Controlled blasting for enhanced safety in the underground environment

by E.J. Sellers*

Synopsis

Good blasting practices based on the careful application of explosive energy lead to significantly safer mining operations. Case studies demonstrate how the selection of the correct charge mass, explosive type and round design are vital for extracting the exact amount of rock required and preventing spurious damage to the remaining rock mass. In other cases the explosives are applied to precondition the rock mass, design rockburst resistant support, prevent damage to important excavations and reduce the exposure of miners to unsafe conditions. The Hybrid Stress Blasting Model is used to understand the explosive rock interaction and to illustrate and contrast good and bad blasting practices. In all cases, good supervision is required to continue safe mining according to the design.

Keywords

Blasting, safety, production, preconditioning, explosives, numerical modelling, drop raising, vibrations, and presplitting.

Introduction

Blasting is a daily practice on the mines and provides the most economical and flexible means of accessing the orebody in a hard rock mine. Because blasting is generally seen as a production activity, little thought is often given to the ways in which good blasting can improve the safety performance of a mine. Studies from the airline industry identified the phenomenon of inattional blindness (Grandin and Johnson, 2005) where professionals do not notice hazards that are immediately obvious to others. This often applies in the mining industry where lack of care and attention to detail, in what is essentially the main production activity, diminishes both the productivity and the safety of the operations. Even if the safety benefits are recognized, it is often difficult to demonstrate the potential hazards that can be formed from poor blasting practice and the possible hazard mitigation measures that can be implemented by good practices and innovative blast designs. In this paper, a number of ways in which good blasting can assist the safety performance of a mine are discussed. Many of these are well known and have been applied with varying success over many years. However, improvements to safety are an important contribution to the industry so it is worthwhile to collate examples from a number of underground projects to demonstrate how improved blasting can enhance safety.

To educate miners about the effects of blasting often requires that the poor practices are contrasted with better options. This is not always possible in practice for safety reasons and due to production pressures. So, in order to illustrate some of the concepts, the newly developed Hybrid Stress Blasting Model (HSBM) is applied to create simple three-dimensional dynamic models of blasting situations that are able to illustrate the safety benefits in an easily accessible manner. The HSBM (Furtney *et al.*, 2010) uses a lattice of particles to represent the rock. Explosive input parameters derived from the AEL detonation code Vixen2009 are applied to generate detonation waves that dynamically interact with the rock particles to predict fracture growth, fragmentation and heave.

A number of blasting practices are considered in this paper and are illustrated using case studies from AEL Mining Services projects where possible. The studies illustrate that blasting can be used to improve conditions and reduce associated hazards, to develop hazard mitigation measures and to reduce the exposure of miners to the hazards, all of which will reduce the safety risks of underground mining.

Importance of correct charging for hangingwall conditions and support

The explosive ANFO is used in many mines due to its ease of use. However, due to the

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loading of the product with pneumatic Lategan loaders there is a temptation to overfill the blastholes as it seems obvious to miners that more explosives will provide better breaking. A study was undertaken to investigate the effect of changing from ANFO to Powergel 813 cartridge explosive (Slabbert, 2005) and it was discovered that the prevailing poor conditions resulted from overcharging the blastholes. Photographs taken of the hangingwall before and after blasting with cartridge explosive are shown in Figure 1. The hangingwall conditions depicted in Figure 1a show the effect of overcharging with ANFO. The blast induced fractures have extended above the hangingwall contact and created a jagged, 'factory roof' effect. The photographs show that the miners have found it difficult to position the support units correctly. The units are not perpendicular to the reef and parallel to each other as would be expected in good support practice. The chances of fall of ground incidents are significantly increased due to the positioning of support on the ragged protrusions from hangingwall, which means that there is no consistent support pressure and the spacing is variable. There is also increased chance of support failure due to buckling as a result of the units not being loaded axially.

Once the charging was done correctly, because the use of cartridged explosive encourages the placing of just the correct amount of explosive in each blasthole, the hangingwall conditions improved significantly as shown in Figure 1b. The hangingwall becomes a flat plane allowing the support units to be installed properly. Thus, there is a double benefit from the lack of loose rocks above the stope and the improved support capacity.

