EVALUATION AND DESIGN OF A MINE VENTILATION SYSTEM FOR MINDOLA DEEP MINE :(5220 L - 6365 L ORE ZONE) ZAMBIA: DETERMINATION OF SIGNIFICANT CONTRIBUTORS AND FACTORS

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ABSTRACT

The paper reports research findings on the evaluation and design of the end of mine life ventilation system for the 5220L – 6365L ore zone of the Mopani Mindola Copper Mine in Kitwe, Zambia. The purpose of mine ventilation is to provide suitable environmental conditions in working places. The objectives of the research were: firstly, the evaluation of the current mine ventilation system for the 4440L – 5220L ore zone, taking into account the aged ventilation system infrastructure and, secondly, building of a ventilation baseline database for design of the end of mine life ventilation system taking into account the planned increase in production, and subsequent increase in depth of mining from 1586m (current depth) to about 1930m (5220L – 6365L ore zone). In evaluating the current ventilation system and designing the end of mine life ventilation system the methodology involved, firstly, collection of the mine and equipment physical details as well as primary and secondary data of the current ventilation system by means of ventilation surveys, analyses, and computations as well as processing of obtained data so as to use it in the design of the end of mine life ventilation system. Findings were: High wet-bulb temperatures (in excess of or close to 31.0 ^{o} C) were recorded in several mining areas below 4440L; Low air volumes of 1.5 – 4.5m^3/s and velocities of 0.6 – 1.5m/s (lower than legal/scientific baselines of 30.0m^3/s and 4.0m/s respectively) were evident in a number of mining areas between 4370L and 4440L; Dust concentrations of ventilation air in almost all mining areas were compliant with the established legal and or scientific baselines of 100 ppm; Obnoxious gas concentrations both in the general body of ventilation air and diesel unit exhaust emissions were well below Zambian legal/global scientific baselines of 1000 ppm CO_2 and 100ppm CO . The total quantity of air leakage was in excess of 224 m^3/s (35 % of total downcast air). In summary, the computed heat loads in the current mining zone (4440L – 5220L) recorded that diesel engine equipment with a total heat load of 1,792.5 kW (49% of the total heat generated) was the highest contributor, indicating it to be the *most important variable. Secondly, eelectrical equipment (fans), exposed rock (strata), metabolism, and others accounted for 1,861.5 kW (51% of the total heat generated). In conclusion, reduction and or replacement of diesel equipment electrical equipment would bring about significant reduction in mine heat load, and reduce or eliminate the need for refrigeration of the Mindola mine workings at depth.*

Keywords: *Ventilation; Environmental; Temperatures; Heat load; Dust; Air leakages; Diesel equipment; Electrical equipment.*

INTRODUCTION

Mindola Copper Mine, Zambia, whose location is shown in Figure 1, is part of Nkana Division of the Mopani Copper Mines (MCM) plc.

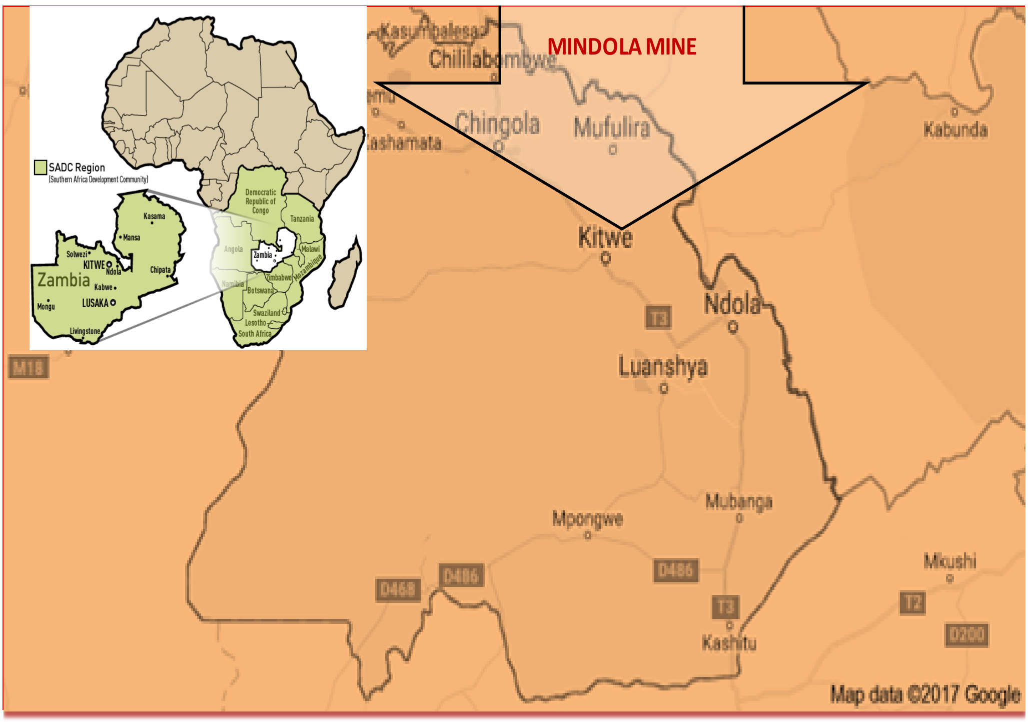


Figure 1: Location of Mopani Mindola Copper Mine, Kitwe, Zambia.

The general layout of the Vertical Crater Retreat mining method is shown in Figure 2. The current mine ventilation system has been in use for over 50 years of mine life. Current mining operations are confined to the 4440L – 5220L ore zone of the mine at a depth of approximately 1586 m, (Guney M. and Bell A.R. 2005).

The mining method currently in use is the mechanised up-dip Vertical Crater Retreat (VCR) with waste rock as post-fill.

