CHAMBER OF MINES OF SOUTH AFRICA

MODEL ANSWER

SUBJECT: CERTIFICATE IN ROCK MECHANICS PAPER 3.3 : MASSIVE UNDERGROUND MINING (HARD AND SOFT ROCK)	**EXAMINER:** J WALLS **MODERATOR:** W JOUGHIN
SUBJECT CODE: COMRME	
EXAMINATION DATE: MAY 2015	**TOTAL MARKS:** [100]
TIME: 3 HOURS	**PASS MARK:** (60%)

QUESTION 1 – MINING METHODS

A high grade steep dipping orebody, 10 m wide is hosted in a highly jointed, weak rock mass. The ground surface and underground aquifers may not be disturbed due to environmental restrictions.

1.1 What mining method would you choose to extract this orebody and why would you use it in preference to other mining methods? (4)

1.2 Describe the mining method, layout and stope access with the aid of sketches. (12)

1.3 Discuss the purpose of sill pillars in this mining method. (4)

[20]

ANSWER 1

(This question is taken from previous papers: MAY, 2009 AND MAY, 2008).

1.1 Brady and Brown, 2004, Ch 12

The orebody is high grade; however due to the weak host rock mass, dilution will be an important consideration. *(1)*

Narrow width of the orebody precludes natural caving methods. *(1)*

Exposure area and duration of exposure of the highly jointed, weak host rock mass will need to be limited. *(1)*

Sub-level caving, sub-level open stoping and long-hole stoping will not achieve this. (why?) *(1)*

The excavation walls will need to be artificially supported with rock, support or backfill. *(1)*

Cut and fill stoping methods will be appropriate in this environment. *(1)*

(*Total possible=6*, Required=4)

1.2 Brady and Brown, 2004, Ch 12.4.3. *(10)*

12.4.3 Cut-and-fill stoping

In the most common form of cut-and-fill stoping, mining proceeds up-dip in an inclined orebody. (An alternative, less common, method involves down-dip advance of mining.) The progress of mining is linked to a closely controlled cycle, involving the sequential execution of the following activities:

a) drilling and blasting, in which a slice of rock, typically about 3 m thick, is stripped from the crown of the stope;

b) scaling and support, consisting of the removal of loose rock from the stope crown and walls, and the emplacement of lightweight support;

c) ore loading and transport, with ore moved mechanically in the stope to an ore pass and then by gravity to a lower transport horizon;

d) backfilling, when a layer of granular material, of depth equal to the thickness of the ore slice removed from the stope crown, is placed on the stope floor.

An important aspect of cut-and-fill stoping is that miners work continuously in the stope, and execute all production activities under the stope crown which is advanced by mining. The success of the method therefore involves achieving reliable control of the performance of rock in the immediate periphery of the work area. This is realised by controlled blasting in the stope crown, application of a variety of local rock support and reinforcement techniques, and more general ground control around the stope derived from the use of backfill.

Cut-and-fill stoping is applied in veins, inclined tabular orebodies and massive deposits. In the last case, the orebody is divided into a set of stope blocks, separated by vertical pillars, geometrically suitable for application of the method. It is suitable for orebodies or stope geometries with dips in the range 35◦–90◦, and applicable to both shallow and deep orebodies. Orebody or stope spans may range from 4 m to 40 m, although 10–12 m is regarded as a reasonable upper limit. The use of a stopescale support system (backfill) *renders cut-and-fill stoping suitable for low rock mass strength conditions in the country rock, but better geomechanical conditions are required in the orebody.*

Cut-and-fill stoping is a relatively labour-intensive method, requiring that the in situ value of the orebody be high. The ore grade must be sufficiently high to accommodate some dilution of the ore stream, which can occur when backfill is included with ore during loading in the stope. On the other hand, the method provides both flexibility and selectivity in mining. This permits close control of production grades, since barren lenses may be left unmined, or fragmented but not extracted from the stope. It is also possible to follow irregular orebody boundaries during mining, due to the high degree of selectivity associated with drilling and blasting operations.

Significant environmental benefits of cut-and-fill stoping are related to the use of backfill. In the mine internal environment, close control of rock mass displacements provides a ventilation circuit not subject to losses in fractured near-field rock. Similarly, maintenance of rock mass integrity means that the permeability and hydrogeology of the mine far field may be relatively unaffected by mining. The advantages of the method, for the mine external environment, include limited possibility of mining-induced surface subsidence. Reduction in surface storage of mined wastes follows from possible replacement of a high proportion of these materials in the stoping excavations. De-slimed mill tailings are particularly suitable for backfilling, since the material may be readily transported hydraulically to working areas. This eliminates the need for extra mine development for transfer of backfill underground.

The amount of stope development for cut-and-fill stoping is small, compared with open stoping. The reasons for this are that the production source is also the working site, and some access openings may be developed as stoping progresses. On the other hand, stope block pre-production development may be comparable with that required for open stoping. Cut-and-fill stoping can commence only after developing a transport level, ore passes, sill incline and sill drift, service and access raises or inclines, and return air raises.

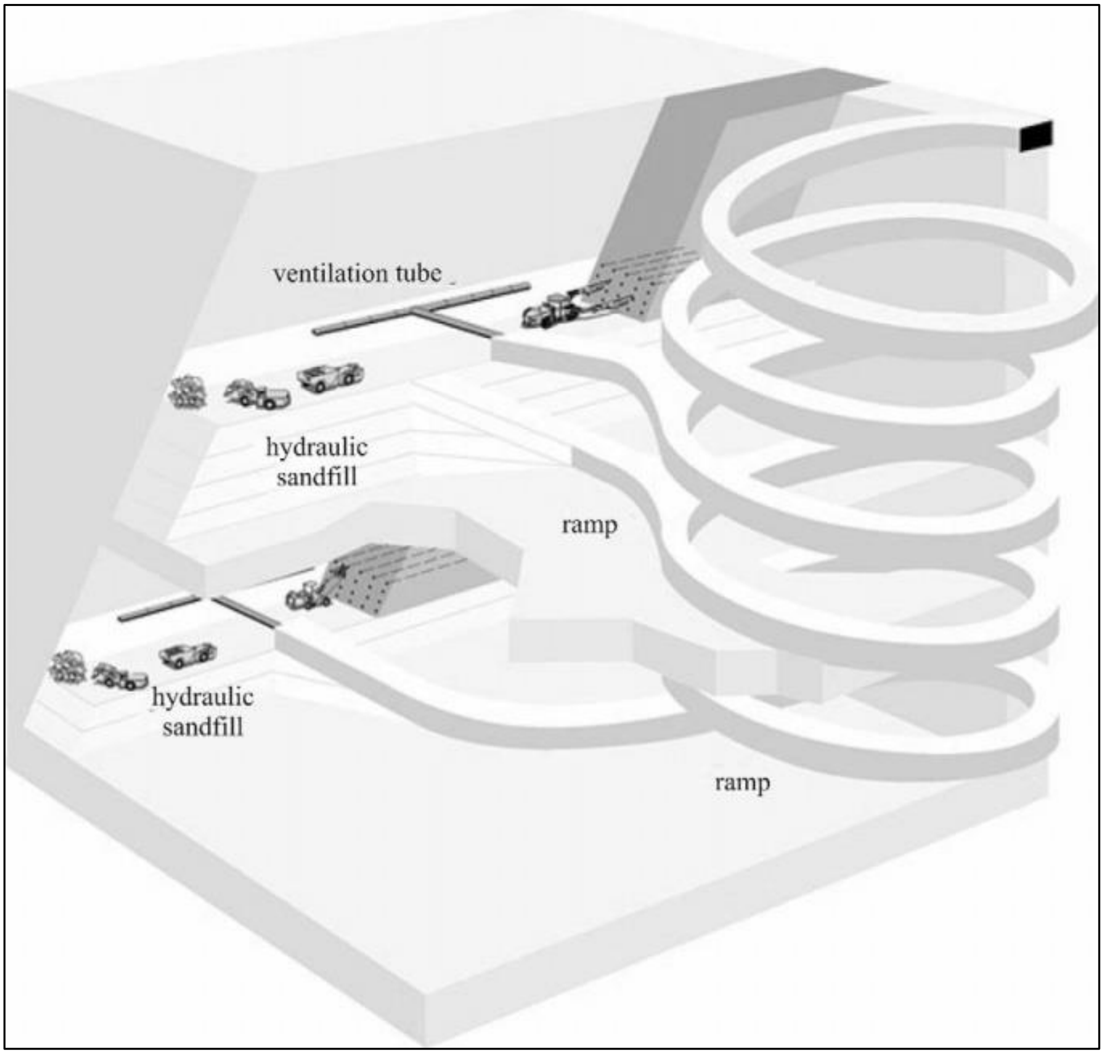
(*Sketch=4, Discussion=12, Total possible=16*. Required=12)

