MEMORANDUM

SUBJECT: CERTIFICATE IN ROCK MECHANICS: PAPER 2 **SUBJECT CODE:** COMRME **EXAMINATION DATE:** OCTOBER 2012 **TIME:** 9:00 – 12:00	**EXAMINER:** LJ SCHEEPERS **MODERATOR:** J LUCAS **TOTAL MARKS:** [100] **PASS MARK:** 60%

NUMBER OF PAGES: 4

SPECIAL REQUIREMENTS:

1. **<u>ALL</u> candidates to answer all 5 questions**
2. References other than those provided are not permitted.
3. Hand-held electronic calculators may be used.
4. Write 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. **<u>Write in ink on the RIGHT HAND SIDE of the paper only (only the right hand pages will be marked).</u>**
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 that is typical of practical conditions.

QUESTION 1 Geotechnical characteristics (15 points)

Transfer and complete the table below into your answer sheet. Describe the 3 rock types (coal, quartzite and pyroxinite) in terms of fundamental class (Igneous, sedimentary or metamorphic), mineral content, geological sequence supergroup, active mining area where it can typically be found and its typical strength.

	Fundamental class	Mineral content	Super-group	Active mining area	Typical Strength [MPa]
Coal	Sedimentary or metamorphic	Carbon	Karoo sedimentary period	Witbank, Evander, Ermelo	5 - 49
Quartzite	Metamorphic	Quartz, feldspar, mica	Witwatersrand Supergroup	Johannesburg, West Wits, Carletonville Vaal River, Orkney	165-190 (220)
Pyroxenite	Igneous	Pyroxene Plagioclase	Bushveld Complex	Rustenburg, Potgietersrus, Lydenburg	100 - 200

[15]

QUESTION 2.Rock and rockmass behavior (25 points)

2.1 Describe and explain how rock quality designation (RQD) may be determined from borehole core (2)

The Rock Quality Designation (RQD) is the ratio of the sum of lengths of completecylindrical core that are greater than 10cm to the total length in a drill run (the smallest distance betweenfractures should be 10cm or greater).

%RQD = (The Sum of Core > 10cm) / TCR x100

2.2 Compare and contrast Barton's Q system, Bieniawski's RMR and Laubscher's MRMR rockmass classification systems and their respective applications. (6)

Barton et al. (1974)) developed this tunnelling quality index system from civil engineering case histories in which all the components of the character of the rock mass were included.

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

where RQD is the Rock Quality Designation,
Jn the joint set number,
Jr the roughness number,
Ja the alteration number,

Jw the water reduction factor and
SRF a stress reduction factor

Bieniawski RMR – catagorise the quality of the rockmass, determine stand-up time, cohesion and friction angles.
The following parameters are used to classify a rock mass using the RMRsystem:
• Uni-axial compressive strength of rock material
• Rock Quality Designation (RQD)
• Spacing of discontinuities
• Condition of discontinuities
• Groundwater conditions

A relationship has been found between RMR and Q as follows (Bieniawski,1989):
RMR = 9 ln Q + 44

Laubscher developed the Mining Rock Mass Rating (MRMR) systemto cater for the differences in underground mining and civil excavations. This system takes into account the same parameters as the Geomechanics system, butcombines the groundwater and joint conditions, resulting in the followingfour parameters prior to modification due to weathering, the stressenvironment, excavation orientation and excavation method:
• rock material strength (UCS)
• RQD
• joint spacing
• joint condition and ground water.

The MRMR rating can now be utilised to estimate the rock mass strength(RMS). The RMS is determined from the UCS of the rock and the MRMRof the rock mass, using the relationship:
RMS = UCS x (MRMR – RUCS)/100
The following also serves as comparison of the application of these systems:
The Q system:
• was developed from civil engineering case histories;
• relates to rock masses where stability is governed mainly by jointing
• was originally applicable to low stress tunnel conditions
• is used for span and tunnel support design;
• is used to determine the Modified Stability Number (N') usedin stablemining span or caving potential estimation

The RMR system:
• was based on shallow civil construction tunnels in sedimentaryrocks and was based on 49 case studies, but was modifiedafter a further 64 case studies;
• does not include a stress-related parameter;
• is to a large extent not used extensively in the RSA underground mines, but

• is used often in open pit slope designs or stand-up time, support and spans in tunnels;
• drawpoint design;

The MRMR system:
• was developed from the RMR system and made it applicableto underground mining excavations and support;
• caters for mining-related issues such as weathering, blasting,excavation orientation w.r.t. jointing and stress;
• applies very well to the mining environment in general andhas been applied to various mining methods, especially massiveand cave mining methods;
• is often used for caving potential estimation;
• is used in unsupported span estimation;
• adjusted MRMR is used for initial slope design;
• drawpoint support design when used to determine the DRMS;

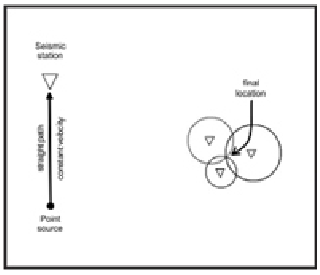
2.3 Define a seismic event and a rockburst. (3)

Seismic event = Sudden release of strain energy due to stresses exceedingthe strength of the rock mass.

