DETERMINATION OF BLAST DESIGN PARAMETERS

Methods to determine blasthole diameter, burden, spacing, stemming, hole depth and powder factor are presented in this chapter. These methods have been developed based on the field studies carried out by the author. Other design parameters such as bench height, hole inclination and subgrade drilling are briefly incorporated from the literature. The influence of initiation sequence and delay timing on blast performance were investigated using a Sequential Blasting Machine. Based on the results and analysis of several case studies, typical diagrams illustrating the problems and solutions related to the initiation sequence and delay timing are also presented.

By comparing the existing blasting practices with the recommended ones and with the international practices, some areas have been identified which can lead to better blast performance at surface mines.

6.1 BENCH HEIGHT

The bench height is decided by mine planners depending on the local geology, production rate, size and type of the loading equipment, mining regulations, and slope stability considerations. For a blasting engineer, the bench height is usually a fixed parameter. Sometimes, the bench height may vary due to topographic changes and nature of the formation.

6.2 BLASTHOLE DIAMETER

The selection of hole diameter is governed by several factors (Dick et al, 1986, Thomas, 1986) such as bench height, critical diameter of charge, drilling cost, production requirement, rock structure (block size), the required degree of fragmentation, environmental constraints, and unit cost of production. Some of these factors require larger diameter holes while others require smaller diameter. Hence, it is necessary to strike a balance and meet the conflicting requirements by selecting an optimum hole diameter.

From the detonation theory of explosives (Persson et al, 1994), hole diameter should be greater than the critical diameter of the charge. However, this is not a problem in surface mines because hole diameters are larger than the critical diameter of the charge.
Large scale operations employ larger blasthole diameter and small scale operations have smaller blasthole diameter. However, this factor does not play a deciding role because production requirement is also directly related to bench height.

The essential factors that decide the selection of hole diameter include bench height, environmental constraints, rock structure (block size) and cost. The first factor is described in detail while the others are discussed briefly. The steps involved in the selection and evaluation of hole diameter are described below. Steps 1 to 3 are to be considered for initial design and Step 4 for optimisation (Fig 31).

**Step 1: Compatibility between hole diameter and bench height**

Practical experience shows that hole diameter increases with an increase in bench height (Fig 61). Large diameter holes in low benches and small diameter holes in high benches are incompatible. Ash (1990) states that the hole diameter should be no larger than 0.015 times the bench height, the limiting point at which the burden dimension will not impose excessive stiffness on the burden rock. A thumb rule, given by Atlas Powder Company (1987), states that the blasthole diameter in inches should be approximately 1/10th of the bench height in feet.

The author conducted blasting studies in both 'compatible' and 'incompatible' conditions. The blast results were different depending on whether the hole diameter and bench height were compatible or not. Fragmentation, drillhole utilisation, powder factor and yield were assessed for different combinations of hole diameter and bench height. Drillhole utilisation is defined as the ratio of charge length to uncharged length, expressed in percentage. Yield is the volume of rock (burden x spacing x bench height) divided by hole depth. The differences in the results could be categorised into three zones, that is Zone A, Zone B and Zone C in Fig 61. The qualitative ratings for the different combinations are listed in Table 61 which indicates that there is little or no benefit when hole diameter is too large (Zone A) or too small (Zone C). A range of acceptable hole diameters has been established by comparison of the results from the different zones and the condition for compatibility is defined by

\[
d_{\text{min}} = 10H
\]

\[
d_{\text{max}} = 16.66H + 50
\]

where

- \(d_{\text{min}}\) = minimum hole diameter (mm)
- \(d_{\text{max}}\) = maximum hole diameter (mm)
- \(H\) = bench height (m)
a) Data collected during 1986 - 89

b) Data collected during 1990 - 93

Fig 6.1 Compatibility between blasthole diameter and bench height
If the drilling machine/bits suitable for the calculated diameter are not available, it may be rounded off to the nearest available size.

**Table 6.1 Blast evaluation for different combinations of hole diameter and bench height**

<table>
<thead>
<tr>
<th>Parameters</th>
<th>Zone A</th>
<th>Zone B</th>
<th>Zone C</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fragmentation</td>
<td>Poor</td>
<td>Good</td>
<td>Good</td>
</tr>
<tr>
<td>Drillhole utilisation (%)</td>
<td>Poor</td>
<td>Good</td>
<td>Good</td>
</tr>
<tr>
<td>Powder factor (kg/m³)</td>
<td>Excess</td>
<td>Normal</td>
<td>Normal</td>
</tr>
<tr>
<td>Yield (m³/m)</td>
<td>Good</td>
<td>Good</td>
<td>Poor</td>
</tr>
</tbody>
</table>

For a certain amount of charge weight per hole, an excessively large hole diameter makes the charge length very short and reduces the drillhole utilisation. On the other hand, if the hole diameter is too small, the charge length will be too long and sufficient room will not be left for stemming. This means drillhole pattern has to be reduced which results in poor yield.

For a given bench height, Equation 6.1 recommends slightly larger hole diameter than given by the thumb rule (Atlas Powder Company, 1987) and Equation 6.2 recommends slightly larger hole diameter than given by Ash (1990). Compared to the international practices, the hole diameters used in Indian mines are slightly larger for the comparable bench heights.

In relation to blasthole diameter used, bench height is termed low and high (Langefors and Kihlström, 1963, Pecsson et al, 1994) and the combination of hole diameter and bench height may be categorised into two zones. In this study, the combination of hole diameter and bench height has been divided into three zones considering their influence on blast results (Table 6.1). The division into different zones is also used later for burden calculation.

Fig 6.1 illustrates the maximum and minimum hole diameter recommended for surface mines along with the plots of 50 sets of data collected from different mines during 1986-89 and 48 sets collected during 1990-93. The points in zone A show that the existing hole diameters are larger than the required. The points in zone C show that the existing hole diameters are smaller than the required. It may be noted that only few cases exist where blastholes diameters are small (Zone C, Fig 6.1), but several cases when they are large. Some of the practical problems which forced the field engineers to work with larger hole diameter (Zone A, Fig 6.1) are:...
• Due to ground vibration problem and restriction on maximum charge per delay or charge per hole, originally planned bench height was reduced. Purchase of a new machine for drilling smaller diameter holes was not possible and the use of existing drilling machine continued.

• Drilling problems also forced the field engineers to select larger diameter holes. For example, 115 mm diameter holes were drilled in complex formation (hard and highly jointed) in an iron ore mine. This was a semi-mechanised mine where the bench height was restricted to 5 m for safety reasons. When drill tools used to get jammed very frequently and drilling rate could not cope up with the required drilling meterage, 152 mm diameter holes were used (Singh and Singh, 1996).

• In coal mines, bench height is restricted by the thickness of coal and parting. It may not be always possible to select hole diameters compatible to the thickness of the strata.

