BLASTING IN MINES – NEW TRENDS

WORKSHOP HOSTED BY FRAGBLAST 10 — THE 10TH INTERNATIONAL SYMPOSIUM ON ROCK FRAGMENTATION BY BLASTING, NEW DELHI, INDIA, 24–25 NOVEMBER 2012

Blasting in Mines – New Trends

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Preface

We take pleasure in writing this short preface to the proceedings of the Workshop on "Blasting in Mines – New Trends" which will precede Fragblast 10. A Workshop, according to the deft definition of Oxford Reference Dictionary, is a meeting at which a group engages in intensive discussion on a particular subject, seeking to enrich and upgade the level of awareness of participants in the subject.

This Workshop hopefully will address the key issues of new trends in blasting practice in mines which have undergone a sea change in the recent past and continue to be honed and reconfigured to meet the demands of today's mining practice. We have sifted the submissions to Fragblast 10 and collated 17 presentations which could meet the Workshop objectives. We have a mix of authors and subjects which highlight the evolving trends in blasting in mines. These range from special techniques of cast blasting, applications of seed wave modelling for improved fragmentation, to design of mass blasts and controlled blasting for stability of pitwalls.

The workshop brings together the diverse perspectives of engineers from Australia, Chile, Brazil, China, India and Germany. We hope that by sharing their experiences, the participants will be able to incorporate the best practices of other countries in their own methods.

We sincerely hope that the in-depth discussions in the Workshop will stimulate the participants to become blasting engineers capable of facing new challenges in developing and defining the best mining practices for the future.

We thank the Convenor of Fragblast 10, Dr. Pradeep K. Singh, for his unstinting cooperation in the selection of presentations and even their editing.

Ajoy K. Ghose
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Seed wave modelling applications for fragmentation, damage, and environmental impact control

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ABSTRACT: Modern mine productivity places a very strong focus on achieving high levels of control over fragmentation, wall control (damage to excavation limits) and environmental impact. The introduction and extensive use of electronic initiation heightens the focus of each one of these blasting outcomes, though systems to permit optimisation are not readily or widely available, and are seldom used in any routine manner in the mining industry. Seed wave modelling, in conjunction with electronic initiation, enables blasting engineers to identify optimum timing configurations for any type of blast, with respect to each of the three mentioned blasting outcomes. Fragmentation, either in the body of the blast or in the stemming zone of blasts, can be increased substantially by optimising timing from the point of view of maximising induced stresses within the rock mass, enabling quantification of a fragmentation index for specific vertical bench sections. Damage, either in the bench batter behind a limits blast, or in the underlying berm, can similarly be minimised by careful delay timing, often more effectively than by costly reduction of blasthole diameter, enabling definition of a Probability of Damage curve. Environmental vibration impacts, either in the medium-field as an impact on vibration-sensitive slopes or mine sectors, or in the far-field as an impact on nearby occupied structures, can also be minimised by careful selection of delay timing. This paper presents the use of seed wave modelling as an everyday tool to enable blasting engineers to make engineering-based decisions regarding the delay timing which will bring maximum control over these blasting impacts.

1 INTRODUCTION

The now-common application of electronic initiation of blasts in open pit mining appears to be delivering the promised benefits of precise and programmable firing times, judging by the plethora of papers extolling the wide range of benefits. However, it also appears that from the user perspective, there remains a technology gap to assist site personnel in choosing the optimum timing to provide best results in terms of fragmentation, damage, or environmental impact control. While electronic initiation enables a much greater flexibility as regards delay timing and blast sequencing, the shot-firer is facing a dilemma of choice as regards a methodology to identify that particular timing which will deliver the desired results.

In the experience of the author, Seed Wave Modelling provides very useful assistance in identifying initiation timing which takes maximum advantage of either constructive or destructive interference opportunities of the vibration waves generated by individual charges as they detonate. By enhancing the constructive interference of waves within the body of the blast, the state of induced tension is maximised and will likely yield an increase in both macro-and micro-fracturing. Modelling assists in identifying those delay times which maximise the constructive interference of waves from individual charges. By maximising the destructive interference of waves at a particular location outside the blast area, the levels of induced vibration may be reduced, with likely benefits in terms of reduced environmental impact (including reduced impact on pit walls and nearby rock slopes). Modelling also assists in identifying those delay times which maximise the destructive interferences of waves from individual charges.

In the context of blast vibration impacts, the Seed Wave Model can be applied to either vibration (ground-borne vibration), or air overpressure (air-borne vibration). Ground vibration applications can involve either velocity-based vibration waves or acceleration-based waves.

2 TECHNICAL BACKGROUND

Seed Wave Modelling is based on the principle of linear wave superposition, i.e. that the net result from the detonation of multiple charges can be found by simple linear summation of the waves generated by each individual charge. The concept and application are well explained by Hinzen (1988). Commonly, it is assumed that each charge produces the same shape of vibration wave, and that the amplitude of each wave is dependent on the weight of explosive associated with each charge. In some applications, individual waves have unique shapes derived from a single measured "parent" wave and related via "parent-sibling" Fourier manipulation. Some landmark studies of the concept of Seed Wave Modelling are worth mention, including those by Blair (1999), Spathis (2006), Yang (2007), Yang & Scovira (2010).

In its purest form, the seed or signature wave from a single charge is fired in the same location as the ensuing multi-hole blast, and its impact (vibration or overpressure) is measured at the point of interest. Modelling then becomes very specific to that geometry, since the shape and characteristics of the recorded seed wave are very dependent on the seismic path. Further, the weight of explosive in the single charge should be equal to that of each of the charges in the multi-hole blast so that no further amplitude adjustment need be performed. In reality, however, the seed wave requires adjustment, sometimes in both shape and amplitude, since the distances between individual holes (or deck charges) and the monitoring point are not constant and not equal to the distance at which the seed waveform was recorded. The angle between the charge and the monitoring point may also vary for long or wide blasts, affecting the relative amplitudes of the horizontal components of a triaxial monitoring system.

Blair (1999) presents particularly comprehensive details on methods of adjustment of seed wave amplitude and shape, drawing heavily on Monte Carlo methods to reflect real variability in both shape and amplitude of seed waves both as they are generated at the blasthole and as they are received at the monitoring location. Yang (2007) also presents methods to adjust seed wave shape and amplitude for both differences in charge weight and propagation distance for both close-in monitoring and far-off monitoring.

It is worth special mention that adjustment of the seed wave amplitude to account for charge weight variability between different holes or deck charges should also have a degree of randomness associated with it to accurately reflect the variability observed in traditional square root scaling regressions involving single charges measured over varying distances. Effectively, the K term of the charge weight scaling Equation (1) should be considered to have a log-normal distribution, with a standard deviation derived from field measurements of single charges.

$$PPV = K \times SD^{-n} \tag{1}$$

Where PPV represents the estimated vibration amplitude, SD represents scaled distance (distance divided by the square root of the charge weight), and K and n are regression parameters.

In its simplest form, the Seed Wave Model can be represented mathematically, for a blast containing N blastholes with individual delay times of d_i, by Equation (2).

$$R(t) = \sum_{i=1}^{N} A_i \times S(t - d_i) \tag{2}$$

where $R(t)$ is the resultant time history waveform representing the sum of the separate seed waves $S(t)$, and A_i represent the amplitudes of each seed wave, calculated using Equation (1) according to the explosive weight and the distance of propagation for each of the individual charges.

As Hinzen (1988), Blair (1999), and Anderson (2008) point out, the same calculation can be undertaken as a convolution in either the frequency or time domain. The calculation can be made significantly more complicated, and probably more realistic, by adding a degree of stochastic behaviour to the shape of the different seed waves, and to the amplitude of each wave, as explained by Blair (1999).

Equation (2) must be applied to each of the three geophone components which comprise a standard vibration monitoring system, so that Equation (2) can be considered to apply to each of the triaxial geophone/accelerometer components, i.e.:

$$\begin{aligned} R_L(t) &= \sum_{i=1}^{N} A_i \times S_L(t - d_i) \\ R_T(t) &= \sum_{i=1}^{N} A_i \times S_T(t - d_i) \\ R_V(t) &= \sum_{i=1}^{N} A_i \times S_V(t - d_i) \end{aligned} \tag{3}$$

where the subscripts L, T and V refer to the Longitudinal, Transverse, and Vertical components of a triaxial configurations of geophones or accelerometers, and where the longitudinal geophone axis is aligned in the direction of the single hole used to generate the seed wave.

The greatest complication, however, comes from the requirement to undertake a coordinate transformation to account for the different angles of incidence of the waves from different blastholes, according to the alignment of the triaxial monitoring transducer, and the lateral spread in blasthole coordinates. This is also explained in considerable detail by Blair (1999), who also points out the special case of the transverse component of the triaxial geophone or accelerometer system. Interestingly, no other worker has described the same irregular behaviour of the transverse component.

The question also arises as to the test conditions which provide the best estimate of the seed wave shape. While the ideal conditions would include a charge weight for the seed wave equal to that typically used in a production hole, and confinement conditions equal to those applicable for the production holes (i.e. a normal free face), it is frequently difficult to achieve such conditions. For example, if the modelling exercise is trying to decide whether to use 1 deck or 2 decks per hole, what charge weight should the single hole use? And many quarry operators do not want to fire a single hole to a free face for fear of creating later problems due to an irregular face, forcing the use of a seed wave obtained under fully confined conditions, away from a free face. Finally, how close to the production blast should the single charge be fired? Ideally, within the same volume, but does that mean the Seed Wave Model must obtain a new seed wave prior to every blast?

3 EXPERIMENTAL CONSIDERATIONS

The answer to the questions above depends quite strongly on the level of precision required of the modelling. If the intention of the modelling is to estimate the vibration characteristics (peak amplitude, frequency and duration) as precisely as possible, then it probably is necessary to obtain a seed wave within a few tens of metres of each production blast. If the objective is to comply with statutory limits (of either vibration or overpressure), then it may be sufficient to simply estimate the levels within a known margin of error such as plus or minus 20%, applying appropriate percentiles to the predictions to ensure statutory compliance. The author's approach is to use the same seed wave until the accuracy of prediction becomes unacceptable, at which time a new seed wave is required.

Bernard (2012) notes that through a process of deconvolution (using the measured production blast waveform and the known time-delayed Dirac delta function) he is able to extract a new seed wave from the previous production blast, obviating the need for repetitive single hole firing. He presents statistics in terms of accuracy of prediction after around 600 applications (an unknown number of which utilised the seed wave extraction method), with 50% of predictions being within 10% of the measured value, and on 28% of occasions, the actual measurements exceed the predicted levels by more than 10%. Further, approximately 70% of the outcomes represented a vibration reduction of at least 30% relative to previous measurements.

Anderson (2008) presented the same deconvolution idea proposed by Bernard (2012), referring to the time-delayed Dirac delta function as a "comb function", though he was much more guarded as regards the success of the process. Personal experience suggests that the deconvolution process to extract a seed wave from a measured seismic record usually produces a wave bearing little resemblance to a single charge wavelet.

As regards the configuration of the charge used to record the seed wave, it is again considered ideal if the charge can as closely approximate a normal production charge as possible, in terms of size, diameter, length, explosive density, and confinement conditions. Anderson (2008) implies the expectation that charge confinement (i.e. a charge fired to a free face with a normal burden or fully confined with infinite burden) will affect vibration levels—an expectation supported by most text books. Interestingly, though, Brent et al (2001) shows that burden confinement appeared to have no effect on peak vibration levels. Ramulu et al (2002) show similar results (i.e. no influence of burden) for the far field (scaled distance >~30 m/kg$^{0.5}$), but note very different behaviour in the near-field (scaled distance <20 m/kg$^{0.5}$). The author's own experience is that seed waves obtained under conditions of infinite confinement (very large stemming columns and no free face) do not necessarily generate higher vibration levels (Figure 1), or show a greater tendency to over-predict vibration impacts, though that experience is limited to scaled distances in excess of 15 m/kg$^{0.5}$.

Figure 1 is not presented to imply that free face vibration levels are higher than those from fully confined charges, but rather that charge

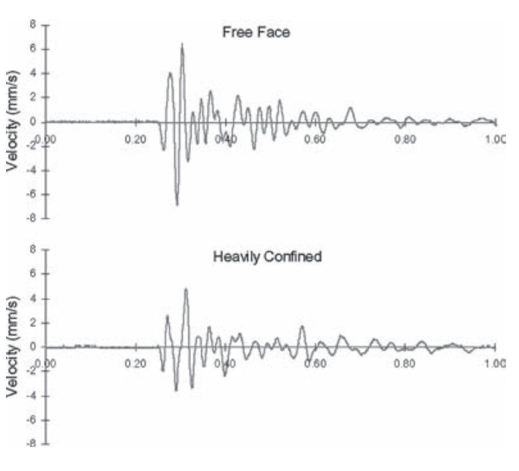

Figure 1. Seed waves from 118 kg charges fired to free face (upper) and fully confined (lower), amplitudes scaled for constant scaled distance of 20 m/kg$^{0.5}$.

confinement in this test did not appear to increase peak vibration level. When considering the differences in amplitude between waves produced by charges of equal weight, it should be remembered that normal amplitude variability appears to have a coefficient of variance (standard deviation divided by mean) of the order of 10%. The source of the variation is unknown, but may relate to differences in explosive coupling, explosive detonation characteristics, or reproducibility of the vibration measurement system.

The author's experience also suggests that the charge weight used to obtain the seed wave can be as small as one third to one half of the size of the production charges being modelled, while still providing reliable vibration predictions. Despite this finding, it is still considered preferable that the seed wave be generated by a charge which closely approximates the size of the production charges (in diameter and weight).

In the author's experience, the more detailed that a seed wave collection campaign can be, the more reliable will be the modelling. An example is cited for the case of modelling conducted for every overburden blast over a period of more than 12 months in an Australian open cut coal mine, based on extensive single hole firing and vibration monitoring. The weight of charge used in the various single holes varied from 80 kg to 200 kg, while production charge weights varied over the range 120 to 300 kg. The seed wave collection campaign enabled accurate determination of the vibration propagation conditions for single charges, as shown in Figure 2.

A complication to the Seed Wave Model seems to lie in the use of permanent monitoring systems. Such systems are installed with a fixed alignment of the geophones, which will generally not be consistent with the assumed convention of the radial (or longitudinal) geophone pointing towards the location of the seed wave charge.

As Figure 3 shows, the use of a permanent monitor which is not well aligned with a blast being modelled may result in the transverse component being better aligned with the single hole charge than the radial component. Considering the comments made by Blair (1999) with respect to amplitude adjustment of the transverse geophone/accelerometer signal, this misalignment will likely cause difficulties in modelling. When using seed wave modelling, it is considered advisable that monitoring always be conducted with the radial geophone oriented with its axis pointing to the centre of the production blast being modelled and monitored. This may mean that seed wave modelling should utilise a well-aligned roving monitor rather than a permanent, poorly aligned monitor.

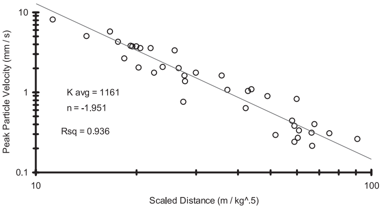

Figure 2. Scaled distance regression for single charges with charge weight varying from 80 to 200 kg.

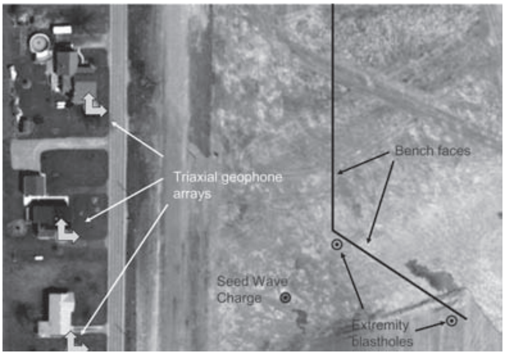

Figure 3. Permanent monitors poorly aligned (arrows show orientation of radial and transverse geophones) for seed wave modelling of single-row blast consisting of 15 holes (extremity holes shown).

3.1 *Overpressure modelling*

Very few papers have presented the results of air overpressure modelling using the superposition principle. Blair (1999) presents some results, and Richards & Moore (2002) present the results of modelling using mathematical wavelets rather than measured wavelets. Richards & Moore (2002) rather the very relevant point that overpressure modelling needs to consider at least two separate sources of overpressure—the collar of each blasthole as well as the free face. When modelling is conducted at a sensitive receiver located forward of the free face, then the program should consider the contribution from each face hole separately from the contributions from every hole collar. Clearly, separate seed waves need to be recorded for each wave type—seeds measured forward of the face will include both collar and free face components (inseparable, though the free face component is expected to dominate in most situations), and seeds measured behind the free face will include only the collar component. When modelling is conducted at a sensitive receiver located behind the free face, then the program should consider only the contributions from the hole collars. While modelling is simplified by the relatively constant shape of the seed wave pulse over long distances, it is complicated by local topography and meteorological conditions such as winds and air temperature gradients which will both act to change the seed wave shape and attenuation in ways which are difficult to model. Overpressure modelling can be further complicated by occasional ejections (blow-outs) from either the free face or the collars of holes.

3.2 *Near-field modelling*

Modelling in the near-field is generally oriented towards controlling the impact of blasting on the surrounding rock excavation, and trying to minimise induced damage and overbreak. The application of the model becomes complicated under these conditions, since the distance from the charge to the modelling point is no longer constant for all points along the charge length. A point which is only 1 metre from the centre of a long column may be 10 metres or more from the extremity of the charge column, so that the amplitude adjustment terms A_i in Equation (2) no longer apply. Further complications arise, including:

1. the shape of the waveform ($S(t)$ in Equation 2) will vary according to the geometry between the modelling point and the charge;
2. the assumption of the same shape of seed wave from each charge within a rather large blast becomes doubtful, due to widely differing propagation distances;
3. coordinate transformation is required in three directions for all three geophone components;
4. charge configurations frequently involve changes in explosive type (density and weight strength) or the inclusion of air decks which exacerbate the variability in impacts along the length of the charge;
5. the strain rates generally exceed the limits considered appropriate for linear superposition.

Heelan (1952) considered a method to estimate the disturbance generated by the detonation of explosives in a cylindrical shot hole. In the discussion of his results, he notes “Conditions in the immediate neighborhood of the shot hole are such that the assumptions upon which our work is founded, in particular the assumption of infinitesimal strain, are not verified in this domain.”. He also qualifies his solution to applications at large distances with respect to the charge length, suggesting that its utilisation to estimate vibration amplitudes within a few metres of long charges lies outside the area of recommended use of the solution.

The application of linear superposition models in the near-field, where strain levels may exceed 1000 micro-strain and rock response can not be considered elastic (cracking is occurring), is likely to be fraught with errors pertaining to the assumption of linear behaviour in a zone of non-linear rock response. Despite this limitation, numerous workers have reported good value and success in applying the principle of linear superposition, or models based on assumed linear elastic behaviour, in the near-field to investigate blast-induced damage (Blair & Minchinton 2006, Villalba & McKenzie 2006, Yang & Scovira (2010), Adamson et al 2011).

Yang & Scovira (2010) present methods to adjust seed wave shape (pulse broadening) according to varying propagation distances as well as propagation through previously blasted ground, when applying the seed wave to near-field damage assessment—a factor which can play an important role in determining the interaction between waves.

4 APPLICATIONS—FAR-FIELD

Environmental vibration impact control appears to be the area of principal application of seed wave modelling. Applications focus on determination of inter-deck, inter-hole and inter-row delay times to either minimise peak particle vibration (PPV) levels, or alternatively to adjust vibration frequencies.

4.1 *Controlling PPV*

The application of seed wave modelling for the prediction of vibration levels from even large and complicated blasts has been well demonstrated (Hinzen et al 1987, Blair 1999, Yang 2007, Yang & Scovira, 2010). This demonstrated ability then becomes the basis for adding a search routine to the models, so that the modelling can be repeated for many different timing combinations. Bernard (2012) is probably the first to publish the results of such search routines, along with measures of the accuracy of prediction, for a statistically large number of blasts. Bernard’s data suggest that, through careful selection of delay times (inter-deck, inter-hole and inter-row), vibration levels can probably be lowered by an average of 30% relative to those achieved with conventional non-electric initiation, and in some case by much greater amounts.

In the author’s experience, using a proprietary Monte Carlo based model, vibration levels for different timing combinations can vary over a factor of at least 4, as illustrated in Figure 4 for 1053 different delay timing combinations (single row blast, 5 decks per hole, inter-deck delays from 8 to 20 ms, inter-hole delays from 20 to 100 ms).

Perhaps the most interesting feature in Figure 4 is the large number of delay combinations which

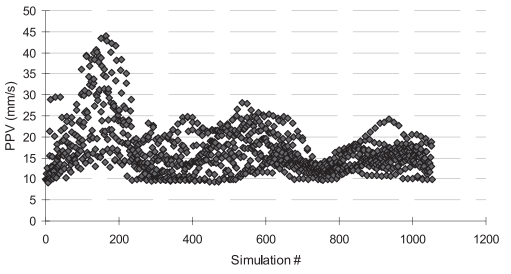

Figure 4. Variability in induced vibration levels for 1053 different delay combinations, for a single-row blast fired within 100 metres of housing.

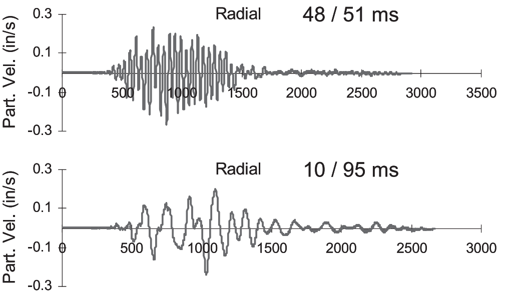

Figure 5. Modelled waveforms obtained from the same blast with different inter-hole and inter-row delay times. Numbers above each waveform show inter-hole/inter-row delay times.

will produce relatively low levels of vibration (~9–10 mm/s), suggesting that there is no unique delay timing combination, and that the blaster will have quite a range of delay times from which to choose a timing appropriate to the other goals of fragmentation, displacement, and overpressure. The data of Figure 4 also highlight the penalty in terms of imprudent or random timing selection, and the need for an engineering tool to assist in selection of times which deliver control over blasting outcomes—a tool desperately needed in the light of the newly-won flexibility offered by programmable electronic initiation. Adoption of the same electronic timing that was previously used with non-electric initiation may well lead to very unsatisfactory results in terms of vibration impacts.

4.2 *Adjusting frequency*

Statutory vibration limits in many countries are frequency dependent, as highlighted in the various vibration standards such as DIN 4150, RI 8507. The "frequency" of a vibration wave as reported by commercial seismographs relates to the time between successive zero-cross points straddling the instant at which the peak particle velocity is recorded on each of the triaxial gauges. While the validity of this measurement as a meaningful measure of frequency can be debated, it is the system adopted for all known commercial blasting seismographs, and it forms the basis for determining compliance in many countries.

In addition to the frequency of the peak vibration, commercial blasting seismographs also display many other amplitude/frequency points determined by finding local maxima/minima between successive zero cross points on each seismograph channel. Figure 5, shows two different possible outcomes for the same blast, depending on assigned inter-hole and inter-row delay timing, and Figure 6 presents the RI 8507 spectra for the same waves. The units for these figures have been kept in the original (US) units for which the study results were obtained.

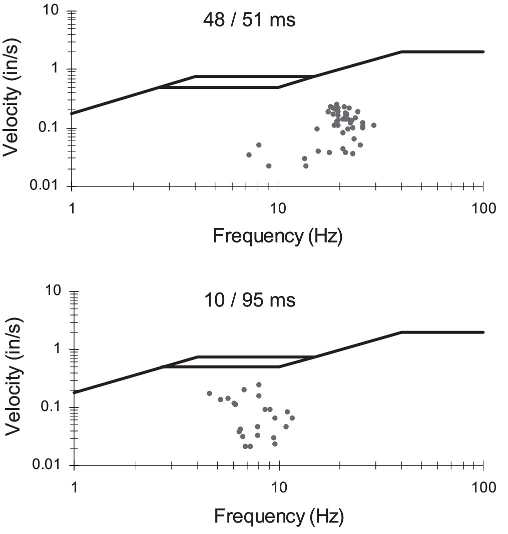

Figure 6. Two different frequency spectra for the modelled waveforms of Figure 5.

In spite of the low amplitudes, the blasting is generating widespread complaint, possibly as a result of the low frequencies of vibration tending to induce building resonances. In order to reduce community perceptions of blasting vibration impacts (and structural resonances), it is possible to search for delay timing combinations which shift the mass of points towards the right, to higher frequencies. As long as this can be done while still remaining compliant with respect to peak vibration amplitudes, blasters are able to choose delay timing which addresses both regulatory compliance as well as community perception.

Searching for a delay combination which maximises frequency requires that the mass of points in the RI 8507 plot be characterised by a single index such as the frequency of the centre of mass. Because of the significance of low frequencies in evaluating the potential for exciting a house resonance, the author prefers an index which focuses on the low end of the distribution of points, and is using the 10% point on the cumulative amplitude spectrum, as illustrated in Figure 7. The cumulative spectrum can be estimated either from Zero Cross spectra using average vibration amplitudes within discrete frequency bins, or from the Fourier spectrum.

Using the cumulative spectra, the two blasts are assigned frequency indices of approximately 5 Hz (10/95 ms) and 17 Hz (48/51 ms).

While delay timing can go some way towards changing personal perception, it must be remembered that the frequency spectrum of vibrations from blasting is controlled principally by the rock mass. Major shifts in frequency by changing delay timing will rarely be possible.

Accepting that human perception of vibration depends on a combination of amplitude, duration and frequency, the issue of personal perception can be taken a step further, by proposing a Perception Index (*PI*), which takes into account the peak particle velocity (*PPV*), vibration duration (T_v) and the derived frequency index (I_f) as:

$$PI = PPV^a \times T_v^b \times I_f^c \tag{4}$$

where PPV can be in either US or metric units, T_v is in seconds, and I_f is in Hertz.

In Equation 4, each term can be derived from analysis of either the modelled or measured seismic waveforms. The exponents a, b and c in the equation can be considered to reflect the relative sensitivities of the different components of the index, and can be adjusted according to community response and feedback, and field experience. Clearly, the exponent c will be negative. Search algorithms can then find the delay times that minimise PPV, maximise the frequency index, or minimise the perception index. The absolute value of the perception index has no particular significance, so can readily be implemented in either US or metric units. For the two blasts of Figure 5, the perception indices were calculated to be 0.47 for the upper waveform, and 1.81 for the lower waveform, for the exponent values of $a = 1$, $b = 0.5$, $c = -1$.

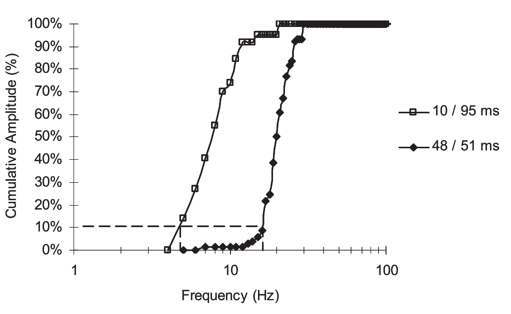

Figure 7. Cumulative amplitude spectra for the two blasts of Figure 5, derived from the RI 8507 graphs of Figure 6. Also shown are the 10% values, representing the frequency indices for each of the waveforms of Figure 5.

5 APPLICATIONS—NEAR FIELD

Seed wave modelling applications in the near-field are also quite commonly encountered in the literature (McKenzie & Holley 2004, Blair & Minchinton 2006, Villalba & McKenzie 2006, Yang 2007, Yang & Scovira 2010, Adamson et al 2011), despite the unknown errors introduced by the assumption of linear behaviour in a zone of non-linear response. Of the seed wave modelling applications encountered (i.e. those capable of considering wave interactions from multiple charges), only those utilising the flawed Holmberg-Persson equation, or modifications thereof, are able to estimate variability in vibration intensity as a function of depth within the bench. Other models are, therefore, unable to address near-field vibration from interactions between multiple charges within the bench volume.

5.1 *Damage control*

The attractiveness of the various techniques lies in the ability to quantify (perhaps roughly) the variability in the severity of impact in adjacent rock masses of the detonation of explosive charges. The technique has obvious application in assisting with drilling and charging design, in particular the contour and near-contour blastholes with respect to the designed excavation limits. Whereas Holmberg & Persson (1979), Hustrulid & Lu (2002) and Blair & Minchinton (2006) proposed methods to estimate vibration contours around cylindrical charges in the near-field, those methods do not seem able to take into consideration the interaction between charges due to delay timing.

Spathis (2001) used conventional seed wave modelling to consider vibration intensity within 30 metres of blasting at the Freeport Grasberg mine in Indonesia, showing the rather strong influence of delay timing on peak vibration levels, and the potential of the induced vibrations to cause damage to the surrounding rock mass. Those authors then went on to present a curve presenting the probability that peak vibration levels would exceed particular threshold values as a function of delay timing and distance from the blast. Their focus was clearly on deriving a blast design for use with large diameter (311 mm and 351 mm) holes in large scale open pit mining. Villalba & McKenzie (2006) presented a similar very concept, but converted the curve to a probability of damage curve using PPV as the damage criteria. The latter authors showed how the probability of damage curves could be substantially altered to reduce the extent of pit wall damage by adjustments to drilling, charging and blasting practices.

In Figure 8, the probability of damage curves relate to the probability, at any distance behind the blast, that the vibration levels, estimated at all different depths within the bench, will exceed a critical level considered to relate to damage—a level which can be derived from theoretical analysis or from field correlations. A Monte Carlos algorithm is used typically to make around 10,000 estimates of levels at various depths in the bench and various distances behind the blast.

Adamson et al (2011) used the seed wave model, with an incorporated search routine, to find delay timing which would minimise the level of vibration induced in a sensitive pit wall, achieving a reduction of almost 50% in the width of the damage zone behind a limits blast pattern. The reduction is noteworthy, since many mining applications achieve similar or less reduction by much more costly and troublesome means.

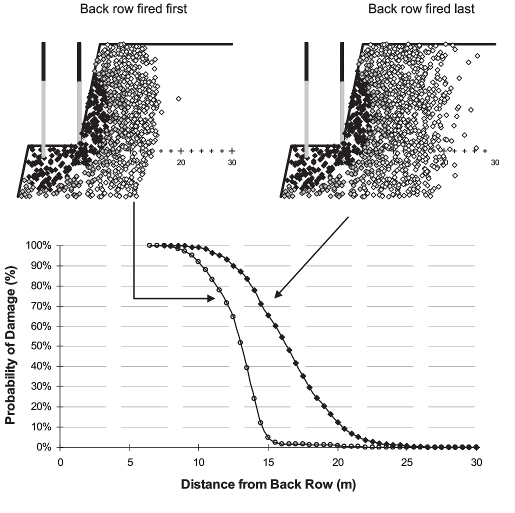

Figure 8. Reductions in extent of high level vibrations induced in pit wall behind limits blast, using seed wave model (after Villalba & McKenzie 2006).

Importantly, seed wave modelling applications are probably the only way to fully evaluate different blast design options in terms of their impact on the excavation limits, since they are potentially able to take into consideration all of the blast design variables (burden, spacing, hole diameter, explosive type, powder factor, stem length, subdrill, initiation system, delay timing, and point of initiation).

5.2 *Fragmentation control*

Ignoring the unknown errors associated with the assumption of linear superposition, there is further benefit in considering the application of the seed wave concept inside the blast. Rossmanith (2002) proposes that maximising stress-wave interaction within the body of a blast should maximise the amount of fracturing occurring, at both the micro—and macro-scale. Various field researchers (e.g. Vanbrabant & Espinosa 2006) have verified the approach, and the benefits form fast timing, and many of the large scale copper mines in Chile apply the concept on a daily basis for ore blasting with large diameter holes (270–341 mm). In particular, the use of very short delays of around 1 to 3 ms between holes has been found to be the most effective means of addressing oversize in the stemming region of patterns when blasting hard rock.

Using an elemental form of the seed wave model, together with site-specific regression data, images of relative vibration levels (such as those presented by Spathis et al 2001) can be obtained as a function of both charge design and delay timing. In a very similar manner to the data shown in Figure 8, which reflect vibration or stress intensity, similar figures can be derived to study intensity as a function of delay timing. The timing recommendations from these analyses agree rather well with field experience, suggesting that the technique can be useful to evaluate not only timing, but also opportunities for pattern expansion to justify the additional cost of electronic initiation.

In the same way that a damage index can be defined when analysing impacts behind the blast in the pit wall, a similarly-defined fragmentation index can be defined when considering rock response inside the pattern, where a particular and frequent interest is the relative fragmentation achieved in the stemming zone.

Figure 9 presents this concept, which has proven very useful to compare different options for oversize reduction including additional stab holes, stemming charges, increased powder factor, reduced stemming lengths, adjustments to explosive density, or changes to delay timing. In Figure 9, the fragmentation index of 100% could be considered as a simple comparison of the aver-

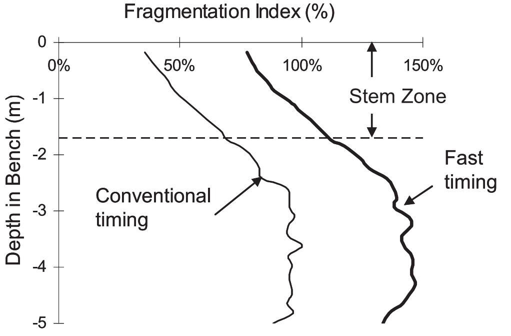

Figure 9. Internal variability in fragmentation with conventional versus fast inter-hole timing. Fragmentation index is expressed relative to conventional timing conditions.

age vibration levels within a series of horizontal layers, expressed with respect to the highest levels which occur, as expected, in the centre of the zone containing the explosive column. The index can be used to compare different explosive types, different powder factors, different drilling geometry, different timing, and even different initiation sequences. The curves are relative to the profiles obtained for some defined base case. In the stemming zone, levels are greatly reduced.

6 CONCLUSIONS

With the advent of electronic initiation, seed wave modelling has become an absolute necessity in order for the benefits of that initiation system to be fully realised in rock fragmentation applications. In a business where incorrect or imprudent delay timing can have very significant impacts on productivity, damage and environmental impact, there appears to be no other way to choose the most appropriate timing from the almost infinite spectrum of possibilities.

Many workers have developed in-house versions of the model, and there are some commercially-available systems as well. A characteristic of each is the requirement for carefully recorded signatures which accurately represent the vibration pulses generated by each independently-delayed charge in the blast. This requires strict control over data gathering programs, and careful orientation of the blasting seismographs.

Modern computers are able to perform thousands of full-blast simulations in periods generally not exceeding a few minutes, and to investigate a very wide range of timing options.

Applications of the model for exploring and quantifying variability in vibration intensity within the bench and around the charge, to account for multiple charge interactions, is needed to complete the set of engineering tools available to design engineers.

The application of the method to address environmental impact control is enormous, and this paper shows that the model can also help address human perception by combining the abilities of seed wave models to accurately estimate both vibration amplitudes and spectra.

REFERENCES

Adamson, W.R., Muñoz, V., & Sarapura, G., 2011. Application of technology for final wall damage control at Cerro Vanguardia, Argentina, *AusIMM, Explo 2011 Conference*, Melbourne, Vic, 27–28 Nov.

Anderson, D.A., 2008. Signature hole blast vibration control—20 years hence and beyond., *34th Annual Conference on Explosives and Blasting Technique, International Society of Explosives Engineers*, Volume 2, Nashville, TN.

Bernard, T., 2012. The truth about signature hole method, 38th Annual Conference on Explosives and Blasting Technique, International Society of Explosives Engineers, Volume 2, Orlando, FL.

Blair, D.P., 1999. Statistical models for ground vibration and air blast, *Int. J. Blasting and Fragmentation,* Vol 3, pp 335–364.

Blair, D.P. & Minchinton, A., 2006. Near-field blast vibration models, in *Proceedings 8th International Symposium on Fragmentation by Blasting—Fragblast 8*, Santiago, Chile, pp 152–159.

G F Brent, G.F., Smith, G.E., & Lye, G.N., 2001. Studies on the Effect of Burden on Blast Damage and the Implementation of New Blasting Practices to Improve Productivity at KCGM's Fimiston Mine, *AusIMM, Explo 2001 Conference*, Hunter Valley, NSW, 28–31 October 2001.

Heelan, P.A., 1952. Radiation from a cylindrical source of finite length, *Geophysics*, 18: 685–696.

Hinzen, K.G., Ludeling, R., Heinemeyer, F., Roh, P., & Steiner, U., 1987. A new approach to predict and reduce blast vibration by modelling of seismograms and using a new electronic initiation system, *13th Annual Conference on Explosives and Blasting Technique, International Society of Explosives Engineers*, Miami, FL, pp 144–161.

Hinzen, K. G., 1988. Modelling of blast vibrations, *Int. J. Rock Mech. Min. Sci, & Geomech Abstr.,* Vol 25, No 6, pp 439–445.

Holmberg, R. & Persson, P.A., 1979. Design of tunnel perimeter blasthole patterns to prevent rock damage, *Proc. IMM Tunnelling '79 Conference*, March 12–16, London.

Hustrulid, W., & Lu, W., 2002. Some general design concepts regarding the control of blast-induced damage during rock slope excavation, in *Proceedings 7th International Symposium on Rock Fragmentation by Blasting—Fragblast 7*, Beijing (Ed: Prof WANG Xuguang) pp 595–604 (Beijing Metallurgical Industry Press).

McKenzie, C. & Holley, K., 2004. A study of damage profiles behind blasts, *Proceedings of the 30th Annual Conference on Explosive and Blasting Technique*, February 1–4, New Orleans, Louisiana, pp 203–214.

Ramulu, M., Chakraborty, A.K., Raina, A.K., Reddy, A.H., & Jethwa, J.L., 2002. in *Proceedings 7th International Symposium on Rock Fragmentation by Blasting—Fragblast 7*, Beijing (Ed: Prof WANG Xuguang) pp 617–624 (Beijing Metallurgical Industry Press).

Richards, A.B., & Moore, A.J, 2002. Airblast design concepts in open pit mines, in *Proceedings 7th International Symposium on Rock Fragmentation by Blasting—Fragblast 7*, Beijing (Ed: Prof WANG Xuguang) pp 553–561 (Beijing Metallurgical Industry Press).

Rossmanith, H.-P., 2002. The use of Lagrange diagrams in precise initiation blasting, Part I: Two interacting blastholes, in *Proceedings Fragblast 6, Sixth Symposium on Rock Fragmentation by Blasting*, Vienna, pp 104–136.

Siskind, D.C., Stagg, M.S., Kopp, J.W., & Dowding, C.H. Structure response and damage produced by ground vibration from surface blasting, *USBM Report of Investigations RI 8507,* United States Department of the Interior, 84 pages.

Spathis, A.T., Smith, G.E., Yacob, I, & Labriola, A., 2001. Wall control at the Freeport Grasberg Opencut Mine: Vibration and gas penetration measurements as a precursor to improvements, *27th Annual Conference on Explosives and Blasting Technique*, International Society of Explosives Engineers, Volume 2, Orlando, FL.

Spathis, A.T., 2006. A scaled charge weight superposition model for rapid vibration estimation, *Int. J. Blasting and Fragmentation*, Vol 10, Nos 1–2, pp 9–31.

Vanbrabant, F., & Espinosa, A., 2006. Impact of short delays sequence on fragmentation by means of electronic detonators: theoretical concepts and field validation, in *Proceedings 8th International Symposium on Fragmentation by Blasting—Fragblast 8*, Santiago, Chile, pp 326–331.

Villalba, I. & McKenzie, C.K., 2006. Evaluating the effectiveness of limits blasts, in *Proceedings 8th International Symposium on Fragmentation by Blasting—Fragblast 8*, Santiago, Chile, pp 290–296.

Yang, R., 2007. Near-field blast vibration monitoring, analysis and modelling, *33rd Annual Conference on Explosives and Blasting Technique, International Society of Explosives Engineers*, Volume 2, Nashville, TN.

Yang, R., & Scovira, D.S., 2010. A model for near and far-field blast vibration based on multiple seed waveforms and transfer functions, *36th Annual Conference on Explosives and Blasting Technique, International Society of Explosives Engineers*, Volume 2, Orlando, FL.

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Design and general practice of mass blast in underground open stopping mining method

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ABSTRACT: The principle of mass blast in underground metal mines is to maximize the void utilization when fragmenting the ore in the commonly used open stoping mining method. For fragmented rock, the swell factor naturally trends to 25%. Thus the size of blast, in terms of volume, can be maximized as long as the undercut in a stope provides over 25% void. Depending on the size of stope, typical stopes can be blasted in 3 blasts or less. In one of the deep gold mines in Canada, even single blast has been successfully used in many stopes. In deep underground mines, less number of blasts per stope means faster cycle time and less damage to the surrounding walls so that the overall productivity is increased. This paper describes the key steps in engineering the mass blast methodology including void evaluation, drill-hole layout and delay timing design around the initial opening of the stope. The application of electronic detonators was a technology breakthrough that made the delay patterns to be practically achievable. This mass blast principle has been widely accepted in underground non-ferrous metal mines in Eastern Canada. Significant production gains have been observed at mining operations with the use of the mass blast principle. It effectively reduces the blasting cycle time, the amount of re-drills and number of re-entries to blasted areas, therefore making it safer for employees who are involved in the re-drill and explosives charging processes.

1 INTRODUCTION

1.1 *Background*

With the booming business in natural resources in the past 10 years, the exploitation of minerals tends to reach more remote locations and/or at greater depth than ever. Consequently, it calls for mining technologies to deal with the harsh environment. In eastern Canada, most underground non-ferrous metals mines are getting deeper and deeper, reaching 3000 m below surface. The high level of horizontal stresses in the tectonic Canadian Shield plays a more important role in the design of mining sequences, stope dimensions, backfill and ground support methods. In drilling and blasting, technology development has been made in all aspects of the process—from drilling equipment for faster penetration and straighter holes to bulk loading of explosives in both down-holes and up-holes and to the application of electronic detonators, etc. For the end-users of such technologies, mining companies have also been active in the application of new technologies to address the new challenges.

For drill and blast under highly stressed conditions, the challenge is how to complete the required activities safely with the shortest exposure possible. First it means how to maintain access to drilled holes prior to charging: how to keep drilled holes open for charging explosives, or how to prevent drilled holes from being squeezed or plugged, and how to reduce re-drill. Secondly, it requires reducing the exposure of workers to blasted areas by reducing the number of re-entries into an active stope. Any blast would cause disturbances to a stressed ground and stress-redistribution or adjustment would inevitably deteriorate ground conditions to a certain degree. Thus the fewer times blasters have to reenter a blasted stope, the better it is for their safety.

This paper is aimed at addressing this challenge from a blasting perspective.

1.2 *General practice*

In the past 15 years, the large and/or deep mining operations in eastern Canada have tried many practices including collar casing for 1.0–1.5 m with plastic tubing, using a wooden wedge to protect the collar, adding a cohesive agent in drilling water to “glue” the cracked rock, and even pre-charging explosives in one or two rings adjacent to a to-be-created void, etc. However, amongst the tested methods, the most effective method is applying the mass blast principle thanks to the availability of electronic detonators in today's initiation market. The idea of this principle is to make the blasts as big as possible in order to maximize the use of existing void to minimize the number of blast in each active stope.

The principal of mass blast offers the following advantages (Liu et al. 2007):

- Shorter blast cycle time: meaning faster turn-around for each stope thus enabling the operation to keep the same production level with fewer number of active stopes.
- Fewer number of blasts per stope: for a typical stope of 20,000 to 40,000 metric tonnes, the whole stope can be completed mostly with 2 or 3 blasts; the maximum can be 4 blasts but there are also some stopes that can be taken out with just one blast—called Single-Shot Stopes.
- Reduced re-drill: many holes can be squeezed and plugged after a blast due to stress redistribution. Fewer blasts can lead to fewer number of holes to be cleaned or re-drilled in an active stope.
- Less re-entry by blasters into a blasted area: thus making it safer for blasting crew.

Applying the mass blast principle generally elevates the mining operation to a higher level of quality: It requires careful engineering, especially on drilling and blasting, as well as production planning. It is also necessary to have disciplined execution of the design by productions personnel. Finally, it is important to have excellent communication and collaboration between engineering and production.

Starting from the engineering side, the mass blast principle relies heavily on the understanding and evaluation of the void available for the blast. Furthermore, the calculated void must be actually available mostly underneath the blast. On the essence, it is critical to understand how much rock swells when estimating the required void in blast design. The next section examines some details of the rock Swell Factor.

2 ROCK SWELL FACTOR

Any fractured rock would swell from its original rock mass. In order words, fractured rocks take a greater volume than its natural rock mass. The swell factor (SF) refers to the percentage of volume increased after the original rock is being fractured or fragmented:

$$SF = \frac{V_f - V_o}{V_o} \tag{1}$$

Where SF = the Swell Factor,%; V_f = Volume of fragmented rock volume), m^3; V_o = Original volume rock, m^3.

From this definition, the nature of swell factor can be related to actual applications in different fields—ranging from muck pile estimation to bulk material handling. In blast design, a question is often asked: what is swell factor of a muck pile? To answer this question properly and mathematically, a simplified model has been considered: Taking a cubical volume of 20 m × 20 m × 20 m, fill up this volume with balls of equal sizes. The volume of air gaps between the balls represents the void due to rock swell. The following graph in Fig. 1 explains the effect of number of balls in this fixed cubical volume on the Swell Factor, SF%. As the volume is filled up with smaller balls, the number of balls tends to be greater and the swell factor decreases but stabilizes just above 24%. To be exact, with balls of 20 cm diameter, the SF is 24.87% and if the balls are 5 mm diameter, the SF is 24.36%. This explains why muck piles on surface are typically measured at 25% swell.

In the real world, rock fracture or fragmentation causes different scales of rock swell. And the Swell Factor differs with different shapes of fragments, size distribution and the size of container in which it is measured.

From fractured rock to fragmented rock pile, the Rock Swell Factor can be roughly divided into the following 5 categories, as listed in Table 1.

Obviously, Category IV (SF = 25% − 35%) is where blast designs are aiming for. Rock swell calibration conducted at a large zinc-lead mine (Mine-B) yielded 33% (or 1.33 as total volume of rock swell). Thus in general, 35% SF can be applied to friable sulphide zinc ores, whereas 30% SF can be used for nickel, copper and gold ores.

In deep mines where stresses are high, it is sometimes required to design blasts with available voids

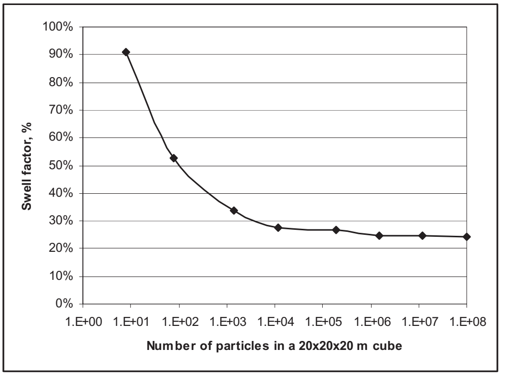

Figure 1. The effect of particle sizes on swell factor.

Table 1. Five categories of rock swell factor.

Category	Swell factor, SF =	Nature of swell	Comments and descriptions
I	1%–5%	Fractured Rock	Rock integrity is lost but rock mass may still be stable if gravity is not pulling it apart.
II	5%–15%	Fragmented rock but the muck pile is "frozen"	The broken rock mass is "frozen" and the muck pile cannot flow freely. De-stress blasting should aim for this rock swell to release the ground stress but the fragmented rock remains in situ.
III	15%–25%	Fragmented rock pile with mucking difficulties	Fragmented rock can be in free-flow state. Initial mucking is possible but the muck pile is packed to a certain degree. Some special efforts are often required such as using "rocket launchers", power-cones, or QuickDraw devices, or water-canon or water-gun to help flushing the muck pile.
IV	25%–35%	Fragmented rock in natural reposing state or free-running state	Fragmented rock is in full free-flow state. The variation of SF in this category depends on the type of ore, degree of fragmentation and the stope width to height ratio. Within this range, higher SF values are often found with very friable massive sulphide ores, more uniformly fragmented rocks and smaller stope width-to-height ratios (narrower and higher stopes).
V	35%–50%	Fragmented rock filled in ore-bins or trucks	Calculation of ore volume in scoops, trucks, ore-bins or ore-passes often needs to use higher SF values. In general, coarser muck in smaller containers tends to have greater SF values and vice versa. This phenomenon is often caused by the "edging effect" where coarser rocks at the edge or corners block the flow of finer materials to fill up the void.

falling into Category III. In this category, the muck pile may flow reasonably well where the stope width to height ratio is high, the rock is well fragmented and the muck pile does not have any oxidation problems. Certain practices at a deep gold mine (Mine-L) are under this category.

Further reduction in void availability may result in "frozen" or "jammed" blast, falling in Category II. In one case at Mine-B, a test stope of 40,880 tonnes was approximately 80% frozen with just 10% void available at the under-cut. The stope dimension was 30 m high, 12 m wide and 20 m long. Although there was 13% void available at the over-cut, it was only partially used.

3 VOID UTILIZATION RATIO

The concept of Void Utilization Ratio is a description of how much void is being filled with fragmented rock in the blast design stage. It is defined as following:

$$R_v = \frac{V_o(1+SF)}{V_o+V_v} \qquad (2)$$

Where R_v = Void Utilization Ratio, %; SF, = Swell Factor, %; V_o = Volume of original rock to be blasted, m^3; V_v = Volume of void available to allow rock to swell, m^3.

There are three important aspects to be considered when applying this equation in blast design:

- Choosing a proper Swell Factor that is suited for the specific mine site, based on either calibration with cavity surveys (measured by CMS—cavity monitoring system) before and after a mass blast, or with consideration of past experiences.
- Control the Void Utilization Ratio (R_v) within a practical range:
  - $R_v < 60\%$ is probably under-utilizing the existing void, unless it is the final blast or side-slash blast.
  - $R_v = 80\%$ is the proper target for most toe-slash blasts such as the first and second blast in a stope, or slot blast.
  - Final mass blast can get close to 100% for R_v or slightly over 100% if needed.
- Calculation of the void:
  - All the volume at the under-cut drift;

- Muck spread at the brow: a typical muck pile has a 45° muck-slope angle. However, if the blast does not break-through to the over-cut, the spread of muck can reach 25° to 30°, in which case more void can be included in the calculation.
- The void in the over-cut drift can only be partially used, typically 25% to 50% of its volume. The reason is simple: the muck only flows down, not up. However, with fast initiation, it is possible to utilize part of the void above the over-cut floor during the initiation process.

It should be mentioned that every mine site may have a different way of expressing the void ratio for blast design. For example, the following equation is used by the deep underground gold mine (Mine-L)

$$V = \frac{V_v}{V_o + V_v} 100 \tag{3}$$

Where V = void ratio, %; V_v = volume of void available to allow rock to swell, m^3; V_o = volume of original rock to be blasted, m^3.

In this calculation, the voids created in the slot raise and the over-cut space are not included. The typical Swell Factor used at Mine-L is about 21% to 22%.

Finally, it is important to ensure that the void calculated is actually available by the time of blasting. A good practice is to have a site visit prior to the design stage and verify again by the blasting supervisor during or after charging the explosives into blastholes prior to the initiation of the blast.

4 BLAST DESIGN CONSIDERATIONS

Blast design is an engineering process about two fundamental aspects:

a. Blast pattern for the geometric positioning of blastholes in the rock mass: Important considerations include: blasthole diameter, burden and spacing.
b. The length of explosive charges and the initiation timing: the weight of explosive charge per specified volume (often referred as the Powder Factor) is to achieve the desired degree of fragmentation. The initiation sequence and delay timing are designed to achieve sufficient relief void for subsequent blastholes to break into. In underground blasting, the delay timing should first allow muck flushing vertically in the slot and then allow muck movement horizontally for burden relief between regular rings to create a loosely packed muck pile.

For the blast pattern, the burden is usually proportional to the blasthole diameter:

$$B = (15 - 30)\phi_H \tag{4}$$

Where B = burden of regular rings, m; ϕ_H = hole diameter, m.

The ratio range from 15 to 30 applies to different types of underground blast with different stope widths. The commonly practiced ratio is 26 for most underground base metal mines. In general, a greater ratio is suited for wider stopes for regular rock breakage while smaller ratios apply to narrower stopes or special blast that must ensure fine fragmentation and effective flushing of void being created. Once the burden is selected, the spacing is usually 0% to 20% greater than the burden with an average about 10%, as following:

$$S = (1.0 - 1.2)B \tag{5}$$

Where S = hole spacing in the same ring, m; B = burden of regular rings, m.

For ring blastholes in a stope, the blastholes are often "fanned" out from the drilling drift (which is typically 4 to 6 m wide) to the under-cut level within the stope boundary which is often around 15 to 20 m wide. When charging such "fanned" holes, the collars (top of uncharged hole) are usually staggered resulting in the charged hole spacing at the collar is 60% to 70% of the toe spacing.

Stemming is also important in any blast. The purpose is to confine the explosives energy inside the blastholes to break the rock rather than being released to the open air. Therefore the theoretical stemming length should be at least the same as the Burden. In underground mining, stemming materials are often transported underground. With the limited supply of the materials, stemming should be at least 1 m in length, usually 1.0 to 1.5 m for 100 mm diameter holes. It is however important to choose the proper material for stemming. In general, crushed rock (sized to <1/6 of hole diameter) is the best material because this size range can provide a bridging effect when being pushed by detonation gases but it can still free-flow when poured into the blastholes.

For the delay timing in the initiation of a blast, the delay time is aimed at giving sufficient relief between the rings. Usually, the delay time is about 20–30 ms per m of burden. Between blastholes in the same ring, the delay time can be much shorter, usually 5–10 ms per m of Spacing.

For the blast pattern in the slot or drop raise, the dimensions depend on the availability of drilling equipment for blastholes as well as reamed holes or raise boreholes to provide the initial void.

In addition, local experience with the workforce is another important factor in the success of blasting in the drop raise or slot. Usually, every mine has its own comfortable pattern to work with for satisfactory length of blast in the slot. However, basic understanding of the blasting physics certainly helps extend the comfortable zone at a mine site in order to achieve more aggressive production targets. In this regard, the deep and stressed underground base-metal mines in eastern Canada have all tested different designs and extended the previous limits in terms of applying the mass blast principle.

Delay design also has two distinguishingly different requirements:

- it must flush out or clean out the slot to make room for subsequent rings to blast into. In other words, if the slot is "frozen" the whole stope is in trouble; and
- the speed of firing with regular rings to avoid jamming of muck and choking at the toe or brow.

With the experience accumulated within the Xstrata group mines, a drop raise delay timing calculator has been made and applied in the past 10 years. Table 2 shows an example of the program. It can be seen that the delay timing depends on

- The muck flushing speed (usually 30 m/s for only downward-moving muck, 20 m/s for slots open to both under-cut and over-cut);
- Length of raise to be blasted and
- The quantity of muck generated by each specific blasthole.
- Design criteria is to keep the slot about 80% full of broken muck whenever a blasthole is fired.

With regular rings, the delay timing has two limits:

- The lower limit is to prevent muck from jamming between rings: about 15–20 ms per meter of burden, as mentioned earlier.
- The upper limit is to prevent choking at the toe or brow. In this case, gravity plays an important role. Depending on the number of rings to be blasted, it can range from 200 ms/ring to 500 ms/ring, relatively faster initiation for more rings and slower for fewer rings.
- Delay between holes varies between 0 ms and 50 ms per hole depending on the quantity of explosives charged in each hole. For fully loaded holes, the delay is typically 25 ms to 50 ms in order to effectively separate the vibration traces for con-

Table 2. Calculation of delay timing in drop raise or slot blasting.

General Specs:	Length of raise to blast	**24.9 m**	Number of holes:	**1**		Muck movement velocity
	Initial void hole diameter:	**0.762 m**	Initial void area:	0.46	m^2	27.43 m/s

Design calculations:

Firing sequence	Hole #	Top area, (m^2)	Bottom area, (m^2)	Average area of rock, (m^2)	Rock vol, (m^3)	Muck vol, (m^3)	Muck gone, (m^3)	Muck left, (m^3)	Total muck, (m^3)	Void available, (m^3)	**INPUT** Next delay, (ms)	**Design**: Delay time, (ms)	Void utilization ratio%	% of old muck
1	**H2**	**0.00**	**0.13**	0.06	1.56	2.34	0.0	0.0	2.3	11.4	190	10	18%	
2	**H5**	**0.21**	**0.41**	0.31	7.67	11.51	0.5	1.8	13.4	19.4	300	200	69%	79%
3	**R2B**	**0.21**	**0.00**	0.11	2.62	3.93	4.4	8.9	12.9	17.3	500	500	75%	67%
4	**H3**	**0.68**	**0.42**	0.55	13.66	20.49	7.1	5.8	26.3	33.0	500	1000	80%	45%
5	**H7**	**0.98**	**0.53**	0.75	18.71	28.06	14.5	11.8	39.9	47.7	400	1500	84%	45%
6	**H4**	**0.18**	**0.98**	0.58	14.47	21.70	17.5	22.3	44.0	55.2	300	1900	80%	56%
7	**R3B**	**0.60**	**0.48**	0.54	13.47	20.20	14.5	29.5	49.7	63.9	350	2200	78%	67%
8	**H6**	**1.26**	**0.98**	1.12	27.94	41.91	19.1	30.6	72.5	91.1	350	2550	80%	62%
9	**H1**	**1.95**	**1.26**	1.61	40.04	60.05	27.9	44.6	104.6	121.8	300	2900	86%	62%
10	**R2**	**1.00**	**0.00**	0.50	12.47	18.71	34.5	70.1	88.8	117.2	200	3200	76%	67%
12	**R3**	**1.50**	**0.90**	1.20	29.93	44.90	19.5	69.3	114.2	147.7	200	3400	77%	78%
13	**R3 A**	**1.00**	**0.70**	0.85	21.20	31.80	25.1	89.1	120.9	155.7	200	3600	78%	78%
	Sum	10.81	7.18											

trolling blast damage to surrounding rock mass. In the case of controlling blast damage in paste-fill, the delay time should be increased to over 75 ms between holes due to the low frequency and low seismic velocity of the paste-fill properties.

