# Critical review of RMR and Q-system of rockmass classification for the design of underground openings

Karanam U M Rao[1], Choon SUNWOO[2], So-keul CHUNG[2] and Sung O.CHOI[2]
[1]Post-doctoral fellow, KIGAM, [2]Geotechnical Division, KIGAM

**Abstract** : In this article a comprehensive review of the Rock Mass Rating and Q-rockmass classification systems is made with reference to their scope with in the constraints of underground mining operations. The modifications suggested by KIGAM for both the RMR and Q for the calculation of a safe unsupported span were tested for Daesung and Pyunghae underground limestone mines. Even though the suggested modifications were site specific, the additional parameters considered in the above classification systems are very significant for a design of stable underground openings, considering any general mining conditions.
**Key words** : Rock Mass Rating (RMR), Q, underground openings

## 1. Introduction

Ever since the time when engineering structures are made in rocks, the rockmass classification has been in existence in some form or the other. Fundamentally rockmass classification is a process of combining certain features of a rock into classes or groups for an easy identification of its nature and behavior in a very broad sense. The diagnostics follow a system and order with information being recorded in a prescribed manner.

Rock by nature is a heterogeneous, anisotropic and inelastic material and it exists in a very wide range with many geological structures built in its greater volume. Thus a rockmass constitutes an assemblage of blocks of rock material separated by various types of geological discontinuities, such as joints, faults, bedding planes. It is this rockmass that makes up a material in which the mining and civil engineering structures are designed and constructed. Therefore the qualitative description of the petrographic details and the quantitative description of the mechanical properties of rockmass are essential inputs for an effective design of engineering structures in rockmass. In the ISRM [1] report it is stated that since many engineering decisions are based on a combination of geologic and rock mechanics data, it is important that a more systematic means of combining and correlating this information be developed. Further, the emphasis was laid on the need for a better documentation and correlation of geological and petrographic data, and corresponding mechanical property data obtained from laboratory specimens and/ or rock masses, together with operating experience.

Rockmass classification constitutes an integral part of empirical mine design. They are traditionally used to group areas of similar geo-mechanical characteristics, to provide guidelines for stability performance and to select appropriate support. In the recent times the scope of rockmass classification is further augmented with the increased applications in conjunction with the numerical and analytical tools.

## 2. Classification as a tool for design and construction of engineering structures in rockmass

The important issue in the design and construction of engineering structures in rockmass is in the selection of significant properties of rockmass. The paradox is that there is no single parameter or index, which can assign the properties of a rockmass. Bieniawski [2] inferred that each property has its' own significance and only if combined can they describe a rockmass satisfactorily. The engineering properties of a rockmass depend more on the system of geological discontinuities within the rockmass than the strength of the intact rock itself [3].Terzaghi [4], Piteau [5] and Tsoutrelis et al [6] have independently concluded that the frequency of discontinuities has more importance than the type of rock material. However, Palmström [7] expressed that although rock properties in many cases are outweighed by discontinuities, it should be brought to mind that the properties of the rocks highly determine the formation and development of discontinuities. Therefore, the complete and reliable estimation of rock properties is also of equal importance.

Kirkaldie [8] mentions a total of 28 parameters present in rock masses which may influence the strength, deformability, permeability or stability behavior of rock masses. It is difficult to correlate so many variables and therefore the need for the development of a classification system became essential in this field of engineering. Many classification systems have evolved as engineers have attempted to apply their experience of rock mass behavior to a wider range of engineering problems (Table 1). The different classification systems place different emphasis on various engineering geological parameters (Table 2).

Terzaghi's classification takes the tunnel dimensions into consideration for calculating rock load to design steel-arch support system for tunnels and is too general to permit an objective evaluation of rock quality and does not provide quantitative information on the properties of rock masses [9]. Lauffer's [10] stand-up time, which is that length of time the tunnel can stand without any support, and Deere's Rock Quality Designation (RQD) [10] have indeed influenced many modern classification systems. Although the RQD is a simple and inexpensive index, alone it is not sufficient to provide an adequate description of a rockmass.

