National Exams - May 2009

98-MMP-B1, Applied Rock Mechanics

3 hours duration

NOTES:

1. If doubt exists as to the interpretation of any question, the candidate is urged to submit with the answer paper a clear statement of any assumptions made;
2. Any non-communicating calculator is permitted.
3. This is an Open Book exam. Note to candidates - you must indicate the type of calculator being used (ie.- write the name and model designation of the calculator on the first inside left hand sheet of the exam work book).
4. Any five questions constitute a complete paper. Only the first five questions as they appear in your answer book will be marked.
5. All questions are of equal value.

QUESTION 1

{10} (1.1) The following triaxial compression failure data was determined on the basis of tests on intact rock core specimens. Using this data, determine the appropriate rock mechanical parameters for this rock material (unconfined compressive strength, cohesive strength and internal angle of friction), as established by both Mohr-Coulomb and Hoek-Brown failure locus plots. Demonstrate, by plotting both Mohr-Coulomb and Hoek-Brown failure envelopes for these results, how the parametric values were obtained.

Axial Failure Stress (σ_1), MPa	Sample Confining Stress (σ_3), MPa
330	12.5
343	28
412.5	37.5
450	50
530	75

{4} (1.2) You are requested to test two additional core samples of similar rock under confining stress levels of 45 and 65 MPa respectively. At what levels of applied axial stress would you ideally expect these intact core specimens to fail?

{4} (1.3) At several sites in an underground mine where these core specimens were recovered, site stress conditions, in terms of (σ_1, σ_3), were measured. For three sites, the major and minor principal stress components were determined to approximate:

Site (a):	(σ_1 = 290 MPa)	(σ_3 = 10 MPa)
Site (b):	(σ_1 = 310 MPa)	(σ_3 = 40 MPa)
Site (c):	(σ_1 = 330 MPa)	(σ_3 = 80 MPa)

Discuss the conditions of stability that exist for these three triaxially-confined rock sites and the relative change in stability conditions which occurs as the level of rock confinement increases. Use the failure locus plots of part (1.1) to illustrate your discussion.

{2} (1.4) For similar intact rock material, unconfined, it is anticipated that a momentary axial stress approximating 350 MPa will be applied. What measure of confinement stress must be applied to this specimen, prior to axial loading, to just prevent failure from occurring at this level of applied axial stress?

QUESTION 2

{10} (2.1) For a section of drill core, whose length and diameter was measured to be 9.8 and 4.6 cm respectively, the appended force-deformation curve was measured during unconfined compression testing in the laboratory. For this core material, determine the following parameters:

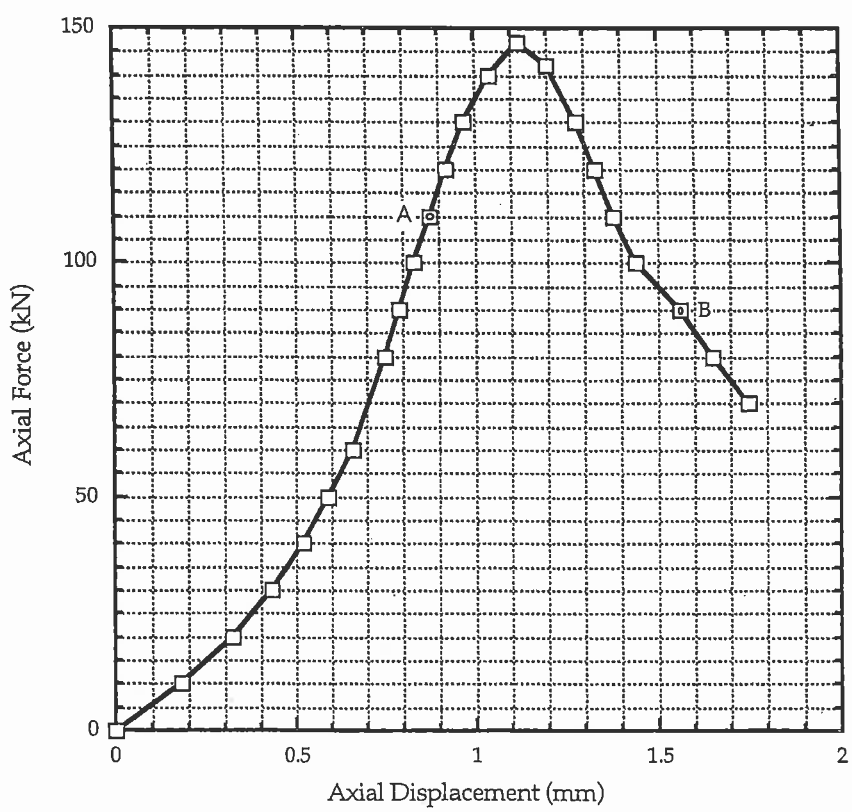
-unconfined compressive strength (UCS)

