CHAPTER 3.2

Rock Blasting

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PURPOSE OF BLASTING IN THE MINERAL INDUSTRY

In most metal mining operations, the in situ ore is separated from the waste rock and is subjected to a series of breakage and separation processes to convert it into a valuable product. Profitability in this industry depends on how efficiently the in situ rock is converted into the final product. Because the breakage and separation processes that take place at the mine and mill are interdependent, traditionally, mining and milling processes are managed as separate cost centers and optimized with little understanding of the impact of one over the other.

Blasting is the first step in that breakage and separation process, but traditionally, the main purpose of drilling and blasting in most metal mines is to break the in situ rock mass into a muck pile of an appropriate size distribution and looseness so that the excavation equipment can load and haul blasted rock efficiently. The fragmentation and muck pile looseness are expected to be achieved without causing uncontrolled blast movement, which poses a safety risk to people and equipment, without causing damage to surrounding rock mass (or walls), and without causing environmental nuisances such as ground vibrations, airblast/noise, and dust.

During the past two decades, several researchers have demonstrated that all the processes in the mine-to-mill production value chain are interdependent and also understand that the impact of blasting outcomes on the overall process efficiency is critical to improve overall profitability.

The purpose of this chapter is to discuss the importance of the drill-and-blast outcomes in comminution and separation processes in the mine-to-mill value chain and then use it as a lever to improve overall profitability rather than a series of simple stand-alone processes. This chapter includes discussion of the following areas:

- Explosive rock interaction during blasting and an explanation about what happens in a blast—how rock fragments and moves in a blast
- Key blasting outcomes and their impact on downstream process efficiency
- Different modeling approaches used to predict blast fragmentation, ore loss, and dilution
- Measurement of blast outcomes
- The traditional blast optimization approach and its drawbacks compared to a holistic mine-to-mill optimization approach
- Case studies to demonstrate the potential value of the mine-to-mill optimization approach
- Challenges and limitations of this approach and further potential of next-generation mine-to-concentrator ideas and technologies

EXPLOSIVE ROCK INTERACTION—WHAT HAPPENS IN A BLAST

The interaction of the surrounding rock mass with the high-pressure and high-temperature gases released by the detonation of an explosive is a very complex process (Fourney 1993). In a blast, the in situ rock mass fragments and moves as a function of the detonation properties of the explosive, the physical-mechanical and structural properties of the surrounding rock mass, and the confinement provided to the explosion gases. The particle size distribution (PSD) or fragmentation in a blast muck pile is formed by

- The new fractures created by the detonating explosive charge,
- Extension of the in situ fractures in combination with the new fractures, and
- Liberation of in situ blocks and the newly formed blocks (Figure 1).

Detonation is the beginning of the rock breakage process in blasting by explosives. An explosive confined within a blasthole is detonated by a pentolite booster that was in turn initiated by a detonator, as shown in the Figure 2. The detonator can be initiated at any prescribed time and the accuracy of its timing depends on whether is it is an electronic detonator or conventional shock tube pyrotechnic detonator. Once initiated, the explosive ingredients within the blasthole are rapidly converted into gaseous products at very high pressures and temperatures. The chemical reaction or detonation front travels along the explosive column generally at a velocity of

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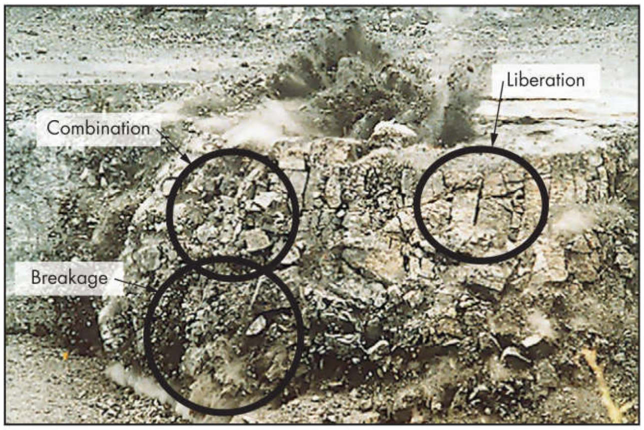


Figure 1 Fragmentation mechanisms in blasting

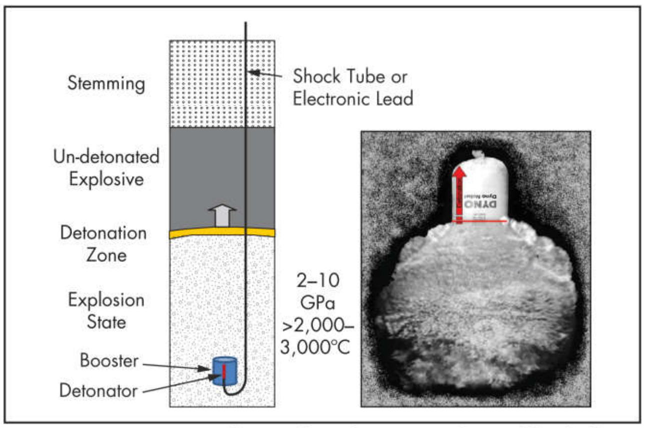


Figure 2 Detonation phase of explosives within a blasthole

Table 1 Properties of commercial and military explosives

Explosive Type	Density, g/cc	Velocity of Detonation, m/s	Explosion Pressure, GPa
ANFO*	0.8–0.85	3,500–4,500	2–4
Heavy ANFO	1.1–1.35	4,500–5,500	6–10
Emulsions	1.1–1.25	5,000–6,000	7–11
Military explosives	1.3–1.4	7,000–7,500	16–20

*ANFO = ammonium nitrate and fuel oil

3,000-6,000 m/s, which is defined as the velocity of detonation (VOD). The pressure of explosion gases within a blasthole for commercial explosives and military explosives such as PETN (pentaerythritol tetranitrate) are given in Table 1.

The explosion pressure of commercial explosives is around 2–10 GPa, whereas the dynamic compressive strength of most rocks will be less than 300 MPa. Therefore, the pressure of explosion gases within the blasthole is much higher (at least 1,000 times) than the dynamic compressive strength of any rock mass.

The sudden impact of the high-pressure explosion gases on the borehole wall transmits a compressive stress wave into the surrounding rock (Figure 3). If the intensity of the stress from this stress wave is more than the dynamic compressive

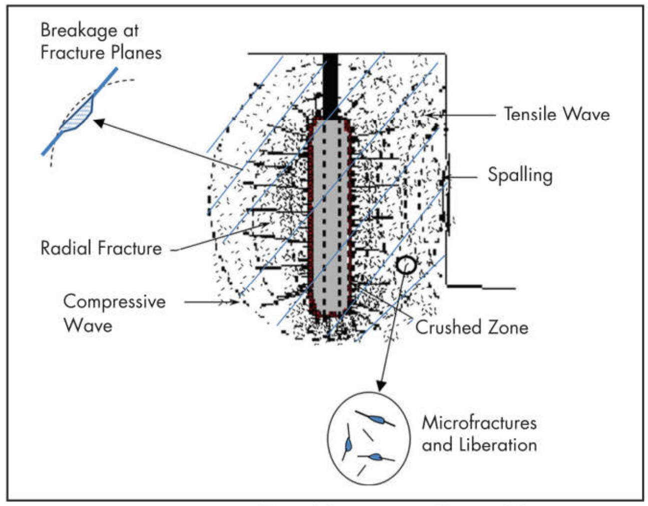


Figure 3 Stress wave–induced fracturing during blasting

strength of the rock, it pulverizes the rock matrix. PSD within this zone is expected to be less than the grain size of the rock matrix.

In ore blasts, the extent of the crush zone can have a significant effect on the efficiency of the grinding circuit because most of the particles within the crush zone require very little additional grinding. The extent of this crush zone depends on the magnitude of blasthole pressure, dynamic compressive strength, and elastic moduli of the surrounding rock mass. In a given rock, high-VOD explosives such as emulsions produce a larger crush zone compared to low-VOD explosives (ANFO [ammonium nitrate and fuel oil] or diluted ANFO). Similarly, a given explosive produces a smaller crush zone in harder rocks compared to a softer rock.

Beyond the crushing zone, the shock wave compresses the material at the wave front and induces a tangential tensile stress known as hoop stress. If the intensity of the hoop stress is more than the dynamic tensile strength of the rock, tensile radial fractures will develop. Both the compressive and tensile components of the wave decay with increasing distance from the blasthole. When the compressive wave hits a free face, nearly all the energy is reflected back as a tensile wave and develops tensile fractures known as spalling.

Microcracks Generation

Since the stress wave is transient, the major radial fractures will not have sufficient time to propagate and hence will branch out by forming microcracks. These microcracks may not propagate very far, but their total surface area in the rock mass may be 10–100 times larger than the total surface of the primary fragments in a blast (Revnivstev 1988). Generally, more microfractures will be formed by an increase in magnitude and the rate of energy release. Therefore, high-energy blasts with high-VOD explosives are expected to produce more intense microfracturing.

Fragmentation and Breakage due to Gas Penetration

During and after the stress wave propagation, the high-pressure gases confined by the blasthole walls and the stemming penetrate into the fractures (in situ fractures and new fractures developed by the stress wave) and expand them further. Some researchers believe that the fracture network throughout the rock mass is completed before the gas pressure phase (Hino 1956; Duvall and Atchison 1957). Other researchers believe that a significant portion of the fracturing process takes place during the gas pressure expansion phase (Kutter and Fairhurst 1971; Brinkmann 1987, 1990).

Impact of Discontinuities on Rock Breakage

Discontinuities and fracture planes in the rock mass play a significant role in the fragmentation mechanism. When the compressive stress wave from the blasthole encounters a discontinuity or a mineralized vein, some portion of the energy is reflected as a tensile wave, creating new fractures along the fracture plane. The propagation of radial fractures formed by the stress wave will be arrested when they encounter a discontinuity, but the energy will be dissipated by forming microfractures along the discontinuity. Subsequent penetration by explosion gases also expands the preexisting fractures, thus liberating the particles bounded by these discontinuities and also by liberating the material within the discontinuities. If the discontinuities are mineralized, then breakage along the discontinuities will have a significant effect on the liberation characteristics of the ore in downstream processes.

Blast Movement

Blast movement is the last stage of the rock-blasting process. In flight, collisions and landing produce minimal rock breakage compared to the shock phase and gas penetration. High-speed photography studies of several large-scale blasts indicated that in many blasting instances the explosion gas remains in the rock mass even after considerable burden movement (Chiapetta et al. 1983; Cameron 1992). The magnitude and direction of rock movement at any location in the blast depend on the confinement conditions, energy intensity, rock mass properties, blast geometry, and timing design. Blast movement can have a significant effect on ore loss and dilution. Mischaracterization of the grade boundaries, both prior to as well as a result of blasting, can lead to significant financial losses because of ore loss and dilution. Ore dilution occurs when waste material is miscategorized as ore and sent for processing diluting the run-of-mine (ROM) head grade and recovery. Ore loss takes place when valuable mineral is miscategorized as waste and sent to the waste dumps, thus never being processed and recovered.

Burden movement also creates a path of least resistance toward the free face; hence, the explosion gas tends to move toward the free face rather than penetrating into the fractures behind the blasthole. This is called relief and is a very important factor for control of blast-induced damage. Proper confinement of explosion gases in the rock mass is necessary to produce the desired fragmentation and displacement. Adequate confinement requires the stemming material to remain in place until the explosion gases release their full potential energy, and this can be achieved by using good stemming material (e.g., crushed gravel instead of drilling cuttings) and by providing sufficient stemming length. Inadequate and poor stemming will result in loss of energy into the atmosphere, reducing the effective heave energy of the explosive energy. It will also result in excessive airblast and noise. On the other hand, if the stemming column is too long, the explosive will be poorly distributed, resulting in the formation of boulders and poor fragmentation from the collar region.

