CHAPTER 69

Backfill in Underground Mining

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69.1 INTRODUCTION

Backfill can fulfill several roles at an underground mine site. It can be used as:

- A construction material
- A major ground-support tool
- A primary mine-waste-disposal method

The materials used for backfill at most underground mines generally consist of mill tailings or waste rock from underground or open-pit mining. At sites where these materials are not available in sufficient quantities to meet the mining-method requirements, alternative sources must be found. Alternative sources such as alluvial sands, quarried rock, or air-cooled smelter slag are used to replace or supplement the mine waste as underground backfill.

As an "underground construction material," mine backfill is used as:

- A floor to mine from
- A wall to mine next to
- A roof to mine under

In most mines, a binder (cement, slag cement, and/or fly ash) must be mixed with the backfill material to give it the strength required for the above construction purposes.

Backfill is the "major ground-support tool" in most underground mines. When placed in an excavation or open stope, backfill provides confinement for the walls and back (when tight filled). This confinement significantly improves the overall stability of the rock mass by limiting closure and preventing unraveling. In most mines, long-term ore removal would be impossible without using backfill to support the rock mass.

Mine backfill, placed as an underground construction material, most often plays a dual role and, consequently, contains a binder. However, if a backfill is being placed only for ground support (i.e., secondary or bulk-mining slopes), a binder is not added. Under certain circumstances, a small amount of binder is occasionally added to paste backfill to guard against liquefaction.

Today, the mining industry uses mine-waste materials for backfill as its primary mine-waste-disposal method. Using this method has become a priority because of the increasing perception among the general public that disposing of mine wastes (tailings, slurry containment, cyanide/arsenic ground water contamination, and acid generating from tailings/waste rock) on the surface is physically and environmentally unstable. The small percentage of surface mine-waste-storage facilities that have failed in recent years have had a substantial negative economic impact on the mining companies involved and on the mining industry in general. As a result of public and political pressure, regulations for surface mine-waste storage have become more onerous, and the potential liability associated with storage-system failure has increased substantially. These factors have caused corporate management and project financiers to push for reducing the storage of mine waste on the surface and to insist on more conservative surface-disposal designs that are more cost effective in the long term

This has led to a greater emphasis in the mining industry on maximizing the return of the waste to the underground work place in the form of backfill. This emphasis has added to the increasing acceptance of the paste-backfill method as an economic alternative to hydraulic slurry and rock backfill methods. The paste-backfill method, unlike hydraulic slurry placement, can use mill tailings without fine particle removal, maximizing the amount of tailings that can be used regardless of the size distribution.

Economic and environmental issues have also increased the acceptance of alternative surface storage methods for mine waste. Dry (filter cake) and semi-dry (thickened) tailings storage and co-disposal with mine-waste rock are gaining prominence as safe and economical waste disposal methods. The increasing economic importance attached to waste disposal is slowly transforming the focus of the mining industry as it evaluates future ventures. Historically, this focus has been towards ore production, but economic and environmental pressures are changing the focus towards long-term waste management.

69.2 TERMINOLOGY DEFINITIONS

Before proceeding with a detailed discussion about mine backfill, it is important to first define the terminology typically used by the mining industry.

Porosity	Volume voids/(volume voids + volume solids)
Wt% solids	Wt. solids/(wt. solids + wt. water) = pulp density
Wt% moisture	Wt. water/(wt. water + wt. solids)
Wt% binder	Wt. binder/(wt. binder + dry wt. backfill)
Wt% passing	Wt. passing given size/total wt. of dry sample
Slump	Slump is a measure of paste consistency. It is the number of inches a paste will slump under its own weight using the standard North American 12-inch-high concrete industry cone slump test (according to CSA standard A23.2-5C or ASTM standard C143-90A).

69.3 HISTORY OF UNDERGROUND MINE BACKFILL

This history of backfill describes the developments as they occurred in North America. While many of the innovations are described from a North American perspective, there is no doubt that similar developments occurred in other hard-rock mining communities throughout the world (Australia, Europe, etc).

In early mining years, typical mining methods required no backfill. The most common methods used in subvertical

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orebodies were open stoping and shrinkage stoping. In sub-horizontal hard-rock mines, the room-and-pillar method was the most common, followed by square-set stoping. In coal mines, seams were essentially self-filling, often resulting in surface subsidence. In the case of open stoping of subvertical orebodies, stopes were also often self-filling, with the wall rocks collapsing into the stope while the ore was drawn out below the barren material. When the waste rock began to appear in the material drawn out of the stope, this section would be closed down and mining would progress along strike and downwards to the next horizon. This mining method often resulted in noticeable subsidence on the surface.

In the case of shrinkage, stoping the wall rocks was generally stronger, and the stopes would remain open after the ore was removed. Once the stope was empty, mining would progress along strike and down to the next horizon, leaving a small pillar between the new stope and the empty one. As the mines became deeper and more extensive along strike, unexpected collapses became increasingly common, and the only solution was to abandon the area surrounding the caving stope and begin mining elsewhere. As the caving areas became larger and more frequent, it became obvious that it would be more cost-effective to fill the shrinkage stopes with waste material of some kind rather than abandon valuable ore resources.

A square-set stoping method was commonly used in shallow-dipping orebodies with wider, more extensive areas of roof or back requiring support. In this method, the roof or back of the stope was supported by timber-sets or timber cribs. Unconsolidated material was introduced into the stopes to confine the timbering and provide a floor for the men to work from. While the timbering provided some support to the roof of the stope, it did not always prevent it from collapsing. However, because most of the stope was filled, the collapse was limited; and it was possible to continue mining, either right on top of the collapsed area or very close to it.

Initially, unconsolidated material such as development-waste rock or surface sand and gravel was supplied through raises and mine cars to the stopes. It was fed into the stopes through raises and then was moved into place manually. This backfill method greatly increased labor requirements, backfill cycle time, and ore recovery

Cost savings and increased revenues resulted in advances in backfill methods. Because cycle time and labor are the main cost factors for backfill, reducing these factors resulted in more advanced systems. Similarly, recovery of ore resulted in backfill advances through increased revenues.

69.3.1 Development of Modern Backfill Systems

One of the first innovations in consolidated backfill technology took place in Canada in 1933. At Noranda's Horne Mine, granulated furnace slag was mixed with pyrrhotite tailings to form a backfill material very similar to today's cemented tailings. The oxidation of the pyrrhotite consolidated the tailings, and it was found that drifts could be driven through the material without support, and stopes could be mined up against it with very little dilution. However, this kind of mixture is very sensitive to the material used, and many other mines were unable to repeat Noranda's success.

The next major step forward in underground backfill took place in the late 1940s when a hydraulic slurry system was introduced into North American hard-rock mines. In this case, the system supplied classified tailings (fine size fraction removed) to square-set stoping areas. Because the volume of fill height was small, the risk of the fill becoming mobilized through liquefaction was minimal, although the rate of drainage was slow by today's standards. However, even allowing for the slow rate of drainage through material with a relatively high fines content, the speed at which stopes could be filled was much greater than the rate of supplying rock fill. Another advantage was that the fill could be directed to different areas of the stope relatively easily by setting up a series of pipes to direct the flow of material.

Mining up against this unconsolidated material remained a problem and became particularly difficult when mining the pillars between old filled stopes. With the introduction of rock bolts, cut-and-fill mining methods largely replaced the square-set stoping method. Stopes became much more open and allowed larger equipment (such as slushers) to be introduced. However, timber was still used to form gob fences or slats on the walls of the stope to restrain the fill when mining the adjacent pillar stopes. In the early 1960s, mines began using an undercut-and-fill method to recover longitudinal stopes, although working under the unconsolidated backfill material required a great deal of timbering.

Around the same time, several mines began tests to demonstrate that a small amount of cement could improve the performance of backfill, and that tailings should be deslimed to achieve a minimum percolation rate of 10 cm/hr. Thereafter, cemented tailings with about 3%–4% cement were used to consolidate fill. In cut-and-fill stopes, a 10% cement layer was in the mucking floor to improve the recovery of the fine (often high-grade) ore material, which became the standard in the cut-and-fill stoping areas. A similar high-cement-content layer was used in underhand cut-and-fill stopes to provide a more stable roof and reduce the amount of overhead timber support. Even very low cement content eliminated the need for gob fences to control fill walls and made it possible for classified tailings fill walls to be self-supporting up to heights of 160–200 ft (50–60 m).

The next major innovation was mechanizing cut-and-fill with jumbo drills and scooptrams, which began to replace jackleg and slusher techniques from the late 1960s. This did not affect backfill, however, as it was essential to have a good mucking floor to support the heavy equipment.

In the 1970s, bulk mining techniques were being introduced. These techniques took advantage of the ability of layered fill to be self-supporting to a height of 200 ft (60 m). This allowed the mining of secondary pillars between primary stopes with little dilution. Since then, primary and secondary mining with bulk techniques has largely replaced selective methods such as cut-and-fill and undercut-and-fill in sub-vertical base-metal mining operations.

The only significant changes to the backfill system in these bulk operations was in the design of stable freestanding backfill heights and in the operational control over the filling process itself. Operational controls include conservative bulkhead designs, drainage system designs, and stope sequencing to allow for percolation and backfill set times. High-grade, shallow-dipping base-metal mines continue to use selective methods profitably, using highly mechanized operations, which take advantage of the greater flexibility and selectivity that these methods offer.

The bulk-mining operations also allowed for supplying various forms of rock fill in combination with cemented tailings and cement slurries without the need to physically rehandle the rock material in the stope. In the 1960s, this technique was introduced at several mines where rock fill was supplied into the stope on top of the ore. As the ore was extracted, the stope filled up with waste. Once the stope was completely filled with rock fill, cement slurry was added to produce a cemented rock fill that was self-supporting and did not result in significant dilution. Experience indicated that the final product performed better if the slurry was added at the same time as the rock fill. These findings led to the development of the cemented rock-fill methods commonly used in many mines today (Kidd Creek in Canada and Mt. Isa in Australia).

