8. Electronic detonators, use and misuse

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Studies of blast damage at the Äspö Hard Rock Laboratory, Sweden

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ABSTRACT: A new tunnel development at the Äspö HRL in Sweden gave opportunities to test new ideas on blast design with the purpose of minimising the excavation disturbed zone (EDZ) caused by the drill and blast method in hard crystalline rock. In addition, tunnelling at this facility with well-defined geoscientific site conditions also gave a unique opportunity to follow up the effect of geological conditions on the environmental impact of the blasting in terms of a vibration monitoring programme. This paper presents the findings concerning the possibility of controlling the EDZ to a depth of within 0.2–0.3 m. The look-out angle for drilling, as well as the distribution of explosives along a round, makes the EDZ discontinuous along the tunnel. The drilling accuracy is an important factor, as well as a simultaneous initiation of the contour. Electronic initiators are promising but were found to be not so easy to handle. The peak particle velocity (PPV) varies, within a 95% confidence interval, by a factor of ±5 from a mean value.

1 INTRODUCTION

A new tunnel was developed at the Äspö Hard Rock Laboratory (HRL), Sweden, during the spring and summer of 2003. The tunnel is located at the 450 m level close to the shaft. The tunnel was specially designed for a rock mechanics experiment, the Äspö pillar stability experiment (APSE) (Andersson 2003). The tunnel is therefore referred to as the APSE tunnel. The main purpose of the tunnel was to provide suitable conditions for a rock mechanics experiment.

The HRL is a facility with extensive R&D activities. The special restrictions on cautious blasting and access times required special concern in planning of the excavation works. Tunnelling at this facility with well-defined geoscientific site conditions also gave a unique opportunity to follow up the effect of geological conditions on the environmental impact of the blasting in terms of a monitoring programme.

It was also concluded during the planning process that the tunnelling would provide some opportunities for studying the possibilities of controlling the development of an excavation damaged zone (EDZ) caused by drill and blast operations (D&B). The new tunnel was to be located only some 300 m away from a tunnel where a study of the EDZ in a D&B had been carried out in 1995 (Olsson & Reidarman 1995). The simple question was whether the current state-of-the-art D&B method, together with an actual design of extreme smooth blasting of the floor by taking the lower part of the cross-section as a separate bench (see section 3), could produce a less pronounced EDZ compared with the results 8 years earlier. This included the possibility of testing a new formula that was recently proposed

to estimate the extension of the EDZ (Olsson & Ouchterlony 2003, Ouchterlony *et al.* 2001).

The present paper is based on two SKB reports (Olsson *et al.* 2004, Nyberg *et al.* 2005).

2 EXCAVATION

2.1 Requirements

The planned pillar stability experiment (APSE) was intended to form a 1.0 m wide pillar by drilling two large-size boreholes of ∅1.8 m close to each other (Andersson 2003) (Fig. 1).

To enable a sufficient stress concentration in the pillar, two geometrical requirements for the tunnel had to be met:

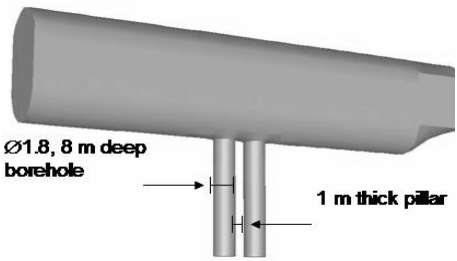

Figure 1. 3D view of the tunnel and the planned pillar.

- The tunnel should have a high height/width ratio in order to concentrate the stresses on the roof and the floor.
- The floor should be circular shaped. A circular floor concentrates the stresses at the centre of the floor, whereas a traditionally shaped tunnel concentrates the stresses at the corners of the walls – floor.

At the location for the planned pillar, the requirements on borehole precision and EDZ were higher than for the rest of the tunnel. A correct circular floor would concentrate the stresses at the centre of the tunnel floor. To achieve a practical excavation, the tunnel was designed with a top heading in a traditionally shaped tunnel, and a bench with a circular floor (see Fig. 2).

Other requirements for the D&B operations were related to vibrations, especially at the nearby shaft station and its furnishings.

2.2 Blast design

The objective with the blast design was to find a general blast pattern that could be used along the entire tunnel. The tunnel was only 70 m long, and there was no time to adjust to new practices of blasting over such a short distance.

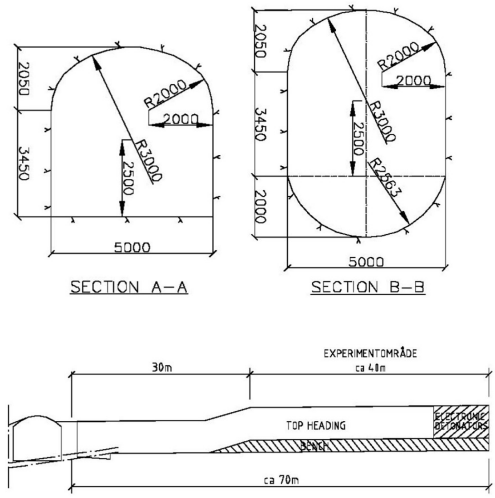

Figure 2. Designed tunnel geometry.

Figure 2 shows the excavation work divided in three different phases. The first phase of the tunnel was an ordinary 26 m² tunnel. After approximately 30 m, a ramp separated the tunnel section into a top heading and a bench (total area 33 m²). The last three rounds were test rounds with electronic detonators.

A normal tunnel round consisted of 120 blastholes with ∅48 mm holes, together with four ∅102 mm empty holes in the cut. The perimeter holes had a spacing of 0.45 m and a burden of 0.5 m. The perimeter holes were charged with ∅17 mm Dynotex (a cartridged explosive from Dyno Nobel), and the helpers with ∅22 mm Dynotex. The other holes were charged with ∅22 mm Dynotex and/or ∅25 mm or ∅29 mm Dynomit/Dynorex. Nonel® was used as the initiation system. The drill pattern was designed primarily for very cautious and careful blasting with small charge weights in order to stay within the restrictions on vibrations and fly rock.

Figure 3 shows the standard pattern for the top heading and the bench. In principle the time delay is 100 ms for each cap number. This means that the delay interval in practice becomes 100 ms up to number 12, 200 ms up to number 20 and thereafter 500 ms up to number 60.

2.3 Equipment

An Atlas Copco rocket boomer drilled the blast, grouting and probe holes. The three-boom 353ES

rig from 1997 was equipped with 16″ BUT35 feed beams with a COP1838/ECS drilling system and Bever guidance control. The outer booms were equipped with an RAS rod-adding system to drill grouting and probe holes.

There are several manufacturers of guidance control systems. Skanskas's rocket boomer was equipped with a system from Bever Control AS. The main features of this jumbo guidance system are: tunnel laser navigation; drill rod direction monitoring and control; drill pattern display and recording; drill depth control; and planning and reporting on a desktop PC.

Charging the blastholes was done from a scaling platform or from the rocket boomer charging basket.

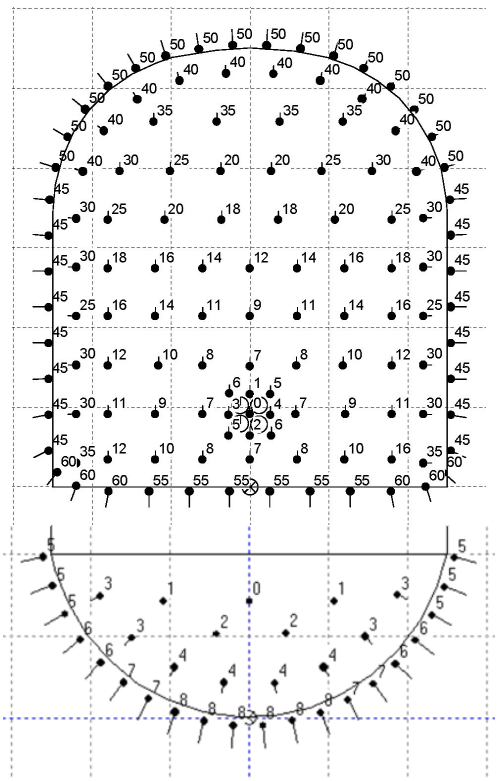

Figure 3. Initiation pattern.

A Cat 966 front loader was used for mucking. When mucking the arched floor, a backhoe was used together with the Cat 966.

The scaling works were done from the same scaling platform that was used for charging.

3 EXPERIMENTAL ROUNDS

How to reduce the damage zone is a very essential task when planning the excavation of nuclear waste deposit tunnels. The smooth blasting technique has to be used. This technique has been a standard in Swedish tunnelling for many years.

In order to find out the factors that influence the damage zone, intensive research has been carried out in Sweden by SveBeFo (Swedish Rock Engineering Research). A large number of holes have been blasted and the cracks have been examined. The coupling ratio (charge diameter/hole diameter), the spacing, the water in the holes, the scatter in the initiation and the influence of different explosives on crack lengths are some of the examined factors. With knowledge of the effects of these factors, the following equation has been proposed by Olsson & Ouchterlony (2003):

$$R_c = R_{co} \times F_h \times F_t \times F_v \times F_b \quad (1)$$

where R_{co} is the measured crack length under standard conditions and F_h, etc., are corrections for the spacing (h), interval time (t), wet holes (v) and type of rock (b) with natural cracks.

According to earlier tests, a simultaneous initiation results in shortest cracks (Olsson & Bergqvist 1997). Simultaneous initiation is only possible with electronic detonators or detonating cord. Pyrotechnic caps, like Nonel, have excessively high scatter in the initiation time and their use results in longer cracks.

In the APSE tunnel, Nonel was used for the blasts except for the last three ones, where it was decided to use electronic detonators in the contour. Three variations of simultaneous detonation were planned. Simultaneous detonations of many contour holes would be preferred to obtain few and short cracks in the remaining rock. However, owing to vibration limitations in the APSE project, only five contour holes could be simultaneously initiated. Consequently, the initiation of the contour holes was divided into different groups. Within each group the caps would detonate simultaneously. Figure 4 shows one example of the initiation pattern for the test rounds. In this pattern the roof and the right wall holes were initiated with electronic detonators, and the left wall holes were initiated with Nonel.

The electronic detonators in this test were i-kon® from Orica (www.i-konsystem.com). The detonators can be assigned a delay of 1–8000 ms, and each detonator has a unique factory-assigned ID number.

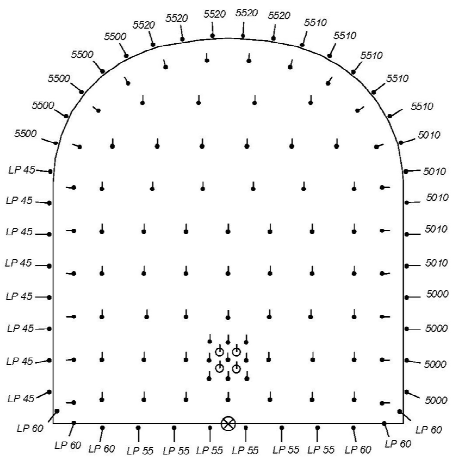

Figure 4. Contour initiation of test round 2.

4 MONITORING DURING EXCAVATION

4.1 Objectives

During the blasting of the APSE tunnel, vibrations were measured for all the rounds. The main objectives directly related to blasting were to:

- make a blast optimisation and a vibration level control at installations and other facilities according to the recommendations;
- determine the decay of peak particle velocity (PPV) with distance;
- measure the electronic cap time intervals for initiation of the contour holes;
- estimate to what extent vibration records could guide in judging the relative extension of an excavation damage zone (EDZ).

4.2 Monitoring

Three datasets were monitored during the blasting.

Bergsäker AB carried out the vibration level control during the three excavation phases. Standard geophones were used at distances of approximately 10–600 m, both in the tunnel system and above ground.

Swebrec, the Swedish Blasting Research Centre at Luleå University of Technology, monitored accelerations in gauge holes near the tunnel wall from the three experimental rounds in order to measure the electronic cap interval timing according to the proposed relation (Equation 1).

Uppsala University Department of Earth Sciences made farfield measurements with three mobile seismic stations positioned above ground at about 600–1600 m distance from the tunnel. The purpose of the seismic measurements was to calculate the farfield amplitude decay from two of the experimental rounds.

4.3 Instrumentation

For vibration level control, surface-mounted standard vertical geophones were used together with a recording unit (AVA95) with a sampling rate of 3000 samples/s. Full waveforms can be recorded, but in this case the recorder usually picked the single maximum value during the blast time sequence. The instrument meets the requirements of the Swedish Standard SS 460 48 66. According to this standard, all recordings were filtered, i.e. frequencies over 350 Hz were filtered out.

Nearfield vibration recordings from blasting have proven to work well if accelerometers are grouted in gauge holes (Nyberg *et al.* 2000, Ouchterlony *et al.* 2003).

The electronic cap timing was measured along the tunnel in two gauge holes 10 m apart and 14–28 m from the front with shock accelerometers (Kistler 8704B5000). The signals were recorded at a DAT with a sampling frequency of 48 kHz.

Two horizontal and one vertical accelerometer were used in each gauge hole. The accelerometers were mounted to aluminium anchors that were grouted at the bottom of drilled holes.

The seismic stations used were highly sensitive and are used as temporary stations of the Swedish National Seismological Network (SNSN). The seismic stations had three-component Lennartz LE-3D/5s geophones and a Nanometrics 'Orion' data logger.

5 RESULTS

5.1 Excavation

The drilling accuracy was very good, perhaps owing to the skilfulness of the operators, with a high number of half-pipes (see Fig. 5).

Figure 5. Final tunnel contour.

In order to study the drilling accuracy, all the visible half-pipes in the walls were surveyed at the starting and end points. The result shows that 95% of all measured half-pipes were within the requirement of a maximum look-out distance of 0.3 m.

