Rock Mass Characterization for Underground Hard Rock Mines

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ABSTRACT: Rock mass characterization is an integral part of rock engineering practice. There are several classification systems used in underground mine design, however, most Canadian mines rely on only one of three classification systems. It is interesting to note that these systems, RQD, RMR and Q system, have their origin in civil engineering. This paper reviews the current state of these classification systems as employed in the mining industry. The first part focuses on the determination of the field parameters, with emphasis on the modifications to each parameter over the last 20 years. The difference between classification parameters which influence rock mass strength estimation and those that influence engineering design is emphasized. The second part of the paper focuses on the design recommendations based on these systems such as maximum span, opening geometry and support recommendations. The paper concludes with reference to errors that may arise in particular conditions.

1 INTRODUCTION

Rock mass classification systems constitute an integral part of empirical mine design. The use of such systems can be either implicit or explicit. They are traditionally used to group areas of similar geomechanical characteristics, to provide guidelines of stability performance and to select appropriate support. In more recent years, classification systems have often been used in tandem with analytical and numerical tools. There has been a proliferation of work linking classification indexes to material properties such as modulus of elasticity, m and s for the Hoek & Brown failure criterion, etc. These values are then used as input parameters for the numerical models. Consequently the importance of rock mass characterization has increased over time.

The primary objective of all classification systems is to quantify the intrinsic properties of the rock mass based on past experience. The second objective is to investigate how external loading conditions acting on a rock mass influence its behaviour. An understanding of these processes can lead to the successful prediction of rock mass behaviour for different conditions.

Despite a plethora of empirical classification systems, only three systems are commonly used for mine design in Canadian mines. The first system is the Rock Quality Designation (RQD) proposed by Deere et al. (1967). Quite often this is the only information readily available at mine sites.

The other two widely used systems in Canadian mines are the Norwegian Geotechnical Institute's Q system, Barton et al. (1974) and the various versions of the Rock Mass Rating System (RMR), originally proposed by Bieniawski (1973). Interestingly, both systems trace their origin in tunnelling. Furthermore, both systems use RQD as one of their constitutive parameters.

The RMR and Q systems have evolved over time to better reflect the perceived influence of various rock mass factors on excavation stability. The introduced modifications have arguably enhanced the applicability of these classification systems, but there are still areas of potential errors and confusion. This paper discusses the evolution of these systems as well as problems associated with estimating the Q, RMR and RQD indexes.

2 ESTIMATION OF RQD, Q AND RMR

Changes associated with the classification systems are of two forms. The first one lies with the actual properties of the systems, the way these are determined on site and the associated weight assigned to each parameter. The second form is the evolution of support recommendations as new methods of reinforcement such as cable bolting and reinforced shotcrete gained acceptance.

2.1 *RQD*

RQD is a modified core recovery index defined as the total length of intact core greater than 100mm in length, divided by the total length of the core run. The resulting value is presented in the form of a percentage, Figure 1. RQD should only be calculated over individual core runs, usually 1.5 metrers long.

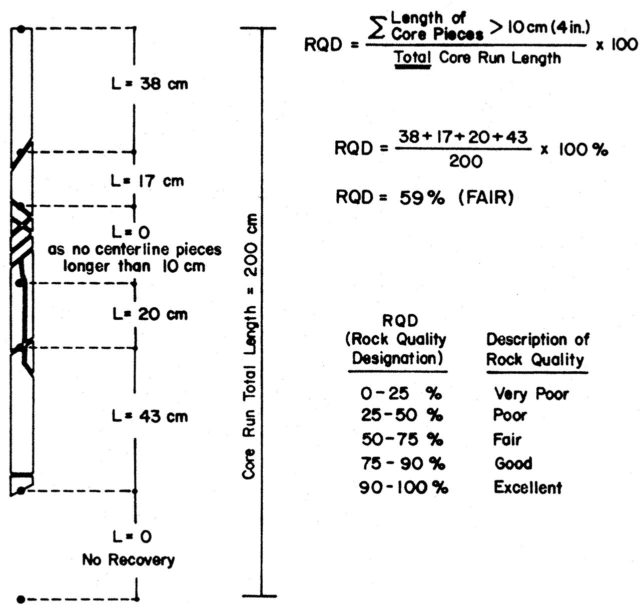

Figure 1. Procedure for determining RQD, after Deere and Deere (1988).

