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## 6. Applying a Blast Load

### 6.1. Introduction

One of the most important aspects of building a numerical model to estimate the effects of blasting on paste fill is to ensure that a representative blast load is applied to the model. A number of methods of applying a blast load are available. The most suitable method of applying a blast load to a numerical model is dependent on the purpose of the model. For this project, it is necessary that the blast loading function is representative of loads applied to the blast hole walls during a blast, and is numerically efficient. The different methods of applying blast loads available in ABAQUS/Explicit and the method chosen for this project are discussed in this chapter.

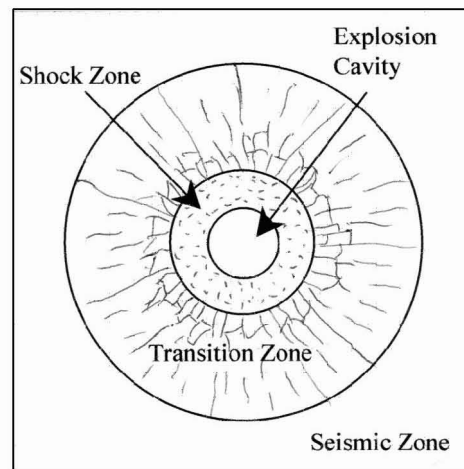
### 6.2. What Happens When an Explosive Detonates?

When an explosive detonates in a blast hole in rock, the chemical reaction produces a gas at a very high temperature and pressure. This gas exerts a very high pressure on the blast hole walls, pushing the walls outwards and shattering the rock surrounding the blast hole. The high pressure sends a stress wave through the rock, which expands radially from the blast hole. The tangential stress from this wave causes radial cracks to occur around the blast hole. The gases then expand into the cracks surrounding the blast hole, opening up the cracks and reducing the pressure of the gas. The response of the material surrounding the blast hole to loading from the detonation of an explosion can be categorized into the following three zones (Brady and Brown, 1993):

1. Shock zone or crushed zone: In the immediate vicinity of the blast hole, the rock behaves mechanically as a viscous solid and the stress wave causes the material to be crushed or extensively cracked. The radius of the shock zone is approximately twice the radius of the blast hole.
2. Transition zone: The transition zone is immediately outside the shock zone. The rock behaves as a non-linear elastic solid subject to large strain in this region. New cracks are initiated in the radial direction in this zone by interaction of the stress wave with the crack population. The radius of this zone is approximately 4 to 6 times the radius of the blast hole.

3. Seismic Zone: Beyond the transition zone, the rock behaves linearly elastically and rock behaviour can adequately be explained by elastic fracture mechanics theory. Crack propagation in this region occurs by extension of the longest cracks in the transition zone.

These zones are shown in Figure 6.1. During blasting in the mine, the paste fill is always located within the seismic zone and never within the shock or transition zone. Therefore the interest in the numerical modelling is mainly with the seismic zone. This can be clearly seen from the field monitoring discussed in Chapter 3. The production blasts monitored consisted of 89 mm diameter boreholes, with a transition zone radius of 0.18 to 0.27 m. The nearest borehole was located 2 m from the paste fill.



**Figure 6.1 – Fragmentation Zones Around a Blast Hole in Rock**

If the stress wave encounters a free boundary, the compressional wave is reflected back as a tensional wave, and cracking known as spalling may occur at the boundary if the tensile stress of the wave is larger than the tensile strength of the rock (Atchison 1968).

As these two energy sources, shock wave and gas penetration, occur both simultaneously and almost instantly, it is difficult to determine which explosive effects occur from each source. A look at the literature shows that authors are divided over which energy source has the greatest effect, and shows that in order to accurately model the damage to the blast hole walls, both effects must be considered. Sarracino and Brinkmann (1990) compared the damage caused by explosives enclosed in metal tubes to prevent gas penetration and normal blast holes. The acceleration and strain measurements showed that the character of shock transmitted from both were the same. Although there is some disagreement between researchers over the effects of the shock wave and the gas penetration, many authors agree that the shock wave stresses and causes acceleration to the surrounding material, while

the majority of fracturing is caused by gases penetrating into crack surrounding the blast hole (Sarracino and Brinkmann, 1990; Daehnke et al., 1996; Liu and Katsabanis, 1997).

The gases produced during the explosion provide the load to the surrounding rock. During detonation, the gases are produced quickly, and so the load would be expected to increase from zero to the maximum load over a very short period of time. When the rock immediately surrounding the blast hole is shattered, the gas expands into the new volume available, reducing the pressure and therefore the blast load. The gas pressure continues to reduce as the gas expands into the cracks produced by the shock waves. This reduction in pressure would be gradual due to the time required for the gas to expand into the cracks. Therefore, the expected shape of the load curve relative to time is a sudden, almost instantaneous increase from zero to maximum load, followed by a gradual decrease.

### ***6.3. Measuring Blast Damage***

In order to model the effect of blasting on paste fill, a measure of blast damage is required. When an explosive detonates, stress waves are generated and travel through the solid, causing ground vibration. This ground vibration causes the particles in the rock mass to run through an elliptical motion, and the highest velocity experienced by the particle is known as the peak particle velocity (ppv). Persson et al. (1994) were able to make reliable predictions of rock damage based on the vibration particle velocities in the rock. Field monitoring results presented in chapter 3 indicate that damage is observed in paste fill when peak particle velocities of the order of 2.5 m/s are experienced.

### ***6.4. Prediction of Peak Particle Velocity***

As discussed in Section 2.3, when an explosive detonates, stress waves are generated and travel through the solid, causing ground vibration. This ground vibration causes the particles in the rock mass to run through an elliptical motion, and the highest velocity experienced by the particle is known as the peak particle velocity (Holmberg and Persson, 1979). The peak particle velocity decreases with distance from the charge, and can be predicted by equation 2.3 for a spherical explosive.

However, explosives are commonly loaded into blast holes, as shown in Figure 2.3. In this figure, the bottom of the blast hole is filled with explosive, and the blast hole is 'stemmed' with an inert material. Holmberg and Persson (1979) modified equation 2.3 to predict the peak particle velocity for

blast waves resulting from columns of explosive, resulting in equation 2.5 which is valid when  $\beta$  is equal to  $2\alpha$ , which is often seen in rock.

As discussed in Section 2.3, Sartor (1999) used this equation to model the vibration induced damage from blasting in rock at Cannington Mine. The constants in the equation were found to be:  $k = 2938$ ,  $\alpha = \frac{1}{2}\beta = 0.66$ . Field tests conducted in paste fill described in Chapter 4 were used to develop a site specific equation for the paste fill at Cannington Mine. The constants were found to be  $k = 1000$  and  $\beta = 1.02$ .

Equation 2.5 can be used to determine the peak particle velocity a loading function should produce at various distances from the charge. This can be used to validate the loading function. The average linear charge density and blast hole length for a set of mass blasts monitored at Cannington Mine is given below in Table 6.1.