The specific charge was 2–2.5 kg/m^3 for the tunnel and 1.2–1.3 kg/m^3 for the bench. The first tunnel round consisted of only a few short drill holes, but after 15 m of tunnelling the advance was 4 m. The holes in the first rounds were stemmed with clay to protect installations from fly rock and air shock. A steel plate was vertically positioned near the front of each round, and rock material was loaded, covering the plate. At the entrance of the tunnel there were heavy rubber mats.

On three occasions, blasting resulted in misfires. The definition of misfire is when a major part of the rock mass is left unbroken after the blast. A major part involves more than 10–20% of the total volume, with many of the blastholes intact. No increase in EDZ owing to the misfires was observed in the wall.

5.2 Vibration monitoring results

This section presents some of the results related to the blasting technique.

Vibrations from blasting in general have a large amount of scatter depending on rock conditions and breakage. However, vibration gauge mounting, band-limited recordings and time scatter in pyrotechnic detonators also cause scatter. This is probably true for our measurements as well.

Figure 6 shows an example of the recordings with different blast intervals from a top heading round. The blasting sequence covers about 6.3 s of vibrations from a vertical geophone at the firewall 56 m from the round. The maximum vertical velocity (9.4 mm/s) from the entire vibration sequence is defined as the peak particle velocity, or PPV.

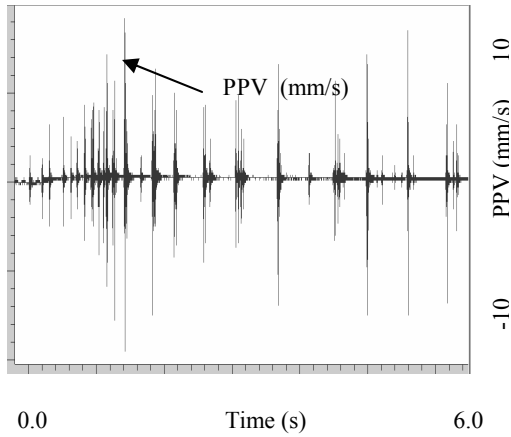

Figure 6. Event from a blasting round with the maximum vibration value defined as PPV (mm/s).

The recorder monitored the PPVs and the associated intervals, and therefore the nominal maximum charge weights (kg/hole) were known. For these blasts, the lifters and production holes were generating most of the PPVs, and the corresponding cooperating charge weights for the intervals varied from 0.4 to 3.7 kg/hole.

Figure 7 below shows the PPV dependence on distance in a log–log scale. The figure shows the maximum vertical velocity for each monitored round for all distances. Note that the recorded velocities at the seismic stations are three-component values.

It is clear that the integrated acceleration values are higher than the standard geophone values at the same distances. This is probably due to the use of the 350 Hz high-cut filter for all the vertical geophone data.

The seismograph values are roughly independent of distance over the range 600–1600 m.

According to the Swedish standard SS 460 48 61, velocities of 0.4–1.0 mm/s will cause a moderate environmental disturbance. The 1 mm/s line is shown in Figure 7. Velocities over 1.0 mm/s will probably cause disturbances.

This occurs closer than 300 m to the blast. Thus, moderate disturbances will very unlikely occur above ground.

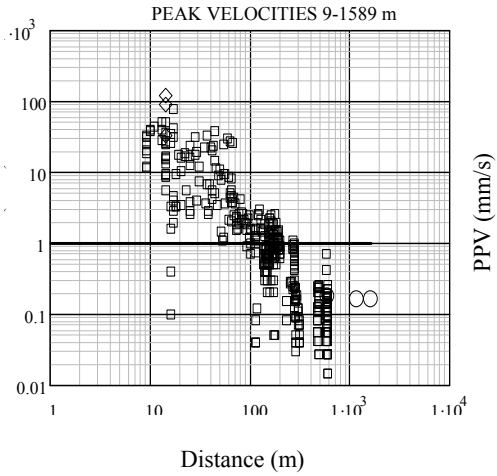

Figure 7. Maximum PPV for each round versus distance (the 1 mm/s level is reached at about 300 m from the blast): □ geophone 1D data; ○ seismograph 3D data; ◊ accelerometer 1D data.

The vibration level control at the firewall showed that the maximum PPV was 49 mm/s at a 13 m distance from round 17. The maximum allowed PPV was 50 mm/s. No damage was found.

One of the goals was to test if the different blasting techniques and round misfires could be monitored at the gauge stations. For a comparison of datasets, a common scaling law such as the one below was used. The square root scaling law is given by:

$$\mathrm{PPV}(R, q_{\mathrm{m}}) = A / \left(R / q_m^{\,0.5}\right)^{\beta} \qquad (2)$$

with intercept A and slope β and $(R/q_{\mathrm{m}}^{0.5})$ as the scaled distance. The scaling law is dependent on distance R (m) and on the maximum cooperating charge weight q_{m} (kg). These values were selected according to the PPV interval number from the data logger.

As mentioned earlier in this section, the PPV from blasting may contain a large amount of scatter. Figure 8 shows a linear least-squares fitting of Equation 2 in a log (PPV) versus log (scaled distance) plane. The number of values is 419 including test rounds with fewer holes. This dataset is used as the reference dataset. The inner 95% confidence bounds apply to the 'true value' and the outer bounds to the 95% prediction intervals, i.e. the next blast will, with 95% probability, generate the PPV ± a factor of 5.

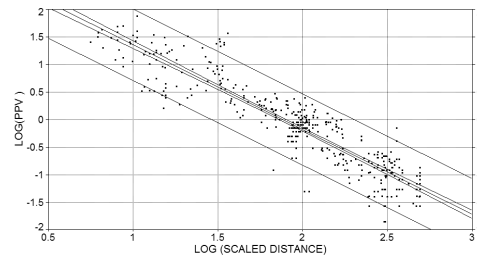

Figure 8. Log (PPV) versus log (scaled distance) for 419 top heading round data.

Figure 9 shows a division of the data into two parts in order to test whether the PPV is dependent on the total number of blastholes per round.

All top heading rounds with a maximum 92 blastholes are marked with black triangles. It can be seen that rounds with fewer blastholes generate lower PPV, especially at long distances.

Blair (1990) has modelled the PPV as a function of the number of blastholes. He found that the PPV increases with the number of blastholes owing to vibration overlap.

Table 1 shows a statistical (95% prediction confidence interval) comparison of the parameters log A and β in Equation 2.

Firstly, a dataset that excluded top heading test rounds and misfired top heading rounds were selected. No differences were found. Secondly, a dataset (T_{299}) from blasts with more than 92 blastholes was selected. This was compared with the reference (T_{419}) in Figure 8. Thirdly, top heading rounds (T_{192}) were compared with the bench rounds (B_{58}).

Table 1. Fitting parameters intercept log A and slope β.

Round	log A	β	r^2
T_{419}	2.91 ± 0.14	1.54 ± 0.06	0.84
T_{299}	3.22 ± 0.17	1.66 ± 0.08	0.84
T_{192}	3.10 ± 0.29	1.61 ± 0.16	0.68
B_{58}	3.14 ± 0.55	1.56 ± 0.31	0.63

Most striking from this test is that the PPVs increase with the total number of holes in the blast.

The most representative values for the entire top heading rounds are those from T_{419}. The parameters are A = 812 mm/s and slope β = 1.54.

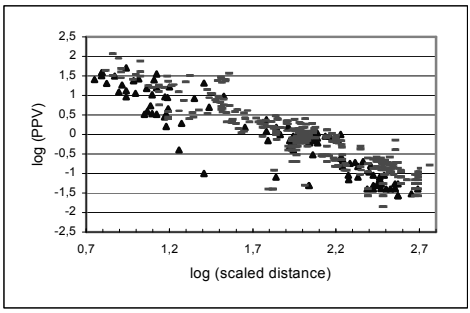

Figure 9. Rounds with a maximum 92 blastholes are marked with black triangles. Full blast rounds, 100–138 blastholes, are marked with grey rectangles.

5.3 Excavation damage

The technique used was developed by SveBeFo (Olsson & Reidarman 1995) and consists of sawing a number of 0.5 m deep cuts in the tunnel wall or floor. After removal of the slices in the cuts, a dye penetrant is sprayed onto the cleaned surface, causing the cracks to appear very clearly.

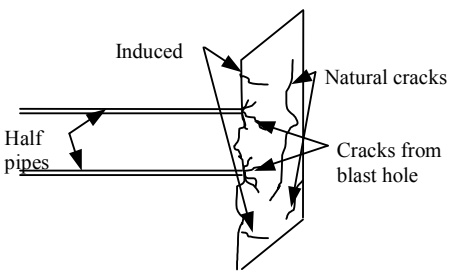

Figure 10. Typical crack pattern.

A typical crack pattern with different types of crack is shown in Figure 10. There are cracks from the blastholes, induced cracks (cracks caused by high stresses and released by the blasting process) and natural cracks (determined by geological conditions).

In the APSE tunnel, five vertical cuts and one horizontal cut were made in the tunnel wall, and two cuts were made in the floor. Figure 11 shows a typical crack pattern from the APSE tunnel. There are only a few cracks originating from the blastholes, and these cracks are relatively short, <15 cm, corresponding to earlier experiences under similar conditions. There are also a number of induced cracks, and finally some natural cracks.

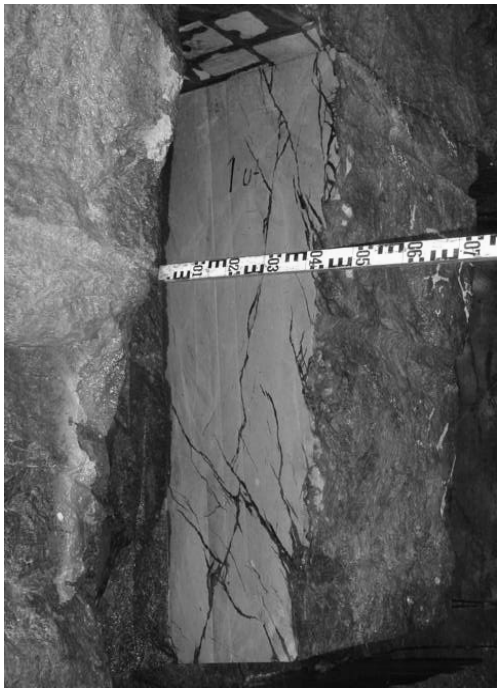

Figure 11. Cracks from the tunnel wall.

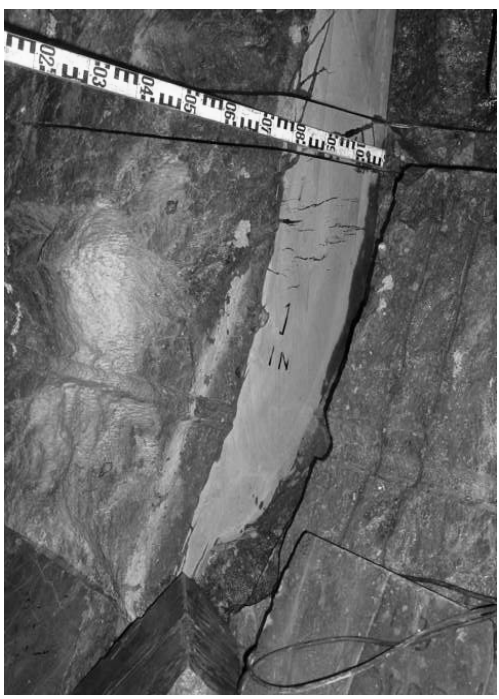

Figure 12. Cracks from a water-filled hole.

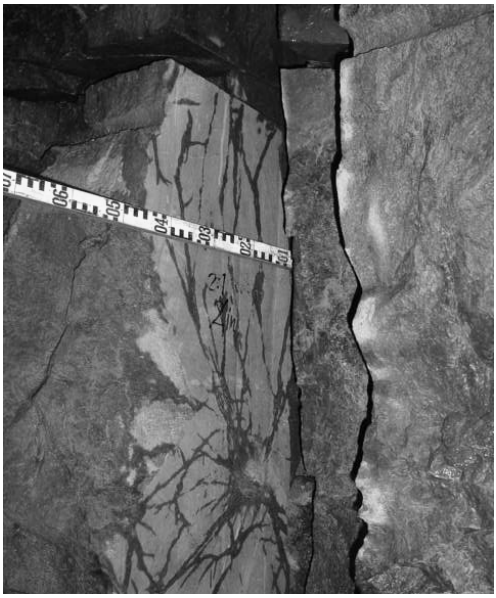

Figure 13. Cracks from a water-filled hole.

The cracks from the bench blasting are generally fewer and shorter, in spite of the fact that Nonel has been used (see Fig. 12). A reason for this might be a different stress situation, straighter holes and maybe less confined rock.

Figure 13 shows the importance of water-filled holes on crack lengths. A perimeter hole had deviated from the tunnel line and was found some 30 cm inside the wall. This hole was found to dip and was probably water filled when blasted. The crack length was >40 cm, which is 3–4 times longer than from a similar charge in dry conditions.

The cracks in the floor were some 20 cm long, which is much shorter than from normal tunnel blasting.

6 DISCUSSION AND CONCLUSIONS

With reference to the short tunnel, there was no time to adjust to new practices of blasting over such a short distance. However, blast optimisation was performed on the basis of the following conditions. The first part of the tunnel was designed exclusively to avoid fly rock and to prevent damage to installations. The design comprised short holes, small charges, clay plugs and short delay times.

The next part of the tunnel subjected to extra care was the passing of the test area. Emphasis was placed on smooth blasting of the tunnel floor. Finally, in the last part of the tunnel, the focus was on optimising the delays times in the contour holes by using electronic detonators.

No optimisation effort was put into reducing the specific drilling and specific charge. The specific charge of 2.0–2.5 kg/m^3 could not be considered particularly low. There is no contradiction in a normal specific charge and a cautious blasting design as long as the explosive is distributed in a correct way.

6.1 Excavation method and control instruments

The drill rig Jumbo used in the APSE tunnel was equipped with the Bever guiding system. One of the functions of this system is to guide the operator when a new collaring of a drill hole is executed. However, the system should be used with a few precautions. Owing to the mechanical characteristics of the rig system – carrier, boom, feeder, hydraulics, etc. – the accuracy of the system is fairly good but not excellent.

