National Exams December 2013

09-Mmp-A5, Surface Mining Methods and Design

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. One only reference sheet, 8.5 x 11 inch, hand written both sides is allowed in the exam. This is a Closed book exam, therefore only the approved Sharp or Casio type calculators are permitted.

3. Compulsory Question 1 and FOUR (4) other questions constitute a complete exam paper.

   Only question 1 and the first four optional questions as they appear in the answer book will be marked. You must select four questions from the "optional" Questions 2 to 7. Be sure you understand that two of Questions 2 to 7 must not be answered.

4. Compulsory Question 1 is worth 40 marks. Each optional question is of equal value (15 marks). Four 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. Use sketches and drawings to illustrate your answers whenever possible.

6. *If you answer Question 2 and/or 5, make sure you write your name on the appropriate figures and hand in with your exam answer booklets.*

Question 1.1 **(6 marks)** ***answer compulsory***

1.1.1 You have been asked by your employer to provide some quick fixed capital cost estimates for the open pit section of a mine. Who would you communicate with to provide a quick estimate +/- 40% within a day.

1.1.2 What do you understand by the "O'Hara Capital Cost Estimation Method" as published in the Canadian Institute of Mining Bulletin, February,1980 and further described by Mular and Poulin in "CapCost, CIM Special Volume 47, 1998" and elsewhere in SME and other publications.

1.1.3 Working today as a capital cost estimator, how would you convert 1980 and 1998 costs to present day (2012) and say 2015 estimates.

(2 marks each 1.1.1 to 3)

Question 1.2 **(5 marks)** ***answer compulsory***

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

1.2.1 What do you understand by the term "discount rate" and give the discounting factor for cash flows 5 and 15 years into the future using a 10% discount rate.

1.2.2 If the "cash flow" is $100 million in each case in (1.2.1), what are the discounted values at year zero.

1.2.3 How does the discount rate affect the design and scheduling of open pit mines.

1.2.4 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 governmental policies.

(1 mark each 1.2.1, 2 and 4 and 2 marks for 1.2.3)

Question 1.3 **(5 marks)** *answer compulsory*

Explain with the use of clear neat diagrams the action of the mechanism for moving large loading and digging equipment using the following three methods. Your answer should include a note on the mobility of the equipment to mine selectively on a short term basis, and the difficulty of moving such equipment from one bench to another. Explain how each type can (or cannot) normally load trucks.

1.3.1 Tracked equipment

1.3.2 "Walking" equipment.

1.3.3 Rubber tire wheeled equipment.

1.3.4 Comment of the use of a close mobile hopper/conveyor to move the mined material replacing the use of trucks.

(one mark each for 1.3.1, 2 and 3, and 2 marks for 1.3.4).

Question 1.4 **(6 marks)** *answer compulsory*

With respect to conventional dragline mining operations, answer the following questions indicating the conditions which make such operating methods necessary and why these methods are used. Use sketch sections/plans where appropriate.

1.4.1 Key cut

1.4.2 Extended bench

1.4.3 Advanced bench

(2 marks each, total 6 marks)

Question 1.5 **(6 marks)** *answer compulsory*

1.5.1 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" through the centre of an orebody.

Question 1.5 continued

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

1.5.2 What were those deficiencies.

1.5.3 Why is the Lerchs and Grossmann algorithm still not "optimal" in the context of an open pit mine with a 10 to 20 year life.

(2 marks each, total 6 marks)

Question 1.6 (6 marks) *answer compulsory*

With respect to truck dispatch, what do you understand by the following and what does each accomplish.

1.6.1 Best path (BP)

1.6.2 Linear program (LP)

1.6.3 Dynamic program (DP)

(2 marks each)

Question 1.7 (6 marks) *answer compulsory*

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

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

1.7.3 What steps can be taken to minimise water hammer.

(2 marks each)

Question 2 (15 marks) One of Four Optional Questions to be Selected

This question involves copper porphyry and epithermal gold deposits.

In Figure 2.1.1 mine ore production is prepared for scheduling. In this operation there is a start-up period shown as (end of year) -1 and 0 where the mine is stripping waste (Figure 2.2.1) to expose Phase 1 ore while the mill is under construction. During these years no ore is mined.