Effect of explosive characteristics and jointing

It is often mentioned in the industry that ANFO is a substantially more 'gassy' explosive than emulsions or watergel and is thought to cause longer cracks and hence damage the hangingwall further. This is in some senses true as ANFO has a lower detonation pressure and a slower delivery of energy than cartridged explosives. This results in less expansion of the blast hole by the shock wave and more energy remaining for driving crack growth and heaving of the fragments that have been created (Cunningham *et al.*, 2006). The longer the driving force acts on the borehole the further the fractures will grow (Sellers and Napier, 2006). However, it is important to note that the densities of ANFO, watergel and emulsion packaged explosives are all different and they deliver different amounts of energy at different stages during their reaction process.

To illustrate the dangers of considering only the charge mass, some results from a project done by COMRO and the CSIR (Toper, 1995) are presented in Figure 2, which shows the degree of fracturing observed for three cases of the same amount of explosive in a 93 mm hole ahead of a development end. Even though the mass is the same, the charge length, applied pressure and rate of delivered energy are quite different and this is reflected in the different fracture extents. Note also that the fracture directions are affected by the prevailing *in situ* stress. The safety of the excavation will not only be determined by the blast induced fractures, but also by the blocks of rock formed by the intersection of the fractures and the slips on pre-existing jointing. Thus, the formation of adverse hangingwall conditions in any given mining scenario can be altered to some degree by the correct choice of explosive type, hole size and round design, but may require more significant changes in mining direction or method. These issues must all be considered for properly controlled blasting.

Limiting of overbreak with good perimeter blasting

Cautious blasting practices such as presplitting and smooth blasting are well known (e.g. Persson *et al.*, 1993), but not always applied in deep level mines. There is a tendency to consider mine tunnels as temporary access ways, but how many mines are using tunnels that are thirty or even fifty years old? In presplitting a line of holes, at closer than normal spacing, is charged with a decoupled charge and set off simultaneously prior to the main blast. The HSBM prediction of the development of a substantial damage zone surrounding line of 5 holes spaced at 0.5 m, but using fully coupled emulsion charges, is shown in Figure3a. When the presplit is performed with decoupled charges, such as the 25 mm diameter charge in Figure 3b or the 18 mm charge in Figure 3c, a single crack forms between the holes and neatly cuts the final wall. Obviously, the damage extent depends on the rock properties and the *in situ* geological conditions, but it is of interest to note the modelled damage extents follow the trends of the experimental measurements of Kilebrant *et al.* (2010). Both the model outputs and the measurements are observed to be less than predicted by the standard Swedish damage prediction formula from Ouchterlony.

In smooth wall blasting, the final row of holes contains a lighter than normal charge, possibly a smaller burden and is ignited after the main charge is completed in order to limit the confinement of the holes and reduce damage back into the sidewalls. Hustrulid and Iverson (2010) note that the success of this type of perimeter blasting hinges on good design of the buffer and helper row holes. Figure 4 compares the shape

Figure 1—(a) Hangingwall conditions when overcharging with ANFO and (b) improvements from using cartridge explosive

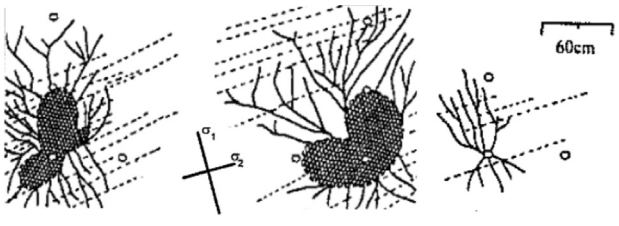

Figure 2—Fracturing from 12 kg of a) ANFO at a density of 1.05 g/cc b) emulsion at density of 1.29 g/cc and c) ANFO at a density of 0.7 g/cc (after Toper, 1995). Dark shaded region represents the extent of the crushed zone

