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:** 10 OCTOBER 2019 **TIME**: 14:30 – 17:30	**EXAMINER:** M. Coelho **MODERATOR:** M. Beukes **TOTAL MARKS:** 100 **PASS MARK:** 60%

NUMBER OF PAGES: 12

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.

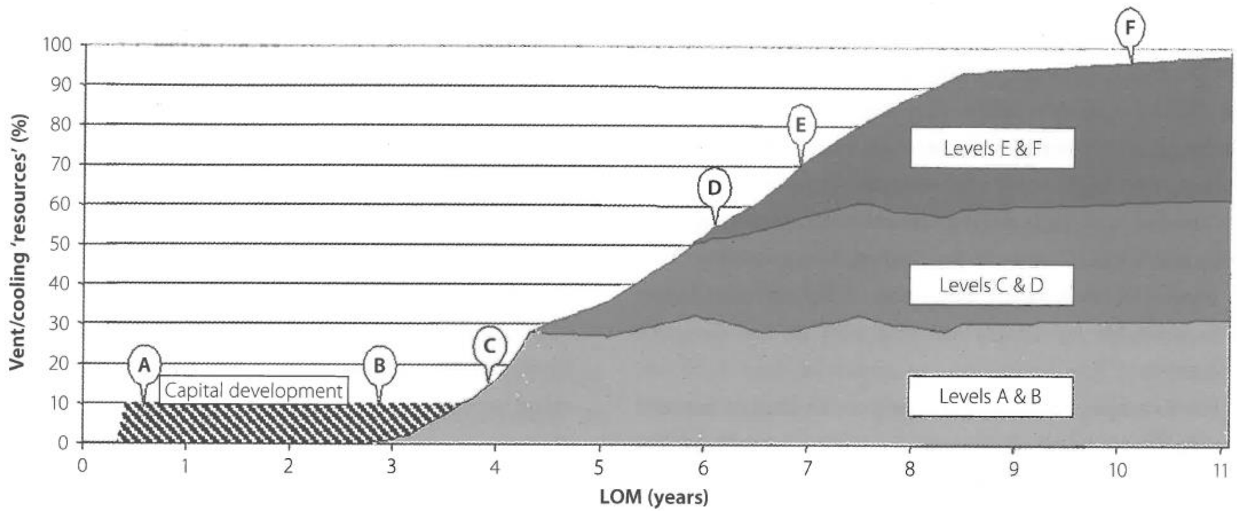
NB: Your name **must not** appear on any answer book or loose sheets.

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 **[20]**

1.1 With aid of a sketch show the environmental strategy for various zones? (5)
1.2 State 5 typical design aspects and criteria? (5)
1.3 State 5 characteristics of the formal feasibility phase? (5)
1.4 The figure below shows a typical example of ventilation/cooling phase-in profile. Explain what is meant by phase-in of capital? (5)



Answers
1.1

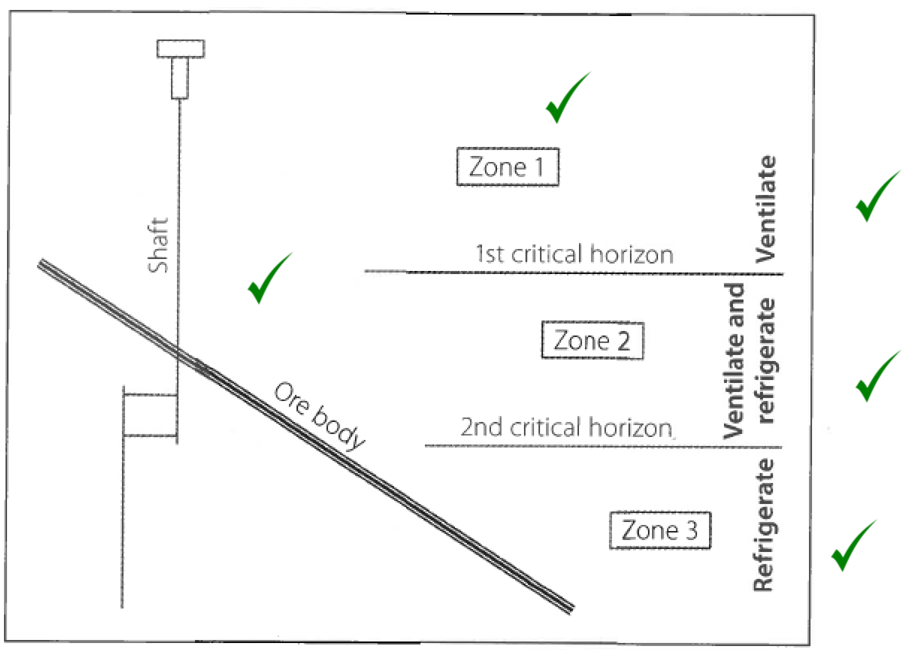


Figure 38.7 Environmental control strategy for various zones

1.2

process. The following list includes some of the typical design aspects and criteria.