The first blast in a stope usually includes the slot or drop raise, plus a few ring holes for toe slashing to expand the void at the under-cut. For the ground stability reasons, the blasted void is usually shaped like a dome by designing the collar of different blastholes. Again, the quantity of ore to be blasted is to maximize the use of voids available. Then the void must be available by the time of blasting—for example the stope should mostly be mucked out prior to blasting.

5 ELECTRONIC DETONATORS

Electronic detonator is perhaps the revolutionary product being applied in rock blasting in the past 15 years. In July 2001, a large scale destress blast in the West Regional Pillar at the Brunswick Mine was successfully carried with over 1500 electronic detonator used to fire over 232,000 kg of bulk emulsion explosives (Liu et al. 2005). Still, the full potential of electronic detonators will be furthered explored by blasting professionals and it will surely take greater share of the blast initiation market. An important part of designing a mass blast is about the initiation of all the charged blastholes.

Electronic detonators, introduced to the Canadian mining business in the past 14 years, provided a reliable means to achieve the delay timing for mass blasts. In general, a mass blast includes the slot area (drop raise or inverse drop raise) and some regular rings. To open up the slot area, it takes a longer time, typically a few seconds depending on the length of the raise. However, once it gets to the part of ring slashing blast, the initiation speed can be much faster in order to achieve better fragmentation. This initiation requirement, namely a slow initiation at the beginning followed by a faster and more precise timing, is exactly opposite to what pyrotechnic detonators can offer. Pyrotechnic detonators are generally more precise for short delay and less accurate for long delay periods. Electronic detonators are very accurate regardless the delay time for the purposes of muck movement consideration. This is why electronic detonators are ideally suited for materializing the mass blast principle. It has been proven that electronic detonator is an important tool for underground open stoping blasting.

At most large underground sulphide mines, production blasts must be initiated from surface by a Central Blasting System. The electronic detonator system therefore must accommodate this requirement. Currently, most of the deep underground mines in Xstrata Canada have implemented the central blasting initiation system with electronic detonators with a few exceptions where telephone lines are used to initiate the logged-and-timed electronic detonators underground.

6 THE EXPERIENCE AT MINE-B

From 1999 to 2001 when Mine-B of Xstrata Canada (then Noranda Mines) implemented the pyramid pillarless stoping sequence to cope with high stresses, stopes were getting smaller and sequencing became more strict. In order to maintain the production level of 10,000 t/day, blast cycle time must be reduced. Otherwise, more stopes had to be active to keep the production level which would increase the production cost, particularly for the development. A Six-Sigma project showed results that shorter blast cycle time can be achieved by better utilization of the voids available in each blast, see Figure 2 (Liu et al. 2001). Using the mass blast principle, the number of blast per stope was reduced from 7 or 8 blasts to 3 or 4 blasts in for average stopes.

In an effort to further apply this principle, Mine-B tried two stopes with one-shot mass blast in 233/4-4 stope for 40,000 tonnes and 513-1 stope for 24,000 tonnes. Figure 3 shows the designed timing delay in the one-shot mass blast in 513-1 stope. This stope was 24 m high, 28 m long and 9 m wide. To ensure the success of this blast, there were three 1.8 m (60″) raise boreholes drilled to serve as initial voids, thus there were two blast centers—one with 2 raise boreholes and another one just with 1 borehole. Roughly the slot raise took about 1.8 seconds to complete and the whole stope took 3.2 seconds. Relatively this firing speed is quite fast but it was designed to flush broken muck into under-cut drifts and it was successful.

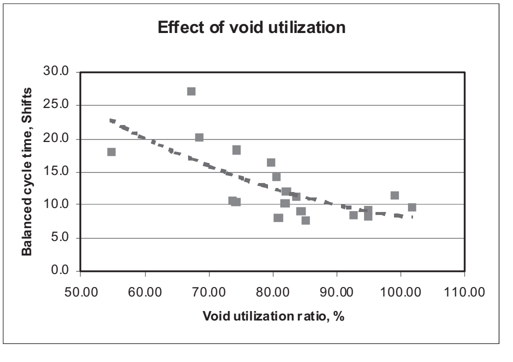

Figure 2. The effect of void utilization ratio on blasting cycle time.

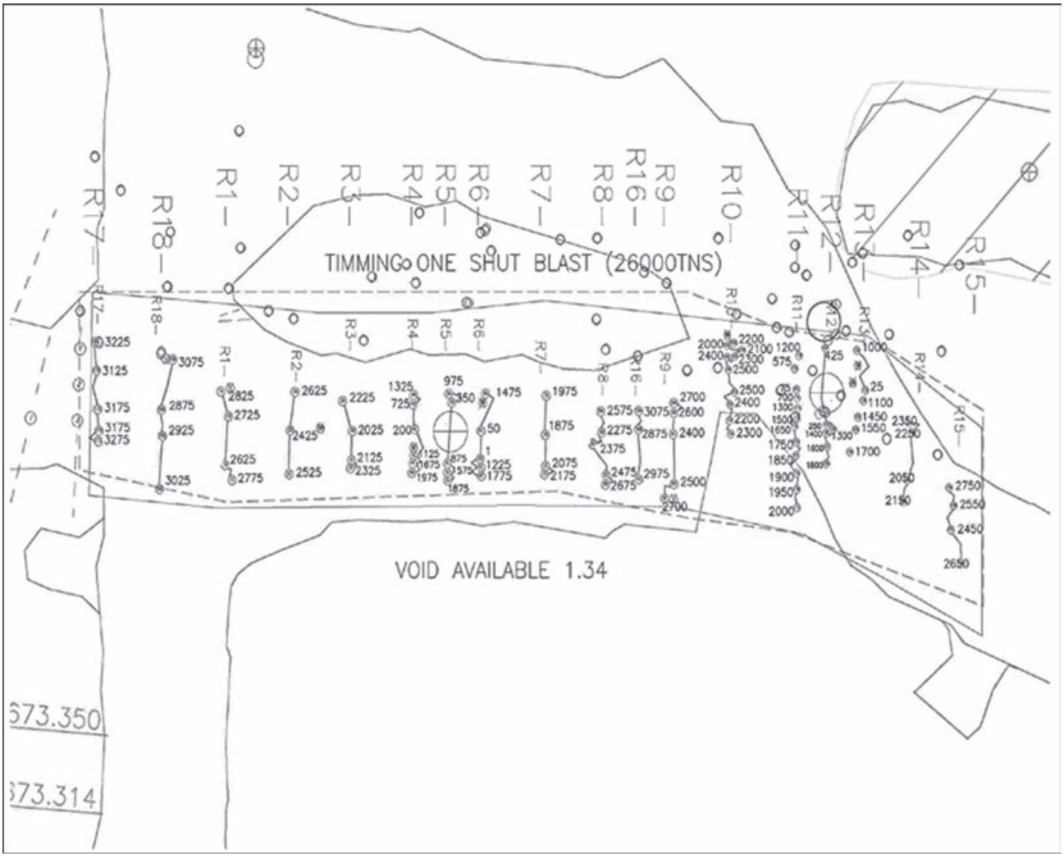

Figure 3. Delay timing of the one-shot stope in 513-1.

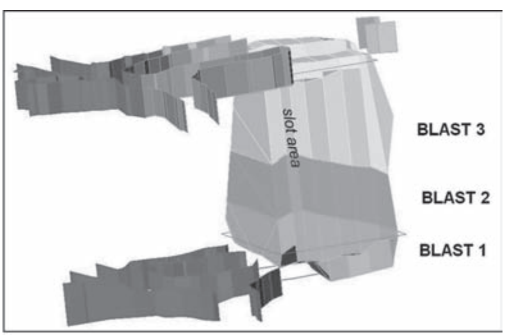

Figure 4. Typical blasting sequence in a stope at Mine-B.

Normally when there is insufficient void to make a one-shot stope, typically 2 or 3 blasts are required to blast a whole stope depending on the availability of voids. Fig. 4 shows a typical stope blast sequence in 3 blasts. At Mine-B, blastholes are all 114 mm (4.5 inches) diameter. For down-holes, a raise bore of 0.71 m (28″) is used as an initial void. For up-hole stopes of 15 m high, an inverse drop raise pattern is used, see Fig. 5.

7 THE EXPERIENCE AT MINE-K

From 2004, Mine-K of Xstrata (then Falconbridge) also implemented a Six-Sigma project aimed at reducing the Blast Cycle time. It was identified that the Blast Cycle time is a function of the number of blast per stope and the tonnage in each blast, which are then further attributed to (1) void utilization ratio and (2) the clear blast design issued by engineers as a production guideline (in form of Blast Letter) for every blast. To maximize the use of void was the key to make blasts bigger and thus it takes less number of blasts to complete a stope. Significant improvement of productivity was achieved as result of this Six Sigma project. Typical stopes take about 3 to 4 blasts to complete, as compared to 8

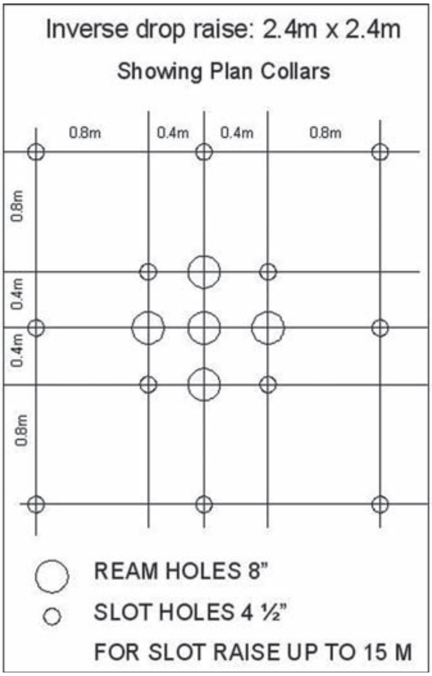

Figure 5. Inverse drop raise pattern used at Mine-B.

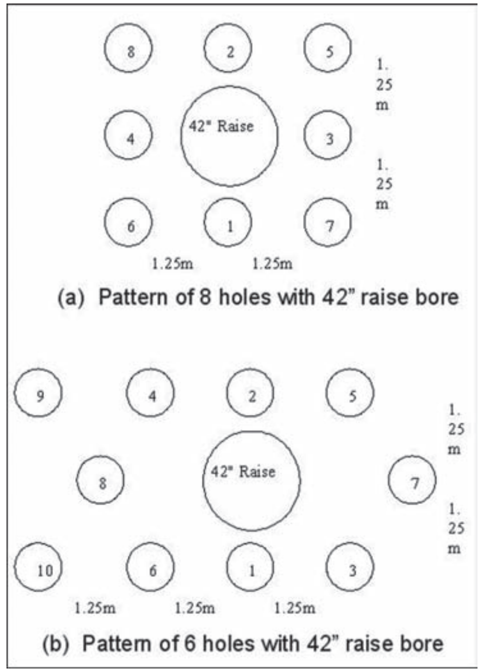

Figure 6. Different drop raise patterns.

or 9 blasts before. In its process-control plan, the following 3 aspects are included:

- All blasts require a blast letter.
- With consideration of mucking capacity in high sulphide and final blast, every blast uses at least 60% of the void available with a target of 80%.
- Database is maintained monthly to ensure project guidelines are being followed.

Further to the Six-Sigma initiative, a Best Practice manual was made, which includes 4 different drill-hole patterns in 8 different mining and blasting scenarios. Each scenario has a different drill and blast pattern, as well as the delay timing in the slot or drop raise. Figure 6 shows two typical drop raise patterns: 8-hole and 6-hole patterns with 42″ raise borehole. For different raise borehole sizes, 42″ or 28″, the basic designs of blastholes are the same.

In order to initiate bigger blasts in each stope, Mine-K tested i-kon™ CBS system of Orica. After over one year of testing and a vigorous Pre-Development Review (PDR), the system was then fully implemented. As a general practice, all production blasts are fired from surface via a telephone line. Application of such new technologies and the new mass blast principle were among the key factors for maintaining the production level at Mine-K when most of the mining activities are reaching over 3000 m below surface.

8 THE EXPERIENCE AT MINE-L

Mine-L is a business partner of Xstrata Canada and an underground gold producer with mining at over 3000 m depth. At Mine-L, the effort of using mass blast principle in stope blasting dates back to 10 years ago. Blasting improvement gradually evolved from 4–5 blasts per stope to just one-shot per stope. Trials took place in the Block 215 pyramid, an area with primary/secondary pyramid mining sequence at deeper part of the mine. Initially, each stope was blasted 4 to 5 times. Problems encountered include:

- Holes to clean after each blast until the final blast;
- Caving in the slot;
- Unable to blast all holes;
- Large blocks in the stope
- Back caving after the mass blast

Obviously, all these problems cannot be addressed by drill and blast design alone. Stope design and ground support modifications were made at the same time, including the following changes:

- Reduced the size of the opening.
- Increased the floor pillar from 10.8 m (35′) to 12.3 m (40′) in the first trial period.

- Then increased the floor pillar to 18.2 m (65') in the second trial period.
- Installed cables around the slot raise.
- Reduced the size of stopes, width changed from 15 m (49′) to 13 m (43′) for primary stopes.

On the drill and blast side, aggressive changes have been made in the following 5 stages:

- Stage-1: Mined 6 stopes with 4 to 5 blasts.
- Stage-2: Mined 22 stopes with 4 blasts;
- Stage-3: Mined 26 stopes with 3 blasts;
- Stage-4: Mined 351stopes with 2 blasts;
- Stage-5: Mined 81stopes with Single-Shot blast, represent 30% of the stopes.

With the single-shot stopes, the average void for the single shot blast is around 12% (min. used 5%). Figure 7 shows a typical ring in the 221-20-43 Stope which had 28,000 tonnes of ore and it was blasted with bulk emulsion explosives and initiated with the i-kon™ electronic detonators. The stope dimension was 20 m long, 12 m wide and 30 m high. There are 8 rings along the stope length. Regular rings were distanced at 3 m (burden) while the ring on the centre of drop raise had 2.5 m to each side. The spacing in each ring was 3 m at the toe. The hole length was 25 m at the centre with B/T (break-through) and 27 m at the sides—leaving a slope on each side for easy flow of fragmented rock pile, as shown in Figure 7. Figure 8 is the firing sequence of this stope.

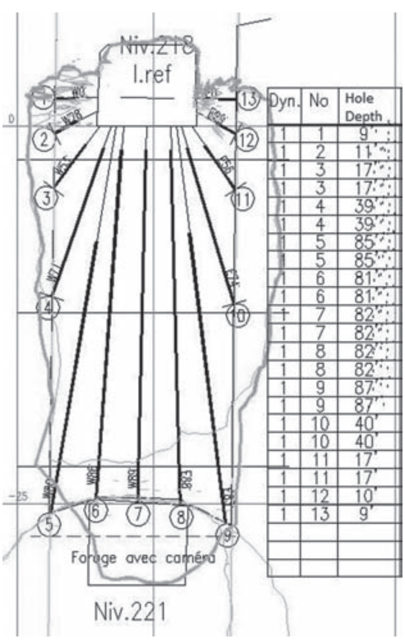

Figure 7. A typical ring in Stope 221-20-43 at Mine-L.

In general, Mine-L has been practicing the mass blast principle in the past 8 years with most stopes mined out in 1 or 2 blasts. With such changes, there has been reduced caving problems, improved drilling cycle time, improved blasting mobilization cycle, reduced number of rehab due to blast damage, reduced risks of drilling crews and blasting crews working in the blasted stopes, improved tonnage availability per stope and mucking productivity due to reduced usage of remote mucking.

Mine-L uses ANFO mostly for the production blast. However, bulk emulsion is also used in wet ground conditions and large stopes in order to reduce the footage of drilling required. The drill-hole pattern varies with different hole sizes and type of explosives:

- For ANFO with 100 mm dia. holes: burden = 2.45 m; spacing = 3.00 m
- For emulsion with 100 mm dia. holes or ANFO with 114 mm dia. holes: burden = 2.85 m, spacing = 3.75 m.
- For emulsion with 114 mm dia. holes: burden = 3.3 m, spacing = 3.75 m.

As for mucking, one-shot stopes are mucked out as fast as possible to reduce the exposure time of an open stope. For 2-blast stopes, fragmented ore from the first blast is mucked out just before the second or final blast in order to prevent side-wall sloughing within a short period of time, typically within 2 or 3 days. In such cases, frequent visits, ideally daily, are required by geologist or production supervisors.

9 CONCLUSIONS

The mass blast principle described in this paper applies to sub-level open stoping mining method. The 3 mining operations given in this paper as examples are among the largest underground operations in eastern Canada, mining at depths from 2000 m to 3000 m below surface. Under such depths, the stress in the rock mass causes many kinds of problems, from ground heat to ground control to material handling, as well as to drilling and blasting. Among the comprehensive measures to keep mining economical at greater depths, the mass blasting principle, as proven in existing operations, is an effective method to cope with highly stressed ground conditions. By better utilization the void available to each blast, the size of blast can be maximized. As a result, the number of blast in each stope can be minimized, or even reduced to just one blast per stope. The application of electronic detonators in the underground environment has been one of the key technology advances in realizing the designed blasts. The practice of this

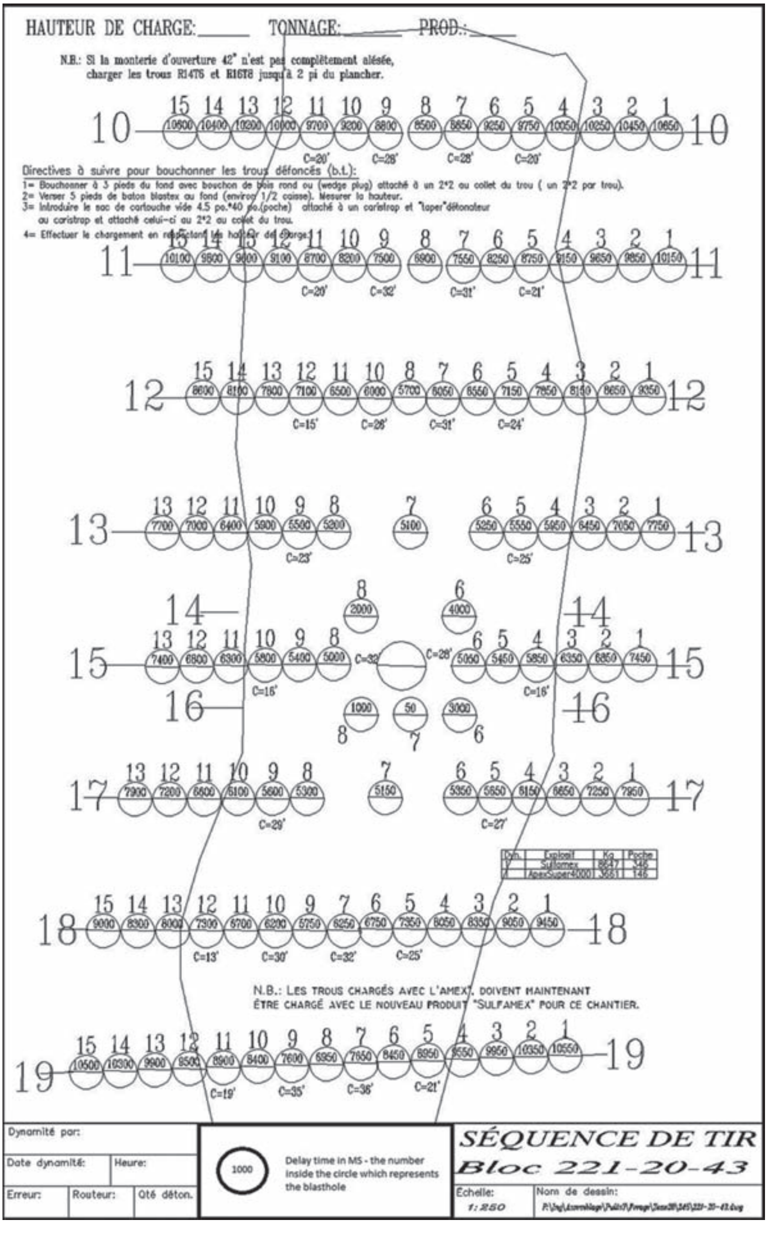

Figure 8. The delay timing sequence of Stope 221-20-43 which is a typical one-shot stope at Mine-L.

methodology has seen significant results in reducing the cycle time of drill and blast and improving the safety of employees working in stopes.

REFERENCES

Liu, Q. & Ellis, B. 2001 "Improvement of Blasting Productivity at Brunswick Mine" *Proceedings of Explo2001*, Hunter Valley, NSW, Australia Oct. 28–31, 2001

Liu, Q., Ellis, B. & Chung, S. 2005. Application of advanced blasting technologies for large scale de-stress blasts at Brunswick Mine, *CIM Bulletin, June/July 2005*, Montreal, Canada.

Liu, Q., Larouche, P., Tolgyesi, S.D. & Roberge, S. 2007. "Application of mass-blast principle for stope blasting in deep mines", Chapter 7, *Challenges in deep and high stress mining,* Y. Potvin, T.R. Tracey and J. Hadjigeorgiou, 2007 Australian Centre for Geomechanics, Perth, Australia.

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Improving pit wall stability by minimizing blast damage vis a vis rock characterization at RAM

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ABSTRACT: Stability of wall slopes is the basic in-gradient for carrying out economic and safe extraction of mineral at any open pit mine. At Rampura Agucha Mine open pit (India) this assumes a greater emphasis as its footwall is achieving its final configuration while hangwall is on dynamic stage. During blasting, back break was the main concern as despite of carrying out pre-splitting, the benches experienced substantial back break adversely affecting the stability. The back break damages the remaining rock and reduces rock mass strength too.

For addressing these issues, an experiment involving 22 blasts was conducted to identify and understand factors causing rock damage. The investigations consisted of collection of ground characteristics (comprehensively termed as Rock Mass Rating) for each test blast site and the blast results including extent of the back break, fractured zone, their orientation, etc were recorded. Based on these observations, a number of measures like modification in firing front orientation, rationale explosive energy concentration vis a vis rock characterization have been initiated commensurate with rock characteristics. It has led to significant improvement in stability of the pit as authenticated by safe standing of the footwall for 25 benches. These measures have instilled confidence to the mine management that the pit will provide sustainable business targeted production till its designed life.

1 INTRODUCTION

Rampura Agucha Mine (RAM), owned by Hindustan Zinc Ltd (HZL)., Vedanta Resources Plc, is a lead zinc open pit mine producing 5.95 mtpa and recently has started its underground part which is slated to produce 4.5 mtpa in near future. It is situated 225 km south-west of Jaipur, in the district of Bhilwara, Rajasthan

RAM works with shovel—dumper combination deploying large sized HEMM like 34 m^3 shovels and 240 tonne trucks navigation with Truck Dispatch System (TDS). The excavation is carried out by a drill fleet of 165 mm Production drill machines and 115 mm diameter drills for trim and pre-splitting purpose. The footwall has reached to its final configuration while its hangwall is under four staged cut-backs.

Normal production blast consists of firing 6–8 numbers of rows involving 140–190 holes. The charge in each hole consists of charging 160–180 kg of explosive with 3–4 m stemming column.

Pyrotechnic and electronic initiations are used for firing blasts. For improved blast performance, reduced ground vibration and noise level, non-electric millisecond down the line delay detonators (250 ms) and non-electric Trunk line detonators (Shock tube initiation) are used. On an average, 4 to 6 blasts are carried out daily in ore and waste.

Depending upon rock types and their condition, spacing and burden is being varied. Typical drill & blast design parameters are as under.

Spacing and Burden

Waste	:	5–6 m × 4 m
Ore	:	3 m × 3 m
Depth of Hole	:	11–12 m
Sub Grade	:	1.0–1.25 m
Primary Charge	:	400 g cast booster (Waste) 500 g emulsion booster (Ore)
Column Charge	:	Site Mixed Emulsion (Bulk)
Charge Length	:	66–70% of hole depth
Charge per hole	:	170 to 190 kg
Drill Pattern	:	Mostly staggered
Blast Pattern	:	Reverse Echelon
MCPD	:	500 kg.
Powder Factor	:	3.7 t/kg (Waste) 1.7 t/kg (Ore)
Holes per round:		60–200 holes

The mine is currently working up to a depth of 250 m from surface (Figure 1). Towing in the line of world class mines, its slopes are monitored by Slope stability Radar.

Figure 1. Rampura Agucha open pit showing various stages of excavation on hangwall.

To maintain the stability of the pit by reducing blast damage, it has initiated many measures in blasting practices e.g. pre-splitting, electronic initiation in geological weaker formations, blast design rational explosive charge concentration, orientation of firing front, etc. Many design combinations have been tried on footwall to contain back break with excellent pre-split results and significant improvement in stability of the pit.

2 GEOLOGY

Rampura Agucha is a stratiform, sediment-hosted Lead Zinc deposit and occurs in Pre-Cambrian Banded Gneissic Complex. It forms a part of Mangalwar complex of Bhilwara geological cycle of Archean age and comprising of magmatites, gneisses, graphite mica schist, pegmatite, amphibolites, and impure marble. The rocks have been subjected to polyphase deformations and high-grade metamorphism.

The deposit is a plunging isoclinal synform while the host rock occupies the core of the synform and plunge in southwestern limit is 65–70° due NE. The Rampura-Agucha mixed sulphide deposit is a massive lens shaped ore body with a NE-SW strike length of 1500 m and a width varying from a few meters to 120 m with an average of 58 m

The host rock for mineralization is Graphite-mica-Sillimanite Gneiss/Schist (GMS). Walls (Figure 2) are composed of Garnet Biotite Sillimanite Gniess (GBSG) and intrusions of Pegmatite and Amphibolite and Mylonite (on footwall only) while GBSG forms the major chunk amounting around 70–80% of the mass.

There are 3 major joint sets on footwall. Foliation is the most prevalent discontinuity 60–80°/N130° affecting the blast damage. Rocks at RAM

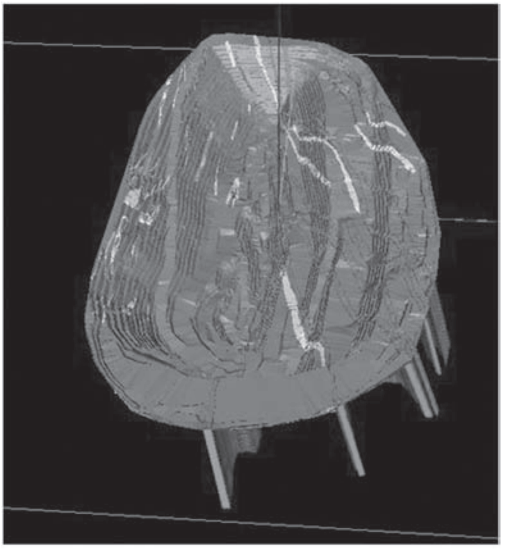

Figure 2. Litho-model of Rampura Agucha mines showing GBSG as golden, Amphibolite as green & Pegmatite as yellow bands.

Table 1. Physico-mechanical properties of the rocks.

Rock type	Uniaxial compressive strength (MPa)	Tensile strength (MPa)	Young's modulus (GPa)	Poisson's ratio
GBSG	33	6.5	10.4	0.14
Amphibolite	74	14	22	0.14
Pegmatite	71	6.2	12	0.05
GMS/Ore	57	9	20	0.12
GBG	43	9	12	0.13

Table 2. Trend of foliation on footwall of the pit (S8/S7 means area between S800 & 700 & 80/141 means joint is dipping at 80° due N 141°).

FW foliation						
S8S7	S7S6	S6S5	S5S4	S4S3	S3S2	S2S1
81/141	80/130	86/137	76/135	78/141	76/135	79/153
NS0N1	N1N2	N2N3	N3N4	N4N5	N5N6	N6N7
82/142	80/139	83/147	85/144	77/149	72/145	81/137

are moderately competent. In order to assess the behavior of these rocks at depth, four major geotechnical holes have been drilled stitching whole of the pit and its core were tested with results produced in Table 1.

Foliation is the main plane of discontinuity which governs the bench profile. The pit strike was divided in 100 m wide zones viz. N100-N200, N200-N300, S100-S200, etc (presented in Table 2). In each segment, joint/foliation surveys were conducted and kinematically analyzed on DIPS software. Prominent pattern, which emerged, is presented in Table 2.

3 INITIATIVES TAKEN FOR MINIMIZATION OF WALL DAMAGE

Blasting usually damages pit wall by shattering (seismic) energy and by heave (gaseous) energy. Damage is adverse inter action of blast energy and rock mass (and its characteristics). The latter varies spatially in the pit (Figure 2).

The 1.5 km long, 0.75 km wide and 250 m deep pit has spatially varying lithological settings (rock type- litho units, geological structures, etc). Under such circumstances, one general/uniform blast design cannot suffice for whole of the pit. Accordingly, a three dimensional litho-model (Figure 2) is generated for RAM. Bench on bench are geotechnically mapped for Rock Mass Rating (RMR) and major geological disturbances. All these attributes are superimposed on one mine model on DATAMINE model of the pit.

RAM adopted a holistic and rational approach while adopting various measures for reducing damage to the pit walls. Under this, blasts were investigated on one front while the pit was discretized into various domains based on rock characteristics i.e. Rock Mass Rating (RMR). Special weak zones (where RMR <40) were demarcated. Signature blasts (single hole firing) were captured, their vibration footprints were studied and natural frequency of rock mass, adjoining dwellings were determined taking assistance from Orica, CIMFR, etc. The study indicated that the natural frequency of the ground varies in range of 14–16 Hz. Based on, inter shot delay of 7–8 ms (or in their multiples) were chosen in firing pattern to avoid resonance and minimize damage to rock mass.

Various blast damage initiatives were adopted and engineered based on merit of the rocks i.e. RMR. Moreover, in geologically disturbed zones e.g. near shear/fault zones, the problem becomes crucial and causes sizeable back break, triggering of pre-existing faults.

3.1 *Re-shaping the firing front*

During staged excavation, normal cut-back width ranges from 100–150 m at RAM. In any cut back, last 20–22 m thick slice of bench is fired separately (called trim blast) with more care and design after firing main production blast. While firing trim blast, a row of pre-splitting holes (inclined at 60^0) is drilled at 1.2 m centre, followed by a row of vertical stab holes drilled at 3.5 m from presplit holes. Then at 3.3 m from stab holes, another row of buffer holes (115 mm) is drilled. The safe stand off of 1 m is kept at toe of buffer holes. Then at 3 m burden, production holes of 165 mm are drilled at 5 mx4 m pattern as shown in Figure.3. Normally, 3–4 rows of production holes are fired making a firing width of 22 m as trim shot.

The trim blasts on footwall were fired with rows initiated parallel to strike of the pit following very flat 'V' pattern (Figure 6).The drilling is carried out at staggered pattern. Normally, firing pattern consists of firing these holes in form of rows parallel to strike of the pit. Firing with this pattern resulted in back break beyond pre-split line and endangered the stability.

To understand the back break mechanism, 22 blasts at various location of footwall were chosen for detailed investigation (Table 3, Figure 5) for a period of 5 months (ranging from November 2009 to March 2010).

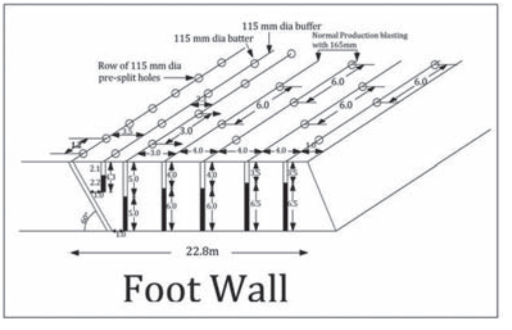

Figure 3. Trim blast design at footwall of the pit.

Figure 4. Failure pattern showing opening of foliation planes.

Table 3. Back break details of test blasts of footwall.

Blast	ID	N/S	E/W	Back break (m)
Nov-09				
366/354	1	N626-N502	W99-W46	4.8, 4.4, 4.2, 3.7, 4.3
320/310	2	S437-S339	W98-W66	3.2, 6.4, 4.4, 5.5, 6.8
Dec-09				
330/320	3	S173-S295	W149-W129	5.1, 5.5, 5.2, 2.3
330/320	4	S353-S491	W136-W118	6.2, 7.9, 6.8, 4.7, 8.1, 8.6
330/320	5	S493-S602	W137-W118	7.8, 5.7, 2.3, 3.0 4.2
330/320	6	S612-S688	W125-W64	3.4, 4.3, 6.0, 4.7, 6.5
330/320	7	S502-S664	W128-W69	4.2, 4.1, 4.4, 6.9, 3.1, 9.0
Jan-10				
330/320	8	S425-S520	W68-W104	5.1, 2.0, 2.1, 2.4, 9.3
330/320	9	S543-S602	W109-W76	5.7, 6.1, 2.8, 5.5, 4.1
342/330	10	S685-S720	W83-W19	1.4, 3.2, 3.1, 2.7, 3.1
342/330	11	S632-S686	W129-W84	6.0, 3.6, 6.6, 6.6, 2.8
342/330	12	S559-S633	W157-W125	3.5, 3.0, 3.7, 4.0, 6.5, 4.2
342/330	13	N713-N502	W2-W85	2.7, 4.6, 7.8, 5.7, 4.5
Feb-10				
342/330	14	N612-N707	W40-W70	3.7, 6.3, 8.5, 8.6, 9.3
340/330	15	N130-N245	W135-W167	2.6, 3.6, 1.3, 3.2, 2.1
300/290	16	S682-S742	E2-E80	3.6, 3.3, 4.9, 4.6, 4.5
290/280	17	S345-S570	W13-W40	4.4, 3.6, 1.6, 2.3, 1.8, 4.5
330/320	18	S120-N103	W111-W72	11.2, 17.3, 6.6, 9.4, 10.0
320/310	19	N128-N273	W70-W51	5.6, 4.7, 4.8, 5.3, 3.1
320/310	20	N363-N638	W63-EW00	9.2, 8.8, 10.7, 13.3, 7.8, 3.7
Mar-10				
310/300	21	N289-N485	W53-W12	9.6, 5.7, 9.2, 8.0, 5.3
280/270	22	S558-S435	W16-E2	5.4, 6.6, 6.8, 6.9, 5.3

It was observed that in spite of firing pre-split holes prior to main blast, back break used to engulf 2 to 17 m behind the pre-split line (Table 4) during the trim blast. It has followed a typical pattern (Figure 7). The geological planes oriented parallel to pit wall N-130° gets opened up leading to massive back break.

It was attributed to firing of the rows parallel to N 130° (the direction of major foliation, Figure 6) causing wave reinforcement due to multiple reflections from foliation planes ultimately leading to a patterned back break failure.

To avoid the patterned back break, remedial measures consisted of changing the direction of advance of blast moving front- i.e. making it oblique to N130° foliation. Therefore, firing pattern was changed from rows parallel to pit strike to "v" and or echelon shape (Figure 7). Three trial blasts were taken on modified pattern at location no. 1 (Table 4) and on investigating them, it was found that the back break was reduced exposing the intact half barrel of pre-split holes. Since then, echelon or "V" shaped firing patterns (Figure 7) are used rather than a very flat 'V' pattern oriented parallel to the foliation.

3.2 *Modifying the explosive energy*

Last 20–22 m width (excluding 5–6 m of berm) of the cut-back (called trim blast) is fired with maximum care. Spatial disposition of the various litho units are presented in Litho model (Figure 2). As clear, GBSG forms the majority of the rock matrix followed by intrusions of Pegmatite and Amphibolite.

The rock characteristics are varying across length and depth of the pit. Prior to these test work, same explosive charge per hole was used in blast designs. It used to lead to sizeable back break in weaker formations and poor fragmentation, toe problems in hard ones. Many hit and trials were conducted in the field at the sites marked in the Figure 5. During each test, depending on the quantum of deterioration of the rock conditions (indexed as RMR),

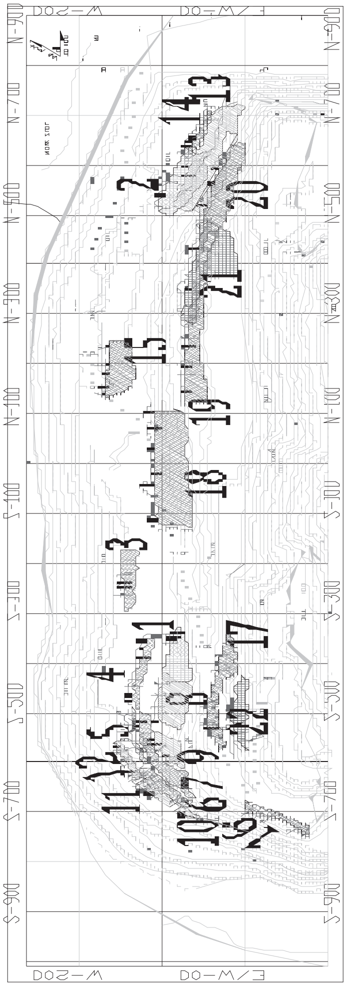

Figure 5. Mine plan showing locations of test blasts.

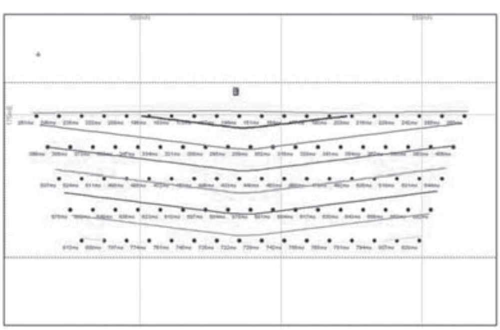

Figure 6. Initiation of rows parallel to the pit strike and foliation.

Table 4. Explosive charge concentration vis-a-vis RMR.

	Rock Characterization (RMR)	Rock condition	Explosive charge reduction
1	>45	Normal Rock, 40–50 MPa	Normal charge
2	30–45	Weak rock	10–15% reduction, use one bamboo
3	30–20	Very weak	15–20% lesser, two bamboos
4	Sheared zone	Powdered mass	No blasting, free dig
5	Pegmatite	Normal strength, 70–90 MPa	Normal charge
6	Amphibolite	Normal strength 90–120 MPa	Normal charge

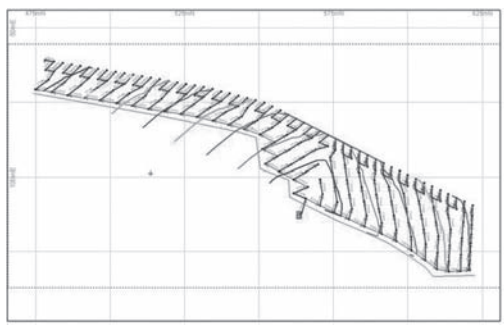

Figure 7. Echelon firing pattern, oblique to foliation orientation.

explosive charge per hole adjusted judiciously and the blast performance was analysed.

After these tests, following outcome was emerged which was used in the field with successful results. In general rock conditions where RMR is more than 45, the normal explosive charge column consists of 170–180 kg per hole. With deterioration of the rock conditions due to weathering, presence of geological discontinuities, characterized by RMR of 30–45, the explosive consumption is reduced by 10–15% of normal charge. It is accomplished by inserting one bamboo stick (having 25–30 mm diameter) continuously in the charge column of each hole. This simple yet effective technique of charge reduction has yielded much desired results as could be seen from visibility of half barrels of pre-splitting holes on benches (Figure 8).

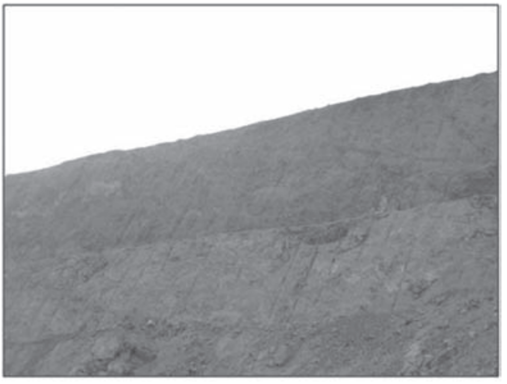

Figure 8. Presplit results showing half barrels on footwall of the pit.

With further deterioration of condition of the rock mass, characterized by RMR of 20–30, the explosive charge is proportionately reduced by 20% of the normal charge. This is again carried out by inserting two bamboo sticks (side by side making about 50 mm diameter) all along the charge column. In fact, each bamboo stick is about 3 m long and for increasing its length, another piece is tied to its upper end while lowering in the hole. This rule of thumb has been established by carrying out a number of blasts at site number (Figure 5).

Strength conditions of Pegmatite and Amphibolite do not vary much with depth and across the pit. Accordingly, normal charge is used for blasting these rocks. Likewise, in shear zones consisting of crushed rocks, blasting is avoided and area is free dug under tight supervision of Geotechnical engineer.

3.3 *Pre-splitting the pit walls*

Production schedule requires every blast to be as large as possible. However, size of the blast has to be optimized within the confines of production requirement and wall damage. Therefore, well proven technique of wall damage control i.e. pre-splitting has been applied at RAM. The theory of pre-splitting is that when shock waves from simultaneously detonating charges in adjoining blast holes collide, tension occurs in the rock, forming a crack in the web between the holes. Firouzadj et al. (2006) concluded that that small diameter holes in pre-splitting row such as 102 mm using decoupled charges is difficult but whereas continuous charging of the pre-split row in such hole yields better results. Among different approaches of continuous charging (decoupling, explosive mixing) decoupling due to its operation difficulties of small diameter holes (102 mm) was rejected by Jimeno et al. (1995).

Carrying out pre-splitting is a costly method and needs full attention specially for establishing its parameters viz. hole dip, diameter, hole spacing and specific charge concentration vis a vis rock characterization. The technique involves, drilling small diameter holes (115 mm) at a close spacing of 1.2 m along the limiting line at an angle of 60^0 to 80^0 as per the designed slopes. Chiappetta, (2001) suggested a good rule of thumb for hole spacing in feet be equal to the hole diameter in inches. Explosive diameter should be 1/2 to 1/3 of the hole diameter and load should be distributed all along the length of the hole except 2–3 m near the collar. These holes are charged lightly with 32 mm cartridges suspended in to holes and axially tied with a detonating fuse without stemming. A charge factor of 0.44 to 0.65 kg/m^2 is maintained depending upon the strata conditions.

These pre-splitting holes are fired instantaneously and are normally fired prior to the production and trim blast in front of limit line. Many trial blasts were taken with varying dip, hole spacing and charge (Singh et al., 2009).

After each trim blast of footwall, remaining rock profile of the berm was observed. The dip and dip direction of the prominent foliation governed the angle of the presplit holes for successful results.

With detailed experimentation, the technique has been now established for the footwall. The half barrels marks visible (Figure 8) on footwall bear testimony of its success. The technology has been established on footwall while it is under experimentation on hangwall. With adoption of the technique, the footwall is standing for a height of 250 m safely.

3.4 *Use of Electronic Delay Detonators (EDD) in weak zones*

The mine uses conventional pyrotechnic initiation (shock tube initiation like EXEL-DET, etc) and Electronic initiation. As is known, firing of pyrotechnic initiation suffers from its inherent shortcomings i.e. higher scatter in firing time, resultant out of sequence firing and inability to assign delay interval other than factory assembled. Firing with conventional pyrotechnic initiation systems witnessed large back break and resultant wall rock damage. The situation gets worsened in geologically weak zones having RMR <40 because of inherent problems of poor burden throw, leakage of gaseous energy through fractures, etc and leading to sizeable back break.

In order to avoid out of sequence firing and tailoring the firing sequence, proper delay gaps to control magnitude and frequency of the ground vibrations, electronic initiation is being used in such places. Near fault zones additional measures like smaller diameter holes (of 115 mm), use of additional row of buffer holes and pre-splitting at half the normal spacing are practiced with mixed results and are in process of fine tuning. After use of EDD, back break has been reduced with visible half barrels of holes on bench face.

The successful results of pre-splitting helped in modification of the blast strategy at RAM (a) to fire all trim blasts, (b) all blasts within 50–60 m from surface, and (c) all other places where RMR is lesser than 40 to be fired with EDD initiation only.

3.5 *Redesigning firing timings: Minimization of over-riding of rows*

During observations of blasted muck profile in trim blasts and wall rock conditions, it was found that there was not proper throw of the blasted muck. Instead there was heaving of latter rows. In such areas, the firing pattern, including inter hole and inter row delays were analyzed. It revealed that the inter row delay was kept constant as 42 or 65 ms, etc with no exponential increase to allow for over-riding in latter rows. It caused more back break beyond trim limits.

Now, after every row, inter row delay is increased exponentially viz. 42, 65, 100, 125 ms, etc. It has remarkably reducing over-riding effect, reduced heaving of muck without burden relief and ultimately back break.

4 CONCLUSIONS

Blasting is one of the major factor adversely affecting the stability. Rampura Agucha mines analysed 22 blasts in detail vis a vis rock characterization i.e. RMR. The journey consisted of:

- Discretization of the pit into different zones based on characterization of ground i.e. Rock Mass Rating (RMR),
- Defining use of mode of initiation (pyrotechnic or Electronic) in critical geotechnical zones,
- Modification in orientation of firing front, firing pattern and firing timings based on rock characteristics including orientation of the major joint/foliation set in trim blasts,
- Defining explosive energy/charging length commensurate with rock characterization,
- Special treatment for area ravaged by major geological structures like faults, shears, etc.

Adoption of these measures have resulted in improvement of stability. Presence of half barrels of presplit holes certifies the success of the measures.The pit walls are real time monitored by Slope Stability Radar (SSR) and authenticates the results of these new practices. It has instilled confidence in mine management that the pit would produce targeted mine production on sustainable basis reaching its designed depth of 372 m.

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REFERENCES

Chiappetta, R.F. 2001. The importance of pre-splitting and field controls to maintain stable high walls, eliminate coal damage and over break, *Proc. 10th High-tech Seminar on State of the Art Blasting Technology., Instrumentation and Explosives Application,* GI-48, Nashville, Tennesse, USA, July 22–26.

Firouzadj, A., Farsangi, M.A.E., Mansouri, H. & Esfahani, S.K. 2006. Application of controlled blasting (Pre-splitting) in Sarcheshmeh copper mine. *Proc. 8th Int. Symp. On Rock Fragmentation by Blasting,* Santiago, Chile, 7–11 May, pp. 383–387. Santiago:Editec.

Jimeno, C.L. Jimeno, E.L. & Carcedo, F.J.A. 1995. *Drilling and blasting of rocks:*252–271. Rotterdam: Balkema.

Singh, P.K., Roy, M.P., Joshi, A., Joshi, V.P., 2009. Controlled blasting (presplitting) at an open pit mine in India, *Proc. 9th Fragblast on Rock Fragmentation by Blasting,* Granada, Spain, 13–17th Sept 2009, pp 481–489.

Blasting in Mines – New Trends – Ghose & Joshi (Eds)

New physical findings revolutionize the drilling and blasting technology as well as the prediction of ground vibrations—Part 1: The new blasting model

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ABSTRACT (Part 1 and 2): Within the last two decades technical innovations led to an improved understanding of scientific relationships regarding optimal detonation, fragmentation and reduction of unwanted ground vibrations. This enables the blast engineer to design and taylor blast operations on a statistically sound basis and to assess the results in a physically objective manner. Results of the analysis of hundreds of open pit and underground blasts performed by the first author have unveiled sonicity (ratii between detonation velocity and the wave speeds) being single dominant parameter which controls the vibration problem in both, above ground and underground operations. Sonicity controls the relationship between detonation, fragmentation and vibration immission. For a constant amount of energy of a blast, a higher degree of fragmentation is accompanied with lower levels of vibrations. The new advanced blasting theory based on sonicity enables the blast engineer to design larger blasts without increasing unwanted vibrations while obtaining optimal fragmentation.

1 INTRODUCTION

During the last decade major improvements in the development and production of explosives were achieved, the design of novel and more sophisticated initiation techniques as well as innovative monitoring and measuring devices were developed. All this led the way to new insight into the mechanics and physics of blasting, particularly with respect to the efficient use of explosives in areas where the increase of fragmention and the reduction of vibration is required.

One of the most important tasks is the improvement of knowledge with respect to the sudden transformation of energy from the explosive into mechanical work in the form of optimal fragmentation and muckpile formation while keeping the vibrations in the near- and farfield at bay. These findings offer the blast analyst new tools for the evaluation of the blasting process on an accepted statistical basis. On the basis of this new knowledge the first author re-evaluated several hundred of the blasting works he performed over a period of more than 40 years. The re-evaluation of surface and underground operations was systematically assisted by a novel type of radar sensor, a device for measuring the detonation velocity, fiber-Bragg-grid based strain sensors, 3-component geophones and a 3-D-laser scanner [Mueller & Pippig 2011b]. The evaluation of the re-investigation of these blasting operations led to revolutionizing results and conclusions. Based on the validity of the sonic effect in blasting and explosive technology, the use of a fictitious detonation pressure and other important physical relations relevant to the blasting process yielded new principles for the dimensioning of blast operations.

The application of the new physically sound advanced blasting method enables the user to tailor and control the fragmentation of the rock mass and, thereby, effectively influence the level of vibrations. In most cases a reduction of blast vibrations is requested. Blast vibration immissions are directly related to the destructive effect of the explosive detonation. New ways to treat vibration immissions result from statistically based and proven relationships between momentum or energy liberated during the blast and the distance between blast site and site of concern as well as vibration velocity and dynamic strains. At the same time the new findings allow an almost unlimited expansion of the blasting operation in terms of number of blastholes, firing pattern, etc. without the risk of increase of vibration levels. An important fact is, that the number of blasts can be reduced and this results in an increasingly environmentally friendly blast operation.

In recent years the advanced drill and blasting technology has been enormously promoted by the development of high power drill machinery, new and safer explosives which can be transported to the blast site in special on-site-mix delivery trucks. In addition, improved and more reliable initation and firing techniques for large blast operations are now readily available. In contrast, the basics of traditional design and blast design calculations rely either on outdated assumptions of the 200 years old crater theory of detonation, or the blast operations are traditionally designed on the basis of empirical grounds, i.e. on experience and/or design rules which in many cases lack of a sound physical basis.

Commonly, the assessment of blast vibrations and induced dynamic strains varies according to the blast designers' knowledge or depends on the rules set by the various institutions. In most cases, the calculation of quantities (such as the charge of explosives per round and the various distance relationships which are or are not based on more or less confirmed statistical estimation methods) rests on statistically unconfirmed methods which are not universally applicable. These methods must be generally valid and should not be dependent on certain limiting restrictive conditions. The classical rules for the design of a blast operation neither take into account the properties of the explosive nor those of the rock mass. This drawback formed the motivation for establishing and developing a physically sound model for explosive blasting which incorporates the recently unveiled relationships between momentum, energy, sonicity, blastability and specific explosive consumption [Mueller 2001].

The Deutsche Bundesstiftung Umwelt (German Federal Institution on Environmental Issues) has been sponsoring the two research projects "Environmetally supportive advanved blasting technology" (Umweltfreundliche Sprengtechnik) and "Sonic Effects" (Sonische Wirkung). Many of the essential findings have been incorporated in this contribution [Mueller et al 2009, Mueller & Pippig 2011b, Mueller et al 2011c, Rossmanith et al 1998a, Rossmanith & N Kouzniak 2004 and Rossmanith & Mueller 2010].

2 THEORETICAL FOUNDATION OF WAVE PROPAGATION AND SONICITY

If a stone is thrown into a pond one will notice circularly crested waves which propagate from the point of impact of the stone on the surface of the water. While these waves expand with a (seemingly) constant velocity, their amplitude diminishes rapidly with radius travelled. Although water is an entirely different material than rock and rock supports a higher complexity of waves than water, this picture quite nicely illustrates and gives an idea of what occurs in a solid body, such as a mountain, a quarry, in the underground or just in a piece of reasonably coherent rock in the form of a specimen in the testing laboratory. In a fairly similar way elastic waves will be expanding from a source in a solid.

Regarding the shape of the source, the point source would be the simplest case, at least in theory. In practice there are no point sources but extended sources, the simplest being the spherical charge. Upon detonation of a spherical charge, a detonation wave expands from the surface of the rock/explosive interface, which gives rise to shock, plastic and elastic waves depending on the behavior of the material which supports wave propagation. If, for small and moderate amounts of explosives, the material behaviour is linear or non-linear elastic, these waves are termed elastic or non-linear elastic stress waves. If some plastic work is associated with the propagation of these waves, the waves are called inelastic waves. If the energy input into the material due to the explosion is beyond a certain limit, the material is deformed like a liquid, the waves associated to this are called shock waves and the theory is called hydro-dynamic.

If the material, in the present case, the rock formation, is homogeneous and isotropic, then the stress wave propagates in all direction with the same speed. Inhomogeneities such as layers of rock with different mineral composition causes the stress wave to change, accelerate or retard, its progress and, hence, the the wave front will be distorted. Homogeneity is usually found in bed rock or competent rock. Rock, however, is hardly isotropic, because due to the genesis of rock formations some anisotropy may be present. Structural geological features such as jointing, faulting etc. have a decisive influence on wave propagation and may lead to complete absence of waves, e.g. zero transmission across an open joint [Daehnke & Rossmanith 1997]. In a solid material such as highly competent rock (theoretically in a continuum) several types of waves will be transmitted: volume waves, surface waves and interface waves, with each of these groups playing important roles under certain circumstances.

Most important are the volume waves which come in two varieties:

a) Primary or longitudinal waves, also simply called P-waves: these are the fastest waves, similar to acoustic waves in air, with wave propagation c_P and particle movement du/dt as well as flow of energy in the same (radial) direction. These waves typically propagate at speeds of 1000 m/s for weak rock up to 7000 m/s for highly competent rock. Stress σ (in rock mechanics: compression positive, tension negative; in contrast to mechanical engineering where tension is positive and compression is negative) and particle velocity $v_P = du_P/dt$ are related by

$$\sigma = \rho \cdot c_P \cdot v_P = Z_P \cdot v_P \tag{1}$$

where the product of the density ρ and the wave speed c_P is termed acoustic impedance, Z_P. The acoustic impedance is the central parameter in the theory of elastic wave propagation.

b) Secondary or shear waves or transversal waves, or simply S-waves: these waves have a speed cS of about 50% of the speed of the P-wave, ($c_S \approx \frac{1}{2} c_P$) typically in the order of 500 m/s for weak rock up to 4100 m/s for very competent rock such as gabbro. The shear stress τ and the particle velocity v_S are normal to the propagation direction of the S-wave, and they are related through

$$\tau = \rho \cdot c_S \cdot v_S = Z_S \cdot v_S \tag{2}$$

where, similar as before, the product of the density and the shear wave speed c_S is also termed acoustic impedance, Z_S

The acoustic impedances Z_P and Z_S play an important role when considering the reflection and transmission of elastic waves across interfaces in dissimilar rock formations and jointed and faulted rock.

The main representative of the surface waves is the Rayleigh-wave, which is an inhomogeneous combination of a P-wave and a S-wave. The amplitude of the Rayleigh-wave decays exponentially with depth, i.e. with distance from the free surface. The velocity c_R is about 90% of the shear wave velocity, i.e. $c_R \approx 0{,}90\ c_S$, with a dependency on Poisson's ratio of the material. Rayleigh waves carry 90% of their energy in a very shallow material layer just beneath the free surface; and it is exactly this property which makes one feel the Rayleigh waves (and not the volume waves!) when a blast has been set off. In addition to volume and surface waves there are also interface waves in jointed and faulted rock formation. The best known examples are Love-waves and Stonely-waves which, however, can only exist for certain material pairings, i.e. ratii of acoustic impedance mismatches.