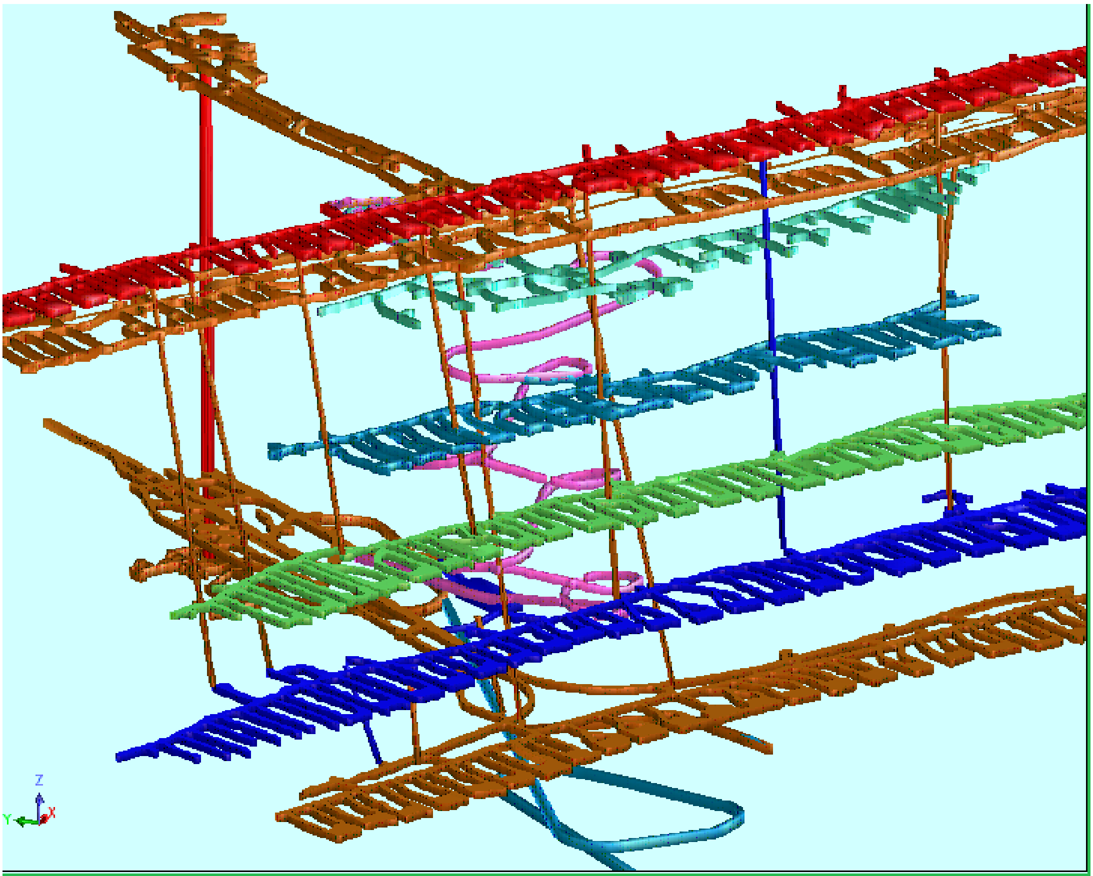
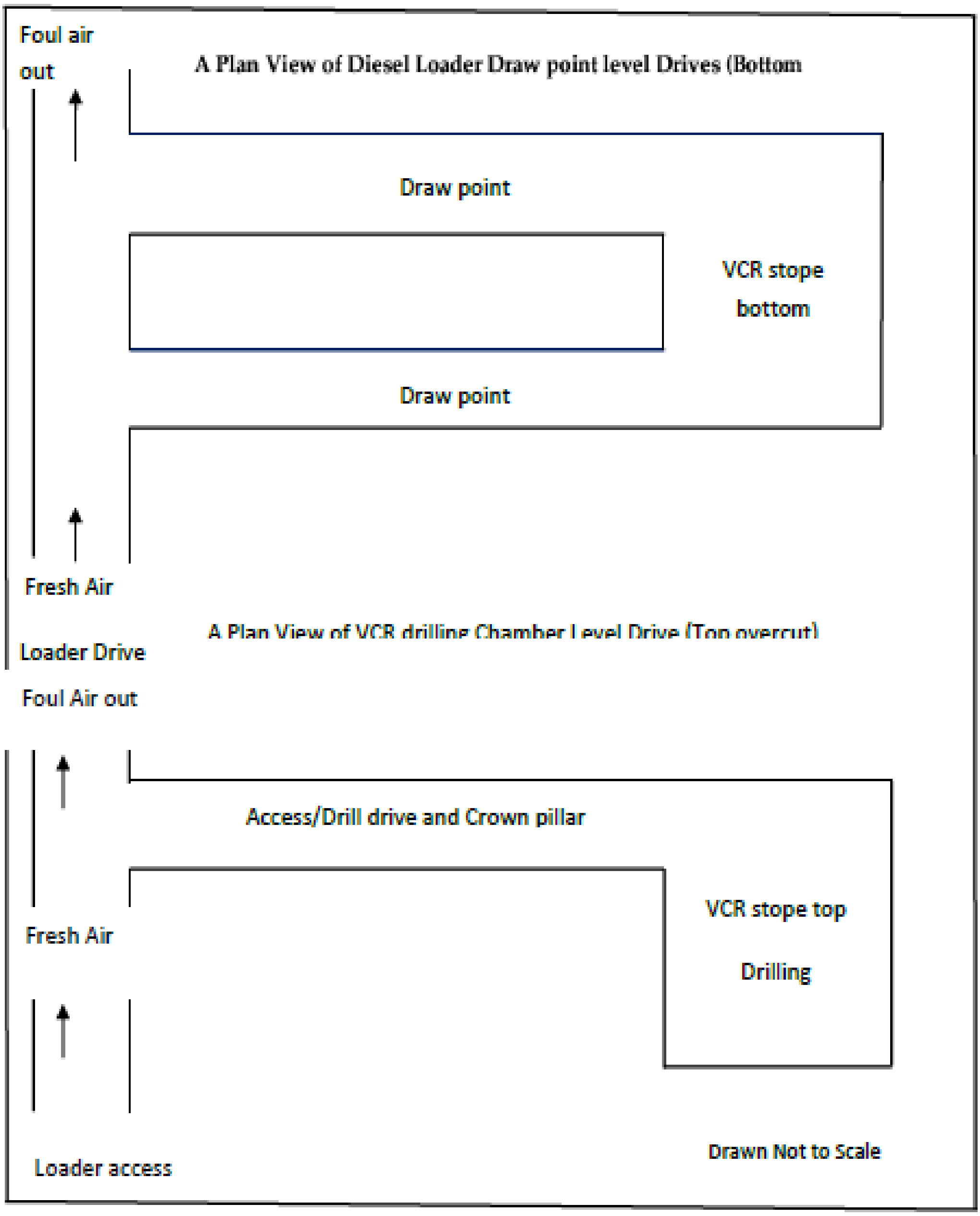


Figure 2: Mindola Sub-Vertical Mine General Layout of Mining Method showing VCR chamber excavations (Mopani Copper Mines - Mindola Mine Planning Department, Report, 2014).

This mining method involves development of haulage for ore transportation, a drill drive, a ventilation drive, and a crown drive for each production retreat. Fresh air enters the top drilling chamber level drive, sweeps the working area and flows out to the main return airway circuit as foul (return) air. In a similar manner, fresh air enters the bottom level draw points through the access drive, sweeps the draw points were diesel loaders operate and foul air returns to the main return airway system, Figure 3, Mutawa H. (2013).



It is planned to extend this mining method to the 5220L – 6365L ore zone of the mine. The Mine plans to increase copper ore production, within the next three years, from 1.1 Mt to about 2.5 Mt per year.

The planned increase in copper ore/waste production (from both development and stopping operations) is shown in Figure 8. This will involve expansion of mining operations to a depth of about 1930 m below surface.

The increased depth of mining will impose challenges on the capacity of the current ventilation system. The geothermal gradient of Mindola mine is about 1.67 °C for every 100m depth (Castillo, D.O. et al, 2002).

At the planned depth of future mining (1,930 m below surface), the virgin rock temperature (VRT) is expected to increase by about 32.3 **°C. The heat generated due to the increase in th**e VRT will result in increased wet-bulb temperature of ventilation air to above the scientific and or legal limit of 31 °C (Mining Regulation 901 (2) (e)), Mutawa H. (2013).

In some parts of the current mining area, wet-bulb temperatures in excess of or close to 31 °C have been recorded. This means that the current ventilation system does not provide adequate air at the required wet-bulb temperature, to maintain a safe and conducive working environment in current and future mining areas, as prescribed by scientific or legal limits.

Previous studies (Castillo et al, 2002) undertaken on the ventilation system of Mindola mine, indicated that high wet bulb temperatures of ventilation air being experienced in many working areas of the mine were due to the geothermal heat resulting from increased depth of mining. The studies recommended refrigeration of downcast air as the only viable solution to combat the excessive heat in order to reduce high wet bulb temperatures to legally/scientifically set limits, Payne, T. and Mitra, R. (2009).

Another study conducted by Mutawa (MCM Report, 2013), however, postulated that increased air quantity, re-designing of the Main Return Airway (MRA), and re-alignment or extension of return air raises, and combined use of booster fans with main upcast shaft fans to increase air pressure, could address the heat at depth problem.

The main aims of the research were to evaluate the current ventilation system of the 4440L – 5220L ore zone (current mining area) and to establish a parametric database for the design of a ventilation system for the end of life of the mine. The ventilation database would subsequently be used for the design of an end of mine life ventilation system for the 5220L – 6365L ore zone. The following were the specific objectives for the present study:

(i) To establish air volume, velocity, and wet-bulb temperature of the current ventilation system in order to test compatibility with baselines, set by scientific or legal standards, for diluting heat, dust, and obnoxious gases.

(ii) To calculate the total heat load generated from different sources in the current mining area in order to determine the volume and velocity of ventilation air that is required to dilute it.

(iii) To derive operating parameters for the design of the future ventilation system by establishing volume, velocity, and wet-bulb temperature of ventilation air that are required to dilute heat, dust, and obnoxious gases in current and future mining areas of the mine.