A global survey of the blasthole diameter and bench height (Table 6.2) indicated that a smaller hole diameter than the recommended one is used if the environmental issues are very sensitive and/or very small size of fragments are required. A larger diameter than the recommended one is used if very high capacity loading equipment is deployed and/or the rock is either soft or highly fractured.

**Table 6.2 Global survey of blasthole diameter and bench height**

<table>
<thead>
<tr>
<th>Mine</th>
<th>Bench height (m)</th>
<th>Hole diameter (mm)</th>
<th>Reference</th>
<th>Recommended hole diameter (mm)</th>
<th>Whether compatible or not?</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ok Tedi Porphyry Copper Mine, Papua New Guinea</td>
<td>15</td>
<td>251</td>
<td>Howell and Gigmai (1992)</td>
<td>150-300</td>
<td>Yes</td>
</tr>
<tr>
<td>Iron ore, Turkey</td>
<td>9.0</td>
<td>194</td>
<td>Bilgin, 1991</td>
<td>90-200</td>
<td>Yes</td>
</tr>
<tr>
<td>Rossmore Uranium Mine, Namibia</td>
<td>15</td>
<td>381</td>
<td>Hunter et al (1990)</td>
<td>150-300</td>
<td>No</td>
</tr>
<tr>
<td>Sydvaranger's Open Pit Iron Ore Mine, Norway</td>
<td>14</td>
<td>311</td>
<td>Nielsen (1983)</td>
<td>140-283</td>
<td>No</td>
</tr>
<tr>
<td>Limestone Quarry, Canada</td>
<td>6.3</td>
<td>75</td>
<td>Mahanty and Chung (1990)</td>
<td>63-155</td>
<td>Yes</td>
</tr>
<tr>
<td>A Stone Quarry near Brisbane, Australia</td>
<td>14</td>
<td>102</td>
<td>Cox and Cotton (1996)</td>
<td>140-283</td>
<td>No</td>
</tr>
<tr>
<td>Coeur Rochester Gold Mine, USA</td>
<td>7.6</td>
<td>165</td>
<td>Zhang et al (1994)</td>
<td>76-177</td>
<td>Yes</td>
</tr>
<tr>
<td>Island Copper Mine, Canada</td>
<td>12</td>
<td>250</td>
<td>Grundstrom (1994)</td>
<td>120-250</td>
<td>Yes</td>
</tr>
<tr>
<td>Mt Whaleback Open Cut Iron Ore Mine, Australia</td>
<td>15</td>
<td>380</td>
<td>Bollaar (1987)</td>
<td>150-300</td>
<td>No</td>
</tr>
</tbody>
</table>

80
**Step 2: Environmental constraints**

When blasting operations are carried out near populated areas, the ground vibration needs to be controlled. The most effective way of controlling the ground vibration is by reducing the charge weight per delay. Since the method of dividing a charge into different delays within a hole makes the initiation sequence complicated and requires the use of in-hole delays, the charge per hole should not exceed the maximum permissible charge per delay. This limitation sometimes restricts the blasthole diameter to decrease the charge per hole. The smaller diameter is also preferable to minimise the risk of flyrock and complaints about air overpressure.

**Step 3: Rock structure (block size)**

Large blocks of rock are not usually broken unless they are intersected by the charge column. The chances of boulder formation can be greater or lesser depending on the block size in the rock mass and blasthole pattern. In a formation with large blocks, smaller diameter holes (Equation 6.1) are recommended to be drilled at closer burden and spacing. On the other hand, larger diameter (Equation 6.2) with larger burden and spacing will be more suitable in a highly fractured rock, with small blocks.

**Step 4: Minimum cost of production**

The hole diameter should be finally selected after taking into account and by minimising the total cost of drilling, blasting, loading, hauling and crushing. If the hole diameter is small, the costs of drilling, priming and initiation are high due to increased number of holes. If it is too large, increased burden and spacing will lead to coarse fragmentation and the costs of loading and crushing will be high.

**6.3 BLASTHOLE INCLINATION**

Blastholes can be drilled vertical or inclined. There are advantages and disadvantages in vertical and inclined drilling (Gregory, 1984, Konya and Walter, 1990). The advantages of inclined drilling are less backbreak, less problems at the grade, more throw, better fragmentation, and stable face. The disadvantages in inclined drilling are difficulties in drilling.

Most of the Indian mines use vertical holes because it is easier and faster to drill. Though drills have facility for drilling inclined holes, their productivity is very low in inclined drilling. Considering the practical aspects, it is preferable to drill vertical holes.
6.4 BURDEN

6.4.1 Calculation of Burden

A large number of parameters influence burden. Table 6.3 summarises the parameters considered by various investigators. The hole diameter or charge diameter is included in all burden formulae. For bulk explosives, charge diameter is equal to the hole diameter. When cartridge explosives are used, the explosive does not fill the entire cross-sectional area of the blasthole, so the charge diameter is therefore less than the hole diameter. The bench height is included only in some formulae. The parameters such as hole depth and charge length are the derivatives of the bench height. Rock and explosive properties are included either directly or indirectly in the form of constants.

Table 6.3 Parameters considered for burden calculation

<table>
<thead>
<tr>
<th>Investigator(s)</th>
<th>Parameters considered</th>
</tr>
</thead>
<tbody>
<tr>
<td>Andersen (1952)</td>
<td>a Hole diameter</td>
</tr>
<tr>
<td></td>
<td>b Hole depth</td>
</tr>
<tr>
<td>Pearse (1955)</td>
<td>a Hole diameter</td>
</tr>
<tr>
<td></td>
<td>b Detonation pressure</td>
</tr>
<tr>
<td></td>
<td>c Dynamic tensile strength</td>
</tr>
<tr>
<td>Fraenkel (1952)</td>
<td>a Hole diameter</td>
</tr>
<tr>
<td></td>
<td>b Hole depth</td>
</tr>
<tr>
<td></td>
<td>c Charge length</td>
</tr>
<tr>
<td>Langelors and Kihlstrom (1963)</td>
<td>a Hole diameter</td>
</tr>
<tr>
<td></td>
<td>b Spacing to burden ratio</td>
</tr>
<tr>
<td></td>
<td>c Density of the explosive in the hole</td>
</tr>
<tr>
<td></td>
<td>d Weight strength of explosive</td>
</tr>
<tr>
<td></td>
<td>e Fixation factor</td>
</tr>
<tr>
<td>Konya and Walter (1990)</td>
<td>a Hole diameter</td>
</tr>
<tr>
<td></td>
<td>b Specific gravity of rock</td>
</tr>
<tr>
<td></td>
<td>c Specific gravity of explosive</td>
</tr>
<tr>
<td>Ruslan (1990)</td>
<td>a Hole diameter</td>
</tr>
<tr>
<td>Berta (1990)</td>
<td>a Charge diameter</td>
</tr>
<tr>
<td></td>
<td>b Specific explosive consumption</td>
</tr>
<tr>
<td></td>
<td>c Density of explosive</td>
</tr>
</tbody>
</table>

One of the factors that has been considered in this study for calculation of burden is the blasthole diameter because it controls loading density, the formation of cracks and breakage around a blasthole. Another factor is bench height, considering the flexural rupture theory applied to blasting (Ash, 1973). Using only the blasthole diameter and bench height one can
arrive at an interval estimate of burden. A point estimate of the burden from this interval is determined by rock mass properties. This method involves the following five steps.

