CHAMBER OF MINES OF SOUTH AFRICA

EXAMINATION PAPER

SUBJECT: CERTIFICATE IN ROCK MECHANICS PAPER 3.3 : MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK) **SUBJECT CODE:** COMRME **EXAMINATION DATE:** 13 OCTOBER 2016 ~~OCTOBER 2016~~ **TIME**: ~~3 HOURS~~14:30 – 17:30	**EXAMINER:** PJ LE ROUX **MODERATOR:** W JOUGHIN **TOTAL MARKS:** [100] **PASS MARK:** (60%)

NUMBER OF PAGES: 23 (incl)

THIS IS NOT AN OPENBOOK EXAMINATION – ONLY REFERENCES PROVIDED ARE ALLOWED

SPECIAL REQUIREMENTS:

1. Answer **all questions**. Answer the questions **legibly** in English.
2. Write your **ID Number** on the outside cover of each book used and on any graph paper or other loose sheets handed in.

NB: Your name **must not** appear on any answer book or loose sheets.

3. Show all calculations **and check calculations on which the answers are based.**
4. **NO** hand-held electronic calculators may be used for this exam.
5. Write **legibly** in ink on the **right hand page** only – **left hand pages will not be marked**.
6. Illustrate your answers by means of sketches or diagrams wherever possible.
7. **Final answers** must be given to an accuracy which is typical of practical conditions, However be careful not to use too few decimal places during your calculations, as rounding errors may result in incorrect answers

   NB Ensure that the correct unit of measure (SI unit) are recorded as marks will be deducted from answers if the incorrect unit is used. (even if the calculated value is correct).
8. In answering the questions, full advantage should be taken of your practical experience as well as data given.
9. Please note that you are not allowed to contact your examiner or moderator regarding this examination.
10. Cell phones are **NOT** allowed in the examination room.

QUESTION 1 – MULTIPLE CHOICE

Only write the question number and the answer. **Example**: 1.1 (c)

1.1 Extension fractures

a) A fracture that develops parallel to the direction of the greatest compressive stress
b) A fracture that develops perpendicular to the direction of greatest stress and parallel to the direction of compression
c) A fracture that develops perpendicular and parallel to the direction of greatest stress and perpendicular to the direction of compression
d) A fracture that runs perpendicular to the direction of smallest stress and parallel to the direction of compression

[2]

1.2 The presence of 'rock flour' (white powder) on structures (shear fractures, joints, faults, etc) indicates

a) the occurrence of substantial pressure on the structures
b) the occurrence of substantial stress on the structures
c) the occurrence of substantial closure on the structures
d) the occurrence of substantial shear displacements on the structures

[2]

1.3 Rockfall height can be defined as

a) the rockburst height of rock that dislodged and collapsed into the mining excavation
b) the rock that dislodged and collapsed into the mining excavation
c) the thickness of the rock that dislodged and collapsed into the mining excavation
d) the length of the rock that dislodged and collapsed into the mining excavation

[2]

1.4 In quasi-static conditions, shallower mines generally have

a) a higher fall out height compared with deeper mines
b) a lower fall out height compared with deeper mines
c) no difference in fall out height compared with deeper mines
d) none of the above

[2]

1.5 The thickness of strata to be supported is 1.6m. The density of the rock is assumed to be 2750 kg/m^3 and the gravitational acceleration is taken as 9.81 m/s^2. The support resistance required to support the above mentioned would be:

a) 16,8 kN/m^2
b) 38,6 kN/m^2
c) 43,2 kN/m^2
d) 42,3 kN/m^2

[2]

1.6 Factors of safety for support design in low risk excavations should be between

a) 1.2 and 1.5
b) 0.9 and 1.6
c) 1.6 and 2.0
d) >2.0

[2]

1.7 The word abutment when used on the mining operations simply means:

a) the 'gully' of the unmined rock around a mined out area
b) the 'edge' of the unmined rock around a mined out area
c) the 'space' between support units of the unmined rock around a mined out area
d) the 'bond' strength of the unmined rock around a mined out area

[2]

1.8 In rock engineering the '45° rule' normally referred to,

a) to ensure that the tunnel will not be placed in the low abutment stress lobes
b) to ensure that the tunnel will be placed in the highly fractured zone
c) to ensure that the tunnel will be placed in the high abutment stress lobes
d) to ensure that the tunnel will not be placed in the high abutment stress lobes

[2]

1.9 In very shallow Bord and Pillar mining (depths < ± 400mbs), in-stope pillars are required to carry the

a) full weight of overburden up to surface due to the absence of horizontal clamping stresses
b) full length of overburden up to surface due to the stress levels sufficient to assist stability

c) full weight of overburden up to surface due to the presence of high stress levels sufficient to assist stability

d) full weight of overburden up to 20m from surface due to the absence of stress levels sufficient to assist stability

[2]

1.10 In shallow mining depth stopes approximately 40mbs, the use of

a) crush in-stope pillars is required
b) non-yield in-stope pillars is required
c) yielding in-stope pillars is required
d) none of the above

[2]

1.11 Hydraulic Radius can be defined as

a) Circumference / Area
b) Area / Perimeter
c) Circumference / Perimeter
d) Perimeter / Area

[2]

1.12 ELOS can be defined as

$$ELOS = \frac{volume\ of\ slough\ from\ stope\ surface}{stope\ height\ x\ wall\ strike\ length}$$

ELOS stands for

a) Equivalent Linear Overbreak Stress
b) Equal Linear Overbreaking Stress
c) Equivalent Linear Over Stress
d) Equivalent Linear Overbreak Slough

[2]

1.13 $I_{s(50)}$ stands for

a) Point Load Index for 500 mm diameter core
b) Point Load Index for 5000 mm diameter core
c) Point Load Index for 50 mm diameter core

d) Point Load Index for 5 mm diameter core

[1]

[25 MA

QUESTION 2

In open stoping backfill forms an integral part of the mining process.

a) With the aid of a sketch discuss the application of backfill and what you would consider as important with reference to placement in an open stope, curing time as well as why stand-up height would be important. [5]

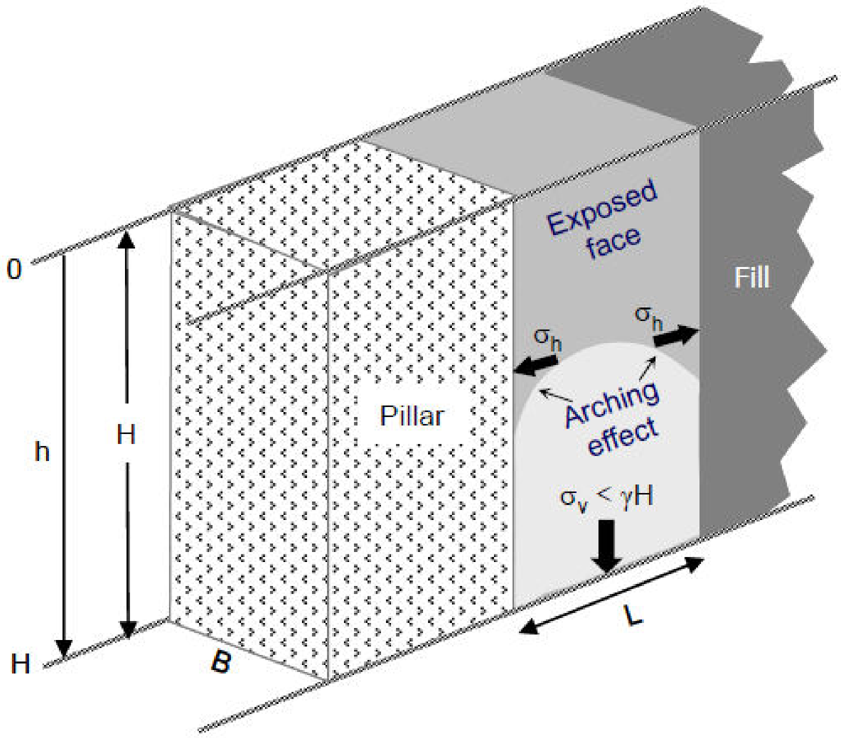
b) Briefly describe the key points to consider when designing a bulkhead. [4]

c) Using the equations below calculate the required Uniaxial Compressive Strength (UCS) in kPa for a given stand-up height of the backfill as shown in Table 1:

