

Exploring the Effect of Blast Design on SAG Mill Throughput at KCGM

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ABSTRACT

Traditionally blasts in mining operations are designed to prepare the in-situ rock mass for excavation and subsequent transport without giving much consideration to the downstream comminution processes. Blasting is in fact the first step in the comminution process leading from in-situ rock to a marketable product and often plays a more important role than just fragmenting the rock for excavation.

This paper describes models to estimate the size distribution rocks resulting from blasting and downstream comminution processes. The models are then used to simulate the impact of changes in blasting practises on the SAG mill throughput at KCGM.

INTRODUCTION

The economics of many operations in the minerals industry depend on the particle size distribution and blasting is usually the first step in creating that size distribution. Traditionally blasting is designed to fracture the in-situ rock mass and prepare it for excavation and subsequent transport. The fragmentation is considered good when it is fine enough and loose enough to ensure efficient digging and loading. While this may be valid from a purely mining perspective, it may not necessarily be optimal from an overall operational viewpoint. Blasting is in fact the first step in the comminution process leading from *in-situ* rock to a marketable product and often plays a more important role than just fragmenting the rock for excavation.

This paper presents models to estimate the fragmentation resulting from blasting and other downstream comminution processes such as crushing and grinding. The models are then used to simulate the impact of changes in blasting practises on the downstream milling operations.

INFLUENCE OF MINE FRAGMENTATION ON DOWN STREAM OPERATIONS

The fragmentation resulting from blasting has a major effect on at least 2 of the subsequent downstream operations:

- Digging and hauling
- Crushing and grinding

Digging and hauling

It is obvious that the fragments created by the blasting operation must not only physically fit into the bucket of the excavator but must do so without unduly reducing the bucket load or fill time. If the size of the fragments in a muck pile is larger than the size that an equipment can handle, it will not

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only reduce the productivity but also increases the secondary blasting costs and equipment maintenance costs.

Even though it is obvious that a finely fragmented muckpile improves the diggability of an excavator it may be very difficult to correlate it to the overall productivity because of the following factors:

- The digging time is only a minor fraction in the overall truck cycle times (Hawkes et al 1995).
- In under-trucked operations any reduction in digging time will be lost in waiting longer for a truck.
- Diggability is also dependant on muckpile shape and looseness.
- The excavator productivity is greatly influenced by the operator's efficiency.

Usually trucks carry several excavator bucket loads of muck, hence they are more tolerant of oversize fragments. However, the fill factor of trucks is influenced by the size distribution of the muck. The payload of a truck loaded with finely fragmented muck is greater than a truck loaded with coarsely fragmented muck.

Crushing and Grinding

Crushing and grinding are the principal comminution stages which are employed to reduce the ROM ore to an appropriate size, whether for direct sale (eg iron ore) or to liberate the valuable minerals for subsequent beneficiation stages such as flotation and leaching. Several recent studies have shown that the capacity and efficiency of such comminution processes are strongly influenced by the ROM fragmentation distribution which in turn is influenced by the blasting (Smith et al 1993, Scott and McKee 1994, Kojovic et al 1995, Nielsen and Kristiansen 1995, Eloranta 1995). This is particularly true of autogenous and semi-autogenous milling which rely on the feed ore for grinding media and hence are relatively sensitive to changes in feed size distribution (Morrell and Morrison, 1996). Given this sensitivity it follows that changes in the ROM size distribution will affect mill performance and hence a potential exists for modifying blast design and practice to benefit the subsequent comminution stages. McKee et al (1995) indicated this through modelling and simulation and showed that by tailoring the blast-induced fragment size distribution autogenous mill throughput could be increased by as much as 20%.

An increasing number of mining companies are now exploring the potential of integrating mining and comminution practices to improve their overall efficiency and hence profitability (Simkus and Dance, 1998; Kojovic et al., 1998). However, field experimentation in this area is not taken lightly due to the cost of implementation and the attendant risk of making changes where knowledge of the likely effects is lacking. A cost effective way of evaluating new blasting/comminution strategies and hence minimising risk is through modelling and simulation. At Kalgoorlie Consolidated Gold Mines blasting and comminution models developed by the JKMRC were used therefore as the first step in determining whether any potential existed to increase mill throughput by changing blast design.

