Chapter 24

Properties of rock masses

24.1 INTRODUCTION

As has been said earlier, the main objective behind blasting a rock mass is to fragment it in such a way that the fragments created in the process are small enough to be easily removed from the blasting site. To achieve this objective the following necessities arise.

1 Choose proper blasthole parameters viz. diameter, depth and inclination.
2 Determine such a layout for the blasthole positions, that after appropriately charging and blasting them the fragments of rock mass are easily loaded and hauled by the equipment chosen for the purpose.
3 Choose apt explosive, efficient method and pattern of charging the blastholes, the sequence and timing of detonating explosives and appropriate accessories for the blast.

As stated earlier, blasting is a very hazardous and dangerous operation. It can become the cause of hundreds of deaths if attention is not given towards minimizing the hazards. Obviously, all the measures to reduce such hazards have become a part of every blasting program.

Several properties of rock specimen were described in chapter 3 of this book. The background behind various tests carried out to determine the magnitude of the properties and the methodology of the test was also elaborated. The reason to take such account early on in this book was that the properties had great influence on the process of rock breakage in drilling the blastholes. In the first part of this chapter the influence of rock specimen properties on blasting is considered.

When it comes to blasting, the properties of the rock mass are of greater importance than rock specimen properties. This is due to the fact that in drilling, only a small portion in the alignment of the blasthole is to be fragmented into very small pieces, whereas in blasting a very large mass of rock is to be fragmented into relatively large pieces. How the properties of rock mass can be taken into consideration while designing a blast has been elaborated in the second part of this chapter.

24.2 ROCK SPECIMEN PROPERTIES AND BLASTING

Rock specimen properties that affect blasting are strength, density and porosity.

24.2.1 Influence of rock strength on blasting

For long it was an observed fact that rocks with higher compressive strength need more explosive energy for fragmentation. It was later noticed that actually the tensile wave reflected from the free surface influences rock breakage to a greater extent.

In the year 1959 Hino postulated that the tensile rock fracture occurs in the form of slabs that are parallel to the free surface. The extent of tensile fractures and the number of slabs so produced depends on the tensile strength of rock (σ_t), amplitude (σ_a) and length (L) of the compressive wave. He concluded that the number of slabs (n) produced by tensile slabbing due to reflected shock waves may be given by

$n \leq \sigma_a/\sigma_t$ or $n \leq L/2t$
where t = thickness of slab.

Hino also noticed a linear relationship between the compressive strength of rock (σ_c) and the amplitude of the compressive stress wave (σ_a) propagated through the rock. Therefore, $\sigma_a \propto \sigma_c$ and hence, $n \propto \sigma_c/\sigma_t$.

Since σ_c/σ_t is proportional to the blasted rock, he proposed it to be called a blasting coefficient.

In the year 1979, Kutuzov correlated the powder factor with compressive strength of rock and gave the tabulated form, as in Table 24.1.

24.2.2 Influence of rock density on blasting

Normally the density of rock is well correlated with its compressive strength. High density rocks require more energy for their deformation and fracture.

Explosives that release a higher volume of gases exert higher pressure on the blasthole walls, and the consequent higher bubble energy is capable of fracturing a larger rock mass.

Table 24.1 Rock mass classification on the basis of joint spacing and bed thickness.

Powder factor in kg/m³		*Mean distance between natural fractures in rock in m*	*Uniaxial compressive Strength MPa*	*Density of rock kg/m³*
Range	*Average value*			
0.12–0.18	0.150	<0.10	10–30	1400–1800
0.18–0.27	0.225	0.10–0.25	20–45	1750–2350
0.27–0.38	0.320	0.20–0.50	30–65	2250–2550
0.38–0.52	0.450	0.45–0.75	50–90	2500–2800
0.52–0.68	0.600	0.70–1.00	70–120	2750–2900
0.68–0.88	0.780	0.95–1.25	110–160	2850–3000
0.88–1.10	0.990	1.20–1.50	145–205	2950–3200
1.10–1.37	1.235	1.45–1.70	195–250	3150–3400
1.37–1.68	1.525	1.65–1.90	235–300	3350–3600
1.68–2.03	1.855	>1.85	>285	>3550

Blasthole pressure is also dependent upon the velocity of detonation of the explosive, and the velocity of detonation is in turn dependent upon the blasthole diameter. For all these reasons it is prudent to use blastholes of larger diameter and use an explosive with higher bubble energy while blasting a denser rock mass.

24.2.3 Influence of rock porosity on blasting

Porosity of rocks is the spread of small inter-particle pores that are created while the rocks are being formed, either by plutonic or sedimentation activities. These pores are spread throughout the rock mass.

