Transforming Underground Blasting Operations into Primary Crushing Operations

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Abstract

There has been significant progress made with narrow vein gold mines in which a second option for crushing has begun to attract attention. To ensure the profitability of a mine that has based its economics on the absence of a crusher station, more than just best blasting practices are required. Using a blasting operation itself as a primary crusher would undoubtedly save enormous capital costs as well as eliminate some of the costs of crushing ore with the result that considerable savings might be realized. The challenge would be to define the conditions necessary to make this type of operation feasible by providing consistent blasting methods that result in a significant paradigm shift with respect to both blasting and current underground crushing methods. A break model has been developed and applied as a technique in designing blasting patterns based on a unit charge blasting geometry and stress reflection from free faces. There are four components to this model and these are 1) unit charge, 2) thermodynamic/energy, 3) stress reflection, and 4) radial break. Some important changes in timing between holes and rows based on rock properties and the percentage of burden moved were also used. On-site testing at a narrow vein gold mine in Quebec Canada, provided measured parameters for the model. No underground crusher was planned - relying only on the fragmentation profile generated by ring blasting. The specification for muck was that the fragmentation profile needed to be 100% < 400 mm (16 in) with no overbreak - with dilution less than 10%.

In this paper, the term "Break Model" refers to the above four components obtained from in-situ testing carried out at a mine site prior to facilitating a blasting pattern design program.

The parameters from in-situ testing were used for input to the model. Details are provided in this paper using new break methodology at Goldcorp's Eleonore gold mine, where some significant improvements have been made over the past year.

Introduction

Surface blasting operations and underground blasting operations can be compared using the following criteria in table 1;

Table 1. Comparisons Between Surface and Underground Blasting Operations

Surface	Underground
• Geometry is not complex - rectilinear and/or staggered	• Complex pattern designs because of complex orebody shapes drilled with oblique holes
• Blast results can be seen quickly	• Blast results cannot be seen after blasting
• Powder factors tend to be about ½ those in underground blasting operations	• Powder factors appear to be higher by twice the value of surface blasting
• Fragmentation larger than underground	• Fragmentation much smaller than surface
• Energy distribution is more uniform – no oblique holes	• Energy distribution is concentrated usually at collar regions because of oblique holes
• Ratio of hole length to burden is roughly 5 (100 mm, 4 in hole) with hole lengths and burden being constant	• Ratio of hole length to burden covers a range of values from 1 for short holes to 10 for long holes
• Access to the open face is available and visible	• Access to open face is not visible or accessible
• Perimeter control/secondary is usually required	• Perimeter control/secondary is often required to prevent dilution

Regardless of the differences shown above, there has always been the problem of devising a suitable means of comparing the vast number of explosive products to an equally large number of rock/ore types in terms of blast pattern design and break. A testing program at Queen's University at Kingston finally provided a means of comparing explosive types as well as rock/ore types for the purpose of employing an explosive/rock/ore type pair used in characterizing criteria to a common configuration for blast pattern design purposes. There are a number of design components used in this paper with the first describing the geometry of a unit charge. All of these components (4) comprise the break model developed specifically for blast design based solely on explosive properties and rock properties and shock attenuation.

Component 1 - Unit Charge Model

The unit charge model provides a means of comparing explosive and rock properties to a common base. The unit charge itself is a geometry that includes the detonation velocity of an explosive along with the P wave velocity of the rock or ore that is undergoing blasting. Therefore, there are three constraints that are placed on the model - every explosive and rock/ore type obey the same rules;

1. determine the P wave velocity for the rock/ore to be blasted
2. from the commercial explosive used to supply energies in the form of shock and gas expansion components, get the detonation velocity of the explosive in the diameter it is used
3. the travel time for the P wave to hit the free face dictates the length of the explosive charge in terms of the detonation velocity of the explosive product

All commercial explosives and all media are handled the same way by the model - which facilitates the blasting design process. Figure 1 below illustrates the geometry a unit charge where P is the initiation point. The unit charge length is red as shown in the figure.

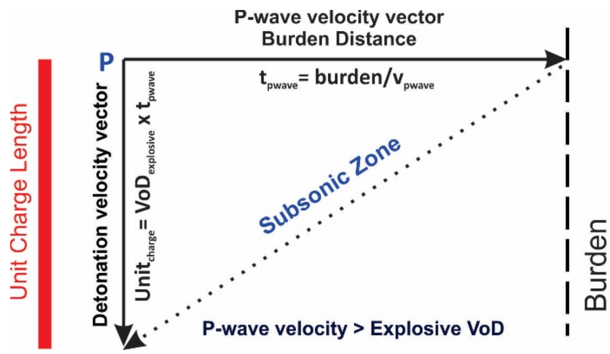


Figure 1. Effective geometry of a unit charge. All explosive/rock/ore combinations are analyzed the same way. There are three configurations – subsonic ($P_{wave} > VoD_e$) which is shown above then sonic ($P_{wave} = VoD_e$), and supersonic ($P_{wave} < VoD_e$). All commercial explosives and all media are handled the same way by the model which facilitates the blasting design process.

Notes Concerning Component 1:
ASSUMPTION 1. Ore/rock will break as a direct manifestation of the unit charge being in direct intimate contact with the ore/rock medium (no air gaps or portions of unloaded holes). No decoupling allowed.
ASSUMPTION 2. The explosive is undergoing detonation for the full length of the unit charge. There are no inert gaps permitted, detonation velocity is assumed to be constant.

To illustrate the use of the unit charge as part of the design process, specific calculations coming from the inputs listed in table 2 are calculated and shown in table 3 using the initial pattern dimensions of 2.3 m (7.5 ft) spacing and 2.0 m (6.6 ft) burden.

