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:** OCTOBER 2015 **TIME**: 3 HOURS	**EXAMINER:** PJ LE ROUX **MODERATOR:** W JOUGHIN **TOTAL MARKS:** [100] **PASS MARK:** (60%)

NUMBER OF PAGES: 6 (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 – MINING METHODS

A high grade steep (80°) to flat (25°) dipping orebody, 10 m to 35 m wide is hosted in a highly jointed, strong rock mass, 1000 m to 2300 m below surface.

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)

Sub-level (Open) Stoping

Considerations:

- Steep dip - the inclination of the footwall must exceed the angle of repose
- Stable rock in both the hangingwall and footwall
- Competent ore and host rock
- Regular orebody geometry

Shrinkage Stoping

Considerations:

- Steep dip
- Comparatively stable hangingwall and footwall
- Regular ore boundaries
- Ore that is not affected by storage in the stope (certain sulphide ores tend to oxidize and decompose when exposed to the atmosphere)

Cut-and Fill Mining

Considerations:

- Steep dip, thin ore body
- Favourable geotechnical characteristics
- Fill material characteristics
- Hangingwall to be cable bolted

Vertical Crater Retreat

Considerations:

- Steep dip - the inclination of the footwall must exceed the angle of repose

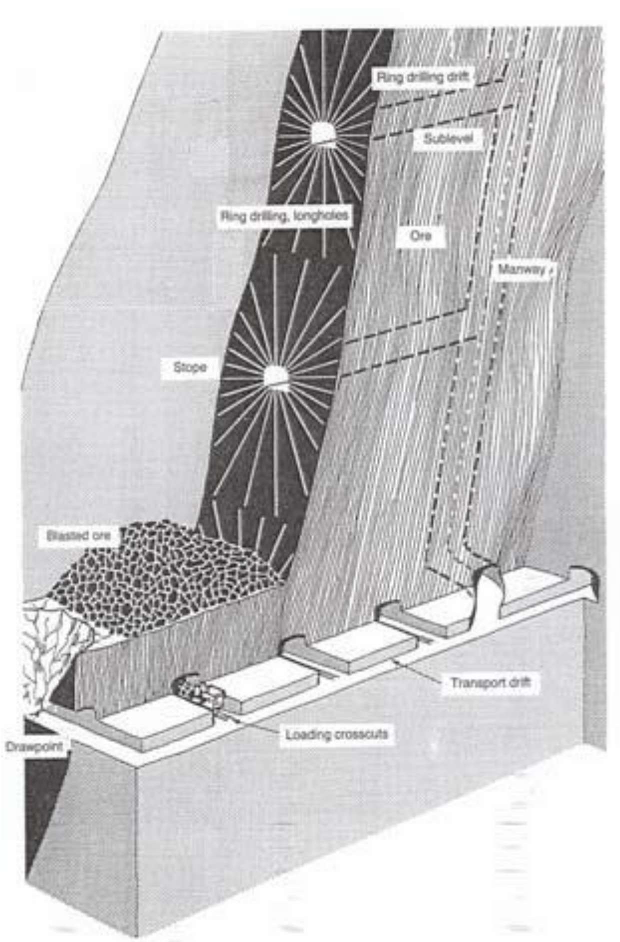
- Stable rock in both the hangingwall and footwall
- Competent ore and host rock
- Regular orebody geometry

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

Sub-level (Open) Stoping

The development for Sub-level (Open) stoping consists of

- A haulage drift is excavated along the ore body at the drawpoint level
- A drawpoint loading arrangement is created underneath the stope
- The stope is undercut
- An overcut access is excavated for drilling and charging



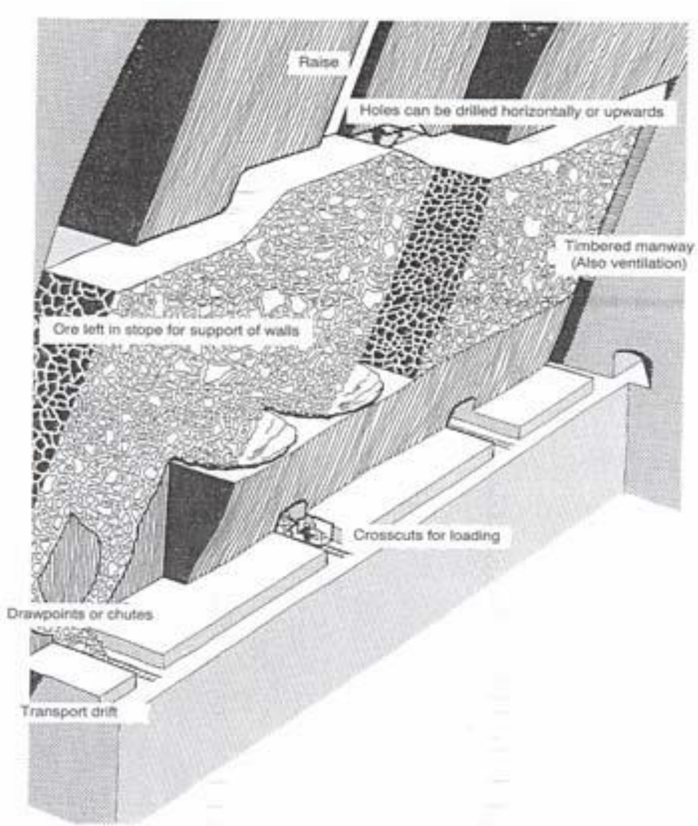
Advantages:

