Blasting for Mine to Mill Optimisation

by

Andrew Scott	Scott Mine Consulting Services Pty Limited
Dr. Sarma Kanchibotla	Julius Kruttschnitt Mineral Research Centre
Dr. Steve Morrell	Julius Kruttschnitt Mineral Research Centre

Abstract

In recent years there has been a growing recognition of the impact that mining practices have on the efficiency of mineral processing operations. Head grade, dilution, particle size distribution and their variability are known to have a significant effect on the throughput and recovery achieved in mineral processing plants. Improved process monitoring and modeling have provided the opportunity to identify and quantify some of these impacts and to define very significant incentives to optimise the overall operation from the mining face through subsequent comminution and processing.

The Julius Kruttschnitt Mineral Research Centre has been undertaking research to optimise "Mine to Mill" performance through a major AMIRA research project and through individual research contracts with mining companies. This work has resulted in a broad range of field and laboratory experience in terms of both the challenges and the opportunities encountered when tailoring blasting operations to suit the overall "Mine to Mill" production chain.

This paper describes the objectives of "Mine to Mill" blasting to optimise the system of rock breakage involved in the blasting, crushing and SAG milling of hard, brittle ores. The design of such blasts falls outside the conventional envelope of experience for most engineers and requires the consideration of some critical factors if the objectives of such blast are to be met and some of the potential problems are to be avoided. Experience to date has also found that the quality of implementation is critical to the success of "Mine to Mill" blasts.

Introduction

The concept of 'doing more breakage in the pit' to minimise the overall cost of a mining operation is often discussed when mine blaster's come together. It is one of those 'intuitive truths' that we know is right, but we don't know how to prove – particularly as it requires deliberately increasing the blasting costs that we have fought so hard to reduce over the years.

Graphs such as that shown on Figure 1 have often been published. Examples include a paper by A.S. McKenzie in 1966, Dinis da Gama 1990, Hunter et al 1990, Cunningham 1990, Eloranta 1995 and Bellairs 1998. The graph shows how reducing blasting costs (increasing fragmentation) increases the cost of excavation. Conversely, increasing blasting costs, reduces the cost of excavation, at least within sensible limits. Kanchibotla (1998) examines this concept in greater detail with cost curves that are sensitive to the nature of the downstream process that is being affected by blasting.

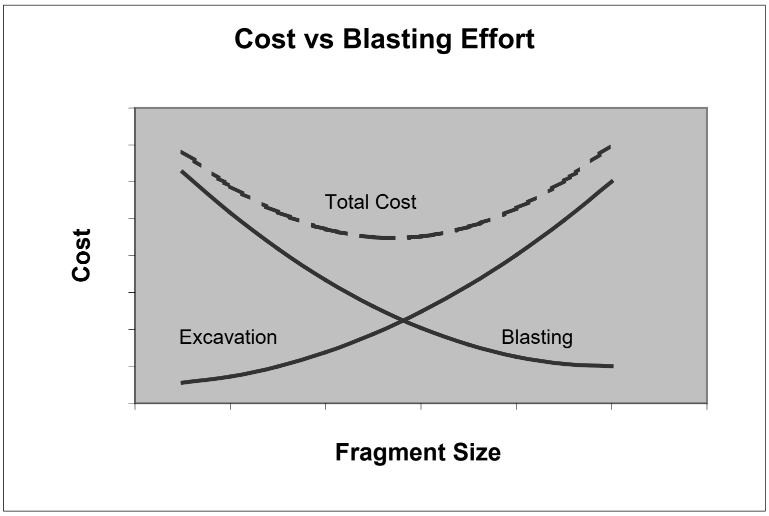

Figure 1 Blasting Effort versus Blasting and Excavation Costs (after Dinis da Gama (1990)

The approach represented by these graphs forms a sub-set of a broader concept where an operation consisting of several components (e.g. mining, crushing and milling) is optimised overall, rather than optimising each operation independently and then just linking them together. If optimisation is regarded as maximising net revenue to the project, then minimising costs is just one of the possible approaches to achieving this objective.

