Question 1

Multiple Choice

Wrong answers are **NOT** negatively marked. Showing calculations can result in partial marks

1.1. **C** (3)

ESS = stress - strength = 15MPa - [0.8MPa+(25MPa*Tan25°)] = 2.54 MPa

1.2. **B** (2)

1.3. **A** (3)

NOTE - spacing is SKIN to SKIN, therefore Area = 2.18m * 1.68m (not 2m*1.5m)

You were instructed to use g=10 m/s2.

SR = (20 * 10)/(2.18*1.68) = 54.6 kN/m2

1.4. **D** (2)

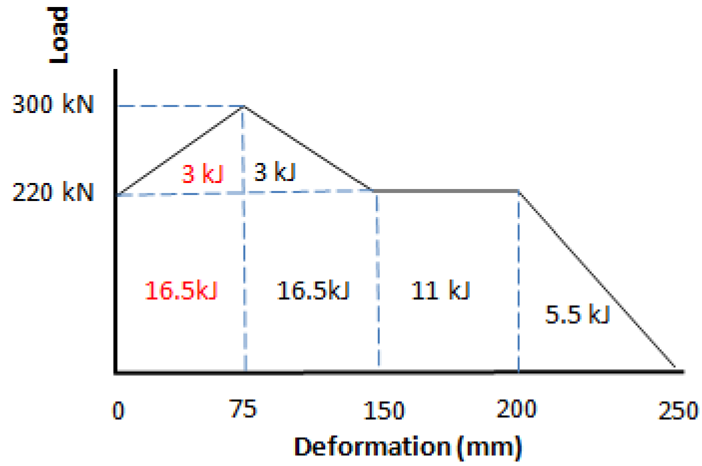
1.5. **D** (3)

EA = Area under graph. **NB.** Load is already given in kN but remember to subtract the initial 75mm yield!

EA = Sum of black area's. The energy represented by the Red areas has already been utilised.

EA = (220)*(0.15-0.075) + 0.5*(300-200)*(0.15-0.075) + (220)*(0.2-0.15) + 0.5*(220)*(0.25-0.2)

= 16.5kJ + 3 kJ + 11kJ + 5.5 kJ = 36 kJ = 36 000 Joule



1.6. **C** (2)

1.7. **D** (3)

y intercept = log(70) = 1.845

Slope (is negative!) = (0 - 1.845) / (2.65 - 0) = - 0.696

1.8. **A** (2)

TOTAL : 20

Question 2

2.1. Theory

2.1.1. a = 0.50, b = 0.75 (hedley and grant with k = 1/3 UCS) (1)

2.1.2. half of the marks for naming the parameters and the other half-of the marks for discussion (3)

DRMS – Design Rockmass strength – a measure of rockmass strength based on both the UCS (insitu strength) and the MRMR (joint condition / blockiness). The higher the DRMS the stronger the pillar.

W – effective pillar width, wider is stronger.

H – Pillar height, higher is less stable (weaker).

2.2. Calculation (10)

Pillar strength: $\boxed{\textbf{Strength = DRMS * We^0.5 / H^0.7}}$

Rating (UCS) = 8 (From the table) **(1/2)**

DRMS = RMS = UCS x (MRMR – RUCS)/100 = 95 * (55 – 8)/100 = 44.65 MPa **(1)**

DRMS = RMS (adjustment factor assumed to be 1.0)

We = 4*area/perimeter = 4*(5*8)/(2*(5+8)) = 6.15 m **(1)**

H = 1.5m

Pillar strength = 44.65 * 6.15^0.5 / 1.5^0.7 = 83.37 MPa **(1)**

Pillar stress:

10° dip is shallow enough to ignore the effect of dip and assume the seam is horizontal. It is however more accurate to include the dip using the full APS formula

Virgin vertical stress Sv = pgh = 3000 kg/m3 * 9.81m/s2 * 250m = 7.36 MPa **(1)**

k=2, thus horizontal stress Sh= 2 * 7.36 = 14.72 MPa **(1)**

Pillar area = 8m * 5m = 40m2

Total area = (10 + 8)*(10 + 5) = 270 m2

Extraction ratio (e) = (270 - 40)/270 = 0.8518 = 85% **(1)**

Average Pillars stress APS = [Sv*Cos^2(10°) + Sh*Sin^2(10°)]/(1-0.85) = 50.55 MPa **(2)**

Also accepted (ignoring the dip)