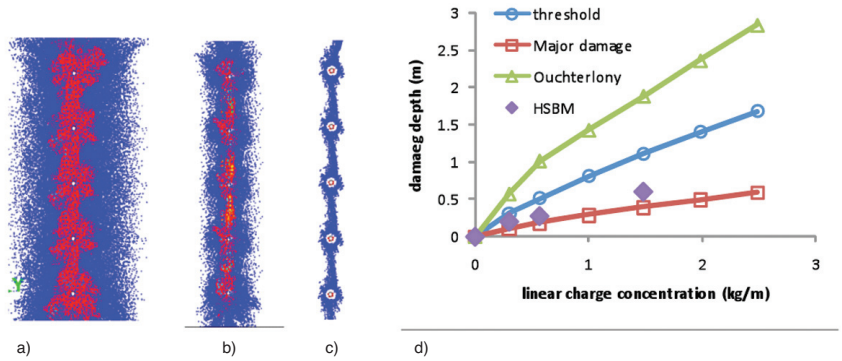

Figure 3—(a) Fracturing from presplit type blasting in 40 mm hole (a) fully coupled with emulsion (b) with 25 mm decoupled charge and (c) with 18 mm decoupled charge. Red lines are though going fractures and blue lines denote a damaged zone. (d) Shows a comparison of the damage extent with measurements for surface blasting in Gneiss by Kilebrant *et al.* (2010)

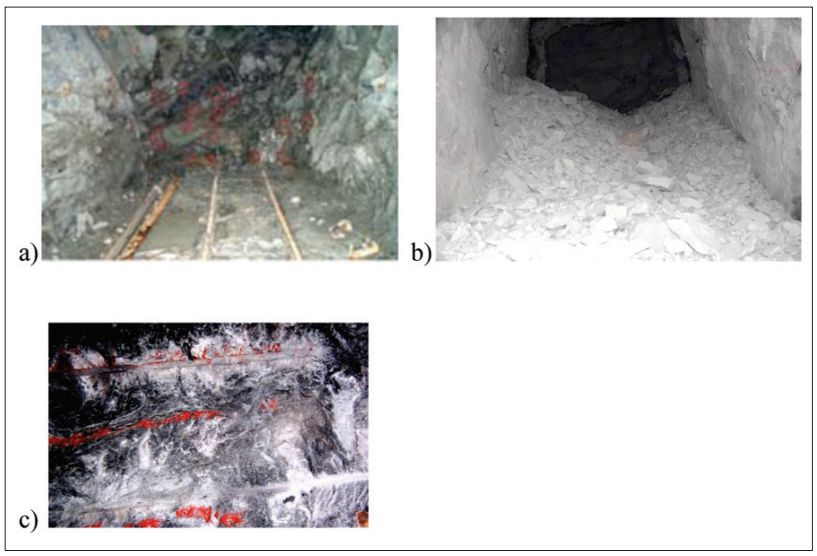

Figure 4—Photographs of a development ends mined using (a) cap fuse and ANFO and (b) using shock tube and emulsion. Photograph (c) shows the barrels in the side wall from the correctly designed shock tube blast (Cross, 2008)

of different development ends using the old method of ANFO initiated with cap fuse with the newer practice of using shock tubes and bulk emulsion (Fourie *et al.*, 2008, 2010, Cross, 2008). With the shock tube and emulsion the square shape of the tunnel is closely achieved with minimum overbreak as confirmed by the blasthole barrels present in the sidewalls of the excavation (Figure 4c).

There are significant number of safety implications if the miner is unable to limit the overbreak, which include:

- Increased spans with a higher probability of falls of ground between units
- Lowered support capacity of support units spaced wider than designed
- Unravelling of the rock between supports requiring regular rehabilitation
- Additional energy imparted to loose rocks during a seismic event (Heal *et al.* 2006)
- Lowered support capacity due to poor installation under difficult conditions.

