EFFECTS OF INCREASING REJECTION TEMPERATURES ON ELECTRICITY DEMAND FOR VENTILATION AND COOLING IN AUTOMATED METALLIFEROUS UNDERGROUND MINES

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A research report submitted to the Faculty of Engineering and the Build Environment, University of the Witwatersrand, in partial fulfilment of the requirements for the degree of Master of Science in Engineering

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DECLARATION

I,……………………………………………………………… declare that this research report is my work except as indicated in the references and acknowledgements. It is submitted in partial fulfilment of the requirement for the degree of Master of Science in the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination in this or any other University.

(Signature of Candidate)

Signed at ……………………………………………………

On the …………………………….. day of ………………………… 20…..

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ABSTRACT

The South African power crisis and corresponding rising costs experienced since 2008, created a paradigm shift in terms of electricity use. The mining sector is the second highest consumer of electricity with metalliferous mines being accountable for 80% of the total power. Recent studies revealed that underground ventilation and cooling accounts for 30% to 40% of total electricity costs in an underground metalliferous mine. Hence the need to look at ways to reduce electricity consumption in ventilation and cooling.

Work has been done on optimising efficiencies of ventilation and refrigeration systems of underground mines. Currently, the high energy consumption is driven by efforts to achieve a thermally acceptable environment for workers (manned) in deep metalliferous mines which is currently between 27ºC (wb) and 29ºC (wb). However, no detailed study has been done looking at increasing thermally acceptable environments for deep level metalliferous mine.

In this study the impact of increasing rejection temperature to 40ºC (db) was assessed in the automated (unmanned) scenario at a maximum depth of 2811 metres. Then the power demand was compared with the manned scenario.

The results proved that automation in an underground mine has the potential of reducing electricity cost of ventilation and cooling by more than 50%. For example, the production rate of about 200 kilo-tons per month yield an annual cost saving of R71 million on electricity. These cost savings can be used to justify automation systems. Automation removes workers from the hazardous environment and replaces them with equipment which can withstand harsher conditions.

The introduction of an automation system in underground mines would not come risk-free. Currently, automation systems have not yet reached a level of removing workers completely from underground.There are instances that would require workers to enter production zones. A Bow-Tie risk analysis was used to show the hazards that workers would be exposed to and prevention controls and responses to mitigate the impact the risks.

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LIST OF ABBREVIATIONS AND UNITS

- ALARP – As Low As Reasonably Practical
- BAC – Bulk Air Cooling
- BIC – Bushveld Igneous Complex
- CNG – Compressed Natural Gas
- CMU – Carnegie Mellon University
- COD – Cooling On Demand
- CoP – Code Of Practice
- COP – Coefficient of Performance
- DMR – Department of Minerals and Resources
- DPM – Diesel Particulate Matter
- eLHD – Electric Load Haul and Dump
- FFR – Fatality Frequency Rate
- FS – Free State
- Heg – Homogeneous Exposure Groups
- HIRA – Hazard Identification and Risk Assessment
- IARC – International Agency for Research on Cancer
- IMVC_2014 – International Mine Ventilation Conference 2014
- LHD – Load Haul and Dump
- MEC – Mineral Economic Centre
- MHSC – Mine Health and Safety Council
- NIHL – Noise Induced Hearing Loss
- NO_X – Oxides of Nitrogen
- NTDE – New Technology Diesel Exhaust
- OEL – occupational exposure limits
- PGM – Platinum Group Metals
- PM – Particulate Matter
- PPE – Personal Protective Equipment
- SDM – South Deep Mine
- ULSD – Ultra Low Sulphur Diesel
- VFSD – Variable Fan Speed Drives
- VOD – Ventilation on Demand

- VRT –Virgin Rock Temperature
- VUMA – Ventilation of Underground Mine Atmospheres

- ^{o}C – degrees Celsius
- kg – kilogram
- km – kilometre
- ktpm – kiloton per month
- kW$_{(R/e)}$ – kilo-Watt (R and e denotes refrigeration and electrical power, respectively)
- kWh(r) – Kilo-Watt-hour
- ℓ/s – litres per second
- m – metre
- m^2 – metre square
- m^3 – metre cube
- mg – milligrams
- ppm – Parts per million
- s – second
- W – Watts
- wb/db – Wet-bulb and Dry-bulb temperature
- µg – micro grams

CHAPTER 1: INTRODUCTION

South African metalliferous underground mines need to use electricity efficiently in order to reduce current energy demand, rising cost and associated carbon footprint. Mine ventilation and cooling systems require electricity to move air and cooling fluids in underground workings to create acceptable environmental standard for workers. These systems account for a significant portion of energy consumption, as mines are ventilated in a brute-force manner. Historically, electricity cost was low and did not have a negative impact on the operating cost of a mine. Hence, the mines could use brute-force ventilation strategy, where all the workings were ventilated and cooled all the time even when there were no employees or activities being performed underground.

The year 2008 can be seen as a defining point for a new era, where the electricity supply and low tariffs could not be guaranteed. The tariffs increased by more than 200% and are expected to increase further in future. This is becoming a threat to the economic viability of mines, especially underground metalliferous mines.

Furthermore, more emphasis is placed on the responsibility and accountability of mine owners in terms of health and safety of which a healthy working environment is a vital component. These challenges and others are forcing the mining industry to look at automation, as it is seen as a holistic approach.

1.1 BACKGROUND

Historically South Africa (SA) had a competitive edge to attract investment due to the abundance of cheap electricity. This was the backbone of the country's economic development which fuelled the booming of the mining sector in the early 20th century. For a very long time in this country, electricity cost never warranted spending time on critically analysing efficiencies and total cost of ownership of equipment. For example, mines used to purchase main fans for ventilation to deal with the 'worst case scenario' (Karsten and Mackay, 2012). Planning departments did not consider that a mine required different amount of air for ventilation throughout its life of mine. The air quantities required for ventilation were overestimated and fans with low efficiencies were used. Basically, this was a very wasteful strategy.

The power crisis and corresponding rising cost thereof experienced since 2008, created a 'paradigm shift' in terms of electricity use. The load shedding strategy implemented by Eskom in that year adversely affected the whole country. All sectors were requested by Eskom to reduce electricity consumption to address shortages of power supply. For example, to prevent blackouts, the mining sector was requested to operate at 10% below their normal electricity consumption. Over and above this, the electricity tariffs were increased to enable Eskom to expand its fleet of power stations (Pooe and Mathu, 2011). It is estimated that the electricity cost has increased by more than 200% in the past six years and it is expected to increase further in the next three years as per the Multi-Year Price Determination 3 (MYPD3) illustrated in Figure 1 below. The MYPD3 shows that in 2017 electricity will cost 89.13 cents per kilo watt hour (c/kWh). Power shortages and tariff increases were unprecedented and it simply means that the country must focus on using electricity more efficiently, to ensure economic growth.

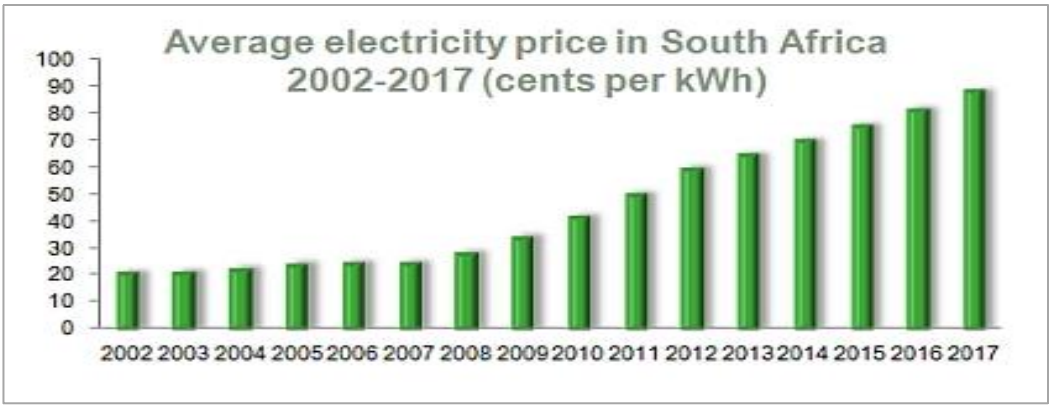


Figure 1: Average electricity price in South Africa (Ramayia, 2013)

1.2 PROBLEM STATEMENT

The mining sector in South Africa is the second highest consumer of electricity produced by Eskom in terms of gigawatt hour. The sector consumes 27.29% of gigawatt hour generated making it heavily dependent on electricity, see Figure 2 below. `

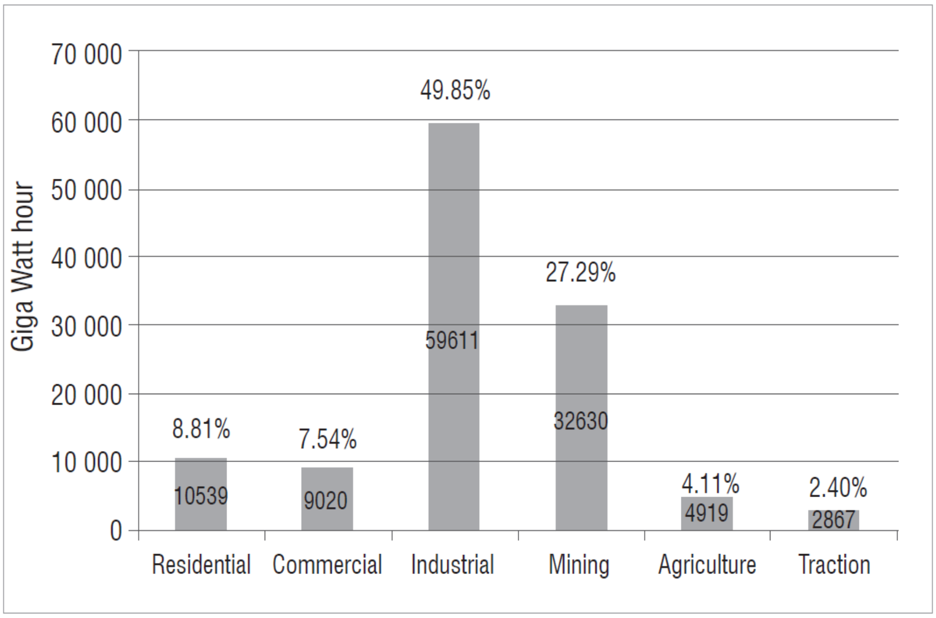


Figure 2: Energy consumption by sector (Thopil and Pouris, 2013)

Different commodities have different power requirement within the mining sector. For example, some commodities such as coal have low exposure to electricity cost as it uses less electricity.

Figure 3 below shows the breakdown of commodities and associated power consumption in the mining sector. It is clear from the graph that gold and platinum mines take about 80% of electricity in the mining sector. This implies that the business models of these commodities are more sensitive to electricity availability and tariff increases. It goes without saying that depth and processing of these commodities attribute to the high demand of electricity.

The shortage and increasing tariffs of electricity will render some of the mining project unviable, especially underground metalliferous mines. This will have a snowball effect on the economy, which relies heavily on mining to earn foreign exchange.

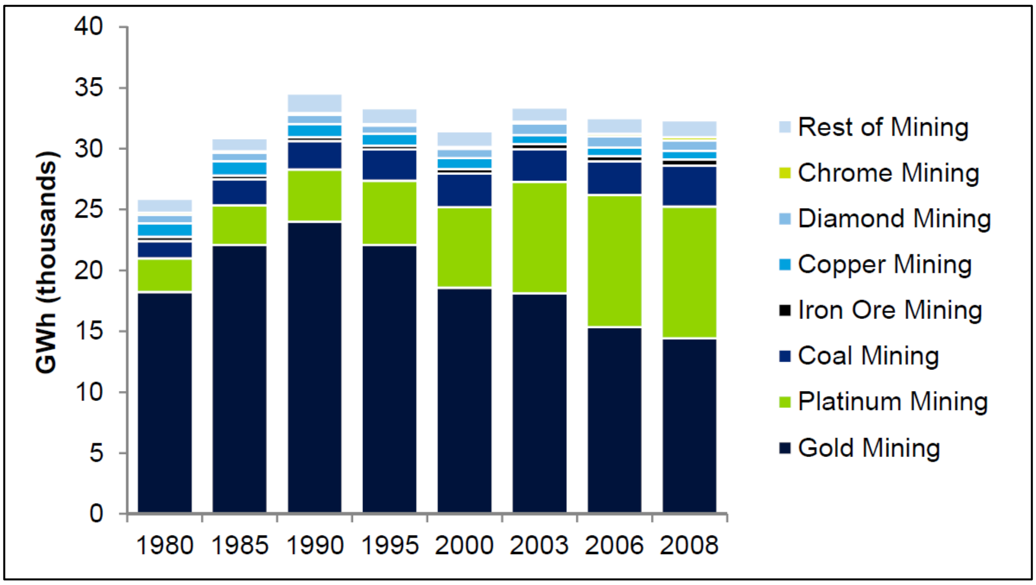


Figure 3: Energy consumption in the mining sector (Deloitte, 2009)

One of the major day-to-day challenges facing South African underground mines is heat. The temperature increases dramatically as the mining depth increases. This causes great difficulty in creating and maintaining comfortable working conditions for both humans and machines (Eskom & Gold Fields, 2010). Hence, the deep level mines adopt unique cooling methods which require power to function. Typical electricity cost drivers in gold and platinum mines are illustrated in Figure 4 below. It is clear from the graph that ventilation and cooling combined is the largest contributor to electricity cost.

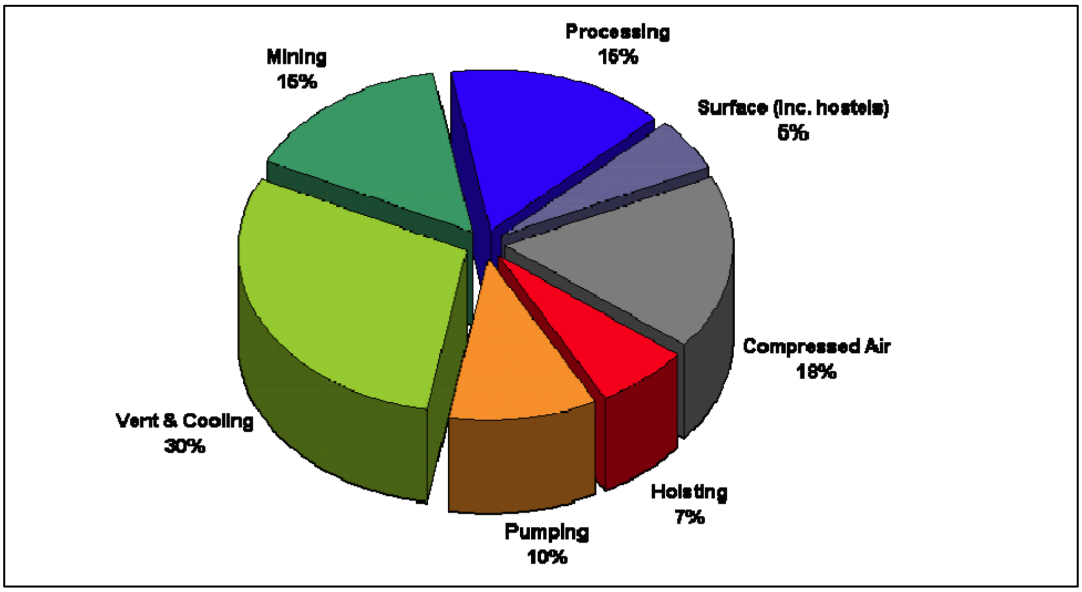


Figure 4: Distribution of energy costs between the various mine processes (Anon., 2008)

The mining industry globally has been looking at innovative ways of using energy efficiently and productively to reduce the cost of electricity for ventilation and cooling. Goldfields together with Eskom (Eskom & Gold Fields, 2010) looked at changing the fan's blade material in order to manipulate the thermodynamics properties and thus reducing energy requirements. It was reported during the 10^{th} International Mine Ventilation Conference in August 2014 (10^{th} IMVC2014) that this was not feasible as composite material could not stand underground environments and also support by Belle (2008). Kevin Lownie, Howden fan specialist, also pointed out that the notion of using lighter material on fan blades will only reduce electricity demand during start-ups. The notion of replacing steel blade with composite material was thought to have a positive impact on energy consumption by the fan. However, further studies have shown that the gain would be at the start-up of the fan and no further benefits would be realised from composite blades material.

Kocsis et al. (2003) looked at Ventilation on Demand (VOD) in the Canadian context. Their work focused on VOD and activity based ventilation which requires tracking and monitoring of workers and equipment underground. The work was found to be feasible as it demonstrated that reduction in energy consumption and cost is possible. However their work cannot easily be adopted in South Africa, because according to Marco Biffi, principal engineer of Anglo American, during a personal discussion on 25 August 2014, the Labour Unions are not yet amenable to tracking of their members during working hours.

It should be noted that apart from high energy cost, the mining industry is facing other challenges such as increasing depth of mining, skills shortages, productivity and pressure to improve 'health and safety' among others. One has to find a holistic approach to address these issues.

Currently the mining industry is looking at automation to address most of these challenges. One of the advantages of automation is that workers can be removed from the hazardous environment. This could create an opportunity to lower the environmental conditions as equipment can withstand a harsher environment compared to employees. This is the basis for this research and its main purpose was to assess the impact of automation and increased rejection temperatures on electricity demand.

1.4 OBJECTIVE

The focal point of this research project was on a gold mine with massive ore deposits similar to the South Deep Mine situated in the Magisterial Districts of Westonaria and Vanderbijlpark (Gauteng Province), some 45km southwest of Johannesburg in the Republic of South Africa.

The main objective of this research was to assess the effect of increasing rejection temperature in an automated underground gold mine on electricity demand. The following was considered:

- Need for automated underground operation
- Suitable environment for equipment
- Ventilation of underground automated operation versus conventional operation
- Ventilation on demand
- Risk assessment required to change to automation

The study was motivated by the following: shortage of electricity supply and associated tariff rise which increase operational costs of mining operations, failure to achieve zero harm and poor productivity level in the SA mining industry.

1.5 SCOPE OF STUDY AND METHODOLOGY

This report was confined to massive gold ore-bodies such as at the South Deep mine. Though platinum was not the main focus, the report briefly touched on its ventilation strategies, which are similar to that of gold mines and would therefore also benefit from this study.

To address the objectives of this study, the following was undertaken:

- Development of an appropriate ventilation model for gold mine using VUMA software.

- Establishment of electricity demand for both manned and automated environments.

The model developed formed the basis for future research and development. This model was mainly based on the South Deep Mine conditions in terms of the deposit and mining method. South Deep is mainly a mechanised operation with pockets of automation. However, the theoretical model that was looked at in this report simulated the underground environment as being fully automated. The reason for simulating is that there are no gold or platinum mine operations in South Africa that are fully automated. The sources of energy for underground mining equipment were limited to diesel and electricity.

The following methods were used to collect information:

- Literature research
- Site visits (South Deep Mine, Kiruna (LKAB) and Boliden mine in Sweden)
- Attendance of conferences and workshops
- Record analysis (electricity consumption, shift utilisation and heat sources) in typical gold and platinum mines
- Conducting interviews
- Making use of computation, qualitative and quantitative methods to present findings.

1.6 DELINIATION

This study does not include the following:

- the development and installation of automated underground control system including communication infrastructure
- capital cost of automation
- quantification of air required for internal combustion of diesel machines
- alternative power sources (e.g. solar, wind, nuclear, battery etc.)
- narrow reefs
- effect of automation on job creation

1.7 STRUCTURE OF THE REPORT

This report is made up of five main chapters:

- Chapter 1 is the introduction of the topic and outlines problem statement, objective, methodology, scope of study and the structure of the report.
- Chapter 2 is the literature review which has subsections focusing on the underground ventilation including ventilation on demand, mine automation, health and safety and risk assessment.
- Chapter 3 is about the methodology used in this study and a detailed background to VUMA software and modelling; mining methods used in underground metalliferous mines and risk assessment
- Chapter 4 provides the results and discussion obtained from the model and risk assessment.
- Chapter 5 is the conclusion and recommendations for future work. .

CHAPTER 2: LITERATURE REVIEW

2.1. BACKGROUND

Mine ventilation is an engineering control system which ensures that acceptable environmental conditions are achieved and maintained in underground operations for employees and equipment (Kocsis, 2009). To make the underground environment thermally acceptable for employees a deep gold or platinum mines spend about 30 to 40% of its electricity cost on ventilation and cooling (Bluhm, 2008) and (Belle, 2008). The figures are expected to increase as mines get deeper. This justifies time and resources to be spent on reducing power demand for underground operations. Hence, this project explored opportunities of increasing the rejection temperature in automated mines.

In depth research has been done on human beings in terms of tolerable ambient temperature and associated heat. The recommended rejection wet-bulb (wb) temperature lies between 25 and 29^{0}C (Burrows et al., 1989).

The shallower resources have been depleted and it will be a challenge to operate mines profitably and safely at ultra-deep levels. Research is being carried out to look at automating mines, especially metalliferous ones, in order to improve health and safety, as well as productivity. Despite the growing interest in underground automation there has been no study that seriously looked at an acceptable environment for machinery.

2.2. UNDERGROUND MINE VENTILATION AND REFRIGERATION

Ventilation is defined as the process of conducting an adequate flow of pure, fresh air along airways, working places and service points underground and refrigeration is a process of cooling air (Kocsis, 2009).

The main aims of ventilation and refrigeration can be summarized as follows:

- “to provide air for underground employees,

- to dilute the concentration of explosive and toxic gases, fumes and radon to environmentally safe levels and to remove them from the mine,
- to dilute the concentration of airborne dust to physiologically acceptable levels and to remove it from the mine workings, and
- to provide a thermally acceptable environment in which persons can work without undue discomfort or any danger of heat exhaustion and to remove heat from the mine as may be necessary"(Bakker, 2010).

2.3. VENTILATION AND REFRIGERATION PLANNING

During the 10th International Mine Ventilation Conference held in South Africa in 2014 (IMVC, 2014), it was established that most of the ventilation practitioners around the world follow a systematic approach in planning ventilation and refrigeration requirements of a mine. This systematic approach was developed by Burrows *et al* (1989) and is outlined schematically in Figure 5 below:

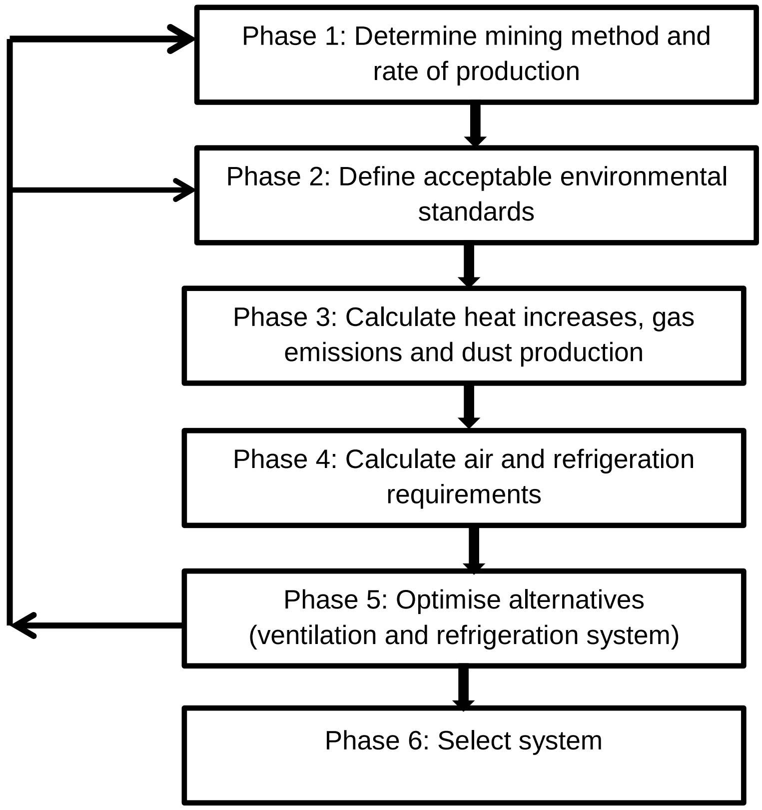


Figure 5: A systematic approach to ventilation planning (Burrows et al., 1989)

2.3.1. MINING METHOD AND RATE OF PRODUCTION

The first phase of ventilation and refrigeration planning is the determination of a mining method and rate of production. This phase is important as it establishes the inherent environmental conditions. Apart from developing a suitable mining method, rate of production and quantifying the life of mine at this phase, potential hazards such as sources of heat, gases, dust, and radiation are identified.

Mining methods depend on the geology of the ore-body. Most of the gold and platinum deposits are deeply buried and require underground mining methods. The ore-bodies can be classified as narrow reef and massive ore-bodies which require different mining methods. The gold deposits have a generally steeper dip-angle than platinum which make them more difficult to mechanise.

The ventilation and cooling strategy depend on the depth of mining. Figure 6 below, shows which ventilation and cooling strategy should be used in relation to critical depth. Critical depth is the depth below surface at which air will exceed the underground target wet bulb temperature solely through auto-compression (Payne & Mitra, 2008).Temperatures below rejection level can be maintained by using ventilation (i.e. air only) to remove excess heat but only to a critical depth in a range of 600 to 800 metres in the Bushveld Igneous Complex (BIC) and around 1250 meters in the Witwatersrand for zone 1.

The second zone is where ventilation and Bulk Air Cooling (BAC) must be applied. This zone start from the critical depth of zone 1 (i.e. 600 to 800 metres) and is up to a critical depth ranging from 1300 to 1500 metres in the Bushveld Igneous Complex and is around 2000 meters in the Witwatersrand. In the BIC, once the mine reaches a depth of 1800m below surface BAC is not efficient due to poor positional efficiency and this leads to the introduction of underground refrigeration plants. The third zone which is around 2000m in BIC and more than 2500 in the Witwatersrand is where only refrigeration (i.e. mainly ice plants) should be applied. These critical depths are based on the rejection temperature for workers and may change for automated mines.

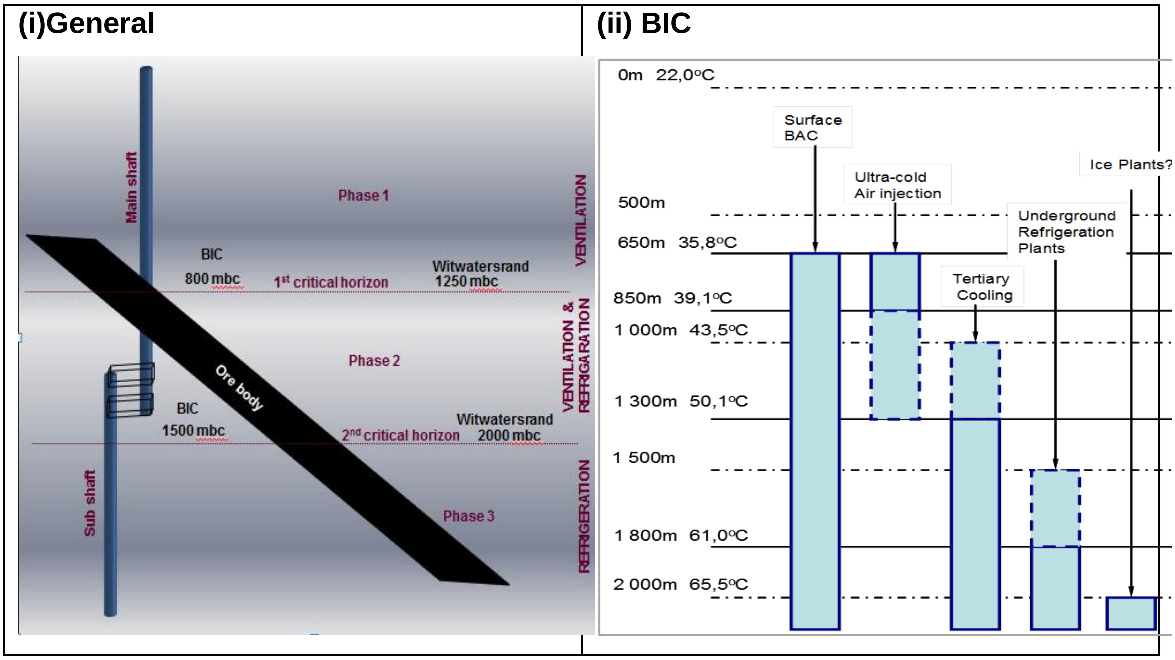


Figure 6: (i) Environmental control strategy for various critical depth zones (Burrows et al., 1989), (ii) Platinum mines(BIC) ventilation and cooling strategy (Biffi, 2014)

The rate of production will assist in estimating the amount of air required using empirical methods as shown in Figure 8 below. When the depths of mining and rejection temperatures are established, the required amount of air can be extrapolated from the graph. Figure 8 is the graph for the Free State (F.S) but can be used for the Witwatersrand as it will give a good estimate.

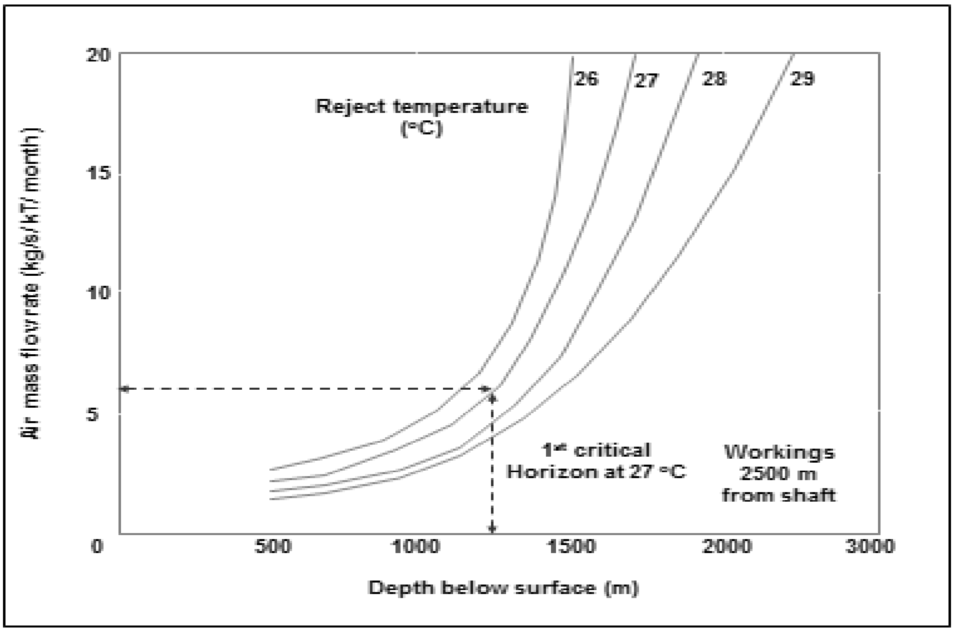


Figure 7 : Air requirements for zone 1 in the F.S (Burrow et al, 1989)

2.3.2. ACCEPTABLE ENVIRONMENTAL STANDARDS FOR EMPLOYEES

The second phase is to define the acceptable environmental standards. In this stage it is important to establish occupational exposure limits (OEL's) for various gases, radiation, rejection temperature, etc. Traditionally, an acceptable environment has the connotation of "conducive environment for employees".

The legislation has a number of guidelines in terms of the OELs of dust and gases. Some of the examples are shown in Table 1 below and ventilation must be used to keep these contaminants below the stipulated OEL levels:

Table 1: Occupational Exposure Levels (OELs) (MHSA, 1996)

Dust	Gold mines (Crystalline silica)	0.1 mg/m^3
	Coal silica < 5%	2.0 mg/m^3
	Platinum silica < 5%	3.0 mg/m^3
	PNOC(no toxin identified)	3.0 mg/m^3(respirable) 10 mg/m^3(inhalable)
Gases	Flammable gases (CH_4,H_2)	1.4%
	Carbon monoxide (CO)	30ppm
	Nitric oxide (NO)	25ppm
	Nitrogen dioxide (NO_2)	3ppm
	Sulphur dioxide (SO_2)	2ppm
	Hydrogen sulphide (H_2S)	10ppm

The other important factor that needs to be addressed in this phase is the rejection temperatures.

2.3.2.1. REJECTION TEMPERATURE

In order to define thermally acceptable environment, previous studies carried out on workers found that the rejection temperature of 27.5^oC wet bulb (wb) was adequate (Burrow et al, 1989). The rejection temperature was based on the heat stress limit for essentially nude un-acclimatized men working in the gold mine (Burrows et al., 1989). Acclimatised men can work up to a temperature of 34^{o}C (wb) depending on metabolic rate of work, see Table 2 for details:

Table 2: Maximum air temperature for various work rate(Le Roux, 1990)

Metabolic Rate of Work (W/m^2)	Maximum wet bulb temperature for acclimatised men (^{o}C)
Hard Work – 240W	32.5
Moderate Work – 180W	33.0
Light Work – 115W	34.0

It is crucial when establishing rejection temperatures to look at productivity of employees. Figure 8 below; demonstrate the relationship between temperature and productivity. When reading the graph(s) it is also important to note that the cooling power of the air also depends on the speed of the air.

