

When the 3.65-m (12-ft) cut height is established, initial 1.8-m (6-ft) rounds are taken toward both the footwall and hanging wall sides. After this, the slusher is lowered and set up. As the cut is being opened up, any of the laminated stringer joints that are exposed are supported with log posts. After the second blast from the orepass is made, a raised grizzly is installed over the borehole. The grizzly is constructed of 127 × 203-mm (5 × 8-in.) timbers, hung by chains from the four posts around the pass and covered tightly with round logs. The top of the grizzly is 1067 mm (42 in.) above the floor of the cut, allowing sufficient space below the logs for the scraper to pass underneath. The manway ladder and nipping compartment are extended to the top of the grizzly.

In mechanized stopes it is not possible to open up a new cut in the conventional manner around the orepass. In order to move an LHD unit from one cut to another, an inclined ramp has to be established on the top sill and carried down through the block. To establish the ramp, a log mat with inclined stringers is installed in one of the slices. The stringers are inclined from the top of the slice at the footwall slot, to the bottom of the slice at the hanging wall. After the logs and screen are in place, as much broken muck as possible is piled underneath the mat and the slice is filled with cemented sand fill. When the ramp is opened up, the face is mined at the normal 3.65-m (12-ft) height, with the stringers being posted in the same manner as when mining a slice on level grade. The ore piled underneath the mat reduces dilution by displacing sand fill and is removed as the ramp is advanced. Because the mat is inclined over a vertical distance of only 3.65 m (12 ft), the grade of the ramp in each individual stope depends entirely on the length of the slice in which it is located. When the ramp reaches the hanging wall, the direction of mining is turned 1.57 rad (90°) and the face advanced to an end slice. On reaching the end slice, the face is turned another 1.57 rad (90°) and mined back toward the footwall until it breaks into an orepass located in the slot. As soon as this pass is ready for dumping muck, the ramp and the slot on the old cut are filled. To make this procedure possible, two orepasses are required in the slot, the first located opposite one of the end slices and the second located near the ramp. The ramp, of course, is established in one of the slices on the opposite side of the stope from where the end orepass is located.

Drilling and Blasting

Once the new cut is established, no drilling takes place on the face until all exposed stringer joints have been posted. Because undercut-and-fill is basically a silling operation, the old lifters are located, cleaned, and plugged with wood. Before drilling commences, the face and walls are scaled and the underside of the mat is checked to remove any loose sand fill not scaled during the slushing operation. Pillars with solid ground on one or both walls may require some bolting to make them safe. Drill-hole remnants are washed clean and all bootlegs are identified by marking them with yellow paint.

Because the ground in most pillars is badly broken up, holes are drilled to suit conditions at the face; the number of holes and the pattern are left to the discretion of the crew. Care is taken in drilling lifter holes to avoid the necessity of popping the bottom to set posts and stringers and also to ensure a relatively

smooth bottom for slushing. To keep the mat from being damaged by the blast, the top holes are drilled flat and at a minimum distance of 0.6 m (2 ft). below the mat. When there is solid rock on both sides, the round is wired to break upward against the log mat, starting at the center of the face to avoid damaging the stringers. The blast is cushioned by the sand fill and any broken muck left under the mat on the previous cut. When there is sand on one or both sides, the blast is wired to break sideways against the fill.

The length of each round is 1.8 m (6 ft), so that only one joint on each stringer is exposed and unsupported at any one time. To prevent posts from being knocked out during the blast and exposing additional joints, the three lines of posts immediately in front of the face are braced with 51-mm (2-in.) planks, one nailed to the underside of the overhead stringer at the top of the post and one wedged tight between the posts at the midpoint.

Drilling is done with airleg drills and detachable 31.7-mm (1 ¼-in.) tungsten carbide insert bits on 0.2 rad (12°) tapered auger steel. The auger steel is more efficient than the regular hexagonal rods in removing drill cuttings from a hole when drilling in broken ground. Holes are charged with ammonium nitrate–fuel oil (ANFO) and detonated with Hi-Strength caps on 2.7-m (9-ft) tape fuse. The caps, which are placed 152 mm (6 in.) from the toe, are positioned and held in the hole by plastic centralizers. Where ground water or seepage from sand fill pours causes wet holes, a water-gelatin explosive in the form of 25 × 305-mm (1 × 12-in.) cartridges is used. Such explosives represent less than 10% of the total consumed.

Mucking Using Slushers

After a blast, the muck pile is thoroughly wetted down and any loose sand fill hanging under the overhead mat is scaled down prior to hooking up the tail sheave. Tail sheaves are never anchored to stringers, mat logs, or posts but are hooked to 12.7-mm (½-in.) chains or cables that are laid under the mat alongside each stringer prior to filling. The chain is exposed as mining progresses and is recovered and used on successive cuts.

During the mucking cycle, dust is controlled at the face by continually wetting the muck pile with a water spray. In long pillars, visibility is improved by illuminating the cut with an air-operated light.

Double drum 15 or 22-kW (20 or 30-hp) air-operated slusher hoists, pulling 1219-mm (48-in.) toothed scrapers, are used for ore removal. The toothed scrapers are very effective for slushing on the rough floor of the cut and from a muck pile that is steeply banked against the face. The scrapers have five teeth on each edge of a reversible backplate. After the teeth on one lip are worn down, the backplate is reversed. When the teeth on both lips are worn out, the backplate is changed and sent to the surface for reconditioning.

Ground Control

Any posts that are knocked off line by a blast or by the scraper during the mucking cycle are straightened and rewedged immediately. If a blast exposes more than one joint on each stringer, the second joint is supported from the

muck pile by a short post before mucking begins. Once sufficient muck has been removed to post under the first joint, the short post is removed and slushing is completed. As soon as the blast is mucked out and the face and walls well scaled, the exposed joints on both stringers are posted with 203- to 254-mm (8- to 10-in.) diam logs. [The minimum allowable diameter for a post is 203 mm (8 in.).] The posts are installed with the large end under the stringer and the small end wedged tightly at the bottom. In order to have sufficient room to build the next set of laminated stringers on the inside of the two rows of posts, the posts are inclined by offsetting the bottom end 381 mm (15 in.) from the vertical.

Ventilation

Because undercut-and-fill pillars have only one access and no positive airflow at the face, they are treated the same as a drift heading and supplied with auxiliary ventilation. Where power is available, electric fans are preferred. Most installations, however, consist of 406-mm (16-in.) compressed air-driven fans equipped with silencers. Fans are installed at the top of the manway on the fresh air side of the raise, and plastic ventilation tubing is carried into the pillar to within 15 m (50 ft) of the face. In addition, a 25-mm (1-in.) plastic air line with the control valve located on the level is extended down the manway and into the pillar. It is turned on after any blast and allowed to blow between shifts. In mechanized stopes where 38.7-kW (52-hp) LHD machines are used, the 406-mm (16-in.) air fans provide adequate ventilation to meet the requirements of the underground diesel code of Ontario.

PROBLEM AREAS

Two problems encountered in undercut-and-fill mining are mat failure and runs of dry fill when mining beside old stopes. Mat failures can occur because of poor blasting practice, poor quality cemented sand fill above the mat, or by the fill mass becoming saturated with water. The most frequent cause is from blasting when one or more posts are knocked from under stringer joints, or where drilling done too close to a stringer results in its being broken by the blast. In the case of fill containing less than the required cement for proper consolidation, or where the fill becomes saturated by drainage or ground water, the resultant weight on the mat and posts can cause the timbers to ultimately fail and the mat to collapse.

Stringers and posts are checked each working shift to determine if any abnormal weight is developing. If and when this occurs, extra posts are installed under the stringers. In cases of extreme weight where stringers and posts begin to break due to excessive pressure, it may be necessary to install auxiliary sets under the mat.

Runs of dry fill occur when old gob fence fails or when pillars break into stopes where no gob fence was installed. Voids in the fill weaken the overhead mats and can lead to mat failure if allowed to go uncontrolled. When a run occurs, an attempt is made to spile off the fill, and if the resulting void is not too extensive, mining is continued. If the fill cannot be stopped, mining is discontinued and a

mat is laid, the stope is filled, and the next cut is opened up. An attempt is made to fill all voids by extending a pour pipe and a breather pipe as far into the void as possible, and backfilling at the same time as the cut is poured.

When a pillar is mined next to a stope where the gob fence is in poor condition and cannot be adequately supported, or is mined next to a shrinkage or blasthole stope where a competent shell of ground cannot be maintained, the usual practice is to decrease the height of the cut from 3.65 m (12 ft) to 3 m (10 ft).

PRODUCTIVITY

Compared to bulk mining systems such as caving and blasthole, undercut-and-fill is an extremely high cost and low productivity method. However, relative to other filling methods, it approaches conventional cut-and-fill in efficiency and cost per ton and has a lower accident frequency. In terms of direct stoping efficiency, undercut-and-fill is capable of producing an average of 12.6 t (14 st) per miner-shift. Productivity will range from a low of 9 t (10 st) per miner-shift for top silling to a high of 14.5 t (16 st) per miner-shift when mining under laminated stringers. Where the method is mechanized by the use of LHD equipment, as in primary stoping blocks or in crown pillar recovery, productivity increases to 16.2 t (18 st) per miner-shift.

Undercut-and-fill stopes are operated on a two-shift basis, except in the case of relatively small pillars, which are commonly worked on a single shift. A stope crew consists of a leader and a driller. They are responsible for keeping the workplace in a safe operating condition by making regular daily inspections, doing repairs as they are required, and by following good housekeeping practices.

Crews work on an incentive system that covers all of the normal mining and sand fill preparation procedures. During the course of completing a slice in an average pillar, the crew will spend approximately 80% of the available working time at mining (drill, blast, slush, and timber), 15% at sand fill preparation (building the stringers and mat), and 5% at pouring sand fill, cleaning up after the pour, and getting ready to open the new cut. Timber and supplies are brought to the top of the manway by miscellaneous crews. The stope crew is responsible for nipping material from that location to the working face. Abnormal situations not covered by the incentive system are handled on an off-contract basis as they occur. This amounts to approximately 20% of the available working time.

MATERIALS AND SUPPLIES

Timber consumption for undercut-and-fill is 40% lower than for square-set mining and averages 0.02 m^3/t (6.1 board ft per ton) for top silling and 0.1 m^3/t (4.8 board ft per ton) when mining under stringers. For the most part, the timber used is the cheapest available, with the majority consisting of rough-sawn logs of jackpine, red pine, or spruce. No framing is required and the only timber that is milled is the 127 × 203-mm (5 × 8-in.) stringer material and the planks used for bracing posts and timbering the manway.

Sand consumption for filling averages 0.9 t (1 st) of sand placed for every 1.8 t (2 st) of ore removed. Cement consumption at a sand-to-cement ratio of 15:1 is 67 kg (134 lb) of cement per ton of sand placed or 34 kg/t (68 lb per ton) of ore removed.

RECOVERY AND DILUTION

At Inco mines, ore recovery for undercut-and-fill mining averages 96%. In stopes where the total width of the pillar is mined, 100% of the ore is recovered. Where a thin shell is left to hold back dry fill, ore recovery can be as low as 85%. Other reasons for less than 100% recovery are abandoned ore due to mat failures or fill runs, ore left under old stopes with no sill floor, or ore left to keep the pillar from breaking into old workings.

As a general rule, undercut-and-fill mining will produce low rock dilution and moderate sand fill dilution. Dilution is defined as the tonnage of rock and sand removed divided by the tons of ore removed. Rock dilution, where rock is defined as anything below the cutoff grade of the pillar, occurs at the footwall and hanging wall ends of the stope. The average dilution rate for undercut-and-fill is 15%, consisting of 5% rock dilution and 10% sand fill dilution.

Approximately 60% of the sand dilution comes from the fill under the mat and 40% comes from the sides next to old gob fence or from adjacent undercut-and-fill slices. A small percentage comes from unavoidable fill runs. Dilution rates will vary according to the width, length, and height of the cut being mined, but they can be estimated for grade control purposes by expecting 0.3 m (1 ft) of sand fill from under the mat and 0.3 m (1 ft) of fill from either side.

An effort is made to control the amount of sand dilution from under the mat by slushing broken ore between the stringers. The last blast in the pillar is left at the face until the stringers are constructed and is then slushed back and leveled off over the length of the stope. This muck combined with the posts laid under the mat keeps the volume of sand poured under the mat to a minimum and helps to diminish the overall dilution.

OPERATING COSTS

The development required for mining the majority of rib pillars by undercut-and-fill is basically the same regardless of their tonnages. As previously outlined, it consists of driving an orepass through the pillar and establishing a control chute on the gangway on the level below. Pillar tonnages vary over a wide range; as a result the development cost per ton for individual pillars varies to the same degree. For purposes of comparison to other mining methods, the cost of boring a 1.5-m (5-ft) diam orepass and constructing a chute is approximately $1.00 per ton (approximate 1979 dollars) for an average size pillar.

Undercut-and-fill stoping costs for mining under stringers in rib pillars range from $8 to $10 per ton of ore removed (approximate 1979 dollars). These are direct costs only and are made up of the following percentages: labor plus

incentives 65%, timber 10%, supplies 10%, and sand fill and cement 15%. The labor cost does not include fringe benefits, and the sand fill cost does not include the cost of the distribution system. The cost per ton for top silling is approximately 20% higher than for mining under the mat. Costs for mechanized undercut-and-fill are approximately 10% lower than for conventional mining with slushers. Because the efficiency of mechanized stopes is 30% greater in terms of tons produced, a greater reduction in cost per ton might be expected. However, the marginal decrease in overall cost per ton reflects the high maintenance cost of operating LHD equipment.

IMPLEMENTATION

The main difficulty in implementing the undercut-and-fill method of mining is in gaining the confidence of being able to mine under fill rather than on top of it. It is imperative that any mine planning to use the undercut-and-fill method have available an adequate source of good-quality sand fill and a reliable plant that will consistently produce the proper ratio of cement to sand.

To learn about and to evaluate the method, choose a pillar that could be abandoned without interfering with future mining plans if the method does not prove acceptable. If at all possible, the initial cuts should be mined in a stope solid on both sides. Once the technique of mining under stringers is mastered, the method can be expanded to mining next to filled stopes.

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CHAPTER 32

Undercut-and-Fill Mining as Practiced by Homestake Mining Co.

A.S. Winters

INTRODUCTION

Homestake Mining Co. owns and operates the Bulldog Mountain mine near Creede, Colorado. Lead-silver concentrates are produced from ores extracted from narrow veins that dip from 1.04 to 1.39 rad (60 to 80°). Annual mill throughput is approximately 90,700 t (100,000 st). Ore shoots average 2 m (7 ft) in width, and the ore is often weak, muddy, and poorly consolidated. Walls can be either strong or highly fractured along the strike of the veins.

Cut-and-fill methods were initially selected to mine the deposit and considerable overhand mining occurred during the initial stages of the mine. Because of unstable ground conditions, mine management recognized that other methods would have to be employed. Undercut-and-fill mining was adapted to extract the loose ore from narrow veins and is presently the primary method used. Approximately 80% of the annual production is extracted by this system.

Loading ramps are positioned along the strike of the vein on 90-m (300-ft) intervals to allow for 45-m (150-ft) long cuts in each direction from the extraction raise. This interval was selected for efficient slusher operation and stope cycling. Loading ramps rather than chutes were selected for ore movement into 4.5-t (5-st) Granby-type cars because of extremely sticky ore conditions.

Mined-out areas are filled with deslimed mill sands. Sand is pumped from the mill into two mine storage dams where the water is decanted. Each dam has a 544-t (600-st) capacity. Sand is pumped into the mine at a 30 to 40% slurry through 2195 m (7200 ft) of 51-mm (2-in.) thick wall pipe. When called for, sand is reslurried within the dams to a density of 55 to 65% and pumped to the particular mine opening requiring fill. Dry cement is added to the slurry prior to pumping. The capacity of the mine fill system is 20 t/h (22 stph).

DEVELOPMENT

Vertical development below the main adit level occurs on 61-m (200-ft) intervals from an internal winze. Main haulage drifts or laterals are 2.7 × 2.7 m (9 × 9 ft) in section and driven parallel to the vein structures. Ground conditions generally dictate that better progress can be realized in the footwall than in the hanging wall. A distance of approximately 21 m (70 ft) is maintained between vein and drift openings to accommodate a smooth 18-m (60-ft) radius curve for crosscuts driven on 91-m (300-ft) intervals. Approximately 18 m (60 ft) of tail room is driven past the vein intersection to accommodate train loading. Ore and waste are hauled using diesel or electric-driven locomotives operating over 18-kg (40-lb) rail [762-mm (30-in.) gage].

Loading ramps are established above each crosscut-vein intersection. Scram drifts are then driven in the vein to determine the width and grade of the ore. If a prior decision to mine the block had been reached, then the scram is driven only far enough [about 12 m (40 ft)] to establish a drawpoint and scraper tail room. In most cases, the drawpoint is a 2.1 × 2.1-m (7 × 7-ft) cutout driven about 2.4 m (8 ft) into the footwall.

On the level above, a raise bore station is established in the footwall to one side of the crosscut-vein intersection. The station is of sufficient size to accommodate a Robbins 41R raise boring machine. Support for the station is accomplished with 1.2- and 1.8-m (4- and 6-ft) roof bolts and wire screening. Once completed, a 152-mm (6-in.) concrete slab is poured and the raise drill set in position to center the 1.5-m (5-ft) diam hole approximately 1.8 m (6 ft) from the vein footwall. A 229-mm (9-in.) pilot hole is then drilled to the scram cutout below and reamed back to full diameter (see Fig. 32.1).

STOPING

Initial mining in a typical undercut-and-fill stope begins from the top of the borehole. The initial cut can be either immediately beneath the track level or the old scram drift, depending upon the particular situation. Cuts vary but are generally 2.4 to 3 m (8 to 10 ft) high. Support is accomplished by square-set timber, overhead caps, or roof bolts and screening, depending upon ground conditions.

Once the initial cut is removed, underhand timber and screen are installed and cemented sand fill placed. The stope is then ready for undercut mining. Preparation work begins with sinking 4.5 to 6 m (15 to 20 ft) on the borehole plus cutting a slot to the vein. As sinking progresses, hanging sets are installed to provide manway and service facilities. Hanging sets 1.7 m (5 ft 8 in.) long and 1.6 × 2.2 m (5 ft 4 in. × 7 ft 4 in.) in section are installed with steel hangers similar to shaft sets. Framed 203 × 203-mm (8 × 8-in.) timbers are made up in advance and shipped to the stope during sinking operations. When manway and service facilities are completed to the floor of cut to be mined, generally 3.65 to 4.3 m (12 to 14 ft) below the previous sand fill, the ore is removed to the hanging wall plus a round each way along the vein. Levelers or long stringers are then installed over the raise opening to support the work deck, slusher, and grizzly opening. All waste and ore from the sinking process is fed

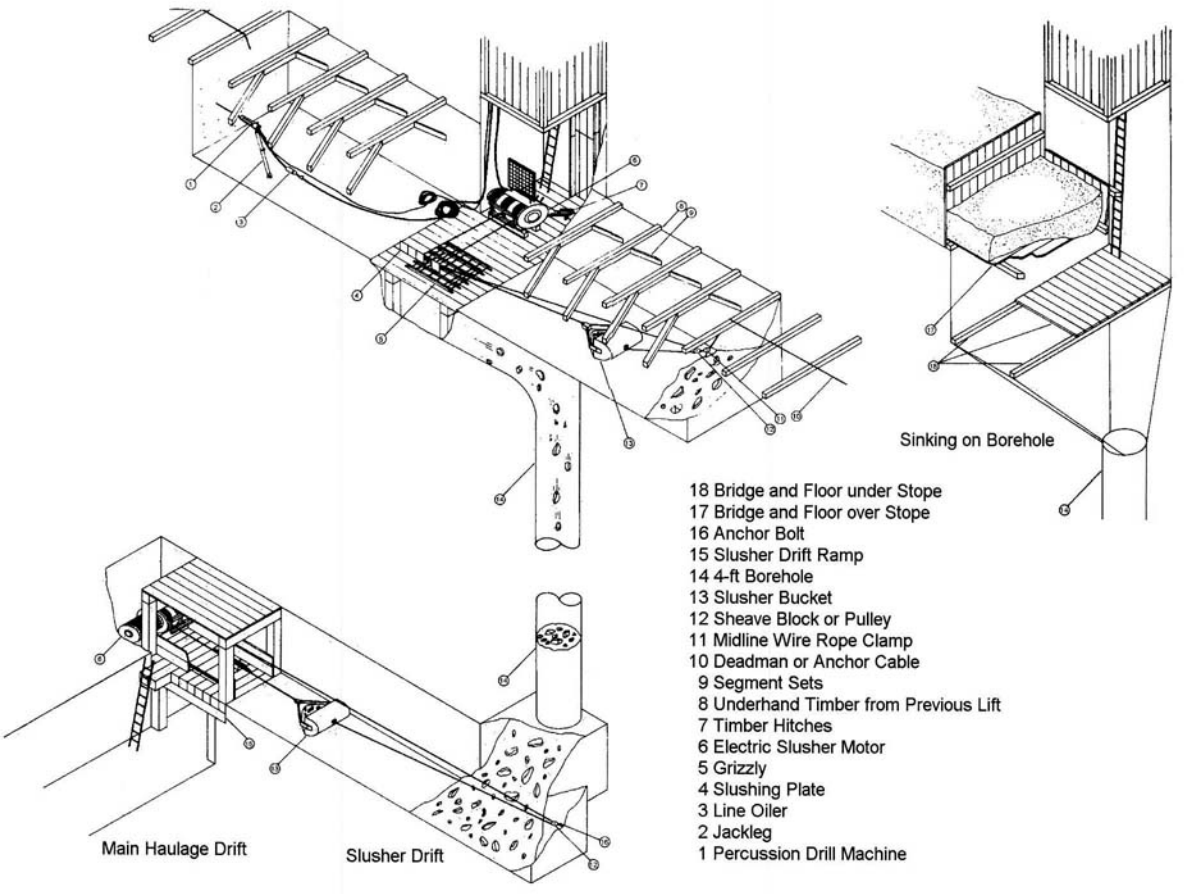


FIGURE 32.1 Isometric view of mining in a stope

by gravity to the raise (see Fig. 32.2). Once the stope is equipped, mining progresses outward in each direction from the grizzly opening. Stope equipment consists of 907 kg (2000 lb) pull air tuggers, timber skips, 11-kW (15-hp) ventilation fans, 15- to 22-kW (20- to 30-hp) electric slushers, 762- to 1270-mm (30- to 50-in.) scrapers, 67-mm ($2\frac{5}{8}$-in.) jackleg drills, as well as other drilling and blasting supplies.

In narrow stopes [generally less than 2 m (7 ft) wide], no support is required for the underhand caps of the previous fill. Where wide sections in the vein occur, segment braces are installed beneath the caps to insure that the sand support cannot slip away from the hanging wall. Sections as great as 6 m (20 ft) wide and 4.5 m (15 ft) high have been extracted but, in general, shorter cuts are scheduled in wider stopes to eliminate extensive wall support during the mining cycle. Wall support is provided by 152 × 152-mm (6 × 6-in.) stulls, or larger, placed at right angles to the dip. Rockbolts are often used in wider stopes.

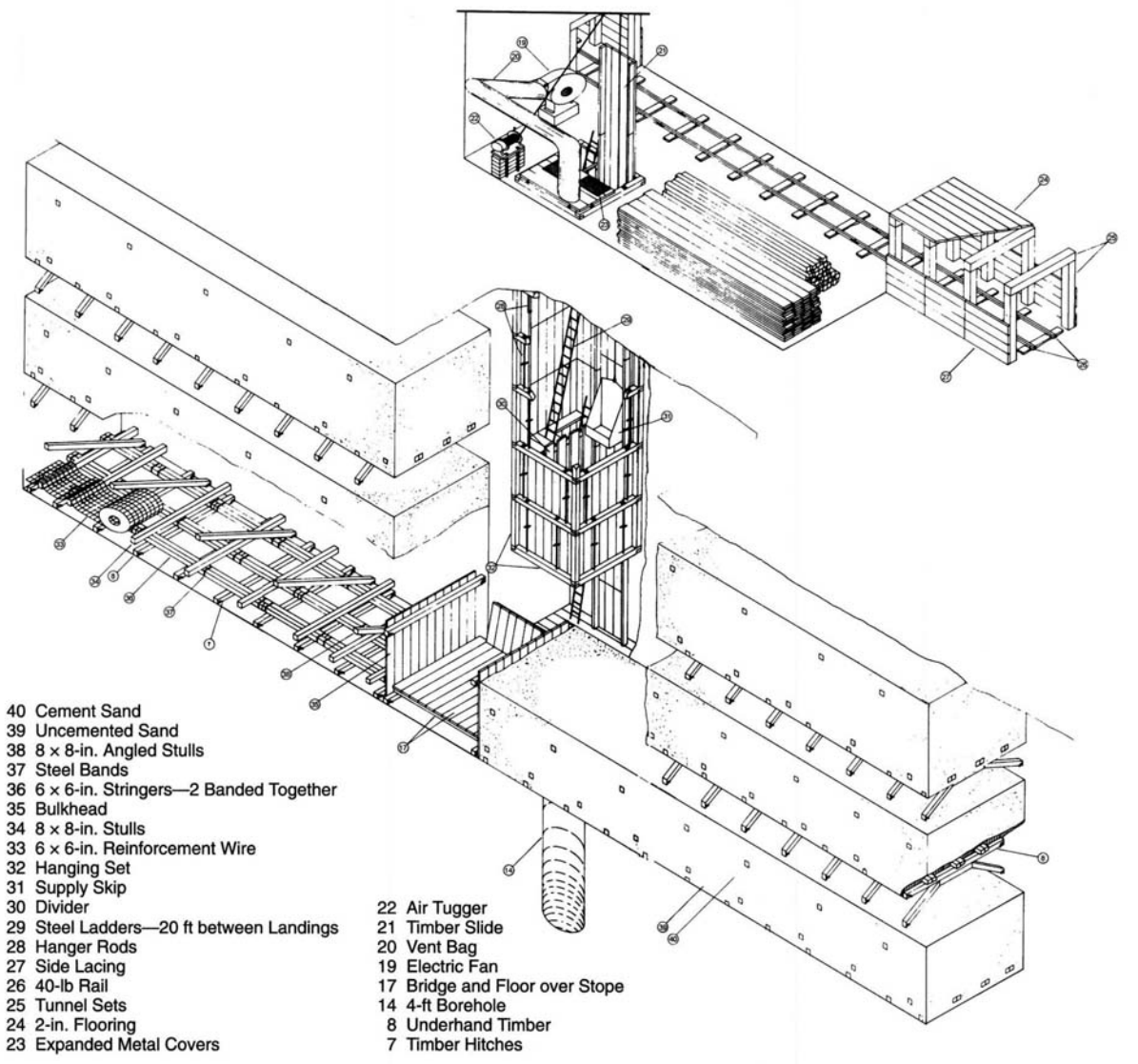


FIGURE 32.2 Isometric view of sand fill preparation

Extraction of the ore is accomplished by blasting with unconventional drill patterns because of ravelly ground, which prohibits charging many of the drill holes. A burn cut is used where the ground is tight and drill holes are kept at least 0.9 m (3 ft) below the sand to prevent damage to support timber above. Miners have learned to quickly adjust to the varying ground conditions found in the vein systems. Regular delay detonators are used with water-gel cartridges for normal blasting in the stopes.

After the stope is mined out and all ore cleaned from the hanging wall, fill preparation begins at the back end of the cut. Sand caps are installed from foot to hanging wall and are supported on each end by 25.4-mm (1-in.) steel saddle

hitches. Installation of 203 × 203-mm (8 × 8-in.) caps is on 0.9- or 1.5-m (3- or 5-ft) centers, depending upon the stope width. Cable ties are placed under the caps and tied back to footwall bolts if additional support is needed. Banded 152 × 152-mm (6 × 6-in.) or 152-mm (6-in.) pole stringers are placed on top of the floor caps parallel to the walls. These stringers are generally positioned on 0.9- to 1.2-m (3- to 4-ft) centers with at least one row near the hanging wall. Reinforcing wire mesh, 152 × 152 mm (6 × 6 in.), is placed to overlie the stringers along the entire length of the stope. Wall-to-wall stulls are later installed to hold the mat in place during the filling process.

Burlap-covered fill walls are built on each side of the raise, and 51-mm (2-in.) distribution lines with valves every 6 m (20 ft) are then installed. Straight sand is poured in the stope until the wire mesh is just covered. An 11 to 12% cement slurry is used for the remaining portion of the fill. Decant water is removed through the milking wall and drained down the raise. Small dams on the level below contain the sand and slime losses from the pour. When the fill is approximately 0.9 to 1.5 m (3 to 5 ft) from the back, sinking is resumed until sufficient room is obtained to permit pouring a cemented sand badge 0.3 to 0.6 m (1 to 2 ft) deep over the previous work platform. This sand bridge is required for hanging wall protection while progressing downward along the raise. Once completed, sinking is resumed to permit installation of manway and service facilities to the floor of the next cut, whereupon the mining process is repeated.

PRODUCTION

In the average stope, a two-person crew will produce approximately 10.8 t (12 st) per miner-shift during the sinking, stope preparation, mining, and fill preparation stages. Productivity, to a large degree, is dependent upon the width of the vein being mined. Dilution is dependent upon the vein and wall rock characteristics. Minimum width for mining is 1.2 m (4 ft). When mining wide veins with competent walls, dilution occurs only from the uncemented portion of the sand fill and varies between 2 and 7%. Ore recovery is essentially 100% except from areas where vein pinches exclude economical recovery.

Major supplies consumed during undercut stope production are as follows: timber, 0.03 m^3/t (11 bd ft per ton); cement, 6 kg/t (12 lb per ton); explosives, 0.7 kg/t (1.4 lb per ton); drill steel, 13.1¢ per ton; and drill bits, 9.3¢ per ton (approximate 1979 prices).

OPERATING COSTS

Direct undercut-and-fill costs for 1976 at the Bulldog Mountain mine were $11.58 per ton (see Table 32.1), and for 1979 were $16.08 per ton (see Table 32.2). These costs per ton include labor and supplies.

TABLE 32.1 Direct undercut-and-fill costs for 1976

Drill, blast, and bar	$ 3.23
Slushing	1.70
Chute drawing	0.02
Timbering, steelwork, manway rockbolting, pipe and ventilation	2.40
Backfilling, sandfill	0.05
Backfilling, pumps	0.26
Backfilling, cement	2.87
Backfilling, sandwalls, brattice, etc.	0.24
Supply handling and equipment	0.21
Supervision	0.60
Total*	$11.58/ton†

*Does not include fringe benefits, equipment maintenance, or overhead charges.

†Metric equivalent: st × 0.9071847 = t.

TABLE 32.2 Direct undercut-and-fill costs for 1979

Mine labor	$ 7.95
Cement	1.92
Supplies	0.97
Explosives	1.01
Timber	1.65
Backfilling	2.58
Total*	$16.08/ton†

*Does not include fringe benefits, equipment maintenance, or overhead charges.

†Metric equivalent: st × 0.9071847 = t.

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CHAPTER 33

Undercut-and-Fill Mining at Falconbridge Mine of Falconbridge Nickel Mines Ltd.

S.A. Tims

INTRODUCTION

The Falconbridge mine orebody extends about 1.6 km (1 mile) in length, and the deepest developed ore is on the 6050 level below surface. The ore zone varies in width from a few inches to more than 30 m (100 ft), and the average width is 4.9 m (16 ft). Access levels are driven in the ore at 53.3-m (175-ft) intervals. The principal method of mining is overhand longitudinal cut-and-fill. Prior to 1962, timber square-set stoping as a secondary extraction method was used for about 15% of the total production. Undercut-and-fill was introduced at Falconbridge in 1962 as a potential replacement for the square-set method in heavy ground. The undercut-and-fill method was developed by Inco in the 1950s, its principal application being to transverse pillar mining. Falconbridge made modifications to this method.

A feature of the mine is the No. 1 flat fault, which dips 0.79 rad (45°) toward the northeast. The main characteristic of the fault is the presence of large swells of ore directly under the plane of the fault. The ground under the fault area is highly fractured and associated with massive sulfides. In the past, the ore under the fault was recovered, with difficulty, by either tight cut-and-fill. or square-set stoping. In the 1970s these methods were supplanted to a large degree by the undercut-and-fill method.

An advantage of the current undercut-and-fill method, which uses cemented fill compared to the cut-and-fill and square-set methods, is the reduction of dilution due to better control of the walls. At Falconbridge mine, it is estimated that the grade of ore produced by undercut-and-fill is improved by approximately 10% over other methods. Where undercut-and-fill is used in very weak ground, a much greater improvement in grade can be expected. Table 33.1 shows mining production for 1974.

The undercut-and-fill method was first used at Falconbridge during 1962. The first longitudinal stope was prepared for undercutting by laying down

TABLE 33.1 Mining production in 1974

	Production, t (st)		% Production	PU* per Miner-shift
Cut-and-fill	548,646	(604,757)	67.9	12.7
Mechanized cut-and-fill	158,900	(175,157)	19.7	26.7
Undercut-and-fill	72,647	(80,080)	9.0	12.3
Square set	19,631	(21,639)	2.4	7.3
Shrinkage	5,693	(6,275)	0.7	
Development	2,559	(2,821)	0.3	
Totals	808,076	(890,729)	100.0	

*PU is a production unit, equal to 0.28 m^3 (10 cu ft), which is approximately equivalent to .907 t (1 st) of ore in place.

laminated beams the length of the stope and installing a lagging mat floor on top of the beams. Unconsolidated tailings fill was poured on top of the mat floor. As the cut advanced under the floor, heavy posts were placed under the laminated beams at 1.8-m (6-ft) intervals. During 1966, a radical change was made to the method when tailings fill, consolidated with portland cement, replaced the unconsolidated fill. This development eliminated the laminated beams and heavy mat floor and greatly improved the stability of the stope. This system, with minor variations, is currently used at Falconbridge mine.

APPLICATION

The undercut-and-fill method is used to mine incompetent ground, sills, or floor pillars under mined-out levels, or a block of ore isolated between levels. It is occasionally used to advantage in sequencing production from various mining blocks. This is done by mining a block of ore by the cut-and-fill method and at the same time mining the ore block directly underneath by the undercut-and-fill method.

The undercut-and-fill mill holes at Falconbridge are either boreholes, stripped timbered raises, or steel mill holes. Boreholes and rock raises tend to slough in heavy and broken ground, which increases dilution when sloughing exceeds the ore width outline and also increases the difficulty of moving down to start the new cut. For example, in one installation, a 1.2-m (4-ft) diam borehole sloughed to a size of 3.7 × 5.5 m (12 × 18 ft).

The undercut-and-fill method usually requires a mill hole extending from the level below the ore to the top horizon of the ore block. The customary methods of providing a mill hole are:

1. A borehole is driven from level to level through the ore block and a chute installed on the bottom level (Fig. 33.1).
2. An existing raise is used as a mill hole. If the raise is timbered, a steel mill hole is installed inside the timber and tailings fill poured around the steel mill hole (Fig. 33.1).

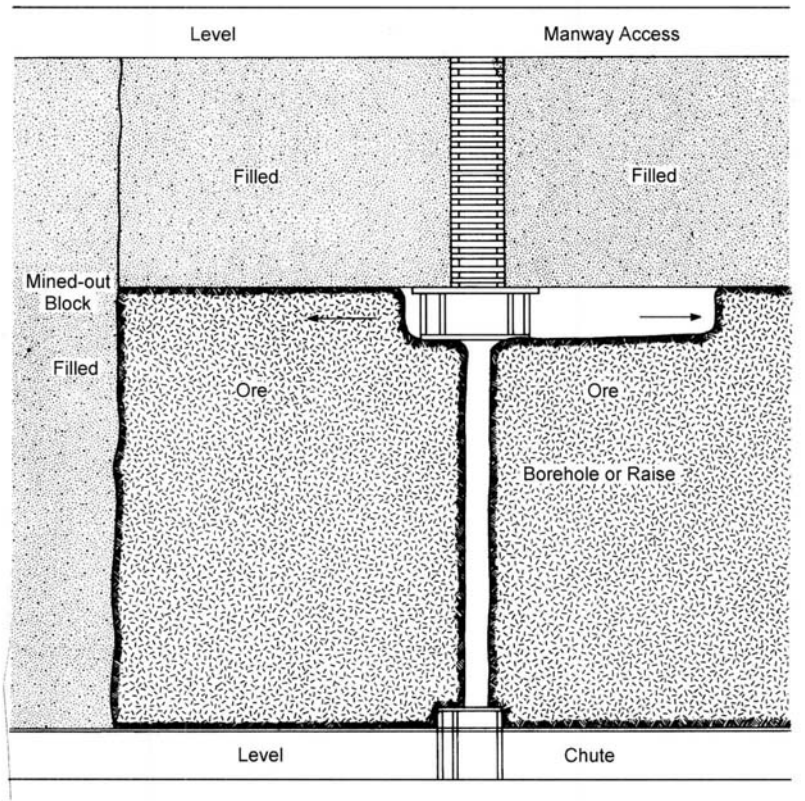


FIGURE 33.1 Standard method

3. An existing steel mill hole, situated at one end of a mined-out stope, is used as the mill hole for an adjacent undercut-and-fill ore block. The mill hole position is determined when planning the mining sequence of the first stope (Fig. 33.2).
4. A stope is designed to be mined by both cut-and-fill and undercut-and-fill. A manway and steel mill hole are raised side by side and situated at the boundary of the cut-and-fill and undercut-and-fill sections. Undercutting to the mill hole begins when the cut-and-fill stope reaches a convenient elevation. Access between the manway and mill hole allows the undercut-and-fill section to be mined independently of the cut-and-fill section (Fig. 33.3).
5. A stope crown pillar (sill) consisting of about 9.1 m (30 ft) of vertical ore extending the length of the stope is undercut to the existing mill holes. A similar technique is used to convert the cut-and-fill method to undercut-and-fill at any elevation between levels (Fig. 33.4).
6. A block of ore, 9.1 to 12.2 m (30 to 40 ft) in height, below a level is undercut-and-fill mined by driving an incline from the level above into the block. The broken ore is slushed up the loading ramp to load into haulage vehicles on the starting level. This procedure is used where mill holes have

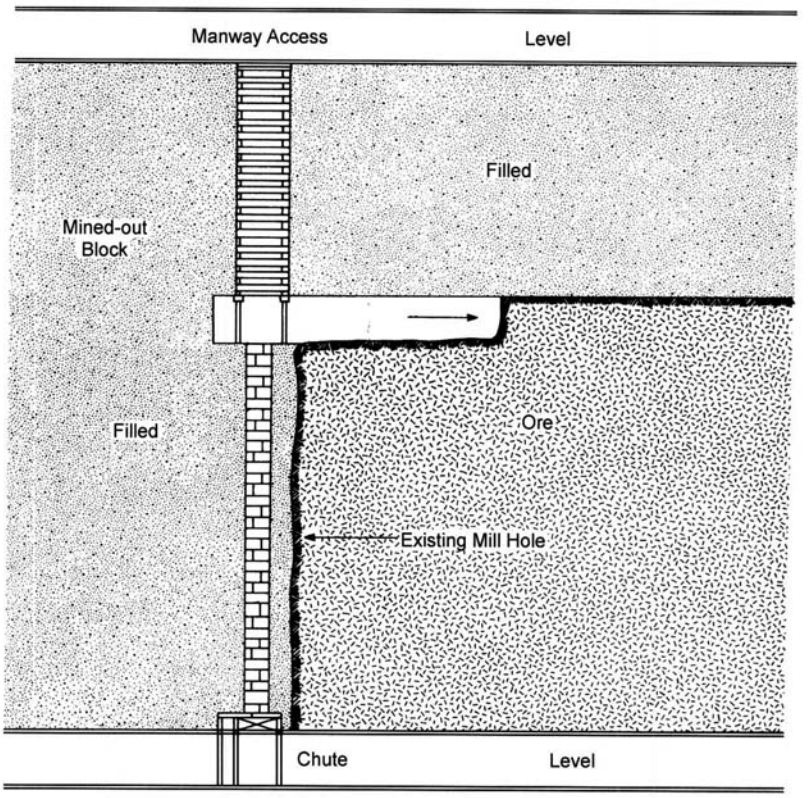


FIGURE 33.2 Mill hole in mined-out stope

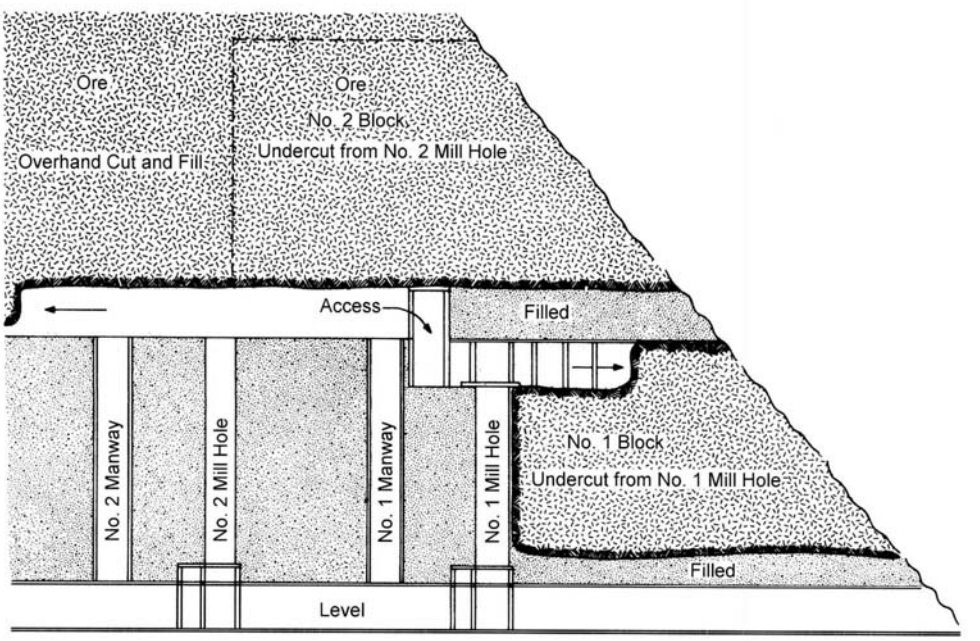


FIGURE 33.3 Cut-and-fill and undercut-and-fill in the same ore block

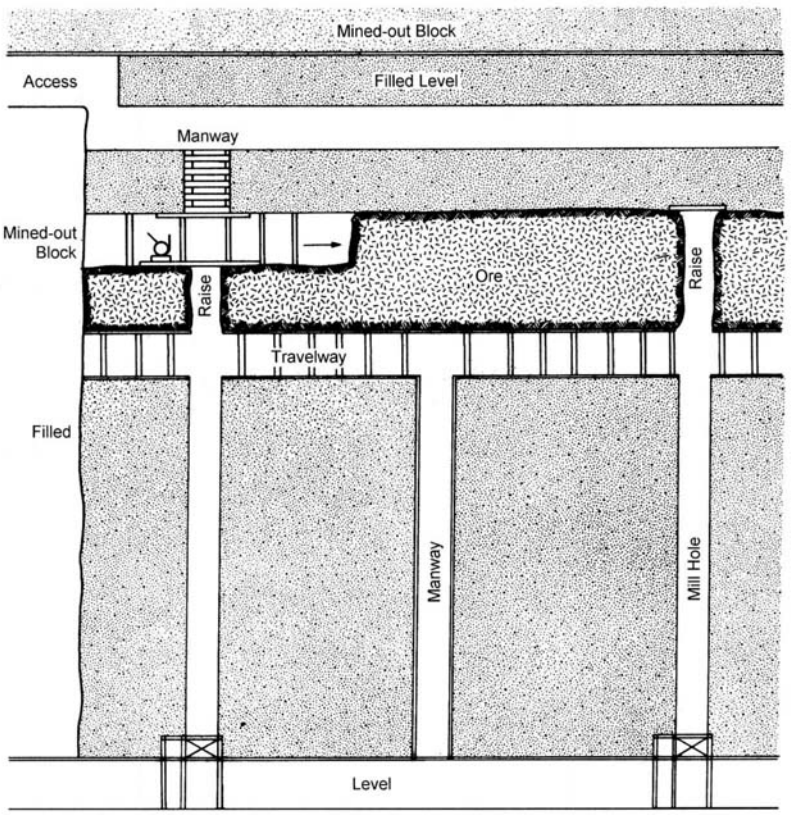


FIGURE 33.4 Undercutting a sill

been lost in underlying mined-out stopes, remnant ore blocks remain below a level floor, or for premining level floor sills in sequencing production from a number of stopes on a given level (Fig. 33.5).

GENERAL METHOD

The mill hole is slashed out from the top down to a depth of 3.4 m (11 ft) to make room to start the new cut. The manway is installed down to the level of the previous floor and posted from the new floor. The area around the manway is filled with tailings fill consolidated with cement. Benches are then driven under the consolidated fill floor from the mill hole to the ore limits. The cut is filled with consolidated fill as soon as completed and the process is repeated until all of the mining above the level of the floor is completed (Fig. 33.6).

STARTING A NEW CUT

The amount of work required to move down and start a new cut depends upon the type and size of the mill hole and the competence of the ground at the mill-hole collar. A wide opening, such as a sloughed raise, requires long steel beams for bridging. Bridging booms of up to 9.1 m (30 ft) in length have been used. These booms were made from 304.8 × 304.8 mm by 4.6-m

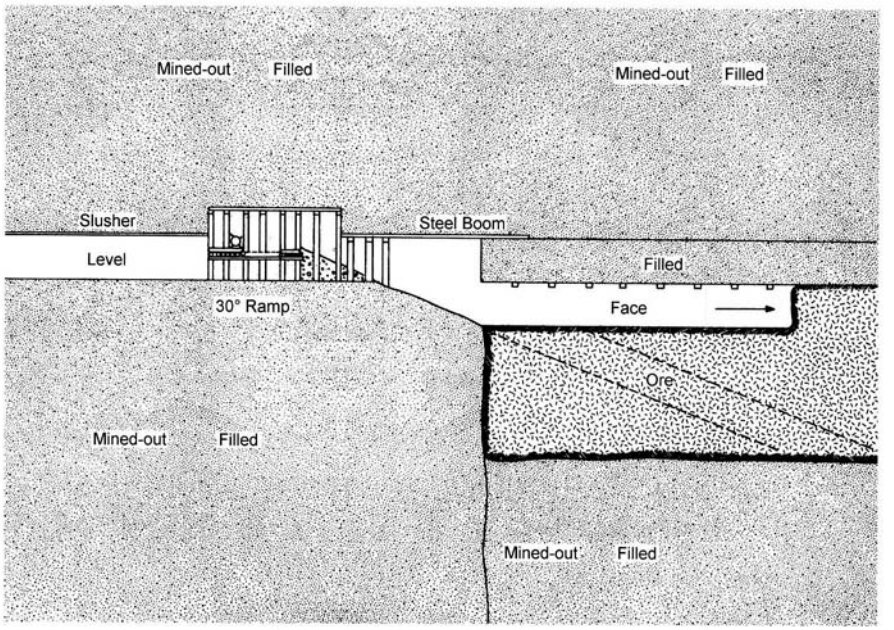


FIGURE 33.5 Silling to level above

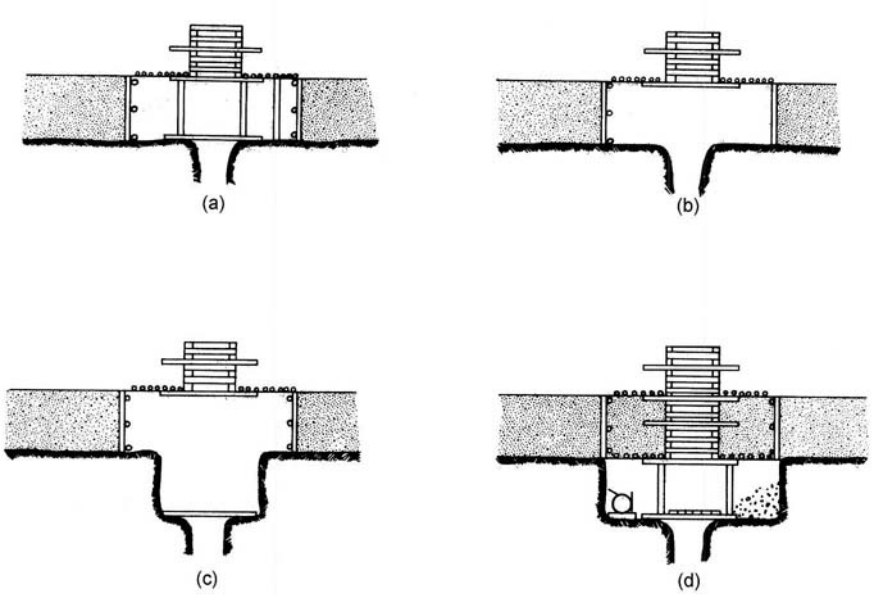


FIGURE 33.6 A general method for undercut and fill: (a) complete mining, fence, and fill both sides to about 4.3 m (14 ft) from manway center line; (b) remove posts, cable up posts, and slusher; (c) slash out mill hole, 3.4 m (11 ft) deep, 2.4 m (8 ft) wide, and 4.6 m (15 ft) long in the direction of the ore; move booms down and block; and (d) hang 2.7-m (9-ft) flats, raise manway, fill around manway, post 2.7-m (9-ft) flats. Take a breast each way. Muck out one side, and lower slusher.

(12 × 12 in. by 15-ft) WF sections spliced end to end. The longest booms currently in use are 254 × 254 mm by 7.3-m (10 × 10 in. by 24-ft) WF beams. When a steel mill hole has been previously installed, there is no bridging problem and booms are not required.

When the mill hole is a raise or borehole, the following procedure is followed:

1. The booms are lifted and secured by cables.
2. The slusher is moved to a convenient corner or lifted up the manway and secured.
3. The mill hole is enlarged by slashing down to a minimum size of 3.4 m (11 ft) deep, 2.4 m (8 ft) wide, and 4.6 m (15 ft) long in the direction of the ore. Holes are drilled and blasted with a light burden and the walls are rockbolted as they are exposed. Mucking is done by hand. In cases where the mill hole has sloughed, it is scaled, bolted, and squared up with short holes. In competent ground, every effort is made to retain a solid ledge on both sides of the mill hole as a base for the steel booms. In some instances, the walls of the raise are bolted below the floor line.
4. The booms are lowered into place and blocked securely 1.8 m (6 ft) apart.
5. The opening is enlarged to a minimum of about 7.3 m (24 ft) by taking a short bench on one or both sides as required. One side is mucked out by hand and the slusher is lowered into place and set up. The other side is mucked out with the slusher. At this point, there is usually room to post the 228.6-mm by 4.3-m (9-in. by 14-ft) manway stringers from the steel booms or from the floor.
6. The manway for the new cut is installed as follows:

    a. Two parallel 228.6-mm by 4.3-m (9-in. by 14-ft) flat stringers support the bottom cribs of the manway. Two 25.4-mm by 1.8-m (1-in. by 6-ft) hanging rods are inserted into predrilled holes in each of the stringers. Two 228.6-mm by 4.3-m (9-in. by 14-ft) stringers are then hung below and in the same direction as the top set of stringers. The process is repeated once more and a second set of stringers is hung in the same way. The final set of stringers rests on two 101.6 × 304.8-mm by 3-m (4 × 12-in. by 10-ft) floor sills that are part of the undercut-and-fill floor.

    b. Manway cribs are 177.8-mm by 2.4-m (7-in. by 8-ft) flats that are dapped in the ends to form 177.8 × 177.8-mm (7 × 7-in.) horns. Commencing with a pair of cribs laid across the bottom set of 228.6-mm by 4.3-m (9-in. by 14-ft) stringers, successive pairs of cribs are laid at right angles to the preceding set to make a manway with 2.4 × 2.4-m (8 × 8-ft) outside dimensions. The cribs are toenailed through the ends. Special length hanging rods are supplied to reach the exact distance.

    For access, the manway is sometimes raised 1.5 m (5 ft) from the bottom stringer. The floor preparation around the manway is then completed and 1.1 m (3½ ft) of consolidated fill poured. The manway is then completed and the remaining space filled with 30:1 consolidated fill.

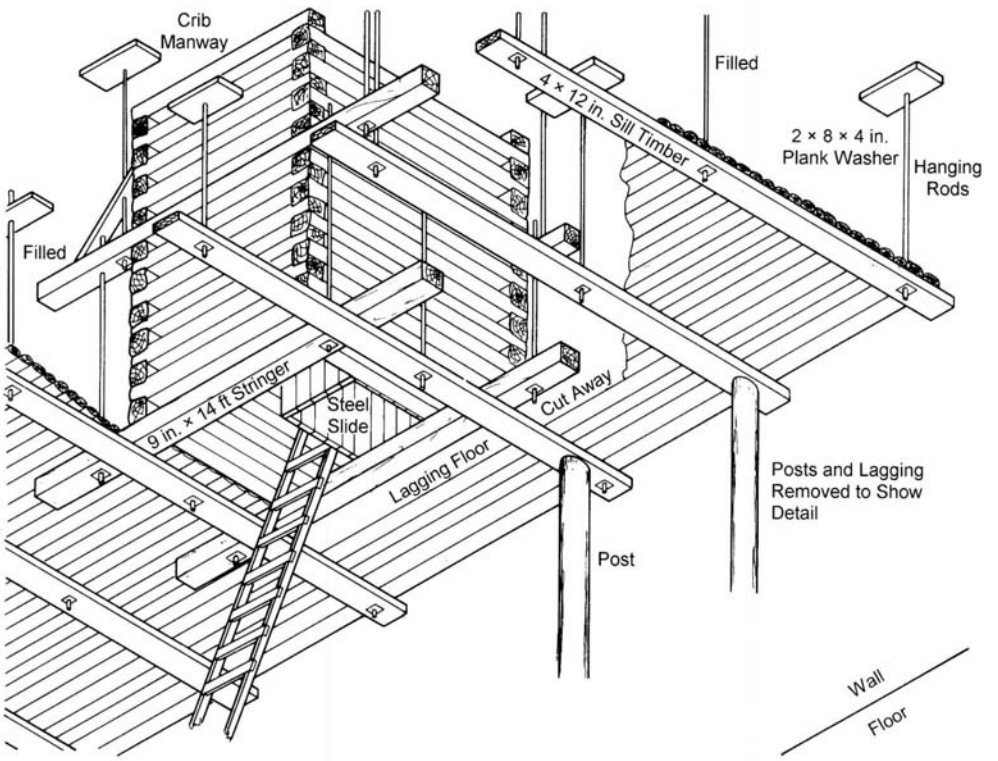


FIGURE 33.7 Manway construction

c. The floor around the manway is prepared for fill by laying 101.6 × 304.8 mm by 3-m (4 × 12 in. by 10-ft) sill planks around the perimeter of the manway and at 1.8-m (6-ft) spacing. The planks are supported by 15.9-mm by 1.2-m (⅝-in. by 4-ft) hanging bolts with 0.6 m by 203.2 × 609.9-mm (2 ft by 8 × 24-in.) plank washers installed on the upper ends. A lagging floor is laid over two sets of sill planks for a maximum distance of 3.7 m (12 ft) from the manway. A 101.6 × 101.6-mm (4 × 4-in.) by 9-gage wire mesh is laid down over the floor. Spaces between the manway and the floor are chinked. The manway is then gobbed with 101.6 × 101.6-mm (4 × 4-in.) 9-gage wire fabric. Both the manway and the floor are covered with a single layer of 24 × 6 weave Tufton (a polypropylene fabric) (see Fig. 33.7).

7. The opening is filled with 1.1 m (3½ ft) of 6:1 consolidated fill and the remaining space with 30:1 consolidated fill.

When a steel mill hole has previously been installed, the procedure is as follows:

1. The mill-hole rings are removed for a vertical distance of 3.4 m (11 ft) by drilling holes in the fill around the rings and blasting lightly. The fill around the segments is hand mucked into the mill hole and the segments

cut up with an oxyacetylene torch for removal. Sufficient room is created to set up a slusher on one side of the mill hole.

2. The manway is installed and the bottom manway stringers posted to the floor of the stope.
3. The floor around the manway is prepared for fill and filled with 1.1 m (3½ ft) of 6:1 consolidated fill and 30:1 consolidated fill. Note that steel booms are not required where a steel mill hole has been provided.

BREAKING AND MUCKING

The standard-size bench is 3 m (10 ft) high, 1.8 m (6 ft) deep, and a maximum of 4.6 m (15 ft) wide. The bench is drilled with jacklegs, 22.2-mm (⅞-in.) drill rods, and 38.1-mm (1½-in.) diam tungsten carbide bits. The face of the bench is drilled with lines of holes from the top down. The first line of holes is drilled at a steep angle to the back (cartwheel). Successive lines of holes are gradually fanned down until a horizontal position is reached. The first holes of the upper lines are drilled with the least burden and break into a line of sawdust bags that were laid during the previous cut-filling cycle. The sawdust bags provide a void to break into. Approximately 22 holes are required to break a 3.7-m (12-ft) wide bench. The holes are loaded with either 25.4 × 304.8-mm (1 × 12-in.) Powermex or ammonium nitrate–fuel oil (ANFO). The shots are ignited with high-strength caps, tape fuse, and igniter cord.

Providing that the bench is properly drilled and blasted, the timber and wire in the floor above will not be damaged.

After the bench has advanced one or two rounds, a hanging staging is installed under the manway to provide better access from the manway to the stope (Fig. 33.8).

Mucking is performed by two-drum air or electric slushers in the 18.6-kW (25-hp) range. Scrapers vary from 914.4 to 1524 mm (36 to 60 in.) in width.

Sheave blocks are hung on 15.9-mm (⅝-in.) cables previously installed under the floor. There is usually little or no scaling. Poor walls are screened with 101.6 × 101.6-mm (4 × 4-in.) 9-gage wire mesh and secured with rockbolts and 50.8 × 203.2 × 609.6-mm (2 × 8 × 24-in.) wood washers.

As the cut progresses, lagging posts are placed under the ends of the 101.6 × 304.8-mm (4 × 12-in.) stringers as required. Narrow sections are not posted. Where the ore is wider than 4.6 m (15 ft), it is either paneled or taken with an adjacent cut. In very wide orebodies, two or more faces may be mined.

The undercut-and-fill method is quite flexible. It is possible to mine around or under waste pillars, to extend or cut back the stope limits, to mine both cut-and-fill and undercut-and-fill at the same time in the same ore block, or to mine several faces at the same time. The method works well in longitudinal stopes and also has been used successfully in transverse stopes and pillar extraction.

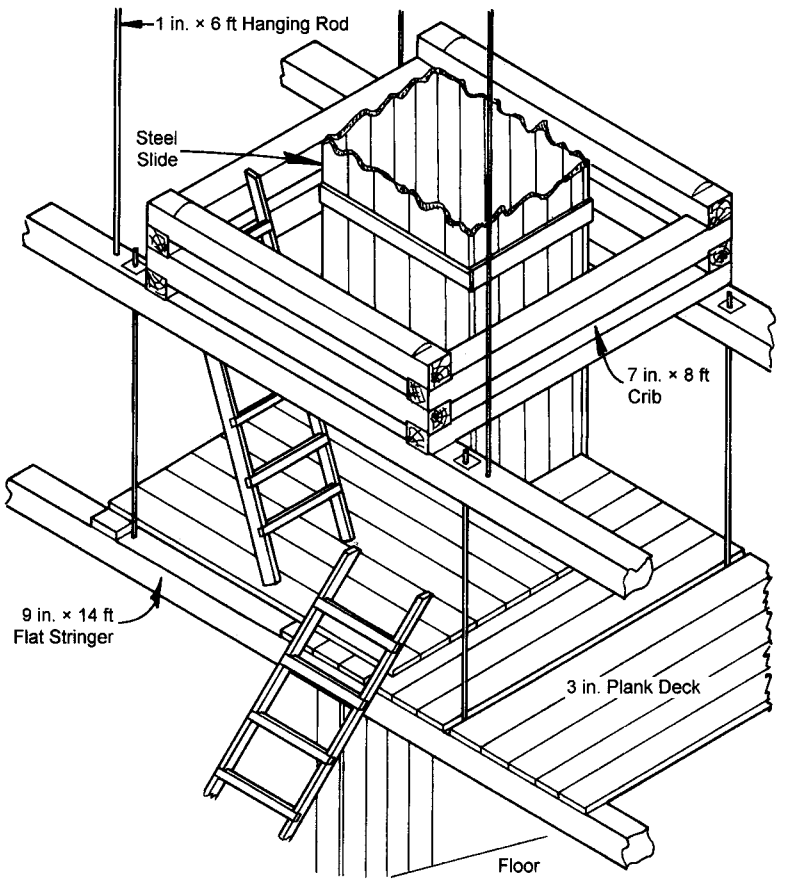


FIGURE 33.8 Hanging staging, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m

Preparation of the Transverse Undercut-and-Fill Stope

The ore blocks on both sides of the undercut-and-fill block were first mined by the overhand cut-and-fill method. As successive cuts in these stopes were filled, a 3-m (10-ft) wide pillar of 30:1 consolidated fill was poured at the stope limit against the undercut-and-fill mining block. In one of these stopes, two 1.4-m (4-ft 8-in.) diam steel mill holes were raised in the 30:1 consolidated fill pillar adjacent to the undercut-and-fill mining block.

The undercut-and-fill stope was mined transversely between the 30:1 consolidated fill pillars. Each cut was mined by a number of comparatively narrow panels. These panels radiated from the steel mill holes that had been previously installed in the adjacent stope (see, for example, Fig. 33.12).

FILL PREPARATION

When breaking is completed and the final muck is slushed out of the stope, it is desirable to leave a few centimeters (inches) of muck for a cushion under the consolidated fill floor. The muck cushion protects the floor from the blast

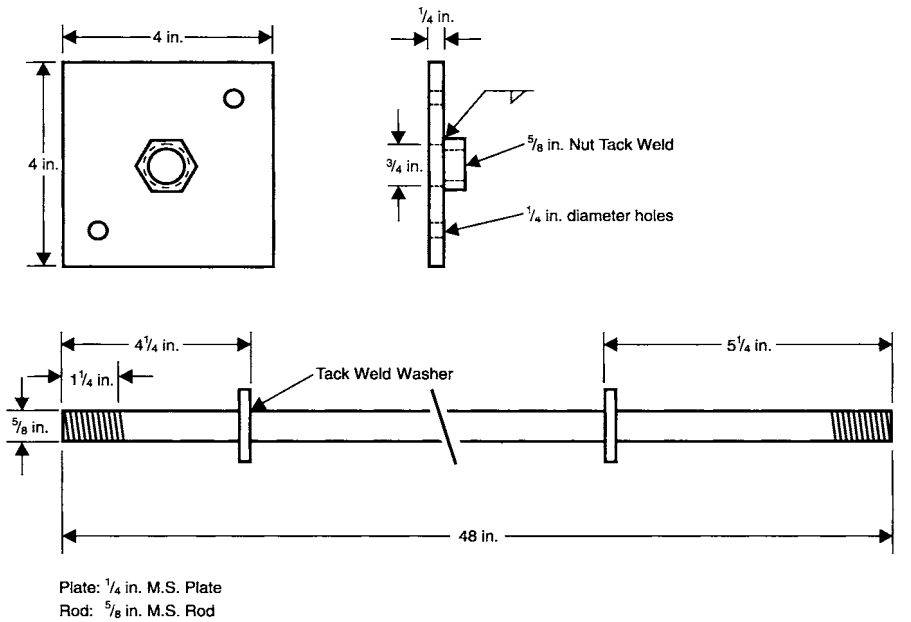


FIGURE 33.9 Support bolt and plate. Note that the bolt and nut must fit snugly. Metric equivalent: in. × 25.4 = mm.

and provides a parting line between solid rock and fill. The muck left under the floor is recovered in the next cut. Holes and dips in the floor are filled with muck and the bottom of the cut made as level as possible. This procedure results in both a level and safe back over the heads of the mining crew.

The next step is to lay a line of sawdust bags down the length of the stope and in the center of the bench. This step is sometimes omitted in cases where the bench breaks easily. Cables, 15.9-mm (5/8-in.) diam, usually two per bench, are laid down and secured to posts or to mucking bolts in the face. The sheave blocks are hung from these cables when mucking out.

The 101.6 × 304.8-mm (4 × 12-in.) sills, usually 3 m (10 ft) long, are laid down 1.8 m (6 ft) apart. The sills may be placed in line or across the orebody as circumstances require. The sills are predrilled with two 25.4-mm (1-in.) diam holes 1.8 m (6 ft) apart. A special support plate-and-nut is nailed over the predrilled hole in the sill. The support plate is installed under the sill (Fig. 33.9). The sills are then covered with 101.6 × 101.6 × 1524-mm (4 × 4 × 60-in.) 9-gage wire mesh, allowing some overlap. Bolts 15.9 mm × 1.2 m (5/8 in. × 4 ft) are inserted through the holes in the 101.6 × 304.8-mm (4 × 12-in.) sills and screwed into the support-plate nuts. Plank washers, 50.8 × 203.2 × 609.6 mm (2 × 8 × 24 in.), are installed on the upper ends of the hanging bolts. The plank washers act as deadmen when buried in the fill.

The sawdust bags and few inches of muck give room for the first holes to break and protect the wire fabric and 101.6 × 304.8-mm (4 × 12-in.) sills from the blast. The 101.6 × 304.8-mm (4 × 12-in.) sills provide a headblock for posts and support the wire mesh. The wire mesh contains any spalling of the

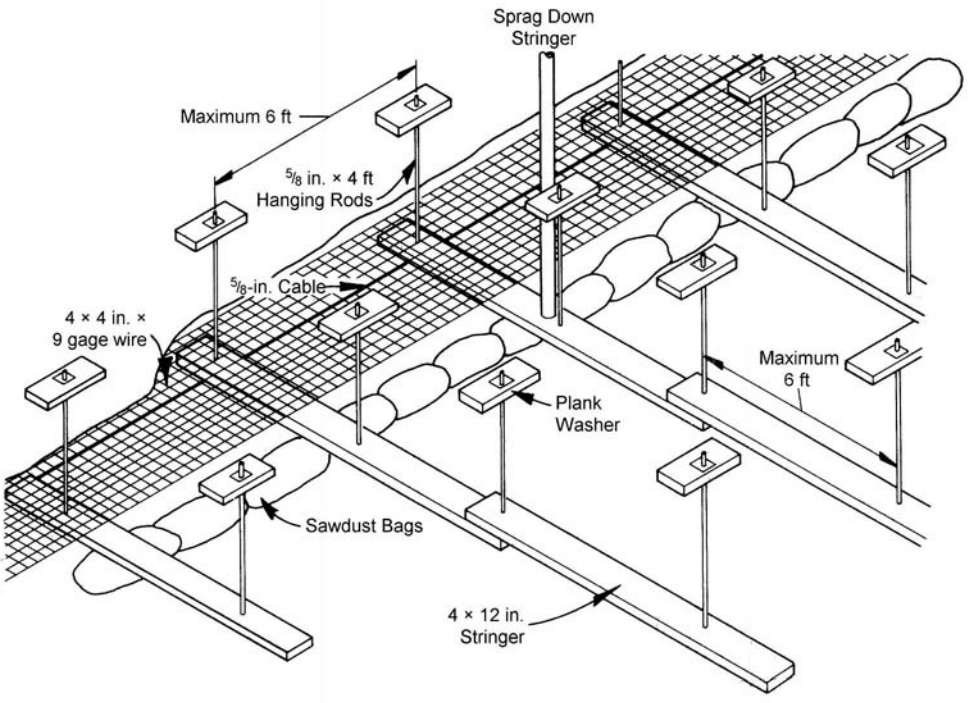


FIGURE 33.10 Undercut-and-fill floor, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m

consolidated fill floor and provides a safe condition over the heads of the mining crew (Fig. 33.10).

Fill fences are made from lagging posts, 25.4-mm (1-in.) plank, and 101.6 × 101.6 × 1524 mm (4 × 4 × 60 in.) by 9-gage wire fabric. The fences are lined with a single layer of 24 × 6 weave Tufton. Intermittent fences are placed at 15.2-m (50-ft) intervals along the length of the stope to regulate the depth of the first enriched consolidated fill pour and to ensure complete filling of the voids under the previous cut. The final fence is placed about 4.3 m (14 ft) from the center of the mill hole. Schedule 40 76.2-mm (3-in.) pipe is used on the levels and in the stope manways. ABS 1.2-MPa (175 psi), 76.2-mm (3-in.) pipe is used for the horizontal discharge lines in the stope. The ABS pipe is installed tight to the back and cannot be salvaged after the pour. Long stopes may require two or more stope lines with right-angle takeoffs.

It is usual to mine out a panel and pour the consolidated fill directly against the rock wall. The only fences are those required to block off the mill hole. When breaking next to a panel of consolidated fill, there is little or no dilution from the fill. Figures 33.11 and 33.12 illustrate the sequencing of panels in undercut-and-fill stopes to provide more than one mining face.

