National Exams May 2015

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 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 four questions from the "optional" Questions 2 to 7.

4. Compulsory Question 1 is worth 40 marks. Each optional question is of equal value (20 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 large neat sketches and drawings to illustrate your answers when possible.

6. If you answer Questions 4 or 6, make sure you hand in Figures 6.1 and 6.2 and/or 4, 4a and 4b with your number/name attached clearly in the space provided.

Compulsory Question 1 (40 marks)

You must answer **all** *of this question, parts 1.1 to 1.6 inclusive*

Question 1.1 (8 marks) *answer compulsory*

1.1.1 What do you understand by the terms "utilization" and "availability" with reference to large open pit mining equipment.

A mine loses 2 hours in every 24 hours because of unavoidable shift change, lunch and coffee breaks. It has therefore decided that utilization and availability values should be based on a 22 hour daily maximum.

1.1.2 If a piece of equipment works for 2 periods of 6 hours in a day, what is the availability and utilization.

1.1.3 If a piece of equipment is under repair for 4 hours and idle for the remainder of the day, what is the availability and utilization.

1.1.4 If a piece of equipment works for 20 hours and is under repair for 2 hours, what is the availability and utilization.

(2 marks each for 1.1.1 through 2.2.4)

Question 1.2 (7 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 1.2.1, 2 marks each 1.2.2 through 1.2.4)

Question 1.3 (6 marks) ***answer compulsory***

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

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

1.3.3 What steps can be taken to minimise water hammer.

(2 marks each)

Question 1.4 (7 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

1.4.4 Chop Down

(2 marks each, 1 mark 1.4.4, total 7 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.

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*

In 1906, Italian economist Vilfredo Pareto created a mathematical formula to describe the unequal distribution of wealth in his country, observing that twenty percent of the people owned eighty percent of the wealth. Pareto's Principle, or Pareto's Law as it is sometimes called, can be a very effective tool to help you manage effectively.

1.6.1 "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). (2 marks)

1.6.2 What conclusions can you draw as a mining engineer with responsibility for reducing shovel loading costs from using Pareto's Law.

1.6.3 Describe what features you would look for in a computerized warehousing-parts-accounting system to ensure that personnel can find the part they want given the type of part or assembly containing it, and quickly locate the part in the warehouse.

(2 marks each, total 6 marks)

Question 2 (20 marks) **One of Three 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 (Figure 2.1.1) while the mill is under construction. During these early years (-1 and 0) no ore is mined.

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 end of year one, 7 million tonnes of ore has been 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.

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.

In the Figures 2.1.1 and 2.2.1 the simple "North West Corner" scheduling method is used as a guide to mine planning and scheduling.

Question 2 Figure 2.1.1

Ore Schedule 7 million/yr starting year 1 to completion (1, 2 ... 8)

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								copy			
1970	5	2			5				2				
1955	9	4			2	7			2	2			
1940	6	6					6			5	1		
1925	3	5					1	2			5		
1910	1	4						1			1	3	
1895	0	3										3	
1880	0	2										1	1
Totals	24	26			7	7	7	3	4	7	7	7	1

2.1.1 What are the timelines which must be met to produce ore in period 1, and what factors are in the plan such that the schedules can or cannot be met by the mine and construction personnel. (2.1.1 through 2.1.5, 2 marks each)

Referring to Figure 2.1.1 ;

2.1.2. Describe how ore on the elevations 1970 and 1955 and Phase (1) ore mining in years 1 and 2 produce the values shown in the table.

2.1.3. Some ore may be mined earlier or later. Describe the periods where the potential for ore stockpiling and stockpile mining might be expected.

2.1.4 Given that the deposit is a copper porphyry and/or epithermal gold, draw a sketch of the 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 variation.

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

Question 2 Figure 2.2.1

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 ... 7)

	Phase One	Phase Two	Phase 1 Year					Phase 2 Year				
Elev-ation	Waste	Waste	-1	0	1	2	3	3	4	5	6	7
2000	5	2	5					2				
1985	8	4	1	7				4				
1970	11	5		5	6			5				
1955	9	11			6	3			11			
1940	7	8				7			1	7		
1925	3	7				2	1			5	2	
1910	0	3									3	
1895	0	1									1	
1880	0	0										
Totals	43	41	6	12	12	12	1	11	12	12	6	

2.2.1 What elevations in Phase 1 and 2 are scheduled for waste removal when deep ore is being mined. What are the safety considerations and will these impact on the ore tonnage mined deep in Phase 2. How can safety be improved when waste blasting throws rock into deeper ore mining elevations. (2.2.1 through 2.2.5, 2 marks each)

2.2.2 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.

