Design methodology for underground ring blasting

I. Onederra* and G. Chitombo

This paper discusses a systematic approach to underground ring design as well as a methodology for the continuous improvement of designs as conditions change. The methodology is applicable to designs for prefeasibility and feasibility stages as well as designs for currently producing mines. The proposed method still recognises the role of experiential guidelines but provides additional and novel empirical techniques to improve the first pass approximations such that they better suit the prevailing geotechnical conditions. The strength of this method is that the designer is able to assess the impact of the design in terms of the expected fragmentation and potential damage to the surrounding rock mass.

Synopsis

As with development or tunnel blasting, underground ring design requires an adequate understanding and description of the prevailing geotechnical and mining conditions; and the likely impact of the designs to the remaining rock mass. A number of published design processes do not always consider these important aspects associated with ring blasting. Current practices are generally based on either 'rules of thumb' or experiential guidelines, which although useful, can only provide first pass approximations at best. Over the years several modelling and simulation tools have been developed to supplement the use of such rules and guidelines. More recently, numerical approaches which link non-ideal explosives detonation codes to geomechanical rock breakage models, have been developed with the aim of modelling the complete blasting process from first principles.[35] Such approaches once validated offer a new opportunity for the industry to move away from the reliance on simple 'rules of thumb'.

This paper discusses a systematic approach to underground ring design as well as a methodology for the continuous improvement of designs as conditions change. The methodology is applicable to designs for prefeasibility and feasibility stages as well as designs for currently producing mines. The proposed method still recognises the role of experiential guidelines but provides additional and novel empirical techniques to improve the first pass approximations such that they better suit the prevailing geotechnical conditions and thus allowing for blast design optimisation. The strength of this method is that the designer is also able to assess the impact of the design in terms of the expected fragmentation and potential damage to the surrounding rock mass, provided the user has the required input parameters for the corresponding models.

Within the context of this paper, underground ring blasting refers to long hole blasting techniques as applied in mining methods such as bench stoping, sub level open stoping and SLC including undercut blasting and drawbell blasting in the case of block and panel caving methods. All of these methods have been fully described in Refs. 5 and 12.

Ring design terminology

In this paper, the following terminology is used:
  (i) ring: this is a collection of blastholes in a given plane forming what is often termed a blasthole fan. Blastholes are generally drilled radially from a central location or pivot point (e.g. drill position), although a ring may also refer to a set of parallel blastholes
  (ii) ring plane: this refers to the plane in which a fan or ring of blastholes resides. It is generally defined by a bearing (dip direction) and a dip angle (inclination). Depending on the size and geometry of the blasted volume, a design may contain one or more ring planes. For example in sublevel caving (SLC) blasting, ring designs reside in only one plane, in contrast to large open stopes or drawbell blasting, several rings may reside in multiple oriented planes
  (iii) ring dump: ring dump refers to the angle of inclination of a ring plane. It is generally applied in confined blasting conditions such as SLC and undercut blasting applications. The angle of dumping ranges from 10 to 20°. There is empirical evidence[16] suggesting that dump angle may have an impact on fragmentation uniformity as this factor may be directly related to the degree of confinement or void volume available for rock displacement

BRC Mining and Geology, Sustainable Minerals Institute, The University of Queensland, Brisbane, Australia

*Corresponding author, email i.onederra@uq.edu.au
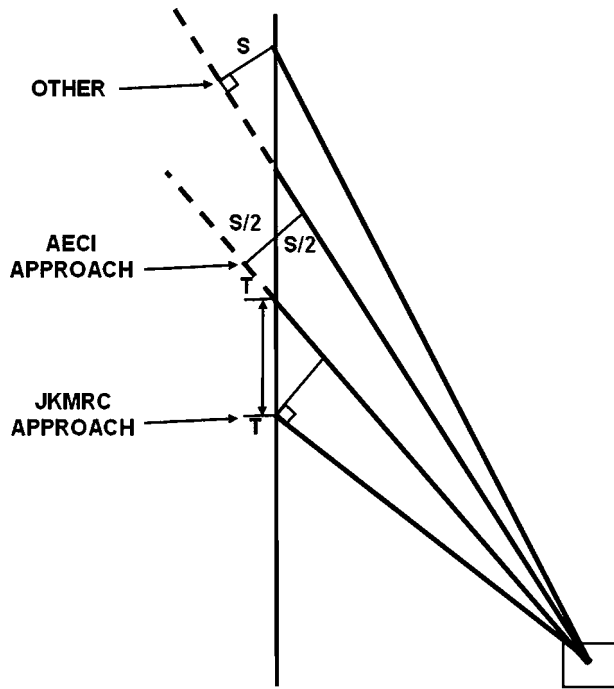
S = True toe spacing
T = Effective toe spacing

1 Definitions of toe spacing

(iv) drill position or pivot point position: this refers to the position inside a development drive where the pivot point of the drill rig is located. In a design it defines the point from which the angle of blastholes is measured

(v) toe spacing: the most commonly used definitions of toe spacing are illustrated in Fig. 1. The definition adopted in the proposed methodology is that defined as the true toe spacing 'S' given by the perpendicular distance from the shortest to the longest hole as shown by the Julius Kruttschnitt Mineral Research Centre (JKMRC) approach

(vi) ring burden: ring burden is defined as the distance between the ring plane and the next free face against which the rock volume is being blasted. Ring burden is also referred to as the design burden

(vii) critical charging distance: refers to the distance between adjacent charges that minimises the likelihood of explosive desensitisation or charge interaction. This is illustrated later in the text

(viii) standoff or offset distance: refers to the distance between a blasthole or toe of a blasthole and an excavation boundary (e.g. stope or drill drive boundary)

(ix) charge length: the length (m) of explosive charge in a blasthole

(x) uncharged collar lengths: the length (m) of hole left uncharged from the collar

(xi) toe energy concentration: toe energy concentration in units of $kg \, m^{-3}$ or $kg \, t^{-1}$ refers to the concentration of energy in the toe region of a ring at a given burden. It is calculated using the 3D explosive energy distribution approach proposed by Kleine[15] which is discussed in subsequent sections of this paper

(xii) critical burden: the minimum burden distance at which no breakage and displacement occurs. At distances equal or greater than the critical burden, the likelihood of freezing increases.

Proposed design methodology

The proposed methodology follows a generic design approach discussed also by the authors in a paper published by Scott *et al.*[37] but further modified for underground production ring blasting. As shown in Fig. 2, the process effectively 'forces' the design engineer to follow a systematic approach to blast design by taking into consideration factors and parameters considered key to successful design, implementation and optimisation of ring blasting in different mining conditions. This paper focuses on the design and analysis process
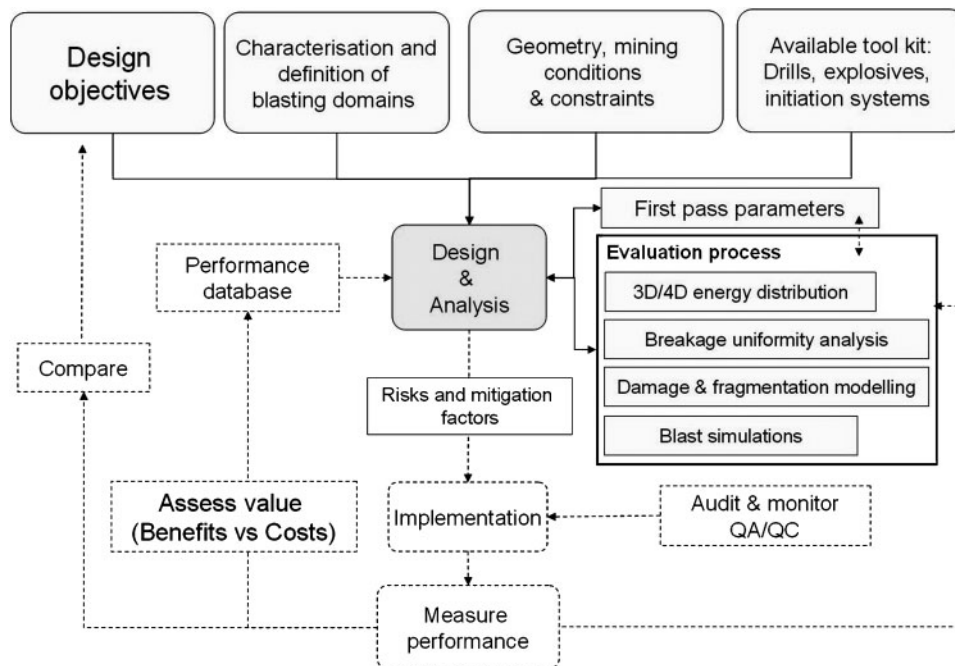
2 General approach to blast design

highlighted in the figure. In this methodology, the application of accepted rules of thumb is supplemented by a 3D and 4D explosive energy distribution analysis, damage and fragmentation models and blast simulation tools. The process also requires identification of risks and mitigation factors prior to implementation. The process is iterative and demonstrates the need to continually review implemented designs as mining conditions change. As with most design approaches, the successful application of this methodology depends on the type, quality and quantity of data used as inputs.

Definition of objectives

When designing a blast, the starting point should be defining the clear objectives or requirements of that blast and they should be measurable or quantifiable. These may include achieving a certain size fragmentation distribution; minimising damage or not exceeding certain vibration thresholds; minimising backbreak, overbreak or dilution; achieving loading and handling productivity targets, consistent drawpoint availability and efficient ore pass performance. In the case of block caving undercut and drawbell design, the design objectives may reflect specific requirements which may include:

(i) rapid undercutting and drawbell extraction
(ii) ease of drilling and charging of the undercut and drawbell rings while at the same time ensuring safe working conditions
(iii) ensuring complete breakage of the undercut and drawbells by achieving suitable explosive energy concentrations and distribution
(iv) achieving suitable fragmentation and throw for subsequent rilling and ore handling
(v) minimising the potential of blast damage to major and minor apex pillars and brows.

Well defined objectives are meant to provide a clear path forward for the definition of preliminary design parameters and support the selection of the most appropriate modelling tools to evaluate these parameters.

