National Exams December 2018

09-MMP-A5, Surface Mining Methods and Design

3 hours duration

FORMAT:

Format: A Casio or Sharp approved calculator is permitted; Closed Book, but one aid sheet written on both sides containing notes and formulae is permitted

Suggested Timing: 30 minutes to read the exam and consider answers. Allow an hour for Question 1. Choose 3 of 5 multiple choice questions. Three multiple choice questions chosen from the five (Q 2 to 6), should take 30 minutes per question, 1.5 hours in all. Total 3 hours.

If you answer Question 5, **Be sure to include your name/number on the Appendices 5.1 and 5.2 and hand in with your exam paper.** If you make major errors on the Appendix copy, you can use the exam pages 13 and 14 and be sure to include your name/number on these pages 13 and 14 and hand in with your exam paper.

Marks are shown

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. One only reference sheet, 8.5 x 11 inch, hand written both sides is allowed in the exam. This is a a CLOSED book exam and an approved Sharp or Casio type calculator is permitted.

3. **Compulsory Question 1** and THREE (3) other questions constitute a complete exam paper. Only question 1 and the first three optional questions as they appear in the answer book will be marked. You must select three questions from the "optional" Questions 2 to 6.

4. Compulsory Question 1 is worth 40 marks. Each optional question is of equal value (20 marks). Three optional questions plus Question 1 constitute a complete exam paper.

5. Many questions require an answer in essay format. Clarity and organization of the answer are important.

6. Always use large (½ page or larger) neat sketches and drawings to illustrate your answers. This is important in obtaining good marks.

09-MMP-A5 Surface Mining Methods and Design

Question 1 A general question on the syllabus

(40 marks, 5 marks / 8 sections)

You must answer all of this question, parts 1.1 to 1.8 inclusive, 5 marks for each part

Question 1.1

What do you understand by the terms "availability" and "utilization" in the context of an 8760 hour year. What other definitions are available to make the mechanical and operating % availability and % utilization look better than they are. 2

Equipment is usually replaced based on hours. Which of the 'hours' would best be used in this context, and what are typical hours of life of mine mobile equipment. 3

Question 1.2

1.2.1 With respect to truck-shovel open pit operations, draw a sketch graph showing shovel production in tonnes per shift (Y axis) versus the number of trucks assigned to the hovel (X axis), and explain any curvature of the resulting relationship. 2

1.2.2 What could happen if the number of trucks assigned was (say) twice the number theoretically required. 1

1.2.3 What do you understand by the terms "spotting", "double back-up" and "drive by" in the context of open pit truck-shovel operations. Neat sketches are expected as are brief comparisons with respect to blast layout configurations. 2

Question 1.3

What do you understand by the term "cost index". 1

Describe two such indexes which might be used for estimating costs in the context of the capital and operating costs of future open pit mines. 2

What are the problems when such indexes are applied to all mine cost sectors and sub-sectors, and applied to periods exceeding (say) 5 years. 2

Question 1.4

With regard to open pit mine scheduling and profitability of typical copper porphyry and epithermal gold deposits;

1.4.1 What do you understand by the term "discount rate" 1

1.4.2 The present worth factors using a discount rate of 10% for 5 and 15 years are 3.791 and 7.606. If the capital cost of a mine is $500 million, what cash will the mine have to generate to retire the debt,

1.4.2.1 in 5 years 1

1.4.2.2 in 15 years 1

1.4.3 How does the discount rate affect the retirement of capital (debt). 1

1.4.4 Can the discount rate be used to quantify the risk of a change from an investment friendly climate to one where assets can be negatively impacted by changes in (usually developing) regions and governmental policies. 1

Question 1.5

1.5.1 In the context of open pit mine dewatering, what do you understand by the term "water hammer". 1

1.5.2 What additions to pipelines are a major cause of water hammer. 1

1.5.3 There are two major effects of water hammer in pipes. What are they. 2

1.5.4 What steps can be taken to minimise water hammer 1

Question 1.6

1.6.1 Describe the development of a block model typically used in studying the feasibility of open pit mines. What single piece of important information is attached to each block. 2

Question 1.6 continued

1.6.2 After applying such as already removed material, wall slope restrictions, rock types, discontinuities, and recoveries, there are two specific rules governing whether a "parcel of ore" (or ore block) will be included in the "optimal" open pit or not.

