Application of Blasting Fundamentals to Argue a Case for Limit Blasting


Abstract

There are several methods that a practitioner can use to evaluate a blast design, be it numerical or empirical. However, regardless of the method, the evaluation of actual field results is fundamental in understanding the response of the rock mass to blasting.

Over the last eight years the open pit at Sunrise Dam Gold Mine has benefited from optimised slope and blast designs. This paper discusses the techniques and procedures the mine has used in blast design to optimise the design of 45m high benches having vertical batters. A series of fundamental equations are presented to quantify the various elements of a blast. Techniques are then introduced for assessing a blast and the effects it has on the rock mass. Finally, procedures are presented that have been used to ensure consistency in blast performance and to manage slope stability and rockfall risk.

1 Introduction

In the mining industry there is the perception that limit blasting is an extra unnecessary cost and is time consuming without providing significant benefits. This perception can only be addressed by developing a site specific blasting solution that considers slope stability and production requirements.

There are several methods for evaluating a blast design; numerical and empirical. Regardless of the method used, the evaluation of field results is fundamental in understanding how the rock mass responds to blasting.

Over the past eight years the open pit at Sunrise Dam Gold Mine has benefited from optimised slope and blast designs. This paper describes the procedures used in blast design to optimise 45m high benches having vertical batters. These procedures are based around the use of controlled blasting techniques. Sites that do not use these techniques need to consider the answers to the following questions:

- “Why should controlled blasting techniques be used?”
- “How much can the slope be steepened if controlled blasting techniques are used?”
- “Can the diameter or spacing of the holes be increased?”
- “Who should design the blast layout?”
- “Should the charge weight be modified?”
- “What is the difference in rock mass damage between pre-split and post-split methods?”
- “How can the required fragmentation be achieved?”
- “How will production be affected?”

These questions need to be addressed to facilitate a site-specific solution. However, site-based geotechnical engineers often do not have the prerequisite background in blast design to do so.

Drill and blast personnel understand that:

- there is a limitation to the maximum size rock that the crushers can handle;
• it is more economical to supply the crusher with finer rock; and
• the crusher can be blocked if the average rock size is too large.

From a production perspective, smaller sized rock ensure:
• better dig rates;
• pit floor grades that are easier to achieve and maintain which has a positive impact on equipment and tyre life; and
• a reduction in secondary blasting of oversize rocks which is difficult, hazardous and expensive.

From a drill and blast perspective, the best outcome is that of a finely fractured muck pile. To achieve these aims, drill and blast personnel often use empirical or numerical methods which are often developed by explosive suppliers.

The geotechnical engineer needs to have a basic understanding of these methods if they are to contribute to the blast design process. Doing so enables them to consider the impact that blast will have on the surrounding rock mass.

As each hole detonates, a shockwave is generated, the intensity of which is proportional to the amount of explosive (i.e. the charge weight). Close to the hole the wave is strong enough to pulverise the rock by forming fresh fractures. Further away it has sufficient energy to dilate existing fractures. At greater distances the vibration attenuates to a level only detectable by instrumentation.

The initiation direction and timing between neighbouring holes and consecutive rows dictates the direction and distance the blasted material moves and the direction in which damage is induced. A parallel can be drawn between the detonation front of each row of holes and an ocean wave. In front of the wave, the material is broken and, behind the wave, the broken material is moved backwards from the direction of propagation. A blast design is required to get the resulting batter as close as possible to the final pit limits without damaging the rock mass within the bench and behind.

The amount of rock mass damage that results when attempting to do so is influenced by the charge weight and also by:
• Water; In highly jointed rock masses and where major structures are present the pressure waves and water vapour gasses dilate weak planes;
• Confinement; In an open-ended blast a significant amount of energy is dissipated by movement of material, which does not occur with a confined blast.
• Timing; Each blast hole and row generates seismic waves. The detonation frequency can result in positive wave interference.