MATERIALS AND METHODS

The research was undertaken through surface and underground surveys and physical inspection of primary and secondary ventilation infrastructure such as main Upcast fans and shafts, downcast shafts, main intake and return airways and raises. In addition, there were internet searches as well as reviews of the Mopani Copper Mines (MCM) technical and consultant reports. The methods and materials used in the collection of both secondary and primary data are shown in Figure 4.

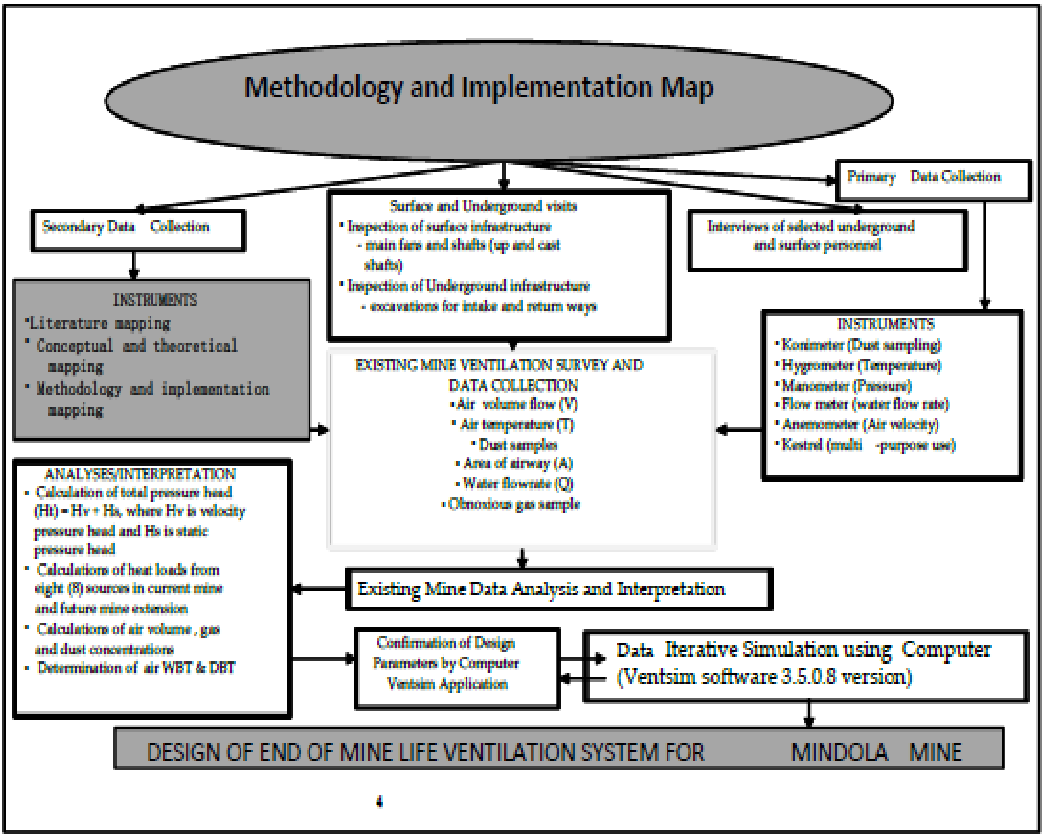


Figure 4: Materials and Methodology map

These were supplemented by consultative visits, and interviews of relevant mine officials. The interviews conducted targeted selected underground workers, engineers, and managers as well as contractors. Both primary and secondary data collected during the study period were deliberately structured to be statistically significant. Surveys conducted involved measurement of air wet and dry-bulb temperatures, main upcast fan pressure, air volume, air velocity and computation of heat from various sources.

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The primary and secondary data collected was then coded, tabulated, analyzed, and then interpreted for final results.

The volume flow rate of air was not measured directly, but velocity and area measurements were made at selected cross sections of the mine airway. Then using scientifically established empirical formulae, air volumes were computed from measured data. The cross sectional areas for various airways were determined by measuring the width and the height of the each airway.

Whillier, A. (1981) has derived empirical formulae for calculating heat flow into mine excavations by applying the theory of fluid flow of Bernoulli's modified equation for ideal fluids. These empirical equations have been used by many researchers in designing deep-level underground mine ventilation networks.

The heat generated by each underground source was calculated by applying the heat flow equations in order to determine the total heat resulting from increased depth of mining.

The calculated total heat was then used to compute the amount of air required to dilute it, by applying psychometric charts. The Air Tonnage Ratio (ATR) method was also used to determine the overall quantity of air that would be required for increased production in the mine. This was obtained by using the current average monthly broken rock tonnage production and the total down cast air in the current production area. The design data was then obtained by carrying out a simulation of data obtained from an evaluation of the ventilation system on the current production area. (Aminossadati, S.M., and Hooman, K. 2004).

A number of trials were conducted using various combinations of existing at the mine and that obtained from other underground operations of similar ventilation configuration. The resulting data from the computer software was the coded and tabulated as design data base.

The values for air quantity obtained from the three (3) methods; Heat Method, ATR method and Computer simulation software method were then compared in order to obtain the desired result. (Hopperstead, K. 2001).

RESULTS AND DISCUSSION

The Ventilation Downcast System

Mindola Sub-Vertical (MSV) mine had two surface downcast shafts, namely, Numbers 1 and 2 shafts (Figure 5). Salient data is summarised in Table 1.

Table 1: Downcast ventilation system salient statistics

Main down cast shafts	No. 1 and No.2
Intake air splits on 2630 and 2880 main levels:	
2630 level total air quantity	370m³/s
Total cross section area	52.0m²
2880L total air quantity to the Sub-vertical (SV) shaft	255m³/s
Total cross section area	25.0m²
From 2880L ventilation air enters the SV service shaft:	
Shaft diameter	7.0m
Cross section area of service shaft	28.0m²
Two raises from 2880L to 4440L with diameter of 5.0m serve as additional down casts to the deeper levels:	
Total area of the two raises	39.0m²

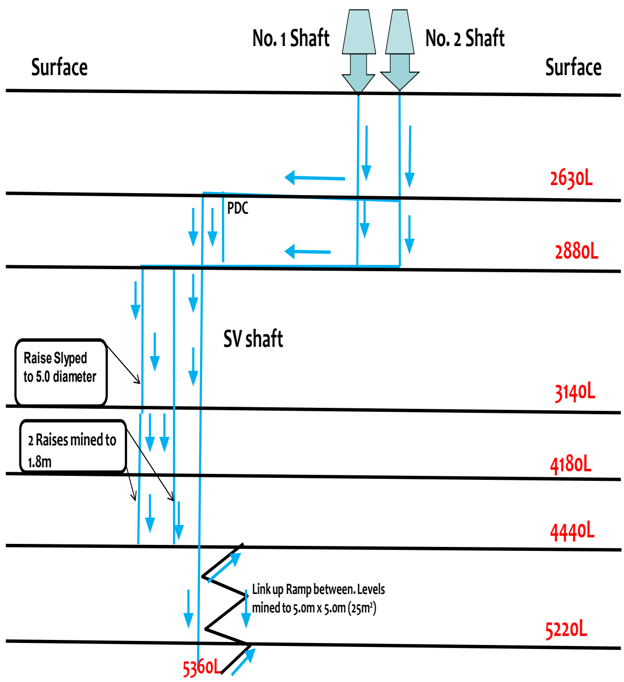

Figure 5: Current Downcast Ventilation System

The Upcast Ventilation System

The Upcast system comprised three principal main Upcast fans namely, V10, V5 and V9 that service the southern, central and the northern parts of the mine respectively, Figure 7. Salient data is summarized in Table 2.