FILLING

Placing the fill is the responsibility of the stope crew. The standard method is to first pour 1.1 m (3½ ft) of consolidated fill with a ratio of six parts of fill to one part of cement. The remainder of the cut is filled with consolidated fill in

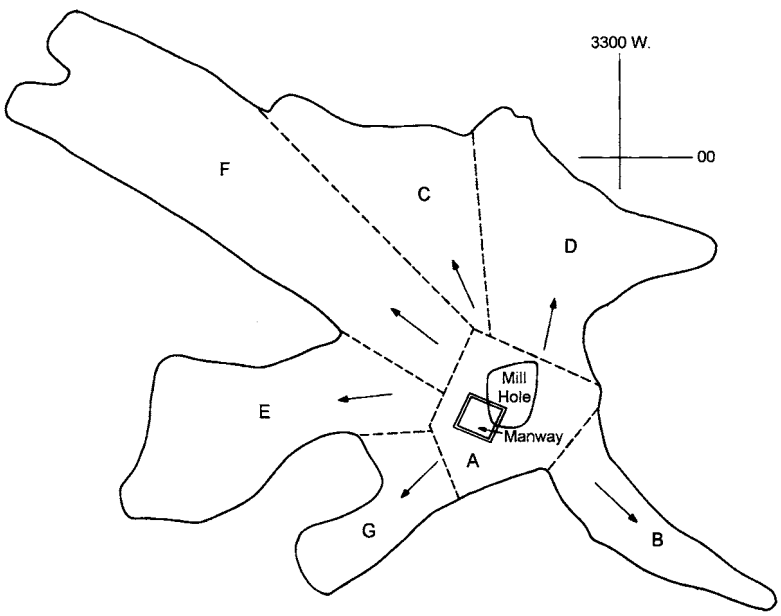


FIGURE 33.11 3602-46-49 stope. Planned mining sequence: Panel A; Panels B and C; Panels D and E; and Panels F and G. Elevation: track 106 ft. Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m.

the ratio of 30 parts of fill to one part of cement. Hydraulic fill contains 70% solids by weight and is delivered at the rate of 90.7 t/h (100 stph). Fill-cement ratios are by dry weights. Up to 10% of screened alluvial sand is added when tailings fill is in short supply. Individual pours are limited to around 24.4 m (80 ft) in length to avoid any possibility of segregation of fill ingredients while being placed.

The 6:1 consolidated fill is poured continuously until the bottom 1.1 m (3½ ft) of the pour is completed. This is done because interrupted pours may cause segregation of the fill. The 30:1 consolidated fill is poured as tightly to the back as possible. Due to drainage water on top of the fill, sometimes there are voids of a few centimeters (inches) left between successive floors. These voids have not caused settling, as was originally feared. At Falconbridge, 1.1 m (3½ ft) of 6:1 consolidated fill and 2 m (3½ ft) of 30:1 consolidated fill will positively support a bench that is 3 m (10 ft) deep and 4.6 m (15 ft) wide.

On occasion, much wider areas have been opened without failure of the floor. The few instances of failure have always been caused by low cement content in the consolidated fill floor. The causes of low cement content are failure or misunderstanding of communications resulting in the wrong fill-to-cement ratio, and poor technique when placing fill, resulting in insufficient depth of 6:1 consolidated fill.

Floor failures are rare and usually small. Such places are supported by timbered roof supports.

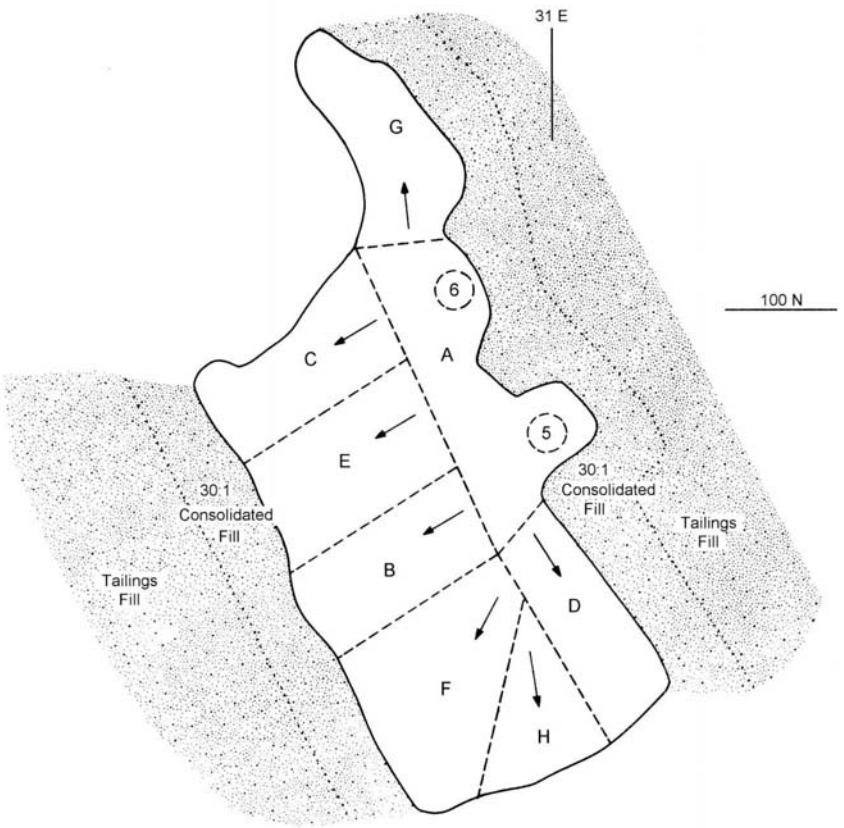


FIGURE 33.12 3602-30 transverse stope. Planned mining sequence: Panel A; Panel B; Panels C and D; Panels E and F; and Panels G and H. Elevation: track 90 ft. Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m.

Fill water drainage is handled in a number of ways. It is desirable to keep boreholes and raises filled with muck to support the walls. The muck in the opening is sealed with a layer of plastic polyethylene film and the drain water pumped to the level above. This procedure also prevents the cement in the drain water from consolidating the muck in the mill hole. Where a steel mill hole is in use, it is pulled empty of ore and the water is drained to the level below.

CONVERTING FROM CUT-AND-FILL TO UNDERCUT-AND-FILL

At Falconbridge, the final 9 m (30 ft) or so under the track (stope crown or level floor pillar) is termed the sill. The undercut-and-fill method is sometimes used to mine the sill under a mined-out level. Ground conditions in a cut-and-fill stope sometimes force a change to the undercut-and-fill method. The process is the same as for silling. The steps are as follows:

1. The level above the stope is prepared for fill. An undercut floor is installed on the level above the stope in the standard way. The gangway timber and track are left in place. When ventilation over the stope must be maintained, a culvert pipe up to 0.9-m (3-ft) diam is installed in the drift. Fences are built at the stope limits and the drift is filled with 1.1 m (3 ½ ft) of 6:1 consolidated fill and the reminder with 30:1 consolidated fill.
2. A travelway is installed at the elevation of the last floor in the stope. This is to provide access to the mill holes and manways.
3. Boxhole raises are driven above the mill holes through the remaining ore block to the filled level. When necessary, a manway compartment is installed in one or more boxholes.
4. A travelway 2.4 m (8 ft) high is driven from the boxholes under the filled level to the stope limits. This travelway gives access, usually at one end only, to the level above. Flats of 228.6 mm (9 in.) are installed at 1.8-m (6-ft) intervals from wall to wall under the filled level for additional support.
5. As soon as the first cut is completed, a second 2.4-m (8-ft) cut is taken.
6. An undercut-and-fill floor is laid and a 2.4 × 2.4-m (8 × 8-ft) undercut-and-fill crib manway is installed over the boxholes for 2.4-m (8-ft) height.
7. The cut is filled with consolidated fill in the usual manner for a height of 2.4 m (8 ft). This creates a travelway 2.4 m (8 ft) high underneath the filled level and a 2.4-m (8-ft) undercut-and-fill floor on top of the remaining ore.
8. Access to the stope is usually from the top level. When desired, existing manways in the stope can be used.

Another application of the undercut-and-fill method is to mine the sill under a mined-out level when there is no mill hole available (see Fig. 33.4). The procedure is as follows:

1. The level above the stope is prepared for an undercut-and-fill floor. Fences are installed at the ore limits and the level filled with consolidated fill in the usual manner.
2. A section of gangway timber is removed and retimbered to inside dimensions of 4 m (13 ft) high, 1.8 m (6 ft) wide, and 8.8 m (29 ft) long. A space of 9.1 m (30 ft) is left between the section of high gangway and the ore limit of the stope.
3. A slusher loading ramp to accommodate two 1.7-m^3 (60-cu ft) cars is built in the high gangway.
4. A 0.52-rad (30°) incline is driven from the end of the ramp to a point just under the start of the filled level.
5. The gangway above the incline is supported by steel beams spliced end to end, by posts, and by cross beams installed wall to wall.
6. The first cut is benched out 3 m (10 ft) high under the filled drift. Additional support to the floor is provided by 228.6-mm (9-in.) flats installed wall to wall at 1.8-m (6-ft) intervals.
7. After the cut has advanced under the floor for a distance of 15.2 m (50 ft), a second slusher is set up to one side and the ore is slushed up to the loading ramp in two stages.

CONCLUSION

From 1962 to 1974, a total of 453,854.5 t (500,289 st) of ore was produced from undercut-and-fill stopes. Miners were able to change from cut-and-fill to undercut-and-fill with little difficulty, and the method was popular with the work force. Although primarily used for poor ground conditions, some undercut-and-fill stopes have achieved productivity efficiencies and costs comparable to cut-and-fill stoping. Experience at Falconbridge shows that the best results are obtained when it is possible to undercut to a previously installed steel mill hole.

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CHAPTER 34

Cost Calculations for Underhand Cut-and-Fill Mining

Gordon M. Pugh and David G. Rasmussen

INTRODUCTION

Underhand cut-and-fill mining is a variation of cut-and-fill mining in which either the wall rock or the vein is too unconsolidated to permit safe mining by conventional cut-and-fill methods.

Basic development consists of driving a 2.4 × 2.7-m (8 × 9-ft) haulage heading laterally in the footwall side of the vein. Crosscuts are driven over to the vein at 91-m (300-ft) intervals. Then the vein, at each intersection, is mined out to a distance of about 9 m (30 ft) above the track level. After the installation of an ore chute, scrams or intermediate drifts are driven in both directions along the vein. Mining then starts immediately below the level bounding the upper limit of the stope. A cut varying between 3 and 4.2 m (10 and 14 ft) in depth and mined for a distance of 45.7 m (150 ft) along the strike is taken from the orepass in each direction (Fig. 34.1).

In this mined-out void, the floor of the cut is equipped with 203 × 203-mm (8 × 8-in.) caps hitched into the wall with steel saddle hitches installed on 1.5-m (5-ft) centers. Double wood stringers, 152.4 × 152.4 mm (6 × 6 in.), are then installed on 1.2-m (4-ft) centers longitudinally down the length of the stope. Reinforcing wire mesh is also placed, overlying each mat of timber. Directly above the stringers, stulls are installed from hanging wall to footwall to hold the timber in place during the sand-filling operation. Then a cement-sand mixture in a ratio of about 1:10 weight is poured to a depth of about 2.4 m (8 ft). The entire cycle is then repeated down to the scram drift.

Another variation of timbering is illustrated in Fig. 34.2. There is a vein width of approximately 5.5 m (18 ft) with the same unconsolidated ground conditions. The primary difference between handling this situation and the previously described undercut-and-fill method is that standard timbered sets are installed in each excavated stoping void rather than stulls. Once again the basic method is to work down the ore block from an upper timbered drift to a

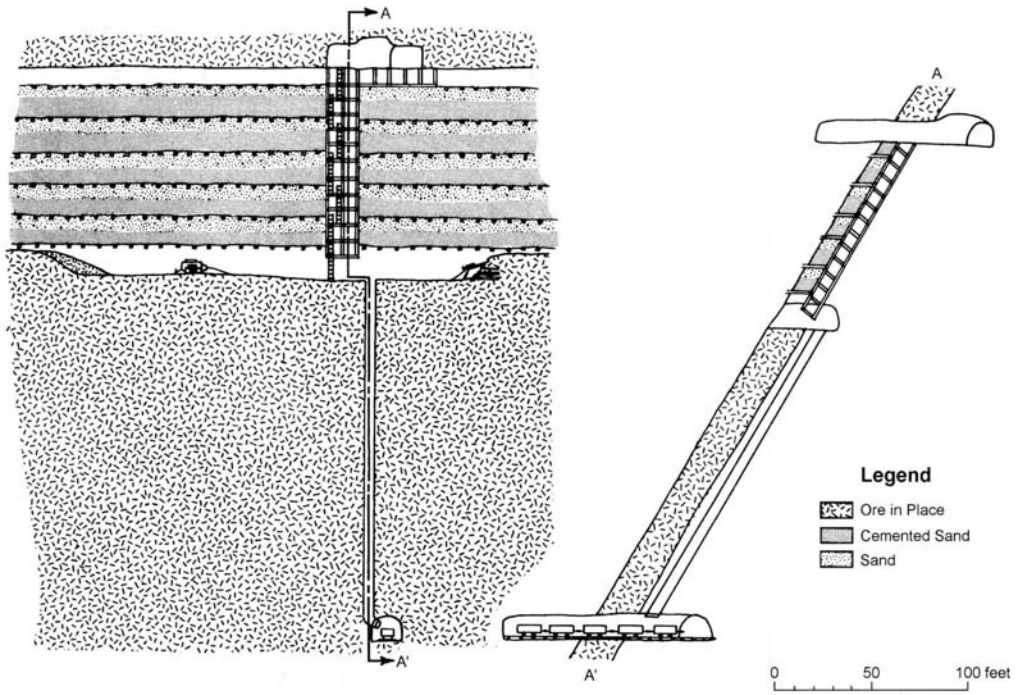


FIGURE 34.1 Underhand stoping method

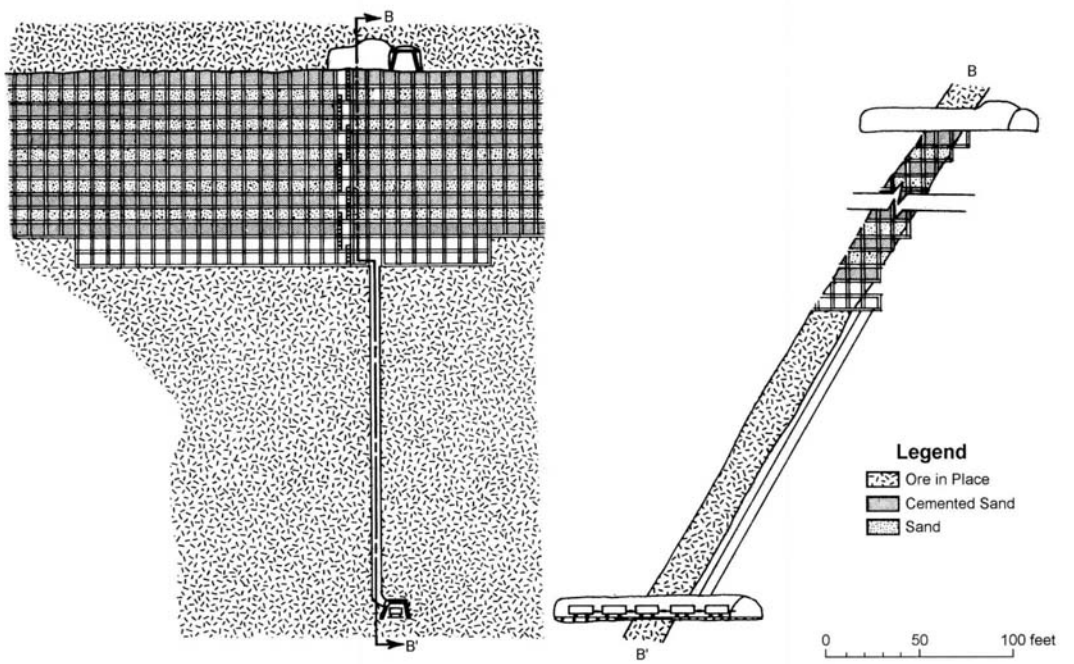


FIGURE 34.2 Variation of square-set timbered underhand stoping

lower timbered drift. Cuts are taken from a timbered raise with a chute on the stope side. Each piece of ground excavated is essentially two sets high, 5.5 m (18 ft), by two sets wide, 5.5 m (18 ft), by the length of the cut along the strike of the vein. The timbered sets are 2.4 × 2.4 × 2.4 m (8 × 8 × 8 ft). The net result is, therefore, two sets of timber on a top floor separated by doubled-up 76-mm (3-in.) lagging from two sets of timber on the bottom floor. The basic advantage of splitting each total stope cut in half is that drilling can be taking place above while mucking with a rubber-tired vehicle or slushing is being done below. Thus the complete cyclic nature of the excavation is lessened. When the endline of the stope is reached, lagging between floors is removed and flooring and matting is put down on the bottom of the lower floor. A 1:12 cement-to-sand mixture is poured for the first 1.2 m (4 ft) and then the balance of the stope excavation is filled with a 1:25 ratio. The stoping cycle is again repeated, starting back at the raise.

COSTS AND PRODUCTIVITIES

The costs derived were calculated based on knowledge of both the productivity and cycle time elements. Costs for equipment, fuel, tires, and other expendables are 1976 figures, and equipment hourly operating costs for each piece of equipment are included at the end of the entire cost section. The labor cost used has been fixed at $10.50 per hour, based on average rates at underground mines in the United States in 1976 and including 35% markup for fringe benefits and 25% for an incentive payment system. For detailed cost calculations, see also Pugh and Rasmussen (1977), Section 2.2, Chapter 13; Section 3.1, Chapter 4; Section 3.2, Chapters 10 and 11; and Section 3.4, Chapter 7.

The total mining cost, based on these assumptions, consists of two parts: development costs and stoping costs. Development costs are determined by first creating a development plan of drifting, raising, and chutes or mill holes. Several specific development plans are considered for each stoping method. Prior to this, appropriate unit costs were applied to the development plan. Total costs calculated from dollar-per-foot figures were then expensed over the ore block directly associated with the development. The development costs do not include shafts, stopes, or other major support systems, but only the costs incurred for the development of the specific tonnage block.

Stoping costs were calculated, using derived unit operation costs, for drilling, blasting, mucking, stope preparation, and ground support installation when required. Selection of proper unit operations were made for specific stoping cases and their related costs compiled to give a stoping cost.

It should be reemphasized that costs given include labor, direct operating costs of equipment, and expendables. These costs have been allocated to the assigned tonnage for the example stoping block. Equipment capitalization and depreciation have not been included.

Costs and productivities for underhand cut-and-fill are given in Table 34.1. Productivity should be approximately 8.6 t (9.5 st) per miner-shift, and the total mining cost is $13.38/t ($12.14 per st).

TABLE 34.1 Underhand cut-and-fill stoping costs and productivities

Unit Operations	Unit Cost, $ per st* of Ore	Vein Width 10 ft* Underhand Cut-and-Fill, Wall Rockbolted, Cemented Sand Fill on Timbered Mat, Ground Condition Poor	Qualifications
Development			
10-ft vein	1.63	1.63	8-hr shift, 6.5 hr effective time at face
Drill and blast			Labor rate $10.50/hr includes bonus and fringe
Underhand breast, 10-ft vein 58.1 st jackleg	2.08	2.08	Unit operation costs include labor
Mucking	1.10	1.10	Equipment costs include maintenance, parts and labor, tires, and fuel
Slusher, 25 hp*	0.78	—	Equipment depreciation is not included
Cavo 511, pneumatic	0.92	—	
LHD, 1 cu yd,* diesel			
Ground support			
Rockbolts, 10-ft wide vein	1.52	1.52	All materials are based on 1976 prices
			Hydraulic sand fill taken to be $1.25 per st of ore extracted
Backfill			
Hydraulic sand	1.25	—	
Hydraulic sand/cement, 10-ft vein	2.61	2.61	
Stope preparation			
Underhand fill preparation, 10-ft vein	3.20	3.20	
Total cost per st		12.14	
Stope crew size		3	
Productivity, st per miner shift		9.5	

*Metric equivalents: ft × 0.3048 = m; cu yd × 0.7645549 = m^3; st × 0.9071847 = t; hp × 0.7456999 = kW.

Timber and filling costs for timbered underhand cut-and-fill are given in Tables 34.2 through 34.4. For additional details on development costs, refer to Pugh and Rasmussen (1977), Section 2.2, Chapter 13; Section 3.1, Chapter 4; Section 3.2, Chapters 10 and 11; and Section 3.4, Chapter 7.

REFERENCE

Pugh, G.M., and Rusmussen, D.G., 1977, "Stope Mechanization–Vein Mining," U.S. Bureau of Mines HO 26024, August, Dravo Corp.

TABLE 34.2 Underhand cut-and-fill stoping sand filling cost, 10-ft vein

Cost of Sand Fill and Stope Preparation*

1. Assume basic hydraulic fill in place is $1.25 per st.
2. Floor of stope to be prepared by:
   a. Placing 6 × 8-in. stringers across stope on 5-ft centers.
   b. Two double rows of laminated stringers (6- × 8-in.) and two single rows of stringers (along walls) for length of stope
   c. 6-in.-diam stulls installed on 5-ft centers on top of longitudinal stringers
   d. 4 × 4-in. wire mesh laid on stulls
   e. Two sealed timber bulkheads per 300-ft stope block
   f. Cement added to fill at 12:1 ratio for 2 ft of fill and 25:1 ratio for 10 ft of fill
3. Materials

a.	Transverse stringers: 60 (6 × 8 in. by 10 ft)	2400 bd ft†
b.	Longitudinal stringers: 6 (6 × 8 in. by 300 ft)	7200 bd ft
c.	6-in. diam stulls: 60 (6 in. diam by 10 ft)	1413 bd ft
d.	Wire mesh: 300 × 10 ft	3000 sq ft
e.	2 bulkheads: 2 × 10 × 12 ft by 3 in.	720 bd ft
f.	Cement: (300 × 10 × 2 ft) by 0.05 st fill per cu ft ÷ 12	25 st
	(300 × 10 × 10 ft) by 0.05 st fill per cu ft ÷ 25	60 st
	Cost preparation —timber 11,733 × $0.28 per bd ft	$3,285.24
	—mesh 3000 sq ft × $0.11 per sq ft	330.00
		$3,615.24
	Miscellaneous materials at 10% of timber and mesh	361.52
	Total	$3,976.76
	Cement cost 85 st at $41.30 per st	$3,510.50

4. Labor

Extra labor is required to pin stringers and stulls, install all timber and mesh

Cycle for 15-ft advance of stope preparation

Raise timber 1.5 hr × 2 workers	= 3 worker-hours
Transport 1.0 hr × 2 workers	= 2 worker-hours
Install 2.0 hr × 2 workers	= 3 worker-hours
Pin timber 2.0 hr × 2 workers	= 4 worker-hours
	12 worker-hours
at 50-min hr	14.4 worker-hours

For 300-ft stope = $\frac{300}{15} \times 14.4$ = 288 worker-hours

Miscellaneous time, travel, and lunch = $\frac{288}{6.5} \times 1.5$ = 66.5 worker-hours

Summary

Short tons extracted = $300 \times 10 \times 12 \text{ ft} \times \frac{162}{2000}$ = 2916 st

Sand Fill Cost

Hydraulic sand	= $1.25 per st
Cement	= $1.36 per st
Total	= $2.61 per st

Stope Preparation

Raise two chutes $\frac{\$135.10 \text{ per ft} \times 12 \text{ ft}}{2916 \text{ st}}$ = $0.56 per st

Preparation for Fill

	Labor	Supplies
Labor, 288 worker-hours at $10.50/hr	$3,024.00	
Supplies		$3,976.76
Travel, lunch, etc., 66.5 worker-hours × $10.50/hr	$ 698.25	
	$3,722.25	$3,976.76

Total = $3722.25 + $3,976.76 = $7,699.01

Cost per st = $\frac{\$7,699.01}{\$2,916.00}$ = $2.64

Total preparation cost per st = $3.20

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; sq ft × 0.0929034 = m^2; cu ft × 0.02831685 = m^3; bd ft × 0.002359737 = m^3; st × 0.9071847 = t.

†Board feet (bd ft) = $\frac{\text{length (ft)} \times \text{width (in.)} \times \text{thickness (in.)}}{12}$

TABLE 34.3 Underhand square-set timbering costs, 18-ft wide vein

Square-Set Timber Installation*

1. Mining by carrying two set high face, 16 ft

2. Install top two sets on muck pile, then mucking out and install bottom two sets

3. Timber required for 8-ft advance (4 sets)

Posts 6 (8 ft by 8 × 8 in.)	256 bd ft
Caps 10 (8 ft by 8 × 8 in.)	428 bd ft
Lagging 2 (8 × 8 ft by 3 in.)	384 bd ft
Cribbing 12 (8 ft by 8 × 6 in.)	384 bd ft
Blocks 4 (12 × 12 × 12 in.)	48 bd ft
Total	1500 bd ft

4. Installation cycle

Hoist timber	1.0 hr × 2 workers =	2 worker-hour
Transport to face	1.5 hr × 2 workers =	3 worker-hour
Stand set	7.0 hr × 2 workers =	14 worker-hour
Clean-up	0.5 hr × 2 workers =	1 worker-hour
		20 worker-hour
Total 50-min hr		24 worker-hour

5. Cost

	Labor	Supplies
Labor, 24 worker-hour × $10.50/hr	$252.00	
Supplies		$420.00
Miscellaneous travel, lunch, etc., 5.5 worker-hour × $10.50/hr	$ 57.75	
Small tools, etc., 6% labor cost		$ 18.59
	$309.75	$438.59
Total	$748.34	

Total short tons extracted = $18 \times 16 \times 8$ ft $\times \frac{162}{2000}$ lb per cu ft = 186.62 st

Cost per st = $4.01

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; bd ft × 0.002359737 = m^3; cu ft × 0.02831685 = m^3; st × 0.9071847 = t.

TABLE 34.4 Underhand square-set filling costs, 18-ft wide vein

Cost of Sand Fill and Stope Preparation

1. Assume basic cost of hydraulic fill in place to be $1.25 per st
2. Floor of stope to be prepared by:
   a. Laminated stringers longitudinally placed
   b. 6-in. diam stull installed on stringers on 5-ft centers
   c. Wire mesh laid on stulls
   d. 4 sealed bulkheads installed on 300-ft stope lift
   e. Cement added to fill at 12:1 ratio for first 4 ft, 25:1 for 12 ft
3. Materials required per stope lift, 16 ft

a. Stringers 3 each × 300 ft × 2 (6 × 8 in.)				7200 bd ft
b. Stulls 60 each × 18 ft by 6 in. diam				2543 bd ft
c. Mesh 300 × 18 ft				5400 ft
d. Bulkhead 4 each × 18 × 18 ft by 3-in. timber				1728 bd ft
e. Cement	(300 × 18 × 4 ft) × 0.05 st fill per cu ft ÷ 12	=		90 st cement
	(300 × 18 × 12 ft) × 0.05 st fill per cu ft ÷ 25	=		130 st cement
Stope preparation	—timber 11,471 bd ft at $0.28 per bd ft	=		$3212.00
	—mesh 5400 sq ft at $0.11 per sq ft	=		594.00
	—miscellaneous materials at 10% of timber and mesh	=		380.60
Total				$4,186.60

Cement cost 220 st at $41.30 per st = $9,086.00

4. Labor

Extra labor to lay stringers, install stulls and mesh.

Cycle for 16-ft stope advance:

Raise timber	1.0 hr × 2 workers	=	2.0 worker-hour
Transport	1.0 hr × 2 workers	=	2.0 worker-hour
Install timber and mesh	3.5 hr × 2 workers	=	7.0 worker-hour
Total			11.0 worker-hour
at 50-min hr			13.2 worker-hour

For 300-ft stope = $\frac{300}{16} \times 13.2 = 247.5$ worker-hour

Travel and miscellaneous worker-hours = $\frac{247.5}{6.5} \times 1.5 = 57.1$ worker-hour

Summary

Short tons of ore extracted = $300 \times 18 \times 16 \text{ ft} \times \frac{162}{2000} = 6998$ st

Stope cement fill per short ton ore

Cement = $\frac{\$9{,}086}{6998} = \$1.30$

Hydraulic sand = $1.25

Total = $2.55

Stope Preparation

Raise two chutes $\frac{\$135.10 \text{ per ft} \times 16 \text{ ft}}{6998}$ $0.31 per st

Preparation for Fill	Labor		Supplies
Labor 247.5 worker-hour at $10.50 per hr	$2,598.75		
Supplies			$4,186.60
Miscellaneous travel and lunch, 57.1 worker-hours × $10.50/hr	$ 599.55		
	$3,198.30		$4,186.60
Total =		$7,384.90	

Cost per st = $\frac{7,384.9}{6998} = \$1.06$ per st

Stope preparation total = $1.47 per st

*Metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; sq ft × 0.09290304 = m^2; cu ft × 0.02831685 = m^3; bd ft × 0.002359737 = m^3.

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SECTION 4

Sublevel Caving

CHAPTER 35

Sublevel Caving: An Introduction

E.W. Cokayne

GENERAL DESCRIPTION OF THE METHOD

The original application of sublevel caving was in ground so weak that it would collapse even in small headings when the support was removed. Heavily timbered drifts were driven across the orebody; the timber was removed at the end of the drift, and the ore caved and slushed out. When dilution became excessive the next set was removed and so on. This method gave high dilution and poor recovery and was slow, but it was the only way to mine this type of orebody at that time.

The method has since been adapted to stronger ground that needs to be drilled and blasted, so it is not a caving method as far as the ore is concerned. However, the method does rely on the walls to cave and so the name sublevel caving has been retained. The method as presently generally applied is shown in Figs. 35.1, 35.2, and 35.3.

Sublevels are established at approximately 7.6- to 12.2-m (25- to 40-ft) vertical intervals. They are usually accessed from a ramp system.

A haulage drift is driven down the strike of the orebody, but in the waste. Production drifts are turned off the haulage drift and driven across the orebody on about 10.7-m (35-ft) horizontal centers. This divides the orebody into a geometrical pattern.

Slot raises are driven at the end of the drifts near the ore-waste contact and the slot is expanded to the shape of the fan drilling. Up holes are drilled to break toward the slot at burdens of 1.2 to 1.8 m (4 to 6 ft). They are drilled past the next sublevel to the production drift immediately above.

When a sufficient number of fans has been drilled, production can start by blasting one or two fans at a time toward the slot. Load-haul-dump (LHD) units are used to muck the ore, which rills out into the drift. At first, when

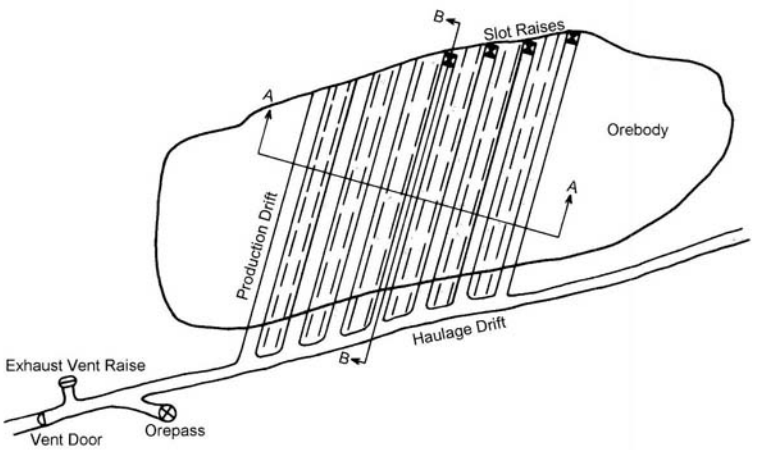


FIGURE 35.1 Typical layout plan

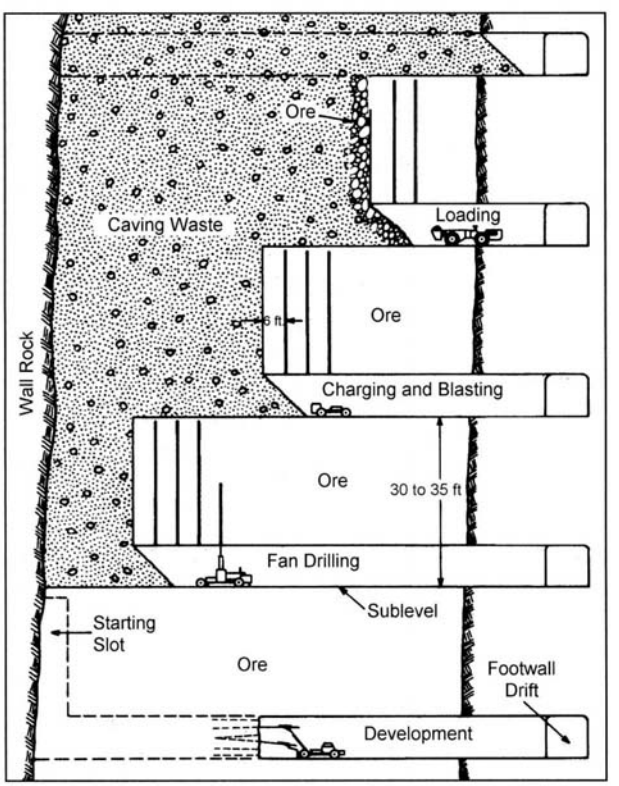


FIGURE 35.2a Diagrammatic representation of a typical sublevel caving operation, showing the four principal operations, metric equivalent: ft × 0.3048 = m

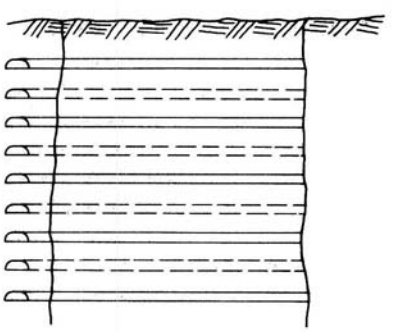


FIGURE 35.3b Typical layout, section B-B

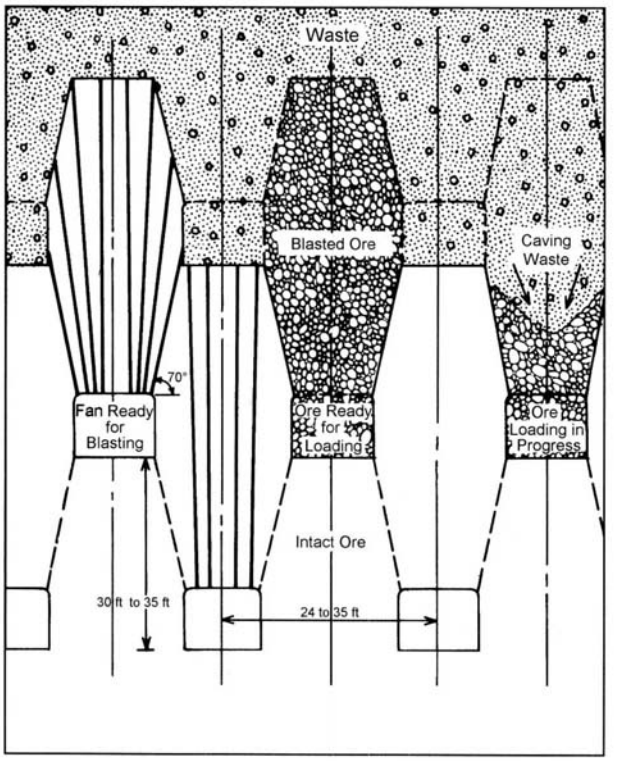


FIGURE 35.3a Diagrammatic cross section of a typical sublevel caving operation showing the sublevel arrangement, metric equivalents: ft × 0.3048 = m; ° × 0.01745 = rad

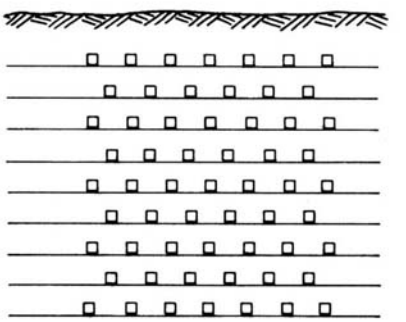


FIGURE 35.3b Typical layout, section A-A

only a small area is mined, the walls may not cave, and the operation will be similar to open stoping. Later, sloughing from the walls will fill any voids created, and the ore after blasting will be contained by the caved waste and gravitate downward as mucking proceeds.

This process is repeated in each of the production drifts as the face is retreated toward the haulage drift. When the top level has been retreated back a safe distance, production can be started from the level below and so on until the orebody is removed. A strict sequence of mining has to be followed.

This general description is common to all sublevel caving, but layouts are modified to conform to the particular shape and dip of the orebodies they are designed to serve.

THE APPLICATION OF SUBLEVEL CAVING

Rock Characteristics

The major problems with sublevel caving are the control and minimization of dilution. From the description previously given, it will be evident that the

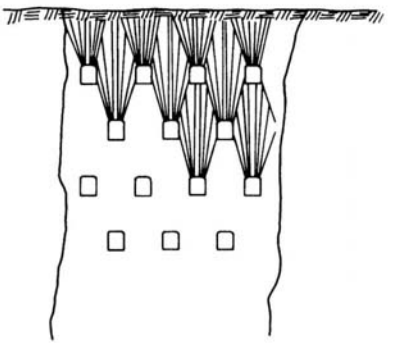


FIGURE 35.4 Steep dip

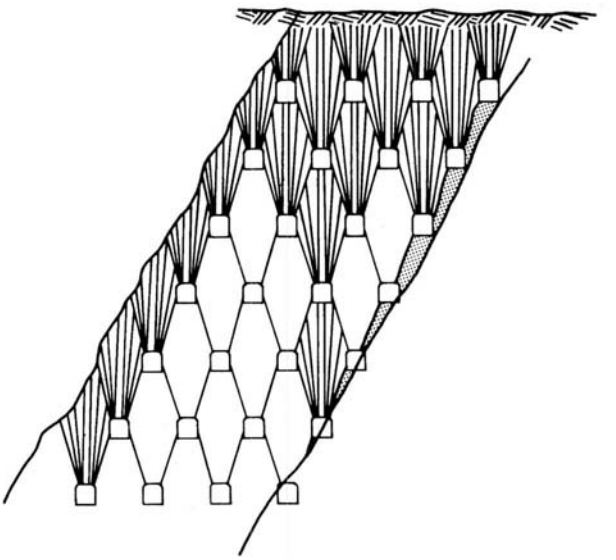


FIGURE 35.5 1.05 rad (60°) dip. Shaded areas are recovered.

broken ore is surrounded on three sides by dilution. Strong brows (the juncture of the blasted fan of drill holes with the back of the drift) and good fragmentation are necessary for the control.

The ore, then, should be strong enough to stand without excessive support because of the large amount of development work, to provide reasonably strong brows, to be suitable for drilling up holes 15.2 m (50 ft) or more in length, and for the drill holes to stay open for loading with explosive. The waste should be weak enough to cave.

Size, Shape, and Dip

The size, shape, and dip also have a strong bearing on the application of the method. Figures 35.4, 35.5, and 35.6 make this apparent. A vertical dip is the best; a medium dip of about 1.05 rad (60°) is quite satisfactory, although not as good; and a flat dip is most unsatisfactory, unless ore width is considerable.

In the case of the flat dip, the following two potential disadvantages should be noted: (1) inefficient development and drilling of hanging wall drifts because of low ore height above the drift; and (2) on the top sublevel, only about 50% of the ore is drawn because the ore takes up its angle of repose and cannot be reached by the loaders. In the case of the vertical dip there is always another level underneath to reclaim the ore left behind (except for the bottom level of the mine), so recovery is relatively good. But for the flat dip, only a small quantity of the lost ore is drawn on levels below, and portions of the orebody

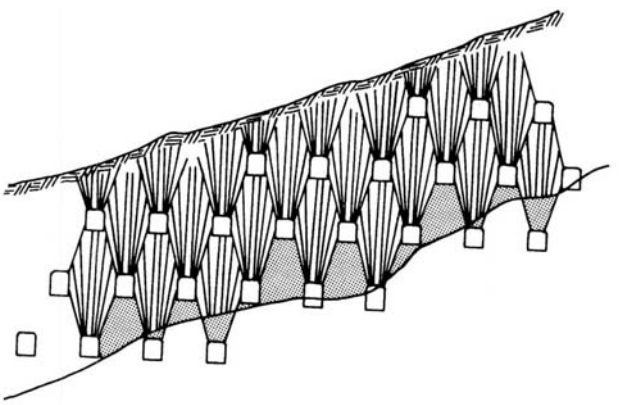


FIGURE 35.6 Flat dip. Shaded areas not recovered.

are not mined at all. This results in high development cost per ton and low recovery.

On the medium dip, geometry is still quite favorable and although a little ore will be lost in the footwall, recoveries are relatively good.

From this explanation, then, it is desirable to have fairly compact ore, weak walls, and a steep dip.

Advantages

The advantages of sublevel caving may be described as follows: it can be applied to both hard and moderately weak ground; it is flexible so it can be applied to irregular orebodies and wide or narrow orebodies down to about 3.7 m (12 ft); all operations take place in drift-size headings that can be well-supported and provide good conditions for accident prevention; it is suitable for a high degree of mechanization; activities can be specialized, simplifying training of personnel and reducing the number of miners required; no pillars are left for subsequent high-cost mining or lost ore; and the method has been successfully applied to pillar recovery.

Disadvantages

The major disadvantages are high dilution and the problems of controlling it. Control includes brow support, good drilling and blasting practice, and an organization for strict draw control. High development cost is also a factor.

Sublevel Versus Block Caving

If the ore is wide enough, steep enough, and will cave, block caving would be selected because the cost is normally lower than that for sublevel caving. However, sublevel caving could be a better choice if the ore was too hard to

cave readily, if the dilution from the hanging wall was excessive due to a flat dip, or if the orebody was too narrow or irregular in shape for the bulk mining associated with caving.

SUBLEVEL CAVING LAYOUT

Introduction

Considerable study has been done on the layout parameters, utilizing model test and theoretical considerations. The use of models is most beneficial in determining these parameters for a particular ore and for generally understanding the principles involved. Later they can be utilized in training both staff and operators. The importance of obtaining the best possible layout and then controlling the draw cannot be overemphasized. Good understanding of the process by engineers, supervisors, and operators is essential. A relatively small change in recovery can have a large influence on revenue.

Reasonable correlation has been found between model tests and actual results, although it is important to understand the limitation of model testing. In this case, the main difference is due to the consolidation that takes place in the mine when ore is blasted against broken muck, whereas the material in the model is loose. In addition, volume and weight are a cubic function, which cannot be properly represented in a model.

Transverse or Longitudinal Layouts

In wide orebodies, transverse layouts are used, as illustrated in Figs. 35.1, 35.2b, and 35.3b. Usually drifts are oriented at an angle to the strike to facilitate turnouts from the haulage drifts into the production drifts.

When orebody widths are below 12.2 or 15.2 m (40 or 50 ft), this layout becomes impractical, and the production drifts are driven down the strike. This may be a single drift or more depending on the width.

Recovery is normally better with a transverse layout because the production drifts cover all the ore from hanging to footwall, whereas the longitudinal drift tends to leave ore behind in an irregularly shaped orebody (see Fig. 35.7).

For transverse layouts, the haulage drift is situated in the waste, preferably in the footwall and about 9.1 m (30 ft) from the ore contact. This insures that blasting will not take place close to the intersection because resulting ground support problems usually cause losses in ore recovery. Previous diamond drilling for ore delineation is necessary to properly locate these drifts. Ore losses result if too close to the ore, and considerable waste development is required if too far away.

In longitudinal layouts, the production drifts have to be turned into the waste where orepasses will be situated at convenient intervals.

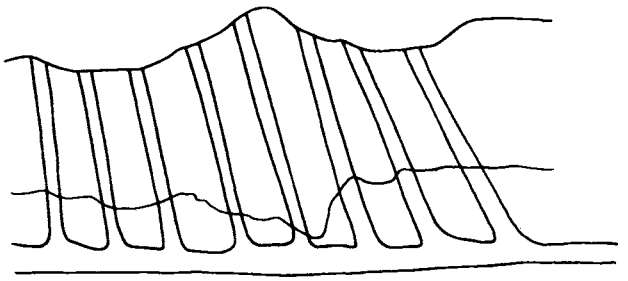


FIGURE 35.7a Transverse layout. All ore recovered.

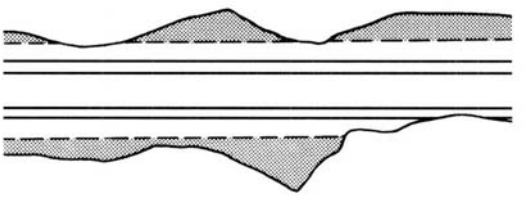


FIGURE 35.7b Longitudinal layout. Hatched areas represent ore not recovered.

Sublevel Height

In theory, the vertical distance between sublevels should be as much as possible. In practice, several factors govern this dimension:

Dip of Orebody. If dip is vertical, there is no constraint. As the dip flattens, the height has to be reduced to avoid drawing the hanging wall waste. See Figs. 35.5, 35.6, and 35.7.

The Ability to Drill, Load, and Break the Rock to a Satisfactory Fragmentation and the Cost of Doing So. Hole deviation and cost increase rapidly with length. If ore is weak, it may be difficult to keep the blastholes open for loading. If ore is not well-fragmented, ore recovery and dilution will be seriously affected. Sublevel heights of 9.1 to 10.7 m (30 to 35 ft) are quite common with maximum hole length of 15.2 to 18.3 m (50 to 60 ft). Heights in excess of this have been tried but have usually been reduced.

Production Drift Spacing and Pillar Width

There is a distinct relationship between the sublevel height and the optimum pillar width and consequent spacing between production drifts. The eccentricity of the cave figure can be estimated from model tests, and a theoretical spacing determined. The width selected for the production drift is also a factor. An ideal arrangement is equal drift width and pillar width, but this is usually impractical. The ground conditions must be considered and also the drill-hole fan layout. With sublevel heights of 9.1 m (30 ft), a 6.1-m (20-ft) wide pillar is satisfactory. This may be increased to 7.6 m (25 ft) if a 6.1-m (20-ft) pillar is not strong enough to support the loading.

Size and Shape of Production Drift

The size and shape of the drift have an important bearing on draw. The drift should be as wide as possible, while still giving good support to the back and brows. For optimum draw, the backs should be flat; in this way the ore will draw evenly across the whole width. If the back is arched, the draw will concentrate in the center and will not move on the sides. Waste will then be pulled down the center before all the ore is recovered. If the backs have to be arched for ground support, the production drifts should be closer, if possible.

When loading, ore remnants are left at the back of the muck pile, out of range of the loader. The quantity increases as the height of the drift increases, so the drift height should be as low as possible, consistent with the size of the drilling equipment and ventilation ducts. About 3.2 m (10 ft 6 in.) is not uncommon.

Fan Drilling

Early layouts included flat-side holes of about 0.78 rad (45°). Their purpose was to reduce the length of the up holes from the level below. However, it was not possible to draw any of the ore broken from these holes from the level from which they were drilled, and so there was no room for expansion when the flat holes in the next fan were blasted. Furthermore, these holes were collared well down the ribs and were often covered with muck from the last blast. When the ore was finally drawn on the level below, it would have been subjected to more dilution. Practice now, as far as possible, is to drill the fan to conform to the ellipsoid of motion.

Side holes should be steeper than 1.22 rad (70°) and they should be collared at the corners of the drift to obtain maximum throat width. Toes of holes are normally spaced at 1.5 to 1.8 m (5 to 6 ft). There are usually eight holes in a fan and approximately 100.6 m (330 ft) in a ring, for a 9.1 m (30-ft) high sublevel. The burden between fans is designed to give good fragmentation and varies between 1.2 and 2.7 m (4 and 6 ft).

In theory, the inclination of the fans is determined by the relative size of the broken rock of the ore and the waste. Because of the tendency of the finer material to infiltrate the coarse, the fan should be inclined backward if the ore is finer than waste and forward if the waste is finer than the ore. Fanning backward is usually not practical because it weakens the brows. There is little difference between vertical and 1.4 rad (80°) forward, and it is usual to drill no steeper than 1.4 rad (80°) because less drilling sludge runs over the drills and the operator at this angle. So drilling usually is inclined forward at 1.4 to 1.2 rad (80 to 70°).

Brow Support

This is a critical factor in the performance of sublevel caving. All the activities of development drilling and blasting, etc., are necessary preparations for the actual drawing of ore from beneath the brows. If the brows are strong and intact, recovery will be at a maximum and dilution will be minimized. Fig. 35.8 illustrates the appearances of a strong and of a collapsed brow.

The effects of brow collapse are as follows:

Ore floods the drift and covers the next row or rows of holes to be blasted. When this happens, the holes are dug out, if possible (a somewhat hazardous operation), or they are lost.

If the brow is uneven (i.e., not flat and horizontal), the ore funnels down through the high spot. This reduces the width of ore flow with consequent ore losses.

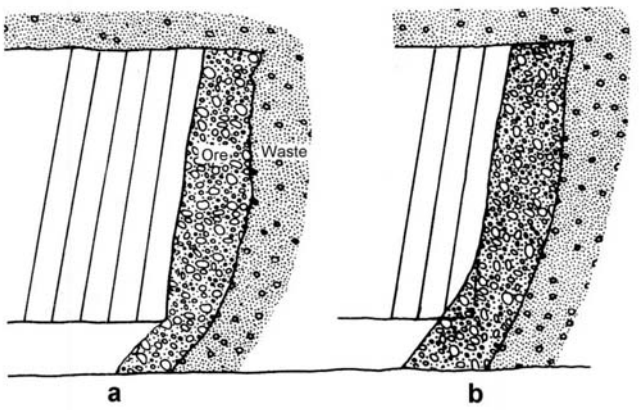


FIGURE 35.8 Brows (a) Strong brow, (b) collapsed brow

When the brow overbreaks, the ore floods further back into the drift than was intended. This prevents the loader from digging in its proper position relative to the broken ore. As a result, a thin draw of ore occurs with waste being pulled down the brow without loading the full depth of ore blasted. In this case, single ring blasts give best results.

When the brow caves back a fair distance or there are high spots from previous sloughing during the development drifting, and it is obvious that the brow will end up in a high spot if only one or two rings are blasted (thus causing ore loss), it may be necessary to blast more than two rings at a time to pass the high area. Recovery will be poor from such a blast, but records are kept to plan for overdraw on lower levels.

TECHNICAL REQUIREMENTS

Mining Sequence

A well-ordered mining sequence is essential to avoid undermining areas still in production, or drilling blastholes into holes loaded with explosive.

Development and Support

Sublevel caving involves a lot of development, with 15% or more of the ore being removed in development headings. In addition, one or more ramp systems are required to provide access to the many sublevels for the trackless equipment.

Ramps should be driven at a maximum of 20% (depending on the equipment in use), but preferably about 18%; on spiral ramps 15% is preferred. Ramps should be flattened off at turnoffs to sublevels. Minimum practical inside radius for a spiral ramp is about 12.2 m (40 ft).

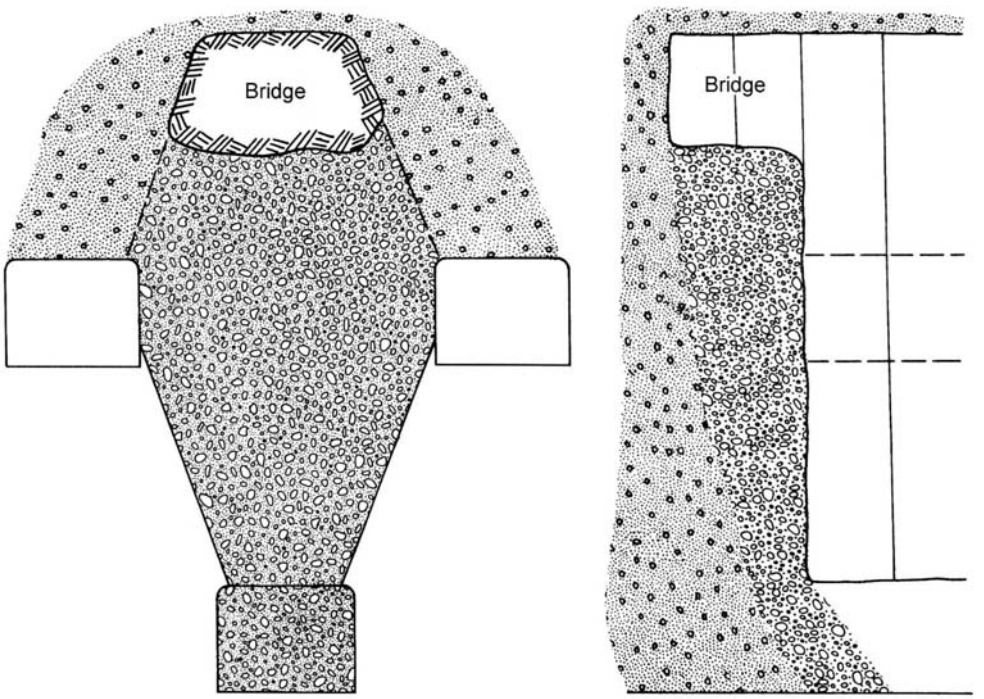


FIGURE 35.9 Formation of bridge

Sublevel headings must be adequately supported for their relatively short life. The intersections of haulage drifts and production drifts require particular care.

The most critical areas for support are the brows, and a satisfactory method must be established. If the ground is strong, extra support may not be necessary. In medium ground, rockbolt, shotcrete, epoxy or grouted rebar bolts, timber, or steel may be required, but the brows must be held up.

Drilling and Blasting

Drillholes must be carefully aligned and drilled accurately. Surveyors should mark up the ribs so that the drill rig can be properly aligned to provide truly parallel fans of holes. A protractor with a bubble should be provided to check the fan inclination and the angle of each drill hole.

The loading and blasting of the holes must be performed with great care. Reliable personnel should do this work, and any incentive paid them should not be based on the footage blasted. This is an invitation for poor work. It is essential that holes should be well-cleaned and loaded right to the top.

If drill holes are not drilled accurately or are not properly loaded and blasted, a solid bridge or arch is left, and only the lower part of the fan is drawn (Fig. 35.9). Usually several more rings are blasted before this condition becomes evident, and

it may be necessary to reslot. Considerable loss of recovery will result if these arches are allowed to form. In addition, coarse fragmentation will allow the waste to dilute the ore.

A characteristic of sublevel caving is that all blasting is against broken muck. The mucking of the previous blast results in the broken muck being loosened up, and it is estimated that this allows an expansion of about 15%. The powder factor needs to be approximately double that required for an open face or from 0.3 to 0.4 kg (0.6 to 0.8 lb) per ton. Obtaining this factor is not sufficient if the explosive is not properly distributed. With a fan pattern, there is always a tendency to overload the collars. It is also necessary to reduce the load at collars to protect the brow.

Loading or Mucking

It is important to use a loading unit that will dig into the muck pile rather than skim it. The depth to which the unit will dig determines the optimum depth of the blast. Diesel-powered load-haul-dump units are the most suitable because their weight and power allow them to dig well into the muck pile. The production drifts are as wide as ground conditions will allow, and the largest LHD unit giving satisfactory clearances would be selected; 3.8-m^3 (5-cu yd) units are commonly used.

When mucking, it is important to alternate from one side of the heading to the other to keep an even draw.

Good roadbeds for the trackless equipment are essential to maximize production and keep maintenance to a minimum. Because of the short life of any sublevel, concreting may not be justified. However, screened crushed rock from the surface can be compacted to make a good roadbed.

It is important to blast hang-ups in the draw as soon as they occur to insure continuous and even flow.

Ventilation

The active areas are dead-end headings, and through ventilation is not possible. Auxiliary ventilation is required. This must be designed in relation to the legal ventilation requirements for the diesel units. On a given sublevel a ventilation door is required at the access to the mining area (see Fig. 35.1). A fan is installed in the door, and ducting is installed to provide air to the end of the active production drifts. Neolon duct is commonly used in the production drifts. This duct can be tied off when not in use.

If mining is taking place close to the surface, and caving has broken through, air will be lost by leakage. To minimize this loss it may be necessary to install a fan in the exhaust raise as well as the fan in the vent door to reduce the pressure differential. This also helps in controlling short circuiting through the orepass.

Drainage

Good drainage in the production drifts and haulage drifts is essential to provide good roadbeds for the trackless equipment. Headings should be driven on a grade to facilitate drainage (i.e., 3%). As equipment tends to be large in relation to the headings, it is usually very difficult to maintain ditches on the sides, so instead the center can be graded to form a ditch.

Considerable water is generated from drilling in addition to any natural water that may be present. Holes can be drilled at convenient locations between sublevels to drain the water down to a lower level for handling to the pumps.

Drilling, Blasting, and Mucking

The importance of good control of drilling, blasting, and mucking has been previously described, and it is emphasized again.

DRAW CONTROL

Cutoff Point

The cutoff point is the control required to maximize the economics of the draw and is associated more particularly with the point at which mucking ceases, and the next fan of drill holes should be blasted. This is a most critical control, as early cutoff results in poor recovery and late cutoff results in excessive dilution.

The first parameter that must be established is the theoretical grade at which the muck should be cut off. This is an economic calculation based on the particular costs of the operation, metal prices, etc., and is usually the grade at which the ore value equals the remaining costs to be incurred.

After this has been established, it is not easy to decide when this point has been reached in the mucking cycle. The two general approaches to this subject are the visual method and the sample and assay system. Both have advantages and disadvantages.

The particular circumstances of any given operation must guide the selection. Features of the two systems are described in the next section.

Sampling Systems

Visual Method. In this case it must be possible to visually distinguish between the ore and waste. As it is difficult to accurately estimate the grade, it is more common to provide the operators with information based on the percentage of ore to waste that should be present at the desired cutoff. The ore grade in situ is determined by sampling the sludge produced from fan drilling, and any assays available from ore delineation, etc. The tonnage blasted is computed from drill-hole layouts. The ore-to-waste ratio is calculated as follows: if in-place grade is 2%, the desired cutoff is 0.5%, and the dilution

carries no values, then mucking should cease when the muck pile appears to consist of 25% ore and 75% waste. The theoretical number of LHD loads to achieve this cutoff point is provided to the production supervisor as well as the calculated desired percentage of ore and waste. If the cutoff appears to occur before the theoretical number of loads has been taken, the operator will take a selected number more (perhaps to a waste pass, if available) and stop the draw if conditions do not improve. If the operator completes the theoretical number and the ore-to-waste ratio is still good, the draw will continue until reaching the cutoff point.

Sample and Assay System. This is usually applied when it is difficult to distinguish between ore and waste, although there are some arguments that it provides better control in either case. A great many samples are taken and they are prepared and assayed underground. By performing this work in the mine, delays in the transportation of samples to the surface are avoided, and the interference to production is kept to a minimum.

As a guide in the choice between these methods, the following points need consideration: (1) it is difficult to collect a representative sample from a muck pile because statistically a large sample is needed; and (2) the muck pile must be left idle until the results are obtained, and then the next muck pile may be significantly different from the ore sampled. If a visual method is possible, estimates can be taken rapidly at frequent intervals and without production delays; however, sampling and assaying, although expensive, provide figures that may be more accurate than estimates. This is such an important subject that each property should closely examine conditions and possibly experiment with both.

RECOVERY, EXTRACTION, AND DILUTION

Recovery

Recovery is the percentage of metal units (copper, for example) extracted to units blasted. It is common to speak in terms of 85 to 90% recovery of units blasted, or even more. This is not the recovery of the in situ ore reserves, which will be less by an amount dependent on ore widths, regularity of ore boundaries, and dip. To estimate the recovery of ore reserves, it is necessary to lay out the pattern of production drifts and measure the volume covered by them; then apply anticipated recovery of units blasted, bearing in mind that ore cannot be completely recovered on any given level; remnants are partially recovered on the level below. Consequently, the remnants on the bottom level are lost.

Extraction

Extraction refers to the percentage of tons drawn to tons blasted.

Dilution

Dilution refers to the barren or low-grade material drawn with the ore, which causes the grade of ore to decline until the cutoff point is reached, and equals

$$\frac{\text{waste}}{\text{ore + waste}}$$

This can be expressed as

$$\frac{\text{grade in place} - \text{grade extracted}}{\text{grade in place} - \text{grade of dilution}}$$

It is apparent from this formula that dilution increases in proportion to the spread between the ore grade and the desired cutoff.

There is a close relationship between recovery extraction and dilution, and an optimum balance must be found. Recovery can be increased at the expense of higher dilution and extraction.

As far as control of the operation is concerned, dilution is not usually used directly because it is often complicated by carrying low values itself. The parameters to watch are the relative grades of blasted and drawn ore, the recovery of metal units, and the extraction rate.

EQUIPMENT AND MAINTENANCE

Production Equipment

Loading and Mucking. Diesel-powered LHD equipment is commonly used and is recommended.

Drilling Equipment. Manufacturers have designed two- and three-boom jumbos especially for sublevel cave drill patterns. Traction can be either air or diesel-powered. Air-powered units are less expensive but have to be towed from one sublevel to another, which often results in damage, and are inconvenient to move to the maintenance shop for servicing and repairs. Diesel-powered traction is recommended.

Blasting Equipment. Diesel-powered vehicles can be mounted with pneumatic loaders for loading ammonium nitrate–fuel oil (ANFO) or slurry.

Service Vehicles

Rugged diesel-powered service vehicles are required for transporting supplies and personnel. It is important that they are properly designed to carry the loads required with due consideration to the rough underground conditions.

Electric traction is also available and should be considered. Advances have been made in the use of hydraulic drills, and these should be seriously considered also.

Maintenance

Maintenance is one of the most important functions in the mine, and a sound system of preventive maintenance must be established and implemented at

the start of the operations. Well-equipped shops of adequate size are essential. Close cooperation is necessary between production and maintenance personnel, so that maximum production over a sustained period can be achieved.

PERFORMANCES

Unit Operations

Every operation is different, but a guideline figure for production drilling with diesel-powered jumbos mounting two booms, and with good drilling, is that 152.4 m (500 ft) per shift or more can be averaged with experienced operators, the drillability of the rock being a big factor. For load-haul-dump, a 3.8-m^3 (5-cu yd) LHD can average 453.5 t (500 st) per shift or more over a 213.4-m (700-ft) one-way haul and under good working conditions.

Overall

Each operation is different, but 36.3 t (40 st) per worker-shift for all underground personnel has been attained.

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CHAPTER 36

The Mechanics and Design of Sublevel Caving Systems

Rudolf Kvapil

INTRODUCTION

Sublevel mining is a mass mining method based upon the utilization of gravity flow of the blasted ore and the caved overlying waste rock mass. As with any other mining method, sublevel caving has advantages and disadvantages that must be carefully considered and evaluated.

The major advantages of sublevel caving are discussed as follows:

1. Because all of the mining activities are executed in or from relatively small openings, sublevel caving is one of the safest mining methods. Drifts, which are the primary working places, are distributed in a uniform pattern on all levels. Normally, the maximum dimensions of the sublevel drifts are about 5 m (16 ft) wide and 3.7 m (12 ft) high. The transportation drifts can have the same section, or the height may be increased to about 4.5 m (15 ft) when trucks are loaded in the transport drifts. The stability and safety of such drifts in competent rock can be easily controlled by smooth blasting or by a combination of smooth blasting with shotcreting. In less competent rock masses, stability can be achieved by combined reinforcing, for example, by a combination of smooth blasting, shotcreting, and rockbolting.
2. The major mining activities can be broken down into three groups–drifting and reinforcing; ore fragmentation (i.e., production drilling and blasting); and ore drawing, loading, and transportation–and all are relatively simple. Because of the repetitive nature of the mining system, one can standardize almost completely all mining activities. This means that a high degree of work efficiency can be achieved.
3. Because the components of mining production in sublevel caving can be standardized, a high degree of mechanization is possible. In modern sublevel caving the sections of drifts and tunnels are sufficiently large to allow the introduction of large trackless mining equipment. The advantages of a

trackless system then can be broadly utilized not only for direct mining but also for all services, including the transportation of mining personnel to the working place.

4. The flexibility of mining is very good. Standardization and specialization of mining activities and equipment on separate levels (lower level or levels in development, upper level or levels in production mining) together with the trackless system yield a high degree of flexibility. This allows a rapid start-up of mining and good flexibility in making production rate changes.
5. The method lends itself to good work concentration, organization, and working conditions. Normally, on the lower levels, various phases of development are under way. Upper levels are in various stages of extraction. Therefore, the work can be easily organized into a system that excludes interference between mining activities. Safety of mining (in small dimension openings), good work organization, high mechanization using large modern mining equipment, etc., comprise very good working conditions. Naturally, such a system enables a high work concentration and rationalization of separate specialized mining activities, and therefore mining by sublevel caving can be effective and relatively inexpensive.

The major disadvantages of sublevel caving, on the other hand, are:

1. There is a relatively high dilution of the ore by caved waste.
2. Various types of ore loss can occur. When the extraction limit (that point yielding the maximum acceptable amount of dilution) is reached, the remaining highly diluted ore represents an ore loss. Some ore is lost in passive zones located on the level of extraction between the active zones of the gravity flow. Part of the ore from these passive zones can be recovered together with ore extraction on the lower sublevel, but some undiluted and often not fragmented ore located in passive zones above the plane of the footwall is lost. In general, these losses are larger as the inclination of the orebody and the footwall is reduced.
3. A relatively large amount of development is required. This includes transport drifts, usually located in the footwall waste rock on each sublevel, and sublevel drifts, which connect the active mining areas to the transport drifts and as a result are partially in ore and partially in the waste rock of the footwall. The waste rock length increases as the inclination of the orebody and footwall decreases. It also includes orepasses, used for transport of the ore or waste from the separate sublevels downward to the main haulage level, and normally driven in waste; and inclined drifts or tunnels, which provide a connection for the trackless equipment between the main haulage level and the separate sublevels and are driven in waste. Finally, there is the destruction of the surface through subsidence.
4. To maximize the ore recovery, minimize the dilution, and achieve a high efficiency of mining by sublevel caving, good data regarding the gravity flow parameters for the blasted ore and the caved waste are of utmost importance. The exact type and amount of data required depend upon the purpose and needs of the study. For the first feasibility study, it may

be sufficient to utilize the data from other sublevel caving operations with similar conditions and circumstances. For any higher level of mine planning it is clear that more exact data, including analytical and experimental analyses up to full-scale in situ testing, are necessary.

5. Basic gravity flow principles and design guidelines for the application of the sublevel caving mining method are presented in the following sections. Although somewhat simplified, they should provide a basis for mine planning and operation. The gravity flow principles described can be effectively applied to other mining situations, with some modification. Also, steep-dipping coal seams can be effectively mined by modified sublevel caving.

PRINCIPLES OF GRAVITY FLOW

The geometry, dimensions, and even operational constraints of sublevel caving cannot be selected at random, but must be planned with respect to the laws of gravity flow of coarse materials, that is, the gravity flow of blasted ore and caved waste rock.

Because of the heterogeneity of coarse materials and a great number of different factors and conditions, the gravity flow of these materials is a very complex process.

It must be emphasized that although flow as used in "gravity flow" means an uninterrupted movement of coarse material, it is a completely different process from the flow of liquids. Hence, knowledge of the flow of liquids cannot be applied to gravity flow of coarse materials.

The laws and principles of gravity flow are independent of the fragment size; they are the same for coarse material as for small granular material (i.e., small gravel or sand).

Gravity Flow Demonstrated

Gravity flow is usually demonstrated on a very simple vertical glass model with horizontally layered white and black sand filling. Such a model represents, for example, a vertical section of a silo taken through the axis of the extraction opening. The extraction opening in the model bottom has minimum dimensions, and yet is sufficiently large for a fluent and uninterrupted flow of material.

The deflection of the originally horizontal thin black layers indicates the active zone, that is, the zone with gravity motion of the material, which attains a form similar to an ellipse with a vertical axis. In reality, this ellipse is a vertical and axial section of an axisymmetric body, namely that of an elongated ellipsoid of revolution whose geometry can be explicitly defined by eccentricity.

Figure 36.1 shows successive phases of loosening, from top to bottom, of the material as a consequence of the extraction of a certain material volume from the model due to gravity flow.

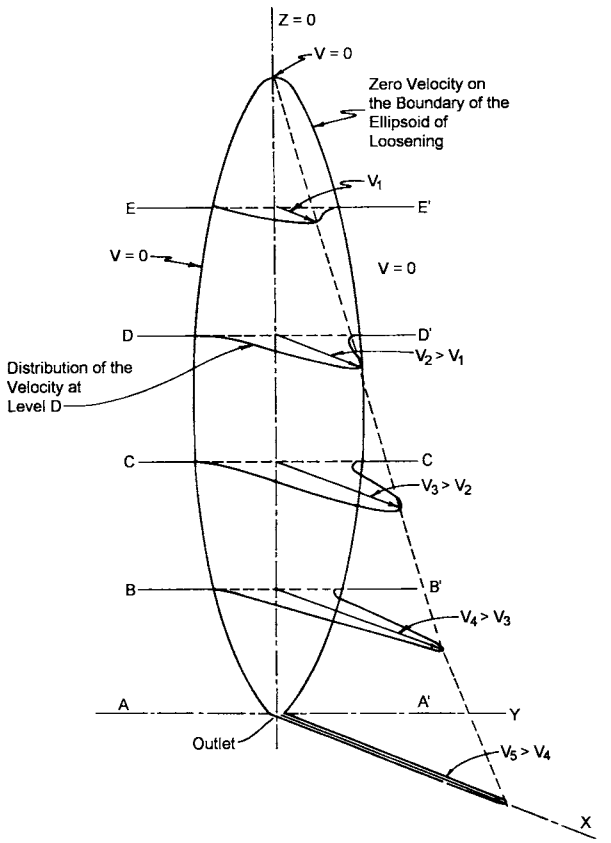


FIGURE 36.1 Velocity distribution in an ellipsoid of loosening

After the extraction of a certain volume, the remaining material replaces this loss by loosening. Although it is evident that a certain relation exists between the extracted material and the loosening, only the visible zone of the ellipsoid of loosening can be defined by this model. The size of the ellipsoid increases as more material is extracted. Depending upon material properties and the conditions acting, the volume of the ellipsoid of loosening can be 14 to 16 times larger than the volume of extracted material.

From observing the pattern in different phases of extraction, it is not possible to define either the location or the shape and dimensions of the zone from which the material, now lying below the discharge opening, has been extracted. Extraction of a certain volume of material results in the development of a certain ellipsoid of loosening. Under these conditions, there is no motion at the boundary of the ellipsoid of loosening and the greatest particle velocity is in the center of the discharge opening. The result of an analysis to determine the velocity distribution on levels E-E ' down to A-A ' is shown in Fig. 36.1. This provides an axonometric representation of the envelope surface of velocity vectors from $V = 0$ to $V_1 < V_2 \ldots < V_5$. For better visualization, the velocity vectors are not constructed in a true direction, but perpendicular to the axial section of the ellipsoid, with zero velocity on the boundary.

