



भारत सरकार / Government of India
श्रम एवं रोजगार मंत्रालय / Ministry of Labour & Employment
खान सुरक्षा महानिदेशालय / Directorate General of Mines Safety



DGMS (Tech.)/S&T/ Circular No.11 of 2026

Dhanbad, dated 28 / 08 / 2026

To

All the Owners, Agents and Managers of Metalliferous Mines

Subject: Guidelines for use of Rock Mass Classifications and Support/Slope Design for Improving Stability of Workings in Metalliferous Mines.

Sir,

Stability of workings is essential for economic and safe exploitation of mineral from mines. The geotechnical assessment, safe support/slope design and monitoring of stability of workings is one of the major challenges in Indian Metalliferous Mines. The stability analysis of mine workings, support/slope design and their monitoring is not yet an integral part of mine planning. In Indian mining conditions, support/slope design guidelines are yet to be formulated for different types of mining practices for maintaining safety and productivity.

However, there is a lot of diversity amongst different mines in understanding and practicing for Rock Mass Classification systems and Support/Slope Designs. Therefore, it was felt necessary to evolve a broad standardization/ guideline for use of Rock Mass Classifications and Support/Slope Design for Improving Stability of Workings in Metalliferous Mines.

In view of the above, a workshop on the subject "Rock Mass Classifications and Support Design for Improving Stability of Workings in Metal Mines" was jointly organised by DGMS and IIT Kharagpur on 28th November 2025 in Jaipur, Rajasthan to deliberate and review the present status of use of different types of Rock Mass Classification systems and Support/Slope Designs. In this workshop standard guidelines are finalized for use of Rock Mass Classifications and Support/Slope Design methodologies for Improving Stability of Workings in Metalliferous Mines. This workshop was attended by about 80 officials and experts from regulatory authorities, academic institutions, research organizations, and various mining industries like DGMS, IITs, NITs, NIRM, CIMFR, M/s HZL, M/s HCL, M/s MOIL, M/s UCIL, M/s IMFA, M/s OMC, M/s Birla Corporation Ltd., M/S Jindal Saw Ltd., M/s JK Cement, M/s JSW Cement, M/s Wonder Cement Ltd., M/s Ultratech Cement, M/s Shree Cement Ltd., M/s Mangalam Cement Ltd., M/s Nuvoco

Vistas Corporation Limited, M/s RK Marble Limited, M/s Shree Cement Limited, M/s UMDS Pvt. Ltd., etc., in which 10 technical presentations were made by different experts giving valuable inputs on the subject matter.

Based on various inputs drawn from the detailed deliberations and discussions made in the workshop, the following guidelines are being made for use of Rock Mass Classifications and Support/Slope Design for Improving Stability of Workings in Metalliferous Mines.

1. **GSI and Q** values shall be calculated by practicing geologists and mining engineers at multiple locations and levels based on the complexity of the geology and geotechnical domain within underground mines. Structured training shall be provided to these personnel for the correct application of these systems.
2. It is time for the Indian metal mining sector to adopt Q and GSI as standard systems for rock mass classification and support design. RMR shall gradually be replaced by Q or GSI, particularly in underground mines and shall be incorporated in technical and scientific study reports. A step-by-step support design procedure using the Q system is provided in **Appendix A**, and the application of GSI in bolt design and numerical modelling is outlined in **Appendix B**.
3. The NGI support design chart shall be used for determining support requirements in drives, drifts, cross-cuts, and other underground excavations. Installed support systems may initially be benchmarked against NGI recommendations, and if any discrepancies found, it shall be scientifically investigated and rectified.
4. In slope stability analysis, overall slope stability shall be assessed using rock mass properties-such as cohesion and friction angle-adjusted through GSI values measured at different slope locations depending on the complexity of the geology and geotechnical domain. Q-slope method or Slope Mass Rating (SMR) shall be used to estimate rock mass quality and bench slope angles to validate the slope stability analysis. A step-by-step methodology is provided in **Appendix C** for Q-slope and in **Appendix D** for SMR.
5. Scientific organisations, academic institutions, and mining companies shall adopt Q- or GSI-based approaches for adjusting rock mass properties to replicate field conditions in numerical models for bringing uniformity in stability assessment and generate data to develop support/slope design charts for site specific conditions.
6. Periodical/ real-time support performance monitoring like load-displacement, pillar stresses and roof displacements is crucial for understanding rock mass behaviour and validating numerical model outcomes. It shall be carried out at various locations near the active stopes, junctions, geologically disturbed areas and a Trigger Action Response Plan (TARP) shall be developed based on numerical and monitoring data.
7. Slope bench movement shall be monitored regularly or in real time to generate heat maps of displacements and validate the slope stability analysis. Zones showing major displacement ("hot zones") shall be identified and TARP shall be developed based on numerical and monitoring data.

8. Mine management shall establish a dedicated Geotechnical Cell, comprising trained geologists and mining engineers with expertise in rock mechanics, ground control, and support/slope design. This team shall be regularly kept updated in the application of GSI, Q, Q-slope and SMR methods, stability analysis of stopes/slopes and monitoring to develop robust mine/support / slope design.
9. In the unorganized sector, small mine owners often lack access to advanced technology, skilled manpower, and proper safety systems. They shall seek technical guidance from nearby mines in the organised sector or any other experts in the concerned field.