RQD was originally introduced for use with NX-size core (54.7 mm) and a threshold value of 100mm is used. Over the years, several correction factors have been introduced to calculate RQD for other core diameters. The most popular approach is to define the threshold value as equal to twice the core diameter. Consequently a threshold core length of 75mm would be used for BQ core and 50mm for AQ size core. The case for a sliding scale of threshold values rests on the greater sensitivity of small diameter core to breaks due to drilling and handling.

It is the authors' opinion that the threshold value of 100 mm should be used for all core sizes since the RQD value should ignore breaks caused by drilling and handling. The only time the use of threshold values other than 100mm may be justified occurs when it is impossible to differentiate between natural breaks and drill induced breaks.

Intact lengths of core only consider core broken by joints or other naturally occurring discontinuities so drill breaks must be ignored, otherwise the resulting RQD will underestimate the rock mass quality.

In practice, a high RQD value does not always translate to high quality rock. It is possible to log 1.5 metres of intact clay gouge and describe it as having 100% RQD. This may be true based on the original definition of RQD, but is very misleading and gives the impression of competent rock. To avoid this problem, a parameter called 'Handled' RQD (HRQD) was introduced, Robertson (1988). The HRQD is measured in the same way as the RQD, after the core has been firmly handled in an attempt to break the core into smaller fragments. During handling, the core is firmly twisted and bent, but without substantial force or the use of any tools.

An estimate of RQD is often needed in areas where line mapping or area mapping has been conducted. In these areas it is not necessary to use core since a better picture of the rock mass can be obtained from line or area mapping. Two methods for estimating RQD are recommended:

(a) For line mapping data, an average joint spacing can be obtained (number of features divided by traverse length). Bieniawski (1989) relying on previous work by Priest and Hudson (1976) has linked average joint spacing to RQD, Figure 2. The ratings in the figure refer to RMR_{89}. It should be noted that the maximum possible RQD based on joint spacing given by Bieniawski actually corresponds to the best-fit relationship proposed by Priest and Hudson. The RQD can be estimated from average joint spacing based on the following equation by Priest and Hudson (1976):

$$RQD = 100\, e^{-.1l}\,(.1l + 1) \qquad (1)$$

where

λ = 1/(joint frequency)

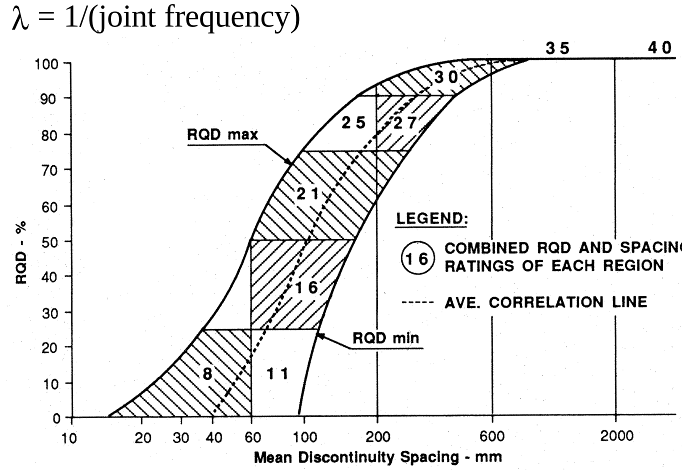

Figure 2. Relationship between discontinuity spacing and RQD, after Bieniawski (1989).

Relating joint spacing to average RQD using Figure 2 will likely lead to conservative estimates. Consequently the use of equation (1) is probably more appropriate. It should be noted, however, that this relationship is also dependent on the direction of the traverse. For a given average joint spacing there is a significant range in possible RQD values. RQD should not be calculated from line mapping based on the same approach used for core (sum of not-jointed mapped distances greater than 100mm). Line mapping distances are seldom accurate enough to warrant this approach.

(b) For area mapping, a more three-dimensional picture of joint spacing is often available. Palmström (1982) defines J_v as number of joints present in a cubic metre of rock:

$$J_v = \sum \frac{1}{S_i} \qquad (2)$$

where

S = joint spacing in metres for the actual joint set.