Table 1. Major Engineering Rock Mass Classifications Currently in Use [10]

| Name of Classification | Originator and Date | Country of Origin | Applications |
|---|---|---|---|
| Rock Load | Terzaghi,1946 | USA | Tunnels with steel support |
| Stand-up time | Lauffer,1958 | Austria | Tunneling |
| NATM | Pacher et al., 1964 | Austria | Tunneling |
| Rock Quality Designation-RQD | Deere et al., 1967 | USA | Tunneling |
| RSR concept | Wickham et al., 1972 | USA | Tunneling |
| RMR | Bieniawski, 1973 (last modified, 1979 USA) | South Africa | Tunnels, mines, Slopes foundations |
| Q-system | Barton et al., 1974 | Norway | Tunnels and Wide openings |
| Basic geotechnical description (BGD) | International Society for Rock Mechanics , 1981 | | General communication |
| Geological Strength Index-GSI | Hoek E-1994 | Canada | Estimation of rockmass strength properties |
| Rock Mass index (RMi) | Palmström, 1995 | Norway | Tunnels , mining openings and other openings in rock mass |

Table 2. Parameters considered in various classification systems (after Palmström, [7] )

| Classification systems | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9** |
|---|---|---|---|---|---|---|---|---|---|
| Rock | | | | | | | | | |
| -origin , name , type | | | | | x | | | x | |
| -weathering | 0 | | * | | | | | | |
| -anisotropy | | | | | | | | | |
| Rock Properties | | | | | | | | | |
| -Unit weight | | | | | | | | | |
| -porosity | | | | | | | | | |
| -rock hardness | | | | | x | | | | |
| -strength | * | * | * | | | x | | x | x |
| -deformation | | | | | | | | | x |
| -swelling | 0 | | * | | | | | | |
| Joint conditions | | | | | | | | | |
| -joint size / length | | | | | 0 | | | | x |
| -joint separation | | | | | | | | | x |
| -joint wall smoothness | | | | | | x | x | * | x |
| -joint waviness | | | | | | x | x | | x |
| -joint filling | | | | | | | x | | x |
| Degree of jointing | | | | | | | | | |
| -Block size | | | | x | x | | | | x |
| -joint spacing/frequency | 0 | * | * | | | x | | x | x |
| -RQD | | | | | | x | x | | |
| -Number of joint sets | | | | | | | x | | |
| Jointing Geometry or structure | | | | | | | | | |
| -joint orientation with respect to excavation | | | | | x | | | | x<br>x |
| -jointing pattern | | | | | | + | | | x |
| -continuity | | * | * | 0 | | | | | x |
| -structure(fold, fault) | * | | * | | + | | | | x |
| External Features | | | | | | | | | |
| -Water condition | | | | | | x | x | | |
| -Rock stress condition | 0 | | * | | | + | x | | * |
| -Blasting damage | | | | | | + | | | |
| -Excavation dimensions | | | * | | | x | x | | |

Classification system number:
1. Terzaghi (1946); 2.Lauffer (1958); 3. NATM (1957-64) 4. Deere (1964); 5.Wickham (1972) 6. Bieniawski (1973); 7.Barton *et al* (1974); 8. BGD-ISRM (1981); 9. GSI (1994)

Legend:
x -well defined ; 0 -very roughly defined * -included but not defined
\+ -used as an additional information(in RMR as adjusted value)

** added to the table given by Palmström A [7].

## 2.1 The Scope of RMR system

The Geomechanics Classification system or the Rock Mass Rating (RMR) system is one of the widely adopted classification systems for mining operations in the present times. In RMR classification five parameters and the rating of each of these parameters aresummarized to give a value of RMR. The rating is an outcome of a supervised classification of each parameter. The calculated RMR value may be used to find which of five pre-defined rockmass classes the rock mass belongs to, from very good rock to very poor rock. All the parameters are measurable in the field and some of them may also be obtained from borehole data. Bieniawski [10] published a set of guidelines for estimating the stand-up time, and for selecting rock support in tunnels, based on the RMR value.

