CHAMBER OF MINES OF SOUTH AFRICA

EXAMINATION FOR THE CERTIFICATE IN ROCK MECHANICS

PAPER 2

METALLIFEROUS OPTION

SUBJECT: CERT. IN ROCK MECHANICS PAPER 2 (METALLIFEROUS OPTION)	**EXAMINER:** L J GARDNER
SUBJECT CODE: COMRME	**MODERATOR:** M K C ROBERTS
EXAMINATION DATE: OCTOBER 2006	**TOTAL MARKS:** [100] **PASS MARK:** 60%

NUMBER OF PAGES: 7

SPECIAL REQUIREMENTS:

1. **Answer *ALL THE* questions**
2. References other than those provided are not permitted. (Refer No. 10)
3. Hand-held electronic calculators may be used.
4. Put your examination 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.

5. Write in ink on the **RIGHT HAND SIDE** of the paper only.
6. Show all calculations on which your answers are based.
7. Illustrate your answers by sketches of diagrams wherever possible.
8. In answering these questions, full advantage should be taken wherever necessary of your practical experience as well as of the data given.
9. Answers must be given to an accuracy which is typical of practical conditions.
10. A list of formulae and any other essential charts and tables will be issued at time of the examination.

QUESTION 1 – GEOTECHNICAL INVESTIGATIONS

A main development end on your mine has intersected poor ground which has slowed advances. An exploratory 60 m long diamond drill hole, with **orientated drill core**, has been drilled ahead of the face to gather information on expected ground conditions. You must analyse the orientated core recovered from the hole.

You record a **total** of 380 joints in the core. There appear to be two distinct sets of strike joints, dipping at roughly 20 and 80 degrees respectively. While the one set is rough and undulating, the other is smooth, planar and slickensided. Traces of micaceous filling, varying in thickness from 0 – 3 mm, are visible in the joint planes. No water seems to be present. There also appears to be a 3.0-metre thick weathered dyke running through the middle of this zone, at an angle of 45 degrees.

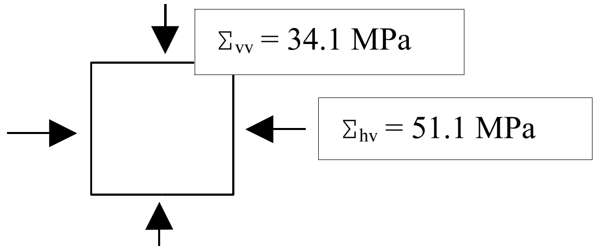
1.1 **Calculate the rock mass quality** based on the Q-system”, using the tables provided in appendix 1. Motivate your reason for each value chosen for use in the rock mass rating. (8)

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

RQD = 115 – (3.3 x (380joints / 60m)) = 94.1, can round up to 95
Jn = 4 (2 distinct joint sets)
Jr = 0.5 (slickensided, planar)
Ja = 2 (Joint filling > 1 mm)
Jw = 1 (dry)
SRF = 2.5 (One shear, fault, dyke or weakness zone)
Q = 2.375

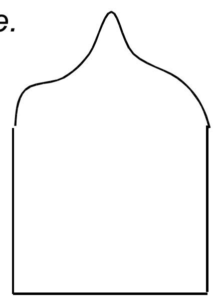
1.2 The development end mentioned above has dimensions of 4.0 m x 4.0 m and is being developed 1120 metres below surface. The host rock has a density of 3.1 t / m^3, and a 120 MPa UCS. Recent stress measurements showed an average k-ratio of 1.5.

a) **By means of an annotated section** through the tunnel, show the magnitudes of the horizontal and vertical stresses acting on the tunnel.



b) Would you expect fracturing of the rock walls surrounding the tunnel? If so, **explain why**, **show where** you would expect it and the likely shape that the tunnel would form. (6)

In the simplest terms, you would expect fracturing as the virgin stress magnitude exceeds 1/3rd of the host rock UCS. Based on the high k-ratio, the fracturing would occur in the hanging wall, resulting in the tunnel forming a "Gothic arch" type profile.