The next innovation was replacing the cement binder in cemented tailings fill with cheaper alternatives. The initial trials involved replacing some of the cement with finely ground iron furnace slag and fly ash, as well as a few others. The slag was cementitious while the fly ash had a pozzolanic effect, but both resulted in low early strength with a higher strength over the long term. Both types of material became common components of cemented tailings slurry and rock-fill systems in moderate-depth mines.

In the early 1980s, the development of paste backfill began with the first active system being commissioned in Germany at Preiessage's Bad Grund Mine. Paste backfill allows full plant tailings with a high solids content to be transported at a paste consistency (7 in. [180 mm] to 10 in. [250 mm] slump) to underground workings. Paste backfill is transported through a borehole/pipeline underground distribution system similar to what is used to transport hydraulic classified tailings backfill. The higher viscosity paste material, as compared to hydraulic backfill, produces a greater resistance to flow and consequently higher pipeline pressures. However, paste backfill has lower cement consumption because it eliminates cement segregation upon placement and because it has a lower water:cement ratio.

While paste backfill requires less cement to produce a given backfill strength, it cannot be placed underground in an uncemented form like classified tailings or rock fill. The high fine-material content (15 wt% passing 800 mesh or 20 microns) required to produce a colloidal water retention necessary for paste transport also makes uncemented paste backfill susceptible to liquefaction. Even with the need for low cement contents in secondary and tertiary stopes, paste backfill generally produced a 30% to 60% reduction in cement consumption over alternative backfill methods.

Developing paste backfill also allowed mines with high fines-content tailings to use their tailings as the primary or sole backfill material. This often resulted in a significant reduction in the use of costly alternative materials (alluvial sand or rock) as an underground fill.

The paste backfill method gained increasing acceptance in the 1990s as a cost-effective alternative backfill method to hydraulic slurry and rock fill and as a method of maximizing mine-waste placement underground. At the present time, there are 23 operating paste backfill plants throughout the world, and several others are under construction. Paste backfill has allowed many operations with the appropriate conditions to reduce fill cost and maximize tailings disposal underground.

The most important innovations in underground mine backfill are listed below:

Prior to 1930s	– Unconsolidated rock fill		
1930s	– First consolidated backfill		
1940s	– Hydraulic slurry backfill systems		
1950s	– Addition of cement to hydraulic slurry backfill		
1960s	– Addition of cement to rock fill		
1970s	– Bulk stoping	– free standing height	
		– bulkhead design	
		– saturation control	
		– alternative binders	– iron blast furnace slag
			– fly ash
1980s	Research and development of paste backfill		
1990s	Implementation of paste backfill		

69.4 UNDERGROUND MINE BACKFILL METHODS

As previously discussed, not all mines or mining methods require backfill. Selecting a mining method for a given mine is influenced by the geometry and grade of the orebody and the stability and properties of the rock mass. The potential requirement of backfill is inherent in any mining method. The type of backfill used by an operation will be dependent on several factors:

- The configuration of the mining process
- The stope sequences and excavation sizes determined by the mining method
- The depth and the orientation of the orebody
- The materials available to use as a backfill, focusing on the mine waste management requirements over the life of the orebody.

TABLE 69.1 Rock fill versus slurry fill versus paste fill

Properties	Rock fill	Slurry fill	Paste fill
Placement State	Dry	60–73 wt% solids	65–85 wt% solids
Transport System	Raise, mobile equip., separate cement system	Borehole/pipeline via gravity	Borehole/pipeline via gravity
Cemented vs. Uncemented	Cemented or uncemented	Cemented or uncemented	Cemented only
Water:Cement (w:c) Ratio	Low w:c ratio, high binder strength	High w:c ratio, very low binder strength	Low to high w:c ratio, low to high binder strength
Placement Rate	100 to 400 tons/ hour	100 to 200 tons/ hour	50 to 200 tons/hour
Segregation	Stockpile segregation, reduced strength and stiffness	Slurry settlement Segregation, low strengths	No segregation
Stiffness	High stiffness if correctly placed	Low stiffness	Low or high stiffness
Tight Filling	Hard to tight fill	Cannot tight fill	Easy to tight fill

The three most important types of modern backfill used in hard-rock mining are rock fill, slurry fill, and paste fill. The choice between these three types is site specific and will depend on the particular requirements of each mining operation. The three types of backfill have different characteristics, which are presented in Table 69.1.

The different backfill systems have different capital and operating costs attached to them. However, these costs must be considered in the context of the cost structure of the particular mining operation. The different characteristics of the three types of fill can be described as advantages and disadvantages in terms of the configuration of the particular mining operation. It is these advantages and disadvantages, along with the mine's overall waste management requirements, that determine the most suitable type of backfill for that mine, rather than the generic operating cost of the system.

69.4.1 Hydraulic Slurry Backfill

The hydraulic-slurry-backfill placement method was developed in the 1940s and has evolved over time to be the most widely used backfill method in the mining industry. It consists of mixing an appropriate-sized granular material with water on the surface to produce a slurry that can be transported underground through a borehole/pipeline distribution system.

The size distribution of hydraulic-slurry placed backfill is governed by two parameters. First is the maximum size and volume of coarse particles that can be transported above the critical flow velocity in a given distribution system; and second, the maximum size and volume of fine particles that will allow an adequate water percolation or drainage rate to maintain the scheduled mining cycle (Thomas, Nantel, and Notley 1978).

Figure 69.1 shows the wt% finer size distribution range of granular materials that are suggested for use with a hydraulic-slurry backfill placement method. Particles above the mesh size

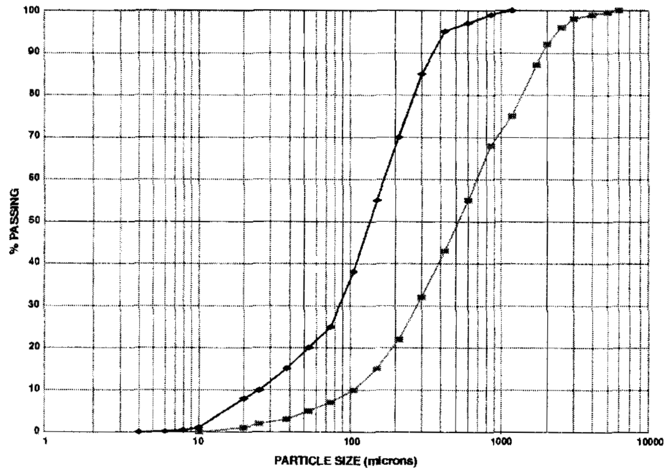


FIGURE 69.1 Size distribution range for hydraulic-slurry backfill materials

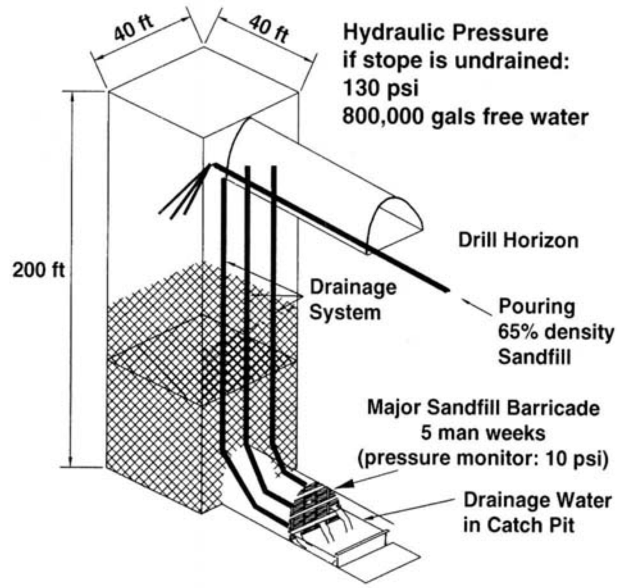


FIGURE 69.2 Hydraulic slurry backfill, auxiliary drainage

are difficult to keep in suspension in slurry, even when transported above the normal critical flow velocity. The coarse particles also increase the dynamic pipe wear exponentially. Decades of operating experience in mines worldwide have led to this recommended restriction on coarse particle size.

The fine particle fraction, as indicated earlier, dictates the rate at which water will percolate through the backfill mass allowing the removal of the excess slurry water that was used to carry the backfill solids to the stope. A binder-free backfill percolation rate of no less than 10 in./hr (2.5 cm/hr) is recommended to allow the water to drain through the fill mass at a sufficient rate to maintain production in most mines. Lower percolation rates will result in water pooling on top of the fill for several shifts after placement has been suspended.

It is common practice to suspend slurry backfill placement in a stope if the water level on top of the fill is greater than 4 ft (1.3 m).

Greater water levels will segregate the granular backfill as gravity settles through the pooled water to the solids/liquid contact in the stope. This is extremely detrimental if a binder has been added to the slurry backfill because bands of concentrated fine binder particles will be formed throughout the fill mass, with other sections being left with little binder content and little strength.

A hydraulic head of pooled water can cause large sections of the fill mass to be totally saturated as the wave of water slowly flows through the granular mass to a drainage location below. This condition could cause an unconsolidated (no binder) fill mass, especially when the fine content is high, to liquefy under dynamic loading from blasting or rock-burst vibrations. A consolidated fill mass, because of the increased cohesion that results from adding the binder, will not liquefy even if it is saturated with water.

Most hydraulic-slurry placed backfill materials (no binder) are sized to have no greater than 8 wt% passing the 20-micron (800-mesh) size range. Higher fine-material content generally reduces the water percolation rate below the accepted 4 in./hr (10 cm/hr). If the fine-material content is kept below this level–allowing a percolation rate or higher to be maintained–placing the backfill into underground stopes without adding binder (unconsolidated) is acceptable. Laboratory testing to confirm this percolation prior to production use of the backfill material is essential.

Unconsolidated hydraulic-slurry backfill is placed in stopes that will not be mined next to, or under. Some mines do use unconsolidated slurry backfill as a mucking floor in low-grade stopes where broken ore is left on the backfill floor to prevent fill dilution during ore removal.

In many mines, especially when a binder is added to the slurry backfill, auxiliary drainage facilities (perforated piping) are installed in the stopes to accelerate the removal of free water (Figure 69.2). While these systems greatly increase water removal rates from the stopes, care must be taken to ensure that excessive amounts of fine solids from the backfill are not removed with the water. This is extremely important when a binder has been added to the backfill, which generally consists primarily of fine particles (>20 microns). Most mines cover the drainage pipes with a fine mesh sock to prevent the fines from entering the drainage system, allowing only water to be removed from the stope.