MODELLING OF BLASTING, CRUSHING AND MILLING

Blast Fragmentation Modelling

A variety of modelling approaches ranging from purely empirical to rigorous numerical models have been used to predict fragmentation from blasting. The Kuz-Ram model is probably the most popular. It was developed by Cunningham (1983) who modified Kuznetsov's equation for ANFO based explosives to estimate average fragment size (X_{50}) and combined it with the Rosin-Rammler equation to predict the entire size distribution. The equations are written as:

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

$$R = 100 - e^{-0.693 (x/x_{50})^n} \quad (2)$$

Where: A rock factor

=7 for medium rocks

=10 for hard, highly fissured rocks

=13 for very hard, weakly fissured rocks

K powder factor i.e. $\frac{Q}{V_o}$, kg/m³

Q quantity of explosive in one blasthole

V_o rock volume broken by one blasthole (burden x spacing x bench height), m³

E relative weight strength of explosive, ANFO=100 and TNT=115

R percentage smaller than x

x size of rock

n uniformity exponent

Cunningham (1987) further developed an equation to estimate the uniformity exponent "n" of the Rosin-Rammler distribution curve from blast design parameters. However, even though the Kuz-Ram model has been used widely for estimating blast fragmentation, it has the following drawbacks:

- The rock quality factor rating is based on a very subjective description such as massive, blocky or friable.
- The energy factor is based on the explosive energies derived from the ideal detonation codes. However, the energy released by an explosive in rock blasting is a function of the rock mass properties, explosive properties and confinement provided by the blast. In some blasting applications the effective energy released by an explosive during blasting can be much different to the theoretical energy estimated by the ideal codes (Sarma 1994).

- Research at the JKMRC and others has demonstrated that the Kuz-Ram model under estimates the contribution of fines (< 100mm) in the ROM size distribution (Kojovic et al.1995, Comeau 1996).

JKMRC Fragmentation Model

In one of the models being developed at the JKMRC an approach similar to that used in the Kuz-Ram model has been adopted but with the following modifications:

- The rock quality factor is based on the *in situ* block size and strength properties of the rock mass similar to that proposed by Grouhel (1992).
- The energy factor is based on the effective energy which is a function of explosive properties, rock properties and the confinement provided in the blast. Effective energy is estimated using the explosive energy model EXEN (Sarma 1994).

In the case of the finer fractions it is hypothesized that they are produced by the pulverizing or crushing action of the explosive adjacent to the blast holes. A cylinder of rock around each hole is therefore defined within which crushing takes place (see Figure 1). The radius of the cylinder, and hence its volume, is determined by calculating the point at which the radial stress around the blast hole exceeds the dynamic compressive strength of the rock.

Currently a working size of 1mm is used to define the coarsest particle that results from crushing. This has been chosen on the basis of results from a number of mines where rom sizings were available. It is expected, however, that this size will vary from rock type to rock type and may be determinable from small scale blast results.

Having determined the crushing zone radius around each blast hole, and hence its volume, and knowing the number of blast holes the volume of crushed material (<1mm) can be calculated (V_{crush}). As the volume of rock blasted (V_{br}) is also known the % of blasted rock smaller than 1mm can be estimated from:

$$\% < 1\text{mm} = 100 \times V_{crush}/V_{br} \quad (3)$$

The uniformity index for the fine end of the distribution is then calculated by substituting the fines percentage in the conventional Rosin-Rammler equation. The uniformity index for the coarse end is estimated using the approach proposed by Cunningham (1987).

