Keeping Geologists, Production Personnel and Contractors Happy – An Integrated Approach to Blasting at Boddington Gold Mine, WA

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INTRODUCTION

Location

The Boddington Gold Mine (BGM) is located approximately 120 km south east of Perth, Western Australia. (Figure 1). Located within the picturesque Darling Ranges, the land was previously used for forestry activities prior to mining commencing in mid-1987. Mineralisation is hosted within deeply weathered felsic and intermediate intrusions and volcanic rocks of the Saddleback Greenstone Belt.

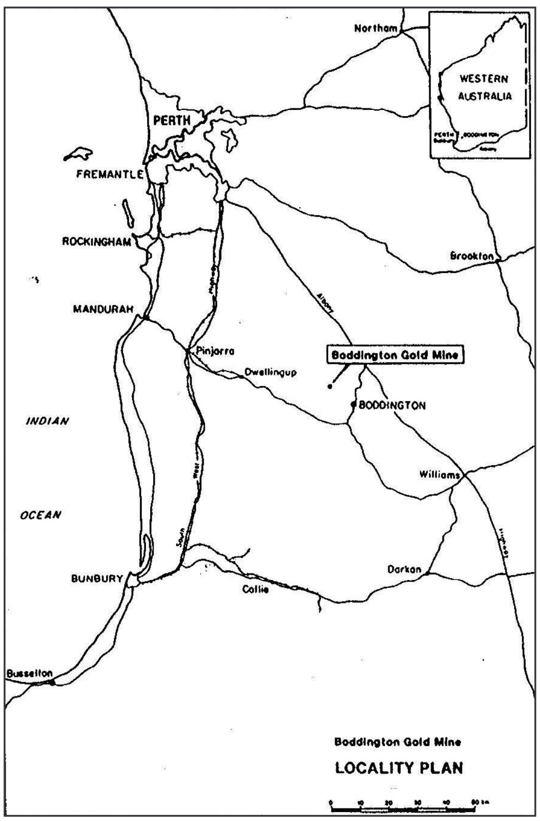

FIG 1 – Location of Boddington Gold Mine.

Current operations

Open pit mining moves approximately 15 Mtpa to recover 420 000 ounces of gold. Underground operations contribute approximately 60 000 ounces annually from 90 000 tonnes of ore. BGM's open pit operation was Australia's second largest gold mine in 1994 (behind Kalgoorlie's Fimiston Pit).

The majority (approximately 70 per cent) of material currently mined in the open pit is derived from the free digging saprolitic horizons of the weathered profile. This profile averages approximately 40 to 50 metres depth over the mine site and consists of kaolinite and smectite clays with varying degrees of ferruginisation and silicification.

Mining is carried out using contractors (Eltin) who utilise three excavators loading a fleet of 14 777C Caterpillar dump trucks with the usual ancillary equipment such as graders, dozers, water carts etc. Little to no blasting is required within the saprolitic horizons. Blasting is required for laterite cap rock, fresh dolerite dykes, and bedrock as encountered. Blasthole drilling is carried out by two Tamrock DHA1000 drill rigs.

With oxide reserves being steadily eroded by consistent mining, attention in recent years has turned to mineralisation in the bedrock. Significant resources have been identified in fresh rock under currently operating oxide pits and potential exists for large scale hard rock mining below the weathering front.

Production recently commenced in one of the bedrock resources (Blackbutt Pit), see Figure 2. Mineralisation is characterised by discrete actinolite and quartz veins in andesite. A significant amount of effort is being made to optimise grade control, blasting, and extraction performance within Blackbutt in order to gain information that will assist in the assessment of the other larger and deeper bedrock resources. This paper aims to explain the evolution of procedures used in Blackbutt as an overview of the issues involved in optimising blasting and extraction.

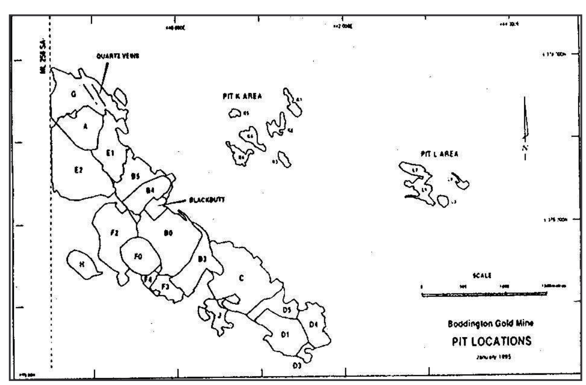

FIG 2 – Location of Blackbutt pit within Boddington Mine site.