**Table 6.1 Blast Parameters for Stope 4760**

<b>Blast Number</b>	<b>Average Blast Hole Length (m)</b>	<b>Average Linear Charge Density (kg/m)</b>
904061	15.7	6.217
904064	2.0	6.223
904065	15.5	6.213
904066	14.7	6.212
904068	20.6	5.563
904072	20.0	5.367
904077	20.1	8.611
904078	20.7	6.300

## ***6.5. How Can Blast Loads be Applied in ABAQUS/Explicit***

A literature search including the ABAQUS (2003) manuals revealed the following four methods of applying blast loads in numerical modelling, which are discussed in detail in the following sections:

- Applying a time varying pressure to the walls of the blast hole column;
- Applying a time varying pressure to the walls of the zone in which cracks occur during a blast;
- Applying an incident wave field; and
- Modelling the detonation of the explosive using the Jones-Wilkens-Lee (JWL) Equation of State (EOS) material model.

### **6.5.1. Applying a Time Varying Pressure to the Walls of the Blast Hole**

Perhaps the simplest method of applying a blast load in the numerical model is to apply a pressure to the walls of the blast hole column. IN ABAQUS, the pressure can be applied as a surface load using the \*DSLOAD command in conjunction with the \*AMPLITUDE function to vary the pressure with time. Since the pressure is applied to the walls of the blast hole, the cracking region adjacent to the blast hole must be modelled. This results in the use of a complex material model and cracking model which causes the model to take much longer to solve. This method is appropriate for situations where the cracking pattern or damage adjacent to the blast hole is required, but is not practical for use in a large scale model due to the time required to solve such a model. An example where this method has been used is seen in the work by Grady and Kipp (1980) who used a combination of experimental work and numerical modelling to develop a fracture model for rock incorporating both rate sensitive features of the fracture process and material wave propagation characteristics.

### **6.5.2. Applying a Time Varying Pressure to the Walls of an “Equivalent Cavity”**

This method involves applying the blast load as a time varying pressure using the \*DSLOAD and \*AMPLITUDE commands as discussed above, however the load is applied at a boundary beyond the cracking zone of the blast hole. This enables simpler material models to be used for the analysis which reduced the computation time for the model. Examples where this method has been used include Jiang et al. (1995), Minchinton and Lynch (1996), Blair and Minchinton (1996), Blair and Jiang (1995) and Potyondy et al. (1996). The applications of these models included studying the

damage zone around a blast hole (Blair and Minchinton 1996) and studying surface vibrations due to a vertical column of explosive (Blair and Jiang 1995).

### 6.5.3. Incident Wave Field

The \*INCIDENT WAVE option is used in ABAQUS to apply incident wave loads such as those experienced due to an underwater explosion or a blast in air. This method is designed to model loads due to external wave sources, which occur in a fluid external to the structure of interest. This fluid is often air or water. Different methods are available depending on whether the user wishes to model the wave in the fluid and the structure, or the wave only in the structure (ABAQUS 2003). Since blasting in an underground mine involves explosives detonated in the rock mass, and not in a fluid, the \*INCIDENT WAVE option was not considered a feasible option for loading the numerical model. Applications of this method include modelling the impact of air blasts or underwater explosions on nearby structures.

### 6.5.4. Modelling the Chemical Reaction using the Jones-Wilkens-Lee Equation of State Material Model

Jones-Wilkens-Lee (JWL) equation of state material model (Lee et al., 1973) is used to model the detonation of non-ideal detonation. It was developed to allow fitting to pressure data obtained using the cylinder expansion test, and has been used in the design of explosive devices and in the modelling of non-ideal detonations (Persson et al., 1994).

This method involves modelling the generation of pressure as the explosive detonates by assigning the JWL material model to the explosive elements and defining detonation points in the explosive. The material model is defined using the \*DENSITY, \*EOS, TYPE=JWL and \*DETONTATION POINT keywords. The JWL equation of state used in ABAQUS can be written in terms of the initial energy per unit mass as follows (ABAQUS 2003):

$$p = J \left( 1 - \frac{\Theta \rho}{\Psi_1 \rho_0} \right) e^{-\Psi_1 \frac{\rho}{\rho_0}} + B \left( 1 - \frac{\Theta \rho}{\Psi_2 \rho_0} \right) e^{-\Psi_2 \frac{\rho}{\rho_0}} + \frac{\Theta \rho^2}{\rho_0} e_{m_0} \quad (6.1)$$

where  $J$ ,  $B$ ,  $\Psi_1$ ,  $\Psi_2$  and  $\Theta$  are material constants

$e_{m_0}$  is the initial energy per mass unit

$\rho_0$  is the density of the explosive

$\rho$  is the density of the detonation products

The arrival time of the detonation wave at a material point is calculated as the distance from the material point to the nearest detonation point divided by the detonation wave speed, and takes into account the detonation delay of the given detonation point (ABAQUS 2003).

This method was used by Yang et al. (1996), Liu and Katsabanis (1996) and Thorne et al. (1990). Applications included the development of constitutive models for blast damage (Yange et al. 1996, Liu and Katsabanis 1996) and investigation of the fundamental mechanisms of cratering (Thorne et al. 1990).

### **6.5.5. Discussion of Blast Loading Methods**

The advantages and disadvantages of the methods discussed above are given in Table 6.2. The method of blast loading to choose for any project depends on the purpose of the modelling and the desired results. The purpose of this project is to assess the effects of blast loads on nearby paste fill material. The blast holes are located in the rock, and the paste fill material is located near by but outside the cracking zone of the blast hole. The lack of symmetry in the region of interest requires the full paste fill stope to be included in the model, resulting in large scale models in 3-dimensions. Due to the large scale of the model, a blast loading method that is numerically efficient is required to ensure that the model can be solved within a reasonable time frame. The results of interest are the velocities experienced in the nearby paste fill as a result of the blast. Therefore, the blast loading method for this project must provide reliable results in the paste fill at a distance from the blast source, but reliable results within the cracked zone are not necessary. The method of applying a pressure to the walls of the zone in which cracks occur (see section 6.5.2) was found to satisfy these requirements.

### **6.6. Concept of an “Equivalent Cavity”**

A cylindrical “equivalent cavity” was used in order to model only the seismic zone, in which elastic wave propagation is expected to occur. This method avoids modelling the region where new cracks in the material are initiated from the shock wave during an explosion. The concept of an “equivalent cavity” was first proposed by Sharpe (1942). Sharpe’s work, and work by other authors (Kutter and Farihurst, 1971; Blair and Jiang, 1995) have shown that the use of an “equivalent cavity” gives reasonable agreement with field measurements. When this concept is used, the blast pressure is applied to the walls of the “equivalent cavity” instead of the walls of the blast hole. The radius of the



**Table 6.2 Advantages and Disadvantages of Methods of Applying a Blast Load**

<b>Method</b>	<b>Advantages</b>	<b>Disadvantages</b>	<b>Typical Use</b>
Applying a time varying pressure to the walls of the blast hole column (see section 6.5.1)	Accurately models the effects of a blast load to the rock mass surrounding the blast hole, including cracking	Numerically expensive due to complex cracking models and small mesh sizes required.	Generally used for modelling the cracking in the region immediately surrounding the blast hole. Not generally used for large scale models.
Applying a time varying pressure to the walls of the zone in which cracks occur during a blast (see section 6.5.2)	Accurately models the effects of a blast load to the rock mass surrounding the cracked region  Numerically efficient	Does not provide accurate model results in the cracked region surrounding the blast hole	Used to model the transmission of the shock wave through the material
Applying an incident wave field (see section 6.5.3)	Models the loading on structures due to explosives blasts in a “fluid” external to the structure	Not applicable for cases where the blast occurs within the structure of interest.	Generally used for loading due to underwater explosions on structures or airblast loading on structures
Model the chemical reaction in a blast hole using the JWL Equation of State Material Model (see section 6.5.4)	Accurately models the explosion in a blast hole and the loading on the rock mass surrounding the blast hole, including cracking	Numerically expensive due to complex reactions in the blast hole and complex cracking model and small mesh sizes required.	Generally used for modelling the cracking in the region immediately surrounding the blast hole. Not generally used for large scale models.

cavity is set a distance from the blast hole wall that ensures that the strength of the material is greater than the stresses experienced from the shock wave, which is typically thought to occur between 3 and 9 times the blast hole radius (Kutter and Fairhurst, 1971). A value of 3 times the blast hole radius was used in this dissertation. This will result in the load being applied in the transition zone shown in Figure 6.1.