-Young's Modulus ($E_{50\%}$)

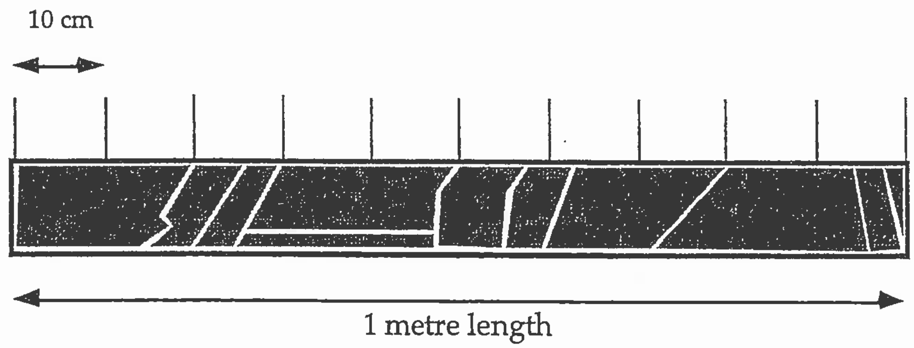
-secant Young's Modulus (E_{secant}), and

-the specimen stiffness conditions at points (A), in the pre-yield loading region, and (B), in the post-yield loading region

For the stiffness condition exhibited by this core specimen at point (B), and knowing that the measured loading frame stiffness approximates 750 kN/mm, explain why the loading system should theoretically be capable of generating controlled, non-violent post-yield failure of similar rock materials.



{10} (2.2) For the following sketch, showing a one meter long section of recovered drill core, what numerical estimate of Deere's RQD would you estimate for the core length and what rock quality quality would you associate with this rock? Provide a brief description of the RQD classification technique, explaining its applicability for assessing rock character in a "stand-alone" fashion and in combination with other commonly-used rock mass classification techniques.



QUESTION 3

Answer each of the following sections of this question by selecting the most appropriate multiple choice answer from the list provided. In order to obtain full marks, you must justify your answer.

{10} (3.1) A room-and-pillar stope is to be developed at a depth below ground surface of 950 meters. The average bulk density of the hanging wall waste rock is 21.53 kN/m^3 (lying between the tabular ore body and surface), and it is known that the host ore exhibits an average unconfined compressive strength approximating 132.5 MPa.

It is planned that an array of square pillars will be left to support the stope roof and that a constant pillar centre-to-centre distance of 25 meters will be maintained between all pillars. Two design cases are to be implemented - one in which pillars are to be constructed at 10 meters width, the other in which pillar widths will be set at 14 meters. For these two cases of development, the differences in post-development pillar stress and extraction ratio conditions will be best reflected by:

(a) (45 MPa, 10%)
(b) (65 MPa, 10%)
(c) (45 MPa, 15%)
(d) (65 MPa, 15%)

{10} (3.2) Three lengths of rock core, of similar composition and at 54 mm diameter, have been subjected to point load testing failure. The failure forces applied for the three tests were found to be (23.5 kN), (27.6 kN) and (21.4 kN). The average Point Load Index (I_S) and calculated unconfined compressive strength (S_C) that can be inferred for the sum of tests conducted on this rock material are best expressed by:

(a) (I_S = 8.05 MPa, S_C = 161.1 MPa)
(b) (I_S = 8.05 MPa, S_C = 193.2 MPa)
(c) (I_S = 8.25 MPa, S_C = 198.0 MPa)
(d) (I_S = 8.55 MPa, S_C = 171.0 MPa)