No guidance system can today meet the demands from many of the Swedish owners of tunnel contracts regarding drilling accuracy. Such high demands may stem from the conviction that these systems are correlated with the demands, which is not true. Manufacturers are aware of this situation and improvements are to be expected in the near future. Atlas Copco presented an article in *Tunnels & Tunnelling* in July 2003, giving their view on the use and misuse of the logging system (Nord 2003).

The requirement concerning increased drilling accuracy for the floor holes raises new demands on drilling equipment and operator skills. Normally the look-out angle in the floor holes is allowed to be larger than in wall and roof holes. The allowed zone of damage is also larger. In the APSE tunnel the same requirements were valid for all contour holes including the floor holes.

The difference in drilling floor holes compared with other contour holes is that the drill steel is not visible to the operator. In order to make the collaring of the hole as close to the previous floor as possible, the feeder beam must be positioned upside down, with the drill steel towards the floor. As long as the on-board systems for positioning and alignment of the drill holes lack the high degree of accuracy needed for a fully non-manual drilling, the result relies on the skill of the opera-

tor. If the operator loses eye contact with the drill steel, the chances of achieving a proper result will be reduced. This situation definitely calls for better guidance tools in the future.

Another practical impact of the smaller look-out angle is problems with preventing water from filling the floor holes. The smaller look-out angle will reduce the volume of the pool in front of the tunnel face, and a smaller volume of water could be contained. The distance between the collar of the hole and the water surface is reduced. Dewatering at the tunnel face must be improved.

6.2 Blasting vibrations

There is a large amount of scatter in the data, and in practice some maximum allowed vibration amplitude must be determined, for example an upper 95% ($\approx +2\sigma$) confidence line instead of a least-squares fit line. The statistics say that a 95% prediction interval spans a given PPV ± a factor of 5.

The environmental impact of this blasting is low. The vibrations, if we consider the scatter, were damped to about 1 mm/s at distances 300 m away. This indicates that moderate disturbances will very unlikely occur above ground.

Note that blasting with heavier charges will increase the average PPV.

The conclusions of the PPV parameter test areas follows:

- Data close to the blast seem to be biased owing to the high-cut 350 Hz filter of the recorder.
- Anomalous rounds such as test rounds and misfire rounds (reduction of 12% in the total dataset) have no effect on the PPV. This may depend on the general data scatter.
- There were only three misfired rounds, and we could not see any clear trend in the vibrations from these rounds.
- At long distances from the blast, values from rounds with fewer blastholes are lower than full blast rounds. This indicates that the amount of explosives that contributes to the PPV increases with the total blasthole number. The scaling law does not account for this.
- The concept of maximum cooperating charges may be a reasonable approximation close to the blast but not at long distances, about 400 m or more in this test.
- Comparison of the parameters for the bench blasting rounds with the parameters for the top heading rounds shows no difference.
- To predict the PPV for a particular blast accurately, the parameters A and β must be known for a relevant type of blast.

6.3 Excavation damage

A higher precision in positioning and drilling is desirable.

Electronic detonators have a high accuracy and a high potential to reduce cracks from blasting in the remaining rock. However, the detonators must be easier to handle.

Access to an electronic detonator could improve tunnelling by:

- A higher safety against simultaneously detonated charges owing to more available delay numbers. The need for surface delay solutions in large tunnel sections with a large number of holes will decrease.
- A better result from contour blasting, a reduced damage zone and a smoother tunnel surface could be achieved by applying simultaneous initiation of contour holes.

No increase in EDZ in the wall owing to misfires was observed. It has also been clearly shown that it is practicable to reduce the EDZ in the floor to the same extent as in the walls and roof by excavation of a top heading and a bench in combination with a reduced charge concentration. The designed circular contour in the floor was purposely to elevate the secondary stresses in the floor. This caused some stress-induced fractures as well. This rock mechanical effect has, of course, also to be considered in the design of a more normal tunnelling project.

Generally, the cracks from blasting were fewer and shorter in the bench rounds than from the tunnel rounds. Another surprising result was that the cracks from the test sections initiated with electronic detonators generally were longer than cracks from holes initiated by Nonel. A probable reason for this is that the holes in the test section were inclined and therefore water filled.

The look-out angle and distribution of specific charge along each round caused a discontinuous EDZ along the tunnel. It is therefore indicated that the impact of the EDZ on hydraulic conductivity along the tunnel is very limited.

Water in the blastholes gives a significant increase in the EDZ. Water could be avoided by drilling the holes pointing upwards.

A large variety of explosive types and dimensions may improve the blasting result but may increase the time for the charging procedure and raise the costs. In order to reduce damage, especially in the floor holes, water must be kept away. To ensure a blasthole almost free of water, special predecoupled charges may have to be manufactured. These charges consist of closed pipes slightly smaller in diameter than the hole itself. Inside the pipe the charge is regularly centred and decoupled. The pipe is placed in the hole and locked to prevent it from slipping out of the hole under water pressure.

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Safety and security - minimising risk with electronic detonators

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ABSTRACT: The need for safer and more secure commercial explosives products is placing an ever-increasing demand on manufacturers from users and authorities alike. For those in the business of developing, manufacturing and selling detonators, this demand translates into creating safer and more reliable products that can only be used by authorised personnel. Interest in electronic detonators is therefore now focusing on technological advancements vis-à-vis reliable, safe and secure application, as well as the blasting and downstream mining benefits that result from precision delay timing. The ability to communicate digitally with detonators opens up a host of benefits, some of which lie outside the blasting and mining process. These include: improved safety on the bench, simplified use to prevent mistakes, always knowing the whereabouts of blasting equipment and the reduced risk of detonators being fired by unauthorised personnel. The paper examines how new features could be incorporated into the latest and future generations of electronic detonator systems to help deliver this added value.

1 INTRODUCTION

Electronic detonator development over the past ten years has focused primarily on delivering millisecond accuracy while providing a level of safety and reliability that is acceptable to the industry it serves.

Hardware and software evolution, particularly during the past three years, however, has developed to the extent that significant new features can be introduced to enhance the basic electronic detonators currently on offer. Now that an intelligent, digitally controlled communications platform for the initiation of explosives has been established, it would be unfortunate if that platform were not advanced relative to the rapid technological developments seen in other digital sectors, such as the Internet, satellite communication, computer networking, and GPS.

For those who are committed, the opportunity exists to expand the capabilities of existing products, providing additional benefits to end-users and society in general.

The base technology has achieved an initial level of success and customers are clearly recognising that value-added system enhancements will differentiate products as the next phase of development progresses.

In this paper, we focus on developments affecting the areas of safety and security. These areas have become increasingly important to end-users, regulatory bodies that control the electronic detonator application environment, law enforcement agencies, and distributors, who are recognising that digital control of the blasting process can provide solutions which were previously unavailable.

The term 'safety' encompasses enhanced safety for users as well as simpler and easier-to-use systems that reduce the chance of user error; while 'security' involves increased security in response to the threat of theft and unauthorised usage, as well as the ability to track and locate blasting equipment.

Certain of the safety and security aspects described in this paper are common to many electronic detonator systems. In other instances, however, they relate specifically to a new range of electronic detonator systems, namely, the 'Shot' range developed by DetNet in South Africa. In particular, reference will be made to DetNet's surface-blasting product: the HotShot™ (trademark of DetNet South Africa (Pty) Ltd) auto-programmable system. This range of products is designed to go beyond providing access to the 'millisecond' and addresses new demands emanat-

ing from a dynamic mining and construction industry, while at the same time seeking to combat threats that exist in an increasingly volatile global environment.

2 SAFETY

This section covers two aspects of safety related to electronic detonator systems: those physically inherent in the systems and those that enhance safety on the bench as a result of software and hardware development capabilities of a digital platform, such as HotShot.

2.1 Safety inherent in the physical components

Current electronic detonator systems offer various sophisticated safety features, which have become a basic requirement for the technology. In those countries where detailed approval testing and certification is required, complying with these safety aspects is a prerequisite to use.

Extremely detailed research has formed the basis for the establishment of strict standards of compliance in South Africa (SANS1717-1), Europe (CEN/TS 13763-27) and Canada (CEAEC). Brief extracts from these regulations are used to highlight the level of assessment that electronic detonator systems can be required to undergo before acceptance.

The common standard safety protection afforded by electronic detonators relates to:

- inherent safety;
- electrostatic discharge (ESD) protection;
- over-voltage protection; and
- electromagnetic immunity.

In order to identify the opportunities for improvements in electronic detonator safety it is pertinent to examine existing standards and to reinforce the robustness of this hard-won knowledge platform.

The introduction of electronics and digital communication on the bench is still a very new and different concept for many who have had years of exposure to pyrotechnic systems.

It is recognised that lack of in-depth knowledge of the complex internal workings of the hardware and software of electronic detonator systems lead to negative perceptions regarding their safety.

There is still concern, for example, about taking an electronic device with a voltage output on to the bench to communicate with detonators that are ultimately fired by a supplied voltage. This paper will therefore attempt, in layman's terms, to raise awareness of key safety features already standardised and implemented in the HotShot system.

2.1.1 Inherent safety

Inherent safety is the term used to describe attributes of the on-bench communications equipment which guarantee that, although they send a voltage to the detonators for purposes of testing and trouble-shooting, the detonators cannot themselves be initiated during the process: an obvious requirement for a user working in close proximity to tons of high explosives.

A way to achieve this is to ensure that the voltage generated by the device always remains at a safe level below the no-fire voltage tested on each individual detonator before shipping, and that the coded firing signal cannot be generated by on-bench equipment.

Figure 1 illustrates inherent safety as it applies to HotShot. Inherent safety is achieved by introducing a series of voltage regulators and clamp circuits which guarantee that the output voltage from the hand-held device (Tagger) cannot exceed Vis. In addition, every detonator is supplied a voltage of Vnf and a complete firing signal sequence during manufacture, and any faulty detonator that is able to fire at this voltage is destroyed. This guarantees that all detonators leaving the plant require more than Vnf, as well as the firing signal sequence, in order to initiate.

If the Tagger cannot deliver more than Vis and the detonators cannot be fired below Vnf, safety is assured.

In the HotShot system, the logic to enable transmission of the coded firing signal is held inside a separate Blast Key which cannot be physically linked to the Tagger as a stand-alone unit. As

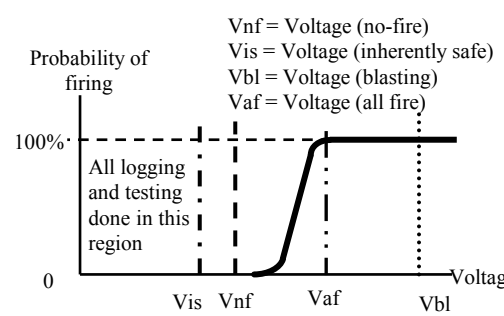

Figure 1. Inherent safety graph.

such, the Tagger in its stand-alone on-bench configuration cannot deliver a firing signal. Since firing voltage and firing signal are required to initiate a blast, double protection is provided against any unauthorised initiation.

2.1.2 Electrostatic discharge (ESD) protection

Electrostatic charges and their resulting discharges can be found in everyday life. When concentrated on equipment or individuals to a high enough level, static can create electric shocks – e.g. touching a metal door knob after walking on non-conductive carpeting.

ESD can create serious risk of ignition of explosives powders within a detonator. It can also damage electronic components such as micro-chips, circuit boards, etc., that are found in electronic detonators. The solution, therefore, is to ensure adequate ESD protection within detonators.

To do this, preferential discharge paths are deliberately included in the electronics module to steer any discharge away from the fuse head and sensitive explosives compositions. The HotShot detonator has withstood multiple discharges from the specified 30kV (1 Joule) test while remaining fully functional.

A question that is frequently asked is whether or not common ESD spikes can cause a detonator to fire prematurely.

In terms of the HotShot detonator, where in-depth knowledge is available, the answer is no. Such initiations are prevented by the following:

2.1.2.1 Construction of the detonator

Figure 2 shows the individual components of a HotShot detonator. The protection structures at the rear of the board provide a preferential discharge path to the copper shell and prevent discharge to the fuse head from the shell. An insulating sleeve, termed an 'H-plug', surrounds the fuse head and acts as ESD protection.

2.1.2.2 Firing signal

A complex digital firing sequence with numerous replies and confirmations that cannot be reproduced by naturally generated electrical impulses is required to initiate the blast.

Sufficient energy must be available from the storage capacitor to fire the fuse head. The transient nature of ESD makes this impossible due to the length of time required to charge the capacitor through the detonator protection circuitry.

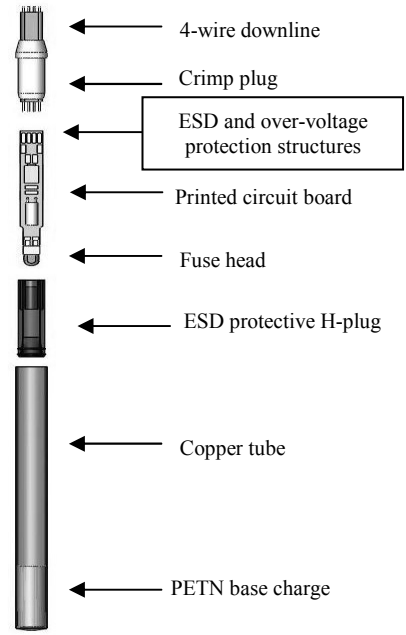

Figure 2. Detonator construction.

2.1.3 Over-voltage protection

Over-voltage protection governs protection against initiation from the deliberate or accidental connection of high electrical output devices such as mains current, electric detonator shot exploders, cap lamps and batteries to an electronic detonator. This test is included in the South African standard of compliance (SANS 1717-1), due to incidences with electric detonators, but is not included in the European Standard (CEN/TS 13763-27). In the HotShot system, damage and premature initiation is prevented by the on-board protection circuitry in the detonator.