Average Pillars stress APS = Sv / (1-e) = 7.36 / (1-0.85) = 49.07 MPa **(2)**

Factor of Safety

FOS = Strength / stress = 83.37 / 50.55 = 1.65 or FOS = 83.37 / 49.07 = 1.70 **(1/2)**

FOS > 1.6. Pillar is stable. **(1)**

NB! Only partial marks are given if units are not presented (i.e. MPa). Method marks allowed provided that the student is not reporting unrealistic numbers (i.e. virgin stress of 100MPa etc)

2.3. Calculation 2 (3)

Pillar strength stays the same = **83.37 MPa**

Pillar stress:

Vertical stress = same = 7.36 MPa

Horizontal stress = same = 14.72 MPa

Extraction ratio (e) = the same = 85%

Average Pillars stress APS = [Sv*Cos^2(35°) + Sh*Sin^2(35°)]/(1-0.85) = 65.21 MPa **(1)**

(No marks given for APS calc if the flat reef formula APS = Sv/(1-e) = 49.07 MPa is used)

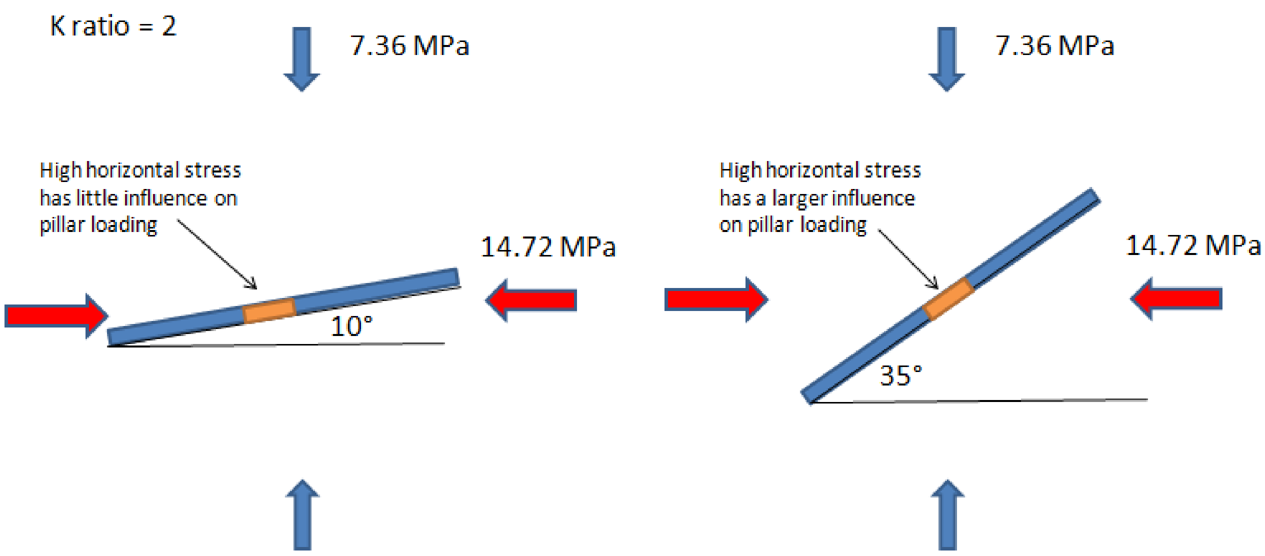
FOS = Strength / stress = 83.37 / 66.02 = 1.28 **(1)**

FOS > 1.0 BUT < 1.6. The pillar would need to be redesigned to have FOS >1.6. **(1)**

NB! Only partial marks are given if units are not presented (i.e. MPa)

2.4. Role played by stope dip (3)

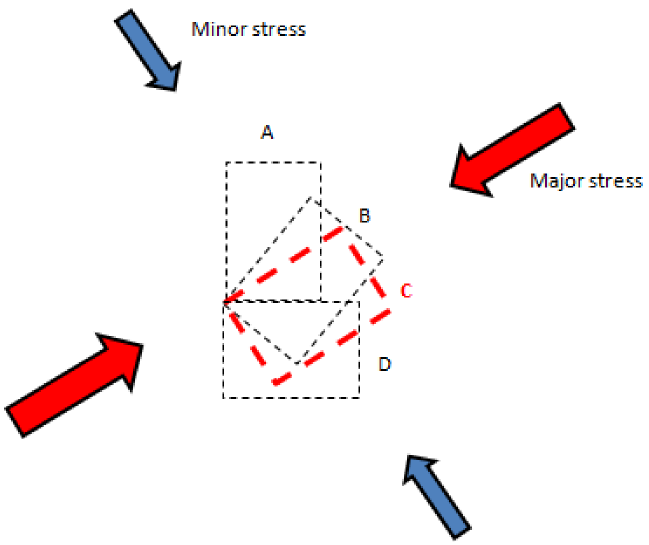
In this case study, the k ratio is 2 meaning that the horizontal stress is twice the vertical stress. As the dip of the chrome seam increases this higher 14.7 MPa horizontal stress starts to play a substantial role in pillar loading.