- Existing mine characteristics and constraints (for brownfields projects)
- Development and project schedule
- Mining method statement and production programme (including clear statement on stoping methods, powering systems, support systems, life-of-mine profiles, etc.)
- Main and secondary shaft requirements and infrastructure[1]
- Inventory of all underground equipment and machinery and work cycles (electrical, diesel and other)
- Geothermal data (virgin rock temperatures, rock thermal properties)
- Mineralogy of ore body and surrounding rock (as related to dust make-up)
- Expectations of fissure water and strata gas emissions
- Underground thermal environmental criteria (temperature, cooling power) and airborne contaminant design criteria (dust, gases, diesel emissions, radiation)
- Service ventilation requirements (including clear statement on workshops, pump stations, crusher stations, conveyors, substations, etc.)
- Service water usage
- Power cost data and expected project rate of return
- Surface weather data

✓ Any 5

1.3

Formal feasibility phase: Characterised by the following (among other issues):

- High-level geological definition, definitive statement on production rates, mining methods, mine and shaft system layouts, etc.
- Definitive statement on ventilation (and cooling) tactics, primary ventilation flows, shaft sizes, secondary ventilation controls, heat loads, refrigeration capacities, equipment lists/schedules
- Definition of life-of-mine phase-in profiles of ventilation (and cooling) needs
- Definition of full project schedules for all disciplines

✓ Any 5

- Engineering design work, drawings, process flow diagrams, equipment and system specifications, etc.
- Definition of control budget estimate (CBE), life-of-mine operating costs and full life-cycle financial evaluations, etc.
- Comprehensive documentation for the purpose of both
  - executive decision on project implementation
  - specifications, guidance, rules and controls for project execution

1.4

3.2 Phase-in of Capital

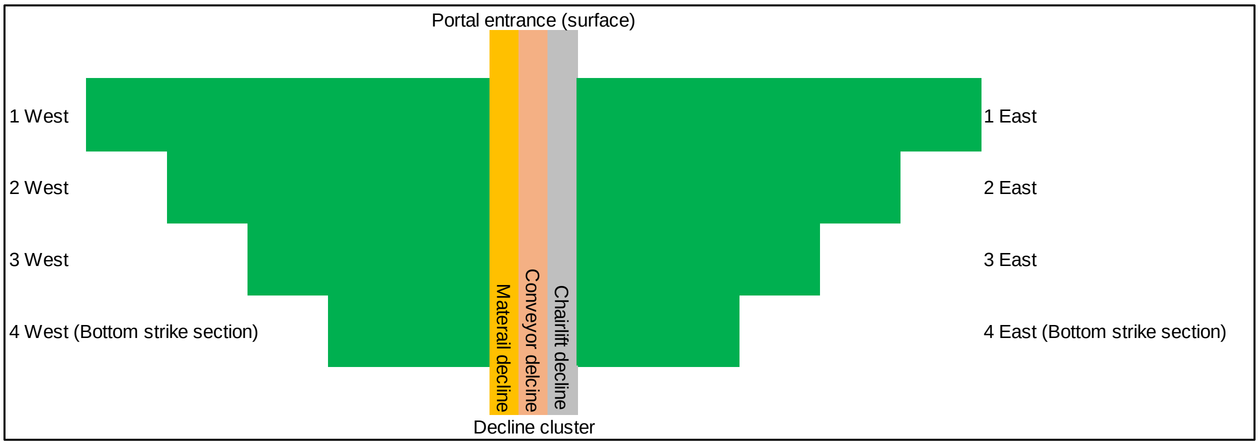
Although most project Capex is utilised in the mine construction stage, often the main ventilation/cooling equipment is installed later. However, the capital costs of the ventilation/cooling equipment can be so high that an optimised phase-in strategy becomes a very important consideration.

