Slope Stability 2013 – P.M. Dight (ed)

doi:10.36487/ACG_rep/1308_93_deGraaf

Blast induced damage mechanism on final walls and the blasting methods to minimise damage

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Abstract

At most open pit mines, profitability is affected appreciably by the final slope angles of the pit. Steep stable and safe pit walls can be formed by accurate final wall drilling and blasting techniques. The violence of the blasting process both leaves damaged surfaces that must be made safe, and breaking beyond the desired limit will increase mining costs. Much can be done to significantly reduce the damage, although the geological conditions and geometry of excavation ultimately limits success.

The aim is to make the transition from well fragmented rock to an undamaged slope or wall in as short a distance possible. The success of final wall blasting techniques depends on the aims being achieved despite the conflict of purpose that may exist in the mine's production environment. Various drilling and blasting techniques are applied to minimise the impact from blasting that causes damage on the perimeter wall. These techniques have a common objective to minimise fracturing and loosening of the rock beyond the excavation line, and include line drilling, pre-splitting, buffer blasting and trim blasting.

In final wall blasting, the degree of confinement of the explosives energy adjacent to the slope plays a key role in the amount of wall damage produced. The energy of the explosives should be directed away from the final wall. Avoid the false notion that the explosives energy must always be minimised to limit blast damage to the walls. The challenge remains to apply explosives energy in ways that limit damage to the walls, but produce the required fragmentation of the ore to enable high productivity. Perimeter blasting is about techniques to limit damage to the walls, by creating split surfaces to define boundaries and terminate crack development, reducing energy against the back wall, and diverting it away from the perimeter zone.

1 Introduction

The creation and integrity of stable pit walls is a primary concern in every mining operation. The safety of personnel and equipment is crucial to all operations and they cannot be required to work near ambitiously steep, unstable walls. On the other hand, conservative low angle walls incur significant cost penalties through unfavourable stripping ratios, making the mining operation uneconomic.

A key characteristic of explosives detonating in a blasthole is that the shockwave portion of the energy is transmitted radially outwards into the surrounding rock mass independently of direction. For standard blasting this is of no serious consequence, and in some cases this is desirable since the objective is to fragment the rock mass. The breakage beyond the blastholes is often referred to as 'free muck'. This however, becomes a problem when blasting in close proximity of the final wall. What is considered as free muck in the context of production blasting becomes over-break in the context of final wall blasting, which can have very expensive and undesirable consequences.

The blast damage transition zone between the blasthole and the wall perimeter should be as small as possible. The width of this zone depends mainly on the final wall blasting practices. As this zone increases in size, the resulting slope angles become flatter, reducing the catchment area. The overall result means excessive scaling, crest loss and impact on production.

It is unreasonable to expect the production blast adjacent to the final wall to fragment and heave the ore material adequately and yet leave the perimeter material undamaged, intact and competent enough to be

stable for years. Most mines adopt a final wall blasting strategy to deal with this situation. The success of perimeter control depends on the aims being achieved despite the conflict of purpose that may exist in the mine's production environment. The success of the final wall is a function of the cost, time and effort put into the appropriate design and implementation thereof.

It should be noted that optimising any of the techniques in isolation will not produce the optimal outcome. For example, a perfect pre-split can be obliterated or rendered ineffective by a careless buffer row or a poorly executed trim blast. Conversely, no amount of timing and energy optimisation in the trim blast can rectify an incomplete an ill-designed pre-split. The continuous improvement process must apply to all the individual components of a final wall blasting system for optimal results.

The success of these techniques depends primarily on the prevailing geotechnical conditions of the rock formation being blasted. In hard massive rock, modified production blasting techniques should produce satisfactory results. In less competent rock formations, for example highly jointed, specialised blasting techniques are fairly successful. In loose unconsolidated formations that do not support themselves, consistently good results may not be possible. As far as practical field applications are concerned, there is still a large element of trial and error involved. This is not surprising considering the geological variables involved in blasting.

2 Final wall control factors

To develop efficient designs, one should have the basic understanding of wall failure mechanisms and understand the limitations of the various wall control procedures. It is of utmost importance that the design be precisely implemented, evaluated and refined on a continuous basis.

The time, cost and effort spent in developing and implementing efficient designs are insurance against future wall failures. According to the ISEE Blasters' Handbook (International Society of Explosives Engineers, 2011), the four main site factors that control pit wall stability include:

- Geology and ground water conditions.
- Slope design.
- Life expectancy of the final slope.
- Value of the excavation.