- *Mechanization – drilling carried out in advance of stoping*
- *Extraction on retreat*

Disadvantages:

- *Extensive development prior to extraction*

Shrinkage Stoping



The development for shrinkage stoping consists of

- A haulage drift along the bottom of the stope
- Crosscuts into the ore underneath the stope
- Finger raises and cones from the crosscuts to the undercut
- An undercut or complete bottom slice of the stope 5 to 10m above the haulage drift
- A raise from the haulage level passing through the undercut to the main level above to provide access and ventilation to the stope

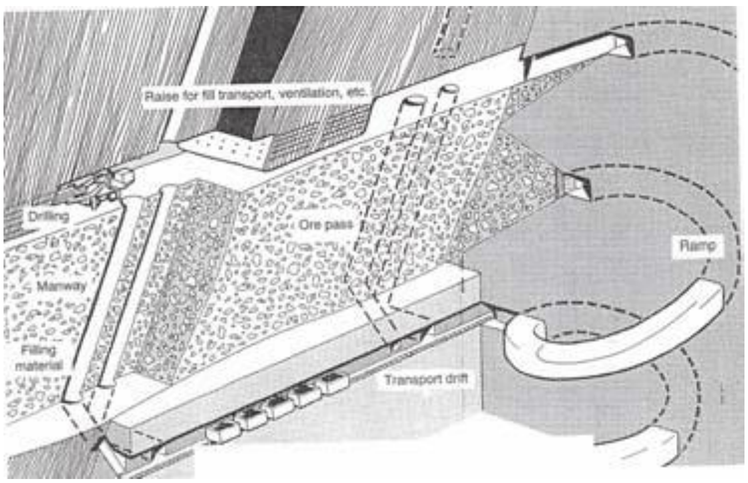
Advantages:

- Mechanization – drilling carried out in advance of stoping
- Only bulk of ore removed after every blast – provide stability for hangingwalls and footwalls

Disadvantages:

- Ore prone to oxidation and hydrolysis

Cut-and Fill Mining



Advantages:

- Can be used in ore bodies with irregular contours
- Offers more mining selectivity
- Total extraction
- Working platform

Disadvantages:

- Orebody is above mining – potential for fall-out if orebody not competent

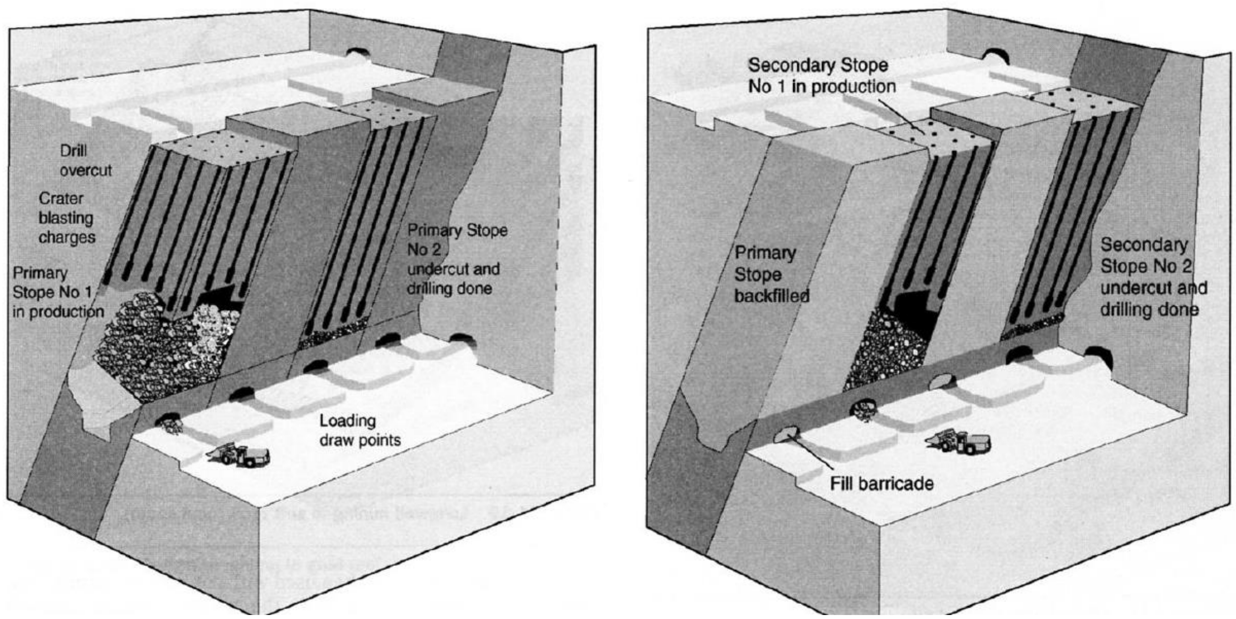
Vertical Crater Retreat

Considerations:

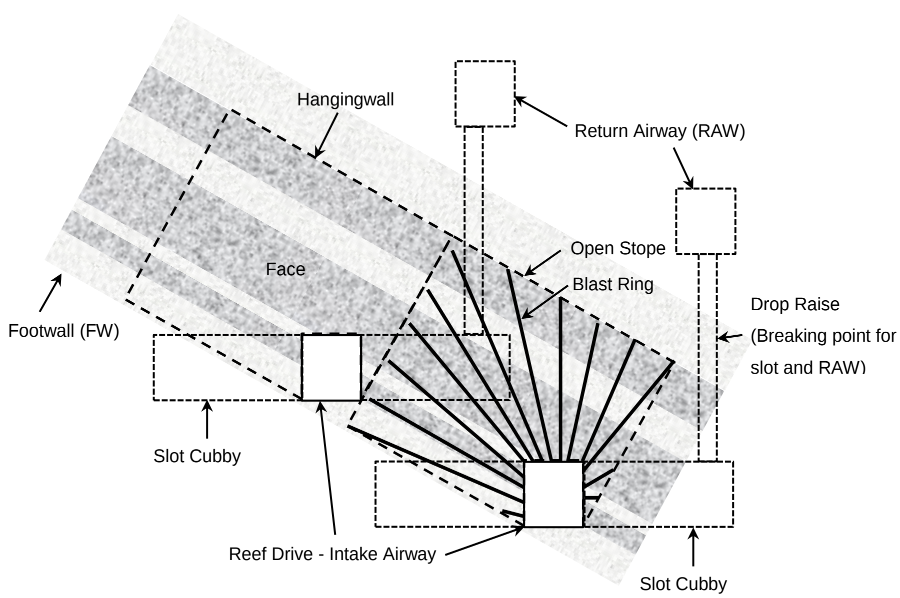
- Steep dip - the inclination of the footwall must exceed the angle of repose
- Stable rock in both the hangingwall and footwall
- Competent ore and host rock
- Regular orebody geometry

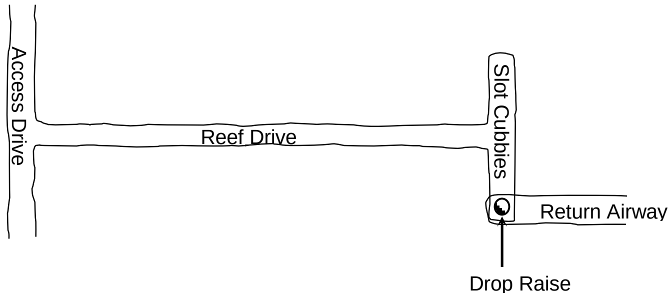
The development for VCR stoping consists of

- A haulage drift is excavated along the ore body at the drawpoint level
- A drawpoint loading arrangement is created underneath the stope
- The stope is undercut
- An overcut access is excavated for drilling and charging

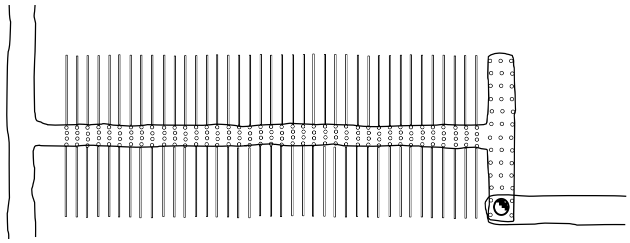


Open stope design with reef widths 10 to 35m and reef dips 80° to 25°

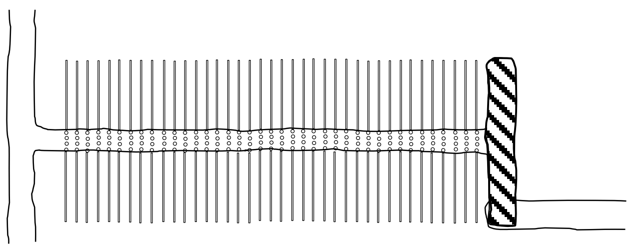




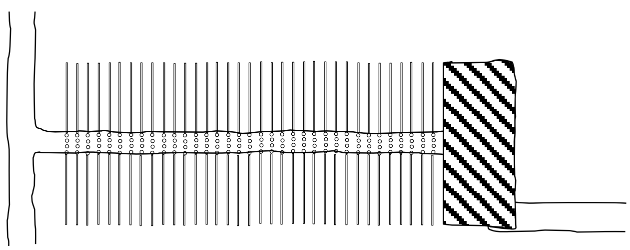
a) Reef drive is developed on strike with slot cubbies and drop raise blasted.