All the Owners, Agents and Managers are advised to prepare Strata/Ground Control Management Plan keeping in view of the above-mentioned guidelines.



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Directorate General of Mines Safety
Chief Inspector-cum-Facilitator of Mines

APPENDIX-A:

UNDERGROUND SUPPORT DESIGN BASED ON Q

Step 1: Estimate Q values at multiple locations in the area

- a) Estimation of parameters based on the tables given in the references (*NGI Publication*¹, *Hoek et al., 1998*², *Edelbro, 2003*³, *Deb and Verma, 2025*⁴). The parameters are i) *RQD* = Rock Quality Designation, ii) J_n = Joint set number, iii) J_r = Joint roughness number, iv) J_a = Joint alteration number, v) J_w = Joint water reduction factor and vi) *SRF* = Stress Reduction Factor. Refer Appendix E for the description and numerical values of each parameter.
- b) The value of $Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$

Step 2: Adjustment of Span or Height based on Excavation Support Ratio (ESR)

- a) Selection of Excavation Support Ratio (*ESR*) based on excavation category (Table 1).
- b) The Equivalent Dimension $D_e = \frac{\text{Span or height in m}}{ESR}$

Table 1. ESR values for different excavation categories (*NGI publication, 2025*)

8	Type underground facility	ESR
A	Temporary mine openings, etc.	3-5
B	Vertical shafts*: i) circular sections ii) rectangular/square sections * Dependant of purpose. May be lower than given values.	2,5 2,0
C	Permanent mine openings, water tunnels for hydro power (exclude high pressure penstocks), water supply tunnels, pilot tunnels, drifts and headings for large openings.	1,6
D	Minor road and railway tunnels, surge chambers, access tunnels, sewage tunnels, etc.	1,3
E	Power houses, storage rooms, water treatment plants, major road and railway tunnels, civil defence chambers, portals, intersections, etc.	1,0
F	Underground nuclear power stations, railways stations, sports and public facilities, factories, etc.	0,8
G	Very important caverns and underground openings with a long lifetime, ≈ 100 years, or without access for maintenance.	0,5

¹Using The Q-System, Rock Mass Classifications and Support Design, NGI, 2025²Hoek E., P.K. Kaiser and W.E. Bawden, 1998, "Support of Underground Excavations in Hard Rock", A.A. Balkema³Cartrin Edelbro, 2003, Rock Mass Strength: A Review, Lulea University of Technology, Sweden⁴Deb and Verma, 2025, "Fundamental and Applications of Rock Mechanics", PHI Publisher

Step 3: Determination of the length of a bolt (L) (Reference: Barton et.al., 1974⁵)

$$L = \frac{2 + 0.15B}{ESR}$$

Where, B is the width of the excavation in m.

* Select the excavation category and the corresponding Excavation Support Ratio (ESR) based on Table 1.

Step 4: Determination of the maximum unsupported span (S), m (Reference: Barton et.al., 1974)

$$S = 2 \times ESR \times Q^{0.4}$$

Step 5: The maximum permanent roof pressure (P_{roof}) in kg/cm^2 (Reference: Barton et.al., 1974)

$$P_{roof} = \frac{2 \times Q^{-1/3} \times \sqrt{J_n}}{3 \times J_r}$$

Step 6: Estimation of support requirement (NGI publication, 2025)

Based on the estimated Q value and D_e , the support requirement is derived from the chart in Figure 1.

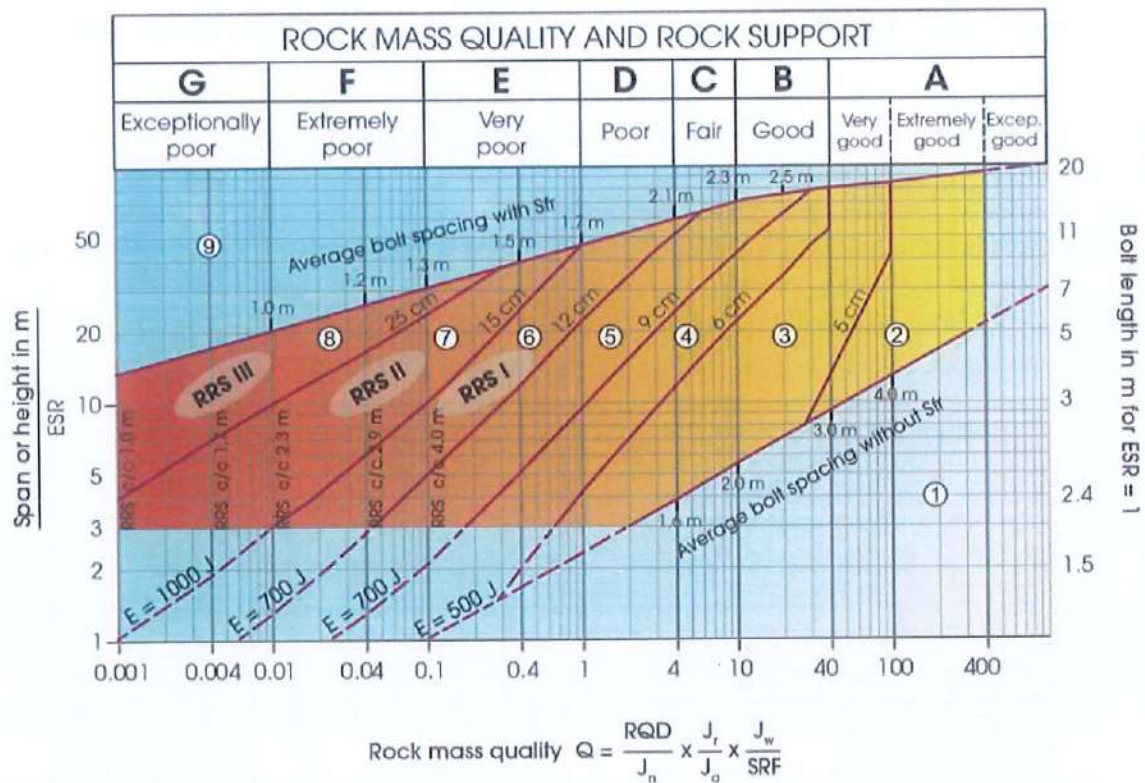
Step 7: Adjustment of Q Value for Wall Support