Since the detonation velocity is always finite and, moreover, the linear charge is of finite length, conical wave fronts as shown in Figure 1 will be generated. When a finite explosive column detonates, the initiation of the detonation will generate a semi-spherical cap front, which continues into a conical part, which is tipped by the transition from the undetonated to the detonated section of the charge. If the detonation front terminates at the end of the linear charge, another semi-spherical cap wave front is radiated. Geometry dictates that the energy density in the conical section diminishes much slower than the one in the end caps, because the volumes covered by the spherical and conical wave front sections in the same time interval are different [Rossmanith et al 1997, Rossmanith & Kouzniak 2004]. The cones formed by a conical

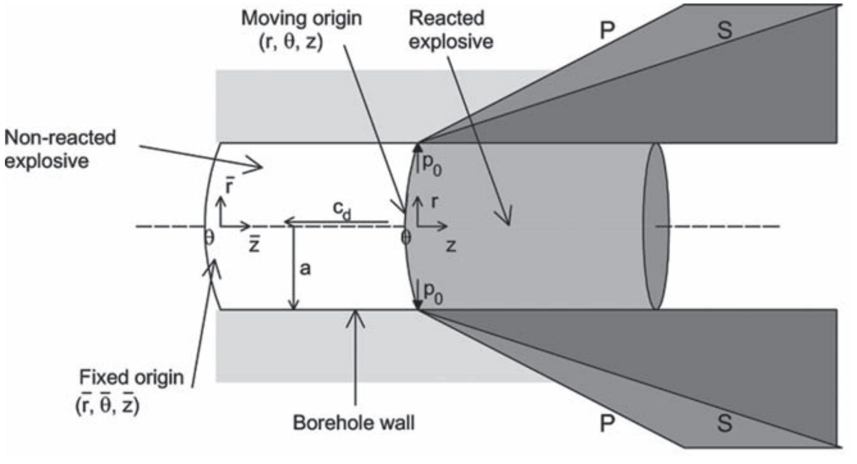

Figure 1. Theoretical Mach stress wave cones in rock due to a supersonically detonating explosive: conical wave ($c_S < c_P < c_P < \infty$).

detonation wave in a rock formation depend on the ratio between the detonation speed and the wave speeds. The higher the ratio c_d/c_P, the more cylindrical-like the conical wave becomes.

The process of wave interaction with an interface gives rise to reflected and transmitted stress waves. Reflection and transmission coefficients can be defined for stresses ($R_{P\sigma}$, $T_{P\sigma}$; $R_{S\sigma}$, $T_{S\sigma}$) and particle velocities (R_{Pv}, T_{Pv}; R_{Sv}, T_{Sv}) by using the acoustic impedances. If Z_{P1} and Z_{P2} are the acoustic P-impedances of the two perfectly jointed materials, reflection ($R_{P\sigma}$) and transmission ($T_{P\sigma}$) coefficients for normal P-wave incidence for the stresses are given by the continuity of stresses and particle velocities. This yields:

$$R_{P\sigma} = \frac{(Z_{P2} - Z_{P1})}{(Z_{P2} + Z_{P1})} \qquad T_{P\sigma} = \frac{2 \cdot Z_{P2}}{(Z_{P2} + Z_{P1})} \tag{3}$$

For a shear wave, replace the index P by an S in equation (3). For acoustic matching, i.e. $Z_1 = Z_2$, all energy is transmitted and nothing is reflected; when the second material is not present (e.g. for a wide open joint, where $Z_2 = 0$, i.e. contact of the joint faces does not occur during wave interaction), then everything is reflected and nothing can be transmitted (up to the point of contact, i.e. closure of the joint) [Daehnke & Rossmanith 1997].

In general, neither the wave fronts nor the joint planes are planar and the level of complexity again is pushed up. The resulting expressions for the general case of non-planar oblique wave incidence become extremely involved. The same holds for wave propagation in jointed and faulted rock.

For monolithic materials and supersonic blasting holds: $c_P > c_S > c_R > c_C$, where c_C is the velocity of a running crack. The interrelationship between the various wave speeds and the crack speed can be graphically demonstrated very nicely in the so-called Lagrange diagram [Rossmanith 2002]. The Lagrangian representation is actually a fairly old method used in all fields of engineering but turns out to be extremely suggestive in blasting. The various wave speeds and the sonic regimes of the rock mass have been indicated in Figure 2, where the supersonic case is the one with the highest wave speeds (shallow black line) and the subsonic one is associated with the lowest wave speeds (steepest black line). The velocity of detonation (the red lines labelled c_d) is indicative of the sonicity of the blast: supersonic, transsonic and subsonic.

In practical work with cartouche-style explosive charges the explosive mass $m_{expl} = V_{expl}\ \rho_{expl}$ (V_{Expl} = volume of explosive, ρ_{expl} = density of

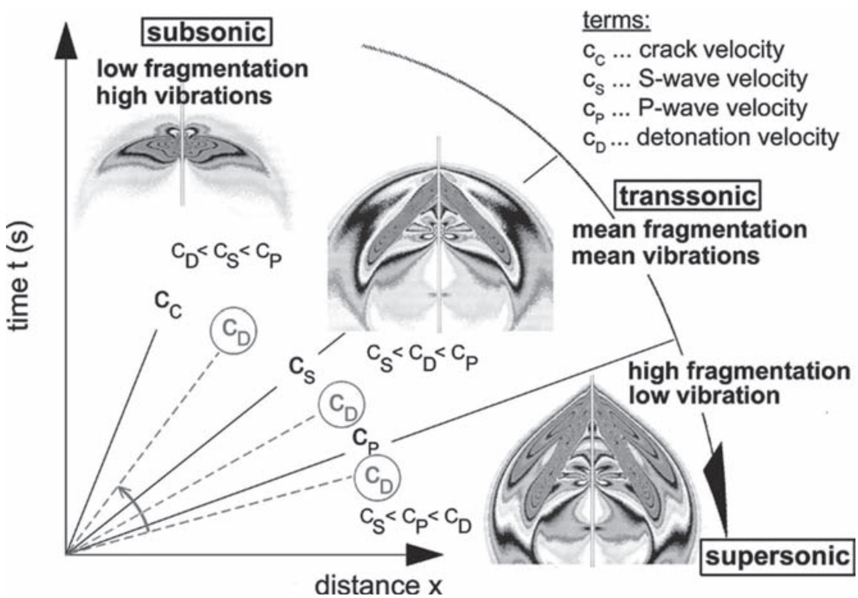

Figure 2. Traces of stress waves and cracks in the time-vs-position diagram and associated save propagation Mach mode: two Mach cones (P and S) for supersonic blasting; one Mach cone (S) transsonic blasting and no Mach cone for subsonic blasting (here: $c_D = c_D$) [Rossmanith et al 1998a, Rossmanith & Mueller 2010, Mueller & Pippig 2011b].

explosive) with $V_{expl} = d_{expl}^2/4\ L_{expl}$ (d_{expl} = diameter of linear charge, L_{expl} = length of charge), in general, is inserted into a cylindrical borehole of defined size (diameter d_{BL}, length L) with volume V_{BL}, whereby the ratio of filling of the borehole is $\xi = V_{expl}/V_{BL}$, and the coupling ratio is $\sigma = (d_{expl}/d_{BL})^2$. The coupling is perfect, i.e. equal to unity for fluid explosives or explosives that can be poured into the blasthole, such as ANFO.

The most important characteristic parameters of an explosive are the density ρ_S and the stationary detonation velocity c_d, or VOD = velocity of detonation. As the velocity of detonation depends on the diameter of the cylindrical charge in the borehole, i.e. $c_d = f(d_{BL})$ and, moreover, a stationary value of c_d is attained only after a short period after initiation, the stationary detonation velocity is given the symbol c_D.

Upon initiation and detonation of a linear charge with velocity c_D, two stress waves will emerge from the detonation front, a longitudinal P-wave and a shear S-wave. Their shape (and, in the supersonic case, cone angles) is (are) dictated by the ratii of c_d/c_P and c_d/c_S.

In a homogeneous and isotropic rock mass there are basically three different modes of sonicity [Rossmanith et al 1997, Rossmanith et al 1998a, Rossmanith et al 1998b] (Fig. 2):

Supersonic detonation: $c_D > c_P > c_S$; the detonation velocity is larger than both wave speeds c_P and c_S of the rock; two so-called conical Mach fronts M_P and M_S will form and sustain as long as the inequality is fulfilled, i.e. $c_D > c_P$.

⇒Result: From optimal to very good fragmentation with comparatively low vibration levels.

Transsonic detonation: $c_P > c_D > c_S$; the velocity of detonation is sandwiched between the wave speeds c_P and c_S of the rock; only one Mach front M_S will be formed with the P-front being a regular quasi-spherical wave similar to the situation where a slower moving source emits continuous energy which is radiated faster then the movement of the source.

⇒Result: Yields average up to good (acceptable) fragmentation within the near region around the blasthole and causes medium level vibrations.

Subsonic detonation: $c_P > c_S > c_D$; the velocity of detonation is smaller than both wave speeds; in this case no Mach cone fronts will be produced.

⇒Result: Poor fragmentation around the blasthole accompanied by very large and intense (unacceptable) vibrations.

The comparison of the various detonation modes is presented in Figure 2 where the supersonic, transsonic and subsonic cases have been incorporated. The transsonic range can be further partitioned into two regimes, one where the c_D is close to the P-wave speed (upper transsonic; $c_S << c_D \leq c_P$) and the other where the c_D is closer to the S-wave speed (lower transsonic; $c_S \leq c_D << c_P$).

The most important case of supersonic detonation is demonstrated in Figs 3a and 3b. Figure 3a shows half of a block with a charged blast hole where a small section of the explosive at the top has not been detonated. The detonation speed, c_d, is larger than any of the wave speeds and the two Mach cones can clearly be identified. The stress wave pattern (distribution of maximum shear stress) was numerically calculated by using a 3D dynamic finite element model for blasting. The 2D counterpart is shown in Fig.3b.

The most important conclusion that can be drawn from this sonicity classification is the realisation that the *mode of detonation of an explosive is not a priori pre-determined*. In other words: The same explosive can detonate supersonically, transsonically and subsonically, the mode depending on the wave propagation speeds of the rock mass in relation to the detonation speed of the explosive. This is called the *principle of sonicity* or the principle of relativity of detonation.

The discovery of the effect of sonicity and its importance in the energy transformation from chemical explosive to shock wave formation upon detonation either in air, liquid or rock mass will have a

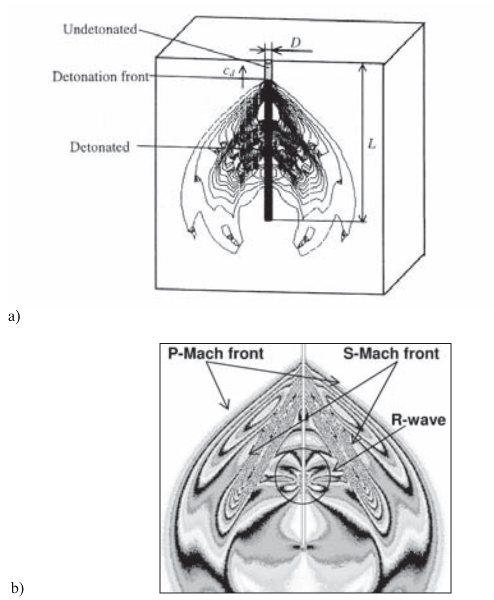

Figure 3. Supersonic detonation: iso-max-shear-stress lines (lines of τ_m = const.) showing two Mach cones for the P-wave and S-wave with $c_S < c_P < c_D$: a) Movement of the detonation front (3D finite element calculation), b) Wave and Mach cone identification (2D finite difference calculation) (Uenishi & Rossmanith, 1998).

dramatic effect in the advanced blasting. Up to now sonicity was not a factor in blast design and operation and hardly a blast enginer was exposed to the physics of stress waves and sonicity. The new physical findings will shape the understanding of the blasting concept and current design rules will be altered and the new knowledge will enter the design codes.

For example, the cutting action of a shape charge is due to the air gap between the charge and the structural component to be cut The cutting jet is formed by the supersonic detonation of the explosive in the air gap. Depending on the type of explosives this process shows cutting speeds of Mach 17-23 (see Figure 4). This rather high Mach number indicates a tremendously strong destructive force of the jet and it is this fact which is most commonly associated with the action of shape charges. In contrast, depending on the detonation velocity of the explosive employed, a supersonic detonation of Mach 3, 5–4 will result when blasting bedrock under water.

These Mach numbers are sufficiently large enough so that the expanding effective Mach stress cones are able to destroy the igniters which are located in the immediate vicinity of the explosive charge, if the initiation sequence under water has not been equipped with instant igniters. The two examples selected show that the sonic effect is always related to a detonative transformation of the explosive in any type of material to be blasted [Rossmanith et al 1998a, b].

3 THE NEW BLASTING MODEL

A new blasting model was derived within the framework of the research projects. It is beased on

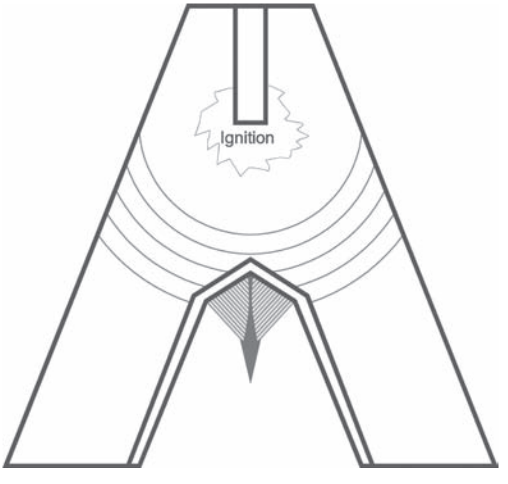

Figure 4. The sonic effect demonstrated with a shape charge.

data acquired before, during and after production blasts performed above ground and underground under realistic in-situ conditions. The blasting model is based on the newly introduced momentum theory as well as on the new physics-based and statistically confirmed theory of sonicity [Mueller & Pippig 2011b, Mueller et al 2011c, Rossmanith & Mueller 2010]. The most remarkable feature of this model is the sonic action and effect around a detonating explosive charge (Figures 2 und 5). For optimal sonic action the explosive ought to be fully coupled in the blasthole.

Practical conclusions:

– With respect to the velocity of detonation the selection of the explosive (for surface and underground blasting operations) is controlled by the speeds of the P-and S-wave of the rock or rock mass (Figure 6).
– The explosive charge should be perfectly, i.e. directly fully coupled to the rockwall in the blasthole. No intermediate tamping is required. Column charges suffer from incomplete contact with the blasthole wall and yield filling ratios $\xi \leq 0{,}75$.

The following notation has been employed in the model shown in Fig. 5:

P_{ZM} = fictitious effectively active detonation pressure of a blast [N/mm²]
P_{ZO} = fictitious effectively active detonation pressure per unit volume [N/mm²]
ξ = ratio of filling of the section of the blasthole containing the explosive charge [–]
ρ_s = density of explosive [kg/m³]
c_d = detonation velocity [m/s]
V_{SO} = volume of explosive charge per unit volume [m³]
V_{SB} = volume of the explosive charge in a blasthole [m³]
λ_s = ratio of dimensions of a blast pattern [–]
w' = effective burden [m]
a_B' = effective spacing [m]
l_{so} = unit length (1 m = const.) [m]
n_v = number of individual volumina [–]

$\left(V_{SB} \cdot \xi\left(\frac{\rho_s \cdot c_d^2}{4}\right)\right)$ = fictitious effective energy $\left[\frac{\mathrm{kg \cdot m^2}}{\mathrm{s^2}}\right]$

ppv_{max} = maximum of vibration velocity [mm/s]
ε_{max} = maximum of dynamic strain [µm/m]
r = distance of blasthole containing the largest explosive charge to the site of vibration measurement
r_o = reference length = 1 m [m]
k, m, n = coefficients appearing in the regression analysis

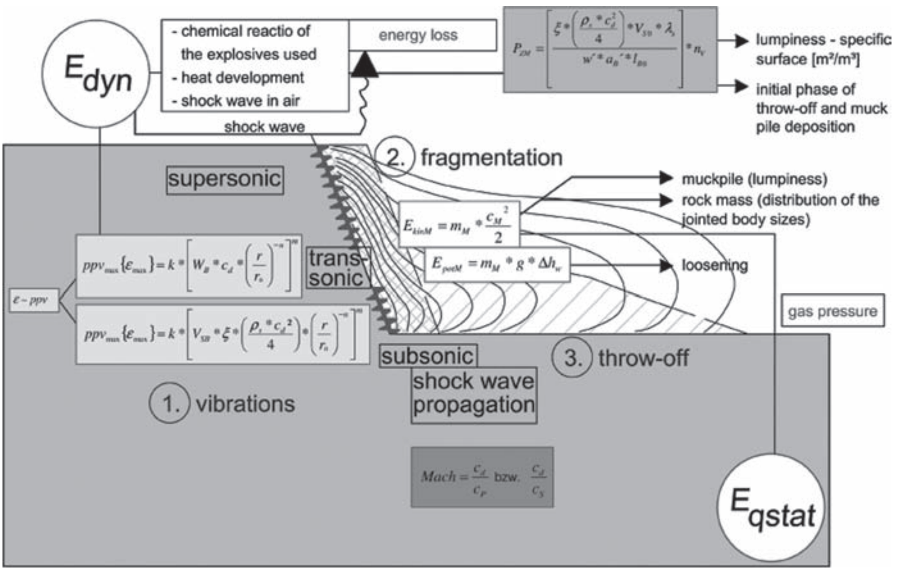

Figure 5. The physico-mechanical model of a blast in a rock mass showing the fundamental objectives such as vibrations, fragmentation and cast (throw off of fragments) of the muckpile (a few relevant mechanical equations containing parameters of advanced blasting theory are inserted in box form).

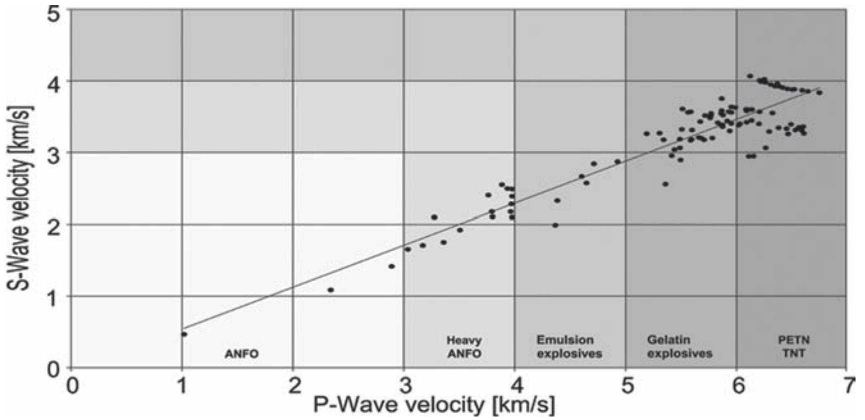

Figure 6. Relationship between longitudinal (P) wave speed and shear (S) wave speed of a variety of hard rocks showing the regions of detonation velocity of the most important explosives (ANFO, Heavy ANFO, Emulsion explosives, Gelatin explosives, PETN, TNT).

Within the framework of the sonicity-based new advanced blasting technology Figure 6 is of utmost importance. Knowing the P- and S-wave velocities of the rock and/or rock mass to be blasted, Figure 6 allows the blast designer or engineer to calculate and optimize the sonic effect for each individual pairing rock versus type of explosive. It is clearly to be seen, that a very competent rock requires a fast explosive in order to generate supersonic blasting conditions whereas ANFO will produce supersonic blasting conditions only in very soft rock where the wave velocities are fairly low.

Granite will be the rock of choice in the particular example problem. The diagram shown in Fig. 7 exhibits the sonic effect of a series of selected commercial explosives when used in conjunction with granite in regular blast jobs. Results shown in [Rrossmanith et al 1998a,b] and in Fig. 3 unveil that the Mach cones around a blasthole have a limited effective range (diameter). The size of the Mach cones depends on the type of explosive, the medium to be blasted, the diameter of the charge and the conditions of the charge in the blasthole.

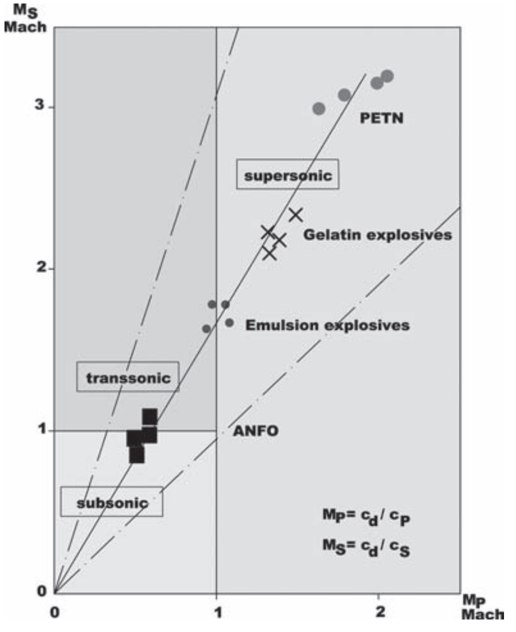

Figure 7. Diagram assisting the assessment of the sonic effect of blasts in competent rock (e.g. in granite).

The practical conclusions to be drawn from the sonic effect are shown in Fig. 8 where the ratio between spacing and burden

$$\lambda_s = \frac{\text{Bohrlochabstand}\,(a_B)}{\text{Vorgabe}\,(w)} = \frac{\text{spacing}\,(a_B)}{\text{burden}\,(w)} \qquad (4)$$

has been selected according to the target of the blast (fragmentation blast or surface splitting blast).

Assume that a series of blastholes of a splitting blast is initiated simultaneously. For narrowly spaced blastholes (smaller than 0.5 m) the "effective" Mach-cones overlap and the result shows a planar splitting action. In this case the vibration analysis must be based on the entire amount of explosive of the split blast. For moderately and widely spaced blastholes (larger than 0.5 m; depending on the blasthole diameter, compared with production and fragmentation blasts) the "effective" Mach cones do not overlap and experience shows that a reliable vibration prediction can be made solely on the basis of the charge per blasthole (see Fig. 8).

Practical conclusions:

- Planar split blasts cause different vibration emission patterns than production and fragmentation blasts and, hence, require differrent assessment.
- The use of the amount of explosive per round (per initiation time) as a basic parameter in the analysis of vibration emissions associated with production and fragmentation blasts is highly

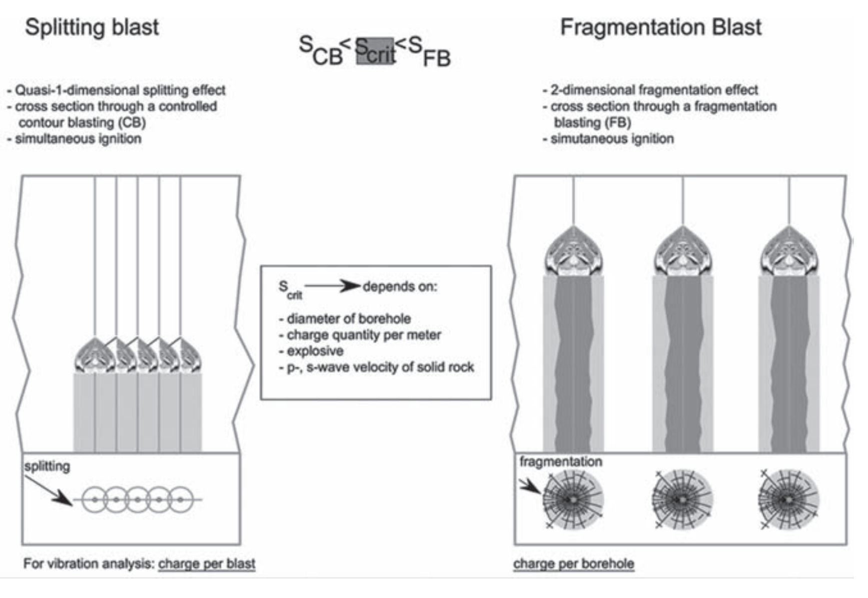

Figure 8. Effect of spacing on fracture network - S_{crit} = critical spacing.

questionable as no influence on the vibration level could be detected.

A major factor which needs to be taken into account is the condition of the rock mass. In massive, sparsely jointed rock the dynamics of the elastic and shock waves is more intense than in heavily jointed rock. Generally speaking, joints tend to reduce wave transmission and also reduce the effective action of the Mach cones [Daehnke & Rossmanith 1997, Rossmanith et al 1997, Rossmanith et al 1998a, b, Rossmanith & Mueller 2010]. Figures 9 and 10 demonstrate the fragmentative action of the rock mass to be blasted with respect to varying degree of jointing for a given sonicity (i.e. the same ratii between the velocity of detonation and the wave speeds). Numerous measurements—performed under identical blasting conditions—indicate and confirm a higher degree of fragmentation in a weakly jointed rock mass than in a strongly jointed one.

Practical conclusions:

- The specific use of explosives is directly related to the frequency of joints and faults of the relevant rock mass [Mueller & Pippig 2011a,b] (Figure 11).
- High specific explosive consumption in heavily jointed rock mass yields a reduced effect due to high energy loss (Figures 9 and 10).

The sonic effect is strongly influenced by the wave dynamic changes of the rock mass [Rossmanith et al 1998b, Rossmanith & Mueller 2010]

The sonic action is of utmost importance in all kinds of blasts: surface (open pit) and underground, under-water blast, shape charge, explosive welding, demolition blasts, etc. There are complex relationships between the various parameters such as the blockability, the fragment size distribution, throw of material, mucking and muckpile formation, as well as the generation of blast ground vibrations.

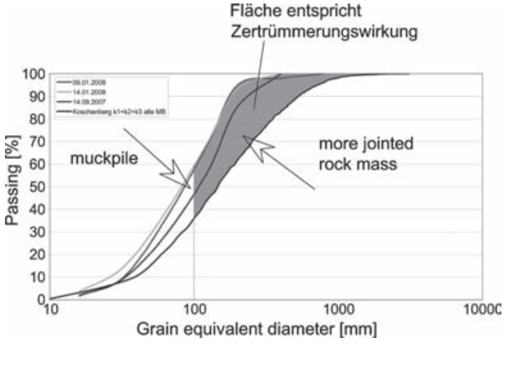

Figure 9. Reduced fragmentation effect associated with a transsonic blasting in a highly jointed rock mass (Meta-Greywacke).

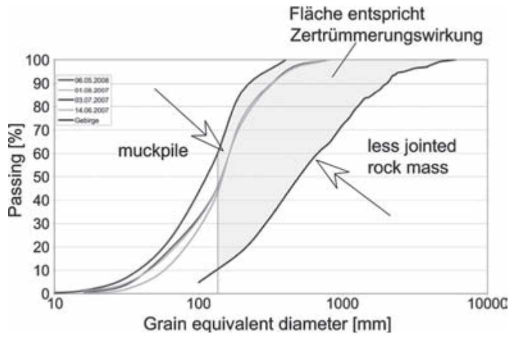

Figure 10. Severe fragmentation as a result of a transsonic blast in weakly jointed rock mass (rock = limestone).

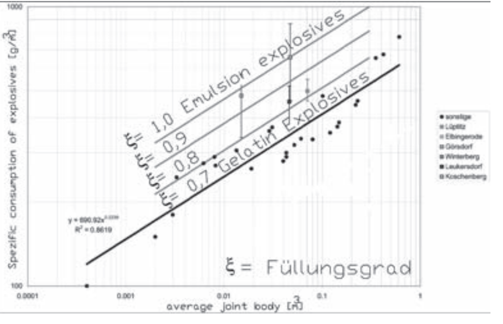

Figure 11. Relationship between average block size, specific explosive consumption and degree of filling for various explosives.

All these relationships could be confirmed by the use of the principle of sonicity and the effective blast energy and blast momentum [Mueller et al 2001, Mueller et al 2009, Mueller & Pippig 2011b, Mueller et al 2011c, Rossmanith & Mueller 2010] (Figure 12). In each blasthole a fictitious effective detonation pressure P_{ZO} is generated which is ultimately responsable for the effective fragmentation of the rock mass (Figures 5 and 12) [Mueller & Pippig 2011b]. The amplitude of the detonation pressure P_{ZO} can be employed as a parameter to control the fragment size distribution, if one considers the unit volume to be blasted (Figure 13).

Practical conclusions:

- Employing the input parameter P_{ZO} the blast operation (production blast) can be tailored and optimized according to need, e.g. to improve the fragment size distribution
- The larger the effective detonation pressure, P_{ZO}, is the more intense the fragmentation will turn out.

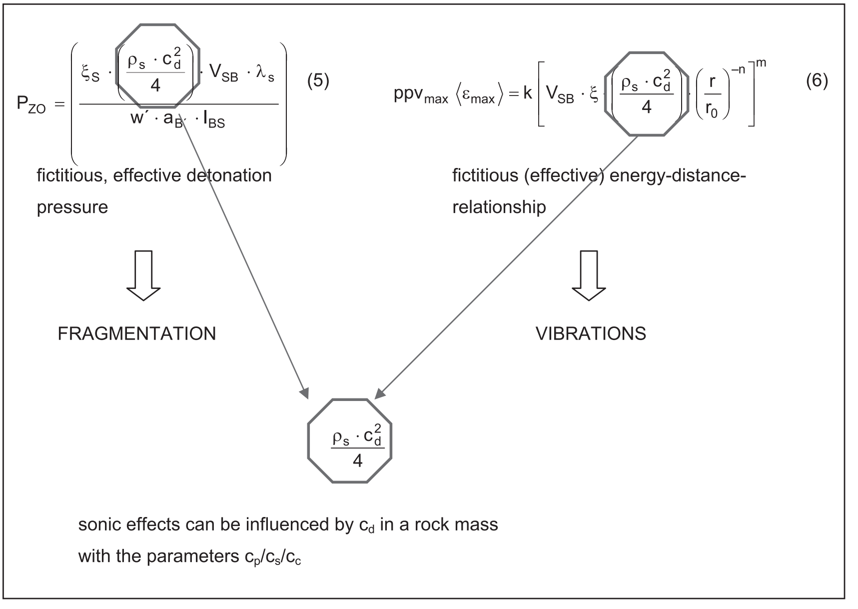

Figure 12. Scheme showing the relationship between fragmentation of the rock mass, excited ground vibrations and the influence of the sonic action (compare Fig. 5)

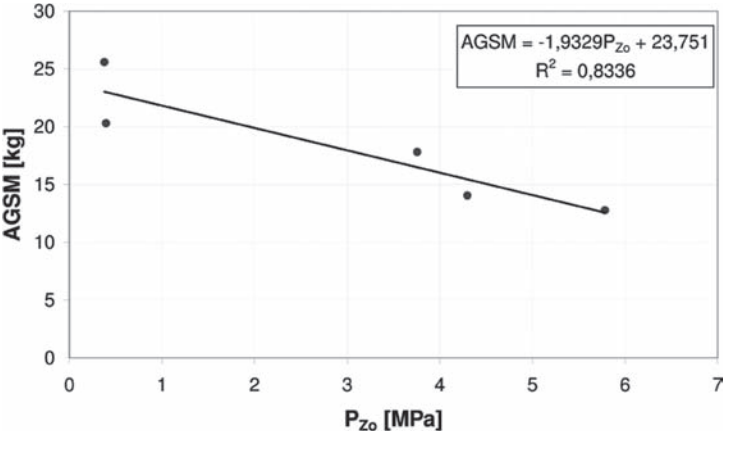

Figure 13. Relationship between fictitious effective detonation pressure P_{ZO} per unit volume (= effective burden × effective blasthole spacing × 1 drill meter) and average mass of fragments of the muckpile (AGSM) [kg].

Extensive measurements of the vibration velocity and of the dynamic strain and regarding the degree of filling, the volume of explosive per blasthole, the density of the explosive and the velocity of detonationl enabled the derivation of statistically confirmed (fictitious) energy-distance relationship for each individual rock mass [Mueller 2001].

The dynamic strain, ε, and the vibration peak particle velocity, ppv, are related by the following physical equations:

$$\sigma = ppv \cdot \rho_G \cdot c_p \tag{7}$$

$$\sigma = \varepsilon \cdot E \tag{8}$$

$$\varepsilon \cdot E = ppv \cdot \rho_G \cdot c_p \tag{9}$$

Where σ is the stress [kN/mm²], ppv is the peak particle velocity [mm/s], ρ_G is the density of the rock mass [kg/m³], c_p is the P-wave velocity of the rock mass [m/s], ε is the dynamic strain [μm/m], and E is Young's modulus [kN/mm²].

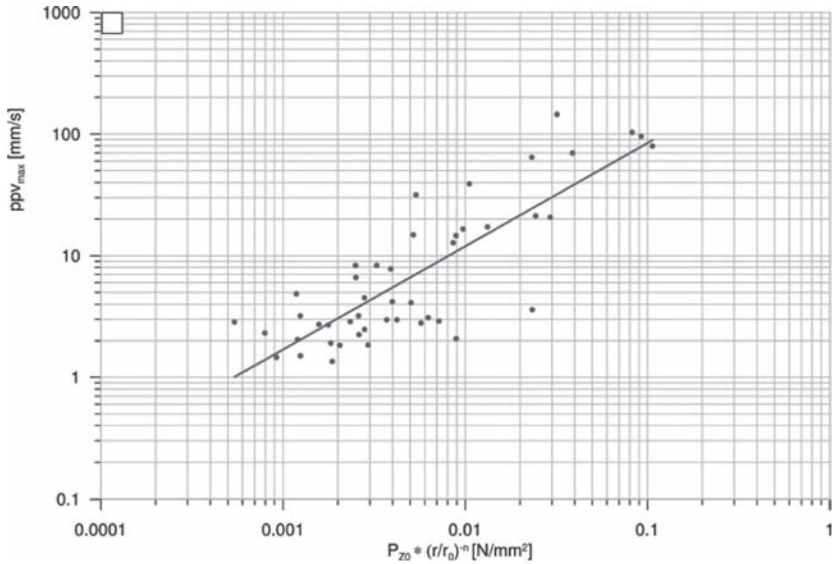

Figure 14. Relationship between peak particle velocity (vibration) and the fictitious, effective detonation pressure (fragmentation) showing the sonic effect of the new advanced blast model.

Based on the eqs (7–9) the use of the dynamic strain instead of the peak particle velocity as parameter in the analysis of blast ground vibrations is promoted and recommended [Baumann & Mueller 2000, Mueller & Pippig 2011b]. In fact, the two quantities, dynamic strain and peak particle velocitiy are on an equal footing.

There is a physically sound linear relationship between the peak particle velocity and the dynamic strain if the transmission conditions and ground conditions during the measurements are the same. The measurements of the dynamic strain by means of the FBG-sensor are by far more precise than measurements of ppv. This holds particularly for the near vicinity of the blasts, i.e. the regime ≤100 m [Mueller & Pippig 2011b]. The new findings whch are based on realistic blasts in various rock masses and which are statistically sound and the physically founded possibilities for an advanced method to predict blast vibrations with the potential for meaningful conclusions with respect to drill and blast technology should exert enough pressure on the decision makers to re-assess and re-evaluate the standards which are currently used for the assessment of blast vibrations [Mueller et al 2011c]. Figure 14 confirms the results of Figure 12, in that the vibrations expressed by the peak particule velocity are directly related to the various different fictitious effective detonation pressure of production blasts if the distance of the measurement site to the detonation site is kept under consideration.

Practical conclusions:

– The current mandatory blast operation design and performance standards (e.g. DIN 42150 in Germany) should be re-assessed and re-evaluated and the modifications must be based on the findings of the new blasting technology which is based on the principle of sonicity. This is a requirement in order to avoid the use of physically unreasonable assumptions in the analysis of predicition of blast vibrations.
– The implementation of the physically proved relationships into practice promises a more environmentally friendly blast practice and, in course, will create a better understanding of the blast requirements of those that are confronted with the vibrations and strains.

REFERENCES

For references see part 2.

Blasting in Mines – New Trends – Ghose & Joshi (Eds)
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New physical findings revolutionize the drilling and blasting technology as well as the prediction of ground vibrations—Part 1: The new blasting model

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ABSTRACT (Part 1 and 2): Within the last two decades technical innovations led to an improved understanding of scientific relationships regarding optimal detonation, fragmentation and reduction of unwanted ground vibrations. This enables the blast engineer to design and taylor blast operations on a statistically sound basis and to assess the results in a physically objective manner. Results of the analysis of hundreds of open pit and underground blasts performed by the first author have unveiled sonicity (ratii between detonation velocity and the wave speeds) being single dominant parameter which controls the vibration problem in both, above ground and underground operations. Sonicity controls the relationship between detonation, fragmentation and vibration immission. For a constant amount of energy of a blast, a higher degree of fragmentation is accompanied with lower levels of vibrations. The new advanced blasting theory based on sonicity enables the blast engineer to design larger blasts without increasing unwanted vibrations while obtaining optimal fragmentation.

1 INTRODUCTION

During the last decade major improvements in the development and production of explosives were achieved, the design of novel and more sophisticated initiation techniques as well as innovative monitoring and measuring devices were developed. All this led the way to new insight into the mechanics and physics of blasting, particularly with respect to the efficient use of explosives in areas where the increase of fragmention and the reduction of vibration is required.

One of the most important tasks is the improvement of knowledge with respect to the sudden transformation of energy from the explosive into mechanical work in the form of optimal fragmentation and muckpile formation while keeping the vibrations in the near- and farfield at bay. These findings offer the blast analyst new tools for the evaluation of the blasting process on an accepted statistical basis. On the basis of this new knowledge the first author re-evaluated several hundred of the blasting works he performed over a period of more than 40 years. The re-evaluation of surface and underground operations was systematically assisted by a novel type of radar sensor, a device for measuring the detonation velocity, fiber-Bragg-grid based strain sensors, 3-component geophones and a 3-D-laser scanner [Mueller & Pippig 2011b]. The evaluation of the re-investigation of these blasting operations led to revolutionizing results and conclusions. Based on the validity of the sonic effect in blasting and explosive technology, the use of a fictitious detonation pressure and other important physical relations relevant to the blasting process yielded new principles for the dimensioning of blast operations.

The application of the new physically sound advanced blasting method enables the user to tailor and control the fragmentation of the rock mass and, thereby, effectively influence the level of vibrations. In most cases a reduction of blast vibrations is requested. Blast vibration immissions are directly related to the destructive effect of the explosive detonation. New ways to treat vibration immissions result from statistically based and proven relationships between momentum or energy liberated during the blast and the distance between blast site and site of concern as well as vibration velocity and dynamic strains. At the same time the new findings allow an almost unlimited expansion of the blasting operation in terms of number of blastholes, firing pattern, etc. without the risk of increase of vibration levels. An important fact is, that the number of blasts can be reduced and this results in an increasingly environmentally friendly blast operation.

In recent years the advanced drill and blasting technology has been enormously promoted by the development of high power drill machinery, new and safer explosives which can be transported to the blast site in special on-site-mix delivery trucks. In addition, improved and more reliable initation and firing techniques for large blast operations are now readily available. In contrast, the basics of traditional design and blast design calculations rely either on outdated assumptions of the 200 years old crater theory of detonation, or the blast operations are traditionally designed on the basis of empirical grounds, i.e. on experience and/or design rules which in many cases lack of a sound physical basis.

Commonly, the assessment of blast vibrations and induced dynamic strains varies according to the blast designers' knowledge or depends on the rules set by the various institutions. In most cases, the calculation of quantities (such as the charge of explosives per round and the various distance relationships which are or are not based on more or less confirmed statistical estimation methods) rests on statistically unconfirmed methods which are not universally applicable. These methods must be generally valid and should not be dependent on certain limiting restrictive conditions. The classical rules for the design of a blast operation neither take into account the properties of the explosive nor those of the rock mass. This drawback formed the motivation for establishing and developing a physically sound model for explosive blasting which incorporates the recently unveiled relationships between momentum, energy, sonicity, blastability and specific explosive consumption [Mueller 2001].

The Deutsche Bundesstiftung Umwelt (German Federal Institution on Environmental Issues) has been sponsoring the two research projects "Environmetally supportive advanved blasting technology" (Umweltfreundliche Sprengtechnik) and "Sonic Effects" (Sonische Wirkung). Many of the essential findings have been incorporated in this contribution [Mueller et al 2009, Mueller & Pippig 2011b, Mueller et al 2011c, Rossmanith et al 1998a, Rossmanith & N Kouzniak 2004 and Rossmanith & Mueller 2010].

2 THEORETICAL FOUNDATION OF WAVE PROPAGATION AND SONICITY

If a stone is thrown into a pond one will notice circularly crested waves which propagate from the point of impact of the stone on the surface of the water. While these waves expand with a (seemingly) constant velocity, their amplitude diminishes rapidly with radius travelled. Although water is an entirely different material than rock and rock supports a higher complexity of waves than water, this picture quite nicely illustrates and gives an idea of what occurs in a solid body, such as a mountain, a quarry, in the underground or just in a piece of reasonably coherent rock in the form of a specimen in the testing laboratory. In a fairly similar way elastic waves will be expanding from a source in a solid.

Regarding the shape of the source, the point source would be the simplest case, at least in theory. In practice there are no point sources but extended sources, the simplest being the spherical charge. Upon detonation of a spherical charge, a detonation wave expands from the surface of the rock/explosive interface, which gives rise to shock, plastic and elastic waves depending on the behavior of the material which supports wave propagation. If, for small and moderate amounts of explosives, the material behaviour is linear or non-linear elastic, these waves are termed elastic or non-linear elastic stress waves. If some plastic work is associated with the propagation of these waves, the waves are called inelastic waves. If the energy input into the material due to the explosion is beyond a certain limit, the material is deformed like a liquid, the waves associated to this are called shock waves and the theory is called hydro-dynamic.

If the material, in the present case, the rock formation, is homogeneous and isotropic, then the stress wave propagates in all direction with the same speed. Inhomogeneities such as layers of rock with different mineral composition causes the stress wave to change, accelerate or retard, its progress and, hence, the the wave front will be distorted. Homogeneity is usually found in bed rock or competent rock. Rock, however, is hardly isotropic, because due to the genesis of rock formations some anisotropy may be present. Structural geological features such as jointing, faulting etc. have a decisive influence on wave propagation and may lead to complete absence of waves, e.g. zero transmission across an open joint [Daehnke & Rossmanith 1997]. In a solid material such as highly competent rock (theoretically in a continuum) several types of waves will be transmitted: volume waves, surface waves and interface waves, with each of these groups playing important roles under certain circumstances.

Most important are the volume waves which come in two varieties:

a) Primary or longitudinal waves, also simply called P-waves: these are the fastest waves, similar to acoustic waves in air, with wave propagation c_P and particle movement du/dt as well as flow of energy in the same (radial) direction. These waves typically propagate at speeds of 1000 m/s for weak rock up to 7000 m/s for highly competent rock. Stress σ (in rock mechanics: compression positive, tension negative; in contrast to mechanical engineering where tension is positive and compression is negative) and particle velocity $v_P = du_P/dt$ are related by

$$\sigma = \rho \cdot c_P \cdot v_P = Z_P \cdot v_P \tag{1}$$

where the product of the density ρ and the wave speed c_P is termed acoustic impedance, Z_P. The acoustic impedance is the central parameter in the theory of elastic wave propagation.

b) Secondary or shear waves or transversal waves, or simply S-waves: these waves have a speed cS of about 50% of the speed of the P-wave, ($c_S \approx \frac{1}{2} c_P$) typically in the order of 500 m/s for weak rock up to 4100 m/s for very competent rock such as gabbro. The shear stress τ and the particle velocity v_S are normal to the propagation direction of the S-wave, and they are related through

$$\tau = \rho \cdot c_S \cdot v_S = Z_S \cdot v_S \tag{2}$$

where, similar as before, the product of the density and the shear wave speed c_S is also termed acoustic impedance, Z_S

The acoustic impedances Z_P and Z_S play an important role when considering the reflection and transmission of elastic waves across interfaces in dissimilar rock formations and jointed and faulted rock.

The main representative of the surface waves is the Rayleigh-wave, which is an inhomogeneous combination of a P-wave and a S-wave. The amplitude of the Rayleigh-wave decays exponentially with depth, i.e. with distance from the free surface. The velocity c_R is about 90% of the shear wave velocity, i.e. $c_R \approx 0{,}90\ c_S$, with a dependency on Poisson's ratio of the material. Rayleigh waves carry 90% of their energy in a very shallow material layer just beneath the free surface; and it is exactly this property which makes one feel the Rayleigh waves (and not the volume waves!) when a blast has been set off. In addition to volume and surface waves there are also interface waves in jointed and faulted rock formation. The best known examples are Love-waves and Stonely-waves which, however, can only exist for certain material pairings, i.e. ratii of acoustic impedance mismatches.

Since the detonation velocity is always finite and, moreover, the linear charge is of finite length, conical wave fronts as shown in Figure 1 will be generated. When a finite explosive column detonates, the initiation of the detonation will generate a semi-spherical cap front, which continues into a conical part, which is tipped by the transition from the undetonated to the detonated section of the charge. If the detonation front terminates at the end of the linear charge, another semi-spherical cap wave front is radiated. Geometry dictates that the energy density in the conical section diminishes much slower than the one in the end caps, because the volumes covered by the spherical and conical wave front sections in the same time interval are different [Rossmanith et al 1997, Rossmanith & Kouzniak 2004]. The cones formed by a conical

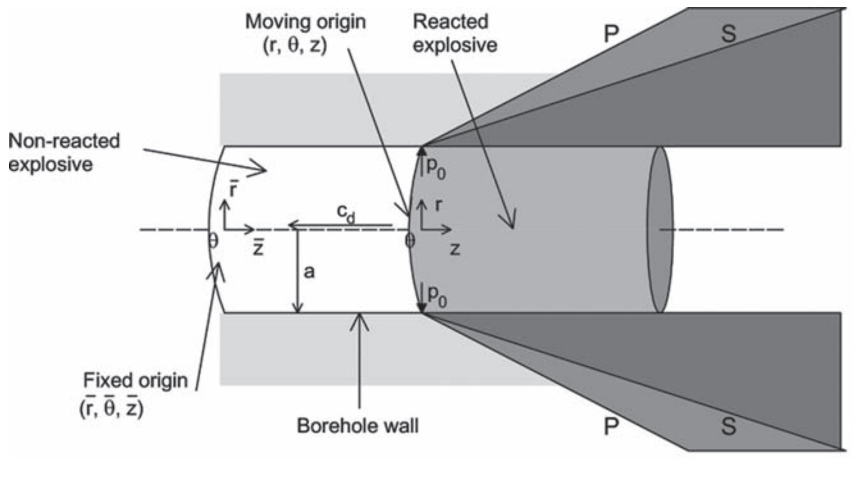

Figure 1. Theoretical Mach stress wave cones in rock due to a supersonically detonating explosive: conical wave ($c_S < c_P < c_P < \infty$).

detonation wave in a rock formation depend on the ratio between the detonation speed and the wave speeds. The higher the ratio c_d/c_P, the more cylindrical-like the conical wave becomes.

The process of wave interaction with an interface gives rise to reflected and transmitted stress waves. Reflection and transmission coefficients can be defined for stresses ($R_{P\sigma}$, $T_{P\sigma}$; $R_{S\sigma}$, $T_{S\sigma}$) and particle velocities (R_{Pv}, T_{Pv}; R_{Sv}, T_{Sv}) by using the acoustic impedances. If Z_{P1} and Z_{P2} are the acoustic P-impedances of the two perfectly jointed materials, reflection ($R_{P\sigma}$) and transmission ($T_{P\sigma}$) coefficients for normal P-wave incidence for the stresses are given by the continuity of stresses and particle velocities. This yields:

$$R_{P\sigma}=\frac{(Z_{P2}-Z_{P1})}{(Z_{P2}+Z_{P1})}$$
$$T_{P\sigma}=\frac{2\cdot Z_{P2}}{(Z_{P2}+Z_{P1})} \tag{3}$$

For a shear wave, replace the index P by an S in equation (3). For acoustic matching, i.e. $Z_1 = Z_2$, all energy is transmitted and nothing is reflected; when the second material is not present (e.g. for a wide open joint, where $Z_2 = 0$, i.e. contact of the joint faces does not occur during wave interaction), then everything is reflected and nothing can be transmitted (up to the point of contact, i.e. closure of the joint) [Daehnke & Rossmanith 1997].

In general, neither the wave fronts nor the joint planes are planar and the level of complexity again is pushed up. The resulting expressions for the general case of non-planar oblique wave incidence become extremely involved. The same holds for wave propagation in jointed and faulted rock.

For monolithic materials and supersonic blasting holds: $c_P > c_S > c_R > c_C$, where c_C is the velocity of a running crack. The interrelationship between the various wave speeds and the crack speed can be graphically demonstrated very nicely in the so-called Lagrange diagram [Rossmanith 2002]. The Lagrangian representation is actually a fairly old method used in all fields of engineering but turns out to be extremely suggestive in blasting. The various wave speeds and the sonic regimes of the rock mass have been indicated in Figure 2, where the supersonic case is the one with the highest wave speeds (shallow black line) and the subsonic one is associated with the lowest wave speeds (steepest black line). The velocity of detonation (the red lines labelled c_d) is indicative of the sonicity of the blast: supersonic, transsonic and subsonic.

In practical work with cartouche-style explosive charges the explosive mass $m_{expl} = V_{expl}\ \rho_{expl}$ (V_{Expl} = volume of explosive, ρ_{expl} = density of

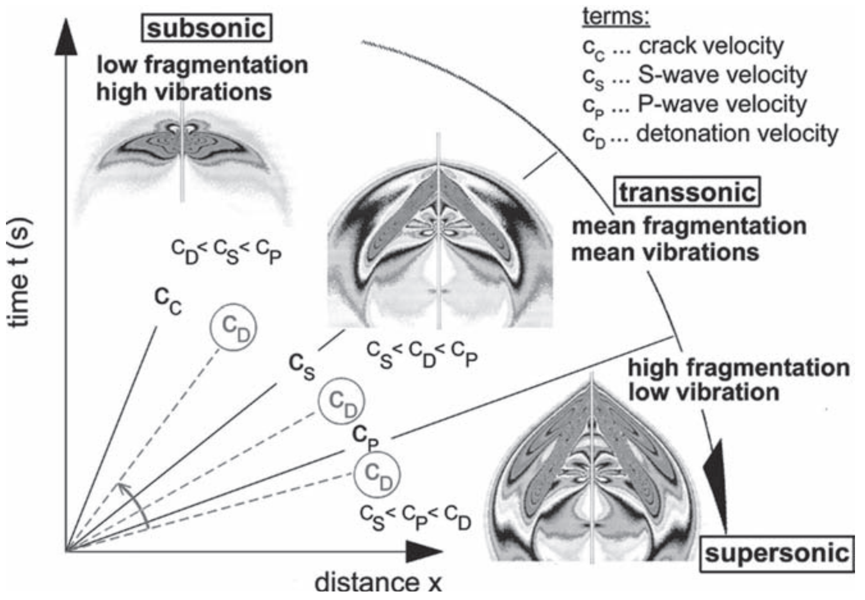

Figure 2. Traces of stress waves and cracks in the time-vs-position diagram and associated save propagation Mach mode: two Mach cones (P and S) for supersonic blasting; one Mach cone (S) transsonic blasting and no Mach cone for subsonic blasting (here: $c_D = c_D$) [Rossmanith et al 1998a, Rossmanith & Mueller 2010, Mueller & Pippig 2011b].

explosive) with $V_{expl} = d_{expl}^2/4\ L_{expl}$ (d_{expl} = diameter of linear charge, L_{expl} = length of charge), in general, is inserted into a cylindrical borehole of defined size (diameter d_{BL}, length L) with volume V_{BL}, whereby the ratio of filling of the borehole is $\xi = V_{expl}/V_{BL}$, and the coupling ratio is $\sigma = (d_{expl}/d_{BL})^2$. The coupling is perfect, i.e. equal to unity for fluid explosives or explosives that can be poured into the blasthole, such as ANFO.

The most important characteristic parameters of an explosive are the density ρ_S and the stationary detonation velocity c_d, or VOD = velocity of detonation. As the velocity of detonation depends on the diameter of the cylindrical charge in the borehole, i.e. $c_d = f(d_{BL})$ and, moreover, a stationary value of c_d is attained only after a short period after initiation, the stationary detonation velocity is given the symbol c_{D}.

Upon initiation and detonation of a linear charge with velocity c_D, two stress waves will emerge from the detonation front, a longitudinal P-wave and a shear S-wave. Their shape (and, in the supersonic case, cone angles) is (are) dictated by the ratii of c_d/c_P and c_d/c_S.

In a homogeneous and isotropic rock mass there are basically three different modes of sonicity [Rossmanith et al 1997, Rossmanith et al 1998a, Rossmanith et al 1998b] (Fig. 2):

Supersonic detonation: $c_D > c_P > c_S$; the detonation velocity is larger than both wave speeds c_P and c_S of the rock; two so-called conical Mach fronts M_P and M_S will form and sustain as long as the inequality is fulfilled, i.e. $c_D > c_P$.

⇒Result: From optimal to very good fragmentation with comparatively low vibration levels.

Transsonic detonation: $c_P > c_D > c_S$; the velocity of detonation is sandwiched between the wave speeds c_P and c_S of the rock; only one Mach front M_S will be formed with the P-front being a regular quasi-spherical wave similar to the situation where a slower moving source emits continuous energy which is radiated faster then the movement of the source.

⇒Result: Yields average up to good (acceptable) fragmentation within the near region around the blasthole and causes medium level vibrations.

Subsonic detonation: $c_P > c_S > c_D$; the velocity of detonation is smaller than both wave speeds; in this case no Mach cone fronts will be produced.

⇒Result: Poor fragmentation around the blasthole accompanied by very large and intense (unacceptable) vibrations.

The comparison of the various detonation modes is presented in Figure 2 where the supersonic, transsonic and subsonic cases have been incorporated. The transsonic range can be further partitioned into two regimes, one where the c_D is close to the P-wave speed (upper transsonic; $c_S << c_D \le c_P$) and the other where the c_D is closer to the S-wave speed (lower transsonic; $c_S \le c_D << c_P$).

The most important case of supersonic detonation is demonstrated in Figs 3a and 3b. Figure 3a shows half of a block with a charged blast hole where a small section of the explosive at the top has not been detonated. The detonation speed, c_d, is larger than any of the wave speeds and the two Mach cones can clearly be identified. The stress wave pattern (distribution of maximum shear stress) was numerically calculated by using a 3D dynamic finite element model for blasting. The 2D counterpart is shown in Fig.3b.

The most important conclusion that can be drawn from this sonicity classification is the realisation that the *mode of detonation of an explosive is not a priori pre-determined*. In other words: The same explosive can detonate supersonically, transsonically and subsonically, the mode depending on the wave propagation speeds of the rock mass in relation to the detonation speed of the explosive. This is called the *principle of sonicity* or the principle of relativity of detonation.

The discovery of the effect of sonicity and its importance in the energy transformation from chemical explosive to shock wave formation upon detonation either in air, liquid or rock mass will have a

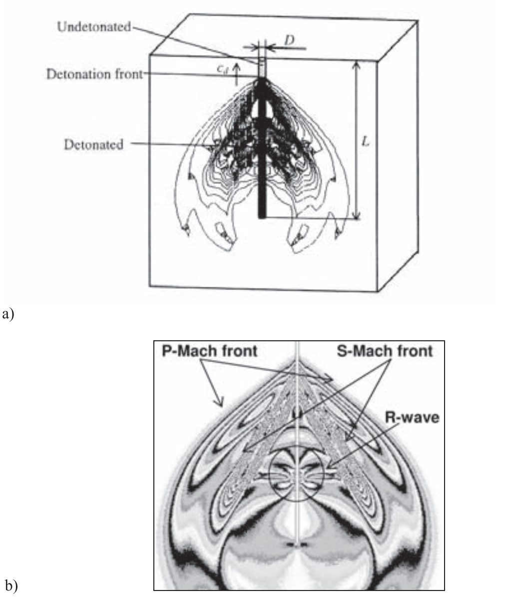

Figure 3. Supersonic detonation: iso-max-shear-stress lines (lines of τ_m = const.) showing two Mach cones for the P-wave and S-wave with $c_S < c_P < c_D$: a) Movement of the detonation front (3D finite element calculation), b) Wave and Mach cone identification (2D finite difference calculation) (Uenishi & Rossmanith, 1998).

dramatic effect in the advanced blasting. Up to now sonicity was not a factor in blast design and operation and hardly a blast enginer was exposed to the physics of stress waves and sonicity. The new physical findings will shape the understanding of the blasting concept and current design rules will be altered and the new knowledge will enter the design codes.

For example, the cutting action of a shape charge is due to the air gap between the charge and the structural component to be cut The cutting jet is formed by the supersonic detonation of the explosive in the air gap. Depending on the type of explosives this process shows cutting speeds of Mach 17-23 (see Figure 4). This rather high Mach number indicates a tremendously strong destructive force of the jet and it is this fact which is most commonly associated with the action of shape charges. In contrast, depending on the detonation velocity of the explosive employed, a supersonic detonation of Mach 3, 5–4 will result when blasting bedrock under water.

These Mach numbers are sufficiently large enough so that the expanding effective Mach stress cones are able to destroy the igniters which are located in the immediate vicinity of the explosive charge, if the initiation sequence under water has not been equipped with instant igniters. The two examples selected show that the sonic effect is always related to a detonative transformation of the explosive in any type of material to be blasted [Rossmanith et al 1998a, b].