In many cases such rocks do not require blasting. They can be fragmented by ripping or other methods.

However, when blasting has to be carried out, explosives with higher bubble energy and low shock energy are found to be more suitable for the fragmentation.

Increasing bubble energy and reducing shock energy can also be accomplished by decoupling the charge and initiation system. More than normal stemming is also a method through which pressure built up in the blasthole can be increased.

24.2.4 Specimen blastability

This blastability index has been proposed by the Norwegian University of Science and Technology (NTNU).

It takes into account many variables such as sonic velocity, anisotropy, density of rock, charge density etc., involved in the process of blasting.

The NTNU Equation for blastability index is as under.

$$S = (0.7364 * I_a^{\ 0.61} * (T)^{0.72})/((C/1000)^{0.4} * (W/C)^{0.25} * \rho^{0.19})$$

where

S = Rock blastability index
I_a = Anisotropy index = C_y/C_z
C_y = Sonic velocity of dry rock parallel to the foliation in m/s
C_z = Sonic velocity of dry rock normal to the foliation in m/s
$C = (C_y + C_z)/2$
ρ = Dry density of rock in kg/L
T = Charging density of explosive in kg/L
W = Detonation velocity of explosive m/s.

Good, medium and poor blastability is indicated by index values of 0.38, 0.47 and 0.56 respectively.

24.3 PROPERTIES OF ROCK MASSES AND BLASTING

In a large area to be excavated, the rock mass is hardly ever homogeneous. Different portions of rock mass have varying mineral contents.

Rock masses also have inconsistencies like voids, folds, unconformities, bedding planes, faults and joints. All these defects result from volcanic, plutonic and tectonic activities, and other processes at the surface of the earth, through which rocks are formed.

24.3.1 Voids

Two types of voids can be found in rock masses.

In some types of rocks, particularly in the softer varieties, very small size voids are homogeneously spread over a very large area. These voids make the rock mass very weak – so weak that there is no necessity of blasting. However, if such a rock mass contains a layer of hard rock or some very large boulders, blasting may be unavoidable.

Sometimes voids of large volume are formed in a rock mass during volcanic or tectonic activities, or by erosion. In such circumstances the gases formed in the explosion rush into the void as shown in Figure 24.1 A. In the process these gases absorb quite a bit of energy. Thus, the energy left for fragmentation of rock mass is reduced and large pieces of rocks are formed.

Similarly, if an easily blastable ground mass contains a large boulder of significantly hard rock as shown in Figure 24.1B, during the fragmentation the rock mass breaks but the boulder does not get sufficient energy to break. This results in the need for secondary blasting.

24.3.2 Folds, unconformities and bedding planes

Sedimentary rock masses cover a considerable part of the earth's surface. When formed, they are in layers lying one upon another. These layers are called beds or strata. The thickness of these beds varies from a few millimeters to several meters. The bounding planes of a bed are called bedding planes.

During the deposition of the sediments that give rise to the beds, often there is a time interval during which no deposition takes place. Therefore, a surface formed during this time interval separates the old beds from new beds. On many occasions, due to a change in the mode of deposition or a change in the type of sediment, such a surface is very distinct and is called an unconformity. The properties of the rock mass on two sides of an unconformity can differ considerably.

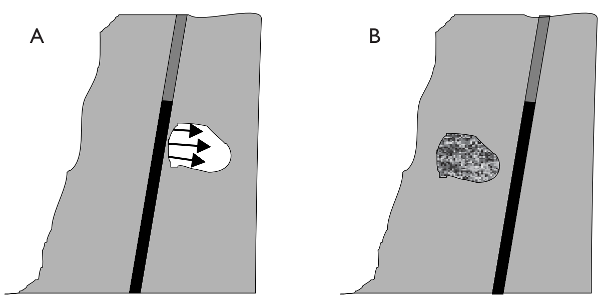

Figure 24.1 Presence of voids and boulders.

Horizontal beds of sedimentary rocks are often distorted by physical forces exerted on them. Such geological activities are termed tectonic activities. Tectonic i.e. structural activities are mainly caused by plutonic activities taking place in the earth's crust and mantle. In tectonic activities beds are often compressed and distorted in such a way that they take the shape of a waveform. Such structures are called folds.

Strike and dip are two terms used to indicate the direction and magnitude of inclination of the bedding planes of the sedimentary layers.