Such a “Mine to Mill” approach has been the focus of a considerable amount of work by the Julius Kruttschnitt Mineral Research Centre in recent years where the following optimisation challenges have been tackled:

- The minimisation of fines in the mining and handling of iron ore
- The management of fines in blasting and crushing for a copper heap leach operation
- The control of blast damage, mining recovery and dilution in open cut coal operations
- The optimisation of mine fragmentation to increase mill throughput in gold and copper operations.

The latter topic has been trialed at a number of mines and is referred to as “Mine to Mill’ fragmentation. This paper discusses issues associated with the design of blasts to meet the requirements of ‘Mine to Mill” fragmentation to increase the throughput of semi autogenous (SAG) mills.

“Mine to Mill Fragmentation”

In classic metalliferous operations the ore undergoes at least three stages of breakage:

- **blasting** to prepare the ore for excavation and transport
- **crushing** to improve its handling characteristics and to prepare the ore for grinding
- **grinding** which is usually undertaken in two stages.

In large open pit mines, ‘rule of thumb’ figures suggest the relationship between the energy requirements and cost for these three stages of breakage are as shown on Table 1.

Table 1 Relative Energy and Cost of Breakage

	Energy	Factor	Cost	Factor
Blasting	0.2 kwh/t	1	$0.15/t	1
Crushing	2 kwh/t	10	$0.75/t	5
Grinding	20 kwh/t	100	$3.75/t	25

In the last 20 years grinding circuit design has undergone considerable change and has seen the rise in popularity of autogenous (AG) and semi-autogenous (SAG) mills to the point where it is rare to find a comminution circuit where they are not used. This trend has enabled unit throughput size to increase enormously, with consequent economies of scale. The downside of these machines is that they are relatively sensitive to the size distribution of the feed, particularly AG mills. This is because the feed provides the media which is responsible for effecting grinding as well as providing the feedstock which is ground. Therefore the feed size distribution has to provide a balance between the amount and size of the media (coarse fraction) and the feedstock (fine fraction). Of course the rock media as it wears in the mill also provides a source of ground product in its own right, but this is a relatively slow process and so coarser rocks tend to accumulate in the mill. Hence if a feed size distribution is produced with too much coarse material/very large top size the mill will tend to overfill with pebble size rocks (25mm-75mm) and limit throughput. If too little coarse material is fed there will be insufficient grinding media to grind the feedstock to the required size. This size is dictated by the size of the apertures in the mill grate and if material cannot pass through it the mill will also tend to overfill with pebble/gravel size material. In both scenarios pebble sized material is found to be the problem and this fraction is often referred to as critical size. Hence it is also found that feeding a size distribution which is rich in pebbles will also limit throughput. It follows from these experiences that the ideal feed size distribution is one which has relatively little pebble size material, does not have excessive coarse material and has as much material in the below-grate size region (10mm-15mm) as possible.

The “Mine to Mill” concept involves increasing the amount of breakage achieved in both blasting and crushing in order to relieve the mill of as much new breakage as possible. In essence the breakage is moved back down the production chain where the energy requirements (and the costs) are lower. Figure 1 shows the concept.

The first objective of “Mine to Mill” blasting is to reduce the top size of the material to improve the ease of excavation and transport within the mine. Reduced top size also allows the primary crusher gap to be reduced, generating material which needs less breakage in the mill. With reduced top size and continuous material supply the crusher can be choke fed without the risk of blockages. Choke feeding allows more inter particle breakage and therefore produces more fines in the crusher product than without choke feeding. The top of the cumulative fragment size distribution curve shown in Figure 1 is moved to the left.

The second objective is to increase the proportion of fines (material less than the grate size in the mill, say 10 mm) because this should pass freely through the mill and require no further breakage. This is represented by lifting the lower end of the size distribution curve. These changes steepen the curve, reducing the proportion of material in the ‘critical’ size range of around 25 mm to 75 mm.

