Chapter 2

REVIEW OF LITERATURE

Various approaches to blast design and the parameters that determine a blast design are reviewed to understand the present state of knowledge on the topic of this thesis and to obtain an indication of the direction that further research in this area should take.

2.1 THEORY OF ROCK BLASTING

2.1.1 Mechanisms of Rock Breakage

Good understanding of the mechanisms of rock breakage is fundamental for blast design. However, the manner in which rock is broken by explosive loading is very complex and is not fully understood (Fourny, 1993). The theories that attempt to explain the mechanisms of rock breakage are:

1) Reflection theory or stress wave theory (Hino, 1956; Duvall and Atchison, 1957)
3) Flexural rupture theory (Ash, 1973)
4) Stress waves and gas expansion theory (Kutter and Fairhurst, 1971)
5) Combined theory (Lang and Favreau, 1972)
6) Nuclei or stress wave/flaw theory (Winzer et al, 1983)

In general, the mechanisms of rock breakage can be classified into two categories, those caused by the shock energy of an explosive and those caused by the gas energy of an explosive. The sequence of events that take place during blasting is illustrated in Fig 2.1.

When an explosive charge confined within a blasthole is detonated, the previously stable condensed ingredients of the explosive are rapidly converted into gaseous products at very high pressure and temperature. The sudden impact of the high pressure gases on the blasthole wall transmits a shock wave into the surrounding rock. The outgoing shock wave crushes the surrounding rock if the intensity of the stress is greater than the dynamic compressive strength of the rock (Hagan, 1979a). The outgoing shock wave also develops radial fractures due to tangential, tensile stresses (Johansson and Persson, 1970). On meeting an open joint or free face, the compressive stress wave gets reflected and spalling occurs if the intensity of the tensile wave exceeds the dynamic tensile strength of the rock (Hino, 1956, Duvall and Atchison, 1957). The reflection of the compressive wave generates tensile and shear waves...
Fig 2.1 Sequence of events in rock breaking process (Sarma, 1994)
which may extend pre-existing cracks (Fourney, 1993)

The high pressure gases remaining in the blasthole after the transmission of the shock wave penetrate into the radial fractures or natural fractures or both. The gas pressure wedges these fractures open and extends them. The explosion gases remaining in the rock mass push the rock mass forward and bend the face causing flexural rupture (Ash, 1973). Shear fracturing occurs when adjoining rock is displaced at different times or at different rates by the high pressure gases (Hagan, 1979a). Some breakage also takes place due to in-flight collision of blasted fragments (Hagan, 1979a).

There is some controversy over the relative importance of shock and gas energy in rock blasting. Recent publications give more importance to the contribution of gas energy. Britton and Skidmore (1989) in their model scale experiments found that the explosive-generated gas pressure was the dominant parameter in breaking the rock in confined blastholes. Brinkmann (1990) conducted experiments with a blasthole liner to separate the effect of gas penetration in blasting. His results showed that the gas pressure induced fractures played an important role in the breakage process. Armstrong et al. (1993) conducted controlled model scale tests and found that the gas penetration into the fractures has a significant effect on the fragmentation process.

Fourney (1993) showed that both stress waves and gas pressurization are important in the fragmentation of rock. Of the two, the energy contained in the gas pressurization phase is much greater than the energy contained in the stress wave component. Nevertheless, the stress waves are useful in preconditioning the rock mass for later action by gas pressurization. Most efficient results can be obtained if proper use can be made of both components.

2.1.2 Partition of Energy

The partition of energy in the process of rock blasting can be described using a simplified model of rock-explosive interaction (Lownds, 1986, Udy and Lownds, 1990). The energy delivered by the explosive during rock blasting is represented by the pressure-volume curve as shown in Fig. 2.2. The area under the curve represents the total energy released by the explosive. The energy is partitioned into different zones which represent the expansion of the explosion gases during the different stages of rock breakage.

Immediately after the detonation of the explosive, the high pressure gases generated by the explosive reaction expand the borehole and transmit a shock wave into the rock. The borehole expansion increases the volume of the borehole with a corresponding decrease in the
The borehole expansion continues until it reaches a quasi-static equilibrium state where the pressure of the explosion gases in the borehole cavity is matched by the stress in the surrounding rock (Lownds, 1986, Udy and Lownds, 1990). Position A on the curve represents the initial explosion state, position B represents the equilibrium state and position C represents the point at which the gases escape into the atmosphere. The line OB represents the response of the blasthole wall to the explosive loading and is dependent on the stiffness of the rock. The energy delivered up to position B is termed shock energy and is represented by area OABD (Lownds 1986, Udy and Lownds 1990).

The shock energy is divided into kinetic component (represented by Zone 1 and area OAB) and strain component (represented Zone 2 and area OBD). The kinetic component of shock energy is considered to be responsible for the crushing of the rock surrounding the blasthole and for generating ground vibrations. The strain component of shock energy is considered to be responsible for the primary fractures necessary for fragmentation.

