Introduction to Open-Pit Mining

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Increasing mining costs, declining average ore grades, environmental considerations, and improved health and safety awareness are some of the main challenges facing the mining industry today (Caccetta and Hill 1999). These challenges emphasize the need for optimization of mining, especially when it concerns large-scale mining associated with open-pit operations.

The underlying message expressed in this chapter is that an open-pit mine is an increasingly complex and interdependent system that can only be optimized by careful coordination, management, and harmonization of its individual elements.

DEFINITION OF OPEN-PIT MINING

Open-pit mining can be defined as the process of excavating any near-surface ore deposit by means of an excavation or cut made at the surface, using one or more horizontal benches to extract the ore while dumping overburden and tailings at a dedicated disposal site outside the final pit boundary. Open-pit mining is used for the extraction of both metallic and nonmetallic ores; application of this mining method in coal is less common. Open-pit mining is considered different from quarrying in that it selectively extracts ore rather than an aggregate or a dimensional stone product. The main difference between strip mining—commonly used in the mining of shallow, bedded deposits—and open-pit mining lies in the overburden disposal. In strip mining, overburden is dumped directly onto mined-out panels rather than outside the final pit boundary, as is typical of open-pit mining.

Production tonnages for open pits range from fewer than 15,000 t (metric tons)/yr in small iron ore operations to more than 360 Mt/yr in large porphyry copper operations such as Escondida in Chile. As of 2008, there are approximately 2,500 industrial-scale open-pit metal mines in the world, which is approximately 52% of all industrial-scale mining operations. Iron ore (44%), copper (38%), and gold (15%) together account for 97% of the total open-pit excavation volume (Raw Materials Group 2008).

TYPICAL DEPOSITS

By definition, ore bodies mined through open-pit mining are located at or near the surface. Although the geometry of ore bodies varies from pit to pit, as a general rule it can be said that open-pit mining favors ore bodies that can be mined on a large scale (e.g., extensive ore bodies with low stripping ratios). Porphyry copper deposits such as Chuquicamata and Escondida in Chile, and Bingham Canyon in the United States, are prime examples of such large low-grade ore bodies. Other common ore-body shapes include stratabound and stratiform deposits such as Western Australia's iron ore deposits and the Zambian copper belt mineralization, diatremes typical of kimberlites (Jwaneng in Botswana) and carbonatites (Palabora in South Africa), and stockworks such as the Kalgoorlie (Western Australia) gold deposits.

OPEN-PIT GEOMETRY

The geometry, or layout, of an open-pit operation is discussed in this section. The main considerations are on those parts of the excavation that have to accommodate the main equipment and their operations, namely the benches, haul roads, and overburden disposal site. Two other subjects related to open-pit geometry—pit expansion and transition to underground mining—are also included in the discussion.

Benches

Benches are possibly the most distinguishing feature of an open pit. They are crucial in an operation as they accommodate the active blasting and excavation areas. Benches can be divided into working benches and inactive benches (Hustrulid and Kuchta 2006). Working benches are in the process of being excavated, whereas inactive benches are the remnants of working benches left in place to maintain pit-slope stability. Between main benches, catch benches are left in place to prevent cascading material from compromising safety in active areas of an operation. Figure 10.1-1 shows a simplified geometry of a typical open pit as well as the layout of some of the crucial elements in more detail.

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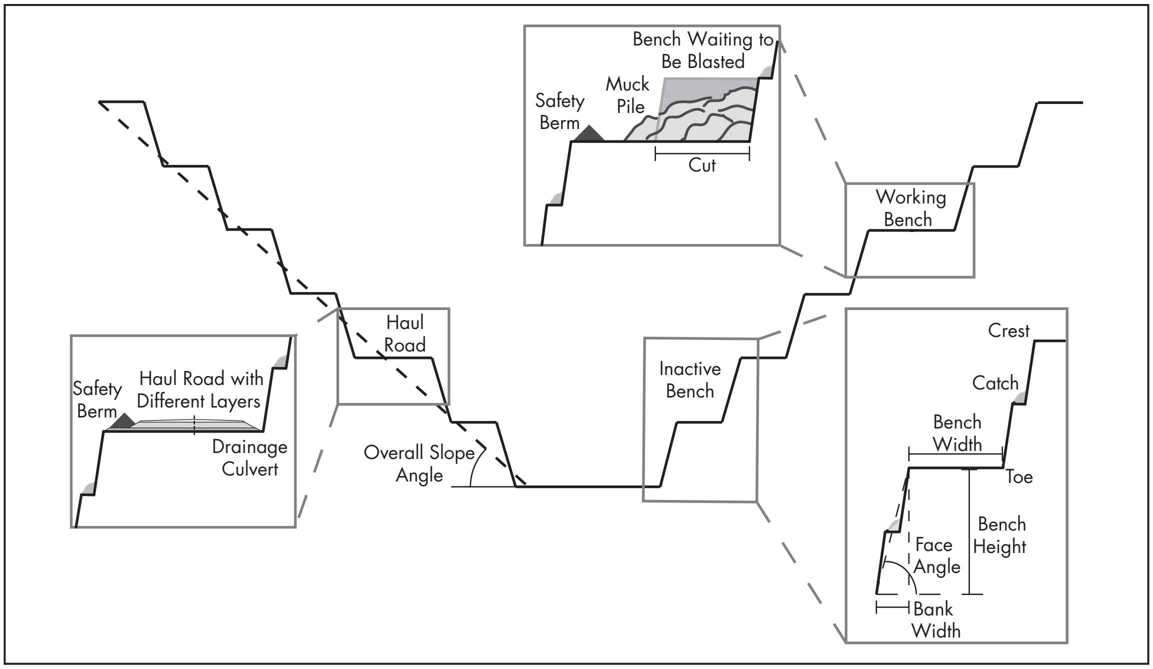


Figure 10.1-1 Typical open-pit geometry

Bench heights typically lie around 15 m. The bench width varies according to equipment size and the type of bench. Working benches should at least be wide enough to accommodate the turning radius of the largest haul truck plus the width of the safety berm (Caterpillar 2006). However, ideally, the bench should at least be wide enough to allow the largest haul trucks to clear the excavator under full acceleration. Depending on the chosen pushback geometry and size of the equipment, the width of a working bench can be anywhere from 30 m to several hundred meters. The width of catch benches is typically between 3 and 5 m but can vary with overall bench height. A small catch berm (~1 to 1.5 m) is usually included at the edge of the catch bench to improve its effectiveness at containing bench-scale rockfalls.

Constituting one of the busiest areas of an open pit, working benches have to accommodate large excavators and dump trucks as well as the muck pile formed after a blast. Therefore, maintaining the quality of a suitable working surface is vital to ensure acceptable safety and productivity levels at an active excavation. As discussed under “Haul Roads,” it is important to balance maintenance requirements against the duration and intensity of traffic. This is no less true for benches than it is for haul roads. Floor maintenance resources such as wearing course (also known as road base or road capping), labor, and road maintenance equipment should be allocated according to the added value they have in the operation as a whole. The importance of good floor maintenance on benches is emphasized in research by Ingle (1991), showing that up to 70% of tire damage may occur in active loading and dumping areas. For this purpose, it is important that a well-drained and smooth surface free of rocks is maintained. Dozers or front-end loaders can be used to aid the main excavator in maintaining good floor conditions. Furthermore, they can increase main excavator efficiency by reshaping the muck pile to increase bucket fill factors and possibly aid in the selective mining of ore and barren rock. For health and safety reasons, safety berms (also known as safety benches or windrows) are constructed along crests of benches in a similar manner to those found next to haul roads. The main goal of berms is to stop equipment from backing over the edge of a crest. Generally, a berm with a height equal to the axle height (e.g., at least half of the wheel height) of the largest truck entering an area is not only a safe design but commonly required by mine safety regulations.

Haul Roads

Haul roads constitute a key element of an open-pit mine, providing the main haulage route for ore and overburden from active excavation areas to the pit rim and beyond. Figure 10.1-1 shows the layout of a typical haul road. In light of a trend toward increased gross vehicle mass and haulage distance, detrimental effects of inadequate haul road design, management, and maintenance are becoming increasingly costly (Thompson and Visser 2006). Possible effects are

- Decreased truck and tire life,
- Loss of productivity,
- Poor ride quality, and
- Excessive fugitive dust generation.

All these factors can result in exacerbated vehicle and road maintenance and operating costs. Furthermore, statistics provided by the National Institute for Occupational Safety and Health (NIOSH) show that, in the United States, haul roads are responsible for 20% of lost-time injuries and 42% of fatalities in surface mines (Turin et al. 2001). Lastly, haul roads can significantly impact pit angles and stripping ratios depending on the adopted design and geometry. As such, sound haul road design and management can have a significant positive influence on the safety record, profitability, and environmental impact of a mine.

Thompson and Visser (2006) argue that optimal performance of a haul road network can only be achieved through an integrated approach incorporating (1) geometric, (2) structural, and (3) functional design as well as (4) the adoption of an optimal management and maintenance strategy. Geometric design covers the basic layout of haul roads based on input criteria such as truck types, traffic intensity, design life of the road, available construction materials, and costs. The next step, structural design, goes into more detail, determining factors such as required materials for road construction material based on the projected design life and traffic intensity. The goal here is to ensure the haul road can accommodate the imposed loads without excessive construction or maintenance costs. Functional design is mainly concerned with providing a safe, vehicle-friendly ride at the best economic performance. Wearing course material selection and performance is crucial during this stage as it is the controlling factor for rolling resistance, fugitive dust generation, ride quality, and surface deterioration rates. The last step, adoption of an optimal management and maintenance strategy, involves developing the most cost-effective approach to maintaining the functionality of the haul road. This subject is covered in more detail later in the "Maintenance and Management" subsection. For a more detailed discussion of design and layout of haul roads, readers are referred to Chapter 10.6 of this handbook.

General Design and Operational Aspects of Haul Roads

Depending on the location and use, haul roads are generally around 3–3.5 and 3.5–4 times wider than the largest truck size on two-way straights and in two-way bends, respectively. Effectively, this places the width of most two-way haul roads between 20 and 35 m, and up to 40 m on bends. For one-way haul roads, a width of 2–2.5 times that of the largest truck size is generally enough. Recommended grades lie between 1 and 8 and 1 and 10 (10%–12.5%), but higher grades are possible when trolley-assist haulage is used. It is important to keep the grade as constant as possible to make truck operation easier and more efficient. Where speeds exceed 15 km/h, corners can be superelevated, although superelevation should not exceed 1 and 10 or 10% (Caterpillar 2006). For better drainage on flat sections, a cross slope of 2% with loaded trucks on the upper part should be considered. On grades, minimum cross slope is required. For safety reasons, angles between roads on intersections should be 90° where possible. Lastly, on two-way sections of haul roads a center berm can be constructed. There is some debate about the effectiveness of such berms—some mines use them; others consider them to be center obstacles without any added value that can cause tire degradation.

Before a shift starts, operators have to perform a basic vehicle check, testing vital systems such as the brakes. Traffic rules on haul roads vary depending on the operation; for example, speed limits for haul roads range from 5 to 40 km/h. Large, loaded trucks in an uphill haul may not be able to achieve the maximum speed. For the rest of a mine site (i.e., workshops, stockyards, crushers, etc.), speed limits generally range from 10 to 20 km/h. In light of safety considerations, trucks traveling in the same direction are normally not allowed to pass, and in most cases passing is also prohibited for light vehicles. In most other aspects, traffic rules on-site closely resemble those of public roads.

Dust

Haul road dust can have a considerable environmental impact, increase maintenance and operations costs, and be a serious safety hazard both in the short term by reducing operator visibility and through long-term exposure, which may cause damage to the respiratory system. The main controlling factors in haul road dust are

- Wind speed at the road surface,
- Traffic volumes and speed on the haul road,
- Particle size distribution of the wearing course material,
- Construction characteristics of the wearing course material, and
- Meteorological conditions at the mine site (Thompson and Visser 2000).

Operator exposure can be decreased significantly by fitting cabs with filtration equipment, airtight seals, and air conditioning as well as increasing the following distance between trucks. Thompson and Visser (2000) state that the most common dust suppression measures include

- The application of a suitable wearing course material,
- Reduction of haulage speeds,
- Regular application of water or chemical suppressants, and
- Sound haul road maintenance.