The approval test passed by HotShot required multiple applications of DC voltages of 4.4V, 12V, 24V, 50V and 1kV as well as an AC test of 250V at 50Hz with no initiations. In developing the protection structure, multiple shot-exploder tests of up to 640 Joules were applied without initiation.

2.1.4 Electromagnetic interference (EMI)

Electromagnetic interference (EMI) emanates from a multitude of sources such as electro-hydraulic mining equipment, two-way radios, wireless networks, mobile phone transmitters, mobile

phones themselves, satellite communication transmitters, 'dirty' power-line emissions, lightning-induced pulses and electromagnetic pulses generated by explosives detonation.

When testing detonators during hook-up (when the user is most exposed to danger), the detonator is qualified to ensure that firing capacitor voltage does not exceed inherently safe testing limits in the presence of EMI.

When blasting (after all people are at a point of safety), the system is assumed ready to fire and EMI protection ensures that only valid test, calibration, programming, firing signals, etc., are received and correctly interpreted by the detonators.

During the development of the HotShot system, extensive use was made of electromagnetic compatibility testing laboratories to improve equipment's EMI immunity and to ensure compliance with stringent European and South African requirements.

It must be noted that should control equipment for whatever reason come into close contact with such EMI sources, the system will fail to safe.

Testing of HotShot to monitor the effects of EMI is undertaken in line with the requirements of SANS1717-1 and CEN/TS 13763-27.

2.2 On-bench safety features

Another range of safety features has evolved with the use of cutting-edge technology. This section highlights new capabilities designed into the HotShot surface system which ultimately improve safety and help reduce the possibility of user error in deployment and blast design. These safety benefits are seen on the bench in the form of system testing, the opportunity to eliminate out-of-sequence firing and safer blasting control. These safety benefits are given in summary below but are explained more fully later:

- *Reducung the probability of misfires.* Pre-blast confirmation of detonator integrity; no cut-offs; system auto-detects installation.
- *Opportunity to eliminate out-of-sequence firing.* Precision timing of detonators; software warning of connection errors and blast design irregularities; firing sequence visible from harness layout; software visualisation of final blast design.
- *Safer blasting control.* No unauthorised blast initiation; incompatibility with other initiation systems; abort ability.

In order to explain these benefits in more detail it is necessary to describe the concept behind the HotShot auto-programmable system. Highlighting the differences that exist between HotShot and existing systems is essential in order to illustrate the trends in technological change.

The main differences are encompassed in the term 'auto-programmability', which is a collective term for three key attributes: daisy-chaining; auto-detection and auto-delay assignment.

2.2.1 Auto-programmability

2.2.1.1 Daisy-chaining

Daisy-chaining is a term used by some to describe existing pyrotechnic delay systems that are connected in the order of firing. Besides initiating themselves, a 'signal' (electrical or shock) is passed on to the next detonator in the line, initiating the same process again; this continues until the last detonator is reached. With HotShot, the same principle occurs whereby the digital signal received by one detonator is actioned and passed on to the next detonator along the string—in other words, one detonator talks to another. The control equipment (in this case the hand-held device called a Tagger), positioned at the start of the installation can itself automatically visualise where each detonator is in a string. This, simply put, means that the user only has to connect one detonator to another along a row in order of firing and the job is complete, as is the case with shocktube systems. There is no longer need to log the position of each detonator. In currently available 2-wire fully programmable systems, a specific deployment methodology is required to know the detonator locations.

2.2.1.2 Auto-detection

Auto-detection is a term used to describe how the hand-held Tagger can detect and visualise the entire blast layout. It can determine the number of detonators, the number of rows (both left and right, if V-cut or closed chevron) and the number of detonators per row. A question often asked is: How can the Tagger self-detect the entire blast layout on its own? The answer is not as complicated as it would seem:

Daisy-chaining, described above, allows the system to scan and see detonators in each row, while specific hardware, such as Row Controllers (Fig. 3), is placed at the start of each row thereby enabling the system to scan and see how rows are

configured. A row is terminated by an End Plug, which is placed in the last connector. Out-of-pattern holes are accommodated by the addition of a Branch Controller within a string of detonators (Fig. 4).

Figure 5 shows a complete HotShot system installation and indicates how detonators are connected to one another along a string, as well as how hardware is placed within the installation to indicate specific locations to the Tagger. The Bench Controller is a communications device and acts as the interface between the 2-wire lead-in and 4-wire cable that links all components on the blast.

2.2.1.3 Auto-delay assignment

Thanks to auto-detection, the Tagger – with its own bird's-eye view of the blast – now only requires the user to enter the timing parameters for the blast. In the HotShot system, this is done by answering a series of simple, easy-to-follow prompts (e.g., what inter-hole delay; what inter-row delay; what inter-deck delay?). The Tagger

Figure 3. Row Controller.

Figure 4. Branch Controller.

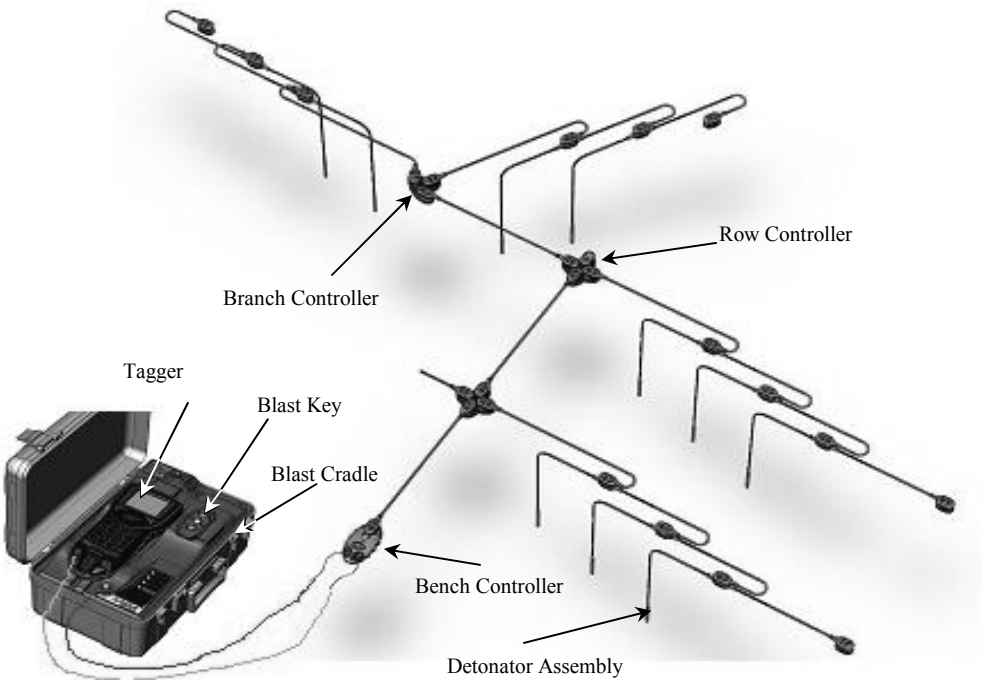

Figure 5. Complete installation.

matches this to its visualisation of the blast layout and automatically assigns the individual delay times required for each detonator. This requires no participation on the part of the blaster as it all happens in the background, and the assigned delays are automatically fed to the detonators as a one-off process prior to blasting. The final blast design generated can be scrutinised on easy-to-view graphical visualisations to ensure correctness.

Given a certain basic understanding of the HotShot system concept, we can now examine the safety benefits listed above.

2.2.2 Safety benefits

2.2.2.1 Reducing the probability of misfires

– *Pre-blast confirmation of detonator integrity.* As is common in most systems where control equipment and detonators can talk to each other (two-way communication), the integrity of the detonator, harness and connectors can be checked by the system. The HotShot system is designed to identify communication errors in detonators by their exact position in the blast (e.g., row 3, right leg, hole 9) via a clear visualisation linked to the blast layout. Figure 6 shows the position of the last working detonator and points the operator to check the next detonator in line, while Figure 7 shows a view of the blast layout with fault locations. Poor connections can be isolated and fixed, while faulty connectors can be replaced. This enables the blaster to ensure full functionality of the system thus minimising the chance of misfires from the blast. Where this capability is not available, a continuity break is not identified and, unless redundancy is available, a misfire will result.

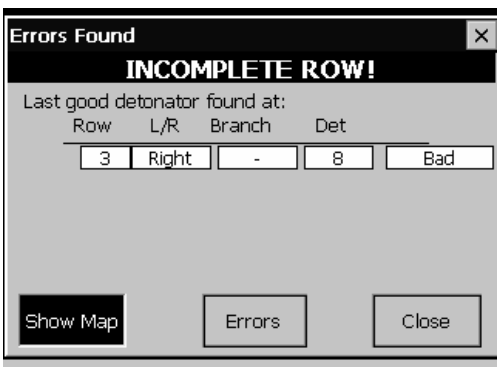

Figure 6. Error message.

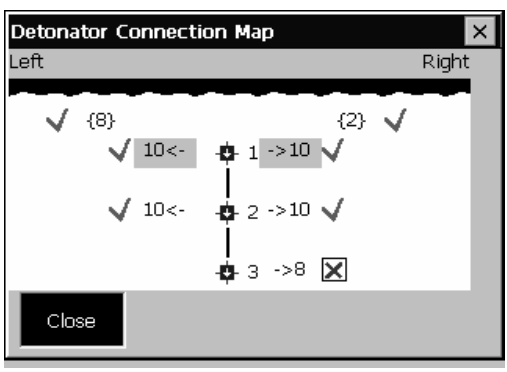

Figure 7. Map showing fault location.

– *Eliminating cut-offs.* As with most electronic systems, when the blast signal is sent, all detonators operate independently of the blasting equipment from that point onwards. As such, any break in the wire harness ahead of explosives firing will have no effect on the detonators' capability to fire—the ultimate 'burning-front'.

– *System auto-detects installation.* Since the HotShot system auto-detects all detonators connected and fires everything it sees, there is no possibility of detonators being added or lost from the installation without the control equipment becoming aware of the fact. The system will warn the user that detonators have been added or deleted since the last time the installation was scanned.

2.2.2.2 Opportunity to eliminate out-of-sequence firing

– *Precision timing of detonators*. This is a benefit well understood by the industry, whereby the precision of the electronic detonator overcomes the risk of out-of-sequence that may occur with pyrotechnic systems where 'cap-scatter' on long down-the-hole delays may overlap with surface timing. In the case of HotShot, the accuracy is within 1ms of program time, based on in-hole VOD measurements taken during field trials.

– *Firing sequence visible from harness layout.* Using the daisy-chain concept, HotShot is connected up in the same way as shocktube systems, i.e. in order of firing. A blaster can now see the order of firing by looking at the harness connections on surface—any errors

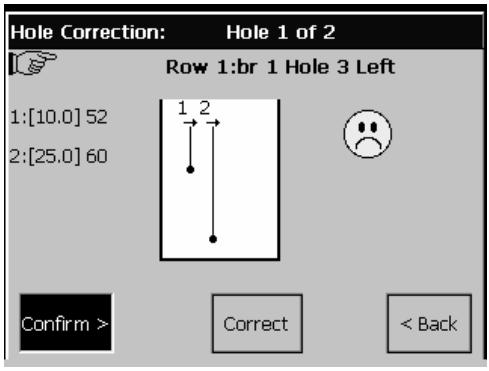

Figure 8. Software warnings of connection errors.

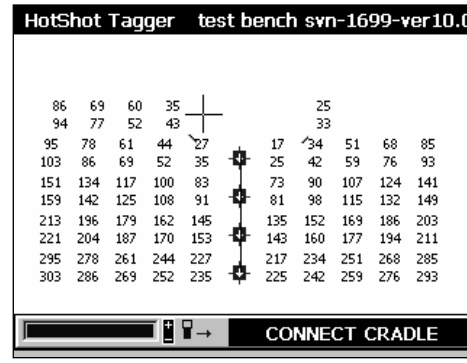

Figure 9. Helicopter view.

are visible and can be corrected. This overcomes a problem whereby incorrect delays may accidentally be assigned to detonators on a random-harness connection resulting in out-of-sequence initiation. Figure 5 can be reviewed to see a typical order-of-firing harness layout.

– *Software warnings of connection errors*. Combining auto-detection of the blast with input on blast parameters, the HotShot system can highlight connection errors graphically with clear messages and correct them without the blaster having to return to the bench. An example is illustrated in Figure 8. The screen displayed to the blaster shows that he has inadvertently connected up a decked hole so that the top deck fires first when the rest of the blast is connected to fire the bottom decks first. Similar warnings occur should the blaster design rows to fire out of sequence which may, in certain circumstances, be warranted.

– *Visual display of blast design*. In line with the rapid development of hardware and software in hand-held PCs, electronic detonator systems now have the ability to provide the blaster far more information thanks to large colour-screen LCD displays. As is the case with any blast, it is critical for the blaster to ensure that his blast design is correct and that he has access to the delays that are automatically assigned by the system. This is available with HotShot as a 'helicopter view' of the entire blast layout (Fig. 9) whereby all assigned delays are displayed for validation. There is also a facility to simulate the firing sequence in slow-motion, for visual validation that the design meets requirements. Any out-of-sequence or unexpected result can then be corrected.

– *Continuous monitoring of system voltages*. Due to advanced communication techniques, key hardware components in the HotShot system, such as Bench Controllers, have their voltages monitored continuously right up until firing point. The system can therefore warn the user should a critical component fail. In the case of equipment that controls individual sectors of a blast, the loss of one such unit if not detected could cause out-of-sequence initiation.

2.2.2.3 Safer blasting control

– *Unauthorised blast initiation*. Strict blasting procedures are crucial when it comes to firing the blast. Software can greatly assist in enforcing such procedures and preventing unauthorised access to the system. In the HotShot system, the blasting sequence is followed as a step-by-step process that cannot be circumvented. The Tagger, used to test and design the blast, is inserted into a Blast Cradle positioned at the point of safety (Fig. 10). Each Tagger has a password specific to its operator, and this would be the first hurdle that an unauthorised user would come up against. At the arming phase, when the voltage is raised above the safe voltage Vis (Fig. 1), the system requires a removable Blast Key (Fig. 11) to be inserted and a password entered, which is linked to that key alone. The Blast Key contains the high voltage circuit, the arm button, both fire buttons as well as the necessary encoded logic to enable the blast to take place. Without the Blast Key in position, no blasting can occur. Even if an unauthorised person had managed to obtain a Blast Key, he would need to know the six-digit password specific to that key.