The geology of the excavation will influence both the slope and the blast designs. It is very important to pay close attention to the prevailing geological conditions of the wall at the blast site to develop blast designs that will limit the damage. The key geological factors are the rock mass structure and the strength. The strength of the rock mass under shear, tensile and compression loading will also dictate the overall stability of the slope. As mining progresses and the pit gets deeper, the final slope design could be modified as more site specific data is gathered and the slope behaviour is understood. In some cases the mine plan changes will also alter the slope design. The slope design parameters include the overall slope height, bench face height, face angle and the static load on the rock mass. Some of these parameters are directly linked to the blast design and these influence the stability of the pit wall.

The blast design parameters that influence the pit wall stability include the blasthole layout, timing, firing sequence, energy concentration adjacent to the final wall and the blast size.

A major factor that controls the wall stability is the quality control and effort by the operation, which includes and is not limited to the face preparation, clearing the muck pile in front of the final wall blast, accurate drilling, charging practices and the accurate initiation sequence of the blastholes.

3 Blast induced damage mechanisms

To better understand what happens during the detonation process and how this process affects the surrounding material, a brief description is given.

A chemical reaction occurs within the blasthole where the solid or liquid explosives are rapidly changed into expanding hot gas. This reaction starts at the initiation point, either through a detonator or primer, situated in the hole. A detonation wave propagates throughout the explosives column as a convex shock wave and interacts with the blasthole wall. Ahead of this reaction zone is undetonated explosives and behind the reaction zone is rapidly expanding hot gasses. The velocity of detonation (VoD) is the rate at which the explosive shockwave detonates throughout the explosive column. A relative slow VoD will apply the energy to the blasthole over a longer period and faster VoD explosives will apply the energy quicker to the blasthole wall but for a shorter period of time. Explosives with a high VoD have a higher shock energy component compared to the gas (or heave) energy component. Conversely, low VoD explosives will generate a higher gas component compared to the shockwave component.

The degree of coupling between the blasthole wall and the explosives will have an effect on how efficiently the shockwave energy is transmitted into the rock mass. For example, bulk explosives will result in better transmission of energy compared to cartridge explosives or de-coupled bulk explosives.

The confinement pressure that builds up in the blasthole depends not only on the explosives' composition but more so the physical characteristics of the rock. Strong competent rock will result in higher pressures compared to weathered material. When the shock wave reaches the blasthole wall the fragmentation process begins. This shock wave which starts out as the VoD of the explosives, decreases rapidly once it enters the rock mass and in a short distance is reduced to the sonic velocity of the rock mass. The compressive strength of the rock mass is typically in the order of 10 times greater than the tensile strength. This means that approximately 10 times more energy is required to crush the rock compared to pulling it apart. When the shock wave first encounters the blasthole wall, the compressive strength of the rock is exceeded by the shockwave and the zone immediately surrounding the blasthole is crushed. As the shockwave radiates outward at declining velocity, the intensity drops and the compressive crushing stops.

The radius of this crushing zone varies with the compressive strength of the rock and the intensity of the shockwave, but seldom exceeds twice the diameter of the blasthole. Beyond this crushed zone, the intensity of the shockwave is still above the material tensile strength resulting in radial cracking. The high pressure hot gas, following the shockwave, expands into the radial cracks and the existing cracks, extending them further. This is the stage where most of the fragmentation process takes place followed by the movement of the rock mass.

As soon as the compressive and tensile stresses caused by the shockwave drop below the tensile strength of the rock, the shock wave becomes the seismic wave that radiates outwards at the sonic velocity of the material through which it passes. At this point it no longer contributes to the fragmentation process, however it creates shaking in the surrounding rock mass. Over an extended time this may result in the dislocation of the remaining intact rock.

4 Minimising blast damage on the wall

When considering ways to reducing unwanted damage behind the perimeter holes, the three abovementioned mechanisms need to be addressed. There are four different possibilities presented from the detonation of an explosives column:

- The length of the longest radial cracks emanating from the blasthole depends on the blasthole pressure. The blasthole pressure is the pressure exerted on the rock wall prior to the rock wall being crushed. The blasthole pressure is generally considered to play a dominant role in the displacing of the rock during blasting.
- The lengths of the cracks have also been shown to be proportional to the diameter of the blastholes. With a reduction in the blasthole diameter, the damage zone will be reduced limiting the blast transition zone.
- The formation of new cracks around the blasthole depends on the detonation pressure. The detonation pressure is associated with a detonation wave moving through the explosives column

and is mainly responsible for the intense compressive cracking near the blasthole. A reduction in the detonation pressure will reduce the cracking.

