Ventilation and Cooling Comparison between Diesel and Electric Mining Equipment

By

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Abstract

Underground mines are faced with economic and health challenges that urge them to move away from diesel equipment and towards electric equipment. However, the effects of such a transition in the design of the underground mine ventilation are yet to be thoroughly understood. The current study underlines how the full replacement of a diesel fleet with an electric fleet will affect ventilation, cooling demand, and relative costs in a conceptual underground metal mine situated in the Northern hemisphere. It focuses on the significance of heat generating mechanisms which contribute to the overall heat load of the mining operation. The study differentiates the significance of heat from auto-compression, machinery, and strata in full transition scenarios. The study includes the simulation of the heating load emitted from auto-compression, diesel, and electric equipment at a conceptual mine site in the Northern hemisphere by the software VentSIM, and ClimSIM. The assessment of the resulting cooling demand is carried out in two steps. The first step is a heating load simulation of the materials handling system of the mine required to reach the maximum daily production rate that has been carried out for two types of scenarios including sole diesel, and solely electric engines. The second step is a simulation of the associated cooling demands of the ventilation network for these two scenarios. The results of the study show that shifting from a diesel fleet to an electric equipment can have moderate to significant impacts, depending on the intensity of equipment utilization, surface conditions, depth and extent of the operations, and geothermal gradient.

Keywords: Underground mining, ventilation, electric equipment, heat load

Résumé

Les mines souterraines sont de plus en plus confrontées à des situations économiques difficiles, et à des restrictions strictes en matière de santé et de sécurité qui les poussent à restreindre l'utilisation des machines diésels aux bénéfices des machines électriques. En revanche, les effets d'une telle transition sur la conception de la ventilation minières doivent encore être étudiés. Cette thèse souligne comment le remplacement complet de la machinerie minière diesel par un une flotte électrique affectera la ventilation, la demande en refroidissement et les coûts relatifs dans une mine de métaux souterraine typique de l'hémisphère Nord de l'Amérique. Cette thèse met l'accent sur l'importance des mécanismes de production de chaleur qui contribuent à la charge thermique globale de l'exploitation minière. L'étude différencie l'importance de la chaleur de l'auto-compression, des machines et des strates géologiques dans les scénarios de transition complète. L'étude comprend la simulation de la charge thermique émise par l'auto-compression et émise par des équipements diesel et électrique sur un site minier sélectionné dans l'hémisphère nord de l'Amériques par les logiciels VentSIM et ClimSIM.

L'évaluation des demandes de refroidissement qui en résultent est réalisée en deux étapes. La première étape est la simulation de la charge thermique émise afin de produire le taux de production journalier maximal qui a été réalisé pour deux types de scénarios incluant la machinerie avec moteurs diesel uniquement, et la machinerie avec moteurs électriques uniquement. La deuxième étape consiste à simuler les demandes de refroidissement associées au réseau de ventilation pour ces deux scénarios.

Les résultats de l'étude montrent que le passage de la flotte diesel à la flotte électrique peut avoir des impacts modérés à significatifs, selon l'intensité de l'utilisation de l'équipement, les conditions de surface, la profondeur, l'étendue des opérations et le gradient géothermique.

Mots clés : Mine souterraines, ventilation, équipement électrique, charge thermique

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Preface and Contribution of Authors

The author of the thesis has written fully and is the sole contributor to this thesis. The numerical analysis has been developed and performed entirely by the author.

Chapter 1
Background

1.1 Purpose and Importance of Mine Ventilation

As astronauts need fresh air and artificial atmosphere in their spacecraft, miners need air to breathe in an underground mine (Howard L. Hartman; Jan M. Mutmansky; Raja V. Ramani; Y.J. Wang 1997). Ventilation of air is responsible for the circulation, quality, and the direction of the air flow (Anon 1993) (McPherson 1993). Having a good quality and sufficient quantity of air in underground mines increases the productivity by allowing comfortable and safe working conditions. The underground air ventilation system is the key to providing a safe working environment for personnel required to work or travel, by diluting air transported particulates created by mechanical equipment, blasting fumes, as well heat created by different sources of heat generation (produced by either the machines or the mine itself) in underground mines (McPherson 1993). For a long time, mining equipment was mostly operated by diesel engines. This popularity of diesel engines is mostly due to their reliability and flexibility, especially in underground environments.

On the other hand, operating these machines in underground mines has been known to have adverse health effects on humans since the 1960's (Jacob 2013). Diesel Particulate Matter which is known as DPM, is emitted from diesel engines the underground mines as a result of fuel impurities and incomplete combustion (de la Vergne 2003) (World Health Organization 2012). These fumes are considered as a Group 1 carcinogen by the World Health Organization. Studies have shown that the health risk associated with DPM are such that miner's life are endangered with the highest DPM concentration emitted from diesel engines (Bugarski, et al. 2004).

Governments of different countries around the world mandate underground mines to provide the specific minimum airflow to meet quality and temperature-humidity standards. These regulations account for a minimum threshold of carbon emission at a Total Weight Average (TWA) for a specific amount of time (8 to 12 hours in a shift). As well, the permissible exposure limit (PLE or OSHA PEL) or short-term exposure limit (STEL) is usually 16 minutes, as long as the time-weighted average is not exceeded (OSHA 2013).

Regulations should follow along best practices for different mines since each mine requires different approaches and consideration based on the mining method and mine's location (Prosser 2016).

1.2 Motivation

Although diesel engines are the most favorable equipment to extract ore from underground mines, electric engines are now becoming more and more competitive based on their sustainability and equivalent production rate. Mining companies are now turning towards machinery electrification. Some advantages are (Chadwick 1992) (McPherson 1993) (Moore 2010) (McCarthy 2011) (Mark 2012) (Paraszczak, Laflamme and Fytas 2013) (Jacob 2013) (Allen and Stachulak 2016):

- Lower heat production: Electric engines produce one third the heat of equivalent diesel engines
- No fumes
- Lower operating ventilation costs since they require less airflow specifications
- Lower power costs of a reduction on capital costs from smaller raises and fans
- Lower maintenance costs

In Canada, big companies such as Goldcorp and Glencore PLC., are considering the electrification of their mining fleet. As Goldcorp's vice-president predicts that the rate of adoption of electrification of diesel fleet is accelerating and expects many underground mining companies will use Battery Electric Vehicles (BEVs) over a diesel fleet by 5 years (Braul 2018). By shifting diesel engines to an all-electric underground fleet, a savings of $9 million dollars per year on diesel fuel, ventilation costs, and carbon footprint costs can be estimated (Braul 2018). Intangible factors of electrification such as the health of the workforce regarding air quality and level of noise are priceless. Thus, the future of BEVs is promising and is expected that most of the mining companies around the world will convert their diesel power to an electric fleet (Moore 2010).

1.3 Issues

There are some issues standing in front of the electrification of diesel power machines. One of the biggest issues is that there are no specific regulations regarding the ventilation and cooling demands of underground mines run by electrical equipment (A Halim, M Kerai 2013). For mines that already use electric engines rather than diesel engines, there is but only one regulation existing created by the Australian government, where a minimum air velocity of $0.25\ m/s$ is required in active faces (Western Australian (WA) Government 1995). Although, the quantity of airflow is not the same for different dimension of working areas (A Halim, M Kerai 2013), thus making this regulation not applicable in all situations. Moreover, a more comprehensive design is required to assess the impact of the mandated velocity for electric engines by the Australian government on heat dilution as well as dust production levels (Allen and Stachulak 2016).

The second issue is that the BEVs need to be recharged often. Recently, Artisan Vehicle System launched the world's first 40 tonne battery electric truck to be used by Kirkland Lake mine (Braul 2018). This type of trucks uses batteries, which need to be exchanged with a fully charged battery at a mechanical shop (Swap Station) in the mine every couple of hours. This method of operation might not be optimal and repercussion on the cycle time of trucks might be affected.

However, the mining equipment company Sandvik has committed to produce a 40-tonne truck to be used by the Borden mine by 2020 which has the ability of on-board charging (Braul 2018). Although it saves loading and unloading time and increases the productivity rate, it needs an infrastructure in the mine to plug into which increase costs.

Choosing the right combination of batteries and motors is mentioned as another issue concerning BEVs, which needs to be properly designed (Braul 2018).

1.4 Objective

The objective of this thesis is to compare the ventilation design and cooling demand for conventional diesel engines and electric engines. A case study is presented on a conceptual mine in the northern hemisphere.

This analysis seeks to quantify a realistic baseline (costs, savings, and environmental impacts) as well as energy savings. The analysis was restricted to the following parameters:

- The analysis was based on the maximum production rate during the life of the mine as a worst-case scenario for the ventilation design
- The materials handling data extracted from the mine is to meet the maximum production rate allowable at the mine

- A critical loop design was constructed in Auto-CAD and imported to VentSIM. A ClimSIM model was built for thermal simulation analysis.
- Surface climate condition used for cooling analysis is considered as summer condition (Worst case scenario).
- All design annotations on drawings and calculation outputs are to be in the SI units

1.5 Contribution to Mining Engineering Knowledge

In this thesis, the original contribution is designing the ventilation and cooling requirement for two different scenarios diesel, and electric machineries in underground mines compared to costs and carbon footprint.

1.6 Thesis Outline

The thesis outline is as follows:

1.6.1 Chapter 2

Chapter 2 presents a literature review on electric and diesel engines in underground mines.

1.6.2 Chapter 3

Chapter 3 presents the major theories of mine ventilation relating to cooling and heating in underground mines.

1.6.3 Chapter 4

In the chapter 4, a case study on ventilation modeling for both diesel and electric scenario will be done.

1.6.4 Chapter 5

Chapter 5 presents the main results for both scenarios presented in chapter 4. The cooling analysis and heat calculation for the two scenarios is presented and explained.

1.6.5 Chapter 6

Chapter 6 is the discussion, conclusion and future work.

Chapter 2
Literature Review

2.1 Health Issues in Underground Mines

Nowadays, one of the most important concerns in all industries is the health and safety of workers. Due to the confined environment of underground mines, the workers are exposed to a greater level of diesel exhaust fumes than other work groups (Fernandez 2015). The underground mine workers are exposed to diesel fumes 3 to 10 times more than surface mine workers (Scheepers, et al. 2003).

Thirty years of research have established diesel fumes as a carcinogenic gas (Moench 2011). Diesel exhaust increases lung cancer among the miners who are exposed to diesel emissions (Attfield, et al. 2011). At 2005, a study estimated that 21,000 people in the United States, who were exposed to diesel particle matters, have a shorter life span (Schneider and Hill 2005).

Also, it has been reported that people who are exposed to diesel fumes, even during a short period of time (less than 1 hour), experience negative effects on the immune system, particularly for people who have allergies or asthma (Hedges, Djukic and Irving 2007). NIOSH (National Institute of Occupational Safety and Health) estimated that 1.4 million workers in the United States were exposed to diesel exhaust between the years of 1981 to 1983 (NIOSH 1983). Pronk et al. estimated that 3 million workers were exposed to diesel exhaust fumes between 1990 and 1993 which is more than over different time periods (Pronk, Clobe and Stewart 2009). This means that the use of diesel engines will continue to increase due to their flexibility and performance in industries (Environmental Protection Agency (EPA) 2002) (Friesen, et al. 2013).

It is reported that 10% of all deaths in the industrialised world are due to lung cancer (K. Hedges, Diesel emission in underground mining 2013). Diesel engines do not only cause negative effects on human lungs through the fumes. Indeed, another side effect related to diesel engines is high decibel (dB) noise that can cause hearing loss (Hartman and Novak 1987). It is estimated that electric engines produce 85 dB while diesel engines produce 105 dB (Moore 2010). Referring to the OSHA standard, protection against noise must be provided when the sound level exceeds 90 dB (OSHA 2013). However, NIOSH considers 85dB as the occupational noise exposure limit, and above that are considered hazardous for employees who work 8 hours TWA (Time Weighted average) in this situation (Franks, Stephenson and Merry 1996).

2.2 Ventilation in Underground Mines

A critical part of any underground mine is a safe and economical ventilation system (Hartman, et al. 2012). Ventilation is described as an organic system in underground mines. One could see the intake airways as being arteries that carry oxygen to the active areas where workers and equipment are working. The return veins would be considered as the return airways, which exhaust the air contaminated with dust, diesel equipment emissions, and polluted air to the outside atmosphere. (McPherson 1993).

Without an effective ventilation system, no underground facility can operate safely (McPherson 1993). Some of the features of underground mine ventilation are (Tuck 2011):

- Providing fresh air for the workers
- Providing oxygen for combustion
- Improving visibility
- Disperse diesel particulate matter (DPM)
- Disperse methane gas from coal mines to avoid explosions
- Disperse dust, heat, and humidity
- Dilute blasting fumes
- Ventilation and cooling of underground mines are two aspects that require high capital cost and operating cost (CIPEC 2005). Ventilation costs are estimated to be over 30% of electrical operation costs (de la Vergne 2003) and was even estimated to be over 40% (Paraszczak, Laflamme and Fytas 2013). A ventilation engineer is supposed to design a primary mine ventilation system at the lowest cost to determine the required airflow at work areas in underground mine (Acuña et Lowndes 2014).

2.3 Regulations

Regarding to health and safety issues in underground mines, mine ventilation practices in mining countries such as Canada, United States of America, South Africa, and Australia are heavily regulated (Tien 1999). Thus, governments around the world mandate regulations on the specific quantity of airflow required to dilute heat, fumes, and airborne dust (Fernandez 2015). The ventilation rates, which are mandated by the government, will differ in practice due to the type of diesel engines, mine design and other factors in underground mines (K. Hedges, et al. 2007). The regulations are different in each country. In Canada, the flow rate ranges from $0.045\ m^3/s$ per kW of diesel engine power to $0.092\ m^3/s$ per kW of diesel engine power, with the average common flow rate at $0.063\ m^3/s$ per kW (100 cfm per brake horsepower) (Stinnette 2013).

In the USA, the regulations mandate mines to provide a specific volume of air in order to remain below a specific maximum exposure level of airborne contaminants (Stinnette 2013). However, in some regions of Canada, mines are allowed to recirculate a specific amount of air as long as the maximum level of contaminants is not exceeded (Government of Ontario 2014)

Table 1 states the minimum air volume, mandated by governments, required for the mines running diesel equipment (Tuck 2011) (Monitoba Government 2014) (Quebec Government Updated to 1 March 2018) (Ministry of Energy and Mines Revised Jun 2017) (Saskatchewan Government 2016) (Nova Scotia Government 2015) (Alberta Government 2018) (Western Australian (WA) Government 1995):

Location	Diesel Airflow Requirement	Comments
Australia	$0.06\ m^3/s$ per kW	
Western Australia	$0.05\ m^3/s$ per kW	
Queensland, Australia	None	Was $0.04\ m^3/s$ per kW
Ontario, Canada	$0.06\ m^3/s$ per kW	
Manitoba, Canada	Min of $0.092\ m^3/s$ per kW for non-approved engine Min of $0.045\ m^3/s$ per kW for multi engines	Ventilation as per CANMET approval or MSHA approval. Uses 100/75/50 rule
Quebec, Canada	Min of $0.092\ m^3/s$ per kW for non-approved engine Min of $0.045\ m^3/s$ per kW for multi engines	Ventilation as per CANMET approval or MSHA approval. Uses 100/75/50 rule

British Columbia, Canada	0.06 m^3/s per kW	Ventilation as per CSA Standard
Saskatchewan, Canada	0.063 m^3/s per kW	Ventilation as per CANMET approval
Alberta, Canada	1.9 m^3/s at active headings, and air velocity of 0.3 m/s	Ventilation as per CSA Standard for coal mines
New Brunswick, Canada	0.067 m^3/s per kW	Engine approval is required for engines above 75 kW.
Nova Scotia, Canada	Min air velocity of 0.33 m/s for coal mines	Engine approval is required.
Newfoundland & Labrador, Canada	0.047 m^3/s per kW	Engine approval is required.
Northwest & Nunavut, Canada	0.06 m^3/s per kW	Engine approval is required. Ventilation as per CANMET or MSHA engine approval
Yukon, Canada	0.06 m^3/s per kW	
United Kingdom	None	
United States	$0.032\ to\ 0.094\ m^3/s$ per kW	
Chile	0.067 m^3/s per kW	
South Africa	0.067 m^3/s per kW	Based on best practice
Indonesia	0.063 m^3/s per kW	
China	0.067 m^3/s per kW	

Table 1) Diesel Engine Ventilation Requirement

2.4 Mine Ventilation - Diesel Engines in Underground Mines

2.4.1 History of Diesel Engines

Rudolf Diesel introduced a new heat engine in 1892 (Mollenhauer and Schreiner 2010). Diesel engines are internal combustion engines that convert chemical energy from diesel fuel to mechanical energy (US Department of Energy 2003). Diesel engines have been playing a vital role in different industries, particularly for heavy-duty functions such as transport, construction, agriculture, and industrial machineries for 50 years (Fernandez 2015). This popularity stems from the high fuel efficiency, capacity and durability (Knothe, Krahl and Gerpen 2010).

