Ground Control and Safety Implications of Blast Damage in Underground Mines

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ABSTRACT

The National Institute for Occupational Safety and Health (NIOSH) is currently performing research to help mine operators minimize the amount of loose or damaged rock surrounding a blasted opening. Improperly designed blast rounds can result in excessive wall rock damage that may reduce the rock mass competency and increase ground support requirements. Unnecessary blast damage can cause many safety and operational concerns including an increased potential for injury from rock falls and other hazards resulting from sub-optimal blast designs. This paper focuses on the ground control and safety implications of blast damage in underground mines. A reduction in rock quality due to blasting can increase the amount of ground support required. Case study data collected by NIOSH in underground metal mines during standard drill and blast production rounds included blast vibration monitoring, detailed three-dimensional laser scanning of the as-built excavation, and geotechnical mapping of development headings.

INTRODUCTION

Blasting practices that are fine-tuned to specific mine conditions can help to avoid a number of ground control problems. Ground falls and injuries sustained during scaling and installation of ground support are a pervasive problem in underground mines. Hazardous conditions are created in underground mines when blasting practices damage the remaining roof, ribs, and walls. Excessively damaged rock at the break limit can fall out unpredictably and is dangerous to personnel and equipment. The National Institute for Occupational Safety and Health (NIOSH) is currently performing blasting research to help mine operators minimize the amount of loose or damaged rock surrounding a blasted opening. The goal of the research is to investigate whether better perimeter control in underground excavations will decrease the number of accidents associated with ground fall injuries as well as decrease the exposure to hazards associated with scaling and installation of ground support.

Blasting is the foundation of underground metal/nonmetal excavation. Drilling and blasting practices or designs that are not optimized to the ground conditions can lead to a multitude of safety and operational problems. The purpose of this paper is to discuss the ramifications of drilling and blasting on ground control safety. Examples are described from recent NIOSH field investigations where current blasting methods were examined. Because of the interrelated complexity of designing, drilling, blasting, scaling, and ground support, we have included descriptions of typical blasting methods, rock damage definitions, and ground support considerations related to blasting that are pertinent to our field investigations.

BLAST ROUND DESIGN

The performance of commercial explosives and the dynamic response of a rock mass are complex but predictable. For a single blast hole, the detonation normally begins at the bottom of the hole where the charge is initiated using a detonator. The detonation progresses along the blast hole towards the collar. The velocity of detonation (VOD) is a measure of how fast this progression takes place. Factors affecting the VOD are confinement, diameter of the charge, and the properties of the explosive. The VOD is a measure of the explosives performance and is used to determine the blast hole pressure and in turn the shock pressure applied to the surrounding rock mass. The shock pressure propagates radially as a pressure wave, at around 5000 m/sec for brittle rock, away from the blast hole. Borehole pressure continues until the pressure from the explosive gas breaks the rock free and expands to atmospheric pressure.

Other factors affecting the performance of an explosive are the dynamic response of the rock based on rock mass strength and the amount of burden. Higher rock strength correlates with less damage. Increased burden distance generally correlates with increased blast vibration energy transmitted into the rock mass, poor fragmentation and increased damage to the periphery.

Blast rounds evaluated in our field investigations were burn cut design. The burn cut is a method to extract the first sections of the blast round where a free face is not available. Forty or more blast holes may be drilled for a single blast round. The first blast hole is detonated at or near the center of the blast round. The free face is supplied by drilling four or five empty holes very close to the first blast hole. These spacings are very short. The burn cut may require several holes detonated sequentially by using detonator delay to extract the center portion of the blast round. The ejected burn cut provides a free face for the production holes that represent the majority of the blast pattern. The production holes are typically equally spaced in a grid pattern and separations are greater. The designed limit of the excavation is determined by the perimeter blast holes. Burdens of the perimeter holes are typically the same as the production holes. A blast round, for example, may involve 15 detonation delays over a 6-second period. Burn cut holes are commonly detonated individually or in pairs, production holes are detonated in pairs or triads, and perimeter holes may be detonated in groups of five or more.

The most common explosive used is ammonium nitrate and fuel oil (ANFO) premixed in a prill form that is blown into the blast holes using compressed air. Emulsion explosive in chub form is used primarily in wet conditions. These explosives are considered to be fully coupled, that is in direct contact with the walls of the drill hole. Special decoupled charges, with void space around the explosive, are sometimes used for perimeter holes to reduce the amount of damage to the excavation. Decoupled charges produce lower shock wave pressures; the high-pressure detonation gas expands to fill the blast hole void and the gas pressure reduces significantly.