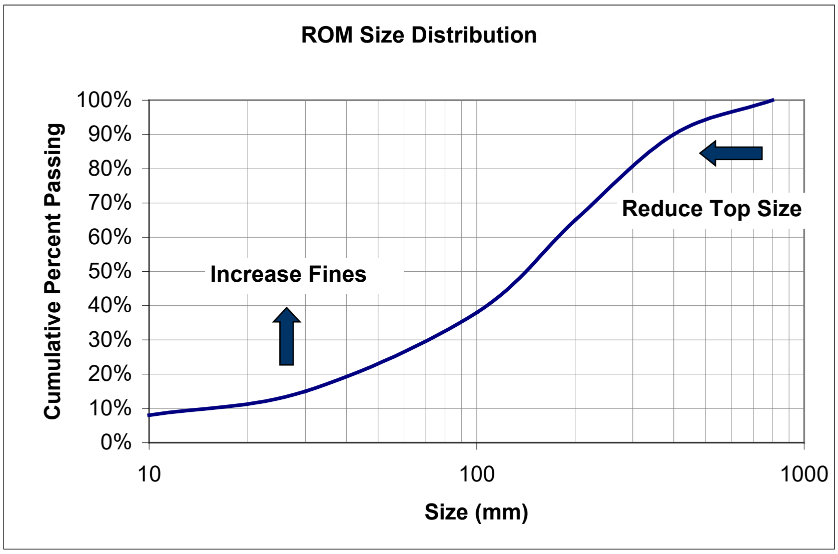

Figure 1 Changes in Size Distribution Sought Through Blasting

An example of the impact of more intense blasting and crushing on the size distribution of the feed to a mill is shown on Figure 2. Modified blasting practices led to a reduction in the top size of the feed to the crusher. This eased the work load of the crusher allowing the closed side setting to be reduced without any reduction in capacity. The crusher then further reduced the top size fed to the mill. The blast and crusher work together as a system to prepare material for the mill – not as independent processes.

The primary crusher adds few fines, so it is up to the blast to increase the proportion of material requiring little or no breakage in the mill. This is seen as the step to the left of the lower end of the curve from 'conventional' to 'modified' blasting on Figure 2. These changes to the size distribution of the mill feed might generate a 10% increase in mill throughput. Alternative operating strategies could see the mill throughput maintained, but a significant reduction achieved in mill power and hence milling cost.