The high pressure gases enter and expand the primary fractures until they escape into the atmosphere. The energy released during this expansion develops the fracture network further and is also responsible for burden movement. Thus the energy in zones 2 and 3 is responsible...
for fragmentation and heave. At the time of gas escape the burden is compressed by the explosion gases in the cracks as strain energy stored in the rock (Zone 4 and area DCE). This energy is regarded as insignificant in fragmentation and heave. The rest of the energy is lost into the atmosphere and the wasted energy is represented by Zone 5. The combined energy represented by Zones 1, 2, 3, and 4 is considered as effective energy released by the explosive during blasting and is called blast energy.

Although this model of energy partition oversimplifies the blasting process, it gives valuable insight into where the energy goes during the various phases of the process. The concept of energy partition has been extended further by Sarma (1994) to calculate explosive energy released during the blasting process.

2.2 BLAST DESIGN

A blast design in its simplest form means the specifications of blastholes (hole diameter, burden, spacing, depth), charging details (type of explosives, charge per hole, stemming) and initiation (priming, sequence, timing). For each blast design, it is necessary to understand the objectives of the blast, ensure that any constraints will be satisfied, and have some confidence that the blasting operation, as designed, will achieve its objectives in the most cost-effective manner (JKMRC, 1993).

The primary objective of a blast design is optimum fragmentation, defined as the blasting practice which gives the lowest combined cost for drilling, blasting, loading, hauling and crushing (MacKenzie, 1966). The blasting engineer naturally wishes to conduct blasting in an optimal way but it must be borne in mind that this not only includes achieving the desired objective at the minimum cost but also minimizing disturbances to the environment (Ash, 1990).

The fundamental variables which exert the predominant influence on the results of blasting are the explosive, blast geometry, and rock mass. These variables are related in terms of energy, mass, and time (Lang and Favreau, 1972).

2.2.1 Blast Design Parameters

The most important blast design parameters are blast geometry, initiation sequence, and delay timing. The blast geometry consists of bench height, blasthole diameter, burden, spacing, stemming, subgrade drilling, and hole depth (Fig. 2.3). Burden is defined as the distance from a blasthole to the nearest free face at the time of detonation. It is conventionally measured and...
recorded in the direction parallel to the free face. Spacing is defined as the distance between adjacent holes, measured perpendicular to the burden. Stemming is an inert material loaded on top of the explosive charge in a blasthole. Subgrade drilling is the depth to which a blasthole is drilled, below the grade level. Powder factor is a relationship between the rock broken and the quantity of explosives used to break it.

![Diagram of blast geometry elements]

**Fig 2.3 Elements of blast geometry**

For a set of blastholes several initiation sequences are possible (Hagan, 1983; Konya and Walter, 1990; ICI, 1993). The initiation sequence significantly affects the blast results in respect of direction of throw, fragmentation and damage to the rock mass. For multitrow blasting, the drilled spacing to burden ratio will change depending on the initiation sequence as illustrated in Fig 2.4. The changed spacing to burden ratio is called the effective spacing to burden ratio ($S_e/B_e$).

Delay intervals between the rows or between the holes in a row or both are an essential part of blast design. Suitable delay intervals result in an improved fragmentation, good displacement, reduced ground vibration and flyrock (Chiappetta et al., 1986). Inadequate delay times cause higher muck piles close to the face (Fig 2.5), excessive flyrock, ground vibration, and air overpressure. The delay intervals between the rows may vary from 10 ms/m of the burden for hard rock to 30 ms/m of burden for soft rock (Olofsson, 1991), from about 5 ms/m of effective burden for strong massive rocks to about 10 ms/m for weak and/or highly fissured strata (Hagan, 1983). In addition to burden and rock type, delay timings are also governed by the desired end results based on their priority (Konya and Walter, 1990).
Fig 2.4 Common initiation sequences
2.2.2 Explosive Characteristics

Explosives are of different types (NG-based, ANFO, Heavy ANFO, slurry, emulsion) and differ in their characteristics. The most important characteristics of explosives are velocity of detonation, density, sensitivity, strength/energy, water resistance, temperature stability, fumes and shelf life (Atlas Powder Company, 1987; Persson et al, 1994; Konya and Walter, 1990; Olofsson, 1991; Dick et al, 1986).

ANFO is the widely used explosives in the world because of its low cost. However, it suffers from poor water resistance and poor bulk energy. Heavy ANFO, a mixture of ANFO and emulsion, can overcome these drawbacks. Fig 2.6 presents a relationship between relative weight strength and relative bulk strength for different aluminised and non-aluminised ANFO, emulsion and heavy ANFO (Crosby and Pinco, 1992). This figure also indicates which products can be used in wet, dewatered or dry blastholes. For a particular bulk strength, there is a range of explosives which can be selected to suit the given condition.