RQD is related to J_v by the following equation:

$$RQD = 115 - 3.3\, J_v \qquad (3)$$

and RQD = 100% when $J_v \leq 4.5$.

This approach averages out some of the anisotropy in the RQD term and gives a more representative value.

The main drawbacks to RQD are that it is sensitive to the direction of measurement, and it is insensitive to changes of joint spacing, if the spacing is over 1m. The main use of RQD is to provide a warning that the rock mass is probably of low quality.

2.2 *RMR*

The RMR classification system, Bieniawski (1989), was developed for characterizing the rock mass and for providing a design tool for tunneling. The system has evolved due to a better understanding of the importance of the different parameters and increased experience. Table 1 summarizes the evolution of RMR ratings as well as the modifications to the weights assigned to each factor. Table 2 provides the most recent version of the RMR system.

The addition of the five ratings gives an RMR value that ranges from 8 to 100. This value can be modified to account for favourable/unfavourable orientation of discontinuities as applied to underground excavation. Figure 3 shows how RMR can be used to predict tunnel stand up time.

The joint orientation adjustment has been developed for tunnelling applications and consists of estimating how favourable the discontinuity orientation is with respect to the tunnel. The rating for the assessment of the joint orientation factor has not changed with time, however, in 1989 the assessment of subhorizontal joints was modified from unfavourable to fair for the effect on stability of tunnel backs.

Table 1. Evolution of RMR Ratings

	1973	1974	1975	1976	1989
Rock Strength	10	10	15	15	15
RQD	16	20	20	20	20
Discontinuity Spacing	30	30	30	30	20
Separation of joints	5				
Continuity of joints	5				
Ground Water	10	10	10	10	15
Weathering	9				
Condition of joints		15	30	25	30
Strike and Dip orientation		15			
Strike and Dip orientation for tunnels	3-15		0-12	0-12	0-12

The main factors that have been changed with the RMR system are the weightings given to joint spacing, joint condition and ground water. In assessing both RQD and joint spacing, the frequency of jointing is included twice. In the 1989 version of RMR, the weighting factor for the spacing term was reduced and the influence of both water and joint condition was increased.

A further important modification to the RMR was in the definition of different rock mass classes (i.e. very good, good rock, etc.). Since 1976 the rock mass classes are divided in intervals of 20, Table 2.

In the latest version of the RMR system, the condition of discontinuities was further quantified to produce a less subjective appraisal of discontinuity Condition, Table 2 section E. This brings RMR closer to the Q-system which allows the assessment of discontinuity condition by two independent terms, J_r and J_a.

Despite efforts to specifically modify the RMR system for mining (Laubscher (1976), Kendorski et al. (1983) etc.) most Canadian mines use one of the versions of RMR given in Table 1. Depending on the required sensitivity and the design method used, this might lead to discrepancies.

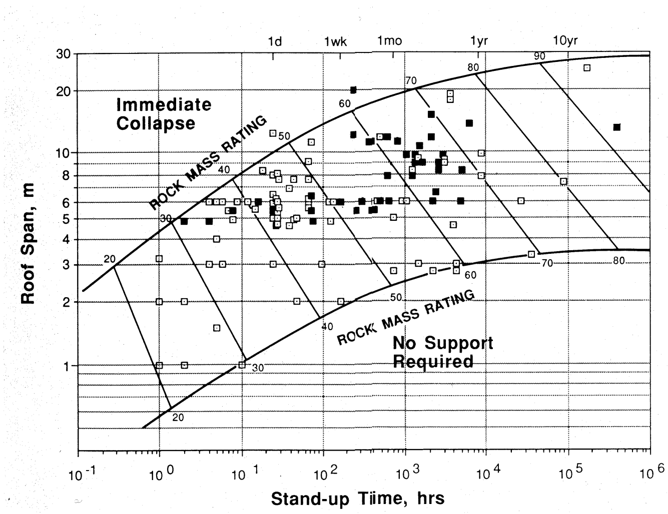

Figure 3. Relationship between Stand-up time, span and RMR classification, after Bieniawski (1989).

The main advantage of the RMR system is that it is easy to use. Common criticisms are that the system is relatively insensitive to minor variations in rock quality and that the support recommendations appear conservative and have not been revised to reflect new reinforcement tools.

