EXAMINATION PAPER

SUBJECT:	EXAMINER:
CERTIFICATE IN ROCK MECHANICS PAPER 3.4 : OPEN PIT	GLEN MC GAVIGAN
SUBJECT CODE:	**MODERATOR:**
COMRME	PETER TERBRUGGE
EXAMINATION DATE:	**TOTAL MARKS:** [100]
October 2012	**PASS MARK:** 60%
TIME: 9:00 – 12:00	

NUMBER OF PAGES: 6

SPECIAL REQUIREMENTS:

1. **Answer *ALL FIVE* questions**
2. References other than those provided are not permitted.
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 (only the right hand pages will be marked).**
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

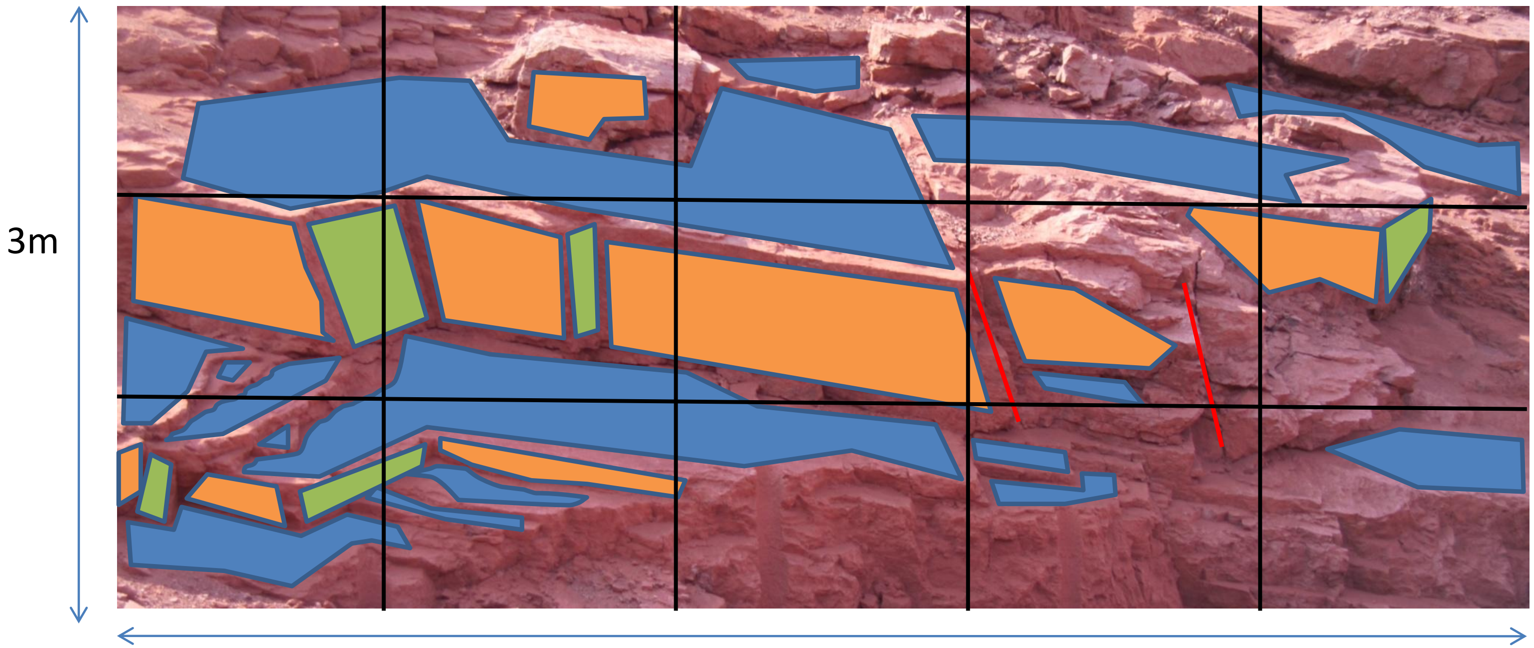
The attached photograph shows an exposure of rock in an open pit mine. Blasting is regarded as poor. The scale of the exposure is defined by the short length of the photograph (addendum 1), which represents a 3m height of exposure and 200m below surface. Mining will proceed to a maximum depth of 450m in similar rock. Logging of exploration core has shown that there is some evidence of clay coating on the joints to a depth of 50m, but there is no evidence of weathering on joint surfaces except for some iron staining at the current mining depth. Joint separations are estimated to be <1mm. The water table is high and it is expected that minor local seepage will be experienced. The uniaxial strength of the rock at mining depth, estimated from point load testing carried out on the borehole core, is 147MPa. In the area the horizontal in situ stress is known to be approximately 50% higher than the vertical overburden stress.