3 THE NEW BLASTING MODEL

A new blasting model was derived within the framework of the research projects. It is beased on

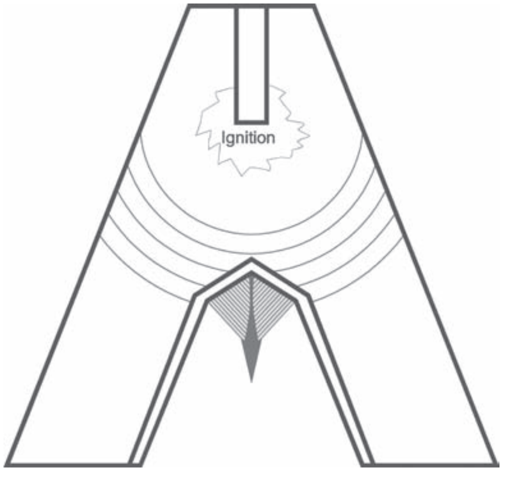

Figure 4. The sonic effect demonstrated with a shape charge.

data acquired before, during and after production blasts performed above ground and underground under realistic in-situ conditions. The blasting model is based on the newly introduced momentum theory as well as on the new physics-based and statistically confirmed theory of sonicity [Mueller & Pippig 2011b, Mueller et al 2011c, Rossmanith & Mueller 2010]. The most remarkable feature of this model is the sonic action and effect around a detonating explosive charge (Figures 2 und 5). For optimal sonic action the explosive ought to be fully coupled in the blasthole.

Practical conclusions:

- With respect to the velocity of detonation the selection of the explosive (for surface and underground blasting operations) is controlled by the speeds of the P-and S-wave of the rock or rock mass (Figure 6).
- The explosive charge should be perfectly, i.e. directly fully coupled to the rockwall in the blasthole. No intermediate tamping is required. Column charges suffer from incomplete contact with the blasthole wall and yield filling ratios $\xi \leq 0{,}75$.

The following notation has been employed in the model shown in Fig. 5:

P_{ZM} = fictitious effectively active detonation pressure of a blast [N/mm²]
P_{ZO} = fictitious effectively active detonation pressure per unit volume [N/mm²]
ξ = ratio of filling of the section of the blasthole containing the explosive charge [–]
ρ_s = density of explosive [kg/m³]
c_d = detonation velocity [m/s]
V_{SO} = volume of explosive charge per unit volume [m³]
V_{SB} = volume of the explosive charge in a blasthole [m³]
λ_s = ratio of dimensions of a blast pattern [–]
w' = effective burden [m]
a_B' = effective spacing [m]
l_{so} = unit length (1 m = const.) [m]
n_v = number of individual volumina [–]

$\left(V_{SB} \cdot \xi\left(\frac{\rho_s \cdot c_d^2}{4}\right)\right)$ = fictitious effective energy $\left[\frac{\mathrm{kg \cdot m^2}}{\mathrm{s^2}}\right]$

ppv_{max} = maximum of vibration velocity [mm/s]
ε_{max} = maximum of dynamic strain [µm/m]
r = distance of blasthole containing the largest explosive charge to the site of vibration measurement
r_o = reference length = 1 m [m]
k, m, n = coefficients appearing in the regression analysis

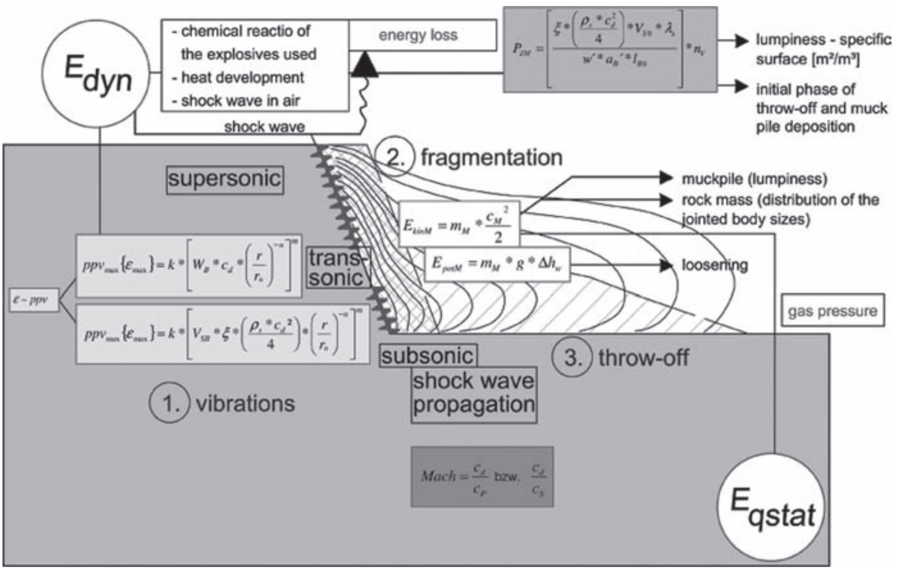

Figure 5. The physico-mechanical model of a blast in a rock mass showing the fundamental objectives such as vibrations, fragmentation and cast (throw off of fragments) of the muckpile (a few relevant mechanical equations containing parameters of advanced blasting theory are inserted in box form).

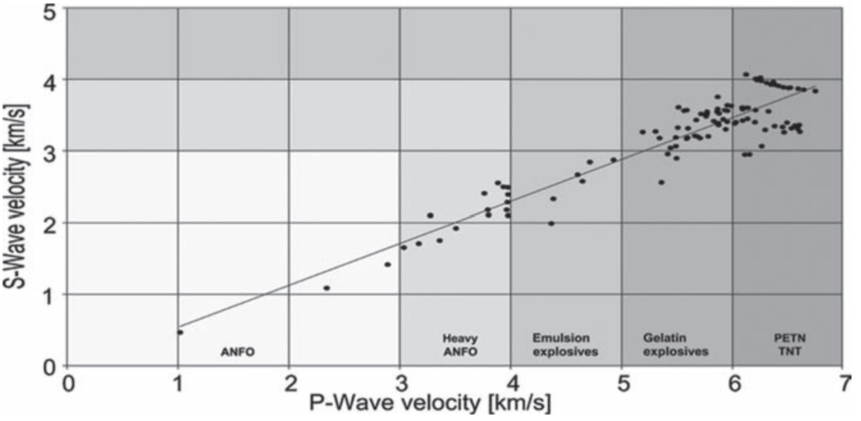

Figure 6. Relationship between longitudinal (P) wave speed and shear (S) wave speed of a variety of hard rocks showing the regions of detonation velocity of the most important explosives (ANFO, Heavy ANFO, Emulsion explosives, Gelatin explosives, PETN, TNT).

Within the framework of the sonicity-based new advanced blasting technology Figure 6 is of utmost importance. Knowing the P- and S-wave velocities of the rock and/or rock mass to be blasted, Figure 6 allows the blast designer or engineer to calculate and optimize the sonic effect for each individual pairing rock versus type of explosive. It is clearly to be seen, that a very competent rock requires a fast explosive in order to generate supersonic blasting conditions whereas ANFO will produce supersonic blasting conditions only in very soft rock where the wave velocities are fairly low.

Granite will be the rock of choice in the particular example problem. The diagram shown in Fig. 7 exhibits the sonic effect of a series of selected commercial explosives when used in conjunction with granite in regular blast jobs. Results shown in [Rrossmanith et al 1998a,b] and in Fig. 3 unveil that the Mach cones around a blasthole have a limited effective range (diameter). The size of the Mach cones depends on the type of explosive, the medium to be blasted, the diameter of the charge and the conditions of the charge in the blasthole.

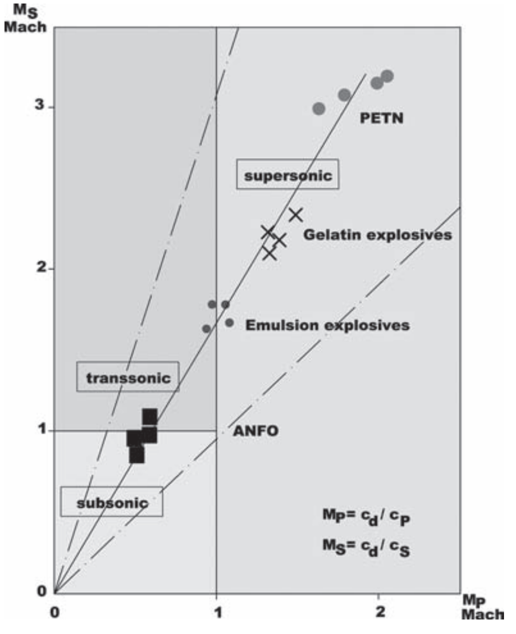

Figure 7. Diagram assisting the assessment of the sonic effect of blasts in competent rock (e.g. in granite).

The practical conclusions to be drawn from the sonic effect are shown in Fig. 8 where the ratio between spacing and burden

$$\lambda_s = \frac{\text{Bohrlochabstand}\left(a_B\right)}{\text{Vorgabe}\left(w\right)} = \frac{\text{spacing}\left(a_B\right)}{\text{burden}\left(w\right)} \tag{4}$$

has been selected according to the target of the blast (fragmentation blast or surface splitting blast).

Assume that a series of blastholes of a splitting blast is initiated simultaneously. For narrowly spaced blastholes (smaller than 0.5 m) the "effective" Mach-cones overlap and the result shows a planar splitting action. In this case the vibration analysis must be based on the entire amount of explosive of the split blast. For moderately and widely spaced blastholes (larger than 0.5 m; depending on the blasthole diameter, compared with production and fragmentation blasts) the "effective" Mach cones do not overlap and experience shows that a reliable vibration prediction can be made solely on the basis of the charge per blasthole (see Fig. 8).

Practical conclusions:

- Planar split blasts cause different vibration emission patterns than production and fragmentation blasts and, hence, require differrent assessment.
- The use of the amount of explosive per round (per initiation time) as a basic parameter in the analysis of vibration emissions associated with production and fragmentation blasts is highly

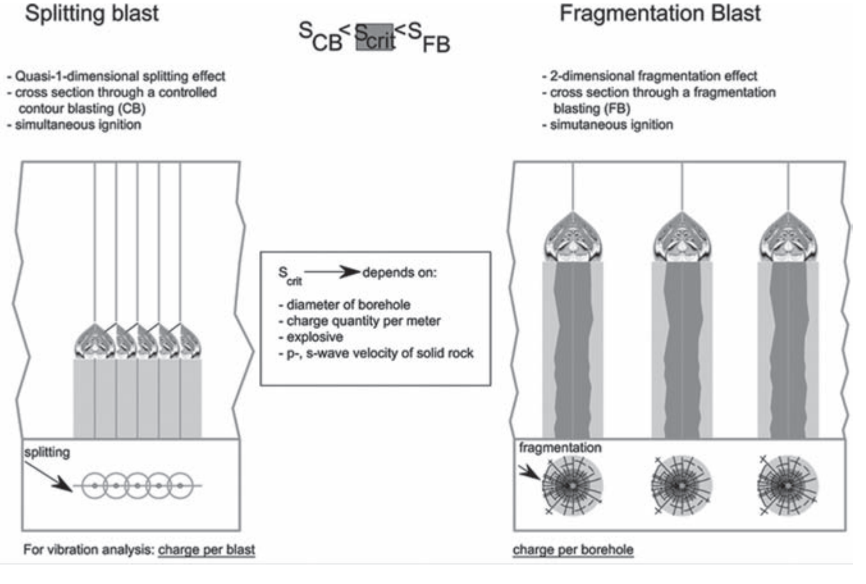

Figure 8. Effect of spacing on fracture network - S_{crit} = critical spacing.

questionable as no influence on the vibration level could be detected.

A major factor which needs to be taken into account is the condition of the rock mass. In massive, sparsely jointed rock the dynamics of the elastic and shock waves is more intense than in heavily jointed rock. Generally speaking, joints tend to reduce wave transmission and also reduce the effective action of the Mach cones [Daehnke & Rossmanith 1997, Rossmanith et al 1997, Rossmanith et al 1998a, b, Rossmanith & Mueller 2010]. Figures 9 and 10 demonstrate the fragmentative action of the rock mass to be blasted with respect to varying degree of jointing for a given sonicity (i.e. the same ratii between the velocity of detonation and the wave speeds). Numerous measurements—performed under identical blasting conditions—indicate and confirm a higher degree of fragmentation in a weakly jointed rock mass than in a strongly jointed one.

Practical conclusions:

- The specific use of explosives is directly related to the frequency of joints and faults of the relevant rock mass [Mueller & Pippig 2011a,b] (Figure 11).
- High specific explosive consumption in heavily jointed rock mass yields a reduced effect due to high energy loss (Figures 9 and 10).

The sonic effect is strongly influenced by the wave dynamic changes of the rock mass [Rossmanith et al 1998b, Rossmanith & Mueller 2010]

The sonic action is of utmost importance in all kinds of blasts: surface (open pit) and underground, under-water blast, shape charge, explosive welding, demolition blasts, etc. There are complex relationships between the various parameters such as the blockability, the fragment size distribution, throw of material, mucking and muckpile formation, as well as the generation of blast ground vibrations.

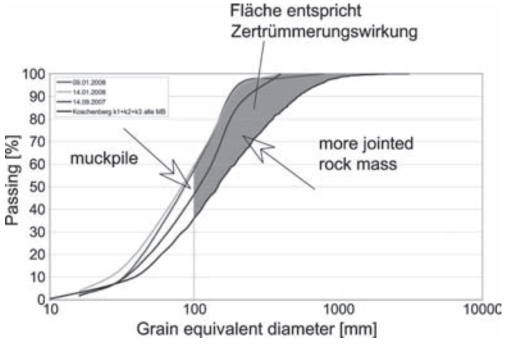

Figure 9. Reduced fragmentation effect associated with a transsonic blasting in a highly jointed rock mass (Meta-Greywacke).

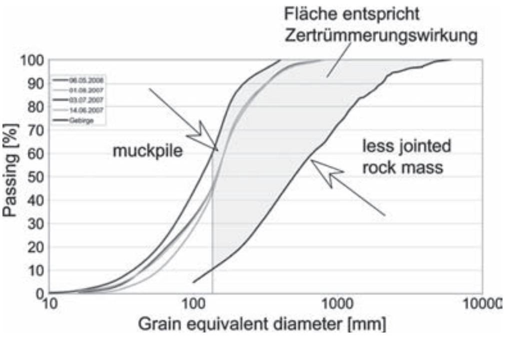

Figure 10. Severe fragmentation as a result of a transsonic blast in weakly jointed rock mass (rock = limestone).

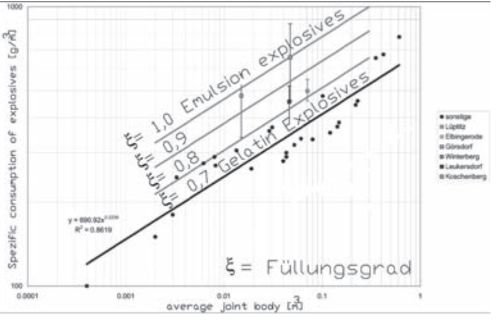

Figure 11. Relationship between average block size, specific explosive consumption and degree of filling for various explosives.

All these relationships could be confirmed by the use of the principle of sonicity and the effective blast energy and blast momentum [Mueller et al 2001, Mueller et al 2009, Mueller & Pippig 2011b, Mueller et al 2011c, Rossmanith & Mueller 2010] (Figure 12). In each blasthole a fictitious effective detonation pressure P_{ZO} is generated which is ultimately responsable for the effective fragmentation of the rock mass (Figures 5 and 12) [Mueller & Pippig 2011b]. The amplitude of the detonation pressure P_{ZO} can be employed as a parameter to control the fragment size distribution, if one considers the unit volume to be blasted (Figure 13).

Practical conclusions:

- Employing the input parameter P_{ZO} the blast operation (production blast) can be tailored and optimized according to need, e.g. to improve the fragment size distribution
- The larger the effective detonation pressure, P_{ZO}, is the more intense the fragmentation will turn out.

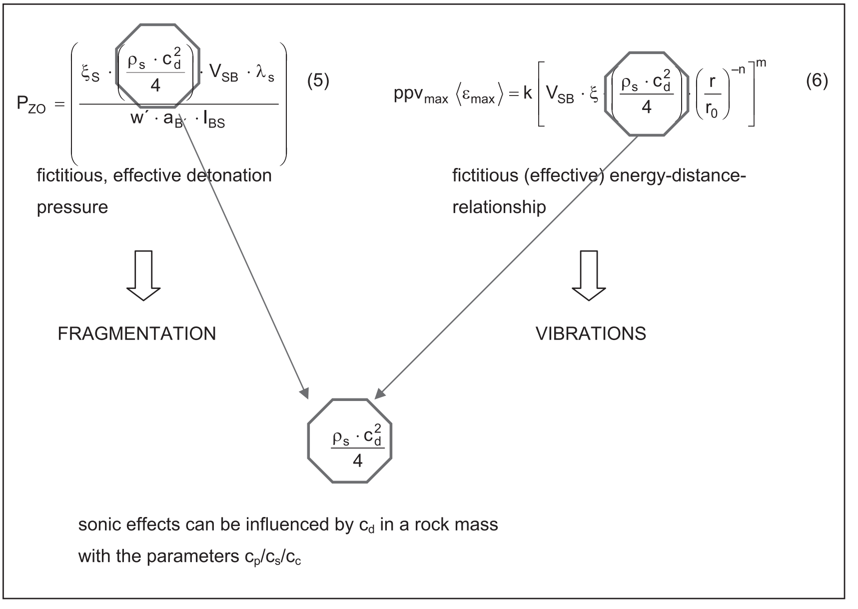

Figure 12. Scheme showing the relationship between fragmentation of the rock mass, excited ground vibrations and the influence of the sonic action (compare Fig. 5)

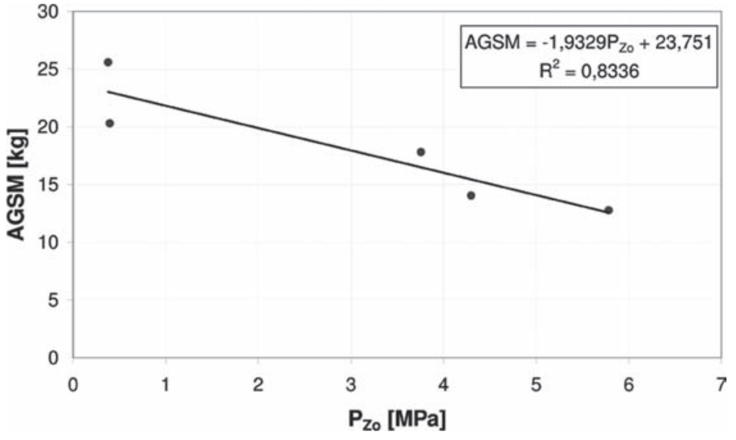

Figure 13. Relationship between fictitious effective detonation pressure P_{ZO} per unit volume (= effective burden × effective blasthole spacing × 1 drill meter) and average mass of fragments of the muckpile (AGSM) [kg].

Extensive measurements of the vibration velocity and of the dynamic strain and regarding the degree of filling, the volume of explosive per blasthole, the density of the explosive and the velocity of detonationl enabled the derivation of statistically confirmed (fictitious) energy-distance relationship for each individual rock mass [Mueller 2001].

The dynamic strain, ε, and the vibration peak particle velocity, ppv, are related by the following physical equations:

$$\sigma = ppv \cdot \rho_G \cdot c_p \tag{7}$$

$$\sigma = \varepsilon \cdot E \tag{8}$$

$$\varepsilon \cdot E = ppv \cdot \rho_G \cdot c_p \tag{9}$$

Where σ is the stress [kN/mm²], ppv is the peak particle velocity [mm/s], ρ_G is the density of the rock mass [kg/m³], c_p is the P-wave velocity of the rock mass [m/s], ε is the dynamic strain [µm/m], and E is Young's modulus [kN/mm²].

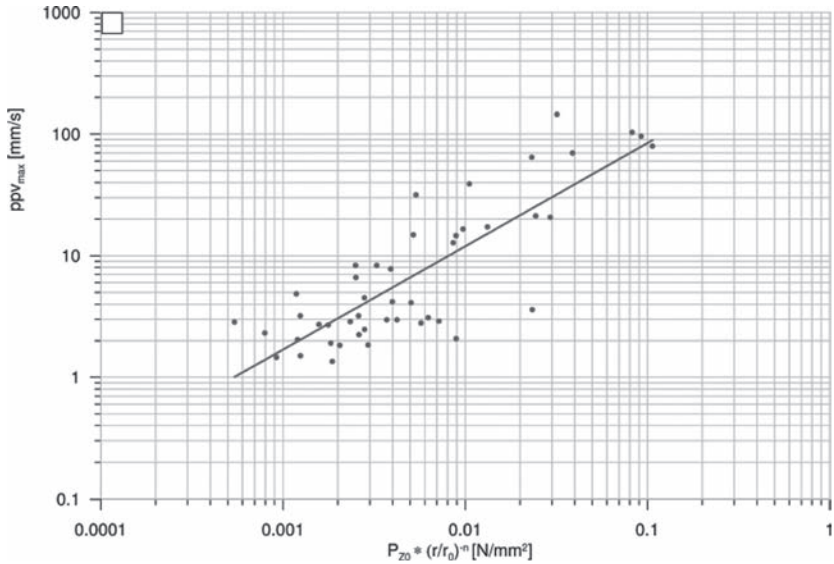

Figure 14. Relationship between peak particle velocity (vibration) and the fictitious, effective detonation pressure (fragmentation) showing the sonic effect of the new advanced blast model.

Based on the eqs (7–9) the use of the dynamic strain instead of the peak particle velocity as parameter in the analysis of blast ground vibrations is promoted and recommended [Baumann & Mueller 2000, Mueller & Pippig 2011b]. In fact, the two quantities, dynamic strain and peak particle velocitiy are on an equal footing.

There is a physically sound linear relationship between the peak particle velocity and the dynamic strain if the transmission conditions and ground conditions during the measurements are the same. The measurements of the dynamic strain by means of the FBG-sensor are by far more precise than measurements of ppv. This holds particularly for the near vicinity of the blasts, i.e. the regime ≤100 m [Mueller & Pippig 2011b]. The new findings whch are based on realistic blasts in various rock masses and which are statistically sound and the physically founded possibilities for an advanced method to predict blast vibrations with the potential for meaningful conclusions with respect to drill and blast technology should exert enough pressure on the decision makers to re-assess and re-evaluate the standards which are currently used for the assessment of blast vibrations [Mueller et al 2011c]. Figure 14 confirms the results of Figure 12, in that the vibrations expressed by the peak particule velocity are directly related to the various different fictitious effective detonation pressure of production blasts if the distance of the measurement site to the detonation site is kept under consideration.

Practical conclusions:

– The current mandatory blast operation design and performance standards (e.g. DIN 42150 in Germany) should be re-assessed and re-evaluated and the modifications must be based on the findings of the new blasting technology which is based on the principle of sonicity. This is a requirement in order to avoid the use of physically unreasonable assumptions in the analysis of predicition of blast vibrations.
– The implementation of the physically proved relationships into practice promises a more environmentally friendly blast practice and, in course, will create a better understanding of the blast requirements of those that are confronted with the vibrations and strains.

REFERENCES

For references see part 2.

Blasting in Mines – New Trends – Ghose & Joshi (Eds)

New physical findings revolutionize the drilling and blasting technology as well as the prediction of ground vibrations—Part 2: Practical applications above ground and underground

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ABSTRACT (Part 1 and 2): Within the last two decades technical innovations led to an improved understanding of scientific relationships regarding optimal detonation, fragmentation and reduction of unwanted ground vibrations. This enables the blast engineer to design and taylor blast operations on a statistically sound basis and to assess the results in a physically objective manner. Results of the analysis of hundreds of open pit and underground blasts performed by the first author have unveiled sonicity (ratii between detonation velocity and the wave speeds) being single dominant parameter which controls the vibration problem in both, above ground and underground operations. Sonicity controls the relationship between detonation, fragmentation and vibration immission. For a constant amount of energy of a blast, a higher degree of fragmentation is accompanied with lower levels of vibrations. The new advanced blasting theory based on sonicity enables the blast engineer to design larger blasts without increasing unwanted vibrations while obtaining optimal fragmentation.

1 BASICS OF BLASTING IN ROCK MASS IN SURFACE OR OPEN PIT OPERATIONS

The design of a blast operation depends primarily on the purpose of the blast operation. In other words, the blast design in terms of borehole pattern (spacing and burden) and firing pattern (intra and inter borehole delay time; delay of rows, etc.) is tailored according to need with the two extremes: split blast and fragmentation blast. Figure 1 shows a schematic representation of the various degrees of fragmentation in a classical bench blast in an open pit operation. The choice of the explosive then depends on the sonicity, the ratio of burden to spacing, λ_s, and on taking into account the critical distance of the blastholes, a_{Bcrit}. Figure 1 unveils that an increasing ratio of burden to spacing and increasing explosive charge of a single blasthole leads to increasing spacial fragmentation of the rock mass. For sufficiently spaced blastholes the effective Mach cones will not interact and hence, the explosive charge in a single blasthole is responsible for the vibrations measured at the site in the far-field.

The fundamental basics of the new blasting technology which need to be considered in the process of design and dimensioning of a blast operation are given in Table 1. Using the parameters which appear in the relationship of the effective active detonation pressure, equ (6) (part 1) (Fig. 12 (part 1)), it is now possible to optimize any fragmentation or production blast operation such that the fragment size distribution of the muckpile wil be improved (Fig. 13 (part 1)). Apart from varying certain parameters the magnitude of the ficititious effective detonation pressure can be altered (decreased or increased) by the sole variation of the initiation sequence. In this way it is possible to achieve an improved destructive action solely on the basis of changing the initiation sequence without the use of additional explosives [Mueller et al 2009, Mueller & Pippig 2011a]. It is, however, to be noticed that in multi-row blast operations, any change of the simultaneous initiation sequence will cause a shortening of the burden and increase the spacing as seen from an initiation technology point of view (Fig. 1)

Practical conclusions:

– Split blast operations must be exclusively performed with supersonically acting explosives, which have a high detonation velocity and

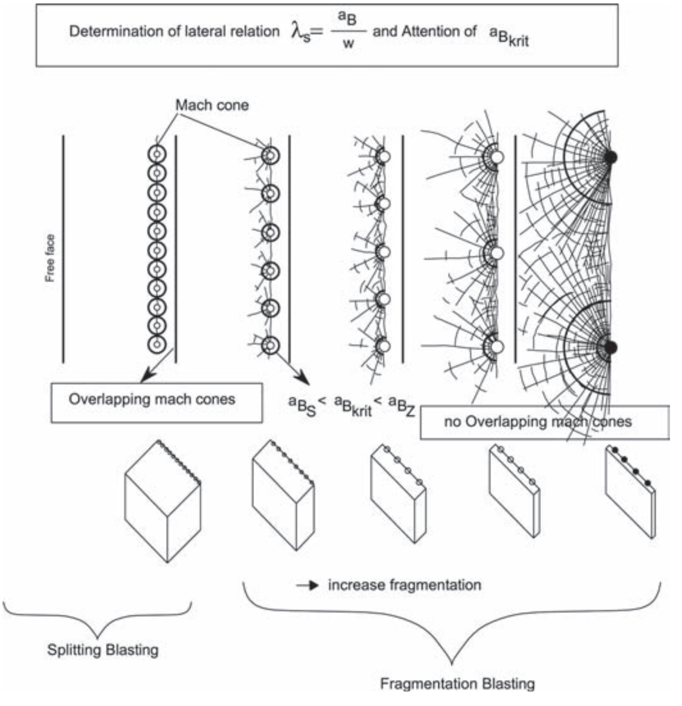

Figure 1. Determination of the ratio burden to spacing according to the blast goal to be achieved paying attention to sonicity and the interaction of the Mach cones.

Table 1. Fundamentals concerning the design and dimensioning of split and fragmentation blast operations in competent rock mass.

Split blast operations	Fragmentation blast operations
Quasi one-dimensional blast operations – planar action and effect	Two-dimensional blast operations – spacial action and effect
Longitudinal uniform explosive charge along the entire length of the blasthole	Full filling (complete coupling) in the entire blasthole except for the stemming section
The specific explosive consumption (according to Fig. 11 (part 1)) is about 1/10; most often it is related to the surfaces of the split gap.	The specific explosive consumption is estimated on the basis of the spacing of the joints and/or the average size of the joint blocks (Fig. 11 (part 1))
Single row blast with simultaneous initiation with wanted interaction of the Mach cones	Multiple row blasts with progressive simultaneous initiation according to the new advanced blasting technology based on the momentum theory; there is no interaction of the individual Mach cones
Supersonic blast is absolutely necessary	Design of blast operation with best possible sonicity
Uniform spacing and inclination of blastholes over the entire length/depth	Harmonic geometrical parameters, uniform distribution of explosive charge
Uniform explosive charge with $c_d > c_p$	Uniform explosive charge with or without booster at the bottom of the blasthole
Prediction of blast vibration on the basis of the total charge of the split blast operation	Prediction of blast vibrations is based on the maximum blasthole charge of a blast operation

density; the charge must be precisely distributed and uniformly arranged.

– Fragmentation blasts should be designed according to the optimum ratio of burden to spacing to achieve the highest possible economic effect by utilizing the principle of sonicity. The charge per blasthole and the type of explosive used are responsible for the prediction of the ground vibrations. The initiation of the blasthole must be controlled by the fundamental issues of the new advanced blasting theory which is based on momentum and effective energy.

Practical examples:

Example 1–Tunnel in open construction
During the excavation of a 12 m deep tunnel (open trench operation) located on the western periphery of the German city of Bautzen, in Saxony, 83 blasts had to be performed in a highly populated area. The rock was granodiorite which is a notoriously difficult rock with very poor blastability. The nearest residential and business buildings were located in close proximity of 3,2–5,6 meters from the detonation site. The excavation work was obstructed and impeded by a gas main at 3 m distance and also by three steel bridges. Considering this complex situation the blast operation was designed on the basis of the new advanced blasting theory. The first part of the blast operation consisted of about 40 supersonic split blasts, each of them had an explosive charge of 7,5 bis 11,5 kg PETN. The second phase began at a depth of 5–6 m below street level. Due to the larger distance from the buildings and in order to save drill meterage this phase was designed a fragmentation blast. The charge per blasthole was 4 bis 8 kg gelatinous explosive.

The ground vibration emissions were registered and monitored by means of several suitably placed geophones and two FBG—strain sensors. The attentive reader will clearly recognize the remarkable differences of the vibrations in the ppv- versus distance diagram (see Fig. 2). These differences are related to the different sonic actions although the same charges of explosives and PETN detonating cord have been used. Figure 3 confirms this statement on the basis of the derived momentum – distance relationship. Hence, the sonic action, with the dissimilar diameters and interactions of the Mach cones, is universally valid and can, therefore, be equally utilized for split blast operations as well as fragmentation blast operations.

The concept developed for the assessment and prediction of the vibration emissions during blasting which takes into account the respective charge corresponds to the real emission conditions. Using the new possibilities offered by the new advanced blasting technology, an improved vibration prediction could be performed and the blasts could be tailored such that in the near-field the buildings did not suffer from excessive strains and surely no

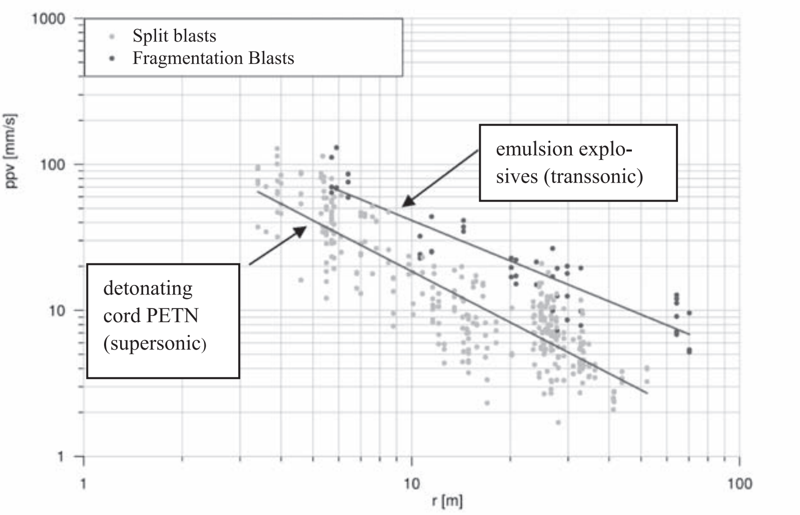

Figure 2. Peak particle velocity (ppv) versus distance (r) diagram and the effect of different types of explosives on the generation of ground vibrations.

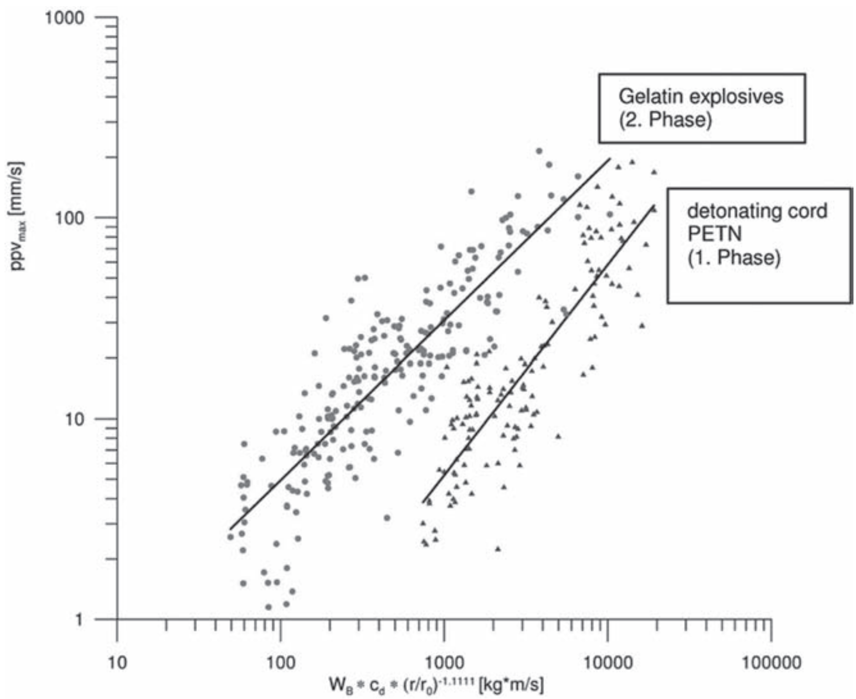

Figure 3. Comparison between fictitious momentum versus distance relationship ($W_B \cdot c_d$) for the gelatinous explosives and detonating cord used showing the validity of the sonic action for split and production blasts (first phase split phase; second phase fragmentation phase).

cracks did develop. Measurements of vibration strain levels were performed in the basements of the buildings and the stresses were of the order of 1,2–16,5 MPa in the near-field of the blasts. These strains and stresses the buldings could easily take without any problems.

Figure 4 shows part of the strain measurements performed during the first phase, i.e. during the split blasts. The results show and confirm the linear relationship between strain and peak particle velocity ppv (see eq (9) (part 1)). The new fundamentals for the design of fragmentation and/or production blast operations alllow an appreciable increase in size of the blast operation without, at the same time, increasing the blast vibrations.

Example 2—Meta-Greywacke Operation in Germany

The application of the results and possibilities offered by the new advanced blasting technique in an open pit Meta-Greywacke operation in Ossling in Germany (Saxony) led to a dramatic reduction of the number of blasts and the blasting technology could be enormously improved. In the past and until now, a variable mixture of ANFO-, emulsion and gelatinous explosives was utilized. After the adaption of the new advanced blasting technology the same operation now uses a unified pumpable emulsion explosive. The daily blasts could be reduced to a single blast every fortnight. The number of blastholes per blast could be elevated from around 10–15 to 200 blastholes and even more.(see Fig. 5).

The vibration emissions remained on the same level and upon request they can systematically be reduced by employing the derived energy-distance quantity. Figure 6 shows the derived relationship between the peak particle velocity (fictitious) and the fictitious energy quantity for the traditional blast operations and those based on the new advanced blasting technology for the open pit operation in Ossling in Germany.

The diagram in Figure 6 reveals that, in comparison to modern blasting, the weak sonicity of the traditional blasts causes higher vibration levels (for a fixed energy input!). As the closest residential buildings are located already at a distance of 150 meters from the blast site in the open pit, the control of the peak particle velocity is of utmost practical importance when production blasts have to be performed in order to be able to exploit the mineral riches of the region. The blasts will now

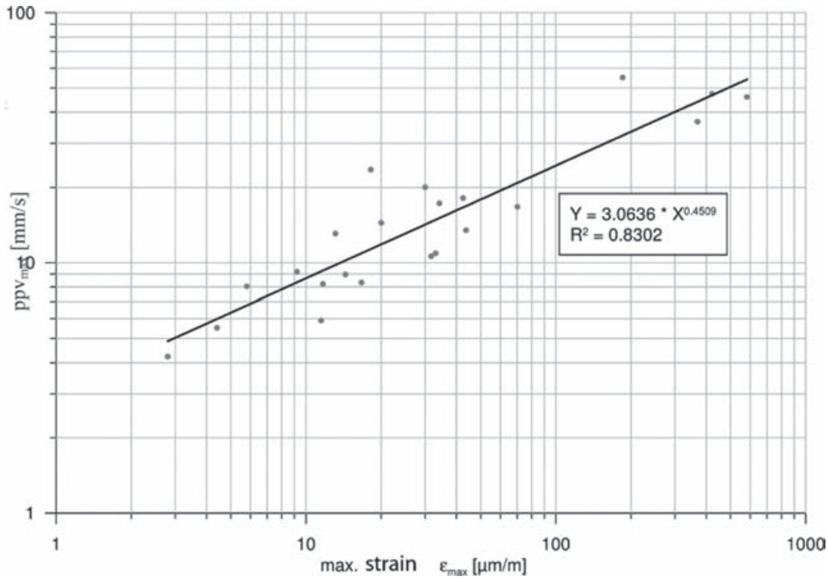

Figure 4. Correlation of the peak particle velocity ppv$_{max}$ versus maximum strain ε_{max} for a split blast at the entrance of the tunnel (open trench method).

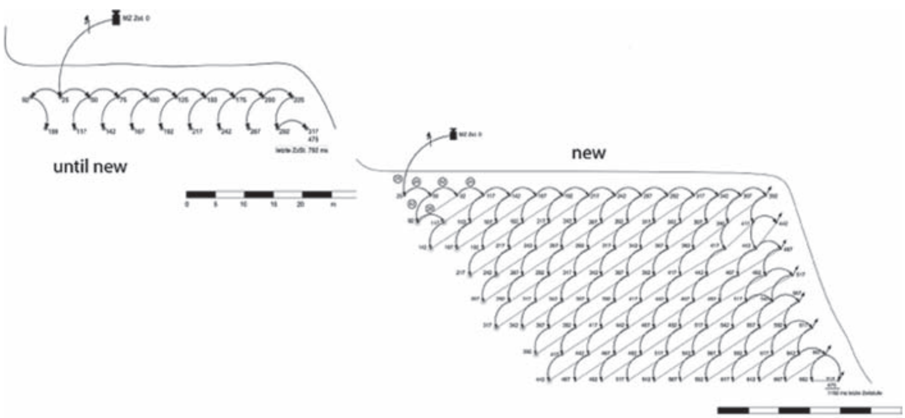

Figure 5. Example showing the evolution of initiation and delay time maps for blast operations (left: traditional old, right: according to the new advanced blasting technology based on effective momentum and the principle of sonicity).

be dimensioned and tailored according to the results obtained while keeping at the same time the admissible recomendations and advancing the stope/bench all the way up to the alllowable limit of the pit.

Practical conclusions:

– In the near-field of blasts (depending on the size of the operation, usually up to 100 meters) the strain meaurements turn out to be more precise and reliable than measurements of peak particle velocity. It is recommended to exclusively perform strain measurements in the near field using FBG-sensor systems.
– The fully supersonic blast actions/effects enables the performance of very near-field blasts where necessary for the solution of very critical blasting problems without running the danger of generating crack-related damage.
– The blast operations can be designed and tailored to almost unlimited size without running the risk of excessive accumulation of vibrations when simultaneously igniting a large number

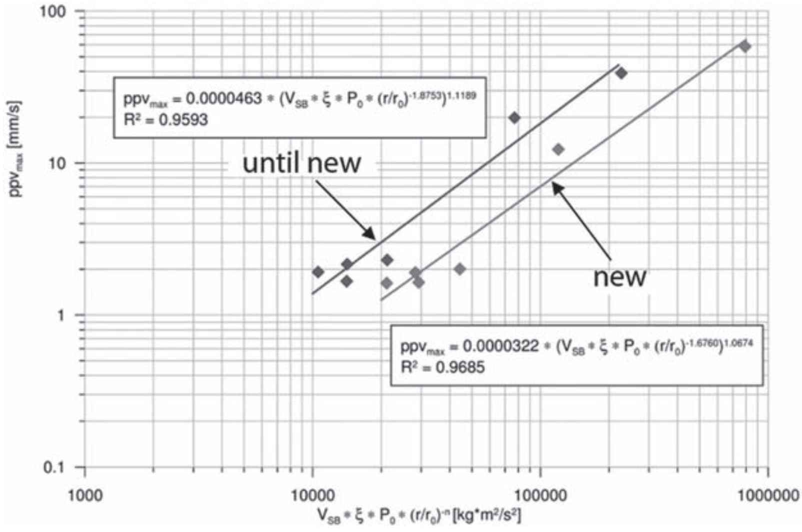

Figure 6. Relationship between peal particle velocity and energy-distance for the traditional and the advanced blasting methods.

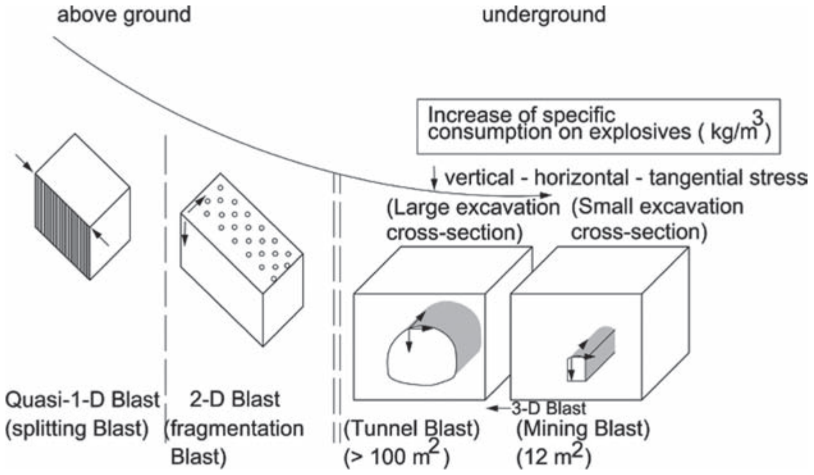

Figure 7. Schematic representation of the increase of the specific consumption of the explosive for various types of blast operations: 1-dimensional, 2-dimensional, and 3-dimensional blasts in competent bedrock (surface and underground operations).

of charges..This allows the blasting engineer to reduce the number of blasts and, thus, actively contribute to environmental protection.

2 THE FUNDAMENTALS OF BLASTING UNDER GROUND IN ROCK MASS

The implementation of the new advanced blasting method to underground operations, i.e. the adaptation of the design and performance of the drill, initiation and blast patterns for underground works in tunneling, mining, quarrying or excavation of underground caverns follows the same procedures as outlined before, the only difference to surface blasts being that underground operations occur under conditions of three-dimensional states of stress and the stress levels are entirely different (Fig. 7).

When blasting underground, the dimensions of the cavern/cavity to be blasted, the size of the stope face and the stress triaxiality play an important role next to parameters, such as rock strength and the parameters characterizing the advance of the stope face [Mueller & Pippig 2011b]. The specific consumption of explosives for three-dimensional subterranean blasts follows from Figure 8. Here, the size of the stope face and the rock quality need to be taken into account.

The selection of the explosive is controlled by maximizing the sonic effect (see Fig. 6 (part 1)), The active boreholes of the break-in section and those within the region of the auxiliary holes should be uniformly loaded with appropriate explosive. The peripheral ring blastholes and the bottom blastholes ought to be designed and treated like the holes in a split blast. In fact, their purpose is to limit the fragmentation zone by arresting any uncontrolled radial cracks that would destabilize the immediate surrounding of the stope. In the design of the number and location of the blast and auxiliary holes, attention has to be given to the special state of stress in the corner regions. The firing pattern should be designed properly such that the break-in should occur in a harmonic fashion with the firing sequence following a spiral pattern from inside to the periphery. An optimal selection of the delay times is achieved when the progressive initiation process works from the center toward the periohery without any jumpy or even out-of-sequence firing.

Under very special circumstances, e.g. in mining of highly sensitive minerals such as diamong mining in kimberlite bedrock, an exception from the rule must be made. In these cases, the mineral structure should not be destroyed and, hence, the blast must be designed as a subsonic blast, regarding the wave speeds c_P and c_S of the diamonds.

In another operation, the Markovec tunnel in Slovenia, the face advance in the 50 m^2 large roof of the tunnel had to be limited such that buildings at a distance of 20 m above the tunnel roof would not suffer from any peak particle velocities larger than 7 mm/sec. Prior to the design and selection of the drill and initiation pattern, rock samples were extracted from the clay and limestone formation and P and S wave speeds measured. Only after these rock dynamics investigations was the selection of the explosive performed: it was decided to use an emulsion explosive manufactured by AUSTIN Powder. This explosive guaranteed supersonic blast conditions. Once the explosive was defined according to the principle of supersonicity, the drill, firing and initiation patterns were developed according to Figure 9. The focus was on minimizing the ground vibrations. The next step was to ckeck the proper design by firing a few trial blasts which confirmed and was in line with the requirements imposed on the blast work. The vibration levels measured 20 m from the blast site were smaller than the limits set by the officials and this confirmed the validity of the selected design of the drill and firiing scheme.

Employing the momentum versus distance relationship of Figure 10 the vibration emissions could

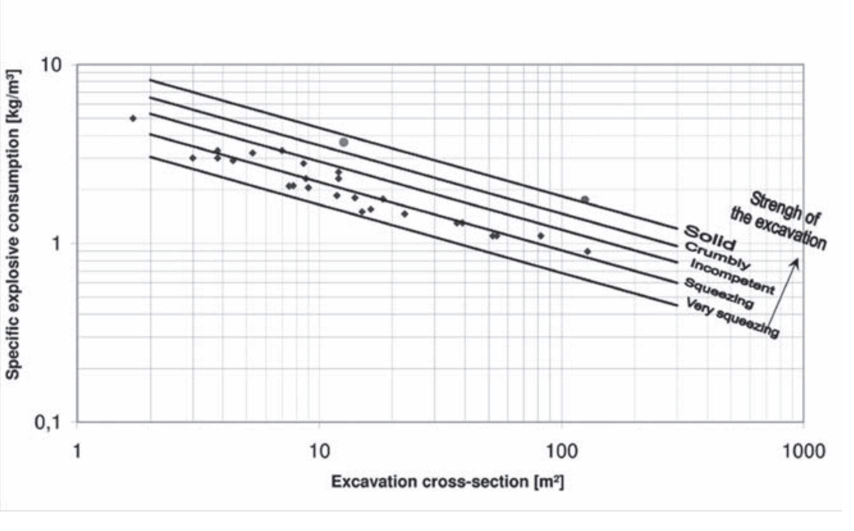

Figure 8. Relationship between specific consumption of explosives and the cross-sectional area of the excavation (stope or tunnel face) as well as the rock fracture strength [Mueller & Pippig 2011b].

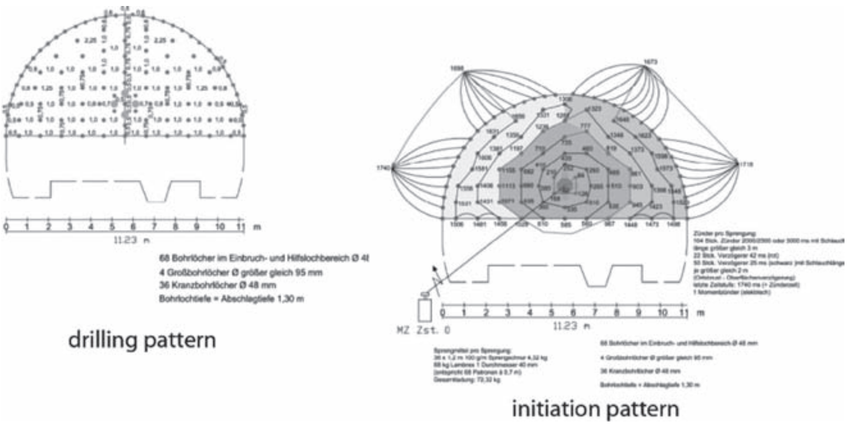

Figure 9. Drill and firing pattern for the roof of the cap of the Markovec tunnel.

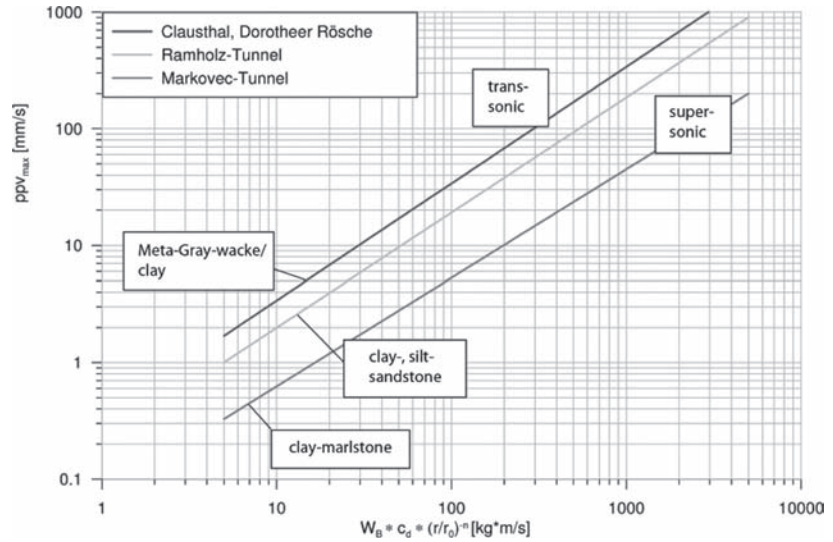

Figure 10. Regression lines in the peak particle velocity (ppv) versus momentum- distance relationship for the underground tunnel advance showing different degree of sonic action.

be adapted without any problems to suit the limits of 7 mm/s at a distance of 20 meters.

Figure 10 shows again that underground blast operations are likewise controlled by the sonicity, i.e. by the ratio of the wave speeds between rock and explosive.

Finally, Figure 11 shows the regression lines of measurements performed in a large number of blast operations in various rock masses, very different blast conditions including surface and underground blast operations.

Practical conclusions:

– The new advanced blasting technology which is based on the effect of sonicity can easily be applied to underground blast operations where the state of stress is three-dimensional and certain special conditions may apply.
– The new predicitive relationships regarding vibration emissions around blasts are valid for all blast works under ground and for all kinds of blast jobs such as tunnels, caverns, large excavations, mining, stoping, block caving, etc. (Fig. 11).

Finally, a most important conclusion has been drawn from hundreds of practical blast works and also supported by theory:

– There does not seem to exist a universal relationship for the assessment and judgement of blast vibration emissions.

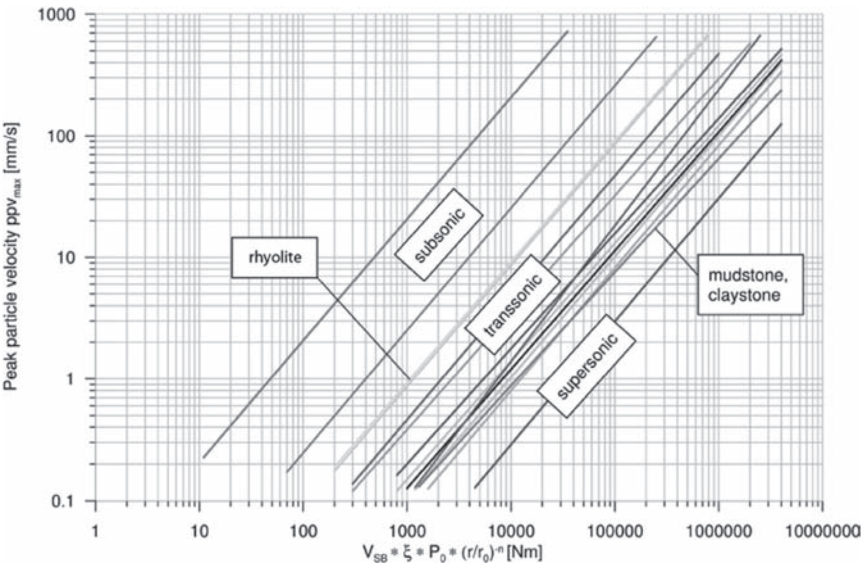

Figure 11. Compilation of regression lines of the fictitious energy versus distance relationship refering to various blast operations and showing the level sonicity of the blast operations.

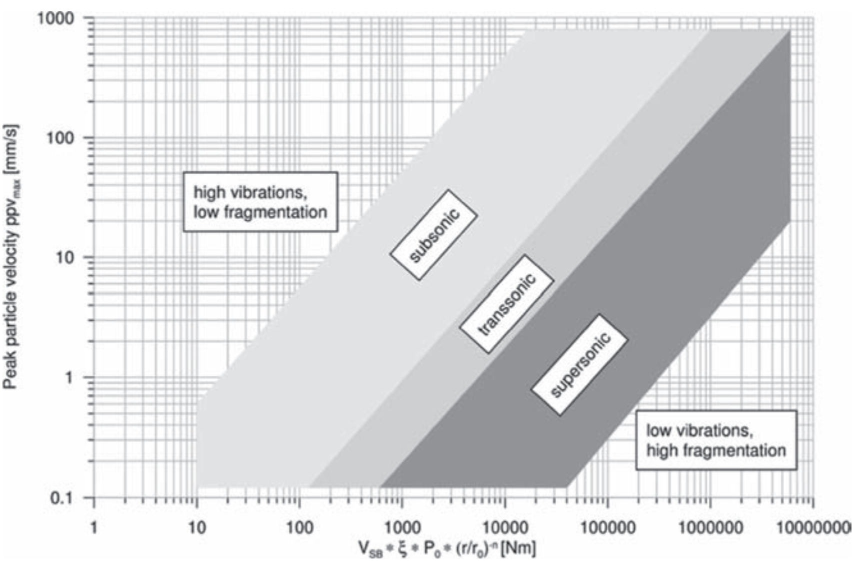

Figure 12. The statistically confirmed dependence of the peak particle velocity (ppv) of the fictitious energy-distance quantity $V_{SB}\ \xi\ P_o\ (r/r_o)^n$ demonstrating the validity of the sonic effect.

3 SUMMARY, CONCLUSIONS AND FUTURE PERSPECTIVES

The aim of this contribution was to present a new advanced blast technology which is based on the exploitation of the principle of sonicity. The strong relationship between fragmentation and vibration emission confirms the importance of the sonic effect in blasting.

It was found from the re-evaluation of numerous blast operation performed in the past, and confirmed by theory that, with increasing sonicity of the blast operation, the blast vibration levels will decrease while the fragmentation of the rock mass will dramatically increase (see Figure 12). The position of the regression lines in Figure 12 is indicative of the level of sonicity. When the regression line moves to the left (right) hand side, the intensity of the blast induced ground vibrations increases (decreases) while at the same time the degree of fragmentation is reduced (enhanced).

The *principle of sonicity* is valid for surface blast operations as well as for underground blast works.

Based on the foregoing practical and theoretical analyses the following conclusions can be drawn:

– The sonic interaction of the explosives with the material to be blasted, in quarrying and mining usually rock, will exercise an influence and fundamentally change the existing traditional blasting technology, primarily the drill and firing methods.
– The new advanced blasting technology allows for almost unlimited enlargement of the blast operations without running the risk of increasing the level of vibration emissions. This implies that the number of blasts can appreciably be reduced.
– Using the new advanced sonicity-based blasting technology the blast engineer can now optimize the blast operation with respect to improving the fragment size distribution, i.e. the muckpile can be optimized and at the same token the level of ground vibrations is reduced.
– Vibration measurements in the near-field (distance smaller than 100 m) should be performed using strain sensors.
– Blast vibration emissions can essentially and systematically be controlled by physically sound and satistically confirmed relationhips which are based on the principle of sonicity.
– The design of surface as well as underground blast operations should be made according the results of the principle of supersonicity.

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REFERENCES (PART 1 & 2)

Baumann, I. & Müller, B. 2000. Neues Messverfahren für die Erfassung von Sprengerschütterungen und anderen dynamischen Einwirkungen im Bauwesen. Spreng-Info 22 (2): 19–32.

Cooper, P.W. 1996. Explosives Engineering. Wiley VCH, USA.

Daehnke, A. & H.P. Rossmanith 1997. Reflection and refraction of plane stress waves at interfaces modelling various rock joints. Fragblast 1:111–231.

DIN Taschenbuch 289 2001. Schwingungsfragen im Bauwesen (Vibrations in civil engineering). Berlin—Wien—Zürich: Beuth Verlag GmbH.

Hustrulid, W. 1999. Blasting Principles of Open Pit Mining—Vol. 1, General Design Concepts. Vol. 2, Theoretical Foundation. Rotterdam: Balkema.

Müller, B. et. al. 2001. A momentum based new theory of blast design. In: 10th High Tech Seminar, 22–26 July 2001, Nashville, USA, V: 3–44.