1.3 Brady and Brown, 2004, Ch 14 – background on stress in crown of c&f stope.

Sill pillars are usually left between levels, to protect the lower workings from rockfalls due to mining activities above (1); allow concurrent production on multiple levels (1); develop drawpoint infrastructure at the base of each stope (1), reduce the mining span (1) and control stability of hangingwall (and footwall) (1) (probably would require rib pillars or *backfill unless extremely competent), prevent large scale failures [(1)], which could damage infrastructure, prevent subsidence in shallow mines [(1)].*

Since the stope above has usually been mined out, the stress damage becomes excessive (protection from the mined-out stope above is required). Also when intersecting the stope above, failure of the backfill may occur, due to a combination of compressive stress damage and unravelling over the exposed span [(1)].

(*Total possible=8*, Required=4)



QUESTION 2– ROCKMASS BEHAVIOUR, SERVICE EXCAVATIONS

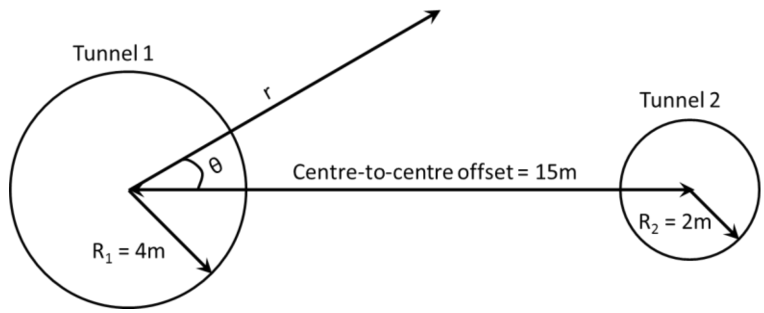
Off-reef drives are required to access the orebody in a massive operation.

2.1 Two horizontal access-ways will be positioned sub-parallel and within close proximity to the other. Assuming Tunnel 1 is extracted first, using the given information and the reference formulas provided, calculate the resultant radial, tangential and shear stresses at the position of the skin of Tunnel 2, before it is developed:

Depth below surface (tunnel centres) = 720 m

$k = \sigma_h / \sigma_v = 1.2$

$\rho = 3\ 200\ kg.m^{-3}$



(6)

2.2 Calculate the percentage change between the stresses due to Tunnel 1 and the virgin field stresses at the skin of Tunnel 2. Is this significant? Explain your answer. (5)

2.3 Development of Tunnel 1 is expected to result in a change in RCF from 0.6 to 1.0 in Tunnel 2. What strategies would you suggest to mitigate against the effects of stress change in Tunnel 2? (4)

2.4 You expect that the tunnel will be affected by seismicity during its lifetime. Discuss types of tendon support or any other support that you would propose to support the excavation. Draw an example of a yielding tendon (5)

[20]

ANSWER 2

(This question is new).

2.1 Ryder and Jager, 2002, Ch 4.2.1; Brady and Brown, 2004, Ch 7.2:

$$\sigma_{rr} = \frac{1}{2}(\sigma_h + \sigma_v)\left(1 - \frac{R^2}{r^r}\right) + \frac{1}{2}(\sigma_h - \sigma_v)\left(1 - 4\frac{R^2}{r^r} + 3\frac{R^4}{r^4}\right)\cos 2\theta$$

$$\sigma_{\theta\theta} = \frac{1}{2}(\sigma_h + \sigma_v)\left(1 + \frac{R^2}{r^r}\right) - \frac{1}{2}(\sigma_h - \sigma_v)\left(1 + 3\frac{R^4}{r^4}\right)\cos 2\theta$$

$$\tau_{r\theta} = -\frac{1}{2}(\sigma_h - \sigma_v)\left(1 + 2\frac{R^2}{r^r} - 3\frac{R^4}{r^4}\right)\sin 2\theta$$

Resultant stress of Tunnel 1 on Tunnel 2 (i.e. stress at the skin of excavation 2):

r = 15m – 2m = 13m [1]
$\theta = 0°$ [1]
σ_{rr} = 24.0 MPa [2]
$\sigma_{\theta\theta}$ = 24.9 MPa [2]
$\tau_{r\theta}$ = 0 MPa [2]

(*Total possible=8*, Required=6)

2.2 *percentage change in stress:*

Relate σ_v to $\sigma_{\theta\theta}$ and relate σ_h to σ_{rr} [1]

Vertical stress change at excavation 2: 10% [1 1/2]
Horizontal stress change at excavation 2: -12% [1 1/2]

The virgin field stress around Tunnel 2 will undergo a significant (>5%) stress change due to the development of Tunnel 1. [1]
The vertical stress will increase by 10% [1]
The horizontal stress will decrease by 12% [1]
Tunnel 2 will be situated within the zone of influence of Tunnel 1 [1]*.*

(*Total possible=7*, Required=5)

2.3 Ryder and Jager, 2002, Ch 7 (p 69) *Precautionary measures may include sequencing of the tunnel excavation (Tunnel 2 will have less influence on Tunnel 1, resulting in less than 5% stress change)* [1] *or increasing the offset between the tunnels* [1]*. Alternatively, pre-reinforcing of the tunnel side-walls to improve confinement of the slabbing (spalling) that will be expected* [1]*. This should include tendons (estimate the depth of fracturing due to the stress change) and areal coverage such as wire mesh and straps or lacing* [1]*. Shotcrete may be included to improve the mesh performance but should not be installed in isolation due to the low tensile strength of shotcrete which will be unable to withstand the bulking effects of a slabbing rockmass* [1]*. Steel arch sets could be considered.* [1]
Monitoring is not a precautionary measure, it is reactionary.
Primary support for 0.7<RCF<1.4: "spot" support where necessary (split sets, rock studs, etc) [1]

Secondary support for 0.7<RCF<1.4: fully grouted tendons >1.5m in length, installed on basic 2m pattern [1]*. Support resistance: 30-50kN/*m^2 [1]*. Rope lacing (sidewalls only?)* [1]*.*