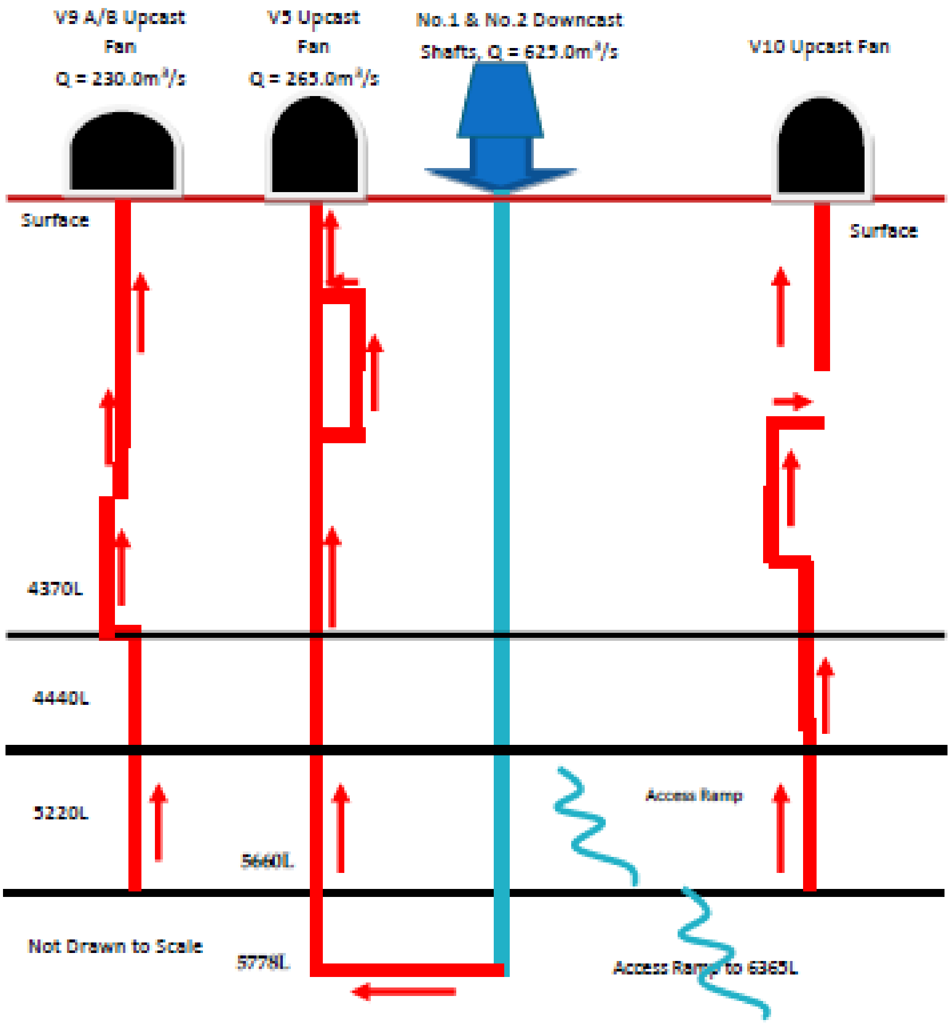

Figure 6: Current Upcast Ventilation System

Figure 7: V10 Upcast Main Fan Mindola Mine-A typical Upcast fan

Table 2: Upcast ventilation system salient statistics

Main Upcast fans	V10, V5, and V9
Total upcast quantity measured 755m³/s at 1.2 kg/m³ (Mass flow of 906kg/s)	
Fan total static pressure	11.9kPa.
V9 Upcast Fan This fan services the northern part of the mine through various levels and internal raises down to 5220L.	
Single inlet, centrifugal fan with a backward curved aerofoil bladed impeller.	
Fan static pressure	4kPa
Air volume	230.0m³/s
Nominal power	1500kW
Shaft diameter	5.5m
V5 Upcast Fan	
Axial flow fan	
Fan static pressure	2.0 kPa
Air volume	265.0m³/s
Nominal power	1500kW
Shaft diameter	4.9m
V10 Upcast Fan (Figure 5) (The shaft runs from surface to 4370L with extension raises to 5770L on the southern part of the mine.)	
Single inlet, centrifugal fan with a backward curved aerofoil bladed impeller.	
Fan static pressure	4.5 kPa
Air volume	260.0m³/s
Nominal power	1500kW
Shaft diameter	4.9m

Planned Ore/Waste Production

The planned increase in copper ore production is shown in Figure 8. The monthly tonnages are assumed to be evenly distributed on all levels. This was done in order to evenly spread the impact of geothermal heat on all operating levels. In this way air quantities can be estimated more uniformly when design parameters are determined.

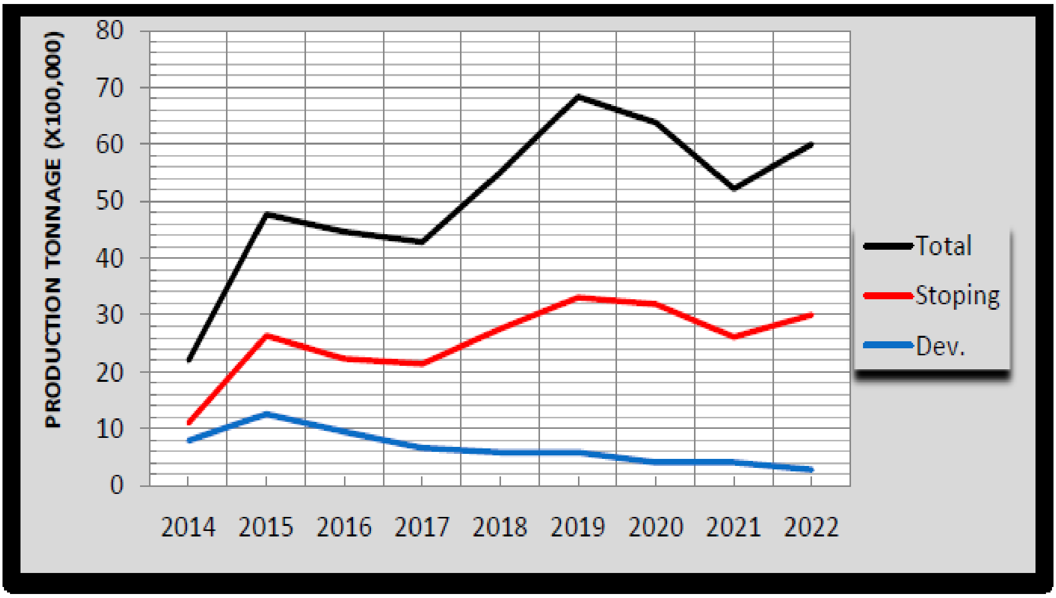

Figure 8: Planned Annual Ore/Waste Production

The analysis of temperatures, volumes, dust, obnoxious gas concentrations measured in the main body air and diesel exhaust fumes yielded results that are shown in Figures 9 and 10. The measured dust and obnoxious gas concentrations have been compared with the Zambia Mines Safety Department's legally/scientific set baselines for diluting such contaminants.

The Geothermal Gradient used in predicting the Virgin Rock Temperatures (VRT's) is 1.67 degrees for every 100 m depth of mining as verified by the thermal water (fissure water) temperatures measure on, 4180L, 4440l and 5360L. The results of the values of VRT for the levels below 4440L were plotted against the Wet-bulb (WB) temperatures to highlight the tendency of the WB temperature of air to increase with increased depth of mining. The resulting graphical data is shown in Figures 12 and 13.