TOTAL : 20

Question 3

3.1. (1)



Option C. Major principle stress lies parallel to the chambers long axis – will reduce stress driven damage. Half mark given for option B.

3.2. (1)

Dominant jointing or other dislocations like faults and dykes weaken the rockmass and can drive instability (wedge and/or block failures). Large excavations should not be planned in areas where they will NOT intersect such structures. If intersection cannot be avoided, then the excavation must be orientated to minimise the possibility of structure related instability. One option is to orientate the excavation so that the shorter sidewalls align with (are parallel to) the strike of the dominant structure.

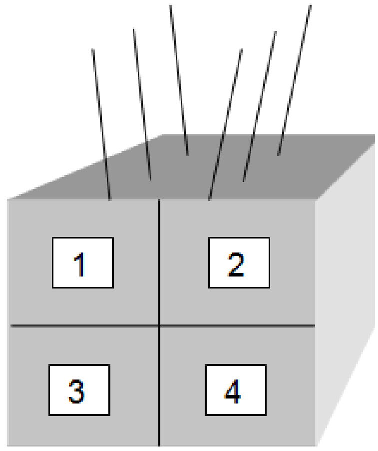
3.3. (4)

Any sensible approach will be accepted. Main points to consider are

- Break the larger excavation into smaller cuts to reduce unsupported spans during development
- A diagonal raise can be developed from the crosscut level to access the top cut.
- Start with a top cut 7.5m high and 7m wide. Install support as you advance (see 1 in sketch)
- It is BEST to install long anchor support now. Otherwise later at *
- Develop the second 7m wide top cut opening the span to 14m. Complete the hangingwall and upper sidewall support (see 2 in sketch).
- (*) Install remaining long anchor support in the hangingwall and in the sidewalls
- Develop the lower cuts (7.5m high) and install sidewall support as you progress. (see 3 and 4)

Key points.

- Reduce spans with smaller cuts (1)
- Start at the top and work down (1)
- Install support as you progress ensuring that the final spans are adequately supported before the full span is opened up (2)



3.4. (3)

As adapted from page 228 of the rock engineering HANDBOOK.....

<u>Explanations</u> (2 marks)

Primary support (which generally involves long grouted anchors) means tendons of sufficient length and density to support and stabilise the exposed rockwalls of an excavation in its **final dimensions**. **Secondary support** (which generally involves shorter but more densely spaced tendons with or without surface fabric) is designed to provide stabilisation and prevention of surface unravelling, both during excavation to protect the development crews and in the long term.

<u>How do they work together</u>? - READ the question! (1 mark)

Secondary support tendons bind the walls of the excavation together forming a strengthened beam. Primary support units (longer and of larger capacity) then anchor this beam into the solid and undisturbed rockmass. Surface fabric support like shotcrete or mesh acts as containment preventing the fallout of key blocks between secondary support tendons.

Note :- The terms primary support and secondary support are often used differently on the mines. Students were not penalised if they referred to long anchor support as being "secondary" provided that their discussion was sensible and clearly laid out.

3.5. (3)

<u>Calculate the required support resistance</u>

Volume to be supported per linear metre = 0.5 *(pi*7m*7m) = 77 m3

Weight to be supported per linear metre = 2.7 t/m3 * 9.81 m/s * 77m3 = 2039.5 kN.

Including the safety factor - Weight = 1.1 * 2039.5 = 2243.4 kN

= 2244 kN / m of excavation length.

Support resistance = 2244kN/m / 14m = 160.29 kN/m2 (1)

<u>Calculate the required long anchor spacing</u>

Each anchor provides 38 tons = 372.8 kN.

Anchors requited per m2 = 160.3 / 372.8 = 0.43

m2 per anchor = 372.8 / 160.3 = 2.33

Anchor spacing = √2.33 = 1.53m. **Therefore, Space anchors 1.5m apart.** (1)

Anchor length should be long enough to anchor above the assumed half-circle failure zone – i.e. should be > 7m. 8m long anchors would be advisable. (1)

(**Note:** in reality a 38ton anchor provides less capacity than 372kN due to stress loads induced on the cable at the anchored ends. Students who made reference to this were awarded one bonus mark. For the sake of this exercise 372.8 kN per bolt is accepted).