2 Fundamental blast design

Usually the production bench heights and hole diameters are dependent on the available drilling and excavation equipment. This dependency forms the basis of most empirical relations used in the industry to define a blast pattern. Table 1 is a summary of these parameters used for various rock strengths (Heilig et. al., 2003). The values in the table only apply to small diameter blast holes (65-165mm), another similar table applies to larger holes (165-450mm).
Table 1. Design parameters for different rock strengths (UCS) related to small diameter (ϕ-mm) blast holes.

<table>
<thead>
<tr>
<th>Design Parameter</th>
<th>Low &lt;70MPa</th>
<th>Medium 70-120 MPa</th>
<th>High 120-180 MPa</th>
<th>Very High &gt;180MPa</th>
</tr>
</thead>
<tbody>
<tr>
<td>[B] - Burden</td>
<td>39 ϕ</td>
<td>37 ϕ</td>
<td>35 ϕ</td>
<td>33 ϕ</td>
</tr>
<tr>
<td>[S] - Spacing</td>
<td>51 ϕ</td>
<td>47 ϕ</td>
<td>43 ϕ</td>
<td>38 ϕ</td>
</tr>
<tr>
<td>[T] - Stemming</td>
<td>35 ϕ</td>
<td>34 ϕ</td>
<td>32 ϕ</td>
<td>30 ϕ</td>
</tr>
<tr>
<td>[J] - Subdrill</td>
<td>10 ϕ</td>
<td>11 ϕ</td>
<td>12 ϕ</td>
<td>12 ϕ</td>
</tr>
</tbody>
</table>

The Kuz-Ram model is often used to assess blast fragmentation. It is based on a Rosin-Rammler distribution (Cunningham 1983, 1987, 2005) and an equation developed by Kuznetsov (1973), which describes the relationship between the mean fragmentation size and the applied blast energy (Powder Factor - PF) as a function of rock type. The original Kuznetsov (1973) equation was adapted by Cunningham (1987) to cater for ANFO instead of TNT. In its basic form the equation is:

\[ X_m = \frac{1}{K^{0.8}} \cdot Q^{1/6} \cdot \left( \frac{115}{RWS} \right)^{19/30} \]  \[ \text{[1]} \]

where,
- \( X_m \) is the mean fragment size (cm);
- \( A \) is the rock mass factor ranges from 4 for low-strength rock to 12 for high-strength rock, (Cunningham, 1983).
- \( K \) is the powder factor or Specific Charge (kg/m³);
- \( Q \) is the total mass of explosive in each blasthole (kg);
- \( RWS \) is the relative weight strength of the explosive expressed as %ANFO.

This equation can be rewritten to calculate the powder factor (K) required for a desired mean fragmentation.

One of the main contributing factors to the rock mass strength in the final exposed excavation is the level of blast damage introduced during the mining process. Although a rockface may appear to have incurred little or no damage on the surface, the vibrations or shockwaves generated during blasting would have induced various levels of damage. The principal driver for these damaging vibrations is the charge weight in each hole (Q).

There are multiple blast design geometries for any given powder factor i.e.

\[ K = \left( \frac{Q}{V_o} \right) \]  \[ \text{[2]} \]

where \( V_o \) is the rock volume broken by a blast hole (m³) = Burden x Spacing x Height of bench.

The Blastability Index (Lilly, 1986) has been adapted to calculate the rock factor A for Kuznetsov’s model to consider density, mechanical strength, elastic properties and structure (Cunningham, 1987) i.e.
\[ A = 0.06 \cdot (RMD + JF + RDI + HF) \]  \[ 3 \]

where,

- **RMD** is the Rock Mass Description (Powdery/Friable = 10; Blocky = 20; Massive = 50). This value approximately equates ½ x GSI;
- **JF** is the Joint Factor = JPS + JPO;
- **JPS** is the Joint Plane Spacing (Close - Average joint spacing \( S_j < 0.1 \text{m} = 10 \); Intermediate - \( S_j < X_0 \) oversize fragment = 20; Wide - \( S_j > X_0 \) oversize fragment = 50);
- **JPO** is the Joint Plane Orientation (Horizontal = 10; Dip out of face = 20; Strike perpendicular to face = 30; Dip into face = 40);
- **RDI** is the Rock Density Influence which is a function of the Relative Density (RD) or specific gravity (tons/m³); If RD > 2 then RDI = 25RD – 50. If RD \( \leq 2 \) then RDI = 1.
- **HF**: Hardness Factor = 0.05 x UCS (uniaxial compressive strength) (MPa).