Blast Damage

Damage will always occur around a blasted excavation. Excessive damage may affect the strength of the remaining rock mass. The break limit is the distance to where the blast round successfully liberated the rock. At the break limit the remaining in situ rock mass will be damaged to a degree commensurate with the amount of explosive charge used in the blast round. Overbreak is the rock that is broken by blasting outside the intended area or line of break. Of specific interest to the authors are 1) the blast overbreak distance beyond the intended perimeter of the excavation and 2) the quality of the rock beyond the break limit. A factor in overbreak is the amount of drill hole deviation from the designed drill pattern. Blast overbreak and drill deviation are important to consider for ground control because both will influence final span width. The blast overbreak is affected most by the perimeter hole detonations but the inner production and burn holes also contribute. Figure 1 illustrates the effect of cumulative damage from production holes and perimeter holes.

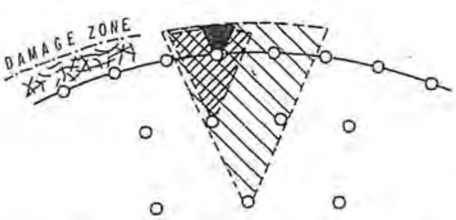

Figure 1. Illustration of how blast damage can be cumulative beyond the perimeter of a blast round due to subsequent blast delays of production and perimeter holes.

Damage can be extensive depending on the amount of energy in the explosive. The energy is directly related to the diameter of the blast hole and the detonation pressure. Kutter and Fairhurst (1) describe the modes of damage from the blast pressure. Crushing occurs in the immediate vicinity of the blast hole from the shock pressure wave. Additional fracturing and radial cracks develop within a zone described as having plastic deformation as the blast pressure wave expands radially. Radial cracks develop into the elastic zone of the rock mass. In a quasi-static mode, expansion of the blast hole cavity from the explosive gas pressure results in growth of the plastic zone and extension of the radial cracks.

The use of coupled charges in the perimeter blast holes result in larger damage zones. Figure 2 after Olsson and Ouchterlony (2) illustrates the expected blast break limit and the extent of measurable damage for coupled charges. The curves are based on the Holmberg-Persson equation integrated over the length of the charge, based on a Dynamex equivalent charge concentrations (kg/m), with assigned peak particle velocity (PPV) limits for damage. The measurable damage limit is 0.725 m/sec and the blast break limit is 8.00 m/sec. Dynamex density is 1400 kg/m^3 and the weight strength with respect to ANFO is 1.13 (3). These damage zone limits reproduce the general trends of data from a large number of blasts in different rock types.

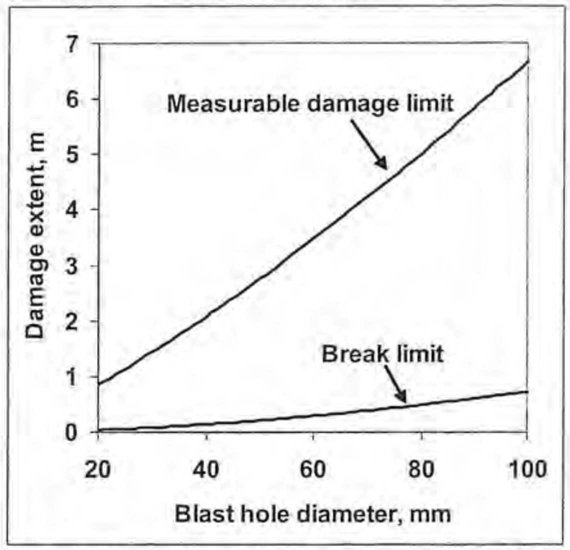

Figure 2. Approximate damage zone extent for fully coupled blast holes calculated from Holmberg-Persson equation using critical PPV values for each damage type (2).

The damage from decoupled charges will be less than the values shown in Figure 2. The degree of decoupling will determine the degree of damage incurred. It is possible to eliminate overbreak using decoupled charges. Zero overbreak is identified by the existence of half barrels from the perimeter holes at the break limit after the blast. Generally, the degree of decoupling will control the measurable damage limit. Olsson and Ouchterlony (2) summarize current methods for determining the extent of damage where damage measured from many field tests was compared to the Holmberg-Persson equation measured at two locations along a blast hole (Figure 3). The amount of damage calculated compared well with their field test data (2).

