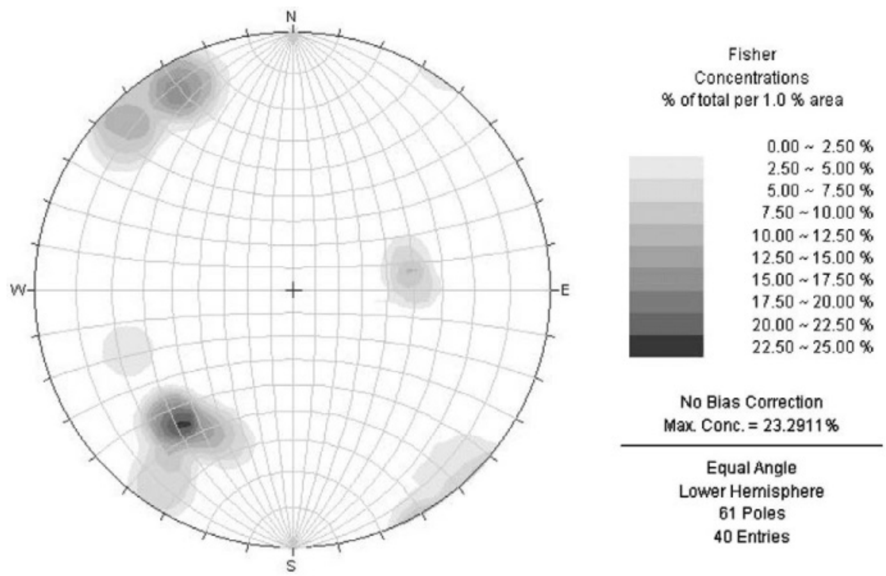


Figure 3.29 Computer generated contoured stereographic projection (after Rocscience, 1999).

3.7 Rock mass classification

3.7.1 The nature and use of rock mass classification schemes

Whenever possible, it is desirable that mining rock mechanics problems be solved using the analytical tools and engineering mechanics-based approaches discussed in later chapters of this book. However, the processes and interrelations involved in determining the behaviour of the rock surrounding a mining excavation or group of excavations are sometimes so complex that they are not amenable to enginerring analysis using existing techniques. In these cases, design decisions may have to take account of previous experience gained in the mine concerned or elsewhere.

In an attempt to quantify this experience so that it may be extrapolated from one site to another, a number of classification schemes for rock masses have been developed. These classification schemes seek to assign numerical values to those properties or features of the rock mass considered likely to influence its behaviour, and to combine these individual values into one overall classification rating for the rock mass. Rating values for the rock masses associated with a number of mining or civil engineering projects are then determined and correlated with observed rock mass behaviour. Aspects of rock mass behaviour that have been studied in this way include the stable spans of unsupported excavations, stand-up times of given unsupported spans, support requirements for various spans, cavability, stable pit slope angles, hangingwall caving angles and fragmentation. A number of these assessments made from geotechnical data collected in the exploration or feasibility study stages of a mining project may provide useful guides to the selection of an appropriate mining method.

Although the use of this approach is superficially attractive, it has a number of serious shortcomings and must be used only with extreme care. The classification scheme approach does not always fully evaluate important aspects of a problem, so that if blindly applied without any supporting analysis of the mechanics of the problem, it can lead to disastrous results. It is particularly important to recognise that the classification schemes give reliable results only for the rock masses and circumtances for which the guide-lines for their application were originally developed. It is for this reason that considerable success has been achieved in using the approach to interpolate experience within one mine or a group of closely related mines, as described by Laubscher (1977), for example.

Hoek and Brown (1980), Goodman (1993) and Brown (2003), among others, have reviewed the considerable number of rock mass classification schemes that have been developed for a variety of purposes. Two of these schemes, the NGI tunnelling quality index (Q) developed by Barton *et al*. (1974) and the CSIR goemechanics or Rock Mass Rating (RMR) scheme developed by Bieniawski (1973, 1976), are currently widely used in civil engineering and in mining practice. Bieniawski's RMR scheme has been modified by Laubscher (1977, 1990), particularly for use in cave mining applications. Because of their widespread use in mining practice, the basic RMR and Q systems will be outlined here. The more recent GSI system introduced by Hoek (1994) and developed further by Marinos and Hoek (2000) will also be discussed.