The diesel engine was introduced to underground mines in Germany during the year 1927. 12 years later, in 1939, diesel engine machines were being utilized in an underground metal mine in Pennsylvania, USA. However, diesel engine machines had not been adapted to the underground situation until 1960's (Kenzy and Ramani 1980).

Table 2 shows a timeline of diesel equipment utilized in underground mines (Kenzy and Ramani 1980).

Year	Description
1882	Diesel Engine Invented
1886	First Gasoline Locomotive in an Underground Mine (Germany)
1897	Diesel Engine Reduced to a Practical Size
1906	First Gasoline Engine in a U.S. Mine
1915	Most States in the U.S. Outlaw Gasoline Engines Underground
1927	First Diesel Engine in an Underground Mine (Germany)
1934	Diesels in Belgian, British, and French Underground Coal Mines
1939	First Diesel Engine in a U.S. Underground Mine (Pennsylvania)
1946	First Diesel Engine in a U.S. Underground Coal mine
1950	Development of the first Diesel-Powered LHD

Table 2) History of Diesel Engines in Underground Mines (Kenzy and Ramani 1980)

Diesel engines are more popular in underground mines than gasoline engines. This popularity comes from the higher efficiency and energy density of diesel fuel compared to gasoline (Varaschin 2016). The heat production of diesel engines is approximately $36\ MJ/L$ rather than $32\ MJ/L$ heating values for gasoline engines. This makes 12% more energy dense than gasoline by volume (Varaschin 2016). Thus, diesel engines provide higher torque at greater efficiency with less carbon monoxide emissions than gasoline engines (Stinnette 2013). Peak operation of diesel engine efficiency is estimated to be 32%, however, they have 20% to 25% efficiency in reality (Smill 2010).

2.4.2 Classification of Diesel Engines Based on Governmental Regulations (U.S EPA: Tier 1 to 4)

In the USA and North America, the Mine Safety and Health Administration (MSHA) supervise and specify the acceptable level of diesel emissions in underground mines. However, diesel emission standards are also set by the US EPA (United States Environmental Protection Agency) (Varaschin 2016).

The US EPA regulation categorizes the non-road diesel engine emissions from Tier I through Tier IV standards (Stinnette 2013). The merged U.S-European standards introduced by the EPA and the US government have resulted in reduced emission levels in underground situations where diesel engines run (DieselNet 2016).

Tier I standard regulation was introduced in 1994 and implemented until 2000. In 1998, Tier I diesel emission standards was adapted to Tier 2 and 3 standards and implemented before 2008 (DieselNet 2012)

In 2004 a new regulation had been placed on the table called Tier 4, for the engines typically found in construction, mining, farming and forestry. (DieselNet 2012) The main focus of this new regulation is to mandate the manufacturing diesel companies to design an engine with the lowest emissions of particulate matter, lower than the reduction planned in Tier 3 standards (Varaschin 2016) (Stinnette 2013) (DieselNet 2012). A summary of current governmental law for non-road diesel engines can be as shown in Table 3 (Stinnette 2013).

	2011	2012	2013	2014	2015
North America & Western Europe					
19-37 kW (26-49 hp)	Tier 4 Interim/ Stage IIIA		Tier 4 Final/ Stage IIIA		
37-56 kW (50-75 hp)	Tier 4 Interim/ Stage IIIA		Tier 4 Final/ Stage IIIB		
56-130 kW (76-174 hp)	Tier 3/ Stage IIIA	Tier 4 Interim/ Stage IIIB			Tier 4 Final/ Stage IV
130-560 kW (175-750 hp)	Tier 4 Interim/ Stage IIIB			Tier 4 Final/ Stage IV	

+ 560 kW (+751 hp) North America Only	Tier 4 Interim		Tier 4 Final
Japan	Tier 3/ Stage IIIA	Tier 4 Interim/ Stage IIIB	Tier 4 Final/ Stage IV
Mexico	Unregulated / Tier 1/ Stage 1		
China	SEPA Stage II Similar to Tier 2/ Stage II		
India (Large Cities)	Bharat (CEV) Stage III/ Tier 2 – Tier 3/ Stage II – Stage IIIA		
Latin America	Unregulated/ Tier 1/ Stage I (These are proposals for Tier 3/ Stage IIA in Brazil and Chile)		
Middle East	Unregulated/ Tier 1/ Stage I		
Africa	Unregulated/ Tier 1/ Stage I		
Russia	GOST R41 96-99 Similar to Tier 1/ Stage I		
Australia	Tier 1/ Stage I		

Table 3) World Diesel Emissions Regulations (Stinnette 2013)

Table 4 shows the stringent emission standards for each Tier mandated by EPA that the manufacturer should use according to design their diesel designs.

Engine Power	Tier	Year	CO	NMHC+NOx	PM
19-37 kW (26-49 hp)	Tier 1	1999	5.5 (4.1)	9.5 (7.1)	0.8 (0.6)
	Tier 2	2004	5.5 (4.1)	7.5 (5.6)	0.6 (0.45)
	Tier 3	2006	4.5 (3.4)		0.36 (0.27)
	Tier 4	2013	5.5 (4.1)	4.7 (3.5)	0.03 (0.022)
37-75 kW (50-100 hp)	Tier 1	1998	-	-	-
	Tier 2	2004	5.0 (3.7)	7.5 (5.6)	0.4 (0.3)
	Tier 3	2008		4.7 (3.5)	
	Tier 4	2013			0.03 (0.022)
75-130 kW (100-175 hp)	Tier 1	1997	-	-	-
	Tier 2	2003	5.0 (3.7)	6.6 (4.9)	0.3 (0.22)
	Tier 3	2007	5.0 (3.7)	4.0 (3.0)	

	Tier 4	2014	5.0 (3.7)	-	0.02 (0.015)
130-560 kW (175-750 hp)	Tier 1	1996	11.4 (8.5)	-	0.54 (0.4)
	Tier 2	2003	3.5 (2.6)	6.6 (4.9)	0.2 (0.15)
	Tier 3	2006		4.0 (3.0)	
	Tier 4	2014		-	0.02 (0.015)
>560 kW (>750 hp) (Except generator sets)	Tier 1	1996	11.4 (8.5)	-	0.54 (0.4)
	Tier 2	2001	3.5 (2.6)	6.4 (4.8)	0.2 (0.15)
	Tier 3	2006		3.8 (2.8)	0.12 (0.09)
	Tier 4	2015		0.4 (0.3)	0.1 (0.075)

Table 4) Non-Road Diesel Engine Emission Standards for Various Tiers, g/kWh (g/bhp.hr) (DieselNet 2017)

2.4.3 Advantages of Diesel Engines in Underground Mines

There are some advantages to running diesel equipment in underground mines compared to electric engines: (Chadwick 1992) (McPherson 1993) (McCarthy 2011) (Paterson and Knights 2012) (Jacob 2013) (Paraszczak, Laflamme and Fytas 2013) (Paraszczak, Svedlund, et al. 2014) (Varaschin 2016):

- Lower capital costs
- More flexibility
- More reliable compared to tethered machines (Trailing cables are not reliable and are more expensive than diesel fuel)
- Faster tramming speed compared to electric equipment
- Less infrastructure required (The electric equipment needs specific infrastructure such as a trolley line, electrical grid, swap station, charging station etc.)

2.4.4 Disadvantages and Issues of Diesel Engines in Underground Mines

2.4.4.1 Gaseous Emissions and Diesel Particulate Matter (DPM)

Diesel exhaust is considered as a complex mixture of gases and particles emitted by diesel-fuelled internal combustion engines (IARC 1989) (Lipsett and Campleman 1999) (Hesterberg, et al. 2005) (Environmental Protection Agency (EPA) 2002). This complexity of diesel exhaust fumes makes them difficult and costly to control and monitor regarding health concerns in underground environments (Fernandez 2015).

Diesel-powered mining equipment produces three major toxic gaseous emissions consisting of carbon monoxide (CO), carbon dioxide (CO_2), and oxides of nitrogen (NO_x) (Stinnette 2013). Table 5 shows features and the negative impacts of diesel emissions on humans (Stinnette 2013).

Gaseous	Features	Negative Impacts		Flammability (Concentration rate %)
		Symptoms of poisoning	Death in higher concentration	
Carbon monoxide (CO)	Odourless-Colourless	Headache, dizziness, fatigue, nausea	Above 1500 pm, death occurs in one hour	Yes. Explosive in range of 12.5% to 74%
Carbon dioxide (CO_2)	Odourless-Colourless	Asphyxiation and toxic effects such as dizziness, nausea and a loss of consciousness at concentration of 3% or above	Rapid death occurs at 20% or above concentration in the air	No
Oxides of Nitrogen (NO_x)	Slight scent and Reddish-brown in higher concentration	pulmonary edema in concentration as little as 1 ppm	90 ppm concentration causes death in 30 minutes	No

Table 5) Gaseous emissions of diesel engines and their impact on humans

Gaseous exposure limits, which are legislated by governments around the world, are shown in Table 6 (Monitoba Government 2014) (Quebec Government Updated to 1 March 2018) (Ministry of Energy and Mines Revised Jun 2017) (Saskatchewan Government 2016) (Nova Scotia Government 2015) (Alberta Government 2018) (Western Australian (WA) Government 1995).

Location	CO (ppm)	CO_2 (ppm)	NO (ppm)	NO_2 (ppm)	SO_2 (ppm)
Ontario	25	5000	25	3	2
Manitoba	20	5000	25	3	---
Quebec	35	5000	25	3	2
British Columbia	25	5000	25	3	2
Saskatchewan	25	5000	25	3	2
Alberta	25	5000	25	3	2
New Brunswick	25	5000	25	3	2
Nova Scotia	25	5000	25	3	---
Newfoundland & Labrador	25	5000	25	3	2
Northwest & Nunavut	25	5000	25	3	2
Yukon	50	5000	25	5	5
United States	50	5000	25	5	---
South Africa	35	9000	30	5	---
Australia	30	5000	25	3	2
China	17	5000	12	3	2
Switzerland	30	5000	30	3	0.5

Table 6) Gaseous Exposure Limits

As diesel engines are the most common equipment ran in underground mines, diesel particulate matter (DPM) exists in underground workings. DPM contains fine particles with a diameter of less than 2.5 μm and ultra fine particles with a diameter of less than 0.1 μm (Olsen, et al. 2014). Due to their small size, DPM aerosols stay much longer in a mine atmosphere than large particles, like dust. DPMs are lighter than air and remain suspended for a long period of time, whereas dusts settle down easily by gravity after a short period of time (Fernandez 2015). Thus, DPMs settle down in the human respiratory system and penetrate deeply into the human lung where gas exchange occurs and can potentially endanger human health (Pietikäinen, et al. 2009) (Morawska, et al. 2005).

Although the diesel engines continue to improve in quality, they are still not exempt from regulation. DPM, atmospheric Total Carbon (TC), atmospheric Elemental Carbon (EC) and atmospheric Nitrogen Dioxide (NO_2) are considered as harmful emissions produced from diesel combustion. The engine manufacturers are mandated to produce engines which comply with Tier 4 emission standards, which regulate harmful emissions from diesel engines (Varaschin 2016).

DPM exposure limits mandated by governments are given in Table 7 (Monitoba Government 2014) (Quebec Government Updated to 1 March 2018) (Ministry of Energy and Mines Revised Jun 2017) (Saskatchewan Government 2016) (Nova Scotia Government 2015) (Alberta Government 2018) (Western Australian (WA) Government 1995).

Location	DPM Exposure Limits (mg/m^3)
Ontario, Canada	0.4
Manitoba, Canada	Uses ACGIH Standards
Quebec, Canada	0.6
British Columbia, Canada	1.5
Saskatchewan, Canada	---
Alberta, Canada	Uses ACGIH Standards
New Brunswick, Canada	1.5
Nova Scotia, Canada	1.5
Newfoundland & Labrador, Canada	Uses ACGIH Standards
Northwest & Nunavut, Canada	1.5
Yukon, Canada	1.5
United States	0.16
South Africa	---
Australia	0.1

Table 7) DPM Exposure Limits in Canada, and Other Places around the World

2.4.4.2 Heat

The heat emission produced by diesel equipment contributes to three different sources. One third of the heat comes from the radiator of machine, the second part is emitted through exhaust gases and the last part is produced when the machine uses shaft power to work against gravity by frictional process (McPherson 1993). A summary of Canadian governmental regulations on heat stress for underground mines where diesel engines operate is provided in Table 8. The list shows a range of acceptable temperature for specific circumstances. In other cases, the American Conference of Governmental Industrial Hygienists (ACGIH) is used as a Threshold Limit Value (TLV) for heat stress or cold stress which is given in Table 9.

Provinces of Canada	Regulation	Temperature
British Columbia	Heat stress regulations Indoor Air Quality Regulation, ASHRE 55-1992 Standard	Limit in WBGT units similar to ACGIH TLVs
	Summer indoor Winter Indoor	23.3-27.2 °C (74-81 °F) 20.5-24.4 °C (69-76 °F)
Alberta	(Guidelines only)	Similar to ACGIH TLVs
Saskatchewan	Thermal environment	Reasonable and appropriate to nature of work
Manitoba	Thermal environment	Similar to ACGIH TLVs
Quebec	Safety in mines: Dryhouse temperature Occupational exposure limits	22 °C minimum Similar to ACGIH TLVs
Nova Scotia	Construction safety regulations: Working chamber	27 °C (80 °F) Maximum ACGIH TLVs for heat stress and cold stress
Prince Edward Island	Occupational exposure limits	ACGIH TLVs for hot and cold environment
Newfoundland & Labrador	Occupational exposure limits	ACGIH TLVs for hot and cold environment

Table 8) Canadian Regulations on Thermal Conditions in the Workplace (CCOHS 2017)

Allocation of work in work / Rest Cycle	Action Limit – Acclimated ($°C\ WBGT$)				Action Limit – Unacclimated ($°C\ WBGT$)			
	Light	Moderate	**Heavy**	**Very Heavy**	Light	Moderate	**Heavy**	**Very Heavy**
75-100%	31.0	28.0	--	--	28.0	25.0	--	--
50-75%	31.0	29.0	27.5	--	28.5	26.0	24.0	--
25-50%	32.0	30.0	29.0	28.0	29.5	27.0	25.5	24.5
0-25%	32.5	31.0	30.5	30.0	30.0	29.0	28.0	27.0

Table 9) ACGIH Screening Criteria for Heat Stress Exposure (Workplace Safety North 2014)

Table 9 assumes eight-hour workdays in a 5-day workweek with conventional breaks. In this table, the terms, light, moderate, heavy, and very heavy work are defined as (Workplace Safety North 2014):

- Light work considers as light hand or arm working, driving and walking.
- Moderate work defines as pushing, pulling or lifting a moderate weight.
- Heavy work means digging, carrying, pushing or pulling heavy loads.
- Very heavy work classifies as a very intense activity at maximum pace.

2.5 Mine Ventilation - Electric Engines in Underground Mines

2.5.1 History of Electric Engines

As early as the 1970's, electric Load-Haul-Dumps (eLHDs) were introduced in underground hard rock mines due to high productivity, low total cost and a lighter environmental impact compared to traditional diesel engines (Emilsson and Sandvik 2015). However, the use of electric equipment in underground mines goes back to the 1990's (Chadwick 1992). By 2010, electric mining machineries were not commonly used in North American hard-rock mines (Moore 2010). In the 1990's, Inco and Kidd Creek mine tested and operated the Kiruna electric haul truck (Chadwick 1992). As of 2010 Coleman-McCreedy, Creighton and Stillwater Mines in North America ran Kiruna electric haul trucks (Moore 2010).