At the end of year one, 7 million tonnes of ore is mined and scheduled annually until the ore is exhausted. The mine operates in 2 Phases and must complete Phase 1 ore before moving to Phase 2 ore. Lengthy shovel moves to and from Phases 1 and 2 take time and excessively wear the shovel(s).

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.

Phase 2 waste stripping (2000 and 1985 benches) has to be limited as far as possible as blasting rock down into Phase 1 from Phase 2 mining should be avoided, while meeting annual ore production. Phase 1 waste should be completed before starting Phase 2 waste, again to avoid lengthy shovel moves between Phases.

Note the nomenclature of the mining year. At the end of the first pre-production year (called minus 1) the waste start up stripping of 6 million tonnes is completed. At the end of year 0, waste pre-production stripping of 12 million tonnes is completed but no ore is mined yet.

At the completion of year 1, the first year of ore production, a total of 19 million tonnes is moved of which 12 million is waste and 7 million ore. This is the maximum annual production.

This scenario is accomplished as shovels and their truck fleets are built and the mill is constructed for full production at the start of year 1 which is also the end of year 0.

2.1.1 On Figure 2.1.1 complete the yearly ore schedule by period for both mining Phases and by the amount of ore mined on each particular bench elevation (1970 down to 1880 in 15 meter benches). This is sometimes referred to as a “North West Corner” solution to the scheduling.
(4 marks)

2.1.2 Draw a sketch of the typical annual head grade mined over the life of the operation and explain the curve(s). No grades are given but remember this is a porphyry copper/epithermal gold type deposit.
(2 marks)

You **must detach the figure(s) which form part of your answer** and place them in your exam answer book with your name printed in the space provided on **all the required figures.**

In Figure 2.1.2 a similar schedule that must be completed for waste at a maximum production of 12 million tonnes per year. The rules are that on any particular bench, waste cannot be mined ahead of ore, so the annual schedule for waste is always one bench above the bench mining ore. In the event of a conflict, waste mined on a bench cannot exceed ore mined on the same bench for that period.

2.2.1 On Figure 2.2.1 complete the annual waste schedule by the amount of waste mined on each bench elevation (2000 down to 1895 in 15 meter benches) and by Phase (1 and 2) and yearly period. This is a constrained or modified north-west corner solution.

(4 marks)

2.2.2 Draw a sketch of the typical truck hours required to achieve the (ore + waste) production required and scheduled.

(2 marks)

2.2.3 How can the truck hours in the schedule be converted to number of trucks required, and how can the number of trucks purchased be minimized while still maintaining the ore production schedule.

(3 marks)

You **must detach the figure(s) which form part of your answer** and place them in your exam answer book with your name printed in the space provided on **all the figures.**

Question 2 Figure 2.1.1 **Your Name/Number** ____________________________

Ore Schedule 7 million/yr starting year 1 to completion (1, 2 ... ?)

Elev-ation	Phase One Ore	Phase Two Ore	Phase 1 Year						Phase 2 Year				
			-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											

Question 2 Figure 2.2.1 **Your Name/Number** ____________________________

Waste Schedule

6 million tonnes per year first pre-strip year (-1)

12 million waste per year second pre-strip year 0

Up to 12 million of waste are mined from year 0 onwards (0, 1 ... ?)

Elev-ation	Phase One Waste	Phase Two Waste	Phase 1 Year					Phase 2 Year				
			-1	0	1	2	3	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										

Question 2 Figure 2.1.1 ***SPARE COPY*** **Your Name/Number ____________**

Ore Schedule 7 million/yr starting year 1 to completion (1, 2 ... ?)

Elevation	Phase One Ore	Phase Two Ore	Phase 1 Year -1	0	1	2	3	4	Phase 2 Year 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											

Question 2 Figure 2.2.1 ***SPARE COPY*** **Your Name/Number ____________**

Waste Schedule 6 million per year first pre-strip year (-1)

12 million waste per year second pre-strip year 0

Up to 12 million of waste are mined from year 0 onwards (0, 1 ... ?)

Elevation	Phase One Waste	Phase Two Waste	Phase 1 Year -1	0	1	2	3	Phase 2 Year 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										

Question 3 (15 marks) One of Four Optional Questions to be Selected

A mining operation is moving ore material such as oil sands from a relatively shallow depth, and the ore is covered by overburden consisting of top soil and barren sands.