However, if the tunnels are blasted carefully, with minimum overbreak, there are a number of associated economic benefits. In Figure 5a it is shown how many more tons of waste rock have to be removed for slight changes in the overbreak. The secondary axis shows the equivalent amount of development lost by poor mining practices. By assuming some typical costs of mesh and lacing and 50 mm shotcrete (Rangasamy, 2010) it can be seen from Figure 5b that the additional area support required can translate into hundreds of thousands of rands per kilometre of development. Also, the additional volume extracted can be costed as equivalent development metres and so even small percentages of overbreak can lead to substantial overheads in development. The extra time required to remove excess rock and to support in poor conditions can result in the loss of advance rate, which adversely affects on the net present value of the project (Ruprecht, 2006). Again, good blasting practice leads to enhanced safety and can significantly improve profitability and the long-term feasibility of a mine.

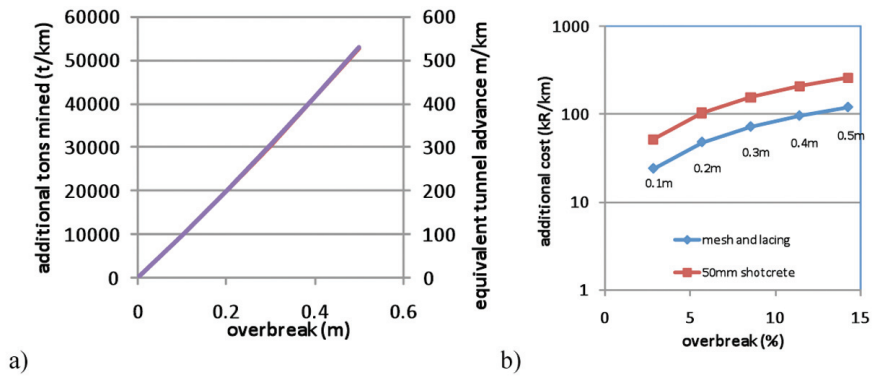

Figure 5—(a) Effect of overbreak on additional tons mined (b) increased cost of support increasing percentage overbreak for 3.5 m x 3.5 m tunnel

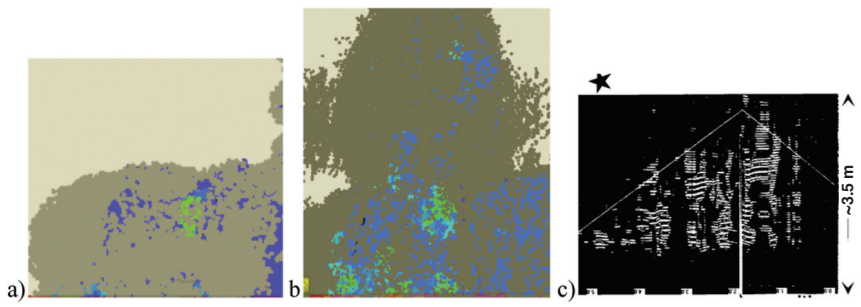

Figure 6—Plan view of damage extent predicted with an HSBM model of a stope production blast using packaged explosives (a) without preconditioning (b) with face parallel preconditioning using an additional 3 m blasthole containing 2 m of ANFO and 1 m of stemming and (c) the extent of preconditioning observed in an actual GPR scan (Toper *et al.*, 1998)

Alteration of highly stressed ground with preconditioning

Preconditioning was developed for deep level stoping during SIMRAC project GAP 336 (Toper *et al.*, 1998), further enhanced in the SIMRAC GAP 811 project (Toper *et al.*, 2003) and is now used frequently in the gold mines to relieve face bursting conditions. The idea of preconditioning is to use explosives to damage the rock mass and reduce its load-carrying capacity. The high stress region ahead of the face migrates into the deeper solid rock and the damaged, preconditioned rock provides a buffer of crushed rock that is less likely to burst at the face. A fully coupled charge of a gassy explosive that is well stemmed is required to create damage and also movement on pre-existing discontinuities. By all accounts this works and provides better drilling conditions if there is minimum delay between the preconditioning and the advance.

The SIMRAC project GAP811 (Toper *et al.*, 2003) summarizes the two suggested methods. If there is an advance gully, a hole can be drilled parallel to, and ahead of, the face. However, this is seldom practical and face perpendicular preconditioning is the most common method. Holes of 36 mm to 40 mm are drilled 3 m into the face at 3 m spacings using normal stope rigs. The last 2 m is charged with ANFO and 1m is stemmed. Each hole uses about 2.5 kg of ANFO.

