Review and Design the Ventilation System of Mindola Sub-Vertical Shaft, Mopani, Kitwe

By

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A dissertation submitted to the University of Zambia in fulfillment of the requirements for the Master of **Mineral Science in Mining Engineering**

DECLARATION

I, Mungalaba Fred, do hereby declare that this is the original work done solely by me and that all sources of information have been duly acknowledged and that this work has never been presented at this university or indeed any other university for academic purposes.

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DEDICATION

This project is dedicated to my lovely wife Nchimunya Ng'andu and my son Chipego Mungalaba. You are my source of happiness and success. I owe you lots you have encouraged and supported me from day one and it shall not fade uncelebrated.

To my brothers Alfred, Emmanuel, Brian and Modley thank you for your love, care and motivation. You have always stood by me during my bad and good times.

Finally, I dedicate this project to all my friends, I thank God for you.

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ABSTRACT

Temperatures and humidity at Mindola Sub-Vertical (MSV) shaft have been rising with depth as mining of lower lying copper reserves is achieved. This rise in temperature has been catalysed by a number factors notably the increase of diesel equipment, geothermal gradient (increase in depth), auto compression and other sources of heat. Productivity, workers morale is affected negatively in these conditions.

The aim of the study was to review the current Mindola Sub-Vertical shaft ventilation system and design a ventilation system that will reduce or control the heat levels and other underground air pollutants. This was achieved by identifying the major sources of heat and quantifying the heat using heat determination methods. The heat from diesel equipment, water fissures, rock strata, explosives, men (metabolism) and electrical machines was also quantified. Subsequently, the air volume required to remove or reduce heat and other mine pollutants like dust and gases was determined.

Findings of the study were that areas at MSV have been experiencing high temperatures that exceed the allowable temperatures of 31°C wet bulb and 28°C dry bulb by the Zambian mining regulation. The temperatures were found to be as high as 35.0°C and 32.5°C dry and wet bulb respectively. The findings reviewed that the old vent raises in the upper levels where mined at 2.4metres diameter while the new raises in the deeps are mined at 3.1metres hence causing improper flow of air.

The study results indicated that total quantity of heat for the mine was 32,033 kW and ventilation air required was 826.45m^3/s compared to the current down casted air of 624.00m^3/s. The study has also established that the V9 shaft main up cast fan should be upgraded by introducing another fan to increase the volume quantity by 202.24m^3/s. The return air raises in old upper levels which were mined at 2.4metres diameter must be slipped to current size of new raises at 3.1metres diameter to avoid choking which results in short circuiting and recirculation of air in both the primary and secondary circuits.

Keywords: *Mine Heat Sources, Heat Loads, Mine Air Pollutant, Wet Bulb Temperature, Dry Bulb Temperature.*

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CHAPTER ONE: INTRODUCTION

1.1 Background

Mopani Copper mines is owned by Glencore (73.1% shareholders), First Quantum (16.9% shareholders) and ZCCM Holdings (10% shareholders). The company has two mining sites on the Copperbelt: the Nkana mine site in Kitwe and Mufulira mine site in Mufulira. Nkana mine site lies immediately to the west of the city of Kitwe while Mufulira is in the north. The Kitwe corporate head office is situated at the corner of Central Street and 5th Avenue in Nkana west, Kitwe.

Nkana mine operates four (4) underground shafts, six (6) Open Pits dotted across the Nkana Oxide Cap, a Concentrator and a Cobalt plant. Mindola Sub- Vertical shaft (MSV), where the project was undertaken is one of the four underground shafts found at Nkana mine. The other shafts are Central shaft, South Ore Body shaft (SOB) and North Shaft. MSV is the largest and deepest of shafts at Nkana mine.

One of the challenges currently facing the Mindola Sub-Vertical Shaft is the difficulty in maintenance of a conducive mine environment as the shaft deepens where workers can work safely. Other than the provision of right quantities of fresh air, to dilute dust and gases, heat and humidity have become a major concern as the shaft deepens. At the current depth of 1,550m, MSV is experiencing high dry and wet bulb temperatures of about 35.0°C and 32.5°C respectively. The location of three shafts is shown in Figure 1.

Figure 1: Location of Central Offices and three Shafts in Nkana (Google Map 2015)

1.2 Nkana Mine history

Nkana Mine was first recorded in 1910 by Mr. J Moffat Thompson when mineralized block shale was discovered along Wusakile stream. In 1914 Mr. Donald Mclean, a railway employee first registered a block of claim known as Nkana copper. In 1916 another railway employee by the name of WC Winiortre pegged Nkana's copper and in 1922 the copper prospect was sold to Bwana Mukubwa. Diamond drilling was carried out under the directive of Mr. Austin J Bancroft and in 1931 Nkana's first copper was produced at Central shaft. In 1933 work had to begin on the mine's second shaft at MSV and in the same year an electrolyte refinery was erected. In 1956, SOB was opened and later on in 1974 another shaft North of Mindola shaft called the Mindola North shaft was officially opened.

By 1964, either Anglo American corporation (AAC) or Roan Selection Trust privately owned all copper mines in Zambia. However in 1972 government nationalized all mining assets together with all other means of production. With nationalization in 1972, all the private mining companies were rearranged into two companies namely Nchanga Consolidated Copper mines

(NCCM) and Roan Copper mines Limited (RCM). NCCM comprised of all mines which were previously owned by Anglo American Corporation while RCM comprised all mines previously owned by Roan Selection Trust.

Nkana which was then called Rokana was under Anglo American Corporation and thus became a constituent of NCCM. In 1982 Government merged NCCM and RCM into one to form the Zambia Consolidated Copper Mines (ZCCM) Limited. Unfortunately ZCCM suffered many challenges resulting in the decline in copper production and revenue. The loss of revenue from copper combined with other economic and political factors adversely affected the country's economic performance leading to a decision in 1989 to switch to a private sector driven economy. This was the privatization that led to the birth of Mopani Copper Mines (MCM) plc and other existing mines.
After privatization it was important that the name be retained with the acronym "MCM" as MCM is the London Metal Exchange (LME) listed name for copper produced at Mufulira refinery. Mufulira copper is considered to be among the purest copper in the world. The name Mopani was chosen by vote among employees. The Mopani is an indigenous Zambian hard wood tree with long life and extreme resilience. (The Zambian Traveler, Jan/Feb 2006 edition)

1.3 Statement of the problem

The mine reserves are almost depleted in the upper section of the Mindola SV hence the need to mine lower reserves below 5220L to 5970L. This means mining beyond the geothermal threshold limit of 1500m below surface (geothermal gradient for MSV is 2°C/100m). This increased depth has resulted in corresponding increase in dry and wet bulb temperatures hence making the mining environment unconducive for personnel.

Furthermore, heat from auto compression, water fissures, working machinery and lights, men (human metabolism) and blasting of explosives, has contributed to the ever increasing temperatures with increase in mining depth. Currently, the working areas in the deeps section of Mindola Sub-Vertical are recording high dry and wet bulb temperatures of about 35.0°C and 32.5°C respectively at high relative humidity. These kinds of temperatures have resulted in 15 reported heat related cases in the last 8 months.

Therefore, the study seeks to come up with the appropriate ventilation design that will significantly control and reduce the amount of heat in the mine atmosphere thereby improving the work environment that is vital to improved productivity.

1.4 Main objective:

To review the current ventilation system and design a suitable system for Mindola Sub-Vertical shaft below 1,500m.

1.4.1 Specific objectives:

The research focused on the following specific objectives:

(i) To determine the total heat pick up at MSV Shaft
(ii) To determine the air quantity and pressure requirements needed to ventilate MSV shaft
(iii) To design appropriate ventilation system
(iv) To investigate the possibility of employing refrigeration methods

1.5 Research questions

In order to evaluate and design an effective ventilation system that will manage high temperatures and humidity, the research aims to answer the following research questions:

1) What are the major sources of heat and other contaminants at MSV shaft?
2) How do the high temperatures being experienced in the working areas affect the working personnel and labour productivity? Are the high temperatures in tandem with the requirements of the mine regulation of Zambia?
3) What are the current air quantities and pressure being supplied to the mine working? Do the air quantities and fan pressures meet the demand?
4) Are mine developments (ventilation network) all mined to the right dimensions?
5) Does the current ventilation system require refrigeration system?

1.6 Aim of study

The principal aim of this study is to review and design an effective ventilation system at Mindola Sub-Vertical Shaft.

1.7 Significance of Study

The outcome of this study will be of immense value to the company and workers in general in that it will lead to control of heat in the deep sections of the mine. Furthermore, reduction of heat in the work environment will result in the decrease the number of heat related disorders in the working personnel and increased productivity of both men and machinery.

1.8 Scope and Limitations of Study

The study concentrated on evaluating the current ventilation system at Mindola Sub-Vertical shaft with the aim of coming up with an appropriate ventilation design capable of controlling heat buildups and removal of air contaminants from the system. The possibility of employing refrigeration methods was also investigated. The study did not however, cover detailed aspects of occupational heat exposure.

CHAPTER TWO: LITERATURE REVIEW

2.1 Introduction

This chapter presents a review of available literature on the following subjects namely: heat Load determination, mine air pollutants, air quantity determination methods, mine ventilation network and refrigeration in deep mines.

2.2 Heat and humidity

Heat and humidity are encountered in tropical locations and in deep underground mines, where the virgin rock temperatures and air temperatures increase with depth, principally due to the geothermal gradient and auto-compression of the air column. Both natural and 'manmade' sources contribute to the underground heat load with blasting and equipment operation also being significant heat contributors (T. Payne & R. Misra, 2008).

2.2.1 Effects of high levels of Heat and Humidity

a) Heat Loss Greater than Heat Generated

When the rate of heat loss to the surroundings (H_{loss}) is more than the rate of metabolic heat generation (H_{gen}), the effects observed can be shown as in Figure 2.

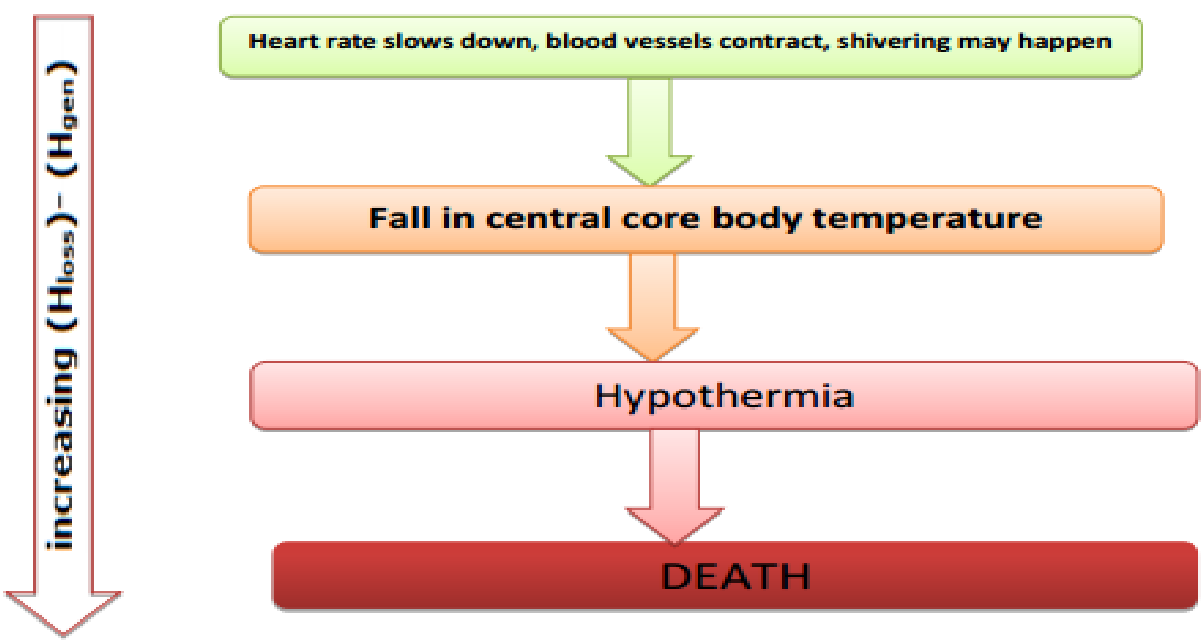

Figure 2: Physiological effects observed when (Hloss) is more than Hgen

b) Heat loss lower than heat generated

This is more common in the mine atmosphere. This condition to a miner/worker may arise due to the combined effect of wet-bulb and dry-bulb temperatures, psychrometric properties of the mine air and velocity of air-current. Besides these factors, the effects observed on an individual are also dependent on 'time duration of exposure' to such condition. This condition generally arises when the dry bulb temperature of the air increases beyond a certain limit such that heat removal from the body is reduced. If the dry-bulb temperature of air/atmosphere goes on increasing, heat removal from body through radiation and convection may reverse the directions (heat may be added to the human body from the atmosphere through the process of radiation and convection). In such cases, the metabolic heat from the body is removed through evaporation of sweat. Sweat, produced by the sweat glands, adds moisture to the surrounding air.

Table 1: Effect of increased dry bulb temperature (Payne & Misra, 2008)

Dry bulb temperature (ºC)	Body temperature (ºC)	Physiological effects
≤25	------	Normal blood circulation, no observable effect, heat removal from the body mainly through radiation and convection
25-29	------	Heat removal rate increases, slight rise in central core body temperature, Vaso-motor control of body increases blood circulation
29 – 37.5	------	Body starts sweating, heat removal mainly through evaporation
≥36.9	36.9	Body temperature equals dry bulb temperature, heat transfer through convection and radiation, reverses the direction, heat removal from body through evaporation only
------	39	Heart beat rises above 140 beats/ min, fatal
------	41	Unconsciousness, coma, may lead to death
------	≥43.3	Sudden death

If the humidity of the air is already high, the evaporation of sweat is reduced and therefore the central core body temperature of an individual rises rapidly. Even if air is moderately humid, sweat glands reduce their capacity to produce sweat with time. A time comes when no more sweat is produced due to fatigue of the sweat glands. If exposure to such environmental conditions continues, it may lead to heat stroke ultimately leading to the death of the person.

2.2.2 Effect of high wet bulb temperature

The difference between the wet bulb temperature and dry bulb temperature decide the relative humidity of the air. The lesser the difference the more the relative humidity. Air at high relative humidity hampers rate of evaporation. At higher wet bulb temperature, the reduction in evaporation of sweat causes body temperature to rise and in extreme case if the rise in temperature exceeds 2^{o}C, heart beat becomes faster, which on continuation may lead to unconsciousness or even death. Table 2 lists the rise in body temperature with wet-bulb temperature.

Table 2: Increase in body temperature with wet bulb temperature (Misra, 1986)

Wet-bulb temperature in K	Rise in body temperature in K
≤302.15	0.11 – 0.66
302.65 – 304.85	0.33 – 0.77
305.35 – 307.65	0.66 – 1.55
≥307.65	1.44 – 1.90

2.2.3 Heat Hazards

The Table 3 summarizes the heat hazards in increasing order of their severity level:

Table 3: Heat hazards in increasing order of their severity level (Tripti Maurya, 2015)

No.	Disorder	Symptoms	Causes	Treatment	Prevention
1	Transient heat fatigue	Reduced performance is seen, particularly in skilled physical work, mental task and those requiring concentration	Discomfort caused from the heat at a lower level which cannot result in other heat illnesses	No treatment is necessary unless other signs of heat illness are present	Acclimatization and training
2	Heat Rash	“prickly heat”, tiny, raised, blister- like rash	a. Skin is constantly wet due to sweating	Keep skin clean and dry	Taking shower after working in hot environment. Keeping skin dry
			b. Sweat gland duct become plugged, leading to inflammation		
3	Heat Cramps	Painful muscle spasms in the arm, legs or abdomen during or after working in hot environment	a. The cause of heat cramp is not very well understood	Resting, drinking water, glucose water and consuming more salt than usual	Adequate water and salt intake (do not use salt Tablets)
			b. But, it is been assumed that it may occur due to loss of salt from sweating		
			c. Dehydration may be an important factor		

Table 3 continues on the next page.

No.	Disorder	Symptoms	Causes	Treatment	Prevention
4	Heat syncope	a. Fainting while standing erect and immobile b. This is variant of heat exchange c. Symptoms of heat exhaustion may precede fainting	a. Due to dehydration blood volume decreases b. Blood pools in dilated vessels of the skin and lower body, making less blood available for the brain	a. Move the victim to cool place b. Make the victim rest and drink fluids	Acclimatization, drinking plenty of water , avoiding stand in one place and intermittent activity to avoid blood pooling
5	Heat exhaustion	a. Fatigue, weakness, dizziness, faintness b. Nausea, headache c. Moist, clammy skin; pale and flushed d. High pulse rate e. Normal or slightly higher body temperature	Dehydration causes blood volume to decrease	Victim should be allowed to rest at cool place and should drink plenty of water	Acclimatization and drink plenty of water.
6	Heat stroke	a. Usually hot, dry skin; red, mottled or bluish b. Sweating may still be present c. Confusion, loss of consciousness,	a. Partial or complete failure of sweating mechanism b. The body cannot get rid of excess heat	a. This is a medical emergence b. Call the doctor and start cooling the victim immediately by taking the	Acclimatization, closely monitoring for signs of heat illness, medical screening and drinking plenty of

		convulsions d. Rapid pulse e. Rectal temperature greater than 104 ºF		victim to a cool place c. Soak the clothes and skin of the victim with cold water and use a fan to create air circulation d. Medical treatment is mandatory	water

2.2.4 Fall of miners' working efficiency

With the rise in temperature and relative humidity of the mine atmosphere, a decrement in miners' efficiencies is observed. Fall of miners' efficiency with varying temperatures is shown in Table 4 and Figure 3.

Table 4: Fall in working efficiency against wet bulb temperature and effective temperature (Misra, 1986)

Effective Temperature (ºC)	Wet bulb Temperature (ºC)	Efficiency %
30.2	29.6	90
32.5	31.2	80
33.7	32.2	70
34.7	33	60
35.5	33.7	50
36.2	34.3	40
36.7	34.9	30
37.2	35.3	20
37.2	35.8	10
-	36.2	0

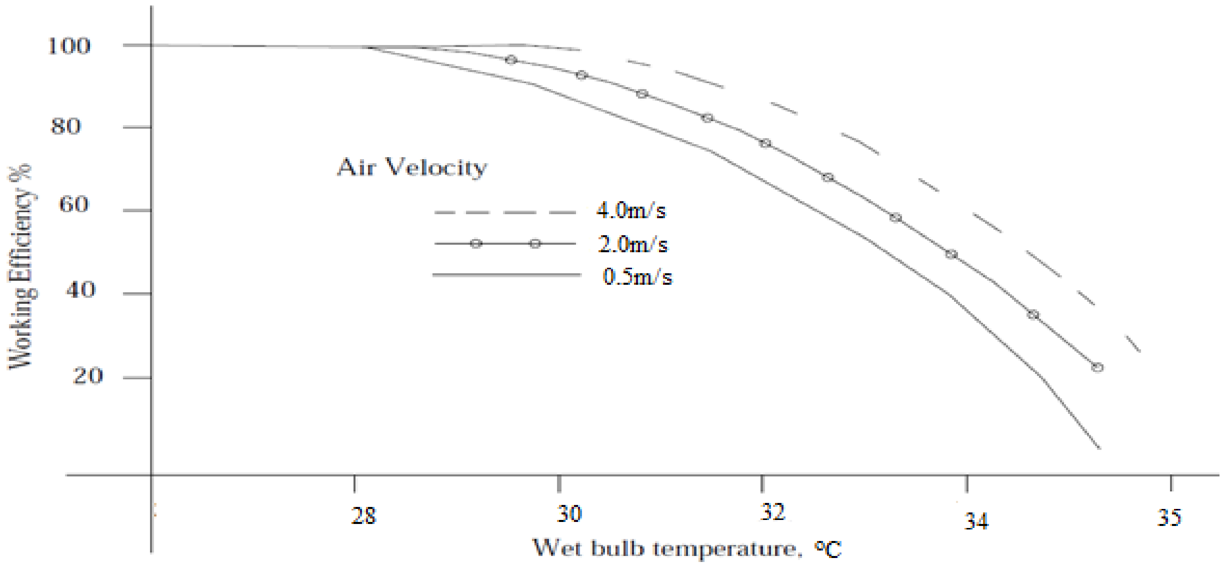

Figure 3: Working efficiency vs. wet-bulb temperature (Le Roux, 1972)

Initially efficiency decreases gently and becomes steeper when the wet bulb temperature goes on increasing.

2.2.5 Sources of heat and quantification

Quantification of the heat emitted into a mine or section of a mine is required in order to assess the airflow needed to remove that heat (McPherson, 2012). It is therefore important during the study of heat flow into mine openings is to classify, analyze and attempt to quantify the various sources of heat.

a) Rock strata heat

Malcolm J. McPherson, (2012) explains that in reviewing the literature for means of determining the amount of heat that will be emitted from the strata, the ventilation engineer is faced with a bewildering array of methods varying from the completely empirical, through analytical and numerical, to computer simulation techniques. The basic difficulty is the large number of variables, often interacting with each other, which govern the flow of strata heat into mine airways. These include:

- The length and geometry of the opening
- Depth below surface and inclination of the airway
- Method of Mining

- Wetness of the airway surfaces
- Roughness of the airway surfaces
- Rate of mineral production or rock breaking
- Time elapsed since the airway was driven
- Volume flow of air
- Barometric pressure, and wet and dry bulb temperatures
- Virgin (natural) rock temperature
- Distance of the workings from downcast shafts or slopes
- Geothermic step or geothermic gradient
- Thermal properties of the rock
- Other sources of heating or cooling such as machines and cooling plant.

Malcolm J. McPherson continues to explain that with such a variety of parameters, it is hardly surprising that traditional methods of predicting strata heat loads have been empirical. Perhaps the simplest and most common of these has been to quote strata heat flux in terms of heat load per unit rate of mineral production; for example, kW per tonne per day. This can provide a useful and rapid guideline provided that all conditions are similar to those on which the value was based. However, as rate of production is only one of the several variables listed above, it is obvious that this technique can lead to gross errors if it is applied where the value of any one of those variables is significantly different from the original sets of measurements used to establish the kW/tonne/day value.

The more sophisticated empirical techniques extend their range of application by incorporating estimated corrections for depth, distance, age, inlet conditions or, indeed, any of the listed variables considered to be of local importance.

A **hybrid method** has grown out of experience in running climatic simulation packages. It is often the case that, for particular conditions, some of the input variables have a very limited effect on the results. By ignoring those weaker parameters it is then sometimes possible to develop simple equations that give an approximation of the heat flow.

In view of these alternative methodologies, what is the mine environmental engineer supposed to do when faced with the practical problems of system design? Experience gained from major planning projects has indicated the following recommended guidelines:

- If the objective is to plan the further development of an existing mine, or if there are neighboring mines working similar deposits at equivalent depths and employing the same methods of working, then the empirical approach (kW per tonne per day) may be adopted for the overall strata heat load on the whole mine or major sections of the mine. This presupposes the existence of data that allow acceptable empirical relationships to be established and verified. Employing past and relevant experience in this way provides a valuable and simple means of arriving at an approximate heat load (McPherson, 2012).
  However, let the user beware. If the proposed mine project deviates in any significant manner from the conditions in which the empirical data were compiled, then the results may be misleading. In particular, great caution should be exercised when employing empirical relationships established in other geographical regions. A phrase commonly heard at mine ventilation conferences is "what works there, doesn't work here."

- The hybrid equations are very useful for rapid approximations of heat flow into specified types of openings. Dr. Austin Whillier of the Chamber of Mines of South Africa,1982 produced many hybrid equations for easy manual application, including:

(1) Radial heat flow into established tunnels

$$q = [3.5\,L\,K^{0.854}\,(VRT - \theta)]/1000\ (kW) \quad (1)$$

Where;

q = heat flux from rock strata (kW)

L = Length of the haulage (m),

K = thermal conductivity of the rock (W/ m ^{0}C),

VRT = virgin rock temperature (^{0}C),

θ = ambient dry bulb temperature (^{0}C)

The above formula is used to calculate heat load from recently mined haulage or from advancing production and development ends within the duration of one month. When an opening is driven in the mine, the temperature of the wall of the opening is close to the virgin rock temperature. Provided the ventilating air is at a lower temperature, heat will be transferred from the rock to the air. The above elements are required in order to calculate the heat load from the advancing ends of heading.

(2) Advancing end of a heading

$$Q = 6k\,(L + 4DFA)\,(VRT - \theta) \quad (2)$$

Where;

L = Length of the advancing end of the heading (m).

This should be not greater than the length advanced in the last month. Equation in (1) may be used for the older sections of the heading.