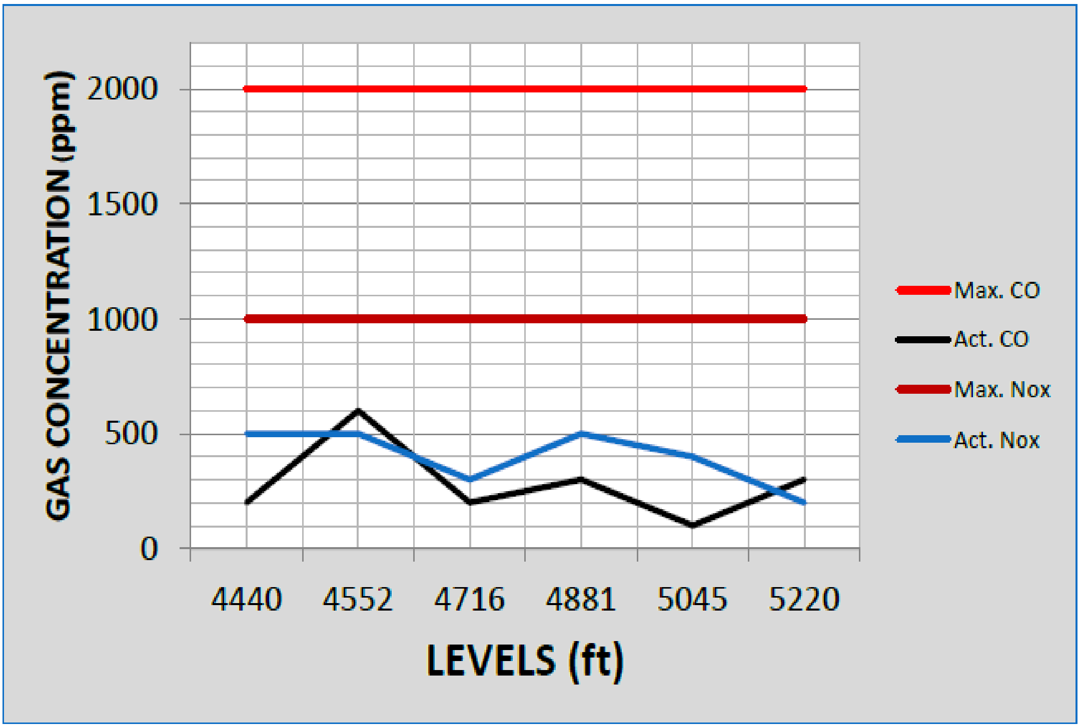

Figure 9: Diesel Equipment Exhaust Gas Concentrations vs Legal Limits

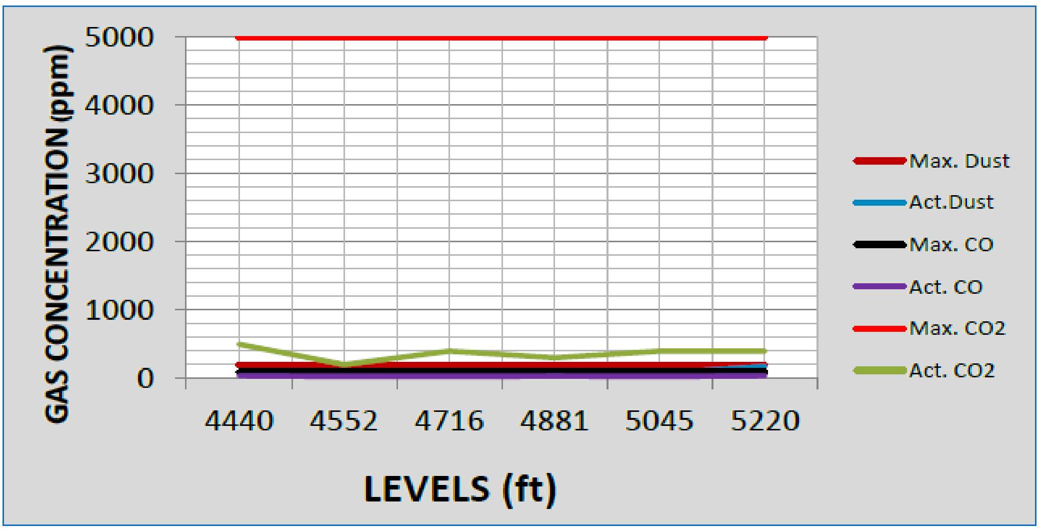

Figure 10: Dust, obnoxious gas concentrations in general ventilation air vs Legal Limits.

Determination of Heat Loads from Various Sources

Heat load generated by auto-compression

When air descends down the mine its internal energy is converted into kinetic energy due to its own weight and acceleration due to gravity. Heat produced, therefore, increases the air temperature.

Enthalpy change for 1 kg dry air with a Thermal capacity of 1.005 kJ/kg = ΔH = g x change in elevation.

But $\Delta H = Ma \times C_{pa} \times \Delta t = C_{pa} \times \Delta t$ for a kg of airflow down the mine.

Where ΔH = Enthalpy due to change in Dry-bulb temperature of air

Ma = the mass of dry air flowing in the mine, in kg.

C_{pa} = Thermal capacity of air (= 1.005 kJ/kg ^{o}C)

g = Acceleration due to gravity (= 9.81 m^2/s)

Δt = Change in air temperature, ^{o}C

Heat generated, Q_c = Mass flow of air x Thermal capacity of air x Change in temperature.

Heat Load generated by exposed strata (rock surface)

Heat from exposed strata (wall rock) in a developing tunnel is determined as follows:

In the zone close to a developing face, say within 50 – 100m, the rate of heat is given by;

$$Q_s = 0.006 \times k \times (L+4 \times DFA) \times (VRT - DB)$$

Where Q_s = Heat from exposed strata (wall-rock), kW

ks = Thermal conductivity of the rock, W/m ^{o} C

L = Distance advanced by the face in a month, m (100.0m)

Heat generated by broken rock (ore & waste

Heat from broken rock is then estimated by the formula;

$$Q_{br} = M_{br} \times (VRT - DB) \times C_{pbr}$$

Where Q_{br} = Heat from broken rock, kW

M_{br} = Mass of rock broken per month (waste and ore), kg

DB = Intake Dry Bulb Temperature, ^{o} C

VRT= Virgin Rock Temperature, ^{o} C

C_{pbr} = Specific heat capacity of broken rock = 0.925 kJ/kg^{o} C

Heat load generated by diesel-propelled equipment

The heat load from diesel-propelled equipment is determined as follows:

By using the total amount of fuel consumed per month for the total equipment fleet, the total Heat Load is estimate as;

$Q_{diesel\ =}$ Average monthly consumption x Fuel Density x Calorific Value of diesel

Where, the average monthly consumption of fuel = 75,360 Litres = 75.36m^3/month

Fuel Density = 845 kg/m^3

Calorific Value of diesel = 45.6 MJ/kg

$Q_{diesel\ equipment}$ = (45.6MJ/kg x 75.36 m^3 x 845= <u>2,9MJ/month</u>

Heat load generated by blasting of explosives

Heat load from explosives consumption is given by the expression:

Heat generated by explosives blasted = Mass of explosives (kg) x Energy released per kg of explosives

Energy (heat) release for ANFO/EMULSION = 2,820 kJ/kg (MCM Engineering Report, 2014)

Heat load from Electrical Equipment (underground fans)

Since fans do not do any useful thermodynamic work, the heat given to air by fans is equal to the thermal equivalent of the electrical energy put into the fan drive motors. This energy is converted into heat which directly affects the intake air. Heat generated by both auxiliary and booster fans is considered to be the most significant for electrically powered equipment. This is because auxiliary and booster fans run, almost on a 24-hour basis.

The heat from electric motor, Qelect. = (Power Rating of motor (kW) x Load Factor)/ (Motor Efficiency)

Load Factor = 0 .75 and Motor Efficiency = 0.80

Heat load generated by human respiration (Metabolic Heat)

Studies conducted in South African gold mines indicated that the metabolic heat produced by the human body varies with the kind of work being performed ("Environmental Engineering in South African Mines", 1982, page 604/5).

Typical figures for the rates of heat produced by each person in a shift are given as follows;

At rest = 90 – 115 W

Light work =200 W

Moderate work = 275 W

Hard work = 470 W

Heat produced by human respiration (metabolism) is thus given by the formula;

Q_{met} = No. of workers (during peak) x Rate of heat

Where Q_{met}= Heat produced by human respiration, kW

Estimation of the heat produced by men working in a mine requires that we use the shift in which population of workers underground is peak, and generally day shift is taken to be ideal.

Peak no. of workers underground is estimated at 800/shift

Heat load generated by fissure (thermal) and service water

Water collected from drilling and face washing down operations is gathered as common water in the drain drive where thermal water flows.

Heat from fissure and service water is estimated by the formula;

$Q_w = M_w \times C_{pw} \times (VRT – DB)$, kW

Where Q_w = Heat from fissure water, kW

C_{pw} = Thermal capacity of water, kJ/C

M_w = Mass flow of water, l/s

DB = Intake Air Dry Bulb Temperature, C

Summary of Heat Loads from Various Sources in the Current Mining Areas.