Initially the mine equipment consists of either;

3.a) Large draglines which advance continually on a wide face along the ore while sitting on the overburden. The ore is placed on the bench beside the dragline and the waste, as mined, is cast back into the area already mined out. Bucket wheel excavators, on the same elevation as the dragline, re-handle the ore and place the material in hoppers feeding a long conveyor which advances ahead of the dragline and feeds the concentrator. No blasting takes place.

3.b) Large bucket wheel excavators which move lightly blasted ore and waste onto separate conveyors which either carry waste to the mined out area, or onto long mobile conveyors which traverse over the mined out area and feed the concentrator.

3.c) When the above equipment reaches the end of its useful life, it is replaced by "conventional" truck/shovel operations.

3.1 Draw a sketch plan and section of each operation (3.a, 3.b and 3.c).

(1 mark each for 3.a, b, and c, total 3 marks)

3.2 Compare the three operations 3.a, 3.b and 3.c from the point of view of

3.2.1 Productivity
3.2.2 Unit Cost
3.2.3 Selectivity of Mining
3.2.4 Grade Control
3.2.5 Concentrator Throughput and Recovery

(2 marks each, total10 marks)

3.3 Discuss why all newer "conventional" oil sand and other similar deposits use "conventional" truck/shovel methods

(2 marks)

Question 4 (15 marks) **One of Four Optional Questions**

4.1 In the context of "conventional" dragline coal mining operations what do you understand by the term "Range Diagram".

(1 mark)

4.2 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.

4.2.1 Draw a neat sketch section showing the dragline tub (20m diameter) and edge of high-wall when the "positioning" is 75%.

4.2.2 What is the distance from the edge of the high-wall to the nearest edge of the tub, and from the high-wall edge to the center line of the dragline.

4.2.3 The dragline reach is the distance from the edge of the highwall to the end of the operating radius. What is the dragline reach.

(1 mark each, total 3 marks)

4.3 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 rather than a percentage.

4.3.1 If a spoil pile swell factor is 0.25, what is the volume of 1 m^3 of unbroken overburden after digging and placing on the spoil pile.

(1 mark)

4.4 Draw a neat sketch section to scale (this enables you to check your answers later in section 4.5) of a dragline side-casting operation showing;

4.4.1 Dragline operating radius 90m

4.4.2 Dragline tub diameter 20m

4.4.3 Stacking height (maximum) 12m

4.4.4 Cut depth 25m

Question 4.4 continued

4.4.5 Positioning factor 75%

4.4.6 Pit width 40m

4.4.7 Depth of overburden (machine can dig to 35m) 25m

4.4.8 High-wall slope angle 63 degrees

4.4.9 Spoil pile angle of repose 35 degrees

4.4.10 Spoil pile swell factor 25%

4.4.11 Coal seam thickness 3m

(total 2 marks)

<u>Calculate</u> (not measure from your section) the following for a 1m wide section of the operation

4.5.1 Cut area (an area per meter wide slice)

4.5.2 Spoil pile area

4.5.3 Spoil pile height above coal seam floor

4.5.4 Height of the spoil pile above base of dragline tub

4.5.5 Operational stacking height (not stacking height specified by dragline)

4.5.6 Horizontal reach factor (crest of overburden to vertical line from top of stacked spoil)

(1 mark each, total 6 marks)

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

(1 mark)

4.7 What auxiliary mining methods and/or extra equipment would be required to accomplish the mining if the dragline and its mining configuration were inadequate for purpose.

(1 mark)

Question 5 (15 marks)

One of Four Optional Questions

Question 5.1 In the optimization of open pits, the first stage is to convert grade to a block model of cash flows.

5.1.1 What do you understand by a block model, and how is each block given a single unique number which can be converted to X, Y and Z co-ordinates and vice versa.

5.1.2 How are block costs broadly defined.

5.1.3 How is a "grade" converted to a revenue.

5.1.4 What is the basis for defining the "cash flow" of a block and how are waste blocks determined and a cash flow allocated.

5.1.5 At some operations "least loss" is used to define whether material is processed or sent to waste or low grade stockpiles. Define the rule base for "least loss".

