A Case Study Examination of Two Blast Rounds at a Nevada Gold Mine

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ABSTRACT

NIOSH researchers collected basic data from two blast rounds at the SSX-Steer Mine as a component of a larger study on controlled blasting, the goal of which is to reduce injuries from falling rocks in underground mines. Drilling and blasting procedures at the mine were observed. Rock mass property data were collected, a program of seismic monitoring and analysis was conducted, and 3-D laser survey scans of the workings before and after each blast were performed. The geologic data showed that the rock mass quality was characterized as poor to very poor. It was essentially uniform within the panels examined. The recorded seismic data indicated that fill material acted as an effective damper to seismic energy. The pre- and post-blast laser scans showed both over-break and under-break conditions, highlighting the problems associated with blasting a weak rock mass. By comparing the achieved results to the intended design, potential improvements in the drilling and blasting practices can be identified. Comparison of these data with similar data from other mines will help define how various combinations of rock and fill behave during blasting. Through studies such as these, data collection procedures and analysis techniques can be refined. An ultimate goal is to develop blast procedures tied to rock mass characteristics that can minimize damage to rock in the perimeter of the opening, limiting the amount of loose rock and improving the safety of miners.

INTRODUCTION

Falls of ground are a significant hazard for underground miners and a leading cause of fatalities. A majority of metal and non-metal mines use drill and blast techniques for drifting and primary development. After blasting, the broken rock is mucked out. This is followed by scaling or "barring down" to remove any loose rock from roof, walls and face. The stability of the workplace is then secured using rock bolts, mesh, shotcrete or other appropriate support systems. The entire process is then repeated. All of these activities are potentially hazardous to the miners that perform them. This hazard level is increased if care is not exercised in protecting the opening perimeter through the use of careful blasting practices. In particular, unwanted blast damage to the perimeter rock increases the hazard to those involved in the scaling and reinforcement tasks. It has been found that a majority of ground fall injuries involve blocks formed within the half-meter of rock immediately surrounding an excavation (Bauer et al., 1999; Mark and Iannachione, 2001). Whyatt et al (2003) summarized methods of assessing blast-induced damage in underground mines and discussed the feasibility of improving ground control results through the use of controlled blasting. Iverson et al (2007) outlined an approach for assessing perimeter blast damage based on rock mass quality, seismic response, and laser scanning.

To assist industry in adapting good perimeter control blasting practices, the Spokane Research Laboratory (SRL), of the National Institute for Occupational Safety and Health (NIOSH), has implemented a program focused on developing practical, yet technically sound, perimeter blasting design procedures. The objective is to provide techniques so that the as-built openings closely resemble those designed. When applied, these will assist mine operators to create stable underground openings with a minimum amount of loose or damaged surrounding rock, thereby enhancing the safety of miners. The overall research and development program involves theoretical, laboratory, field and modeling studies. The results are being integrated into a user-friendly blast design software package.

Field studies are an important component of the program. The purpose of the field studies conducted by NIOSH is to fully document the results of a number of blast rounds carried out under a wide range of rock conditions. The documentation includes: the blasting patterns, the actual versus the as-designed drillhole locations and orientations, the final excavation profile, the seismic records from the blast, the rock mass quality, the ground support, and a number of other parameters. The results of this study will provide valuable baseline information on fragmentation practices and add to data compiled on mining conditions in different geologic settings. This paper presents some preliminary results of studies carried out at the SSX-Steer Mine.

STUDY SITE AND METHODS

Background

The SSX-Steer Mine is located about 60 km (40 mi) north of Elko, Nevada in the Jerritt Canyon mining district. Queenstake Resources Ltd., the sole owner of the mine during November 2006 when the NIOSH study was conducted, merged with YGC Resources, Ltd. in May 2007 to form the Yukon-Nevada Gold Corp. (Yukon-Nevada Gold, 2007), the current owner and operator.

Disseminated gold deposits were discovered at Jerritt Canyon in 1972. Open pit mining operations involving several small open pits began in 1981 (Queenstake, 2006). Underground mining began in 1992 as the surface operations began to reach their economic limits (Ash, 2006). The SSX deposit (figure 1) was discovered in the early 1990s by following structural trends from adjacent properties. Accessed from a portal, it lies at depths ranging from 140 to 300 m (450 to 1000 ft) below surface. Mining commenced in 1997.

In 2004, the Steer mine portal was collared from the old Steer pit. A drift connecting the Steer mine to the SSX deposit was completed in the latter half of 2005. Since that time, the two mines have been operated as a single unit referred to as the SSX-Steer Complex. Gold mineralization at the Steer deposit has been identified in an area stretching eastward from the old Steer pit to halfway along the connection drift to the SSX deposit, a length of approximately 910 m (3000 ft). The connection drift facilitated mining in the Steer deposit by providing ventilation, secondary access, and shared infrastructure. The present study was conducted in the Steer portion of the complex known as Zone 7. Specifically, information was collected from one blast round each in crosscuts XC11 (November 7, 2006) and XC7 (November 9, 2006) on the 7170 level (figure 2). The XC11 and XC7 drifts were driven

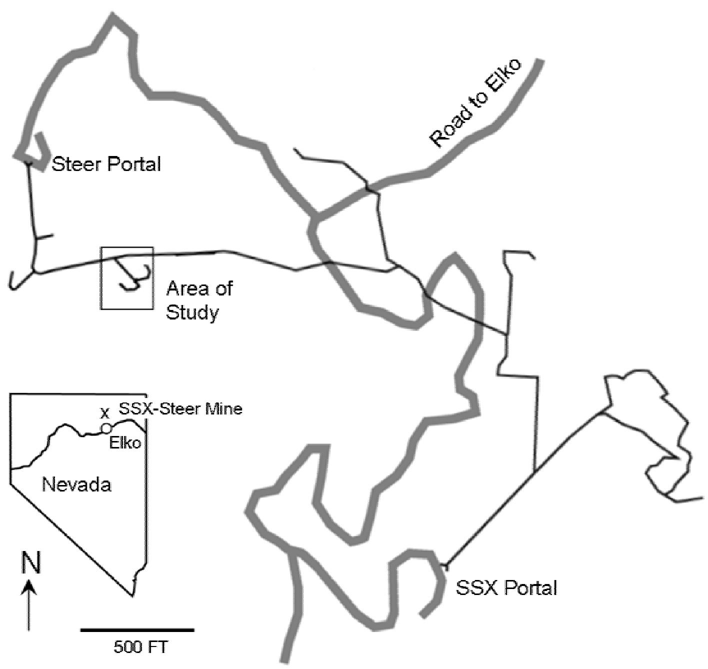

Figure 1. The SSX-Steer Complex showing generalized underground workings.

to remove ore remaining between primary extraction drifts. After mining, these primary drifts were filled using cemented fill. Access for this study was by way of the Steer portal.