Detailed evaluation of critical scenarios and interpolation between these confers an understanding of the life-of-mine needs and helps to build up the phase-in picture. Figure 38.3 presents a hypothetical but typical phase-in profile of ventilation/cooling resources for a greenfields

project. It should be noted that the investment in the ventilation/cooling equipment is well spread over the life of the mine. Indeed, the final main fan modules can be delayed to year 6 and the final refrigeration systems to year 10. Understanding and predicting the ventilation/cooling Capex delay is very important in successful planning. Effective planning deserves the effort required to produce this type of profile for both Capex and Opex requirements.

QUESTION 2 **[25]**

You are the ventilation engineer of a platinum mine in the eastern limb for which the ventilation requirements are to be determined. The figure below shows the mining layout of a portion of the mine.



The table below shows the diesel fleet that will be active per strike section

Vehicle type	Engine Output (kW)	Number of
LHD	256	2
Support equipment (includes drill rigs , bolters, UV, etc)	40	N/A

2.1 What requirements must be met for a diesel dilution factor of 0.06m^3/s per rated kW at point of application to be used for ventilation planning? (2)

2.2 Calculate the air requirement per strike section? (assumes 20% leakage) (4)

2.3 Three (3) declines are to be developed from surface, the decline cluster will have 2 LHDs operational during a working shift. Calculate the air requirement for the decline cluster? (assumes 20% leakage) (3)

2.4 Calculate the air requirement for bottom entry strike section provided that the bottom entry requires 1.5 the air requirement of a strike section? (2)

2.5 An underground workshop 8m wide and 5m high requires a minimum velocity 0.5m/s. Calculate the air requirement for this commitment? (2)

2.6 Calculate the air requirements for the mine if there will 6 active strike sections together with the decline cluster? (assume 30% strike section leakage) (6)

2.7 The decline excavation dimension and maximum carrying velocities are shown in the table below. Calculate the maximum volume that the declines can accommodate? (4)

Excavation	Wide (m)	High (m)	Maximum velocity (m/s)
Conveyor belt decline	4.5	3.5	3
Machine decline	6	3.5	7
Chairlift decline	5	4	5

2.8 Is there a shortfall? (2)

Answers

2.1 Tier 3 or Tier 4 engines

50ppm or 10 ppm sulphur content in diesel

2.2 Q per strike = (2x256 +40) x 0.06 =33.12 m^3/s

Add 20% in section leakage

Qstrike= 33.12 / 0.8 = 41.40 m^3/s

2.3 Qdecline = 2 x 256 x 0.06 = 30.72 m^3/s

Add 20% section leakage

Qdecline = 30.72 / 0.8 = 38.4 m^3/s

2.4 Qbottom =Qstrike x 1.5 = 41.40 x 1.5 = 62.1 m^3/s

2.5 Qworkshop = 8 x 5 x 0.5 =20 m^3/s

2.6 Qtotal = 4 x Qstrike +2 x Qbottom + 1 x Qdecline + 1 x Qworkshop

= 4 x 41.4 + 2 x 62.1 + 1 x 38.4 + 1 x 20

= 348.2 m^3/s

Add 30% strike section leakage

Qtotal = 348.2 / 0.7

= 497.4 m^3/s

2.7 Q max conveyor = 4.5 x 3.5 x 3 = 47.25 m^3/s

Q max machine = 6 x 3.5 x 7 = 147 m^3/s

Q max chairlift = 5 x 4 x 5 = 100 m^3/s

Q max total = 294.25 m^3/s

2.8 Yes there is shortfall of 203.2 m^3/s (497.4-294.25). Additional intake infrastructure will be required to meet the ventilation requirements

QUESTION 3 **[20]**

3.1 Define total owning cost? (2)

3.2 If you were to invest the sum of R20000 in a bank at an interest rate of 11% compounded annually. What would the value be after 5 years? (3)

3.3 A ventilating duct system will have a useful life of 5 years. The prevailing interest rate is 12%. The capital cost of this system is R 37 750 000. At the end of its life this system will have a resale value of 20% of the capital cost. Determine through calculation the net present value cost of the installation? (5)

3.4 You are required to choose between 2 fans designed to operate at 300m^3/s at 4000 Pa. Fan A costs R 25 000 000 and has an efficiency of 83%. Fan B costs R 19 600 000 and has an efficiency of 73%.If the life of the project is to be 20 years and the average, cost of electricity is R 0.78c/kWh. Which fan is the most economical? Assume the fan will operate continuously for 24 hours per day and 365 days per annum. The value of R 1 per year over 20 years at 12% interest is 7.47. (10)