**Step 1: Burden from the blasthole diameter**

Most of the relationships between the burden and hole diameter are linear. The non-linear relation (Rustan, 1990) is possible when bench heights are not scaled up as the diameter increases. At this stage of calculation, it is proposed to calculate the burden from the hole diameter using the following equation

\[ B = C_1 d \]  

where

- \( B \) = burden (m)
- \( d \) = hole diameter (m)
- \( C_1 \) = constant of proportionality (burden to hole diameter ratio)

The minimum and maximum values of \( C_1 \) reported by various investigators are given in Table 6.4. For all types of rock and explosives, the value of \( C_1 \) is more or less established and varies between 15 and 40. Too small a value of \( C_1 \) will cause excessive air overpressure and flyrock whereas too large a value will result in poor fragmentation, toe, and excessive ground vibration. Fig 6.2 shows histograms of burden to hole diameter ratios as observed in different mines. Fig 6.2(a) is a plot for 51 sets of blast data collected during 1986-1989 and Fig 6.2(b) represents 108 sets collected during 1990-93. From these histograms, it is seen that the burden to hole diameter ratio in Indian mines varies from 15 to 40 except for a few cases. The ratios are usually within the limits reported by various investigators (Table 6.4). However, \( C_1 \) was found to be less than 15 when the hole diameter was relatively large (Zone A in Fig 6.1) and greater than 40 when the hole diameter was small (Zone C in Fig 6.1).

### Table 6.4 Minimum and maximum values of constant \( C_1 \)

<table>
<thead>
<tr>
<th>Investigator(s)</th>
<th>Minimum value</th>
<th>Maximum value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hagan (1983)</td>
<td>20</td>
<td>35</td>
</tr>
<tr>
<td>ICI (1993)</td>
<td>19</td>
<td>43</td>
</tr>
<tr>
<td>Konya and Walter (1990)*</td>
<td>15</td>
<td>37</td>
</tr>
<tr>
<td>Dick et al (1986)</td>
<td>20</td>
<td>40</td>
</tr>
</tbody>
</table>

* Considering the maximum influence of explosive density and rock density on burden (Rustan, 1990)
Fig 6.2 Histograms of burden to blasthole diameter ratio

N = 51

a) Data collected during 1986 - 89

N = 108

b) Data collected during 1990 - 93
At smaller burden, stress wave energy is better utilised in causing spalling fractures. However, gas energy does not get enough time before it is vented to cause radial cracks and their bifurcation. At larger burdens, the role of stress waves is considerably decreased in fragmentation but it is then transferred to the surrounding rock, thereby causing increased vibrations. At optimum burden, fragmentation is good, throw is acceptable and vibrations are low, that is, energy utilisation is better in rock fragmentation.

In order to narrow down the range of calculations, burden can be calculated using the following values of $C_1$ for different zones (Fig 6.1):

- $C_1 = 15-35$ for Zone A
- $C_1 = 20-35$ for Zone B
- $C_1 = 20-40$ for Zone C

**Step 2: Burden from the bench height**

It is also proposed to calculate burden from the bench height using the following equation

$$B = C_2H$$

where

- $B =$ burden (m)
- $H =$ bench height (m)
- $C_2 =$ constant of proportionality (burden to bench height ratio)

For satisfactory blasts, $C_2$ varies from 0.25 to 0.50 (Adhikari et al, 1990). It was determined by analysing the burden and corresponding bench height data from several surface mines. The interest to analyse the data was created after going through the flexural rupture theory (Ash, 1973). The reciprocal of $C_2$ is equivalent to the stiffness ratio given by Konya and Walter (1990). They recommended that the stiffness ratio for a good blast design should be between 2 and 4. There is a good agreement between these two works, which were done independently and perhaps by different approaches.

Fig 6.3 shows histograms of burden to bench height ratio as observed in different mines by the author. Fig 6.3(a) shows a plot for 77 sets of blast data collected during 1986-1989, whereas Fig 6.3(b) represents 108 sets collected during 1990-93. It may be noted that the actual practice deviates a little from the recommended values. If the selected hole diameter is large for the given bench height, operators try to increase burden as much as possible which...
a) Data collected during 1986-89

b) Data collected during 1990-93

Fig 6.3 Histograms of burden to bench height ratio
increases the ratio greater than 0.5. If the selected hole diameter is small, the burden to bench height ratio may be less than 0.25 because blasthole pattern will be limited by the charge that can be accommodated in the hole. The frequency of burden to bench height ratio greater than 0.5 is more because there are a number of cases in Zone A (Fig 6.1). The frequency of this ratio smaller than 0.25 is less because there are few cases in Zone C (Fig 6.1).

**Step 3: Burden for a given blasthole diameter and bench height**

An attempt was made to derive an empirical equation for calculation of burden as a function of the hole diameter and bench height. Although it was expected to be a generalised equation, it remained site-specific. Finally a logistic method was evolved which served its purpose. The method is different from the existing ones and is described below.

Burden calculated from the hole diameter gives a range. Similarly burden calculated from the bench height gives another range. When the two ranges are compared, three cases are possible as illustrated in Fig. 6.4.

1) The burden range from the hole diameter may have greater values than the range calculated from the bench height (Fig 6.4a) but the two ranges overlap.
2) The burden range from the hole diameter is smaller than the range calculated from that of bench height (Fig 6.4b) but the ranges overlap.
3) The ranges do not overlap when the selection of the hole diameter is inappropriate.

![Fig 6.4 Interval estimate of burden from hole diameter and bench height](image)

As illustrated in Fig 6.4, the common (overlapping) range is narrower than the individual range computed either from hole diameter or bench height. The common range eliminates the chances of inappropriate burden for a particular situation. If the two ranges do not overlap, then this combination of blasthole diameter and bench height should not be used even if it satisfies the conditions set by Equation 6.1 and Equation 6.2.

Given a hole diameter of 165 mm and bench height of 12 m, the burden with regard to the hole diameter may vary from 3.3 to 5.7 m, while with regard to the bench height it may vary...
from 3 to 6 m. Hence, the minimum and maximum burden for the given combination of hole diameter and bench height is 3.3 m and 5.7 m, respectively.

**Step 4: Initial value from the burden range**

A suitable burden should be chosen from the common range considering explosives and rock properties. A simple method that does not require the detailed knowledge of rock and explosives properties is described below.