2.5.2 Advantageous of Electric Engines in Underground Mines

The advantages of electric engines in underground mines have been discussed by some studies; the following can summarize them compared to running traditional diesel engines in underground mines (McPherson 1993) (Varaschin 2016) (Braul 2018):

- Lower operating costs due to inexpensive electrical power cost rather than diesel fuel cost
- Lower noise production (Electrical vehicles make 85 dB compared to 105 dB for diesel.)
- No carcinogen emissions
- Lower heat emission
- Reduced ventilation and cooling operating costs
- Reduced capital costs on ventilation, cooling, and mine design
- Reduced fog and better visibility due to less exhaust particles

2.5.3 Classification of Electric Engine Machines Run in Underground mines

Nowadays, there are three different types of electric engine machines running in underground hard rock mines include:

- Trolley-Electric Haul Trucks
- Tethered Electric LHDs
- BEV

Among these types of electric machines, only trolley-electric haul trucks and tethered electric LHDs are tried, tested, and commercially available in bulk underground hard rock mines (Varaschin 2016). However, due to the flexibility of BEVs in confined environments, it has been expected that many underground mines around the world will choose battery-powered vehicle over the diesel fleet over the next 10 years (Braul 2018).

2.5.3.1 Trolley-Electric Haul Trucks

These types of machines are the most common electric trucks used in underground mines.

Figure 2) Trolley Electric Truck (Queen's University, Electric equipment 2017)

Figure 1) Kiruna Trolley Electric Truck (womp 2013)

Electric haul trucks are used in order to gain the following advantages compared to diesel machineries (Willick 2010) (Chadwick 1992):

- High availability (Over 85%)
- Reduced ventilation costs
- No exhaust gases
- Reduce carbon footprint
- Long asset life of 60,000 hours
- Lower cost per tonne
- High production rate by providing faster speed going up the ramps (Shorter cycle times)

The major disadvantages for these electric machines are high capital costs and maneuverability limitation movement along the trolley lines. For a trolley system, an overhead cable should be installed with the infrastructure cost per truck, which is estimated 75% of the total truck price (Paraszczak, Svedlund, et al. 2014).

Other issues can be summarized as follows (Paraszczak, Svedlund, et al. 2014) (Queen's University, Electric equipment 2017):

- Not ideal for shallower mines due to added high capital costs
- Selected only if the diesel trucks are not suitable
- Impact on the flow of traffic on the ramp because of high speed ability
- Space Occupied from the tunnel's ceiling for the trolley system

2.5.3.2 Tethered Electric LHDs

LHDs take care of mucking ore materials from stopes and hauling them to dumping points or loading them onto a haul truck (Jacob 2013).

Overhead power LHDs or Trolley eLHDs are impractical in underground mines because of the maneuverability limitation created by the trolley line. Another type eLHD, which is more common in underground mines are Tethered LHDs. A tethered LHD provides more maneuverability due to the tether trailing cable. This type of LHD is suitable for short hauling distances and repetitive operations (Aggregates and Mining Today 2010)

Figure 3) Tethered Electric LHD (Mining Magazine 2017)

Some advantages of tethered eLHDs are (Jacob 2013) (Chadwick 1992) (Varaschin 2016) (Queen's University, Electric equipment 2017):

- Zero emission, better visibility and better working conditions
- Reduced noise in working area
- Reduced heat production in working area
- Cost savings in ventilation, fuel consumption, and carbon footprint

Despite the high capital costs of tethered electric LHDs, the control of the trailing cable is difficult and restrains the mobility, versatility, and limits in travel distances (Jacob 2013) (Mining Magazine 2017). Due to this issue and to reduce the risk of backing over the trailing cable, a study recommends that tethered electric LHDs are suitable for caving mining methods such as block caving, panel caving, inclined draw point caving and front caving (Paterson and Knights 2012).

Other issues with this type of equipment can be summarized as follows (Jacob 2013) (Chadwick 1992):

- Added cost of cable and reel
- Relocation and reposition of power feeds is difficult and time consuming

2.5.3.3 Battery Powered Vehicle

Battery powered vehicles are the newest technology for running electric vehicles in underground mines. The BEVs provide more flexibility for the machines in the confined environment such as underground mine.

Although battery powered equipment saves on capital costs, reduces infrastructure demand and removes operational limitations compared to other types of electric machines (Varaschin 2016), there are some issues that stand in front of battery electric vehicles. These issues include repercussion on the cycle time of trucks in order to swap the batteries at a swap station (Braul 2018).

One of the biggest challenges of the BEV is the limited battery life and the limited size. For a battery eLHD, 1.5-2 tonnes of batteries are required, and it only keeps a charge for 2-2.5 hours with an estimated recharge time of 2 hours (Jacob 2013). Thus, the machine availability decreases to 50% due to the recharging process (Jacob 2013). However, the new technology is going to tackle this problem by on-board charging (Braul 2018). With this new technology, the vehicle does not need to stay at the charging station or travel to the swap station for swapping a charged battery.

General underground mobile equipment is divided into two categories: high dynamic use, and low dynamic use. Some of the equipment operating in underground mines has low relative demands in production rate. So, they have short utilization and they can work for a long period of time without charging. As well, the charge time duration does not affect the production rate. An example of low dynamic use would be jumbo drill. Unlike low dynamic use equipment, high dynamic use equipment plays a key role to maximize the productivity rate of the mine. So, they require high relative demands, need to work continuously, used over a long duration and require the minimal charge time. LHDs and trucks are examples of high dynamic use (Conklin 2017).

Thus, the dynamic use predicts what battery charging methodology is optimized for the specific application in the mine. There are two different charging methodologies, on-board charging and off-board charging. The battery charger in on-board charging is integrated into the equipment. So, the battery will be charged for the equipment by plugging into the electric grid. However, the battery charger on off-board charging is independent of the equipment. Thus, the battery should be swapped at charging bays.

Three common ways for charging an electric equipment include: fast charging, slow charging, and charging at a battery swap station. In quick charging, the equipment can be charged within 15 minutes for every two hours of operation. Thus, it has the lowest impact on cycle time and mine production rate, but the quick method of charging adds heat to the mine during the charging process and affects the cooling system and ventilation network. Slow charging does not affect the cooling system, but it increases cycle time for underground mining equipment, which decreases the productivity rate. For high dynamic use equipment in underground mines such as the LHD, the optimum charging method is on-board charging with fast charging (Conklin 2017). In the case of off-board charging, a swap station needs to be constructed and this affects the capital costs as well as heat production in an average of 114,306.8 BTU/hr (33.5 kW) per charg bay (Mayhew 2017).

The summary of advantageous and disadvantageous of battery charging methods is given in Table 10.

Charging Method	Advantageous	Disadvantageous
Fast Charging	Low impact on cycle time and production rate	Adds heat to the mine environment, thus, increases the cooling power
Slow Charging	Low impact on ventilation and cooling system	Large impact on cycle time and production rate
Battery Swap Station	Low impact on cycle time and production rate	Requires infrastructure, thus affects on capital costs

Table 10) Advantageous and Disadvantageous of Different Battery Charging Methods

2.6 Factors Affecting Electric Equipment Selection

Studies show that for either electric or diesel engines an evaluation needs to be done by mining companies to ensure that the mining equipment will meet the company's goal and profit (Varaschin 2016). This study mentions that the equipment should be chosen based on characteristics which include the capital costs of equipment, operating costs, the equipment's ability to meet the estimated production rate, the equipment's effect on mine design such as drift and tunnel design, equipment reliability, and comfortability and familiarity of employees with new technology (Varaschin 2016)

2.7 Carbon Tax in Canada

A carbon tax is a tax which is charged by the government on the carbon content of fuels called fossil fuels such as coal, petroleum, and natural gas. In fact, by combusting the fossil fuels, the carbon is converted to carbon dioxide (CO_2) which categorises as Green House Gases (GHG). GHG emissions cause negative effects on the climate system by trapping heat in the atmosphere (Amanda, Huddleston and Rudenstein 2008). The objective of the carbon tax is to reduce the amount of carbon dioxide emissions, thereby retardation of climate change and its negative effects such as global warming.

In Canada, the carbon tax proposed in 2008. Although, there was no federal carbon tax before 2016, the Canadian provinces do have carbon taxes. In 2016, the Canadian government announced that the federal tax is set at a minimum of \$10 per tonne of CO_2 equivalent in 2018, rising \$10 annually to \$50 per tonne in 2022.

Table 11 shows carbon tax plan in different provinces of Canada.

Provinces	Year	
	2018 (\$/tonne of CO_2 equivalent)	2022 (\$/tonne of CO_2 equivalent)
Alberta	\$30	\$50
British Columbia	\$30	\$50
Ontario	\$15.9	\$21.6
Quebec	\$15.9	\$21.6
Manitoba	\$25	Not planned yet
Other Provinces	Federal Tax plan will be implemented	

Table 11) Carbon Tax in Different Provinces of Canada (Good 2018)

2.8 Conclusion on Literature Review

At present, diesel equipment in underground mines is creating more issues by increasing adverse health effects on human, progressively challenging mine regulations, producing extra heat, etc. These factors are made even worse by increasing mine depth (Paraszczak, Svedlund, et al. 2014).

Electric engines only produce heat (McPherson 1993) however, diesel engines emit carbon dioxide, carbon monoxide, oxides of nitrogen, water, DPM and heat (Stinnette 2013). This is the reason why electric engines are more popular than diesel engines among underground mining industries.

The diesel exhaust result in acute health issues and death and has been classified by the International Agency for Research on Cancer (IARC) as a Group I carcinogenic gas to humans since 2012 (IARC 2012). Moreover, the US Occupational Safety and Health Administration (OSHA) describes that diesel emissions increase the risk of cardiovascular, cardiopulmonary and respiratory disease within the human body (OSHA, Hazard Allert: Diesel ExhaustéDiesel Particulate Matter 2013).

DPMs emitted by diesel engines, can carry diesel soot, exotic compounds (organic and inorganic) and heavy metals through the lungs (OSHA, Hazard Allert: Diesel ExhaustéDiesel Particulate Matter 2013). Regarding diesel exhaust adverse health effects on humans, DPMs are considered as one of the main disadvantages of running diesel engines in underground mines.

Chapter 3
Ventilation

3.1 Mine Ventilation Theory

3.1.1 Background

If one looks at the background theory of mine ventilation, it can be found in fluid mechanics and thermodynamics. Some equations play an important role, such as the Bernoulli equation, the steady-flow energy equation, and last but not least, the Chezy-Darcy law which is used for frictional pressure drop in pipes (Tuck 2011).

In 1854, John Job Atkinson wrote his paper on the theory of mine ventilation titled “On the theory of the ventilation of mines”. The classical paper was 154 pages long and was published by the North of England Institute of Mining Engineers (McPherson 1993). In this paper, he simplified and developed the principles of the theory of mine ventilation which are still used today (McPherson 1993).

Atkinson introduced several simplifying assumptions to form the basis of the mine ventilation theory. One of the most important assumptions was the incompressibility of air. In shallow mines (less than 500 meters in depth), the changes in air density are benign (McPherson 1993). However, the incompressibility assumption is not valid for deep mines, for the variability of air density increases with the depth (Tuck 2011). If the change of air density is greater than 5% due to changes in elevation and temperature, the assumption of incompressibility must be avoided (Tuck 2011).

3.1.2 The Mine Airway Resistance

The reason air moves between two places is because of the differential pressure between them. The volume of the air going through a mine tunnel, or a duct, depends on both the magnitude of the pressure differential and the resistance of the airway that the air passes through. The airflow resistance is a function of the tunnel size and surface roughness.

The mine ventilation system is designed by setting the all energy sources within the system to overcome the pressure loss. This might occur naturally, which is called Natural Ventilation Pressure (NVP), or artificially, by fans located on the surface. (Howes 2011).

The mine airway resistance equation, which is also known as Atkinson's Law, comes from the French hydraulic engineers Chezy-Darcy. This equation is for pipes and is stated in Equation 1

$$\mathrm{P} = f\mathrm{L}\frac{Per}{A}\rho\frac{u^2}{2} \quad \mathrm{Pa} \qquad \text{(Equation 1)}$$

Where;

P is pressure drop (Pa)

f is Coefficient of friction (Dimensionless).

L is the length of pipe (m)

Per is the perimeter of the pipe (m)

A is the cross-sectional area of the pipe (m^2)

ρ is the density (kg/m^3)

And u is the velocity (m/s)

Atkinson assumed that the air behaviour is considered as a fully turbulent flow in underground mine airways. Thus, the coefficient of friction f in Equation 1 is considered as a constant value for any given airways.

Thus, the constant factors can be collated together and introduced as a single factor, which is called the Atkinson friction factor and is given in Equation 2.

$$k = \frac{f\rho}{2} \quad kg/m^3 \qquad \text{(Equation 2)}$$

The Chezy-Darcy equation then becomes

$$\mathrm{P} = k\mathrm{L}\frac{Per}{A}u^2 \quad \mathrm{Pa} \qquad \text{(Equation 3)}$$

Equation 3 is known as Atkinson's equation. The difference between Atkinson's equation and the Chezy-Darcy equation is that the Atkinson friction factor (K) is a function of density, however the Chezy-Darcy friction factor is a dimensionless parameter.

Atkinson's equation can be written in terms of air flow quantity Q (m^3/s), where $Q = uA$, giving

$$P = kL\frac{Per}{A^3}Q^2 \quad \text{Pa} \qquad \text{(Equation 4)}$$

By considering constant density, the friction factor varies only with the roughness of an airway's surface for fully developed turbulent flow. Thus, all of the factors can be collated together to define a new single variable called airway resistance. This can be given in the following equation

$$R = kL\frac{Per}{A^3} \quad N\,s^2/m^8 \qquad \text{(Equation 5)}$$

By comparing Equation 4 and 5, the Square Law of mine ventilation can be defined as the following equation:

$$P = RQ^2 \quad \text{Pa} \qquad \text{(Equation 6)}$$

The Square Law of mine ventilation is the most commonly used formulas in underground mine ventilation system design due to its simplicity and acceptable precision.

If the air density is not a standard density (i.e. not 1.2 kg/m^3), the Atkinson resistance is calculated with the following formula:

$$R = k_s L\frac{Per}{A^3}\frac{\rho}{1.2} \quad N\,s^2/m^8 \qquad \text{(Equation 7)}$$

Where k_s is the friction factor at air density of 1.2 kg/m^3.

The Atkinson friction factor can be determined in a number of ways (McPherson 1993):

- By analogy with similar airways from ventilation surveys
- By direct measurement
- From design table (Table 5.1, (McPherson 1993))
- From geometric data

The concept of airway resistance is very important in underground ventilation planning. The ventilation cost of an underground mine varies directly with the resistance of the airway that air passes through. This can be estimated by determining the air power required in an airway ($Air\ power = P \times Q$). The pressure loss is determined by the Square Law, resulting in $Air\ power = R \times Q^3$. Thus, the resistance is directly influencing the ventilation cost.

The Square Law (Equation 6) shows that the airway resistance has a constant of proportionality between pressure drop and the square of the airflow at a specific air density. Figure 4 shows the airway resistance curve as a parabolic form of the square law in a $P - Q$ plot.

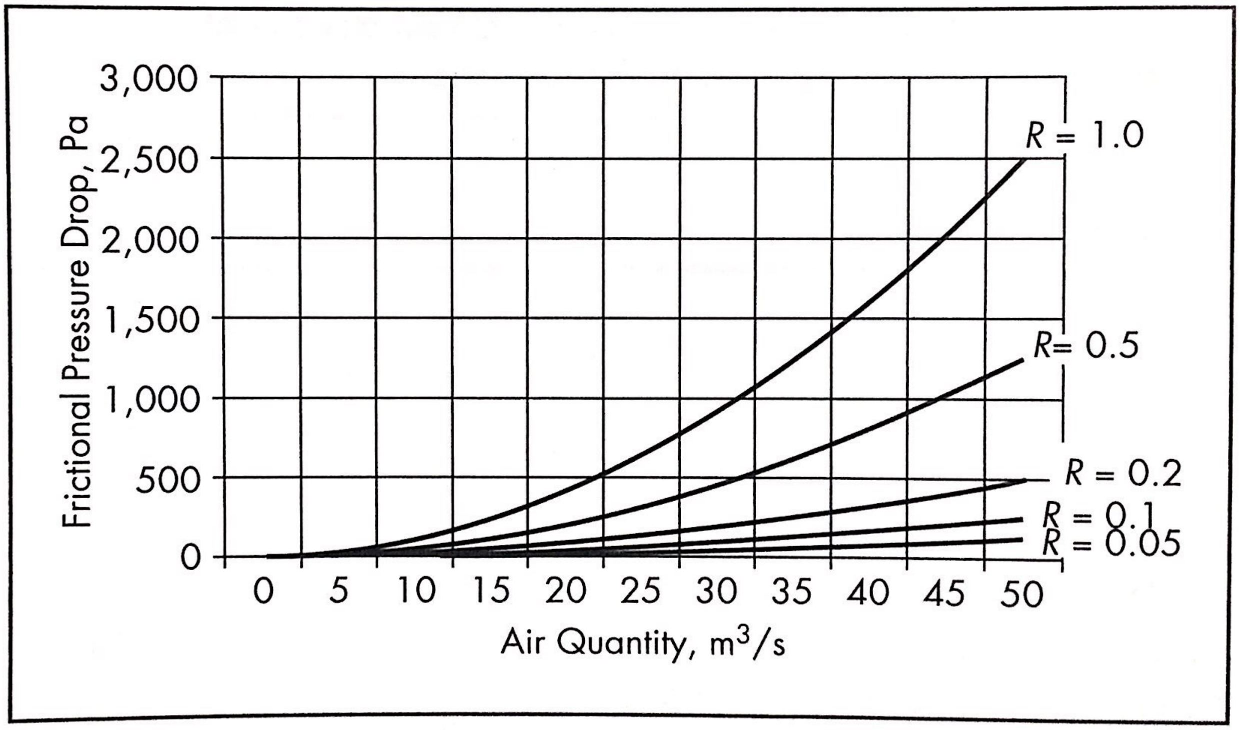

Figure 4) Mine Airway Resistance Curve (McPherson 1993)

3.1.3 Shock Losses

In addition to frictional resistance, additional energy losses occur whenever an airflow's direction changes. These occur at bends, junctions, and changes in airway cross section due to enlargement or contraction. These losses consume energy and create frictional resistance.