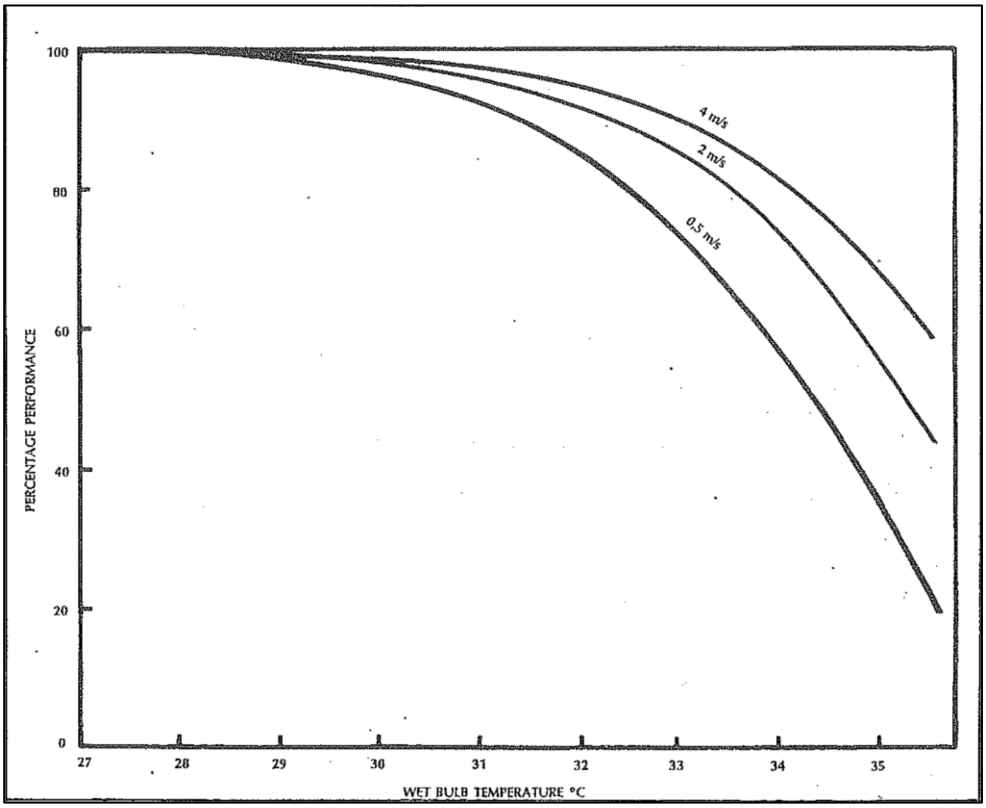


Figure 8: Performance vs Wet Bulb Temperature(Le Roux, 1990)

The graph illustrates that when workers are exposed to underground conditions, the rejection temperature of 27.5^{o}C (wb) would enable both acclimatised and un-acclimatised workers to perform at 100%. The performance of workers decreases when the wet bulb temperature reaches 30^{o}C (wb) and is seriously affected when the temperature is above 32^{o}C (wb). Maintaining rejection temperatures of working areas below 27.5^{o}C comes at an exorbitant cost given the electricity price hikes. This explains why most mines carefully choose a ventilation strategy which sets the rejection temperature at the upper scale, in a range of 28 to 29.5^{o}C (wb).

South African mines are required to draft and implement a mandatory Code of Practice (CoP) based on guidelines established by the chief inspector of mines in terms of section 9.2 of the Health and Safety Act. The mandatory thermal stress CoP is summarises in Table 3 below, it shows temperature ranges, interpretation and actions required.

Table 3: Thermal stress Code of Practice (DME, 2002)

Category	Temperature range (°C)	Interpretation	General Action
A	*twb* >32,5	Abnormally hot. Unacceptable risk of heat disorder	Risk assessment required to do the work
B	29 < *twb* ≤ 32.5	Potential heat disorder	Heat stress Management (HSM) mandatory
C	27.5<*twb*≤ 29	Economic range for acclimatized workers	HSM mandatory
D	twb ≤ 27.5	Risk of heat disorders is minimal for both un-acclimatised and acclimatised workers	No special precautions. Temperature monitoring

The mandatory CoP on thermal stress is a measure to limit employees to hot environment in order to avoid risk associated with heat.

2.3.2.2. HAZARDS OF HEAT STRESS

Heat stress is the combination of all the internal and the external heat factors which cause the body to become stressed. Internal factors that determine the level of heat stress on the body include core body temperature, acclimatisation, natural heat tolerance and metabolic heat generated by the workload. External factors include the surrounding air temperature, radiant heat, air velocity and humidity. Apart from heat reducing productivity of workers, it also affects their health and safety. There are various illnesses that may occur when workers are exposed to high levels of heat stress and it is crucial to be able to identify signs and symptoms of these illnesses (Burrows, et al., 1989).

2.3.2.2.1. HEAT DISORDERS

Heat stress causes conditions known as heat disorder. Heat disorders caused by hot environments vary from irritations, such as prickly heat, to fatal conditions. The most common types of heat disorders in the mining industry are:

Heat Stroke – is the most serious heat disorder that one can experience. It occurs when the body fails to control its temperature. It can be fatal if the person is not attended to urgently. Some of the symptoms are rising body temperature, sweating, confusion and hallucination. The major cause of heat stroke is strenuous work followed by suspected heat intolerance and dehydration and excessively hot environment (DMR, 2002).

Heat Cramps – These occur after sweating a lot while performing a strenuous task. This leads to excessive loss of salts and moisture in the body. Typical symptoms are muscle pains, spasm in the arms and legs.

Heat exhaustion – This occurs when the body tries to respond to loss of salts and moisture by sweating. The common symptoms are dizziness, nausea, feeling weak and headaches.

Heat Rash – This is a skin irritation caused by excessive sweating as a result of hot temperatures. The symptoms are red clusters of pimples or small blisters.

Heat Syncope – This is a fainting episode or dizziness caused by dehydration and lack of acclimatization. Typical symptoms are light headedness, dizziness and fainting.

In South African mining industry the major focus has always been on heat stress. However, it is also is also important to monitor cold environments such as down cast shafts and intake airways. Employees working in such environment may suffer from cold stress. The human body is constantly adapting to changes in temperature in the surroundings. The body requires a core temperature of ±37ºC. A human-being is comfortable when the environmental temperature is between 18ºC and 22ºC and the humidity is 45%. When temperature drops below 18ºC, the body tries to produce heat through muscle movement, this is when shivering occurs. Heat loss from the parts of the body such as hands, feet, limbs and head occurs (OHCOW. Inc., 2005).

When designing a mine it is important to ensure that all the workings have thermally acceptable environments by addressing both cold and hot environments.

2.3.3. ENVIRONMENTAL CONDITION FOR EQUIPMENT

The equipment that is being used in the mining industry currently are predominantly diesel and electrically powered machines in both mechanised and automated environments. These equipment, like people, require ventilation and cooling. However, machines can stand harsher conditions that human-beings and in most cases air ventilation is sufficient for them. Diesel machines require air for internal combustion and cooling while electrically powered equipment needs air for cooling only. However, the air quality does not have to be the same as the one required for workers, as machines have filters to clean the air which are replaced regularly. In spite of this there is a limit as poor air quality may have a detrimental effect on equipment. There are three elements among other that must be looked at in terms of the air required for machines namely; temperature, humidity and dust.

The International Council of Combustion Engine Working Group (2009) put together a position paper where they outline the following:

- High ambient temperature influences the performance of engine indirectly by negatively affecting the performance of the turbocharger and the engine cooling system.
- High humidity has a positive impact on the environment as it reduces the emissions of oxides of nitrogen (NO_X). However, it affects the efficiency of the engine in a negative way as it slows down the combustion speed as well as reducing the maximum combustion temperature (CIMAC, 2009).

- Dirty air will wear out a diesel engine rapidly. “Less than two tablespoons of dirt can rapidly wear internal engine components. ” (Anon, n.d.).

A diesel machine requires an environment that would supply adequate, clean and cool air. Diesel machines emit heat, moisture and dust that require ventilation air to dilute and remove to create a conducive environment.

2.3.4. CALCULATE HEAT LOAD, GAS EMISSIONS AND DUST PRODUCTION

The third phase is to quantify the heat, gas, radiation and dust that were established in the first phase. It is very difficult to estimate dust and toxicity at the planning stage, but experience from similar mines can be used to determine potential controls. Heat is one of the main challenges for underground operations and it is important to quantify the amount of heat load that the mine experiences in order to determine the cooling required.

2.3.5. CALCULATION OF THE HEAT LOAD

There are two different approaches which can be used to calculate the heat load. The first approach is based on theory and consists of adding up all sources of heat. The second one is an empirical method, which uses data from existing mine workings to develop appropriate models to predict conditions. The empirical method is widely used because it is based upon actual measurements from the mines (Burrows et al., 1989).

2.3.6. SOURCES OF HEAT

There are various sources of heat in mining but this report will focus on four main sources, which are:

- geothermal gradient
- equipment
- blasting
- auto-compression

These sources of heat in underground mine workings will be looked at in detail in the subsequent sub-sections. The critical parameter in defining the geothermal gradient is the virgin rock temperature (VRT). VRT is a function of surface rock temperature and depth, and it differs from area to area. An indication of the virgin rock temperatures, at specific depths, in some parts of South Africa is given in Figure 9 and in other parts of the world including South Africa is given in Figure 10. From the two figures, it is clear that the VRT in ultra-deep (>3500m) mining is in excess of 75^{o}C for the F.S and the (BIC) while Witwatersrand approaches 70^{o}C.

The formulae for calculating VRT for these areas can be summarised as follows:

Witwatersrand: $\mathrm{VRT} = 18 + (9.3 \times d)$............................***Equation 1***

Carletonville: $\mathrm{VRT} = 16 + (10.5 \times d)$..........................***Equation 2***

Free State: $\mathrm{VRT} = 20 + (14.5 \times d)$..........................***Equation 3***

Bushveld Igneous Complex: $\mathrm{VRT} = 18 + (20 \times d)$............................***Equation 4***

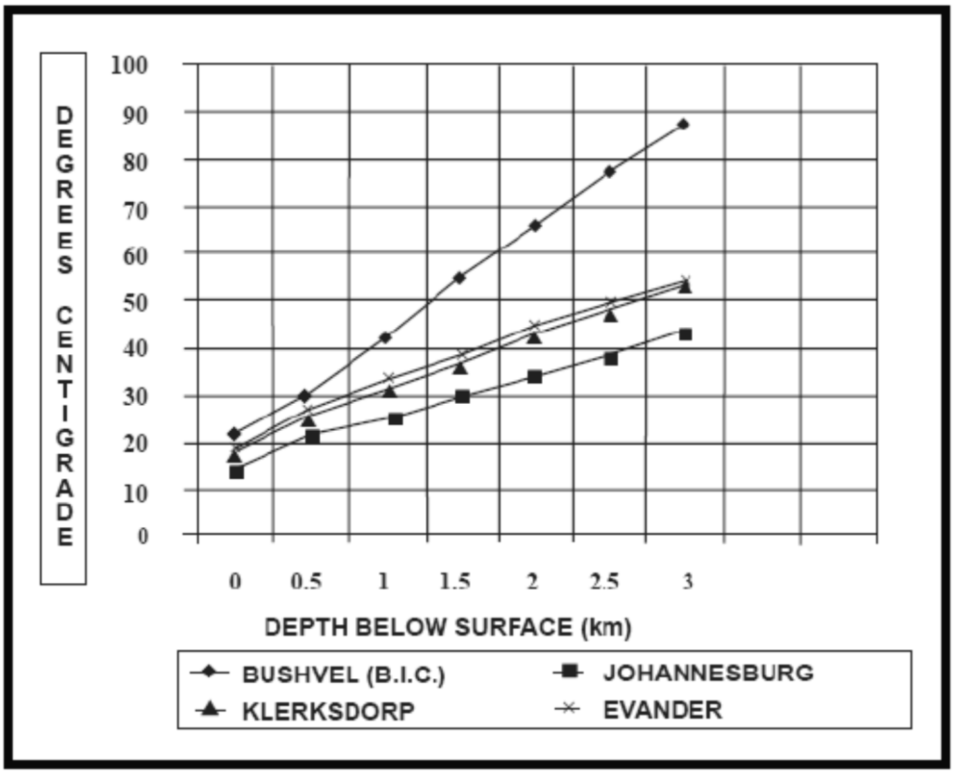


Figure 9: Virgin Rock Temperatures at depth (Stanton, 2004)

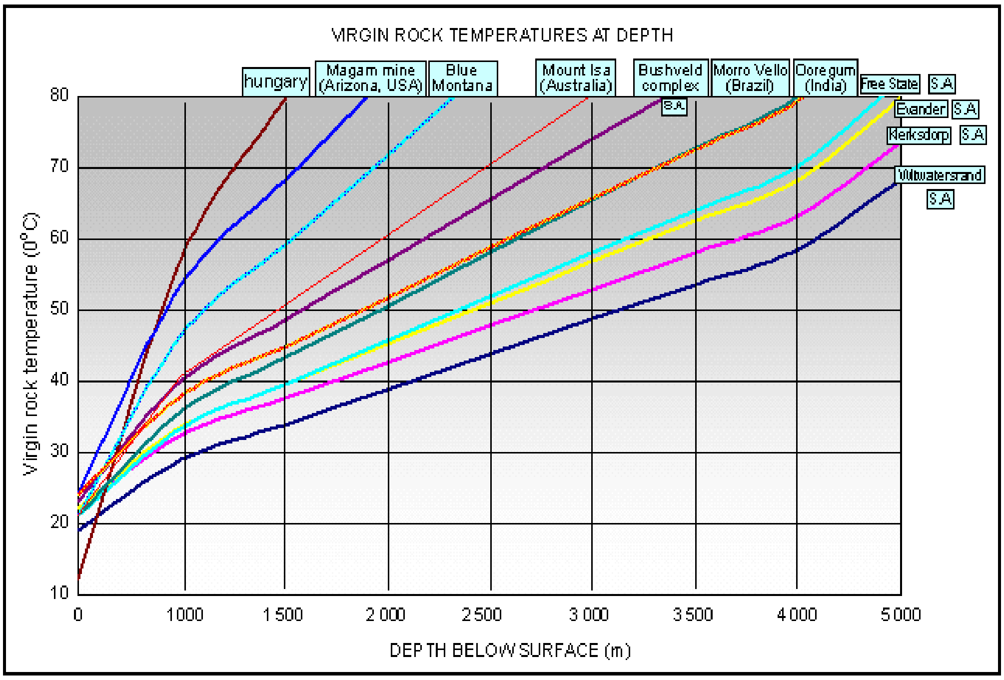


Figure 10: Virgin Rock Temperatures at Depth (Bakker, 2010)

2.3.6.1. GEOTHERMAL GRADIENT

The geothermal gradient is the rate at which the virgin rock temperatures increase with increasing depth. “The geothermal flow of heat emanating from the earth's core and passing through the earth’s skin has an average value of 0.05 to 0.06 W/m^2” (Mcpherson, 1992). “In practical utilization, it is often inverted to give integer values and is then referred to as the Geothermal step = dD/dθ and units are m/°C” (Mcpherson, 1992).

The heat is transferred from the rock to the air through conduction and convection (water fissures). This heat transfer is greater in newly established developments. However, the rock surface will cool gradually to an equilibrium point where the rock and air temperature are almost the same (Kocsis, 2009).

2.3.6.2. MINING EQUIPMENT

Mining uses electrical and diesel powered equipment and most of the energy consumed is converted into heat, equipment convert electrical energy and chemical energy (diesel) into heat directly through power losses and indirectly through friction when doing actual work (Burrows et al., 1989). Heat from equipment is one of the major sources in South African mines as the use of equipment is on the increase across the mining industry. Diesel equipment emits more heat than electrical. It is estimated that diesel will produce three times more heat than electrical equipment due to its overall efficiency of around 30% (Burrows et al., 1989).

2.3.6.3. EXPLOSIVES

Most of the chemical energy contained in explosives is converted into heat during blasting and can be considered as one of the significant sources of heat. Some of the heat enters the broken rock and is released over a long period (Burrows et al., 1989).

2.3.6.4. AUTO-COMPRESSION

This concept is not strictly speaking a source of heat. Auto-compression occurs when potential energy of the fluid is converted into thermal energy. This is extremely important in mine ventilation systems as it relies heavily on circulating fluids to remove excess heat within the underground workings. Auto-compression reduces the cooling capacity of the air. The increase in heat due to auto-compression is typically 9.79kJ/kg for every 1000m depth (Burrows et al., 1989). The temperature increase of the air can be calculated by using the following formula:

$\Delta T = \frac{(z_1 - z_2)g}{C_p} \,^\circ C$ ....***Equation 5***

Where;

- z = height above datum point (m) (subscripts 1 and 2 refer to inlet and outlet respectively)
- g = gravitational acceleration
- C_p = specific heat capacity

2.3.7. CALCULATE AIR AND REFRIGERATION REQUIRMENTS

The fourth phase is to calculate the air and refrigeration required to adhere to the acceptable environmental standards. It is important to note at this stage that the surface climate and weather conditions are also established as this will define psychrometric properties of the air that is taken underground. Mining depth, method, equipment and heat transfer will also attribute in quantifying air and refrigeration required (BBE, 2014).

2.3.8. AIR QUANTITY

The quantity (Q) of air supplied underground is a function of area (A) and air velocity (V), the formula is expressed as:

$Q = A \times V$...***Equation 6***

There are guidelines in determining air velocity for an airway. Excessive air speed may cause dust, discomfort and high operating cost. The other issue that must be considered is the fire hazard in the conveyor belt excavations.

Underground ventilation is made up of two components primary and secondary. Primary ventilation consists of main ventilation infrastructure such as main fans, main inline booster fans, up-cast and downcast shafts, raise bore holes, main airways, bulk air coolers, etc. Secondary ventilation is concerned with control of ventilation in working areas such as in-stope and development auxiliary fans, ducting systems, service ventilation of pump stations, tips, workshops, etc.

The Table 4 shows the design speeds as recommended by the workshop that was held during the 10th International Mine Ventilation Conference in August 2014 in South Africa (IMVC, 2014)

Table 4: Air speeds for various airways (IMVC, 2014)

Airway type	Commodity	Speed range (m/s)
Intake air ways	metal	5 - 7
	coal	2 - 5
Return airways	metal	Up to 12
	coal	3 - 5
Vertical unequipped shafts	all	16 - 22
Vertical equipped shafts	all	10 - 12
Inclined shafts	all	6 - 8
Conveyor belt excavation	all	2 - 3
Up-cast shaft not in range (critical velocity)	all	7 - 12

Quantifying the required amount of air for underground mine has never been an easy task. There have been many attempts to develop a scientific method to predict air requirements in a mine with little success. Most of the ventilation practitioners globally use empirical methods and computer softwares such as Ventilation of Underground Mine Atmospheres (VUMA) and Ventsim to determine the quantity of air and cooling required. The complexities of calculating air require is due to a number of factors that must be taken into account such as stoping width, face advance, wetness of the airway, thermal conductivity and diffusivity of rock, frequency of blasting, time of residence of broken rock, temperature and humidity of intake air way, length of face, etc. The use of computer models has made it possible to calculate future air requirement to a reasonable accuracy.

In South African metal mines after many years of experience the air quantity is estimated by multiplying a ventilation factor of between 3 to 6kg/s per 1000 tons mined per month (3 – 6kg/s/ktpm) by production rate (Burrows et al., 1989). Presenting one of the keynote addresses at the 10th International Mine Ventilation Conference, held in South Africa in August 2014, Mr. Marco Biffi pointed out that this factor has been increasing over the years. He referred to the work done by Belle (2012) which is shown in the Figure 11 below:

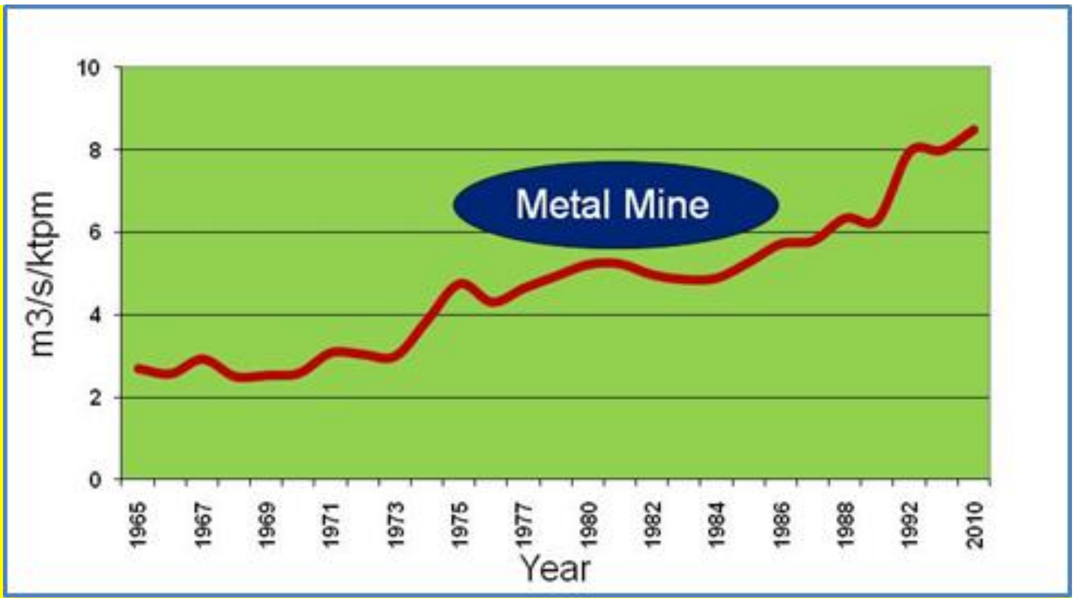


Figure 11: Timescale of ventilation demand in gold mine (Belle, et al., 2014)

Clearly from the above graph in Figure 11, it is important to take note of the units as the graph uses volumetric instead of mass flow rate. One has to do the conversion and this is achieved by multiplying the volumetric flow rate by density. For example, the graph shows that around 2010 the volumetric flow rate was 8.2m^3/s/ktpm, using a density of 1.2kg/m^3 the mass flow rate factor is 9.84 kg/s/ktpm. The explanation for the increase was, apart from metal mines adopting mechanisation, air leakage as the workings get further away from the shafts. He went further to say that the mines need to deal with the air leakages in order to reduce electricity cost.

The increased ventilation factor could also be the reason why gold and platinum mine in general are operating below a wet-bulb temperature of 32.5^{o}C, as shown in Figure 12 below. It is very interesting to note that the shallower deposits such as coal, diamond and copper have areas with wet-bulb temperature of more than 32.5^{o}C, though very small which is in a range of 3% to 22%. Gold, copper and platinum (PGM) have high exposure in rejection temperature of between 29^{o}C and 32.5^{o}C.

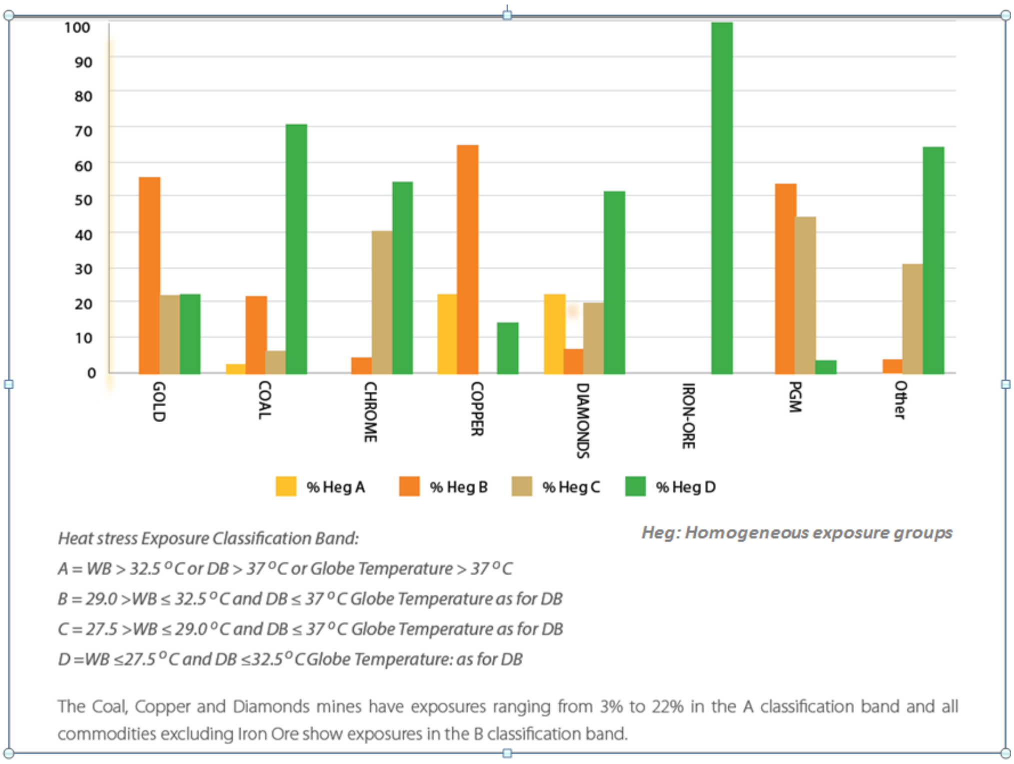


Figure 12: Heat stress exposure (DMR, 2013)

Power has a cubic relationship with air quantity and any amount of air reduction will result in substantial saving on electricity. This issue will be dealt with in detail in the later section of this chapter. The other important issue when dealing with heat is refrigeration as it is also one of the high cost drivers of electricity.

2.3.9. REFRIGERATION

Underground cooling can be done in a number of stages but the common known three stages are; primary, secondary and tertiary air cooling. Primary bulk air cooling constitutes large air coolers on surface or underground in main intakes where the air is cooled by chilled water for the first time. Secondary bulk air cooling constitutes mainly underground coolers in main intakes where the air is cooled by chilled air for the second time. Tertiary air cooling is mainly air coolers underground where the air is cooled by chilled water for the third time in its path.

Power estimation for refrigeration can be calculated by subtracting the net air cooling capacity from the heat load.

The rejection temperature is important in calculating heating load and cooling capacity. Hence, it is important to establish the rejection temperature when defining acceptable environmental conditions.

There are two factors that contribute in determining the actual capacity of refrigeration plant namely: location of the plant (underground or surface) and line losses. Sensible heat from underground workings is removed by cooled air, chilled water and, in some cases, by ice.

2.3.10. OPTIMISE ALTERNATIVES

The fifth phase is to optimise ventilation and refrigeration systems which lately have become very important due to shortages of electricity and its associated costs.

2.4. VENTILATION ON DEMAND AND COOLING ON DEMAND

Ventilation on Demand (VOD) is a concept of supplying adequate air at the required place when it is needed while saving on energy (Kocsis, 2009). There are two VOD strategies namely event and quality based. Event based is also known as activity based which means the air is supplied for the activity or event in a working area. Quality based is when the fans are switched on when the contaminants reach a certain level. This report will look at both because the machines must be supplied with air and occupational exposure limit of contaminants must be kept at acceptable level in the underground workings.

The various case studies on VOD focus on secondary (auxiliary) ventilation system as supported by O'Connor (2008) and Belle (2008). In these case studies variable fan speed drives (VFSD) are used to increase or reduce air quantity. Little focus has been put on using variable fan drives on the main fans. During IMVC (2014), it was established in a workshop that VOD has a limited application on main fans. The challenge is that when there is too much variation on speed of main fans at irregular intervals the mechanical stresses may result in premature fatigue failures of the impeller. It was recommended that before applying variable fan speed drives on main fans trade-off studies must be done between energy cost savings and reduced life-spans of impeller (Lownie, 2014) and this is supported by Belle (2008).
There are also drawbacks in using VFSD on auxiliary fans as they are expensive and need good maintenance (Lownie, 2014).The work done by Marco Biffi also concluded that VFSDs are expensive, usually very challenging to apply to motors greater than 1 MW and the technology is still not widely used in South Africa.

There are mines that do not use VFSD given the cost and maintenance requirements. When a mine is scheduled in such a way that all activities are known and when they will take place, then fans can be switched on and off, using event based VOD, for example in mines where there are no challenges with flammable gases and using mining method such as block-caving. The fans can be switched on when the machine drives into a tunnel and off when it leaves.
Cooling on Demand (COD) follows the same principle as VOD. The difference is that COD focus on regulating ambient temperature to create acceptable environment for equipment and workers.
The author believes that VOD and COD need a very strong communication and tracking system whereby workers and equipment location are known on a real time basis. Over and above to this, sensors must continuously feed into the communication system to inform environmental condition of working areas. When all this information is known then VOD and COD can be applied optimally and safely.

In accordance to the fan laws, there is a cubic relationship between the supplied air power and the airflow. The formula for calculating power is represented by the following equation:

As a result, even small reductions of airflow in the auxiliary ventilation systems have the potential to generate significant savings in operating costs.

2.4.1. SELECT SYSTEM

The last stage is to select the system that is sound from an economical and practical point of view. Important factors that are considered among others are: total cost of ownership of fans and ventilation requirements from commissioning phase, ramping up, steady state production ramping down and associated mining activities.

2.5. AUTOMATION

The mining industry around the globe is pursuing automation to address multiple challenges of productivity, environmental impact, skills, standardisation, costs and health and safety. Automation was defined by Parasuraman et al. (2000) and Thorogood et al. (2009) as generally referring to the full or partial replacement of a function previously carried out by human operators, either physically or mentally. “Automation can be characterised by continuum levels rather than all or none concept” (Sheridan, 2002).Table 5 below show levels of automation:

Table 5: Levels of Automation (Sheridan, 2002)

Levels of automation	Characteristics
Lower level automation	Warning system such as proximity detection, technologies that signal maintenance of equipment etc.
Mid-level automation	Line of sight control, collision detection technology etc.
Full automation	Equipment controlled remotely using computer screens, joysticks and other controls and other displays.

There is a rapid growth in the development and uptake of automation. In future mining, there will be no human presence in the production areas. All activities will be remotely controlled or automated and this is supported by these two companies Codelco and ABB (Classon & Inostroza, 2010), Figure 13 below illustrates where the mining industry comes from and where it is going. In the past traditional mining was labour intensive and all workers had to be underground. Presently the industry has adopted ‘real time mining’ where some operations are automated but the emphasis is more on mechanisation although some processes are being automated. The future which is what this report is concerned about is: ‘automated underground operations’ where there will be few workers underground working in safe vaults. The operations in future will be continuous with automated equipment and processes. The maintenance of robots and repair will be done in structurally safe underground vaults where there will be localised cooling.

The future of mining is lean production and automation. Bicheno and Holweg (2008) explain “lean” as a system which is more than the sum of its components and it involves a question of how to recognise and kill off inappropriate tools whilst developing new and stronger ones.

The mining industry has to embrace change (i.e. kill off inappropriate tools) in order to survive. It is the opinion of the author that mining methods that expose workers to hazards at the production zones must be avoided. Some of the mining companies are already looking at new ways of mining to improve productivity and health and safety. “Anglo American has signed a five year contract with US-based Carnegie Mellon University (CMU) to develop automated technologies that would improve safety in the mining industry”(Creamer, 2013).

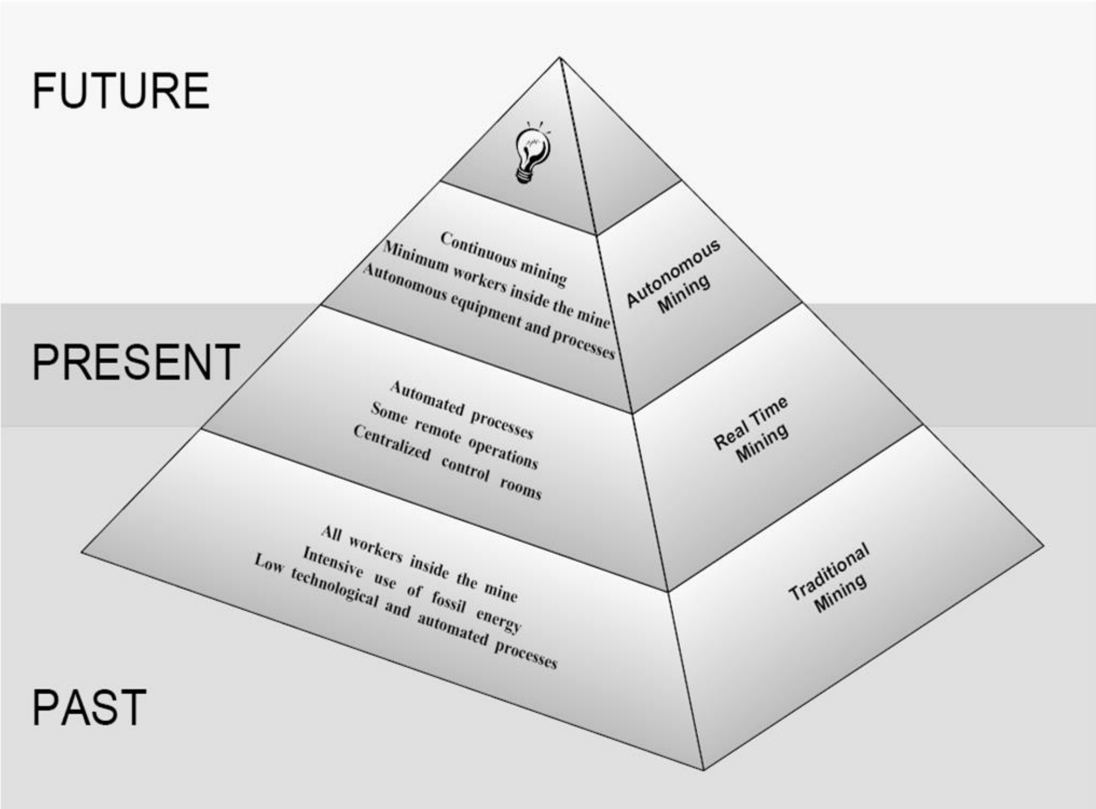


Figure 13: Future underground mining (Classon & Inostroza, 2010)

Not everyone thinks that automation will be a solution for future mining as it has a long history marked by successes and failures. Sarter and Woods (1995) in their paper titled 'how in the world did we ever get in that mode', argued that when automation is introduced to eliminate human error, the result is sometimes new and often more catastrophic errors. This statement concurs with Sheridan (2002) who also argued that automation fails because the role of the person performing the task is often underestimated particularly their ability to compensate for the unexpected.

