Model Answers for Paper 3.3 2012-10

Q1.

a) *List five reasons for carrying out geotechnical investigations? [5]*

Geotechnical investigations are required for the following reasons:

- To understand the characteristics of the rock mass in which mining will take place,
- To determine the stability of proposed mining excavations
  - Maximum dimensions of stable excavations
  - Support requirements for excavations,
- To determine the cavability of the orebody in caving operations
  - Minimum footprint dimensions
- To identify potential failure modes,
- To anticipate the expected rock mass behaviour.

Any five logical reasons

b) *Describe how you would determine the orebody and host rock properties in a geotechnical investigation programme. 5 marks*

Geological drilling programme to be defined by geologist.Geological logging and orebody definition by geologist.

Sample appropriate number of boreholes for geotechnical logging (core orientation as necessary).Geotechnical logging to be carried out to determine rock mass characteristics, geological weaknesses and joint orientations.

Mapping of rock exposures where available (eg open pit or outcrops). Joint orientations and characteristics, rock mass classifications)

Laboratory testing of intact rock (UCS, TCS, shearbox on joints)

Empirical estimation of stable stope dimensions or cavability - required for mining method selection.

Empirical evaluation of tunnels and service excavation stability and support requirements.

Numerical analysis as required.

c) *Laubscher's Mining Rock Mass Rating and Barton's Q rating are commonly used to characterise the rock mass. Describe how you would use the information in b) to characterise the rock mass using these systems. List and discuss the additional* *information that you need to collect and how it is applied in the two systems. [15]*

Laubscher RMR = IRS + FF + Jc

Or

Laubscher RMR = IRS + RQD rating+ Js + Jc

IRS = Intact rock strength, FF =Fracture frequency rating, RQD = Rock quality designation, Js = Joint spacing Rating, Jc = Joint condition rating. (2)

Adjustments (blasting, stress, orientation, weathering) are applied to the RMR to determine the MRMR (1)

Q = RQD/Jn x Jr/Ja x Jw/SRF

Jn = Joint set number, Jr = Joint roughness, Ja = Joint alteration, Jw = Joint water, SRF = stress reduction factor (2)

The information in b) provides the RQD, fracture frequency and number of joints. This is used for the FF, RQD, Js (Laubscher), RQD/Jn (Q). Ratings are obtained using the relevant tables (3)

The uniaxial compressive strength can be determined from laboratory testing. This is used to determinedLaubscher's IRS using the relevant table. Barton's Q does not take rock strength into consideration (3)

Joint characteristics are determined from mapping exposures or boreholes (1)

Joint roughness represents the surface roughness of joints (undulating, planar, smooth rough). The Lausbscher macro roughness (large scale) and micro roughness (small scale) and Barton's Jr are determined from relevant tables (2)

Joint wall alteration and infill characteristics must be determined. These are used to determine Laubscher'sJc and Barton's Ja from relevant tables. (2)

Joint water can be determined from observed seepage in underground exposures, drilling records and ground water investigations. This is applied in LaubschersJc and Barton's Jw using the relevant tables (2)

The rate of weathering is estimated by observing slaking in core or underground or carrying out slake durability tests. This is used in Laubscher'sweathering adjustment (1)

Shear zones observed during core logging or in underground exposures can be used to determine the SRF in the Q system (1)

The expected stress state can be determined from stress measurements and numerical analyses. This is applied in Laubscher's stress adjustment and in Barton's SRF (2)

Joint orientations can be determined from underground mapping or oriented core and applied in Laubscher's orientation adjustment (1)

23 marks available only 15 required.