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Background

The Blackbutt pit is a cut back into the exposed basement along the north side of pit B0, see Figure 2. The total resource is 425 000 tonnes @ 3.55 g/t. Current benches average 45 000 tonnes with a strip ratio of 1.04:1. Mining was initially carried out on three-metre deep benches. This was mainly a function of previous experience (benches in the oxide pits are 3 m deep) rather than any operational or economic consideration.

The ore zones are reasonably consistent but lack significant visual control. Ore definition in early benches was carried out by blocking assay results from blast hole sampling and surveying in the boundaries after blasting.

Initial blast hole patterns were set out using 89 mm diameter vertical holes on a staggered 2.5 metres spacing with 2.5 metres burden, and 3.5 metres depth (the bench height being 3 metres with 0.5 metres for sub-drill).

Number 8 Nonel detonators and 125 gram boosters were used down each blast hole and the surface tied in with 5 gram detonator cord. This was normally done row by row with each row delayed using a 65 millisecond (ms) DRC or 35m/s DRC. Every second hole was delayed with a 15 ms DRC, and a 100 m/s Nonel delay was used around the back of the shot as a safety line (see Figure 3a).

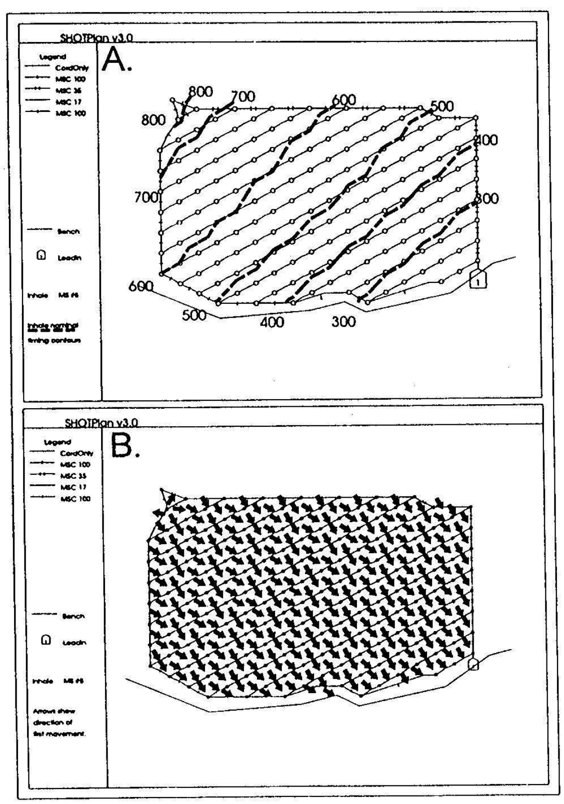

FIG 3 – Models of previous blasting practices. (A) plan view of tie in. (B) Plan view of predicted movement of shot (arrows indicate movement)

The holes were charged using either ANFO (dry holes) or slurry (2560) (wet holes). The powder factor was normally around 0.64 kgs/BCM and the holes were stemmed with the drill cuttings from around the hole.

Pre-splitting was carried out using 76 mm diameter holes, generally angled at 70 degrees, on a spacing of 1.8 to 2 metres, and charged with 25 mm diameter traced powershear with the bottom half metre of charge doubled over to 'kick out' any toe. The pre-splits were usually fired before production drilling commenced. Due to their large spacings, the pre-splitting sometimes left a lot to be desired (see Figure 4).

FIG 4 – An early pre-split wall of the Blackbutt Pit.

ISSUES RELATED TO BLASTING

With the commencement of mining in Blackbutt a number of problems relating to excessive shot movement, fragmentation size, and dilution were encountered. The result was loss of ore, misdirection of waste as ore, unnecessary additional explosives costs, need for secondary breakage, and delays to digging as a result of toe and heave. Figure 3b illustrates the movement pattern resulting from the early blasting practices. Figure 5 shows a general view of an early shot profile.