The other reason why full automation has not been successful is the view by Funk et al (2009), and Endsley and Kaber (1999) who are advocating that full automation will come with other challenges such as loss of situation awareness and boredom due to the repetitiveness of operations, vigilance task and deskilling. In line with this research, other studies have shown that mid-level automation is preferable to keep operator awareness at a higher level and to allow performance of critical functions and this view is supported by work done by Kaber and Endsley (1999) and Endsley and Kiris (2005). There is a counter argument to this view by Kaber et al. (2009) in their paper titled ‘On the design of adaptive automation for complex system’ saying adaptive automation which refers to the dynamic allocation of system control functions to a human operator over time will preserve controller awareness. It is believed that this will facilitate a better match between task demands and cognitive resources. This is supported by previous work done by Kaber and Riley (1998) where they argued that operator awareness and preparedness would be enhanced by automation.

In developing countries such as South Africa, one of the major concerns with regard to automation is jobs. It is viewed that automation will have a negative impact on job creation.

This issue has already been addressed by several authors; Danelle and Horberry (2011) in their paper titled “Human factor issues with automated mining equipment” argued that the assumption that automation replaces workers are not correct and that rather, it changes the nature of the work that employees do. They supported their statement by saying, not all possibilities can be foreseen by automation designer in a complex environment and hence, operators are required to exercise their experience and judgement. However, the paper is silent in terms of the number of jobs that automation can create compared to conventional mining. A different view shown by McNab and Garcia-Vaquez (2011) who were very explicit in their report, titled “Autonomous and remote operation technology in Australia”, that automation does not only change the type of jobs that employees do but also reduces the mine workforce significantly. The report further quantified that the operations which adopt full scale automation the workforce may reduce by up to fifty percent (50%).

Job creation in mining industry is beyond the scope of this project but it is interesting to note that in a lecture on beneficiation at the University of the Witwatersrand 2012, Dr. Mtegha pointed out using a mining value chain, as shown in Figure 14 below, that the peak of job creation in South Africa is at the mining stage which is an anomaly; primary industry such as mining and agriculture should be mechanised and automated with minimal employment (Mtegha, 2012). He further noted that South Africa must not stop at the concentrate and extraction phase but must do full beneficiation to realise job creation and other benefits associated with it. The writer agrees with this view because less decent jobs can be done by machines and descent jobs can be created downstream.

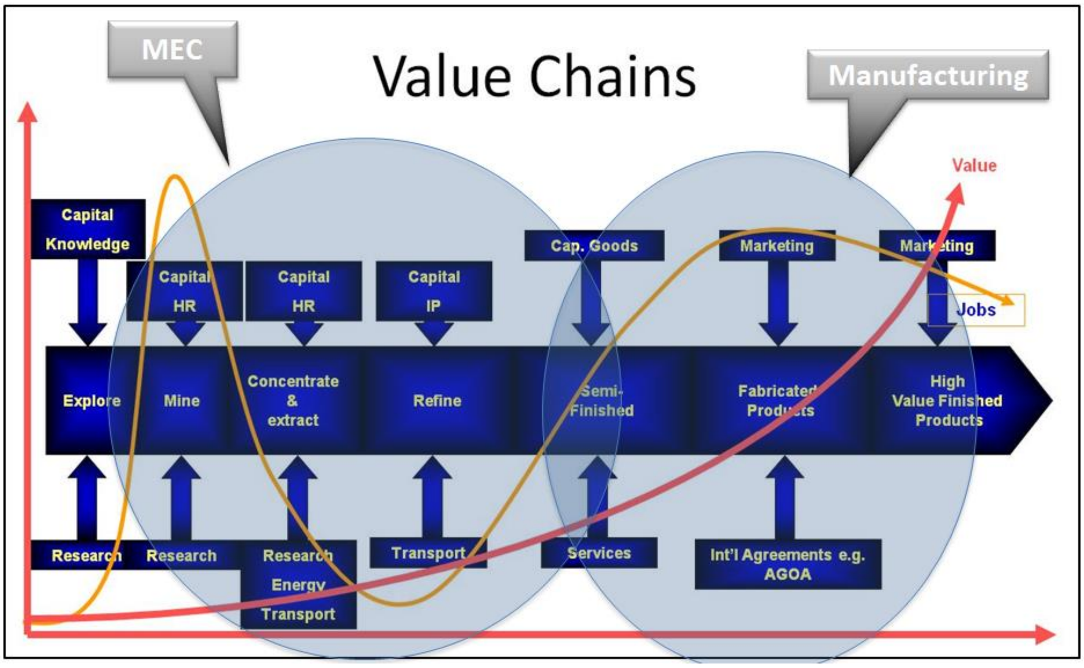


Figure 14 : Mineral value chain (Mtegha, 2012)

The jobs that are created in automated mines require skilled employees as it generally increases the complexity of a job. This view is supported by McNab and Garcia-Vaquez (2011) who also stated that mining projects in Australia are looking at education and training requirements for future automated mines.
Automation in the mining industry is imminent and unavoidable. South African gold and platinum mines have to look at how automation will impact on ventilation and cooling. This is very important for these two commodities as ventilation costs can render some of the deep level mining projects unviable.

Automated mines would require ventilation and cooling but at a reduced level as compared to conventional mines and this is supported by investigations that have shown that the overall air volumes required in the underground automated mine conditions are less than those required in a conventional mine (Kocsis et al., 2003). However, because of the need to control the heat generated during the tele-remote and automated processes, the air volumes are still significant (Kocsis et al., 2003). Simulations were done in this research to establish if there are benefits in increasing rejection temperature in automated mine.

According to Kocsis, Hall and Hardcastle (2001), the benefits of automation can be summarised as follows:

- “The temperature design criteria throughout would be 40°C dry bulb for equipment as opposed to the order of 28°C wet bulb for humans
- At shallow depths, a tele-remote operation can require up to 75% less air than the current diesel exhaust design standards
- With the removal of humans, controlled recirculation of air would be more acceptable
- Controlled re-circulation has the benefit of reducing the amount of air brought underground, this fact alone is significant as this is often the most expensive part of the ventilation system
- At depth, controlled recirculation can be combined with refrigeration to avoid increasing the mine intake airflow
- At shallow depth or where refrigeration is applied, natural or convection ventilation may be possible in local vertical circuits
- Tele-remote mining has the potential to reduce the number of vehicles underground and hence the required air volume
- Automation would readily facilitate the optimization of auxiliary ventilation, so permitting 'ventilation-on-demand' as opposed to the full-out norm.”

From the advantages of automation listed above, of great importance to the current study is the design temperature. Although Kocsis et al. (2003), suggested design temperature of 40°C for the equipment, the report lacked supporting technical data.

The recent available data supplied by leading equipment manufactures indicate that the design operating temperature can go up to 55°C, as shown in Table 6 below.

Table 6: Maximum operating ambient temperature of equipment

Manufacturer	Equipment	Maximum Temperature (^{0}C)	Reference
Atlas-copco	Electric Scooptram EsT2D	high	(Atlas-Copco, 2013)
Atlas-copco	Drill rig LIC	52	(Atlas-Copco, 2013)
Atlas-copco	Scooptram st1600LP	52	(Casteel, 2009)
Atlas-copco	Scooptram st7	52	(Atlas-Copco, 2013)
Atlas-copco	Minetruck MT5020	52	(Atlas-Copco, 2013)
Atlas-copco	Compressor	55	(Atlas-Copco, 2013)
Atlas-copco	Simba M6 C-ITH	40	(Atlas-Copco, 2013)
Sandvik	LH410	High temp	(Sandvik, 2013)
Sandvik	DI550 drill rig	55	(Sandvik , 2013)
Sandvik	LH514	54	(Sandvik, 2013)

2.6. HEALTH AND SAFETY

At a tripartite summit of the Mine Health and Safety Council (MHSC) in 2003, the parties agreed to certain milestones that the industry would have to achieve to ensure an improvement in mine health and safety.

Hermanus had enunciated the milestones as follows:

> “The sector target for safety is zero fatalities and injuries. The milestones associated with this target are:

- *In the gold sector*—To achieve by 2013, safety performance levels at least (i.e. the average of the safety performance of mines in the US, Australia and Canada) equivalent to current international benchmarks for underground metalliferous mines
- *In the platinum, coal and other sectors*—to achieve by constant and continuous improvement, at least equivalent performance levels to current international benchmarks.

One of the sector's health targets is to eliminate silicosis. The milestones associated with this target are to:

- By December 2008, reduce 95% of exposures to below the occupational exposure limit for respirable crystalline silica of 0.1 mg/m3 (these results are individual readings and not average results)
- After December 2013, using present diagnostic techniques, cause no new cases of silicosis to occur among previously unexposed individuals (previously unexposed individuals are workers who would not have been exposed to silica prior to 2008, for example workers who are new entrants to the industry in 2008 or who have worked on mines or in occupations in which silica exposures were absent).

The second health target, which is also the final target of the sector, is to eliminate noise-induced hearing loss (NIHL). The present noise exposure limit specified in regulation is 85 dB (A). The milestones associated with this target are that:

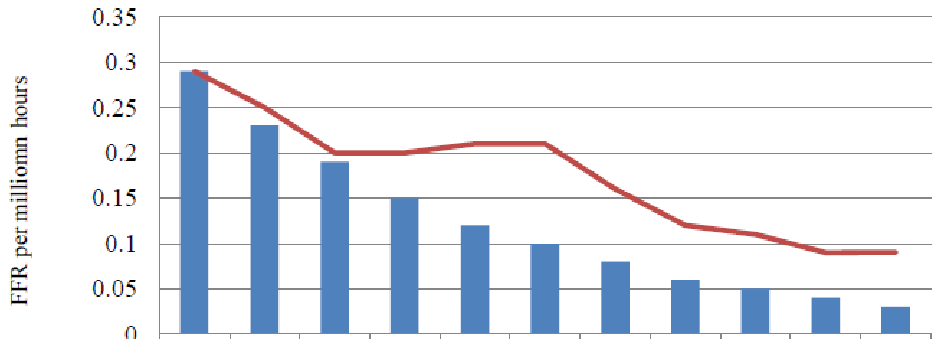
- After December 2008, hearing conservation programmes must ensure that deterioration in hearing are no greater than 10% amongst occupationally exposed individuals.
- By December 2013, the total noise emitted by all equipment installed in any workplace must not exceed a sound pressure level of 110 dB (A) at any location in that workplace" (Hermanus, 2007)

These milestones are more clearly indicated by Table 7 below.

Table 7: Milestones of the MHSC (Singh, 2012)

Occupational Safety – Fatalities and injuries - Milestones	
Industry Target: Zero rate of fatalities and injures	In Gold Sector: By 2013, achieve safety performance levels equivalent to at least current international benchmarks for underground metalliferous mines
	In the platinum, Coal and Other Sectors: By 2013, achieve constant and continous improvement equivalent to at least current international benchmarks
Occupational Health- Milestones	
Industry Target: Elimination of silicosis	By December 2008, 95% of all exposure measurement results will be below the occupational limit for respirable crystalline silica of 0.1mg/m^3 (these result are individual readings and not average results)
	After December 2013, using present diagnostic techniques, no new cases of silicosis will occur amongst previously unexposed individuals (Previously unexposed individuals unexposed prior to 2008, i.e. equivalent to a new person entering the industry in 2008)
Industry target: Elimination of Noise Induced Hearing Loss (NIHL)	After December 2008, the hearing conservation exposed individuals programme implemented by the industry must ensure that there is no deterioration in hearing greater that 10% amongst occupationally
	By December 2013, the total noise emitted by all equipment installed in any workplace must not exceed a sound pressure level of 110dB(A) at any location in that workplace (includes pieces of equipment)

With 2013 being the definitive target for the compliance of the milestones, one wonders whether the targets have in fact been met. The Chamber of Mines have indicated compliance with the target as follows:



	2003	2004	2005	2006	2007	2008	2009	2010	2011	2012	2013
Industry milestones	0.29	0.23	0.19	0.15	0.12	0.10	0.08	0.06	0.05	0.04	0.03
Actual industry	0.29	0.25	0.20	0.20	0.21	0.21	0.16	0.12	0.11	0.09	0.09

Figure 15: Actual Frequency fatality Rate compared with MHSC milestones (Phakathi, 2013)

From Figure 15 above, it is clear that the industry has fallen far short of the expected targets and has effectively failed in its commitments in terms of the milestones. Having said that, the graph does indicate that a significant improvement in the industry has been achieved, and this should be celebrated. It is however imperative that the objectives and purport of the milestones is not forgotten and every attempt be made to achieve the milestones. The causes of fatalities and associated percentages in mining industry are shown in Figure 16 below:

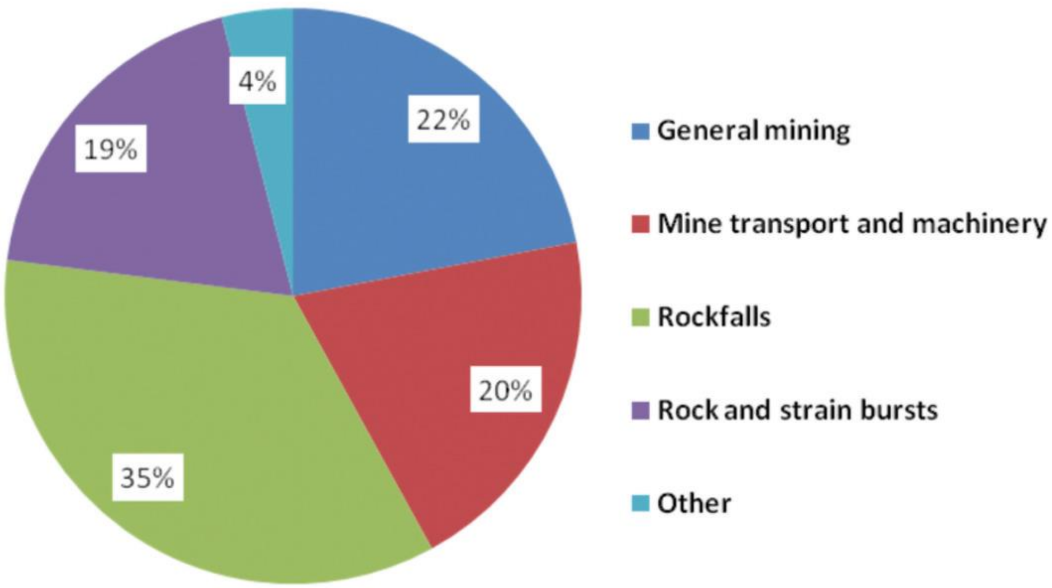


Figure 16: Fatality causes within South African Mines (2008)

The exposure of workers in production zones underground is the root cause of fatalities. The interaction of machinery, workers and underground environment is the biggest source of danger, if not managed properly. Human errors are the biggest contributor in most accidents.

Clearly with all the interventions being put in place and targets not being met, it is the author's opinion that the gap between the industry milestone and actual industry's achievement can be closed by introduction of automation as it removes employees from the hazardous environment.

Exposing workers to hazardous production does not only create safety issues but also health challenges. Table 8 below shows occupational diseases for year 2011 and of interest is the contribution of gold and platinum combined. These two commodities have contributed 87% of overall occupational diseases in year 2011. There are only two diseases where these commodities do not dominate and it is expected. These diseases are coal workers pneumoconiosis and asbestosis. These statistics reinforces the view that employees must be removed from the hazardous production zone.

Table 8: Occupational dieses redrawn from Mineral Resource Annual Report (DMR, 2012)

Occupational Diseases	**Commodity**						
	Gold	**Coal**	**Platinum**	**Diamond**	**Other Mines**	**Total**	**Gold& Platinum %**
Silicosis	1095	2	129	0	60	1286	95
Pulmonary Tuberculosis (PTB)	1696	249	1005	6	92	3048	89
Silico-Tuberculosis (Sil+TB)	553	0	0	0	2	555	100
Noise induced Hearing Loss(NIHL)	560	158	367	11	105	1201	77
Coal Workers Pneumoconiosis(CWP)	0	87	2	0	1	90	2
Asbestosis (Asb)	2	3	1	0	4	10	30
Other Diseases	145	22	13	3	14	197	80
Total	4051	521	1517	20	278	6387	87

Furthermore, limiting the exposure of in production zones will ensure that mining companies do not face future litigation. The classical example is the silicosis class action faced by gold mines. It is estimated that there are 288 000 of compensable silicosis cases in South Africa with an estimation of US$3.5 billion (Reuters, March 2012). The other issue that is highly topical and the mines need to be prepared for is diesel particulate Matter (DPM).

2.7. DIESEL PARTICULATE MATTER

When the milestones were developed in 2003, Diesel Particulate Matter was not a major concern. This could explain why there is no regulatory Occupational Exposure Limit (OEL) on DPM in South Africa till to date. However, the situation has changed and DPM has become a topic of common discussion at health and safety and ventilation conferences around the world.

DPM has been classified as a Group 1 Human carcinogen by the International Agency for Research on Cancer (IARC) in June 2012 and this is an impetus in driving governments to establish OEL's for DPM. Mining companies are also establishing the current exposure levels and improve the environment as they are compelled by law to conduct risk assessment on all hazards.

During IMVC_2014, it was established in a mine ventilation planning workshop that DMR proposed milestone for DPM will be as follows:

- A DPM exposure control value of 350 µg/m^3 [TC] up to December 2015
- A DPM exposure control value of 200 µg/m^3 [TC] up to January 2017; and
- A DPM exposure control value of 160 µg/m^3 [TC] in January 2018
- Universal availability of ULSD diesel fuel (10ppm) planned from mid-2017

Emission of low DPM requires better engines such as tier-IV and ultra-low sulphur diesel. It is no surprise that the OEL of 160 µg/m^3, which is the standard in most of the developed countries, will be instituted after the ULSD diesel is made easily accessible across the whole country.

Presenting one of the key note addresses at Mine Diesel Exhaust Conference (MDEC), 3 October 2012, Roger Mclellan suggested that New Technology Diesel Exhaust (NTDE) could have been classified as category 3 had IARC agreed to provide a separate evaluation for NTDE. Category 3 simply means that the agent is not classifiable as to carcinogenicity to humans. According to him recent research shows that Particulate Matter (PM) of NTDE are more than 100 fold lower than pre-regulation levels of diesel emission. He further said the PM of NTDE is much closer in composition to PM found in Compressed Natural Gas (CNG) and gasoline (i.e. petrol) exhausts.

The International Labour Organization, labour unions and government are putting mining industry under enormous pressure to improve health and safety. The shallow deposits are very limited and current studies indicate that lucrative ore-bodies, especially in the gold mines are at a depth of around 5000 metres (ultra-deep). The mines have not reached most of the health and safety targets mining at shallower depths. The reality is that it will be more risky to make employees to work in ultra-deep mines in terms of health and safety. This is leaving mine owners with no choice but to look at alternative ways of extracting the minerals such as automation.
The current equipment technology available being it diesel or electric powered requires ventilation and cooling.

2.7.1. DIESEL VS ELECTRICAL MACHINES

The mining industry around the world is looking at electric equipment more favourably, as occupational exposure regulations are becoming stricter, particularly on DPM. There are a number of reasons giving electric machines a competitive advantage over diesel machines namely: ‘zero-emission’, lower noise level, use about 40% less energy and produce 40% less heat. The advantages are supported by Jacobs (2013), MDEC (2012) and (E&MJ, 2013).

The electrically powered machine does not come without challenges. The machines are powered by reel-cables which present issues such as cable safety, flexibility, handling and control (E&MJ, 2013).

Despite the competitive benefits of electrically powered machines, mines are still reluctant to change. The main reasons could be the flexibility benefits due to mobility of the diesel equipment. Recent research by Poirier et al (2008) shows that hybrid diesel-electric LHD could be a solution to both eLHD and diesel LHD challenges. The hybrid does not have reel-cable and use diesel to generate electricity for the electric motor of the LHD (Poirer et al, 2008).

2.7.2. TEMPERATURE OF THE AIR

It has been stated earlier in this report that the rejection temperature conducive for employees 27.5°C (wb) and diesel equipment can be designed to handle ambient temperature of up to 55°C. The intake temperature of the air above the design specification can result in higher emissions of NO_X.

2.8. RISK MANAGEMENT

Risk management is defined by Biffi (2012) as the identification, assessment, and prioritization of risk followed by coordinated and economical application of resources to minimize, monitor, and control the probability and/or impact of unfortunate events or to maximize the realization of opportunities. It is based on adopting a reasonably practical approach to eliminate, mitigate or even accept risks. Risks can only be managed after being assessed. Therefore, risk assessment is defined as the systematic identification of undesired events and their causes and analysing their likelihood and potential consequences in order to make a valued judgement as to the acceptability or tolerability.
There are three types of risks assessment:

- Baseline risk assessment
- Continuous risk assessment
- Issue based risk assessment

2.8.1. BASELINE RISK ASSESSMENT

Baseline risk assessment is concerned with a process in a geographically defined area, with a start and end activity not necessarily in the same geographic location and it usually involves large scale areas. The outcome of this assessment is a full list of risks/ potential risks, ranked in order of severity, some of which may require further investigation.

2.8.2. ISSUE BASED RISK ASSESSMENT

This is a risk assessment process which require in-depth investigation and deal with specific issues such as noise, airborne pollutants, slope stability, seismicity and when major changes happen. The change could be as a result of new machine brought in, modified machine, change in work arrangements, after an accident or "near miss" as well as when new knowledge or information is made available.

2.8.3. CONTINUOUS RISKS ASSESSMENT

This is done periodically to assess whether the conditions, activities and controls are as planned. It is also done to check the status of noise levels (exposure) and control status; radiation levels (exposure status) and control status; and strata related risks exposure and control status. It is often done when updating the risk assessment (not usually part of the periodic assessment) when conditions have changes and review of risk assessment (this could also form part of Issue based assessment).
The more suitable assessment for this study is the issue based risk assessment as it deals specifically with change in work arrangements to full automation in production zones. South African mining industry, particularly gold and platinum are still predominantly using conventional mining methods and it is only in recent years that they are migrating to mechanised mining method.

The risk assessment process can be facilitated by using the risk management model developed by MDG 1O10 (2011). This risk management model has been utilized for number of years and is enjoying world-wide acceptance. Figure 17, shows schematic representation of the model.

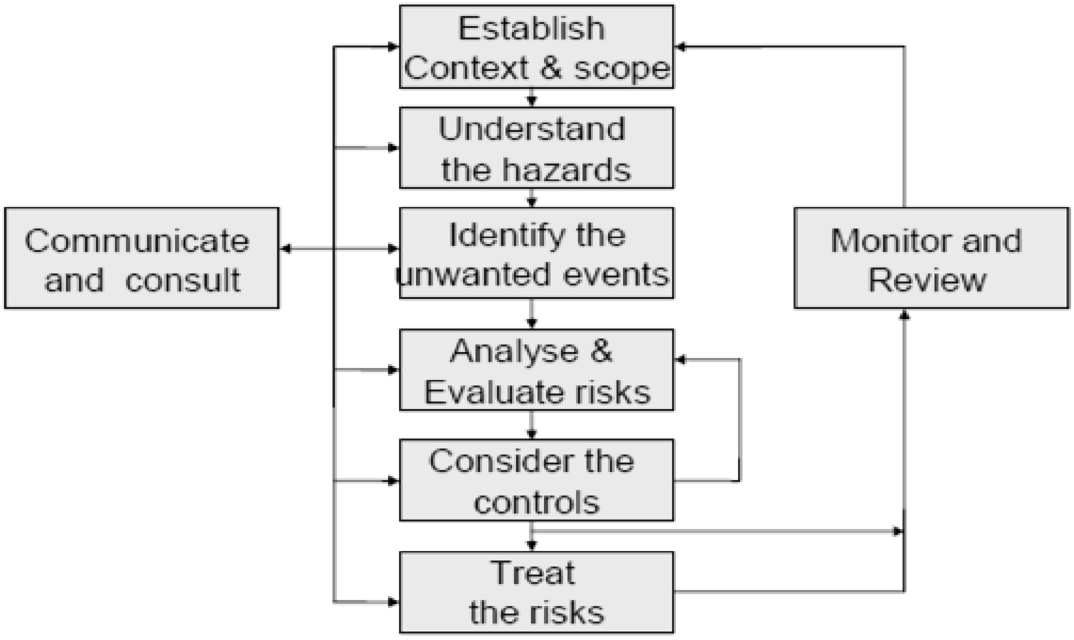


Figure 17: Typical risk management model (MDG 1010, 2011)

This model is important to the mining industry as it can be applied to major corporate decisions as well as day to day decisions. It gives a step-by-step process that can be incorporated in most decision-making where there are threats to safety, community, production etc.

2.8.4. COMMUNICATION AND CONSULTING

This is a very critical process in all the steps of the model. All the stakeholders must be involved and have open communication between them to ensure successful implementation of this model.

It has been established in this model that automation does not necessarily mean removing employees completely from underground. There will be time when employees are required to go into production zones for a very short time in rare cases. For example, if machines cannot be retrieved by an automated recovery vehicle. Employees would be required to enter the production zone and get the machine to the workshops, as this operation is risky , the employee must be aware of all the risks involved.

2.8.5. ESTABLISHING THE CONTEXT AND SCOPE

This is the first step where risk acceptability criteria are defined which considers internal and external factors. The scope is concerned about assessment objective, boundaries, methodologies, resourcing and timeframe.

2.8.6. UNDERSTANDING THE HAZARDS

In this step, all potential sources of harm are identified and understood. The main objective is to identify all sources of energies in an environment and possible harm to workers. For the purpose of this study, the sources of energy will be limited to heat.

2.8.7. IDENTIFY THE UNWANTED EVENTS

This step is to ensure that a thorough systematic approach is applied in examining the issue, process and design.

2.8.8. ANALYSE AND EVALUATE RISKS

This step involves combining likelihood and consequence to quantify the risk.

2.8.9. CONSIDER THE CONTROLS

This is the step where risk management process is applied to reduce the risk to acceptable levels. The hierarchy of control, which has been incorporated in many regulatory approaches, is used to address the risk.

According to the mine Health and Safety Act of South Africa, 29 of 1996: “Employers must ensure, as far as reasonably practical, that the mine is commissioned, operated, maintained and decommissioned in such a way that employees can perform their work without endangering the health and safety of themselves or of any other person.” It is very clear from this statement that the main purpose of the law is to protect the employees. Furthermore, the Act gives a clear guidance of addressing risks by providing hierarchy of controls in section 11 of the Act and can be easily illustrated by Figure 18 below:

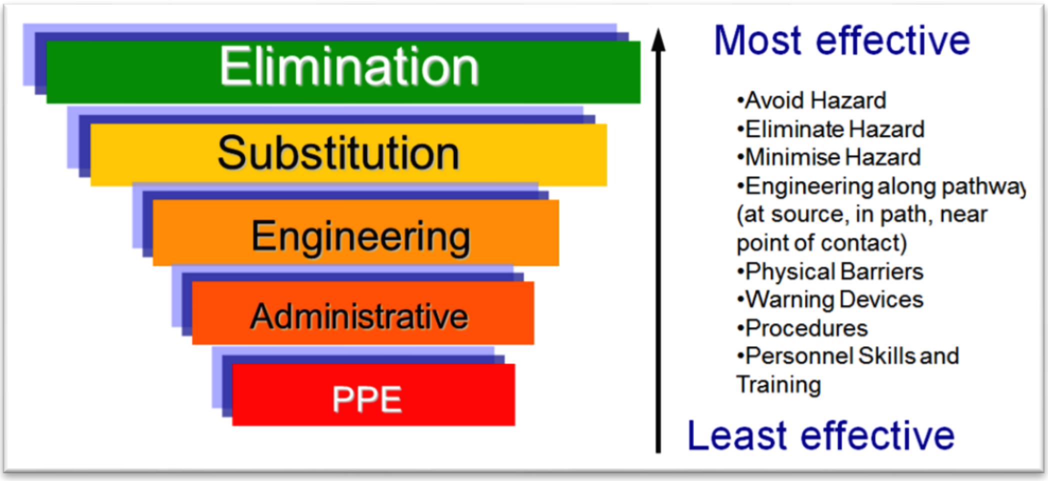


Figure 18: Hierarchy of control (Bakker, 2011)

The hierarchy of control risks associated with underground mining can be summarised by saying that the first measure to do when addressing risk is to eliminate the hazard. When the hazards cannot be eliminated, the next measure is to substitute or use alternatives. If the risk is still high, then put engineering controls, such as ventilation. The risk that exists after putting engineering controls can be addressed by introducing administrative controls and when administrative fails, the last resort is to use personal protective equipment (PPE). Using PPE can be interpreted as admission of failing to control a hazard. Ventilation is one of the engineering controls to provide environmentally acceptable standards.

This project looked at “higher order” controls such as risk elimination and process substitution. This was achieved by replacing employees with machines. However, where these machines break down irretrievably, they need human intervention. The first step in such cases would be to make the production zone where the machine broke down conducive for worker by switch on coolers and supply adequate ventilation. Furthermore localised cooling would be implemented in the form of air conditional cabs of the recovery vehicle and ice jackets in case of emergency. The localised cooling has two forms air conditioned cabs are engineering controls, and ice jacket is PPE.

2.8.10. TREAT THE RISK

This step is about implementing effective controls that have been tried and tested. There are two maintenance strategies namely preventative and corrective maintenance. The diagram below (Figure 19) depicts the maintenance strategies according to the EN13 306 standard of 2001. Furthermore, the engineering control measures will be used in the current study by selecting an appropriate maintenance strategy.

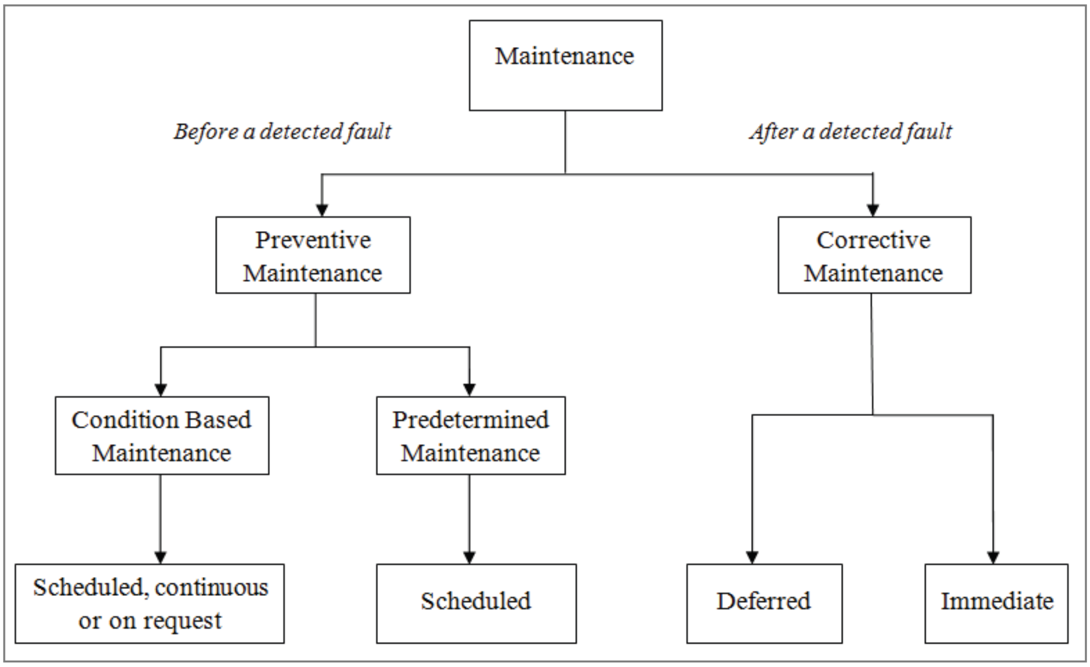


Figure 19: Maintenance overview chart according to EN13 306 (Mkemai, 2011)

Corrective maintenance is a reactive strategy, as it only replaces a component of a machine after a breakdown. If a component that has failed is readily available maintenance can be done immediately and if not maintenance will be deferred. This can be a very expensive strategy especially when there is an urgency to fix a machine.

Preventive maintenance is proactive, as it replaces components of a machine based on condition or predetermined duration. This can be easily scheduled and increases the availability and utilisation of a machine.

2.8.11. MONITOR AND REVIEW

This is a continuous process where audits are performed to check that the controls are in place and whether the hazard or conditions affecting the risk have not changed. This will be achieved by having a robust communication system. The effectiveness of this step can be facilitated by installing sensors in underground production zones and on equipment alerting the control room system, on real time basis, about environmental conditions of a tunnel, status and location of a machine. The communication will enable the operator to monitor and review the hazard and conditions on a continuous basis.