The pressure loss caused by shock losses can be calculated as

$$\mathrm{P}_{Shock} = \mathrm{X}\rho(u^2/2) \quad Pa \qquad \text{(Equation 8)}$$

Where,

X = Shock loss factor which are listed in Chapter 5 (McPherson 1993)

$\rho = air\ density\ (kg/m^3)$

$u = air\ velocity\ (m/s)$

The shock loss resistance can be calculated with the following equation

$$\mathrm{R}_{Shock} = R_s = \mathrm{X}\rho/2A^2 \quad Ns^2/m^8 \qquad \text{(Equation 9)}$$

Equation 9 can also be turned into an equivalent length. By definition, the equivalent length expresses the additional shock loss resistance, R_{Shock}, in terms of the length of the straight airway with the same value of shock resistance.

$$R_s = (KL_{eq}per/A^3)\,\rho/1.2 \quad Ns^2/m^8 \qquad \text{(Equation 10)}$$

Or

$$L_{eq} = 1.2 \text{ X } A/(2K\,per) \quad m \qquad \text{(Equation 11)}$$

3.2 Mine Ventilation Systems

3.2.1 Background

Regardless of the type of mine and mining method, its geometry/layout, size, geology, and pollutants, ventilation systems can be classified as:

- Mine systems
- District systems
- Auxiliary ventilation systems

Figure 5 illustrates a simplified schematic of a mine ventilation system. In this figure, fresh air enters via a downcast shaft to feed air to working areas along the intake airways. Contaminated air returns back to the surface along return airways and the upcast shaft. Doors, stoppings, air-crossings, and regulators control the airflow quantity, but also the quality of air while preventing fresh air from mixing with polluted air.

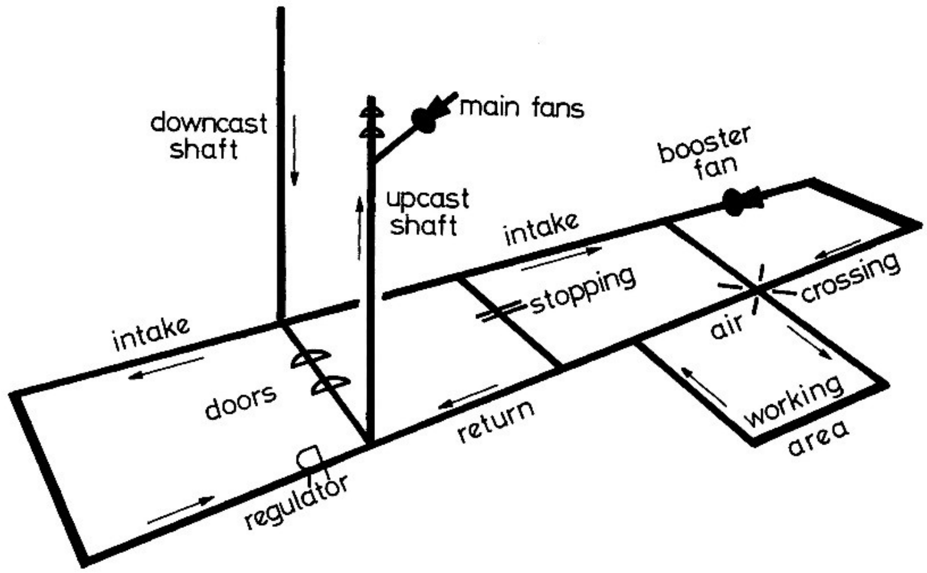

Figure 5) Ventilation System in Underground Mine (McPherson 1993)

3.2.2 Kirchhoff's Laws

A German physicist named Gustav R. Kirchhoff (1824-1887) recognized a fundamental relationship that governs the behaviour of electrical current within a network of conductors. This basic relationship is also applicable to closed ventilation networks at steady state.

Kirchhoff's first law states that the mass flow entering a junction equals the mass flow leaving that junction:

$$\sum_j \dot{M} = 0 \qquad \text{(Equation 12)}$$

As we know, $\dot{M} = Q\rho$, hence:

$$\sum_j Q\rho = 0 \qquad \text{(Equation 13)}$$

By considering the air density as a constant variable in any single junction:

$$\sum_j Q = 0 \qquad \text{(Equation 14)}$$

Equation 14 gives an accurate airflow measurement taken around a junction.

Kirchhoff's second law states that the algebraic summation of all pressure drops around a closed path must be zero. The effect of fans and/or natural ventilation pressures must be taken into account.

Kirchhoff's second law for compressible flow is

$$\sum_{m}(P - P_f) - NVP = 0 \quad Pa \qquad \text{(Equation 15)}$$

Or

$$\sum_{m}(R\ Q\ |Q| - P_f) - NVP = 0 \quad Pa \qquad \text{(Equation 16)}$$

Where,

$P_f = \rho W =$ Frictional pressure drop rise in total pressure across a fan

Q = Airflow (m^3/s)

|Q| = The absolute value of airflow

R = Atkinson resistance ($N\ s^2/m^8$)

NVP = Natural Ventilation Pressure

In Equation 16, $|Q|$ ensures that the frictional pressure drop has the same sign as airflow.

3.2.3 Ventilation Network Solution

One of the most challenging issues in underground mines is the ventilation design and analysis to provide acceptable environmental conditions in working areas. Determining the airflow requirement, distribution of airflows, fan design, fan location and duty is considered an essential component in mine ventilation design.

There are two different methods to analyze a fluid network: analytical methods and numerical methods. The analytical methods solve the fluid network by applying governing laws and formulating them into sets of equations. The numerical methods use iterative procedures to solve equations and these procedures continue until a solution is found within a specified accuracy. Most of the simulation programs utilize the numerical methods to solve equations.

The first step in the numerical method analysis is to distribute pressure and airflow through the ventilation network.

3.2.3.1 Hardy Cross Technique

This method has been introduced by Professor Hardy Cross from the University of Illinois in 1936. However, this has been modified and developed for mine ventilation systems by D.R. Scott and F.B Hinsley at University of Nottingham in 1951. Most of the ventilation simulation programs use this method to solve the ventilation network analysis.

The primary purpose of ventilation network analysis is to provide airflow distribution through the system. First, the airflow, Q, is unknown. By assuming a new airflow Q_a which is less than the true value, we have:

$$Q = Q_a + \Delta Q \qquad m^3/s \tag{Equation 17}$$

By applying the Square Law to Equation 17:

$$P = R(Q_a + \Delta Q)^2 \qquad Pa \tag{Equation 18}$$

Equation 18 can expand to:

$$P = RQ_a^2 + 2RQ_a\Delta Q + R(\Delta Q)^2 \qquad Pa \tag{Equation 19}$$

In Equation 19, the frictional pressure drops related to the assumed airflow, Q_a, is:

$$P_a = RQ_a^2 \qquad Pa \tag{Equation 20}$$

A differential pressure drop can be introduced as 'error':

$$\Delta P = P - P_a = 2RQ_a\Delta Q + R(\Delta Q)^2 \tag{Equation 21}$$

By assuming ΔQ is a very small number, $R(\Delta Q)^2$ can be ignored from Equation 21.

When $Q_a \rightarrow Q$, then

$$\frac{\Delta P}{\Delta Q} \rightarrow \frac{dP}{dQ} = 2RQ \tag{Equation 22}$$

Then,

$$\Delta Q = \frac{\Delta P}{2RQ_a} \tag{Equation 23}$$

With the Newton-Raphson method the roots of Equation 23 are estimated.

The composite value of ΔQ is estimated by the following equation. For more information, refer to (McPherson 1993).

$$\Delta Q_m = \frac{-\Sigma(RQ_a|Q_a^{n-1}| - P_f - NVP)}{\Sigma(nR|Q_a^{n-1}| + S_f + S_{nv})} \qquad \text{(Equation 24)}$$

Where,

$P_f = fan\ pressure$

$NVP = Natural\ Ventilation\ Pressure$

$S_f = The\ slope\ of\ P - Q\ Characteristic\ curve\ for\ the\ fan$

$S_{nv} = The\ slope\ of\ P - Q\ characteristic\ curve\ for\ the\ natural\ ventilation\ effects$

However, in practice, $S_{nv} = 0$, i.e., it is assumed that natural ventilation effects are independent of airflow.

For most of underground ventilation system, Equation 25 would be applicable with acceptable accuracy:

$$\Delta Q_m = \frac{-\Sigma(RQ_a|Q_a| - P_f - NVP)}{\Sigma(2R|Q_a| + S_f + S_{nv})} \qquad \text{(Equation 25)}$$

The Hardy Cross procedure can be summarized based on (McPherson 1993)

1. A network schematic is drawn and closed meshes are chosen, which are included in all the branches.
2. The initial estimate of airflow, Q_a, for each branch is made.
3. On one mesh, the mesh correlation factor, ΔQ_m is calculated.
4. On the same mesh and same direction, the airflow with ΔQ_m is adjusted.
5. Steps 3 and 4 for each mesh are repeated.
6. Steps 3, 4, and 5 are repeated until the value of $-\Sigma(RQ_a|Q_a^{n-1}| - P_f - NVP)$ is close to zero (Kirchhoff's Second Law is satisfied with an acceptable degree of accuracy)

3.3 Heat and Humidity

3.3.1 Background

Figure 6 demonstrates the workers' performance based on temperature. As Figure 6 shows, at 22 °C effective temperature in the underground workings, the workers have the highest efficiency and by increasing temperature, the worker efficiency is decreased. This effect is more pronounced on workers' performance if the effective temperature exceeds 30 °C.

Above 32 °C wet bulb temperature, no one can work.

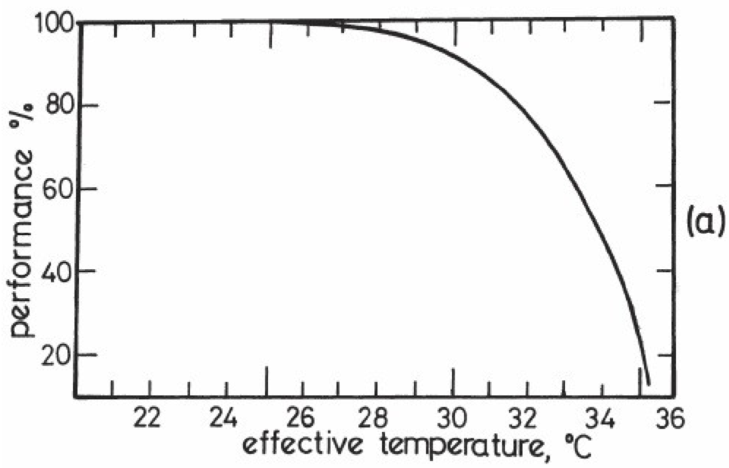

Figure 6) Underground Mine Climate Effects on Workers (Poulton 1970)

In Figure 7, the relationship between wet bulb temperature, air velocity, and workers' performance in an underground mine has been shown. It indicates that the work performance decreases as wet bulb temperature increases above 30 °C. By these explanations, the comfort zone for the underground miners can be considered around 28 °C wet bulb temperature or less.

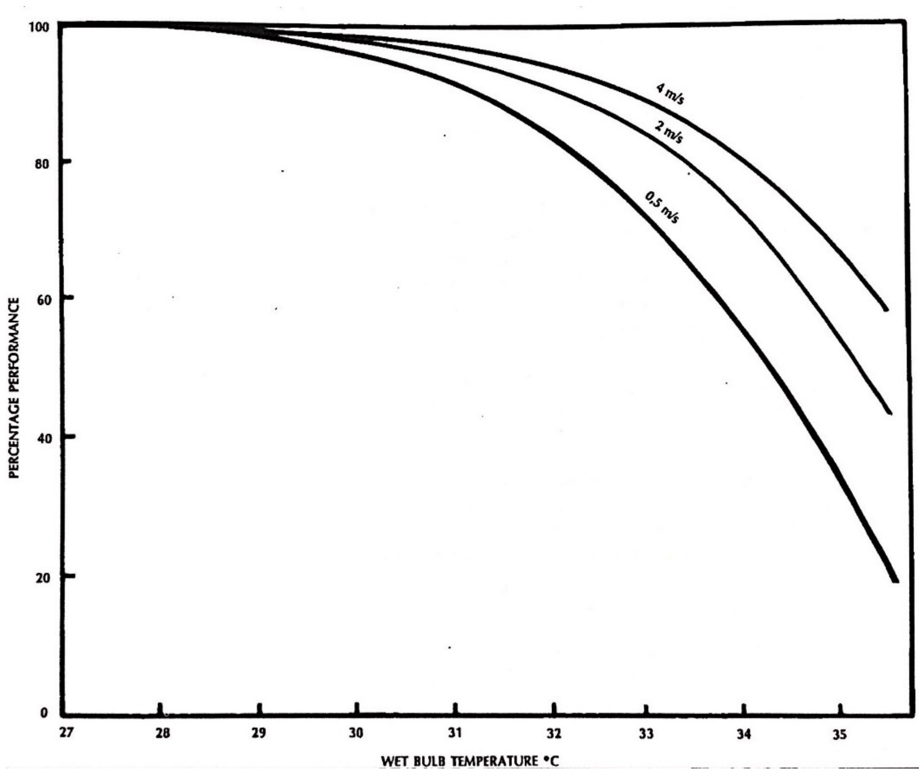

Figure 7) Underground Mine Workers Performance in Different Wet Bulb Temperature and Air Velocity (Le Roux 2008)

The number of mines suffering heat problems because of the deepening of the working area is increasing. It is widely known that high temperature and humidity in an underground workplace result in reduced performance and impaired attention of workers. Thus, countries around the world define a specific amount of time that a miner can work safely in a hot situation. The permissible working hours are shown in Figure 8.

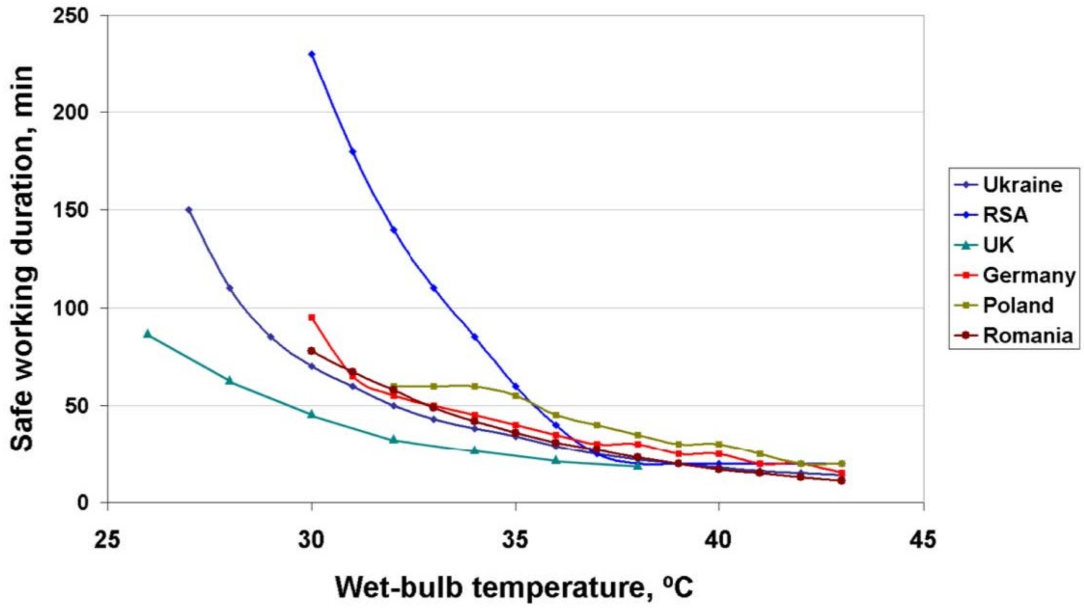

Figure 8) Permissible Work Duration at High Wet Bulb Temperature in Different Countries (Workplace Safety North, 2014)

3.3.2 Heat Load Sources in Underground Mines

The major sources of heat production in deep underground mines are auto-compression, diesel or electric equipment, and geothermal rock strata. Minor sources of heat production such as heat released from compressed air, and underground water can be negligible. In fact, in the heat balance calculation for a hot mine, the amount of heat generated by body metabolism, and electrical cables is relatively insignificant. It can be considered offset by the cooling effect of releasing compressed air –without sacrificing accuracy of the heat balance, as a whole.