Classify the rock mass, with respect to the mining activities at the depth of the rock exposure, using the Q, GSI and RMR systems. Calculate the equivalent RMR from the other classification systems where possible and compare the numbers you obtain. Annotate your photograph to illustrate the basis of your classification and provide a brief explanation for your decisions on the input parameters for each classification (NB annotated photographs to be handed in with your script).

Students are not overly penalised for calculation errors and are awarded points based on the principles of applying rock mass rating systems.

[20 MARKS]

OPEN PIT



Approx 6m

Four joint sets are present
J1 – Bedding plane (Blue)
J2 – Green
J3 – Orange
J4 – Red

For the purposes of the calculation assume the unit volume with the most joints (i.e. most conservative), therefore the bottom left unit volume (ideally one should determine a range of joint spacing's e.g. min, max and mean)

RQD defined by Palmstrom (1974) as RQD = 115 – 3.3Jv

Where Jv is the number of joints per one m^3

J1: 6 joints = 0.166m

J2: 2 joints = 0.5m

J3: 2 joints = 0.5m

J4: not in unit volume but may be included

$Jv = 1/0.166 + 1/0.5 + 1/0.5$

$= 10$

$RQD = 115 – 3.3 (10)$

= **<u>82%</u>**

NB = evaluate rock mass at exposure depth – 200m

RMR

Assume critical joint set to be J1/Bedding. It is the most closely spaced, persistent and slightly rough to rough. It also daylights in the pit face and is likely to have the greatest impact on stability:

- RQD = 82% (17)
- UCS = 143MPa (12)
- Spacing = J1, 0.166 (8)
- Discontinuity conditions: (17)
  - Persistance = 2 – 3m (4)
  - Separation = 1mm (4)
  - Roughness = Slightly rough – rough (3)
  - Infill/gouge = staining (0)
  - Weathering = unweathered (6)
- Groundwater = minor seepage (wet to dripping) (4)

RMR= 58

Q

Assume critical joint set to be J1/Bedding. It is the most closely spaced, persistent and slightly rough to rough. It also daylights in the pit face and is likely to have the greatest impact on stability:

- RQD = (82%)
- Jn = 4 joint sets (15)
- Jr = Rough or irregular/planar or smooth planar (1.5)
- Ja = unaltered/surface staining (1.0)
- Jw = Dry/Minor to Medium (1 – 0.66), assume 0.66 (0.66)
- SRF = (2.5)

Q = RQD/Jn x Jr/Ja x Jw/SRF
Q = 2.15

RMR ≈ 9ln Q +44
RMR ≈ 51

GSI

Very blocky (4 or more joint sets) to Blocky/Disturbed/Seamy (bedding), joint conditions are good (rough, iron stained):

GSI : 50 – 60

Comparison

RMR ≈ 9ln Q +44
RMR ≈ 51

GSI = RMR – 5
= 58 – 5
GSI = 52

QUESTION 2

2.1 Discuss the fundamentals of the following concepts and their application in open pit mining. Make use of basic formula and sketches in your answer:

REFER to HOEK:
http://www.rocscience.com/hoek/corner/8_Factor_of_safety_and_probability_of_failure.pdf

- Factor of Safety [5]

  The classical approach used in designing engineering structures is to consider the relationship between the capacity C (strength or resisting force) of the element and the demand D (stress or disturbing force). The Factor of Safety of the structure is defined as F = C/D and failure is assumed to occur when F is less than unity (1).

  Include a measure of acceptable levels of FoS for open pit mining linked to the consequence of failure (i.e. slope carrying infrastructure should have a higher FoS).