2.3 *Q-Tunnelling index*

The Q or NGI (Norwegian Geotechnical Institute) classification system was developed by Barton, Lien and Lunde (1974), primarily for tunnel design work. It expresses rock quality, Q, as a function of 6 independent parameters:

$$Q = \frac{RQD}{J_n} x \frac{J_r}{J_a} x \frac{J_w}{SRF} \qquad (4)$$

where:

RQD = Rock quality designation
J_n is based on the number of joint sets
J_r is based on discontinuity roughness
J_a is based on discontinuity alteration
J_w is based on the presence of water
SRF is the Stress Reduction Factor

It has been suggested that RQD/J_n reflects block size, J_r/J_a reflects friction angle and J_w/SRF reflects effective stress conditions.

The main advantage to the Q classification system is that it is relatively sensitive to minor variations in rock properties. Except for a modification to the Stress Reduction Factor (SRF) in 1994, the Q system has remained constant. The descriptions used to assess joint conditions are relatively rigorous and leave less room for subjectivity, compared to other classification systems. Table 3 provides the latest version of the Q system, after Barton & Grimstad (1994).

One disadvantage of the Q system is that it is relatively difficult for inexperienced users to apply. The J_n term, based on the number of joint sets present in a rock mass, can cause difficulty. Inexperienced users often rely on extensive line mapping to assess the number of joint sets present and can end up finding 4 or more joint sets in an area where jointing is widely spaced. This results in a low estimate of Q.

An important asset of the Q system is that the case studies employed for its initial development have been very well documented. The use of the Q system for the design of support has also evolved over time. In particular Barton has introduced a design chart that accounts for the use of fibre reinforced shotcrete. This has been based on increased experience in tunnelling. For most mining applications, however it is common to rely on the design chart shown in Figure 4.

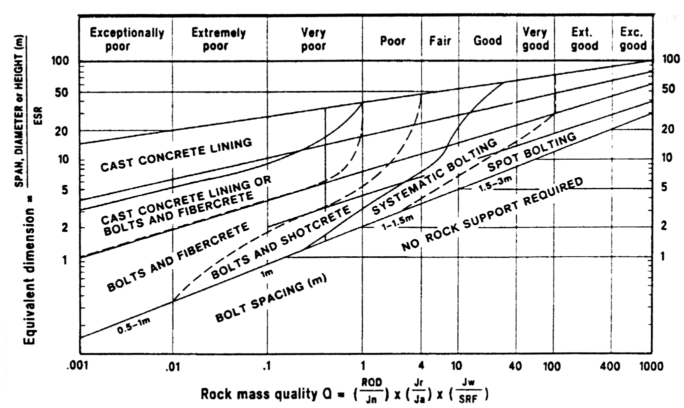

Figure 4. Design of Excavations based on the Q-system, after Barton & Grimstad (1994).

In mining the use of the ratio of Excavation Span/Equivalent Support Ratio (ESR) is limited. In open stope design this term is replaced altogether by the hydraulic radius. Alternatively one can assign different ESR values dependent on the type of opening (e.g. 5 for non-entry stopes, 1 for Shaft etc.). As there are limited documented case studies this involves considerable judgment.

The next section looks at the weightings given to the different parameters used in the Q and RMR classification systems, and how the two systems are related.

2.4 Comparative Rock Mass Property Weightings

Both the Q and RMR classification systems are based on a rating of three principal properties of a rock mass. These are the intact rock strength, the frictional properties of discontinuities and the geometry of intact blocks of rock defined by the discontinuities. For the Q system, the intact rock strength is only a factor in the context of the induced stress in the rock as defined by the SRF term.

In order to investigate the influence of these parameters, the approximate total range in values for RMR and Q are used as a basis of comparison. Table 4 shows the degree by which the three principal rock mass properties influence the values of the Q and RMR classification.

Table 4. Influence of Basic Rock Mass Properties on Classification, after Milne (1988).