What are these two rules. 3

Question 1.7

1.7.1 With regard to conventional truck-shovel mining operations, explain why in hard rock open pit mines the displacement monitoring of wall slopes is essential, even after extensive rock mechanics design procedures have been employed. 2

1.7.2 Bieniawski is noted for his work on rock mechanics. Describe, and discuss the achievements and shortcomings of Bieniawski's work in this regard. 3

Question 1.8

1.8.1 Clearly define the steps taken (rule base) in the development of the moving (or floating) cone with reference to a simple two dimensional "vertical cut" section through the centre of an orebody. 3

In 1964 Lerchs and Grossmann introduced the first "true" three-dimensional algorithm to resolve deficiencies in the moving cone method.

1.8.2 Why is the Lerchs and Grossmann algorithm still not "optimal" in the context of an open pit mine with a life of 10 to 20 years or more. 2

Question 2 optimizing mining truck operations

2.1 Truck-shovel operations make up the majority of ore and waste transportation in open pit mines.

2.1.1 In the context of conventional open pit mining, what percentage of total operating cost is attributable to truck haulage. 1

2.1.2 List the 5 most likely delays to which trucks are subject to in the haulage cycle. 1

2.1.3 A conventional truck-shovel operation has a shovel which takes 6 minutes to load a truck. The complete full cycle time for the trucks, (loading, full haul, dump and return) is 45 minutes. What is the theoretical number of trucks which can be utilised in the operation. 1

2.1.4 The "Match Factor" (MF) is an important consideration in judging truck-shovel productivity. In the context of truck-shovel open pit mining what do you understand by the term "match factor", and define this in terms of multiple shovels and trucks. 1

2.1.5 Using the example of a shovel taking 6 minutes to load a truck and a fleet of five trucks with a full cycle time of 45 minutes, what is the MF. If the number of trucks calculated in 2.1.3 were used what would be the MF. 1

2.1.6 Is it possible for the MF at an operation to exceed 1 and what are the advantages. Draw a diagram of Match Factor (X) from zero to 2 and Theoretical Operational Efficiency (Y) from zero to 100%. Explain your diagram. 1

2.1.7 In the more complex case of several shovels of various types and a truck fleet of dissimilar capacity and manufacture, how is the match factor for the operation calculated. Using an example of 40 trucks and 5 shovels with a weighted average full cycle time of 45 minutes and a weighted average load time of 4 minutes, what is the MF. 1

2.2 Most open pit operations using shovels and trucks have invested in truck dispatch systems and software despite the large ongoing costs. Not only is productivity improved, but accounting and equipment operating statistics are better managed.

Figure 2.1 shows a sketch with two shovels operating. One shovel is digging waste which must be trucked to the waste dump and the other shovel is loading ore which must be trucked to the crusher.

The times taken by the trucks to travel to and from the shovels and dumps are shown below, as are the load and dump times. All times are in minutes.

• Shovel 1 to Crusher	12 minutes
• Crusher to Shovel 1	8
• Shovel 2 to Waste Dump	12
• Waste Dump to Shovel 2	8
• Crusher to waste Shovel 2	4
• Waste Dump to ore Shovel 1	3
• Loading Ore at Shovel 1	3
• Dumping Ore at Crusher	1
• Loading Waste at Shovel 2	3
• Dumping at Waste Dump	1