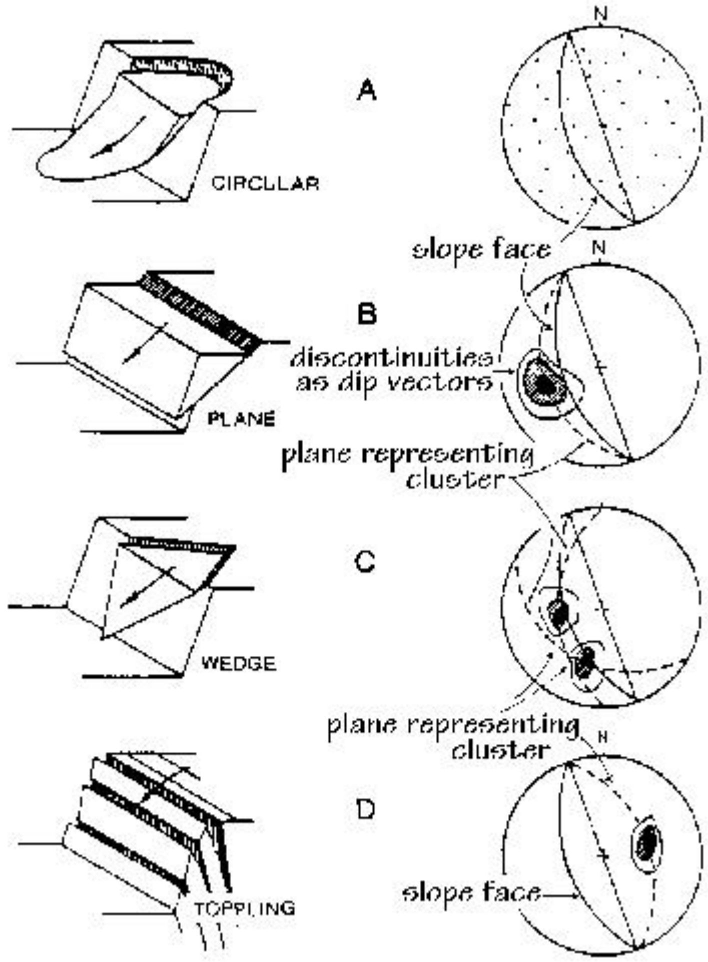
- Probability of Failure [5]

  REFER to HOEK:
  http://www.rocscience.com/hoek/corner/8_Factor_of_safety_and_probability_of_failure.pdf

  POF = [FoS < 1)

  Include in your discussion random variables, statisitical distributions, sampling techniques and a measure of acceptable PoF linked to consequence of failure.

2.2 Discuss the conditions for planar, wedge and toppling failure. Make use of hand drawn stereonet sketches in your answer. Clearly label your sketches. [10]



[20 MARKS]

QUESTION 3

Discuss the following numerical analysis methods with their applicability to the open pit environment:

- Continuum modelling
- Discontinuum modelling
- Hybrid/Coupled modelling

In your answer comment the following:

- Fundamentals of each method
- Critical input parameters
- Advantages

- Limitations
- Examples of commercially available codes for each of the methods.

Analysis method	Critical input parameters	Advantages	Disadvantages
Continuum modelling (finite element, finite diference e.g. FLAC, PHASE2).	Representative slope geometry, constitutive criteria (e.g elastic, elasto-plastic, creep etc), groundwater, characteristics,, shear stress of surfaces, insitu stress state.	Allows for material deformation and failure. Can model complex behaviour and mechanisms. Capability of modelling 3D. Con model the effects of ground water and pore pressure. Able to assess effects of parameter variations on instability. Recent advances in computing hardware allow complex models to be solved in reasonable run times. Can incorporate creep deformation and dynamic analysis.	User must be well trained, experiencesd and observe good modelling practice. Need to be aware of modelling limitations (e.g. boundary effects, mesh aspect ratios, symmetry. Availability of input data generally poor. Required input parameters generally not measured. Inability to model highly jointed rock (more suitable to massive, weak and soil like rocks). Can be difficult to perform sensitivity analysis due to run time constraints. 2D models assume plain strain conditions which may not be frequently valid in inhomogeneous rock slopes off varying structure, lithology and topography.
Discontinuuim modelling (distinct element and discrete element , e.g. UDEC, 3DEC).	Representative slope geometry and discontinuity geometry, intact, constitutive criteria (e.g elastic, elasto-plastic, creep etc), discontinuity stiffness and shear strength, groundwater, characteristics,,	Allows for block deformation and movement of blocks relative to one each other. Can model complex behaviour and mechanisms (combined material and discontinuity behaviour with hydro mechanical and dynamic analysis). Able to assess effects of parameter variation on instability.	As above, experienced user required to observe good modelling practice. General limitations similar to those listed above. Need to be aware of scale effects. Need to simulate representative discontinuity geometry (spacing, persistence etc). Limited data available on joint properties (e.g. jk_n, jk_s)

	shear stress of surfaces, insitu stress state.		
Hybrid/Coupled modelling (e.g. ELFEN)	Combination of input parameters listed above for standalone models	Coupled finite element/distinct element models able to simulate intact fracture propagation and fragmentation of jointed bedded media.	Complex problems require high memory capacity. Comparatively little practical experience in use. Requires ongoing calibration and constraints.