- The presence of pre-existing cracks at near perpendicular angles to those being created or extended by the blast, causes the blast cracks to be terminated.

For a multi blasthole blast, the importance of timing sequence cannot be ignored. The flexibility of the timing design will assist the blasting operator in minimising the damage on the final wall through directing the energy away from the wall. This is achieved through two actions:

1. The direction of the thrust of the detonating blastholes should be parallel to the final wall and not perpendicular to the wall. This will also assist in the clearing of the rock sticking to the final wall, leaving a clean face.
2. The delay between holes needs to be sufficient to allow for the relief of the rock mass thus reducing the choking effect which in turn creates unwanted energy directed into the final wall.

5 Final wall blasting methods

Specialised blasting methods aim to reduce the energy transmission into the final wall and have been around for many years and applied with great success. The techniques described below are not a cure-all, and a continuous improvement final wall blasting control program should be implemented and closely monitored. The challenge remains to apply explosives energy in ways that limit damage to the final wall with little or no adverse impact on fragmentation and productivity.

Figure 1 is an example of the poor quality of final walls where perimeter wall blasting techniques were not practiced, which contrast with Figure 2 where all efforts were made to apply appropriate final wall blasting techniques.

Figure 1 No final wall blasting techniques followed (Photograph courtesy A. Karzulovic)

Figure 2 Final wall blasting techniques followed (Photograph courtesy A. Karzulovic)

5.1 Line drilling

Line drilling, as shown in Figure 3, involves a single row of closely spaced small diameter holes drilled along the perimeter of the excavation, providing a weak plane into which the adjacent blast can break (de Graaf, 2010). These holes are usually less than 75 mm in diameter and the spacing is two to four times the diameter of the hole. Drilling accuracy is very important for good results. Any deviation from the planned position will have an adverse effect on the final result. The blastholes adjacent to the line of holes are usually more closely spaced than the rest of the holes in the pattern. In some applications the holes can be lightly charged compared to the bulk of the blastholes. The best results are obtained in homogeneous rock formations with a minimum number of natural discontinuities. In jointed rock formations, pre-splitting will give better results. The disadvantages of line drilling include high drilling costs, very time consuming due to excessive drilling, and a slight deviation in drilling accuracy will cause poor results. This method of perimeter control is not used frequently in surface mining operations.

