Measurement of blast damage

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Abstract

There is no straightforward and systematic method for adequately measuring blast damage in on-going mining operations. This paper considers various techniques available for the measurement of blast damage, making particular reference to recent underground hard-rock mine studies. This paper concludes by indicating the importance of understanding and accounting for geology and emphasizes the need for a robust damage-audit procedure.

Introduction

Although blasting remains the most inexpensive method of fragmentation in hard-rock mining, the significance of the costs of blast-induced rock-mass damage in terms of mining efficiency and safety is becoming increasingly recognized. Blast-damage measurement must account for the natural in situ state of the rock mass, as well as the damage subsequently caused by blasting or mechanical excavation, and must account for the redistribution of ground stresses. Rock-mass damage has received relatively little attention compared to blast-vibration analysis in structures such as buildings. A growing recognition of the impact of rock-mass damage by the mining industry raises the issue of its effective measurement by simple, reliable and robust techniques. Visual inspection and traditional-measurement parameters are not well documented, and inconsistencies in the application and interpretation of data have been observed.

Geophysical methods offer the potential for greater resolution and penetration in three dimensions, but these methods are now limited in their practical applicability. Blast-vibration analysis to assess and control rock-mass damage has not been studied extensively, and rock-mass-damage criteria related to vibration levels and frequencies are not entirely clear.

Only through effective measurement can damage mechanisms be identified and related to the blast design and geological environment. This should facilitate the integration of damage-audit procedures into routine mine-production control to maintain blasting quality for mining efficiency and safety. It is evident from prior work that measurement techniques are not ideal or well established, and geology is seldom accounted for when characterizing and understanding the damage and its genesis. Few mines practice routine damage monitoring and quality control.

Mining-induced damage and its impact

Damage to a rock mass is considered to be the reduction in its integrity or quality. This paper distinguishes between inherent rock-mass damage, which arises from natural processes during its evolution, and mining-induced damage, which is inflicted subsequently by the mining process itself. Mining-induced damage may relate to blasting, to mechanical excavation or to the redistribution of ground stresses in response to the excavation process. Damage is any deterioration in rock-mass strength due to the presence of newly generated or extended fractures or opening and shearing along cracks and joints. A rock mass may sustain considerable damage and yet be able to perform structurally for the period required by the mine. Holmberg and Persson (1980) presented a damage model that was correlated against the difference between pre- and postblast fractures in core drilling.

Damage was considered to occur when the number of cracks after the blast was measurably greater than before the shot. Oriard (1982) defines damage to include "not only the breaking and rupturing of rock beyond the desired limits of excavation but also an unwanted loosening, dislocation and disturbance of the rock mass. Frequently, there is damage from the overbreak due to poor drilling control, the dislocation of rock due to venting of explosive gases and the loosening or the dislocation of rock due to the influence of seismic waves (ground vibrations)."

In surface mining, blast damage has a significant impact on slope stability. Bauer (1982) noted that, if backbreak is not controlled, a decrease in the overall pit-slope angle will ultimately be necessary, and undesired consequences, such as reduced ore recovery and increased waste-to-ore ratios, will result. Greater amounts of loose face rock will be produced and planned safety berms will be less effective or nonexistent.

The benefits of reduced rock-mass damage include (Calder, 1977; Mohanty and Chung, 1986; Persson et al., 1994):

- the stripping ratio can be increased;

- mechanical support, scaling and secondary blasting costs can be reduced;
- the berm interval can be increased, because the pit walls and berms are more sound;
- costly damage to buildings or tunnels can be prevented by controlling vibrations from blasting;
- rockfall hazards to miners and equipment can be reduced (pit walls are smoother and less fractured, so rockfalls are reduced);
- safety berms will be more effective in catching rockfalls, because they have not been degraded by overbreak or crest fracture from production blasting; and
- ground shock or blasting vibrations can be reduced to improve stability.