Q2.

a) *Describe with the aid of sketches the method of longhole open stoping with post filling*with primary and secondary stopes. *[10]*

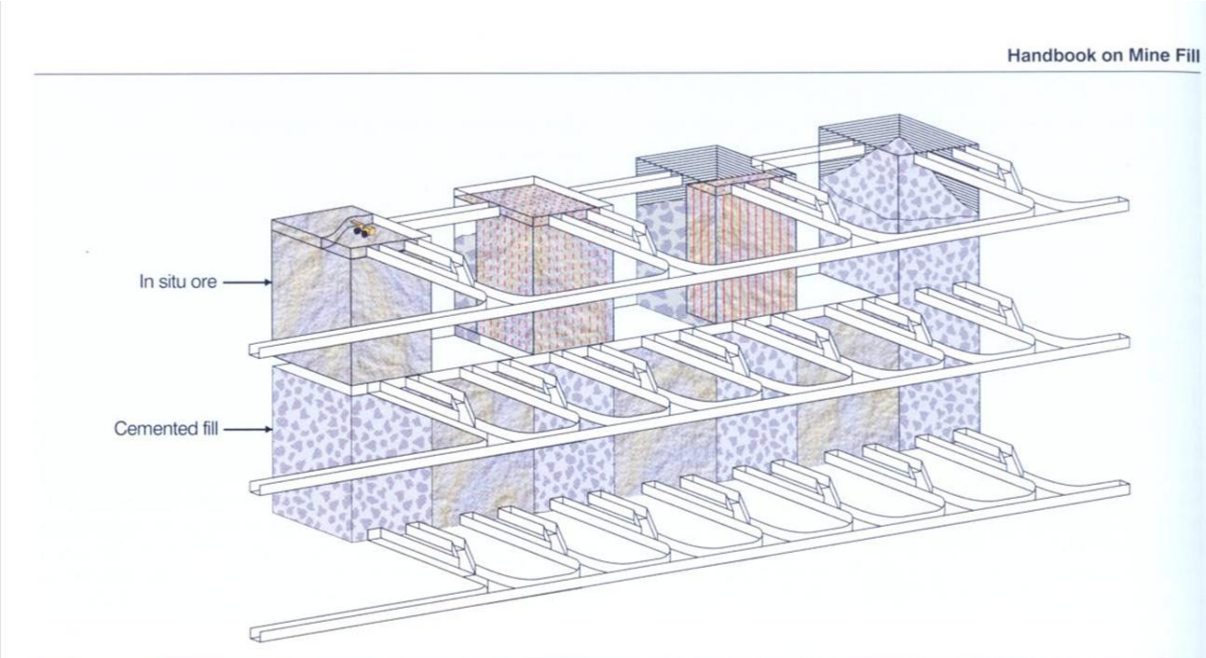


FIGURE 1.4 Schematic three dimensional view of open stope mining showing the different stages of mining, starting from left to right with pre-production, development of overcut, drilling, blasting and filling (after Potvin, 1988)

b) *Why are open stopes post filled? [3]*

Post filling is done to maximise extraction of the orebody. When the orebody is considerably larger than the maximum stable dimensions, it is necessary to divide the orebody into stope blocks, which will be stable. Each stope is filled after the ore has been extracted and this enables the mining of adjacent stope blocks.

c) *Calculate the required uniaxial compressive strength and cement content for backfill (unit weight = 18kN/m2, porosity 50%), using a factor of safety of 1.5 in stopes of the following dimensions::*

- *50m long 15m wide and 15m high*
- *50m long 20m wide and 30m high* ***[12]***

Stope length (B) = 50, stope width (D) = 15 and stope height (H) = 15

$$UCS = \frac{F.B\gamma_b(2H - D)}{2H + 2B - D}$$

$$UCS = \frac{1.5 \times 50 \times 18 \times (2 \times 15 - 15)}{2 \times 15 + 2 \times 50 - 15}$$

UCS = 176 kPa *(3)*

$$UCS = \frac{27(c/n)^{1.54}}{F}$$

$$c = n \times (UCS/27 \times F)^{(1/1.54)}$$

$$c = 50\% \times (0.176/27 \times 1.5)^{(1/1.54)}$$

c = 2.5% *(3)*

Stope length (B) = 50, stope width (D) = 20 and stope height (H) = 30

$$UCS = \frac{F.B\gamma_b(2H - D)}{2H + 2B - D}$$

$$UCS = \frac{1.5 \times 50 \times 18 \times (2 \times 30 - 20)}{2 \times 30 + 2 \times 50 - 20}$$