Table 2. Inputs for the Initial Blasting Pattern of 2.3 m (7.5 ft) Spacing and 2.0 m (6.6 ft) Burden

Parameter	Input Values and Units	
Drill Hole Diameter:	100 mm	4.00 in
Spacing Distance:	2.3 m	7.54 ft
Burden Distance:	2.0 m	6.56 ft
Explosive Product:	Subtek Intense	Subtek Intense
Detonation Velocity:	5723 mps	18778 fps
Rock/Ore Type	Graywacke	Graywacke
P wave velocity:	6010 mps	19718 fps

Table 3. Unit Charge Parameters – Sonic Case

Parameter	Calculated Unit Charge Values and Units	
Time to Free Face	0.33 MS	0.33 MS
Unit Charge Length	1.91 m	7.19 ft
Unit Charge Internal Area	0.60 m^2	6.44 ft^2
Unit Charge Volume	0.01 m^3	0.53 ft^3
Unit Charge Weight	17.95 kg	39.57 lbs
Unit Charge energy	51.19 MJ	51.19 MJ
Unit Charge Impulse Value	2.00 MN-s	450490 lbf-s

Component 2 – Thermodynamic or Heat Model

In addition to the unit charge model, a thermochemical or heat model is required, based on the critical or minimum diameter of an explosive, as well as an explosives ideal diameter (the maximum diameter in which the velocity of detonation remains constant). The heat model considers the amount of energy that is liberated by the detonation reaction determined by the percentage of ingredients reacting in the detonation head. The ratio of the square root of the detonation velocity at the critical or minimum diameter divided by the detonation velocity at the ideal diameter is defined as the volumetric extent of reaction. The detonation velocity - charge diameter curve is given in Figure 2.

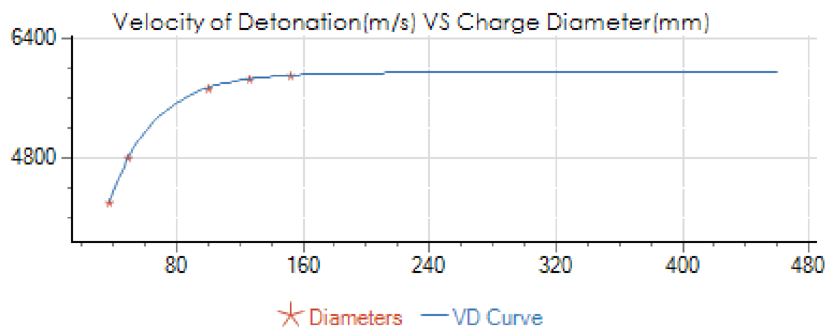


Figure 2. The detonation velocity - charge diameter curve for Subtek Intense.

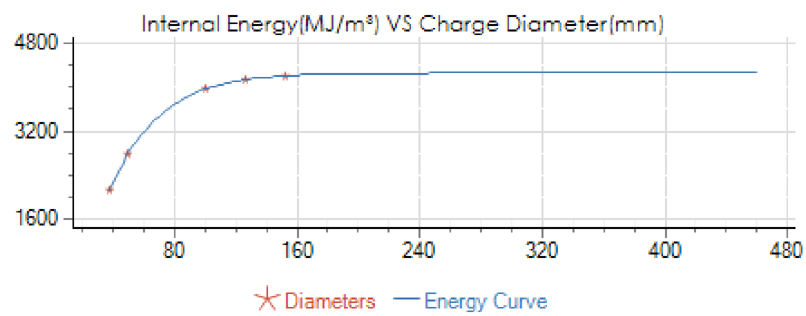


Figure 3. The heat variation with detonation velocity is included above.

Note from the above curve in figure 3 that maximum energy output occurs when the explosive is detonating at its ideal velocity, otherwise the energy will be reduced by the volumetric reaction extent at other diameters of charge less than the ideal.

Notes Concerning Component 2:
ASSUMPTION 1. The maximum energy generated by the detonation reaction will be at the ideal detonation velocity for any specific explosive that undergoes detonation at the ideal diameter of the explosive.
ASSUMPTION 2. The volumetric reaction extent is equal to the ratio of the critical velocity at the critical diameter of the explosive, divided by the detonation velocity measured in the ideal diameter of the explosive.

The third component of the break model includes a very important property for any rock/ore type. This is the decay ability of a solid medium to attenuate shock as it moves from blasthole to free face.

Component 3 - Stress Reflection Model

The stress reflection model considers the attenuation of shock pressure as it is transmitted from the wall of a borehole that contains the detonating explosive charge. At the borehole wall, the detonating pressure due to the impact of detonation shock, begins to decay exponentially according to the properties of the host rock/ore. The expanding shock wave front is compressive until it hits a less dense medium; at an open face it will be reflected back into the burden as a tensile wave. If the dynamic tensile strength is exceeded at the free face (rock/ore/air interface), scabbing back into the host rock/ore will occur.

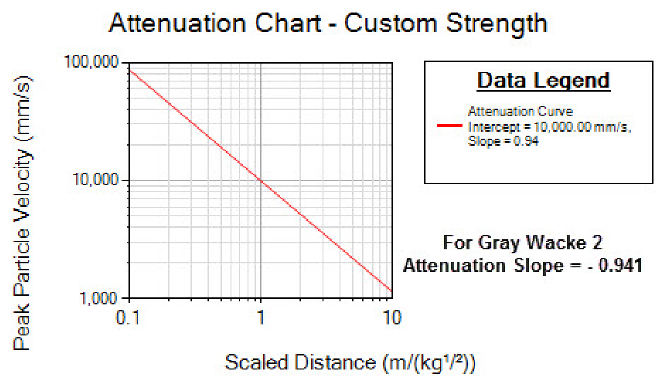


Figure 4. This graph represents the shock attenuation property for the rock/ore type Graywacke.

This part of the model requires the above attenuation graph (figure 4) that shows exponential decay of peak stress or peak particle velocity at radial distances away from a single charge detonating in a blasthole. Attenuation curves are obtained using either near field or far field monitoring using seismographs and sensors used for determining PPV versus scaled distance characteristics from regular blasting operations. This requirement indicates where shock is decayed slowly or quickly depending on the internal structure of a rock mass. As noted in the graph, the decay constant was -0.941.

