Minerals Council South Africa

MEMORANDUM

SUBJECT: CHAMBER OF MINES OF SOUTH AFRICA – CERTIFICATE IN MINE ENVIRONMENTAL CONTROL PAPER 3	**EXAMINER:** M. Coelho **MODERATOR:** A. Oosthuizen
SUBJECT CODE: COMMEC3– ECONOMICS, PLANNING AND TECHNICAL LITERATURE	**TOTAL MARKS:** 100
EXAMINATION DATE: NOVEMBER 2020	**PASS MARK:** 60%
TIME: 14:30 – 17:30	

NUMBER OF PAGES:15

SPECIAL REQUIREMENTS:
THIS IS NOT AN **OPEN BOOK EXAMINATION**. **FORMULA HANDOUT IS AVAILABLE**
INSTRUCTIONS TO CANDIDATES:

1. Answer all questions **legibly** in English.
2. Write your **ID 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.

3. Show all calculations **and check calculations on which the answers are based.**
4. Hand-held electronic calculators may be used for calculations.
5. Write **legibly** in ink on the **right-hand page** only – **left hand pages will not be marked**.
6. Illustrate your answers by means of sketches or diagrams wherever possible.
7. Answers must be given to an accuracy which is typical of practical conditions.
   NB Ensure that the correct unit of measure (SI unit) are recorded, as marks will be deducted from answers if the incorrect unit is used (even if the calculated value is correct).
8. In answering the questions, full advantage should be taken of your practical experience as well as data given and clearly state any assumptions you make.
9. Please note that you are not allowed to contact your examiner or moderator regarding this examination.
10. Cell phones are **NOT** allowed in the examination room.

QUESTION 1 **[30]**

1.1 Mention 4 control methods for centrifugal fans and with the help of simple sketches show what impact it will have on the power curves associated with each control method. Which of these methods are the most economic in terms of energy consumption and why? (10)

Answer 1.1

See drawings on page 171 EE. The most effective is the speed control as the efficiency of the fan is not affected and remains constant with the increase or decrease in fan speed.

8 CONTROL METHODS FOR CENTRIFUGAL FANS

There are basically four methods for controlling the output of a centrifugal fan:

1. System dampers.
2. Inlet damper control.
3. Inlet vane control.
4. Speed control.

8.1 System dampers

One or more dampers can be installed anywhere in the duct system. Such a damper simply acts as a throttling device. The fan pressure/volume and power curves remain unchanged.

In Figure 7.9(a) AB is the fan pressure curve. OC is the system resistance curve. With the damper in the full open position the fan will operate at point D where the system resistance line OC intersects the fan curve. It will consume power P_1 and deliver volume Q_1.