DFA = Daily face advance (m)

(3) One Dimensional Heat Flow toward Planar Surface

Assuming good convective or evaporative cooling of the surfaces,

$$q = [A\,(K\,\dot{\rho}C)\,0.5\,(VRT - \theta) / t^{0.5}] /1000 \text{ (kW)} \quad (3)$$

Where;

A = Surface area of the opening (m^2),

$\dot{\rho}$ = Density of the rock (kg/m^3),

C = Specific heat of the rock (J/kgºC)

t = Time since the mine haulage was mined (s)

K= Thermal conductivity of the rock (W/mºC)

θ = Ambient air dry bulb temperature (ºC)

The above formula is used to calculate heat load to the mine by examining the surface area of the opening and the age. As the duration of the exposure of the rock surface to the ventilating air increases, the rate of heat transfer to the mine opening is reduced and the temperature of the rock will be approximately that of the ventilating air. Since most of the openings at Mindola SV shaft are very old, the heat transfer to the mine is negligible. Hence heat due to planer surface will not be considered in this project.

b) Geothermal Gradient

The temperature of sub-surface rock rises steadily with depth. In most climates, the so-called virgin-rock temperature (t), ceases to be affected by surface temperature changes and is taken as constant for reference purposes at about 50m beneath the surface. It then increases with depth at approximately a uniform rate in a given locale and rock formation; the rise is termed the geothermal gradient, or the temperature change per unit depth Δt/ΔZ. However, even within a single mining district, it varies and seldom can be considered a constant (Hartman, 1997).

Although it varies in different mining districts and can be determined accurately only by field measurement, the earth's heat flux q/A (heat flow per unit area) ranges from 0.04-0.06 W/m² and averages 0.05 W/m². Knowing the heat flux and the thermal Conductivity k (W/mºC) of the various rock formations encountered, the geothermal gradient in ºC/100 m for a given formation can be calculated from conductive heat-transfer theory:

$$q = kA\left(\frac{\Delta t}{\Delta Z}\right) \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots (4)$$

Where;

q = Strata heat

K = Thermal Conductivity

A = Unit Area

Δt = Change in temperature

ΔZ = Change in depth

Re arranging the equation, the overall gradient can then be determined by cumulating the gradients for the individual formations.

$$\frac{\Delta t}{\Delta Z} = \frac{q/A}{k} \quad (5)$$

c) Auto Compression

Strictly speaking Auto-compression is not a source of heat, for it results in an increase in temperature of either air or water as a result of the conversion of potential energy into enthalpy, and not as a result of a flow of heat from an external source. However Auto-compression is of such importance in mine ventilation and cooling that it must be included in any discussion on sources of heat in mines. (Hartman, 1997)

The term Auto-compression should, strictly speaking, only be applied to compressible fluids; in the mining context this means either the ventilation air or the compressed air. However the principle involved, the conversion of potential energy into enthalpy, applies to any fluid and as this effect is of importance when water flows into a mine. Auto-compression is dealt with in this section as it affects both air and water.

The steady flow energy equation is used to obtain the following expression for the increase in enthalpy which occurs when any fluid flows from a higher to a lower elevation, for example in a mine shaft;

$$H_2 - H_1 = g\,(Z_1 - Z_2) + q \quad (6)$$

Where H = enthalpy (J/kg)

Z = height above datum (m)

g = gravitational acceleration (m/s^2) and

q = heat added from surroundings (J/kg)

(Subscripts 1 and 2 refer to the inlet and outlet ends of the airway respectively).

The enthalpy thus increases by 9. 79 kJ/kg for every 1,000m decreases in elevation. As it is the fluids (the ventilation air, the compressed air, and water) circulating through the mine which provides the only means for removing heat from the mine, and as the temperature at which these fluids leave the working areas of the mine is limited by the design reject wet-bulb temperature, this enthalpy increase serves to reduce the amount of heat which the fluid is able to remove. It is important to realize, however, that irrespective of the fluid the enthalpy increase is always 9.79 kJ/kg per 1,000m decrease in elevation (Hartman, 1997).

d) Fissure water and channel flow

Groundwater migrating through strata toward a subsurface opening can add very considerably to the transfer of geothermal heat through the rock. Such water may continue to add heat to the ventilating airstream after it has entered the mined opening.

Fissure water is often emitted at a temperature close to the virgin rock temperature (VRT). In circumstances of local geothermal activity or radioactive decay, it may even be higher. The total heat load on the mine environment can be calculated from the flow rate and the drop in temperature of the water between the points of emission and effective exit from the mine ventilation system (McPherson, 2012). The formula for Heat flow from fissure water is given by:

Heat flow from fissure water (q) =

(M) x (C) x (VRT –WB) (Ramsden and Von Glenn, 2012)……………………………….…. (7)

Where:

M= Flow rate

C= Specific heat of water

The rate at which heat is emitted into the airstream depends upon the difference between the temperature of the air and the water, and whether the water is piped or in open channels. In the latter case, cooling by evaporation will be the major mode of heat transfer and will continue while the temperature of the water exceeds the wet bulb temperature of the air. Hot fissure water should be allowed no more than minimum direct contact with intake air. The most effective means of dealing with this problem are:

(a) To transport the water in closed pipes and

(b) To restrict water flow routes to return airways.

A less effective but expedient measure close to the points of water emission is to restrict air/water contact by covering drainage channels by boarding or other materials.

Where chilled service water is employed, condensate run-off may be at a temperature below that of the local wet bulb temperature. While this continues, the condensate will continue to absorb both sensible and latent heat, thus providing a cooling effect on the airflow.

In, or close to the working areas of many mines, it is often inevitable that open drainage channels will be used. Similarly, the footwalls or floors may have areas that are covered with standing or slowly moving water.

e) Heat from machinery and lights

The operation of all mechanized equipment results in one, or both, of two effects; ***work is done against gravity*** and/or ***heat is produced***. A conveyor transporting material up an incline, a shaft hoist and a pump are examples of equipment that work, primarily, against gravity. Vehicles operating in level airways, rock breaking machinery, transformers, lights and fans are all devices that convert an input power, via a useful effect, into heat. (McPherson, M. J. 1993)

It is known that it is hard to design a machine with 100 % efficiency. Thus, all machines add heat to the mine air by non-mechanical work produced by them. Different types of machinery used in mines can be classified on the basis of their source of power supplied. They are:

- Electrical equipment
- Diesel engines
- Compressed air run equipment/machinery

f) Electrical equipment

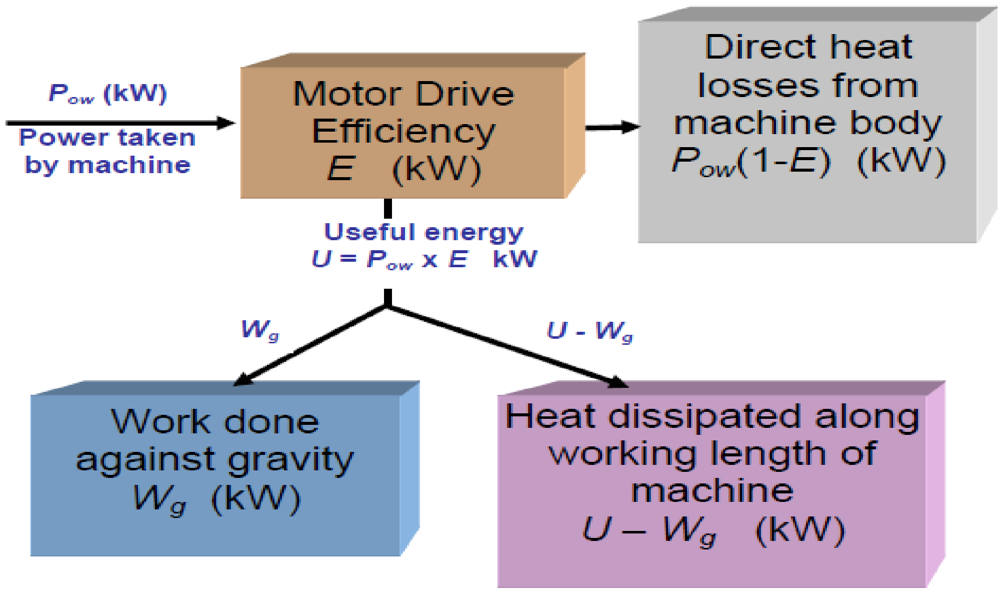

Figure 4: Electrical machine power utilization (McPherson, 1993)

Figure 4 illustrates the manner in which the power taken by an electrical machine is utilized.

The machine efficiency is relevant, within this context, in two ways. First, the total amount of heat produced can be reduced only if the machine is replaced by another of greater efficiency to give the same mechanical power output at lower power consumption. For any given machine, the total heat produced is simply the rate at which power is supplied, less any work done against gravity.

Secondly, the efficiency of the machine determines the distribution of the heat produced. The higher the efficiency, the lower the heat produced at the motor and transmission, and the greater is the percentage of heat produced at the pick-point, conveyor rollers, along the machine run or by any other frictional effects caused by the operation of the device. (McPherson, 1993)

Fan is also an eminent source of heat in underground mines. Some of the work done by a fan is converted into kinetic energy while a large part of it is converted to heat. Due to fan the rise in temperature of the mine air is 0.83 K per kPa of power developed by it (Misra, 1986). That is why it is logical to install booster fan in return and prefer exhaust fan on the surface instead of forcing fan. Forcing fans add heat to the air at the initial stage itself thereby increasing the heat load on the air supplied to the workings. On the other hand, exhaust fans add heat mostly at the out bye end of the mine workings. It is advisable to install booster fans in the return in case of necessity so as to reduce the heat load on the ventilation system.

g) Diesel equipment

Diesel equipment are less efficient compared to electrical units, they have approximately 1/3 efficiency (McPherson, 2012). As a thumb rule, the heat generated by a diesel engine is taken to be 2.8 to 3kW per kW of the rated power of the equipment (Banerjee, 2003). Heat load from a diesel engine is also calculated on the basis of average rate of fuel consumption by it in a shift. The heat produced by a diesel engine appears in three ways each of which may be roughly of the same magnitude:

1) Heat from the radiator and machine body (mostly sensible heating).
2) Heat in the exhaust gases (as latent heat of water vapour and sensible heat of other by-product gases).
3) The remaining as useful shaft power. It is further converted to heat by frictional processes as the machine performs its tasks (as sensible heat).
   Thus, diesel equipment do not only add heat load but also add moisture to the mine air.

Stinnette and De Souza (2013) states that calculating the heat production from a diesel-powered machine can be practically accomplished through the following process:

First, the ***Total Heat*** is determined based on the fuel consumption rate:

$$Q_T = \frac{f_c + C_{diesel}}{3600} \qquad (8)$$

Where:

Q_T = Total Heat (kW)

f_c = fuel consumption (litres/hr)

C_{diesel} = Calorific value of diesel (kJ/litre)

Next the ***Latent Heat*** is calculated:

$$Q_I = \frac{V(H2O) + I(H2O)}{3600} \qquad (9)$$

Where:

Q_I = Latent Heat (kW)

$V_{(H2O)}$ = Volume of water production (litres/hr)

$I_{(H2O)}$ = Latent heat of vaporization of water (kJ/kg)

The ***Sensible Heat*** generated is simply the difference between the Total Heat and the Latent Heat:

$$Q_s = Q_T - Q_I \qquad (10)$$

Where:

Q_s = sensible heat (kW)

Q_T = total heat (kW)

Q_I = latent heat (kW)

The generic criterion that is used most widely for initial estimates of required airflow is based on rated output power of the diesel equipment:

Heat load (q) = R_e (Power rating for diesel units)

For design purposes, many ventilation planners employ 6 to 8 m^3/s of airflow over the machine for each 100 kW of rated diesel power, all equipment being cumulative in any one air split. Stinnette and De Souza (2013)

h) Compressed air run engines

In general, compressed air machinery does not add any heat to the mine air. It is because the heat added due to the frictional work done by compressed air machinery gets compensated by the heat absorbed by the exhaust of the compressed air unit. However, addition of heat to mine air takes place when hot compressed air is taken down the shaft in pipelines.

i) Oxidation

Oxidation, in general, is always exothermic in nature. Oxidation of coal, timbers, Sulphur in metal mines liberate significant amount of heat. We can make an estimate of heat addition to the mine air by calculating depletion of oxygen in the air-current. A 0.1% depletion of oxygen between intake and return can cause rise in temperature of mine air by 7℃ provided that all oxygen is utilized in oxidation of coal (Banerjee 2003). In operating airways, the heat generated by oxidation is added directly to the ventilating air. But, heat produced by oxidation in waste areas, gob, etc. is not added totally to the mine air. A part of the heat produced is retained in the areas and in worst case the temperature rise in such areas may be such that, it may be the cause of spontaneous fire.

j) Blasting operations

It is estimated that as much as 90-95% of the energy released in blasting eventually finds its way into the underground environment as heat. It is probable that some of the heat produced will be carried away with the blasting fumes out of the development end and some will remain in the broken rock which may be released prior to and during rock removal. The proportion of heat removed by each process is a function of rock fragmentation, the ventilation arrangements and the mining cycle. (T. Payne & R. Misra 2008)

k) Rock movement

Subsidence of strata in underground mines is common in part of mines where extraction of ore has been done. In this case rock falls to the floor and the potential energy associated with the rock is almost totally converted into heat. It is fortunate that only a small part of this heat is added to the ventilating air. Most of the heat is trapped in the broken rock itself. It is said that about 1% of the total heat addition to mine air is contributed by falling rock (after, Misra, 1986). It may reach to almost 9% if no heat is trapped inside the broken rock and all of them are added directly to the main ventilating air.

On the other hand, in underground metal mines, ore/mineral descending down through an ore pass or vertical bunker may immediately be exposed to a ventilating air stream. If no heat is transferred to the ventilating air and surrounding, the temperature of the falling rock is calculated by the equation:

$$(T_2 - T_1) = \frac{\Delta Z g}{Cr} \text{ °C} \quad \ldots (11)$$

Where,

T_2 = Temperature of rock just after impact/fall (°C)

T_1 = Temperature of rock before impact/fall (°C)

ΔZ = falling distance (m)

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

C_r = specific heat of rock (J/kg°C)

l) Human metabolism

Due to metabolism, human beings produce heat even when they are at rest. The heat produced is dependent on various factors such as:

- Manual work being done
- Body surface area of the man
- Level of mental stress, etc.

Heat produced is proportional to all these factors. Men at work produce more heat and the amount of heat produced goes on increasing as the work becomes more vigorous. A man transfers heat to the surroundings in three different ways:

- through heat loss from body surface
- through respiration
- through frictional work

Table 5 gives the average heat produced by human body under different situations.

Table 5: Average heat produced by human body in different situation (Misra, 1986)

Nature of Work	Light work (e.g. winch driving)	Moderate work (e.g. fitting)	Hard work (e.g. Shoveling)

Metabolic Heat rate, W/m2	90	180	270

In general, the body surface area of a miner/worker can be taken as 1.8-1.9 m. We can see from Table 5 that the heat added by miner/worker is not a significant one. It can be significant only when the local working of the mine (e.g., face) is not properly ventilated.

2.3 Mine air pollutants

Broadly defined, *a* ***contaminant***, as used in ventilation and air conditioning, is any undesirable substance not normally present in air or present in an excessive amount. Liquid particulate contaminants include mists and fogs, and solid contaminants include dust, fumes, smoke, and organisms (bacteria, pollen, etc.). The most common types of air contaminants found underground are **gases** and **dust** (*Hartman 1997)*. Due to depth (geothermal gradient) and confinement, **heat** or **high temperatures** and **humidity** are also present in most underground mines (*Donoghue, 2004)*, (*Wang et al 2012).*

For this study, heat and humidity will be covered in depth; gases and dust will be covered in brief.

2.3.1 Dust

Throughout the mining and processing of minerals, the mined ore undergoes a number of operations ranging from crushing, grinding, cleaning, drying, and product sizing as it is processed into a marketable commodity. These operations are highly mechanized, and both individually and collectively these processes can generate large amounts of dust. If control technologies are inadequate, hazardous levels of respirable dust may be liberated into the work environment, potentially exposing workers (*Cecala et al 2012*). Dust has been defined as solid particles <100 μm in size that can be disseminated and carried by air in suspension (*Darling, 2011*).

The respirable dust in the mine air is measured at regular intervals by ventilation staff (MR 09). The common instruments used to measure dust concentrations are the Konimeter and the Gravimetric dust sampler (Figure 5).

Figure 5: Gravimetric Dust Sampler (Jay Colinet, 2010)

The Konimeter is the instrument used and suffers from one major disadvantage of taking a sample in one fifth of a second - hence it is not truly representative. The conventional gravimetric sampler is a good device for measuring dust because it is the instrument used for compliance measurements. This dust sampler consists of an air pump, a small cyclone that separates out the respirable size fraction of the dust cloud, and a filter to collect the respirable dust, (*Kissell, 2003)*

The Zambian Government Mines Inspectorate has set a maximum limit of 350 particles of dust per cubic centimeter (350 ppcc).

2.3.1.1 Physiological effects of dusts

McPherson 2012 explains the classification of dusts with respect to potential hazard to the health and safety of industrial workers into five categories.

1. **Toxic dusts**. These can cause chemical reactions within the respiratory system or allow toxic compounds to be absorbed into the bloodstream through the alveolar walls. They are poisonous to body tissue or to specific organs. The most hazardous include compounds of arsenic, lead, and uranium and other radioactive minerals, mercury, cadmium, selenium, manganese, tungsten, silver and nickel.

2. **Carcinogenic (cancer causing) dusts**. The cell mutations that can be caused by alpha, beta and gamma radiation from decay of the uranium series make radon daughters the most hazardous of the carcinogenic particulates. A combination of abrasion of lung tissue and surface chemical action can result in tumour formation from asbestos fibres and, to a lesser extent, freshly produced quartz particles. Exposure to arsenic dust can also cause cancers. Diesel exhaust particulates are a causative factor in lung and other types of cancer.
3. **Fibro-genic dusts:** The scouring action of many dusts causes microscopic scarring of lung tissue. If continued over long periods this can produce a fibrous growth of tissue resulting in loss of lung elasticity and a greatly reduced area for gas exchange. The silica (quartz, chert) and some silicate (asbestos, mica, talc) dusts are the most hazardous of the fibrogenic dusts and may also produce toxic and carcinogenic reactions. Welding fumes and some metalliferous ores produce fibrogenic dusts. Long and excessive exposure to coal dusts also gives rise to fibrogenic effects.
4. **Explosive dusts**. These are a concern of safety rather than health. Many organic materials, including coals other than anthracite, become explosive when finely divided at high concentrations in air. Sulphide ores and many metallic dusts are also explosive.
5. **Nuisance dusts**. All dusts can be irritating to the eyes, nose and throat and when in sufficiently high concentration may cause reduced visibility. Some dusts have no well-defined effects on health but remain in the category of a nuisance dust. These include the evaporites (halite, potash, gypsum) and limestones. The soluble salts of halite ($NaCl$) and potash (KCl) can occasionally cause skin irritations, particularly around hatbands or tightly fitting dust masks.

2.3.1.2 Methods of dust control

For an existing mine and a new mine, there are four main methods of controlling the production, concentration and hazards of airborne dust.

a) Suppression by water

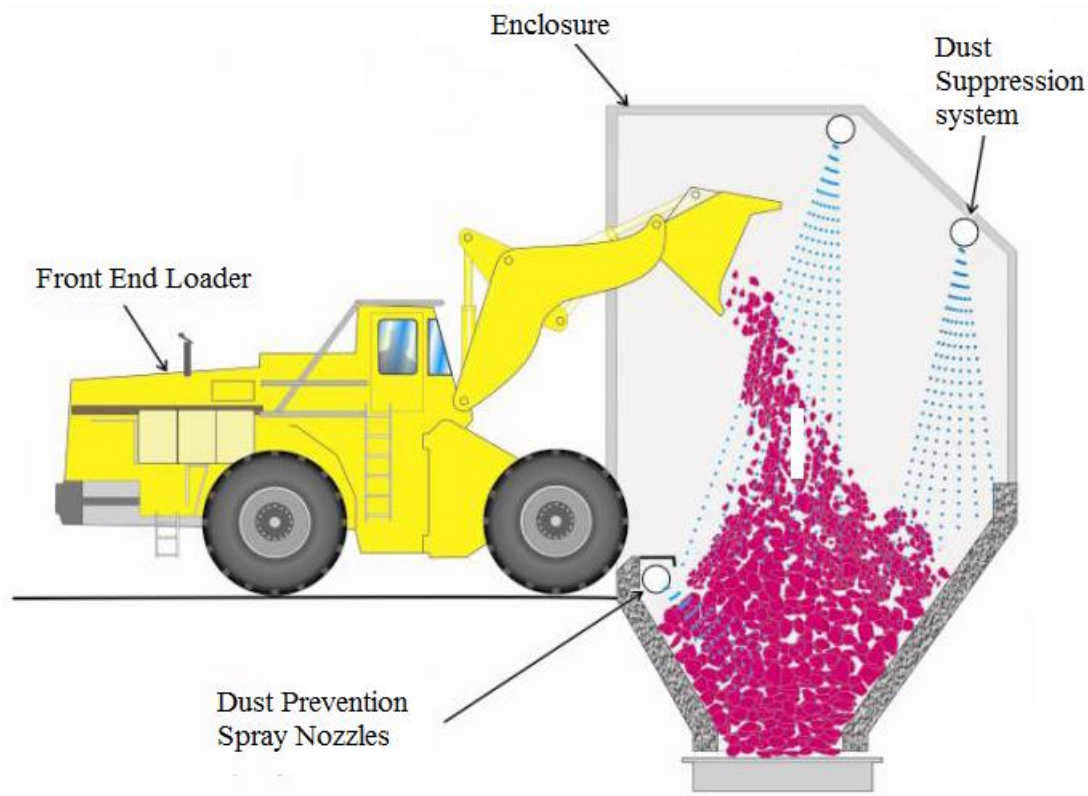

Figure 6: Typical loader dump dust control application (Schultz, 2012)

All efforts should be concentrated on the prevention of the generation and liberation of dust at source. If this is not possible efforts should be directed towards reducing the possible exposure of employees to the airborne dust through appropriate control measures (*Stanton et al 2006*). Figure 6 shows a typical loader dump dust control application in mining.

b) Filtration and scrubbing

Air filtration systems (Figure 7) remove dust of the inspirable sizes from the air circulation. Scrubbers are used in dust control at continuous miner sections in coal mines where the main dust sources are continuous miners and roof bolters.

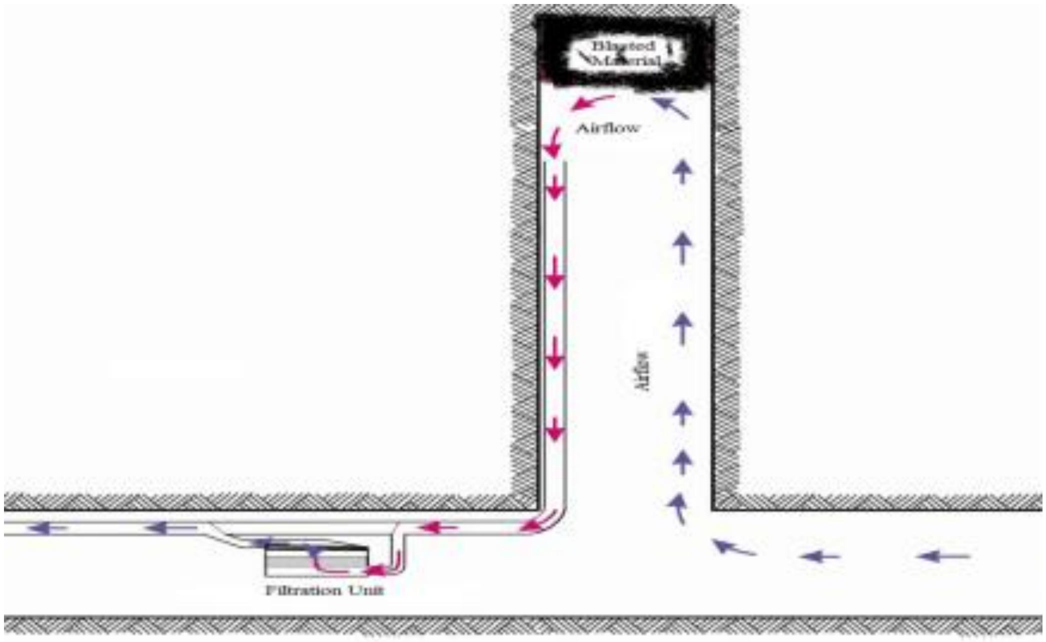

Figure 7: Filtration Unit

c) Dilution by airflow

Dilute and transport dust from mine atmosphere –Direct dust away from workers.

Mine-wide – Main fan to establish airflow on a mine-wide basis

Local – Booster fans for more direct and controlled air volume. The Figure 8 shows an example of a booster fan installed underground.