Ground Support Considerations

While there are many opinions about the blast damage mechanisms and varying definitions as to what constitutes damage, it is generally agreed that careful perimeter blasting of mining excavations is beneficial even though the effect of cracking on the ultimate stability of the underground opening is difficult to quantify. The improvement in rock quality due to careful blasting will positively affect other outcomes including decreases in scaling, rock handling, and ground control requirements. Increased drilling and the added expense of including a decoupled explosive charge are negative outcomes with measurable costs.

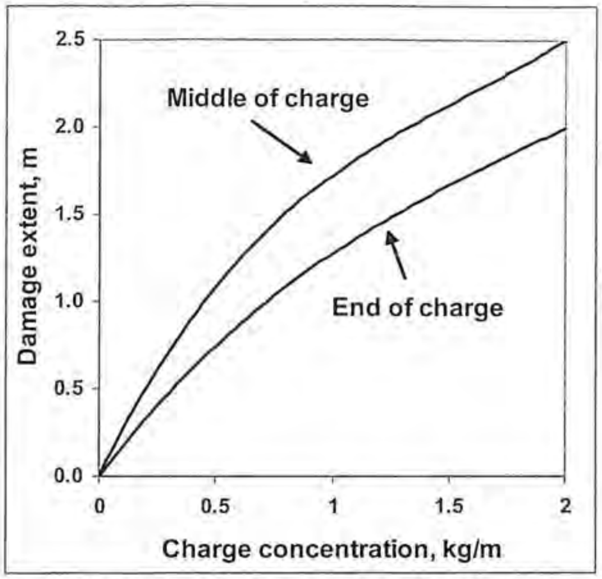

Figure 3. Relationship between decoupled blast hole charge concentration and damage extent using Holmberg-Persson PPV equation (2). Curves are based on a Dynamex equivalent charge concentration.

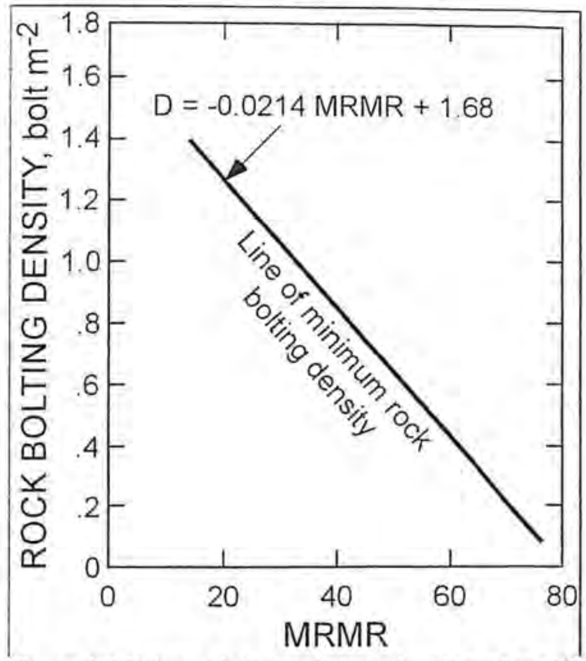

Figure 4: Minimum bolt density as a function of modified rock mass rating (MRMR) after Choquet and Charette (7).

Ground control requirements and determining the stability of the openings are directly related to the strength of the rock mass. Two empirical methods were used to quantify the integrity of the rock mass. The rock mass rating system (RMR) uses five parameters to classify rock quality: unconfined compressive strength of the rock, drill core quality (RQD), spacing of discontinuities, condition of discontinuities, and groundwater conditions; an adjustment can be made for the favorability of discontinuity orientations (4). RMR ratings range from less than 20 for very poor rock conditions to 100 for the best possible rock. The rock mass quality Q system uses six parameters to represent rock block size, interblock shear strength, and active stress; the main difference from the RMR system is the introduction of stress parameters. The Q system ratings are logarithmic-scaled values that range from 0.001 for exceptionally poor ground to 1000 for exceptionally good, practically unjointed ground (5).

Our field investigations included measurement of rock quality using RMR. RMR classification was determined for the two ribs and the back at intervals approximating the length of each blast round along the working, generally 3 to 5 m. Q ratings were also determined where the stress parameter was likely to play a role in rock stability.

According to Whyatt et al (6), poor use of conventional blasting would require an RMR adjustment to 80 pct of pre-blasting rock quality whereas use of smooth-wall blasting would require minimal RMR adjustment. Whyatt et al (6) further describe rock bolting density requirements as a function of RMR where bolting requirements decrease with increase in RMR. Figure 4 provides a method to calculate rock bolting density based on a modified rock mass rating. An alternative method of determining ground control needs based on Q values and span is shown in figure 5.