Answers

3.1 TOC is the total amount of money it costs to own a scheme at today's value ✓✓

3.2 A=P(1 + (i/100)^n

Thus A = 20000x(1 +11/100)^5 ✓

=20 000x(1.11)^5 ✓

=R 33 701.16 ✓

3.3 Capital cost to purchase ducting =R 37 750 000

Resale value in 5 years = R 37 750 000 x 0.2

= R 7 550 000 ✓

A=P(1 + (i/100)^n

Therefore

P = A/(1 + (i/100)^n)

= 7 550 000/((1 + (12/100))^5) ✓

= R 4 284 073 ✓

Actual capital cost installed cost = R 37 750 000 - R 4 284 073 ✓

= R 33 465 927 ✓

3.4 Electrical power cost R/kW/annum = 0.78x24x365

=R 6 833/kW/annum ✓

FAN A

Capital cost = R25 000 000

Air power =PQ/1000

=(4000x300)/1000

=1200 kW ✓

Power consumed =(1200 x 100)/83

= 1 446 kW

Annual power cost = 1446 x 6833

= R9 880 518

PV of power cost = R9 880 518 x 7.47

= R 73 807 470

FAN A TOC = R 25 000 000 + R 73 807 470

= R 98 807 470

FAN B

Capital cost = R19 600 000

Air power =PQ/1000

=(4000x300)/1000

=1200 kW

Power consumed =(1200 x 100)/73

= 1 644 kW

Annual power cost = 1 644 x 6833

= R11 233 452

PV of power cost = R11 233 452 x 7.47

= R 83 913 886

FAN B TOC = R 19 600 000 + R 83 913 886

= R 103 513 886

The TOC of FAN A < TOC FAN B

FAN A would be more economic to purchase

QUESTION 4 **[20]**

The ventilation requirements for a coal mine with sparse methane must be determined. The following aspects should be considered:

- *Mining Method*: Bord and Pillar
- *Road width*: 6.5 m
- *Seam Height*: 3.2 m

- *Pillars sizes*: 12 x 12 m
- *Air velocity – conveyor belt*: 2.5 m/s
- *Air velocity – intake road*: 3,5 m/s
- *Air velocity – return road*: 6 m/s
- *Air velocity – downcast shaft*: 8 m/s
- *Air velocity – up cast shaft*: 10 m/s
- *Standard Panel length*: 1,2 km
- *Assume air utilisation*: 80 %
- *Air commitments*: 150 m^3/s
- *Design velocity in last through road*:1,2 m/s
- *Production ton/continuos miner/month*: 60 000 t
- *Production /year*: 4 320 000 t

4.1 Calculate the total area per road (1)
4.2 Calculate the volume required for the conveyor road (1)
4.3 Calculate the volume required for the return road (1)
4.4 Calculate the volume required for the intake road (1)
4.5 Calculate the volume required in the last through road? (1)
4.6 Number of continuous miner sections required? (2)
4.7 Calculate the total volume required for the mine? (4)
4.8 The diameter of the downcast shaft? (2)
4.9 The diameter of the up cast shaft? (2)
4.10 Calculate the friction pressure loss in the downcast shaft? (2)
4.11 Calculate the pressure loss in the intake roadways per km? (3)

Answers

Answer - EE pg 971

4.1 Total area per road = 3.2 x6.5 = ✓ 20.8m^2

4.2 Airflow volume on conveyor road (QB) = 20.8 x 2.5 = 52 ✓ m^3/s

4.3 Airflow volume return road (QRAW) = 20.8 x 6 = 125 m^3/s ✓

4.4 Airflow volume intake road (QINTAKE) = 20.8 x 3,5 = 73 m^3/s ✓

4.5 Minimum volume air in last through road (QR) = 20.8 x 1,2 = 25 m^3/s ✓

4.6

Total tons/month = 4 320 000/12

= 360 000 tons/month ✓

Sections required = 360 000/60 000

= 6 ✓

4.7

Minimum airflow volume (QR) = 25 m^3/s ✓

Airflow volume for 6 sections = 6 x 25 = 150 m^3/s ✓

Airflow volume for 6 sections (including utilisation factor) = 150/0.8 = 187.5 m^3/s ✓