3.6. (2)

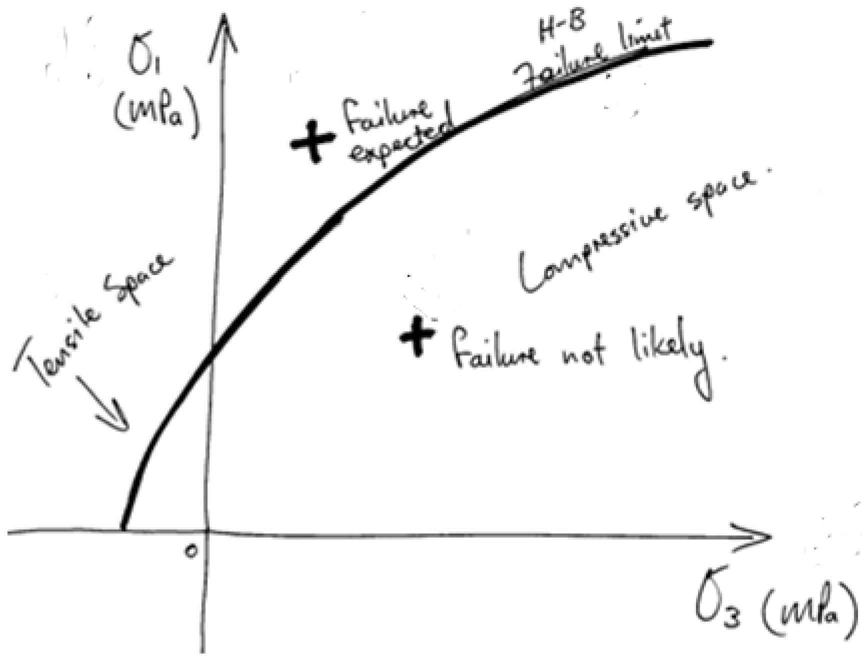
<u>Rule of thumb</u> suggests 7m long anchors spaced 3.5m apart.

<u>Calculation</u> suggests >7m long anchors **spaced 1.5m apart.**

The rule of thumb leads to insufficient support capacity because the anchor spacings are large. The rule of thumb is more accurately applied to smaller excavations (3 – 5m in width) and does not serve large excavation design to a sufficient degree of accuracy.

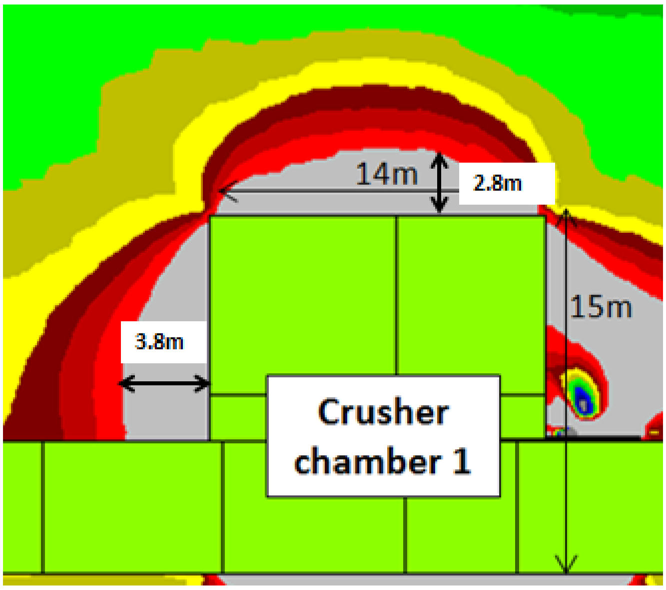
(**Note:** the applied calculation (half-circle failure zone) is however usually very conservative and it is better to apply other means when designing support for large excavations – see questions 3.7 – 3.9.