From the above, the average loading time for a truck, including truck positioning, is 3.0 minutes, ore or waste. The time taken for the trucks to reach the crusher is 12.0 minutes. Backup and dumping take 1.0 minute and the empty return 8.0 minutes to ore shovel 1 and 4 minutes to waste shovel 2. The time taken for the trucks to reach the dump is 12.0 minutes. Backup and dumping take 1.0 minute and the empty return 3.0 minutes to ore shovel 1 and 8.0 minutes to waste shovel 2,

2.2.1 With respect to truck routings in a truck-shovel mining operation, what do you understand by the terms "closed out" and "dispatched". Referencing Figure 2.1 showing all possible routings, draw two neat diagrams showing the respective closed out and dispatched routes and times. 1

2.2.2 What is the total "theoretical" truck cycle time and number of trucks required, and using these truck numbers, what is the match factor for either circuit, shovel 1 or 2 working independently of each other. 1

2.2.3 What are the theoretical numbers of trucks required for a "dispatched" operation included in Figure 2.1 to obtain maximum production. 1

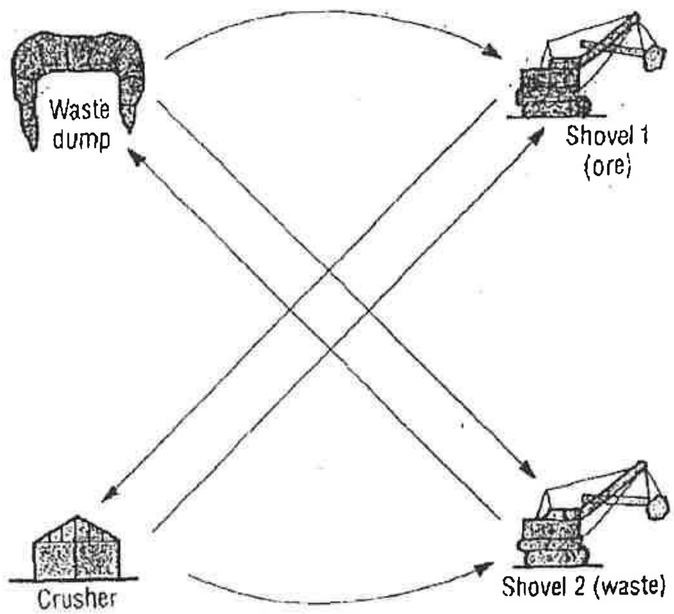

Figure 2.1 All Possible Truck Routings

2.2.4 Which trucking configuration is the most efficient, closed out or dispatched. 1

2.2.5 For full productivity, how many truckloads are delivered to the crusher and to the dump in an 8 hour shift. Theoretically how many trucks are required for full productivity, and are these values realistic. Note the 8 hours does not include 1.5 hours mealtimes, 3 by 0.5 hour snack breaks and 1.0 hour for combined start up and shut down making a 12 hour shift.

2.2.5.1 Closed out	number of truckloads	number of trucks	1
2.2.5.2 Dispatched	number of truckloads	number of trucks	1

2.2.6 What efficiencies has dispatching provided for the operation and estimate the total cost saving and the savings components. 1

2.3 All large open pits use the principles shown in figure 2.1 and in the text of 2.2.

2.3.1 Briefly, with respect to truck dispatch, what do you understand by the following and what does each accomplish 2

2.3.1.1 Best path (BP)
2.3.1.2 Linear program (LP)
2.3.1.3 Dynamic program (DP)

2.3.2 Briefly describe the mathematical algorithm which makes the truck assignments, and the software that communicates these assignments. Discuss the computer hardware employed on the shovels and trucks, what information is given to the computer and what information is received back from the computer. 2

2.3.3 Briefly discuss how the following are built into the dispatching algorithm which is run at frequent intervals during the shift. 2

2.3.3.1 Maximum production
2.3.3.2 Stripping ratio
2.3.3.1 Grade control

Question 3 **open cast dragline operations**

There are major marks for the neatness and clarity of the plan(s) and section(s) required to fully answer this question. Candidates could loose more than half marks for lack of clarity and neatness of drawings, especially in answering this question.