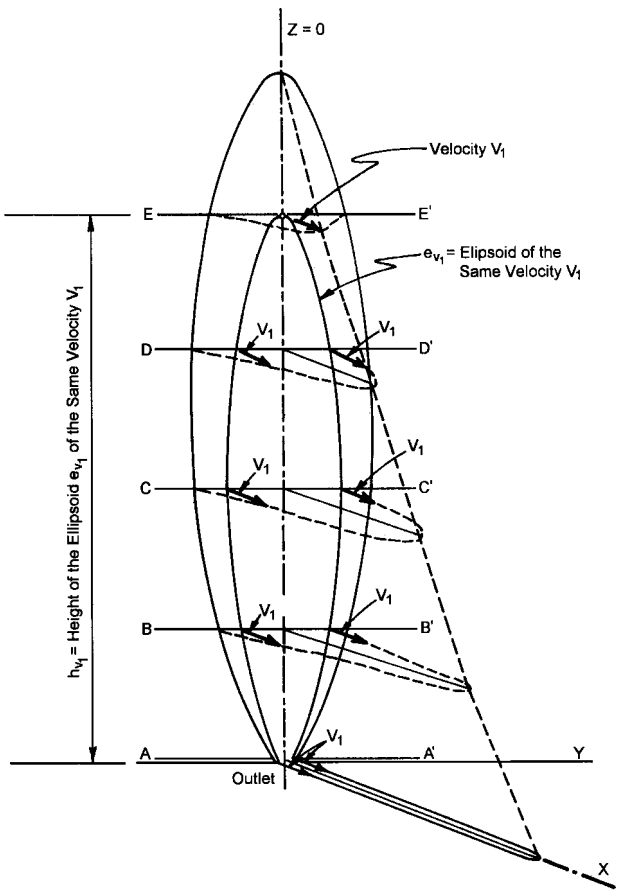


FIGURE 36.2 Ellipsoid of the same velocity

From Fig. 36.1, one can easily derive the zones of the same particle velocity V_1 defined on the boundary, shown in Fig. 36.2. The line that connects the particles of the same velocity forms an elliptic figure in a vertical section, and in space, an ellipsoid of the same velocity.

Evidently, the shape of the gravity flow zones is controlled by a specific distribution of the velocity of motion, resulting in ellipsoids of the same velocity.

Therefore, not only does the zone of loosening have the shape of an ellipsoid (boundary of the ellipsoid of loosening has the velocity = 0), but so also does the zone from which the discharged material was extracted. This zone, similar to the elongated ellipsoid of revolution, is called the ellipsoid of extraction.

Mathematical Description. The existence of the extraction ellipsoid can be demonstrated by using several different methods. One possibility is a three-dimensional model in which marks are located in a certain pattern in the granular material. Knowing the original location of the marks (defined by space coordinates) before the extraction, the marks extracted together with the material will define the extraction ellipsoid.

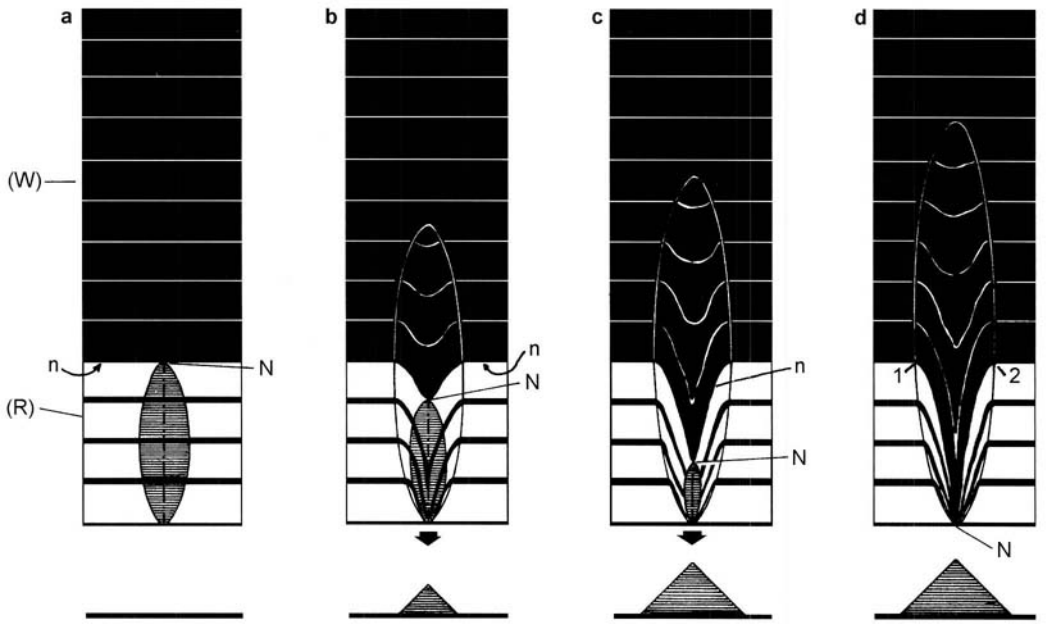


FIGURE 36.3 Mechanism of gravity flow

The gravity flow mechanism, showing the flow of material from the extraction ellipsoid and development of the ellipsoid of loosening, is shown diagrammatically in Fig. 36.3.

From Fig. 36.3d it is evident that, with complete extraction of the extraction ellipsoid, the original horizontal plane n passing through the apex point N is deflected downward, forming the outflow funnel 1, N, 2. The points 1 and 2 intersect the ellipsoid of loosening at a height equal to the height of the extraction ellipsoid. It means that the outflow funnel diameter 1, 2 is equal to the horizontal section of the ellipsoid of loosening measured at the height of the apex point (N) of the extraction ellipsoid. The volume of the outflow funnel is the same as the volume of the extraction ellipsoid.

As shown in Fig. 36.4, for a certain extraction height h_n, the mechanism of gravity flow is defined by explicit relationships between the discharged material, the ellipsoid of extraction, the outflow funnel, and the ellipsoid of loosening.

Using the nomenclature defined in Fig. 36.4, the characteristic relations for gravity flow can, by introducing certain simplifications, be expressed by the following formulas:

$$V_{EE} \approx V_c, \qquad \textbf{(EQ 36.1)}$$

that is, the volume of the extraction ellipsoid is approximately the same as the volume of discharged material.

$$V_{EE} \approx V_c \approx V_F, \qquad \textbf{(EQ 36.2)}$$

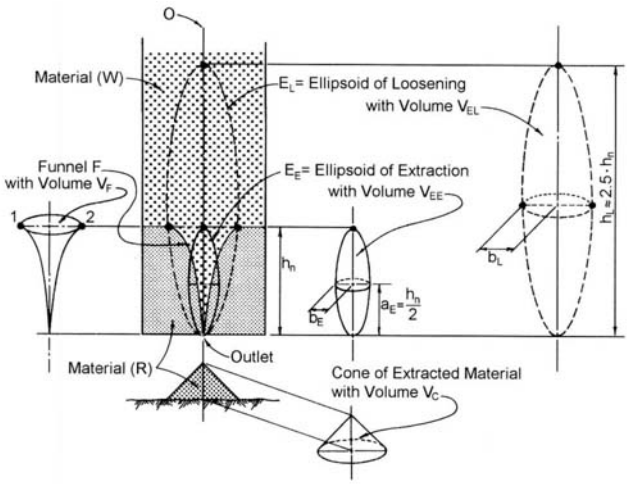


FIGURE 36.4 Relationships in the gravity flow mechanism

that is, the volumes of the extraction ellipsoid, discharged material, and outflow funnel are approximately the same.

$$V_{EL} \approx 15\ V_{EE} \approx 15\ V_c \approx 15\ V_F, \qquad \textbf{(EQ 36.3)}$$

that is, the volume of the ellipsoid of loosening is about 15 times larger than the volume of the extraction ellipsoid, the volume of discharged material, or the volume of the funnel.

Assuming approximately the same eccentricity of the ellipsoid of loosening and the extraction ellipsoid, and knowing that $V_{EL} \approx 15\ V_{EE}$, then

$$h_L \approx 2.5 h_n\,. \qquad \textbf{(EQ 36.4)}$$

Hence, the height h_L of the ellipsoid of loosening is about two and a half times bigger than the height h_n of the extraction ellipsoid.

These relations, which are generally valid, form the basis for understanding, explaining, and justifying the different phenomena involved in gravity flow.

Consider, for example, the situation in which the lower part of the material with height h_n is the ore, denoted by R in Fig. 36.4, and the upper part is the waste rock W.

As is evident from Fig. 36.4, the maximum volume of pure ore that can be discharged without any waste rock (i.e., without any dilution) is only that from the extraction ellipsoid with height h_n. Any additional material discharge will result inevitably in a rapid increase in dilution because the outflow funnel, equal to the discharged extraction ellipsoid of height h_n, is now filled by waste rock.

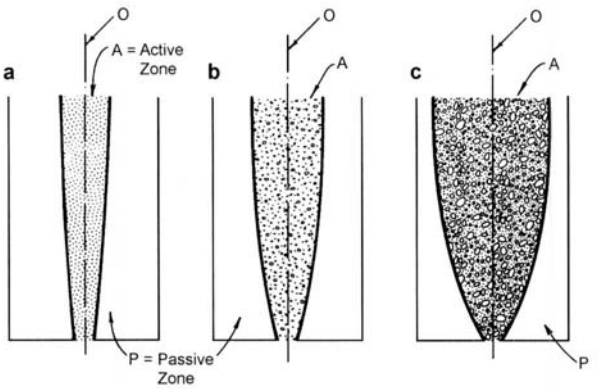


FIGURE 36.5 Influence of particle or fragment size on the shape of active (gravity flow) zone: (a) fine material; (b) granular; and (c) coarse material

For the same extraction height h_n the extraction ellipsoid can have a different volume in different materials because the eccentricity ε of the ellipsoid of extraction (and loosening) depends upon the material. The formula for eccentricity of the ellipsoid is

$$\varepsilon = \frac{1}{a}\sqrt{(a^2 - b^2)},$$

where a is the semimajor axis and b the semiminor axis of the ellipsoid (see also Fig. 36.4).

Naturally, the greater the eccentricity, the slimmer the ellipsoid of extraction (or loosening) and the smaller the volume.

It is very well known that in fine materials (cement, fine sand, etc.), the active zone is very slim, and in coarse material, the active zone is broad. This is shown in Fig. 36.5a for very fine material, in Fig. 36.5b for granular material, and in Fig. 36.5c for coarse material.

The eccentricity of the ellipsoid of extraction and loosening also increases with the height of the ellipsoids. This effect, which is relatively small in sublevel caving, has a much greater importance in block caving because of the very large block heights.

The eccentricity depends not only upon the size of the particles (and from the gravity flow height) but also on many other factors, such as shape or form of the particles (spherical, irregular, etc.), surface roughness of the particles (smooth, rough), angle of friction (small, large), density (high, low), percentage of components with a lubricating effect (high, low), extraction or discharge rate (high speed, low speed), and material properties of the particles (strength, moisture, etc.).

All of these factors result in a certain behavior, which can be expressed in terms of the mobility of granular or coarse materials.

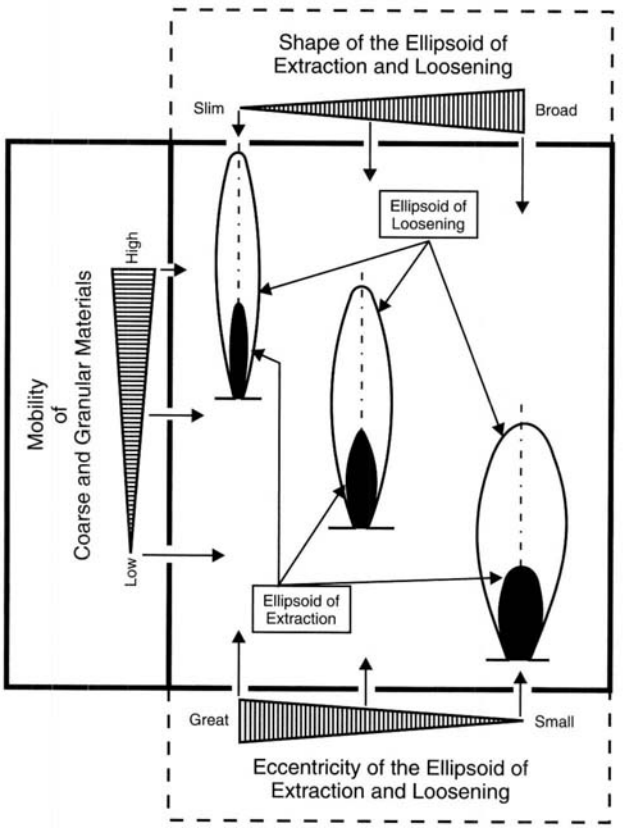


FIGURE 36.6 Shape and eccentricity of the ellipsoid of extraction and loosening as a function of material mobility

A greater material mobility results in easier gravity flow and a higher eccentricity of the ellipsoids. This means that in materials with high mobility, the ellipsoids are slim, and in materials with low mobility, the ellipsoids are broad and contain large volumes.

Although an absolute scale is not yet available, the mobility of materials [classified between the limits of high (= very good) to low (= poor mobility)] can be used as a qualitative comparison of the flow behavior. From this indication of the flow behavior, one can deduce the approximative geometry of the ellipsoids. Figure 36.6 shows the shape and eccentricity of the ellipsoid of extraction and loosening as a function of material mobility.

Specific Application to Sublevel Caving

For a vertical sublevel front, the sublevel drift forms a vertical opening, located in the plane of the sublevel front.

Therefore, on a vertical section A-A ' as shown in Fig. 36.7, the geometry of sublevel caving is similar to a bin with discharge openings located not on the bottom, but in the vertical wall of the bin.

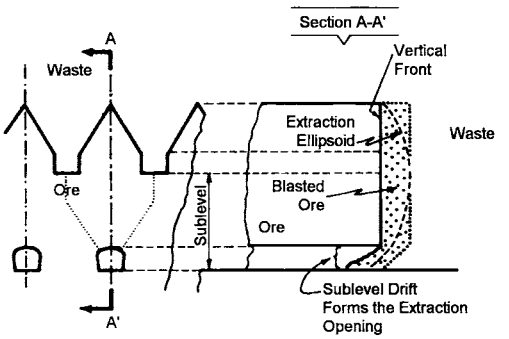


FIGURE 36.7 Simplified geometry of sublevel caving

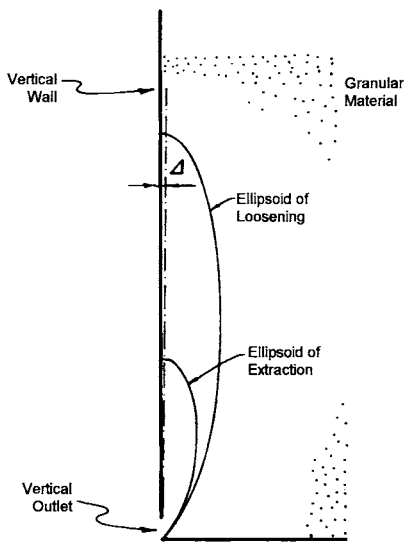


FIGURE 36.8 Scheme of ellipsoid of extraction and loosening by extraction through the opening located in the vertical wall. Vertical wall cuts off about half of both ellipsoids.

The gravity flow zone is cut off by the vertical bin wall, but is otherwise unchanged. It means that the vertical wall cuts off the ellipsoid of extraction and loosening as shown in Fig. 36.8.

The axis of gravity flow on this section deviates from vertical by a certain angle *A*. This angle increases as the friction along the vertical wall increases.

Neglecting this deviation, one can assume that the vertical wall with vertical discharge opening cuts off half of the ellipsoid of extraction and loosening.

Naturally, when this half of the ellipsoid is inscribed in a prism, then its volume is 50% of this prism. Therefore, in sublevel caving, if half of the extraction ellipsoid is inscribed in the body of blasted ore, as shown in Fig. 36.7, then a maximum of 50% of the blasted ore volume can be extracted without dilution.

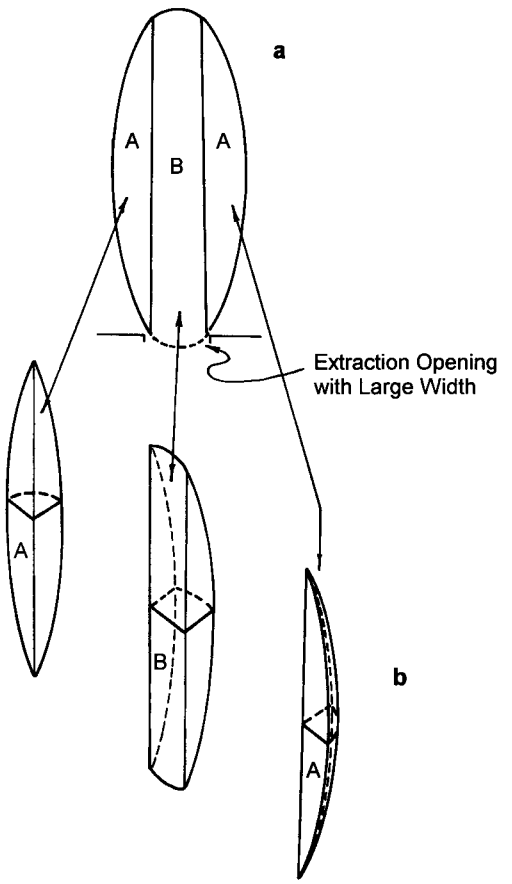


FIGURE 36.9 Simplified shape of the extraction zone by sublevel caving, caused by a large extraction width

For the foregoing description of the gravity flow to be representative, the material must be discharged through a minimum size opening sufficiently large for a fluent outflow.

In sublevel caving, the width of vertical discharge is formed by the width of the sublevel drift, which is greater than the width of a minimum size opening.

A greater opening width produces certain changes in the gravity flow, which can be utilized for better ore extraction. The central part of the gravity flow zone above the discharge opening moves downward more or less as a column. This kind of gravity movement is called mass flow.

In sublevel caving, the opening at the level of the sublevel drift roof has the shape of a slot. Its length is equal to the roof width of the horizontal drift. In such a situation, the extraction zone (which has the same function as the extraction ellipsoid in a minimum size opening) no longer has the shape of an elongated ellipsoid of revolution, but forms an ellipsoid approximately similar to that shown in Fig. 36.9a.

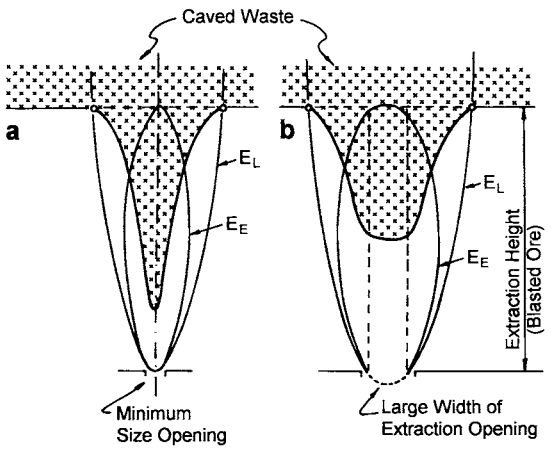


FIGURE 36.10 Influence of the extraction width on the gravity flow pattern: (a) small width and (b) large width

With certain simplifications, one can assume that this extraction ellipsoid is formed in three parts, as shown in Fig. 36.9b. Gravity flow consists of a partial mass flow (*B*) which occurs only on contact with the vertical plane of the sublevel front, whereas the remaining zones of the ellipsoid (*A*) undergo normal gravity flow. It means that mass flow occurs only in the zone marked *B* in Figs. 36.9a and 36.9b.

It is clear that as the width of extraction opening increases, the width, and therefore the volume of the extraction zone, increases. At the same time, a wide extraction opening will also result in an increase in the volume of undiluted extracted ore, because the outflow funnel will reach the extraction opening after the extraction of more undiluted ore than with a minimum size opening. This effect is shown in vertical section along the plane of the vertical front of the sublevel in Fig. 36.10. Here Fig. 36.10a represents the situation for a minimum size opening, and Fig. 36.10b shows the situation with a large width extraction opening.

It must be emphasized that the effective width of the extraction opening depends not only upon the width of the sublevel drift, but also upon the shape of the sublevel drift roof.

When the roof is arched, as shown in Fig. 36.11a, the slope of blasted ore will form a cone in the drift. Ore extraction from any place on the periphery of the cone toe will induce the flow of ore, which will follow the surface lines of the cone. Therefore the cone will form a very small width outflow opening in the zone of the apex of the arched roof.

Such a situation is unfavorable for sublevel caving because the effective extraction opening width will be very small. The gravity flow of blasted ore in the slice and flow of waste will have the pattern shown in Fig. 36.10a.

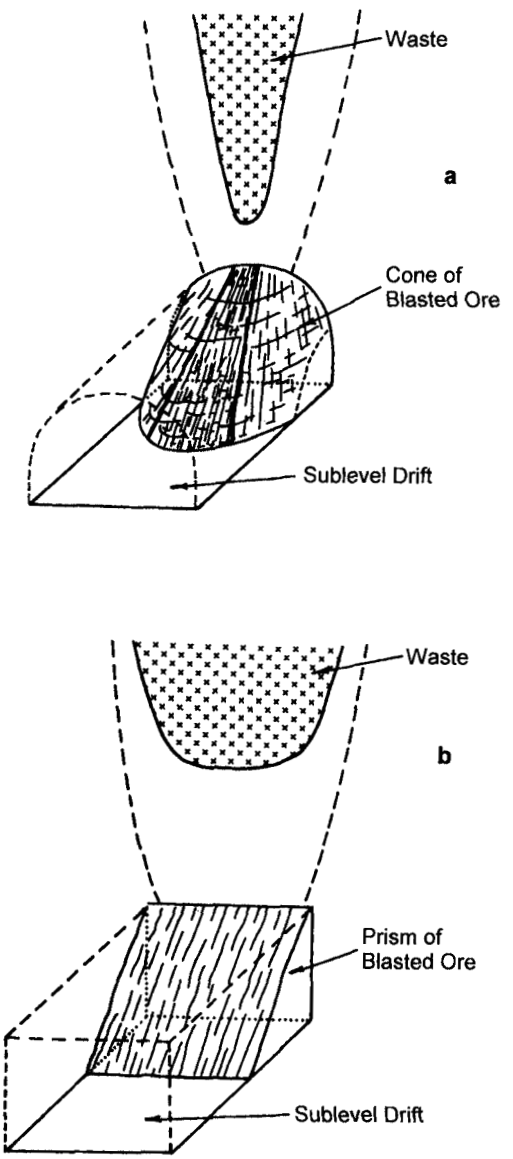


FIGURE 36.11 Gravity flow pattern as a function of the shape of sublevel drift roof: (a) arched roof and (b) flat roof

When the roof is horizontal (or slightly arched; see Fig. 36.11b), then the blasted ore will form a prism in the sublevel drift. Extraction from the slope toe of this prism will have a parallel flow on the slope, and therefore almost the total width of the sublevel drift can be utilized as an effective width of the extraction opening. Naturally, such a situation is favorable for sublevel caving because the effective width of extraction will be large and the gravity flow of ore (and waste) will have the pattern shown in Fig. 36.11b.

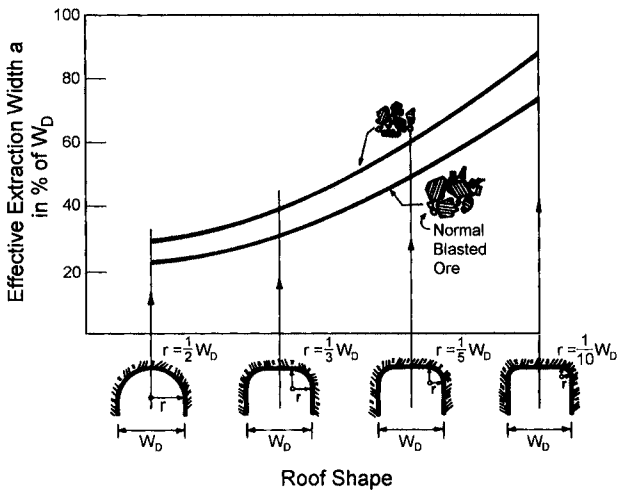


FIGURE 36.12 Approximate effective extraction width as a function of the shape of sublevel drift roof as a percentage of sublevel drift width W_D

Figure 36.12 indicates the approximate effective extraction width as a percentage of the width of the sublevel drift W_D as a function of the roof shape.

Correct ore extraction demands not only a large extraction width, but also a satisfactory thickness or depth of the outflow zone. This depth depends upon how deep the loader can dig into the slope. Naturally, when the digging depth is small, the depth of the outflow zone is also small, and one can utilize only a small part of the sublevel drift height for extraction.

According to Rankin's theory, the maximum stress trajectories are neither vertical, nor perpendicular, nor parallel to the slope inclination, but are inclined away from the vertical by $\beta = (90° - \phi)/2$ (see Fig. 36.13).

In this figure, ϕ is the natural angle of repose, h_D is the height of the sublevel drift, point 1 represents the theoretical slope toe, 3 is the point at the edge formed by the intersection of the drift roof and sublevel front, point 2 represents the intersection of trajectories from point 3 with the floor of the sublevel drift, and X is the distance between points 1 and 2.

With material extraction from the toe, the slope angle will approach the theoretical limit defined by the trajectory 3-2. This is the theoretical situation in which the slope 3-2 will have a stability coefficient equal to 1.0. This means that such a slope is just ready to fail.

The logical conclusion is that to utilize the complete height h_D of sublevel drift for extraction, the digging depth must be

$$X = h_D \operatorname{c\,cotg}\phi - h_D \tan \frac{90 - \phi}{2}.$$

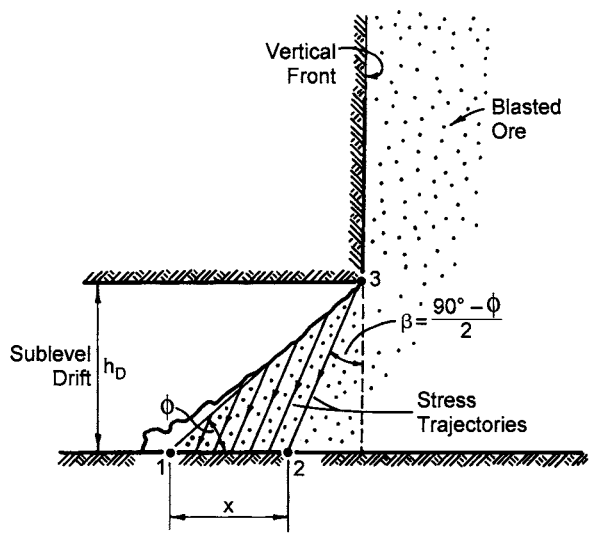


FIGURE 36.13 Mechanics of the slope from fragmented material in the sublevel drift

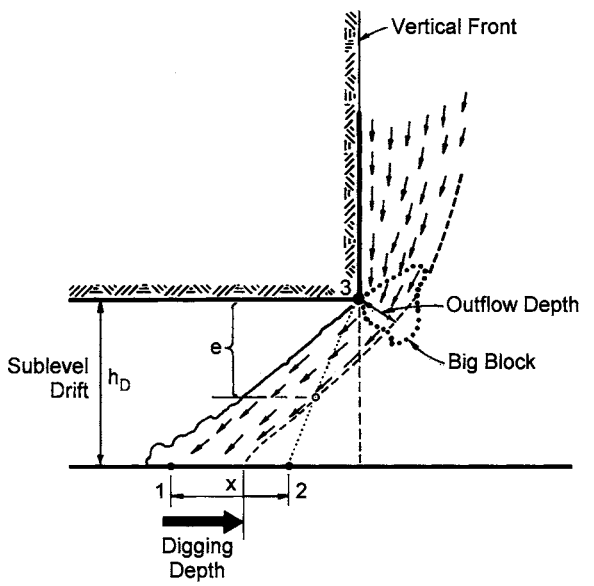


FIGURE 36.14 Outflow depth into the sublevel drift

Naturally, the digging depth of a loading machine, usually about 1 to 1.3 m, is much smaller than the theoretical depth x. This means that only a certain upper part (e) of the sublevel drift height will be utilized for normal extraction (Fig. 36.14). The remaining lower part, although not utilized for a normal extraction, also has a function—allowing the extraction of blocks with dimensions larger than the depth of the outflow zone on the level of the sublevel drift floor.

Even in normal extraction, the slope in the sublevel drift does not have a constant angle. The slope can vary between the limits, defined in Fig. 36.13 as points 1 and 2. It means the slope toe (point 1) can be close to point 2. In some cases, point 1 will be identical with point 2 and the slope will be very steep. The closer the slope is to the plane 3-2 (see Fig. 36.13), the more dangerous is its abrupt failure. Therefore, for safety reasons, it is useful to define the limiting angle of plane 3-2 and to induce the slope failure before this limiting condition occurs.

The extraction can be stopped by arching of the material over the extraction zone. Most of the time, arching occurs over the outflow depth of the extraction zone (see Fig. 36.14), because this is the smallest dimension of the extraction opening. Although arching can have many variations, one can distinguish the following basic types: arching caused by a group of blocks that forms an arch structure, arching caused by a compacted material (including the effect of apparent cohesion and/or plasticity), and arching caused by a combination of these two.

When ore extraction is interrupted, the blasted ore (and caved waste) starts a process of settlement that can result in some compaction of the material. This occurs primarily when plastic components and/or fine particles and water are present.

Therefore, arching (mainly of the second type) is more frequent after holidays or after any longer interruption of a fluent extraction. Therefore, in materials with a relatively high rate of settlement and compaction, it is best to prevent this interruption. In practice, this means that total ore extraction from blasted slices of a sublevel should be completed before any long interruption.

Naturally, the best method to avoid or minimize arching caused by big blocks is through controlled drilling and blasting.

Another question is how to safely and efficiently remove the different types of arches. It is impossible to define a universal method that will satisfy all types of arching under different conditions and circumstances. It can only be recommended that before any removal work commences, it is very important to understand the type, structure, and probable behavior of the arch and to define the critical zone or spot whose displacement (or excavation) will induce the failure of the arch. Otherwise, the work can be unsuccessful and even very dangerous.

As has been mentioned, the extraction ellipsoid (for a minimum size opening) does not have the shape of an elongated ellipsoid of revolution whose upper half is exactly identical to its lower half. Rather, the real shape has an upper half that is wider than the lower half (see Fig. 36.15).

For fine materials, the difference between the real and simplified shape is very small and can be considered negligible. The difference starts to be more pronounced in coarse materials and for extraction openings with widths greater than the minimum size. Lower mobility of coarse material, an apparent interlocking of the large fragments, together with partial mass flow in the

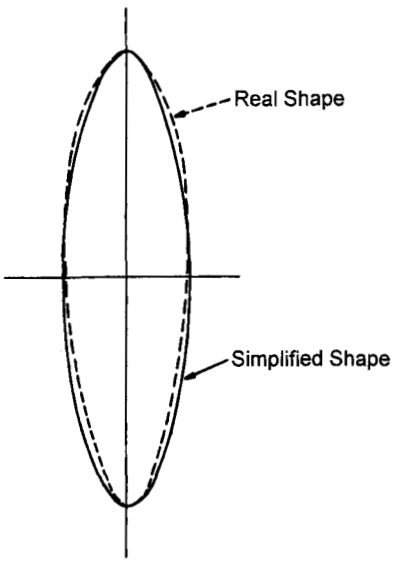


FIGURE 36.15 Difference between the idealized (simplified) and real shape of the ellipsoid by a minimum size extraction opening

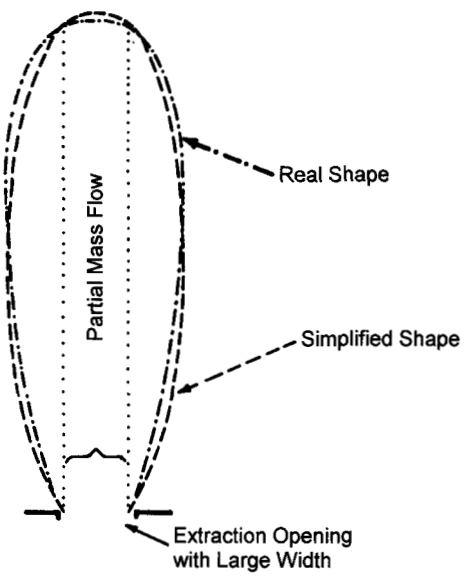


FIGURE 36.16 Simplified and real shape of ellipsoid by large width of the extraction opening

central zone, results in a steepening of the lower part. Here the loosening and gravity motion is easier because it is directed toward the extraction opening. The material in the upper part is remote from the opening, has lower loosening and lower mobility, and therefore needs a wider zone for gravity flow (see Fig. 36.16).

The change of shape is even more evident in sublevel caving because, in addition to the coarse and blocky material and greater width of the extraction opening, there are some additional factors due to ore blasting.

In sublevel caving, the vertical slice of the ore is blasted against the caved waste.

The final swell of the blasted ore varies over the height of the slice. Naturally, the greatest swell and loosening is in the lower part because a certain volume of the blasted ore falls and flows into the sublevel drift. This lower portion of the slice also has finer fragmentation due to the closer spacing of the drill holes in the fan drilling pattern. (Even with a parallel drilling pattern, the lower part of the blasted slice contains a higher percentage of small fragments and small particles than the upper part, because they can penetrate downward into the cavities between the larger fragments.)

The swell and mobility of blasted ore in the slice decrease upward (away from the sublevel drift), and the fragments (and blocks) are bigger because of the larger spacing of drill holes in the upper part of the fan drilling pattern. Naturally, in the upper parts, the confinement of the blasted ore in the slice is higher than in the lower part, which certainly results in a lateral prestressing of the blasted ore. An additional factor that can decrease the mobility of the upper parts is the penetration of a certain volume of overlying waste between the blasted slice and the ore front at the moment of blasting, when for a short while the blasted slice is separated from the sublevel ore front. This can also be combined with a certain back break in the remaining ore front of the sublevel.

Therefore, with certain simplifications, it can be said that the blasted ore in the slice has the highest mobility in the lower part, and the mobility decreases in the upward direction. Naturally, the smaller the mobility, the greater the width necessary for gravity flow.

The extraction ellipsoid will be slimmer in the lower section, with its maximum width being realized somewhat above half of its height, as is shown in Fig. 36.17a.

The shape of the extraction zone (in vertical section A-A', perpendicular to the sublevel front) is constructed in Fig. 36.17b. Because the surface of the sublevel ore front is very rough, the friction along this plane is high and the ellipsoid axis will deviate from vertical. The deviation increases with increasing friction.

Determination of this deviation is difficult, but it can be assumed that at a sublevel height of about 11.0 m (36 ft) the horizontal distance between the apex of the extraction ellipsoid and the vertical plane representing the sublevel front can be about 1.0 m (3 ft) or even more.

Blasted ore in the slice usually represents a very heterogeneous material.

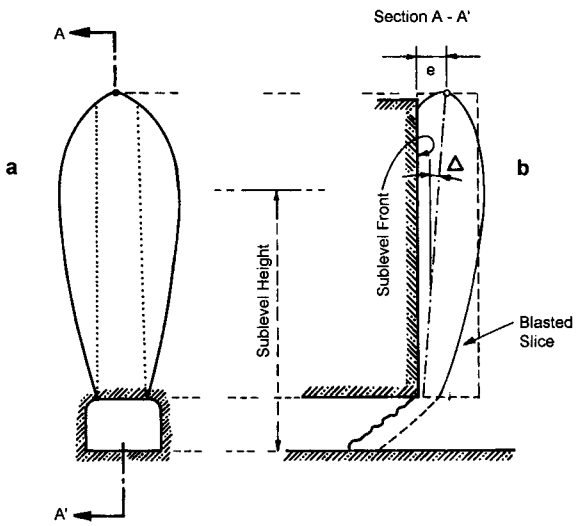


FIGURE 36.17 Simplified shape of the extraction zone by sublevel caving

Even with perfect drilling and blasting, the variations induced by so-called blasting tectonics can result in the creation of zones with large fragments, or zones with fine material. Big blocks of ore (or waste) occur at random, mainly in the upper part of the slice. The migration of these large blocks or even pockets of fine ore (or waste) will cause disturbances in the gravity flow. Many other disturbing factors can also enter, for example, interruption of ore extraction from the slice, which can result in settlement and compaction of the blasted ore; discontinuous failure and caving of waste rock masses, which can change the stresses acting on the slice of blasted ore; progressive failure and caving of other kinds of rock types at depth, which can produce another type of fragmentation of caved waste than experienced in the upper zones; and increased size of the progressive failure zone and increased area of subsidence, leading to the collection of water, which can cause undesirable changes in the properties and behavior of the blasted ore and therefore difficulties in ore extraction, etc.

It is evident that because of these factors, the extraction ellipsoid in sublevel caving is not regular, as might be expected with a homogeneous coarse material extracted without these disturbing influences.

Laboratory Studies and In Situ Testing

Research directed toward understanding the shape of the extraction ellipsoid in sublevel caving has been executed on models in two types of in situ testing. Each of these research systems has certain advantages. In the laboratory, even very sophisticated models cannot simulate the natural conditions correctly. In the field, two types of testing are used.

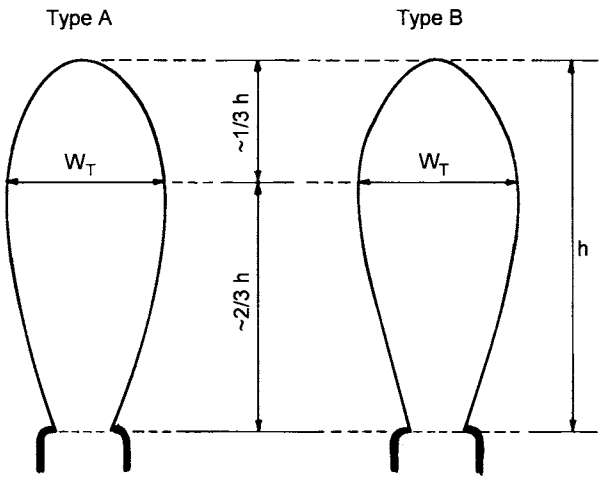


FIGURE 36.18 Generalized geometry (types A and B) of extraction zone

The first type of in situ testing is based on the utilization of a large number of marks, located in drill holes in the ore slice before blasting. After blasting, the marks are dispersed in the rubble and their new location is not exactly known. The second type is based upon the location of marks in the fragmented ore, that is, in the blasted ore of the slice. This method is difficult, laborious, very expensive, etc., and therefore only a few marks can be installed safely in blasted ore.

Analyses of laboratory experiments and of in situ tests suggests two shapes for the extraction ellipsoids in sublevel caving.

The generalized geometry of both shape types is shown in Fig. 36.18. Although the differences are relatively small, it is possible to distinguish that type A has a more rounded shape, whereas type B is more angular.

The cause of a type A or a type B shape is still questionable because the data from models (with limited precision) and from in situ tests (small number of tests) are still insufficient for solution.

In both types the maximum width W_T of the extraction ellipsoid is about $\frac{2}{3}\,h$ above the roof of the sublevel drift, where h is the extraction height above the sublevel drift (see Fig. 36.18). The maximum depth d_T (or thickness) of the extraction ellipsoid perpendicular to the sublevel front also occurs at the same height.

Because of the large width of the extraction opening, the structure of the extraction ellipsoid is analogous in principle to that shown in Fig. 36.9.

For these reasons (disturbance factors and difficulties in testing), the exact shape of the extraction ellipsoid in sublevel caving is still not explicitly defined. Although knowledge of the shape is sufficient for engineering practice, it is still insufficiently precise for the development of an explicit theory.

PRACTICAL DESIGN GUIDANCE

The main question in the design of a sublevel caving mining method is the determination of a mining geometry, which will satisfy as much as possible the parameters of gravity flow. This means that one needs to first determine the width and thickness of the extraction ellipsoid for a certain extraction height.

Naturally, the parameters can be determined by in situ tests, but usually the test data are not available in time for the mine design.

Until now, no explicit method for making engineering calculations of these parameters has been available. Because of the heterogeneity of coarse materials and the complexity of factors involved in gravity flow, certain empirical formulas will be introduced here, which can serve as guidelines for the approximate determination of gravity flow parameters and for the geometry of the method.

Dimensions of the Extraction Ellipsoid

Coarse material is actually a mixture of different sizes. The presence of a small proportion of fine particles and fragments decreases, often markedly, the effect of the coarse fragments. As a result, the gravity flow zone in coarse material is sometimes surprisingly slim.

Because the eccentricity of the ellipsoid increases with its height, the greater the height, the slimmer the flow. (This is very well known in block caving. With big block heights, the gravity flow above a single drawpoint can form a chimney with vertical walls.)

With the same fragmentation, the gravity flow of a high density material (i.e., blasted iron ore) will be slimmer than the flow of a low-density material (i.e., blasted copper ore).

The width of the gravity flow depends also upon the size of the extraction opening. (See also Figs. 36.9, 36.10, 36.11, 36.12, and 36.16.)

To exclude the variable factor of different extraction opening sizes, the data from analytical and model research, in situ tests, and from observations in sublevel caving, operations were normalized for the determination of a theoretical width W' of the extraction ellipsoid, assuming extraction of material through a minimum size opening.

Assuming a normal fragmentation of the blasted ore, the theoretical minimum size opening is about 1.8 m.

For high-density blasted ore (iron ore) the approximate theoretical width W' of the extraction ellipsoid is shown in Fig. 36.19 as a function of the extraction height h_T.

In sublevel caving, the total extraction height (h_T) in the ore is normally between 15m (50 ft) and about 26 m (85 ft).

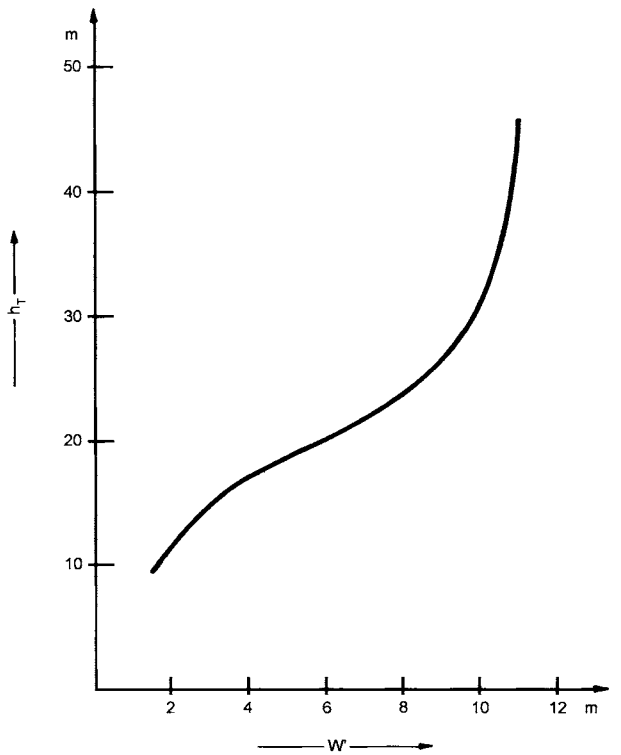


FIGURE 36.19 Theoretical width W' of an extraction ellipsoid (for high density ore) as a function of the extraction height h_T

The theoretical width W' of the extraction ellipsoid, corresponding to the different total extraction heights h_T, within these limits [15 to 26 m (50 to 85 ft)] is shown in Fig. 36.20 for low- and high-density blasted ore.

The effective extraction width a (see also Fig. 36.12) is usually larger than the width of a minimum size opening [1.8 m (6 ft)], and therefore the total width W_T of the extraction ellipsoid in a sublevel caving operation will be bigger than that shown in Fig. 36.20.

A very approximate value of the total width W_T and total depth d_T of the extraction ellipsoid for a given height h_T can be calculated in meters (m × 3.2808 = ft) using the following empirical formula:

$$W_T \approx W' + a - 1.8 \qquad \textbf{(EQ 36.5)}$$

$$d_T \leq W_T/2 \qquad \textbf{(EQ 36.6)}$$

where W' is the total extraction height h_T (see Fig. 36.20) and a is the effective width of the extraction opening dependent upon the shape of the sublevel drift roof (see Fig. 36.12).

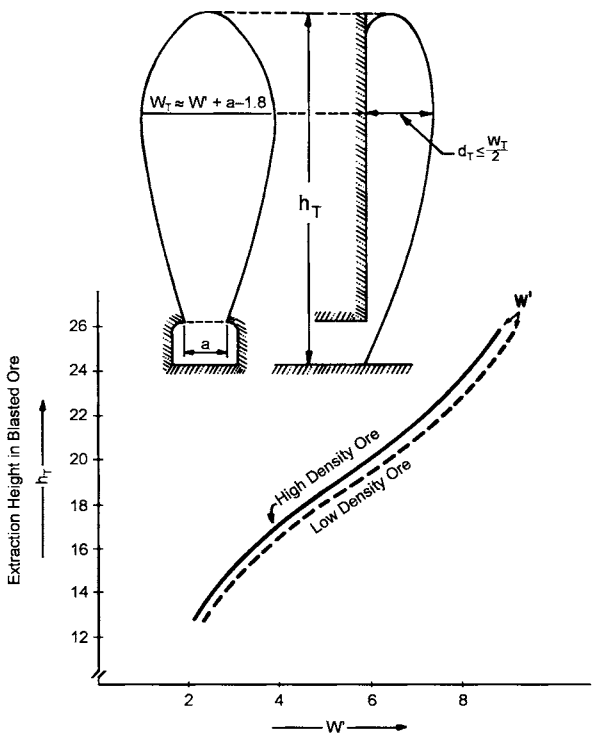


FIGURE 36.20 Very approximate width W' of the extraction ellipsoid in high or low density ore as a function of the extraction height h_T

Vertical Spacing of Sublevel Drifts

The sublevel caving extraction drifts should be located in a checkered pattern conforming to the pattern of gravity flow.

In the vertical direction (see Fig. 36.21), the sublevel drifts should be located in the zone where the extraction ellipsoid has its maximum width W_T. This is at about $\frac{2}{3}\,h$ (h is the extraction height above the roof of the lower sublevel drift). In principle, this location determines the height (h_s) of the sublevel.

After ore extraction, an intact (not blasted) ore pillar having a triangular shape (transverse section) formed by the limiting inclined drill holes of the fan drilling pattern remains between the sublevel drifts on the upper level.

This pillar is usually covered by blasted ore remaining in a passive zone above the pillar after ore extraction. The thickness of this passive zone of blasted ore can be large or small depending upon the inclination of the limiting drill holes in the fan drilling, the width W_D of sublevel drifts, the spacing S_D of sublevel drifts (see Fig. 36.21), and the properties and gravity flow parameters of the coarse material, which controls the shape of the passive zone.

The blasted ore in the passive zone can be partially recovered by extraction on the lower sublevel. Therefore, the total extraction height h_T is the distance

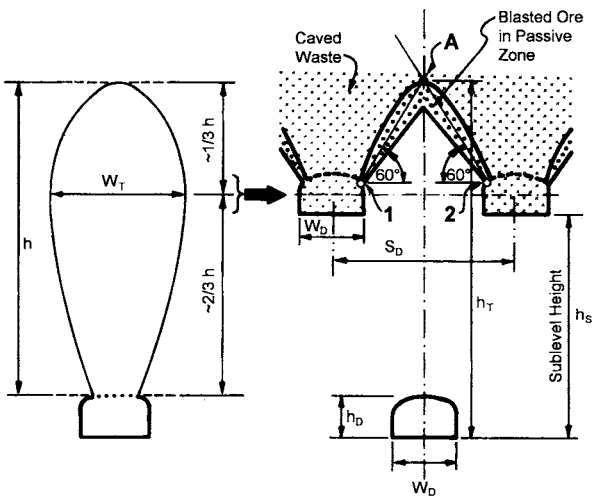


FIGURE 36.21 Vertical location of sublevel drifts conforming to the pattern of gravity flow

between the floor of the lower sublevel drift and the apex *A*, formed by the remaining blasted ore in the passive zone.

Because it is difficult to define the shape and dimensions of this passive zone, one must usually estimate the total extraction height h_T, using experience from sublevel caving operations with similar conditions.

As a rule of thumb, the location of the apex *A* can be very approximately defined as the intersection of 1.05 rad (60°) inclined planes from points 1 and 2, as shown in Fig. 36.21.

Horizontal Spacing of Sublevel Drifts

Knowing H_T and W_T, one can determine an approximate horizontal spacing S_D of the sublevel drift axes. This determination is based upon utilizing the idealized relations between the ellipsoids of extraction and loosening described in Fig. 36.4. Assuming similarity of eccentricity of ellipsoids and that the loosening ellipsoid is 2.5 times bigger than the ellipsoid of extraction, the width of the extraction ellipsoid is 40% of the width of the appropriate ellipsoid of loosening. (For these idealized conditions, one can calculate the width of the ellipsoid of loosening on any horizontal section of extraction ellipsoid.)

In sublevel caving, one needs to determine the width W_L of the loosening ellipsoid on a horizontal section just at the level where the extraction ellipsoid has its maximum width W_T (see Fig. 36.22).

The width of the loosening ellipsoid on this level indicates the approximate horizontal spacing S_D of the sublevel drifts.

Assuming that the relations and principles of idealized gravity flow (described in Fig. 36.4) can be applied to sublevel caving, the total width W_T of the

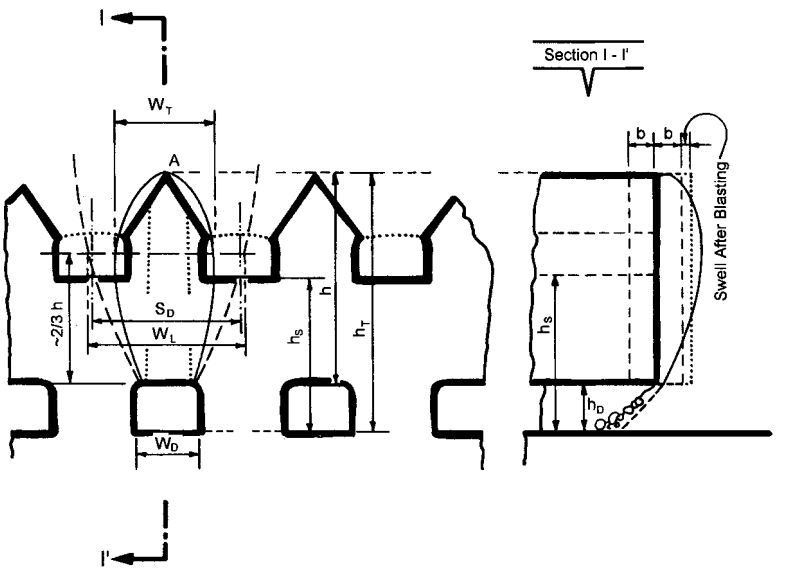


FIGURE 36.22 Simplified principle of sublevel caving geometry

extraction ellipsoid in sublevel caving is about 60 to 65% of the width of the loosening ellipsoid on the level where the extraction ellipsoid has its maximum width W_T.

The width is about 60% for heights up to about h_T = 18 m (59 ft). Over 18 m (59 ft), the width W_T is about 65%.

Hence, the approximate horizontal spacing S_D of sublevel drifts is:

For extraction heights $h_s \leq 18$ m (59 ft)

$$S_D < \frac{W_T}{0.6} \qquad \textbf{(EQ 36.7a)}$$

For extraction heights $h_s > 18$ m (59 ft)

$$S_D < \frac{W_T}{0.65} \qquad \textbf{(EQ 36.7b)}$$

The basic geometric unit of sublevel caving is in principle defined by the relation between the horizontal spacing S_D of sublevel drifts and vertical height of the sublevel h_s.

In conventional sublevel caving (see Fig. 36.22) this relation is

$$S_D \leq h_s$$

which means that the basic geometric unit has a square shape, or deviates only slightly from square.

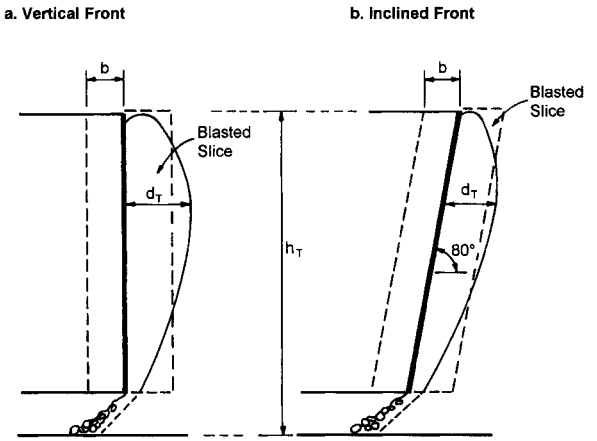


FIGURE 36.23 Vertical (a) and inclined (b) fronts of sublevel caving. Inclined front increases the dilution.

Improved precision of longhole drilling in recent years has resulted in a tendency to increase the height of the sublevels with the goal of decreasing the volume of development.

Ore draw can continue over heights greater than the extraction height h_T in the ore, but the dilution then rapidly increases.

Thickness of the Blasted Slice

An approximate guide for the thickness of the blasted slice (burden spacing) on the front of the sublevel is usually

$$b \leq \frac{d_T}{2} \qquad \text{(EQ 36.8)}$$

where d_T is from Equation 36.6.

Inclination of the Front

The sublevel front is usually inclined at 1.4 rad (80°). This inclination is favorable not only for drilling and charging of drill holes, but also for minimizing the dilution.

The effect of the sublevel front inclination is evident from Fig. 36.23, where Fig. 36.23a has a vertical front and Fig. 36.23b has an inclined front. The extraction height h_T is the same in both cases.

With a vertical front (Fig. 36.23a) the extraction ellipsoid intrudes deeper into the caved waste rock. This is particularly true when the sublevel is high.

The inclination [1.4 rad (80°)] of the front shown in Fig. 36.23b will cause a change of the gravity flow shape, and the extraction ellipsoid will be much slimmer than with a vertical front. With certain simplifications, it can be said

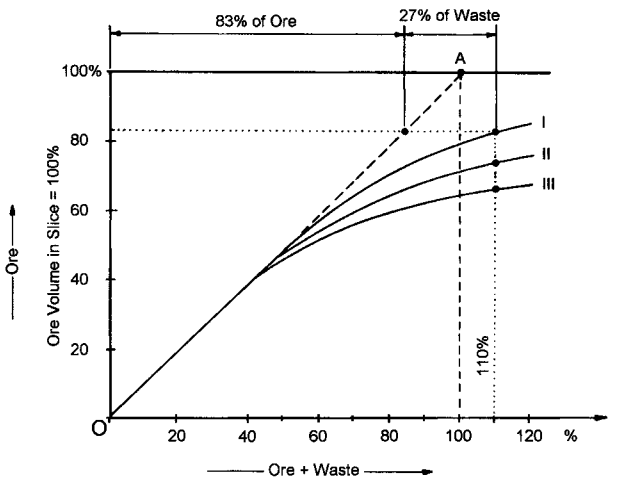


FIGURE 36.24 Generalized process of dilution development in sublevel caving. Curve I represents good extraction and curve III represents bad extraction with high dilution.

that in this case the extraction ellipsoid has a tendency to be inscribed within the blasted ore slice. The intrusion of the extraction ellipsoid into the caved waste is smaller, and therefore the dilution, with ore extraction over the same extraction height h_T, will be smaller than with a vertical front.

Extraction and Dilution

The dilution process in sublevel caving can take many different anomalous, unexpected, and surprising forms, for example, the outflow of a certain quantity of caved waste rock almost at the beginning of ore extraction, which is then followed by normal ore flow; or the random occurrence of waste rock pockets at different stages of the ore extraction, etc. The cause of such phenomena is still not defined satisfactorily. In ore that tends to arch over the extraction opening, one can expect a higher probability of anomalous dilution.

The normal process of dilution development in sublevel caving is illustrated in Fig. 36.24 as a function of the volume of the ore and the volume of extracted material (consisting both of ore and caved waste). It is expressed as a percentage of the volume of ore slice, where the total volume of the ore slice is 100%.

Theoretically, the best extraction will be defined by the line OA in Fig. 36.24. This means one will extract 100% of the ore without any waste. Evidently, such a situation is not possible in practice.

Figure 36.24 shows three different cases of ore extraction with different dilutions (i.e., with different volumes of waste, extracted together with the ore).

The optimum total extraction (ore and waste) can have different values depending upon the value of the ore and the overall economics. Figure 36.24 assumes that the extraction will stop after drawing 110% of the slice volume.

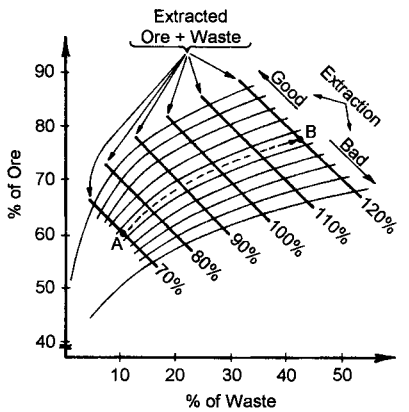


FIGURE 36.25 Simplified chart of dilution development

Curve I in Fig. 36.24 represents good extraction because ore recovery is 83% with only 27% waste.

Curve II demonstrates a case in which the distribution is about 75% ore and 35% waste, for a total extraction of 110%. Of course, this is not as good as the previous example, but in certain conditions, primarily in cases when a simple and cheap process of waste separation is applicable, it can be acceptable.

Curve III characterizes a bad extraction. In this case, for a draw of 110% total material, only 65% of ore with waste as high as 45% is recovered.

As a guide for evaluating the extraction (assuming 110% of material [ore and waste] extraction) the following classification can be used:

Class I. Extraction is good because the ore is at least 80% and the waste is less than 30%.

Class II. Extraction is acceptable in certain cases because the ore is at least 75% and the waste is no more than about 35%.

Class III. Extraction is poor because the ore is about 65% and the waste is 45% or more.

Total dilution development is already defined at the beginning of the dilution process. The approximate relations are evident from the chart in Fig. 36.25.

For example, if after a material extraction of 71%, the ratio is 60% ore and 11% waste (see point A in Fig. 36.25), one can expect that with 120% material extraction (see point B in Fig. 36.25), the ratio will be about 77% ore and at least 43% waste.

The limit between good and bad extraction can be different in different ores with different ore grades.

The approximate boundary between good and bad extraction is indicated in Fig. 36.25 by the line A-B.

Opening Stability

The opening of the sublevel caving structure is usually very good, and so this method can be safely utilized even in relatively soft ores and rock masses. Nevertheless, it is necessary to understand that the checkered pattern of sublevel drifts generates a special stress distribution, which determines the stability limits of the structure. This checkered pattern of drifts results in high shear stress concentrations in the rock lying between the closest corners of drifts on the upper and lower levels.

Under the same loading conditions, the stress concentration increases with decreasing sublevel height h_s and drift spacing S_D and with increasing sublevel drift width W_D and height h_D. See Fig. 36.22.

Under critical loading, the failure of the sublevel structure is characterized by shear cracks between the closest corners of sublevel drifts on the upper and lower levels. Detailed stress analyses can be provided using the finite element or other methods.

It is necessary to emphasize that the introduction of sublevel caving at depth requires a careful stress and stability analysis for definition of a safe geometry.

With regard to the checkered location and relatively short distance between sublevel drifts, damage to the rock mass resulting from poor drifting practice is very undesirable for good stability. Therefore, good smooth blasting is very necessary, particularly in soft ores and rocks and at greater mining depths.

Example Problem

This problem involves the determination of an approximate geometry for sublevel caving given the following information:

Width of the sublevel drift W_D = 5 m (16 ft)

Height of the sublevel drift h_D = 3.5 m (11½ ft)

Blasted ore has a high density

Shape of the drifts: flat roof

To determine the parameters and geometry: Total extraction height is estimated to h_T = 21 m (69 ft). Shape of the sublevel drift with flat roof is favorable for ore extraction, and the effective extraction width (according to Fig. 36.12) is about 70% of W_D, that is, $a = W_D \times 0.7 = 5 \times 0.7 = 3.5$ m (11½ ft).

With respect to the location of sublevel drifts in the vertical direction (see Fig. 36.21), the approximate sublevel height is h_s = 12.5 m (41 ft).

Theoretical width W' of the extraction ellipsoid for a total extraction height h_T = 21 m (69 ft) is determined from Fig. 36.19 to be W = 6.8 m (22 ft).

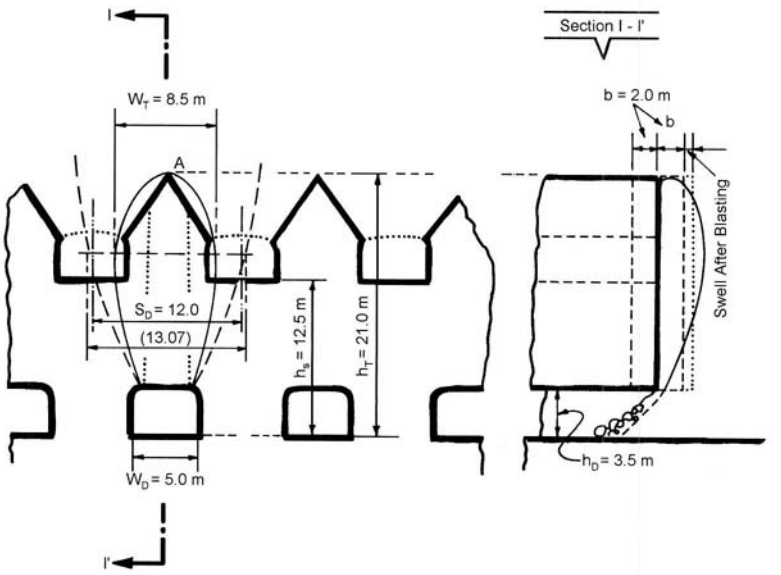


FIGURE 36.26 Geometry of an example problem

Approximate total width and depth of the extraction ellipsoid, using Equations 36.5 and 36.6, is

$$W \approx 6.8 + 3.5 - 1.8 = 8.5 \text{ m (28 ft)}$$

$$d_T \leq \frac{8.5}{2} \quad ; \quad d_T \leq 4.25 \text{ m (14 ft)}.$$

Spacing of the burden b using formula (36.8) is

$$b \leq \frac{4.25}{2} \quad ; \quad b \leq 2.12 \text{ m (7 ft)}.$$

Because the total extraction height $h_T = 21$ m > 18 m (69 ft > 59 ft), the horizontal spacing of sublevel drifts is calculated using Equation 36.7, that is,

$$S_D < \frac{W_T}{0.65} = \frac{8.5}{0.65} = 13.07 \text{ m (43 ft)}; \; S_D \leq 13.07 \text{ m (43 ft)}.$$

For good ore extraction, it is favorable when the adjacent zones of loosening at the elevation of the maximum width W_T of the extraction ellipsoid intersect. Therefore, the spacing becomes:

$$S_D = 12 \text{ m} < 13.07 \text{ m (39 ft} < 43 \text{ ft)}.$$

As shown in Fig. 36.26, the complete geometry of the sublevel caving in the example is:

Sublevel height h_s = 12.5 m (41 ft).

Horizontal spacing of sublevel drifts S_D = 12 m (39 ft).

Sublevel drifts section h_D = 3.5 m (11½ ft), W_D = 5 m (16½ ft).

Total width of the extraction ellipsoid W_T, assuming a total extraction height h_T = 21 m (69 ft), is W_T = 8.5 m (28 ft).

Spacing of the burden b = 2 m < 2.12 m (6½ < 7 ft).

It must be emphasized that the system for the determination of sublevel caving geometry presented in this section is very simplified and should serve as a guide only. It can be used for developing a preliminary concept of the mining method geometry, especially in cases where no other data are available.

SURFACE EFFECTS FROM SUBLEVEL CAVING

Concerning environmental constraints, the basic condition for utilization of sublevel caving is that an inevitable and drastic destruction of the surface is permissible.

Sublevel caving will induce progressive failure, caving, and subsidence of the overlying waste rock masses, undercut by mining. In this section, the extent of the damage zone is considered.

Shape, Size, and Extent of Disturbance

The shape, size, and course of the progressive failure and caving depend on many factors. Probably the most important are inclination of the orebody (β_0); the orebody width (W); the depth (H_1) of the bottom level of future underground mining; the height (H_2) of the caved waste rock zone, which generates the arching forces that resist failure; shear strength of the rock masses involved in progressive failure, defined by effective cohesive strength (c) and friction angle (ϕ); and density of the waste rock masses (γ).

When the orebody inclination creates a stable footwall slope, the progressive failure and caving will occur only in the hanging wall, as shown in Fig. 36.27.

Mining of a vertical or very steep dipping orebody will induce progressive failure both in the hanging wall and the footwall (see Fig. 36.28).

A simplified mechanism of progressive failure is characterized by the break plane B at angle α_B and the sliding plane S with angle α_S. The magnitude of these angles is a function of certain configurations of H_1, W, H_2, C, ϕ, γ, and α.

As is evident from Figs. 36.27 and 36.28, the devastation zone DZ is limited by the intersection of the break plane B with the surface. Between the intersection of break plane B and sliding plane S with the surface is a so-called transition zone TZ. Even this zone is not sufficiently stable for the location of any permanent construction.

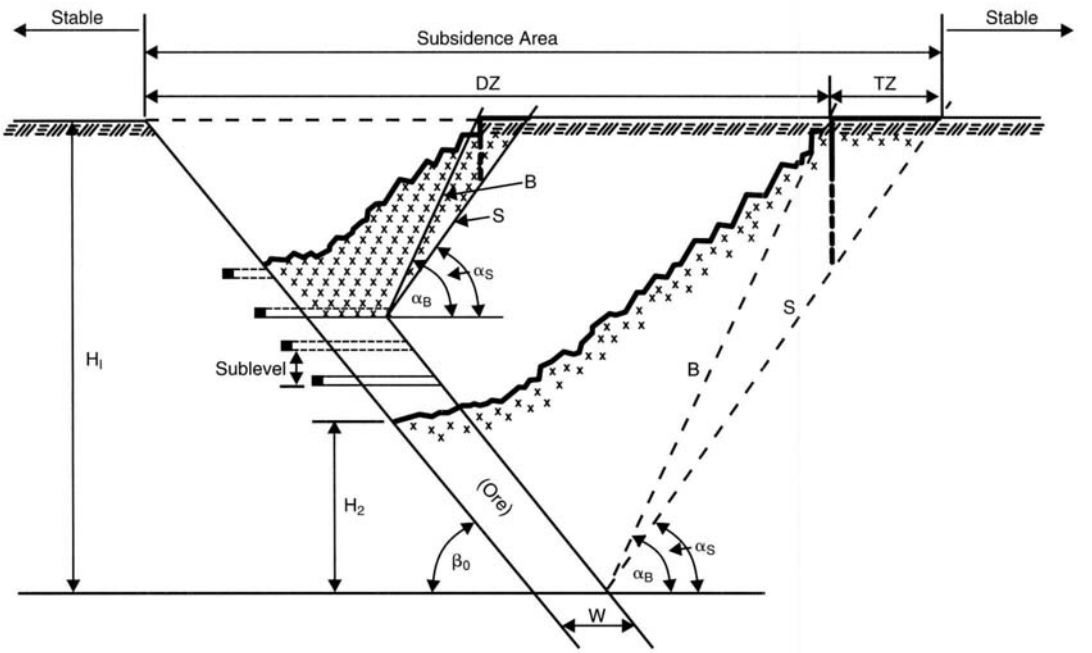


FIGURE 36.27 Progressive failure and subsidence induced by sublevel caving in the hanging wall

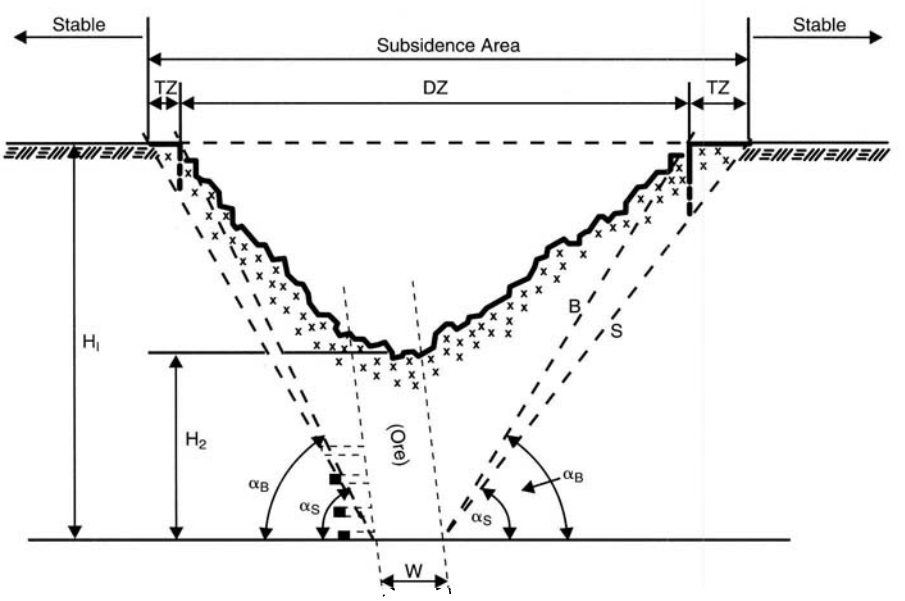


FIGURE 36.28 Progressive failure and subsidence caused by sublevel caving in the hanging wall and footwall

The intersection of the sliding plane *S* with the surface represents the boundary of the subsidence area. Naturally, the stable surface is outside the subsidence area.

For the safe location of permanent structures on the surface (shafts, concentrator, buildings, tailings dams, etc.) and underground (crushing plant, pumping station, main ore and waste rock passes, permanent tunnels and drifts, etc.), it is necessary to define the consequence of sublevel caving; that is, it is necessary to determine the unstable zones of progressive failure (limited by sliding planes) and the subsidence area. All important constructions must be located outside of these zones.

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CHAPTER 37

Sublevel Caving at Craigmont Mines Ltd.

R.A. Baase, W.D. Diment, and A.J. Petrina

INTRODUCTION

In 1957, diamond drilling on a magnetic anomaly indicated an extensive zone of copper mineralization on what is now the Craigmont Mines property. By mid-1958, drilling established a copper orebody.