2.2.3. Will the purchase/rental of used trucks help any problems envisaged in 2.2.2 above.

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

2.2.5 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.

There is a 2 mark bonus for a good answer to 2.3.1

2.3.1 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.

Question 3 (20 marks) **One of Three Optional Questions to be Selected**

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 3.1. 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.

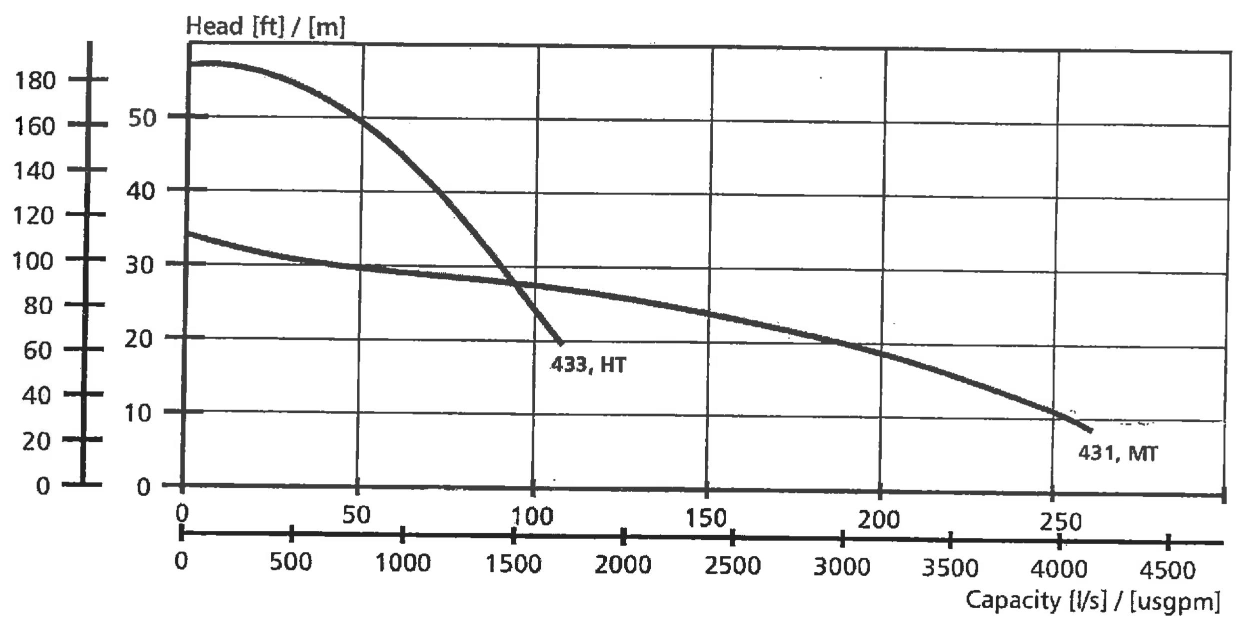

Figure 3.1 Pump characteristics (head and flow rate) for high head (433, HT) and high volume (431, MT) versions.

Question 3.1 (11 marks)

3.1.1 From the graph, Fig. 3.1, will the high volume pump configuration (431, MT) be used for both pumps.

3.1.2 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).

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

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

3.1.5 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 marks each for 3.1.1 to 3.1.4, 3.1.5, 3 marks, total 11 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 8^0C 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 3.2 (9 marks)

An alternative to the pumping system described in 3.1 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;

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

3.2.2 How the pumps might be laid out in plan and the pump depth determined. (2 marks)

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

3.2.4 The operational advantages quantified from a pit operations perspective. (2 marks)

Question 4 (20 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".

(2 marks)

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)

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;

(3 marks)

4.4.1 Dragline operating radius 70m

4.4.2 Dragline tub diameter 20m

4.4.3 Stacking height (maximum) 12m

4.4.4 Cut depth 25m

4.4.5 Positioning factor 75%

Question 4.4 continued

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

<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 4.5.1 to 4.5.6, 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. (3 marks)

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. (2 marks)

Question 5 (20 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 total costs broadly defined for ore and waste.

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".