Geology and Rock Mass Classification

The host rock for the gold mineralization is the Hanson Creek Formation (hc3, Silurian and Ordovician), which consists of black, fine-grained, carbonaceous limestone (Muntean and Henry, 2006). The limestone has thin, wavy laminations and commonly contains pods and lenses of black chert. Brecciation and stringers of white calcite, realgar, and orpiment are typical.

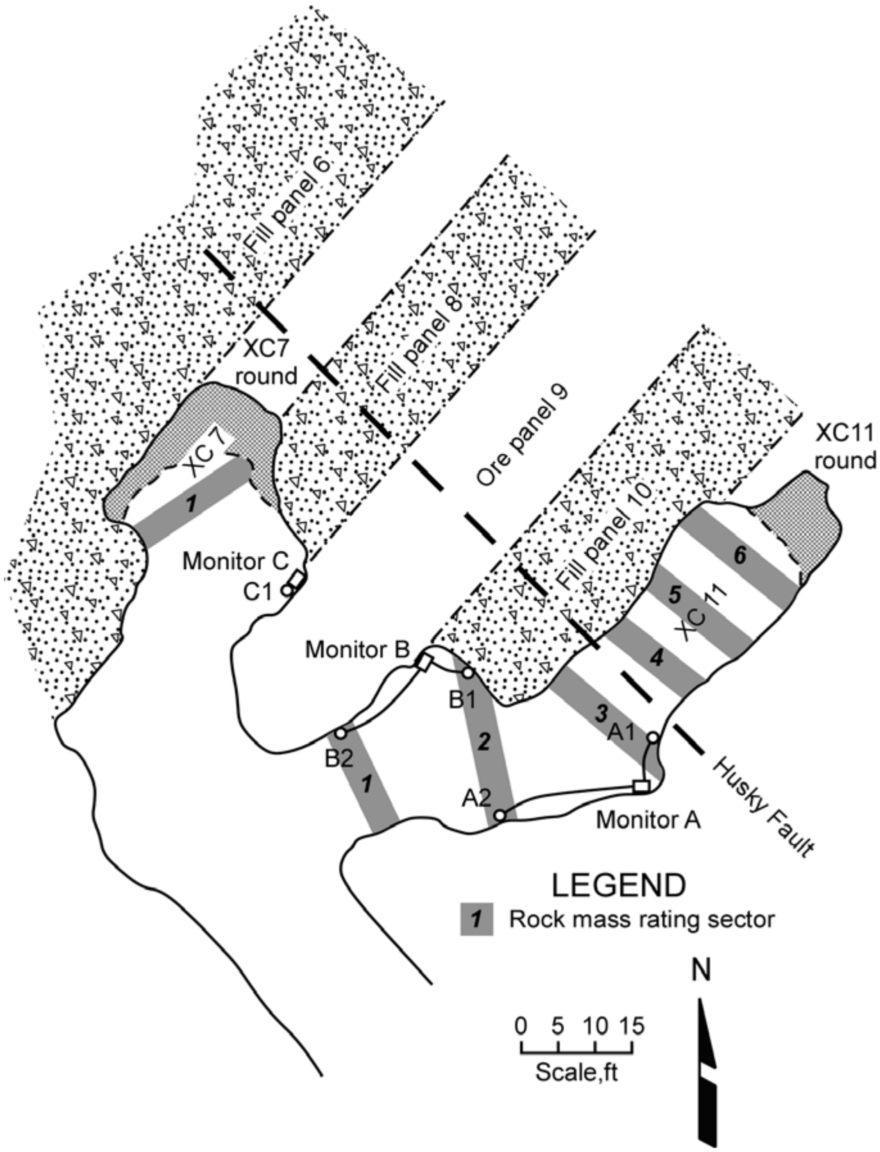

Figure 2. Plan map of Zone 7-7170 level, SSX-Steer Mine showing the locations of XC11 and XC7, the seismic monitoring stations and the rock classification sections.

The strength of the rock mass is a critical factor in determining the stability of the openings. Two empirical methods, the Q system and the rock mass rating (RMR) system, were used to classify the integrity of the rock mass. Both systems use geologic, geometric and engineering parameters for classification. The RMR system involves five parameters. These include intact rock strength, drill core quality (RQD), spacing of discontinuities, condition of discontinuities, and ground water conditions. There is an adjustment for discontinuity orientation. The RMR value based upon these factors is expressed as a score that ranges from less than 25 for the worst rock conditions to 100 for the best possible rock (Bieniawski, 1976). The Q system involves six parameters to represent rock block size, interblock shear strength, and the active stress. The Q system ratings are logarithmic-scaled values that range from 0.001 for exceptionally poor ground to 1000 for exceptionally good, practically un-jointed ground (Barton et al, 1974). For this study, the rock mass was classified using both methods and was applied to the two ribs and the back at intervals of approximately 3.7 m (12 ft) along the drift.

Seismic Monitoring

The seismic signals generated by each blast were recorded using two Instantel MiniMate Plus seismic monitors. Each monitor was connected to one or two triaxial geophones with channels to collect vertical, longitudinal, and transverse vibrations. Thus, each monitor collected and stored up to six channels of information. The monitors, which were set to a sampling rate of 8192/s, would begin recording when triggered by a motion exceeding 10 mm/s (0.39 in/s). The geophones were mounted on aluminum brackets and anchored to the ribs of the entries (figure 3) with steel masonry bolts. Each geophone was then connected to a monitor with shielded copper cables. The monitors themselves were hung from the steel mesh along the ribs to avoid damage from blast effects and from mining machinery during the mucking and bolting cycles. Where blast effects were difficult to avoid, a steel-plate box that could protect one monitor and one geophone (figure 4) was used.

Figure 3. Installing a triaxial transducer and bracket on the rib in XC 11.

Figure 4. Seismic monitor C installed in an armored steel box in XC7

Seismic data were analyzed using Instantel Blastware software (version 4.37). The software generates waveforms, selects peak values, and calculates statistical parameters. One key value is the peak particle velocity (PPV). Particle velocity was recorded for each of the three axes at each point in time sampled. The PPV is taken to be the largest of the three values for each vibration event. The vector sum is calculated by combining the velocities for each of the three axes for each data sample. The peak vector sum (PVS) is the maximum resultant particle velocity. The PVS may or may not coincide with any of the peak particle velocities for the individual channels.