Characterisation and definition of blasting domains

Blasting domains are zones within a mining area that have a similar response to blasting. These can be delineated by lithology, structure, alteration or any other property that significantly controls blasting performance. Domains may or may not coincide with domains delineated for geotechnical purposes and hence they should be jointly delineated by the geology, geotechnical and blasting departments. For underground ring blasting, the delineation of domains should include the following parameters which are known to influence blast performance and therefore results:

(i) unconfined compressive strength and tensile strength. Both of these parameters are used as input for the damage and fragmentation models in the proposed methodology
(ii) rock material and rock mass stiffness defined by sonic velocities or static and dynamic young's modulus. These parameters are also used as input for damage and fragmentation models
(iii) rock density. This parameter is used in the 3D and 4D explosive energy distribution analysis,

and more specifically in calculating the energy concentrations at the burden or ring toe sections
(iv) degree of fracturing (fracture frequency, location and orientation of major structures). These are used as input into the fragmentation models but also qualitatively help identify critical boundaries that may influence overbreak and dilution outcomes
(v) a general description of the in situ stress regime. This information is currently used qualitatively as the impact of stress in the blasting process is still not well understood. Numerical models are currently being developed to quantify in-situ stress regimes on blast performance.[35]

Definition of geometry and boundary conditions

Boundary conditions include stope outlines, drilling drive outlines and major geological features. These influence various aspects of design patterns including the definition of drill (pivot) positions, standoff distances and explosive charge concentration.

Computer aided design and mine planning software tools provide the platforms for managing geometrical and geotechnical information. Laser based cavity monitoring systems are now systematically used to define adjacent stope volumes and boundaries. This is important for example, when designing rings against filled or old stopes.

Blast design essential tool kit

In terms of ring design, the key design tools that also influence the choice of the design parameters are drilling equipment, range of explosive types and initiation systems. The design engineer needs to have a thorough appreciation of the capabilities and limitations of each one of these enabling technologies. The key aspects to consider include:

(i) drilling equipment
- size and shape of drilling drives
- type and capability of available drill rigs
  ○ range of hole diameters that the rig can drill
  ○ maximum hole length that can be drilled and drilling accuracy
  ○ alignment and drill stabilising systems
  ○ control systems available such as rotary actuators and pendulum arms
  ○ pivot positioning constraints with respect to excavation boundaries
  ○ overall drilling performance (penetration rates and wear)

(ii) explosives, initiation systems and delivery systems
- type of explosive and critical diameter (e.g. ANFO or gassed emulsions)
- ability to vary the density of the product
- for pumpable products, the capability of the delivery systems with respect to blasthole diameter and length
- sensitivity of product in situ (e.g. sleep times and dead pressing)
- range of delays available
- accuracy of delays.

At prefeasibility and feasibility stages, the blasting engineer is able to influence the specification and therefore choice of the design tool kit, however in an operating mine, the engineer is forced to design and optimise blasts within the constraints of the existing or
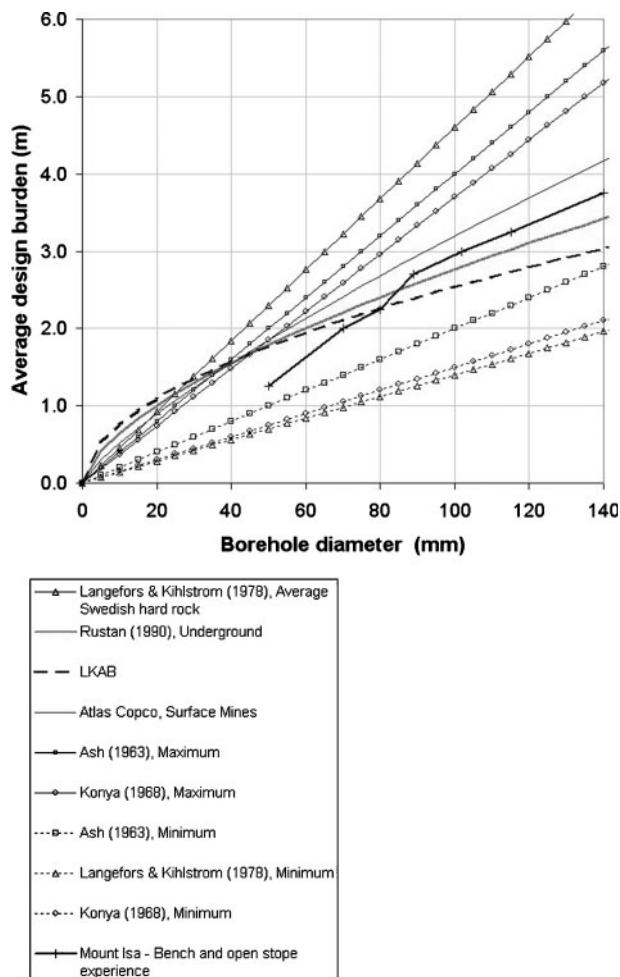
3 Comparison of blasthole diameter versus average design burdens (after Rustan)[36]

available design tool kit. Thus the importance of a thorough appreciation of the capabilities of the blast design engineers tool kit during design and implementation is essential.

Design and analysis process

In the proposed methodology, first pass parameters are initially derived from local experience or 'rules of thumb'. Those often used by the authors are described in this paper. The suitability of these parameters to the prevailing conditions is then evaluated through the static 3D explosive energy distribution analysis, dynamic 4D energy distribution analysis where appropriate; breakage uniformity analysis; fragmentation and damage modelling; and blast visualisation and simulations. These steps and techniques are described in the following sub sections.

First pass parameters

A number of rules of thumb for blast design for ring blasting have been proposed over the years. The bulk of the rules still in use are those developed through controlled tests by the US Bureau of Mines during the 1970s and 80s and those developed in competent granites by Swedish research groups. These rules have mainly been derived from correlations between single parameters or a set of parameters such as blasthole diameters, burdens and toe spacings. Such rules are site specific and should where possible be supplemented by local experience. Examples of such rules for determining blasthole diameter, design burden, toe spacing, explosives charging, and timing include the followings.

Blasthole diameter

In underground blasting literature, little has been published with regards to 'rules of thumb' for determining blasthole diameter; and this is often left to equipment suppliers or by evaluating the capabilities of the available drilling equipment. However in practice, the choice of diameter is usually influenced by the mining or blasthole stoping method used, the maximum drill length and orientation of blastholes (e.g. up hole versus down hole). Blasthole diameter may also be influenced by the type of explosives most likely to be used, e.g. ANFO versus bulk emulsions. Currently, for long hole blasting, the most common diameters used range from 64 to 115 mm, with the most commonly used diameters being 89 and 102 mm, particularly in up hole drilling. In such cases, these diameters can easily be charged with both ANFO and Emulsion products. Smaller diameter holes (64 or 76 mm) are generally restricted to activities such as drawbell blasting where the design objective is to minimise damage to pillars.

Design burden

For the common hole diameters used in ring blasting, design burdens typically range from 1·8 to 3·5 m and rarely 4·0 m. Available rules of thumb include:
  (i) after Myers *et al.*;[21] $B=3.15d \, (SGe/SGr)^{1/3}$ where *B* is the optimal burden distance in feet, *SGe* is the specific gravity of the intended explosive, *SGr* is the specific gravity of the rock (ore) and *d* is the diameter of the explosive (or blasthole) in inches
  (ii) after Rustan;[36] where design burden is a function of blasthole diameter *d* given by the following relation $B=11.8d^{0.63}$. Other relations of this kind are given in Fig. 3.
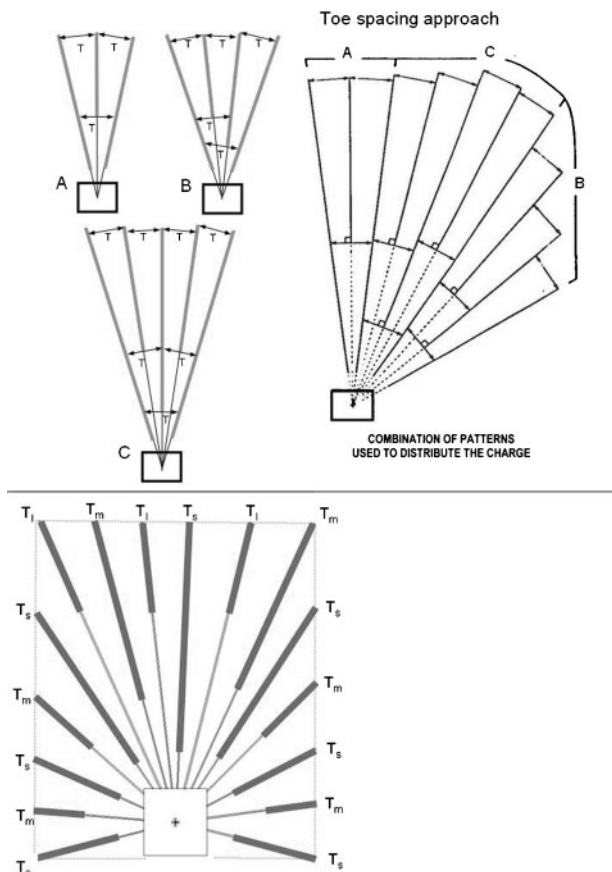
Toe spacing

The most commonly used rules for toe spacing calculations are based on spacing/burden ratios (S/B) and include:
  (i)   The Julius Kruttschnitt Mineral Research Centre's Experience over wide range of rock masses defining S/B ratios in the range of 1·1 to 1·4 for 76–115 mm diameter holes[7,8]
  (ii) Rustan[36] who recommends S/B ratios of 1·5 to 2·0
  (iii) Myers *et al.*[21] recommend ratios in the range of 1·4 to 2·0
  (iv) Cunningham[9] who recommends S/B ratios of 1·3 to 1·5.

Explosive charging guidelines

In terms of explosives charging, the distribution of explosive charges within the blast volume is considered critical. The distribution of explosives is meant to ensure even breakage of the blast volume, in particular at the extreme (toe) regions of a ring.

In underground production blasting, explosive densities are generally in the range of 0·8 to 1·2 g cc^{-1}. Guidelines generally suggest the use of higher densities at the toes of holes to ensure complete breakage, with reduced densities at the collar and boundary holes to

Toe spacing approach

COMBINATION OF PATTERNS
USED TO DISTRIBUTE THE CHARGE

4 Explosive charging guidelines (toe spacing approach and AEL approach): stemming sequence, T_s T_l T_m T_l etc.; T_s=shortest uncharged length=20 charge diameters; T_m=intermediate uncharged length=50 charge diameters; T_l=longest uncharged length=125 charge diameters; stemming not permitted to exceed two-thirds of hole length

minimise damage to brows, pillars and the perimeter of excavations.

The recommended charging guidelines are summarised in Fig. 4. They follow the critical charging distance approach which is also referred to as the toe spacing criteria[13] and the AEL approach,[9] which is based on the definition of the shortest, intermediate and longest uncharged collar lengths in a ring. The toe spacing criteria shown in Fig. 4 is an adaptation of a method developed and used in the Mount Isa mine stopes.[13] As will be discussed later, both of these approaches can be supplemented with the application of static 3D and dynamic 4D explosive energy distribution analysis.