The underground borehole/pipeline distribution system of a hydraulic-slurry backfill is generally designed to allow a transport flow velocity above 5.0 ft/s (1.5 m/s) or higher to be maintained. This is typically the critical flow velocity between laminar and turbulent flow transport for most granular material suitable for hydraulic-slurry-backfill placement. Below this flow velocity, solids settle out of slurry suspension and eventually plug the borehole/pipeline transport system.

Transporting hydraulic-slurry backfills typically uses the hydraulic head generated by gravity to provide the energy required in the given diameter distribution system to move the slurry above the critical flow velocity. In some shallow layering or laterally extensive orebodies, the vertical head is insufficient to provide adequate energy. Centrifugal pumps (in the form of booster pumping stations) are generally used to add energy to the system on the surface or underground.

Designing all underground hydraulic-slurry transport systems must be conducted with a sound understanding of the specific slurry-transport properties. The friction-loss flow properties, potential transport-system wear problems, and energy versus borehole/pipeline diameter versus flow velocity balance must be defined to ensure trouble-free transport to all the work places throughout the mine. A design error can result in a hydraulic-slurry backfill system that is plagued with transport problems and can result in major production losses and eventual destabilization of the entire rock mass.

Mill Tailings as a Hydraulic Slurry Backfill. The majority of hydraulic-slurry backfill placed in underground mines throughout the world utilizes classified mill tailings as the fill material. All mill tailings have too high a fine-material content to satisfy the recommended percolation rate for hydraulic backfill. Consequently, the tailings are classified through cyclones to remove the fine fraction. In many cases, however, the tailings have such a high fine-material content that, after removing the fine fraction to produce a fill material with only 8 wt% of its particles finer than 20 microns, there is an insufficient volume of tailings to meet the underground demand for backfill. In these situations, a mine will implement an alternative backfill method or supplement the lack of available coarse mill tailings with an alluvial sand source or turn totally to alluvial sand as the hydraulic-slurry backfill material.

When mill tailings are used as the backfill, the material must be carefully analyzed for chemical and mineral composition. There are several minerals (zinc, lead, and some pyrites) that can affect the binder reaction, resulting in strength retardation, reduction, and long-term deterioration. Laboratory short- and long-term binder-strength testing is required before the tailings can be used underground as a production backfill.

Consideration must also be given to the health and safety of underground workers when a potential tailing material is analyzed for use as a backfill. Many tailings contain health-hazardous contaminants (i.e., cyanide, arsenic, etc.) from the milling process. There are health and safety standards in most countries for underground backfill, and for underground contaminants in general, that must be reviewed to determine if they will be exceeded by using a given tailings material as an underground backfill.

Another concern with pyritic materials is the exothermic potential of certain tailings. Some forms of pyrrhotite and pyrite can chemically react under the proper underground moisture and oxygen conditions, internally heating to temperatures that can ignite their sulfur content and produce a self-sustaining underground fire. The sulfide gas produced from such fires can be very toxic and hazardous to health in a confined underground environment.

Alluvial Sands as a Hydraulic Backfill. Many mines are remote from the milling operations. Others are associated with a milling process that produces high fines-content tailings. After cyclone classification, these tailings cannot produce a sufficient quantity of coarse tailings suitable for hydraulic-slurry backfill placement (only 8 wt% finer than 20 microns). Such mines supplement or replace the tailings with alluvial sand from a natural deposit accessible to the mine site.

Used for this purpose, alluvial sands (whether blended with mill tailings or used as a stand-alone hydraulic-slurry backfill) must still have a size distribution (Figure 69.1) suitable to allow trouble-free borehole/pipeline slurry transport and adequate in-place water percolation or drainage rates. When using alluvial sand for a slurry backfill deposit, those containing clays and/or micas should be avoided. Both of these minerals are flat and platy in particle shape, which tends to reduce percolation rates. Furthermore, because of their long surface area, such particles generally require higher binder contents to produce a given backfill strength.

There are other minerals that are found naturally in alluvial deposits that can also affect hydraulic-slurry backfill drainage and binder strength. Any materials being considered for use as a slurry backfill (whether it be tailings or an alluvial deposit) must undergo vigorous laboratory testing. These tests must define the material's transport, drainage, and binder-strength properties before it is used underground as a mine backfill.

69.4.2 Rock Fill

Rock fill has been used to provide underground support in the mining industry for more than 80 years. The main attraction of this material has always been its availability at mine sites. In the 1960s, cemented rock fill was introduced to the industry and has since had a great impact on many mining operations. Immediate benefits were realized through improvements to bulk-mining practices such as pillar recovery and void-filling operations. Cemented rock fill is now being used successfully in many mines worldwide.

Rock fill generally refers to waste rock used for underground filling. Rock fill may be used in the form of cemented (CRF) or uncemented (URF) material. To give the rock sufficient cohesion to be self-supporting, CRF is generally placed in an underground stope when mining will be conducted either next to or under the stope. In some high-grade orebodies, CRF will also be used in a stope floor to insure maximum ore recovery with minimum fill dilution during ore removal.

CRF strength is controlled by a number of factors including grading of fill particles, cement content, rock type, angularity of fill particles, time in place before exposure, placement techniques, segregation, and moisture content (i.e., excess water). The required fill strength is a function of the mining methods in use, the dimensions of the stopes, and the stope cycle times. CRF strength is mainly controlled by the amount of cement added to the rock fill. The strength, however, can be influenced by the self weight of the fill, the degree of arching between solid rock walls, blast damage during adjacent stope mining, abrasion or attrition of the rock during raise transport, and rock-mass ground movements.

The quantity of cement in a rock fill is the main strength modifier that can be easily controlled in the fill system. Typically, the cement content for rock fills lies in the range of 4% to 8% by weight. However, because cement is expensive, there are economic incentives to find acceptable substitute materials, such as slag and fly ash, which are used at a number of operations.

Swan (1985) has examined the effects of cement content and aggregate size using test results available from several mines (see Figure 69.3). He determined and proposed that, for any given backfill material, the 28-day laboratory unconfined compressive strength (σ_c) is related to cement content (C_v) by volume through the expression:

$$\sigma_c = \alpha\, C_v^{2.36}$$

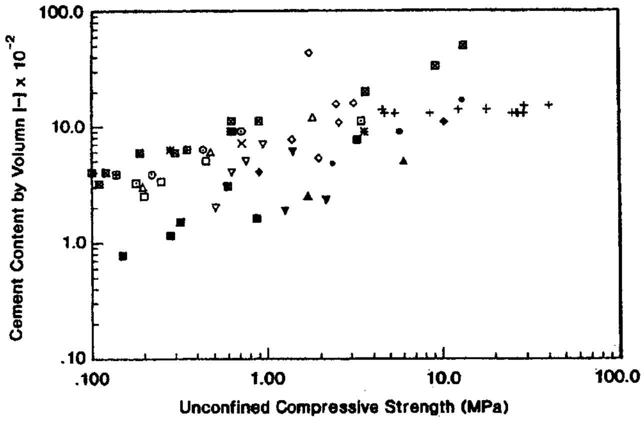
Typical strength and deformation characteristics determined from both small- and large-scale tests on cast samples of cemented rock fill are summarized below:

- Unconfined compressive strength: 1 to 11 MPa
- Unconfined deformation modulus: 300 to 1,000 MPa

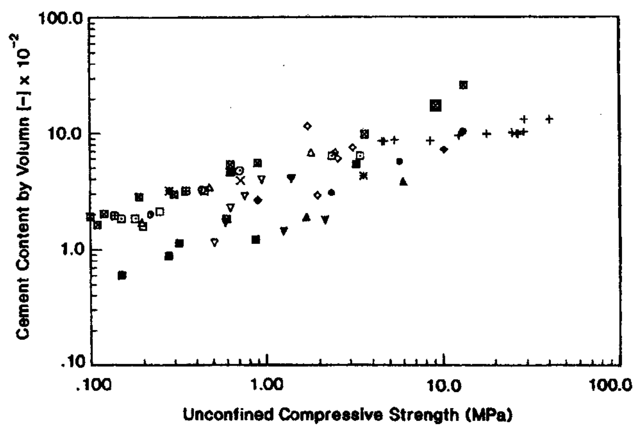
Experience at a number of mining operations has shown that the fill strength obtained in sample tests can be significantly greater than that of the placed CRF. This difference is due to a number of factors including scale effects, segregation, and quality control. A variety of studies (e.g., Barrett and Cowling 1983) suggest that the in-situ strength of a CRF may be about 50% of that measured in laboratory tests.

A number of other factors also influence the strength of CRF. Fill grading is an important factor in determining fill strength, as illustrated in Figure 69.4. An excessive fraction of coarse particles will result in a "loose" material that is susceptible to blast damage because it relies largely on point-to-point contact. It may also allow excess water to percolate through the fill mass, washing out cement. Excessive fines tends to consume cement due to the large surface area:weight ratio. Because of the high relative consumption rate, fills with high fine contents often have poor particle-to-particle bonding.

The grain-size distributions for cemented rock fills used at a number of mines throughout the world are shown in Figure 69.5. Generally, the upper boundary for particle size, as delivered to the top of the fill raise, is set within the 150-mm to 200-mm range.

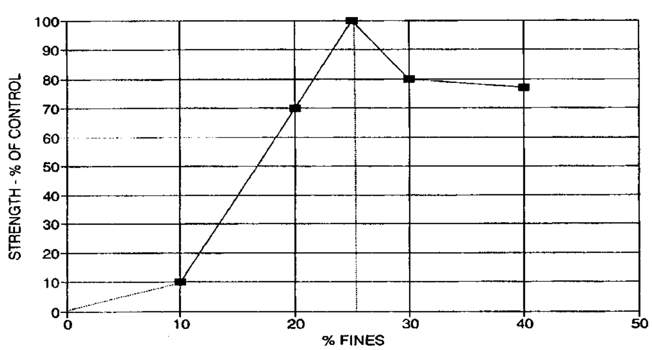


Unconfined compressive strength of selected mine backfills and concretes versus cement content by total weight per cent. Filled symbols designate cemented rockfill (see Table 69.2). Curing conditions: 28 days, 100% humidity.