NB. discuss fundamentals of finite element, finite difference as applied to continuum and discontinuum modelling. Include basic principles that underlie each method.

[20 MARKS]

QUESTION 4

Presplitting is widely regarded as the industry benchmark in terms of limiting the damage caused by blasting. Discuss the concept of presplitting, focusing on the following:

- Effect of rock-mass properties on the results of presplitting

  Presplitting rarely gives good results in closely fissured rock. When over charged, presplitting can damage appreciable volumes of rock as explosion gasses vent through the fissures. Under favourable geological conditions (massive rocks of moderate strength), prespliting can provide improved results over postsplitting.
- Limitations

  Limitations are general restricted to ground and air vibrations produced by firing the presplit. The levels of ground vibration per kilogram of explosives are much higher than that of a normal production blast because the rock that is remote from the bench is relatively undialated.

  In theory, there is no limit to the length of a presplit that can be fired in a single blast. In practice however, the possibility of rock properties changing and the charge either failing to produce the split or causing excess damage is good reason not to fire excessively long presplits too far in advance of the regular pit-wall blasts (generally 2 – 3 spacings).

- Hole diameter

  There is no limit to the diameter of pre-split holes, successful splits have been conducted with 406mm holes (in favourable geology). To achieve economies of scale, there is an incentive to use larger diameter pre-split holes. Unfortunately the diameter of *angled* presplit holes is limited by the rigs that can drill angled blast holes beneath the body of the drill or at 90 degrees to the tracks. Most large drill rigs can only drill angled holes where there is sufficient berm width to provide tail room for the drill. Therefore, maximum economies of scale can often only be achieved there vertical presplit blast holes are geotechnically acceptable.

- Explosive types

  Ideally presplit holes should be charged with cartridges attached to the detonator cord down the hole in an attempt to obtain better charge energy distribution. The development of continuous colums of small diameter explosives, specifically for presplits has resulted in optimal energy distribution. However, because their energy yield per meter of charge length is constant, their use does not allow small changes in powder factor which are achieved with spaced cartridges down hole.

  For larger diameter holes, presplitting can sometimes be carried out using low-density bulk explosives. The energy concentration of these explosives are too high for long, continuous, full coupled charges but are usually acceptable for fully coupled charges that have a long column of air above them. With these charges, the best splits are achieved by:

    - Lowering the density and energy concentration
    - Top priming
    - Avoiding using stemming.

- Stemming

  If there is no need to control air vibrations, presplit holes should not be stemmed allowing the explosive gasses to jet into the air, resulting in less damage to the rock mass along joints as well as into cracks that intersect the upper wall of the blast hole. The latter, reduces the damages to the crests of final berms.

- Timing

  Optimum results are obtained when charges are initiated simultaneously. This is achieved by joining all of the down lines to a detonator cord trunkline (or through electronic detonators). Where ground vibrations are of a concern, delays should be inserted into the trunkline in order to obtain consecutive firing of groups of holes. The number of blast holes in each group should be sufficient to achieve a satisfactory splitting action while not exceeding the maximum charge weight that can be fired per delay.

[20 MARKS]

OUESTION 5

You are the Principal Rock Engineer for STABLE mining and have been requested to develop a monitoring strategy for a new mine to be constructed in the mountainous area of Central Africa. You have read the feasibility report and highlighted the following:

- Pit is rectangular in shape and with its long axis orientated NE-SW
- There is a single access ramp into the pit.
- Pit depth – 500m
- Overall slope angle - 42 deg
- Bench height 15m
- Bench face angle – 80 deg
- Stack height - 60m
- Stack angle - 55 deg
- Upper 50m of the slope will be constructed in highly weathered material
- Remainder of the rock mass has an average RMR of 67 and may be regarded as brittle.
- The average joint spacing is 20cm
- Several large wedges were identified from orientated core and stereographic analysis
- Annual rain fall of 2500mm with over 200 days of rain per year
- Dense fog is experienced in the early morning during the winter months

- On average it takes 3 days travel to site from Johannesburg
- You have a budget of U$ 450,000

You must consider all of the above aspects in developing your strategy which you must present to the CEO for approval. In developing your strategy be sure to highlight the advantages and disadvantages of each method that you have selected.