3.1 With respect to large walking draglines operating in open pit mines with relatively flat coal seams, and with the aid or neat sketch plans and sections, describe the following:- 6

3.1.1 Range diagram
3.1.2 Side casting
3.1.3 Advanced bench
3.1.4 Pullback
3.1.5 Extended bench
3.1.6 Multiple pass operations

3.2 With regard to side casting there are two essential tasks for a walking dragline.

3.2.1 Key cut 2

3.2.2 Deadheading 1

Explain each of the terms, how tasks are carried out and the importance of the results. Neat sketches are expected.

3.3 The following are the specifications of a walking dragline mining overburden covering a 3m thick flat single coal seam using simple side casting. The machine is working about half way along the cut in an established mine.

Dragline specifications

Tub diameter	20m
Operating radius	90m
Positioning factor	75%
Stacking height	12m
Digging depth	35m

The mined material specifications are as follows.

High wall angle (from horizontal)	63 degrees
Coal face angle (from horizontal)	90 degrees (to simplify calculations)
Spoil pile slope (from horizontal)	35 degrees
Spoil pile swell factor	0.25
Depth of overburden	25m
Pit width	40m

3.4 The term "dragline positioning" is used to describe the position of the dragline tub with respect to the edge of the high-wall. The term defines the percent of tub diameter from the dragline centerline to the edge of the high-wall. Draw a neat sketch section showing the dragline tub (20m diameter) and edge of high-wall when the "positioning" is 75%. 1

What is the distance from the edge of the high-wall to the maximum stacking distance, and from the high-wall edge to the center line of the dragline. 1

3.5 Draw a neat sketch section of a dragline side-casting operation showing; 2

- 3.5.1 Dragline operating radius
- 3.5.2 Stacking height
- 3.5.3 Cut depth
- 3.5.4 Dragline reach (reach factor)

3.6 Indicate on your neat sketch section 1

- 3.6.1 Pit width
- 3.6.2 Depth to bottom of coal seam
- 3.6.2 High-wall slope angle
- 3.6.3 Spoil pile angle of repose
- 3.6.4 Coal seam and thickness
- 3.6.5 Very simple 2 or 3 line schematic of dragline in position

3.7 The term "spoil pile swell factor" is the increase in volume of unbroken overburden when stacked as broken rock (spoil). It is usually stated as a decimal increase rather than a percentage. If a spoil pile swell factor is 0.25, what is the volume of 10 m^3 of unbroken overburden after digging and placing on the spoil pile. 1

3.8 Calculate the following for a 1m wide section of the operation 2

- 3.8.1 Cut area (a volume per meter wide slice)
- 3.8.2 Spoil pile area
- 3.8.3 Spoil pile height above coal seam floor
- 3.8.4 Operational stacking height (above dragline floor)
- 3.8.5 Horizontal difference between dragline dumping distance and top (peak) of spoil pile

3.9 Will the dragline be capable of mining the overburden and uncovering the coal as planned in question 3.8, i.e. can the dragline complete the operation of the mine without modifications to the mining method or machine. 1

3.10 There are many alternative operational changes that can be made if the dragline cannot uncover an area of coal that can be easily mined. Choose one method, and describe how the operation can be revised to achieve an adequate coal area for mining and still achieve the objective of moving the overlying overburden to the opposite side of the pit and into the area where the coal has already been extracted. Neat plans and sections are expected as part of your answer. 2

Question 4

equipment costing

4.1 "Pareto's Law" can be used to find the major cost components of the operating cost of drills, trucks or shovels in a typical truck/shovel hard rock open pit mine. Given that you have access to all accounting information for such a mine, how would you apply Pareto's Law to quickly reduce the operating costs of items such as drilling (for blast-holes), trucks (for rock haulage) and shovels (for loading trucks).

For each of the following three mine cost centres, list the two major consumables cost items (not maintenance, grease, fuel, power or operators, etc.) and their percentage of the total for that cost sector.