The support requirements for the wall are not as rigorous as the roof support. The equivalent dimension is calculated based on wall height. Then, the Q value is adjusted based on Table 2 and the Step-2 to Step-6 are followed.

Table 2. Adjustment of Q value for wall support (NGI publication, 2025)

9 Dimensioning of wall support		
In rock masses of good quality	$Q > 10$	Multiply Q -values by a factor of 5
For rock masses of poor-fair quality	$1 < Q < 10$	Multiply Q -values by a factor of 2.5. In cases of high rock stresses, use the actual Q -value
For rock masses of poor quality	$Q < 1$	Use actual Q -value
Wall height > span width	Applies for all Q -values	Use actual Q -value

⁵N. Barton, R. Lien, and J. Lunde, 1974, "Engineering Classification of Rock Masses for the Design of Tunnel Support", Rock Mechanics 6, 189--236 (1974)@ by Springer-Verlag 1974



Support categories

- ① Unsupported or spot bolting
- ② Spot bolting, **SB**
- ③ Systematic bolting, fibre reinforced sprayed concrete, 5-6 cm, **B+Sfr**
- ④ Fibre reinforced sprayed concrete and bolting, 6-9 cm, **Sfr (E500)+B**
- ⑤ Fibre reinforced sprayed concrete and bolting, 9-12 cm, **Sfr (E700)+B**
- ⑥ Fibre reinforced sprayed concrete and bolting, 12-15 cm + reinforced ribs of sprayed concrete and bolting, **Sfr (E700)+RRS I+B**
- ⑦ Fibre reinforced sprayed concrete >15 cm + reinforced ribs of sprayed concrete and bolting, **Sfr (E1000)+RRS II+B**
- ⑧ Cast concrete lining, **CCA** or **Sfr (E1000)+RRS III+B**
- ⑨ Special evaluation

Bolts spacing is mainly based on Ø20 mm

E = Energy absorption in fibre reinforced sprayed concrete

ESR = Excavation Support Ratio

Areas with dashed lines have no empirical data

RRS - spacing related to Q-value

I **SI30/6 Ø16 - Ø20 (span 10m)**
D40/6+2 Ø16-20 (span 20m)

II **SI35/6 Ø16-20 (span 5m)**
D45/6+2 Ø16-20 (span 10m)
D55/6+4 Ø20 (span 20m)

III **D40/6+4 Ø16-20 (span 5 m)**
D55/6+4 Ø20 (span 10 m)
 Special evaluation (span 20 m)

SI30/6 = Single layer of 6 rebars,
 30 cm thickness of sprayed concrete

D = Double layer of rebars

Ø16 = Rebar diameter is 16 mm

c/c = RSS spacing, centre - centre

Figure 1. Support chart for the selection of support requirements based on Q and D_e (NGI publication, 2025)

APPENDIX-B:

APPLICATIONS OF GSI IN BOLT DESIGN AND NUMERICAL ANALYSIS

Step 1: Estimate GSI at Multiple Locations in the Study Area (Reference: V. Marinos P. Marinos E. Hoek, 2005)

- Conduct a detailed field survey of rock faces.
- Estimate the maximum and the minimum GSI at the study area
- GSI is estimated based on the structure of the rock mass and surface conditions.
- The GSI chart, as shown in Figure 1A, is to be used for the determination of GSI.
- If the surface is wet, Figure 1B must be used to modify the GSI value