FIELD DAMAGE ASSESSMENT METHODS

Blast vibration measurement

Blast vibration is directly related to rock damage. Peak particle velocity (PPV) near the detonation is high and depends on the magnitude of the shock pressure wave. The shock pressure wave decreases exponentially with distance from the charge and the limit of damage can be correlated to the attenuated PPV. PPV data from a blast round can be identified to a specific blast hole or group of holes having a burden, explosive type, charge weight, charge length, coupling ratio, distance, and orientation with respect to the measurement location. The PPV data was plotted vs. distance and a regression was calculated using the Holmberg-Persson equation (Equation 1, Persson et al, 1994). This equation was used because it takes into account the charge length and the observation radial and longitudinal distances.

$$PPV = K\left[q \int_0^H \frac{dz}{\left[(r_0)^2 + (z_0 - z)^2\right]^{\beta/2\alpha}} \right]^{\alpha} \quad (1)$$

where

PPV = Peak particle velocity, m/sec

q = Charge concentration, kg/m

K, α, and β are considered site constants

r_0 = lateral offset distance of the measurement point from the blast hole, m

z_0 = longitudinal distance from the measurement point to the bottom of the charge, m

H = charge length, m

z = variable distance along charge length, m

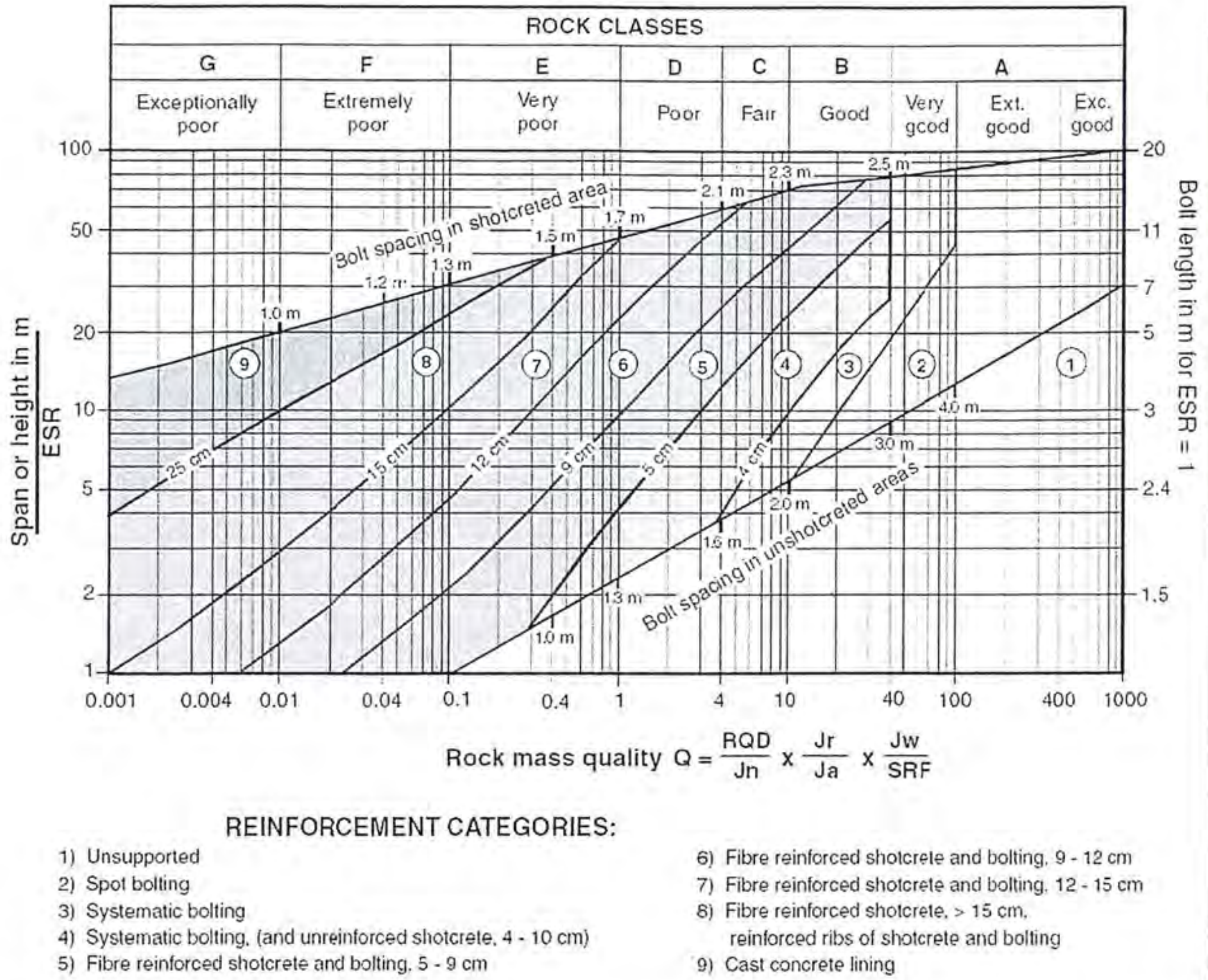

Figure 5. Ground control support categories based on span versus rock mass quality Q after Grimstad and Barton (8), reproduced from Palmstrom and Broch (9). Factors for calculating Q: RQD=Rock Quality Designation, Jn=Number of joint sets, Jr=Joint roughness, Ja=Joint alteration, Jw=Joint water, SRF=Stress reduction factor (5).

Figure 6 is an example of selected blast vibration PPV values from a development heading at a field site. Geophones were attached to the rib of the development entries at mid height (Figure 7). Geophones were placed at various distances from the blast face, and additional geophones were added following successive blast rounds. The PPV values represent the vector sum of the three orthogonal channels for each geophone. Overall maximum PPV values were determined for each distance.