(*Total possible*=*7*, Required=4)

2.4 *experiential knowledge*

yielding cables [1]*, yielding tendons* [1]*, yielding straps* [1]*, reinforced shotcrete (preferably fibre, not steel as steel is too rigid)* [1]*, wire mesh (not weld mesh – too rigid)* [1]*, lacing, yielding arches* [1]*; no rigid tendons* [1]*; grouted / resin bonds will break; however, this is acceptable provided that the rockmass is successfully arrested (confined) to minimise bulking. Illustration* [2]

(*Total possible*=*9*, Required=5)

QUESTION 3 – GEOTECHNICAL CHARACTERISATION AND SUPPORT FOR OPEN STOPING

The stability of open stopes depends on the quality of the rock, excavation dimensions and support installed. With reference to the Matthews/Potvin stability method illustrated in the figures below:

3.1 Describe how N' is determined. (7)

3.2 Describe the concept of hydraulic radius. (2)

3.3 Illustrate the maximum HR on the graphs provided for an unsupported and a supported excavation if N' = 5. Annotate the graphs and include them with your script. Give an example of the possible dimensions of such an excavation surface (4)

3.4 Draw and explain cable bolt reinforcing patterns for open stope backs/crowns and hanging walls (geological). (7)

[20]

ANSWER 3

(This question is taken from previous papers: OCT, 2010 AND OCT, 2009).

3.1 Hoek, Kaiser, Bawden, p188; Hutchinson and Diederichs, pg 221; Brady and Brown, 2004, pg 266, Ch 9

N' = Q' x A x B x C *(1)*

N' is determined for each wall of the stope.

Q' = RQD/Jn x Jr/Ja. Parameters are determined from underground mapping or core logging. *(1)*

The factor A is a measure of the ratio of intact rock strength to maximum compressive stress parallel to the wall in question (1). The value of A reduces as the stress approaches the values of the intact rock strength (1), to reflect the related instability due to rock yield. The intact rock strength is determined from laboratory testing (1/2). The virgin stress state is determined from stress measurements (1/2) and the compressive stress parallel to the wall can be determined using numerical modelling (1/2). *(2)*

The factor B is a measure of the relative orientation of dominant (critical) jointing w.r.t. the orientation of the excavation surface [(1)]. *The smaller the angle, the lower the factor and the less stable the wall* [(1)]. *The joint orientations should be determined from mapping or oriented core* [(1/2)]. *(2)*

The factor C is a measure of the influence of gravity on the stability of the wall (face) in question [(1)]. *Overhanging stope faces (backs) or walls with joints that will allow sliding into the excavation are weakest* [(1)]. *The excavation geometry and the orientation of structural weaknesses are required* [(1)]. *(2)*

(*Total possible*=*11*, Required=7)

3.2 *Hydraulic Radius = Area/Perimeter* [(1)]

It is a single parameter which describes the dimensions of an excavation and takes cognisance of the aspect ratio (effect of size and shape) [(1)]. *It is used together with an estimate of the rock mass characteristics (N') to determine the stability of excavations.* [(1/2)]

(*Total possible=2½*, Required=2)

3.3 *Graphical illustration unsupported* [(1)]*; supported* [(1)].

For N' = 5, HR ~ 4.2 (unsupported) and HR ~ 8.8 (supported).

Possible excavation dimensions:

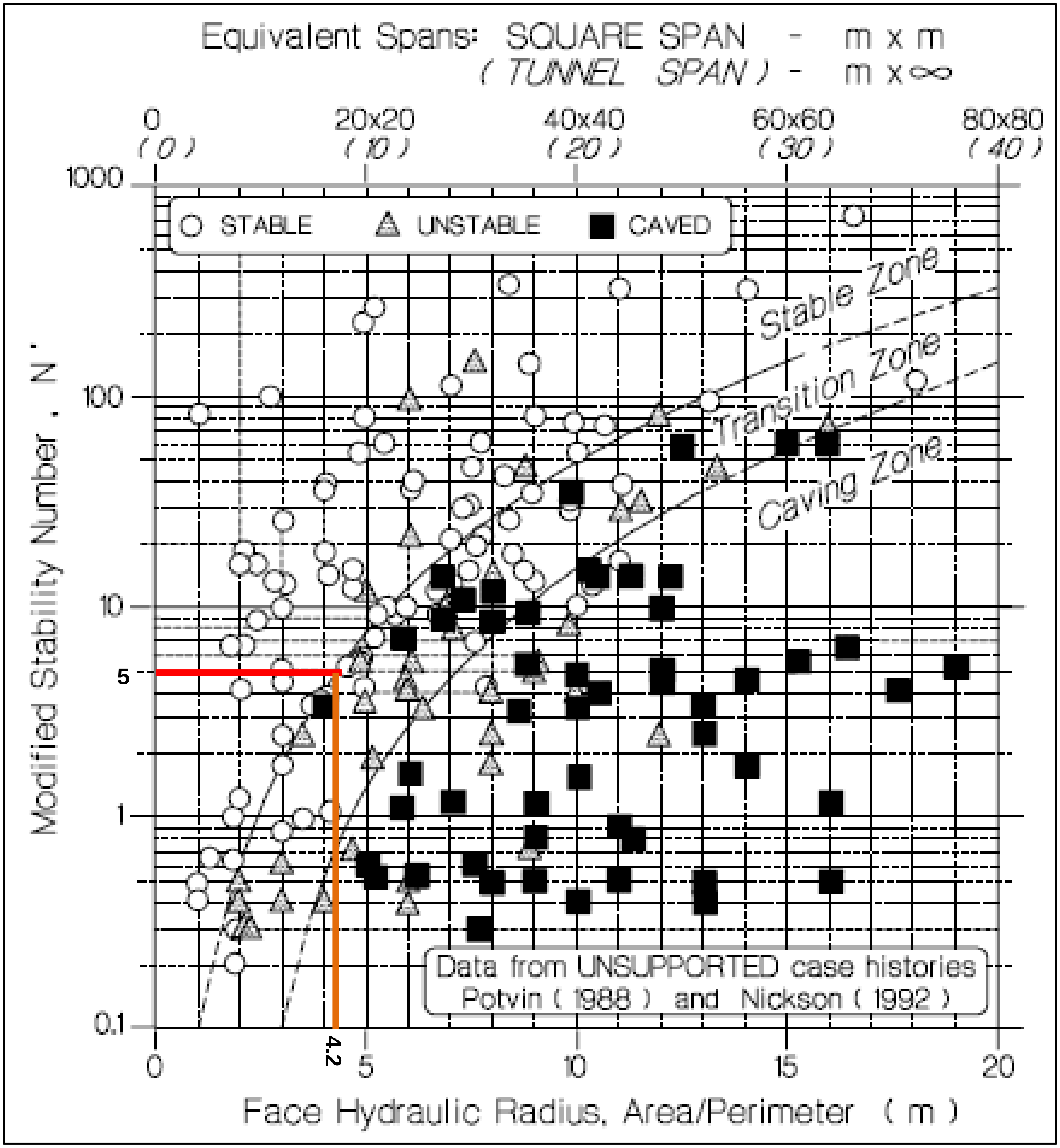
*a = 4*4.2 = 16.8 m (square excavation, unsupported)* [(1)],

*a = 2*4.2 = 8.4 m wide (infinitely long excavation, unsupported)* [(1)]