Damage in underground mining has received comparatively less attention. Before attempting to control blast damage, it is important to distinguish between damage related to blasting (vibrations and high-pressure gases) and damage caused by ground-stress redistribution. Although the impact of damage is perhaps more evident underground, the causes of the overall rock-mass damage, such as inherent damage caused by tectonic forces prior to mining or damage from stress redistribution in deep openings, are sometimes harder to distinguish. McKown (1986) states that the resulting overbreak and damage to remaining rock can lead to safety problems and additional costs due to the following:

- rock falls that result in additional mucking;
- extra backfill material to fill overbreak (expensive shotcrete, in the case of a lined tunnel or a foundation to be poured against rock);
- additional rock reinforcement (e.g., rock anchors, steel sets and wire mesh), which may be required due to damage to rock walls;
- additional pumping or grouting, which may be required (if below the groundwater table) if joints or other discontinuities are opened by explosive gases; and
- additional maintenance (e.g., scaling) of exposed rock walls.

Underground mining efficiency is strongly affected by ore loss and dilution, as well as rework, ground support and restricted access. Damage may necessitate the design of larger pillars, reduced stope sizes and ore loss. Damaged ground presents hazards to both personnel and equipment.

Damage measurement techniques

The relative merits of the various techniques described below should be evaluated in light of the fact that some of them were used for specific scientific studies, while others were developed and applied in on-going routine excavation work. The work of the authors has been aimed at being functional within a conventional mine-operating environment, in terms of acceptable reliability, simplicity and robustness.

Assessing preblast, inherent damage. Prior to any blast, it is important to characterize the existing integrity of the rock mass. This can only be achieved through defining the geology of the mine's rock masses and their component rock units. It is then possible to sensibly construct a geomechanical classification to provide a basic reference for postblast observations and audit. Standards and recommended procedures exist, such as those provided by the International Society of Rock Mechanics (ISRM) (Brown, 1981).

Fragmentation and damage-assessment studies have often suffered from inadequate supporting rock-property measurements, perhaps due to a lack of facilities, expense and the difficulty in correlating damage with any one specific rock property. It is important to distinguish static from dynamic properties. Because blasting induces dynamic loads, dynamic strengths can be many times higher than static ones, and there appears to be no quantitative correlation between the dynamic strength and static strength or any other easily measured elastic property (Mohanty, 1987).

It has been demonstrated that even microcracks caused by blasting can measurably reduce the rock strength (Persson et al., 1994). It should also be noted that, even if there is no change in the measured strength of the intact rock, it in no way means that the rock mass has not been damaged, because this is more likely to occur in the "weakest links", i.e., in discontinuities in the rock mass.

Visual inspection and surveys. Visual inspection of a blast site can provide a great deal of information, albeit qualitative, on the performance of a blast and the associated pre- and postblast damage. It is important to inspect the site prior to the blast to assess the site conditions to identify inherent damage features (i.e., joints, cracks and major discontinuities) and any prior damage that may have existed. The inspection of nearby fixed installations, such as mine support, shotcrete or concrete structures, is particularly useful, because these installations are often more susceptible to damage at lower thresholds and because they are usually more visible than damage to the rock mass. Information regarding the blast design, the actual explosives loaded and the actual firing sequence should also be collected prior to the blast. In open-pit blasting, the number of free faces and the confinement of the last two rows of holes to be fired are factors to be noted, as well as the relative direction of the blast in relation to the orientation of the major discontinuities or preferential orientation of the joint sets.

Visible half-cast holes consequent to the use of wall-control blasting techniques can indicate the extent of blast damage. Backbreak, circumferential cracking and radial cracking show that damage will not extend uniformly and can align preferentially with rock-mass discontinuities or as a function of the blasting direction. Observations should be aimed at providing insight into the damage mechanisms operative in each geological setting.