Figure 8: Booster fan used to transport and dilute dust (Chekan, 2010)

d) Isolation

Isolation consists basically of placing a barrier between the dust source and the workers. It can be applied at the source or beyond the source, at any point up to the immediate surroundings of the workers. Isolation of the source implies a barrier between the hazard source and the work environment, while isolation of the workers means, for example, a loader operator in a ventilated cabin (*Ennis DE, 1999).*

2.3.2 Mine Gases

Underground mine air is a mixture of several gases. The air found underground seldom contains the exact concentrations of gases because, as it circulates through the mine, it loses some of its oxygen and gains other gases from various sources such as the strata, blasting, and internal combustion engines (*Howard L. Hartman, Jan M. Mutmansky, Raja V. Ramani, Y. J. Wang, 2012).* Table 6 shows the various gases found in the mine environment.

Table 6: Properties of Mine Gases (Hartman 1997)

Name	Symbol	Specific Gravity (Air = 1)	Specific Weight lb/ft³ (kg/m³)	Other Physical Properties	Harmful Effects	Primary Sources	TLV-TWA, %	TLV-STEL, %	TLV-C, %	Explosive Range, %
Oxygen	O_2	1.1056	0.083 (1.33)	Odorless, colorless, tasteless	Nontoxic	Normal air	—	—	—	—
Nitrogen	N_2	0.9673	0.073 (1.17)	Odorless, colorless, tasteless	Nontoxic, simple asphyxiant	Normal air, strata	—	—	—	—
Carbon dioxide	CO_2	1.5291	0.115 (1.84)	Odorless, colorless, slight acid taste	Asphyxiant, increased respiration	Breathing, strata, fire, IC engines, blasting	0.5	3.0	—	—
Methane	CH_4	0.5545	0.042 (0.67)	Odorless, colorless, tasteless	Asphyxiant, explosive	Strata, blasting, IC engines	—	—	—	5–15
Carbon monoxide	CO	0.9672	0.073 (1.17)	Odorless, colorless, tasteless	Toxic, explosive	Fires and explosions, IC engines, oxidation	0.0025	0.04	—	12.5–74
Hydrogen sulfide	H_2S	1.1912	0.89 (1.43)	Rotten egg odor, colorless, acid taste	Toxic, explosive	Strata, strata water, blasting	0.001	0.0015	—	4–44
Sulfur dioxide	SO_2	2.2636	0.170 (2.72)	Irritating, colorless, acid taste	Toxic	Burning sulfide ore, IC engines	0.0002	0.0005	—	—
Oxides of nitrogen	NO, NO_2, N_2O	1.5895	0.119 (1.91)	Irritating odor, red-brown color, bitter taste	Toxic	Blasting, IC engines	0.0003	—	0.0005	—
Hydrogen	H_2	0.0695	0.005 (0.08)	Odorless, colorless, tasteless	Explosive	Water on a hot fire, batteries	—	—	—	4–74
Radon	Rn	7.665	0.575 (9.21)	Odorless, colorless, tasteless	Radioactive	Strata	1 WL	—	—	—

a) Oxygen, O_2

Human beings and, indeed, the vast majority of the animal kingdom are completely dependent upon the oxygen that comprises some 21 percent of fresh atmospheric air.

As muscular activity increases, so also does the rate of respiration and the volume of air exchanged at each breath. However, the percentage of oxygen that is utilized decreases at heavier rates of breathing. Table 7 indicates typical rates of oxygen consumption and production of carbon dioxide while Table 8 shows effects of oxygen depletion.

Table 7: Gas exchange during respiration (McPherson, 2012)

Activity	Breath/min	Inhalation rate	Oxygen Consumption	Carbon dioxide Produced
		Litres/s	Litres/s	Litres/s
At rest	12 - 18	0.08- 0.2	≈0.005	≈0.004
Moderate work	30	0.8 – 1.0	≈0.03	≈0.027
Vigorous work	40	≈1.6	≈0.05	≈0.05

Table 8: Effects of oxygen depletion (McPherson, 2012)

Percent Oxygen in air	Effects
19	Flame height on a flame safety lamp reduced by 50 percent
17	Noticeable increase in rate and depth of breathing- this effect will be further enhanced by an increased concentration of carbon dioxide
16	Flame lamp extinguished
15	Dizziness, increased heartbeat
13 to 9	Disorientation, fainting, nausea, headache, blue lips, coma
7	Coma, convulsions and probable death
Below 6	Fatal.

Oxygen deficiency implies an increased concentration of one or more other gases (McPherson, 2012)

The following are gases that maybe found in underground mines:

b) Nitrogen, N_2

Nitrogen constitutes approximately 78 percent of air and is, therefore, the most abundant gas in a ventilated system. It is fairly inert and occurs occasionally as a strata gas, usually mixed with other gases such as methane and carbon dioxide. *(McPherson, 2012)*

c) Methane, CH_4

Methane is colourless, odourless, and tasteless. It is lighter than air – Specific Gravity 0.55. It will migrate towards the roof and may accumulate in cavities or form layers. It is explosive in air in the range 5% to 15%. It is also an asphyxiant as it displaces oxygen. Methane occurs naturally and is present in most coal seams. It is released as the coal is mined and also when coal is heated. It may release under pressure as an outburst. Methane can be detected by readily available portable gas detectors *(Safe work Australia 2011).*

d) Carbon dioxide, CO_2

Carbon dioxide appears in subsurface openings from a variety of sources including strata emissions, oxidation of carbonaceous materials, internal combustion engines, blasting, fires, explosions and respiration. Physiological effects of carbon dioxide are shown in Table 9.

Table 9: Physiological effects of Carbon Dioxide (McPherson, 2012)

Percent Carbon dioxide in air	Effects
0.037- 0.038[2]	None, normal concentration of Carbon dioxide in air
0.5	Lung ventilation increase by 5 percent
2	Lung ventilation increase by 50 percent
3	Lung ventilation doubled, panting on exertion
5 to 10	Violent panting leading to fatigue from exhaustion, headache
10 to 15	Intolerable panting, severe headache, rapid exhaustion and collapse

Stagnant mixtures of air in sealed off areas often have an increased concentration of carbon dioxide and decreased oxygen content. Such mixtures are sometimes called blackdamp. When carbon dioxide has an increased concentration a number of effects can be observed. The Table below lists some of the physiological effects of carbon dioxide concentration.

e) Carbon monoxide, CO

The high toxicity of carbon monoxide coupled with its lack of smell, taste or colour make this one of the most dangerous and insidious of mine gases. It has a density very close to that of air and mixes readily into an airstream unless it has been heated by involvement in a fire, in which case it may layer with smoke along the roof.

Carbon monoxide is a product of the incomplete combustion of carbonaceous material. Although colourless, it has the traditional name of *whitedamp*. The great majority of fires and explosions in mines produce carbon monoxide.

Physiological reactions to carbon monoxide depend upon the concentration of the gas, the time of exposure and the rate of lung ventilation (Table 10). Although variations exist between individuals, the following list provides a guideline of the progressive symptoms.

Table 10: Guidelines of the progressive symptoms due to carbon monoxide exposure (McPherson, 2012)

Blood Saturation (Percent CO. Hb)	Symptoms
5 - 10	Possible slight loss of concentration
10 - 20	Sensation of tightness across forehead, slight headache
20 - 30	Throbbing headache, judgment impaired
30 - 40	Severe headache, dizziness, disorientation, dimmed vision, nausea, possible collapse
40 - 60	Increased probability of collapse, rise in rates of pulse and respiration, convulsions
60 - 70	Coma, depressed pulse and respiration, possible death
70 - 80	Fatal

f) Sulphur dioxide, SO_2

This is another highly toxic gas but one which, fortunately, can be detected at very low Concentrations both by its acidic taste and the intense burning sensation it causes to the eyes and respiratory tracts. The latter is a result of the high solubility of the gas in water to form sulphurous acid. Physiological effects of SO_2 are shown in the Table 11.

$H_2O + SO_2 = H_2SO_3$, this, in turn, can oxidize to sulphuric acid, H_2SO_4.

Table 11: Physiological reactions to Sulphur dioxide in parts per million (McPherson, 2012)

Concentration of Sulphur dioxide (ppm)	Effects
1	Acidic taste
3	Detectable by odour
20	Irritation of eyes and respiratory system
50	Severe burning sensation in eyes, nose and throat
400	Immediately dangerous to life

g) Oxides of nitrogen, NOx

The Table 12 shows Progressive symptoms of nitrogen oxide exposure.

Table 12: Progressive symptoms of nitrogen oxide exposure (McPherson, 2012)

Concentration of Nitrogen dioxide (ppm)	Effects
40	May be detected by smell
60	Minor throat irritation
100	Coughing may commence
150	Severe discomfort, may cause pneumonia later
200	Likely to be fatal

Three oxides of nitrogen are Nitric oxide, NO, nitrous oxide, N_2O, and nitrogen dioxide, NO_2, are formed in internal combustion engines and by blasting. The proportion of nitrous oxide is likely to be small. Furthermore, nitric oxide converts rapidly to nitrogen dioxide in the presence of air and water vapour.

h) Hydrogen sulphide, H_2S

The presence of this highly toxic gas is readily detected by its characteristic smell of bad eggs. This has given rise to the colloquial name ***stinkdamp***. Unfortunately, hydrogen sulphide has a narcotic effect on the nervous system including paralysis of the olfactory nerves. Hence, after a short exposure, the sense of smell can no longer be relied upon.

Physiological effects of hydrogen sulphide may be listed as shown in Table 13:

Table 13: Physiological effects of hydrogen sulphide (McPherson, 2012)

Concentration of hydrogen sulphide (ppm)	Effects
0.1 to 1	Detectable by smell
5	Beginning of toxicity
50 to 100	Slight irritation top eyes and respiratory tract, headache, loss of odour after 15 minutes
200	Intensified irritation of nose and throat.
500	Serious inflammatory of eyes, nasal secretions, coughing, palpitations, fainting
600	Chest pains due to corrosion of respiratory system, may be fatal
700	Depression, coma, probable death
1000	Paralysis of respiratory system, very rapid death

Hydrogen sulphide is produced by acidic action or the effects of heating on sulphide ores. It is formed naturally by bacterial or chemical decomposition of organic compounds and may often be detected close to stagnant pools of water in underground mines. Hydrogen sulphide may occur in natural gas or petroleum reserves and migrate through the strata in a weakly acidic water solution. It can also be generated in gob fires. (*McPherson, 2012)*

A victim who recovers from hydrogen sulphide poisoning may be left with longer term conjunctivitis and bronchitis.

i) Hydrogen, H_2

Although non-toxic, hydrogen is the most explosive of all the mine gases. It burns with a blue flame and has the wide flammable range of 4 to 74.2 percent in air. Hydrogen can be ignited at a temperature as low as 580 °C and with ignition energy about half of that required by methane.

Hydrogen occasionally appears as a strata gas and may be present in afterdamp at about the same concentrations as carbon monoxide. The action of water on hot coals can produce hydrogen as a constituent of water gas. Dangerous accumulations of hydrogen may occur at locations where battery charging is in progress. Hydrogen has a density only some 0.07 that of air. It will, therefore, tend to rise to the roof. Battery charging stations should be located in intake air with a duct or opening at roof level that connects into a return airway *(McPherson, 2012)*

j) Radon, R_n

This chemically inert gas is one of the elements formed during radioactive disintegration of the uranium series. Although its presence is most serious in uranium mines, it may be found in many other types of underground openings. Indeed, seepages of radon from the ground into the basements of surface buildings have been known to create a serious health hazard.

Radon emanates from the rock matrix or from ground water that has passed over radioactive minerals. It has a half-life of 3.825 days and emits alpha radiation. The immediate products of the radioactive decay of radon are minute solid particles known as the radon daughters. These adhere to the surfaces of dust particles and emit alpha, beta and some gamma radiation (*McPherson, 2012*)

2.4 Mine Ventilation system

A well designed and properly implemented ventilation system will provide beneficial physiological and psychological side effects that enhance employee safety, comfort, health, and morale. In planning a ventilation system, the quantity of air necessary to circulate to meet all health and safety standards must be decided at the outset. Once the quantity required has been fixed, the correct size of shafts, number of airways, and fans can be determined. Figure 9 shows a typical example of a ventilation system.

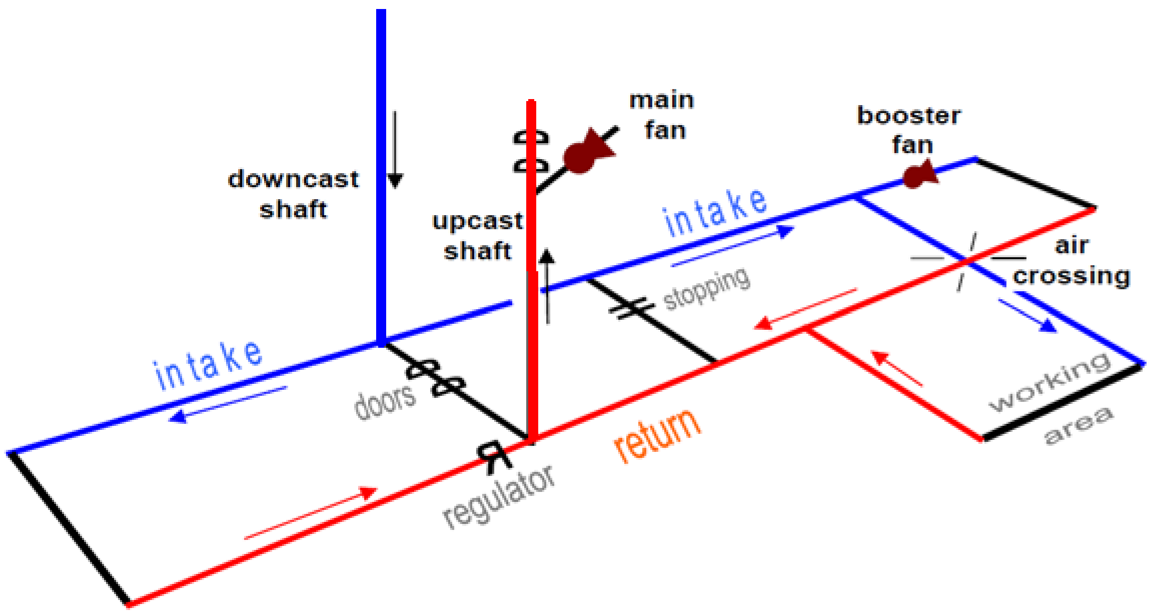

Figure 9: Typical elements of a main ventilation system (McPherson 2012)

Fresh air enters the system through one or more downcast shafts, drifts (slopes, adits), or other connections from surface. The air flows along intake airways to the working areas or places where the majority of pollutants are added to the air. These include dust and a combination of many other potential hazards including toxic or flammable gases, heat, humidity, and radiation. The contaminated air passes back through the system along return airways. In most cases, the concentration of contaminants is not allowed to exceed mandatory threshold limits imposed by law and safe for the entry of personnel into all parts of the ventilation system including return airways.

The intake and return airways are often referred to simply as intakes and returns respectively. The return air eventually passes back to the surface via one or more upcast shafts, or through inclined or level drifts *(McPherson 2012).*

In summary pollutants enter a ventilation airstream from a variety of sources. The aim of any ventilation system is to dilute any pollutants to a safe level and then remove them. The range of pollutants that can be released into a ventilation system depends on the situation under consideration. *(Peter Darling, 2011)*

2.4.1 Secondary Ventilation

This type of ventilation employs auxiliary fans to ventilate development ends. The system normally used includes exhaust, forcing and overlap. The choice between forcing and exhausting arrangements depends mainly upon the pollutants of greatest concern, dust, gases or heat.

2.4.1.1 Exhaust-Overlap System

A small fan and small duct is used to force air to the development end and an exhaust column of bigger diameter connected to a big exhaust fan in order to remove used air from the development end. An overlap of 10 to 15 metres must be maintained as shown in Figure 10 as mining advances to prevent re-circulation of used air. The quantity of air supplied to the face must not exceed that exhausted otherwise the system will not be able to contain larger quantity of air.

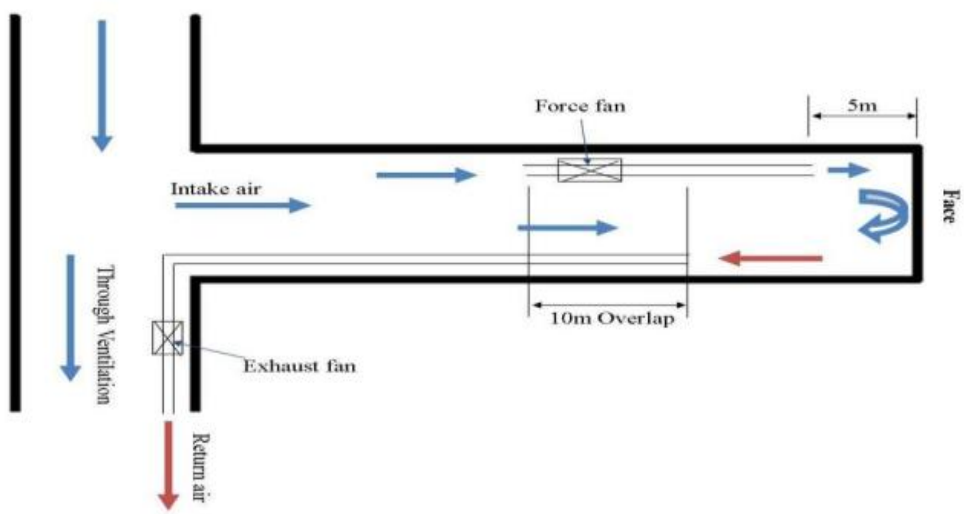

Figure 10: Exhaust-Overlap System (Burrows J. et al, 1982)

2.4.1.2 Forcing System

This system installs a fan to draw clean air and deliver it to the working end. The intake is at least 10 metres upstream in the nearest through ventilation and 5 metres is maintained between the tail and the force column as shown in Figure 11 in order to maintain sufficient air at the tail.

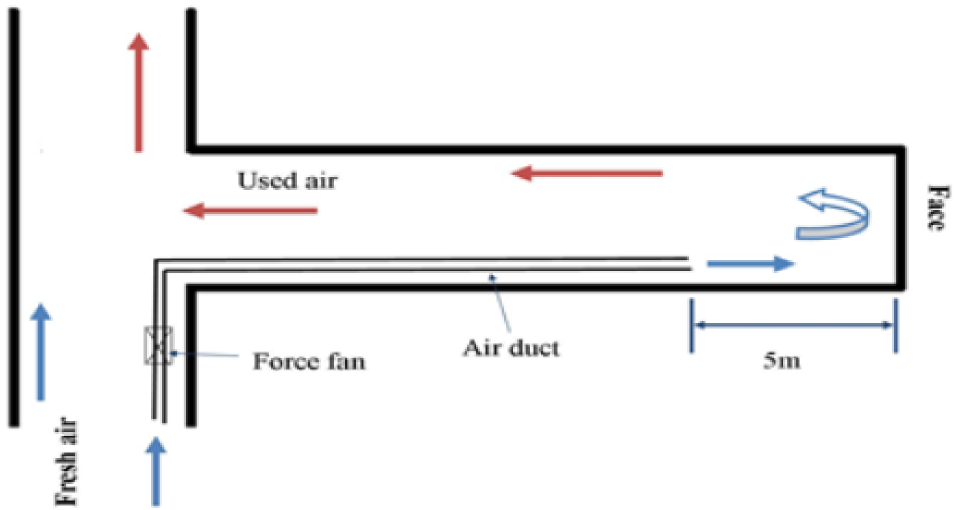

Figure 11: Force ventilation system (Burrows J. et Al, 1982)

2.4.1.3 Ventilation of Twin Ends

In this system (Figure 12), one of the haulages is used as a return and the other as the intake.

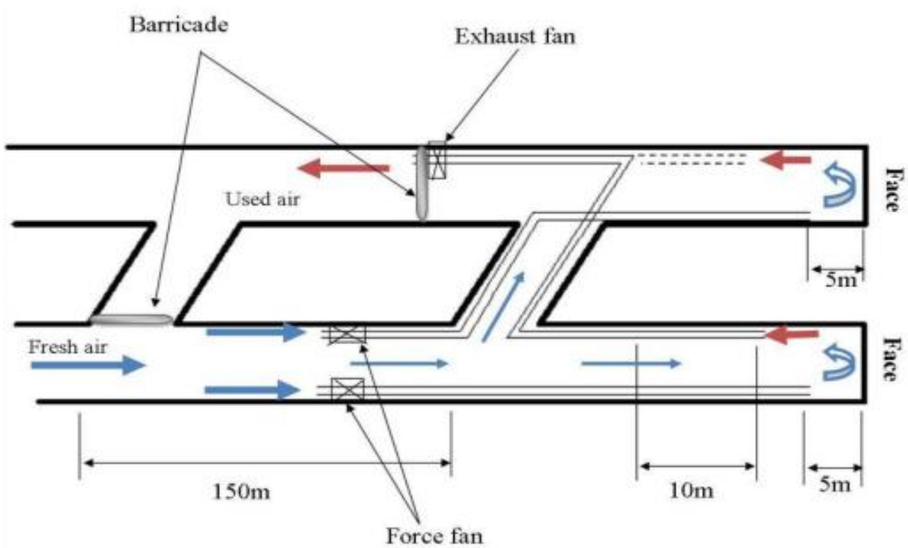

Figure 12: Ventilation of twin development ends (Burrows J. et Al, 1982)

The distance between the two haulages is usually 15 to 30 metres with connecting cross cuts at intervals of 150m to 200m. The intake air haulage is used for access, transporting material and ore while the other haulage is used as the return airway and drainage is usually unequipped

2.4.1.4 Atkinson's Equation

Dynamically, mine ventilation systems are treated almost exclusively as systems of incompressible fluid flow and are described most often through Atkinson's equation, commonly given by: $\Delta p = [\frac{KO(L+Le)}{A^3}]Q^2$……………………………………………………………………(12)

Where:

$\Delta\text{p} = \text{Pressure Difference}$

$\text{k} = \text{Friction factor, kg/m}^3$

$\text{O} = \text{Perimeter, m}$

$\text{L} = \text{Length, m}$

$\text{Le} = \text{Equivalent length to accounbt for shock losses, m}$

$\text{A} = \text{Cross} - \text{sectional area, m}^2$

$\text{Q} = \text{Volumetric flow rate, m}^3\text{/s}$

The parameters in the brackets of the Atkinson's equation above are generally jointly referred to as resistance, R, with units of Ns^2/m^8 (Hartman et al. 1997). Airway resistance (R) is an important concept in ventilation engineering. The square law states

$\Delta p = [R]Q^2$………………………………………………………………………… (13)

This shows the resistance to be a constant of proportionality between the frictional pressure drop, Δp and the square of the airflow, Q, at a specified air density. The parabolic form of the plot *on a p*, Q graph is known as the airway resistance curve (Figure 13).

The cost of passing any airflow through an airway varies directly with the resistance of that airway. This can be determined by evaluating the air power consumed in an airway using air power

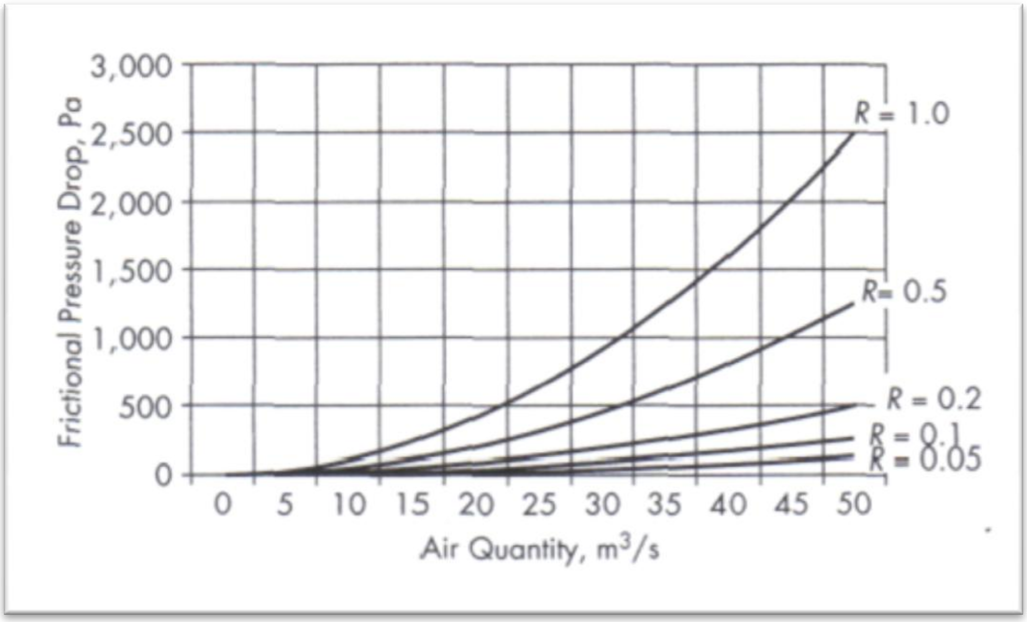

Figure 13: Airway resistance curve (McPherson 2012)

The volumetric flow rate is generally calculated using:

$$Q = V.A \quad (14)$$

Where: $Q = Volumetric\ flow\ rate, m^3/s$

$V = Velocity, m/s$

$A = Cross\ sectional\ area, m^2$

Additionally, in mine ventilation systems, the velocity of the air flowing through the airways is generally measured using vane anemometers and the velocity of air flowing through the fan inlet or outlet duct is calculated from velocity pressure measurements using:

$$Pv = \frac{\rho V^2}{2} \quad (15)$$

Where: $Pv = Velocity\ Pressure, Pa$

$\rho = Fluid\ Density\ kg/m^3$

$V = Velocity, m/s$

Based on Atkinson's equation, resistance is directly proportional to airway length and inversely proportional to airway cross-sectional area. Over 70 % of typical mine resistance is due to friction between air and rock that forms the airways.