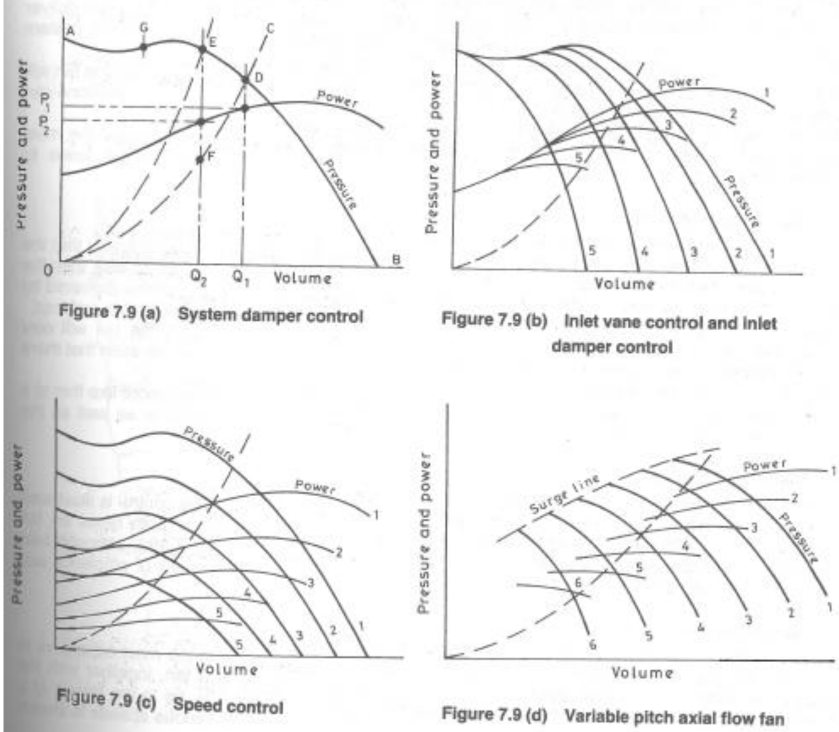


Figure 7.9 (a) System damper control

Figure 7.9 (b) Inlet vane control and inlet damper control

Figure 7.9 (c) Speed control

Figure 7.9 (d) Variable pitch axial flow fan

1.2 Prove that one advantage of siting a refrigeration plant on surface is that the input power to the compressor should be less than the input power to the compressor for an underground plant. Illustrate your answer by referring to a 10 000 kW plant and assume the following values. (10)

Evaporating temperature is constant at	5ºC
Condensing temperature surface plant	35ºC
Condensing temperature underground	45ºC
Overall cycle efficiency is constant at	60%

Answer 1.2

The main difference of siting a refrigeration plant on surface as apposed to underground, is that the underground plant will have a much higher condensing temperature. Assume the following:

Evaporating temperature is constant at	5ºC
Condensing temperature surface plant	35ºC
Condensing temperature underground	45ºC
Overall cycle efficiency is constant at	60%

Surface plant

Surface plant

$$\text{Carnot C.O.P} = \frac{T_1}{T_2 - T_1} = \frac{278}{30} = 9{,}27$$

$$\text{The overall compressor C.O.P} = \text{Carnot C.O.P x cycle efficiency} = 9{,}27 \times 0{,}6 = 5{,}56$$

$$\text{Compressor motor input power} = \frac{\text{evaporator duty}}{\text{overall compressor C.O.P}} = \frac{10\,000}{5{,}56} = 1\,798 \text{ kW}$$

$$\text{Carnot C.O.P} = \frac{T_1}{T_2 - T_1} = \frac{278}{40} = 6{,}95$$

Underground plant

$$\text{The overall compressor C.O.P} = \text{Carnot C.O.P x cycle efficiency}$$
$$= 6{,}95 \times 0{,}6$$
$$= 4{,}17$$

$$\text{Compressor motor input power} = \frac{\text{evaporator duty}}{\text{overall compressor C.O.P}}$$
$$= \frac{10\,000}{4{,}17}$$
$$= 2\,398 \text{ kW}$$

These calculations show that an underground compressor requires a great deal more power than a surface compressor. Expressed as a percentage as follows:

$$\text{\% difference} = \frac{2\,398 - 1798}{1798} \times 100\%$$
$$= 33\% \text{ more power required}$$

1.3 What two main methods are available for determining the psychrometric properties of moist air. What are the advantages of each? (5)

Answer 1.3

Psychrometric data can easily and quickly be obtained from Barenbrug's psychrometric charts. What they may lack in accuracy, they more than make up in versatility, especially when dealing with more complex problems, for example, the calculation of psychrometric data in air mixtures etc. The order of accuracy of the results is sufficient for most mining applications.

Psychrometric may also be calculated from first principles using the basic psychrometric equations. These give more accurate answers but require, in the first place, a knowledge of the various formulae which are difficult to remember and secondly a basic understanding of mathematics.