Whatever the inclination of a bedding plane may be, it is always possible to draw a straight line on each of the bedding planes in such a way that the line will be horizontal. The direction of such a line is called the strike. The strike can be defined by noting its bearing with north or any other well-defined direction. If the bedding plane is really a plane and not a curved surface, all the lines parallel to the strike and lying on that bedding plane will also be horizontal.

A line lying in the direction perpendicular to the strike and also lying on that bedding plane, will have maximum inclination with the horizontal. The angle of inclination of this line with the horizontal is called the dip. The direction of such a line is called the direction of dip. Any line on the bedding plane, but not lying in the direction of dip will have lesser inclination than the dip. It is, therefore, termed as apparent dip. The strike and dip of a bedding plane are shown in Figure 24.2.

The orientation of bedding planes with respect to the blasthole alignment can have considerable influence on the outcome of blasting because the bedding planes are weak, and parting of the rock mass along these bedding planes is rather easy.

Based on the relationships between the orientations of bedding planes and blastholes, three cases, as under, are usually considered.

1 Shooting with the dip
2 Shooting against the dip
3 Shooting along the strike

An elaboration of these is given below.

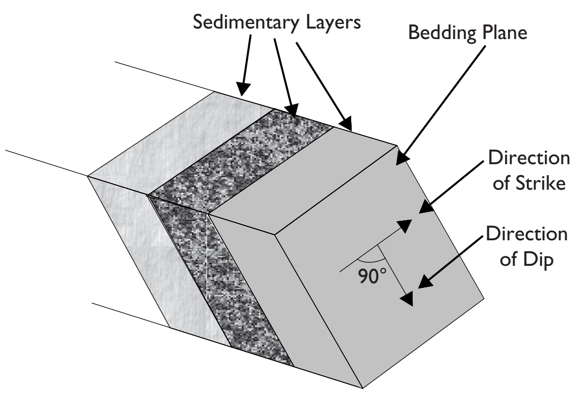

Figure 24.2 Strike and dip of a bedding plane.

24.3.2.1 Shooting with the dip

In this situation, shown in Figure 24.3, the lines of intersections of the bedding planes and bench floor or bench top are parallel to the bench crest.

For the following reasons, the blast results are neat.

Toe problems are less hence the resultant bench floor is smooth.

As planes of weakness parallel to the bench face already exist, the failure by flexure is easier. For this reason the fragmentation tends to be more satisfactory and the throw of the fragments is farther away from the bench face. Such a muckpile is easy for loading operations.

However, a blast in such circumstances tends to give backbreak problems.

The inclination of blastholes in the direction of dip (as shown in Figure 24.3) is one of the remedies in reducing backbreak problems.

24.3.2.2 Shooting against the dip

In this situation also the lines of intersections of the bedding planes and bench floor or bench top are parallel to the bench crest, as shown in Figure 24.4.

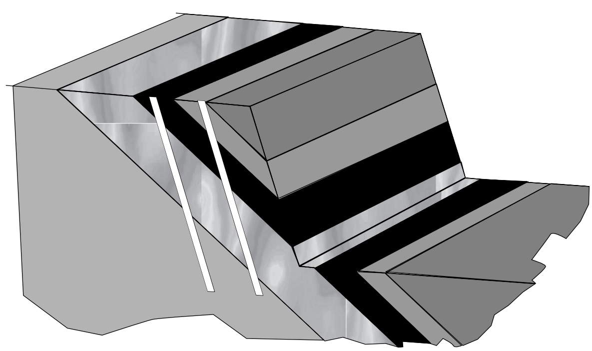

Figure 24.3 Shooting with the dip.

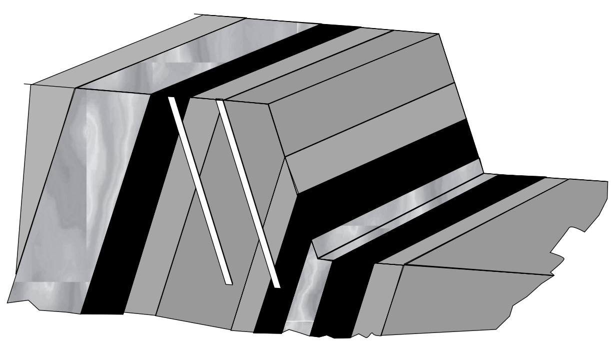

Figure 24.4 Shooting against the dip.

The results of blasting can have one or more of these undesirable features.

The booster charge at the bottom forms cracks along the bedding planes and the magnitude of breakage of rock on the two sides of such planes is often different. For this reason there can be many ups and downs in the newly formed bench floor after blasting. The toe can have large size stumps. There are more oversize rock pieces.

