MODEL ANSWER
PAPER 3.3
OCTOBER 2014

QUESTION 1

To determine design criteria for stable underground excavations and select the most suitable mining method for economic extraction of an orebody, the orebody and host rock characteristics, amongst other parameters, must be investigated.

1.1 Describe the geotechnical parameters required for rock mass classification using Laubscher's Mining Rock Mass Rating system. (3)

1.1 The following geotechnical parameters are applicable:

Laubscher RMR = IRS + FF + Jc
Or
Laubscher RMR = IRS + RQD rating+ Js + Jc (1)

- 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)

(**4 marks available, 3 required**)

1.2 Describe a site investigation program that you would undertake to obtain the data and how the parameters are applied to the classification system. (10)

1.2 The following approaches are applicable:

- Geological drilling programme to be defined by geologist. Geological logging and orebody definition by geologist. (1)
- 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. (2)
- Mapping of rock exposures where available (eg open pit or outcrops). Joint orientations and characteristics, rock mass classifications (1)
- Lausbscher macro roughness (large scale) and micro roughness (small scale) are determined from relevant tables (1)
- Joint wall alteration and infill characteristics must be determined. These are used to determine Laubscher's Jc from relevant tables. (1)
- Laboratory testing of intact rock (UCS, TCS, shearbox on joints) (1)
- UCS is used to determined Laubscher's IRS using the relevant table. (1)
- Joint water can be determined from observed seepage in underground exposures, drilling records and ground water investigations. This is applied in Laubschers Jc 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's weathering adjustment (1)
- The expected stress state can be determined from stress measurements and numerical analyses. This is applied in Laubscher's stress adjustment (2)

- Orientation adjustment based on orientation of excavation with respect to joint set data. Joint set orientations can be determined from exposure mapping or orientated core logging (1)
- Blasting adjustment can be determined from the blasting program (1)

(**15 marks available, 10 required**)

1.3	Discuss any other factors that influence selection of a suitable mining method.	(3)

<u>1.3</u> Other factors that influence the selection of a suitable mining method are:

- Geometric configuration of the orebody (Brady & Brown, Chapter 12) – veins, lodes, tabular, massive (seam, placer or stratiform) – geological origin (1)
- Disposition and orientation – dip, dip direction, depth below surface, shape, continuity
- Size – absolute and relative dimensions (1)
- Geomechanical setting (1.1) and (1.2) (0)
- Engineering environment – groundwater, topography, surface constraints, ventilation and radiation (environmental) constraints (1)
- Grade distribution and absolute grade (1)
- Mechanical and labour resources (1)

(**5 marks available, 3 required**)

1.4	How you would apply the parameters obtained in (1.1) to (1.3) to the selected approach? Discuss any assumptions you must make and how this affects the design.	(4)

<u>1.4</u> Parameters can be applied as follows:

- Empirical estimation of cavability using Laubscher's tables and selection of a mining method based thereon. (1)
- Empirical evaluation of tunnels and service excavation stability and support requirements. (2)
- Based on stress orientation data, orientation of tunnels and excavations favourable and unfavourable to tunnelling and stoping (caving) respectively (1)
- Empirical evaluation of rock mass strength (stress:strength ratio) and pillar strength factors (1)
- Where there is limited information, further testing and investigation should be undertaken
- Assumptions may need to be made wrt dominant joint set orientations if orientated core or exposures are not available for mapping. This will affect the placement (orientation) of large excavations which in turn can lead to serious disruptions during execution if not clarified sufficiently far in advance.
- Assumptions may need to be made wrt to the principal field stresses (orientation and magnitude) due to insufficient resources during the pre-feasibility or feasibility phase. This affects design of large excavations and estimation of cavability which in turn affects selection of the mine design and must be clarified early in the study.
- Comment on excavation stability in low/high stress environment as appropriate.
- Failure criteria for numerical analysis may need to be assumed. These must be determined through laboratory testing (Poisson's ratio, Young's modulus) as required.

(5 marks available, 4 required)

<u>TOTAL MARKS: [20]</u>

QUESTION 2

A massive, near-vertical copper orebody has been extracted to a depth of 600 m by surface mining. Underground extraction is planned by cave mining where the undercut is situated at 1 050 m below surface. A horizontal section through the orebody is roughly oblong, spanning approximately 150 m in width and 600 m in length. The orebody is fairly well jointed with a Mining Rock Mass Rating (MRMR) approximately 50, hosted within slightly less jointed pyroxenite with an MRMR approximately 70.

2.1 Determine the cave angle and the extent of the zone of failure at 800 m below surface. Comment on the stability of an access ramp that approaches to within 100 m from the limit of the orebody at this depth? Illustrate your answer using a sketch. (7)