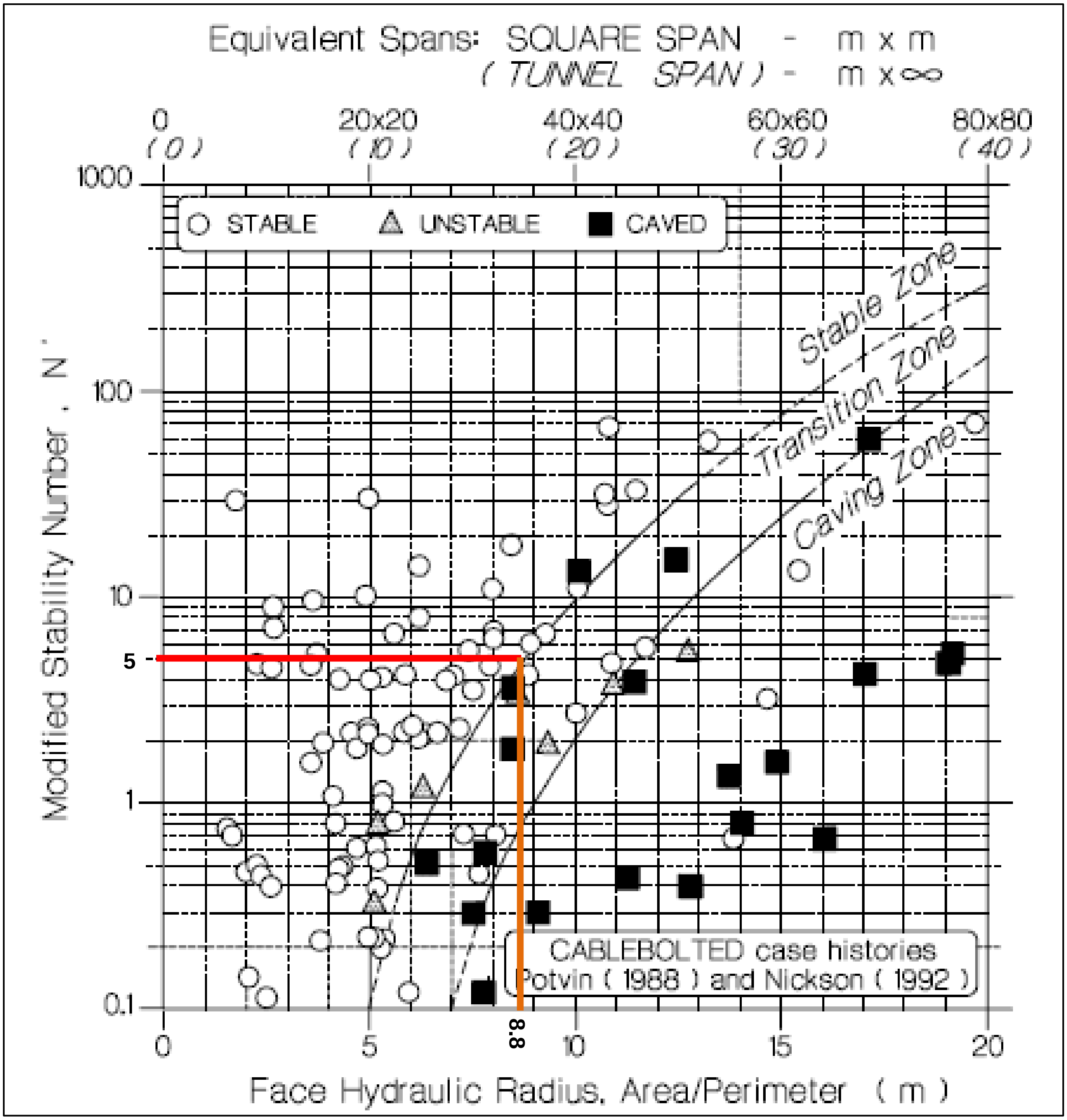
*a = 4*8.8 = 35.2 m (square excavation, unsupported)* [(1)],

*a = 2*8.8 = 17.6 m wide (infinitely long excavation, unsupported)* [(1)]

(Total possible=6, Required=4)



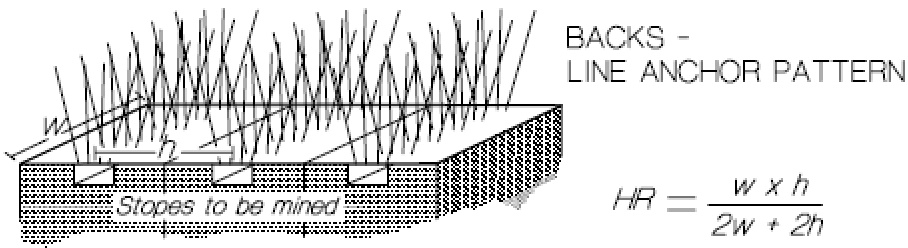
Equivalent Spans: SQUARE SPAN - m x m
(TUNNEL SPAN) - m x ∞
0
(0)
20x20
(10)
40x40
(20)
60x60
(30)
80x80
(40)
1000
100
10
5
1
0.1
STABLE
UNSTABLE
CAVED
Stable Zone
Transition Zone
Caving Zone
Data from UNSUPPORTED case histories
Potvin (1988) and Nickson (1992)
Modified Stability Number , N'
4.2
0
5
10
15
20
Face Hydraulic Radius, Area/Perimeter (m)



3.4 Hoek, Kaiser, Bawden, p188; Hutchinson and Diederichs, pg 221; Brady and Brown, 2004, pg 266, Ch 9

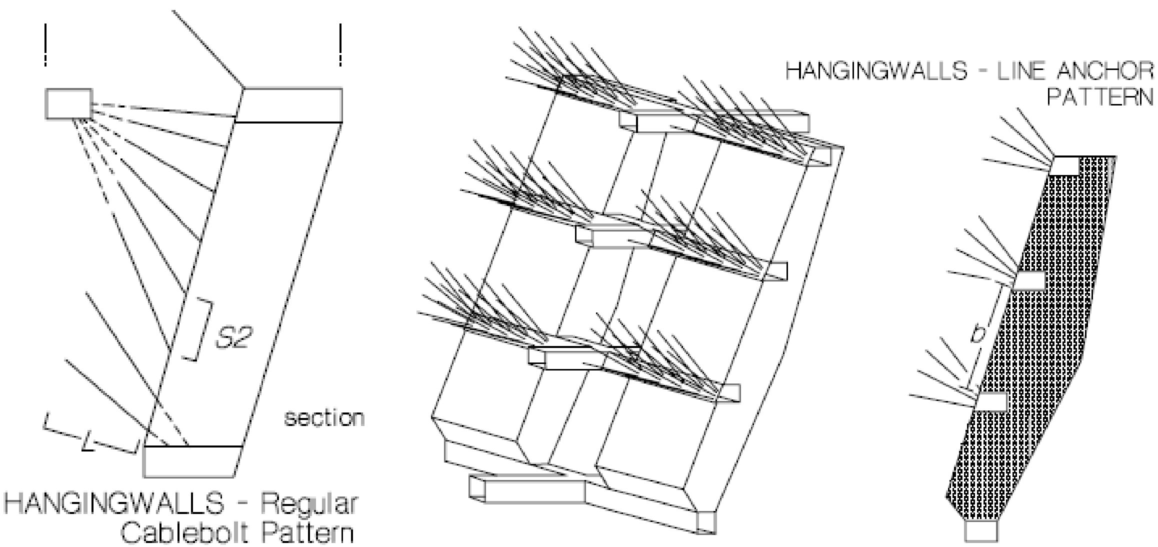
Open stopes can be supported using regular (uniform) arrays for the excavation as a whole or a line or point array where access is limited [(1)]. *Line or point arrays should only be used in rock-masses dominated by a single lamination parallel to the stope face or joint sets perpendicular to the face (few oblique joints)* [(1)]. *Cable spacing should be uniform* [(1)]. *Cable lengths are specified for cablebolts installed within 30°-40° of perpendicular to the stope face* [(1)]. *The length refers to the perpendicular distance between the face and the* *end of the cables [1]. Actual cable length will depend on the cable angle [1]. Neighbouring cables must be within 40° of being mutually parallel [1].*