Notes Concerning Component 3:

ASSUMPTION 1. The shock wave magnitude occurring at the borehole wall is derived from the well-known equation $P = \rho DW$ where ρ is equal to the explosive density, D is equal to the explosive detonation velocity and W is equal to the particle velocity of the explosive which is about 0.25D. P is detonation pressure.

ASSUMPTION 2. The impedance mismatch between a shock traveling though a rock/ore body as a P wave or sound wave and impacting a free face exposed to air is substantial. Shock is totally reflected in tensile mode

The next component of the break model is based on explosive energy coming from both the detonation state and explosion state regimes of a specific explosive product.

Component 4 - Radial Break Model

The convention of calculating powder factor using rectilinear geometries does not work for radial break. A single hole facing a rectilinear figure does not represent the true physical view of break from a geometry standpoint. In a borehole containing a detonating explosive, large cracks are generated well after shock transit, by hoop stress, but initially cracks are generated from shock from the detonation reaction. This shock produces a continuous rolling crack front from the borehole wall to a free face due to impedance mismatch and stress reflection . This process generates outbound cracks and upon reflection from the free face, extends them further, by changing mode from compressive to tensile. Powder factor is a poor term for defining the concentration of an explosive within a specific volume or rock/ore since explosives have different heat energies. Therefore, one kg (lb) of emulsion explosive does not equate to one kg (lb) of ANFO that has a completely different detonation velocity and heat energy. For the radial break model, explosive charges were confined away from any free face for the purpose of determining the break radius of the charge. Blasting patterns were set up to use pressure gauges for monitoring shock waves as well as observe gas expansion into cracks as shown in figures 5 and 6. In hole seismic transducers were used to obtain dynamic modulus and correlate shock attenuation with increasing distance from the blasthole.

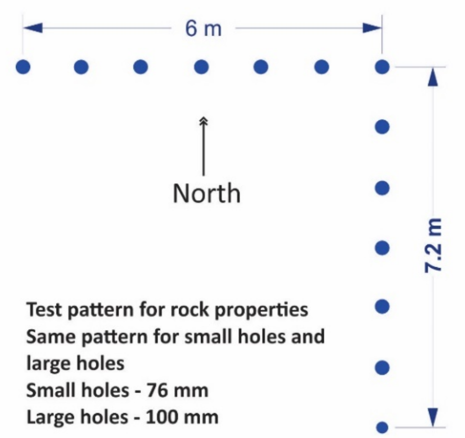


Figure 5. Blasting pattern for rock properties.

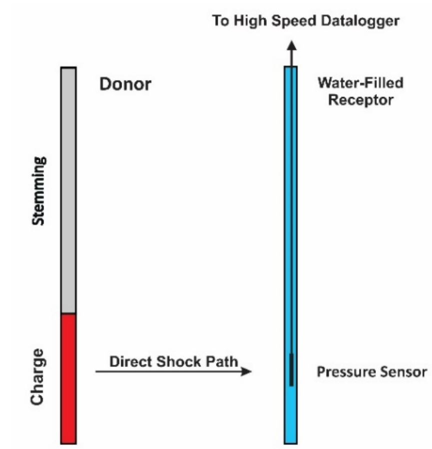


Figure 6. Instrumentation setup.

Notes Concerning Component 4:
ASSUMPTION 1. The radial break from a detonating blasthole, break will expand outwards radially as an expanding cylinder or ellipse.
ASSUMPTION 2. It was determined that the rebounding stress waves are hyperbolic in nature, outbound as a compressive wave front and inbound changing mode to a tensile wave front after reflection from an open face.
ASSUMPTION 3. For the shock wave attenuation, it is assumed that the reflective tensile portion from the open face may produce tensile scabbing at the free face based on the value of the dynamic tensile strength of the rock/ore medium as well as its impulse value. Shock magnitude is time dependent with maximum values at the borehole wall being steadily attenuated and stretched out traveling to the free face. The scabbing action will continue inward until the dynamic tensile stress in has decayed below the characteristic impulse value of dynamic tensile strength.

Using the Unit Charge Break Model as a Design Methodology

Eleonore mine in northern Quebec, a Goldcorp property, contracted BBA to develop a blasting design program that would not only provide a fragmentation profile limited to 400 mm (16 in) or less but also limit damage and dilution to less than 10%. The economics of the mine was based on the absence of an underground crusher. Underground testing provided rock/ore properties (for Graywacke including dynamic modulus, strength parameters and shock attenuation along with explosive detonation state properties for Orica's Subtek Intense pumpable emulsion. These properties are listed in table 4 below.

Table 4. Properties for Both Graywacke Ore and Subtek Intense Pumpable Emulsion

Rock/Ore Type - Graywacke			Explosive Type - Subtek Intense		
P wave velocity	6010 mps	19718 fps	Charge Diameter	100 mm	4.0 in
S wave velocity	3400 mps	11155 fps	Density	1.20 g/cm^3	1.20 SG
Bulk density	2.74 g/cm^3	171 lb/ft^3	Detonation Velocity	5723 mps	18778 fps
Unconfined compressive strength	101 MPa	14602 psi	Detonation Pressure	10.1 GPa	101 kBar
Dynamic compressive strength	252 MPa	36558 psi	Bulk Internal Energy	3957 MJ/m^3	26778 kcal/ft^3
Dynamic tensile strength	15 MPa	2237 psi	Volumetric reaction extent	0.94	0.94
Attenuation factor	-0.941	-0.941			

Previous blasting practices were based on using 100 mm (4 in) diameter blastholes and a blasting pattern spacing of 2.3 m (7.5 ft) and burden of 2.0 m (6.6 ft). Results were consistent with overbreak and dilution along with not meeting the < 400 mm (16 in) fragmentation profile specification. The natural response of the engineering department was to consider pulling the pattern in further with the realization that powder factors would increase. It was hoped that increasing the powder factor would solve the fragmentation concerns. When fragmentation is poor and found to be undesirable, the natural response of many engineering departments, is to increase the powder factor and thus the overall available explosive energy to mitigate problems. Alternatively, it has been proven that this action usually exacerbates blasting problems. To appeal to the Eleonore engineering department for devising an alternative blasting plan, it was necessary to provide a means of illustrating this particular blasting problem using the unit charge model and with in-situ testing that was conducted to acquire explosive and rock/ore properties listed in table 4 above.