Timing guidelines

There is limited information regarding the calculation of optimum interhole timing for underground ring blasting. General published rules include interhole delays in the range 3–15 ms m^{-1} burden or usually <5 ms m^{-1} blasthole spacing. A general rule based on the 'cooperation' between adjacent charges and their potential impact on fragmentation is summarised in Fig. 5. In reference to this analysis, Guest *et al.*[10] demonstrated that under 'cluster blasting' conditions or when the delay time between holes is short or approaching zero, there is a tendency for fragmentation to become coarser
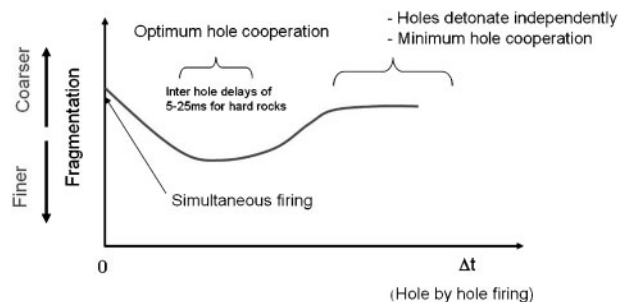
5 Schematic diagram of effect of interhole timing on breakage and fragmentation in underground ring blasting

with a pronounced bimodal distribution (i.e. coarse materials mixed with very fine materials). This is usually referred to as the 'crushing' and 'presplitting' effect. Longer interhole times result in individual hole firing with effectively no cooperation between adjacent charges. In such cases fragmentation is assumed to be mainly structurally controlled and/or controlled by the explosive energy. Controlled experimental work conducted by Stagg and Rholl[40] also supports the cooperation time concept and the hypothetical shape of the curve is shown in Fig. 5. As depicted in this curve, in underground hard rock metalliferous operations, optimum hole cooperation in ring blasting is achieved at a range of interhole times that generally lies between 5 and 25 ms.

Design evaluation process

The design evaluation process involves the application of analytical methods to evaluate the suitability of first pass parameters to the prevailing mining conditions. The analytical tools available in this process include: the static 3D explosive energy distribution analysis, dynamic 4D energy distribution analysis; breakage uniformity analysis; damage modelling; fragmentation modelling and blast visualisation and simulations.

Static 3D and dynamic 4D explosive energy distribution

Kleine[15] developed the concept of static 3D explosive energy distribution as an integration or summation of the energy contribution of all explosive charges at a point in space. Calculation (energy) contours are determined on a defined plane, usually the ring burden plane. The 3D energy distribution approach in its simplest form (also referred to as 3D powder factor) is limited to calculations that only require the pattern geometry, blasthole diameter, charging quantities and the density of the explosive and rock mass; and thus is calculated in units of kg m^{-3} and/or kg t^{-1}. A more detailed description of the calculation method is given in 'Appendix'. All of the main algorithms used to calculate and display energy distribution contours have been incorporated into the JKSimBlast design and analysis software described by Onederra *et al.*[24]

Figure 6 is an example of explosive energy distribution contours for two similar ring patterns, using 64 and 76 mm diameter holes respectively. From a design analysis perspective, the aim is to ensure that the concentrations at the toe sections of rings (expressed in this case in terms of kg t^{-1}) are those that are known through experience to produce acceptable breakage and fragmentation outcomes.

Calculation burden at 1.8 m

▓	0.100 to 0.233 kg/t
░	0.233 to 0.367 kg/t
□	0.367 to 0.500 kg/t
▓	> 0.500 kg/t

64 mm   76 mm
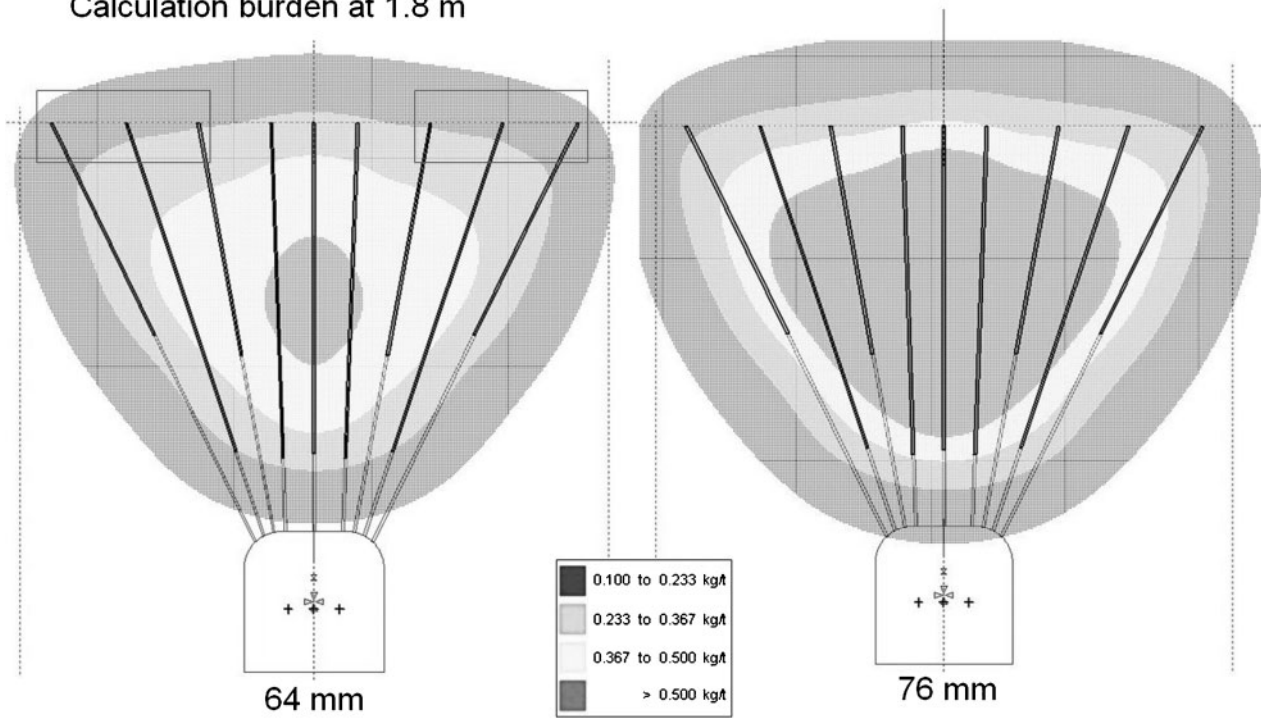
6 Static 3D explosive energy distribution of 64 mm versus 76 mm blasthole configurations

An extension of this graphical analysis is shown in Fig. 7, where the average toe energy concentrations are calculated and plotted for a range of ring burdens. Results from this type of analysis allow for the calculation of the range of practical burdens that are likely to achieve complete breakage. In this case, the breakage threshold was defined at the toes of rings to be in the range of 0·25 to 0·35 kg t^{-1}. As shown in this example, adjustments in the concentration of energy can be made through changes in diameter (i.e. 89 mm versus 102 mm) or explosive density (i.e. 1·0 g cc^{-1} versus 1·2 g cc^{-1}). The application of the 3D explosive energy distribution concept is also discussed by Guest *et al.*,[10] Onederra *et al.*[22] and Pierola *et al.*[32]

A noted limitation of the 3D explosive energy distribution analysis is that it does not consider the impact of delay timing on breakage and fragmentation. A modified approach was introduced in the late nineties by the JKMRC,[34] to address this problem. The approach involves the concept of 'cooperation time' as an index or input parameter that weights the 3D explosive energy contribution of each detonated charge in a blast pattern. Details of the 4D energy distribution analysis have been made available by Riihioja[34] and are also described in 'Appendix'. It is important to note that cooperation time has been directly associated with minimum response or burden movement time. Methods to estimate minimum response time have been discussed by Onederra and Esen[26] and Onederra.[28]

Energy contours derived from the dynamic 4D energy distribution analysis are akin to a 'dynamic powder factor' calculation. Figure 8 shows the result of 4D energy distribution analysis for a stope ring where interhole timing is varied from 25 ms, using pyrotechnic delays, to 10 ms, using electronic delays. As shown, the energy concentrations at both the toes and centre sections of the ring are increased by allowing the explosive charges to 'cooperate'. In other words, the

overall dynamic powder factor for the 10 ms interhole blast is higher than for the 25 ms case. In this example, the cooperation between charges is achieved by the reduced inter hole delay (i.e. 10 ms), which is 5 ms less than the cooperation time (i.e. 15 ms) established for this geometry and rock mass condition using the approach discussed by Onederra.[28] One practical and beneficial implication of the analysis presented in this example is that the quantity of explosives used; and thus the concentration of explosive charges may be reduced when using 10 ms interhole delays, without significantly affecting breakage performance.