The summary of the calculated Heat Loads from the various heat sources of the mine were computed using the individual formulae shown in 4.1.1 – 4.1.8. The resulting data has been presented in graphical form in Figure 11.

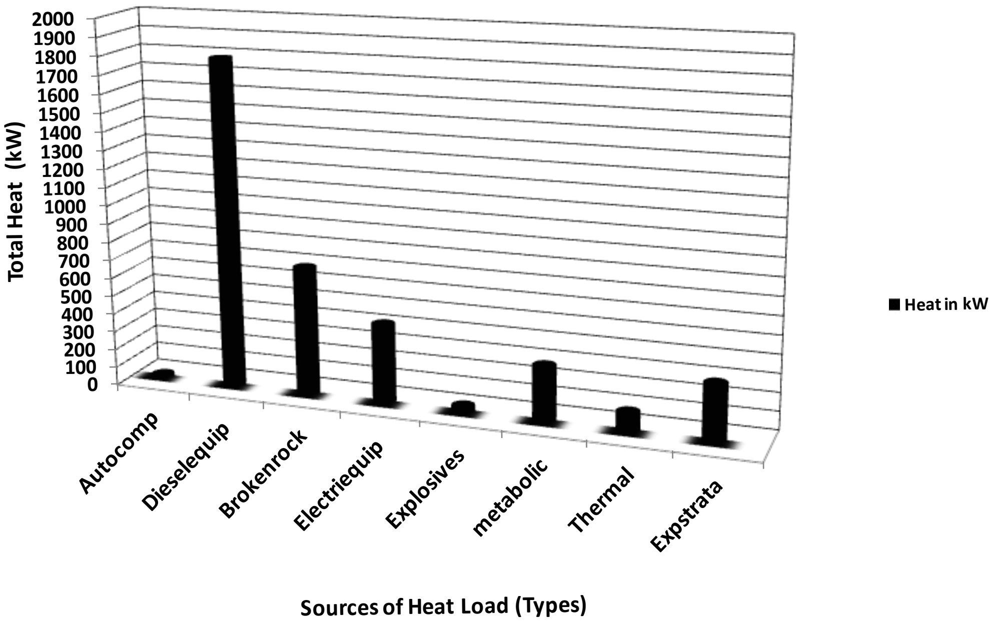

Figure 11: TOTAL HEAT LOADS FROM VARIOUS SOURCES (4440L – 5220L Ore Zone)

Current Air Volumes Measured against Produced Tonnages of broken rock

For the purpose of ventilation planning, the factor of air quantity per tonne of broken rock has considerable significance. This index helps one to predict the air requirements of a new area or mine. The quantity of air to be circulated underground is measured in m^3/s per 1000 tonnes of broken rock. This is then related to the Virgin Rock Temperature obtaining at any operating level when estimating the total air required to dilute the generated heat from broken rock.

Virgin Rock Temperature, Dry and Wet – bulb Air Temperatures versus Depth of mining.

The Virgin Rock Temperature (VRT) that was calculated for each main level was plotted against the depth of mining in order to determine the effect of heat at various depths of the mine. The resulting graphical data shows a steady rise in both wet bulb and dry bulb temperature as the depth increases, Figure 12.

The dry and wet-bulb temperatures that were measured at each increasing depth of mining were also plotted against the depth of mining in order to determine the effect of heat on the dry and wet-bulb temperatures of underground ventilation air. The graphical data revealed that both the wet and dry-bulb temperature of air steadily increased with depth.

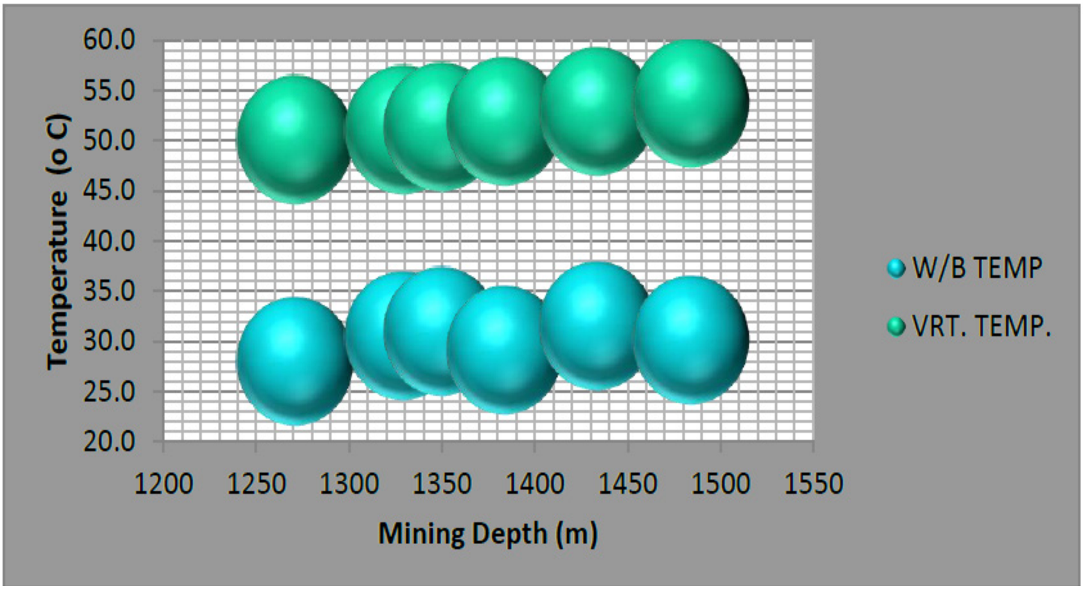

Figure 12: Virgin Rock Temperature as measured on a linear scale against depth of mining

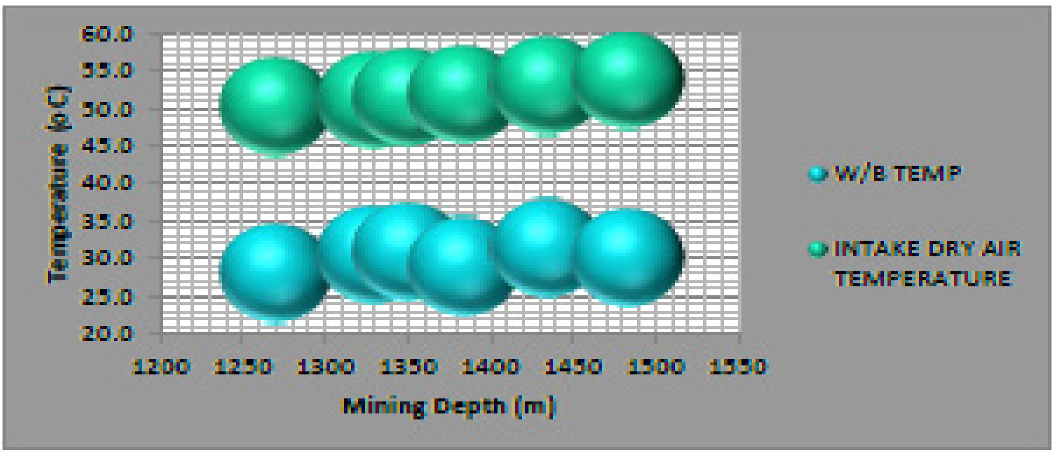

Figure 13: Intake Dry air Temperature/Wet-bulb Temperature Depth of mining

Air Leakage Pattern and Analysis

The results of the analysis of air leakage both in the primary and secondary ventilation circuits yielded results which show that a substantial amount of intake air leaks into the Upcast system.

The analysis showed that a total quantity of 224m^3/s (about 36%) of ventilation air leaked both in the primary and secondary ventilation circuits. The secondary ventilation circuit contributed about 159 m^3/s which is 72% of total air leakage, while the primary circuit contributed 65 m^3/s (29% of total air leakage).

The air leakage in the primary ventilation circuit is due to collapsed old areas on levels such as 450, 650, 870, 1127 and 1380 respectively in the upper levels of the mine connecting the main Upcast fan system. These areas are inaccessible. The air leakage in the secondary ventilation circuit is due to poor installations of vent canvases, seals and regulators and in some cases absence of the same.