3.3.2.1 Auto-Compression

Auto-compression occurs when air goes down the shaft (either naturally or through a man-made ventilation system) and it experiences a physical compression. This means that although the volume of air has been reduced, the amount of heat remains the same resulting in hotter air. Auto-compression is independent of the quantity of air and its factor may be altered depending on moisture content in shafts.

The temperature rises of air that is going down the shaft, assuming that there is no interchange in heat and moisture content in the shaft, can be calculated from the following equation:

$$\frac{T_2}{T_1} = (P_2 - P_1)^{[\frac{\gamma-1}{\gamma}]} \qquad \text{(Equation 26)}$$

Where,

T is absolute dry-bulb temperature, (℃)

P is atmospheric pressure, (Pascal)

γ is the ratio of the specific heats of air at constant volume and pressure,

1 and 2 denotes the initial and final conditions respectively.

In fact, auto-compression is a conversion of potential energy within the air of thermal energy by increasing the depth. Thus, some of the potential energy of the fluids (i.e. air) is going to be converted to enthalpy creating an increase in pressure, internal energy and hence, temperature:

$$\mathrm{H} = \mathrm{PV} + \mathrm{U} \tag{27}$$

Where,

H is enthalpy (J/kg)

P is pressure (Pascal)

V is specific volume

U is specific internal energy.

Equation 28 describes auto-compression in mathematical language:

$$\mathrm{C_{p,m}}(\mathrm{T_2} - \mathrm{T_1}) - \mathrm{L\Delta X} = \mathrm{g}(\mathrm{h_1} - \mathrm{h_2}) \tag{28}$$

Where:

$C = Specific\ Heat$

$T_2 = Temperature\ in\ underground\ mine$

$T_1 = Temperature\ on\ surface$

$\Delta X = Increase\ in\ vapor\ content\ due\ to\ evaporation$

$L = Latent\ heat\ of\ evaporation$

$L\Delta X = Conversion\ of\ sensible\ heat\ to\ latent\ heat$

As Figure 9 and Figure 10 show, the dry bulb and wet bulb temperature of the air increases as the air falls through a downcast shaft,

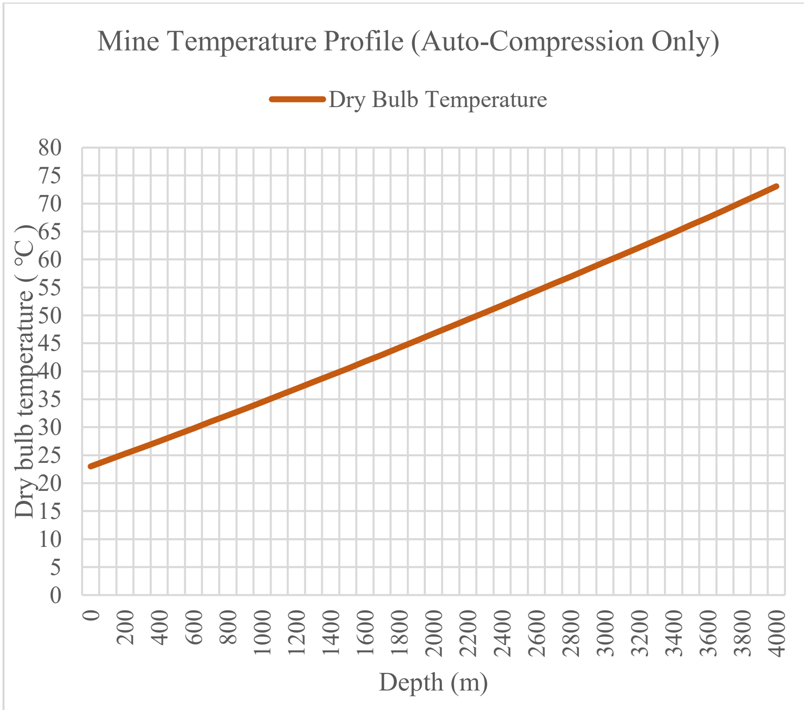

Figure 9) Auto-Compression Effects on Dry Bulb Temperature by Increasing Depth

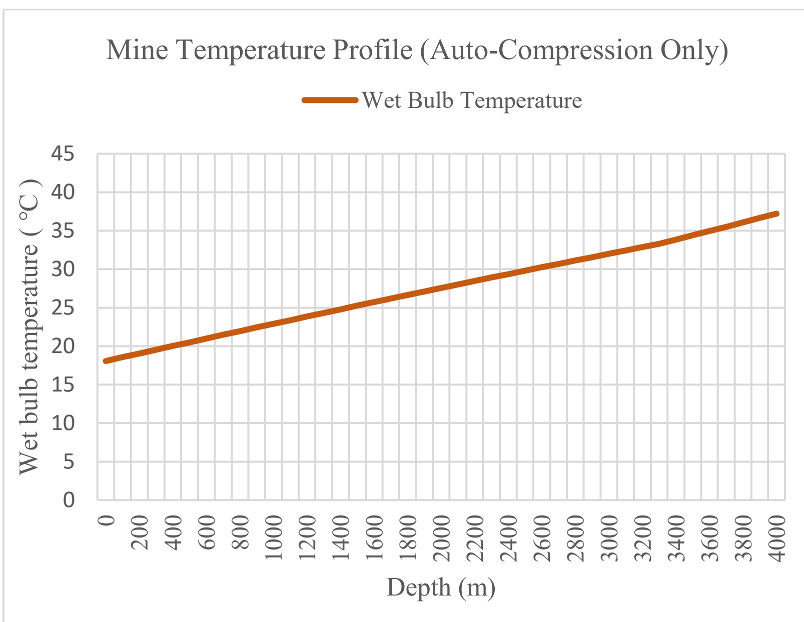

Figure 10) Auto-Compression Effects on Wet Bulb Temperature by Increasing Depth

The effect of auto-decompression occurs in the exhaust shaft where polluted air is returned to the surface. In this airway, the temperature drops, and it causes fog and condensation in this airway. The negative effect of fogging in the return airway creates corrosion on fan impellers or stalls on the surface fan until the suspended blanket of water cascades down when the fan starts up again. To prevent this situation, the maximum velocity of the air should not exceed 10 m/s in return airway.

3.3.2.2 Geothermal Heat from Strata

The rock temperature at 50 m depth in the earth's crust is approximately equal to the surface air temperature. Beyond this depth, the temperature increases over the surface temperature. This changing temperature of the rocks in an underground mine is known as a geothermal gradient and it varies with both tectonic and thermal properties of the rocks. This type of heat production in underground mines is the most complex to analyse in a quantitative manner . For practical purposes, the average value of a geothermal gradient for metalliferous mines is 1.8 ℃ per 100 m (McPherson 1993).

An underground mine behaves like a thermostat (McPherson 1993), and the heat transfer occurs between the air and rock and vice versa. During the cold seasons, the rock strata is warmer than the surface air, thus the heat transfers is from the rock strata of the ambient air in the mine. During warmer seasons, it is the opposite. The air going down the shaft gains heat from auto compression and engines in the mine. The air is thus warmer than the rock strata and the heat transfer is from the air to the rock.

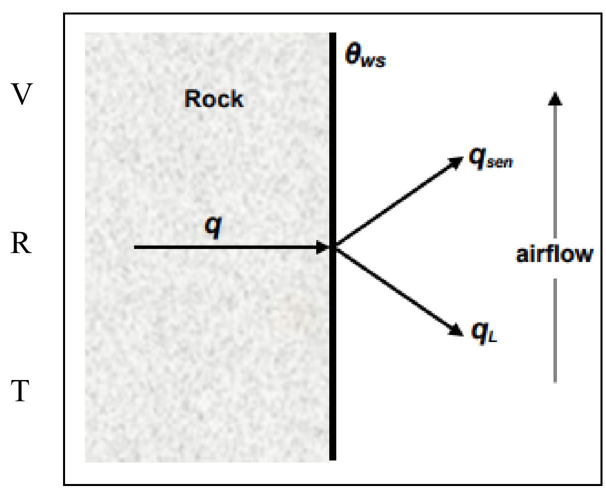

Figure 11) Heat Transfer in Rock Strata (McPherson 1993)

The type of heat transfer in the rock is by conduction. Sensible heat (q_{sens} in Figure 11) is a combination of convection and radiation. Latent heat (q_l in Figure 11) exists only if there is wetness (X) on the walls.

For a dry tunnel, the heat flux can be determined from the following formula:

$$\frac{q}{A} = h\frac{G}{B}(VRT - t_{bulk}) \qquad \text{(Equation 29)}$$

For a wet tunnel, the heat flux can be determined in the following formula:

$$\frac{q}{A} = h\left[\frac{G}{B}(VRT - t_{bulk})\left(1 - \frac{\chi}{100}\right) + (t_{ws} - t_{bulk})\frac{\chi}{100}\right] \qquad \text{(Equation 30)}$$

Where,

G: Dimensionless temperature gradient and requires Gibson's algorithm.

t_{ws}: Wet surface temperature; requires iteration

VRT: Virgin Rock Temperature

B: Biot number

3.3.2.3 Mechanized Equipment

This equipment produces heat in two ways which include (McPherson 1993):

- Heat production from their engine
- Heat generation from their operation working against gravity (e.g. travelling up a ramp)

The mechanized equipment can be divided into two groups: diesel and electric.

3.3.2.4 Diesel Equipment

These types of equipment have an average thermal efficiency of 33%. This means that only 1/3rd of the fuel consumption is converted into mechanical work (McPherson 1993). Thus, the remaining 2/3rd of the heat is released to the underground environment. The overall efficiency of diesel engines is one third of electric equipment with the same power rate. Hence, diesels generate three times more heat than electric engines (McPherson 1993). For diesels, the rate of fuel consumption is 0.31 litres per kW per hour (McPherson 1993). If we consider 34000 kJ/J as the calorific value of diesel fuel, the emitted heat can be determined from the following formula:

$$\frac{0.31}{60 \times 60} \frac{litres\ diesel}{kW\ Output \times Second} \times 34000 \frac{kJ\ Heat}{litres\ diesel} = 2.83\ kJ/s\ (or\ kW) \quad \text{(Equation 31)}$$

Diesel engines produce both latent heat and sensible heat, which are emitted in three ways: from the machine’s radiator, exhaust gases, and frictional process in the motor engines.

3.3.2.5 Electric Equipment

Electrical equipment has an efficiency of more than 90% and they release less heat to the underground environment. Electric engines only produce sensible heat.

3.3.2.6 Explosives

The studies show that as much as 90% to 95% of the energy release from blasting operations at an underground mine is converted to heat. Parts of this heat generation contain blasting fumes which are emitted from blasting holes. The rest of the heat produced will remain in the broken rocks and will be released to the mine environment gradually over the life of the mine (T. Payne & R. Mitra 2008).

3.3.2.7 Other Heat Sources

Other sources of heat generation in underground mines include metabolic heat from the human body, dewatering sumps, battery swap stations, underground water (ground water and mine water), cemented back fill, mechanical processes and lights, compressed air, main/auxiliary/ booster fans, pumps, chargers, etc.

3.3.2.8 Heat Source Percentages

Mines in different parts of the world will not have the same heat source percentages. However, as Figure 12 shows, auto-compression remains the top heating load as mine depth becomes higher (Queen's University, Ventilation 2017).

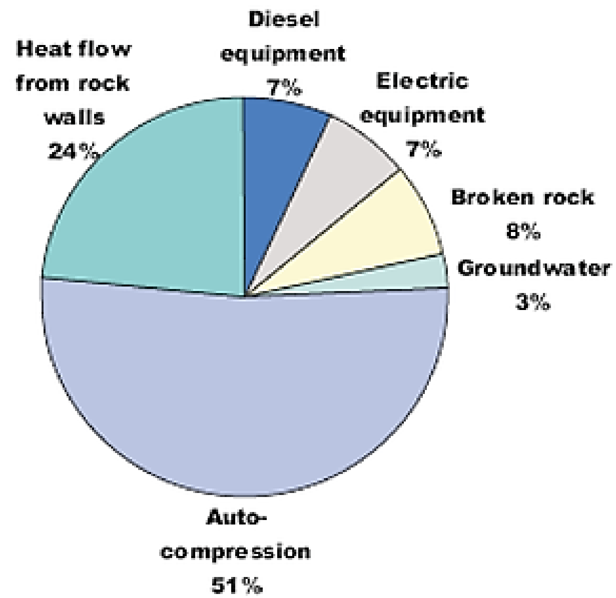

Figure 12) Major Heat Sources Distribution in Underground Mines

3.3.3 Basic Principles of Mine Heat

3.3.3.1 Fundamentals

The study of mine heat and cooling requires one to have a solid foundation in Psychrometry. Psychrometry is a field in thermodynamics that studies systems or processes involving both dry air and water vapour. The combination of dry air and water vapour is called moist air. In the combination of mixed air with vapour, the mass of dry air remains constant, but only the mass of water vapour can increase or decrease.

3.3.3.2 Dry Bulb Temperature

Dry bulb temperature is the temperature that we always read from weather forecast TVs or websites.

3.3.3.3 Relative Humidity

Relative humidity is the ratio of partial pressure exerted by the water vapour in the moist air to the partial pressure of saturated vapour (pure water) at a given temperature:

$$\varphi = \frac{P_v}{P_g} \qquad \text{(Equation 32)}$$

Where,

P_v is partial pressure exerted by the water vapour in the moist air.

P_g is partial pressure of saturated vapour (pure water).

And φ is the relative humidity

P_g is estimated with Equation 33:

$$P_g = 0.6106 \times e^{\left(\frac{17.27 \times T_{db}}{237.3+T_{db}}\right)} \qquad \text{(Equation 33)}$$

Where, T_{db} is dry bulb temperature.

3.3.3.4 Absolute Humidity

Absolute humidity is a molar mass ratio of water vapour to dry air and it gives a direct measure of the moisture content.

From the ideal gas equation:

$$pV = MRT \qquad \text{(Equation 34)}$$

Where,

R = specific gas constant

M = molar mass of gas $[kg/kmol]$

Absolute humidity is determined by the following formula:

$$\omega = \frac{Molar\ mass_v}{Molar\ mass_a} = 0.622\frac{P_v}{P - P_v} \qquad \left[\frac{kg\ of\ vapor}{kg\ of\ dry\ air}\right] \qquad \text{(Equation 35)}$$

3.3.3.5 Enthalpy (h)

Enthalpy measures the energy content of the moist air. Ideal gas assumption enables us to simplify enthalpy analysis by setting $h = h(T)$

$$\begin{gathered} h = h_a + \omega h_v \\ h = c_{p,a}T_{db} + \omega h_g \end{gathered} \qquad \text{(Equation 36)}$$

Where,

$$c_{p,a} = specific\ heat\ capacity\ of\ dry\ air \ldots \frac{kJ}{kg°C}$$

$$h_g = enthalpy\ of\ saturated\ water\ vapor \ldots\ \frac{kJ}{kg}$$

Thus, Enthalpy is given in equation 37:

$$h = 1.005T_{db} + \omega(2501.2 + \underbrace{1.84}_{c_{p,v}} T_{db}) \ldots \left[\frac{kJ}{kg_a}\right] \qquad \text{(Equation 37)}$$

3.3.3.6 Sigma Heat

Sigma heat (σ) measures the energy content of the moist air mixture at saturation, excluding liquid condensate energy. It depends only on the wet bulb temperature of air for any given barometric pressure (McPherson 1993)

The sigma heat of the air is calculated with the following equation:

$$\sigma = 1.005T_{db} + \omega(2501 - 2.386T_{db}) \ldots \left[\frac{kJ}{kg\ of\ dry\ air}\right] \qquad \text{(Equation 38)}$$

3.3.3.7 Wet Bulb Temperature

Wet bulb temperature is a temperature that would be measured if moist air were cooled to saturation by the sole effect of evaporation. At saturation, the dry bulb and wet bulb temperatures are equal. Thus, the saturation conditions in the sigma heat would be evaluated at wet bulb temperature.