Previous Blasting Practice Design Critique

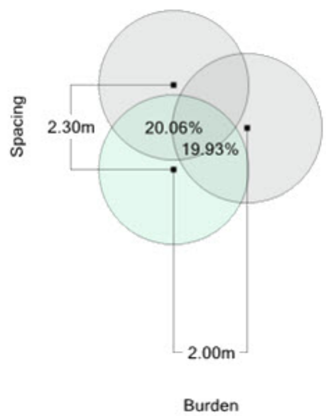
To determine the evaluation parameters concerning the past blasting pattern having a spacing of 2.3 m (7.5 ft) and burden 2.0 m (6.6 ft), the first step is to determine the radial break for a specific unit charge geometry. The radial break is calculated for multiple holes to determine the radial break overlap which is the measure of break coverage between holes to ensure that there are no gaps in energy distribution. There are five zones that are used to predict the radial break based on stress due to the shock from a detonating explosive. These are listed in Table 5. This table shows variances in tensile strength between -30% and +30% of the mean value. The bolded value of 2.08 m (6.8 ft) is the expected break for a 1.91 m (6.3 ft) charge that is completely confined in the Graywacke rock/ore type as determined for a pattern radial break based on both explosive and rock properties.

Table 5. Break Radii for Different Zones Away from the Blasthole for a Unit Charge of 1.90 m (6.3 ft)

Name	Units	-30%	-15%	Mean	15%	30%
Crush Extent Radius	m, ft	0.34, 1.12	0.30, 0.98	0.28, 0.92	0.26, 0.85	0.25, 0.82
Shatter Extent Radius	m, ft	0.58, 1.90	0.52, 1.71	0.48, 1.57	0.45, 1.48	0.42, 1.38
Minimum Extent Radius	m, ft	0.96, 3.15	0.87, 2.85	0.80, 2.62	0.74, 2.43	0.70, 2.30

Pattern Extent Radius	m, ft	2.49, 8.17	2.26, 7.41	2.08, 6.82	1.94, 6.36	1.82, 5.97
Maximum Extent Radius	m, ft	4.02, 13.19	3.65, 11.97	3.37. 11.05	3.14, 10.30	2.95, 9.68

The expected break overlap for the above spacing and burden dimensions of 2.3 m (7.5 ft) and 2.0 m (6.6 ft) is given in figure 7 (a). Figure 7 (b) is the results table for determining the range values to evaluate this tight pattern design.



(a)

Statistics	Range	Value	Gauge	Result
Break Overlap Burden	2% - 12%	19.9%		
Break Overlap Spacing	2% - 12%	20.1%		
Reflection Point	15.42 MPa - 252.06 MPa	312.77 MPa		
Ratio of Radial Break ...	45% - 75%	104.06%		
Break Angle	80° - 90°	87.61°		
Powder Factor Mass	0.4 kg/tonne - 0.8 kg/tonne	0.90 kg/tonne		
Energy Factor Mass	1.32 MJ/tonne - 2.64 MJ/tonne	2.98 MJ/tonne		

The red zone indicates a bad result. The yellow zone indicates that improvements can be made – results are within +/- 5% tolerance. The green zone confirms that a result falls within the acceptable range.

(b)

Figure 7 (a). Break overlap for the 2.3 m (7.5 ft) x 2.0 m (6.6 ft) pattern. Figure 7 (b). The results table is used to appraise a blasting design based on explosive and rock/ore properties and strength including attenuation factor. This pattern produce large muck as well as extensive overbreak with dilution.

Discussion

From figure 7 (a), note that the radial break overlap extends almost to the blastholes which indicates that there is too much energy located within a very small area and hence volume. This is supported in the results table indicating that the powder factor is 0.9 kg/tonne (1.8 lb/ton). This is too high even for underground powder factor standards. Underground blasting operations use powder factors that are typically twice open pit values because of oblique drill hole geometry and stope geometry. Figure 7 (b) provides a fit for only one range variable and that is the break angle of 87.6 degrees. All other parameters are out of range. Obviously, there is room for this blasting pattern to be improved.

It now becomes useful to determine the break characteristics of this tight blasting pattern using stress reflection and the hyperbolic break in the presence of a free face. Most, if not all mining professionals skilled in blast design acknowledge the fact that the propagating shock wave from a detonating blasthole will reflect at an open face changing mode from compression to tension. The reflection point for the tight blasting pattern is shown below in figure 8.