The equation constants K, α, and β (Equation 1) were calculated to be 1.18, 0.42, and 0.84, respectively for this specific mine site. An α:β ratio of 2:1 was used. The regression best fit was determined by the least sum of the squares method. PPV values plotted for various distances are shown as a curve in Figure 8. For comparison, the equation constants K, α, and β as described by Olsson and Ouchterlony (2), related to the damage formulations described earlier, are 0.7, 0.7, and 1.5, respectively and represent expected PPV for given distances typical of hard rock masses (3).

Based on the regression, PPV values were calculated for a range of distances radial to the midpoint and end of the blast hole (Figure 9). At a radial distance of 4 m, the PPV decreases to 0.7 m/sec for both curves, which is the PPV limit of the damage zone (2). Table 1 lists the estimated PPV values at the break limit midpoint from the charge based on the regression analysis.

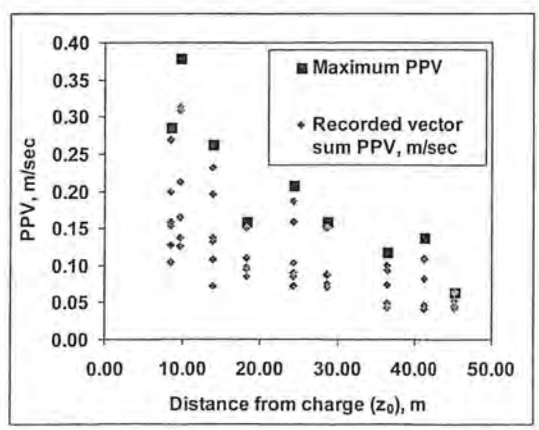

Figure 6. Peak particle velocity versus scaled distance from ANFO perimeter detonations in three subsequent blast rounds at a field site. Maximum values are determined by calculating the mean values for each distance and adding two standard deviations.

Figure 7. Triaxial geophone to measure blast vibration mounted to left rib of development heading at field site.

Table 1. Peak particle velocity values determined from regression analysis for break limit distances measured beyond perimeter holes where ANFO was detonated.

Break limit distance, mm	PPV, m/sec
540	2.7
425	3.1
391	3.3
351	3.5
394	3.2
544	2.7
461	3.0

Laser scanning

Three-dimensional survey scans for each blast round were collected using a Leica Geosystems HDS 3000 laser system. The scanner was operated using a laptop computer; independent batteries powered the scanner and computer.

Scans were acquired before and after each round for a series of rounds. Pre-blast scans required a higher level of detail in that they involved collecting collar locations and estimating blasthole orientations whereas post-blast scans need only gather sufficient information to represent the size and shape of the excavation.