The backbreak may be less but there are more cracks on the top near the crest of the newly formed bench. Large overhangs increase the possibility of a rock fall from the high wall.

The blasted fragments are not thrown farther away from the high wall. This results in a tall heap of broken material.

The following measures may reduce the problems to some extent.

Extended subdrilling length
Higher booster charge
Use of explosive with higher brisance at bottom
Decking of explosives in a blasthole
Use of small diameter satellite holes near the crest

24.3.2.3 Shooting along the strike

In this situation the lines of intersection of the bedding planes and bench floor or bench top are perpendicular to the bench crest, as illustrated in Figure 24.5.

From the viewpoint of blasting this is the worst situation for the following reasons.

Since different types of rocks outcrop on the floor of the newly formed bench floor, due to the differences in their properties the bench floor can be highly uneven.

Backbreak resulting from the blast can be very irregular.

Field experience has shown that this situation is the worst amongst the three. Reorienting the bench face, and blasthole inclination, may be the only viable solutions.

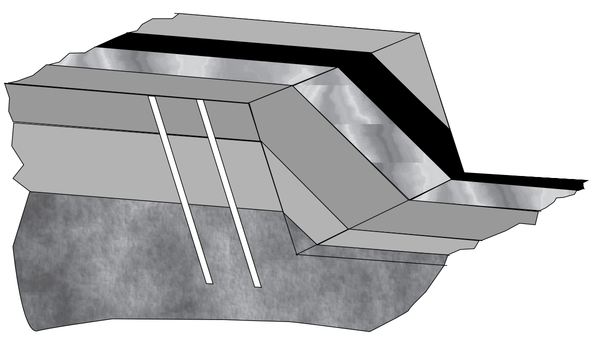

Figure 24.5 Shooting along the strike.

24.3.3 Faults and joints

Both faults and joints are fractures in the ground mass. Their size is very large. They are essentially planes of separation formed during plutonic or tectonic activities.

When there has been observable movement of the rock mass on the two sides of the fracture plane, they are termed as faults, otherwise they are called joints.

Joints are usually small in thickness and may have intrusion of fine clayey particles.

Faults are usually much thicker. During formation of the fault, when two sides of the rock mass rub against each other, many pieces of rock are formed and remain inside the fault zone. Later the intrusion of fine particles into the fault zone takes place and rocks like breccia or conglomerate are formed.

As far as blasting in large surface mines is concerned there is no distinction between faults and joints.

These fracture surfaces can be found in almost any inclination and direction. The frequency of their occurrence is often very high.

Fracture surfaces i.e. joints, require to be given great attention, not only in the realm of blasting but any other type of excavation and construction in or above the ground. For this reason very much research has been done in respect of the effects of joints in the rock mass during blasting or excavation.

The next sections of this chapter have been devoted to some details of the effects of joints on blasting.

24.4 CLASSIFICATION OF ROCK MASSES

Rock masses are classified either through visual observations or indexes proposed on the basis of diverse parameters of the rock mass.

24.4.1 Classification by visual observation

Terzaghi proposed a rock mass classification in the context of tunnel support design in 1946. The concepts behind the classes are very important. Classes and their description proposed by Terzaghi are given in Table 24.2.

In 1963, Deere proposed a classification of rock mass based on the spacing between joints found in the cores obtained in exploratory diamond core drilling and the thickness of the beds encountered in the rock mass. This conceptual description is presented in Table 24.3.

Both these classifications are now superseded by other classification schemes that are more specific.

They are based on absolute factors and do not leave much scope for different interpretations. Many such schemes take into account many more factors than UCS of rock or joint frequency.

24.4.2 Classification by index

Some systems of rock mass classification that are based on statistical or empirical indices include:

Table 24.2 Rock mass classification based on field and core sample observations.

Rock mass	*Description of the rock mass*
Intact Rock	It contains neither joints nor hair cracks. Hence, if it breaks, it breaks across sound rock. On account of the injury to the rock due to blasting, spalls may drop off the roof several hours or days after blasting. This is known as spalling condition. Hard, intact rock may also be encountered in the popping condition involving the spontaneous and violent detachment of rock slabs from the side of roof.
Stratified Rock	It consists of individual strata with little or no resistance against separation along the boundaries between the strata. The strata may or may not be weakened by transverse joints. In such rocks the spalling condition is quite common.
Moderately Jointed Rock	It contains joints and hair cracks, but the blocks between joints are locally grown together or so intimately interlocked that vertical walls do not require lateral support. In rocks of this type, both spalling and propping conditions may be encountered.
Blocky and Seamy Rock	It consists of chemically intact or almost intact rock fragments which are entirely separated from each other and imperfectly interlocked. In such rock, vertical walls may require lateral support.
Crushed but Chemically Intact Rock	It has a character of crusher run. If most or all of the fragments are as small as fine sand grains and no re-cementation has taken place, crushed rock below the water table exhibits the properties of water-bearing sand.
Squeezing Rock	This type of rock slowly advances into the tunnel without perceptible volume increase. A prerequisite for squeeze is a high percentage of microscopic and sub- microscopic particles of micaceous minerals or clay minerals with a low swelling capacity.
Swelling Rock	This type of rock advances into the tunnel chiefly on account of expansion. The capacity to swell seems to be limited to those rocks that contain clay minerals such as montmorillonite, with a high swelling capacity.