Table 1. Required stand up heights for open stopes

Height	5m	10m	15m	20m	25m

The open stope width is 20m and the backfill internal friction angle and cohesive strength are 25° and 20 kPa respectively. The backfill bulk unit weight is 22 kN/m^3 and a minimum safety factor of 1.5 is required.



Make use of the attached table and graph in the question paper to show your results. Plot the UCS results on the graph and discuss the obtained results.

$$UCS_{design} = \frac{1.25B}{2K\tan\phi}\left(\gamma - \frac{2c}{B}\right) \times \left[1 - \exp\left(-\frac{2HK\tan\phi}{B}\right)\right] FS$$

where

$$K = \frac{1+\sin^2\phi}{\cos^2\phi + 4\tan^2\phi} = \frac{1}{1+2\tan^2\phi}$$

B = stope width

K = Backfill pressure coefficient

c = Backfill cohesive strength (kPa)

$\emptyset$ = Backfill internal friction angle in degree

γ = backfill bulk unit weight (kN/m^3)

H = Backfill height (m)

FS = Factor of safety [16]

[25 ~~MARKS~~]

QUESTION 3

A large crusher chamber is being designed for an underground mine. The excavation is 30 m long, 12 m wide and 15 m high and the long axis is orientated at 150°. Geotechnical mapping has indicated that there are two major joint sets in the area. There orientations are as follows:

	Dip	Dip direction
J1	80°	60°
J2	35°	240°

The joint friction angle has been estimated as 32°. Cohesion can be considered negligible. The unit weight of the rock is 27kN/m^3. The excavation is to be supported with 250 kN grouted cable anchors to a safety factor of 1.5.

a) Draw the excavation in plan view and show the joint traces (clearly show the orientation of the excavation and joints). Draw a vertical cross-section showing the largest wedges that can be formed in the roof and sides of the excavation (approximately in proportion). Calculate the height and weight of each of the wedges and indicate these on the drawing. [12]

b) Calculate the number of cable anchors required to support the roof wedge. Suggest an appropriate pattern of cable anchors for the roof based on this calculation. [5]

c) Determine which of the two side wedges is more unfavourable and calculate the number of cable anchors required to prevent it from sliding. Suggest an appropriate pattern of cable anchors for the side based on this calculation. [7]

d) How could you improve the design in order to reduce the support requirements? [1]

[25 ~~MARKS~~]

QUESTION 4

As part of a mining study, rock mass characterisation must be carried out to understand the geotechnical conditions in which mining will take place.

a) Briefly outline the types of data you would investigate to characterise the rock mass, the relevance of the data to the geotechnical investigation, and what approaches you would use to obtain this data. [5]

b) From the rock mass exposure shown in the photograph, Slide 1, identify the joint sets, estimating the joint spacings, joint orientations (in relation to the directions shown in the exposure), and calculate the RQD. Using the approach of Laubscher to classify the rock mass, obtain a value for the MRMR. Outline assumptions that you need to make and would need to be addressed to improve the results by making use of the attached diagrams.

A copy of the exposure has also been provided on the last page of the examination paper. The page should be handed in together with the answer booklet, showing the selected joint sets.

[15]

c) An excavation 35 m long and 12 m wide will be developed in this ground. Calculate the hydraulic radius and comment on the stability of the excavation. Discuss how you might improve the excavation stability by making use of the attached diagrams. [5]

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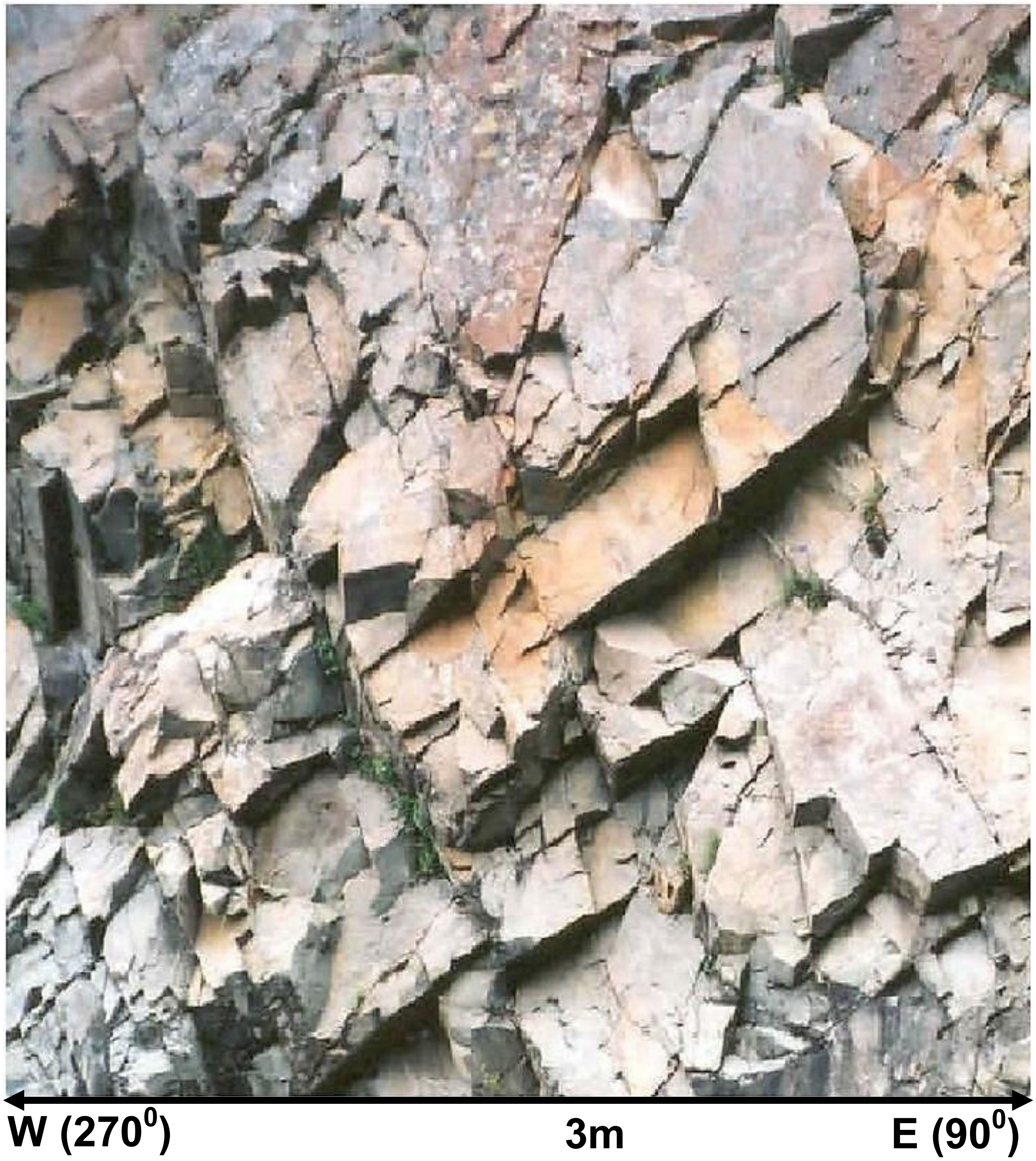