The stope back can be supported from pre-developed drifts at the top of the stopes to be mined (illustration) [1]:



(2)

The hanging walls of open stopes can be supported either from a hangingwall drive [1] (a fan of cables is installed from the drive towards the stope) [1] or from a drive within the orebody [1] on the hangingwall side (a fan of cables is installed outwards from the drive) [1]. (illustration):



(2)

(Total possible=16, Required=7)

QUESTION 4 – SUBSIDENCE, MONITORING

Subsidence engineering is an important aspect of mine design.

4.1 Block caving results in discontinuous subsidence which may influence the surface. Discuss the orebody characteristics as well as and any other influencing factors that determine the extent of surface expression. (6)

4.2 Subsidence effects on surface are expressed in terms of tilt, horizontal displacement and strain. Define tilt, horizontal displacement and strain, giving the correct units of measurement for these quantities. (4)

4.3 Discuss the effects of tilt, horizontal displacement and strain on linear structures such as pipelines and conveyor belts. (4)

4.4 To monitor the surface expression of subsidence, you are required to record the principal strains across a surface element as shown in the illustration. The following information is available:

$l_A = 10\ m.$

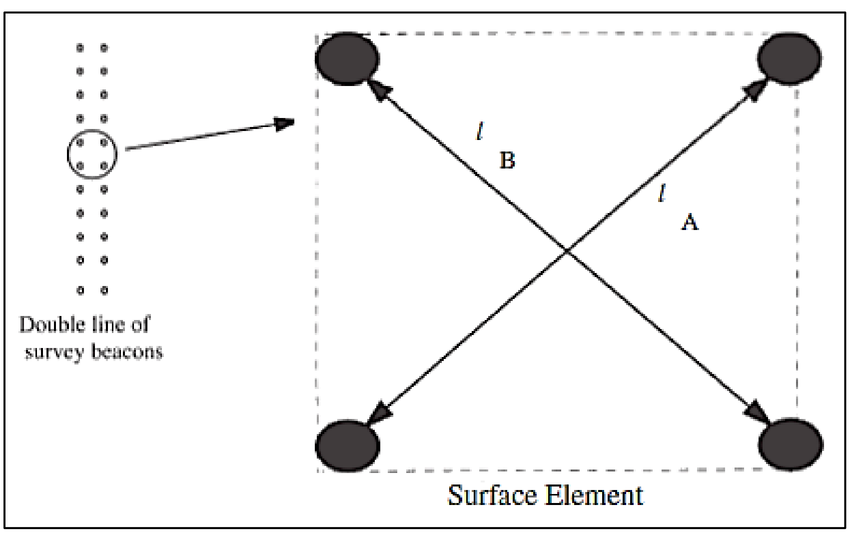
$l_A^1 = 9.6\ m.$

$l_B = 11\ m.$

$l_B^1 = 11.5\ m.$

$\phi = 15°.$

Use the reference equations provided to evaluate the major and minor principal strains. Evaluate the direction of ε_1 relative to the direction of ε_2. (6)



[20]

ANSWER 4

(This question is taken from previous papers: MAY, 2013 AND OCT, 2008).

4.1 Brady and Brown, 2004, pg 496, Ch 16

A vertical orebody with well-defined cut-off between the orebody and surrounding country rock will cave vertically to the surface with inclined surface slopes forming only in weak or weathered surface layers (1). Other influencing factors include:

- *dip of the orebody (1)*
- *shape of the orebody (plan section) (1)*
- *strength of the orebody (RMC, UCS) (1)*
- *strength of the surrounding country rock (RMC, UCS) (1)*
- *strength of the overburden or cap rock (RMC, UCS) (1)*
- *presence of major structural features, e.g. faults and dykes intersecting the cap rock that may contribute to, or inhibit cave propagation and associated subsidence (1)*
- *depth of mining defined by the undercut level and associated* in situ *stress field (1)*
- *slope of the ground surface (1)*
- *any prior surface mining (1)*
- *placement of fill in pre-existing or newly produced crater (1)*
- *nearby underground excavations (1)*

(Total possible=12, Required=6)

4.2 v d Merwe and Madden, 2002, p117, p214

Tilt is defined as the relationship between the amount of vertical subsidence between two points and the horizontal linear displacement between the same two points (1), measured in mm/m (1).

$$\boldsymbol{T} = \frac{\Delta \boldsymbol{S}}{\boldsymbol{d}}$$

$$\Delta \boldsymbol{S} = \boldsymbol{S_i} - \boldsymbol{S_{i+1}}$$

$$\boldsymbol{d} = |\boldsymbol{x_i} - \boldsymbol{x_{i+1}}|$$

Horizontal displacement occurs when a point on the surface does not follow a straight line to its final position, measured in m (1). Horizontal displacement may occur in a straight line, a curved arc or a figure of eight (1).

Strain is defined as a change in length relative to the original length, more correctly described as "deformation" in the context of subsidence (1), measured in mm/m (1). Strain *may be compressive or tensile [(1)]. The transition from tensile to compressive strain occurs at the position of maximum tilt [(1)].*

(Total possible=8, Required=4)

4.3 v d Merwe and Madden, 2002, p125

Increased tilt on a conveyor belt can result in slippage, backup and loss of the transported material [(1)].

Horizontal displacement results in deviation from the designed alignment and the belt will not run true on the rollers, resulting in spillage of the transported material [(1)].

Strain may be either compressive or tensile. Compressive strain could result in buckling of the conveyor [(1)], with associated loss in transported material, while tensile strain may cause structural damage and increase tension in the conveyor [(1)].

Similarly, with rigid pipelines, tilt can cause a pipe to pinch, damaging the pipe material [(1)].

Horizontal displacement can cause shear and rupture (tearing) of the pipeline, particularly in a buried service [(1)].