The selection and application of a suitable wearing course, combined with regular application of water and chemical suppressants, are the most feasible and effective options. Applying a suitable wearing course should be the preferred control measure as it is a preventive rather than mitigating action and it has many beneficial effects in other areas of haul road management and maintenance. The feasibility of watering and chemical suppressants should be evaluated through a cost–benefit analysis. Watering often seems the cheapest alternative, although in light of its short effectiveness and, possibly, limited water supplies, other alternatives may be more feasible (Thompson and Visser 2000). Chemical suppressants are more effective at long-term dust suppression than water. However, as production haul roads are highly dynamic in nature, chemical suppressants generally do not last very long in these environments as continual watering and grading decreases their effectiveness. Furthermore, they do not mitigate effects of material spillages during haulage and are normally more costly. Therefore, continual watering and grading is in many cases the main type of dust suppression for production haul roads, although in some cases a combination of both systems might be the most viable alternative. For auxiliary roads that are more permanent in nature, dust suppressants are a more viable alternative to continual watering and grading.

In addition to the measures discussed previously, avoiding spillages also plays an important role in fugitive dust suppression and, possibly more importantly, prolonging tire life. Chemical suppressants are not effective at suppressing fugitive dust generated from spillages as they are not applied regularly (Thompson and Visser 2000). Watering haul roads will mitigate this problem but suffers from the problems discussed in the previous paragraph. Considering the drawbacks of watering and the effect spillages can have on tire life (as discussed in the "Tire Management" section later in this chapter), avoiding spillages through adequate load placement by the excavator should be preferred over more regular watering.

Maintenance and Management

Rolling resistance is the resistance to motion that a haul truck experiences because of friction. The main contributors are wheel load and road conditions and, to a lesser extent, tire flexing and internal friction. Minimum rolling resistances of

1.5% (radial and dual assemblies) to 2% (cross-ply or single-wheel assemblies) are quoted for rear-dump trucks. Estimates of rolling resistance related to tire penetration indicate an increase in rolling resistance of 0.6%/cm of tire penetration into the haul road (Thompson and Visser 2003). Thompson and Visser also report that similar resistances can arise from road surface deflection or flexing. As haulage is one of the main cost generators in an open-pit mine; rolling resistance reduction can lower capital and operating costs considerably.

Some of the main contributors to a high rolling resistance and bad haul road performance are inadequate wearing course construction and application as well as haul road defects such as potholing, rutting, loose material (dust and stones), corrugations (commonly referred to as washboards), surface cracking, and insufficient drainage. If these defects are left unattended, they may reduce haul road productivity, impede safe vehicle operations, and damage equipment. Five of the main routine maintenance activities for a haul road are

1. Dust suppression measures,
2. Routine surface maintenance,
3. Clearing material spillages,
4. Replacing the wearing course material, and
5. Drainage culvert and shoulder maintenance (Paige-Green and Heath 1999; Thompson and Visser 2003).

All these measures are aimed at maintaining or improving road quality and will mitigate most haul road defects. However, because of limited resources and complex maintenance requirements, a practical maintenance strategy is more effective at mitigation and prevention of these defects.

Semipermanent and temporary haul roads are commissioned and decommissioned during a mine's life. It is vital that the costs of constructing these haul roads are balanced against their design life (Thompson and Visser 2003). Under-expenditure of resources on permanent, high-volume haul roads or over-expenditure on short-term, low-volume haul roads can have serious detrimental effects related to premature failure, compromised health and safety, high maintenance costs for permanent haul roads, or an excessive drain on resources for haul roads with a short, active life span.

Wearing or surface course material plays a major role in the productivity and maintenance requirements of a haul road and the equipment using it. Therefore, the selection, application, and maintenance of wearing course materials are paramount to the good functional performance of a haul road during its operational life. To determine its influence on overall haul road functionality and road user costs, the performance of wearing course materials should be analyzed and benchmarked (Thompson and Visser 2006). Visual inspection of haul roads has traditionally been the main method of determining the maintenance needs for haul roads. However, recent advances in the use of high-precision Global Positioning Systems (GPSs), communications, and equipment monitoring have led to the development of tools for real-time qualitative assessment of haul roads by haul trucks. The integration of accelerometers in haul truck–mounted road monitoring systems may provide quantitative indicators of haul road quality, and as such add a whole new dimension to haul road management (Thompson and Visser 2006).

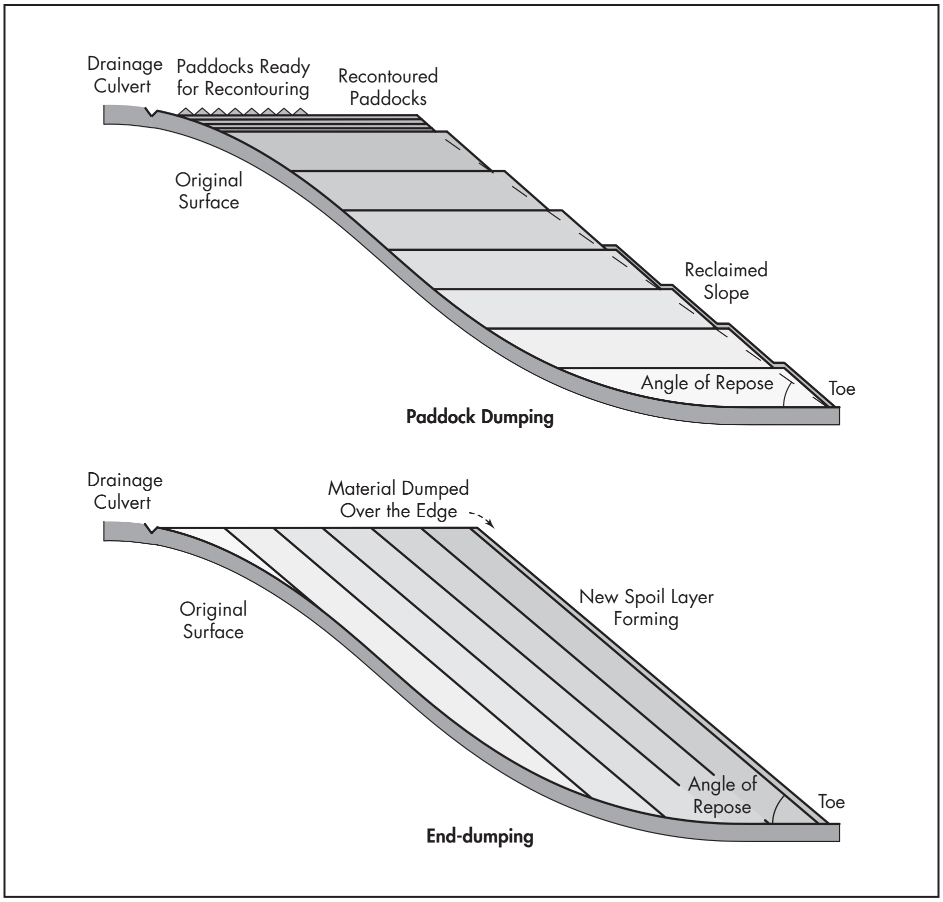
During the scheduling of haul road maintenance, it is important to bear in mind traffic intensity (preferably expressed as gross vehicle mass), function, and projected life span of the road, as well as its maintenance requirements. Routine maintenance and grading may seem like best practice, but when traffic intensities and maintenance requirements are not taken into account, it becomes apparent that this practice may incur excessive maintenance costs on sections of the haul road system with low-traffic intensities. Therefore, optimum maintenance intervals should be determined for haul road maintenance activities, based on a cost–benefit analysis of the relationship between maintenance costs and vehicular cost-incurring factors related to road maintenance, according to Thompson and Visser (2003). They conclude that balancing the optimum maintenance intervals for haul road maintenance and dust suppression with available resources, in conjunction with visual inspection or, ideally, real-time monitoring of haul road performance, should form a solid basis for a practical maintenance strategy for all haul roads in a mining complex. When this proactive, holistic approach to haul road management is implemented, significant gains in operating and maintenance costs, as well as road construction costs, can be achieved.

Two other important elements of a haul road that need maintenance are drainage culverts and safety berms. Similar in size and purpose to safety berms on working benches, safety berms are constructed at the side of a haul road. Drainage culverts are constructed where climatic and hydrogeological conditions dictate their usefulness. They can be located at the pit wall side of the haul road (e.g., where there is a high influx of water) or at both sides of the haul road (e.g., for drainage in areas of high precipitation). Sufficient height of the safety berm and maintaining unobstructed flow in the drainage culverts are essential in ensuring safety on haul roads.

Overburden Disposal

Overburden forms, by far, the largest volume of material produced by most open-pit mines. As overburden generally does not generate any revenue, handling is kept to a minimum. Furthermore, it can contain sulfides or other substances that are potentially damaging to the environment. Consequently, selection of the most suitable site for the overburden embankment involves a trade-off between handling costs related to overburden disposal and the environmental impact of the overburden at a particular site. Minimizing costs involves selecting an overburden embankment site in close proximity to the mine where the environmental impact is as small as possible. Preferably, this site is as close as possible to the projected final pit limit, at the same or a lower elevation as the excavation to minimize upslope haulage costs while maintaining the lowest possible cycle time. With these considerations in mind, optimization of overburden management at a mine site can have a considerable positive influence on the environmental impact and economic viability of a mine.

Overburden is deposited either top-down or bottom-up. Both methods are illustrated in Figure 10.1-2. End-dumping (or top-down dumping) of overburden involves dumping the material over an advancing face. During operation of the dump, only limited reworking of material by dozers is required. Recontouring starts after the end of the dump life. In paddock dumping (also known as bottom-up dumping), the layers of overburden are stacked by dumping on top of the dump, followed with spreading by bulldozers to form relatively thin layers. Paddock dumping is favored from a geotechnical point of view because it allows for more control over the angle of repose of the dumped material and provides a better homogeneity of the material and ultimately better stability (Spitz and Trudinger 2008). Furthermore, paddock dumping provides the possibility



Source: Spitz and Trudinger 2008.

Figure 10.1-2 Comparison between paddock dumping and end-dumping

of concurrent rehabilitation of the overburden embankment and more control over the encapsulation of potentially acid-generating overburden. The lack of homogeneity in top-down dumping provides more potential for settlement and creation of zones of different permeability, both of which can cause more pronounced erosion and eventually instability. Moreover, the lack of homogeneity in rock size can also encourage oxidation of sulfides and consequently acid-mine drainage (Department of Resources, Energy and Tourism, Australia 2006). Paddock dumping does not markedly decrease truck cycle times (Turin et al. 2001). The main advantage of end-dumping is that it is significantly cheaper than paddock dumping due to considerably lower rehandling of overburden both during operation and rehabilitation of the dump site. However, most importantly, paddock dumping is far superior in terms of safety because there is less risk of edge failure and a dump truck falling over an edge when backing up. This point is illustrated by Turin et al. (2001) in an analysis of 10 years of lost-time injuries and fatalities at overburden embankments in the United States. They found that "backing up and falling over an edge" accounts for 73% of fatalities at dumps. Three of the main contributing factors to this are edge failure (23%), the lack of a berm or barrier (35%), and driving through a berm or barrier (31%). This not only shows that proper barrier construction is vital, but also that dump site stability is an important factor. Furthermore, these statistics highlight that, from a safety perspective, paddock dumping, with its better geotechnical control, is the preferred option. In order to further increase safety on an overburden embankment, it is good practice for haul trucks to approach the dumping face from left to right so the operator can inspect the dump berm and dump surface for any tension cracks. The next step is stopping and reversing the haul truck to the dump edge, using the berm as a marker rather than a stopping block prior to tipping.

Pit Expansion

Expansion of an open-pit mine is done in a series of phases, often referred to as pushbacks or cutbacks. From a planning standpoint, a pushback should be aimed at maximizing the financial return from a mine. When planning a pushback, this means taking into account not only the grade of a material but also the costs of development, mining, processing, and marketing (Hall 2009).

