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Measurement-while-drilling technique and its scope in design and prediction of rock blasting

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ABSTRACT

With rampant growth and improvements in drilling technology, drilling of blast holes should no longer be viewed as an arduous sub-process in any mining or excavation process. Instead, it must be viewed as an important opportunity to quickly and accurately measure the geo-mechanical features of the rock mass on-site, much in advance of the downstream operations. It is well established that even the slightest variation in lithology, ground conditions, blast designs vis-à-vis geologic features and explosives performance, results in drastic changes in fragmentation results. Keeping in mind the importance of state-of-the-art measurement-while-drilling (MWD) technique, the current paper focuses on integrating this technique with the blasting operation in order to enhance the blasting designs and results. The paper presents a preliminary understanding of various blasting models, blastability and other related concepts, to review the state-of-the-art advancements and researches done in this area. In light of this, the paper highlights the future needs and implications on drill monitoring systems for improved information to enhance the blasting results.

1. Introduction

Blasting is the most frequent, versatile and often the most economical method of breaking rocks. Owing to their dependence on a complex interaction of intact rock and rock mass properties, blast geometry, hole deviation, explosive properties and initiation sequences, rock blasting is considered as a truly complicated operation. For application of MWD for investigation and value addition in blasting, the emphasis must be laid on quantitative as well as qualitative ascertainment of intact rock and rock mass properties. The influence of intact rock and rock mass properties on blasting efficiency has been at the center of much research, from the beginning and has been carried out by numerous researchers [1–10]. These citations are small in comparison with the sustained research in this field. Though insufficient, these references necessarily serve useful purpose in understanding the complexities involved in rock breakage by blasting.

The influence of intact rock and rock mass properties has been incorporated into blasting in various forms, such as 'blastability coefficient', 'rock factor', 'Blastability Index (BI)' and 'bond work index' [11–14]. Nevertheless, selecting just a few parameters, such as rock properties, to represent the resistance of the rock mass to fragmentation by blasting has been a major limitation in describing the ease of fragmentation. Furthermore, by defining these parameters based on a few rock samples, often representing an entire mine, has further reduced the possibilities of detailed blast design. This is, perhaps, the most significant reason as to why blast designs are largely empirical and mostly governed by rules of the thumb. Since the rock mass properties are significant in any blasting program, their proper characterization is critical for effective design and usage of explosive energy. The need is to rationalize blast designs and practices vis-à-vis the influence of intact rock property, in-situ rock mass properties, discontinuity structures and their interactions by use of modern, state-of-the-art techniques, methods and procedures.

It is in this context that the ability of the MWD technique must be fully exploited for its useful application in blast designs and explosive loading patterns. It may be appropriate to mention the ability of MWD to define the variations in bench geology. A continuous monitoring and read-out of the drill parameters are capable of providing useful information on variability of the rock mass.

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The monitored drill hole data, in conjunction with knowledge of the geology of area, defined from core logging or surface maps, can be correlated with the rock variability on mine benches, which in turn could be of immense use in improved blast design to enable selection of proper explosives, charge distribution, stemming primer positions as well as burden, spacing and other related blast design parameters.

2. Models for assessment of fragmentation by blasting

In blasting studies, there has been continued development and improvement in the assessment of fragment size by use of photographic and image analysis techniques. In view of this continued growth, it becomes imperative that researchers expand their horizons in the field of post-blast rock fragmentation assessment. It is worthwhile stressing the use of drill monitoring data and its integration with fragmentation assessment. Eloranta [14] expressed this by stating that the key parameters in fragmentation are drill monitoring and optical image analysis. These should focus on finding clues in the drilling data to predict fragmentation. Yin et al. [30] used the data from the Thunderbird-Pacific system at Minntac mine to develop useful relationships among the drill monitoring data to yield necessary information on fragmentation. The algorithms of image analysis software are based on fragmentation evaluation models. As such, concurrent with the use of the image analysis approach, the empirical relationships to predict fragmentation and its distribution in the blasted muckpile have also grown at a rapid pace. Hence, an understanding of various models for predicting the fragment size and its distribution in the muckpile is necessary.

2.1. Kuznetsov equation

The Kuznetsov equation [12] relates the mean fragment size in the blasted muckpile to the quantity of explosive needed to blast a given volume of rock. The equation is expressed as:

$$K50 = A[V/Q]^{0.8}Q^{1/6} \tag{1}$$

where $K50$ = mean fragment size (cm), A = rock factor, V = volume of rock broken per hole (m^3), Q = mass of TNT equivalent explosive per hole (kg).