BLAST OUTCOMES AND THEIR IMPACT ON DOWNSTREAM PROCESSES

Traditionally in metal mining, the key objectives of the blasting process are to

- Eliminate any safety hazards such as misfires and fly rock,
- Achieve sufficient fragmentation for efficient digging and primary crushing,
- Achieve an appropriate muck-pile shape and looseness suitable to the loading equipment,
- Not cause any excessive damage beyond the blast boundaries,
- Not exceed statutory ground vibration and airblast limits, and
- Minimize unit drill-and-blast cost per metric ton of rock.

The effects of blasting outcomes on grinding and separation were not considered and limited to the primary crusher. This approach works if profitability of the operation depends only on moving the rock from place A to place B and requires very little downstream processing (e.g., a quarry). However, in most operations, the blasted rock undergoes several other processes before it is sold as the final product; therefore, the traditional approach may not necessarily lead to optimum overall process efficiency. In addition to safety, the other key blasting outcomes that will impact the mine-to-mill value chain are

- Fragmentation,
  - Run-of-mine particle size distribution,
  - Quality of blast fragments (ore softening due to microcracks),
- Blast damage, and
- Blast movement (ore loss and dilution).

Run-of-Mine Particle Size Distribution

Run-of-mine particle size distribution (ROMPSD) has a direct influence on the performance of mining production equipment, crushers and grinding mills, and separation, as shown in Figure 4. The definition of oversize depends on the downstream process conditions and equipment size. Large equipment can tolerate bigger oversize compared to small equipment.

Muck piles with finer fragmentation not only improve the dig rates of excavators but they also reduce maintenance costs due to less stress on loading equipment. The fill factor of trucks is influenced by the size distribution of the muck, as Michaud and Blanchet (1995) have shown that the payload of a truck loaded with finely fragmented muck is greater than a truck loaded with coarsely fragmented muck. Primary crusher throughput and power consumption are strongly influenced by the ROM oversize. Eloranta (1995) and Elliot et al. (1999) demonstrated that reducing the oversize in the ROM ore can increase crusher efficiency.

Semiautogenous grinding (SAG) and autogenous grinding (AG) mills operate most efficiently when they are fed with a bimodal feed size distribution. To run SAG and AG mills efficiently, ore blasts should aim to maximize fines (–20 mm) generation and minimize the critical size (20–100 mm) and maintain the media (+100 mm) around 10%–15% (Figure 5A). The amount of rock media has a more pronounced effect in AG mills than SAG mills. AG mills require around 10%–15% of rock media for efficient grinding, whereas in SAG mills, some of this rock media can be compensated by adding steel media. In highly fractured ores, maintaining rock media may

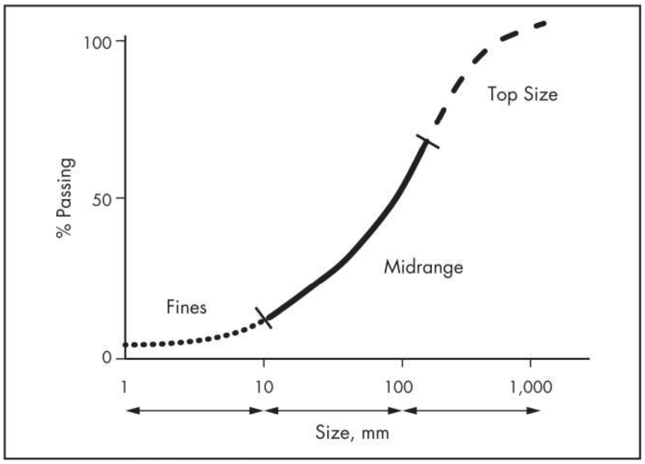


Figure 4 ROMPSDs and their impact on downstream processes

be difficult because it is governed by the in situ size distribution. Hence, blasting strategy in these ores should aim to increase fines and retain in situ grinding media as much as possible (Figure 5B).

In hard and massive ores, generally there is enough rock media; therefore, by adjusting blast patterns and primary crusher settings, it is possible to reduce the rock media to less than 15% (Figure 5C). Primary crushers are mostly used to reduce the top size from the ROM ore. Therefore, it is advantageous to generate as much fines as possible in blasting to maximize mill throughput. Control of critical size during blasting may be difficult, but the negative impact of critical size on mill efficiency can be minimized by optimizing the grates (pebble ports) and crushing the rejected pebbles using conventional crushers.

In some operations, excess fines may not help downstream processes. For example, fine coal is difficult to handle, suffers low yield, carries excessive moisture, and often attracts a lower sales price. Iron ore fines are sold at a reduced price or must be pelletized to regain value. Quarry products are sold on the basis of size and shape and are particularly prone to a reduction in value from over-fragmentation. In certain heap leaching operations, excessive fines can affect acid percolation and impact leach recovery (Scott et al. 1998). In some operations, it is important to understand the material handling properties of fines and their impact on all downstream processes, otherwise excess fines may result in bogging of crushers and storage bins.

Blast Damage: Impact on Separation

Separation of waste and ore starts at the mine, where strip ratio (waste/ore) determines the amount of waste that needs to be removed to expose the ore. Any breakage that takes place beyond the blast boundary limits is considered as damage. Poorly designed and implemented controlled blasts can cause damage near the crest and leave excessive toe near the floor. A combination of crest damage and excess toe will result in the loss of catch berm, which in turn can lead to sterilization of ore and pose a safety risk to both machinery and human life (Figure 6). In addition, the actual pit slope will become flatter, thus increasing the amount of waste.

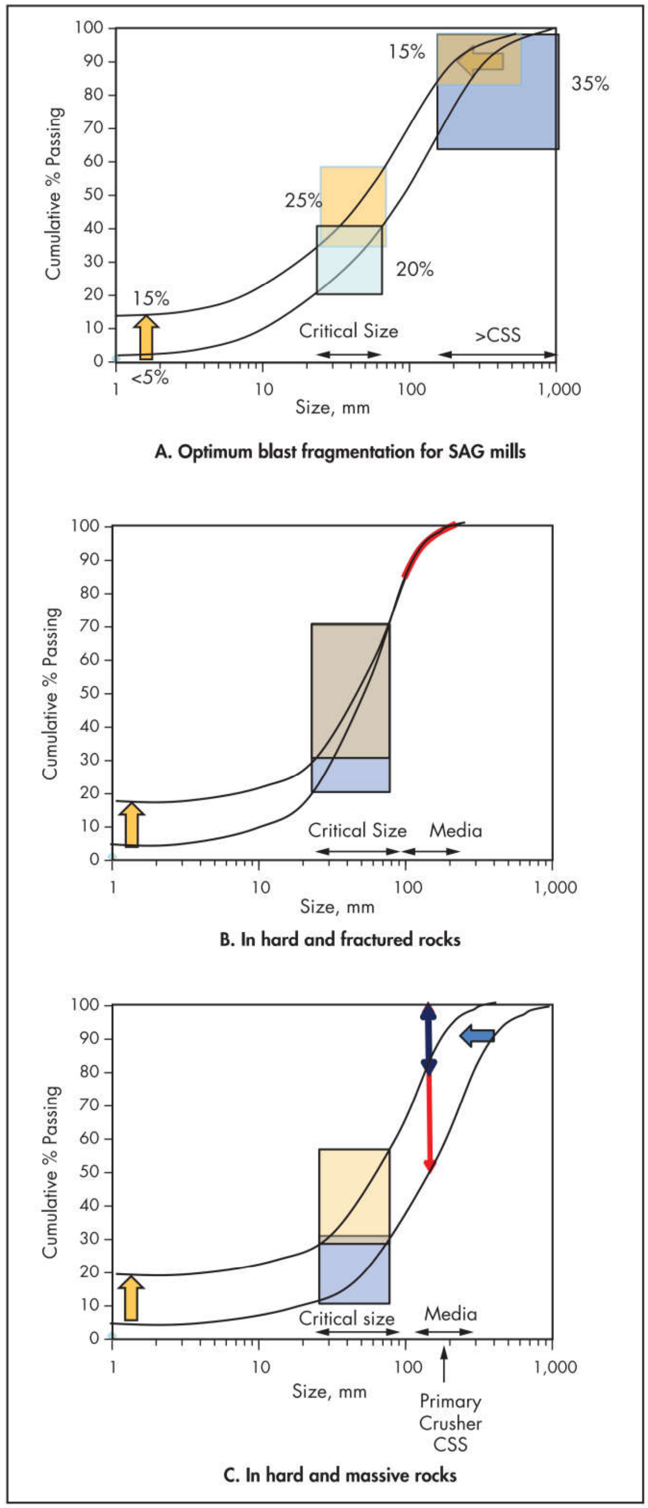


Figure 5 Blasting strategies to control feed size for SAG mills

General thinking in the mining industry is that high-energy blasts produce greater damage than low-energy blasts. This is why most mines implement controlled blasts adjacent to the final walls and interim walls to minimize the blast damage to these walls. These controlled blasts are fired after the

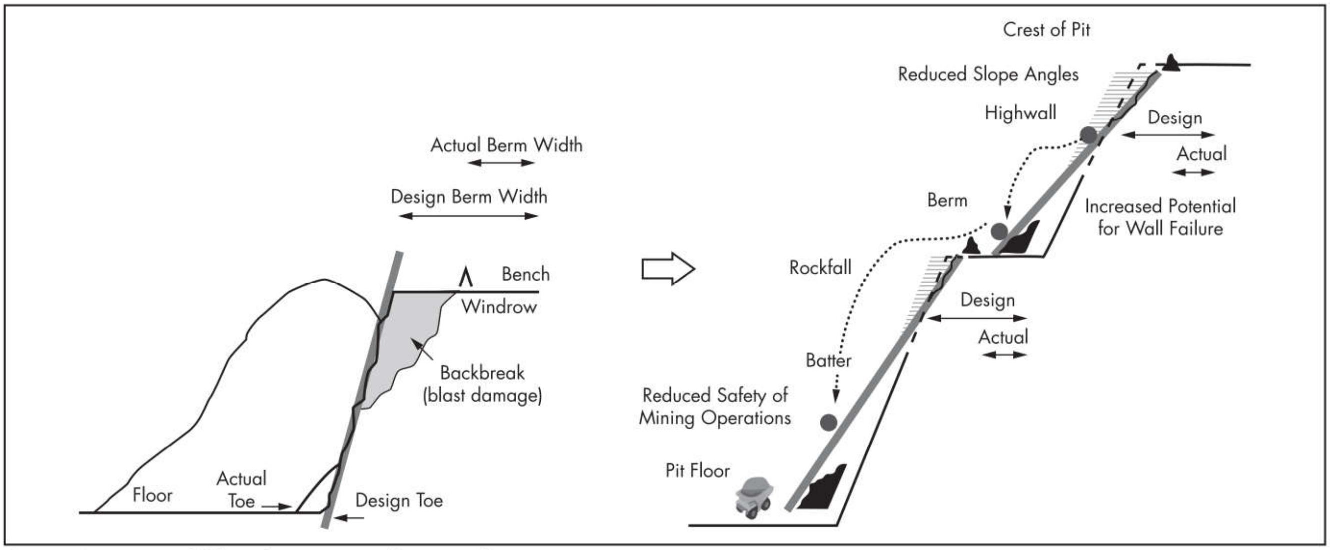
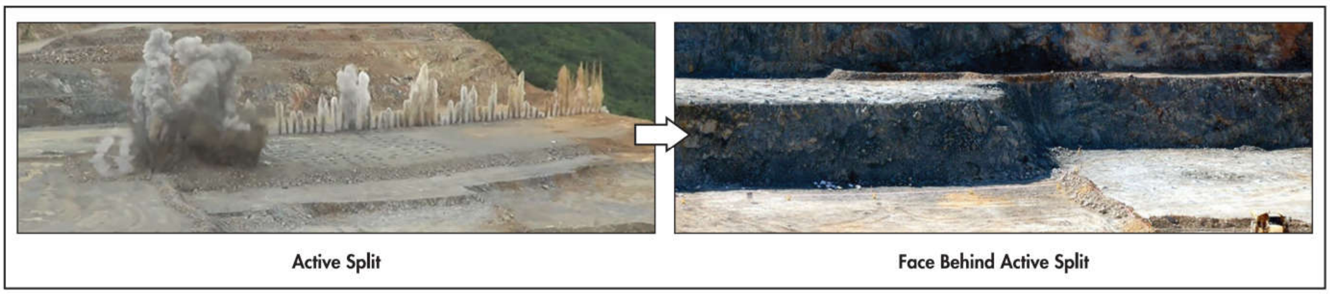


Figure 6 Impact of blast damage on slope angle



Source: Gaunt et al. 2015, reprinted with permission from the Australasian Institute of Mining and Metallurgy

Figure 7 Active split trials and results

production blasts and are generally designed with low energies and good free-face conditions to provide decent relief. It is true that sometimes the fragmentation and blast-movement outcomes from these controlled blasts may not be optimum for downstream comminution and separation processes.