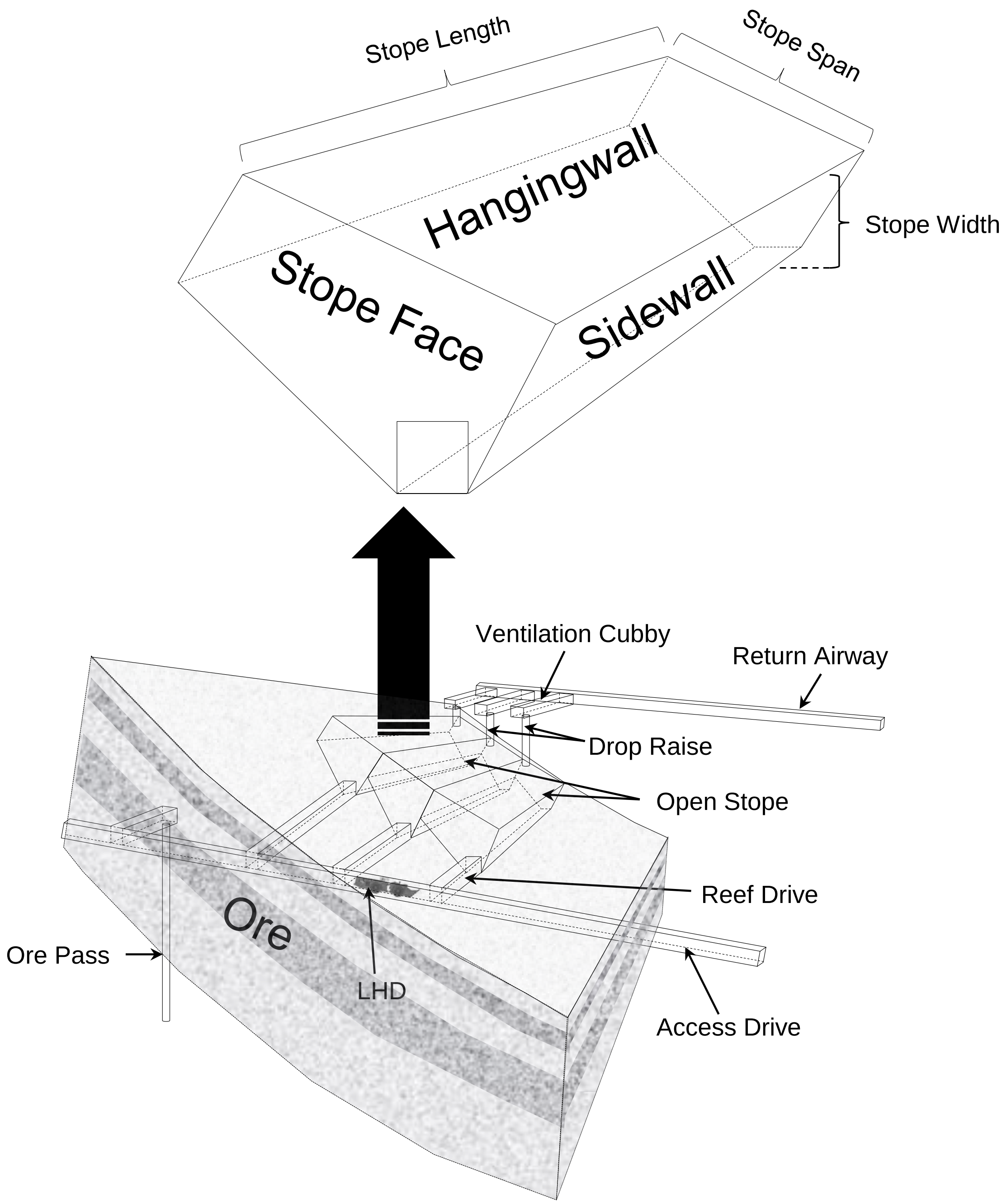
b) Slot and blast rings are drilled.



c) Slot is blasted and cleaned with remote loading.



d) A maximum of four rings are blasted and cleaned at one time with remote loading.


Stope Length
Stope Span
Hangingwall
Stope Face
Sidewall
Stope Width
Ventilation Cubby
Return Airway
Drop Raise
Open Stope
Reef Drive
Ore
Ore Pass
LHD
Access Drive

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

Brady and Brown, 2004, Ch 14.

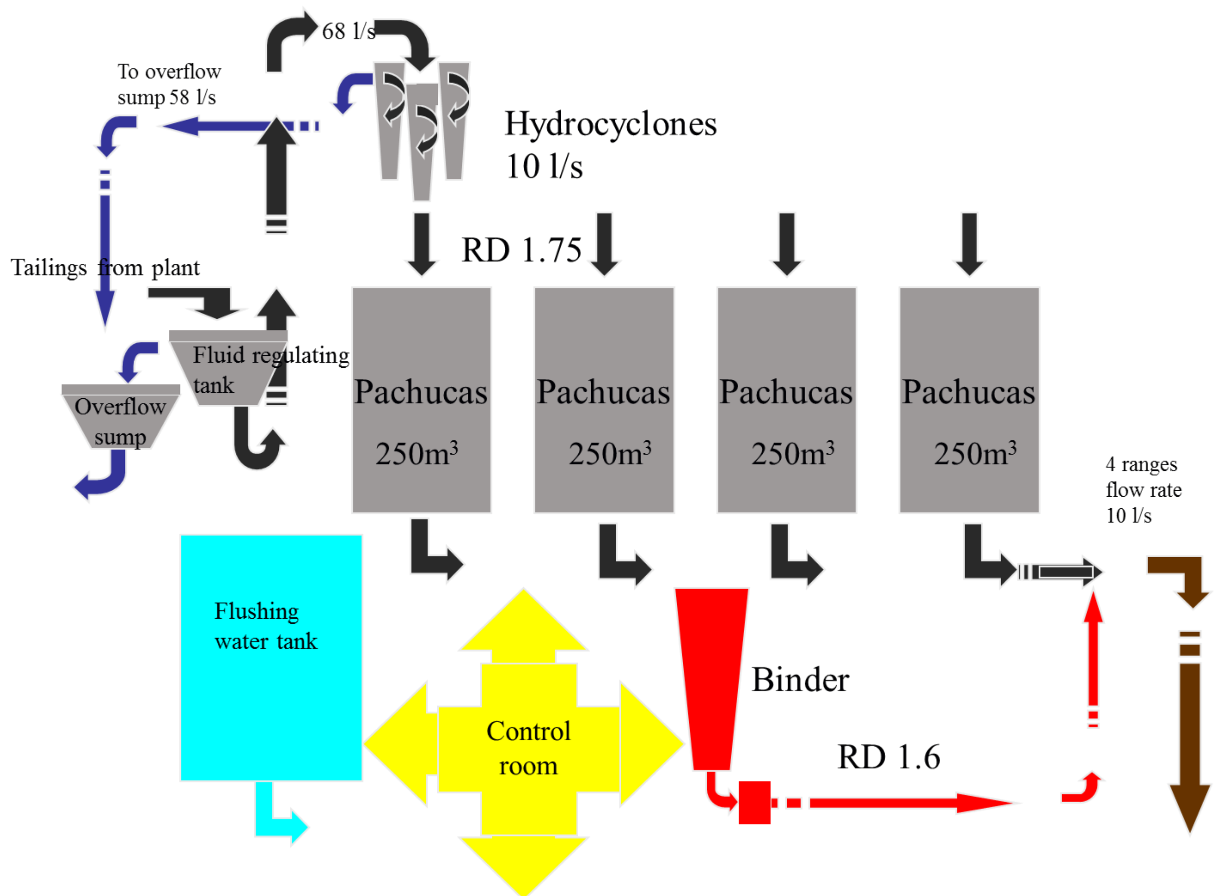
Sill pillars are usually left between levels, to protect the lower workings from rockfalls due to mining activities above (1); allow concurrent production on multiple levels (1); develop drawpoint infrastructure at the base of each stope (1), reduce the mining span (1) and control stability of hangingwall (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).