Unconfined compressive strength of selected mine backfills and concretes versus total cement content by volume per cent (see Table 69.2 for symbol code).

FIGURE 69.3 Strength of cemented backfills



C.R.F strength as a function of the fines content. Control samples had a gradation of 75% coarse/25% fines. Minus 150 mm rockfill aggregate utilizing 450 mm diameter cylinders with 5% cement content by weight of aggregate.

NOTE: Fines defined as particles less than 10 mm size (from Quesnel and de Ruiter, 1989)

FIGURE 69.4 Effect of gradation on rockfill strength

A number of mines have observed rock-fill attrition when transporting the fill from the surface to the stope. This reduces the size of the fill particles and increases the fines content. In general, the maximum particle size is reduced by approximately 50% per 300 m of vertical distance traveled. The primary factors

TABLE 69.2 Key to mine backfills

Symbol	Mine, material*	Reference
+	Concrete	(1), (8)
X	Black Mount, CTSF	(5)
⊕	Warrego, CGF	(19)
⊠	Strathcona, CHF & CTSF	(12), (14)
⊞	Garson, CSF	(21)
⌶	Levack, CSF	(21)
⊙	INCO, CHF	(14)
◇	Lockerby, CHF	(14)
△	Falconbridge, CHF	(13)
▽	Mount Isa, CHF	(16), (18)
□	Selbaie, CHF	(7)
⊡	Kiena, CTSF	(22)
★	Coal Washery, CRF	(20)
▲	Kidd Creek, CRF	(11)
▼	Mount Isa, CRF	(16), (17)
■	Selbaie, CRF	(7)
●	Uludag, CRF	(6)
◆	Cavorrano, CRF	(15)

*NOTE: CTSF = cemented tailings–sand fill
CGF = cemented gravel fill
CSF = cemented sand fill
CHF = cemented hydraulic (tailings) fill
CRF = cemented rock (tailings secondary component) fill

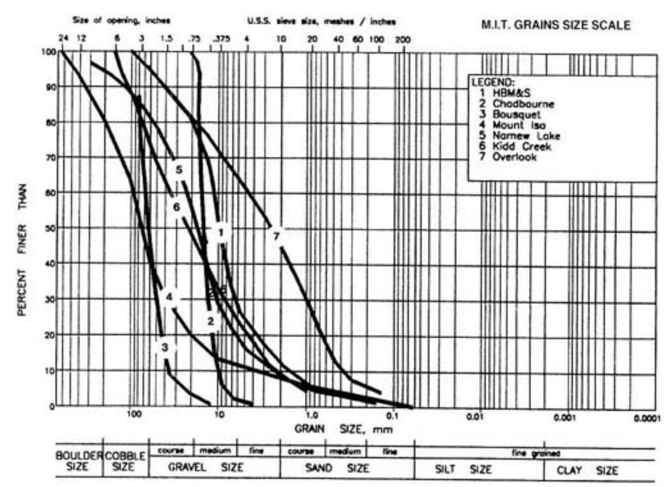


FIGURE 69.5 Grain size distribution of rockfills used at a number of mines

influencing the degree of attrition are raise inclination, distance of free fall, and rock type (Gignac 1978). As shown in Figure 69.6, similar results were found at the Kidd Creek Mine (Yu 1989).

The water content or amount of free water in the fill mass has a strong influence on strength. If excess water exists, it will tend to percolate through the fill mass upon placement, removing cement in suspension. Excess water may be present in the fill for a variety of reasons. Common reasons include the groundwater in the fill raise wetting the rock fill, water seeping into the stope, and poor quality control during mixing. Any of these reasons can cause excess water and adversely affect the fill properties.

Water quality of both the mix water and any potential inflow water must be checked for potential chemical reactions that may affect the cementing process.

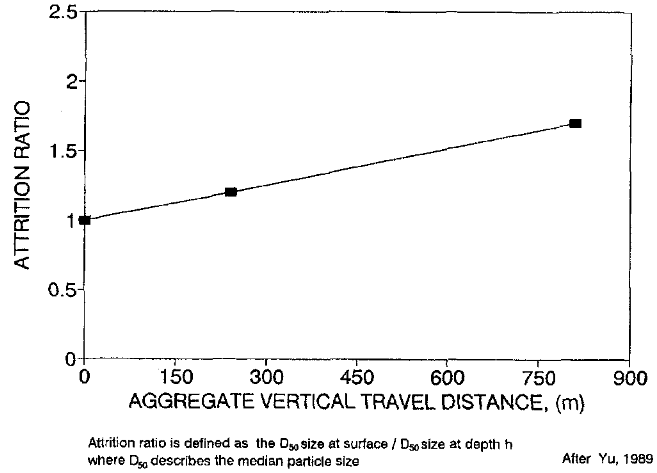


FIGURE 69.6 Attrition rate of aggregate in a rockfill raise

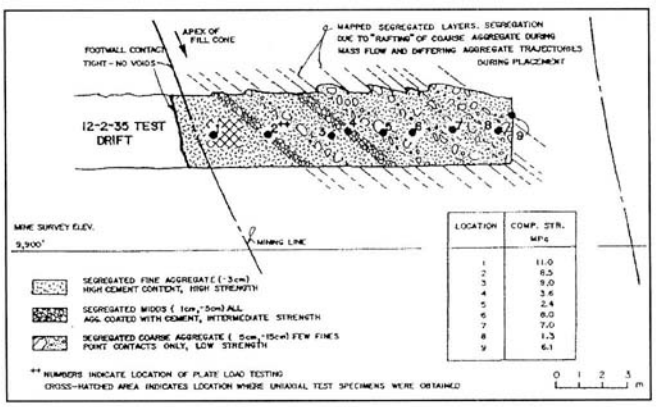


FIGURE 69.7 Rockfill segregation

The strength of the rock aggregate influences the basic fill strength. If it is too weak, the rock will break down during the mixing, transportation, or placement process and result in a poor-quality fill. Blasted rock tends to give stronger fills than those derived from natural gravel because of the effect of particle angularity.

Segregation cannot be avoided when fill is dumped into a stope. Larger particles generally travel farther, having greater momentum. The result is a gradual coarsening away from the dump point. As shown in Figure 69.7, studies at the Kidd Creek Mine (Yu and Counter 1983) found "that a zone of fine aggregate tends to occur near the impact area (i.e., below the dump point). Most of the cement is consumed in this zone leaving a low cement content rock fill at the perimeter of the fill cone." Adopting good placement techniques (multiple dump points) and controlling the water content can minimize segregation.

Rock aggregate is usually transferred underground dry. When truck transportation to the stope is used, mixing is generally carried out as the rock fill and cement slurry are discharged into the truck at the underground batch plant. When conveyors are used, mixing is generally carried out at the edge of the stope as the aggregate discharges from the conveyor.

Most of the mines using CRF send the cement binder underground in slurry form, via a borehole. The pulp density of this slurry typically ranges from 55% to 60%. The correct proportion of cement is obtained in the fill when the cement

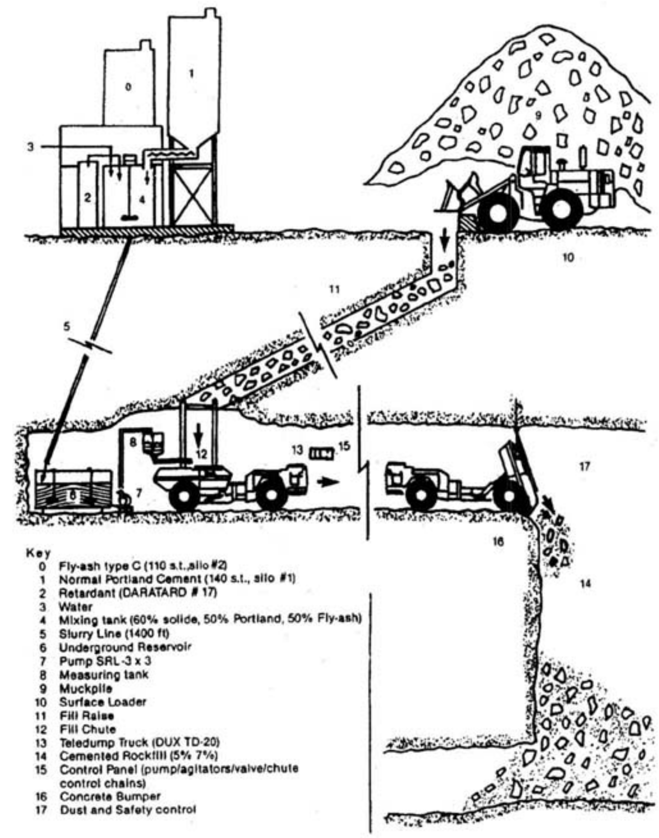


FIGURE 69.8 Gravity placement of rockfill

slurry is added to the rock fill. It is critical that no free water drainage from the fill occurs once it is placed in the stope.

Some surface cement batch plants use PLCs (programmable logic controllers) to allow the entire batching process to be controlled from underground by the operator at the slurry/fill mix point. Experience at a number of these mines indicates that there is a considerable learning curve during start-up. It is important that rigorous quality control be maintained during start-up to identify and resolve "bugs" within the system.

Placement methods can be divided into three groups:

- Spread placement
- Gravity placement
- Consolidation placement.

Spread placement refers to directly dumping the fill in the stope and spreading it using a bulldozer. This method usually results in an evenly compacted fill. This approach tends to be used in cut-and-fill, drift-and-fill, and low-lift bench mining.

Gravity placement involves dumping the fill (either from a truck or conveyor) down a raise or from an access drift and allowing the fill to fall over some vertical distance. This approach is the most common placement method, particularly for high-volume filling of large open stopes (Figure 69.8). Unfortunately, segregation invariably occurs. This segregation is controlled by the location of the dump point with respect to the stope geometry and the angle that the fill enters the stope. Ideally, these stopes would be leveled and topped-up by spread placement.