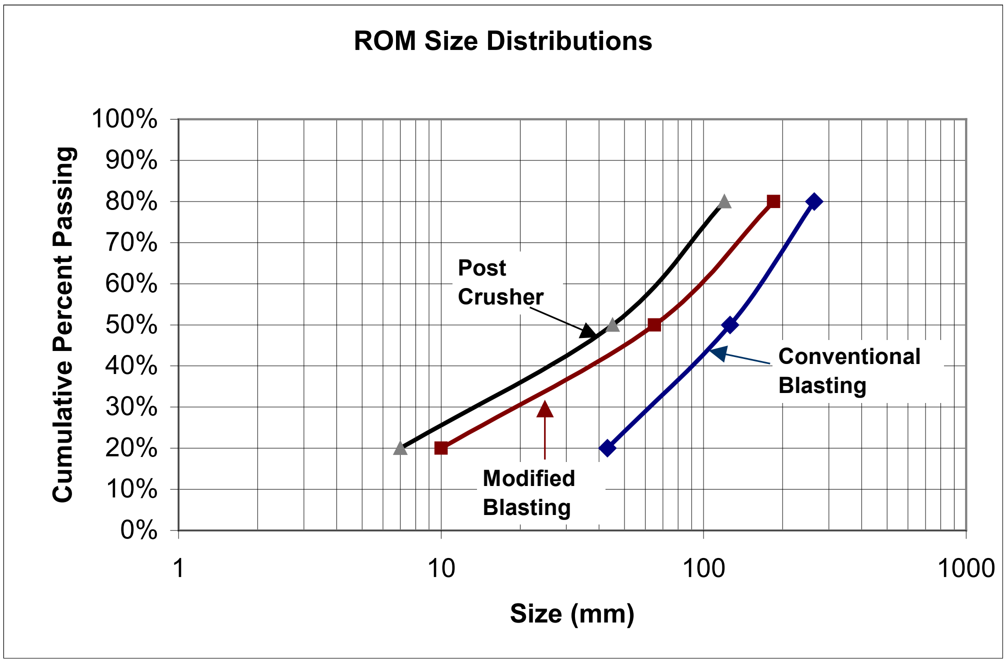

Figure 2 Impact of Increased Blasting and Crushing Effort

The impact of this change can be simply evaluated. If the additional blasting cost was $0.10/t and mill throughput increased by 10%, then according to the figures on Table 1, the average milling cost would fall by $0.38, or four times the extra investment made in the blast. Of course this simple approach ignores many factors including likely benefits to the mining operation itself from higher excavation productivity, higher availability and reduced maintenance costs for equipment. If the increased revenue generated from additional sales made possible by the higher production rates is factored into the equation, the rewards can become very significant.

Highland Valley Copper is the best example that has been published (Simkus and Dance 1998). At Highland Valley the mine introduced larger blast holes in order to reduce the unit cost of drilling. The resulting muck could still be excavated and hauled to the in-pit crusher but milling rates fell by 10%. The cause was not immediately obvious, but when it was understood to have been the result of a change in blasting practices it was decided to explore how blasting might be used to improve mill performance over and above the previous levels. A further increase of 7.5% was achieved through fairly modest blasting changes.

The President of Highland Valley Copper, Mr. David Johnson, told the AusIMM Mine to Mill Conference in Brisbane in October 1998 that this turn around was worth $US35 million per year to the operation. The potential savings in drilling and blasting that had caused the problem were worth $US1.5 million – indicating a return of 2,300%.

Underground blasting costs are higher than in open pit mines. They also vary significantly depending on the mining method, ground conditions and scale. The objective of implementing modified blasting and crushing practices in an underground operation could be to achieve savings in milling costs (through greater throughput, reduced energy, or both) that are sufficiently greater than any increase in mining costs to justify the investment.

So how should intense “Mine to Mill” blasts be designed and implemented? Experience to date suggests that a few traps can be involved in significantly increasing the energy contained within mine blasts. The following discussion covers some basic aspects of the design and implementation of energetic blasts designed to increase production in hard rock open pit mines.

Blast Design

Blast design has traditionally had two objectives:

1. The creation of a loose, readily excavated muckpile
2. Minimum cost.

In the author’s experience the second point often over-rules the first. The ‘Mine to Mill’ concept adds an additional objective which leads to the re-definition of which cost is to be minimised. This additional objective is:

3. To produce a fragment size distribution that minimises the overall mining and processing cost.

In general, the recipe for achieving these objectives are to:

- improve the distribution of the explosive

- increase the proportion of shock energy delivered by the explosive by increasing its density and velocity of detonation

- increase the amount of explosive per unit of rock.

In essence this requires going against a number of the “rules of thumb” used in the conventional optimisation of blasting to minimise costs.

What then, should be the rules for the design of an intense blast? These rules have yet to be derived and fully tested, but the following discussion highlights some of the issues and suggests a path. The discussion is focussed on a classic “Mine to Mill” optimisation of mill throughput for hard, brittle ores. Other types of ore and mining objectives may require quite different strategies than those described below.

It is argued that the same basic principles for conventional blast design apply to more intense “Mine to Mill” blasts. However, some modifications are necessary. The following discussion deals with each of the major decisions encountered when designing an intense blast. These are:

- Confinement
- Powder Factor
- Explosive Selection
- Blasthole Diameter
- Burden and Spacing
- Stemming
- Initiation Sequence
- Back-break and Damage
- Dilution
- Far Field Vibration and Airblast.

Confined or Free Face?

Confined or choke blasting is popular in metalliferous mines because it:

- allows the generation of extensive blasted inventory ahead of excavation equipment

- allows mining in confined spaces without undue disruption to surrounding equipment and operations

- is believed to encourage the generation of a higher proportion of fines

- is believed to avoid the ‘loosening’ of natural boulders from the front burden of an unconfined blast.