Slide 1: Rock mass exposure

[25 ~~MARKS~~]

TOTAL MARKS: [100]

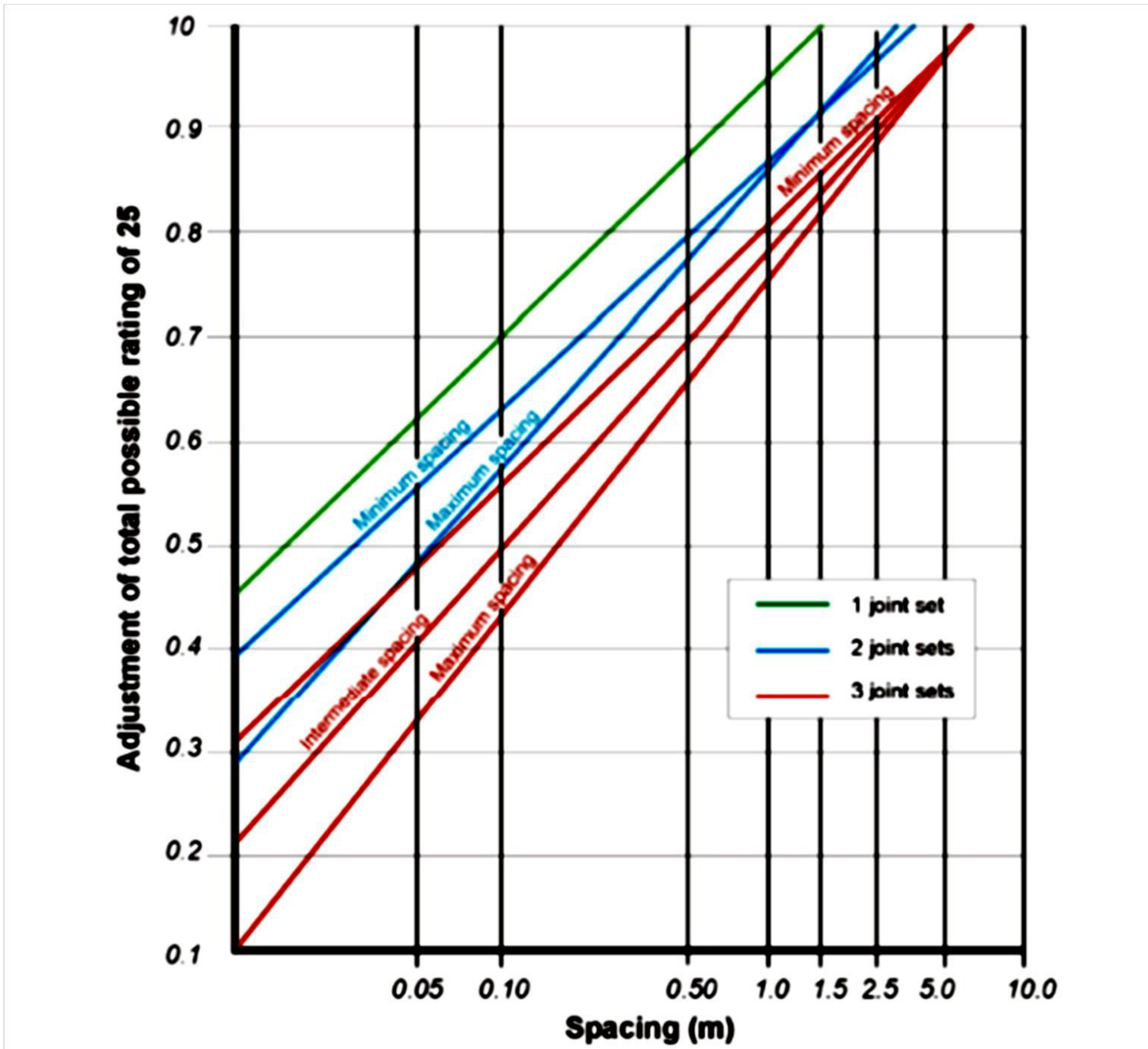
Tables of Information

	Parameter	Range of Values										
1	*RQD*	100-97		96-84	83-71	70-56	55-44	43-31	30-17	16-4	3-0	
	Rating (= *RQD* x 15/100)	15		14	12	10	8	6	4	2	0	
2	*UCS* (MPa)	185	184-165	184-165	164-145	144-125	124-105	104-85	84-65	44-25	24-5	4-0
	Rating	20	18	16	14	12	10	8	6	4	2	0
3	Joint Spacing	Refer Figure 9										
	Rating	25						0				
4	Joint Condition Including Groundwater	Refer Table 9										
	Rating	40						0				

IRS–MPa rating %		RQD rating %		Joint spacing m	Fracture frequency, FF/m: Average per metre	Rating: 1 set	2 set	3 set
>185	20	97–100	15	0<—>25	0,1	40	40	40
165–185	18	84–96	14	See Fig. 2	0,15	40	40	40
145–164	16	71–83	12		0,20	40	40	38
125–144	14	56–70	10		0,25	40	38	36
105–124	12	44–55	8		0,30	38	36	34
85–104	10	31–43	6		0,50	36	34	31
65–84	8	17–30	4		0,80	34	31	28
45–64	6	4–16	2		1,00	31	28	26
35–44	5	0–3	0		1,50	29	26	24
25–34	4				2,00	26	24	21
12–24	3				3,00	24	21	18
5–11	2				5,00	21	18	15
1–4	1				7,00	18	15	12
					10,00	15	12	10
					15,00	12	10	7
					20,00	10	7	5
					30,00	7	5	2
					40,00	5	2	0
					ALLOW FOR CORE RECOVERY			

Parameter	Description		Accumulative % adjustment of possible rating of 40			
				Adjustment, %		
			Dry	Moist	Mod. pressure 25–125 1/m	High pressure >125 1/m
A	Multi wavy directional		100	100	95	90
	Uni		95	90	85	80
Large-scale joint expression	Curved		85	80	75	70
	Slight undulation		80	75	70	65
	Straight		75	70	65	60
B	Rough stepped/irregular		95	90	85	80
	Smooth stepped		90	85	80	75
Small-scale joint expression	Slickensided stepped		85	80	75	70
	Rough undulating		80	75	70	65
200 mm × 200 mm	Smooth undulating		75	70	65	60
	Slickensided undulating		70	65	60	55
	Rough planar		65	60	55	50
	Smooth planar		60	55	50	45
	Polished		55	50	45	40
C Joint wall alteration weaker than wall rock and only if it is weaker than the filling			75	70	65	60
D	Non-softening and sheared material	Coarse	90	85	80	75
		Medium	85	80	75	70
		Fine	80	75	70	65
Joint filling	Soft sheared material, e.g. talc	Coarse	70	65	60	55
		Medium	60	55	50	45
		Fine	50	45	40	35
	Gouge thickness < amplitude of irregularities		45	40	35	30
	Gouge thickness > amplitude of irregularities		30	20	15	10

Example: A straight joint with a smooth surface and medium sheared talc under dry conditions gives A = 70%, B = 65%, D = 60%; total adjustment = 70 × 65 × 60 = 27%, and the rating is 40 × 27% = 11.