The Rosin-Rammler formula is used to estimate the fragmentation size distribution i.e.

\[ R_m = 1 - e^{-\left(\frac{X}{X_c}\right)^n} \]  \[ 4 \]

where,

- **\( R_m \)** is the proportion of fragments passing through on a screen/sieve;
- **\( X \)** is the screen/sieve size (mm);
- **\( X_c \)** is the characteristic size which is a scale factor usually for \( R_m = 63.9\% \);
- **\( n \)** is the index of uniformity.

The characteristic size (\( X_c \)) is the sieve size through which 63.9% of the fragments pass. If this value and the index of uniformity (\( n \)) are known, a typical fragmentation distribution curve can be plotted. The index of uniformity describes the steepness of this curve estimated by the equation (Cunningham, 1983) i.e.

\[ n = \left( 2.2 - 14 \frac{B}{D} \right) \cdot \left( 1 + \frac{S}{2B} \right) \cdot \left( 1 - \frac{W}{B} \right) \cdot \frac{L_{\text{Tot}}}{H} \cdot \left( \frac{L_{\text{BC}} - L_{\text{CC}}}{L_{\text{Tot}}} + 0.1 \right) \]  \[ 5 \]

where,

- **\( B \)** is the burden (m);
- **\( S \)** is the spacing (m);
- **\( D \)** is the hole diameter (m);
- **\( W \)** is the standard deviation of drilling accuracy (m);
- **\( L_{\text{Tot}} \)** is the total charge length (m);
- **\( L_{\text{BC}} \)** is the length of the bottom charge (m);
- **\( L_{\text{CC}} \)** is the length of the column charge (m);
- **\( H \)** is the bench height (m).

A thorough QA/QC programme is required to reduce the variance between the planned blast pattern and the pattern produced in the field. The engineer compares the patterns, charge weights, detonation sequences and assesses the muckpile profile and rockmass damage. The results are then used to validate the blast design and back calculate site specific input parameters and constants.
3 Fundamental blast damage assessment

The primary pressure (P) wave and secondary shear (S) wave velocities $V_p$ and $V_s$ respectively are estimated as follows:

$$V_p = \frac{E(1-\nu)}{\sqrt{\rho(1+\nu)(1-2\nu)}}$$  \[6\]

and

$$V_s = \frac{E}{2\rho(1+\nu)}$$  \[7\]

where,

- $E$ is Young’s (Elastic) modulus applicable to the intact rock, (Pa);
- $\nu$ is Poisson’s ratio applicable to the intact rock;
- $\rho$ is the density of the intact rock (kg/m³).

The maximum vibration level or peak particle velocity before tensile failure occurs is estimated by:

$$PPV_{\text{Max}} = \frac{V_p \cdot \sigma_t}{E}$$  \[8\]

where,

- $V_p$ is the P-wave velocity, (m/s);
- $E$ is Young’s (Elastic) modulus, (Pa);
- $\sigma_t$ is the tensile strength of the intact rock (Pa).

Table 2 summarises the velocities and maximum vibration levels per rock type for a sample data set. It lists the minimum vibration level $PPV_{\text{CE}}$ at which crack growth can occur. This value is assumed to be 0.25$PPV_{\text{Max}}$.