Choke blasts also tend to generate more fly-rock, suffer greater back-break and higher far field vibrations than free face blasts. The combination of significant bench height (greater than 10 metres) large diameter blast holes (greater than 270 mm) and designs that minimise the cost of drilling and blasting alone is almost certain to lead to large fragments in the stemming zone of the blast and tight digging at the toe. The cost of these negative aspects of low cost, deep, confined blasts is rarely calculated and so rarely considered in the blast optimisation equation.

If the constraint of minimising the cost of blasting is removed, choke blasts can achieve more of their objectives and suffer fewer of their draw-backs. This is because sufficient energy can be made available to achieve significant breakage in the lower half of the bench, allowing efficient excavation and assisting in the generation of a greater proportion of fines than would otherwise be the case. However, particular care will be required to manage the ejection of stemming, control the cratering of the true burden (surface) and manage progressive relief throughout the blast if the other negative aspects are not to increase. Figures 3 and 4 show some contrasting performance in this regard.

Figure 3 The Controlled Containment of Energy

Figure 4 Failure to Contain Explosion Energy

Free face blasts provide the opportunity to manage relief within the blast. Because movement is provided for in the basic design there is less pressure on both the stemming zone and the boundary of the blast. However, control of the dynamic burden throughout the blast is essential if excessive burden movement velocities are to be avoided. The objective of a high energy "Mine to Mill" blast is to increase fragmentation, not to throw the rock large distances.

Powder Factor

In conventional blast design, the optimal powder factor is generally derived through trial and error, but is found to relate to the material strength and structure of the rock mass and the target fragmentation (Scott 1999). The target fragmentation in conventional blasts aims to achieve efficient loading and hauling. In the case of a high intensity mine to mill blast the target fragmentation aims to increase the mill throughput and would be regarded as being ‘over-powered’ by conventional standards. Whether it is 150% or 160% ‘over-powered’ is probably not as critical as how effectively that energy is controlled. The actual choice of powder factor is therefore less critical in mine to mill blasting.

Theoretical considerations and the author’s experience suggest that there is a law of diminishing returns with regard to elevated powder factors. Each increment in powder factor above the ‘norm’ is likely to yield a lower benefit than the initial increase. In most cases, it is likely that a 50% increase in powder factor will provide very obvious benefits to mill performance. It is also likely that a 100% increase in powder factor will produce a greater, but not much greater improvement. An optimum probably occurs after which any further increase in blasting intensity either fails to produce additional benefits (such as fines) or leads to an increase in negative factors (such as more critical size material). There is insufficient experience with the cost – performance relationships to define the optimum point for any situation. Each case will have unique attributes and require individual analysis. Figure 5 shows the form of the expected relationship.

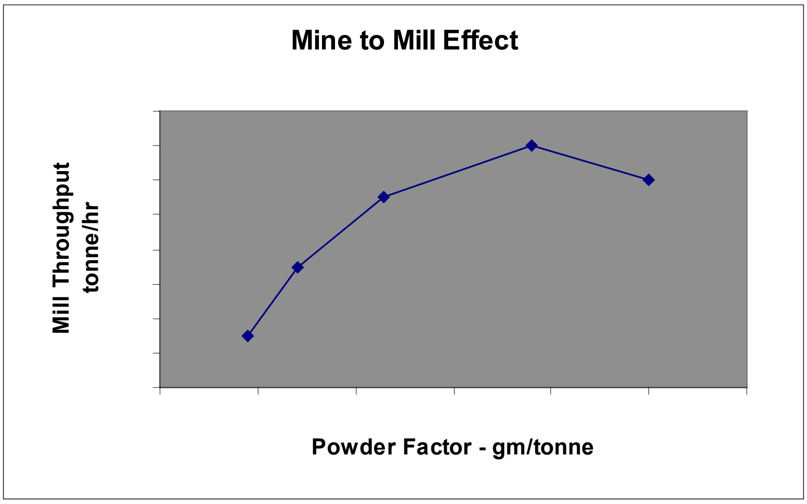

Figure 5 Effect of Powder Factor on Mill Throughput

In trials conducted to date, increases of 50% in powder factor have resulted in more than 10% increase in mill throughput. This improvement probably arises from better utilisation of explosive energy through more rigorous design and field implementation of the blasts as well as from the increase in the amount of energy provided. These results will be material and process dependent and should not be relied upon as a general rule.