3.7.2 Bieniawski's geomechanics classification

Bieniawski (1973, 1976) developed his scheme using data obtained mainly from civil engineering excavations in sedimentary rocks in South Africa. Bieniawski's scheme uses five classification parameters.

1 **Strength of the intact rock material**. The uniaxial compressive strength of the intact rock may be measured on cores as described in section 4.3.2. Alternatively, for all but very low-strength rocks, the point load index (section 4.3.9) may be used.
2 **Rock Quality Designation** (*RQD*) as described in section 3.3.
3 **Spacing of joints**. In this context, the term joints is used to describe all discontinuities.
4 **Condition of joints**. This parameter accounts for the separation or aperture of discontinuities, their continuity or persistence, their surface roughness, the wall condition (hard or soft) and the nature of any in-filling materials present.
5 **Groundwater conditions**. An attempt is made to account for the influence of groundwater pressure or flow on the stability of underground excavations in terms of the observed rate of flow into the excavation, the ratio of joint water pressure to major principal stress, or by a general qualitative observation of groundwater conditions.

The way in which these parameters are incorporated into Bieniawski's geomechanics classification for jointed rock masses is shown in Part (a) of Table 3.5. For various ranges of each parameter, a rating value is assigned. The allocation of these rating values allows for the fact that all parameters do not necessarily contribute equally to the behaviour of the rock mass. The overall Rock Mass Rating (RMR) is obtained by adding the values of the ratings determined for the individual parameters. This RMR value may be adjusted for the influence of discontinuity orientation by applying the corrections given in Part (b) of Table 3.5. The terms used for this purpose are explained in Table 3.6. (When falling or sliding of blocks of rock from the roof or walls of an excavation is a possibility, this approach should not be relied upon. A wedge analysis of the type described in Chapter 9 should be used.) Part (c) of

Table 3.5 Geomechanics classification of jointed rock masses (after Bieniawski, 1989).

(a) Classification parameters and their ratings

	Parameter		Ranges of Values						
1	Strength of inftact rock material	point–load–strength index(MPa)	> 10	4–10	2–4	1–2	for this low range, uniaxial compression test is preferred		
		uniaxial compressive strength (MPa)	> 250	100–250	50–100	25–50	5–25	1–5	< 1
	rating		15	12	7	4	2	1	0
2	drill core quality *RQD* (1%)		90–100	75–90	50–75	25–50	< 25		
	rating		20	17	13	8	3		
3	joint spacing (m)		> 2	0.6–2	0.2–0.6	0.06–0.2	< 0.06		
	rating		20	15	10	8	5		
4	condition of joints		very rough surfaces, not continuous, no separation, unweathered joint wall rock	slightly rough surfaces, separation < 1 mm, highly weathered walls	slightly rough surfaces, separation < 1 mm, higly weathered walls	slickensided surfaces or gouge < 5 mm thick or separation 1–5 mm continuous joints	soft gouge > 5 mm thick or separation > 5 mm. continuous joints		
	rating		30	25	20	10	0		
5	groundwater	inflow per 10 m tunnel length ($\ell\ \mathrm{min}^{-1}$)	none	< 10	10–25	25–125	> 125		
		or $\frac{\text{joint water pressure}}{\text{major principal stress}}$	0	< 0.1	0.1–0.2	0.2–0.5	> 0.5		
		or general conditions	completely dry	damp	wet	dripping	flowing		
	rating		15	10	7	4	0		

(b) Rating adjustment for joint orientations

strike and dip orientations of joints		very favourable	favourable	fair	unfavourable	very unfavourable
Rating	tunnels and micro	0	−2	−5	−10	−12
	foundations	0	−2	−7	−15	−25
	slopes	0	−5	−25	−50	−60

(c) Rock mass classes determined from total ratings

Ratings	100 ← 81	80 ← 61	60 ← 41	40 ← 21	< 20
Class no.	I	II	III	IV	V
Description	very good rock	good rock	fair rock	poor rock	very poor rock

(d) Meaning of rock classes

Class no.	I	II	III	IV	V
average stand-up time	20 years for 15 m span	1 year for 10 m span	1 week for 5 m span	10 hours for 2.5 m span	30 minutes for 1.0 m span
cohesion of the rock mass (kPa)	> 400	300–400	200–300	100–200	< 100
friction angle of the rock mass	45°	35°–45°	25°–35°	15°–25°	< 15°

Table 3.6 The effects of joint strike and dip in tunnelling (after Bieniawski, 1989).