Wet bulb temperature is calculated with the following formula:

$$T_{wb} = \frac{(1.005 + 1.84\omega_1)T_1 - 2501.2(\omega_2 - \omega_1)}{1.005 + 1.84\omega_2 - 4.186(\omega_2 - \omega_1)} \qquad \text{(Equation 39)}$$

In mine ventilation and cooling design, wet bulb temperature always is considered. The reason is that human comfort depends on the evaporation of sweat from the skin and the rate at which moisture can be taken up by the atmosphere (Hartman, et al. 2012).

3.3.3.8 Cooling Capacity of the Air

Cooling capacity or free cooling is the term to describe the moist air's ability to absorb the heat with respect to a design temperature before mechanical cooling is required.

To calculate free cooling, first, we require an estimate of the sigma heat of design (σ_{design}) which uses a threshold value of wet-bulb temperature. This threshold value is normally stated by the mine or governmental regulations.

Second, the sigma heat local (σ_{in}) uses ambient wet bulb temperature, which can be at surface condition or at level entry condition.

Then, the free cooling is estimated by Equation 40:

$$Q_{free} = \dot{m}_a(\sigma_{design} - \sigma_{in}) \quad \text{(Equation 40)}$$

If the cooling capacity of the air is negative, it means we need cooling for the workers in the specific working area. If it is positive, it means the air has room to absorb additional heat on that specific area.

Calculating the cooling capacity of the air is very important and it needs an iterative simulation to be estimated. Indeed, air flow volume requirements should be estimated by the cooling capacity of the air. In fact, knowing how much air flow volume one needs in each active working area, the cooling capacity of each of the active working area should be estimated.

3.4 Theory of Cooling and Refrigeration

3.4.1 The Need for Cooling Plants in Deep Underground Mines

The importance of keeping the air temperature within human comfort levels at deep underground mines has been discussed in section 3.1.1. To provide this comfort zone, wet bulb temperature should be kept around 27°C and must not exceed 30 °C (Hartman, et al. 2012).

When active working areas become too hot and humid, some methods might be applied to overcome this situation such as increasing air flow or drying out intake airways. However, these methods are not always economical, and/or the mine's ventilation system may not be able to remove the heat from the mine. Then, installing a refrigeration plant must be considered (Le Roux 2008) (Tuck 2011) .

Thus, it is necessary to cool the whole mine or some part of it when ventilation alone is insufficient to keep adequate comfort conditions in hot working areas.

Two alternative methods are utilized to cool the deep underground mines: direct and indirect heat exchanger cooling system.

3.4.2 In-Direct Heat Exchanger Cooling Systems

A heat exchanger is a device used to transfer heat between two or more fluids. If the fluids are separated by a wall to prevent mixing, this is called in-direct heat exchanger.

3.4.2.1 Theory of Refrigeration

The mechanical process of heat absorption from one place and the transfer or discharge to another place is called refrigeration. Refrigeration is considered as an indirect heat exchanger since the two fluids do not combine with each other. The refrigerant alternates between the liquid and vapour phases. Thus, this process is called vapour refrigeration or change of state (Howard L. Hartman; Jan M. Mutmansky; Raja V. Ramani; Y.J. Wang 1997).

A vapour refrigeration plant consists of four basic systems, including an evaporator, compressor, condenser, and expansion valve. Figure 13 shows a schematic of the vapour refrigeration system.

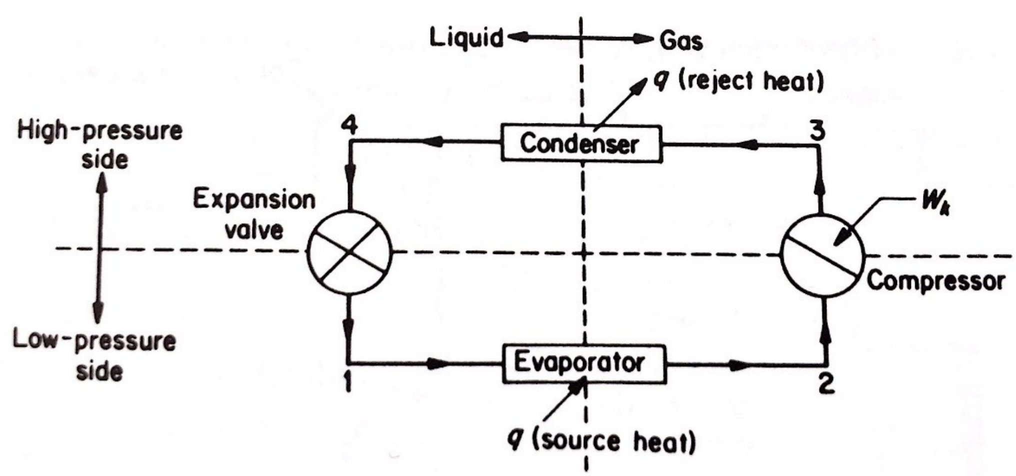

Figure 13) Schematic of Vapour Refrigeration System (Howard L. Hartman; Jan M. Mutmansky; Raja V. Ramani; Y.J. Wang 1997)

As Figure 13 shows, by absorbing heat from a heat source at the evaporator, the refrigerant changes state from liquid to gas, with no change in temperature. At the compressor, the refrigerant flows at vapour state where work is done to compress it. The vapour condenses to liquid again and releases heat at condenser without a temperature change. Then, the liquid goes through the expansion valve where the temperature and pressure of liquid drop due to expansion. The evaporator and condenser are typically shell-and-tube heat exchanger where the refrigerant flows in the shell and the water inside the tubes.

Chillers are commonly used in underground cooling projects, and usually operate using the vapour-compression or the ammonia absorption refrigeration cycles (McPherson 1993).

3.4.2.2 Coil

A cooling coil is another type of in-direct heat exchanger and is constructed with metal tubing and fins. A coil can be used as a closed loop, and this gives some advantages such as: balanced pumping heat, elimination of water-control tools, no flooding danger in operation levels, and dual use for summer cooling and winter heating (Hartman, et al. 2012). However, the fouling and corrosion in the coils requires the higher installation and maintenance costs, while large space requirement makes them unpopular in underground mine cooling operations.

3.4.3 Direct Heat Exchanger Cooling Systems

In a direct heat exchanger, two or more fluids are in contact with each other to directly transfer heat. The different types of direct heat exchanger cooling plants which are common in underground mines include spray cooling, cooling tower, and bulk air cooling.

3.4.3.1 Spray Cooling

There are two types of spray cooling in underground mines: a chilled water spray chamber, and a cooling tower. The principle of heat transfer in spray cooling is evaporative cooling, which is the conversion of sensible heat to latent heat by the addition of moisture and without changing the total heat content of the air.

3.4.3.1.1 Chilled Water Spray Chamber

There are two different types of spray chambers for underground mines, large spray coolers for bulk cooling, and portable small spray chambers suitable for localized areas in underground.

Large spray coolers are either designed vertically or horizontally. The vertical design is similar to cooling towers, but the spray water is supplied by cold water. They may have heat transfer duties up to 20 MW. The horizontal spray chambers have less capacity than vertical one, but they are more convenient for underground areas where the space is limited. As Figure 14 indicates, the chilled water spray in the way of warm air and makes it cool. The warm water is dumped to the dewatering system.

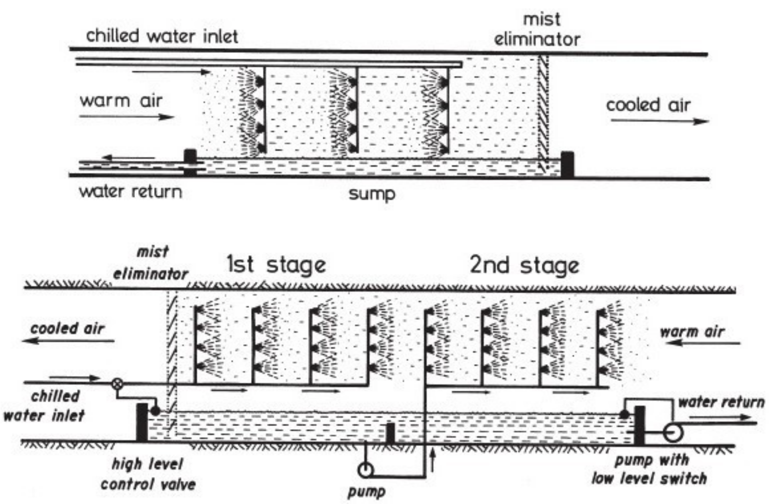

Figure 14) Schematic of Spray chamber (single and two stages) (McPherson 1993)

3.4.3.1.2 Cooling Towers

A cooling tower is a direct heat exchanger device that cools water by a combination of heat and mass transfer. In cooling tower process, the water in wet bulb temperature of the air is sprayed into the air. Then air vaporizes the water and the sensible heat of the air is converted to latent heat. By this process, the air becomes cool. For, there is no adding and removing heat to the air, this process is called adiabatic saturation (Hartman, et al. 2012). The main application of cooling towers in underground mines is to produce cold water on the surface for using in cooling air directly in underground heat exchangers.

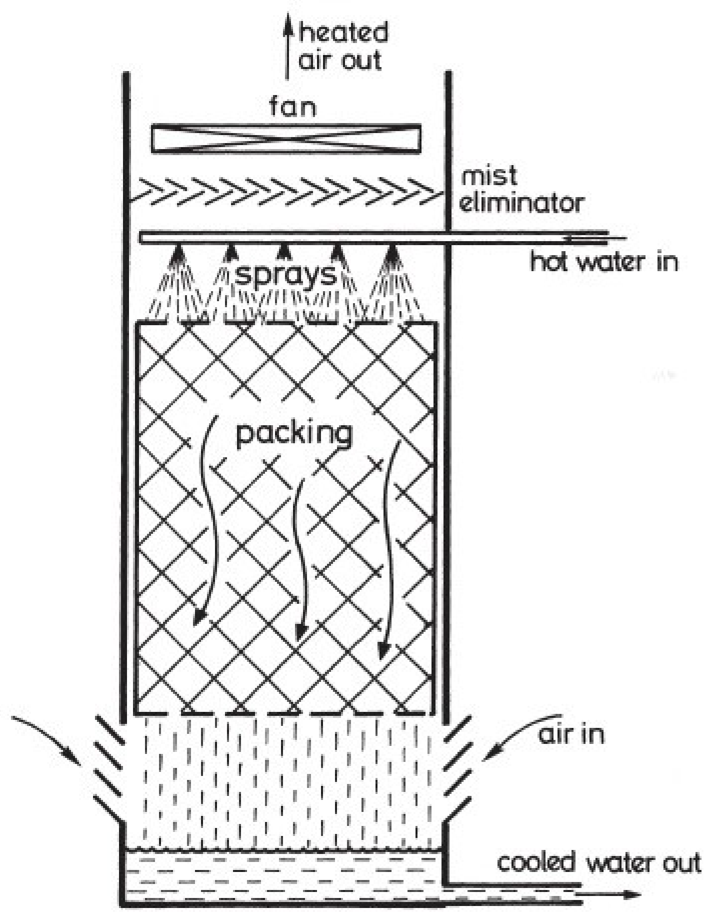

Figure 15) Schematic of Cooling Tower (McPherson 1993)

As Figure 15 indicates, the hot water is sprayed down in a counter-flow direction to rising air, which is forced up by the fans located on the top of the tower. By passing water from packing material, the contact area and time with water and air increases.

The temperature of sprayed water is 45 ℃ which cools to about 35 ℃ before returning back to refrigeration room. The total heat transfer in water side (Equation 41) should equal to the total heat transfer in the air side (Equation 42) (Hartman, et al. 2012)

$$H_w = M_w C_p \Delta t \quad kw \qquad \text{(Equation 41)}$$

$$H_a = M_a \Delta h \quad kw \qquad \text{(Equation 42)}$$

Where,

H_w and H_a are total heat transfer in water side and air side, respectively.

M_w and M_a are the mass flow of water, and air, respectively. (kg/s)

C_p is the specific heat of water, 4.18 kJ/kg ℃

Δt is the water temperature differential on inlet and outlet

Δh is the specific enthalpy differential on entering and exiting of air. kJ/kg

3.4.3.2 Bulk Air Cooling

Bulk air cooling is an evaporative cooling type of direct heat exchanger, similar to the cooling tower. But the difference is that the heat transfer direction in BAC is the inverse of cooling tower, which means the inlet water is colder than the intake air wet bulb temperature. Thus, the air is cooled to a lower wet bulb temperature.

BACs are classified into two categories: Bulk air cooling on the surface, and bulk air cooling underground. The bulk air surface cooler is used in warm-climate mines to provide a cold situation in underground year-round. These coolers are less expensive in installation and maintenance. They work continuously without any disruption and provide output temperature of 6 ℃ to 8 ℃. Ammonia machines can be used in surface bulk air coolers (Tuck 2011).

Unlike surface bulk air cooler, underground bulk air cooler is more expensive. However, they have higher positional efficiency as it is close to working area. In these machines, the refrigerant must be non-toxic (Tuck 2011). Figure 16 shows surface and underground bulk air coolers.

(a) (b)

Figure 16) Surface Bulk Air Cooler (a) and Underground Bulk Air Cooler (b) (BBE Group 2018)

Chapter 4
Project statement

4.1 Introduction

In the previous chapter, the heat generation in underground mines have been introduced and the importance of heating sources has been highlighted. Among those sources, the sole source that can be regulated and controlled is the heat production from mining equipment. In fact, heat generation of auto-compression and geothermal rock strata is out of human's hands.

This chapter presents a heating load simulation and cooling demand estimation for two different scenarios: electric mining equipment fleet scenario, and diesel traditional mining equipment fleet scenario. A thorough analysis and comparison will be presented, putting forward the need for change in the current mining industry toward a more sustainable future and enhanced health and safety regulations.

In this thesis, the worst-case scenario is considered to estimate the capital and operating costs. The reason is to consider the conservative design and safety point of view which is accounted by the mining industries.

4.2 Project Specifications

The studied mine is a conceptual underground mine located in the Northern hemisphere of America. The following is the principal assumptions in the project of this mine:

1. The conceptual underground mine consists of 4 mining horizons at work simultaneously. Zone first is located 1600 m below surface. Zone second is located at 2100 m below surface. Zone 3 and 4 are located at 2700 m below surface.
2. The fresh air will be coursed in shaft #1 with a diameter of 24 feet (7.32 m). Shaft #1 provides fresh air to the working area in first two zones. The fresh air for zone 3 and 4 will be provided by a downcast air raise located at the bottom of zone 2 in the diameter of 12 feet (3.66 m). The polluted air is going back to surface by shaft #2, an exhaust shaft with 22 feet (6.71 m) in diameter. A schematic of this mine is shown in Figure 17.

3. The hydraulic diameter of the tunnel is assumed 5.07 meters.
4. To keep the temperature safe for the workers who are working in active areas, a design wet bulb temperature of 28 °C is considered (Refer to section 3.4.1)

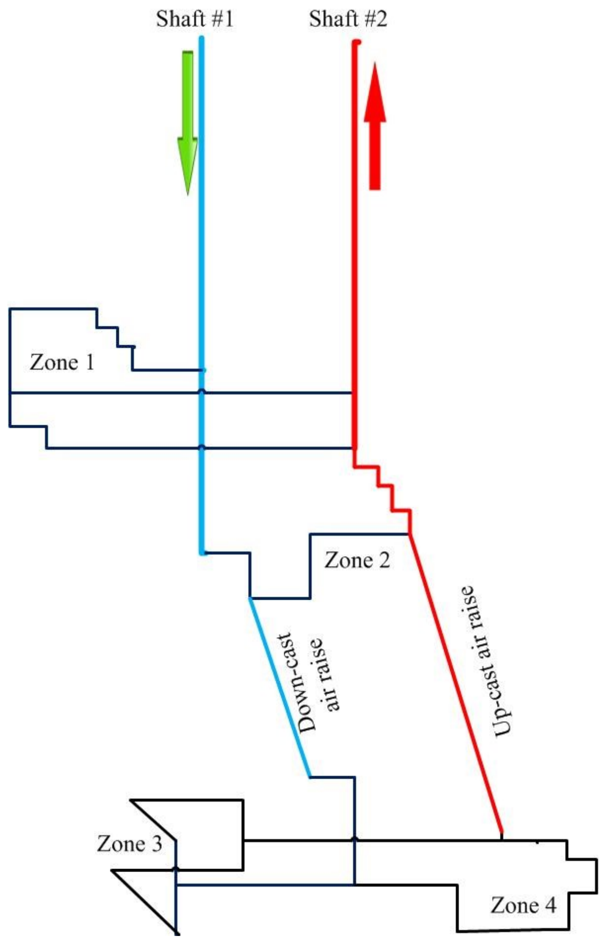

Figure 17) Schematic of the Conceptual Underground Mine Ventilation Plan

4.3 Materials Handling

The materials handling has been calculated to meet the specific daily production rate. The details of this materials handling have been given in Table 12.