2.1 The following information and methodology apply: **(7 marks available, 7 required)**

Given:
Host rock pyroxenite MRMR = 70
Depth of base of block cave = 1 050 m

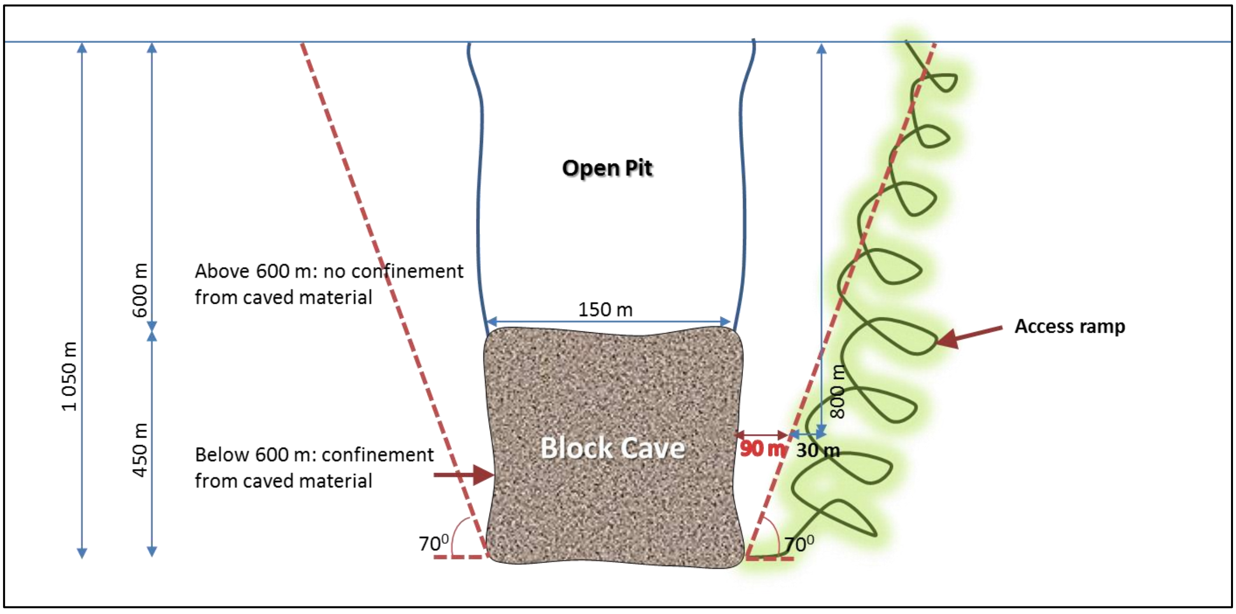
Determine the Cave Angle: (Laubscher, 1990)
Lateral confinement is provided by the broken ore in the cave
Cave angle = 70° (read off table) (1)

Extent of failure zone
We are interested in the lateral (horizontal) extent of failure at depth (>500 m), i.e. underground at 800 m.
Extent = 30 m (read off table) (1)

Horizontal distance from limit of orebody to a position at 800 m below surface:
D = 250 / tan 70° + 30 m = 90 m + 30 m = 120 m (2)

Stability of the access ramp:
The access ramp will be situated within the failure zone. (1)

Illustrative sketch



(2)

2.2 Discuss how stress-driven damage can be averted on the production level by strategic sequencing of the undercut. Use sketches to illustrate your answer (7)

2.2 Describe and sketch either pre-undercut or advanced undercut:

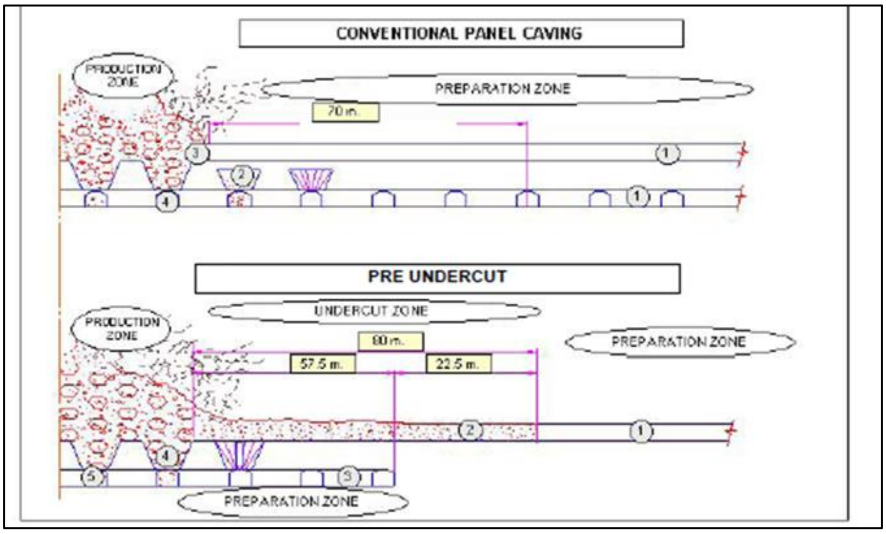
(Brady and Brown pg. 472 to 474. Brown (Block caving handbook) pg 193 to 195)
NOTE: Post-undercut (conventional undercut) does not apply because in this case the undercut level is developed to follow-behind the production level. This is only applicable in low-stress environments.

Note: at 1 050 m below surface this is no longer a shallow operation. Moderate stresses apply, and may be high stresses.

Pre-undercut: (**8 marks available, 7 required**)

- Undercut is mined ahead of extraction level development (variant of advance undercutting. (1)
- Minimum horizontal offset between extraction level and undercut is often the separation distance between the two levels – sometimes referred to as the 45^0 rule. (1)
- Provision for higher stresses should be made, i.e. larger offsets may be necessary. (1)
- Even with the 45^0 rule, the extraction level excavations may not be in a **full stress shadow zone** or may be subjected to some **abutment stress concentration**.
- In high stress fields, the extraction level may be placed under distress.
- **The extraction level is developed in a de-stressed environment** (1)
- The **undercut can be mined independently of the extraction level** (mining consideration)
- **Support and reinforcement requirements on the extraction level are generally lower than for post-undercutting** (1)
- **Broken ore in the undercut level acts as rock fill reducing and redistributing the abutment loads on the undercut face to a measure**. (Note: this can have the inverse effect if blasted ore on the undercut level becomes compacted and applies "remnant" type stresses on the extraction development). (1)