Examine the rock property test result sheet shown in Appendix 2 and **determine**:

a) The UCS of the sample – *125 MPa*

b) The axial strain at failure – *900 microstrains*

c) The Elastic Modulus at 50% of the UCS – *140 GPa*

d) The Poisson's ratio at 30% of UCS – *0.22*

e) Does the sample exhibit linear, strain-hardening or strain-softening behaviour? **Motivate** your answer (6)

Sample exhibits linear behaviour – stress / strain curves are generally a straight line over the length of the test, also shown in consistency of the Youngs modulus and Poisson's ratio curves

20 MARKS

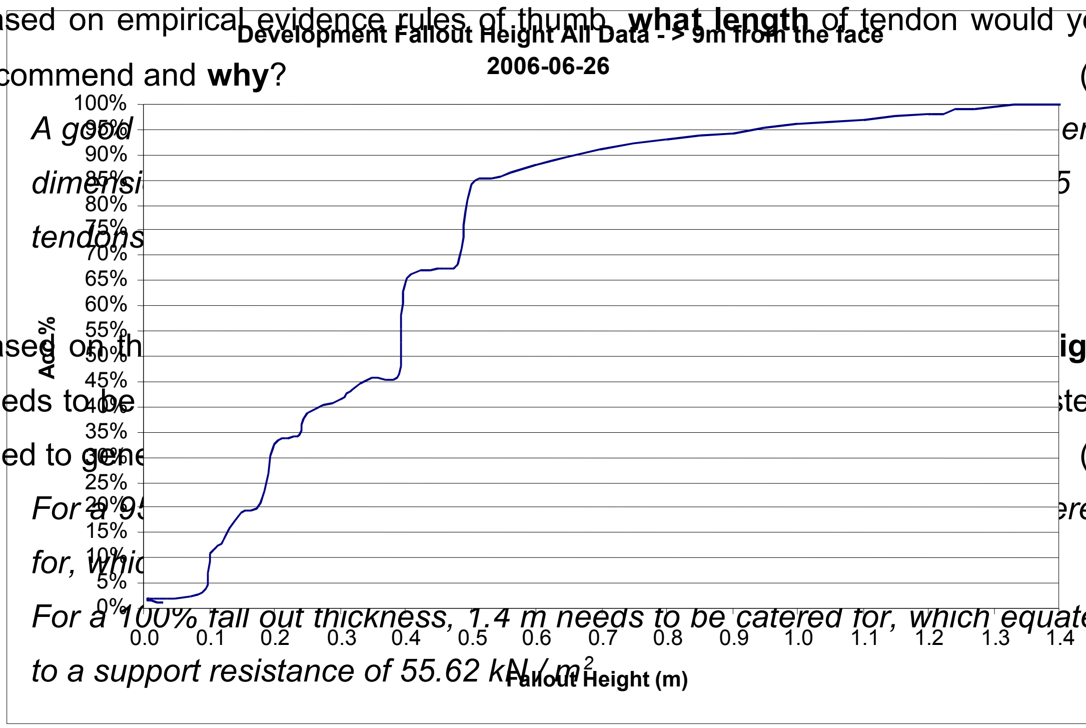
QUESTION 2 – SUPPORT DESIGN

You have been contracted by the manager of a **small, shallow mine with a low profit margin** to design a suitable tendon support system for their development ends. The following information has been provided:

- The mine manager wishes to have an active support system on the face, as there have recently been a number of Fall of Ground accidents.
- The rock in which development takes place is moderately to heavily jointed, and most falls are bounded by geological discontinuities.
- The average adjusted rock mass rating in present development ends is 45, using the Laubscher MRMR system.
- The average development end size is 3. 0 m wide x 3.5 m high.
- A search of development end Falls of Ground from the mine's FOG database (+/- 500 FOGs), provided the information shown on the graph below.

2.1 Based on empirical evidence rules of thumb, **what length** of tendon would you recommend and **why**? (2)

A good ... end dimensi ... 5 m tendons ...



2.2 Based on th... ight needs to be ... stem need to gene... (2)

For a 9... ered for, whic...

For a 100% fall out thickness, 1.4 m needs to be catered for, which equates to a support resistance of 55.62 kN / m^2.