Müller, B., Hausmann, J. & Niedzwiedz, H. 2009. Sprengtechnik nach neuesten Erkenntnissen—Grundsätze, Möglichkeiten und Konsequenzen (Blasting technology based on recent findings—Fundamentals, possibilities and consequences). Spreng-Info 31.

Müller, B., Lange, P. & Pippig, U. 2011. Die sonische Wirkung von Sprengungen—Spreng-Info 33(3):8–19.

Müller, B. & Pippig, U. 2011a. Praktikable Klassifikationen von Festgesteinen und Festgebirgen für das Bauwesen sowie den Bergbau. Felsbau Magazin 2011/1:10–31.

Müller, B. & Pippig, U. 2011b. Physikalische Zusammenhänge revolutionieren die Bohr- sowie Sprengtechnik und ermöglichen eine statistisch gesicherte Erschütterungsprognose. Felsbau Magazin 2011/4:253–272.

Rossmanith, H.P., Uenishi, K. & Kouzniak, N. 1997. Blast wave propagation in rock mass—Part. 1: Monolithic medium. Fragblast 1:317–359.

Rossmanith, H.P. et. al. 1998a. Der Einfluss der Detonationsgeschwindigkeit auf das dynamische Verhalten des Gebirges und die Bruchentwicklung. Spreng-Info 20(2):27–34.

Rossmanith, H.P., Uenishi, K., Kouzniak, N. & Daehnke, A. 1998b. Der Einfluss der Detonationsgeschwindigkeit auf das dynamische Verhalten des Gebirges und die Bruch-entwicklung (The influence of velocity of detonation on the dynamic behavior of the rock mass and the development of the fracture network). Spreng-Info 20(2):27–34.

Rossmanith, H.P. 2002. The use of Lagrange diagrams in precise initiation blasting—Part 1: Two interacting blastholes. Fragblast 6(1):104–136.

Rossmanith, H.P. & Kouzniak, N. 2004. Supersonic detonation in rock mass—Part 2: Particle displacements and velocity fields for single and multiple non-delayed and delayed detonating blastholes. Fragblast 8(2):95–117.

Rossmanith, H.P. & Müller, B. 2010. Success in Advanced Blasting on the Basis of sonicity—or—What a Blaster Should Know about Wave Dynamics in Rock. Blasting and Fragmentation 4(1):1–34.

Uenishi, K. & Rossmanith, H.P. 1998. Blast wave propagation in rock mass—Part 2: layered media. Fragblast 2:39–77.

Vanbrabant, F., Chacon, E.P. & Quinones, L.A. 2002. P and S Mach waves generated by the detonation of a cylindrical explosive charge—Experiments and simulations. Fragblast 6(1):21–35.

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Assessing a risk analysis methodology for rock blasting operations

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ABSTRACT: Rock Blasting is a primary activity in mining and civil excavations, carried on every day in thousands of locations around the world. Risk Management in blasting activities involves personal Safety and Health as an imperative primary concern, but other kind of risks are involved in the employ of explosives. An Economical Risk appears as a concern when dealing with productivity targets or with mine assets value: incorrect fragmentation result or disrespect of excavation contours of the blast can affect the downstream productivity; equipment damage due to flyrock impact can lead to premature depreciation of the mining assets. Environmental Risk, moreover, is a daily problem dealt by companies that operate with blasting activities in an anthropic environment: ground vibration and airblast overpressure control are getting ever more a central aspect in blast planning and management. Modern blast planning and design are evolving to face these challenges in risk management. The present paper proposes a risk management approach to rock blasting activities. Statistics of blast-related errors are presented as a basis for calculating the frequency of risk-related occurrences. System deviations are described through the main risk analysis techniques, such as HAZOP. Inherent risk is therefore calculated, and Rick Matrixes are created. The most common initiation systems are finally analyzed on the basis of how each one can deal and prevent risk-related deviations and errors.

1 INTRODUCTION

A mining and civil rock blast is a sudden activity, characterized by a large release of energy in a small amount of time, typically in the order of fraction of seconds, and is the results of a longer previous activity of planning, drilling, charging and connecting operations. Due to its rapid happening, there is no time for remediation when the firing signal is given, and the result of the preparing operations can only be a success or a threat at the blasting time, with no mid-term solutions. According to the Blaster´s Handbook of the International Society of Explosives Engineers (ISEE, 2011), a preeminent international reference for Blasting Activities, "With blasting, the two main possibilities are the successful and uneventful completion of the project or an unplanned event (an accident/occurrence) that puts the job in jeopardy". This peculiar condition gives to the blasting operations a critical risk condition. The imperative necessity is to undertake every risk avoidance policy at the time of planning and preparing the blast, since no remediation is possible afterwards. Unplanned consequences in rock blasting mainly lead to three types of risk:

a. Health and Safety Risk: an imperative primary concern, that can cause injuries and/or fatalities due accidents directly related with explosives (e.g. detonation of blasting caps during handling, premature blast with operators still on the site…) or indirectly related to the explosive itself (e.g. operators being hit by flyrocks)
b. Economical Risk: the result of the blast has an impact on all the downstream operations. When dealing with productivity targets, incorrect fragmentation result or disrespect of excavation contours of the blast can result in cost overruns and project delays; when dealing with mine assets value, equipment damage due to flyrock impact can lead to premature depreciation of the mining assets
c. Environmental Risk: the environmental impact of the blast is a daily problem dealt by companies that operate within anthropic environments: exceeding thresholds in ground vibration and airblast overpressure can result in citations from regulatory agencies, legal actions, mandatory refund costs and negative publicity.

According to the ISEE Blaster´s Handbook, the methods of handling blasting risk are: i) Risk Rejection: the decision to avoid to take a risk or to afford a risk-potential situation (the so-called "zero-solution"); ii) Risk Transfer: the legal transfer of the risk to third figures, such as external contractors; iii) Risk Reduction: the process of adopting procedures and employing assets to prevent the occurrence or reduce the impact of an unplanned event; iv) Risk Assumption: assuming the residual risk, after having taken all the feasible precautions to reduce the risk as low as possible. Risk Reduction and Management is the job of the explosives engineer, in order to be able to handle every possible tool at the time of the decision-making. Modern blast planning and design are evolving to face theses challenges in risk management. The present paper reviews the main applications of vanguard intelligence in Blasting Risk Management.

2 LITERATURE REVIEW OF BLAST-RELATED INJURIES

Various studies have been lead in the past applying risk analysis techniques to aspects related to blasting in mining activities. The starting point for this kind of study should be a reliable statistic base of data about common quarry accidents with their magnitude and frequency. In this sense, great importance have the database of OSHA website (United States department of labor) and in particular the one of NIOSH (National Institute for Occupation, Safety and Health) where an entire portion is dedicated to the Mining industry. Another important database that can be used for the elaboration of statistics is the one of the Australian Queensland Government, Mining and Safety section. Verakis (2006, 2011) summarized blasting accident data of the period 1978–2008 for all types of surface and underground mining operations in the United States. The review of blasting accidents is intended to emphasize the continuing need for safe blasting practices, education, development of improved technology, and the observance of Federal, State, and local requirements for mine blasting operations. Bajpayee et al. (online) analyzed the issue of blasting area security. After collecting statistics of 1978–2003, they noticed how 80–90% of injuries can be avoided with training of workers and improving the procedures necessaries to clearly define and evacuate the blast area bounds and to protect the blasters with adequate shelters. Another study of Bajpayee et al. (2002, 2004) focused more or less on the same aspects, reviewing injuries occurred during the years 1978–1998. Also in this case a prevention approach, consisting of behavioral/educational, administrative/regulatory, and engineering interventions was seen as the most effective way to achieve mitigation. Little (2007) studied the problem of rock fragments´ propulsion during the blasting analyzing all the most common mechanisms and the consequent vulnerability of the nearby areas through a wide series of case studies. Kecojevic and Radomsky (2005) studied the same problem with particular attention to the possible causes in blasting design. According to the authors the best way to reduce the risk consists in the workers' training and in applying the state-of-the-art technology. Zhou et al. (online) applied the fault tree analysis (FTA) method is as a tool to analyze the risk associated with blasting flyrock. The flyrock accident is considered as a combination of events and relations components, and the fault trees are established to delineate the interrelationships of these components. Mainiero et al. (online) analyzed the problem of toxic fumes that follow a detonation, with particular attention to the possible measures adoptable for the reduction of the risks associated to this phenomenon. Potvin (2009) analyzed the risk associated with mining induced seismicity, one of the major threats to the safety and sustainability of deep underground mines. He described techniques (basically ground support and re-entry time) that allow site practitioners to efficiently control such risks in mines and reported some case studies referred to Australian mines. Maier (2000) presented a list of the most common risks that can occur in mining activities; in fact the first step for prevention and mitigation consists in a clear identification of the biggest threats. He then proposed a series of simple and intuitive measures that should be followed in the practical activity in order to lower the risk level. Ehnes et al. (2000) collected a series of statistics referring almost to the entire 20th century from a trader association in order to acquire information useful to improve the already existing laws and regulations. Taylor (2011), starting from collection of statistics from Canada and USA, underlined the importance of communication and training of the workers in order to prevent explosive-related accidents in mines.

3 RISK ANALYSIS AND HAZOP

Risk analysis can be defined as a technique used to evaluate and analyze risk, defined as a technical measure able to assess the distance from safety of a system or situation. A unique and worldwide valid definition of risk is really hard to find, as underlined by Kaplan (1997), because its meaning can vary according to the subject and the field analyzed. The common accepted definition of risk (R) is the product of frequency of occurrence of a negative event (F) and its damage (D) (in terms of safety, economical loss, reputation…), as expressed in Equation 1.

$$R = F \times D \tag{1}$$

In order to obtain a quantitative risk output is therefore necessary to have numerical values of frequency and damage. For a quantitative stage of risk evaluation, tables can be created to define generic levels of probability and damage, and associate numerical values to them. For example, in the a table defining the frequency of occurrence of a risk-related occurrence, it is possible to divided four different levels of frequency: very low = 1, low = 2, high = 3, very high = 4. Similar tables can be created to define the magnitude of the damage consequent to the risk occurrence, and this damage can be further subdivided in different typologies. In our case, for example, it has been divided into safety-related, environmental, and economical damage. The result of the product between the two parameters mentioned above can be clearly shown through a Risk Matrix. In these matrixes, conventionally frequency is plotted on the Y axis and the magnitude of damage on the X axis. It is hence possible to define three different risk's zones. In the example of a 4×4 matrix (four levels of probability of occurrence and four levels of magnitude of damage), the risk zones are:

– High risk zone, for values from 9 to 16, usually identified with the red color
– ALARA (As Low As Reasonably Achievable) zone, for values from 4 to 8, usually identified with the yellow color
– Low risk level, for values under 4 usually identified with the green color.

This matrix is extremely helpful to underline the most critical threats and better direct the prevention and mitigation actions. Various techniques have then been developed in the last decades in order to lead in a complete and systematic way the following step of the qualitative analysis, as to say hazard identification; one of the most used is the "hazard and operability study" (HAZOP). A HAZOP can be defined as a structured and systematic examination of a planned or existing process or operation in order to identify and evaluate problems that may represent risks to personnel or equipment. The HAZOP technique, born in 1983, was initially developed to analyze chemical process systems, but has later been extended to other types of systems and also to complex operations and to software systems. The essential steps of this technique can be resumed as follows:

– *Definition of the system*: Identification of the main processes and/or working phases that take place in the system;
– *Identification of control parameters*: Definition of a certain number of parameters useful to control the state of the operation considered;
– *Identification of the key word*: For each control parameter it is possible to individuate key words to describe the condition of the parameter itself (i.e. 'high', 'low', 'yes', 'no', 'open', 'closed')
– *Identification of possible deviations*: Referring to the two previous steps, it is then necessary to describe in what all the possible deviations from the ideal condition consist;
– *Causes-Effects*: After the clear identification of the deviation from the optimum condition, is necessary to list both the possible causes that can lead to that and the possible consequences;
– *Frequency*: According to the frequency tables previously elaborated on the basis of statistic, is necessary to assign a value of frequency rate of occurrence to the event analyzed. The frequency considered usually is the "Frequency of the cause", but when this data is not available is possible to refer to the "Frequency of the effect", as explained in Table 2;
– *Damage*: As in the previous point, it's necessary to assign a numerical value to the damage consequent to the deviation: for an higher detail, it is possible to divide damage into Safety, Economic and Environmental;
– *Risk assessment*: As specified above, the final result of the analysis should be the achievement of the risk value associated to the operations and processes considered. This value is obtainable through the product between frequency and the various types of damage considered;
– *Countermeasures*: This step, one of the most important, consists in the individuation of all the existing ways in which is possible to avoid the deviation or at least mitigate the negative effect that it induces on the system.

In the following paragraphs the HAZOP technique will be applied to the blasting operations in quarries in order to underline the biggest threats and suggest how the risk can be prevented or mitigated.

4 STATISTICAL ANALYSIS OF BLAST-RELATED ACCIDENTS

The database used for this analysis is the extensive and detailed one made available by the Mining and Safety section of the State of Queensland Government of Australia. This database contains very detailed information of blast-related accidents per year, including a description of the event. For the most frequent type of accident (misfire, see Table 1), the causes of it are reported as well (see Figure 1 and 2 of the previous page). It appears evident that the most common cause of misfire is related with the initiation system: either a failure of it, or an error in the initiation or firing procedure.

Table 1. Statistics of blast-related incidents per type of deviation. Both injury and on-injury related accidents are accounted.

Year	2005 Number	2005 %	2006 Number	2006 %	2007 Number	2007 %	2008 Number	2008 %	2009 Number	2009 %	Average per year
Misfire	46	77	43	93	156	92	190	91	223	92	132
Fly rocks	11	18	2	4	11	7	13	6	1	0	8
Uncontrolled shock wave	1	2	0	0	0	0	0	0	0	0	0
Premature blast	1	2	1	2	1	1	3	1	2	1	2
Fumes	0	0	0	0	1	1	2	1	15	6	4
Air overpressure/ vibration	1	2	0	0	0	0	0	0	1	0	0
Total	60	100	46	100	169	100	208	%	242	100	

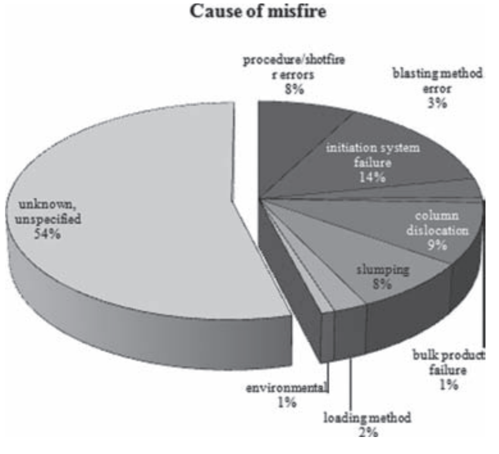

Figure 1. Causes of misfire events (Source of data: Australian Queensland Government, Mining and Safety section).

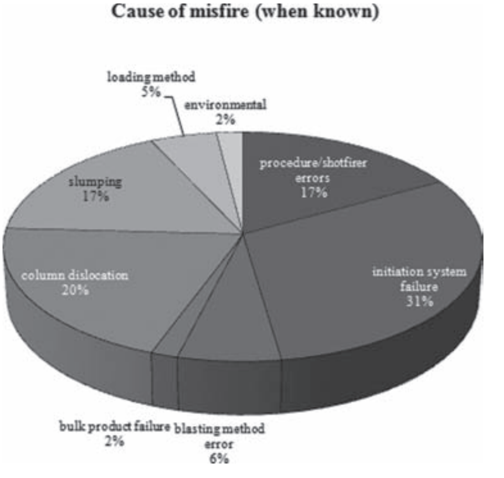

Figure 2. Known causes of misfire events (Source of data: Australian Queensland Government, Mining and Safety section).

5 HAZOP OF BLASTING OPERATIONS

In order to perform a HAZOP analysis of blasting operations, some definitions must be given. The most important difference with normal risk analysis operations is the definition of two types of risk: a risk related to the cause of the event, and a risk related to its effect. The reason of this distinction is the availability of consistent statistics. The Australian database, although being very extensive, contains statistics on the cause of a deviation only when this is the most frequent one: misfire (as specified in paragraph 4 and Figure 3). For all the other cases, statistics on the frequency of occurrence are only available on the effect of the deviation (the content of Table 1). Hence, the necessity of the two definitions of risk, that are resumed Table 2.

As specified in paragraph 3, qualitative classes of frequency and damage must be given class numerical values (generally from one to four) in order to use them in the risk calculation. The definitions and numerical values of frequency and magnitude of damage used for the HAZOP performed in this study are given in tables 3 and 4.

In order to simplify the HAZOP procedure, codes have been assigned:

- in the column "cause" roman numerals correspond to the causes of misfire reported in Figure 3 (in the same sort order). Words are used to describe the cause when the event is not a misfire
- in the column "Effect on the system" arabic numerals correspond to the events of table 1 (in the same sort order as well). Words are used to describe the event when this is not contemplated in the list of Table 1.

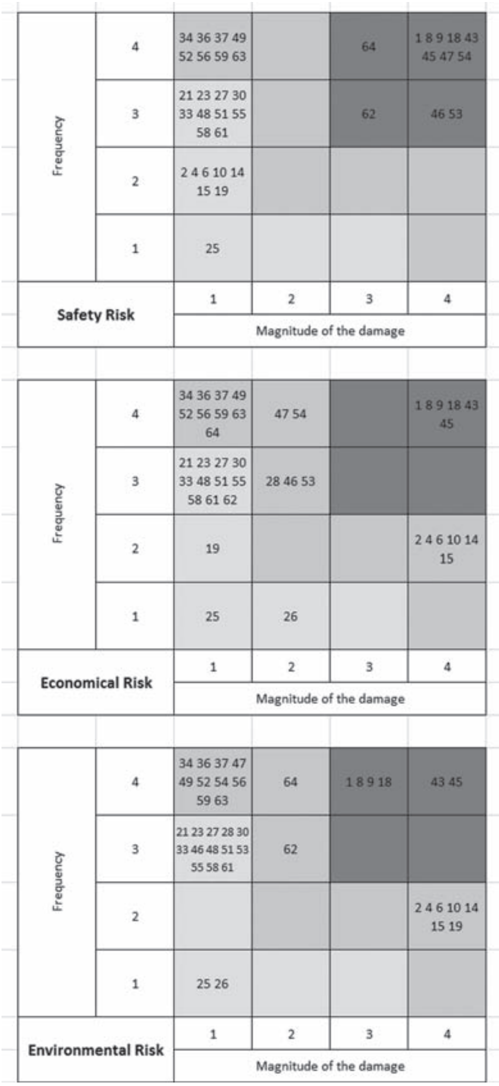

Figure 3. Risk matrixes: the numbers refer to the deviation indexes of the HAZOP spreadsheet (ANNEX1).

The HAZOP spreadsheet is very large to be shown in the body of the article; it has been attached at the end of the paper and adapted to be readable. Its results have been plotted in the form of risk matrixes. According to the definitions of paragraph 1 and tables 2, 3 and 4 three types of risk have been calculated, and therefore three risk matrixes have been created (Figure 3): a matrix of safety risk, a matrix of economical risk and a matrix of environmental risk.

Looking at the three matrixes showed above, it appears how the biggest number of high-risk

Table 2. Types of risk considered in this analysis.

Type of risk	Symbol	Definition of risk	Risk calculation
Risk due to the Cause	R_C	Risk of the effect happening, due to a specified cause	Frequency of the cause times Damage of the effect
Risk of the effect	R_E	Risk of the effect happening, without considering a specific cause	Frequency of the effect times Damage of the effect

Table 3. Definition of frequency classes.

		Events expected in one year	
Value	Frequency	Frequency of the cause	Frequency of the effect
1	Very low	5	0
2	Low	<10	<1
3	High	<15	<4
4	Very high	<20	>4

events are related to the safety aspects (12 events compared to 6 events both for environmental and economical aspects).

The processes that imply the higher risks are the Fire impulse signal phase and the Design one, followed by the Blast area security; lower risks are related to the Priming, Charging and Connection phases (see Annex 1).

Being the biggest risks related to safety, it's clear how the prevention through design and the careful work's execution (that can be achieved through workers' formation, information and training) are essential not only to avoid economical losses and environmental damages but most of all to prevent workers' injuries.

As underlined above, the risk level is the product between Frequency and Damage: in order to reduce it is necessary to try to lower these two components. Since blasting activity is characterized by an enormous energy release in few milliseconds, it appears very difficult to manage to mitigate the entity of damage once the chain of events has started; for this reason great importance is acquired by the activities aimed at reducing the Frequency, as to say be sure to avoid as much as possible the dangerous events.

The last column of the HAZOP has been dedicated to the detection of countermeasures that can be applied in order to deal with the deviation and reduce the risk (in terms either of reducing

Table 4. Definition of damage classes.

Value	Safety damage	Economical damage	Environmental damage
1	Neglectable (no harm to people)	Neglectable (Less than one working shift lost)	Neglectable (No significant effects)
2	Slight damage to people	Damage to moveable equipment or less than 3 workdays lost	Small-scale (local) effects (inside the working area)
3	Serious damage to people	Damage to company's structures (no legal issues with third parties) or more than 3 workdays lost	Medium-scale effects (outside the perimeter of the working area)
4	Death	Damage to third party's property and legal issues	Large scale effects (far away from the working area)

Table 5. Definition of the types of countermeasure.

Type of countermeasure	Definition
Operational	Countermeasure dealing with the phases of execution of the work
Design	Countermeasure dealing with the design of the blast
Initiation	Countermeasure dealing with the type of the initiation circuit
Procedural	Countermeasure dealing with the phases and order of procedures
Safety practices	Countermeasure dealing with job safety practices

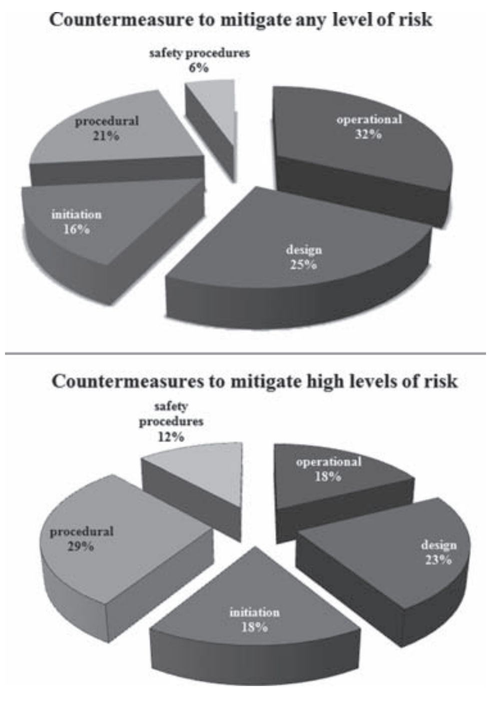

Figure 4. Distribution of countermeasures to mitigate the risk (per type).

the frequency of occurrence or mitigating the damage). These countermeasures have been classified per type. The definition of the types is given in Table 5. Their distribution of occurrence is shown in Figure 4, analyzing separately the occurrence of solutions that mitigate high levels of risk (the red zone in the risk matrixes). Both for any level and for high levels of risk, the most frequent countermeasure to mitigate it appears to be the improvement of the operational phases and of the blast design. On the other side, little space is given to procedures directly related to safety. As to say: risk reduction is achieved at the root of the process. Trying to ameliorate the execution of the job site work and a good and careful project can strongly reduce the need of further interventions of mitigation.

6 CONCLUSIONS

Due to its rapid nature, blasting activities give no time for remediation: when the firing signal is given, only a success or a threat at the blasting time can happen, with no mid-term solutions. From here comes the imperative necessity of a careful and dedicated evaluation of the risks of the preparing operations. Starting from a series of statistic data collected by the Queensland Government of Australia, statistics have been elaborated in order to underline the incidence of accidents of various types over 5 years (2005–2009). Since the higher number of accidents emerged to be related to misfires, the causes of misfires itself have been investigated; in this way emerged that initiation system failure is the most common known cause.

The HAZOP method has then been applied, in order to clearly understand the process deviations with their causes and effects, and to underline the frequency of occurrence of the event considered and the damage (safety, economical and environmental) associated with each of them. The risk values resulting from the HAZOP have been analyzed through Risk matrixes to identify the higher weakness of the system. In this way it was possible to assume the most effective countermeasures that could be adopted in order to prevent the process deviation.

It emerged that focusing on operational, design and procedural countermeasures can avoid more than 3/4 of the accidents: this underlines how proper training of the workers and careful working processes are fundamental, together with a suitable projectapproach adapted step by step to the specific characteristics of the site.

REFERENCES

Bajpayee, T.S., Harry, A., Verakis, H.C., Thomas E. Lobb. Blasting Safety—Revisiting Site Security. In *http://www.cdc.gov/niosh/mining/pubs/pdfs/bsrss.pdf*

Bajpayee, T.S., Rehak, T.R., Mowrey, G.L., Ingram, D.K. 2002. Summary of Fatal Accidents Due to Flyrock and Lack of Blast Area Security in Surface Mining, 1989 to 1999. In *Proceedings of the 27th annual conference on explosives and blasting technique, vol. I ISEE,* Cleveland, USA.

Bajpayee, T.S., Rehak, T.R., Mowrey, G.L., Ingram, D.K. 2004. Blasting Injuries in Surface Mining with Emphasis on Flyrock and Blast Area Security. In *Journal of safety research 35*: 47–57.

Ehnes, H., Friesen, G., Pichl, W. 2000. Explosives accidents in the federal republic of Germany-An analysis of the accident rate and conclusions for the body of regulation. In *Explosives & Blasting technique*: 49–51. Holmberg (ed.), Rotterdam: Balkema.

Kaplan, S. 1997. The words of Risk Analysis. In *Risk Analysis, Vol. 17, No.4.*

Kecojevic, V., Radomsky, M. 2005. Flyrock phenomena and area security in blasting-related accidents. In *Safety Science. Vol. 43, Issue 9:* 739–750. Elsevier.

Little, T.N. 2007. Flyrock risk. In *EXPLO Conference*: 35–43. Wollongong, NSW.

Maier, A.A. 2000. Evaluation of the kind of hazards and risks encountered with explosives when blasting in quarries-A simple concept. *Explosives & Blasting technique*: 59–61 Holmberg (ed.), Rotterdam: Balkema.

Mainiero, R.J., Harris, M.L., Rowland III, J.H. Dangers of Toxic Fumes from Blasting. Available at: *http://www.cdc.gov/niosh/mining/pubs/pdfs/dotff.pdf*

Occupational Safety & Health Administration, United States Department of Labor: *http://www.osha.gov/oshstats/work.html*

Potvin, Y. 2009, Strategies and tactics to control seismic risks in mines, *Journal of the Southern African Institute of Mining and Metallurgy*, 109, 3, pp. 177–186.

Taylor, K. 2011. The safety of explosive handling and blasting in the mining industry. In *Proceeding of the thirty-seventh annual conference on explosives and blasting Technique, ISEE*: 411–418. San Diego, CA, USA.

The National Institute for Occupational Safety and Health (NIOSH) http://www.cdc.gov/niosh/

Verakis, H.C. 2006. An Examination of Mine Blasting Accidents Over a Quarter of a Century. In *International Society of Explosives Engineers 2006 Volume 2*. 1–11.

Verakis, H.C. 2011. Flyrock: a continuing blast safety threat. In *Proceeding of the thirty-seventh annual conference on explosives and blasting Technique, ISEE, San Diego, CA, USA:* 731–739.

Zhou, Z., Li, X., Liu, X., Wan, G. Safety Evaluation of Blasting Flyrock Risk with FTA Method. Available at: *http://www.lib.hpu.edu.cn*

ANNEX 1.
HAZOP SPREADSHEET OF BLASTING OPERATIONS.

Process	Parameter	Key Word	process deviation	index	Cause	Effect on system	Frequency of cause	Frequency of effect	Safety Damage	Economical Damage	Environmental Damage	Safety Risk	Economical Risk	Environmental Risk	Countermeasure	Type of countermeasure
Design	holes inclination	high	high inclination of	1	Drilling or design error	2		4	4	4	3	16	16	12	# careful drilling operations	operational design
		low	bad transmission	2	Drilling or design error	3		2	1	4	4	2	8	8	# careful drilling operations	operational design
	subdrilling	high	-	3	Drilling or design error	-		-							# careful drilling operations	operational design
		low	bad transmission of detonation wave to rock	4	Drilling or design error	3		2	1	4	4	2	8	8	# careful drilling operations	operational design
				5	Drilling or design error	bad fragmentation and profiling				2	1	0	0	0	# careful drilling operations	operational design
	burden	high	too high distance of maximum resistance of	6	Drilling or design error	3		2	1	4	4	2	8	8	# careful drilling operations	operational design
				7	Drilling or design error	bad fragmentation and profiling				2	1	0	0	0	# careful drilling operations	operational design
		low	insufficient distance of	8	Drilling or design error	2		4	4	4	3	16	16	12	# careful drilling operations	operational design
	powder factor	high	too high energy release	9	Design or charging error	2		4	4	4	3	16	16	12	# careful charging operations	operational design
				10	Design or charging error	3		2	1	4	4	2	8	8	# careful charging operations	operational design
		low	insufficient energy release	11	Design or charging error	Bad fragmentation				2	1	0	0	0	# careful charging operations	operational design
	delay time	high	bad cooperation	12	Design or execution	Bad fragmentation and profiling				2	1	0	0	0	# careful execution # redesign the pattern	operational design
		No	no fragmentation	13	Design or execution	Bad fragmentation				2	1	0	0	0	# careful execution # redesign the pattern	operational design
		low	overposition of shockwave	14	Design or execution	3		2	1	4	4	2	8	8	# careful execution # redesign the pattern	operational design
	charge per delay	high	too high energy release	15	Design or execution	3		2	1	4	4	2	8	8	# careful execution # redesign the pattern	operational design
		low	-	16	Design or execution							0	0	0	# careful execution # redesign the pattern	operational design
	stemming	high	surface layer with bad	17	Design or execution	Bad fragmentation				2	1	0	0	0	# careful execution # redesign the pattern	operational design
		low	insufficient containing of gas pressure	18	Design or execution	2		4	4	4	3	16	16	12	# careful execution # redesign the pattern	operational design
				19	Design or execution	6		2	1	1	4	2	2	8	# careful execution # redesign the pattern	operational design

Priming	blasting cap - cartridge coupling	good	-	20	-	-						0	0	0		
		bad	Bad transmission of shockwave - possibility for	21	I	1	3		1	1	1	3	3	3	# check the ignition system # careful procedure's execution	initiation
	integrity of cap line	good	-	22		-			1	1	1	0	0	0		procedural
		bad	No initiation signal to	23	I	1	3		1	1	1	3	3	3	# check the ignition system	initiation procedural
Charging	explosive column integrity	yes	-	24		-			1	1	1	0	0	0		
		no	incomplete detonation of the explosive column	25	IV	1	1		1	1	1	1	1	1	# careful charging procedure	operational
				26	IV	bad fragmentation and profiling	1			2	1	0	2	1	# careful charging procedure	operational
				27	V	1	3		1	1	1	3	3	3	# careful charging procedure	operational
				28	V	bad fragmentation and profiling	3			2	1	0	6	3	# careful charging procedure	operational
	integrity borehole walls	yes	-	29		-						0	0	0		
		no	incomplete detonation of the explosive column	30	VI	1	3		1	1	1	3	3	3	# careful charging procedure # careful drilling	operational
				31	VI	bad fragmentation and profiling				2	1	0	0	0	# careful charging procedure # careful drilling	operational
Connection	connections integrity	yes	-	32		-						0	0	0		
		no	No initiation signal to blasting cap	33	I	1	3		1	1	1	3	3	3	# careful procedure's execution	procedural
				34	II	1	4		1	1	1	4	4	4	# check the ignition	initiation
	execution of connections	good	-	35		-						0	0	0		
		bad	No initiation signal to blasting cap	36	I	1	4		1	1	1	4	4	4	# careful procedure's	procedural
				37	II	1	4		1	1	1	4	4	4	# check the ignition system	initiation

Blast area security	alarm sound	yes	-	38		-						0	0	0		
		no	Unalarmed personnel	39	Safety procedure	Unsecured Blast area			4	2	1	0	0	0	# careful evacuation and safety procedure	safety procedural
	personnel's evacuation	good	-	40		-						0	0	0		
		bad	Unalarmed personnel	41	Safety procedure	Unsecured Blast area			4	2	1	0	0	0	# careful evacuation and safety procedure	safety procedural
	equipment dislocation	high	-	42		-						0	0	0		
		low	Unprotected equipment in	43	Safety procedure	2		4	4	4	4	16	16	16	# careful evacuation and safety procedure	safety procedural
	equipment protection	good	-	44		-						0	0			
		bad	Unprotected equipment in	45	Safety procedure	2		4	4	4	4	16	16	16	# careful evacuation and safety procedure	safety procedural
Firing impulse signal	pyrotechnic (fuse)	soon	premature detonation	46	I	4	3		4	2	1	12	6	3	# careful procedure's execution	procedural
				47	II	4	4		4	2	1	16	8	4	# check the ignition system	initiation
		late	delay in detonation	48	I	1	3		1	1	1	3	3	3	# careful procedure's execution	procedural
				49	II	1	4		1	1	1	4	4	4	# check the ignition system	initiation
	electric (analogic or digital)	high	-	50	-	-						0	0	0		
		low	Missed detonation signal to some holes	51	I	1	3		1	1	1	3	3	3	# careful procedure's execution	procedural
				52	II	1	4		1	1	1	4	4	4	# check the ignition system	initiation
		soon	premature detonation	53	I	4	3		4	2	1	12	6	3	# careful procedure's execution	procedural
				54	II	4	4		4	2	1	16	8	4	# check the ignition system	initiation
		other than	AC instead of DC signal	55	I	1	3		1	1	1	3	3	3	# careful procedure's execution	procedural
				56	II	1	4		1	1	1	4	4	4	# check the ignition system	initiation
	shockwave (NONEL-type)	yes	-	57		-						0	0	0		
		no	No initiation signal to blasting cap	58	I	1	3		1	1	1	3	3	3	# careful procedure's execution	procedural
				59	II	1	4		1	1	1	4	4	4	# check the ignition system	initiation
	shockwave (detonating cord)	high	-	60		-						0	0	0		
		low	Deflagration instead of detonation	61	I	1	3		1	1	1	3	3	3	# careful procedure's execution	procedural
				62	I	5	3		3	1	2	9	3	6	# careful procedure's execution	procedural
				63	II	1	4		1	1	1	4	4	4	# check the ignition system	initiation
				64	II	5	4		3	1	2	12	4	8	# check the ignition system	initiation

Blasting in Mines – New Trends – Ghose & Joshi (Eds)

Novel blasting techniques for productivity improvement in hard rock underground dolostone mine

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ABSTRACT: The pace of developments in underground blasting technology is insignificant in contrast to developments in opencast blasting. Blasting practice in underground mines has limited flexibility due to many safety criteria, statutory and field constraints. This has narrowed the scope of major modifications in the solid blasting design vis-à-vis the efficiency and yield per round. Central Institute of Mining & Fuel Research (CIMFR), Regional Centre, Nagpur developed and devised two new underground blasting techniques with simple modifications in explosive loading patterns. They are: (i) in-hole delay solid blasting technique (ii) bottom hole decking technique. The first technique was in-hole delay cut blasting technique by inserting multiple delay detonators in cut holes to improve the solid blasting efficiency. The conventionally available resources are used in this technique without violating the statutory guidelines prescribed by Directorate General of Mines Safety (DGMS). CIMFR conducted extensive experimental blasts at hard rock dolostone ore of Tummalapalle Underground Mine for M/s SMSIL to implement the new blasting techniques. The trial blast results indicated improvements in all the parameters like pull, yield per round, powder factor. The overall improvement in pull per round was 28%. The technique also resulted in reduction of ground vibration intensity by 30–40%, which resulted in substantial reduction of rock mass damage. Another blasting method developed was 'bottom hole decking technique' incorporating air-decking at the bottom of the blastholes. In this technique a spacer is to be placed at the bottom of the hole and remaining portion of the hole is conventionally charged. The length of spacer is equal to 10–12% of depth of blasthole. A wooden spacer or any plastic pipe can also be used for decking. The technique was applied in the periphery holes of medium hard to hard rock formations. The trial results with blasts with bottom decking indicated improvements in control of overbreak and rock mass damage. The overall reduction in overbreak was achieved as 40–50%. Application of both the techniques resulted in reduction of rock mass damage by 25–30%. The new blasting techniques also resulted in decrease of specific charge and specific drilling by 14% and 30% respectively. The application of new blasting techniques helped in enhancement of both safety and productivity of hard rock dolostone mine.

1 INTRODUCTION

It is high time to focus upon underground mining vis-à-vis blasting, as the cost of opencast mining is going to increase in near future due to higher stripping ratio as well as environmental concerns. The ever growing demand for metals and minerals is pressing the need for progress of underground blast rounds. But, there are some technical constraints associated with underground blasting. The chances of explosive malfunctioning are high in solid blasting due to close proximity of charges. In coal mine blasting statutory provisions indicate that desensitization of blastholes was very frequent in coal mine when the blastholes were closer than 0.6 m. Katsabanis & Ghorbani (1995) found that the sympathetic detonation might occur if different charges are separated at less than 8 times the hole diameter. Ramulu et al. (2005) applied the in-hole delay solid blasting technique successfully for blasting productivity improvement in coal mines.

There is a restriction for longer rounds of blasts in rock tunnels or drifts due to confinement proportional to area of cross section of the opening. However, plenty of new ideas and efforts are being experimented to improve the yield per blast round and implemented in coal and rock tunnels. It is known that in solid blasting, a cut is blasted initially towards which the rest of the shots are fired. The confinement, which is maximum in the cut holes in absence of any free face, is released to a great extent one the cut is developed and hence, the balance holes are blasted with minimised confinement. The efficiency of a blasting round vastly depends on the success of cut development. Innovations in various explosive accessories like relays, shock tubes and others are applied in opencast blasting not yet introduced in underground metal mines due to field constraint. Hence, blast rounds deeper than 3 m, are not common in India considering the prevalent restrictions. The pull to hole depth ratio also lies in a mediocre range of 0.6–0.7. This paper deals with the case study of successful implementation of two new blasting methods for improved pull as well as pull percentage in a hard rock metal mine.

2 IN-HOLE DELAY SOLID BLASTING

In view of large number of restrictive conditions, an innovative in-hole delay pattern was evolved by the authors to improve the solid blasting efficiency in the coal mines. This essentially includes the use of multiple delay detonators in a single hole so that total permissible explosive quantity is distributed or segmented in different delays which are fired sequentially from the top, where the confinement is originally smaller, to provide less confinement to the charge being fired in the next delay situated in the bottom part of the hole and having originally a larger confinement. Further, multiple delays provide additional time for the burden to be displaced more efficiently. Though this type of delay arrangement may be tried in all the holes for better fragmentation and output, but is especially useful in the cut holes or toe holes, where the confinement is larger than other holes in a round, to reap the major benefits in case of limited availability of delay detonators. The technique is briefly explained in Figure 1, which resembles to the in-hole delay initiation method used in opencast blasting using shock tubes.

The uniqueness of the technique is that it abides by all the existing safety criteria and uses the conventional electric delay detonators, without demanding for extra resources. As the confinement in the cut holes are maximum and the

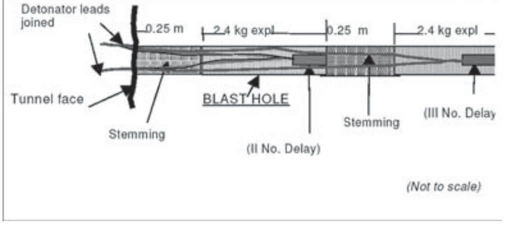

Figure 1. In-hole delay solid blasting technique—a schematic diagram.

blast performance in the underground openings depend mainly on the development of the cut portion, the in-hole delay were used only in the cut holes. The salient features of the in-hole delay pattern are:

– The collar portion of the hole was blasted prior to the bottom portion. Thus, the confinement at the hole bottom was less during firing.
– Mid-column decking between the two charges in a hole was kept at least 0.6 m to avoid sympathetic detonation. This decking provided confinement for the bottom charge.

3 BOTTOM HOLE DECKING TECHNIQUE

The mining industry is striving to enhance the productivity by improving fragmentation to reduce the system cost. In order to achieve this objective, development of new techniques and their application is essential. Ramulu et al. (2002) conducted experiments on bottom decking technique with Plexiglas models and enumerated the advantages in comparison to the other decking methods. The authors at CIMFR, experimented a blasting technique called 'bottom hole decking technique' to achieve the objective of blasting productivity improvement of the mining industry. The technique consists of air decking at the bottom of the blasthole in dry holes by means of a wooden spacer or a closed PVC pipe. Although, practice of air decking is not new thing in blastholes, the concept of inserting bottom hole decking below the explosive column is relatively new. Explosives provide a very concentrated source of energy, which is often well in excess of that required to adequately fragment the surrounding rock material. Blast design, environmental requirements and production requirement limits the degree to which the explosive energy distribution within the blasthole can be significantly altered using variable loading techniques. Use of air-decks provide an increased flexibility in alteration and distribution of explosive charge in blastholes. Attempts were made by Indian researchers to apply the air-decking technique for controlled blasting as well as production blasting to improve the blast fragmentation (Chakraborty & Jethwa 1996, Jhanwar et al. 1999, Chiappetta 2004).

The bottom hole air-decking was developed to avoid the general disadvantages of middle air decking and to simplify the complex charging procedure, associated with it. The design aspects of the technique are explained in the following sections. The bottom hole decking consists of air decking at the bottom of the hole in dry holes by

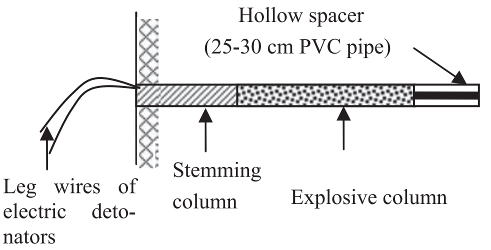

Figure 2. Blasthole charge design for production blasts with bottom air-decking.

means of a spacer or a closed PVC pipe, covered at the upper end. The fume characteristics of the spacer are to be tested before applying in underground coal mine. If blastholes are wet, water decking will be created at the bottom by means of a spacer with a weight attached to it for sinking to the bottom. The diameter of the spacer should be preferably one third of the blasthole diameter for easy lowering and not allowing the charge to go to bottom side while loading. The reported values of air-deck length was taken as basis for optimum bottom deck length which was about 10% of the hole depth (Mead et al. 1993).

The hole contains explosive and stemming column as in conventional loading but with a spacer at the bottom. The principle of bottom hole air decking in achieving optimum explosive energy interaction on rock mass is given below:

i. Reduced shock energy around the blasthole due to cushioning effect of air decking, which otherwise would result in crushing.
ii. Explosive energy-rock interaction is more at the bottom due to relative relief zone existing at that zone.
iii. Effective toe breakage is due to striking and reflection of shock waves at the bottom face of hole.

The procedure and sequence of blasthole loading and initiation for the bottom hole decking are given below:

i. Inserting the spacer in to the hole bottom by stemming rod and loading the primer explosive cartridge attached by delay detonator. The rest of the column charge is loaded conventionally.
ii. Stemming of the hole by proper stemming material, preferably sand mixed clay cartridges or coarser sand cartridges.

The advantages of the bottom air decking technique in comparison to the conventional middle air decking include: The highly confined toe is free of explosive charge but exposed to high concentration shock energy, resulting in good toe breakage and low vibration intensity. The reduced overall peak shock reduces the back break and damage. Blasthole charge design for production blasts with bottom air-decking is Figure 2.

4 FIELD APPLICATION OF NOVEL BLASTING METHODS

In-hole delay cut blasting technique using delay electric detonators and bottom hole decking methods using plastic spacers were used in Tummalapalle Mine Project to improve the pull per round and to reduce the ground vibrations as well as overbreak. The details of the mine and general blasting practice are explained in the following sections.

4.1 *Details of the test site*

Tummalapalle Mine Project is located in Kadapa district of Andhra Pradesh. The area where the Dolostone deposits were found is covered in Survey of India's topographic sheet Nos. 57 J/3 and 57 J/7. From the geological section of the deposit two parallel ore band are established designated as hang wall load and foot wall load. Hang wall load is more uniform in grade, thickness & extent. These two bands are separated by a uniform lean zone. The ore bands are tabular strata bound, non-transgressive in nature with limited variation in grade and thickness along strike as well as in dip direction. Strike direction of the ore body is N 680 W and S 680 E. the average dip of ore body is 150 to 170 due N220 E.

4.2 *Geotechnical information*

From the geological study of the deposit, the mineralization has taken place in Dolostone and the overlying rock is purple shale. Dolostone is stronger than purple shale having a Rock Quality Designation index (RQD) of 83% to 95%. Purple shale has a RQD between 67% and 88%. The average specific gravity of dolostone is 2.80. Compressive strength of dolostone is ranging from 64.7–70 MPa and tensile strength is in the range of 5.6–6.5 MPa. Shear strength of dolostone is about 30 MPa. There are three number of joint sets with joint inclination of 90°. Joint spacing is about 10–20 mm and the openings are sometimes filled by calcite/quartz, veins and hematite.

4.3 *Method of working*

In Tumalapalle underground mine the ore is exploited by breast stopping with ramp in the apparent dip. There are three declines namely east decline, central decline and west decline. Development of east decline started with breast stopping along strike and dip direction but remaining two were worked as tunnel. The method of working for extracting ore is by the by Room and Pillar with a dimension of Advance in Strike Drive (ASD) of 4.5 m width by 3.0 m height and the dimension of decline is 5.0 m x 3.0 m (width x height).

4.4 *Drilling and blasting*

The brief details of drilling and blasting parameters are given below:

- Drilling machine: Hydraulic Jumbo drilling machine (Tire Mounted).
- Boom Length: 5.11 m
- Drill rod length: 3.7 m
- Drill Depth: 3.25 m
- Drill bit Diameter: 45 mm
- Relief hole bit type: Button Bit & chisel bit type
- Relief hole Diameter: 89 mm and 102 mm
- Drilling pattern: Burn cut pattern with 3 to 4 relief holes.
- Type of Explosive: Emulsion
- Diameter of Explosive: 40 mm
- Weight of a cartridge: 0.39 kg
- Strength of Explosive: 80% and 90%
- Velocity of Detonation: 4000 m/s
- Type of Detonators: Long delay
- Length of leg wires: 5 m
- Blasting circuit connection: Series connection

The mine management used to practice a set of blast design parameters at Advance Strike Drive (ASD) before the optimization trials started in the mine. The design and output parameters prevailing at the mine are given in the Table 1. The prevailing conventional blast pattern is shown in Figure 3. Test blasts were conducted by applying the in-hole delay cut blasting and the bottom hole decking simultaneously.

Table 1. Blast design and output parameters prevailing at the Tummalapalle Mine Project.

Parameter	Value
Diameter of blasthole	45 mm
Total no. of blastholes	43
Charge per round	115 kg
Maximum Charge per delay	14.4 kg
Velocity of detonation	3900 m/s
Avg. pull per round	2.5 m
Pull% (Pull/Hole depth)	76%
Specific charge	3.4 kg/m^3
Specific drilling	4.6 m/m^3
Blast vibrations attenuation equation	$V = 1480 (D/3\sqrt{Q})^{-1.8}$
Overbreak	0.4–0.5 m

Ten trial blast were conducted at ASD 4E continuously to test the consistency of the optimized blast pattern. The face dimensions of the ASD were 4.5 m width and 3 m height. The modified in-hole delay cut blasting pattern is shown in Figure 4. Long delay with half second delay was used as delay timing between each delay as it was like a small tunnel. The in-hole delay is applied for the I, II, III, and IV delays of cut holes and bottom decking is applied in the rib and back holes of IX delay as shown in Figure 4. The actual view of blasting face with cut hole charging is shown in Figure 5.

5 TEST BLAST RESULTS

The test blasts with new blast pattern with in-hole delay and bottom decking techniques yielded encouraging results as described in the Table 2. There is substantial overall improvement in all the blast performance parameters (Table 2). The burn cut portion was blasted without any socket as shown in Figure 6. There was hardly 0.1 m sockets at side holes and stopping holes. This might be because of slight angular deviation of periphery holes.

The following inferences are made from the outcome of test blasts with in-hole delay method and bottom-hole decking technique:

i. Test blasts with 3 number of relief holes of 102 mm diameter (reamers) yielded the same results as in case of 4 relief holes of 89 mm diameter, without compromising the over-all progress of parallel cut, which obviously reduces time and efforts of drilling.
ii. In-hole delay cut blasting could give complete pull of blasthole depth with pull to hole depth ratio of 0.95.
iii. Longer cut-holes (3.25 m depth in place of 3.0 m depth) yielded very good results in breaking of maximum burn-cut portion, encouraging to go for deeper holes for more progress per round.
iv. Bottom hole decking with spacer of 0.15 m (PVC pipes) in periphery holes resulted in reduction of overbreak by 0.3–0.4 m and substantial reduction in the intensity of ground vibrations.
v. Application of the productive and controlled blasting techniques resulted in improvements in pull by 0.5–0.6 m, decrease in Specific charge and specific drilling by 0.5 kg/m^3 and 0.6 m/m^3.
vi. There was substantial reduction in ground vibrations from 8.5 mm/s to 4.4 mm/s at 50 m distance. The overbreak was also reduced by 0.3–0.5 m.

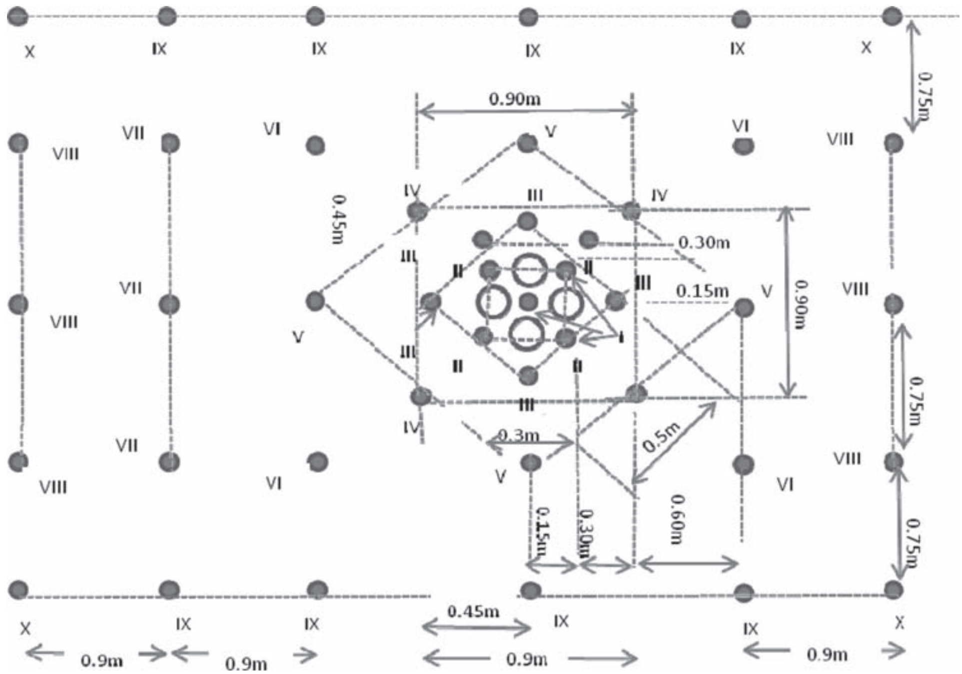

Figure 3. Prevailing conventional blast pattern at Tummalapalle Mine Project (the Roman numbers indicate delay numbers).

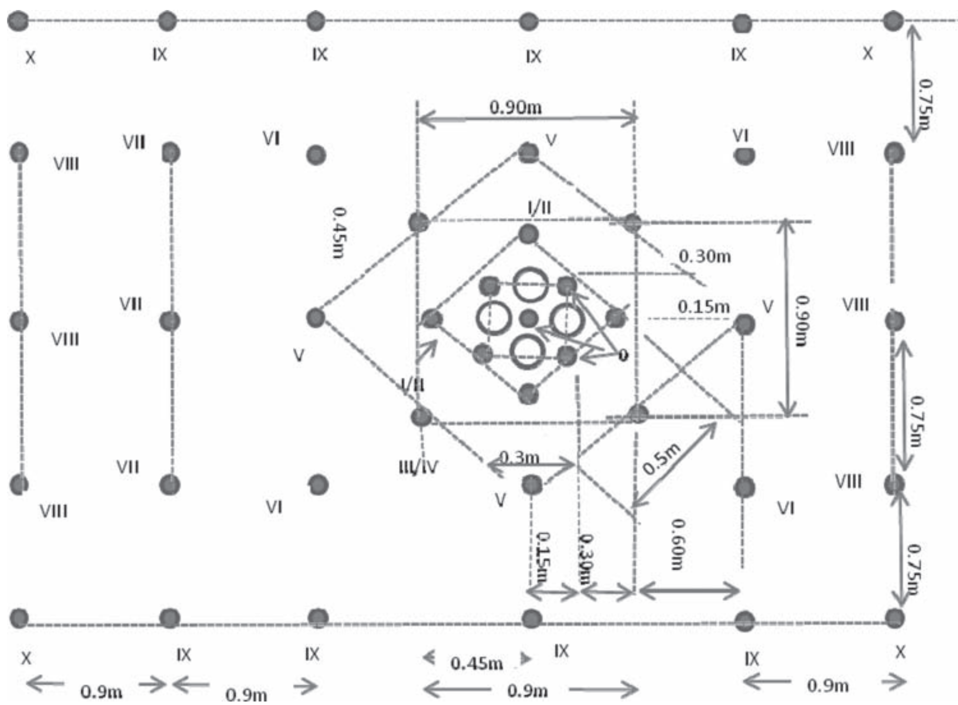

Figure 4. Modified in-hole delay cut blasting pattern at Tummalapalle Mine Project (the Roman numbers indicate delay numbers).

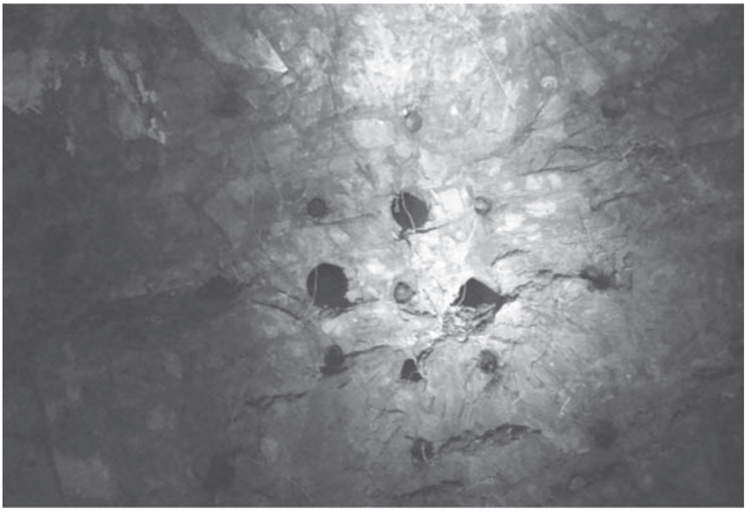

Figure 5. View of application of double delays in cut holes and 3 relievers at Tummalapalle Mine Project.

Table 2. Modified blast design and output parameters with in-hole delay cut and bottom hole decking at Tummalapalle Mine Project.

Parameter	Value
Diameter of blasthole	45 mm
Total No. of blastholes	45
Charge per round	119 kg
Maximum Charge per delay	14.4 kg
Velocity of detonation	3900 m/s
Avg. Pull per round	3.1 m
Pull% (Pull/Hole depth)	95%
Specific charge	2.9 kg/m^3
Specific drilling	3.8 m/m^3
Blast vibrations attenuation equation	$V=1410(D/3\sqrt{Q})^{-1.9}$
Overbreak	0–0.2 m

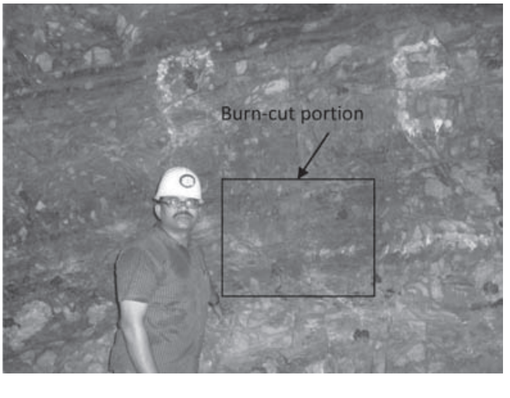

Figure 6. Blasted face with modified pattern without socket at burn-cut portion.

6 CONCLUSIONS

CIMFR developed and applied an innovative in-hole delay cut pattern and bottom hole decking techniques at Tummalapalle Mine Project using electric delay detonators without violating the existing safety criteria. The technique deploys multiple electric delay detonators in a hole, which adds time for the burden displacement, to partition the total charge and firing sequentially from collar to bottom in order to provide less confinement for the bottom charge to pull a greater depth. The improvements observed in all the blast performance indicators like pull, yield per round and powder factor, with 3.2 m deep rounds. The progress per round was improved by 28%, specific charge was improved by 15% and specific drilling was improved by 18%. There was an added advantage of reduction in ground vibration intensity by 30–40%, which obviously results in improving ground control and roof support aspects of underground mine. The overbreak/sidebreak was reduced by 40–50% due to increase relief and reduced vibration intensity. These techniques were also resulted in reduction of rock mass damage by 25–30%, which obviously reduced roof control problems. The major breakthrough of these new blasting methods is the outstanding increase of pull% by 95% in the most confined and hardrock conditions. Therefore, the application of novel and innovative blasting techniques helped in enhancement of both safety and productivity of hard rock dolostone mine at Tummalapalle Mine Project in India.