The scanner is capable of collecting x, y, and z information to a precision of 1.2-mm. However, a full 360° scan at the highest density would require approximately four hours to complete. NIOSH experiments to optimize data collection demonstrated that a scanner positioned 6 m from the face using a scan density of roughly 7.5 x 5 mm provided ample detail and could be completed within 45 minutes. In the field, a 45 - minute scan could be

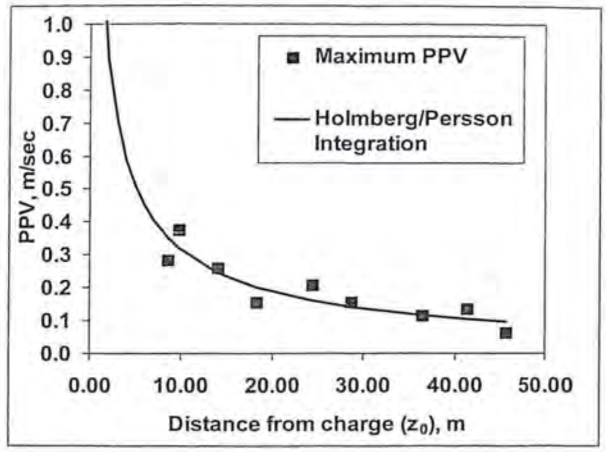

Figure 8. Peak particle velocity versus scaled distance from ANFO perimeter detonations in three subsequent blast rounds at a field site. Maximum values for each distance are compared to a regression fit of the Holmberg-Persson equation.

completed while the development crew was cycling equipment, retrieving explosives from the magazines, or conducting other tasks.

An array of readily distinguished laser targets, both NIOSH and Leica Geosystems designs, were placed within the scan scene. Using geo-referencing, these targets allowed the processing software to merge adjacent scans. Target locations were also surveyed by the mine surveyors so the scan data could be referenced to the local mine coordinate system.

Once the face was drilled, but before the drill holes were loaded, a detailed face scan was performed to locate the hole collars and to estimate the drill hole orientations. To accomplish this, scan aids

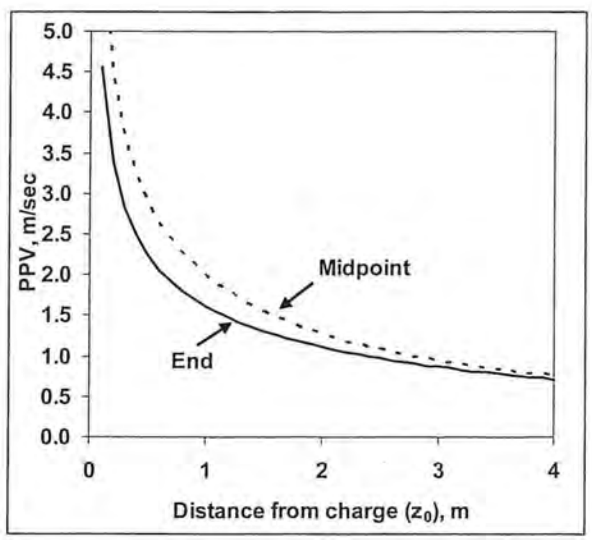

Figure 9. Peak particle velocity versus scaled distance from ANFO perimeter detonations in three subsequent blast rounds at a field site. Values are calculated from the Holmberg-Persson integration based on calculated site constants for distances radial to the end and midpoint of the blasthole.

that consisted of two-foot long pieces of plastic PVC pipe were inserted in each drill hole, and the collar of each drill hole was spray-painted. The paint enhanced the reflected signal of the laser at the collar location of each drill hole and made the point cloud display easier to interpret. The pipes provided an approximation of the hole orientation that could later be displayed in three dimensions using the processing software.

Post-blast scans were made after the heading had been mucked out but prior to installation of ground support. This allowed the freshly blasted face to be scanned from a distance while ensuring personnel remained under supported ground. Scaling is an ongoing process in most metal mines and is directly influenced by blast damage imparted to rock of the ribs and back. Initial mechanized scaling is done during mucking, which is followed by additional manual or machine-assisted scaling during support installation. The post-blast scans for this study captured the results of initial scaling, which was not specifically differentiated from blast damage.