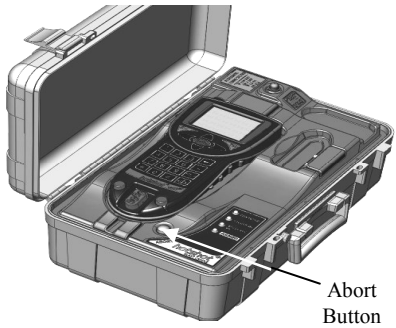

Figure 10. Tagger docked.

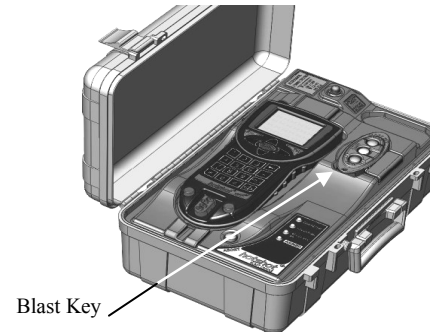

Figure 11. Tagger and Blast Key docked.

This number is only issued to the certificated HotShot user allocated the key. Once past this stage, firing of the blast requires the deliberate action of firing two blast buttons on the Blast Key simultaneously.

– *Abort ability.* The Blast Cradle (see Figs 10-11) carries an abort button, which if pressed at any point during the blasting sequence up until the point of firing itself, causes the system to shut down immediately. This could be actioned should a person be seen entering the danger area. Besides the abort button, there are other ways to abort a blast, e.g. removing the Blast Key, removing the 2-wire lead-in from the Blast Cradle or removing the Tagger from the Blast Cradle.

3 SECURITY

Banks and governments are usually at the forefront of implementing technological develops in the computer and electronic fields, particularly when these bring about enhanced security. Owing to the nature of explosives detonation, applying security enhancements to detonators is not as straightforward. A number of factors are involved here:

– *Cost*. Most electronic detonator manufacturing companies have already invested a substantial amount of money in developing a design that has been implemented in a dedicated microchip. These manufacturers therefore attempt to recover development costs before major new microchip work is undertaken.

– *Low volume.* The current non-electronic detonator market remains very large and many sites must still be converted to electronic detonators before secure detonator systems can be introduced. The step change necessary for the mass market conversion to electronic blasting has still to occur.

– *Emerging technology*. There is limited availability of technologically secured detonators (e.g. passwords, coded signals and encryption) and an unproven track record of their features and capabilities.

A combination of these factors has resulted in electronic detonators not being as developed, in security terms, as one would have expected given their inherent capability in this regard.

3.1 Security advantages with current electronic detonators

Electronic detonator systems that are currently on the market already have major security enhancements compared to other non-electronic detonator systems. This is derived from the way most electronic detonators operate, requiring both a command sequence and energy to initiate detonation. All non-electronic detonators need only energy to initiate detonation; however, energy sources to start the initiation of non-electronic detonators are readily available and it is possible, with intent, to find a means other than the supplied equipment to initiate the detonators.

Electronic detonators such as HotShot are designed only to be initiated by firing equipment specifically designed for the purpose. Thus the only way to initiate an electronic detonator such as HotShot is to supply the right energy level for the right amount of time at the right location in a complex sequence of commands. This is extremely difficult to achieve without the correct firing equipment, making accidental or wrongful initiations by unauthorised people virtually impossible.

It is, however, not impossible to reverse engineer the communication sequence of standard electronic detonators and to build an electronic circuit that emulates the correct command sequence and energy levels to start initiation of the electronic detonators. But to do this, the person must have malicious intent to use the detonators wrongfully and must have the necessary electronic tools and knowledge to complete such a task.

Current electronic detonators do not solve this problem; it is only overcome using encrypted electronic detonators.

3.2 Security advantages with future electronic detonators (encrypted detonators)

In the future, electronic detonators might be equipped with encryption keys and all commands to the units will be encrypted with the correct key(s) in the control equipment. For the purpose of this paper we will refer to these electronic detonators as encrypted detonators.

These keys can be matched so that only specific dedicated blasting equipment can communicate or initiate an encrypted detonator. In this way, total control over the authorisation of a blast can be achieved.

Encryption security is directly linked to the size of the encryption key length used: the longer the key sequence the less chance there is to crack the code and use the detonator maliciously. High-level encryption with long cryptographic keys is so secure that banks and governments literally entrust their whole businesses to these encoded systems. The larger and more complex the encryption process the more effort it takes to break into the encrypted system. All systems can be cracked, it is just the amount of effort required to do so that distinguishes the more secure systems from the less secure ones.

There is no need at present to implement an extremely high level of encryption on encrypted detonators. There must be a balance between the complexity of the encryption and product use. One's first thoughts would probably be that detonators are dangerous and can be used for malicious intent and must therefore be very securely encrypted; however, this is not the case. The encryption complexity must match the complexity of building a detonator or opening a detonator and bypassing the electronic module.

Complex encryption systems in encrypted detonators will not make these detonators more secure than lesser, but adequate, encrypted systems if both systems require the same level of effort to open and bypass the physical electronic components. Thus encryption levels only have to increase as more secure mechanical ways of securing detonators or new and more complex initiating systems are introduced inside the detonators.

In areas where non-electronic detonators are readily used there is no gain by introducing an encrypted detonator, since the person with malicious intent will use these easy-to-detonate non-electronic products rather than encrypted detonators.

3.3 Next level of security and control

Having secure detonators is only the first step in a totally secure and authorised blasting environment. In order to close the security loop, the control equipment used to initiate a blast must also be fully monitored. The advances made with GSM communication systems and the availability of GPS, mean that a dedicated authority can easily control blasting systems. The following models can be used:

- *Blast per location*. With the introduction of a GPS or any other location device in the control equipment, a blasting system can be 'locked' down to operate only in a dedicated geographical area. In other words, certain blasting equipment will only function at a particular mining site. Removal of the equipment by unauthorised personnel will render the equipment useless, thereby limiting theft and malicious use. Reverse engineering of the equipment is, however, still possible if a person with malicious intent has the necessary funding, skills and electronic equipment.
- *Assignment of electronic detonators to specific customers*. Regional or customer codes can be assigned to a particular batch of electronic detonators. If the control equipment is then configured to operate only with a specific code or range of codes, control can be implemented such that detonators can work only with dedicated equipment.
- *Get blast authorisation before blasting*. With the introduction of a GSM cellular module or any other means of communication in the control equipment, blast authorisation can be obtained via the GSM network over a secure link to an authorisation centre before any blasting is initiated. A timed window of blasting can also be issued with such a system, authorising a blast to occur within a particular time period.
- *Combine GSM and GPS systems*. Such a system would incorporate all the security

features of the GPS and GSM or communications-linked systems. Thus only authorised blasts within an assigned time window at a particular location can be initiated. Dedicated electronic detonators assigned to a specific blasting site would be the best security model currently proposed with existing electronic detonators such as HotShot.

3.4 The secure future

With the introduction of encrypted detonators and the combined GSM/GPS security system as described above, blasting systems could become extremely secure. Such a system could operate as follows:

The batch code of the detonators would be scanned or read in by the control equipment. The control equipment would ask for authorisation from the authorisation site via a secure link over the communication medium (most likely the GSM network). The encrypted request message from the control equipment to the authorisation site would have the batch number for the detonators, GPS location of equipment, blaster identification, etc. The authorisation site would verify if a particular batch code is allowed to be used at a particular location. It would also verify if the blaster is a valid blaster and if the person is authorised to blast at that location.

If all the criteria are met, the authorisation site would issue the decryption key of that particular batch code over the secure communication link within a specific time window in which the blast has to occur.

Such a system would assist in preventing unauthorised use of detonators or use for purposes with malicious intent.

4 CONCLUSION

The digital communications platform underlying electronic detonator technology means that the opportunity now exists to provide an initiating system that can deliver enhanced blasting benefits, while at the same time offering new levels of safety and security. In terms of the latter, electronic detonator systems have the capability to minimise safety risks on the bench as well as to reduce the opportunity for unauthorised and malicious use away from the bench.

Continued volatile global conditions will demand far more from detonator systems of the future. Secure encrypted detonators which can only be fired by legitimate blasters, at legitimate times and from legitimate locations could be a common sight on the blasts of tomorrow.

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A digital surface remote blasting system

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ABSTRACT: This paper describes a newly developed Surface Remote Blasting System (SURBS) for electronic detonators, which is currently being introduced into surface blasting applications. The rationale for developing the SURBS was to meet key needs of mine operations, such as minimizing the shutdown time and the capability to conveniently initiate blasts in difficult terrain from a safe distance. Increased distance from the blast site, combined with a good overview of the site, enables maximum safety during blasting. The system also removes the inconvenience of running long firing cables. The SURBS consists of the field-proven detonator and equipment technology of an existing electronic detonator system, a Surface Remote Blasting Box and a Surface Remote Blaster. The Surface Remote Blaster is the Master unit and controls the Remote Blasting Box via encrypted radio communication using a sophisticated safety technology, which guarantees that the highest safety standards are maintained.

1 INTRODUCTION

Five years ago Orica Mining Services launched its first electronic blasting system i-kon™. Since then many mines have successfully deployed the system achieving significant advantage in terms of safety, cost and environmental compliance. The application of this original system requires a firing cable between the 'Blaster' at the point of safety and the 'Loggers' and detonators at the blast site.

The new Surface Remote Blasting System makes hardwiring between the point of safety and the blast site a thing of the past and enables a remote blast initiation with the known precision and flexibility of the existing system. Instead of a single Blaster, which connects to the Loggers and detonators via the firing cable, the Blaster function is divided onto two new pieces of control equipment: the Surface Remote Blaster (Blaster2400R) and the Surface Remote Blasting Box (SURBB). The Blaster2400R is the Master of the system and controls the SURBB via radio on a licensed and approved frequency. The SURBB is a Slave to the Blaster2400R and supplies the firing energy to the attached Loggers and detonators.

All communications between the two units are encrypted for complete security with a digital single-use code. The single-use encryption code is carried on a Smart Dongle. The line-of-sight operating distance between the Blaster2400R and the SURBB is up to 8200 feet or 2500 m. The units are small, compact, portable and ruggedly designed to stand up to the demanding conditions of surface mining. The maximum system capacity is 2400 detonators in a single blast.

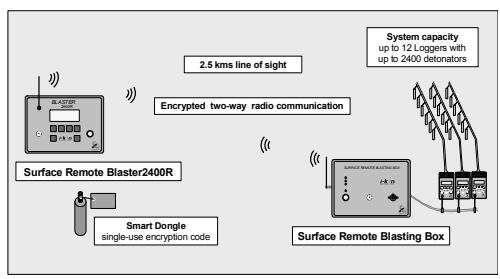

Figure 1. Surface Remote Blasting System.

The SURBS also has a Test Mode, which allows the strength and quality of the radio signal from various pit locations to be tested.

The benefits to using the SURBS in a quarry or open cut operation are clear. Blasting will be safer, because the blast can be initiated from a safe distance well away from unstable ground and fly rock. Blasting will also be more convenient,

because it will be no longer necessary to run a long length of firing cable under time pressure, often in difficult terrain, crossing haulage roads, etc. This will also contribute to cost savings, because with SURBS, the shutdown time for blasting will be significantly reduced, and mine and quarry efficiency will improve.

2 SYSTEM COMPONENTS AND OPERATION OF THE SURBS

2.1 Detonators and Logger(s)

The fully programmable electronic detonator can be programmed from 0 to 15000 milliseconds in 1 millisecond increments. The detonators are capable of two-way communication. Each detonator has a unique factory assigned ID-number, which is written to the detonator microchip. This is the basis for full two-way communication between detonators and control equipment, allowing the operator to test the detonators at all stages during preparation of a blast.

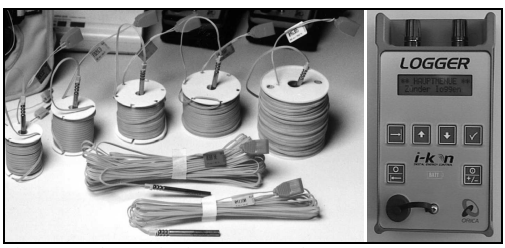

Figure 2. Electronic detonators and Logger.

During hook-up of the detonators to a twin twist harness wire, the desired delay times are assigned to individual detonators by means of a Logger. The detonator is attached to the harness wire with a clip connector.

The Logger is capable of storing data for up to 200 detonators. For assigning delay times to the detonators it can be operated in four different modes (Manual, Auto, Delay numbers or SHOT-Plus®-i), thus enabling the operator to choose the most convenient way for the application.

2.2 Surface Remote Blasting Box

After the detonators have been logged and tested, the Loggers are connected in parallel to the Surface Remote Blasting Box, which must be positioned a safe distance from the blast. A single SURBB can fire up to 2400 detonators programmed via a maximum twelve Loggers.

The SURBB provides the higher energy level required for programming and firing a blast. It is operated via encrypted two-way radio communication from the point of safety.

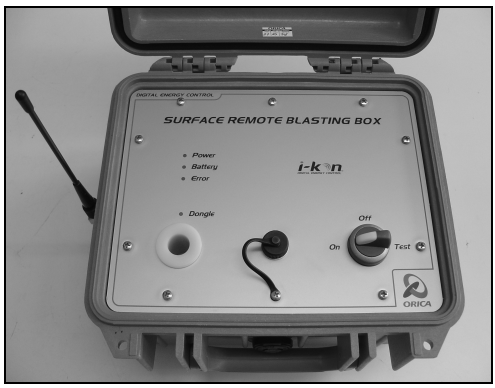

Figure 3. Surface Remote Blasting Box.