The burden range is divided into three equal intervals, the interval with smallest to highest values corresponds to hard, medium and soft rocks, respectively. For example, if the burden range of 3.3 - 5.7 m is divided into three equal intervals, burden can vary from 3.3 to 4.1 m for hard, from 4.1 to 4.9 m for medium and from 4.9 to 5.7 m for soft rock. The mean value of each interval is the burden for the corresponding rock.

**Step 5: Correction for the influence of the block size**

The burden calculated from the previous step is to be multiplied by a correction factor to take into account the influence of block size. Following the minimum uncertainty principle (Bieniawski, 1992) the correction factor for different block size is restricted to 0.9-1.1 (Table 6.5), though it may be higher or lower in very favourable or very unfavourable conditions. By introducing the correction factor, more weightage is given to the influence of joints than the strength of rocks. For similar strength of rock, burden has to be increased if the block size is small and decreased if the block size is large. The burden calculated from Step 4 might go out of the burden range determined in Step 3. In that case, the corrected burden should be restricted to the maximum or minimum burden specified by Step 3.

**Table 6.5 Correction factor for the block size**

<table>
<thead>
<tr>
<th>Number of joints per cubic metre</th>
<th>Block size</th>
<th>Correction factor</th>
</tr>
</thead>
<tbody>
<tr>
<td>&lt; 3</td>
<td>Large</td>
<td>0.9</td>
</tr>
<tr>
<td>3 to 10</td>
<td>Medium</td>
<td>1.0</td>
</tr>
<tr>
<td>&gt; 10</td>
<td>Small</td>
<td>1.1</td>
</tr>
</tbody>
</table>

In order to simplify the procedure for burden calculation a flow-chart is presented in Fig 6.5. This flow chart was used to write a computer program.
Calculates burden from hole diameter and bench height

Calculation of interval estimate of burden

Point estimation of burden from rock properties

Burden correction for influence of joints

Fig 6.5 Flow chart for calculation of burden
6.4.2 Comparison With Other Burden Formulae

The proposed method for calculation of burden is similar to that of Langefors and Kihlstrom (1963) and Konya and Walter (1990) in that all are empirical and all consider the blasthole diameter as one of the important parameters. Unlike other methods, parameters such as the spacing to burden ratio, powder factor, and number of rows, which are determined after the burden, do not enter in the proposed step-by-step procedure for burden calculation.

Langefors and Kihlstrom (1963) have considered the rock properties through a rock constant. Konya and Walter (1990) have used the density of rock and two correction factors for rock deposit and geologic structure. The rock properties such as specific surface energy used by Berta (1990) are not common and difficult to determine. The proposed method considers the compressive strength and a correction factor for the block size in the rock mass.

Langefors and Kihlstrom (1963) consider the weight strength of an explosive which is now obsolete (Persson et al., 1994). Konya and Walter (1990) consider the density of an explosive which may mislead the calculation with low density explosives. Since blasting performance cannot be rated by one or two properties of explosives, it may not be justified to include just one or two properties for the calculation of burden. The calculated burden is applicable for the selected explosive as discussed in Section 4.2.

Konya and Walter (1990) have used a correction factor for the number of rows, so do Persson et al. (1994). They suggest to reduce the burden as the number of rows increases. In multiple row blasts, the harder it will be to fragment and displace the broken rock mass in front of each new row. However, this can be controlled by providing adequate delay intervals for later firing charges.

The burden formula of Langefors and Kihlstrom (1963) is valid only for blasthole diameters in the range of 30-89 mm (Rustan, 1990). The method suggested by Konya and Walter (1990) gives larger burden for certain conditions (Zone A, Fig. 6.1). According to Berta (1990), burden decreases as the powder factor increases whereas the powder factor is almost constant for burden from 1 to 10 m, usually used in blasting (Rustan, 1990, Persson et al., 1994). At small burdens, the powder factor increases because the specific surface area is increased. At very large burdens also, the powder factor increases because the volume increases rapidly and more and more energy is needed for the throw of the fragmented material.
6.4.3 New Burden When Hole Diameter/ Bench Height is Changed

A change in hole diameter or bench height will necessitate a change in the burden. Usually, trials are conducted to optimise the burden for the changed condition. An attempt is made to evolve a suitable method to calculate the new burden so that the number of field trials can be reduced.

If the burden has been optimised and later if the hole diameter is changed but the rock and explosives remain unchanged, a new burden can be determined using the theorems of proportionality. From Equation 6.3 it is seen that burden is proportional to the hole diameter. For the previous hole diameter we can write

\[ B_1 = C_1 d_1 \]

For the new hole diameter

\[ B_2 = C_1 d_2 \]

The constant of the proportionality \( C_1 \) is the same because rock properties and explosive properties remain the same. Then

\[ \frac{B_2}{B_1} = \frac{d_2}{d_1} \]

or

\[ B_2 = \frac{d_2}{d_1} \times B_1 \]  

(6.5)

where

- \( B_1 \) = previous burden
- \( B_2 \) = new burden
- \( d_1 \) = previous hole diameter
- \( d_2 \) = new hole diameter

This equation is same as that given by Konya and Walter (1990) but differs from Equation 6.6 which is given by Corsby and Pinco (1992)

\[ B_2 = \left( \frac{d_2}{d_1} \right)^{2/3} \times B_1 \]  

(6.6)

The powers of \( d_2/d_1 \) in Equation 6.5 and 6.6 do not agree and the calculated burden will be different. Both of these equations do not consider the bench height.
Using Equation 6.4, another equation can be derived for the change in bench height, with other parameters being the same:

\[
\begin{align*}
B_1 &= C_2 H_1 \\
B_2 &= C_2 H_2 \\
\frac{B_2}{B_1} &= \frac{H_2}{H_1} \\
or \quad B_2 &= \frac{H_2}{H_1} \times B_1
\end{align*}
\]  

(6.7)

where

\[
H_1 = \text{previous bench height} \\
H_2 = \text{new bench height}
\]

The value of \( C_2 \) remains constant as all other parameters remain unchanged. Equation 6.7 considers the bench height but does not consider the hole diameter. A need was then felt for a relationship which incorporates both hole diameter and bench height. The basis for calculation is already available from Section 6.4.1.