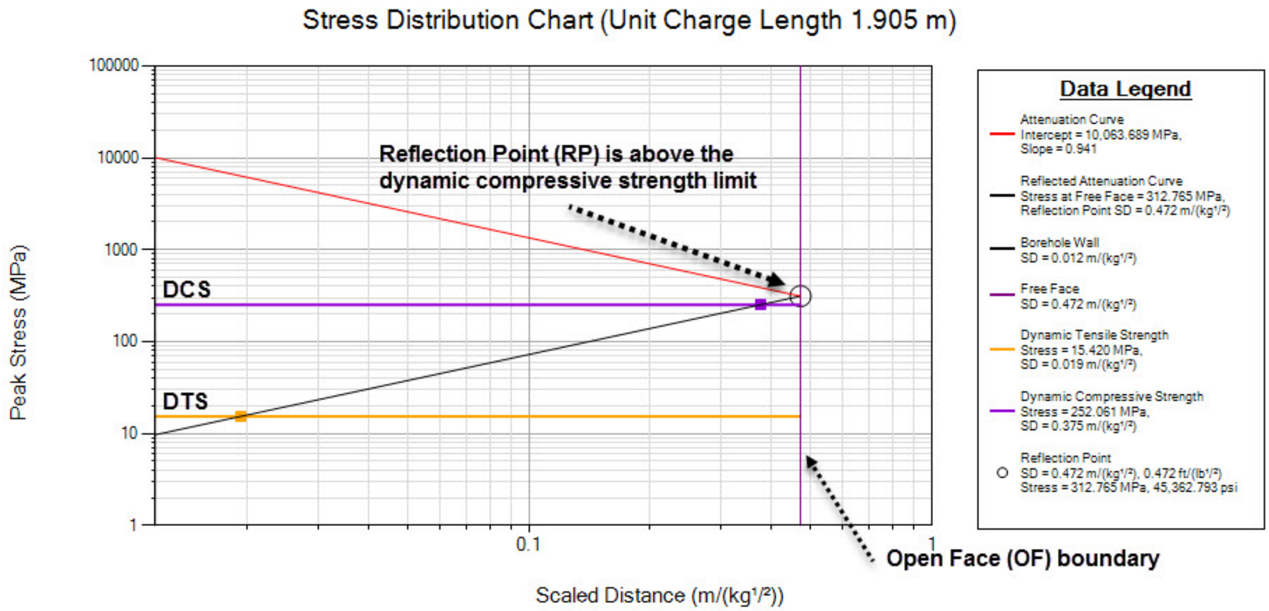


Figure 8. The stress attenuation chart decays detonation shock (red line) from blasthole to the open face (OF) using an attenuation factor of -0.941. The reflection point (RP) is above the dynamic compressive strength limit (DCS). The RP should be between the dynamic tensile strength limit (DTS) and the DCS limit. This region is called the 'sweet spot'.

The calculated results for the hyperbolic break are presented in table 6. The important parameters are bolded.

Table 6. Hyperbolic Break Parameters for the 2.3 m (7.5 ft) x 2.0 m (6.6 ft) Pattern Using a Unit Charge of 1.91 m (6.2 ft)

Parameter	Value	
Radial Break Distance	**2.08 m**	**6.83 ft**
Breakout Angle	**87.61 °**	**87.61 °**
Hyperbolic Area	3.81 m^2	41.03 ft^2
Hyperbolic Volume	7.26 m^3	256.39 ft^3
Hyperbolic Tonnage	19.89 tonne	21.93 ton
Hyperbolic Powder Factor - Volume	2.47 kg/m^3	0.15 lb/ft^3
Hyperbolic Powder Factor - Mass	**0.90 kg/tonne**	**1.8 lbs/ton**
Hyperbolic Energy Factor - Volume	8.15 MJ/m^3	55.18 kcal/ft^3
Hyperbolic Energy Factor - Mass	**2.98 MJ/tonne**	**643.15 kcal/ton**

The hyperbolic break is illustrated in figure 9. Note the position of the radial break that extends past the burden distance. This indicates that the gas expansion (thermochemical borehole pressure) will have very little work to do since the shock stress is so high. Gas expansion should be about one-half times the burden distance. If the burden is too light, the next ring to detonate may choke causing extensive backbreak.

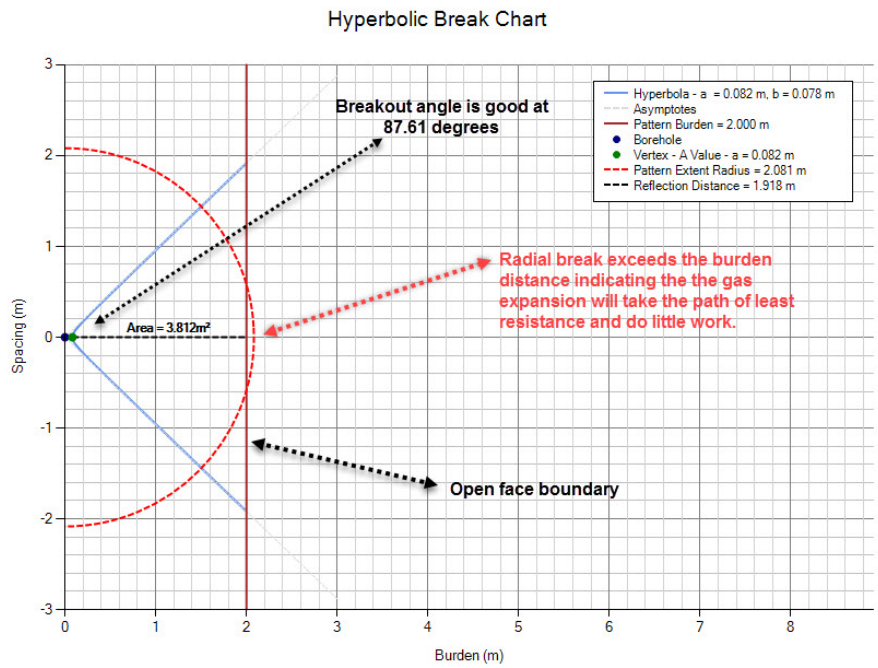


Figure 9. The radial break extends past the open face which is a sure sign that the 2.3 m (7.5 ft) x 2.0 m (6.6 ft) pattern was too tight. There are actually three possible solutions – 1) change the explosive, 2) change the hole diameter and 3) expand the pattern. The third alternative was chosen.

Damage to screen and bolts as well as hanging-walls and foot-walls would be expected. The engineering group decided on expanding the pattern to 2.8 m (9.2 ft) x 2.5 m (8.2 ft). A picture showing the damage caused by the 2.3 m (7.5 ft) x 2.0 m (6.6 ft) pattern is shown in figure 10.

Figure 10. This picture shows the damage produced by the tight pattern (2.3 m, 7.5 ft x 2.0 m, 6.6 ft).