UCS = 386 kPa *(3)*

$$UCS = \frac{27(c/n)^{1.54}}{F}$$

$$c = n \times (UCS/27 \times F)^{(1/1.54)}$$

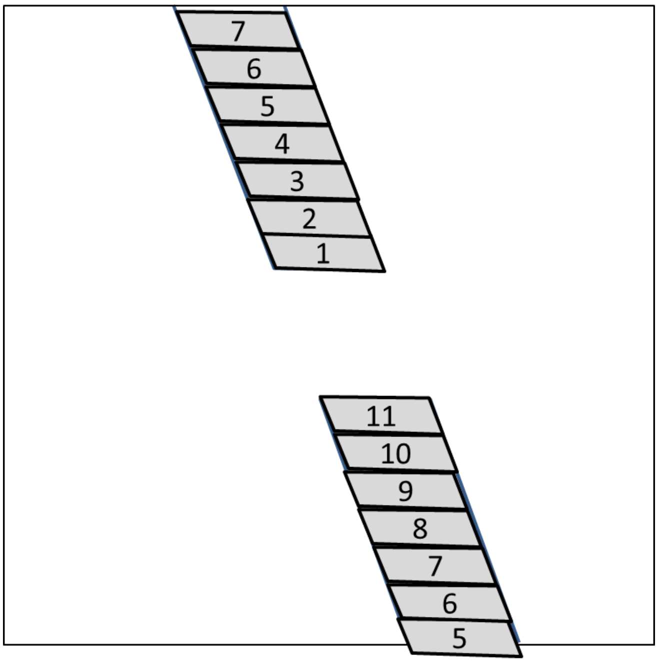
$$c = 50\% \times (0.386/27 \times 1.5)^{(1/1.54)}$$

c = 4.1% *(3)*

Q3.

a) *Sketch the stoping sequence that you think was mined. Use a realistic lift height in your drawing. Provide a brief explanation.* *[5]*

Lift height = 5m, reach of equipment.



b) *Calculate the virgin stress at this depth. Determine the rockwall condition factor under virgin stress conditions. Comment on the appropriateness of the installed support at the time of development.* *[5]*

$\sigma_V = \rho g h = 2600 \times 10 \times 600 = 15.6$ MPa

$\sigma_h = \sigma_V \times k = 15.6 \times 1.5 = 23.4$ MPa

RCF = $(3\sigma_1 - \sigma_3)/(F\sigma_c) = (3 \times 23.4 - 15.6)/(0.5 \times 150) = 0.7$ Good ground conditions (F=0.5 due to extreme jointing)

Yes, but perhaps mesh or shotcrete should have been used for highly jointed conditions

Table 6.4.1 Support recommendations for good ground conditions (RCF < 0.7).

Case	Primary support (Typical requirements)	Secondary support (Typical requirements)
Static stress conditions	'Spot' support where necessary (rock bolts, etc.)	'Spot' support where necessary (rock bolts, etc.). Light mesh where ground conditions are inherently poor.

c) *Did the elastic modelling include the haulage? Explain your answer.* [2]

No. The stress contours are not affected by the tunnel.

d) *Calculate the rockwall condition factor after mining and formation of the sill pillar. Sketch the anticipated damage in the haulage. What support should have been installed?* [10]

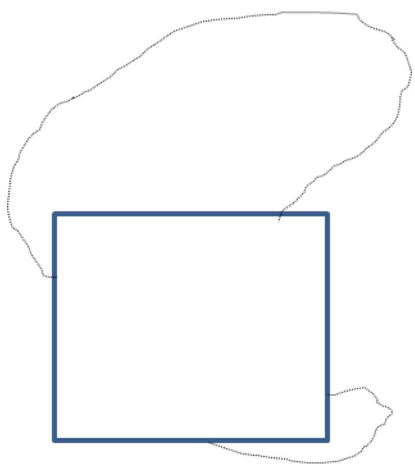
From results:

σ_1= 75 MPa

σ_3= 8 MPa

RCF = $(3\sigma_1 - \sigma_3)/(F\sigma_c)$ = (3x 75-8)/(0.5 x150) = 2.9. Poor ground conditions (F=0.5 due to extreme jointing)

Damage in crown or roof and floor. Slightly offset, due to orientation of σ_1



Select support for stress changes.