2.3 Barton recommends tendon lengths based on the formula:

L = <u>2 + (0.15 x B)</u>

ESR

ESR

For an excavation / span ratio of 2, **what length** of tendon would be required according to this formula? (2)

L = (2 + (0.15 x 3.0)) / 2.0 = 1.225 m

2.4 In light of the above calculations, **compile** a tendon support system for the mine, **specifying**:

- Type of unit to be installed

Must be an active support type, so can be one of:

– *mechanically-anchored rockstuds*
– *resin-grouted rebar with fast-setting capsules*
– *Hydraulically-prestressed tendon (X-Pandabolt or Swellex)*

Based on cost and ease of use, the simplest choice would be a mechanically-anchored rockstud, which can be post-grouted if required

- Length of unit

Depending on the approach adopted, the initial length can vary from 1.0 m (for a mechanically-anchored rockstud installed at 90^o to hold up the 96% fall out height) to nearly 3.0 m (for a hydraulically prestressed unit with an 0.8 m bond length, installed at 55^o to hold up the "rule of thumb" 1.75 m). Two realistic scenarios would be the mechanically-anchored rockstud with an 0.15 m anchor length installed at minimum 70o, intending to keep up either the 0.9 m 95% fall out height, or the 1.225 m proposed by the Barton formula – this gives tendon lengths of 1.1 - 1.45 m. A wide range of answers up to 1.75 m was accepted, based on the angle of installation

- Spacing of tendons

Although no information o the sizes and orientation of falls is given, the RMR of 45 and statement regarding "moderate to heavily jointed" rock mass implies that tendons should densely, rather than widely, spaced. An initial spacing of 1.0 m x 1.0 m is a good start. This should be backed up by further underground investigation.

- Minimum angle of installation

  While a number of candidates stated a 90^o installation angle, this is practically impossible on a mine employing conventional jackhammers and airlegs, as drilling water and dust falls into the rockdrill operator's eyes. A 70^o angle is a practical, good starting point. The implementation of the "90o to strata" rule also gained marks.

- Necessary quality control measures

  Visual inspection, planned task observation, pre-load indicators, checking tension by spanner or torque wrench, workplace audits, etc, all gained marks

Give an indication of what the system will cost in terms of R / m^2. In all cases your answers must be properly motivated (10)

Answers ranging from R 25-00 / m^2 – R 50-00 / m^2 were accepted

2.5 **What additional** on-the-face support measures / systems can the mine manager call on to ensure worker safety, should ground conditions deteriorate further? (4)

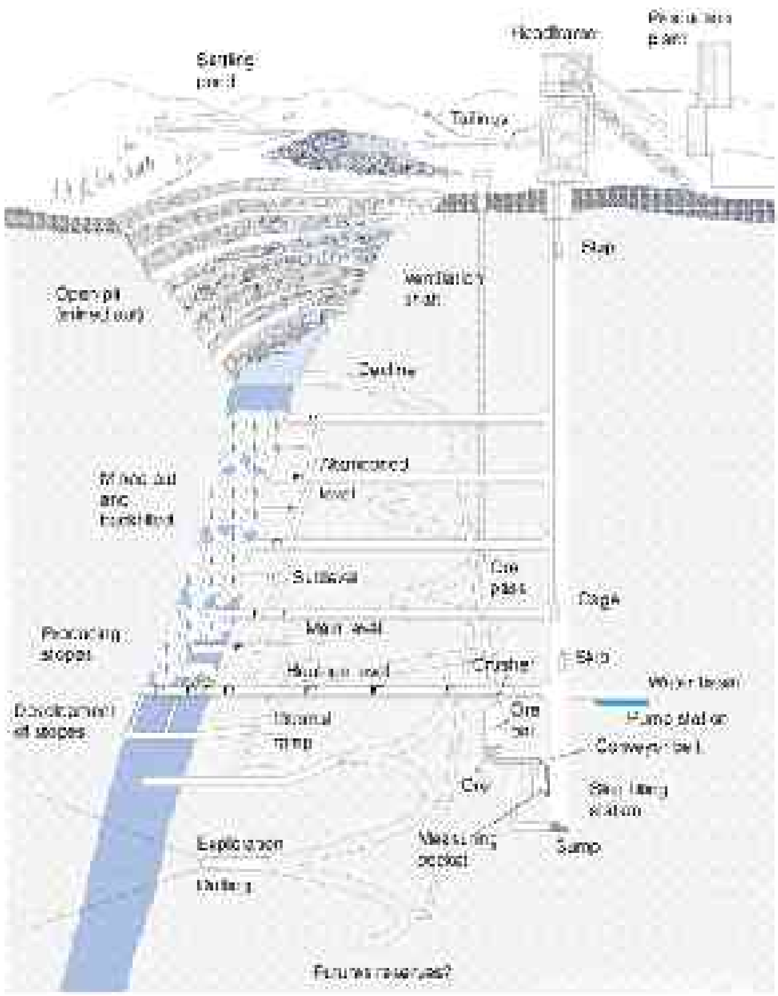
Temporary support, on-face nets, straps / mats or wire mesh sheets, areal support such as Thin Sprayed Liners or shotcrete, follow-on meshing and lacing, cable anchors, spiling, steel sets, etc, etc… Marks were given for any of the above. Marks were NOT given for changes in blast round, smooth wall blasting, etc, as the question specifically refers to support systems.