Geomechanical line surveys are a systematic method of site inspections to assess the rock mass prior to and after blasting, highlighting particularly the discontinuities that are the "weakest links" in the rock mass. Basic data for each fault, or joint and bedding set, is comprised of orientation, spacing, persistence, roughness, wall strength, aperture, filling and water seepage. Line surveys can serve the following three purposes:

- the assessment of the rock-mass structural features, which may affect the response to blasting;

- the basic information for compiling the discontinuities into sets; and
- the required information for rock-mass classification systems.

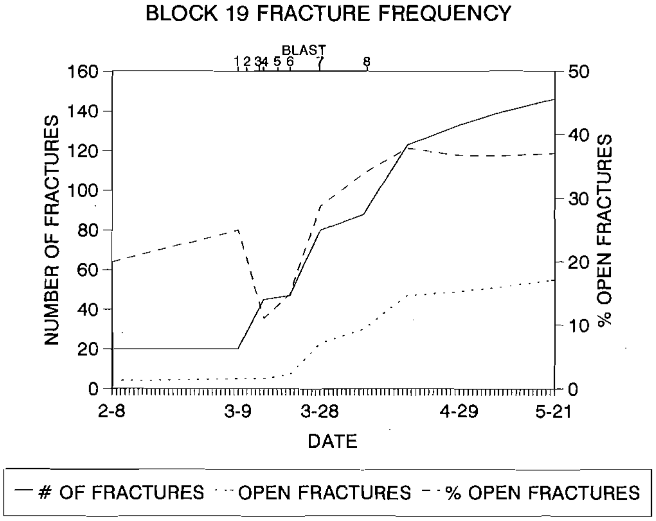
Differentiation between natural discontinuities and blast-induced fractures is important in mapping or logging but may not be straightforward.

Borehole surveying, including the assessment of core, is the most common and useful three-dimensional method of blast-damage evaluation. Borehole cameras for relatively small holes, 60 mm (2.5 in.), are now quite common, and they have proven their worth in several rock-mechanic stability evaluations. In a recent study, Inco (1994) used a borehole camera for a blast-damage-assessment study at a mine in Manitoba, Canada.

Figure 1 illustrates the compilation of a borehole survey, which indicates fractures and open fractures, as a function of both time and the blast number. The camera was inserted into boreholes in the hanging wall of a blasthole stope. The results were compared to surveys from a blasthole stope with lightly loaded holes. Thus, an attempt was made to differentiate blast fracturing from that due to stress redistribution. Scoble et al. (1987) employed a borehole dilatometer to measure the rock-deformation modulus in witness holes at varying distances from blastholes underground. Fractures were mapped by adapting the dilatometer as a fracture-impression packer. A correlation was derived between the modulus and the blast-fracture intensity, in an attempt to develop destress blast-design guidelines.

FIGURE 1

Borehole survey data from a stope blast sequence (after Inco, 1994).



Traditional observation methods. These methods do not directly measure damage, and what may be judged a "good" result at one site might be considered a "poor" result in another geology or blast geometry. The half-cast factor (HCF) is a popular parameter to assess damage. Controlled-blasting techniques, when effective, will leave part of the perimeter blast holes intact, referred to as "half-casts" or "half-barrels." HCF is computed as the total length of visible half-casts divided by the total length of perimeter holes drilled, expressed as a percentage (McKown, 1986). It is not an absolute value, but it is only an indirect measure of efficiency. Identical techniques can also produce very different HCFs from one blast to another, simply because of slight variations in rock-mass structure. Worsey (1985) reported that half-cast measurements do not always agree with the rock quality established by diamond drill-core fracture-density analysis. Other details can be recorded in addition to the condition of the half-barrels, such as the crushed zone lengths produced by the toe loads. The percentages of half-barrels can then be computed with the crushed zone lengths subtracted from the total hole lengths, as shown in Fig. 2 (Paventi, 1995).