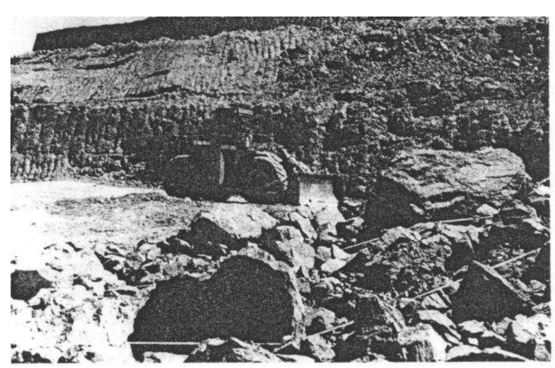

FIG 5 – an early shot profile (Loader for scale) showing oversize.

Interaction between all parties involved was traditionally limited and often ineffective. However, a more integrated approach was taken with increased communication between the various groups involved at each stage. This ensured that all parties' objectives were taken into consideration.

Blast performance is greatly affected by the following factors (Cameron and Kennedy, 1993):

1. Production requirements;
2. Rock properties;
3. Explosive properties;
4. Blast geometry;
5. Initiation sequence and delay timing; and
6. Work practices.

These issues were each addressed in turn in order to identify areas in which improvement could be made. At the BGM the key issues affecting cost performance were identified as:

1. Matching blasting to geological properties of the host rock;
2. Allowing a degree of selective mining due to the style of mineralisation;
3. Working around physical constraints due to the location of the pit;
4. Optimising blasting when a free face is not possible;
5. Timely return of grade control results to allow prediction of ore boundaries; and
6. Meeting the various agendas of the different groups involved, ie the contractor's aim to achieve optimum digability, Production's aim to achieve low cost, and Geology's aim to minimise dilution and get maximum ore recovery.

Figure 6 illustrates the interdependence of each process that occurs prior, during and after a blast. The issues outlined above are addressed in the order in which they occur within the production cycle.

Identification of rock properties

Table 1 lists the rock properties for the material being mined in Blackbutt. In general the material is considered to be very strong.

TABLE 1

Rock properties within the Blackbutt Pit.

Property	
Uniaxial compressive strength	160 Mpa*
Young's Modulus	22.5
Density	2.75*
Water content	0.12%
Material index (mi)	17 (igneous rock)

* (Dight and Bieser, 1993)

Structural mapping identified four main fracture sets (N, NNE, WNW, and sub horizontal) with the inclusion of up to 17 distinct fault zones within the Blackbutt pit area (Dight and Bieser, 1993). Fractures occur regularly across relatively small intervals (1 to 1.5 metre spacing).

The rock mass can be described as consisting of very hard material with numerous planes of weakness. Blasting should aim to exploit these natural weakness planes to assist with fragmentation.

Rock properties are consistent across the entire Blackbutt Pit so blast parameters do not need to be adjusted due to any rock type change.

Blasthole spacing, design, and drilling

The blasting patterns in use were costly, produced a lot of oversize and toe, were slow to dig, and resulted in mixing of ore and waste. Discussion between the contractors, geologists, and production personnel, highlighted the fact that existing blasting practices didn't meet anyone's requirements! It was decided to refine the blast designs.

Firstly, only one aspect of the blast parameters was to be changed at a time so each change could be assessed and quantified. Video taping of blasts was commenced to provide a record of what was happening during the blast in relation to stemming ejection, fly rock and to some extent the initiation sequence. Computer simulation of the tie in and shot movement commenced using SHOTPLAN software. In turn each of the following parameters was modified and assessed:

1. The pattern size was changed to 2.5 metres spacing by 2.2

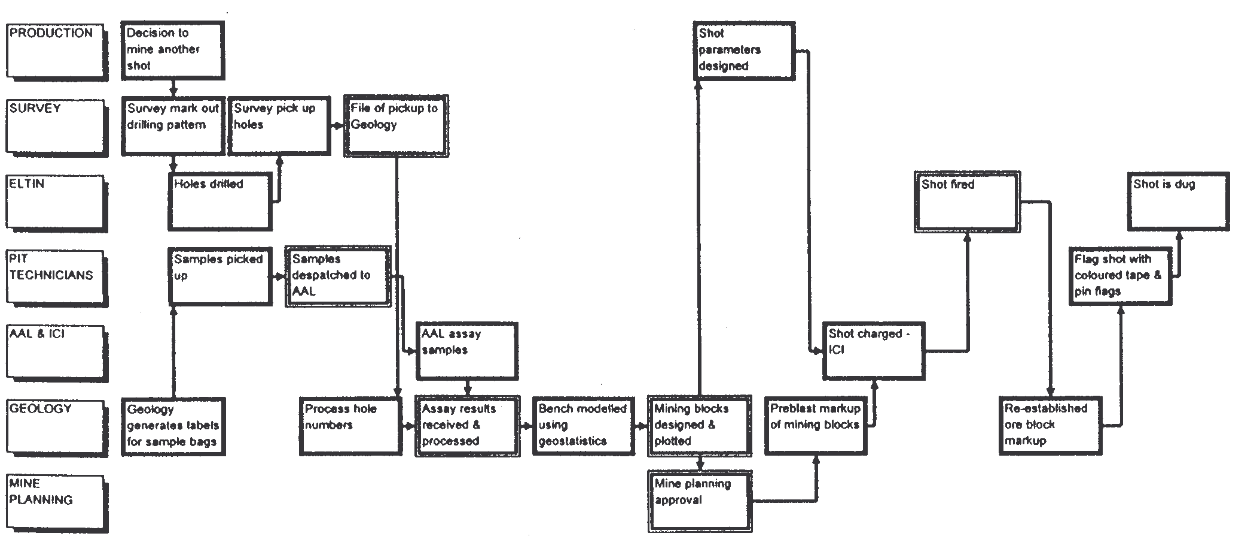

FIG 6 – Flow diagram of events relating to blasting and digging.

metres burden, thus giving us an equilateral staggered pattern. Even this small change in pattern size produced noticeably less oversize and toe.

2. The stemming was changed from drill cuttings to crushed blue metal which maintains more energy in the blast.
3. The powder factor was then dropped from 0.64 kg/BCM to 0.42 kg/BCM. The use of ANFO in dry holes was stopped, and all holes were charged with 2560 slurry. When the blast was fired there was a lot less stemming ejection and fly rock plus it was easy to dig with very little oversize or 'toe' on the floor.

The use of 2560 slurry as opposed to ANFO resulted in higher shock energy which produced better results in the hard andesite. The slurry also gives a greater density per metre of hole than ANFO. This allowed the same hole diameter to be used with more stemming while at the same time lowering the shock energy down the column.

All these shots were tied in as described earlier (see Figure 3a). The results of course were very pleasing to all groups. However, it was still feasible to make improvements especially in controlling shot movement and heave.

The next major step was to change the bench height. Blasting was now resulting in greater fragmentation and thus better digability. A larger bench height would give greater burden across the shot and so provide even better results in terms of stemming ejection, fly rock, and general movement of the material. However, the equipment used on site is not amenable to digging deeper benches. Using the current fleet a change to five metre benches would necessitate two passes over the material during digging, or double benching.

Geology had significant reservations about grade controlling and mining five metre benches in two portions (double benching). A number of issues could result in significant misrepresentation of the ore and dilution:

1. Multiple samples from a single blast hole introduces a high risk of contamination and sampling error.
2. Movement of the material during blasting to a free face creates problems identifying ore boundaries within the mass. Ore mark-ups become calculated guess work.
3. Rilling of material could potentially result in mixing of different ore categories.

It was decided to increase the bench height but to satisfy grade control requirements not differentiate different horizons within the blast. That is single samples would be taken over the blast holes and ore mark-ups would present material for the entire bench.

For general purposes the maximum bench height can be expressed as 60 to 120 times the hole diameter, depending on rock strength (Boucher, 1994). The blast holes are 89 mm diameter and our rock is hard so the bench height works out to 5.3 metres.