2.9. CONCLUSION

The reality is that the mining industry around the world is facing similar serious challenges namely: increasing depth, increasing electricity cost, low productivity and shortage of skills. The industry is looking at automation to address these challenges.

The power crisis and associated rising tariffs since year 2008 are being an impetus to Eskom and South African mining industry in seeking solutions to reduce energy consumption, especially in ventilation and cooling. A number of initiatives were found not feasible. For example, replacing steel blade with composite material was unsuccessful because of corrosion and VOD which need VFSD is economically unviable.

The industry is putting resources in automation research as it is seen as a holistic approach to these challenges. Despite the growing interest in underground automation there has been no study that seriously looked at increasing rejection temperature in automated mines for future operations in order to reduce electricity cost.

CHAPTER 3: METHODOLOGY

The reduction of electricity demand on ventilation and cooling in the South African mining sector, especially in underground metalliferous mines, has been discussed at various conferences and seminars. Section 1.2 presented some of the initiatives that were implemented or contemplated to reduce electricity demand. However, full potential has not been realised due to the following reasons:

- Initiatives not being suitable in the South African context
- Unable to reduce amount of air ventilation and cooling because of the presence of employees in production areas.

The second reason led to this research whereby a fully automated mine was modelled to explore the possibilities of reducing the electricity demand in underground mines. Throughout this research work and discussion with ventilation specialists, it became apparent that automation can assist in reducing electricity demand.

A hypothetical mine was modelled in collaboration with a South Deep Gold Mine ventilation specialist to quantify the benefits of increasing rejection temperatures in automated mines. The model focused on massive ore-bodies where mechanisation has been proven. The modelled mine was based on a simplified version of South Deep Mine. Two scenarios were simulated namely mechanised (manned) and full automation (unmanned), and a comparison was made in terms of ventilation and cooling required. The mechanised scenario takes into account that there are employees operating equipment in production areas while in automation only machines are in the production areas.The software used to develop this model was VUMA.

Though the work presented in this report is theoretical, the input and assistance was obtained from industry ventilation specialists and support from Bluhm Burton Engineering (BBE).

3.1. SIMULATION

3.1.1. VUMA

The acronym VUMA Stands for Ventilation of Underground Mine Atmospheres and is windows-based software. Its capabilities among others are simulation of air flows, temperature, humidity, dust and gas concentration throughout a mine ventilation network. VUMA can be used as a planning tool in most mining methods for narrow-reefs, massive ore-bodies and colliery layout with different levels of mechanization and automation (Bluhm, 2008).

This planning tool is important to predict and optimise future ventilation and cooling requirements. It is especially useful when dealing with complex networks that cannot be reduced to simple series or parallel circuits. However, it must be noted that VUMA is not a magic panacea. It does not replace a good working knowledge of basic ventilation theory and practice.

3.1.2. ANALYSIS OF NETWORKS

There are a few terms that must be defined before proceeding with analysis of networks, namely:

- Node is a point defined by x, y, and z coordinates
- Branch is a line that joins two nodes. It start and end with a node, depicting network components such as shafts, stopes, tunnel, raise-bore holes etc.
- Network is made up of branches that are interconnected and form a closed system
- Mesh is a building block of a network which has a close path with connected branches. It is as a closed loop within a network.

To arrive at a solution of a complex network, each and every airway characteristics must be determined. Two laws of Kirchhoff's, which were originally applied to electrical networks, are applicable (Burrows, et al., 1989):

The first law states that the algebraic sum of all mass flow rates at any junction is zero, as shown in equation 7 below.

$\sum_{i=1}^{n} M_i = 0$..*Equation 7*

Where;

M_i = mass flow rate = W_iQ_i (kg/s)

W_i and Q_i are the air density and volume flow rate respectively in the i-th branch connected to the junction, there being n such branches.

The second law states that the algebraic sum of all energy transforms which take place within the airflows around any closed mesh is zero and is expressed as follows:

$\sum_{i=1}^{n}(\mathrm{Pi} - \mathrm{Pfi}) - (\mathrm{nvp}) = 0$...*Equation 8*

Where;

Pi = frictional pressure drop, in Pascal (Pa)

Pfi = Difference in total pressure across the fan (Pa)

nvp = natural ventilating pressure (Pa)

3.1.3. THE LAWS OF AIRFLOW

The software incorporates pressure drops when the flow remains fully turbulent. The pressure drop is proportional to the square of the volume flow and directly proportional to the density. This is critical as it is a corresponding equation that represents the accuracy of the processes.

$P = RQ^2$ ....*Equation 9*

Where;

P = Pressure drop (Pa)

R = Resistance (Nm^8s^2)

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

3.1.4. ITERATIVE TECHNIQUES

The technique that the model uses is Hardy Cross method of analysis (Burrows, et al., 1989) in order to improve reliability and efficiency in dealing with mine ventilation networks. The technique involves making an initial estimate of the flow distribution and calculating an approximate correction to be applied to the flow in each branch and then iterating the correction until acceptable results have been achieved.

When considering the volume of fluid, Q, passing through an airway with resistance R and obeying a law, $P = RQ^n$, the true value of Q an estimated value, Qa is first assumed and summed with the error involved in the assumed value which is ΔQ. The formula is express as follows:

$Q = Qa + \Delta Q$..*Equation 10*

The challenge is to obtain ΔQ. The best method is to plot a graph representing $P = RQ^n$ and obtain the gradient at a point. ΔQ will be an error for quantity and ΔP will be an error for pressure. The Hardy Cross method is represented graphically in Figure 20 below.

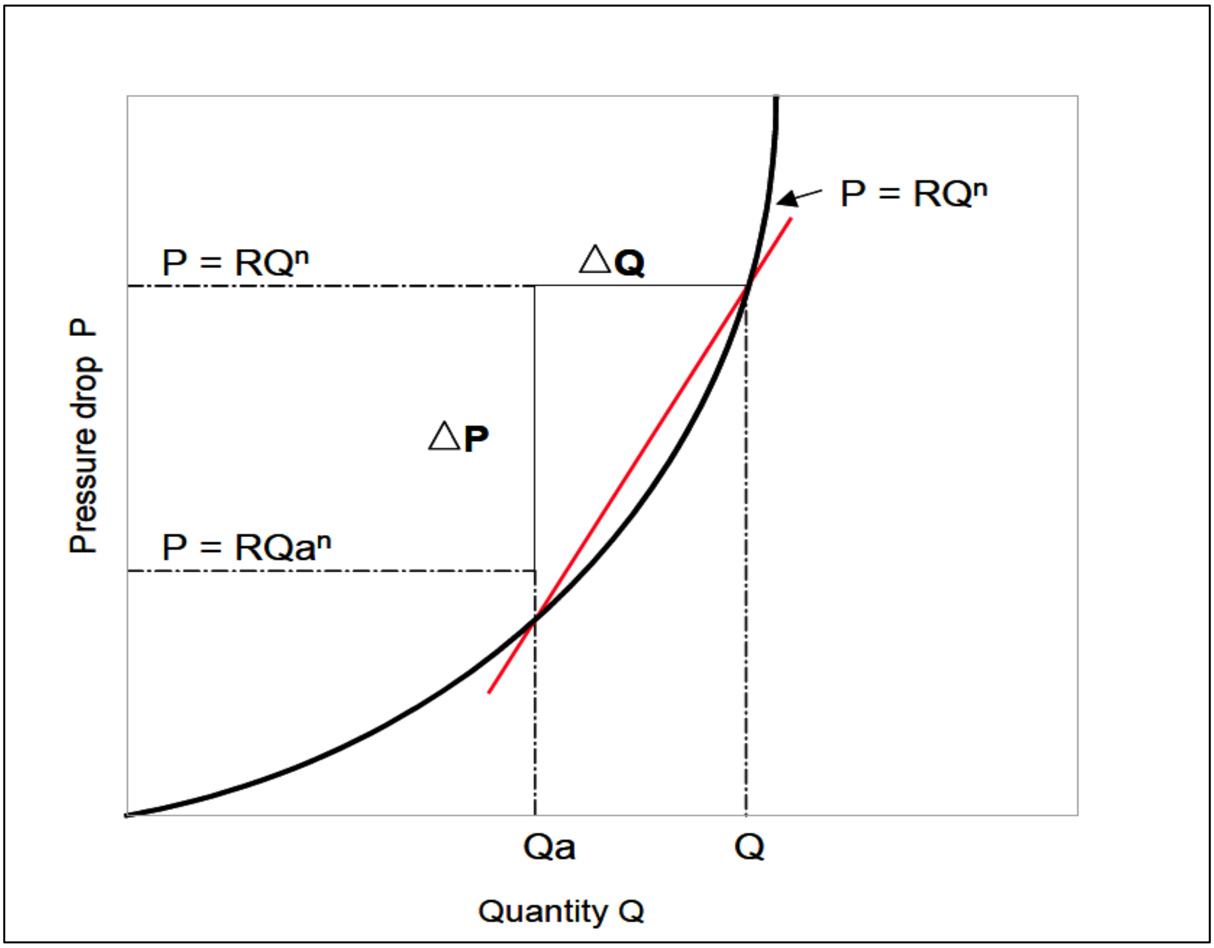


Figure 20: Graphical representation of Hardy Cross method (Burrows, et al., 1989)

Differentiating P= RQ^n will give a slope of the curve

$\frac{\Delta P}{\Delta Q}$ **= nRQn-1** ....*Equation 11*

Therefore;

$\Delta\, Q = \frac{\Delta P}{nRQ^{n-1}}$ ....*Equation 12*

But;

$\Delta\, P = RQ^n - RQ_a{}^n$...*Equation 13*

Therefore;

$\Delta\, Q = \frac{RQ^n - RQ_a{}^n}{nRQ_a{}^{n-1}}$ ....*Equation 14*

which is out of balance equation in a single duct.

Considering a network with b branches and applying Kirchoff's laws and assuming index n=2 for a fully turbulent flow a mesh correction factor of Q_m is determined by (Burrows, et al., 1989)

$\Delta\, Q_m = \frac{-\left\{\sum_{i=1}^{b} RQ_{ia}\left|Q_{ia} - P_{ia}\right.\right\}}{\sum_{i=1}^{b}(2R_{ia}R|Q_{ia}| - Sfi)}$...*Equation 15*

Where:

$|Q_{ia}|$ is the absolute value and S is the slope of the fan characteristic.

The procedure for the use of this method is described as follows (AMC Consultant, 2005):

a) Estimate the quantity of air flow
b) Decide of meshes pattern
c) Evaluate mesh correction factor
d) Correct the flow in each branch
e) Repeat c) and d) until $\Delta\, Q_m$ is below prescribed value
f) Repeat b) and e) for each number of changes

3.1.5. MODEL INPUTS

The required parameters and design elements to simulate underground operation are as follows:

- Data defining each branch to calculate the air pressure drop, air thermodynamic and contaminant level changes in a specific branch.
- Input data for a start node such as barometric pressure, virgin rock temperature, wet and dry bulb temperatures.
- Schematic representations of a closed circuit ventilation network
- Data defining mechanical ventilation devices such as BACs, regulators and fans

Departure point in developing a ventilation model is to understand the geological formation of the ore-body as it dictates the mining method.

3.2. GEOLOGY

The study will be confined to the Witwatersrand gold deposits. The word “Witwatersrand” is an Afrikaans name which means “White Water Ridge”. The Witwatersrand basin is mainly an underground geological formation and hosts the largest known gold reserves in the world. The size of the Witwatersrand basin is estimated to be 350km long that stretches east and west of Johannesburg and roughly 200km wide (Viljoen, 2009). Some of the gold mines found in this area are the deepest mines in the world such as Mponeng and Tautona at an average depth of 4km.

The gold bearing deposits are generally narrow. However, towards the east they diverge and thicken up to 130m, as shown in Figure 21 below.

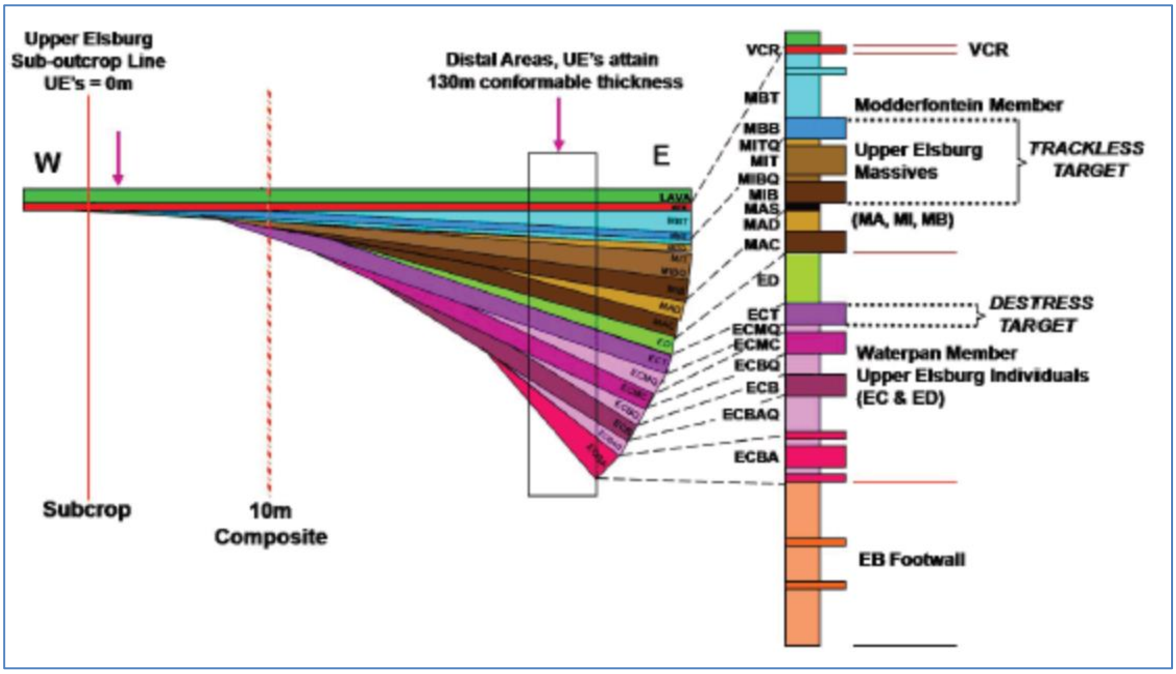


Figure 21: Generalised Witwatersrand East-West section showing the stratigraphy of the ore-body (Jones, 2003)

The base of the Witwatersrand is granite greenstone overlain by the Dominion Group which is around 2km thick. The Dominion Group is overlain by the Witwatersrand Supergroup which is 7km thick and is made up of two groups namely the lower West Rand Group and Central Rand Group. The West Rand Group consists of alternating quartzite and shale units and one lava unit, while the Central Rand Group consists almost entirely of quartzite and conglomerate, with one prominent shale unit being developed locally. In some goldfields, the Witwatersrand succession is capped by the Venterspost Formation, a thin layer of quartzite and conglomerate (the Ventersdorp Contact Reef), which has not formally been assigned to either the Witwatersrand or Ventersdorp Supergroups (Jones, 2003).

The simplified stratigraphy of the Witwatersrand is illustrated on Figure 22 below.

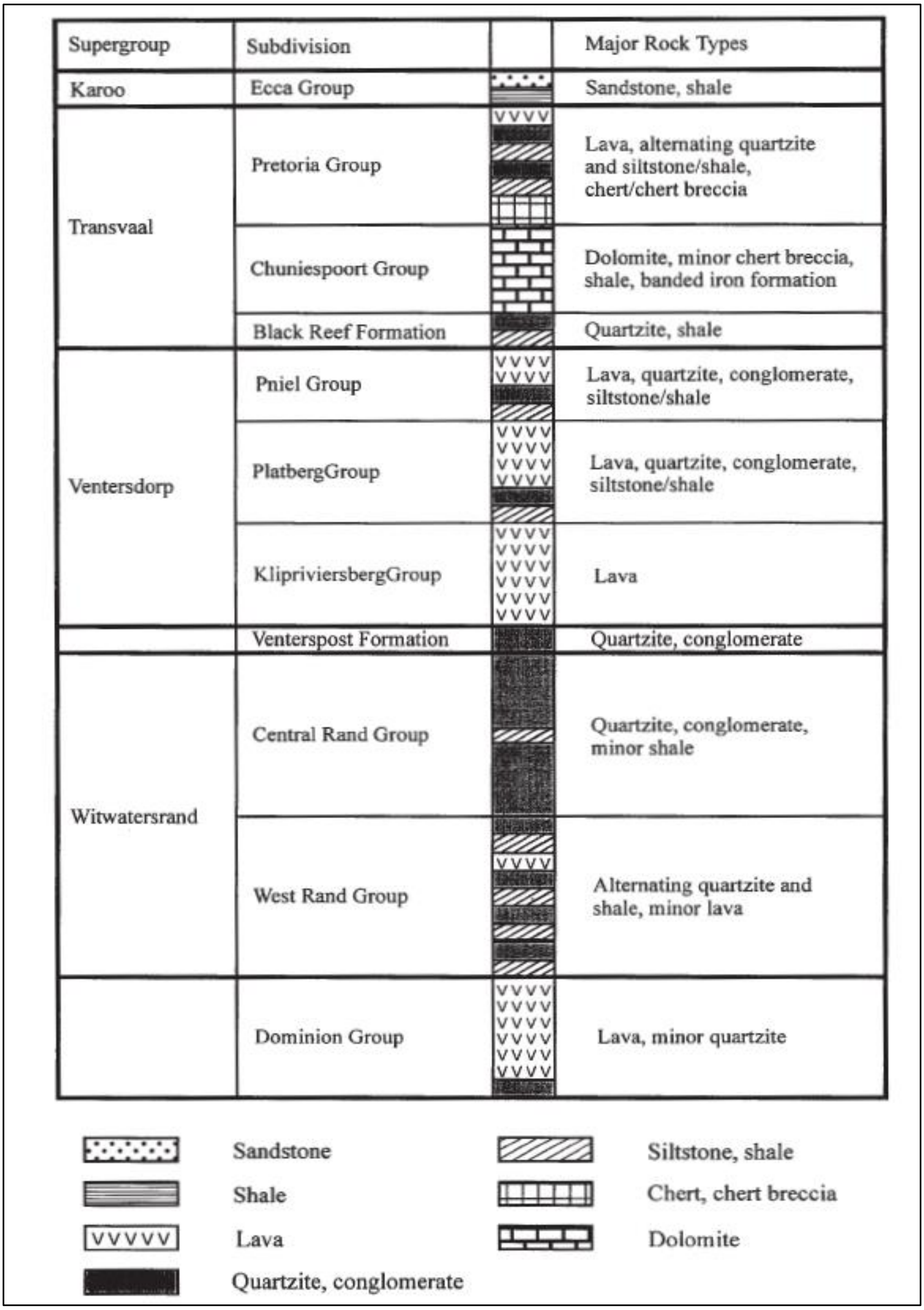


Figure 22: Schematic stratigraphic column for the Witwatersrand Basin (Jones, 2003)

In developing the model, the following assumptions were made on geology as shown in Table 9 below:

Table 9: Geological assumptions

Rock type and density (kg/m^3)	Quartzite	3000
	South deep strata	2670
	South deep reef	2850
Surface rock temperature (^{0}C)	18	
Geothermal gradient (^{0}C/m)	0.012	

3.3. MINING METHOD

The mining method will be geared for a thick deposit on the eastern part of the Witwatersrand basin as shown in Figure 21. There are two production sections termed corridor 1 and 2. Typical ore extraction in corridors incorporates a number of mining methods such as destress, long-hole stoping and bench and drift. Backfilling is used extensively to address rock mechanics challenges but also assist in increasing air utilisation and efficiency (Chadwick, 2011).

The model takes a snap shot at a steady state production of around 200 000 tons per month including development tonnage for both manned and unmanned scenarios. The model assumes the following assumptions shown in Table 10 for mining.

Table 10: Mining assumptions

Parameters	**Dimensions**
Depth (km)	2.5 to 3
Steady state Production per corridor (kton/month)	90
Number of corridors	2
Steady state production (kton/month)	180
Tunnel sizes (m)	5.5 x 5.0 and 5.5 x 5.5

3.3.1. DESTRESS MINING

The destressing is specifically for the main stoping horizon which is achieved by mining a 2m block through the reef horizon to ensure a destress window of 50 to 60m above or below the associated stope. The destressed part of the ore-body is accessed through a spiral decline. Main access drives are developed horizontally from the decline. The dimension for destress tunnels are 5.0m wide by 2.5m high for the model. The sequencing of the tunnel development is critical to mitigate rock-burst and stress damage. This destress tunnels are subsequently sliped to 5.5m x 5m behind the destress face and used as long hole drives. After mining the stopes the tunnel are backfilled as shown on Figure 23 below:

(i)Plan view

(ii)Cross-section view

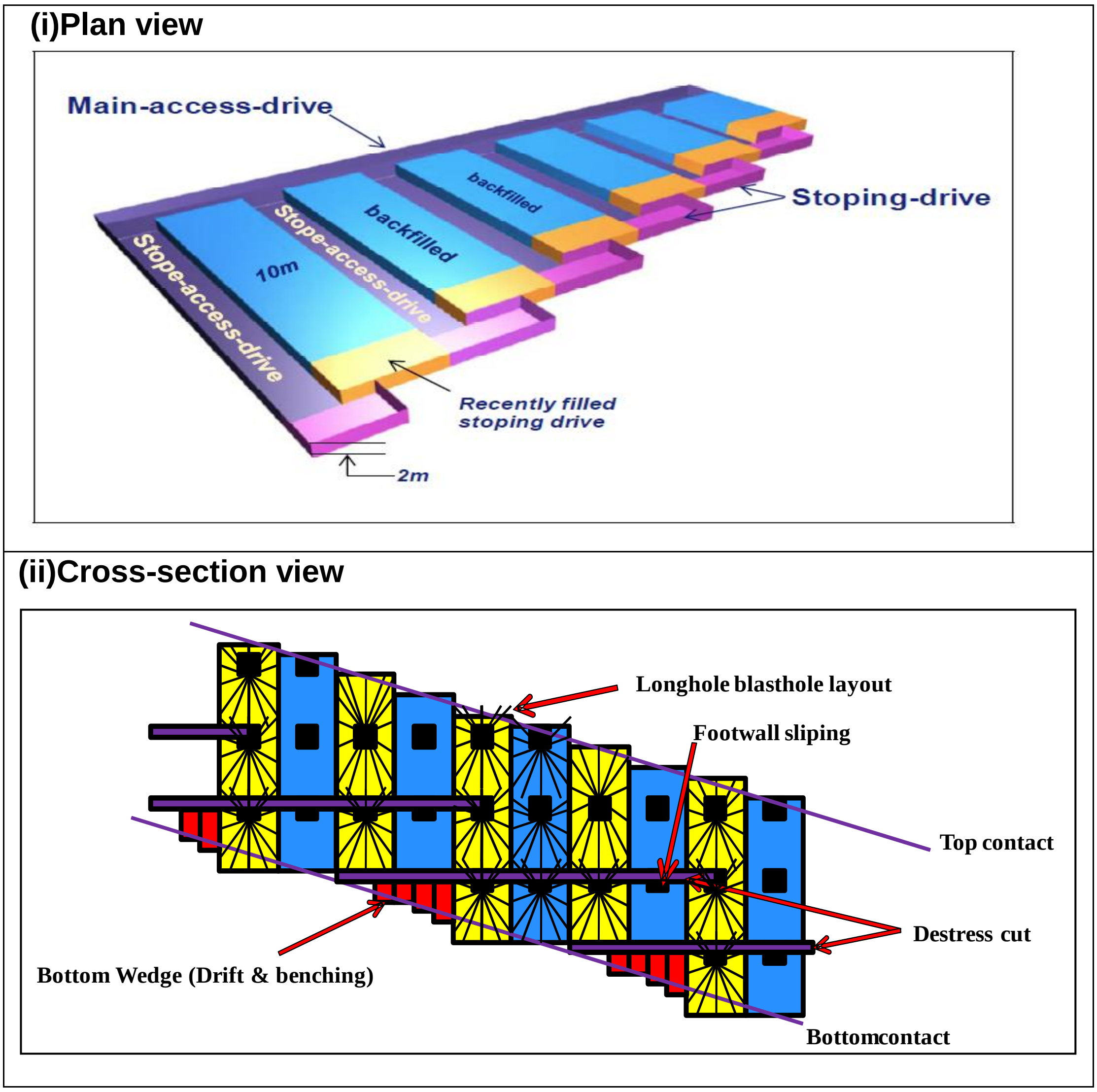


Figure 23: Destress mining (Chadwick, 2011)

The diesel equipment required in destress are shown in Table 11 below, and have been used as inputs to the model development.

Table 11: Destress equipment

Equipment	**Number**	**Rated power (kW)**
Drill Rig (S1L Boomer)	3	174 (each @ 58)
LHD (ST600LP)	3	408 (each @ 136)
Roof bolter	1	50
Transporter	1	85
Supervision vehicle	1	45
General Utility vehicle	1	30
Lubrication vehicle	1	84
Charging unit	1	84
Total	**12**	**960**

3.3.2. BENCH AND DRIFT MINING

The reefs that are 5m to 10m thick are mined using bench and drift method and the mined out areas are backfilled. The extraction using this method occurs below the de-stressed mining block. Drifts and benches are accessed through drift and bench access drives respectively. Drift and bench excavations are towards each other (i.e. in an opposite direction). It must be highlighted that benches are at the lower elevation and where they intersect they form one joint excavation. Drifts and benches advance at approximately 24m per month. Benches are mined using vertical holes and greater volumes are mined per blast. The sequence and mining of bench and drifts are illustrated in Figure 24 below:

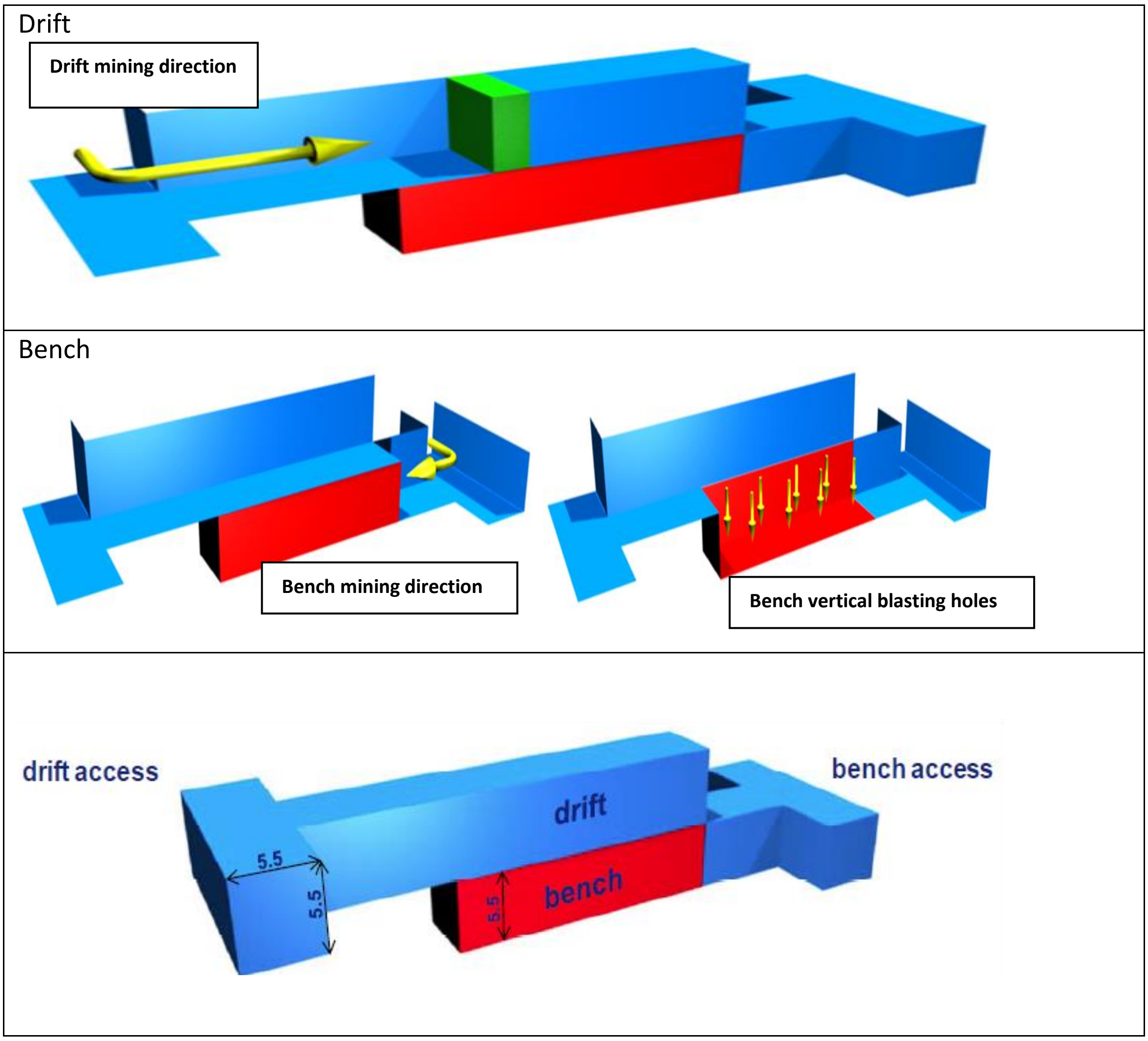


Figure 24: Drift and bench mining (Chadwick, 2011)

Bench cleaning is an automated process and use line of sight control. Employees are not allowed in working areas during cleaning of benches (i.e. remote loading). Once the bench is mined and backfilled the neighbouring drifts and benches are mined.

The diesel fleet required for bench and drift mining method are shown in Table 12 below and have been used as input to the model development.

Table 12: Equipment for Bench and drift

Equipment	Number	Rated power (kW)
Drill Rig (RB282)	1	58
LHD (ST1530)	1	298
Roof bolter	1	50
Transporter	1	85
Supervision vehicle	1	45
General Utility vehicle	1	30
Lubrication vehicle	1	84
Charging unit	1	84
Total	**8**	**734**

The dilution factor of 0.06m3/s/kW is used to determine the amount of air required in areas where machines operate (Kocsis, 2009).

3.3.3. LON-GHOLE STOPING METHOD

This mining method is aimed to extract the thickest portion of the ore-body which is greater than 10m. It is typically for blocks of the following dimensions; height of 17 to 35m; width of 15m and length of 60m. Access drives are developed at the top and bottom of the block for long-hole stoping. Rings are drilled either from the top or bottom access. A raise bore hole is rimmed to connect bottom and top access to create a free breaking zone. Each ring will blast about 1.5m of the length of the long-hole stope, yielding an estimate of 1280 tons with a height of about 15m. Figure 25 below show the long-hole stoping mining layout:

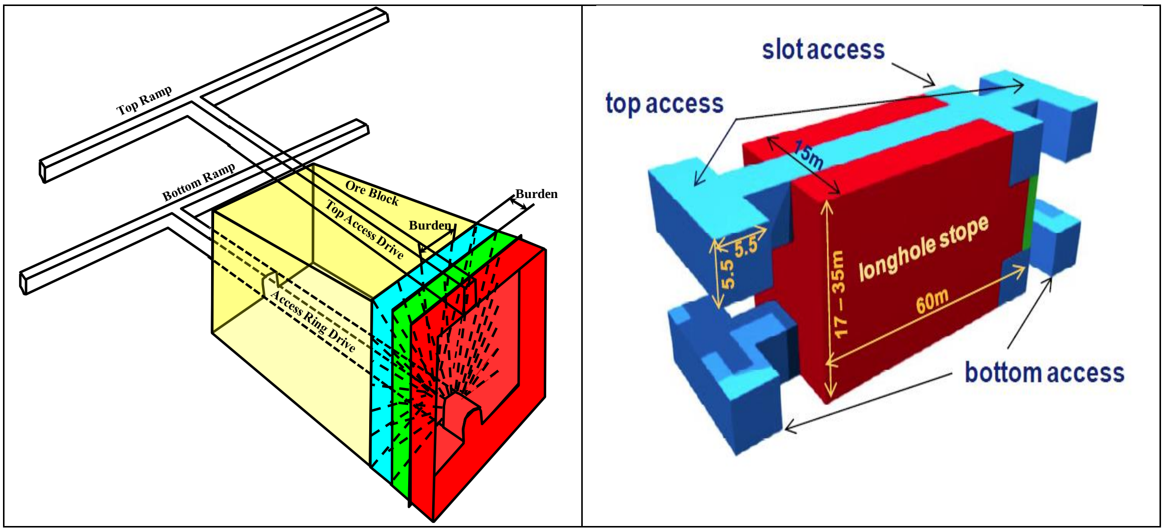


Figure 25: Long-hole stoping (Chadwick, 2011)

Once the slot has holed between the upper and the bottom access drive, there will be 'through ventilation' in long-hole stope. The equipment for long-hole stoping are shown Table 13 below and have been used as input to the model development.

Table 13: Equipment for long-hole stoping

Equipment	Number	Rated power (kW)
Drill Rig (RB282)	1	58
LHD (ST1530)	3	894
Roof bolter	1	50
Transporter	2	85
Supervision vehicle	1	30
Charmec (charging unit)	1	170
Lubrication vehicle	1	84
Truck(MT436B)	1	298
Charmec (scaler)	1	120
Total	**12**	**1789**

3.3.4. WORKSHOP AND CONTROL ROOM

It was decided to place the workshop and control room for both mechanised (manned) and automated (unmanned) scenario underground for easy logistics. The control room is used for different purposed for both scenarios. In mechanised mining control room is used for operating haulage and vertical hoisting of ore while for automated scenario it will be used to operate all the automated processes such as loading using LHD, drilling, and support operations.