Consolidation placement is used in a limited number of cases and refers to either blowing the fill into position (pneumatic placement) or throwing it into position using belt-slinger methods. At the Meggen Mine in West Germany, for example, cemented rock

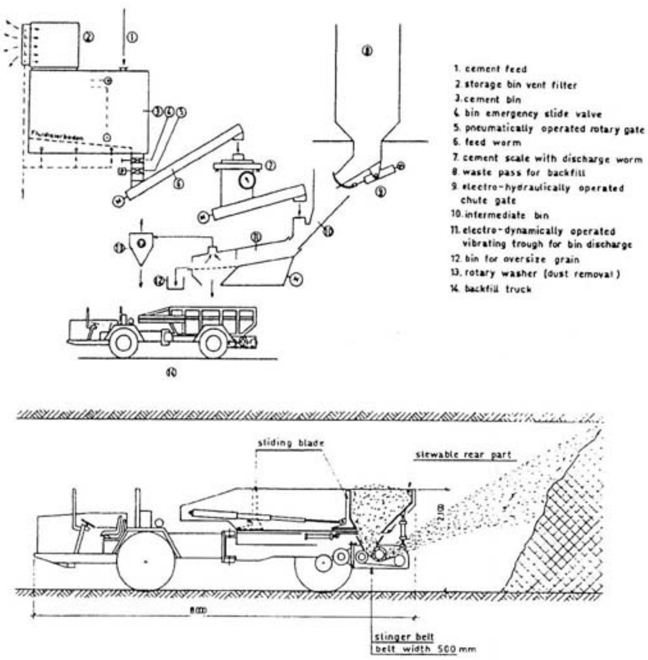


FIGURE 69.9 Rockfill placement using a slinger truck

TABLE 69.3 Design criteria for fill raises

Ratio of orepass dimension to particle	Relative frequency of dimension interlocking
D/d > 5	Very low; almost certain flow
5 > D/d > 3	Often, flow uncertain
D/d <3	Very high, almost certain no-flow

where: D = minimum span/diameter fill raise/borehole
d = particle dimension

fill is placed in room-and-pillar stopes by 6 and 10 m^3 trucks, using belt-slinger technology (see Figure 69.9). This produces a reasonably uniform, compact fill material.

The fill raise must be sized so that transferring the material from the surface to underground is not restricted. The size of the opening required is a function of the size of fill to be transferred and the volume of fill required. Rock-fill raises from surface to underground are typically 2 m and 3 m in size. Two types of blockage can occur:

- Hangups due to interlocking arches
- Hangups due to cohesive arches.

Interlocking arches form as a result of large-sized boulders becoming wedged together to form an obstruction. Such blockages will occur at changes of direction or reductions in cross-sectional area of raises or boreholes. Screening oversize boulders on the surface is the main method of minimizing this type of blockage. Hambley (1987) presented empirical design criteria for avoiding interlocking arches due to oversize as outlined in Table 69.3.

Cohesive arches form as a result of sticky fine particles adhering to each other. Whether a cohesive arch is formed depends upon the span of the raise. If the opening is sufficiently large, gravity forces will exceed the cohesive and frictional forces, and an arch will not form.

Hambley (1987) proposed that, to prevent cohesive arching, the minimum dimension of a raise be determined from:

$$D > 2(G/\delta)(1 + 1/r)(1 + \sin \phi)$$

where D = ore pass dimension
G = cohesion of fines (fines defined as particle size less than 0.25 mm)
δ = density of fines
r = length/width ratio of opening
ϕ = internal angle of friction of lines

An important aspect of the raise is water inflow. Above all else, the potential for blockages to occur in a raise increases as both the percentage of fines and the water content of the fill material increase. Furthermore, if the raise makes any water, this will increase the moisture content of the rock fill and result in a change to the water:cement ratio if not otherwise adjusted for. Excessive water in the raise can cause a poor quality CRF.

Water flow into the raise can also lead to an uncontrolled out-rush of rock if a hydraulic head of water is allowed to build above the rock and/or the rock becomes saturated and liquefies due to dynamic vibration from blasting or rock bursts. Such conditions are very dangerous and have resulted in fatalities in several mines.

Raises can be operated in a choked condition, an empty condition, or somewhere between these two limits. Choked raises reduce the wear and attrition of the fill materials, but the risk of blockages increases. Because of the need to maintain surge capacity and the difficulty of monitoring the fill level within a raise, some compromise is usually reached, with the fill level fluctuating between one-third and two-thirds full.

Quality control testing and monitoring is required to assess the performance of a rock fill system. Routine laboratory and in-situ material testing should become an integral part of any rock fill system design. Such testing is the only means of quantifying the actual "in-place" material properties and the impact of placement processes (drainage, segregation, curing, and dynamic loading).

69.4.3 Paste Backfill

A paste is a granular material mixed with sufficient water to fill the interstices between the particles so that the material behaves as a fluid. The granular material retains all the water between the particles because of its colloidal electrical particle charge that bonds the solid particles to the water molecules. In this state, paste can be transported through a pipeline but has no critical flow velocity (i.e., the velocity at which the solid and liquid components separate into two distinct phases). If more water than can be held between the particles is added to the paste, it becomes slurry, and the material does have a critical flow velocity. In this state, the material will flow through the pipeline but will separate out into two distinct phases if the pipeline velocity drops below the critical value.

In general, a granular material must have at least 15 wt% of its particles finer than 20 microns for the colloidal properties of the material to retain sufficient water to form a paste. Granular materials with less than 15 wt% of fine material will not possess the colloidal properties to form a paste and cannot be transported as such. The colloidal properties of a material are governed not only by the size of the particles, but also by their chemical content and mineralogical composition. This means that different materials will form a paste with different size distributions. Each granular material must be tested independently to determine its properties and its behavior as a paste. Both the production plant and the pipeline distribution system have to be designed according to these specific properties.

Paste backfills should never be placed underground without adding a binder. Uncemented paste backfill is very prone to liquefaction and will remain in a fluid state for days, weeks, and even years after being placed underground. If the ore being

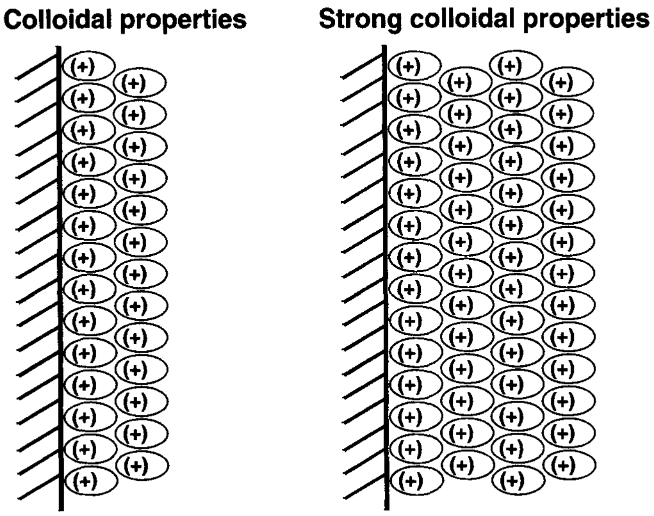


FIGURE 69.10 Paste backfill colloidal chemistry

mined will not economically support binder addition to the backfill, then paste backfill should not be used in that mine.

Full Plant Tailings Mix Design. There are three size-distribution categories for paste-backfill-mix design for most hard rock mine tailings throughout the world. These are coarse, medium, and fine tailings (Landriault 1995).

Coarse tailings contain between 15 wt% and 35 wt% minus 20 micron content. At a 7-inch slump, paste backfill will have a pulp density ranging from 78 wt% to 85 wt% solids depending on the specific gravity of the tailings material. For a given cement content, the high solids content provides a good water:cement ratio and results in backfill strengths at least double that of comparable hydraulic-slurry backfills.

Medium tailings contain between 35 wt% and 60 wt% minus 20 micron content. At a 7-inch slump, paste backfill will have 70 wt% to 78 wt% solids, again depending on the solids' specific gravity. These tailings generally produce a good paste fill, but typically have lower strength than the coarse tailings because of a higher water:cement ratio.

Fine tailings, which come from a mill process that grinds for recovery, contain 60 wt% to 90 wt% finer that 20 microns. High water retention is expected with these tailings, which usually produces a paste fill that is good for flow transport but poor for strength because of a very high water:cement ratio. Fine tailings at a 7-inch slump will have pulp densities between 55 wt% and 70 wt% solids.

The amount of water required to produce a given slump consistency in a paste backfill is dependent on the colloidal properties of the particles. These properties are a measure of the Zeta potential charge of the fine particle in the paste (Figure 69.10). The Zeta potential charge causes the water molecules to bond to the fine particles, giving the paste the ability to retain water within its particle matrix. The stronger the Zeta potential charge, the greater the water retention of a paste material at a given slump. As the fine material content increases, the amount of water retained by the paste at a given slump increases.

Soft-rock tailings generally contain much a higher water content (a 7- to 10-inch slump consistency) regardless of their size distribution. This is primarily because of their mineralogy, which often consists of high calcite, clay, or similar highly colloidal, high water-retention minerals. These tailings can range from 50 wt% to 30 wt% solids at a 7-inch slump consistency depending on the fines material content and mineral composition.

As with hydraulic-slurry backfill, the mill tailings used in paste backfill must be analyzed for mineral content (zinc, lead, and some pyrites) that can affect the binder reaction. Certain minerals can produce strength retardation, reduction, and long-term deterioration. Laboratory testing of the short- and long-term binder strength is required before a backfill mix consisting of such tailings is used underground as a paste backfill.

The health and safety of the underground workers must be considered when designing a paste backfill for a given mine. Tailings can contain health-hazardous contaminates (i.e., cyanide, arsenic, etc.) from the milling process. There are health and safety standards in most countries for underground backfill, and underground contaminates in general, that must be reviewed.

When pyritic tailings are used as a paste backfill, the exothermic properties must be investigated. As stated earlier, some forms of pyrrhotite and pyrite can chemically react under the proper underground moisture and oxygen conditions. Such reactions can cause internal heating to temperatures that will ignite the sulfur content of the material and result in a self-sustaining underground fire that produces toxic sulfide gas.