Laser Scanning

Three-dimensional survey scans for the two blast rounds in the XC7 and XC11 ore panels were conducted using a Leica Geosystems HDS 3000 laser system. A laser beam reflected from any surface in the beam path (rock, fill, equipment, etc.) is used to calculate the distance and direction to that surface. The result is a three-dimensional point cloud representing the size and shape of the excavation. It has an accuracy of 6 mm over a range of 50 m (0.23 in

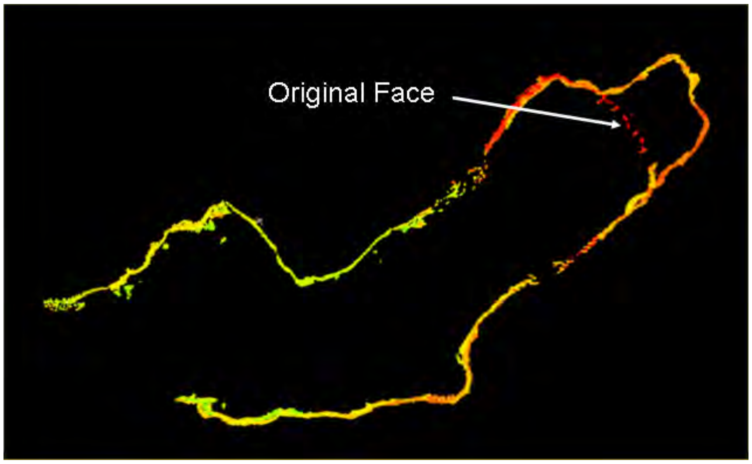

Figure 5. Plan view of XC 11, showing the original face and the area extracted by the shot on 11/7/06.

at 164 ft). The scan density was set to a sample grid of 7.5 mm vertical x 5 mm horizontal (0.3 in x 0.2 in) at a range of 6 m (20 ft). Each scan was completed within approximately 45 minutes.

Two scans were required to assess blast round results. The first scan provided the pre-blast configuration of the drilled face including the back, ribs, and blast-hole collar locations. The second scan provided a survey of the excavation after the round was shot and mucked (figure 5). Successive laser scans were merged into a single three-dimensional survey using the associated software. Point cloud accuracy was improved by placing survey targets within the scanning area. Several 15-cm (6-inch) diameter spherical plastic targets designed by NIOSH were mounted along the ribs. The targets were screwed into reusable holes that had been drilled and tapped in rock bolt plates. The targets were removed during blasting to prevent damage and remounted for subsequent scans.

Short lengths of PVC pipe were inserted into the blast holes as scan aids so that the laser scanner could accurately capture collar locations as well as hole orientations. Software would later be used to estimate the 3D location of each drill hole in space to its full depth.

XC11 BLAST ROUND

The XC11 entry was the end ore panel on the 7170 level. Panel number 10 containing cemented fill was located along the left side while the right side was formed by rock that will not be mined (see figure 2). The design for the November 7 blast is shown in figure 6. The blast holes were 2.2 cm (1-7/8 in) in diameter and the relief holes were 7.6 cm (3 in) in diameter. All holes were drilled to a depth of 3.7 m (12 ft). Perimeter holes were drilled along the back and upper left rib on nominal 0.3 m (1 ft) centers in an attempt to limit the extent of breakage. These were not charged with explosives. Except for the lifters, all blast holes were charged with ANFO. The lifter holes were charged with sticks of Unigel.

The rock mass quality was evaluated for six sectors in the XC11 ore panel. Due to the presence of bedding-planes and crosscutting fractures, the exposed bedrock appeared highly fractured. The fractures were predominately north-trending with shallow inclinations. In addition, west-trending fractures with shallow dips to the south, and steeper (75-81° NE) fractures with northwest strikes were exposed. The south wall and back had a uniform RMR rating of 35 for all six sectors evaluated in XC11. Q ratings were a consistent 0.55 in the back and in sectors 1 through 4 in the south wall. The rock in sectors 5 and 6 was even more highly fractured than the rest of the crosscut and Q values in the south wall were rated at 0.34. Fill in the north wall was not rated. Based upon the Q values and RMR ratings, the rock in the XC11 ore panel was classified as very poor.

A seismic monitor serving two geophones was installed on each wall of the entry for the November 7 round in the XC11 ore panel (see figure 2). Both geophones (A1 and A2) located along the right hand wall were attached to solid rock. On the left wall, one geophone (B1) was attached to the fill wall of panel 10 and the second geophone (B2) was attached to rock (panel 9 between fill panels 8 and 10).

The Minimate monitors were triggered by the blast at 17:27 and began sampling. The seismic waveforms from monitor A showed peaks for each delay in the blast pattern over a duration of about 7 seconds (figure 7). As expected, the peak velocities recorded at geophone A1 located 9.8 m (32 ft) from the blast face, were significantly higher than those recorded at geophone A2, which was 16.2 m (53 ft) from the face. The seismic waveforms recorded by monitor B revealed velocities that were clearly attenuated by the presence of the fill. For geophone B1, located 14.0 m (46 ft) from the blast face, the peaks for several blast delay intervals were much-reduced or essentially missing when compared to the signals from geophone B2 (19.5 m or 64 ft from the face) attached to rock.

The PPVs for each channel of geophone A1 were near or exceeded 100 mm/s (3.9 in/s) as can be seen in table 1. The peak velocities for the three channels occurred at three different blast delay intervals, suggesting a