Users of explosives are frequently confused with strength of explosives. The confusion arises from the different ways of rating explosive strength, and from the practical experience that an explosive does not always perform according to the strength quoted by the supplier (Lownds, 1986). The Swedish weight strength concept of an explosive (Langeфорs and Kihlstrom, 1963) strongly overestimates the blasting strength of high-density, high-energy explosives such as aluminised TNT slurries and plastic dynamites with high nitroglycerin content. It also strongly underestimates the blasting strength of the low flame temperature, low density explosives such as ANFO and pure emulsion blasting agents (Persson et al, 1994).
There are some deficiencies in the calculation of effective strain wave energy and effective heave energy (Hagan and Duval, 1993). When the explosion energy is wasted in crushing or plastically deforming rock immediately around a charge, currently calculated values of effective strain wave energy are too high because they include the wasted energy. For rocks which are weak or highly fissured, currently calculated values of heave energy are too low because they do not include the useful energy which is liberated by the explosion gases at pressures between 100 MPa (the generally accepted cut-off pressure for calculated effective energy) and around 30 MPa. These deficiencies in calculating and predicting effective energies lead to overstating the performance of high-pressure explosives like emulsions on one hand and under-estimating the performance of low-pressure explosives like ANFO on the other.

The explosive manufacturers usually rank an explosive based on its chemical efficiency and energy release. Although total energy factors are easy to calculate and have been used widely, they are not sufficient to describe the blasting process (Katsabanis, 1995). Energy partition into shock and gas energy is important and must be considered. The understanding of partitioning of energy has opened new horizons for selection of explosives. The explosives are tailored to produce a certain partitioning of energy that match the ground type and the

Fig. 2.6 Regulating strength and water resistance of ANFO with emulsion and aluminum
operating environment. The tailoring of explosive performance (Rainbird, 1995) generally involves three major methods (other than changing the bulk density and sensitivity): i) replacing fast burning reactants with slower burning reactants or vice versa; or ii) leaving the explosives intact and diluting the entire explosive with slow burning or 'inert' material; or iii) changing the concentration of the fuel oil/oxidiser mix.

In the area of ANFO explosives, low density explosives have been developed to control shock energy output, while reducing the total energy output of the product. This has been accomplished by using mixes of ANFO with various amounts of low density ingredients to obtain densities between 0.4 and 0.7 g/cc (Wilson and Moxon, 1989). ANRUB, a mixture of ammonium nitrate and rubber (Harries and Gribble, 1993) serves the purpose of controlling the shock energy, while keeping the total energy content approximately constant. ANFO minipills (Adams and Irwin, 1994) provide more shock energy, due to the enhanced reaction before the CJ plane, stemming from the reduced particle size, while microporous ammonium nitrate-fuel mixes (Vuillaume and Bouvet, 1993) do the same, due to reduction of the effective particle size resulting from the increase in porosity and mixing intimacy.

In the areas of slurry explosives, the development of low density explosives follows a similar trend to that of ANFO. Low density slurries have been developed to minimise unwanted damage in wet conditions. Densities of the order of 0.4-0.7 g/cc have been applied for this purpose (Jackson, 1993). While slurries have been known to be typical non-ideal explosives, their degree of non-ideal performance can be modified by reducing the effective particle size of the mix, which is achieved by the use of solids with small particle sizes and enhanced mixing.

Emulsion explosives, due to their ideality of detonation, deliver a large amount of shock energy and consequently less heave energy, resulting in good performance in hard rock and rather poor performance in soft or highly fractured rock (Katsabanis, 1995). The introduction of Heavy ANFO solved the poor energy partition problem and introduced a powerful explosive system. By mixing an ideal ingredient (emulsion) with various amount of non-ideal ingredient (AN or ANFO prills), varying degrees of non-ideality can be achieved. Thus the shock and heave energy partitions, as well as total energy levels, can be changed by simply changing the proportions of various ingredients. With controlled non-ideal performance, Heavy ANFO formulations offer the advantages of flexibility and adaptability.

2.2.3 Rock Mass Characteristics

It is well established that the rock mass characteristics have a significant influence on
The principal characteristics that influence blast results are the intact rock properties and the structural discontinuities. The intact rock properties include compressive strength, tensile strength, density, velocity of wave propagation, porosity, Young's modulus and Poisson's ratio. However, intact rock properties do not truly indicate whether the rock mass is easy or difficult to blast because structural discontinuities overshadow the influence of physico-mechanical properties of rocks (Gnirk and Pfleider, 1968).

The structural discontinuities of a rock mass are joints, bedding planes, foliation, faults, etc., which may be called joints in general. They are characterised by their orientation, number of sets, spacing, continuity and filling material. Structural discontinuities divide rock masses into a collection of separate blocks. The size and shape of the blocks are controlled primarily by the orientation and spacing of the discontinuities. These blocks exert a significant control over rock fragmentation by blasting (La Pointe and Ganow, 1986, Goodman and Shi, 1985). Figure 2.7 illustrates that blasting has a tendency to separate the blocks along the joints. Using the concept of fractal geometry, Ghosh et al. (1990) showed that about 50 to 75% of the blasted fragments were formed by the structural discontinuities. Attempts were made to develop computer models to estimate the complete block size distribution and to study its influence on fragmentation (Da Gama, 1977; Maynard, 1990, Wang et al, 1992, Villaescusa and Kleine, 1990, Aler et al, 1996). Since these models are basically research tools, a simpler method that can indicate the representative block size in the rock masses would be extremely useful to the blast designer.