Blended Paste-Fill-Mix Design. Some mines have found that it was to their advantage to blend a coarse material with their mill tailings to produce a blended paste backfill (Landriault 1995). This advantage comes from the strength gains produced by the lower water:cement ratio created with blended paste backfills. By adding coarse material to the tailings, the particle size distribution of a paste fill is widened, which produces a lower material porosity. This results in less water being required to fill the lower amount of voids between the particles. This also allows a 7-inch slump to be obtained at a much higher solids content, producing a lower water:cement ratio and greater strength for a given cement content. Blended paste backfills are most attractive when the mine produces only fine tailings, which generally yield low cemented-paste backfill strengths.

Blended paste backfills are used in some mines because of their improved ground-support properties. The lower porosity created in blended paste fill results in a higher-modulus material. This improves the load response of the backfill as the result of rock-mass closure. Thus, less closure can occur before the backfill begins to carry some of the stress. This stress would normally be transferred through pillars and mining abutments.

In small mines with low production rates, blended paste fill allows them to produce backfill at a rate higher than their mill tailing production rate. This allows these mines to backfill stopes at a faster rate than if they were using a full-plant-tailings paste backfill. In mines with ground support problems, backfilling open stopes as quickly as possible is very important to keeping the rock mass stable.

If tailings are used as part of the blended paste backfill, the strength and health and safety concerns associated with the chemical/mineralogical composition of the underground backfill material (lead, zinc, cyanide, arsenic, exothermic properties) should still be determined.

69.4.4 Paste Backfill Rheology

Paste backfills produce a plug flow when transported through a borehole or pipeline (Figure 69.11). The annulus of fine particles that naturally forms in the plug flow condition acts as a lubrication layer along the pipe wall, reducing flow resistance and pipe wear. The coarse particles are naturally forced to the center of the pipeline and are transported in the fine-material carrier. This natural phenomenon allows very coarse material to be transported through a borehole/pipeline by a fluid material with paste flow properties.

Most paste materials can be categorized as Bingham fluids, but several have also demonstrated pseudoplastic flow properties (Landriault 1992). They all possess a yield stress that varies greatly in magnitude for different tailings paste materials.

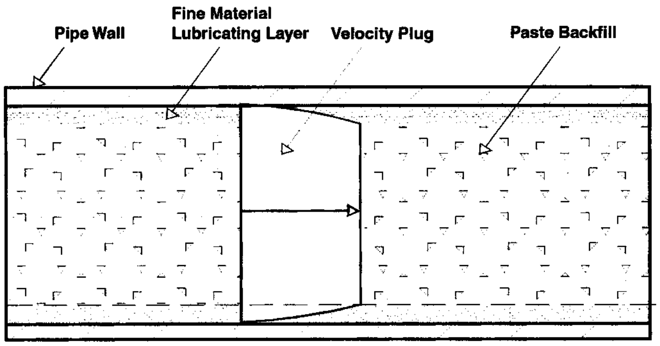


FIGURE 69.11 Velocity profile for plug flow

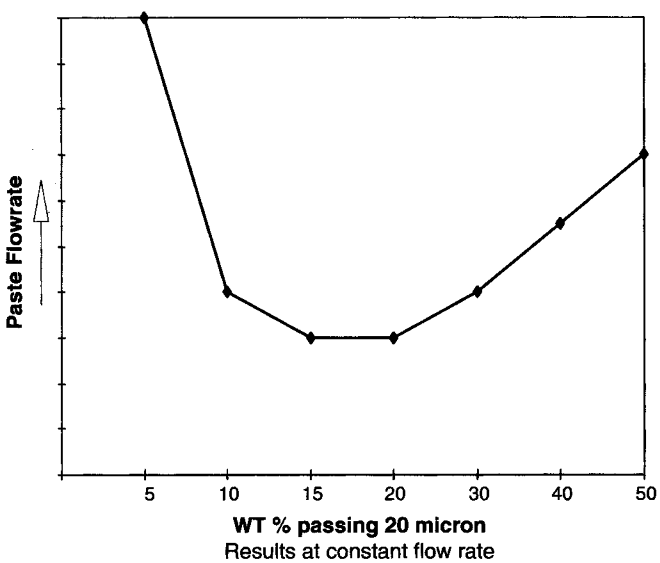


FIGURE 69.12 Paste backfill resistance versus 20

In general, a granular material must have at least 15 wt% finer than 20 microns to produce sufficient colloidal water retention to create paste-flow properties. Many paste materials demonstrate an increase in flow resistance with increasing 20-micron content (Figure 69.12). The resistance to flow, as with all fluids, increases with decreasing transport borehole/pipeline diameter.

Paste material properties, such as water retention and flow resistance, are not controlled by particle-size distribution alone. The chemical content and mineralogical composition of the paste material are as important to paste-material behavior as fine-material content and particle-size distribution.

69.5 MINING METHOD BACKFILL STRENGTH REQUIREMENTS

As previously described, backfill is used with several different mining methods. The backfill strength required underground is dependent on the construction and geotechnical needs for the specific mining method and rock mass in that given section of the mine. The following section outlines the basic approaches used to determine the backfill properties required to safely mine the orebody using different mining methods.

69.5.1 Bulk Mining (Long Hole, Shrinkage, Vertical Retreat)

Backfill in bulk-mining stopes is placed primarily as a ground support tool to minimize hanging-wall dilution, to reduce foot wall/hanging wall closure, and to stabilize the overall rock mass. Backfill also plays a major role as a construction material fulfilling three functions in most bulk stopes.

The first function is to act as a consolidated (cemented) backfill bulkhead in the mucking level draw point to contain the fill that will be placed above the draw-point horizon. With hydraulic-slurry and paste backfills, the backfill bulkhead can also eliminate the potential for liquefaction of freshly placed cemented fill.

If rock fill is used to backfill the bulk stope rather than hydraulic slurry or paste backfill, the need for a backfill bulkhead is not necessary because the rock fill is placed unsaturated and will not liquefy. Consolidated hydraulic slurry and paste backfills, even at low cement concentrations, typically produce sufficient strength after 24 hours of curing to contain the weight of fill above the draw point and to prevent liquefaction.

A typical backfill bulkhead is constructed by building a timber and/or waste-rock barricade in the draw point drifts accessing the bulk-mining stope. These barricades must be water retaining, and they usually incorporate an internal stope-drainage system to remove excess slurry water through the barricade and bulkhead out into the draw-point drift. If paste backfill is used, the barricades do not have to be water retaining, and auxiliary stope drainage is not required because paste backfill does not produce any noticeable water runoff. The high viscosity of the paste backfill will allow development rock to be used as the barricade without the potential of leakage from the stope.

The barricade is normally located back from the draw-point brow a distance that is twice the largest drift dimension. The purpose of the barricade is to act as a low-strength (40 kPa) construction form designed to contain the consolidated backfill until it sufficiently hardens to the strength required to withstand the potential head created by freshly placed backfill above it.

Backfill is placed into the stope from the drill drift on the level above or, if slurry or paste fill is used, through a borehole accessing the stope from some upper level. Slurry and paste fills are allowed to build up behind the barricade to a height of approximately 3 meters above the draw-point brow. The backfill bulkhead can normally be placed in one to three days, depending on stope size, and is typically allowed to cure to a uniaxial strength of 25 psi (170 kPa) before any subsequent backfill is placed in the stope. The time period required to reach this backfill strength depends on the type of backfill and its cement content.

The second function of backfill in bulk-mining stopes is as a freestanding wall that will be self supporting while the adjacent stope is mined. Prior to secondary stope removal, the primary stope is filled with cemented-paste backfill. After a period of time (at least 28 days), the secondary pillar will be taken. To ensure minimal dilution and to improve the regional ground support after the mining of that area is complete, the backfill must remain self-supporting when exposed.

The design requirements of backfill serving this function, regardless of type, are dependent on the dimensions (length, height, and depth) of the freestanding backfill wall that will be exposed during subsequent mining. The method used for calculating the necessary backfill properties for a competent freestanding backfill wall for the various bulk-mining stopes dimensions are based on work conducted by R. J. Mitchell of Queens University in Ontario, Canada.

Figure 69.13 shows the proposed stope size and the extent of the exposed face that must remain intact.

The equation used to determination the strength required is based on a wedge-type failure of the backfill. The parameters required are the stope dimensions, density of the fill, and the internal angle of friction. The internal angle of friction is generally determined through triaxial tests of consolidated backfill, but does not have a great influence on the final strength

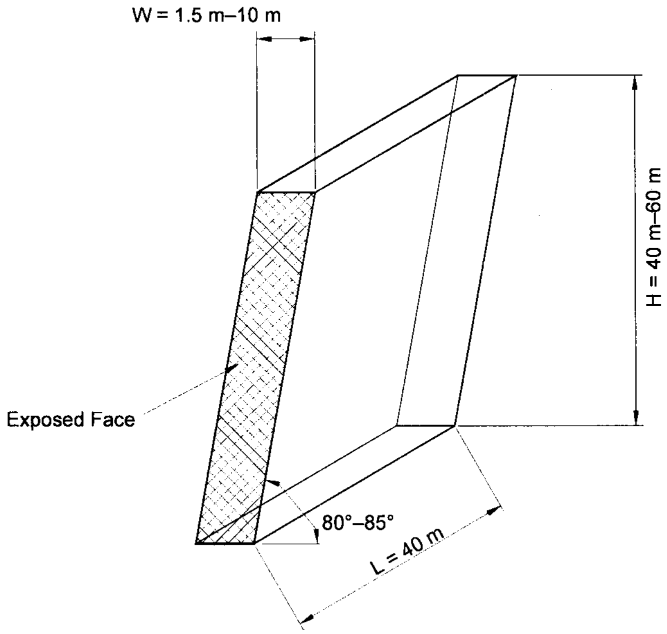


FIGURE 69.13 Proposed stope size and the area that must remain intact

given the above geometry. Most consolidated backfill has an angle of internal friction that ranges from 20–35 degrees.

