EXAMINATION PAPER

SUBJECT: CERTIFICATE IN ROCK MECHANICS PAPER 3.1 : HARD ROCK TABULAR **SUBJECT CODE:** COMRME **EXAMINATION DATE:** May 2013 **TIME:** 9:00 – 12:00	**EXAMINER:** L J GARDNER **MODERATOR:** JJ THOMPSON TOTAL MARKS: [100] PASS MARK: 60%

NUMBER OF PAGES: 10

SPECIAL REQUIREMENTS: Answer any *FIVEof the SIX* questions. Only the first **FIVE** questions answered will be marked. References other than those provided are not permitted. Hand-held electronic calculators may be used. 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.** Write in ink on the RIGHT HAND SIDE of the paper only (only the right hand pages will be marked). Show all calculations on which your answers are based. Illustrate your answers by sketches of diagrams wherever possible. In answering these questions, full advantage should be taken wherever necessary of your practical experience as well as of the data given. Answers must be given to an accuracy that is typical of practical conditions.

QUESTION 1 – GEOTECHNICAL INVESTIGATION

The mine on which you are employed is planning to extend its current operations, which presently stop at 1200 metres below surface, by means of a sub-level decline system. As part of this project, a large chamber must be blasted to accommodate an underground refrigeration plant. When complete, the chamber will measure 100 m long x 12 m wide x 12 m high. The area in which the chamber will be located is presently accessed by a 3 m x 3 m development crosscut. As the mine's rock engineering practitioner, you are requested to define the geotechnical environment as input into further design of the chamber.

1.1 **By means of short notes and sketches**, **describe** how you would go about this task, including:

a) Geology-related investigations (4)

b) Geotechnical investigations (7)

c) Rock property testing (4)

a – Geological investigations (ANY 4 OF THE FOLLOWING WITH SKETCHES)

- *Site mapping and / or core logging*
    - *Basic rock type (sedimentary, igneous, metamorphic)*
    - *Stratigraphy*
    - *Middling to reef horizon/s*
    - *Weathering / oxidation*
    - *Presence of ground water*
    - *Structures*
        - *Faults*
        - *Dykes*
        - *Shear zones*
        - *Unusual features / zones*
- *From data gathering, produce various models (in 2- and 3-D, with dip and strike where applicable):*
    - *Stratigraphic column*
    - *Structures*

b – Geotechnical investigations (ANY 7 OF THE FOLLOWING WITH SKETCHES)

- *Site mapping and core scrutiny and logging*
    - *RQD*
    - *Rock mass ratings (extra ½ marks for sub-sections)*
    - *Discontinuity frequency, orientation, nature*
    - *Joint / fracture angle distribution*
    - *Local k-ratio (from blast hole sockets)*
- *Borehole camera inspections*
    - *Presence of prominent layers/features in proximity of current excavation*
    - *Depth of fracturing around current access excavation*

- *Stratigraphy*
    - *Relative strengths*
    - *Thickness and consistency*
    - *Nature of boundaries / contacts*
- *Produce models:*
    - *Rose diagrams of joint / fracture angles*
    - *Stratigraphic isopachs*
    - *Geotechical zones / areas*
    - *Boundaries between different horizons*

c – Rock property tests (ANY 4 OF THE FOLLOWING WITH SKETCHES)

- *Field tests*
    - *Point load tests*
    - *Schmidt hammer*
- *Laboratory tests*
    - *Uniaxial Compression tests (to verify field testing)*
    - *Uniaxial or Induced Tension tests*
    - *Triaxial Compression, at 3 / 4 suitable confining pressures*
    - *Shear tests on joints / stratigraphic boundaries*
    - *If needed, swell and slake tests*
- *Use input from above to determine failure criteria*
- *From laboratory tests, also determine other mechanical properties:*
    - *Density*
    - *Young's Modulus*
    - *Poisson's Ratio*

1.2 As part of the rock property testing, you must submit samples for testing. **List** five aspects that you must pay attention to in the selection of samples (you may assume that the samples come from drilled borehole core) (5)

- *Representative of intended test area / rock type*
- *Orientation of samples in relation to layering or bedding, particularly in cases where anisotropy may play a role*
- *Core to be of suitable size (preferably approx 54 mm or more in diameter)*
- *Core sample length 2.5 – 3 x core diameter*
- *Core diameter / sample size to be greater than 10 x largest grain size*
- *Core / sample sides to be smooth and straight*
- *Samples to be free of visible discontinuities*

(20 marks)

QUESTION 2 – MINING LAYOUTS

As part of the design team planning a new intermediate depth shaft for a new mine, you are required to conduct an early shaft pillar extraction in order to minimise the complications associated with a conventional shaft pillar. The shaft is intended to be 10m in diameter, concrete-lined and fully equipped to hoist personnel, material and rock. The shaft's final depth will be 2000m, but it will intersect the tabular orebody (1.5m channel width, 10^{o} dip) at 1500m.