Laubscher [11] modified RMR classification for mining applications involving asbestos mines in southern Africa. The modifications featured a series of adjustments for RMR values to accommodate the effects of the original (in situ) and induced stresses, changes in stress, the effects of blasting and weathering. Cummings et al., [12] and Kendorski et al., [13] also modified RMR for mining applications in U.S. block caving copper mines and it has been identified as MBR ( modified Basic RMR) system. This MBR value is used in the same fashion as RMR for determining support requirements. Venkateshwarulu [10, 14] modified RMR [10] for estimating roof conditions and support in Indian coal mines. The modification was called CMRI Geo-mechanics classification. However, RMR cannot be used as the only indicator, especially when rock stresses or time dependent rock properties are of importance for the rock engineering issue. In a modification to RMR suggested by the rock mechanics division of Korean Institute of Geo-sciences and Mineral Resources (KIGAM), the effect of time dependent factor was considered for calculating the dimensions of the safe unsupported span (Eq.1) [15].

$$W = 0.24 \quad Total.\ RMR - 0.0013\ T_p + 1.0 \tag{1}$$

where,

$W$ = the width of the unsupported span (m)

$T_p$ = total stand-up time of the opening (in days).

The Eq.(1) was developed by KIGAM based on the joint survey data of Daesung and Pyunghae underground limestone mines. The calculation of safe unsupported span using Eq. (1) involves the first term (0.24 Total RMR), which takes care of all the mining factors effecting the stability and the second term (0.0013Tp), takes care of the negative effect of unsupported time of the excavation and a constant (1.0) which is a minimum unsupported span independent of RMR and time effect.

The RMR value has also been used to estimate rockmass property, Bieniawski [2,10] and Serafim and Pereira [16] have given a relationship between the RMR value and the rockmass deformation modulus. Kalamaras and Bieniawski [17] suggested a relationship linking the compressive strength of a rock mass and intact rock and a very similar expression was proposed by Ramamurthy [18].

## 2.2 The Scope of Q system

The Q-system of rock mass classification system was developed in Norway in 1974 by Barton, Lien and Lunde [19]. The Q-system is based on a numerical assessment of the rock mass quality using six different parameters. The six parameters are grouped into three quotients to give the overall rock mass quality Q.

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF} \qquad (2)$$

The rock quality can range from $Q$ = 0.001 to $Q$ = 1000 on a logarithmic rock mass quality scale. The ratio $RQD/J_n$ represents the intact rock mass block size with a maximum value of 200, $J_r/J_a$ represents the relative frictional strength and the third quotient, $J_w/SRF$, represents the active stress situation. This third quotient is the most complicated empirical factor and has been debated in several papers. It indeed represents four groups of rock masses: stress influence in brittle blocky and massive ground, stress influence in deformable (ductile) rock masses, and stress influence in weakness zones and swelling rock.

Barton [20] suggests modifications to Q values by considering the influence of uniaxial compressive strength of the intact rock ($\sigma_{ci}$) in the following form:

$$Q_c = Q\sigma_{ci}/100 \qquad (3)$$

and recommended $Q_c$ values for estimating the compressive strength and modulus of elasticity of rock mass.

The $Q$-system is normally used as an empirical design method for rock support. Together with the ratio between the span or height of the opening and an excavation support ration ($ESR$), the $Q$ values define the rock support. The original $Q$-based empirical equation for underground excavation support pressure [20], when converted from the original units of kg/cm$^2$ to MPa, is expressed as:

$$P_r = \frac{J_r}{(20 \times Q^{1/3})} \qquad (4)$$

Barton's [20] classification has further been modified to account for the tunnel deformation (i.e. convergence) and *Q*-value data and a collection of *Q/SPAN* versus deformation data was published, having both axes as log scale [20]. The supporting data for the plot of log *Q/SPAN* and *log convergence* was obtained by Chen et al [21] from tunneling projects in Taiwan. A generalized expression suggested by Barton is:

$$\Delta = \frac{SPAN}{Q} \tag{5}$$

While for vertical ($\Delta_v$) and horizontal ($\Delta_h$) deformations are as follows:

$$\Delta_v = \frac{SPAN}{100Q}\sqrt{\frac{\sigma_v}{\sigma_c}} \tag{6}$$

$$\Delta_h = \frac{HEIGHT}{100Q}\sqrt{\frac{\sigma_h}{\sigma_c}} \tag{7}$$

and the approximate value for $\sigma_h/\sigma_v = k_o$ is given as:

$$k_o = \left(\frac{SPAN}{HEIGHT}\right)^2\left(\frac{\Delta_h}{\Delta_v}\right)^2 \tag{8}$$

Units in the above Eq. (4)-(6) are as follows: *SPAN, HEIGHT*, $\Delta_v, and\ \Delta_h$ is each in millimeters, while rock stresses and rock strength are in MPa.