Mitchell's method of determining the backfill properties required for a given bulk-mining situation have been used in the North American mining industry for more than 20 years and have proved to be very reliable in more than 100 mines.

The basic formula requires the height, length, and depth of the freestanding wall to be defined along with the in-place bulk density or weight of the backfill to be determined. The angle of internal friction of the backfill is also required for input into the equation. Based on hundreds of triaxial tests conducted on mine backfills throughout the world, a generic internal friction value of 30° is used since the values of the tests varied only between 28° and 30°.

The third function of backfill is as a cohesive soil in the secondary or tertiary stopes of a bulk-mining operation. The backfill placed in these peripheral stopes is primarily a ground-support tool. These stopes will not have the fill exposed, but the fill must still have sufficient cohesion to prevent liquefaction. Paste backfill is the only fill type that cannot be placed in the secondary or tertiary stopes in an uncemented form. The natural water retention properties of the high fine-material-content paste backfills (15 wt%, 20 micron content or greater) make them susceptible to liquefaction if placed under a dynamic load from blasting, rock burst, or a major ground fall. Hence, paste backfill placed in the peripheral stopes must have sufficient binder added to give the fill the necessary cohesion to eliminate any potential of liquefaction. The traditional paste backfill design strength to eliminate liquefaction potential in underground North American mines is 25 psi (170 kPa) after 28 days of curing.

When it is necessary for the efficient disposal of mine waste, uncemented rock will be dumped into the primary bulk-mining stopes as consolidated hydraulic slurry or paste fill is being placed. If conducted properly, this is a common and successful method of waste-rock disposal at most mines.

Waste rock should be dumped into the stope only while slurry or paste backfill is being poured. The rate of waste-rock placement should not exceed the rate of slurry/paste fill placement into the bulk stope. The rock should be dumped into a pool of fluid fill and should not be allowed to build up above the level of the paste in the stope against a stope wall that will later be exposed by subsequent mining. Slurry and paste backfills will not penetrate the interior of a dry pile of waste rock. Exposing the unconsolidated waste rock will result in wall failure and dilution of the ore in the adjacent stope.

69.5.2 Drift-and-Fill, Cut-and-Fill, and Uppers-Retreat Mining Methods

Mining using the drift-and-fill, cut-and-fill, or uppers-retreat method requires the backfill to act as the mucking floor for each sequential cut. With the bottom-up mining approach using these methods, stopes are often filled to within 0.3 to 0.5 meters from the back. This opening will be left to provide a free face for the next cut of ore to be blasted into.

Rock fill can generally be placed to within 1.5 meters from the back with a loader and then requires dozing and/or ramming to place it closer to the back of the drift. Producing a level mucking floor with rock fill would be difficult, if not impossible, using this placement method. Adding a moderate amount of cement would be required to produce a hard-rock-fill mucking floor of 150 psi (1 MPa). The uneven/non-smooth nature of the floor, however, would not be conducive to a high/undiluted recovery of high-grade ore fines that will be produced from blasting in higher-grade stopes. No backfill barricades are required for rock fill placement.

Hydraulic-slurry and paste backfills can be easily placed to within 0.5 meters from the drift back and, with the addition of a moderate amount of cement, can produce a hard mucking floor of 150 psi (1 MPa). The normal method of placing these types of backfills tight to the drift back utilizes pipelines that are secured approximately 0.5 meters from the back of the drift. A small raise can be created near the end of the drift to allow the pipeline to discharge at an elevation slightly higher than the normal drift back.

As paste backfill is discharged into the drift, it builds up with an angle of repose that can range from 1–10 degrees, which will be controlled by the slump of the paste fill being placed at any given time. Floors are poured at a high slump, resulting in floors from 1° to 3°. The paste fill will flow like lava from the discharge point back to the backfill barricade, which is normally placed about 60 meters apart to accommodate the placement angle of the paste-fill floor.

Hydraulic-slurry backfill typically produces a floor angle of 1–2 degrees, but because of the excess slurry-water removal from the stopes, substantial segregation of the particle sizes occurs. The result is a high fine-material and binder content near the drain locations and a variation in the backfill floor throughout the stope.

These mining methods are also used where the backfill is to be placed 2.4 meters from the back. The distance from the backfill to the stope back must be sufficient to allow the up-holes for the subsequent round to be drill, loaded, and blasted. As with the drift-and-fill methods, the same basic backfill mucking floor requirements (150 psi) are necessary.

69.5.3 Mining Under Backfill (Undercut-and-Fill, Sill-Pillar Recovery)

To facilitate production requirements, mining generally progresses on several different horizons in the mine. This results in mining that progresses from lower stopes up into areas that have already been mined. This will create a situation where sill-pillar ore will have to be removed from under the previously backfilled stopes, creating the need for backfill sill pillars.

This will create a similar situation that exists with undercut-and-fill mining, where men and/or equipment are exposed under a backfill head cover. Undercut-and-fill mining traditionally uses timber post and lagging support to contain the backfill but, with the introduction of paste backfill, some mines are using unsupported backfill with this mining method.

To safely recover the ore and to minimize dilution, the backfill above the ore being mined must be strong enough to be self-supporting over the span where it will be exposed. To determine the required strength, a safety factor of 1.2 is generally used if only equipment (remote mucking) is exposed under the backfill; a safety factor of 2.0 is used if both men and equipment will be working below the fill.

With these mining methods, the possible failure modes of the exposed fill are associated with arching, flexural or bending failure, block sliding due to side-shear failure, rotational failure at the hanging wall contact, and sloughing of wall rock. The stability of fill backs has been researched by Mitchell and Roettger (1989) and calibrated with laboratory testing. The following back-stability calculations are based on their work. The analytical design methods are based on typical sections. If there are significant variations in the geometry of the orebody, the calculations become very complex and should be implemented with extreme care and confirmed by numerical modeling.

69.5.4 Mining Through Backfill

Access into secondary mining areas often requires some development through in-situ backfill. Where this access has been anticipated, high-strength backfill can be placed to accommodate drift development through the fill and safe access to the ore with minimal mechanical support being required (typically friction bolts with welded wire mesh screen for a span of 3.0 meters or less). In areas where access has not been anticipated, and where the fill strengths are lower or drift spans of greater than 3.0 meters are excavated, a combination of shotcrete reinforced with friction bolts and welded wire-mesh screen will be used to support the backfill.

Typically, a 50-mm layer of shotcrete is applied to the backfill over a 2-meter advance. Wire-mesh screen is placed over the shotcrete and held in place with 0.3-meter friction bolts. A second 50-mm layer of shotcrete is then applied over the screen to provide added support. This approach to backfill support has been successfully used in North American mines for more than a decade.

69.6 BACKFILL AS GROUND SUPPORT

One of the most important functions of backfill is to provide support to the surrounding rock mass. It is important to establish at the outset what kind of support the backfill can and cannot be expected to provide.

Backfill can provide confinement to pillars and to stope walls and help prevent the progressive deterioration of exposed and unsupported rock surfaces. Backfill has a limited direct effect on the mining-induced stresses.

69.6.1 Effect on Stresses

Backfill cannot significantly alter the mining-induced stress conditions because of three factors. These factors include the contrast between the deformation modulus of the backfill and the excavated rock, the time lapse between the excavation of the rock and the placement of the backfill, and the tightness of the placement of the backfill.

Except in a few particular circumstances, the kinds of backfill that are placed in typical underground hard-rock mines are too weak to carry the kinds of load previously borne by the excavated material. Typical mine backfill, even cemented backfill, provides only passive support, so its capacity to resist wall convergence increases as it is compressed. In most cases, a great deal of the total convergence occurs before the backfill can be placed and consolidated. However, the compressibility of the fill also has an effect: cemented backfill provides more confinement than uncemented slurry fill which, in turn, provides more support than loose sand and gravel fill.

One case where backfill can carry significant stresses is in a narrow, steeply dipping orebody. The amount of convergence is dependent on the stress normal to the orebody, the properties of the wall rocks, and the area of exposed hanging wall and footwall. The resistance of the fill is determined by the total amount of void space in the fill, which determines how quickly the fill will begin to resist the convergence. In orebodies wider than 15 or 20 ft., the fill never gets to the point where it will offer significant resistance to convergence; therefore, the adjacent rib pillars and sill pillars become highly stressed. Only in the narrow orebodies can the fill offer resistance and reduce the stress concentrations in the adjacent pillars. The support pressure offered by backfill in such cases can be as high as 700–800 psi (5 MPa).

The deformation modulus of fill increases with the cement content of hydraulic-slurry backfill and is the most important factor in resisting wall convergence. Cemented hydraulic-slurry fill has a modulus of around 7,500 psi (50 MPa). Cemented rock fill has a higher stiffness or deformation modulus, ranging from 15,000–150,000 psi (100 to 1,000 MPa), depending on the density and cement content. Paste fill can achieve a modulus equal to the low end of cemented rock fill. A modulus of 150,000 psi (approximately 1 GPa) is sufficient to alter the energy balance and reduce stress concentrations in adjacent areas so the strain energy stored in the surrounding rock is reduced.

69.6.2 Effect of Confinement

Backfill enhances the behavior of the surrounding rock mass by providing a confining pressure to the exposed rock surfaces. The most obvious example of improving the capacity of rock by applying confinement is the case of post-pillar cut-and-fill mining. In this case, an array of pillars with a width:height ratio of about 1 are left in place to support the roof or back of the stope. The extraction ratio of post-pillar cut-and-fill stopes is about 85%–90%. Once the first cut of ore has been removed, the floor is filled with cemented tailings and the second cut extracted, leaving an identical array of pillars on top of those of the previous cut. With each successive cut, the width:height ratio decreases so that the pillars inevitably fail, usually on the second or third cut (width:height ratio = 2 or 3). However, the fill maintains the integrity of these failed pillars and, as the pillars dilate during the failure process, the surrounding fill begins to resist the dilation. The confining stress applied by the fill maintains the residual strength of the pillars as the width:height ratio becomes precariously small. Left unconfined, these pillars would undoubtedly collapse on the third or fourth cut, but in actual stoping situations, they continue to provide support for cuts as high as 14 or 15.