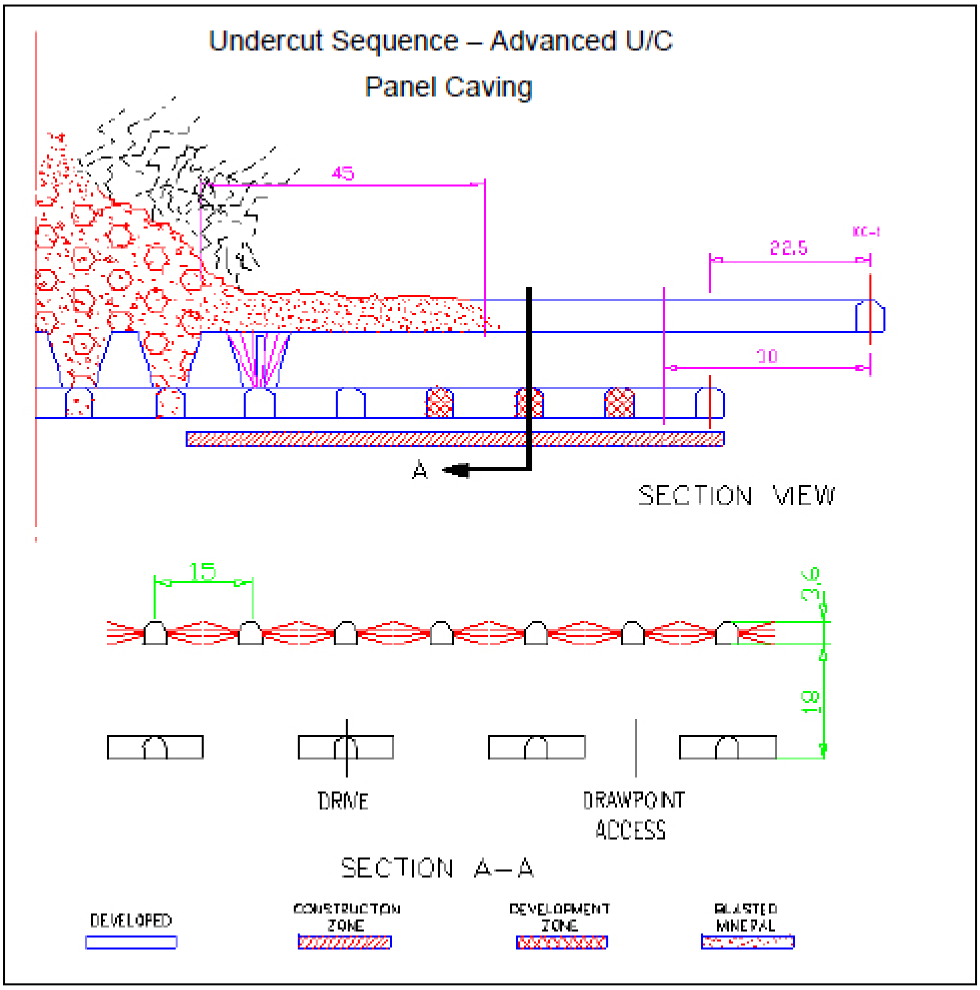
(Disadvantages: - separate ore handling on undercut level; possible sequencing problems between extraction horizon and undercutting; drawbells are developed into broken rock; possible high stress remnants from compaction of blasted undercut ore on the undercut level; drawpoint hang-ups resulting from ore compaction; slower initial production – both as a function of the sequence of development as well as the potential adverse effects as outlined).



(2)

Advanced undercut (El Teniente): **(8 marks available, 7 required)**

- Partial development of extraction level (1)
- Undercut follows behind partially developed extraction level.
- Extraction level comprises extraction drifts only or extraction drifts and drawpoint drifts. (1)
- Drawbells are never prepared ahead of the undercut, i.e. drawbells are always prepared in the de-stressed zone lagging behind the undercut – usually the 45^0 rule applies. (1)
- Advance undercutting is essentially a compromise between post- and pre-undercutting strategies. (1)
- Extraction ratio on the extraction level and associated damage are reduced compared with post-undercutting (i.e. less waste compared to post-undercutting) – mining consideration.
- Production is quicker than pre-undercutting – this reduces problems such as ore compaction associated with increased development times; similarly, the probability of forming stress-inducing remnants due to muck pile compaction is reduced. (1)
- Separate ore handling for undercut level is still required but more limited than for pre-undercutting. (mining consideration)
- Slower than post-undercutting because some extraction level development must still be completed after the undercut has advanced. This is partially due to drawbell development that must be developed into broken ore.
- Time-consuming and costly repairs to the extraction level drifts are almost inevitably required with post-undercutting but largely obviated with advance undercutting. (1)



(2)

2.3 The orebody is intersected by high strength dolerite intrusions which result in uneven distribution in fragmentation and drawdown of the caved material as well as large block sizes. As a result, the production level is expected to undergo damage. Discuss strategies other than sequencing of the undercut that you could recommend to protect the production level. (7)

<u>2.3</u> : (**8 marks available, 7 required**)

- Draw control (1)
- Drawbell spacing: if the location, orientation and thickness of the dolerite dykes are known, drawbell spacing can be planned to accommodate larger block sizes. Larger block sizes require larger drawbell spacing. ***(1)*** This in turn increases pillar stability between drawbells due to the increased drawbell spacing. The increased pillar sizes further assist in redistributing the load of caved material. ***(1)*** (A drawback of increased drawbell spacing may be experienced if caved material contains largely smaller fragments which will therefore undergo compaction in the large spacings between drawpoints). (2)
- Drawbell (drawbell brow) support. –
  - Must be appropriate to maintain drawpoint stability for the full period of extraction (1)
  - Must be able to withstand the effects of mechanical action during ore loading and secondary blasting
  - Level of support required is guided by mining environment stress distribution (1) (should take into account potential for uneven stress distribution due to uneven fragmentation and difficult drawpoint control) (1)
  - Level of support is guided by Laubscher's DRMS which relates mining stress to DRMS and associated support requirements. (1)
- Secondary blasting to reduce block sizes and minimise damage to drawpoints / uneven load distribution within the caved material (1)

<u>TOTAL MARKS: [20]</u>

QUESTION 3

Longhole open stoping is being used to mine a 12 m wide orebody, dipping at 80^0 to the west with a known strike length of 600 m. Mining will commence at a depth of 150 m below surface. The individual stopes will be 40 m long and 30 m high. Elastic modelling has been carried out to determine the stress distribution around the stope (results are included with the tables of information). A geotechnical investigation has revealed that the rock mass quality is quite good (Q' = 14) and the intact rock strength is 120 MPa. Two major joint sets, striking roughly parallel to the strike of the orebody, influence the rock mass surrounding the excavation, illustrated below.