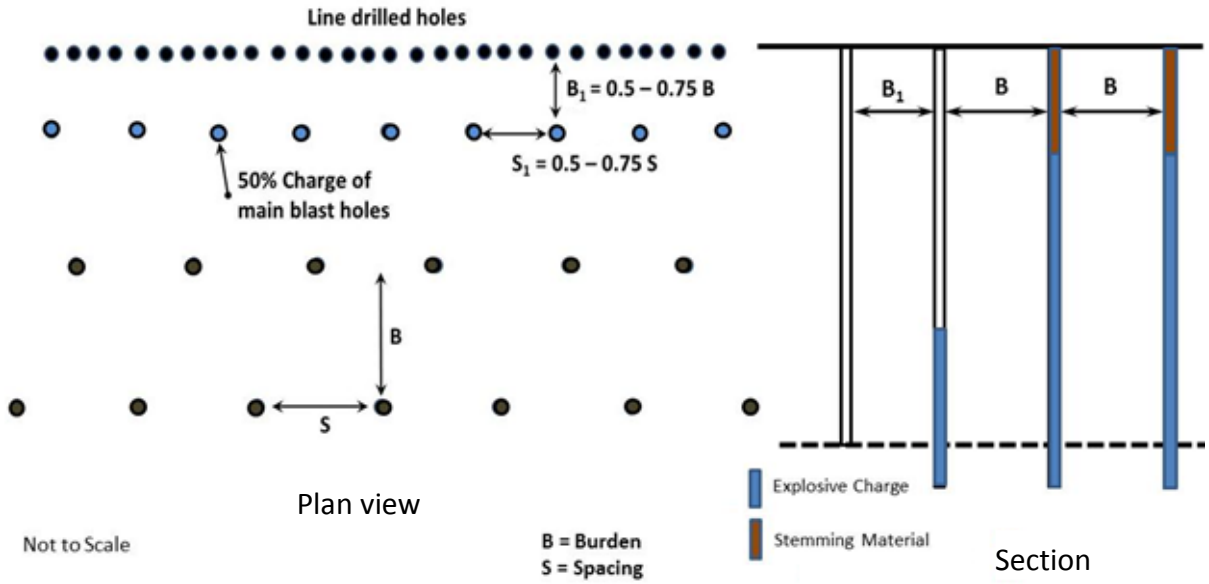

Figure 3 An example of line drilling and adjacent blasthole pattern

5.2 Pre-splitting

This is the most successful and widely adopted blasting method for perimeter control. The idea behind pre-split is to isolate the blasting area from the remaining rock formation by creating an artificial crack along the wall perimeter. A line of closely spaced holes is drilled along the perimeter. These holes are lightly charged and initiated simultaneously before the main blast (St J Tose, 2006). The presplit blast transmits compressive shock waves, which creates a zone of tension between the holes when they collide that fractures and shears the rock. For this reason it is very important that the charges are detonated simultaneously. For the best results, detonating cord or electronic detonators should be used to initiate the charges.

Figure 4 shows a schematic diagram of the split formation in the solid rock mass through instantaneous initiation of closely space holes drilled at the perimeter.

The majority of the shockwaves from the subsequent main blast are reflected against the pre-split face, preventing them from being transmitted into the remaining rock formation, thus reducing damage to the final wall. Drilling precision is very important and even small deviations may adversely affect the final pre-split result.

In incompetent rocks, the result may be improved by drilling guide holes between the charged holes to promote the split along the intended perimeter. Unloaded guide holes between the charged holes give better final result in all rock formations, but are seldom used due to the increased drilling costs and time to drill the additional holes.

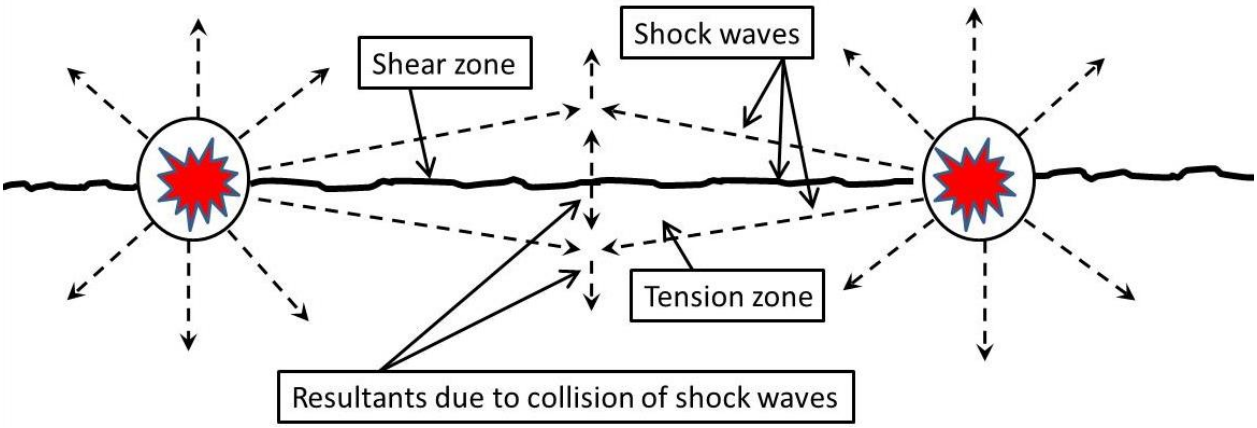

Figure 4 Pre-splitting theory illustration (St J Tose, 2006)

Ground vibration, airblast and noise are the main environmental concerns during pre-splitting. Short delays may be placed between batches of blastholes to reduce the ground vibration and airblast.

Generally the holes are not stemmed to reduce the restriction of the expanding gasses entering the final wall. However, in environmentally sensitive areas a small stemming plug may be inserted or the blast may be covered with old conveyor belting to reduce the airblast and noise.

Figure 5 shows an example of a final wall with the blasthole barrels clearly visible. The half barrels clearly show the accuracy of drilling.