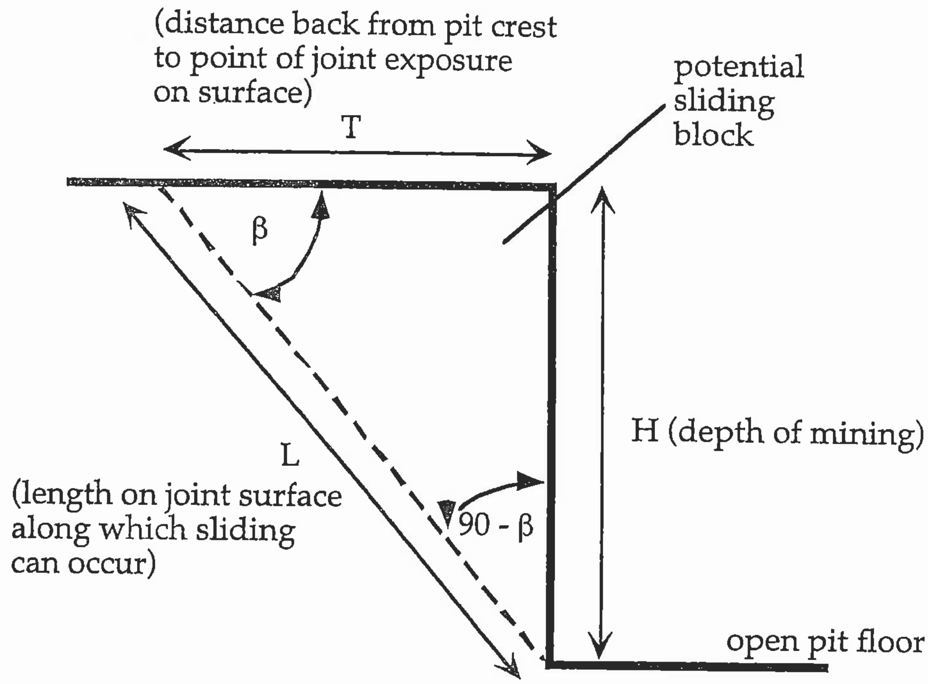
QUESTION 4

{10} (4.1) A simplified view of a three-dimensional rock wedge that outcrops on a vertical open pit face is illustrated in the accompanying sketch. This wedge is formed by the intersection of two joint families and the vertical wall of an open pit.

One joint strikes perpendicular to the pit face, dips vertically and exhibits a regular spacing interval (W, metres) between successive joints in a direction that is parallel to the pit face. No cohesive strength (ie.- cohesion = 0) is assumed to be exhibited by this joint feature.

The second joint system strikes parallel to the pit face, dips directly into the pit at an angle (β), and also exhibits a regular repeat interval spacing that is unknown. Some measure of cohesive strength is assumed to exist along these joint features. The vertical geometry of the second joint system is illustrated in the section view through the pit face, which is shown in the sketch.

Derive an equation, in terms of the excavation and block wedge geometry, that relates the block Factor of Safety (FS) against sliding versus depth of mining (H) when the base of the potential wedge is first exposed.



VERTICAL SECTION THROUGH PIT WALL
SHOWING POTENTIAL SLIDING WEDGE

{10} (4.2) In a situation such as that which was described in part (4.1), the following physical conditions are known to exist:

- -dip of joint plane (β) = 45°
- -rock density of wedge material = 28.35 kN/m^3
- -internal angle of friction of potential sliding plane (ϕ) = 35°
- -joint plane cohesion (C) = 137.28 kN/m^2
- -vertical joint system spacing (W) = 15 m

At what minimum depth of mining below the pit crest can sliding wedge failure, for the physical and geometric conditions shown, be expected to occur?

QUESTION 5

Briefly answer the following, using sketches to illustrate your answers:

{10} (5.1) Explain how the presence of water and ground acceleration, induced by either blasting or seismic events, can affect the stability of open pit walls structures. For each of the two influences (exposure to groundwater and ground vibration), present two different cases, and show by illustrative example, how wall stability can be (a) detrimentally affected and (b) beneficially affected (or remain unchanged).

{10} (5.2) Describe the major functions of ground support media used to reinforce underground mine excavations. Provide examples of typical reinforcement media, and differentiate between support methods that have been adopted to (a) reinforce the structural capabilities of near-excavation rock zones and (b) provide purely surface or areal support enhancement for underground excavations.

Explain and differentiate between the mechanisms of support that can be developed using passive and active rock dowel systems. In your response, provide descriptions of the methods of installation and an explanation of the processes whereby bolt or dowel tensions are developed for each system.

QUESTION 6

In point form responses only, briefly describe the following:

{5} (6.1) Two methods, and their procedures for installation and, operation, by which rock mechanics or geotechnical staff can determine in-situ <u>uniaxially-directed</u> or <u>biaxially-directed</u> stress conditions from a single test

{5} (6.2) Some of the common problems that are associated with conducting overcore stress measurements in the field, and how these problems can be overcome.

{5} (6.3) Two types of instrumentation are commonly utilized by mining and geotechnical staff for the purpose of measuring both loading force and rock deformation conditions. Both types of instrumentation make use of strain gauge technology to provide such measurements. Relate the methods by which strain measurements are used by each type of instrument to provide output of force and deformation conditions.

{5} (6.4) Principal methods by which area support can be provided for underground mining excavations, with a statement for each which details the manner in which effective resistance to rock displacement is developed by them and significant variations in support capabilities offered by each.

Marking Scheme

1. 20 marks total (1 item for 10 marks, 2 for 4 marks and one for 2 marks))
2. 20 marks total (2 items at 10 marks each)
3. 20 marks total (2 items at 10 marks each)
4. 20 marks total (2 items at 10 marks each)
5. 20 marks total (2 items at 10 marks each)
6. 20 marks total (4 items at 5 marks each)