Rate of weathering and adjustments (%)					
Description of weathering extent	6 months	1 year	2 years	3 years	4 + years
Fresh	100	100	100	100	100
Slightly	88	90	92	94	96
Moderately	82	84	86	88	90
Highly	70	72	74	76	78
Completely	54	56	58	60	62
Residual soil	30	32	34	36	38

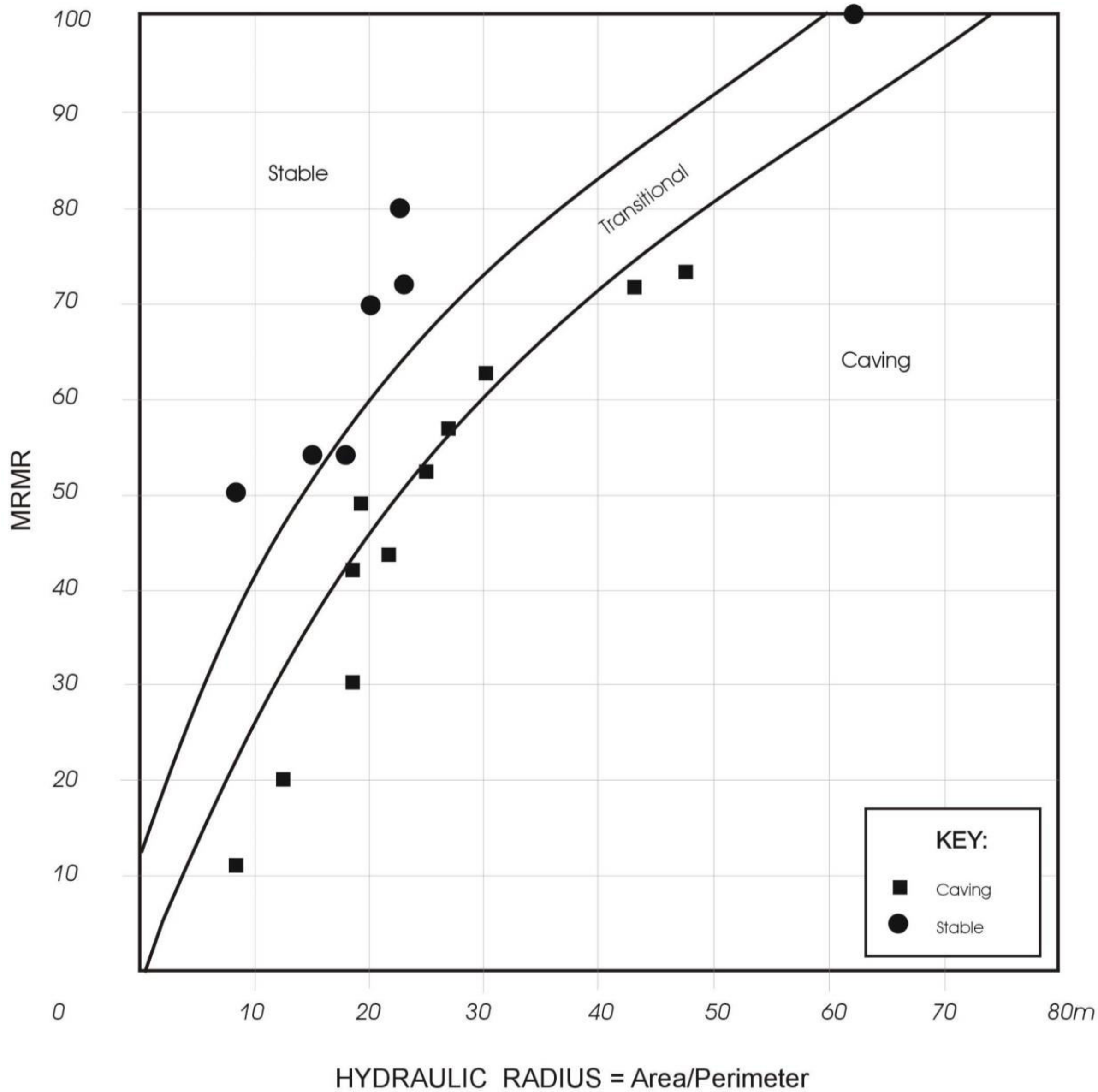
Excavation Technique	Adjustment (%)
Boring	100
Smooth wall blasting	97
Good conventional blasting	94
Poor blasting	80

Table 11 Adjustments to MRMR due to joint orientation

Number of joints defining the block	Adjustment (%) Number of faces inclined away from the vertical				
	70	75	80	85	90
3	3	-	2	-	-
4	4	3	-	2	-
5	5	4	3	2	1
6	6	5	4	3	2 or 1

Laubscher's table for stress adjustments required .

Stress adjustments range between 60% and 120%. The adjustments for mining induced stresses are essentially based on judgement. Good confinement enhances stability and the maximum positive adjustment is 120%. Poor confinement, associated with numerous, closely spaced joint sets, does not promote stability, and the maximum negative adjustment is 60%. High stress to strength ratio causes fracturing and reduces stability.



Laubcher's Stability graph

Cave angle and failure zone (Laubscher 1990)

	Condition	Depth (m)	Adjusted *MRMR*				
			100-81	80-61	60-41	40-21	20-0
Cave angle	No lateral restraint	100	80	70	60	50	40
		500	70	60	50	40	30
	Lateral restraint from caved material	100	90	80	70	60	50
		500	80	70	60	50	40
Extent of failure zone	Surface	100	10 m	20 m	30 m	50 m	75 m
		500	10 m	20 m	30 m	50 m	75 m
	Underground	100	10 m	20 m	30 m	50 m	100 m
		500	20 m	30 m	50 m	100 m	200 m

Table A1

Rock quality designation		RQD (%)
A	Very poor	0–25
B	Poor	25–50
C	Fair	50–75
D	Good	75–90
E	Excellent	90–100

Notes: (i) Where RQD is reported or measured as $\leqslant 10$ (including 0), a nominal value of 10 is used to evaluate Q. (ii) RQD intervals of 5, i.e., 100, 95, 90, etc., are sufficiently accurate.

Table A2

Joint set number		J_n
A	Massive, no or few joints	0.5–1
B	One joint set	2
C	One joint set plus random joints	3
D	Two joint sets	4
E	Two joint sets plus random joints	6
F	Three joint sets	9
G	Three joint sets plus random joints	12
H	Four or more joint sets, random, heavily jointed, 'sugar-cube', etc.	15
J	Crushed rock, earthlike	20

Notes: (i) For tunnel intersections, use $(3.0 \times J_n)$. (ii) For portals use $(2.0 \times J_n)$.