With the shift to five metre benches another pattern size had to be designed as all the relationships between burden and hole spacing changed with the increase in depth. From our previous experiments it was decided to use an equilateral staggered pattern blue metal stemming and single hole initiation.

The following parameters were applied to each shot (after Boucher, 1994):

- Sub-drill required is five to ten times the hole diameter ie 0.8 metres;
- Burden should be 1.4 times the stemming height ie 3.2 metres; and
- Spacing should be to 1.15 times the burden which equates to 3.7 metres.

These guidelines gave us a 3.7 m x 3.2 m pattern (3.7 metres equilateral).

Hole depth is critical and so each hole in the pattern was surveyed in with drill depth, then re-measured by the driller and again by the shot firer.

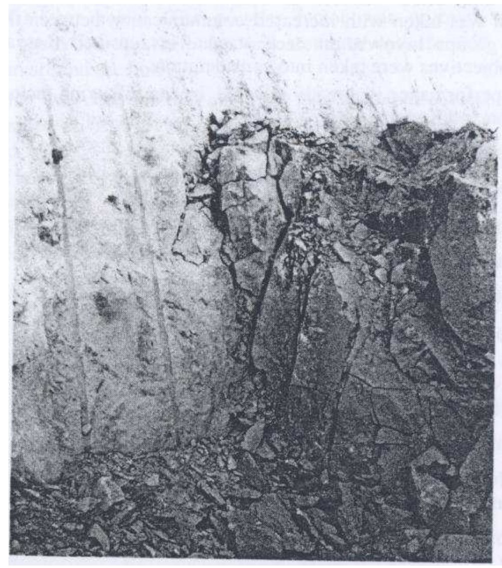

FIG 7 – Pre-split face after closing hole spacing to 1.2 metres.

Pre-splitting was again carried out using 76 mm diameter holes, generally at 70 degrees, but spacing was decreased to 1.2 metres and charged with 25 mm diameter traced powershear with the bottom half meter of charge doubled over to 'kick out' any toe. Figure 7 shows a considerable improvement in pre-splitting results.

Blasthole sampling for grade control

The assay results used for definition of ore/waste boundaries are derived from sampling the blast holes. Sampling procedures were derived from the methods developed by Pierre Gy as documented in Pitard (1992 and 1993). The minimum size of the lumps carrying gold was calculated using Gy's classic sampling formula (Pitard, 1993) to be 610 micron (Morley, 1994a). Average gold grade of ore on previously mined benches was 2 g/t. Using the sampling nomogram (see Figure 8) a total sample weight of 4 kg is required to achieve a fundamental error

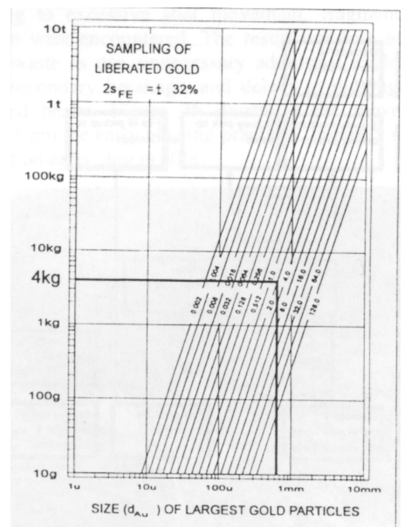

FIG 8 – Sampling nomogram for calculation of sample size (after Pitard, 1993) variance of +/ 32 per cent (ie an acceptable margin of error variance).

A purpose built sampling tray was constructed to collect the sample. A deflector plate and shroud secured to the drill mast adaptor (see Figure 9) directs sample around the collar and into the tray. This technique provides a statistically acceptable sample (Pitard, 1993) in a relatively simple manner. Risks of contamination or mistakes are minimised by the simplicity of the technique.

Sample bags are numbered with computer generated labels, a duplicate of which is attached to a plastic cup which is filled with drill chips and left beside the hole. The numbers on cups allow navigation across the bench using a plot of sample numbers. It is planned to include bar codes on these labels to speed laboratory processing and simplify handling procedures. The samples are despatched in batches of up to 200 for 30 g fire assay, with results available on a 24-hour turn around.