For the previous combination of hole diameter and bench height, the interval estimate of burden is given by \( B_{\text{min1}} \) and \( B_{\text{max1}} \). For the new combination, the interval estimate of new burden is given by \( B_{\text{min2}} \) and \( B_{\text{max2}} \). In addition, an optimum burden for the previous condition is also known. This information is sufficient to derive a relationship for calculation of new burden using the method of linear interpolation. As illustrated in Fig. 6.6, \( B_{\text{min1}} \) corresponds to \( B_{\text{min2}} \) and \( B_{\text{max1}} \) to \( B_{\text{max2}} \). Since the optimum burden for the previous combination lies within the specified interval, the burden for the changed condition should also lie within another specified interval, which means \( B_2 \) can be estimated by

\[
B_2 = B_{\text{min2}} + \frac{B_{\text{max2}} - B_{\text{min2}}}{B_{\text{max1}} - B_{\text{min1}}} (B_1 - B_{\text{min1}})
\]

(6.8)

where

\[
\begin{align*}
B_{\text{min1}} &= \text{minimum calculated burden for the previous combination} \\
B_{\text{max1}} &= \text{maximum calculated burden for the previous combination} \\
B_1 &= \text{optimum burden for the previous combination} \\
B_{\text{min2}} &= \text{minimum calculated burden for the new combination} \\
B_{\text{max2}} &= \text{maximum calculated burden for the new combination} \\
B_2 &= \text{calculated burden for the new combination}
\end{align*}
\]
Equation 6.5 is simple and gives reasonable values provided the hole diameter is compatible with the bench height. When the hole diameter is large, it gives higher values for the new burden. Equation 6.7 is also partially successful in calculating the new burden as it does not consider the hole diameter. Equation 6.8 is applicable for a wide range of situations because it considers both bench height and hole diameter. This equation also allows to calculate a new burden when there is a change in the hole diameter or in the bench height or in both.

6.5 SPACING

Spacing is mainly a function of burden (Pugliese, 1972, Gregory, 1984). It is also governed by initiation pattern (Konya and Walter, 1990, Dick et al., 1986) and orientation of joints (Bhandari and Badal, 1990). Therefore there are three major factors that decide spacing. They are burden, orientation and spacing of joints, and initiation pattern.

Spacing is calculated from the established optimal spacing to burden (S/B) ratio for bench blasting situations, which varies between one and two (Ash, 1990, Pugliese, 1972, Gregory, 1984). Wide spacing technique (Langefors and Kihlstrom, 1963) was successful in some operations when the orientation of joints was favourable, and unsuccessful in other operations when joints were unfavourable (Bhandari and Badal, 1990). Keeping the minimum uncertainty principle (Bieniawski, 1992), S/B can vary between 1 and 2. S/B < 1.0 is not recommended because it causes premature splitting of holes and early release of gaseous pressure, losing the contribution of gaseous energy in rock fragmentation. S/B > 2.0 is also not recommended because it may result in incomplete breakage between the holes and poor fragmentation.

The optimal S/B ratio has also been studied in model and half scale blasting tests (Langefors and Kihlstrom, 1963, Bhandari et al., 1975, Bergmann, 1983, Bhandari and Badal, 1990, Dojcar, 1991). A summary of their findings is given in Table 6.6. The optimal ratio is greater than 2 and does not agree with the S/B ratio established from the field studies. In model blasts, optimal ratio was determined mostly for optimal fragmentation without considering the environmental impacts of blasting which is an important consideration in field blasting. The model blasts were conducted with limited number of holes while multi-row...
blasting is common in the field. However, the results of the model blasts correlate with the effective S/B ratio (Fig. 2.4). Hence, wide spacing technique for bench blasts is recommended for effective spacing to burden ratio and not for drilled spacing to burden ratio.

Table 6.6 Optimal spacing to burden ratio according to model and half scale blasts

<table>
<thead>
<tr>
<th>Investigator(s)</th>
<th>S/B range studied</th>
<th>Optimal S/B ratio regarding fragmentation and breakage at the toe</th>
</tr>
</thead>
<tbody>
<tr>
<td>Langefors and Khilstrom (1963)</td>
<td>1.0-8.0</td>
<td>Larger the S/B ratio finer the fragmentation for staggered blasthole pattern. For straight blasthole pattern S/B = 2.0</td>
</tr>
<tr>
<td>Bhandari et al (1975)</td>
<td>2.0-4.0</td>
<td>S/B = 3.0 The optimum is flat for S/B &gt; 3.0</td>
</tr>
<tr>
<td>Bergmann (1983)</td>
<td>1.0-2.0</td>
<td>S/B = 2.0</td>
</tr>
<tr>
<td>Bhandar and Badal (1990)</td>
<td>1.0-4.0</td>
<td>S/B = 2 to 4, depending on the orientation of joints</td>
</tr>
<tr>
<td>Dojcar (1991)</td>
<td>0.5-8.0</td>
<td>S/B ≥ 2</td>
</tr>
</tbody>
</table>

When the field data collected by the author during 1986-1989 were analysed, the most common S/B ratio for satisfactory blasts was found to be 1.2 (Adhikan et al, 1990). Therefore the initial estimate of the spacing is equal to 1.2 times the burden. This ratio can be varied provided that the area (burden x spacing) is kept constant and spacing is maintained between 1 and 2 times the burden. The condition of constant area is introduced because blast performance is a result of combined influence of burden and spacing. An adjustment is necessary to take advantage of joint orientation and delay pattern. Furthermore, it is convenient for marking the holes if burden and spacing are specified, for example, as 4 m x 5 m instead of 3.84 m x 5.21 m.

Fig 6.7 shows the histogram of S/B ratio for a total of 108 blasts observed in eleven surface mines during 1990-93. Although most of the blasts were conducted with S/B ratio between 1 and 2, some mines allowed deviations from it. S/B > 2.0 may be explained from the wide spacing technique but it is difficult to justify for S/B < 1.0.
In order to further analyse the data, 12 records with S/B less than 1.0 and greater 2.0 were deleted because they do not fall within the recommended range. The remaining 96 records were sorted by three common hole diameters, namely 100-115 mm, 150-165 mm and 250 mm. The descriptive statistics of S/B ratio, given in Table 6.7, shows that the range as well as the mean values of S/B ratios decrease with an increase in hole diameter. When using large diameter blastholes in surface mining, geological discontinuities became an important consideration since many pronounced joints can occur between blastholes. The joints can cause fracture between holes to stop prematurely and produce coarse fragmentation. In view of greater risk of coarser fragmentation the S/B ratio is smaller for larger diameter.

**Table 6.7 Descriptive statistics of S/B ratio for commonly used blasthole diameters**

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Blasthole diameter 100-115 mm</th>
<th>Blasthole diameter 150-165 mm</th>
<th>Blasthole diameter 200-250 mm</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mean</td>
<td>1.46</td>
<td>1.42</td>
<td>1.22</td>
</tr>
<tr>
<td>Median</td>
<td>1.33</td>
<td>1.45</td>
<td>1.29</td>
</tr>
<tr>
<td>Mode</td>
<td>1.33</td>
<td>1.50</td>
<td>1.29</td>
</tr>
<tr>
<td>Standard deviation</td>
<td>0.23</td>
<td>0.17</td>
<td>0.14</td>
</tr>
<tr>
<td>Range</td>
<td>1.2-2.0</td>
<td>1.0-1.8</td>
<td>1.0-1.43</td>
</tr>
<tr>
<td>Number of observations</td>
<td>24</td>
<td>37</td>
<td>35</td>
</tr>
</tbody>
</table>
6.6 SUBGRADE DRILLING

The basic purpose of subgrade drilling is to overcome the greater resistance encountered at grade level by providing extra drill length below the grade. At the grade level, only a small portion of the stress wave is reflected in tension and therefore rock must be broken mainly in shear. When subgrade is small, the rock is not sheared off completely at the grade level. On the other hand excessive subgrade drilling causes wastage of drill meterage and explosives resulting in an increased ground vibrations and undesired shattering of the floor (Ash, 1990).