[20]

QUESTION 2– BACKFILL

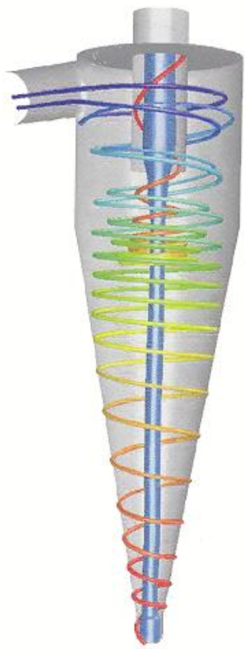
Backfill forms an integral process of massive mining. As the Rock Engineer on the mine you are tasked with the design of the backfill system for this mine.

2.1 Draw a typical layout of a classified tailings backfill plant making used of sketches. Also name and discuss the different units of the backfill plant in your sketches. (10)



Discuss the following:

- Fluid regulating tank
- Overflow sump
- Hydrocyclones



- Pachuca
- Flushing water tank
- Binder silo
- Control room
- Backfill ranges and if it's an open ended or closed system and type of pipes to be used

2.2 Why would backfill be required in massive mining operations? (4)

- Hangingwall stability
- Increased extraction
- Dilution Control
- Regional Support
- Pillar recovery
- Working floor
- Reducing mining cost
- Protect mine workings
- Reduce rockburst damage
- Surface Subsidence control

- Rock support substitute
- Environmental protection
- Fire control
- Improved ventilation

2.3 Discuss and name two of the methods used for calculating the required backfill strength. (2)

1) The stability of a free standing backfill pour can also be determined using equations developed from physical model tests. Based on centrifugal modeling tests conducted by Mitchell (1983), the stability can be related to the unconfined compressive strength of the backfill. The critical unconfined compressive strength (UCS) is given by:

$$UCS_{design} = \frac{\gamma LH}{L+H} FS$$

where:

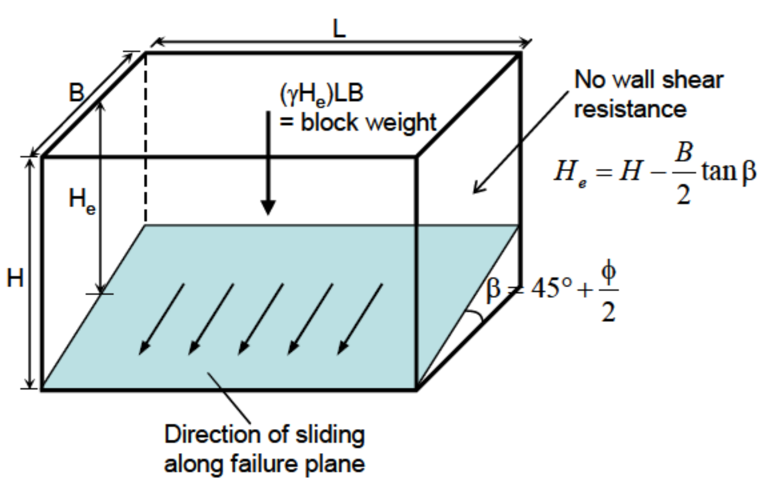
UCS = unconfined compressive strength, kPa

H = total height of fill (m)

L = strike length of stope (m)

γ = fill bulk unit weight (kN/m^3)

FS = factor of safety



2) Australian Centre of Geomechanics Backfill UCS (Design) for massives

$$UCS = \frac{\gamma L g}{1 + \left(\frac{L}{H}\right)} SF$$

where:

UCS = unconfined compressive strength, kPa

H = total height of fill (m)

g = 9.81m/s^2

L = strike length of stope (m)

γ = fill bulk unit weight (kN/m^3)

FS = factor of safety

3) CSIR - Backfill UCS (Design) for Massive

$$UCS = \gamma g H * SF$$

where:

UCS = unconfined compressive strength, kPa

H = total height of fill (m)

g = 9.81m/s^2

γ = fill bulk unit weight (kN/m^3)

FS = factor of safety

2.4 How would you ensure compliance to the required design strength? (2)

Quality control making use of:

- Collecting samples from ranges for testing
- Drill core samples for UCS tests on a regular basis
- Check relative density of backfill at plant and underground daily
- Check particle distribution of tailings on a regular basis

2.5 How would you select the type of binder to be used in cemented backfill? (2)

The strength, hydraulic conductivity, and cost should be considered.