The effect of face perpendicular preconditioning is illustrated with an HSBM model in Figure 6 where the addition of a longer hole ahead of the face causes slip on the bedding planes surrounding the stope and hence increases the buffer of broken rock ahead of the face. The predicted zone affected by the preconditioning corresponds closely to ground penetrating radar observations (Toper *et al.*, 1998).

There is very little information on preconditioning in development. Toper (1995) describes some preconditioning in a development end at West Driefontein in the early 1990s. Adams and Geyser (1999) preconditioned a tunnel through a dyke at Kloof mine by drilling four additional 4 m holes into the tunnel face. The holes were tamped with 2.5 m of clay and charged with 1.5 m of ANFO type explosives. Following some minor footwall strain bursting after the first preconditioning blast an additional two preconditioning holes, angled towards the footwall from the centre area of the tunnel face were included in each preconditioning round. Sellers and Hattingh (2007) described the preconditioning of a 'pillar' formed at the end of decline where high stresses were induced due to misalignment of the intersection with two tunnels. The end was 5 m wide and 4 m high. The last 8 m of development had to be completed when strain bursting began. Two holes of 60 mm diameter were drilled through the pillar. The far end was stemmed for 2 m, and then the central 4 m was charged with ANFO and centre primed with cartridges. The last 2 m was stemmed. These two holes were fired simultaneously. The preconditioning was reported to be successful and the decline development was completed with no further strain bursting.

Removal of people from hazardous places—inverse drop raising

During conventional boxhole development, a number of people must climb the excavation in order to perform the blasting to advance the end. The shift supervisor enters the boxhole to assess the rock conditions and to bar down and mark out the next blast. A surveyor may also enter to check the direction of development. Two drillers must then climb the raise on chains, hauling their dills with them. A platform of wooden planks is built each day to support the working and then the drilling proceeds. The end advances about 1m per day and due to the lifting of the end of the drill rigs the line of the raise tends to flatten, which causes choking of the ore once the raise is in use (Pall 2010). There are a number of potential safety hazards involved in this operation ranging from the possibility of rock falls to injuries due to manipulation of equipment. At least four people are required to enter this hazardous environment on a daily basis.

A safer alternative, inverse box hole blasting, has been successfully used in platinum mining and AEL Mining Services recently partnered with EBJ Mining Construction to successfully execute mechanized, inverse box hole blasting at Harmony Gold's Elandsrand mine in South Africa (Fourie, 2010). This method (see Figure 7a) entails drilling a stub upwards at an angle of 55° using an automatic drilling machine. This 15 m long section is removed in a single blast. The vertical portion is blasted in 2 m sections using a retreat method. Risk can be expressed in the equation

$$Risk = Hazard \times Exposure \quad [1]$$

and so by using controlled and well designed blasting, the exposure of miners to the hazards of excavating the boxhole can be minimized and even eliminated and hence the safety risk to the mine is reduced substantially. The smooth wall blasting as shown in Figure 7b reduces the chances of rock slips and rock related injuries. As with many of the other examples in this paper, the increased safety is accompanied by improved production efficiency in the short term with the reduction in excavation time required from 12 days to 7 days. Long-term production efficiencies are generated by the superior performance of the orepass due to the smooth wall blasting.

Improved support performance by protecting gully shoulders

Proper control over drilling and blasting in the stope environment has significant effects on the performance of support units. With the introduction of shock tube initiating systems the faster ejection of rock has required that the drilling directions are changed from the conventional 70° to the stope face to being drilled perpendicular to the stope face. This reduces damage to the support units. The reduced scatter in the initiating systems permits the increase of the burdens due to the reduction in the possibility of out of sequence firing.

One small detail in the design of a stope round can make a significant impact on the gully support and can easily be overlooked. As shown in Figure 8a, the recommended stope blast design (AEL, 2007,2010) uses opening holes (also called gut holes) as the first two or so holes next to the gully. These holes have reduced burden and charge to protect the stope footwall on the sides of the gully.