Scaling time: Scaling time is another empirical blast-damage-assessment parameter. In a specified rock mass, scaling time should, in theory, be a function of the area and the quality of the blast. Sutherland (1989) used scaling time to evaluate perimeter blasting in development headings, establishing a correlation between scaling time

FIGURE 2

Surveying of half-casts (after Paventi, 1995).

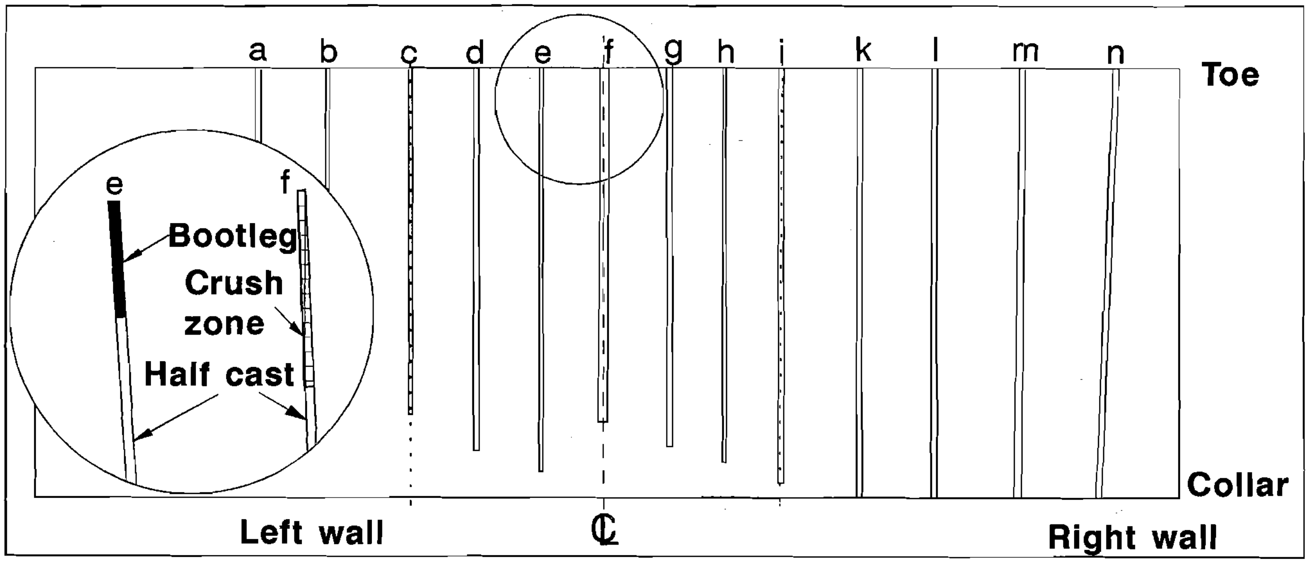


FIGURE 3

Cavity-monitoring system (after Inco, 1994).

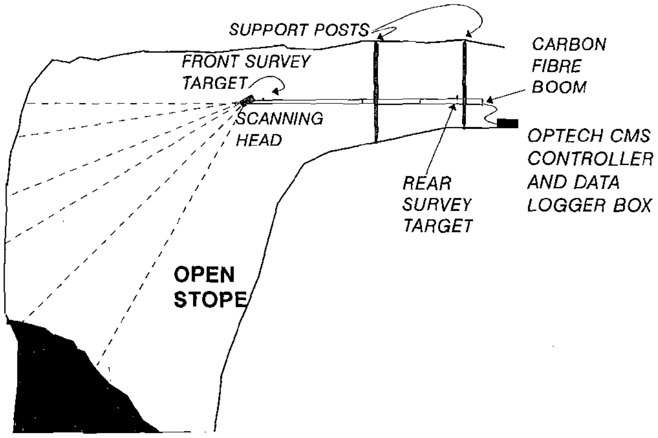


FIGURE 4

The effect of structures and their condition on the computation of DI.