Activity	Equipment	Power (hp)	Power (kW)	Utilization Factor	Quantity (1st Zone)	Quantity (2nd Zone)	Quantity (3rd Zone)	Quantity (4th Zone)
Production	6yd LHD	279	208	100%	1	1	1	1
	Emulsion Loader	138	103	40%	1	1	1	1
	Jumbo	160	119	100%	0	0	1	1
	Production Drill	160	119	40%	1	1	0	0
	Toyota Jeep	128	95	40%	1	1	1	1
Development	jumbo	160	119	40%	0	1	1	1
	Toyota Jeep	128	95	40%	0	1	1	1
	Bolter	154	115	40%	0	2	1	1
	6yd LHD	279	208	100%	0	1	1	1
	Backhoe	46	34	40%	0	1	1	1
Ramp	36t Truck	400	298	100%	2	2	2	2
	Boom Truck	201	150	40%	1	1	1	1
	Scissor Truck	147	110	40%	1	1	1	1
	Mobile Drill	160	119	40%	0	0	1	1
	Fork Lift	46	34	40%	2	2	1	1
	Grader	146	109	40%	1	0	1	1
	Toyota Jeep	128	95	40%	4	1	2	2

Table 12) Assumed Materials Handling

4.4 Ventilation for Diesel Equipment Scenario

4.4.1 Regulation Used in Airflow Requirement for Diesel Scenario

Based on the Occupational Health and Safety regulation in Canada, the airflow requirement for an underground mine which uses diesel equipment must be at least $0.06\ m^3/s$ to the maximum of $0.092\ m^3/s$ per kW of engine's power rate (Subsection 183.1(3) of Reg. 854, (OH&S, Mines Safety and Inspection Regulations 1995)).

In this thesis the minimum value of $0.075\ m^3/s$ air volume flow requirement (An average between minimum and maximum airflow requirement in Ontario) for each kW of diesel engine (159 cfm for each horsepower) has been applied to dilute gas emissions from diesel engines in underground mines. The reason for considering this value is based on (Stinnette 2013) and considering a conservative environmental situation for underground mines.

4.4.2 Airflow Calculation for Diesel Scenario

To calculate the required amount of airflow in active areas in underground mines for diesel scenario, $0.075\,m^3/s$ based on average requirements in Ontario's regulation (refer to section 4.4.1) is multiplied by adjusting[1] kW of diesel equipment operate underground that are given in Table 12. Thereafter, 15% leakage has been added to the subtotal calculated airflow of production and development activities on each zone. Then, 15% as a primary leakage and another 15% as a design allowance to the planned active mining levels to prevent recirculation are added to the total required airflow. At the end, the total calculated airflow should be calibrated based on local density. This density is taken as an average density in each zone from VentSIM software. Table 13 shows the calculated airflow required for each zone based on governmental regulation which are calibrated in local density. Refer to Chapter 8, Table 30 to Table 33, for detailed calculations.

[1] $Adjusted\ power\ rate\ (kW) = Number\ of\ equipment \times Power\ rate\ \times Utilization\ factor$

Zone	Air flow requirement in m^3/s for Diesel Scenario
Zone 1	154.6
Zone 2	170.3
Zone 3	181.0
Zone 4	181.0
Total Air Required	686.9

Table 13) Airflow Requirement in Each Zone for Diesel Scenario

4.5 Ventilation for Electric Equipment Scenario

4.5.1 Methodology to Estimate Airflow Requirement for Electric Machines

To calculate airflow requirements for electric engines, it is assumed that the thermal efficiency of diesel and electric engines are 35% and 90% respectively. Then, the adjusting factor to calculate air flow required for electric engines should be:

$$Adjustable\ factor = \frac{Thermal\ efficiency\ of\ diesel}{Thermal\ efficiency\ of\ electric} = \frac{0.35}{0.90} = 0.39 \quad \text{(Equation 43)}$$

So, the required airflow for electric engines can be determined:

$$Required\ airflow\ for\ each\ kW\ of\ electric\ engine = 0.39 \times 0.076 = 0.03\ m^3/s \quad \text{(Equation 44)}$$

After considering the safety factor of 20% for air leakage and mine development, the required airflow for electric engines should round up to $0.036\ m^3/s$ which equals to 76 $cfm\ per\ hp$.

Although we roughly estimate an average volume flow of air for electric engines from diesel engine regulation, but as it is mentioned (section 1.3), there is one regulation in the world which is ruled by Australian government for underground mines which use electric engines and it says, the minimum velocity of $0.25\ m/s$ air must provide in active area where vehicles or locomotives powered by electricity are used (Regulation 9.34, Electric Vehicles Underground, P. 199 (OH&S, Mines Safety and Inspection Regulations 1995)).

4.5.2 Airflow Calculation for Electric Scenario

In this thesis, an assumption has been considered regarding to the materials handling for electric engines. It is assumed that the materials handling, and production rate does not change based on time loss due to charging or battery swap. Detailed calculation of airflow for electric engines is given in Table 34 to Table 37, located in Appendix A. Like in diesel airflow calculation, power rate should be adjusted. As electric engines are smaller than diesel engines (30% smaller) (Kerai and Halim 2012), so a factor of 0.7 is multiplied by diesel engine power rate to get the electric power rate. Thereafter, the calculated electric power rate is multiplied by its utilization factor to achieve the adjusted power rate. Then, estimated required airflow for each kW of electric power rate is multiplied by the adjusted power rate to get the airflow requirement in each zone. Like diesel engine airflow calculation, 15% leakage for activities in each zone (production and development), 15% as a primary leakage and another 15% as design allowance are added to the total required airflow. At the end, the total calculated airflow is corrected based on the local density simulated by VentSIM software.

Table 14 shows the calculated airflow required for each zone.

Zone	Air flow requirement in m^3/s for Electric Scenario
Zone 1	64.12
Zone 2	57.21
Zone 3	60.82
Zone 4	60.82
Total Air Required	243

Table 14) Airflow requirement in each zone for electric scenario

4.6 Ventilation Design

4.6.1 Methodology

The methodology for the ventilation design and numerical modelling can be summarized:

1- Airflow requirement coursed on the surface is estimated (Based on regulation)

2- A critical ventilation loop with AutoCAD is modeled (based on giving assumptions and mine heat loads)
3- The AutoCAD model is imported into the VentSIM ventilation software.
4- The friction factors of tunnel and shaft, as well as shock losses are imported to the software,
5- The total fan pressure is calculated by the software.
6- The shaft power required for the fan is introduced.

4.6.2 Auto-CAD Modeling

A simplified critical loop as a ventilation design drawing has been modeled in Auto-CAD for preliminary work where all the dimensions were inputted into the shafts, tunnels, ramps… This model was drawn based on the mine ventilation plan (refer to Figure 17). Figure 17 shows the mine ventilation critical loop for the mine.

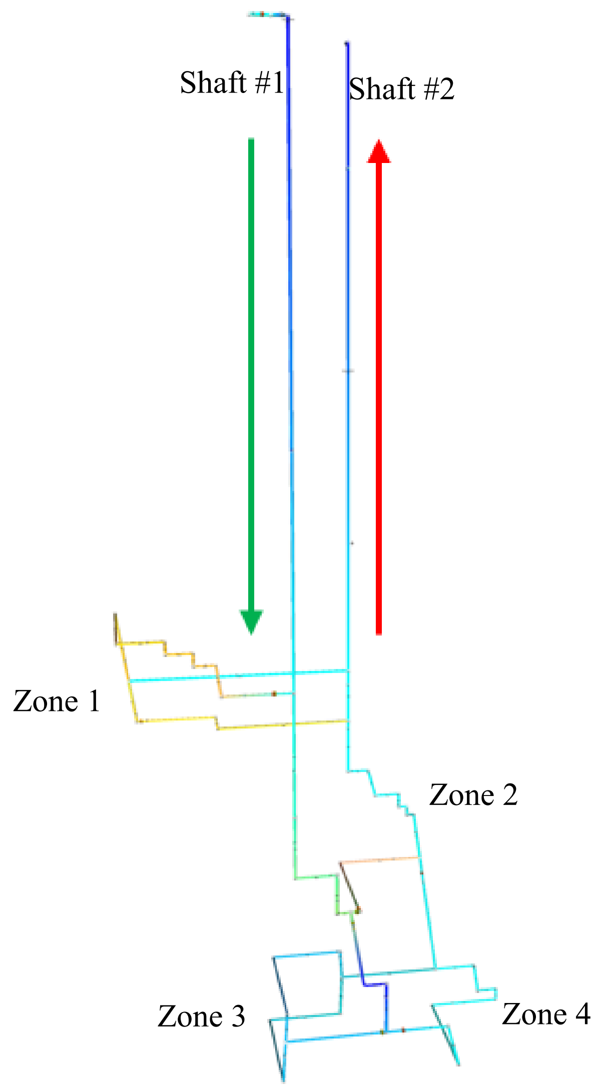

Figure 18) Critical loop ventilation design in Auto-Cad

4.6.3 Ventilation Simulation in VentSIM

The simplified critical loop CAD file is then imported into VentSIM to simulate the airflow in the ventilation network of the mine for both, diesel and electric engine scenario. Before running the airflow simulation, the parameters in Table 38 are imported into VentSIM.

Figure 19 shows the critical loop of the mine ventilation network imported to VentSIM software.

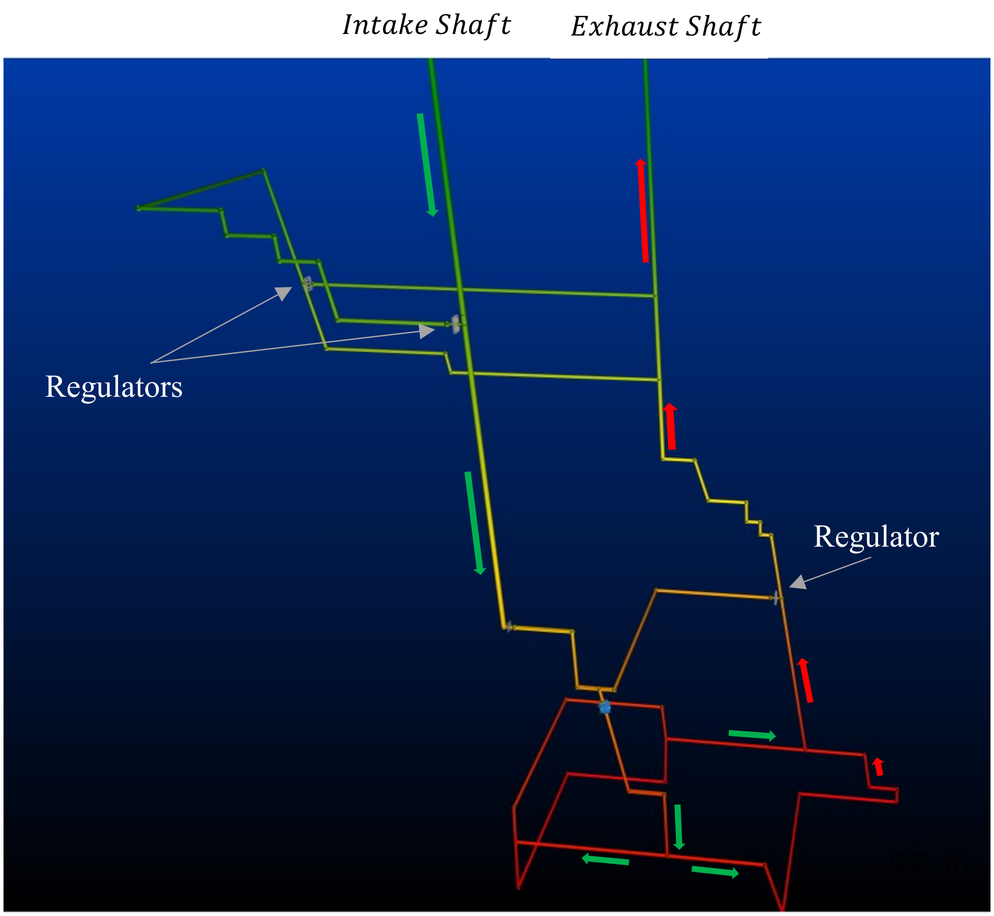

Figure 19) Ventilation Network on VentSIM

In VentSIM software, air is controlled by regulators to distribute the air volume flow in each level based on calculated airflow for diesel and electric engines given in Table 13 and Table 14, respectively. After simulating the network, the fan total pressure and shaft power of the ventilation network is calculated by the software. The fixed flow method has been used to simulate the required fan power in exhaust shaft (Pull system) for both scenarios.

The VentSIM ventilation software also has been running for the electric engine scenario to simulate the fan total pressure and shaft power rate. After simulation, the results show as of Table 15. Also, operating costs have been calculated in this table based on industrial electrical costs in Canada which is $CAD\$ \ 0.09/kW$. To estimate operating costs, it is assumed that the fan is working 24 hours a day, 365 days a year. Due to lack of available field data, the model is not validated with field measurement, however, the model is benchmarked with other ventilation software such as Vuma 3D network.

	Diesel Scenario	Electric Scenario
Total Required Volume Airflow (m^3/s)	687	243
Total Fan Pressure (kPa)	35.6	3.85
Fan Shaft Power (MW)	33.93	1.25
Fan Electrical Power, 85% Efficiency (MW)	35.7	1.31
OPEX (M$/year)	**28.15**	**1.03**

Table 15) Simulation of Fan Operation without Cooling in Two Different Scenarios

The mine characteristic curve for both, diesel and electric scenario, is shown in Figure 20.

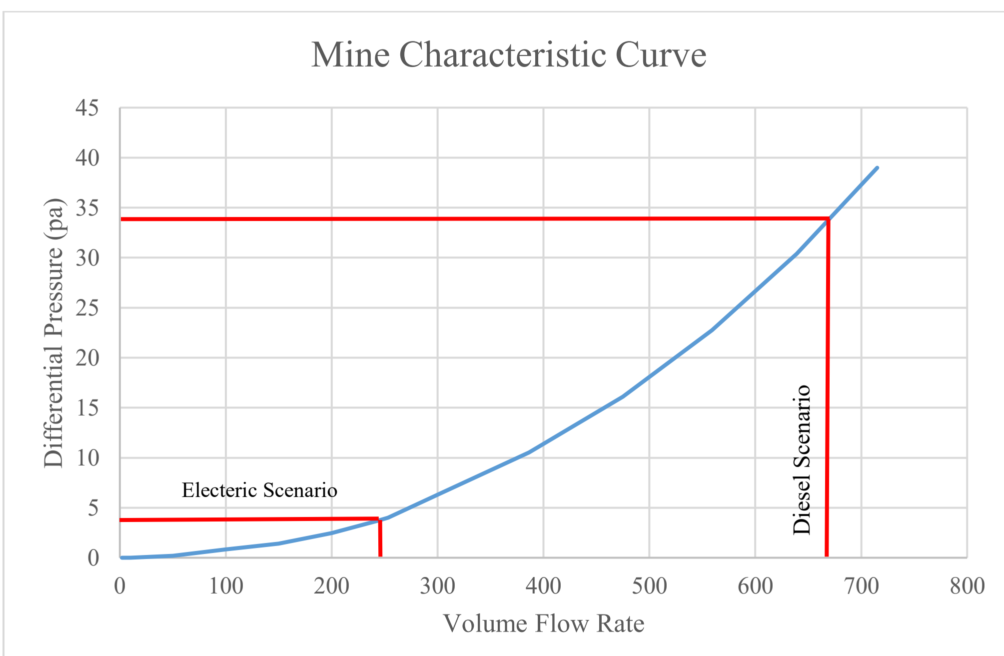

Figure 20) Mine Characteristic Curve

Chapter 5
Cooling Aspect

5.1 Thermal Simulation Results

In this section, three different cooling scenarios are introduced and simulated with the ClimSIM software for the diesel and electric fleet scenarios in order to determine the minimum cooling requirements to provide the previously stated wet bulb temperature of 28 ℃ (refer to section 3.4.1).