Order	Delay no.	Delay, ms	No. of holes
1	*0*	0	2
2	*1*	500	2
3	*2*	800	2
4	*3*	1,100	2
5	*4*	1,400	2
6	*5*	1,700	2
7	*6*	2,000	2
8	*7*	2,300	2
9	*8*	2,700	2
10	*9*	3,100	4
11	*15*	5,900	5
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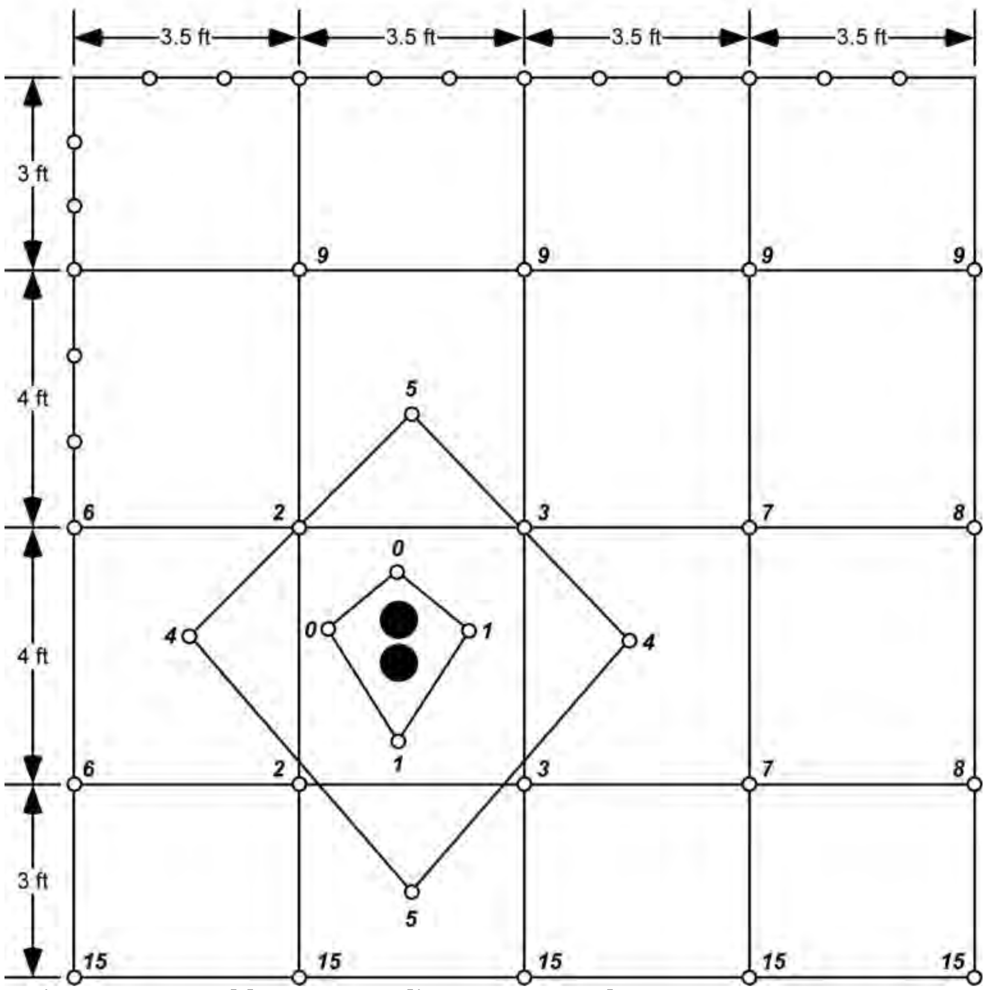

Figure 6. XC11 blast pattern diagram, November 7, 2006.

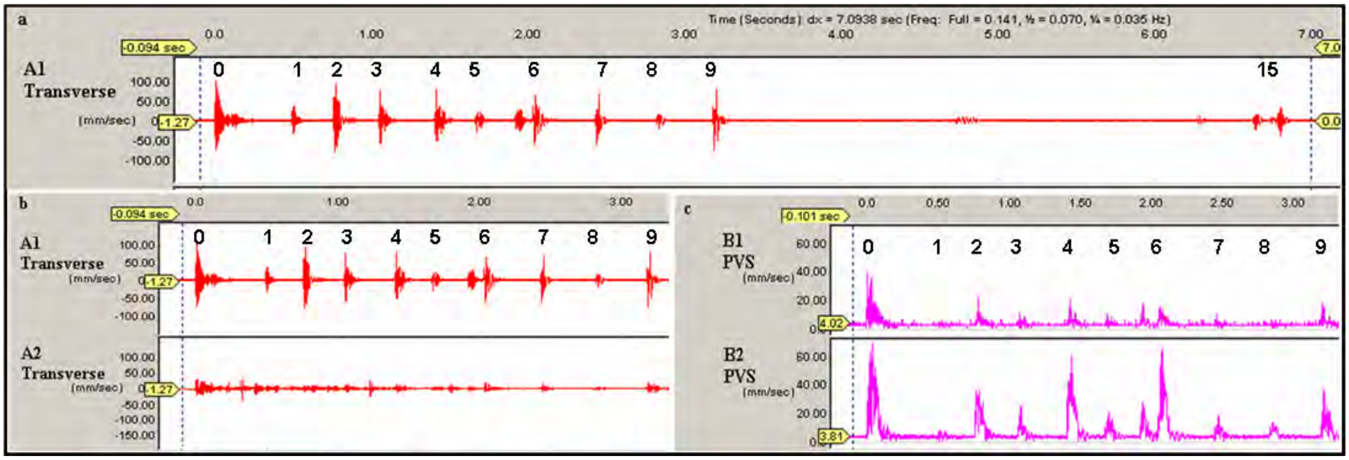

Figure 7. Seismic waveforms from XC 11. a- particle velocity vs time for the duration of the blast (detonator delays 0-15), Monitor A1, Transverse channel; b- particle velocity vs time (detonator delays 0-9) showing higher particle velocities in the A1 Transverse channel compared to A2 Transverse channel; c- particle velocity sum (three channels) vs time (detonator delays 0-9) showing relatively higher velocities in rock (B2) compared to fill (B1).

Table 1 Seismic data summary for the blast in the XC11 ore panel (November 7, 2006), showing PPV for each channel, their PVS at each geophone, and the timing delay of peak velocities.

Geophone (Distance from face, ft)	Channel	PPV/PVS (mm/s)	Time (s)
A1 (32)	Transverse	102	0.015
	Vertical	99.1	3.201
	Longitudinal	171	2.051
	Vector Sum (PVS)	190	2.052
A2 (53)	Transverse	41.9	0.332
	Vertical	39.4	0.321
	Longitudinal	88.9	0.321
	Vector Sum (PVS)	192	2.052
B1 (46)	Transverse	24.1	0.031
	Vertical	17.8	0.004
	Longitudinal	39.4	0.005
	Vector Sum (PVS)	39.6	0.005
B2 (64)	Transverse	57.1	1.440
	Vertical	35.6	0.069
	Longitudinal	61.0	0.044
	Vector Sum (PVS)	69.0	0.044

relatively uniform response to the blast pattern. Farther from the blast face at A2, the velocities were under 100 mm/s (3.9 in/s) and concentrated at one time interval. On the left wall, all but one PPV was associated with the first hole to be shot in the burn. The exception was the transverse sensor, where the PPV was associated with the fifth timed hole (delay #4). The PVS value for the first hole shot was 74% higher in the rock panel geophone than in the fill panel geophone, although the fill panel was nearer to the blast face.The pre-blast laser scan (November 7) showed that the overall panel width was 1.8 m (6 ft) or more wider than the planned drift blast. The remaining ore between the left drift wall and the fill had been removed during previous blast/muck cycles (figure 8). The designed face would be about 18.2 m^2 (196 ft^2), but the pre-blast face was about 27.9 m^2 (300 ft^2).