For activation of the SURBB the shotfirer needs a Smart Dongle. As part of the activation sequence the SURBB generates a single-use encryption code and writes it to the Smart Dongle. A successful activation sequence is indicated by the LEDs on the front panel, after which the SURBB is in standby mode. The standby-time can be set to a maximum of four hours by the operator. The blast must be fired within the standby-time. If the standby-time is exceeded the SURBB shuts down automatically and the single-use encryption code becomes invalid. During standby mode no power is switched to the output terminals. In standby mode the SURRB can only be addressed via radio by the Blaster2400R with the valid encryption code. With the start of the blasting sequence the two-way communication between the two units is continuously monitored. If the data transfer is disrupted, the SURBB will automatically default to standby mode. After firing, the SURBB shuts down automatically.

2.3 Surface Remote Blaster2400R

After activation the shotfirer must transport the Smart Dongle with the valid encryption code to the safe blasting location. This location will be at a safe distance from the blast but close enough that the shotfirer has a clear line-of-sight view of the blast and the approaches.

The shotfirer then inserts the Smart Dongle in the Blaster2400R, which reads the encryption code. It is only after the Smart Dongle has been read that the Blaster2400R can communicate with the SURBB.

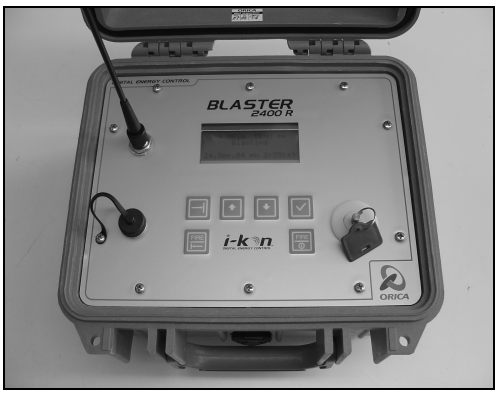

Figure 4. Surface Remote Blaster2400R.


Figure 5. Firing from a safe distance.

The whole blast sequence is controlled via full two-way communication from the Blaster2400R, and the shotfirer can continuously verify that all the detonators are responding and fully functional, right up to blast initiation. Shotfirers will now be able to easily position themselves to clearly view every blast. No longer will they have to rely on third-party information to verify that everything is clear before and after firing. This remote blasting system enhances the safety and convenience of blasting.

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Prediction of the damaged zone for ground support around underground openings formed by blasting

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ABSTRACT: This technical paper discusses a new non-destructive acoustic testing method – the impulse response (IR) testing method – for the damage zone around underground mine openings. The IR test can quickly map fractures and rock quality (such as fragmented, delaminating and cracking rock), as well as provide information on variations of geology in rock mass. This new method was applied to measure the damage zone of blasting used in the production drift. The experimental tests used pyrotechnic delay and electronic delay detonators, both of which have very similar timing and loading patterns. The impulse response (also known as the 'impact response') is a good testing tool for quick validation of rock stability and quality (mobility) around the new opening before and after each blast. Further validation in different rock properties should be explored in the future.

1 INTRODUCTION

1.1 Background

Drilling and blasting is a very common technique for hard rock mines. Over the last few decades, the damage (cracks in the vicinity of the new surface, overbreak, backbreak) due to blasting was only able to be indirectly forecasted by using vibration data from the local geophone stations. Natural, uncontrollable variables such as geological conditions affect the blast results directly. Controllable variables such as choice of explosive and initiation systems could greatly influence the extent of the blast damage zone (see Fig. 4) Fractures and cracks in the vicinity of the new surface of the support wall around the new opening are a serious safety hazard for miners and mining equipment.

In order to develop better protection for miners and equipment, as well as introduce new tools to minimize the actual blast damage due to production blasting, a new preblast and post-blast surveying method of the drift walls was introduced. This test method measured the impact of a single blast as well as continuously measuring the impact of repeated blasting inside the same drift during the progression of the drift. This new method is not only a very simple and efficient means of collecting data, it also validates the data. Each test measurement took less than 30 min. In addition, this work also quantitatively compared the performance of electronic and pyrotechnic detonators in underground production drift development using blasting.

The new post-blast surveying method utilised the impulse response (IR) test method. The IR method is a non-destructive, stress wave test that is widely used in the aircraft industry (Davis, 1974–2003). This method has not yet found common use in underground construction or mining applications. Hence, the IR method was applied to underground mining and testing. It measures the dynamic mobility and void index of the drift walls, as these enable an estimate of the blast damage to be made.

The basic theory of dynamic mobility developed up to this point has not changed. However, its range of application to different structural elements has expanded to incorporate many practical day-to-day problems. Figure 1 illustrates the use of the test unit and gives an example of test results using a contour plot for graphical analysis of the rock wall.

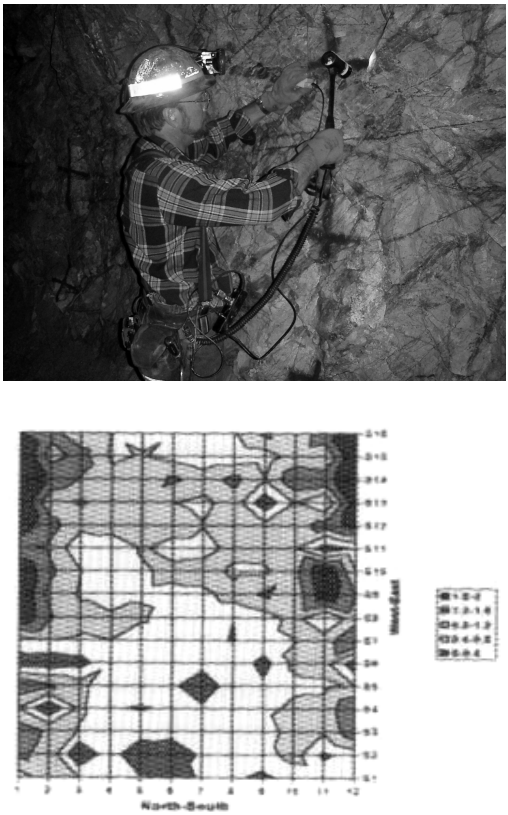


Figure 1. Use of the test equipment and an IR data contour plot from test results (no units).

1.2 Impact loading model

The IR apparatus model can be represented by the discrete mass–spring model shown in Figure 2. The mass is released and strikes the top of the spring at time t_o. As the spring compresses, the structure begins to displace downwards, relieving the force in the spring. Therefore, the force that the impact response imparts is actually a function of the mass and stiffness of the structure as well as the IR mass and drop height. To examine this force, it is necessary to consider the case in which impact load tests are performed on a non-deforming surface. Assuming that point B in Figure 2 is fixed, the total downward displacement of point A after time $t = t_o$ is given by the following equation:

$$y(t) = \frac{v_o}{\omega}\sin\omega t + \frac{W_m}{K}(1 - \cos\omega t) \quad (1)$$

where $y(t)$ = downward displacement at point A (m); v_o = velocity of mass at $t = t_o$ (m/s); w = natural frequency of oscillation (Hz); W_m = weight of the falling mass (kg); K = equivalent spring constant (kg/m); and t = time elapsed from impact (s).

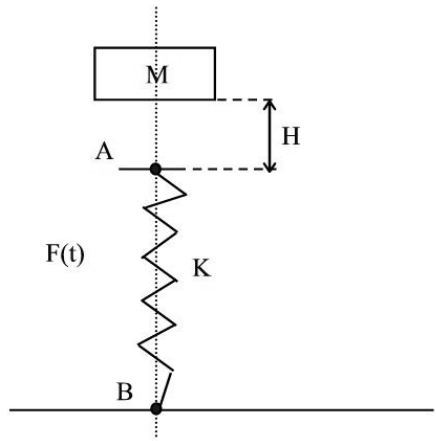


Figure 2. Single-degree-of-freedom mass–spring model for the IR system.

The velocity of the mass at impact, v_o, is given by the equation:

$$v_o = \sqrt{2gH} \quad (2)$$

where g = acceleration due to gravity (m/s^2), and H = drop height of mass (m).

The natural frequency of the IR system, ω, is defined as:

$$\omega = \sqrt{\frac{K}{M}} \quad (3)$$

where M = falling mass. Equation 1 can be written as follows:

$$y(t) = \left[\frac{2gHM}{K}\right]^{1/2}\sin\omega t + \frac{W_m}{K}(1 - \cos\omega t) \quad (4)$$

The second term in the equation represents the displacement at point A that is caused by the dead weight of the mass. For a drop height ranging from 20 cm (8 in) to 35 cm (14 in), this term accounts for less than 3% of the total displacement. Therefore, it can be ignored, and the equation for the downward displacement at point A essentially becomes:

$$y(t) = \left[\frac{2gHM}{K} \right]^{1/2} \sin \omega t \quad (5)$$

The force that the IR imparts to the structure at point B is determined by the following relationship:

$$F(t) = Ky(t) \quad (6)$$

Therefore, the equation for the load as a function of time is as follows:

$$F(t) = (2gHMK)^{1/2} \sin \omega t \quad (7)$$

which is valid for impact load tests on non-deformable structures for the time range in which the falling mass is in contact with the springs and the springs are in compression, that is, $0 < t < \pi/\omega$.

Equation 7 is useful for experimentally determining the equivalent stiffness of the test apparatus. From Equation 7, the peak load occurs at time $t = \pi/2\omega$ and is given by the following equation:

$$F_{\max} = (2gHMK)^{1/2} \quad (8)$$

The mass and drop height are known, and $F_{\max}$ is measured by the geophone positioned next to the falling weight (hammer). The only unknown is the equivalent stiffness, K, of the IR. By selecting a test point that experiences very small displacements, such as a concrete block or solid rock, the equivalent stiffness of the IR apparatus can be determined. Tests have shown that the equivalent stiffness, K, increases as the mass or drop height increases. This increase can be attributed to the material non-linearity of the testing material and the geometric changes in the cross-section of the material as deformation occurs. Therefore, if a constant equivalent stiffness is used, it must be determined experimentally for each mass and drop height combination.

2 EXPERIMENT

2.1 Introduction

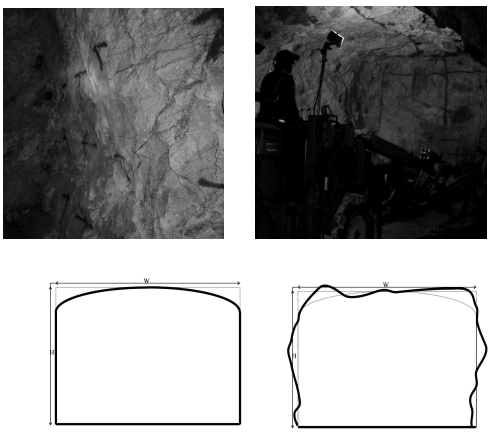
A test study was carried out at the Henderson Mine in Empire, Colorado, property of Climax Molybdenum Company, during May 2004. The objective of the test was to demonstrate that new blasting technology could reduce the risk of post-blasting damage to surrounding walls via the utilisation of new electronic detonators. The post-blast damage was quantitatively measured using non-destructive impulse response (IR) (also known as 'impact echo') testing equipment. There were three test blasting rounds fired for each type of detonator – pyrotechnic and electronic.

An already existing drift wall (which will be used for future mining production) was selected as the test site. The new development was excavated with the following dimensions: width 4.2 m (14 ft), height 3.9 m (13 ft), calculated cross-sectional area 26.9 m^2 (289.5 ft^2), approximate length of total excavation 40 m (130 ft), with each round pilling to depth of 3.7 m (12 ft). The series of six total blasts (37–42 blastholes per test) was performed in relatively similar and uniform rock strata about 915 m (3000 ft) below the surface on the 720th level. The number of holes per test was variable owing to communication barriers between different drillers and different shifts.

The cross-sectional area of the new drift opening has a trapezoidal shape with a circular roof (see Fig. 3).

Non-destructive test surveying has been carried out in the drift to study the dynamic mobility of the right side of the drift wall before blast and after blast. The mobility of the rock mass represents the harmonic loading when different angular frequencies are applied at different locations, and measurement of the response makes it possible to identify the eigenfrequencies and eigenmodes. The mobility is defined as the maximum velocity at a point, divided by the maximum load. In other words, the IR method uses a low-strain impact to send stress waves through the tested element. The impactor is usually a 1 kg sledgehammer with a built-in load cell in the hammerhead. The maximum compressive stress at the impact point in rock is directly related to the elastic properties of the hammer tip. A typical peak stress level is 5 MPa for a hard rubber tip. Response to the input stress is measured by a velocity transducer (geophone) placed adjacent to the impact point (direct mobility). Both the hammer and the geophone are linked to a portable field computer for data acquisition and storage. The bending mode, in a 0–800 Hz frequency range, generated by the tip of the hammer and the stress wave are processed in the field computer using a fast Fourier transform (FFT) algorithm. The ratio of the Fourier transform of the maximum velocity and the Fourier transform of the maximum load is determined as the direct mobility of the rock. Field experience shows that the average mobility value

over the 100–800 Hz range is related to the density and the thickness (i.e. stiffness) of the rock wall. A reduction in rock wall thickness corresponds to an increase in average mobility of the rock wall. For example, when the rock wall is debonded in the form of sandwiched front and back parts of the face, the mean mobility reflects the thickness of the front part. Also, any cracking in the rock wall will increase the mobility over the tested frequency range.



(a) (b)

Figure 3. Drift profile: (a) theoretical cross-section; (b) real cross-section (not to scale).