As much as possible, the sampling and ore definition process is designed to limit the amount of time the blast holes remain open. Ore blocks are designed utilising local geological knowledge derived from mapping and examination of previous benches in conjunction with the assay results. ENVISAGE and MEDSYSTEM software is utilised to model the bench and produce ore blocks.

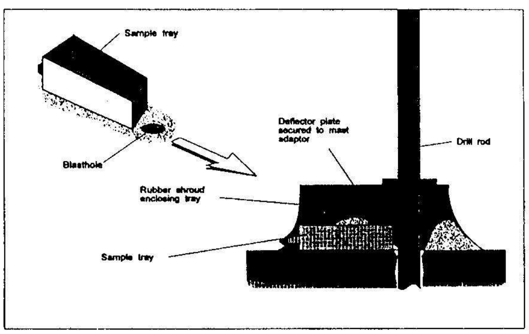

FIG 9 – Blast hole sample technique (after Newcrest Mining Limited, 1994).

Choice of area to blast and blast design

Due to the size of the pit and the relatively simple shapes of the mineralised zones, blasts are designed where possible to match the ore zones. The decision of which holes will actually be fired is not made until after the grade control assay results have been received. This allows a significant amount of flexibility that is often not possible in large open pit operations where blast patterns are normally set prior to the grade control information being available. This is a luxury that may not continue once mining commences in some of the larger bedrock pits that require tighter scheduling.

The benefits of having grade control results when designing the blast is a reduction in dilution, and simplification of the digging. This is mainly due to shot boundaries matching the ore/waste boundaries. Disadvantages include the loss of holes drilled but not charged and some requirement for redrilling. In general, however, the benefit of segregating ore from waste outweighs the loss of a small number of holes at the face.

Where possible blasts are designed so that:

1. Paddock blasts are limited to areas of all waste, or all ore;
2. Where possible blasts with ore are fired to a free face;
3. If a mixture of ore and waste is contained in a single shot, movement is designed to occur long strike;
4. When movement must occur across strike the shot comprises of either all waste or all ore (not a mixture).

Pre-blast mark-up of ore

Once the blast pattern has been decided on, and if more than one ore category exists within the proposed blast, a pre-blast mark-up will take place.

The ore blocks are superimposed onto a plot showing sample numbers that correspond to the labelled cups next to each hole. This plot is used to navigate across the blast pattern. Coloured flagging tape is placed on the shot to delineate boundaries. The tape is secured to fist size lumps of rock spaced approximately five metres apart along the boundaries. In some larger shots additional holes are drilled (to full depth) and poly pipe is installed. The pre-blast and post blast positions are surveyed to allow blast movement and direction to be quantified.

The advantage of utilising preblast boundary tapes is that they move with the rock mass during blasting, giving an accurate location of the material either side of that boundary after the blast. The practice of surveying in blocks after a blast without any adjustment for movement results in significant dilution of ore and misdirection of ore as waste.

The disadvantage of utilising preblast tapes is their sometimes elusive nature after the blast. If unanticipated heave or rifling of holes has occurred the tapes can become so shredded that precinct together the line becomes close to impossible. However, with good blast design this can be largely avoided.

Charging and firing of the shot

Gold-dets were used the first time the new pattern was fired. This particular blast had no free-face and so a centre lift tie in was used, with the ‘centre’ on the left hand side, two holes in from that edge (see Figure 10). The explosive used was ICI 2560 slurry (1.15 density). Blue metal stemming was used in the top 2.3 metres of the hole.

The powder factor using this pattern size was quite low, (0.39 kg/BCM) and when initiated produced virtually no fly rock or stemming ejection and there was very little noise. The fragmentation was very good with no oversize and the movement in the shot was very uniform which satisfied grade control requirements (see Figure 11).

Ore mark-up

After the blast has been cleared, water is used to remove/settle dust and to allow any visual characteristics of the ore/waste to be recognised. Ore mark-up tapes are established based on the new locations of the preblast tapes and adjusted to account for visual discrepancies observed by the geologist. Pin flags are also used to indicate which ore category is contained within the boundary.

Digging of the shot

The small bench size does not lend itself to the use of a face shovel, and the rock characteristics suggest significantly hard wear on a loader. For these reasons the shots are dug with back hoe excavators. Flattening of the blasted shots is limited to dozing of material within single ore categories only. As much as possible, blasting is designed to minimise the need for dozing of the top of the blast.