Air recirculation in Secondary Ventilation Circuit

In the secondary ventilation circuit, air recirculation due to loose connections between auxiliary fans and vent ducts was observed in various sections of the mine. Air recirculation was also due to mismatched installed fans in some cases, where a volume flow of intake air of say, 8m^3/s was matched with a 45 kW fan of air capacity 15 m^3/s in an advancing development face. A 45 kW fan generates a substantial amount of heat through air recirculation around the intake and its discharge point well in excess of 45 kW. This can contribute to the heat load generated in the working areas of the mine. An auxiliary fan of 22 kW duty would suffice for an intake air volume of 8 m^3/s.

The use of a correctly sized auxiliary fan in a development face, in this case, would result in about 50 per cent reduction in the heat load generated by electrical equipment. This results in improved positional cooling efficiency of the fan at the working face.

Establishing Design Parametric Database

Calculating the total volume of ventilation air required to dilute total generated heat;

Intake Air parameters

Temperature (summer)	=	24.0 ºC /28.0ºC WB/DB
Barometric Pressure (Surface)	=	88.5Kpa

Using the psychometric formula below

Apparent Specific Volume (ASV):	=	0.970m^3/kg
Sigma Heat	=	**70.5kj/kg**

Return Air Conditions (Based on comfortable temperature)

Temperature:	=	29.0/33.0ºC WB/DB
Barometric Pressure: (1940mL)	=	120.5Kpa

Using the psychometric formula below

Apparent Specific Volume (ASV): = **0.810m³/kg**

Sigma Heat = **80.6kj/kg**

(read from *psychometric* Chart)

Therefore, ΔSigma Heat = (Heat of Return air - Heat of Intake air)

= (80.6 – 70.5)

= 10.1kj/kg of V air

Therefore **1 kg** of ventilating air will remove 1**0.1kj** of heat.

Total Heat Load generated in the mining block = **4, 743.1kW**

Hence the minimum quantity of ventilation air required to dilute heat generated in the mining block is; = (4,743 x 0.810)/ (10.1)

= **380.4m³/s**

Design Parameters

A computer simulation (Figure 14) was conducted on the mine ventilation network system using design parameters that were obtained from the excel spreadsheet calculation. The design data is shown in Table 12. The input data included main upcast fan air quantities for the three upcast fans (V5, V9, and V10), airway resistance, sizes (cross sectional areas) and airway distances.

The profile shown in Figure 13 shows the designed ventilation system for the current (4440 L – 5220 L and future (5220 L – 6365 L) mining areas of Mindola mine. The profile shows the three (3) air intake shafts, No.1, No. 2 and the new service shafts with the existing main exhaust shafts, V 5, V 9, and V 10 respectively. The proposed MRAW is on 4370 L and the eight (8) main vertical return air raises from 5960 L to 4370 L (MRAW).

Table 13: Design Parametric Data for the current (4440L – 5220L) and future (5220L – 6365L) mining ore zones

Type of opening	Size of opening	Current Figures (m^3/s)		Design Figures (m^3/s)		Remarks
	(Area- m^2)	Volume (Q)	Velocity (V)	Volume (Q)	Velocity (V)	Location
Main Shaft (intake)	20.0 – 25.0	370.0	7.11	250	10.0	Shaft Collar
Main Shaft (service)	20.0 – 25.0	255.0	10.2	500	20.0	Shaft Collar
Main Haulage	14.0 – 20.0	17.7	1.2	63.0	4.5	Underground
Drives (drill/ acc.)	14.0 – 20.0	19.0	1.3	56.0	4.0	Underground
Main Ramp	14.0 – 20.0	22.0	1.3	56.0	4.0	Underground
Cross Cut (access)	14.0 – 20.0	17.3	1.2	56.0	4.0	Underground
Raise (intake)	4.0 – 6.0	7.2	0.8	8.8	2.0	Underground
Raise (return)	4.0 – 6.0	5.4	0.6	8.8	2.0	Underground
Diesel Workshop	14.0 – 20.0	17.7	1.2	28.0	2.0	Underground
Workshop (other)	14.0 – 20.0	14.7	1.0	28.0	2.0	Underground
Conv. Belt tunnel	7.0 – 9.0	13.5	1.5	17.5	2.5	Underground
Crusher Chamber	14.0 – 20.0	36.0	1.8	70.0	5.0	Underground
Draw Point X/C	14.0 – 20.0	15.2	1.1	17.5	2.5	Underground
Loading Box	14.0 – 20.0	12.0	1.0	14.0	2.0	Underground
Pump Chamber	14.0 – 20.0	8.5	1.3	17.5	2.5	Underground
Diesel equipment. Dilution		Actual = 0.05m^3/kW		Design = 0.08 m^3/kW		Rated Power

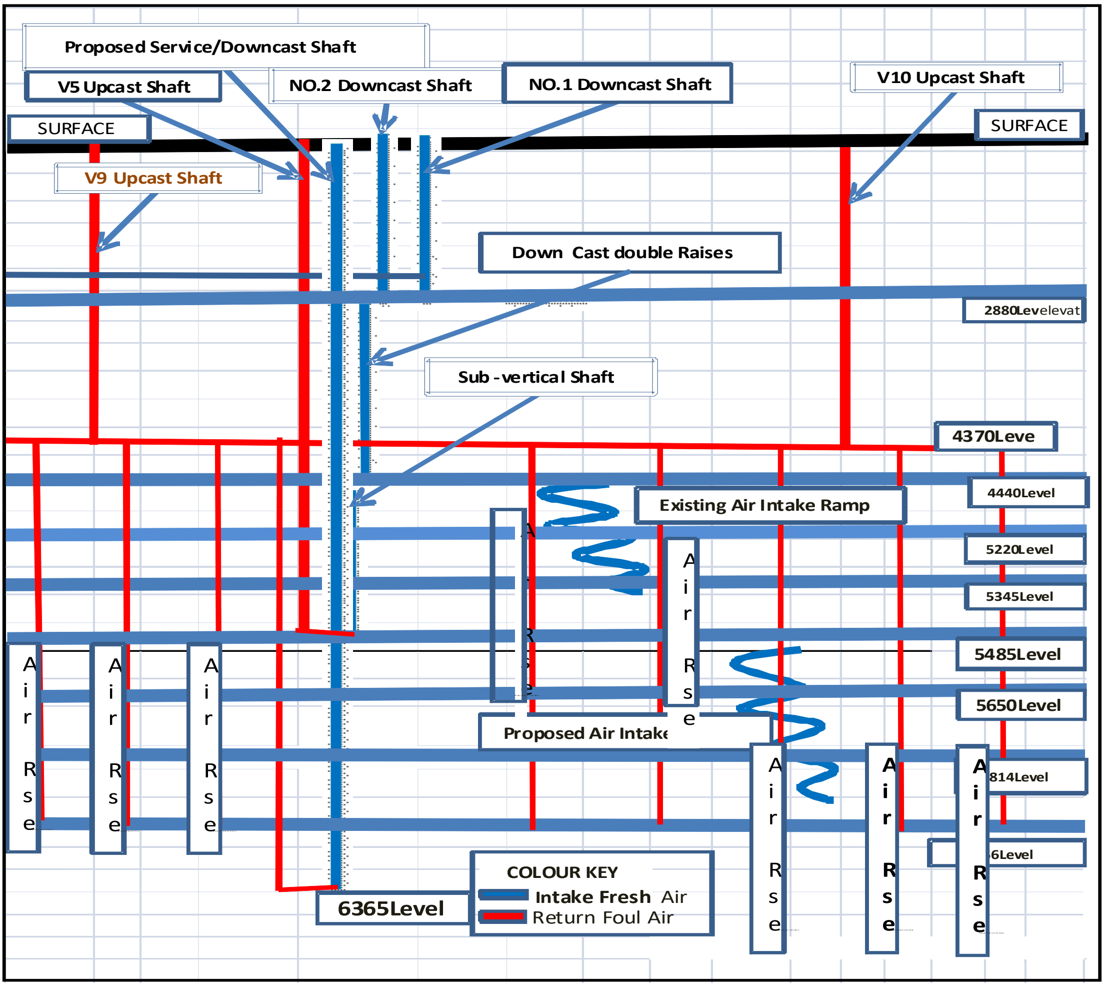

Figure 14: Profile of Designed Parametric Data obtained from the computer simulation of mine ventilation network.