5.3 Buffer blasting

Buffer blasting is carried out with the main production blast. One or two rows of blastholes adjacent to the pre-split or final wall are charged lighter and the delay timing is increased to fire just after the bulk of the main production blastholes. The explosives concentration in the buffer blastholes is low enough to avoid crushing damage to the final wall, yet the explosives is distributed to ensure adequate breakage and removal of loose rock from the pre-split plane. Figure 5 shows an example of a one row buffer line which

was fired shortly after that of the production holes. The buffer row holes were left un-stemmed to reduce the gas energy build-up in the buffer row.

Figure 5 Pre-splitting practice in the field

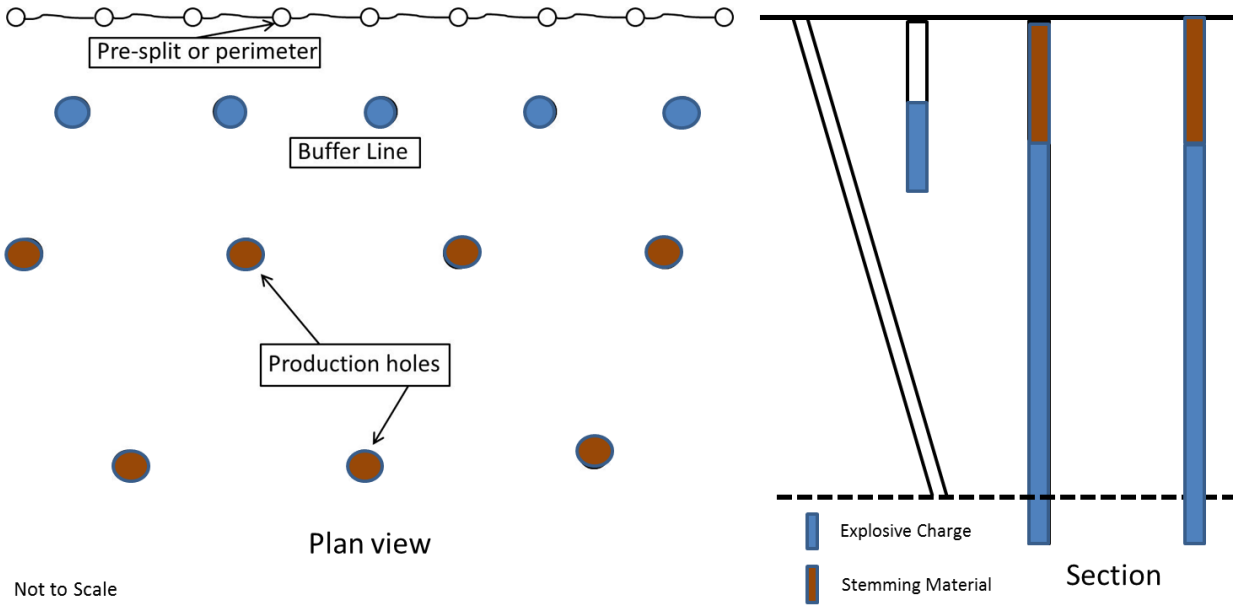

Figure 6 Buffer blast showing one row of buffer holes, lightly charged and unstemmed

5.4 Trim blasting

A trim blast is a relatively small blast designed to remove the final rock mass adjacent to the final wall (International Society of Explosives Engineers, 18th Edition). It is usually fired against the pre-split or the planned wall perimeter. The blast consists of two to four rows of holes with different drilling and charging layouts. Smaller diameter holes are often drilled on a smaller burden and spacing and the holes are drilled to grade, and in some instances short of grade. The explosive loading procedures are different in that the holes contain less explosive and one to two rows of blastholes adjacent to the wall are not stemmed. The explosives gasses are allowed to escape through the collar of the blasthole to minimise the damage to the final wall.

The powder factor is fairly similar to that of the production blast to maintain consistency in the fragmentation throughout the blast. The blast timing is controlled through single-hole firing, and maintaining a thrust parallel to the new perimeter wall. Figure 7 shows a three row trim blast, using smaller diameter holes compared to the production blastholes. The row closest to the pre-split blastholes (final perimeter) are lightly charged with no stemming added in the collar area of the blastholes. The holes are also drilled to grade to reduce the breakage below the grade, thus maintaining the integrity of the berm on the lower bench. The adjacent rows of holes are stemmed as per conventional blasting practices.