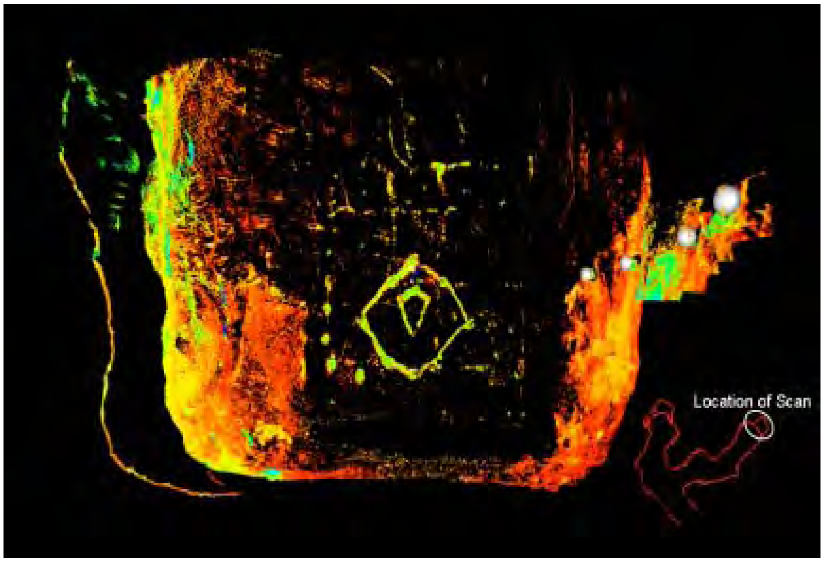

Figure 8. Laser scan collected November 7, 2006 (face view) in the XC11 ore panel showing old rock bolts hanging from the fill as well as new mesh (upper left side), the painted blast hole pattern, and the scan-aid spheres (right side).

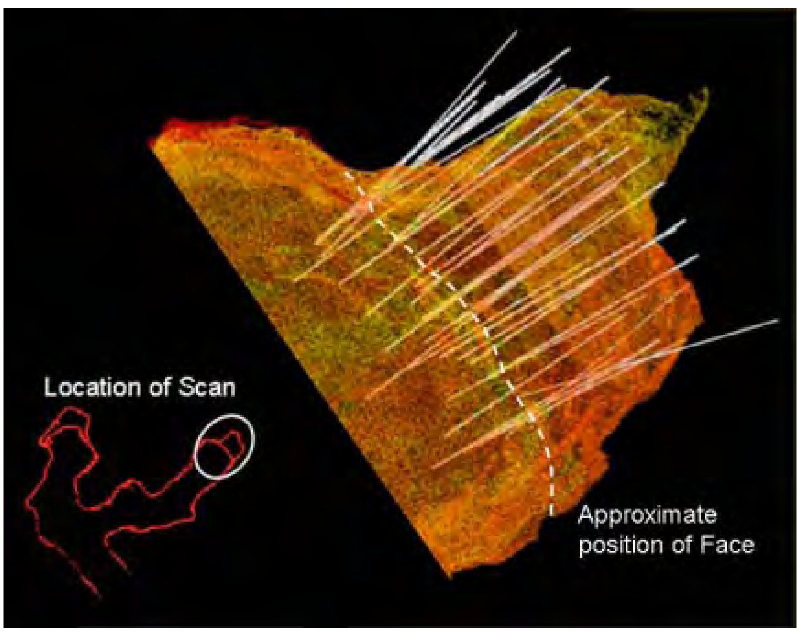

Figure 9. Plan view of the XC 11 ore panel blast round (November 7, 2006), showing pre- and post-blast face configurations and blast holes as projected from collar orientations. Uncharged holes are shown outside the resulting excavation

The plan-view projections of the blast holes are shown in figure 9 based on the combined pre-blast and post-blast scans. The drill pattern had been positioned to minimize over-break on the left rib with uncharged perimeter holes intended to limit over-break along the fill as had occurred in previous rounds. The line of uncharged holes placed along the left rib were still present after the blast. The post-blast configuration indicates that rock broke well beyond the ends of the blast holes on the center-left side.

Figure 10a is a vertical, longitudinal section taken through the post-blast scan. There is considerable breakage above the roof holes near the blasted face. The cross-section areas as determined using slices taken parallel to the face on one foot intervals are indicated in table 2. At the 1.5-m (5-ft) distance, the area is about what was desired. Figure 10c is vertical section at taken at mid-round. The projected positions of the holes are shown with distances from the charged holes to the final profile. Along the back, the distances from the uncharged holes to the final perimeter have also been determined. The line of uncharged roof holes was apparently not effective in controlling the profile. For both walls, the final position seemed to correspond quite closely to the positions of the charged holes.