The conditions to be monitored vary from soft rock (large deformations over time) to brittle rock (small deformations over a short time) and therefore need to cater for both situations. A typical set up could be:

- Real time prism monitoring for brittle rock masses and long term history. Approx 2 systems will be required for each side of the pit. Prisms should be systematically installed and densified on critical wedges and on main access into the pit. Remote access (for the Principal Geotech) should be arranged considering the remoteness of the site.
    - Advantages – accurate, long term, semi real time.
    - Disadvantages – discrete points (considering 20cm average joint spacing), prisms difficult to maintain and install. May be affected by the fog. Atmospheric effects are likely to be high considering temperature and humidity variations; therefore it may only be possible to compare readings of the same time of day with each other.
- A series of inclinometers should be installed on critical Geotech domains to monitor large structures etc. Data may be linked remotely.
    - Adv – very accurate, gives information on the rock mass and specific structures. Real time and can be linked remotely
    - Diadvantages – single point, if existing holes are not used costs of drilling dedicated holes are high.
- Extensiometers should be installed and access maintained. May be linked remotely.
    - Advantages: cheap, easy to install, mobile, can be measured by lower qualified workers. Real time and can be linked remotely
    - Diadvantages: single point, not three dimensional.

- Visual inspections – a systematic visual inspection program is required. The visual inspections should be carried out be personnel trained in the identification of rock related hazards ranging from foremen to geotechnical engineers. Standard forms and procedures should be developed to ensure consistency in visual monitoring.
- Rainfall and groundwater monitoring – considering the blockiness of the rock mass, rock falls may occur after and during heavy rainfall. The monitoring of the intensity of rainfall linked to a TARP is good practice. Because of the high rainfall groundwater recharge is likely to be high and should be monitored with piezometers.
- Technicians must be trained in the maintenance of the equipment as it takes three days to travel from Johannesburg. Critical spares should be kept on site.
- Considering the high occurrence of fog and the reliance on optical methods of monitoring (e.g. visual and prism) you could motivate a radar system for the high rainfall and fog months. Alternatively the monitoring procedure (prism) could be modified for this period to collect data in the afternoon when the fog is less of a problem.

Other systems that may be considered:

- Laser
- INSAR for weathered material

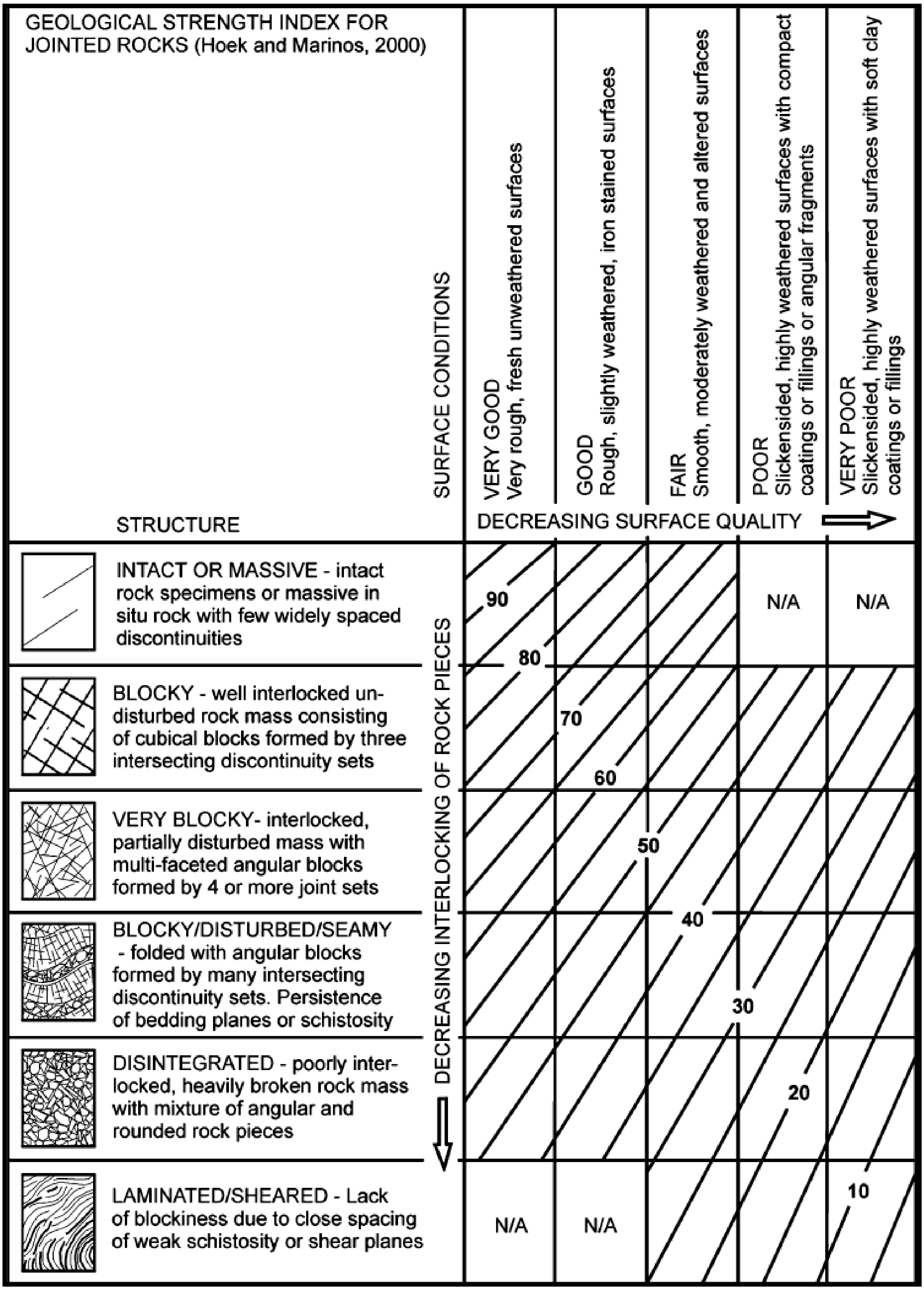
[20 MARKS]