As mines develop, the airway lengths and therefore the resistance increase, which increases the pressure necessary to produce sufficient air quantity. Additionally, it is typical for coal mines to develop multiple parallel airways including intake, neutral, and return airways. Most coal mines have a minimum of three parallel airways, one intake, one neutral, and one return; in the main development sections, multiple intake, neutral, and return airways are often used. However, deep coal mines, such as those in the western United States, experience high rock stresses. To manage the stresses and prevent ground failure, these mines often use only two parallel airways and the airways tend to have small cross-sectional areas. Because of a limited number of airways and airways with small areas, deep mines have high resistance. High resistance in extensive and deep mines is one factor that contributes to the need for substantial fan pressure to deliver adequate air quantity.

In addition to airway length and area, resistance is also dependent on the friction factor; *K*. Friction factors are related to the roughness of an airway and in practice they are determined experimentally based on pressure and quantity measurements. *(McPherson 1993)*

2.5 Air quantity determination

The amount of air needed to ventilation a given section or an underground mine can be determined using the following methods:

2.5.1 Heat Loads determination,

To determine the overall mine heat load the contribution from all mine heat sources must be taken into account. (Maurya. et al (2015). The sources of heat as earlier explained are from geothermal gradient, Auto-compression, mechanized equipment (electrical equipment and diesel equipment), Explosives and blasting, Mechanical processes and lights, Underground water and man (metabolism). Quantification of the heat emitted into a mine or section of a mine is required in order to assess the airflow needed to remove that heat. Therefore the heat energy (q) from all sources is summed up to get **total heat load.**

The heat energy content of air, defined in terms of kilojoules of heat associated with each kilogram of dry air, is known as **Sigma Heat.**

The stages of determining the airflow required removing heat from a mine or section of a mine is as follows:

(a) Evaluate the sigma heat of the air at inlet, S_1, using equations or charts

(b) Evaluate the highest value of sigma heat, S_2, that can be accepted in the air leaving the mine or section of the mine. This threshold limit value may be specified in terms of one of the indices of heat stress or simply as a maximum acceptable (cut-off) value of wet bulb temperature.

(c) Estimate the total heat flux, q_{12} (kW), into the air from all sources between inlet and outlet. This may involve simulation studies for additions of strata heat. *(McPherson 2012)*

The required airflow, Q, is then given as:

$$Q = q_{12} / (\rho \times (S_2 - S_1)) \ (m^3/s) \quad \ldots\ldots (16)$$

Where:

ρ = mean density of the air (kg/m^3)

The factor $\rho(S_2 - S_1)$ is sometimes known as the Heat Removal Capacity (HRC) of a given airflow, Q.

2.5.2 Air – tonnage ratio (ATR)

The second method of determining airflow requirement is the Air – Tonnage Ratio. For the purpose of ventilation planning, the factor of air quantity per ton broken has considerable significance. Based on experience on many mines in which rock temperatures where high, *Lambrecht and Barcza (1958)* devised an empirical relationship which has proved its worth over the years. Recent advances in use of computers in heat- flow calculations have resulted in the introduction of quantitative methods which largely confirm the accuracy of the empirical air – tonnage ratio.

Air requirement for a mine defined by the production rate, this rate in addition to the rock and ore characteristics depends on the type and size of mining equipment used on average the flow requirement is about 2.985 or 3 tonnes of air per ton of ore extracted *(Bandopadhyay & Ganguli, 2004).* Table 14 shows air quantity and tonnages for different mines from 1940 to 1971.

Table 14: Air Quantities and tonnages 1940-71 (Burrows, 1974)

	Total Air	Air/man	Air/Tonne	
Year	Quantity m^3/s	$m^3/s/man$	$m^3/s/1000t$	No. of Mines
1940	8348	0.0314	1.27	41
1950	11145	0.049	1.87	46
1960	17158	0.059	2.18	52
1970	24307	0.085	2.91	46
1971	24548	0.087	**2.985 (3)**	44

2.5.3 Widely Accepted charts.

The Figure 14 shows a relationship between airflow and production rate for 42 mines across Africa and Australia.

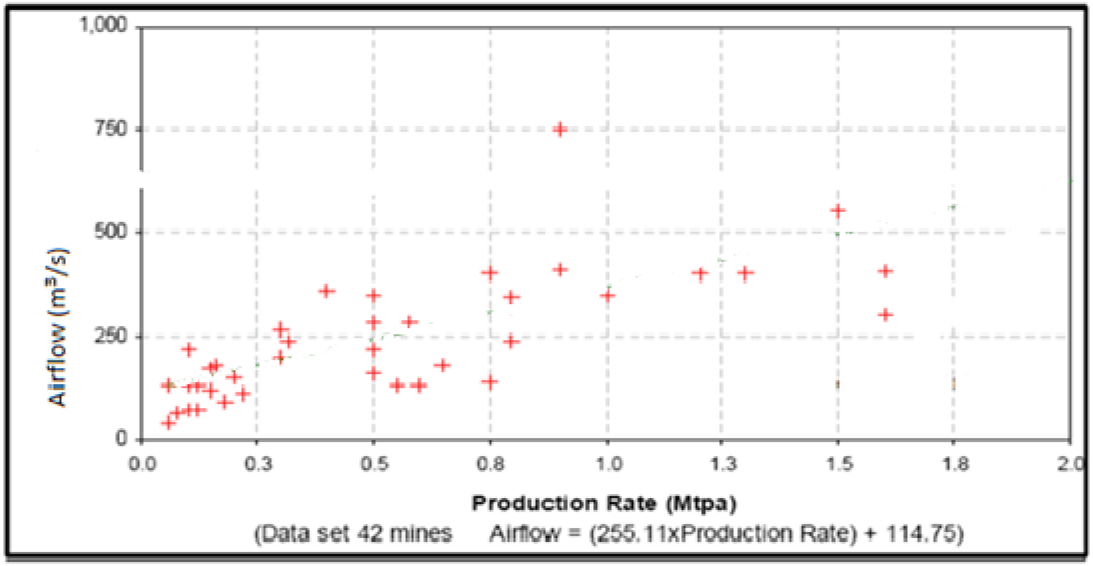

Figure 14: Airflow - Production rate graph

CHAPTER THREE: RESEARCH METHODOLOGY

3.1 Introduction

Various techniques were used to gather information for the study as summarised in Figure 15. The sources of information included detailed literature review and field data collection from the mine in particular from mine planning and mine ventilation offices. Structured interviews were also used to obtain specialized data from various sections of the mine.

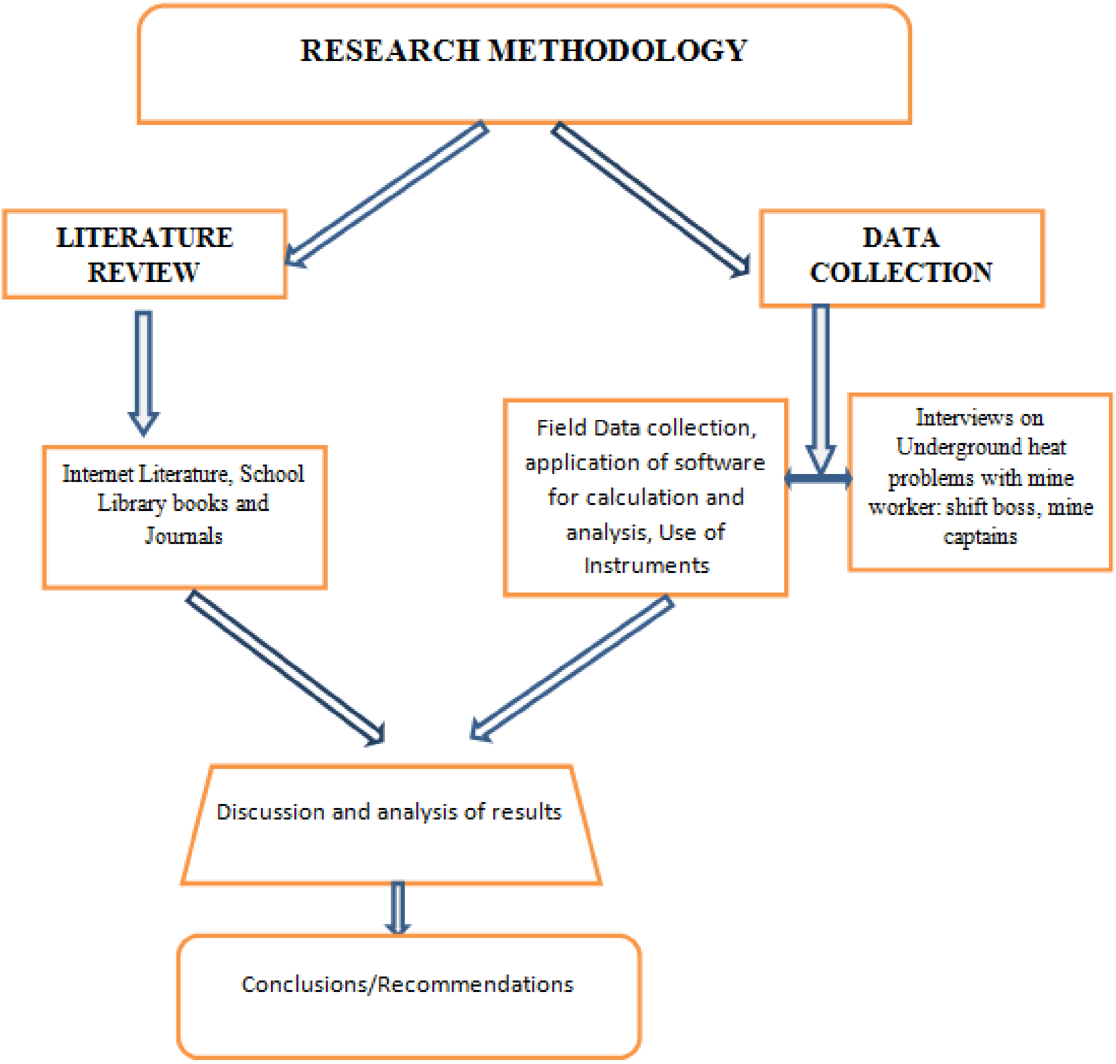

Figure 15: Research methodology summary

3.2 Literature review

Extensive literature review was done in order to understand the subject at hand and look at different ways of heat load determination. Literature review was also done for the purpose of establishing knowledge gaps in the study.

The main objective throughout the review stage was to identify likely sources of heat and other air pollutants in underground mines relevant in the calculation of air quantities required for optimal ventilation to take place. The review of material on refrigeration in deep mines gave insights on the when and how it is employed.

2.5.4 Sources of Information

Journals

Journal articles with up-to-date information offered relatively concise, up-to-date information for research.

Text books

Various text books were useful in literature review as they are intended for teaching and offered a good starting point to the study of ventilation because of more detailed explanations and illustration.

Conference proceedings

Conference proceedings were useful in providing the latest information on the topic of ventilation and air quantity determination.

Theses and dissertations

Theses and dissertations were generally available from the library shelf or through inter-library loan.

3.3 Field Data Collection

Interviews

Structural interviews were conducted prior to the main study. The interviews were aimed at obtaining preliminary data that gave insight to the challenges of heat being experienced at Mindola Sub Vertical shaft. The interviews were conducted with mining officials that frequently work underground like the PICs (person in charge), section bosses and mine captains.

2.5.5 Field Measurement

Gathering of data from the field was through measuring parameters related to air quality in an established systematic manner to aid in answering relevant questions. The goal for field data collection was to capture quality evidence that translated to rich data analysis and allowed the building of a convincing and credible answer to questions of heat challenges.

2.5.5.1 Instruments Used

Air Flow Measurement

The following Air flow measuring instruments were used in observing the direction and magnitude of the flow:

The Vane Anemometer

The velocities of air in airways were measured with an instrument called the **Vane Anemometer** (Figure 16) in conjunction with the stop watch.

Figure 16: Vane Anemometer

Suitable measuring stations were selected and the traversing was done in one plane and at right angles to the air flow while the stop watch was set to count for a stated period of traversing. The selected station was regular, reasonably straight and unobstructed. Where bends were present, measurements where made upstream of them.

In order to compute air volumes, the anemometer reading was initially divided by the time taken to traverse the airway to get the velocity and then multiplied by cross section area of the mine air way.

Calculation of airflow Quantity

The quantities of air flowing through different airways and sections were obtained by obtaining the product of the cross sectional areas in different airways and the air velocities measured. The quantity of airflow is thus given by the formula:

$$Q = \text{Velocity of Air} \times \text{Cross sectional Area}$$

$$Q = V \times A$$

Where Q = quantity of air in m^3/s

V = Velocity of air in m/s and

A = area of cross section in m^2

Measuring Tape

The tape (Figure 17) was used to measure the distances and dimensions. From these areas were calculated. Areas of airways were calculated according to their different shapes using measuring tape. The areas of regular cross sections were calculated from measurements of the width and height.

Figure 17: Measuring Tape

Stop Watch

The stop watch (Figure 18) was used in conjunction with the vane anemometer in determining the air velocity in an airway during traversing.

Figure 18: Stop Watch

The smoke tube

The smoke tube or an instrument commonly known as dust disperser or simply a puffer, which is a small flexible container filled with shinny aluminums powder, with the needle size opening on the lid shown in Figure 19 was used to determine the direction of flow of air. The container is squeezed in order to release the shiny aluminum powder into the air and let it be carried by the airflow in its direction.

Figure 19: Smoke tube

The smoke tube was also used to measure velocities of less than one meter per second. This was done by Timing the speed of a visible cloud of smoke or dust over a known distance. A portion of an airway of regular cross sectional area and remote from bends and obstructions was selected and two points of at least three metres apart marked on the side walls. An assistant provided with dust disperser aligned the beam of lamp at right angles to the direction of airflow at the upstream mark and then released a cloud of dust in the center of the airway at arm's length upstream of the mark.

The observer stationed opposite the downstream mark timed the dust cloud enters the beam of the assistant's lamp and stopped the watch as the dust cloud enters the beam of the observer’s lamp aligned at right angles to the downstream mark. This procedure was repeated for at least three times to get accurate data.

The Whirling Hygrometer

The whirling hygrometer which consists of wet and dry bulb thermometers (Figure 20) was used to determine the wet-and dry bulb temperatures of air. The wet bulb which is covered in thin cotton sleeve called ***muslin film*** was kept wet by adding water into a small container attached to the hygrometer located near the web bulb thermometer. Measurement of air temperatures was done by facing the air stream and holding the instrument at arm’s length in front of the body. Whirling of the hygrometer was performed at a rate approximately three revolutions per second to give an air velocity past the bulbs of about 3m/s. The two temperatures were read as rapidly as possible after whirling the hygrometer for 30 seconds starting with wet bulb temperature because it rises when whirling ceases.

Figure 20: Whirling hygrometer

CHAPTER FOUR: VENTILATION SYSTEM AT MINDOLA SV

4.1 Existing infrastructure at MSV

4.1.1 Primary Ventilation Systems

The ultimate goal of Mindola Sub-Vertical shaft is to provide adequate ventilation conditions in compliance with the Mine Regulation IX that demands among other things a temperature of not more than 31.0 ^{o}C wet bulb.

4.1.2 Up-Cast Ventilation shafts:

The shaft is served by three (3) principal main Up-cast fans namely, V10, V5 and V9 serving the southern, central and the northern part of the mine respectively. The total up-cast quantity measured is 657.0m^3/s at 1.2 kg/m^3, the total mass flow is 778kg/s at static pressure of 11.9kPa.

4.1.2.1 V 9 Upcast shaft

This is a single inlet, centrifugal fan of a backward curved aerofoil bladed impeller with a duty of 5.4kPa static pressure handling a quantity of 138.0m^3/s representing 21% of the total upcast quantity. The drive is direct coupled with nominal power of 1500kW, sitting on the 5.5m diameter shaft. This saves the northern part of the mine running through various levels and internal raises up to 5045L.

4.1.2.2 V5 Upcast shaft

This is an axial flow fan with a duty of 2.0kPa static pressure handling a quantity of 236.5m^3/s representing 36% of the total upcast quantity. The nominal power of 1500kW, sitting on the 4.9m diameter shaft. The shaft runs through center of ore body strike length and saves the shaft complex areas.V5 extends down to the shaft bottom (5660L).

4.1.2.3 V10 Upcast shaft

These are two fans with a single inlet, centrifugal fan of a backward curved aerofoil bladed impeller with a duty of 4.5kPa static pressure handling air quantity of 283.0m^3/s representing 43% of the total upcast quantity. The drive is direct coupled with nominal power of 1500kW, sitting on the 4.9m diameter shaft. The shaft extends up to 4370L on the southern part of the mine. Though there is no direct link to the global network circuit at 4440L. Arrangement of the V10 main upcast fan is shown in Figure 21 while Figure 22 shows the longitudinal projection of Mindola SV shaft.

Figure 21: V10 Main Upcast Fan

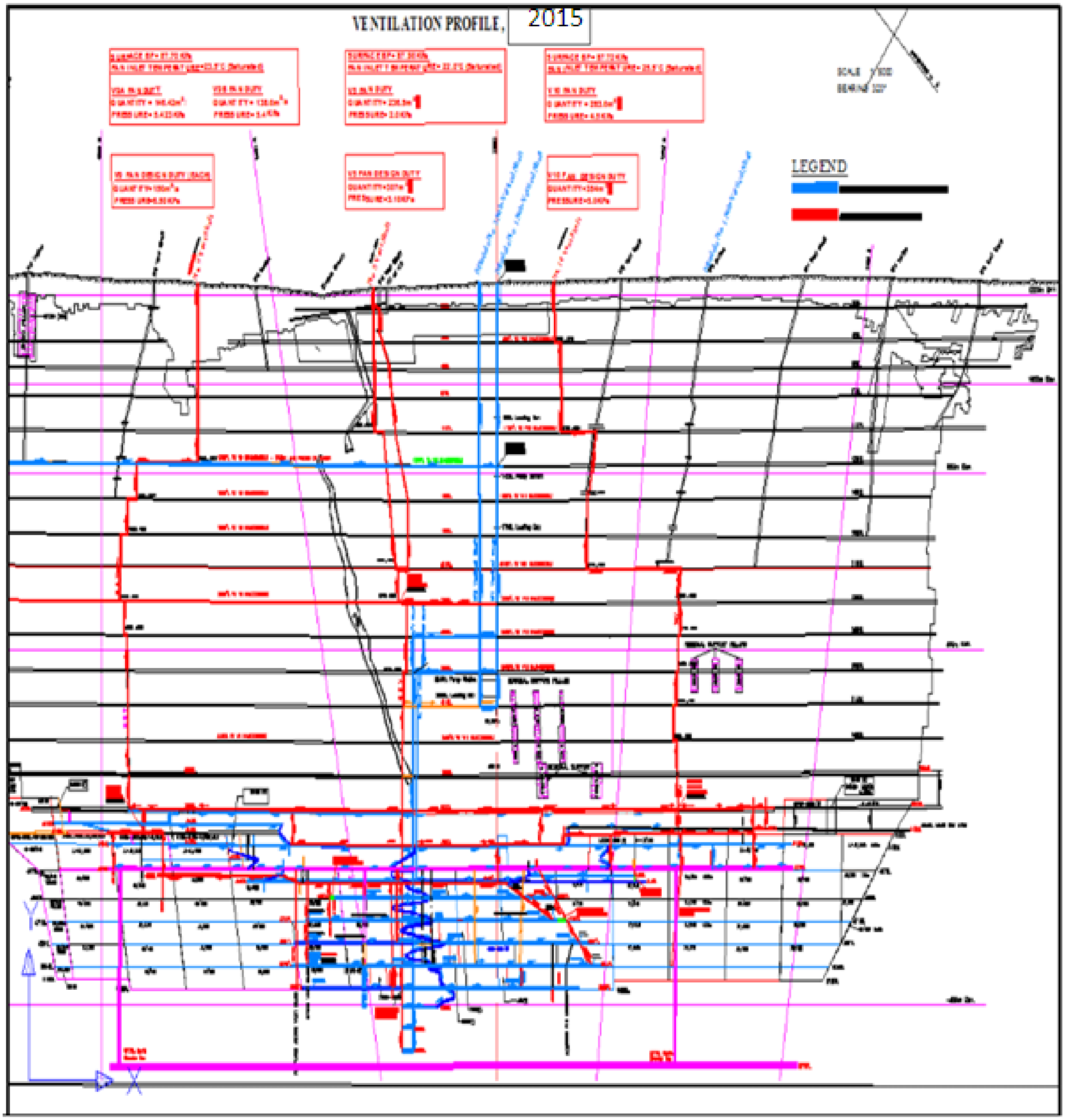

Figure 22: Vertical Longitudinal Projection (Mine Planning, Mindola Sub-Vertical, 2015)

Table 15 shows the pressure and volumetric measurement on the main upcast fans at MSV shaft.

Table 15: Measurements on Main Upcast Fans

Fan	Static Pressure (kPa)		Air Volume Measurement (m^3/s)			
	Design Duty	Actual	Design Duty	Actual	U/G point	Level
V10	5	4.5	354	283	244.8	3920
V9	6.5	5.4	150	138	134.4	3920
V5	3.1	2	307	224.5	224.7	2380
		Total	**811**	**645.5**	**603.9**	

4.1.3 Downcast shaft

MSV has two surface downcasts namely number 1 and 2 shafts. The intake air splits at 2630L and 2880L, with former further splitting through three air haulages (trunks) handling a total $369m^3/s$ on its available cross section area of $51.7m^2$. The remainder of $255m^3/s$ goes through the cross section area of $24.5m^2$ at 2880L main crosscut to Sub-Vertical Shaft (SV). From 2880L, the SV shaft is mined at 7.1m diameter with available area for ventilation estimated at 70% i.e. $27.7m^2$. Note also that the sand raises are being slipped from 1.8m to 5.0m diameter from 2880L to 4440L and will give a total area of $39.3m^2$. Figure 23 shows the down cast system.