(Total possible=6, Required=4)

4.4 v d Merwe and Madden, 2002, p215

$$\varepsilon_A = \frac{l_A - l_A^1}{l_A} = \frac{10-9.6}{10} = 0.040 \; ^{(1)}$$

$$\varepsilon_B = \frac{l_B - l_B^1}{l_B} = \frac{11-11.5}{11} = -0.045 \; ^{(1)}$$

$$\Gamma = \frac{1}{2}\tan\phi = \frac{1}{2}\tan 15 = 0.134 \; ^{(1)}$$

$$\varepsilon_{1,2} = \frac{1}{2}\left(\varepsilon_A + \varepsilon_B\right) \pm \sqrt{\Gamma^2 + \frac{1}{4}\left(\varepsilon_A - \varepsilon_B\right)^2}$$

$$\varepsilon_1 = \frac{1}{2}(0.040 - 0.045) + \sqrt{0.134^2 + \frac{1}{4}(0.040 + 0.045)^2}$$

$$= -0.0025 + \sqrt{0.017949 + 0.00180625} = 0.138$$

$$= 138\frac{mm}{m}\ (compressive)$$

(2)

$$\varepsilon_2 = \frac{1}{2}\ (0.040 - \ 0.045)\ - \ \sqrt{0.134^2 + \frac{1}{4}\ (0.040 + \ 0.045)^2}$$

$$= \ -0.0025\ - \ \sqrt{0.017949 + 0.00180625} = -0.143$$

$$= -143\,\frac{mm}{m}\ (tensile)$$

(2)

$$\tan 2\theta = \ \frac{2\ \Gamma}{\varepsilon_A - \ \varepsilon_B}$$

$$\theta = \frac{1}{2}\arctan\frac{2\ \Gamma}{\varepsilon_A - \ \varepsilon_B} = \frac{1}{2}\arctan\frac{2 * 0.134}{0.040 - (-0.045)} = \frac{1}{2}\arctan 3.152$$

$$= 36.2°\ (direction\ of\ \varepsilon_1\ relative\ to\ \varepsilon_2$$

(2)

(Total possible=9, Required=6)

QUESTION 5 –SEISMICITY

Seismic activity provides useful data relating to cave propagation while presenting a significant safety and stability hazard in cut and fill and open stoping operations.

5.1 Discuss the hardware components of a seismic monitoring network. (5)

5.2 Draw a typical seismogram, indicating the P- and S- arrivals (3)

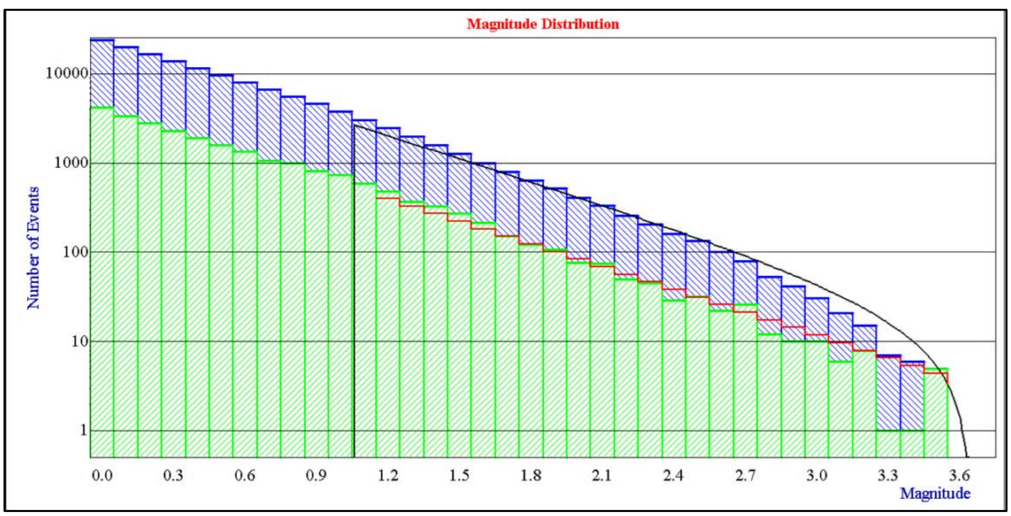
5.3 Refer to the diagram and data provided below for an operation.

5.3.1 Discuss the seismic hazard estimation provided in the given data below. What is the distribution called? What limitations are there to using this approach? (6)

5.3.2 How many events per month above magnitude M = 2.0 are in the database? (1)

5.3.3 What is the average recurrence period for events above magnitude M = 2.0? (1)

5.3.4 What is the probability for an event above magnitude M = 2.0 to occur within 3 months? (1)



Mean Return Periods					Probabilities of Seismic Event						
Mag.	Number of Events /Month	/Year	Return Period months	years	Mag.	1month	3months	6months	1 year	2 years	5 years
1.5	6.89	82.7	0.145	0.01	1.5	99.9%	100.0%	100.0%	100.0%	100.0%	100.0%
2.0	2.53	30.3	0.396	0.03	2.0	92.0%	99.9%	100.0%	100.0%	100.0%	100.0%
2.5	0.89	10.7	1.125	0.09	2.5	58.9%	93.1%	99.5%	100.0%	100.0%	100.0%
3.0	0.27	3.3	3.649	0.30	3.0	24.0%	56.1%	80.7%	96.3%	99.9%	100.0%
3.5	0.04	0.5	23.176	1.93	3.5	4.2%	12.1%	22.8%	40.4%	64.5%	92.5%

5.4. Explain the term "PPV". Discuss how PPV is related to support design. (3)

[20]

ANSWER 5

(This question is wholly new).

5.1 Jager and Ryder, 1999, Ch 2.5.1

A standard seismic network comprises a minimum of four, but in practice at least six to 30 operational geophones, arrayed around an area of interest. For acceptable vertical location accuracies, a minimum number of these must be situated well below and/or above the reef horizon. Geophones detect *ground velocities* incident from a seismic event. Signals are transmitted over telephone wires or radio links and storage and processing of data are carried out on central mini- or micro-computer systems.

Geophones (1) with three orthogonal directional sensors (1)

Minimum of three (ultimate minimum), recommend minimum four (1)

Seismometer (1)

Location accuracy – vertical vs horizontal array (1)

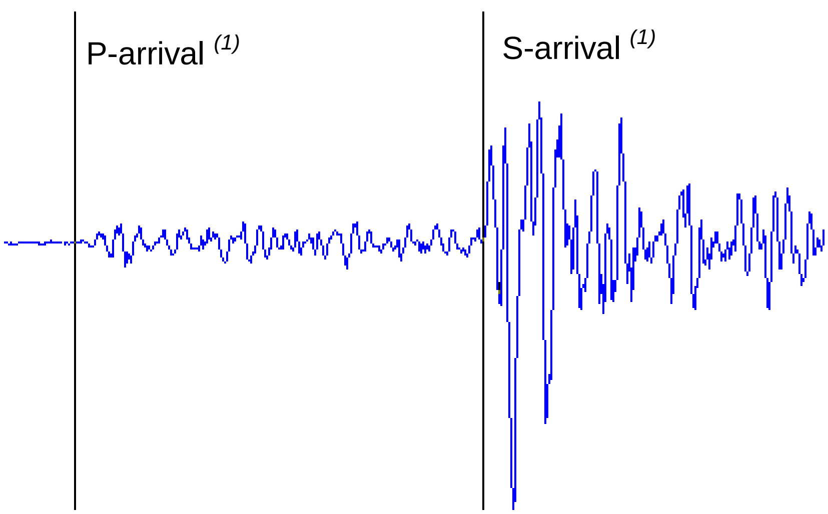
Area of interest (1)

Communication link – wires, radio, modem hub (1)

Storage centre (computer) and processing hub (1)

(Total possible=8, Required=5)

5.2 Ryder and Jager, 2002, Ch 5



(1)

(Total possible=3, Required=3)

5.3 Jager and Ryder, 1999, Ch 9, Mendecki, 1997, Ch 10.2

5.3.1 Frequency-magnitude (1) (Gutenberg-Richter (1)) distribution; Power-law (1) event size distribution, Used to estimate mean recurrence (1) intervals over a certain period (1) of observation; The given data does not specify the given period (1), Used to indicate M_{max} (1); The most significant deviations of observations from the F-M relation are at the largest observed magnitudes (1); Recurrence times beyond the timespan of the data set should be treated with caution (1); Due to larger magnitude events being fewer in the dataset, the upper truncation is poorly defined (1); Period of observation determines the data contained in the dataset (1).