QUESTION 2 **[20]**

Answer Q 2.1

2.1 Discuss the mechanism of a coal dust explosion and refer to the difference between a deflagration and an explosion. Also comment on the most common initiator that causes coal dust to explode and comment on preventative measures to prevent coal dust explosions. (10)

MECHANISM OF A COAL DUST EXPLOSION

Finely divided coal will explode if a supply of oxygen is present and there is a source of ignition. The flame from a flammable gas ignition heats the atmosphere, causing a sudden rise in pressure. This rapid rise causes great turbulence, mixing the gases and exploding, resulting in a flame spread of up to five times the volume of the original mixture. The fast flow of air disperses dust from the roof, sides and floor of the roadway and the primary dust cloud is ignited by the flame front. The further rapid rise in the pressure caused by the heat of the burning coal and the turbulent flow of gases, lead to a pressure wave, raising coal dust which is subsequently ignited by the flame front lagging behind. These factors perpetuate and the mine with its contents can be destroyed in a few seconds.

Explosions can be divided into deflagrations and detonations.

In a deflagration the shock waves travel at the speed of sound (about 330 m/s) in the cloud of unburnt dust which has not yet contributed to the explosion, while the flame front follows at a slower speed. There is, therefore, a time lag between the arrival of the shock wave at a given point and the arrival of the flame. This interval can be used to advantage in that the shock wave can be made to trigger flame quenching devices which will arrest the progress of the flame. Obviously as the distance from the epicentre of the explosion increases, so does the interval between the arrival of the shock wave and the flame front. This, in part, determines the location of flame quenching devices, particularly those that have only a short duration efficacy.

The violence of an explosion depends on the maximum pressure generated and the maximum rate of pressure build-up. Dust explosions have produced maximum pressures of 1 MPa and pressure build-up rates of 100 MPa/s.

In a detonation the shock wave and the flame front travel together at approximately 1000 m/s in the gases produced by the explosion and will thus continue until all the available dust has been consumed. Most industrial dust explosions are deflagrations with flame speeds of up to 100 m/s, because the initial energy of the igniting source is small and the scale of operation is also relatively small. In coal mines, however, the igniting source is usually an explosion in itself i.e. a flammable gas explosion, and flammable gas may continue to augment the effects of the dust. In a fully developed coal mine dust explosion, the speed of sound may be exceeded. Work by Landman, amongst others, has shown that, in the presence of only small concentrations of flammable gas, the minimum ignition energy, the minimum explosible concentration and the energy generated by a coal dust explosion all change.

Unlike dust explosions in surface plants, dust explosions in mines are confined in that the surfaces of the airways restrain the explosion and force it to travel along the airways. The pressure build-up is, therefore, compounded in all directions.

Rapid cooling of the heated gases and mine surfaces causes a vacuum, which leads to a reversal of forces that can be as destructive as the primary pressure wave. These phenomena are not as marked in bord-and-pillar workings as in concentrated longwall workings.

The ignition sensitivity of a type of dust can be expressed in terms of the minimum temperature or the minimum energy required to create an ignition. In practice the two are not independent and both must be taken into account. Most dusts also have different ignition sensitivities depending on whether the dust is in a cloud or a layer. Additional factors which will affect the ignition sensitivity are size distribution, volatile content, concentration, turbulence and moisture content.

INITIATION OF COAL DUST EXPLOSIONS

The most common initiator of a coal dust explosion is an explosion of flammable gas. Rare cases have, however, been recorded of coal dust explosions initiating and propagating in the absence of flammable gas.

Volumes of flammable gas/air mixture of as little as 4m³ have been known to initiate a coal dust explosion.

The best defence against flammable gas explosions is to dilute the flammable gas as soon as it is released, to below its lower explosive limit by ensuring sufficient ventilation. All places where flammable gas can accumulate must be adequately ventilated. Once mixed with air, flammable gas remains diluted.

In addressing this aspect in the Code of Practice reference should be made to the guidelines dealing with the ventilation Code of Practice and the prevention of explosions should form an integrated system.

2.2 Layering of methane is a common phenomenon that can occur in underground mines and the layering number can be determined by the following formula.

$$LN = \frac{U}{\sqrt[3]{g\ x\ \frac{1-SG}{SG}\ x\ \frac{R}{D}}}$$

Where U	=	air velocity in tunnel
R	=	methane emitted in m^3/s
D	=	width of the airway
SG	=	SG of gas emitted
LN	=	layering number

A layering number of less than 5 is an indication that methane accumulation is likely to take place on the hanging wall. If the methane concentration in a raise is 0,4% and the width and height is 1,5 m and 2,4 m respectively, determine the minimum air velocity and subsequent air quantity required, to ensure that methane layering will not take place in the raise. Perform a cross check to see whether your answer is correct. Assume g as 9,79 m/s2.