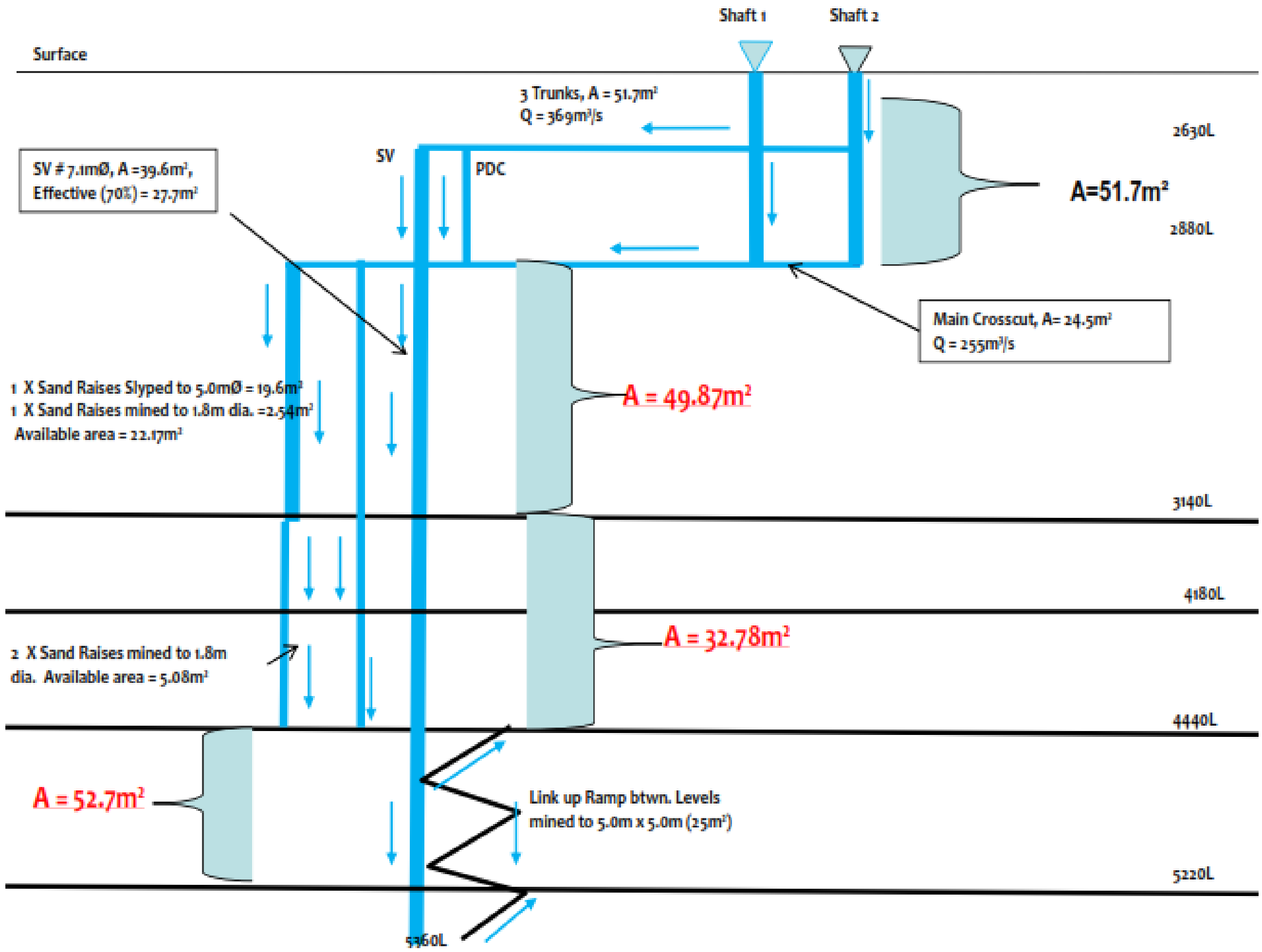

Figure 23: Current ventilation downcast systems (Mine Planning MSV)

4.1.4 Fans in use

4.1.4.1 Main Fans

Both axial and centrifugal fans are used to provide air circulation in mine ventilation system with efficiencies of over 70% being achieved. The selection between axial and centrifugal flow for main fans depend on cost, size, pressure, efficiency, robustness and performance variations. In mines where fan failure may result in dangerous gas emissions additional fan capacity is installed to ensure continuity of ventilation. Where it is not so critical and with twin fan installation about two thirds of mine air flows are used for low pressure applications and centrifugal fans are high pressure system. Either selection is suitable for the intermediate pressure. When robustness is

required such as exhaust with air velocities above the critical range and water droplets are carried up and out of the system, a centrifugal fan has more reliable selection.
The critical air velocities range from 7.5m/s to 13m/s where the water droplets may stay in suspension depending on their size. With this range the amount of suspended water can build up and increase the system pressure until fan stalls.

This is region where some of the air recirculation occurs around the blades and the fan operation becomes unstable. Though non desirable for any fan type, the possibility of centrifugal fan blade failure significantly occurs less than axial blade failure in the region of flow fluctuations.
It is rare that a main fan is required to operate at the same duty point over the life of the mine and effective method of varying fan performance is desirable.

Although variable speed results in most efficient operation for both axial and centrifugal fans, the cost particularly for large fans is high. The performance of an axial flow fan can be varied by adjusting the blade angle and this can be carried out either when the fan is stopped or at a significant higher cost when it is rotating. By imparting a swirl to the air entering the fan casing variable inlet vanes, the performance of a centrifugal fan can be varied while it is running. The efficiency of a centrifugal fan away from its design point falls off more rapidly than that of an axial flow fan and if a high performance is required over a wide range of operating points and the pressure is suitable, the axial flow fan can be selected.
The main fans are generally installed on surface and this has the advantage in that no heat is added to the air underground or possibility of recirculation of foul air, easy accessibility in case of fires and easy to service *(Howes M.J. 2010).*

4.1.4.2 Booster fans

Booster fans (Figure 24) are widely used in metal mines but are usually prohibited in coalmines. In metal mines booster fans are used either to divert air flow or give additional air movement power to a place where it is needed. Booster fans are used in areas which are widely scattered and cannot be served by the main fan. Booster fans will be required for air distribution to remote parts of the mine or a large fan will be required to pressurize all areas. If large main fan is used, restrictions will then be needed to balance the pressure in non-remote areas. The second solution may be unnecessary expense when booster fans are applicable. Below is a picture of booster fan.

Figure 24: An Underground booster fan

4.1.4.3 Auxiliary fans

These are relatively smaller fans used in directing air to individual faces or smaller areas. Where dead end faces are to be ventilated, the air is delivered to the faces with ducts connected to the fan.

In case of large openings, auxiliary fans are often hanged in main air streams to divert the air into these openings with little or no duct connected to the fan. These are jet fans.

4.1.5 Fan laws

There are certain laws which are applicable to fans making it possible to calculate characteristic curves for a fan at different speed and air density. For varying air speed but with constant density:

1. Quantity varies directly as speed
2. Pressure varies as the square of the speed
3. Power varies as the cube of the speed while
4. Efficiency remains constant

For varying density but with constant speed, quantity remains constant. Pressure varies directly as the density. Power varies directly as the density. Efficiency remains constant. It is important to know the speed of the fan during fan testing. Any change in the fan speed will result in a very different performance characteristic. Hence when the curves of the fan are plotted from the fan test, results are varied only at the speed which the fan was running during the test. If the fan is to be used at the speed other than that which is specified in the curve, then the curves have to be corrected to the relevant speed. The fan performance curves can be altered using fan speed *(Howes M.J. 2010).*

4.1.6 Air distribution and control

The distribution of air throughout the mine in right amounts is as important as the total quantity circulated. If fresh air is not taken down in its clean state to areas where it is needed in the right amounts, it is of little value.

The distribution system should deliver the minimum quantity of fresh air necessary to maintain a health working environment direct to the working areas and return airways.
Admittedly the ideal system is rarely feasible; it requires a close cooperation and commitment between production, engineering and ventilation departments.

Controlled split methods of ventilation is vital for mines with multiple working sections as it ensures improved efficiency, low ventilation costs and minimize the accumulation of contaminants. Individual air currents ventilate each active section from where they are directed to primary returns. Figure 25 shows air distribution system at Mindola SV shaft.

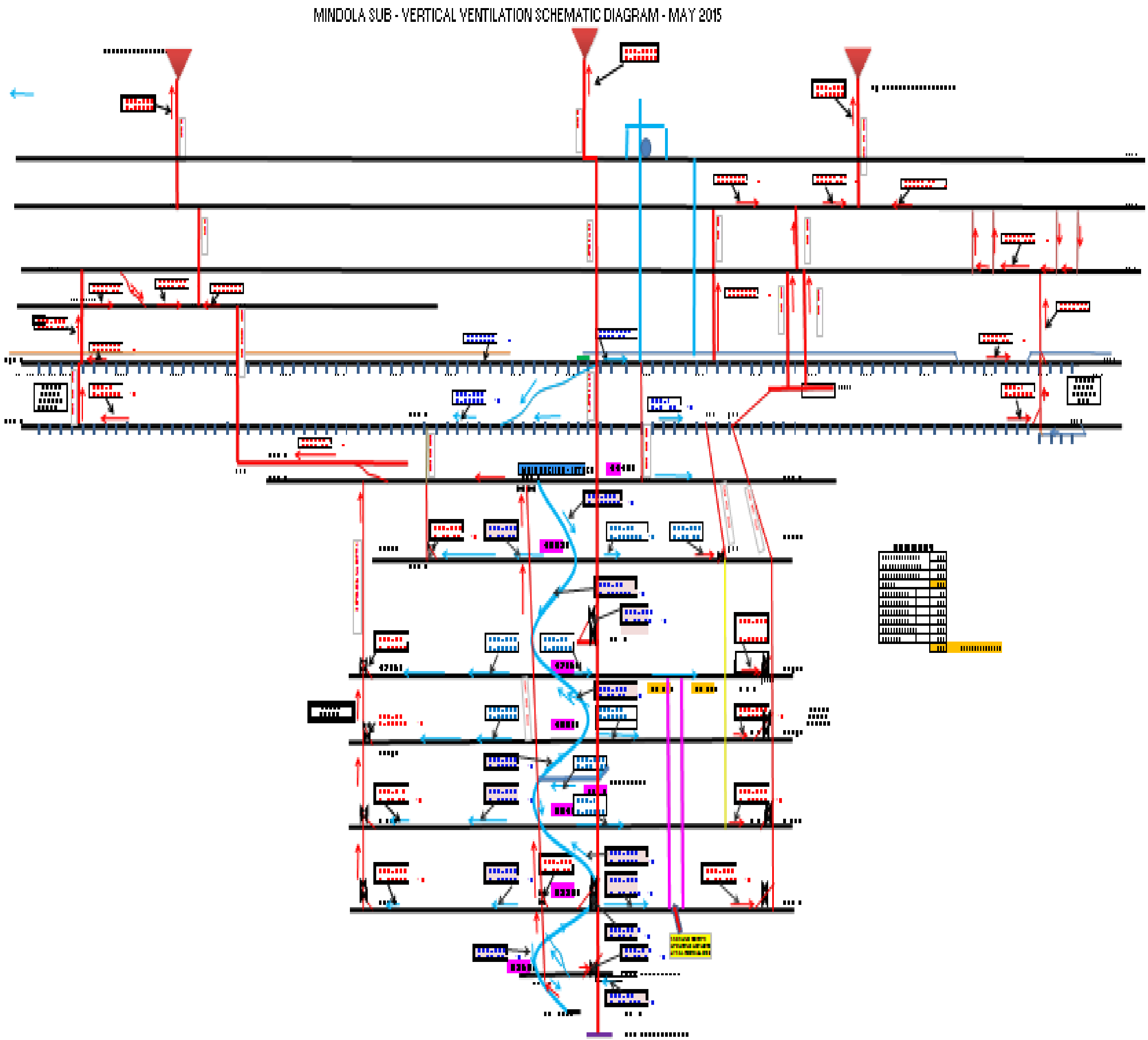

Figure 25: Mindola Sub- Vertical Ventilation Schematic diagram (Mine Planning, 2015)

CHAPTER FIVE: DATA COLLECTED

5.1 Introduction

This chapter provides the data that was collected and used in solving the problem of ventilation at Mindola sub vertical shaft.

5.2 Air- Temperature surveys

Air quality determination is carried out periodically at interval not exceeding three months by a competent person. Determinations include Air Quantity, temperature (wet bulb and dry bulb) and amount of dust present in the general body of the air (Mining Regulation IX). The maximum air quantity delivered to a stope at Mindola Sub vertical is planned to 20m^3/s. In development ends the planned air quantity is 25m^3/s. This air quantity is adequate and ensures temperatures are within the Mopani standard of 28^{o}C and 32 ^{o}C wet and dry bulb respectively. 20m^3/s must be adequate to ensure that dust levels are below 200 ppcc (MR 09)

The following Tables, Table 16, 17, 18, 19, 20 and 21 show air temperature surveys in production and Development areas carried out in three months August, September and October 2015.

Table 16: Air- Temperature surveys (Production areas) August 2015

Ref. No.	Name of Development heading	Available conditions				
		Volume	Temperature		Dust	Leakage
		m^3/s	WB °C	DB °C	ppcc	%/100m
Standards		> 20	< 28	< 32	<200	< 15.0
1	4180L 5095S Stope	18.0	**31.0**	**33.0**	52.0	**34.2**
2	4370L 5095S Stope	24.0	**32.0**	**34.0**	55.0	**17.0**
3	4370L 5870N Stope	22.0	**31.5**	**33.0**	47.0	6.9
4	4440L 2180S Stope	13.0	**31.0**	**33.4**	54.0	**31.4**
5	4716L 3160N Stope	32.5	**33.5**	**34.5**	N/A	**27.6**
6	4716L 1380S Stope	27.8	**30.0**	**32.5**	N/A	13.6
7	5045L 1610S Stope	26.9	**30.0**	**33.3**	54.0	**38.5**
Average		**23.5**	**31.3**	**33.4**	**52.4**	**24.2**

Table 17: Temperature surveys (Production) September 2015

Ref. No.	Name of Development heading	Available conditions				
		Volume	Temperature		Dust	Leakage
		m^3/s	WB °C	DB °C	ppcc	%/100m
Standards		> 20	< 28	< 32	<200	< 15.0
1	4180L 5095S Stope	17.8	**30.5**	**32.5**	44.0	**22.7**
2	4370L 5095S Stope	22.0	**33.0**	**34.5**	52.0	**15.9**
3	4370L 5875N Stope	23.0	**32.0**	**33.0**	N/A	7.7
4	4716L 3160N Stope	24.3	**32.0**	**34.5**	N/A	**22.8**
5	5045L 3250N Stope	31.6	**31.5**	**33.0**	N/A	**32.8**
6	5045L 1610S Stope	23.6	29.0	31.5	68.0	**34.2**
Average		**23.7**	**31.3**	**33.2**	**54.7**	**22.7**

Table 18: Temperature surveys (Production) October 2015

Ref. No.	Name of Development heading	Available conditions				
		Volume	Temperature		Dust	Leakage
		m^3/s	WB °C	DB °C	ppcc	%/100m
Standards		> 20	< 28	< 32	<200	< 15.0
1	4370L 5870N Stope	23.0	**32.0**	**33.0**	43.0	**19.6**
2	4440L 2280S Stope	16.0	31.0	32.4	43.0	13.1
3	4716L 3160N Stope	26.4	**33.0**	**34.5**	N/A	**24.5**
4	5045L 3250N Stope	33.4	**29.5**	**33.0**	N/A	**20.8**
Average		**24.7**	**31.4**	**33.2**	**43.0**	**19.5**

Table 19: Temperature surveys (Development) August 2015

Ref. No.	Name of Development heading	Available conditions				Leakage
		Volume	Temperature		Dust	Leakage
		m^3/s	WB °C	DB °C	ppcc	%/100m
Standards		> 25	< 28.0	< 31.0	<200	< 15
1	4180L 5810N Crosscut 1	16.5	**30.5**	**33.0**	60.0	52.4
2	4180L 5195S Crosscut 1	18.0	**31.0**	**33.0**	52.0	**33.3**
3	4370L 5810N Stub drive	22.0	**31.5**	**33.0**	47.0	**27.8**
4	4370L, 5195S Crosscut 2	24.0	**32.0**	**34.0**	55.0	12.7
5	4440L 2180S Footwall drive	13.0	**31.0**	**35.0**	54.0	**21.8**
6	4552L 2980N Decline	18.7	**29.0**	**32.0**	N/A	13.3
7	4552L3070N Footwall drive	18.7	**29.0**	**32.0**	N/A	5.3
8	4716L 1720S Crosscut 1	27.8	**30.0**	**32.5**	N/A	**21.2**
9	4716L 3340N Decline	31.6	**33.0**	**34.5**	N/A	11.9
10	4881L 3420N Footwall drive	24.3	**31.7**	**34.0**	62.0	13.4
11	4881L 3420N Stub drive	24.3	**31.7**	**34.0**	62.0	**22.2**
12	5045L 2980N Decline	34.2	**29.0**	**32.5**	56.0	**17.7**
13	5045L 3420N Footwall drive	35.1	**29.4**	**33.0**	58.0	**18.6**
14	5045L 1860S Stub drive	26.9	**30.0**	**33.3**	54.0	**20.9**
15	5150L Incline	14.2	23.0	28.0	60.0	14.9
16	5220L 3340N Crosscut 1	18.5	29.0	32.0	N/A	**54.3**
17	5220L 1720S Footwall drive	34.9	30.0	34.0	N/A	**15.2**
18	5500L RAW	50.1	25.0	**29.0**	N/A	6.4
19	5500L Vent crosscut	21.2	30.0	**37.0**	N/A	12.1
20	5650L Main Decline	50.1	25.0	**29.0**	N/A	13.6
Average		**26.2**	**29.5**	**32.7**	**56.4**	**20.5**

Table 20: Temperature surveys (Development) September 2015

Ref. No.	Name of Development heading	Available conditions				
		Volume	Temperature		Dust	Leakage
		m^3/s	WB °C	DB °C	ppcc	%/100m
Standards		> 25	< 28.0	< 31.0	<200	< 15
1	4100L 5810N Footwall drive	18.0	**32.0**	34.0	N/A	22.5
2	4180L 5810N Crosscut 1	17.5	**30.5**	**32.5**	52.0	25.0
3	4370L 5875N Stub drive	23.0	**31.5**	**33.0**	N/A	6.0
4	4440L 2490S Footwall drive	17.0	**30.0**	**32.0**	N/A	13.8
5	4716L 1495S Crosscut 2	18.9	**31.5**	**34.5**	N/A	14.3
6	4881L 3340N Crosscut 1	27.6	**31.5**	**33.0**	N/A	**22.6**
7	4881L 1610S Footwall drive	7.3	30.5	33.0	N/A	**19.3**
8	4881L 1610S Crosscut 2	7.3	30.5	33.0	N/A	**27.7**
9	5045L 1860S Footwall drive	26.3	29.0	31.5	68.0	**20.5**
10	5150L Incline	14.9	24.5	29.0	84.0	**19.8**
11	5220L 3340N Access crosscut	20.6	29.0	32.0	N/A	**29.1**
12	5500L Vent crosscut	28.6	28.5	**34.0**	N/A	14.8
Average		**18.9**	**29.9**	**32.6**	**68.0**	**19.6**

Table 21: Temperature surveys (Development) October 2015

Ref. No.	Name of Development heading	Available conditions				
		Volume	Temperature		Dust	Leakage
		m^3/s	WB °C	DB °C	ppcc	%/100m
Standards		> 25	< 28.0	< 31.0	<200	< 15
1	4370L- 5940N Stub drive	23.00	**32.0**	**33.0**	43.0	**23.1**
2	4440L 2280south Stub Drive	16.00	**30.5**	**32.0**	50.0	**15.3**
3	4716L 1720S Stub drive	15.30	**32.0**	**34.5**	N/A	**24.6**
4	4716L 1495S Crosscut 2	15.30	**32.0**	**34.5**	N/A	**26.3**
5	4881L 1495S Crosscut 2	32.60	30.0	32.5	N/A	**19.0**
6	4881L 1610S Crosscut 1	32.60	30.0	32.5	N/A	**32.8**
7	5045L 1860S Footwall drive	17.60	29.5	33.0	76.0	**23.7**
8	5150L Incline	14.70	25.0	30.0	70.0	**20.1**
9	5220L 3420N Stub drive	17.80	29.5	32.0	68.0	13.5
10	5220L 1720S Crosscut 2	33.70	31.0	34.0	N/A	**15.3**
11	5345L 900N	10.13	28.5	**33.5**	N/A	22.0
12	5485L 900N	51.30	31.0	**36.0**	N/A	2.9
13	5500L Vent crosscut	48.77	28.5	**35.0**	N/A	2.8
14	5360L 1580N Main decline	48.77	28.5	**35.0**	N/A	6.7
Average		**27.00**	**29.9**	**33.4**	**61.4**	**17.7**

Table 22 and Figure 26 shows the mean temperatures for 12 months at Mindola sub vertical shaft (I.e. dry and wet bulb temperatures).

Table 22: Mean temperatures for 12 months at Mindola sub vertical shaft

	WB ^{0}C	DB ^{0}C
Jan-15	29.90	32.20
Feb-15	30.50	33.90
Mar-15	31.20	33.40
Apr-15	31.40	33.20
May-15	30.40	33.10
Jun-15	28.60	32.60
Jul-15	29.90	33.40
Aug-15	30.40	33.05
Sep-15	30.60	32.90
Oct-15	30.65	33.30
Nov-15	31.50	33.90
Dec-15	29.90	33.20

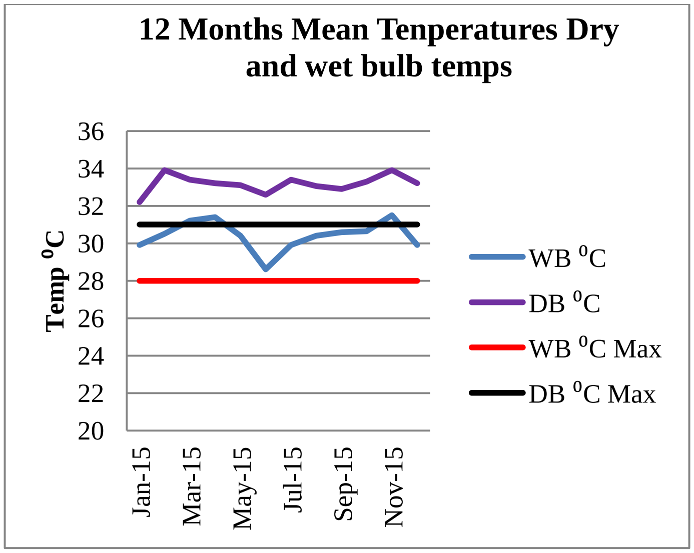

Figure 26: Mean Temperatures for 12 months

5.3 Exhaust Gases

Sampling of the exhaust gases for carbon monoxide and oxides of nitrogen on self-propelled diesel engine is systematically carried out at Mindola SV using the gas tube and pump (Figure 27). The manager, according to mining regulation part IX, shall ensure that the exhaust gases are does not contain 0.2 per centum by volume of carbon monoxide or 0.1 per centum by volume of oxides of nitrogen. Tables 23 and 24 show concentrations of carbon monoxide and oxides of nitrogen recorded from exhaust of diesel engines:

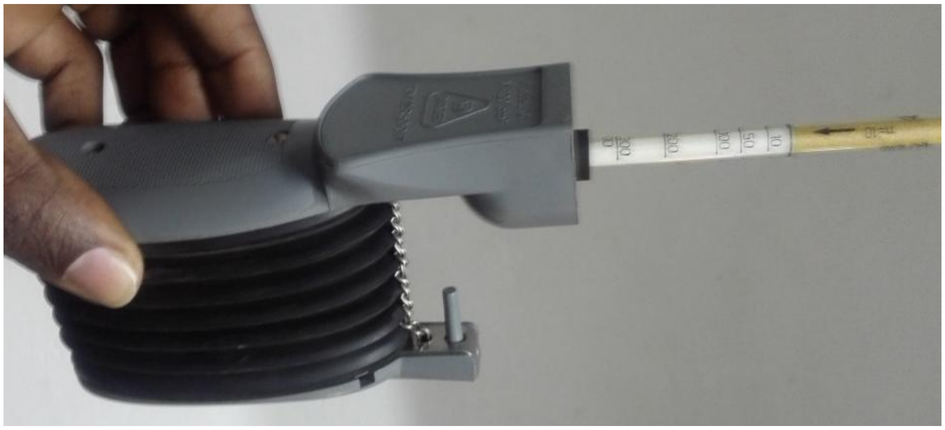

Figure 27: Gas tube and pump

Table 23: Concentrations for carbon monoxide and oxides of nitrogen- August 2015

Equipment Number	Equipment Description	Position	CO	NO
			%	%
14	CAT	4552 LEVEL -2630N POSITION	0.02	0.03
19	CAT	DECLINE TO 4552 LEVEL North	0.03	0.04
20	CAT	ACCESS TO 4716 LEVEL-900N/S	0.01	0.01
48	AD	5220 LEVEL-1270S POSITION	0.03	0.04
7	AD	4881 LEVEL-3250N POSITION	0.02	0.02
58	CAT	RE-FUELING BAY-5220 LEVEL	0.01	0.02
51	CAT	RE-FUELING BAY-5220 LEVEL	0.04	0.03
6	UDT	5220 LEVEL W/BAY	0.03	0.04
30-06	AD	5220 LEVEL W/BAY	0.01	0.01
30-07	AD	5345 LEVEL-900N POSITION	0.01	0.02
45	CAT	ACCESS TO 5485 LEVEL	0.03	0.04
49	CAT	5345 LEVEL DECLINE-5650L	0.02	0.01
59	CAT	5220 LEVEL SHAFT STATION	0.01	0.03

Table 24: Concentrations for carbon monoxide and oxides of nitrogen- September 2015

Equipment Number	Equipment Description	Position	CO	NO
			%	%
16	CAT	4960LEVEL DRIVE TO DECLINE	0.01	0.02
20	CAT	4881LEVEL 1495SOUTH POSITION	0.01	0.01
14	CAT	4881LEVEL 2895N POSITION	0.01	0.01
30 - 06	AD	5045LEVEL DECLINE-5220LEVEL	0.02	0.02
30 - 07	AD	5220LEVEL SHAFT STATION	0.02	0.03
51	CAT	5220LEVEL SHAFT STATION	0.02	0.02
59	CAT	5220LEVEL LOADER WORKSHOP	0.01	0.01
6	UDT	5220LEVEL LOADER WORKSHOP	0.03	0.04

5.4 Hydrogeology

The Ore Formation is sandwiched by aquifers of different magnitude. These aquifers have been designated names in relation to the Ore Formation position.