4.1.1 Drilling (for Blast-holes) 2

4.1.2 Truck (Haulage) 2

4.1.3 Shovel (Loading) 2

(an example answer for the blasting cost centre might be 30% ANFO, 25% Slurried Explosive, 20% Wages and Benefits, 10% Blast Hole Dewatering, 10% Detonators and Accessories and 5% "Other").

4.1.4 Discuss how inflation cost indexes could improve cost estimates of future operations. 2

4.2 A publication has produced capital cost indexes for a variety of mining purchases based on the equipment size for the year 1997. Such information allows the mining engineer to estimate capital costs (US or CDN $) for a potential mining project by the various cost centres.

4.2.1 For drills the relationship is

$P=a X^{b}$ where P is the capital cost of a drill, X is the pull-down force in pounds (lbs.), and a and b are constants, 4,000 and 0.67 respectively.

What is the 1997 cost of a 55,000 kg pull-down force rotary drill. 2

4.2.2 For cable shovels the relationship is

$= a X^{b}$ where P is the capital cost of a shovel, X is the shovel bucket capacity in cubic yards and a and b are constants, 5,350,000 and the power 0.74 respectively.

What is the 1997 cost of a 53 cubic meter bucket size shovel. 2

4.2.3 For mechanical drive trucks the relationship is

$P = a X^b$ where P is the cost of a truck, X the truck capacity in short tons and a and b are constants, 20,000 and the power 0.90 respectively

What is the 1997 cost of a 300 tonne truck. 2

(1 lb = 0.4536 Kg 1 m^3 = 1.308 yd^3 1 tonne = 1.102 ton)

4.3 A cost index estimates that a 1997 dollar is now worth 2 dollars
What are the costs of buying the drill, shovel and truck today, and are the values realistic. 2

4.4 T.A, O'Hara has developed an empirical 'mine life' rule used by producing operations in the minerals industry.

What is the 'rule'. 2

Explain why you feel this rule to be accurate/inaccurate in its use in feasibility studies involving the 'time value of money'.` 2

Question 5 **mine planning and scheduling**

This question involves copper porphyry and epithermal gold deposits mined with an initial pit and followed by surrounding or partially surrounding phases or shells. Often the phases or shells are started before the previous excavation has been completed, and scheduling to maintain ore production while maintaining stripping ratios is essential. This can cause mining of the later phase to blast rock down onto the previous phase which is the source of scheduled high grade ore, and some thought and planning are necessary to maintain ore production and the stripping ratio.

In Figure 5.1.1 mine ore production is prepared for scheduling. Working Figures 5.1.1 and 5.2.1 are attached at the end of the exam as Appendices 5.1 and 5.2.

In this operation there is a start-up period shown as (end of year) -1 and (end of year) 0 where the mine is stripping waste to expose Phase 1 (the initial pit) ore (Figure 5.1.1) while the mill is under construction.

Elev-ation	Phase 1 Ore	Phase 2 Ore	Year One Phase						Year Two Phase				
			-1	0	1	2	3	4	4	5	6	7	8
2000	0	0											
1985	0	0											
1970	5	2											
1955	9	4											
1940	6	6											
1925	3	5											
1910	1	4											
1895	0	3											
1880	0	2											
Totals	24	26											

Ore Schedule 7 million tonnes per year starting Year One to completion

Figure 5.1.1

At the end of year one, 7 million tonnes of ore have been mined and scheduled annually until the ore is exhausted, Figure 5.1.1 and Appendix 5.1.

The year shown at the top of the figure refers to the end of the year. For example, year "0" ends at the start of the ore mining and mill operations which have been under construction during years "-1" and "0". Year "1" is material mined from mill start-up lasting for one year.

The mine operates in 2 Phases and management demands the "ore' production, normally Phase 1 ore before moving to Phase 2 ore. The critical objective of the mine is to produce 7 million tonnes of ore annually starting in year 1 and continuing each year until ore depletion and end of mine, or development of Phase 3 etc.