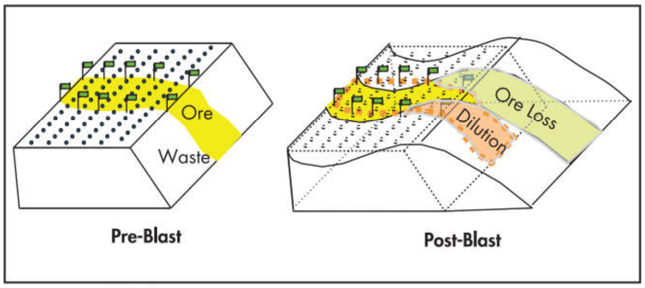
In some cases, production blasts are fired along with the controlled blasts for scheduling and production purposes. In those cases, blast damage can extend to the final walls from the production blasts. However, with proper understanding of the damage mechanisms and with appropriate blast designs, it is possible to minimize blast damage even in high-energy blasts. Gaunt et al. (2015) demonstrated that implementing an active presplit between the high-energy production blast and trim blast can prevent the activation of joints into the wall, and reduce the damage behind the high-energy blasts (Figure 7). Another option is to change the blast sequence such that a carefully designed trim shot is fired before the high-energy production blast, creating a broken rock filter between the production shot and the final wall. This broken rock absorbs the energy from the high-energy production blast and thus limits blast damage to the wall (Scott et al. 1999).

Blast Movement: Impact on Dilution and Ore Loss

In many mines, ore and waste are often blasted together, and the rock mass within the blast volume is fractured and displaced relative to one another. Blast movement can be detrimental to the accurate delineation of post-blast grade boundaries if the movement is not consistent and properly understood (Tordoir 2009). This issue is demonstrated in Figure 8, where the post-blast grade boundaries are assumed to be the same as pre-blast boundaries, and so no account is made for movement.

The consequence of movement can be blast-induced ore loss and dilution. *Ore loss* takes place when valuable mineral is sent to waste dumps, and *ore dilution* occurs when waste rock is mischaracterized as ore and sent to the mill for further processing. Unintended mixing of different ores can also have other negative effects on the quality of the final product. For example, if a high-arsenic ore is mixed with the clean ore, the final concentrate quality will be affected. Similarly, if oxidized clay ores are mixed with fresh sulfide ores, this can result in lower recovery.

Generally where dilution is a major concern, production blasts are designed with low powder factors and choked conditions to reduce any lateral movement during blasting (Crone et al. 1973; Little and Van Rooyen 1988; Davis et al. 1989). Such an approach may reduce blast-induced movements, but it can have significant impact on fragmentation and muck-pile looseness, thus affecting the efficiency of downstream operations. More importantly, this simplistic approach does not address the fundamental cause of ore loss and dilution: Is it blast movement or a lack of understanding of blast movement that causes ore loss and dilution?



Source: Kanchibotla 2014

Figure 8 Blast-induced ore loss and dilution

Blast-movement studies have shown that the economic impact of blast-induced ore loss and dilution on a mining operation's profit is significant. Taylor et al. (1996) monitored ore control grade versus crusher discharge grades for Coeur Mining's Rochester mine in Nevada (United States) from 1992 to 1994 and estimated that if the mine had accounted for blast-induced movement during 1994 alone, the mine could have produced an additional 200,000 oz of silver and 4,560 oz of gold and would have added some US$2.7 million in annual revenue (assuming a gold price of US$380/oz and silver price of US$4.80/oz). Blast-movement results at an open pit gold mine operated by Placer Dome Inc. indicated that the economic impact of blast-induced ore loss was greater than US$1.5 million and dilution of about US$0.5 million (Thornton et al. 2005). Engmann et al. (2013) from Ahafo gold mine in Ghana estimated that the cost of ore loss and dilution due to unaccounted blast movement of ore polygons was around 33% and approximately US$160,000 for three production blasts. Adjusting ore blocks for blast movement is an effective way to reduce blast-induced ore loss and dilution.

In addition to blast movement, several other factors will affect overall dilution. Some of those factors are shape of ore body, accuracy of ore body modeling, excavator size and shape, production sequence, and so on. Although ore loss and dilution have a significant impact on the overall profitability of a mining operation, very few mines have a real understanding of their current dilution level and the cause.

Impact of Microcracks and Preferential Breakage on Comminution and Separation

The laboratory experiments conducted by several researchers suggest that the microcracks developed during blasting action can reduce the impact breakage resistance of post-blast fragments (Revnivstev 1988; Chi et al. 1996; Nielsen and Kristiansen 1995; Kojovic and Wedmair 1995; Michaux 2005; Kim 2010; Parra 2013; Parra et al. 2016). In some mineral deposits, the valuable minerals are deposited along cracks and fissures, and preferential breakage along these fissures can liberate mineral-rich material. Because of its relatively low to moderate strength, a material deposited within fractures tends to become fragmented into small size fragments (fines). Bulk sampling and screening tests of ROM ore from the Bougainville mine in Papua New Guinea showed better upgrade potential in ores where mineralization was mainly on fracture planes compared to highly disseminated ores (Burns and Grimes 1986). Laboratory blast trials at the Julius Kruttschnitt Mineral Research Centre (JKMRC) showed the beneficial effect of blast-induced preferential breakage and upgrading of the fines fraction from two ores (JKMRC 2002).

Generally, marginal ores and mineralized wastes are blasted and removed or stockpiled without further treatment. However, by exploiting this selective breakage mechanism, it may be possible to further process the high-grade fine fractions. Considering that the amount of low-grade or marginal ore is frequently larger than the amount of ore above the cutoff grade, the proposed approach offers an avenue for substantial increase of a mine's economically recoverable and treatable mineral reserves. Increased reserves and successful implementation of the proposed technology will ultimately result in increased profitability and increased value for mining companies. In recent years, the Cooperative Research Centre for Optimising Resource Extraction in Australia had developed and implemented Grade Engineering technology at several large open pit operations by deploying a range of coarse waste rejection solutions (Walters 2016).

BLASTING MODELS—HOW TO PREDICT BLAST OUTCOMES

A variety of modeling approaches have been used to describe the rock-blasting process and to predict blasting outcomes. They range from highly computationally intensive numerical models to reasonably simple empirical models. Over the years, several numerical models were developed to estimate either breakage and fragmentation or movement. The hybrid stress blasting model (HSBM) is one of the recent numerical models developed in collaboration between the University of Queensland, Itasca Consulting Group, and the Cambridge, Imperial, and Leeds Universities (Sellers et al. 2012). HSBM is a rock-breakage engine to model detonation, wave propagation, rock fragmentation, and muck-pile formation. It also incorporates an ideal and a nonideal detonation performance and predicts fragmentation, displacement, and damage. The main limitation of most numerical models is that the amount of computational power and time required to simulate production blasts is quite large. That is why empirical and semi-empirical models are more popular to predict blast outcomes; some of them are discussed in the following sections.

Blast Fragmentation Models

Kuz–Ram Model

One of the most popular blast fragmentation models is the Kuz–Ram model. It is an empirical model that uses the equation developed by Kuznetsov (1973) to estimate the mean fragment size:

$$x_{50} = A \cdot K^{-0.8} \cdot Q^{0.167} \cdot \left(\frac{115}{E}\right)^{0.633} \qquad \textbf{(EQ 1)}$$

where

x_{50} = 50% passing fragment size, cm
A = rock factor:
7 for medium-strength rocks
10 for hard and fractured rocks
13 for very hard and massive rocks
K = powder factor, kg/m^3
Q = mass of main explosive per blasthole, kg
E = relative weight strength of explosive (ANFO = 100)

Cunningham (1983) combined Kuznetsov's model with the Rosin–Rammler equation to calculate the entire size distribution (Equation 2) and developed an empirical equation to estimate the uniformity index "n" using Equation 3.

$$P = 1 - e^{-0.693\left(\frac{x}{x_c}\right)^n} \quad \textbf{(EQ 2)}$$

where

P = cumulative weight passing size (x)
x = diameter of fragment
x_c = characteristic size (mean size)
n = uniformity index (relates to the slope of the distribution)

$$n = \left(2.2 - 14\frac{B}{D}\right)\left[\frac{1+\frac{S}{B}}{2}\right]^{0.5}\left(1 - \frac{W}{B}\right)\left(\frac{L}{H}\right) \quad \textbf{(EQ 3)}$$

where

B = burden, m
D = charge diameter, mm
S = spacing, m
W = standard deviation of drilling accuracy, m
L = charge length, m
H = bench height, m

Lilly (1986) modified the description of the rock factor (A), which is derived from properties of the rock mass:

$$A = 0.06\ (RMD + JF + RDI + HF) \quad \textbf{(EQ 4)}$$

where

A = rock factor
RMD = rock mass description
JF = joint factor
RDI = rock density influence
HF = hardness factor

Research at the JKMRC and elsewhere has demonstrated that the Kuz–Ram model underestimates the contribution of fines in the ROM size distribution (Kojovic et al. 1998; Comeau 1996; Kanchibotla et al. 1999). The model estimates were reasonable for quarry blasting applications where the impact of fines is not as important as in metal mining operations where the ore needs to undergo crushing and grinding processes.

JKMRC Blast Fragmentation Models

JKMRC researchers have investigated many alternative approaches to improve the fines prediction of fragmentation models (JKMRC 1998). One of the approaches is the crush zone approach developed by Kanchibotla et al. (1999), and the other is a two-component model (TCM) developed by Djordjevic (1999). Both models assume that fragmentation in a blast takes place in two different mechanisms—(1) creation of new fractures and (2) liberation of existing fractures—and therefore estimate the coarse end and fines end of PSD separately. Both models estimate the coarse end of the size distribution similar to the Kuz–Ram with slight modifications to the rock factor estimate.

The main difference between the JKMRC models and Kuz–Ram model is how they estimate the fines end of size distribution. The TCM uses experimentally derived parameters from small-scale blast chamber tests to estimate the fines region of the model. One of the main drawbacks of the TCM is the requirement to conduct a series of small-scale laboratory blasting tests to derive the crushed zone parameters, and the crush zone model has been developed to overcome that limitation.