Table 24.3 Rock mass classification on the basis of joint spacing and bed thickness.

Joint spacing	*Bed thickness*	*Spacing in inches*
Very Close	Very Thin	Less than 2 in.
Close	Thin	2 in. to 1 ft.
Moderately Close	Medium	1 ft. to 3 ft.
Wide	Thick	3 ft. To 10 ft.
Very Wide	Very Thick	More than 10 ft

Rock Quality Designation i.e. RQD
Rock Mass Rating i.e. RMR
Rock Tunnel Quality Index i.e. RTQI.

Descriptions of these systems are given below:

24.4.2.1 RQD based classification

RQD was proposed by Deere in 1969 and is defined on the basis of core samples of the rock mass obtained in diamond core drilling carried out during exploration stage. It is defined by a very simple formula as under.

$$RQD = (100 * L)/D$$

where

L = Total cumulative length of core pieces obtained in a diamond core drilling run, each with 100 mm or longer length.
D = Total length drilled in the run.

Since L is dependent upon the diameters of drill rods, core barrel and the type of core barrel used in drilling, these parameters must be as follows:

1 The cores must be NW or larger size in diamond core drilling i.e. the core diameter should be at least 54.7 mm or 2.15 inches.
2 Double tube swivel core barrel must be used.
3 RQD value should be calculated immediately after recovering the core before the pieces break in subsequent rehandling.
4 Core drilling operations should be carried out with the intention of getting maximum core recovery. From this viewpoint the drill rods used should be at most one size smaller than the size of core barrel. If the size of drill rods used for drilling is smaller than such size difference, heavy vibrations are caused in the drill string and the core samples get broken easily.

Table 24.4 Standard dimensions in diamond core drilling.

Size designations	*BW*	*HW*	*HW*
Diameter of the Drilled Hole mm	59.6	75.3	98.8
Outside Diameter of Drill Rod mm	54.1	66.8	89.1
Outside Diameter of Core mm	42	54.7	76.2

Table 24.5 Classes of rock quality designation.

RQD value	*Rock quality class*
Less than 25%	Very Poor
25%–50%	Poor
50%–75%	Fair
75%–90%	Good
90%–100%	Excellent

Relevant sizes of diamond core drilling equipment are given in Table 24.4.
Rock quality classes according to their RQD values are given in Table 24.5.

24.4.2.2 RMR based classification

Classification based on RMR, proposed in 1973 by Bienawski, is also called the Geotechnical Classification. It is based on a few basic parameters relating to the geometrical and mechanical properties of the rock mass as listed below:

1 Uniaxial compressive strength of intact rock
2 Rock Quality Designation
3 Spacing of discontinuities in the rock mass
4 Condition of the surfaces at the discontinuity
5 Groundwater conditions
6 Orientation of discontinuities relative to the engineered structure

Of these, the rating values for the first five parameters are to be appropriately chosen from Table 24.5. To the sum of these values an adjustment value for the orientation of discontinuities, chosen from Table 24.6, is to be added. The resulting sum will give the RMR rating that can be interpreted from Table 24.7 to give the class assigned to the rock mass. It is easy to observe that, whenever the rating value of a parameter is higher, the condition is favorable. Whenever conditions are very near the boundary of a particular criteria, the values can be interpolated or the graphs presented by Bienawski can be used. One such graph is shown in Figure 24.6.

Table 24.6 Values of variables to be chosen for calculating blastability index proposed by Ghose.