2.1 Using diagrams and short notes, explain how you would propose to go about this task. Your notes should include:

a) Location of the reef cut in relation to other levels and infrastructure (5)

Reef should ideally be located ½ way between 2 levels, or at least a reasonable distance (20 m?) above / below the nearest level

Excavations such pump and hoist chambers or fridge plants should be located well away from the reef horizon, on either upper or lower levels

Provision must be made for either overstoping or placing within a pillar such excavations as station tips, station crosscuts, main ore passes and drain holes

Main intake and return airways should be located well away from the inner pillar extraction area

Potential to pre-extract pillar before shaft reaches reef, such as at South Deep

b) Dimensions, extraction sequence and support strategy for the inner cut (5)

The inner cut will have to be cut from the shaft itself during the sinking phase, which impacts on sinking time.

A minimum of 5 m should be cut around the shaft, to give a cut of between 20m x 20m and 30m x 30m.

The cut should be extracted either in a circular or square shape, using some logical sequence that can be blasted and cleaned (establish small gullies and faces, etc).

Typical support should consist of tendons combined with yielding elongates, followed by yielding cementitious packs or some form of soft fill (e.g. low strength, but cemented backfill).This must NOT be placed directly on the shaft edge, but be located a small distance off it.

c) General sequence for mining the outer section of the pillar area (5)

The sequence for the outer pillar should aim to mine radially away from the shaft in a consistent fashion. This can be achieved in a number of ways, the simplest being to establish raises either side of the inner pillar from which to stope.
Of importance is that faces should “bombshell” away from the shaft at the same time in an orderly fashion, with minimal leads & lags, or deviation from the planned pattern.

d) Location and design criteria for satellite pillars (5)

The aim of the satellite pillars is to reduce the amount of closure across the shaft pillar area, and absorb the stress generated by the extraction without transmitting it back onto the shaft barrel.
The satellite pillars should thus obviously not be located over major infrastructure, and should ideally be symmetrically laid out around the shaft
Design criteria could include the following:

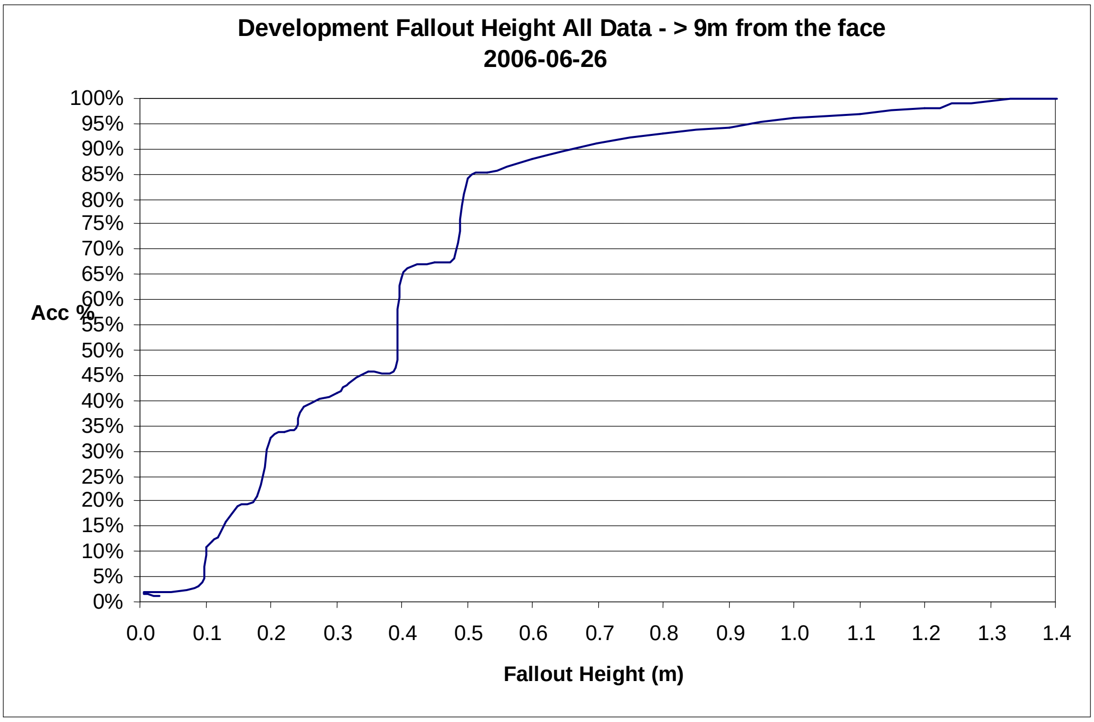
- *Minimum width: height ration of 10:1 (must be stable pillars)*
- *Maximum stress level not to exceed 2.5 x UCS of host rock*

(20 marks)

QUESTION 3 –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 (joint spacing 0.75m – 1.2m), and most falls are bounded by joints.
- The average adjusted rock mass ratingin 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.
- Development takes place via conventional means using jackhammers and air legs
- A search of development end Falls of Ground from the mine's FOG database (+/- 500 FOGs), provided the information shown on the graph below.