GEOLOGICAL STRENGTH INDEX FOR JOINTED ROCKS (Hoek and Marinos, 2000) From the lithology, structure and surface conditions of the discontinuities, estimate the average value of GSI. Do not try to be too precise. Quoting a range from 33 to 37 is more realistic than stating that GSI = 35. Note that the table does not apply to structurally controlled failures. Where weak planar structural planes are present in an unfavourable orientation with respect to the excavation face, these will dominate the rock mass behaviour. The shear strength of surfaces in rocks that are prone to deterioration as a result of changes in moisture content will be reduced if water is present. When working with rocks in the fair to very poor categories, a shift to the right may be made for wet conditions. Water pressure is dealt with by effective stress analysis.		SURFACE CONDITIONS				
STRUCTURE		DECREASING SURFACE QUALITY →				
	 INTACT OR MASSIVE - intact rock specimens or massive in situ rock with few widely spaced discontinuities	VERY GOOD Very rough, fresh unweathered surfaces	GOOD Rough, slightly weathered, iron stained surfaces	FAIR Smooth, moderately weathered and altered surfaces	POOR Slackensided, highly weathered surfaces with compact coatings or fillings or angular fragments	VERY POOR Slackensided, highly weathered surfaces with soft clay coatings or fillings
	 BLOCKY - well interlocked undisturbed rock mass consisting of cubical blocks formed by three intersecting discontinuity sets	90	80	70	N/A	N/A
	 VERY BLOCKY - interlocked, partially disturbed mass with multi-faceted angular blocks formed by 4 or more joint sets		60	50		
	 BLOCKY/DISTURBED/SEAMY - folded with angular blocks formed by many intersecting discontinuity sets. Persistence of bedding planes or schistosity			40	30	
	 DISINTEGRATED - poorly interlocked, heavily broken rock mass with mixture of angular and rounded rock pieces				20	
	 LAMINATED/SHEARED - Lack of blockiness due to close spacing of weak schistosity or shear planes	N/A	N/A			10

Figure 1A. Classification of rock mass based on interlocking and joint conditions (V. Marinos P. Marinos E. Hoek, 2005⁶)

⁶V. Marinos P. Marinos E. Hoek, 2005, "The geological strength index: applications and limitations", Bull Eng Geol Environ (2005) 64: 55–65 DOI 10.1007/s10064-004-0270-5