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>$E$ (GPa)</th>
<th>$\nu$</th>
<th>$V_p$ (m/s)</th>
<th>$V_s$ (m/s)</th>
<th>$PPV_{Max}$ (mm/s)</th>
<th>$PPV_{\text{CE}}$ (mm/s)</th>
</tr>
</thead>
<tbody>
<tr>
<td>QF Gneiss</td>
<td>84</td>
<td>0.25</td>
<td>6052</td>
<td>3494</td>
<td>901</td>
<td>225</td>
</tr>
<tr>
<td>Dolerite</td>
<td>40</td>
<td>0.25</td>
<td>3777</td>
<td>2181</td>
<td>1152</td>
<td>288</td>
</tr>
<tr>
<td>Garnet</td>
<td>86</td>
<td>0.30</td>
<td>5521</td>
<td>3235</td>
<td>912</td>
<td>228</td>
</tr>
<tr>
<td>Chert</td>
<td>66</td>
<td>0.25</td>
<td>5187</td>
<td>2995</td>
<td>892</td>
<td>223</td>
</tr>
<tr>
<td>Schist</td>
<td>84</td>
<td>0.25</td>
<td>5268</td>
<td>3041</td>
<td>559</td>
<td>140</td>
</tr>
</tbody>
</table>

The following formula can be used to estimate the PPV’s at a set distance for different explosive weights when evaluating the blast impact and by doing so calculate the depths and damage levels that a blast design will imposes on a rock mass.

$$PPV = k\left(\frac{d}{\sqrt{w}}\right)^{\alpha}$$  \[9\]

where,

- $w$ is the explosive weight, (kg);
- $d$ is the distance from the source, (m);
- $k$ is a site constant (In a confined blast k=5000);
- $\alpha$ is a site constant.
Assume blast holes are 10m deep and 165mm diameter and an explosive weight of 205kg. The site specific values are assumed to be $k=3750$ and $\alpha=-1.6$. The resulting tensile failure depth and crack extension depth are listed in Table 3. Blast damage is shown to extend a significant distance into the rock mass, effectively weakening it in close proximity to the established pit limit.

Table 3. Theoretic tensile failure and crack extension zones.

<table>
<thead>
<tr>
<th>Rock Type</th>
<th>$PPV_{\text{Max}}$ (mm/s)</th>
<th>$PPV_{\text{CE}}$ (mm/s)</th>
<th>Tensile failure depth (m)</th>
<th>Crack extension depth (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>QF Gneiss</td>
<td>901</td>
<td>225</td>
<td>35</td>
<td>80</td>
</tr>
<tr>
<td>Dolerite</td>
<td>1152</td>
<td>288</td>
<td>30</td>
<td>70</td>
</tr>
<tr>
<td>Garnet</td>
<td>912</td>
<td>228</td>
<td>35</td>
<td>80</td>
</tr>
<tr>
<td>Chert</td>
<td>892</td>
<td>223</td>
<td>35</td>
<td>80</td>
</tr>
<tr>
<td>Schist</td>
<td>559</td>
<td>140</td>
<td>45</td>
<td>105</td>
</tr>
</tbody>
</table>

4 Influence of blasting on rock mass strength

The shear strength of a highly discontinuous rock mass can be expressed in terms of the Mohr-Coulomb criterion (Hoek, 2002) as defined in terms of a cohesive strength and friction angle. These parameters can be determined as functions of the Hoek-Brown parameters. These parameters include a Disturbance Factor (D) which considers stress relaxation resulting from blast damage.

Following on from the blast damage assessments in Table 3, consider three rock masses comprising a high strength rock (eg. Garnet Gneiss), medium strength rock (eg. Chert) and low strength rock (eg. Schists). The influence of blast damage, on the reduction of shear strength of these rock masses can be considered by varying the D parameter using the RocLab software (Rocscience, 2007). Figure 1 illustrates this variation.

Consider two slopes:

- Slope A involves limit blasting i.e. $D=0.6$. A reduction in shear strength parameters of $c=900\text{kPa}$ and $\phi=4^\circ$ are obtained in a high strength rock mass and $c=400\text{kPa}$ and $\phi=9^\circ$ in a low strength rock mass.