Table A3

Joint roughness number		J_r
(a) *Rock-wall contact, and (b) rock-wall contact before 10 cm sear*		
A	Discontinuous joints	4
B	Rough or irregular, undulating	3
C	Smooth, undulating	2
D	Slickensided, undulating	1.5
E	Rough or irregular, planar	1.5
F	Smooth, planar	1.0
G	Slickensided, planar	0.5
(b) *No rock-wall contact when sheared*		
H	Zone containing clay minerals thick enough to prevent rock-wall contact.	1.0
J	Sandy, gravely or crushed zone thick enough to prevent rock-wall contact	1.0

Notes: (i) Descriptions refer to small-scale features and intermediate scale features, in that order. (ii) Add 1.0 if the mean spacing of the relevant joint set is greater than 3 m. (iii) $J_r = 0.5$ can be used for planar, slickensided joints having lineations, provided the lineations are oriented for minimum strength. (iv) J_r and J_a classification is applied to the joint set or discontinuity that is least favourable for stability both from the point of view of orientation and shear resistance, τ(where $\tau \approx \sigma_n \tan^{-1}(J_r/J_a)$.

Table A4

Joint alteration number		ϕ_r approx. (deg)	J_a
(a) *Rock-wall contact (no mineral fillings, only coatings)*			
A	Tightly healed, hard, non-softening, impermeable filling, i.e., quartz or epidote	—	0.75
B	Unaltered joint walls, surface staining only	25–35	1.0
C	Slightly altered joint walls, non-softening mineral coatings, sandy particles, clay-free disintegrated rock, etc.	25–30	2.0
D	Silty- or sandy-clay coatings, small clay fraction (non-softening)	20–25	3.0
E	Softening or low friction clay mineral coatings, i.e., kaolinite or mica. Also chlorite, talc, gypsum, graphite, etc., and small quantities of swelling clays	8–16	4.0
(b) *Rock-wall contact before 10 cm shear (thin mineral fillings)*			
F	Sandy particles, clay-free disintegrated rock, etc.	25–30	4.0
G	Strongly over-consolidated non-softening clay mineral fillings (continuous, but <5 mm thickness)	16–24	6.0
H	Medium or low over-consolidation, softening, clay mineral fillings (continuous, but <5 mm thickness)	12–16	8.0
J	Swelling-clay fillings, i.e., montmorillonite (continuous, but < 5 mm thickness). Value of J_a depends on per cent of swelling clay-size particles, and access to water, etc.	6–12	8–12
(c) *No rock-wall contact when sheared (thick mineral fillings)*			
KLM	Zones or bands of disintegrated or crushed rock and clay (see G, H, J for description of clay condition)	6–24	6, 8, or 8–12
N	Zones or bands of silty- or sandy-clay, small clay fraction (non-softening)	—	5.0
OPR	Thick, continuous zones or bands of clay (see G, H, J for description of clay condition)	6–24	10, 13, or 13–20

Table A5

	Joint water reduction factor	Approx. water pres. (kg/cm^2)	J_w
A	Dry excavations or minor inflow, i.e., < 5 l/min locally	< 1	1.0
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, considerable outwash of joint fillings	2.5–10	0.33
E	Exceptionally high inflow or water pressure at blasting, decaying with time	> 10	0.2–0.1
F	Exceptionally high inflow or water pressure continuing without noticeable decay	> 10	0.1–0.05

Notes: (i) Factors C to F are crude estimates. Increase J_w if drainage measures are installed. (ii) Special problems caused by ice formation are not considered. (iii) For general characterisation of rock masses distant from excavation influences, the use of $J_w = 1.0$, 0.66, 0.5, 0.33, etc. as depth increases from say 0–5, 5–25, 25–250 to > 250 m is recommended, assuming that RQD/J_n is low enough (e.g. 0.5–25) for good hydraulic connectivity. This will help to adjust Q for some of the effective stress and water softening effects, in combination with appropriate characterisation values of SRF. Correlations with depth-dependent static deformation modulus and seismic velocity will then follow the practice used when these were developed.

Table A6

	Stress reduction factor			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 of excavation ⩽50 m)			5
C	Single weakness zones containing clay or chemically disintegrated rock (depth of excavation > 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 of excavation ⩽50 m)			5.0
F	Single shear zones in competent rock (clay-free), (depth of excavation > 50 m)			2.5
G	Loose, open joints, heavily jointed or 'sugar cube', etc. (any depth)			5.0
		σ_c/σ_1	σ_θ/σ_c	SRF
	(b) *Competent rock, rock stress problems*			
H	Low stress, near surface, open joints	> 200	< 0.01	2.5
J	Medium stress, favourable stress condition	200–10	0.01–0.3	1
K	High stress, very tight structure. Usually favourable to stability, may be unfavourable for wall stability	10–5	0.3–0.4	0.5–2
L	Moderate slabbing after > 1 h in massive rock	5–3	0.5–0.65	5–50
M	Slabbing and rock burst after a few minutes in massive rock	3–2	0.65–1	50–200
N	Heavy rock burst (strain-burst) and immediate dynamic deformations in massive rock	< 2	> 1	200–400
		σ_θ/σ_c	SRF	
	(c) *Squeezing rock: plastic flow of incompetent rock under the influence of high rock pressure*			
O	Mild squeezing rock pressure	1–5	5–10	
P	Heavy squeezing rock pressure	> 5	10–20	
		SRF		
	(d) *Swelling rock: chemical swelling activity depending on presence of water*			
R	Mild swelling rock pressure	5–10		
S	Heavy swelling rock pressure	10–15		