3.1 Calculate the modified stability number (N') for the stope crown and walls. 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. (7)

3.1 : **(7 marks required)**

UCS, MPa	**120**	**Q'**	**14**

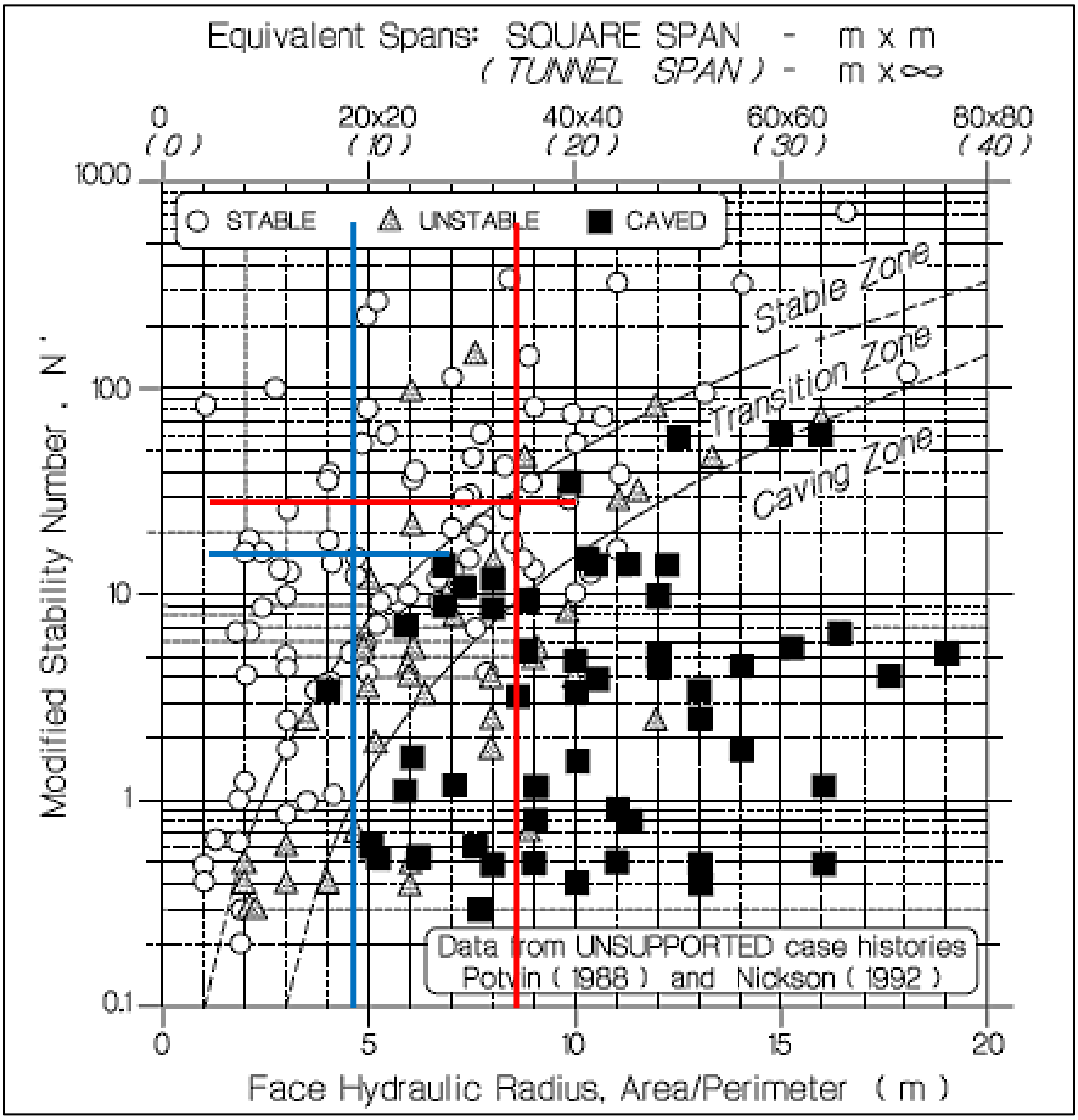
Excavation Surface	**Stress, MPa**	**Strength : Stress**	**A**	**Critical joint**	**Angle to Stope Face**	**B**	**Mechanism**	**Angle (Dip of Stope Face)**	**C**	**N'**	**Comment**
Crown	14	8.6	0.9	J2	50	0.6*	Gravity Fall	10	2.1	15.8	
Hanging-wall	2	60	1.0	J1	0	0.3	Slabbing	80	7.0	**29.2**	**Take the worst-case scenario**
Hanging-wall	2	60	1.0	J2	40	0.5	Sliding	80	5.0	35.0	
Foot-wall	2	60	1.0	J1	0	0.3	No failure	80	8.0	33.6	

* proportionally distributed value between 0.8 and 0.5

3.2 Calculate the hydraulic radius for the crown and walls *(2)*. Determine whether the stope crown and walls are stable, transitional or caving and estimate the ELOS *(3)*. Plot these values on the modified stability and ELOS graphs and hand in the annotated graphs with your script. (6)