Crush Zone Model

The crush zone model (CZM) assumes that fines in a blast are generated predominantly by the crushing of rock around the blasthole due to compressive and shear failure. The coarse fragments are generated primarily by tensile failure, by extending the in situ fractures and by liberating in situ block. The CZM uses a rock calibration factor (RCF) instead of a constant 0.06 to estimate the rock factor.

$$A = RCF \times (RMD + JF + RDI + HF) \quad \textbf{(EQ 5)}$$

The CZM also calculates the RMD and JF differently from the Kuz–Ram model; a detailed description of their calculations is given in the AMIRA International report (JKMRC 1998). The extent of the crushed zone is determined by calculating the point at which the radial stress around the blasthole exceeds the dynamic compressive strength of the rock. The stress around the blasthole is estimated by assuming that explosion gases apply pressure on the blasthole walls quasistatically by using the following equation (Jaeger and Cook 1979):

$$\sigma_x = p_b\left(\frac{r}{x}\right)^2 \quad \textbf{(EQ 6)}$$

where

σ_x = radial stress at a distance, x, from the center of the blasthole
p_b = borehole pressure
r = radius of the blasthole

The borehole pressure for a fully coupled explosive is estimated by the following equation:

$$p_b = K\,\rho_e\,VOD^2 \quad \textbf{(EQ 7)}$$

where

K = constant dependent on explosive density (between 0.25 and 0.125)
ρ_e = density of the explosive
VOD = velocity of detonation of the explosive

For decoupled charges, the explosion gases expand adiabatically and hence the blasthole pressure will decrease. The rock matrix within the crushed zone is assumed to be totally pulverized and the particle size within this zone is understood to be less than the grain size. The default value is assumed as 1 mm. By using the crushing zone radius, the volume and the percentage of fines within the crushed zone is estimated using Equation 7:

$$\% -(\text{grain size}) = 100 \times V_{crush}/V_{br} \quad \textbf{(EQ 8)}$$

where

V_{crush} = volume of crushed zone
V_{br} = volume of blasted rock per blasthole

The uniformity index and the coarse end of the size distribution are estimated by using Kuz–Ram Equations 1 through 3. The fine uniformity index and fine end of the distribution is then calculated by substituting the % (grain size or –1 mm value) and characteristic % (X_c%) using the following equation:

$$n_{fine} = \frac{LN\left[LN - \frac{(1 - \text{fines}\,\%)}{-X_c\%}\right]}{LN\left(\frac{1}{X_c}\right)} \quad \textbf{(EQ 9)}$$

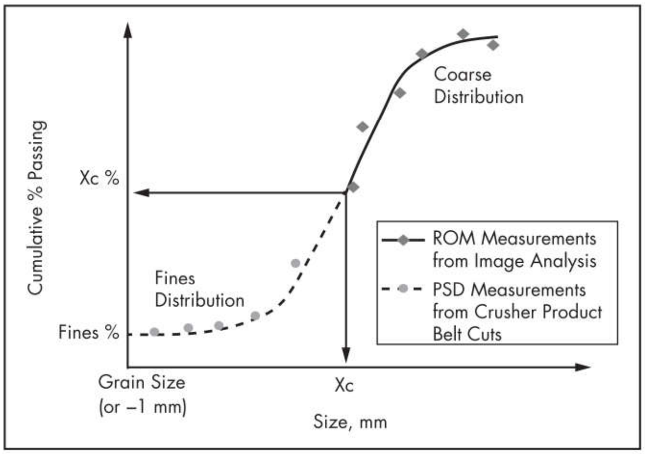


Figure 9 Fines and coarse size distribution in ROM

The X_c% is where the coarse and fine curves join, and initially this was chosen as the mean size or X_{50} (Figure 9). However, subsequent trials indicated better results with a characteristic size of X_{90} with very soft rocks (uniaxial compressive strength [UCS] < 10 MPa). It is likely that for intermediate-strength rocks, the point where the two distributions are joined will vary between the X_{50} and X_{90} values. It is therefore necessary to calibrate the CZM with the measured PSDs. The coarse end of the size distribution is calibrated with the ROM measurements obtained from the muck-pile images or at the back of trucks. Primary crushers are not expected to produce many fines (–25 mm); hence, most of these fines in the primary crusher product or SAG feed are assumed to have come from ROM ore. That is why the fine end fragmentation is calibrated to match the size distributions from the primary crusher product or SAG feed belt cuts. Once the model is calibrated for a given ore, then it is useful to run several what-if simulations to predict the entire ROMPSD.

An example comparison between the JKMRC CZM and Kuz–Ram model is provided at the end of this chapter.

KCO Model

The KCO model (Ouchterlony 2005) is an extended version of Kuz–Ram model and it essentially replaces the Rosin–Rammler function used in Kuz–Ram with a new function, the Swebrec function, as shown in the following equations:

$$P(x) = \frac{1}{\left\{1 + \left[\dfrac{\ln\left(\dfrac{x_{max}}{x}\right)}{\ln\left(\dfrac{x_{max}}{x_{50}}\right)}\right]^b\right\}} \quad \textbf{(EQ 10)}$$

$$b = \left[2 \cdot \ln 2 \cdot \ln\left(\frac{x_{max}}{x_{50}}\right)\right] \cdot n \quad \textbf{(EQ 11)}$$

where

$P(x)$ = percent of material passing size (x)
x_{max} = maximum fragment size (top in situ block size, spacing, or burden)
x = fragment size
x_{50} = mean (50%) passing size (same as Kuz–Ram)
b = fragmentation curve undulation parameter
n = uniformity index (relates to the slope of the distribution)

Limitations of Empirical Blast Fragmentation Models

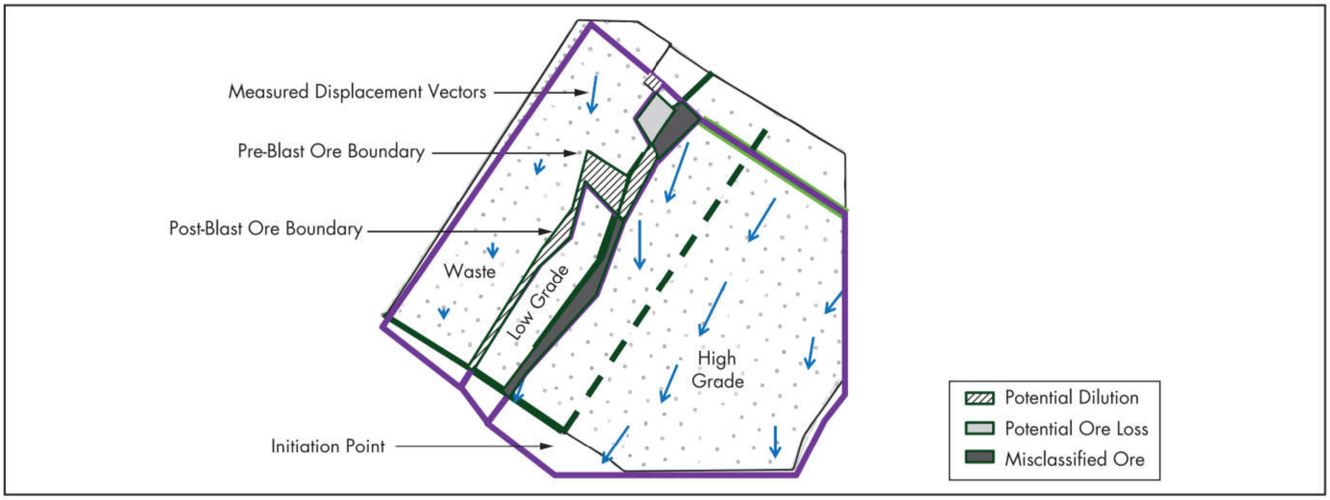
The following list describes the limitations of empirical blast fragmentation models:

- They require field calibration before they are used for any design purposes.
- They are single-hole models and the rock mass around the blasthole within a bench is considered one material. The impact of any variation in the rock properties across the blasthole is not considered.
- They do not take any blasting mechanisms explicitly. Only the CZM uses an analytical equation to estimate the crushed zone and fines around the blasthole.
- The input variables (e.g., rock mass properties and blast design parameters) used in these models has been assumed to be constant and uniformly distributed within a blast, whereas in reality, these parameters are not uniformly distributed.
- The impact of delay timing between the blasthole is not considered.

Blast Movement Models

Various modeling approaches have been developed to estimate blast movement; however, they have had very limited use in the context of production grade control. Yang et al. (1989) developed a two-dimensional (2-D) kinematic model to predict final muck-pile shape for bench blasts with free faces. Preece (1994) utilized advances in computing power to develop a 2-D model that represented the rock mass as discs. The explosive charge in the blasthole acts upon these discs and their trajectory, and final position defines the final muck pile. Tucker et al. 2006 used the particle flow code, a discrete element model to predict blast movement and muck-pile shape. Use of blast movement models in the production environment is limited because of their need for significant computational power and complexity. Production-grade control typically requires the use of simple and effective techniques that can be implemented on a day-to-day basis without impacting the time constraints imposed by the production environment. An alternative to computationally intensive numerical modeling is a relatively simple movement template approach (Rogers et al. 2012). This approach consists of translating grade boundaries based on average movement vectors measured from previous displacements of blasts with similar design properties (blast type, rock type, initiation type; Figure 10).

W.H. Bryan Mining and Geology Research Centre (BRC) of the University of Queensland has been involved in the development of a three-dimensional (3-D) blast movement model that uses an external "physics engine" (Tordoir 2009). In this model, the blast volume is discretized into equidimensional cubes and are assigned initial conditions based on the energy distribution and detonation times of surrounding explosive decks. Initial velocity vectors and first movement time are calculated for each block. These blocks have only a notional mass, and changing block size does not have a logical impact of the model behavior. The cubes are then moved using a physics engine, which handles all inter-cube collisions and landing logic. The explosive energy in each block is converted to kinetic energy and an energy loss factor. An energy loss factor scales the explosive energy in the blast to match the proportion of work it does in moving the rock. Each block can be assigned with specific attributes such as grade, mineralogy, or hardness, and their post-blast location can be tracked.



Source: Rogers et al. 2012

Figure 10 Template to adjust ore boundaries for blast movement

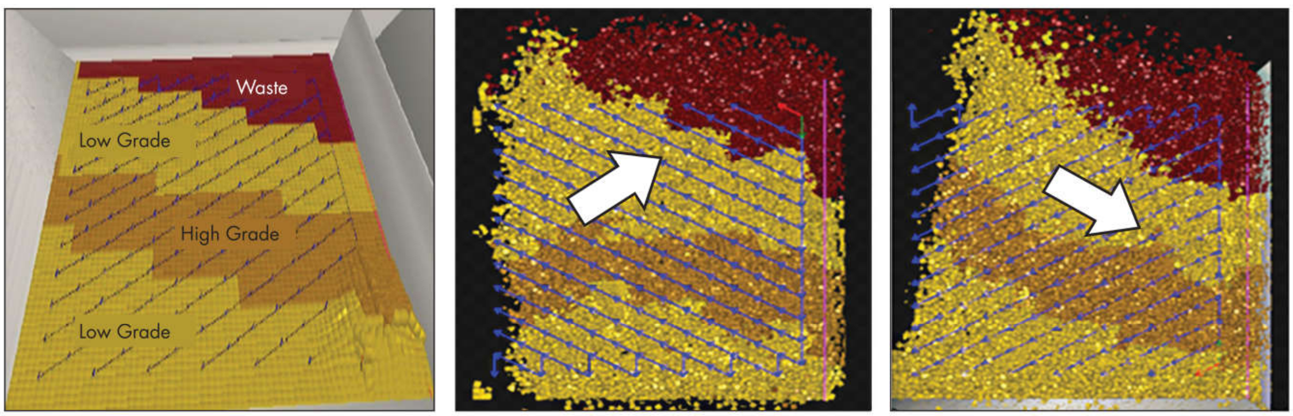


Figure 11 Blast movement with different timing from the BRC model

Figure 11 is an example of ore block movements simulated using the BRC model with different delay configurations. The simulation results clearly show that promoting rock movement parallel to the strike of ore body results in less disruption to the boundaries compared to movement perpendicular the strike of the ore body, which is consistent with field observations.