The method of using an equivalent cavity allows the behaviour of the material in the seismic zone to be modelled without the need to model the cracking that occurs around the blast hole. For large scale models such as those used in this work, this method dramatically reduces the solving time of the models, as complex material models and cracking models are not required. As the paste fill is not in the area immediately surrounding the blast hole where cracks are expected to occur, the development of the cracks surrounding the blast hole does not need to be modelled. It is the shockwave travelling through the ore and into the paste fill which causes the damage to the paste fill and threatens the stability of the stope. In ABAQUS, if a sufficient load is applied to the model, the program will model the transmission of the shock wave through the material. Hence, an initial load is required that will transmit an equivalent shock wave through the model as that measured in the field. Since the purpose of this work is to model the behaviour of paste fill rather than the behaviour of the rock in the immediate vicinity of the blast hole, the use of an equivalent cavity is appropriate. However, the use of an equivalent cavity would not be suitable if the behaviour of the rock in the vicinity of the blast hole was required.

### ***6.7. Explosives Being Modelled***

The loading function applied to the numerical model will be dependent on the explosive used in the given application. Emulsion type explosives were used in both the field tests and the production blasts at Cannington Mine. In the test blasts in paste fill, cartridges of the explosive Powergel Powerfrag were used, and in the production blasting that was monitored, the explosive Powerbulk VE, at a density of approximately  $1.0 \text{ g/cm}^3$  was used. Both explosives were supplied by Orica. Information about these explosives is given in the Tables 3.1 (Powerbulk VE) and 4.2 (Powergel Powerfrag).

Powergel Powerfrag is a high strength, detonator sensitive packaged explosive. It is designed for priming applications, such as the initiation of explosive columns, and for use as a medium density column explosive in mining and general explosive work. Powergel Powerfrag is supplied in cartridges. Powerbulk VE is a primer sensitive bulk emulsion explosive that has been designed for

used in underground blasting applications. Powerbulk VE is a fluid with a viscosity similar to that of heavy grease. It is pumped into boreholes and can be used for boreholes of up to 35 m length. The explosive can be detonated using either a primer or a Powergel packaged explosive cartridge in conjunction with a detonator.

### **6.8. Form of the Loading Function**

As discussed in section 6.2, the pressure from an explosive blast is expected to increase to maximum pressure quickly, and slowly dissipate as the gasses penetrate the cracked region. In ABAQUS, a time-varying pressure is applied using the *\*DSLOAD* function in conjunction with the *\*AMPLITUDE* function. When the load is applied, the data lines on the *\*DSLOAD* function include the name of the amplitude function, and the maximum pressure. The maximum pressure is then multiplied by the amplitude function. Therefore, a 'unit' amplitude function is required.

The literature was consulted at this point of the development of the blast loading function to determine the expected shape for a blast loading function, or the functions used in numerical models by other researchers. A summary of loading functions used to model explosive materials is given in Table 6.3. These loading functions were used to model a variety of scenarios. Only the functions used to model blasting in a mine were considered as a possible loading function for use in this dissertation. Jiang et al. (1995) used the function given as equation 2.10 to model an emulsion explosive in rock, by applying the pressure to the walls of an equivalent cavity. Emulsion type explosives are also used at Cannington Mine. Therefore, the loading function represented by equation 2.10 was used to apply blast loads to the numerical models. The unit function is shown in Figure 6.2 for  $n = 1$  and  $\beta = 2000$  used by Jiang et al. (1995). These values for  $n$  and  $\beta$  were adopted for this work. The calculation of the initial peak pressure is discussed in section 6.8.1.

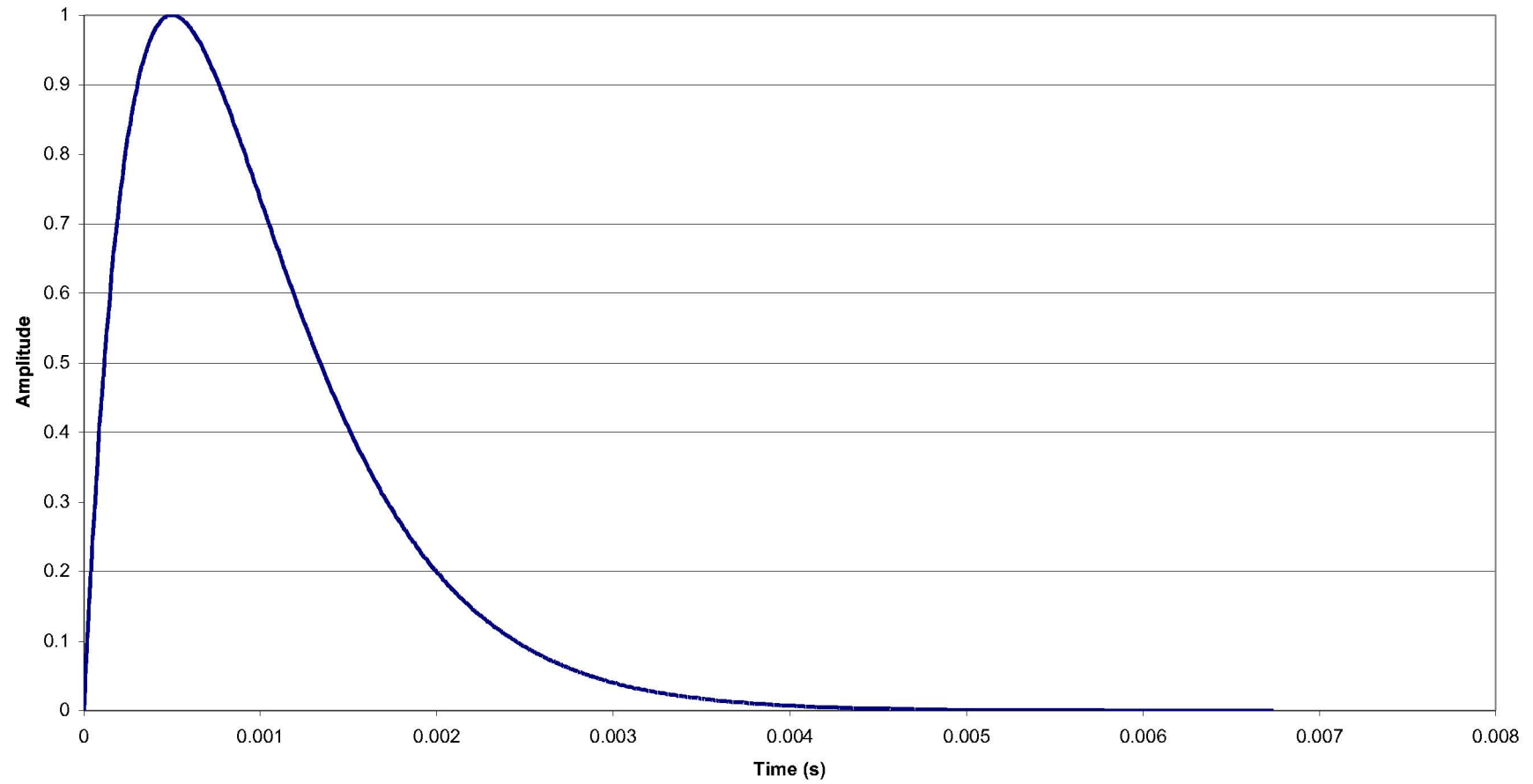
**Table 6.3 Loading Functions found in Literature**