- Slope B involves no limit blasting i.e. $D=0.9$. A reduction in shear strength parameters of $c=1500\text{kPa}$ and $\phi=7^\circ$ are obtained in a high strength rock mass and $c=600\text{kPa}$ and $\phi=16^\circ$ in a low strength rock mass.

The example illustrates that it is possible to control the depth of the blast damaged zone by effective design. It is also necessary to establish procedures to ensure that the designs are properly implemented.
Figure 1. Reduction of rock mass strength due to blast damage.

5 Procedures and controls

In this section the procedures used to ensure consistency in blast performance and controls to manage slope stability and rockfall risks are discussed, with reference to Sunrise Dam open-cut mine (Cowan et al, 2007). At this site the ground control philosophy is to maintain the stability of mine development sufficiently to enable continued safe and efficient production.

5.1 Primary blasting

The drill and blast department designs production blasts one bench at a time with assistance from the geotechnical engineers. Geotechnical input primarily consists of utilising lithological and structural models to identify the major rock units and structures. These models are used for the design of perimeter blasting and to assist with determining the initiation direction by considering the structural characteristics, thereby reducing damage to the final walls.

Presplit holes (Figure 2) are 127mm diameter and drilled on a 1.3m spacing (i.e. ~10\(\phi\)). They are loaded using a 25mm packaged emulsion. The single pass pre-splits are normally drilled over three 7.5m high production benches that combine to form a 22.5m bench. In areas of the pit where higher benches (30m or 45m) can be achieved, the first 7.5m or 15m presplit is drilled and fired. Then the adjacent 1 x 7.5m or 2 x7.5m production shots are drilled, fired and excavated. Finally the standard 22.5m single pass presplit is completed to form the next three production benches. A small lip may occur where the two presplit interfaces meet; however, the effect is negligible where an offset of 1m or less is achieved. Pre-split rows are fired prior to the production rows.

The production blastholes are 165mm diameter and drilled through a single 7.5m bench. The holes are drilled on a regular staggered pattern that is aligned with the mine grid. They consist of 127mm diameter vertical batter holes and two buffer holes in conjunction with 165mm diameter production holes. The batter and buffer holes are drilled along lines parallel to the final wall. On the bench above a berm, modified (shortened) production holes are drilled where the holes are within a defined distance above the crest to be formed below. Doing so
reduces the likelihood of incurring crest damage and later crest loss during excavation. The holes are loaded with bulk emulsion delivered down the hole. Blast initiation takes into account the direction of muckpile throw to minimise ore dilution and firing direction to protect the final walls. The sequencing is designed by using software and firing is done remotely with a radio system.

Perimeter blasting generally result in high quality final walls, However, the structural orientations can impact the final result as illustrated in Figures 3 and 4.

Figure 2. A typical site specific 22.5m blast hole layout & sequence for the hardrock design (not to scale).
5.2 Secondary blasting

Localised secondary blasting has proved to be a cost effective and successful way to eliminate hazardous bench scale wedges. The key aspect is the identification of adversely dipping structures in the initial 7.5m face height. In order to develop a suitable blast hole design and predict the extent of any unfavourable wedges that may be adversely impacted by the blast, the geotechnical engineers use a combination of face mapping data, structural interpretation and extrapolation to construct representative 3D numerical models for site specific stability analyses.
Temporary jump-up access pads are constructed for blast hole rigs in locations where wedges are to be removed along final wall limits. Designated holes are drilled, charged and fired before wall scaling commences. If wedges are not recognised in the initial 7.5m bench, the necessity to construct jump-up ramps from lower benches becomes costly and may impact significantly on the production schedule.

5.3 Wall scaling

After mining the three flitches comprising the 7.5m production bench, the wall is scaled to expose the presplit barrels and remove loose material on the crest and batters.

Wall scaling forms an integral part of the production process and hence is only done by experienced operators. Scaling uses a dedicated small excavator (Komatsu PC 1100SP - 110 tonne) equipped with a hardrock toothed bucket. Larger excavators (e.g. 180t and 360t) have been found to damage the batters.