BLAST MEASUREMENT—HOW TO MEASURE BLAST OUTCOMES

To quantify the impact of blast outcomes on downstream processes and then to sustain the benefits, it is very important to measure the blast outcomes (PSD, blast movements, dilution, ore loss, damage, etc.) in an accurate and timely manner.

Fragmentation Measurement Techniques

Application of digital image processing (DIP) techniques to measure and/or assess fragmentation has become popular in the last few decades. The two primary types of DIP systems are desktop and online. The desktop systems are those that involve the collection of images from a blasted muck pile. The particles in these images are delineated and their sizes are measured using image analysis software. In desktop systems, there can be manual intervention to delineate the particles. In online systems, images are captured automatically at regular intervals and PSD is measured automatically. The online systems can be installed at the crusher feed, moving conveyors, shovel bucket, and so forth, where real-time fragmentation analysis is desired.

One of the main drawbacks of current image analysis systems is their inability to accurately measure fines because fine particles are not always present on the surface of the material on a truckload or conveyor belt because of segregation. Figure 12 shows the plan and cross section of particles on a belt conveyor where it can be clearly seen that most of the fines are at the bottom of the pile. Since the images are generally taken on the top, the fine particles are not seen on the image, and hence not accounted for or measured. Secondly, even if the fine particles were visible on the surface, the individual fragments are too small to be delineated properly because of resolution limitations of the imaging systems.

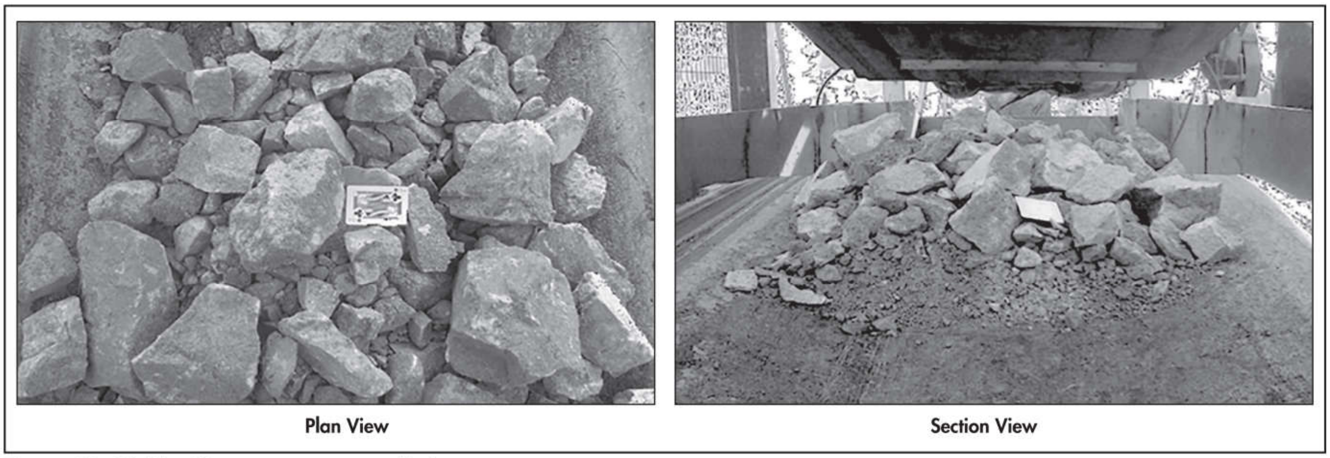


Figure 12 Hidden fines on a conveyor belt

Noy 2006 suggested using 3-D imaging to improve the accuracy of particle delineation. Thurley (2009) developed a system using lasers and demonstrated that a laser system is more accurate than 2-D image analysis systems. However, even these systems have the same limitation in assessing hidden fines.

Blast Movement Measurement Techniques

Drill-and-blast engineers have been using several methods to measure rock movement during blasting. These methods can be categorized into direct and indirect techniques. Indirect techniques such as high-speed video analysis and mobile radar systems measure rock mass movement (displacement, velocity, and shape) of muck-pile surfaces during blasting. The movement data from the muck-pile surface is used to estimate or infer the subsurface movement inside the exposed surface.

The need to measure rock movements beyond the exposed surface led to the development of several direct methods, such as passive markers (PVC pipes, colored stemming) and interactive remote transmitters (e.g., radioactive, magnetic, and electronic). The simplest direct method consists of marking out or delineating the areas of interest on the bench's horizontal surface with paint, chalk, or flagging tape (Davis et al. 1989; Morely and McBride 1995). Excessive heave, cratering, or stemming ejection could damage these markers and make them untraceable, and they only measure surface movement. The second method consists of installing passive visual markers such as polyethylene pipe, chains, wooden stakes, or colored stemming into specially drilled holes or into the stemming regions of the blastholes such that they protrude from the surface (Davis et al. 1989; Morely and McBride 1995; Scott et al. 1996; Zhang 1994). This method requires surveying of pre- and post-blast locations to estimate the rock movement and thus can disrupt production.

The next generation of markers are those that involve remote detection methods. They include radioactive markers and magnetic markers (Gaidukov 1970; Harris et al. 2001; Gilbride 1995). Usage of radioactive markers is restricted because of health and safety concerns, whereas magnetic markers have limited detection range and poor recovery.

The latest measurement systems involve Blast Movement Monitors (BMMs) developed at the JKMRC at the University of Queensland. The BMM is an electronic target that can be programmed at different frequencies and placed in different areas of the blast at variable depths. After the blast, it is necessary to walk over the muck pile with a radio frequency detector and global positioning system (GPS) to establish post-blast locations of the BMMs (Thornton et al. 2005).

The post-blast depth of the monitor is calculated from its signal strength, and its location is measured using conventional surveying techniques of a GPS. Using this information and pre-blast coordinate data, 3-D movement vectors can be calculated prior to muck-pile excavation (Figure 13).

BLAST OPTIMIZATION

Although blasting is the first step in the comminution and separation process of any mining operation, traditionally the blasting process is grouped with the mining processes and optimized to minimize the total mining costs. In this approach, blast results are considered good when they ensure good digging and loading operations while maintaining safety and environmental standards. That is why graphs (shown in Figure 14) are often used to estimate the optimum blasting effort (McKenzie 1965; Dinis da Gama 1990; Eloranta 1995).

The aforementioned approach may be appropriate if the sole purpose of the operation is to move the in situ rock mass from one place to another. In this traditional approach, the mine and concentrator are managed as separate cost center "silos" and each cost center is optimized independently, with little understanding of the real effect on downstream processes and overall efficiency. However, in the case of most metal mining operations, the in situ ore is separated from the waste rock and is subjected to a series of breakage and separation processes to convert it into a valuable product. Profitability in the mineral industry is the difference between the price of the final product when sold and the costs to produce it and can be estimated as

$$\text{profit} = \text{revenue} - \text{total operating cost} - \text{fixed cost}$$

where

$$\text{revenue} = (\text{grade} \times \text{recovery} \times \text{price}) / (1 + \text{dilution})$$

total operating cost = cost of drilling, blasting, loading, hauling, crushing, grinding, and liberation

fixed cost = capital and overhead cost per annum / total production per annum

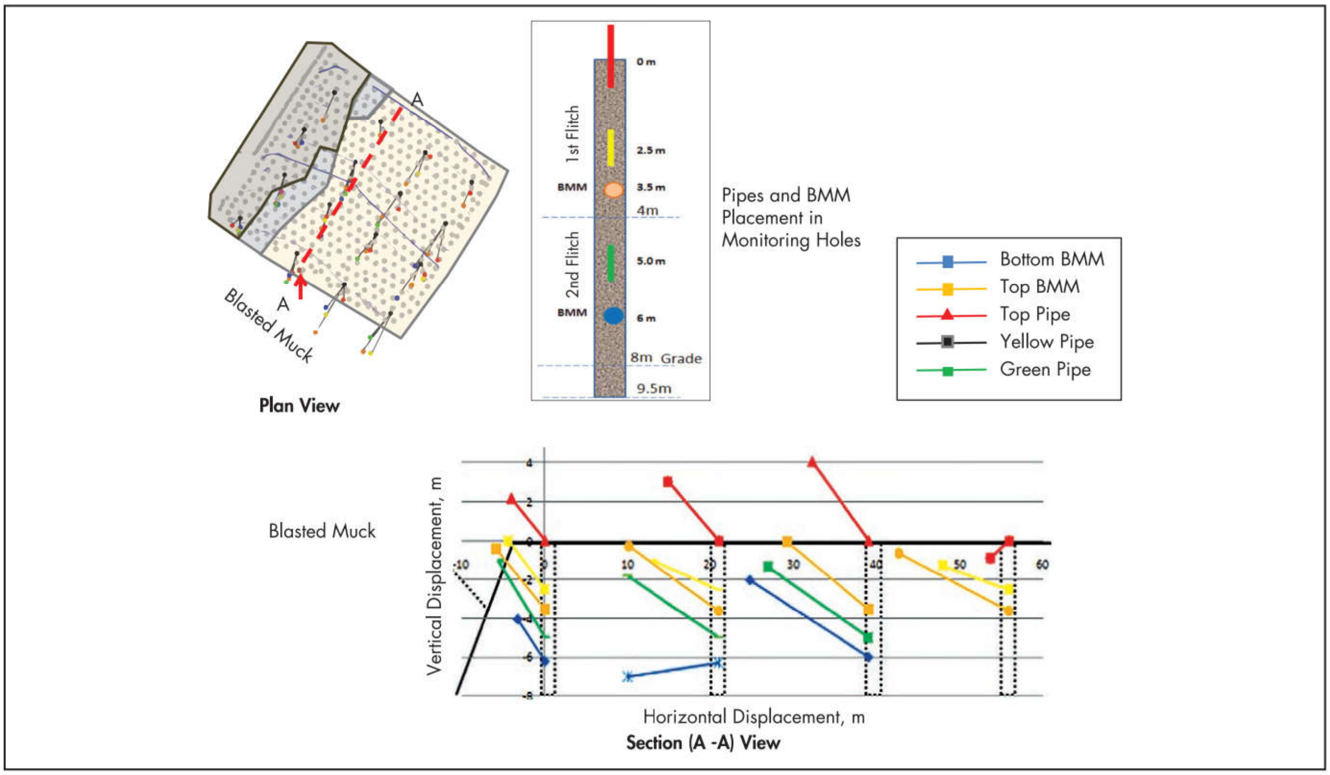


Figure 13 Blast movement measurements in different portions of a blast

In most cases, fixed costs are a significant portion of overall costs, and the increase in throughput without additional capital will reduce the fixed cost per ton of final product. Since blasting cost is an operational cost, any increase in final product throughput should be weighed against the decrease in unit fixed costs. In cases where there is demand for the final product, it is important to consider this aspect. Finally, in mine-to-mill blast optimization, the optimum blasting effort is where overall profit per ton of ore is maximized. In this approach, the aim is to leverage blasting outcomes to maximize the overall profits of operations rather than reduce the unit mining cost or unit operating cost.