## 2.3 Modifications to rockmass quality Q-system based on width-height ratio of opening

The instability of underground mines is affected by many factors and of which some of the important factors are the height of the mined-out area, width of unsupported mine roof, the depth of the mine from surface, strength of the rock mass, pillar dimensions, hydrological conditions of the mine along with the frequency and condition of joints and lastly the life time of the mine. The modifications to the rock mass quality *Q* are suggested by KIGAM [22] duly considering the influence of width-height factor on stress and strength conditions of rockmass surrounding underground openings, the joint orientation and the hydrological condition of the mine.

The stability number *N'* suggested by Potvin [23] which is basically a modified *Q* system, includes the following parameters:

$N' = Q' \times A \times B \times C$ ( $A = \frac{\sigma_{ci}}{\sigma_\theta}$ , $B$=joint orientation, and $C$=orientation of the opening) (9)

$$N' = \left[\frac{RQD}{J_n} \times \frac{J_r}{J_a}\right] \times \frac{\sigma_{ci}}{\sigma_\theta} \times \text{Joint } orientation \times Orientation\ of\ the\ opening$$

If $Q''$ represents the modified rockmass quality index, then

$$Q'' = \left[\frac{RQD}{J_n} \times \frac{J_r}{J_a}\right] \times \frac{J_{ort} \times \sigma_{ci}}{\sigma_\theta \times (w/h)} \quad (10)$$

The above Eq. (10), in fact includes stress reduction factor ($SRF$) value of the original Barton's classification system which is modified to suit the mining conditions and is given as follows:

$$SRF = \left[\frac{SPAN}{HEIGHT}\right] \times \left[\frac{\sigma_\theta}{\sigma_{ci}}\right] = \frac{\sigma_\theta \times (w/h)}{\sigma_{ci}} \quad (11)$$

$\sigma_{ci} = uni-axial\ compressive\ strength\ of\ a\ rock\ sample$

$\sigma_\theta$ = *the tangential stresses on the opening boundary.*

The values of uni-axial compressive strength ($\sigma_{ci}$) are calculated from the standard laboratory tests while the magnitudes of tangential stresses ($\sigma_\theta$) is computed from the Krish solutions [24] for stress analysis. Similarly, the factor $J_{ort.}$ in the Eq. (10) is equivalent to factor $B$ of Potvin's classification; however the values are further classified into a range from avery favorable condition to a very unfavorable condition in five stages based on dry to saturated condition of the joints. The values are given in Table 3.

Table 3 Ratings for the joint orientation ($J_{ort.}$) in terms of wetness condition

| Orientation of the Joint | $J_{ort.}$ Rating (For dry condition) | $J_{ort.}$ Rating (For wet condition) | $J_{ort.}$ Rating (For fully water saturated condition) |
|---|---|---|---|
| Very Favourable | 1 | 0.95 | 0.80 |
| Favorable | 0.95 | 0.85 | 0.75 |
| Fair | 0.85 | 0.80 | 0.60 |
| Unfavorable | 0.80 | 0.75 | 0.50 |
| Very unfavorable | 0.75 | 0.50 | 0.25 |