Rockburst = a seismic event that results in damage to excavations

2.4 How is a seismic event located? (5)

Locating a seismic source is a matter of triangulation using the P- and S- arrivaltimes recorded at a number of sensors and determining the point inspace that matches the recorded times.
Processing of seismic waveforms yields the time of P- and S-arrival of seismicwaves at each of the seismic stations. The distance between source and station can be determined by using the time difference between P- and S- arrival and estimated wave velocity.Therefore, each seismic station delivers a sphere on which the source centre should have been located. The location algorithm then determines that point where all spheres intersect (3 spheres intersect in a single point, therefore minimum 3 stations are required for a location).



2.5 How do the rockmass properties influence the location of an event? (3)

Waves do not travel on a straight path due to refraction, deflection etc at contact planes between rock compartments with different rockmass properties and around excavations. The wave travel distance is therefore different to the straight-line distance assumed in the location algorithm

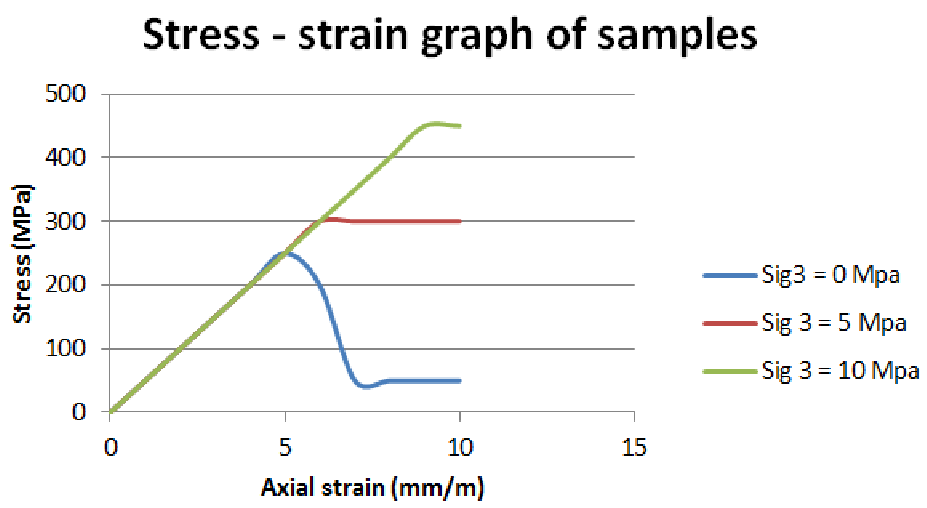
Waves do not travel with constant velocity through rock with different rockmass properties as assumed in the location algorithm; therefore the distance will be incorrectly calculated from the arrival times.

2.6 Explain the difference between the terms hypo-centre and epicentreused in describing the location of seismic events (2)

hypocentre = the point in space where the rupture took place (3D coordinatesrequired)

epicentre = projection of the hypo-centre onto the surface (2D co-ordinates suffice)

2.7 Sketch and describe the complete stress-strain graph for rock in a triaxial compression test. (4)



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QUESTION 3. Mining layouts (20 points)

3.1 Describe and explain how vertical shafts are protected from the effects of mining induced stresses and displacements (3)

To prevent excessive stress levels, all mining practices should be kept away from the barrel (including issues such as stoping and development).Shaft pillars are designed to keepstress levels around the shaft barrel as low as possible. By leaving thesesubstantial areas of unmined ground around the barrel as a shaft pillar,stress levels on large and essential service excavations normally situatedaround the barrel are also reduced.
Pre-extraction slot –where possible the ground where the shaft is to be located can be mined to allow rock movements to occur and the rockmass to be de-stressed prior to the establishment of the shaft barrel, therefore no need to leave a shaft pillar.