Subgrade drilling is about 0.3 times the burden (Atlas Powder Company, 1987, Langeform and Kihlstrom, 1963, Duck et al. 1986, Konya and Walter, 1990, Persson et al., 1994). However, it is not required in horizontally bedded deposits, where clear partings exist. The natural parting of the formation itself gives a clean breakage. The available literature on subgrade drilling has been accepted in the guided approach.

6.7 HOLE DEPTH

The conventional formula for calculation of depth for vertical holes is

\[
\text{Hole depth} = \text{Bench height} + \text{subgrade drilling}
\]

The conventional method does not consider allowances for hole collapse and Reduced Level (RL) correction. Blastholes are to be drilled longer than the calculated depth to give some allowances for falling of loose pieces into the hole due to natural collapse of the hole, movement of the drilling machine, explosive van and other vehicles. The excess length of the hole should be backfilled during charging. It is suggested to record and analyse hole depths over a period of time which is normally the time allowed before charging of these holes. The average fallback into the hole should be determined such that excess depths are not drilled and no redrilling is required. When the hole depth is insufficient, it is a good practice to redrill it. Loading extra amount of explosives in the hole cannot solve the problem.

When the bench surface is not leveled, hole depths are to be determined individually with known RL of the intended bench floor and the RL of hole collar. RL correction is required only when the surface is uneven, as illustrated in Fig. 6.8.
Hole depth = bench height + subgrade drilling + allowances for hole collapse ± RL correction

6.8 STEMMING

The basic purpose of stemming is to confine the explosion pressure in the blasthole. The resistance to explosion pressure is provided by the mass of the stemming and friction. The mass of the stemming material can be calculated by

\[ w = A \rho \]  \hspace{1cm} (6.9)

where

- \( w \) = mass of the stemming material (kg)
- \( A \) = cross-sectional area of the blasthole (m²)
- \( l \) = length of the stemming column (m)
- \( \rho \) = density of the material (kg/m³)

The density of air and water, being less than the drill cuttings, are not better stemming material than the drill cuttings. Frictional forces can be increased by increasing the friction between blasthole wall and stemming and by means of interlocking effect between the particles.

The cross-sectional area of a blasthole being the same, the forces due to weight can be increased by increasing the length of stemming column. If the stemming length is inadequate, the explosive gases will vent prematurely, reducing blasthole pressure and resulting in poor displacement and a tight muckpile. It will also result in flyrock and air overpressure. Longer stemming length gives good confinement but it may lead to poor fragmentation from the stemming region.

The stemming length is calculated in terms of burden (Atlas Powder Company, 1987, Konya and Walter, 1990, Ash, 1990, Pearson et al, 1994, Dick et al, 1986, ICI, 1993). This method was adopted in this study also but the empirical coefficients were derived from the case studies collected by the author. From the analysis of the data collected during 1987-89,
the stemming was found to be equal to 0.7 to 1.2 times the burden (Adhikan et al, 1990). From the data collected during 1990-93, it was found to be 0.7-1.3 times the burden (Table 6.8) From Section 5.3.3 it was found that the stemming length should be 0.7-1.2 times the burden. These findings are consistent with those available in the literature.

Table 6.8 Recommended stemming length for surface mines

<table>
<thead>
<tr>
<th>Case</th>
<th>Rock at the collar</th>
<th>Flyrock problem</th>
<th>Stemming length (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Hard</td>
<td>Yes</td>
<td>(1.0-1.2) x Burden</td>
</tr>
<tr>
<td>2</td>
<td>Hard</td>
<td>No</td>
<td>(0.7-1.0) x Burden</td>
</tr>
<tr>
<td>3</td>
<td>Soft</td>
<td>Yes</td>
<td>(1.0-1.3) x Burden</td>
</tr>
<tr>
<td>4</td>
<td>Soft</td>
<td>No</td>
<td>(0.7-1.0) x Burden</td>
</tr>
</tbody>
</table>

If the stemming section of a blasthole comprises soft rock, there would be no problem of fragmentation, where stemming can be 1.3 times the burden. This is also applicable for rocks which are highly jointed and the natural blocks are smaller than the maximum allowable fragment size. In hard rock, stemming length greater than 1.0 times the burden may cause fragmentation problem from the stemming region. Unless the stemming is maintained as 1.0-1.2 times the burden, flyrock is difficult to control. In such cases, stemming efficiency can be enhanced using suitable stemming material or using blast control plugs. In the front row, stemming length may be adjusted on a hole-by-hole basis depending on the face profile. It may be necessary to consider the stemming length for the back row separately to contain overbreak. These recommendations are made from the analysis of blasts using drill cuttings as stemming material and detonating cord as downline initiation system. Use of bottom initiation and proper stemming material will provide better confinement, hence better control of fragmentation and flyrock.

It is difficult to optimise stemming length by simply observing blasts in the field. However, gun barrel like discharge from the collar of the hole shows inadequate stemming. The stemming length can be evaluated easily using a high resolution video camera. Unlike high-speed photography, the ejection velocity cannot be calculated from video records but it has been found to be a simple and effective tool for evaluating the stemming performance.

Worsey (1990) studied the effect of stemming on burden movement by using high-speed videography. His results suggest that the burden rock movement is inversely proportional to the stemming ejection velocity. The results of his tests showed that the burden velocity was reduced by a factor of three when stemming performed poorly. Armstrong et al (1993) studied the effect of confinement due to stemming on the fragmentation and rock movement in model.
scale experiments. Their results showed that the degree of fragmentation and rock movement improved with the increase in stemming retention.

Several kinds of material have been experimented abroad. Water, mud, soil, wet clay and drilling dust are easily ejected. On the other hand, dry angular material under the effect of the impulsive gas pressure tends to form a compaction arch which locks into the wall of a blasthole, thus increases its resistance to ejection. Stemming materials, ranked in the order from the least efficient to most efficient are air, water, wet drill cuttings, wet sand, paper cartridge of rock dust, clay dummies, wet crushed stone, dry drill cuttings, dry sand, dry crusher run stone, dry screened stone (Lappincott, 1992).

In water filled holes, gravel and crushed rock quickly settle to form a plug of stemming. The behaviour of drill cuttings can be quite different. They are converted to a sludge which has little to offer, other than its own mass, in opposing explosion gas pressure.

Proper particle size of the stemming material is also important when the stemming length is short. Materials containing fine and coarse particles are not suitable for stemming. Fine particles are easily ejected and coarse particles have a tendency to bridge the hole and can also damage initiating devices. The optimum particle size of the stemming material is 0.05-0.08 times the hole diameter (Konya and Walter, 1990; ICI, 1993).