The four aquifers (4) noted at the shaft are as follows;

- Footwall Aquifer
- Near Water Aquifer
- Far Water Aquifer
- Ultra-Far Water Aquifer

5.4.1 Footwall Aquifer

This aquifer is confined in the Footwall Sandstone and Footwall Conglomerate and lies on the footwall side of the Ore Formation. The Footwall Sandstone ranges from 20 to 30m and is made up of grey Arkoses to reddish sandstone with argillaceous intercalations. The Lower Conglomerate is 5 to 15m thick arkoses to sandstone pebble-cobble –boulder clasts set in a sandy-silty to argillaceous matrix. The water is generally associated with fissure and joints leading to local leaching. The water normally drains off upon Mining development.

5.4.2 Near Water Sediments Aquifer

This aquifer lies about 6m to 60m above the ore body. It is confined to the Near Water Sediments unit. This is a succession of interbedded sandy Dolomitic Argillites. The unit has two sub units called the **Lower near Water** Sediments and the **Upper near Water** Sediments. A thin aquitard, the Middle Quartzite, separates the two units.

The water is stratigraphically controlled with poor continuity. It is also controlled by tension fissures and fault zones radiating from the No.3 Granite at about 5000N and extending northwards. Elsewhere the aquifer fades away. Dewatering of this aquifer is carried out routinely from dewatering crosscuts spaced at 60m intervals. Lying on top of the Near Water Sediments is the Upper Quartzite unit. This unit forms an aquiclude between the Near Water Sediments Aquifer and the Far Water Aquifer.

5.4.3 Far Water Aquifer

This aquifer lies about 60 to 120m above the orebody. It is localized in the Dolomite Argillite Sequence, Far Water Dolomite and Far Water Sediments. This is a succession of Dolomites with local intercalations of Grits, Argillites and Limestone. The water is lithologically controlled in part and in some cases structurally controlled. The aquifer has poor strike and dip continuity. The major source of this aquifer water lies north of 4000N. Current Dewatering in this area indicates that the aquifer bottoms out between 4100 and 4130 feet below surface. Hence it is expected that no Far Water will be encountered below this level throughout the mine. The region between 4000N and 4000S is generally dry with occasional patches between 1700S and 2300S. Another isolated water portion is noted between 3600S and 5000S position. Dewatering of this aquifer is done routinely from dewatering crosscuts spaced at 60m intervals.

5.4.4 Ultra-Far Water Aquifer

This unit has a thickness of between 160m to 200m and lies about 400 to 500m above the Ore Formation. The succession is made up of Dolomites and sandy Dolomite. The water is mainly confined in the sandy dolomite layers. Current Dewatering practice does not dewater this aquifer as it is very far off the projected angle of 600 of the cave cracks. *(Geology, MSV 2015)*

5.4.4.1 Aquifer Monitoring and Dewatering Strategy

The Footwall, Near Water and Far Water Aquifers are routinely dewatered before mining in these areas are scheduled. Table 25 shows pumped out volumes of water in the month of May 2015 while Figure 28 is a schematic diagram of water management. The average temperature for the water pumped out is 33.5 ºC.

Table 25: Pumping Figures in month of May (Geology, Mindola SV 2015)

Week	**Water quantity pumped (m^3/day)**
Week One	55,474
Week Two	56,750
Week Three	54,578
Week Four	55,509
Average	**55,577**

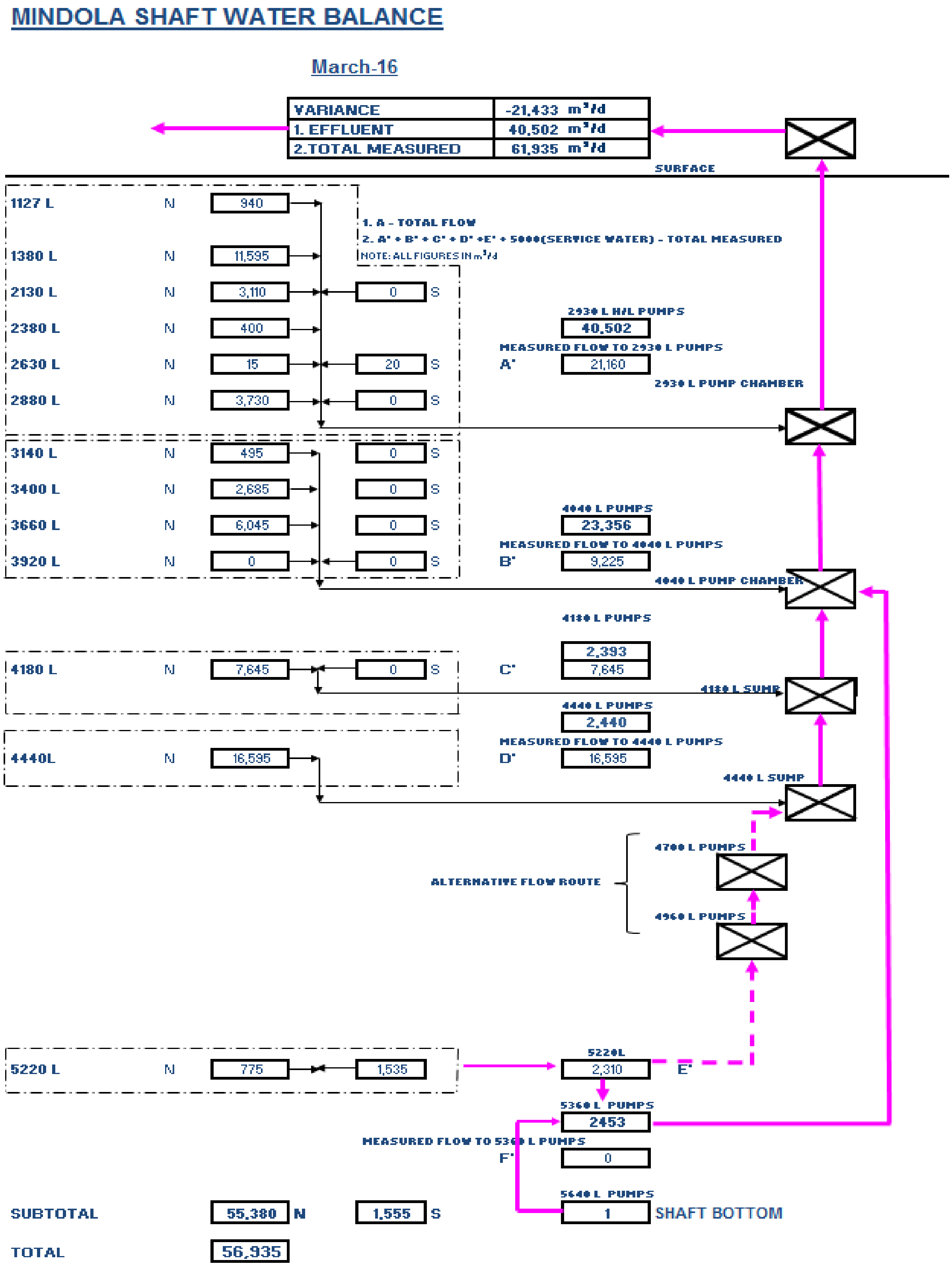

Figure 28: Water balance at Mindola Sub-Vertical Shaft (Geology, Mindola SV 2015)

5.5 Electrical power consumption

Table 26: Electrical power consumption July 2014-June 2015 (Electrical Power Distribution, Mindola SV)

MONTH	kW
July	16705.70
August	15797.50
September	14783.00
October	14461.20
November	15201.30
December	16545.00
January	14175.20
February	14478.30
March	16159.60
April	16110.10
May	17070.00
June	16179.60
Total	**187,666.40**

Table 26 shows electrical power consumption for the month of July 2014 to June 2015.

5.6 Mobile Equipment

The introduction of more diesel equipment in underground confined environment due to mechanisation demands sufficient ventilation. This is to cater for the exhaust gas and dust and primarily, the heat production.

The mining regulations require a minimum airflow of $0.05m^3/s$ per kilowatt of engine power (Mining regulation 902 (2) (f).

At present the following machinery shown in Table 27 are being used in the many different operations that are taking place underground.

Table 27: Diesel Units at Mindola SV

Type	Unit No.	Power rating (kW)
LHD	CAT12	186
LHD	CAT16	186
LHD	CAT19	186
LHD	CAT51	186
LHD	CAT11	186
LHD	CAT13	186
LHD	CAT14	186
LHD	CAT15	186
LHD	CAT20	186
LHD	CAT45	186
LHD	CAT48	186
LHD	TNK36	255
LHD	TNK58	255
LHD	TNK59	255
LHD	TNK37	255
LHD	TNK38	255
LHD	TNK39	255
LHD	TNK40	255
LHD	TNK41	255
RIGS	CUBEX01	55
RIGS	CUBEX02	55
RIGS	SOLO-06	55
RIGS	SOLO-O5	55
RIGS	CUBEX03	55
RIGS	CUBEX04	55
RIGS	R/BOOMER-02	55
RIGS	R/BOOMER-05	55
RIGS	R/BOOMER-11	55
RIGS	R/BOOMER-04	55

Continuation of Table 27

Type	Unit No.	Power rating (kW)
RIGS	R/BOOMER-04	55
DUMP TRUCKS	UDT06	255
DUMP TRUCKS	UDT07	255
DUMP TRUCKS	UDT08	255
DUMP TRUCKS	UDT09	255
DUMP TRUCKS	EJC522	186
DUMP TRUCKS	EJC533-15	186
DUMP TRUCKS	EJC533-15	186
DUMP TRUCKS	EJC533-15	186
DUMP TRUCKS	EJC533-14	186
DUMP TRUCKS	EJC533-06	186
DUMP TRUCKS	EJC533-05	186
UTILITIES	NORMET-05	55
UTILITIES	NORMET-LUB 03	55
UTILITIES	NORMET-LUB 02	55
UTILITIES	LAND CRUISER-03	50
UTILITIES	LAND CRUISER-02	50
UTILITIES	LAND CRUISER-01	50
DIESEL LOCOMOTIVES	CL 01	50
DIESEL LOCOMOTIVES	CL 03	50
DIESEL LOCOMOTIVES	CL04	50
	TOTAL	**7,423**

5.7 Mindola SV workforce

Coming up with amount of labour required by a mining company to meet its business objectives can be crucial. The Mindola Mine New Shaft has a total labour force of 815 and comprises of 468 from mining and 347 from engineering. This can be seen in Table 28 and Table 29.

Table 28: Labour force (Mine Planning, Mindola SV 2015)

SECTION	CURRENT
MINING	
Mining Production Labour	255
Mining Services	111
Mine Technical	54
Mining Administration	48
Total Mining	**468**
ENGINEERING	
Shafts	103
Mobile	66
Pumps	84
Crushers and Conveyors	62
Electrical	32
Total Engineering	**347**
Grand Total	**815**

Table 29: Average number of men working/shift at Mindola SV, 2015

Day	Day Shift	Afternoon Shift	Night Shift
Monday	428	175	179
Tuesday	535	179	181
Wednesday	530	178	180
Thursday	508	176	179
Friday	477	170	176
Average	**496**	**176**	**179**

5.8 Production Schedule

Table 30 shows the production schedule for years 2017 to 2021. Due to expansion project at Mindola SV production will be increased from 1,440,000 tonnes to 2,000,000 tonnes per annum.

Table 30: Mindola Sub-Vertical production schedule

Mindola Sub vertical Shaft	Units	Value
Tonnes Hoisted	Tonnes	2,000,000
Cu Grade	%	1.88
Co Grade	%	0.12
Copper in Ore Hoisted	Tonnes	37,600
Cobalt in Ore Hoisted	Tonnes	2,400

5.9 Explosive Consumption

Table 31 shows the monthly explosive consumption for the period 1st to 30th April, 2015.

Table 31: Monthly Explosive Consumption (01st – 30th April, 2015) Source: Blasting Office, Mindola SV

Explosive Name	Units	Consumption
Anfex 25kg	Bag	456.0
Magnum Buster -50x270mm	Case	257.0
RS100G Emulsion	Case	655.1
Explo Smooth- 38x550mm	Case	181.0
Total	Case/bag	1549.1

5.10 Existing exhaust ventilation raises

The existing exhaust ventilation raises and a number of levels served are shown in the Figure 29 and Table 32 respectively.

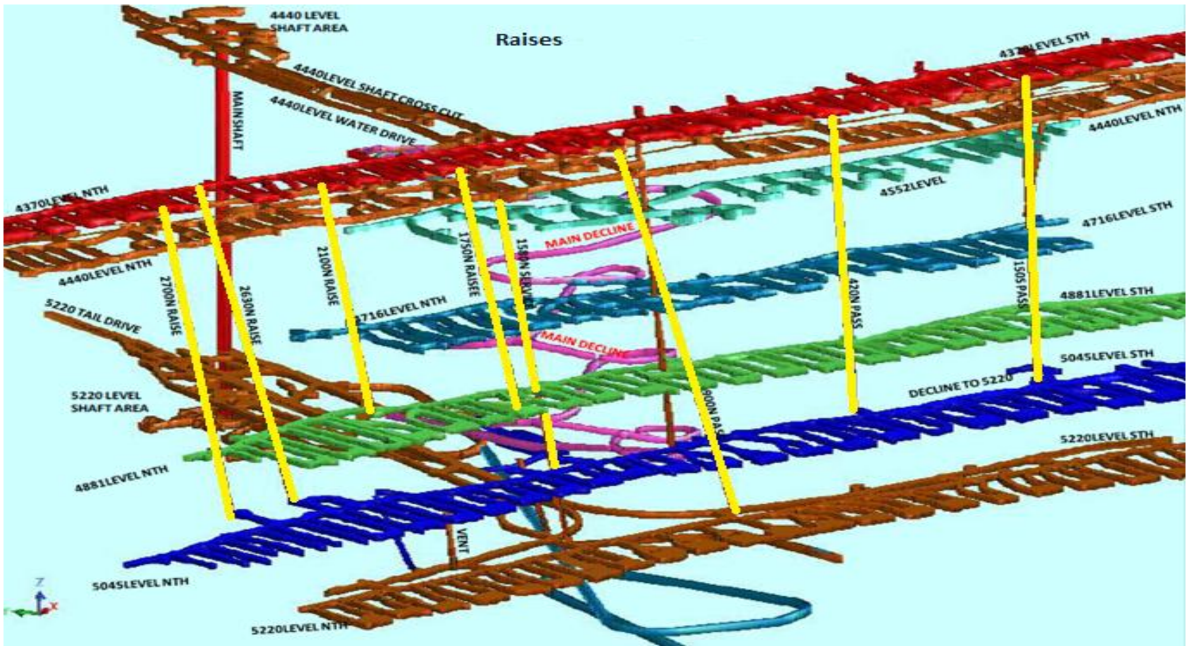

Figure 29: Existing exhaust ventilation raises (Mine Planning, Mindola SV 2015)

Table 32: Ventilation raise and number of levels served

No.	Vent raise location	Diameter (m)	Area (m²)	Air velocity (m/s)	Maximum air carrying capacity (m³/s)	Number of levels / sections served	Air volume requirements per section per level	Expected air volume for the raise (m³ /s)
1	**2710N** exhaust vent raise	2.4	4.52	18	81.4	1	32.8	32.8
2	**3160N** exhaust vent raise	3.1	7.55	18	135.9	5	32.8	164
3	**1050S** exhaust vent raise	2.4	4.52	18	81.4	4	32.8	131.2
4	**1495S** exhaust vent raise	3.1	7.55	18	135.9	1	32.8	32.8

CHAPTER SIX: DATA ANALYSIS

6.1 Introduction

This chapter provides the data analysis and calculation for the mine heat load and air quantity needed for adequate ventilation at Mindola Sub-Vertical shaft (MSV).

6.2 Air Quantity Determination methods

The air that is required to ventilate MSV shaft is determined using either the three methods namely:

1) Heat Loads method,
2) Air – tonnage ratio and,
3) Widely accepted charts.

6.2.1 Heat Loads calculation

Calculation parameters for Heat Loads

Below are parameters necessary for heat load and air volume calculation for the mine;

- KJ/s is equivalent to kW. *(Le Roux, 1990)*
- Density of quartzite rock is 2690 kg/m^3 *(Cornelis Klein, 2016)*
- Thermal capacity (C) and conductivity (K) of quartzite are 837kJ/kg ºC and 5.74W/m ºC respectively. *(Cornelis Klein, 2016)*
- Standard density of air is 1.2kg/m^3
- Quantity of down cast air is (shaft 1 and 2= 369.0 + 255.0) = 624m^3/s
- Level of consideration is 1,726m (5660L)
- Mopani domestic efficiency is about 75%
- Average Virgin Rock Temperature (VRT) is 42.0ºC
- Ambient temperatures (θ) are 24.0/25.0ºC. Maximum desirable temperature are 31.0/34.0ºC

- For maximum air volume calculation assume that all the diesel units are operating in one room at efficiency 80%
- Considering 26 working days in one month

6.2.1.1 Heat from Rock Strata

Radial Heat Flow from Established Haulages

- Total length of the entire main mine openings is 13,200m.

$q = 3.35LK^{0.854}$ (VRT- θ) *(McPherson, 2012)*

$= 3.35 \times 13{,}200 \times 5.74^{0.854}(42.0-25.0)/1000$

= ($m \times W/m^{\circ}C \times {}^{\circ}C$)/1000

= 3343.32 kW

Where; L = Total length of all the haulages (m)

Advancing End of the Heading

- Daily face advance (DFA) is 1.8 m x 2= 3.6m
- Length of the advancing end of the heading is 3,600m

$q = 6K (L+4DFA) (VRT- \theta)$

$= 6 \times 5.74(3600+ (4 \times 3.6)) (42.0 - 25.0)/1000$ **=** ($W/m\ {}^{o}c \times m \times {}^{o}c$)**/**1000

= 2116.16 kW

Where;

L = Length of the advancing end (m)

DFA = Daily face advance (m)

= **3343.32 kW +2116.16 kW**

=**5459.48 kW**

6.2.1.2 Heat of Auto-Compression

- Auto- Compression of the Ventilating air
- Depth of the mine level of consideration is 1,726m (5660L)
- Down cast quantity (shaft 1 and 2= 369.0 + 255.0) = 624m^3/s
- Assuming Ambient density of 1.2 kg/m^3
- Taking the increase in sigma heat per 1000m as 9.79kJ/kg

Formula; $H_2 - H_1 = g\,(Z_1 - Z_2)/1000$ *(Hartman, 1997)*

Where: $H_2 - H_1 = S$

$S = 9.79 \times \Delta Z/1000$ kJ/kg

$= (9.79 \times 1726)/1000$

= 16.90 KJ/kg

$M = Q \times w$ (kg/s) (Burrows, 1974)

$= 624 \times 1.2$

= 748.80 kg/s

$q = S \times M$ (kW)

$= 16.90 \times 748.8$

= 12,654.7 kW

Where;

Q = Quantity of the down cast air (m^3/s)

w = Density of the ambient air (kg/m$^{3)}$

ΔZ = Difference in elevation between 1,726m (5660L) and surface

S = Sigma heat at 1,726m (5660L)

M = Mass flow of the down cast air (kg/s)

6.2.1.3 Heat from Fissure Water

Provided the mine pumps **35,577.75m^3** from the mine every day and the weekly borehole measurements of water is as given above. 35,750m^3/day can be assumed as the amount of water added to the mine working every day. Average Fissure Temperature = 33**.5 ºC**

- Average volume of water measured from the boreholes is 35,577.75m^3/day.
- Taking 1Litre = 1Kg and 1000Litres = 1m^3
- Average water fissure temperature is 33.5 ºC.

Formula; q = (M × C × (W_f - θ) / t (kW) *(Ramsden, F. von Glenn, 2012)*

= [(35577.75×1000×4.187× (33.5-25)]/ (24×30×3600)

= (Kg ×kJ/kg ^{0}C× ^{0}C)/s

=977.00 kW

Where; q = heat from water fissures

M = mass of the water fissures pumped out of the mine per day (kg)

W_f = water fissure temperature (^{0}C)

t= seconds in one month

6.2.1.4 Mechanized Equipment

Electrical Equipment

Electrical power consumption for Mindola Sub vertical shaft for 12 months is given as 187,666.4/12 kW as shown in Table 26 in section 5.5. The average consumption can be calculated as shown below:

Average power consumption =Total/12 =187,666.4/12 **=**15,638.87kW

- Using Mopani domestic efficiency to be 75 %, the heat from Electrical consumption is given as: **Formula; q = P×I** *(McPherson, 1993)*

=15,638.87kW ×0.25 **= 3,909.7 kW**

Where;

P = Power consumption (kW)

I = Mopani domestic inefficiency (%)

q = heat (kW)

Diesel Units

Taking the total input power rating of the diesel units as **= 7,492kW** (Table: 27 section 5.6)

Formula; $q = R_e$ (*Stinnette and De Souza, 2013)*

=7,492kW

Where;

R_e = Total input power rating for all the diesel units (kW)

q = Heat load from diesel units (kW)

6.2.1.5 Heat from workers

The average number of workers in the mine can be calculated using day shift hence average 496 men (Table 29 section 5.7) is used for heat calculation due to workers.

- Given that on average a worker produces $260 W/m^2$ when performing hard labor.
- Taking the average mass and height of a miner as 60.5kg and 1.688m respectively. *(J Burrows, 1982)*

Area of man, $A = 0.202\ Mt^{0.425}\ Ht^{0.725}$ (m^2) (McPherson, 2012)

$= 0.202\ (60.5^{0.425})\ (1.688^{0.725})$

$= 1.814 m^2$

$q = N \times A \times R_M = 496 \times 1.814 \times 260$

= **234 kW**

Where;

Mt = Mass of a worker (kg)

Ht = Height of man (m)

N = Number of workers

A = Area of a worker (m^2)

R_M = Metabolic heat rate (W/m^2)

6.2.1.6 Heat from Broken Rock

According to production schedule in Table 30 section 5.8 average rock broken per month at 20% waste inclusion = 200,000t/m tones.

Formula; q = M× K× (VRT – θ)/t (*Le Roux, 1990)*

= (200,000 ×1000×0.837× (42.0-25.0))/ (26×24×3600) = (kg × (kJ/kg ^oC) × ^oC)/s

=**1266.83 kW**

Where; M = Mass of rock broken per month (kg)

K= Thermal Capacity of the rock (kJ/kg ^oC)

6.2.1.7 Heat from Explosives

Monthly explosive consumption for the period of 1st to 30th April, 2015 was 1549 cases (Table 31 section 5.9). Provided each bag and case weighs 25kg, the total monthly consumption in kg was;

= 25kg×1549.0
= 38,727kg

- 60 % of the heat from the explosives is returned to the rock
- Heat content of ANFO explosives is 3820kJ/kg

Formula; q = (0.6× M ×H)/t

= (0.6×38,727 ×3820)/ (26×24×3600)

= (kg × kJ/kg)/s **= 39.5 kW**

Where;

M = Mass explosives used by the mine (kg)

H = Heat content of the explosives (kJ/kg)

Table 33: Summary and total of all the heat loads

Heat Source	**Quantity (kW)**
Rock Strata	**5,459.48**
Auto-Compression	**12,654.70**
Fissure water	**977.00**
Electrical Equipment	**3,909.70**
Diesel Units	**7,492.00**
Men	**234.00**
Broken Rock	**1,266.83**
Explosive	**39.50**
Total	**32,033.21**

The following parameters are needed to calculate the volumetric requirement for the mine;

- Taking the Barometric pressure at 1,726m (5660L) as 107.3Kpa
- Using the 101.325kPa Psychometric chart in Figure 30, the sigma heat at the ambient temperatures of 24.0/25.0 ^{o}C is 73.5kJ/kg and at maximum allowable temperatures of 32.0/34.0 ^{o}C is 111.5kJ/kg.