Question 2.2 – Answer

The area of the raise = width x height

= 1,5 x 2,4

= 3,6 m^2

The quantity of air in the raise is (A x U), but in our case the velocity U is unknown, say 'y' m/s.

This means that the quantity of the air Q_{air} in the raise can be expressed as (3,6 x U) The percentage methane in the raise is given as 0,4%

From the formula for the gas %, the following:

$$\%CH4 = \frac{QCH4}{QCH4\ x\ Qair}\ x\ 100$$

$$0.4 = \frac{QCH4}{QCH4\ x\ 69.1667}\ x\ 100 \qquad \text{(equation B)}$$

From equation A above a value for the layering number will be put into the equation and the value U as shown in equation B must be substituted in A.

$$LN = \frac{U}{\sqrt[3]{g\ x\ \frac{1-SG}{SG}\ x\ \frac{R}{D}}}$$

$$5 = \frac{69,166667\ q_{CH_4}}{\sqrt[3]{9,79\ x\ \frac{1-0,55}{0,55}\ x\ \frac{q_{CH_4}}{1,5}}}$$

solve for q_{CH_4} = 0,0449 say 0,045 m^3 /s

Substitute q_{CH4} = 0,045 in B

Ψ U = 3,1125 say 3,1 m/s

then Q = U x A

= 3,1 x 3,6

= 11,2 m^3/s

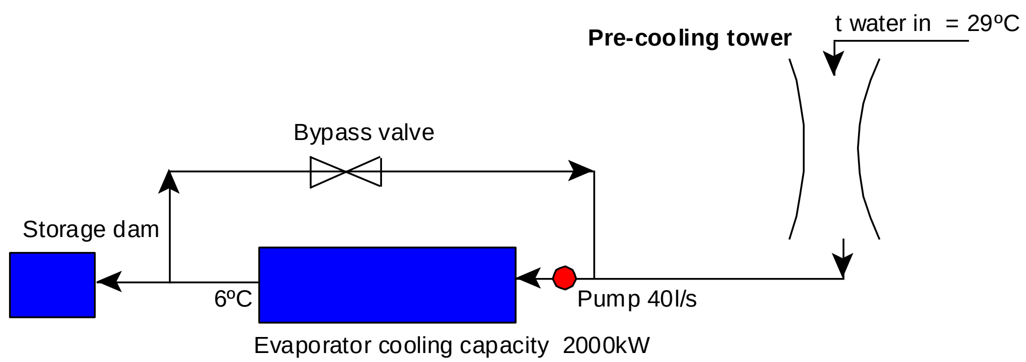
The velocity of the air in the raise to prevent layering will therefore be a minimum of 3,1m/s.

Check the answer:

$$\%CH4 = \frac{QCH4}{QCH4\ x\ Qair}\ x\ 100$$

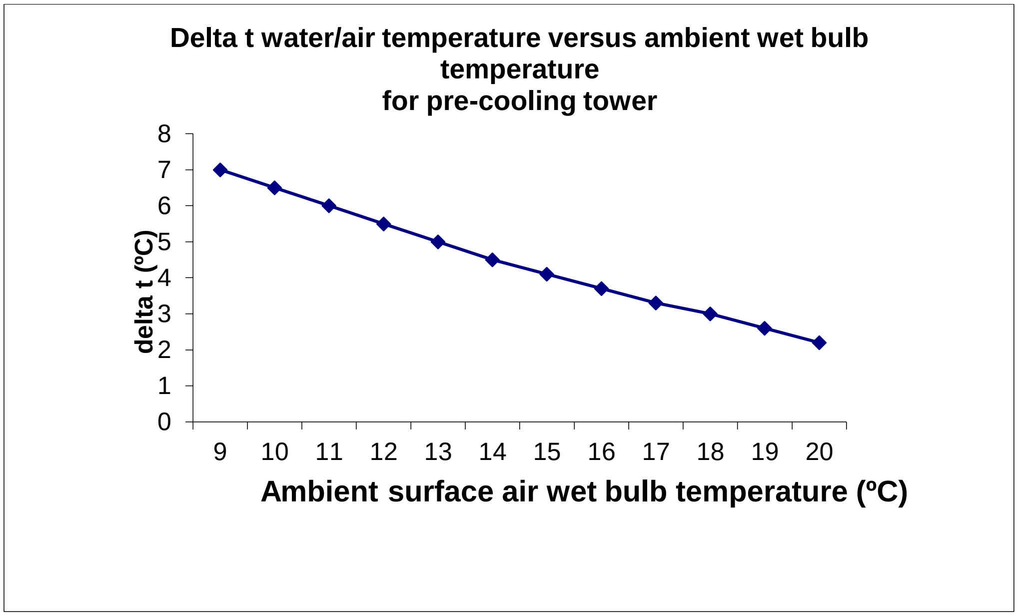
$$\%CH4 = \frac{0.045}{0.045\ x\ 11.2}\ x\ 100$$
$$\%CH4 = 0.4\%$$

Question 3 **[15]**



The sketch above shows the evaporator of a surface cooling plant, which receives its water from a pre-cooling tower and delivers it to a dam. The rated capacity of the plant is 2 000kW of refrigeration. The water leaving the plant is to be maintained constant at 6ºC. The water flow through the evaporator pump is constant at 40 ℓ/s.

When the temperature of the water from the cooling tower exceeds the design figure, the temperature of the water leaving the plant will rise above 6ºC. To prevent this, an automatically controlled by-pass allows some of the cold water to re-circulate and mix with the water from the pre-cooling tower as indicated on the sketch. The curve below shows the relationship between the "$\Delta t_{water/air}$ in tower" and the ambient surface wet-bulb temperature.



NOTE: *"$\Delta t_{water/air}$ in tower" is the difference between the ambient surface wet-bulb temperature and the temperature of the water leaving the pre-cooling tower (temperature of water approaching the evaporator, assume no losses).*

The mean ambient surface wet-bulb temperature for various months are as follows.

February	16.5ºC
March	14.6ºC
April	13.0ºC
May	9.5ºC
August	11.0ºC
September	13.0ºC

3.1 For the month of February the following must be determined:

(i) How much water will flow through the by pass?
(ii) How much water will flow to the dam?
(iii) If water enters the cooling tower at 29ºC, how much cooling is being supplied
- by the pre-cooling tower
- by the refrigeration plant

(iv) How much cooling would have been needed by the fridge plant if no precooling tower was available to still deliver 6ºC of chilled water?

3.2 During which months will the by-pass be closed or virtually closed

3.3 During which months will the refrigeration plant be producing less than its full load? (not working at full capacity)
Hint: Calculate for each month the temperature of the water leaving the pre-cooling tower.

(15)

Answer Q 3

3.1 Determine the temperature of the water entering the evaporator (assume 1 litre=1kg).

Q = m x c x Δt
Δt = q/ (m x c)
= 2000/ (40 x 4.187)
= 11.9 °C

The temperature of the water entering the plant can now be determined:

Water entering temperature = water leaving temperature + Δt

= 6,0 + 11,9

= 17,9ºC

(i) When using the curve supplied, it should be noted that the water ***cannot leave the pre-cooling tower at a lower temperature*** than the temperature of the air (ambient surface wet-bulb) being used in the cooling tower to cool the water.

This is the reason why the *"$\Delta t_{water/air}$ in tower"* temperature is added to the ambient surface wet-bulb temperature to obtain the temperature of the water coming from the cooling tower, flowing towards the evaporator.