20 MARKS

QUESTION 3 – MINING LAYOUTS

The diamond mine on which you are employed is mining an 800 m diameter diamond pipe by means of an open pit. The pit is reaching the end of its economic life, and the mine is planning to continue mining by means of underground operations.

As part of the planning team, you are required to **make diagrams and notes** concerning the rock mechanics and rock engineering considerations related to the different areas of the projects.



Your notes should include:

3.1 Considerations for locating a vertical shaft system (4)

Beyond the zone of influence of the open pit and orebody, balance between location in stable ground and distance to access orebody, outside the orebody in country rock, reasonably adjacent to the present infrastructure, in competent ground

3.2 Methodology for accessing the orebody from the shaft system (4)

See diagram above – can use different levels, or spiral ramp, or combination of these two approaches

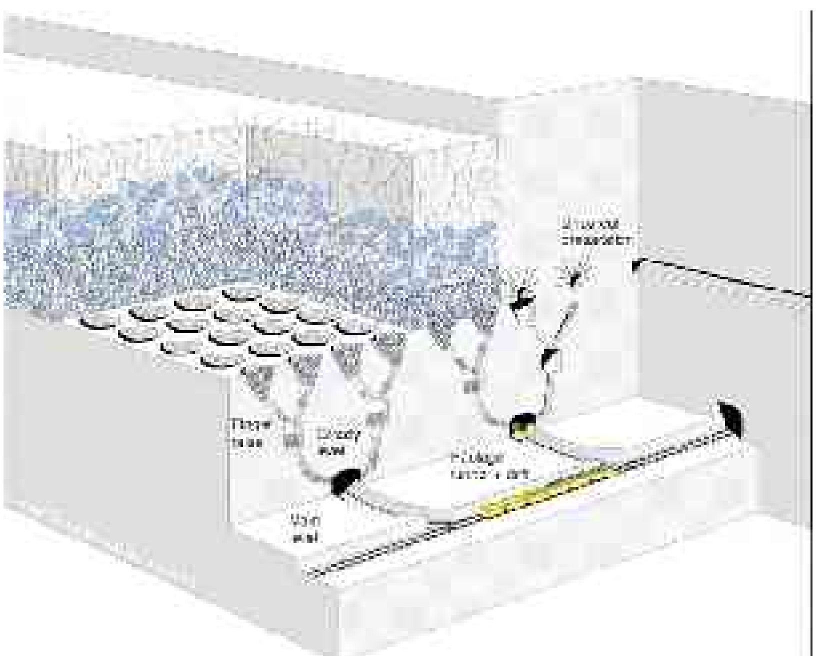
3.3 Proposed mining method (sketch / diagram essential) (6)

See notes below

3.4 Appropriate support measures for different excavations (6)

Should have mentioned:

- *Shaft - Split sets, cladding, shotcrete if needed, concrete lining*
- *Access and orebody tunnels – tendons, areal support such as strapping, shotcrete, TSLs, or meshing and lacing, cable anchors, steel sets*
- *Drawpoints – tendons, cable anchors, strapping, shotcrete, steel sets*