Loading Function	Description	Reference
$p(t) = p_0 \left( 1 - \frac{t}{t_0} \right)$ <p>where <math>p_0</math> = initial peak pressure <math>t_0</math> = duration of blast loading</p>	<p>Modelled the loading from a spherical source in air on aluminium cladding using LS-DYNA.</p>	Hanssen et al. (2002)
$p(t) = p_0 \left( \frac{t}{t_0} \right) e^{-t/t_0}$ <p>where <math>p_0</math> = initial peak pressure <math>t_0</math> = duration of blast loading</p>	<p>Modelled a cylindrical blast by applying the pressure to the blast hole wall. Interested in the explosive fracture in oil shale.</p>	Grady and Kipp (1980)
$p(t) = p_0 t^n e^{-\varphi t}$ <p>where <math>p</math> = pressure <math>p_0</math> = initial peak pressure <math>t</math> = time <math>n</math> = constant <math>\varphi</math> = constant</p>	<p>Modelled explosive in rock, applying pressure load to walls on an equivalent cavity. Some authors used a spherical source, others a cylindrical source.</p>	<p>Jiang et al. (1995) Minchinton and Lynch (1996) Blair and Minchinton (1996) Blair and Jiang (1995)</p>

Loading Function	Description	Reference
$p(t) = p_{ra} \left[ 1 - (t - t_a) \right] e^{-\frac{t-t_a}{b/T_s}}$ <p>where <math>p_{ra}</math> = resulting peak overpressure  <math>t_a</math> = arrival time  <math>b</math> = waveform parameter  <math>T_s</math> = positive phase duration</p>	<p>Modelled the explosion of a bomb in a city street using small-scale experiments and numerical simulation. This was a spherical source from an air blast.</p>	Smith et al. (2000)
$v_r(t) = 0 \quad 0 \leq t \leq t_a$ $v_r(t) = \frac{v_r^{\max}}{0.1t_a} (t - t_a) \quad t_a \leq t \leq 1.1t_a$ $v_r(t) = v_r^{\max} \left[ 1 - 0.4 \left( \frac{t - t_a}{t_a} \right) \right] e^{-0.4 \left( \frac{t - t_a}{t_a} \right)} \quad 1.1t_a \leq t$ <p>where <math>v_r</math> = radial velocity relative to the centre of charge  <math>v_r^{\max}</math> = peak radial velocity  <math>t_a</math> = arrival time of ground shock time, <math>t_a = \frac{r}{c_p}</math>  <math>r</math> = distance from centre of charge  <math>c_p</math> = longitudinal seismic velocity  <math>t</math> = time after explosion</p>	<p>Modelled ground shock wave propagation using FLAC to consider the effect of explosive loads in the design of underground civil defence structures. A spherical explosive source was assumed.</p>	Olofsson et al. (1999)

Loading Function	Description	Reference	
$p(t) = p_m \frac{t}{t_r}$	$t < t_r$	Applied a time-varying pressure to the outer surface of a cylinder three times the diameter of the actual blast hole, avoiding simulation of the crushed zone. A cylindrical source was used.	Potyondy et al. (1996)
$p(t) = p_m$	$t_r \leq t < t_f$		
$p(t) = p_m (0.76 + 2000t)^{-2}$	$t \geq t_f$		
<p>where <math>p_m</math> = maximum pressure  <math>t_r</math> = rise time  <math>t_f</math> = fall-off time</p>			
$p(t) = p_0 \sin(\omega t)$	Modelled a sill mat in a mine which was subjected to seismic loads	O'Hearn and Swan (1989)	
	Modelled structural response to sinusoidal excitation and real blast vibration transients	Todo and Dowding (1984)	

Loading Function	Description	Reference
Linear Pressure Pulse  maximum pressure at $t = 0$ s  zero pressure at $t = 0.00108$ s	Modelled sacrificial claddings under blast loading from an air blast explosion (spherical source)  Noted the spherical pressure front can be approximated as a plane pressure front if the distance of explosion is high compared to the dimensions of the plane surface that is subjected to the blast loading.	Guruprasad and Mukherjee (2000)
JWL Equation of State	Modelled cratering due to an explosive charge, using a new damage model for rock.	Yang et al. (1996)  Liu and Katsabanis (1996)  Thorne et al. (1990)



*Figure 6.2 – Unit Loading Amplitude Applied to ABAQUS/Explicit Models*



### 6.8.1. Calculating the Initial Peak Pressure

The calculation of the initial peak pressure used in the loading function (equation 2.10) is discussed in this section. The initial peak pressure was calculated for two cases, the explosion of a Powergel Powerfrag cartridge in paste fill, and the explosion of Powerbulk VE in rock (see Tables 4.2 and 3.1). The method used in this calculation is outlined below. The finite element models created for this dissertation were created through a three stage modelling process. The first stage of the modelling process involved validating the numerical model results of the transmission of blast waves through paste fill against data from the field instrumentation tests. The models created for stage 1 consisted of a single blast hole located in paste fill. Since the stage 1 model was based on the field instrumentation tests in which one cartridge of explosive was loaded into the blast hole, the pressure blast calculated for the explosion of a Powergel Powerfrage cartridge was applied to the model.

The second stage of the process involved validating the numerical model results of the transmission of blast waves through rock against the peak particle velocity predictions obtained from equation 6.3 and the stage 2 model consisted of a single blast hole located in a body of rock. The third and final stage of the modelling process involved modelling blast holes located in rock at various distances from a paste fill stope. The blast pressure calculated for a column of Powerbulk VE was applied to both the stage 2 and stage 3 models. The problem definition and geometry of the models from all three stages are discussed in section 7.3.