Milling commenced in September 1961 at 4536 t/d (5000 stpd), and by the end of October 1977 the mine had produced 339,662.04 t (374,363.9 st) of copper. At present, two-thirds of the mill feed is derived from underground operations and one-third from low-grade surface stockpiles.

Craigmont Mines is situated 209 km (130 air miles) northeast of Vancouver (see Fig. 37.1), 16 km (10 miles) west of the town of Merritt, a logging, ranching, and mining community of about 7000 people. It is serviced by paved highways, Canadian Pacific Railway, British Columbia Hydro, and Inland Natural Gas Co. Water is pumped from the Nicola River, a distance of 6 km (4 miles) and a lift of 244 m (800 ft).

In March 1967, the open-pit mining operations at Craigmont Mines reached their economic limit and were suspended. Before this, it had been decided that a sublevel caving method of underground mining would be used to supply ore to the concentrator after the cessation of open-pit production. This chapter describes the factors influencing the choice of mining method, some of the problems encountered, mining practices, and results.

GEOLOGY

The orebodies of upper Triassic age are located in a limy horizon striking east-west, closely paralleling the intrusive Guichon batholith, bounded on the south by rhyolites and on the north by graywackes, and dipping steeply to the south (Figs. 37.2a, b).

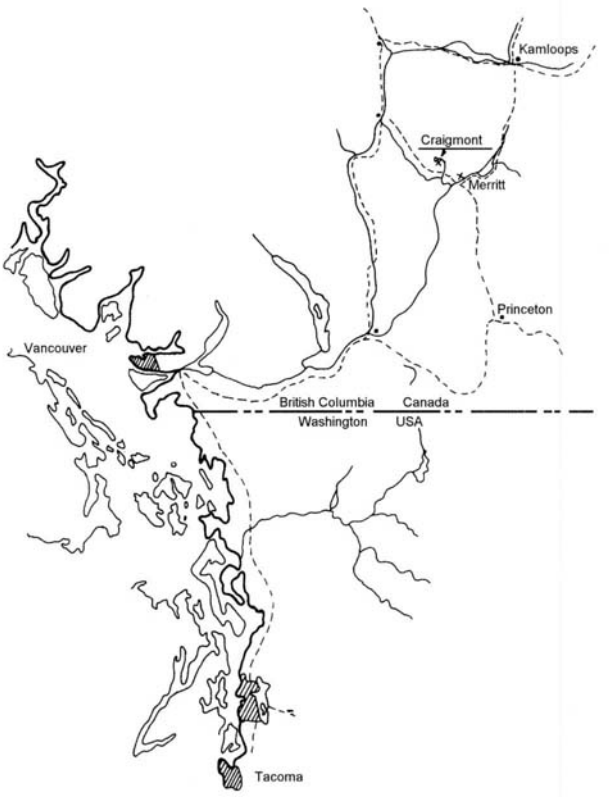


FIGURE 37.1 Map showing the location of Craigmont

The orebodies are relatively narrow with a maximum width of 79 m (260 ft), a combined strike length of 853 m (2800 ft), and a vertical extent of 610 m (2000 ft). Chalcopyrite is virtually the only copper mineral, and 20% of the ore zone consists of acid-soluble magnetite and hematite.

The area has been subjected to considerable faulting and brecciation, which is a major factor in the mining operation. Total geological reserves, at 0.7% Cu cutoff, for the deposit were 22,316,743 t (24,600,000 st) at 1.89% Cu. An additional 5,236,270 t (5,772,000 st) at 0.6% Cu were mined from the open pit.

Ground Conditions

The waste rocks—graywacke, andesites, and diorite—are relatively incompetent due to the high degree of fracturing and jointing, and all require varying degrees of support.

The ore zones are somewhat less fractured; ground support is still required, however, although to a lesser extent than in the country rock. Ground conditions in the main orebody are better than in the smaller, narrower orebodies.

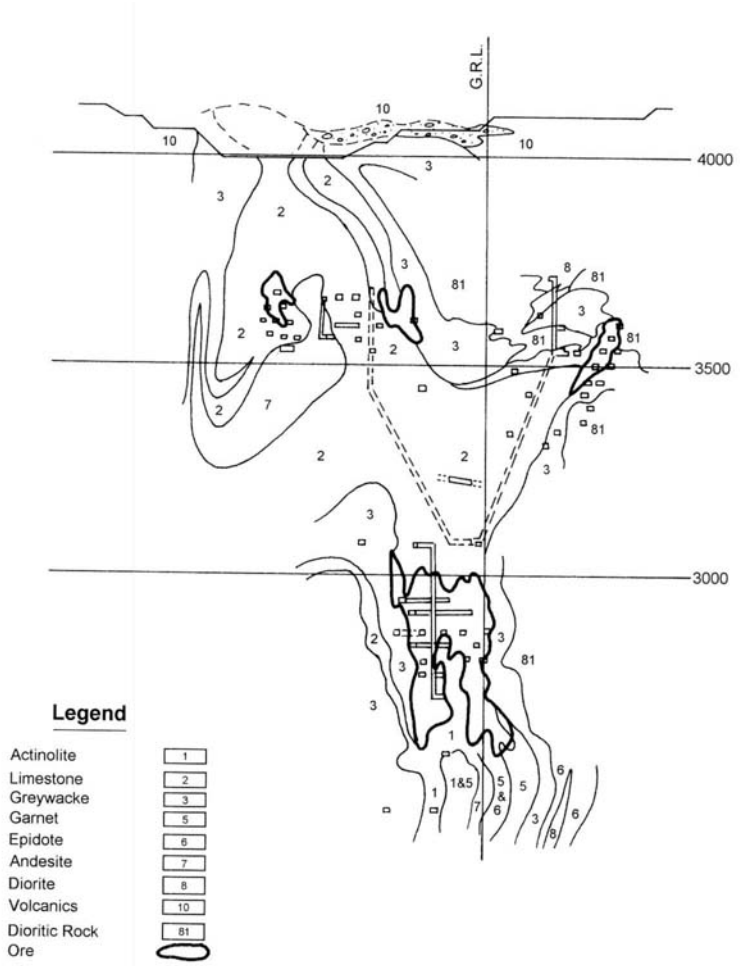


FIGURE 37.2a Section 7615 (original scale 1 in. = 300 ft)

Clayey fault gouge is present in most of the faults; gouge zones may be up to 6 or 9 m (20 or 30 ft) wide. The main ground problems are associated with local weakness rather than pressure.

Shape of Orebodies

Figures 37.2a and b and 37.3a and b show the Craigmont Mines orebodies. The main No. 1 orebody is approximately 244 m (800 ft) long and 46 m (150 ft) wide. It extends vertically from the original top of the open pit at 4200 elevation to just below the 3060 level.

The No. 2 orebody is approximately 304 m (1000 ft) long, varies from stringer width at the extremities up to 79 m (260 ft) wide, and extends from 3060 level to 2400 level. Both these orebodies have extensions resulting in additional small irregular bodies.

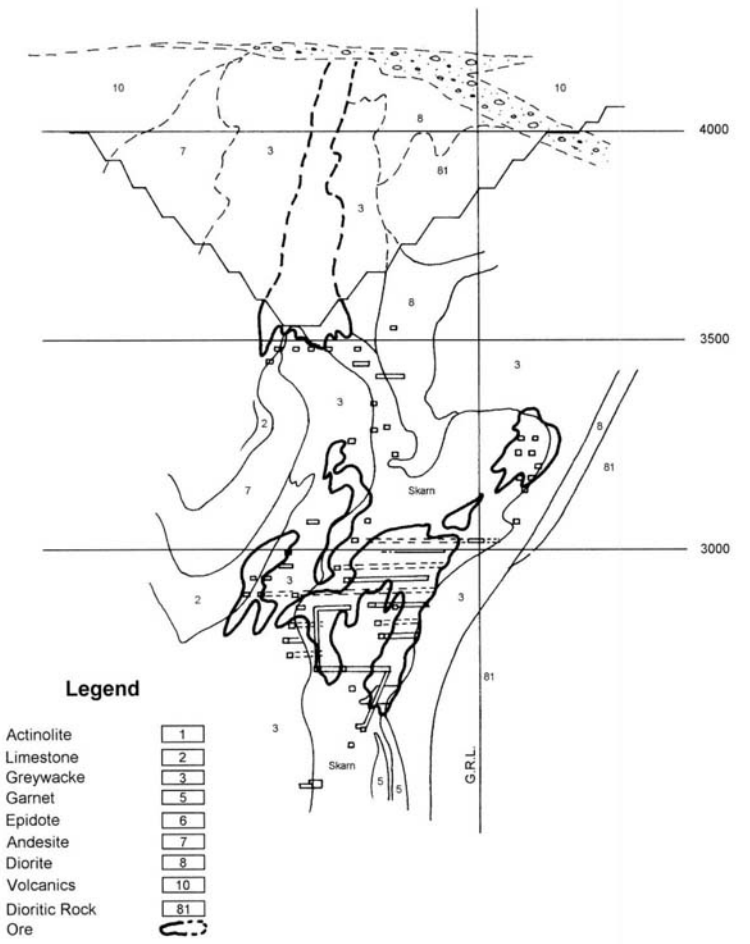


FIGURE 37.2b Section 8215 (original scale 1 in. = 300 ft)

Orebodies are mostly steep dipping, although part of the Wing orebody, an extension of No. 2 orebody, dips at 0.87 rad (50°). This orebody varies in size, but is approximately 122 m (400 ft) long, 21 m (70 ft) wide, and about 213 m (700 ft) high.

No. 1 Limb orebody is a narrow extension of the No. 1 Main with a vertical extent of 137 m (450 ft), average width of 18 m (60 ft), a strike length of 152 m (500 ft), and dips steeply at 1.4 rad (80°).

No. 1 East is an eastern extension of the No. 1 Main with a vertical extent of 183 m (600 ft), a strike length of 91 m (300 ft), an average width of 30 m (100 ft), and dips at 1.2 to 1.4 rad (70 to 80°).

No. 1 South is at the upper west end of the open pit with a vertical extent of 76 m (250 ft), a strike length of 183 m (600 ft), an average width of 24 m (80 ft), and near vertical dip.

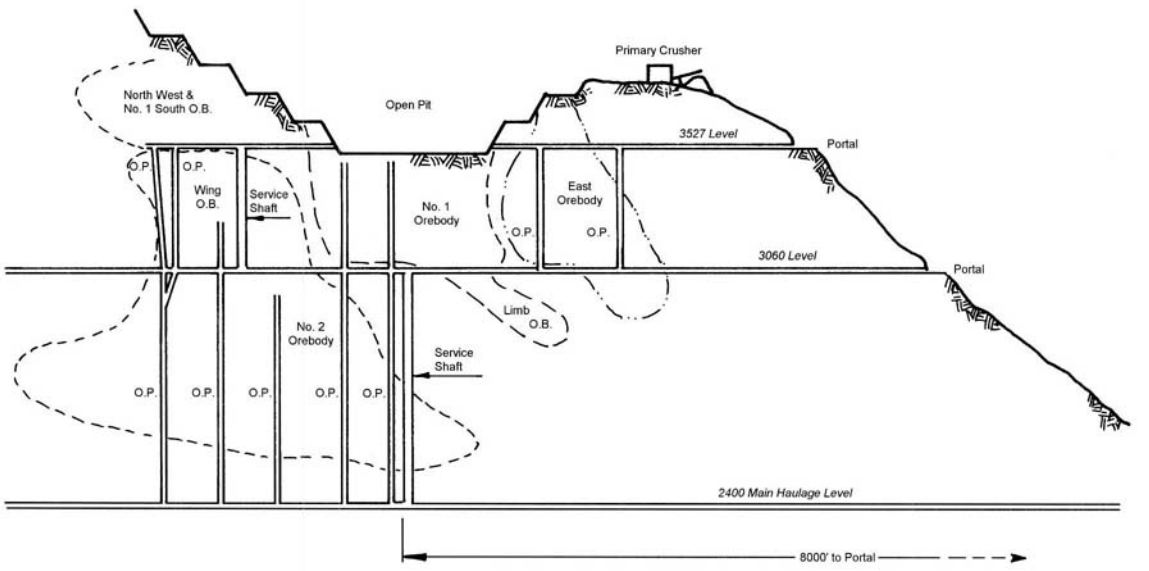


FIGURE 37.3a Schematic section (looking north) of the Craigmont mine, metric equivalent: ft × 0.3048 = m

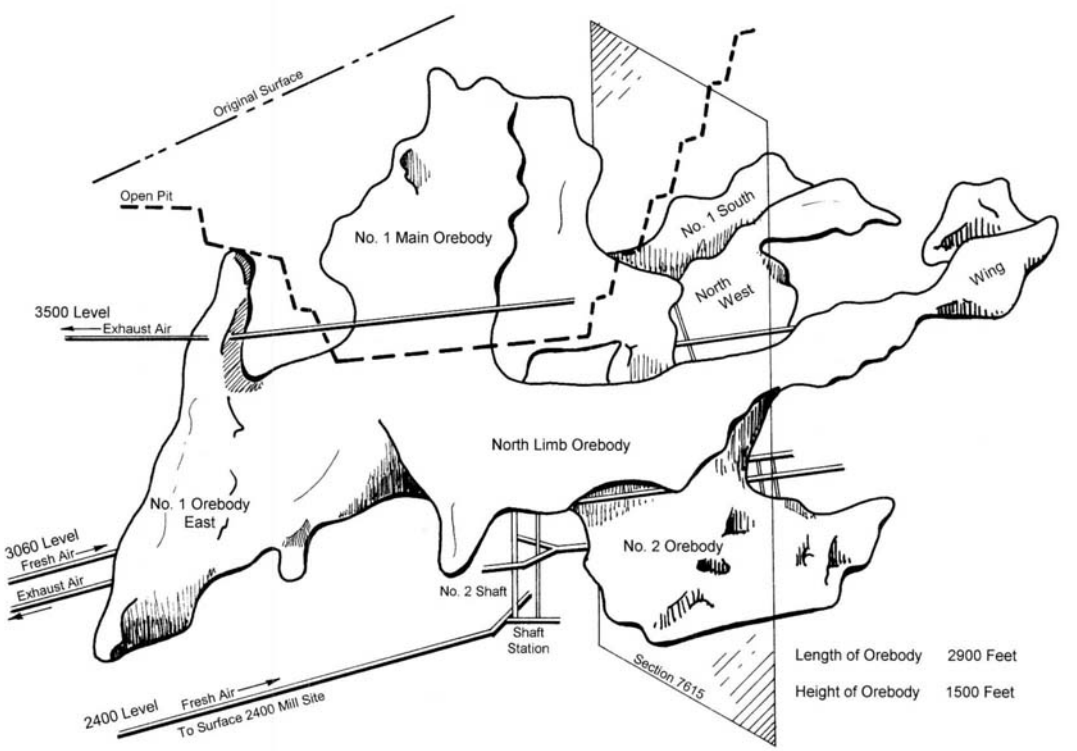


FIGURE 37.3b Model of Craigmont orebody (looking southwest), metric equivalent: ft × 0.3048 = m

The North West orebody is erratic with a vertical extent of 152 m (500 ft), a strike length of 107 m (350 ft), an average width of 15 m (50 ft), and dips at about 1.2 rad (70°).

MINING PRACTICE UP TO 1967

Open Pit Mining

The pit produced ore from March 1961 to March 1967—7,399,906 t (8,157,000 st) at 1.77% Cu and 5,236,270 t (5,772,000 st) at 0.60% Cu. Total material moved including waste and overburden was 79,054,796 t (87,143,000 st), which gave a waste-to-ore ratio of 6.3 to 1. When open pit mining ceased, several million tons of low-grade ore remained on surface stockpiles, permitting continuous operation of the concentrator until the underground mine reached design capacity.

Underground Mining

While the open pit was operating, exploration and development work continued underground. In anticipation of ground problems, it was decided to commence underground work before completion of the pit, partly to gain information on the orebody, and partly to supplement mill feed with higher-grade ore. No mining could be done near the open pit, and no method could be used that permitted subsidence in the pit.

Two variations of blasthole stoping were tried: benching from sublevels with vertical rings and horizontal ring drilling from corner raises; in both instances, the stopes were backfilled with pit waste.

Transverse cut-and-fill mining was used in the No. 2 orebody and in the North Limb orebody with varying degrees of success. Even though stoping width was limited to only 8 m (25 ft), ground support was a major problem.

About 763,850 t (842,000 st) were mined underground by these methods. In the fall of 1965 the decision was made to adopt the sublevel caving system of mining as soon as possible after the cessation of open pit mining.

REASONS FOR THE CHOICE OF SUBLEVEL CAVING

Experience from both open pit and underground extraction gave engineers and operators valuable knowledge of the physical characteristics of the Craigmont ore and the adjacent ground. These characteristics and the probable extraction effects on adjacent ground were the major factors in determining the underground mining method to be used. As previously mentioned, variations of blasthole and cut-and-fill stoping were tried in initial underground operations with limited success. Mining efficiency was low, and costs were high. Some stoping areas gave excessive dilution and sometimes hazardous conditions due to the weak character of both ore and wall rock. Due to the physical weakness of part of the ore deposit and the lack of competent ground

adjacent to the ore, it was logical to turn toward caving methods of ore extraction where the tendency of ground to cave could be utilized instead of resisted.

The Craigmont orebody is relatively small and contains many irregular, narrow ore shoots surrounded by weak, readily cavable waste rock. Because of the small size and irregular shape of most of the Craigmont ore deposit, any type of a block or panel caving system would lack the selectivity required to obtain satisfactory ore recovery without excessive dilution. In addition, high development costs plus poor fragmentation from the harder, tougher parts of the ore deposit ruled out extraction by this type of cave mining.

On examination, a sublevel caving method of mining offered certain advantages for the extraction of the Craigmont ores:

1. It allowed selective mining of a small, irregular orebody containing ore of both weak and strong physical characteristics, adjacent to extremely weak, fractured, cavable country rock on all sides. It is quite flexible and offers variations that enable it to be adapted to small irregular ore lenses or large massive deposits.
2. It was a method that took advantage of the caving tendency of the wall rocks. Resulting surface subsidence would be acceptable because open pit operations would be terminated by the time underground mining operations were started.
3. The method did not require large stope openings but only extraction openings of drift width through the ore and adjacent wall rocks, the size of which would be a compromise between ground stability and productivity of the equipment used.
4. Sublevel caving allows the continuous mining of an ore deposit of high vertical dimension, without tying up a substantial proportion of its ore in pillars. Sublevel caving does not require pillars. Mining starts at the top of the orebody and progresses down, consuming the development openings as it progresses. The method does not require perpetual retention of development openings once the ore on that horizon has been mined.
5. Sublevel caving allows relatively efficient underground production and is well adapted for mechanization with highly productive equipment, limited in size only by the dimensions of the extraction drifts. It allows the use of equipment similar to that used in efficient room-and-pillar methods of mining.
6. Mining costs and efficiencies, although not as good as block caving, large-scale room-and-pillar, or blasthole operations, are superior to other methods of extraction where there is a problem in ground support.

The major disadvantage of sublevel caving is the high dilution inherent in the method, and this is the price paid for using it. The percentage of ore recovery depends on the amount of dilution tolerated in the extraction process. The higher the dilution allowed, the better the ore recovery; these factors in turn are dependent on the cutoff grade, that is, the predetermined grade at which ore draw is to cease. Higher-grade ores can stand more dilution in approaching the draw cutoff grade and hence will obtain a better recovery of the

minable ore. When you consider that no major pillars of ore have to be left in an ore deposit extracted by this method, a total ore recovery of 90% of minable reserves compares very favorably to the recovery achieved by the best of other mining methods.

In Craigmont's case, sublevel caving was the most economic method of ore extraction per pound of copper produced. Due to adverse ground conditions it was the only method that could be used at all, without resorting to a type of square-set or undercut-and-fill method of mining with their high operating costs and low productivity.

PLANNING FOR SUBLEVEL CAVING

In the fall of 1965, it was decided to prepare the mine for sublevel caving. Swedish methods had been studied that summer, and mine planning started.

The first major decision was choosing between small low-productivity air-powered equipment in small headings or large high-productivity diesel-powered equipment in large drifts.

Because of poor ground conditions, it was originally planned to keep drifts narrow [about 2.4 by 2.7 m (8 by 9 ft) high] and use air-powered autoloaders and air shuttle cars delivering to closely spaced orepasses. This was the Swedish practice in mines of similar size with comparable ground conditions.

There was concern over the following points:

1. Because of short haul range, many orepasses at close intervals would be required. Raw orepasses at Craigmont had given a very short life; after passing about 181,437 t (200,000 st) they would cave out, and have to be abandoned.
2. A good many headings would be required to obtain the initial tonnage target of 2722 t/d (3000 stpd).
3. Opportunities for a high degree of mechanization were limited.

In March 1965, experiments had begun with the use of shotcrete as a means of ground support in lieu of timber. Preliminary tests indicated that shotcrete would adequately support a 3.6 × 3.9-m (12 × 13-ft) heading, considered to be the minimum size for use of high-productive diesel loaders and 914-mm (36-in.) ventilation ducts.

Diesel-powered equipment has a greater haul range and would require fewer orepasses than short-haul equipment. However, the orepasses would require a sturdy lining to stand up to the greater tonnage. In spite of this, economic studies favored the larger, more productive diesel equipment, where the orebodies were readily accessible to a ramp system, service shaft, ore and waste passes. At first it was decided to use diesel equipment only in the larger and more competent Main orebody, and small air equipment in the narrow orebodies; finally, in the interests of uniformity, it was felt that diesels should be used throughout. Some of the relative pros and cons are given in Table 37.1.

TABLE 37.1 Comparison between diesel equipment and small air equipment

	Diesel, large units	Air, small units
Productivity	High	Low
Automation	High	Low
Maintenance costs	High	Low
Number of orepasses required	Minimum	Maximum
Ground control (drifting and production brows)	Maximum required	Minimum required
Ventilation	Maximum required	Minimum required

To sum up, although it was known that ground control and ventilation would be a problem with the larger diesel equipment, the reduction in orepasses, the higher potential productivity, and the proposed use of shotcrete for ground support favored the selection of the highly productive diesel equipment. Smaller ore blocks requiring considerable vertical access with short hauls to orepasses were to be mined with the smaller air equipment. While this planning was proceeding, the blasthole and cut-and-fill stopes were phased out.

Parameters

Scale-model tests were carried out using crushed ore and multiple drifts to determine optimum development patterns. It was understood that model tests could only give indications and could not be relied upon for statistical results. As a result of the tests and guided by the judgment and experience of the operators in regard to ore fragmentation, recovery, dilution, and ground support, and consideration of practices in Sweden and other parts of the world, it was decided to use: (1) vertical heights of 9 m (31 ft) floor to floor between sublevels; (2) production drifts 4 m (13 ft) wide by 3.2 m (10 ½ ft) high; (3) production drifts on 11-m (37.ft) horizontal centers to give 8-m (25-ft) pillars; and (4) 1.29 rad (74°) side holes.

The 9-m (31-ft) level interval was changed slightly to conform with existing workings. The original choice for pillar width was 6 m (20 ft), but was modified to 8 m (25 ft) because of ground conditions (see Figs. 37.4 and 37.5). At this time, Swedish practice was increasing the sublevel interval to 12 or 14 m (40 or 45 ft). At Craigmont, difficulty with fragmentation was anticipated at the greater heights. Swedish practice has since returned to about 10 m (33 ft). Good fragmentation and an even ore flow when loading in the extraction drifts are of vital importance to optimize recoveries and keep dilution to minimum levels.

Drift Layouts

It was tempting to place as much of the workings in ore as possible because this is in somewhat better ground, reduces development, and minimizes costly waste handling. This practice was not uncommon in some mines studied. However, in incompetent ground, ore losses would result at drift

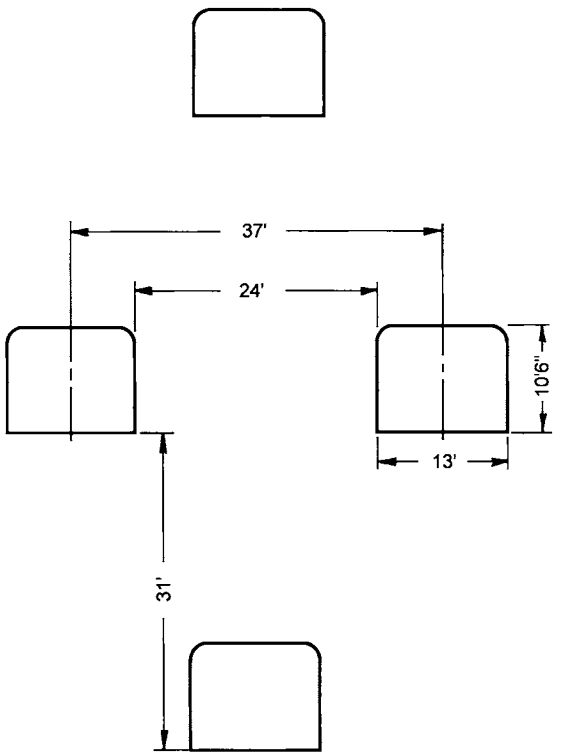


FIGURE 37.4 Drift spacing for transverse sublevel caving, metric equivalents: ft × 0.3048 = m; in. × 25.4 = mm

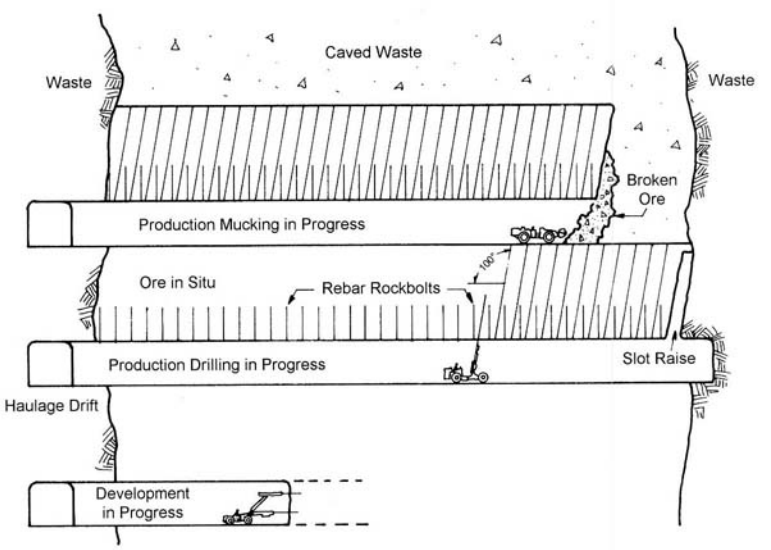


FIGURE 37.5 Transverse sublevel caving

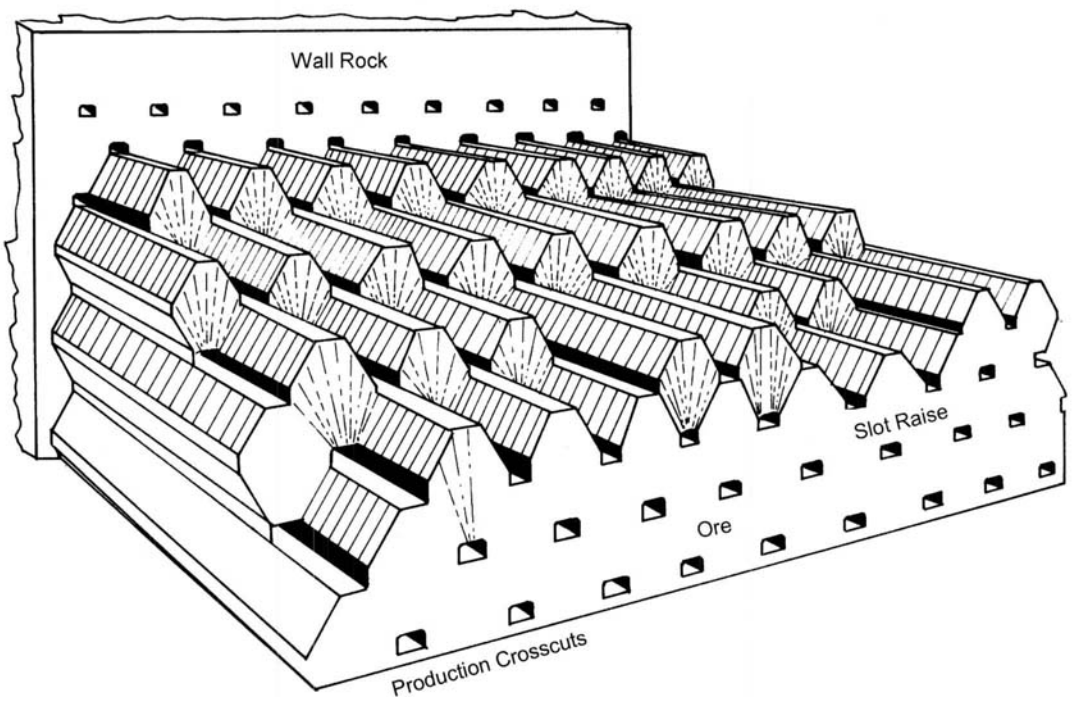


FIGURE 37.6 Perspective view of transverse sublevel caving with waste-cover removed (original scale: 1 in. = 45 ft)

turnouts where ground conditions would deteriorate as mining approached, and it would be necessary to blast multiple rings together. In these cases, ore recovery would be very poor.

In the wider sections (i.e., No. 1 Main orebody), a transverse system of extraction openings was planned (Fig. 37.6), whereas in the narrow orebodies it was necessary to use longitudinal layouts. In this latter case, the location of the drifts is critical, and much predevelopment drilling is required to properly define ore boundaries in order to locate extraction drifts correctly. All levels were to be interconnected by ramps at a maximum grade of 20% for easy movement of supplies, equipment, and personnel.

Orepasses

Previous experience with raw orepasses had shown that even in the best Craigmont ground, they cave out after about 181,437 t (200,000 st) has been passed.

It was decided to provide two armor-plated orepasses with bins above the chute for the Main orebody from 2400 haulage level to 3432 level, which would handle about 4,535,924 t (5,000,000 st), and another for the westerly orebodies from 2400 to 3060 level to handle about 1,814,369 t (2,000,000 st). The top portion of the westerly orepass could be driven raw because it was largely in competent limestone.

All tramming would be on 2400 level using two 23-t (25-st) trolley locomotives with twelve 7-m^3 (256-cu ft) Granby cars per train.

Choice of Equipment

Load-Haul-Dump Units. Scooptrams, Scoopmobiles, and transloaders were considered. The bottom dump feature of the transloader would have meant a rather complicated and costly orepass dump design. Headroom and operating characteristics were factors in choosing the Scooptram. The ST4 was chosen in preference to the ST5 because of much lower ventilation requirement without serious reduction in productivity.

Drilling Equipment. For development, Gardner-Denver Universal jumbos with DH 123 drills were chosen. For blastholes, Atlas Copco Simba 26s with BBC 120 drills and Gardner-Denver fan drills with DH 123 drills were chosen.

Service Vehicles. Service vehicles should be compact to fit onto a 2.4 × 3.6-m (8 × 12-ft) cage, diesel-powered, sturdy with good gradability under load, and must have a minimum load capacity of 0.9 t (1 st). Mercedes Benz Unimogs and Getman Scootcretes were chosen.

Shotcrete Equipment. Model H True Gun-all machines were chosen because of compact size, simplicity, portability, wet-mix machine, good maintenance costs, and relative high productivity.

Repair Shop

The success of the mining method would depend to a very great degree on the availability and condition of the equipment. It was therefore decided to build a fully equipped repair shop underground close to the mining area in order to perform all work on equipment, except major engine overhauls.

Ventilation

The 2400 haulage level and 3060 incast adit were to be used to bring fresh air into the mine. The 3060 outcast adit and 3500 outcast adit would be used for exhaust. Auxiliary ventilation would be a pressure system using 914-mm (36-in.) steel duct along the main entrance and 762-mm (30-in.) neolon duct in extraction drifts.

CRAIGMONT PRACTICE

Equipment

Table 37.2 lists the major items of equipment in use underground at Craigmont.

Layouts

Sublevels are 9 m (31 ft) apart vertically, and are joined by a 15% ramp system; every third sublevel is connected to the service shaft. Extraction drifts are driven at a 3% grade to ensure good drainage.

TABLE 37.2 Equipment used underground at Craigmont

9	ST5A Wagner Scooptrams with V8 Deutz engines
1	3-boom Gardner-Denver development jumbo (44-hp* engine, DH 123 rock drills)
1	2-boom Gardner-Denver development jumbo (44-hp engine, DH 123 rock drills)
3	Gardner-Denver fan drills with 2 PR 123 drills
2	single-boom John Deere "Skidder" truck conversions (DH 123 drill used mainly to drill rebar holes and test holes; LAL93 drill used mainly for hole cleaning)
2	Model H True Gunall trailers
2	Galion Graders—Model 503A with Deutz engine.
11	service trucks with John Deere "Skidder" conversions (Deutz and Mercedes engines)
2	25-ton* trolley locomotives (1 Goodman, 1 Clayton)
32	256-cu ft* Granby cars
Main Vent Fans*	
3	72-in. Sheldon Vaneaxial fans with 430-, 450-, 500-hp motors
1	72-in. Jeffrey Aerodyne fan with 150-hp motor
53	auxiliary ventilation fans ranging from 17½ to 75 hp

*Metric equivalents: st × 0.9071847 = t; cu ft × 0.0283168 = m^3; in. × 25.4 = mm; hp × 0.746 = kW.

Initially, production drifts were 3.6 m (12 ft), wide on 11-m (37-ft) centers, giving 8-m (25-ft) pillars. These are 4 m (13 ft) wide, still on 11-m (37-ft) centers, giving 3-m (24-ft) pillars. In a few areas where the ground is stronger, pillars have been reduced to 6 m (20 ft).

Development

Ramps are driven on a maximum grade of 20%, with 15% favored particularly on curves. A flat spot is provided at level horizons for turnouts. Maximum radius of curvature on ramps is 12 m (40 ft) inside. Haulage drifts and production drifts are driven at a +3% grade to provide adequate drainage. Safety cutouts are established at 30-m (100-ft) intervals along all travelways.

Waste development is 3.6 m (12 ft) wide in haulage drifts and ramps. Ramps 3.6 to 3.9 m (12 to 13 ft) high and haulage drifts 3.9 to 4.2 m (13 to 14 ft) high require 762- or 914-mm (30- or 36-in.) metal ducts. Production drifts are 4 m (13 ft) wide and 3.5 m (11½ ft) high in transverse mining, but could be up to 4.2 m (14 ft) high in longitudinal mining. Development work in waste is arched, but production drifts in ore are flat backed to give the best draw characteristics.

Rounds are drilled off with Gardner-Denver jumbos using DH 123 machines, 3353 × 31.7-mm (132 × 1¼-in.) round steel, and 47.6-mm (1⅞-in.) bits. Original machines mounted three booms, but it was found that two machines were sufficient in the relatively soft ground, so a new jumbo was made up using the center booms.

All rounds were originally blasted with stick powder. Practice during most of the years of sublevel caving was to load Amex over electric caps in the cut and all holes except for 31.7-mm (1¼-in.) 75% Forcite in the lifters, and Xactex [15.8-mm (⅝-in.) stick powder] in the perimeter holes. Current practice, because of the small amount of development, has reverted to blasting with stick powder. Perimeter holes are drilled no more than 610 mm (24 in.) apart, and blasting is light to avoid shattering the back and walls.

Transverse Layouts. Standard practice for transverse layouts (see Fig. 37.6) is to locate a haulage drift 9 m (30 ft) outside the ore zone where the ore is 13.7 m (45 ft) or more wide. Production drifts are then oriented across the orebody at approximately 1.9 rad (110°) to the haulage drift.

The main advantage is the production flexibility obtained through a large number of headings available per level. The main disadvantage is the increased cost associated with additional development, essentially in waste, and additional slots for each additional production drift.

Longitudinal Layouts. Standard practice, whenever the ore zone is less than 13.7 m (45 ft) wide, is to drive along the strike of the orebody with one or more drifts. Depending on the width and dip of the ore zone, the level interval could vary from 9 to 18 m (30 to 60 ft).

The main advantages of this system are the lower development cost and reduced number of slots required. The main disadvantages are the reduced flexibility of a lesser number of production drifts and the lead time necessary to begin production if access onto the level is from one end of the orebody.

Slot Drift. On occasion in a transverse mining situation, a production drift driven from the haulage drift (located in waste outside the hanging wall) through the orebody to the footwall is then turned and advanced longitudinally along the footwall. While the slot drift is being advanced along the footwall, the other transverse production drifts are being driven across the orebody until they are within 3 m (10 ft) of the slot drift (see Fig. 37.7). Upon completion of the slot drift development, production proceeds as normal from a slot at the face.

After production in the slot drift has retreated sufficiently, the transverse drifts are successively brought into production by blasting the 3-m (10-ft) breakthrough rounds. Some difficulty is experienced in getting the cave to move initially, so that considerable bombing is required for the first four to six production ring blasts.

The main consideration in using slot drifts at Craigmont is the poor ground conditions prevailing along the footwall of the orebody. With transverse mining, and the haulage located at the hanging wall, it is then necessary to drive the production drift slots to +15-m (50-ft) height (two levels) in poor ground at the footwall. If the slots are located back from the footwall in better ground, considerable ore (at better than average grade) is lost. Even though the slot drift is driven in poor ground along the footwall, the drift can be readily supported with timber sets and rebars. Instead of many +15-m (50-ft)

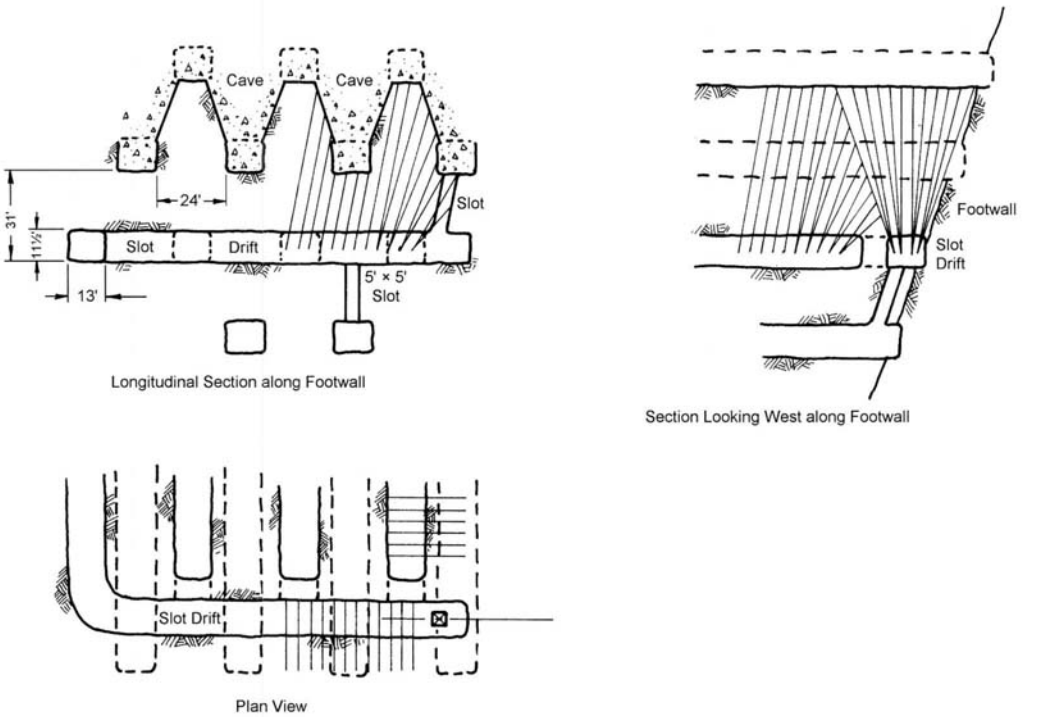


FIGURE 37.7 Slot drift (original scale: 1 in. = 40 ft), metric equivalent: ft × 0.3048 = m

slots, only one relatively short 6.7-m (22-ft) slot is required for the slot drift, and on the next level down, the transverse production drifts will require only 6.7-m (22-ft) slots breaking into the slot drift above. The next slot drift would then be driven two levels below the first slot drift and so on. The mining of these slot drifts, even though timber is required along the entire drift, has turned out very well, with good recoveries of high-grade ore that would otherwise have been lost.

Production Drilling

The original drills used were two Simba 26 air motor traction jumbos purchased from Atlas Copco. Early performance in the confined area of No. 1 orebody was good; however, the need for highly maneuverable equipment as well as the high cost of maintenance has ended their usefulness.

Three Gardner-Denver drill assemblies with two PR123 machines each are mounted on John Deere timber skidders. These skidders are a center-articulated rugged piece of equipment, designed for hard service in the lumbering industry.

Original practice was to drill off as much of a level as possible before starting production, but on many occasions, holes have caved and had to be redrilled or cleaned. A single telescopic boom jumbo with a 93 LAR Gardner-Denver

machine mounted on a John Deere skidder is used for this purpose. A second similar unit with a DH123 Gardner-Denver machine is used for drilling rebar holes, test holes, and some production drilling.

Tungsten carbide cross-bits were originally 47.6 mm (1⅞ in.) in diameter, with 31.75-mm (1¼-in.) round steel of 1.2-m (4-ft) length on Copco jumbos and 1.5-m (5-ft) length on Denvers. Couplings are Copco. Shattered ground around the walls of the open pit where sublevel caving was first applied required an increase to 57.15 mm (2¼ in.). Subsequently, these were reduced back to 47.6 mm (1⅞ in.) and returned to 51 mm (2 in.), which is standard except for 57.15 mm (2¼ in.) in shattered ground. Bits are supplied with the center hole plugged and two flushing holes forward and two back. This arrangement has done much to reduce mudding in the hole with the attendant excessive loss of rods.

The three production jumbos all use 31.75-mm (1¼-in.) round steel in 1.5-m (5-ft) lengths. The hole-cleaning machine uses 47.6-mm (1⅞-in.) bits and carbonized steel rods 1524 by 22 mm (60 by ⅞ in.). The rebar and test-hole machine uses 1.2-m (4-ft) round steel rods, and 47.6-mm (1⅞-in.) bits when drilling rebars and 51-mm (2-in.) bits for test holes and rings.

Various fan inclinations ranging from 1.13 rad (65°) forward toward the cave, through vertical, to 1.4 rad (80°) backward have been tried with 1.4 rad (80°) toward the cave being standard. Because the initial level mined in an ore zone tends to void, the practice is to incline the production drill fans forward at 1.2 rad (70°) in the expectation that a better break at the toe of the holes will occur.

Fan burdens have varied from 1.06 to 1.5 m (3½ to 5 ft) with blasts of one or two fans at a time. Much of the No. 1 Main orebody was drilled at 1.06-m (3½-ft) burdens and two fans blasted together (Fig. 37.8a). Mining practice is to drill fans with 1.2-m (4-ft) burdens and then blast the fans one at a time (Fig. 37.8b). Also, the short side holes in Fig. 37.8a, designed to open up the throat, have been discontinued with the increase from 3.6 to 4 m (12 to 13 ft) in the width of the production drifts.

Slotting

Several different layouts for slotting have been used:

1. Originally two short raises were driven with stopers up to each of the sublevels 9 m (30 ft) above. Rings were drilled up to the top sublevel kicking into the raises and into the cave above. After some initial success, the top section often failed to break, and the method was abandoned.
2. A fan method was tried in one area, with holes fanning up from the almost horizontal 1.2 rad (70°), slashing down into the drift. Trouble was experienced after about 9 m (30 ft) of height was attained; a considerable drift length was required to effect the slot, and ore was left that could not be drawn.

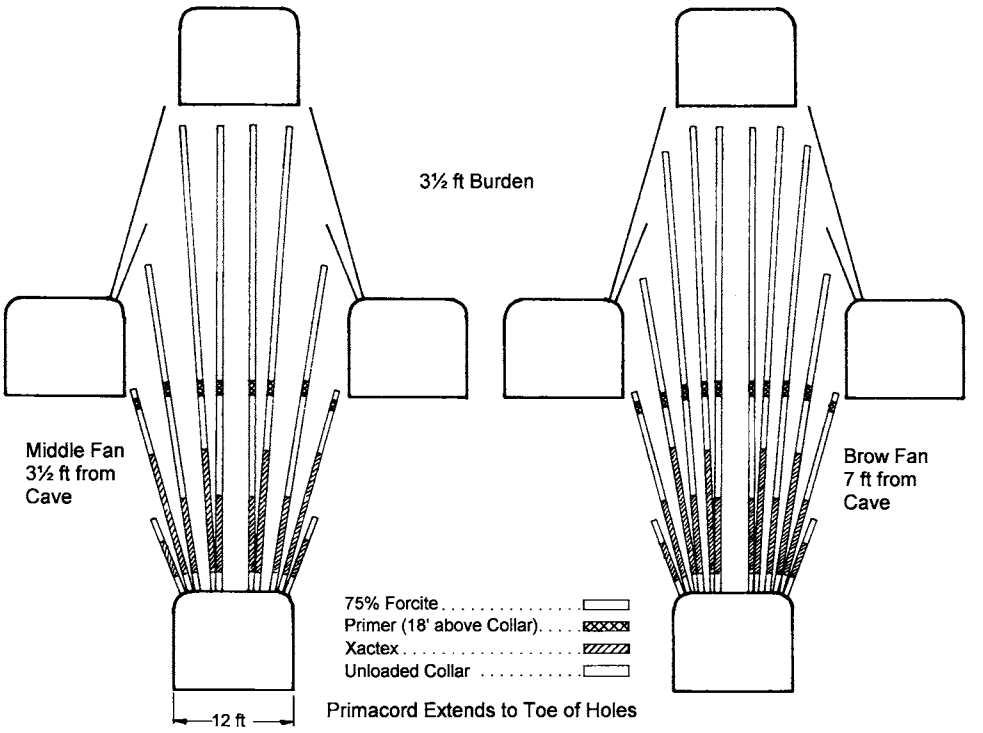


FIGURE 37.8a Early practice—two-fan blast (used in No. 1 Main orebody), metric equivalent: ft × 0.3048 = m

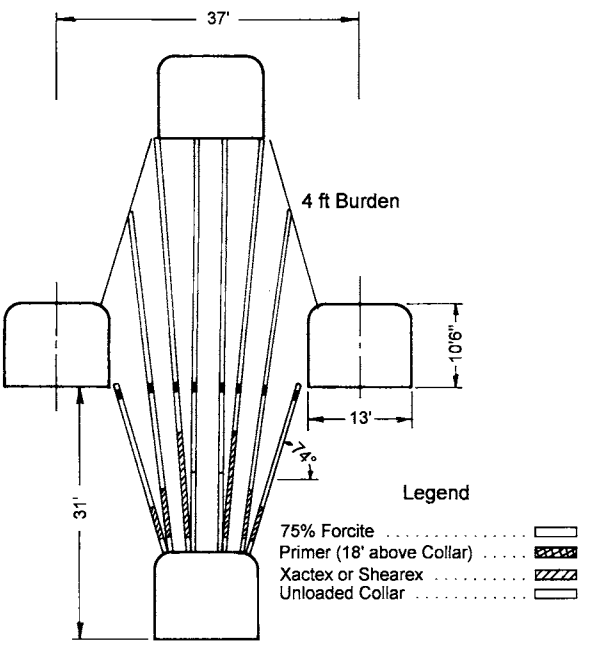


FIGURE 37.8b Current practice—single-fan blast, metric equivalents: 1 ft × 0.3048 = m; 1 in. × 25.4 = mm

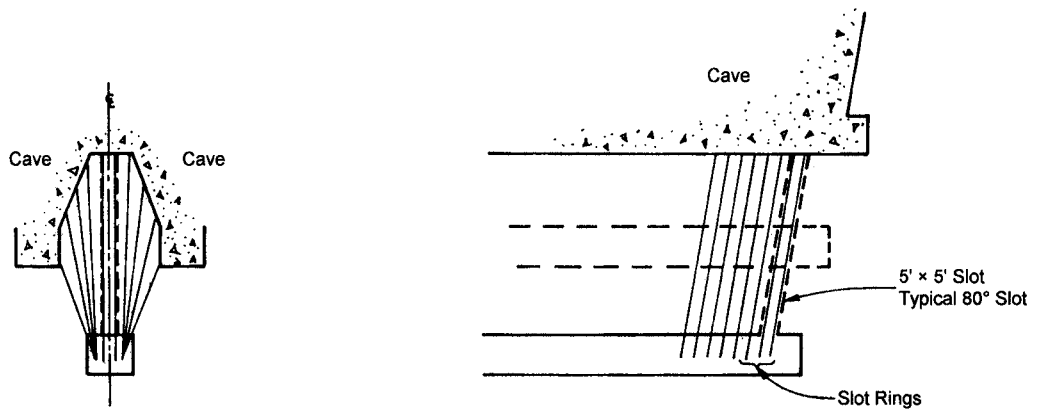


FIGURE 37.9a Early practice, metric equivalent: ft × 0.3048 = m

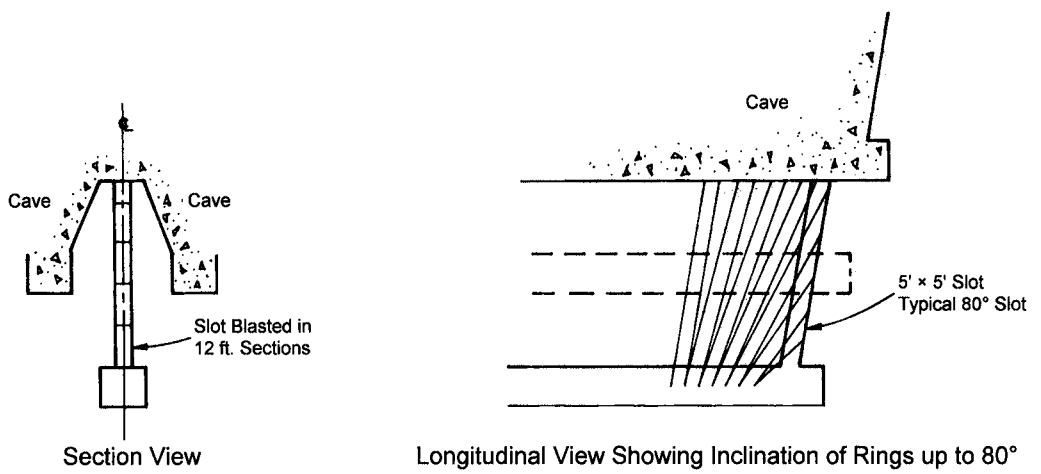


FIGURE 37.9b Current standard slot, metric equivalent: ft × 0.3048 = m

3. A 1.5 × 1.5-m (5 × 5-ft) slot is drilled by the production jumbos up the middle to the top subdrift [about 15 m (50 ft) high] and then blasted in sections up to 3.6 m (12 ft) at a time. Early practice (Fig. 37.9a) was to drill around and parallel to the slot to open it up to fan width. The last round of the slot was blasted with the fan holes.

Because of bridging problems with the breakthrough round and caved holes around the raise, this particular method has been modified. As shown in Fig. 37.9b, holes are fanned forward much like the fan method previously mentioned, except that the raise is used as an opening and holes can be steepened much faster. Also, the raise round is blasted through and mucked if necessary. Although some ore is lost near the slot because of the relatively flat fan holes, the method has proved satisfactory and is now standard.

TABLE 37.3 Cumulative powder factors to October 31, 1977

	Wing	No. 1 South North West	No. 1 Main and Limb	No. 1 East	No. 2 Main	Total Mine
lb per st* drawn	0.62	0.76	0.50	0.49	0.36	0.47
lb per st blasted	0.74	0.84	0.76	0.74	0.63	0.72

*Metric equivalents: lb × 0.4535924 = kg; st × 0.9071847 = t.

Blasting

Apart from the importance of good fragmentation, the protection of the brows requires close attention. Brow collapse was the first major problem encountered, resulting in recoveries of 45% and the difficulty of loading rings already buried in broken rock. Initially the brows were supported mechanically with timber sets.

Early blasting practice was to use ammonium nitrate–fuel oil (ANFO) except in wet holes and to bring the explosives fairly well down the hole to fragment the brow. To reduce the shattering effect on the brow, 6-m (20-ft) leads were used on millisecond delay blasting caps, and explosives were kept well up from the collar. The same Xactex used in development was used to gently blast near the collars. Because of the shattered ground, Primacord was used in all holes to ensure complete detonation. At least two rings were preloaded ahead of the blast to minimize the problems of flooding the rings with broken rock when the brow overbroke.

Later the use of ANFO in production blasting was abandoned for the following reasons:

1. It was necessary to preload, and the holes were not blasted right away.
2. There were problems with shattered ground. Sometimes far more ANFO could be loaded into the hole than was required.
3. There were many ineffective blasts.
4. Problems were encountered with water in the holes.

Standard practice (see Fig. 37.8b) is to load the greater portion of each blasthole with 31.75 × 406-mm (1¼ × 16-in.) cartridges of 75% Forcite and then near the collar with Shearex. Originally, 38.1-mm (1¼-in.) Forcite was used, but the smaller size gives satisfactory fragmentation and is easier to load in Craigmont's rough holes. The primer is located 5.4 m (18 ft) up the collar, and Primacord is used in all holes to ensure complete detonation. Sufficient preloaded fans (generally four) are maintained so that loading can be accomplished from scissors-deck service trucks, thereby eliminating the use of wooden stagings. See Table 37.3 for cumulative powder factors.

Load-Haul-Dump System

The LHD system needs no description because it is well-known. Productivity has increased from about 272 t (300 st) per miner-shift, over a round trip average distance of 549 m (1800 ft) in 1970, to 454 t (500 st) per miner-shift, over

396 m (1300 ft) in 1972, to 523 t (577 st) per miner-shift in 1977 over 444 m (1458 ft). All Scooptrams have been converted to ST5s with buckets enlarged from 5.4- to 6.8-t (6- to 7.5-st) capacity and engines from 78 to 116 kW (105 to 156 hp).

Good roads have contributed to increased productivity. At first, roads were far from good, partly due to the soft ground; the Scooptrams made holes at turnouts that would fill with water. The mine's own waste (or even ore) was too soft to use for ballast. Due to the narrow width of the drifts, it was not possible to maintain ditches on the side of the drifts, and drainage was a real problem. Drifts are driven at +3%. Crushed and screened diorite is brought in from the surface to use as ballast. This is oversize from the screening plant, which provides sand for shotcrete and aggregate for concrete. It is crushed in the secondary crusher and screened to 38.1 mm (1½ in.). This is distributed on the roadbed by Scooptrams and graded with Galion graders. A ditch is maintained in the center with the grader, and as long as the water is run off this way, the sides are packed hard with the Scooptram tires.

Holes 203 mm (8 in.) between sublevels are drilled for drainage with a GD133 mounted on an Air Trac. Engine life on Scooptrams is currently about 2000 hr with 5500 hr as a maximum.

Tires used are smooth (18.00 × 25) 24-ply Firestones and Bridgestones with Michelin tubes. Originally tires were recapped, with new tires lasting 980 hr and recaps 635 hr. That practice changed to using new tires only, as average tire life is now 1600 hr, and recapping provides no cost saving. Scooptram availability is approximately 75%.

ST5 Scooptram V8 engines are rated at 116 kW (156 hp) at 241 rad/s (2300 rpm), and 6.6 m^3/s (14,000 cfm) of fresh air must be provided. Oxycatalyst scrubbers on the equipment control all gases but the oxides of nitrogen, which must be flushed from the drift atmosphere by adequate ventilation. Third and fourth gears have been removed from the transmission.

Ground Control

Ground control has been a major problem in Craigmont's relatively poor ground. Originally, about 38.1 to 51 mm (1½ to 2 in.) of shotcrete were expected to adequately support the ground and this was supposed to replace timber. In point of fact, grouted rebar bolts, timber, and steel are used in various sections of the mine as well as shotcreting all the headings. Shotcrete is used in 100% of the headings; rebars on all production drifts and about half the waste headings; timber on 15% of the headings; and steel in special areas, usually turnouts or where headings have had to be driven through fault zones.

True-Gunall Model H machines are used to apply wet shotcrete as headings are driven—usually a round at a time—but in stronger ground several rounds may be taken before gunning. Wet shotcrete is used because of the problems of keeping the sand dry in underground storage raises. In one section of the mine, more accessible from surface, dry shotcreting has been used effectively

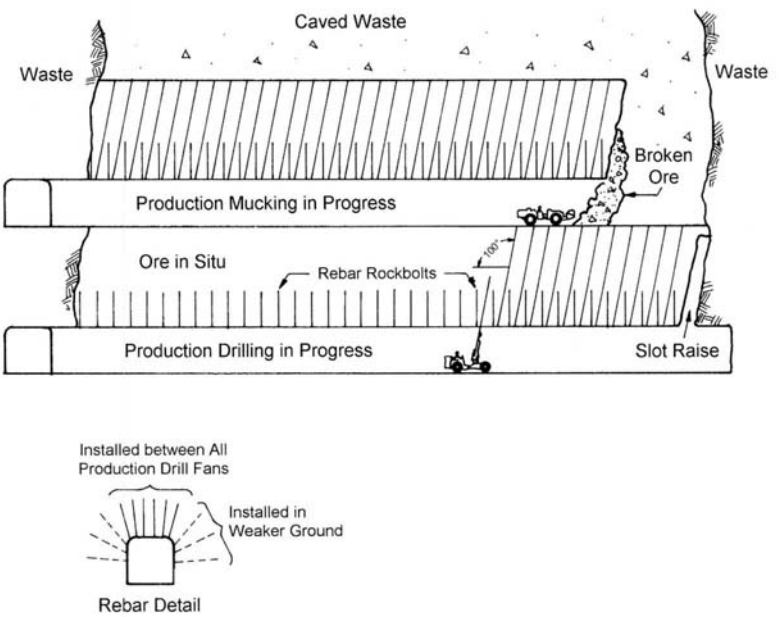


FIGURE 37.10 Transverse sublevel caving

with an Aliva machine and a batch plant feeding self-unloading cars. With +610 m (2000 ft), of development advance, an average of 535 m^3 (700 cu yd) is placed each month; 2 m^3 (3 cu yd) are blown per miner-shift. Approximately 22 rebars are installed per miner-shift (including drilling) in development headings where installation is usually on a round-by-round basis, and 35 rebars are drilled and installed per miner-shift in production drifts where large numbers are installed at a time.

Shotcrete is found to be very effective in weak ground where no movement or pressure is applied. The thin skin of concrete can seal and bear the weight of loose ground and hold it together. However, if there is any movement along fault planes or any appreciable pressure, the concrete is subjected to tension or more compression than it can stand. As headings are driven, supervisors judge if they need rebarring as well as shotcreting, or if it is necessary to go straight to timber. After the heading is driven, it is often necessary to do repair work, such as reshotcreting or adding rebars or timber. Sometimes, due to deterioration of the drift, all three are used; of course, the rebars would have been omitted if the final timber stage could have been predicted.

Rebars (see Fig. 37.10) are installed in a pattern of fanned 2.4-m (8-ft) deep holes and consist of 19.05-mm (¾-in.) reinforcing bars grouted into the holes using a 2:1 sand-to-cement mix placed with the True-Gunall machine. Rockbolts with expansion shells have too weak an anchorage in the broken ground. Two major factors in maintaining mine headings are time and mining sequence. As far as practical, the headings should be driven only just before they are required for production preparation. Mining sequence should be strictly adhered to in order to avoid throwing stresses on other areas still in operation.

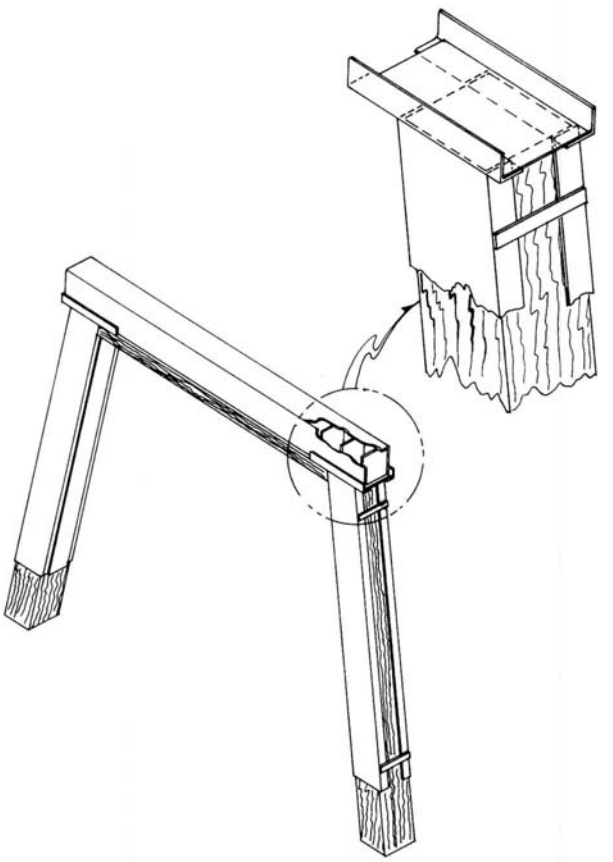


FIGURE 37.11 Steel set

Timber sets are constructed using double 254 × 254-mm (10 × 10-in.) fir caps standing on 254 × 254-mm (10 × 10-in.) posts. Posts are battered at 0.09 rad (5°) to help resist side pressure. Two complete sets are installed per three miner-shifts, using scissor-lift platforms on skidder chassis. Where space is available, cribs of 254 × 254-mm (10 × 10-in.) timber may be used.

Special steel sets (see Fig. 37.11) are used where timber cannot bear the loads. Caps consist of 254 × 254 × 11.4-mm (10 × 10 × 0.45-in.) wall hollow structural steel tubing stiffened near each end by 12.7-mm (0.5-in.) steel plates. Posts are steel clad 254 × 254-mm (10 × 10-in.) fir timbers. Two 254-mm (10-in.) channels are welded together with plate and the post driven through. The purpose here is to provide strength to resist side pressure. These sets, although expensive, have been very effective. Many have been salvaged and reused.

There are many mines with worse ground, where much more support is required; Craigmont's problem is associated with providing the minimum support to satisfy safety and draw control requirements. Most of Craigmont's workings cave shortly after abandoned.

Draw Control

When Craigmont mining first started, recovery was 45%, and the draw was out of control. The primary problem was the collapse of the brows after blasting. Numerous remedies were tried to control this failure, namely: (1) presplitting of first 3 m (10 ft) of fan; (2) use of Xactex near collars of fans; (3) variations in drill patterns; (4) expanding shell rockbolts between fans (including bolting timbers to the back); (5) rebar bolts installed with epoxy; (6) full-length caps supported on bullhorns and blocked to the back; and (7) rebar bolts installed with mortar grade concrete.

The full-length caps on bullhorns were effective only about 50% of the time. The best and most economical results were obtained by grouting a row of six 2.4-m by 19-mm (8-ft by $\frac{3}{4}$-in.) rebar bolts between each fan of blastholes.

Presplitting proved ineffective. Xactex or Shearex near the collars is used. Various fan inclinations burdens and drill patterns were tried, ranging from 1.08 rad (65°) to 1.57 rad (90°) toward the cave and from 1.06- to 1.5-m ($3\frac{1}{2}$- to 5-ft) burdens. Fans are inclined at 1.4 rad (80°) toward the cave and a burden of 1.2 m (4 ft) [1.5 m (5 ft), in one section of the mine] is used. Usually one ring is blasted at a time.

The use of brow blocks [254 × 254-mm (10 × 10-in.) timber caps supported by bullhorns] improved recoveries considerably and provided the opportunity to experiment with other methods. Rebar bolts installed with epoxy were successful, but the use of mortar was cheaper, easier to apply, and equally effective. Brows are not always perfectly square after blasting, but recoveries are acceptable.

The effect of brow collapse is as follows:

1. Ore floods the drift and covers the next row or rows of holes to be blasted. When this happens, the holes are dug out (a somewhat hazardous operation), if possible, or they are lost.
2. If the brow is uneven (i.e., not flat and horizontal), the ore funnels down through the high spot. This reduces the width of ore flow with consequent ore losses.
3. When the brow overbreaks, the ore floods farther back into the drift than was intended. This prevents the Scooptram from digging in its proper position relative to the broken ore. As a result, a thin draw of ore occurs, with waste being pulled down the brow without loading the full depth of ore blasted. In this case, single ring blasts give best results.
4. When the brow caves back a fair distance, or there are high spots from previous sloughing during the development drifting, and it is obvious that the brow will end up in a high spot if only one or two rings are blasted (thus causing ore loss), it may be necessary to blast two or more rings at a time to pass the high area. Recovery will be poor from such a blast, but records are kept to plan for overdraw on lower levels.

As well as trying to keep square brows, the following steps are taken to control draw:

1. Primary blasting is supervised to ensure good fragmentation for an even and continuous flow of ore, that is, ensuring that holes are properly cleaned and loaded to the toes and that proper blasting procedures are followed. Bridging also results from improper loading and blasting.
2. During the draw, hang-ups or partial hang-ups are blasted immediately to assure continuous flow.
3. Information is provided for supervisors and operators so that they may recognize the desired cutoff points. This is closely supervised.

One of the most important duties of the production shift bosses is to make the decision as to when ore recovery from any given blast has reached the optimum in terms of dilution and recovery.

1. Grade of ore in situ is determined by sampling the sludge produced from blasthole drilling; the samples are taken from the drift floor after drilling.
2. Tonnage blasted is computed from drill-hole layouts.
3. With this information, the ore-to-waste ratio is calculated for the appropriate cutoff. For example, assuming the waste dilution has no copper values, the ore in place is 2% Cu, and the desired cutoff is at 0.5% Cu, then mucking should stop when the muckpile appears to consist of 75% waste and 25% ore. The theoretical number of Scooptram buckets to achieve this cutoff is provided to the supervisor in advance. Distribution of copper values in Craigmont orebodies is very erratic, and wide fluctuations in grade are common; consequently, the theoretical number of buckets to achieve a given cutoff grade can vary widely from fan to fan.
4. If the supervisor finds after drawing the theoretically correct number of buckets that the pile still appears to be above cutoff grade, the draw will naturally continue. If the opposite condition occurs, that is, cutoff grade is reached substantially too early, an extra ten buckets will be drawn as waste in the hope of getting the ore to run again.
5. It is considered impractical to sample and assay to determine cutoff because it is difficult to get a reasonable sized representative sample of a muckpile usually consisting of fine waste and relatively coarse ore; three or four buckets can change the grade of the pile considerably, so that sampling would have to be done at very frequent intervals to be meaningful; and a great number of working places would be necessary to maintain production while awaiting assay results.

Orepasses

Orepasses have always been a major problem at Craigmont due to the fact that raw raises will cave out rapidly when used to pass rock. Raises are steel-lined and concreted with bins above the chute. Ore packs and hangs up if the raises are filled, so they are operated empty with a high maintenance cost. This subject is covered in detail by Cokayne (1969).

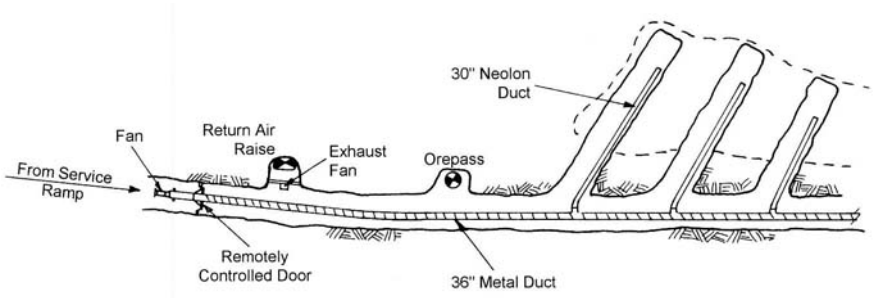


FIGURE 37.12 Typical sublevel auxiliary ventilation, metric equivalent: in. × 25.4 = mm

Ventilation

When sublevel caving was first started, approximately 71 m^3/s (150,000 cfm) of air was passed through the mine using one 298-kW (400 hp) 1854-mm (73-in.) axial flow in the exhaust drift. Auxiliary ventilation in the sublevels (Fig. 37.12) was provided by 24-kW (32-hp) fans with 762-mm (30-in.) flexible ducts. Due to air losses through the cave, increasing use of diesel equipment, and the raising of the standards of acceptable ventilation conditions, it was necessary to make major changes involving driving additional drifts and substantially increasing mechanical ventilation.

Primary ventilation passes 179 m^3/s (380,000 cfm) of fresh air using two adits as intakes. Each adit is equipped with a two-stage 1854-mm (73-in.) vane axial direct driven variable-pitch 373-kW (500-hp) 125.6-rad/s (1200-rpm) fan capable of delivering 90 m^3/s (190,000 cfm) at 342-mm (13.5-in.) SWG (standard water gage). Fans are installed in airlocks. Air is exhausted through two routes, one equipped with the original 298-kW (400-hp) fan, and the other with a 112-kW (150-hp) axial fan. Secondary ventilation is provided by a suitable power fan, or fans, through 914- or 762-mm (36- or 30-in.) metal duct in the haulage drift, and 762-mm (30-in.) flexible duct in the production drifts. On levels below the cave, fans are also installed in the exhaust raise to balance pressure on the level and minimize leakage through the cave (Press and Johnstone, 1976).

Maintenance

The vital operation of maintenance is carried out almost completely in an underground shop 85 m (280 ft) long, 6 m (20 ft) wide, and 4.8 m (16 ft) high. It is serviced by two 4.5-t (5-st) overhead cranes. Improvements in maintenance procedures have done much to improve the mine efficiency (Croteau, 1969).

Productivity

Originally, planned production from sublevel caving was anticipated to be on the order of 2722 t/d (3000 stpd). By late 1969, the early mining problems of poor fragmentation, poor brows, and bridged slots had largely been solved, and it was apparent that production in excess of 2722 t/d (3000 stpd) could be reached if the ventilation system was improved.