The exact geometry of a pushback is very site-specific and depends on a range of factors including ore-body geometry, financial goals, geotechnical consideration, mining equipment, production goals, and long-term planning. Pushbacks can be either conventional or sequential (McCarter 1992). Essentially, both methods push back a pit shell the same distance horizontally; however, a sequential pushback does this through a number of smaller, active benches pushed simultaneously at several levels, whereas a conventional pushback mines the whole horizontal extent of a pushback level before progressing to the next level. Different zones of sequential

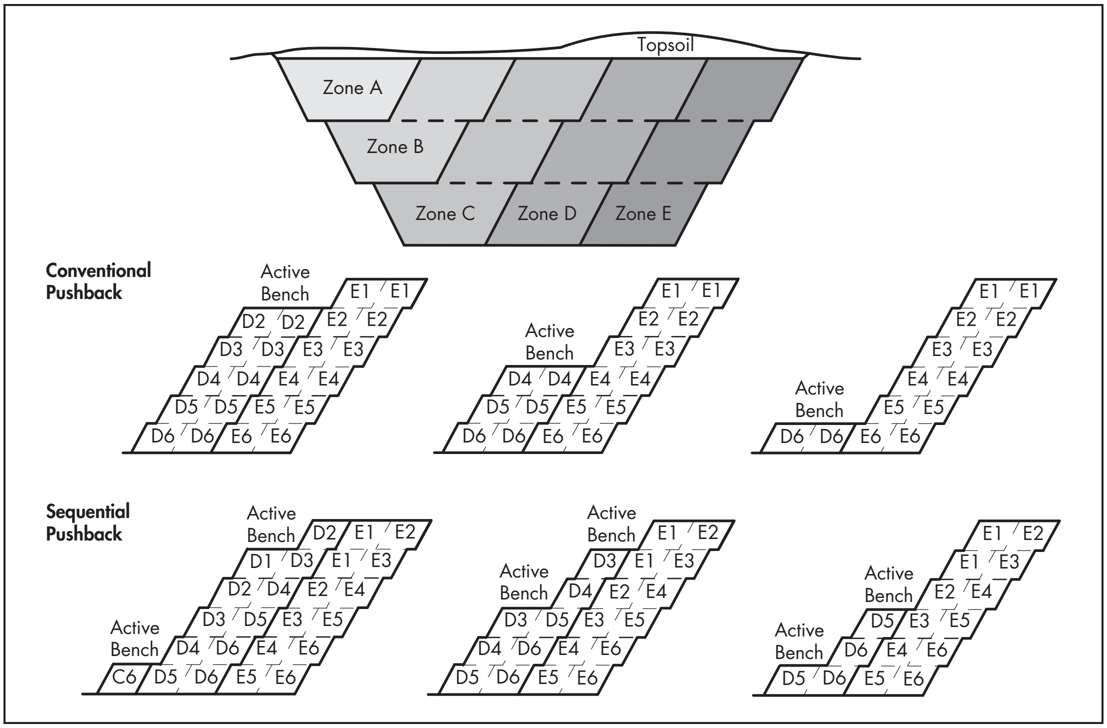


Figure 10.1-3 Conventional and sequential pushbacks

pushbacks are divided by haul roads. Figure 10.1-3 illustrates the differences between sequential and conventional pushbacks. The general pit layout is shown in the top of the figure, divided into different pushback zones (Zone A, Zone B, etc.).

The bottom of the figure illustrates how both methods would mine the same pushback differently. The letter in the blocks denotes the zone, and the number indicates the mining sequence (i.e., mining commences with Block D1, then D2, etc., until the last block is mined).

In a conventional pushback, all activity is concentrated on one level as compared to multiple active levels in a sequential pushback. This means that sequential pushbacks require more complex planning than conventional pushbacks. However, developing several areas simultaneously means there is more flexibility during excavation, allowing superior control over production planning and blending of ore. Furthermore, as sequential pushbacks contain a number of active benches on the pit slope, the working slope angle is lower than the final pit slope angle, which does not have these wider, active benches, resulting in improved slope stability. Conventional pushbacks, on the other hand, are less complex to schedule and the larger active area at a given time allows more working faces within a pushback. Disadvantages include less flexibility in scheduling and blending as well as a greater vulnerability to operational problems.

Presplit blasting is often done at the final reach of a pushback or pit shell. By allowing more control over face angle and back break, this improves long-term stability of the pit face. Sequential pushbacks are more commonly seen in large-scale open pits, whereas conventional pushbacks are more common in shallow and small-scale operations.

Transition to Underground Mining

In some cases, notably with vertically extensive ore bodies, it can be profitable to continue mining by underground methods after the final pit limit has been reached. In recent years several big open pits, such as Palabora, have commenced underground production with others, such as Chuquicamata and Debswana's Jwaneng mine, announcing plans to go underground. The transition from open-pit to underground mining presents a set of unique geotechnical, planning, and management challenges.

With regard to management, the first challenge is deciding on the feasibility of underground mining. Most comparison methods between open-pit and underground mining rely on establishment of a break-even stripping ratio and a comparison of the net present value for the next feasible open-pit pushback to that for an underground mine. After feasibility of underground mining has been proven, timing of underground mining is the next issue to be decided. There are two major considerations in the timing of the transition. First, to maintain continuity of the operation it is important that the underground mine can supplement and eventually take over production from the open pit without major permanent changes in tonnages of ore shipped to the mill. Differences between the open pit and underground ore grade and composition may complicate this issue. A production overlap between open-pit and underground mining is therefore common to allow for a smooth transition from open-pit to underground mining. Second, while a smooth transition requires a production overlap, neither of the two operations should compromise the other's safety. Geotechnical analysis should provide insight into the interaction between the two adjacent operations. Safety in the surface operations could be jeopardized as a result of crown pillar failure or due to mining subsidence-induced slope failure (e.g., when a caving method is used). On the other hand, surface blasting-induced vibrations could compromise safety underground.

Siting of the portal is another major consideration in the transition from open-pit to underground mining. Most importantly, it is crucial that the stability of the underground mine entrance should not be compromised in any way by surface operations. This is as true for a decline portal within an operating pit as it is for a shaft located near a pit wall. Additionally, when a decline portal is sited within the open-pit walls, surface traffic interactions relating to both excavations should be kept to a minimum. This includes not only keeping haulage routes for both operations separated to the highest extent possible, but also considering the effects of blasting.

UNIT OPERATIONS

In mining, unit operations can be defined as those basic steps necessary for the production of payable material from a deposit, and the auxiliary operations used to support the production (Hartman and Mutmansky 2002). Unit operations in an open-pit production cycle include access, drilling and blasting, excavation practices, and haulage methods. Auxiliary operations and other production-related activities include overburden and topsoil removal, ancillary operations and mine services, and equipment monitoring and maintenance.

Access

The first step in the development of a new mining operation or a pushback is gaining access to the area to be mined. Especially when working in remote areas, an appropriately planned and designed access road with sufficient capacity can make a substantial difference in the initial success of a new operation.

After permanent access to the new mining area has been secured, removal of topsoil and overburden can begin. In mining, overburden refers to all unprofitable material that needs to be excavated to access an ore deposit, including topsoil and overburden. Topsoil refers to the layer of unconsolidated material at the surface that is suitable for sustaining plant growth. Because of the unconsolidated nature of topsoil, it often requires different excavation techniques. Depending on climate, topography, and bedrock geology, topsoil can vary from anywhere between centimeters and tens of meters thick. Overburden refers to the consolidated material underlying the topsoil and generally overlying the ore body. If overburden is encapsulated between two layers of ore, it can be referred to as interburden.

Topsoil Removal

After the initial pit outline has been staked out, vegetation should be removed and any surface water courses should be diverted away from the site. As topsoil is generally free digging, scrapers, bulldozers, front-end loaders, and small hydraulic excavators are the most common equipment used in topsoil stripping. Bulldozers can be used for pushing material onto piles for further excavation by front-end loaders or hydraulic excavators. Alternatively, they can support scraper operation by ripping soil or by pushing scrapers along where they do not have enough traction. Graders are mostly used for precision applications such as haul road construction. Haulage distance is an important consideration in choice of equipment. At short haul distances, scrapers and bulldozers are more economic, whereas a more conventional excavator/truck haulage operation tends to be more economical at longer haul distances.

Before the bedrock is reached, the exposed soil may be a large potential source of dust, and a water bowser or tank truck (a mining truck that has been adapted for the distribution of water) may be required for dust suppression. In many operations, topsoil storage is required for reclamation purposes at the end of mine life. In some cases separate storage of different topsoil and subsoil layers may be necessary to ensure quality of the material. Depending on the duration of topsoil storage, revegetation and erosion control may be required.

In some operations, the excavation may never progress into consolidated material and the practices discussed here may also apply to the excavation techniques used for ore.

Overburden Removal

Removal of overburden is often required before extraction of ore can begin. Regarding materials handling, there are three important differences between ore and overburden:

1. Overburden is not beneficiated and will generally not generate any revenue.
2. Overburden tonnages almost invariably exceed ore tonnages in an open-pit mine.
3. The rock mass characteristics are often different from that of the ore.

The first two points imply that handling of overburden and related costs should be kept to a minimum. To minimize costs related to the handling of overburden, it is often blasted to a coarser fragmentation than ore, and in many cases excavated and hauled by larger-capacity equipment. As a result, in many operations the stripping of overburden is contracted out. Drilling, blasting, excavation, and haulage practices for consolidated material will be discussed in more detail in the following sections.

Drilling and Blasting

Drilling and blasting comprise the first two of the four main stages in the production cycle of an open-pit mine and the most common method of rock breaking. Other rock-breaking methods such as mechanical breaking and surface miners can generally not compete in terms of either production rate or economy and will not be discussed in this chapter.

Blast Design

Blast design is the first and most crucial step in drilling and blasting. First and foremost, blast design is an iterative process, where important factors such as the required fragmentation, production, and muck pile shape are used as a starting point for determining optimal drill-hole diameter, depth and inclination, subdrilling, explosives type, and detonation timing (Hopler 1998). Importantly, operating costs of both the mine and the processing plant are directly related to the fragmentation achieved during blasting (Bhandari 1997). Bench height and subdrilling requirements dictate drill-hole depths. *Subdrilling* is the term used for the length a blasthole extends beyond the excavation level. This is done to reduce the risk of equipment damage as a result of poor floor conditions, which in turn is governed by geological features and the strength of the rock mass. Figure 10.1-4 shows a basic blast and blasthole geometry.

Bench height is generally fixed by ore-body characteristics and geometric and geotechnical considerations and is taken as the starting point for blast design. Selection of a suitable drill-hole diameter is a complex process taking into account a host of factors related to production requirements, rock mass characteristics, environmental considerations, and equipment selection (Hopler 1998). In general, it can be said that larger drill-hole diameters can be operated at larger burdens and spacings. However, this does result in larger fragments compared to smaller-diameter holes at the same powder factor. Assuming a fixed bench height, a good indication of hole diameter, D, is given by Bhandari (1997) as

$$D = H/120$$

where H is the bench height in meters.

When determining the blasthole diameter, it should be taken into account that there is an inverse relationship between hole diameter and operational costs (Bhandari 1997). Typical values range from 83 to 350 mm depending on the scale of the operation.

Drilling patterns (e.g., burden/spacing ratio) vary substantially in size, and patterns are tailored to specific operations. There are many ways of calculating spacing (S) and burden (B), most of which are based on hole diameter. The following rules are considered a good starting point for the estimation of the spacing and burden (Hustrulid 1999):

$$B = K \times D$$

where K is a constant and D denotes hole diameter in meters, and

$$S = 1.15 \times B$$

where B denotes burden in meters.

The constant K is dependent on the strength and density of the explosive, as well as rock "blastability." It ranges from 20 in dense rock with light explosives to 40 in light rock with heavy explosives, but typically it lies between 25 and 35 (Hustrulid 1999). In strong or blocky rock and to achieve optimal fragmentation, conservative values should be used when calculating the burden. Typical values for the burden range from 3 m in small-scale operations to 10 m in very large-scale operations.

The relationship between burden and spacing—the burden/spacing ratio—can be varied according to the chosen blasting pattern. Square patterns ($S \sim B$) are easy to lay out but result in poor charge distribution (Bhandari 1997). Elongated patterns ($S > B$) are preferred in hard-breaking rock and when there are problems with back break (Hustrulid 1999). Staggering a pattern further complicates layout but, due to superior blasting energy distribution, results in better fragmentation. If there are problems with back break or when superior face angle control is necessary, presplitting can be done with a smaller drill rig.

Stemming is placed on top of the explosive column to ensure efficient use of the explosive energy and to reduce air overpressure. Drill cuttings are often used for stemming. This is a cheap alternative to inserting specialized stemming material, but it is less efficient at containing blast energy, possibly resulting in unsatisfactory blasting results (e.g., vertical flyrock and oversize blocks). Where possible, angular material is preferred as by nature it tends to lock in place better during the detonation process, further improving confinement of the explosive pressure. Appropriate stemming-chip size lies in the range of 10% of the blasthole diameter. A good approximation of stemming depth is 0.7–1 $\times$ B, and is commonly between 2 and 7 m.