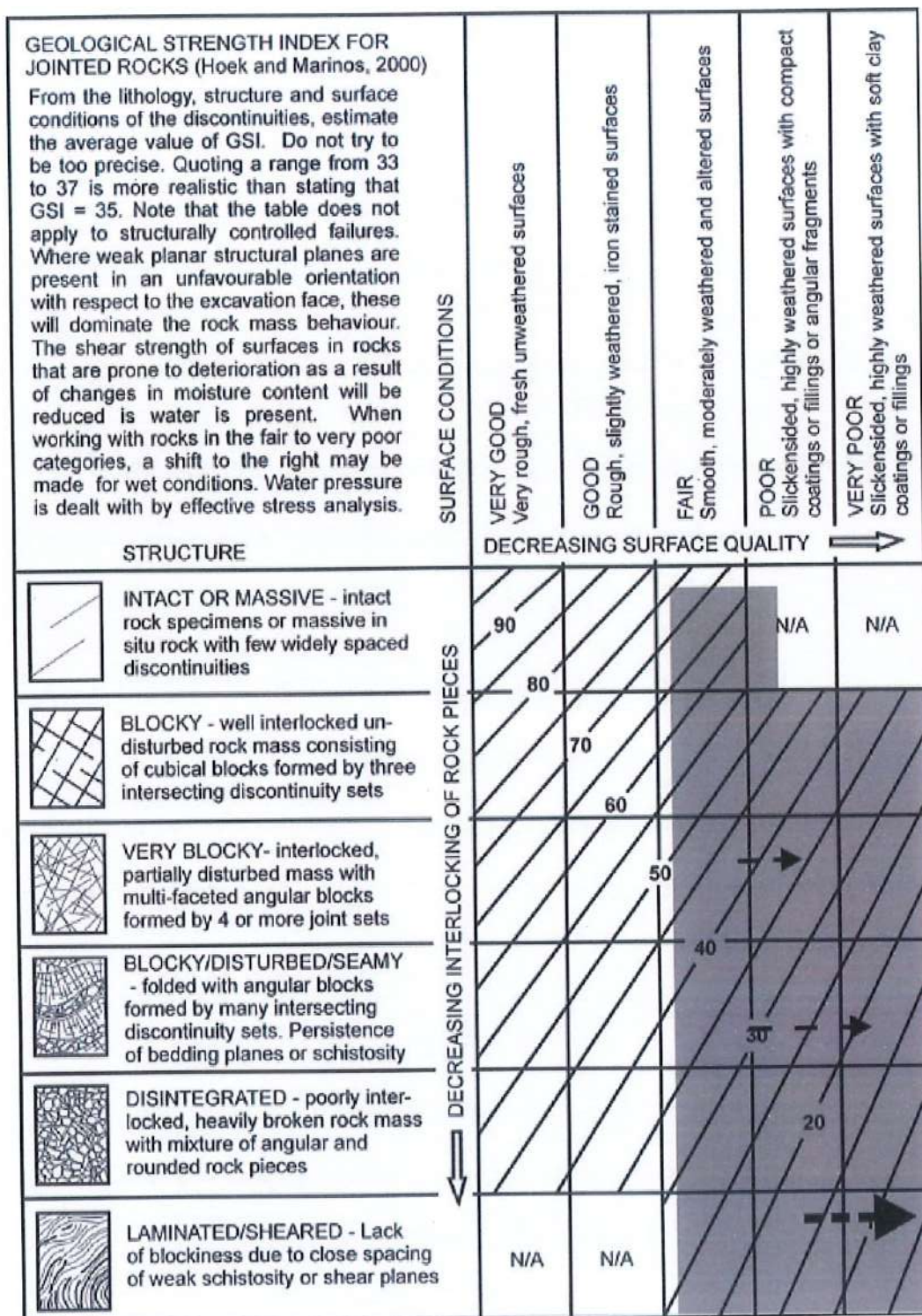


Figure 1B. A shift towards the right due to wet condition (*V. Marinos P. Marinos E. Hoek, 2005*)

Step 2: Estimation of Bolt Load and Spacing

Step 2.1: For determination of the Length of a Bolt (L), refer to step 3 of Appendix A.

Step 2.2: Determination of Rock Load Height (h_l) (Bieniawski, 1989⁷)

Rock load height in m for the excavation width, B is given by

$$h_l = B \times \left(\frac{100 - GSI}{100} \right)$$

Ensure that L is at least $h_l + 0.5 m$ for proper anchorage. (Note: In the above reference, equation contains RMR. Here, it is replaced by GSI).

Step 2.3: Determination of Rock Load (R_L) (Reference: Deb and Verma, 2025)

The number of rock bolts, B as the width of the excavation, and $s \times s$ as the square spacing is

$$N = \frac{B}{s}$$

Rock load per bolt, in kN , is estimated for γ_{rock} , as the unit weight of a rock in kN/m^3 as

$$R_L = s^2 \times h_l \times \gamma_{rock}$$

Step 2.4: Determination of the Allowable Bolt Load (B_L) (Reference: Deb and Verma, 2025)

The allowable bolt load, in kN , for σ_y as the yield strength of the bolt rod in kPa , d as the diameter in m , and SF , as the required safety factor, is estimated as

$$B_L = \frac{f \times \sigma_y \times \frac{\pi d^2}{4}}{SF}$$

Where f is the bolt efficiency factor, typically taken between 0.7 and 0.9.

Step 2.5: Determination of Bolt Spacing (Reference: Deb and Verma, 2025)

From the above two equations, the spacing of the bolt is determined as

$$s = \sqrt{\frac{f \times \sigma_y \times \pi d^2}{4 h_l \times \gamma_{rock} \times SF}}$$

⁷Bieniawski Z.T., 1989, "Engineering Rock Mass Classification: A Complete Manual for Engineers and Geologists in Mining, Civil and Petroleum Engineering, Wiley, New York

Step 3: Application of GSI in Numerical Analysis

Step 3.1: Determine intact rock properties such as bulk density, compressive and tensile strength, Hoek-Brown parameter, m_i , cohesion, friction angle, elastic modulus and Poisson's ratio in the laboratory.

Step 3.2: Adjust the strength, cohesion, friction angle and elastic modulus based on *GSI* value, m_i and blast disturbance factor D , using the procedure outlined in references *Hoek et al., 2002*⁸ and *Deb and Verma, 2025*⁹ (page numbers: 72-73 and 265-266).

Step 3.3: In the numerical analysis, apply the adjusted rock mass properties for determining displacements and stresses.

Step 3.4: Perform the analysis using elasto-plastic material models.

⁸ Hoek E., Torres C.C. and Corkum B, 2002, "Hoek and Brown failure criterion", 5th ed. North American Rock Mechanics Sym. And 17th Tunnelling Association of Canada Conf, 267-271

⁹ Deb and Verma, 2025, "Fundamental and Applications of Rock Mechanics", PHI Publisher



APPENDIX-C: SLOPE STABILITY BASED ON Q-SLOPE AND NUMERICAL MODELLING

Step 1: Estimate Q-Slope values at Multiple Locations in a Bench

- i) For each slope bench, estimate Q_{slope} based on six parameters as given below.
- ii) Estimation of parameters based on the tables given in the references (*Bar and Barton, 2025*¹⁰). The parameters are i) RQD = Rock Quality Designation, ii) J_n = Joint set number, iii) J_r = Joint roughness number, iv) J_a = Joint alteration number, v) J_{wice} = Environmental and geological condition number and vi) SRF_{slope} = Maximum of three strength reduction factors a, b , and c , (SRF_a : Physical condition number, SRF_b : Stress and strength number and SRF_c : Major discontinuity number). Refer **Appendix E** for the description and numerical value of each parameter.
- iii) The value of $Q_{slope} = \frac{RQD}{J_n} \times \left(\frac{J_r}{J_a}\right)_{O-factor} \times \frac{J_{wice}}{SRF_{slope}}$.
- iv) Inter-block shear strength: least favorable $\left(\frac{J_r}{J_a}\right)$ or average shear strength in the case of wedges is to be adjusted by the discontinuity orientation factor (O -factor). Refer to Table 1 for the estimation of the O -factor.