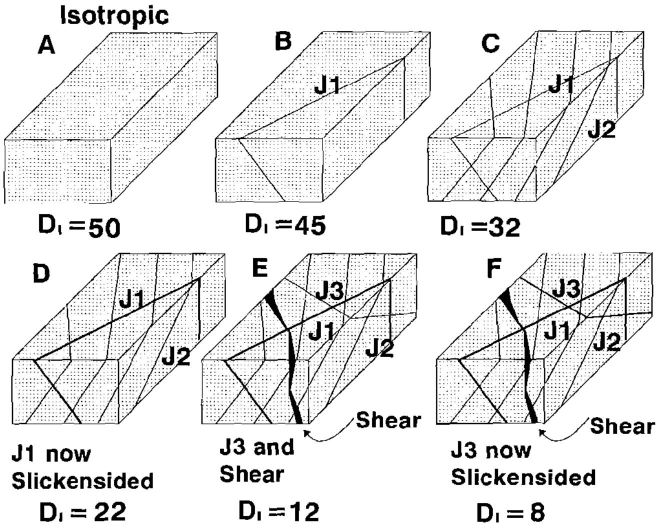
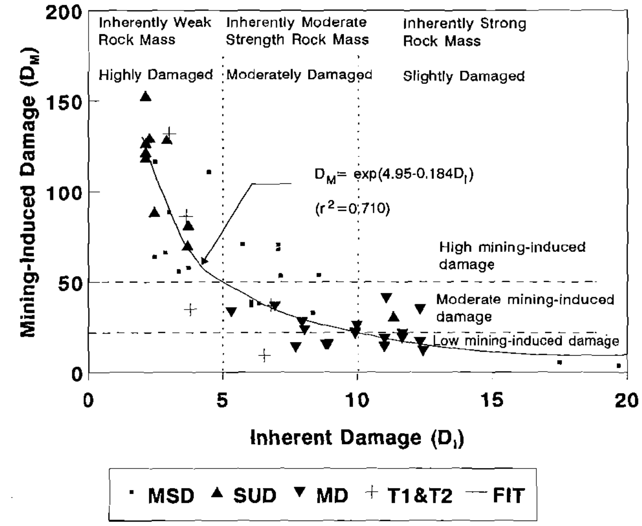


FIGURE 5

Inherent vs. blast-induced damage in different rock masses (after Paventi, 1995).



and the HCF. This can be useful for estimating blast damage on a routine basis without having to measure HCFs. To account for size effect in assessing tunnel damage, Paventi (1995) used a scaling time that was weighted by tunnel perimeter area.

Overbreak and underbreak: Forsyth (1993) defined overbreak as the breakage or significant reduction in rock quality beyond the design perimeter of the excavation. He reported on changes in blast design that reduced overbreak in tunneling from 25% to <5% with a decrease in both the time for shotcrete application and the total volume of shotcrete required. Developments in laser and ultrasonic technologies now make the measurement of face, stope cavity and development drift profiles more convenient and feasible than traditional surveying or photographic profiles. Surveys can relate planned profiles to overbreak, underbreak and sloughing. Measurements taken with a laser cavity-monitoring system during a blast-damage study were useful in monitoring the cavity shape throughout a stope life (Inco, 1994). This can be inserted into a stope on an 8-m- (26-ft-) long beam or through a 200-mm- (8-in.-) diam borehole, as illustrated in Fig. 3.