(Total possible=10, Required=6)

5.3.2	*2.5*	*(1)*
5.2.3	*0.4 months*	*(1)*
5.2.4	*99.9% probability*	*(1)*

5.3 Ryder and Jager, 2002, Ch 5, Ch 6, Mendecki, 1997, Ch 1

Peak particle velocity (1) (PPV) is a measure of strong ground motion (1). PPV is a measure of the velocity of displacement of the rock mass / rock surface at a particular point in space at a given distance from the source of the event (1).

PPV dissipates with increasing linear distance from the event source (1)

Energy absorption requirements (1) of a support system are related to the downward hanging-wall displacement and a function of the square of the PPV (1).

(Total possible=5, Required=3)

Extras:

PPV in the near-field is about 1/10 of the shear stress drop taking place on the rupturing plane (1).

PPV in the far-field is a function of distance from the source, rock density, corner frequency, seismic moment and S-wave radiation pattern (1)

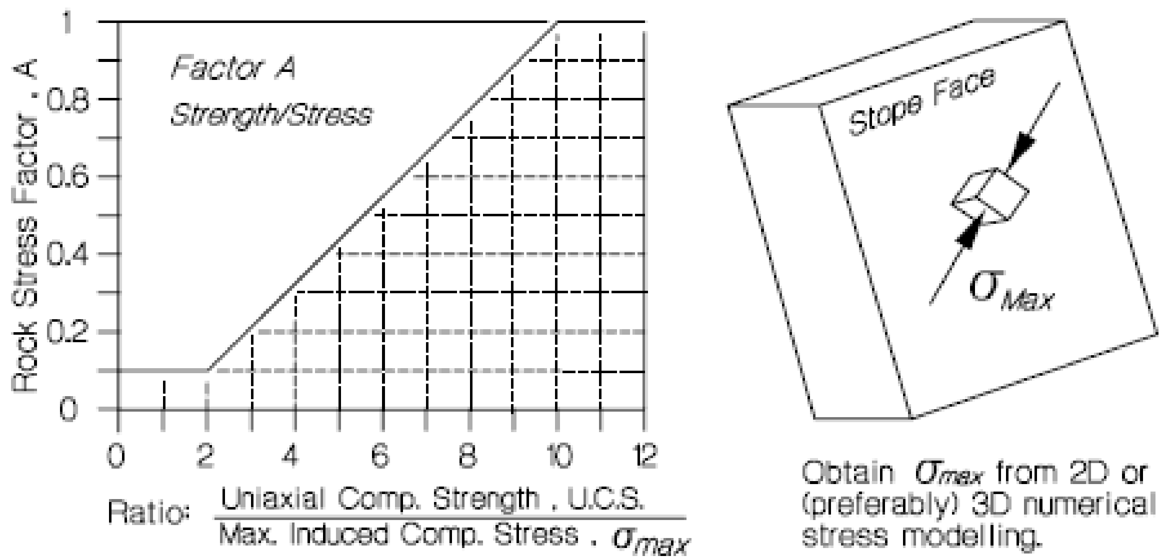
PPV is a function of the S-wave radiation pattern (transverse motion) and not P-wave radiation (compressive) as the S-wave motion is associated with the greatest amplitude, which is relevant for sensor specifications, support specifications and any other construction considerations (1).

TOTAL MARKS: [100]

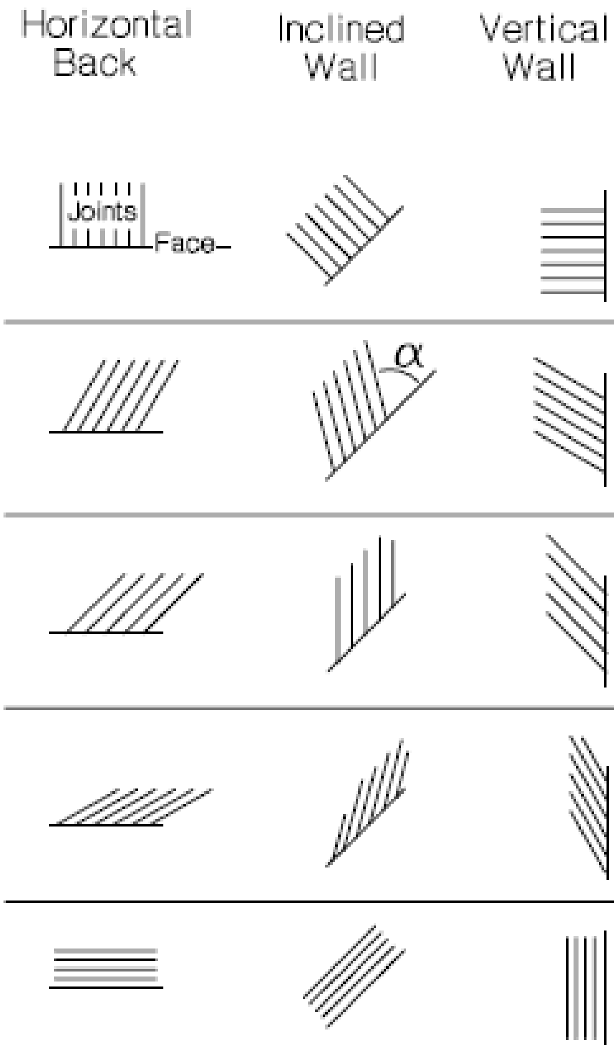
REFERENCES / FORMULAE

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

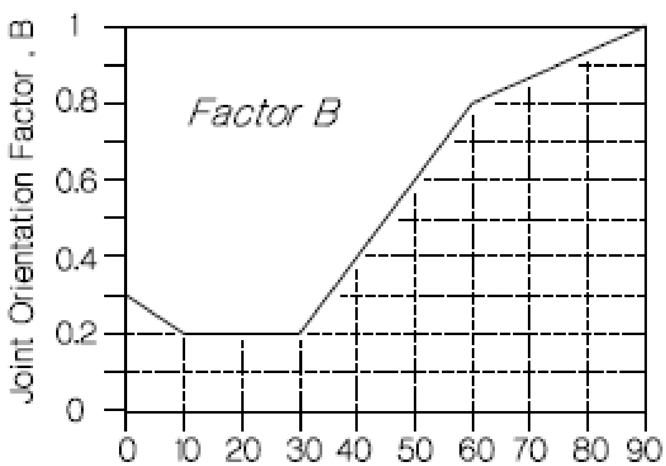
Determine maximum induced tangential stress (compressive) acting at the centre centre of the stope face being considered. Obtain uniaxial compressive strength strength for the intact rock. Evaluate Stress Factor, *A*, using the graph below:



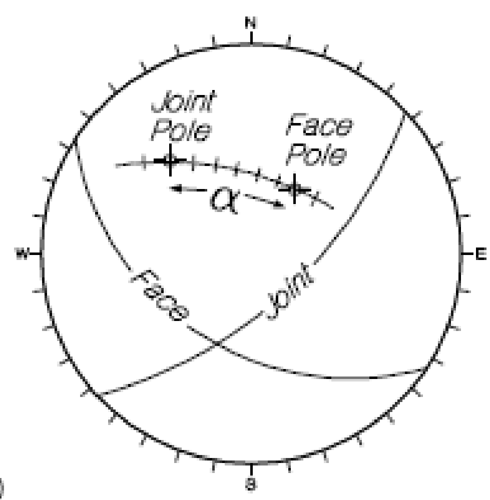
Obtain σ_{max} from 2D or (preferably) 3D numerical stress modelling.