An HSBM model was run using a standard square or twin box pattern and Figure 8b shows how the lower hole next to the gulley has broken off the edge of the gully and significantly damages the rock to some depth below the footwall. Correct practice would be to rehabilitate the gully sidewall prior to setting the pack though in many cases, to maintain production, the gully packs would merely be placed further into the stope. This shortcut can widen the span of the gully by nearly 1 m, which has serious negative side effects for the safety of the miners in the gully. The effective gully roof span is significantly increased leading to higher loads on the anchor support units in the gully hangingwall reducing their load carrying capacity. The increased span can lead to a greater potential for rock falls between the support units (Joughin, 2010).

Blasts to design rockburst resistant support

Blasting has been used to assist rock engineering researchers with the development of rockburst resistant support units. Hagan *et al.* (2001) simulated the effect of a seismic event to understand the ejection of rock from tunnel sidewalls. Heal *et al.* (2006, 2007) have taken this work further and used

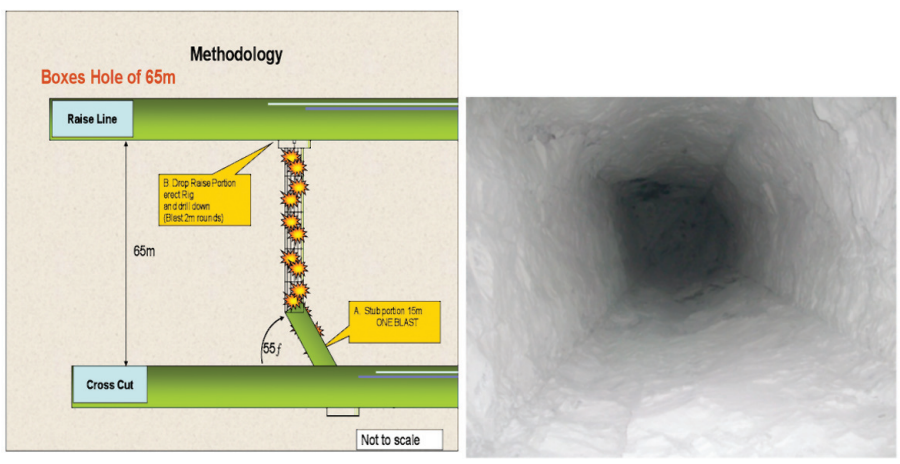

Figure 7—(a) Diagram of an inverse drop raise and (b) the final product showing barrels in the wall of the excavation

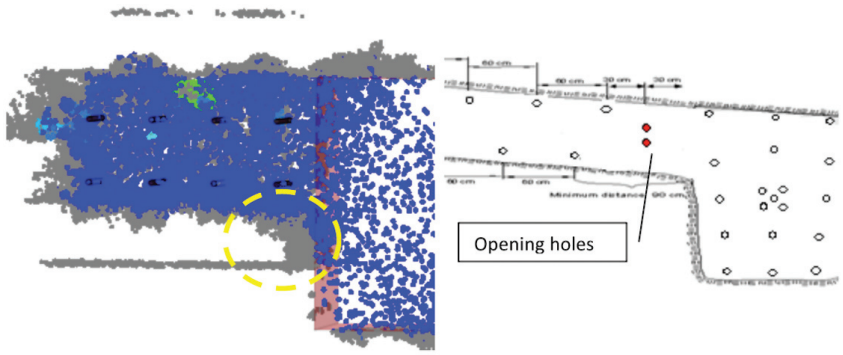

Figure 8—(a) HSBM model showing the damage to the gully sidewalls (circled) due to poor blasting practice and (b) standard stope blast design (AEL, 2010)