Table 1. Description of O -factor (*Bar and Barton, 2025*)

O-factor description	Set A	Set B (where applicable)
Very favourably oriented	2.0	1.5
Quite favourable	1.0	1.0
Unfavourable	0.75	0.9
Very unfavourable	0.50	0.8
Causing failure if unsupported	0.25	0.5

Step 2: Determination of Bench Slope Angle (*Bar and Barton, 2025*)

Apply the following equation for the estimation of the bench slope angle in degrees

$$\beta = 20 \log_{10} Q_{slope} + 65$$

¹⁰Q-Slope: Rock Slope Engineering 10 Years on, Rock Mechanics and Rock Engineering (2025) 58:11453–11474, <https://doi.org/10.1007/s00603-024-04064-5>



Step 3: Estimation of the safety factor of the overall slope (*Bar and Barton, 2025*)

Develop a numerical model of the overall slope using the estimated bench slope angle and the bench geometries, such as width, height and material properties. Friction angle of slope may also be estimated as follows:

$$\varphi = \tan^{-1} \left(\frac{J_r}{J_a} \right)$$

Determine the safety factor of the overall slope based on the developed numerical model.



APPENDIX-D: SLOPE MASS RATING (SMR)

The 'Slope Mass Rating' (SMR) is obtained from RMR, estimated according to Bieniawski's 1979 RMR chart, and then four additional factors are used for the adjustment^{11, 12}. Three factors depend on the joint-slope relationships, and the fourth factor is added based on the method of excavation. SMR is calculated as

$$SMR = RMR + (F1 \times F2 \times F3) + F4$$

The RMR is computed by adding rating values for five parameters: (i) strength of intact rock (ii) RQD (iii) spacing of discontinuities (iv) condition of discontinuities and (v) water inflow through discontinuities (estimated in the worst possible conditions). RMR has a range between 0 and 100. Then the adjustment is applied as given below.

Step-1: The factor $F1$ depends on parallelism between joints and slope face strikes. Its range is from 1.00 (when both are near parallel) to 0.15 (when the angle between them is more than 30°) (Table 1). Alternatively, $F1$ can be computed as

$$F1 = (1 - \sin A)^2$$

Where, angle A denotes the angle between the strikes of the slope face and the joint.

Table 1. Factors $F1$, $F2$ and $F3$

Case		Very favorable	Favorable	Fair	Unfavorable	Very unfavorable
P	$ \alpha_j - \alpha_s $	$> 30^\circ$	$30-20^\circ$	$20-10^\circ$	$10-5^\circ$	5°
T	$ (\alpha_j - \alpha_s) - 180^\circ $					
P/T	F_1	0.15	0.40	0.70	0.85	1.00
P	$ \beta_j $	$< 20^\circ$	$20-30^\circ$	$30-35^\circ$	$35-45^\circ$	45°
P	F_2	0.15	0.40	0.70	0.85	1.00
T	F_2	1	1	1	1	1
P	$\beta_j - \beta_s$	$> 10^\circ$	$10-0^\circ$	0°	0° to 10°	$< -10^\circ$
T	$\beta_j - \beta_s$	$< 110^\circ$	$110-120^\circ$	$> 120^\circ$	-	-
P/T	F_3	0	-6	-25	-50	-60

P, plane failure; T, toppling failure; α_j , joint dip direction; α_s , slope dip direction; β_j , joint dip; β_s , slope dip

¹¹M. Romana, R. Tomás and J. B. Serón, 1985, "Slope Mass Rating (SMR) Geomechanics Classification: Thirty Years Review"

¹²M. R. Romana, 1993, "A Geomechanical Classification for Slopes: Slope Mass Rating", COMPREHENSIVE ROCK ENGINEERING Principles, Practice & Projects, Chief Editor: J.A. Hudson, Volume 3

Step-2: The factor, F_2 refers to joint dip angle in the planar mode of failure. Its value varies from 1.00 (for joints dipping more than 45°) to 0.15 (for joints dipping less than 20°). Alternatively, F_2 can be approximated as

$$F_2 = \tan^2 \beta_j$$

where β_j denotes the joint dip angle. For the toppling mode of failure F_2 remains 1.00.

Step-3: The factor, F_3 reflects the relationship between the angle of slope face and joint dip. The values of F_3 are shown in Table 1. Conditions are fair when slope face and joints are parallel. When the slope dips 10° more than joints, very unfavourable conditions occur.

Step-4: The adjustment factor for the method of excavation (Factor F_4) has been fixed empirically as shown in Table 2. Details of this factor are available in the literature mentioned above.

Table 2. Factor F_4

Method	Natural Slope	Presplitting	Smooth blasting	Blasting or mechanical	Deficient blasting
F_4	+ 15	+ 10	+ 8	0	- 8

Step-5: Table 3 provides classes and conditions of slope based on SMR value. Figure 1 shows tentative support requirements (Romana, 1985).