Explosive

The objective of increasing the intensity of the blast is to create greater breakage at both the coarse and fine ends of the fragmentation curve. Both of these objectives require shock energy rather than additional heave energy. In fact, the greatest challenge facing the design of such blasts is the containment of the higher levels of heave energy than encountered in routine blasting. Decking with lower energy explosives at the top of the charge column or varying the density of the explosive up the column may move the energy distribution closer to the ideal. The trick is to design a blast where the shock energy is maximised to create new breakage but heave energy is not excessive but sufficient enough to create a loose muckpile.

Hole Diameter

In open pit mines one of the greatest drivers for reducing costs has been to increase the diameter of the blast holes. This, of course allows patterns to be expanded leading to a reduction in drilling cost per tonne of ore broken. However, this trend also results in a less even distribution of explosive energy and greater reliance on the heave of the blast to break and loosen the rock between blastholes. The thickness of the stemming zone also increases, further reducing likely fragmentation. The objective of an intense “Mine to Mill” blast is the opposite – an even distribution of explosive is desired with shorter distances between charges to maximise the shock breakage of material between holes.

Burden and Spacing

Burdens and spacings must be selected to suit the blasthole diameter and the energy factor required to achieve the target size distribution to optimise mill throughput. The high energy levels will tend to reduce burden movement times and increase burden velocities. In energetic free face blasts care may be required to select burdens that will limit the burden movement velocities to avoid excessive material displacement or throw. There is no experience available to suggest that the relationship between burden and spacing should be any different for a more energetic blast than for standard blasting operations.

Stemming

In an energetic blast, the requirement to contain the explosion gases is even more important than for conventional blasting. Any premature ejection of stemming will not only reduce the energy available to do useful work within the blast volume, but offers the potential for the generation of fly-rock.

The authors' experience to date with the implementation of energetic blasts suggests that conventional lengths of high quality stemming are capable of containing the energy from quite high energy blasts and that a dramatic increase in stemming length is not required on this basis alone.

However, the cratering of the stemming horizon and the subsequent ejection of large quantities of fly-rock into the air is a critical aspect of the design of energetic blasts, particularly those without a free face. The thickness of stemming zone required to contain the explosion gases will depend on the strength and structure of the material and the detailed design of the blasts. Cratering theory may provide some guidance in the absence of any field experience.

Figure 6 Cratering of Blasthole Collars

A classic quandary exists: where the surface is the only free face, without some movement there is no swell and hence no looseness to provide the necessary digability of the resulting muck. Any movement in the surface burden creates the opportunity for the premature release of explosion gases reducing the energy available for breakage and ejecting material into the air as shown on Figure 6. The thickness of the stemming zone is a critical parameter in the design of energetic blasts.

To manage this it is likely that the stemming length may need to be varied within a blast design. Greater stemming zone thicknesses may be needed in the areas of greatest confinement such as along the initial row of a choked blast or the central area of a tight V tie-up. In blocky ground is may be necessary to utilise stab holes within the pattern to create adequate fragmentation in the stemming zone. The extra cost of such a scheme must be balanced against the benefit provided to the whole mining and processing operation.

Initiation Sequence

The best initiation sequences for intense blasts are those that will allow and even progression of the detonation front with time, but limit the free travel distance of each burden. In free face blasting a tight V is recommended so that the mobilised burdens are not free to travel far. The result will be a fairly heaped muckpile with considerable swell, rather than a shallow muckpile strewn over a considerable area.

In choke blasting situations the ordered vertical displacement of the burden must be encouraged while limiting its velocity. A herringbone centre lift design is preferred over designs initiated on an edge because there are fewer rows to each side of the pattern, reducing the confinement and hence damage on these boundaries.

Figure 7 Initiation of a Herringbone Centre Lift Design

Although damage and far field vibration aspects should influence the actual delays chosen, it appears that inter-row delays for energetic blasts should be shorter than for conventional blasts in the same material. This is because the additional explosive energy will reduce the time required to mobilise each burden during the blast.

