Minerals Council South Africa

MEMORANDUM

SUBJECT: CHAMBER OF MINES OF SOUTH AFRICA – CERTIFICATE IN MINE ENVIRONMENTAL CONTROL PAPER 3 **SUBJECT CODE: COMMEC3– ECONOMICS, PLANNING AND TECHNICAL LITERATURE** **EXAMINATION DATE:** 9 MAY 2019 **TIME**: 14:30 – 17:30	**EXAMINER:** M. Coelho **MODERATOR:** B.A. Doyle **TOTAL MARKS:** 100 **PASS MARK:** 60%

NUMBER OF PAGES:11

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 **[17]**

1.1 Discuss the impact of air speeds and nominal carrying capacities for intake and return airways and upcast and downcast shafts. (6)

1.2 Provide typical air design speeds for:

1.2.1 Intake airways

1.2.2 Return airways

1.2.3 Vertical unequipped shafts

1.2.4 Vertical equipped shafts (4)

1.3 With the aid of a sketch show the generic theoretical phases of introducing mine cooling. (5)

1.4 Give two (2) requirements of a good technical report. (2)

ANSWER:

1.1 Blue book pg. 785

Excessive air speeds in airways will cause dust entrainment, general discomfort and high ventilation operating costs. However, intake airways normally represent a substantial investment and the potential ventilation carrying capacities of these excavations must be well used. The greater the cross-section, the higher the excavation cost but the lower the velocity for the same airflow. Fan power costs vary proportionally to the cube of the air velocity, and thus for shafts (and airways) that are dedicated to ventilation use, economic optimum velocities can be determined. This is an important issue containing overall investment and running costs. The economic optimum speeds depend on power costs, excavation costs and other factors, and can change significantly from project to project (and country to country). Typical optimum air speeds in dedicated ventilation shafts may vary from 16-22 m/s. There are many non-ventilation dedicated excavations in which the velocity is constrained by other issues such as access for personnel, conveyance stability and dust entrainment.

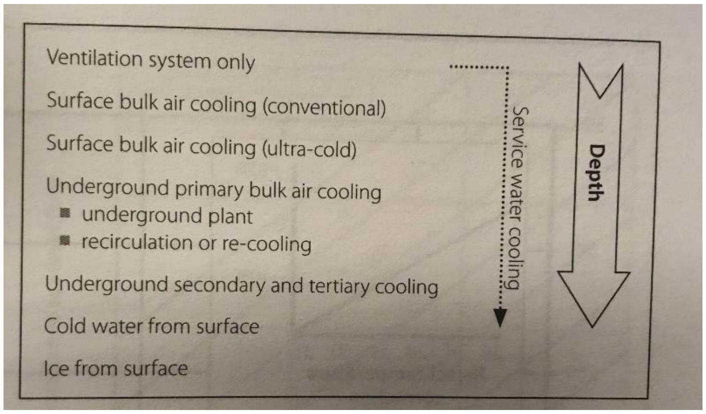
1.2.1 5 to 7 m/s

1.2.2 6 to 9 m/s

1.2.3 16 to 22 m/s

1.2.4 10 to 12 m/s

1.3 Blue book pg. 787 (10 x 0.5 marks = 5 marks)



1.4 any 2

The following points are essential in a good technical report

- Determine the true objective and write the report to achieve this aim and therefore avoid non-essential consideration.
- All facts should be accurate and logically analysed
- Have sound recommendations and conclusions
- Brevity is preferred and irrelevant information excluded
- Subject matter must be broken down into small steps and be presented logically and systematically
- Keep the readers interested

QUESTION 2 **[33]**

A desktop study for a new mining project in Africa to exploit the zinc/copper rich orebody using a sub - level caving method is to be undertaken. You are the consulting ventilation engineer and it is required to determine the ventilation and cooling requirements. The table below indicates the production ramp up and diesel fleet phase in for the period 2020 to 2025.