Table 3. Description of SMR classes

Classes →	V	IV	III	II	I
SMR	0-20	21-40	41-60	61-80	81-100
Description	Very bad	Bad	Normal	Good	Very good
Stability	Completely unstable	Unstable	Partially stable	Stable	Completely stable
Failures	Big planar or soil-like	Planar or big wedges	Some joints or many wedges	Some blocks	None
Failure probability	0.9	0.6	0.4	0.2	0



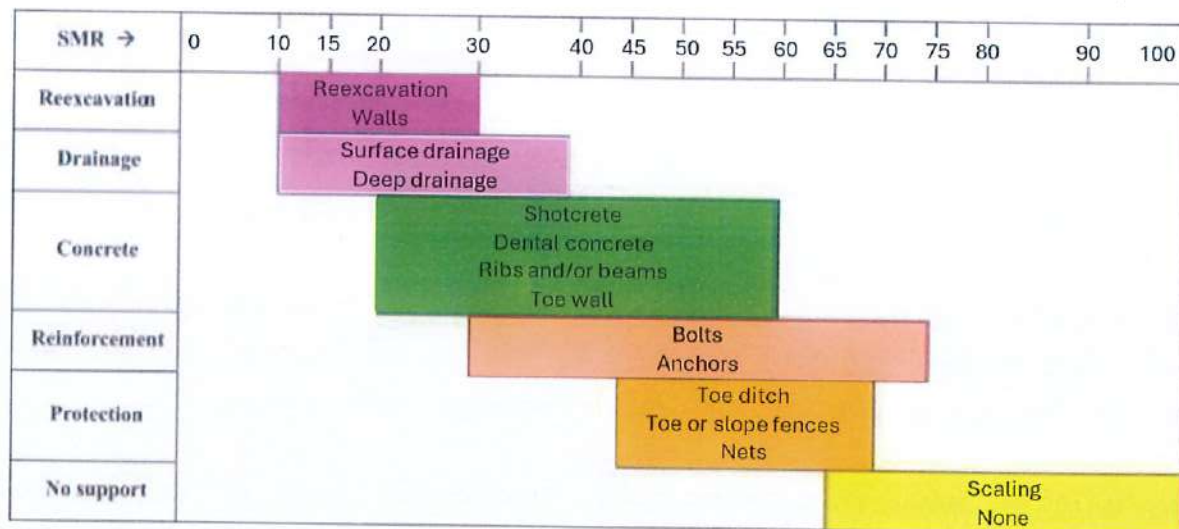


Figure 1. Slope support guidelines based on SMR (Romana, 1985).

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APPENDIX-E

Tables of Parameters for Q and Q-Slope

Table 1: The RQD Classification

RQD (%)	<25	25-50	50-75	75-90	90-100
Rock Quality	Very poor	Poor	Fair	Good	Very good

Table2: The Joint Set Number (J_n)

Class	J_n
A Massive, no or few joints	0.5-1
B One joint set	2
C One joint set + random joints	3
D Two joint set	4
E Two joint set + random joint	6
F Three joint set	9
G Three joint set + random joints	12
H Four or more joint sets	15
J Crushed rock	20

- (i) For tunnel intersections: $3 \times J_n$
- (ii) For portals: $2 \times J_n$

Table 3: The Joint Roughness Number (J_r)

Class	J_r
(a) Rock-wall contact, and (b) rock –wall before 10 cm shear	
A Discontinuous joints	4
B Rough, undulating	3
C Smooth, undulating	2
D Slicken sided, undulating	1.5
E Rough, planar	1.5
F Smooth, planar	1

G	Slicken sided, planar	0.5
	(c) No rock-wall contact when sheared	
H	Zone containing clay minerals thick enough to prevent rock-wall contact	1
J	Sandy, gravely or crushed zone thick enough to prevent rock-wall contact	1

1. Add 1 if the mean spacing of the relevant joint set is greater than 3m.
2. $I_r = 0.5$ can be used for planar, slicken sided joints having lineations, provided that the lineations are oriented for minimum strength.

Table 4: The Joint Alteration Number (J_a)

	Class	ϕ_r	I_r
	(a) Rock-wall contact (no mineral fillings)		
A	Tightly healed, hard, non-softening, impermeable filling		0.75
B	Unaltered joint walls, surface staining only	25-35	1
C	Slightly altered joint walls, non-softening mineral coatings, sandy particles.	25-30	2
D	Silty or sandy clay coating, small clay friction	20-25	3
E	Softening or low friction clay minerals coating	8-16	4
	(b) Rock wall contact 10 cm shear		
F	Sandy particles, clay-free disintegrated rock	25-30	4
G	Strongly over-consolidated non-softening clay mineral fillings (continuous, <5 mm thickness)	16-24	6
H	Medium or low over-consolidation, softening, clay mineral fillings (continuous, <5 mm thickness)	12-16	8
J	Swelling-clay fillings	6-12	8-12
	(c) No rock-wall contact when sheared (thick mineral filling)		
KLM	Zones or bands of disintegrated or crushed rock and clay	6-24	6, 8, 12
N	Zones or bands of silty or sandy clay, small clay fraction (non-softening)		5
OPR	Thick, continuous zones or bands of clay	6-24	10,13-20

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Table 5: The Joint Water Reduction Number (J_w)