After a drilling pattern is established, a delay sequence should be fitted to this pattern. The first consideration in determining delay intervals is the availability of free faces. A blast should be initiated at the free face and aim at maximizing the use of the free face throughout the blast. When there

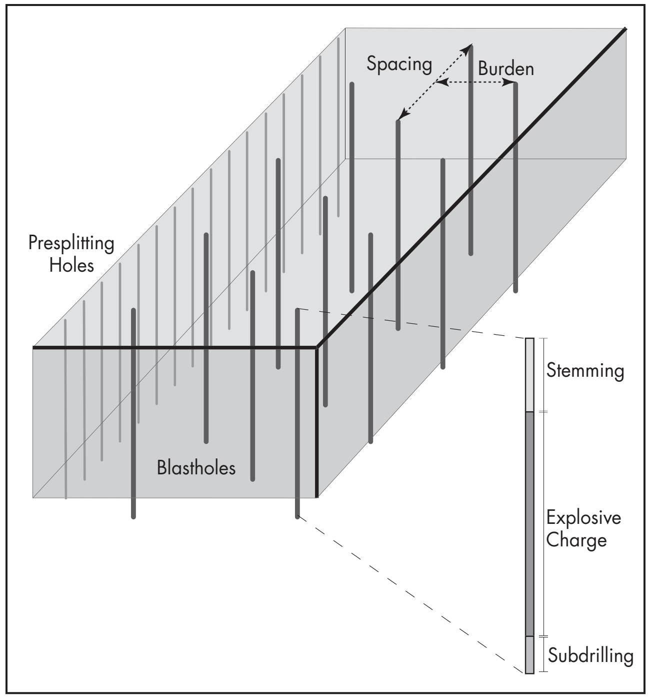
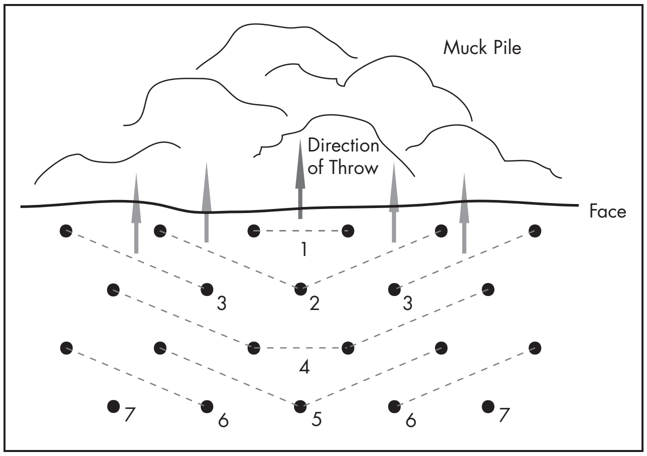


Figure 10.1-4 Blast and blasthole geometry

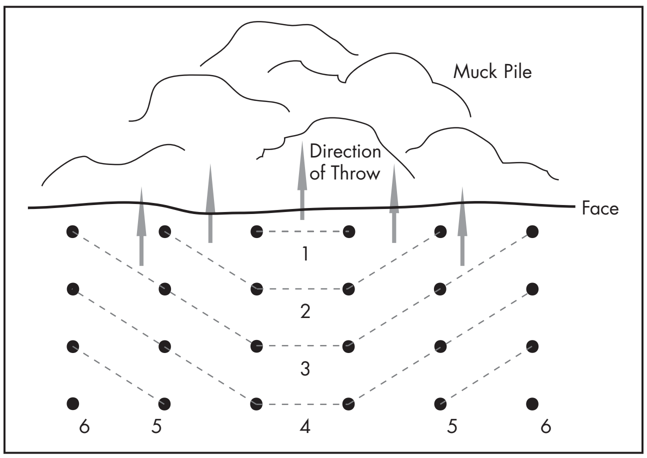
are no free faces available (e.g., a box cut or sump blast), a diamond cut is the best option, displacing rock upward. In the case of one free face, a chevron pattern (V-pattern) is recommended, although a row-by-row pattern can also be used (Bhandari 1997). The angle of the V can be varied according to the local geological conditions and the desired blasting result. When there are two free faces, an echelon pattern usually produces the best results. Figures 10.1-5 through 10.1-8 show different standardized blast designs with relative detonation sequencing of the rows. Other cuts exist but are generally used in more specialized applications. After a delay sequence has been established, appropriate delay intervals can be assigned to rows or individual holes. Regarding delay intervals, Bhandari (1997) recommends 3–6 ms/m of effective burden (i.e., at the time it is blasted and not when it is drilled). Apart from the drilling pattern, other important considerations during the selection of suitable delay timings are safety of the blast, geology, prevention of surface cutoffs, vibration reduction, and fragmentation requirements. Delay timings should be customized to prevailing geological conditions, even within the same operation.

The shape of the muck pile required by the excavator is another consideration during blast design. High, compact muck piles are generally preferable for rope shovels and hydraulic excavators, whereas a low, flat muck pile is better for front-end loaders (Hustrulid 1999). The delay pattern, point of initiation, and number of rows are the main influences on muck pile shape. The higher the number of rows in a blast, the larger the vertical component of rock movement and the higher the resulting muck pile (Bhandari 1997).

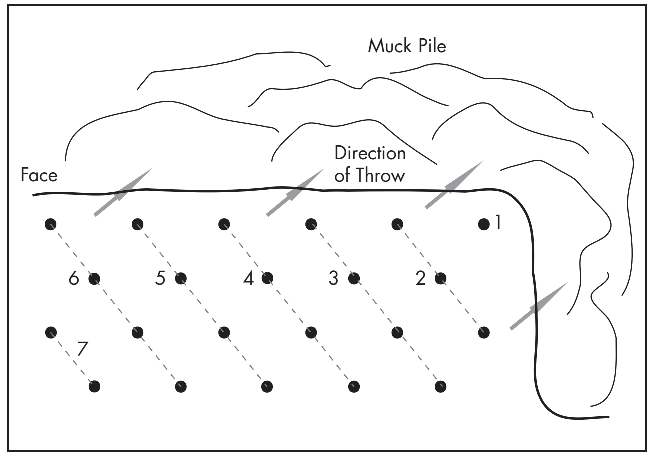
The mining industry is becoming increasingly aware of the benefits of achieving a good fragmentation during blasting. Most importantly, drilling and blasting is, comparatively, the cheapest method of comminution. Therefore, achieving good fragmentation at this early stage can have a significant positive



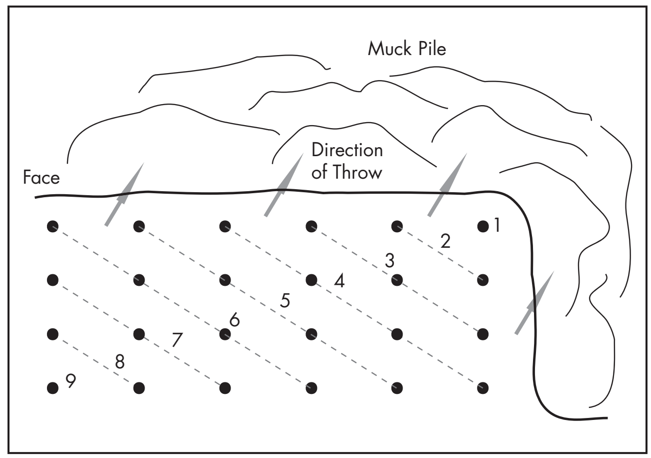
Source: Hustrulid 1999.
Figure 10.1-5 Face blast chevron staggered



Source: Hustrulid 1999.
Figure 10.1-6 Face blast chevron rectangular



Source: Hustrulid 1999.
Figure 10.1-7 Corner blast echelon staggered



Source: Hustrulid 1999.
Figure 10.1-8 Corner blast echelon rectangular

impact on the efficiency and costs of downstream comminution processes (Borquez 2006). Furthermore, better fragmentation allows for better use of the capacity of the excavation and haulage equipment in a mine. On top of that, optimization of blasting at a specific site may suggest that the same results can be obtained by using less explosives, and it may reduce the amount of oversize boulders produced. With these considerations in mind, it becomes clear that determining the optimal fragmentation is a function of not only the effectiveness of the drilling and blasting process but also of excavation, haulage, and the downstream comminution processes.

The first step toward obtaining optimal fragmentation results is adopting a blast design aimed at producing the best fragmentation. Generally, this means implementing a more closely spaced drilling pattern (especially a lower burden) with smaller-diameter blastholes and more accurate timings, but the exact blast design depends on a host of site-specific parameters. Ongoing research is aimed at establishing the most influential factors in fragmentation and how blast design can be geared toward optimized fragmentation using these factors.

Application of high-precision GPSs, accurate drill guidance, and drill monitoring have made it possible to drill blastholes with very little deviation. Together with the use of electronic detonators, these two fairly recent advancements in drilling and blasting have made it possible to consistently produce optimal fragmentation during a blast. These developments are further aided by the introduction of imaging software that can analyze the fragmentation of blasted material, allowing for an iterative approach toward optimal fragmentation.

Drilling

Production drill rigs are usually truck- or crawler-mounted and are powered either by a diesel engine or an electric drive. Pull-down and hoist forces are applied by either hydraulic or chain-hoist systems. A range of systems is available for monitoring machine health and the drilling process.

Production drill rigs are divided into rotary, top hammer, and down-the-hole (DTH) hammer drill rigs. Rotary drill rigs rely on a pull-down force transmitted through a rotating drill string usually with a tricone bit for the cutting action (Australian Drilling Industry Training Committee 1997). Rotary drill rigs are generally most efficient in medium to hard rock and in holes with a diameter larger than approximately 170 mm. Hole depths can extend to more than 80 m in extreme cases. Top hammer drill rigs transmit the hammering force from the drill rig through the drill string down the hole. They are preferred for hole diameters up to 140 mm, depths down to 20 m, and mostly used in small-scale operations and precision applications such as secondary breaking. Hole straightness and energy loss at drill string joints are the main factors limiting the use of this type of drill rig for deeper and larger-diameter holes. DTH hammer drill rigs rely on compressed air for operating a piston at the end of the drill string to provide the hammering action. Common applications include presplitting, dewatering holes, and other applications where high accuracy is required, although they are also used for blasthole drilling. DTH hammer drill rigs are the most efficient drill rig type in hard to very hard rock. Drill-hole diameters for DTH hammer drill rigs commonly lie between 140 and 170 mm, and depths of up to 40 m are feasible. Selection of the most suitable hole size and drill type for a particular mining operation is a function of balancing projected operating and capital costs with the required rock fragmentation, wall stability, grade control, and production requirements.

Generally, drilling and blasting is the bottleneck in terms of time consumption in the drilling and blasting cycle. It is absolutely essential for the overall productivity of a mine that muck pile volumes are sufficient to keep excavators and the haulage system used to the highest degree at all times (Hopler 1998). Drilling productivity is dependent on rock hardness, drill rig and bit selection, bailing air volume, and engine capacity. Rock hardness is the main determinant in the drillability of a material, and the drill rig and bit should be selected for the prevailing conditions. Engine capacity is the most important factor in fitting a drill rig to the drillability of the material as it determines torque, rotation speed, and pull-down force of the rig. Bailing air, provided by an onboard compressor, is used to clear broken rock from the bottom of the hole. The airflow should be sufficient to clear rock chippings out of the hole but not so high as to cause excessive fugitive dust generation, wear on the drill string, and excessive fuel/electricity consumption.

Blasting

Ammonium nitrate-fuel oil (ANFO) is the most common and cheapest surface blasting agent, followed by emulsions and slurries. The ingredients making up the explosive substance are carried to the blasting location in separate compartments of a specialized truck. For safety reasons the ingredients making up the explosive substance are generally not mixed together by the explosives truck until it is on-site, loading the blastholes. Optimal fragmentation is usually achieved when explosives are distributed so that the lower third of the hole depth contains half of the explosive charge (Hopler 1998). If high drilling costs or problems with fragmentation are experienced, it can be advantageous to use stronger explosives or deck charges, or add additives to increase explosive energy (Bhandari 1997). Shock tube, detonating cord, and trunkline/downhole combinations of these two systems are the most common surface detonation systems. Pyrotechnic millisecond connectors are the most common delay mechanisms, although in recent years electronic detonators have rapidly been gaining ground because of unrivalled delay timing accuracy. In most applications, a detonator is used to set off a primer or booster that will in turn initiate the explosive charge.

Before charging a blasthole, it is good practice to check for water (stagnant or influx) and unexpected voids and to verify drilling was done to the planned pattern, hole depth, and inclination. If water is found, the hole should be dewatered; a polyethylene liner can be inserted and/or a water-resistant explosive can be used.

Secondary blasting may be required to break oversize boulders that are too large for the primary crusher. Mudcapping and blockholing are the two most common secondary blasting methods. Mudcapping involves molding an explosive to a rock surface and covering it with mud. Blockholing requires drilling a hole in the rock and charging it with explosive cartridges. Both processes are expensive and may well produce excessive air overpressure and flyrock. It is essential to be able to see all sides of the boulder in the case of blockholing due to the possible presence of a misfire from the initial blast. Alternatively, mechanical breakers can be used to break oversize boulders, but this technique is often inefficient when dealing with competent rock types.