Table 6.4.3 Support recommendation for poor ground conditions (RCF > 1.4).

Case	Primary support (Typical requirements)	Secondary support installed by specialist crews close to face (Typical requirements)
Static stress conditions	(If necessary) shotcrete to face, then fully grouted tendons or steel ropes, length > 1.8 m on basic 1.5 m pattern, as close to face as possible. Support resistance: 80 - 110 kN/m^2.	Moderate strength steel wire mesh usually with additional tendons but may be integrated with primary support, optional shotcrete.
Stress changes anticipated	(If necessary) reinforced shotcrete to face, then fully grouted steel ropes or yielding tendons, length > 1.8 m on basic 1 m or double 2 m pattern, as close to face as possible. Support resistance: 120 - 230 kN/m^2.	Medium strength rope lacing and moderate strength wire mesh with additional tendons but may be integrated with primary support. Add integral shotcrete in long life tunnels. If necessary, additional hangingwall support comprising grouted steel ropes.
Seismic activity anticipated	(If necessary) reinforced shotcrete to face, then fully grouted steel ropes or yielding tendons, length > 2.3 m on basic 1 m pattern, as close to face as possible. Support resistance: 220 - 290 kN/m^2.	Strong rope lacing and strong, stiff wire mesh with additional tendons but may be integrated with primary support. Add integral shotcrete in long life tunnels. If necessary, additional hangingwall support comprising grouted steel ropes.

e) *What should have been done to the backfill to enable complete extraction of the sill pillar? [3]*

Strengthening and reinforcement of the bottom 10m of backfill in the bottom lift above the sill pillar. This is to increase the compressive and tensile strength of the backfill to enable mining beneath it. usually 10% binder and various arrays of mesh and support reinforcement.

Q4.

a) *Calculate the modified stability number (N') for the stope crown and walls for the two geotechnical domains. Present your answer in a table and list the stress values,* strength to stress ratio, critical joint set, *relevant angles, failure mechanism and the values of the A, B and C parameters. [10]*

UCS	100	MPa
Q'	11	

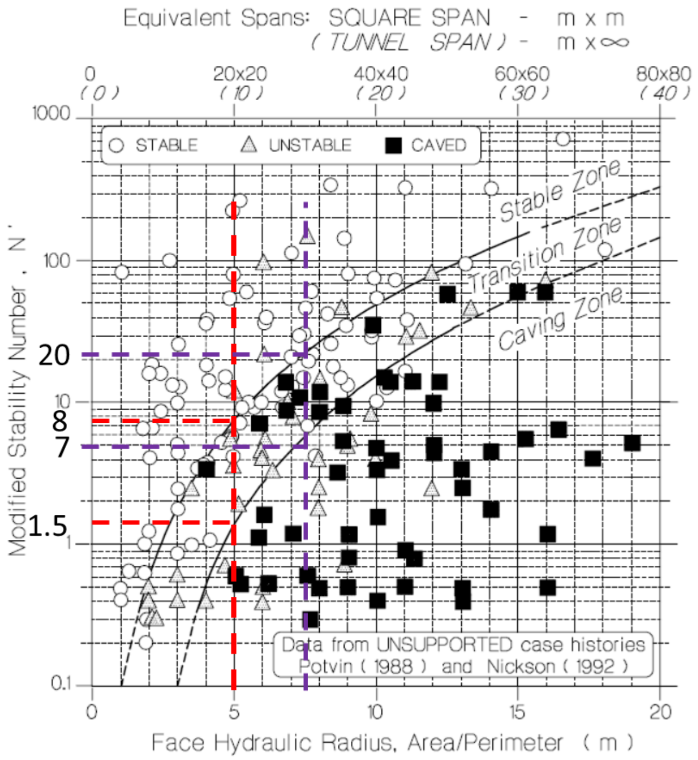
Geotechnical Domain	Excavation Surface	Stress	Stress ratio	A	Critical joint	Angle	B	Mechanism	Angle	C	N'
Domain 1	Crown	12	8.3	0.73	J1	0	0.3	Slabbing/G	0	2	4.8
	L Wall	3	33.3	1.00	J2	0	0.3	Slabbing	90	8	26.4
	R Wall	3	33.3	1.00	J2	0	0.3	Slabbing	90	8	26.4
Domain 2	Crown	12	8.3	0.73	J2	20	0.2	Slabbing/G	0	2	3.2
	L Wall	3	33.3	1.00	J1	20	0.2	Slabbing/G	90	8	17.6
	R Wall	3	33.3	1.00	J1	20	0.2	Sliding	70	4	8.8