[20]

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 Potvin Modified Stability Number method:

3.1 Describe how N' is determined. (7)

The modified stability number N' is calculated as:

N′ = Q′ x A x B x C

Q' = modified from the NGI (Q) rock quality index (Barton et al.,1974) with the stress reduction factor set to 1 (Potvin, 1988).

$$Q' = \left(\frac{RQD}{Jn}\right)\left(\frac{Jr}{Ja}\right)$$

RQD = Rock Quality Designation

Jn = Joint Set Number

Jr = Joint Roughness Number

Ja = Joint Alteration Number

A is the stress factor that was modified by Potvin (1988) from a rule of thumb by Mathews et al (1981), which was an attempt to account for the effect of stress in open stope design.

The A-factor can be expressed as the relationship between the intact rock Uniaxial Compressive Strength (UCS) C_{ucs} and induced stress in the hangingwall to account for any compressive failure. If the obtained value for A is 1, the hangingwall is assumed to be in relaxation or tension.

A is given by:

A = 1.125R – 0.125 1>A>0.1

where *R* is the ratio of the C_{ucs} of the rock material to the maximum induced compressive stress. The maximum induced compressive stress is determined by numerical stress analyses.

B is a factor, which describes the ease of keyblock fallouts. *B* is given by the following equations:

$B = 0.3 – 0.01$	$\alpha < 10$
$B = 0.2$	$10 < \alpha < 30$
$B = 0.02\alpha - 0.4$	$30 < \alpha < 60$
$B = 0.0067\alpha + 0.4$	$60 < \alpha < 90$

where α is the true angle between the hangingwall surface of the excavation and the joint plane. In the case of numerous joint planes, the smallest angle is applicable. The true angle between the hangingwall surface of the excavation and the joint plane is generally determined using a stereonet.

C is the gravitational adjustment factor. In the case of gravity falls and slabbing where sliding on joints is not applicable, the factor is given by the following equation:

C = 8 – 6 Cos (Dip of stope face)

If sliding on joints can be expected, the gravity adjustment factor is given by the following equations:

C = 8 Dip of critical joint < 30^o

C = 11 – 0.1 (Dip of critical joint) Dip of critical joint > 30^o

3.2 Describe the concept of hydraulic radius. (2)

Hydraulic radius is commonly used in massive mining operations as a measure of the size of the extraction area in plan view where the stability for a given rock mass with certain geotechnical characteristics is estimated. The hydraulic radius of an open stope can be calculated as the area of the hangingwall divided by its perimeter. As the hydraulic radius of an open stope increases, the larger the exposed roof area and the more unstable the hangingwall beam becomes. The reason for this is that the beams become less self-supporting, become unstable and eventually fall out under gravity.

3.3 Discuss the limitations of the Modified Stability Number method. (6)

- The method assumes the wall in question is fully bounded and thus the method cannot be used when adjacent stopes are not tightly filled or for intersections;
- Discrete wedges, shears, and delamination zones are not accounted for and are more appropriately assessed by analytical methods; and
- Corners or bulges in the stope wall are not accounted for in the hydraulic radius calculation and will dominate the stope stability.

There are other significant problems associated with obtaining representative rock classification values which include the following:

- Limited HW exposure
- Difficulty interpreting core data
- Defining Structural Domains/Variable HW rock quality
- Multiple zones of HW rock (layered)
- Difficulty interpreting discrete features such as shears and dykes
- Using the graphs for design outside their limitations (specific to the mines and range of data applied to develop the empirical system)
- Limited database from weak rock masses
- Methods do not account for undercutting, stress, faulting, blasting, time, and other mine specific factors
- Difficult to assess irregular opening geometry
- The failure to realise that the graphs are for one rock type rating and don't include an adjustment for rock contacts and structure