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

A good starting point would be 1.5 m long tendons, based on the "½ the end dimension" rule of thumb, although a case could be made for 1.75 m tendons based on the higher side walls

3.2. Based on the information provided by the FOG database, **what fall out height** needs to be catered for, and **what support resistance** does the support system need to generate if the average rock density is 4.05 t / m^3 (2)

For a 95% fall out height (as per COP guidelines), 0.9 m should be catered for, which equates to a support resistance of 35.76 kN / m^2

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

3.3. Barton recommends tendon lengths based on the formula:

$$L = \frac{2 + (0.15 \times B)}{ESR}$$

For an excavation support 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

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

- Type of unit to be installed (2)

  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 (3)

  Depending on the approach adopted, the initial length can vary from 1.0 m (for a mechanically-anchored rockstud installed at 90° 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° 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 70°, intending to keep up either the 0.9 m 95% fall *out height (some candidates used the full 1.4m 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 from 1.2 to 1.8 m will be accepted, based on the angle of installation.*

- Spacing of tendons (2)

  Although no information of 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.Students may decide to do a SR calc based on the required 36 kN/m2 from Q3.2 above. For 100kN (10t) units the required spacing would be approximatelysqrt(100/36) = 1.69m x 1.69m, which is obviously not compatible with the joint spacing. Some students used the simple formula of 0.5 x length, which is also acceptable provided tendon lengths are reasonable.

- Minimum angle of installation (2)

  While candidates may state 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 "90^o to strata" rule will also gain marks.

- Necessary quality control measures (2)

  Visual inspection, planned task observation, pre-load indicators, checking tension by spanner or torque wrench, workplace audits, pull tests, etc, will all gain marks. Simply stating "monitor…" will not gain 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 (1)

Answers ranging from R 50-00 / m^2 – R 100-00 / m^2 will be accepted

3.5. **What additional** on-the-face support measures / systems can the mine manager call on to ensure worker safety, should ground conditions deteriorate further? (2)

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 will be given for any of the above. Marks will NOT be given for changes in blast round, smooth wall blasting, etc, as the question specifically refers to support systems.

(20 marks)

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*
- *Nearby seismic activity / rock bursting – loosen and/or dislodge rockwalls*
- *Orientation – poor orientation to weak layers could produce weak areas*
- *Proximity to other rock passes - increased stress levels*
- *Proximity to geological structures – possible planes of weakness*
- *Secondary blasting –concussion could loosen walls of rockpass*
- *Material falling, not rolling or sliding – impact could loosen walls of rockpass*
- *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 – REGIONAL SUPPORT

When mining tabular orebodies at depth, regional support is generally provided by one or more of the following:

- Strike-orientated regional pillars
- Dip-orientated regional pillars
- Backfill

5.1 Briefly describe the functions of such regional support systems? (6)

Any 6 of the following:

- *Compartmentalise the mine (1), to reduce the effect of large-scale events such as flooding or seismic activity (1)*
- *Reduce the open spans in stopes (1), cutting down on "seismic resonance" (1)*
- *Reduce face stress levels (1), the intensity of stress fracturing (1) and the potential for facebursting (1)*
- *Control closure levels in stopes (1), with effective reductions in energy release levels (1) and overall seismicity (1) (6)*

5.2 Using a table, list the application, advantages and disadvantages of each of the different systems when used in isolation (10)

5.2 Major aspects, other minor ones also marked correct: (10)

Aspect	Strike pillars	Dip pillars	Backfill
Typical applic-ation (1 mark)	Longwall mining	Sequential grid mining	Scattered mining, pillar extraction
Advantages (½ marks)	Parallel to drives, less influence on tunnels Advance concurrent with faces Constant face stress levels	Only becomes seismically active at end of stope life Greater potential to include geological structures	Less exposed span Better access control Better vent control
Disadvantages (½ marks)	Adjacent gully has consistently poor conditions Seismicity along entire length of gully – major access way	Influence on off-reef development Requires longer setup time	High capital setup cost High operating cost Water hazard Backfill placed on sweepings

5.3 What benefits can be gained by combining two of these systems? (4)

Any four of:

- *Better extraction as pillar sizes can be reduced*
- *Reduction of open stope spans*
- *Reduction in in-panel support requirements*
- *Better ventilation and access control*
- *Greater flexibility when planning mining operations*
- *Potential use of backfill "ribs", not full fill (4)*

(20 marks)

QUESTION 6 – REGIONAL SUPPORT

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

6.1 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.Plot all the curves on the same set of axes so that the differences are clearly evident. (6)

See figure on page 56 of Jager and Ryder's "Rock Engineering Practice for Tabular Hard Rock Mines"

6.2 **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 (1)*	*3 – 5:1 (1)*	*10:1 or more (1)*
Peak strength	*Approx same as UCS - 100 MPa (Merensky)(1)*	*220 MPa (Merensky)(1)*	*Infinitely strong (1)*
Residual strength	*Approx 10% of peak strength - 20 MPa (Merensky) (1)*	*Should be close to peak strength - 200 to 220 MPa (Merensky) (1)*	*Not applicable (1)*
Typical application / depth range	*Support tensile zone / weak parting as in-stope support (2)*	*Support tensile zone / weak parting as in-stope support (2)*	*Barrier pillars, regional pillars (1)*

(20 marks)