ACKNOWLEDGEMENTS

The authors thankfully acknowledge the Scientist-in-Charge, CIMFR Regional Centre, Nagpur and M/s SMSIL, Nagpur and Tummalapalle mine management for their cooperation during the studies. The authors owe the credit of the novel ideas explained in this paper to their mentor Late Dr. A.K.Chakraborty, former Scientist-in-Charge, CIMFR Regional Centre, Nagpur, who strived for the productivity improvement of mining industry till his last breath. The views expressed in the paper are those of the authors and not necessarily of the organizations they represent.

REFERENCES

Chakraborty, A.K & Jethwa, J.L. 1996. Feasibility of air-deck blasting in various rock mass conditions-A case study. *Fifth International Symposium on Fragmentation by Blasting*, Montreal. Rotterdam: Balkema.

Chiappetta, R.F. 2004. Blasting Technique to Eliminate Subgrade Drilling, Improve Fragmentation, Reduce Explosive Consumption and Lower Ground Vibrations. *General Proc. of Annual Conf. on Explosives and Blasting Research*. Cleveland, OH: International Society of Explosives Engineers.

Jhanwar, J.C., Chakraborty, A.K, Anireddy, H.R. Jethwa, J.L. 1999. Application of air-decks in production blasting to improve fragmentation and economics of an open pit mine. *Geological and Geotechnical Engineering* 17:.37–57.

Katsabanis, P.D. & Ghorbani, A. 1995. A laboratory study of explosives malfunction in blasting. *General Proc. of Annual Conf. on Explosives and Blasting Research*. Cleveland, OH: International Society of Explosives Engineers.

Mead, D.J., Moxon, N.T., Danell, R.E. Richardson, S.B. 1993. The use of air-decks in production.

Ramulu, M., Chakraborty, A.K., Raina, A.K. Jethwa, J.L. 2002. Blast performance assessment in Plexiglas models by measurement of specific surface area-a new approach. *Journal of Mines Metals and Fuels* 35: 68–75.

Ramulu, M., Raina, A.K., Choudhury, P.B. Chakarborty, A.K. 2005. Application of innovative in-hole delay solid blasting technique for productivity enhancement in underground coal mine—a case study. *Journal of Mining Technology* (June): 68–75.

Blasting in Mines – New Trends – Ghose & Joshi (Eds)

Application of a blast-based mine-to-leach model in Barrick Zaldivar

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ABSTRACT: This work presents the results of the application of a blast based Mine-to-Leach (M2L) model implemented at Barrick Zaldivar, Northern Chile, based on the positive results obtained in a previous M2L project developed during 2008–2009, where a decrease of P80 size at muck piles (>6″ to 4″) allowed substantial improvements in effective shovel performance, increasing crushing throughput (90 tonnes per hour) and also allowed raising the percent of Cu recovery in dump leach (1.5% in a 3-year leaching cycle). Barrick Zaldivar is currently attracted to further optimize mine fragmentation to achieve a target of P80 = 3″. Efforts to meet this target have made by applying a novel blast design and techniques, leading to potential benefits of increasing crushing performance (70 tph), a higher shovel loading performance and an increase of at least 1.5% of copper recovered in dump leach. The potential benefits are estimated as 3,520 additional fine copper tonnes per year by means of an increase of plant throughput and 1,560 additional fine copper tonnes per year by means of dump leach, giving to Barrick Zaldivar a total of 5,080 additional fine copper tonnes per year. These extra benefits are achieved through Advanced Blast-based Services provided by a multidisciplinary Orica Mining Services team working together with Barrick Zaldivar mine-plant staff.

1 INTRODUCTION

It is well-known that a decrease of P80 resulting from blasting optimizes the mine-plant processes, enhancing the value generation in downstream results. This concept is known as "Mine to Leach" (M2L) (Menacho and Olivero, 2005; Muñoz *et al.*, 2010). This paper presents the results of application of a blast based M2L model in Barrick Zaldivar in conjunction with Orica Mining Services (OMS). The objective was to create value in mine-plant processes from the results of blasting operations. The pursued value was focused on the result of a consistent fragmentation, improved loading performance of shovels, a higher primary crusher throughput, increased dump leach kinetics and wall control which actually acts as a restriction. The development of M2L process was divided into three stages. First, a baseline was established in order to evaluate historical performance of blast process. Second, a modification to blasting-applied standards (demoblast) was carried out in order to achieve the fragmentation target. And third, an economical evaluation of potential benefits was made (not included in this article).

2 MATERIAL AND METHODS

During the development process of demo blast, in a period of four months (September–December 2011), a new concept in blasting denominated Advanced Blast-based Services (BBS) was applied. A multidisciplinary OMS team of experts in blasting, geotechnical, metallurgical and technology areas worked in conjunction with Barrick Zaldivar mine-plant staff in order to apply novel blast design and techniques and define KPI´s of continuous control on results. Blast design considered the modification of drill patterns and the application of HA 50/50 (Fortan Advantage 150) and special emulsion (Flexigel 110) in a double charge by hole and the use of double prime with Ikon electronic detonators (Figure 1 and Table 1). Charge configuration and delays time were defined by advanced modelling and simulation (Shotplus-Ipro, BDA and MBM softwares).

A total of 1,1 Mton of demoblast was considered as a representative sample for evaluate changes in shovels performance, crusher throughput and dump leach fragmentation (Table 2). Dispatch and PI system was used for collecting data from

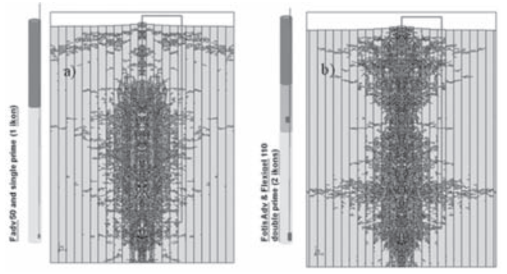

Figure 1. Charge configuration applied in Barrick Zaldivar. a) Standard design, b) Demoblast design.

Table 1. Modification of drill patterns according to target in fragmentation.

Disenos	Diseno pozos production	
	Burden (m)	Espaciamiento (m)
Actual	8	9.2
M2L(6″ ≥ 4″)	9	10.4
M2L(4″ ≥ 3″)	8.2	9.4
Actual	8.5	9.8
M2L(6″ ≥ 4″)	9	10.4
M2L(4″ ≥ 3″)	8.5	9.8
Actual	9	10.4
M2L(6″ ≥ 4″)	10	11.5
M2L(4″ ≥ 3″)	9	10.4
Actual	10	11.5
M2L(6″ ≥ 4″)	10	11.5
M2L(4″ ≥ 3″)	10	11.5

Table 2. Total minerals of different grades blasted with demo blast.

A (High grade) + HL (heap leach grade)	803,384
DL (dump leach grade)	350,570
Total	1,153,954

shovels and crusher respectively. Fragmentation was measured directly on the muckpile by PSieve and calibrated by macro scale sampling. The effect of fragmentation in the recovery of dump leach was evaluated using the Kelsall kinetic model (Mavros & Matis, 1991), which was supported by pilot-scale leaching tests developed in the previous M2L project (Muñoz *et al.*, 2010).

Figure 2 shows the methodology applied for demoblast as an advanced BBS, which consisted of a series of factors that determine demoblast results. Long-term geology and drilling data allowed recognizing lithology-alteration-mineralization and

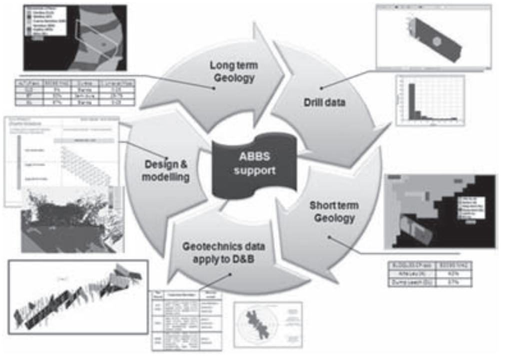

Figure 2. Methodology applied for demoblast design.

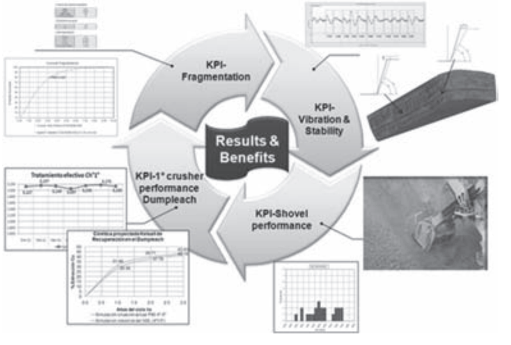

Figure 3. KPI evaluation for demoblast.

drilling velocity, respectively, which indicated oxide graded-areas and relative rock resistance. Short-term geology and geotechnical data permitted a conciliated mineral identification and structural control in the mine. Finally, after collection and analyzing of mentioned data, demoblast was designed by Shotplus Ipro, BDA and MBM.

Figure 3 shows results and benefits chain reached from demoblast. All KPI (fragmentation, vibration—stability, shovel performance, crusher performance and dump leach results) are obtained following a traceability to the demoblast areas from the mine to the plant, starting with a vibration near-field test for wall control, followed by a image sampling for P80 determination on the muckpile once the shovel meets demoblast, next shovel performance and effective crusher throughput are obtained from dispatch and PI System, respectively, after tracing mineral destination.

3 RESULTS

Fragmentation measured on the muck piles showed that the P80 obtained by demoblast was 2.9″–3.3″, which was very close to the P80 target of 3” (Figure 4). The effective shovel performance averaged 3,704 tonnes per hour (PH4100, 58 yd^3), considering those data when shovels loaded at least 50% of demoblast ore (Figure 5), which compared to baseline is more than 80 tonnes per hour. Primary crusher effective throughput in demoblast period was considered from certain days of the study period, when over a 70% of plant feeding contribution came directly from a demoblast blasted ore only and the rest of the days was considered as a blend and was added to the baseline data. Effective throughput of the primary crusher (Gyratory Crusher 54”x75”) averaged 3,174 tonnes per hour (Figure 6). Dump leach potential benefits due to enhanced kinetics shows up to 42% of Cu recovery at the end of the 3-year leaching cycle resulting of a feeding mineral of P80 3” (Figure 7).

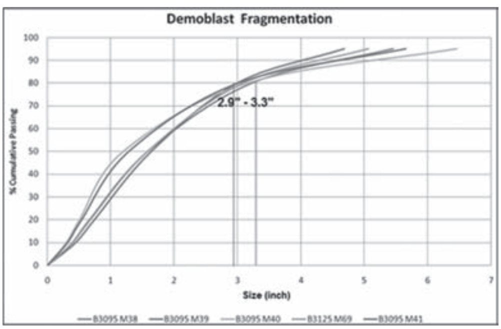

Figure 4. Fragmentation curve for demoblast mineral.

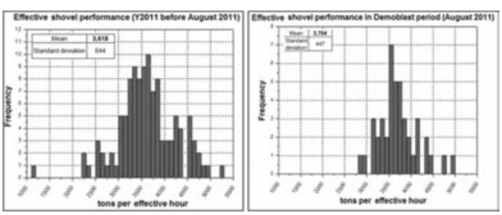

Figure 5. Shovel performance with demoblast (3,700 tph) compared with standard D&B design (3,600 tph).

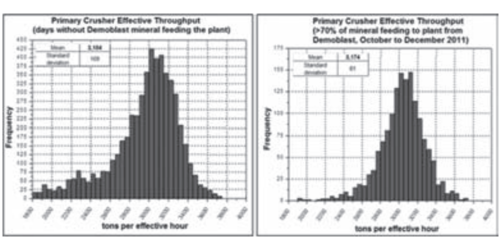

Figure 6. Primary crusher effective throughput for baseline (3,100 tph) and demoblast (3,174 tph).

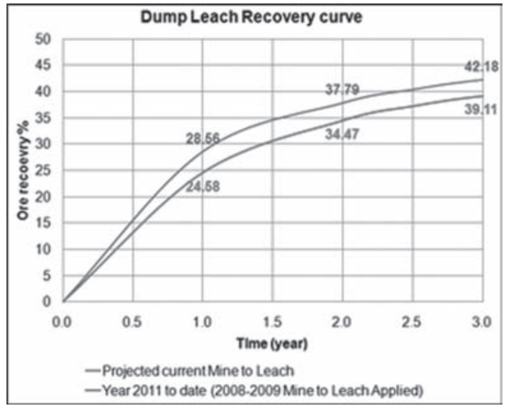

Figure 7. Estimation of potential increase in DL by blasting mineral to a P80 of 3″.

Table 3. Demoblast result in wall control.

Statistics		Value
Pre-split filter (%)	Obtained by current D&B standard in TRIM	39%
	Obtained by Demo blast novel D&B design in TRIM	53%
Difference (inches)		14%
% increase (or decrease) with respect to current standard		35.9%

In respect to wall control demoblast demonstrated that damage was reduced in a 36% with respect to current standard.

4 CONCLUSIONS

Demoblast demonstrated that can actually meet the mine target fragmentation of P80 3”, as originally required. It also shows the positive difference of 86 and 70 tonnes per hour over the results obtained by current D&B standard in terms of shovel performance and primary crusher throughput, respectively.. Table 4–6 represents the results of demo-blasts in terms of fragmentation of P80 3”, shovel performance and primary crusher throughput.

Dump leach potential benefit is pending for quantification. However, based on previous M2L experience, all other results indicate that even a small improvement of 1% in copper recovery leads to over 1,500 fine copper produced. The potential benefits are estimated approximately as 1,560 additional fine copper tonnes per year by means of

Table 4. Results of demo-blast in terms of fragmentation of P80.

Statistics	Fragmentation P80 (inches) Obtained by current D&B standard in production	Obtained by demoblast novel D&B design in production	Difference (inches)	% increase (or decrease) with respect to current standard
Minimum	4.3	2.9	–1.4	–32.6%
Maximum	5.5	3.3	–2.2	–40.0%

Table 5. Results of demo-blast in terms of shovel performance.

Statistics	Shovel performance (tonnes per hour) Obtained by current D&B standard in production	Obtained by demoblast novel D&B design in production	Difference (tonnes/hour)	% increase (or decrease) with respect to current standard
Mean	3618	3704	86	2.4%
Standard deveaition	644	447	–197	–30.6%

Table 6. Results of demo-blast in terms of primary crusher throughput.

Statistics	Primary crusher throughput (tonnes per hour) Obtained by current D&B standard in production	Obtained by demoblast novel D&B design in production	Difference (tonnes/hour)	% increase (or decrease) with respect to current standard
Mean	3104	3174	70	2.3%
Standard deveaition	169	81	–88	–52.1%

dump leach. That is why, efforts must be continued for leaching tests to recognize and meet the value. On the other hand, an additional of 3,520 fine copper tonnes per year is related to increased plant throughput, which means a total potential benefits by 5,080 additional fine copper tonnes per year for Barrick Zaldivar.

REFERENCES

Mavros, P. & Matis, K.A. 1991. Innovations in flotation technology. *Proceedings of the NATO Advanced Study Institute on Innovations in Flotation Technology, Kluwer Academic Publishers, Kallithea*, Chalkidiki, Greece, 12–25 May of 1991, p 197.

Menacho, J. & Olivero, P. 2005. Rentabilidad Mine To Leach v/s tronadura. *In: Proceedings of III Hydro-Copper, Sanatiago-Chile.*

Muñoz, C., Andrádes, A., Toro, D., Palape, R. & González, L. 2010. Impacto de las prácticas de voladura en la gestión de proyectos Mine To Leach. *In: Proceedings of ASIEX 9, Santa Cruz-Chile.*

Blasting in Mines – New Trends – Ghose & Joshi (Eds)
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Cast blasting for improved mine economics

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ABSTRACT: Ever growing demand of minerals has compelled the mine owners to adopt smart ways to increase production, productivity, efficiency and recovery of minerals. The end of XIth plan (2007–2012) has envisaged demand of coal in India at 713.24 Mt, which will be growing to 1055.0 Mt at the end of XIIth (2012–2017) plan (the economy of India is based in part on planning through its five year plans, which are developed, executed and monitored by the planning commission, Government of India). It may be observed that due to revised coal production projections in the terminal year of XI plan (i.e. 2011–12) there is a production shortfall of around 50.09 Mt., indicating a growth of 7.89% in coal production against 9.60% envisaged initially. The opencast mining accounts for about 90% of the produced coal in Coal India Limited. Since more open cast mines are coming up in the coal sector, the projected target of coal production by surface mining will necessitate removal of larger volume of overburden. In view of increasing volume of overburden judicious selection of the stripping method becomes more important. Even a small reduction in unit cost in overburden removal would result in enormous savings in a large-scale open cast operation. Conventionally, over—burden is removed by draglines, shovels or loaders. Cast blasting has emerged a cheaper alternative to the conventional method for the removal of overburden in opencast coal mines.

This paper describes systematic approach of design and implementation of cast blasting at an Indian operating mine with modified blast design pattern for achieving enhanced productivity. Total of 25 side cast blasts were conducted with an increase in drill and blast cost of 25% while the reduction in operating cost of dragline was 20%. Total cost saving for the overburden handling for two years was 17.98 Million Indian Rupees for 3.59 Mm^3 of overburden removal.

Keywords: cast blasting, productivity improvement, heavy blasting, dragline blasting

1 INTRODUCTION

In India, coal production in the past was mainly coming from underground mining. But currently, the share of opencast to the total coal production is estimated to be around 90 percent in 2011. Since more opencast mines are coming up in the coal sector, the projected target of coal production by surface mining will demand removal of larger volume of overburden. In view of increasing volume of overburden judicious selection of the stripping method becomes more important (Runge, 1981; Atkinson, 1992; Chironis, 1981). Even a small reduction in unit cost in overburden removal would result in enormous savings in a large-scale opencast operation. Conventionally, overburden is removed by draglines, shovels or loaders. Globally cast blasting has become a cheaper alternative to the conventional method for the removal of overburden in opencast coalmines. Now cast blasting is a regular practice in most of the operating open-cut mines of USA, Australia and Canada (Chaoji & Dey, 2000; Learmont, 1983; Reddy & Uttarwar, 1999).

The extra explosive energy used in cast blasting reduces the amount overburden to be handled by the machinery but increases the risk of damage to the underlying coal seams. This damage may lead to reduced coal recovery and may reverse much of the benefit sought from cast blasting damage and loss. A large amount of explosives is detonated during cast blasting with complex time series which creates problems in terms of increased vibration, unstable post blast high-wall and in even some cases loss of revenue for excessive dozing if not planned well (Sahai, 1996; Worsey & Giltner, 1987; Brent, et. al., 2003; Sanchidrian, et. al., 2007).

2 CAST BLASTING

Cast blasting is a specific case of directional blasting. It is also called explosive over burden casting or blast casting or casting overburden by blasting or simply throw blast and it also has been referred to as "Controlled trajectory blasting". But in the decade that has followed blast casting or cast blasting has become popular. It can be defined as a technique of blasting to cast over burden material directly into the de-coaled area without re-handling. This technique is employed primarily in surface coalmines.

The primary objective of cast-blasting technique is to fragment and remove the over burden material directly into the spoil pile in a single operation. The percentage of cast is governed by geological conditions of the site and the parameters of the blast applied. A properly designed blast in a favourable condition will lead to higher casting percentage and reduced overall operating cost of mine (Mishra, 2000; McDonald, et. al., 1982; Chiapetta, et. al., 1990; Chiapetta, et. al., 1988; Scott, et. al., 2010).

2.1 *Cast blasting design parameters*

Although cast blasting has increased dramatically in Indian surface coalmines in recent years, there is no clear idea on what weightage to assign to the various blasting factors to make a cast successful. The parameters affecting castings can be discussed in the following three major groups:

- Characteristics of over burden
- Type and Characteristics of explosive
- Blast geometry and initiation sequences

2.2 *Characteristics of over burden*

The geology of the blast area exerts great influence on casting of over burden by blasting. The mechanical properties of rock, discontinuities, stratigraphy and hydrogeology are important parameters to be considered for designing the cast blasting.

2.3 *Type and characteristics of explosive*

In cast blasting, explosives other than ANFO can be used under watery conditions. The use of slurries and emulsion would lead to high borehole energies and better casting results. A study on explosive casting with high-speed photography has shown that burden could be increased when going from ANFO to higher energy explosives, by maintaining a constant hole diameter (Mishra, 2000). The type of explosive is important not only for the energy it develops but for its actual yield. The ratio between burden and explosive energy governs the ejection velocity of material from the face and, as a consequence, the distance reached.

2.4 *Blast geometry and initiation sequences*

The blast design can be adjusted for a given situation with proper selection of the following variables:

- Bench height and pit width
- Blast hole inclination
- Blast hole diameter
- Burden and spacing
- Charge factor
- Stemming and decking
- Initiation sequences
- Detonator accuracy

The following paragraphs deal with a mine scenario where throw blasting was not practised. It was introduced with certain changes in blast design and compared with the conventional blast in terms of saving to the mine.

3 OVERVIEW OF THE MINE

3.1 *Details of opencast project-A*

Opencast project-A, Godavari Khani falls within the South Godavari lease hold of the Singareni Collieries Company Limited. The estimated total reserve is about 54.4 million tonnes and the annual production from the mine is about 2 million. The topography of the quarry area is flat and gently undulating and is covered with a thin mantle of subsoil. The coal seams are gently sloping on both sides of the property from 9 degree to 16 degree.

Almost half of the reserves of No. 3 and 4 seams combine to make a composite seam of 14 m. The overburden consists of massive grey white medium to coarse grained felspathic sandstone inter collated in some horizons with thin bands of shale, clay and carbonaceous sand stone.

Conventional opencast mining method using shovel—dumper is adopted in this mine. EKG 4.6 m^3 shovels in conjunction with 50T dumpers are used for hauling the waste rock/coal from the mine. Rotary drills of 250 mm diameter are used for production blasts. A walking dragline of 24/96 is deployed to work in extended bench method with a cut width of 60 m with a bench height of 24 m.

Opencast Project-A, covering an area of about 3 sq. km is situated in the south western extremity of Godavari Basin in Andhra Pradesh. It is about 245 kmfrom Hyderabad. This was the first major mechanised opencast mine of SCCL.

The southern mine boundary is limited up to 140 m upthrow Archean Fault which is also limiting the extent of Barakar measures. The northern boundary is adjacent to the underground workings of GDK No. 9 incline. The eastern side is flanked by the underground Blasting Gallery Workings of GDK No. 10 Incline mine while in the western side, Surface structures for infrastructure facilities like Office, Stores, Silo, Diesel bunker, Workshop Quarters etc are present. The property is bifurcated in to almost two equal halves and twisted by a Scissors Fault running in North–South direction. Due to this the seams are sloping in North direction in the Eastern part and South direction in the Western part at 60 to 160. Major portion of the mine area contains combined 3 & 4 seams while in the rest portion a stone parting varying from 2 m in the middle to 6 m on the northern side separates the said two seams. There also exist 1.5 m thick 3 A seam over No. 3 seam, which blocks about 1.2 m coal within the property. No. 3 seam is 27 m below the floor of No. 3 A seam. Average gradient of the seams are 1 in 12 and average grade of coal is 'E' containing 30% to 36% ash and 5.6% to 6.1% moisture. Calorific value of coal varies from 16861.5 to 19141.8 kJ/kg of coal and the specific gravity is 1.5. Crushing strength of coal is 503 kg/cm^2.

3.2 *Geology*

The fault F 46, trending in WNW–ESE direction (N60 W) with a throw of 1000 m towards NNE, is a major boundary fault, where the Gondwanas abut against the Metamorphics. The liner extent of this fault is about +30,000 m and it forms the southern limit of OCP-A block and also Ramagundam coalfield.

The OCP area reflects two plunging synclinal structures on either side of the line joining the borehole Nos.185 and 491. The eastern synclinal structures plunge towards ENE, where the strike of the coal seams varies from NNE (Northern limb) to WNW (Southern limb) and slopes at a gradient of 1 in 20 to 1 in 6.

3.3 *Rock type and strength*

The rock type in OCP-A block represents, kaolinisedfeldspathic sandstones of Barakar formation pertaining to Lower Gondwana sediments. These sandstones were highly weathered and brownish in colour generally up to a depth of 20 to 30 m from surface and represented weathered mantle, overlain by about 3 m thick soil cover. The strata below the weathered mantle represented un-weathered white/grey—white feldspathic sandstone. The strata consist of II, IIIB, IIIA, III and IV (or III & IV combined seams). The coal seams generally become clayish up to a depth of 20 to 30 m from surface due to weathering. No. III and IV seams were merged together over a width of about 1 km, parallel to the major fault.

4 METHOD OF WORKING

A block size of 100 m length, 60 m cut width and 26 m average depth has been considered for comparative study and carrying out the techno economic analysis of the side casting practice.

4.1 *Drilling geometry*

For a normal blast, the geometry of blast holes are shown in the Figure 2. In case of cast blasting, additional holes of full depth wererequired for large horizontal displacement of the blasted

Figure 1. Site specified for cast blasting.

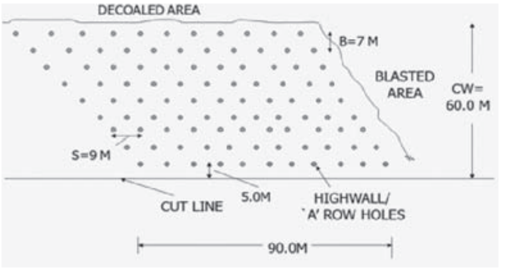

Figure 2. Layout of the holes in a normal blast.

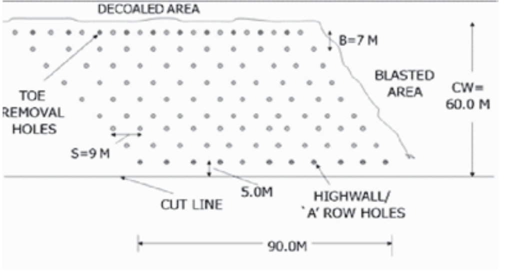

Figure 3. Layout of blast holes in side casting blast.

material. These had been called as staggered holes or toe removal holes and such hole had been drilled in between the normal production holes (Figure 3) towards the decoaled area.

There was no change in main production holes which were drilled at a burden and spacing of 7 m and 9 m respectively.

The High Wall 'A' Row holes were also drilled in normal fashion with a burden of 7 m lying within 5 m of the cut line having depthof 13 m.

4.2 *Explosive geometry*

Explosives were charged without any decking in all the holes. A varying degree of stemming height was maintained in each row for best possible movement of the blasted material. The energy of the bottom load was packed in such a way that it imparted sufficient energy for maximum horizontal movement of the blasted material.

The staggered holes were charged with the bottom load only and a large stemming height of 14 m was maintained whereas for 'A' Row holes stemming length of 6.5 m to 7.0 m was maintained (Figure 5).

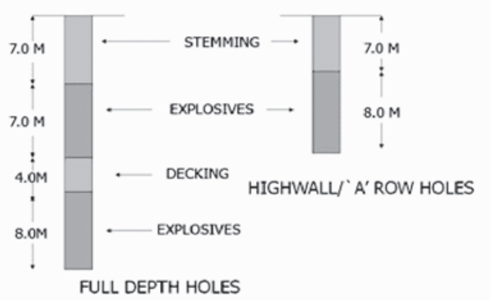

Figure 4. Explosive loading chart in a normal blast holes.

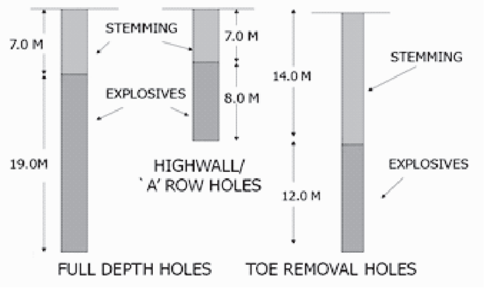

Figure 5. Explosive loading chart in side casting blast holes.

4.3 *Firing sequence*

The planned displacement of the blasted material was achieved by increasing the delay interval between the rows so that each row of holes could get sufficient time to complete their flight. This would have provided proper burden relief for subsequent holes. However, due care was taken to restrict the time gap such that there is no premature release of gas energy of the succeeding row holes. The delay period was based on burden rock response time studied with high speed digital camera of Red Lake Imaging and Motion analysis software of MREL.

4.4 *Muck profile*

A typical cross section of the Dragline Bench after normal blast & side casting blast is shown in Figure 8 and 9 respectively. It is observed that incase of normal blast the material thrown in the de-coaled

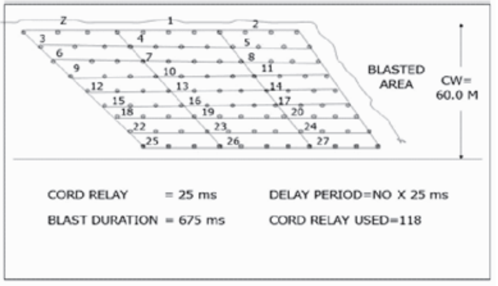

Figure 6. Firing sequence for normal blast.

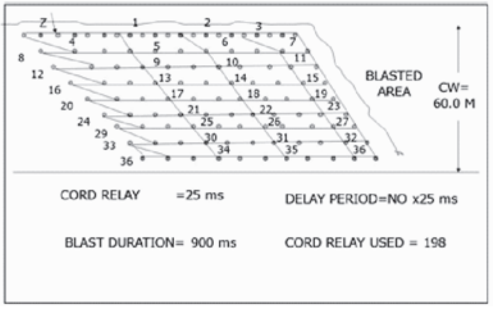

Figure 7. Firing sequence for side casting blast.

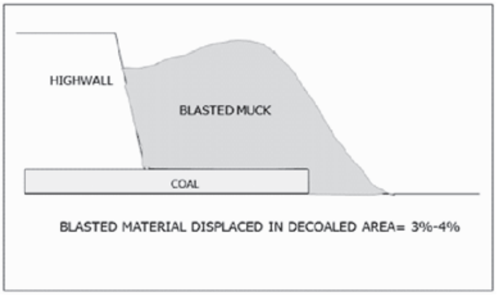

Figure 8. A model of dragline bench after normal blast.

area is restricted to about 4% against 20%–25% in case of side casting blast.

4.5 *Observation*

The powder factor of 0.61 kg/m^3 could be achieved with a side casting of 20% against the powder factor of 0.45 kg/m^3 with conventional blasting. The working in detail is described in the Table 1.

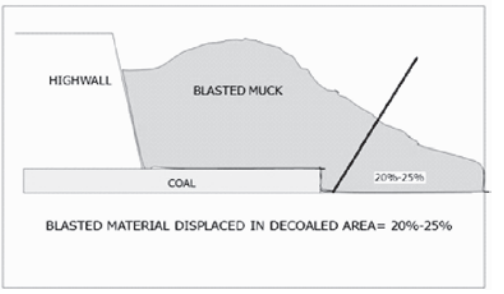

Figure 9. A model of dragline bench after side casting blast.

Figure 10. Plate showing blasted muck after side cast blasting at site.

5 RESULTS AND ANALYSIS

In order to evaluate the advantage of side cast blasting over normal blasting, a detailed working illustrated in Table 2 was carried out with the assumption that cost of drilling per meter with 250 mm diameter holes is Rs.250 per meter and the Dragline excavation cost is Rs. 20 per m^3.

It is seen that, in spite of increased explosive consumption as well as drilling cost, there was a net saving in Dragline operating cost by Rs. 7,80,000/- for every such block of 100 m length.

Table 3 shows a summary of 25 cast blasts conducted during the study. The total savings to the mine management was of 17.90 million Rupees for handling the over burden of 3.59 Mm^3 with average powder factor of 0.51 kg/m^3 and percentage of casting of 20% respectively.

Table 1. Comparison between normal blast and blast with side casting in the dragline bench.

Sl. no	Description	Normal blast (BCM)	Side casting blast (BCM)
A	**BLOCK SIZE**		
	Cut width (m)	60.00	60.00
	Depth (m)	26.00	26.00
	Length (m)	100.00	100.00
	Volume (m^3)	**156000.00**	**156000.00**
B	**DRILLING GEOMETRY**		
1	*Full Depth*		
	Burden (m)	7.00	7.00
	Spacing (m)	9.00	9.00
	Depth (m)	26.00	26.00
	No. of Holes	88.00	88.00
	Decking (m)	4.00	0.00
	Stemming height (m)	7.00	7.00
2	*For Toe Burden*		
	Burden (m)	–	7.00
	Spacing (m)	–	9.00
	Depth (m)	–	26.00
	No. of Holes	–	10.00
	Decking (m)	–	0.00
	Stemming height (m)	–	14.00
3	*High Wall/'A' Row*		
	Burden (m)	7.00	7.00
	Spacing (m)	9.00	9.00
	Depth (m)	15.00	15.00
	No.of Holes	11.00	11.00
	Decking (m)	0.00	0.00
	Stemming height (m)	7.00	7.00
C	**EXPLOSIVE GEOMETRY**		
1	*Full Depth*		
	Indogel Series (kg)	66000.00	83600.00
	Indocast (kg)	132.00	176.00
	Sub Total (kg)	66132.00	83776.00
2	*For Toe Burden*		
	Indogel Series (kg)	0.00	6000.00
	Indocast (kg)	0.00	12.50
	Sub Total (kg)	0.00	6012.50
3	*High Wall/'A' Row*		
	Indogel Series (kg)	4752.00	4752.00
	Indocast (kg)	8.25	8.25
	Sub Total (kg)	4760.25	4760.25
	Indogel Series (kg)	70752.00	94352.00
	Indocast (kg)	140.25	196.75
4	**Total Explosives**	**70892.25**	**94548.75**
D	**Powder Factor—In-Situ (kg/m^3)**	**0.45**	**0.61**
	G.P.F. (m^3/kg)	0.45	
4)	*EXPLOSIVE ACCESSORIES*		
A)	FULL DEPTH		
	Indocord-10 (m)	2289	2289
B)	FOR TOE BURDEN		
	Indocord-10 (m)	0	261
C)	HIGH WALL/ STABILITY		
	Indocord-10 (m)	166	166
D)	Surface layout(m)	1660	1660
	TOTAL INDOCORD (m)	4115	4376

Table 2. Cost analysis between normal blast and blast with side casting in the dragline bench.

Sl no	Description	Normal blast	Side casting blast
1	BLOCK SIZE		
	Cut width (m)	60.00	60.00
	Hole Depth/Bench height (m)	26.00	26.00
	Blast Length (m)	100.00	100.00
2	Volume In-Situ (m^3)	156000.00	156000.00
3	No. of Holes		
	Full Depth—26 m	88.00	98.00
	High Wall holes—15 m	11.00	11.00
4	Cost of Drilling @ Rs. 250/m (Rs)	613250.00	678250.00
5	Powder Factor-In-Situ (m^3/kg)	2.201	1.650
6	Total Quantity of Explosives (kg)	70892.25	94548.75
	Indogel Series (kg)	70752.00	94352.00
	Indocast (kg)	140.25	196.75
7	Indogel Series @ 16.023/kg	1133659.296	1511802.10
	Indocast @ 18.26/kg	2560.965	3592.655
	Total Explosives cost (Rs.)	1136220.261	1515394.75
8	Drilling and Blasting cost (Rs)	1749470.261	2193644.75
9	Dragline Benefit @ 20% Casting (m^3)	0.00	31200.00
10	Dragline Excavation Cost/m^3 (Rs)	25.00	25.00
	Dragline Operating cost (Rupees)	3900000	3120000
11	**Savings to Dragline cost (Rs.)**	**0.00**	**780000.00**
12	Total Operation Cost (Rs)	5649470.26	5313644.75
13	**Saving for the Blasting Block (Rs)**	**0.00**	**335825.51**

Table 3. Total savings during the study.

Blast ID no.	Block volume (m^3)	Powder factor kg/m^3	% of Casting	Savings (Rs)
278	84182	0.582	18	378819
279	222514	0.551	18	1001313
256	162508	0.582	17	609405
257	153296	0.551	17	651508
261	105870	0.524	17	449947.5
438	179270	0.530	17	761897.5
439	98244	0.446	18	442098
205	235337	0.498	20	1176685
206	108112	0.555	20	540560
208	151635	0.552	20	758175
209	25931	0.370	20	129655
97	174988	0.518	20	874940
98	147020	0.517	20	735100
99	124217	0.468	20	621085
100	232126	0.529	20	1160630
101	128060	0.500	21	672315
102	129480	0.508	20	647400
327	176460	0.517	20	882300
328	170595	0.484	20	852975
355	141475	0.535	20	707375
381	182975	0.509	24	1097850
382	83000	0.556	22	456500
383	129139	0.476	24	774834
402	94900	0.492	25	593125
403	148053	0.510	25	925331.3
Total	3589387			17901823

6 CONCLUSION

The Side Cast Blasting methodology at opencast-A mine improved the productivity of the Dragline on account of following:

1. Reduction in the volume of the material to be handled by Dragline by 20% thus saving in the operation cost.
2. Improved efficiency of the Dragline on account of loose muck profile, better fragmentation & smooth digging.
3. Enhanced rate of coal exposure because of higher utilization of Dragline.

The cast blasting is considered a cost effective blasting technique. The cost competitive market has compelled every mine operator to adopt the latest technological advancement for creative solution of mine problem. Enhancement of productivity is on the top of priority list of every mine operator. Proper implementation of cast blasting could accrue the savings of 17.90 Million Rupees for overburden production of 3.59 Mm^3 with savings to the mine. Extra hours were avoidable for dragline to increase production and uncover of coal. During the study this could not be evaluated quantitatively in field. There is still scope for further improvement with usage of electronic detonators and variable explosives energy.

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REFERENCES

Atkinson, 1992. Surface mining—past, present and future. Journal of Mines, Metals & Fuels, Vol.30(6), India: 241–252.

Brent, G.F., Edmondson, M. & Goswami. T. 2003. High performance throw blasting with i-kon electronic detonators in an environmentally sensitive area at Stratford Coal, NSW, Australia, Fifth Large open pit mining conference.

Chaoji, S.V. & Dey, B.D. 2000. Dragline operation in mines—an overview. Journal of Mines, Metals & Fuels, Vol.48(5), India: 84–93.

Chiappetta, R.F., Mammele, M.E. & Postupack, C. 1988. Causes and Recommendations for Controlling Coal Damage When Blasting Overburden. Proceedings of the 14th Conference on Explosives and Blasting Technique, Anaheim, CA.

Chiappetta, R.F., Srihari, H.N. & Worsey, Paul, N. 1990. Design of overburden casting by blasting—recent developments. Journal of Mines, Metals & Fuels, Vol.38(9) India: 194–208.

Chironis, P. Nicholas, 1981. Mines Take on Explosive Casting. Coal Age, October: 130–134.

Learmont, T. 1983. The Walking dragline and its application. Proceedings of International Symposium on Surface Mining and quarrying, Institution of Mining and Metallurgy, England.

McDonald, K.L., Smith, W. & Crosfy, W.A. 1982. Productivity improvements for Dragline operations using controlled blasting in a single and multi-seam operation at Reitspruit, South Africa, Annual Meeting, CIMM, Quebec, Canada.

Mishra, A.K. 2000. Evaluation and design of blast using high-speed video camera in coal measure rocks: Ph.D. Thesis, Indian School of Mines, Dhanbad.

Reddy, A.H. & Uttarwar, M.D. 1999. Potential of Blast Casting to Improve Economy of Operating Surface Coal Mines in India. Proceedings of International Conference on Mining: Challenges of the 21st Century (Editors: A.K. Ghose and B.B. Dhar), APH Publishing Company, New Delhi: 245–260.

Runge, I.C. 1981. Mining design for deep open cut mines. Proceedings of the Aus IMM Conference on Strip Mining-45 m and beyond, Rockhampton: 37–42.

Sahai, B.N. 1996. Cast Blasting in Opencast mine, Proceedings of 3rd National convention of Mining Engineers and seminar on Productivity in Opencast Mines. Institution of Engineers, India: 227–238.

Sanchidrian, J.A., Segarra, P. & Lopez, L.M. 2007. Energy components in rock blasting. International Journal of Rock Mechanics & Mining Sciences 44 (2007): 130–147.

Scott, B., Ranjith, P.G., Choi, S.K. & Khandelwal, M. 2010. A review on existing opencast coal mining methods within Australia. Journal of Mining Science, Vol. 46(3).

Worsey, P.N. & Giltner, S.G. 1987. Economic & Design Considerations for Explosive Overburden Casting. International Journal of Mining & Geological Engineering, Vol.5: 93–108.

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Blasting and explosive application in India—Past experience and future trends

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ABSTRACT: Explosives are a source of concentrated chemical energy which can be harnessed for a number of applications. The shock wave generated upon detonation of the explosive followed by the expanding gases, both generated in a very small fraction of time, fragments and displaces the medium surrounding it. During the late sixties and early seventies, there were debates whether blasting was an 'art' or 'science'. With better understanding of the blasting mechanisms using high speed photography/videography and instrumented model scale tests and of regular production blasts, it can be said that blasting has emerged as a full fledged science today. Applications of explosives are many which this paper reviews in brief, and past experiences and emerging trends in the Indian scenario.

1 INTRODUCTION

Blasting—Art or Science was an often asked question. Many considered it an art since good blasts could be achieved with a common sense approach. The importance of safety, compatibility of explosive with rock characteristics, importance of providing proper confinement, plastic rocks absorbing shock, elastic rocks transmitting shock, correct shot design for rock movement were the aspects discussed in a paper published in Mining Congress Journal in 1969 (Grant 1969). The paper also concluded that blasting was truly undergoing a change from an art to a science.

The twenty first century has witnessed many new developments in the field of explosives and blast initiation systems. From humble beginnings of development of Safety Fuse by William Bickford in 1831 and Plain Detonators in 1864, the 1990s witnessed digital technology introduced into detonator shells to achieve super-accurate electronic detonators. Nitro-Glycerine based dynamites which ruled the roost for almost a century was replaced by Ammonium Nitrate based Slurry (Watergel) explosives and more recently the emulsion explosives which are known to have 'near-ideal' detonation characteristics.

The explosive properties of Ammonium Nitrate became evident with two untoward incidents. In 1921, at the end of the war, a large surplus of Ammonium Nitrate was left when the first ammonia synthesis plant was closed down. A huge pile of the compound was stored in an open field. In attempts to break apart the pile for removal, explosives were charged into holes drilled in the pile. Contrary to what was expected, the whole mound of 4,500 tons detonated killing 600 people. In the other incident known as the Texas city disaster, a ship carrying Ammonium Nitrate based fertilizer docked in the port caught fire and exploded causing large scale death and destruction. This incident occurred on April 16, 1947.

2 THE MANY USES OF EXPLOSIVES

Commercial explosives are used for a variety of applications in agriculture, civil, mining, prospecting for oil, medical, electrical and metallurgical applications. The various uses are detailed below:

a) Mining Applications
   - Bench blasting.
   - Box-cut excavation.
   - Overburden casting.
   - Tunneling and Shaft sinking in mines.
   - Stoping using small & large diameter drillholes.
   - Drop Raising using VCR technique.
   - Dimension Stone excavation

b) Civil Applications
   - Ground leveling.
   - Tunneling and Shaft sinking for rail/road.
   - Storage caverns.
   - Trenching/Ditching.
   - Controlled demolition of structures.
   - Fill settlement (compaction of landfills).
   - Generation of Armour Stone (Rip-Rap).
   - Coyote Blasting.
   - Underwater blasting for increasing draught.

c) Agriculture
   - Spreading fertilizer/manure on the field.

- Removal of tree stumps, cutting branches/logs.
- Breaking and removal of embedded boulders.
- Excavating/deepening of wells.
- Sectional blasting of bore wells (for increasing water yield.

d) Petroleum & Natural gas
- Seismic prospecting work.
- Shaped charges for oil well casing perforation.
- Quenching oil well fires.

e) Miscellaneous uses of blasting
- Explosive excavation.
- In-situ Leaching.
- De-stress blasting in underground mines.
- Contour Blasting—Smooth blasting & Pre-splitting in surface and underground, mining and construction.
- Avalanche control.
- Removing hard accretion/slag inside kilns.

f) Metallurgical & Electrical applications
- Metal cladding (bonding metals with explosive).
- Surface hardening of metals (shovel bucket teeth, railway frogs etc).
- Compaction of ceramic and metal powders.
- Explosive Forming.
- Structural & Electrical Transition Joints.
- Explosive Welding

g) Metal cutting/Demolition
- Linear Shaped Charges (for cutting sheet metal, pipes in undersea installations, demolition of structures etc.

h) Medical applications
- Pulverizing kidney stones

i) Art & Entertainment
- Explosive Engraving.
- Explosive 'carving'.
- Special effects in Movies and in Events

j) Automobile & Aircraft
- Aircraft seat ejection, cutting of canopy
- Air bags in automobiles

Few of the unconventional applications mentioned in the above list are briefly described.

2.1 *Fill settlement*

It may become necessary to locate highways, rail tracks through swampy and water logged areas. In such cases it is necessary to undertake special measures to provide a competent foundation. Detonating an explosive charge embedded in the fill provides a firm base for the fill material to settle to the bottom.

2.2 *Coyote blasting*

This is a specialized method of blasting to provide large quantity of broken material for civil works. This technique is usually deployed in hilly terrain where it is difficult to employ conventional drilling and blasting methods. The technique involves driving of tunnels and cross tunnels underneath a hill. The tunnels are filled completely with explosives and blasted. The explosion causes the entire burden to lift promoting a crushing and fragmenting action. Ideal conditions for a coyote blast are where the rock formation has joints in cubical or columnar nature. Most adverse condition is when the bedding planes are almost horizontal.

2.3 *Sectional blasting of bore wells*

This is a technique adopted to increase the water flow into a bore well by inducing fractures in the aquifer. Prior knowledge of location of aquifer zone is necessary to place the explosive charge at the desired horizon. Location of aquifer is obtained either from the driller (in case of new bore well) or through resistivity survey inside the bore well in case of old ones. Small charges are set off, and repeated if required. Detonating a large quantity of explosive can cause damage/collapse of bore well and also eject out the casing pipe.

2.4 *Vertical Crater Retreat (VCR) blasting*

VCR blasting utilizes 'spherical charge' concept. A 'spherical charge' is one whose length: diameter ratio is equal to 6. Upon detonation, a spherical charge in a single drillhole with infinite burden creates a crater. The crater volume is dependent on the depth of burial of charge, the explosive properties and rock properties.

Cratering concept has been adapted in underground metal mines for creating a raise, ore pass between two levels. Drillholes in a square grid with an optional central drillhole are drilled from the drilling level up to the undercut or extraction trough level below. The drillholes are drilled downward, breaking through into the undercut viz are open at both ends. Charging of explosive is also carried out from this level. The open end of drillhole is plugged at a specified distance from the drill level, a spherical charge is placed and stemmed. The detonation creates crater-shaped openings. When several drillholes are detonated at same level, a horizontal slice is blasted. The ore is excavated by such horizontal slices, starting from the stope bottom and advancing (retreating) upwards and hence the name VCR.

The drillhole depth is measured and the drillhole is blocked at proper height above the bottom end. Explosive charges are lowered, stemming of sand and water placed on top of the charge. Holes are grouped with charges at same elevation and distance to rock surface. Explosive charges co-operate in breaking rock, normally loosening a 3.0 m slice of ore, falling into the space below. Vertical drillholes are preferred wherever possible. Drillhole diameters used are 150 or 165 mm, but even 205 mm (8″) holes have been tried in a few mines. The most common drillhole diameter is 165 mm, allowing holes to be spaced 4.0 m × 4.0 m. Charging requires training of the charging crew for successful blast result. The ore produced by crater blasting falls down in the open space underneath.

2.5 *Explosive excavations*

This method utilizes large scale use of explosives for excavating canals, harbours, waterways, widen channels, and create earthen dams. Both conventional explosives and nuclear devices have been used as can be seen from Table 1.

2.6 *Directional blasting for construction of rockfill dam*

Creation of earthen dam between two hillocks to impound a river using explosive energy to cast the soft earth into place has been reported in Russia. (Ignatyuk 1970). For the first time in the world, on March 29, 1968 the Soviet Union effected the construction of a rockfill dam with the use of directed blasting across the Vaksh River in Tadjikistan. A dam of height 55 m and crest length 170 m was executed by blasting 11 chambers provided in the rocky slopes of the Baipaza gorge. The total weight of explosive detonated was 1860 Mt. The total volume of earth exploded and placed into position was 1.5 million cubic metres.

2.7 *Explosive carving*

Controlled use of small charges of explosive, mainly pieces of detonating cord of varying core loads commensurate to drill diameter and spacing created the 'Crazy Horse' Memorial in the USA. Drillholes of 38, 42 and 48 mm diameter of depths varying from 1.5 m to 10 m depths were used with spacing as low as 15 cm up to 1.2 m. After the desired contours were achieved, a jet finishing torch is used for the final finishing of the surface of the mountain carving. The jet removes drill marks and smoothes the final surface. The torch runs on diesel fuel and compressed air. The 3,300°F jet flame causes tiny fragments of rock to flake off as the result of heat expansion, leaving a polished surface.

2.8 *Kidney stone removal*

Japanese researchers have developed a new method of removing kidney stones in the human bladder by using mini-explosive charges such as Lead Azide. A newly developed cysto-scope with an explosive chamber at its end is inserted into the bladder through the urethral canal. The explosive chamber is placed close to the kidney stone and remotely set-off which pulverizes it. The patient's bladder is usually filled with water to protect it from shock waves and flying fragments. The method eliminates the use of surgery.

3 EXPERIENCES OF BLASTING INNOVATIONS AND EXPLOSIVE APPLICATIONS IN THE INDIAN SCENARIO

The Indian scenario has witnessed introduction of commercial explosives and blast initiation systems products in line with world trends and application of innovative blasting techniques for specific field needs. Details are:

Table 1. Examples of excavation.

Project name	Purpose	Excavation dimension	Type & quantity of explosive used
Pre-Gondola Phase II & III	Row cratering to connect water reservoir	Height—1370′ Width—100′–200′ Depth—13′–39′	Nitro methane—140 MT AN Slurry—70 MT
Tugboat	Berthing basin	Length—240′ Width—240′ Depth—12′	AN Slurry—100 MT
Trinidad	Railway cut	Length—400′ Width—40′ Depth—20′	AN-FO & AL-AN Slurry—44 MT
Dannyboy	Experimental crater	Crater radius—107′ Crater depth—62′	Nuclear explosion—0.5 kiloton
Sedan	Experimental crater	Crater radius—604' Crater depth—320'	Nuclear explosion—100 kiloton

3.1 *Products*

- Acid emulsion explosives (large diameter packaged).
- Slurry explosives (small diameter, large diameter and permitted, pumpable).
- Slurry explosive in couplable plastic tubes (upto 52 m hydrostatic head)
- Emulsion explosives (small diameter, large diameter and permitted, pumpable).
- Emulsion Booster.
- Heavy AN-FO & Doped Emulsions.
- Shock tube based non-electric initiation system.
- Fully field programmable electronic detonators.
- Sequential Blasting Machine.
- Shaped charges for underwater blasting.

3.2 *Blasting techniques*

- Drop Raising using VCR technique.
- Blast hole stoping using large diameter drillholes (VCR raise, slot removal and stoping).
- True bottom hole initiation in bench blasting using non-electric shock tube detonators.
- Overburden casting in opencast coal mines
- Creating a Box-cut.
- Coyote Blasting.
- Underwater blasting using OD Technique and use of Shaped Charges.
- Removal of solidified slag (accretion) in kiln.

Late seventies and early eighties saw innovations and adaption of blasting techniques and explosive applications in India.

3.3 *Application of VCR technique*

3.3.1 *Drop raising*

VCR technique has been adopted by Hindustan Copper Limited and Vedanta Group (formerly Hindustan Zinc Limited) for drop raising followed by cutting of slot and subsequent stoping using 165 mm diameter drillholes. While VCR blasting is carried out with detonating fuse downlines initiated at the collar using millisecond electric delay detonators, multi-deck charging within a drillhole in blasthole stoping was carried out using non-electric shock tube based millisecond delay detonators.

3.3.2 *Adaptation of VCR technique for inducing caving of hard roof at Bijuri colliery*

The hard sandstone roof in Bijuri underground colliery of Hasdeo Area South Eastern Coalfields Ltd (previously Western Coalfields Ltd) was not caving causing concern. A sudden collapse of a large mass of roof into the goaf could cause airblast and damage. It was decided to induce caving artificially by blasting. Ideas such as blasting the roof using upward drillholes, creating s pre-split for the entire thickness of roof at one of the block were considered. Blasting specialists from IDL suggested cutting a notch in the solid sandstone roof above the goaf at one end of the block and another row perpendicular along bottom gate edge. Concept of VCR technique was adopted successfully. 10 drillholes of 150 mm diameter and approx 60 m depth, spaced 6 m apart were drilled in a row 10 m away from goaf edge to reach the goaf from surface. 5 drillholes were drilled perpendicular to it. The drillholes were plugged at the bottom (as carried out in VCR method) and charged with large diameter high strength cap-sensitive packaged slurry booster along with a single pellet of 500 g cast booster. The booster was lowered with the help of 20 g/m detonating cord till the drillhole bottom. Since the drillholes were very watery, extra care was taken to ensure that the cartridges reached the desired horizon inside the drillhole.

Blast was carried out and the caving occurred over a period of time, a testimony to the success of the blasting technique adopted for the project.

3.4 *Blast technique for recovery of large pieces of Asbestos and to reduce fines*

An underground Asbestos mine was generating lot of fines and small pieces during primary blasting and losing out on value. The mine wanted a technical solution for generating large pieces of asbestos. IDL provided a blasting solution in the form using low strength explosives (permitted category) and pre-splitting the asbestos—host rock interface through appropriate drilling and delay sequencing of drillholes during blast. The recovery of large pieces of asbestos helped the mine considerably.

3.5 *Sill pillar blasting using electric detonators*

For the first time in the history of Indian underground metal mining, IDL helped the mine to carry out a sill pillar blast using millisecond delays supplied with pure copper wire with thicker insulation for better insulation and protection in conductive ore body.

3.6 *Underwater blasting for deepening approach channel using OD technique and shaped charges*

IDL helped Dredging Corporation of India (DCI) to carry out underwater blasting of limestone at Tuticorin port, in Tamil Nadu, India for increasing the draught of the approach channel from the sea to facilitate entry of larger ships.DCI owned a drilling pontoon with three drills which could drill 6 rows of drillholes of 63/76 mm through a casing pipe. Assistance was provided to DCI by IDL by supplying suitable explosive, specially designed detonators for underwater application and pneumatic cartridge loader for charging. While sea bed was to be lowered by 1.5 m, drilling and blasting of 3 m deep drillholes was carried out.

IDL also provided indigenously designed and fabricated Shaped Charge canisters filled with aluminized cap-sensitive slurry booster. Array of canisters were lowered and placed on sea bed by divers employed by DCI. Each canister was primed with 10 $g.m^{-1}$ detonating cord. About 30–40 cm of cord was bundled into a ball and pushed into the slurry inside the canister for providing an overdrive to the explosive. The free end (pigtail) was brought out through a hole in the threaded cap provided with the canister. All pigtails were connected to a trunkline of 10 g/m cord and set off. Rock was shattered to a depth of 0.8 to 1.0 m.

Broken material was excavated using a Cactus Grab, loaded onto barges and discharged into the deep sea away from the approach channel.

3.7 *Coyote blasting*

Extension of outer harbor at Vishakapatnam, in the state of Andhra Pradesh India required 2,06,300 cubic meters of rock of different grades (sizes) and proportions for construction of southern and eastern breakwaters about 2 km from shore (Rath, 1979). The rock consisting of garnetiferous sillimanite gneiss of khondalites group was blasted at Lova Garden quarry.

Before undertaking the main coyote blast, a pilot blast was carried out at Dasarimetta quarry, about 18 km from Lova Garden quarry which was successful and gave confidence for major blast. A number of coyote blasts were carried out in the quarry using adit of 1.2 m × 1.5 m size and of 10–18 m length. Cross-cuts were driven at the end of the adit for a length of 19 m to 46 m. Explosive quantity used varied between 3 MT to 11 MT. Powder factor achieved was 0.24–0.33 kg/m^3. Sand filled bags and rock pieces were stacked in the adit as stemming.

The cross cuts were filled with conventional packaged slurry explosive Aquadyne and Energel and Pentolite Booster (manufactured by IDL). The shot was initiated using 10 $g.m^{-1}$ detonating cord D Cord-II which was laid from the mouth of adit till the ends of the cross cuts.