	Class	Water pressure (kg/cm^2)	J_w
A	Dry excavations or minor inflow, i.e., < 5 l/min locally	< 1	1
B	Medium inflow or pressure, occasional outwash of joint fillings	1-2.5	0.66
C	Large inflow or high pressure in competent rock with unfilled joints	2.5-10	0.5
D	Large inflow or high pressure	2.5-10	0.33
E	Exceptionally high inflow or pressure at blasting, decaying with time	>10	0.2-0.1
F	Exceptionally high inflow or pressure	>10	0.1-0.05

Table 6: The Stress Reduction Factor (SRF)

	Class	SRF
	(a) Weakness zones intersecting excavation, which may cause loosening of rock mass when tunnel is excavated	
A	Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock, any depth)	10
B	Single weakness zones containing clay or chemically disintegrated rock (depth < 50 m)	5
C	Single weakness zones containing clay or chemically disintegrated rock (depth > 50 m)	2.5
D	Multiple shear zones in competent rock (clay-free), loose surrounding rock (any depth)	7.5
E	Single shear zones in competent rock (clay-free, depth < 50 m)	5
F	Single shear zone in competent rock (clay-free, >50 m)	2.5
G	Loose open joints, heavily jointed or sugar cube (any depth)	5

1. Reduce these values of SRF by 25-30% but only if the relevant shear zones influence does not intersect the excavation

Class	SRF	
(b) Competent rock, rock mass problems	σ_c/σ_1	σ_θ/σ_c

H	Low stress, favourable stress condition	>200	<0.01	2.5
J	Medium stress, favourable stress condition	200-10	0.01-0.3	1
K	High stress, very tight pressure. Usually favourable to stability	10-5	0.3-0.4	0.5-2
L	Moderate slabbing after > 1h in massive rock	0.65-1	50-200	5-3
M	Slabbing and rock burst after a few minutes in massive rocks	0.65-1	50-200	3-2
N	Heavy rock burst and immediate dynamic deformation in massive rock	>1	200-400	<2
(c) Squeezing rock; plastic flow of incompetent rock under the influence of high rock pressure			σ_{θ}/σ_c	
O	Mild squeezing rock pressure		1-5	5-10
P	Heavy squeezing rock pressure		>5	10-20
(d) Squeezing rock; chemical swelling activity depending on presence of water				
R	Mild swelling rock pressure			5-10
S	Heavy swelling rock pressure			10-15

Table 7: The Rock Mass Classification based on *Q*-System

Q-value	Class	Rock mass
400-1000	A	Exceptionally good
100-400	A	Extremely good
40-100	A	Very good
10-40	B	Good
4-10	C	Fair
1-4	D	Poor
0.1-1	E	Very poor
0.01-0.1	F	Extremely poor
0.001-0.01	G	Exceptionally poor

Table 8: Discontinuity orientation factor O factor

O factor description	Set A	Set B
Very favourably oriented	2.0	1.5

Quite favourable	1.0	1.0
Unfavourable	0.75	0.9
Very unfavourable	0.5	0.8
Causing failure if unsupported	0.25	0.5

Table 9: Environmental and geological condition number

I_{wice}^a	Desert environment	Wet environment	Tropical storms	Ice wedging
Stable structure; competent rock	1.0	0.7	0.5	0.9
Stable structure; incompetent rock	0.7	0.6	0.3	0.5
Unstable structure; competent rock	0.8	0.5	0.1	0.3
Unstable structure; incompetent rock	0.5	0.3	0.05	0.2

^aWhen drainage measures are installed, apply $I_{wice} \times 1.5$, when slope reinforcement measures are installed, apply $I_{wice} \times 1.3$, and when drainage and reinforcement are installed, apply both factors $I_{wice} \times 1.5 \times 1.3$

Table 10: physical condition

Description	SRF_a
A Slight loosening due to surface location, disturbance from blasting or excavation	2.5
B Loose blocks, signs of tension cracks and joint shearing, susceptibility to weathering, severe disturbance from blasting	5
C As B, but strong susceptibility to weathering	10
D Slope is in advanced stage of erosion and loosening due to periodic erosion by water and/or ice-wedging effects	15
E Residual slope with significant transport of material downslope	20

Table 11: SRF_b : stress and strength

Description	σ_c/σ_1^a	SRF_b
F Moderate stress-strength range	50-200	2.5-1
G High stress-strength range	10-50	5-2.5
H Localized intact rock failure	5-10	10-5
J Crushing or plastic yield	2.5-5	15-10
K Plastic flow of strain softened material	1-2.5	20-15

^a σ_c =unconfined compressive strength (UCS), σ_1 =maximum principal stress

Table 12: SRF_c : major discontinuity

SRF_c	Favourable	Unfavourable	Very unfavourable	Causing failure of unsupported
L Major discontinuity with little or no clay	1	2	4	8
M Major discontinuity with $RQD_{100} = 0^a$ due to clay and crushed rock	2	4	8	16

N	Major discontinuity with $RQD_{300} = 0^b$ due to clay and crushed rock	4	8	12	24
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^a $RQD_{100} = 1\ m$ perpendicular sample of discontinuity, ^b $RQD_{300} = 3\ m$ perpendicular sample of discontinuity