3.2 Describe and explain how the ore around a vertical shaft may be mined early to avoid the necessity of leaving a shaft pillar (5)

The ore body is mined prior to the sinking of the shaft: If access to the ore body is possible from another shaft, the ore body can be extracted before the shaft is sunk. This allows the stress levels to change to very low levels prior to sinking (mining de-stresses the barrel), while displacements will be allowed to occur and will terminate before the barrel is developed. It is critical that the mined-out area is sufficiently large to ensure that stress-level increases at the position of the barrel are prevented.

The ore body is mined soon after the shaft sinking has been completed:
Removal of the shaft pillar at this stage will limit stress levels to the absolute minimum, but will still expose the barrel, steelwork and lining to displacements. To limit the impact of these displacements so early in the life of the shaft, the lining around the ore body intersection may be left out (only mesh and shotcrete or fibre reinforced applied to the walls) and the steelwork may be suspended or not installed until after the extraction of the shaft pillar. Each of the options has disadvantages, but conditions will be better than extraction at the end of the life of the mine.

3.3 Describe a mining strategy in an ultra-deep level mine. Explain your designed placement of access development (6)

Sequential Grid mining: controlled mining spans (180 m) separated with wide (30 m) dip stability pillars as well as bracket pillars on major geological structures resulting in an extraction ratio less than 70%. Grid development ahead of mining will experience stress changes and

must therefore be positioned deep in the footwall or hangingwall (100 m) away from the mining abutments.
Or mini-longwall mining with controlled mining spans (200 m) separated with wide (30 m to 40 m) strike stability pillars as well as bracket pillars on major geological structures resulting in an extraction ratio less than 70%.
Follow-on development positioned in the de-stressed zone 30 m to 60 m below or above the mined out mini-longwall.

3.4 Describe a mining strategy in a shallow mine. Explain your designed placement of access development (6)

Room and pillar mining: This method recovers the mineralisation in open stopes, leaving pillars of ore to support the roof. These pillars must be able to support the full overburden. Alternatively large regional pillars are left to support the overburden and smaller in-stope pillars are left to support the immediate hangingwall.
Tendon support is used to control the hangingwall conditions in the rooms.
Little consideration is required for development ends as the rooms are used for access ways and off-reef access ways are typically not exposed to high stress conditions.
Where poor rockmass conditions prevail, appropriate support must be installed.
In weak rock, stress related instabilities may occur and development must be positioned away from abutments and pillars.

[20]

QUESTION 4.Mining support design (20 points)

4.1 Describe an appropriate support strategy and explain your support design for haulages in an ultra-deep mine implementing the sequential grid mining strategy. (5)

The access ways will be subjected to high stress and stress changes due to moving mining abutments. This will result in densely spaced stress fractures. The stress fractures will overshadow other types of discontinuities like joint sets and therefore rockmass ratings are less important in support design. Primary support with spacing of 1 m by 1 m or less will be required to control the fractured rock. The support tendons must be integrated with area coverage support like mesh or skin support as close to the development face as possible. Tendon support length is based on the rule of thumb of half the diameter of the tunnel. Assuming 3.5m square tunnels, tendons must be at least 1.75 m in length. To cater for deformation (bulking) associated with fracturing due to stress changes at the time of mining, the support tendons must be able to yield. To cater for seismic conditions, densely spaced (1 m square spacing) high yielding support with good energy absorption, integrated with mesh and lacing is required. Shotcrete or TSL without mesh and lacing is not sufficient under seismic loading conditions. The tendons must be able to yield at least 200 mm at a yield load of 100 kN. The support design to cater for seismic loading conditions will be sufficient to also cater for the fractured rock.

4.2 Describean appropriate support strategy and explain your support design for follow-on development in an ultra-deep mine implementing the longwall mining strategy. (5)

The follow-on tunnels will not experience high stress or stress changes and therefore little stress fracturing and support design must be based on the rockmass conditions (rockmass ratings) and rules of thumb. In good ground conditions no secondary support is required. Primary support length is based on the rule of thumb of halt excavations span. For the 3.5m square tunnels, the tendons length of 1.75m minimum will suffice. Tendon spacing can be based on the rule of thumb of half tendon length and therefore tendon spacing of 0.8m is required.

4.3 Describe an appropriate support strategy and explain your support design for access ways in a shallow mine implementing the open stoping mining strategy. (5)

Shallow mine – rockmass instability will be driven by gravity and discontinuities in the rockmass like joints, faults and dykes. Little stress related fracturing is expected. Spot bolting in specific places, where potential instability has been identified, may be sufficient. High horizontal stress is a possibility – under such conditions arch failure can be expected in the hangingwall and the hangingwall support requirements will be more than for the sidewalls. Support design is mostly determined through kinematic assessment of individual potential unstable blocks. Wedge failure (from the weak side) at discontinuities is the biggest hazard. Support must cater for the dead weight loading (gravity) and sufficient anchorage beyond the instability plane must be ensured. For a starting point support length can be determined using the ½ span rule of thumb and support spacing on the ½ length rule of thumb. Rockmass ratings should be used to refine the support design using several tables and graphs that indicate appropriate length and spacing of support. Pre-stressing of tendons is important (active support).