Overbreak example in low-strength rock. Figures 10 and 11 serve as an example of overbreak in low strength rock. This crosscut in ore was between cemented backfill on the left rib and rock on the right rib. Exposed bedrock was highly broken by bedding-plane and crosscutting fractures with both shallow and steep inclinations. The ribs and back had an RMR rating of 35 (Poor) and Q of 0.55 (Very Poor). Blast holes were drilled at 0.9 to 1.2 m spacing for a nominal 4.3-m-square entry. All holes were drilled to a depth of 3.7 m. The blast holes were 4.8 cm in diameter and the relief holes were 7.6 cm in diameter. Holes in the back and left rib were line-drilled on 0.3 m centers and were not loaded. These holes were intended to limit the extent of the break. All of the blast holes were loaded with ANFO except the bottom row

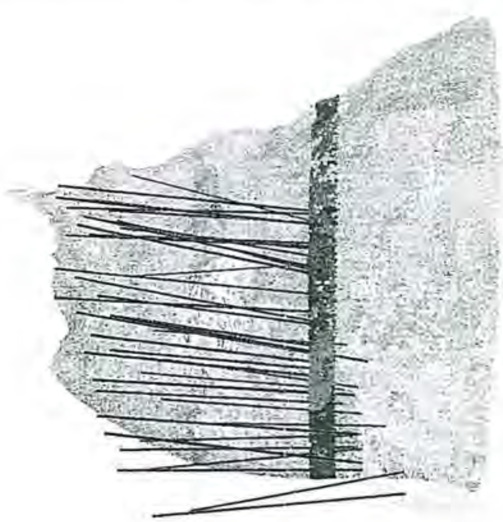

Figure 10. Vertical, longitudinal section through a blast round in weak rock. The profile was generated from two laser scans "before" and "after" the blast. Black rods represent drill holes.

(lifter holes) which was loaded with Unigel sticks.

The pre-blast laser scan showed that the entry at the face was at least 1.8 m too wide where fill in the left rib had come down during previous blast/muck cycles. The designed face would be about 18 sq m, but the pre-blast face was about 28 sq m. The post-blast configuration showed that rock broke well above the top row of unloaded holes. Overbreak ranged from 0.5 to 1.4 m, for an overbreak area of 4.3 sq m above the top row of holes in the first 0.3 m from the face. This was reduced to 2.5 sq m at the midpoint of the round (1.8 m from the face). This represents an overbreak above the drill pattern of 24% near the face and 14% at the midpoint of the round.

Overbreak example in good-quality rock. Figure 12 shows a development lateral in good quality rock. The entry was being widened to form a muck bay on the left side. Rock strength was in the 100-120 MPa range, RQD 75-90%, joint spacing 0.6 m to more than 2 m, discontinuities were rough to locally gouge-filled, and conditions were dry. RMR ratings in the vicinity averaged 69. Blast holes were drilled on about 0.7 m spacings to depths of 3.7 m. The round was loaded with ANFO in all the holes except those along the back, which were loaded with a decoupled trim-type explosive. The laser-generated cross-section makes it apparent that the use of trim powder in the back holes created very little breakage past the designed height of the opening even in a span as wide as this. The ANFO-loaded holes in the ribs show breakage past rib holes averaging 0.4 m. Breakage beyond the perimeter holes on both sides represent a total area of about 3.2 sq m, or approximately 11% of the area of the resulting face. This example demonstrates that close break limit control can be achieved by selective adjustment of the type of powder used.

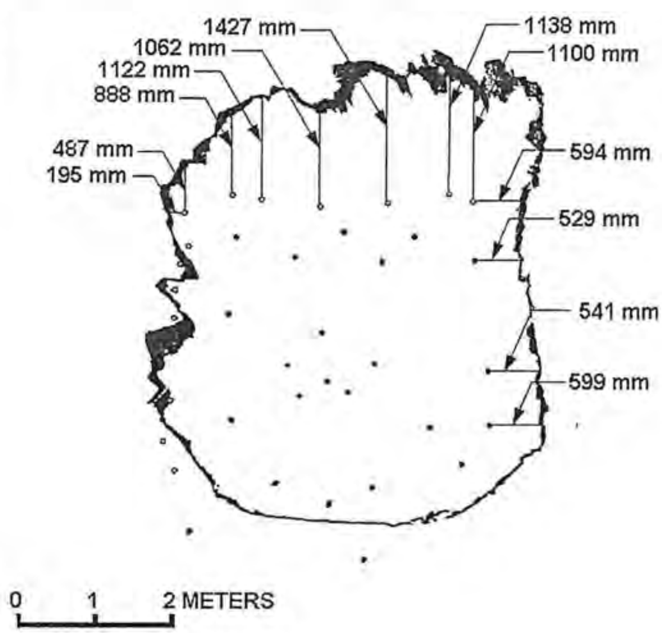

Figure 11. Cross-section of a drift in weak rock from scanning laser data. Solid points represent drill holes loaded with explosives, and open points represent unloaded trim holes. Leaders show distances from perimeter holes to the final surface.