During these early years (-1 and 0) no ore is mined, and if any is inadvertently encountered it would be stockpiled.

Note the nomenclature of the mining year in Figures 5.1.1 and 5.2.1. At the end of the first pre-production year (called minus 1) the waste start up stripping of 6 million tonnes is completed with one shovel and sufficient trucks.

At the start of year 0, waste pre-production stripping of 12 million tonnes with at least two extra shovels and truck fleets is started and further shovel and truck fleets are under construction to mine ore, but no ore is mined as yet, Figure 5.2.1. (Appendix 5.2)

Elevation	Phase 1 Waste	Phase 2 Waste	Year One Phase -1	0	1	2	3	4	Year Two Phase 3	4	5	6	7
2000	5	2											
1985	8	4											
1970	11	5											
1955	9	11											
1940	7	8											
1925	3	7											
1910	0	3											
1895	0	1											
1880	0	0											
Totals	43	41											

Waste Schedule
6 million tonnes per year in first pre-production Year -1 (minus One)
12 million tonnes per year second pre-production Year 0 (zero)
Up to 12 million tonnes of waste mined per year thereafter (1, 2, . . ?)

Figure 5.2.1

In the Figures 5.1.1 and 5.2.1 the simple "North West Corner" scheduling method is used as a guide to mine planning and scheduling. This ensures an orderly operation, starting at the highest working elevation in the earliest period. The method works down to fulfil the production quota then right to the next mining period to fill the next annual quota.

When mining ore, the bench(es) above should have removed enough waste to uncover the ore to make the ore mineable. To ensure waste is not removed too early, incurring a cost which could have been delayed and improving profits, the rules are that in any given year, the amount of waste removed on a bench should not exceed the amount of ore removed. The rules on mining waste neither too late nor too early are the responsibility of the mine planner/scheduler.

First the ore schedule must be constructed which mines 7 million tonnes per year starting in year one mining the first phase and then moving to the second. The choice of moving ore from both phases when the first phase is near completion is again the responsibility of the mine planner/scheduler.

5.1.1 In Figure 5.1.1, complete the ore schedule and use Appendix 5.1 to tabulate your answer. **Be sure to include your name/number on the Appendix 5.1 and hand in with your exam paper.** If you make major errors on the Appendix copy, you can use the copy on the exam page 13 and ***be sure to include your name/number on the page 13 and hand in with your exam paper.*** 2

5.1.2. Describe how ore production moves from Phase 1 to Phase 2, including the elevations and tonnages at the transition. Discus how the mine personnel accomplishes the transition given that the shovels are large and electrically powered. 2

5.1.3. Based on the porphyry and epithermal geology, some ore may be stockpiled for later mining. How, and in what circumstances, would you build and later mine the ore stockpile. 1

What relative grade values would be stockpiled, and under what circumstances would grades mined in the pit be less than those stockpiled. 1

A neat sketch diagram is expected in 5.1.3 (and possibly 5.1.2) to help in explaining your answer.

5.1.4 Given that the deposit is a copper porphyry and/or epithermal gold, draw a sketch of the relative head grade to be expected over the life of the deposit. How will the mill and concentrate load out have to adjust to handle such variability. 1

5.1.5 If the availability and productivity of the shovels reduces significantly due to poor blasting practices, what could happen as Phase 1 comes to completion. 1

5.2.1 In Figure 5.2.1 and Appendix 5.2, complete the waste mining schedule based on the production expected and the completed ore schedule in Figure 5.1.1. Ensure that the rules governing waste mining are followed, **Be sure to include your name/number on the Appendix 5.2 and hand in with your exam paper.** If you make major errors on the Appendix copy, you can use the copy on the exam page 14 and ***be sure to include your name/number on the page 14 and hand in with your exam paper.*** 3