Excavation

Excavation is the third main stage in the production cycle in a mine. Depending on the size of the operation and the type of haulage system, electric rope shovels, hydraulic excavators, or in some cases large front-end loaders are used in open-pit mining operations.

Equipment

Rope shovel bucket capacities have risen to just over 100 t. The largest bucket capacities for hydraulic excavators currently available are slightly lower than that of rope shovels, topping at 90 t. Front-end loaders normally have capacities of around 36 t, although there are larger models with capacities of up to about 90 t.

Until recently, rope shovels were the sole players in the 60 to 100-t range. However, there are now several hydraulic excavators competing directly with rope shovels in this size range. Rope shovels are still the standard in high-production, low-cost mines because of their reliability and long life. In the intermediate size range (30 to 60 t), the choice of equipment is to a large degree site-dependent, opting either for the flexibility of a hydraulic excavator or the reliability of a rope shovel. In the smaller size range (<30 t bucket capacity), hydraulic excavators account for almost all orders and are slowly replacing rope shovels.

The net digging force in hydraulic excavators is a combination of break-out or curling force (bucket-tilt cylinders), boom force (boom cylinders), and the crowd force (stick cylinders). In rope shovels, the net digging force consists of the crowd force (crowd machinery) and rope pull (hoisting machinery).

For hydraulic excavators, face shovel and backhoe configurations are available. Face shovel configurations are preferred in harder rock and with higher rock faces. Backhoe configurations allow for more selective digging and faster cycle times as swing angles can be reduced when loading a truck on a lower level. Front-end loaders are the preferred choice for specialist jobs where their mobility and flexibility can be used to a maximum extent. Examples include blending operations, road and infrastructure construction, working in the confined space of a drop cut, and as support or backup for larger excavators.

Excavation equipment can be evaluated in terms of productivity (metric tons per hour) and efficiency (cost per metric ton). The bucket fill factor is an important consideration in the overall productivity of an excavator and is dependent on the truck coverage, "diggability" of the material, operator skills, and net digging forces from the excavator. Other important factors in achieving acceptable productivity and efficiencies from excavators are matching the trucks to excavator sizes (ideally three or four loading passes), selecting the right excavator for the bench height, and providing enough working space for the excavator and trucks to operate (Hustrulid and Kuchta 2006).

Excavation Practices

Excavation of a new level often begins with a drop cut. Alternatively, when excavations are still above topography (i.e., on a hillside), a new excavation is begun from an access road without the need for a drop cut. The confined working space in a drop cut often requires the use of a backhoe excavator from the top level or a front-end loader driving in and out of the drop cut. Alternatively, a truck can turn and reverse into position next to a small excavator in the drop cut. Lateral extension of the bench is done through frontal or parallel cuts. Frontal cuts involve cutting adjacent niches into the muck pile rather than having an excavator moving with the muck pile parallel to the bench wall as done in parallel cuts (Hustrulid and Kuchta 2006).

As shown in Figures 10.1-9 through 10.1-11, trucks can drive by or stop and reverse into a position next to the excavator. Hustrulid and Kuchta (2006) have written a comprehensive description of different possible modes of truck–excavator operation. According to them, available working space, the necessary swing angle of the excavator, and truck positioning time are the major considerations in the selection of the type of operation. Drive-by operations are most suitable for parallel cuts, optimizing efficiency by reducing positioning time of the truck. Frontal cuts can require excessive swing angles by the loader, making it inefficient. The disadvantage of drive-by operations is that they require larger working areas and ideally a separate ingress and egress route from the loading area. Stop-and-reverse operations can be employed in combination with both parallel and frontal cuts. They require less operating space and are more efficient from the excavator point of view. When there is sufficient space on the bench, a truck can turn without the need for reversing.

With regard to operating techniques, it is generally considered good practice to excavate the farthest or hardest-to-dig material in the first pass while waiting for the truck to position. The intermediate passes can be positioned where the operator deems fit and the last pass can be used for floor cleanup. This is only necessary when there is no dedicated support equipment for floor cleanup. Correct load placement is crucial in avoiding spillages and excessive truck wear. Ideally, loads are centered on the center line of the body (longitudinally) above the hoist cylinders of the truck (laterally) with no material on the headboard and enough freeboard on the sides and rear (Caterpillar 2006). Considering optimal loading of dump trucks, an original equipment manufacturer (OEM) recommends what they call the 10/10/20 payload policy. This policy states that "no more than 10% of loads may exceed 10% over the target payload and no loads may exceed 20% of the target payload" (Caterpillar 2006). Average cycle times range from 25 to 27 seconds for hydraulic shovels in backhoe and face shovel configuration, respectively, to an average of 35 seconds for rope shovels and 38 seconds for large front-end loaders (Caterpillar 2006). In well-fragmented rock, rope shovels commonly have bucket fill factors around 100% to 105%, for backhoe excavators this lies between 80% and 110%, and large front-end loaders generally can attain bucket fill factors between 90% and 110%. Bucket fill factors of more than 100% are achieved by heaping material in the bucket. Backhoe excavators are most efficient on benches no higher than the length of the stick with short swing angles, loading a truck on a lower level.

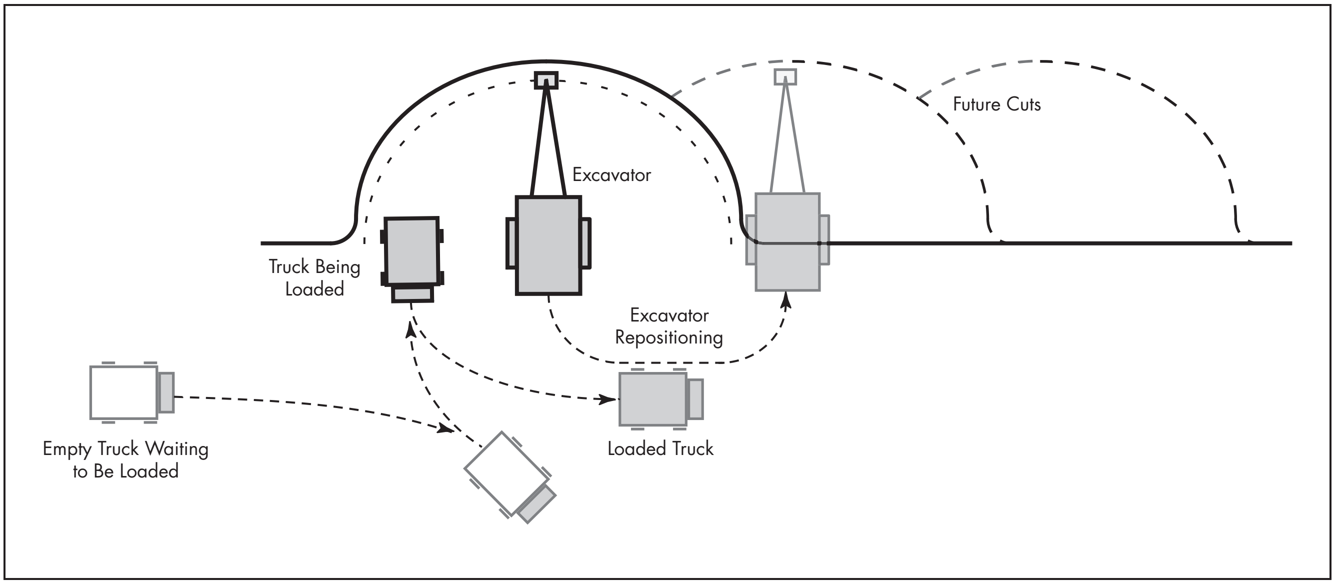
Some of the main productivity losses for excavators are idle time while waiting for trucks, excavator relocation (especially with rope shovels), poor operator skills, bad digging conditions (i.e., poorly fragmented rock, wet material, poor underfoot conditions, incorrect muck pile shape and dimensions), and unplanned downtime. For front-end loaders, there are the additional requirements of good floor condition maintenance and good drainage to prevent excessive tire damage. Careful scheduling can increase productivity by reducing idle time and excavator relocation time. Reduction of excavator cycle times can also improve the productivity of a mining operation markedly. To illustrate this point, in a typical operation, with all other factors unchanged, a reduction of 5% in the excavator cycle time can equate to up to 40 extra truckloads per day. Implementing cycle time improvements suggested by personnel, OEMs, or consultants is crucial in capitalizing on possible cycle time reductions. Among the possible improvements are better floor condition maintenance, adequate operator training, and excavator-optimized blast design to ensure the correct fragmentation and muck pile shape. Furthermore, excavator manufacturers play a pivotal role in providing technological advances such as increased engine performance, monitoring, and automation to reduce possible cycle time.

Grade Control, Reconciliation, and Selective Mining

Open-pit mining has historically been considered a bulk mining method with low selectivity. However, a global trend toward increasingly challenging open-pit mining conditions combined with the need to control costs and optimize mill performance have accentuated the need for optimum fragmentation, selective mining, and improvement in grade control. In line with this need, technological advances make selective mining increasingly feasible by decreasing the selective mining units: the minimum volume of material that can be extracted by an excavator without significant dilution. The ultimate goal of grade control and selective mining is to ensure a constant mill feed, as well as minimizing ore loss and dilution. Depending on the operation, profits from improved grade control, grade reconciliation, and selective mining can be larger than from any other operational improvement and as such they deserve thorough attention (Sinclair and Blackwell 2002).

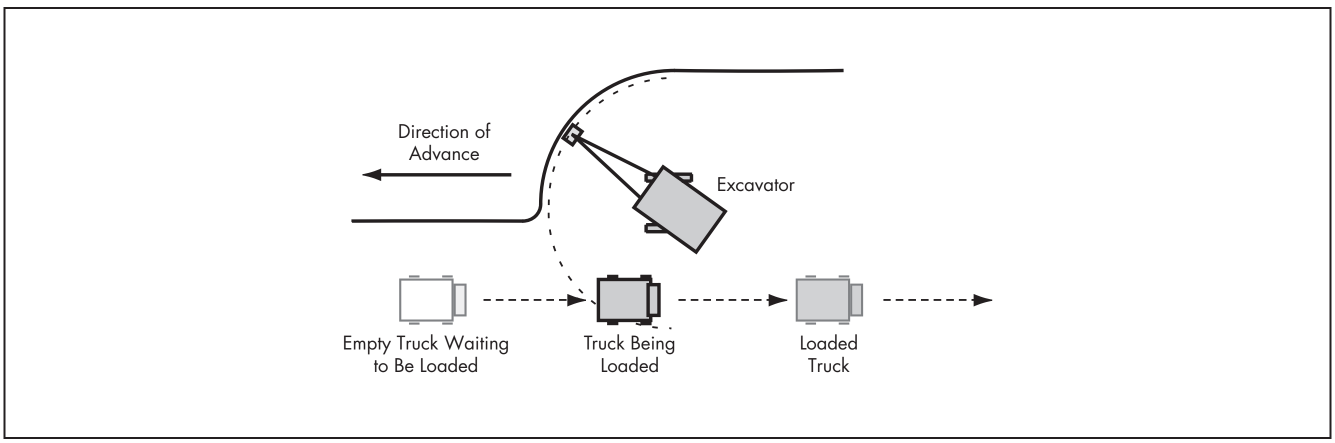
Grade control requirements and practices are largely dependent on the commodity. First, the commodity price controls the implications of ore loss, and management has to justify the extra expenditures relating to selective extraction. Second, the increased mill performance due to lower dilution must justify any additional actions required during the mining processes. Last, the style of mineralization, often commodity-specific, dictates whether grade control is geared more toward ore/barren rock discrimination or whether it is focused on grade and stockpile control (Davis 1992).