				2020	2021	2022	2023	2024	2025
Production									
Stope tonnes broken	ktpm			125	153	189	189	189	189
Development tonnes broken	ktpm			70	55	55	55	59	56
Total tonnes broken	ktpm			195	208	244	244	248	245
MRBD	m			430	470	540	580	620	660
Diesel Fleet	**Type**	**Rated kW**							
Sandvik (TH663)	Truck	565	No. of units	8	9	9	9	11	11
Sandvik (LH621)	LHD	352	No. of units	9	9	10	10	12	12
Jumbo Sandvik (DD421)	Drill Rig	110	No. of units	2	2	3	3	4	4

2.1 What is the diesel dilution factor that can be applied when ultra-low sulphur (10ppm) diesel and Tier IV engines are used for the diesel fleet? (1)

Please note for all the questions that follow use 0.06m^3/s per rated kW at point of application. Density is assumed to be 1.0 kg/m^3.

2.2 The orebody will be accessed from a 6.5m wide x 5.0m high **twin** ramp system from surface. During cleaning operations at each ramp development a truck and LHD will operate at the ramp development face. Calculate the fan intake volume requirements for total ramp development (Include 20% leakage). (3)

2.3 During stoping activities only the LHD will operate at the face as the truck will remain in the last point of through ventilation. Calculate the volume required for stoping activities if 3 levels will be active. (3)

2.4 Footwall and ore drive development on a typical level will consist of a LHD, drill rig and truck. The truck will not enter beyond the last point of through ventilation. Calculate the fan intake air volume requirements for level development? (Include 20% leakage). (2)

2.5 What is the total air volume required if there are 4 active development levels? (1)

2.6The following commitments need to be accounted for

- Workshop = 50m^3/s
- Crusher ventilation system = 40m^3/s

From the above calculate the total air volume required for the mine assuming a 10% primary system leakage. (3)

2.7 If the maximum ramp design velocity is 6m/s, determine if there is additional intake infrastructure required. (5)

2.8 Calculate the diesel fleet heat load for the year 2025. Use the following work rate factors: (4)

- 1.5 x rated kW for LHDs
- 1.2 x rated kW for trucks
- 0.25 x rated kW for drill rigs

2.9 Calculate the heat from rock use a factor of 10kW/ktpm for year 2025. (2)

2.10 Calculate the effect of auto compression. (3)

2.11 Calculate the total mine heat load if other sources of heat in year 2025 amounts to 3928 kW. (2)

2.12 Calculate the shortfall of cooling capacity of ambient air, for 2025, using a factor of 8.9 kJ/kg. (2)

2.13 Calculate the amount of air cooling required for 2025. (2)

Answers

Since a density of 1.0 kg/m^3 is assumed all answers are therefore in kg/s as well

2.1 0.05m^3/s per rated kW

2.2

Q = (565+352)x0.06 = 55m^3/s at the face

With 20% leakage

Q = 55 x 1.2 =66 m^3/s

Q for ramps = 2 x 66 = 132 m^3/s

2.3

Q = 352 x 0.06 = 21.1 m^3/s at the face

With 20% leakage

Q = 21.2 x 1.2 = 25.3 m^3/s

For 3 half levels Q= 25.3 x3 = 76 m^3/s

2.4

Q = (352 + 110) x 0.06 = 27.7 m^3/s

With 20% leakage

Q = 27.7 x 1.2 =33.3 m^3/s

2.5

Q= 33.3 x 4 = 133.1 m^3/s

2.6 Q_{total} = (132 + 76 + 133.1 + 50 +40) x 1.1

= 474.2 m^3/s

2.7

A = 6.5 x 5 = 32.5m^2

Total area for both ramps = 32.5 x 2 = 65 m^2

Air carrying capacity = 65 x 6 = 390 m^3/s

Short fall of 474- 390 = 84 m^3/s

Therefore, additional infrastructure or larger cross section ramps will be required

2.8

kW from trucks = 11 x 565 x 1.2 = 7458 kW

kW from LHDs = 12 x 352 x 1.5 = 6 336 kW

kW from drill rigs = 4 x 110 x 0.25 = 110 kW

Total heat from diesel equipment = 13 904 kW

2.9

kW from rock = 245 x 10 = 2450 kW

2.10

kW from auto compression = 474 x 9.79 x 660/1000

= 3062.7 = 3063 kW

2.11

kW total = 13 904 + 2450 +3063 + 3928 = 23 345 kW

2.12

Cooling capacity from downcast air = 474 x 8.9

= 4219 kW

2.13

Cooling required = 23345 – 4219

= 19126 kW

Question 3 **[15]**

The mean dust count on a mine is 185 particles per ml and the annual dust levy for this mine is R 3 660 000, which remains constant for the 10 year time period. The dust levy varies directly as the (mean dust count)^2. Three alternative schemes are proposed below, which is the most economical scheme:

Scheme 1

To accept the situation as it stands.