Table 4 Safe span values based on RMR, Q, and Q"classification system

| Site | Basic RMR | Q | Q" | Measured Width(m) | Safe unsupported span | | | Remarks |
|---|---|---|---|---|---|---|---|---|
| | | | | | W(RMR) | W(Q) | W(Q") | |
| A-1 | 56 | 4 | 6.1 | 15.5 | 14 | 12.79 | 14.86 | Marginally exceeded |
| A-2 | 47 | 3.9 | 2.9 | 13.3 | 8.5 | 9.48 | 10.34 | ,, |
| A-3 | 47 | 1.5 | 2.9 | 11.1 | 10.8 | 11.67 | 13.80 | With in safe limits |
| A-4 | 56 | 6.1 | 4.6 | 10.8 | 11.6 | 14.27 | 14.44 | ,, |
| A-5 | 45 | 3.9 | 4.6 | 9.7 | 9.4 | 12.6 | 14.45 | ,, |
| Main | 63 | 8.8 | 13.3 | 6.1 | 13 | 15.75 | 16.06 | ,, |
| B-1 | 61 | 5.5 | 8.8 | 12.5 | 14.3 | 13.88 | 15.4 | ,, |
| B-2 | 60 | 10.6 | 11.3 | 17.7 | 10.8 | 16.5 | 15.8 | Marginally exceeded |
| B-3 | 61 | 4.7 | 3.8 | 14.3 | 10.9 | 13.3 | 14.16 | ,, |
| B-4 | 59 | 8.3 | 6.6 | 17.8 | 9.7 | 5.3 | 11.23 | Largely exceeded |
| B-5 | 60 | 4.8 | - | 20.1 | 10.4 | 10.03 | - | ,, |
| B-6 | 47 | 2.5 | 5.0 | 9.6 | 10.7 | 14.02 | 18.20 | With in safe limits |
| B-7 | 62 | 8.3 | 12.5 | 20.7 | 15.2 | 6.1 | 6.40 | Largely exceeded |
| C-1 | 42 | 0.5 | 1.0 | 20.0 | 5.4 | 15 | 4.96 | Largely exceeded |
| C-2 | 59 | 6.6 | 5.6 | 16.0 | 10.6 | 5.8 | 5.85 | Largely exceeded |
| C-3 | 49 | 3 | 3.9 | 9.0 | 9.5 | 11.78 | 14.21 | With in safe limits |
| C-4 | 56 | 8.7 | 9.3 | 12.0 | 12.0 | 19.64 | 19.36 | With in safe limits |
| D-1 | 60 | 4.1 | 3.5 | 12.0 | 11.4 | 12.82 | 14.07 | With in safe limits |
| D-2 | 55 | 3.3 | 4.3 | 7.6 | 12.4 | 15.12 | 17.91 | With in safe limits |
| D-3 | 59 | 12.4 | 17.0 | 13.2 | 13.4 | 17.28 | 16.50 | ,, |
| D-4 | 61 | 4.1 | 6.3 | 17.1 | 15.2 | 15.36 | 18.63 | |
| 60-1-1 | 69 | 5.3 | 4.5 | 11.5 | 13.6 | 13.74 | 14.41 | |
| 60-1-2 | 73 | 42.5 | 42.5 | 25.0 | 15.6 | 30.13 | 22.55 | |
| 60-2 | 62 | 4.0 | 6.0 | 27.0 | 12.8 | 19.82 | 18.54 | Largely exceeded |
| 60-3 | 52 | 0.7 | 2.7 | 10.0 | 10.4 | 14.69 | 13.69 | |
| 60-4-1 | 41 | 1.7 | 2.5 | 9.0 | 8.1 | 4.49 | 5.4 | Marginally exceeded |
| 60-4-2 | 56 | 6.6 | 10.0 | 9.0 | 11.7 | 6.5 | 6.2 | |
| 60-5 | 55 | 7.0 | 10.6 | 9.5 | 11.9 | 6.62 | 6.3 | |
| 60-6 | 65 | 11.2 | 17.0 | 25.0 | 15 | 23.5 | 20.6 | Marginally exceeded |
| 60-7-1 | 42 | 5.0 | 7.5 | 11.0 | 9.4 | 6.04 | 6.06 | ,, |
| 60-8 | 45 | 1.9 | 2.8 | 14.0 | 10.8 | 14.5 | 17.18 | With in safe limits |
| 60-9 | 55 | 2.8 | 4.3 | 11.5 | 13.4 | 12.95 | 14.3 | ,, |
| 60-10 | 61 | 9.4 | 14.2 | 12.0 | 11.9 | 22.4 | 20.2 | ,, |
| 80-2 | 55 | 4.7 | 3.5 | 10.0 | 7.9 | 14.8 | 14.07 | ,, |
| 80-3-1 | 56 | 3.3 | 2.5 | 9.0 | 8.8 | 16.91 | 16.98 | ,, |
| 80-3-2 | 52 | 3.1 | 4.7 | 9.0 | 10.8 | 13.3 | 14.47 | ,, |
| 80-4-1 | 49 | 2.3 | 3.4 | 8.5 | 10.2 | 15.23 | 17.54 | ,, |
| 80-4-2 | 38 | 1.4 | 2.2 | 10.0 | 7.6 | 10.84 | 13.41 | ,, |
| 80-5 | 60 | 6.6 | 10.0 | 24.0 | 14.6 | 20.4 | 19.51 | Largely exceeded |
| 80-6 | 50 | 2.3 | 1.8 | 20.0 | 7.8 | 12.3 | 13.11 | ,, |
| 80-7-1 | 52 | 3.6 | 5.4 | 27.0 | 13 | 13.81 | 14.68 | Largely exceeded |
| 80-7-2 | 62 | 53.1 | 53.1 | 27.0 | 15.4 | 17.17 | 23.06 | Largely exceeded |
| 80-8 | 50 | 3.3 | | 12.5 | 12.3 | 13.52 | 14.56 | With in safe limits |