1. The following numerical models of a single blast hole were used:
  - a. Paste Fill: The stage 1 model, described in section 7.3.2, was used to model the detonation of a cartridge of explosive located in paste fill. This model was based on the field instrumentation tests described in chapter 4. The axisymmetric model consisted of a 3 m deep borehole, which was assumed to contain a 0.2 m long cartridge of explosive material. An area 15 m wide and 6 m deep was modelled, with infinite elements along the side and base of the model in order to provide a non-reflecting boundary. The mesh for the stage 1 model is shown in Figure 7.3.
  - b. Rock: Model 2, described in section 7.3.3, as used to model an explosive column in rock. The axisymmetric model consisted of a 3 m deep borehole, which was assumed to consist of 2 m of explosive material and 1 m of stemming material. An area 15 m wide and 6 m deep was modelled, with infinite elements along the side and base of the model in order to provide a

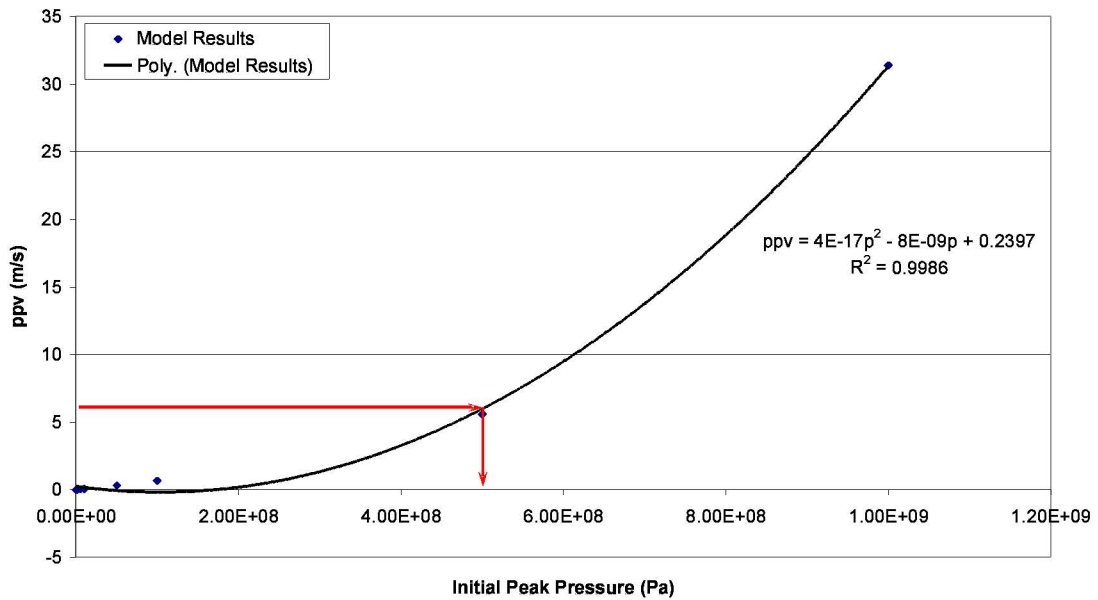
non-reflecting boundary. The mesh for the stage 2 model is shown in Figure 7.5.

2. The model was solved for a variety of blast loads using the time varying amplitude specified by equation 6.5 and initial peak pressures ranging between 100 kPa and 1000 MPa. For each loading case, the peak particle velocity (ppv) was computed at a number of points in the model that were located between 0.5 m and 14.0 m from the centreline of the blast hole.
3. The results from all of the different loading cases were collated. For each point at which the ppv was calculated, the ppv calculated by the model was plotted against the initial peak pressure applied to the model. This created a separate plot for each point of interest which showed the effect of the peak initial pressure on the ppv induced in the surrounding material. Trend lines were fitted to the model results. The plots for the points located 2.0 m and 5.0 m from the centreline of the blast hole in rock can be seen in Figure 6.3. The plots created in this step were used in steps 4 and 5 to estimate the peak initial pressure that should be applied to the model to produce the peak particle velocities shown in the experimental data.
4. For each plot created in step 3, the ppv was predicted using the formulae presented in section 6.4. The initial peak pressure to be applied to the model to calculate this ppv was then read from the plot as shown in Figure 6.3.
5. The initial peak pressure value to apply to the models was the average of the values obtained in step 4.

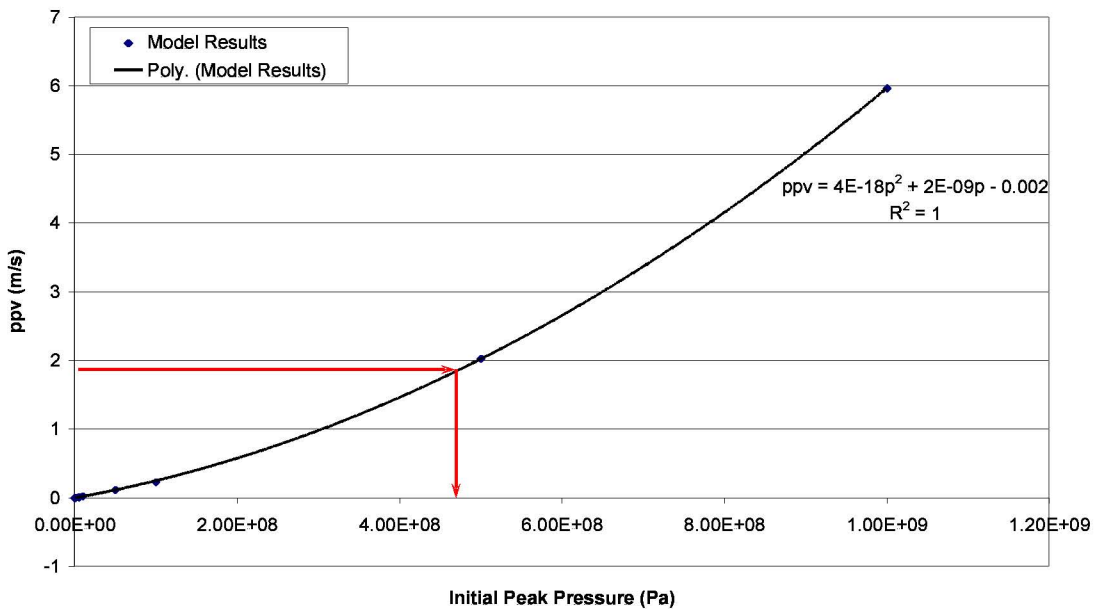
A pressure of 502 MPa was used for explosions in rock, and 45.1 MPa was used for explosions in paste fill.

### ***6.9. Validation of the Loading Function***

In order to validate the loading function, the model of the explosive cartridge in paste fill and the model of the explosive column in rock were run with the blast loads described in Section 6.8 applied. The results of these models were compared against the predicted ppv values for these scenarios. The results of the explosive cartridge in paste fill are shown in Figure 6.4 and the results of the explosive column in rock are shown in Figure 6.5.