Strike perpendicular to tunnel axis				Strike parallel to tunnel axis		
Drive with dip		Drive against dip				
Dip 45°–90°	Dip 20°–45°	Dip 45°–90°	Dip 20°–45°	Dip 45°–90°	Dip 20°–45°	Dip 0°–20° irrespective of strike
very favourable	favourable	fair	unfavourable	very unfavourable	fair	fair

Table 3.7 Determination of rock mass rating.

Parameter	Value or description	Rating
1. strength of intact rock material	150 MPa	12
2. *RQD*	70	13
3. joint spacing	0.5 m	10
4. condition of joints	slightly rough surfaces separation < 1 mm slightly weathered joint wall rock	25
5. groundwater	water dripping	4
	Total RMR	64

Table 3.5 sets out the class and description assigned to rock masses with various total ratings. The interpretation of these ratings in terms of stand-up times of underground excavations and rock mass strength parameters is given in Part (d) of Table 3.5. The variation with RMR of the *in situ* strengths and deformabilities of jointed rock masses will be discussed in section 4.9.

As an example of the application of Bieniawski's classification, consider a granitic rock mass for which the RMR is determined as shown in Table 3.7. An adit is to be driven into the granite oriented such that the dominant joint set strikes roughly perpendicular to the adit axis and dips at 35° against the drive direction. From Table 3.6 this situation is described as unfavourable for which a rating adjustment of – 10 is obtained from Part (b) of Table 3.5. Thus the final RMR is reduced to 54 which places the rock mass in Class III with a description of fair.

3.7.3 The NGI Q system

This classification was developed by Barton *et al.* (1974) as a means estimating support requirements for hard rock tunnels in Scandinavia as a function of an index of rock mass quality, defined as

$$Q = \left(\frac{RQD}{J_{\mathrm{n}}}\right) \times \left(\frac{J_{\mathrm{r}}}{J_{\mathrm{a}}}\right) \times \left(\frac{J_{\mathrm{w}}}{\mathrm{SRF}}\right) \tag{3.11}$$

where

RQD is the **Rock Quality Designation** discussed in section 3.3;

$\mathbf{J_n}$ is the **Joint Set Number** which represents the number of joint sets in the rock mass, varying from 0.5 for a massive rock mass with no or few joints to 20 for crushed or diaggregated rock;

$\mathbf{J_r}$ is the **Joint Roughness Number** which represents the roughness of the structural features in the rock mass, varying from 0.5 for slickensided, planar surfaces to 5 for non-persistent structures with spacings larger than 3 m;

J_a is the **Joint Alteration Number** representing the condition or degree of alteration of the structures in the rock mass, varying from 0.75 for wall-wall contact in unaltered rock or for joints containing tightly healed, hard, non-softening, impermeable filling to 20 for structures with thick fillings of clay gouge;

J_w is the **Joint Water Reduction Factor** representing the groundwater conditions, varying from 0.05 for exceptionally high inflows or for water pressure continuing without noticeable decay to 1.0 for dry conditions or minor inflows; and

SRF is the **Stress Reduction Factor** which is a coefficient representing the effect of stresses acting on the rock mass, varying from 0.5 for high stress but tight structure conditions in good quality rock to 400 for heavy squeezing rock pressures or heavy rock burst conditions and immediate dynamic deformations in massive rock.

The three quotients in equation 3.11 may be taken to represent the block size, the inter-block frictional shear strength and the "active stress", respectively. The details of how the six parameters in the Q system are determined are given by Barton *et al.* (1974), Hoek and Brown (1980), Priest (1993) and Barton (2002), for example. Except for some changes to the SRF parameter introduced to account for rockburst conditions, the original Q system has remained essentially unchanged since it was first developed. Possible Q values range from 0.001 to 1000 on a logarithmic scale. The system defines nine geotechnical classes of rock mass ranging from exceptionally poor ($Q \leq 0.01$) to exceptionally good ($Q \geq 400$). The application of the Q system in underground mining rock mechanics will be discussed at various points in this book. It should be noted that some applications use the parameter Q' which is the value of Q with the active stress term J_w/SRF, put equal to unity.