20 MARKS

Block-caving is a large scale production mining method applicable to:
Low grade, massive orebodies with:

1. *Large dimensions both vertically and horizontally*

2. *A rock mass that behaves properly, breaking into manageable size blocks.*

3. *Surface which is allowed to subside.*

These rather unique conditions limit block-caving applications to special mineral deposits. Looking at world-wide practise, we find block-caving used in iron ore, low grade copper and molybdenum mineralisations, and diamond-bearing kimberlite pipes. Large tonnage produced by each individual mine makes block-caving mines real heavy-weights when compared to other mines

Description
Block caving is based on gravity combined with internal rock stresses, to fracture and break the rock mass in pieces, which can be handled by miners. The drilling and blasting required for ore production is minimal, while development volume is massive. "Block" refers to the mining layout, which divides the orebody in large sections, blocks, with areas of several thousand square meter. Caving of the rock mass is induced by undercutting the block. The rock section underneath the block is fractured by blasting, which destroys its ability to support overlaying rock. Gravity forces, in the order of millions of tons, act on the block. Fractures spread to affect the whole block. Continued pressure breaks rock into smaller pieces, passing the drawpoints, where the ore is handled by LHD-loaders.

Development
Development for block-caving applying conventional gravity flow involves:

1. *The undercut, where rock mass underneath block is fractured.*
2. *Drawbells underneath the undercut, gathering rock into finger raises.*
3. *Finger raises, collecting rock from draw-bells to the grizzlies.*
4. *A grizzly level, where oversize blocks are caught, and treated.*
5. *A lower set of finger raises, channelling ore from grizzlies to chutes for loading.*
6. *The lowermost level prepared for train haul age and chute loading.*

The finger raises are arranged as branches of a tree, gathering ore from a large area at the undercut level, further channelling material to chutes at the haulage level.
Openings underneath the block are subject to high internal rock stresses. Drifts and other openings in the block caving mine are excavated with minimum cross

sections. Still, heavy concrete lining and extensive rock bolting is necessary, to secure the integrity of mine drifts and drawpoint openings.

Production

After completion of the undercut, the rock mass above begins fracturing. The blocks are gathered by draw-bells/crates and funnelled down through finger raises. The intention is to maintain a steady draw from each block. Miners keeping records of volumes extracted from individual draw-points. Theoretical, no drilling and blasting is required for ore production. In practise, it is often necessary to assist the rock mass fracturing, by longhole drilling and blasting in wide spaced patterns.

Boulders which must be treated with drilling and blasting, is a frequent disturbance. Large blocks cause hang-ups in the cave, which are difficult and dangerous to tackle.

Ore handling

Original block-caving techniques relies to 100 % on gravity, to deliver ore from the cave, into rail cars. The ore funnelled through a system of finger raises and ore passes, ending trough chutes at the main haulage level. As chute loading requires controlled fragmentation, rock has to pass the grizzly before entering the ore pass. The grizzly-man, with his sledgehammer, used to be the bottleneck in old-style block-caving mines. Today, hydraulic breakers are installed to treat boulders, and with LHD loaders assisting ore handling, dealing with oversize rock becomes more efficient. Today, block-caving mines have adapted trackless mining, using LHD-loaders to handle the cave in drawpoints. As a consequence a ventilation level is added to development preparations, to clear the production level from diesel exhaust. The LHD loaders are able to handle large size rock, oversize boulders blasted in drawpoints.

QUESTION 4 – ROCKPASS REHABILITATION

As the rock engineering practitioner on a deep-level gold mine, you are notified that one set of the shaft's main rockpasses have holed into one another as a result of excessive scaling and deterioration.