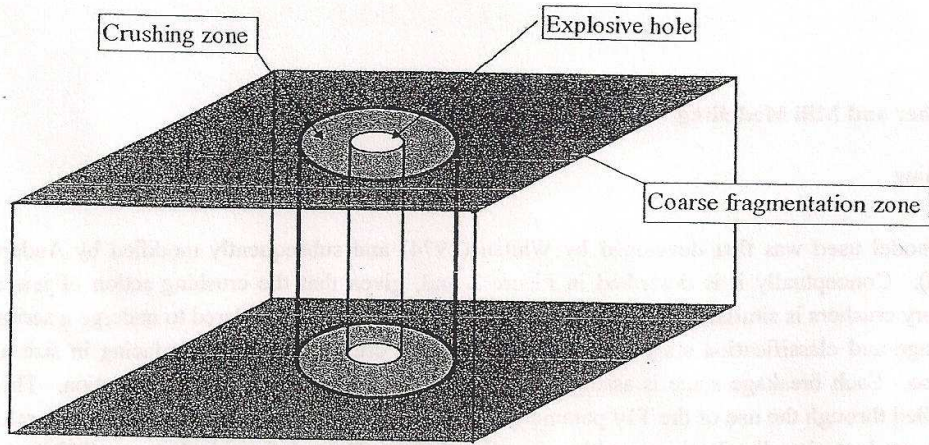


Figure 1 - Schematic of the Crushed Zone Approach

To illustrate the ability of the new model to predict the entire ROM size distribution it was used on basalt data (Kojovic et al, 19995) and compared with the conventional Kuz-Ram prediction. The results are shown in Figure 2.

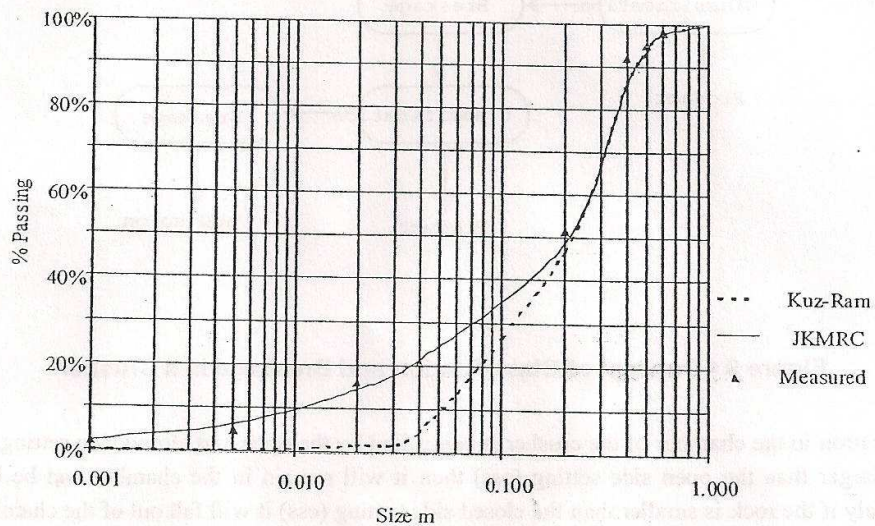


Figure 2 - Measured ROM Size Distribution vs Kuz-Ram and JKMRC Model Predictions

Crusher and Mill Modelling

Crushing

The model used was that developed by Whiten (1974) and subsequently modified by Andersen (1988). Conceptually it is described in Figure 3 and, given that the crushing action of jaw and gyratory crushers is similar, can be used for both devices. Feed is considered to undergo a series of breakage and classification stages as it passes down the crushing chamber, reducing in size as it does so. Each breakage stage is assumed to produce the same geometric size reduction. This is modelled through the use of the T10 parameter (Narayan and Whiten, 1988) which in turn is related to the product size distribution resulting from breakage. The relationship between T10 and the breakage size distribution is determined experimentally by breaking representative rock specimens using the JKMRC's drop-weight tester.

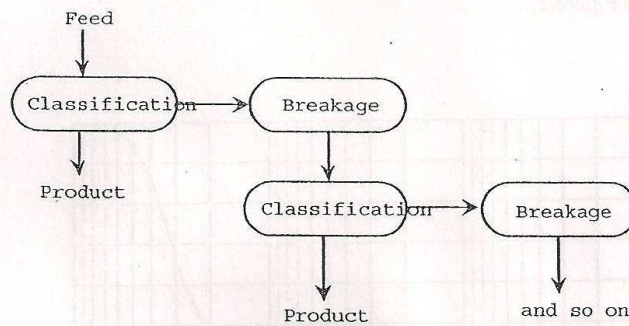


Figure 3 - Concept of Classification and Breakage in a Crusher

Classification in the chamber of the crusher is controlled by the open and closed side setting. If the rock is larger than the open side setting (oss) then it will remain in the chamber and be broken. Conversely if the rock is smaller than the closed side setting (css) it will fall out of the chamber and not be crushed any further. For rocks which are in between the css and oss in size a probability exists for them to either remain or pass out of the crusher