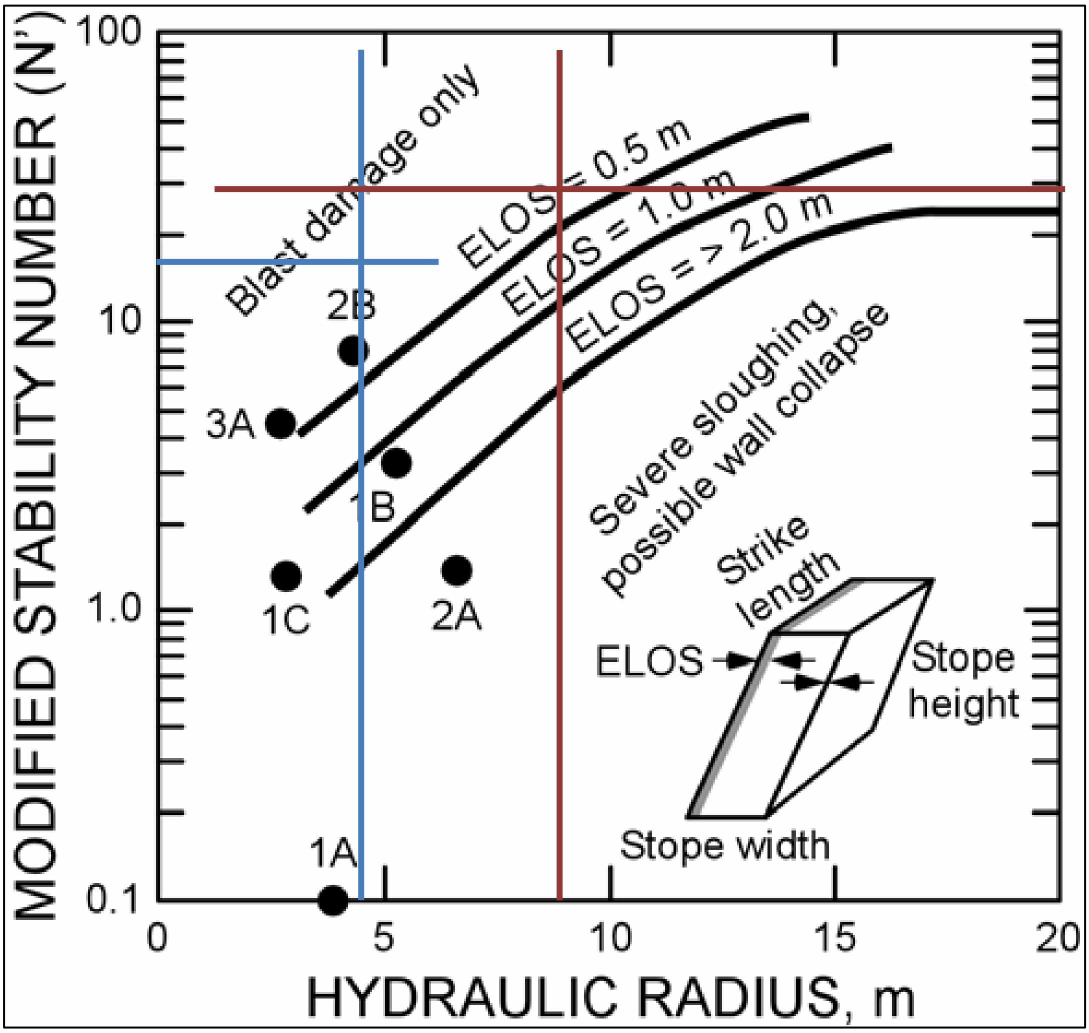
3.2 : **(6 marks required)**

Excavation Surface	Height	Width	Length	HR	N'	Stability	ELOS
Crown	-	12	40	4.6	15.8	Stable	~0.4 m (blast damage only)
Hanging-wall	30	-	40	8.6	29.2	Transitional / Stable	~0.5 m (blast damage only)
Foot-wall	30	-	40	8.6	29.4	Transitional / Stable	~0.5 m (blast damage only)



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

(1)



(1)

3.3 Calculate the percentage dilution and indicate what action is required, if any. Present your results and comments in a table. (7)

3.3 : **(7 marks required)**

Excavation Surface	N'	ELOS	Overbreak (Volume)	Initial Excavation Volume	Dilution (%)	Total Overbreak (Volume)	Total Dilution (%)	Action	Comment
Crown	12.3	~0.4 m (blast damage only)	192	14400	0%	1392	8%	Cable Support	Affects next stope
Hanging -wall	29.2	~0.5 m (blast damage only)	600		4%			May require backfill if dilution is excessive.	Total dilution is less than 10%. To determine whether this is tolerable will depend on ore grade and profit margins.
Foot-wall	29.4	~0.5 m (blast damage only)	600		4%				

TOTAL MARKS: [20]

QUESTION 4

Underground mining in hard-rock settings is highly dependent on successfully executing efficient blasting procedures.

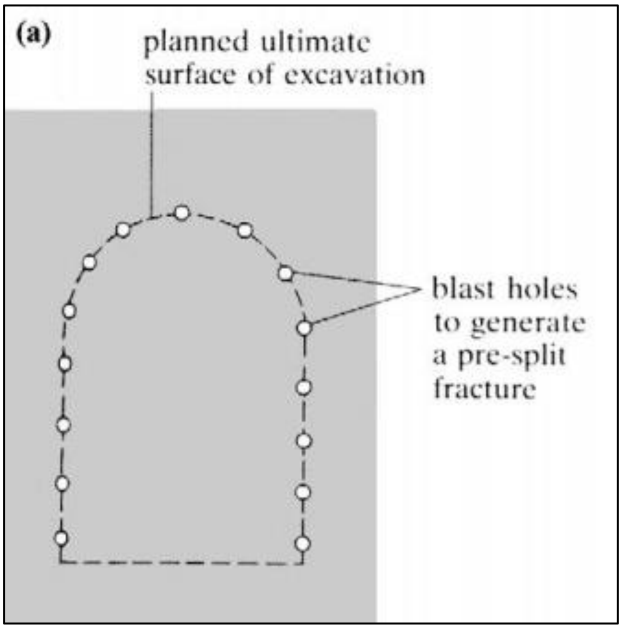
4.1 In ground conditions which are susceptible to adverse deterioration when subjected to blasting, it may be necessary to produce geometrically precise and relatively undisturbed ultimate surfaces. With the aid of sketches, discuss and compare the principles of perimeter blasting by pre-splitting and smooth blasting. (8)

4.1 : **(20 marks available, 8 marks required)**

- Both methods are based on the use of decoupled charges, in which the **objective is to restrict the development of a rose of cracks around a hole**. (1)