4.4 In an ultra-deep level narrow tabular mine using hydraulically placed classified tailings backfill placed in geotextile bags, a large number of bag failures have been reported. Discuss the potential causes that you would investigate to solve this problem. (5)

High percentage of fines – the fines will clog the apertures in the geotextile bags, reducing drainage of water and potentially causing a high hydraulic head – this may result in bag failures. Proper cylconing may fix this problem.

High flow-rate – with a high flow rate, the bag is filled much faster than the rate of water drainage, resulting in a high hydraulic head and potential bag failure. Smaller diameter delivery hoses may fix this problem.

Low RD; i.e. excessive water in the tailings will improve the flow rates but may result in high hydraulic head. The correct mixture of water and slimes is important.

Mismatch between geotextile bag and backfill properties – a range of geotextile bags must be tested with the backfill to choose the optimum bag that will ensure fast drainage without excessive losses of particles. If the apertures are too small, draining will be hampered, resulting in a large hydraulic head and potential bag failure.

Low quality geotextile bags – follow up on quality control

Insufficient placement of support units (elongates or packs) at the bottom of the panel to create a proper bulk head.

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QUESTION 5. Monitoring, numerical modeling and risk assessment (20 points)

5.1 Describe the rock parameters which may be measured and the appropriate instruments to be used in a monitoring program in an aseismic mine. (5)

Possible instabilities to be monitored are related to large blocks of ground becoming unstable under gravity as well as support effectiveness. Possible pillar failures, both quasi-static and dynamic should be monitored.

Monitoring equipment to be used includes extensometers to monitor bed-separation, closure stations to monitor possible onset of accelerated closure. Tell-tales in access ways to warn of possible large deformations.Tension indicators on anchors to monitor pre-stressing and smart cables to monitor rock deformation after installation of support.Borehole camera to monitor possible rockmass deformation on discontinuities or weak planes.

5.2 Identify, describe and explain the objectives of a seismic monitoring program in a seismically active mine. (6)

Record large events to direct search and rescue efforts

Record large events for the purpose of back analysis with the aim of improving designs that will prevent large events

To determine the seismic levels on the mine and use for mine design strategies

To determine seismicity trends and guide mine design

To do short-term, medium-term and long-term seismic hazard assessments and provide early warning of increased hazard

To determine ground control districts (if parts of the mine is active and parts not

To have sufficient data for research

5.3 Describe and explain for which rock engineering application the following numerical modelling techniques are suitable (4)

5.3.1 Finite element methods

Finite element solutions are codes where all results at all points in themodel are calculated. The model creates a mesh (grid) throughout thewhole modelled area.Mining tunnelsand excavationsassociated withunderground rockmass that canconsist of variousmaterial typesand can be elasticor assume plasticfailure. Miningsteps can alsobe simulated.Surface layoutssuch as trenches,slots or slopescan also be evaluatedfor stability.Support unitsand areal coveragesupportsuch as shotcretecan be appliedand evaluated.

5.3.2 Finite difference methods

In the finite difference model (such as the FLAC code), grids are createdthroughout the model similar to the mesh generated in finite elementcodes. These codes are based on a cycle of calculations, performed foreach grid point in the model.Mining layoutsassociated withtabular and massiveexcavationsin rock mass thatcan consist of variousmaterials. Itcan simulate severalmining stepsand can be usedto back analyseprevious miningconducted or predictchanges tobe expected withplanned mining.Tunnels and shaftscan be constructedwithin the model.

5.3.3 Boundary element methods

The Boundary element technique creates elements along the boundaryof the excavations included into the model only, instead of includingelements within the complete model. This allows for fast solution / calculationtimes due to a smaller number of elements and one-dimensionallinear elements

This is suitable for narrow tabular orebodies and when large areas are included in the model. Stress redistribution is easily modelled, but failure has to be interpreted.

5.3.4 Keyblock methods

Keyblock analysis methods (Goodman and Shi, 1985) are used to evaluatewhether blocks are removable and whether the chosen support willbe sufficient to ensure stability.

5.4 List at least 5 parameters to be included in a short-term panel hazard assessment to be used in the planning process on a shallow coal mine. (5)

Face shape

Face length

Pillar dimensions

Host rock type (properties)

Stope width

Rockmass rating

[20]