Figure 15 shows approximate unit costs of key processes within the mine-to-mill value chain for a large open pit base metal operation treating hard ore. In this operation, a 100% increase in drilling-and-blasting cost can be compensated by a 4% to 5% reduction in downstream grinding costs. Since drill-and-blast is the first step in comminution and separation, its impact will be felt in all the downstream processes.

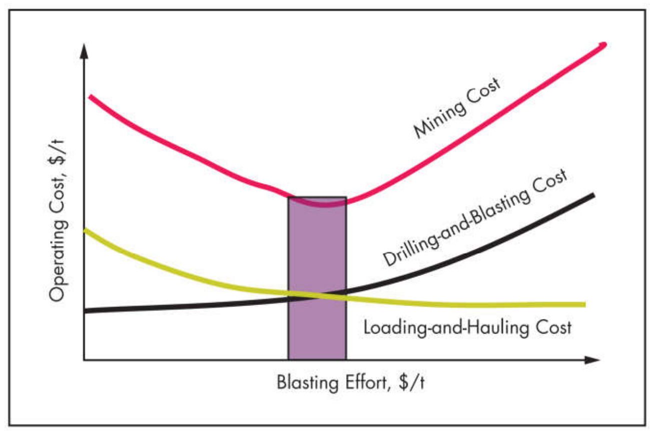


Figure 14 Traditional blast optimization

CASE STUDIES OF MINE-TO-MILL BLAST OPTIMIZATION PROJECTS

The theory of using blasting outcomes to improve the downstream processes is not new and has been implemented at several operations. However, most of the earlier case studies were limited to a large extent to improving the productivity of load and haul or the primary crushing process. During the AMIRA/JKMRC mine-to-mill research project P483A, *Optimization of Mine Fragmentation for Downstream Processing* (JKMRC 1998), the concept has been extended to understand the impact of blasting outcomes on grinding operations. The positive effect of finer feed size distribution on mill throughput, especially on SAG and AG mills, has been demonstrated at several mining operations around the world: Highland Valley Copper in Canada; Minera Alumbrera in Argentina; Cadia Hill, Kalgoorlie Consolidated Gold Mines, and Ernest Henry in Australia; Batu Hijau in Indonesia; Porgera gold mine in Papua New Guinea; and Candelaria in Chile. Most of these operations include conventional SAG-mill/ball-mill circuits and to a large extent were SAG-mill constrained. In all these operations, the blast designs were modified with higher energy, better energy distribution, and better confinement to produce finer fragmentation. Finer blast fragmentation coupled with tighter primary crusher operation produced much finer feed size distribution to the SAG mill, resulting in a throughput increase of between 5% and 25%. In most of the cases,

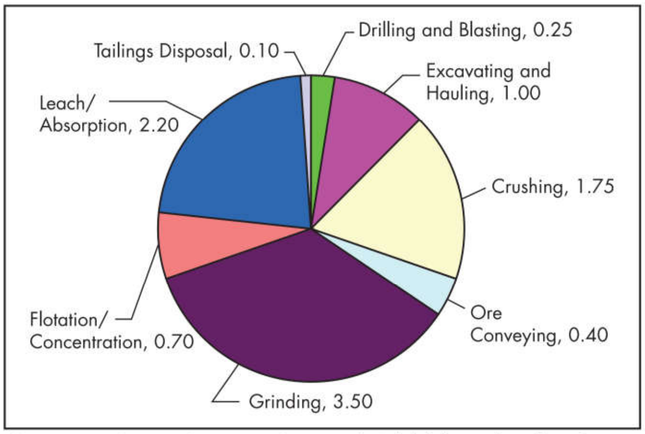


Figure 15 Approximate unit costs (US$/t) for a hard rock open-pit base metal operation

the ball mills have enough power to take care of additional throughput. In operations where there is a limitation of ball-mill power, either the throughput was decreased or the grind size was coarsened, whichever resulted in maximum metal production. A list of operations where finer ROMPSD from blast optimization improved grinding circuit performance is given in Table 2.

Case Study 1—Impact of Blast-Induced Dilution and Ore Loss (taken from Engmann et al. 2013)

Newmont's Ahafo operations in Ghana realized the impact of blasting outcomes, mainly the fragmentation and dilution, on the overall profitability of their operations and therefore implemented a blast optimization program to improve fragmentation and minimize blast-induced dilution. The blast optimization project was implemented in two phases. The first phase resulted in much finer fragmentation and allowed operation of the primary crusher with a much tighter gap. These changes resulted in much finer feed to the SAG mill, resulting in increased throughput in primary ores by up to 30% (Mwansa et al. 2010; Dance et al. 2011). In the second phase, a comprehensive blast movement monitoring program, as shown in Figure 16, was implemented to minimize blast-induced ore loss and dilution. A detailed description of this study is given by Rogers et al. (2012) and Engmann et al. (2013). Based on the understanding of blast movement dynamics from the study, alternative strategies were developed and implemented to reduce blast-induced ore loss and dilution. These strategies included

- Implementation of blast designs to promote consistent movement along the strike of the ore body and minimize inconsistent movement from edge effects, uneven free faces, and cratering, especially along the ore/waste boundaries; and
- Adjusting the post-blast ore boundaries to account for expected blast movement and to ensure that excavation follows the adjusted polygons. The blast movement vectors were estimated based on blast movement measurements.

The alternative strategies developed from this study have been incorporated in standard site operating procedures. Reconciliation data show improved agreement between the mine and mill grade and reduction in diluted tons since the implementation of the blast movement study in April 2011 (Engmann et al. 2013; Figure 16).

In addition to blast movement adjustments, several other improvements, such as increased blasthole sampling and pit supervision, have already been implemented. The geologists at Ahafo mine consider the biggest gains to have been made from post-blast movement adjustments. Post-blast polygon adjustments have enabled ore to be recovered correctly. This project has also improved focus on heave mining and excavation control to ensure that grade blocks are captured correctly in dispatch and the correct sampling techniques are being followed.

Case Study 2—Potential Impact of Microcracks on Mill Throughput (taken from Kanchibotla et al. 2015)

Paddington mill in Australia was expected to treat a significant portion of fresh ore from the Navajo Chief open pit, which was harder compared to the then blend of altered ore. This was expected to have a negative effect on mill performance; hence, a blast optimization project was implemented to improve the feed size distribution from fresh ores. The optimization involved advance blasting practices with changes to the blast energy, energy distribution, type of energy, and, more importantly, priming and delay timing. The advance blast designs resulted in finer fragmentation and a 36% increase in mill throughput. The different factors and their relative contributions in increased throughput are shown in Figure 17.

The results highlighted that the most significant contributor to the increase in throughput between the baseline and validation surveys was the change in ore hardness, followed by the increased blasting intensity (finer ROMPSD) and some improved circuit parameters. The drill penetration rates and the point load tests data indicated that the pre-blast hardness of ore (as measured by the drill penetration rates and point load tests) from the advanced blast designs was harder than the baseline blasts (Figure 18). However, the comminution properties measured as from Bond tests and JKMRC drop weight tests showed that the ore from the advanced blast designs was softer than that from the baseline blasts (Figure 19). The JKMRC drop weight test measures the impact breakage of ores by two parameters, A and b, and $A \times b$ is considered the impact hardness of ore (Napier-Munn et al. 1996).

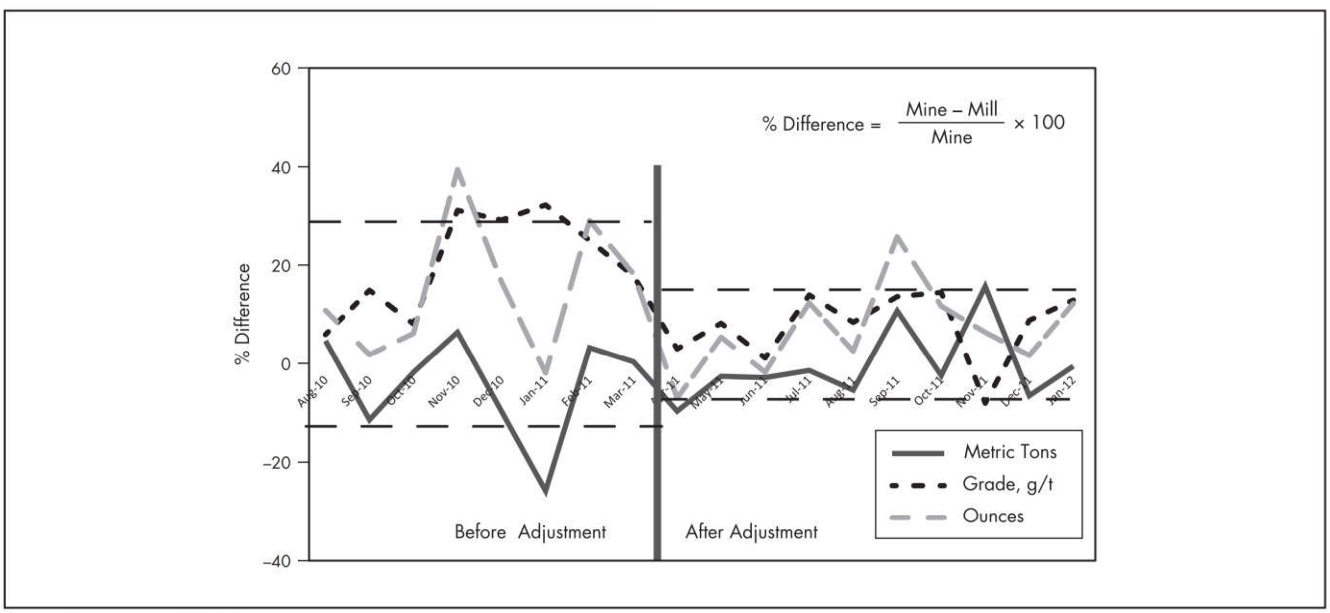
The study results suggest that the high-intensity advanced blasts with multiple primers and fast timing may have created more microcracks and apparently softened the ore. This phenomenon was observed in several laboratory blasting experiments and few controlled blasting trials (Revnivstev 1988; Chi et al. 1996; Nielsen and Kristiansen 1995; Kojovic and Wedmair 1995; Michaux 2005; Kim 2010; Parra 2013; Parra et al. 2016). This hypothesis needed to be further investigated; consequently, research is being conducted at the JKMRC in this area. However, if this hypothesis is proven to be correct and the advanced blasts can consistently soften the mill feed, it can have a significant impact not only on grinding circuit performance but also on recovery.