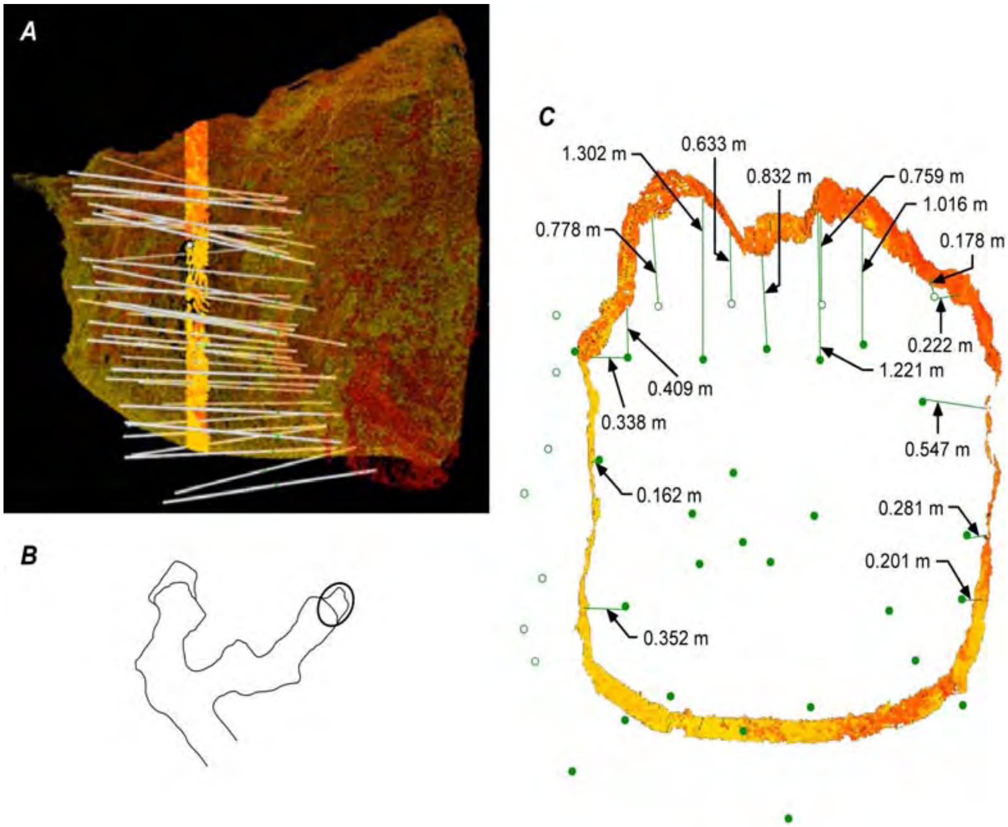

Figure 10. A. Vertical longitudinal section through XC 11 from before and after laser scans; showing projected drill pattern and position of the mid-round cross-section. B. Outline of the workings showing location of sections. C. Vertical slice at mid-round showing drill hole intercepts with distances to the ribs and back, solid points were loaded holes and open points were unloaded perimeter holes.

Table 2. Area of the round taken at 1-ft slices from the face.

Distance from Face (ft)	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Area (sq ft)	279	252	229	204	183	166	156	138	137	125	113	88	55	31

XC7 BLAST ROUND

As indicated earlier, XC7 was intended to extract the ore pillar remaining between previously mined and filled panels 6 and 8. The November 9 blast round was not a standard blast round but was intended to turn a corner between the two pillars of fill. Hence, the blast holes were angled to the right from the collars shown in the drill plan (figure 11). The hole lengths indicated for each vertical row of holes decreased from 3.7 m (12 ft) near the left rib to 1.8 m (6 ft) on the right. The initial opening is a combination fan-and-burn cut. As with the XC11 blast round, the blast and relief hole diameters were 2.2.cm (1-7/8 in) and (3 in), respectively and the same explosive types were used. A third seismic monitor (C) equipped with one geophone was attached to the rock on the right rib of the entry into the XC7 ore panel (see figure 2). One monitor (B) remained in the left rib of the XC11 ore panel with one geophone (B1) in fill (panel 10) and the second geophone (B2) in solid rock (panel 9 between fill panels 8 and 10).

The rock mass was characterized based on observations made at a single sector in the XC7 ore panel. Since cemented fill composed both ribs at the drill face, rock exposure was limited. Steeply dipping (81° NE), northwest striking fractures were present. Rock exposed in the back had an RMR of 45 and Q of 2.42. Both the Q and RMR values for the XC7 ore panel indicated that the rock mass strength was notably better than that in the XC11 ore panel and was classified as poor to fair.

Order	Delay no.	Delay, ms	No. of holes
1	***0***	0	2
2	***1***	500	2
3	***2***	800	2
4	***3***	1,100	2
5	***4***	1,400	2
6	***5***	1,700	2
7	***6***	2,000	2
8	***7***	2,300	2
9	***8***	2,700	2
10	***9***	3,100	2
11	***11***	3,900	6
12	***13***	4,900	6
13	***15***	5,900	6
			38

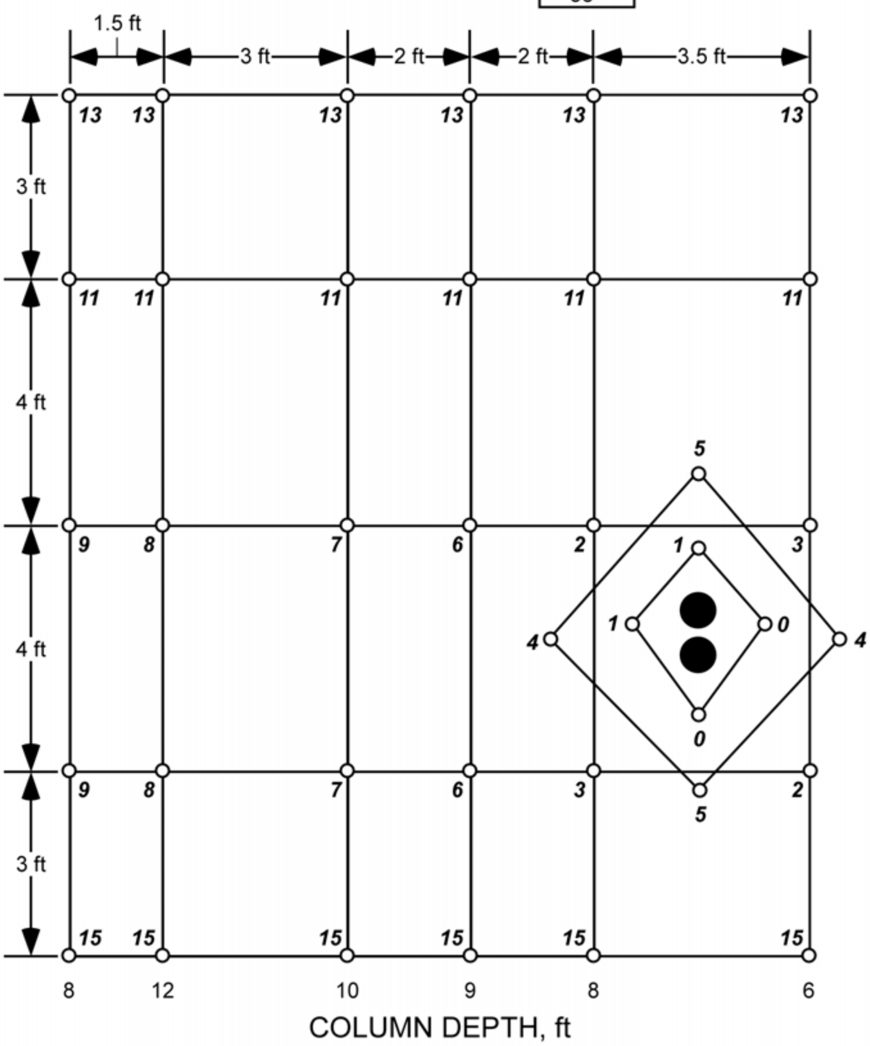

Figure 11. XC7 ore panel blast pattern, November 9, 2006

The blast triggered seismic sampling at 17:21. The seismic waveforms from monitor C showed peaks for each delay in the blast pattern over a period of about 7 seconds (figure 12). Peak particle velocities at geophone C1, 6.7 m (22 ft) from the blast face, were sharply defined and well dispersed throughout the blast sequence. Waveforms from monitor B demonstrated velocities that were again attenuated in the fill (B1) but somewhat better defined in the rock panel (B2). For the monitor nearest to the blast face (C), both PPV and PVS approached or exceeded 100 mm/s (3.9 in/s) as shown in table 3. Monitor B (located on the opposite side of the rock panel) recorded particle velocities that barely reached the 10 mm/s (0.39 in/s) trigger threshold.