Scaling is conducted in multiple passes, generally working at a 45° angle from the wall depending on the dip of the structures and the locations of blocks on the crest. A hydraulic rock breaker is used to remove wall flares and frozen toes in front of the presplit line.

In highly fractured or sheared ground, a water cart equipped with a high-pressure water cannon is used on the crest and batter to remove smaller rocks left behind after the excavator has completed scaling.

5.4 Bench inspection

The final wall inspection and formal signoff procedure for all active mining faces commences once a production blast pattern adjacent to a final wall has been blasted. A bold green hatched line is digitally placed on the daily production plan to signify changes to the final wall limits. This plan is prepared from geology, engineering and geotechnical data and is distributed to all production supervisors and technical disciplines. It forms the basis for discussions at the daily morning production meeting at which the geotechnical engineer raises any concerns about particular areas.

After scaling, the wall is inspected by a geotechnical engineer to determine if any additional blasting, scaling or alternate remedial work is required prior to permitting any further drilling or mining activities in the immediate area. Following approval by the engineer, the general mine foreman completes the formal signoff on the daily plan for the area. The green hatched line is removed from the plan and production can continue with personnel and drill rigs able to approach the final wall limits.

5.5 Hazard zoning

Two types of visual aids are used to demarcate wall control related hazards in active working areas.

Green cones are used to restrict access on berms below unscaled final walls. The geotechnical department places the cones before scaling commences. The location of the cones is replicated as the bold green hatched line on the daily plan. Only approved equipment (scaling excavator and water carts) are allowed into these areas and the access must be associated with a visual Job Hazard Assessment (JHA) form which considers the level of risk.

Symbols and arrows are painted directly onto the face to target areas where specific remedial work is required. Typically letters are used to show $F$: frozen material, $S$: shotcrete required, $W$: water cart required and $B$: secondary blasting required.

Post excavation amelioration work is required in areas where there is an increase in rockfall hazard due to excessive crest loss, adverse ground conditions or local design change. Options considered include:

- Catch fences – Several 2 m high x 50 kJ Geobrugg Tecco catch fences at ~AUD$300,000/100m
- Shotcrete – Shotcrete or fibrecrete has been used on crests due to the availability of underground equipment at AUD$20,000/100m
• Draped mesh curtains – Free-hanging Rocklink mesh curtains are used to reduce rockfall risk under areas over the high wall that are inaccessible at AUD$4,000/100m

• Energy dissipation pads – This low cost option reduces energy of falling rocks and prevents them from travelling further down the slope. Installation requires sufficient space being available on a bench.

6 Conclusions
This paper summarised some of the theory, techniques, procedures and controls used by the mining team at Sunrise Dam to achieve a final pit depth of 460m with 59° inter-ramp (stack) angles over 180m high. High quality blasting procedures and an understanding of the impact that blasting has on the rock mass has enabled benches at the base of the pit to be operated with 45m heights. The procedures have included:

• Consistent, good quality presplit blasting of the final walls to ensure the optimum catch capacity of the benches, to minimise the rockfall risk and to achieve the final slope design. Doing so has required a rigorous QA/QC programme.
• Batter angle drilling has aimed for consistency to prevent/reduce the risk of undercutting.
• Domain plans show boundaries between areas having different rock mass strengths and major structures. These plans influence presplit and pit limit blasting designs.
• Presplit and pit limit blasting trials have been carried on interim pit walls to provided valuable information as to how a particular blast design will impact the rock mass and how structures having various orientations will respond to blasting.
• Grade control holes are designed so they do not intersect the final pit limit to restrict the ingress of gasses from production blasts. Crest loss and batter damage has been evident in areas where these holes have intersected production holes.

Figure 5 shows the western high wall where the bench heights have been increased from 22.5m to 45m.

7 Acknowledgements
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8 References


Figure 5. Evolution of bench heights from 22.5m to 45m vertical.