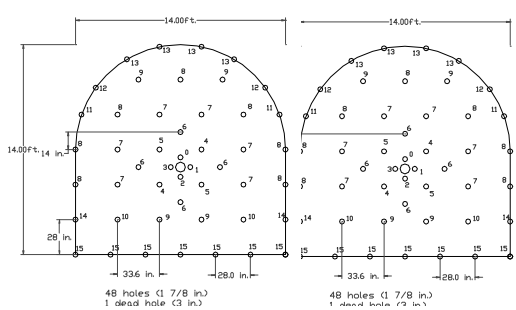
Therefore, the more solid the rock mass, the lower the mobility will be by comparison with weak or loose rock. The quality of the rock mass at this drift included an assessment of both the average mobility and the void index, which showed minor fractures and minor surface spalling of the tested rock wall. The new surveying method was able quantitatively to map the rock damage after each blast test. Figure 3 shows the theoretical drift profile versus the excavated real profile with many overbreaks and underbreaks.

As a result of certain safety restrictions enforced by the mine manager, access to the post-blast drift was allowed only after the muckpile had been excavated, all the drift walls had been scaled and washed and, in some instances, new rock bolts had been installed prior to our surveying.

2.2 *Drill pattern*

In each round, between 38 and 42 holes of 4.5 cm (1.78 in) diameter and 3.7 m (12 ft) length were drilled. The holes were drilled parallel to the drift, with a small drilling deviation (20 ft of drill hole length with 1 in deviation) and 0° angle. The drilling pattern used a four-square burn cut. The production holes were loaded pneumatically with ANFO bulk explosive, for a length of 2.7 m (9 ft). The choice of explosive was based on its high gas energetic partition. At a distance of 0.9 m (3 ft) from the drift face, a plastic cone was inserted into the drill hole and refilled with ANFO, in order to maximize the time of gas retention. Control holes (also known as perimeter holes) of 3.7 m (12 ft) length were drilled as the production holes. The control holes were loaded pneumatically with ANFO bulk explosive (density 1.0 g/cm^3) for a length of 0.3 m (1 ft) and 1 m (3 ft) air decking, plugged with plastic cone and reloaded with ANFO. The bottom holes (lifters) were hand loaded with emulsion explosives [Magnafrac Plus HW 32 mm × 400 mm (1.25 in × 16 in), density 1.13 g/cm^3, velocity of detonation (VoD) 4.7 m/s and relative bulk strength (RBS) with respect to ANFO 133 g/cm^3, with excellent water resistance (15.4 ft/s) and stemmed with a plastic cone].

The only difference between the shots was a change to the electronic detonators in the initiation sequence. Figure 3 shows the improved time sequence for production blasting of the test drift. The time sequence in Figure 4a is a 200 ms down-hole delay, which represents detonator No 8. Figure 4b shows the improved sidewall time sequence using No. 14, which is a 350 ms down-hole delay. Non-electric shock-tube pyrotechnic delay detonators of the EXEL™ LP series were used for the first part of testing. New precise electronic i-kon™ detonators were used for the second part of the testing.



38 – 42 holes 1 7/8 in

(a) (b)

Figure 4. Improved time sequence for production blasting of the test drift: (a) time sequence of sidewall No 8 (200 ms); (b) improved time sequence of sidewalls using No. 14 (350 ms).

2.3 *Testing*

The test area was divided into a rectangular grid mesh on the top surfaces of the new drift wall, as shown in Figure 5. The first mesh consisted of eight columns by four rows, and, after every test blast, this mesh was extended by another eight columns and four rows. The mesh was a rectangular shape, and each rectangle was 1 m (3 ft) by 0.75 m (2.5 ft). The grid mesh density was not increased because the testing required was fulfilled (see Fig. 6), and the second step consisted of three test runs with electronic delays. A total of six test results were evaluated. This range of tests was sufficient to derive the standard statistical results.

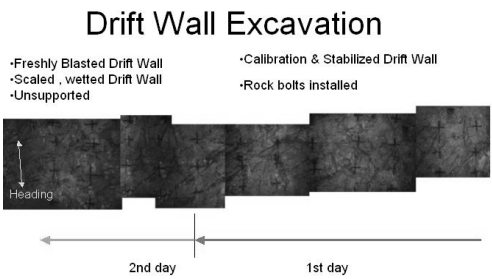


Figure 5. Drift wall excavation and expansion of the IR surveying.

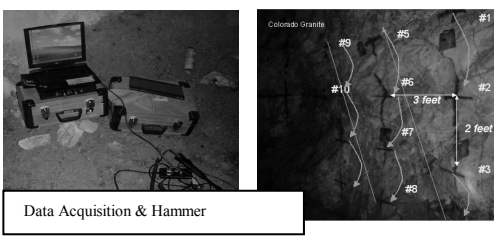


Figure 6. Test equipment: a data acquisition system with an amplifier built inside the laptop computer, with an impact hammer and geophone for data collection. Experimental rectangular grid mesh on the top surface (red crosses).

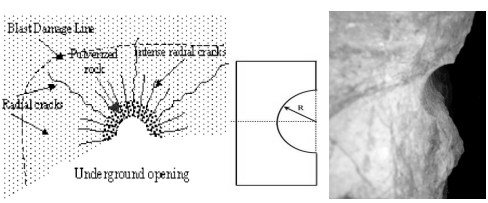


Figure 7. Typical half-barrel after the test blast: (a) sketch of half-barrel; (b) blast result of half-barrel.

Figure 7a shows the theoretical mechanisms of rock breakage around the control blasthole, and Figure 7b shows the result for the half-barrel during our experiments.

As shown in Figures 8a, b, the blast results of cross-sectional profiles using electronic and pyrotechnic delay differ in the number of half-barrels around the surface of the underground opening.

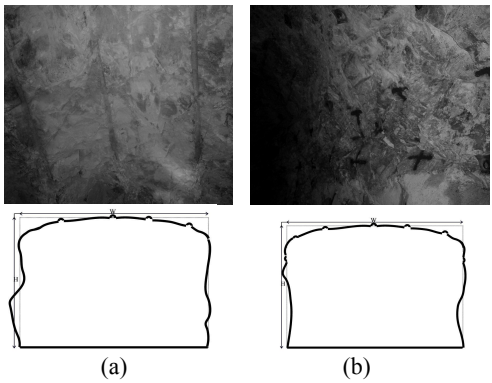


Figure 8. Half-barrel profiles around the opening: (a) typical roof profile when pyrotechnic delay detonators were used; (b) typical roof and top side of wall profile when electronic detonators were used.

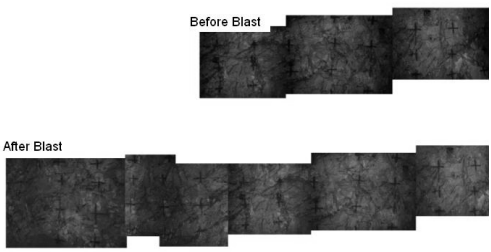
2.4 *Discussion and conclusion*

All the blastholes were initiated at the same time in order to prevent cut-offs. Long time period pyrotechnic and electronic detonators were used for sequential blasting to validate the test results. Owing to time and personal constraints, no attempt has been made to measure vibration for individual blast tests or to correlate the blast results with additional vibration analyses.

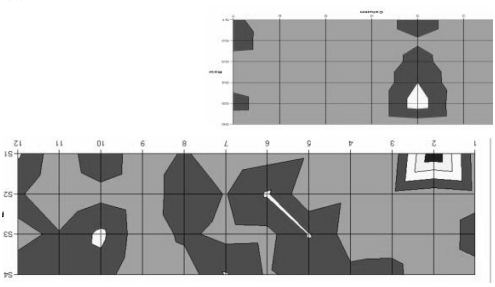
Inspection of the test drift after the blast showed that the number and location of half-barrels increase when electronic detonators are used (Fig. 8). Therefore, some improvements of post-blast damage have been observed from the visual inspection. Figure 9 shows the different test results observed during testing with pyrotechnic delay detonators. The number of half-barrels was 3–4, located only on top of the roof of the drift. More importantly, post-blast damage was observed around the drift contour. Compared with electronic delay detonators, we found that the number of half-barrels increased appreciably to 5–7, located not only on the roof but also on upper parts of the drift walls. Elsewhere, the post-blast damage was relatively minor, considering the power of the blast and the proximity of the walls.

The post-blast damage was limited to a light spalling of the walls. In this part of the drift the geology was more complicated and very fractured. Even in the portion that had unfavourable geology, we saw an improvement when we used electrical detonators.

The results show the difficulty in measuring the efficiency and the degree of post-blast damage. The objective of IR surveying is to reduce the risk of post-blast damage and the risk of uncontrolled failure affecting miners or mining equipment. The precision of the instruments used allowed the post-blast damage to be assessed, as well as the effects of repeated blasting in the same drift. The test survey results show that the rock wall was continuously losing integrity, and surface spalling was observed. Monitoring of the blasting activity in the drift during the next few months could add more information.



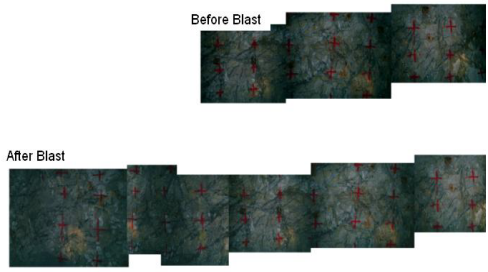
(a)



(b) Average mobility

Figure 9. Test results before and after blast: (a) wall condition before and after blast; (b) test measurements of average mobility of drift wall before and after blast.

The IR technique helps us to visualize the rock mass properties before and after the blasting. These test results show that, for each blast that was heavily loaded, a loss of integrity in the drift surface was observed. Areas of rock wall damage and decrease in ground wall stability were detected during each blast. Some of the loose boulders and fractured rock wall and area were removed during each blast or during scaling of the roof and walls. The IR method was the primary test method used in this surveying.



(a)

(b) Void Index

Figure 10. Test results before and after blast: (a) wall condition before and after blast; (b) test measurements of void index of drift wall before and after blast.

During production blasting, the right drift wall was surveyed for damage due to blasting on the drift wall. As can be seen in Figure 9, the test results show an increase in mobility. The increase in mobility of the drift wall is due to deformation and damage to the walls during testing.

A possible way to improve this blasting practice for production blasting is to adjust the blast design in order to reduce the number of holes and reduce the amount of explosive per blast by 10–15%, as well as improve the blast time sequence using shorter time delays between the holes. Electronic delay detonators can be used to improve the blast time sequence. Also, an additional alternative is to try one of the presplitting techniques or smooth blasting techniques, perhaps combined with bulk emulsion explosives. These combinations may help reduce the potential post-blast damage due to standard ANFO loaded blastholes. To be able to troubleshoot this blasting practice, more time is required to validate the suggested changes. The present author believes that in the near future this new technique will be validated for different rock types.

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Interference patterns - predicting vibration concentration for precise detonators

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ABSTRACT: Precise electronic detonators have significant advantages in vibration control. The precision allows timing designs that take advantage of destructive interference effects to limit vibration amplitudes. However, the opposite can occur, where constructive interference leads to an unexpected increase in vibration amplitude. The risk is more acute when precise detonators are used, where zones of constructive interference can occur in certain directions, even though delays have been chosen that predict destructive interference. This paper presents a graphical computer method for testing the radiation interference patterns that are generated on the surface around a blast. The model takes into account the blast timing and the geometric positions of blastholes. The model is based on the assumption that radiation occurs concentrically around a blasthole. The paper shows how seismic measurements that have been made around blasts confirm the computer-generated patterns. The sensitivity to variations in wave velocity and timing delay changes are presented.

1 INTRODUCTION

Vibration prediction has been achieved, in most cases, by applying a scaled distance model, where the charge mass fired per delay is scaled as a function of the distance from a blast (Borg et al. 1987). This procedure is relatively easy to apply, and with site-specific characterization, reasonable predictions have been made. With precise electronic delay detonators becoming more commonly used, it is possible to use the precision to alter vibration amplitudes and frequencies through destructive interference of waves from different holes in a blast. Achieving effective destructive interference between holes requires sophisticated computer models to decide on the best combination of delays for a particular blast geometry.

This paper presents a method that has been incorporated into the BlastMap (Rorke 2004) blast design code for predicting vibration concentrations using wave interference patterns generated by blast designs using the code. The technique presented here shows how vibration concentrations caused by a firing sequence can concentrate in particular directions.

2 GAINS AND RISKS OF PRECISE FIRING TIMES

Precise detonators are defined as units that fire at accuracies of 1 ms or less. This means that wave interactions for holes with precise firing times occur in a predictable fashion, provided certain simplifications are considered.

Destructive interference can be achieved by choosing timing delays so that waves meet each other out of phase, and amplitudes are thus reduced. There is risk, however, that constructive interference can occur, if the wrong delays are chosen. Figure 1 shows simple sinusoidal curves that are used to illustrate the effect of delay leading to destructive or constructive interference.

The illustration in Figure 1 is greatly simplified, because it considers only two waves, both emanating from the same point in space. In blasting, the situation is significantly more difficult because the vibration waves generated by blastholes are complex and blastholes are separated in space in varying rock structure, thus resulting in different interference behaviour at different places around a blast.

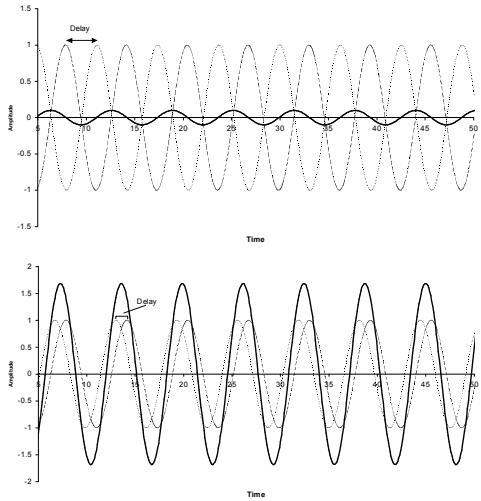

Figure 2. A simple illustration showing destructive (top) and constructive (bottom) interference. The dotted and dashed sinusoidal curves represent waves at different phase angles (the equivalent of delays in a blast). The wave resulting from the combination of the two is shown as a solid curve. By changing the delay, it is possible either to reduce the resultant amplitude of the combined waves or increase it.