4.1 **Name** four factors that possibly contributed to this occurrence, and **describe** how each factor could contribute to the deterioration of the rockpass surfaces? (8)

- *Stress – stress-induced fracturing*
- *Geology – loss of cohesion, planes of weakness*
- *Abrasion – removal of loose rockwall and increase in dimension*
- *Water – reduce rock strength, lubricate joints and fractures*
- *Time-related deterioration – increase effect of above factors*

4.2 **With the aid** of sketches**, describe** three possible methods for rehabilitating the rockpasses and **comment** on the suitability of each method for ensuring long-term stability in view of the circumstances listed above (9)

- *Reinforcement of present excavation – normally employed in cases where deterioration is detected early, or where sufficient middling exists between the ore pass and adjacent excavations to provide stability. The orepass is reinforced to accommodate its enlarged size and shape, not restored to its original dimensions.*
- *Installation of tube and backfill – normally employed in cases where deterioration is severe or ongoing, or where the ore pass has inadvertently holed into another excavation. The use of a tube means that the ore pass is returned to a size similar or possibly smaller than its original dimensions, normally in the same position as original ore pass.*
- *Fill and redevelopment through fill – a variation on complete replacement, this method entails filling the ore pass with cemented waste rock and then redeveloping through the fill, usually by raiseboring.*

4.3 To prevent similar occurrences in the future, **list** three practical warning signs that would provide an indication of ore pass deterioration (3)

- *Obvious increases in ore pass dimension*
- *Irregular ore flow through the ore pass*
- *Large waste rocks blocking the control chutes at bottom of the ore pass*
- *Mixing of reef and waste in supposedly separate ore passes due to ore passes holing into each other*

20 MARKS

QUESTION 5 – PILLAR PROPERTY COMPARISON

Pillars are traditionally classified as crush, yield or rigid pillars, based on their behaviour under load.

5.1 Using the attached graph paper, **sketch** the typical load / deformation curves of these three pillar types. Make sure that you **show and name** important points on each of the curves. (6)

See figure on page 46 / 146 of Jager and Ryder's "Rock Engineering Practice for Tabular Hard Rock Mines"

5.2 **By means of a table, compare** the following aspects of the three different pillar types:

- Typical width: height ratio
- Typical values for peak strengths (give relevant rock type)
- Typical values for residual strengths (give relevant rock type)
- Typical mining application or depth range (14)

Pillar type	***Crush***	***Yield***	***Rigid***
Width: height ratio	*2:1*	*3 – 5:1*	*10:1 or more*
Peak strength	*Approx same as UCS - 100 MPa (Merensky)*	*220 MPa (Merensky)*	*Infinitely strong*
Residual strength	*Approx 10% of peak strength - 20 MPa (Merensky)*	*Should be close to peak strength - 200 to 220 MPa (Merensky)*	*Not applicable*

Typical application / depth range	*Support tensile zone / weak parting as in-stope support*	*Support tensile zone / weak parting as in-stope support*	*Barrier pillars, regional pillars*

20 MARKS

APPENDIX 1 – ROCK MASS RATING VALUE TABLES

	STRESS REDUCTION FACTOR	
A	No shear, faults, dyke or weakness zone	1
B	One shear, fault, dyke or weakness zone	2.5
C	One shear, fault, dyke or weakness zone with blocky ground conditions	4
D	Curved joints or dome structure approaching a pothole, reef roll or OPL's	7.5
E	Curved joints or dome structure approaching a pothole, reef roll, OPL's with blocky ground conditions	8
F	Many faults, dyke and weakness zones.	9
G	Wide shear zone	10

	JOINT ALTERATION NUMBER	
A	Tightly healed, hard rockwall joints, no filling	0.5
B	Slight infill, coating < 1mm	1
C	Joint filling > 1 mm	2
D	Joint filling > 3 mm	4
E	Zones or bands of disintegrated or crushed filling, open joints.	8

	JOINT SET NUMBER	
A	Massive, no to few joints	1
B	One joint set	2
C	One joint set plus random joints	3
D	Two joint sets	4
E	Two joint sets plus random joints	6
F	Three joint sets	9
G	Three joint sets plus random joints.	12
H	Four or more joint sets, random, heavily jointed.	15
I	Crushed rock	20

	JOINT WATER	
A	Dry	1
B	Dripping water	0.5

	JOINT ROUGHNESS NUMBER	
A	Discontinuous joints	4

B	Rough or irregular undulating	3
C	Smooth undulating	2
D	Slickensided, undulating	1.5
E	Rough or irregular planar	1.5
F	Smooth planar	1
G	Slickensided planar	0.5

APPENDIX 2 – ROCK PROPERTY TEST