On the five metre deep shots two passes are made, and ore categories remain consistent throughout the vertical profile.

RECONCILIATION

Blackbutt ore is normally blended through the BGM’S basement plant with a number of other ore streams (underground ore, open pit quartz, and other open pit bedrock). This makes grade control predictions and monitoring the occurrence of dilution for any single stream difficult. In late-1994 a batch parcel of ore from Blackbutt Pit was treated to provide some measure of grade control performance and dilution. Results of this trial are shown in Table 2. These are obviously pleasing results.

TABLE 2

Reconciliation of Blackbutt grade control to milling (after Morley, 1994b).

Grade control estimate	Mill assay head results	**Mine call factor**
27 241 tonnes	24 963 tonnes	0.92
2.08 g/t	2.27 g/t	1.09
56 661 grams	56 666 grams	1.00

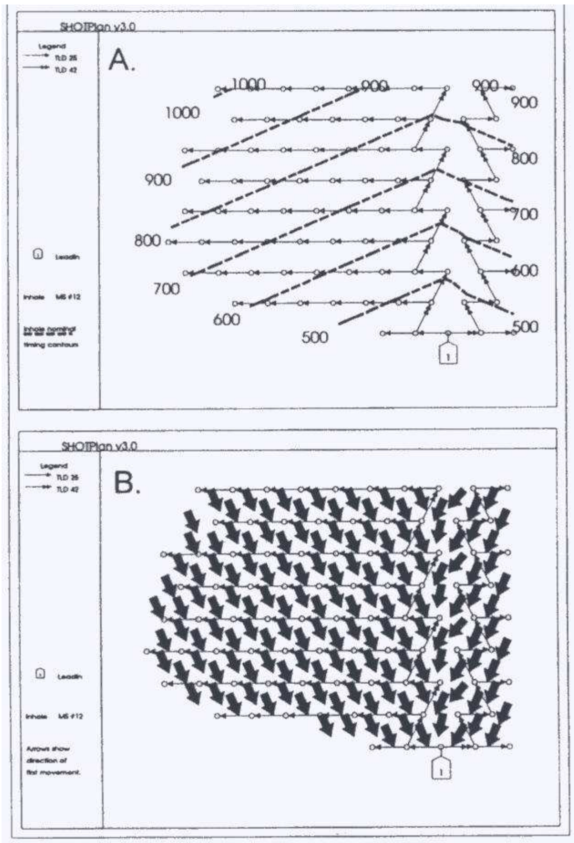

FIG 10 – Models of current blasting practices. (A) plan view of tie in. (B) Plan view of predicted movement of shot (arrows indicate movement).

CONCLUSIONS

Table 3 presents a comparison of the parameters prior to this review with current practices. Significant changes have been made to most practices associated with extracting ore from Blackbutt Pit.

Fundamental to the success in Blackbutt was effective communication from all people involved. Blasting is an integral part of the mining process and must be examined and assessed as part of the total process.

This is by no means the definitive example of efficient and effective blasting. However, the results are worth noting as they represent the practical application of ideas from across a number of areas of expertise. With all parties having input into the final solution cost effective blast results are achieved while keeping geologists, engineers, production personnel, contractors, and of course the owners satisfied.

FIG 11 – Profile of blasted shot using revised blasting parameters (hard hat for scale)

Table 3

Comparison of the parameters prior to this review with current practices.

Parameter	Previous	Current
Hole size	89 mm	89 mm
Pattern	Staggered	Equilateral (staggered)
Bench height	3 m	5 m
Spacing	2.5 m	3.7 m
Burden	2.5	3.5 m
Hole depth	3.5 m	5.8 m
Sub-drill	0.5 m	0.8 m
Initiation consumables	8 Nonel dets 125gm booster 15m/s DRC 65 m/s DRC	Gold dets 125gm booster
Explosive	ANFO (dry) 2560 (wet)	2560
Powder factor	0.64 kg/bcm	0.39 kg/bcm
Stemming	1 m for ANFO 1.8 m for 2560 of drill cuttings	2.3 m Blue metal

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