From the graph, in February, the temperature of the water leaving the cooling tower will be as follows:

$16,5 + 3,5 = 20,0°C$

For the evaporator to remove 2 000 kW, the water flow rate at this temperature should be:

$$M = q/ (\Delta t \times c)$$
$$= 2000/ ((20-6) \times 4.187)$$
$$= 34.1 \ \ell/s$$

The by-pass will carry = 40,0 – 34,1 = 5,9 litres/second

(ii) 34,1 litres/second will flow to the storage dam

(iii) The pre-cooling tower supplies the following cooling:

$$q = m \times c \times \Delta t_{(twi - two)}$$
$$= 34,1 \times 4,187 \times (29 - 20)$$
$$= 1\ 285 \text{ kW}$$

and the refrigeration plant supplies 2 000 kW (this remains constant).

(iv) If there was no pre-cooling the cooling needed would have been 3 285 kW (2000+1285) to be able to deliver the same water temperature of 6ºC.

3.2 The temperature of the water leaving the pre-cooling tower and entering the evaporator in the following months:

(i) April
=13 + 4,95
=17,95ºC

(ii) May
=9,5 + 6,8
=16,30ºC

(iii) August
= 11,0 + 5,8
=16,90ºC

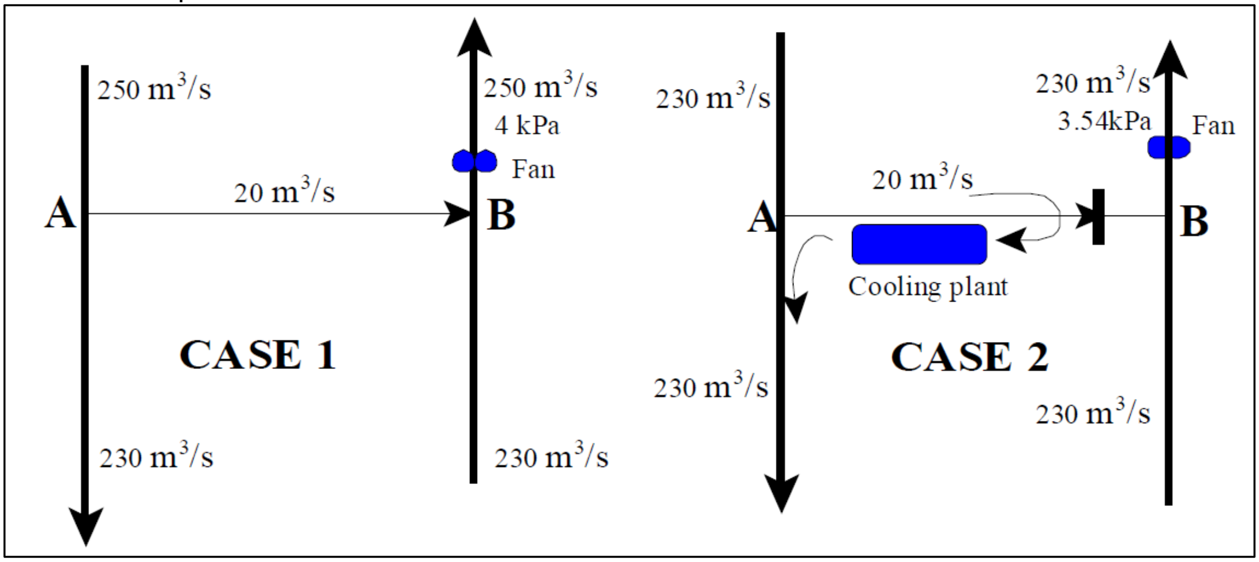
(iv) September
= 13,0 + 4,95
=17,95ºC

The temperature of the water entering the evaporator when the flow rate of the water is 40 litres/second (meaning the by-pass valve is closed) is 17,9ºC, this means that for the above mentioned months the by-pass valve will be closed or virtually closed.

3.3 The plant will be producing less than normal load from June to August (assuming that the temperature trend remains constant for June and July at less than 11ºC ambient surface wet-bulb temperature.