Parameter	*Range of variation for the variable and value to be chosen for the range*				
Density of the Rock Mass in t/m^3	<1.6	1.6–2.0	2.0–2.3	2.3–2.5	>2.5
Value for Density Ratio	20	15	12	6	4
Discontinuity Spacing in m	<0.2	0.2–0.4	0.4–0.6	0.6–2.0	>2.0
Value of Discontinuity Spacing Ratio	35	25	20	12	8
Point Load Strength Index in MPa	<1	1–2	2–4	4–6	>6
Value for Point Load Strength Index Ratio	25	20	15	8	5
Joint Plane Orientation	Dip into Face	Strike at an Angle to the Face	Strike Normal to the Face	Dip Out of Face	Horizontal
Joint Plane Orientation Ratio	20	15	12	10	6
Adjustment Factor 1 – for Highly Confined Condition					–5
Adjustment Factor 1 – for Reasonably Free Condition					0
Adjustment Factor 2 – for Hole Depth/Burden Ratio >2					0
Adjustment Factor 2 – for Hole Depth/Burden Ratio 1.5–2					–2
Adjustment Factor 2 – for Hole Depth/Burden Ratio <1.5					–5

Table 24.7 Relationship between blastability index proposed by Ghose and the powder factor.

Range of Blastability Indices	30–40	40–50	50–60	60–70	70–85
Powder Factor in kg/m^3	0.7–0.8	0.6–0.7	0.5–0.6	0.3–0.5	0.2–0.3

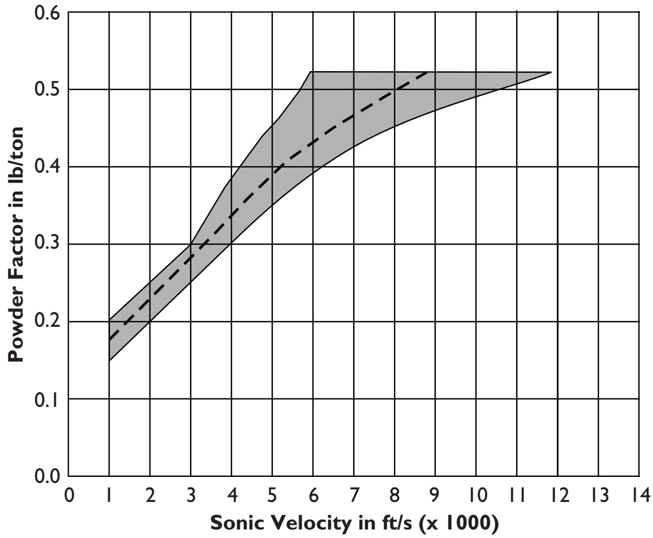

Figure 24.6 Relationship of powder factor with sonic velocity.

Many other correction values and subsequent interpretations are required to be applied to these RMR ratings to get a more purposeful picture of the rock mass from the civil engineering viewpoint. For this purpose additional tables have been presented in books written on this subject. Presenting all such information is beyond the scope of this book.

24.4.2.3 RTQI based classification

Rock Tunnel Quality Index was proposed by Barton and others at the Norwegian Geotechnical Institute on the basis of several case histories of tunnels. Therefore, it is referred to as NGI Q Index.

Since this index is applicable for tunnel construction, where a long stretch of rock mass in limited cross section is under consideration, it does not have much relevance on blasting in large surface mines. Therefore, it has not been discussed at length in this book.

24.4.3 BI index based classification

Blastability index is applicable to rock mass as well as rocks. Blastability index for rock mass aims at its direct correlation to fragment size distribution resulting from a blast. Therefore, when it comes to blasting, it is far more important than the basis of rock mass classification described earlier in this chapter.

Several researchers have attempted to correlate many different properties of rock and rock mass to arrive at an index that will give an indicative idea of the results of blast in the rock mass. Some formulae have been described below.

24.4.3.1 Index proposed by Hansen (1968)

After carrying out experiments at Morrow Point Dam, Hansen proposed an equation, as below, for estimating the quantity of explosive required for optimum fragmentation of the rock mass.

$$Q = B_2 * (0.236 * (h/B + 1.5) + 0.1984 * C * (h/B + 1.5))$$

where

Q = Total charge in a single blasthole with free burden in kg
B = Burden in m
H = Height of free face on m
C = A rock constant to be estimated by tests

He also proposed that the total charge Q computed by the above equation be corrected by the following equation.

$$Q_c = 0.8 * (F/E) * (S/B)$$

where

F = Fixation factor depending upon blasthole inclination
E = Explosive factor depending upon the explosive
S = Spacing
B = Burden