Semi Autogenous Milling

The most recent version of the sag mill model was used and incorporates the effect of ball load, feed size and speed (Morrell and Morrison, 1996) as well as the effects of grate design (Morrell and Stephenson, 1995). In addition it also contains a power draw model (Morrell, 1996a, 1996b). Conceptually the model is represented in Figure 4.

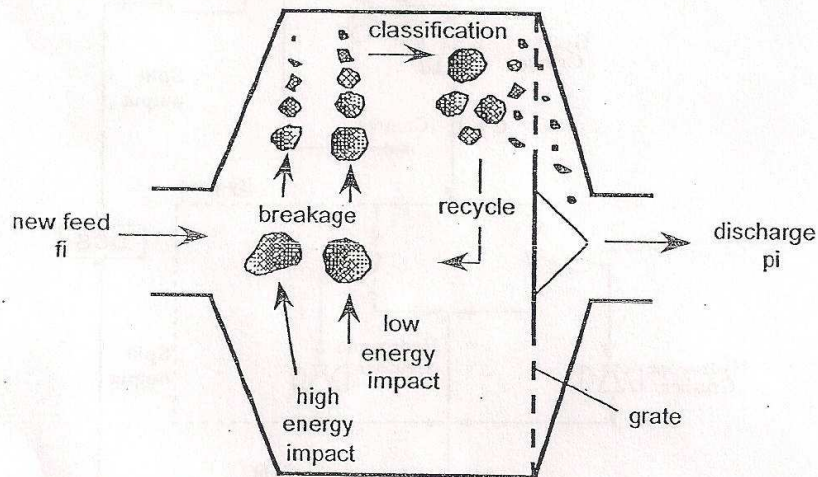


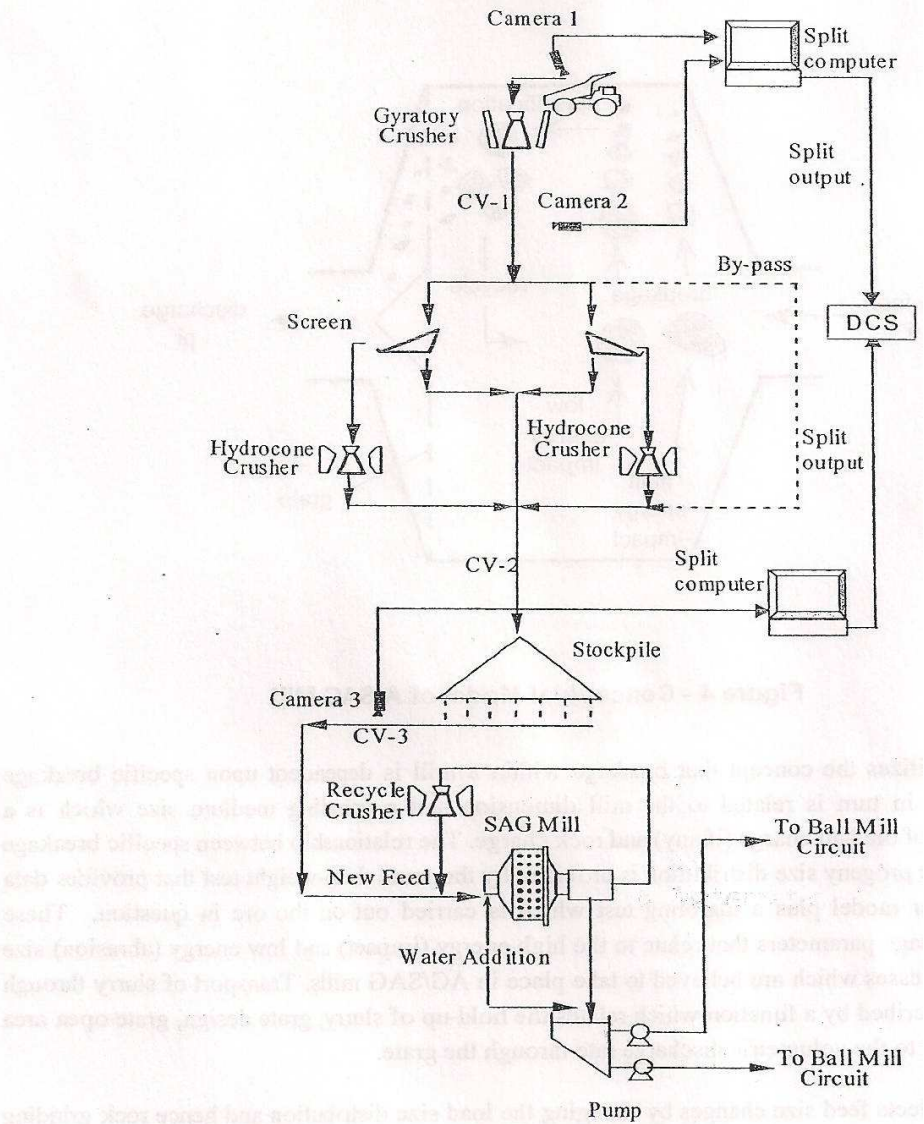
Figure 4 - Conceptual Model of A SAG Mill