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

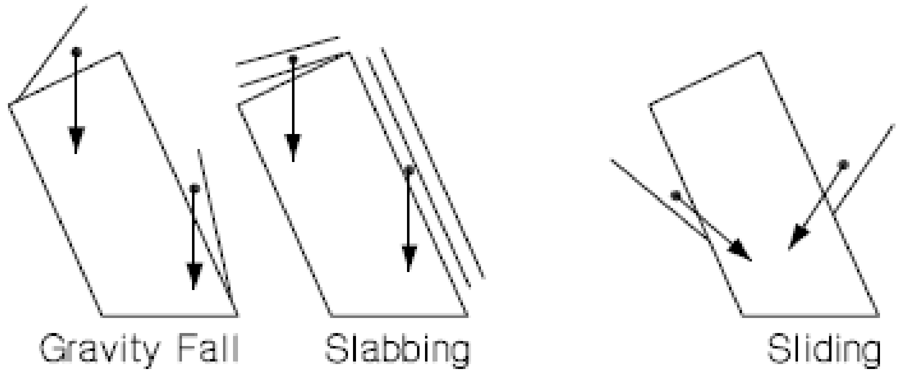


True Angle Between Face and Joint
(Angle α Between Poles)

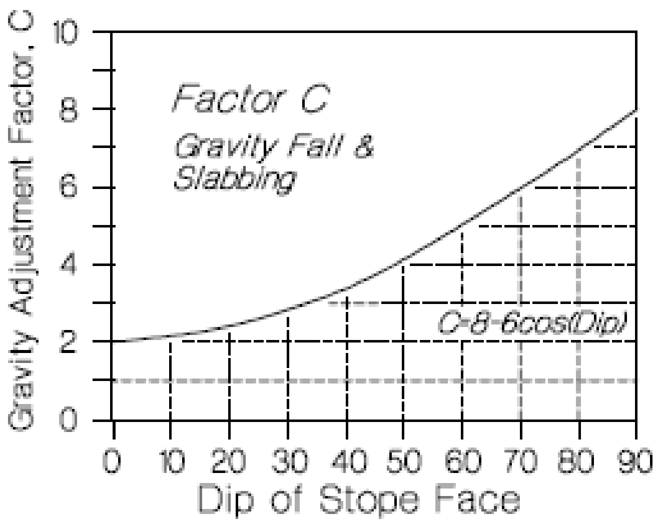


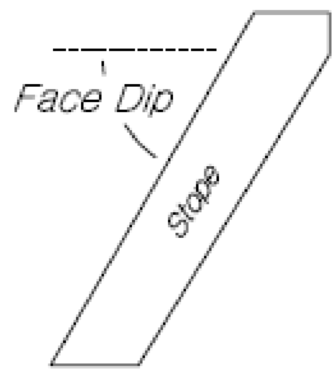
Determination of the minimum or true angle between two planes = Angle α between poles

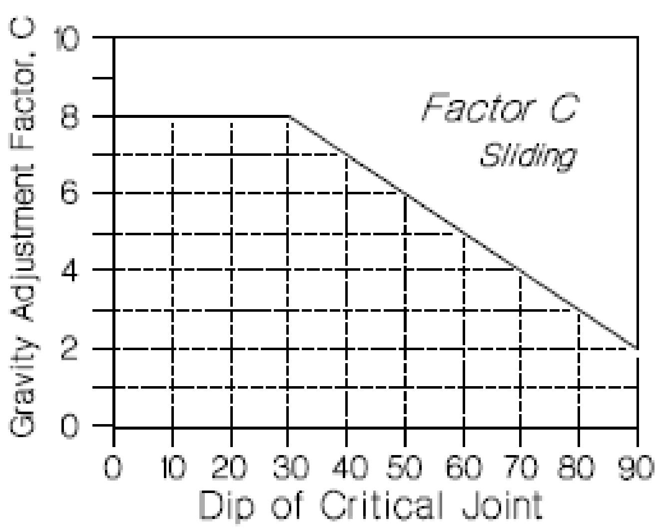
1) Determine the most likely mode of structural failure in case study using the figures below:

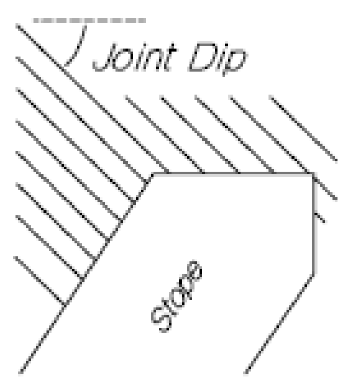


2) Next determine the gravity adjustment factor, C, based on the failure mode using the appropriate chart below.









Kirsch equations:

$$\sigma_{rr} = \frac{1}{2}\,(\sigma_h + \sigma_v)\left(1 - \frac{R^2}{r^r}\right) + \frac{1}{2}\,(\sigma_h - \sigma_v)\left(1 - 4\frac{R^2}{r^r} + 3\frac{R^4}{r^4}\right)\cos 2\theta$$

$$\sigma_{\theta\theta} = \frac{1}{2}\,(\sigma_h + \sigma_v)\left(1 + \frac{R^2}{r^r}\right) - \frac{1}{2}\,(\sigma_h - \sigma_v)\left(1 + 3\frac{R^4}{r^4}\right)\cos 2\theta$$

$$\tau_{r\theta} = -\frac{1}{2}\,(\sigma_h - \sigma_v)\left(1 + 2\frac{R^2}{r^r} - 3\frac{R^4}{r^4}\right)\sin 2\theta$$

Tilt:

$$T = \frac{\Delta S}{d}$$

$$\Delta S = S_i - S_{i+1}$$

$$d = |x_i - x_{i+1}|$$

Subsidence (strain):

$$\varepsilon_A = \frac{l_A - l_A^1}{l_A}$$

$$\varepsilon_B = \frac{l_B - l_B^1}{l_B}$$

$$\varepsilon_{1,2} = \frac{1}{2}\,(\varepsilon_A + \varepsilon_B) \pm \sqrt{\Gamma^2 + \frac{1}{4}\,(\varepsilon_A - \varepsilon_B)^2}$$

$$\Gamma = \frac{1}{2}\tan\phi$$

$$\tan 2\theta = \frac{2\,\Gamma}{\varepsilon_A - \varepsilon_B}$$

$$l_A = distance\ across\ first\ diagonal\ before\ subsidence$$

$$l_A^1 = distance\ across\ first\ diagonal\ after\ subsidence$$

$$l_B = distance\ across\ second\ diagonal\ before\ subsidence$$

$$l_B^1 = distance\ across\ second\ diagonal\ after\ subsidence$$

$$\varepsilon_{1,2}\ are\ the\ major\ and\ minor\ principal\ strains, respectively$$

$$\varepsilon_{A,B}\ are\ the\ strains\ in\ the\ A\ and\ B\ directions, respectively$$

$$\emptyset = angular\ distortion$$

$$\theta = direction\ of\ \varepsilon_1\ relative\ to\ \varepsilon_2$$