The pre-blast laser scan in the XC7 ore panel included drill hole collar locations and hole orientations (figure 13). The post-blast scan survey showed that the drilling pattern and loading plan was effective in excavating the corner near the right rib fill with minimal over-break into the fill. Some over-break did occur along the left rib rock/fill contact.

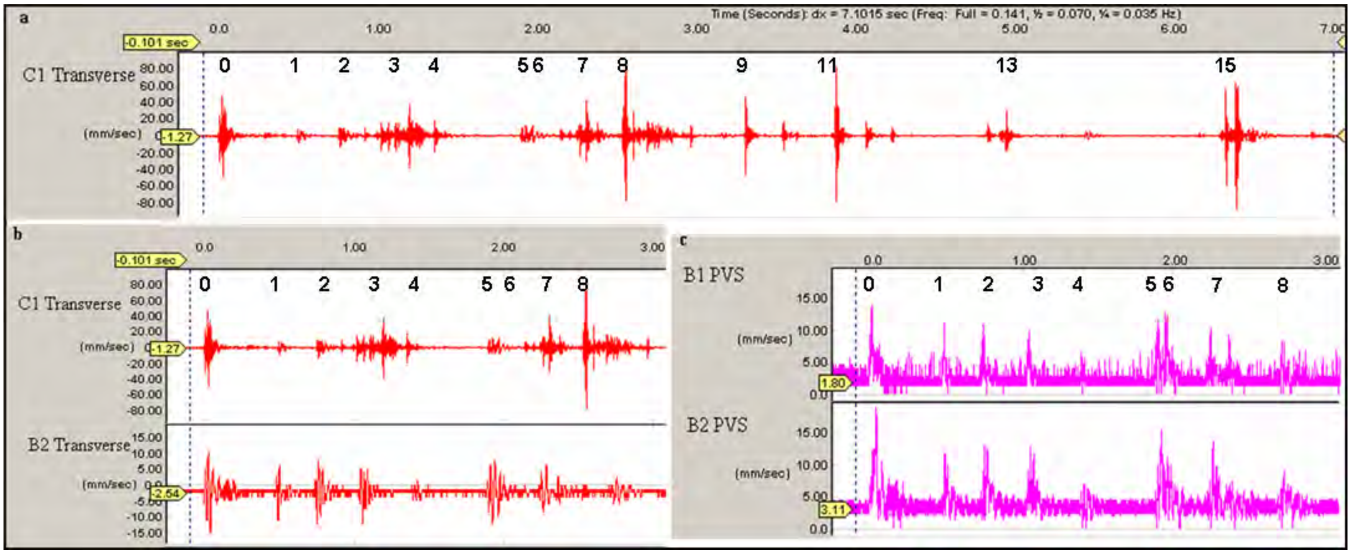

Figure 12. Seismic waveforms from XC 7. a- particle velocity vs time for the duration of the blast (detonator delays 0-15), Monitor C1, Transverse channel; b- particle velocity vs time (detonator delays 0-9) showing higher particle velocities in the C1 Transverse channel compared to B2 Transverse channel; c- particle velocity sum (three channels) vs time (detonator delays 0-9) showing relatively higher velocities in rock (B2) compared to fill (B1).

Table 3. Seismic data summary for the blast in the XC7 ore panel, November 9, 2006, showing PPV for each channel, their PVS at each geophone, and the timing delay of peak velocities.

Geophone (Distance from face, ft)	Channel	PPV/PVS (mm/s)	Time (s)
B1 (46)	Transverse	10.2	1.951
	Vertical	7.62	1.041
	Longitudinal	12.7	0.001
	Vector Sum (PVS)	14.0	0.010
B2 (41)	Transverse	15.2	0.038
	Vertical	8.89	0.033
	Longitudinal	12.7	0.033
	Vector Sum (PVS)	18.9	0.037
C1 (22)	Transverse	90.2	6.394
	Vertical	102	3.881
	Longitudinal	92.7	2.556
	Vector Sum (PVS)	154	3.881

DISCUSSION AND CONCLUSIONS

The two blast rounds monitored at the SSX-Steer mine were part of their secondary recovery program. XC11 was located on the southeast side of the mining section. The panel immediately to the northwest of this panel had been previously mined and filled. The 4.3-m (14-ft) wide drift was being driven leaving a 1.8-m (6-ft) wide rib pillar on the left hand side. The function of this rib was to limit damage to the fill and dilution during the mucking operation. In previous rounds, both ore and fill had broken out so that the final overall room width was about 6 m (20 ft).

XC7 was being driven to extract the 3.7-m (12-ft) wide pillar remaining between ore panels 6 and 8. Again, the idea was to leave an initial skin of rock near the fill walls to limit the potential for dilution. Since the round was one of the first in the overall extraction process, it was not a standard design. In both cases, the rock mass was classified as poor to very poor.

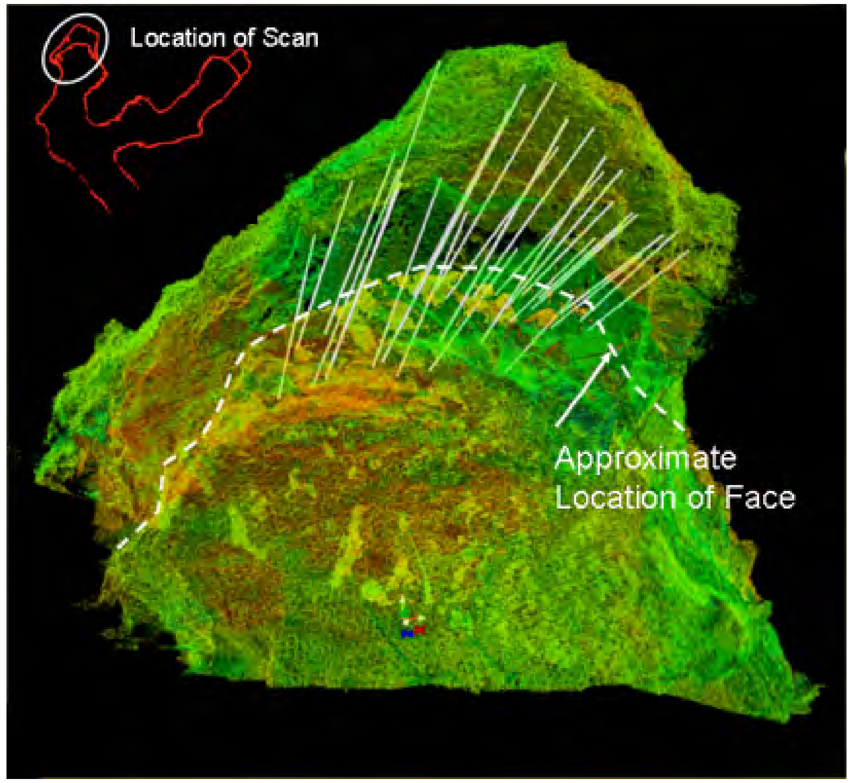

Figure 13. Plan view of XC7 ore panel blast round with projected drill holes and pre- and post-blast face configurations.