In addition to the direct use of rock mass properties, attempts were also made to characterise the sites based on 1) seismic survey (Broadbent, 1974, Hemen and Dimock, 1976, Muftuoglu et al, 1991), 2) geophysical logging of production holes (Bellaars, 1987), 3) monitoring of drilling parameters of blastholes (Leighton et al, 1982, Muftuoglu et al, 1991; Jimeno and Hevia, 1987), and 4) Blastability Index (Lilly, 1986). The first and second methods need suitable instrumentation and specialists to carry out the investigations and are therefore not common. Penetration rate of drilling can provide useful information about the strength of rock and therefore is generally used. Blastability Index is determined from five parameters, namely rock mass description (powdery, blocky or massive), joint spacing, joint orientation, density and hardness. Since the orientation of joints and their spacing govern the rock mass description, it would be appropriate to characterise the rock mass in terms of the representative block size.
2.3 VARIOUS APPROACHES TO BLAST DESIGN

Various approaches have been used to blast design and the literature is very vast in this area. Only the most accepted ones are briefly discussed.

2.3.1 Empirical

Empirical approach depends largely on experience factors and is based on recommended practices and inter-relationships that exist amongst the numerous blast design parameters. The various thumb rules still in use fall in this category. This approach has limitations as it lacks the basic information on material response to explosive action. Empirical relationships are safe to apply only in the context for which they have been formulated. It is most popular among the practising mining engineers to calculate the initial blast design parameters.
The best known of all empirical approaches are that developed by Langefors and Kihlstrom (1963) The formula for burden calculation is.

\[ B_m = 0.9581 \sqrt{\frac{\rho_e \cdot s}{(S/B) \cdot c_0 \cdot f}} \]  \hspace{1cm} (2.1)

where

- \( B_m \) = maximum burden for good fragmentation (m)
- \( d \) = hole diameter (m)
- \( \rho_e \) = density of the explosive in the borehole (kg/m³)
- \( s \) = weight strength of the explosive
- \( f \) = confinement of the blasthole
- \( S/B \) = spacing to burden ratio
- \( c_0 \) = corrected blastability factor (kg/m³)
  - \( = c + 0.05 \) for \( B_m = 1.4 - 15 \) m
  - \( = c + 0.07/B \) for \( B_m < 1.4 \) m
- \( c \) = rock constant

Konya and Walter (1990) have suggested the following two equations for burden calculation

\[ B = 3.15 \cdot d_e \left[ \frac{\gamma_e}{\gamma_r} \right]^{1/3} \]  \hspace{1cm} (2.2)

\[ B = \left[ 2 (\gamma_e/\gamma_r + 1.5) \right] d_e \]  \hspace{1cm} (2.3)

where

- \( B \) = burden (ft)
- \( \gamma_e \) = specific gravity of explosive
- \( \gamma_r \) = specific gravity of rock
- \( d_e \) = diameter of explosive (in)

They apply three correction factors as

\[ B_c = K_d \cdot K_s \cdot K_r \cdot B \]  \hspace{1cm} (2.4)

where

- \( B_c \) = corrected burden (ft)
- \( K_d \) = correction factor for rock deposition (Table 2.1)
- \( K_s \) = correction factor for geologic structure (Table 2.2)
- \( K_r \) = correction factor number of rows (Table 2.3)
Table 2.1 Correction factor for rock deposition

<table>
<thead>
<tr>
<th>Nature of deposition</th>
<th>$K_d$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bedding steeply dipping into cut</td>
<td>1.18</td>
</tr>
<tr>
<td>Bedding steeply dipping into face</td>
<td>0.95</td>
</tr>
<tr>
<td>Other case of deposition</td>
<td>1.0</td>
</tr>
</tbody>
</table>

Table 2.2 Correction factor for geologic structure

<table>
<thead>
<tr>
<th>Geologic structure</th>
<th>$K_s$</th>
</tr>
</thead>
<tbody>
<tr>
<td>Heavily cracked, frequent weak joints, weakly cemented layers</td>
<td>1.30</td>
</tr>
<tr>
<td>Thin well cemented layers with tight joints</td>
<td>1.10</td>
</tr>
<tr>
<td>Massive intact rock</td>
<td>0.95</td>
</tr>
</tbody>
</table>

Table 2.3 Correction factor for number of rows

<table>
<thead>
<tr>
<th>Number of rows</th>
<th>$K_r$</th>
</tr>
</thead>
<tbody>
<tr>
<td>One or two rows of holes</td>
<td>1.0</td>
</tr>
<tr>
<td>Third or subsequent rows</td>
<td>0.95</td>
</tr>
</tbody>
</table>

Konya and Walter (1990) have suggested different equations for spacing calculation depending on the mode of initiation and $L/B$ ratio (Table 2.4) where $L$ is the length of hole, $B$ is the burden and $S$ is the spacing.