The model utilizes the concept that breakage within a mill is dependent upon specific breakage energy. This in turn is related to the mill dimension and a grinding medium size which is a characteristic of the ball charge (if any) and rock charge. The relationship between specific breakage energy and the progeny size distribution is provided by the same drop-weight test that provides data for the crusher model plus a tumbling test which is carried out on the ore in question. These generate breakage parameters that relate to the high energy (impact) and low energy (abrasion) size reduction processes which are believed to take place in AG/SAG mills. Transport of slurry through the mill is described by a function which relates the hold-up of slurry, grate design, grate open area and mill speed to the volumetric discharge rate through the grate.

The model reflects feed size changes by changing the load size distribution and hence rock grinding media size. In addition the size-by-size breakage rate distribution, which is central to the operation of the model, is modified by using an empirical correlation based on a large data base of operating mills.

Comminution Circuit

The primary comminution circuit at KCGM is illustrated in Figure 5.



It comprises a primary gyratory crusher which feeds a 36 ft sag mill in closed circuit with a pebble crusher. Originally secondary crushers were used but have now been largely phased out (Nelson et al., 1996). Of note are a number of “Split” image analysis size distribution measurement systems installed in the circuit which are currently being trialled/developed (Atasoy et al 1998).

Model Development / Validation

Following characterisation of one of KCGM’s principal ore types the JKMRC’s modified Kuz-Ram model was used to estimate the ROM size distributions using their current design. Their current blast design details are given in Table 1.

Table 1: Current blast design parameters

Burden (B)	5
Spacing (S)	5.8
Hole Depth (L)	11.3
Hole Diameter (D)	165
Column Charge Length (CCL)	7.2
Explosive	Energan 2620
Density	1100
VOD	4550
Powder Factor	0.58

The ROM predictions were then compared with the results from the analysis of images of the muck in 2 trucks. The results are shown in Figure 6 indicate the predicted ROM follows the coarser part of one distribution and the finer part of the other.