CHALLENGES IN IMPLEMENTATION

Even though the benefits of the mine-to-mill blast optimization methodology are obvious, there are many challenges to sustain the benefits in the long term. One of the key challenges is to convince one part of the value chain (e.g., blasting and mining) to increase their costs to realize the benefits at other parts (e.g., grinding) of the value chain. It requires monitoring

Table 2 Select mine-to-mill blast optimization case studies

Mine	Mine-to-Mill Program Description	Process	Reference
Highland Valley Copper, Canada	Blast optimization included pattern changes and stemming practices.	Demonstrated an increase in SAG mill throughput by controlling feed size.	Djordjevic 1999; Dance et al. 2011
Minera Alumbrera, Argentina	Blast designs were matched to ore types, and the primary crusher closed side was reduced.	Mill throughput increased by at least 13%. Increased blasting costs would have been offset by a 5% increase.	Valery 2000
Placer Dome's Porgera gold mine, Papua New Guinea	Blast designs were modified to produce a finer feed size distribution to the SAG mill.	Finer run-of-mine particle size distribution (ROMPSD) from the modified blast increased SAG mill throughput by up to 25%.	Grundstrom et al. 2001; Lam et al. 2001
Barrick/Newmont's Kalgoorlie Consolidated Gold Mines operation, Australia	Blast optimization and primary crusher operations were improved. Powder factor increased 29%.	Mill throughput increased by 13%.	Kanchibotla et al. 1998; Nelson et al. 1996
Cadia Hill, Australia	A finer feed size distribution was produced through modified blasting practices. SAG mill and pebble crushing operating parameters were modified.	Demonstrated an improvement in SAG mill throughput by up to 10% from finer feed size.	Kanchibotla et al. 1999; Hart et al. 2001
Antamina, Peru	Blast designs were modified in two interventions in harder ores to produce finer feed to the mill. The primary crusher and mill operating practices were modified to take advantage of finer feed.	Finer ROMPSD improved the primary crusher, and SAG mill throughput increased by more than 25%.	Rybinski et al. 2011; Kanchibotla and Valery 2010
Newmont's Ahafo gold mine, Ghana	Blasting practices and primary crushing operations were modified to produce finer mill feed.	SAG mill specific energy was reduced by 25% and the throughput in primary ores was increased by up to 30%.	Dance et al. 2011
Newmont's Ahafo gold mine, Ghana	Blast movement was monitored and a system was implemented to adjust ore blocks for blast movement.	Blast-induced dilution was reduced.	Engmann et al. 2013
PanAust's Ban Houayxai mine, Laos	Blasting practices in fresh ores resulted in lower mill throughputs. A blast optimization project was implemented that required changes in blast geometry and energy.	Changes in blasting practices resulted in finer mill feed and in mill throughputs by more than 40% than the design for fresh ores.	Symonds et al. 2015
Paddington gold operations, Australia	Blasting practices with advanced priming and timing designs were modified for harder ores to produce finer feed. This was coupled with changes in digging practices to reduce dilution and in the mill to take advantage of finer feed.	SAG mill throughput was increased by 35%, and energy consumption was reduced by 27%.	Kanchibotla et al. 2015

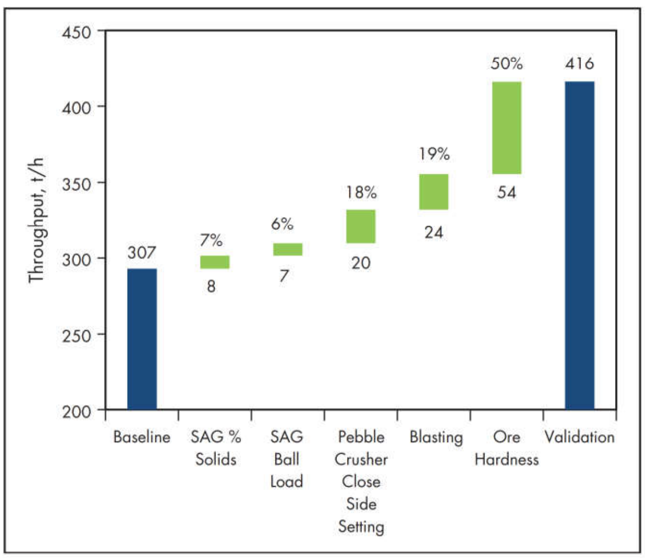


Source: Engmann et al. 2013

Figure 16 Grade reconciliation between mine and mill

of key performance indicators (KPIs) at each stage and effective communication between the key processes.

Another important challenge is to understand the characteristics of the ore body, its variations, and its response to key processes and then implement processes to improve the overall performance of the mine-to-mill value chain. In most operations, rock mass is characterized and tested by geologists, drill-and-blast engineers, and metallurgists separately for different purposes with very little communication between different groups. In a majority of these cases, available information is either late or inappropriate to use for operational decision making. For example, many mines collect blasthole chips and assay them to estimate the ore-grade boundaries, but the ore-grade boundary estimates from these data are not made available to drill-and-blast engineers before the blast. Therefore, reliable online techniques to measure the changes in the geology are very important for successful implementation of this approach. New technologies such as big data analytics, measure while drilling (MWD) techniques, and other geophysical testing methods can assist in this area. Further research is needed to understand the relationships between the MWD parameters and rock mass characteristics to develop process performance indices such as blastability, grindability, and separability using these data.



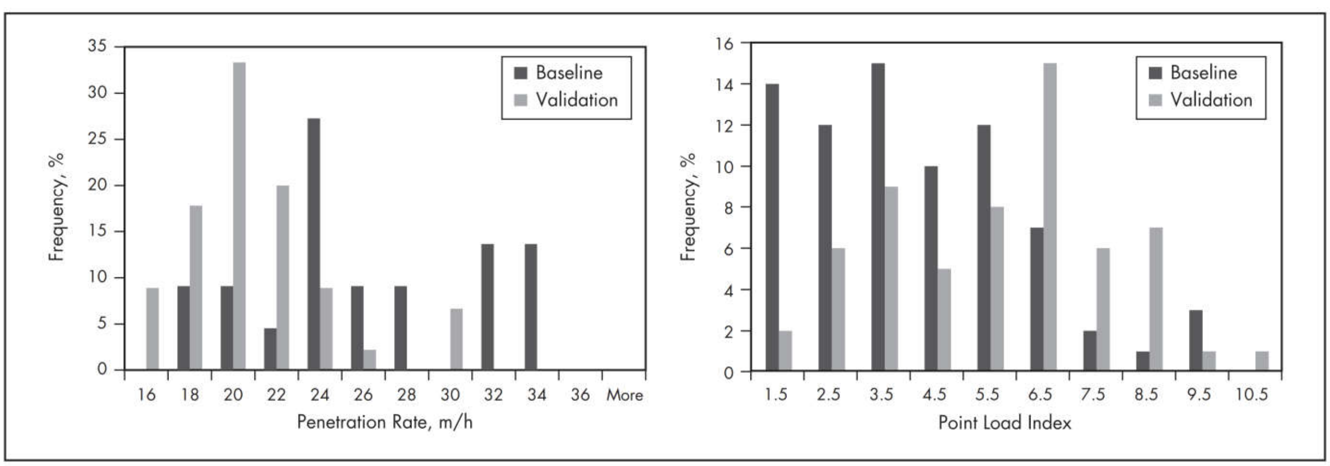
Source: Kanchibotla et al. 2015

Figure 17 Relative contribution of changes to throughput increase (estimates from simulation)

A good understanding of interactions between the key processes and the ability to leverage them to optimize the overall performance is another important element. Reliable models of key processes within the value chain would help not only to understand these interactions but also to control and optimize them. Current models represent only one or two processes within the value chain, and most of the current models represent only one rock/ore. These models cannot handle the variability even within each rock/ore characteristic. Stochastic approaches are needed to understand the impact of natural variability of rock mass on the process performance. Further work is also needed to integrate the models of all key processes in one platform so that the operations can simulate the key mine-to-mill processes and understand the interaction between them.

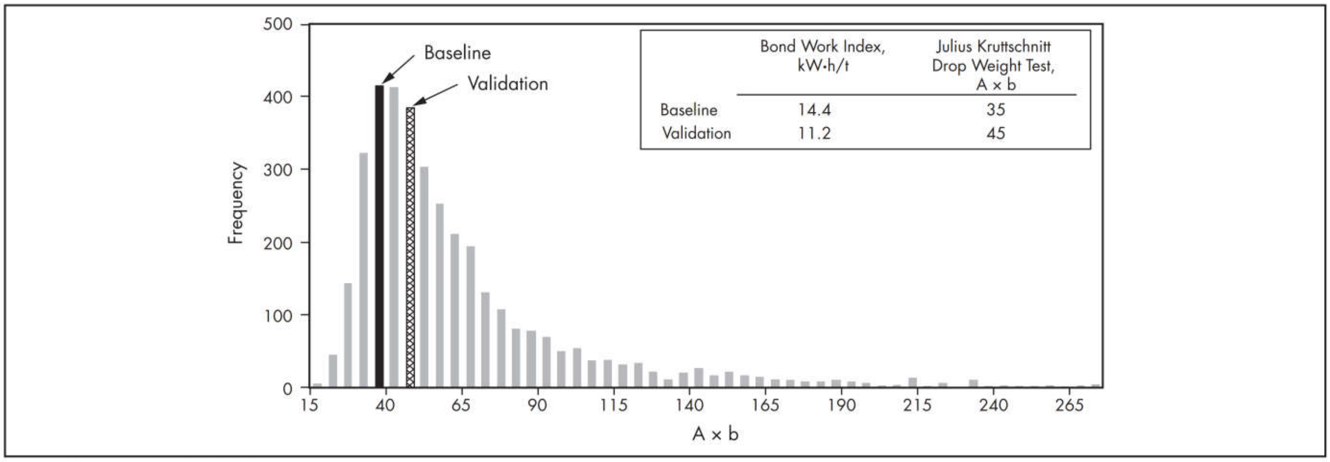
Establishment of appropriate KPIs and reliable systems to monitor these KPIs accurately and report them on a daily basis is another important element. For example, reliable measurement of ROMPSD, especially fines, is important to quantify the impact of fragmentation, but the current image analysis systems have limitations in this regard. It is therefore important to develop reliable measurement systems that can measure the relevant KPIs in a timely and accurate manner so that the operations can use them to not only detect the changes but to take appropriate actions to mitigate the negative effects. Data from these online measurements can be used to calibrate the process models and then use them for day-to-day process control as well as optimization purposes. A schematic view of such an approach is shown in Figure 20.

It is important that the new methodology, designs, and operating procedures are embedded in the day-to-day standard operating procedures and all key site personnel are well trained to implement them. The measurement systems and/or models need to be embedded in the operational work flow rather than stand-alone separate systems so that the operators are familiar with them and can use them routinely for diagnostic as well as optimization purposes.