Literature on the application of 4D analysis has been limited to site specific optimisation studies where detailed monitoring has shown that interhole timing can have a significant effect on breakage and fragmentation outcomes. An example is given by Guest *et al.*[10] In general, the 4D energy distribution concept provides a practical graphical representation of the distribution of energy as a function of delay timing, and may be used as
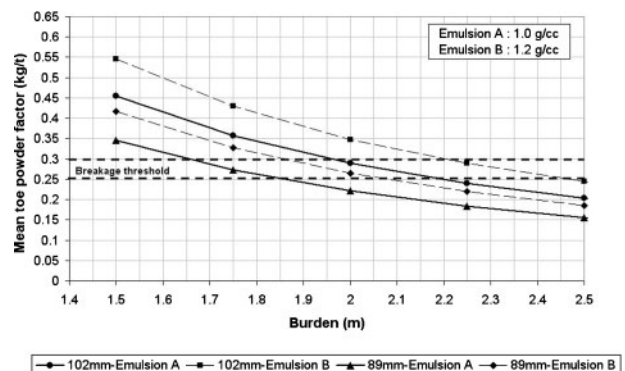
7 Example plot of toe energy concentration versus burden distance for changes in diameter and explosive density
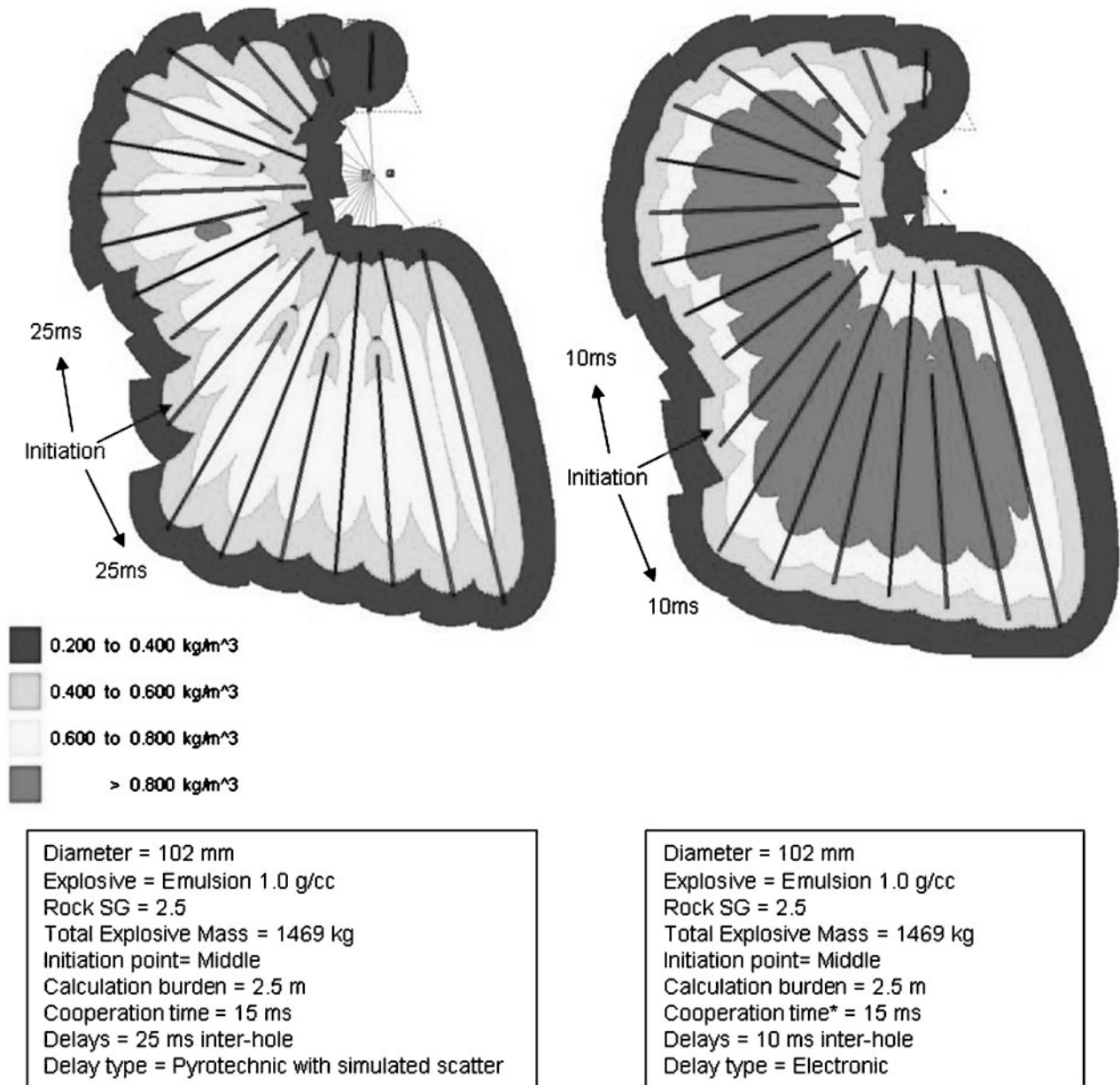
8   Example output of 4D explosive energy distribution analysis (cooperation time is equivalent to minimum response time and was estimated with the approach proposed by Onederra)[28]

a reference for developing empirical relations between interhole timing, energy distribution and subsequent breakage and fragmentation performance. This has become particularly important with the wide spread application of electronic delay detonators.

Breakage uniformity analysis

During the design process, it is important to predetermine the critical burden or the range of burden distances at which the likelihood of poor breakage or possible 'freezing' increases. A methodology based on an empirical blast damage and breakage model has been developed to assist in the determination of critical burden conditions for ring blasting. The methodology involves the definition of breakage indices at a range of design burdens to produce a breakage uniformity curve. The same methodology is used to determine fragmentation modelling parameters.[27] Figure 9 illustrates the components of the breakage uniformity curve which also defines three practical breakage zones:

(i) the first zone is the 'non-uniform breakage zone' which identifies burden distances in which poor breakage and fragmentation is expected to occur and where the likelihood of freezing increases. In the example shown, the boundary is identified at burdens $>2\cdot3$ m. In this example, this corresponds to spacing to burden ratios (S/B) of $<1\cdot30$. Point B lies well inside this zone and can be identified as a point in which inadequate breakage is almost certain

(ii) the second zone is defined as the 'transitional zone'. It lies between the maximum attainable uniformity (point A) and the critical burden boundary (point C). This zone identifies burdens at which breakage is expected to occur and thus provides a way of defining a 'practical design burden' (i.e. point D). In the proposed approach, this is defined as the mid point between A and C. For the example shown, the practical design burden corresponds to $1\cdot9$ m. In terms of cost savings, if
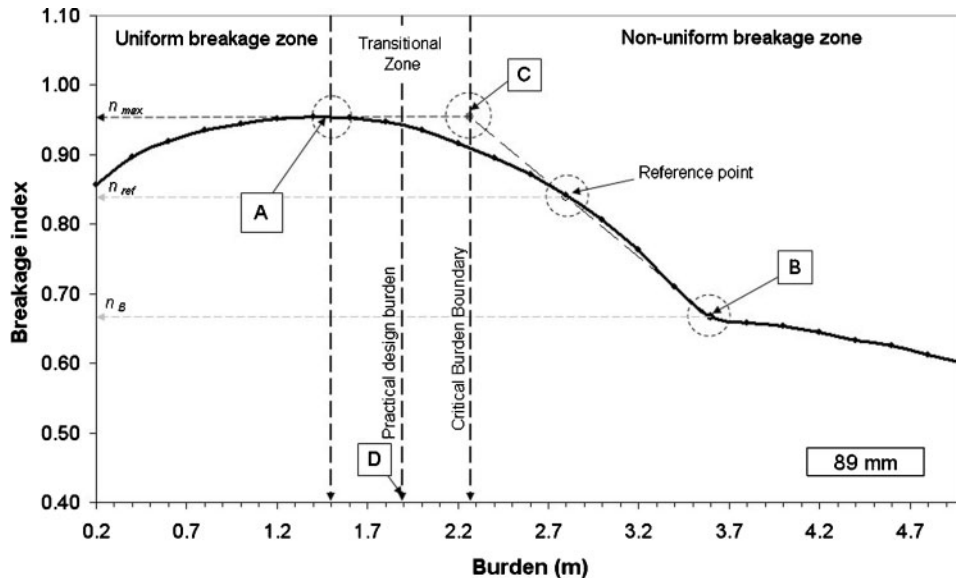
9 Example plot of breakage uniformity curve

fragmentation is not the principal design criteria, there is a clear advantage in choosing burdens within the region bounded by the practical design burden and the critical burden (e.g. in the range 1·9–2·3 m for the 89 mm example case). However, the likelihood of poor breakage is expected to increase as the chosen burden approaches the critical boundary. Good quality control procedures are essential in this zone to minimise the risk of poor breakage or even freezing

(iii) the third zone is the 'uniform breakage zone' which lies to the left of the maximum attainable uniformity (point A) and identifies burden conditions at which efficient breakage and fine fragmentation is expected. In the 89 mm example case, this corresponds to burdens of <1·5 m.

For a given ring design layout, the aim of the proposed approach and the definition of these key regions is to provide engineering approximations of the possible range of design burdens that would allow for effective breakage. The analysis takes into account rock strength and stiffness parameters through peak particle velocity (PPV) attenuation and damage thresholds. Indices of incipient damage and breakage are estimated from knowledge of static tensile strength of the rock (Pa), P wave velocity (m s^{-1}) and static Young's modulus (Pa).

Damage modelling

Rock mass damage in underground mining can be usually associated with either a consequence of induced stresses or blasting. Blast damage is defined as the creation and extension of new fractures and/or the opening of pre-existing geological discontinuities in the rock mass. Blast induced damage weakens a rock mass, potentially leading to stability problems when the excavation size is increased and it is therefore an important component of the analysis process.[4]

Empirical evidence from underground open stoping suggests that near field peak particle velocity levels from blasting can be linked to rock mass disturbance and damage. A relationship between the critical peak vibration velocity and rock mass damage in the near field (within a charge length) can be determined by correlating measured vibrations and the damage determined with geotechnical instrumentation.[11,2,17] The magnitude of the vibrations depends on the nature of the rock mass, the blasting agent, the hole diameter used, the drilled pattern (burden, spacing, hole angle and distance of the holes to the exposed walls) and the hole deviation. Results of back analysis work at Mount Isa Mines has suggested that blasting may control up to 15% of the overall stope hangingwall behaviour.[44]

One of the most widely used engineering methods to model the attenuation of blast waves in a rock mass is the Holmberg–Persson approach.[11] This approach models blast wave attenuation by defining two rock mass and explosive specific constants (K and α). From a fundamental point of view it has however been argued that models which only consider contours of damage, based purely on peak vibration, may be in error if directly related to stress or strain.[3,30] In spite of this, the application of site specific damage criteria based on PPV thresholds has been well documented and supported by several independent studies.[2,11,17–20,29,43–45] Provided that the limitations of the adopted attenuation models are well understood, the above studies have demonstrated that PPV as an engineering index can still be successfully applied to infer the extent of damage. Table 1 provides a summary of published field work that has been conducted to define the Holmberg–Persson attenuation constants (K and α), together with estimates of damage and breakage thresholds.

The index of incipient damage or critical peak particle velocity PPV_{crit} values shown in Table 2 may be estimated from direct in situ measurements of damage such as those described by Villaescusa *et al.*[45] or as a first approximation, by assuming the simple equation for the stress in a plane sinusoidal wave in an infinite medium[31]

$$PPV_{crit} = \frac{T_s v_p}{E_{st}} \qquad (1)$$

where T_s is the static tensile strength of the rock (Pa), v_p is the compressional wave velocity (mm s^{-1}) and E_{st} is the static Young's modulus (Pa).

Table 1   Holmberg–Persson attenuation parameters and damage/breakage thresholds for different rock types

Rock type and reference	K	α	PPV_{crit}, mm s^{-1}	$PPV_{breakage}$, mm s^{-1}
Massive granite[31]	700	0·7	1000	>4000
Andesite[19]	200	0·9	600	>2400
Strong sandstone[19]	400	0·78	450	>1800
Strong shale[19]	175	1·25	350	>1400
Strong shales (across bedding)[45]	456	1·12	848	>3400
Ridgeway volcanics[23]	470	0·94	1200	>4800
Medium/coarse grained quartz diorite[17]	150	0·87	840	>3360
Bronzewing[25]	332	1·0	1100	>4400

Figure 10 shows an example of analysis to identify disturbed zones using PPV as an index to predict the extent of damage. With this analysis, an evaluation of standoff distances, explosive charge densities and charging configurations can be made with the view to minimise the extent of damage beyond the excavation boundaries.