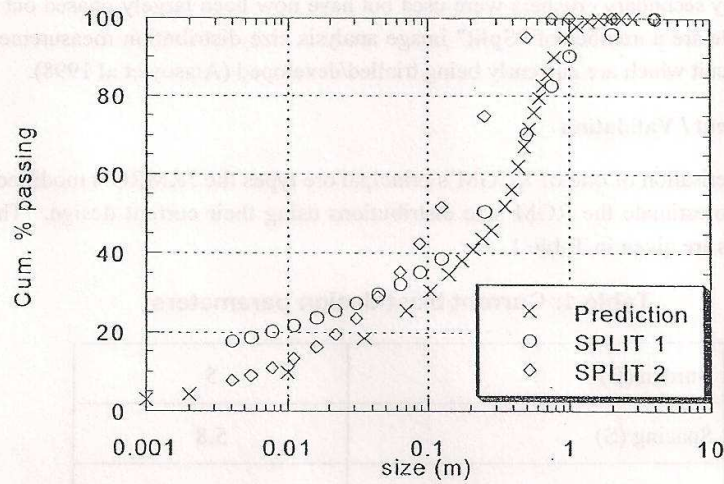


Figure 6 - ROM Prediction vs Image Analysis Results from 2 Trucks

In the next step the predicted ROM was used to simulate the performance of the gyratory and sag mill circuits. The models for these circuits were developed from a series of surveys which were conducted by KCGM and JKMRC staff (Nelson et al., 1996). The results of this simulation are shown in Table 2 and show good agreement with the current performance characteristics of the circuit.

Table 2 - Observed vs Predicted Performance of the Primary Grinding Circuit

	Observed	Predicted
Feed Rate (t/hr)	1250	1250
Feed F80 (mm)	130	118
Trommel undersize 80 (mm)	3.06	3.21
Recycle crusher feed rate (t/hr)	276	345
Recycle crusher P80 (mm)	15.0	15.5
Mill filling (%)	29.5	30
Mill power (kW)	12717	12750

Simulating Changes in Blast Design

Having validated that the models can reproduce current blasting and comminution circuit performance the next step was to simulate changes in blast design and determine whether they would affect sag mill performance. Two different blast designs are used in this paper and are given below together with the current design (design 1). The resultant ROM predictions are shown in Figure 7. The 2-part nature of the model is apparent in the change in gradient in the central part of the curve. Ideally the curves should give a smooth transition between the coarse and fine parts of the distribution. Research is currently underway to effect this.

From Figure 7 it is clearly seen that the increase in powder factor and change to a higher vod explosive of design 2 has resulted in a ROM with more fines and a small change in the coarser end of the distribution. In the case of design 3 the much higher powder factor has a marked effect on the coarse end of the distribution as well as increasing the amount of fines.

Table 3. Blast design parameters

Hole dia mm	Design 1	Design 2	Design 3
Burden m	5	5	4
Spacing m	5.8	5.8	5
Hole Depth m	11.3	11.3	11.3
Explosive	HANFO	Emulsion	Emulsion
Explosive density kg/m ³	1100	1250	1250
VOD m/s	4550	6000	6000
Powder factor kg/m ³	0.58	0.66	0.96

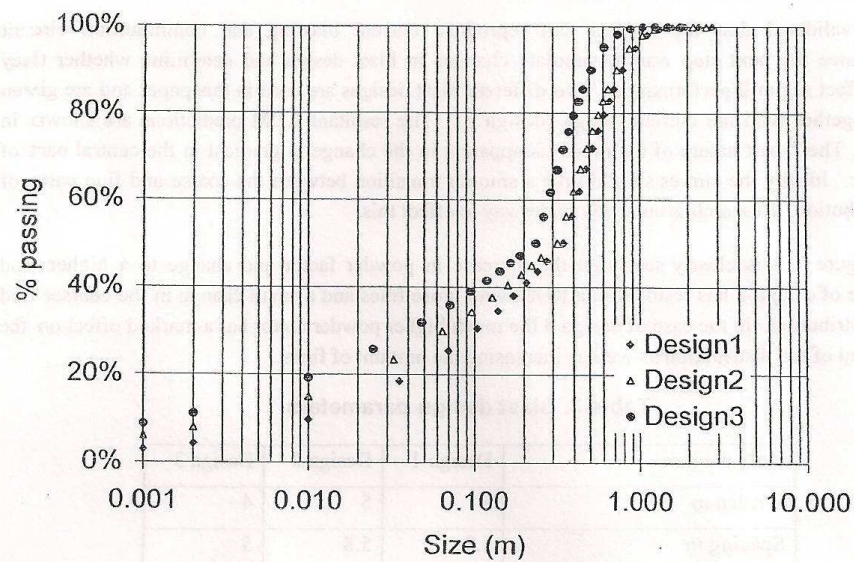


Figure 7. ROM size predictions for different blast designs

The new rom distributions were then used to simulate the primary crusher product, the results of which are shown in Figure 8. It is seen that the primary crusher product has considerably reduced the differences apparent in the rom size distributions. However significant differences in the fines and intermediate size fractions are still apparent. Apart from size distribution effects, the reduced amount of coarse material in design 3 resulted in less work for the primary crusher ie it generates additional capacity which, if required can be used for additional throughput or used to decrease the gap setting and give further benefits to the sag mill.