Pre-split blasting

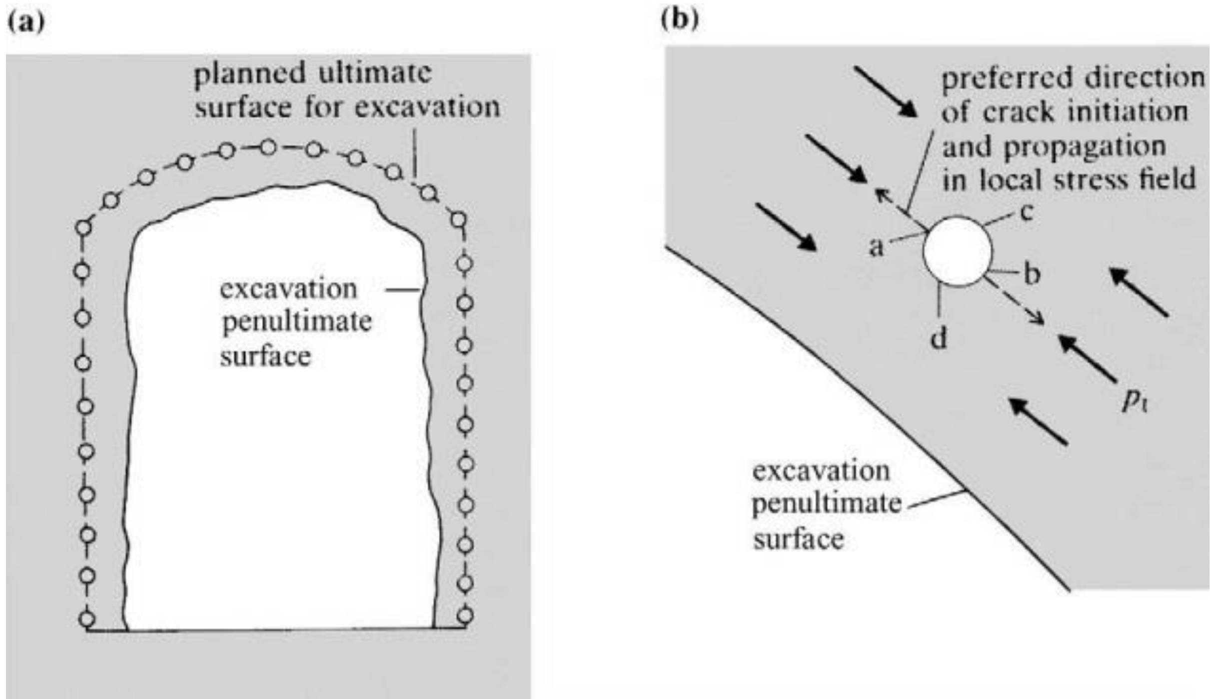
- A continuous fracture will form the final surface of an excavation (1)
- The fracture is generated in the absence of a local free face (1)
- Parallel, close-spaced holes, which will define the excavation perimeter, are drilled in the direction of face advance (1)
- Charges are detonated nearly simultaneously (1)
- Fracture development does not occur when either there is a long delay between the imitation of adjacent holes (independent detonation) or when adjacent blast holes initiate simultaneously. (1)
- Pre-split blasting shows variable results in a stressed medium, depending on the orientation of adjacent pairs of holes relative to the field stresses, and the process becomes less effective as the fields stresses increase. As a result, pre-splitting may be successful in near surface development work, but at even moderate depth, it may be completely ineffective. (2)
- In stratified rock, the preferred direction of crack development is parallel to the stratification, exploiting the natural micro-structure as guide cracks. Pre-split fractures may develop in any anisotropic rock parallel to a dominant fabric element. Fracture development perpendicular to the fabric element may be difficult or practically impossible. (2)



(2)

Smooth blasting

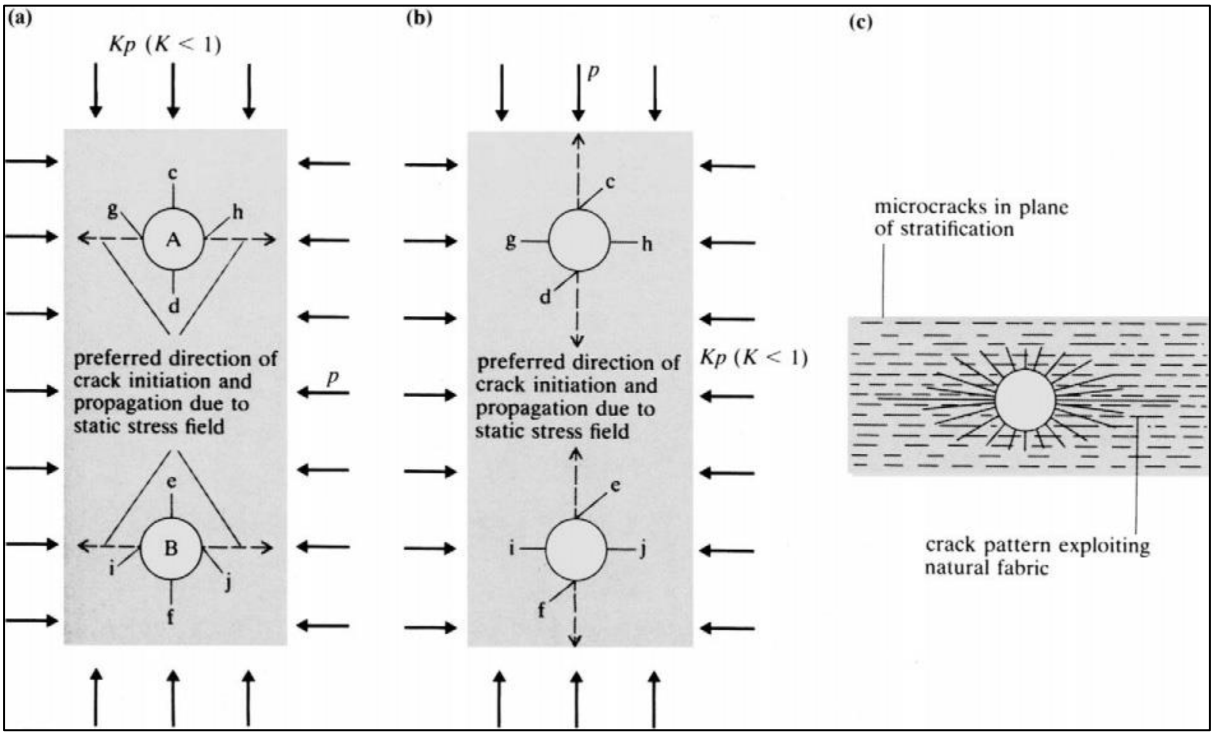
- The ultimate surface is developed by controlled **blasting in the vicinity of a penultimate free face** (1)
- Holes are initiated with short delaty between adjacent holes (1)
- The burden on holes exceeds the spacing (1)
- The mechanics of smooth blasting may be understood by examining the local state of stress around the penultimate boundary of the excavation (1)
- The design of the opening should achieve a compressive state of stress in the excavation boundary and adjacent rock (1)
- The local stress field is virtually uniaxial and directed parallel the penultimate surface (1)
- This generates tensile boundary stresses around the blast hole (at points in the blast hole perimeter where the tangent to the blast hole perimeter is normal to the direction of compressive stress within the penultimate excavation perimeter) and compressive stresses at points where the tangent to the borehole perimeter is parallel to the compressive stress field. (2)
- Thus the stress wave emitted by detonation of the charge in the hole initiates radial fractures which propagate preferentially parallel to the local major filed stress (which is orientated parallel to the penultimate excavation perimeter) (1)
- Generation of fractures parallel to the penultimate surface of the excavation is favoured (1)