Fragmentation modelling

Applications of empirical methods to underground production blasting have been reported by Stagg *et al.*[41] using site specific formulae to predict fragmentation for simple underground pattern geometries. Applications by Adamson and Lund[1] and Trout[42] have also been reported; and these have been based on modified Kuz–Ram modelling procedures. These approaches however are unable to properly consider the three dimensional distribution of explosive charges which is characteristic of the more complex geometries found in underground ring blasting conditions.

There have been a few cases of mechanistic models being directly applied to underground fragmentation modelling, they include the approach proposed by Kleine[15] and coded into the FRAGNEW program.[34] The SABREX model developed by ICI explosives and described by Kirby *et al.*[14] and the model proposed by Preston[33] embedded in the DynACAD-3D software package. The application of SABREX and more specifically the ICRAX component of this model at the underground Denison Mine was reported by Sheikh and Chung.[38] This modelling work was however restricted to blasting conditions involving parallel hole

Table 2   Parameters used in evaluation of SLC designs

Scenario 1 (base case)	Scenario 2
9 holes	9 holes
102 mm diameter	89 mm diameter
Emulsion, 1·1g cc^{-1}	Emulsion, 1·1 g cc^{-1}
Range of burdens	Range of burdens
Competent rock	Competent rock
K=500 mm s^{-1}	K=500 mm s^{-1}
α=0·9	α=0·9
$PPV_{breakage}$=4500 mm s^{-1}	$PPV_{breakage}$=4500mm s^{-1}
Rock SG=2·5 g cc^{-1}**	Rock SG=2·5 g cc^{-1}
UCS=150 MPa*	UCS=150 MPa
T_s=10 MPa	T_s=10 MPa
P wave=4500 m s^{-1}	P wave=4500 m s^{-1}
E_d=50 GPa	E_d=50 GPa
Medium fracturing	Medium fracturing
(X_{insitu}=0·5 m)	(X_{insitu}=0·5 m)

*UCS is Unconfined Compressive Strength.
**SG is Specific Gravity.

patterns. To date, there is no documented evidence of this approach being applied to more complex underground ring blasting geometries, and the package is proprietary to ORICA explosives.

The fragmentation modelling approaches proposed by Kleine[15] and Preston[33] are the only ones found in literature that have originally postulated frameworks to model underground ring blasting conditions. Kleine's approach consists of the interaction of three independent models which address the determination of in situ block size distributions; the calculation of vibration energy contributing to breakage at predefined points in a volume of rock; and the application of comminution theory to define the breakage characteristics of the rock at these points. With the exception of Chitombo and Kleine,[6] blasting literature is limited with regards to the application of this particular model. It was recognised that calibration requirements were its main disadvantage and the reason for the lack of acceptance in practice.

Recently an alternative modelling framework has been proposed. Designated as FRAGMENTO, this model has an open structure and has been fully described by Onederra.[27] The user input requirements for the application of this model are summarised in Fig. 11. Explosive performance input parameters can be directly obtained from *in situ* monitoring or data available from the explosive supplier. Intact rock input parameters can be obtained from standard geotechnical tests and are readily available at mine site operations. In operating mines, core samples for specific domains can be obtained and appropriate tests conducted for any given domain. If access and adequate exposures are available, geotechnical mapping can be conducted to define rock mass conditions and determine the likely distribution of in situ block sizes. Near field peak particle velocity monitoring can also be conducted to obtain the required attenuation modelling constants and define both damage and breakage thresholds. All geotechnical parameters are incorporated into the model through the application of what are defined as 'near field' and 'far field' fracturing models.

The application of the model in prefeasibility or feasibility studies require the definition of input parameters from past experience or environments similar to those being investigated. The application of this model is discussed and demonstrated later.
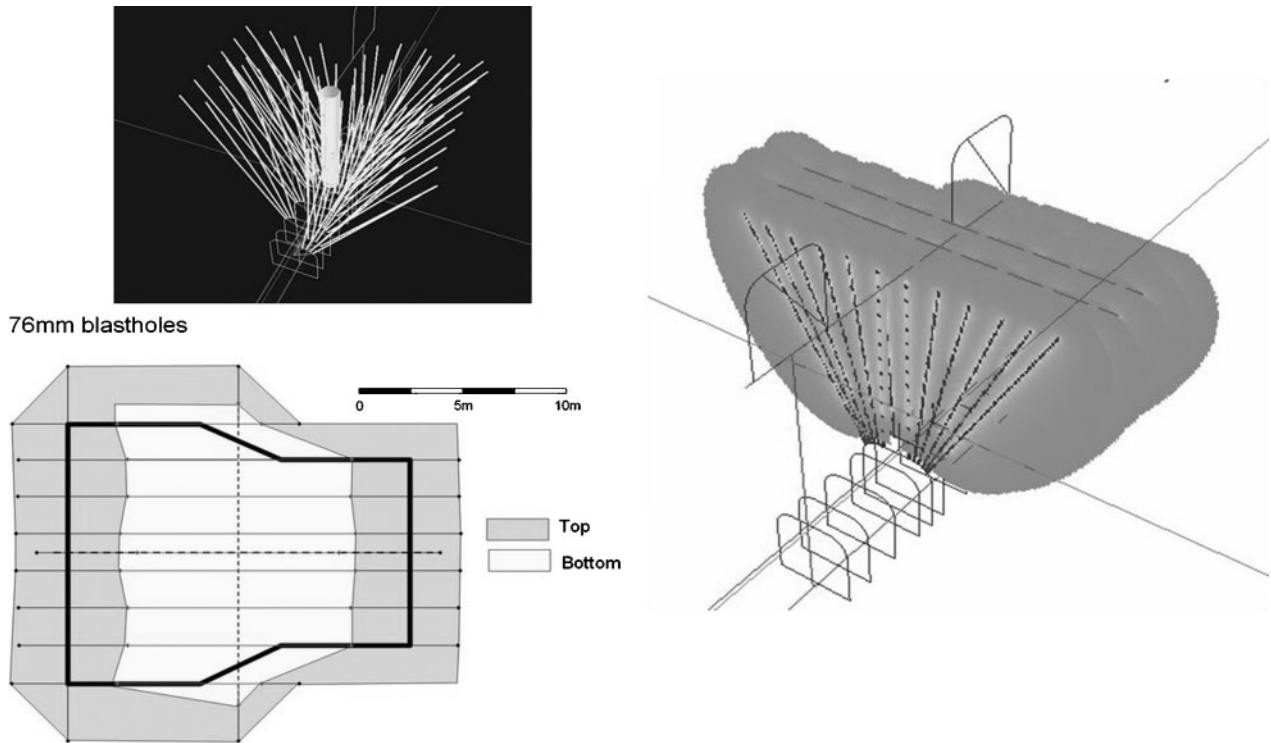
Blast sequence simulations

Blast sequence simulations allow for a visual assessment of the sequence of detonation of a blast prior to actual blasting. This is particularly important for complex geometries such as mass blasts and drawbells. Current software (such as that used in this work) can simulate

76mm blastholes

Top
Bottom

0   5m   10m

10   Example of disturbed zones in drawbell blasting using PPV as index of damage
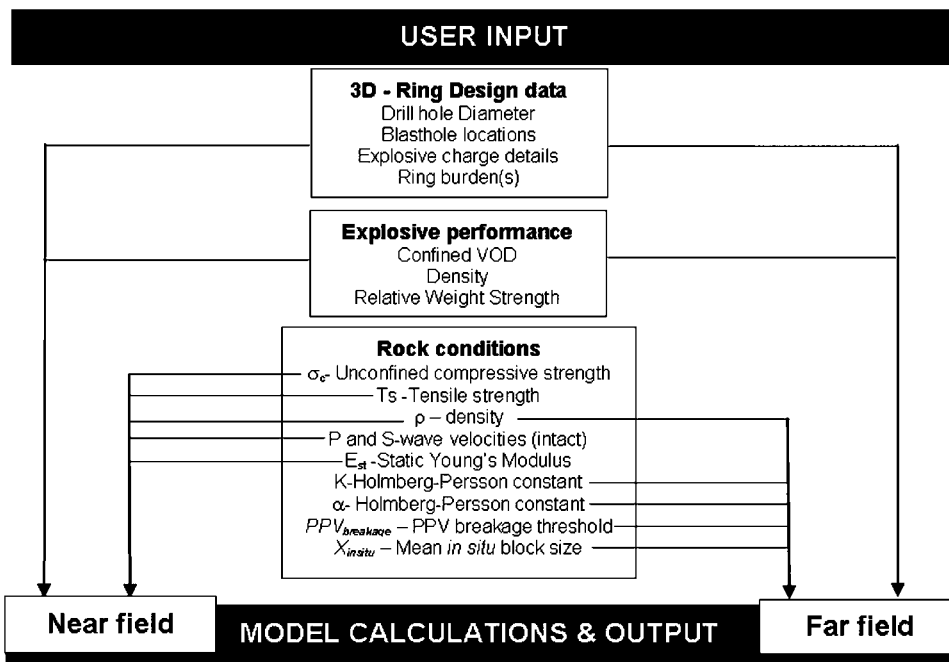
either electronic or pyrotechnic systems. Delay scatter factors can be assigned to pyrotechnic delays and Monte Carlo simulations run on the detonation time statistics defined by these factors.[24]

A simulation of the mass blast detonation sequence using pyrotechnic delays is shown in Fig. 12. In this case the analysis had, as the main objective, the identification of potential areas of 'crowding' (i.e. areas where an unacceptable number of holes detonate at the same time or in a given time window); and the determination of the expected explosive consumption rate (kg/unit time) as a measure of energy release. Another example is given in Fig. 13, in this case the simulation tool is used to describe the detonation sequence of a drawbell blast.

Example applications

The following examples are based on actual case studies. These are meant to help illustrate the application of the methodology for ring blasting as described in the previous sections.