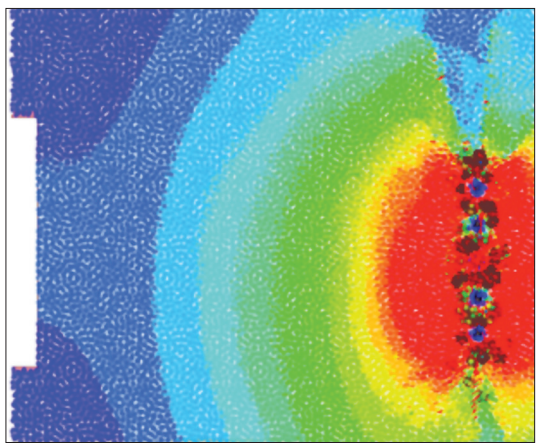

Figure 9—HSBM model of the Kopanang simulated rockburst experiment (Hagan, 2001) showing the dynamic stress waves emanating from the blast

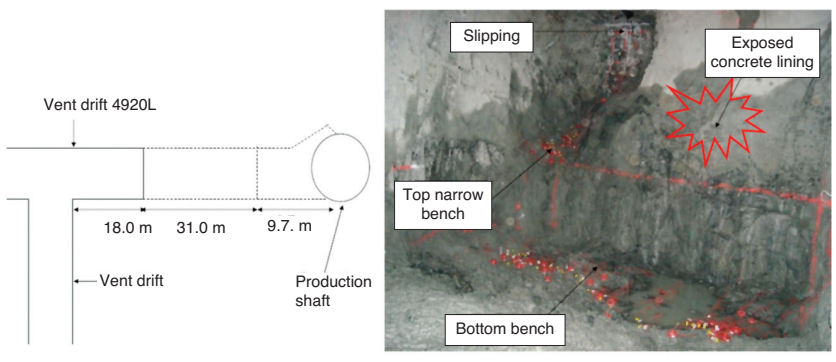

Figure 10—(a) plan of vent drift holing (b) drift prior to final holing into shaft (Mkumbo, 2009)

blasting to simulate a number of rockbursts. The HSBM model is currently being applied to understand the role of fracturing, and rock mass conditions on the peak particle and ejection velocities (Figure 9) and to calibrate the model for further planned experiments.

Mining towards a shaft

Safety in the underground environment can also be enhanced when mining adjacent to service excavations such as shafts and storage chambers. Because of the critical nature of these structures, blast vibrations can lead to local failures that, even if small in nature, can have serious consequences. For example, the spalling of a small piece of shotcrete from a shaft wall can cause serious damage to the shaft infrastructure and injury to people travelling or working in the shaft. An example of how controlled blasting can lead to a successful shaft holing, with no damage to the shaft was provided by Mkumba (2009). As shown in Figure 10a a vent drift had to be holed into a shaft at a mine in Tanzania. The peak particle velocity (PPV) is most commonly used as a

measure of damage potential and a model for the attenuation with scaled distance was determined from a few blasts on the full 5 m x 5 m development end. By altering the design to have shorter rounds, a top heading with a following footwall lift and by moving from shocktube to AEL Smartdet electronic detonators, it was possible to design and fire just the required amount of explosives within any single time window. The drift was blasted to within 2 m of the shaft wall (Figure 10b) with PPVs reduced from over 100 mm/s to around 15 mm/s and absolutely no damage was observed at the shaft lining.

Conclusions

The examples in this paper demonstrate how correct blasting practices using rigorous design procedures and maintaining care in the daily application of explosive energy can lead to significant safety improvements. Good blasting with correct selection of the explosive type, charge mass and round design can improve rock mass conditions and maximize the useful capacity of support units. By making the effort to implement cautious blasting practices in tunnel development the amount of overbreak is limited, which improves rock mass conditions and support integrity with the spin-off of reduced support and remediation costs and better project feasibility. The application of modern initiation system technologies such as electronic detonators further empowers the miner to mine safely by being able to exactly achieve designed effects.

Controlled blasting also provides a number of hazard mitigation measures including the:

- Alteration of highly stressed ground by preconditioning (development ends, stopes and pillars)
- Removal of people from hazardous places e.g by inverse drop raising
- Provision of proper foundations for support unit such as on gully shoulders
- Providing inputs for the design of rockburst resistant support.

By applying appropriate quality control and understanding of the implications of proper blast design, blasting operations can not only improve the safety on the mine, but will most likely improve productivity and the long-term sustainability of the mine as well.

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