Worsey and Nixon (1988) developed and patented a mechanical stemming aid for efficient stemming. They have shown substantial reductions in stemming ejection velocity or ejection elimination whilst reducing the stemming up to 35% over conventional practices. Eloranta (1994) evaluated stemming performance in 406 mm diameter blastholes with different sizes of stemming materials, concrete plugs, and an decking. His results showed that concrete plugs in conjunction with coarse stemming reduced stemming ejection velocities by 40%. Blast control plugs are now commercially available. It is reported that these plugs significantly improve blasting efficiency while also eliminating most flyrock and greatly reducing air overpressure and dust (JEE, 1996).

Stemming is one of the most neglected design parameters in Indian mines. Some consideration is given to stemming length but no regard is given to the stemming material and the particle size of the stemming material. There is great potential in these mines for improving blast performance by using suitable stemming material and blast control plugs.
6.9 POWDER FACTOR

A number of approaches have been made to calculate powder factor including, 1) based on seismic wave velocity (Broadbent, 1974, Hennen and Dimock, 1976, Muftuoglu et al, 1991), 2) based on drilling data (Leighton et al, 1982, Muftuoglu et al, 1991, Jimeno and Hevia, 1987), 3) based on rock properties such as uniaxial compressive strength, tensile strength and density of the rock (Muftuoglu et al, 1991), 4) based on rock mass properties (Ashby, 1981, Schoeman, 1986), 5) based on the blastability index (Lilly, 1986), 6) based on the single hole full scale test (Rustan and Nie, 1987), and 7) based on energy balance concept (Berta, 1990)

Terms such as specific charge, powder factor, specific explosive consumption, charge ratio, charge factor used in blasting with different units of measurement have resulted in confusion In this study, only the term powder factor is used and is expressed in kg/m³

For surface mining powder factor can vary from as low as 0.15 to as much as 1.5 kg/m³, with 0.3-0.6 kg/m³ being the most common (Ash, 1990)

Blasting being a repetitive operation, the evaluation of full scale production blasts has been the most direct and reliable method This method was used by the author at various mines Using the data generated from the field, empirical relationships are derived for calculation of powder factor

It is recognised that powder factor is mainly a function of the rock mass properties The type of explosive and its characteristics also influence powder factor It may not be considered for the calculation of powder factor if the explosive is selected as suggested in Section 4.2 Powder factor is controlled by a fixation factor (Persson et al, 1994) It is different for bench blasting, crater blasting and boulder blasting Since the calculations concern only with bench blasting, the fixation factor can be ignored Powder factor also depends on the purpose of the blast In a throw oriented blast, a higher powder factor is required In some special situations like Neyveli lignite, where blasting is carried out to loosen the strata for easy handling of the material by bucket wheel excavator, a very low powder factor (0.10 kg/m³) is sufficient Such special applications cannot be excluded for calculation of normal bench blasting operations

The suggested method involves three steps First, powder factor is calculated for intact rock material depending on rock type and density Then two correction factors are introduced to account for the influences of block size and the bench height This method is simple and does not require elaborate testing for rock properties
Step 1: Powder factor depending on rock type and density

Muftuoglu et al (1991) had derived a correlation for calculation of powder factor with the density of rock for Turkish lignite mines. When the data for powder factor and density were examined, it was found that some rocks like granite with a density of 2.5-2.7 gm/cc would require twice the powder factor as that of limestone with nearly equal density. Therefore, rock type in addition to density was used to derive an empirical equation for calculation of powder factor:

\[ q = K + B \rho \]  

where

- \( q \) = powder factor (kg/m\(^3\))
- \( K, B \) = constants of the equation (Table 6.9)
- \( \rho \) = density of rock (gm/cc)

### Table 6.9 Empirical constants for determination of powder factor from rock type and density

<table>
<thead>
<tr>
<th>Rock type</th>
<th>Value of K</th>
<th>Value of B</th>
</tr>
</thead>
<tbody>
<tr>
<td>Coal</td>
<td>-0.400</td>
<td>0.500</td>
</tr>
<tr>
<td>Sandstone</td>
<td>-0.470</td>
<td>0.410</td>
</tr>
<tr>
<td>Shale</td>
<td>-1.043</td>
<td>0.670</td>
</tr>
<tr>
<td>Limestone, dolomite</td>
<td>-0.733</td>
<td>0.400</td>
</tr>
<tr>
<td>Ore/waste in metal mines, granite, basalt, amphibolite</td>
<td>-1.066</td>
<td>0.666</td>
</tr>
</tbody>
</table>

The constants \( K \) and \( B \) were derived in the following manner:

1) The powder factors for different rocks were collected from the blast records, histograms were plotted and variations within the same rock type noted.
2) The density of rock for the same blast records were collected and variations were noted through histograms.
3) For the given range of powder factor and the density of rock, powder factor was assumed to increase linearly with the increase of density. Assigning the lowest value of powder factor to lowest density and highest powder factor to highest density of rock, straight lines were drawn from which coefficients were determined.
Table 6.9 shows the values of constants K and B for coal, shale, sandstone, limestone and for waste rock or ores in metal mines having density up to 3.0 gm/cc

**Step 2: Considerations for the influence of joints**

Since the powder factor is significantly influenced by structural discontinuities, it is necessary to consider the influence of joints by introducing a correction factor as suggested in the following empirical equation

\[ q_c = \frac{q}{j} \]  \hspace{1cm} (6.11)

where
- \( q_c \) = corrected powder factor (kg/m\(^3\))
- \( q \) = powder factor calculated from the previous step
- \( j \) = correction factor depending on the block size (Table 6.5)

The correction factor incorporates the fact that the powder factor required to achieve the same degree of fragmentation is less for a highly jointed rock (small block size) compared with a rock mass containing large blocks.

**Step 3: Considerations for incompatible conditions**

In mines where the hole diameter is large for the bench height (Zone A in Fig. 6.1), the powder factor has to be increased in order to achieve the desired degree of fragmentation. In other cases, Step 3 may be ignored.