3.3.5. MINING ACTIVITIES

Ventilation optimisation studies require a full understanding of mining processes, as it provide a foundation for improvement. The typical mining cycle for de-stress cut, long-hole drives, drift and bench development is shown in Figure 26 below:

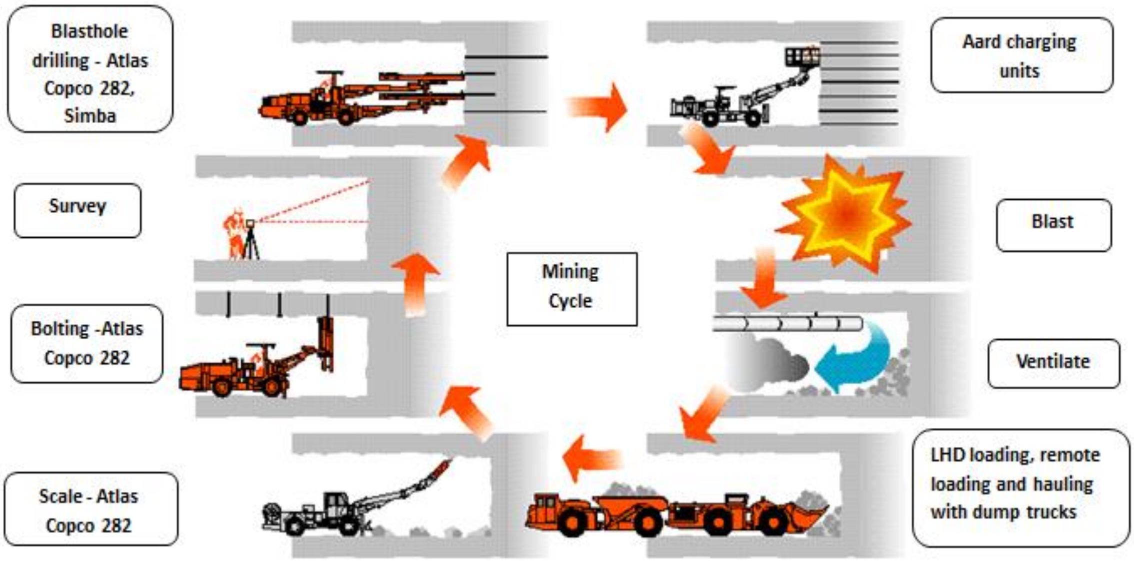


Figure 26 Mining cycle for distress, drift and fill and long hole stopping (Chadwick, 2011)

3.3.5.1. SHIFT UTILISATION

Mines use different shift configurations to achieve production target. This project has assumed two 12 hour shift, working throughout the year. There are various activities that take place during the shift requiring varying amount of air ventilation. The focal point will be on the peak production time or effective shift time. The shift effective time is the time that is taken to do production and can also be seen as the 'worst case' scenario for ventilation. This is the period where heat, dust, DPM and other contaminants are produced at a maximum potential level and maximum ventilation is required to operate at its optimum level to create acceptable environment. A typical two 12 hour shift configuration used to determine the effective working hours.

Time studies were done on shift time utilisation at South Deep Mine and the summary is shown in Table 14 below.

Table 14: Time Studies (South Deep Mine, 2011)

Categories	**Day shift**	**Minutes**
Start of shift procedure	Clocking in	20
	Shaft-Traveling to waiting place	55
	Waiting place procedure	15
	Travel to work station	15
	Pre-check list	30
Effective working shift time	Marking of, Drilling, charge up, loading, service, water control, backfill, survey, Housekeeping, Maintenance	365
End of shift procedure	Parking at EOS, travel to waiting place, end of shift procedure, blasting and reporting	60
	Travel to shaft	30
	Shaft-Traveling to surface	175
	Total	765

The two-12 hour shifts mirror each other as shown in Figure 27 below:

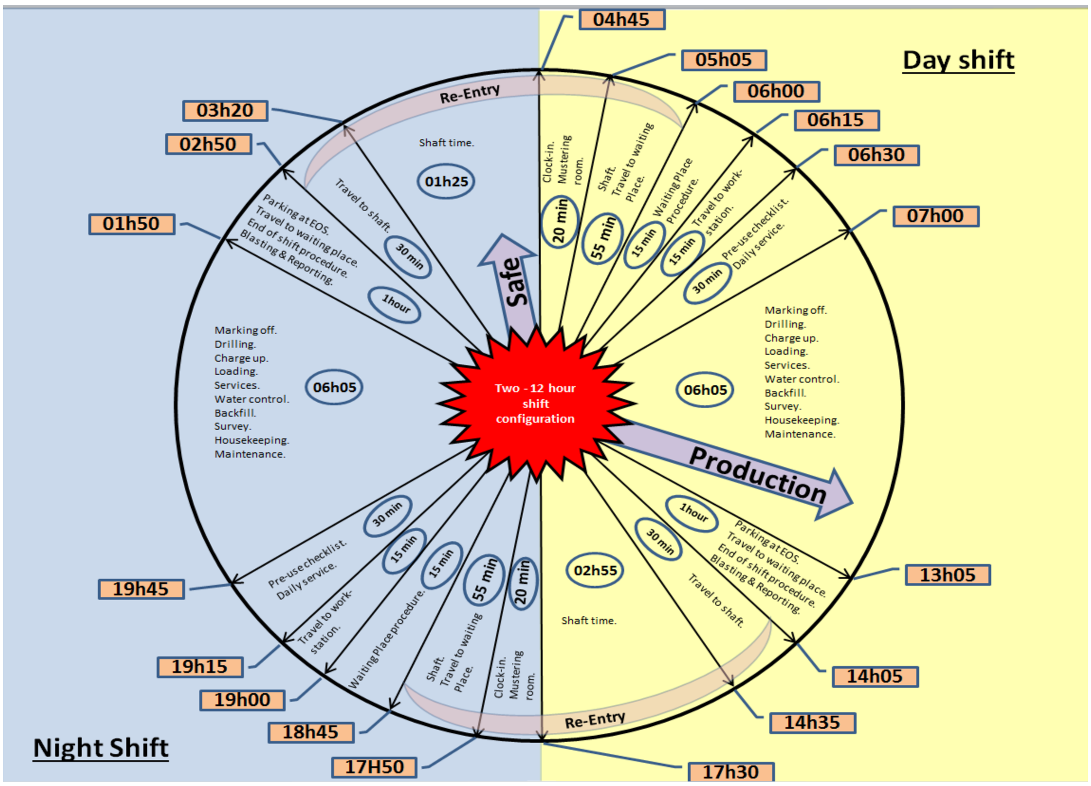


Figure 27:Two 12 hour Shift configuration

3.4. ACCEPTABLE ENVIRONMENTAL CONDITION

The project focuses on a thermally acceptable environment for both mechanised and automated scenarios. The rejection temperature (wb) for the mechanised mine (i.e. manned) is set at 27.5^oC (wb) and 32^oC (db). The automated mine (unmanned) rejection temperature was set at 40^oC (db) in the working areas as the machines are not affected by wet-bulb temperatures. However, the travelling ways to the workshop and the workshop itself has been set at 27.5^oC (wb) and 32^oC (db) because of two reasons namely; employees working in the workshop and for equipment in the control room.

3.5. VENTILATION AND COOLING

3.5.1. PRIMARY VENTILATION CIRCUIT

The primary ventilation circuit chosen for the model was a hybrid of series and parallel circuits. The parallel circuit generally adopts a design philosophy of "one pass, flow through" air. The air is directed from the primary air intake circuit through the workings and exhaust to the return air way circuit. The main advantage of this system is that it reduces the re-entry period provided the return airway does not form part of the entry access for workers. The parallel system was mainly used in the underground workshop and access levels (AMC Consultant, 2005).

The series circuit was chosen for production zones (or corridors). This is a circuit normally used for narrow reef ore-bodies. It must be pointed out that South Deep Mine, which is used for model validation, uses this circuit in its massive mining. Hence, this circuit was adopted in the current study for production zones. This type of circuit has the advantage of simplicity of control and less development for ventilation (AMC Consultant, 2005).

3.5.2. SHAFTS

The modelled mine for both manned and unmanned scenarios, has two shafts namely the down cast which will be used to transport rock, employees and material and up-cast which is dedicated for ventilation only.

3.5.3. MAIN FANS

The assumption made was that the mine has multiple main fans in parallel placed at the top of the up-cast shaft. This would assist the mine to maintain some air flow in the event that one fan or all fans fail. In an event that all fans fail the underground workings would have lower pressure than the atmosphere which will create air movement.

3.5.4. COOLING

In the model air was cooled in stages. The first stage which is primary cooling was done on surface. Secondary and tertiary cooling was done underground.

3.6. MODEL VALIDATION

The hypothetical model of the underground operations was validated by comparing the results generated by VUMA software and calculated figures of the VRT and auto-compression. It was also validated by using actual measured values from South Deep Mine.

The air velocities in various airways were gauged against the recommended criteria. The percentages of heat sources in the model were compared with the one in South Deep Mine.

3.7. SIMULATION OUTPUT PARAMETERS

The following output parameters are expected from the model developed:

- Heat sources and amount of heat produced
- Virgin Rock Temperature (VRT)
- Impact of cooling strategies
- Air cooling power in the workings
- Pressure losses
- Tonnage output
- Shaft sizes

The output parameters stipulated above were used to calculate energy consumption and associated costs for both manned and unmanned mining scenarios, and energy savings thereof.

3.8. RISK MANAGEMENT

The Mine Health and Safety Act requires mine owners to identify hazards, perform risk assessment and implement control measures to address risks.
The process outlined below in Figure 28, illustrate four basic principles of risk assessment which were used in this project:

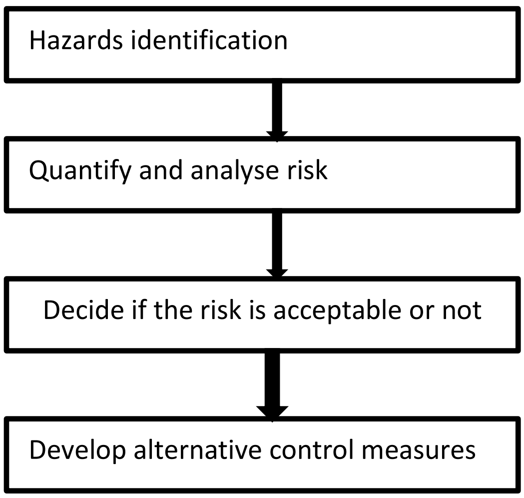


Figure 28: Risk Management framework (MHSA 29, 1996)

3.8.1. HAZARD IDENTIFICATION

The most important step in risk assessment is hazard identification because it informs the subsequent step of quantification and analysis of risk. There are many systems that can be used to identify hazards. The type of risk assessment to be done dictate the type of hazard identification method to be used. The methods can be broadly categorised into two groups namely: high level or macro and detailed hazard identification. The one that has been chosen in the current study is high level. This method does not go into detailed hazard identification rather provides, for example, structured 'What-if' checklist.

3.8.2. QUANTIFY AND ANALYSE RISK

Once the hazards have been identified, it is necessary to prioritize them. The hazards are analysed in terms of the risk that can be realised from them. Risk is a function of likelihood (or frequency) and consequences (or severity).

Likelihood can be defined as a chance of the event taking place and is a function of exposure and probability. Exposure is the actual time period that the resource is at risk while probability is the chance of loss, damage or injury taking place during the exposure time. From these two components likelihood or frequency of a scenario occurring can be quantified. There are a number of ways of determining likelihood. For the current study it was quantified as follows:

Table 15: Likelihood

Likelihood value	Descriptor	Description
1	Rare	May occur in exceptional circumstances
2	Unlikely	Could occur at some time
3	Possible	Might occur at some time
4	Likely	Will probably occur in most circumstances
5	Almost certain	Is expected to occur in most circumstances

Consequence is a positive or negative outcome that may result from the event and was quantified as follows:

Table 16: Consequence

Consequence value	Descriptor	Description
1	Insignificant	No injuries, low financial loss
2	Minor	First aid treatment, on-site release immediately contained, medium financial loss
3	Moderate	Medical treatment required, on-site release with outside assistance, high financial loss
4	Major	Extensive injuries, loss of production capability, off-site release with no detrimental effect, major financial loss
5	Catastrophic	Death, toxic release off-site with detrimental effect, huge financial loss

From Tables 15 and 16, a matrix shown in Table 17 below could be developed which will be used in this project.

Table 17: Risk Matrix

	Likelihood				
Impact score	1	2	3	4	5
	Rare	Unlikely	Possible	Likely	Almost certain
5 Catastrophic	5	10	15	20	25
4 Major	4	8	12	16	20
3 Moderate	3	6	9	12	15
2 Minor	2	4	6	8	10
1 Negligible	1	2	3	4	5

Score	Risk Level	Recommended Response
15 – 25	High Threat	Immediate action or detailed planning to be included within implementation plans
8-14	Medium Threat	Measures to be included into action plans and monitored
1-7	Low Threat	Limited action and review will be undertaken

3.8.3. RISK ACCEPTABILITY

Once the risk has been determined using Table 17, a difficult decision has to be made whether it is acceptable or not. There are broadly three categories of risk acceptability in terms of “As Low As Reasonably Practical (ALARP)” illustrated in Figure 29. The South African legislation does not use” ALARP” but “As far as is reasonably practical”. Broadly acceptable risk can be referred to as negligible risk which implies that nothing further needs to be done to mitigate the risk. Tolerable risk is when controls are required to reduce the risk to acceptable level or the risk is tolerated only if the benefits are highly desired. Intolerable risk means that the risk is unacceptably high, irrespective of the benefits achieved from taking the risk.

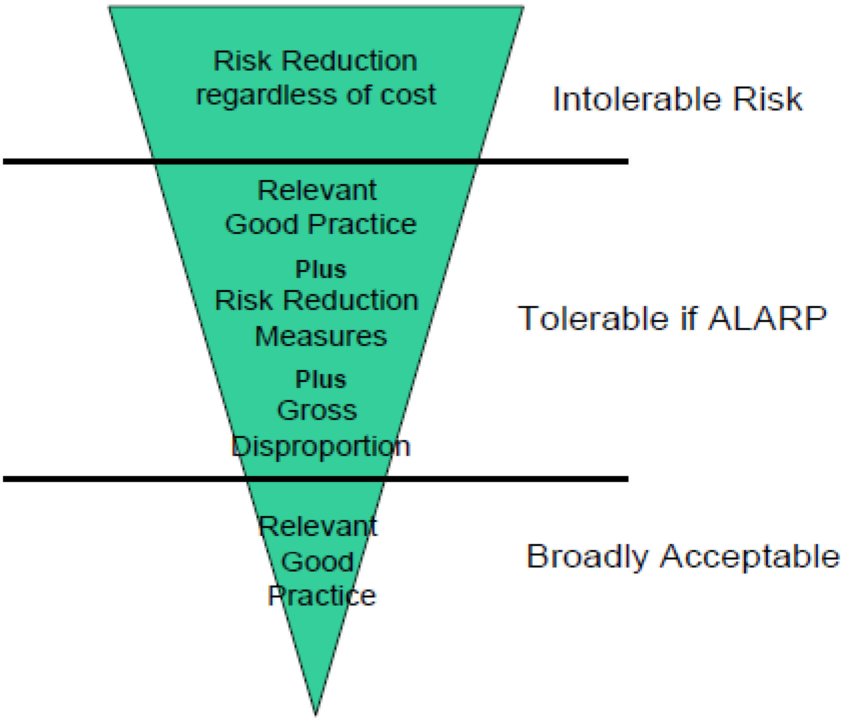


Figure 29: Risk acceptability (Trade & Investment Mine Safety NSW government, 2011)

3.8.4. CONTROL MEASURES

Control measures are actions required to lower the consequences of the risk or reduce the likelihood of the risk materialising or both of these. Figure 18 in chapter 2 depicts possible measures which were used.

CHAPTER 4: RESULTS AND DISCUSSION

This chapter discusses the results obtained from the model for both manned and unmanned mining processes scenarios. It followed a systematic approach of planning mine ventilation. This process provides the amount of air and cooling required for both scenarios and an estimation of electrical power was calculated using various formulas. Then the electricity tariff was applied on the estimated electrical power required for both scenarios to obtain the cost of electricity. Comparison was done on both scenarios by looking at kilo-Watt-hour per ton and cost per ton for the two scenarios.

Simulation has become a vital part in most businesses including mining in solving real life challenges. Multiples of 'what-if' scenarios can be generated and analysed to provide answers to real life challenges. The real life situation can be compared to a conceptual process or system before being implemented. Simulation generates the real characteristics of the existing system and can also show the impact of process changes on the system.

VUMA software model has made it possible to simulate underground conditions similar to South Deep Mine and compare manned and unmanned scenarios, with regard to acceptable thermal conditions.

4.1. MINE DESIGN

A hypothetical mine was designed to produce a total monthly tonnage of about 200 000. The tonnage is produced from two corridors and development headings. The corridors produced about 90 000 tons each from destressing, long-hole stoping and bench and drift mining. The balance was from development headings. There were critical infrastructures required to meet this target and these included shafts, intake and return airways for ventilation and cooling and production zones. These infrastructures were to handle tonnage output, equipment, air and cooling required. For the purpose of this study, dimensions of the mine infrastructures were kept to be the same for both manned and unmanned scenarios. A number of iterations were run on the VUMA model for the manned scenario to achieve optimum mine design which was used as a base design for the unmanned scenario. This made comparison with a real life situation possible and was also used to validate the model.

4.1.1. RATE OF PRODUCTION

Table 18 below shows the split of production tonnage in order to achieve a target of about 200 000 tons for manned scenario. The tonnage was obtained from destress, bench and drift, long-hole stoping and development.

Table 18: Production rate for the modelled mine

	Width (m)	Height (m)	Advance per month(m)	Density (kg/m³)	Number of active production zones per corridor	Number of corridors	Tonnage per month
Destress	5	2.5	30	2.85	10	2	21 375
Bench and drift	5	10	14	2.85	12	2	47 880
Long hole stoping	15	15	16	2.85	6	2	123 120
Development headings(quartz)	5.5	5	15	3	1	2	2 475
Development headings	5.5	5	15	2.65	1	2	2 186
Total							**197 036**

The tonnage produced was as per the two-12 hour shift configuration shown on Figure 27 in chapter 3 for the manned scenario with an effective shift utilisation of around 50%.

For the unmanned scenario, the author assumed that from the six hours which are used for the start and end of the shift procedure in the manned scenario, three hours will be converted into productive time. This translated to nine effective working hours which yielded 75% of shift utilisation. This assumption was informed by the possibility of real 'hot-seat' change in the control room. The shift schedule for the unmanned scenario converted the following activities of the manned scenario into productive time; waiting place procedure (15 minutes), travel to work station (15 minutes),pre-use check list (30 minutes), parking and travelling to the waiting place (1 hour), travel to shaft (30 minutes) and 45 minutes of shaft time. The improved effective working hours meant that equipment could be used productively and have overall production improvement of 50%. Hence, the tonnage production set for unmanned scenario was 295 000 tons.

4.1.2. SHAFTS

Both manned and unmanned mine models had down-cast and up-cast shafts. A number of iterations were performed with the VUMA software and the optimum design of the down cast shaft was determined to be a 9.5m diameter, concrete lined shaft with streamlined buntons and an Atkinson friction factor (k) of $0.025Ns^2/m^4$ was used (Burrows, et al., 1989). The advantage with this shaft size diameter is that it will make the logistics of sending equipment underground easy. From the literature review in section 2.3.8, the recommended maximum air speed in a shaft is 12m/s which means the maximum air carrying capacity of the downcast was calculated as $850m^3/s$ by using Equation 6.

From a number of iterations the up-cast shaft was designed to be 7.5m in diameter; concrete lined with no steelwork and an Atkinson friction factor (k) of $0.004Ns^2/m^4$ was used (Burrows, et al., 1989). The maximum air carrying capacity of the up cast shaft was the calculated to be $927m^3/s$.
Table 19 below, show the results of maximum air carrying capacities of the shafts and design capacity.

Table 19: Shafts design and maximum air carrying capacities

		Manned scenario		**Unmanned**	
		Air Speed (m/s)	**Air Quantity (m^3/s)**	**Air Speed (m/s)**	**Air Quantity (m^3/s)**
Down cast shaft	Max	12	850	12	850
	Design	11.7	830	14	618
Up-cast shaft	Max	22	927	22	927
	Design	20	886	14	14

The maximum depth of the shaft was designed to be 2811m (level 92). The mine was designed to have 4 levels namely 86, 87, 90 and 92. The air moved from surface through the downcast shaft to various shaft stations and the design used twin-intake airways (travelling ways) as conduits to various working areas. The air then travelled through return airways (RAW) where workers access is limited, and eventually was exhausted through the up-cast shaft.

4.1.3. TUNNELS

The intake airway and return airway tunnels dimensions were set at 5.5m x 5m for the model with a maximum air carrying capacity of 137.5m^3/s at a maximum speed of 5m/s. Figures 30 and 31 below show the designed mine layout with intake airways shown in blue and return airways in red. The mechanical workshop and control room for both scenarios, manned and unmanned, were placed underground level 90. The refrigeration machines, using refrigerant R134a, for both scenarios were placed closer to the up-cast shaft to avoid leakage of a refrigerant which could eventually finds its way into the combustion of diesel engines and produce toxic gases. According to Brake (2001) when the diesel engine intake air is mixed with refrigerant R134a and gets into the combustion chamber, has a potential to be converted into a highly toxic and corrosive gas, hydrogen fluoride.

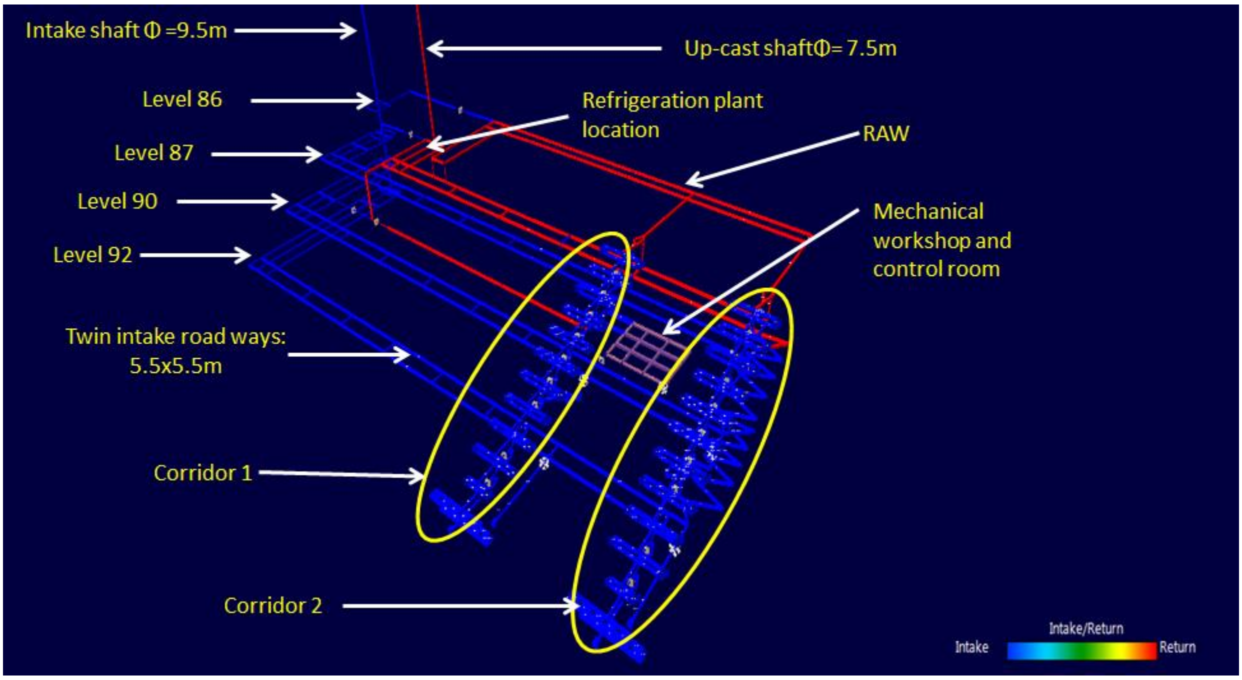


Figure 30: Schematic drawing of modelled mine- air-intakes (blue) and RAW (red)

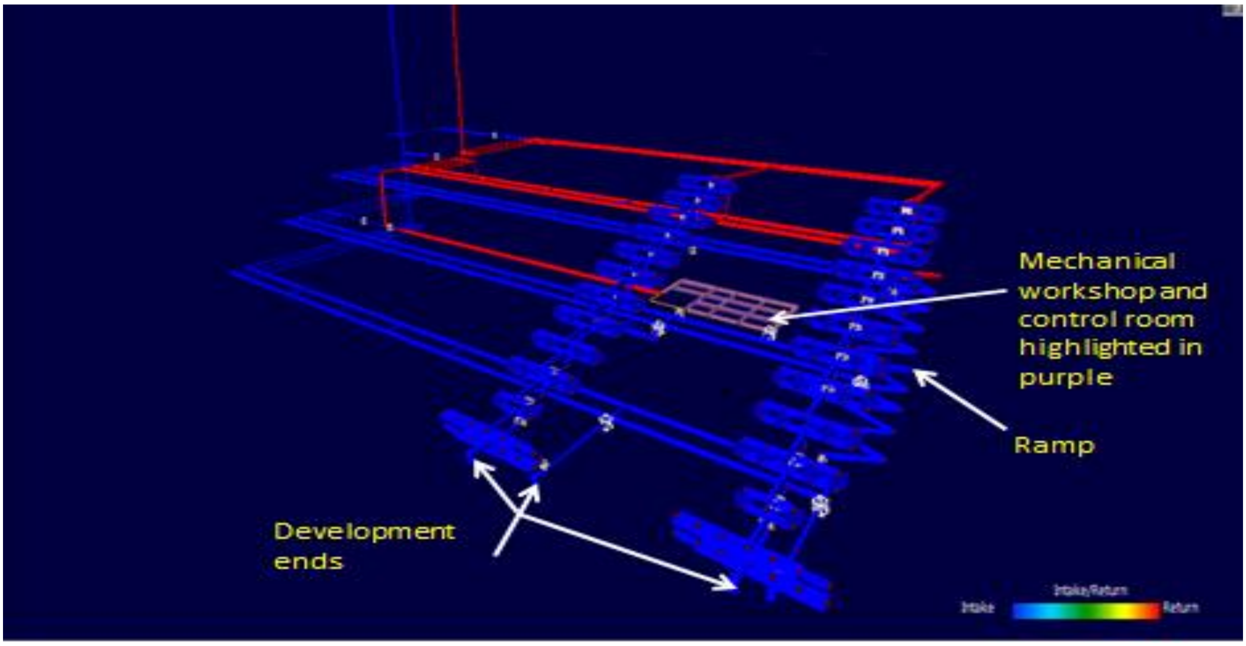


Figure 31: Tilted view showing development ends

4.2. ACCEPTABLE ENVIRONMENT

The manned mine was ventilated and cooled to achieve ambient operating temperatures of below 27.5^{0}C (wb) in all working areas including production zones (corridors), workshop, underground control room and travelling ways where the presence of workers is expected.

For the unmanned scenario, two rejection temperatures were used for areas where the presence of workers was expected and for production areas which were dedicated for machines only. The rejection temperature of travelling ways, workshop and control room was set at 27.5^{0}C (wb), while for production zones it was set at 40^{0}C (db).

The VUMA software model was used to establish the amount of air and cooling required for the two scenarios. From Table 6 in chapter 2, it is clear that a manufacturer of mining equipment can make equipment that work above ambient working temperature of 50^{0}C. However, for purpose of this project it would mean that no cooling would be required because the maximum VRT is around 50^{0}C. This scenario was considered but was discarded as the results from the model showed that air cooling power would be very low, in some cases reaching a value of zero. This scenario cannot allow workers to work anywhere underground because all travelling ways would be above thermally acceptable environment. Most of old diesel and electric equipment such as

Simba-M6-C-ITH drill rig shown in Figure 6 can operate in a maximum ambient temperature of 40^{0}C. A Number of studies carried out on automation has come to the conclusion that automation requires some level of human intervention and this is supported by Lynas and Horberry (2011) and Kumar et al.(nd)

4.2.1. HEAT INCREASES

The total heat load for manned and unmanned mining scenarios obtained from the model were around 48 000kW and 40 000 kW respectively. Both manned and unmanned scenarios had the same heat sources with varying degree of contribution as shown in Figure 32 below:

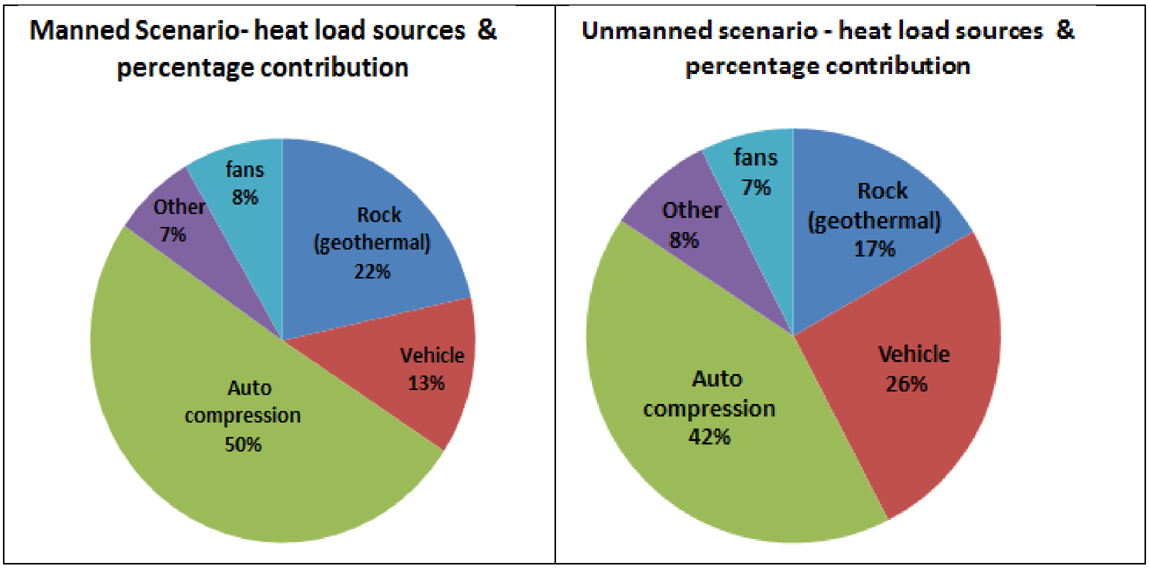


Figure 32: Heat load sources and contribution for manned and unmanned

4.2.1.1. AUTO-COMPRESSION

From the results obtained and illustrated in Figure 32 above it is clear that auto-compression contribution differs for the two scenarios. The difference between manned and unmanned scenarios is due the fact that heat generated by auto-compression is a function of mass of air taken underground. The amount of air required in the unmanned scenario is less than that required for the manned scenario. Hence, heat contribution due to auto-compression in unmanned is less compared to manned scenario. The cooling due to auto-decompression experienced in both scenarios was taken into account. The results shown are the resultant impact of auto-compression.

4.2.1.2. GEOTHERMAL

The geothermal results contribute differently towards heat load for the two scenarios as shown in Figure 32 above. This is due to different prevailing surfaces of airways temperature for both scenarios. The difference of temperatures of the rock and surface of airways is a function of heat conduction transfer and is represented by a formula shown in Equation 16 below. This equation illustrates that the heat transfer is directly proportional to the change in temperature. Hence, the results were as expected.

$$q = kA(t_1 - t_2)/b \quad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \textbf{\textit{Equation 16}}$$

Where; q= heat transfer

k= thermal conductivity (W/m^{0}C)

A= Area (m^2)

t = temperature (^{0}C), 1 is for hot and 2 for cold surface

b = thickness of the transferring medium (m)

For both scenarios all the parameters were kept constant except the temperature of the airways. The change in temperature (Δt) of the manned scenario airways rejection temperature and the rock was larger compared to the unmanned scenario. Hence, the results of geothermal contributed 22% of total heat load for manned compared to 17% on the unmanned scenario.

4.2.1.3. VEHICLE HEAT

The results obtained from the model show the percentages of the heat load contribution from vehicles were different for manned (13%) and unmanned (26%) as shown in Figure 32 above. During modelling, heat contribution of the machines was taken into account. Equation 17, which gives a relationship between heat and power rating, load on the engine as well as utilisation of the machine was used.

$$\mathrm{Heat(kW)} = \frac{\mathrm{Engine\ power\ rating(kW)}}{\mathrm{Thermal\ efficiency(\%)}}\ \mathrm{x\ Engine\ load(\%)\ x\ Utilisation(\%)} \ldots \textbf{\textit{Equation 17}}$$

Where; Thermal efficiency is 30% (typical)

Engine load (60% for light, 75% for average and 90% for heavy)

Utilisation as per shift schedule as shown in chapter 3 which is around 50% for manned and assumed to be 75% for unmanned.