ADDENDUM 1 – PHOTOGRAPH FOR QUESTION 1

ID NUMBER:____________________________ EXAM NUMBER:______________________

NB: INCLUDE WITH YOUR SCRIPT

OPEN PIT


GEOLOGICAL STRENGTH INDEX FOR JOINTED ROCKS (Hoek and Marinos, 2000)
SURFACE CONDITIONS
VERY GOOD Very rough, fresh unweathered surfaces
GOOD Rough, slightly weathered, iron stained surfaces
FAIR Smooth, moderately weathered and altered surfaces
POOR Slickensided, highly weathered surfaces with compact coatings or fillings or angular fragments
VERY POOR Slickensided, highly weathered surfaces with soft clay coatings or fillings
STRUCTURE
DECREASING SURFACE QUALITY
INTACT OR MASSIVE - intact rock specimens or massive in situ rock with few widely spaced discontinuities
BLOCKY - well interlocked un-disturbed rock mass consisting of cubical blocks formed by three intersecting discontinuity sets
VERY BLOCKY- interlocked, partially disturbed mass with multi-faceted angular blocks formed by 4 or more joint sets
BLOCKY/DISTURBED/SEAMY - folded with angular blocks formed by many intersecting discontinuity sets. Persistence of bedding planes or schistosity
DISINTEGRATED - poorly inter-locked, heavily broken rock mass with mixture of angular and rounded rock pieces
LAMINATED/SHEARED - Lack of blockiness due to close spacing of weak schistosity or shear planes
DECREASING INTERLOCKING OF ROCK PIECES
90
80
70
60
50
40
30
20
10
N/A
N/A
N/A
N/A

Q system

DESCRIPTION	VALUE	NOTES
1. ROCK QUALITY DESIGNATION	*RQD*	
A. Very poor	0 - 25	1. Where *RQD* is reported or measured as $\leq$ 10 (including 0),
B. Poor	25 - 50	a nominal value of 10 is used to evaluate *Q*.
C. Fair	50 - 75	
D. Good	75 - 90	2. *RQD* intervals of 5, i.e. 100, 95, 90 etc. are sufficiently
E. Excellent	90 - 100	accurate.
2. JOINT SET NUMBER	J_n	
A. Massive, no or few joints	0.5 - 1.0	
B. One joint set	2	
C. One joint set plus random	3	
D. Two joint sets	4	
E. Two joint sets plus random	6	
F. Three joint sets	9	1. For intersections use ($3.0 \times J_n$)
G. Three joint sets plus random	12	
H. Four or more joint sets, random, heavily jointed, 'sugar cube', etc.	15	2. For portals use ($2.0 \times J_n$)
J. Crushed rock, earthlike	20	
3. JOINT ROUGHNESS NUMBER	J_r	
a. Rock wall contact		
b. Rock wall contact before 10 cm shear		
A. Discontinuous joints	4	
B. Rough and irregular, undulating	3	
C. Smooth undulating	2	
D. Slickensided undulating	1.5	1. Add 1.0 if the mean spacing of the relevant joint set is
E. Rough or irregular, planar	1.5	greater than 3 m.
F. Smooth, planar	1.0	
G. Slickensided, planar	0.5	2. J_r = 0.5 can be used for planar, slickensided joints having
c. No rock wall contact when sheared		lineations, provided that the lineations are oriented for
H. Zones containing clay minerals thick enough to prevent rock wall contact	1.0 (nominal)	minimum strength.
J. Sandy, gravely or crushed zone thick enough to prevent rock wall contact	1.0 (nominal)	
4. JOINT ALTERATION NUMBER	J_a	ϕr degrees (approx.)
a. Rock wall contact		
A. Tightly healed, hard, non-softening, impermeable filling	0.75	1. Values of ϕr, the residual friction angle, are intended as an approximate guide
B. Unaltered joint walls, surface staining only	1.0	25 - 35 to the mineralogical properties of the
C. Slightly altered joint walls, non-softening mineral coatings, sandy particles, clay-free disintegrated rock, etc.	2.0	25 - 30 alteration products, if present.
D. Silty-, or sandy-clay coatings, small clay-fraction (non-softening)	3.0	20 - 25
E. Softening or low-friction clay mineral coatings, i.e. kaolinite, mica. Also chlorite, talc, gypsum and graphite etc., and small quantities of swelling clays. (Discontinuous coatings, 1 - 2 mm or less)	4.0	8 - 16