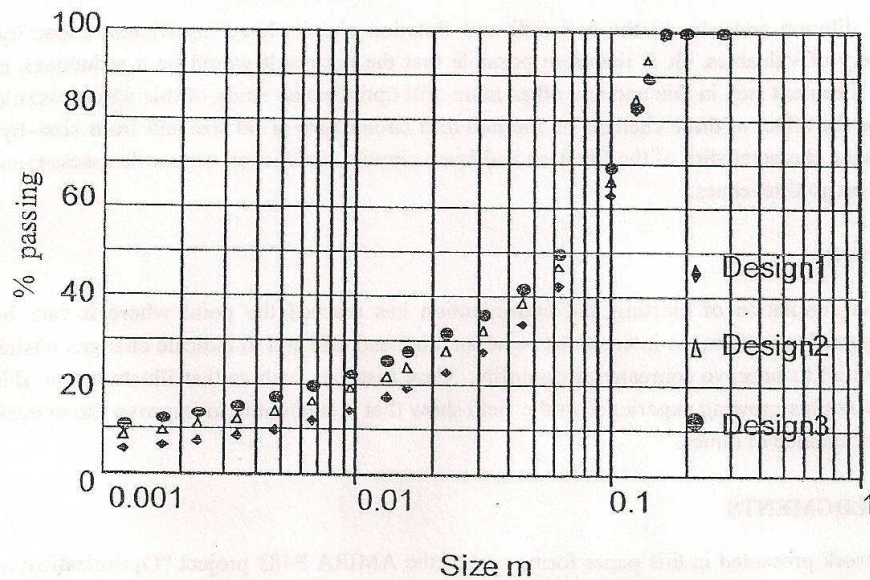


Figure 8 – Predicted Crusher Product Size Distributions

Having simulated the gyratory crusher the product was then simulated through the sag mill. Mill performance is usually evaluated in terms of throughput, specific energy (kWh/t) and final grind size. For these simulations the mill was run at a constant mill filling and speed, and hence constant power, so that the effect on throughput could be gauged. The results are shown below (Table 4) for the 2 new designs (2&3) together with the current design (1).

Table 4 – SAG Mill Simulation Results

Design	Throughput (t/hr)	P80(mm)	P50(mm)	Power(kW)	kWh/t
1	1250	3.21	0.69	12750	10.2
2	1420	3.13	0.65	12755	9.0
3	1480	3.11	0.63	12749	8.6

As can be seen a significant increase in throughput is obtained with Design 2 with further, though proportionately much less, improvement with Design 3. This is despite the 50% increase in powder factor in Design 3 compared with Design 2. The reasons for the increased throughput are twofold. Firstly both new designs generate more -10mm material which does not need to be ground and is essentially “free” throughput for the mill. Secondly, the changes in the +50mm size range lead to marginally lower amounts of so-called “critical” size in the feed and a narrower distribution in the +100mm media size range, with less unnecessary oversize.

Although the results show that potential exists for increasing sag mill throughput by changing blast design this may not necessarily be desirable as there may be an economic penalty from the

viewpoint of dilution and, should the ball mill and flotation circuits have insufficient capacity, poorer recovery of valuables. It is therefore possible that the net result would be a reduction in profitability. The next step in this and any other mine-mill optimisation study of this nature would be to simulate the effect of these changes on the ball mill circuit final grind size and from size-by-size performance characteristics of the flotation and leach circuits, conduct an economic assessment of the impact on gold revenues.

CONCLUSIONS

Modelling and simulation of blasting and comminution has reached the point where it can be constructively used to explore the interactions between mine and mill and to indicate changes which have the potential to improve company profitability. Case histories such as that illustrated in this and other papers plus growing experience in the field show that it is possible to improve the overall economic performance of mines.

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