- The Density of air, at the maximum allowable temperatures and pressure of 107.3Kpa is 1.02kg/m^3.
- The total quantity of heat load for the mine **32,033 kW**

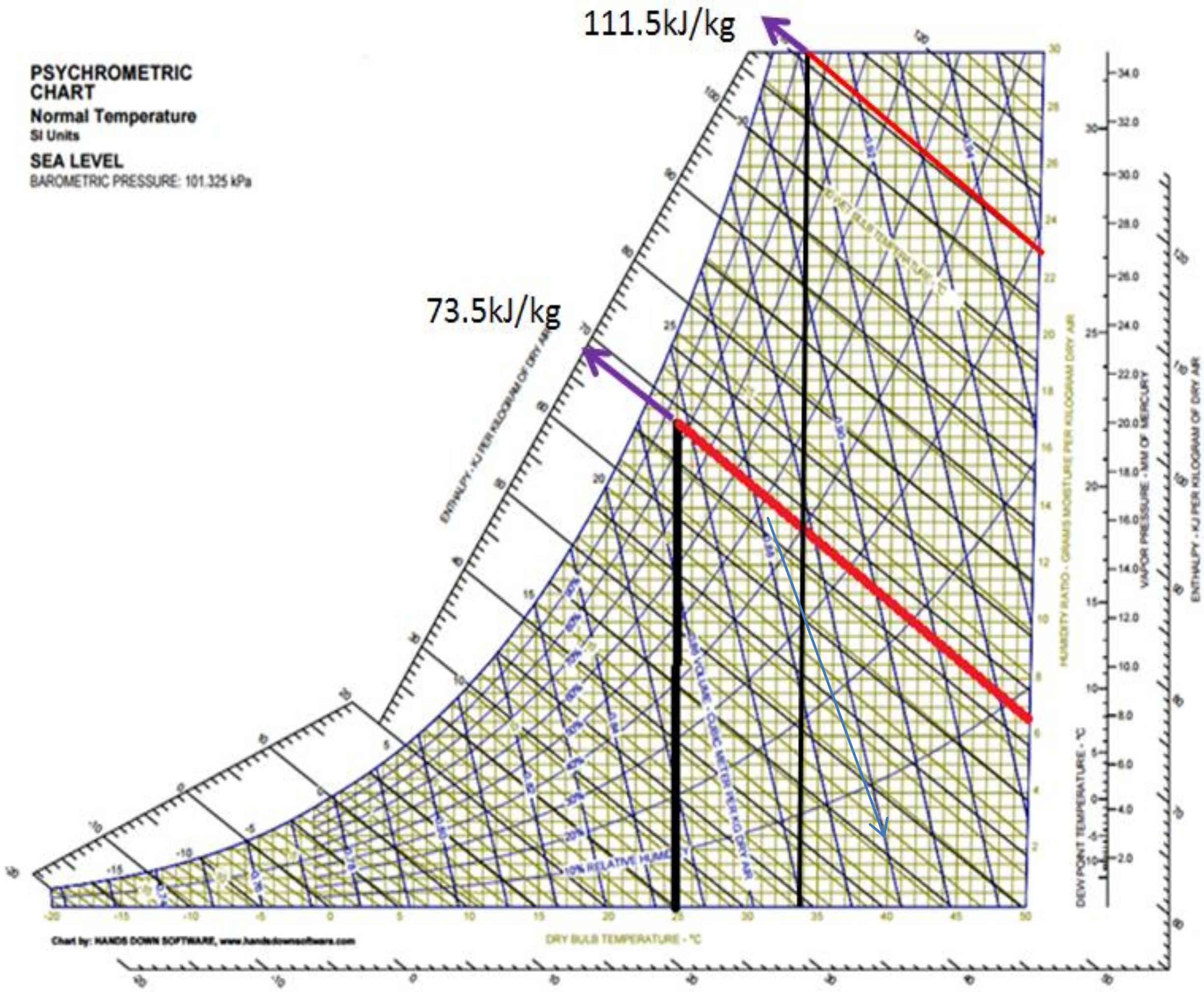

Figure 30: Psychrometric Chart (A. Barenberg, 1974)

Formulae; Q = q/ (p × (S_1 – S_2)) (m³/s) *(McPherson, 2012)*

= (kJ/s) / (kg/m^3 × kJ/kg)

= **32,033** / [1.02 × (111.5-73.5)] **= 826.45m^3/s**

Where;

q = total heat load (kW)

P = density of air at the maximum temperature (kg/m^3)

S= sigma heat (kJ/kg)

Q = quantity required to dilute the heat load (m^3/s)

6.2.2 Air Tonnage Ratio

This is a compilation of statistical data relating air volume requirement to the tonnage broken per month. Air – Tonnage Ratio of 3.0m^3/s/1000t/month will apply, *(Burrows, 1974).* Beyond the year 2017, mining operations will concentrate below the 5220L at an annual rate of 2.0 million tonnes of ore.

Therefore using 2million tonnes/ year and 3m^3/s/1000t/month, air required is

$$Q = \frac{(3\ X\ 2000000)}{12*1000}$$

$$= \mathbf{500m^3/s}$$

6.2.3 Worldwide Accepted Charts

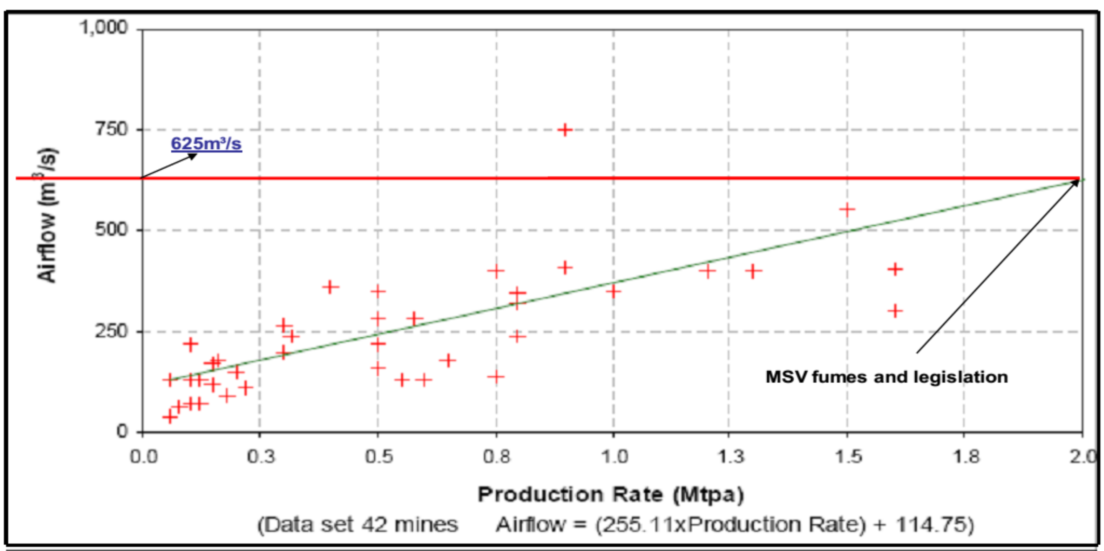

Figure 31: Airflow- Production rate chart (Burrows, 1974).

Using the Airflow- Production rate chart (Figure 31) by plotting a straight line in the centre of the cross marks, the air quantity required to produce 2 million tonnes of copper at Mindola SV is **625m^3/s**

Summary of Air volumes calculated using different methods

From the above calculations using the three methods to determine air required to ventilate MSV shaft, **Heat Loads method (826.45m^3/s)** is chosen since it gives a high value of air quantity required. Summary of air volumes calculated using different methods is given in Table 34.

Table 34: Summary of Air volumes calculated using different methods

Determining method	Air volume calculated (m^3/s)	Remarks
Heat Loading calculations	826	Adopted
Air – Tonnage Ratio	500	
Ventilation Charts	625	

CHAPTER SEVEN: PROPOSED DESIGN OF VENTILATION SYSTEM

7.1 Introduction

This chapter explains the proposed design and takes into account the calculated parameters of heat loads, dust and gases. The design is an extension or an addition to the current ventilation network.

7.2 Underground Ventilation Design

Underground mining operations at Mindola Sub-Vertical Shaft are currently confined between 4440 Level and 5220 Level, with the re-deepening of the shaft addition levels down to 6000 Level will be included. This will require ventilation parameters to be adjusted in order to meet the new demand.

The results of the calculation of heat loads from various contributors of heat to the underground working area can be summarized as shown in Table 35:

Table 35: Summary of Heat loads

Heat Source	Quantity (kW)
Rock Strata	5,459.48
Auto-Compression	12,654.70
Fissure water	977.00
Electrical Equipment	3,909.70
Diesel Units	7,492.00
Men	234.00
Broken Rock	1,266.83
Explosive	39.50
Total	32,033.21

Using the formulae in the previous chapter for airflow determination, the air required to ventilate or dilute the heat load is calculated as 826.45m^3/s. therefore the current air being upcasted from the mine of 624m^3/s will not provide adequate ventilation. The upcast fans need to be upgraded to provide additional 202m^3/s needed to dilute heat loads from underground workings. Figure 32 shows the proposed design of the ventilation system.

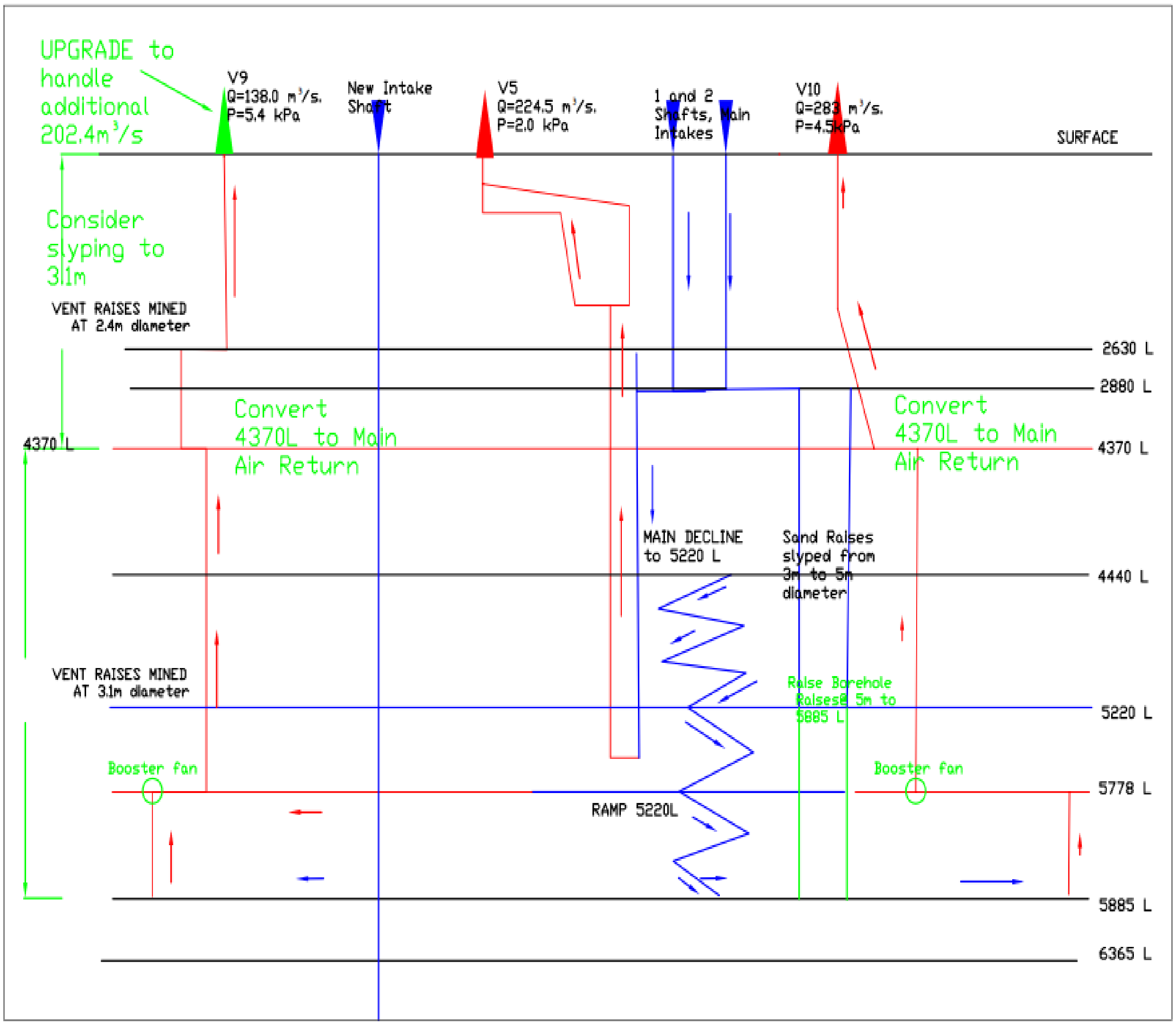

Figure 32: Proposed design of the Ventilation system at MSV

The calculated air volume of 826.45m^3/s based on the heat load formula, will require to be pushed through the new downcast shaft, Sub -Vertical shaft, and the Raise Boreholes raises from 5220L to 5880L.

7.3 Intake air quantity

7.3.1 Sub-Vertical shaft

The Sub-Vertical shaft was mined at 7.1m diameter giving total area of 39.6m² due to shaft installations likes pipes for compressed air and water and the cage the available area for ventilation is 70% (39.6) =27.7 m². At the recommended minimum velocity of 10.0m/s, the shaft will handle a total volume of 277m³/s

7.3.2 New Shaft

The new shaft at Mindola SV is being mined at 6.1 diameters which give a total area of 29.22m². The available area for ventilation is 70% (29.22) =20.45 m². At the recommended minimum velocity of 10.0m/s for shafts, the shaft will handle the total volume of 204.6m³/s

7.3.3 Raise Bore Hole 2880L / 4440L

A 2 x 5.0m diameter gives a total area of 39.28m². Available area for ventilation at 100% *(39.28) = 39.28 m². At the recommended minimum velocity of 10.0m/s, these will handle the total volume of 392.80m³/s

7.4 Proposed Features

7.4.1 4370L as Return Airway (RAW)

A return is required to effectively return the polluted air from working areas below 4440L at the recommended velocity of 15m/s. since there will be no production at 4370L, 4370L will need to be converted into the main return airway. The foul air off the haulages, the Central Ramp and the sublevels will return through the established raises to the 4370L RAW.

7.4.2 900L and 1075L RBH Raises

The 900L and 1075L 2.4m diameter raises that end at 5045L must be extended from 5045L to 5885L to supply fresh air and facilitate the mining of the central ramp between 5220L and 5885L.

7.4.3 Vent raises

The 3.1m diameter RBH return raises will be mined at 150m (Zambia mining regulation) spacing on either side i.e. North and South. These raises will finally link to the MRAW 4370L.

The existing internal raises in the production block of 4440L / 5220L should link to the new mining block below 5220L.

7.4.4 Extension of the Main upcast shafts influence by use of booster fans at 5778L

It is not always that an inefficient existing main upcast fan is replaced; at times there is need to boost the system with a booster fan. Table 36 shows the measurements conducted on the Main Upcast Fans to determine the existing efficiencies. The efficiencies on V10 and V9 are low therefore there is need to install Booster Fan to assist the main upcast fan in providing primary ventilation current.

The specifications should be similar to the main fan installed on surface, i.e. Single Inlet, Centrifugal backward curved aerofoil bladed design and duty. The pull-push is preferred.

Table 36: Measurements on main uncast fans

FAN	Static Pressure (kPa)		Air Volume Measured (m^3/s)			Level	System Efficiency (%)	Fan Efficiency (%)
	Design Duty	Actual Duty	Design Duty	Actual Duty	U/G Point			
V10	5.0	4.5	354	283	245	3920	87	80
V5	6.5	5.4	150	138	134	3920	97	92
V9	3.1	2.0	307	237	225	2630	95	77

CHAPTER 8 PROPOSED REFRIGERATION SYSTEM

8.1 Introduction

This chapter highlights the selection criteria for refrigeration plant.

When simple methods of controlling the mine climate do not make an effect on mine environment artificial cooling of the air must be considered (Burrows *et al.*, 1989). This is done by refrigeration of mine air. Installation of refrigeration plants is designed to produce tolerable environment condition throughout the year. The introduction of air cooling and refrigeration on surface can allow hot underground mines to exploit deeper reserves, extend the maximum limits of mining and increase production where underground air temperatures are the limiting factor. *(Wilson and Pieters, 2008)*

8.2 Causes of high temperatures

The major heat load contributors were found to be auto compression, diesel machines and rock strata. The temperature increase was found to be more as the depth increased. The number of operational diesel-powered equipment, the increase in geothermal heat and the handling of freshly blasted ore at a temperature often exceeding 50°C resulted in rapid heat energy absorption, both in terms of sensible and latent heat. This resulted in the wet-bulb temperature exceeding the 28°C Mopani standard.

8.3 Relation between mining depth and type of refrigeration system

Refrigeration and the associated cooling installations at depth are essential to ensure that legal thermal requirements are maintained. Work done by *Bluhm and Von Glehn (2010)* indicates the relationship between mining depth and type of refrigeration and cooling system required shown in Figure 35.

The depth of Mindola Sub-Vertical shaft new mining block is at 5220L to 6000L, therefore changing the 5220L into feet is equal to:

1ft = 0.305metres

5220ft Level = (5220 * 0.305)

= 1,592metres below surface

The depth of the new mining block is below 1,500m, therefore refrigeration need to be considered as indicated in Figure 33.

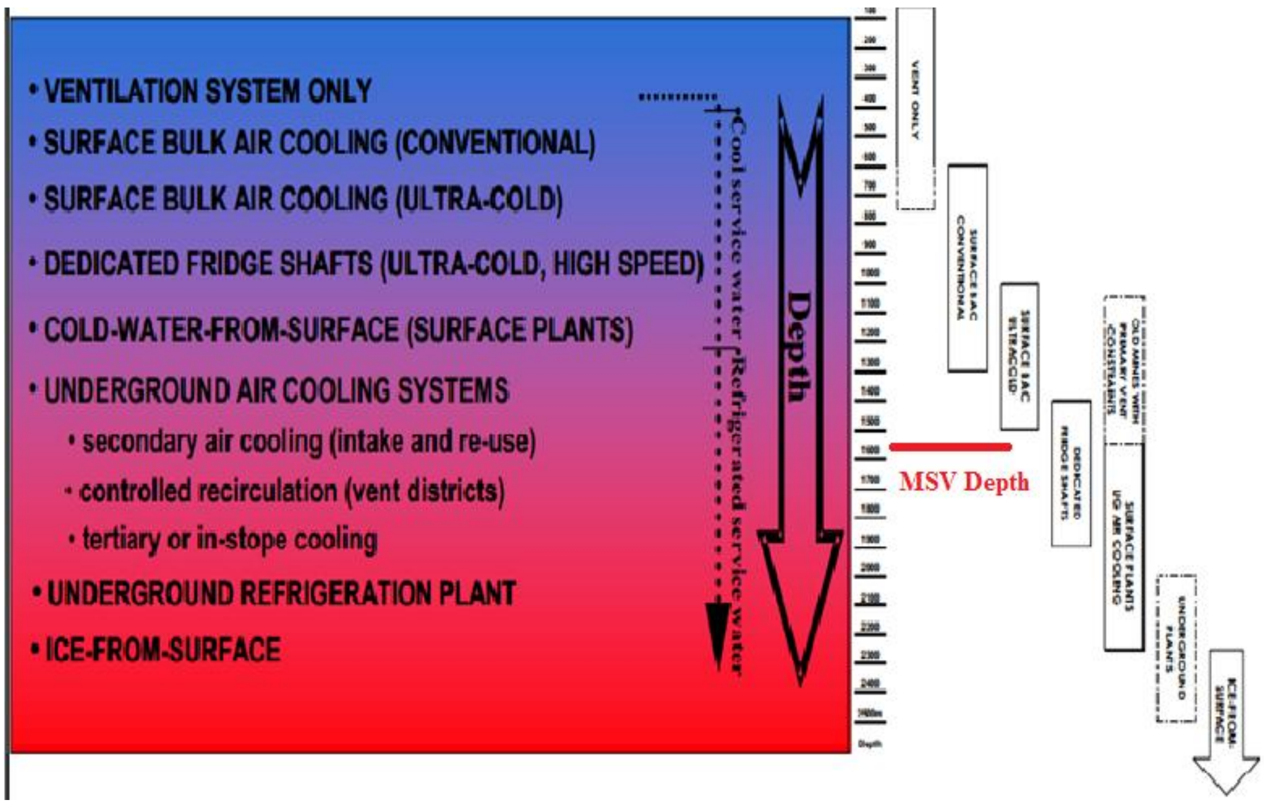

Figure 33: Relation between mining depth and type of refrigeration system (Bluhm and Glehn, 2010)

8.4 Surface Cooling Plant

The study proposed that MSV should have surface refrigeration plant (Bulk air heat exchangers), with a capacity of 25,500 kW of refrigeration, to feed a two-cell bulk air cooler in a closed circuit.

Bulk air heat exchangers in underground operations are used either to cool air when maximum legal return temperatures from the working areas (reject temperatures) are reached, or to cool water from underground refrigeration plants (heat rejection). These heat exchangers form part of two underground water refrigeration circuits, namely the evaporator circuit where chilled water is distributed to air coolers and the condenser circuit where heat is rejected to return air *(Burrows et al., 1989).*

Surface installations are normally first choice due to the following advantages:

- Generally surface installations have lower CAPEX and OPEX.
- Surface installations are easier to maintain.
- Construction time for surface installations is less
- Normally there are fewer size constraints on surface and Infinite amount of air available for heat rejection *(Russell Ramsden Frank von Glehn, 2012)*

Figure 34 shows layout for surface refrigeration plant.

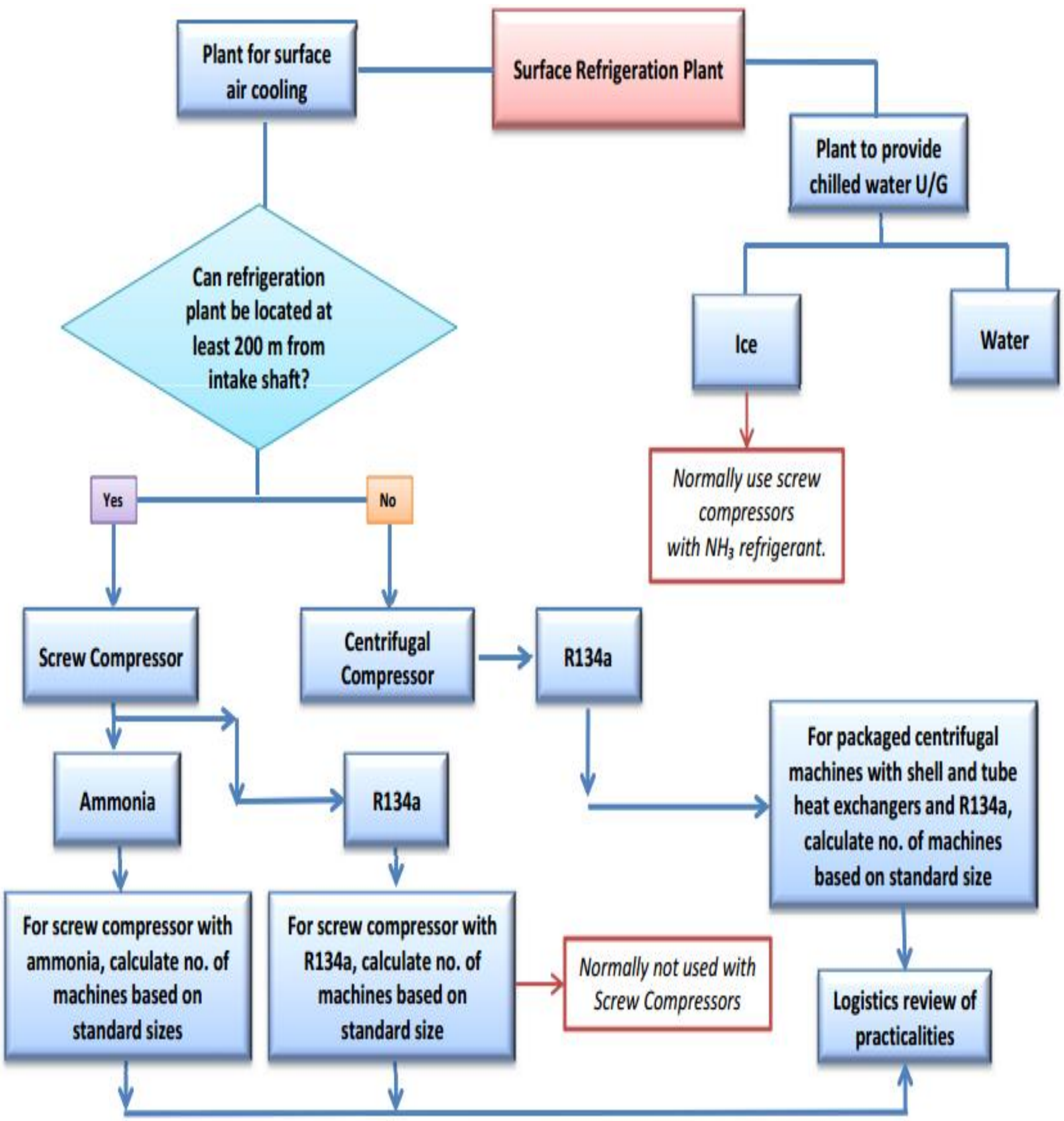

Figure 34: Location of refrigeration machines (Russell Ramsden and Frank von Glehn, 2012)