(1 mark each, total 5 marks)

Figure 5.2 below shows a simple 2 dimensional section of an open pit being mined with 15m^3 blocks (15 x 15 meter square blocks in single section 2D drawing) and a 45 degree wall slope.

The cash flow per block is indicated showing ore and waste as positive and negative values respectively. A potential expansion is outlined and shaded on the right hand pit wall. The example is "contrived" to demonstrate the problem. A spare copy is included as part of your written exam.

The lower part of the Figure 5.2 is available for calculations and a spare copy of the figure is attached. **Make sure you hand in your Figure 5.2 with your name/number shown clearly.**

Figure 5.2 Proposed Pit Expansion Your Name/Number ____________________

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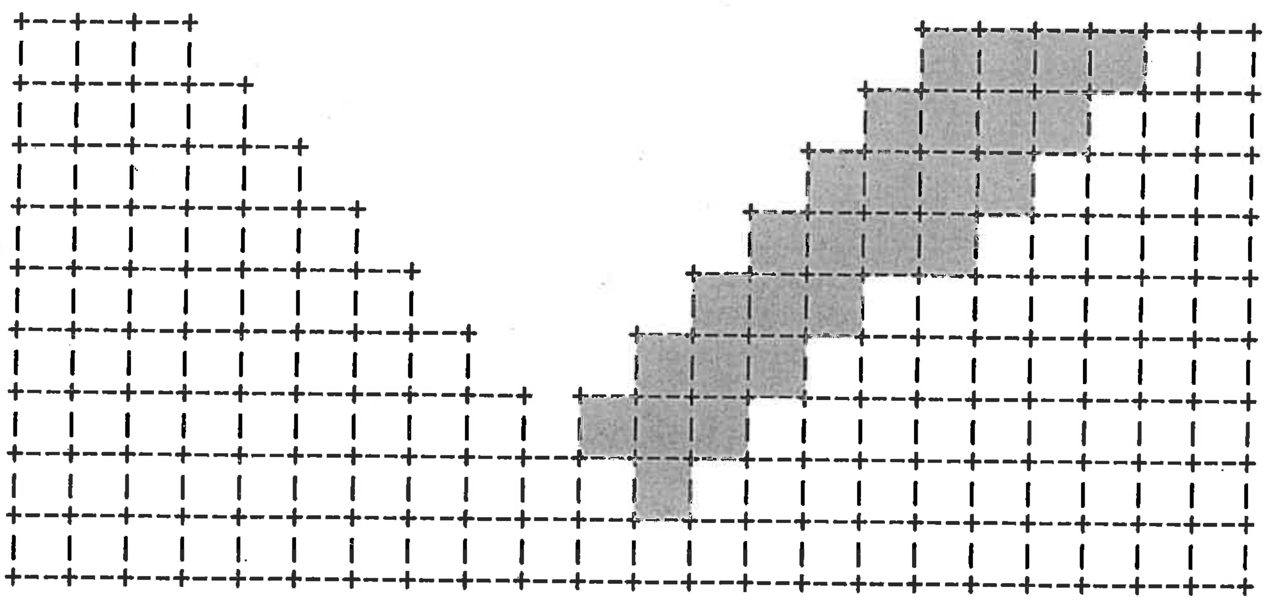

Figure 5.2. Top part shows the cash flow for each block on a single two dimensional section for a typical pit expansion. The middle and lower parts of the figure are spaces to calculate the Lerchs-Grossman "optimal" pit expansion. Blocks are 15x15m and wall slope 45 degrees.

Figure 5.2 Proposed Pit Expansion *SPARE COPY*   Your Name/Number ___________

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Figure 5.2. Top part shows the cash flow for each block on a single two dimensional section for a typical pit expansion. The middle and lower parts of the figure are spaces to calculate the Lerchs-Grossman "optimal" pit expansion. Blocks are 15x15m and wall slope 45 degrees

On the section Figure 5.2, a pit expansion to the right is indicated as shaded. Use your two dimensional moving (floating) cone definitions from your answer to question 1.5.1 to explain your answers at each stage;

For the optimal pit outline;

5.2.1 Develop the "optimal" pit based on your answer to question 1.5.1

5.2.2 How many waste and ore blocks are mined

5.2.3 What is the total "cash flow"

5.2.4 Is this the "optimal" pit expansion, and if not, what is.

5.2.5 Neatly show your "optimal" pit outline on your Figure 5.2. Do not forget to place your name/number on the Figure 5.2 and hand it in with your answer booklet. An extra copy of Figure 5.2 is provided in case you need to revise your "optimally" mined out area.