4, JOINT ALTERATION NUMBER	J_a	ϕr degrees (approx.)	
b. Rock wall contact before 10 cm shear			
F. Sandy particles, clay-free, disintegrating rock etc.	4.0	25 - 30	
G. Strongly over-consolidated, non-softening clay mineral fillings (continuous < 5 mm thick)	6.0	16 - 24	
H. Medium or low over-consolidation, softening clay mineral fillings (continuous < 5 mm thick)	8.0	12 - 16	
J. Swelling clay fillings, i.e. montmorillonite, (continuous < 5 mm thick). Values of J_a depend on percent of swelling clay-size particles, and access to water.	8.0 - 12.0	6 - 12	
c. No rock wall contact when sheared			
K. Zones or bands of disintegrated or crushed	6.0		
L. rock and clay (see G, H and J for clay	8.0		
M. conditions)	8.0 - 12.0	6 - 24	
N. Zones or bands of silty- or sandy-clay, small clay fraction, non-softening	5.0		
O. Thick continuous zones or bands of clay	10.0 - 13.0		
P. & R. (see G.H and J for clay conditions)	6.0 - 24.0		
5. JOINT WATER REDUCTION	J_w	approx. water pressure (kgf/cm^2)	
A. Dry excavation or minor inflow i.e. < 5 l/m locally	1.0	< 1.0	
B. Medium inflow or pressure, occasional outwash of joint fillings	0.66	1.0 - 2.5	
C. Large inflow or high pressure in competent rock with unfilled joints	0.5	2.5 - 10.0	1. Factors C to F are crude estimates; increase J_w if drainage installed.
D. Large inflow or high pressure	0.33	2.5 - 10.0	
E. Exceptionally high inflow or pressure at blasting, decaying with time	0.2 - 0.1	> 10	2. Special problems caused by ice formation are not considered.
F. Exceptionally high inflow or pressure	0.1 - 0.05	> 10	
6. STRESS REDUCTION FACTOR ***a. Weakness zones intersecting excavation, which may cause loosening of rock mass when tunnel is excavated***		*SRF*	
A. Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock any depth)		10.0	1. Reduce these values of *SRF* by 25 - 50% but only if the relevant shear zones influence do not intersect the excavation
B. Single weakness zones containing clay, or chemically dis-tegrated rock (excavation depth < 50 m)		5.0	
C. Single weakness zones containing clay, or chemically dis-tegrated rock (excavation depth > 50 m)		2.5	
D. Multiple shear zones in competent rock (clay free), loose surrounding rock (any depth)		7.5	
E. Single shear zone in competent rock (clay free). (depth of excavation < 50 m)		5.0	
F. Single shear zone in competent rock (clay free). (depth of excavation > 50 m)		2.5	
G. Loose open joints, heavily jointed or 'sugar cube', (any depth)		5.0	