## 3. Conclusions based on a case study

Using the Eq. (1), (2) and (10), the safe limits of the underground openings at Daesung and Pyunghae mine were calculated and are given in Table 4.

The safe span calculations by *RMR*, *Q*, and *Q″* for majority of the cases are with in comparable limits. The differences are mainly due to the selection of *ESR* value in *Q* system and in the case of RMR calculations, the negative influence of stand-up time Eq.(1) is found to have a significant influence on the values of safe spans.

In Daesung mine the measured width in 475 ML at location A1 is exceeding by 1.5, 2.7, and 0.64m based on *RMR*, *Q* and *Q″* systems respectively. The span has been standing without support for 274 days. Similarly the only other location in 475ML at which the safe limits are exceeded is A2, where it is exceeded by 4.8m (RMR) and 3.8m (Q) and 2.96m (Q″) the stand up time of the span has been for 639 days. Unlike a tunnel opening not all locations have equal importance in mining operations. Since the area under the influence at location A1 belongs to a level drive of certain importance compared to the temporary opening at A-2, it is suggested that sufficient roof bolting may be planned around A1 location to secure stability of the opening before any failure occurs. Since rock bolts are effective when the roof is still stable, it is essential to drive rock bolts at the earliest time. In case of the locations in 450 ML, at points B4, B5 and B7 the width is exceeding on an average by 8m from RMR calculations and 12m by Q and 6m by modified Q″system and the openings have been standing unsupported as long as 2010 days to 1600 days. The points belong to a room-and-pillar stope where the stope has been widened following the high grades values. Since it is a worked out stope, though the safe limits are far exceeded, it is suggested to fence the old workings as abandoned areas. The other important observation is that the stand up time of the drive in 425 ML has been between 1888 to 2161 days. The safe span limits therefore are exceeded by 14 m by RMR. It is further noticed that a RMR and Q″span dimensions are very close while a large difference exists for the safe span limits by Q, of the order of 10m.

The recommendation from the present study is to reinforce the roof strata by systematic rock bolting. The safe span limits in 405 ML are found to be with in reasonable limits.

Pyunghae mine is relatively an older mine. The main production is concentrated within the drives at 80 and 60 ML. The level drives have been widened following room-and-pillar operations at selective places on the basis of the grade of limestone. In the level drive at 60ML, the computed span widths by RMR, Q, and Q″at location 60-1-2 are 12.8, 19.8, and 18.54m respectively with the measured width exceeding by 14.2, 7.2, and 8.46m for RMR, Q and Q″ respectively. Similarly in the level drive 80 ML, the safe span limits are exceeded at 80-5, 80-6, 80-7-1 and 80-7-2. Since all these points belong to different mined out room-and-pillar stopes and hence no active support is recommended however for the safety purposes the old mine out may be fenced to restrict the accessibility.