3.4 Draw and explain how the Modified Stability Number method works and how you would apply it to a mine. (4)

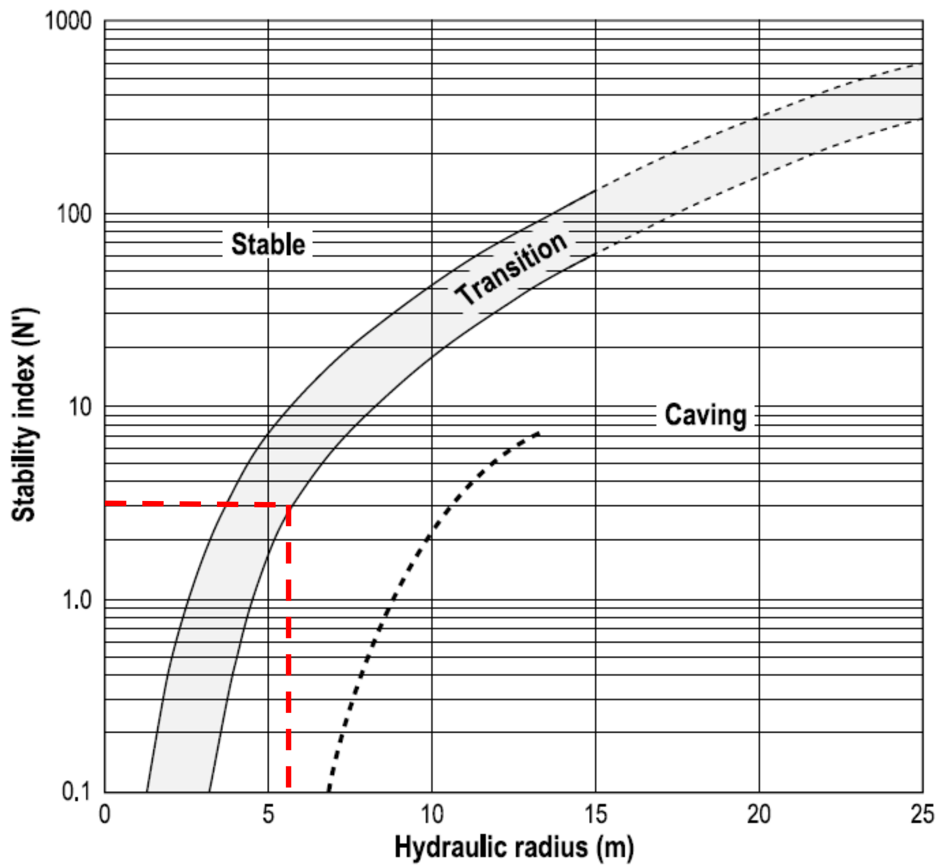


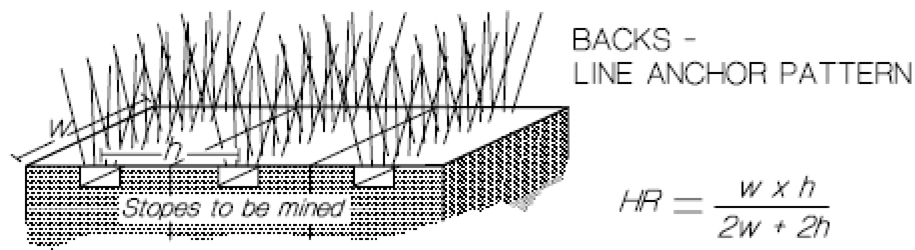
Figure above shows the Modified Stability Graph (After Potvin, 1988, From Potvin and Hadjigeorgiou, 2001)

For this example if the Modified Stability Number (N') = 3 and the HR = 6 the stope will fall on the boundary of the transitional zone with possible caving, thus the hangingwall will have to be supported. The graph should be amended for each mine depending on actual stope data.

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

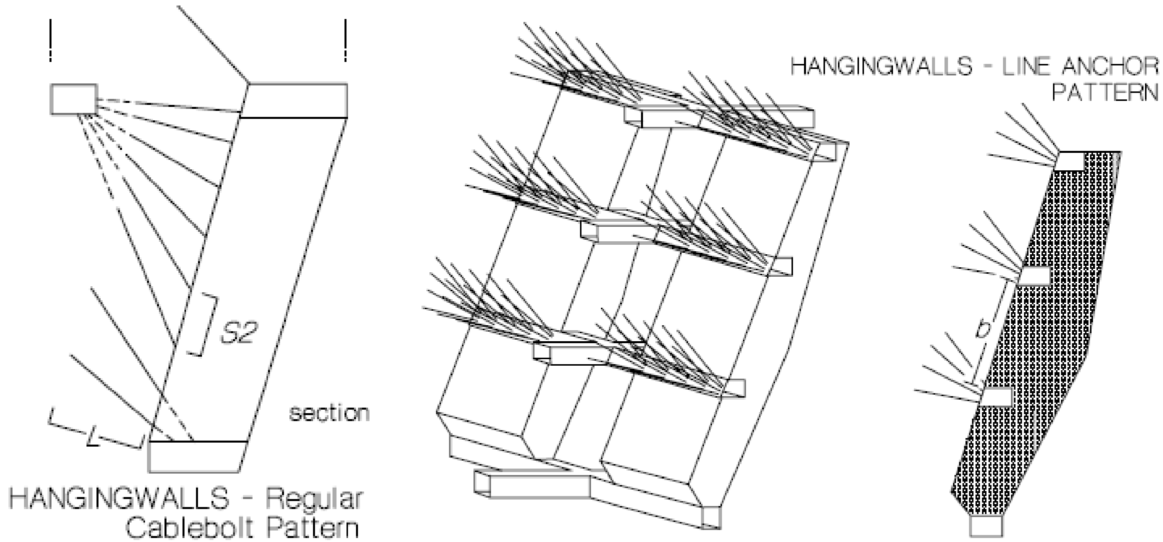
Open stopes can be supported using regular (uniform) arrays for the excavation as a whole or a line or point array where access is limited [1]. Line or point arrays should only be used in rock-masses dominated by a single lamination parallel to the stope face or joint sets perpendicular to the face (few 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):