	Q	RMR_{76}
Basic Range in Values	0.001 to 1000	8 to 100
Strength as % of the Total Range	19%	16%
Block Size as a % of the Total Range	44%	54%
Discontinuity Friction as a % of the Total Range	39%	27%

Table 4 shows the surprising similarity between the weightings given to the three basic rock mass properties considered. Despite this it should be noted that there is no basis for assuming the two systems should be directly related. The assessment for intact rock strength and stress is significantly different in the two systems. Despite these important differences between the two systems, it is common practice to use the rating from one system to estimate the rating value of the other. The following equation proposed by Bieniawski (1976) is the most popular, linking Q and RMR:

$$RMR = 9 \ln Q + 44 \qquad (5)$$

Referring to Table 5, it is evident that equation (5) does not provide a unique correlation between RMR and Q. Depending on the overall intact rock and discontinuity properties and spacing, different relationships between Q and RMR can be expected.

Another difference between RMR and Q is evident in the assessment of joint spacing. If three or more joint sets are present and the joints are widely spaced, it is difficult to get the Q system to reflect the competent nature of a rock mass. For widely spaced jointing, the joint set parameter J_n in the Q system appears to unduly reduce the resulting Q value.

Table 5. Correlation between RMR and Q, after Choquet and Hadjigeorgiou (1993).

Correlation	Source	Comments
RMR = 13.5 log Q + 43	New Zealand	Tunnels
RMR = 9 ln Q + 44	Diverse origin	Tunnels
RMR = 12.5 log Q + 55.2	Spain	Tunnels
RMR = 5 ln Q + 60.8	S. Africa	Tunnels
RMR = 43.89 - 9.19 ln Q	Spain	Mining Soft rock
RMR = 10.5 ln Q + 41.8	Spain	Mining Soft rock
RMR = 12.11 log Q + 50.81	Canada	Mining Hard rock
RMR = 8.7 ln Q + 38	Canada	Tunnels, Sedimentary rock
RMR = 10 ln Q + 39	Canada	Mining Hard rock

3 ROCK MASS CLASSIFICATION FOR MINING

One of the fundamental differences between tunnel and mine design approaches to rock mass classification is the large variation in the engineered openings in mining applications. For tunnel projects, tunnel orientation, depth and stress conditions are usually constant for significant portions of a project. In mining, none of these conditions can be assumed to be constant.

Due to the relatively constant engineered conditions in tunnelling, the stress condition has been included in the Q classification system and the relative orientation between the tunnel and critical joint set has been included in the RMR system. This approach has not been widely adopted in the mining industry because it would result in the same rock mass having dozens of classification values throughout the mine, depending on drift orientation, mining level and the excavation history. This would lead to significant confusion and render the rock classification values useless.

Two general approaches have been taken to allow these classification systems to be applied to mining conditions. The first approach was to try and create a complete design method from the classification systems by including other engineering and loading condition factors. One example of this is Laubscher's MRMR system (Mining Rock Mass Rating) proposed in 1977, which added factors such as blasting, weathering, multiple joint orientations and stress to obtain a term called the design rock mass strength. This term represents the unconfined strength of the rock mass in a specific mining environment, Laubscher (1990). The assessment of these added factors is quite subjective and requires an experience practitioner. The use of design classification systems such as this is not widespread in the Canadian mining industry.

The second approach consists of simplifying the classification system to only include factors dependent on the rock mass and to ignore environmental considerations such as stress and drift orientation. The resulting rating is solely dependent on the rock mass, and will give the same assessment for the same rock conditions at different depths and drift orientations within a mine. This simplified rock classification approach has been applied to both the RMR and Q systems. Q' is the modified Q classification with SRF = 1 and RMR' drops the joint orientation factor.

The Q' and RMR' classification values have been used in many different mining situations. With these design approaches, factors such as stress and the influence of joint orientation have been added as steps in the design process, which do not influence the classification values. Some of these design methods are mentioned in the following sections.

Many mine design approaches have been developed from the Q' and RMR' classification systems. In many cases these design approaches account for the influence of stress and joint orientation in the design steps and it would be incorrect to assess these factors twice by including them in the rock mass classification.

3.1 *Empirical Open Stope Design*

The Stability Graph method for open stope design, Potvin (1988), plots the stability number versus the hydraulic radius of a design surface, Figure 5. The stability number N is based on a Q' rating adjusted to account for stress condition (Factor A), joint orientation (Factor B) and the surface orientation of the assessed surface (Factor C). Based on an extensive database it is possible to predict the stability of an excavation.