(2 to 3)

4.2 Sketch the blast fracture pattern that you would observe around a blast socket in an environment where there is a significant difference between the major and minor principal stresses. Clearly indicate the direction of principal stresses in your sketch. (3)

4.2 : **(3 marks required)**



(3)

4.3 List and discuss measurement techniques that can be used in the field to evaluate the performance of explosives. (6)

4.3 : **(14 marks available, 6 marks required)**

(Brady and Brown, Chapter 17.9)

- Velocity of detonation (1)
- Near-field vibration monitoring (1)
- Muckpile surveying (1)
- Fragmentation measurement (1)
- High-speed photography (1)
- Gas penetration measurement (1)
- An underwater test is commonly used to estimate the energy released from a charge. (1)
- In **test blasts** in rock, each trial explosive should be sampled as it is being loaded into the blast hole and its physical and chemical properties determined from comparison against specifications. (1)
- Damage to surrounding rock by a blast may be assessed by a variety of techniques including the direct measurement of fractures in blast holes and on rock surfaces and indirect methods including cross hole or tomographic seismic scans, in-hole acoustic or seismic profiling, ground probing radar, micro-seismic emission monitoring, blast vibration measurements and in situ permeability testing. (3)
- Blast damage might be expected to reduce the static and dynamic Young's moduli of the rock surrounding the blast site and some methods therefore rely on direct and indirect measurements of these properties. (2)
- Ground motions in the far field, including at ground surface, may be measured using systems for monitoring micro-seismic activity. (1)

4.4 List and briefly discuss rock properties that determine the performance of a particular explosive in the medium. (3)

<u>4.4</u> : **(4.5 marks available, 3 marks required)**

(Brady and Brown, Chapter 17.2)

- Young's modulus (½): the capacity of the rock mass to transmit energy (½) is related in part to the Young's modulus of the rock medium.
- The ease of generating new fractures (½) in the medium is a function of the strength properties of the rock material, which may be represented for convenience by the uniaxial compressive strength (½).
- The unit weight (½) of the rock mass affects both the energy required to displace (½) the fragmented rock (muckpile) and the energy transmissive properties (½) of the intact medium.
- Jointing (½) – directional fracture propagation (½)
- Field stress (½) – directional fracture propagation (½)

<u>TOTAL MARKS: [20]</u>

QUESTION 5

Backfill can be generated from several types of constituent materials and fulfil various functions at an underground mine site.

5.1 Compare and contrast the characteristics, advantages and disadvantages of three types of backfill systems. Present your answer in a table. (10)

5.1 : **(15 marks available, 10 marks required)**

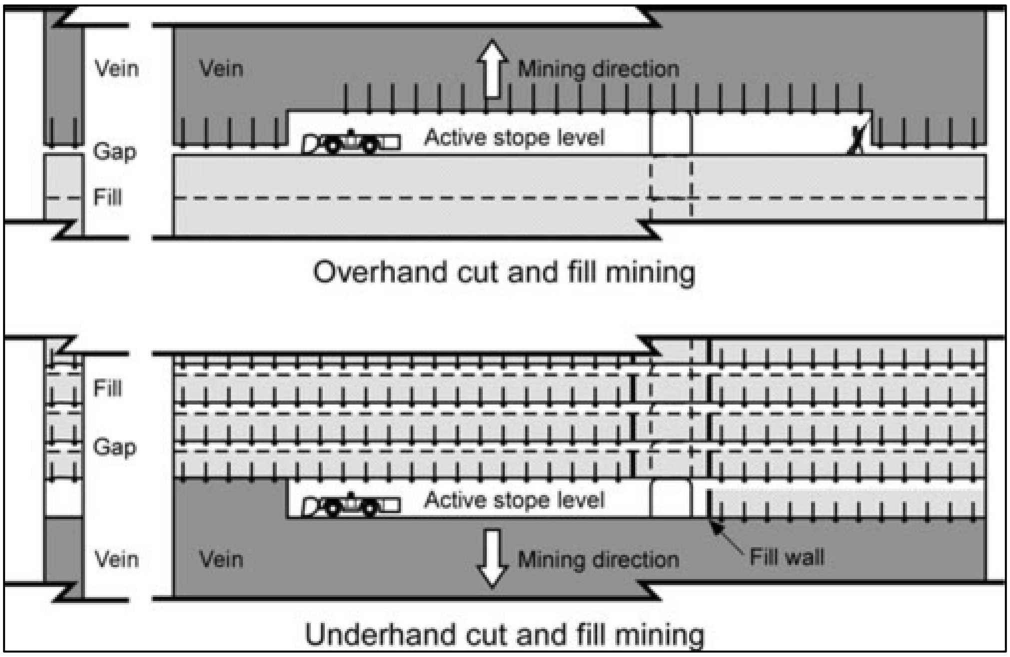
SIMRAC, 2001 – Best practice rock engineering handbook for "other" mines, T R Stacey
Ryder and Jager – A textbook on Rock Mechanics for tabular hard rock mines, pp 274-275