Total airflow volume required = (187.5 + 150)/0.8 = 421.88 m^3/s ✓

4.8

421.88/8 = 52.73m^2 ✓

Diameter = ≈ 8.2 m ✓

4.9

420.31/10 = 42.18m^2 ✓

Diameter = ≈ 7.33 m ✓

4.10

$$Friction\ pressure\ loss \quad = 2.5\ x \frac{wv^2}{2}$$ ✓

=2.5 x ((1 x8^2)/2) (Assume w =1)

= 80 Pa ✓

4.11

Rule of thumb EE pg 974 = velocity pressure loss of 1Pa for each pillar. ✓

Therefore, a velocity pressure loss of 1 Pa = 17 m or 58.82 Pa per km ✓

The pressure loss for intake roadways → 58.82 x $2.5^2/2$ = 183.81 Pa (Assume w=1) ✓

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

"In-Stope dust control at Beatrix Gold Mine" by J.J.L. Du Plessis and M.I. van der Bank – Associate Professor, UP and Environmental Engineering Manager, Beatrix Gold Mine October/December 2018 Volume 71 No 14

The authors discuss field trial results of In-Stope dust control using atomising spray. Elaborate on their key findings of the field trial results? (15)

NB! Elaborate on this section of the paper only

Answers:

6. FIELD TRIAL RESULTS

The personal sampling determining exposure measurements for the rock drill stope operator (RDO), winch operators and the in-stope crew are shown in Table 2 (van der Bank, 2013). The mass concentrations were calculated using equations 1 and 2 and averaged for the number of measurements.

Table 2. Average measured dust concentrations

Occupation	Before concentration (mg/m^3)	After concentration (mg/m^3)	Exposure reduction (%)
RDO	1.163	0.231	80.1
Winch operator	1.358	0.200	85.3
Stope team	1.450	0.317	78.1

From Table 2 it is clear that the average respirable dust concentration as determined from personal exposure monitoring results decreased significantly, with the lowest reduction being measured for the stope team workers at 78.1% and the highest reduction of 85.3% being measured at the winch operator.

In previous studies (See Figure 2 above) the winch operators were found to be more exposed than other in-stope workers, but these measurements showed that the general stope workers and RDOs were more exposed than the winch operators.

The time-weighted average (TWA) quartz corrected exposure measurements for the RDOs, winch operators and in-stope crew are shown in Table 3. The TWA concentrations were calculated using equation 3 and averaged for the number of measurements. The historical average quartz percentage for Beatrix is between 22 - 28% (personal communications, Van der Bank, 2015) but for the project average quartz percentage was between 11 - 18.8%. This was attributed to the fact that mining had just started in the shaft pillar area.

Table 3. Average measured TWA quartz dust concentration

Occupation	Before concentration (mg/m^3)	After concentration (mg/m^3)	Exposure reduction (%)
RDO	0.129	0.02	79.8
Winch operator	0.249	0.0366	85.3
Stope team	0.168	0.046	72.6

From Table 3 it can be seen that an average reduction of between 72.6 and 85.3% in the TWA quartz concentration was achieved during the trial, with the highest reduction being achieved for winch operators. The combined average reduction in the TWA quartz concentration across all occupations was calculated to be 79.2%. When comparing the different occupations, the in-stope team members are now the occupation with the highest average exposure. Although this is still well within the current OEL, it is close to the new proposed OEL of $0.05mg/m^3$.

A summary of additional field observations made during the study within the study area is as follows:

- Water covers the total face area from a fixed point.
- There is no need to enter into any unsafe area to water down.
- The water spray washes the blasted face, assisting in exposing misfires.
- Induces airflow with a measured air quantity of up to $2.5m^3/s$.
- There is an increased face velocity measured 15m from the water-blast of 3.3m/s.
- There is an increased face velocity measured 30m from the water-blast of 1.6m/s.
- The measured water flow rate was 0.75l/s or 45l/min.
- No additional watering down was done during the trials.

No compressed air measurements were taken during the study. It is the opinion of the authors that it is important to measure the compressed air use at different air pressures and the impact on water use with different water pressures available. This should be done independently as well as when both of the feed components will be entering the nozzle at the same time as the usage will be influenced by the entering pressure of each other. This will greatly assist in estimating the operational cost associated with the implementation of the intervention.

TOTAL MARKS: [100]