Considering all the above; Equation 17 can then be simplified as follows:

$$\text{Machine heat(kW)} = \text{Engine power rating (kW)x C} \ldots\ldots\ldots\ldots\ldots\ldots \textbf{\textit{Equation 18}}$$

Where C is a constant and can be simplified as follows as shown in Table 20, as derived from VUMA software model (BBE).

Table 20: VUMA heat load machine categories and multiplying factors (BBE)

Category	Description	C value
VUMA Category 1	Light utilisation with machine on very light load. For example, mobile unit working on flat ground, with unit being used infrequently and carrying almost no load; or a stationary, utility-type unit used infrequently	0.5
VUMA Category 2	Moderate utilisation with machine on moderate load. For example, mobile unit working on gentle inclines with unit being used occasionally and carrying a minimal load; or a stationary utility-type unit used occasionally, or TBM working below rated production.	1
VUMA Category 3	Average utilisation with machine on average load. For example, mobile unit working on moderate inclines with unit being used regularly and carrying a medium load; or a stationary utility-type unit used continuously, or continuous-miner working at rated production.	1.5
VUMA Category 4	Heavy utilisation with machine on heavy load. For example, mobile unit working on steep inclines with unit being used continuously and carrying near-maximum load, or road-header working at above rated production.	2
VUMA Category 5	Very heavy utilisation with machine working on very heavy continuous full load rating. For example, mobile unit working on maximum inclines [as per unit specification] with unit in continuous use and carrying maximum design load, or road-header working at significantly above rated production. [It is unlikely that mining vehicles or other equipment will work at this rate continuously for full shift cycle times [exception would be haul trucks on long ramps] but user may wish to examine part of cycle.	2.5

4.2.1.4. AUXILIARY FANS AND OTHER HEAT SOURCES

The heat from fans and other heat sources remained constant at around 16% combined for both scenarios, though the number of fans required for the manned scenario was 79 and 108 for the unmanned. The reason was that the power rating of fans used in unmanned scenario was lower. The sum of fan power ratings was 3985kW and 2940kW for the manned and unmanned scenarios respectively. The input power to these fans is converted into heat. Hence, the heat produced by fans and other equipment proportional to the total heat load contributed around 16%.

The auxiliary fan electrical power required in the manned scenario is almost 50% of the main fan. The author believes that this would be a suitable case to look at the financial viability of using VOD and given that the shift utilisation is 50%. This cannot be contemplated for the unmanned scenario as the shift utilisation is high and air flow is required all the time to create a conducive environment for vehicles.

4.3. AIR REQUIRED

The total amounts of air required to create a thermally acceptable environment were 923kg/s (1000m^3/s) at a pressure drop of 5.3kPa for the manned scenarios and 652 kg/s (706m^3/s) at a pressure drop of 2.2kPa for the unmanned respectively. The pressure drops and associated air quantity assisted in validating the model as they obey the fan law, $P = RQ^2$, which is Equation 9 in chapter 3. The ventilation factors for the manned scenario were determined as 4.7kg/s/kt/month and for the unmanned 2.2kg/s/kt/month.

Four main fans were selected with twin speed to be able to supply air for both manned and unmanned scenarios and were located on top of the up-cast shaft. This is different from the current practice of installing fixed flow main fans. A variable fan speed drive was considered. Apart from prohibitive costs, the variable speeds drives are not recommended for main fans as discussed in chapter 2. Hence, twin speed fans were chosen. This would ensure an air flow at all times underground as the likelihood of all four fans mechanically failing at the same time would be unlikely. Centrifugal fans were chosen for this study as they have a reputation of being robust and reliable. These fans are designed to cater for both the manned and unmanned scenarios. This would be of advantage especially for the unmanned scenario in an event of emergency whereby workers expected to enter the production zone. The fan speed would be increased to the top to create an acceptable environment for workers. The fan speed for manned and unmanned scenario was found to be 740 rpm and 510 rpm respectively. A company by the name of Air Blow Fans provided fan curves for this project in order to validate the calculated electrical power required for main fans. Figure 33 show the system operating point when all four fans are running for both scenarios. Figures 34 and 35 show each fan operating point for the manned and unmanned scenarios respectively. The results are as expected as they are in line with fan laws when connected in parallel. The pressure remained the same and the sum of air volume of all fans correlated with the volumetric flow rate of system operating point for both scenarios. The graphs confirmed that the calculations for electrical power required were correct.

In addition, auxiliary fans were installed in working areas to ensure a supply of adequate air. The manned scenario required a total number of 79 auxiliary fans with total input of 3985kW while the unmanned required 108 with a total motor rating of 2940kW. The number of auxiliary fans required was due to the number of working areas and use of smaller fans in the unmanned scenario.

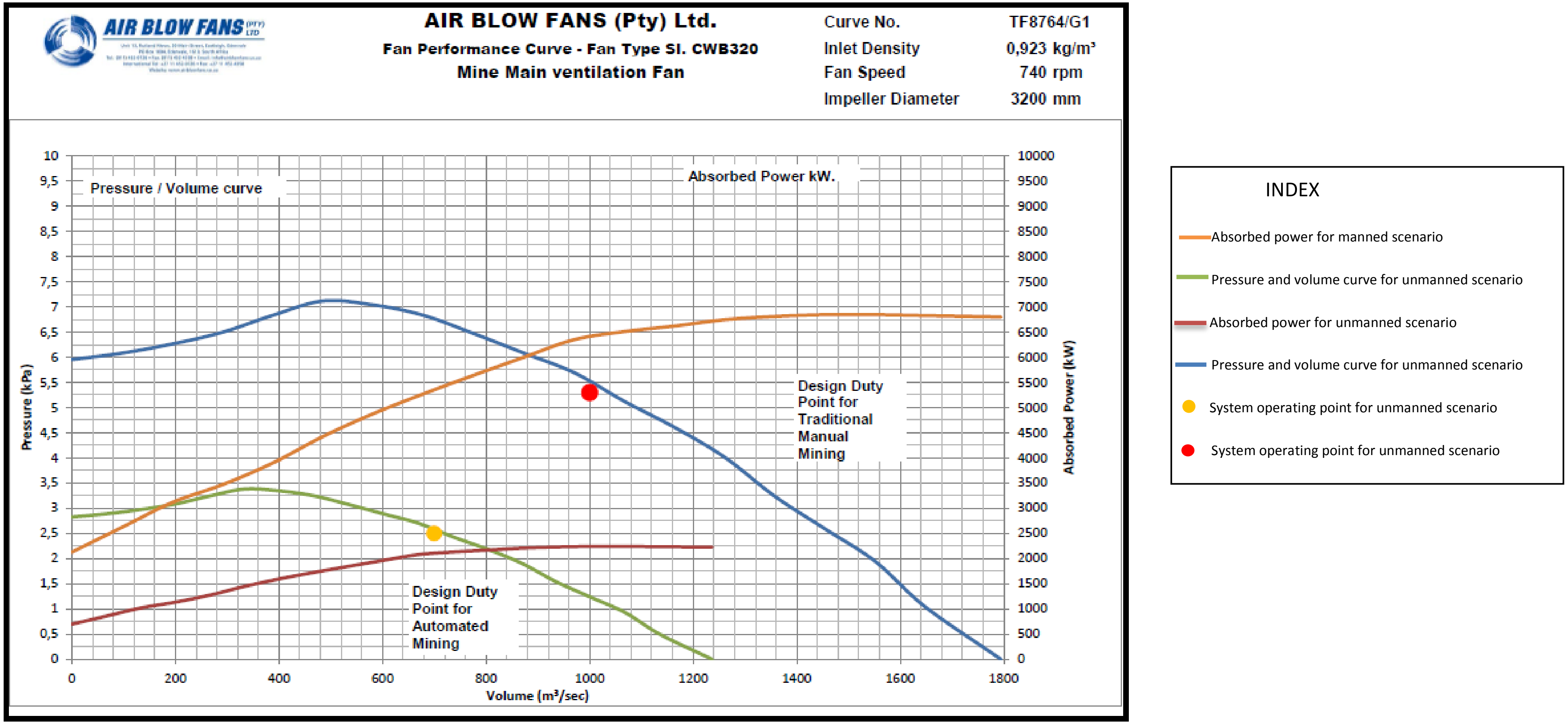


Figure 33: Fan curves - System operating points for automated (unmanned) and traditional mining (mechanised)

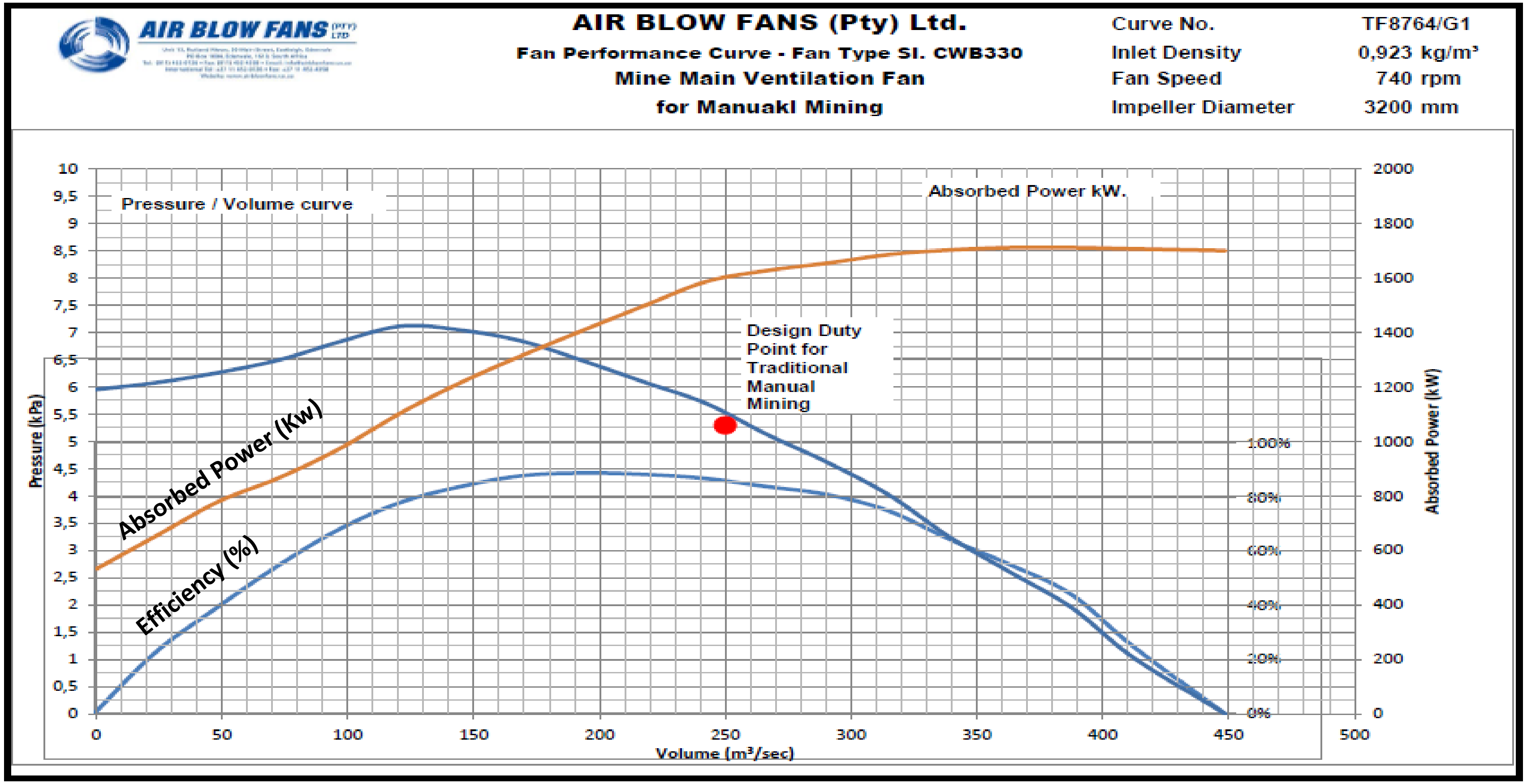


Figure 34 Fan curves – single fan operating point for traditional mining (mechanised)

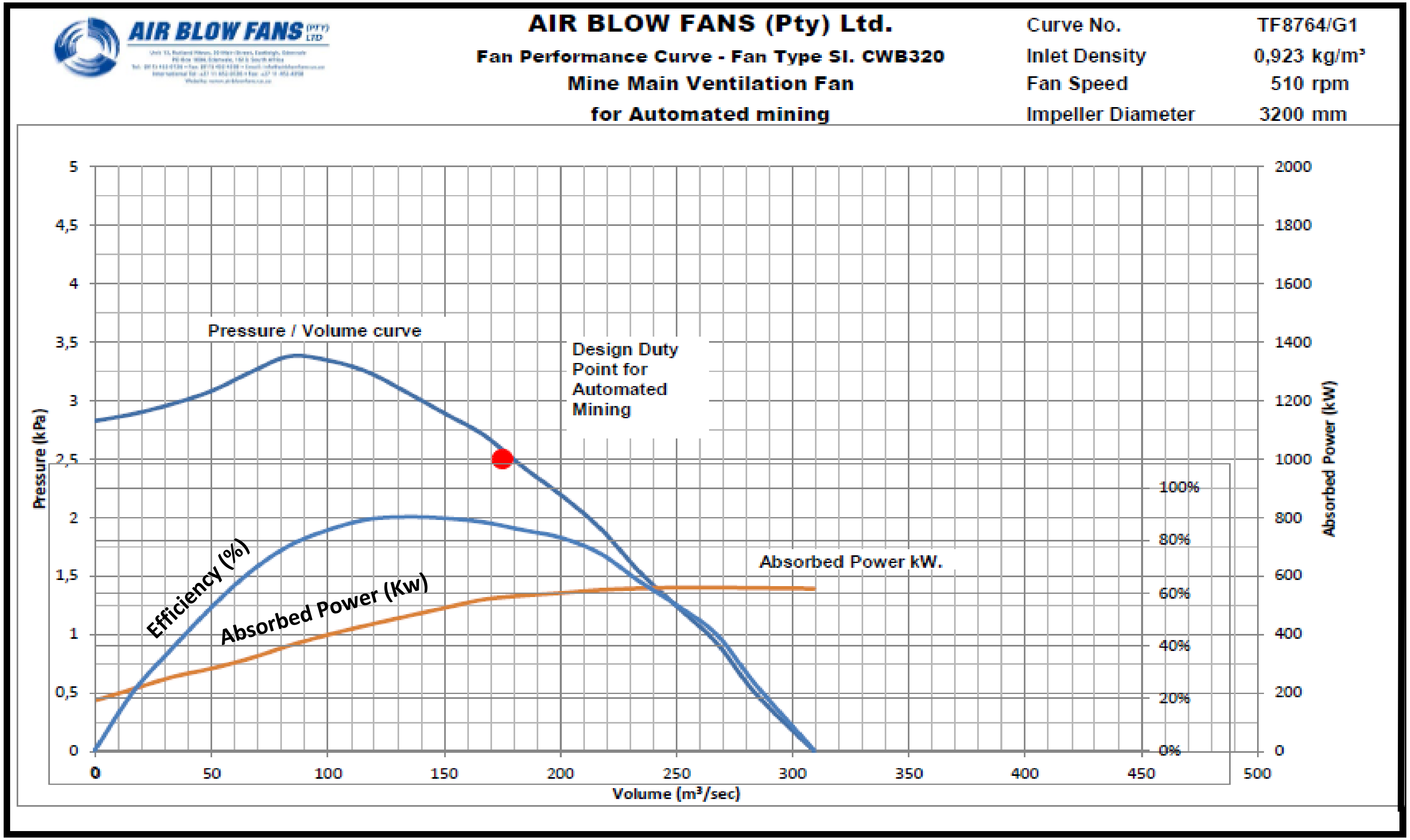


Figure 35: Fan curves – single fan operating point for automated mining (unmanned)

4.4. COOLING REQUIREMENTS

The results from the model revealed that the manned scenario required a total cooling capacity of 36 000kW and the unmanned 11 000kW. The strategy that was adopted was to first cool the bulk of the air on surface (primary cooling) to overcome auto-compression and create winter conditions during summer. Secondly cool the air underground to derive maximum benefits of positional efficiency (secondary and tertiary cooling). The cooling stages and splits of the cooling are shown in Table 21 below.

Table 21: Cooling stages and capacities

Cooling stages	Manned	Unmanned
Primary(kW)	25 000	10 000
Secondary (Kw)	10 500	1 000
Tertiary (Kw)	500	0
Total	36 000	11 000

The surface refrigeration plant was meant to only operate in hot seasons as the winter season provides cold air for underground mine. This means it operates for nine months and is switched-off for three months. A similar cooling system was used for unmanned with a smaller capacity.

Two refrigerants were considered namely ammonia and R134a. Ammonia has the advantage that it can cool water to 1^0C cost effectively. However, the risk is its toxicity in the event that it leaks. Moreover, the rule of thumb is to put the plant 200m away from the working place and considering the toxicity factor, a number of stringent safety systems must be put in place (Wilson & Pieters, 2008).

Horizontal spray chamber with two stages of cooling has been chosen for this project. This chamber is widely used in the mining industry due to its superior thermal performance, flexibility, capital and running cost (Wilson & Pieters, 2008). Figure 36 on the next page shows the process flow diagram of a surface cooling system for manned scenario The BAC depicted is a direct contact cooler (i.e. the air and water are in direct contact with each other).

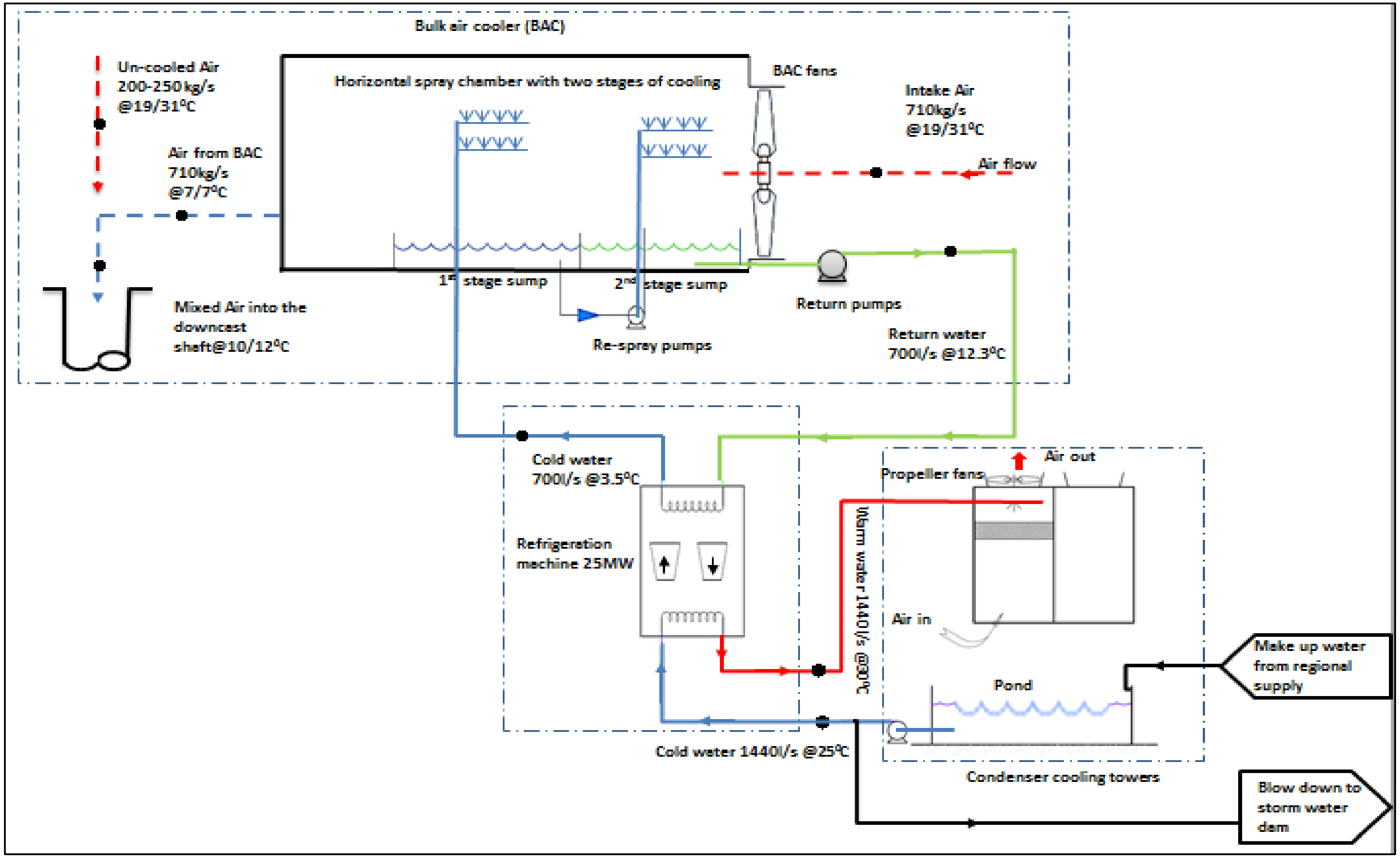


Figure 36: Process flow diagram of surface BAC (25MW) for manned scenario

The combination of the air quantity required and cooling yielded thermally acceptable environments for both manned and unmanned scenarios. The wet-bulb (wb) and dry-bulb (db) temperatures for the manned scenario was generally below rejection temperature of 27^{0}C (wb) and 32^{0}C (db) respectively. Figure 37 below depicts the wet (a) and dry bulb (b) temperatures of production areas for the manned scenario.

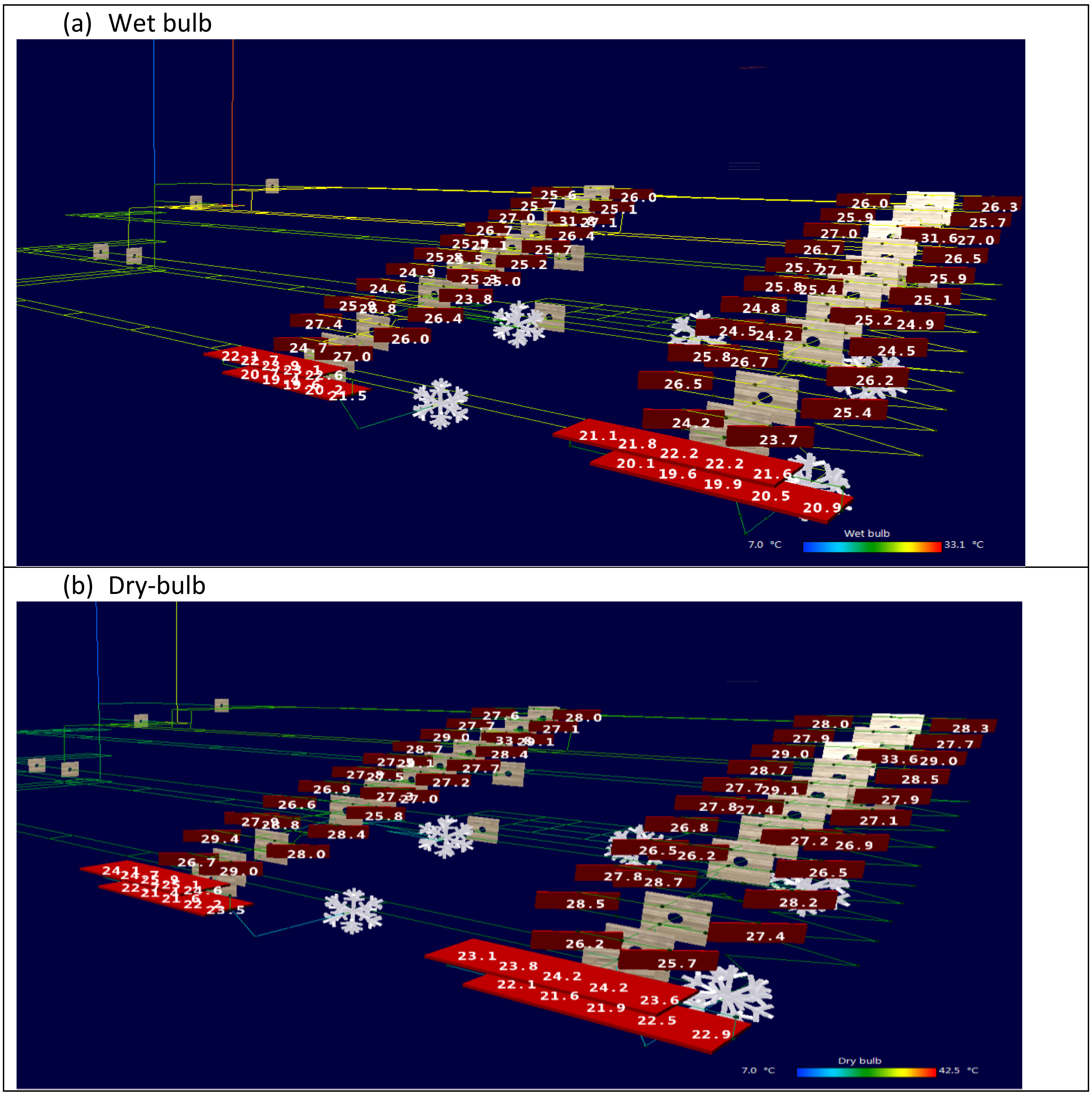
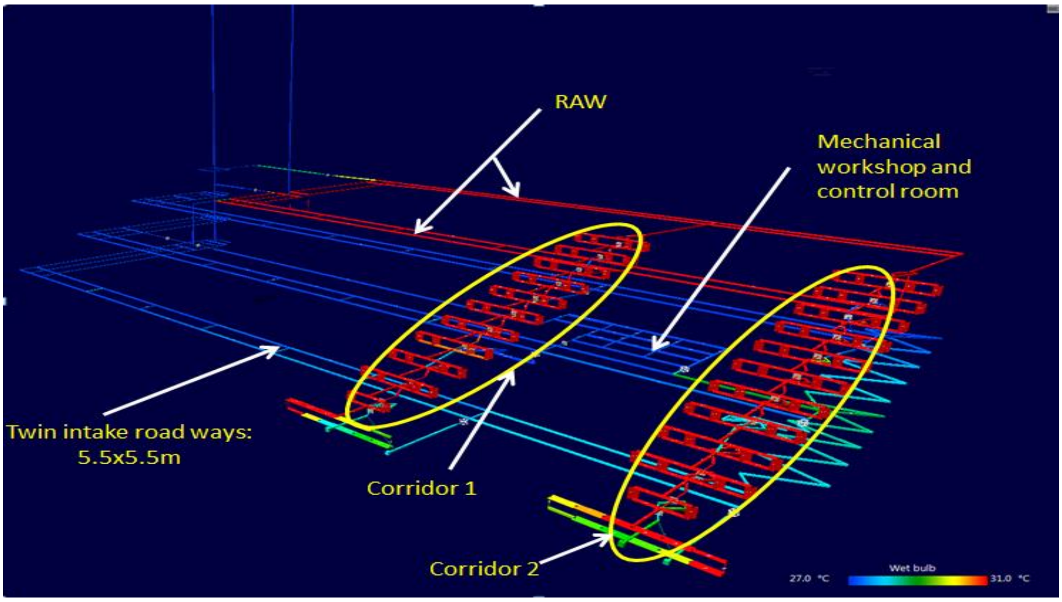


Figure 37: Production temperature (a) wet bulb and (b) dry bulb temperatures of manned scenario

In the unmanned scenario, underground workshop and control room rejection temperature were set at 27^{0}C (wb) and 32^{0}C (db) similar to the manned scenario. The dry-bulb rejection temperature for the corridors (i.e. production zones) was set at 40^{0}C. Figure 38 below, illustrates that both dry-bulb and wet-bulb temperatures for workshop and control room are around 27^{0}C shown in blue lines while the temperatures for the corridors are above 31^{0}C (wb) and around 36^{0}C (db). The environmental conditions in the unmanned conditions are not thermally acceptable for employees. Hence, no workers would be allowed in the corridors in the unmanned scenario.

(a) Wet bulb



(b) Dry bulb

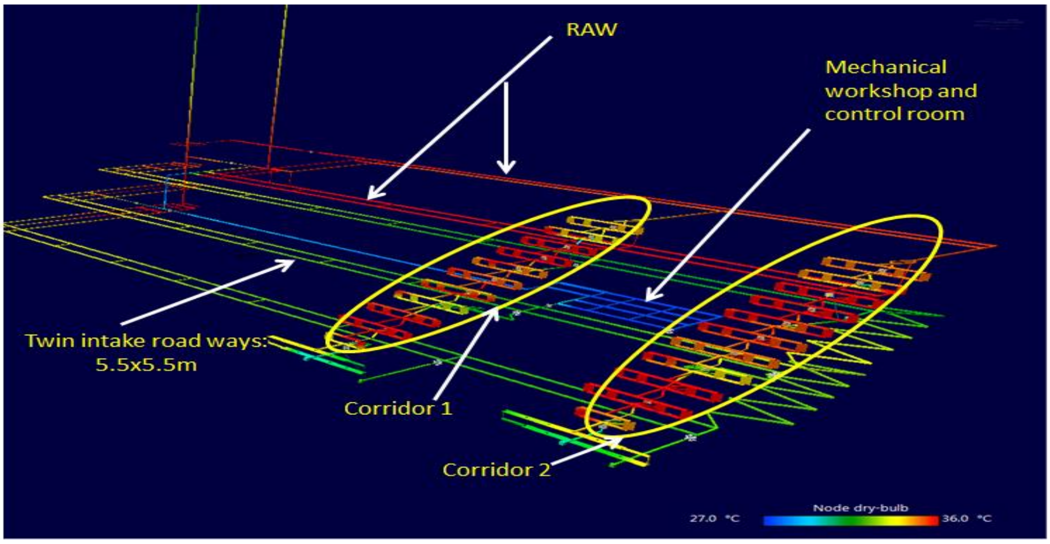


Figure 38: (a) wet-bulb and (b) dry-bulb temperature in production zones (Corridors) and workshop for unmanned scenario

The unmanned environmental conditions from the model are not suitable for workers as they fall under the category of an excessively hot environment. The mandatory code of practice (CoP), thermal stress guidelines (Hermanus, 2002) defines an excessively hot environment as one where the wet-bulb temperature is greater than 32.5^{0}C or dry-bulb is greater than 37^{0}C. The historic analyses done as per the Mandatory CoP indicate that the fatality rate due to heat stroke doubled where such thermal conditions prevailed. The wet bulb temperature ranges between 29 and 36^{0}C which in most cases is above 32.5^{0}C, see Figure 39 for details.

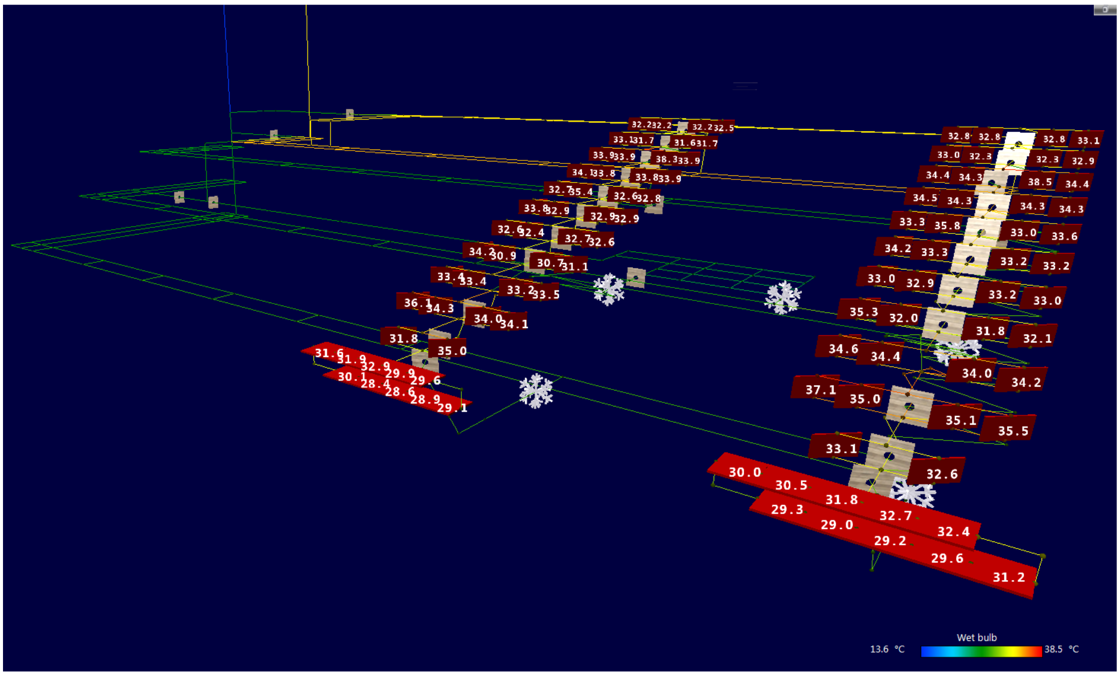


Figure 39: Wet-bulb temperature in production areas in unmanned scenario

According to the definition of an excessively hot environment, the above can be defined as such using the wet-bulb or dry-bulb temperature threshold. The unmanned production zones fit this definition using either of the two temperature limits. The dry-bulb temperature ranges between 31^{0}C and 40^{0}C which in most instances is close to 37^{0}C as shown in Figure 40. Hence, no workers are allowed in the production zones(i.e.in the environment where machine are operating).