QUESTION 4 **[20]**

The sketches below show a section of a mine with its own fan and a pump chamber (between A and B). The air used to ventilate the pump chamber can be passed direct to return and through the fan (case 1) or it can be cooled by a refrigeration plant to remove the heat picked up in the pump chamber and allowed to rejoin the intake air (case 2). There are four pumps in the chamber each with a mechanical input of 940 kW, i.e. the pump motor output is 940 kW. The motor efficiency is 95% and 5% of the pump input enters the pump chamber ventilation as heat.
In case 1 the pressure difference between A and B was determined as 1 kPa.



4.1 How much heat is added to the air from the pumps and motors?
4.2 What will be the pressure difference between A and B in case 2?
4.3 Given the following information:

- The PV of R1 per annum is R 11,10 for the period and rate of interest concerned
- The capital cost of the cooling plant is R 3500 per kW of refrigeration output.
- Capital cost to speed up the fan R 200 000
- Electricity cost is R 1 200/kW/annum
- The maintenance cost for the refrigeration plant is R 380/kW/annum.
- The overall efficiency of the fan is 75%.
- To cater for case 1, the fan must be speeded up and
- Fan maintenance cost is estimated at R 380/kW/annum

Calculate which case within the scenario’s will be the more economical scheme?

Question4 Answer

4.1
The pump motor output power is 940 kW and the motor efficiency is 95% (given)

The input pump motor power $= \text{kW}/\ \text{efficiency}_{\text{pump motor}}$
$= 940/0.95$
$= 989.5\ \text{kW}$

The heat given off by the pump motors $= (989{,}5 - 940) \times 4$
$= 198\ \text{kW}$

The pump efficiency is given as 95%, which means that here also heat is given off to

the air in the chamber, therefore 5 % of 940 kW is given off as heat as well:
The total heat given off by the 4 pumps = (0,05 x 940) x 4
= 188 kW

The total heat given off by the pumps and motors = 188 + 198
= 386 kW

4.2
As the quantity of air flowing through the section remains constant at 230 m^3/s, the pressure drop from A to B will remain at 1 kPa.

4.3
Assuming that a 400 kW plant is available (the heat generated by the pumps and motors is 386 kW and the cooling required should be higher than the total amount of heat added). This value of 400 kW indicates a very high positional efficiency of 96,5% (386/400 x 100%).

(i) **Scheme 1**
The capital cost for speeding up the fan = R 200 000
The air power required = PQ
= 4 x 250
= 1 000 kW

Fan motor input power = fan overall/fan motor input efficiency
=1 000 / 0.75
= 1 333.3 kW

The PV of fan maintenance cost = fan motor kW x R/kW/annum x PV factor
= 1333,3 x R 380 x 11,10
= R 5 623 859

The PV of the fan running cost = R (1333,3 x 1 200 x 11,10)
= R 17 759 556

The total owning cost (TOC) for scheme 1 = R (17 759 556 + 200 000 + 5 623 589)
= **R 23 583 415**

(ii) Scheme 2
The capital cost for fridge plant = R (3 500 x 400)
= R 1 400 000

The PV of the plant power cost = R (400 x 1 200 x 11,10)
= R 5 328 000

The PV of fan fridge plant maintenance. cost = kW fridge plant x R/kW/annum x PV factor
= 400 x R 380 x 11,10
= R 1 687 200

The air power required = PQ
= 3,54 x 230
= 814,2 kW

Fan motor input power = Air power/ $\text{efficiency}_{\text{fan overall}}$

=814.2/0.75
=1 085,6 kW

The PV of the fan running cost = R (1 085,6 x 1 200 x 11,10)
= **R 14 460 192**

The PV of fan maintenance cost = fan motor kW x R/kW/annum x PV factor
= 1085,6 x R 380 x 11,10
= **R 4 579 061**

The total owning cost (TOC) for scheme 2, adding the above values:

TOC for scheme 2 = R(1 400 000 + 5 328 000 + 1 687 200 + 14 460 192 + 4 579 061)
= **R 27 454 453**

Scheme 1 is the cheaper option to the value of **R 3 871 038.**

QUESTION 5 **[20]**

Elaborate on the relevant section in the article related to mine environmental control and occupational hygiene, which appeared in the following publication

"Good practice guidance on occupational health risk assessment" by C.J. Badenhorst – Anglo American, October/December Volume 72 No 4

The author discusses good practice guidance on occupational health risk assessment. Discuss in detail the section that covers the purpose of the guideline? (20)

NB! Elaborate on this section of the journal only

Where possible, alignment has also been attempted with the ISO 45001 international standard "Occupational health and safety management systems - requirements with guidance for use". This was in draft at the time of review but it is not anticipated that major changes will take place in the final version.