3.7. (2)



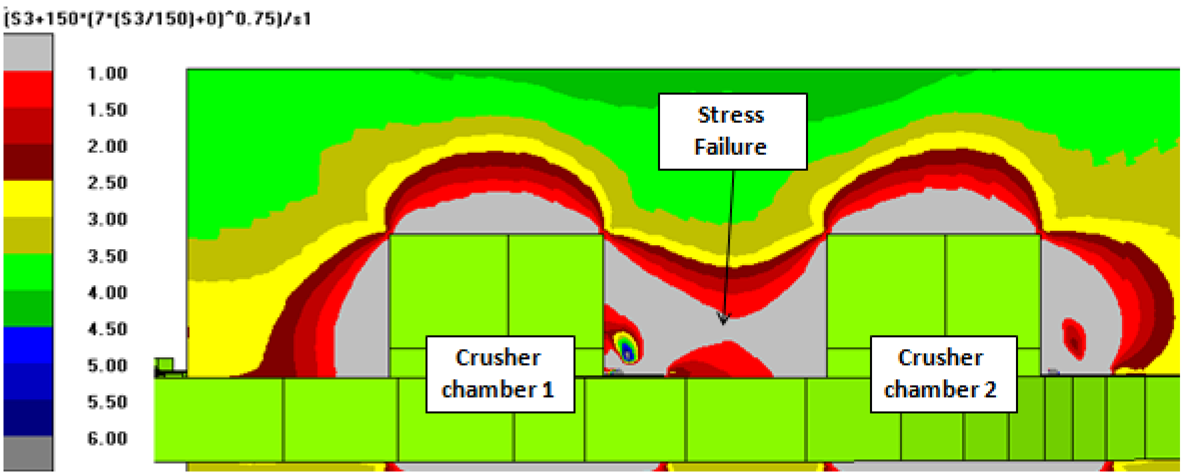
3.8. **(2)**

Using the drawing (figure 2) to scale using the provided width dimension, the depth of failure (H-B factor < 1.0 – the white grey zone around the chamber) is estimated at about 2.8m and 3.8m in the hangingwall and sidewall respectively as shown below. This is far less than suggested by the industry "half-circle" guideline which infers 7m in the hangingwall.



3.9. **(2)**

Stress interaction is expected between chambers 1 and 2. The chambers are too close together and should be moved further apart until failure is no longer predicted throughout the middling between them.



TOTAL : 20

Question 4

4.1. See HANDbook pgs 74, 163, TEXTbook pgs 273 and 384 (6)

As regional support backfill (any 3) **(3)**

- limits elastic deformation effectively reducing volumetric closure
- reduced volumetric closure results in
  - reduced energy release rate (ERR) on the face as well as
  - reduced size of excess shear stress (ESS) lobes that affect the stability of nearby geology.
- reduces stresses acting on stabilising pillars
- helps increase percentage ore extraction by allowing larger mining spans

As Local support backfill (any 3) **(3)**

- Stabilises the hanginwall by providing areal coverage on a large scale - Stops the dislocation of back area key blocks
- Absorbs energy (thus reducing damage in the stope) from nearby seismic events.
- Reduces closure - thus allowing elongates to work more effectively (also confines them making them stronger and able to provide active load longer)
- Prevent unwanted access into the back areas (forces early sweepings)

4.2. (4)

The **Particle size distribution** describes the spread of particle sizes within the solids from which the backfill is made. **Porosity** is a measure of the non-solid volume (voids and water) within the fill material expressed as a percentage of the total volume. (1)

As a general rule backfill stiffness and strength increases with reduced porosity. (1)

The sample size distribution directly affects the porosity of the material. The lowest porosity (and thus the strongest best performing backfill) is achieved when there is an optimum mix of both larger and smaller particles within the backfill medium. Small particles can fill the space between larger particles thus reducing the overall porosity. (2)

Note that only having larger particles or only having smaller particles does not on its own affect the porosity % much - there must be an optimum balance and blend of both smaller and larger solid particles.

4.3. (1)

Mix A has the lower porosity. Its stress-strain curve shows that it is stiffer and stronger than Mix B.

4.4. (3)