Table 2.4 Equations for calculation of spacing

<table>
<thead>
<tr>
<th>Mode of initiation</th>
<th>$L/B &lt; 4$</th>
<th>$L/B \geq 4$</th>
<th>$L$ = hole depth</th>
</tr>
</thead>
<tbody>
<tr>
<td>Instantaneous</td>
<td>$S = \frac{L+2B}{3}$</td>
<td>$S = 2B$</td>
<td>$B$ = burden</td>
</tr>
<tr>
<td>Delay</td>
<td>$S = \frac{L+2B}{8}$</td>
<td>$S = 1.4B$</td>
<td>$S$ = spacing</td>
</tr>
</tbody>
</table>

2.3.2 Trial and Error

In this method, the blast design is decided by experimenting with different patterns and modifying the designs until the right degree of fragmentation and explosive consumption is attained. This is the most common method in Indian mines.
Singh et al (1994) used the trial and error method for optimisation of blast design parameters at Bailadila Project, India. In these trials, no consideration was given to the introduction of a new initiation system or bulk explosives. The trials were conducted with the detonating cord downline and cartridge explosives. Since the production of the mine is about 5.4 million tonnes of iron ore and the hole diameter is 250 mm, bulk explosives would have been ideal. Though rock properties such as compressive strength, tensile strength, bulk density and Young's modulus were determined, they were not used to design the blast.

Sagmania limestone quarry (Singh, 1992) was being worked till 1977 in three benches of 7 m, 3.5 m and 4 m, respectively. For 100 mm diameter, burden and spacing used were 3 m x 3 m irrespective of the bench height. The holes were initiated with millisecond delay detonators. This practice resulted in boulders and toe.

Realising the low bench height as the cause for fragmentation problem, the quarry was developed in two benches with bench heights of 10-11 m and 5-6 m. Without any proper justification, the hole diameter was changed from 100 mm to 150 mm whereas 100 mm would have been better for the bench height of 5-6 m. Various patterns, number of rows and different initiation sequences with a Sequential Blasting Machine and detonating relays were again tried to increase the size of the blasts without increasing the vibration levels.

Several trial blasts with total charge of 274 - 850 tonnes were conducted in dragline benches at an open cast coal mine of Singrauli (Malchanda, 1990), which produces 10 million tonnes of coal per annum. Though blast results were satisfactory, Manehanda (1990) suggested the followings to meet the future challenges at Singrauli Coalfields.

1) Inclined drilling to reduce toe burden,
2) Use of bulk explosives,
3) Use of advanced initiation system (shock tube system),
4) Use of an equipment to measure toe,
5) Use of computer models for blast design, and
6) Regular monitoring and control of ground vibration.

From these suggestions, it can be inferred that large mines are interested in scientific studies, introduction of new products, blast monitoring and computer models.

2.3.3 Cratering

If a vertical hole is drilled to a horizontal surface of rock and charged with explosive, it
may blow out a conical crater Utilising this phenomenon, a test was proposed by Livingston (1956) This test is conducted using charges of constant weight and detonating them at different depths of burial The information needed for a blast design are the critical depth and optimum depth. The critical depth is the distance from the surface to the center of gravity of charge at which no breakage occurs Optimum depth is the distance from the surface to the centre of gravity of charge corresponding to maximum crater volume Livingston developed a strain energy equation for crater charges as given below.

\[ N = EW^{1/3} \]  \hspace{1cm} (2.5)

where
\[ N = \text{critical depth of a charge (m)} \]
\[ W = \text{weight of charge that would just cause the rock surface to fail (kg)} \]
\[ E = \text{strain energy factor, derived empirically} \]

Livingston modified this equation by reducing the charge depth to give good fragmentation by expressing it in the following form

\[ d_0 = \Delta_0 EW^{1/3} \]  \hspace{1cm} (2.6)

where
\[ d_0 = \text{optimum depth (m)} \]
\[ \Delta_0 = d_f/N = \text{optimum depth ratio} \]
\[ W = \text{weight of charge (kg)} \]

![DOMING OF THE SURFACE](image_url)

Fig 2.8 Cratering phenomenon of a constant charge at varying depth in the same formation
Fig. 2.8 is a schematic illustration of the effect of varying depth on a constant charge in the same formation. At shallow depth, most of the energy is transmitted to the atmosphere in the form of air overpressure and flyrock. At greater depth most of the energy is used in ground vibration with little crushing around the charge. At optimum depth, the rock is completely broken with minimum dissipation of energy in unwanted forms.

Crater data can be plotted in a number of ways (Atlas Powder Company, 1987; Bauer, 1961). In India, crater tests were used for explosives selection (Ghose, 1986) and for determination of burden and powder factor (Singh et al, 1987) Though no explicit material or explosives properties are required in this method, cratering method is not suitable because of the different geometry of bench blasting.