Rock-mass-classification methods. The empirical rock-mass-quality rating systems, e.g., Barton et al. (1974), Bieniawski (1976) and Laubscher (1977), are now widely used, but they were not designed specifically to address issues of blast damage. Their role is limited to inherent damage assessment, but they usually do not consider discontinuity orientation. The blast-damage rating system was devised by Colchester-Steele et al. (1991) as a specific method to evaluate blast damage. It is based on simple field measurements of scaling, HCF and overbreak, which are obtained by summing the three components to give a rating between zero and nine. Paventi (1995) extended this approach in underground drifting by adopting a measure of preblast rock-mass damage, using an inherent-damage index (D_I) based on the evaluation of rock strength, fabric, meso- and macroscale structures. Figure 4 illustrates the influence of structural variation in a sample rock mass on D_I. An index of blast-induced damage (D_M) was also used in this study to quantify the amount of postblast damage inflicted on a rock mass. It is derived from the product of the following five parameters:

- reduction in intact rock strength,
- postscaling half-cast factor,
- drift condition (sounding the back and walls with the use of a scaling bar),
- normalized scaling time and
- direction of the structure with respect to the drift direction.

These parameters were selected and combined after statistical evaluation of several available parameters, against known damage situations.

Sounding of the back and walls indicates the looseness of the rock-mass blocks defined by inherent discontinuities and mining-induced damage. Sounding with a scaling bar is somewhat subjective. The results may differ depending on the skill and experience of the individual. A drift-condition rating based on sounding was investigated by Forsyth and Moss (1991). This was adopted by Paventi (1995) as a component of D_M. The use of this type of parameter was discussed by Jowsey et al. (1991) in connection with a prototype device, mounted on a hydraulic scaler, that potentially can sense and assess the looseness of a rock mass.

Another D_M component was the drift direction with respect to the direction of the prevailing meso- and macrostructure, accounting for the most persistent and abundant structures encountered in the drifts. Figure 5 shows D_I and D_M plotted for 54 monitored drill-and-blast rounds, all broken with the same design and blast practice. It clearly shows a correlation between the two indices and their distinct variation according to the type of rock mass. The metasedimentary rock mass (MD), according to this rating system, is inherently strong, sustaining the least damage as a result of mining. This was followed by the massive-sulfide rock mass (MSD), while the most damaged by blasting was the weak serpentinized ultramafic rock mass (SUD).

The D_I-D_M relationship enables damage to be classified according to intensity. The data suggests that the relationship could enable the prediction of damage from a particular blast design in another type of rock mass. This would require further study of the manner in which variation in the blast design affects the relationship within different rock-mass types. The required data in this type of approach is also readily obtained with simple tools for field measurement.

Geophysical methods. It is anticipated that new technology will provide improved tools in the future. These methods may then become more applicable to assessing mining-induced damage, e.g., seismic tomography, ground-penetrating radar and loose-rock detection sensors. At present, such tools lack either sensitivity, require specialist support, or are limited in terms of cost, size or site preparation requirements. Spathis et al. (1987) demonstrated the use of high-frequency cross-hole seismic methods for surface and underground experimentation. The main measures selected for analysis are velocity, amplitude and rise time. Stachura and Cumerlato (1989) used seismic-refraction tomography in experimentation to select better explosives for highwall stability. Data from a series of refraction shots were collected both before and after the test blasts and analyzed tomographically.

Ouchterlony et al. (1993) used geophysical methods to measure zone depths in various smooth blasting patterns in underground development. Borehole methods were used in properly positioned and angled holes, both before and after blasting. The most sensitive method was the sonic log. The most useful method in shallow holes was the electric-resistivity log. It could detect both single fractures and porositylike effects, and it gave similar results in hammer- and core-drilled inspection holes. The damage zone in the contour consisted mainly of single fractures that produce distinct log anomalies. The damage zone in the floor had a continuous deterioration towards the floor, indicating closely spaced fractures and/or microcracking. Ground-penetrating radar was used by Adams et al. (1993) before and after preconditioning blasts to quantify the changes in fracturing in deep rock burst-prone stopes. The radar penetrated 10 m (33 ft) to identify the postblast extension of preexisting fractures and new fractures.