Confirmation of Design Parametric Data using Ventsim Computer Software

Figure 14 shows the computer simulation output of the designed ventilation system based on the data used in designing the ventilation network. Three separate ventilation network runs were conducted, the network errors were corrected and the final run yielded the results that are recorded in Table 12.

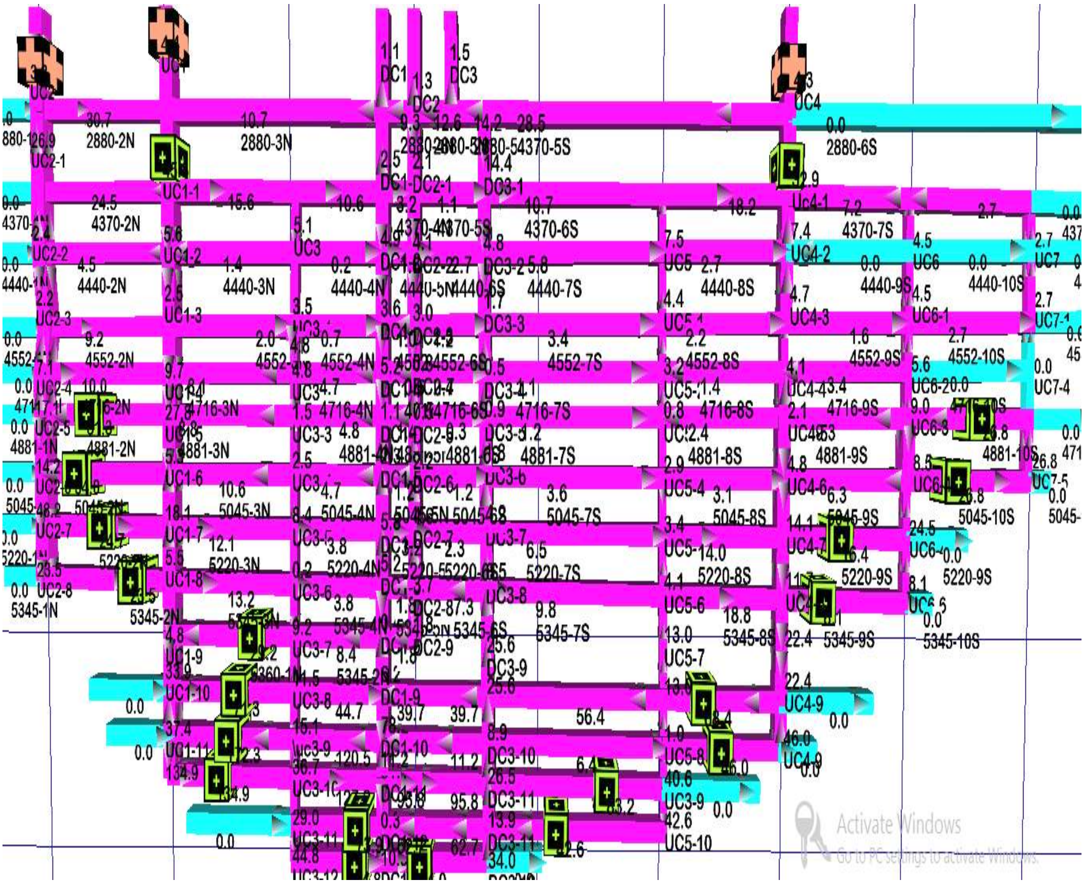

Figure 15: Ventsim Computer Software Simulation of Ventilation Network

Table 14: Comparative Analysis of Air Quantities by Methods of Calculation

Type of method used in determining air quantity	Quantity (m^3/s)
Heat and personnel	942.6
Air Tonnage Ratio (ATR)	960.4
Computer Software Simulation (VENTSIM)	923.85

Data Analysis by Computer software simulation (VENTSIM 3.5.0.8. Software)

A computer simulation was conducted using design parameters that were obtained from the excel spreadsheet calculation. The input data included main upcast fan air quantities for the three upcast fans (V5, V9, and V10), airway resistance, sizes (cross sectional areas) and airway distances. A copy of the actual simulated ventilation network is shown above in Figure 13.Three separate ventilation network runs were conducted, network errors corrected and the final run yielded the following results (Table 12).

It is important to note, however, that the software used in the simulation exercise was limited to the simulation of air quantity in the current and future mining areas and pressure drop in the three main upcast air return raies. It was not necessary to carry out a simulation of heat loads, as these were obtained using empirical formulae for the current mining area (4440L – 5220L ore zone) and then extrapolating the calculated heat loads using the calculated ATR for the future mining area (5220L – 6365L ore zone). The required volume of air for diluting the generated heat in the mine was determined using the calculated total heat load. The simulation of heat would only be necessary if the option of refrigeration of air was considered in the study. But this is not the case in this study. The comparison of the total quantity of air by the three methods shows that the quantity of air in the current and future working areas of the mine, as determined by the three methods are in agreement within the acceptable limits of ventilation design.

The amounts determined by Heat/Personnel and the ATR methods were 942.6 and 960.4 m^3/s respectively, are higher than that determined by the Computer Software Simulation (923.85 m^3/s). This is due to the fact that the total quantity of air determined by the Computer Software (Simulation Method) did not take into account the geothermal heat content of the air.

The total pressure shortfall in the primary return circuit of the mine as determined by the Computer software simulation is 1.08 kPa. The shortfall in pressure as calculated using empirical formula in the excel spreadsheet is 1.1 kPa. This again confirms that the value obtained from the Computer software simulation and the excel spreadsheet are compatible. Consequently, it is clear from the above comparative analysis that Booster fans are required at the bottom of the main upcast fans in order to compensate for the shortfall in pressure. The comparison of the total quantity of air by the three methods shows that the quantity of air in the current and future working areas of the mine, as determined by the three methods are in agreement within the acceptable limits of ventilation design.

The amounts determined by Heat/Personnel and the ATR methods (942.6 and 960.4 m^3/s) respectively, are higher than that determined by the Computer Software Simulation (923.85m^3/s). This is due to the fact that the total quantity of air determined by the Computer Software (Simulation Method) did not take into account the geothermal heat content of the air. The total pressure shortfall in the primary return circuit of the mine as determined by the Computer software simulation is 1.08 kPa. The shortfall in pressure as calculated using empirical formula in the excel spreadsheet is 1.1kPa. This again confirms that the value obtained from the Computer software simulation and the excel spreadsheet are compatible. Consequently, it is clear from the above comparative analysis that Booster fans are required at the bottom of the main Upcast fans in order to compensate for the shortfall in pressure.

CONCLUSION

The findings made from the research study yielded the following conclusions;

(i) The total quantity (volume) of downcast air in the current areas of the mine (4440L – 5220L) ore zone) is 625m³/s.

(ii) The quantity (volume) of downcast air that is required in the current and future mining areas is about 942.6 m³/s (an increase of about 50%).

(iii) The amount of air that flows in a number of current areas of the mine though sufficient to dilute dust and obnoxious gases, is not adequate to dilute heat generated in the mine in order to keep the Wet-bulb temperature of ventilation air below the scientific/legal baseline of 31.0 °C.

(iv) The geothermal gradient (increase in VRT due to depth of mining) of the mine has directly affected the wet-bulb temperature of mine air.

(v) The estimated total heat load generated by various sources in the current mining (4440L – 5220L ore zone) is about 3,654 kW.

(vi) Diesel equipment, geothermal gradient, and electrical equipment are the major sources of heat in the mine contributing about 88 per cent of the total heat load while the rest contributed 12 %. This heat is significant to the ventilation system design.

(vii) Diesel equipment with the highest amount of heat contributed about 1,792.5 kW (49% of total heat in the mine).

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