3.8 *Quenching gas well fire*

A blowout followed by a fire occurred at the Oil & Natural Gas Corporation's (ONGC) well No.19 near Pasarlapudi in East Godavari district of Andhra Pradesh on January 8, 1995. Nearly 1 million cubic metres of gas was gushing out every day and flame reached upto a height of 200 feet. The oil well fire was successfully quenched by detonating approximately 400 kg of plastic explosive near the mouth of the gas-spewing well. The detonation of the explosive near the raging fire depletes the surrounding area of oxygen causing the fire to die out. A water umbrella is created over the flame by continuously spraying water to bring down the temperature near the well before placing the primed explosive charge using an Athey Wagon, a track mounted vehicle with a long boom (Jain et al. 2012). The well was capped successfully on March 14 1995, 62 days after the blowout had occurred. The capping was carried out by Neal Adams Firefighters.

3.9 *Use of electronic detonators in India*

India has not lagged behind in the use of electronic detonators. Imported detonators were initially used for mass pillar blasting in underground metal mine. Otherwise its use was mainly to carry out cautious blasting for ground vibration control in sensitive areas. IDL developed electronic detonators under the brand name e-DET through its own R&D work and using indigenous in-house technology and first field blasting was carried out in 2004.

In India, use of electronic detonators is still confined to vibration control and critical blasting works. Factory programmed delays, factory programmed delay number field programmed delay interval (desired delay interval is set on the exploder and upon firing, delay intervals are generated viz. delay number multiplied by the delay interval set on the exploder) and fully field programmable electronic detonators are the versions introduced into the Indian market. Large scale use of electronic detonators for optimum overall cost benefit is still in a very nascent stage, though used regularly in few mines for primary blasts.

3.10 *Blasting underground coal pillars along with overburden (OB)*

Several underground coal mines with Bord & Pillar workings are converted as opencast mines. Often drillholes in the OB penetrate into the goaf or into coal pillars. During blasting, the coal pillars get damaged and during excavation of OB the coal gets contaminated. The mine wanted a technique to blast coal and OB simultaneously without coal getting contaminated. IDL suggested a design using two delay detonators in a drillhole, one in the coal pillar and the other in OB, separated by an inert deck. The delay sequence and time interval between bottom deck and top deck chosen would ensure the drillholes in coal pillar getting blasted first and the broken coal swells up and fills the voids (roadways). The upper deck is initiated after a few hundred milliseconds. During loading, the OB is handled separately and then the broken coal. The success of the method is dependent on:

- Accuracy of firing time of the two in-hole delays.
- Accuracy of survey for positioning drillholes in OB with respect to coal pillars and depth.

3.11 *Miscellaneous details*

The largest opencast overburden blast in India consumed approximately more than 800 Mt of explosive. Blast was carried out using bulk explosives and detonating cord downlines and trunklines delayed using detonating relays. Some mines carry out pre-splitting with large diameter drillholes (150 and 165 mm diameter) using small diameter cap-sensitive explosive cartridges of 25–50 mm diameter taped to a single downline of 10 g/m detonating cord. The drillholes are charged with a small quantity of explosive at the bottom to assist breakage at the drillhole bottom and drillholes are unstemmed. Some mines fill the drillhole with soft earth after ensuing that the explosive charge is placed axially in the drillhole. Concept of airdeck has also been utilized in mines to reduce charge per drillhole either to reduce costs or ground vibrations. Several innovations to create the airdeck such as bamboo basket held by nylon rope and lowered to the desired level or a wooden stick with two circular discs nailed at the ends have been tried successfully. Ammonium Nitrate Fuel Oil (ANFO) mixed with saw dust or rice husk is used in limestone quarries which rely more on gas (heave) energy.

3.12 *Explosive energy measurement using Underwater Method*

During the late seventies, IDL established facilities for measurement of explosive energy. Explosive energy was computed using computer software specially developed for the purpose and measured by ‘underwater test’ method (Pond Test). This method is considered a near-accurate simulation of the blasting action of explosives in rock. The shock energy and gas (bubble) energy of test sample are measured and compared with a standard explosive such as Pentolite (PETN:TNT, 50:50).

4 BLASTING TRENDS

The primary objective of using explosives for excavation is to fragment and displace the material for optimum downstream operations of loading, hauling and crushing when required. The blasts should also not generate boulders (oversize), should not create back break, side break and flyrock, ground vibration and airblast beyond threshold limits.

Fragmented rock in the muckpile is mainly from three sources:

- Fragments formed by new fractures created by the detonation process (compressive shock wave).
- Fragmentation formed by the extension of existing in-situ fractures, combined with newly generated fractures.
- In-situ boulders and cracked rock mass present in burden (caused by bad blasting practices) that get pushed into muck pile without any further breakage.
- Rock mass properties important to blasting performance:
- High-density rocks require more blast energy to loosen and displace it vis-à-vis low-density rock.
- Strong rocks require greater blasting effort than weaker rocks.
- Soft/Plastic rocks tend to ‘absorb’ explosive energy (mainly shock energy) resulting in less effective blasting.
- Absence of fractures or discontinuities increases the blasting effort required to achieve desired fragmentation.
- Dynamic compressive strength, which controls the crushing that, occurs at the blasthole wall.
- Dynamic tensile strength of the rock, which influences the extent of new, fracture generation in both shock and gas pressure phases.
- Density of rock mass which affects the inertial characteristics and hence how the rock mass moves in response to the forces applied during blasting.
- Rock mass stiffness which controls distortion of blasthole wall and hence the pressure developed inside the blast hole.
- Attenuation properties of the rock mass which control how far the stress waves travel before energy falls below the levels that cause primary breakage.
- In-situ fracture frequency orientation and character which together define the in-situ blocks, the attenuation of shock wave and migration/venting of gases.

Planning and monitoring of blasting operations can be divided into three distinct event based diagnostic entities (Sarathy 1994). They are:

a) Pre-blast monitoring
b) In-blast monitoring
c) Post-blast monitoring

a) Pre-blast

Parameters that can be evaluated are:

- Rock characteristics:
  Geophysical Logging:
  - Density.
  - Hardness.
  - Compressive strength.
  - Tensile strength.
  - Poisson's ratio.
  - Young's Modulus.
  - Longitudinal wave velocity (sonic velocity).
  - Grain size.

  Drill Monitoring:
  - Penetration rate.
  - Torque.
  - Pull down pressure.
  - Presence of hard and soft bands.
  - Presence of vughs, solution cavities, joints
- Explosive characteristics:
  Leadblock expansion.
  Ballistic Mortar.
  Underwater Test.
  Computer Codes:
  - Density.
  - Velocity of detonation (VOD).
  - Detonation Pressure.
  - Minimum Booster.
  - Strength/Energy- Weight and Bulk
  - Gas volume.
- Laser profiling of bench face:
  - Real frontal burden (crest and toe).
  - Presence of under cuts.
  - Presence of buffer.
- Survey of structures
  Detailed mapping of buildings for existing cracks, subsidence of foundation, etc.
- Borehole camera.

b) In-blast

- In-hole VOD of explosive.
- In-hole detonation and borehole pressure.
- High speed photography/videography.
  Quantitative:
  - Firing time of drillholes.
  - Burden 'response' time.
  - Burden 'move-out' velocity.
  - Bench top uplifting velocity.
  - Venting velocity.

  Qualitative:
  - Misfires/malfunction of initiator.
  - Gas energy loss from discontinuities.
  - Stemming ejection.
  - Flyrock generation.
  - Monitoring ground vibration and airblast.

c) Post-blast

- Quantitative:
  - Muckpile Profile and its looseness.
  - Fragment Size Analysis.
  - Shovel/Dragline performance monitoring.
  - Crusher performance.
  - Hauling productivity.
  - Boulder count.
  - Secondary blasting.
  - Total cost evaluation.
- Qualitative:
  - Misfires/missed hole detection.
  - Flyrock range, probable source and cause.
  - Presence of toe, humps on bench floor.
  - Backbreak, overbreak and side-tear.
  - Pitwall stability.
  - Damage to structures.
  - After-blast fumes in pit and in atmosphere
  - Dust generation and its movement.

4.1 *Comparison of true bottom priming (point initiation) vis-à-vis multi-point initiation in bench blasts*

Till the late seventies and early eighties, most Indian surface mines utilized 10 g.m^{-1} detonating cord downlines for carrying out blasts. The drillholes were initiated at the collar using millisecond delay detonators or detonating cord trunklines delayed using delay detonators or detonating relays.

IDL introduced concept of true bottom hole initiation in bench blasting using its non-electric shocktube based pyrotechnic millisecond delay detonator Raydet during early eighties. Many studies were carried out in Limestone and Iron ore mines by practicing mining engineers, scientists and academicians to compare the results achieved with non-electric detonators vis-à-vis detonating cord downlines. IDL also sponsored a research project with the Mining Engineering Department of National Institute of Technology, Surathkal, Karnataka, India.

The various studies concluded significant advantages with the use of bottom hole initiation mainly superior fragmentation, reduced boulders, improved shovel productivity, reduced loading cycle time, reduced flyrock and airblast generation and absence of stemming ejection. The cost per tonne also reduced in spite of shock tube based systems being 3–4 times more expensive than detonating cord on cost per metre basis.

Some quantified data of studies in Iron ore mine are in Table 2

Studies in Limestone mines gave the following results with shock tube based non-electric detonator.

Mine 1:

- 19.2% reduction in digging time, 12% time in loading time and 7.27% in total cycle time.
- Boulders reduced by 33%.

Table 2. Output from Iron ore mine.

Project name	Purpose
Oversize	2.2% with Raydet 10% with detonating cord
Shovel productivity	275 MT/ hour with Raydet 225 MT/hour with cord
Cost per tonne	Rs.5.21 per MT with Raydet Rs.5.78 per MT with cord

- Average fragment size reduced by 31%.
- Noise levels reduced by 10%.

Mine 2:

- Shovel digging time: Reduced by 19.5%.
- Shovel loading time: Decreased by 12%.
- Cycle time: Reduced by 7.27%.
- Cost per tonne: Less by 18.75%.

Similar studies by firing the blasts with precise timings using electronic detonators are the need of the hour and are envisaged in the coming years. With increasing demand, the price of electronic detonators would reduce compared to present costs.

5 ELECTRONIC DETONATORS—OVERSEAS EXPERIENCES

Electronic detonators were introduced into the world market during the early nineties. It was observed that precision timing achieved with electronic detonators could deliver consistent blast-to-blast results previously unobtainable with traditional pyrotechnic blasting systems. By accurately controlling timing delays (firing times of drillholes), use of electronic detonators demonstrated (Kuhar 2005):

- Increase rock fragmentation (reduce average fragment size).
- Reduce oversize.
- Lower ground vibration levels.
- Lessen the potential of flyrock.

These translated into faster excavation times and improve downstream processing costs for the aggregate operation by increasing throughput, reducing crusher wear, and lowering power consumption and maintenance costs.

Controlled tests in a Pennsylvania quarry documented significant benefits using electronic detonators in place of pyrotechnic based non-electric delay initiation in production blasts, even without optimizing the blast design. The researchers reported the following results:

- 32% decrease in the mean size of rock in the post-blast muckpile.
- 37% increase in rock of 8 inches minus screen passing size.
- 25% reduction in digging time to excavate the muck pile.
- 6–10% savings in primary crushing costs, measured by power consumption.

Optimization of the blast design to take greater advantage of the electronic detonators' precision was expected to expand the blast pattern and reduce the powder factor without negatively affecting stone production.

Ground Vibrations were measured for 25 blasts as part of a project at Prospect Quarry (Flanagan, 2002) which revealed a 21% reduction in ground vibrations and 41% reduction in wall vibrations.

In another study in Commercial Stone Rich Hill Quarry (Bartley, 2003), results achieved with electronic detonators were as under:

- 43% increase in mean fragment size.
- 17% increase tonnage throughput at primary Crusher.
- Ground vibrations within regulatory limits with improved frequency (Hz) content, without a need for scaling down the size of blasts.

A study in quarry industry (Miller & Martin, 2007) reported:

- 23% increase in load and haul rates.
- 18% increase in crusher throughput
- 43% reduction in oversize
- 13% overall decrease in operational cost, despite increased blasting cost incurred through use of electronic detonators.

Benefits accrued through the use of precise electronic delay detonators to the user such as 52% reduction in ground vibration and 3% reduction in airblast noise (dB), 30% increase in fines and 50% reduction in boulder count, significant improvement in downstream operation cost in spite of using electronic detonators at higher input costs, have been reported in International Symposia.

6 NEED IN INDIAN MINES

Hagan (1983) suggested the use of 'stepped drillholes' with a rig and drillbit capable of drilling a drillhole having two or more different diameters along its length. The advantages of this revolutionary idea are:

- Rig will be used to drill a blasthole of desired drill diameter commensurate to the rock characteristics and properties of explosive in use.

A lower drillhole diameter at the collar area would reduce cost of drilling and would also require lesser stemming material. Stepped drillhole would offer better confinement to gases generated upon detonation of the explosive in the drillhole.
– Where a drillhole has excess burden or hard bands are encountered, the drill diameter can be increased in the zone to assist loading more explosive charge quantity or a stronger product to break the excessive burden or fragment the hard band.

Blast design and explosives application is a complex matrix comprising of:

– Average fragment size desired.
– Selecting suitable loading machine bucket size commensurate to production rate planned. Also to be considered are breakout force required and matching excavator with the haul truck.
– Choosing appropriate drillhole diameter. Use of large diameter drillholes in shallow benches causes mismatch in blast geometry and hence should be avoided. As per Indian mining law, bench height should not be more than the maximum reach of the loading machine deployed.
– Study the rock properties and choose explosive properties appropriately. Concept of 'impedance matching' would be a useful tool.
  Ramulu et al. (2012) conducted impedance matching studies in three different benches in an opencast coal mine and concluded that each bench needed explosive with different velocity of detonation viz 3400, 3700 and 4100 m/s.
– Maximize the use of delay initiators and provide the desired delay interval along spacing between drillholes and across the burden, commensurate to the blast geometry used and burden response time. Burden response time may be defined as the time interval from detonation of the explosive in the drillhole to changes observed in the burden and start of initial rock movement. This is observed using high speed videography of production blasts. Burden response time is dependent on the properties of rock being blasted and explosive characteristics.
– Signature hole tests provide adequate information that can be used as inputs in blast design, drill patterns, initiation sequence and vibration control.
– Especially with bulk explosive supplies, single weight energy product is mostly carried and only the bulk energy is varied through density control. The burden and spacing are changed depending on the rock type being blasted. Powder factor (Cu.m.kg^{-1}) gets reduced while blasting medium hard to hard strata as lower burden and spacing are used. Attempting to keep higher burden and spacing in hard rocks with low energy explosive results in poor fragmentation, boulder generation, and reduced movement of blasted mass, tight muckpiles and toe formation. Use of higher energy explosives provides scope to expand the drill pattern in order to maintain the powder factor.
– Blasts should be designed based on the explosive's energy and rock properties. Powder factor based blast designs adopted by mining companies should be dispensed with.
– Adopt cast blasting as a regular practice wherever possible and reduce the extent of mechanical handling of overburden.
– Evaluate total cost of operations and not individual cost of blasting (explosives and blast initiation systems) or on the basis of powder factor.
– Introduce the use of electronic detonators in regular primary blasts and study the improvements in efficiency of downstream operations.

7 CONCLUSIONS

A number of parameters optimally used are essential for achieving for good blast results. The first step in the excavation process is drilling. Proper choice of drillhole diameter commensurate to the loading machine deployed is very essential in the Indian scenario since mining law restricts bench height to the maximum reach of loading machine in use. The primary blast designs should generate desired fragment size commensurate to the bucket size of the loading machine in use. Muckpile profile and looseness required also vary as per the loading machine viz Dragline, Rope Shovel, Hydraulic Excavator, Back-Hoe or Front End Loader. Some opencast coal mines also use in-pit crushing technology in which the blasted OB is further crushed inside the pit using crusher and carried through conveyor belts to the dumps. Achieving average fragment size commensurate to need in primary blasts itself requires scientific application of blast inputs.

Proper inputs result in good blasts having the desired fragmentation and muckpile looseness for efficient downstream operations of loading, hauling and crushing. Blast designs and explosive selection based on scientific measurements and utilizing information gathered through study of impedance and signature hole analysis would enable carry out optimum blasts. Mines would have to use explosives having different VODs and energy content for blasts in various types of strata encountered in the benches in a mine.

Firing times of drillholes and sequence of initiation in primary blasts has a direct impact on the resulting fragment size and muckpile looseness. Use of electronic detonators ensures precise control of drillhole firing times and initiation sequence. Further, electronic detonators initiate the drillholes truly at the bottom and does not disturb the stemming or desensitize the explosive charge. The explosive energy generated inside the drillholes gets optimally utilized for good blast results.

Indian mines have to carry out blasting in a scientific manner both in underground and on surface keeping environment and safety issues in mind. Need for monitoring effect of after blast fumes and dust generated from surface blasts would become a responsibility of surface mines in the future. Mines will benefit by carrying out studies on optimum blast and total cost, rather than attempt to lay emphasis the cost of explosives and powder factor.

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REFERENCES

Bartley, D.A. 2003. Further field applications of electronic detonator technology. Davey Bickford; Field Applications downloads.

Flanagan, M. 2002. Application and performance of electronic detonators; Thesis Bachelor of Engineering; School of Mining Engineering, University of New South Wales.

Grant, C.H. 1969. Blasting—An Art or A Science? *Mining Congress Journal*: 79–81.

Hagan, T.N. 1983. Some recommended features of drilling equipment—a blasting engineer's view. *Second international surface mining and quarrying symposium*, Bristol. October 4–6.

Ignatyuk, G.L. 1970. Rockfill dam construction in the Soviet Union with the use of directed blasting.

Jain, C.K, Yerramilli, S.S, Yerramilli, R.C. 2012. A case study on blowout and its control in Krishna-Godavari (KG) basin, east coast of India: safety and environmental perspective. *Journal of Environment and Earth Science* 2(1): 49–59.

Kuhar, M.S. 2005. *Pit & Quarry* (January 01).

Miller, D. & Martin, D. 2007. A review of the benefits delivered using electronic delay detonators in the quarry industry. Proceedings Quarrying 2007, Australia.

Ramulu, M, Sangode, A.G, Sinha, A. 2012. Blast optimization with in-situ rock mass characterization by seismic profiling at an opencast coalmine in India. *2012 Coal Operators' Conference*, University of Wollongong.

Rath, B.K. 1979. Coyote Blast helped construction company to solve its production problem. *National Seminar on Explosives and Blasting Techniques*, Singrauli,

Sarathy, M.O. 1994. Recent advances in surface blast monitoring and evaluation techniques—A review. *The Indian Mining & Engineering Journal* (March): 45–56.

Blasting in Mines – New Trends – Ghose & Joshi (Eds)

Application of bunch-hole blasting in recovering residual ore in irregular goaf groups

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ABSTRACT: Random mining sequence in vein-belt of Tongkeng tin mine, China, resulted in irregular goafs above 560 m level, leaving a number of residual ore and causing serious ground pressure hazard. It was very difficult to recover the residual ore and solve the hazard due to unidentified boundaries of goafs. Zonal mass caving with bunch-hole blasting was carried out. In the scheme, the total explosive load was 150 t, resulting in 770,000 t of ore broken and 6,500 m^2 of caved areas. Continuous caving of overlying rock that progressed from the blast site to the surface basically eliminated the impending disaster of underground pressure and helped safe recovery of the ore resources.

1 INTRODUCTION

Extraction of the vein-belt orebody is one of the main operations at Tongkeng tin mine. Sublevel caving without sill pillar method of mining had left 135,000 m^3 irregular goaf groups without filling, and many of them had already collapsed, with more than 3 million tons of residual ores.

Further collapse of these goafs might have caused great hazards, resulting in equipment damage and personal injury.

Spontaneous combustion occurred since 1976 due to high sulphur content in the ore. To ensure production safety and prevent fire from spreading, fire-proof ore pillars were provided from 625 m to 650 m level.

To extinguish the fire and handle the impending disaster of underground pressure, fire sources of high sulphur ore were to be fully and efficiently eliminated.

The recovery of residual ores was also beneficial from mine service life and economic considerations. The recovery was carried out usually together with goaf group treatment (Sun, 2006). To prevent interference to residual ore pillars and worsening of goaf groups conditions, small-scale operation should be avoided. The suitable method of recovery should be characterized by fast, effective and centralized operation (Wang, 2008).

2 METHODOLOGY

2.1 *Exploitation current situation of the vein-belt ore body*

The ore is in vein-belt, thick plate or plate status and mineralization occurs mainly along fractures. It is mainly in block structure, stringer structure or disseminated structure with the strength factor of $f = 8 \sim 12$ and the density of 2.5 g/cm^3, no large structural fracture.

Stopes #15 and #16 are located to the east of the treatment and stope #11 is located to the west (Figure 1). The mined out stopes were partially filled with waste rocks. Spontaneous combustion fire area was above 650 m level, and causing high rock temperature near 650 m level.

2.2 *Solutions*

Zonal mass caving was used to deal with the impending disaster. Most of the goafs were to be caved with deep long bunch-holes. Medium long hole were used additionally in those locations where long hole could not be drilled. Small cham-

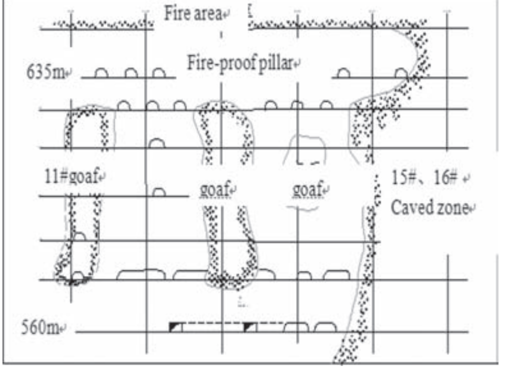

Figure 1. Goaf groups.

ber explosion was used to destroy the fire-proof pillar. The goaf groups above 560 m, stopes #11, #12, #13, #14 and the fire-proof ore pillar were to be caved at one time.

In the layout of 12# and 14# stopes, there were branch roadways used for sublevel stoping method. A large number of branch roadways, shallow mined-out area and local caving were the disadvantages for overall caving. There were 714,000 m^3 mined-out area, space of roadways 178,000 m^3, the total volume was 892,000 m^3, as well as 770,000 t residual ores. The treatment scheme demanded for tight blasting with a small compensation space.

2.3 *Technical introduction of bunch-hole*

The basic concept of bunch-hole blasting is that, one bunch-hole is formed by several nearby holes arranged in parallel, and distance between the single holes is 3~8 times the diameter, holes (diameter d) of the bunch-hole are detonated at the same time. The blasting effect to the wall rock may be equivalent to a larger diameter hole (Blasthole diameter D) as shown in Figure 2 (Sun, 1984).

Compared with the common single-hole, the stress field and stress wave of bunch-hole blasting are in certain "thickness", the destruction of stress wave to rock continues longer time, with larger impulse and broader range, which makes broken effect much better.

The advantage that the number of holes in bunch-hole and the way of holes arrangement can be properly adjusted according to burden, makes the technology highly flexible and applicable.

As to the common single-hole blasting, the stress field and stress wave formed by bunch-hole blasting are in certain thickness and the rock is under the action of stress wave, with longer time, larger impulse and broader range of action; therefore, the broken effect is better (Sun, 1984).

2.4 *Mining project*

As drifts and ore pillars of 584 m, 596 m, 613 m, 625 m level were seriously damaged, it was impossible to carry out caving in these levels for poor safety conditions. Caving chamber was to be set only in 635 m level where working conditions were better and most of the preparatory works centralized in 635 m level.

All the stopes and roadways in each level were composited together to find places where caving chambers could be arranged. Make sure the bunch-hole can control all the residual ore from 570 m to 635 m (Figure 3 and Figure 4).

According to the compound analysis, the blasting holes were arranged in 635 m, the holes should be punched to 570 m level, and some of them might fall to the goaf group. The diameter of hole was 165 mm. The bunch-holes consist of 3~9 single holes according to the burden, and the bunch-hole patterns were linear, rectangular square, circular or triangular type (Figure 6).

Altogether there were 24 bunch-holes, consisting of 177 single holes and the total length of holes was 9,855 m. Many of the holes were terminated at the boundary of goaf to prevent from falling into goaf and mutual penetrating. The hole deviation should be strictly controlled while punching.

In 570 m and 584 m levels, upper-medium-deep holes were set at a few places where could not be controlled by holes from 635 m.

Figure 2. Bunch-hole with an equivalent larger diameter.

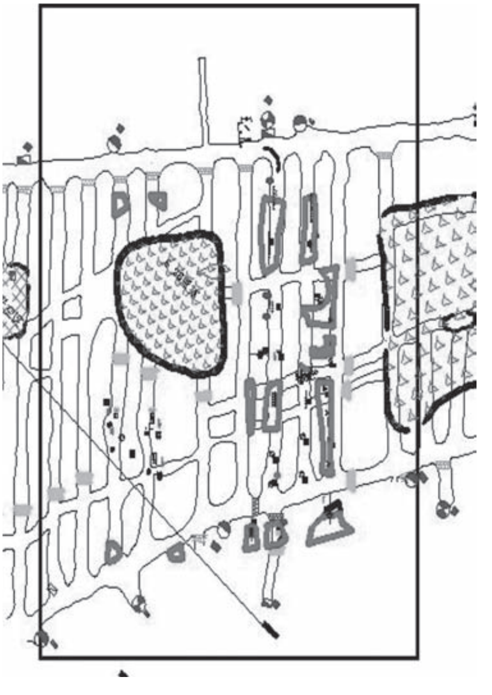

Figure 3. Locations (shown in red) in 635 m showing whereholes could be drilled to 570 m.

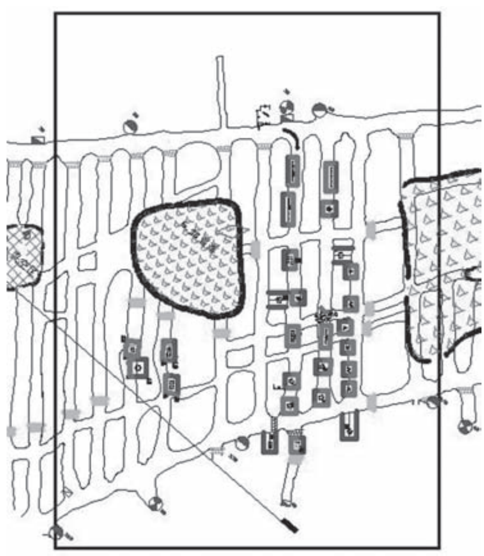

Figure 4. Arrangement of blasting-hole in 635 m level.

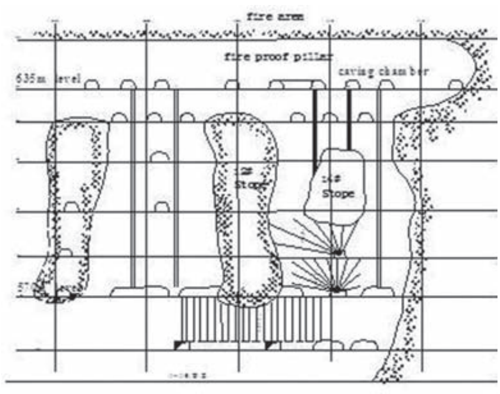

Figure 5. Profile map of bunch-holes.

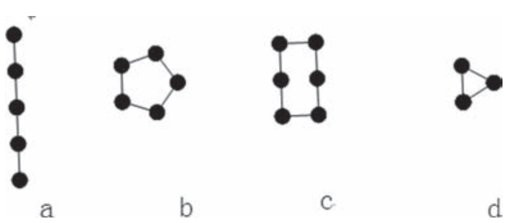

Figure 6. Different forms of Bunch-hole; a—Linear bunch-hole; b—Circular bunch-hole; c—rectangular bunch-hole.

Due to fire the temperature of fire-proof ore pillar was very high. To avoid blasting in high temperature, five small chambers were set in 635 m instead of holes to cave the fire-proof ore pillar.

Trench base structure was located at 560 m level, and a scraper was used to remove ores (Sun, 1992).

2.5 *Blasting scheme*

The irregular goaf groups were taken as the compensation space for blasting, so the burden of blasting holes varied according to the irregularity boundary. The number of holes for each bunch-hole and charging pattern of bunch-hole also varied depending on the burden.

2.6 *Safety measures*

Multi-section millisecond delayed detonation was used. To reduce superposition of blasting vibration, the detonation of hanging wall and foot wall was done alternatively. Shockwave-resistance walls were set in key places to prevent the impact wave from damaging the engineering equipment (Sun, 2006).

3 RESULTS

The caved blasting area was 6,500 m^2, the caved ore was 770,000 t and the overall explosive quantity used was 150 t. Blasting was in form of mini-interval detonation in 20 sections and overall detonation duration was 2 seconds. This was the largest underground mine blast with large diameter holes ever conducted in China.

The whole blast area was completely caved in after blasting but caused no damage to the adjacent stopes and engineering facilities. The impending disaster due to underground pressure and fire were eliminated. Thus, favorable conditions were created for the recovery of the ore resources (Figure 8).

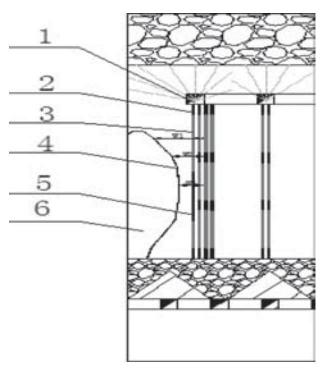

Figure 7. Schematic diagram of charging structure with changing burden; 1—Caving chamber; 2—Stemming; 3—Explosive; 4—varying burden; 5—decking; 6—Irregular goaf.

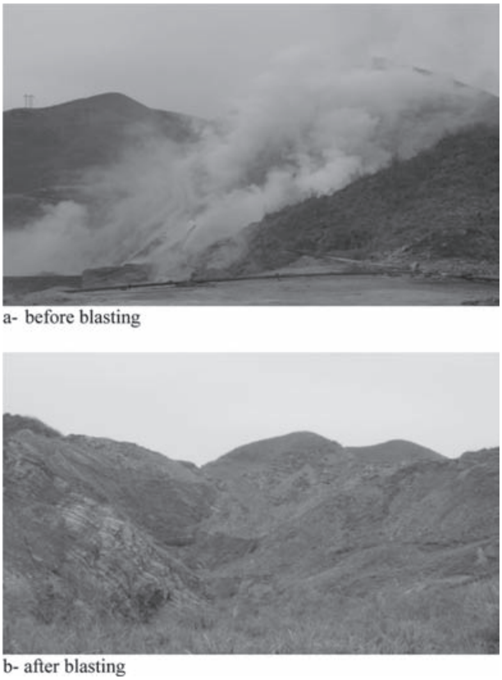

Figure 8. Smoke of the fire area.

4 CONCLUSIONS

Roadway-type caving chamber supported by large size continuous ore pillars was formed in drilling level, creating a safe operation environment.

The operations were centralized in the caving level and ore removal level, which improved the management of operation and the quality of ventilation.

The available goaf group should be fully utilized as compensation space for blasting.

The technology could be applied to deal with a large number of irregular goaf groups and safe recovery of residual ores.

REFERENCES

Sun, Z.M. 1984. Experimental investigation on VCR mining method, *China Nonferrous Metals*, Volume 2: pp. 7–16.

Sun, Z.M. & Chen, H. 2006. Application of new mass caving technology of bunch-hole equivalent spherical charge. Collected Works of Forum on New Advances in Mining Science and Technology of Metal Mines of Mining Research and Development, pp. 4–6.

Sun, Z.M., Chen, H & Wang, H.X. 2006. Huge ore caving technology with spherical explosive cartridges and cluster drilling holes, *A Collection of Papers of Forum on New Advances in Mining Science and Technology of Metal Mines*, pp. 4–6.

Sun, Z.M., Qiu, X.T. & Rao, Q.L. 1992. Industrial experiment on bunch deep hole phase caving and continuous ore withdrawal and transportation mining, *China Nonferrous Metals*, Special Issue: pp. 63–71.

Wang, H.X., Chen, H. & Sun, Z.M. 2008. Recovery method of residual ore in underground mine, *Mining and Metallurgy*, pp. 24–26.

Blasting in Mines – New Trends – Ghose & Joshi (Eds)

Investigation into the influence of air-decking on blast performance in opencast mines in India: A study

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ABSTRACT: A study was conducted to evaluate the influence of air-decking on blast performance and blast economics and its feasibility in the production blasting of jointed rocks in an opencast manganese mine and sandstone rocks in an opencast coal mine. A series of blast trials were conducted in 6–15 m high benches with varying spacing and burden using mid-column single and double air-decks and conventional charge blasting. The analysis of blast results indicated that the effectiveness of air-decking was more pronounced in very low to low strength rocks with medium to large in-situ blocks than in medium strength rocks with small size blocks. The overbreak, throw and ground vibration were reduced by 50–86%, 10–35% and 30–90% respectively in air-deck blasting as compared to conventional charge blasting.

Air-deck blasting maximised the fragmentation and produced a more uniform fragmentation. The mean fragment size was reduced from 0.68–1.0 m to 0.27–0.51 m in jointed rocks. The fragmentation index was increased from 1–1.47 to 1.96–4.34 in sparsely jointed rocks and from 1.38 to 1.57 in closely jointed rocks. Based on this study, the air-deck length as a fraction of original charge length was recommended in the ranges of 0.1–0.2, 0.2–0.3 and 0.3–0.4 for different rock masses as defined by RMR in the ranges of 45–65, 35–45 and 20–35 respectively. Feasibility of air-deck blasting was defined as excellent in very low to medium strength sandstone rocks, very good in very low to low strength sparsely jointed rocks and good in medium strength blocky sandstones and closely jointed rock masses. The study demonstrated that air-deck blasting could be effectively used in routine production blasting in opencast mines to improve techno-economic efficiency of blasting. This paper presents the details of blast trials conducted, discusses the blast performance parameters and present the results.

1 INTRODUCTION

In rock blasting operations, explosives provide a concentrated source of energy, which often exceeds the level that is required to cause adequate breakage in the surrounding rock mass. Charge configurations play a significant role in achieving required blasting performance. The different charge designs commonly used are: full column—fully coupled high explosive charges, full column—fully coupled low density low VOD charges, full column—decoupled charges and fully coupled decked charges using either air or solid decks (Jhanwar, 2011).

In concentrated charge blasting, as a full column of explosive detonates, the tremendous initial pressure that arises in explosion products greatly exceeds the strength of the rock mass, so that a strong shock wave begins to propagate into the medium, crushing it and breaking it into extremely small particles. Because of this intense, excessive crushing of the rock, a large portion of the explosive energy is wasted in an area near the charge (Chiappetta & Memmele, 1987 and Moxon et al., 1993).

In air-deck blasting, the presence of air gap allows the explosion product gases to move and expand into the air gap, thus decreases the initial bore hole pressure. The shock waves oscillate in the bore hole, interact mutually and also with stemming column and/or hole bottom. The repeated interactions result in the generation of reinforced secondary shock front and allow shock waves to act over the surrounding rock mass for a longer period (Mel'Nikov & Marchenko, 1971, Fourney et al., 1981 and Moxon et al., 1993. Since the shock waves oscillate repeatedly within an air-gap, their velocities and pressure at the wave front are governed by the length they travel within the air column. Air-deck length is, therefore critical to the fragmentation. The effectiveness of this technique is also controlled by the rock mass structure and its strength (Jhanwar, 2011).

Mel'Nikov & Marchenko (1971) and Mel'Nikov et al. (1979) reported that regardless of the rock strength and explosive type as well as blasting procedures, the use of air-deck charges substantially improved the degree and uniformity of fragmentation. Chiappetta & Memmele, 1987 reported full scale trials of air-deck charges in a coal mine to characterize their effects in a production environment. Mead et al. (1993) reported the use of air—decks in production blasting in three cases one each of copper, iron ore and coal mine. The air-decks were used to provide more even explosive distribution within the hole. Explosive consumption was reduced by 15 to 35% without any adverse effects on the diggability of the material.

Over the period, air-deck blasting technique has been applied in a variety of applications like presplitting, controlling ground vibrations and fly rock, reducing fines and improving blast economics in opencast mines across the world. The mechanism of rock breakage in air-deck blasting is not fully understood and its use does not necessarily improve blast results in all types of rock mass and other geo-mining conditions. In order to evaluate the influence of air-deck blasting on blast performance a series of blast trials were conducted in opencast manganese and coal mines. The results of these trials are discussed in this paper.

2 GEOMINING DETAILS OF THE MINES

The opencast mines (Mine A and Mine B), where the field studies were conducted were situated near Nagpur in the state of Maharashtra, India.

2.1 *Mine A*

This is an opencast manganese mine with an annual overburden excavation and ore production of 0.3 million m^3 and 0.18 million tonne respectively. Shovels and hydraulic excavators are deployed in combination with 25 and 15 t dumpers for the excavation of overburden and ore respectively. The ore body at this mine is lensoid in shape, consisting of aluminium minerals of braunite, pyrolusite and sillimanite. The footwall consists of muscovite schists and the hangwall rocks are biotite-gneisses and quatzitic-gneisses, grading into schistose rocks. The strata, in general, strike due east-west, with a southerly dip varying from 55 to 60°. The ore body width varies from 6 to 35 m, with a strike length of 500 m and thickens towards the east.

The rock mass in both the footwall and hangwall sides of the mine consists of four joint sets, including one set of schistocity. The volumetric joint count (J_v) in the footwall varied from 5.8 to 9.0, which indicated medium sized in-situ blocks. The J_v in the hangwall side varied from 11 to 21, which indicated small-sized in-situ blocks. The rock mass rating (RMR) as suggested by Bieniawski (1973) was determined to assess the rock mass quality at this mine. The RMR varied from 24 to 65, which indicated poor to good rock mass conditions. The uniaxial compressive strength of footwall rocks varied from 4 to 40 MPa, which indicated very low to low strength rocks, and it varied from 58 to 65 MPa in hangwall rocks, which indicated medium strength rocks (Jhanwar, 1998 and Jhanwar et al., 2000).

2.2 *Mine B*

This is an opencast coal mine forming a part of the Wardha Valley Coalfield in central India. Annual overburden excavation and coal production at this mine are at 4.9 million m^3 and 1.6 million tonne respectively. The overburden at this mine consists of coarse grained ferruginous sandy soil and black cotton soil of up to 2.3 m thickness, followed by fine to medium grained yellowish and white sandstones, which measure 35–45 m in thickness. The coal seam is 17–20 m thick with a dip of 1 in 8 at S 54°30′ E. The coal seam floor consists of 3.25 m thick grey sandstone. A typical geological section is shown in Figure 1 (Jhanwar & Jethwa, 2000).

The rock mass at this mine was broadly homogeneous and was free of regular jointing. The RMR of sandstone rock mass varied from 20 to 60, which indicated poor to good rock mass conditions (Jhanwar, 1998).

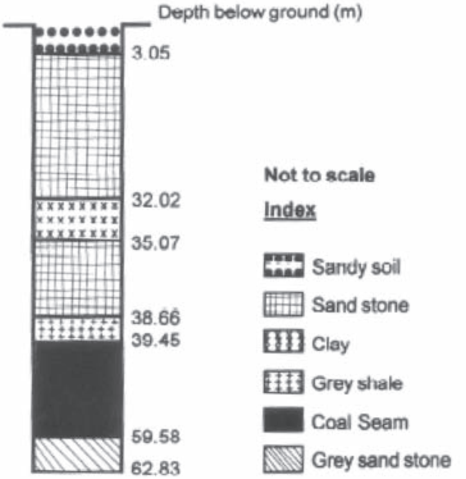

Figure 1. Geological section of a typical bore hole (After Jhanwar & Jethwa 2000).

3 BLASTING AT A MANGANESE MINE (MINE A)

A series of blast trials, which included blasting with solid decks and air-decks were conducted in the jointed rocks of footwall and hangwall side benches. Each blast was monitored for parameters like fragmentation, throw, back break, ground vibration, etc (Jhanwar, 1998).

3.1 *Conventional charge blasting at mine A*

A total of six blasts were conducted using solid decks of drill cuttings at various locations in the overburden benches. The charge pattern in a typical conventional blast is shown in Figure 2. The variations of different parameters in these blasts were as follows; Number of holes in each blast: 9–24, Hole depth: 6.0–10.5 m, Spacing: 2.5–3.0 m, Burden: 2.0–2.5 m and Powder factor: 0.37–0.40 kg/m^3.

3.2 *Air-deck blasting at mine A*

A total of eleven blasts were conducted at different locations in footwall and hangwall side benches (Jhanwar, 1998 & Jhanwar et al., 2000). The variations of different parameters in these blasts were maintained as follows; Number of holes: 6–26, Hole depth: 6.0–10.75 m, Spacing: 2.5–3.5 m, Burden: 2.0–2.5 m, Charge per hole: 11.12–34.56 kg, Powder factor: 0.28–0.89 kg/m^3 and Air-deck length: 0.9–2.4 m. The ratio of air-deck length and original charge length (ADL) was maintained in the range of 0.20–0.40. The charge pattern in a typical air-deck blast is shown in Figure 3.

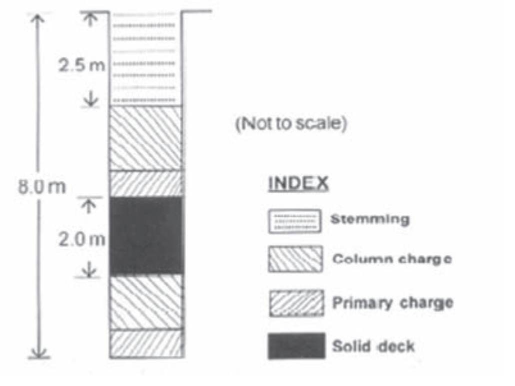

Figure 2. Charge pattern in a conventional blast at mine A. After (Jhanwar et al. 2000).

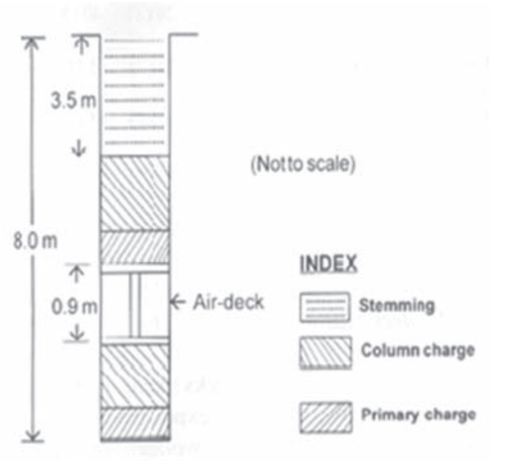

Figure 3. Charge pattern in an air-deck blast at mine A (After Jhanwar 1998).

3.3 *Influence of air-decking on blast performance*

The influence of air-decking was assessed in terms of mean fragment size (MFS), fragmentation index (FI), back break, throw, ground vibration and blasting cost. The analysis of blast results indicated that air-deck blasting produced a more uniform fragmentation as compared to conventional blasting. The difference in MFS between conventional and air-deck blasts was prominent in footwall rocks as compared to hangwall rocks, which signified that air-decking was more effective in very low to low strength moderately jointed rocks than in medium strength highly jointed rocks. The FI, which defined blast induced reduction in in-situ block size, was estimated as the ratio of average in-situ fragment size to average muck pile fragment size. The FI increased from 1.38 to 1.58 and from 1.0–4.7 to 1.97–4.34 in hangwall and footwall rocks respectively. The advantages of air-deck blasting in terms of different parameters are shown in Table 1.

4 BLASTING AT A COAL MINE (MINE B)

A series of blast trials, which included blasting with solid decks and with air-decks were conducted in the sandstone rock masses of this mine (Jhanwar, 1998; Jhanwar & Jethwa, 2000). Cartridge slurry explosive was used for blasting. The blast hole diameters were 150 and 250 mm. Initiation was done using electric delay detonators on the surface Each blast was monitored for various

Table 1. Advantages of air-deck blasting in manganese mine.

Parameter	Improvement
Fragmentation	
MFS	Near optimum size
Secondary blasting	Almost eliminated
Shovel efficiency	Increased by 50–60%
Ground vibration	Reduced by 44%
Backbreak	Reduced by 60–70%
Toe problem	Almost eliminated
Throw of muck pile	Reduced by 65–85%
Explosive cost	Reduced by 31.6%
Loading cost	Reduced 36.3%

parameters like fragmentation, throw, backbreak, ground vibration, etc.

4.1 *Conventional charge blasting at mine B*

In this case, solid decks of drill cuttings were placed to separate the explosive charge in a blast hole. The deck length varied in the ranges of 2–3 m and 4–5 m for 8–10 m and 16 m deep blast holes respectively. The charge pattern in a typical conventional blast is shown in Figure 4.

4.2 *Air-deck blasting at mine B*

Ten numbers of blast trials were conducted using air-decks in the blast holes. Air-decks were introduced in the blast hole in the middle of the explosive column using a wooden spacer. The variations of different parameters in these blasts were maintained as follows; Number of holes: 8–28, Hole depth: 6.0–15.0 m, Spacing: 5–7 m, Burden: 4.5–6.0 m, Charge per hole: 46–137.5 kg, Powder factor: 0.14–0.29 kg/m^3 and Air-deck length: 1.5–3.0 m. The ADL was maintained between 0.20 and 0.47. The charge pattern in a typical air-deck blast is shown in Figure 5.

4.3 *Influence of air-decking on blast performance*

The assessment of fragmentation was made by studying the shovel loading cycle and visual analysis of muck pile during the shovel loading operation. The impact of air-deck blasting as compared to conventional charge blasting in terms of different parameters is shown in Table 2.

In air-deck blasting due to the reduction in explosive charge, the shock energy responsible for crushing was significantly reduced. Further, the mid column air-deck allowed expansion of explosion products into the air gap and induced repeated oscillations of shock waves in the medium. As a result, the initial bore hole pressure was reduced and the duration of shock wave acting on the surrounding rock mass was increased.

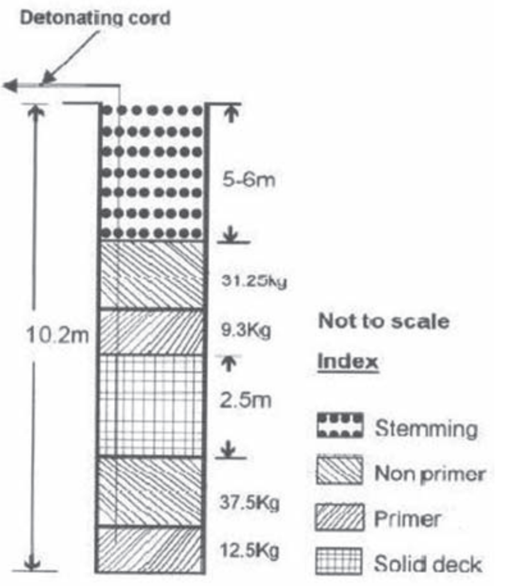

Figure 4. Charge pattern in a conventional blast at mine B.

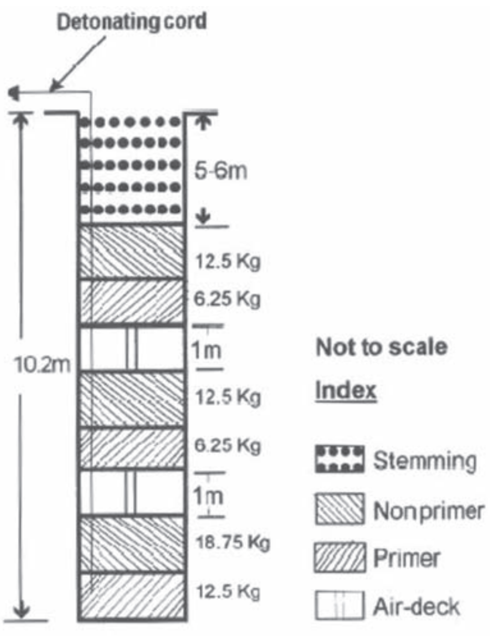

Figure 5. Charge pattern in an air-deck blast at mine B.

Table 2. Advantages of air-deck blasting in a coal mine.

Fragmentation	
• Became uniform	
• Fines reduced by up to 70%	
Shovel efficiency	Increased by 20–40%
Ground vibration	Reduced by 30–94%
Backbreak	Reduced by 50–80%
Throw of muck pile	Reduced by 65–85%
Explosive cost	Reduced by 10–35%
Shovel loading cost	Reduced 36.3%

Table 3. Feasibility of air-deck blasting (after Jhanwar 1998).

Type of rock mass	Feasibility
Very low to medium strength Sedimentary rock	Excellent
Very low to low strength Sparsely jointed rock	Very good
Medium strength sedimentary rock	Good
Closely jointed rock	Good

Table 4. Air-deck length for different rock masses (after Jhanwar 1998).

RMR (Bieniawski's, 1989)	ADL*
20–35	0.30–0.40
35–45	0.20–0.30
45–65	0.10–0.20

*Air deck length as a fraction of the original charge length.

5 GUIDELINES ON AIR-DECK BLASTING

Based on the investigation, a feasibility index for air deck blasting as shown in Table 3 and guidelines on ADL for various rocks masses in opencast mines as shown in Table 4 were proposed (Jhanwar, 1998 and Jhanwar, 2011).

The author feels that this technique is more suitable in highly jointed rocks (the in-tact rock type may be strong), soft and medium strength rock types which require less amount of shock energy and more of gas energy i.e. situations where blasting is required to induce little additional cracks and to shake the rock mass for heaving only. As the mechanism involves the interaction of two gas fronts leading to a reinforced stress field and explosive action for a prolonged period, this basic phenomenon is therefore expected to help in almost all blasting situations.

The techno-economic feasibility of this technique was found to be governed by the rock mass structure, air-deck size and the desired blast results besides other design parameters. In the case of a medium strength and sparsely jointed overburden rocks, if the blasted muck is worked by a larger bucket size, the resulting fragmentation is not of much concern for the systems productivity and thus long air-deck could be used and if the blasted muck is loaded by a small bucket size, the concern for fragmentation overrides other considerations and hence air-deck length has to be cautiously selected. Still, some generalizations can be made as to the reasonable range of air deck length (Jhanwar, 2011).

6 CONCLUSIONS

The air-deck blasting technique is significantly effective in very low to medium strength rock masses. It causes more uniform fragmentation with minimum fines and oversize as compared to the conventional charge blasting.

The effectiveness of air-deck technique in improving fragmentation is more significant in very low to low strength moderately jointed rocks than in medium strength highly jointed rocks.

Air-deck blasting maximizes the fragmentation in jointed rock masses as indicated by the reduction in MFS and increase in FI.

Better utilization of explosive energy in this technique offers other advantages in terms of reduced back break, ground vibration and throw.

Since the gaseous products expand into an air gap, the gas pressure in air-deck blasting reduces and consequently the throw of muck pile is also reduced.

Improved explosive energy utilization in air-deck blasting induces improvement in blast economics through reduction in explosive cost and increase in shovel loading efficiency due to better fragmentation.

Based on this study, the ADL was recommended in the ranges of 0.1–0.2, 0.2–0.3 and 0.3–0.4 for different rock masses as defined by RMR in the ranges of 45–65, 35–45 and 20–35, respectively. Feasibility of air-deck blasting was rated as excellent in very low to medium strength sandstone rocks, very good in very low to low strength sparsely jointed rocks and good in medium strength blocky sandstones and closely jointed rock masses.

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REFERENCES

Bieniawski, Z.T. 1973. Engineering classification of jointed rock masses. Transactions of South African Institute of Civil Engineers. 15 (12), pp. 335–344.

Bieniawski, Z.T. 1989. Engineering rock mass classifications. John Wiley and Sons, p. 251.

Chiappetta, R.F. & Memmele, M.E. 1987. Analytical high—speed photography to evaluate air-decks, stemming retention and gas confinement in pre-splitting reclamation and gross motion studies. In: Proceedings of the Second International Symposium on Rock Fragmentation by Blasting. Society for Experimental Mechanics, Bethel, CT, USA, pp. 257–301.

Fourney, W.L., Barker, D.B. & Holloway, D.C. 1981. Model studies of explosive well simulation techniques. Int. J. Rock Mechanics & Mining Sciences and Geomechanics. Abstracts, 18, pp. 113–127.

Jhanwar, J.C. 1998. Investigation into air-deck blasting and its influence on blast performance and economics in open-pit mines. Unpublished M. E. Thesis, Department of Mining Engineering, Visvesvaraya Regional College of Engineering, Nagpur University, Nagpur, India, p. 142

Jhanwar, J.C. 2011. Theory and practice of air—deck blasting in mines and surface excavations—A review. Geotechnical and Geological Engineering, Vol. 29, No. 5, pp. 651–663.

Jhanwar, J.C. & Jethwa, J.L. 2000. The use of air-decks in production blasting in an open-pit coal mine. Geotechnical and Geological Engineering. 18: 269–287.

Jhanwar, J.C., Jethwa, J.L. & Reddy, A.H. 2000. Influence of air-deck blasting on fragmentation in jointed rocks in an open—pit manganese mine. Engineering Geology: 57, pp.13–29.

Mead, D.J., Moxon, N.T., Danell, R.E. & Richardson, S.B. 1993. The use of air-decks in production blasting. In: Proceedings of the 19th Annual Conference on Explosives and Blasting Technique, International Society of Explosives Engineers, Cleveland, Ohio, USA, pp. 219–226.

Mel'Nikov, N.V. & Marchenko, L.N. 1971. Effective methods of application of explosive energy in mining and construction. In: Twelfth Symposium on Dynamic Rock Mechanics. AIME, New York, pp. 350–378.

Mel'Nikov, N.V., Marchenko, L.N., Seinov, N.P. & Zharikov, I.F. 1979. A method of enhanced rock blasting by blasting. IPKON AN SSSR, Moscow, Translated from Fiziko-Tekhnicheskie Problemy Razrabotki Poleznykh Isko-Paemykh (Journal of Mining Science), No. 6, pp.32–42.

Moxon, N.T., Mead, D. & Richardson, S.B. 1993. Air-decked blasting techniques: Some collaborative experiments. Trans. Inst. Min. Metall. (Sec. A: Mining Industry), 102, Jan–Apr., pp. A25–A30.

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Air spring vibration absorber blasting technology research and application

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ABSTRACT: The set of air column in the blasthole is similar to the energy storage component in hydraulic vibration impact system, which is gas accumulator. The main principle is the use of large energy storage capabilities of the gas storage chamber to absorb impact, eliminate pulsation and recover energy. When explosive column exploded in the hole, the air columns placed in the hole absorb the peak impact energy, and convert into smooth transitions clipping compression energy, to improve energy efficiency, reduce peak vibration. Using this principle, the Chongqing Nanping Central Transportation Hub Project 550,000 m^3 rock blasting engineering has been completed successfully, and format a mature engineering method by combining with process and technology.

1 INTRODUCTION

The Chongqing Nanping Central Transportation Hub Project is underground four-story highway-light rail dual use structure, foundation pit length 660 m, width 48 m, the maximum excavation depth is 33 m, total blasting excavation is 550,000 m^3. The project is located in Nanping CBD, excavation surrounding clouds of high-rise buildings, the nearest place's distance less than 2 m, and raised extreme demands of blasting vibration effects control. Facing the unprecedented complexity foundation pit blasting, the design uses active dynamic control and auxiliary, isolation vibration, such as energy storage-by-hole damping, group control of amplitude and frequency, blasting sequence adjustment in main vibration direction, and wall control blasting,etc. Fully control the project safety, quality and duration.

2 BLASTING VIBRATION CONTROL ANALYSIS

Rock blasting vibration control in the CBD area is a comprehensive, systematic design, should take the initiative dynamic control and auxiliary damping, vibration isolation both in terms of consideration. The dynamic control is vibration source interference control, including important factors such as frequency and amplitude, frequency and blasting sequence, power transmission process, etc. Auxiliary passive control is the safety isolation and protection, the most common method is presplitting and decay, observation, monitoring and control, reinforcement and protection, etc.

2.1 *Blast hole storage damping*

The set of air column in the blast hole is similar to the energy storage component in hydraulic vibration impact system, which is gas accumulator. The main principle is the use of large energy storage capabilities of the gas storage chamber to absorb impact, eliminate pulsation and recover energy. When explosive column exploded in the hole, the air columns placed on the bottom and top of the hole absorbs the peak impact energy, and convert into smooth transitions clipping compression energy, to improve energy efficiency, reduce peak vibration. Assuming the dynamite in blast hole is the same nature, density is uniform and continuous distribution, therefore the pulse of initial pressure and the explosion time can be calculated:

$$P_m = \frac{1}{2} \cdot \frac{1}{K+1} \cdot \frac{1}{1 + {}^{l_a}\!/_{l_c}} \rho_o D^2 \tag{1}$$

$$t = \frac{2(K+1)(1 + {}^{l_a}\!/_{l_c})}{\rho_o D^2} I \tag{2}$$

Where P_m = explosion pulse of initial pressure; K = explosion products isentropic coefficient, K = 3; ρ_o = explosive density; D = explosive detonation velocity; T = duration of action; I = blasting impulse; l_a = air column length; and l_c = charge length.