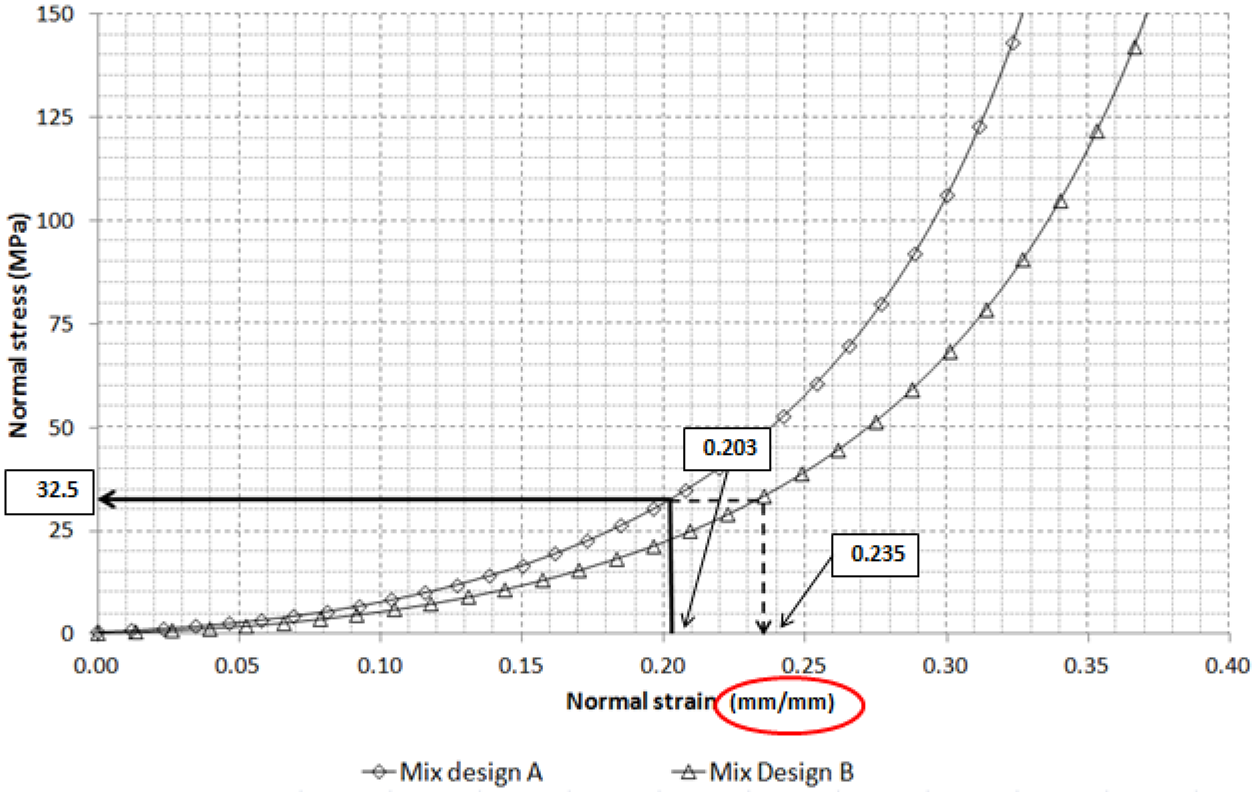
To determine the stress within the backfill at 20m behind the face the following process is followed.

- SW at time of placement 6m behind the face = 1.5m - (6*20mm) = 1.38m (1)
- Closure between 6m and 20m behind the face = 14m * 20 mm/m = 280mm (1)
- Strain in fill = 280mm / 1.38m = 203 mm/m or 0.203 mm/mm (0.5)
- Reading 0.203 strain off from the graph one notes that Mix A provides approximately 32.5 MPa of normal stress at 20m behind the face. (0.5)

See illustration on the graph below.

4.5. (1)

Reading 32.5 MPa stress off from the graph one notes that Mix B needs to strain 0.235 mm/mm to provide the same normal stress as mix A. See illustration on graph below.



Note The graph provided in the exam labelled the strain axis as %. This is an obvious error as these percentages are too low. The strain is actually shown in mm/mm (units of strain). Although it is expected that students should have found this obvious, students were not penalised for any confusion caused here.

4.6. (3)

To determine the required stope width we can work our previous calculation (4.4) backwards.

- From Q 4.5, the required strain when using Mix B is 0.235 mm/mm.
- Closure rate stays the same, thus the closure that takes place between 6m and 20m behind the face remains 280mm and the closure that takes place before backfill placement remains 6 * 20mm = 0.12m. (1)
- Strain = Closure induced / Stope width (at placement)
  - thus SW (at placement) = 280mm / 0.235 = 1191.5mm or 1.19m (1)
- Add the closure which takes place before backfill placement (from 0m to 6m) gives
  - SW (blasted) = Stope width (at placement) + 0.12m

    = 1.19m + 0.12m = **1.31m** (1)

The required blasted SW would thus be approx 1.3 metres, 0.2m less that if we used mix A.

4.7. **(2)**

The following can be done

- Smaller stope width (Q4.6)
- Install the backfill sooner (closer to the face)

The following conditions will help

- A higher closure rate in the stope.