USER INPUT

3D - Ring Design data
Drill hole Diameter
Blasthole locations
Explosive charge details
Ring burden(s)

Explosive performance
Confined VOD
Density
Relative Weight Strength

Rock conditions
σ_c- Unconfined compressive strength
T_s -Tensile strength
ρ – density
P and S-wave velocities (intact)
E_{st} -Static Young's Modulus
K-Holmberg-Persson constant
α- Holmberg-Persson constant
$PPV_{breakage}$ – PPV breakage threshold
X_{insitu} – Mean *in situ* block size

Near field   **MODEL CALCULATIONS & OUTPUT**   **Far field**

11   Input requirements of FRAGMENTO's single ring model
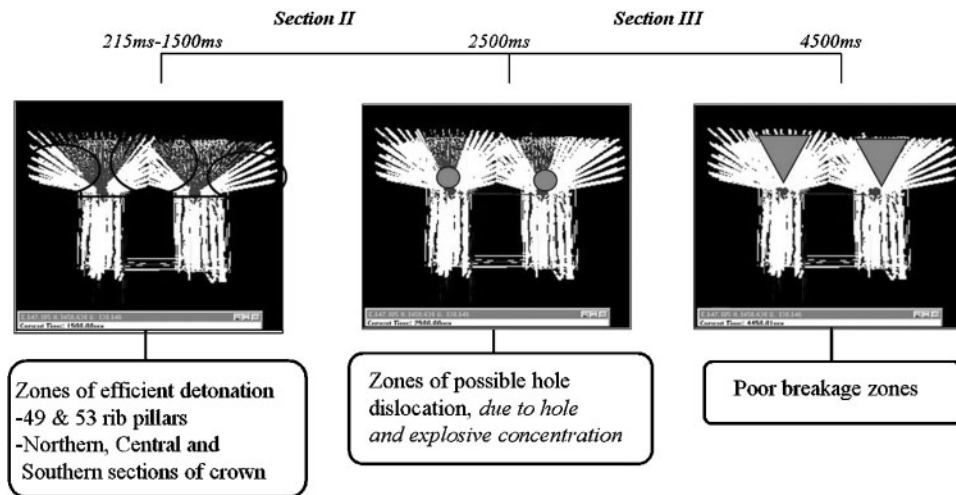
12 Example simulation of rib and crown pillar mass blast[22]

Case study 1: evaluation of critical burden and fragmentation outcomes of alternative SLC design configurations
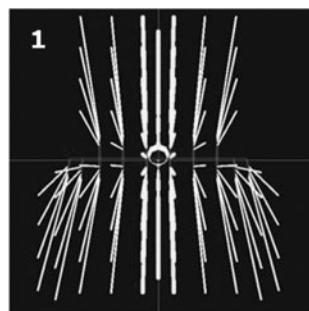
An existing SLC operation was interested in evaluating ring design burdens and relative changes in fragmentation as a result of changes in blasthole diameter (i.e. from 102 to 89 mm). The base case design consisted of a nine hole pattern and its parameters are summarised in Fig. 14.

The characteristics of the two evaluated scenarios are summarised in Table 2.

The results of breakage uniformity analysis for these scenarios are shown in Fig. 15. The results indicate that for the assumed conditions, critical design burdens should be approximately 3·0 and 2·2 m for the 102

and 89 mm diameter configurations respectively. Practical design burdens should be of the order of 2·6 m for the 102 mm configuration; and 2·0 m for the 89 mm configuration.
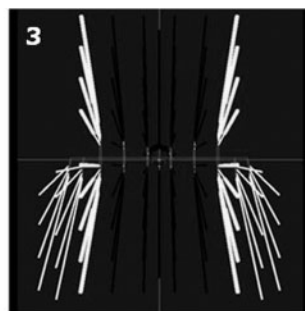
The modelled relative changes in the expected fragmentation for a range of burden configurations are summarised in Fig. 16. As shown, for a given burden, relative changes in intermediate to mean fragmentation outcomes defined by the P50 value (i.e. 50% passing fraction) are not as significant as coarse fragmentation outcomes defined by both the P80 and P90 fractions. For the same burden (i.e. 2 m) rings drilled with 89 mm holes are expected to produce coarser fragments than with rings drilled with 102 mm diameter holes.
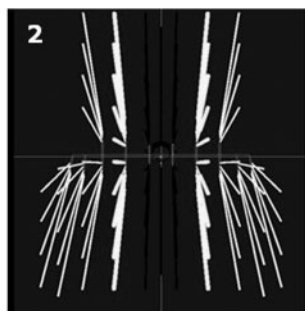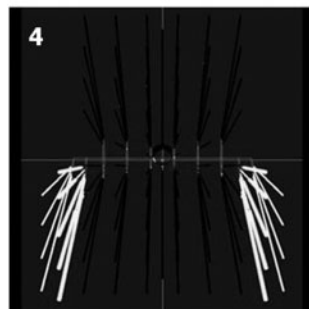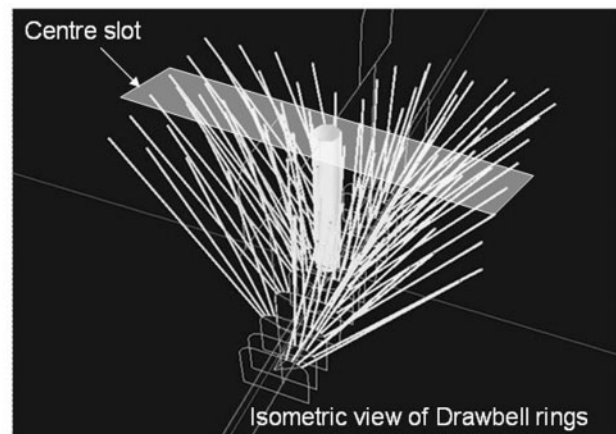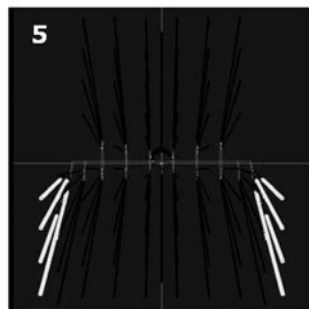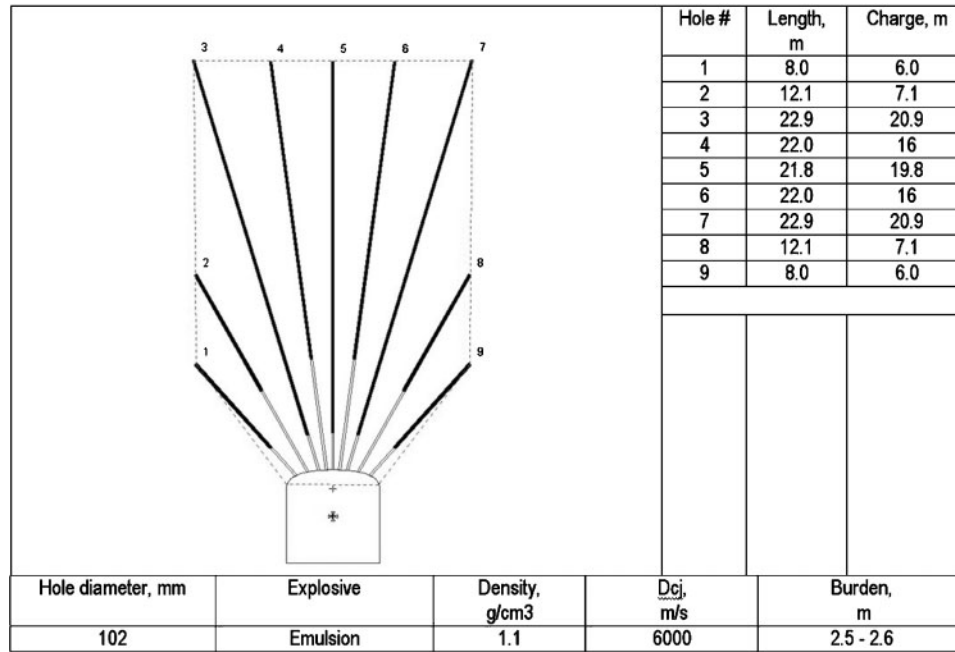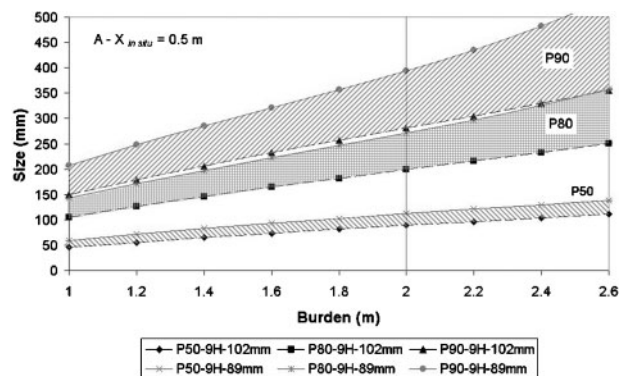

13 Example simulation of drawbell detonation sequence

14   Design parameters of base case SLC design (scenario 1)
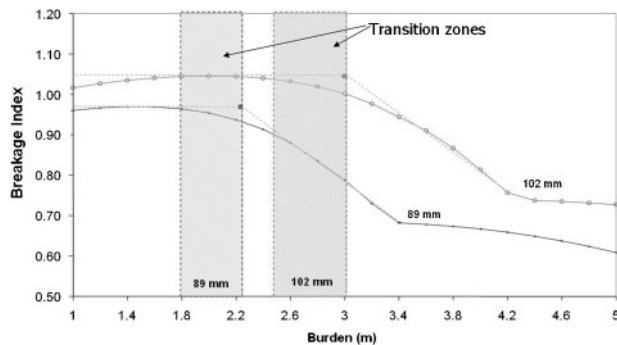
15   Breakage uniformity analysis of 89 and 102 mm SLC patterns

16   Results of fragmentation modelling from 89 and 102 mm SLC configurations

Case study 2: design parameters for narrow/inclined undercut rings
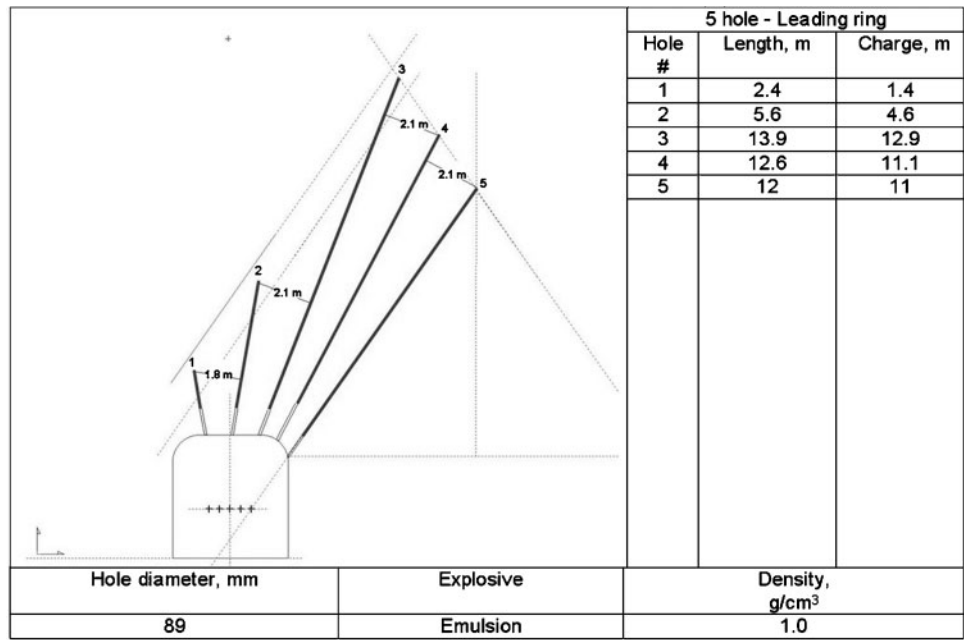
As part of the feasibility study of a block caving operation, design parameters for a narrow inclined undercut were evaluated. Figure 17 describes the parameters for the originally proposed five hole layout. It should be noted that the rock mass and explosive input parameters used in the analysis were based on the geotechnical information available at the time and corresponded in general terms to a 'hard rock' with the characteristics summarised in Table 3.