Source: Kanchibotla et al. 2015

Figure 18 Drill penetration rates and point load strength of rock from standard and advanced blasts



Source: Kanchibotla et al. 2015

Figure 19 Comminution properties of ore from standard and advanced blasts

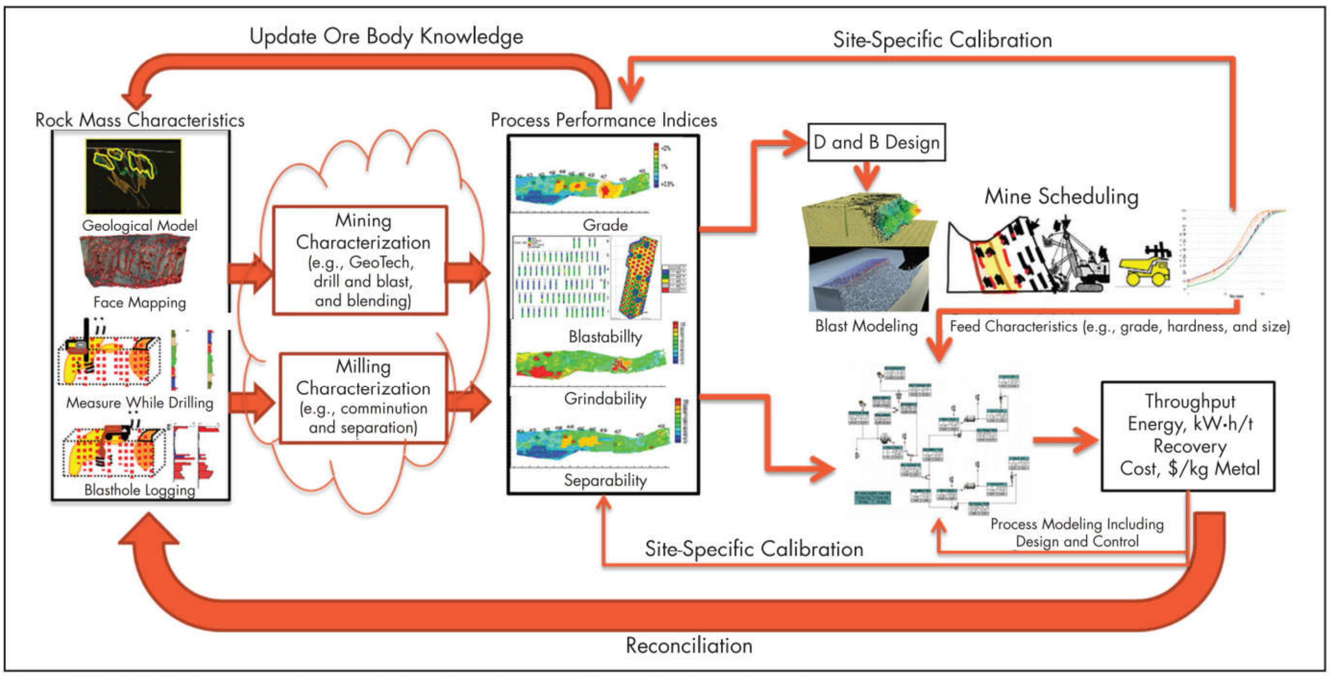


Figure 20 A holistic approach to optimize the mine-to-mill production value chain

CONCLUSIONS

The traditional approach of optimizing mining and milling operations separately can fail to recognize the potential benefits that can otherwise be achieved by optimizing the entire process. All the processes in the mine-to-mill production value chain are interdependent, and the results of the upstream mining processes can have a significant impact on the efficiency of downstream processes. A new holistic approach is needed wherein mining activities are designed to supply the mill with ore characteristics to increase the overall energy efficiency and productivity of the total production value chain rather than an individual unit process without adversely affecting safety, slope stability, and environmental risks. Blasting plays an important role in this approach, because it is the first step in comminution and separation processes within the mine-to-mill value chain.

Several case studies demonstrate the value of mining and concentrator personnel working together to reduce the overall cost of an operation rather than reducing their individual operation costs. The benefits derived from designing blasts to improve the efficiency of crushing and grinding stages far outweigh the additional blasting costs. Ore softening and grade increase in finer fractions with advanced blasting practices can further increase the leverage of blasting on the production value chain. Possible negative side effects, such as ore loss, dilution, and blast damage, can be controlled with improved understanding of the blasting process based on systematic monitoring. Effective communication between the key process areas and reliable tools to measure the KPIs in a timely manner are essential for successful implementation of this approach. A good understanding of ore-body characteristics and the ability to model their impacts on the performance of key processes in the value chain is important to sustain the benefits.

COMPARISON OF FRAGMENTATION ESTIMATES FROM THE KUZ–RAM AND JKMRC CRUSHED ZONE MODELS (adapted from Kanchibotla et al. 1999)

A comparison of fragmentation estimates between the Kuz–Ram model and JKMRC CZM is shown using the data from a blast trial conducted at Cadia Hill, a large copper–gold open pit mine in Australia. The rock mass properties, blast design parameters, and explosive properties for this blast are given in Table 3. ROM size distribution was estimated from 10 muck-pile images and 18 images taken from the back of trucks using Split image analysis software (Split Engineering 2017). Primary crusher product size distribution was obtained by taking a belt cut.

Calculations of the Kuz–Ram Model

The following calculations pertain to the Kuz–Ram model:

- Rock factor (A) is estimated as 10, because the rock is considered to be hard and fractured.
- Powder factor (K) = explosive weight / (burden × spacing × bench height) = 593 / (6 × 7 × 15) = 0.94 kg/m^3
- Explosive charge weight (Q) = 593 kg
- Energy factor or relative weight strength (RWS) = 80
- Uniformity index (n) according to Equation 3 = 1.52
- Particle size distribution:
  - Mean fragment size (X_{50}) or 50% passing according to Equation 1 = 0.38 m
  - Cumulative % passing 1 mm or 0.001 m consistent with Equation 2 = 0%
  - Cumulative % passing 10 mm or 0.01 m according to Equation 2 = 0.3%
  - Cumulative % passing 100 mm or 0.1 m consistent with Equation 2 = 8.5%
  - Cumulative % passing 1,000 mm or 1 m according to Equation 2 = 95%

Calculations of the JKMRC Crushed Zone Model

The following calculations pertain to the JKMRC CZM:

- Rock factor (A) is similar to Lilly's blastability index shown in Equation 5 with some modifications:
  - Rock mass description (RMD) is estimated as function of in situ block size (I_{50}). In Figure 21, with an I_{50} of 1.4 m, RMD is estimated to be 50.
  - The joint factor (JF) is based on the assumption that rock is easier to blast if it has many fractures or no fractures in between two holes and is estimated as a function of the number of joints between blastholes, as shown in Figure 22. In this example, with an I_{50} of 1.4 m, the number of joints between two holes is estimated as five and the JF as 33.
  - RDI (rock density influence) = 25*(rock density – 2) = 25 × (2.6 – 2) = 15
  - HF (hardness factor) = Young's modulus (YM) if YM < 50 GPa
    = YM/3 if YM > 50 GPa
    = UCS (uniaxial compressive strength) /5 if 50 GPa > YM < 50 GPa
    = In this case, HF is 25.4 (UCS/4) because YM is 62 GPa
- Rock factor A according to Equation 5 = 0.6 × (50 + 33 + 15 + 25.4) = 7.4
- Powder factor and energy factor are the same as the Kuz–Ram model

Table 3 Rock properties, blast design parameters, and explosive properties for trial blast no. 760043

Rock Mass Properties	
Rock type	Monzonite
Density	2.60 t/m^3
Young's modulus	62.00 GPa
Uniaxial compressive strength	127.00 MPa
Rock quality designation	100
Mean in situ block size (I_{50})	1.4 m
Blast Design Parameters	
Burden	6 m
Spacing	7 m
Hole depth	16.5 m
Hole diameter	229 mm
Explosive weight	593 kg
Bench height	15 m
Stemming length	4.50 m
Subdrill	1.50 m
Explosive Properties	
Explosive	Titan 2,070 g
Density	1.2 g/cc
Velocity of detonation	4,950 m/s
Relative weight strength (ANFO* = 100)	80

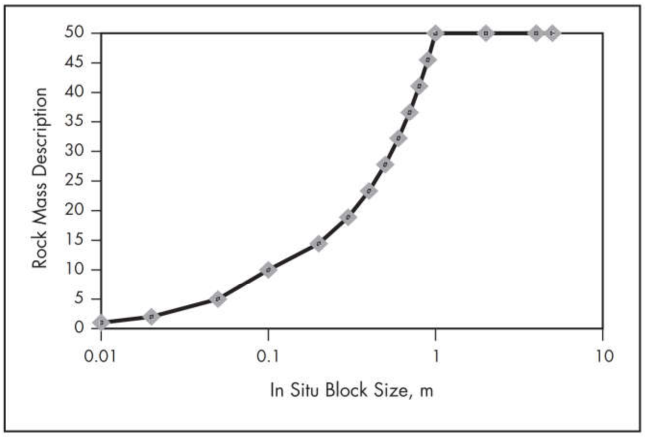
*ANFO = ammonium nitrate and fuel oil

- Mean fragment size (X_{50}) or 50% passing using Equation 1 = 0.28 m
- This model uses two uniformity indices.
  - Uniformity index for the coarse end is the same as the Kuz–Ram and in this example it is 1.52.
  - Uniformity index for the fines end is estimated using Equations 6 through 9.
    - Borehole pressure P_b = 0.25 × 1,200 × 4,000^2 = 7.3 GPa using Equation 7
    - Crushed zone radius according to Equation 6 = 0.87 m
    - Fines uniformity index according to Equations 8 and 9 with a grain size of 1 mm = 0.48
- Particle size distribution:
  - Mean fragment size (X_{50}) or 50% passing according to Equation 1 = 0.28 m
  - Cumulative % passing 1 mm (based on volume of crushed zone) = 4.5%
  - Cumulative % passing 10 mm or 0.01 m = 13%
  - Cumulative % passing 100 mm or 0.1 m = 34%
  - Cumulative % passing 1,000 mm or 1 m = 99%

A comparison of ROM predictions from the two models (Kuz–Ram and JKMRC CZM) with the fragmentation measurements from Split Engineering (2017) and crusher product belt cuts is shown in Figure 23.

The results showed good comparison between the Kuz–Ram estimate and Split measurements without fines correction, whereas the JKMRC CZM showed good comparison with crusher product belt cuts and Split measurements with fines correction. It is very well accepted that most image analysis systems underestimate fines.

The Kuz–Ram model estimates very little or no fines (–25 mm), whereas the crusher product belt-cut measurements



Source: JKMRC 1998

Figure 21 Relationship between RMD and in situ block size

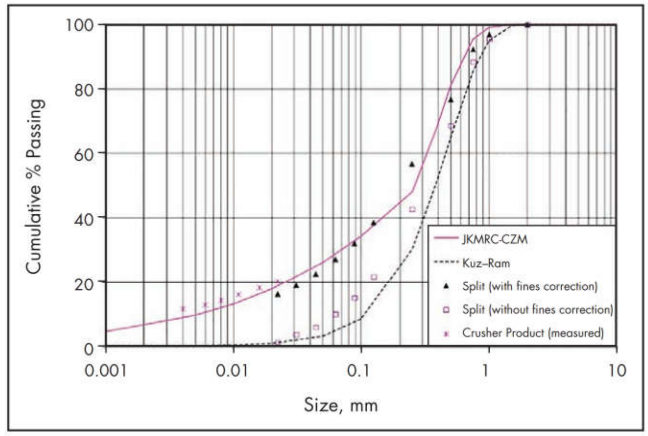
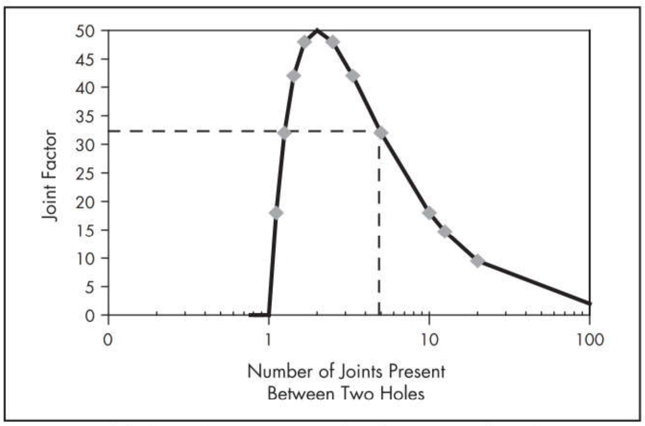


Figure 23 Comparison fragmentation estimates from the models and measurements



Source: Kanchibotla et al. 1999, reprinted with permission from the Australasian Institute of Mining and Metallurgy

Figure 22 Relationship between JF and number of joints between holes

clearly show around 20% of –25 mm particles. It is very well known that primary crushers do not generate much fines (–25 mm) and most of the fines in the primary crusher product would be coming from the blast. Therefore, it can be concluded that the conventional Kuz–Ram blast fragmentation model underestimates fines (–25 mm) compared to the JKMRC CZM.

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