Many organisations may be aligning their internal documentation with this standard in time, and some of the terminology in the new ISO standard has changed.

2. PURPOSE OF THE GUIDELINE

This guide is an information resource for conducting occupational health risk assessments (HRAs). It is intended for mining and metals managers and advisers who are responsible for ensuring the occupational health and well-being of employees and third party contractors. Though the guidance focuses on the occupational health risks to employees and contractors in a mining and metals operation, it is important to note that these risks can also affect the wider community living around that operation.

HRA is an integral part of the process of health risk management and often the two may be indistinguishable.

It is important to note that health risk management is not synonymous with hazard identification and risk assessment. The latter forms only part of the more comprehensive management approach.

The aim of occupational HRAs is to systematically and proactively identify health hazards, assess their potential risks to health, prioritise these, including the identification of material unwanted events (MUEs), and determine appropriate control measures (including the identification of critical controls to prevent MUEs) to protect the health and well-being of workers. The HRA process is a partnership between occupational health advisers, occupational hygiene advisers, managers and operational staff with each - depending on the circumstances - using their knowledge, experience and skills to support the HRA process. A key component to the success of health risk management is the commitment and visible leadership from senior and executive management.

HRAs within the mining and metals sector are especially complex because of the breadth and range of the mining life cycle, which includes:

- exploration
- design
- construction
- operation/extraction
- processing
- engineering services and maintenance
- closure
- rehabilitation/remediation.

This life cycle also encompasses the movement of products, equipment and personnel by road, rail, air and sea and the associated transportation networks and distribution facilities (e.g. ports and warehouses), as well as the manufacturing, recycling and disposal of goods made from the metals and minerals extracted from mines.

Workers are an important and valued part of the mining and metals sector and that places a moral obligation on the sector, alongside the legal obligations placed on it, to protect the health and well-being of its workers.[1] This moral obligation is increasingly being embedded within the sector through the adoption of the vision of zero harm (i.e. zero exposures above occupational exposure levels) and zero serious illness or fatal events from occupation health related exposures within a wider health and wellbeing at work policy.

This vision encompasses four key aspects:

- developing a workplace culture across an organisation that recognises that the prevention of long-term serious disease is just as important as the prevention of serious safety events
- making a consistent and sustained effort to ensure that there are no repeat occurrences of occupational diseases in any workplace setting of an organisation
- setting and implementing a simple, consistent and non-negotiable set of health and safety standards across an organisation that aim to prevent occupation-related illnesses
- for businesses to identify their 'material unwanted health events' and manage these in accordance with ICMM's *Health and safety critical control management: good practice guide* (2015).

In addition to the cost of occupational ill health in terms of preventable human suffering, which affects not just workers but their families and communities, work-related illness also directly impacts on the productivity and bottom line of companies in the mining and metals sector. This is usually through:

- higher presenteeism and absenteeism
- lower worker morale
- higher turnover rate
- loss of skilled and experienced workers
- loss of investment in training and development
- difficulties in recruiting new high-quality workers.

Alongside this, companies in the sector will also have to bear the costs of:

- healthcare for the affected workers
- compensation and/or damages to sick or disabled workers or to the families of workers that are killed
- higher insurance premiums
- legal advice

1 See ICMM Sustainable Development Principle 5 - www.icmm.com.

- regulatory fines
- damage to premises and equipment
- disputes and protracted negotiations with trade unions, public authorities and/or local residents
- loss of reputation
- loss of business
- loss of competitiveness
- in high-profile cases the complete or partial loss of the licence to operate.

TOTAL MARKS: [100]