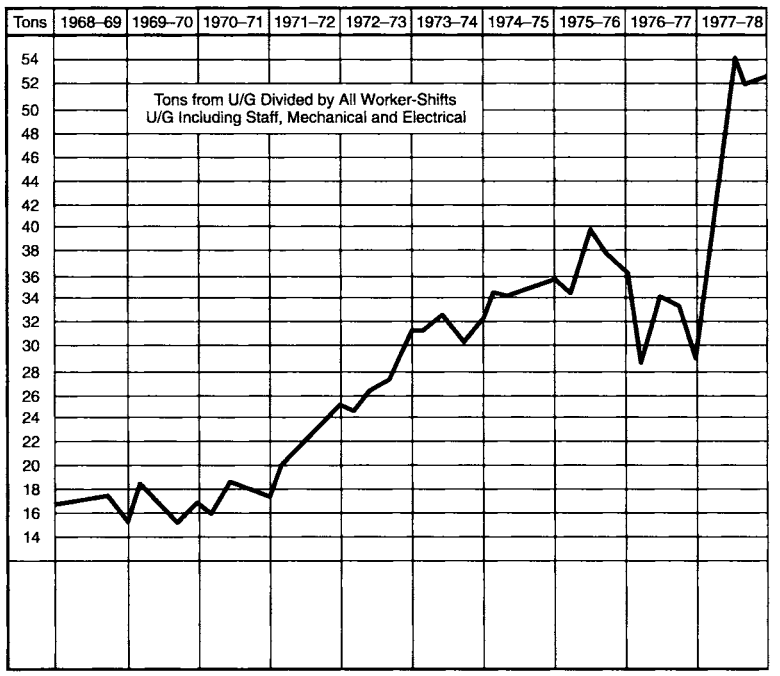


FIGURE 37.13 Underground tons per miner-shift, metric equivalent: st × 0.9071847 = t

By 1971, fresh-air capacity had been increased from 94 to 179 m^3/s (200,000 to 380,000 cfm). About the same time, a fully equipped underground maintenance shop had been centrally established at the 3000 level, together with ready access to a ramp system connecting all the active production areas. With these improvements, production in 1971 averaged 3629 t/d (4000 stpd).

In 1972, the addition of three lined orepasses with offset points and surge bins above the chute virtually eliminated the wear and hang-up problems that had been experienced to that time. Also, a number of ST5a Scooptrams with an average 6.8-t (7.5-st) bucket capacity were purchased. The existing ST4a Scooptrams [5.4-t (6-st) capacity] were then gradually converted to ST5a Scooptrams. These improvements plus further refinements in production drilling, improved brow support, better reserve mining grades, and increased numbers of production headings all contributed to bring production to 4264 t/d (4700 stpd) between 1972 and 1978. Best production was attained in 1975 and 1976 with an average 4899 and 4808 t/d (5400 and 5300 stpd), respectively, when the greatest number of production headings were available.

Productivity per miner-shift (see Fig. 37.13) increased from 14.5 t (16 st) to a recent high of 50 t (55 st). Tons per miner-shift had remained relatively constant until the latter part of 1971. With the improvements mentioned, productivity gradually increased until mid-1976. Then development footages gradually decreased from +732 m (2400 ft) per month without a similar decrease in crew requirements. However, by mid-1978 with development

below 152 m (500 ft) per month, the work force was allowed to decrease through attrition with a consequent increase in productivity above 45 t (50 st) per miner-shift.

ANALYSIS OF SUBLEVEL CAVING RESULTS

The following equations apply to the sublevel caving ratios contained in Tables 37.4 and 37.5:

$$\text{Extraction} = \frac{\text{tons drawn}}{\text{tons in place}} \times 100$$

$$\text{Recovery} = \frac{\text{copper units drawn}}{\text{copper units in place}} \times 100$$

$$\text{Dilution} = \frac{\text{grade in place} - \text{grade drawn}}{\text{grade in place}} \times 100.$$

Although the Craigmont orebodies are surrounded by a halo of mineralization, the grade of diluting material drawn into a muckpile is unknown and cannot be estimated with any degree of confidence. Therefore, dilution figures at Craigmont assume that the diluting material carries no copper values.

Table 37.4 is a summary by orebody of sublevel caving mining statistics to June 1978. Development ore includes all material above 0.4% Cu encountered during drifting and is therefore well below the production-in-place grade.

Production draw tonnages and grade are determined for each production drift from the adjusted mill statistics so that recovery and dilution figures for each orebody are essentially assigned values. Orepass tonnages are distributed between drifts using mucking reports as a guide. Copper units drawn are assigned to each production drift using the monthly extraction and the in-place grade as a guide in dividing up the total units. Therefore, there could be a built-in error in both recovery and dilution figures where an orepass serves more than one orebody. Combined figures for North West and No. 1 South and for No. 1 Main and Limb are therefore provided.

Table 37.5 compares the mining statistics of Table 37.4 to the geological mineral inventory (at 0.7% Cu). The in-place ratio of development plus production tons blasted as a percent of geological reserves indicates how much of the mineral reserve we actually attempted to mine. The draw ratio of development plus production tons drawn indicates how much of the geological inventory was actually mined.

North West Orebody

The North West orebody is narrow, dips at 1.57 to 1.22 rad (90 to 70°), and is mined longitudinally.

TABLE 37.4 Sublevel caving—summary to June 30, 1978

	Development			Production in Place			Production Drawn					
	Tons*	Grade	Units	Tons*	Grade	Units	Tons*	Grade	Units	Rec, %	Ext, %	Dil†, %
North West	51,389	1.27	65,080	345,516	1.44	496,365	397,961	1.02	407,219	82	115	29
No. 1 South	56,757	1.11	62,856	295,101	1.40	412,218	316,777	1.17	372,093	90	107	16
NW and No. 1 South	108,146	1.18	127,936	640,617	1.42	908,583	714,738	1.09	779,312	86	112	23
No. 1 East	81,716	1.25	102,051	458,653	1.63	746,396	658,211	1.16	760,710	102	144	29
Wing	167,365	1.28	214,052	989,198	1.92	1,899,127	1,179,106	1.31	1,539,134	81	119	32
No. 2 Orebody	780,114	1.83	1,429,817	3,305,285	2.65	8,762,829	6,006,816	1.43	8,583,301	98	182	46
No. 1 Main	558,462	1.45	807,010	2,697,692	2.16	5,837,330	4,365,827	1.29	5,652,818	97	162	40
No. 1 Limb	130,372	1.71	223,381	890,527	2.38	2,118,636	1,133,177	1.51	1,715,209	81	127	37
No. 1 Main/No. 1 Limb	688,834	1.50	1,030,391	3,588,219	2.22	7,955,966	5,499,004	1.34	7,368,027	93	153	40
Total mine	1,826,175	1.59	2,904,247	8,981,972	2.26	20,272,901	14,057,875	1.35	19,030,484	94	157	40

*Metric equivalent: st × 0.9071847 = t.

†Diluting material assumed to be barren of copper values.

TABLE 37.5 Comparison of sublevel caving to geological reserves at 0.7% Cu cutoff

	Geological Reserves*			Development and Production in Place Total			Development and Production Draw Total			Development and Production in Place as % of Geological Reserves		Development and Production Draw as % of Geological Reserves		
	Tons†	Grade	Units	Tons†	Grade	Units	Tons†	Grade	Units	Tons, %	Units, %	Extr, %	Rec, %	Dil‡, %
North West	798,612	1.44	1,152,984	396,905	1.41	561,445	449,350	1.05	472,299	50	49	56	41	27
No. 1 South	528,761	1.49	789,990	351,858	1.35	475,074	373,534	1.16	434,949	67	60	71	55	22
NW and No. 1 South	1,327,373	1.46	1,942,974	748,763	1.38	1,036,519	822,884	1.10	907,248	56	53	62	47	25
No. 1 East	817,974	1.42	1,158,867	540,369	1.57	848,447	739,927	1.17	862,761	66	73	90	74	18
Wing	1,443,604	1.66	2,391,234	1,156,563	1.83	2,113,179	1,346,471	1.30	1,753,186	80	88	93	73	22
No. 2 Orebody	4,423,741	2.11	9,336,462	4,085,399	2.49	10,192,646	6,786,930	1.48	10,013,118	92	109	153	107	30
No. 1 Main	3,442,101	1.96	6,742,255	3,256,154	2.04	6,644,340	4,924,289	1.31	6,459,828	95	99	143	96	33
No. 1 Limb	1,356,450	2.02	2,743,519	1,020,899	2.29	2,342,017	1,263,549	1.53	1,938,590	75	85	93	71	24
No. 1 Main No. 1 Limb	4,798,551	1.98	9,485,774	4,277,053	2.10	8,986,357	6,187,838	1.36	8,398,418	89	95	129	89	31
Total mine	12,811,243	1.90	24,315,311	10,808,147	2.14	23,177,148	15,884,050	1.38	21,934,731	84	95	124	90	27

*Geological reserves apply only to areas mined to June 30, 1978.

†Metric equivalent: st × 0.9071847 = t.

‡Diluting material assumed to be barren of copper values.

Table 37.4. Near vertical configuration that caved readily, surrounded by ±0.6% Cu diluting material resulted in fair recovery (82%) at low extraction (115%) and low dilution (29%).

Table 37.5. Low in-place grade and erratic mineralization resulted in 56% extraction and 41% recovery of the geological inventory at 27% dilution.

No. 1 South Orebody

The No. 1 South orebody is narrow, dips at 1.57 rad (90°), and is mined longitudinally.

Table 37.4. The orebody is overlain by a strong limestone cap that did not readily cave. The top three levels voided with the balance of the levels obtaining cave after breaking into the open pit. Other than the obvious safety hazard from the large void the results were good with recovery of 90% at very low extraction (107%) and dilution (16%).

Table 37.5. With low in-place grade and erratic mineralization, much of the geological reserve was not mined, resulting in only 71% extraction and 55% recovery.

No. 1 East Orebody

The No. 1 East orebody is medium in size [30 m (100 ft) width], dips at 1.22 to 1.40 rad (70 to 80°), and is mined transversely in two sections with 76-m (250-ft)-high blasthole stopes in between.

Table 37.4. The massive shape allowing transverse mining, the ±0.5% Cu halo of mineralization around the orebody, and the recovery of units left from No. 1 Main mining of the pillar between old blasthole stopes resulted in a recovery of 102% at a medium extraction (144%) and low dilution (29%).

Table 37.5. Although the orebody was mined transversely and should have had a high recovery, only 74% of the geological reserves were recovered. Mineralization was generally continuous, but a lower than average grade section encountered at a time of low copper prices plus poor ground above some old blasthole stopes resulted in the poor recovery.

Wing Orebody

The Wing orebody is fairly narrow [±21 m (70 ft)], dips at 0.87 to 1.22 rad (50 to 70°), and is 80% mined longitudinally.

Table 37.4. Much of the orebody was narrow [±12 m (40 ft)], dipped flatly at 0.87 rad (50°), and was overlain by readily caved waste nearly barren of copper values. As a result, mining recoveries were only 81%, even though in-place grade was considerably better than the North West and No. 1 South orebodies.

Table 37.5. With a better in-place grade than either the North West or No. 1 South, more of this narrow orebody (88% of the copper units) was mined.

With the flat dip of the orebody [0.87 rad (50°)] and nearly barren readily caved waste, only 73% of the geological reserve was recovered.

No. 2 Orebody

The No. 2 orebody is massive [79 m (260 ft) wide], dips irregularly at ±1.22 rad (70°), and is mined 90% transversely.

Table 37.4. The massive, nearly vertical configuration of the orebody overlain by a large halo of ±0.4% Cu mineralization resulted in high extraction (182%), high recovery (98%), and high dilution (46%).

Table 37.5. For the same reasons as those previously mentioned, the mineral reserve was fully exploited. Because the No. 2 orebody contains several high-grade zones, 107% recovery of geology reserves can be attributed to the geologic practice of cutting all high-grade assays to 6%.

No. 1 Main Orebody

The No. 1 Main orebody is massive [46 m (150 ft) wide], dips at 1.57 rad (90°), and is mined 95% transversely, situated below the open pit.

Table 37.4. Although early problems of poor fragmentation, poor brows, and bridged slots initially gave poor results, the massive shape and vertical extent of this orebody gave a high recovery (97%) at near average extraction (162%) and average mine dilution (40%).

Table 37.5. For the reasons previously mentioned, most of the orebody was mined (98% of the geology copper units). At 143% extraction and 33% dilution, 96% of the geological reserve units were recovered.

Limb Orebody

The Limb orebody is narrow [±18 m (60 ft)], dips at 1.4 rad (80°), and is mined 80% longitudinally.

Table 37.4. The interference of old stopes and the irregularity of the orebody resulted in a low recovery of 81%, even though the in-place grade was the highest of all the narrow orebodies.

Table 37.5. For the reasons previously mentioned, recovery of geological reserves was only 71%.

SUMMARY

Mining methods tried at Craigmont between 1961 and 1965 met limited success. Variations of blasthole stoping produced reasonable costs and were relatively safe, but dilution was serious and unpredictable. Cut-and-fill stoping was high cost with low productivity, and in the No. 2 orebody it was extremely hazardous. Both methods left 50% of the mineral reserve tied up in pillars that would be extremely difficult to mine.

A block caving system of mining was ruled out because the orebodies were small and irregular, and many areas were composed of competent ground that would be poorly fragmented.

In contrast, sublevel caving has provided a safe, low-cost, highly productive method to mine the Craigmont orebodies once the early problems of poor fragmentation, poor brows, bridged slots, frequently hung-up orepasses, inadequate ventilation, and low productivity had been overcome.

In summary, sublevel caving has proved successful for mining the Craigmont orebodies for the following reasons:

1. It allowed selective mining of small, irregular lenses as well as larger, massive orebodies.
2. The caving tendency of the weaker country rock allowed mining orebodies of weak or strong physical characteristics.
3. The halo of low-grade copper mineralization surrounding the orebodies, by providing free units in the diluting material, has led to high recovery (94%) of mining reserves.
4. The high vertical dimension of the orebodies allowed continuous mining without tying up 50% of the deposit in pillars or requiring maintenance of development once the ore was removed from a level.
5. The method is efficient and well adapted to highly mechanized productive equipment to provide a low mining cost.

Certainly the overall recovery of 90% of the geological mineral inventory (0.7% Cu cutoff) at 27% dilution compares well with the 85% recovery and 30% dilution envisaged in the original sublevel caving proposal of March 1965; and as stated earlier, although dilution is high, sublevel caving has provided the most economical method of mining per pound of copper produced. Because of the adverse ground conditions and relatively low reserve copper grade, it is the only method that allowed full exploitation of the Craigmont deposit.

ACKNOWLEDGMENTS

Much of the material contained in this chapter has been taken directly from reports published earlier, especially the two reports "Sublevel Caving at Craigmont" by C.L. Pillar, A.J. Petrina, and E.W. Cokayne, published in April 22, 1970, and "Some Practical Problems of Sublevel Caving at Craigmont Mines" by E.W. Cokayne, published in August 1972. A. Mosley prepared the illustrations for this chapter.

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CHAPTER 38

Sublevel Caving at the Stobie Mine, Inco Metals Co.

G.W. Johnston

INTRODUCTION

For many years mining at the Stobie mine of the Inco Metals Co. consisted mainly of fill-type mining for higher-grade ores and block caving or blasthole open stope mining for lower-grade ores. Open stope mining, although quite favorable in the initial stoping phase, has always presented problems in the pillar recovery phase, especially at depth.

In 1966, sublevel caving was introduced. This method is ideally suited for low-grade ore and can be applied very successfully to intermediate or higher-grade ores providing proper controls are enforced.

Sublevel caving has three distinct advantages: it is highly adaptable to mechanization; it recovers the ore in one operation; and it provides adequate ground support and safety regardless of mining depth.

This method does, however, make extra demands, particularly for new types of supporting services. The constant presence of rock makes quality control of the grade of the ore being extracted essential. Increased development, different types of ventilation, roadways, and maintenance are only a few of the many changes required. The care and maintenance of the fleets of diesel equipment and the modification of equipment present a constant challenge.

CHOICE OF SUBLEVEL CAVING

The question arises: Why mine by sublevel caving? Figure 38.1 shows a typical problem encountered by the open stope or blasthole mining method. Near the surface there is little or no ground support problem. As mining progresses to an intermediate depth, however, ground failure starts to occur in the supporting pillars. If open mining is carried beyond this point, the critical depth is soon reached, accompanied by hanging wall caving, extensive pillar failure, and the danger of losing control of the mining cycle. The rock caved from the

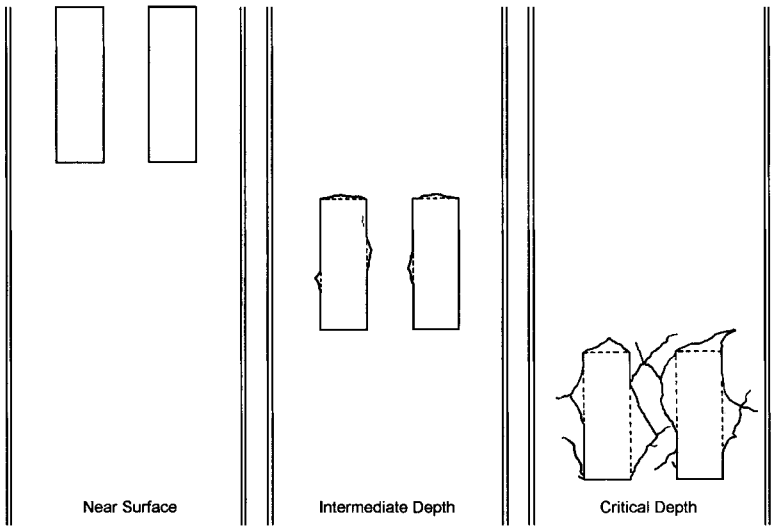


FIGURE 38.1 Open stoping ground conditions at three different depths

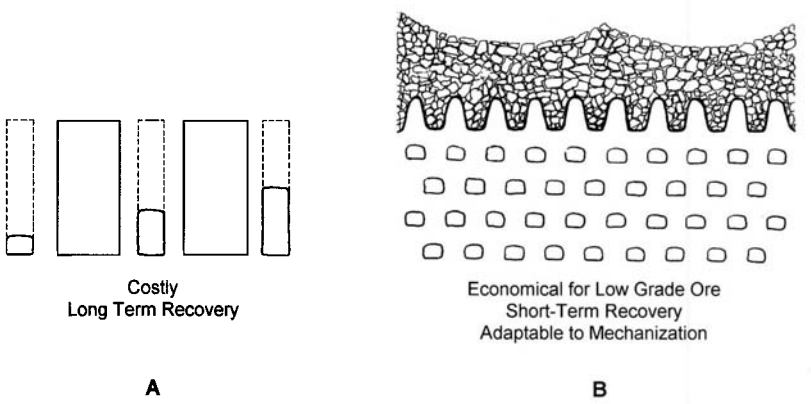


FIGURE 38.2 Ground support alternatives: (A) fill mining for high-grade ore (sand fill support); and (B) sublevel caving for low-grade ore (broken rock support)

hanging wall mixes with and dilutes the ore being extracted from the blast-hole stopes. Ore losses occur when pillar failure prevents drilling and development of part of the ore. The interruption to the mining cycle causes serious curtailment in production until ground failure is controlled and the mining method changed to adapt to the new conditions.

Because of the limitations of open stope mining, alternatives were sought. The first alternative (Fig. 38.2) was to fill the open stopes with sand fill before extensive pillar deterioration occurred, recover the pillar ore by fill mining, and then convert to total fill mining below. If the ore grade is high enough it

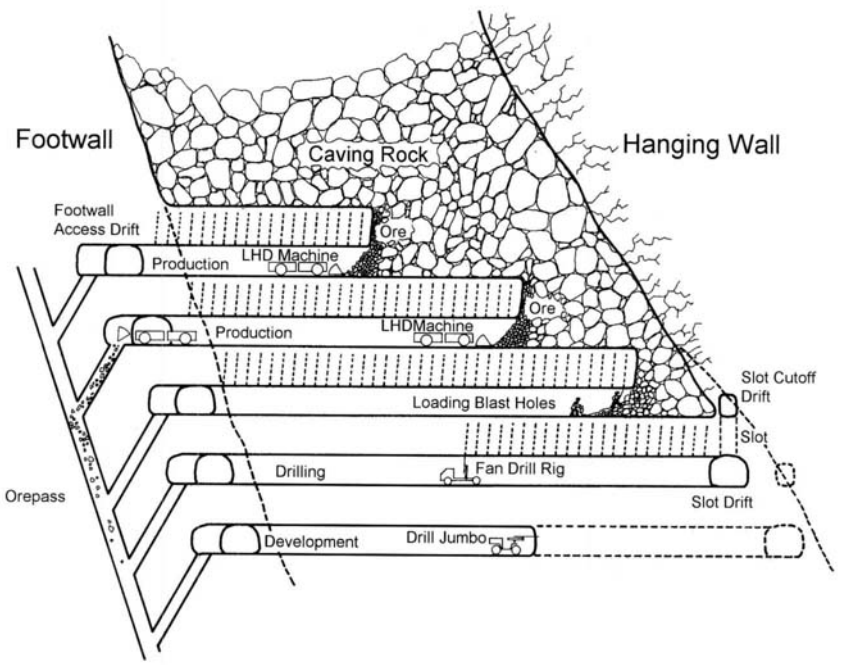


FIGURE 38.3 Cross-sectional view of sublevel caving

can support the costly fill method. Recovery will be at a much slower rate and will therefore extend over a longer period of time.

On the other hand, sublevel caving gives complete primary mining recovery in a short time and is very adaptable to mechanization, which is now a necessity to combat spiraling labor costs and give greater productivity per miner.

Sublevel caving is ideally suited as a mining method for low-grade orebodies, which eventually reach a critical depth for open stope mining.

DEVELOPMENT

The development phase comes first and, of course, its main purpose is to gain access to the ore and provide a route for ore removal from the mine.

Initially a 20% inclined service ramp is driven in the footwall rock (Figs. 38.3, 38.4, and 38.5). It is preferable to start at the top of a mining block and drive the ramp downgrade so gravity can assist in mucking the blasted development round, and also access can be gained more quickly to the initial production area. The ramp normally turns at 3.1 rad (180°) about every 152.4 m (500 ft) to keep it reasonably close to the rock disposal pass. The result is an elongated spiraling downgrade service tunnel that stays within the limits of the mining block.

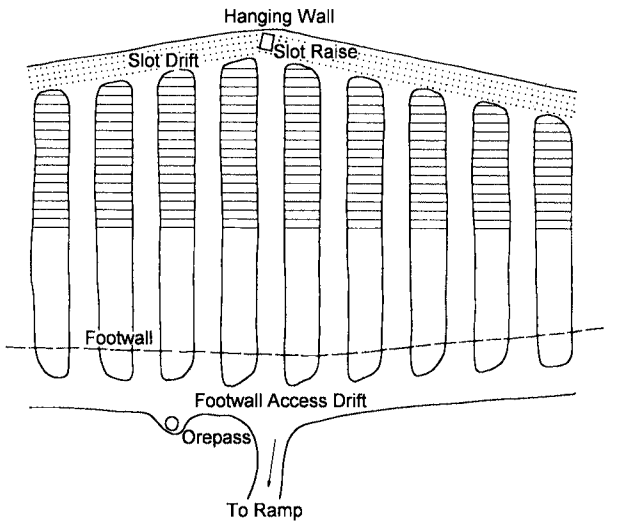


FIGURE 38.4 Plan of level showing location of slot and drilling

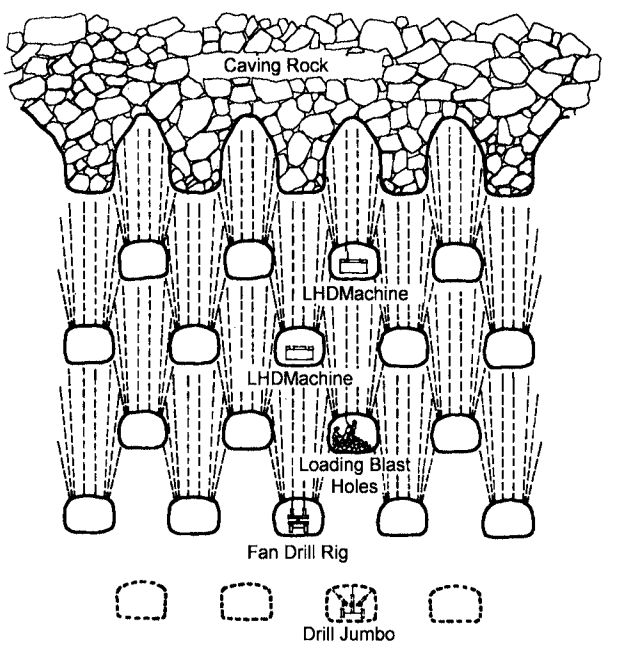


FIGURE 38.5 Longitudinal sectional view of sublevel caving

Sublevels are established by driving horizontal access drifts off the ramp approximately every 9 m (30 ft) vertically. A haulage drift is then driven in the footwall rock about 10.7 m (35 ft) from the footwall ore contact. Another option is to position this drift in the footwall ore with the final production retreat along the footwall drift toward the level access drift. Parallel ore extraction drifts 11 to 12.2 m (36 to 40 ft) apart are driven from the footwall

drift across the orebody to the hanging wall contact. A slot drift is then driven in ore along the hanging wall contact interconnecting all of the extraction drifts at the hanging wall as well as the footwall.

In general, drifts in rock are 4.3 m (14 ft) wide by 3.7 m (12 ft) high, whereas in ore they are 4.9 m (16 ft) wide by 3.7 m (12 ft) high—a wider drift improves recovery.

Slot raises are drilled from the slot drift to the level above [about 6.1 m (20 ft) long] about every third or fourth crosscut.

At the footwall access drift an orepass is established every 152.4 m (500 ft) along the drift. This is normally a 2.1-m (7-ft) diam bored raise, which is intersected at each level by drilling and blasting an inclined finger into the raise.

One rock pass serves each ramp system. The ramp is centered around this rock pass in order to gain access to the pass on each level as quickly as possible. Until the pass is available on a new level, all rock must be hauled up the ramp to the level above and then hauled along this level to the rock pass.

One repair area about 6.1 m (20 ft) wide, 6.1 m (20 ft) high, and 15.2 m (50 ft) long is excavated and constructed for each ramp system, usually near the top of the ramp. This repair area, commonly called a garage, is equipped with a traveling crawl with a 7.3-t (8-st) electric hoist, and with electric welding equipment and other miscellaneous tools. Another alternative to the fixed crawl beam installation is 4.5-t (5-st) movable gantries. These are rolled manually on the concrete floor and positioned over the equipment to be lifted.

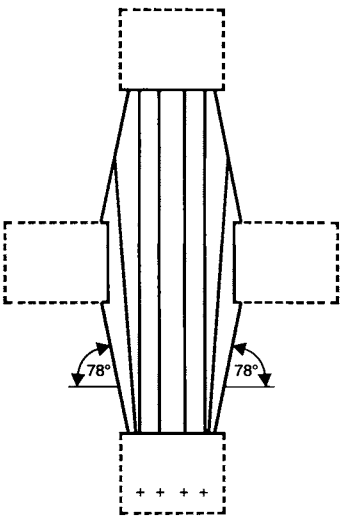
The mechanized development equipment consists of a three-boom development drill jumbo, which drills about 40 holes in each drift round in about 3 hr, and load-haul-dump (LHD) units, Scooptrams, to remove the broken development material. A diesel-powered scissor truck with an adjustable platform for bolting and screening is a definite asset.

The rate of providing development presents the biggest single challenge to sublevel caving. The current development level must be three to four levels ahead of the producing levels to allow proper organization of the other phases of mining. This lead development must not only be attained quickly in new mining blocks but also all development consumed by the constantly advancing production phase must be replaced. Each 305 mm (1 ft) of sublevel caving development results in approximately 91 t (100 st) of production ore from the drill ring above.

PRODUCTION

Drilling

The function of production drilling is to distribute the explosives through the solid ore in such a manner that when blasted, the broken ore will be suitable for extraction and handling.



Parameters for Sublevel Caving

Sublevel Height	33'
Pillar between Drift	20'
Side Slope Angle	78°
Blast Retreat Distance	5'
Drill Ring Inclination	85°
Drift Center to Center	36'
Drift Height	12½'
Drift Width	16'
Throat Width	13½'
% Ore as Development	20%

Drilling and Blasting Statistics

Toe Burden	5'
Ring Burden	5'
Drill Footage	340'
Holes/Ring	8
Hole Size	2⅛'
Maximum Hole Length	53
Drill Ring Inclination	85°
Tons/Ring	500
Tons/Drilled Foot	1.47
Load Footage	316'
Pounds Explosives	415
Powder Factor	0.83

FIGURE 38.6 Sublevel caving drill ring, metric equivalents: in. × 25.4 = mm; ft × 0.3048 = m; lb × 0.4535924 = kg; st × 0.9071847 = t; ° × 0.01745329 = rad

The extraction drifts on any level are staggered or offset with respect to the level above. Therefore when up-hole drilling is done from a drift, the holes can extend through the level above and terminate at the second level above. The side holes are angled to cover the area between the drifts on the level above, and this results in a diamond-shaped drill area above each extraction drift, which extends from hanging wall to footwall. A series of diamond-shaped drill rings or slices are drilled at a uniform burden or blast retreat distance from hanging wall to footwall in each extraction drift (Fig. 38.6).

In addition to this, closely spaced up holes are drilled from the hanging wall slot drift to the level above along the entire length of the slot drift and around the slot raises described in the development phase.

The machine used is an air-powered fan drill, which has two drills, one mounted on each side of the machine. The machine is a rubber-tired vehicle equipped with leveling jacks. It can be towed into a drill drift, but when attached to the compressed air line the unit can be air propelled within the limits of the attached air hose. It is moved from one ring to another in this manner.

Because blasting will be against broken rock, the amount of explosive used is higher than for open stope blasting. For this reason, we set our powder factor at 0.36 to 0.41 kg/t (0.8 to 0.9 lb per st). From this we can design the drill ring to determine the ring burden distance and the number of holes required.

Initially, rings were drilled with a 1.5-m (5-ft) burden. Statistics shown in Fig. 38.6 are based on a 1.5-m (5-ft) burden single-ring blast. Based on additional model testing and field experience, however, the optimum recovery

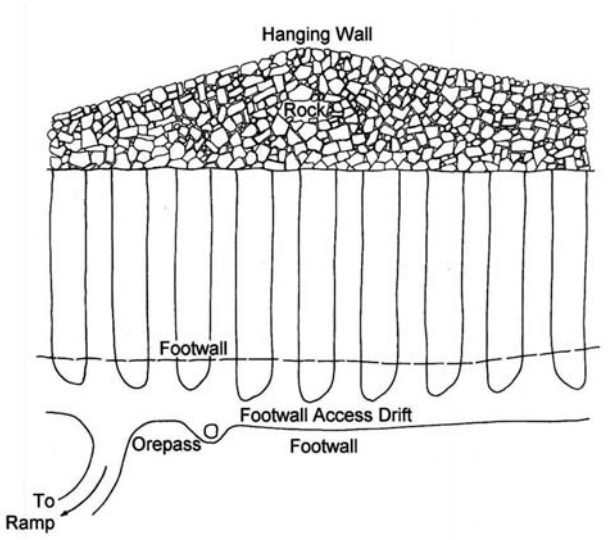


FIGURE 38.7 Plan of level in production

and minimum dilution can be attained with an 8-hole ring, 54-mm (2⅛ in.) diam hole, drilled at 1-m (3½-ft) burden. The ring is inclined forward at 1.5 rad (85°). Ring design can be changed to adapt to field conditions.

Blasting

Before production can begin, the ore must be blasted to change it from its solid state to pieces that can be extracted. The slot on the level is blasted first by starting with the holes around a slot raise. A series of small separate blasts with excess ore removal between blasts will eventually complete the opening of the slot so systematic ring blasting can begin.

Each hole in the drill ring is charged with blasting agent by blowing it into the hole with compressed air through a 19-mm (¾-in.) antistatic hose. Before the hose is fully charged with explosives, a nonelectric delay cap is positioned in the hole. The hole charging is then completed.

Two rings are blasted at one time. The delay caps in each hole in the two rings are held together with B-line. The blast is initiated with one electric cap.

Loading and Hauling

A load-haul-dump machine enters an extraction drift in which a ring has been blasted and begins to remove the ore (Fig. 38.7). It digs its bucket into the toe of the ore pile, hauls it to and along the footwall drift, and then dumps the ore into an orepass. It repeats this sequence until the ore has been extracted. The ore from the blasted ring, which is retained by the caved hanging wall rock, falls by gravity each time a bucket of ore is removed. The rock follows the ore down from the top and dilution appears mostly from the rock on top of the blasted ring rather than from the vast quantity behind it.

The time to stop pulling and blast a new ring is regulated by the amount of rock showing at the production rate. This percentage of rock allowed varies with the grade of the orebody, and because this grade is known, the cutoff percentage of rock is established before production begins. Because of the large number of production faces available in sublevel caving, the load-haul-dump machine will move to a blasted face without any interruption to the production cycle when the blasting crew moves in.

Production, therefore, is continuously advancing toward the footwall and downward, thus setting the pace for the other mining phases that must advance at the same rate to keep out of the way of the never-ending production cycle.

The Scooptram, just as the development drill, has many coordinated activities associated with it. Preparation for production involves development, drilling, and blasting preceding the actual ore removal, and then a vast team of personnel and equipment to move the ore out of the orepass to ensure continuing availability of space to receive the produced ore. This team will tram the ore from the orepass to the crusher, crush the ore, convey it to the shaft, hoist it to the surface, and then convey it to the mill.

PLANNING GUIDELINES

Extensive preparatory work is necessary before any ore is extracted and there is a thorough committment to planning judgments made far in advance of actual production. In addition to experience and knowledge gained by others, judgment guidelines are supplemented with laboratory experimentation.

The scale models are made of plexiglass. The front panel of the model has openings cut to scale to represent the extraction drifts on four to five levels in a vertical plane. A column of ore—dark, granular magnetic sulfide ore that has been crushed and screened to represent the scale of actual ore pieces—is loaded at the face of the model and this is backed with white quartzite representing the caved hanging wall rock.

Ore is extracted from each drift, and records are kept that, when summarized, show the effectiveness of ore recovery and the extent of rock dilution. Other experiments are conducted by changing the parameters: different thickness of ore column, level intervals, or extraction drift spacing. The experiment producing the best result on recovery and dilution indicates the development and drilling layout to be used.

A composite of all experiments was also done. In this test the model was loaded entirely with quartzite, and marked pieces were placed in the model while loading in a closely spaced coordinate system. By drawing from one opening at the bottom of the model until no more marked stones are extracted, a series of master cave figures can be plotted on paper (additional rock is added while drawing to keep the model full). From the master cave figures, specific parameters are chosen and tested individually for verification. The cave figure is elliptical in shape. The best extraction is obtained by designing a blasted ring that comes closest to filling the theoretical cave

figure. Any rock inside the cave figure must be removed; any ore outside the cave figure will be lost.

Of course, model experimentation alone should not be relied on fully. It does, however, provide very useful guidelines, especially when no other guidelines are available. Model testing presents the opportunity to simulate the underground operation in the laboratory and thus provide the required production behavioral patterns quickly and cheaply well in advance of developing the ore.

Underground testing is constantly being done in one form or another. Research teams can monitor around the clock, if required by the production, the effectiveness of various combinations of ring burdens.

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CHAPTER 39

Sublevel Caving at LKAB's Kiirunavaara Mine

H. Heden, K. Lidin, and R. Malmström

INTRODUCTION

LKAB was formed at the turn of the century when the railways between the ore fields and harbors were built. The mines are situated north of the Arctic Circle in Sweden. Kiirunavaara is the largest mine, with a production capacity of approximately 30 Mt/a (33 million st per year). Malmberget has a capacity of about 12 Mt/a (13 million st per year). Both are underground mines. Svappavaara, the only opencast mine currently in operation, has a capacity of about 4.5 Mt/a (5 million st per year). The ore harbors are at Narvik for the Kiruna and Svappavaara mines and at Luleå for the Malmberget mine (see Fig. 39.1).

Production in the form of finished products is shown in Fig. 39.2a, b, and c. Due to poor ore markets, production and shipments had been cut back, but there was a positive swing in 1978.

LKAB has been 96% state-owned since 1957. In addition to the iron mines in northern Sweden, LKAB has subsidiaries in central Sweden, which produce sulfide ores, quartz sand, and uranium (Fig. 39.3).

The mining methods employed are opencast bench stoping at Svappavaara and sublevel caving at Kiirunavaara and Malmberget.

GEOLOGY

The Kiirunavaara orebody is 4000 m (13,100 ft) long and averages about 90 m (300 ft) wide, with variations in width from a few meters (≈10 ft) to around 150 m (500 ft). The ore has been explored by deep drilling down to about 1000 m (3300 ft), but magnetic measurements indicate a depth of at least 1500 m (5000 ft). The Kiirunavaara ore is oriented in a north-northeasterly

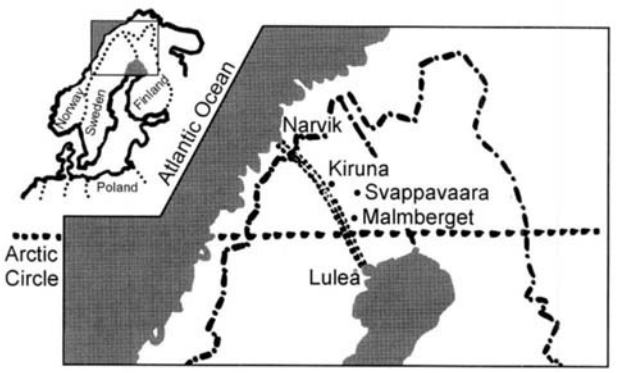


FIGURE 39.1 Geographic situation

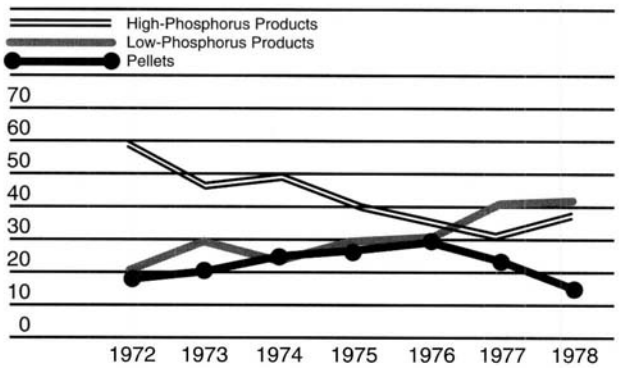


FIGURE 39.2a Production at LKAB

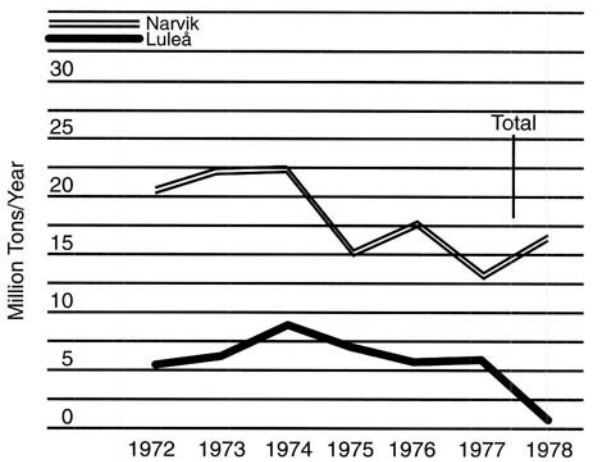


FIGURE 39.2b Railway transport at LKAB

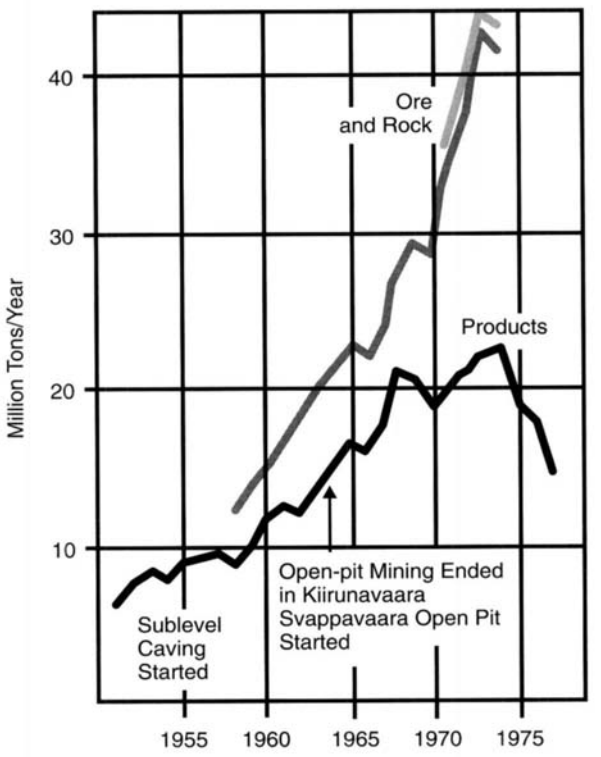


FIGURE 39.2c Production at Kiirunavaara

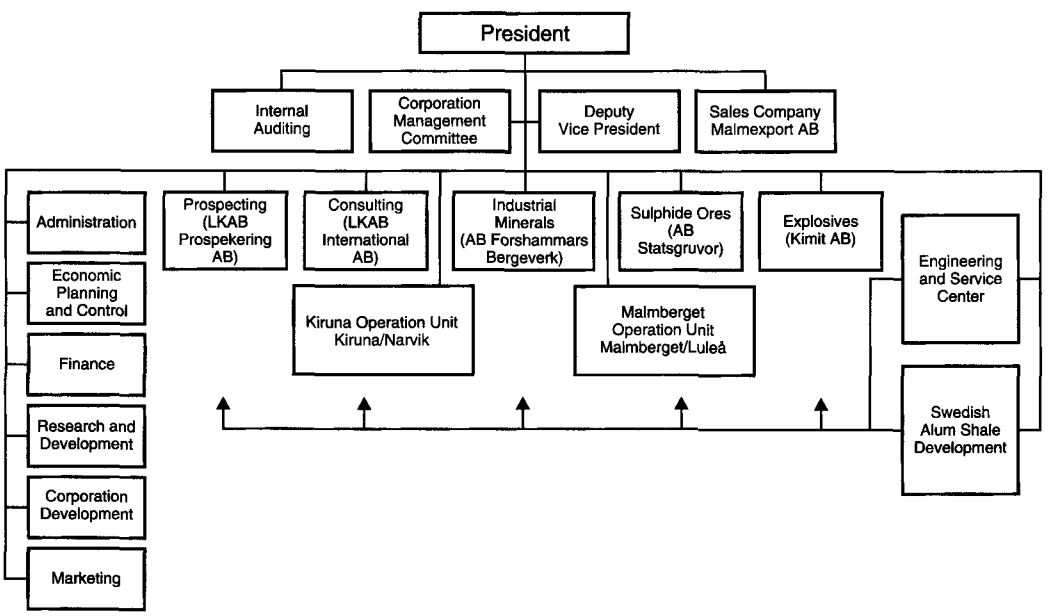


FIGURE 39.3 Organization of LKAB

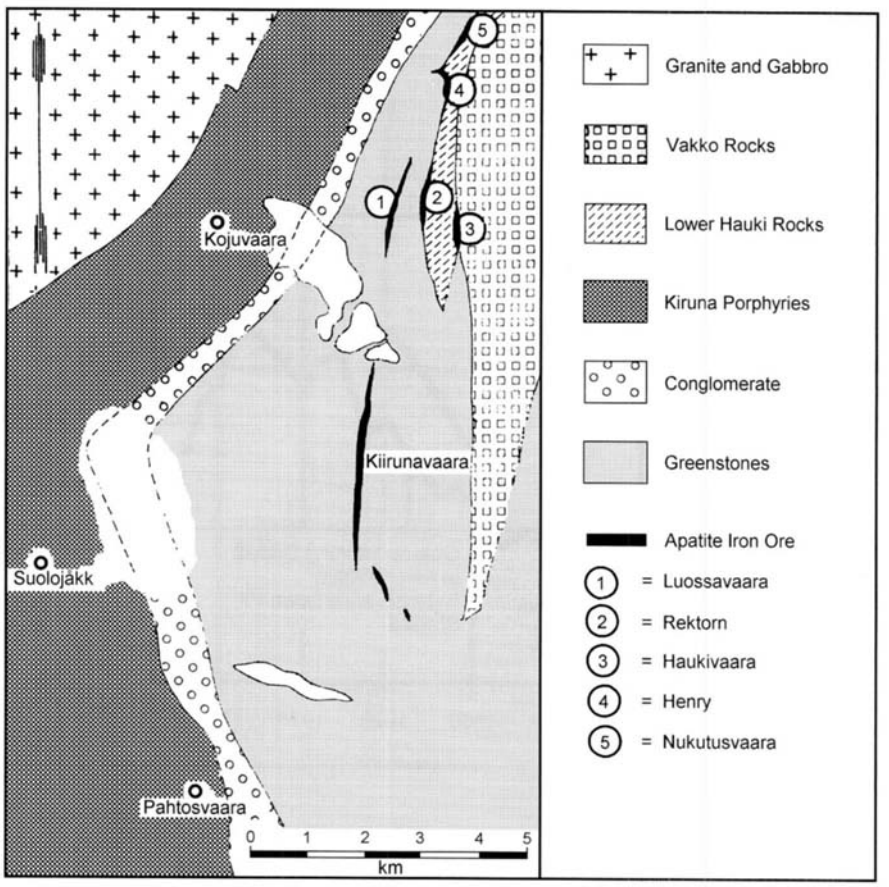


FIGURE 39.4a The geology of the Kiruna region (courtesy SGU C 709 T Parak 1975)

direction with a number of ores in the boundaries between the wall rock types. The ore in Kiirunavaara is mainly a fine-grained magnetite with a varying content of fine-grained apatite. The apatite content of the ore decreases with increasing depth.

The other ores are all apatite-bearing, and a couple are hematite ores (Fig. 39.4a, b).

The ores are sedimentary, tilted, and tectonically deformed. They have a lateral dip of 0.87 to 1.2 rad (50 to 70°). The contacts between ore and wall rock are distinct and sharp, but both the hanging wall and the footwall are brecciated and impregnated with magnetite.

The footwall of the Kiirunavaara ore consists of syenite porphyry with a low quartz content and relatively good strength; the hanging wall consists of quartz porphyry. Both types are known as Kiruna porphyries. The ore at the boundary with the hanging wall is often rich in kaolin. There are a few veins of syenite porphyry running through the ore, which has a few major faults (Fig. 39.4c, d).

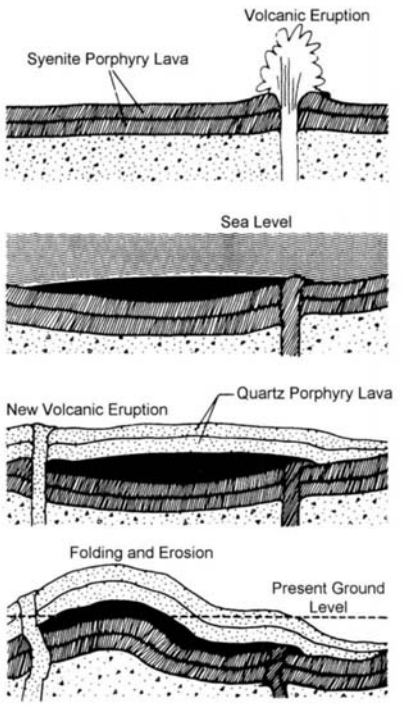


FIGURE 39.4b Ore genesis of Kiruna

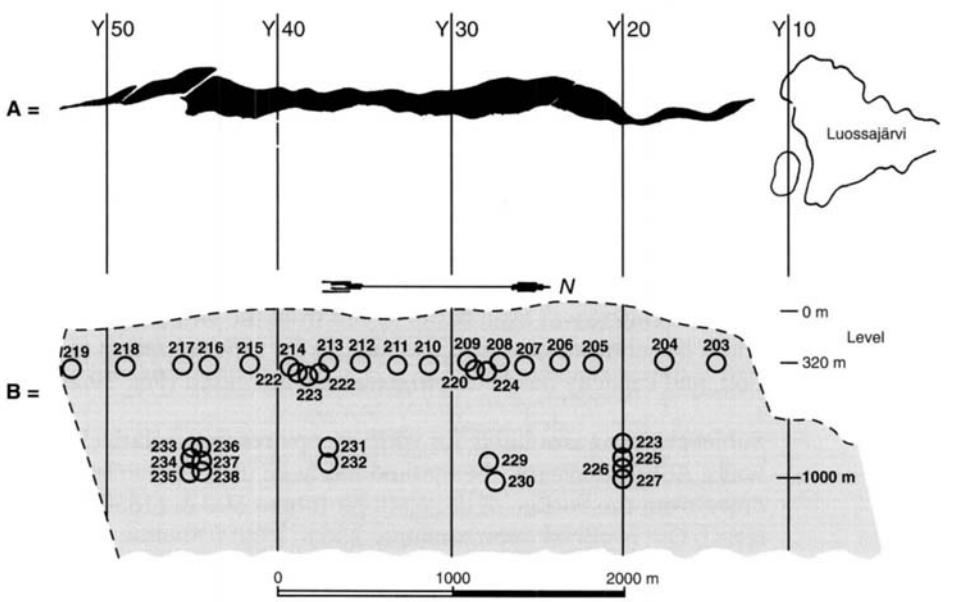


FIGURE 39.4c The Kiirunavaara orebody, A = horizontal section and B = longitudinal section (Courtesy SGU C 709 T Parak 1975)

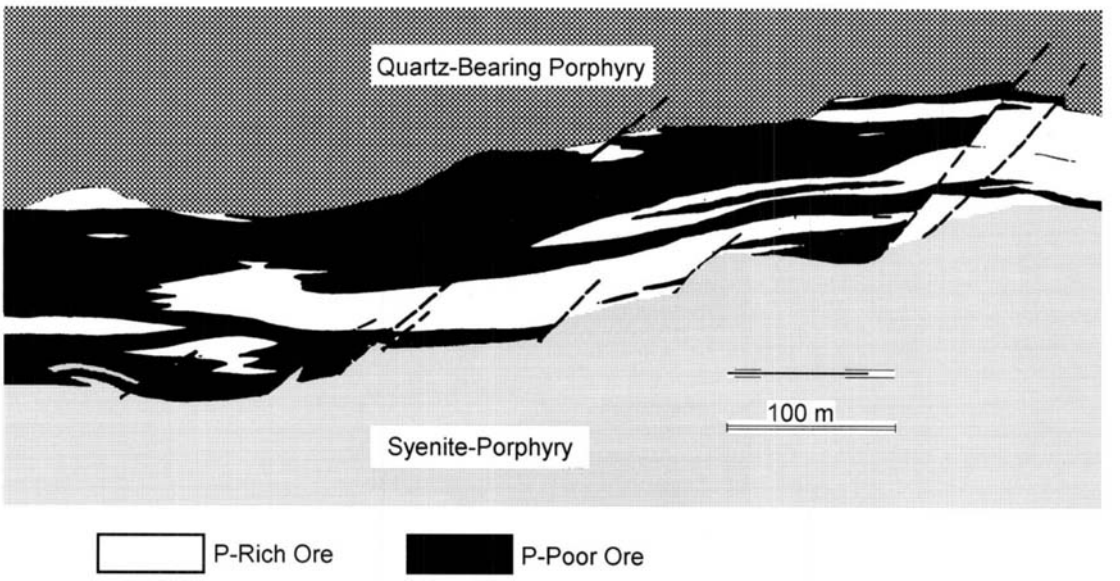


FIGURE 39.4d Horizontal section of ore grades

There are two sharply delineated ore grades in Kiirunavaara: low phosphorus ore with less than 0.1% phosphorus (about one-third of the ore) and high phosphorus ore with greater than 0.5% phosphorus. The iron content is 67% in the low phosphorus ore and 60% in the high phosphorus ore. These ore types are sold as different grades and so are kept separate during mining.

MINING AT KIIRUNAVAARA

The Kiirunavaara and Luossavaara ores were first mined using opencast bench stoping in 1902. When mining costs rose owing to higher waste removal, underground methods were adopted. In the mid-1950s, the ores were still being mined on the surface at the same time as shrinkage stoping had started underground. It was found, however, that these ores required a higher extraction capacity and selectivity, so sublevel caving was introduced. Sublevel caving was even better suited from the viewpoint of rock mechanics, and it has allowed greater opportunities for rationalization and mechanization, and capacity has been progressively increased (Fig. 39.2c).

Sublevel caving is suitable for wide, steep ores with relatively weak hanging walls. At Kiirunavaara, the method has been used for the levels extending down from the bottom of the open pit [about 500 m (1650 ft) above sea level]. Ore is mined approximately 250 m (820 ft) below this level.

The method is also being used in Malmberget, where the working level is approximately 250 m (820 ft) below the ground surface.

Refined and modified sublevel caving will probably be used at even greater mining depths, because it is a safe and reliable method.

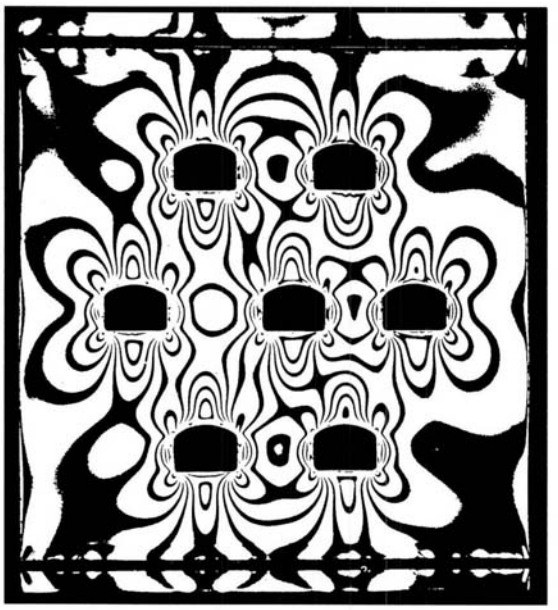


FIGURE 39.5a Model test showing the effect of drift placement errors on the stress field

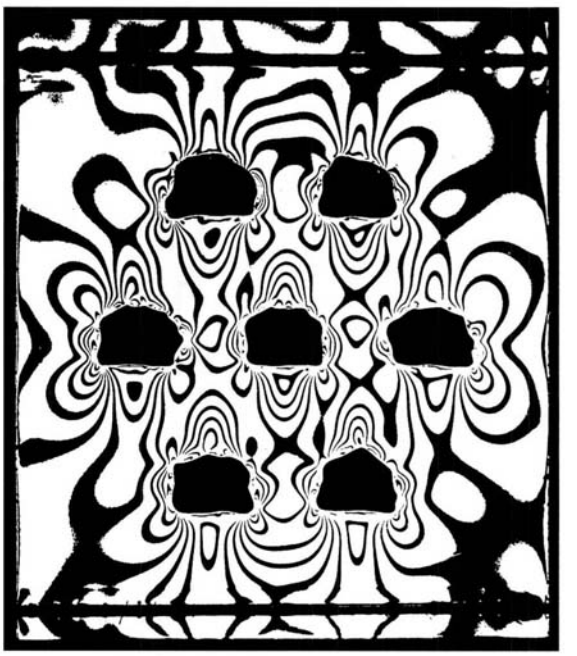


FIGURE 39.5b Model test showing the effect of irregular drift contour on the stress field

Some advantages of sublevel caving are that it is well-suited to selective ore extraction, it permits high mechanization and flexible capacity, it is safe compared with methods using open stopes, and it is relatively simple to plan and not particularly affected by variations in ore width and grade.

Disadvantages include the fact that it requires relatively high investments and relatively extensive development work, and it can lead to rock mechanics problems at great depths.

ROCK MECHANICS AT KIIRUNAVAARA

Rock mechanics is important in determining the choice of mining method and the design of drifts with increasing mining depth and thereby increasing rock pressure. The modifications already planned—greater slice height (vertical distance between sublevels) and greater crosscut spacing—are partially a result of such considerations. Strength tests, fracture frequency analyses of drill cores, and model experiments are used by the mine to predict rock mechanics conditions. The strength of the wall rock is relatively good with a tensile strength of 14 MPa (2000 psi) and a compressive strength of 230 MPa (33,500 psi). The tensile/compressive strength of the ore is 10/120 MPa (1450/17,500 psi). Stress conditions around rock caverns are being studied in model experiments. Figs. 39.5a and b illustrate the distribution of stresses around development drifts. Fig. 39.5a shows the effect of minor errors in horizontal placement, and Fig. 39.5b of uneven drift contours. Rock mechanics conditions are also followed by means of measurements made at the surface

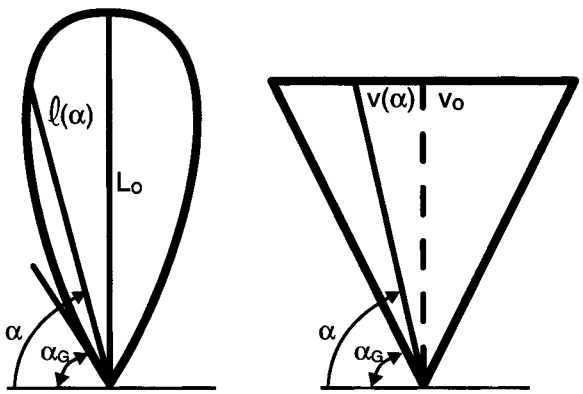


FIGURE 39.6a Idealized representation of the body of motion (drawbody). The surface is defined by the particles that reach the drawpoint at the same time.

as well as by seismic and other measuring methods. The main haulage level currently being planned, at 1000 m (3300 ft), will probably be the last main level where the current modified sublevel caving method will be used. Below this level, it is likely that rock mechanics conditions are such that sublevel caving will be excessively expensive, and other methods may entail lower costs.

SUBLEVEL CAVING THEORY

Sublevel caving theory has been studied in operational experiments and model experiments by the Mining Technology Department of LKAB. The parameters of importance to the method are slice height (vertical distance between sublevels), distance between crosscuts, location of access drifts, and drift dimensions. The geometric accuracy achieved in development is of great importance for fragmentation and thus for the movement of the caved material during mining. The movement of the caved material is difficult to study. Studies of this movement have been conducted using models in which the boundary layer angle and ore yields have been observed for different blasting layouts.

The "drawbody" is an imaginary body (see Figs. 39.6a, b, c, and d) whose surface is made up of particles that reach the drawpoint at the same time.

It is assumed that there is an angle α_G (the value depends on the type of material and probably the stress situation) outside which the material will never move, no matter how long one draws. If the draw is assumed to be concentrated at one single point, the situation in Fig. 39.6a happens.

The figures represent a single phase in the drawing where $l(\alpha)$ is the distance covered by the particles drawn at this phase. L_o represents the maximum value of $l(\alpha)$ for the phase. In the same way, particle velocities $v(\alpha)$ may be assumed.

In reality, this concentrated draw is not achieved, and the drawbody in Fig. 39.6b is an attempt to get closer to what really happens. The draw width

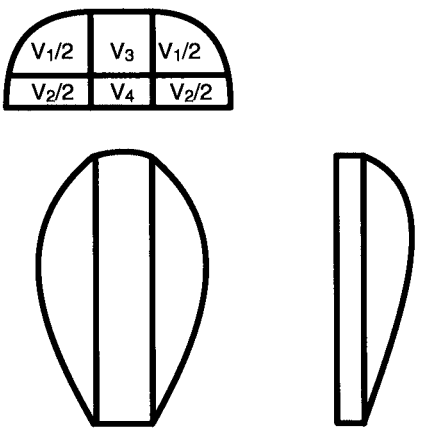


FIGURE 39.6b Drawbody realized with typical sublevel caving geometries

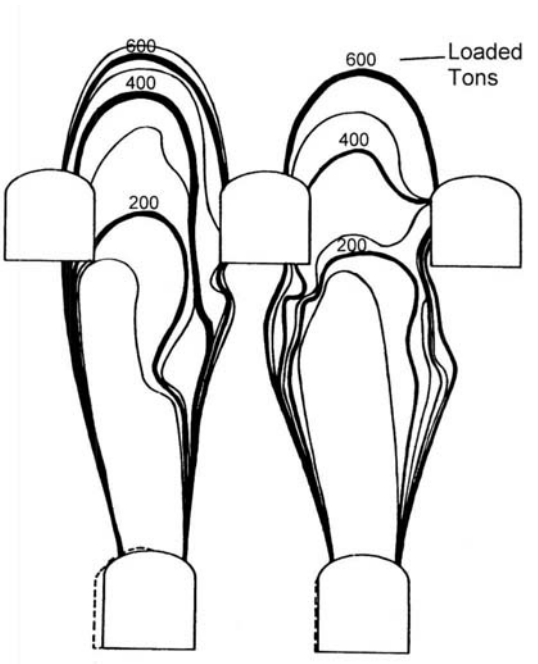


FIGURE 39.6c Full-scale trial at Grängesberg. One round with 1.5-m (5-ft) burden represents 500 t (550 st). (Courtesy Janelid tests in Grängesberg. Kiruna Symposium, 1975.)

is the width at the brow along which there really is a draw. Its value depends upon the shape of the drift, the size of the bucket, and the loading strategy.

Mathematical formulas for the volume of the drawbody (see Figs. 39.6a, b, and c) on the basis of the boundary layer angle (α_G) are:

$$V = \frac{V_1}{2} + V_2 + V_3 + V_4$$

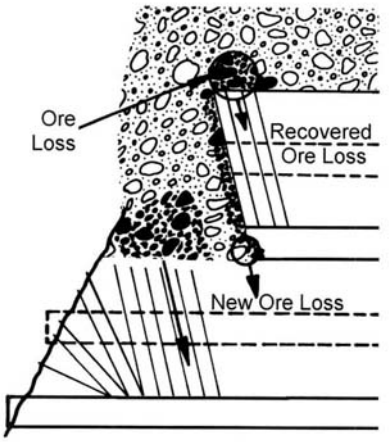


FIGURE 39.6d Recovered ore losses. This means that ore lost on the upper slice is recovered on the next slice and so on.

$$V_1 = \frac{\pi}{6} L_o^2(1 - \sin \alpha_G)$$

$$V_2 = \text{draw depth} \times \text{area}$$

$$V_3 = \text{draw width} \times \frac{\text{area}}{2}$$

$$V_4 = L_o \times \text{draw width} \times \text{draw depth.}$$

$$\text{Area} = \frac{L_o^2}{(1 - \sin \alpha_G)^2} \cdot \left[\left(\frac{\pi}{2} - \alpha\frac{\pi}{180°}\right)\left(\frac{1}{2} + \sin^2 a_G\right) - \frac{3}{2} \sin \alpha_G \cos \alpha_G\right]$$

On the basis of theories and experiments and in consideration of the rock strength problems encountered at great depths, the slice heights have been gradually increased from 8 to 12 m (26 to 40 ft). A slice height of 12 m (40 ft) is used at Kiirunavaara and the center-to-center distance between crosscuts is 11 m (36 ft) (Fig. 39.7a). When the present main haulage level at 540 m (1800 ft) is passed, the crosscut spacing will be increased to 16.5 m (54 ft) (Fig. 39.7a, b).

The calculated values for recovery and dilution are method dependent. At Kiiruna, the recovery is 80 to 90% of the in-place ore, and dilution by ore-bearing rock is 20 to 30%. Consider the drawing of one single round in sublevel caving.

From the drawing one gets M_a t of waste rock and M_u t of ore. In Kiruna, drawing is stopped when the average dilution is 25% by weight.

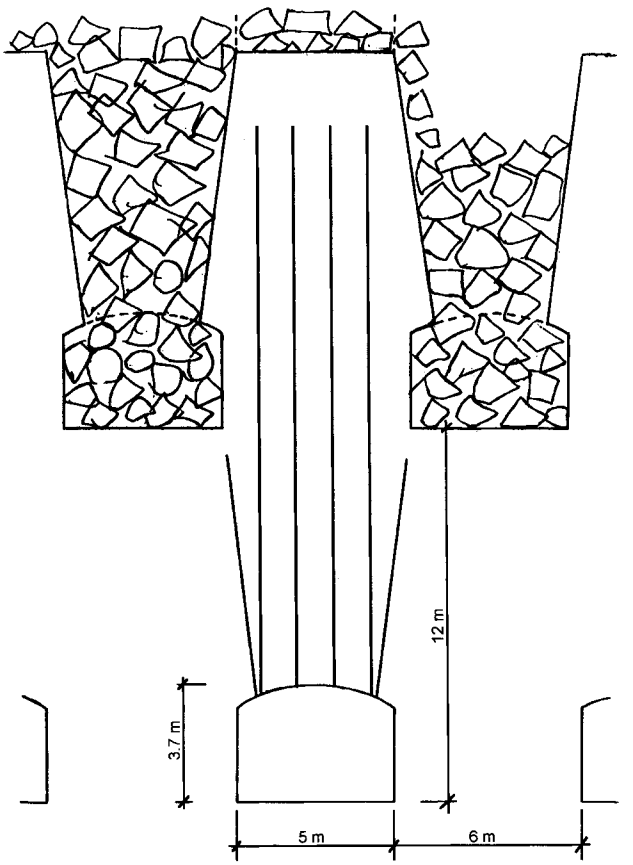


FIGURE 39.7a Silo layout for sublevel caving

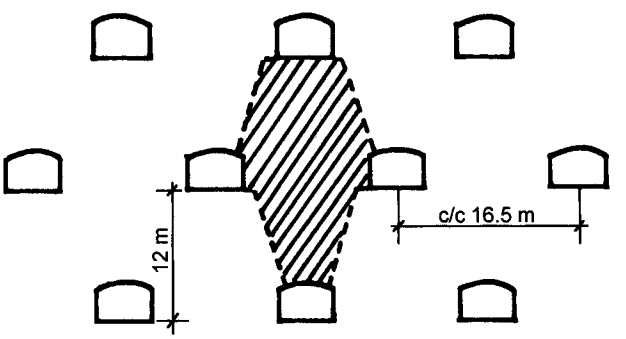


FIGURE 39.7b Sublevel layout below 550-m (1800-ft) level

Thus we have

$$\frac{M_a}{M_a + M_u} = 0.25$$

or

$$M_a = \frac{M_u}{3} \qquad \text{(EQ 39.1)}$$

However, M_r t of caved masses contain β% (weight) of ore (from earlier ore losses) and thus

$$M_r = \frac{M_a}{1-\beta} \qquad \text{(EQ 39.2)}$$

Combining Equations 39.1 and 39.2, one finds that

$$M_r = \frac{M_u}{3(1-\beta)} \qquad \text{(EQ 39.3)}$$

This means that from the caved masses we "re-recover" (see Fig. 39.6d) a quantity

$$M_r - M_a = M_u \cdot \frac{\beta}{3(1-\beta)} \qquad \text{(EQ 39.4)}$$

As a whole, we draw M_u t of ore from a round weighing M_o t. Of this tonnage, one part

$$M_u \quad \frac{\beta}{3(1-\beta)}$$

is from the caved masses surrounding the round and the remainder (M_{us})

$$M_{us} = M_u - M_u \cdot \frac{\beta}{3(1-\beta)} \qquad \text{(EQ 39.5)}$$

emanates from the blasted round.

The total recovery is therefore

$$\text{Total recovery} = \frac{M_u}{M_o} = 1 - \gamma \qquad \text{(EQ 39.6)}$$

where γ is the ore loss in weight percent.

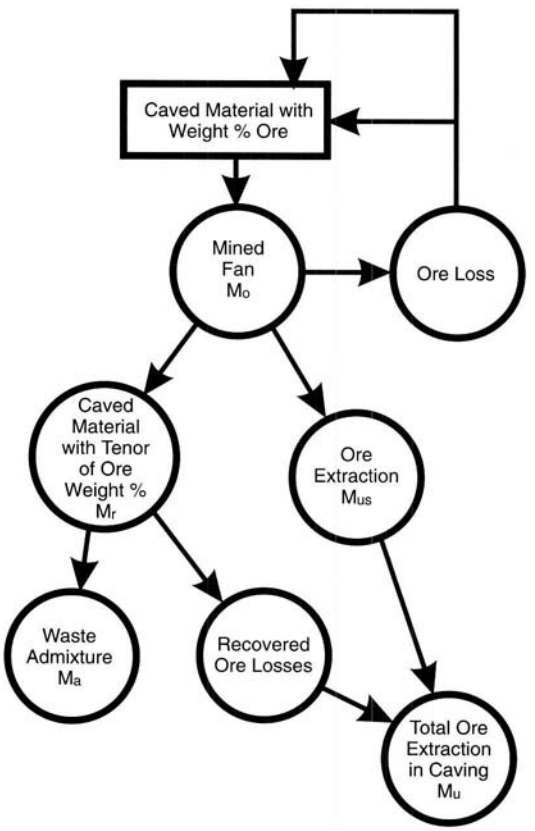


FIGURE 39.8a Flow pattern for mining (courtesy Internal Report, L. Hermansson, 1977)

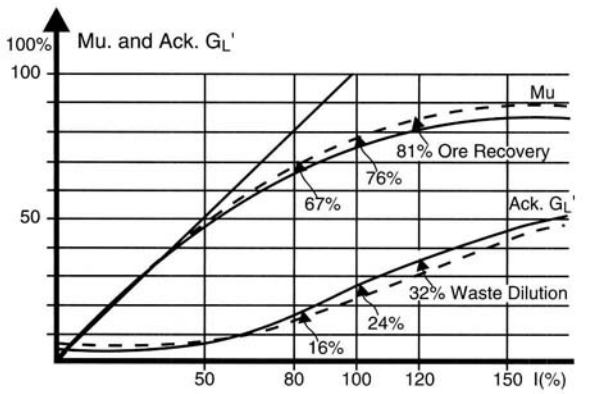


FIGURE 39.8b Example of ore recovery and waste dilution in mining as a function of degree of loading (*I*). Ack.G_L' = waste dilution and M_u = ore recovery.