Although the rock quality was essentially uniform within the panels examined, the rock newly exposed by both blast rounds showed signs of larger block sizes and potentially greater hazards from rock falls. The fill material was relatively friable but stood up well where exposed in secondary drifts. Seismic data indicated that the fill material acted as an effective damper to seismic energy. This may have helped to limit blast damage to the remaining rock panels.

A more detailed look at the results of the standard blast round in XC11 can provide some insight regarding drill-blast performance. The post-blast longitudinal and plan sections show that the excavated opening tapered down from the drilled face, which was 53% larger than designed, to about the designed area at the midpoint of the round, and to much less than designed (about 45%) at the drilled depth of 3.7 m (12 ft).

Figure 14 is a cross-section of the round generated by slicing a 0.30-m (1-ft) section of the laser scan. The drill hole locations projected to mid-round from the hole collars at the drilled face were used to align the designed blast round with the resulting excavation. Deviations from ideal positions for these locations reflect both the actual drilled deviations and errors in projecting the drill holes from the scan aids used to collect hole orientation.

Although the total cross-sectional area at mid-round was approximately as designed, the laser section at that point shows that over-break in the back accounted for 15.4% of the designed area. Over-break ranged from about 18 to 78 cm (7 to 31 in) in the back and averaged 59 cm (23 in) for a total of 2.8 m^2 (30.2 ft^2). Less significant over-break along the right rib, based on perimeter hole locations, amounted to about 0.9 m^2 (9.3 ft^2), with a range of 20 to 28 cm (8 to 11 in), and an average of 24 cm (9 in) or 4.7% of the designed area. At the same time, under-break occurred along the left rib as shown by the retained, unloaded perimeter holes. The under-break amounted to about 2.1m^2 (22.4 ft^2), with a range of 32 to 55 cm (13 to 22 in), and an average of 46 cm (18 in) or 11.4% of design. The remaining apparent under-break, about 10%, was in mostly broken material on the floor of the mucked opening.

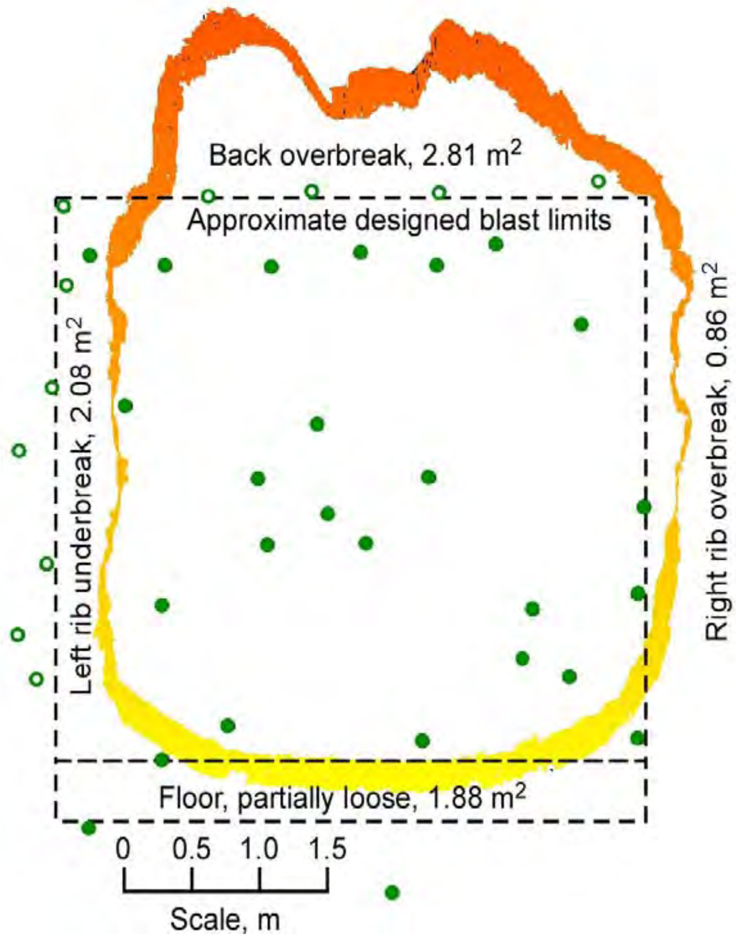

Figure 14. Cross section of the XC11 round at midpoint, showing over-break and under-break in relation to drill holes.

The unloaded perimeter holes across the back of XC11 failed to prevent over-break in the blast round examined. The closely spaced line of holes across the back appear to have been accurately drilled, but an arched configuration for these holes might have helped to retain more of the back. Gravity certainly played a role in the failure as it acted on crosscutting fractures in the very poor quality rock mass where scaling of loose rock is likely to extend beyond design limits. At the same time, the unloaded holes along the left rib were still standing after the blast, indicating that they were also ineffective in controlling the final configuration. The blast round in XC7, with a relatively complex pattern in a tight space, was successful in continuing the excavation with only small amounts of over-break.

SUMMARY

Two blast round surveys were completed at the XC11 and XC7 ore panels in the Zone 7–7170 level. Rock quality was classified as very poor and poor for the XC11 and XC7 ore panels, respectively. PPV measurements near the blasts were much lower in the fill than in the rock. The laser surveys showed both over-break and under-break conditions in the XC11 ore panel. Generally, the laser surveys illustrate the problems associated with blasting a weak rock mass. Study of additional blast rounds can be used to document changes in rock mass characteristics or to verify correlations between blasting methods and wall rock damage. Data from successive blast round surveys would provide a basis for the design and testing of new blast designs. Expanding on the use of laser data will allow more detailed comparison between blast designs and final configuration of the resulting excavations.

Seismic data and blasting outcome case studies will be used to calibrate new models for predicting blast damage to the perimeter of underground openings. Drilling and blasting methods that minimize blast damage will reduce the hazards of loose, broken rock underground and enhance the safety of miners.

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