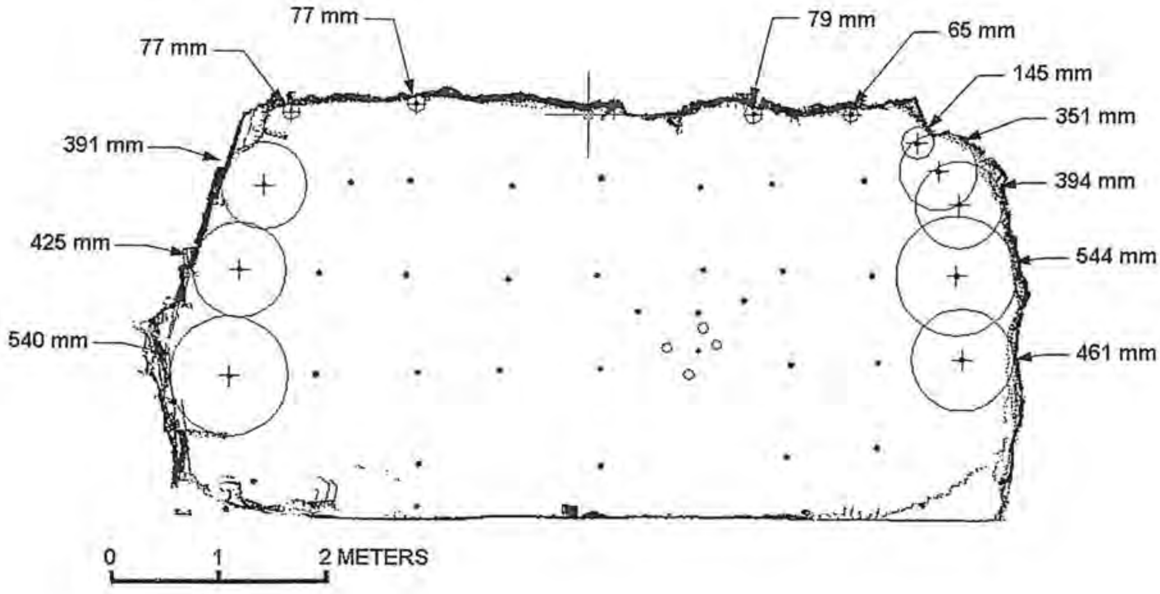

Figure 12. Cross-section of a footwall lateral generated from scanning laser data. Solid points represent drill holes loaded with explosives, and the four open points represent unloaded relief holes. The larger circles show radius distances from perimeter drill holes to the final surface.

DISCUSSION

The measurable damage limit from blasting may extend significant distances past the planned excavation perimeter. Both measurable damage and overbreak can be reduced by using careful perimeter blasting techniques. Mines that use careful perimeter blasting, regardless of rock quality (Q or RMR ratings), would see improved stability in their workings and fewer rock falls from blast damaged rock. In turn, the improved rock quality will reduce span of openings, reduce the ground control needs, and improve the safety of underground excavations.

Images showing blast hole orientations from the eight complete pre-blast scans collected so far show that, in general, blast hole drilling was highly accurate. Deviations occurred mainly where jumbo booms were required to be turned at high angles from the length of the entry or where holes were drilled tightly against the back. A quantitative assessment of the importance of drill hole deviation to overbreak and wallrock damage awaits additional scans and analysis.

The Holmberg-Persson PPV integration used in this report is well known. Unfortunately, an error was found recently in the derivation of the integral (10). The integral works however because the constants are site determined and damage limits based on the integration are accurately correlated.

Future research

Blast-induced damage is difficult to measure. Two methods to measure damage are proposed for future field investigations (1). Drilling of inspection boreholes normal to the ribs and back would allow use of a borehole camera to document visually the number, location, and orientation of cracks surrounding the entry (2). A micro-velocity probe used in the same inspection holes could measure changes in seismic velocity with depth of penetration and correlate with degree of damage. The seismic velocity is expected to be low in damaged rock near the collar and increase to normal background velocity at depth.

Current methods provide measurement of seismic blast vibrations, but extrapolation of PPV data to the damage zone using the Holmberg-Persson integration and regression has shortcomings. A better approach would be to measure dynamic strain very close to the perimeter hole detonations. This method is preferred because, for engineers, strain is more easily related to rock failure and the measurement is taken within the damage zone, while PPV values are extrapolated from a measurement at some distance. Dynamic strain measurement methods are currently being tested at the NIOSH Spokane Research Laboratory.

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