$J_v = 1/J_1+1/J_2+1/J_3+1/\text{Random}$

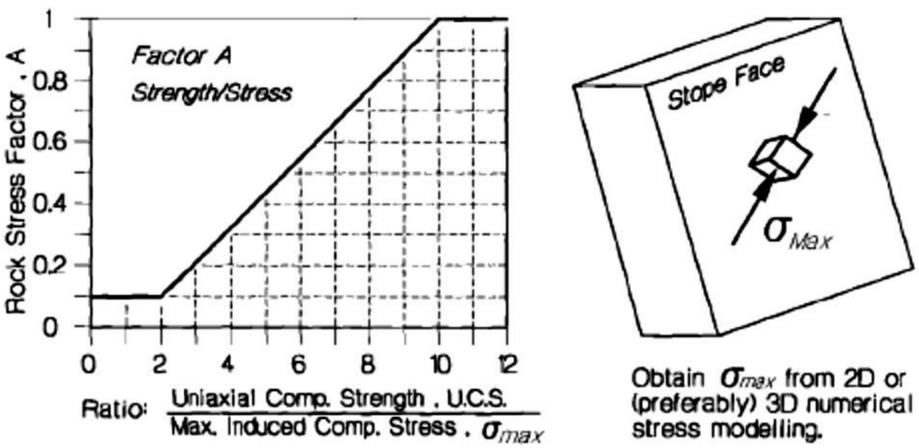
$RQD = 115-3.3J_v$

$Q = (RQD/J_n).(J_r/J_a).(J_w.SRF)$

$N' = Q' \times A \times B \times C$

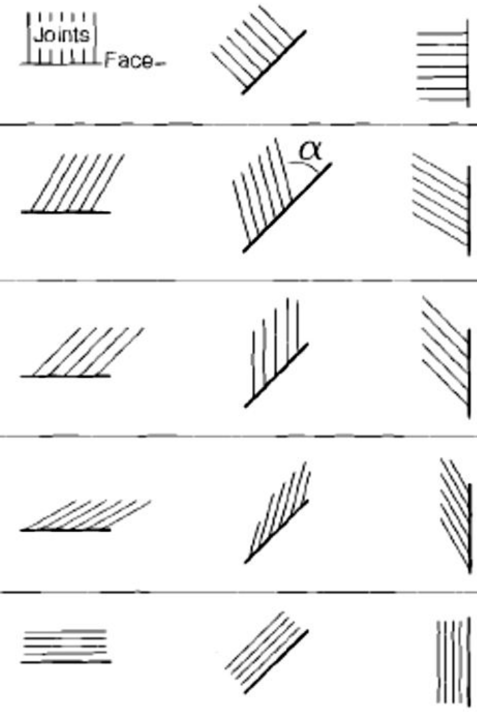
Rock Stress Factor A

Determine maximum induced tangential stress (compressive) acting at the centre centre of the stope face being considered. Obtain uniaxial compressive strength strength for the intact rock. Evaluate Stress Factor, *A*, using the graph below:

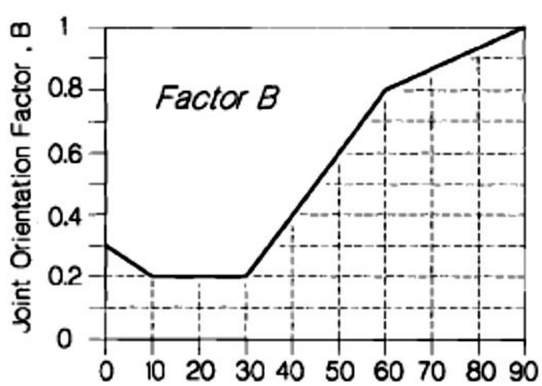


Obtain σ_{max} from 2D or (preferably) 3D numerical stress modelling.

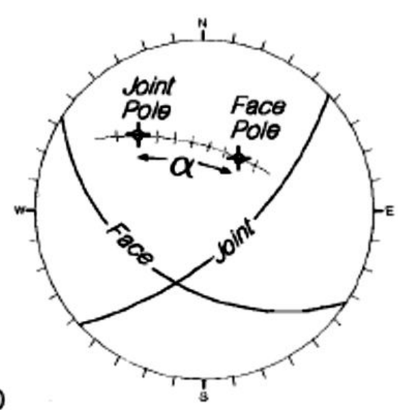
Joint Orientation Factor, B



Horizontal Back	Inclined Wall	Vertical Wall	True Angle between Face & Joint	Potvin Factor *B*
			$\alpha = 90°$	1.0
			$\alpha = 60°$	0.8
			$\alpha = 45°$	0.5
			$\alpha = 30°$	0.2
			$\alpha = 0°$	0.3



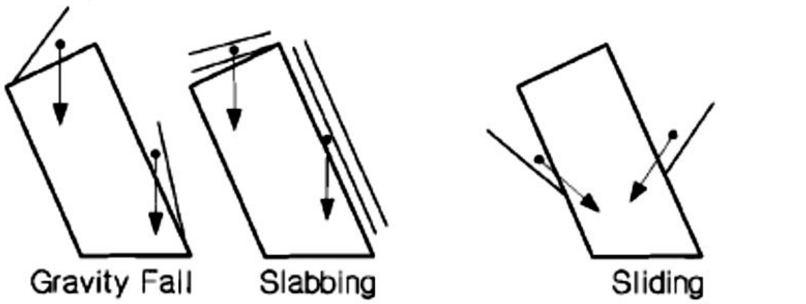
True Angle Between Face and Joint
(Angle α Between Poles)



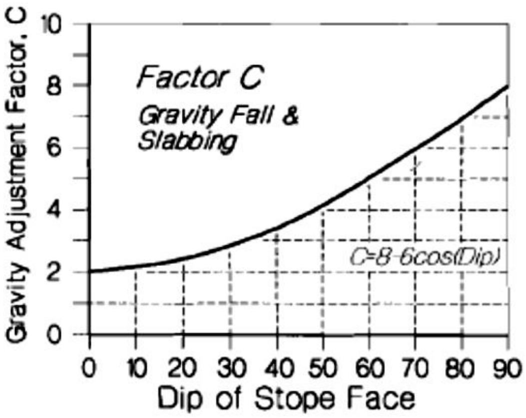
Determination of the minimum or true angle between two planes = Angle α between poles

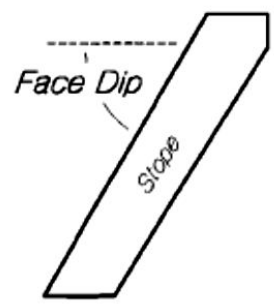
Gravity Adjustment Factor, C

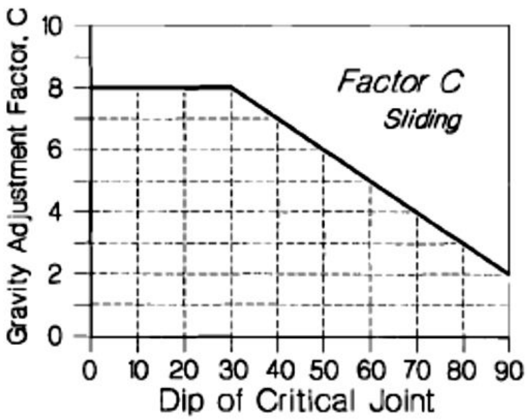
1) Determine the most likely mode of structural failure in case study using the figures below:

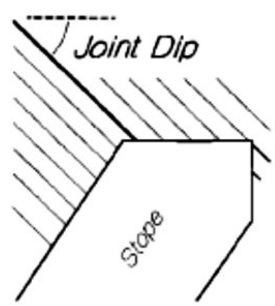


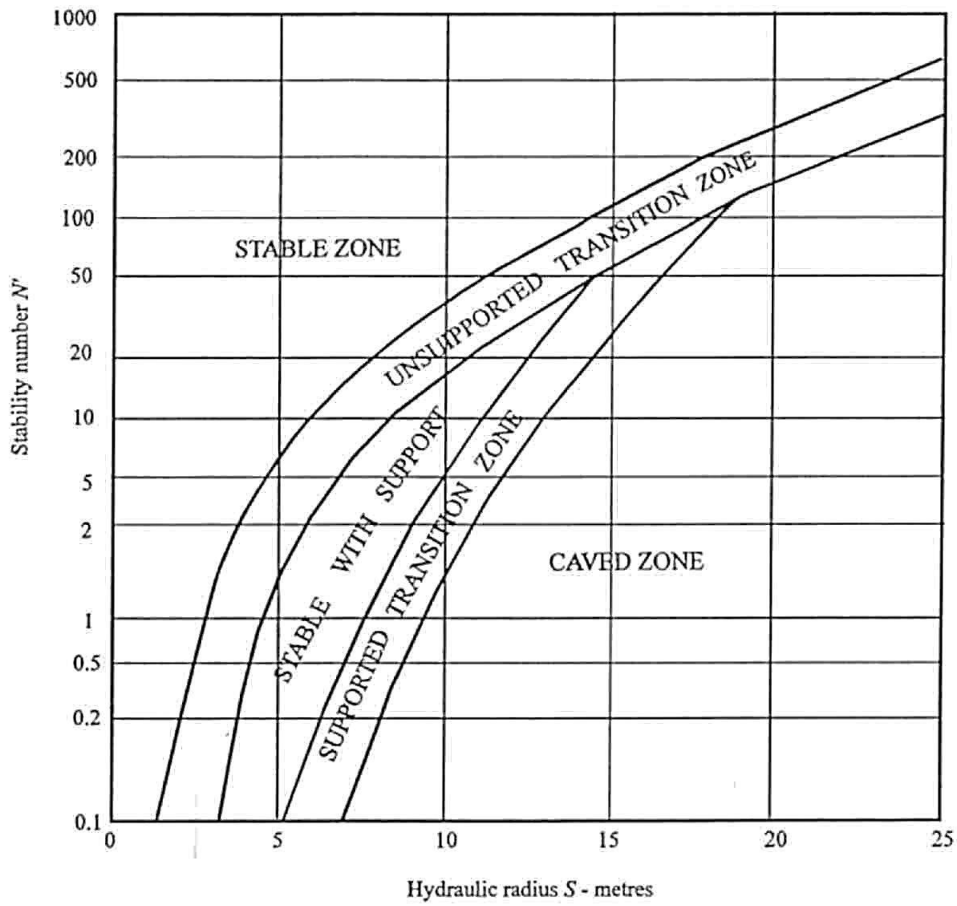
2) Next determine the gravity adjustment factor, *C*, based on the failure mode using the appropriate chart below.



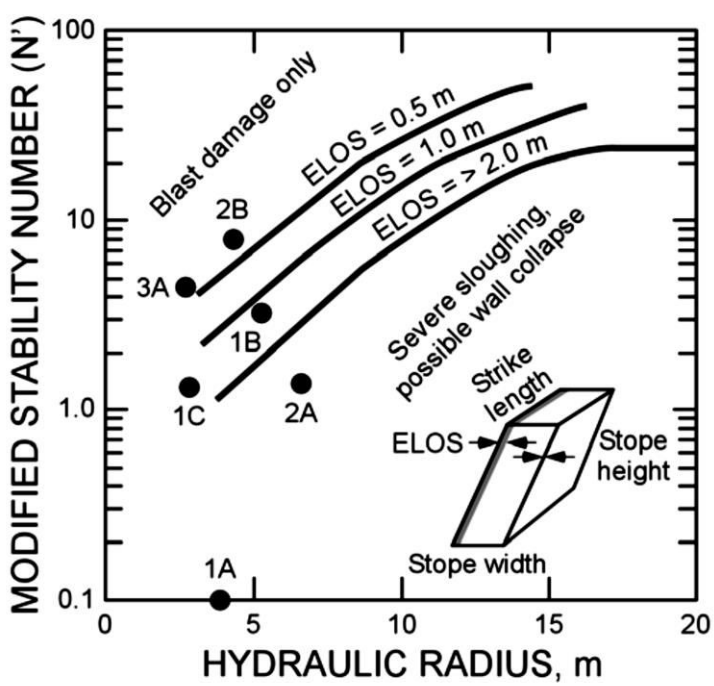




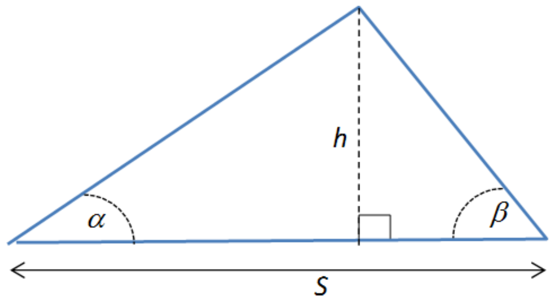




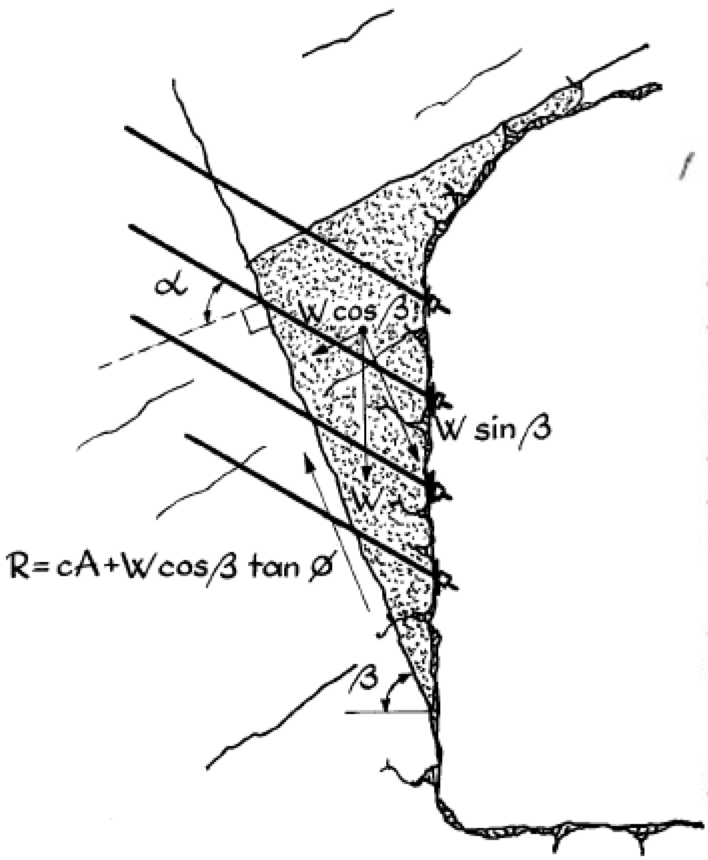
1000
500
200
100
50
20
10
5
2
1
0.5
0.2
0.1
0
5
10
15
20
25
Stability number N'
Hydraulic radius S - metres
STABLE ZONE
UNSUPPORTED TRANSITION ZONE
STABLE WITH SUPPORT
SUPPORTED TRANSITION ZONE
CAVED ZONE



MODIFIED STABILITY NUMBER (N')
100
10
1.0
0.1
0
5
10
15
20
HYDRAULIC RADIUS, m
Blast damage only
ELOS = 0.5 m
ELOS = 1.0 m
ELOS = > 2.0 m
Severe sloughing, possible wall collapse
2B
3A
1B
1C
2A
1A
Strike length
ELOS
Stope height
Stope width



$$h = \frac{S \times \tan\alpha \times \tan\beta}{\tan\alpha + \tan\beta}$$



f - 1.5 for grouted bolts
2.0 for non-grouted bolts

Definitions:

- N = number of cablebolts
- W = weight of the wedge
- f = factor of safety
- B = cablebolt capacity
- b = dip of sliding surface
- c = cohesive strength of sliding surface
- A = base area of sliding surface
- a = angle between the plunge of the bolt and the normal to the sliding surface
- ϕ - joint surface angle of friction

$$N = \frac{W(f\sin\beta - \cos\beta\tan\phi) - cA}{C(\cos\alpha\tan\phi + f\sin\alpha)}$$

NAME:

EXAMINATION NUMBER:

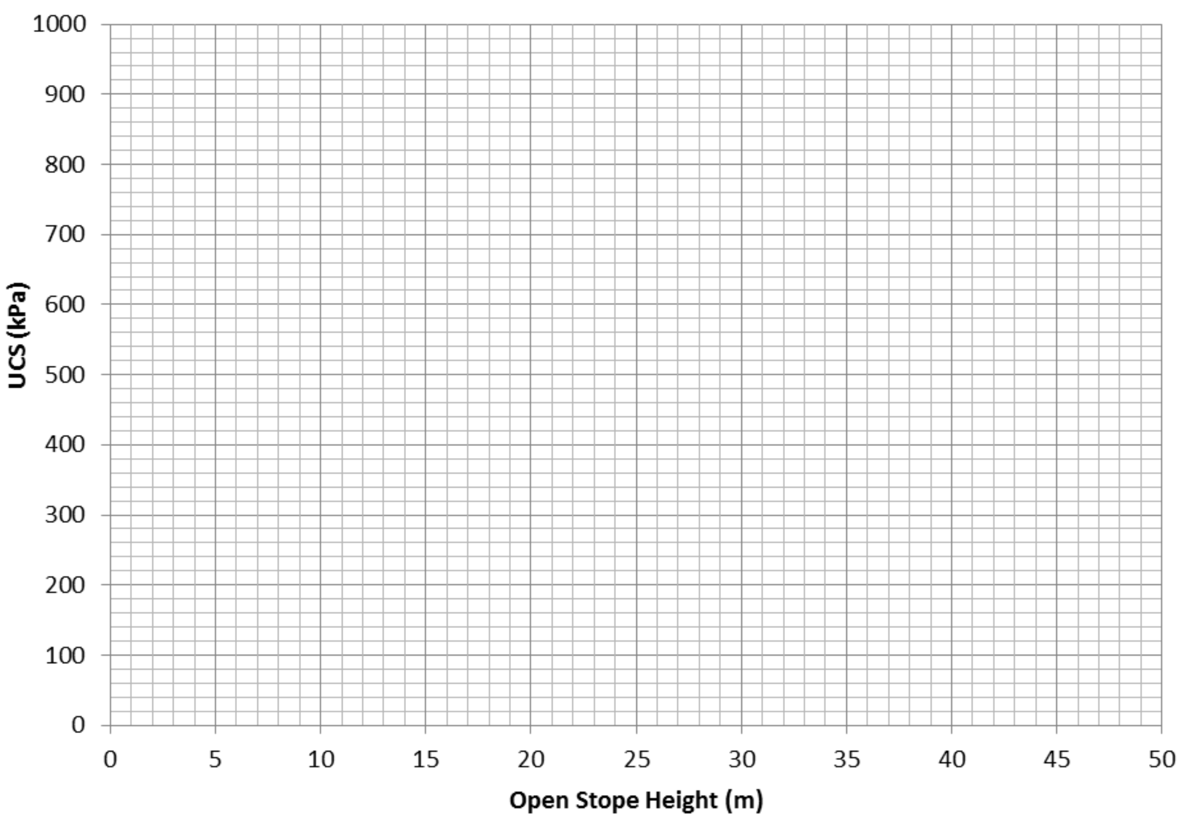
PAPER:

DATE:

Submit together with the answer sheet.

Question 2 (c)

Height	Uniaxial Compressive Strength (UCS) in kPa
5m	
10m	
15m	
20m	
25m	
30m	
35m	
40m	
45m	



NAME:

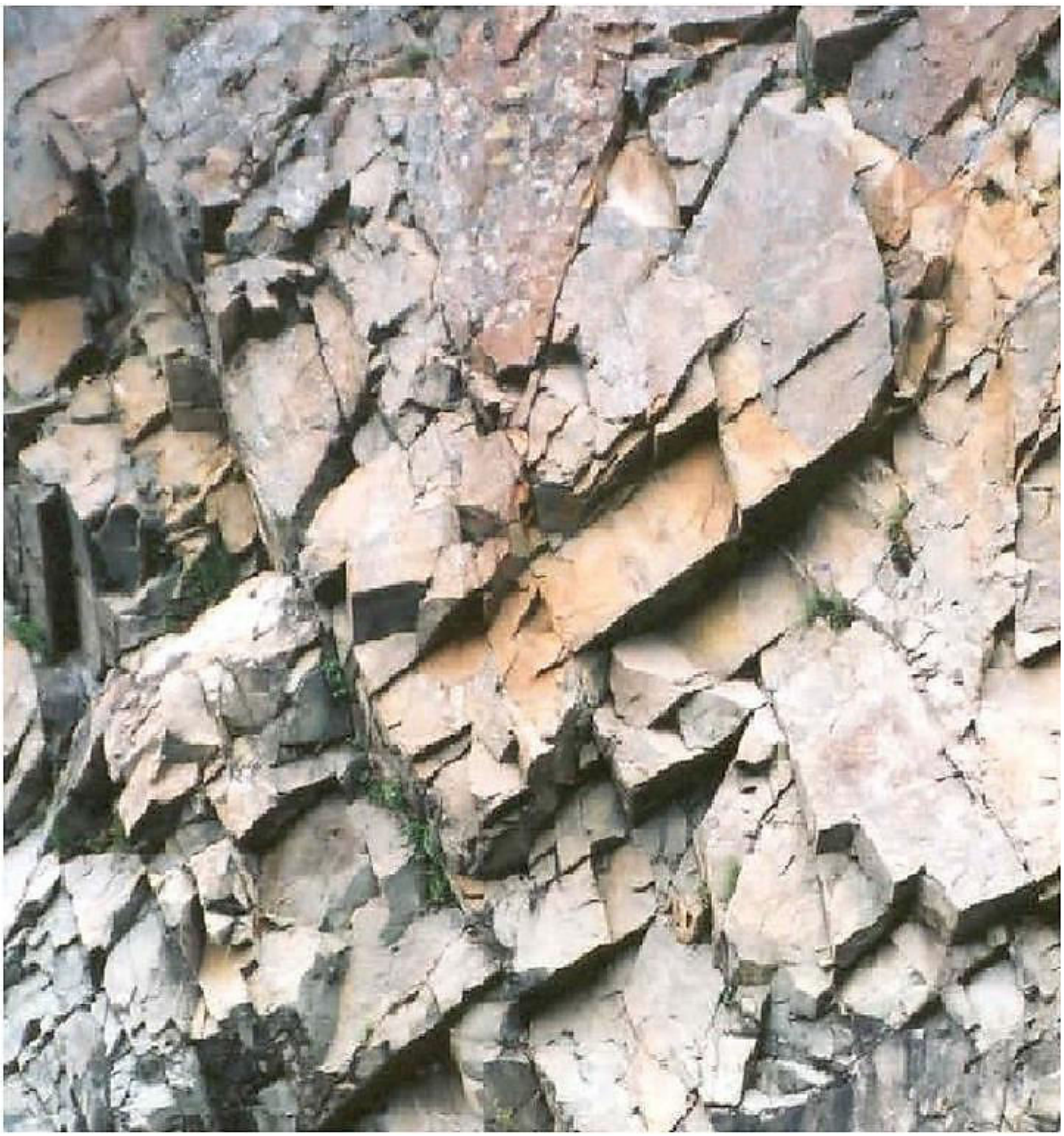
EXAMINATION NUMBER:

PAPER:

DATE:

Submit together with the answer sheet.

Question 4



W (270^0) **3m** **E (90^0)**