The above statements can be justified by looking into the relationships between burden and length to diameter ratio of the charge, required for satisfactory fragmentation. Fig. 6.9 shows the plots of burden against length to diameter (l/d) ratio of the charge for different hole diameters. Empirical equation of the following type was derived by regression analysis:

\[ B = \alpha(1 - e^{-\beta l/d}) \]  \hspace{1cm} (6.12)

where
- \( B \) = burden (m)
- \( l \) = length of the charge (m)
- \( d \) = diameter of the charge (m)
- \( \alpha, \beta \) = empirical constants (Table 6.10)
Table 6.10 Results of the regression analysis between burden and the length to diameter ratio of the charge

<table>
<thead>
<tr>
<th>Hole diameter (m)</th>
<th>Number of data</th>
<th>Constant $\alpha$</th>
<th>Constant $\beta$</th>
<th>Correlation coefficient</th>
<th>$\alpha/d$</th>
</tr>
</thead>
<tbody>
<tr>
<td>0.10</td>
<td>22</td>
<td>2.91</td>
<td>0.102</td>
<td>0.81</td>
<td>29.1</td>
</tr>
<tr>
<td>0.15</td>
<td>37</td>
<td>4.40</td>
<td>0.109</td>
<td>0.71</td>
<td>29.3</td>
</tr>
<tr>
<td>0.25</td>
<td>22</td>
<td>7.15</td>
<td>0.108</td>
<td>0.87</td>
<td>28.6</td>
</tr>
</tbody>
</table>

From column 6 of Table 6.10, it can be seen that for all the three blasthole diameters, $\alpha/d = 29$ or $\alpha = 29d$ and $\beta = 0.1$. Thus a generalised equation can be derived

$$B = 29d \left(1 - e^{-0.1 \frac{1}{ld}}\right) \quad (6.13)$$

Equation 6.13 shows that burden depends on the blasthole diameter and is comparable with Equation 6.3.

From Equation 6.3, $B = 27d$ ($C_1 = 27$, an average value for 15-40)

From Equation 6.13, $B = 26.6d$ (for $l/d = 25$)

The best-fit lines in Fig. 6.9 show that burden increases very marginally for $l/d > 25$ and decreases significantly for $l/d < 20$. Drilling being a costly operation, burden is increased as much as possible by increasing the charge length. This explains why the explosive consumption increases when the hole diameter is large for the bench height. Under such a condition, powder factor can be reduced by resorting to deckung (Singh, 1991) or air deckung (Pompanna and Chikkareddy, 1993). These measures reduce the drillhole utilisation as mentioned in the second column of Table 6.1. The problems of poor drillhole utilisation and excessive powder factor can be avoided, provided that the blasthole diameter and bench height are compatible. Previous workers (Hagan, 1983; Perssson et al, 1994) have also noted the effect of low bench height. Hagan (1983) has mentioned that the burden is reduced appreciably for low bench height whereas Perssson et al (1994) have suggested two separate equations for low and high benches.

Though there is no advantage when $l/d > 70$ or when $l/d < 20$, there are several points for $l/d < 20$. The discrepancy between the recommended and operating practices is due to the use of larger blasthole diameters than the required (Fig. 6.1).
Fig 6.9 Burden versus length to diameter ratio of the charge

Blasthole diameter - 100 mm

Blasthole diameter - 150 mm

Blasthole diameter - 250 mm
The influence of initiation sequence and delay timing on blast performance was studied in detail at limestone quarries using Sequential Blasting Machine (SBM). Short delay detonators and detonating relays have fixed delay intervals and do not provide flexibility in the selection of delay timings. SBM is a solid state condenser discharge blasting machine that can initiate up to 10 individual blasting circuits in a sequence with a desired time interval between the circuits. The system includes timer, terminal board, extension cable, and circuit tester. SBM in conjunction with short delay detonators can yield many independent delays within a blast.

This section discusses the problems and solutions associated with typical patterns in which holes along the same rows are connected in the same circuits. Row-to-row delay interval is a function of the timer setting, while the hole-to-hole delay interval is attained by the nominal firing times of the detonators. These patterns are designed based on the results and conclusions of several case studies from limestone quarries.

The time required to deliver electric current to all circuits and the time at which the first hole detonates is very important. If all the circuits are energized prior to detonation of any hole, then it is a good design and should be the goal in designing a circuit diagram. Partially activated circuits also minimize the problems. When zero delay detonator is connected to the first circuit, cutoff of detonator leads is possible before the second circuit gets energized. If there is cutoff in any of the circuits, the machine will not supply current to the remaining circuits causing misfires. It is better to use detonators starting from number 3 or 4 so that the first initiation will take place at 75 ms or 100 ms. This allows some of the circuits to energize while others are at safer distances or when the circuits are energized three rows of holes ahead of the firing times.

A new problem is encountered even if the circuit begins with detonator number 4 on account of nonuniform delay intervals. The detonators supplied by one of the major manufacturers have delay numbers ranging from zero to ten with a nominal time interval of 25 ms between successive delay numbers from 1 to 6, 50 ms for 7 and 8, and 75 ms for 9 and 10. The firing times of blastholes connected by broken lines (Fig 6.10) show that the holes in the front row fire prior to the holes in the back rows up to number 6 detonator. From number 7 onwards, the holes in the back rows fire earlier than the holes in the front row. This change in sequence will adversely affect the blast results.
In an attempt to overcome the above problem, delay intervals were increased up to 60 ms which resulted in frequent misfires, mostly from the middle and back rows. These misfires were mainly due to cutoff as 60 ms between rows (about 15-20 ms/m of burden) was too long.

Free face

Circuit 1 (Instant)
8/250 17/200 6/150 5/125 4/100

Circuit 2 (33 ms)
8/283 17/233 6/183 4/133 5/158

Circuit 3 (66 ms)
8/316 17/266 6/216 5/191

Circuit 4 (99 ms)
8/349 17/299 6/249

Fig 6.10 Improper order of detonation due to nonuniform delay intervals

The above problems were eliminated by using detonators of another manufacturer which also supplies zero to ten series but with uniform delay intervals of 25 ms (Fig 6.11). The problem as mentioned in Fig 6.10 does not arise here as the desired sequential firing times are secured. This pattern is usually recommended for blasting up to 130 holes (13 holes per row x 10 circuits), with two holes per delay.

Free face

Circuit 1 (Instant)

Circuit 2 (33 ms)

Circuit 3 (66 ms)
8/316 7/276 6/226 5/181

Circuit 4 (99 ms)
8/349 7/299 6/249 5/204

Fig 6.11 Recommended pattern for blasting with two holes per delay
Initially these sequences were designed to control ground vibration. However, it was found that fragmentation improved in the hole to hole sequencing compared to V or VI type of initiation (Fig. 2.4). The delay principles and results of SBM should be valid for other initiation systems.

When blasting near a village or any vulnerable structure, the limitation on maximum charge per delay may not permit to fire more than one hole per delay. For such a condition, the pattern as shown in Fig. 6.12 is recommended in which only one hole goes off at one time.

**Fig 6.12 Recommended pattern for blasting near dwellings**

For a particular case, the delay interval is determined after conducting a few trial blasts by gradually increasing or decreasing the timing between circuits. The delay interval of 33 ms and 42 ms (8-10 ms/m of burden) were found to be satisfactory in terms of fragmentation, flyrock and ground vibration.

The initiation sequences discussed above provided some examples of the application of fundamental principles of delay blasting. In Chapter 7, problems were noted due to initiation sequence and delay timing. In the area of initiation, there is a wide gap between the technology available and the technology used in Indian mines and there has been much delay in the introduction of shock tube system, which is well proven abroad.