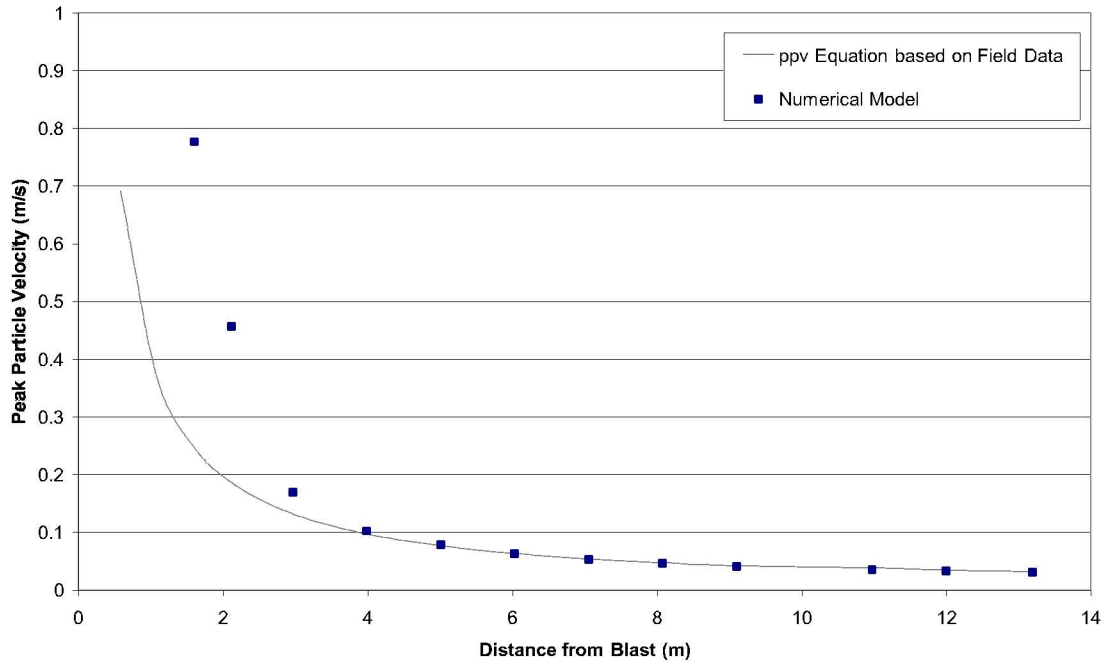
(a)



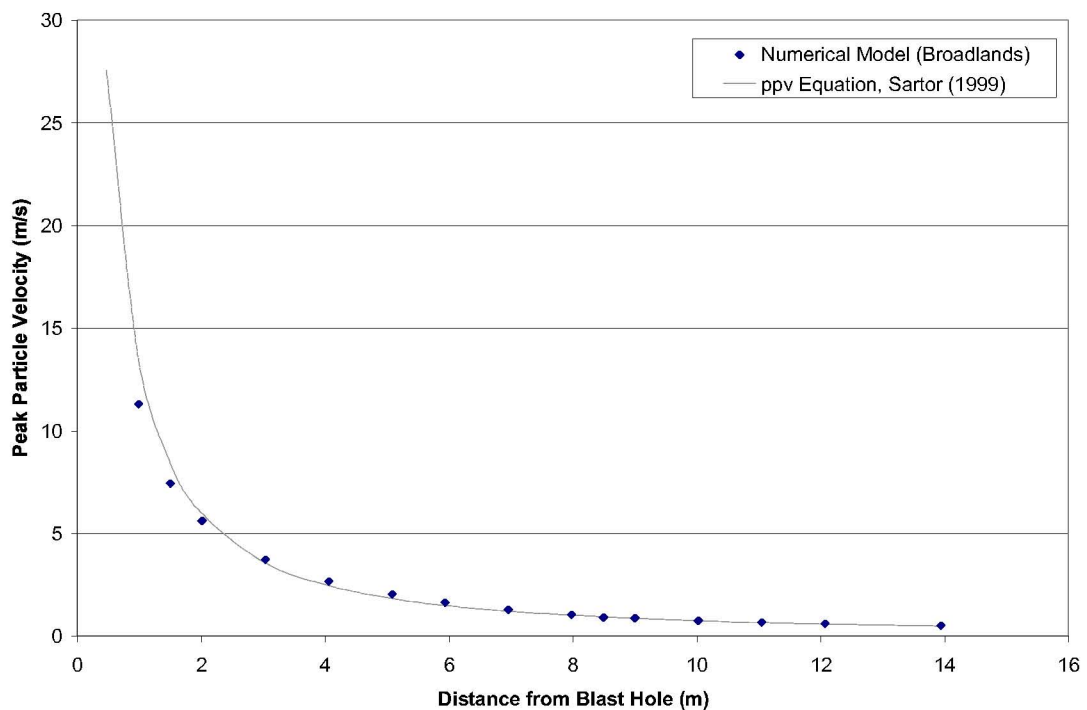
(b)

**Figure 6.3 – ppv Versus Pressure for 2 m Long Explosive Column in Rock**  
 (a) 2 m from Blast Column (b) 5 m from Blast Column

As seen in Figure 6.4, for distances greater than or equal to 2 m from the explosive in paste fill the model matches the predicted ppv quite closely, although the ppv are overestimated at distances less than 2 m from the source. Similar results are seen in rock.



**Figure 6.4 – Validation of the Stage 1 Model - Cartridge of Explosive in Paste Fill**



**Figure 6.5 – Validation of Stage 2 Model - Explosive Column in Rock**

The model overestimates the ppv close to the explosive source as the crushing and cracking mechanisms that occur in the vicinity surrounding the borehole are not considered in the numerical model. As discussed in section 6.5.5, since the purpose of the modelling for this

dissertation is to predict the ppv in paste fill as a result of blasting in adjacent rock, the effect of the stress wave is of interest rather than the crushing and fracturing which occurs around the blast hole. Since the paste fill is not within 2 m of the blast hole in the stage 3 model of blast holes in rock adjacent to paste fill, the results shown these models were found to produce acceptable results.

### ***6.10. Variables Which Affect Blast Loading***

The models discussed previously in this chapter consisted of a single explosive in a rock or paste fill mass. However, the production blast for the extraction of a stope in an underground mine consists of multiple blast holes which are detonated with millisecond delays between the detonation of each blast hole. There are many variables that can exist for a production blast, including the type of explosive used, the size of the blast, the blast hole pattern and the properties of the rock mass. These variables affect the stresses and peak particle velocities induced in a rock mass due to the blasting that occurs during the extraction of a stope. Some of these variables are discussed in the following sections.

#### **6.10.1. Types of Explosives**

The properties of explosives vary depending on the type of explosive. Commercial explosives used for mining applications can be broken into two groups, low and high explosives. Low explosives, such as black powder (or gun powder), are explosives that can be initiated by a flame, while high explosives are explosives which require shock or impact for detonation. High explosives consist of the following four main groups:

- **Ammonium Nitrate-Fuel Oil (ANFO):** As the name suggests, ANFO explosives consists of a mixture of ammonium nitrate and fuel oil. The performance of ANFO decreases when the explosive is exposed to water and mixtures containing more than 10 % water may fail to initiate. ANFO explosives have a velocity of detonation in the range of 2200 – 4000 m/s (Sen, 1995).
- **Watergels:** Watergels consist of a mixture of a gel base with ammonium nitrate and sometimes aluminium powder. These explosives have a gelatinous consistency which makes them suitable for use in wet conditions. Watergel explosives have a velocity of detonation in the range of 3500 – 5000 m/s (Sen, 1995).
- **Emulsions:** Emulsion explosives consist of fine microscopic droplets of oxidiser salts, finely dispersed into the continuous phase of fuel oil. These explosives have excellent

water resistance and can be detonated in deep wet blast holes. Emulsion explosives have a velocity of detonation in the range of 4500 – 6100 m/s (Sen, 1995).

- **Gelignites:** Gelignites are explosives which are based on nitroglycerine. They can be manufactured in gelatinous or semigelatinous form depending on the power, density and waterproofing requirements. Gelignite explosives have a velocity of detonation in the range of 3500 – 5500 m/s (Sen, 1995).