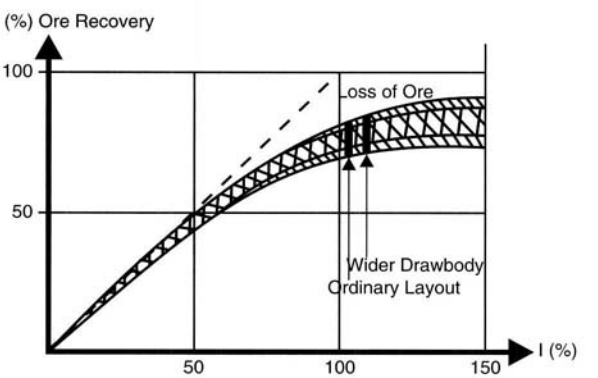


FIGURE 39.8c Example of ore recovery as a function of degree of loading (*I*), as determined from a model trial

Combining Equations 39.5 and 39.6 one finds that

$$\text{Total recovery} = \frac{3(1-\beta)}{3-4\beta} \cdot \frac{M_{us}}{M_o} \qquad \text{(EQ 39.7)}$$

With current slice geometry, it is not theoretically possible to avoid a total ore loss of 10 to 20% and an increase in dilution with increasing mucking (ore extraction) (see Figs. 39.8a, b, c). Figure 39.8c shows, in addition to ordinary sublevel caving geometry, the effect of a wider drawbody, that is, a greater distance between crosscuts, and a wide silo drilling pattern. High- and low-phosphorus ores are produced in Kiirunavaara and are kept separate throughout the entire ore flow. Owing to differences in value between these main types of ore, it is desirable that the selectivity of the method be kept high and

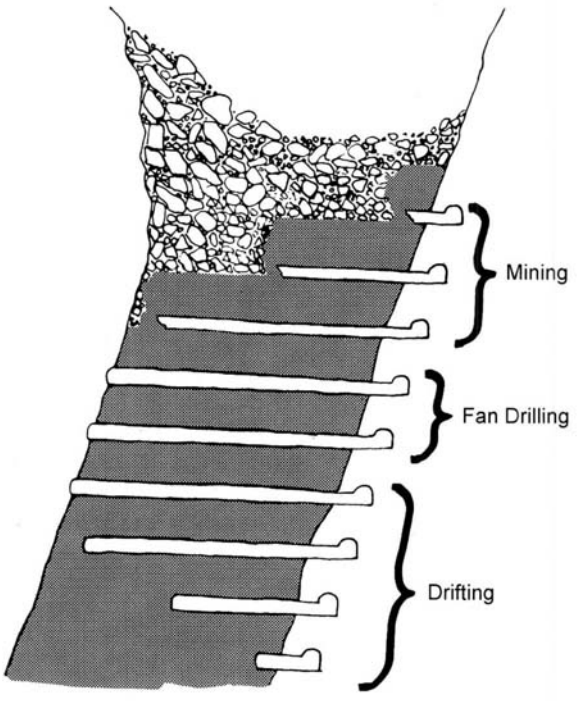


FIGURE 39.9 Cross section showing the present mining system

be improved. The important factors in this regard are accuracy in development and drilling and then accuracy in sampling.

MINING SYSTEM

The underground portion of the Kiirunavaara mine has a communications system that is reminiscent of an open-pit mine. From the main entrance, car and bus traffic to the various horizontal levels is via seven main ramps. There are a number of additional ramps, including spiral ramps, in the ore. All of the working areas in the mine can be reached by car. The main levels are at 370, 509, and 740 m (1200, 1670, and 2430 ft). The main haulage levels are at 540 and 775 m (1800 and 2550 ft). At these levels, there are workshops, dining rooms, and offices. These facilities, as well as shafts for rock and ventilation, are constructed in advance. Mining can be divided into three phases: development, slice drilling, and loading of caved material (Fig. 39.9). Normally, two levels are mined at the same time, while two levels are being drilled and three levels are developed.

In development, entry cuts are made from the ramp to the slice level, from which an access drift along the footwall of the ore and crosscuts into the ore are driven. At the same time, a ventilation shaft is opened to supply fresh air to the working (Fig. 39.10).

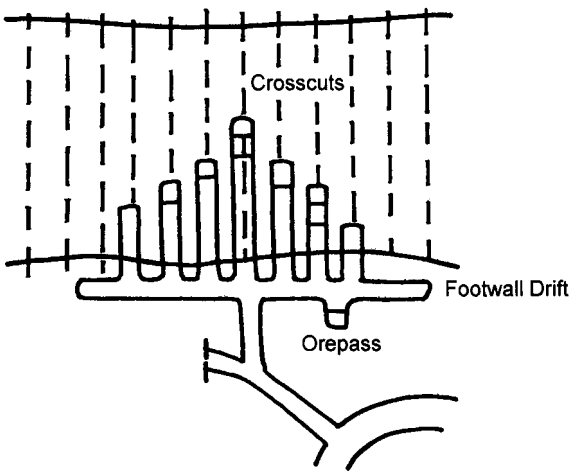


FIGURE 39.10 Drifting technique

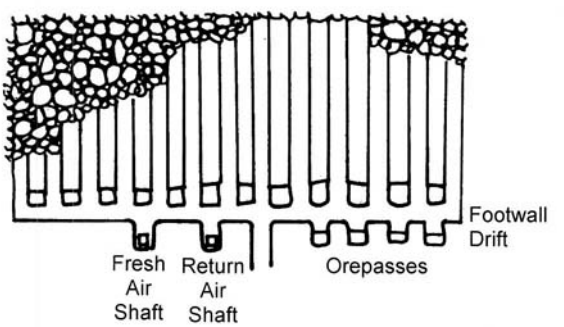


FIGURE 39.11 Caving technique

Slice drilling is begun from the hanging wall as soon as development is completed.

As shown in Fig. 39.9, the caved material is mined out as the slices are gradually backed out toward ore chutes and ramp entry cuts (Fig. 39.11).

All rock from the various phases of caving is drawn off via ore chutes–some 15 groups with five to seven chutes per group–to the main haulage levels. On the main haulage level, which is fully computer operated, the ore is transported by train to crushers that crush it to −100 mm (−4 in.). The ore is hoisted in skips to sorting plants, after which it is transported by train to the harbor for shipment to customers (Fig. 39.12).

The ventilation system (see Table 39.1), which is continuously being expanded as the number of diesel-driven machines increases, is built up around a basic system with some 50 shafts equipped with fans. This system is gradually extended as the workings reach greater depth (Fig. 39.13a). The total airflow capacity is around 2000 m^3/s (4.2 million cfm) with about

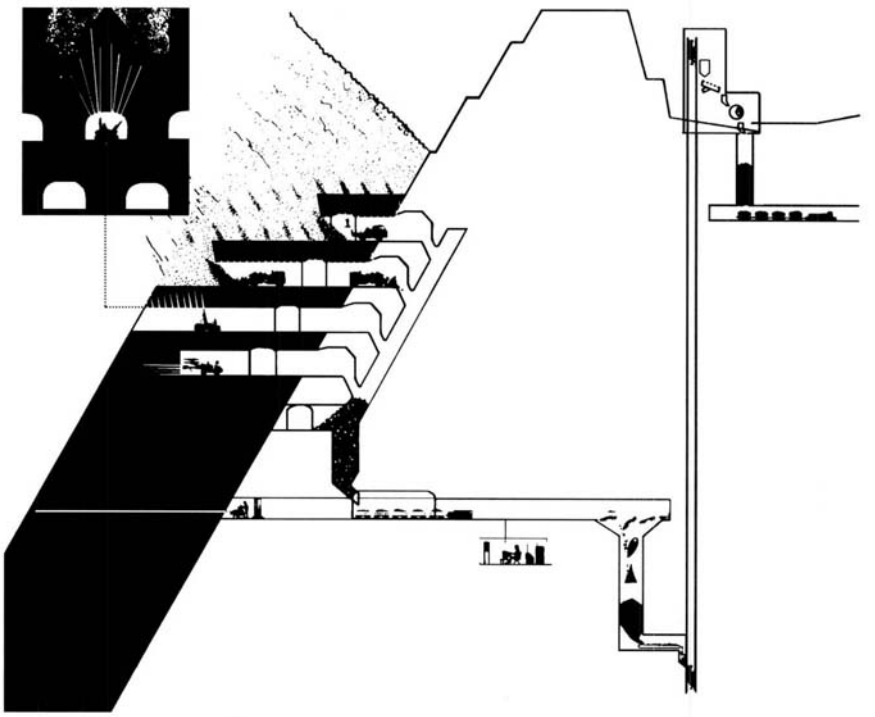


FIGURE 39.12 Main haulage level where ore is transported by train to crushers, then hoisted in skips to sorting plants, and finally transported by train to the harbor for shipment

TABLE 39.1 Basic data for sublevel caving system

Ramp	8 × 5 m, approx. 300 km
Slice height	12 m
Access drift	5.5 × 4.2 m
Crosscut	5 × 3.7 m
Pillar width	6 m, increased to 11.5 m below the 550-m level
Ore chutes	9 m^2, approx. 90 chutes to the 540-m level
Distance between ore chute groups	200 to 250 m
Distance between entry cuts	400 to 500 m
Inclination of drilled holes	1.4 rad (80°) to the horizontal and the hanging wall
Number of holes per fan	10 or 6
Burden	1.8 m
Drilled meters per fan	Approx. 90
Drilled meters per m^3 ore	0.5
Hole diamter, standard	51 mm
Large main ventilation stations	7 for intake air and 7 for exhuast air
Small main ventilation stations	25 for intake air and 25 for exhuast air
Installed fan capacity, main system	Approx. 10 MW
Installed fan capacity, local system	Approx. 7 MW

Metric equivalents: m × 3.2808 = ft; km × 0.62 = mile; m^2 × 1.196 = sq yd; m^3 × 1.307 cu yd; mm × 0.039 = in.; MW × 1340.4 = hp.

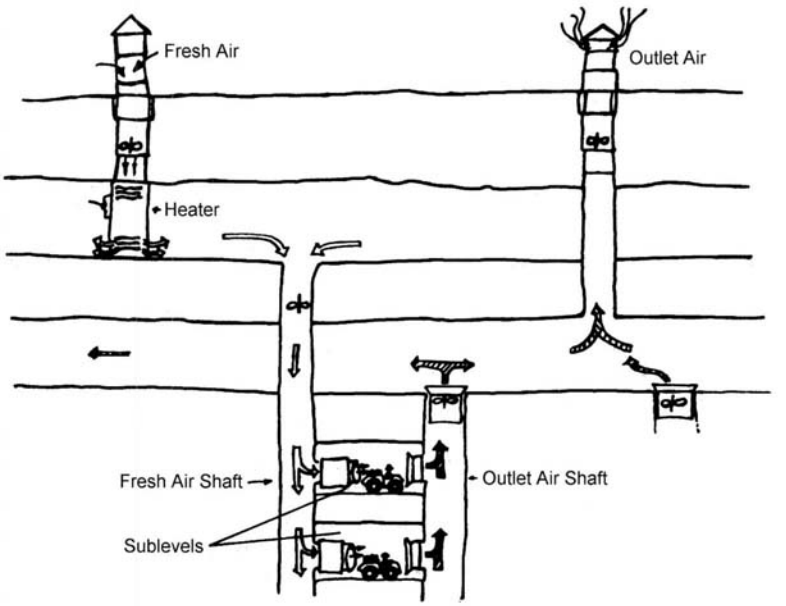


FIGURE 39.13a Basic ventilation system (courtesy B. Aaro, International Sublevel Caving Symposium, Stockholm, 1972)

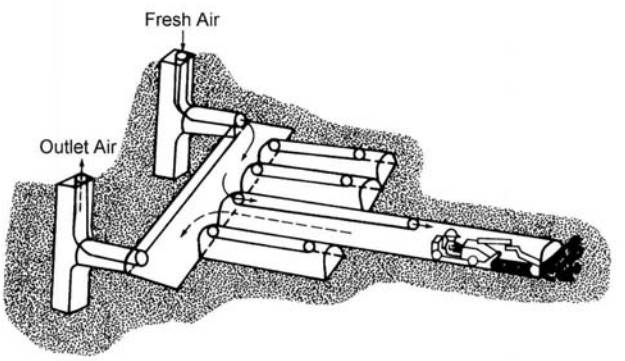


FIGURE 39.13b Crosscut ventilation (courtesy B. Aaro, International Sublevel Caving Symposium, Stockholm, 1972)

60 MW (80,400 hp) for heating the air. In addition, there are local systems (crosscut ventilation) at the working levels (Fig. 39.13b).

PLANNING PRINCIPLES

Five-year planning of mining operations is carried out on the basis of market and sales forecasts with consideration of planned investment volume and available capacity for the period. The five-year plan includes a shipment plan; production plans; and product outcome plans for mining activities, hoisting, sorting, and ore shipment. The one-year plan contains the same data as the

five-year plan as well as capacity and grade distribution per month. The five-year and one-year plans are prepared with the aid of a computer and are based on the latest drill hole data.

The weekly plan, or fine plan, is a flow plan that covers all stages of production from the ore in the ground to the customer. The weekly plan is run via a computer system and takes into account grade deviations, available mechanical and labor resources, and the shipment of finished products as well as the inventory situation. For coordination and operative production control, there is a flow control center where all information from production staffs both under and above ground is collected. This center can direct all production units and can correct for disturbances that occur in the flow of ore from the workings to the harbor.

The production unit is organized in three sections: working, main haulage levels, and mine service. Working consists of four areas with three production groups in each area. There is one production group for development and two for slice drilling and mucking. Each production group contains 40 to 50 workers and is responsible for drilling, mucking, blasting, and service. Main haulage levels include operation of the levels as well as maintenance of systems and rolling stock. Mine service includes work with ventilation, compressed air, electricity and water, and rock reinforcement.

PRACTICAL OPERATION

Main Development

Sublevel caving requires a relatively extensive amount of initial construction. Ramps and main levels as well as ore chutes and skip shafts are driven in the footwall in preparation for when the workings approach the current main level. The haulage level at 540 m (1800 ft) commenced operation in 1970 and the next main level at 775 m (2550 ft) commenced operation in 1977. Planning of the main level at 1000 m (3300 ft) was begun in 1976. With the exception of certain ventilation raises where raise boring is also employed, the type of equipment largely used in the construction work is the same as in production.

Drilling

Development drilling is done with Gardner-Denver AD 1025 and Atlas Copco Boomer 131 drill jumbos, which have diesel engines for transport. The rigs have three air drills and are controlled by one operator. The blasting round is drilled with parallel ϕ 43-mm ($1\frac{5}{8}$-in.) diam holes and one ϕ 100-mm (4-in.) opening hole. The advance per round is 3 m (10 ft), and productivity is approximately four rounds per two shifts (Fig. 39.14a, b, c).

Slice drilling is done with Atlas Copco Boomer 323 air-operated rigs equipped with three drills. The round is drilled in a fan or silo pattern. Silo-pattern drilling was introduced in order to improve caving, but it has been found that fan drilling is more appropriate at greater pillar widths. Each row contains ten 51-mm (2-in.) holes having a total length of 90 to 100 drilled m (300 to

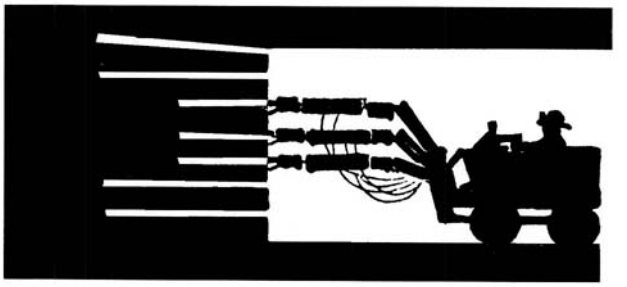


FIGURE 39.14a Drilling for drifting

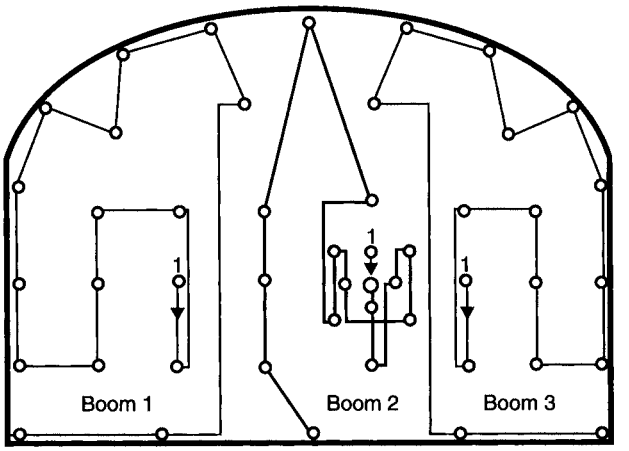


FIGURE 39.14b System for moving the three booms

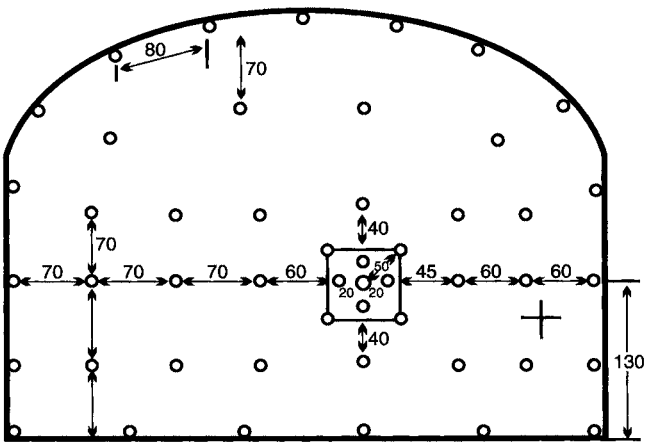


FIGURE 39.14c Parallel hole blasting round with large opening holes

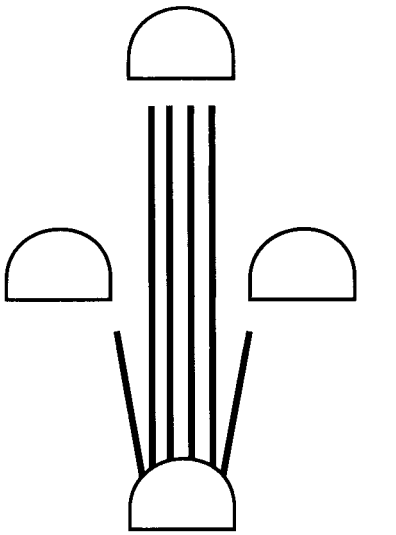


FIGURE 39.15a Silo drilling

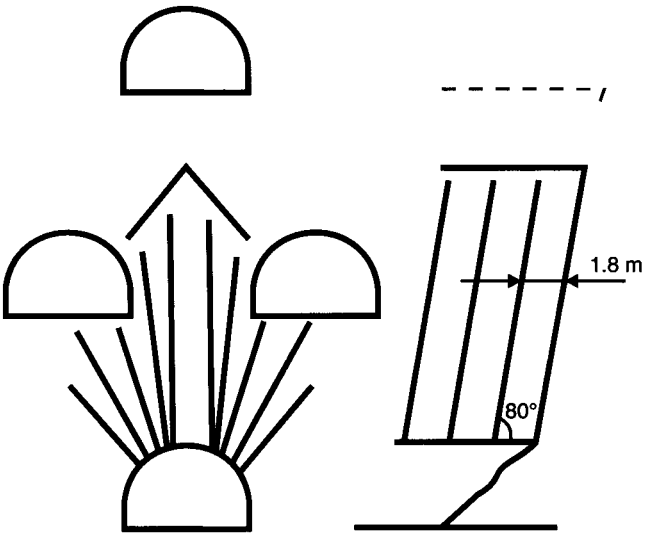


FIGURE 39.15b Fan drilling

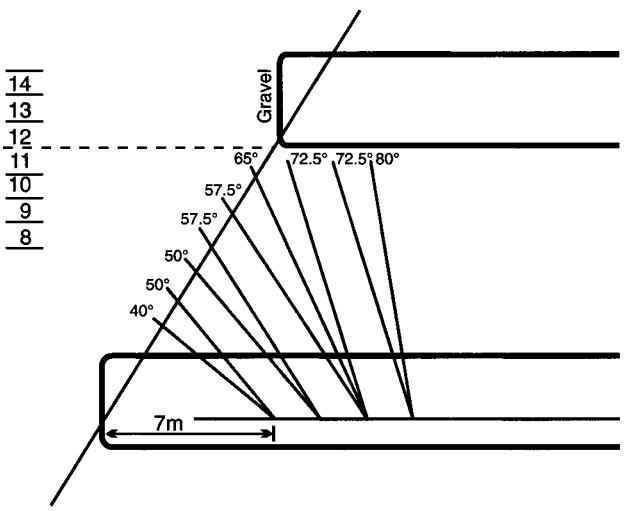


FIGURE 39.16 Slot fan drilling, metric equivalent: ° × 0.01745329 = rad

330 drilled ft). The productivity is 600 to 800 drilled m (1970 to 2620 drilled ft) per rig for two shifts (Figs. 39.15a, b, and Fig. 39.16).

Blasting

Ammonium nitrate–fuel oil (ANFO) and Kimit are used for blasting. Both explosives are manufactured in Kiruna and distributed via underground stations to charging trucks. There are special personnel within the production groups with specially built diesel-driven vehicles for pneumatic charging. The

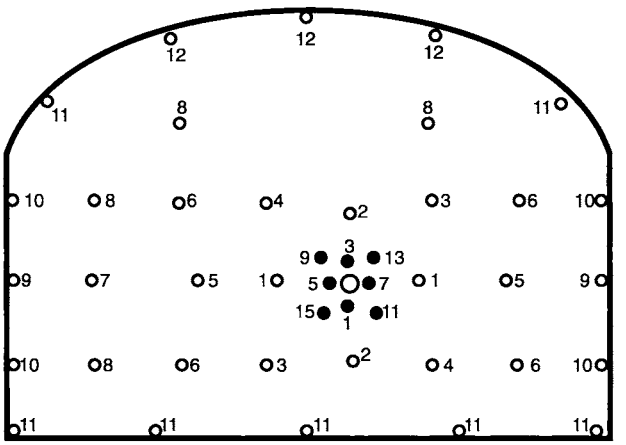


FIGURE 39.17a Delay blasting in drifting. Closed circle denotes short interval and open circle denotes one-half second interval delay.

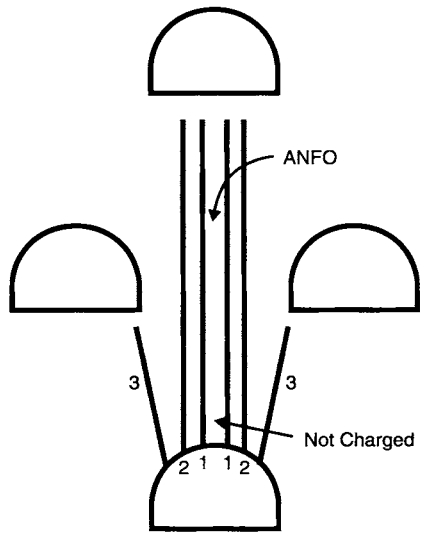


FIGURE 39.17b Silo fan charging with ANFO

rounds are detonated electrically after the end of the afternoon shift from central shooting stations. A development round contains about 200 kg (440 lb) of explosive, and a caving round about 150 kg (330 lb) of explosive. About 30 kg (66 lb) of Kimit is used in development rounds and as a primer in the caving round (Fig. 39.17a, b).

Loading and Transport

At the start of underground mining, loading was carried out by air-operated loaders. These were later replaced by electric Joy loaders. In order to boost capacity, diesel-driven front loaders and trucks were introduced in the 1960s. The current loading and haulage equipment consists of Cat 980Bs and Wagner ST-8s as well as a few Kiruna trucks. Kiruna trucks are used for long haulage distances. The productivity of the loading units is 1200 to 1500 t/shift (1300 to 1650 st per shift). Loading of the caved material continues to the point where the ratio of ore to waste is approximately 50%.

Haulage Levels

The required production capacity is approximately 100,000 t/d (110,000 st per day) of rock. In order to achieve this capacity, a trackbound system has been chosen for the main levels. The 540-m (1800-ft) level, which is now the large main haulage level, has a capacity of 25 Mt/a (27.5 million st per year). Rock is transported by 13 to 15 trains, each of which hauls 360 t (400 st) from some 90 ore chutes divided into 15 groups to seven centrally located stations. The trains are computer controlled and driverless. The trains can be loaded remotely if desired (and are in some cases). For grade-control purposes, the ore chutes are classified according to grade and phosphorus content. Chemical analyses are carried out in field laboratories (see Fig. 39.18a, b).

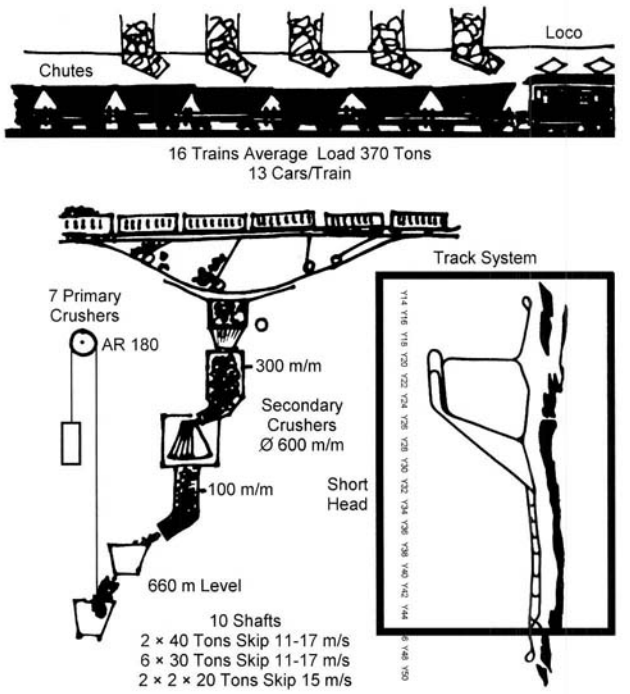


FIGURE 39.18a Transport system at Kiirunavaara on the 540-m (1800-ft) level. Capacity is 30 Mt/a (33 million st per year).

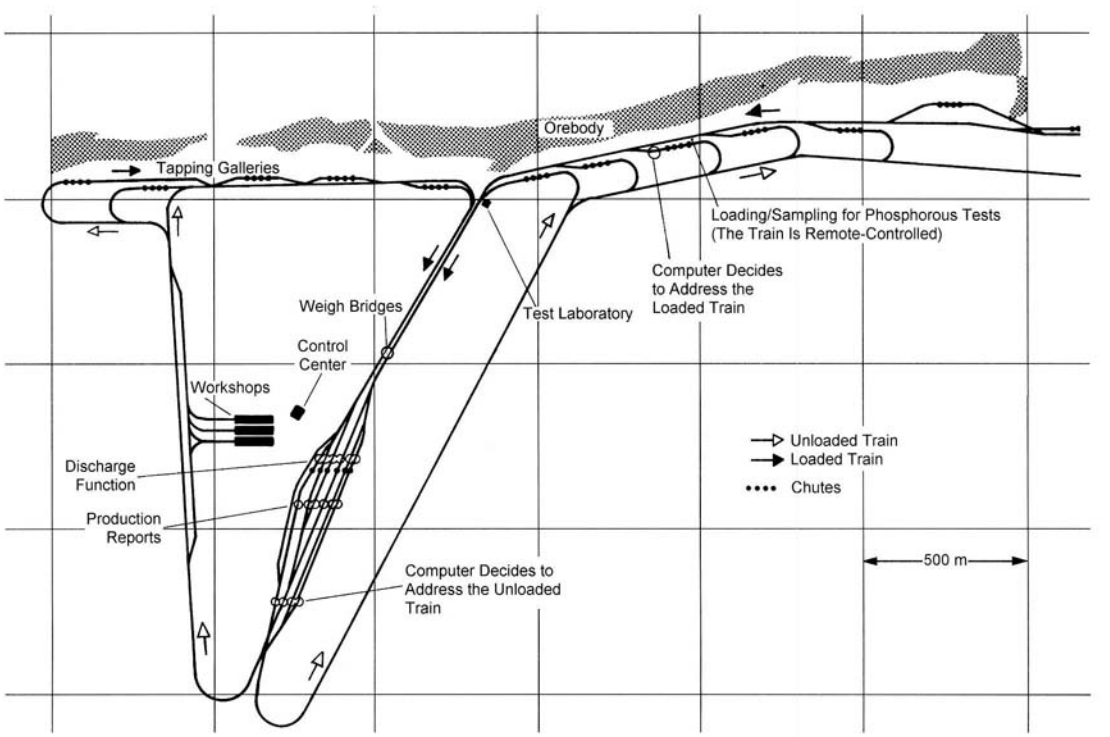


FIGURE 39.18b Transport system on the 775-m (2550-ft) main haulage level

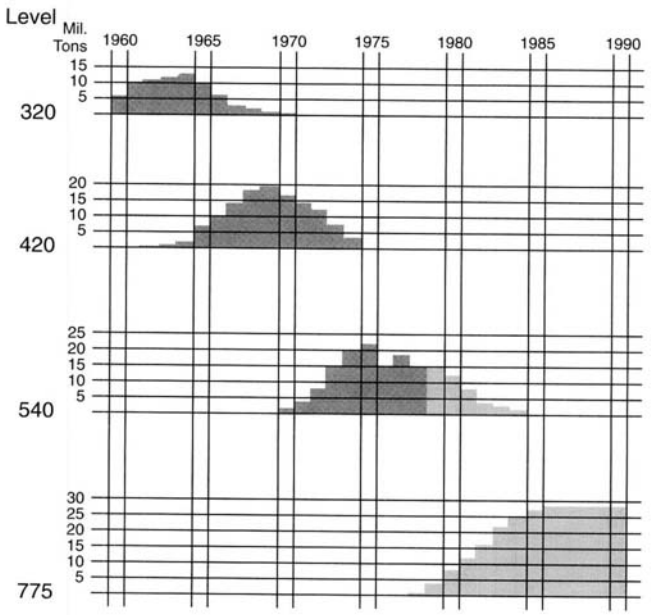


FIGURE 39.18c Millions of tons hauled on main levels in Kiruna

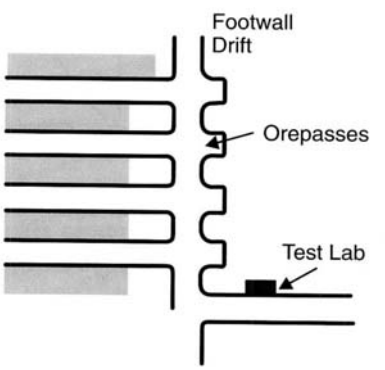


FIGURE 39.19 Quality testing in the drifts

The new haulage level at 775 m (2550 ft) (Fig. 39.18b), which is the fifth main level since the start, was commissioned in 1977 and will have a higher capacity—14 trains with a load of 460 t (500 st)—and more refined technology than level 540. Train loading and grade control are automated and will permit higher capacity than the 540-m (1800-ft) level. Fig. 39.18c shows a production chart for the main levels in the Kiirunavaara mine.

Grade Testing

Ore grade is centrally controlled in the production system. Grade test drilling, with holes spaced at intervals of 25 m (82 ft), is carried out prior to extraction in order to provide a basis for planning. During extraction operations, grade testing is carried out at the working levels (Fig. 39.19) so that ore grade in the

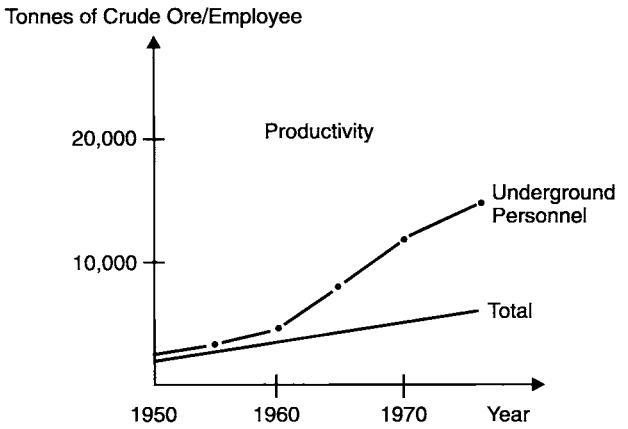


FIGURE 39.20 Productivity at LKAB (courtesy B. Hedberg, IXth World Mining Congress, 1976)

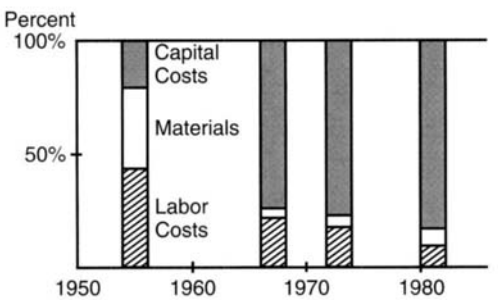


FIGURE 39.21a Cost distribution for main haulage level (courtesy B. Hedberg, IXth World Mining Congress, 1976)

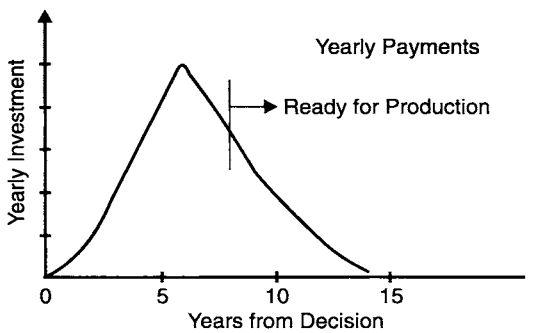


FIGURE 39.21b Spending rates as a function of time required for the development of a new main haulage level

ore chutes can be maintained at the desired level. On the main haulage levels, trains are checked to make sure they are headed for the right crusher and crusher pocket.

ECONOMIC ASPECTS

As was mentioned earlier, sublevel caving permits high flexibility, and extraction capacity can be increased by the use of additional machinery and personnel, providing that sufficient capacity exists in subsequent stages of the ore flow. By eliminating bottlenecks in the ore flow and by investing in machinery with better performance, productivity at LKAB has gradually been increased over the past two decades (Fig. 39.20). The influence of capital costs has therefore also increased during the same period. The main haulage levels can serve as an example (Fig. 39.21a, b). The relation between capital costs and other costs shows that capital costs have less influence at high capacities (Fig. 39.22).

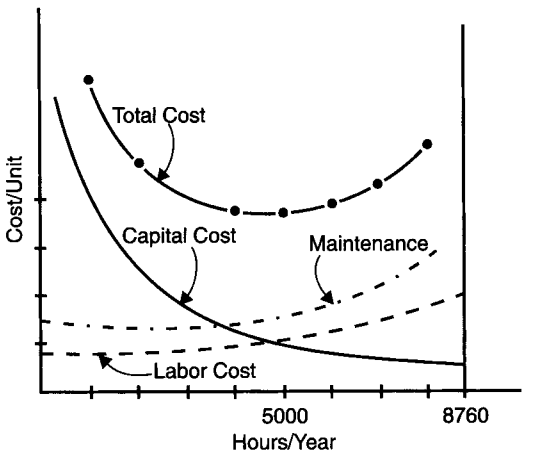


FIGURE 39.22 Cost relationships (courtesy B. Hedberg, IXth World Mining Congress, 1976)

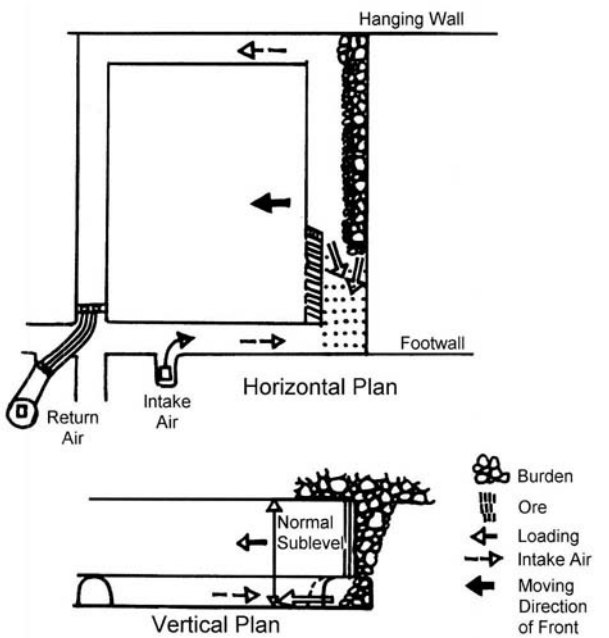


FIGURE 39.23 Longwall sublevel caving (courtesy Hanzagi, internal report, LKAB, 1970)

DEVELOPMENT AT LKAB

LKAB has been experimenting with modifications and improvements of the sublevel caving method for a number of years. Methods that have been tested include longwall sublevel caving, high sublevel caving, and sublevel shrinkage caving.

Longwall sublevel caving was intended to facilitate and improve ventilation, reduce development, and improve loading and rock mechanics. However, the method was found to be of no practical interest (Fig. 39.23).

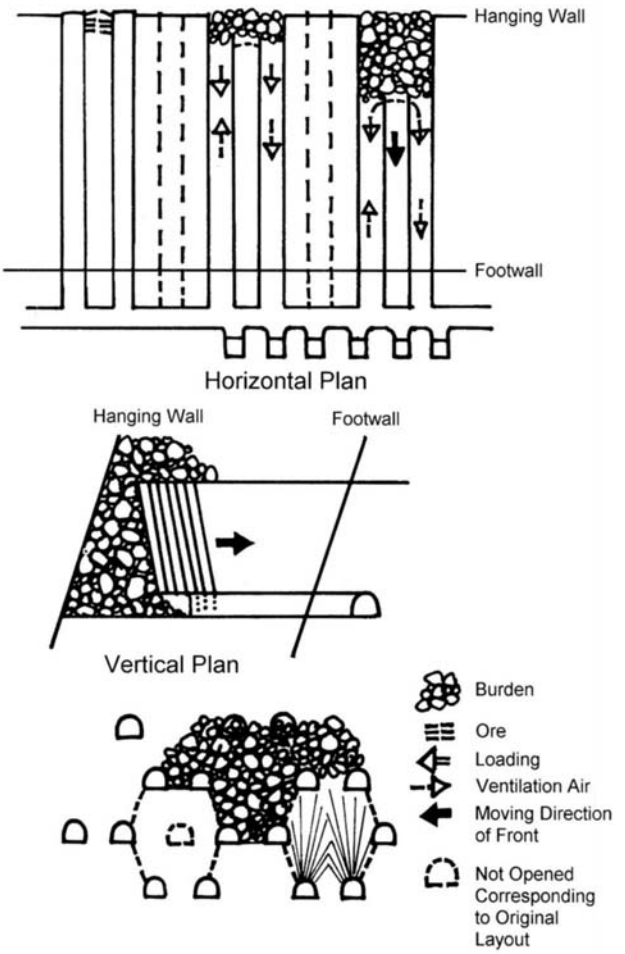


FIGURE 39.24 High sublevel caving (courtesy L. Hermansson, internal report, LKAB, 1974)

High sublevel caving was intended to facilitate ventilation, give a wide mucking face, and reduce development. However, the method was found to give rise to rock mechanics problems owing to the great width of the face (Fig. 39.24).

These methods have been tested in full-scale pilot trials and are ready for practical application. However, the results were not significantly better than those for present sublevel caving methods at Kiirunavaara.

The sublevel shrinkage caving method has been tested in the Kiirunavaara and Konsul ores, a smaller orebody south of Kiirunavaara. The method involves shooting a block of seven slice levels, after which the excess volume of ore is taken out and the rest is left behind. Thus, the breaking method is the same as in ordinary sublevel caving, but the mining operation is interrupted at an early stage and most of the ore is left behind. When the entire block has been blasted, all the rock is drawn out from a single loading level with careful control of the load intensity. The intention is to draw the entire block evenly downward and not allow waste to get mixed in with the ore

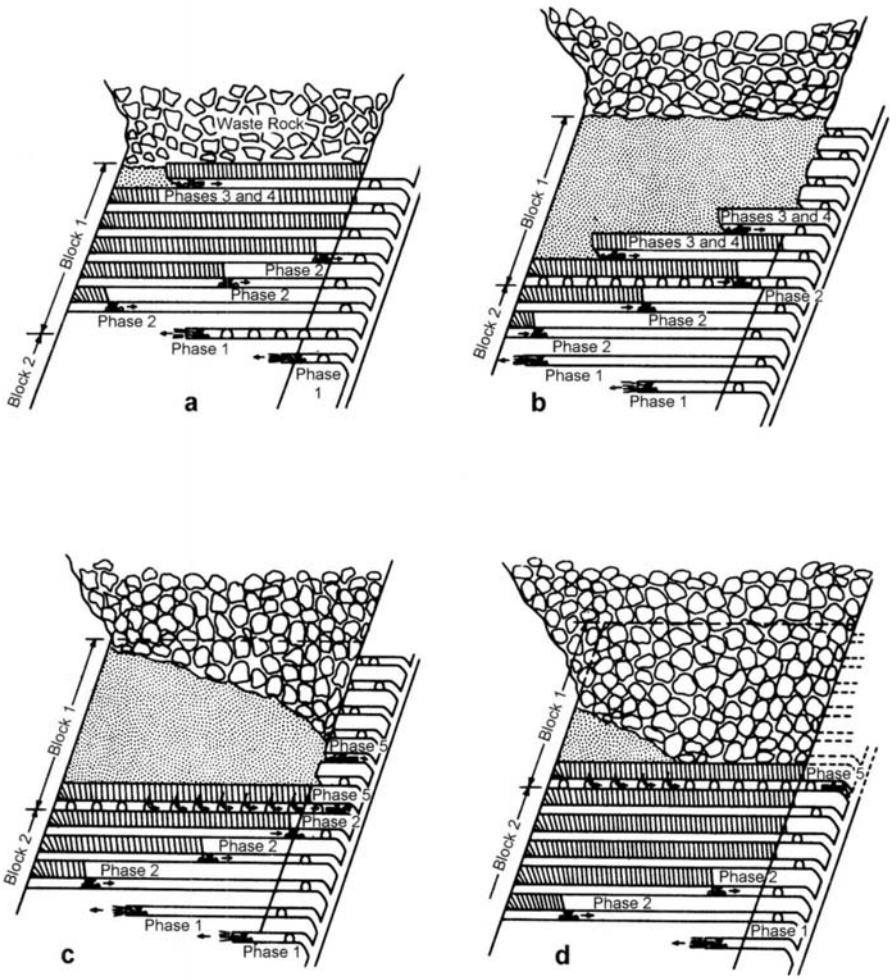


FIGURE 39.25 Sublevel shrinkage caving

(Fig. 39.25a, b, c, d). In the Konsul ore, mining is carried out lengthwise but using the same technique as in Kiirunavaara.

Development and testing of the next generation of machines is under way for the next main level, at 1000 m, and for the drilling and loading work. The new generation of drills being tested are electro-hydraulically driven. The loaders are diesel powered with higher capacity as well as electric loaders. Product development work, primarily on pellet products, is also being pursued within LKAB.

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CHAPTER 40

Planning Economics of Sublevel Caving

Dan Nilsson

INTRODUCTION

There are many mine planning factors in sublevel caving, as in the other mining systems, which when varied can substantially alter mining costs and profitability. This chapter addresses the following four topics with regard to economic optimization of sublevel caving: production planning, haulage level spacing, orepass spacing, and extraction cutoff. The costs used here are the right order of magnitude for 1979, and they should be considered only as examples. The objective is to demonstrate the techniques that can be used. In a real mine, the evaluations must be done using actual current costs and conditions. In addition, all examples are performed in the metric (SI) system without conversion so the calculations can be followed more easily.

It is difficult to predict exactly the costs involved, and therefore it is valuable to perform a sensitivity analysis to evaluate the effect of making incorrect assumptions. Because the mining industry is very capital intensive, the effect of the interest rate must also be studied.

PRODUCTION PLAN FOR THE IRON ORE DEPOSIT

Introduction

The first and most important thing to do before designing an underground mine is to establish a long-range production plan. Such a plan is necessary for all economic evaluations and should provide information about the lifetime of the mine, how much ore and waste must be handled per year, how much development per year is required, etc. An example is given in the following section.

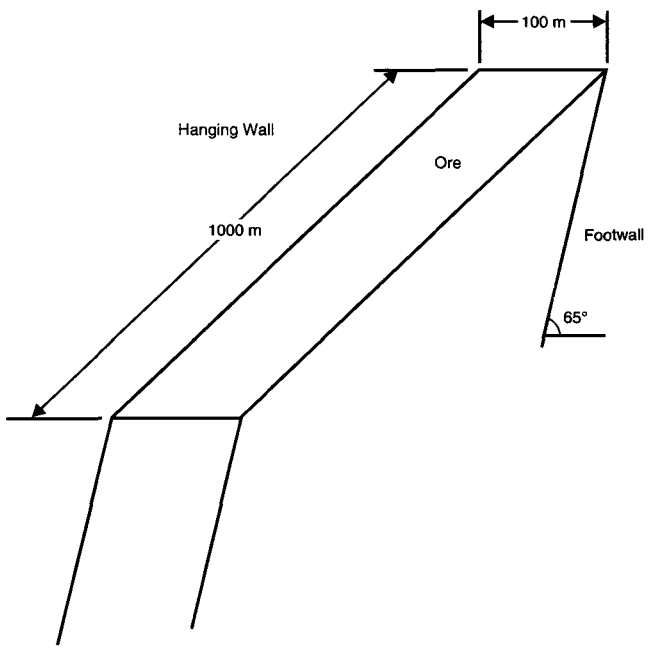


FIGURE 40.1 Simplified picture of orebody

Example

In an iron ore mine, sublevel caving is used. The iron content is 42%, and the mine supplies a pelletizing plant with an annual capacity of 3 million t/a. The orebody is shown in Fig. 40.1. The length of the orebody is 1000 m and the width is 100 m.

Each slice is 10 m high, and there are 10 m between crosscuts, each of which has an area of 20 m^2. The density of the ore is 3.5 t/m^3. The spacing between rings is 2 m, and the extraction is 100%. The iron content is 66% in the pellets and 6% in the tailings. The problem is to develop a detailed production plan. A typical sublevel caving sequence is shown in Fig. 40.2.

Solution

Amount of ore per meter of crosscut: $20 \times 3.5 = 70$ t

Number of crosscuts per slice = 100

Length of crosscuts per slice: $100 \times 100 = 10{,}000$

Ore from crosscuts per slice = $10{,}000 \text{ m} \times 20 \text{ m}^2 \times 3.5 = 700{,}000$ t

Number of sublevel caving rounds per slice: $10{,}000/2 = 5000$

Area for each blast in sublevel caving: $10 \times 10 - 20 = 80 \text{ m}^2$

Amount of ore per blast: $80 \text{ m}^2 \times 6 \text{ m} \times 3.5 = 560$ t

Extraction: 100% (see extraction curve Fig. 40.11 on page 736)

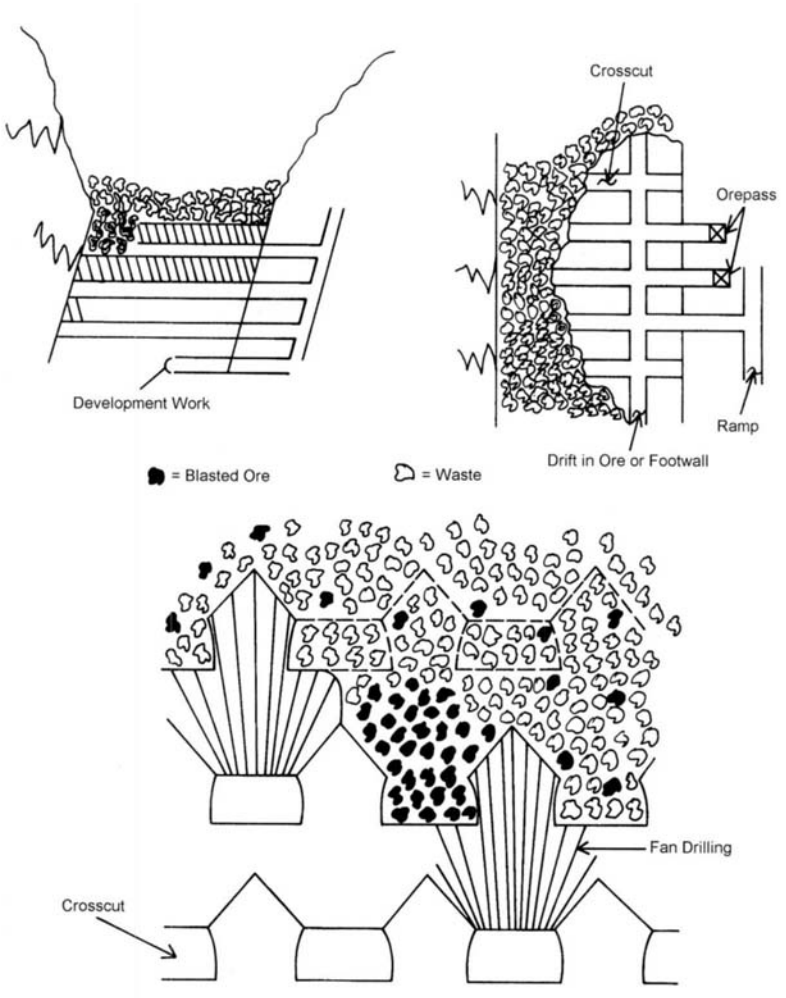


FIGURE 40.2 Typical sublevel caving sequence

Loaded ore per blast: 75%, or 420 t

Loaded waste per blast: 25%, or 140 t

Total: 560 t

Total amount of rock from each blast in the sublevel caving:

Ore: 560 t, of which 2 × 70 = 140 t are from development work and 420 t are from sublevel caving

Waste: 140 t

Total: 700 t

The ore needed per year is $\frac{3 \times (66 - 6)}{42 - 6}$ = 5 million t

Necessary number of blasts per year = 5,000,000/560 = 8929

Distance to develop per year = 8929 blasts per year × 2 m per blast = 17,858 m

Total amount/year:

Ore from development work: 17,858 m × 70 t/m	= 1.25 million/ta
Ore from sublevel caving: 8929 blasts per year × 420 tons per blast	= 3.75 million t/a
	5.00 million t/a
Waste rock dilution: 8929 blasts per year × 140 tons per blast	= 1.25 million t/a
Total amount to hoist	6.25 million t/a

The amount of ore from development work will increase a little if the horizontal drift is placed in the orebody and not in the footwall. In this example, the ore loss is 20%, and the waste rock dilution is also 20%. After taking the ore loss into account, the lifetime for 100 m of the orebody will be:

$$\frac{100\text{ m} \times 100\text{ m} \times 1000\text{ m} \times 3.5\text{ t/m}^2 \times 0.8}{5\text{ million t of ore/a}} = 5.6\text{ years}$$

The waste rock dilution means extra costs for handling 1.25 million t of waste rock per year.

OPTIMUM INTERVAL BETWEEN HAULAGE LEVELS

When mining underground, the orebody is often divided into a number of blocks or slices under which there probably is a transportation system with trains, trucks, conveyors, etc. When all the ore within such a block is extracted, a new transportation system must be built. Fig. 40.3 shows a cross section through a hypothetical mine with the main haulage levels, each with a crushing station.

Many times the configuration of the orebody means that there is no problem in deciding where to place the new level. Fig. 40.4 illustrates an orebody where the bottom is known, thus making it easy to decide where to build the new level. In other mines the mining method used decides where new levels are built, and sometimes the rock conditions determine this. But when the orebody is large it is possible to divide the rest of the orebody into different numbers of levels with different depths (see Fig. 40.5). If, for example, one has an orebody with a depth of 1000 m, it can be divided into ten levels of 100 m or five levels of 200 m. To place the level at the right depth means that one can minimize the capital costs. However, the depth also decides at what time construction work must begin and the amount of money that must be spent on the project (see Fig. 40.6).

Higher intervals between the levels also mean that more fines will be formed in the orepasses. If the mining company wants to sell lump ore, this is a disadvantage; if the ore must be milled on the surface, it is an advantage. Higher intervals mean higher hoisting costs during the beginning of the lifetime of the new level, a longer lifetime for orepasses, etc., and perhaps higher

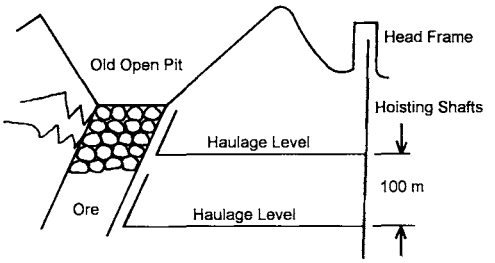


FIGURE 40.3 Cross section through orebody with main haulage levels

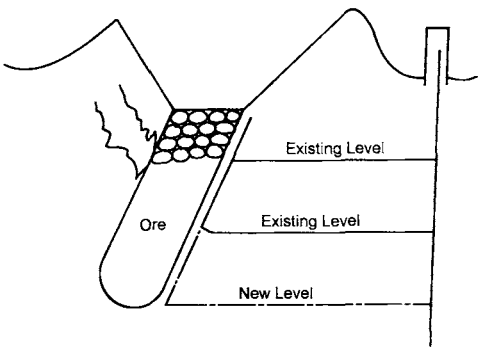


FIGURE 40.4 Cross section through orebody where the bottom is known, with new haulage level

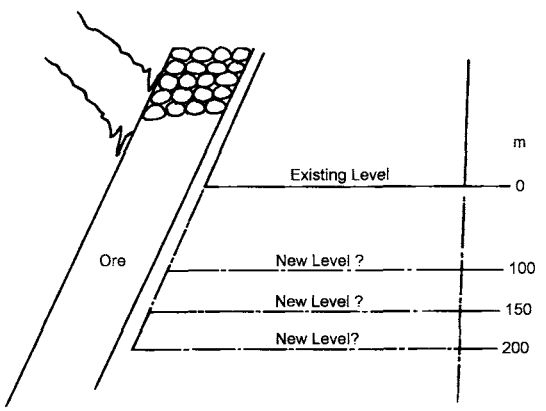


FIGURE 40.5 Cross section through orebody with alternative depths for new haulage level

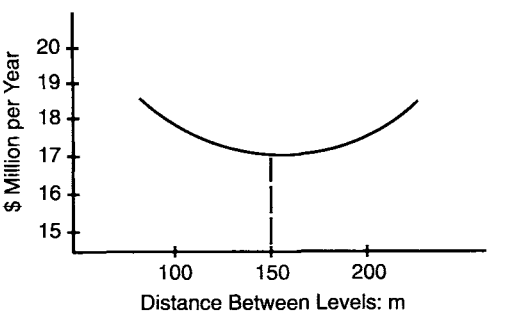


FIGURE 40.6 Capital costs for new haulage level as a function of the interval spacing

maintenance costs. Longer lifetime also means taking the risk of tying up the production for a longer time to the currently available technology.

The total investment for a new level in a mine can be divided into fixed investments (independent of depth) and those that are proportional to the depth. Examples of fixed investments are crushing stations, workshops, the horizontal haulage level, etc. Those costs proportional to depth are orepasses, ventilation shafts, hoisting shafts, ramps, etc.

If the expected future lifetime for the mine is short, it is necessary to calculate the capital value of the future income, minus future operational maintenance and investment costs, etc., for the whole mine as a function of level interval. If the expected future lifetime is long, it is often sufficient to compare the annual costs for different alternatives such as capital costs, etc.

Example

A deposit has a width of 50 m, a length of 2000 m, and a depth of 1000 m. The average ore density is 3.5 t/m^3. The mining company has to build a new haulage level and asks if the interval shall be 100, 150, or 200 m. The production is estimated at 4 million t/a of ore. The required fixed investment is estimated at $50 million, and the variable at $25 million per 100 m. The interest rate is 15%. The problem is to find the best depth for the new level. In this example, only the capital costs are studied.

Solution

Amount of Ore:

100 m: $50 \times 2000 \times 100 \text{ m} \times 3.5 = 35.0$ million t

150 m: $50 \times 2000 \times 150 \text{ m} \times 3.5 = 52.5$ million t

200 m: $50 \times 2000 \times 200 \text{ m} \times 3.5 = 70.0$ million t

Ore loss: 20%

Lifetime of the Level:

100 m: 35.0 million t × 0.8/4 million t/a = 7.0 years

150 m: 52.5 million t × 0.8/4 million t/a = 10.5 years

200 m: 70.0 million t × 0.8/4 million t/a = 14.0 years

Required Total Investment:

100 m: $50 million + 1.0 × $25.0 = $75.0 million

150 m: $50 million + 1.5 × $25.0 = $87.5 million

200 m: $50 million + 2.0 × $25.0 = $100 million

Average Capital Costs per Year:

100 m: $75.0 million × 0.240 = $18.0 million/year

150 m: $87.5 million × 0.195 = $17.1 million/year

200 m: $100 million × 0.175 = $17.5 million/year

On this basis, the optimal depth for the new level would seem to be about 150 m below the existing level.

OPTIMUM NUMBER OF OREPASS LOCATIONS PER LEVEL

The ore from development work as well as the ore and the waste rock dilution from the rounds are loaded and hauled in the crosscuts and drifts either directly to crushers located on the surface or underground or to bins or orepasses in the footwall. From the bottom of such orepasses the ore is normally hauled with trains or trucks to crushers.

One important problem is to determine the optimal number of groups of orepasses along the orebody (Fig. 40.7). To do this, one compares transportation

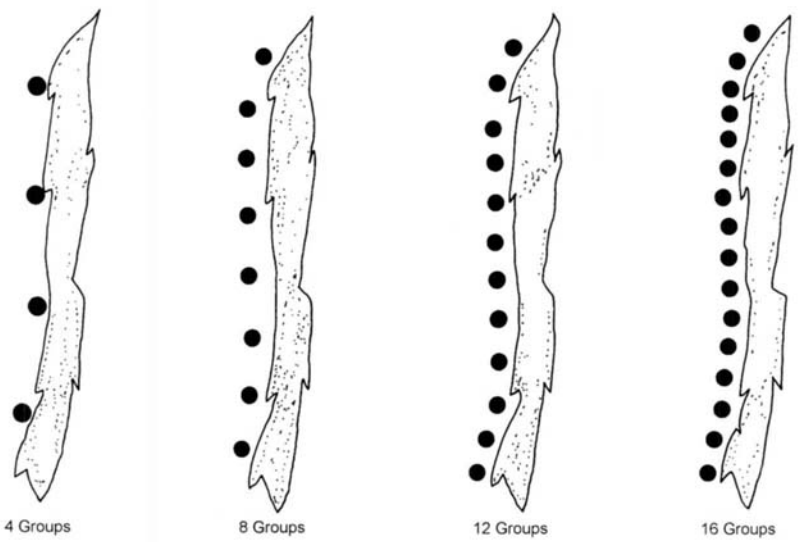


FIGURE 40.7 Number of groups with orepasses

costs with investment. More groups mean a higher investment but shorter haulage distances in the ore. The following is an example of how to do such an optimization.

Example

Consider a deposit with a length of 4000 m, a width of 100 m, and an inclination of 1.05 rad (60°). The deposit is to be mined by sublevel caving. Each slice is 15 m high. The costs for drifts on each slice is $300/m. The ore is loaded and hauled with front loaders to groups of orepasses, situated 10 m into the footwall. There will be on average five orepasses in each group because of the different ore grades to be handled. Each orepass costs $1000/m. Tunnel, loading equipment, etc. for trains at the bottom of each group cost $1 million. It will take 30 min to load a train. Each train carries 500 t. For loading and transport of the ore, front loaders with 10-t scoops will be used. Time and feasibility studies show that:

Average tons/scoop: 8.8

Speed on average: 10 km/hr

Time to load, unload, etc.: 1.5 min/scoop

Working time: 6 hr/shift, 2 shift/day, 5 days/week, 150 days/year

Price: $150,000

Lifetime: 5 years

Energy costs: $3/hr

Interest rate: 15%

Miners' cost: $20,000/year

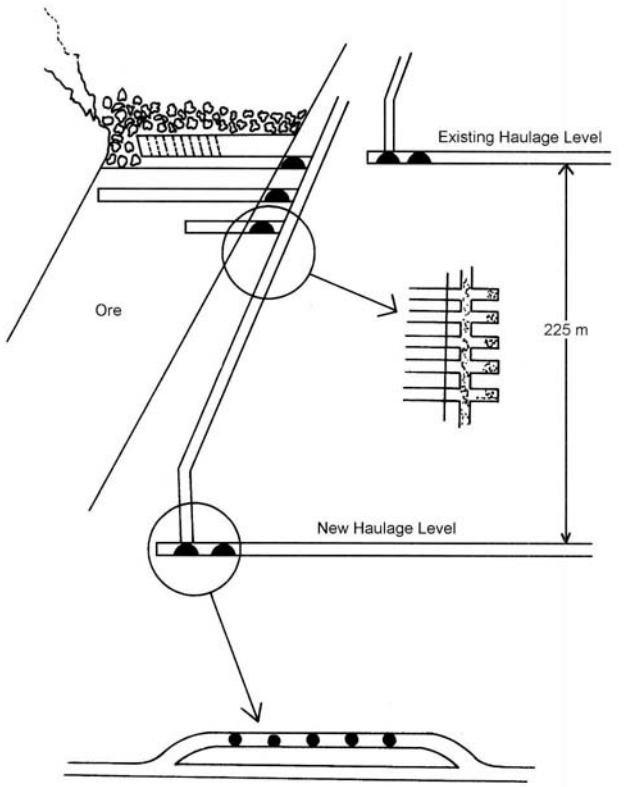


FIGURE 40.8 Orepass

The mine production is planned to be 20 million t/a. The company has to build a new haulage level and asks how many groups with orepasses they will build: 4, 8, 12 (Fig. 40.7), or 16. The new level (see Fig. 40.8) will have a height of 225 m, which means a lifetime of about 15 years. The optimum number of groups of orepasses is calculated in the following section.

Solution

Average Haulage Distance in the Ore:

$$\text{4 groups:} \quad 50 + 10 + \frac{4000}{4 \times 2 \times 2} = 310 \text{ m}$$

$$\text{8 groups:} \quad 50 + 10 + \frac{4000}{8 \times 2 \times 2} = 185 \text{ m}$$

$$\text{12 groups:} \quad 50 + 10 + \frac{4000}{12 \times 2 \times 2} = 143 \text{ m}$$

$$\text{16 groups:} \quad 50 + 10 + \frac{4000}{16 \times 2 \times 2} = 122 \text{ m}$$

Capacity for a Front-Loader:

Distance:	310 m	185 m	143 m	122 m
Fixed time, min	1.5	1.5	1.5	1.5
Haulage time, min, round trip	3.7	2.2	1.7	1.5
Total time, min, round trip	5.2	3.7	3.2	3.0
Loads per shift	69.2	97.3	112.5	120.0
Tons/shift	609	856	990	1056
Thousands of tons/year	183	257	297	317

Annual Cost for a Front-Loader:

Capital: $150,000 × 0.30	=	$ 45,000/year
Miners: 2 × $20,000	=	$ 40,000/year
Energy: 300 shifts × 6 hr × $3	=	$ 5,400/year
Maintenance: 300 shifts × 6 hr × $10	=	$ 18,000/year
		$ 108,400/year

Total Haulage Cost:

	4 groups	8 groups	12 groups	16 groups
Per ton, $	0.59	0.42	0.36	0.34
Per year, million $	11.8	8.4	7.2	6.8

Investment in Orepasses per Group: 5 × 270 m × $1000 = $1.35 million

The investment in tunnel, etc., under each group is given to $1 million, but will the capacity in such a tunnel be enough when the number of groups decreases?

The number of trains arriving to each group per shift will be:

4 groups: 20 million t/(500 t × 300 shifts × 4 groups) = 33 trains/shift

8 groups: 20 million t/(500 t × 300 shifts × 8 groups) = 17 trains/shift

12 groups: 20 million t/(500 t × 300 shifts × 12 groups) = 11 trains/shift

16 groups: 20 million t/(500 t × 300 shifts × 16 groups) = 8 trains/shift

But the capacity for one loading tunnel under each group will be 6 hr/0.5 hr per train = 12 trains per group.

Therefore, the capacity under each group will have to double if the number of groups is eight and triple if the number is only four. This can be difficult to

achieve in reality. (In reality, a computer simulation is recommended to study what will happen with the total capacity on the haulage level if the number of orepasses is decreased).

The total investment will be:

4 groups: $\$4 \times (1.35 + 3 \times 1.0)$ million = \$17.4 million

8 groups: $\$8 \times (1.35 + 2 \times 1.0)$ million = \$26.8 million

12 groups: $\$12 \times (1.35 + 1 \times 1.0)$ million = \$28.2 million

16 groups: $\$16 \times (1.35 + 1 \times 1.0)$ million = \$37.6 million

The capital cost (15 years, 15%):

4 groups: $\$17.4 \times 0.17$ = \$3.0 million per year

8 groups: $\$26.8 \times 0.17$ = \$4.5 million per year

12 groups: $\$28.2 \times 0.17$ = \$4.8 million per year

16 groups: $\$37.6 \times 0.17$ = \$6.4 million per year

It is also necessary to examine whether the capacity for unloading the front-loaders on each slice is high enough.

Assuming that 50% of the ore production will be extracted from one slice requires the following number of front-loader trips to each group per shift:

4 groups: 10 million t/(8.8 t × 300 shifts × 4) = 947

8 groups: 10 million t/(8.8 t × 300 shifts × 8) = 473

12 groups: 10 million t/(8.8 t × 300 shifts × 12) = 315

16 groups: 10 million t/(8.8 t × 300 shifts × 16) = 236

Assuming that unloading and other movements in front of the orepasses take 1 min per front-loader and that not more than one front-loader can unload at the same time, then the maximum unloading capacity of each group will be 360 front-loaders per shift. (In reality, queuing problems can also arise, which will decrease this capacity.)

Therefore, special arrangements must be made on each slice if the number of groups is four or eight. A solution is shown in Fig. 40.9. The length of the drifts at each slice per group will be: 4 groups, 150 m; 8 groups, 100 m; 12 groups, 50 m; and 16 groups, 50 m.

The cost per slice will be:

4 groups: 4×150 m × \$300 = \$0.22 million

8 groups: 8×100 m × \$300 = \$0.24 million

12 groups: 12×50 m × \$300 = \$0.18 million

16 groups: 16×50 m × \$300 = \$0.24 million

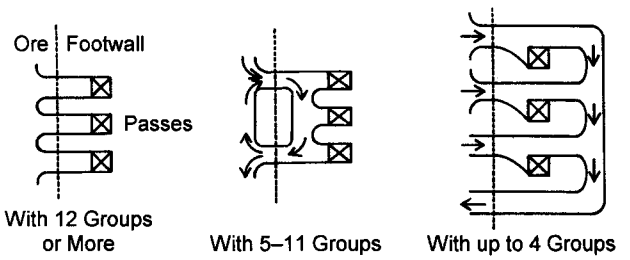


FIGURE 40.9 Design of discharging stations

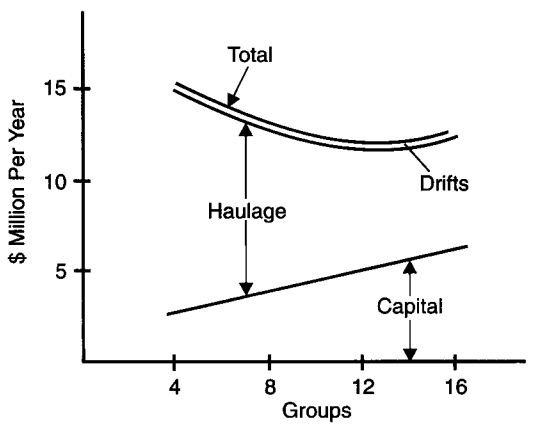


FIGURE 40.10 Optimal number of groups of orepasses

Each year one slice will be developed. One can now finally sum annual costs in millions of dollars:

	Capital	Haulage	Drifts	Total
4 groups	3.0	11.8	0.2	$15.0 million
8 groups	4.5	8.4	0.2	$13.1 million
12 groups	4.8	7.2	0.2	$12.2 million
16 groups	6.4	6.8	0.2	$13.4 million

The alternative with 12 groups seems to be the best. The result is shown in Fig. 40.10. It is also possible to take other factors into account such as ventilation and investment and operating expenses on the haulage level.

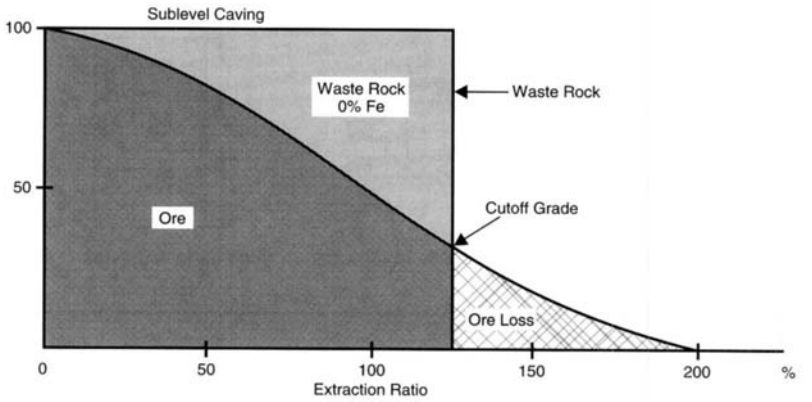


FIGURE 40.11 Extraction curve for sublevel caving round

TOTAL OPTIMUM EXTRACTION

The low costs for open-pit mining in comparison with underground mining make it possible to mine deposits and parts of deposits that would not be profitable to mine by underground methods. Even in those parts of a deposit that are profitable to mine by underground methods, the ore losses will be higher than in an open pit.

The ore losses underground depend upon the mining method used. Cut-and-fill methods often mean low losses, whereas room-and-pillar mining makes it necessary to leave ore in supporting pillars. Often 20 to 40% has to be left.

With sublevel caving the proportion of ore loaded is initially much higher than that of waste rock (see Fig. 40.11). As more rock is removed, the proportion of waste rock increases, while at the same time an increasing proportion of the total ore content can be recovered. Through an economic optimization, the break-even ore loss can be found.