System	Advantages	Disadvantages	Remarks
Crushed Waste Fill	- Available waste rock on surface can be used if tipping through a pass system is appropriate - Hoisting time on shaft is reduced if waste is crushed underground - Immediate access is possible on the fill by heavy equipment	High cost of crushing underground High cost of transport Confinement provided to pillars not as good as with some other types of backfill	Sufficient waste must be available or generated
Classified cycloned tailings (CCT)	- Simple method of conveying large volumes of fill from surface directly into areas to be filled - Has good drainage characteristics	Large volumes of excess water have to be handled underground Use of only the coarser fraction, leaving the finer fraction for tailings disposal creates potential problems with the building of slimes dams More difficult to pump than full plant tailings	Sufficient backfill material must be available
Cemented Full Plant Tailings	- Simple method of conveying large volumes of fill from surface directly into areas to be filled - Maximum use of tailings product - Very good pumping characteristics	Poor drainage characteristics High moisture content requires greater quantities of binder	The binder material assists by hydrating the excess water retained by the fines
Paste Fill	- Relatively small amounts of cement (3% to 5%) produce stiff backfill (1.5 to 3.5MPa) - Reduced tailings impoundment requirements - Good support properties - Reduced spills underground - The entire tailings stream can be placed underground	Costly system which requires expensive pumps, pipes and dewatering equipment Conveying distances limited due to high pressure gradients (approximately 1km horizontally) Good quality control is necessary	Pumping difficulties can be alleviated by pumping the fill underground as a slurry and then dewatering it close to the stopes
Slurry Fill	- Very little run-off water - Can be used with full plant tailings - The total cost of this system could be less than for cemented CCT	Requires availability of a special binder Accelerator is costly and requires accurate dosing	Technically, this system is good

5.2	Discuss the use of backfill in the mining method illustrated below. Would you recommend the backfill be cemented or uncemented? Discuss critical performance requirements. (5)

5.2 : **(10+ marks available, 5 marks required)**

Ryder and Jager – A textbook on Rock Mechanics for tabular hard rock mines, pp 274-275
"*Underhand Cut-and-Fill Mining at the Lucky Friday Mine*" – Peppin et al, SME Publication (2001), "Underground Mining Methods".



Overhand:

- Must be able to carry machinery
- Strength and consolidation of backfill dependent on final purpose – i.e. reduce remnant pillar sizes, increase extraction in which case, crushed waste fill might be acceptable with a cement cap to carry machinery
- High strength not necessary (unless intended to provide support in place of pillars or be free-standing)

Underhand:

- The underhand cut-and-fill system depends on **cemented backfill** (UCS > 1.5 MPa and < 4.5 MPa). (1)
- i.e. **the backfill must be self-supporting and be able to resist damage under high stress or seismic conditions**. (2)
- Fill with compressive strength less than 1.5 MPa will not be self-supporting for more than 1 to 3 days at widths over 2.5 m. ()
- Must be able to safely undercut backfill (1)
- May require reinforcing (1)
- Backfill is placed in successive pours – may result in cold joints that can fail in tension when undercut. Can be avoided by using paste fill or reinforcement (1)
- **High-strength backfill placed with good quality control** can improve production when undermining the backfill by increasing the span of production headings and minimising ground support installation requirements (Murray Mine, SME 2001, pg 335). (2)
- The backfill must have the capacity to sustain blast-on. (1)

Generic:

- The addition of cementitious binders to backfill results in a significantly stiffer stress-strain response in the low strain region (early stiffness). (1)
- Fill with UCS > 4.5 MPa is susceptible to damage from seismic events and stope closure, causing it to break into large blocks. ()
- Rapid early stiffness for production and safety purposes are required. ()
- Tailings and/or crushed waste is available for the manufacture of the backfill due to off-reef development associated with this mining method. (1)
- It is probable that CCT is employed instead of FPT due to the high potential risk to safety and production associated with poor drainage characteristics of FPT. (1)
- Backfill is required to reduce stress on sill and or remnant pillars. (1)
- The use of backfill optimises extraction by reducing the necessary pillar sizes. (1)
- Paste backfill might be recommended due to the higher placement quality that can be achieved which is particularly necessary for safety of access in underhand mining. (1)

5.3 Access into secondary mining areas often requires some development through or undercutting beneath backfill such as drift development or crown pillar removal. Discuss hazards and failure mechanisms you would anticipate, with suggestions for reinforcement strategies that you would propose. (5)

5.3 : **(10 marks available, 5 marks required)**

Reference: "*Backfill in Underground Mining*" – D Landriault, SME Publication, 2001 – Underground Mining Methods

- The backfill in the stope back must be strong enough to be self-supporting over the span where it will be exposed. (1)
- To determine the required strength, a safety factor of 1.2 is generally used if only equipment (remote mucking) is exposed under the backfill. (1)
- A safety factor of 2.0 is used if both men and equipment will be working below the fill. (1)
- The possible failure modes of the exposed fill are associated with arching, flexural or bending failure, block sliding due to side-shear failure, rotational failure at the hanging-wall contact and sloughing of wall rock. (3)
- Cold joints resulting from successive pours present a delamination hazard. (1)
- High-strength backfill can be placed to accommodate development (drift development) with minimal mechanical support.
- Friction bolts with welded wire mesh screen can be used to support the backfill. (1)
- A combination of shotcrete reinforced with friction bolts and welded wire-mesh screen can be used (North-American mines). (1)

TOTAL MARKS: [20]