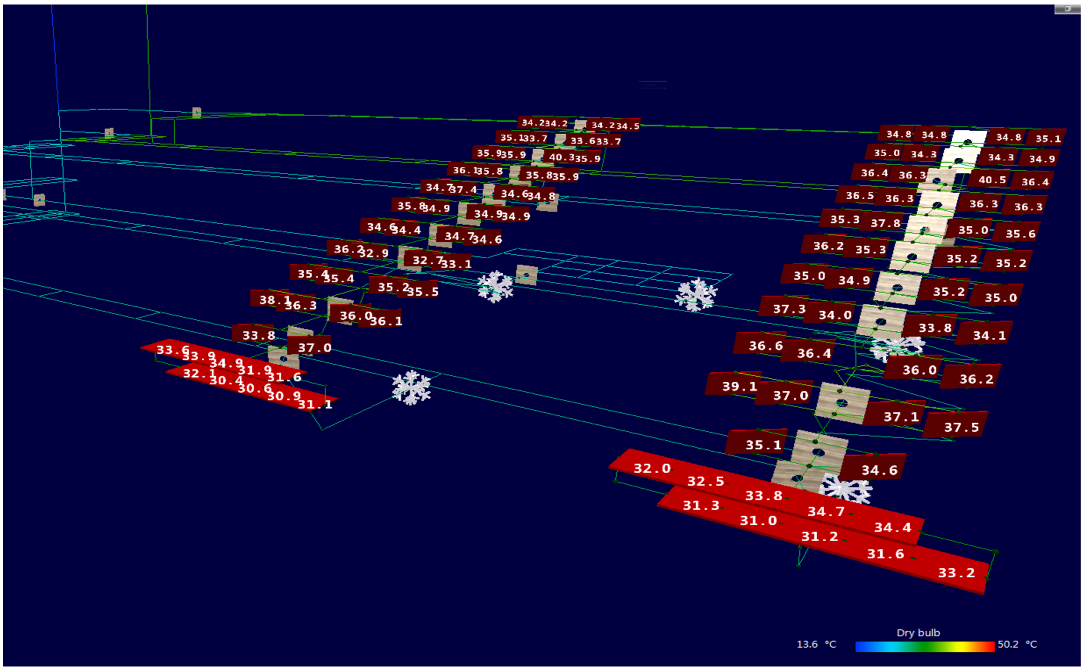


Figure 40: Dry-bulb temperature in production areas in unmanned scenario

4.4.1. AIR COOLING POWER

The air cooling power is critical for the health of workers. It was considered during modelling that the air cooling power in the environment where the presence of workers is expected should be more than the metabolic rate of workers as per their functions. There is a direct relationship between the type of work that a worker performs and the metabolic rate. Figure 41 shows work categories and expected metabolic rate for a mechanised operation.

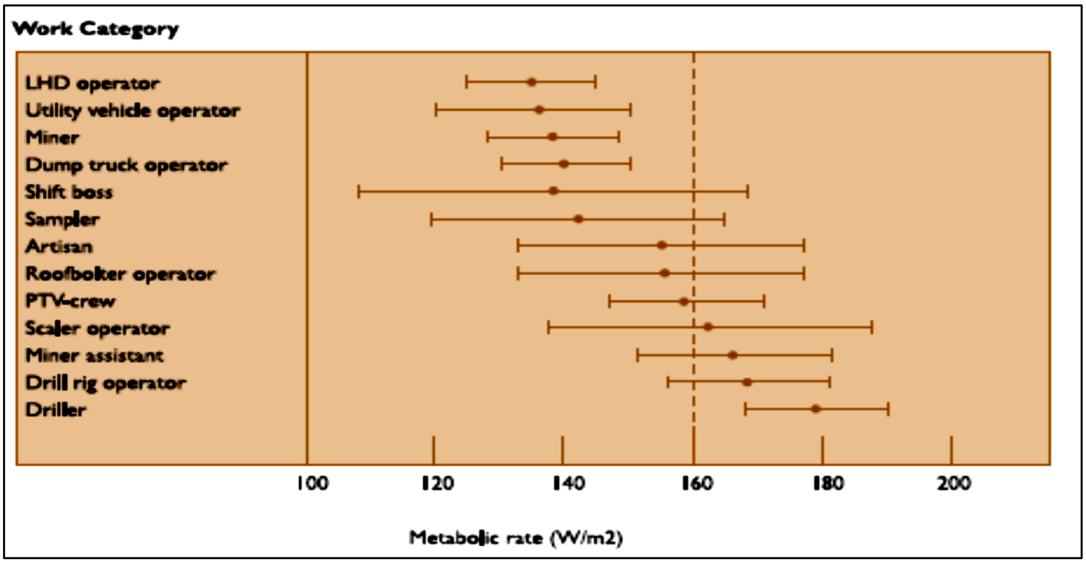


Figure 41: Work category and expected metabolic rate

When designing ventilation and cooling systems of a mine, it is important to design in such a way that the cooling power exceeds the metabolic rate of the job category expected to be performed in each working environment. The mandatory Code of Practice based on guidelines issued by the Chief Inspector of mines (Hermanus, 2002) on thermal stress considers strenuous work as any work that associated with mean metabolic rate of 160W/m^2. As may be noted in Figure 41 above a number of occupations exceed this rate. Hence, the design criteria was taken as 180W/m^2 as a minimum in the manned scenario.

Generally, the working areas in corridors (production zones) for manned has air cooling capacity higher than 200 W/m^2, as shown in Figure 42 below. On the other hand the air cooling power in production zones of the unmanned operation is lower than 160W/m^2, as, no employees are allowed in the production zones of unmanned scenario. The general cooling power for this scenario is around 55 W/m^2 as shown in Figure 43.

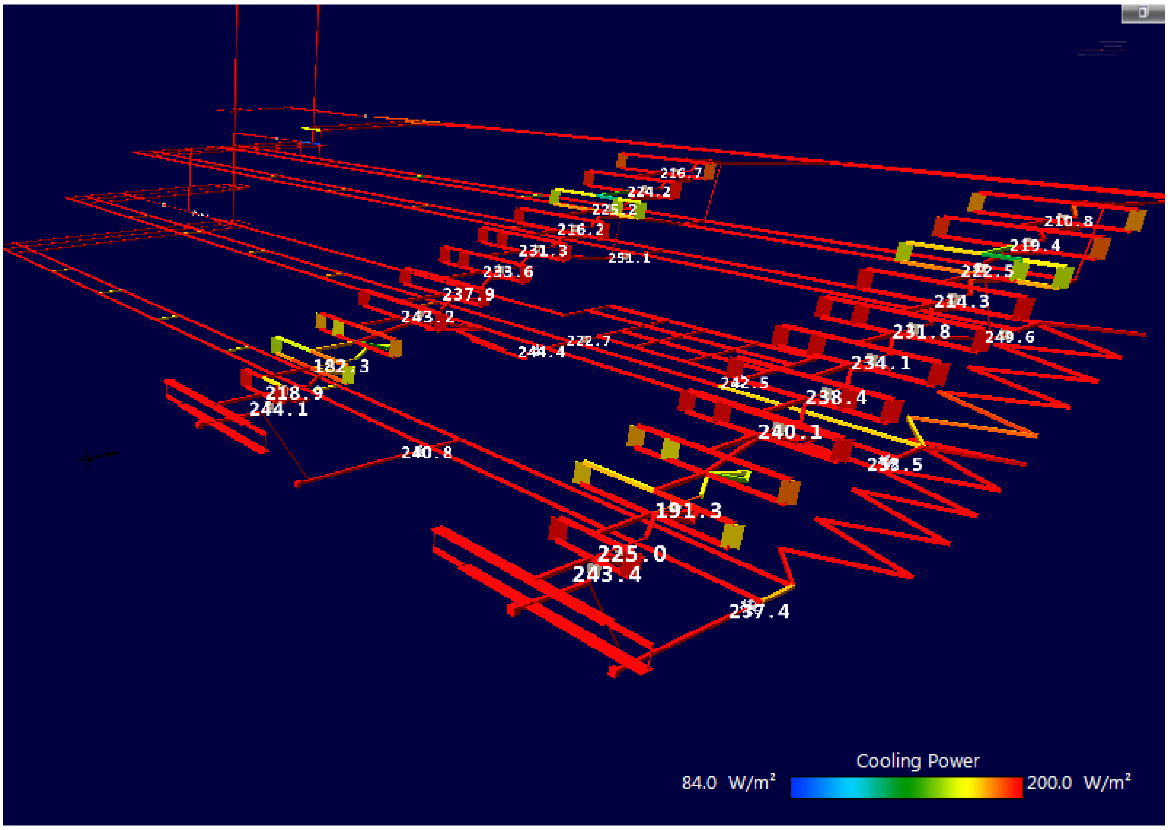


Figure 42: Air cooling power in manned scenario

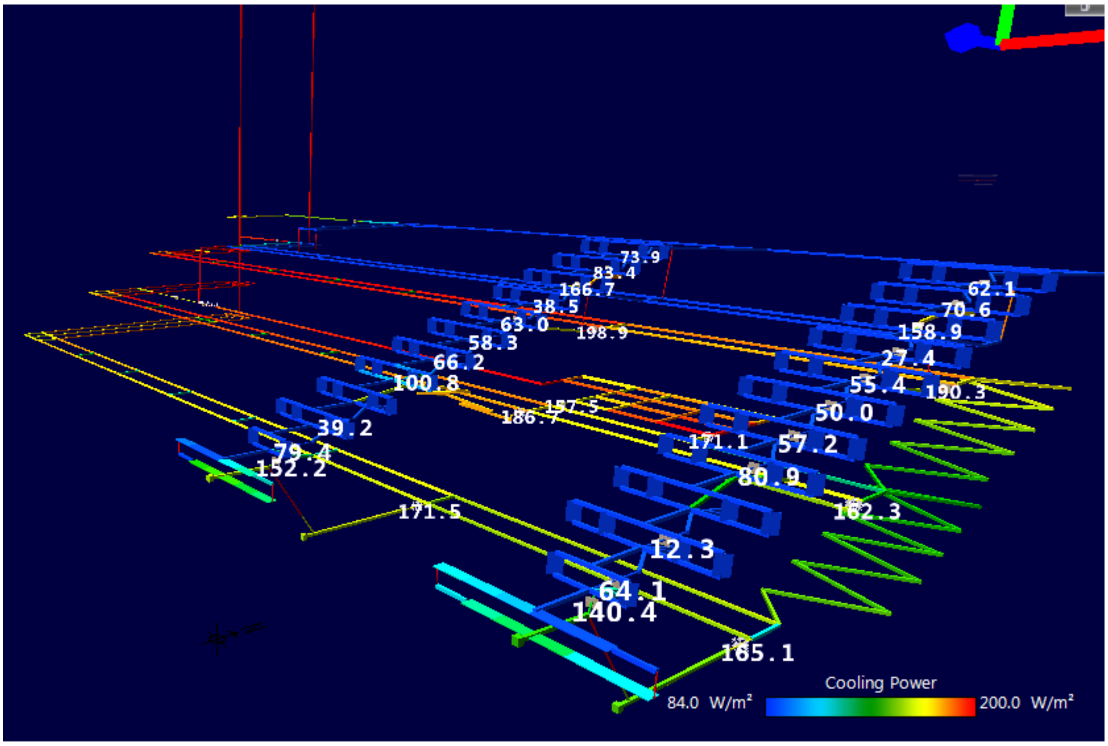


Figure 43: Air cooling power in unmanned scenario

4.5. VENTILATION AND REFRIGERATION COSTS

The Eskom mega flex tariff structure, shown in Figure 44 below, was applied to the results obtained from the model in order to calculate cost for ventilation and cooling. Low season tariff is R0.54/kWhr from January to March. From April the MYPD annual increase of 8% was effected and from June to August high season rates of 0.87/kWhr was applied and low season tariff were applied from September to December. The low season means that the electricity demand is low and there is no strain on the electricity grid, while high season means that there is high electricity demand which is normal during winter season.

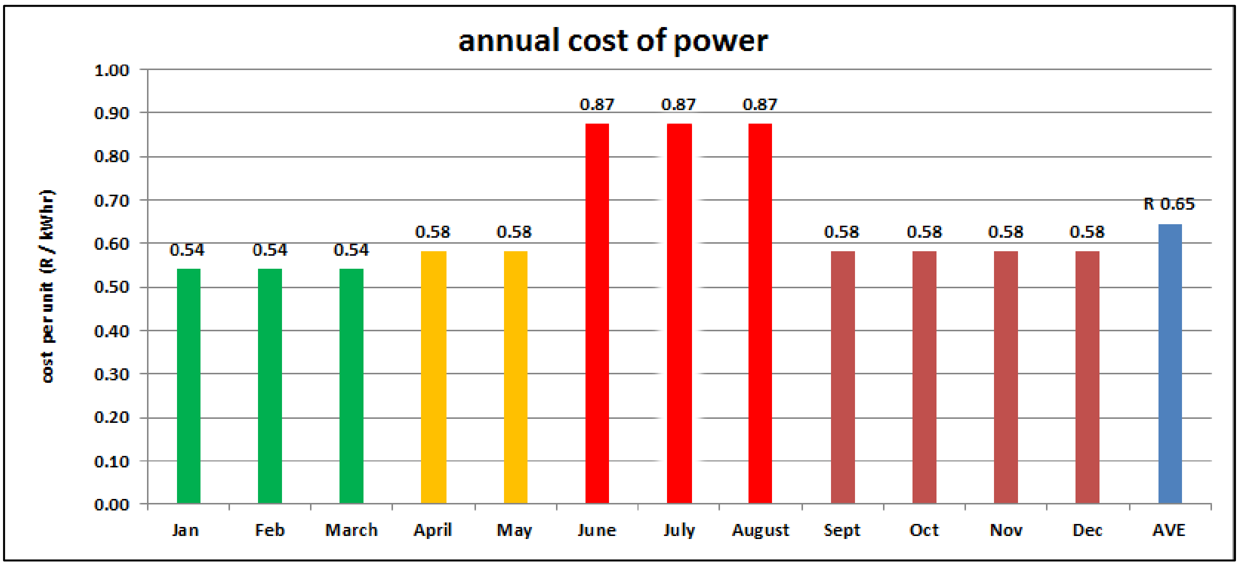


Figure 44: Simplified Eskom mega flex tariff structure (Eskom, 2014)

4.5.1. MAIN FANS ELECTRICAL POWER REQUIRMENTS

In order to obtain input power to main fans, the pressure drop obtained from the results of the model was multiplied by the air quantity, motor efficiency and fan efficiency. The motor and fan efficiencies were obtained from a personal discussion with Nico Du Preez, Engineering Manager of Air Blow Fans on 11 November 2014. The motor and fan efficiency were estimated to be 93% and 95% respectively. The electrical power for the manned scenario was calculated to be 6705 kWe, while for the unmanned scenario it was 2037 kWe. The summary of input data to calculate the required electrical energy is shown in Table 22 below for both scenarios.

Table 22: Air and electrical power required for main fans

	Units	manned	Unmanned
Total fan Volume	m^3/s	1000	700
Total Fan mass flow	kg/s	923	651.8
Fan pressure	kPa	5.3	2.3
Air density	kg/m^3	0.93	0.93
Motor efficiency	%	93	93
Fan efficiency	%	85	85
Overall efficiency	%	79	79
Motor input power	kWe	6705	2037

4.5.2. MAIN FANS ELECTRICITY COST

The monthly electricity tariff was multiplied by the motor input power to obtain the hourly cost of electricity. Then the hourly cost was multiplied by 24 hours to obtain the daily cost. The monthly cost was derived by multiplying the daily cost by 365 days and the product was divided by 12 months. The total annual cost was obtained by adding all the 12 months' electricity cost. The annual cost of electricity for main fans is R38 million manned versus R11.5 million unmanned scenario. The results of surface main fans for manned and unmanned scenarios are shown in Tables 23 and 24.

MINE FULLY MANNED (FULLY COOLED): SURFACE MAIN FANS (100% speed)

The following data was used to calculate the electrical input power which was subsequently used to calculate electricity costs in Table 23 on the next page. The motor input power was calculated by using the equating below:

$Power\ (kW) = fan\ Pressure\ (kPa)\ x\ air\ quantity(m^3/s)$...***Equation 19***

The power obtained from equation was then divided by overall efficiency which was calculated from the formula below:

$Overall\ efficiency(\%) = motor\ efficiency\ (\%)\ xfan\ efficiency\ \ (\%)$..***Equation 20***

Total fan Volume	=	1000	m^3/s
Total Fan mass flow	=	923	kg/s
Fan pressure	=	5.3	kPa
Air density	=	0.923	kg/m^3
motor efficiency	=	93	%
fan efficiency	=	85	%
overall efficiency	=	79.05	%
motor input power	**=**	**6705**	**kWe**

Table 23: Operational cost of main fans for manned scenario

	Jan	Feb	March	April	May	Jun	July	Aug	Sept	Oct	Nov	Dec	AVE
R/kWhr	0.54	0.54	0.54	0.58	0.58	0.87	0.87	0.87	0.58	0.58	0.58	0.58	**0.65**
Power consumed at surface main fans	6 705	6 705	6 705	6 705	6 705	6 705	6 705	6 705	6 705	6 705	6 705	6 705	**6 705**
hourly cost of power consumed at surface main fans	3 620	3 620	3 620	3 910	3 910	5 865	5 865	5 865	3 910	3 910	3 910	3 910	**4 326**
daily cost of power consumed at surface main fans	86 892	86 892	86 892	93 843	93 843	140 765	140 765	140 765	93 843	93 843	93 843	93 843	**103 836**
monthly cost of power consumed at surface main fans	2 642 960	2 642 960	2 642 960	2 854 397	2 854 397	4 281 595	4 281 595	4 281 595	2 854 397	2 854 397	2 854 397	2 854 397	**3 158 337**
annual cost of power consumed at surface main fans	**37 900 049**												

MINE UNMANNED (PARTIALLY COOLED): SURFACE MAIN FANS (100% speed)

The same method used to calculate motor input power for the manned scenario using equations 19 and 20 was used for the unmanned scenario and the results were used to calculate electricity cost shown in Table 23 on the next page.

Total fan Volume	=	700	m^3/s
Total Fan mass flow	=	651.8	kg/s
Fan pressure	=	2.3	kPa
Air density	=	0.93	kg/m^3
motor efficiency	=	93	%
fan efficiency	=	85	%
overall efficiency	=	79	%
motor input power	**=**	**2037**	**kWe**

Table 24: Annual cost electricity of main fans for unmanned scenario

	Jan	Feb	March	April	May	Jun	July	Aug	Sept	Oct	Nov	Dec	AVE
R/kWhr	0.54	0.54	0.54	0.58	0.58	0.87	0.87	0.87	0.58	0.58	0.58	0.58	**0.65**
Power consumed at surface main fans	2 037	2 037	2 037	2 037	2 037	2 037	2 037	2 037	2 037	2 037	2 037	2 037	2 037
hourly cost of power consumed at surface main fans	1 100	1 100	1 100	1 188	1 188	1 782	1 782	1 782	1 188	1 188	1 188	1 188	1 314
daily cost of power consumed at surface main fans	26 395	26 395	26 395	28 507	28 507	42 761	42 761	42 761	28 507	28 507	28 507	28 507	31 543
monthly cost of power consumed at surface main fans	802 861	802 861	802 861	867 090	867 090	1 300 636	1 300 636	1 300 636	867 090	867 090	867 090	867 090	959 419
annual cost of power consumed at surface main fans	**11 513 034**												

4.5.3. UNDERGROUND AUXILIARY FANS OPERATING COST

The above process of estimating electricity cost for surface fans was followed in establishing the underground auxiliary fans costs. The annual cost of auxiliary fans for manned was R22.5 million and R16.6 million for unmanned. The detailed electricity costs for both manned and unmanned scenarios are shown in Tables 25 and 26 respectively.

OPERATIONAL COST OF UNDERGROUND AUXILLARY FANS - MANNED

Table 25: Annual cost of electricity of underground auxiliary fans for manned scenario

	Jan	Feb	March	April	May	Jun	July	Aug	Sept	Oct	Nov	Dec	AVE
R/kWhr	0.54	0.54	0.54	0.58	0.58	0.87	0.87	0.87	0.58	0.58	0.58	0.58	0.65
Power consumed all underground auxiliary fans	3 985	3 985	3 985	3 985	3 985	3 985	3 985	3 985	3 985	3 985	3 985	3 985	3 985
Hourly cost of power consumed all underground auxiliary fans	2 152	2 152	2 152	2 324	2 324	3 486	3 486	3 486	2 324	2 324	2 324	2 324	2 572
Daily cost of power consumed all underground auxiliary fans	51 646	51 646	51 646	55 777	55 777	83 666	83 666	83 666	55 777	55 777	55 777	55 777	61 716
Monthly cost of power consumed all underground auxiliary fans	1 570 887	1 570 887	1 570 887	1 696 558	1 696 558	2 544 837	2 544 837	2 544 837	1 696 558	1 696 558	1 696 558	1 696 558	1 877 210
Annual cost of power consumed all underground auxiliary fans	22 526 520												

OPERATIONAL COST OF UNDERGROUND AUXILLARY FANS - UNMANNED

Table 26 : Electricity cost of underground auxiliary fans for unmanned scenario

	Jan	Feb	March	April	May	Jun	July	Aug	Sept	Oct	Nov	Dec	AVE
R/kWhr	0.54	0.54	0.54	0.58	0.58	0.87	0.87	0.87	0.58	0.58	0.58	0.58	**0.65**
Power consumed all underground auxiliary fans	2 940	2 940	2 940	2 940	2 940	2 940	2 940	2 940	2 940	2 940	2 940	2 940	2 940
Hourly cost of power consumed all underground auxiliary fans	1 588	1 588	1 588	1 715	1 715	2 572	2 572	2 572	1 715	1 715	1 715	1 715	1 897
Daily cost of power consumed all underground auxiliary fans	38 102	38 102	38 102	41 151	41 151	61 726	61 726	61 726	41 151	41 151	41 151	41 151	45 532
Monthly cost of power consumed all underground auxiliary fans	1 158 948	1 158 948	1 158 948	1 251 664	1 251 664	1 877 496	1 877 496	1 877 496	1 251 664	1 251 664	1 251 664	1 251 664	1 384 943
Annual cost of power consumed all underground auxiliary fans	**16 619 314**												

4.5.4. REFRIGERATION POWER

The optimum temperature for air sent underground was achieved by mixing airways from the cooled air in the BAC and uncooled air. The design wet-bulb temperature of air below the collar elevation for the manned scenario was set at around 10^0C while for the unmanned was around 14^0C during summer. During winter the BAC does not operate for 3 months. The electricity power required for surface BAC to achieve cooling duty of manned (25MW) and unmanned (10MW) was 5938kWe and 2793kWe respectively.

Table 27 illustrate the electrical power required in the refrigeration plant to cool the air for both scenarios. The following formula and assumptions were used.

$$P = \frac{M_w \times H \times g \times \rho}{\eta \times 1000}$$ ……………………………………………………………………***Equation 21***

Where:

P = Power of the pump (kW)

M_w = Mass flow rate of water (kg/s)

g = specific gravity (9.79m/s^2)

ρ = Density of the water (1000kg/m^3)

η= Efficiency of the pump (%)

H= Head of the pump (m)

$$\text{Condenser duty} = \text{Evaporator duty(kW)} + \text{Compressor power(kW)}$$ ………..***Equation 22***

And

$$\text{condenser flow rate(l/s)} = \frac{\text{Condenser duty(kW)}}{\Delta t \times C_{PW}}$$ ……………………………………..***Equation 23***

Where:

Δt = Change in temperature of the water – normally 5^0C

C_{pw} = Thermal capacity of the water (4.187kJ/kg)

Assumptions made based on current industry practices:

- BAC pump head = 35m
- BAC fan pressure drop = 500Pa
- Condenser pump head = 25m

- Condenser fan pressure = 160Pa
- Condenser fan efficiency = 70%
- Control and instrumentation Lighting (C&I, Lighting)

Table 27: Estimation of electrical power required for surface refrigeration plants for both scenarios

			Units	Manned	Unmanned
Evaporator	water	flow rate	kg/s	700	560
		Inlet water temperature	°C	3.5	3.5
		Outlet water temperature	°C	12.3	8
	Air	Mass flow rate	kg/s	710	560
		Barometric pressure	kPa	84.0	84.0
		Temperature (wb)	°C	19	19.0
		Temperature (db)	°C	31	31.0
		Temperature of air leaving the coil	°C	7	7
		Density	kg/m³	0.94	0.94
		BAC DUTY	kW	25000	10000
		Condenser water flow rate	kg/s	1440	590
		Condenser air flow rate	kg/s	1440	590
Ai mixing		Uncooled air	kg/s	210-220	
		Mixed air temperature(wb/db)	°C	10.4/12.4	
Electrical Input power		Total Evaporator duty	kW	25772	10608.85
		COP		6	6
		Compressor input power	kWe	4295	1768
		BAC pumps	kWe	300	240.1
		BAC fan	kWe	472	368.75
		Condenser pumps	kWe	441	180.6875
		Condenser fans	kWe	329	134.8571429
		Other (C&I, lighting)	kWe	100	100
		Total power	**kWe**	**5938**	**2793**

Subsequent to establishing the air and cooling required, the electrical power and then associated costs were calculated. The annual cost of the manned scenario was around R20 million versus the unmanned, which was R9.3 million. The detailed calculated electricity costs for manned and unmanned are shown in Tables 28 and 29 respectively.

SURFACE BAC FOR MANNED

Table 28: Electricity costs of surface refrigeration plant for manned scenario

	Jan	Feb	March	April	May	Jun	July	Aug	Sept	Oct	Nov	Dec	AVE
R/kWhr	0.54	0.54	0.54	0.58	0.58	0.87	0.87	0.87	0.58	0.58	0.58	0.58	**0.65**
Power consumed all underground auxiliary fans	5 938	5 938	4 750	4 453	3 563	-	-	-	3 563	4 453	4 750	5 938	4 816
Hourly cost of power consumed all underground auxiliary fans	3 206	3 206	2 565	2 597	2 078	-	-	-	2 078	2 597	2 770	3 463	2 729
Daily cost of power consumed all underground auxiliary fans	76 953	76 953	61 562	62 332	49 865	-	-	-	49 865	62 332	66 487	83 109	65 495
Monthly cost of power consumed all underground auxiliary fans	2 340 645	2 340 645	1 872 516	1 895 922	1 516 738	-	-	-	1 516 738	1 895 922	2 022 317	2 527 896	1 992 149
Annual cost of power consumed all underground auxiliary fans	**19 921 488**	(NB: surface refrigeration plant completely stopped for the three winter months)											

OPERATIONAL ELECTRICITY COST OF SURFACE BAC FOR UNMANNED

Table 29: Annual cost of surface refrigeration plant for unmanned scenario

	Jan	Feb	March	April	May	Jun	July	Aug	Sept	Oct	Nov	Dec	AVE
R/kWhr	0.54	0.54	0.54	0.58	0.58	0.87	0.87	0.87	0.58	0.58	0.58	0.58	**0.65**
Power consumed all underground auxiliary fans	2 793	2 793	2 234	2 094	1 676	-	-	-	1 676	2 094	2 234	2 793	2 265
Hourly cost of power consumed all underground auxiliary fans	1 508	1 508	1 206	1 221	977	-	-	-	977	1 221	1 303	1 629	1 283
Daily cost of power consumed all underground auxiliary fans	36 191	36 191	28 953	29 315	23 452	-	-	-	23 452	29 315	31 269	39 087	30 803
Monthly cost of power consumed all underground auxiliary fans	1 100 818	1 100 818	880 654	891 662	713 330	-	-	-	713 330	891 662	951 107	1 188 883	936 918
Annual cost of power consumed all underground auxiliary fans	**9 369 183**	(NB: surface refrigeration plant completely stopped for the three winter months)											

4.5.5. SECONDARY AND TERTIARY REFRIGERATION PLANT ELECTRICITY COST

The manned scenario required five bulk air cooling plants (BACs) underground at the following locations.

- Workshop and control room: 500kW
- Level 90 cross cut (x/c) corridor 1 : 3500kW
- Level 90 cross cut (x/c) corridor 2 : 3500 kW
- Level 92 cross cut (x/c) corridor 1 : 1750kW
- Level 92 cross cut (x/c) corridor 2 : 1750Kw

The unmanned scenario requires only one secondary BAC underground of 1000kW for the workshop and control room. Figure 45 below shows the capacity and location of the BACs for both manned and unmanned scenarios.

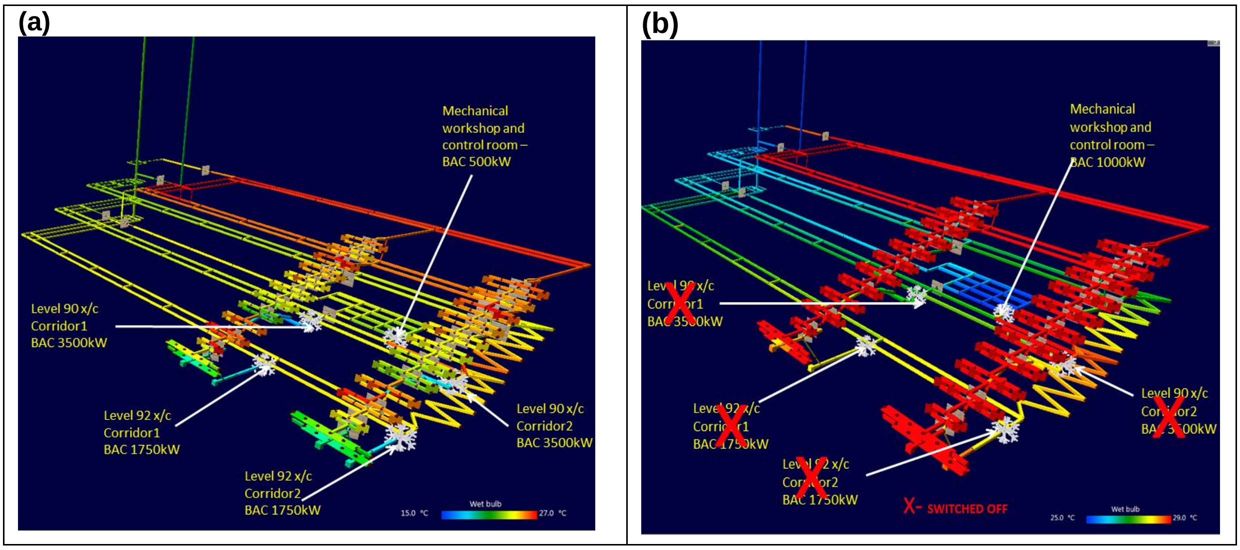


Figure 45: The location of BAC's and capacities on manned (a) and unmanned (b)

The total electricity required was 4913kW for the manned and 508kW for the unmanned scenario. The unmanned scenario require little electrical input power as it only needs 1000kW cooling capacity for the workshop and control room while other BACs remain switched off. In calculating the electrical power required the assumption and formulas used to estimate power requirements for surface BAC were adopted with the exception of COP which is typically 3.5 for underground refrigeration plants.

Table 30 below is a summary of the calculation of the power required for all the BACs underground. The table has combined all the elements of all the five BACs required underground. For example, to arrive at a chilled water flow rate of 235 l/s all the chilled water flow rates in five BACs were added.

Table 30: Electricity consumption of underground BACs for both manned and unmanned

	Units	**Manned**	**Unmanned**
Evaporator duty	kW	12161	1082
COP		3.55	3.55
chilled water flow rate	ℓ/s	235	20
combined BAC duty	Kw_R	11000	1000
compressor input	kW_e	3426	305
BAC fans	kW_e	345	31
Chilled water distribution. pumps	kW_e	672	46
evaporator pumps	kW_e	58	5
BAC spray pumps	kW_e	86	0
condenser pumps	kW_e	227	21
other (C&I, lighting)	kW_e	100	100
Total u/g condenser water flow rate	**kg/s**	**740**	**70**
TOTAL POWER	**kW_e**	**4913**	**508**

These electricity requirements were used to calculate the total cost of electricity of operating underground BACs. The costs for manned and unmanned scenarios were R16.4 million and R1.6 million respectively. Tables 34 and 35 provide detailed calculations.

The workshop and control room needed for the manned scenario was 500kW to achieve temperatures of around 22^{0}C (wb) and 25^{0}C (db) while for the unmanned scenarios required 1000 kW to achieve temperatures of around 24^{0}C (wb) and 26^{0}C(db) after mixing of cooled air with uncooled air quantities as shown in Table 31 for both scenarios.

Table 31: Underground workshop and control room required temperatures

			Units	Manned	Unmanned
workshop BAC	water	flow rate	kg/s	15	15
		Inlet water temperature	°C	9	9
		Outlet water temperature	°C	21	15
	Air	Mass flow rate	kg/s	15	40
		Barometric pressure	kPa	110.0	110.0
		Inlet temperature (wb)	°C	24	27.0
		Inlet temperature (db)	°C	29	32.0
		outlet temperature(wb/db)	°C	13.1/13.1	17/17
		Density	kg/m³	1.2	1.2
		Cooler duty	**kW**	**500**	**1000**
Air mixing		Uncooled air	kg/s	70	48
		Mixed air temperature(wb/db)	°C	22/25	24/26

Two BACs with each having cooling capacity of 3500kW were installed for each corridor at level 90 as shown in Figure 45 above. During modelling it was required to keep the BAC operational to maintain a thermally acceptable environment for the manned scenario while it was switched off for the unmanned scenario. Table 32 below, shows the required amount of air to be cooled and mixed with uncooled air to achieve a temperature of 18^{0}C before being sent to production zones for the manned scenario and it also shows that it was switched off for the unmanned scenario.