5.2.2 What elevations in Phase 1 and 2 are scheduled for waste removal when Phase 1 deep ore is being mined. What are the safety considerations and will these impact on the ore tonnage mined deep in Phase 1. How can safety be improved when waste blasting throws rock into deeper ore mining elevations. 1

5.2.3 Draw a sketch of the truck hours required over the life of the mine. How would you incorporate this information in an overall mine plan that attempts to maintain constant truck hours. 1

5.2.4. Will the purchase/rental of used trucks help any problems envisaged in 5.2.3 above. The example Figures 5.1.1 and 5.2.1 assume that truck productivity is constant. 1

What effects do you expect "time" and "depth" to have on truck productivity and discuss the acquisition of equipment (not necessarily more trucks) to offset any shortfall. 1

5.2.5 What is the maximum and minimum annual stripping ratio, and describe how this knowledge will help your attempt to maintain constant truck hours. 2

5.2.6 Is there any year(s) where waste mining is started/completed before ore is mined. What rule in open pit mining is broken in this event. 1

5.2.7 Will your schedule have mining on the upper levels of Phase 2 working above ore shovel(s) working in Phase 1. Draw a mine plan at this juncture and explain how safety and productivity can be assured in this instance. 1

5.2.8 In your opinion, has the simple mine scheduler been successful in giving the short/medium/long term planning engineer a solid background to make short term scheduling through long term planning scenarios. If not, what would you recommend. 1

Question 6 **open pit dewatering**

Question 6.1

A mine has chosen to use in-pit pumps in a sinking cut sump to remove water. There are two of the same type of pump available with the characteristic curves shown in Figure 6.1. below. The pumps can be used in high volume (MT) or high head (HT) configuration. The pumps can also be used submersible (with a screen Question 6 open pit dewatering
adaptor around the intake) or in series (tandem) with a pipe adaptor at the inlet such that the pump inlet may have a positive pressure.

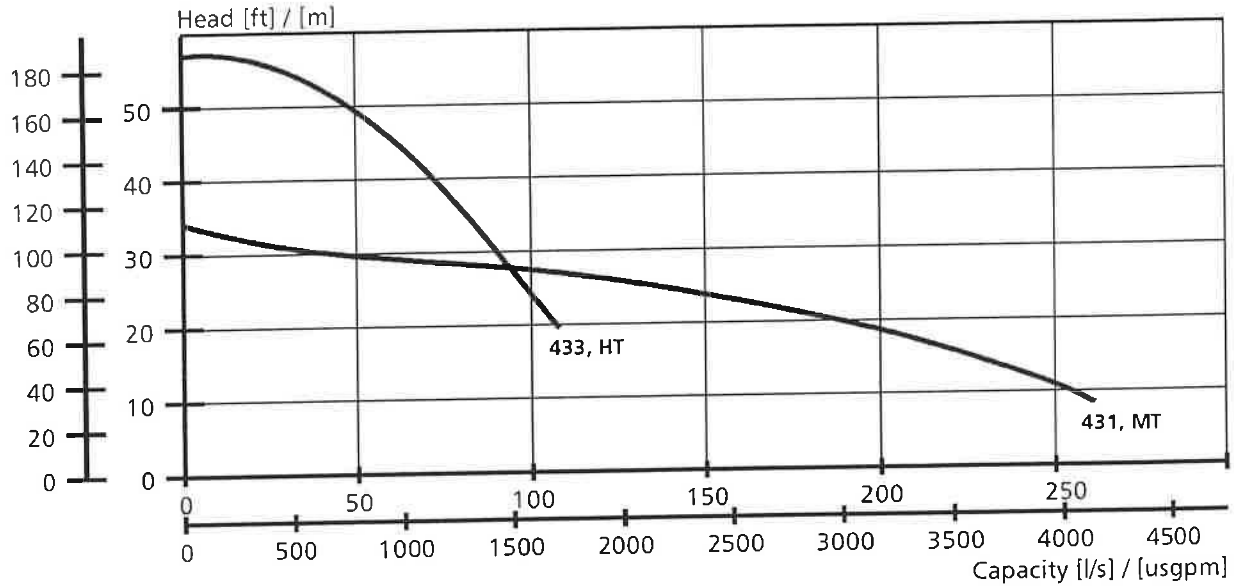

Figure 6.1 Pump characteristics (head and flow rate) for high head (433, HT) and high volume (431, MT) versions. Sown US imperial and metric.