2.3.4 Single Hole

The effect of single hole blasting at different burdens on rock fragmentation and displacement is illustrated in Fig. 2.9 (Berta, 1990).

a) If the burden is large, detonation results in pulverisation and crushing of the rock in the immediate vicinity of the hole with several radial cracks between the blasthole and the free face. However, the breakage is limited to the immediate vicinity of the hole without displacement of rock (Fig. 2.9a).

b) If the burden is optimum, it is possible to get the maximum amount of broken material, good fragmentation, and good displacement (Fig. 2.9b).

c) If the burden is small, the amount of rock broken is small with excessive throw and flyrock (Fig. 2.9c).

Rustan and Nie (1987) proposed a method based on single hole blasts in full scale. The tests are conducted with different burdens up to and greater than the critical, with that hole diameter, depth of hole, explosive and stemming desired to be used in the real operation. This method requires determination of fragment size distribution, the backbreak, throw and the damage to the remaining rock for each burden tested.

Good correlation was obtained between burden and angle of breakage (Table 2.5). As the hole approaches to the bench face (burden decreases) the angle should approach to $180^\circ$ but it is not always the case because structural discontinuities play a major role in the breakage and control the breakage angle.
Fig. 2.9 Effect of varying burden distance on fragmentation and displacement of rock
Berta (1990)
Table 2.5 Angle of breakage as a function of burden based on single hole tests

<table>
<thead>
<tr>
<th>Investigator(s)</th>
<th>Name of mine</th>
<th>Formula</th>
<th>Correlation coefficient</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ruslan and Nie (1987)</td>
<td>Storungs limestone, Sweden</td>
<td>$\theta = 165.7 - 6.88B$</td>
<td>0.93</td>
</tr>
<tr>
<td>Bilgin (1991)</td>
<td>Divriği iron ore, Turkey</td>
<td>$\theta = 206.89 - 18.02B$</td>
<td>0.99</td>
</tr>
<tr>
<td>Bilgin et al (1993)</td>
<td>Tamtas limestone, Turkey</td>
<td>$\theta = 180-21.63B$</td>
<td>0.99</td>
</tr>
</tbody>
</table>

$\theta =$ angle of breakage (degrees), $B =$ burden (m)

The angle of breakage determines the broken rock volume and in turn the powder factor. The broken volume increases as the burden increases, and the volume reaches a maximum when the burden is optimum. For a burden value beyond optimum, the broken volume decreases rapidly. This phenomenon is similar to that of crater blasting.

Single hole method determines the burden and powder factor for predicted fragmentation, throw and damage. However, single hole method does not provide information about multihole blasts. It does not take into account the initiation system and delay timing which are vital elements in the blast design.

Several researchers (Anderson et al., 1985; Hinzen, 1988; Cronwelge, 1987) have used single hole blasts to predict and control ground vibration. The seismic signature produced by the detonation of a single hole charge is recorded. It is assumed that the signature is determined primarily by the site geology and each hole in that area will produce similar signature. Vibrations produced by blasts are simulated using the linear superposition of the signature delayed by a given time interval.

2.3.5 Instrumented Field Trials

Instrumented field trials are usually conducted with the help of consultants or explosive suppliers in a mine where the management is open-minded to new methods, products or technology. Using instruments and analysis tools, the guess work involved in blast designs and explosive performance assessments can be eliminated (Chiappetta, 1991; Cunningham, 1990). Full scale blasts can be monitored to evaluate the explosive performance, determine optimum delay intervals, measure burden velocity and stemming ejection velocity. The monitored data can be analysed and used to optimise the site specific blast designs.

Cox and Cotton (1996) reported a case study related to the evolution of blasting practices at a stone quarry in Australia. A series of trials were conducted and monitored. The blast
monitoring included burden velocity, in-the-hole VOD, vibration, air overpressure, muckpile profile and blasthole deviation. As a result of several trials with a range of drill patterns, firing sequences, explosives types and initiation systems, the quarry significantly increased the size of blasts and reduced the frequency of firing without increasing the ground vibration and air overpressure levels. It resulted in significant improvements in terms of both the cost effectiveness of its operation and in terms of its environmental impact on nearby residents.

In an opencast mine in South Africa, problems were experienced with blasting results (Ladds et al., 1993) Very large boulders and portions of completely unfragmented rock were commonly encountered. A full scale blast monitoring programme was undertaken to optimise blasting operations at the mine. Eight blasts were monitored using VOD recorders, high speed photography and close-in seismic monitoring. The results showed an incompatibility problem between the explosives being used and the in-hole initiation system. Variable detonation results were also identified in some of the blasts and a change in the design of the surface tie-up was found to be necessary because the required individual firing of holes and sequential firing was not occurring. The performance of various stemming material under different blasting conditions was also measured. It was found that stemming performance was controlled mainly by the amount of water in the stemming material.

2.3.6 Computer Modelling

Computer modelling of rock blasting continues to be an active research area. It holds the promise of predicting the results of a blast before it occurs, thus allowing blast to be simulated first on a computer before an expensive blast is attempted. The object of computer blast modelling is to apply the principles of physics and mechanics to a model geometry that represents the real geometry to be blasted. The application of mechanics makes the model dependent on measurable material properties. The models are developed with frequent comparison of computer simulations and field data. Some of the models are listed in Table 2.6. Use of these models on many real problems has shown many strengths and weaknesses (ISRM, 1992).