Table 32: BACs capacity installed at level 90 and required mixed air

			Units	Manned	Unmanned
90-C1X/C AND 90-C2 X/C BAC'S	water	flow rate	kg/s	70	
		Inlet water temperature	°C	9	
		Outlet water temperature	kg/s	21.0	
	Air	Mass flow rate	kg/s	100	Switched-OFF
		Volume flow	m^3/s	83	
		Barometric pressure	kPa	110.2	
		Inlet temperature (wb)	°C	24	
		Inlet temperature (db)	°C	29	
		outlet temperature	°C	12.2	
		Density	kg/m^3	1.2	
		Cooler duty	**kW**	**3500**	**0**
Air mixing		Uncooled air	kg/s	70	
		Mixed air temperature (wb/db)	°C	18/19	

Furthermore, two additional BACs with each having a cooling capacity of 1750kW were installed for each corridor at level 92 as shown in Figure 45 above**.** Once again, the BAC had to be in operation for the manned scenario and be switched off for the unmanned scenario. Table 33 below, shows the required amount of air to be cooled and mixed with uncooled air to achieve a temperature of 18^0C saturated air before being sent to production zones for the manned scenario and it also shows that it was switched off for the unmanned scenario.

Table 33: BACs capacity installed at level 92 and required mixed air temperatures

			Units	Manned	Unmanned
92-C1X/C AND 92-C2 X/C BAC'S	water	flow rate	kg/s	40	
		Inlet water temperature	°C	10	
		Outlet water temperature	kg/s	21.0	
	Air	Mass flow rate	kg/s	50	Switched-OFF
		Volume flow	m^3/s	38	
		Barometric pressure	kPa	111.2	
		Inlet temperature (wb)	°C	24	
		Inlet temperature (db)	°C	29	
		outlet temperature(wb/db)	°C	12.4/12.4	
		Density	kg/m^3	1.3	
		Cooler duty	**kW**	**1750**	**0**
Air mixing		Uncooled air	kg/s	30	
		Mixed air temperature	°C	18	

UNDERGROUND COOLING COST - MANNED

Table 34: Operating cost of underground refrigeration plant for manned scenario

	Jan	Feb	March	April	May	Jun	July	Aug	Sept	Oct	Nov	Dec	AVE
R/kWhr	0.54	0.54	0.54	0.58	0.58	0.87	0.87	0.87	0.58	0.58	0.58	0.58	0.65
Power consumed at surface refrigeration plant	4 896	4 896	3 917	3 672	2 938	-	-	-	2 938	3 672	3 917	4 896	3 972
hourly cost of power consumed at surface refrigeration plant	2 644	2 644	2 115	2 142	1 713	-	-	-	1 713	2 142	2 285	2 856	2 250
daily cost of power consumed at surface refrigeration plant	63 459	63 459	50 767	51 401	41 121	-	-	-	41 121	51 401	54 828	68 535	54 010
monthly cost of power consumed at surface refrigeration plant	1 93 (Thopil & Pouris, 2013)0 197	1 930 197	1 544 157	1 563 459	250 767	-	-	-	1 250 767	1 563 459	1 667 690	2 084 612	1 642 812
annual cost of power consumed at surface refrigeration plant	**16 428 118**	(NB: surface refrigeration plant completely stopped for the three winter months)											

UNDERGROUND COOLING COSTS- UNMANNED

Table 35: Operating cost of underground refrigeration plant for unmanned scenario

	Jan	**Feb**	**March**	**April**	**May**	**Jun**	**July**	**Aug**	**Sept**	**Oct**	**Nov**	**Dec**	**AVE**
R/kWhr	0.54	0.54	0.54	0.58	0.58	0.87	0.87	0.87	0.58	0.58	0.58	0.58	0.65
Power consumed at surface refrigeration plant	484	484	387	363	290	-	-	-	290	363	387	484	392
Hourly cost of power consumed at surface refrigeration plant	261	261	209	212	169	-	-	-	169	212	226	282	222
Daily cost of power consumed at surface refrigeration plant	6 269	6 269	5 015	5 078	4 062	-	-	-	4 062	5 078	5 416	6 770	5 335
Monthly cost of power consumed at surface refrigeration plant	190 669	190 669	152 535	154 442	123 553	-	-	-	123 553	154 442	164 738	205 922	162 280
Annual cost of power consumed at surface refrigeration plant	**1 622 802**	(**NB:** surface refrigeration plant completely stopped for the three winter months)											

4.5.6. TOTAL COST OF VENTILATION AND COOLING

The overall results of the calculated costs of electricity required in relation to the tonnage are shown in Table 36 for the two scenarios. It is clear from the Table that the total cost of ventilating and cooling the underground environment for manned mine to maintain thermally acceptable environment is R97 million per annum compared to R39 million for the unmanned scenario. When considering that the tonnage production for the two scenarios are different it is important to consider ratios such as kWh per ton and ventilation and cooling costs per ton produced. The kWh and the electricity costs per ton for manned scenario were calculated as 63kWh and R41 respectively. The results for unmanned was 17kWh per ton and cost of electricity was R11 per ton.

Based on the results and only looking at the electricity cost, a mine can realise cost saving of about R58 million per annum with automation and increased rejection temperature. The results also show that this strategy is still beneficial when one becomes conservative and assume the same rate of production as the manned scenario of 197 036 tons per month. Applying an electricity cost per ton of R11 of the unmanned scenario result in annual cost of R26 million. This translated to an annual cost saving of R71 million.

The results demonstrated that automation and increased rejection temperature will still be beneficial in terms of electricity cost savings. It is important to note that the overall profitability of a mine cannot be based on this study alone because it excludes the capital cost of the automation system. However, it suffices to say the payback period will be short and profitability will be improved.

The ventilation factor for the manned scenario was 4.7kg/s per kt/month which is within the acceptable range of 3 to 6kg/s per kt/month (Burrows, et al., 1989) . On the other hand, the unmanned ventilation factor was 2.2 kg/s per kt/month which is roughly 50% of the manned scenario. This makes logical sense as only a small quantity of air is required in the workshop, control room and travelling ways to maintain a rejection temperature of 27^{0}C (wb). In addition to this, the dry-bulb rejection temperature of 40^{0}C also required less air to maintain acceptable environment for machinery and also reduces the heat transfer from the rock.

Table 36: Summary of ventilation and cooling electricity costs for all scenarios

Manned		
SUMMARY	**Power manned (KW)**	**Manned Annual cost(ZAR)**
Surface Main fan	6705	37 900 049
Underground auxiliary fans	3985	22 526 520
Surface BAC	5938	19 921 488
Underground BACs	4896	16 428 118
Total Annual cost		96 776 174
Average monthly cost		8 064 681
Average electricity price (R/kWh)		0.65
Average kWh per month		12 407 202
Tonnage per month		197 054
Average kWh per ton		63
Elec. Cost per ton		40.93
Unmanned		
SUMMARY	**Power manned (KW)**	**Unmanned Annual cost(ZAR)**
Surface Main fan	2037	11 513 034
Underground auxiliary fans	2940	16 619 314
Surface BAC	2792	9 369 183
Underground BACs	484	1 622 802
Total Annual cost		39 124 333
Average monthly cost		3 260 361
Aver electricity price(R/kWh)		0.65
Average kWh per month		5 015 940
Tonnage per month		295 379
Average kWh per ton		17
Elec. Cost per ton		11.04
Unmanned with same production as manned (worst case)		
Average monthly cost		3 260 361
Aver electricity price(R/kWh)		0.65
Average kWh per month		5 015 940
Tonnage per month		197 054
Average kWh per ton		25
Elec. Cost per ton		16.55

These savings would not come without a risk. There are risks that must be addressed.

4.6. RISK ASSESSMENT

The project is about increasing the rejection temperature in automated mines (unmanned scenarios) where workers are not permitted in the production zones. The risk assessment is largely applicable for an automated system.

The conceptual automated mine has the potential of causing significant risks that requires assessment and management. During modelling it became apparent that with the current known technology it is almost impossible to replace workers completely with machinery. As a point of departure the control room and mechanical workshop were placed underground for strategic reasons which means there will be an interaction among equipment, workers and underground environment though limited compared with mechanization (manned) scenario.

In the literature review it was revealed that when automation is introduced to eliminate human error, the results are sometimes new and often more catastrophic errors (Woods, 1995). It is for this reason that a risk assessment was carried out for this project.

The risk assessment was conducted to determine whether there is any likelihood of a potentially hazardous situation that can cause damage to assets, injury and death to workers. Evaluating automation system risks would help determine the most serious hazards which would enable development of action plans to prevent or mitigate them.

4.6.1. HAZARD IDENTIFICATION

Underground full automation systems are still in its conceptual stage. Nevertheless a risk assessment was done through analysis of its anticipated operation activities. The assessment began by identifying possible hazards that could occur during automation of the operation. The process of identifying system hazards was broken down into three broad categories as indicated below:

- Identification of hazards that can cause the automation system to fail
- Identification of hazards that can cause unsafe environmental condition
- Identification of hazards related to operations and maintenance of the automated system.

The hazard identification process was followed by quantification of risk using the risk matrix in Table 37.

4.6.1.1. CAUSES OF AUTOMATION SYSTEM TO FAIL

Since automation is a process of remotely controlling machinery that are operating underground and relying on communication infrastructure; the system may fail due to one of the following hazards:

- Fall of ground
- Fire
- Unskilled operators
- Lack/shortage of electricity

4.6.1.2. UNSAFE ENVIRONMENTAL CONDITIONS

During the operation of an automated system, there is a likelihood that an unsafe environment may prevail and impede the production operation.

- Poor visibility due to dust ,smoke from fire and water vapour
- Flooding
- High ambient temperature
- High humidity
- Mud-rush
- Failure of ventilation and refrigeration system
- Gases

4.6.1.3. OPERATIONAL AND MAINTENANCE HAZARDS

There are also other automation system hazards which result from operational and maintenance issues as indicated below:

- Falling of objects from overhead crane in a workshop
- Workers in a close proximity to automated machinery being injured during maintenance and after breakdowns
- Failure or malfunctioning of any automated equipment
- Mechanical failure of equipment such as brakes
- Collision of automated machinery with each other or with the side wall
- Lack of skilled labour

- Poor quality spare parts
- Unauthorised entry of workers in production zones
- Poor maintenance of equipment
- Noise

4.6.1.4. EXISTING HAZARD CONTROL MEASURES

Automation systems have integrated built-in safety features to prevent unwanted events from happening. The safety features are incorporated into the programmable logic control (PLC) software which communicates with all the sensors. In most cases, for each threat, one or more control measures are in place to prevent the risk from materializing.

Table 37 provides details of identified automation system hazards, available controls and risk value. Since, the automation system is still in its conceptual stage which means there is lack of actual data, the likelihood of an event happening and was based on research, consultations and the author’s judgement. The risk values were used in ranking the risks and the appetite for risk (acceptable risk) was set at 12, as shown in Figure 46. The major risks which were above the acceptable level were associated with lack or inadequate skills.

Occupational Health and Safety Risk Assessment

Table 37: Risk assessment

	Likelihood				
Impact score	1	2	3	4	5
	Rare	Unlikely	Possible	Likely	Almost certain
5 Catastrophic	5	10	15	20	25
4 Major	4	8	12	16	20
3 Moderate	3	6	9	12	15
2 Minor	2	4	6	8	10
1 Negligible	1	2	3	4	5

Risk Assessment Form

Activity	Hazard no.	Hazard identified	Consequence	Available Controls	Likelihood	Severity	Risk value	Risk Classification
Maintenance	1	Noise	hearing loss, hearing handicap and hearing disability	Hearing protection	2	4	8	
All activities	2	Fall of ground	damage to communication system, equipment and injuries and fatalities to workers	Support system and barring procedure	3	5	15	
Blasting and tramming	3	Dust	contraction of illness by workers	Dust suppression system, re-entry period after the blast	2	4	8	
Blasting and tramming	4	Dust	high cost maintenance ,damage to engine and poor visibility	Dust suppression system, re-entry period after the blast	2	5	10	
Use of diesel equipment	5	Diesel Particulate Matter	Cancer illness resulting from exposure to carcinogenic DPM	Adequate ventilation and no entry for workers in production zones	2	3	6	
Transportation	6	Vehicle accidents	damages to equipment, injuries and fatality to workers resulting from vehicle to vehicle, vehicle to human , and vehicle to equipment	Proximity detector system and collision avoidance system	2	3	6	
Procurement	7	Poor quality of spare parts (skills)	Early failure of equipment leading to loss of production	Use recommended spare parts from OEMs	3	5	15	
All activities	8	ground vibrations	damage to refuge bay and serious injures	Support system standard	2	5	10	

	Likelihood				
Impact score	1	2	3	4	5
	Rare	Unlikely	Possible	Likely	Almost certain
5 Catastrophic	5	10	15	20	25
4 Major	4	8	12	16	20
3 Moderate	3	6	9	12	15
2 Minor	2	4	6	8	10
1 Negligible	1	2	3	4	5

Risk Assessment Form

Activity	Hazard no.	Hazard identified	Consequence	Available Controls	Likelihood	Severity	Risk value	Risk Classification
Loading, drilling & barring	9	Dust	silicosis , headaches, irritation eye, nose and throat	Dust suppression system, re-entry period after the blast and PPE	2	5	10	
All activities	10	Fire	damage to property, injuries and fatalities to workers	Fire suppression and warning system	1	5	5	
	11	Flooding	drowning workers	Drainage system	1	5	5	
Services/maintenance	12	Water leakage	increasing humidity which reduce performance of diesel equipment	Drainage system	1	3	3	
Loading, drilling & barring	13	High ambient temperature	Heat disorders and poor performance	Adequate ventilation and no entry for workers in production zones	1	5	5	
Loading, drilling & barring	14	High ambient temperature	High emission of NOX from diesel engines and poor engine performance	Adequate ventilation and no workers in production zones	4	2	8	
vehicle maintenance	15	Lack of / poor skills(maintenance)	poor maintenance resulting in vehicle breakdowns	Training and development system and motivation	4	5	20	
Automation System and mine design	16	Lack of / poor skills(Design)	loss of production, accidents & build-up of gasses	Training and development system and motivation	3	5	15	
Maintenance	17	poor drainage system	increasing humidity and heat transfer from the rock resulting in heat disorders	Monitoring system	2	4	8	
All activities	18	Poor lighting	Poor visibility and may results in accidents and demotivation of employees	Maintenance system	3	4	12	
All activities	19	Electricity shortage/	Poor ventilation, no production & build-up of gases	Generator backup	2	4	8	

	Likelihood				
Impact score	1	2	3	4	5
	Rare	Unlikely	Possible	Likely	Almost certain
5 Catastrophic	5	10	15	20	25
4 Major	4	8	12	16	20
3 Moderate	3	6	9	12	15
2 Minor	2	4	6	8	10
1 Negligible	1	2	3	4	5

Risk Assessment Form

Activity	Hazard no.	Hazard identified	Consequence	Available Controls	Likelihood	Severity	Risk value	Risk Classification
Loading, drilling & barring	20	Mechanical failure of mobile equipment	Accidents and poor production	Preventative maintenance	2	5	10	
Handling/storing/use of explosives	21	Uncontrolled explosions	result in fatal injuries and damage to equipment	Sensors to monitor build-up of gases, Adherence to procedures	2	5	10	
Loading, drilling & barring	22	Unauthorised entry in production zones	Exposure to hazardous environment	Human tracking system	1	5	5	
All activities	23	Mechanical failure of main fans	Poor ventilation	4 fans instead of 1, Emergency procedure with critical spare parts	1	2	2	
Loading from ore passes	24	Mud rush	Poor production, damages to equipment	Drainage system-keeping water away from rock handling system	2	2	4	
All remotely controlled vehicles	25	Automation system failure	Poor production, accidents and poor ventilation	Training and development system and motivation	4	4	16	
All activities	26	Fatigue/substance abuse/boredom	Poor production, accidents and poor health	electronic Sensors, aptitude test, psychometric and medical screen, breathalyser test	2	4	8	
All activities	27	Air leakage/ damage to ventilation controls	Poor and expensive ventilation	Backfill, control and monitoring	2	3	6	

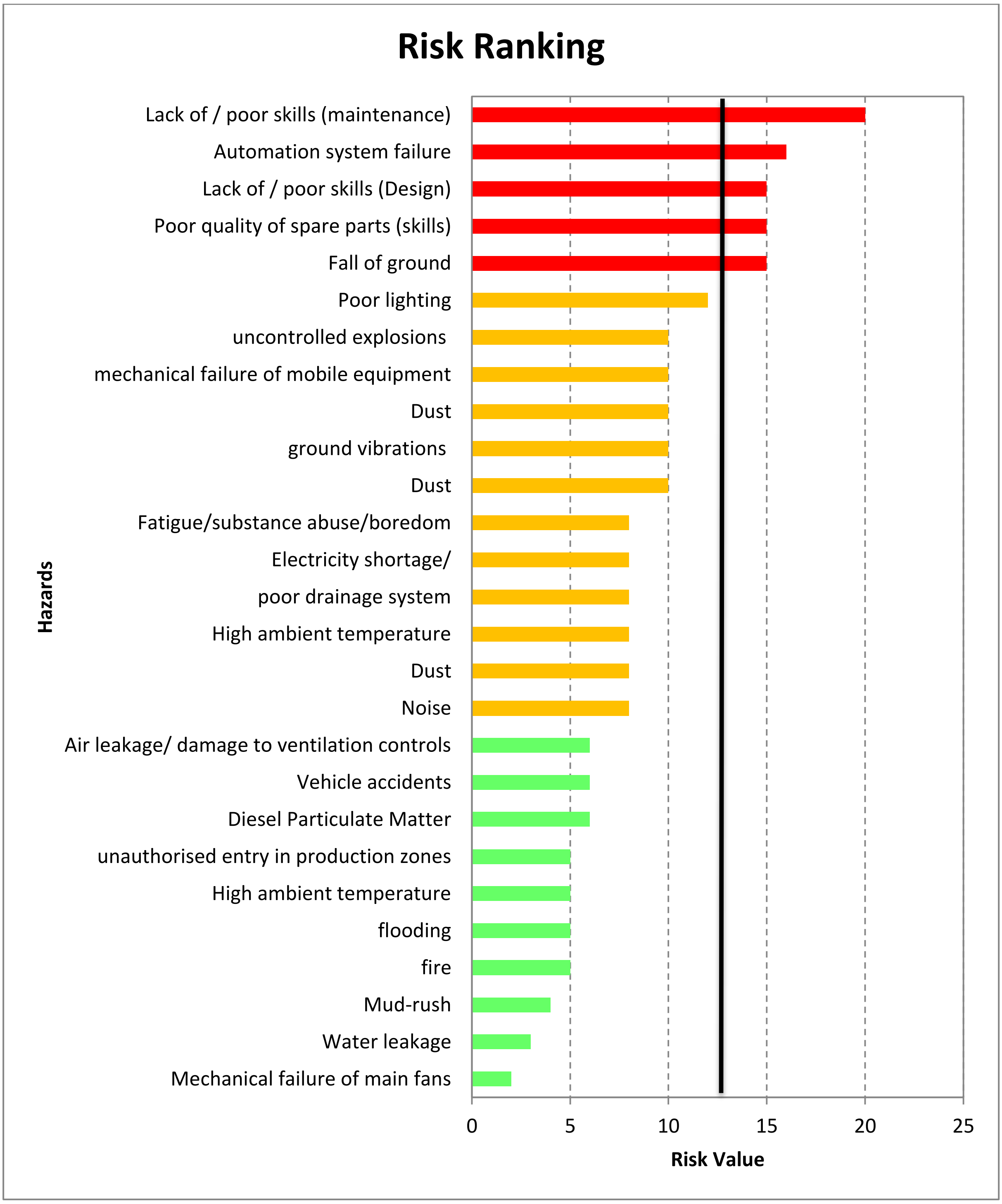


Figure 46: Risk Ranking

Score	Risk Level	Recommended Response
15 – 25	High Threat	Immediate action or detailed planning to be included within implementation plans
8-14	Medium Threat	Measures to be included into action plans and monitored
1-7	Low Threat	Limited action and review will be undertaken

Ernest and Young report (2012) identified the following top ten business risk according to their order:

- Resource nationalisation
- Skills shortage
- Infrastructure access
- Cost inflation
- Capital project execution
- Maintaining a social licence to operate
- Price and currency volatility
- Capital management and access
- Sharing the benefits
- Fraud and corruption (Ernest & Young, 2012)

The results of a high level risk assessment revealed that the major risks are associated with inadequate or lack of skills. This is the second highest on the top ten business risks facing mining industry

A correlation could be drawn among some of the top ten risks. For example, some of the strong labour unions constituencies are unskilled labourers which have the power to withdraw social licence to operate if their members' jobs are at risk due to introduction of automation.

Furthermore, a correlation could be drawn between the social licence to operate and sharing benefits. Should there be a common feeling among members of the community that a mine is contemplating to employ other workers outside their community let alone from outside the country, the social licence to operate could be withdrawn.

Automation is seen as a solution to alleviate a shortage of skills as mines can be operated remotely with fewer workers. However, the risk assessment done contradicts that notion for the South African context, as automation would largely replace the unskilled labour and require more skilled employees.

Mine automation is mainly a process control system which requires highly skilled employees particularly in the fields of technical, procurement and maintenance. Technical skills are crucial in ensuring that the system is reliable as the future of the mine depends on it. The other major challenge is that in most cases poor automation systems are evident when it is too late and high capital cost required for correction. Hence, highly skilled employees are required to ensure effective and efficient systems right from the onset.

The maintenance strategy that has been chosen for this project is reliable maintenance. It is very important to be able to accurately predict the next maintenance for a machine, which means procurement, must not only use the cost as a decision maker but quality as well. This would also require highly skilled employees because once the machine is sent to a production zone there must be quality assurance that it will work for the specified hours before being serviced again.

One of the impetuses for mining industry to strongly drive research on automation is the perceived benefits on health and safety. Indeed automation may reduce the risk in the mining industry as it reduces the exposure of workers to the hazardous environment. One of the main reasons as to why the South African mining industry is unable to achieve the MHSC milestones, is because current mining methods expose workers to hazardous environments.

Automation can be seen as eliminating hazards or substituting control measures which are both in the higher order of hierarchy of controls, as it removes workers from dangerous environments and replace them with machines. It is the opinion of the author that automation is the solution to achieving zero harm in the mining industry. This can be the instrument to assist the South African Mining industry to achieve the MHSC milestones. This does not mean automation comes free of risks in terms of occupational health and safety.

When changing to automation, hazards would remain the same as energies would not change. The only thing that will change is the quantification of risks. When applying the risk matrix to quantify the risk, most of health and safety risks for employees will fall under the acceptable risk because the likelihood will be low. However, it has been mentioned that there would be instances where workers have to go into the production zones when machines cannot be recovered remotely. Though the likelihood is low the consequence would be high which require major hazard control strategies which have the potential to control major risks. The risk matrix, in many instances, is used to indicate that the likelihood and severity are managed in such a way that the risk under discussion falls within the acceptable zone. However, the purpose of the risk matrix is to prioritize risk. The main object of Hazard Identification and Risk Assessment (HIRA) is to obtain effective controls for the identified risk. Hence, the Bow-Tie risk analysis was chosen to address all the hazards that could materialize when a trigger event of compelling workers to enter hazardous production zones is initiated. Bow-tie is an analysis too that is mainly used to address single unwanted events.

The major hazard control framework has four pillars namely: prevention control, monitoring control, first response and emergency response.

Figure 47 on the next page illustrates the Bow-Tie risk analysis for this project. It shows all the threats when workers enter the production zone, first and second preventative controls, first response when controls have failed, emergency response and potential consequences when risks materialize.

The modelled unmanned mine required regular monitoring of preventative controls to test their effectiveness. Emergency evacuation dry runs are to be carried out on regular intervals to ensure that worker are aware of emergency procedures and know where there are refuge bays. This project required that all the workers are to be trained on first aid, firefighting, refresher training on emergency numbers and be aware of emergency signals as these would reduce the effects of the hazard.

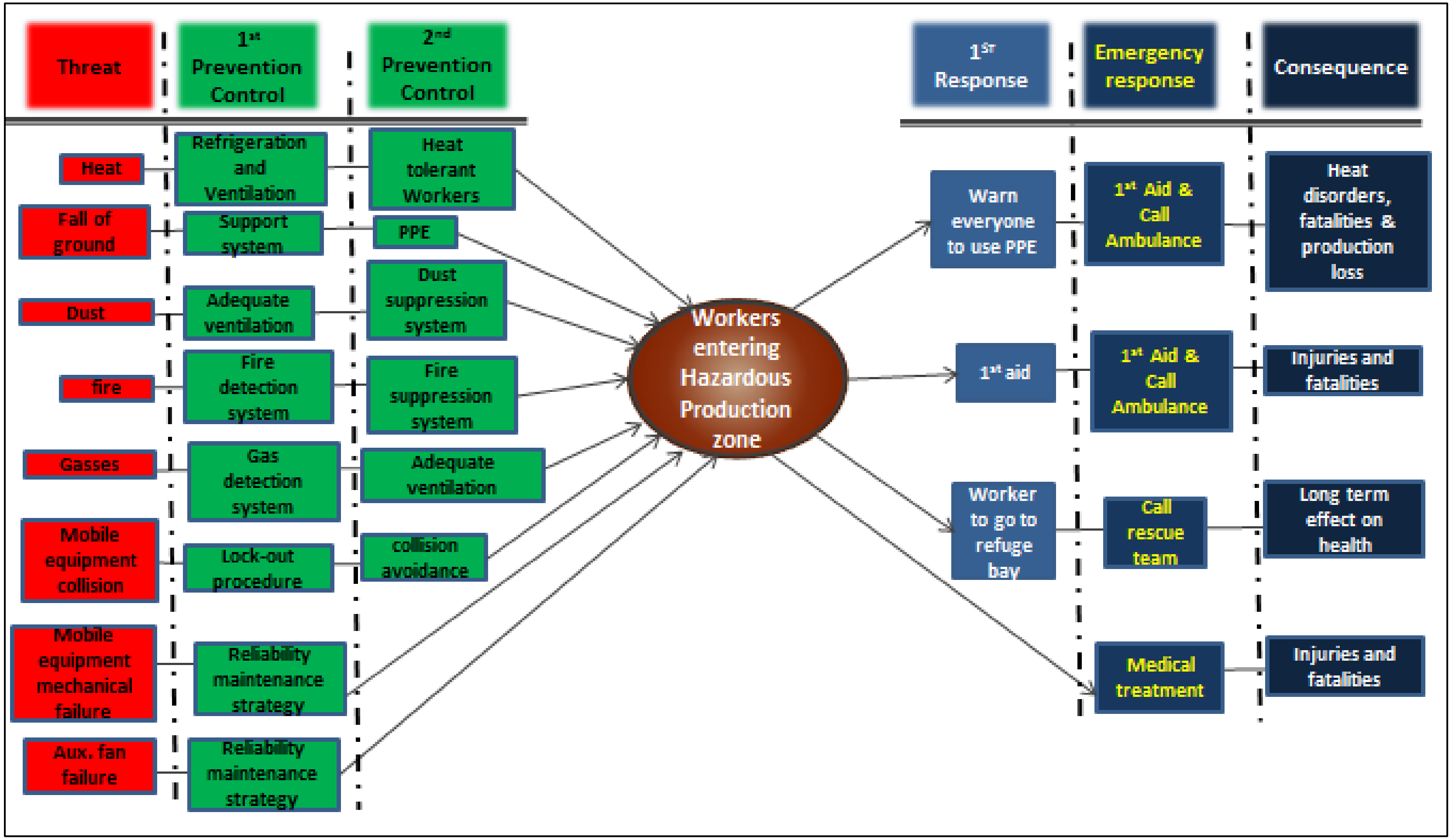


Figure 47: Bow-Tie analysis model

CHAPTER 5: CONCLUSIONS

South Africa has had a history of having a relatively affordable and an abundance of electricity. However, since 2008, the situation has changed as there is a shortage of electricity and the tariffs have increased by more than 200% and expected to increase further in future. According to MYPD in 2017 the electricity would cost 89c/kWh which would affect the electricity bill of a mine by 37% when comparing with the current average price 65c/kWh. The challenge with electricity is not only the cost but the security of supply as well. Electricity challenges could affect the mining industry negatively if not addressed

A literature study was conducted into the field of energy savings specifically focusing on underground gold mines ventilation and cooling. Recent research indicates that ventilation and cooling accounts for 30 to 40% of the total electricity cost of a metalliferous underground mine.

Electricity is required to power fans and refrigeration plants to maintain an acceptable environment for workers underground. The classification of DPM as a carcinogen may indirectly mean an increase of power requirement to supply clean fresh air to dilute and remove it from the workings. This makes it difficult to reduce electricity demand as it is part of essential needs for operations with workers in production zones.

Heat has a detrimental effect on workers' health and productivity. The temperature which is conducive for workers is 27.5^{0}C (wb). Maintaining this rejection temperature comes at an exorbitant price. Hence, most deep metalliferous mines use rejection temperature on the upper scale of the economic range which is between 27.5^{0}C and 29^{0}C (wb).

A literature review also revealed that the current practices and major focus towards reducing electricity costs of ventilation and cooling are on VOD, COD and use of different materials on fan blades. These initiatives are not yet suitable for South

African underground metalliferous mines due to cost, political and environmental conditions.

The purpose of this study was to assess the impact of increasing the rejection temperature to 40^{0}C (db) on electricity demand on an automated mine using VUMA software. The advantage of using VUMA was that the holistically integrated ventilation and cooling systems could be simulated and assessed under a number of 'what if' scenarios.

Though automation has been a highly topical issue in recent years, no one has seriously looked at how it will impact on the electricity demand on ventilation and cooling. This study has presented a different approach in reducing electricity cost in underground ventilation of gold and platinum mines by looking at the introduction of automation and increasing rejection temperature.

A conceptual mine similar to South Deep Mine was modelled at a maximum depth of 2811 metres, consisting of a downcast shaft 9.5m in diameter, and a 7.5m diameter up-cast shaft. The conceptual mine had four levels and produced around 200 kilo-tons per month from two corridors, three mining methods were used namely: destressing, long-hole stoping and 'bench and drift'.

The effect of different rejection temperatures on the electricity cost was evaluated for two scenarios, namely manned and unmanned.

The thermally acceptable environment for workers was set at 27.5^{0}C (wb) in manned and machinery was set at 40^{0}C (db) for the unmanned scenario. The study using the VUMA model has confirmed that the heat transfer between the rock and surface of the airways is less in unmanned scenario by 5% compared to the manned scenario, as the rejection temperature is closer to VRT. Furthermore, auto-compression was 8% less in unmanned versus the manned scenario due to less air required underground. Vehicle heat load was double in the unmanned scenario because the work load and shift utilisation were increased.

To maintain the underground environment below rejection temperatures in each scenario, ventilation was combined with cooling. A number of iterations were carried out to establish the total air quantity, refrigeration capacity rating, shaft sizes, tunnel size and ventilation controls.

The results obtained from the model were air quantity, fan sizes in terms of kilo-Watts and refrigeration cooling capacity for surface and underground. These results were then used to calculate the power requirement and associated cost using the mega-flex cost structure of Eskom.

The results have shown significant cost savings which could justify the acquisition of automation systems when using increased rejection temperatures and automation. The annual electricity cost of ventilation and cooling to produce at a rate of 200 kilotons per month was R97 million in manned compared to R26 million in the unmanned scenario. This translated to a cost saving of R71 million per annum when using automated mining processes which could be used to reduce the payback period of automation system. It is likely that production in the automated mine would increase which would result in overall productivity and profitability improvement of a mine.

However, these savings would not come without a risk. This research has revealed that a lack of or inadequate skills on technical, procurement and maintenance as the major risks. Skills shortage is ranked second on the top ten business risks facing mining. Automation by its nature would require up-skilling of workers which would require time before implementing full automation.

Even though automation is intended to remove workers from hazardous environment, the study has revealed that workers would not be completely removed from production zones as there would be instances when workers are required to enter them. The interaction of machinery, workers and environment would be limited in the unmanned scenario and this would reduce the likelihood of risk materialising. The risk analysis method used in this study was the Bow-Tie methodology and it showed that a situation where workers are required to go into the production zones must be avoided. In order to improve health and safety the study suggested that preventative controls must be effective even if the risk is low. This would ensure that the exposure time to hazardous environments is limited. However, the study also suggested the mitigating control to minimise the impact if the risk is realised.

For automation and increased rejection temperatures to work effectively would require strong communication systems with reliable sensors. The study has also shown that it would be important to track equipment and workers underground to improve productivity and safety.

5.1. FUTURE WORK

While this report successfully demonstrates the cost saving benefits of increasing rejection temperature and is able to predict power requirement, the model is only a first step towards the development of an accurate full dynamic model of an underground automated mine. The first priority in future work is to evaluate the automation sensors and communication systems. Secondly work must be done on tribology to assess wear and tear of equipment in a hot environment. Thirdly a detailed study must be done in terms of skills and training required prior to the implementation of automation.

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