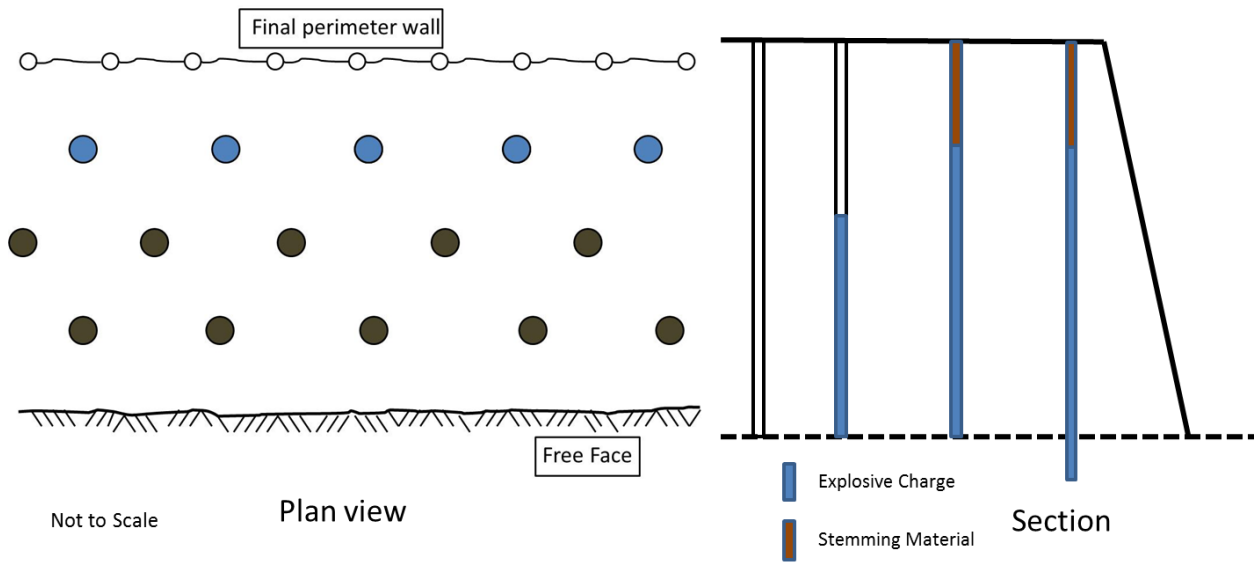

Figure 7 Trim blast showing three rows of holes

6 Case studies

The busy production schedules at many large open pit operations often result in a lack of focus on final wall blasting techniques. This results in unsafe highwalls with localised failures and in extreme cases major failures.

Mine 1 (Figure 8) is characterised by a hard competent rock mass with densities typically 3,000 kg.m^{-3} and hardness values typically 150–200 MPa uniaxial compressive strength. The rock material is well jointed; joint planes being planar in nature, with little infill. The structural environment is typical to an igneous mafic layered intrusion, and comprises of many sub-horizontal joint sets, but also some vertical to sub-vertical sets. The stability of the rock slopes is structurally controlled, with the orientation of the joint planes with respect to the pit orientation being the key factor. The predominant mode of failure is a combination of wedge and planar failures, depending on the local geometry.

In a structurally controlled slope, it is imperative that an effective wall control technique is adopted, even if the rock itself has a high strength. Figure 8 shows the result of no wall control blasting. The energy from the production blast resulted in a damage zone and loose or unstable blocks of rock occurring on the highwall.

Towards the left of the photograph, the orientation of the joints with respect to the pit wall suggests that small scale toppling failure could occur where the sub-vertical inward dipping joints are intersected with sub-horizontal planes. A combination of the free surface, stress relaxation, and aggravation from the production blast allows the joint planes to open and continuous spalling of the rock mass to occur in the damaged zone. This leads to compromised safety with an increased risk to people and equipment on lower benches by falls of ground. It also results in a compromised catchment berm for any falls of ground from the benches above.

Figure 8 Final wall – no specialised blasting techniques were followed. Left of the photograph shows potential failures which may occur after some time

The catchment berms are designed to an appropriate width in the bench design in order to catch falls of ground based on the original in situ rock mass characteristics. The bench design incorporates an adjustment factor in the Laubscher MRMR system (Laubscher, 1990) to accommodate different blasting practices, but the factors can be increased in order to maintain the maximum slope angles and to minimise waste stripping. Loss of crest width due to a combination of spalling in the damaged zone, and back break/ over break from lack of control of blast energy decreases the effectiveness of the catchment berms.