The mine has to move water from the sinking cut sump at 1650 meters to the pit crest discharge at 1740 meters, a 90 meter lift. The 15 meter benches from 1695 meters and up are accessible for the laying of high density polyethylene pipe, but are not suitable for pump infrastructure such as power lines and generators or for pump etc. maintenance.

The cost of a pipe line buried in the 10% ramp (900+ meters) is regarded as far too expensive and not easily repaired and therefore ignored.

One pump will be placed in the sump (1650m) and run as a submersible unit with moveable pipe laid on the ramp edge. The other pump will be placed as far as possible up the ramp at the 1675 meter elevation. At this upper pump, a tandem fitting will be coupled directly to the pipeline coming up from the sump. The upper tandem pump must deliver sufficient combined pressure from a pipe suspended on the pit wall to directly discharge water at the pit crest. You may assume that the friction loss in the large diameter polyethylene pipe used is small over the total length in question.

6.1.1 From the graph, Fig. 6.1, will the high volume pump configuration (431, MT) be used for both pumps, or the (433, HT), or a combination of both. Note that the 2 pump types are essentially the same, but changing from one configuration to the other is skilled work for the manufacturers distant representatives mechanics. 3

6.1.2 What will be the maximum volume of water (liters per second) discharged at the pit crest based on the best pump configurations (HT and/or MT). 2

6.1.3 What will be the inlet and outlet pressures at each of the pumps in meters of water. 2

6.1.4 What schedule of pipe will be required at the outlets of the two pumps 2

6.1.5 If a check (gate) valve is fitted at the outlet of the top pump to stop water returning to the sump, what modifications will be required. 2

Assume the water is "clean" and temperature/pressure have no effect on density.

Some conversion factors that may or may not be of use,
1 m water at 8^0C is 1.41 psi 1 ft water at 62^0F is 0.43 psi 1 psi = 6.9 kPa 1 ft water = 3 kPa

Question 6.2

An alternative to the pumping system described in 6.1 is a series of deep well pumps around the pit perimeter and possibly on the ramp inside the pit.

From this perspective, described the work by Jacob, Theis and others, and discuss;

6.2.1 How the transmissibility and storage constants of the pit wall rocks are found using a graphical solution and experimental wells. 3

6.2.2 How the pumps might be laid out in plan and the pump depth determined. 2

6.2.3 How the feasibility and cost of such a system might be estimated. 2

6.2.4 The operational advantages quantified from a pit operations perspective. 2

End of the Exam **(Appendices 5.1 and 5.2 Follow)**

Elev-Ation	Phase 1 Ore	Phase 2 Ore	Year One Phase -1	0	1	2	3	4	Year Two Phase 4	5	6	7	8
2000	0	0											
1985	0	0											
1970	5	2											
1955	9	4											
1940	6	6											
1925	3	5											
1910	1	4											
1895	0	3											
1880	0	2											
Totals	24	26											

Ore Schedule

7 million tonnes per year starting Year One to completion

Figure 5.1.1

Appendix 5.1

Your Name/Number ________________

Elev-Ation	Phase 1 Waste	Phase 2 Waste	Year One Phase -1	0	1	2	3	4	Year Two Phase 3	4	5	6	7
2000	5	2											
1985	8	4											
1970	11	5											
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1925	3	7											
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1895	0	1											
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Waste Schedule

6 million tonnes per year in first pre-production Year -1 (minus One)
12 million tonnes per year second pre-production Year 0 (zero)
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Figure 5.2.1

Appendix 5.2

Your Name/Number ____________