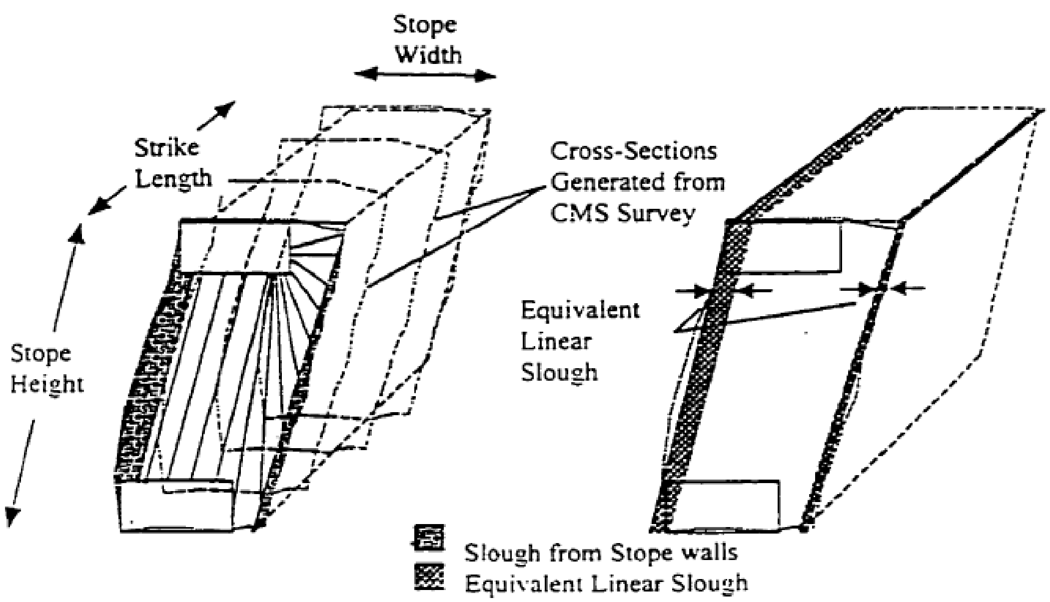


(2)

3.5 What does ELOS stand for? (1)

The potential for dilution can be determined from design charts proposed by Clark and Pakalnis (1997) or Capes (2009) respectively. The Equivalent Linear Overbreak Slough (ELOS) is graphically illustrated in the Figure below and is defined as:

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



Clark and Pakalnis (1997) developed the dilution graphs, which were then improved by Capes (2009) as an empirical design method. The dilution graph is based on the Modified Stability Graph after Potvin (1988) and has been empirically calibrated so that the degree of stability is represented as the average metres of slough (ELOS) that can be expected to fail from the open stope hangingwall. An estimate of dilution is determined by plotting the modified stability number, N' versus the hydraulic radius of the open stope hangingwall being assessed (Clark and Pakalnis, 1997).

[20]

QUESTION 4 – SHOTCRETE

With reference to the research project for the evaluation of the performance of shotcrete with and without fibre reinforcement under dynamic and quasi-static loading conditions (SIM 040204):

4.1 Name the steps in the process to determine the inputs required for the design of shotcrete. (6)

- Field stress and stress changes
- Seismicity
- Rock mass characteristics
- Excavation characteristics and requirements
- Tendon support design
- Shotcrete characteristics

4.2 What inputs are required for elastic modelling. (4)

- Virgin stress state
- Elastic parameters
- Mine geometry
- Mining sequence and layout

4.3 The rock mass characteristics should be determined for the geotechnical domain or ground control district in which the tunnel is situated. Name the important aspects to consider. (6)

- Type of rocks
- Rock strength
- Geological structures
- Bedding and joint characteristics
- Weathering
- Ground water

4.4 It is important to consider not only the size of excavations, but the function and importance of an excavation. Name the important aspects to consider. (4)

- Actual height and width
- Life of excavation
- Exposure of personnel
- Access and importance
- Minimal functional dimensions for tramming
- Accessibility for maintenance and rehabilitation

[20]

QUESTION 5 – NUMERICAL MODELLING

Numerical analysis form an integral part of Rock Engineering when evaluating excavations, mining layouts, support requirements etc.

5.1 Discuss the theory of the Boundary Element Method. (6)

Ryder, J.A. and Jager, A.J. (2002) A Textbook on Rock Mechanics for Tabular Hard Rock Mines, The Safety in Mines Research Advisory Committee (SIMRAC), Johannesburg, 2002. pp. 403 - 409

5.2 Discuss the application of the Boundary Element Method. (4)

Ryder, J.A. and Jager, A.J. (2002) A Textbook on Rock Mechanics for Tabular Hard Rock Mines, The Safety in Mines Research Advisory Committee (SIMRAC), Johannesburg, 2002. pp. 409 - 410

5.3 Discuss the theory of the Finite Element Method. (6)

Ryder, J.A. and Jager, A.J. (2002) A Textbook on Rock Mechanics for Tabular Hard Rock Mines, The Safety in Mines Research Advisory Committee (SIMRAC), Johannesburg, 2002. pp. 410 - 413

5.4. Discuss the application of the Finite Element Method. (4)

Ryder, J.A. and Jager, A.J. (2002) A Textbook on Rock Mechanics for Tabular Hard Rock Mines, The Safety in Mines Research Advisory Committee (SIMRAC), Johannesburg, 2002. pp. 413 - 414

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TOTAL MARKS: [100]