When the air column length is l_a = A, l_a = 0, according to (1) and (2) that:

$$P_m|_{l_a=A} < P_m|_{l_a=0} \tag{3}$$

$$t|_{l_a=A} > t|_{l_o=0} \tag{4}$$

Therefore, when set the air cavity in blast hole, the pulse peak pressure in the hole after the explosion can be reduced. At the same time, the duration of action will be extended. The principle of action in the blasting process is mainly reflected in the following three aspects:

a. Unloading, reducing the initial explosion pres sure.
   In the initial phase of explosive column detonation in the hole, the explosion pressure quickly unloading to the air column, reducing the initial explosion pressure and the detonation shock, significantly reduce the peak vibration of the surrounding rock mass.
b. Accumulator, increasing the blasting impulse, weakening of blasting vibration.
   When blasting, rock crushing follow the impulse conservation, i.e. I = ∫pdt. When explosive column exploded in the hole, part of the energy stored in air column in the hole cavity, and accompanied by the main compression wave direct effect in the rock continuous release, increased the destructive effects time on the rock and the pulsation period, improved blasting effect, weaken the vibration peak.
c. Pressure break rock delay, fracture failure strengthening.
   The peak explosion energy of explosive column in the hold can be converted to compression after absorption of air column (Due to the blasting time is very short, it can be regarded as the adiabatic process), shock compression can be reduced to a certain extent when the medium generate sufficient crack, compression air column continues to release energy, formatting two dynamic rock breaking combination, which are explosion shock and pressure destruction, extended blasting effect time, the fissures created by explosion waves and stress waves are in the further expansion of the splitting effect by the high-pressure gas, and blasting vibration is slowed down.

Reasonable emplacement of air cavity in the hole, not only can achieve the purpose of reduce peak vibration energy and control of flying rocks, but also improve blasting efficiency and reduce the unit consumption of rock explosive. Comprehensive analysis, reduce vibration and improve efficiency is complementary, the basic consideration is focusing and optimizing the explosive energy to the rock breaking power.

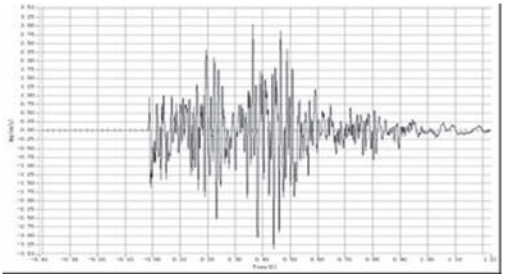

Figure 1. The blasting vibration wave monitored by blasting frequency control technology.

2.2 *Blasting vibration frequency control*

Regulation of blasting vibration frequency can adjust the burden distance, in addition, it is more important to take full advantage of the physical characteristics of group blasting improve the source frequency of blasting area and response frequency of the protected object. Usually the natural frequency of ground protection facilities $f_N \le 6$ Hz, therefore, rock blasting vibration frequency modulated to more than 40 Hz can significantly change the protected object's response of blasting seismic waves. When designing, control delay blasting interval $\Delta t \le 25$ ms, to generate the blasting fundamental frequency $f_0 \ge 40$ Hz. At the same time, as far as possible to ensure sufficient delay segment series and formation of the main vibration fundamental wave by a combination of delay blasting in the blast area.

Another way to control blasting vibration frequency is to optimize the initiation sequence according to the principle of the Doppler Effect, fully use of the relative motion relationship between source and protected object to generate the Doppler shift, to further increase the main blasting vibration fundamental frequency received by the protected object, improve the dynamic response state. Calculated as follows:

$$f = \frac{u}{u - u_s} f_s$$

f is receiving frequency, f_s is explosion source frequency, u is shock wave propagation speed, u_s is explosion source moving velocity.

3 BLASTING PROCESS DESIGN

3.1 *Blast sequence and network design*

Take the most difficult part of the project as an example, the blast sequence and network design of

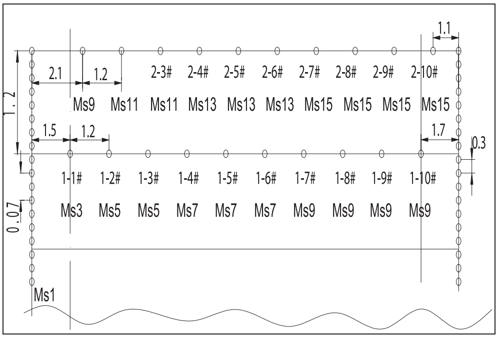

Figure 2. Direct motion blasting network diagram (unit: meter).

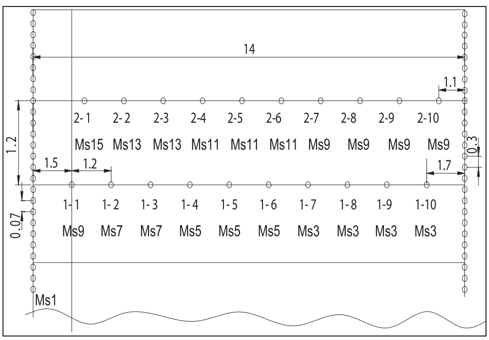

Figure 3. Inverse blasting network diagram (unit: meter).

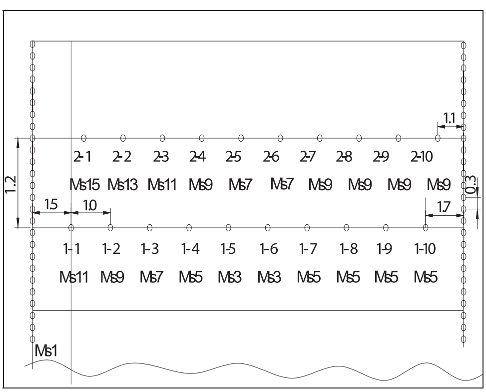

Figure 4. Complex network diagram (unit: meter).

the 14 m width pit at west side of the light railway trough.

3.2 *Low vibration pre-splitting blasting technology*

The use of pre-splitting blasting technology is a common method to passively control the wave propagation. But when blasting area and protected object interval of only a few meters, the waves generated by pre-splitting blasting will become an important factor to control the blasting vibration. Combined with the results of comprehensive analysis, the short delay pre-splitting blasting technology of the intensive hole with an empty hole successfully solves this problem. Description of the pre-splitting blasting design process according to the situation of 2 m distance between protected building basis and excavation pit edge. Selection of pre-splitting blasthole $d = 50$ mm, hole spacing $a = 300$ mm, hole depth $h = 3500$ mm, fill the pre-splitting explosives column in the hole, average charge weight per meter $q = 100$ g/m, delay interval $\Delta t \leq 25$ ms, total delay control in $\Delta T \leq 50$ ms, zoning combining.

3.3 *Process control*

a. Ensure hole distribution meets the design requirements, reserved 200–500 mm air cavity at the hole bottom, can be formed by PVC pipe, bamboo or cardboard tube placed in the hole bottom.
b. According to design requirements, by-hole charge and also retain 300–500 mm air cavity in the hole top, then complete stemming.
c. Pre-splitting blasting hole, the smooth blasting hole can also set an air cavity in order to weaken the impact on bedrock.
d. After completion of interconnection and protection according to design requirements, emplace monitoring device.
e. After initiation, timely processing of monitoring data, analyze and summarize.

4 APPLICATION SUMMARY

The air spring vibration absorber blasting technology can reduce the peak of blasting vibration, reduce blasting noise and weaken the distance of flying stone, it is the preferred process of blasting excavation in complex environments. The project practice shows, the total length of top and bottom air cavities should be controlled within 20 to 30 percent of the total length of explosive column. In which, the bottom air cavity should be 60 to 70 percent of the entire air cavity, compared with a fully coupled continuous charge, blasting efficiency can be increased by 30%, explosives consumption can be reduced by 10 to 20 percent.

Control the initiation sequence and the delay interval can change the dynamic response of the protected object. When the need to increase blasting vibration frequency, the protected object as basis point, blasting sequence should be from far to near, delay interval should be less than 25 ms. Through blasting vibration monitoring and analysis to guide the use of active control source and auxiliary damping, vibration isolation technology is a proven engineering method with a wide range of value.

REFERENCES

Cheng, Kang. & SU, Wei-qian. 2011. Analysis influence of contour boreholes space with air-decked charge on smooth blasting effect. *[C], Blasting*, December, pp. 15–19.

Shi, Fu-qiang. 2011. Blasting vibration frequency control technology research and application [C]. *New Development on Engineering Blasting (The Asian-Pacific Symposium on Blasting Techniques, 2011)*. Xiamen: Metallurgy industry press.

Shi, Fu-qiang & Wang, Ping. 2009. Research and Application of Quantitative Safety Estimate on Blasting Vibration Impact, Engineering Blasting.

Shi, Fu-qiang & Wang, Ping. 2009. Safety Demonstrative Research on Mining Blasting in the Vicinity of Rail Way Trunk Line, *Engineering Blasting*, June, pp. 82–86

Tan, Yuan-jun. & Chen, Ji-fu. 2011. Theory research of powder destroying with cylinder charge in rock under the condition of air interval decoupling charge along diameter. *[C], Blasting*, March, pp. 58–60.

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Linking relationship between parameters of rock mass and ground vibrations

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ABSTRACT: There is an increasing trend towards opencast mining particularly in India where blasting is one of the main operations. Opencast blasting generates ground vibrations and noise which causes inconvenience to the surrounding habitats and cause damage to the buildings and other nearby structures. These ground vibrations are mainly dependent on a number of parameters. Among them physico-mechanical properties of rocks through which the waves travel are important. During the present study, 27 blast events have been recorded in an opencast iron ore mine consequent to 13 blasts. Field investigations were carried out for collection of representative samples of rock mass/ore for determination of physico-mechanical properties of rock mass parameters especially for determination of density, uniaxial compressive strength, uniaxial tensile strength, shear strength, modulus of elasticity and angle of friction and calculation of combined Rock Mass Rating (RMR).

Field investigation reveals that the extent of blast vibrations in three orthogonal directions, i.e., Longitudinal, Vertical and Transverse wave influenced by the litho units of the rock mass through which it propagates. The path through which the waves traveled encounter rock types like Laterite, Manganiferrous clay, Phyllitic clay, Limonitic clay, Dyke, Lumpy ore, Powdery ore, Limonitic ore and Banded Ferruginous Quartzite. USBM equation for predicting the peak particles velocity has been modified based on the recorded data and a new mathematical model for prediction of peak particle velocity has been developed. The co-relation between the blast event parameters and physico-mechanical properties of rock mass including Rock Mass Rating has been discussed.

1 INTRODUCTION

There is an increasing trend towards opencast mining particularly in India where blasting is one of the main operations. Many open pit mines exploiting low grade minerals at comparatively greater depth are presently economically viable mainly due to use of better blasting technology. The explosive energy is not fully utilized for rock breaking. Only a part of the energy is used in doing the useful work, and remaining is spent in undesirable phenomenon such as ground vibration, noise and air blast. Thus, ground vibrations are a matter of concern as they cause inconvenience to the surrounding habitats by way of noise and shock vibration and cause damage to the buildings and other nearby structures (USBM-RI-8507, 1980; USBM-RI-8896, 1984). These ground vibrations are mainly dependent on a number of parameters (Giri etal., 1996; IBM, 1993). Among them physico-mechanical properties of rocks through which the waves travel are important (Thote & Singh, 1996). A study on the impact on propagation of the ground waves of the rock mass through which it passes through will be of much help in taking effective measures to minimise the damage caused to these structures.

Factors influencing ground vibration can be grouped into uncontrollable and controllable factors. Factors like geological condition, structure, lithology and distance and location of structure come under category of uncontrollable factors (Winzer & Ritter, 1980; Sinha, 2000). Parameters like type of explosives used, burden and spacing, geometry of shot, sub-drilling, stemming, delay timing, charge weight per delay, direction of blast initiation can be grouped together as controllable factors (Stagg & Engler, 1980; IS, 1982). The intensity of vibration increases as the quantity of charge detonated increases per delay. The selection of suitable delay interval is extremely important in multi-row blasts (Sinha & Nath, 2005).

2 EXPERIMENTAL WORK

The aim of the investigation is to study the effects of rock mass parameters and rock mass rating on blast vibration. The investigations have been carried out in a large opencast iron ore mine where 13 blasts at different periods of time have been monitored for ground vibrations at different locations. Longitudinal, vertical and transverse peak particle velocities have been recorded along with frequencies giving a total of 27 blast events. The maximum peak particle velocity, charge weight and scaled distance were also recorded. These are given in Table 1. The lithological and structural characteristics of the rock mass through which the vibration waves passed have been recorded. Physico-mechanical properties of rocks in travel path of waves and combined RMR have been obtained from field measurements.

An attempt has been made to study the effect of travel path rock characteristics including rock mass rating on peak particle velocity of different modes of vibration and establish co-relation between some physico-mechanical properties and peak particle velocity, frequencies as well as scaled distance.

Table 1. Blast events recorded during field investigation.

Monitoring station	PPV (mm/s)	Frequency (Hz)	Max^m charge per delay (kg)	Distance (m)	Scaled distance (m/kg$^{1/2}$)
MS1	10.7	9.0	210	120	8.28
MS2	3.68	6.2	210	190	13.11
MS3	3.94	7.0	250	360	22.76
MS4	2.67	4.3	250	180	11.38
MS5	10.0	4.0	280	160	9.56
MS6	4.32	4.7	280	150	8.96
MS7	1.27	8.0	240	460	29.69
MS8	2.03	3.2	240	350	22.59
MS9	5.59	4.0	225	212	14.13
MS10	7.11	3.8	225	170	11.33
MS10A	2.41	3.3	225	380	25.33
MS11	15.2	3.0	434	160	7.68
MS12	46.2	5.2	434	100	4.80
MS13	7.11	4.0	110	190	18.11
MS14	9.4	4.2	110	170	16.20
MS15	9.27	3.0	480	300	13.69
MS16	8.13	3.6	480	190	8.67
MS17	0.905	5.0	263	450	27.74
MS18	1.27	5.0	263	280	17.26
MS19	1.3	5.0	180	325	24.22
MS20	10.3	5.6	180	100	7.45
MS21	6.18	6.0	280	190	11.35
MS22	1.4	6.1	280	460	27.49
MS23	6.54	5.0	40	85	13.43
MS24	1.65	4.1	40	250	39.52
MS25	6.7	4.0	40	85	13.43
MS26	1.27	5.5	40	250	39.52

2.1 *Field investigations*

Blast site is mostly located on Laterite (Footwall and Hanging wall) and Banded Ferruginous Quartzite. Litho unit of blast vibration monitoring station consists of Laterite, Dyke, Manganiferrous clay and phyllitic clay. It is observed in some of the sections that litho units encountered limonitic clay, limoniticore, powdery ore and Banded Ferruginous Quartzite (BFQ) between blast site and monitoring station.

Geological sections were drawn along the longitudinal path of the vibration wave from blast site to monitoring station showing location of bore holes, litho log and RMR values. One of the typical section is shown in Figure 1 and abbreviation used is as under:-

LAT: Laterite, CP: Phyllitic clay, CMn: Manganiferrous clay, CL: Limonitic clay, LO: Limonitic Ore, OP: Powdery ore, B. F. Q.: Banded Ferruginous Quartzite, Dy: Dyke/Altered intrusive, RL: Reduced Level, MS: Monitoring Station, BH: Borehole.

2.1.1 *Monitoring of ground vibration*

Blast induced ground vibrations were monitored at various monitoring stations along different geological sections and blast events recorded in all the three orthogonal directions i.e. transverse, vertical and longitudinal along with respective frequencies.

Blast vibrations were recorded with geophone and microphone sensors attached to the seismographs and calibrated periodically. These instruments are a micro-processor based instrument which receives analyses and displays/prints peak particle velocities, peak sound pressure levels, and frequency summary tables. Each record furnishes

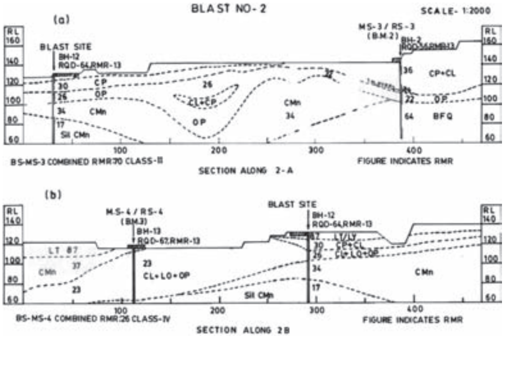

Figure 1. Geological sections showing location of bore holes, litho log and RMR values.

peak values in three orthogonal directions and their vector sum.

2.2 *Laboratory investigation*

During the field investigations representative samples of rock mass/ore consist of laterite, phyllitic clay, Manganiferrous clay, Dyke, Banded ferruginous quartzite, Powdery ore, Limonitic ore and Limonitic clay were collected and tested for density, uniaxial compressive strength, tensile strength, young's modulus and angle of friction as per ISRM standards. These are given in Table 2.

2.3 *Determination of Rock Mass Rating (RMR)*

Geomechanics Classification of Rock Masses for determination of the rock mass rating system has been taken into consideration for determination of rock mass rating of various litho units encountered in geological sections drawn. Rock mass rating has been determined based on physico mechanical property of rock mass parameters such as (1) strength of intact rock material, (2) drill quality RQD (Rock Quality Designation), (3) spacing of discontinuities, (4) condition of discontinuities and (5) ground water. Strength of intact rock material has been determined in laboratory and its results have been given in Table 3. Drill quality (RQD) has been determined with % core recovery. Total 45 boreholes have been taken into consideration for determination of RQD and RMR.

RMR is evaluated as per geo-mechanical classification of rock masses considering litho log of bore-holes. Ranges of RMR is given in Table 3.

Combined RMR(C) values have been calculated with weightage RMR lengthwise by dividing total length in the travel path. Values of combined RMR and blast parameters are given in Table 4.

3 RESULTS & DISCUSSION

Field investigations reveal that the extent of blast vibrations in three orthogonal directions, i.e., Longitudinal, Vertical and Transverse are influenced by the litho units of the rock mass through which they are propagating. The path through which the waves traveled encounter rock types like Laterite, Manganiferrous clay, Phyllitic clay, Limonitic clay, Dyke, Lumpy ore, Powdery ore, Limonitic ore and Banded Ferruginous Quartzite in the present investigations.

USBM equation for predicting the peak particles velocity:

$$V = K\left(\frac{D}{\sqrt{Q}}\right)^{-B} \tag{1}$$

$$K = V\left(\frac{D}{\sqrt{Q}}\right)^{B} \tag{2}$$

Table 2. Rock Characteristics.

Sampling station no.	Rock type	Density (g/cm^3)	U.C.S. (MPa)	T.S. (MPa)	S.S. (MPa)	Y.M. (MPa × 10^4)	Friction angle
RS-1	Laterite (Lumpy)	1.71	7.22	1.03	1.85	1.46	38°
Rs-2	Laterite (Lumpy)	1.71	6.83	1.02	1.54	1.43	39°
Rs-3	Phyllitic Clay	1.60	3.5	0.465	0.92	1.12	14°
Rs-3A	Phyllitic Clay	1.60	3.55	0.426	0.71	1.12	14°
Rs-4	Mangani-ferrous Clay	1.72	7.25	1.25	2.05	1.46	32°
Rs-5	Laterite	1.69	6.03	0.89	1.16	1.37	40°
Rs-6	Laterite	1.69	6.13	0.79	1.36	1.37	41°
Rs-7	Dyke	2.07	56.0	7.35	12.85	3.06	37°
Rs-8	Dyke	2.08	59.5	6.395	19.00	3.13	22°
Rs-9	BFQ	2.09	62.0	7.983	13.08	3.17	38°
Rs-10	Powdery ore	1.98	0.05	0.0006	0.001	0.242	32°
Rs-11	Limonitic Ore	1.33	0.45	0.056	0.09	5.35	23°
Rs-12	Limonitic Clay	1.02	0.027	0.0033	0.005	0.194	22°
RS-13	Phyllitic Clay	1.61	3.58	0.388	0.66	1.16	18°
RS-14	Mangani-ferrous Clay	1.73	8.05	1.35	2.15	1.52	35°
RS-15	Mangani-ferrous Clay	1.72	7.75	1.30	2.10	1.48	34°

Note: U.C.S- Uniaxial Compressive Strength (MPa); T.S.-Tensile strength (MPa); S.S-Shear Strength (MPa); Y.M.: Young's Modulus (MPa × 10^4).

Table 3. Range of rock mass rating.

Rock types	Strength of intact rock material UCS strength (MPa)	RQD	Spacing of discontinuities	Condition of discontinuities	Ground water	Total Rock mass rating (RMR) range
Laterite	2	13–20	20	25–30	7–10	70–87
Phyllitic clay	1	8–20	5	0	4–7	16–36
Limonitic clay	0	8–20	5	0	4–10	21–33
Limonitic ore	0	8–20	5	0	4–10	23–32
Powdery ore	0	3–20	5	0	4–10	18–32
Magani-ferrous clay	2	3–20	5	0	4–10	20–37
Sil. Mangani-ferrous clay	2	0–3	5	0	7–10	17–20
Dyke	7	0–20	5	0	4	36–39
Hard BFQ	7	3–20	20	25–30	4–7	64–87

Table 4. Values of combined RMR and Blast parameters.

Blast site	Monitoring Station (MS)	Moni-toring section	Blast site to MS distance (m)	Com-bined RMR	Maximum charge per delay (Kg)	Peak particle velocity (mm/s)	Type of wave component	Remarks
Blast No.1	MS-1	1-A	120	75	210	10.70	Longitudinal	
	MS-2	1-B	190	40	210	3.68	Vertical	
Blast No.2	MS-3	2A	360	70	250	3.94	Longitudinal	
	MS-4	2B	180	26	250	2.67	Vertical	
Blast No.3	MS-5	3A	160	81	280	10.00	Vertical	160 m void area
	MS-6	3B	150	33	280	4.32	Longitudinal	
Blast No.4	MS-7	4A	460	29	240	1.27	Vertical	
	MS-8	4B	350	36	240	2.03	Longitudinal	350 m void area
Blast No.5	MS-9	5A	212	65	225	5.59	Vertical	
	MS-10	5B	170	67	225	7.11	Vertical	
	MS-10A	5B	380	48	225	2.41	Vertical	1700 m void area
Blast No.6	MS-11	6A	160	87	434	15.2	Longitudinal	
	MS-12	6B	100	87	434	46.2	Vertical	
Blast No.7	MS-13	7A	190	87	110	7.11	Vertical	
	MS-14	7B	170	87	110	9.4	Vertical	
Blast No.8	MS-15	8A	300	87	480	9.27	Transverse	200 m void area
	MS-16	8B	190	60	480	8.13	Vertical	190 m void area
Blast No.9	MS-17	9	460	20	263	0.905	Transverse	460 m void area
	MS-18	9	280	18	263	1.27	Transverse	280 m void area
Blast No.10	MS-19	10	325	25	180	1.3	Longitudinal	325 m void area
	MS-20	10	100	66	180	10.3	Vertical	100 m void area
Blast No.11	MS-21	11	190	58	280	6.18	Transverse	
	MS-22	11	430	28	280	1.4	Vertical	260 m void area
Blast No.12	MS-23	12/13	85	72	40	6.54	Long/Long	80/85 m void area
	MS-24	12/13	250	49	40	1.65	Long/Trans	240/245 m void area
Blast No.13	MS-25	12/13	95	82	40	6.7	Long/Long	80/85 m void area
	MS-26	12/13	245	37	40	1.27	Long/Trans	240/245 m void area

This equation has been further modified by replacing K with RMR(C).

$$C = V\left(\frac{D}{\sqrt{Q}}\right)^{B} \tag{3}$$

where V = Peak particle velocity (PPV), mm/sec; D = distance between the point of blasting and the measuring station, m; Q = maximum charge weight per delay, kg; C = Combined rock mass rating.

From the above equations, it is evident that the left side is function of Rock Mass Rating and right side is function of blast parameter. Site constants **B** is evaluated with Equation (3) for each 27 blast events parameters and combined Rock Mass Rating of the travel path. Arithmetic mean value of B (27 events) determined as 0.92322758, and the equation is subsequently modified.

$$C = V\left(\frac{D}{\sqrt{Q}}\right)^{0.92} \tag{4}$$

95 percentile value of B is 1.17.

Thus, Co-relation between RMR and blast vibration parameters has been established.

It is observed that good co-relation exists between the blast parameter (x) and Rock Mass Rating (y). The various relationships obtained are as follows in different mathematical forms:

(i) In the form of Power
$y = 1.9888x^{0.8128}$
$R^2 = 0.9308$

(ii) In the form of Exponential
$y = 5.2233e^{0.0104}x$
$R^2 = 0.6584$

(iii) In the form of Logarithm
$y = 39.1\ell n(x)^{-99.311}$
$R^2 = 0.9403$

(iv) In the form of Polynomial
$y = -0.0049x^2 + 1.4296x - 7.5567$
$R^2 = 0.9898$

It is observed from the results of ground vibration monitoring that longitudinal waves component predominates if it travels through elastic medium comprising of Class I & II types of rock like Laterite and Banded Ferruginous Quartzite. It is noticed that if longitudinal lines between blast site and monitoring station pass through excavated portion (void area) of the quarry, the longitudinal waves diminish. It is also observed that ppv of longitudinal wave diminishes while traveling through Class IV & V types of rocks heterogeneous like Manganiferrous clay, Phyllitic Clay, Limonitic clay and Dyke.

Vertical wave component predominates if it travels through inelastic medium comprising of Class IV & V types of rocks like Manganiferrous clay, Phyllitic Clay, Limonitic clay and Dyke.

Transverse wave is predominant if it travels through different inelastic media comprising of Class IV & V types of rocks like Manganiferrous clay, Phyllitic Clay, Limonitic clay and Dyke and existence of Class IV & V types of rock below blast site and monitoring station. It is noticed that if longitudinal lines between blast site and monitoring station pass through excavated portion (void area) of the quarry and waves travels through elastic media comprising of Class I & II types of rock like Laterite & BFQ, the Transverse waves are predominant over the Longitudinal waves, provided monitoring station is located on free face of the quarry.

It is also observed that Vertical wave vibration was absorbed to some extent in higher compressive strength rock (class I & II types) like Banded Ferruginous Quartzite (B.F.Q) and diminished in intensity while traveling through the rock media.

The results of physic-mechanical properties of rock mass and blast events showed good co-relation exists between the compressive strength of rock mass and frequency of blast events for class I & II types of rock like Laterite while no co-relation exists in case of class IV & V types of rock.

Some co-relation exists between the compressive strength of rock mass and peak particle velocity of blast events for class I & II types of rock like Laterite as stated below however, no co-relation exists in case of class IV & V types rock.

A better co-relation exists between the compressive strength of rock mass and Scaled Distance of blast events for types of rock like Laterite.

It is observed that for nearly class I & II types of rock like Laterite, a good Co-relation exists between the compressive strength of Rock Mass and frequency of Blast events recorded at monitoring stations as R^2 is varying from 0.92 to 0.97. The different relationships and co-relation coefficients for Laterite in the different mathematical form are:

(i) In the form of Power
$y = 4.3192x^{0.2381}$
$R^2 = 0.9552$

(ii) In the form of Exponential
$y = 5.2233e^{0.0375}x$
$R^2 = 0.9213$

(iii) In the form of logarithm
$y = 1.5721\ell n(x) + 3.8188$
$R^2 = 0.9599$

(iv) In the form of polynomial
$y = -0.0409x^2 + 0.7863x + 3.4645$
$R^2 = 0.9737$

Where Y is compressive strength (MPa); and x is frequency of blast events (Hz).

It is observed that in class IV & V types of rocks, no co-relation between compressive strength of rock mass and Scaled Distance of blast events exists as R^2 is varying between 0.035 to 0.49.

It is observed that there is no co-relation exists between shear strength and peak particle velocity. However, for Phyllitic clay R^2 is varying between 0.60 to 0.788. It is observed that there is no co-relation between shear strength and frequency of all types of rock except in case of Laterite where R^2 is varying between 0.94 to 0.98. The different relationship and co-relation coefficients for Laterite in diffrent mathematical form are:

(i) In the form of Power
$y = 0.5604x^{0.5491}$
$R^2 = 0.9767$
(ii) In the form Exponential
$y = 0.8687e^{0.0864}x$
$R^2 = 0.9419$
(iii) In the form of logarithm
$y = 0.08207\ell n(x) + 0.0503$
$R^2 = 0.9906$
(iv) In the form of polynomial
$y = -0.0137x^2 + 0.3107x + 0.1609$
$R^2 = 0.9893$

Where Y is shear strength (MPa); and x is frequency of blast events (Hz).

It is observed that no co-relation exists between shear strength and Scaled Distance. However, in case of Manganiferrous clay R^2 is varying between 0.51 and 0.64. It is observed that there is no co-relation between Young's Modulus and peak particle velocity as R^2 is varying between 0.51 and 0.64. It is observed that there is a good co-relation between Young's Modulus and frequency in case of Laterite as R^2 is varying between 0.92 and 0.96. It is also observed in case of Phyllitic clay that R^2 is varying between 0.96 and 0.98. The different relationship and co-relation coefficients for Laterite in different mathematical forms are:

(i) In the form of Power
$y = 1.2115x^{0.0861}$
$R^2 = 0.9553$
(ii) In the form of Exponential
$y = 1.2977e^{0.0135}x$
$R^2 = 0.9213$
(iii) In the form of logarithm
$y = 0.1215\ell n(x) + 1.1962$
$R^2 = 0.957$
(iv) In the form of polynomial
$y = -0.0033x^2 + 0.0632x + 1.1618$
$R^2 = 0.9734$

Where Y is Young's Modulus (MPa10^4); and x is frequency of blast events (Hz).

For Phyllitic clay:

In the form of Power
$y = 1.209x^{-0.0366}$
$R^2 = 0.9818$
In the form of Exponential
$y = 1.1846e^{-0.0071}x$
$R^2 = 0.961$
In the form of Logarithm
$y = -0.0418\ell n(x) + 1.2072$
$R^2 = 0.9818$
In the form of Polynomial
$y = -0.002x^2 + 0.0298x + 1.2339$
$R^2 = 1$

Y is Young's Modulus (MPa104) and x is frequency of blast events (Hz).

It is observed that there is no co-relation between Young's Modulus and Scaled Distance. However in case of Manganiferrous clay R^2 is varying 0.737 to 0.845.

4 CONCLUSIONS & RECOMMENDATIONS

The relationship between blast parameters and Rock Mass Rating has been established and USBM equation for prediction of peak particle velocity has been modified for Iron ore formation:

$$V = C\left(\frac{D}{\sqrt{Q}}\right)^{-0.92}$$

$$C = V\left(\frac{D}{\sqrt{Q}}\right)^{-0.92}$$

The results of physic-mechanical properties of rock mass and blast events showed good co-relation exists between the compressive strength of rock mass and frequency of blast events for nearly class I & II types of rock like Laterite while no co-relation exists in case of class IV & V types of rocks.

A better co-relation exists between the compressive strength of rock mass and Scaled Distance of blast events for class I & II types of rock.

The present investigation is limited to study of rock mass associated with iron ore formations. It is suggested that further investigation in other minerals and associated rocks may be taken up to substantiate the results of the study. More number of blasts should be monitored at different litho units.

Dynamic properties of the rock mass should be determined in the laboratory to see their influence on wave propagation and vibration of rocks.

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REFERENCES

Giri, A.K., Singh, T.N. and Singh, D.P. 1996. *Evaluation of Rock Fragmentation due to Blasting, Proc. Nat. Symp.* on Drilling and Blasting, Sept. 21–22, Bhubaneswar, pp. 200–202.

IBM. 1993. Ground vibrations due to blasting. *Information Circular, No. IBM-IC-01*, Nagpur, India.

Indian standard. 1982. Criteria for safety and design for structures subject to Underground blasts, *Indian Standards Institution, New Delhi*. pp. 1–8.

R.I. 8507. 1980. United States Bureau of Mines.

R.I. 8896. 1984. United States Bureau of Mines.

Sinha, B.P. 2000. Monitoring of ground vibrations data for regression analysis for estimation of safe charge. *Mining Engg. Jr. Vol. 1*, pp. 26–31.

Sinha, B.P. & Nath, R. 2005. "Effect of Rockmass Parameters on Ground Vibrations due to Blasting." *International Symposium on Advances in Mining Technology and Management*, pp. 265–271.

Stagg, M.S. and Engler, A.J. 1980. Measurement of Blast Induced Ground Vibrations and Seismograph Calibrations. US Bureau of Mines, Report of Investigations 8506.

Thote, N.R. and Singh, D.P. 1996. Controlled Blasting for better Ground Conditions, *Proc. Nat. Conf. on Ground Control in Mining,* Jan. 19–20, Varanasi, pp. 383–391.

Winzer, S.R. & Ritter, A.P. 1980. The role of stress waves and discontinuities in rock fragmentation: a study of fragmentation in large limestone block. *Proceedings 21st US Symp. Rock Mech.*, Rolla, pp. 362–370.

Blasting in Mines – New Trends – Ghose & Joshi (Eds)

Performance enhancement by adopting improved blasting techniques in a limestone mine: A case study

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ABSTRACT: At our Nimbeti Limestone Mines, we were facing some challenges like, Toe problem, Fly Rocks, Blast Induced Vibrations and Poor Fragmentation etc. To overcome with these problems & to improve cost efficiency we have done number of experiments in Blasting practices like use of True Bottom Initiation, Bottom Air Decking, Use of Polymer Beads with ANFO & Bulk Mixing Delivery Van. By adopting these practices the Toe problem is almost eliminated, achieved better fragmentation results, blasting cost per ton reduces significantly as well as increases overall safety.

1 INTRODUCTION

Nimbeti Limestone Mines of M/s Shree Cement Ltd. is a highly mechanized Limestone mines having 15 million tones Rock handling per annum consuming 2200 Tonnes of Explosive per annum. The Mine is situated near Village Ras, in Pali District, Rajasthan, India & have Mining lease area of 750 hectares. Limestone is metamorphosed in nature and highly fractured & structurally folded. The beds are having dip varying from 45° to 60°. The pegmatite is intruded in the weak zones both across & along the dip/strike. The Mine is presently feeding seven of M/S Shree Cement Ltd Cement Plants situated at Ras & Beawar.

To overcome the problem of Toe formation & poor fragmentation as well as fly rock & vibration due to blasting in highly fractured/jointed & folded strata of intricate nature limestone deposit we have adopted some of the latest techniques successfully which are as under:

2 BULK MIXING & DELIVERY VEHICLE (BMD)

It consists of a heavy truck chassis; and a system of tanks/bins, pumps, motors, hydraulic transmission network, electronic/mechanical controlled devices, safety devices and delivery equipments mounted on the chassis. It carries ingredients in its bins and tanks to blast sites and mixes them in the desired proportion. It is a need of today and tomorrow in Indian Mining operations. We {Shree Cement Ltd. (SCL)} are the first explosive consumer company in India to get the license for BMD Vehicle (Fig. 1). The main basic requirement for operating Bulk -ANFO system are as follows:

- NOC from DGMS (Director General of Mines Safety) for use of ANFO in mine.
- Licensed BMD Vehicle from CCE (Chief Controller of Explosives)
- Qualified blasters & system operators.

2.1 *Salient features of bulk ANFO system*

There are only two raw materials for preparing ANFO i.e. Low density prilled AN as oxidizer & Diesel oil as fuel (FO). The BMD system possesses two main storage bins/tanks, for the two main raw materials for preparing ANFO i.e. a bin for Prill AN & a separate tank for fuel. The salient features of BMD used at SCL are:

Figure 1. View of a BMD system.

- Licensed Capacity, Granted by Explosive Department: Max 24000 T/Yr.
- Storage Capacity of Ammonium Nitrate: 6 T
- Storage Capacity of Fuel oil: 560 Ltr
- Mechanical mixing is safe.
- Fuel Oil mixed optimally as small quantities are mixed at any instance.
- Mixing is very fast, 100–200 kg/min.
- Uniform mixing of ANFO.
- Optimum explosive energy of ANFO is achieved.
- Delivery is mechanized and hence fast.
- Bigger blast size which was earlier getting restricted for various reasons.
- Manual mixing is totally eliminated & in turn highly safe.

2.2 *Advantages of using BMD vehicle*

- Mechanical mixing of ANFO ensures perfect mixing & delivery.
- The actual ANFO mixing is done just before loading in holes, ensures better safety, & productivity.
- As only small quantity of AN & FO is mixed at any given time (in the system), offers best possible FO distribution, best absorption & retention.
- This ensures better oxygen-balance, & results to optimum blasting energy. This also helps to eliminate generation of brown fumes. All in all, optimizes best fume characteristics.
- Faster mixing, delivery & down-the-hole loading is possible with least complicated chemical composition & reactions. This save time, money as well as ensures relatively high safety.
- Involvement of manpower in the actual mixing & delivery process is almost nil (no direct contact is required), which makes the practice chemically very safe.
- It can load at a rate of 100–150 kg/min and is very mobile to serve multi-location
- It handles explosive ingredients and is very safe in operation.
- It support large scale blasting and mining operations
- Bigger Size of Blasts is possible.
- Cost effective

Table 1 clearly represents the result obtained using BMD mixing in comparison to Manual mixing.

Table 1. Manual Mixing Versus BMD Mixing.

S. no.	Parameters	Manual mixing	BMD mixing
1	Time taken (per ton of mixing)		
1(a)	Loading into truck	10 minute	10 minute
1(b)	Transportation up to ANFO mixing shed	10 minute	Not required
1(c)	Unloading at ANFO mixing shed	10 minute	Not required
1(d)	AN & FO mixing	40 minute	8–10 minute
1(e)	Loading into bags & truck	15 minute	Simultaneously with mixing
1(f)	Transportation up to blasting site	10 minute from ANFO mixing shed	10 minute from AN storage room
1(g)	Unloading at blasting site & distribution at holes	20 minute	10 minute
	Total time taken in mixing process	1 hour 55 minute	40 minute
2	Man power required (per ton)	2.4 persons (2500 Kg mixing by 6 person)	1 person (6000 Kg mixing by 6 person)
3	Mixing cost (Man power/ton), Rs.	600	250
4	Licensed capacity	2.5 tonne/day	6 to 8 tonne/day

3 USE OF POLYMER BEADS WITH ANFO

Poly-Sterene (Polymer) beads are mixed with ANFO (10%–12%, by volume), as column explosive in a hole. The polymer beads in ANFO mix acts as fillers and reduces the Density of Explosive, thereby reducing excessive quantity of ANFO, for a particular Column & Type of Rock to be blasted (Jimeno et al. 1995). Density of Polymer Beads is 0.0165 gm/cc where as Density of AN is 0.791 gm/cc. Figure 2 depicts the actual site photograph of Ammonium Nitrate mixed with Polymer Beads.

The main advantages of the mixing of polymer beads with ANFO are mentioned below. Table 2 clearly shows the result obtaining using Polymer Beads with ANFO in comparison to ANFO only.

- Uniform distribution of Explosive energy throughout the explosive column.
- Less explosive consumption & less ground vibrations & noise.
- Better fragmentation.
- Powder factor increased.
- Reduction in Blasting cost per Ton.

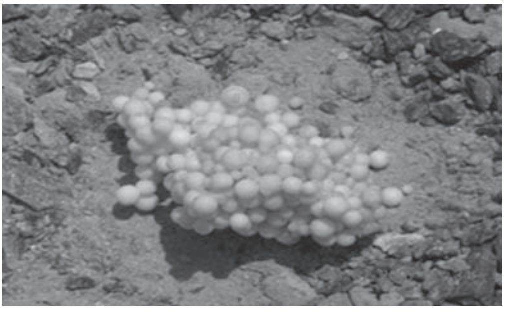

Figure 2. View of the Ammonium nitrate mixed with Polymer Beads at mine site.

Table 2. Polymer Beads with ANFO versus ANFO only.

S. no.	Process	ANFO	ANFO with Polymer Beads
1	Hole dia., mm	165	165
2	Hole depth, m	12	12
3	Burden, m × Spacing, m	4.5 × 6.5	4.5 × 6.5
4	Tonnage of rock broken per hole, tonne	870	870 T
5	Powder factor, tonne/kg	5.85	6.45
6	Stemming, m	3.0	3.0
7	Sub-grade drilling, m	Nil	Nil
8	Total charge per hole, kg	150	136
9	Cost of explosive (ANFO), Rs.	3291/-	2974/-
10	Cost of explosive (primer), Rs.	1100/-	1000/-
11	Cost of Polymer beads, Rs.	Nil	32/-
12	Total cost of explosive per hole, Rs.	4391/-	4006/-
13	Cost of explosive per tonne, Rs.	5.04/-	4.60/-
14	Saving per tonne, Rs.	0.44/-	

4 BOTTOM AIR DECK

The fractured & dipping strata (45° to 60°) is very much prone to toe problems as the explosive energy releases through fractures/joint planes. To combat this problem Bottom air decking is adopted.

When an explosive detonates in a borehole, the high temperature by-products of the detonation will always take the path of least resistance. The bottom hole air deck will first be subjected to an intense shock wave traveling through it. When the initial shock wave front hits the bottom of the hole, the shock wave speed decreases, reflects from the hole bottom and increases the pressure at that point. At this instant of time, a separate secondary impact from the explosion products adds another impulse to the bottom of the hole. The combined effect is that the resulting pressure at the hole bottom can be increased from 2–7 times relative to initial pressure. The increased point source pressure is sufficient to create a planar split and fragmentation at the hole bottom. In essence, the sum of the primary shock wave energy and secondary explosion products are much more efficient than a concentrated continuous cylindrical charge at the hole bottom, but only when the bottom hole air deck length is properly designed for the given field conditions and explosives system (Choudhary et. al, 2008). Figure 3 represents bottom air deck technique used in the mine and Figure 4 depicts the charging Pattern with and without botttom air deck.

The main outcome resulted by using bottom air deck techniques has eliminated (almost) Toe problem and it is presented in Figure 5. The bottom Air Deck of 0.5 m to 1.0 m also helps in eliminating toe as well as undulating floor problems. It also reduced the explosive consumption with improved fragmentation which ultimately reduced the blasting cost per tonne and it is presented in Table 3.

4.1 *True bottom initiation*

The Shock Tube Detonators provides True Bottom initiation[3]. When detonation takes place at the bot-

Figure 3. View of the bottom air deck technique.

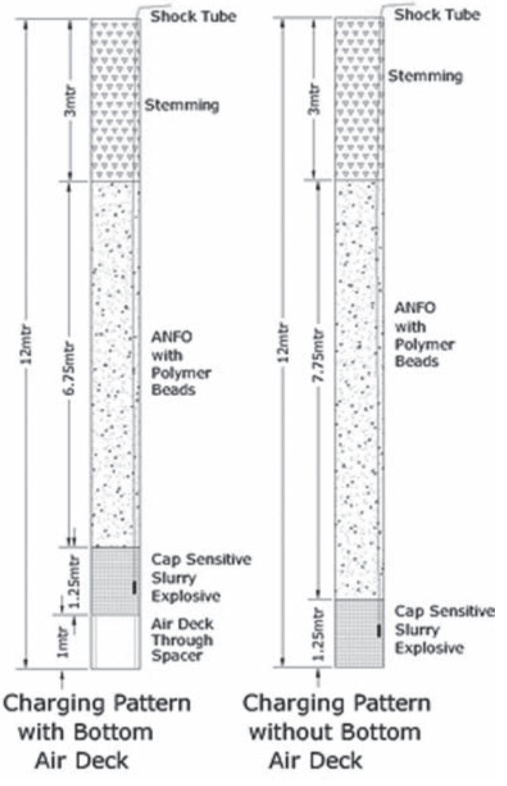

Figure 4. Schematic diagram showing Charging Pattern with & without bottom air deck.

Figure 5. Toe elimination after using bottom air deck technique.

Table 3. Bottom air deck v/s without bottom air deck.

S. no.	Process	Bottom air deck	Without bottom air deck
1	Hole dia., mm	165	165
2	Hole depth, m	12	12
3	Burden, m × Spacing, m	4.5 × 6.5	4.5 × 6.5
4	Tonnage of rock broken per hole, tonne	870	870
5	Powder factor, tonne/kg	6.55	5.85
6	Stemming, m	3.0	3.0
7	Sub-grade drilling, m	Nil	Nil
8	Total charge per hole, kg	134	150
9	Cost of explosive (ANFO), Rs.	2952/-	3291/-
10	Cost of explosive (primer), Rs.	965/-	1100/-
11	Total cost of explosive per hole, Rs.	3917/-	4391/-
12	Cost of explosive per tonne, Rs.	4.50/-	5.04/-
13	Saving per tonne, Rs.	0.53/-	

tom, the rock between the blast hole and free vertical face get displaced horizontally. This also creates the space for the displacement of balance rock and thus flying of rock can completely avoided. The blasting with bottom initiation and free face is just like cutting of cake except displacement starts from bottom towards the ground surface. The reduction of noise & fly rock is directly reflected during blasting, as there was no complaint from nearby villagers & land owners at the foot hill of mines & there was no requirement of paying any compensation to them, while neighbouring mines has to give compensation to villagers during the same period.

4.2 *Summary of improvement achieved by adoption of the above techniques*

The Table 4 clearly indicates the improvement in Powder Factor achieved, as well as cost saving. The saving illustrated is a combined effect of all above techniques applied time to time and based upon actual costing data.

Table 4. Summary of savings achieved and improvement in powder factor using above techniques.

S. no	Particulars	Blasted tonnage (Lac MT)	Kelvex 800 explosive (Lac Kg)	Ammonium nitrate (Lac Kg)	Diesel (Lac Litre)	Powder factor (tonne/Kg)
A	Explosive consumption before implementation of new technology (08–09)	96.5	3.11	12.12	0.93	6.04
B	Explosive consumption during experimenting period of new techniques (09–10)	145.1	4.12	16.25	1.26	6.78
C	Explosive consumption after fully implementing new Techniques (11–12)	152.3	4.42	16.55	1.28	6.92
D	*Applicable rates in Rs/Kg or Rs/Ltr	–	33.25	21.3	33.65	–
E	Calculated saving of explosive cost w.r.t. Rock handling in period (B) based on PF of previous period (A) in Lac Rs	64.35/- Lac Rs				
F	Calculated saving of explosive cost w.r.t. Rock handling in period (C) based on PF of previous period (A)	78.98/- Lac Rs				

* Note: To neutralize the effect of Price hike the rate of Slurry explosive & ANFO have considered constant.

5 CONCLUSIONS

By adopting these practices the Toe problem is almost eliminated, achieved better fragmentation, blasting cost per ton reduces of significantly as well as the overall safety increased significantly.

A total saving of Rs. 64.35 Lac was achieved during the trial & experimenting of one by one techniques & their various combination. A saving of Rs.78.98 Lac has been achieved after regularly use of these techniques in combination.

REFERENCES

Bhandari, S. 1997. Engineering Rock Blasting Operation. *Published by A.A. Balkema.*.

Choudhary, P.B., Raina, A.K., Ramlu, M. & Kumar, P. 2008. Optimisation studies & investigation into Ground vibration, Air over Pressure & Fly Rock for recommended Productive & safe Blast Design at Nimbeti Limestone Mine of M/s Shree Cement Ltd. *Report submitted by CMRI Regional centre, Nagpur in January 2008.*

Jimeno, C.L., Jimeno, E.L. & F.J.A. Carcedo, 1995. Drilling and Blasting of Rocks. *Published by Rotterdam: Balkema.*

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Precision delay timing—a tool for improving mining efficiency: A case study

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ABSTRACT: Mining of minerals involves breaking the rocks/minerals to the desired sizes and removing them in an efficient and safe way. Overall mining cost directly depends on fragmentation which in turn depends largely on right delay time and their precision, assuming all other parameters like blast parameters, explosive properties are held constant. Mining involves rock breaking, excavation, transportation and crushing. The efficiency of each activity depends on fragmentation. Rock breaking being the first activity, the industry takes a lot of care. However, due to the lack of flexibility in the choice of delay time while using factory made delays, the blasting engineers' best efforts may not always result in good fragmentation. In this context, we carried out nine blasts using Precision delays (Electronic Detonator—e-Det-ft) manufactured by Gulf Oil Corporation Limited and found the results highly encouraging. This case study is about precision detonators (electronics detonators) and their role in improving mining efficiency.

Trials with a total of nine blasts were carried out at a limestone mine. Initially few blasts were carried out on the prevailing drill and charging patterns while the sole factor that varied was delay time. After seeing the results like fragmentation, throw and excavator efficiency, the drill patterns were increased gradually, initially spacing by 12%, latter burden by 16%, and finally increasing both spacing and burden each by 12 & 16% respectively, keeping charge per hole and the type of explosive constant. When compared to blasting practices using normal factory made non electric delays, the blast results like fragmentation, throw, excavator efficiency and hauling efficiency, with electronic detonators—e-Det-ft were found highly encouraging. The excavator and hauling efficiency increased by over 10%. The excavator output per hour increased from 500 TPH to + 550 TPH. The additional costs incurred on electronic detonators about Rs 650/hole were fully recovered.

1 INTRODUCTION

Kotputli Cement Works, Ultratech is located on the Delhi—Rajastan National Highway, about 150 km from Delhi. The project is designed to produce 6 million tonnes per annum ROM, from two shift operations, using Hydraulic drill of 110 mm (ROC-L6, Make: Atlas Copco), front end shovels of 6.5 cum bucket capacity (Komatsu) and 55 MT capacity dumpers (Komatsu). The limestone deposit available is a combination of plain and hill deposit. The excavator efficiency was around 500 tons/hour and drilling efficiency was 25 m/hour. The daily output from two shift working was ranging 16000–18000 MT.

2 PREVAILING BLAST PRACTICES

The mine is being operated in five benches of 10 m bench height. The rock is having compressive strength ranging from 100–140 MPa. Rock type is massive at places instead of stratified. The blast design in practice 3 m × 4 m × 10 m, was suggested by CFMRI after extensive trials. The time taken to drill a hole of 10 m depth is around 24 minutes, indicating rock toughness. The drill holes are marked for drill pattern accuracy. The mine is carrying out blasting mostly with ANFO explosives using non-electric initiating devices. The charge column length used to range from 6.5–7.0 m using charge of 52–58 kg/delay, and stemming column length varied from 3–3.5 m.

From the mine boundary, sporadic habitation exists within 300 m. The Cement manufacturing plant is located at around 500 m, and the residential colony is located around 1000 m. Mines office is located around 250 m aerial distance from nearest operating bench. The project authorities like to ensure full control of blasting at all times.

The objectives of blasts included:

- Control of fly rock, ground vibrations and noise without foregoing much on the fragmentation front.
- Increasing mining efficiency.

3 METHODOLOGY

- A presentation is made for project team regarding features, usage and likely benefits of precision detonators in blasting.
- Drill pattern planning is done and drilling is carried as per plan.
- Each electronic detonator is checked before usage and used for priming booster charge.
- Pre-decided charge quantity is used in each hole.
- Holes are stemmed adequately. After all holes are stemmed, the point of initiation is decided, based on habitation to be protected from vibrations and desired throw direction. From the point of initiation, holes are programmed for the holes in same row. Second row holes programming was started from the hole immediately behind point of initiation. Similarly for the third row holes.
- While programming each hole, the hole number and delay time given are marked on the blank tag provided for the purpose.
- Delay time is programmed for each hole considering rock type, charge length, burden, spacing, desired muck profile. It was varied from 4.5–7.25 milli- second per meter for the holes in same row (spacing), and 10.85-15.30 milli-second per meter between rows (burden).
- All the holes are connected after delay time programming is complete.
- The connectivity of the circuit is tested finally, checking the holewise delay time and connectivity.
- After the circuit is found ok, the one end of polarizer is connected to circuit with a lead wire, and the other end to blasting cable. The polarizer can be connected at any point.
- Blasting cable was drawn to a safe place where it connects to e-exploder, before blasting.
- After withdrawing men, machinery to safe distances, posting guards at key locations, placing a seismograph near the habitation, a signal is conveyed for connecting the blasting cable to the e-exploder.
- E-exploder is powered and blast is fired ultimately. Blast is visually observed from safe distance jointly by IDL & Project team.
- Blast site is visited for post-observations before an excavator is re-deployed.
- A blast report is made.
- After using all detonators, a consolidated report is made jointly, and a presentation is made to the project team.

4 BLAST OBSERVATIONS

In all, nine blasts were conducted and the observations are given below:

- Fragmentation was found to be within the acceptable range by project authorities. It ranged from 250 mm–400 mm size.
- Excavator efficiency was improved from 500 TPH to Plus 550 TPH. Hauling efficiency has also gone from 65 Trips to Plus 75 Trips during the shift
- Back break was found in the range of 0.5–1.0 m against the normal range of 1.5–2.0 m.
- There was no fly rock, beyond the blasting block limits. It was around 10 m.
- The peak particle velocity measured near habitation was found to be less than 1.0 mm/s.
- The throw towards frontal portion ranged from 15–30 m.
- In each blast, the depth of cut in the last row ranged from 3–5 m.
- Excavator operators, along with other project team, expressed satisfaction and happiness with the blasts.
- There were no misfires or any damage to men or machinery during the period.
- The additional cost spent on precision detonators got fully recovered in drill and blast cost alone.
- The blast economics and details are given separately in the Annexure.

5 CONCLUSIONS

Precision delay time using electronic detonators having field programming capability features are found to have immense advantages in improving mining efficiency and blast control. For establishing right delay time in a particular bench, a few blasts are required to be carried out in the given bench keeping blast objectives and the constraints in view. A mine can establish right delay time in all the benches over a month time effectively. The additional cost incurred on electronic detonators usage can be fully recovered in limestone mines if the excavator efficiency can improve even by 10–12%.

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REFERENCES

Designing surface blasting rounds—DMVP-Slovenia.

Case studies Published on Electronic delays usage- Detnet Solutions.

Hustrulid, W. "*Blasting Principles for Open Pit Mining*".

Initiator firing time and their relationship to blasting performance -Winzer, S R, Furth, W & A Ritter. 1079. 20th US Symposium on Rock mechanics, 4–6 June 1979, Austin, Texas.

ANNEXURE

Blast Economics

Drilling costs for a hole depth of 10 m with different patterns tried				
Drilling cost	Rs 120/m	10 m deep holes	Rs 1200 per hole	
	Existing	Improved -1	Improved-2	Improved-3
Drill Pattern, m	3 × 4	3.0 × 4.5	3.5 × 4	3.5 × 4.5
Tonnage/hole @ 2.5 density	300	337.5	350	393.75
Cost/cum-Rs	4.00	3.56	3.43	3.03
	Maximum savings envisaged—0.97/cum, ie 25%			

Initiating device costs for a hole depth of 10 m with different patterns tried				
Non-electric	Rs 12/m	16 m/hole	Rs 192/hole	Ex-works
e-det—ft	Rs 850/No	20 m/hole	Rs 850/hole	Ex-works
	Existing	Improved -1	Improved-2	Improved-3
Drill Pattern, m	3 × 4	3.0 × 4.5	3.5 × 4	3.5 × 4.5
Tonnage/hole @ 2.5 density	300	337.5	350	393.75
Cost/cum-Rs	0.64	2.52	2.43	2.16
Additional cost	Minumum—Rs 1.52/cum & Maximum—1.88/cum			

Explosive costs for a hole depth of 10 m with different patterns tried				
ANFO	Rs 25.6/kg	57 kg/Hole	Rs 1459/hole	
BOOSTER	Rs 175/kg	1000 gm/hole	Rs 175	
	Existing	Improved -1	Improved-2	Improved-3
Drill Pattern, m	3 × 4	3.0 × 4.5	3.5 × 4	3.5 × 4.5
Tonnage/hole @ 2.5 density	300	337.5	350	393.75
Max charge/hole-kgs	57	57	57	57
Tons/kg-	5.27	5.93	6.14	6.90
Cost per Cum Rs	5.45	4.85	4.68	4.15
	Maximum savings envisaged—Rs 1.30/cum—24%			

Summary of Savings in consumable quantity thereby reduced handling—improved convenience

	The ROM for the mine is considered as 60 Lakh tons per annum			
	Existing	Improved-1	Improved-2	Improved-3
Drill Pattern, m	3 × 4	3.0 × 4.5	3.5 × 4	3.5 × 4.5
Tonnage/hole	300	337.5	350	393.75
No of holes-Lak holes	0.30	0.27	0.26	0.23
Tonnage/m	30	33.75	35	39.37
Tonnage/kg	5	5.63	5.84	6.57
Drilling-Lakh m	3	2.67	2.58	2.29
Explosives-Lakh kg	18	15.99	15.41	13.70
Initiating system-Lakh Nos	0.60	0.54	0.52	0.46

Summary details of Delay timings used in the blasts

Sl no	Number of holes	Spacing, (m)	Burden, (m)	Total delay time (ms)	Spacing-(ms/m)	Burden-(ms/m)	PF (T/kg)	Remarks
0	20	4	3	250	4.25	14.00	5.00	Nonel
1	22	4	3	332	7.25	13.33	5.37	Edet-ft
2	32	4.5	3	298	5.25	15.30	5.70	Edet-ft
3	25	4	3.5	301	5.25	10.85	6.13	Edet-ft
4	17	4	3	128	4.75	9.00	5.09	Edet-ft
5	24	4	3	191	4.75	14.30	5.22	Edet-ft
6	29	4.5	3	323	4.50	13.50	5.50	Edet-ft
7	16	4.5	3.5	239	6.0	14.28	7.40	Edet-ft
8	17	4.5	3.5	203	4.66	14.00	6.36	Edet-ft
9	15	4	3	169	5.40	14.75	5.6	Edet-ft

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