DESCRIPTION	VALUE			NOTES
6. STRESS REDUCTION FACTOR			*SRF*	
b. Competent rock, rock stress problems				
	σ_c/σ_1	$\sigma_t\sigma_1$		2. For strongly anisotropic virgin stress field (if measured): when $5 \leq \sigma_1/\sigma_3 \leq 10$, reduce σ_c to $0.8\sigma_c$ and σ_t to $0.8\sigma_t$. When $\sigma_1/\sigma_3 > 10$, reduce σ_c and σ_t to $0.6\sigma_c$ and $0.6\sigma_t$, where σ_c = unconfined compressive strength, and σ_t = tensile strength (point load) and σ_1 and σ_3 are the major and minor principal stresses.
H. Low stress, near surface	> 200	> 13	2.5	
J. Medium stress	200 - 10	13 - 0.66	1.0	
K. High stress, very tight structure (usually favourable to stability, may be unfavourable to wall stability)	10 - 5	0.66 - 0.33	0.5 - 2	
L. Mild rockburst (massive rock)	5 - 2.5	0.33 - 0.16	5 - 10	
M. Heavy rockburst (massive rock)	< 2.5	< 0.16	10 - 20	3. Few case records available where depth of crown below surface is less than span width. Suggest *SRF* increase from 2.5 to 5 for such cases (see H).
c. Squeezing rock, plastic flow of incompetent rock under influence of high rock pressure				
N. Mild squeezing rock pressure			5 - 10	
O. Heavy squeezing rock pressure			10 - 20	
d. Swelling rock, chemical swelling activity depending on presence of water				
P. Mild swelling rock pressure			5 - 10	
R. Heavy swelling rock pressure			10 - 15	

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

RMR = 9lnQ + 44

OPEN PIT

RMR 89

A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

	Parameter		Range of values						
1	Strength of intact rock material	Point-load strength index	>10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range - uniaxial compressive test is preferred		
		Uniaxial comp. strength	>250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill core Quality *RQD*		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of discontinuities		> 2 m	0.6 - 2 . m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of discontinuities (See E)		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensided surfaces or Gouge < 5 mm thick or Separation 1-5 mm Continuous	Soft gouge >5 mm thick or Separation > 5 mm Continuous		
	Rating		30	25	20	10	0		
5	Groundwater	Inflow per 10 m tunnel length (l/m)	None	< 10	10 - 25	25 - 125	> 125		
		(Joint water press)/ (Major principal σ)	0	< 0.1	0.1, - 0.2	0.2 - 0.5	> 0.5		
		General conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		

B. RATING ADJUSTMENT FOR DISCONTINUITY ORIENTATIONS (See F)

Strike and dip orientations		Very favourable	Favourable	Fair	Unfavourable	Very Unfavourable
Ratings	Tunnels & mines	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	

C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

Rating	100 ← 81	80 ← 61	60 ← 41	40 ← 21	< 21
Class number	I	II	III	IV	V
Description	Very good rock	Good rock	Fair rock	Poor rock	Very poor rock

D. MEANING OF ROCK CLASSES

Class number	I	II	III	IV	V
Average stand-up time	20 yrs for 15 m span	1 year for 10 m span	1 week for 5 m span	10 hrs for 2.5 m span	30 min for 1 m span
Cohesion of rock mass (kPa)	> 400	300 - 400	200 - 300	100 - 200	< 100
Friction angle of rock mass (deg)	> 45	35 - 45	25 - 35	15 - 25	< 15

E. GUIDELINES FOR CLASSIFICATION OF DISCONTINUITY conditions

Discontinuity length (persistence)	< 1 m	1 - 3 m	3 - 10 m	10 - 20 m	> 20 m
Rating	6	4	2	1	0
Separation (aperture)	None	< 0.1 mm	0.1 - 1.0 mm	1 - 5 mm	> 5 mm
Rating	6	5	4	1	0
Roughness	Very rough	Rough	Slightly rough	Smooth	Slickensided
Rating	6	5	3	1	0
Infilling (gouge)	None	Hard filling < 5 mm	Hard filling > 5 mm	Soft filling < 5 mm	Soft filling > 5 mm
Rating	6	4	2	2	0
Weathering	Unweathered	Slightly weathered	Moderately weathered	Highly weathered	Decomposed
Ratings	6	5	3	1	0

F. EFFECT OF DISCONTINUITY STRIKE AND DIP ORIENTATION IN TUNNELLING**

Strike perpendicular to tunnel axis		Strike parallel to tunnel axis	
Drive with dip - Dip 45 - 90°	Drive with dip - Dip 20 - 45°	Dip 45 - 90°	Dip 20 - 45°
Very favourable	Favourable	Very unfavourable	Fair
Drive against dip - Dip 45-90°	Drive against dip - Dip 20-45°	Dip 0-20 - Irrespective of strike°	
Fair	Unfavourable	Fair	