Unbiased sample acquisition, meticulous sample processing, and accurate sample analysis on a short time scale are keys to effective grade control. In hard-rock operations, sampling blasthole cuttings is the most effective option for grade control, although sometimes a drill rig is dedicated solely to grade-control drilling (Annels 1991). In this case, timing of



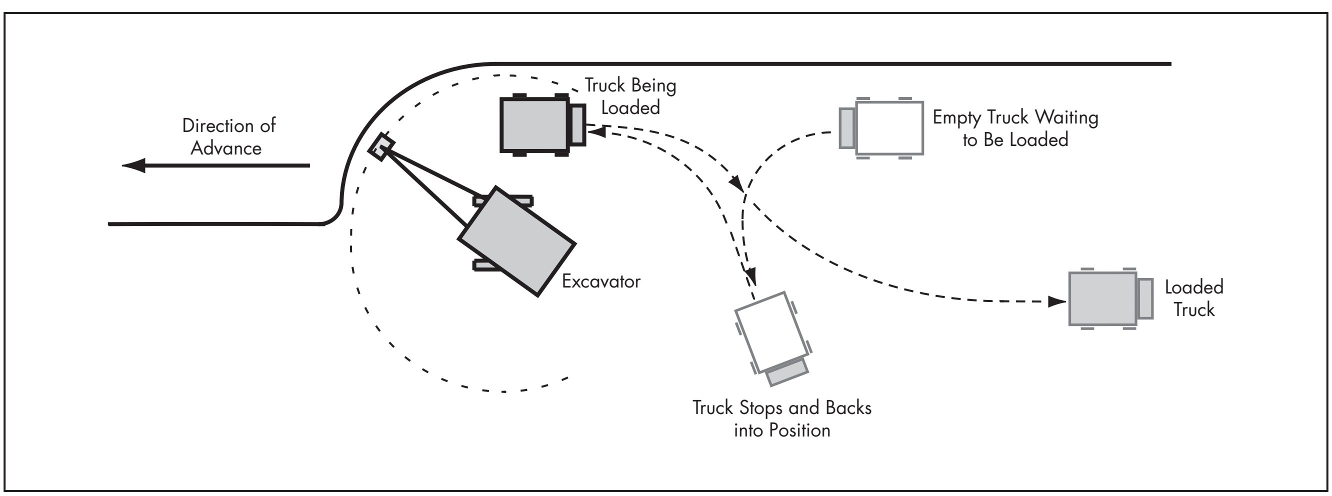
Source: Hustrulid and Kuchta 2006.

Figure 10.1-9 Frontal cut



Source: Hustrulid and Kuchta 2006.

Figure 10.1-10 Drive-by operation



Source: Hustrulid and Kuchta 2006.

Figure 10.1-11 Stop-and-reverse parallel operation

sample acquisition is crucial because of the risk of contamination during drilling. Operations mining unconsolidated material often employ a continuous trencher (commonly referred to as a ditch witch) to produce samples. Best results are obtained by trenching at right angles to the predicted ore-body orientation. Sample results should be used to model grade distributions in planning or geostatistical software, while taking into account the minimum selective mining unit of an excavator. On the basis of the produced models, a clear demarcation of ore and barren rock in active operating areas can be provided, often done using colored flags or pegs. Hanging colored ribbons down the face can further aid excavator operators in discerning ore and barren rock.

The next step in grade control is the selective mining of material. A distinction should be made between free-digging material (e.g., laterites) and consolidated material, because drilling and blasting aggravates potential ore losses and dilution (Davis 1992). In free-digging material, excavators often work across the strike of the ore body. *Flitches* refer to the steps or thin "lifts" in which a bench is mined. Depending on ore-body geometry and the minimum selective mining unit of an excavator, they can be as small as 1 m, although 2.5 m is a more common size. Ideally, mineralized portions of the flitch are removed prior to the barren rock. To minimize risk of dilution and ore loss in a drilling and blasting operation, it is important that excessive intermixing and movement of blasted material is inhibited. When a whole blast is located in ore, spacing and burden can be reduced to increase fragmentation and subsequently reduce comminution costs. Alternatively, when a blast only breaks up barren rock, increased spacings and larger burdens reduce fragmentation and decrease production costs. If ore and barren rock are intermixed on a subblast level, they are blasted together before demarcation of both zones. Bulldozers can be used to separate ore and barren rock before excavating, although this can result in "smearing" of the ore or barren rock margin. In any case, supervising excavations closely is important to ensure correct dispatching of trucks and, possibly, to visually discern ore/barren rock contacts during excavation. After the ore is loaded into a truck, it should be dispatched to the dump or the correct stockpile. The use of modern equipment dispatching systems greatly aids in dispatching trucks to the correct destination.

Apart from ore/barren rock discrimination and assigning metallurgical grades to material, grade control also provides a basis for reconciliation of mill production figures, geostatistical models, and pit production tonnages and grades (Davis 1992). Discrepancies between the mine production and mill production can serve as an indication of poor mine or mill performance, a lack of communication between different departments in a mine, sampling error, lab error, or errors in the geostatistical model.

Haulage Systems

The fourth and last main stage of the production cycle in a mine is haulage. Rigid-frame haul trucks have dominated haulage in open-pit mining operations for decades, although in some cases articulated dump trucks (ADTs) have proven a viable alternative, and sometimes rail haulage is still being used. Furthermore, longer haulage distances in many large pits, availability problems with haul trucks, and improvements in technology have revived interest in in-pit crushing and conveying (IPCC).

Rigid-frame trucks, possibly in combination with trolley-assist hauling, are the preferred choice for haulage in most open-pit mines. Compared to ADTs, they are far superior in payload capacity, achieve better speeds in most road conditions, and have lower maintenance requirements. However, in certain cases such as small-scale operations and operations that struggle to maintain adequate surface conditions, the flexibility and versatility of ADTs can pay dividends. IPCC systems are another alternative to conventional truck haulage. Whether IPCC is economically viable is a function of production, duration of the operation, and the distance and vertical lift of the haulage route.

Trucks

Payload capacities for rigid-frame haul trucks currently range from around 25 t to just over 360 t. Depending on the truck size and manufacturer, there is a choice between mechanical-drive and electrical-drive systems. Mechanical-drive systems use a diesel engine in combination with a mechanical power train, whereas diesel–electric systems rely on a diesel-powered alternator to generate electricity for electric motors (mostly AC). Diesel–electric AC systems dominate the larger truck sizes (>150-t payload), while payload capacities for ADTs generally do not exceed 50 t.

Truck cycle times depend on the type of excavator, capacity of the truck, and haulage distance. Assuming good truck–excavator capacity matching and good digging conditions, trucks can be loaded in approximately 100 to 180 seconds, although this can be longer for front-end loaders. Spotting at the excavator typically takes between 40 and 60 seconds.

In the highly interactive system of today's open-pit mine, productivity of the haulage system is largely dependent on the performance of other activities in the mine, notably the haul road and dump maintenance and the excavator efficiency. For that reason, problems with haul truck productivity and reliability can often be traced back to poor performance of other parts of the operation. The operational interdependence of haul trucks and excavators comes to light during synchronization of their use and during loading. Production scheduling can reduce idle times for trucks and excavators. Payload monitoring and good communication between truck and excavator operators is important when trying to achieve optimal loading. Similarly, good operator training as well as a high degree of coordination, communication, and visual confirmation on the part of both the excavator and the truck operator is required for adequate truck spotting. Ideally, excavator operators should communicate the correct position for a truck to the truck operator, rather than relying on the judgment of the latter to get the truck in the right place. Good communication is especially crucial when double-spotting trucks. This method has the potential to reduce excavator and truck idle time but it is more complex than single-spotting or drive-by operations. Lastly, effective dispatching can maximize the use of trucks and excavators by providing a better synchronization between the two.

Other considerations in dump truck productivity are haul road performance, floor conditions in active loading and dumping areas, and retention of material in the truck's bed. Poor haul road performance (e.g., haul road defects and high rolling resistance) can reduce productivity and reliability of dump trucks. Likewise, poor floor conditions in active loading and dumping areas (e.g., benches and dumps) can also affect productivity and reliability of dump trucks and especially their tires. In both cases, management of the floor conditions in a manner similar to that of haul roads can have a positive impact on haul truck performance. Additionally, retention of material in both the excavator bucket and the truck bed can pose a challenge to operations, especially in arctic conditions or where material has a high clay or moisture content. Adaptation of the truck bed design (e.g., rubber floor mats or circulation of exhaust fumes through the bed to prevent freezing) can alleviate this problem.

Tire Management

Tires are a basic constituent of a dump truck and in recent years tire shortage has been a major challenge for the mining industry. Prolonging tire life can result in big savings, considering that off-the-road tires for large dump trucks can cost more than $50,000 (2009 estimate), while excessive downtime of trucks can result in even higher costs.

More than 80% of tires fail before they wear out. Of all failures, approximately 45% are caused by cuts from spilled material and 30% by impacts with large rock fragments (Caterpillar 2007). Maintaining the correct inflation pressure, maintaining good floor conditions, and ensuring good truck handling and tire awareness by operators throughout the mine are some of the main aspects that can increase tire life. For truck operators, especially when it comes to removing spillages, good communication between truck operators and the road maintenance department is crucial. Analysis of scrap tires and monitoring of tire performance can provide valuable insight into causes of tire failure and possible prevention of premature tire failure. Furthermore, as a response to tire shortages, several major tire manufacturers now offer tire management systems. These systems use software, sensors integrated into tires, and dedicated, handheld tire-monitoring devices to measure and benchmark tire performance.

Trolley-Assist Haulage

Trolley-assist haul trucks are haul trucks that have been adapted from the standard diesel–electric system to a system that relies on pantographs to connect to an electrified overhead line for power supply. Historically, use of this system has been concentrated in southern Africa, but with rising diesel prices, interest in trolley-assist hauling from other parts of the world is increasing. Major infrastructures that need to be installed for the use of trolley-assist haulage include the overhead lines, truck conversion packages, and trolley substations. These conversion packages can be fitted on the majority of diesel–electric trucks.

Increased power supply from the overhead line, compared to a truck-based diesel generator, allows maximum use of the capacity of the electric motor in a truck. As a result, reduced cycle times are possible, and consequently, truck fleets can be reduced. This allows for productivity increases and possibly reduced capital investment costs related to purchasing fewer trucks. Furthermore, as the diesel engine is idling while on trolley assist, fuel consumption and ultimately time between engine overhauls can be reduced significantly. Lastly, energy can be recovered into the trolley supply grid when trucks are moving downgrade.

There is no generic way to determine whether an operation would benefit economically from a trolley-assist system. Savings related to trolley-assist usage are directly proportional to the number of kilometers traveled on the trolley-assist line by the entire truck fleet (Hutnyak Consulting, personal communication). Other important factors to consider in determining the possible savings related to trolley-assist haulage are the truck fleet size, designed life of a ramp, vertical lift of the haulage route, and traffic densities. As such, trolley systems are not suitable for every operation. First, the number of trolley-assist kilometers must be large enough to offset additional investments required for the trolley-assist infrastructure. This is most likely in large mines with a long life and an extensive diesel–electric haul truck fleet. Second, the difference between fuel costs and electricity costs must justify the transition to trolley-assist haulage. Lastly, trolley-assist haulage inherently reduces the flexibility of a haul road system by fixing haul routes. Therefore, trolley-assist hauling is economically most attractive to extensive mines with long, permanent, uphill haul roads and a large truck fleet in regions with high diesel costs relative to electricity costs.

Higher haul truck speeds, more traction, and highly fixed routes increase the chance of rutting and other haul road defects. The main adverse effects of this are reduced productivity and possibly pantograph damage when trucks are rejected from the trolley line. Therefore, haul road maintenance is even more important on trolley-assisted haul roads. The recent adoption of AC drives now allows trucks to connect to and run on trolley assist at variable speeds, negating one of the main disadvantages in the past when they relied on DC drives. Furthermore, sensors can now aid operators in staying underneath the overhead line.

In conclusion, it can be said that trolley-assist hauling can be economically viable, especially now that technological advances have solved some former DC-related disadvantages of trolley systems. However, because of mine-specific circumstances, economic viability of trolley systems should be evaluated on a mine-by-mine basis.

In-Pit Crushing and Conveying Systems

IPCC systems typically rely on gyratory, impact, cone, or jaw crushers to feed an overland conveyor belt that transports material to the mill or overburden embankment. They can be classified into mobile and semimobile systems. Mobile systems are crawler mounted and are often fed directly by an excavator. Having capacities of less than 1,500 t/h, these systems are usually found in small open-pit mines or quarries. Semimobile systems are mostly based on gyratory crushers fed either directly from trucks or from truck-fed apron feeders. They can only be moved with specialized equipment, hence the name semimobile. Having far higher capacities (up to 14,000 t/h) than fully mobile systems, these systems are suitable for mines with very large production tonnages.