3.7.4 Geological strength index (GSI)

As part of the continuing development and practical application of the Hoek-Brown empirical rock mass strength criterion to be discussed in section 4.9.1, Hoek (1994) and Hoek *et al.* (1995) introduced a new rock mass classification scheme known as the Geological Strength Index (GSI). The GSI was developed to overcome some of the deficiencies that had been identified in using the RMR scheme with the rock mass strength criterion.

The GSI was developed specifically as a method of accounting for those properties of a discontinuous or jointed rock mass which influence its strength and deformability. As will become apparent in Chapter 4, the strength of a jointed rock mass depends on the properties of the intact pieces of rock and upon the freedom of those pieces to slide and rotate under a range of imposed stress conditions. This freedom is controlled by the shapes of the intact rock pieces as well as by the condition of the surfaces separating them. The GSI seeks to account for these two features of the rock mass, its structure as represented by its blockiness and degree or interlocking, and the condition of the discontinuity surfaces. Using Figure 3.30 and with some experience, the GSI may be estimated from visual exposures of the rock mass or borehole core.

It will be noted that the GSI does not explicitly include an evaluation of the uniaxial compressive strength of the intact rock pieces and avoids the double allowance for discontinuity spacing as occurs in the RMR system. Nor does it include allowances for water or stress conditions which are accounted for in the stress and stability analyses with which the Hoek-Brown criterion is used. Although the origin and petrography of the rock are not represented in Figure 3.30, the rock type will usually constrain the range of GSI values that might be encountered in rock masses of that type. Marinos

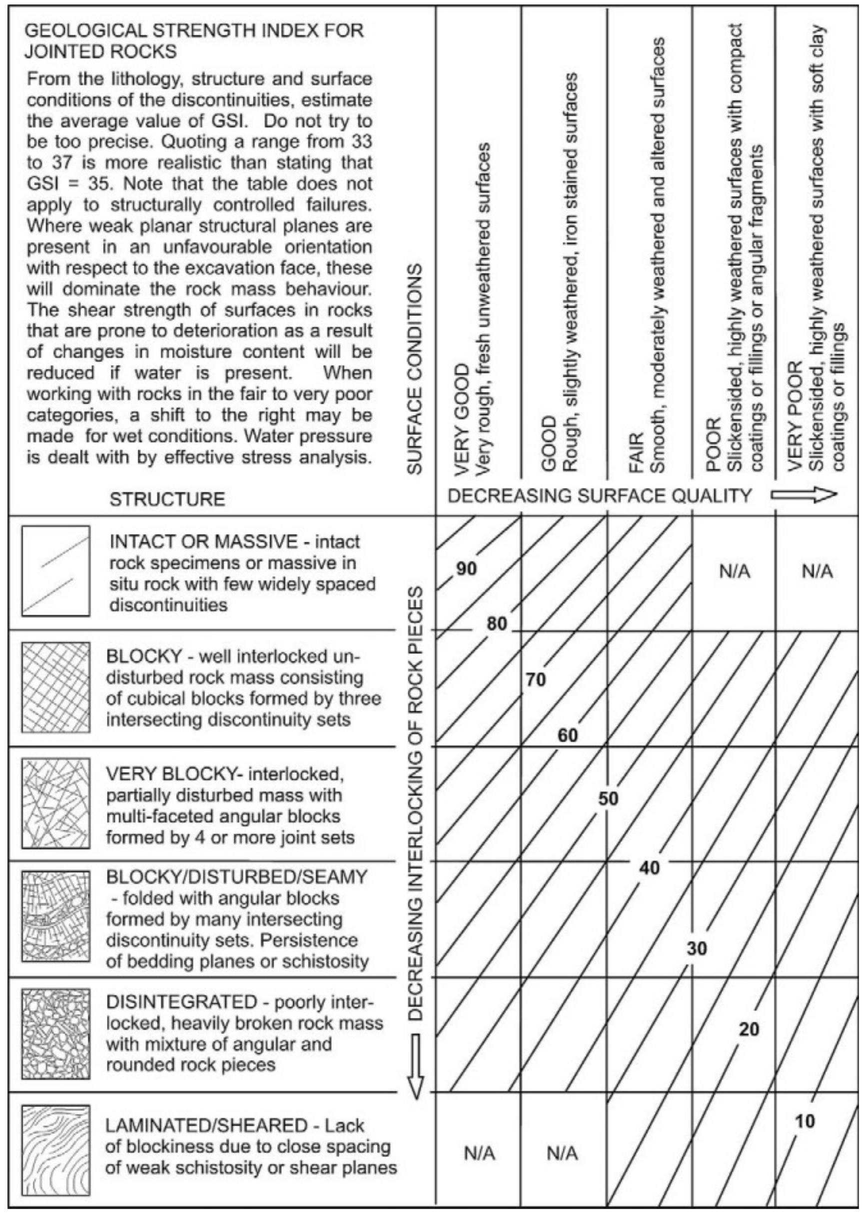


Figure 3.30 Geological Strength Index (GSI) for jointed rock masses (after Hoek, 2003).