(2 marks each, total 10 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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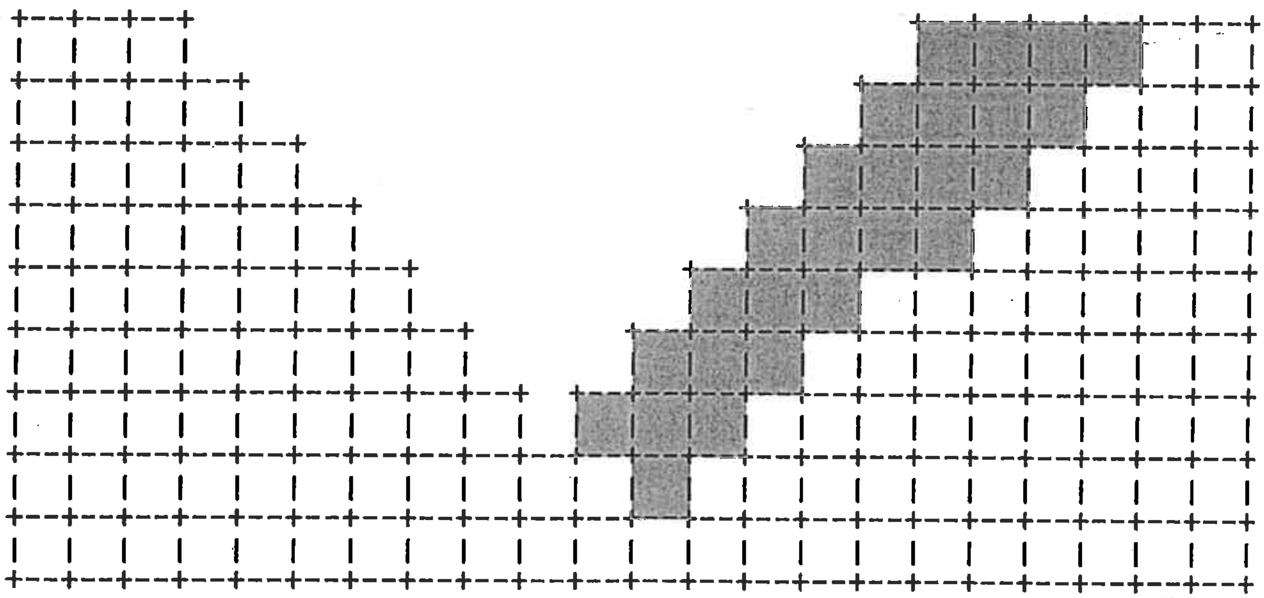

Figure 5.2. Top part shows the cash flow for each block on a single two dimensional section for a typical pit expansion. The middle and lower parts of the Figure are spaces to calculate the Lerchs-Grossman "optimal" pit expansion. Blocks are 15x15m and wall slope 45 degrees.

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

-1	-1	-1														-1	-1	-1	-2	-2	-2
-2	-2	-2	-2												-2	0	-1	-2	-2	-2	-2
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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 (20 marks) **One of Three Optional Questions**

6.1 For each of the following three mine cost centers, list the four major cost items and their percentage of the total of that particular cost sector.

6.1.1 Drilling (for blast-holes; excludes explosives etc.)

6.1.2 Truck (Haulage)

6.1.3 Shovel (Loading)

(2 mark each, total 6 marks)

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

6.2 What do you understand by the terms "cost index" and "inflation index".

Describe two inflation indexes which might be used for estimating "todays" costs in the context of the capital and operating costs of future open pit mines. What are the problems when such indexes are applied to all mine cost sectors.

Discuss how inflation indexes could be used to improve cost index estimates of future operations based on your answers to questions 6.1.1, 6.1.2 and 6.1.3. and Blasting)

(total, 4 marks)

6.3 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 for a potential mining project by the various cost centres. The Question uses US $ throughout and the answers are expected to be in US $ where CDN/US $ = 1.0, i.e. parity. (2 marks each 6.3.1 through 6.3.3, total 6 marks)

(There is no need to convert CDN/US $, US $ are assumed in 6.3 and 6.4 and US/CDN $ = 1.0 i.e. Parity)

6.3.1 For drills the cost index 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, 400 and 0.67 respectively.

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

6.3.2 For shovels the cost index relationship is

$P = a X^b$ where P is the capital cost of a shovel, X is the shovel bucket capacity in cubic yards and a and b are constants, 540,000 and the power 0.75 respectively.

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

6.3.3 For trucks the cost index 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.

6.4 An inflation index estimates that a 1997 dollar is now (2015) worth $2.00 dollars

What are the costs of buying the drill, shovel and truck today. Are the values realistic and what errors could be expected.

(4 marks)

Do not forget to hand in your Figure 5.2 with your name/number if you attempted Question 5

End of the Exam