Suggested Pattern Improvement Using the Unit Charge Break Model

Moving to the new pattern of 2.8 m (9.2 ft) x 2.5 m (8.2 ft), table 7 was generated using the new unit charge calculations;

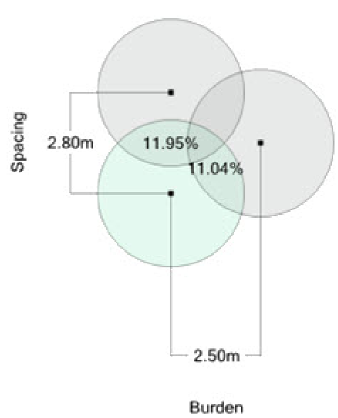
Table 7. Inputs for the Expanded Blasting Pattern of 2.8 m (9.2 ft) Spacing and 2.5 m (8.2 ft) Burden

Parameter	Input Values and Units	
Drill Hole Diameter:	100mm	4.00 in
Spacing Distance:	2.8 m	9.18 ft
Burden Distance:	2.5 m	8.20 ft
Explosive Product:	Subtek Intense	Subtek Intense
Detonation Velocity:	5723 mps	18778 fps
Rock/Ore Type	Graywacke	Graywacke
P wave velocity:	6010 mps	19718 fps

Table 8. Unit Charge Parameters – Sonic Case for the Expanded Pattern

Parameter	Input or Calculated Result	
Time to Free Face	0.42 ms	0.42 ms
Unit Charge Length	2.38 m	7.81 ft
Unit Charge Internal Area	0.75 m^2	8.05 ft^2
Unit Charge Volume	0.02 m^3	0.66 ft^3
Unit Charge Weight	22.44 kg	49.47 lbs
Unit Charge energy	73.99 MJ	73.99 MJ
Unit Charge Impulse Value	3.13 MN-s	703891 lbf-s

Since there were no changes to the explosive product, hole diameter as well as rock/ore type, strengths and attenuation, the break radius value of table 5 was used again in the analysis. It was decided to expand the pattern to 2.8 m (9.2 ft) spacing and 2.5 m (8.2 ft) burden. This time the break overlap and results table produced much better range values as shown in figure 11 (a) and figure 11 (b) respectively.



Statistics	Range	Value	Gauge	Result
Break Overlap Burden	2% - 12%	11.0%		
Break Overlap Spacing	2% - 12%	11.9%		
Reflection Point	15.42 MPa - 252.06 MPa	253.53 MPa		
Ratio of Radial Break ...	45% - 75%	83.25%		
Break Angle	80° - 90°	87.00°		
Powder Factor Mass	0.4 kg/tonne - 0.8 kg/tonne	0.59 kg/tonne		
Energy Factor Mass	1.32 MJ/tonne - 2.64 MJ/tonne	1.93 MJ/tonne		

The red zone indicates a bad result. The yellow zone indicates that improvements can be made – results are within +/- 5% tolerance. The green zone confirms that a result falls within the acceptable range.

(a) **(b)**

Figure 11 (a). Break overlap for the 2.8 m (9.2 ft) x 2.5 m (8.2 ft) pattern. Figure 11 (b). The results table was used to appraise a blasting design based on explosive and rock/ore properties and strength including the attenuation factor for the Graywacke rock/ore type. Powder factor has

been reduced to 0.59 kg/tonne (1.18 lb/ton). Energy factor has been reduced to 1.93 MJ/tonne (1.75 MJ/ton).

From figure 11 (b), there is improvement shown. The reflection point is just within tolerance whereas the radial break ratio is still out of range. The stress reflection characteristics are presented in figure 12.

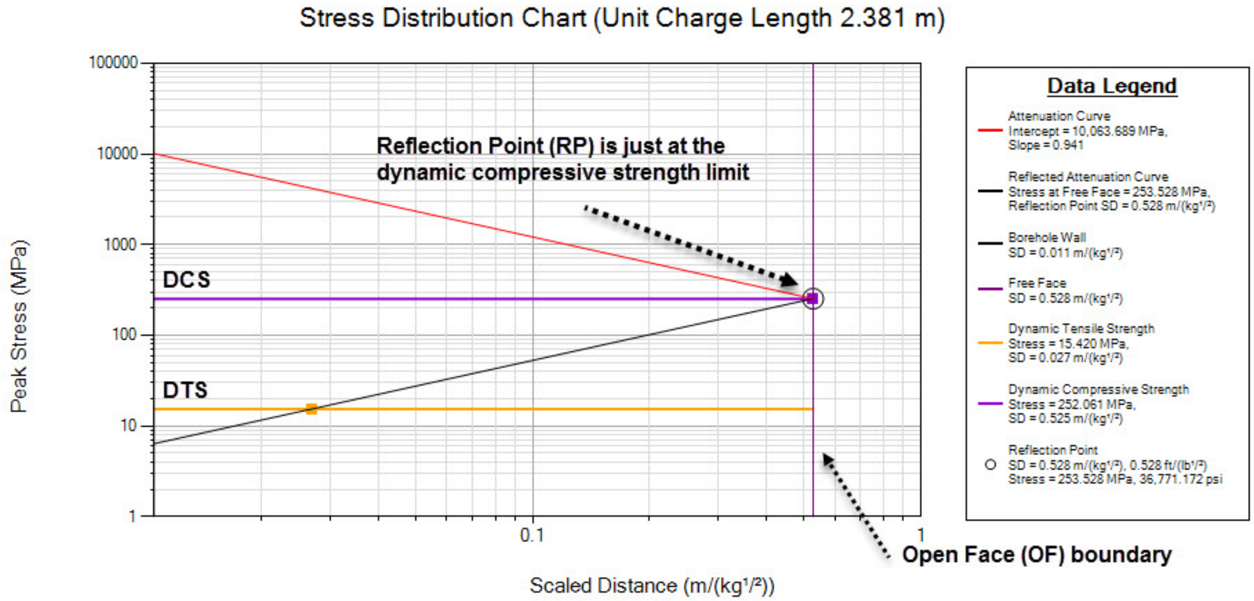


Figure 12. The stress reflection characteristics of the new pattern 2.8 m (9.2 ft) x 2.5 m (8.2 ft).