Hard rock massive pits typically have much steeper design slope angles due to the strong competent rock mass. The bench is designed more aggressively in order to achieve these steep slope angles. Catchment berms are in the order of 5–8 m, and it is imperative that controls are put in place to protect the brittle jointed rock mass, minimise the damage zone and to protect the crest of the bench to maintain full catchment width.

Mine 2 (Figure 9) shows the outcome from final wall blasting, however with mixed success. It also represents a hard competent rock mass as discussed at Mine 1 in a layered mafic igneous complex. The same structural mode, and key slope stability factors apply. In this case, the orientation of the joints is outward dipping into the pit, and has flatter angles relative to the high wall. The wedge planes therefore daylight into the pit, and is also intersected by a sub-horizontal joint set. The cohesion of the joints making up these wedge formations is decreased with disturbance due to blasting, and results in the blocks failing, and further loss of the catchment berms. In the case of a dominant joint set in a trim blast that is critical to the stability of the highwall, it is important to change the timing of the blast to propagate the energy in the most favourable direction relative to the direction of the jointing.

On the lower bench some half barrels are visible. The rock mass has been damaged through blasting practices and in particular the sub-drill from the upper benches causing reduced berm widths.

Figure 9 Final wall with specialised blasting techniques was followed with partial success

The waste rock from Mine 3 (Figure 10) is in basalt, with good rock mass conditions. Overall rock mass rating (RMR) values are between 65 and 75, and a mining rock mass rating (MRMR) of between 50 and 65. The intact rock strength varies from 120 to 160 MPa, with densities averaging 2,600 kg.m^{-3}. The main contributing factor to instability is the kinematics and associated joint strengths and orientations. The rock mass is essentially massive, with joints, typical to a large volcanic flow environment, forming blocks and wedges (sub-horizontal flow boundaries as well as vertical to sub-vertical orthogonal jointing).

Pre-split blasting was used as a control to prevent damage to this competent brittle rock mass (Figure 10). It has not been very effective, and a cumulative loss of catchment berm has resulted in what appears as a double bench effect. Poor trim blast techniques have left a portion of the ground frozen against the highwall with unknown integrity, as a split was originally created behind the frozen face. This creates a fall of ground risk for future benches below. The toes also present a problem, as the net result is a decreased catchment berm width, and therefore a reduced catchment berm carrying capacity. Combined with the crest loss due to excessive over break and damage zone, the catchment berm is almost non-existent. Significant width has been lost and the purpose of the slope design has not been achieved.

Figure 10 Final wall – pre-split not effective and the catchment width significantly compromised by crest damage and toe formation

Figure 11 illustrates an attempt that has been made to salvage the loss of catchment width by marking holes to be blasted at the toe of the slope in an attempt to remove frozen ground. This can result in an even further damaged and weakened rock mass at the toe of the slope.

Figure 11 Final wall – attempts made to remove the frozen ground from a poor trim blast against the pre-split, which can result in further damage at the toe of the slope

7 Conclusion

No component in the final wall blasting program can be optimised in isolation; the only effective approach involves consideration of pre-splitting, production blasting and trim blasting as a whole.

The rock mass structure can be the most important influence in the success or failure of the final wall blasting program and cannot be ignored.

The geometry of the blast needs to favour energy release away from the critical final wall. Blasthole timing influences this by directing the ground reaction away from the perimeter wall.

It is not possible to completely prevent blast-induced damage to the final wall due to the destructive nature of the explosive charge detonation. Simple and cost effective means have been described to minimise this. The aim is to prevent such damage from compromising the safety of the operation or adding significantly to the cost.

Rules of thumb, guidelines and computer tools can provide useful guidance but are not definite answers. There is no substitute for thorough and consistent field trials and experience. Close monitoring of blasting results is crucial to achieve sound designs for each domain of final wall blasting. The designs should be continuously refined to strive for continuous improvement of the final wall blasting technique for an effective and efficient final wall control programme.

An efficient assessment program must be developed that involve engineers, geologists and miners, and adherence to input in order to maximise the return from the field trials. A successful final wall control programme can only be achieved if all personnel involved give total commitment, with the overriding consideration being the overall slope stability and worker safety.

Acknowledgement

The authors acknowledge the support and assistance given by University of Pretoria and AEL Mining Services in the development of this study and approval to publish this paper. In particular, the assistance of Ewan Sellers of AEL Mining Services is much appreciated.

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