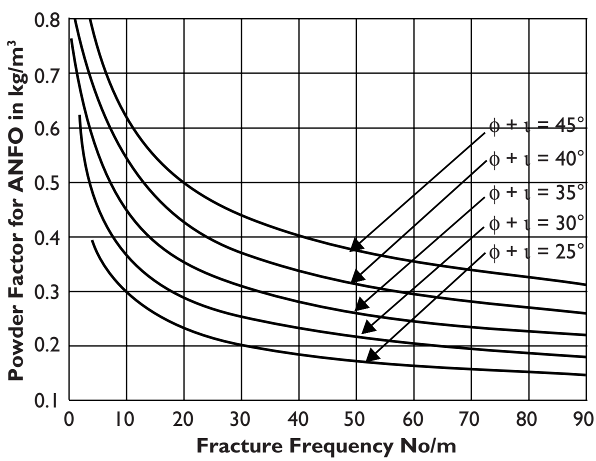

Figure 24.7 Relationship between powder factor and fracture frequency.

24.4.3.2 Index proposed by Hainen and Dimock (1976)

While working in a copper mine in Nevada in the USA, Heinen and Dimock investigated the correlation between powder factor and the velocity of sound wave in the rock mass. For the investigation they studied several blasts of rectangular patterns measuring 18 × 21 ft, 21 × 24 ft, 24 × 27 ft, 27 × 30 ft and 30 × 33 ft.

The results obtained by them are plotted in Figure 24.6. In the figure the dotted line represents the mean of the rock mass sonic velocities for which the powder factor is valid.

24.4.3.3 Index proposed by Ashby (1977)

Ashby proposed an empirical equation for powder factor from his observations in Bougainville Copper Mine. The variables in his equation were fracture frequency which, in effect, meant the density of fractures in the rock mass, and the friction angle which is related to the joint shear strength. The equation is as follows:

$$Q = (0.56 * \tan(\phi + \iota))/D^{0.333}$$

where
Q = Powder factor for ANFO in kg/m^3
ϕ = Friction angle in °
ι = Roughness angle in °
D = Frequency of fractures in number/m

The graph constructed for the above equations is shown in Figure 24.7.

24.4.3.4 Index proposed by Langefors (1978)

As per the concept proposed by Langefors, for every rock mass there is certain powder factor C_0 for which there is no appreciable throw. He then proposed that for satisfactory breakage of the rock mass the powder factor Q should be taken as $Q = 1.2 * C_0$.

For the brittle crystalline granite rock in which he carried out his research the value of C_0 was found to be 0.17 kg/m^3. For other rocks the value of C_0 lies between 0.18 to 0.35 kg/m^3.

24.4.3.5 Index proposed by Lilly (1986)

Lilly developed an equation for the blastability of rock mass based on five parameters related to the site conditions. This equation is as under:

$$BI = 0.5 * (RMD + JPS + JPO + SGI + H)$$

where
BI = Blastability index
RMD = Rock mass description

JPS = Joint plane spacing
JPO = Joint plane orientation
SGI = Specific gravity influence
H = Rock hardness on Moh's scale

The values to be used for the above variables are as below.

ROCK MASS DESCRIPTION

The RMD values are

10 – for powdery/friable rockmass
20 – for blocky rockmass
50 – for totally massive rockmass

JOINT PLANE SPACING

The JPS values are

10 – for joints with spacing <0.1 m
20 – for joints with spacing 0.1 to 1 m
50 – for joints with spacing >1 m

JOINT PLANE ORIENTATION

The JPO values are

10 – when the joint orientation is less than 10° with the horizontal plane.
20 – when the absolute difference between joint dip angle and face dip direction is less than 30°.
30 – when the absolute difference between joint dip angle and face dip direction is more than 60°.
40 – when the absolute difference between joint dip angle and face dip direction is between 30° and 60°.

SPECIFIC GRAVITY INFLUENCE

The SGI values are to be calculated as:

$$SGI = 0.25 * SG - 50$$

where SG = rock mass specific gravity in kg/m^3

ROCK HARDNESS ON MOH'S SCALE

The value to be used for H is the hardness value lying between 1 and 10 on Moh's scale of hardness. These values are given in an appendix at the end of this book.

From the value of blastability index proposed by Lilly calculated by formula given above one can calculate the powder factor Q and energy E factor by using equations as under.

$$Q = 0.004 * BI$$
$$E = 0.015 * BI$$

where
Q = Quantity of ANFO in kg/ton of rockmass
E = Energy required in MJ/ton of rockmass

Similarly, the rock factor to be used in predicting fragment size distribution in the Kuz Ram model, proposed by Cunningham, can be calculated by multiplying BI (i.e. Lilly's blastability index) by 0.12.