If the pillars are made of weak rock, a fill with little or no cement will increase both the peak and residual strength of the pillars. If the pillars are made of strong rock, such as a quartzite, a high-cement-content fill is required to increase the residual strength. It is possible to generate a residual strength of around 70% of peak with confinement provided by a 10% cement backfill. The same pillar surrounded by uncemented-tailings fill will have a residual strength of only 25% of peak, while an unconfined pillar will have almost no residual strength.

The strength of the confining material also has an effect on how violently the pillars fail. In highly stressed conditions, pillars confined with weak backfill will fail violently, while those surrounded by much stronger, high-cement-content backfill fail passively. In general, increasing the confining pressure on any failing rock mass allows it to behave in a less brittle, more ductile way, releasing less seismic energy during the failure process. The effect of high-strength backfill has important implications for mining in highly stressed conditions.

This confining effect is less obvious in bulk-stoping operations, but is equally effective. First, the backfill prevents the progressive collapse of the hanging wall. In plan, hanging wall

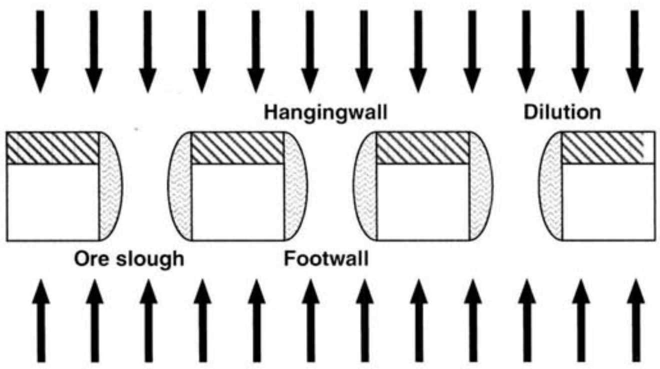


FIGURE 69.14 Plan of bulk stopes

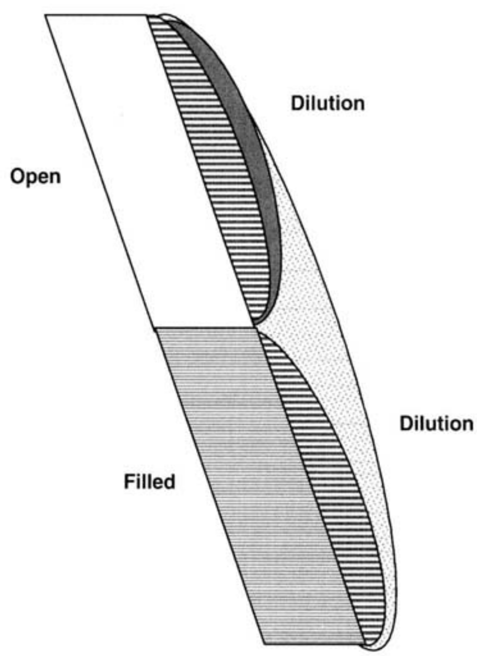


FIGURE 69.15 Cross section of bulk stopes

collapse changes the slenderness ratio of the ore pillars in a horizontal stress field and will contribute to slough ore off the pillar walls (Figure 69.14). In many cases, this is not considered a problem because the additional material is ore. However, this over-break causes the stope walls to become concave, and when these expanded stopes are filled and later mined, the fill walls are convex in shape, which results in fill dilution during mining of the pillars.

Second, this slough extends the failed zone into the hanging wall, and when this occurs at each of the primary stopes, the hanging wall of the secondary stopes become less stable, even before mining begins. The same is true for the hanging wall of the primary stopes on the next upper horizon. Hanging wall dilution in the lower stope effectively undercuts the hanging wall of the stope on the next horizon (Figure 69.15). Excessive dilution off the hanging wall of the primary stopes creates a patchwork of failed zones extending into the hanging wall and causes even greater dilution when these areas are mined.

There is little effective difference in the amount of confinement provided by the different types of backfill as long as they are cemented. Uncemented fill has a much lower stiffness and is relatively easily compressed and provides little support. Cemented rock fill can provide the greatest support, but it is also the most variable of the three cemented-backfill types because of the degree of percolation of cemented slurry through the rock fill and the potential for segregation during placement.

69.6.3 Backfill Rate of Delivery

A major difference between the types of backfill is the rate at which the backfill begins to provide support. There is virtually no difference between the rate of delivery of slurry fill and paste fill. The material handling setup required for cemented rock fill placement (haulage or conveying) is much more extensive than slurry or paste pipeline placement. If the material handling system is in place on the mining level, the placement rates can be twice as high as pipeline transport. However, if the placement system is not in place, a considerable delay could occur, resulting in more stope dilation or convergence occurring before the rock fills can provide any support at all.

In addition to the delivery rate, there is the time required for the backfill to become capable of carrying load (set-up time). Both slurry and paste fill can be supplied at about the same rate (100–200 ton/hour). However, the actual rate of delivery is often not the most critical factor. In bulk-mining operations, using slurry fill requires a major effort to construct water-retaining barricades, which contain the saturated material while it drains. The construction time and the relatively slow pouring rate in the early stages of backfilling are the most important factors. In the case of cemented rock fill, unless the entire backfill system is set up with conveyors or a large vehicle haulage system, the delayed and/or low rate of rock fill placement to the stopes will produce the same result. Paste fill requires minimal barricade construction to contain the material, and there is no water entrained in the system, except the amount required to hydrate the cement.

While the pouring rate of slurry and a paste fill is almost the same, the overall rate of delivery is much slower for the slurry fill. This is because most slurry-fill operations have to stop pouring until the saturation level of the slurry falls below the critical level as the material drains. Because there is virtually no free water in a paste-fill system, there is no need to delay the fill pours. Finally, the consolidation rate of a slurry fill is much slower than for paste fill and rock fill. Both fill types reach the required strengths days or weeks before that of slurry fill because of their higher water:cement ratios.

The rate of completely filling a stope has an effect on the stability of the rock mass surrounding it. During the backfill preparation period and the early stages of placement, the stope walls will be deteriorating; and the cumulative effect of this deterioration will result in even more dilution in later stopes. It is possible to design the stopes slightly more conservatively to ensure that the rock mass remains stable during filling, but more conservative designs are less cost effective than optimally designed stopes. In the case of slurry fill, the excess water lubricates the joint systems and further weakens the rock mass. In intensely jointed conditions, minor collapses during the fill cycle are common in cut-and-fill stopes. Even when the stope is filled with slurry fill, it takes some time before the fill gains sufficient strength to resist the dilation of stope walls or pillars.

69.6.4 Mining in High-Stress Conditions

One of the main reasons behind the development of paste fill was the expectation that because of its overall lower porosity it would reduce the release of seismic activity in deep, highly stressed mines. Unfortunately, it has now been established that, apart from narrow orebodies, no backfill material can provide sufficient resistance to wall convergence to prevent rock bursts. However, implementing high backfill-placement rates (i.e., paste or rock) in deep, highly stressed mining operations can

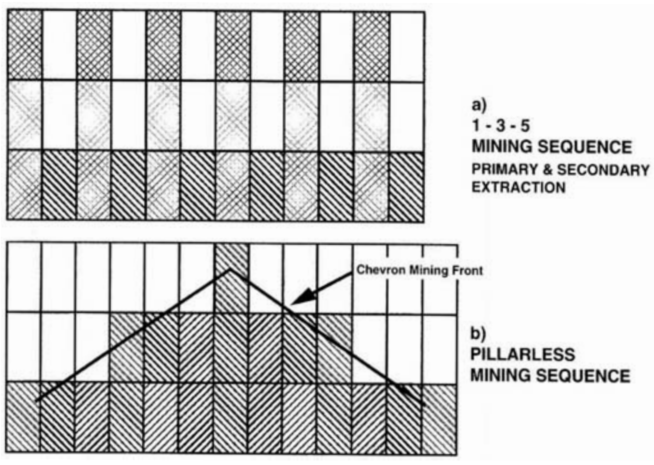


FIGURE 69.16 Alternative stope sequences

significantly improve the management of rock burst problems by allowing for a more appropriate mining sequence.

The most economic mining sequence for bulk stopes is the primary-secondary stope-pillar sequence, with equal-sized stopes and pillars (Figure 69.16). However, in highly stressed situations, it becomes very difficult to mine the secondary pillars because they have become so highly stressed that violent failure is inevitable as soon as production blasting begins. The situation is worse if the stopes and pillars are unequal in size. The best solution to this problem has been the introduction of the center-out mining sequence, where the next stope to be mined is immediately adjacent to the most recently filled stope. The disadvantage of this sequence is the small number of stopes that can be brought into production early in the schedule and the time delay in mining the next stope in the sequence. In a slurry-fill operation, this sequence is viable only if there are several blocks or areas of the mine available for production at any one time. A paste- or rock-fill system can allow for a much shorter cycle time and makes it easier to adopt a center-out mining strategy in highly stressed conditions.

Reducing the cycle time in a mining operation by placing backfill at a high rate has the potential to radically change the approach to all mining operations, not just those in highly stressed conditions. In the past, mining operations have attempted to achieve economies of scale by mining a few very large stopes at one time. As mines become deeper and extraction ratios increase, the stability of large stoping blocks and the surrounding rock mass decreases. Unfortunately, reliance on these stopes is relatively high, compared to the selective small-scale mining operations that were prevalent 20 years ago (drift-and-fill, cut-and-fill). A large stoping area can represent as much as 20% of the mine's production and any interruption to the flow of ore from these stopes has serious implications for the mine as a whole.

At the same time, the very large size of each individual stoping area generally allows less control over the production process rather than more. The cost of conservative designs for ground support and blasting tends to erode the economies of scale but leads to problems with stope stability and fragmentation. The variations in the mining process (i.e., dilution and secondary blasting) necessitate costly remedial activity and elaborate contingency plans to cover for production short falls. The economies of scale that were essential because of the long cycle time of an individual stope are now being eroded by the increasing cost of contingency planning and remedial activities. As the physical conditions in deeper, older mines deteriorate, these problems will worsen and reduce the cost effectiveness of this approach to bulk mining.

69.7 ACKNOWLEDGMENTS

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