Note that the reflection point (RP), is almost at the level of the dynamic compressive strength, indicating that the pattern might be able to be expanded slightly further. The radial break limit should be between around one-half the burden distance (within a tolerance of +/- 5%). Also observe that the unit charge length has increased to 2.381 m (7.8 ft) in table 8 because of the increase in burden distance. Table 9 shows the results for the expanded pattern.

Table 9. Hyperbolic Break Parameters for the 2.8 m (9.2 ft) x 2.5 m (8.2 ft) Pattern Using a Unit Charge of 2.38 m (7.8 ft) - Burden Distance has Increased

Parameter	Values	
Radial Break Distance	**2.08 m**	**6.83 ft**
Breakout Angle	**87.00 °**	**87.00 °**
Hyperbolic Area	5.87 m²	63.23 ft²
Hyperbolic Volume	13.99 m³	493.89 ft³
Hyperbolic Tonnage	38.32 tonne	42.24 ton
Hyperbolic Powder Factor - Volume	1.60 kg/m³	0.10 lb/ft³
Hyperbolic Powder Factor - Mass	**0.59 kg/tonne**	**1.17 lbs/ton**
Hyperbolic Energy Factor - Volume	5.29 MJ/m³	35.80 kcal/ft³
Hyperbolic Energy Factor - Mass	**1.93 MJ/tonne**	**418.62 kcal/ton**

The hyperbolic break is shown in figure 13 on the following page, illustrating the reduction in radial break but still retaining a break angle of 87 degrees.

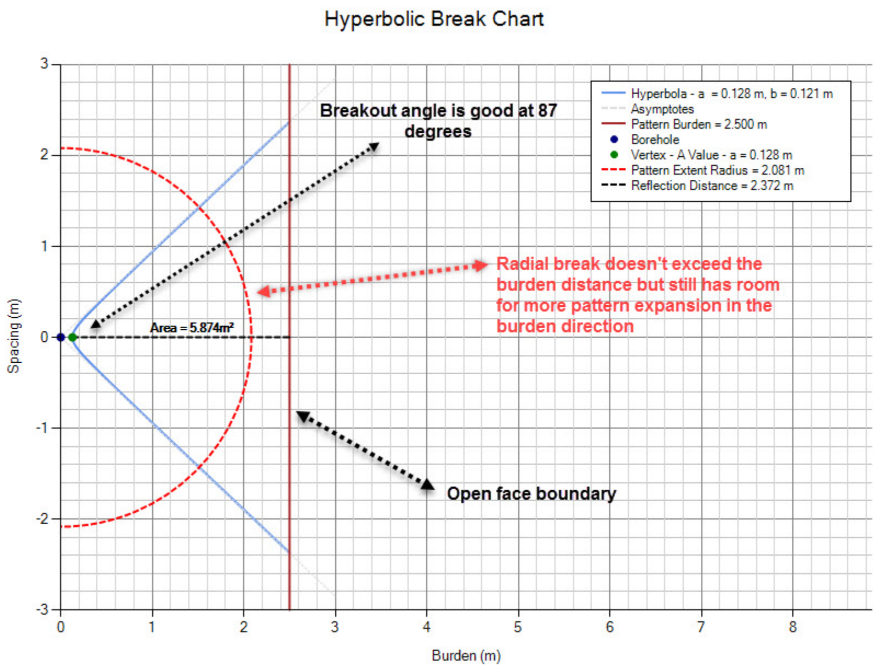


Figure 13. This graph shows the position of the new radial break along with the breakout angle of 87 degrees. For best design, the radial break should be about one-half the burden distance +/- 5%.

Figure 14. Improvements made to the narrow vein stope at Eleonore mine in Quebec. Note the reduction in stope damage with hanging wall intact. The fragmentation specification of muck < 400 mm (16 in) was also met. Dilution was drastically reduced to less than the original specification of 10%.

Figure 14 above clearly shows that improvement. Note the keyhole shape of the topsill and ore vein.

Overbreak was eliminated with much better recovery along with the following statement that;

"Given that the mine does not have a primary crusher underground, the fragmentation had to be less than 400mm, the opening dimensions of their grizzly. As a result of these steps, the blast fragmentation was impeccable throughout the mucking process. The stope was excavated under the barrier of the coveted 10 per cent dilution." – Daniel Roy, BBA Drilling and Blasting Engineering Expert (Roy, 2018).

Timing and Sequencing

Electronic detonator timing was derived from measured P wave velocities for these specific rock masses including S wave and estimated crack velocities under typical powder factors/energies representative of underground mining operations. Table 10 lists the properties used along with travel times;

Table 10. Delay Calculations Based on an Equation for Dynamic Motion

Rock/Ore Property	Velocity		Transit Time	
P wave Velocity	6010 mps,	19718 fps	0.166×10^{-3} spm ,	0.051×10^{-3} spf
S wave Velocity	3400 mps,	11155 fps	0.294×10^{-3} spm ,	0.089×10^{-3} spf
Crack Velocity	2160 mps,	7085 fps	0.463×10^{-3} spm ,	0.141×10^{-3} spf
Burden Movement Velocity at 30% detached	55 mps,	180 fps	18.182×10^{-3} spm ,	5.556×10^{-3} spf

*spm – seconds per meter. *spf – seconds per foot

Delays were adjusted between holes and rows according to the timing constraints presented in table 11. From high speed cameras used in open pit face movement monitoring, burden movement velocities range from 15.2 mps (50 fps) to 30.5 mps (100 fps) roughly at 30% detachment. Little work has been done in underground blasting regarding the measurement of face movement with the exception of development. Since powder factors for underground mining are typically double those for surface mining, an assumption was made that the burden movement velocities would also double. The delay times for the inclusion of dynamic events was determined from the following equation 1 (for single holes detonated). Between rings timing was 3 times this value or roughly 65 ms;

$$T_d := T_{uc} \cdot \frac{E_{cl}}{C_{ucl}} + \sqrt{\left(\frac{B_p}{P_w}\right)^2 + \left(\frac{B_p}{S_w}\right)^2 + \left(\frac{S_p + B_p}{C_v}\right)^2} + N \cdot \left(\frac{B_p}{G_v}\right) = 21.526\ ms$$

…. Equation 1

Table 11. Parameters Used to Calculate Delays Between Holes.