Back-Break and Damage

It is logical to expect a more intense, energetic blast to generate greater back-break or damage than a conventional blast. Advantage can be taken of increased fragmentation without causing unnecessary damage. To achieve this requires an understanding of the damage mechanisms appropriate to the design and materials involved and an appropriate criterion for limiting damage at the boundaries of the blast.

The idea is to contain an ‘energetic core’ within a softer boundary zone around the blast. This softer boundary zone is similar to designing a dynamic limit blast at the edge of the production blast. This can be as simple as reducing explosive density or changing the type of explosive, reducing the effective burden or providing more time for the final burdens to move. Perhaps pre-splitting techniques could be used to limit damage. Any additional costs required to control damage must be taken into account in assessing the overall economics.

Dilution

Dilution can have a major impact on the economics of mineral operations and is a major driver for the adoption of confined blasting techniques in gold operations. The additional energy provided in a 'mine to mill' blast will render the muck from choke blasts more 'diggable' and overcome toe problems. To minimise dilution it is imperative that the explosion energy is effectively confined and not released in the form of cratering.

Far Field Vibration and Airblast

The criteria for the design and management of environmental factors such as far field vibration and airblast will be the same for high intensity blasts as for normal blasts. Closer attention may be required to the detailed design and implementation of the blast, but provided the additional energy contained in each hole is allowed for, established design criteria should provide adequate guidance.

Conclusions

Substantial increases in mill throughput and/or reduction in mill power can be achieved by moving some of the breakage back down the production chain to the primary crusher and mine. There are a growing number of case studies where these relationships have been demonstrated in the field. However, carefully controlled trials and routine production blasting are not entirely the same thing and there are a number of areas where the design and management of more intense "Mine to Mill" blasting operations need careful attention.

The distribution of explosive energy and the control of confinement are critical to the successful design of an intense blast. The control of the parameters follows the same principles required for conventional blasting, but greater levels of sophistication are required in both the detailed design and implementation of such blasts if the potential negative outcomes of fly-rock, damage vibration and noise are to be effectively managed.

There has been only limited field experience with the implementation of energetic "Mine to Mill" blasts but the indications are that their theoretical impact on milling performance can be achieved in practice, if not exceeded. The ancillary benefits of improved digability, reduction in secondary blasting, and wear and tear on equipment remain difficult to cost but become apparent to those associated with the day to day operations.

Over the next few years it is expected that improved design rules and practices will be developed to readily tune energetic blasts to the requirements of the rest of the mining and processing system. This will be a valuable addition to the Blast Engineering Tool Kit.

References

Bellairs, P. (1998) "Optimum Drill and Blast", AusIMM Mine to Mill Conference, Brisbane, October

Cunningham C. (1990) "The Role of Blasting Instrumentation in Promoting Mining Profitability", The Third International Symposium on Rock Fragmentation by Blasting, AusIMM, Brisbane, August.

Dinis da Gama C. (1990) "Reduction of Cost and Environmental Impacts in Quarry Rock Blasting" The Third International Symposium on Rock Fragmentation by Blasting, AusIMM, Brisbane, August.

Eloranta J. (1995) "Selection of Powder Factor in Large Diameter Blast Holes", AusIMM EXPLO 95 Conference, Brisbane, September.

Hunter G., Sandy D., Miles N. (1990) "Optimisation of Blasting in a Large Mine", The Third International Symposium on Rock Fragmentation by Blasting, AusIMM, Brisbane, August.

Kanchibotla S.S (1998) "Importance of Mine Fragmentation on Downstream Operations", Proc. Blast 98, IIR Conference.

Scott A. (1999) "Effective Blast Engineering", AusIMM EXPLO 99 Conference, Kalgoorlie, November.

Simkus, R and Dance, A. (1998) "Tracking Hardness and Size: Measuring and Monitoring ROM Ore Properties at Highland Valley Copper", AusIMM Mine to Mill Conference, Brisbane, October