TOTAL : 20

Question 5

5.1. (2)

Chapter 5.1 Locality

A brief description and locality map to indicate the location of the mine in relation to towns, existing infrastructure and any other relevant features such as mines sharing a common boundary, dams, rivers and any other topographical features which could influence the strategies adopted.

Chapter 5.2 Geological Setting

Geological structures, such as faults and dykes and stratigraphy, around individual orebodies or seams must be described and any hazardous conditions highlighted. A typical geological section of the mine must also be included. A detailed geological assessment may not be necessary but a map showing major geological features in relation to mining outlines and shafts must be included.

Chapter 5.3 Mining environment

The mining environment describes major subdivisions of an orebody, which dictates specific fundamental extraction strategies and occurs on a regional scale thus crossing mine boundaries.

Chapter 5.4 Ground Control Districts

The location and extent of ground control districts must clearly be described in the COP. The nature of the virgin stress field in which mining is to take place, as well as the occurrence of significant pore water and any other local geological features, must be included here. All ground control districts must be indicated on a Ground Control District Plan

Chapter 5.5 Mine Rock Fall and Rock Burst Incident Analysis

The COP must contain a tabulation of the mine's five year history of rock-related casualties (fatals, reportables and disabling incidents) and non-casualty incidents (where available), categorised according to rock bursts and rock falls per 1000 employees at work for both surface and underground operations. From this information the risk associated with each hazard can be established. These statistics should be normalised with respect to production tonnage in the different ground control districts.

Students to correctly name and briefly discuss at least 2 of the above. Marks deducted if no discussion (Read the question)

5.2.1. a) (1)

The terms ***hazard*** and ***risk*** are defined in section 102 of the Mine Health and Safety Act as

Hazard - Means a source of or exposure to danger.

*(Alternatively, a **HAZARD is something that has the potential to cause HARM.** This includes substances, machines, methods of work or other aspects of work Organisation; (The term danger is not defined in the Act))*

Risk - Means the likelihood that occupational injury or harm to persons will occur.

*(Alternatively, RISK **is the LIKELIHOOD that the harm from a particular hazard will occur**; The extent of the risk depends on not only the severity of the harm to a person but also the number of people who will be harmed. Risk therefore reflects both the likelihood that the harm will occur and its severity in terms of the degree of harm and the number of people harmed).*

5.2.1. b) (1)

Existing controls - Means those controls (physical, managerial or procedural etc) that are currently in place specifically to reduce the likelihood of harm due to a given hazard.

Additional controls - Means those controls (physical, managerial or procedural etc) deemed to be necessary (following the conducted hazard identification and risk assess (HIRA) process) and being in addition to those controls that are already in place, aimed at further reducing the risk to a more acceptable level.

5.2.2. a) (2)

Mine management to involve representatives from all applicable departments. Most important is :-

- Safety rep (of U/G crews) and representatives of the general work force NB!.
    - NB! This is essential to ensure chosen controls are practical, acceptable and achievable. **(Mark deducted if workforce representation is not mentioned)**
- Mine management (Mine manager and upper level management)
    - NB! This is essential to ensure chosen controls are practical, acceptable and achievable.
- Rock Engineer
- Safety Manager / Officer and other relevant persons

5.2.2. b) (2)

The team is to identify and assess all rock related hazards. This is often best done by classing hazards according to the various mining operations (drilling, blasting etc.)

A Risk Matrix type approach is often used whereby those who are carrying out the risk assessment categorise the **consequence**s of the hazard, and its **likelihood** separately and then combine them on a matrix to produce a priority based on a Risk Rating.

Each hazard is assigned a Risk Rating (RR) where

$$\text{RR} = \underbrace{\text{Probability (P) x Exposure (E)}}_{\boxed{\text{Frequency / Likelihood}}} \text{ x } \underbrace{\text{Consequence (C)}}_{\boxed{\text{Consequence}}}$$

Probability = Can/will the hazardous situation occur?

Exposure = Will people / machinery be exposed?

Consequence = Assuming a hazard causes harm to men or machinery, how severe will this harm be?

5.2.2. c) (2)

The objective is to **classify and rank rock related hazards in terms of risk** i.e. to identify those hazards that currently pose the highest risk to the operation. **RANKING** of the risk is a key objective. Furthermore the purpose is to :-

Determine all necessary control measures necessary to either **eliminate, control at the source or minimise** the risk; and insofar as the risk remains, provide for personal protective equipment; and institute a programme to monitor the risk to which employees may be exposed.