Parameter	Symbol	Input Values and Units	
Length of unit charge	C_{ucl}	2.38 m	7.80 ft
Detonation time of unit charge	T_{uc}	0.42 ms	0.42 ms
Length of explosive column	E_{cl}	30 m	98.42 ft
Pattern Spacing	S_p	2.8 m	9.19 ft
Pattern Burden	B_p	2.5 m	8.20 ft
P wave velocity	P_w	6010 mps	19717 fps
S wave velocity	S_w	3400 mps	11155 fps
Detonation velocity	D_e	5723 mps	18776 fps
Crack velocity	C_v	2160 mps	7087 fps
Gas expansion velocity	G_v	55 mps	180.fps
Burden Percent Detached	N	30%	30%
Time delay for this hole	T_d	21.53 ms	21.53 ms

Normally, long holes are found near the center of the ring so the delay time for the longest hole would be used throughout the ring with 65 ms used between rings.

Summary for Blast Using the New Pattern

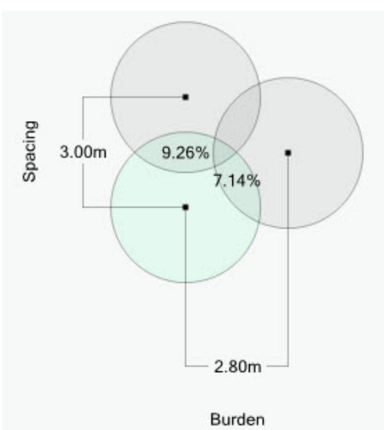
The use of the unit charge break model was successful in terms of comparing an explosive product and rock/ore type pair for the purpose of defining hyperbolic break using explosive and rock/ore properties along with shock attenuation. Table 12 shows the results from a blasting operation which was summarized with the hanging-wall intact. The complete stope was recovered in two production blasting operations.

Table 12. Stope Statistics Using Expanded Pattern – 2.8 m (9.2 ft) Spacing by 2.5 m (8.2 ft) Burden

Parameter	Value
Workplace	650-5050-251
CMS	Front and Back
Blast	Final
Date of last blast	30 July 2017
Planned tonnage	15699 tonnes
Planned dilution tonnage	17269 tonnes
Tonnage recovered	15668 tonnes
Tonnage remaining	0 tonnes – 100% recovery
Hanging-wall condition	Hanging-wall undamaged

Recommendations for Future Work

There have been discussions of improving the current 2.8 m (9.2 ft) x 2.5 m (8.2 ft) design further. Some preliminary work has been completed to expand the pattern further to 3.0 m (9.8 ft) x 2.8 m (9.2 ft). The break analyzer software was run again for these new pattern dimensions giving the result below;



(a)

Statistics	Range	Value	Parameter Gauge	Result
Break Overlap Burden	2% - 12%	7.1%		✓
Break Overlap Spacing	2% - 12%	9.3%		✓
Reflection Point	15.42 MPa - 252.06 MPa	227.88 MPa		✓
Ratio of Radial Break ...	45% - 75%	74.33%		✓
Break Angle	80° - 90°	86.63°		✓
Powder Factor Mass	0.4 kg/tonne - 0.8 kg/tonne	0.47 kg/tonne		✓
Energy Factor Mass	1.32 MJ/tonne - 2.64 MJ/tonne	1.55 MJ/tonne		✓

The red zone indicates a bad result. The green zone confirms that a result falls within the acceptable range. From the above chart, all criterion are within range.

(b)

Figure 15 (a). Break overlap for the 3.0 m (9.8 ft) x 2.8 m (9.2 ft) pattern. Figure 15 (b). The results table was used to appraise an additional blasting design based on explosive and rock/ore properties and strength including the attenuation factor for the Graywacke rock/ore type. Powder factor would be reduced further to 0.47 kg/tonne (0.94 lbs/ton). Energy factor would be reduced to 1.55 MJ/tonne (1.41 MJ/ton).

A summary table shown in table 13 below presents the possible additional improvement that could be realized if the blasting pattern was increased to 3.0 m (9.8 ft) spacing and 2.8 m (9.2 ft) burden. Important parameters are bolded;

Table 13. Hyperbolic Break Parameters for the 3.0 m (9.8 ft) x 2.8 m (9.2 ft) Pattern Using a Unit Charge of 2.67 m (8.8 ft) - Burden Distance has Increased

Parameter	Values	
Radial Break Distance	**2.08 m**	**6.83 ft**
Breakout Angle	**86.63 °**	**86.63 °**
Hyperbolic Area	7.31 m^2	78.64 ft^2
Hyperbolic Volume	**19.48 m^3**	**687.97 ft^3**
Hyperbolic Tonnage	**53.38 tonne**	**58.84 ton**
Hyperbolic Powder Factor - Volume	1.29 kg/m^3	0.08 lb/ft^3
Hyperbolic Powder Factor - Mass	**0.47 kg/tonne**	**0.94 lbs/ton**
Hyperbolic Energy Factor - Volume	4.25 MJ/m^3	28.79 kcal/ft^3
Hyperbolic Energy Factor - Mass	**1.5 MJ/tonne**	**336.59 kcal/ton**

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