The most common conveyor belt configuration is a standard trough-type conveyor. However, this type of conveyor belt suffers from limited curve radii (minimum of ~400 m for large overland conveyor systems) and slope angles (maximum of 16°–18°) it can scale. Pipe conveyors are a relatively recent development used to negotiate tighter curves. They are essentially rubber conveyor belts folded into a pipe shape with idler rollers. As idlers constrain the belt from all sides, far tighter curves can be negotiated. A further advantage is the reduction of spillages and fugitive dust generation. The disadvantage of a pipe conveyor is its limited capacity. To overcome the slope angle limitation, conveyors can be led up switchbacks or a dedicated trench can be excavated for the conveyor belt at the desired angle. Alternatively, one of several high-angle conveyor systems can be used such as the sandwich design or the pocket-lift design. The sandwich design, as the name suggests, sandwiches materials between two conveyor belts kept in place by idlers. The pocket-lift design relies on material being carried in pockets created by wrinkling the belt. A system similar to the pipe conveyor can also be used for high-angle conveying, provided that material is sufficiently confined.

As mentioned earlier, the economic viability of IPCC depends on production tonnages, duration of the operation, haulage distance, and vertical lift. As a general rule, it can be said that if production exceeds approximately 100,000 t/d, when haulage distances surpass 5 km or when the vertical lift exceeds 250 m and if the installation can be in operation for at least 7–8 years, the benefits of IPCC can offset the higher capital costs of this installation. The economic benefits of IPCC rely on the potential to significantly reduce truck haulage distances and consequently, fuel consumption, haul road and truck maintenance costs, and labor requirements.

Furthermore, operations using IPCC are less prone to tire, equipment, or labor shortages. Other advantages over conventional truck haulage systems include lower carbon emissions and improved safety. The resulting reduction in overall mining costs has led to a revived interest in IPCC. IPCC, especially semimobile systems, reduces the flexibility of a mining operation with respect to pit expansion and pushbacks. Additionally, crusher moves and unplanned downtime of the conveyor belt can have serious impacts on the overall productivity of the system. Careful selection of crusher locations should minimize downtime due to crusher relocation.

Advantages of IPCC systems are best realized in large, high-volume open pits with a long mine life. However, regardless of mine life and size, materializing the potential cost savings still requires a detailed economic feasibility study incorporating site-specific operational, geological, and economical aspects.

Ancillary Equipment and Mine Services

Ancillary equipment and mine services, also referred to as auxiliary operations, are "all activities supporting but not directly contributing to the production of ore" (Hartman and Mutmansky 2002). Among the more prominent and important auxiliary operations in an open-pit mine are power/fuel supply and distribution, haul road construction and maintenance, in-pit water management, and the communications infrastructure. These activities do not generate revenue directly. Nonetheless, it is critical for the overall efficiency of a mining operation that these activities are given adequate attention. Bulldozers, front-end loaders, graders, water bowsers, and fuel/lube trucks are some of the most important pieces of ancillary equipment. Graders and water trucks are absolutely essential to haul road maintenance, which in turn is one of the most important elements of an efficiently operating surface mine. The main role of bulldozers and front-end loaders is maintaining active loading and dumping areas, preventing tire damage, and ensuring effective loading and dumping.

The increasing awareness of health and safety within the mining industry has raised attention for mine rescue service. Many large open-pit mines have one or more ambulances, a fire truck or firefighting equipment, and some have advanced rescue trucks with highly trained crews available 24 hours per day to respond to emergencies. Many operations also have an airstrip or heliport locations for emergency life-flight services if needed.

With many large excavators running on electricity, the power system is undoubtedly the most important mine service. Electricity can be supplied either by a utility company (often more economical) or by an on-site generator (in remote areas). Typically, an open-pit primary distribution system consists of a ring bus or main that partially or completely encloses the pit. The distribution voltage is normally 4.16 kV, but 7.2, 6.9, or 13.8 kV are sometimes used (Morley 1990). Radial ties complete the circuit from the bus to the switchhouses located in the pit. Where necessary, substations are employed where equipment voltages are lower than the main power loop. The power distribution evolves throughout a mine life; substations and other components of the power distribution system are mobile so they can follow mining equipment into a new working area. More details on mine power supplies are found elsewhere in this handbook.

In-pit water management, lighting, and communication infrastructure are three other important mine services. The benefits of efficient in-pit water management are discussed in the "In-Pit Water Management" section in this chapter. Flood lights are important as they enable around-the-clock production, which markedly increases productivity at a modern open-pit mine, especially at high latitudes.

The communication infrastructure in mining operations is rapidly evolving into an essential part of the operation as a response to the ongoing increase in complexity of open-pit mining operations. While initially used mainly for radio communications, the communication infrastructure now also carries information essential for equipment dispatching and monitoring (discussed in more detail in the next section). A well-implemented communications structure in a surface excavation can improve safety, efficiency, and productivity by enabling real-time dispatching and monitoring of mining equipment as well as voice and video communications. Additionally, mine services such as the pumping system can now be monitored and controlled remotely; slope-stability monitoring can be centralized to a large degree; and cameras can be located in critical areas such as the digging face, primary crusher, and other key areas. Several different types of networks are available for surface excavations and the choice of system is site-specific. Open standard wireless local area networks such as IEEE 802.11b/g/n are becoming commonplace as they are cheaper and more versatile than proprietary radio systems, satellite phones, or terrestrial phones. Depending on the application, wireless networks can comprise two or more discrete points (i.e., point-to-multipoint networks) or as a more flexible, all-encompassing mesh network (e.g., Bluetooth, Zigbee, IEEE 802.15.4, or similar standards).

Equipment Dispatching and Monitoring

Advances in GPS positioning, mechanical health monitoring, and production monitoring are contributing to an ever-increasing trend toward automation of open-pit mines. Truck dispatching is now commonplace in most operations, and the benefits and potentials of various other monitoring systems are increasingly being recognized.

All major OEMs include a basic machine management package with their equipment. Their most important function is to monitor the health of vital machine functions such as the power train, suspension, and brakes to detect abnormal conditions or impending failure. By enabling proactive rather than responsive maintenance, operator safety can be improved while significantly reducing downtime and maintenance costs. The option exists to have OEM engineers analyze the data and make maintenance recommendations remotely. These packages can be expanded with additional features to increase productivity, such as payload monitoring and road analysis in trucks; bucket load monitoring and motion tracking for excavators; and vibration reduction feedback systems, GPS-based drill positioning systems, and drill progress monitoring for drill rigs.

Equipment dispatching systems are offered by most OEMs and several third parties. Originally, these GPS-based systems were geared toward dispatching of dump trucks to the correct areas. However, by applying the same principles to drill rigs, and to ore blending, auxiliary, and other mobile support equipment, modern systems are better described as equipment management systems. Integration of payload monitoring, cycle times, and other production-related variables have further revolutionized equipment management in mines. Tracking these parameters over time allows benchmarking and bottleneck identification in the production cycle. By providing benchmarks and highlighting previously unnoticed bottlenecks in the production cycle, modern equipment management systems can prove a valuable tool in the optimization of an open-pit operation.

Extensive research is being done into the use of autonomous surface mining systems, and the first pilot-scale tests on the use of haul trucks and drill rigs are now under way. However, the use of autonomous surface mining systems is still in its infancy, and the realization of the huge potential of these systems will be an ongoing process in the next few decades. An in-depth discussion of current developments in autonomous mining systems is found in Chapter 9.8 of this handbook.

Production Planning

Adequate production planning is absolutely essential in achieving the highest possible return from a mining operation (Caccetta and Hill 1999). It is a complex process for evaluating a host of variables including ore-body geometry, mill-feed requirements, and equipment fleet-related factors to achieve the required production. Accordingly, in recent years a large amount of research has been dedicated to improving mathematical and financial principles behind production planning. On the basis of these principles, pit optimization and planning software have been developed and included in major mining software packages to aid in production planning. In many cases, production planning is not simply a case of planning the mining sequence for an ore body. Rather, it should be a cost–benefit analysis not only on the extraction of ore but also on the required development and installation of infrastructure required for ore extraction and the interaction between the two in terms of economic performance, safety, availability of labor, and availability of equipment (Hall 2009).

Depending on the type of production planning, a host of factors such as staff and equipment fleet dispatching and availability, pit expansion, processing alternatives, and commodity prices come together in production planning to meet targets set by the management of a mining corporation. It may be contracted out to a consultancy or it can be done in-house by engineers at the operation or sometimes at a dedicated office.

Production planning can be divided into long-term and short-term planning. Long-term (or strategic) planning concentrates on defining a goal and as such is undertaken on a multiple-years to life-of-mine time scale (Kear 2006). Short-term (or tactical) planning, on the other hand, is more geared toward achieving a goal and is done on a time scale ranging from day-to-day planning to multiple-year production plans. Inherently, long-term planning is more concerned with providing frameworks for pit design and expansions (e.g., pushbacks), production schedules, rehabilitation plans, and equipment selection, whereas short-term planning focuses on equipment use, maximization of productivity, and ultimately meeting productions targets. In modern operations, equipment-dispatching systems play an important role in short-term production planning. Short-term production schedules should have a degree of flexibility in them to respond to unanticipated changes in the production environment. These changes range from simple matters such as a haul road defect to complicated issues such as a slope failure. Assessment and, where necessary, adjustment of short-term production schedules as a result of these changes should be implemented in a timely fashion.

OPEN-PIT MINING AND THE ENVIRONMENT

As mentioned in the introduction of this chapter, the influence of mining on the environment is becoming an increasingly important consideration before, during, and after the mining process. As such, discussions of the environmental impacts, as well as mine closure and rehabilitation of open-pit mines, are warranted. Furthermore, to connect several other deliberations within this chapter, a discussion of geotechnical considerations and slope monitoring is included together with a section on in-pit water management.

Geotechnical Considerations and Slope Monitoring

Throughout this chapter, there have been several references to the importance of adequate geotechnical design and slope-stability monitoring of an open-pit mine. Subsequently, a generic discussion of these topics is appropriate. Readers are referred to Chapters 8.3 and 8.5 of this handbook for a thorough discussion of the subjects of geotechnical instrumentation and slope stability.

Rock mass parameters are an important consideration from the feasibility study onward; they not only affect pit layout and geometry, but also blasting practices and, to a lesser extent, mining equipment selection and the layout of the comminution circuit (Wyllie and Mah 2004). Most importantly, rock mass parameters directly determine the steepest possible slope and face angle while maintaining an acceptable factor of safety, thereby having a major influence on the profitability of an operation. Furthermore, they can rule out the use of certain areas of an excavation for important infrastructure such as the main haul road.

In an active excavation, continual slope monitoring is crucial in predicting and preventing slope failures and, when failure is imminent, mitigating the effects of a slope failure. A comprehensive slope-stability monitoring program reduces the risk of major production delays or even sterilization of part of a reserve permanently as a consequence of a slope failure. Moreover, it ensures overall safety of personnel and equipment in an operation. A survey showed oversteepened slopes, failure to appreciate the effects of water, and rockfalls to be the main causes of injuries, fatalities, and damage to equipment (Sullivan 2006). Adequate design and monitoring can largely prevent these situations from occurring. Another situation where pit-slope monitoring is important is when there are active underground workings in close proximity to an open-pit mine. Crown pillar failure or caving-related subsidence can permanently cease surface excavation activities.

Slope-stability monitoring techniques can be divided into surface and subsurface monitoring techniques (Wyllie and Mah 2004). Surface monitoring techniques include visual survey, direct measurement techniques, prism monitoring, laser systems, and radar systems. Visual survey includes visual inspection of slopes as well as mapping of structural discontinuities (and more recently, photogrammetry). Although the discontinuity mapping and photogrammetry can provide valuable insights into failure mechanisms that cannot be gained using other techniques, they do not provide quantitative data on slope stability and therefore should be supplemented by at least one of the methods discussed in the following paragraphs.

Direct measurement techniques include crack width meters, tilt meters, and other similar devices. These are low-tech methods to provide accurate indications of minor displacements in a possible failure zone. Prism monitoring relies on the use of total stations at permanent, stable reference positions to determine the distance to prisms mounted in areas of instability. From the change in spatial coordinates of targets over time, the displacement velocity and direction can be calculated. It is a very cost-effective method but it is vulnerable to atmospheric conditions such as excessive dust or mist. Laser systems rely on a laser scanner to produce a three-dimensional point cloud model of a slope. The higher density of points compared to prism monitoring makes laser scanning more comprehensive than conventional surveying techniques. A further advantage of laser scanning is that it can aid in photogrammetry and the mapping of discontinuities. Radar systems are similar to laser systems but provide higher accuracy. The drawbacks of radar systems are that they can only monitor one single area at a time compared to a broader picture as gained through laser scanning, and they are generally more costly. However, because of the unrivaled accuracy (submillimeter), they are often used to monitor the highest risk areas, such as working faces or areas of known instability.

Subsurface monitoring techniques include time–domain reflectometry, borehole probes, extensometers, and inclinometers. These techniques rely on measurement of changes of the inclination or other characteristics of a borehole that could indicate deterioration of stability. Additionally, seismic monitoring techniques are used. These rely on geophones registering acoustic emissions associated with failure events.