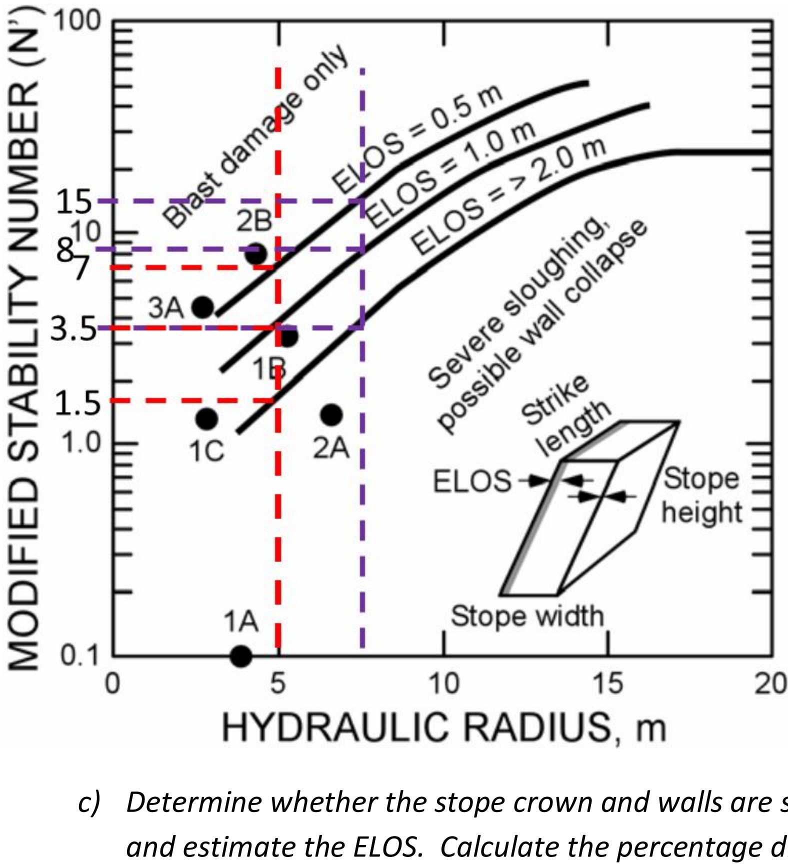
b) *Calculate the hydraulic radius for the crown and walls. Plot these values on the modified stability and ELOS graphs and determine the threshold N' values for the stability zones and ELOS values. Hand in the annotated graphs with your script. [5]*

Height	30	m
Width	15	m
Length	30	m

Treshold Values

	Hydraulic Radius	Stability Zone			ELOS		
		Stable	Transition	Caved	0.5	1.0	2.0
Walls	7.5	20	7		15	8	3.5
Crown	5	8	1.5		7	3.5	1.5





c) *Determine whether the stope crown and walls are stable, transitional or caving and estimate the ELOS. Calculate the percentage dilution and indicate what action is required, if any. Present your results and comments in a table.*

[10]

Note that the stope crown is on ore, so there is no dilution. Instability in the crown will affect the next stope.

Geotechnical Domain	Excavation Surface	Stability Zone	ELOS	Wall Dilution	Total Dilution	Action	Comment
Domain 1	Crown	Transition	1.0		7%	Cable Support	Affects next stope
	L Wall	Stable	0.5	3%		None	Dilution tolerable
	R Wall	Stable	0.5	3%		None	Dilution tolerable
Domain 2	Crown	Transition	2.0	13%	10%	Cable Support or reduce dimensions	Affects next stope
	L Wall	Transition	0.5	3%		None or possible cable support	Dilution tolerable
	R Wall	Transition	1.0	7%		Support if accessible from tunnel	Dilution moderate