The breakage uniformity results for the proposed five hole ring shown in Fig. 18 indicated the critical burden
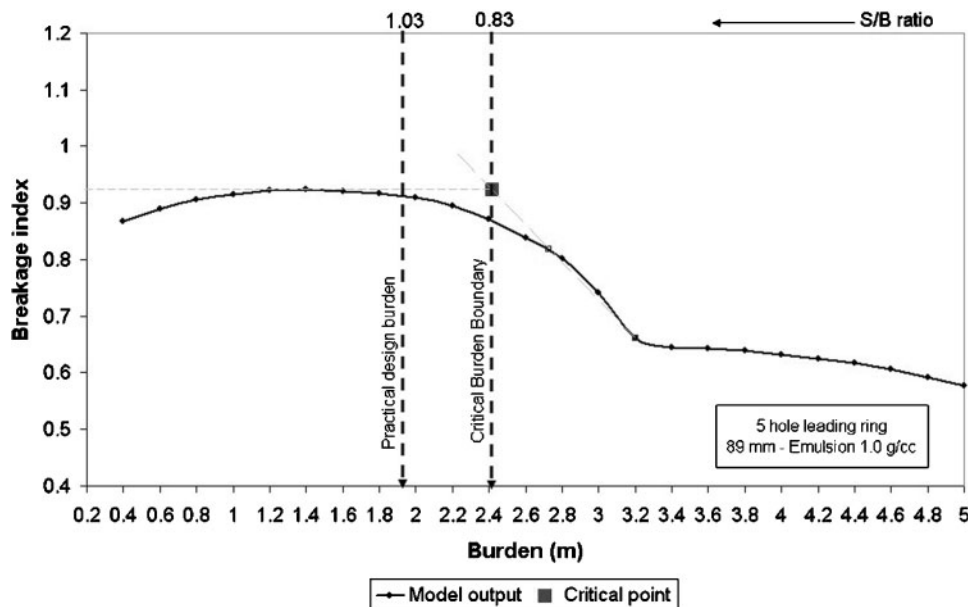
boundary to be $\sim 2{\cdot}4$ m, corresponding to an S/B ratio of $0{\cdot}83$. Burdens in the estimated transitional zone ranged from $1{\cdot}8$ to $2{\cdot}4$ m. The evaluation dictated ring burdens to be strictly $<2{\cdot}4$ m. Given that possible deviations from 'as designed' conditions can occur in practice, it was considered that a $1{\cdot}8$–$2{\cdot}0$ m burden configuration would be an adequate starting condition. In this configuration, average toe spacings were selected after conducting an evaluation of the three dimensional distribution of explosive energy for the volumes defined by the ring burdens.

Table 3   Mechanical and breakage properties of orebody

Rock material properties		Rock mass attenuation constants and breakage threshold		Rock mass structural characteristics	
UCS, MPa	180	K	500	Fractured X_{insitu}	0·5 m
T_s, MPa	14	α	0·9		
v_p, m s^{-1}	5300	$PPV_{breakage}$	>4000 mm s^{-1}		
v_s, m s^{-1}	2900				
E_d, GPa	58				
Density, kg m^{-3}	2680				

5 hole - Leading ring		
Hole #	Length, m	Charge, m
1	2.4	1.4
2	5.6	4.6
3	13.9	12.9
4	12.6	11.1
5	12	11

Hole diameter, mm	Explosive	Density, g/cm³
89	Emulsion	1.0

17   Design parameters of narrow inclined undercut ring

18   Breakage uniformity curve for evaluated five hole inclined undercut ring
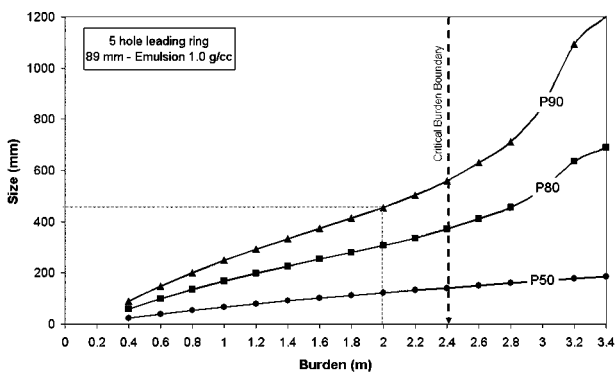

19   Fragmentation modelling results of 89 mm inclined undercut ring

Figure 19 shows the results of fragmentation predictions for the proposed five hole ring configuration. As shown, for a 2 m burden configuration, modelling results indicated that 90% fragments are expected to be <450 mm.

During the actual extraction of the undercut and as more data became available, the five hole inclined ring layouts were further modified by the operation and optimised to three hole configurations. Ring design burdens in these cases were maintained at 2·0 m and complete extraction of the undercut was successful.[39]

A back analysis of the three hole inclined ring configuration was conducted for the geotechnical conditions assumed at the feasibility stages. Figure 20 shows the results of this analysis. As shown, for the three hole configuration and the originally assumed 4000 mm s^{-1} PPV breakage threshold, critical burden
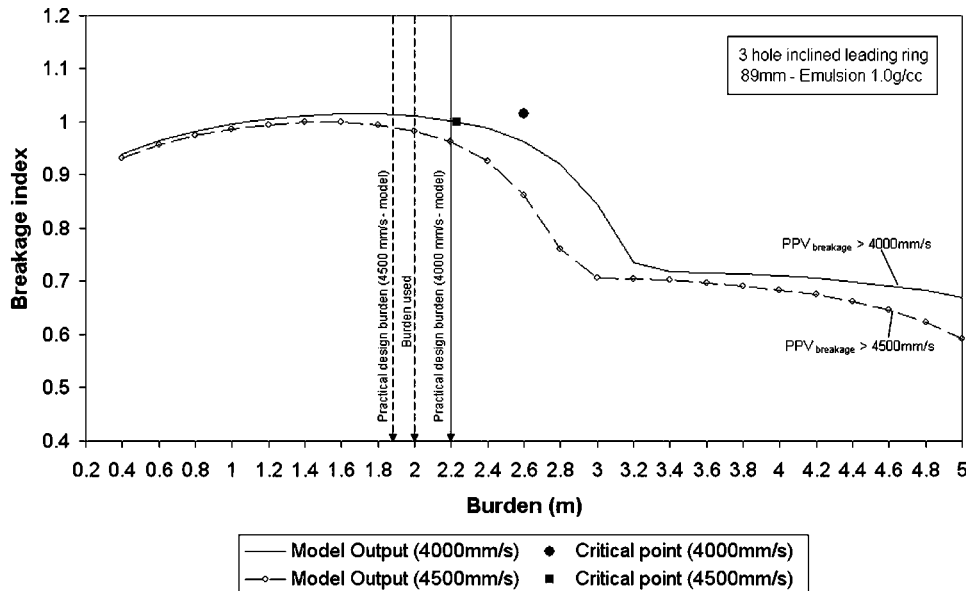
20 Breakage uniformity analysis of three hole undercut ring configurations

conditions were estimated to be reached at 2·6 m, the results also give a transition zone ranging from 1·8 to 2·6 m.

To simulate 'harder' or more competent rock conditions, the analysis also considered the assumption that breakage would be achieved at values >4500 mm s^{-1}. Results from this analysis indicated that critical burdens could be reached at values of 2·2 m, with the transitional zone in the range of 1·5 to 2·2 m. In both cases, modelling results confirmed that breakage would be achieved at burdens of 2·0 m.

The results of this analysis confirmed that for preliminary design evaluations, the originally adopted rock mass breakage parameters were appropriate, however a breakage threshold of 4500 mm s^{-1} appears to give a narrower and more appropriate estimation window of the transitional zone. This indicates that rock mass conditions were slightly 'harder' or more competent than anticipated and first assumed at the feasibility stage. The analysis demonstrates that, once in operation, input parameters should be reviewed and further refined as described in the proposed design and analysis methodology.

Conclusions

Current design methodologies applied to underground production blasting still rely on rules of thumb to provide first pass approximations. Over the years several modelling and simulation tools have been proposed which can be used to supplement the use of rules of thumb. This paper has introduced a methodology which enables a more systematic approach to underground ring design.

The proposed design methodology can be applied for feasibility studies or for operating mines as well as for blast optimisation. The process requires the definition of key objectives and supporting design information such as the characteristics of blasting domains; mining and geometry constraints and the available tool kit. This paper placed particular focus on the design and analysis process which is supported by the application of experiential guidelines, 3D computer aided design

techniques, static and dynamic explosive energy distribution analysis, damage and fragmentation models and blast simulation tools.

Example applications based on actual case studies have been presented and demonstrate the requirement to continually review recommended designs as information becomes available and mining conditions change.
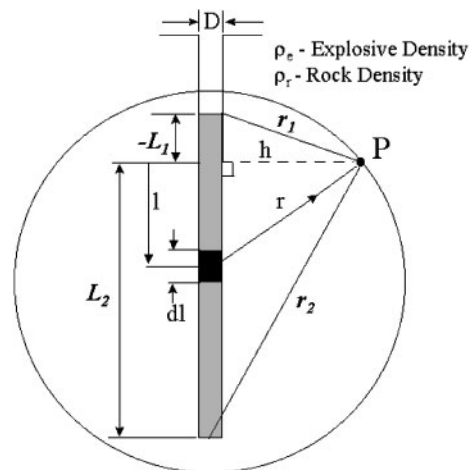
Appendix
Static 3D explosive energy distribution

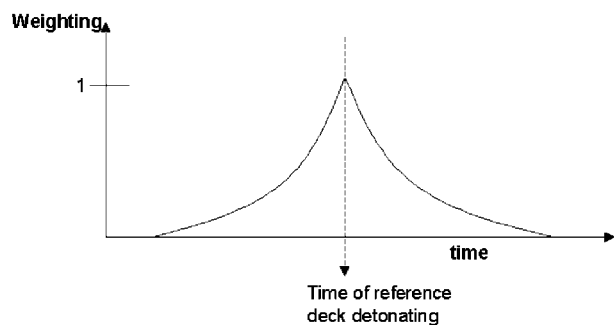In reference to Figure 21 below, the 3D explosive energy distribution calculation for one hole/deck is given by

$$P = \int_{L1}^{L2} \frac{1000\rho_e \pi \left(\frac{D}{2}\right)^2}{\rho_r \frac{4}{3}\pi(h^2+l^2)^{2/3}} dl \tag{2}$$

Equation (2) can be integrated and rewritten as

$$P = 187 \cdot 5 \frac{\rho_e}{\rho_r} D^2 \frac{1}{h^2}\left(\frac{L_2}{r_2} - \frac{L_1}{r_1}\right) \tag{3}$$

21 Schematic diagram of the 3D explosive energy distribution analysis (after kleine 15)

22 Weighing function of 4D analysis (After Riihioja 34)

The above calculation can be explained as an extension of the traditional powder factor calculation by writing the equation for the resulting explosive concentration at a point 'P' for a sphere centred at the charge segment. Special conditions apply to the above relationships at the charge axis (i.e. $h=0$) and at very large distances (i.e. $h=$ infinity). The explosive concentration at any point in 3D is determined by solving the appropriate integrated form of the equation for each explosive charge and summing the values. Important points to note about this calculation:

(i) the result is not strictly a powder factor since a sphere of rock is used. However its has been shown that an approximation to powder factor is obtained when calculations are conducted at the burden plane

(ii) the result is a true 3D representation of the explosive distribution.