8.5 Refrigeration plant Capacity

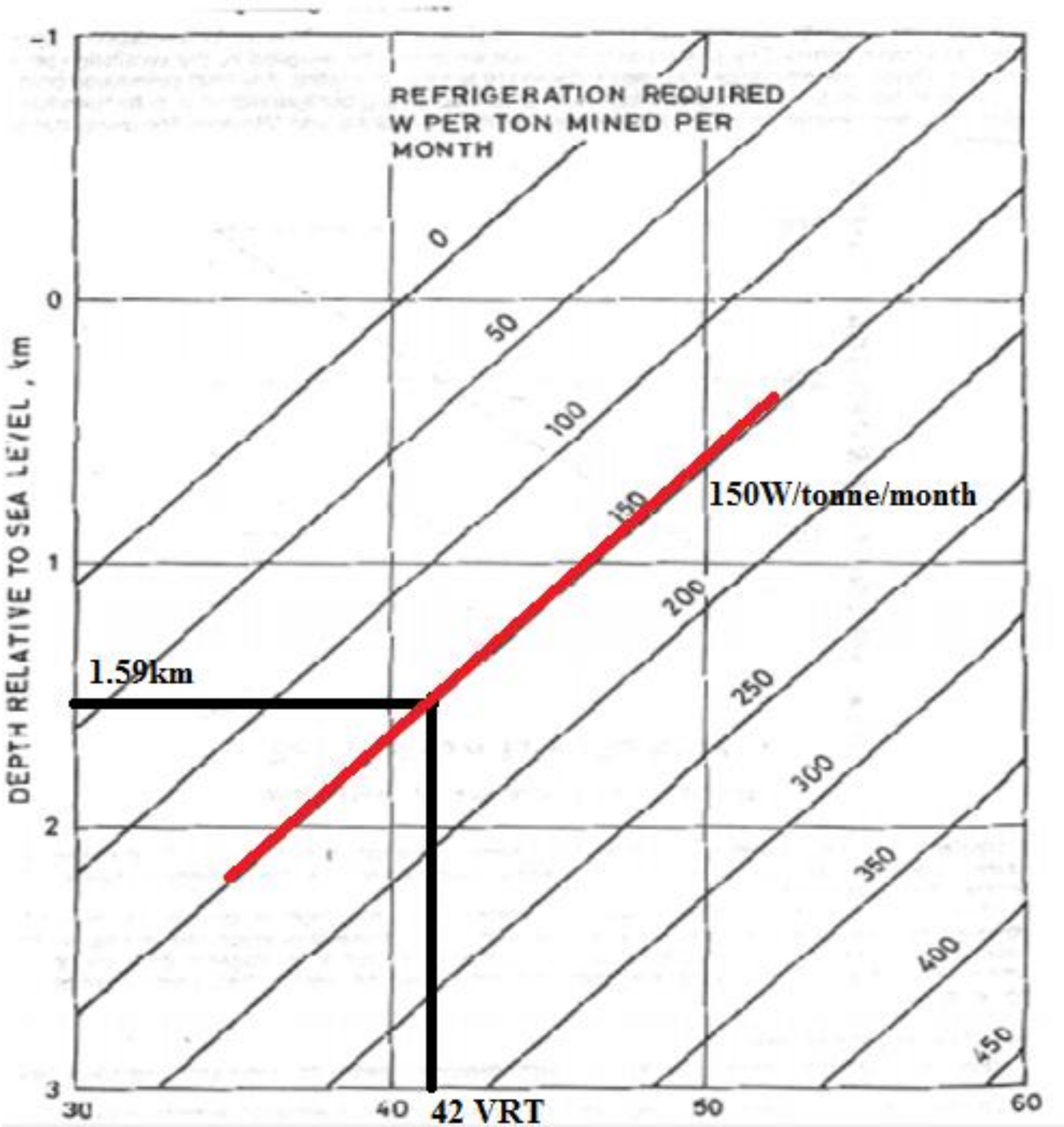

Figure 35: Method for estimating refrigeration requirement proposed by Whillier 1974

Given the following parameters as depth, virgin rock temperature and production rate, the W per month for MSV can be determined using the refrigeration requirement method in Figure 35:

- The depth of MSV=1,592 metres below surface
- Virgin rock temperature = 42 ^{o}C

Production = 2,000,000 tonnes/12months

- Production/month= 170,000 tonnes/month

Using Figure 35, W per tonne per month is 150W/tonne/month. If the production is 170,000tonnes/month then refrigeration capacity, W/month is (150*170000) = **25.5 MW**

The refrigeration plant must be on surface as shown in Figure 36.

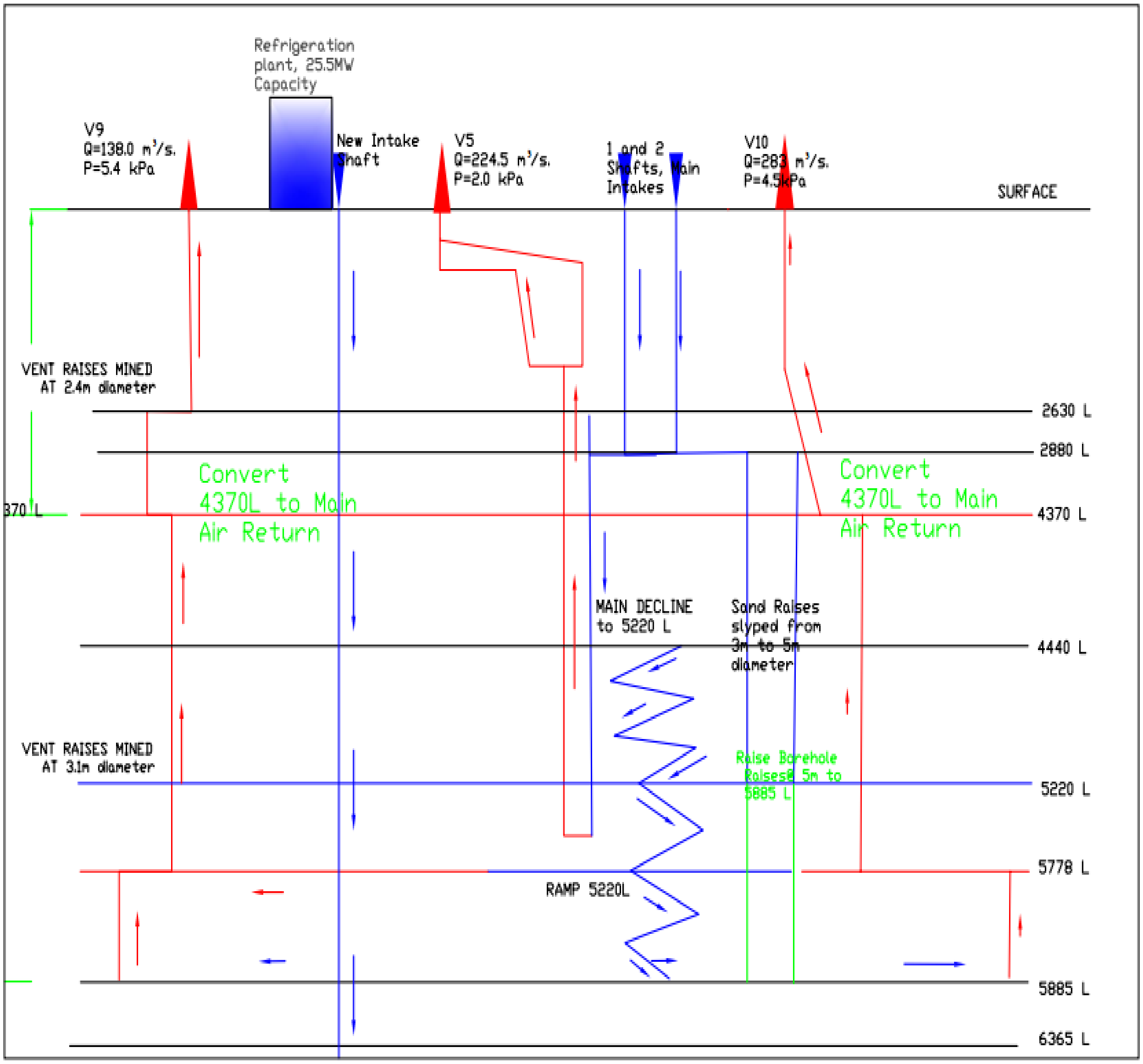

Figure 36: The refrigeration plant on surface

A mine surface cooling system forms an important part of the integrated water reticulation system. The basic layout of a generically simplified cooling system as integrated with a mine water reticulation system is shown in Figure 37.

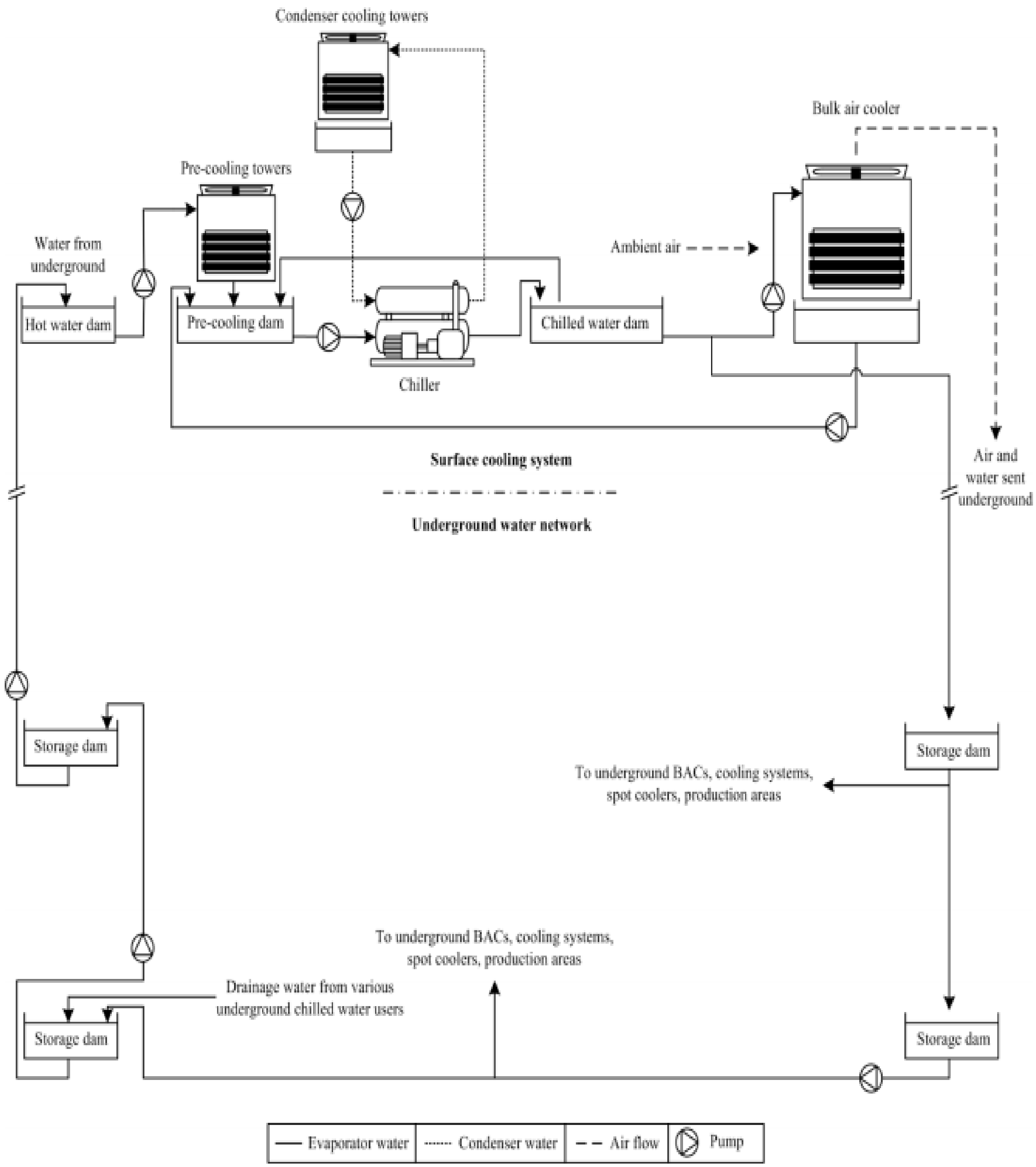

Figure 37: Schematic layout of a typical deep-mine cooling and water reticulation system, (Schutte, 2014)

An example of refrigeration plant that Mopani must consider is shown in Figures 38, 39 and 40.

Figure 38: Surface refrigeration supplying cold water underground (Ramsden & Von Glehn, 2012)

Figure 39: Surface refrigeration plant (Ramsden & Von Glehn, 2012)

Figure 40: Surface refrigeration supplying cold water underground (Ramsden & Von Glehn, 2012)

CHAPTER NINE: FINDING AND DISCUSSION

9.1 Introduction

- The findings of the study indicated that the major contributors of heat to the mining environment is Auto compression, Diesel Units and Rock strata or Geothermal heat as can be seen from the bar chart shown in Figure 41.

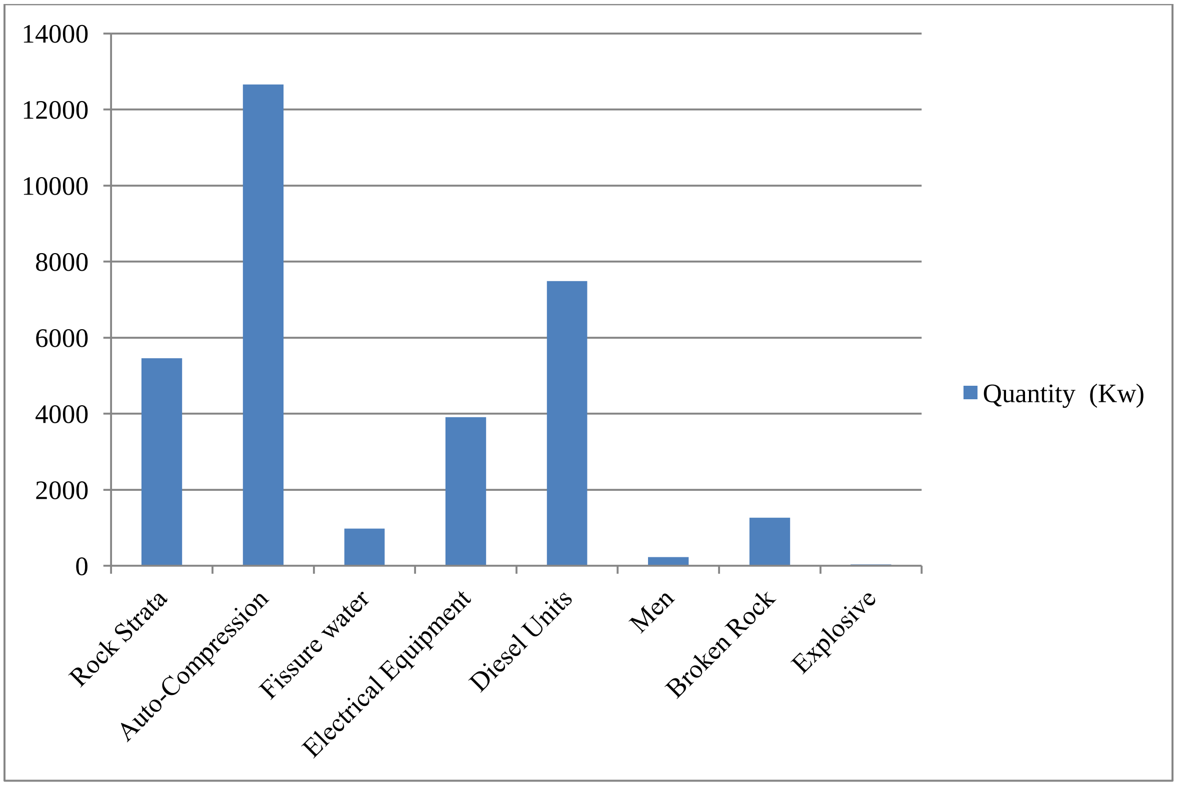

Figure 41: Summary of heat loads

- The findings of the study were also that the average temperatures at Mindola Sub-Vertical shaft were as high as 35.0°C and 32.5°C dry and wet bulb respectively. The Zambian Mine Regulation IX demands amongst other things, temperature of not more than 31.0°C wet bulb. This clearly indicates that temperatures are higher than the allowable temperatures by the Mining regulation.

- The study results indicated that total quantity of heat for the mine was 32,033 kW and ventilation air required was 826.45m^3/s. The study further established that the current up casted from the mine of 624.00m^3/s was not enough to provide adequate ventilation conditions. The mine further requires 202.24m^3/s to be uncasted for adequate ventilation conditions for both men and machinery operation. The V9 shaft main up cast fan should be upgraded by introducing another fan to increase the volume quantity by 202.24m^3/s.

- The findings reviewed that the old vent raises in the upper levels where mined at 2.4metres diameter and the new raises in the deeps are mined 3.1metres hence causing improper flow of air.

- The findings of the study also established that the Mindola sub vertical shaft require refrigeration to be installed. Due to the depth at which mining at MSV is carried out of more than 1500 metres, the virgin rock temperature of 42 ^{o}C and the production rate of 170000 tonnes per month, refrigeration of plant capacity of 25.5MW is highly recommended.

- Furthermore the finding established that there is recirculation of air in the vent network causing foul air to mix with fresh air. Leakage points observed (unwanted openings) where as shown in Table 37 as at 06/08/2015.

Table 37: Showing leakage points

LEVEL	TOTAL SEALS REQUIRED	SEAL DONE	VARIANCE
4552'L	27	5	22
4716'L	41	17	24
4881'L	42	1	41
5045'L	41	7	34
5220'L	43	1	42

CHAPTER TEN: CONCLUSIONS & RECOMMENDATIONS

CONCLUSION

1. The study has established that the total quantity of heat generated in the mine is 32,033.21kW and the quantity of air required to dilute this heat is 826.45m^3/s. The said quantity is 202.24m^3/s more than the current down casted air of 624.00m^3/s.
2. Based on point (1), a new ventilation system has been designed (Figure.32). The calculated air quantity of 826.45m^3/s will be pushed through the new downcast shaft, Sub-Vertical shaft, and the raise boreholes raises from 5220L to 5880L.
3. Based on the study, there is need to employ refrigeration system at MSV due to increased depth of mining currently standing at 1600m and increased heat load being generated (32,033 kW). The typical design with a capacity of 25.5MW has been proposed based on methods for estimating refrigeration requirement proposed by Whillier, 1974.

RECOMMENDATIONS

In order to reduce or control high temperatures at Mindola SV the following must be done:

- . The V9 shaft main up cast fan should be upgraded by introducing another fan to increase the volume quantity by 202.24m^3/s.
- Installation of booster fans on 5778'L to operate in series within the main Upcast fans so as to improve on suction pressure at lower levels.
- Conversion of the 4370L into the main return airway (MRAW) in order to increase on the positional efficiency of the upcast primary circuit.. Opening up of restrictive points in the circuit such as the 9 x 2.4m diameter connecting raise currently chocked up with ground to effectively link up the deeps to the primary circuit and the also the lashing out the restrictive ground. This must be followed with installation of brick walls to cut off old extraction drives from down casting of foul air into MRAW.
- The return air raises in old upper levels which were mined at 2.4m diameter must be slipped to current size of new raises at 3.1m diameter to avoid choking which results in short circuiting and recirculation of air in both the primary and secondary circuits.

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APPENDICES

1.0 Ventilation Standards

		Mindola Surbvertical	
	WASTE DEVELOPMENT	**DEVELOPMENT**	**PRODUCTION**
Force delivery quantity ($m^3/s/m^2$ of face)	>0.15	>0.15	N/A
Force distance to face (m)	<20.0	<20.0	N/A
Force column leakage (%) per 100m	<10 %	<10 %	N/A
Force fan position (m)	9m from bull nose	9m from bull nose	N/A
Exhaust intake quantity ($m^3/s/m^2$ face)	2 x more than force quantity	2 x more than force quantity	>0.15
Exhaust distance to face (m)	<35.0	<35.0	
Exhaust column leakage (%) per 100m	<10 %	<10 %	
Exhaust fan position (m)	9m from bull nose on return side	9m from bull nose on return side	
Force exhaust overlap (m)	>10.0	>10.0	
Face temperature (°C WB)	<31.0	<31.0	
Intake dust (ppcc)	<200	<200	
Face dust (ppcc)	<350	<350	
Service water col. distance (m)			
Electrical switch gear position	Through ventilation on intake side	Through ventilation on intake side	
Force exhaust system	Both sides of fan	Both sides of fan	

2.0 Air Contaminants Design Criteria

This is in compliant with the Mining Regulation 903 and as prescribed by the Chief Inspector of		
Mines (CIM)		
Description	**Compliance Limit**	
Respirable dust	200 p.p.c.c. at intake	
	350 p.p.c.c. at return	
Carbon Dioxide (CO_2)	7500ppm	
Carbon Monoxide (CO)	100ppm (2000ppm on diesel exhaust)	
Nitrous Fumes (NO_X)	10ppm (1000ppm on diesel exhaust)	
Sulphur Dioxide	20ppm	
Hydrogen Sulphide (H_2S)	20ppm	
Diesel dilution:	Air quantity of $0.05m^3/s$ per kW	

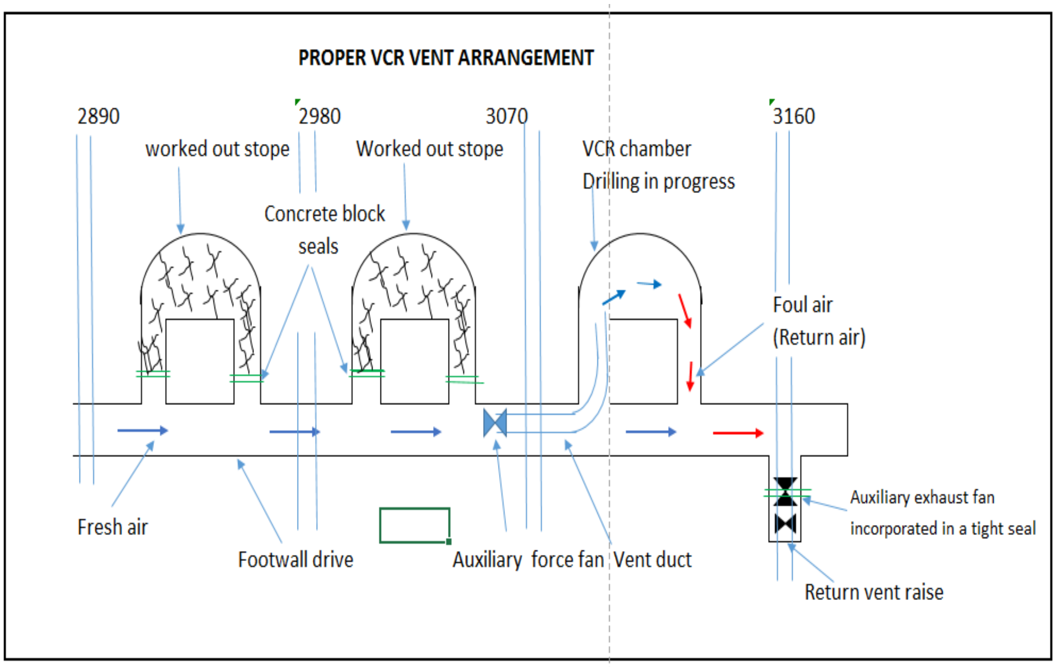
3.0 Acceptable Environmental Standards

The following are the general acceptable ranges of velocities in Primary airways that could create acceptable environmental conditions in the Mine		
Description of Area		**Velocity m/s**
Development ends /draw points		**0.5 - 4.0**
Ramp Raise		**2.5 - 8.0**
Conveyor Tunnels		**2.5 - 8.0**
Main Haulages		**2.5 - 8.0**
Main Shaft Crosscuts		**4.0 - 8.0**
Down Cast Shafts		**8.0 - 10.0**
Upcast Shafts (Concrete lined)		**18 - 22.0**
Upcast Shafts (Rough Rock		**18 - 22.0**
Return Ventilation raises (Concrete lined)		**15.0 - 20.0**
Return Ventilation raises (Rough Rock)		**10.0 - 15.0**

4.0 Existing exhaust ventilation raises

No	Vent raise location	Diameter(m)	Area (m^2)	Air velocity (m/s)	Maximum air carrying capacity (m^3/s)	Number of levels / sections served	Air volume requirements per section per level	Expected air volume for the raise (m^3/s)
1	2710N exhaust vent raise	2.4	4.52	18	81.4	1	32.8	32.8
2	3160N exhaust vent raise	3.1	7.55	18	135.9	5	32.8	164
3	1050S exhaust vent raise	2.4	4.52	18	81.4	4	32.8	131.2
4	1495S exhaust vent raise	3.1	7.55	18	135.9	1	32.8	32.8

5.0 Proper vent arrangement for VCR Mining method



6.0 Improper vent arrangements (prevailing in most section at MSV)