If a single blasthole is considered, on detonation, a shock wave radiates from it that turns into an elastic compression wave front that travels at a velocity controlled by the rock properties. For example, a brittle hard quartzite will transmit a compression wave at about 6000 m/s whereas weathered shale may transmit a compression wave at about 3500 m/s. This wave front travels out through the rock from the detonating blasthole as a sphere. Where it intersects a two-dimensional surface, such as the earth's surface, it will appear as a circle, as illustrated in Figure 2.

For different rock properties, distances between holes, the number of holes and the timing design of the blasts, the wave fronts emitted by each hole will combine together to form a different pattern of circles that is referred to as an interference pattern. An example is given in Figure 3.

Electronic detonators provide precise initiation, which means that the interference patterns generated from a blast are predictable and repeatable. This is not the case with pyrotechnic initiation systems, where firing times vary over a range that is large enough to scatter the wave fronts and create a random and unpredictable interference pattern. Therefore, the advantage of precise firing times is that delays can be chosen so that:

- destructive interference occurs between waves;
- wave front concentrations can be directed away from structures sensitive to vibration.

The risk with precise detonators is that concentrations of energy do occur along certain directions, and if these are not taken into consideration, blast vibration may actually be amplified.

An example is shown in Figure 4, where the distance between holes and the precise delay times that were applied show a concentration in vibration in certain directions, and a dispersion of wave fronts in other directions. For limiting vibration, a dispersion of wave fronts is desirable.

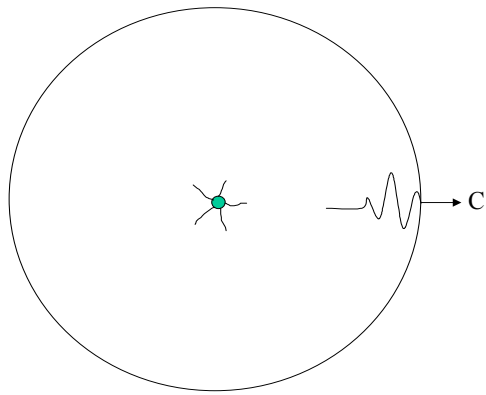

Figure 3. View from the top of a blasthole illustrating how vibration energy is radiated from a hole detonation in a rapidly expanding sphere. This sphere expands at a velocity C that is a function of the rock properties. The sphere is represented on the surface in two dimensions as an expanding circle, similar to ripples in a pond caused by a pebble dropped into it.

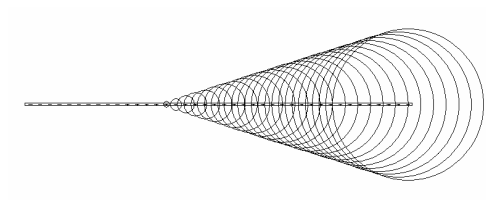

Figure 4. A set of wave fronts generated from a line of holes detonating at 1 ms intervals from right to left.

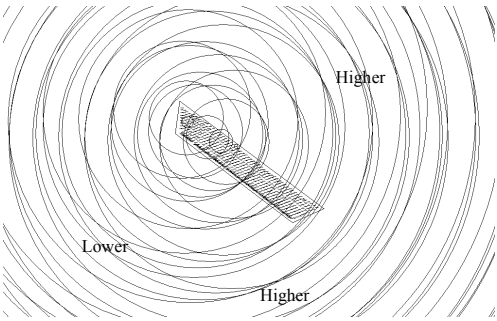

Figure 5. A wave-front interference pattern for a large electronically timed blast. Note the concentrations of wave fronts in certain directions compared to others. Where wave fronts cross each other, constructive interference will be maximized, and higher vibration amplitudes are likely.

3 CALIBRATION OF THE MODEL

Models need to be tested for accuracy. Two blasts were specifically monitored to check the effects of vibration amplitude relative to position. The results are shown in Figures 5 and 6.

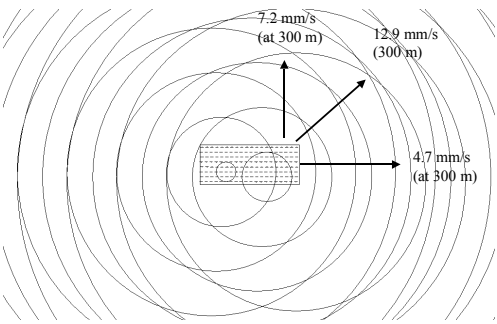

Figure 6. Electronic blast in an overburdened coal-mine blast showing peak vector sum measurements made at the positions indicated.

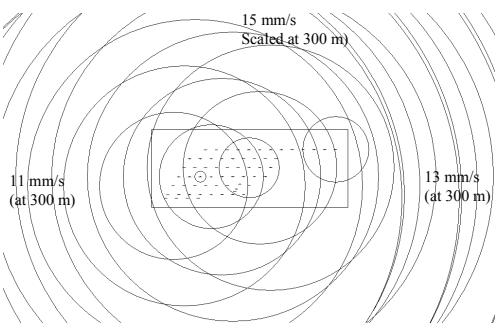

Figure 7. Electronic blast in a large copper mine showing peak vector sum measurements made at the positions indicated.

The data indicates higher vibration amplitudes in areas where there is a concentration of wave fronts, therefore, for a simple visual comparison, the interference patterns can indicate where higher vibration amplitudes can be expected. However, quantification is necessary so that a prediction of amplitude can be achieved with confidence.

At this stage, developments are in place to combine signature traces according to the wave interference patterns so that the frequency and duration of the waves can be taken into account. This will involve an additional step of measuring a number of signature traces, which is not trivial and sometimes not practically possible.

Consider each wave front to be the equivalent of vibration generated by a single hole detonation. Where these wave fronts are close to each other, higher amplitudes should be expected according to the frequency of the wave. In other words, if three wave fronts superimpose exactly, an increase in amplitude should be expected according to an equivalent of three holes firing at the same time in the same place. The measured results show an increase in amplitude where wave fronts superimpose, which correlates approximately with values predicted by a scaled distance model for multiple hole detonation. An example is given in Table 1.

Table 1. Measured peak particle velocities compared to values predicted by a scaled distance calculation based on the number of superimposed wave fronts from the model. The example used is the case in Figure 5.

Holes firing according to model	1	2	3
Measured (mm/s)	4.7	7.2	12.9
Scaled distance prediction (mm/s)	5.8	10.3	14.4

The scaled distance prediction in Table 1 is based on constants of 1143 and 1.65 and a charge mass of 150 kg of explosive per hole.

Frequency is important, because constructive interference will continue to occur, until a point, at different distances apart. This is because waves have a significant wavelength according to Equation 1.

$$\lambda = \frac{v}{f} \qquad (1)$$

where v = the wave velocity and f = frequency. Therefore, for a 15 Hz wave with a wave velocity of 3600 m/s, the wavelength will be 240 m.

Hence, even though wave fronts may appear to be further apart on the wave interference patterns, there may still be constructive interference occurring. The most obvious thing the wave interference patterns show is the zones where waves are perfectly in phase and strong constructive interference takes place. The least constructive interference (lowest vibration amplitudes) will occur when wave fronts are a half wavelength apart. The complexity of blast-generated waves makes this a simplification, but the technique provides a guide that is approximately correct.

4 SENSITIVITY OF WAVE INTERFERENCE PATTERNS

The wave interference patterns are a function of the spatial distances between holes, blast timing and the speed of the wave front through the rock mass. It is not always easy to measure the wave velocity, so a sensitivity study was carried out to determine whether blast timing or wave velocity has the greatest influence on the wave interference pattern. The influence of the distances between blastholes was not examined as these are normally a function of the hole diameter and the energy requirement for fragmenting the rock.

Simulations were carried out on a number of blasts, and the trend outlined in Figure 7 shows the results most commonly found. These are:

- A small increase or decrease in wave front velocity has the least impact on the wave interference pattern.
- A change in delay period produces a notable change in the pattern.

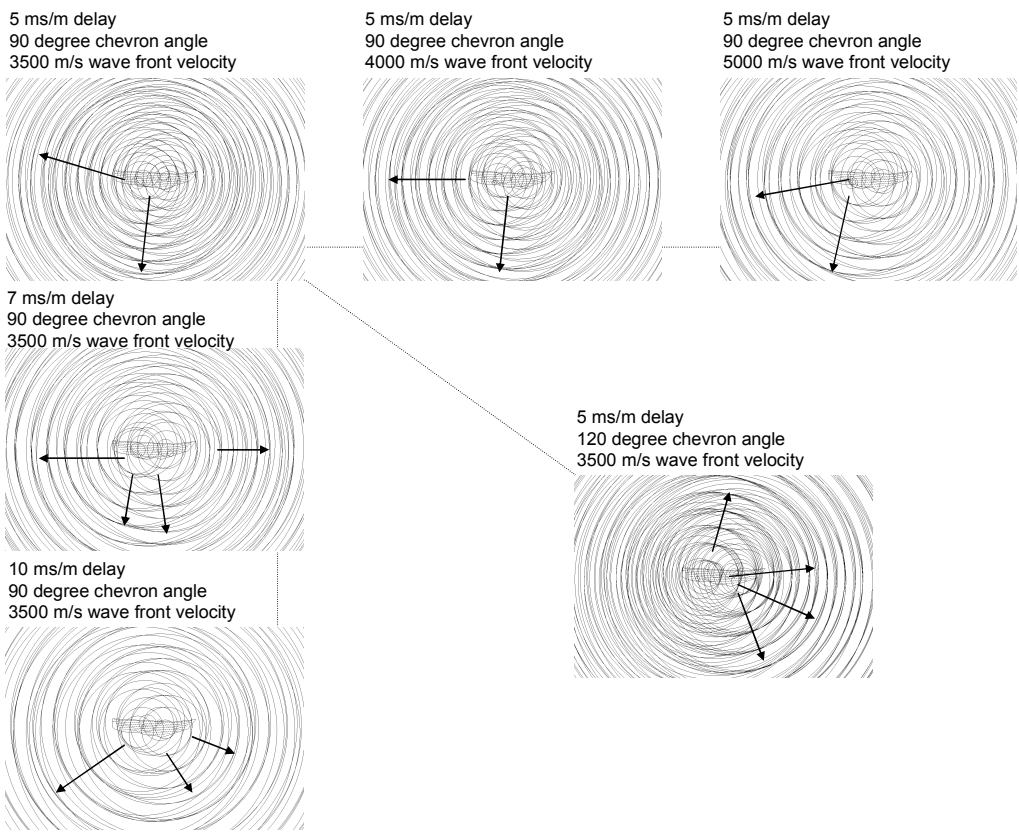

Figure 8. Composite diagram showing wave interference patterns for the same blast. The row across the top has a constant timing design (5 ms/m of burden), but a changing wave velocity. The vertical column on the left comprises the same wave velocity (3500 m/s), but an increasing delay period. The pattern on its own is for a wave velocity and delay period that is the same as for the top left-hand pattern, but with a 30° wider chevron angle.

– A change in the blasting angle (chevron angle) has a significant influence on the interference pattern.

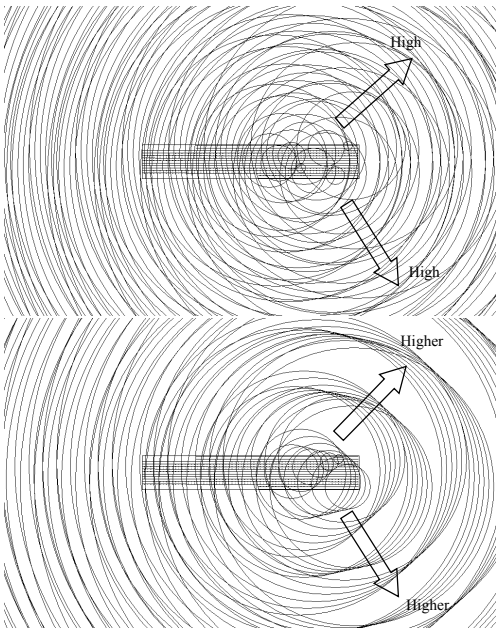

Figure 9. The top interference pattern was generated from a timing design where 13 ms delays were used between holes (east–west) and 40 ms delays were used between rows (north–south). The bottom interference pattern shows much more intense concentrations of waves. It is generated from the same blast, but delays were changed by a mere 2 ms to 11 ms between holes in the east–west direction. In the north-south direction, delays between rows remained at 40 ms. Surface wave velocity was set at 3600 m/s in the simulation.

This means that the model will produce approximately correct results with a guess at the velocity of the wave front.

In some cases, only a small change in the delay period has a significant effect on the intensity of wave concentrations. This is particularly the case when shorter delay periods are used. An example of this is shown in Figure 8.

5 VALIDITY WITH DISTANCE FROM SOURCE

The interference patterns become less defined with increasing distance from the source. This is mainly because the blast starts to approximate a point and distances between holes become insignificant compared to the wave path distances. Although the model does not show it, dispersion within the rock medium will become more dominant with increasing distances, thus making the prediction less accurate. The model is therefore only realistic for distances within a two to three blast dimension. These are the critical distances where electronic delays are generally applied for vibration control.

With larger distances from a blast, the traditional scaled distance model is still an acceptable choice for predicting and controlling vibration. Under these circumstances, the best choice of the delay period to be used in a blast design is outlined by Bernard (2004), who presented a unique chart that shows how certain delays can be safely chosen dependent on the dominant frequency of the wave, to disassociate one hole's signal with another and thus avoid constructive interference.

6 CONCLUSION

This paper has presented a method for viewing wave interference patterns to determine the areas of higher vibration concentration. The simulation model is generated by a computer design code for use with precise electronic delay detonators.

Although the concept is more useful if an accurate knowledge of the wave velocity is known, it was found that the model is less sensitive to small changes in wave velocity and most sensitive to the direction of the blast. In other words, the model is most sensitive to the chevron angle. The model is also sensitive to changes in delay period, with certain delay periods leading to concentrated bands of constructive interference in a particular direction.

Field measurements have been carried for checking the model and these appear to support the predictions.

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