Scheme 2

To spend R 2 000 000 now and R 500 000 per year on better dust control to reduce the mean dust count to 160 particles per ml.

Scheme 3

To spend R 1 000 000 per year on better dust control to reduce the mean dust count to 165 particles per ml.

The life of mine is 10 years and the present value of R 1 per year over 10 years at 12% interest is R 5.65. Which is the most economical scheme?

Answer

The dust levy is directly proportional to the (mean dust count)^2. This implies the following:

$$\frac{DL_2}{DL_1} = (\frac{MDC_2}{MDC_1})^2$$

$$DL_2 = DL_1\, x (\frac{MDC_2}{MDC_1})^2$$

Where:

DL = dust levy

MDC = mean dust count

Scheme 1 (accept the dust count as 185 ppml)

The dust levy is R 3 660 000 per year.

The PV of the total cost over 10 years =

R 3 660 000 x 5,65

= R 20 679 000

= say R 20,7 million

The total owning cost for Scheme 1 **= R 20,7 million**

Scheme 2 (reduce the dust count to 160 ppml)

The new annual dust levy can be calculated as follows:

$$DL_2 = DL_1\ x(\frac{MDC_2}{MDC_1})^2$$

$$DL_2 = 3\ 660\ 000\ x(\frac{160}{185})^2$$

$$DL_2 = R\ 2\ 737\ 648$$

The PV of the levy cost over 10 years =

R 2 737 648 x 5,65

= R 15 467 711

= say R 15,5 million

The PV of the dust control measures = R (500 000 x 5,65) over 10 years =R 2 825 000

The Capital cost, dust control measures = R 1 000 000 = R 1 million

The total owning cost for Scheme 2 = R (15,5 + 2,8 + 1)

= R 19,3 million

Scheme 3 (reduce the dust count to 165 ppml)

The new annual dust levy can be calculated as follows:

$$DL_2 = DL_1\ x(\frac{MDC_2}{MDC_1})^2$$

$$DL_2 = 3\ 660\ 000\ x(\frac{165}{185})^2$$

$$DL_2 = R\ 2\ 911\ 424$$

The PV of the levy cost over 10 years = R 2 911 424 x 5,65

= R 16 449 548

= say R 16,4 million

The PV of the dust control measures = R (1 000 000 x 5,65) over 10 years

= R 5 650 000

= say R 5,7 million

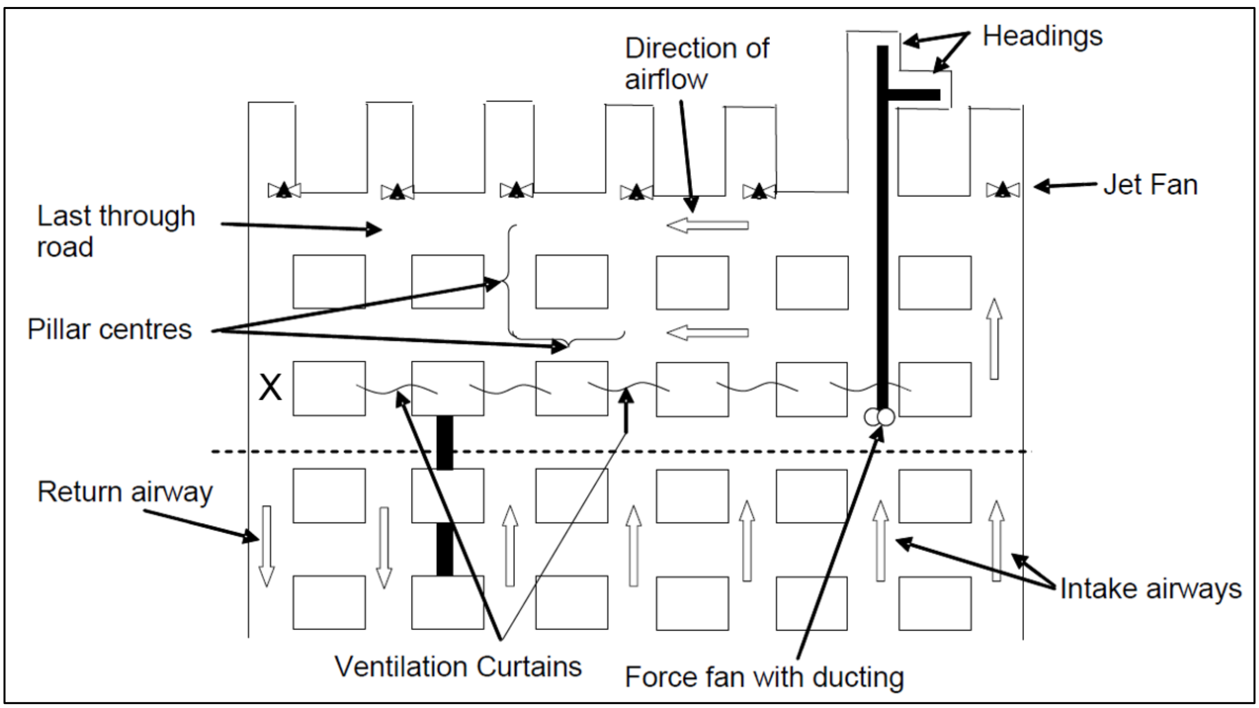
The total owning cost for Scheme 3 = R (16,4 + 5,7)