5.3. (Deep Level minign) (6)

Management of closure, seismicity, ERR and ESS **(4 marks)**

Refer to Chapter 3 (p 45 - 53) of the HANDBOOK.

a) How are closure, ERR, ESS and seismicity related? (1)

- In deep environments virgin stresses are high and lead to high closure and thus very high face / abutment stresses after mining. High induced stresses can then lead to seismicity both on the faces and on nearby structures. Optimum stoping layouts designed to reduce face stress levels are thus vital in deep level mining environments.
- The most effective way of controlling face stress (and thus seismicity) is to reduce or control **mining span.** Managing or controlling mining span reduces elastic convergence (and thus closure) which in turn reduces the likelihood of severe seismic activity. ERR and ESS are engineering concepts used to quantify the risk of face related and structure related seismicity respectively.
  - Energy Release Rate (ERR) is an engineering criterion used to assess average face stress levels and is an indicator of possible seismic incidence in deep **geologically-undisturbed** mining situations. Lower values of ERR are better (typically ERR < 30 MJ/m2 is targeted).
  - Excess Shear Stress (ESS) is an engineering criterion used to give a semi-quantitative assessment of the affect of stope layouts on the seismic potential and stability of geological structures. Positive ESS suggests that shear failure of a structure is possible. The stope layout must limit the size (area on the fault plane) and magnitude of positive ESS lobes on geological structures.

Students were lucky - I could have made part a) above count 2 marks as I clearly asked for this! Students DONT READ the question!

b) Possible controls(any 3 as long as you say what the control helps to control) (3)

Agent (being controlled)	Control
Mining Span	Stabilising pillars / regional support pillars ▪ The deeper the mining (and the higher the stress) the wider these pillars must be to reduce pillar stresses. Typical widths range between 20m and 40m. ▪ The deeper the mining (and the higher the stress) the closer together these pillars should be spaced to reduce the mining spans. Typical spacings range between 180m and 240m. ▪ Fault losses and other "unpay" pillars also contribute to regional support.
Volumetric Closure	Stabilising pillars / regional support pillars Backfill Monitoring - closure / Ride stations **(NB! not packs or other stope support!!!!!)**
ERR	Stabilising pillars / regional support pillars Backfill **Preconditioning** Numerical modelling
ESS	Stabilising pillars / regional support pillars Backfill **Bracket pillars on structure** **Seismic network (geophones + accelerometers)** Numerical modelling

Protection of vertical shafts and off reef development **(2 marks)**

The following possible discussion points :-

- Shafts are protected either by pre-mining shaft pillar extraction or by the leaving a sufficiently large shaft pillar so that induced stresses and strains at the shaft locations are below critical limits and will not result in damge to the shaft lining or steelwork. Shaft are further protected by
  - Proper placement of shaft pillars (away from major structure)
  - Mining away from the shafts
- Off reef LOM tunnels are protected by placing them deep enough in the footwall (or hangingwall) so that they are not influenced by induced abutment stresses - typically 60 - 100m below/above the reef in deep environments. Furthermore the location and orientation of tunnels are planned so as to **minimise the risk of stress and/or seismically driven damage** :-
  - Proper approach to structures (not parallel to or too close to major structures)
  - Siting of tunnels in the most competent areas of the geological sequence (strongest rock type)
  - Proper tunnel support to cater for both static and dynamic conditions.

5.4. (Shallow) (4)

Mining approach to geological features **(2 marks)**

The location and orientation of excavations are planned to **minimise risk of structure related instability**:-

- Proper approach to structure. - Do not develop dives parallel to major joint sets or structures as this will result in possible instability given that the structure will be exposed in the tunnel hangingwall over excessive lengths. Mining direction perpendicular to the strike of the structure.
- Mining direction to be opposite to the dip of the structure.
- Siting of tunnels in the most competent areas of the geological sequence (strongest rock type and away from major **shallow dipping** structures)
- Proper tunnel support to cater for static conditions. Proper support of the weak side of exposed structures to prevent wedge failures.
- Shallow dipping structures / jointsets pose a higher risk - support wedges.

Protection of declines and LOM excavations **(2 marks)**

Declines and LOM excavations in shallow mining environments are typically on-reef and are protected by

- Proper tunnel support design based on the rockmass quality to ensure stability over the long term. Typically longer tendons with areal support where necessary.
- The leaving of large decline protections pillars (either side of declines) which protect the declines from the effects of instope collapses and back breaks.

TOTAL : 20