Emulsion type explosives were used at Cannington Mine for the field tests, and the production blasts which were instrumented and monitored for this project.

### **6.10.2. Rock Properties**

The rock structure and material properties will generally have a greater effect on the performance of blasting than the explosive properties. For the purpose of this modelling, it has been assumed that the rock mass is an isotropic and homogenous body, however, this is generally not the case. Rock masses contain joints and bedding planes which effect the transmission of the blast wave through the rock. These joints and beddings complicate the prediction of peak particle velocities and stresses in a rock mass due to the reflections of the blast wave which occur.

Rock material properties such as dynamic compressive strength, elastic modulus, density, porosity, internal friction, water content and in situ static stress effect the transmission of the blast wave through the rock and the subsequent damage to the rock mass.

### **6.10.3. Mining Methods**

The mining method which is used in a particular ore body depends on the type of rock and the geometry of the ore body. The various underground mining methods include longwall, open stope (also known as long hole stoping), cut and fill, room and pillar, shrinkage, sublevel caving and vertical crater retreat. The blasting pattern used for a particular mine will be dependent on the mining method.

At BHP Billiton's Cannington Mine, the open stope mining method is used with post placed back fill, where the ore is mined in blocks referred to as stopes. In open stope mining, access to the top and bottom of the ore block is set up with tunnels and a vertical hole is excavated from the top to the bottom of the stope. Blast holes are then drilled in order to excavate vertical slabs off the ore block (e-Gold Prospecting & Mining n.d.). The broken ore from the stope is then loaded into trucks from draw points at the bottom of the stope and hauled to the surface (BHP Billiton n.d.). Once the stope has been extracted, the void is filled with paste fill.

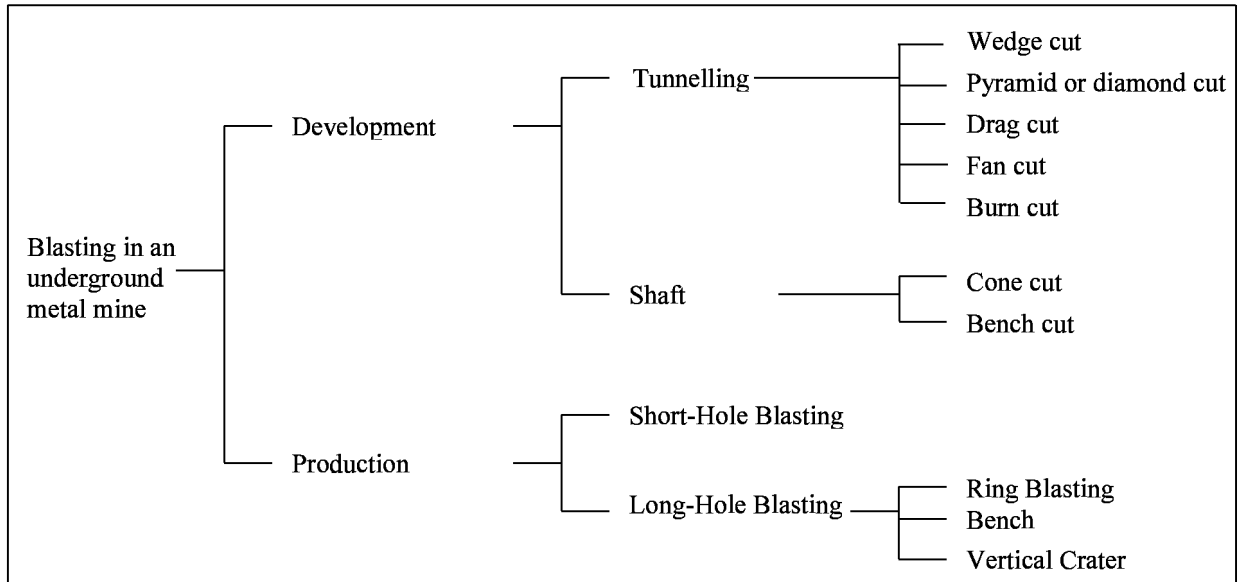
#### 6.10.4. Blasting Pattern

The blast load applied to a rock mass is dependent on many factors including the geometry of the ore body, the number of blast holes, the placement of the blast holes and the detonation order and delay timing. The blast pattern chosen for an ore body depends on the application of the blasting, the mining method being employed, the shape of the ore body, the depth and geological characteristics of the ore body.

The following definitions are useful for the discussion of blast hole patterns.

- Burden: The distance from the blast hole to the nearest free face
- Spacing: The distance between adjacent blast holes, measured perpendicular to the burden
- Charge length: The length of explosive in a blast hole
- Stemming: An inert substance, such as sand, filled between the explosive charge and the collar of the blast hole to confine the explosion gases. Materials such as water, drill cuttings, sand, mud and crushed rock are often used as stemming
- Decking: A technique of dividing the explosive column into two or more charges in the same blast hole, separated by stemming material.
- Charge density: The charge mass distributed in the column of a blast hole, measured in kg/m

A number of mining methods used in underground metal mines are shown in Figure 6.6, categorised by mining application. The blasting methods shown in the figure are broadly categorised into development and production blasts. Development blasts are the blasts required to access the ore body and transport the material after excavation. This type of mining includes tunnelling, shaft sinking, cross cutting and raising. Descriptions of the blasting patterns used in development blasts are given in Sen (1995).



**Figure 6.6 – Blasting Methods Used in Underground Metal Mines**

Production blasts are the blasts used in mining of the ore body. The long-hole blasting methods are of interest to this work. The three long-hole blasting systems are:

1. Ring Blasting

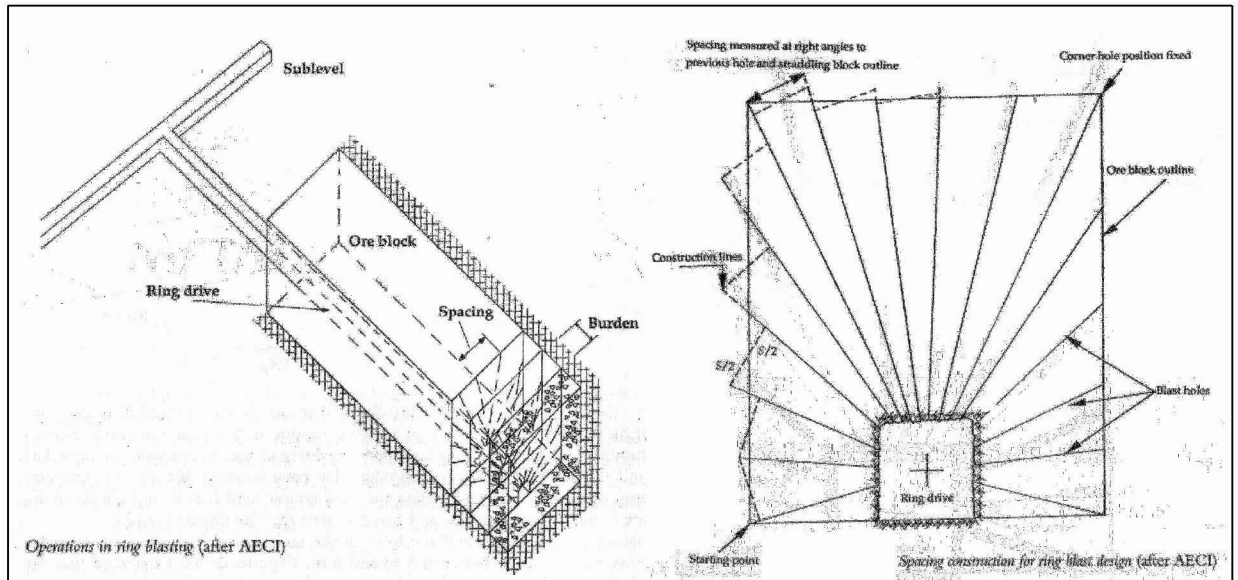
Ring blasting is the name given to the mining method that involves radial patterns of blast holes. It is used in the mining of massive ore bodies. Ring blasting involves the three steps listed below. These steps can be seen in Figure 6.7.