Among these computer models, SABREX has been used in India (Bhushan and Srilari, 1990). This model requires rock properties such as density, compressive strength, tensile strength, Young's modulus, Poisson's ratio, P- and S-wave velocities, crack attenuation factor, and Rock quality factor. Blast geometry is defined by blasthole diameter, bench height, hole depth, burden, spacing, drilling pattern, and delay hook-up. Explosive properties are calculated by a companion program called CPEX, a non-ideal detonation code. Given the chemical composition of the explosive, its density and detonation velocity at three different
charge diameters, CPEX calculates explosives performance at any diameter and confinement. The essential properties of explosives used are density, detonation velocity, detonation pressure, explosion pressure, shock energy, and three coefficients of pressure-volume curve. CPEX creates a file on calculated explosive properties which can be directly assessed by SABREX.

Table 2.6 Computer programs for blast design and analysis

<table>
<thead>
<tr>
<th>Program</th>
<th>Developed by</th>
</tr>
</thead>
<tbody>
<tr>
<td>3x3o-PRO</td>
<td>JKMRC, Australia (Cameron et al, 1991)</td>
</tr>
<tr>
<td>SABREX</td>
<td>ICI Explosives Group (Kirby et al, 1987)</td>
</tr>
<tr>
<td>BLASPA</td>
<td>Favreau (1980)</td>
</tr>
<tr>
<td>DMC</td>
<td>Sandia National Laboratory, USA</td>
</tr>
<tr>
<td></td>
<td>(Taylor and Preece, 1989)</td>
</tr>
<tr>
<td>SOROBLAST</td>
<td>Lulea University, Sweden (Kou and Rustan, 1993)</td>
</tr>
<tr>
<td>DYNOVIEW &amp; BLASTEC</td>
<td>Dyno Nobel, Inc (Hopler, 1994)</td>
</tr>
</tbody>
</table>

A reference blast is first defined and all comparisons are made with respect to this blast. The cost is calculated on the basis of input geometry, unit cost of explosives, accessories and drilling. Two types of fragmentation calculations are carried out by SABREX: absolute and relative. Absolute fragmentation is in terms of size distribution of broken rock. Fragmentation at the toe level, along the charge column and in the stemming region is given in relative terms. For relative comparison, a value of 100 is given for the reference blast. Any changes in fragmentation due to changes in geometry or charges are then calculated as relative numbers compared to the reference blast. A value less than 100 indicates poor breakage, and more than 100 means improved fragmentation.

Similar to fragmentation, throw is calculated in absolute and relative terms. For absolute prediction of throw, it requires additional inputs like angle of repose of the broken rock, number of rows blasted and delay between the rows. Given these inputs, velocity of rock movement is calculated. Flyrock prediction is only in relative terms.

If the measured fragmentation and velocity of rock movement is available, the model can be calibrated and used as a tool for blast optimisation. The predicted results are displayed in colour graphs and in tables.

Due to complexity of rock structure, difficulty in collecting data and a lack of understanding of the dynamic response of rock to the explosive loading, any predictions made
by the model should be treated as a guideline instead of an accurate prediction (Sarma, 1994).

2.3.7 Expert Systems

Knowledge based systems or expert systems are sophisticated, interactive computer programs which use all forms of knowledge (thumb rules, empirical formulae, engineering judgement and past experience) in some narrow problem domain to solve a complex problem in that domain. Problems are solved by means of an information containing rules and data from which inferences are drawn on the basis of human experience and previously encountered problems. Expert systems have been applied to blasting problems (Schech et al., 1987; Cheimanoff et al., 1990; Ghose et al., 1990, Jiang and Little, 1990, Paul and Gershon, 1989).

A group of researchers (Schech et al., 1987) have developed a surface mine blast design and consultant system to help with blast design and blast vibration analysis. The software consists of two modules: one uses theoretical and empirical formulae and procedures to design a blast based on user supplied geological and mechanical data, while the other is an expert system that analyses the blast vibration problems and recommends remedial action using knowledge based rules.

Jiang and Little (1990) attempted to develop a knowledge based system to assist the user with the selection of blast technique and blast design process for open pit gold mining. It was meant to select the appropriate blasting strategy for ore/waste or high grade/low grade ore separation, and predict fragmentation, recovery, and cost.

Ghose et al. (1990) developed a computer model called BESTPOL (Blasting Expert for Surface Mines Through Process Of Learning) using both case based reasoning and rule based reasoning. The basic retrieval process of case histories has been organised to search for relevant case by a traversal strategy, which is then subjected to appropriate modifications using rule-based reasoning. Each problem solving experience also contribute to refinement, modification, and confirmation of the problem-solving knowledge already available.