Values of A = 7 for medium rock, 10 for hard, highly fissured rocks and 13 for hard, weakly fissured rocks were suggested. Since TNT is not used as the standard explosive for comparison, an equivalent quantity of any explosive (Q_c) was related to TNT as:

$$Q = Q_c[E_c/1090] \tag{2}$$

where E_c = absolute weight strength of explosive (cal/g) and the factor 1090 is the absolute weight strength of TNT. Eqs. (1) and (2) can be simplified and rewritten as:

$$K_{50} = Aq^{-0.8}Qc^{1/6}[E_c/1090]^{-19/30} \tag{3}$$

where q is the inverse of V/Q_c, defined as the powder factor (kg/m^3).

From the above relationships, it is clearly evident that the rock factor, A, has greatest influence on mean fragment size since it bears the highest exponent value. Hence, it may not be desirable to propose a rough estimate for it. Instead, the value of the rock factor has been suggested to be precisely evaluated by considering important rock parameters as given in Eq. (4), proposed by Cunningham [7] as:

$$A = 0.006[RMD + JF + RDI + HI] \tag{4}$$

where RMD = Rock Mass Description, JF = Joint Factor, RDI = Rock Density Influence.

The range of values for these parameters was described for varying rock mass conditions (Table 1).

2.2. Rosin–Rammler equation

The Rosin–Rammler equation was adopted from a coal comminution approach by Cunningham [5] for analysis of fragment size and its distribution in the blasted muckpile. The equation is given as:

$$R = e^{[-X/X_C]n} \tag{5}$$

where R = fraction of material retained on screen (cm), X = screen size, X_c = constant called the 'characteristic size', n = constant called the 'uniformity index'.

The uniformity index has a typical range of 0.8–2.2. A value of 0.8 means that the muckpile is non-uniform, while a value of 2.2 indicates that the muckpile has a majority of fragments close to the mean fragment size.

2.3. Kuz-Ram model

This model was proposed by Cunningham [5] by combining the Kuznetsov and Rosin–Rammler equations by assuming $X = K50$ in the Rosin–Rammler equation, which means $R = 50\% = 0.5$. This assumption modifies Eq. (5) as:

$$0.5 = e^{-[X/X_C]n} \tag{6}$$

This implies that $K50$ can be determined from the Kuznetsov equation and the characteristic size can be computed if n is known. Furthermore, if both X_c and n are known, the distribution can be known from the Rosin–Rammler distribution. The resulting model is known as the Kuz-Ram model. Cunningham proposed the following equation for estimation of n:

$$n = [2.2 - 14[B/d]][1 - W/B]\left[\frac{1 + S/B}{2}\right]^{0.5}\left[\frac{L_b - L_c}{L_b + L_c} + 0.1\right]^{0.1} L/H \tag{7}$$

where B = burden (m), d = hole diameter (mm), W = standard deviation of drilling accuracy (m), L = charge length above the grade (m), L_b = bottom charge length (m), L_c = column charge length (m), H = bench height (m).

2.4. TCM and CZM

The Two-Component Model (TCM) and the Crushed Zone Model (CZM) were evolved to overcome the limitations of the Kuz-Ram model. When a blast hole is detonated, rock breakage occurs in two different stress regions: compressive and tensile. In the first region, the compressive stress waves form a crushed zone in the immediate vicinity of the blast holes. The second region, namely the cracked zone, occurs outside the crushed zone and consists of radial cracking. The widely used Kuz-Ram model does not recognize these two different blast regions. In the case of hard rocks or blasting where the extent of the crushed zone is minimum, the Kuz-Ram model may give a reasonably good description. However, experience has revealed that the Kuz-Ram model is capable of predicting the coarser range quite precisely, but tends to significantly eliminate the amount of fines, which are generated from the crushed zone [15,16]. Since there are numerous blasting situations where the amount of crushing plays a vital role, modeling of rock fragmentation with a single distribution function is not appropriate. The JKMRC developed two blast fragmentation models, as part of their mine-to-mill project, to overcome the limitation posed by the Kuz-Ram model. These models TCM and CZM were developed by Djordjevic et al. [15] respectively. These models are preferred over the Kuz-Ram model due to their improved capability