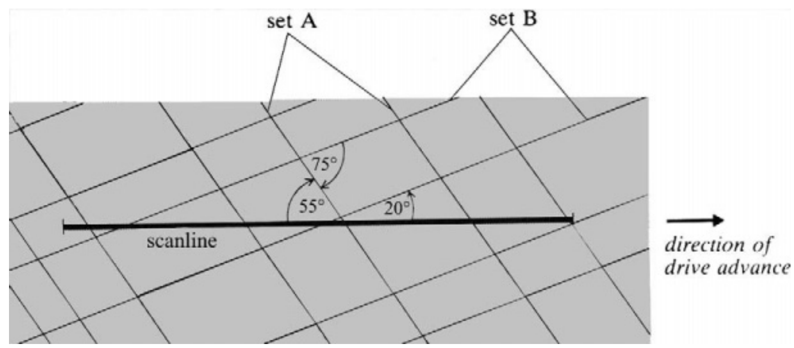
and Hoek (2000) present a series of indicative charts which show the most probable ranges of GSI values for rock masses of several generic rock types.

Problems

1 A scanline survey is to be carried out on the vertical wall of an exploration drive. The rock mass contains two sets of parallel discontinuities whose traces on the wall are mutually inclined at 75° as shown in the diagram. The traces of set A make an

angle of 55° with a horizontal scanline. Large numbers of measurements give the apparent mean spacings of sets A and B along the scanline as 0.450 m and 0.800 m, respectively.

(a) Calculate the mean normal spacing of each set.
(b) What is the mean spacing of all discontinuities in the direction of the scanline?
(c) Assuming that the combined discontinuity spacings follow a negative exponential distribution, estimate the *RQD* of the rock mass in the direction of the scanline.



2 A preliminary borehole investigation of a sandstone produced 30 m of core that contained 33 drilling breaks, 283 iron-stained joints and 38 other discontinuities of uncertain origin.

(a) Calculate the bandwidths within which the overall mean discontinuity spacing in the direction of the borehole axis lies at the 80% and 95% confidence levels.
(b) What approximate additional length of borehole is required to provide a value in a bandwidth of ± 8% of the true spacing value at the 95% confidence level?

3 Plot on the stereographic projection the great circle of the plane with the orientation (dip direction/dip) 110/50. What is the apparent dip of this plane in the direction 090°?

4 What are the trend and plunge of the line of intersection of the planes 110/50 and 320/60?

5 Plot the great circles and poles to the planes 156/32 and 304/82. What is the acute angle between these planes? In what plane is it measured?

6 A tunnel of square cross section has planar vertical sidewalls of orientation 230/90. The lineation produced by the intersection of a planar joint with one sidewall plunges at 40° to the north-west. The same joint strikes across the horizontal roof in the direction 005° − 185°. What is the orientation of the joint plane?

7 A reference line is scribed on drill core for use in correctly orienting discontinuities intersected by the core. A certain planar discontinuity has the apparent orientation 120/35 measured with the reference line vertical. If the actual trend and plunge of the

borehole axis, and of the reference line, are 225° and 60° respectively, determine the true orientation of the discontinuity.

8 Assume that, for the rock mass described in Problem 1, both sets of discontinuities strike perpendicular to the drive axis. The intact rock material has a uniaxial compressive strength of 120 MPa, the joint surfaces are slightly rough with an average separation of 0.2 mm and, although there is water in the joints, the flow into the excavation is quite small.

Determine the basic CSIR geomechanics classification for this rock mass (Table 3.5). How does application of the adjustments for joint orientations for tunnelling given by Tables 3.5 and 3.6 affect this classification?

Is the adjusted RMR value likely to provide a satisfactory guide to roof stability in this case?