Vibration analysis. An explosive detonation shock wave in a blasthole eventually manifests itself as blasting vibrations. This seismic wave undergoes attenuation, multiple reflection, refraction and diffraction from boundaries and other discontinuities. Mohanty and Chung (1986) found that backbreak is essentially a "near-source" phenomenon, with the zones of crushing, compressive failure and tensile failure confined to the immediate vicinity of the exploding boreholes. The former two failure zones in hard rock seldom exceed 20 borehole diameters. In contrast, the damage potential from blasting vibrations extend much beyond these zones and, therefore, may be considered a far-field phenomena. This includes the phenomena of spalling, crack extension and block sliding. Blast-vibration analysis to assess and control rock-mass damage has not been studied extensively and specific rock-mass damage criteria related to vibration levels and frequencies are not entirely clear. Mojtabai and Beattie (1996) reported on blast-vibration monitoring in open-pit bench blasting. They related observed degrees of damage to monitored PPV and scaled distance within four rock units of varying strength.

Singh (1993) reviewed the damage criteria based on vibration levels, peak particle velocities (PPVs), reported by others. The following investigators reached these conclusions:

- Langefors and Kihlström (1973) proposed the following criteria for tunnels: PPVs of 12 in./sec (305 mm/s) result in the fall of rock in unlined tunnels, and PPVs of 24 in./sec (610 mm/s) result in the formation of new cracks.
- Calder (1977) observed that no fracturing of intact rock will occur with a PPV of 10 in./sec (254 mm/s). However, PPVs of 10 to 25 in./sec (254 to 635 mm/s) result in minor tensile slabbing, and PPVs of 25 to 100 in./sec (635 to 2,540 mm/s) would cause strong tensile and some radial cracking. The break up of a rock mass will occur at a PPV of 100 in./sec (2,540 mm/s).
- Oriard (1982) proposed that most rock masses suffer some damage at a PPV above 25 in./sec (635 mm/s).

Damage can be observed at measured PPVs in the near-field, while it is difficult to predict damage from PPVs measured further away, based on exceeding the strain level of the intact rock close to the exploding charge. Again, damage relates to the properties of the "weakest link" within the rock mass. Blast-vibration monitoring, while providing some indication of damage to the rock mass is, unfortunately, not a complete solution to the problem of measurement.

The measurement of seismic velocities can provide an indication of the relative change in rock-sample elasticity. This can be extrapolated to rock-mass assessment in the field, considering that damage would produce a change in the rock-mass modulus (E_m). If it is only possible to measure V_P in the field, then rock with a higher seismic velocity will have a higher modulus for a constant density (McGaughey et al., 1994). Seismic velocity is also affected by other factors, such as changes in the stress field.

Conclusion

It is paramount to understand and account for geology in assessing and controlling rock-mass damage. The geology should be defined by identifying the rock units encountered in the mine and grouping them into their associated domains or rock masses. Structural features at the rock unit and rock-mass scales are very important to establish. The geological classification thus established can be the basis for a geomechanical classification, summarizing the physical and mechanical properties of the environments in which blasting will occur. This should facilitate a procedure to quantify the inherent rock-mass damage, i.e., the in situ integrity of the medium. Observations should also relate the geology to the observed forms of damage mechanisms and their relative intensity. Only at that stage should a procedure and measurement technique to quantify blast damage be selected. This should be able to relate damage to both geology and blast design. The capability only exists to proceed further and relate damage to mining method, sequence and ground support.

The experience gained in this sequence of work should be exploited by formally developing a damage-audit procedure that can be integrated into routine mine-production control. Controlled blasting may then be optimized according to damage tolerances, geology and blast design. An underlying requirement is to convince all personnel of the benefits of damage control for mine safety and efficiency. This paper has reviewed the available damage-measurement techniques. At this point in time, it is felt that simple observational procedures to derive inherent and mining-induced damage indices, integrated with geological mapping, represent the most effective approach to damage monitoring. Nevertheless, the ultimate aim should be to develop a robust geophysical technique capable of extending these simple practical procedures to improve the resolution and volumetric coverage of damage characterization. ■

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