## References


1. International Society for Rock Mechanics (1971), Commission on standardization of laboratory and field tests, Int. Soc. Rock Mech.(ISRM), Lisbon.
2. Bieniawski Z.T. (1984), Rock mechanics design in Mining and Tunneling, A.A. Balkema, Rotterdam, 272p.
3. Lama R.D. and Vutukuri V.S. (1978), Handbook on mechanical properties of rock, Trans Tech Publications, Clausthal, Germany, 1650p.
4. Terzaghi K. (1946), Introduction to tunnel geology, In rock tunneling with steel supports, Publishers, Proctor and white, pp. 5-153.
5. Piteau D.R. (1970), Geological factors significant to the stability of sloped cut in rock, Proc. Symp. on planning Open pit Mines, Johannesburg, South Africa, pp-33-53.
6. Tsoutrelis C.E., Exadactylos G.E. and Kapenis A.P. (1990), Study of the rock mass discontinuity system using photo analysis, Proc. Of Symp. on Mechanicsof Jointed and faulted rock, pp. 103-112.
7. Palmström A. (1995), a rock mass characterization system for rock engineering purposes, PhD thesis, Oslo Univ., Norway.
8. Kirkaldie L (1988) Rock classification systems for engineering purposes, STP 984, ASTM, 167pp.
9. Cecil O.S. (1970), Correlations of rock bolt-shotcrete support and rock quality parameters in Scandinavian tunnels. PhD Thesis Univ. of Illinois.
10. Bieniawski Z.T. (1989), Engineering rock mass classification, John Wiley & sons , New York, pp.251
11. Laubscher D.H. (1986), Geomechanics classification of jointed rock masses-mining applications, Trans. Inst. Min. Metall.pp.A1-A7.
12. Cummings R.A., Kendorski F.S., and Bieniawski Z.T. (1982), Caving rock mass classification and support estimation, USBM-RI:J0100103.
13. Kendorski F.S., Cummings R.A., and Bieniawski Z.T. (1983), A rock mass classification scheme for the planning of caving mine drift supports., Proc. Rapid Excavation Tunneling conf., AIME, New York., pp.192-223.
14. Venkateshwarulu V. (1986) Geomechanics classification of coal measure rocks vis--vis roof support, PhD, thesis Indian School of Mines, Dhanbad, 251p.
15. Karanam U.M.Rao, Choon SUNWOO, So-keul CHUNG, Sung O.CHOI and Yang

Soo Jeon, (2003), A suggestion of rockmass classification system for stability of underground limestone mines- a case study, J. of Korean Soc. Rock. Mech, Tunnel and Underground Space, Vol.13(6), pp.421-433
16. Serafim J.L and Pereira J.P. (1983), Consideration of the Geomechanics classification of Bieniawski, Proc. Int. Symp. on Engineering Geology and Underground construction, pp.113-1144.
17. Kalamaras G.S. and Bieniawski Z.T. A rock strength concept for coal seams incorporating the effect of time. Proc. Of the 8th Int. Congress on rock Mech. Vol.1, 1995. Pp.295-302.
18. Ramamurthy T. (2004), A Geo-engineering classification for rocks and rock masses, IJRM 41, 89-101.
19. Barton N., Lien R. and Lunde J.(1974), Engineering classification of rock masses for the design of rock support, Rock Mechanics, Vol.6, 189-236
20. Barton N. (2002), Some new Q-value correlations to assist in site characterization and tunnel design, IJRM, Vol.39, 185-216.
21. Chen C.N, Guo G.C.(1997), Rockmass classification and guidelines for tunnel convergence, J. China Inst. Civil Hydraulic Eng., Taiwan, Vol.9(3), pp.359-367.
22. SUNWOO C. et al. (2003), A study on the safety and environmental control of underground opening in limestone quarries, KIGAM Report, KR-03(c)-17, 121p.
23. Potvin Y (1988) Empirical open stope design in Canada, PhD thesis, The Univ. of British Columbia, 350p.
24. Hoek E and Brown E.T (1980), Underground excavations in rock, Institution of Mining and Metallurgy, London, 527p.