24.4.3.6 Index proposed by Ghose (1988)

Ghose proposed a blastability index based on many properties of rock and rock mass. His approach was somewhat similar to that of Lilly. The equation for the blastability index is as under.

$$BI = (DR + DSR + PLR + JPO + AF_1 + AF_2)$$

where
BI = Blastability index
DR = Density ratio
DSR = Discontinuity spacing ratio
PLR = Point load index strength ratio
JPO = Joint plane orientation ratio
AF_1 = Adjustment factor 1
AF_2 = Adjustment factor 2

The values for these factors are to be chosen from Table 24.6.

Once the value of blastability is determined, it can be used to find out the powder factor from the correlation between the two as given in Table 24.7.

24.4.3.7 Index proposed by Gupta (1990)

Gupta et al suggested the following equation for charge factor based on their field observations.

$$CF = 0.278 * B^{-0.407} * F^{0.62}$$

where
B = Effective burden in m
F = Protodyakonov strength index

The Protodyakonov strength index is to be calculated from the following equation.

$$F = 1.06 * C^2/E$$

where
C = Compressive strength in kg/cm^2
F = Modulus of elasticity in kg/cm^2

24.4.3.8 Index proposed by JKMRC (1996)

The Julius Kruttschnitt Mineral Research Center in Australia has developed a blast fragmentation model through the efforts of many researchers working in the organization.

A rock factor used in the fragmentation analysis can be used for prediction of the powder factor.

Exact details and the formulae used for the calculation are not divulged but are built into software prepared by the organization.

Factors taken into consideration while calculating the powder factor include:

strength, density and Young's modulus of the rockmass, average in situ block size, influence of structure, target fragment size, heave desired, confinement provided, scale of operation and groundwater.

For appropriate input in the software the values of the variables are obtained as below:

1 Strength, density and Young's modulus by laboratory tests.
2 Block size through field measurements of exposed rock surface.
3 Target fragment size is the desired value.
4 For all other factors the input value is to be chosen from a scale of 1 to 9. A value of 5 is to be treated as neutral, values 4 to 1 are progressively favorable and 5 to 9 are progressively adverse.

Cast blasting needs the highest heave.

A front end loader needs larger heave to have a spread (instead of a heap) of the blasted material.

For a bench with an open free face the confinement is neutral.

The JLMRC fragmentation model is gaining wider acceptability.

24.4.3.9 Index proposed by Han, Weiya and Shouvi (2000)

These researchers have used an Artificial Neural Network approach for determining rockmass blastability through a computer program. The logic of the equation is based on the following Expression:

$$K = f\{d_{cp}, L, S, R_{cd}, E_d, P_c\}$$

Table 24.8 Blastability calculation by JKMRC software.

Parameter	*Dragline Operation*	*Dragline Operation with Cast Blasting*	*Shovel Operation*	*Shovel Operation in Wet Conditions*	*Front End Loader Operation*
Rockmass Parameters					
Rock Strength in MPa	60	60	50	50	40
Density in g/cc	2.51	2.51	2.47	2.47	2.42
Young's Modulus in GPA	12	12	10	10	10
Structural Parameters					
Block Size in m	2	2	2	2	0.3
Structural Favorability*	5	5	5	5	3
Design Parameters					
Target Fragment Size in m	0.5	0.5	0.3	0.3	0.15
Heave Desired*	5	10	5	5	7
Confinement of Blast*	5	5	5	5	7
Scale of Operation*	3	3	5	5	7
Environmental Parameters					
Groundwater Presence*	1	1	1	5	1
Output Given by Software					
Strength	0.30	0.30	0.25	0.25	0.20
Breakage	0.08	0.08	0.13	0.13	0.06
Heave	0.25	0.51	0.26	0.26	0.36
Modifier	–0.02	0.03	0.00	0.08	0.02
Powder Factor in kg/ton	0.18	0.24	0.17	0.21	0.16
Powder Factor in kg/m^3	0.44	0.61	0.42	0.52	0.39

where

K = Output parameter i.e. blastability Index
d_{cp} = Mean fragment size in mm
L = Total length of fractures in a block measuring 2×2 m
S = Mean distance of fractures in the 2×2 m block
R_{cd} = Dynamic compressive strength of rock in MPa
E_d = Dynamic elastic modulus of rock in GPa
P_c = Percentage of unqualified blocks in %

Nowadays, many rotary blasthole drills are equipped with a system to measure various rock characteristics based on the penetration rates and the power required in terms of rotary speed, weight on the bit and torque, while the drilling is being carried out.

Analysis of the data obtained through such systems by means of software enables planning of a blast in a much more effective and accurate manner than the use of the different equations for blastability mentioned above.