(2 marks for each of 5.2.1 to 5.2.5, total 10 marks)

Space has been allocated on the figure for calculations and a spare duplicate copy is included in case you need to revise your answer.

You **must detach the figure(s) which form part of your answer** and place them in your exam answer book with your name printed in the space provided on **all the figures.**

Question 6 (15 marks) **One of Four Optional Questions**

6.1.1 With respect to truck dispatching used in mines, what do you understand by the terms "closed out" and "dispatched".

Truck dispatch has now become an integral part of the operation of open pit mines. Explain the methodology used in such a computerised system when applied to the following headings;

6.1.2 Maximized production

6.1.3 Minimized equipment (trucks)

6.1.3 Optimal mill head grade

6.1.4 Maintaining the stripping ratio

Question 6.1 continued

6.1.5 Provision of a computerised data base for production, maintenance, control of consumables and future truck purchases

(1 mark each for 6.1.1 to 6.1.5 total 5 marks)

6.2 Figure 6.2 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.

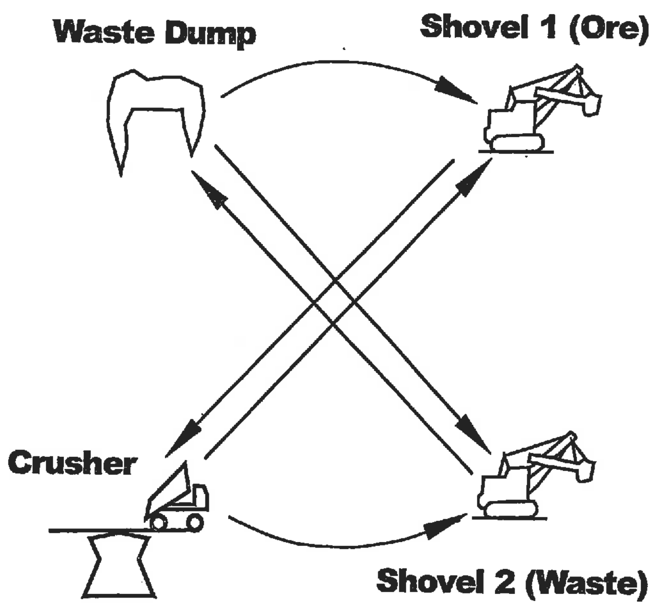

Figure 6.2 All Possible Truck Routings

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 Shovel 2 4

Question 6.2 continued

•	Waste Dump to Shovel 1	3
•	Loading at Shovel 1	3
•	Dumping Ore at Crusher	1
•	Loading Waste at Shovel 2	3
•	Dumping at Waste Dump	1

It is not necessary to convert numbers to integers for the following question 6.2.

6.2.1 Referencing Figure 6.2 showing all possible routings, draw two neat diagrams showing the respective "closed out" and "dispatched" routes. (2 marks)

6.2.2 What are the theoretical numbers of trucks required for "closed out" and "dispatched" operation in Figure 6.2 to obtain maximum production. (2 marks)

6.2.3 Which trucking configuration is the most efficient, closed out or dispatched. (1 mark).

6.2.4 Using the most efficient configuration, how many truckloads are delivered to the crusher and dump in an 8 hour shift and is this value realistic. (1 mark)

6.2.5 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 shovel (X axis), and explain any curvature of the resulting relationship. (2 marks)

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

(1 mark each for 6.2.1 to 6.2.6 total 6 marks)

Question 6.3 relates to general knowledge of truck/shovel operations

6.3.1 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 regard to blast layout configurations.

6.3.2 In the context of truck-shovel mining, what do you understand by the term "match factor".

6.3.3 The average loading time for a truck, including truck positioning, is 3.0 minutes. 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.

What is the total "theoretical" truck cycle time and match factor.

6.3.4 All large open pits use the principles shown in Figure 6.2 and in the text of question 6 above.

Briefly describe the computer hardware employed on the shovels and trucks to achieve maximum trucking efficiency in such large pits. You may use "bullet" form headings and a brief explanation for each "bullet" in order to answer this part (6.3.5) of question 6.3.