= R 22,1 million

From the above Scheme 2 seems to be the most economical.

QUESTION 4 **[20]**

Thick dolerite intrusions have necessitated the establishing of an independent satellite ventilation shaft system which will serve 3 mechanical mine bord and pillar sections. The sketch below shows a typical section layout:



The following information is given:

Planned production rate	60 000 tons/month/section
Shifts per month	72
Bord dimensions	6.5m W x 3m H
Pillar centres	30m
Number of roadways per section	7
Number of headings per roadway during first split development	2
Minimum air quantity required in headings	$0.15m^3/s/m^2$ of heading
Total leakage and air commitments	$90m^3/s$
Minimum design last through road velocity	1.0 m/s
Number of last through roads open	2

The method of ventilation employs auxiliary fans and ducting to be used during the first split development (apply a factor of 1.8 per heading to cater for duct leakages and to prevent recirculation at fans).

4.1 Calculate the air quantity to be supplied for:

4.1.1 Heading ventilation for the first split development

4.1.2 Assuming that ventilation controls are airtight at the face state the quantity of air at x

4.1.3 Total air quantity per section assuming 80% section air utilisation (12)

4.2 Determine the total air quantity required for this ventilation district and calculate the respective circular shaft sizes using the following optimum air velocities

- Downcast velocity 8m/s
- Upcast velocity 12m/s (8)

Answer:

4.1.1

Heading ventilation for first split development

Area of 1 heading = 6.5 x 3.0 = 19.52 m^2

Air volume / heading = .15 x 19.22 = 2.92 m^3/s

Apply factor of 1.8 = 2.92 x 1.8 = 5.26 m^3/s per heading

Air volume / roadway (2 headings) = 5.26 x 2 = $\underline{10.53\ m^3/s}$

4.1.2

Last through road area = 6.5 x 3.0 = 19.52 m^2

Air quantity to attain 1.0 m/s in both last through roads = 1.0 x 19.52 x 2 = 39.0m^3/s

4.1.3

Total air quantity per section intake = 39.0 + 10.53 = 49.53m^3/s (1)

Air volume required for one section (80% section air utilisation) = 61.91 m^3/s (1)

Air volume required for 3 sections = 61.91 x 3 = 185.74 m^3/s (1)

Leakage and commitments = 90 m^3/s

Total air volume to be circulated = 164.5 + 90 = 275.74 m^3/s (1)

Area required downcast shaft = (275.74/8) = 34.47 m^2

Diameter for equivalent area = 6.6 m

Are required upcast shaft = 275.74/12 = 22.98 m^2

Diameter for equivalent area = 5.4 m

QUESTION 5 **[15]**

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

"Block cave mine ventilation optimisation techniques" by M. Hooman, W. Marx and F.H von Glehn – BBE Consulting, January/March Volume 71 No 1

The authors discuss ventilation optimisation techniques during the capital development phase. There are several aspects that need to be included in ventilation planning of block cave mine; elaborate on their key findings in the article? (15)

NB! Elaborate on this section of the journal only

4. VENTILATION OPTIMISATION TECHNIQUES DURING CAPITAL DEVELOPMENT PHASE

During ventilation engineering planning, it is essential that the mine layout and plan are correctly understood for both the capital development and steady state production phases. Depending on the mining schedule and design, ventilation engineering challenges include high temperatures and airflow profiles that typically have peaks during capital development when the apex and/or undercut, production, ventilation and ore transport levels require many development ends to be ventilated simultaneously. There are a number of aspects that need to be included in the planning and these are discussed below.