Example

An iron ore deposit has an iron content of 65% and an area of 50 × 4000 m = 200,000 m^2. The density is 4.5 t/m^3. The deposit is mined with sublevel caving. Each slice is 10 m. There are 10 m between crosscuts, each of which has an area of 20 m^2. When loading, front-loaders with 10-t buckets are used. The extraction curve is shown in Fig. 40.11. Each ring has a 2-m burden. The mined ore is hauled to an orepass, after which it is transported by train on a main haulage level, and then hoisted to the surface. The mining company builds main haulage levels on 200-m intervals. The waste rock dilution is removed from the ore by magnetic separation. The ore is then sold as lump ore and fine. The mining company can sell 10 million t/a of ore. Find the optimal extraction if: a main haulage level costs $100 million; a crosscut costs

$300/m; a round costs $300 to drill and blast; loading, transportation, hoisting, and ore treatment cost $5/t of rock (in reality, waste rock will probably cost more than ore because of the lower density, etc.); and the price for the ore products is $10/t or $20/t.

Solution

Number of crosscuts/slice: 400

Length of crosscuts/slice: 400 × 50 m = 20,000 m

Ore from crosscuts/slice: 20,000 m × 20 m^2 × 4.5 = 1.8 million t

Number of blasts/slice: 20,000/2 = 10,000

Area of each blast in sublevel caving: 10 × 10 m − 20 m^2 = 80 m^2

Amount of ore/blast in sublevel caving: 80 m^2 × 2 m × 4.5 = 720 t

Extraction alternatives for evaluation: 50%, 100%, 150%, and 200%

Ore from each blast in sublevel caving: for 50% extraction, 324 t; 100% extraction, 540 t; 150% extraction, 684 t; and for 200% extraction, 720 t

Waste rock from each blast in sublevel caving: for 50% extraction, 36 t; 100% extraction, 180 t; 150% extraction, 396 t; and for 200% extraction, 720 t

Ore from development/meter: 20 m^2 × 4.5 = 90 t

Total amount of ore/blast in the sublevel:

	Development	Sublevel	Total (t)
50% extraction	180	324	504
100% extraction	180	540	720
150% extraction	180	684	864
200% extraction	180	720	900

Number of blasts/year in sublevel caving

50% extraction	10,000,000/504 = 19,841
100% extraction	10,000,000/720 = 13,889
150% extraction	10,000,000/864 = 11,574
200% extraction	10,000,000/900 = 11,111

Distance to develop/year

50% extraction:	2 × 19,841 = 39,682 m
100% extraction:	2 × 13,889 = 27,778 m
150% extraction:	2 × 11,574 = 23,148 m
200% extraction:	2 × 11,111 = 22,222 m
Ore/meter of crosscut:	10 × 10 m × 4.5 = 450 t

Ore losses:

50% extraction: $1 - 504/2 \times 450 = 44\%$

100% extraction: $1 - 720/2 \times 450 = 20\%$

150% extraction: $1 - 864/2 \times 450 = 4\%$

200% extraction: $1 - 900/2 \times 450 = 0\%$

Total amount to load in each blast in sublevel caving:

50% extraction: 324 t ore + 36 t waste = 360 t

100% extraction: 540 t ore + 180 t waste = 720 t

150% extraction: 684 t ore + 396 t waste = 1080 t

200% extraction: 720 t ore + 720 t waste = 1440 t

Total annual amount of ore and waste rock:

	Ore			
	Development	Sublevel	Waste Rock	Total, Million t
50% extraction	3.57	6.43	0.72	10.72
100% extraction	2.50	7.58	2.50	12.50
150% extraction	2.08	7.92	4.58	14.58
200% extraction	2.00	8.00	8.00	18.00

Waste rock dilution:

50% extraction: 0.72/10.72 = 6.7%

100% extraction: 2.50/12.50 = 20.0%

150% extraction: 4.58/14.58 = 31.4%

200% extraction: 8.00/18.00 = 44.4%

Number of blasts in sublevel caving/200 m: 20 slices × 10,000 blasts/slice = 200,000.

Lifetime for main haulage level (total number of blasts per 200 m/annual need; can, of course, also be estimated another way):

50% extraction: 200,000/19,841 = 10.1 years

100% extraction: 200,000/13,889 = 14.4 years

150% extraction: 200,000/11,574 = 17.3 years

200% extraction: 200,000/11,111 = 18.0 years

Annual development costs:

50% extraction: 39,682 m × \$300/m = \$11.9 million/year

100% extraction: 27,778 m × \$300/m = \$8.3 million/year

150% extraction: 23,148 m × \$300/m = \$6.9 million/year

200% extraction: 22,222 m × \$300/m = \$6.7 million/year

Annual costs for drilling and blasting in sublevel caving:

50% extraction: 19,841 blast/year × $300/blast = $5.9 million/year

100% extraction: 13,889 blast/year × $300/blast = $4.2 million/year

150% extraction: 11,574 blast/year × $300/blast = $3.4 million/year

200% extraction: 11,111 blast/year × $300/blast = $3.3 million/year

Annual costs for loading, etc., in sublevel caving:

50% extraction: (6.43 + 0.72) million t/a × $5/t = $35.8 million/year

100% extraction: (7.58 + 2.50) million t/a × $5/t = $50.4 million/year

150% extraction: (7.92 + 4.58) million t/a × $5/t = $62.5 million/year

200% extraction: (8.00 + 8.00) million t/a × $5/t = $80.0 million/year

Total annual costs:

50% extraction: $53.6 million/year

100% extraction: $62.9 million/year

150% extraction: $72.8 million/year

200 % extraction: $90.0 million/ year

To assess annual income, investigate the effects of ore values at $10/t and $20/t, which gives $100 million or $200 million/year, respectively.

Gross annual profit

	$10/t	**$20/t**
50% extraction	$46.4 million/year	$146.4 million/year
100% extraction	$37.1 million/year	$137.1 million/year
150% extraction	$27.2 million/year	$127.2 million/year
200% extraction	$10.0 million/year	$110.0 million/year

Present value of profit (15%):

	$10/t	**$20/t**
50% extraction	$232.9 million/year	$734.9 million/year
100% extraction	$214.1 million/year	$791.1 million/year
150% extraction	$165.1 million/year	$772.1 million/year
200% extraction	$ 61.3 million/year	$674.3 million/year

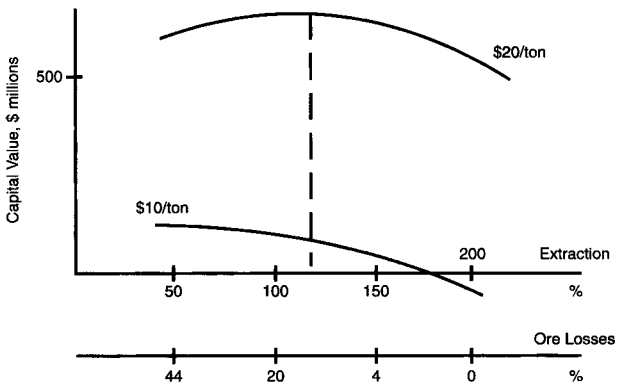


FIGURE 40.12 Capital values for different extraction ratios

Net present value:

	$10/t	$20/t
50% extraction	132.9	634.9
100% extraction	114.1	691.1
150% extraction	65.1	672.1
200% extraction	–38.7	574.3

At the lower price an extraction of only about 50% (or perhaps lower) seems to be the best. That should mean an acceptable ore loss of 44%! The higher price results in an optimal extraction of about 120%, which means an acceptable ore loss of 10 to 15%. However, Fig. 40.12 reveals that the curves have very flat peaks. Because the extraction curves can be different for different blasts, in reality it can be difficult to determine the optimal extraction.

In addition, the acceptable ore loss depends on how large the ore reserves are. If a company has a very large amount of ore at its disposal, it can accept higher ore losses than if the ore reserves are limited. A large ore reserve means that an ore loss today will result in a loss of income far in the future. Such a loss has very small value today. A small ore reserve means that this loss will occur earlier and is therefore more important to take into account. Unlimited haulage, hoisting, and ore treatment capacity mean that it can be profitable with a higher extraction than if these capacities are bottlenecks.

It must be kept in mind that costs used are hypothetical. Rapid price increases in recent years have made many of the actual costs used inaccurate.

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SECTION 5

Block and Panel Caving

CHAPTER 41

Block Caving: General Description

Ray L. Tobie and Douglas E. Julin

INTRODUCTION

Block caving is a distinct caving method applied mostly to large, massive orebodies because of its inherent low cost and high production capabilities. Areas of sufficient size are removed by undercutting so that the mass above will cave naturally. Drawing of the caved ore at the bottom of the ore column causes the caving action to continue upward until all of the ore above the undercut level is broken into sizes suitable for handling. When properly applied, block caving results in a lower mining cost per ton than any other underground method.

There are three distinct forms of block caving: (1) dividing the horizontal area into rectangular or preferably square or nearly square blocks, drawing evenly over the entire area to maintain an approximately horizontal plane of contact between broken ore and caved capping; (2) dividing the horizontal area into panels across the orebody, retreating by undercutting manageable areas from one end of the panel to the other and maintaining an inclined plane of contact between the broken ore and caved capping (thus the name panel caving); and (3) no division of horizontal area of the orebody into definite blocks or panels (this is termed mass caving). Undercutting may be from wall to wall, retreating from one end of the orebody to the other, maintaining an inclined plane of contact between the broken ore and caved capping. The total active caved area is determined by the size of block that will not produce undue stress on workings below the undercutting level and by total production requirements. This type of operation is also referred to as panel caving at some properties.

SUITABLE OREBODIES

Block caving in its various forms is applicable to deposits of various shapes and ores of various strengths. Its success is governed by rigid requirements and limitations. In unsuitable deposits or where improperly employed, the loss of ore may exceed that of any other mining method. Good planning, systematic work procedures, careful supervision, and good judgment contribute to its success.

Ore Characteristics

Included as a necessary characteristic in an orebody suited to a successful block-caving operation is a proper fracture pattern. Ore hardness should be another governing factor, and the toughness or softness of the ore should be considered. There must be sufficient horizontal area available for expansion of the undercut, if necessary, to start the caving process. Large, massive orebodies usually meet these conditions. Veinlike deposits must be wide and dip rather steeply to conform to the block-caving limitations. The capping must be of such a nature as to cave when the underlying ore is being removed because the weight of the overburden aids in crushing the ore. The most favorable capping characteristic is that it breaks into coarse pieces and resists attrition as the block is drawn. Soft capping, by breaking into fine pieces and sifting into the ore or channeling through the ore to the drawpoints, can seriously dilute the ore and hinder the extraction of clean ore from the blocks.

Types of Ore

In block caving, a fairly uniform distribution of values in the ore is necessary. Grade values may range from low grade to high grade, but most often the system is applied to low-grade ores. Wide veins, thick beds, or massive deposits of homogeneous ore overlain by ground that will cave constitute suitable orebodies. The ore must be such that it can be supported while blocks are being developed and undercut but breaks up readily when caved. Some applications include porphyry copper, disseminated molybdenite, hematite in the Lake Superior iron district, limonite deposits in northern Spain, the diamond mines of South Africa, asbestos and nickel in Canada, and massive magnetite in Pennsylvania.

Size and Shape

Outline of the orebody should be fairly regular, and the sides of the orebody should dip steeply. It may not be economical to mine small portions of ore extending into the walls of the deposit, and low-grade inclusions in the ore cannot be left unmined. As mining progresses, some dilution of the ore by low-grade or barren capping is inevitable and, unavoidably, some clean ore is left between drawpoints.

Determining Cavability

The use of rock mechanics is sometimes advantageous in helping to determine the cavability of an ore deposit. The U.S. Bureau of Mines has information and testing ability that can assist in determining the cavability of ore

TABLE 41.1 Drawpoint spacing

	% of Ore Size, ft*				
Mine	+5	2–3	1–2	–1	Drawpoint Spacing, m
Creighton	30	30	40	—	9 × 12
Thetford	20	25	25	30	7.6 × 7.6
Mather	5	10	15	70	4 × 8.5
Miami Copper	—	—	—	—	5 × 5.7
El Salvador		Varies			43.6–148.6†
San Manuel		Fine ore			4.6 × 5.3
Climax	7	24	23	46	10 × 10.4
Grace	50	20	20	10	6 × 9
Cornwall	10	20	20	50	7.6 × 12
Urad	40	15	15	30	9 × 9
Henderson	—	—	—	—	12 × 12
De Beers	—	—	—	—	4.6 × 6.8

Source: *SME Mining Engineering Handbook*, 1973.

*Metric equivalent: ft × 0.3048 = m.

†43.6 m for fine ore; 148.6 m for coarse ore.

types. Several competent consulting organizations knowledgeable in rock mechanics can also assist in evaluating the cavability of an orebody. Some recent work has been done to better define the cavability index of various orebodies; an example of how this was determined is discussed by McMahon and Kendrick (1969).

The intensity of the fracture pattern is a critical parameter to be analyzed. The question to be answered is, "How intense must the fracture pattern be?" Table 41.1 presents the sizes of rock fragments that appear at the drawpoint of various block-cave mines around the world. These are indicative of the fracture-pattern spacing, and, as can be seen, they vary substantially from the almost sandy material at the Mather mine to fairly sizable blocks at the Grace mine.

Several sets of fractures are essential to promote good caving. Ideally, two vertical sets at nearly right angles to each other and a third set nearly horizontal are required to insure a good caving orebody. Because initial caving of an ore mass is almost always by the action of gravity acting on these planes of weakness, it is doubtful that any orebody would cave without them unless the rock approached the consistency of sand. One or even two sets of vertical fractures probably would not allow caving unless the rock had an extremely low tensile strength. Nearly horizontal fractures would probably act as a bedded deposit and, even though the ore would break in tension or shear, the blocks would probably be too large to handle through the drawpoints and would certainly result in high secondary blasting costs.

In the initial evaluation of the orebody for mining methods, the deposit should be studied for several sets of fractures oriented in such a fashion that the orebody will break into sizes that can be readily handled at the drawpoint. The more closely spaced the fractures, the more readily the ore will cave. In the ideal orebody, all the fragments will break small enough to pass through the drawhole. This type of orebody is probably nonexistent so the next best thing is one in which a sufficient percentage of ore fragments will pass through the drawhole so that secondary blasting costs do not become exorbitant. At least 50% of the ore fragments should break to 1.5 m (5 ft) dimensions or less. The highest secondary blasting costs seem to be incurred during the first 30% of the draw. During this time, breaking depends almost entirely on the action of gravity on the fracture planes. As the ore column is drawn down, stresses are placed on the individual blocks within the ore column, and attrition helps to reduce the size of the ore fragments by tensile and shear stresses.

The fracture patterns in an orebody can be evaluated by two methods: from diamond-drill core or by studying development openings in the orebody. By either of these methods, the orientation of the fractures and the interval between fractures can be determined. The only method of doing this, as far as is known, is by visual inspection and physically counting the fractures within given increments, usually 3 m (10 ft). Two methods have been used. One method, rock quality designation (RQD), is to count all core fragments 100 mm (4 in.) or larger. The other is to count only the fractures that are geologic features. All fresh conchoidal fractures and separations due to disking in the hole would be ignored. Either method is acceptable. The first method is somewhat less tedious.

Choice Among Various Forms of Block Caving

Whether an orebody should be mined in blocks, panels, or as a mass depends to a great extent on rock competency. Where the ore is soft or highly fractured, breaks into fine particles, and a fairly high rate of draw can be attained, the block system can be used. In this type of rock, excessive weight may damage the workings if the mining block size is too large. This system is no longer widely used. When the fracture pattern is somewhat more dispersed and the rock in situ has more strength or when the expansion of a block is limited in one direction, as in a steeply dipping vein, a panel caving system may be more suitable. The necessity of leaving a pillar between panels depends upon the strength of the rock and whether the waste in the block, after having been drawn to extraction, will reconsolidate to permit recovery from the pillar. If the orebody is massive and fairly competent, having a fairly widely spaced fracture pattern, then mass caving should be considered. In this type of orebody, it may be necessary to undercut a large horizontal area before caving will start, and some type of boundary cutoff is recommended. Experience at other properties could guide the initial work, but the final plan for a large block-cave operation is attained by experimenting under local conditions. An additional choice must be made among a full-gravity grizzly system, a slusher system, or a system using mobile equipment.

TABLE 41.2 Daily production rates

	Daily Production, t*
Cornwall No. 3	1814
Cornwall No. 4	3629
Grace	6350
Mather	7676
San Manuel	58,967
Creighton	12,700
Thetford	3629
El Salvador	23,587
Climax	42,638
Urad	5715
Hecla's Lake Shore mine, oxide	5851
sulfide	9979
Atlas Consolidated	19,958
Bultfontein	2550
El Teniente	36,287

Source: *SME Mining Engineering Handbook*, 1973.

*Metric equivalent: t × 1.102 = st.

Productive Capacity

Caving is best suited for high rates of production because it is usually used for low-grade and high-tonnage-reserve mines. Daily production covers a large range, as shown in Table 41.2.

The success of a caving operation is dependent on fairly high productivity. Production crews should be expected to produce in the range of 136 to 317 t (150 to 350 st) per worker-shift. Productivity for the entire mine personnel including service departments and management ranges from 18 to 36 t (20 to 40 st) per worker-shift.

PRINCIPLES AND PRACTICE

Block caving can be used where rock mass has sufficient fractures, planes of weakness, or earthiness, as in some types of hematites, so that the mass will break if the support of an area of sufficient size is removed by some method of undercutting. The material caves from the bottom of the block, broken material is drawn off, and the caving of the mass progresses upward through the ore. There is a limit to the rate that this caving action progresses, which is in relation to the structure of the material being caved.

Arching

If the ore is drawn faster than the caving progresses, a void over the caved material will be created that could result in a very dangerous situation. The uncaved portion or a large part of it might drop suddenly into the void between caved and uncaved ore, causing a very destructive airblast through the extraction openings.

It is also possible to form a stable arch, if the rock mass has sufficient strength, so that it is difficult to promote further caving. In this case, the block should be enlarged until caving starts or some means of inducing a cave must be employed. A rectangular area, if enlarged in one direction only, still may not cave even though a very long cave is opened up because arching will occur across the shorter dimension.

Rate of Draw

The caved material should be drawn rapidly enough to avoid packing in the new undercut and permit the caving to proceed upward through the rock mass. The rate of drawing the broken material may vary, depending upon cavability of the ore, from 152 mm (6 in.) to 1.2 m (4 ft) per 24-hr day. Material from newly undercut pillars should be pulled immediately following an undercut blast to be certain no small parts of pillars remain as support between the roof and undercut floor. If pillar remnants are not removed at the time of undercutting, these remnants can support the back of the undercut and interfere with proper caving or transmit damaging weight to the extraction openings below the undercutting level.

Ore caves and crushes by its own weight and that of the overlying capping, preferably into sizes suitable for handling. Oversize chunks must be blasted to pass through draw openings. Caving usually extends through the overburden to the surface with the overburden subsiding as the underlying ore is removed. Ore drawing continues at predetermined rates until the ore values of the material from the drawpoints decreases to an economic limit. Draw cutoff point can be determined by assay values and visual inspection of the material being drawn.

Dilution Control

Well-controlled draw is essential to minimize effects of dilution. A dilution zone that breaks into the same size as the ore or larger is best for minimum dilution. Where waste is finer than ore, greater amounts of dilution can be expected but can be controlled with good draw practices.

The recovery of ore and amount of dilution are important factors in the success of a caving system, and there is no known method of predetermining the end results. Experience at other mines could be a good indicator. Table 41.3 shows some typical results.

TABLE 41.3 Ore recovery and dilution

Mine	% Ore Recovered	% Dilution
Cornwall	100	17
Grace	85	20
Mather	67	10
Thetford	100	20
Creighton	95	15
Climax	92.5	15
San Manuel	101.5*	12.2
Urad	101*	15
De Beers	100	20
El Teniente	90.95	10–20

Source: *SME Mining Engineering Handbook*, 1973.

*Actual recovery of calculated ore.

Height of Blocks

High-column blocks are desirable because the development cost per ton is inversely proportional to the column height, and if the high-column blocks are properly drawn, the ratio of capping to clean ore can be minimized. If capping breaks much finer than the ore, the amount of dilution may increase in proportion to clean ore in high-column blocks because finely broken capping will infiltrate and mix with ore as it is being drawn. Good ore-drawing procedures are necessary to obtain optimum results from high-column blocks. Where capping breaks into coarse chunks, the dilution problem is greatly reduced; however, good draw practice must be followed here as well. Maximum practical height of ore above the extraction level depends on the total height of the deposit, dip of the orebody, and character of ore and capping (Table 41.4).

Among porphyry copper mines using block caving, the height of blocks has increased to 183 m (600 ft) and higher, with the total rock column above the extraction level of ore and capping being 457 m (1500 ft). The total rock column in mining areas at San Manuel, Arizona, in mining blocks can be as high as 1067 m (3500 ft) above the extraction level. The Henderson mine near Empire, Colorado, is caving with ore heights up to 152 m (500 ft) and a total rock column above the extraction level up to 1219 m (4000 ft).

Horizontal block area should be limited to what can be drawn at the desired vertical inches of solid per 24-hr day as well as what will give sufficient flexibility of draw and haulage area and require a minimum of repair. The area must be large enough to cause the ground to cave with relative ease but small enough not to exert excessive weight on extraction openings.

TABLE 41.4 Block size in block caving

Mine	Type of Deposit	Block Size, m*	Height, m*
Thetford Mines, Asbestos Corp. Ltd.	Asbestos igneous stockwork	49 × 49	122 and less
Miami Copper Co., Tennessee Corp.	Porphyry copper	30 × 49	183
El Salvador, CODELCO-Chile	Porphyry copper	29 × 36 to 48 × 74	49–213
Mather, Cleveland-Cliffs Iron Co.	Hematite	27 × 27	61–76
Creighton, International Nickel Co.	Disseminated norite	Mass cave	152
San Manuel, Magma Copper Co.	Disseminated chalcocite Altered granite	Panel cave	183 and greater
Grace, Bethlehem Steel Corp.	Massive magnetite	46 × 46	34
Cornwall Nos. 3 and 4, Bethlehem Steel Corp.	Magnetite	Panel cave	30–61
Climax, AMAX	Disseminated molybdenite	Mass cave	91
Urad, AMAX	Disseminated molybdenite	Mass cave	91
De Beers Cons. Mines, Ltd.	Porphyritic volcanic stock	Panel cave	122–183
El Teniente, CODELCO-Chile	Massive sulfide copper	60 × 90	100–180
Henderson, AMAX	Disseminated molybdenite	Mass cave	122

Source: *SME Mining Engineering Handbook*, 1973.

*Metric equivalent: m × 3.2808 = ft.

Drawpoint Spacing

The zone of disturbance affected by one drawhole is a function of size of the material expected at the drawhole. The zone of disturbance of one drawhole should slightly overlap all adjacent zones of disturbance so as not to leave pillars of ore that do not move when the ore is being drawn. The more finely the rock breaks, the closer the drawpoint spacing must be. Coarser rock particles can have wider drawpoint spacing. There is a practical limit, however, because sufficient access must be maintained into the undercutting area for blasting down large slabs and loosening high hang-ups. Spacing probably should not exceed 15 × 15 m (50 × 50 ft). Table 41.1 shows typical drawpoint spacing.

Drawpoint spacing is best determined by experience at other mines working under similar conditions and experimentation at individual mines after production has started. Most mines that have been in operation for a period of time have varied drawpoint spacing from time to time to meet changing conditions.

Weight Problems

As ore is removed and voids are created, the weight of overlying rock that has not caved is transmitted to the perimeter of uncaved ore. Unusual stresses can be transferred to fringe workings causing excessive damage. Fringe workings are best protected by placing them outside the area of excessive stress. This is difficult to predetermine and is not always practical. The configuration of the advancing cave face can help to minimize the effects of this transferred weight. The configuration best suited to a particular mine is learned by

discussions with operating mine personnel and through experience. Isolated blocks of ore within the cave area can act as supports for the subsiding ore and also transmit weight to the workings below. Weight within the active caved area is best avoided by maintaining a steady and even draw and removing during undercutting any remnants that could act as pillars. Opening up only sufficient area to maintain desired production is an important feature of this type of control.

The area necessary for production is a matter best learned through experience. Some factors to be considered are: (1) the rate of vertical draw that can be expected per day; (2) the maximum size pieces to arrive at the drawpoint, which determine amount of secondary blasting; (3) availability of drawpoints with time allowed for repair; (4) loading time for transportation units (it is possible to multiple-load if loading points are properly spaced); (5) number of haulageways necessary to maintain tonnage requirements; and (6) total production desired.

DEVELOPMENT

Development must be suited to the characteristics of the orebody and form of block caving used. For example, in relatively hard ground a minimum of ground support in the extraction level would be required, and drawpoints could be spaced on a wider interval. Where the ore is expected to break into relatively small chunks, more ground support would be needed, and extraction drawpoints should be spaced on closer intervals. In any case, a fairly long period of development work and a substantial investment of money are required before production can be realized.

The details of development will vary to some extent from one mine to the next and are also dependent on which of the three variations of caving is being used.

Production Level

In general a caving mine will require a haulage or production level or possibly both. The gathering or scram drifts and the drawpoints are usually on the same level and lie above the haulage level. The main haulage units to move the ore from mine to crusher or mill are almost always ore trains. Because the production is always run-of-mine ore, other means of transportation are not suitable until some size reduction has been accomplished by crushing.

Production from the drawpoints is accomplished by gravity feed into loading chutes, electrically operated scrapers moving the ore directly to ore cars or diesel-operated load-haul-dump units transporting the ore to orepasses.

Undercut Level

The undercut level lies directly and some distance above the production level. Undercutting consists of a series of parallel drifts at the undercutting level from which a series of longholes are drilled. When the longholes are loaded and blasted, a thin horizontal slice of ore is removed, thus removing the

vertical support from the ore column above. As the blasted rock is drawn through the drawpoints, the ore column above begins to collapse by gravity. Continued draw gradually brings down the entire ore column.

Caving Limits

Caving in most cases extends to the surface, with the surface subsiding as the underlying ore is removed. The horizontal limits of caving are determined initially by the size of block being undercut. In weak ore, the initial surface expression may lie almost directly over the undercut boundaries. In stronger ores, because of the arching tendency, the initial surface expression may be significantly smaller in dimensions than the undercut area. It will always lie within the limits of the undercut. Eventually the caving will work outward from the undercut boundaries. The total surface disturbance will lie within an area described by drawing a 0.79 rad (45°) line from the undercut elevation to the surface. No permanent surface structures or shafts should be placed inside this line because they could eventually be damaged or lost.

Boundary Weakening

The lateral extent of caving in some cases can be temporarily controlled by block boundary weakening where a formal isolation system of drifts or raises or both is driven along the block boundary, thus limiting the horizontal extent of caving for the block. An aid to boundary weakening in hard ground is attained by excavating narrow shrinkage stopes at the virgin corners of the block. These stopes can extend horizontally in two directions along the length of the block. These excavations can be completed before undercutting in the block has begun or in conjunction with the undercut advance. Boundary weakening is also an aid in promoting the initial cave.

In some cases where the ore is fairly competent, the amount of cutoff may be quite extensive and placed along adjacent sides of the initial cave area. Once the caving action starts in a block, the ore caves and crushes due to its own weight.

Equipment

Equipment must also be designed for high production rates. Development equipment should be as highly mechanized as the mine layout will allow. Multiboomed drill jumbos and high-tonnage mucking units should be used where possible. Where production loading is by chute, they should be designed for rapid flow and high tonnage. Production scrapers should be as large as the mine opening will allow, and the distance for scraping ore should be minimized. Load-haul-dump units should be 3.8 to 6 m^3 (5 to 8 cu yd), although smaller units have been used. Because load-haul-dump units are not suitable for long hauls, and trucks are usually not amenable to a caving layout, train haulage is used as some part of the transporting process. Mine cars should be at least of 9-t (10-st) capacity and larger where possible. Locomotives should be capable of handling 20- to 30-car trains. Multiple or continuous loading to the cars should be a consideration. If the distance to the mill facility is very far, the relative merits and cost of train haulage versus underground crushing and conveying should be considered.

Ventilation

Clean air can be brought to the working areas through the main haulage system or by an entirely separate drifting system. The clean air is directed to the working areas and once it has removed the dust and gases produced by the mining activity, it is captured in a separate exhaust system and returned to the surface. The volume of air must be adequate to insure proper working conditions for the workers and, in the case where electrically or air-powered equipment is used, very little additional air is required. Where diesel-operated equipment is used, a substantially greater amount of air is required to remove the noxious gases produced by the diesel motors. The amount required is controlled by state or federal regulations and is dependent on the amount of horsepower being used.

GENERAL CONSIDERATIONS

Costs

The cost of bringing a caving mine into production is significant, as is the time required for the initial construction and development. Based on 1976 costs, $16,500/t ($15,000 per st) produced per day for mine, crushing, and milling facilities is a realistic figure. Mine operating costs should lie in the range of $3.00 to $4.40/t ($2.75 to $4.00 per st) exclusive of development costs. This cost per ton will depend on the height of ore being developed but should average around $1.65 to $2.20/t ($1.50 to $2.00 per st).

Advantages of Block Caving

The advantages of block caving are: (1) mining is inexpensive compared to other systems because relatively little drilling, blasting, and timbering are done per ton of ore and the amount of development work per ton is small; (2) centralized production permits efficient supervision, resulting in higher worker productivity and a good control of safe working conditions; (3) control of ventilation is less complex compared to other mining systems; (4) production can be maintained at a relatively high rate; and (5) the system is suitable to orebodies of relatively low grade.

Disadvantages of Block Caving

The disadvantages of block caving are: (1) preparing blocks for caving requires more time and money than other systems before production is attained; (2) maintaining drifts in the draw area can be costly and will interfere with production; (3) varying production under increased demand for the product requires a long time interval to prepare additional blocks for production; (4) stoppage of ore drawing for a considerable time may result in complete loss of the development openings in the area involved if it is subject to weight concentrations; (5) ore recovery can be low due to adverse conditions, and there is constant danger of losing large amounts of ore if good draw practice is not followed; and (6) the method is inflexible once started—a change to another underground method is difficult.

Block Versus Sublevel Caving

The relative advantages of block caving are: (1) mining cost is less expensive per ton; (2) it gives larger daily output from a given area; (3) it requires less development work per ton; and (4) ventilation is more easily controlled. Sublevel caving is possible in hard ores and smaller orebodies, but should be considered only where gravity caving might be questionable because of sparse fracturing or where large blocks might result in high secondary blasting costs.

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CHAPTER 42

Climax Panel Caving and Extraction System

John C. Gould

GENERAL DESCRIPTION

The Climax mine is located 161 km (100 miles) southwest of Denver on Fremont Pass astride the Continental Divide. Climax employs 2800 people to produce 42,670 t/d (47,000 stpd) of molybdenum ore. The Storke level, the main underground level, produces 19,970 t/d (22,000 stpd); the 600 level, the lowest active underground area, produces 7260 t/d (8000 stpd); and the open pit presently produces 15,430 t/d (17,000 stpd) (see Fig. 42.1). A panel caving method is possible because of the favorable geology and geometry of the Climax orebody.

Geology

The Climax deposit is composed of three separate orebodies formed in a massive Tertiary stockwork, which penetrates Precambrian granites and schists. The orebodies are elliptical in plan with overall dimensions of 914 × 1219 m (3000 × 4000 ft). In section the orebodies are arcuate with an overall height of 457 m (1500 ft), although the depth is not firmly ascertained. Ore grade decreases rapidly into a highly siliceous central core and more gradually toward the outer boundaries of the deposit. The mineralization is relatively continuous in the ore zone and contains few dilution-causing waste inclusions.

Molybdenite is contained in pre-ore fractures, which were largely recemented during the three mineralization pulses. Post-ore fractures, with a spacing of a few centimeters (inches) to several meters (feet), render the deposit amenable to caving when it is undercut. The peripheral rock caves similarly to the ore, but the central core is competent and is difficult to cave.

Caving Feasibility

Successful caving action requires an undercut area of approximately 122 by 122 m (400 by 400 ft). When the undercut area exceeds this minimum

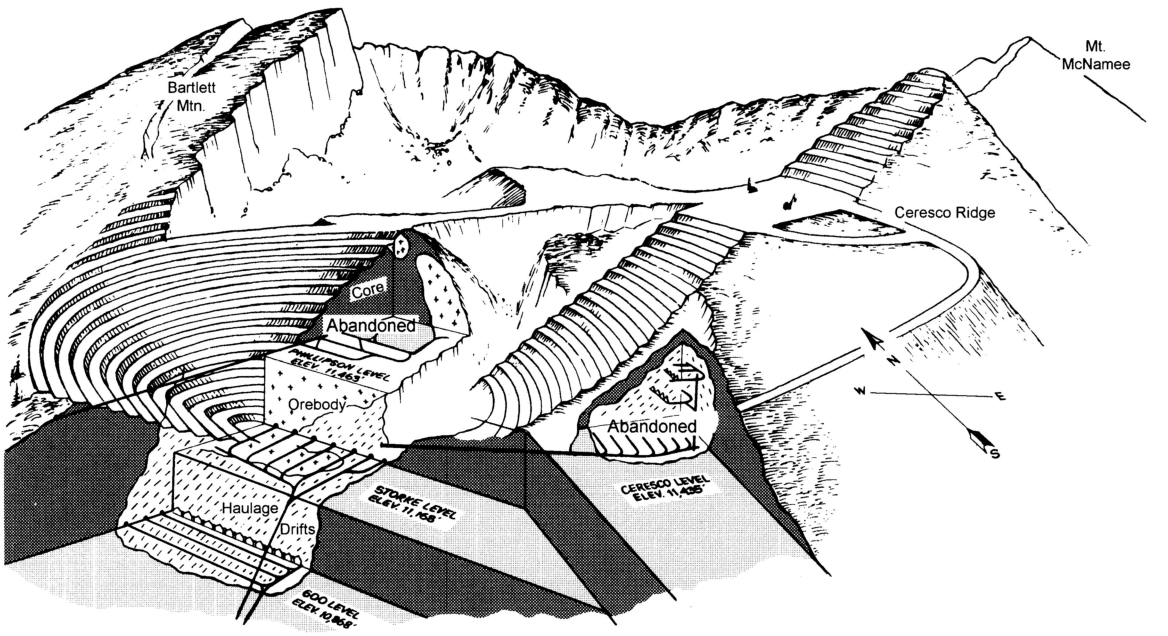


FIGURE 42.1 Orthographic projection of mining levels, Climax mine

distance, the arching effect is lost, and the ore column collapses. Ore extensions from the main cave area less than 122 m (400 ft) wide usually cannot be caved. Waste or low-grade areas within the deposit can be left if they are large enough to carry weight around openings passing through them. Experience indicates that these areas should be more than 30 m (100 ft) in length and width. Smaller unprofitable units of waste, such as post-ore dikes, are mined to avoid weight problems and result in ore dilution. Successful caving and ore recovery are dependent upon timely ore removal from the undercut zone because columns of immobile ore inhibit caving and concentrate weight onto the underlying workings. Climax experience indicates that a finger (drawpoint) spacing[1] averaging 10.4 m (34 ft) yields an excellent caving action and a 92.5% ore recovery at the estimated grade. An additional 15% of the ore is recovered at a diluted grade (Fig. 42.2).

Mine Layout

The slusher drift is the primary production facility (Fig. 42.3). The ore is undercut and caved into the fingers, scraped to the drawhole and into the ore cars. The specific dimensions and spacing of the fingers depend on a number of considerations involving ore recovery, development cost, and operating conditions. The primary factor influencing the mine layout is the finger spacing required for good ore recovery. The spacing of the fingers along the slusher drift varies from 9 m (31 ft) to 12.8 m (42 ft) across the loading drift. This spacing in a 30.5-m

1. Modeling studies have shown the relationship between rock size and the horizontal distance rock will travel to the drawpoint (McNicholas, Rogers, and Walker, 1946).

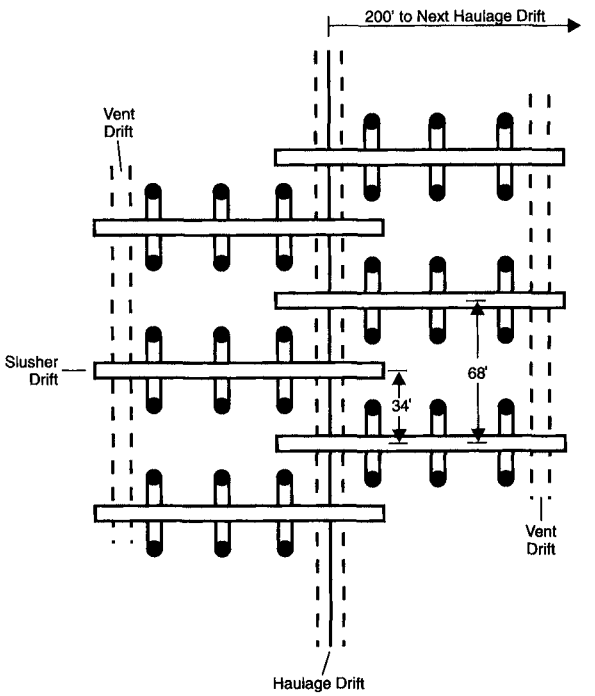


FIGURE 42.2 Plan of slusher drills and finger spacing, Climax mine, metric equivalent: ft × 0.3048 = m

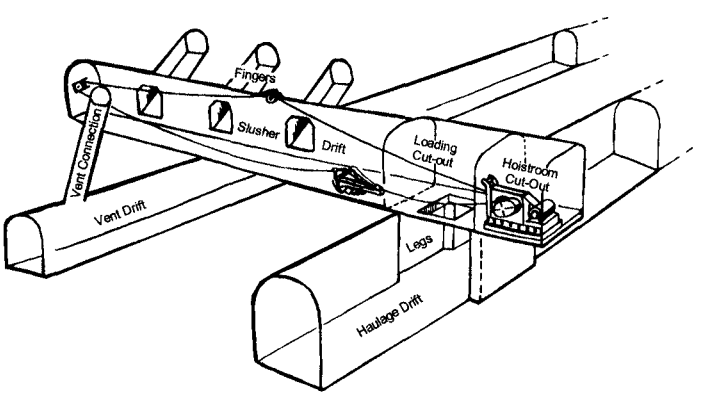


FIGURE 42.3 Perspective of slusher drift, Climax mine

(100-ft) slusher drift yields a six-fingered slusher drift with three fingers on each side. Longer slusher drifts tend to lower per-ton costs for development and equipment but tend to increase production problems and costs. Train loading times increase due to the greater slushing distance and poor visibility when loading from the back of a longer slusher drift. Operators also tend to draw the nearest fingers, resulting in poor draw control.

The distance between slusher drifts is controlled by the finger design; this permits simultaneous loading of several cars spotted under adjacent drawholes. A 10.4-m (34-ft) drawhole spacing allows a car to be spotted beneath

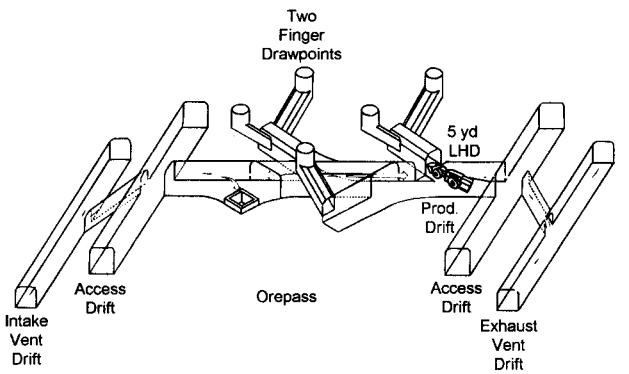


FIGURE 42.4 Alternative mining method, Climax mine. Lower levels production system.

adjacent drawholes with two cars between the drawholes. The slusher drifts are driven perpendicular to and on alternate sides of the haulage drifts, which are on 61-m (200-ft) centers. Ventilation drifts spaced halfway between loading drifts provide air exhaust capabilities for the slusher and haulage drifts.

Alternate Mine Layout

More recent production-level layout will be able to take full advantage of diesel-powered rubber-tired equipment and increase the stability of underground excavations in both development and production work. The caving technique of undercutting the ore and caving into the fingers is the same as the slusher system. The production level consists of 3.7-m (12-ft)-wide production drifts with 3-m (10-ft)-wide drawpoint turnouts every 12 m (40 ft). Each drawpoint is fed by two 2.4-m (8-ft)-diam fingers spaced 12 m (40 ft) apart (Fig. 42.4).

The ore is loaded at the drawpoints by load-haul-dump units and hauled through the production drift to an orepass. The orepass carries the muck to a loading chute at the haulage level located beneath the production level. Trains load the ore from the chutes and haul it to the crusher.

DEVELOPMENT AND PREPARATION SEQUENCE

Development of a new production level begins with the establishment of access through either a shaft or tunnel. The access is planned to accommodate the efficient removal of development rock and for long-term material supply to the level.

Ventilation Laterals

Upon gaining access to the production area of the mine, the exhaust ventilation laterals are driven (Fig. 42.5). The exhaust laterals are large-diameter openings of 4.3 × 4.3 m (14 × 14 ft), requiring considerable time to mine. By driving the exhaust laterals early in the development sequence, fill-in

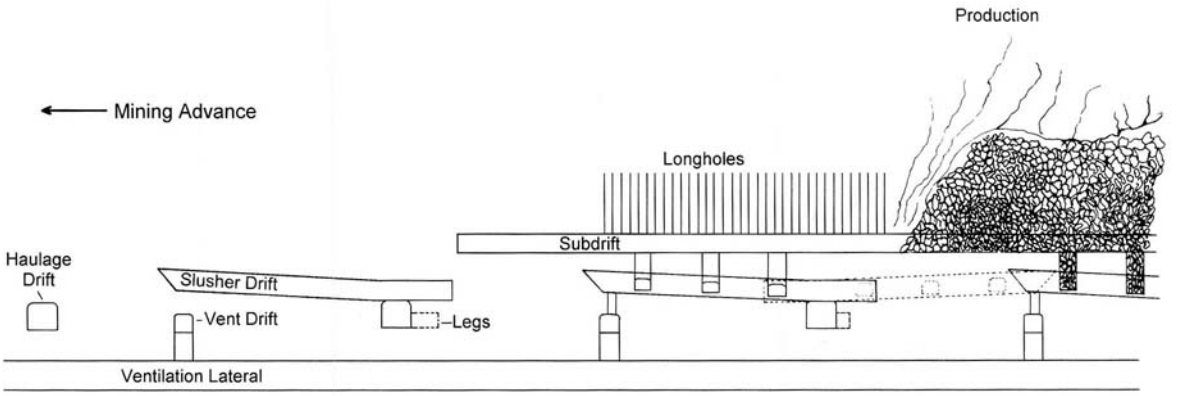


FIGURE 42.5 Section through mining level, Climax mine

diamond drilling can be performed to better define the ore boundaries. More precise definition of the ore boundaries and grade values permits detailed engineering of the mine layouts. The mining of exhaust laterals also enables good ventilation to be established as the development progresses. Early development of the ventilation drifts is possible because these are mined from the exhaust laterals.

Main Haulage and Loading Drifts

The main haulage drifts are mined after the excavation of the ventilation laterals. These are driven around the perimeter of the deposit and through the center of the level. As mining advances, fresh air is brought in through the haulage drifts and the contaminated air exhausted through the laterals.

Upon completion of the primary haulage drifts, loading drifts are mined on 61-m (200-ft) centers. When the loading drift is complete, storage cutouts, powder magazines, switch vaults, and manways to slusher drifts are developed in the loading drifts. After mining, these openings are lined with concrete. An opening is left in the back above the legs for mining the loading cutout of the slusher drift. Simultaneously with mining the loading drifts, the vent drifts are driven from the vent laterals.

Slusher Drifts

Once the mining activities in the loading drifts are complete, work on the slusher drifts can begin (Figs. 42.3 and 42.6). The slusher drifts are mined and a short 0.6-m (2-ft)-diam raise, called the vent connection, is driven from the vent to the back end of the slusher drift. This raise permits air circulation from the loading drift, through the slusher drift to the vent drift. The slusher drift and vent connections are formed and lined with concrete, leaving openings in the concrete for the mining of the fingers. A fan station is established over the exhaust laterals in the vent drifts to draw the contaminated air from the slusher drifts. The fingers are then mined and lined with concrete.

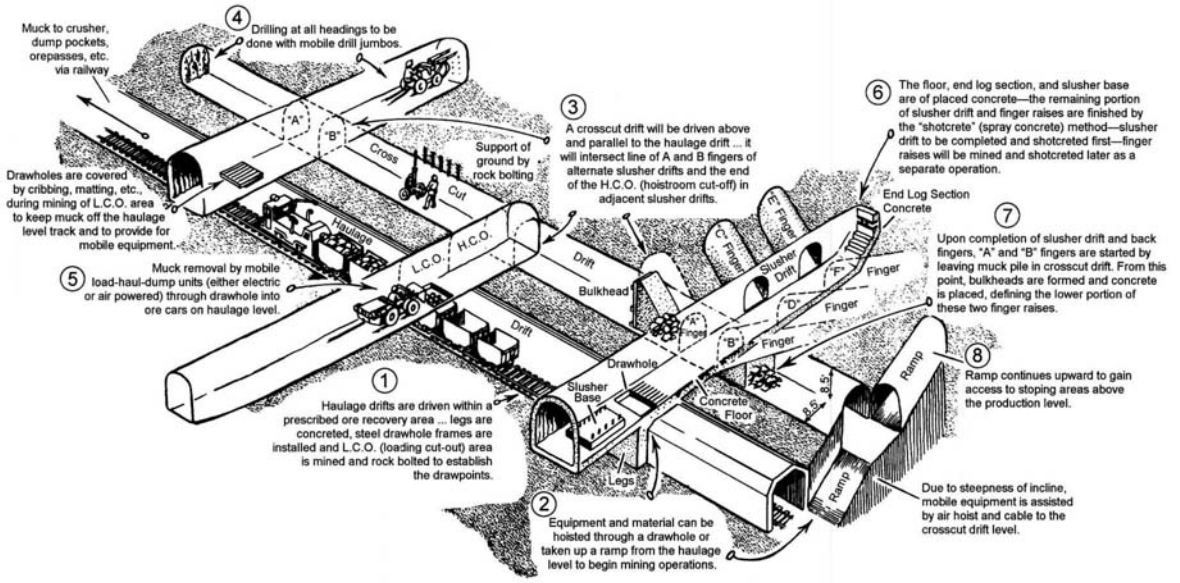


FIGURE 42.6 Mining method, Climax mine

Undercutting

Preparation for undercutting the orebody follows completion of the slusher drifts (Fig. 42.7). This phase of mining is called stoping. Subdrifts are driven across the tops of the fingers parallel to the slusher drifts to establish access for the ring drills. Crosscuts are driven between subdrifts at regular intervals to provide equipment access to several faces. Rock from the subdrifting is disposed of through the fingers into the slusher drifts by means of short finger raises. These finger raises are mined as muck disposal requirements dictate.

The undercut is then ring drilled from the subdrifts in a fan pattern as illustrated in Fig. 42.7 so that the major and minor apexes are defined. The last stage in the sequence is to blast the fan holes and bell out the tops of the fingers before turning the slusher drift over to the production crews.

Sequence of Work

The continuous sequence of work required to prepare areas for production necessitates that concurrent mining activities occur in four adjacent loading drifts. Loading-drift headings are driven while legs are mined in the adjacent completed loading drift. Slusher drifts are prepared in the next drift toward the production area while stoping is advanced adjacent to the production area. During the initial development and prior to production from a new level, the facilities to handle production ore are constructed. An underground crushing and conveying system is installed in the lower levels to move ore to the tertiary crushers and milling facilities on the surface.

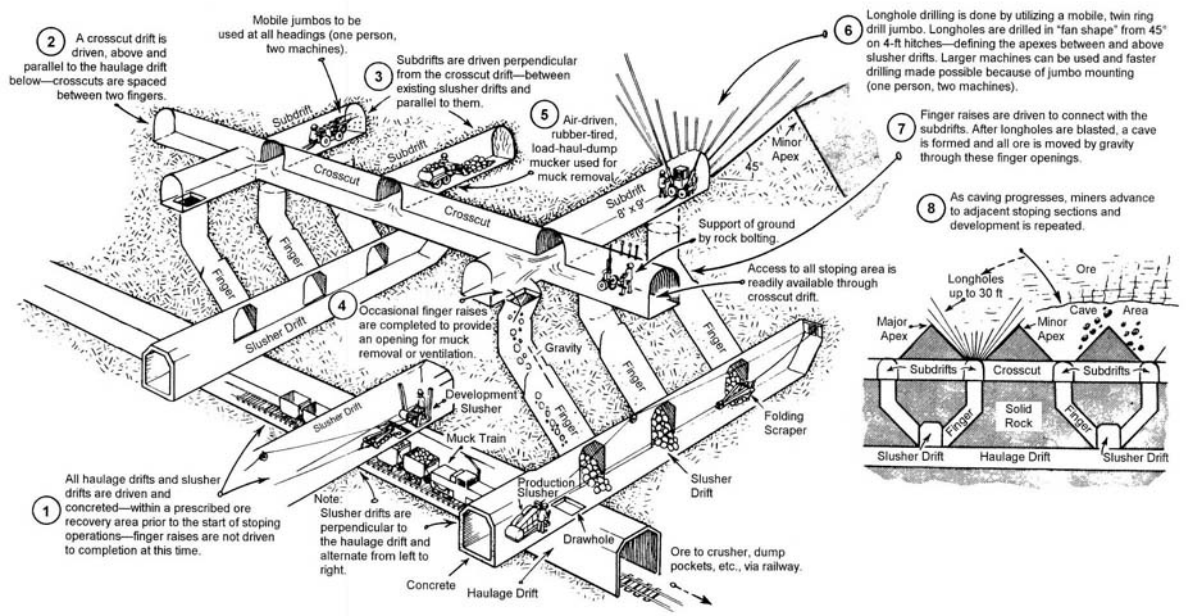


FIGURE 42.7 Undercutting method, Climax mine

MINING PRACTICES

Mining is classified into three specific activities at Climax: development, preparation, and stoping.

Development

Development at Climax includes all mining and other activities involved with establishing shafts, adits, primary ventilation intake and exhaust systems, haulage and loading drifts, underground crushing and conveying systems, permanent underground shops, lunchroom, and storage areas. The most common development activity is the driving of loading drifts.

The mining of these openings is done with air- or diesel-powered rubber-tired equipment. The drilling is done with small rubber-tired jumbos that are equipped with two 3-m (10-ft) long screw-driven feeds mounted with 89-mm ($3\frac{1}{2}$-in.) drifters that drill 50-mm (2-in.) holes. The loading drifts are 3.6 m (12 ft) high, 3.6 m (12 ft) wide at the spring line, and 4.3 m (14 ft) wide at the hitch. The drift is driven 457 mm (18 in.) below grade so that drainage flumes and proper track grades can be installed. A standard round consists of 36 to 40 holes with an average pulled depth of 2.1 m (7 ft).

Blasting is carried out using ammonium nitrate with a Nonel down-the-hole delay detonating system and boosters. Watergel explosives are being evaluated as an alternative explosive to ANFO because of their better fume characteristics and direct detonation properties. Development blasting is done either at lunchtime or shift change to allow the working areas adequate time to clear of smoke and dust.

Local practice requires all openings to be supported with rockbolts. The minimum support is provided by rockbolts spaced 762 mm (30 in.) apart in a bolt fan with the fans spaced on 1.8-m (6-ft) centers. The rings are drilled and bolts installed from the muck pile using 76-mm (3-in.) jacklegs, drilling 38-mm (1½-in.) holes. A special rockbolting jumbo will soon replace the jacklegs. While expansion-type anchor bolts are still used, a conversion to either grouted or epoxy bolts is being made. Mucking is done with 1.5- and 3.8-m^3 (2- and 5-cu yd) diesel-powered load-haul-dump equipment. Headings are advanced three shifts per day, seven days per week, using two workers per heading.

Preparation

Preparation at Climax includes all mining activities for establishing legs, slusher drifts, fingers, and vent drifts.

Legs. Legs are openings in the loading drift that support the slusher drift activities above. This includes establishing a timber boxout and powder magazine for every four slusher drifts, a switch vault for every two slushers, and a manway cutout for every slusher drift. The haulage drift is widened beneath each slusher drift to make space for the concrete that supports the drawhole steel. A complete set of legs requires the mining of 46.8 m^3 (1652 cu ft) of rock.

Slusher Drifts. Slusher drifts are mined with trackless equipment. Trackless equipment requires a crosscut from the hoist cutout of one slusher drift to the front set of fingers of the adjacent slusher drift so the mobile equipment can readily move between headings. Temporary support is provided by rockbolts and wire mesh and, when access can be closed between slusher drifts, the crosscuts are backfilled with rock and concrete.

Finger Cutouts. Finger cutouts are mined with jacklegs and 45-kW (60-hp) slushers, after the slusher drifts are concreted. Two-person crews work in a slusher drift advancing fingers.

Vent Drifts. Vent drifts are mined prior to the development of the slusher drift to establish immediate exhaust ventilation from the slusher drifts. These openings are mined 2.7 × 2.7 m (9 × 9 ft) using jacklegs and 0.76-m^3 (1-cu yd) diesel LHD units. The average round of 32 holes is 1.8 m (6 ft) deep. The holes are loaded with watergel explosives and detonated with Nonel caps. Fresh air is delivered to the face with 711-mm (28-in.) ducting and 7.5-kW (10-hp) in-line fans.

Preparation Concrete

The concrete section is responsible for building forms, delivery and placement of concrete, and stripping and cleaning up after the concrete is placed in the preparation areas. The major concrete work is in legs, slusher drifts, and fingers.

Legs. Legs are formed in groups of three at a time by form crews consisting of five people. The legs include switch vaults, powder magazines, and timber box-outs. A combination of prefabricated reusable forms and wooden custom-made forms are used. No rebar is placed in the legs. A set of three legs requires about 382 m^3 (500 cu yd) of concrete. A concrete crew consists of seven people including the motor operator. The concrete is delivered in two 4.6-m^3 (6-cu yd) transit mixers per train drawn by an 8-t (8.8-st) locomotive. Two trains are used to shuttle the concrete from the batch plant to the placing site. The concrete is hauled wet mixed and is designed for 1814 kg (4000 lb) compressive strength. The concrete is placed in the forms using pumps. This method of delivery and placement of concrete is common to all the concrete projects except shotcrete.

Slusher Drifts. Slusher drifts are poured in three sections: the loading cut-out, the hoist cutout, and the slusher drift section. Slusher drifts are formed with a crew of five people using prefabricated reusable metal forms and then concreted with 306 m^3 (400 cu yd) of concrete.

Fingers. After the six finger cutouts are mined in the slusher drift, they are formed by a five-person crew and then concreted using 160 m^3 (210 cu yd) of concrete.

Stoping

Stoping at Climax includes all those mining activities carried on above the finger including subdrifting, crosscutting, finger raises, finger bells, ring drilling, and blasting.

Subdrifts. Subdrifts are openings, 2.4 m (8 ft) wide and 2.7 m (9 ft) high, running parallel to the slusher drifts across the tops of the fingers. These subdrifts provide the ring-drill access. The subdrifts are driven with air-powered, rubber-tired, two-machine jumbos. Each round requires about 35 to 40 50-mm (2-in.) holes, 2.4 m (8 ft) deep.

The holes are bottom primed with half a stick of 40% dynamite into which is inserted a Nonel delay cap. The hole is loaded with ammonium nitrate and shot at the end of the day shift. Mucking is accomplished using 1.5- and 3.8-m^3 (2- and 5-cu yd) diesel LHDs. There are 12 of these units assigned to stoping. Muck from the subdrift rounds is disposed of through the fingers into the slusher drifts.

Ring Drilling. In order to cave the ore above the slusher drift, fans or rings of 89-mm (3½-in.) holes are drilled out of the subdrifts. These holes fan upwards from 0.79 rad (45°) to vertical and vary from 4.6 to 11.3 m (15 to 37 ft) in length. The fans are spaced 1.5 m (5 ft) apart. The ring-drill machine is equipped with two valveless 121-mm (4¾-in.) machines mounted on skid steering carriages. One operator runs the two machines.

Finger Raises and Bells. The finger raise is mined to allow for disposal of rock from the subdrifts. This raise is 2.4 m (8 ft) high from the end of the finger to the subdrift. The bell is an enlargement of the finger raise to form a funnel top to the finger. An additional 18 holes are required to form the bell, these holes are drilled at the same time as the raise. The bells are shot with

the ring-drilled holes overhead so that the floor of the subdrift can be maintained until the last possible moment. Finger bells and raises are mined with stopers drilling 38-mm (1½-in.) holes. The holes are loaded with 40% dynamite and detonated with electric delay caps. The rock from the shot drops into the slusher drift for removal by a 45-kW (60-hp) slusher.

Blasting. The rings are loaded with ammonium nitrate, which is blown into the holes using standard ANFO air pots. The detonating system consists of using long-period Primadetes bottom-hole delay system and Detaprime boosters. Each hole is delayed to provide maximum relief. Shooting takes place at the end of the day shift and is initiated from the main blasting circuit.

Shooting the longholes advances the cave line. The sequence of shooting the longholes requires consideration of a number of interactions discussed in the next section.

Cave-Line Advance

The completion of the stope mining activities leaves the ore prepared for caving. The cave-line advance is achieved by shooting the fans of 89-mm (3½-in.) holes. Fans are usually shot three at a time in a given subdrift. The cave line is advanced across haulage drifts on the bias so that high stresses do not occur along the entire length of an opening. This is accomplished by advancing the cave farthest in the center of the orebody and then winging back two to five rings along each subdrift outward from the center. The net effect is to create a long, arcuate, somewhat staggered cave line across the entire width of the orebody. Prior to shooting each set of rings, it is very important to draw the swell rock from the previous set of shot rings. Without adequate swell rock relief, the rings may not break cleanly, leaving pillars of intact rock internal to the cave. Such pillars are also caused by cutoff of holes and misfires. These pillars act like punches, concentrating immense stress on a very limited area. These high-stress points simply punch the back out of the underlying openings, causing interruption in the overall cave advance program. Drawing the swell muck after shooting each set of rings also provides access to inspect the success of the previous shot. If pillars are found, it is still possible to shoot them with paste shots prior to advancing the cave further.

The caving causes the stress-strain relationships along the line between the caved and uncaved regions to readjust, occasionally in a very destructive manner. The successful advance of the cave line must take into account the influence of prominent geological structures on the weight redistribution, the behavior of the stress front in relationship to the geometrical shape of the cave line, the need to establish efficient extraction conditions, and the need to maintain adequate width in the active cave zone for proper draw control.

The main stress buildups lead the cave line by approximately 9 to 12 m (30 to 40 ft). When the cave line is undercut, the stress advances along the front of the undercut region. The weight advance has a time delay of from several hours to several days.

The stress advance is measured by monitoring the movement induced in the openings along the cave front. The movement is measured using a spring-loaded telescoping rod with a direct reading scale on the side. The cave must be advanced rapidly until the stress is brought to rest over major openings paralleling the cave line. When the cave line is brought to rest, the weight will build up until a new equilibrium condition is achieved. Fault zones can create very damaging weight concentrations beyond the usual zone of stress concentrations or within the region of the stress front.

Ground Support

Most standard types of ground support are used at Climax. During driving of headings, it is mandatory to place a minimum of three rockbolts in a fan across the back at intervals every 1.8 m (6 ft) down the drift. Extra bolts, wire mesh, or timber are used as required. Shotcrete is frequently used to prevent air slacking. A great portion of the ground-support effort at Climax is directed toward controlling deformation induced by the cave advance. In areas where poor ground conditions and significant structures cross slusher drifts or haulages, additional support is placed prior to caving. The support may take the form of epoxy bolts, grouted bolts, yieldable steel arches, or reinforced concrete. It is preferable not to use steel support in the stopes but rather bolts, wire mesh, shotcrete, and timber because of the problems caused by steel when attempting to pull it through the fingers after initiating the cave.

PRODUCTION

Slusher Drifts

Production from a slusher drift begins as soon as caving of the stope is initiated. Slusher drifts are equipped with 111.75-kW (150-hp) double-drum, tandem slushers. These are equipped with a 1.8-m (6-ft)-wide folding scraper capable of moving 2.27 t (2½ st) per pass. To prevent dangerous backlashes from parting nose or tail cable, 32-mm (1¼-in.) prestretched (used) cables are employed on the slushers. The slusher operator can see to the back of the slusher drifts by means of spotlights and can see the car being loaded through the opening under the drawhole apron.

Secondary Blasting

Production blasting must be carefully controlled. Experienced senior employees, known as hang-up people, are responsible for assessing the amount and placement of explosives to be used in shooting. The hang-up person is responsible for the assignment of guards and making the shot "hot." Production blasting is done using specially designed 3.75-kg (8⅓-lb) bags of 45%-strength semigelatin powder. The explosives are placed on 25- × 50-mm (1- × 2-in.) by 4.9- to 6-m (16- to 20-ft) long sticks and strategically located under the hang-up. Bags are wrapped with detonating cord, which is in turn tied to a detonating-cord trunk line laid to the top of the manway. A number of fingers in a single slusher drift can be shot simultaneously but only one slusher drift

at a time. The trunk line is detonated with a 134-cm (53-in.) safety fuse and cap. The fuse is ignited using a pull wire spitter. Production blasting is carried on throughout the shift as required.

Ventilation

Dust and explosive fumes control in production areas is essential. The primary air-quality control is provided by the flow of approximately 142 m^3/s (5000 cfm) of fresh air from the haulage through the drawhole and into the slusher drift. This air flow draws most of the dust and fumes from production or blasting into the exhaust system. Bazooka sprays [500-mm (19 ½-in.) pipe mixing high-pressure air and water] are used for added dust control during blasting. A spray curtain between the first set of fingers and the drawhole provides additional wetting for dust suppression during mucking. A second spray is used to control dust generated by the fall of muck into the cars during slushing. This control helps suppress dust that might otherwise eddy back up around the slusher operator.

Haulage System

The rail haulage system is a 0.9-m (36-in.) gage track using 41-kg (90-lb) rail for the mainline trackage and 27-kg (60-lb) rail for the loading drift and less-traveled areas. All rail joints are bonded to provide a grounding system for the 293-V ac trolley system. Production equipment consists of a mix of 19- and 20-t (21- and 22-st) haulage locomotives. The ore cars consist of 5.7-m^3 (200-cu ft) heavy-duty Granby-type side dump cars. These cars carry 7.7 t (8.5 st), and a train is made up of 18 to 27 cars depending upon the haulage condition. The 20-t (21-st) locomotives pull 27 cars. The average train hauls 182 t (200 st) of ore. The round-trip distance varies from 4 to 4.8 km (2.5 to 3.0 miles).

Productivity Considerations

High productivity from a loading drift is dependent not only upon the production crews and the muck size but also on the condition of the loading drift and the ability to cycle trains through the loading areas. One of the key factors to cycling trains is maintaining a low dump time at the crusher. To maintain high mechanical availability of the crushers, three preventive maintenance shutdowns are scheduled per week. Each shutdown is approximately 2 hr long. Advantage is taken of the shutdown period to clean up spilled muck in the loading drift using skid-steer, rubber-tired loaders. Continued cleanup is important in the loading drifts so that time-consuming delays caused by derails from muck can be avoided. The loader can follow a train into a loading drift and muck into the last car.

Preventing oversized rock from reaching the crusher and causing hang-up delays is important to high production rates. Proper sizing of the rock is established by the size that will pass through the finger. If the finger is worn, the rock may be too large to pass through the drawhole. If either situation occurs, the rock is blasted prior to being loaded into a car. Care must be taken not to load large rocks on the top of partially full cars or the car will be derailed,

timber sets pulled down, or trolley wire stripped out of the drift as the rock jams against the back of the drift.

Fully loading the cars is also an important factor in achieving high productivity. Wet sticky muck, which tends to build up in the bottom of cars, is cleaned using specially designed cleaning forks. These car-cleaning devices are located at the crusher and of necessity tend to slow down the cycling of trains through loading drifts. Loading-drift productivity is also dependent upon the timely support of mechanical, electrical, and supply crews. Items such as the rapid replacement of broken slusher cables and resupply of powder magazines are critical for continuous high productivity.

Draw Control

The profitability of a block-caving operation is highly dependent upon minimizing dilution of ore with waste from the rock overlying the orebody. To minimize the inclusion of waste with ore, close control is exercised over the amount of ore drawn from the fingers. The draw is controlled so that adjacent columns of ore are pulled to maintain no more than a 10% difference between them in total draw. This results in a maximum of 10% draw next to the uncaved ore and 90 to 100% draw at the exhaustion line. This results in the ore waste line lying at approximately 0.7 rad (45°) as the finger spacing is approximately 10 m (33 ft) and the ore column 91 m (300 ft) high. The active production area should therefore be maintained in excess of 91 m (300 ft) wide.

The production engineer and production supervisors control the tonnage drawn from each producing stope by increasing or decreasing draw rates, closing fingers, or closing entire slusher drifts. Visual inspections are made to see that off drawpoints are not being pulled. The ore grade is monitored by grab samples taken by the hang-up people. Fingers approaching exhaustion frequently show reddish waste dilution and may be closed for draw by inspection.

Mine Repair Crew

An ongoing program of repairing slusher drifts, haulage drifts, ventilation, and haulage systems is required. Considerable effort is necessary to repair damage caused by excessive weight buildups in slusher drifts. This effort requires remining areas that are caving in, additional ground support in the form of bolting, setting steel arches, or forming and replacing concrete. Repair of fingers worn by the flow of ore is also an ongoing requirement so that slusher drifts can be kept in a safe and productive condition. When a safe high hang-up is achieved in a finger scheduled for repair, a concrete bulkhead is placed above the brow of the finger. The finger repair usually occurs only once during the life of a finger unless the method of shooting hang-ups has been incorrect, resulting in destruction of the brow, or if weight problems have created excessive finger damage. This repair work is accomplished by crews of 272 employees distributed on three shifts, seven days per week. Where construction of a bulkhead is required in areas of bad ground, it is carried out by an especially qualified miner called the "hang-up timberman."

Maintenance of track and other facilities is done by a day-shift repair crew. The day-shift repair crew has 110 people assigned to it and works only five days per week, one shift per day.

Repair Concrete

Considerable concrete is poured to repair worn fingers and damaged openings. These pours are usually small and inefficient because of the two-shift setup and pullout time. The forms for these repair areas are usually built by the repair crews but are poured by the concrete personnel.

Considerable support work is accomplished using shotcrete. Shotcrete crews consist of five people and use one or two transit mixers per shift. The areas shotcreted are usually small and involve considerable move-in and move-out time. Depending on the size of the area to be shotcreted, the placement rate varies widely. The mix is brought in dry for the shotcrete application, and water is added 4.6 m (15 ft) from the end of the nozzle. The rebound losses in shotcrete placement vary from 25 to 55%, averaging about 40%.

MINE SERVICES

Ventilation

The ventilation system at Climax is designed to maintain a healthy work environment. The system must control the levels of gases, dust, blasting and welding fumes, and diesel combustion products in all work areas.

The primary air supply is provided by two 3.05-m (10-ft), 933 kW (1250-hp), axial vane fans. These two fans supply 566 m^3/s (1,200,000 cfm) of air via an 8.5-m (28-ft)-diam shaft and 488-m (1600-ft) long intake lateral 4.6 × 4.6 m (15 × 15 ft) in section. The intake air is supplied to the 600 level and from there redistributed to the Storke level. An additional 33 m^3/s (70,000 cfm) of air is supplied through the Storke portal by the negative pressure developed from the exhaust fan system. All intake air must be heated during the winter months. The air is heated to 4°C (40°F) from a designed low of −29°C (−20°F). This is accomplished by direct-fired natural-gas heaters. The intake air is distributed into the haulage drifts at different points throughout the mine and then from the haulage drifts to the work areas.

The contaminated air is exhausted from the slusher drifts through the vent connections and into the vent drifts. Vent drifts servicing full production areas are equipped with two 137-cm (54-in.), 112-kW (150-hp), axial vane fans to maintain an adequate pressure drop from the haulage to the vent drift. The vent drifts exhaust into the exhaust lateral and raise system. The exhaust system is equipped with two 2.7-m (9-ft), 112-kW (300-hp) fans, and one 2.13-m (7-ft), 224-kW (300-hp) fan, and one 24.3-m (8-ft), 373-kW (500-hp) fan. Air is supplied to the stope and preparation areas from the haulage drifts and exhausted into the vent drifts by means of the vent connections. Within the stopes, air is moved by means of 500-mm (20-in.) diam vent ducting and 7.5 kW (10-hp) in-line fans. Where diesel-powered equipment is operated, the ventilation is designed to exhaust 0.32 m^3/s per kW (100 cfm per

horsepower) operating in the area. Underground shops are generally ventilated by moving fresh air through the areas. Where welding or cutting is done, point pickups are used to remove the fumes. The point pickups are equipped with flexible lines and portable collection hoods, which are placed 457 mm (18 in.) from the welding or cutting site.

Mechanical Repair Crews

Equipment maintenance is performed by crews located both underground and on the surface. The main surface shop is staffed with 54 employees distributed on three shifts per day, seven days per week. Major repair of all equipment that can readily be brought out from underground is performed in the main shop. This work includes rebuilding and major repairs on muck cars, slushers, transit mixers, mobile equipment, fans, and pumps. Facilities for maintenance of jacklegs, drill steels, bit sharpening, blacksmithing, and steel fabrications are included in the surface shop area.

Underground, the mechanical crews are responsible for the maintenance of pumps, fans, slushers, shafts, skips, and numbers of other pieces of equipment. They are also responsible for moving slushers, setting drawhole steel, and crawl beams in the slushers. A total of 80 workers are employed on a two-shift, seven-day-a-week basis to service the mine equipment.

The maintenance of the mobile equipment is performed underground by special crews of mobile equipment mechanics. There are several temporary underground shops for this purpose. These shops are kept close to the preparation and stoping areas and moved from time to time as the mining advances. There are 66 workers employed in the mobile-equipment maintenance program, who are assigned to two shifts per day, five days per week.