(1 mark each for 6.3.1 to 6.3.4, total 4 marks)

Question 7 (15 marks) *One of Four Optional Questions*

Question 7.1 (8 marks)

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 7.1. 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 adaptor around the intake) or in series (tandem) with a pipe adaptor at the inlet such that the pump inlet may have a pressure.

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, 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 and the other placed as far as possible up the ramp at the 1695 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 pressure to discharge water at the pit crest. You may assume that the friction loss in the large diameter polyethylene pipe used is small over a 125 meter length on a 45^0 pit wall.

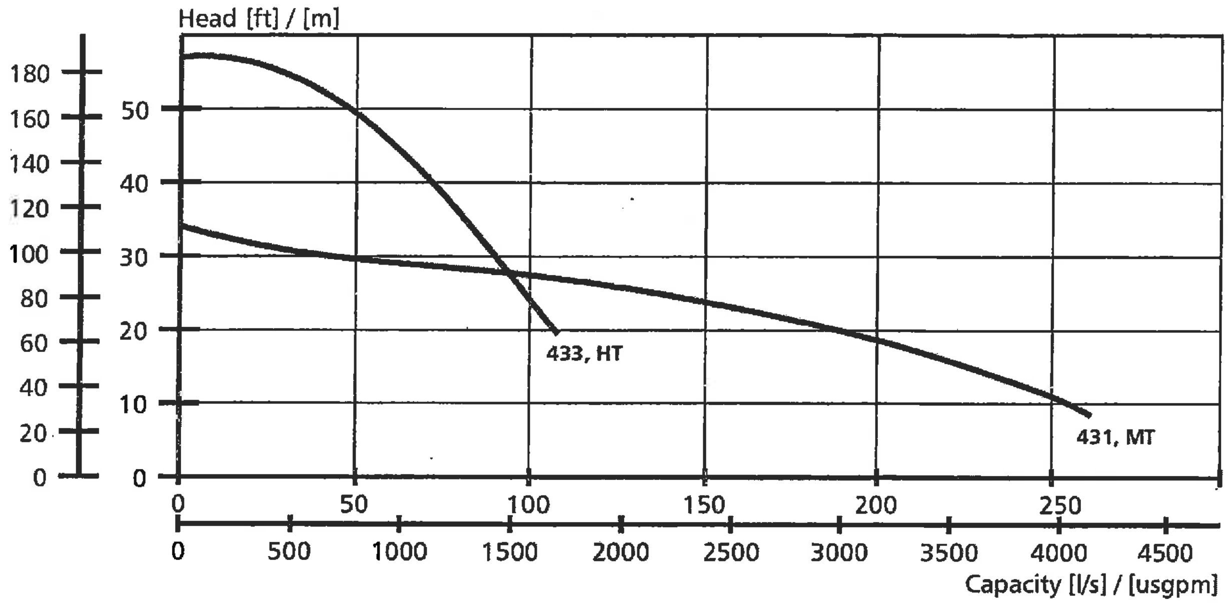

Figure 7.1 Pump characteristics (head and flow rate) for high head (433, HT) and high volume (431, MT) versions.

7.1.1 From the graph, Fig. 7.1, will the high volume pump configuration (431, MT) be used for both pumps.

7.1.2 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).

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

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

7.1.5 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.

(1.5 marks each for 7.1.1 to 7.1.4 and 2 marks for 7.1.5 , total 8 marks)

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

Some conversion factors that may or may not be of use,

1 m water at 8^{0}C is 1.41 psi   1 ft water at 62^{0}F is 0.43 psi   1 psi = 6.9 kPa

1 ft water = 3 kPa

Question 7.2 (7 marks)

An alternative to the pumping system described in 7.1 is a series of deep well pumps around the pit perimeter and possibly on the ramp inside the pit.

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

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

7.2.2 How the pumps might be laid out in plan and the pump depth determined. (1 mark)

7.2.3 How the feasibility and cost of such a system might be estimated. (2 marks)

7.2.4 The operational advantages quantified from a pit operations perspective. (1 mark)

End of the Exam

Do not forget to put your name on Figures 2.1.1 , 2,2,1 and 5.2 and hand in with your exam paper.