- a. Excavation of the “ring drive”, which is a tunnel running the full length of the stope
- b. Excavation of the “slot”, which is an empty space located at the end of the ring drive
- c. Drilling of sets of “rings” parallel to the slot

In ring blasting, the burden is defined as the distance between two consecutive rings, while the spacing is defined as the distance between the ends of adjacent holes in the same ring, measured as shown in Figure 6.7.

The collars of the blast holes are close together in ring blasting. As a result, a variable stemming length is used to avoid overcharging the ore body in this region.





**Figure 6.7 – Ring Blasting (Sen, 1995)**

## 2. Bench Blasting

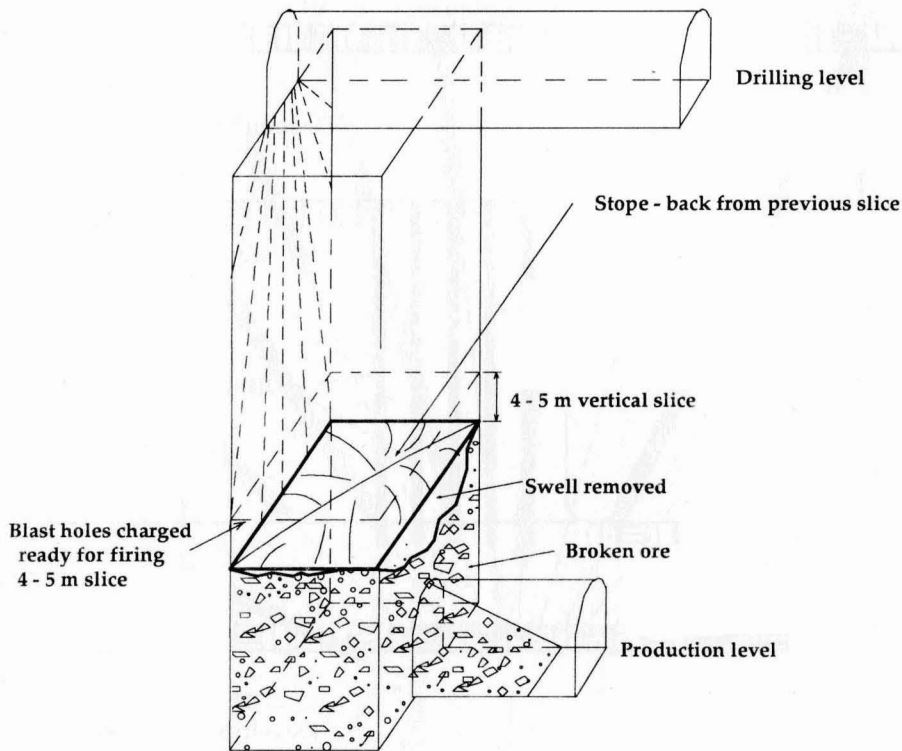
Bench blasting involves the use of a series of parallel blast holes, and consists of the following steps:

- a. Excavation of a development heading at the top of the sublevel to provide drilling space; and
- b. Drilling of blast holes. The blast holes can be horizontal, vertical, or inclined.

Either square drill patterns or staggered drill patterns can be utilised with bench blasting. Staggered patterns produce a more uniform blasting effect through the rock mass than square patterns. The effectiveness of a particular blasting pattern is dependent on the order of detonation.

## 3. Vertical Crater Retreat

In the vertical crater retreat method, the stope is mined from the bottom up. Blast holes are drilled downward from the top level to the bottom level. A slice of ore body is excavated from the lower level upward using the same blast holes for the different levels. Spherical charges are used in this method, and gravity assists the excavation process. The vertical crater retreat method is shown in Figure 6.8.



*Vertical crater retreat method (after Agnew Nickel)*

**Figure 6.8 – Vertical Crater Retreat Blasting (Sen, 1995)**

A combination of bench blasts and ring blasts were used in the production blasts monitored at Cannington Mine.

### 6.10.5. Initiation Patterns

Initiation patterns can vary greatly, with the optimal initiation depending on the blasting application, and geology. Initiation patterns are chosen to ensure that the blasting pattern is always working towards a free face.

### 6.10.6. Delay Intervals

Delay intervals between the detonation of blast holes in a blast round are used to reduce the ground vibrations and to increase fragmentation. The short delays are designed to allow the rock of previous blasts to move away and the free face of the next blast hole to be uncovered so that the blasting pattern is always working towards a free face. The delays are kept short enough that the rock from previous rows is still hanging in the air at the time of detonation of the next row, and is able to stop rock fragments from the second row from moving with greater speeds than average. The optimal delay time for a blast is dependent on the burden.

### **6.10.7. Blast Size**

The prediction of ppv from an explosive blast was discussed in section 6.4 and equations were presented from which to predict ppv based on the mass of explosive and the distance from the source. From equation 6.1 and 6.3 it can be seen that an increase in the linear charge density (mass per unit length) of explosive will result in a corresponding increase in ppv. Therefore, the ppv and consequently damage to the surrounding material can be decreased by reducing the mass of explosive detonated at any one time. This is achieved through the use of delay intervals.

### **6.10.8. Velocity of Detonation**

When an explosive detonates, the detonation is initiated at a point in the explosive material and a detonation wave travels through the explosive material. The rate at which this wave travels through the explosive is known as the velocity of detonation. The velocity of detonation is used in determining explosive performance, and often depends on both the explosive type and the blast hole diameter. Explosives with lower velocities of detonation tend to release gas energy over a longer period of time than explosives with higher velocities of detonation. This results in less fracturing around the blast hole and more heaving action. Therefore, the velocity of detonation of a given type of explosive can be used to determine which explosive is required for a given application.

## ***6.11. Summary***

A blast loading function was identified in this chapter. In order to identify a method of applying a blast load to the numerical model, the methods used by other authors and the mechanics of a blast were investigated. The typical use and the advantages and disadvantages of the different methods was considered along with the intent of the models created for this dissertation to determine the most appropriate method of load application for this work. The blast load was applied to the numerical model by applying a time-varying pressure pulse to the walls of an “equivalent cavity”. The form of the time-varying pressure pulse was based on the function used by Jiang et al. (1995) to model the detonation of the same type of explosive. The initial peak pressure for the time-varying pressure was set to the value required to produce peak particle velocities similar to those observed in field tests. At the end of the chapter, variables which affect the stresses and peak particle velocities induced in a rock mass due to the blasting were discussed.