The most cost-effective approach to slope-stability monitoring is generally a combination of several of these techniques where they are used to complement one another. For instance, laser systems or prism monitoring can be used to determine overall stability of pit slopes and identify possible failure zones. If instability of a slope is detected, extensometers or radar systems can be used for more precise determination of movements in this area.

A last important consideration in slope stability is the presence of groundwater. Phreatic levels in and around a mine are crucial for maintaining pit-wall stability, especially in areas with clayey material or where the rock is heavily affected by structural discontinuities (Wyllie and Mah 2004). If the climate has periods of prolonged frost, the freezing/thawing cycles can further aggravate the negative effects of groundwater on slope stability. Piezometers are the main tool for determining groundwater levels. These, together with rain gauges, can act as an early warning system and serve as a basis for adjustment of the rate of water extraction from dewatering wells to prevent groundwater-induced failures. Apart from the geotechnical implications of groundwater in and around a pit, there are also major production-related considerations associated with in-pit water. These are discussed in the following section.

With mining being a business enterprise, geotechnical design, monitoring, and stabilization of an open-pit mine is ultimately a matter of economics: balancing the benefits and costs of stabilization against the costs and implications of a slope failure (Pine 1992; Wyllie and Mah 2004). This is in sharp contrast to civil engineering where the social and financial consequences of failure can be far more extensive (Pine 1992). A combination of this different approach to risk management together with the bigger scale of potential slope failures means that civil engineering solutions to slope stabilization are generally not feasible in surface excavations. Furthermore, surface excavations can possibly tolerate a degree of slope failure that would be unacceptable in civil engineering applications (Wyllie and Mah 2004). As a result of this, lowering of the water table and decreasing the slope angle are often the only practical options in surface mining, although in some cases applying civil engineering solutions has been economically viable (Wyllie and Mah 2004).

In-Pit Water Management

Responsible water management is essential in minimizing the environmental impact of most mining operations. Furthermore, there are considerable geotechnical, operational, and economical advantages to in-pit water management (i.e., more stable pit walls and a lower stripping ratio). A sound approach to in-pit water management requires the development of drainage strategies for both surface and groundwater and continuous monitoring of the performance of the water management plan (Department of Resources, Energy and Tourism, Australia 2008). For this, a thorough assessment of local geology, rock mass characteristics, hydrogeology, surface hydrology, and local climate are required. When deciding on dewatering methods, it is important to consider not only the above-mentioned factors but also the logistics related to open-pit mine dewatering. This includes the interaction between the chosen dewatering method, dewatering-related infrastructure (i.e., power supply and water transport from the well) and unit operations in a mine (Atkinson 2000).

To maximize the advantages of dewatering, sections of an open-pit mine must be dewatered before mining begins. Ideally, wells, drain holes, pump lines, and other dewatering infrastructure are situated such that they do not require rerouting as mining progresses. Additionally, the destination of water removed from the pit is an important consideration. It can be reinjected elsewhere, used in mineral-processing operations, or it can be treated and discharged into surface water courses.

There are several advantages to a correctly implemented dewatering program. First and foremost, dewatering pit slopes improves and maintains slope stability. This results in safer working conditions and allows for steeper slopes, lowering the stripping ratio. Second, a lower moisture content of blasted material increases diggability and reduces haulage costs as dry material has a lower mass than wet material, and there is less retention of material in the excavator bucket and the truck bed. Detrimental effects of wet haul roads include unsafe traffic conditions, more tire cuts, and increased rolling resistance. Lastly, water influx into blastholes is decreased, reducing the need for blasthole dewatering or the use of more expensive water-resistant explosives.

Rain can constitute one of the main influxes of water into an open pit. As such, a thorough understanding of the typical local climate and its extremities (e.g., monsoons) is invaluable when formulating the water management plan for an operation. Rain gauges can be a further aid, monitoring rainfall and providing an indication of increased influx and possibly the demand to adjust the pump rate.

Stream diversion and dewatering wells are the main water management activities outside a pit perimeter. Inside the pit perimeter, dewatering wells, sumps, horizontal drain holes, and in some cases drainage adits or grouting are used for water management. Stream diversion, both of permanent and ephemeral streams, is done during the development stage of a mine before or concurrent with topsoil removal. Depending on the hydrogeology, dewatering wells can be situated within pit boundaries or outside of them (Atkinson 2000). Where the direction of flow is mainly lateral, dewatering wells outside the pit perimeter are generally more effective. A further advantage of dewatering wells outside the pit perimeter is that they can be installed before mining commences. However, they are less effective at preventing vertical inflow through the pit floor. To mitigate this, in-pit (vertical) dewatering wells are more effective (Atkinson 2000). This type of well creates more drawdown in the pit than dewatering wells outside the pit perimeter, but they cannot be installed prior to mining, and it generally also requires more complicated logistics. In-pit horizontal drain holes are used for locally depressurizing targeted areas (Atkinson 2000). They are an inexpensive method that can significantly increase slope stability because the small scale of these holes allows them to be installed quickly, targeting specific problem areas. However, they can only be installed after mining begins, they suffer from freezing effects in arctic areas, and water removal usually requires a sump downstream of the drain holes. Sumps are catchment basins at the base of a pit that serve the purpose of collecting in-pit water so it can be pumped back out. Grouting refers to the injection of chemicals that block pores in the rock to provide a barrier that prevents groundwater influx. This is a costly option that is only effective if there is a very well-defined geological feature producing the majority of water influx in a pit. Drainage adits around a pit shell serve the same purpose as drain holes. Because of the labor involved, it is a very costly and inflexible option that is becoming increasingly scarce.

Environmental Issues

In recent years, the pressure on mining companies to minimize the environmental impact of their operations has increased considerably. As a result, prevention and mitigation of detrimental effects to the environment are now high on the agenda of modern operations. Some discussion of environmental considerations related to mining in general is available elsewhere in this handbook, so this chapter focuses on environmental issues specific to open-pit mining.

One of the largest environmental impacts associated with open-pit mining stems from the physical change of the landform as a result of the vast quantities of material moved (Spitz and Trudinger 2008). The physical change of the landscape encompasses both the excavation itself and the disposal sites for overburden and tailings. The excavation will have a large impact on the visual amenity of the landscape, drainage patterns, and groundwater levels. Furthermore, the higher stripping ratios and generally lower ore grades mean that surface mines vastly exceed underground mines in the overburden volume that is generated, both from ore processing and overburden. As a result, a larger surface footprint is taken up by the overburden embankments and tailings impoundment generated by open-pit mining. Associated with the larger volume is not only a larger aesthetic impact on the landscape, but also a higher potential risk of spillage of toxins into the environment. Additionally, the changes in topography make the site more susceptible to erosion. Concurrent or postmining revegetation and placement of geotextiles on erosion-prone surfaces can provide sufficient protection against both wind and water erosion.

Chemical contaminants, increased turbidity, changes in flow patterns, and higher susceptibility to flash floods are the main effects of open-pit mining on the surface water regime (Spitz and Trudinger 2008). Chemical contamination and suspended solids can generally be removed in treatment plants. Channeling or rerouting streams away from vulnerable areas can be done to prevent erosion and mobilization of contaminants. Lowering of the groundwater level in an area due to mining may result in vegetation losses (and consequent changes in fauna), ground settlement, and lower flow rates from spring-fed surface water. Most detrimental effects of groundwater level control can only be effectively mitigated after mine closure.

Effects of blasting include excessive vibration and air overpressure, as well as dust, fumes, and possibly flyrock. The maximum allowed peak particle velocity (PPV) depends on the vicinity of populated areas and national or regional regulations. The U.S. Bureau of Mines found that cosmetic damage to houses can start at a PPV of 12 mm/s at a frequency of 10 Hz. The onset of damage is dependent on both vibration frequency and construction quality of a building, but in general it can be said that a higher frequency needs a higher PPV to be damaging (Siskind et al. 1989). Flyrock should be avoided altogether. Correct blast design should minimize environmental effects from blasting, and good communication with local residents can reduce perception of blasting by the public.

In populated areas, dust, noise, and road traffic effects are more pronounced than at remote mine sites. Noise causes disturbance of wildlife and annoyances both with residents and operators. The emphasis on noise management should be on reducing noise and limiting exposure time (Department of Consumer and Employment Protection of the Government of Western Australia 2005). In this context, hearing protection is regarded as an interim noise-protection measure unless other measures are demonstrably impractical. Cabs on modern equipment are designed to reduce noise exposure and as such play an important role in limiting exposure time. Lower perception of noise by the public can only be achieved by taking noise reduction into account during production planning, control of noise at the source, and noise barriers. Typical exposure limits imposed by legislative bodies range from 80 to 90 dB(A) for average exposure levels and peak exposure levels from 135 to 140 dB(A) (NIOSH 1998).

Depending on the particle size, shape, and chemical composition, dust can cause physical or chemical contamination of equipment and soils as well as respiratory and dermatological problems, reduced visibility, and coating of vegetation

(Department of Resources, Energy and Tourism, Australia 2008). The extent of the impacts of dust is highly dependent on climatic conditions and dust composition. Dust is generated by drilling and blasting, excavation, haulage, dumping, and processing of material, or it can emanate from poorly vegetated or bare areas like tailings impoundments in combination with wind. Dust generated from drilling can be suppressed by adding water to the bailing air from the drill hole, and by employing drill deck shrouds that envelop the drill stem. Haul roads are the most significant contributor of dust in surface mine operations, emitting between 78% and 97% of all dust.

As discussed earlier in the section on haul roads, dust suppression measures include spraying with water or chemical dust suppressants, and compacting or changing wearing course material. If loading, hauling, and dumping causes significant dust release, wetting the material before excavation can be considered, although this has detrimental effects on the efficiency of excavation and haulage. Water sprays around other sources such as stockpiles and the mill can be used to prevent fugitive dust from these areas. In Australia, maximum exposure levels for PM10 (dust with a median diameter <10 μm) lies at 50 μg/m^3 on a 24-hour average (Department of Resources, Energy and Tourism, Australia 2008). Toxicity of the particulate matter is dependent largely on silica content and exposure levels should be adjusted accordingly.

An increase in mining-related surface traffic on public roads may cause congestion, damaged or polluted roads, dust, noise, and unsafe situations. Where unacceptable situations occur, other forms of haulage, careful route planning, and mitigation of effects can relieve pressure on the road network, but often the problems cannot be solved completely.

Mine Rehabilitation and Closure

The main objective of mine closure is to ensure physical and chemical stability, as well as restoration of the ecosystem in all areas disturbed by the mining operation (Spitz and Trudinger 2008).

Before closure of a mine, ore robbing or scavenging can be done. This practice involves extraction of residual ore tied up in haul roads and other areas that did not allow extraction of the ore for operational reasons. One example of ore scavenging was done at Palabora, South Africa, to supplement underground ore production. It is important to realize that ore scavenging may reduce factors of safety for slopes to dangerous levels and should only be done after careful analysis of the probability, extent, and consequences of slope failure. Furthermore, it may inhibit access into the pit, should this prove necessary in the future.

Mine closure aspects specific to open-pit mining mostly arise from the large surface impact and large volumes moved (Spitz and Trudinger 2008). Careful planning during the development and operational stage and taking into account postmining use of an operation can significantly reduce postmining site disturbances and costs. Generally, it is attempted to put the postmining landscape to an equal or better use compared to the premining landscape. Stabilization and erosion protection of steep slopes and unstable faces are crucial for ensuring physical and chemical stability. This usually involves recontouring to stabilize pit faces and embankments to approximate original contours of the landscape, followed by establishment of vegetation or liner placement for erosion protection. Flooding of the excavation allows it to be more successful at blending in with the landscape and natural ecosystem, it prevents oxidation of sulfides and subsequent potential acid mine drainage, and it aids in the prevention of reestablishment of drainage patterns. Backfilling is another option for rehabilitation of an excavation but may be a very costly option that is generally only considered when there is a depleted open-pit mine nearby, when there is an active excavation or other supply of large volume of material in close proximity to the pit to be backfilled, or when backfilling is explicitly demanded by environmental regulations.

Maintaining chemical stability of overburden and tailings typically involves the prevention of acid mine drainage and subsequent metal leaching, as well as immobilization of any other toxic chemicals left over from the mining process (Spitz and Trudinger 2008). Flooding or sealing any potentially acid-generating material prevents oxidation of sulfides and the resulting acidification of water. If mine drainage is expected to contain any contaminants, a treatment plant should be erected with sufficient capacity to treat all effluent water and possible storm surges.

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