4.1. Develop critical path to achieve through-ventilation

The critical path is the shortest path that can be mined to ensure that through-ventilation is achieved on the extraction level. Through-ventilation will enable the mine to achieve multi-end development and even multi-blast or fixed-time multi-blasting opportunities.

During this phase, the mine planning team, development schedule team and ventilation planning team need to be perfectly aligned to ensure that each department's key milestones are met. It is only when through-ventilation is established that adequate development can commence.

4.2. Capital development ventilation-on-demand

Ventilation-on-demand is an engineering control system that stops or reduces air supply to an inactive end and redirects this air to another active mining location. This reduces the overall air flow that needs to be delivered from surface. Ventilation-on-demand during the development phase can only be achieved when the mining and ventilation operations teams plan mining activities together.

In practice, this is typically achieved using force-ventilation tubing in development ends that will be tied with ropes to control the air distribution.

Alternatively, the system can be automated with open/close ventilation valves by either activating a local on/off switch or with the use of Programmable Logic Controllers (PLCs) via a Supervisory Control and Data Acquisition (SCADA) system. In some cases automated valves are used when multiple crosscuts on the undercut and extraction levels are in development and production to ensure loading crosscuts are ventilated appropriately.

4.3. Capital development cooling

Development ends are typically hot environments. The heat load is mainly caused by fresh hot broken rock, diesel loaders and/or trucks, ground water and fan heat. The overall mine cooling system needs to be carefully planned to ensure that cooling is available during all critical phases of the life-of-mine.

During the capital development phase an opportunity exists to cool the ends using localised cooling coil cars served by small temporary refrigeration machines providing chilled water. Spot cooling is an ideal solution for short-term use while larger permanent refrigeration machines with long lead-times are procured. However, spot cooling systems are inefficient and difficult to maintain over prolonged periods, and if the temporary refrigeration machines are located underground, they require access to return airways to enable heat from the refrigeration machine to be rejected.

Figure 5. Bank of cooling coils

4.4. Thermal design criteria

Thermal design criteria are specified to ensure acceptable working conditions and efficient operation during the life-of-mine. Capital development is the worst period in a block cave mine's life in terms of pressures on the thermal environment. During this period extensive development and production ramp-up take place but the required infrastructure (such as refrigeration equipment and main fans) has not yet been fully established.

To assist with the setting up of the mine's infrastructure and to enable steady state production to be established sooner, owners/operators can consider a temporary relaxation of underground reject temperature criteria. Application and risk management procedures have to be drafted and presented to the regulatory authorities for approval. The risks associated with any temporary relaxation need to be acknowledged and can include heat stress and heat stroke incidents. These risks need to be included in a heat stress management plan.

4.5. Heat stress management

HSM involves heat tolerance screening, work-rest cycles, nutrition and hydration regimes, medical surveillance, acclimatisation, etc. Heat stress management (HSM) is generally a feature in hot deep mines where auto-compression, high rock geothermal properties, high production rates, etc. are encountered.

HSM enables the mining operation to operate in high reject temperatures (typically above 27.5°Cwb) and needs to be carefully monitored, measured and recorded. As part of a HSM plan, shift cycles should also be assessed to ensure that safe and healthy practices are employed for both short and long-term operations (Kielblock 1992). Functional work assessment could be implemented in addition to the above to ensure overall physical work fitness.

4.6. Economic airway and vertical hole design

Capital development includes the development of vertical fresh air and return air passes between surface and the mining footprint. These vertical holes are then connected to the block cave by means of horizontal airways.

Economic velocities of the vertical holes and the horizontal airways need be determined to ensure that capital/development costs and operating cost provide a positive net present value.

TOTAL MARKS: [100]