Q' is used as opposed to Q because stress is assessed in the A factor. Furthermore, the stability graph method is based on an empirical database which does not include case histories where there was significant water inflow and should therefore be used with caution if water is present. The SRF term in the Q system includes descriptions for multiple shear or fault zones where it is felt the rock mass will be relaxed at any depth. Under these conditions the SRF value could arguably be included in the Q' classification because the presence of multiple shear zones is a description of the rock mass, not the stress condition.

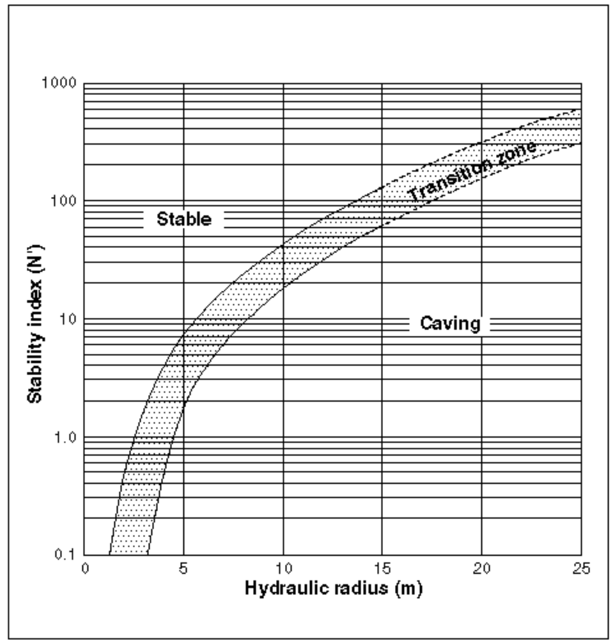

Figure 5. The Stability Graph, after Potvin (1988).

3.2 *Span Design*

The critical span design method for entry mining was developed for cut and fill mining Lang et al. (1991), Figure 6.

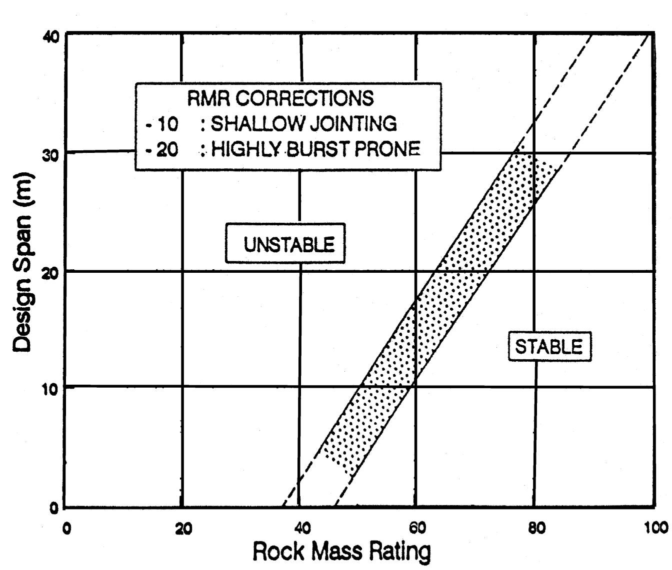

Figure 6. Design Span versus RMR, after Lang et al.

(1991).

The critical span is defined as the diameter of the largest circle that can be drawn within the boundaries of the exposed back. This span is then related to the RMR_{1976} value. The joint orientation factor is not used, however, reduction of 10 is given to the RMR value for joints dipping at less than 30°.

Under high stress, burst prone conditions a reduction of 20 is assigned to the RMR value. Figure 6 summarizes this method. The approach is suitable for zones that have local support, but have not been cable bolted. Stability is limited to a 3-month duration and the analysis is based on a horizontal back. The back is assumed to be in a relaxed state unless high stress/burst prone conditions are encountered.

3.2 *Failure Criteria*

That there is some link between the properties of a rock mass and its rock mass characterization rating would appear logical. Expressing this relationship in a definitive manner is however extremely difficult. Referring to Table 2 it can be shown that different classes of rock as defined by RMR have different frictional properties. For example an RMR of 60-81 would indicate cohesion of 300-400 kPa and an angle of friction between 35-45°. The case studies that support these relationships however are not known.