Electrical Maintenance

Electrical maintenance is carried out around the clock, seven days per week by a crew of 16 electricians with 4 electricians working per shift. These electricians are primarily assigned to deal with minor problems as they arise. A crew of 41 mine electricians work straight day shift, five days per week. Their duties include inspecting all main haulage and loading drifts once every two weeks, performing scheduled preventive maintenance, and correcting minor problems. Major repairs are scheduled with operations for planned shutdown periods. These electricians also perform maintenance work on the electric LHDs.

An additional 28 employees are assigned two shifts Monday through Friday with skeleton weekend crews to perform locomotive repairs. There are 78 locomotives at Climax in the 8-, 10-, 13-, 15-, 19- and 20-t classes. The production locomotive units are given a complete preventive-maintenance check and lubrication once every ten days, whereas the smaller service units are checked and lubricated every three weeks.

Electrical Supply and Distribution

The mine is supplied with 13.8 kV power via two separate routes. The primary power is carried underground to permanent substations. This power is transformed to 2400 V for the operation of large fans and pumps or to 440 V for the operation of the production slushers. The trolley system is supplied with 292 V dc by means of four 500-kW (670-hp) rectifiers. The underground electrical demand is 11,000 kW (14,740 hp). Electrical consumption is approximately 2,160,000 MJ (6,000,000 kW-hr). The average underground consumption is 19.8 MJ/t (5 kW-hr per st) of ore.

Compressed-Air Supply

The underground compressed-air supply is provided by two 5.2-m^3/s (11,000-cfm) centrifugal Joy compressors—one powered with a 1492-kW (2000-hp) motor and the other with a 1641-kW (2200-hp) motor. An Ingersoll-Rand XHE 0.4-m^3/s (870-cfm) compressor is used for standby.

Drainage System

The water from the drilling, the cave zone, and other ground-water sources is collected in the haulage-drift flumes. This water is dropped into the exhaust laterals and drained to a sump below the 600 level. Water is passed from the Storke level to the 600 level by bored drain holes. Some of this water is recycled for drilling, but the majority of it is pumped to the tailing pond. The water cannot be released into local streams because it is acidic. The main mine pumps are two Worthington, seven-stage, vertical turbine pumps, 448 kW (600 hp) each, with a capacity to pump 11,350 L/m (3000 gpm) against a 183-m (600-ft) head. The average pumping rate is 2650 L/m (700 gpm). The pump sumps must be designed to facilitate cleaning of the silt and sand that accumulate in them.

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CHAPTER 43

San Manuel Mine

R.L. Tobie, L.A. Thomas, and H.H. Richards

GENERAL DESCRIPTION

Since the beginning of operations, with the exception of a small tonnage mined by slushing, ore extraction has been by full gravity caving. Formerly, a checkerboard sequence of block undercutting was followed with the even-numbered blocks in one panel and the odd-numbered blocks in the adjacent panel being mined. As these blocks were depleted, the intermediate or pillar blocks were mined (Fig. 43.1). Following this checkerboard, the mining sequence went through a number of changes, finally evolving into diagonal retreat panel caving by blocks (Fig. 43.2). The numbers in Fig. 43.2 indicate the sequence in which blocks were undercut. Gaps in the numbering sequence indicate undercutting on the level outside the illustrated area.

Geology

The orebody is a low-grade deposit of chalcopyrite mineralization disseminated throughout structurally weak, highly fractured, strongly altered granitic host rocks. It takes the shape of a gently dipping elliptical cylinder consisting of an ore shell of variable thickness surrounding an interior waste core. Major and minor axes of the mineralized cylinder are 1524 m (5000 ft) and 762 m (2500 ft), respectively, and length approximates 2438 m (8000 ft). Ore is sufficiently fractured to break readily into medium-coarse size.

The igneous rock complex containing the orebody is covered by a wedge-shaped blanket of Tertiary conglomerate, which was brought into place by faulting along the major regional structure of the San Manuel fault. Thickness of the conglomerate cap varies from only 9 m (30 ft) at the east end of the orebody to more than 610 m (2000 ft) at the west. Structurally, the conglomerate is much more competent than the igneous host rocks and, when caving, it breaks into massive chunks. Conglomerate boulders seen in drawpoints underground are very coarse.

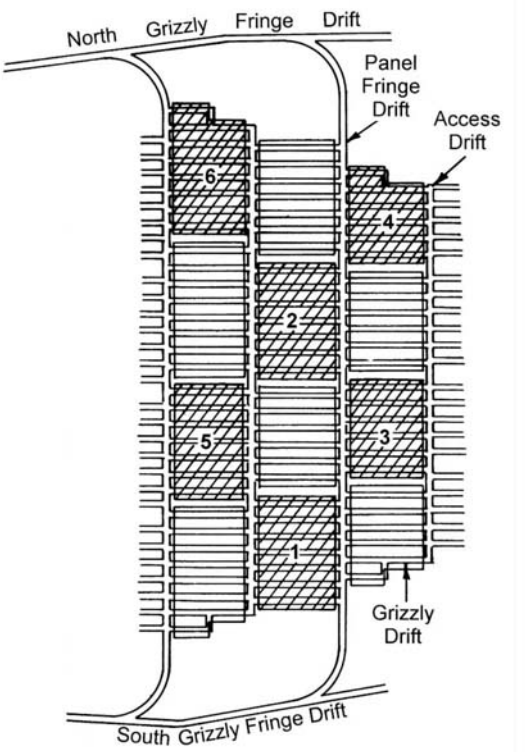


FIGURE 43.1 Sequence of block undercutting, original method

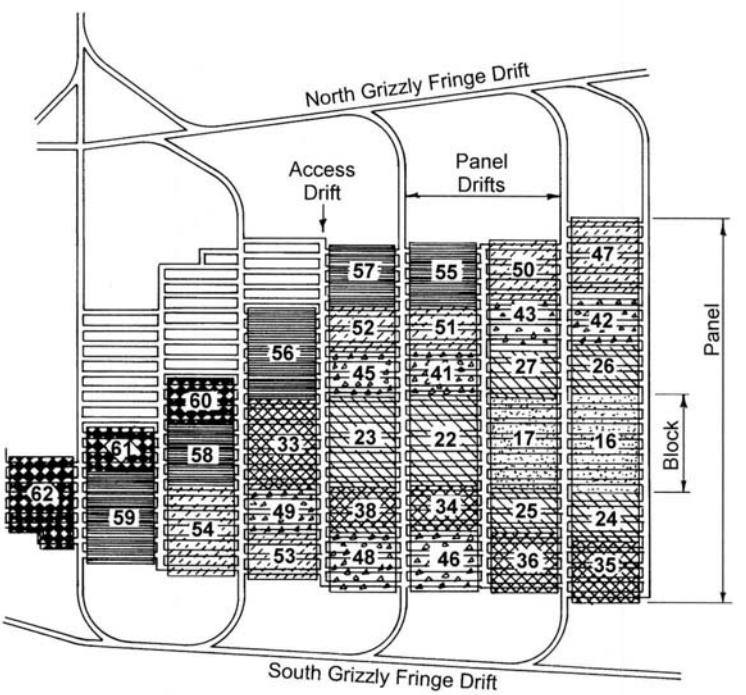


FIGURE 43.2 Diagonal retreat panel caving by blocks

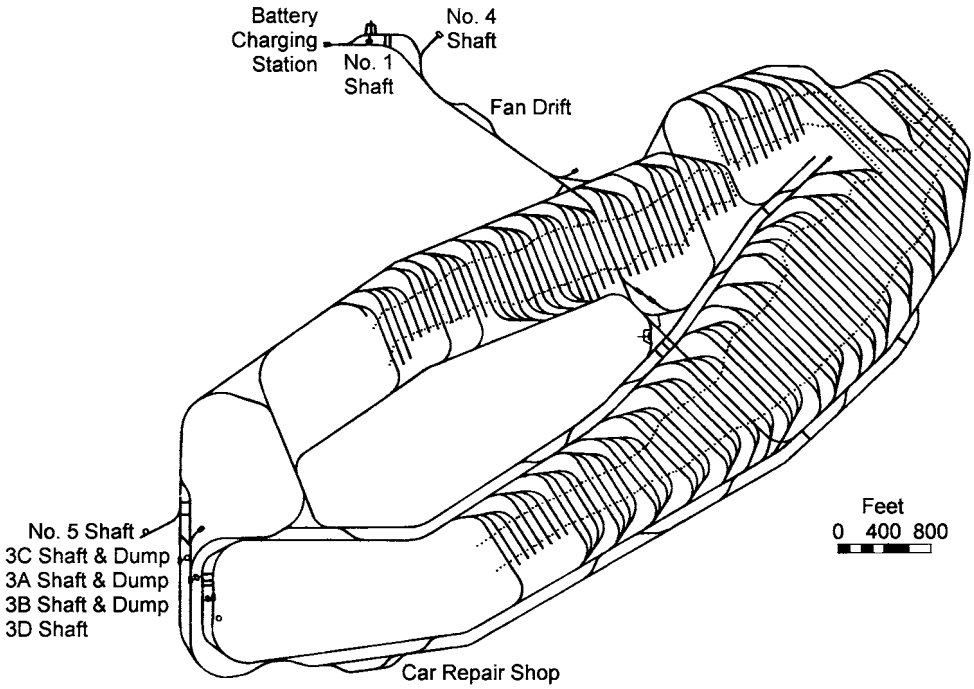


FIGURE 43.3 Main orebody, 2075 haulage, metric equivalent: ft × 0.3048 = m

The total rock column over the initial mining area of the 1415 grizzly level is 354 m (1160 ft), of which 122 m (400 ft) are ore, 79 m (260 ft) are leached igneous capping, and 152 m (500 ft) are conglomerate above the San Manuel fault.

Diamond Drilling. From 4572 to 7620 m (15,000 to 25,000 ft) are drilled annually from underground workings to delineate the orebody.

MINE DEVELOPMENT

Haulage Level

In the south, or main, orebody (see Figs. 43.3 to 43.6), with the exception of the draw and transfer raises, all the extraction openings are concreted (Seaney and Tobie, 1965). The haulage panel drifts, which are 18 m (60 ft) below the grizzly drifts, are first driven with preconcrete ground support. The drift, which has an arched section, then is concreted using mobile collapsible steel tunnel forms. The haulage drifts leading from the panels to the hoisting shafts are not concreted. After the panel drifts have been concreted, the raise stations from which transfer raises will be driven are constructed, and the raise-station ore-drawing chute is installed. The chute is prefabricated of A-36 steel with undercut guillotine gates made of abrasion-resistant 2.5-cm (1-in.) steel plate powered by 20-cm (8-in.) air cylinders installed on each side of the raise station.

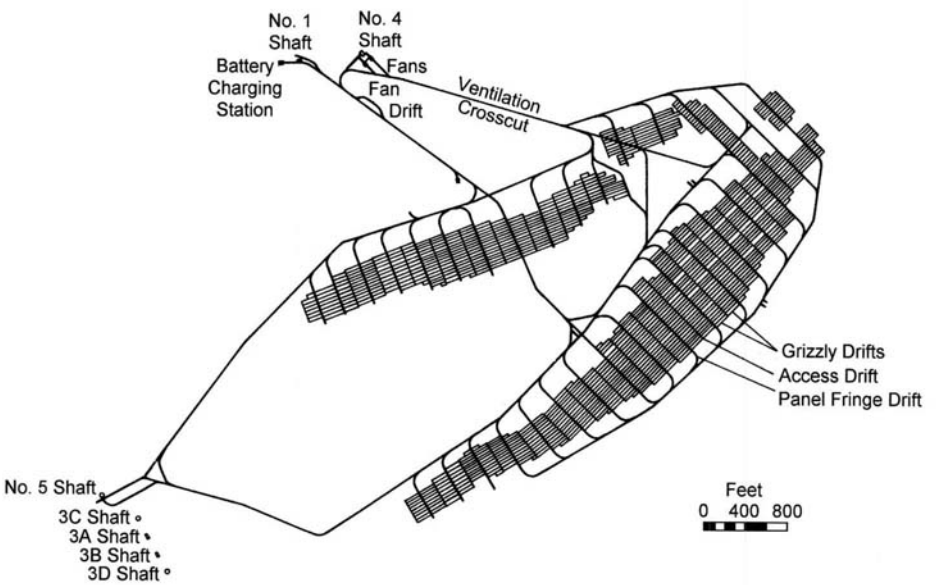


FIGURE 43.4 Main orebody, 2015 grizzly

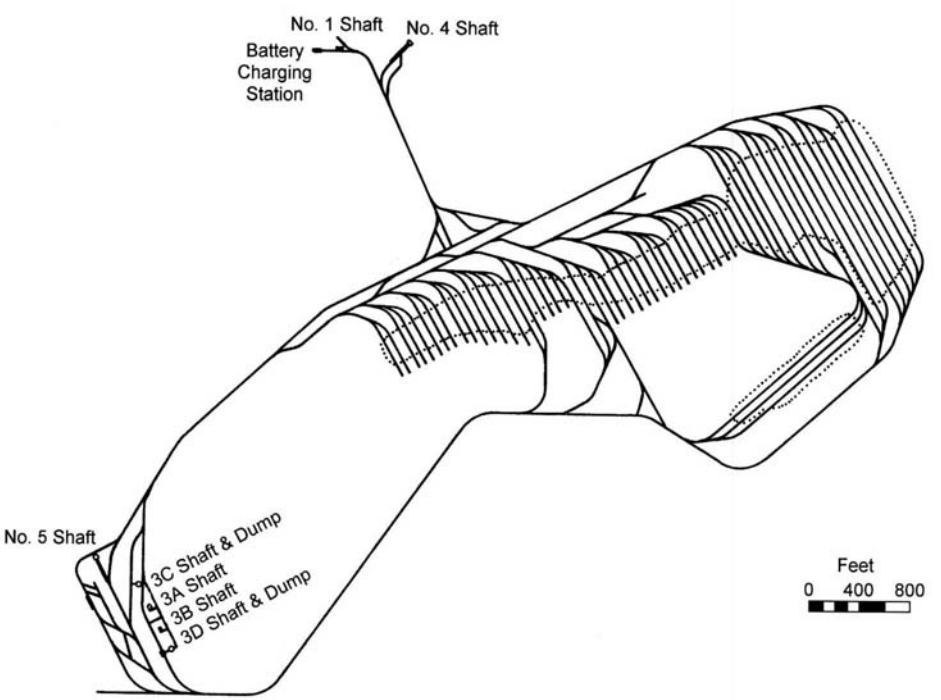


FIGURE 43.5 Main orebody, 2375 haulage

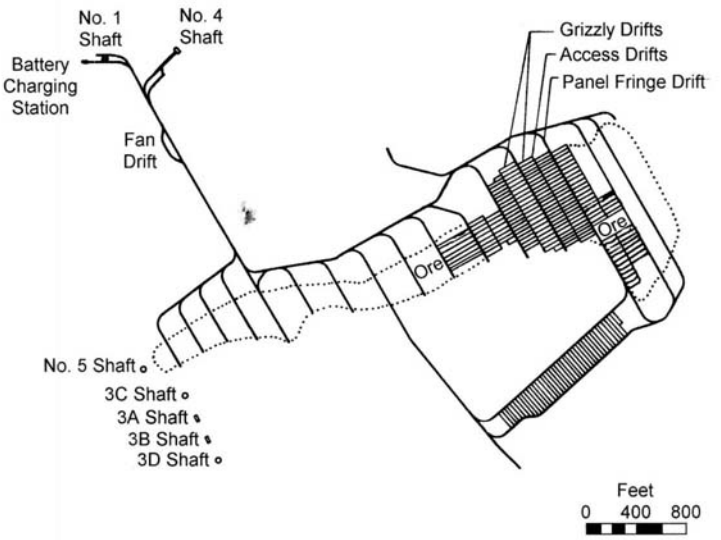


FIGURE 43.6 Main orebody, 2315 grizzly

Transfer Raises

The transfer raises are lined with 15 × 20-cm (6 × 8-in.) cribbing and are 1.22 m (4 ft) in the clear. Each cribbing is protected from wear by a high carbon steel angle, which is nailed onto the cribbing. The transfer raises are driven from each side of the raise station on an angle of 1.1 rad (63°). Each raise consists of a main and a backover branch. The transfer-raise driving crew consists of two men working one shift only.

Grizzly Drifts

After the transfer raise reaches the grizzly level, the grizzly drift can be driven. The grizzly drifts are spaced at 10.6-m (35-ft) centers and are driven parallel to the long axis of the orebody (see Fig. 43.2). This drift is driven by a two-person crew working on one or more drifts at a time using feed-leg machines. The eight grizzlies in the 42.7-m (140-ft) long drift are spaced at 5.3-m (17.5-ft) centers. Practically no hand mucking is required in driving the drift. The preconcrete drift size is 2.7 m (9 ft) high and 2.4 m (8 ft) wide. The ground is supported by rockbolts and wire mesh prior to concreting. The grizzlies are installed as the drift progresses. They consist of 3.6-m (12-ft) 203 × 203-mm (8 × 8-in.) timber sills placed parallel to the drift. The five grizzly rails are 41 kg/m (60 lb per yd) and are spaced to provide a 356-mm (14-in.) opening between rails.

After the grizzly drift is driven, the tops of the transfer raises are formed to fit the transfer-raise top to the grizzly opening. Concrete then is poured around the forms to give support to the grizzly sills.

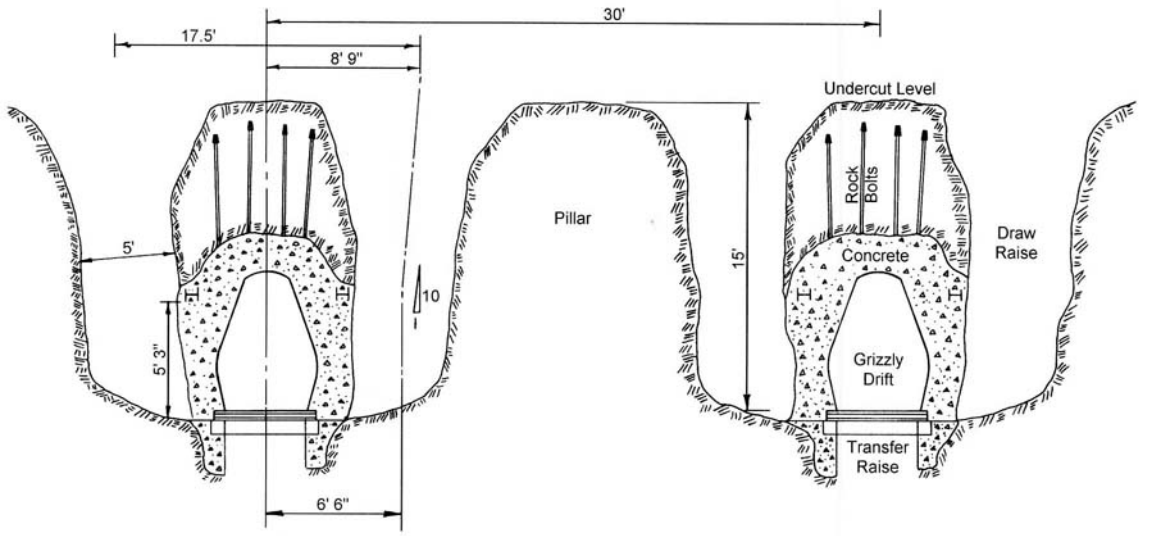


FIGURE 43.7 Section of grizzly

Grizzly drift forms are constructed of plastic-impregnated plywood braced with 51 × 102-mm (2 × 4-in.) wood ribs. The forms are built in sections, which are easily handled by one person. As the drift is formed, a 152-mm (6-in.) pipe is suspended from the back through which concrete is pneumatically placed. The concrete placed is 127- to 152-mm (5- to 6-in.) slump and is vibrated behind the forms. Drift forms are removed no sooner than 16 hr after the drifts have been poured.

Draw Raises

The draw raises are driven from the grizzly draw-raise opening to the undercutting level (Fig. 43.7). The first round from the draw-raise opening is drilled 1.5 m (5 ft) horizontally with a feed-leg machine, after which the 1.5-m (5-ft)-diam raise is driven steeply with an upward stoper to the undercutting level. The vertical distance from the floor of the grizzly level to the floor of the undercut is 4.6 m (15 ft).

Primary Blasting

In the drift headings, carbamite primed with a 32 × 305-mm (1¼ × 12-in.) stick of Amogel or watergel is used. Detonation is with regular delay electric blasting caps. Raises and undercuts are blasted with 60% Amogel, also blasted electrically. Blasting is done at the middle and end of each shift.

Drilling Equipment

Haulage drifts are drilled with a San Manuel–designed jumbo with three or four 2.4-m (8-ft) booms, mounting 76.2-mm (3-in.) drifters with 1.8-m (6-ft) feed shells. The booms are hydraulically operated. Smaller headings use feed-leg drills. Stopers with 457-mm (18-in.) steel change are used on raises.

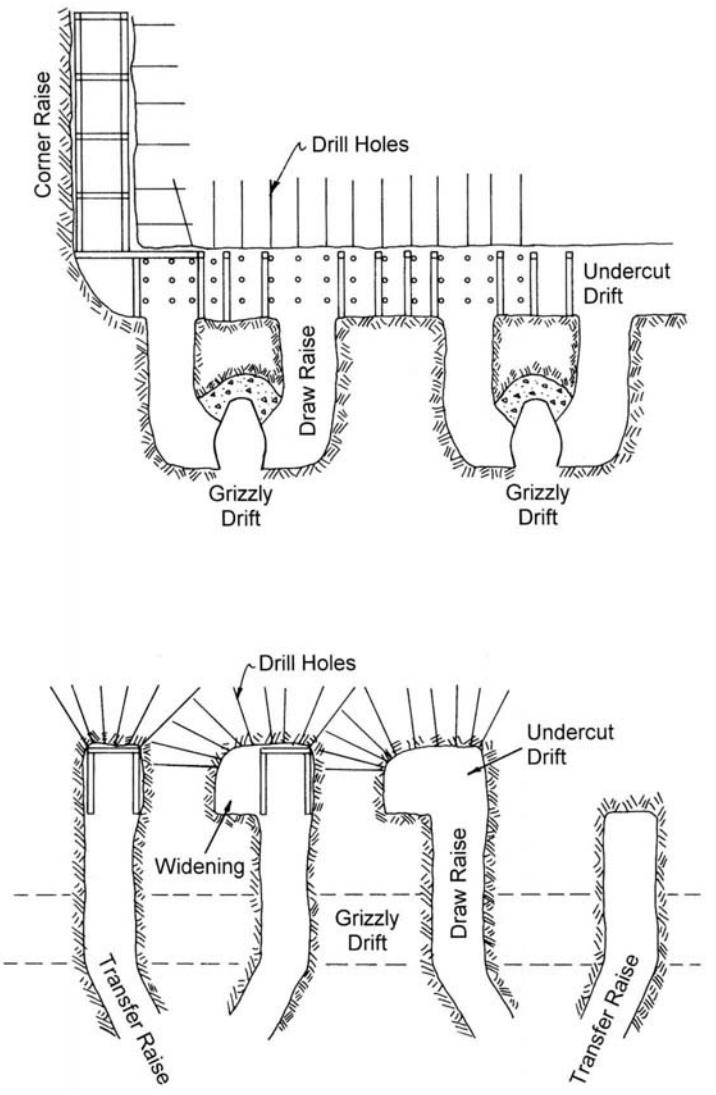


FIGURE 43.8 Undercut drilling

Mucking Equipment

Mucking is done with either a Hagglund mucker or a rocker shovel having a steel flight conveyor dumping into 9-t (10-st) bottom-dump development cars. The cars are switched by a 7-t (8-st) storage battery locomotive.

Undercutting

For the undercutting procedure, see Figs. 43.8 and 43.9.

The undercut drifts are 1.5 × 2.1 m (5 × 7 ft) timbered with 152 mm (6 in.) round posts and 152 × 203-mm (6 × 8-in.) timber caps. The drifts are driven

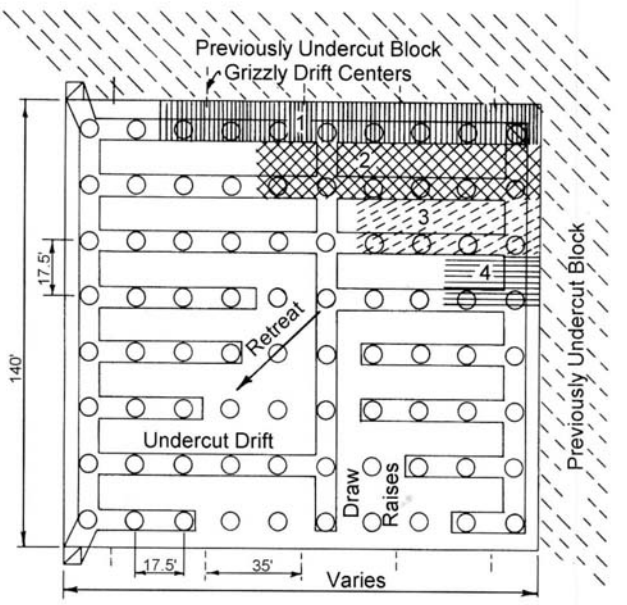


FIGURE 43.9 Pillar blasting

over the tops of the draw raises and at right angles to the grizzly drifts. Access undercut drifts are driven parallel to the grizzly drifts over the tops of the northernmost or southernmost draw raises. In blocks of five or more grizzly lines, an additional access undercut drift is usually driven across the center of the block. Corner raises are driven only in relatively hard rock and only on the virgin corners of the block. Undercutting usually is started against a caved block, retreating diagonally toward solid corners. The cave is started by drilling and blasting a pillar between drifts or at a block boundary. Before each pillar is blasted, the drift is widened on one side about 1.2 m (4 ft); the back and the remaining 2.4-m (8-ft) pillar beyond the widened drift are drilled out. The back is drilled out to a height of 4 m (13 ft) above the undercut floor. The timber supporting the undercut drift is drilled with wood augers and shot with the undercut round. For pillar size and sequence of blasting pillars, see Fig. 43.9. Generally, a 5.3-m (17.5-ft) section along the pillar from one draw raise to the next is taken with each blast. Millisecond-delay electric caps are used in pillar blasting and are wired in series parallel with a maximum of 25 primers in any series. Circuits are tested with a galvanometer before being connected to the power source. The undercut blasting circuit is independent of all other blasting circuits. After the undercut blast, sufficient muck is drawn from the blast area to be visually certain the pillar is completely broken.

Timber. Timber includes round posts and 152 × 203-mm (6 × 8-in.) caps with 152 × 203-mm (6 × 8-in.) stringers or sills over raise tops; 51 × 305-mm (2 × 12-in.) back and side lagging; and raise tops temporarily covered with 51-mm (2-in.) lagging.

TABLE 43.1 Production table (to December 1979)

Level	Tons Mined		End of Mining
	t	st	
1415	63,475,184	69,983,665	1965
1715	28,206,970	31,092,864	1972
2015	172,139,400	189,789,866	Active
2315	30,339,311	33,450,177	Active
2615	1,226,834	1,352,628	Active

Crews. Two-person crews drill 10 to 18 feed-leg holes and 20 to 30 stoper holes per shift, depending upon ground conditions. Undercutting proceeds on a three-shift/five-day-per-week basis.

Explosives. Holes for pillar blasting are primed and loaded using 60% Amogel and millisecond electric blasting caps. The undercut shots are fired from the central blasting station along with the other blasting circuits at lunchtime or at the end of the shift. A separate and independent feeder is used for all undercutting blasts.

PRODUCTION

In 1979, an average of 55,273 t (60,940 st) were produced per day. Production is from three levels (see Figs. 43.3 and 43.4). Production systems have a capacity of 56,700 t/d (62,500 stpd). Average ore grade is 0.65% recoverable copper with 0.25% molybdenite. Tons per worker-shift are: 91 t (100 st) for mine development; 73 t (80 st) for mine production; and 36 t (40 st) for mine total.

Tons of ore per pound of explosive used in secondary blasting average 18 t (20 st). To assure good flexibility for production of 56,700 t/d (62,500 stpd), 2000 active drawpoints, or an area of 56,907 m^2 (612,500 sq ft), are needed. Average drawdown of this area is 457 mm (18 in.) per day (Table 43.1).

Mining cutoff grade on the 1415 level was 0.60% Cu. Currently, cutoff grade is 0.50% sulfide Cu. Total production to the end of 1979 including development was 311,062,820 t (342,957,912 st) at 0.760% total Cu, 0.053 oxide Cu, and 0.707% net sulfide ore.

Secondary Blasting

Watergels and ammonium-nitrate–based explosives in 227- and 340-g (½- and ¾-lb) plastic bags, initiated with zero-delay electric blasting caps, are placed on large rocks in the draw raises or on the grizzlies.

Draw Control

Control of the draw is maintained at the grizzly with ore drawn in accordance with daily written orders issued by block engineers. A close check is kept on

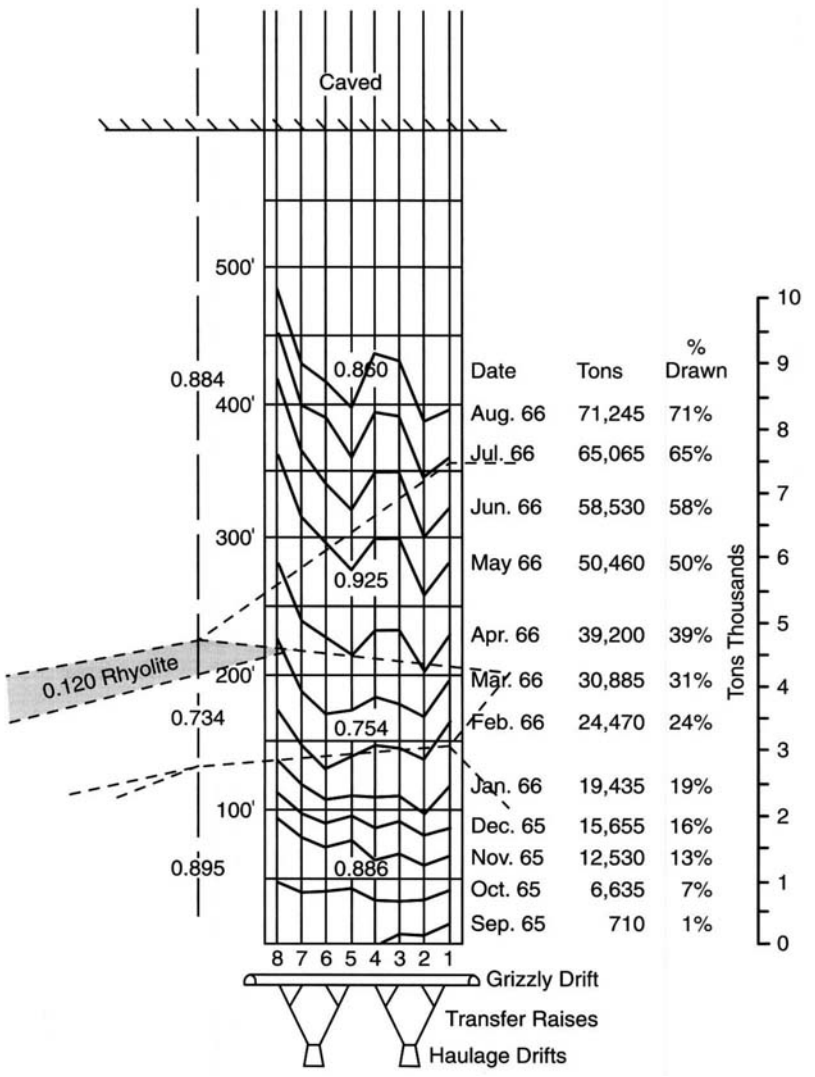


FIGURE 43.10 Stope draw chart

tonnage from each draw raise by filling and drawing the calibrated transfer raise between the haulage and grizzly levels. The ore is fed from the draw raise through 356-mm (14-in.) grizzly openings into the transfer raise by a chute tapper. The chute tapper is closely supervised by a draw boss who is responsible for carrying out the draw orders (Tobie and Staley, 1967).

The ore drawn from each line of draw raises is plotted by the tonnage department once a month, as shown in Fig. 43.10. These charts are used by the block engineers in maintaining the desired angle of draw over the individual grizzly lines.

Subsidence

Initial surface manifestation of subsidence came when draw from the first blocks had reached 8 to 12%, which on a drawdown rate of 457 mm (18 in.)

per day required nearly 5 months. First surface cracks were bedding-plane slips in the conglomerate, and these were followed shortly by measurable sag over the mining area of 183 mm (7⅛ in.). The sag mechanism generated a series of arcuate peripheral tension cracks, which eventually became the perimeter of a large oval area inside of which subsidence occurred relatively rapidly and outside of which changes occurred very slowly. The differential movement thereby gradually produced a steep scarp line around the actively subsiding area. For example, at 12 months' draw, maximum subsidence within the oval had reached 8.8 m (29 ft), and at 24 months' draw, with additional blocks undercut, maximum subsidence had reached 41 m (134 ft) within the large oval.

Occasionally during the first five years of operation, small independent "chimneys" appeared at the surface within the larger subsiding perimeter. These independent collapses were normally structurally controlled by steep faulting, and their surface area was always somewhat smaller than the area of the undercut that had produced them. None ever produced a noticeable disturbance in the mine as they occurred.

The vertical angle measured from the edge of a given undercut to the outermost known tension crack is a considerable variable, which is strongly affected by in situ rock-mass physical characteristics, major structure, and dimensions of the undercut perimeter. Average angles as low as 0.7 rad (40°) and as high as 1.1 rad (65°) have developed, dependent on the geology of the rock mass through which they rise.

To mid-1976, the mine had produced 249,302,520 t (274,809,000 st) of ore over 21 production years. Calculations relating the volume of subsidence to the volume of ore withdrawn have not been done for many years, but the ratio equalled about 90% and had apparently leveled off at the time of the last computation.

MINE SERVICES

Shafts

Shaft locations are shown on accompanying level plans, Figs. 43.3 to 43.6 inclusive. Shafts 1, 4, and 5 handle personnel and supplies and provide intake ventilation capacity. Shafts 3A, 3B, 3C, and 3D provide muck-hoisting capacity and exhaust ventilation sufficient to balance the intake capacity. See Table 43.2 for additional details.

Automation

Skip loading and hoisting at the production shafts are fully automated. Skip loading and dumping are viewed on closed-circuit television (Tobie, 1971).

Primary Crushing Plant

Coarse-ore bins with a diameter of 18.3 m (60 ft), height of 20.2 m (66¼ ft), and bin live-load capacity of 6350 t (7000 st) are placed at each shaft into which the two skips dump. A 2.4-m (8-ft) pan feeder conveys the ore from the

TABLE 43.2 Shaft data

Type*	1 Shaft Service	4 Shaft Service	5 Shaft Service	3A Shaft Ore Hoisting	3B Shaft Ore Hoisting	3C Shaft Ore Hoisting	3D Shaft Ore Hoisting
Size, m	7.8 × 1.8	8.1 × 4.3	7.6 diam	8.8 × 2.1	8.8 × 2.1	6.7 diam	6.7 diam
Depth, m	862	832	1257	704	704	871	1132
Ventilation	Intake	Intake	Intake	Exhaust	Exhaust	Exhaust	Exhaust
Hoisting compartments	3	2	4	3	3	3	3
Headframe height, m	31	33	48	55	55	61	61
Sheave size, m	3.6	4.3	4.3	4.3	4.3	4.6	4.6
Main Hoist							
Type (drum)	Double	Double	Double	Double	Double	Double	Double
Capacity	12 people	100 people	100 people	19.0 t	20.9 t	26.3 t	26.3 t
Rope size, cm	2.86	5.72	5.40	5.72	5.72	5.72	5.72
Power, hp	400	1400	2000	6000	6000	7000	7000
Speed, mph	4.06	7.62	9.14	14.22	14.22	14.22	14.22
Guides	Timber	Timber	Timber	Steel	Steel	Steel	Steel
Secondary Hoist							
Type (drum)	Single	—	Double	Single	Single	Single	Single
Capacity	6 people	—	5 t	18 people	18 people	9 people	9 people
Rope size, cm	2.5	—	3.5	2.5	2.9	2.5	2.5
Power hp	200	—	1000	250	250	250	250
Speed mph	2.77	—	9.78	1.27	1.27	1.27	1.27
Guides	Timber	—	Timber	Timber	Timber	Timber	Timber

*Metric/English equivalents: m × 3.2808 = ft; t × 1.102 = st; cm × 0.39 = in.; hp × 0.7456999 = kW; mph × 1.6 = km/h.

coarse-ore bin to the 1067 × 1651-mm (42 × 65-in.) gyratory crusher. Capacity of the crusher is 1361 t/h (1500 stph). Conveyor belts of 1219 mm (48 in.) transfer the crushed ore from the crusher building to transfer towers and thence to the fine-ore bins. Skip dumping, fine-ore bins, and transfer belts are viewed by closed-circuit television. The entire automated crushing-transfer system is monitored from control panels at each crusher location.

From the fine-ore storage bins, the ore is loaded through air-operated drop-bottom gates into 90.7 t (100 st) air-operated bottom-dump cars and hauled 11.2 km (7 miles) on a standard-gage railroad to the mill ore bins.

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CHAPTER 44

The Henderson Mine

William G. Doepken

GENERAL DESCRIPTION

The Henderson mine utilizes a highly mechanized, continuous panel-caving system to extract ore from a deep, massive, molybdenum deposit. The mine site, located 80.5 km (50 miles) west of Denver, is situated at 3170 m (10,400 ft) above sea level in a steep valley on the eastern side of the Continental Divide. The mill site, located on the western side of the divide, is linked to the mine via a 15.5-km (9.6-mile) railroad tunnel. Mine production was scheduled to reach 27,215 t/d (30,000 stpd) from the uppermost production level within the four years of an intensive ten-year preproduction development program in the late 1970s. A second production level, due to come on stream in later years, would expand production capabilities to 31,751 t/d (35,000 stpd). The mine, which employs approximately 2000 people, has blocked-out ore reserves of 220 Mt (242 million st) at an average grade of 0.42% MoS_2.

Geology

The Henderson deposit is composed of two partially overlapping orebodies that lie 1080 m (3600 ft) beneath the peak of Red Mountain. The orebodies are entirely within a Tertiary rhyolite porphyry intrusive complex that has intruded Precambrian granite. The deposit is elliptical in plan with overall dimensions of 670 × 910 m (2200 × 3000 ft). In section it is arcuate with an average thickness of 185 m (600 ft). The top of the deposit is at an elevation of 2610 m (8700 ft), whereas the lower limits range from 2340 m (7800 ft) on the west to below 2100 m (7000 ft) on the east. The mineralization is relatively continuous in the orebodies and consists of molybdenite and quartz in random to subrandom, intersecting, closely spaced veinlets.

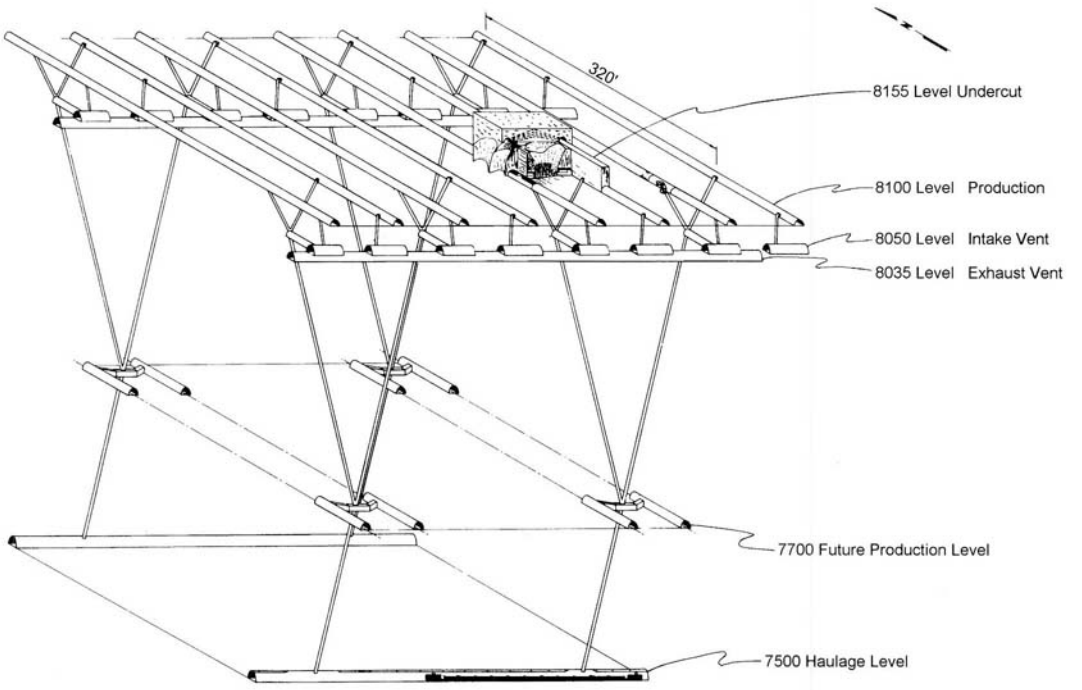


FIGURE 44.1 Isometric of mining levels, Henderson mine

Cavability

Rock quality designation [RQD; percentage of core in pieces greater than 100 mm (4 in.)] values measured from Henderson drill core and compared with the RQD values from the Climax and Urad block-cave mines indicated that the Henderson ore would cave but with difficulty, and that a high percentage of the cave material would be blocky. Initial caving experience shows the Henderson ore caves easier and with greater fragmentation than was indicated by the RQD values. Possible reasons for this are:

6. The greater lithostatic load at the Henderson has a greater effect on caving and fragmentation than anticipated.
7. The relief of confining pressure due to caving action and brittleness of the rock is enhancing breakage.

Mine Layout

The Henderson mine layout was designed from its inception for a large-scale, rubber-tired, continuous panel-caving mining system. The orebody is vertically divided into three major mine levels (Fig. 44.1). The topmost level (8100 level), which is 2469 m (8100 ft) above sea level, was the mine production level in the late 1970s. The intermediate or 7700 level provides subsequent production capabilities. The bottom level (7500 level) is the rail-haulage level. All three levels are provided with both shaft access and interlevel ramp access. Each mine production level has five associated sublevels as follows: an undercut level

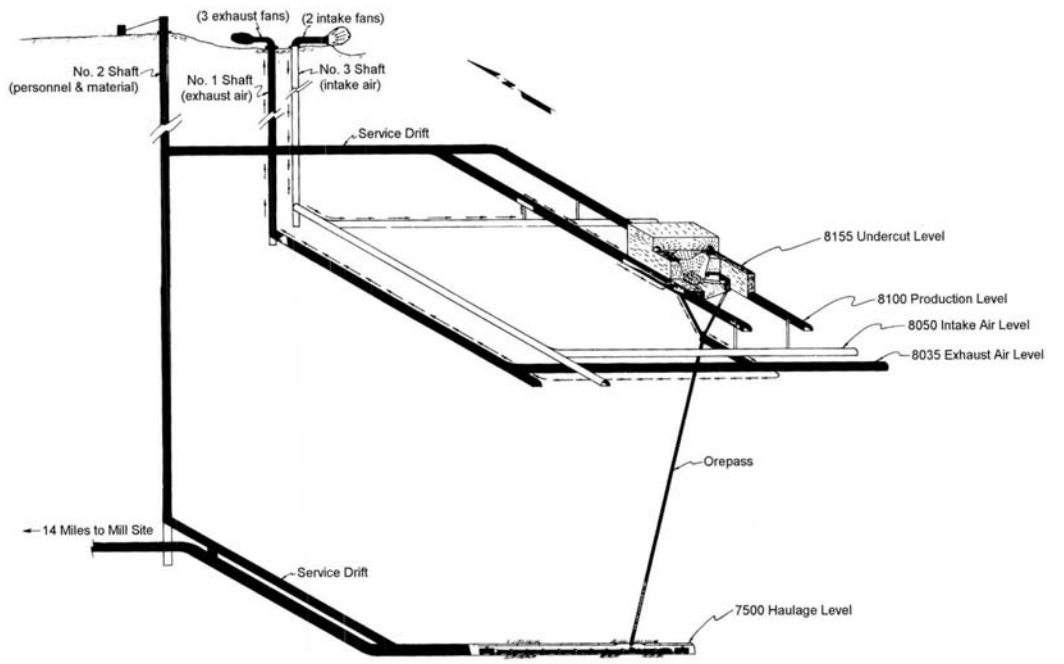


FIGURE 44.2 Ventilation system

located 16.8 m (55 ft) above the production level, two boundary cutoff levels located 44.2 m (145 ft) and 62.5 m (205 ft) above the production level, and intake and exhaust ventilation levels located 15.2 m (50 ft) and 19.8 m (65 ft), respectively, below the production level (Fig. 44.2).

Access for ore extraction is provided by production drifts located on 24.4-m (80-ft) centers across the orebody (Fig. 44.3). Between these drifts, drawpoint crosscuts are driven on 12.2-m (40-ft) centers into which funnel-shaped bells are developed to receive ore from the cave. Drawpoints are mined at 0.8 rad (45°) to the production drift to facilitate load-haul-dump (LHD) access to caved ore while maintaining an adequate rock pillar for ground support.

The cave is initiated from undercut drifts located directly above each production drift. From these drifts, which form the top of the major apex, longhole rings are drilled to develop two adjacent bells and the included undercut. Bell development and undercutting proceed in a sequential and continuous manner, advancing the cave line to bring new bells into production as old ones are exhausted.

Caved ore is loaded with LHDs from drawpoints on either side of a production drift and trammed to orepasses located alongside the drift every 97.5 m (320 ft) or eight drawpoints. These orepasses, which are interconnected between production drifts in a "Y" arrangement, transfer the ore to lower-level train loading drifts for rail haulage to the mill (Fig. 44.1).

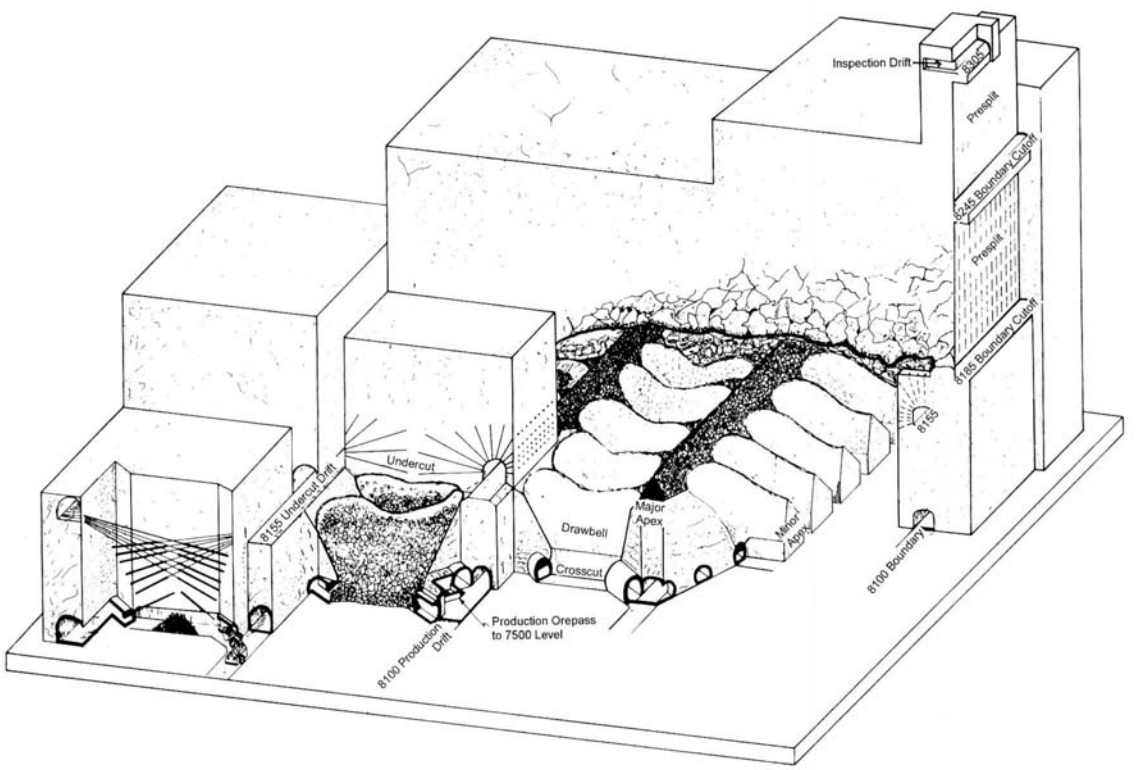


FIGURE 44.3 Isometric of mining plan

Ventilation for dust control and diesel exhaust dilution in the production area is provided by raises from intake and exhaust ventilation drifts. The orepass itself acts as the exhaust raise, whereas a separate intake raise is provided.

Mine Layout Modifications

Two major modifications to the Henderson mining plan are in progress. The first involves drawpoint layout and the second draw-bell development.

Initially, drawpoints were mined in a modified chevron pattern with a 6.1-m (20-ft) offset between left- and right-hand drawpoint entries in a production drift (Fig. 44.4). This offset was designed to minimize the amount of unsupported production drift in a given section. However, because of significant weight damage to drawpoint entries and normal to major tectonic horizontal stresses in the orebody, a new "Z"-style pattern with both entries parallel to the principal stress has been developed.

Draw bells were originally developed from the drawpoint crosscut drift by mining a slot raise and longhole drilling a full bell fan pattern (Fig. 44.5). However, changing ground conditions necessitated the virtual elimination of the overhead belling method. The new-style belling method requires only

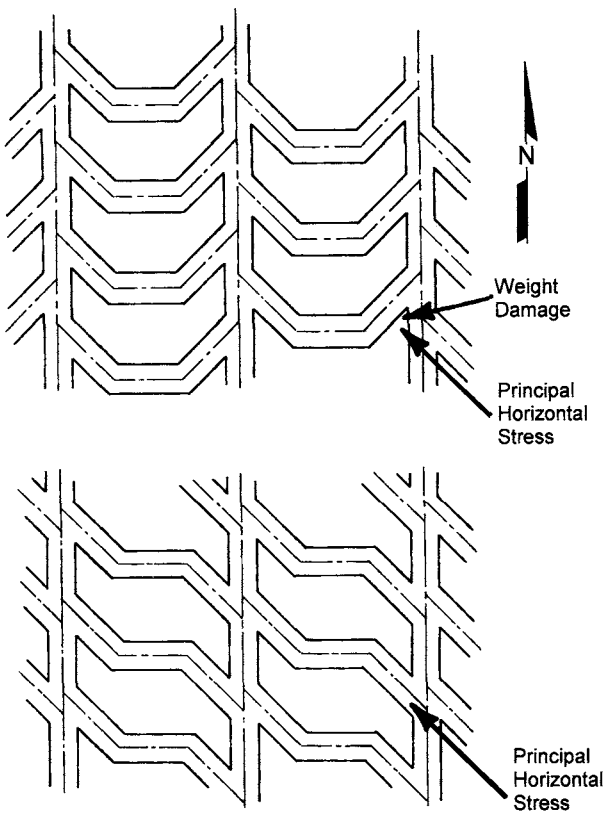


FIGURE 44.4 Drawpoint configuration. Top is old type drawpoint, bottom is new type drawpoint.

V-slot development from the drawpoint crosscut, and the major part of the bell is developed by longhole drilling from the undercut drift.

DEVELOPMENT AND PREPARATION SEQUENCE

The mining plan to bring a production level on stream requires three distinct but overlapping mining phases spanning a period of eight to ten years. The first of these phases, referred to as development mining, establishes access drifts to the production level and associated sublevels. Once shaft access or interlevel ramp access is established, a second mining phase, referred to as preparation mining, develops the production drifts, drawpoints, bells, ore-passes, and ventilation system required for ore extraction. The third phase, referred to as undercutting, provides the drifts, longhole drilling, boundary drifts, and boundary-cutoff longhole drilling necessary to undercut the ore-body and initiate caving.

Ventilation Level

The exhaust ventilation drifts are initially driven beyond immediate production requirements to establish diamond-drilling stations. Both intake- and exhaust-ventilation-level mining advances continue well in advance of

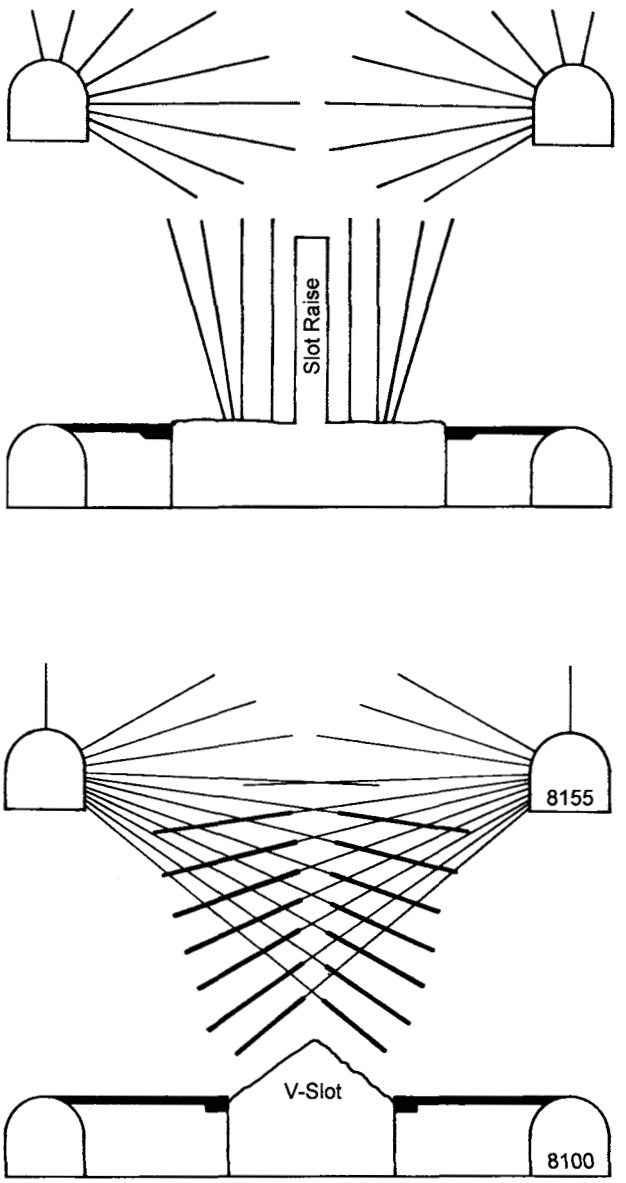


FIGURE 44.5 Draw-bell drilling. Top is old style bell development, bottom is new style bell development.

production scheduling to provide ventilation for other preparation mining areas. The exhaust level is also required for early access to mine orepass raise-boring stations.

Production Level

Following ore-zone definition by diamond drilling, fringe and production-drift development begins in the initial production area. After a production drift is completed, allowing access from both ends, concrete drift linings are poured between drawpoint intersections. Drift lining is immediately followed

by the sequential mining of drawpoint turnouts and crosscuts. Completed turnouts are concrete-lined, and concrete or steel brows are installed. The mining of connecting raises for orepasses and intake ventilation closely follows concreting operations. After the raises are completed, concrete dump pockets are installed around the orepass. This operation is followed by the final preparation mining stage on the level, that of mining bell V slots.

Undercut Level

The mining of undercut access and undercut drill drifts occurs in the same area and at the same time as production-drift mining. Ramps are mined to the boundary-cutoff levels to allow boundary development to proceed at the same rate as undercut development. When an undercut drift is completed, ring drilling for bell and undercut development commences. Ring-drilling operations are concurrent with the drawpoint concrete lining operations.

Ore-Pass Development

During the time frame of production-drift development, access drifting for orepass raise boring is being developed on the intermediate production level and the train haulage level. By the time that raise-boring stations have been developed on the exhaust level, access is available to the 7500 level and the 7700 level.

MINING PRACTICES

Drift Mining

Two standard drift cross sections are used in the majority of development, preparation, and undercut drifting applications. A 3.7 × 3.7-m (12 × 12-ft) arched drift section is utilized for boundary, undercut, sublevel ramp, ventilation, and train-loading drifts, whereas a 4.2 × 4.1-m (14 × 13.5-ft) arched drift section is utilized for production, drawpoint, interlevel ramp, shaft access, and perimeter drifts. However, some larger drift cross sections are mined for main ventilation laterals connecting ventilation levels to the intake and exhaust shafts. With the exception of ramps, which are driven at grades of up to 15%, all drifting is done at a nominal grade to allow drainage.

All drift drilling is accomplished with two- and three-boom diesel-powered jumbos equipped with 3.7-m (12-ft) feeds and 139.7-mm (5.5-in.) pneumatic rock drills. Jumbos drill 44.5-mm (1.75-in.), 50-hole, 3.3-m (10-ft) rounds in the smaller drift sections and 55-hole rounds in the larger drift sections.

Explosives charging is carried out from lift-deck powder trucks. The trucks are equipped with 136-kg (300-lb) capacity ammonium nitrate–fuel oil (ANFO) pressure vessels to pneumatically load blastholes. Contour blasting, utilizing 22 × 610 mm ($\frac{7}{8}$ × 24-in.) trimming powder in arch perimeter holes and ANFO in the balance of the round, reduces drift overbreak. While the majority of drift blasting is performed with ANFO, wet areas are blasted with 32 × 406-mm (1.25 × 16-in.) stick powder. A bottom-prime detonating system utilizes standard, delay-period, nonelectric caps and 17-g (266-grain) boosters. All

rounds loaded during a shift are tied into a central electric blasting circuit and shot at shift change.

Muck from drift rounds is handled by 3.8-m^3 (5-cu yd) LHDs, which tram the muck to the nearest development orepass. Where possible, tramming distances are held to within 457 m (1500 ft) to maintain mucking efficiency. Double handling the muck is not uncommon, and transfer raises are utilized frequently.

Bell and Undercut Mining

Both the draw bell and undercut are developed by longholes drilled from the undercut drift. Longholes are drilled with twin-boom, air-powered, rubber-tired jumbos equipped with 1.2-m (4-ft) feeds and 140-mm (5.5-in.) pneumatic rock drills. Jumbos drill 76-mm (3-in.) longholes up to 18 m (60 ft) in length using 1.2-m (4-ft) coupled drill steel. Longholes are drilled in 23-hole rings with hole inclinations set to maintain a 1.7-m (6-ft) spacing between hole bottoms. Holes below the horizontal develop bells, whereas those above develop undercut (Fig. 44.3). The rings, spaced on 2.1-m (6.7-ft) centers along the undercut drift, are drilled two at a time. Six half-rings spanning 12 m (40 ft) in each of two adjacent undercut drifts are required to develop each draw bell.

Bell blasting is accomplished with powder trucks equipped with twin 136-kg (300-lb) ANFO pressure vessels. The blastholes are loaded and shot in stages to allow shot rock to be removed to provide room for swell muck from the next shot. Each hole is bottom-primed with a nonelectric cap and booster, and a 486-mg (7.5-grain) detonating cord brought to the collar of the hole provides the tie line. Only the bottom 20 to 30% of each hole is loaded with ANFO. The tie lines are connected to a 1.6 g (25-grain) trunk line, which is hooked into the central blasting circuit.

Undercut blasting, which occurs at the cave line, is done in three-ring increments. Rings are shot only into fully developed bells to allow for shot swell muck. The undercut longhole rings are primed, loaded, and shot using the same method as the bell longholes.

Boundary-Cutoff Mining

The orebody boundary cutoff is developed by vertical longholes drilled from high boundary drifts. During initial cave development, the first two lifts of boundary cutoff were developed as continuous break slots using a 1.5 × 1.5-m (5 × 5-ft) trenching drill pattern with 76-mm (3-in.) blastholes. The third lift, drilled 18 m (60 ft) vertically up from the topmost boundary drift, was developed as a presplit with holes on 1.5-m (5-ft) centers. With a more mature cave, the break slots were eliminated and all three lifts, with the exception of short break slots at boundary corners, are presplit.

The first two lifts are drilled down, and the third lift is drilled up with air-track drills utilizing 1.7-m (6-ft) coupled drill steel. The presplit blastholes are shot instantaneously in long sections to maximize the presplit effect.

Raise Mining

Raise boring is utilized to develop production orepasses to transfer ore from the production drifts to the train haulage level. These orepasses, which include a 168-m (550-ft) primary branch and a 76-m (250-ft) feeder branch, are bored from drill stations located adjacent to exhaust ventilation drifts. Using a 149.2-kW (200-hp) dc raise-boring machine, a 311-mm (12.25-in.) pilot hole is drilled at approximately 1.2 rad (70°) to intersect the target level. A 16-cutter reaming head is then attached, and the pilot hole is backreamed 2.4 m (8 ft) in diameter to within 4.6 m (15 ft) of the boring machine. After the borer has been removed from the drill site on an air-track carrier, the top of the orepass is shot in.

Drop raising or longhole raising is used to develop the orepass connections between the production drifts and the tops of the bored orepasses. This raising method, which utilizes an air-track drill, is also required for ventilation connection raises and development-muck transfer raises. These raises require ten to sixteen 76.2-mm (3-in.) downholes to develop, and they are blasted in 3.3-m (10-ft) increments from the bottom by lowering explosive charges down the holes.

Ground Support

Two distinct types of ground support are required for development- and production-drift openings. The minimum support requirement for development drifts is nine rockbolts and three 2743 × 305-mm (108 × 12-in.) corrugated steel mats. Both 2.1-m (7-ft) expansion-shell and 1.5-m (5-ft) split-tube friction rockbolts are used. Where additional support is required to prevent local spalling, shotcrete is applied.

Support of production-drift and drawpoint openings requires concrete lining and in some areas steel sets in concrete to withstand cave abutment loading. After the cave line has moved across a production area, the concrete linings often require reinforcement with rockbolts and repair with shotcrete.

Concrete Operations

Concrete-drift and drawpoint linings are formed with steel pan-type forms, which provide a 305-mm (12-in.) minimum lining thickness. The concrete is wet-batched in a surface plant and sent via 203-mm (8-in.) shaft slicklines to the mine levels. Concrete transit trucks with a rated capacity of 4.6 m^3 (6 cu yd) are loaded at the shaft station. Concrete delivered to the work sites is pumped into the forms with the twin-piston concrete pumps.

Shotcrete is dry-batched, dropped through the slickline, and loaded into 2.3-m^3 (3-cu yd) transit trucks. At the shotcreting site the dry batch is mixed with water, the mixing vessel is pressurized, and the shotcrete is sprayed with a boom-mounted nozzle. Typical single-pass shotcrete depths are 25.4 mm (1 in.) on the back and 50.8 mm (2 in.) on the ribs.

TABLE 44.1 Development and production equipment fleet size

	Units
Production Equipment*	
5-cu yd* LHD	26
3-cu yd LHD	6
Hang-up drill jumbos	5
Hydraulic rock breakers	3
Hang-up powder trucks	3
Development Equipment*	
5-cu yd LHD	25
3-cu yd LHD	4
3-boom jumbo	10
2-boom jumbo	5
Rock-bolt jumbo	9
Ring-drill jumbo	7
Fan-drill jumbo	4
Track-drill jumbo	7
Drifting powder truck	6
Long-hole powder truck	3
6-cu yd concrete truck	7
3-cu yd concrete truck	8
Shotcrete trucks	6
Concrete pumps	3
Road maintenance equipment	11
Service trucks	21
Utility trucks	10
Personnel trucks	36

*Metric equivalent: cu yd × 0.7645549 = m^3.

Mining Equipment

A highly mechanized fleet of diesel mining equipment is utilized in all phases of mine development and production. This fleet, which includes 43 units of production and 182 units of development equipment, will further expand to 275 units to reach and maintain a 31,751-t/d (35,000-stpd) production rate. The development and production equipment fleets for a 27,215-t/d (30,000-stpd) production rate are shown in Table 44.1.

Maintenance of this equipment fleet is performed in a well-equipped, 36-space, main underground shop and three smaller remote satellite shops. These shops perform all preventive, scheduled, and breakdown maintenance functions. Diesel fuel and oil are pumped directly from surface facilities to a central fueling and lubrication bay and are trucked from there to remote fueling stations.

Additional surface maintenance facilities maintain all surface plant and equipment. These shops also perform the complete overhauls of underground equipment. Control of all maintenance functions is provided by the maintenance program utilizing a computerized record-keeping system.

Mine Ventilation

Mine ventilation is provided by a unique push/pull system of intake and exhaust air. The main fans, located on the surface, supply fresh air down No. 3 shaft and exhaust contaminated air up No. 1 shaft. Each of the two intake and three exhaust fans is 3 m (10 ft) in diam powered by a 932.5-kW (1250-hp) dc motor and capable of moving 378 m^3/s (800,000 cfm) of air. By creating a slight negative pressure underground, fresh air is also supplied down No. 2 shaft for lower-level ventilation.

Separate intake and exhaust levels provide various mine workings with ventilation through strategically located raises. Booster fans underground are utilized to aid and direct airflows. Development headings are generally the only mine areas that require the use of ventilation ducting to provide necessary airflows.

Ventilation of the ore haulage tunnel is provided by air intake from both the mine and the tunnel portal. Tunnel air is exhausted through No. 4 shaft, a 483-m (1585-ft) deep, 3.4-m (11-ft) diam concrete-lined opening located at the midpoint of the tunnel. This shaft is equipped with two underground 94 m^3/s (220,000-cfm), 373-kW (500-hp) fans.

Carbon monoxide monitors coupled to automatic louvers are utilized to redirect certain airflows in case of a fire. A remote monitoring system on some fan and air-door installations indicates failures and provides stop and restart capabilities.

PRODUCTION

To maintain daily production rates, a fleet of unique high-production 3.8-m^3 (5-cu yd) LHDs trams broken ore from assigned drawpoints a maximum 49 m (160 ft) on concrete roadways to orepasses. Under normal production conditions, 320 drawpoints are available for draw. Each operating LHD services 8 to 12 drawpoints and averages 816 t (900 st) per shift with a 4.2-t (4.6-st) carry factor.

Secondary Blasting

Secondary blasting to bring in hang-ups and reduce oversize ore, even in the early stages of drawdown, is infrequent. Many apparent hang-ups are broken up with the LHDs. When true hang-ups occur, they are drilled and shot, whereas oversize ore is removed to a nearby off-draw drawpoint to be drilled and shot or reduced with a hydraulic rock breaker.

Draw Control

Ore draw control is closely administered by assigning the number of LHD buckets to be drawn from each drawpoint every day. Computerized records are maintained to provide accurate and current records of total tons drawn and the current ore grade for each drawpoint. Grab samples taken from drawpoints after every 907 t (1000 st) of production establish operating grade, whereas more frequent 91-t (100-st) interval samples are used to determine drawpoint cutoff when the economic grade limit is reached.

Draw-control engineers balance the rate of draw and the rate of cave, which is approximately 0.3 m/d (1 ft per day), to minimize the hazards of large voids or packing in the cave. To minimize dilution, draw is scheduled to maintain an approximate 0.98 rad (56°) ore-waste contact between uncaved ore and exhausted drawpoints. This ore-waste profile is maintained by progressively increasing the total ore column draw in 15% increments in each row of drawpoints working away from the cave line.

Cave-Line Management

The coordination of all cave-line–related mining activities is critical to the success of the mining operation. The preparation, undercutting, and production schedules must be maintained to keep the cave line advancing in an optimum, stepped, concave front. Experience indicates that large irregularities in the cave line cause dramatic increases in weight concentrations and serious drift and drawpoint damage. To minimize the effects of weight concentrations, the steps in the cave line between production drifts are maintained in a range of 6 to 12 m (20 to 40 ft).

Rail Haulage System

A unique rail haulage system is used to transport ore from the mine to the mill via a 15.5-km (9.6-mile), twin-track, railroad tunnel with a +3% grade to the portal and an additional 6.4 km (4 miles) of twin-track surface road. Each of the nine trains required for full mine production consists of three 45-t (50-st), 1400-V dc, electric locomotives and twenty-two 12-m^3 (424-cu ft) bottom-dump ore cars. The trains are continuously loaded out of orepass surge bins with side-loading drop-lip chutes. The rate of flow of ore is controlled by pneumatically actuated flow chains. Each car averages 17.4 t (19.2 st), and a train hauls 383 t (422 st) per trip. The 48-km (30-mile) haulage cycle requires 150 min to complete at a 40 km/h (25 mph) train traveling speed.

MINE SERVICES

Electrical Supply

Electrical power is supplied to the mine site at 115 kV through a closed-loop system to eliminate power interruptions. From the mine-site substation, a 13.8-kV loop transmits power to underground substations, which provide 4160-V, 480-V, and 120-V power for underground usage.

Compressed-Air Supply

A surface compressed-air plant provides 1430 m^3/s (30,000 cfm) of compressed air to underground workings through a loop distribution system. This capacity is developed by four water-cooled, four-stage, centrifugal compressors.

Access Shafts

Personnel and material access into the mine is via No. 2 shaft, a 945-m (3100-ft) deep, 8.5-m (28-ft) diam concrete-lined shaft. Hoisting capabilities are provided by two counterweighted double-drum hoists with 4.6-m diam by 2.4-m face (15-ft diam by 94-in. face) grooved drums holding a double wrap of 57 mm (2.25-in.) wire rope. A three-compartment shaft arrangement provides a 7.9 × 2.7-m (26 × 9-ft) personnel/material cage (18 t or 100 people), a double-deck personnel cage (100 people), and a small shaft inspection cage (4 people).

Two additional mine-site shafts provide airways for intake and exhaust ventilation. No. 1 shaft is 744 m (2440 ft) deep, and No. 3 shaft is 698 m (2290 ft) deep. Both shafts are 7 m (23 ft) in diam inside concrete.

Mine Dewatering

The mine dewatering system handles 7570 L/min (2000 gpm) through a complex of twin inclined settling sumps. These settlers progressively clarify the mine water. Eight 9463-L/min (2500-gpm) pumps lift the water in two stages from the 7500 level to the main, 8100-level, pump station. From this level, water is pumped in a single 701-m (2300-ft) lift to the surface with four 1.3 MW (1750-hp), 6283-L/min (1660-gpm), six-stage, horizontal, centrifugal pumps. At the surface, the water is cooled by aeration and further settled before it is discharged into the watershed.

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