The diagnosis of a blasting problem relies heavily on personal knowledge and experience of an expert to reach a conclusion. If this expertise is embedded in an expert system, it could prompt the user to describe the blast results and the computer could reach conclusion based on the stored information.
Berta (1990) has proposed a concept of total energy balance and demonstrated its applications in blast design. According to him, the energy transmitted to the rock is distributed as follows:

- a) Fracturing \textit{in situ} < 1 \%
- b) Breakage 15 \%
- c) Displacement 4 \%
- d) Crushing in the vicinity of the hole 15-20 \%
- e) Flyrock < 1 \%
- f) Deformation of solid rock beyond the shot < 1 \%
- g) Ground vibrations 40 \%
- h) Air overpressure 38-39 \%

The transfer of energy to the rock is a function of both the characteristics of the explosive and the rock. The energy transferred is influenced by impedance factor ($\eta_1$) and coupling factor ($\eta_2$)

\[ \eta_1 = 1 - \frac{(I_e - I_r)^2}{(I_e - I_r)^2} \]  
\[ \eta_2 = \frac{1}{e^{d/dc} - (e-1)} \]

where

- $I_e$ = impedance of explosive = density of explosive x detonation velocity (kg/m² s)
- $I_r$ = impedance of rock = density of rock x seismic wave velocity (kg/m² s)
- $d$ = hole diameter (m)
- $dc$ = charge diameter (m)

Powder factor and burden are calculated by

\[ q = \frac{s \varepsilon d}{\eta_1 \eta_2 \eta_3 \varepsilon} \]  
\[ B = dc \sqrt{\frac{\pi \rho e}{4 q}} \]

where

- $q$ = powder factor (kg/m³)
In order to facilitate the use of formulae 2.9 and 2.10, Berta (1990) has given the values of \( \rho_e \), \( I_e \) and \( \varepsilon \) for some explosives and the values of \( \rho_r \), \( I_e \), \( \varepsilon_{ss} \) and seismic velocity for common rock types.

Equations 2.7 and 2.8 indicate that the energy transfer is maximum when \( I_e = I_r \) and \( d = d_c \). From Equation 2.9 it is evident that when all other parameters are equal, powder factor increases with increasing new surfaces created by breakage or with the decrease in the size of the broken rock.

2.3.9 Powder Factor/ Energy Factor

One of the approaches to blast design is based on powder factor value derived by experience (Bhushan and Srnari, 1990). Depending on the diameter and depth of holes, quantity of explosives per hole is determined. Using the powder factor value, volume of rock that can be broken per hole is calculated. From that volume, burden and spacing are derived. This approach is considered not suitable for blast design (Lang and Favreau, 1972; Bhushan and Srnari, 1990).

A similar approach but substituting powder factor by energy factor has been applied for blasthole pattern expansion using high energy explosives (Kate and Sarma, 1996). In this method, the existing drillhole pattern, powder factor, and explosives used are recorded. From the available energy of explosives and powder factor, the energy factor for the existing pattern is calculated. Considering this as the base, possibility of pattern expansion to about 20% using high energy explosives is worked out. A few case studies are also presented to demonstrate the benefits of high energy explosives in Indian mines.

With the development of new concepts such as partition of energy and explosive rock interaction, the energy factor approach to pattern expansion is questionable. In most strata, blasthole patterns for more powerful explosives cannot be enlarged to the stage where
reduction in mining costs outweighs the increase in the cost of explosives, with the exception of a very few hard/strong formations (ICI, 1993)

2.4 CONCLUSIONS FROM THE LITERATURE REVIEW

The most important parameters that need to be addressed in a blast design are blasthole diameter, burden, spacing, hole depth, stemming, powder factor, initiation sequence and delay timing. Despite many years of fundamental work aimed at describing the role of rock mass properties in blasting, these properties are not satisfactorily included in blast design. There is a need for characterising the site by simple and measurable parameters and incorporate them in the design of blasts. The characteristics of explosives alone are inadequate for selection of explosives as the performance depends on the rock mass properties and application conditions.

Since about 78-79% of the energy transferred to the rock mass is dissipated in the form of ground vibration and air overpressure, it is essential to improve the utilisation of explosive energy in rock breakage.

Some of the approaches to blast design such as trial and error, cratering, powder factor/energy factor are not suitable for large scale blasts in surface mines. The formulae to calculate burden and powder factor based on the energy balance approach involve rock and explosive properties that cannot be easily determined; hence they are hardly used in Indian mines. The empirical method continues to be the most common method for calculations of initial design parameters. Computer simulation is a very promising method but it does not incorporate the blast design process and hence cannot be considered as a complete approach. Available computer models for prediction of fragmentation, muckpile profile and vibration can assist the blast designer but their role in the blast design process has to be defined. Thus, an integration of empirical, computer modelling, and instrumented field trials appears to be the state-of-the-art in blast design. However, various steps in the blast design process are not clearly defined. The empirical formulae developed elsewhere have not been verified for Indian conditions. They may need some modifications or new relationships have to be developed.

Realising the need for a suitable approach for blast design, a guided approach to blast design is proposed and described in the next chapter.