Dynamic 4D explosive energy distribution

The 4D explosive energy distribution differs from a 3D calculation, in that a deck's detonation time is considered. The model is based on the 3D analysis and incorporates a weighting factor which is a function of the time a deck detonates and a rock mass specific factor called 'cooperation time'.

As part of the 4D energy distribution analysis, a timing simulation must be carried out first in order to estimate the average detonation time of each deck. The approach can be used to simulate both pyrotechnic and electronic delays. With pyrotechnic delays, a scatter factor can be defined and Monte Carlo simulations run to simulate the potential deviation of detonation from nominal times.

The 4D energy distribution tessellates points on a plane specified by the user just like the 3D energy distribution. For each calculation point, the nearest charged deck is found. The time at which this deck detonates is used as a reference time. The weighting function is determined based on the cooperation time and the detonation time of charges. For every explosive deck in the timing simulation the 3D explosive energy value is calculated and multiplied by the following term

$$W = e^{\left(\frac{-|t_d - t_n|}{t_c}\right)}$$

where t_d is the time the charge deck detonated, t_n is the time the nearest deck to the calculation point detonated and t_c is the cooperation time. The graph of this weighting function is shown in Fig. 22.

References

1. W. R. Adamson and A. S. Lund: Proc. Conf. EXPLO 2001, Hunter Valley, NSW, Australia, October 2001, The Australasian Institute of Mining and Metallurgy, 207–223.
2. P. Andrieux, C. McKenzie, J. Heilig and A. Drolet: Proc. 10th Symp. on 'Explosives and blasting research', Austin, TX, USA, January 1994, ISEE, 107–119.
3. D. Blair and A. Minchinton: *Int. J. Blast. Fragm.*, 1997, **2**, 59–72.
4. E. T. Brown, G. P. Chitombo, T. Li and P. A. Walker: Proc. ISRM Int. Symp. on 'Integral approach to applied rock mechanics', (ed. M. van Sint Jan), Vol. I, 289–301, 1994, Santiago, ISRM.
5. E. T. Brown: 'Block caving geomechanics', ICS study 1997–2000, University of Queensland, Queensland, Australia, 2003.
6. G. P. Chitombo and T. H. Kleine: Proc. 2nd Int. Symp. on 'Rock fragmentation by blasting', Keystone, CO, USA, August 1987, Society for Experimental Mechanics, 657–671.
7. G. P. Chitombo: 'A review of the proposed front cave undercut rings for Koffiefontein', JKMRC/AMIRA/BART P447 report, Brisbane, Australia, 1996.
8. G. P. Chitombo and I. Onederra: 'Drill and blast designs and parameters for a narrow inclined, advanced undercut of the NPM E26 lift 2 block cave', JKTech report no. 99257, 1999.
9. C. Cunningham: Personal communication, Documents from the African Explosives Limited (AEL) handbook, 2005.
10. A. Guest, G. Chitombo and H. Grobler: Proc. Conf. EXPLO 1995, Brisbane, Australia, September 1995, AusIMM, 75–80.
11. R. Holmberg and P. A. Persson: *Trans. Inst Min. Metall. A*, 1980, **89A**, 37–40.
12. W. Hustrulid: 'Underground mining methods: engineering fundamentals and international case studies', (ed. W. A. Hustrulid and R. L. Bullock), 3–14; 2001, Littleton, CO, Society for Mining, Metallurgy and Exploration.
13. B. King: 'An existing ring design program (RDS)', Internal JKMRC-Mining Report, Mount Isa Mines, Brisbane, Australia, September 1987.
14. I. J. Kirby, G. H. Harries and J. P. Tidman: Proc. 13th Conf. on 'Explosives and blasting technique', Miami, FL, USA, February 1987, SEE, 184–194.
15. T. H. Kleine: 'A mathematical model of rock breakage by blasting', PhD thesis, JKMRC-The University of Queensland, Queensland, Australia, 1988.
16. T. Law: 'Fragmentation analysis: determining the influence of dump angles on fragmentation results', Internal report, Newcrest Mining Ltd, Ridgeway Mine, Australia, 2001.
17. T. LeBlanc, J. Heilig and J. Ryan: Proc. 6th High-Tech Semin. on 'The state of the art in blasting technology instrumentation and explosives applications', Blast Analysis International Inc. and International Society of Explosives Engineers, Boston, MA, USA, July 1995, 225–291.
18. Q. Liu and R. Proulx: Proc. Conf. NARMS 1996: 'Rock mechanics tools and techniques', 599–608; 1996, Rotterdam, Balkema.
19. C. McKenzie, C. Scherpenisse, J. Arriagada and J. Jones: Proc. Conf. EXPLO 95, Brisbane, Australia, September 1995, AusImm, 285–292.
20. T. Meyer and P. G. Dunn: Proc. Conf. NARMS 1996: 'Rock mechanics tools and techniques'; Vol. 1, 609–617; 1996, Rotterdam, Balkema.
21. T. Myers, R. Lundquist and C. Konya: Proc. 16th ISEE Annual Conf., Orlando, FL, USA, February 1990, ISEE, 43–52.
22. I. Onederra, J. Player, P. Wade and G. Chitombo: *Int. J. Blast. Fragm.*, 1999, **3**, 1–23.
23. I. Onederra: 'Near field vibration monitoring of SLC ring blasting in XC11 of the 5305 level undercut', JKMRC-BARTII Project Report, Newcrest Ridgeway, Ridgeway, Australia, November 2001.
24. I. Onederra, D. La Rosa, K. Riihioja and G. Power: Proc. Conf. EXPLO 2001, Hunter Valley, Australia, October 2001, AusIMM, 237–244.
25. I. Onederra and D. La Rosa: 'Introduction of engineering tools for the continuous improvement of drilling and blasting practices', Internal JKMRC Report, Newmont-Bronzewing operation, Australia, December 2002.
26. I. Onederra and S. Esen: Proc. EFEE 2nd World Conf. on 'Explosives and blasting technique', Prague, Czech Republic, September 2003, EFEE, 269–275.
27. I. Onederra: 'A fragmentation model for underground production blasting;, PhD thesis, The University of Queensland, JKMRC, Australia, 2005.

28. I. Onederra: *Min. Technol. (Trans. Inst. Min. Metall. A)*, 2007, **116**, 7–15.
29. F. Ouchterlony, C. Sjoberg and B. Jonsson: Proc. 9th Ann. Symp. on 'Explosives and blasting research', San Diego, CA, USA, January–February 1993, ISEE, 189–197.
30. F. Ouchterlony, M. Olsson and I. Bergqvist: Proc. Conf. EXPLO 2001, Hunter Valley, NSW, Australia, October 2001, AusIMM, 281–303.
31. P. A. Persson, R. Holmberg and J. Lee: 'Rock blasting and explosives engineering', 259–264; 1994, Boca Raton, FL, CRC Press.
32. M. Pierola, M. Diaz and W. Adamson: Proc. 8th Int. Symp. on 'Rock fragmentation by blasting (Fragblast-8)', Santiago, Chile, May 2006, Chilean Association of Explosives Engineers, 279–289.
33. C. Preston: Proc. Conf. EXPLO 95, Brisbane, Australia, September 1995, AusIMM, 1–12.
34. K. Riihioja: Personal communication, 2004.
35. M. Ruest, P. Cundall, A. Guest and G. Chitombo: Proc. 8th Int. Symp. on 'Rock fragmentation by blasting (Fragblast-8)', Santiago, Chile, May 2006, Chilean Association of Explosives Engineers, 140–151.
36. P. A. Rustan: Proc. 3rd Int. Symp. on 'Rock fragmentation by blasting (Fragblast 1990)', Brisbane, Australia, August 1990, AusIMM, 303–310.
37. A. Scott, I. Onederra and G. Chitombo: Proc. 8th Int. Symp. on 'Rock fragmentation in blasting (Fragblast-8)', Santiago, Chile, May 2006, Chilean Association of Explosives Engineers, 232–238
38. A. M. Sheikh and S. H. Chung: Proc. 2nd Int. Symp. on 'Rock fragmentation by blasting', (ed. W. L. Fourney and R. D. Dick), 521–530; 1987, Keystone, CO, International Society of Explosives Engineers.
39. A. Silveira: Proc. Int. Conf. MassMin 2004, Instituto de Ingenieros de Chile, Santiago, Chile, August 2004, 410–414.
40. M. S. Stagg and S. A. Rholl: Proc. ISEE General Conf., Anaheim, CA, February 1988, ISEE, 318–330.
41. M. S. Stagg, R. E. Otterness and F. Djahanguiri: Proc. 10th Ann. Symp. on 'Explosives and blasting research', Austin, TX, USA, January–February 1994, International Society of Explosives Engineers, 197–208.
42. P. Trout: Proc. Underground Operators' Conf., Townsville, Australia, July 2002, AusIMM, 107–117.
43. A. M. Tunstall, N. Djordjevic and H. A. Villalobos: *Trans. Inst. Min. Metall. A*, 1997, **106A**, 42–46.
44. E. Villaescusa, C. Scott and I. Onederra: 'Near field blast monitoring at Hilton', Mount Isa technical report no. RES MIN 78, 1997.
45. E. Villaescusa, I. Onederra and C. Scott: *Int. J. Blast. Fragm.*, 2004, **8**, (1), 6–29.