A popular empirical criterion in rock engineering has been proposed by Hoek and Brown (1980):

$$s_1 = s_3 + \sqrt{m(\frac{s_3}{s_c}) + s} \qquad (6)$$

where

$_1$ is the major principal effective stress at failure
$_3$ is the minor principal effective stress at failure
$_c$ is the uniaxial compressive strength of the intact rock
m & s are material constants

The determination of m and s has also been linked to rock mass classification ratings. When estimating the m and s values, the RMR' value should be used which does not include the joint orientation factor and the ground water factor has been set to 10, for dry conditions (Hoek et al., 1995). The m and s failure criteria and the equations relating m and s to rock classification are given below:

For undisturbed rock

$$m = m_i e^{(\frac{RMR - 100}{28})} \quad where\ m_i = m_{intact} \qquad (7)$$

$$s = e^{(\frac{RMR - 100}{9})} \qquad (8)$$

For disturbed rock

$$m = m_i e^{\frac{(RMR - 100)}{14}} \qquad (9)$$

$$s = e^{\frac{(RMR - 100)}{9}} \qquad (10)$$

It should be noted that the m and s values for disturbed and undisturbed rock could also be derived by using the equations for disturbed rock and adjusting the RMR classification values. The increase in the RMR classification between disturbed and undisturbed conditions can be calculated based on the equation (11).

$$RMR_{undisturbed} = RMR_{disturbed} + (50 - .5RMR_{disturbed})$$

(For RMR > 40) (11)

It is suggested (Hoek et al., 1995) that the Q classification system can also be used to estimate m and s. The joint water and SRF terms are set to 1.0 and the RMR value can then be estimated from the equation (5). This equation was developed to relate the original RMR and Q values when the joint water, SRF and joint orientation terms were all included in the classification. Section 2.4 describes how unreliable it is to relate the Q and RMR classification systems. It is doubtful if Equation (5) can be used with any confidence, especially when the joint water and SRF terms are ignored. It is more prudent to independently determine the RMR' and Q' values.

4 CONCLUSIONS

Rock mass classification is one of the only approaches for estimating large-scale rock mass properties. In the mining industry the Q and RMR classification system form the basis of many empirical design methods, as well as the basis of failure criteria used in many numerical modelling programs.

Classification systems have evolved as engineers have attempted to apply rock classification to a wider range of engineering problems. Rock mass classification is one of the only approaches available to estimate large-scale rock mass properties. It forms the basis of many empirical design methods as well as forming the basis of some failure criteria used in numerical modelling design approaches.

This paper attempts to highlight some of the potential problems when using the Q and RMR systems. Practitioners should be aware that classification and design systems are evolving and that old versions of classification systems are not always compatible with new design approaches. Some of the problems that can be encountered are outlined below:

a). More than one relationship has been suggested for relating joint spacing to RQD. These approaches do not all agree and the users should use more than one method. An estimate within 5% is more than adequate for RQD.

b) Practitioners sometimes estimate one classification and then derive a second classification from empirical relationships. Relating Q and RMR makes for an interesting comparison between classifications and may improve our understanding of the rock mass, however, the two systems should always be derived independently. There are many published relationships between Q and RMR, however, it is likely that no one relationship would work for all rock mass conditions. Relationships such as equation (5) provide a useful check. Nevertheless, there is no reason why the Q and RMR systems should be directly related.

c) Care must be taken when using classification systems with empirical design methods. The user must be sure that the classification system used matches the approach taken for the development of the empirical design method. A design method based on RMR_{76} cannot be expected to give the same results as RMR_{89}. Under these circumstances, for purposes of continuity, it is sometimes necessary to continue using an earlier version. Design methods which do not rely on case histories or past experience, do not have the same constraints.

d) Mining applications of the Q and RMR system have tended to simplify classification systems to only included factors dependent on the rock mass, ignoring environmental and loading conditions. This has resulted in the Q' and RMR' which ignore factors such as stress and joint orientation. This approach to classification is warranted in complex mining situations. Serious errors can result if these simplified classification systems are applied to the empirical civil tunnel design approaches such as the Q support graph.

Despite their limitations, the reviewed classification systems are still in use as they provide an invaluable reference to past experience.

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