The cooling scenarios are: 1) solely surface cooling, 2) solely underground cooling, and 3) combination of surface and underground cooling.

The mine heat loads are determined by adding up the contributions of heat from major sources such as the equipment, auto-compression, and rock strata.

The auto-compression and rock strata heat loads are simulated by the VentSIM software which is given in Table 17.

The equipment heat load for the diesel scenario is calculated in Equation 45. The factor 2.83 is the total heat produced by the equipment per kW (calculated in section 3.3.2.4).

$$Q_{Total} = 2.83 \times power\ (kW) \times (\%\,used/100) \times 1000\ W/kW \quad \text{(Equation 45)}$$

As it is mentioned in section 3.3.2.5, the rated power of the electric engine is considered as a sensible heating load. The results of a heat load comparison between electric and diesel equipment has been done and can be seen in Table 16. Table 14 shows that electric engines create 75% less heat than diesel engines. For additional details concerning the estimation of the heating load of diesel/electric equipment, refer to Appendix A.

Zones	Equipment Heat Load Production (kW)	
	Diesel Scenario	Electric Scenario
Zone 1	3,564	881.4
Zone 2	4,246.8	1,050.5
Zone 3	4,444.4	1,099.3
Zone 4	4,444.4	1,099.3
Total Equipment Heat Load	**16,699.0**	**4130.5**

Table 16) Comparison of Heat Load Production (in kW) in Different Scenarios

As Table 16 shows, the heat load from auto-compression in the diesel scenario is more compared to the electric scenario. By coursing more airflow into the intake shaft, the total differential pressure will be increased and by referring to Equation 26, auto-compression will be increased. Also, it is obvious that a higher volume airflow takes heat from rocks by increasing the Nusselt number and thereafter, the rate of heat transfer between the air and rock will be increased. Thus, the more heat will extract from rocks. That is why heat loads from rock strata in diesel scenario is less than electric scenario. If one compares the total heat loads given in Table 17 for both scenarios, there is 82% less heat with the electric scenario than diesel scenario.

Major Heat Loads	Diesel Scenario (kW)	Electric Scenario (kW)
Rock Strata	424.45	1,168.1
Auto-Compression	20,870.2	6,631.2
Equipment	16,699.0	4130.5
Total	**37,994**	**11,930**

Table 17) Major Heat Loads in Two Different Scenarios

5.2 Sizing the Cooling Plant

5.2.1 Solely Surface Cooling Scenario

In this scenario, the input temperature of surface cooling has been considered as 18 °C (refer to Table 38). The output cooling wet bulb temperature is considered in a range of 4 °C to 16 °C for both diesel and electric scenarios. Then, the maximum wet bulb temperature in underground mine is simulated by ClimSIM software. At the end, the results are plotted for different cooling output temperature and the maximum wet bulb temperature at stopes. Then the optimum output temperature for the surface cooling is selected and the cooling duty is estimated. A sensitivity analysis has been done to estimate the optimum output temperature where the lowest required cooling duty should meet the environmental regulation/standards which is the maximum 28 °C wet bulb temperature at stopes in this thesis.

5.2.1.1 Surface Cooling for Diesel Fleet Scenario

Figure 21 shows the output cooling wet bulb temperature range of 4°C to 16°C for the diesel scenario. It indicates that by lowering the output wet bulb temperature in the cooling system, the maximum wet bulb temperature at stopes decreases. However, it requires more cooling power at the surface.

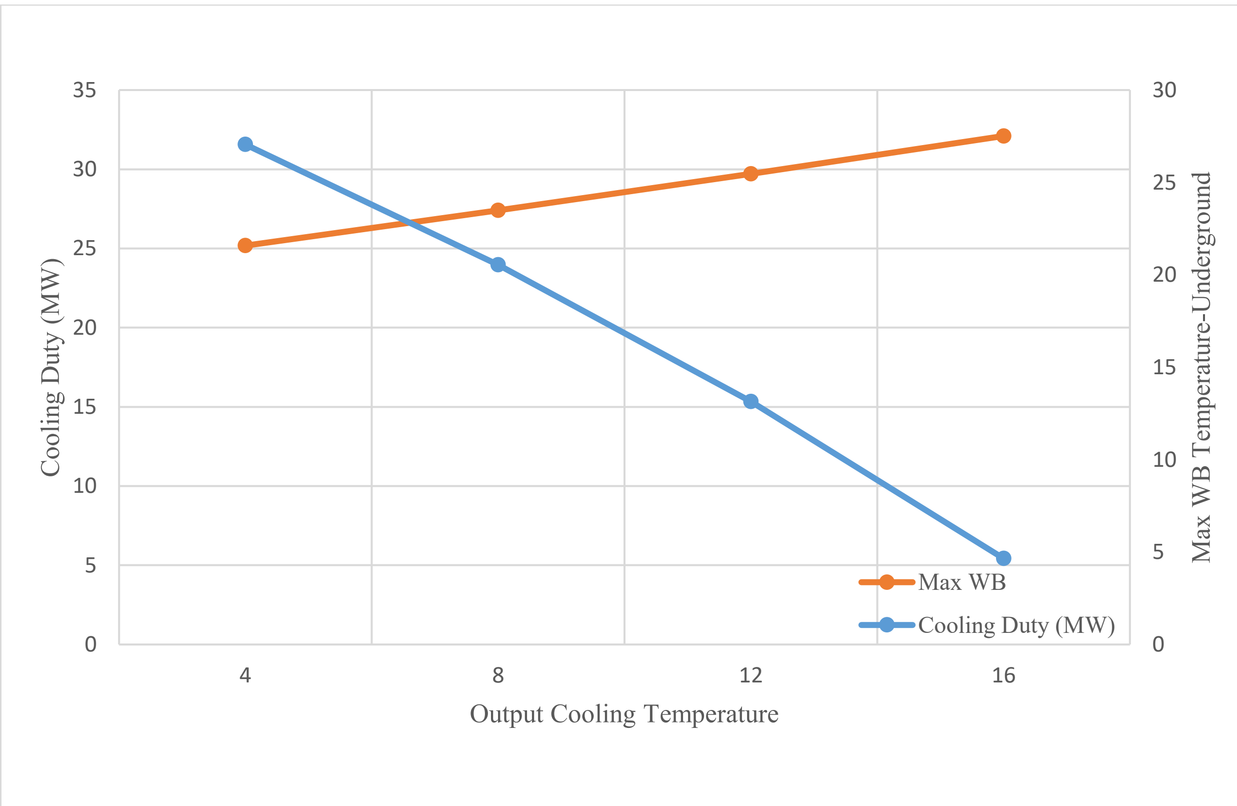

Figure 21) Different Output Temperature for Surface Cooling Plant- Diesel Scenario

Figure 22 shows the maximum wet bulb temperature in the shaft after introducing different output temperatures to the surface cooling system. As it is seen, the maximum wet bulb temperature never exceeds the designed wet bulb in the intake shaft. But, this should be investigated in the zones and at stopes where mining employees work.

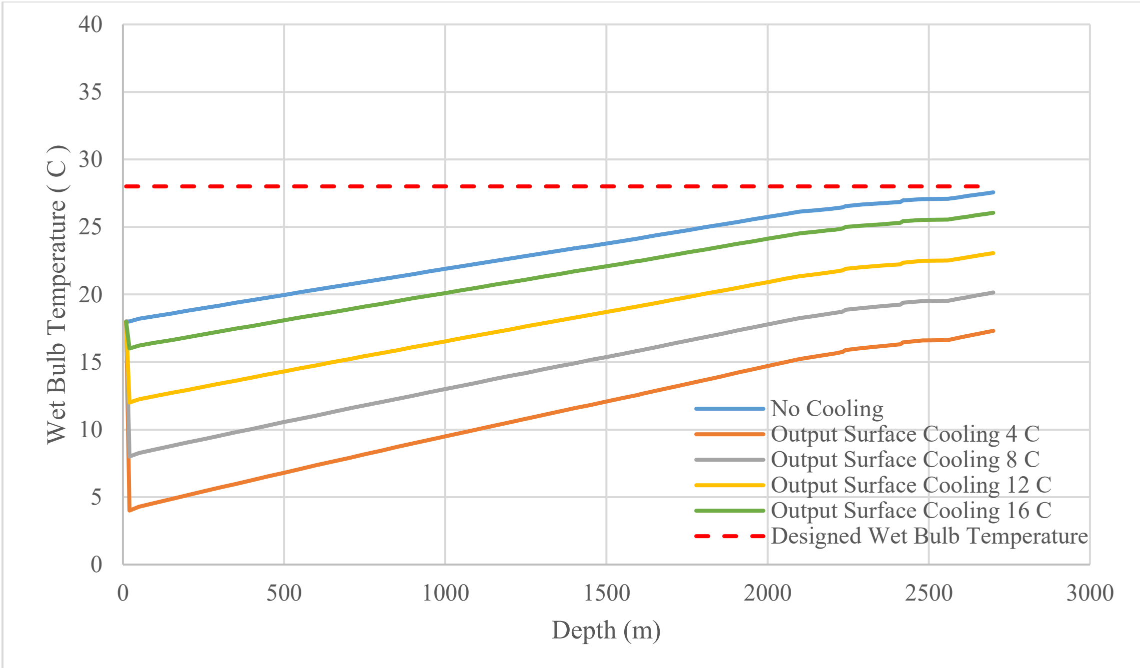

Figure 22) Temperature Distribution in Shaft for Different Ranges of Output WB Temperature - Diesel Scenario

The maximum wet bulb temperature at stopes in each zone was simulated for 4 different ranges of output wet bulb temperatures for surface cooling, and the results were plotted in Figure 23 to Figure 26. As it is seen in these figures, the wet bulb temperature exceeds 28 ℃ in each zone if no cooling is provided. By considering 16 ℃ as the output wet bulb temperature, only Zone 1 can meet the designed wet bulb temperature with other zones exceeding 28℃. An output wet bulb temperature of 12 ℃ only meets the designed criteria for Zone 1 and 2. Thus, like 16 ℃ output wet bulb temperature, 14 ℃ output wet bulb temperature is not feasible as a surface cooling for the diesel scenario. As Figure 25 and Figure 26 show, the output cooling wet bulb temperature of 8 ℃ would be feasible for surface cooling in diesel scenario as it meets the designed criteria. An output wet bulb temperature of 4 ℃ for surface cooling meets the designed wet bulb temperature, but it creates too much cooling duty for diesel scenario and increases capital and operating costs.

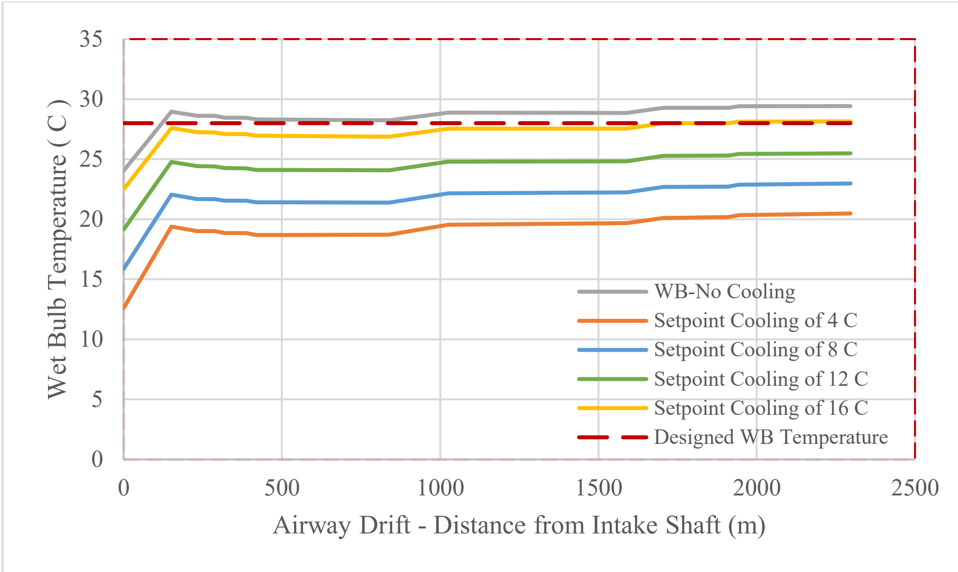

Figure 23) Wet Bulb Temperature Distribution at Zone 1- Solely Surface Cooling Plant – Diesel Scenario

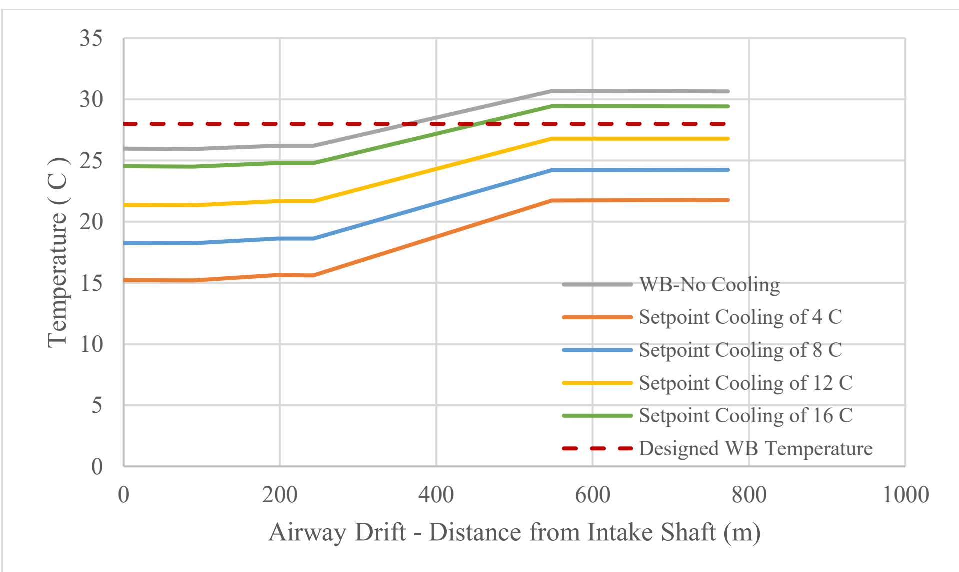

Figure 24) Wet Bulb Temperature Distribution at Zone 2- Solely Surface Cooling Plant – Diesel Scenario

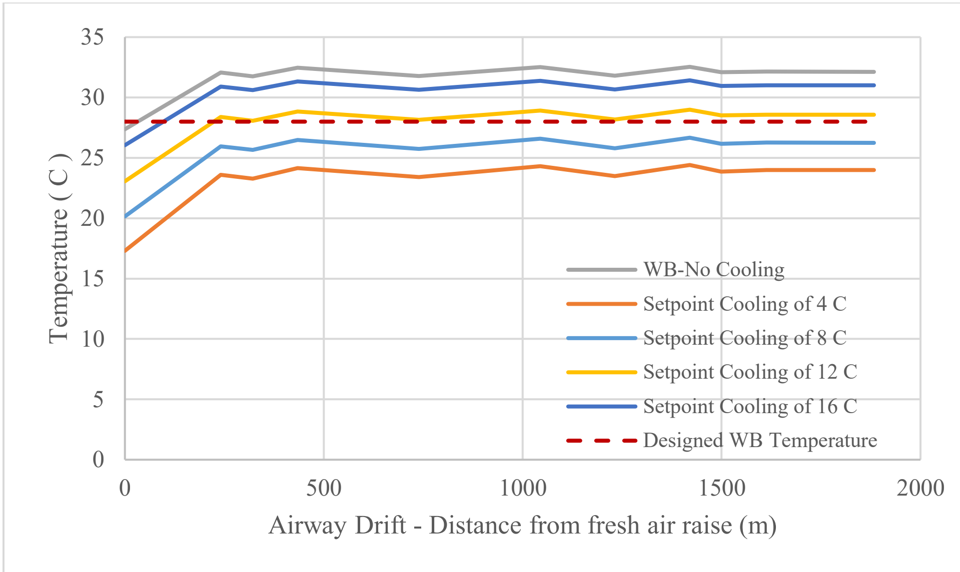

Figure 25) Wet Bulb Temperature Distribution at Zone 3- Solely Surface Cooling Plant – Diesel Scenario

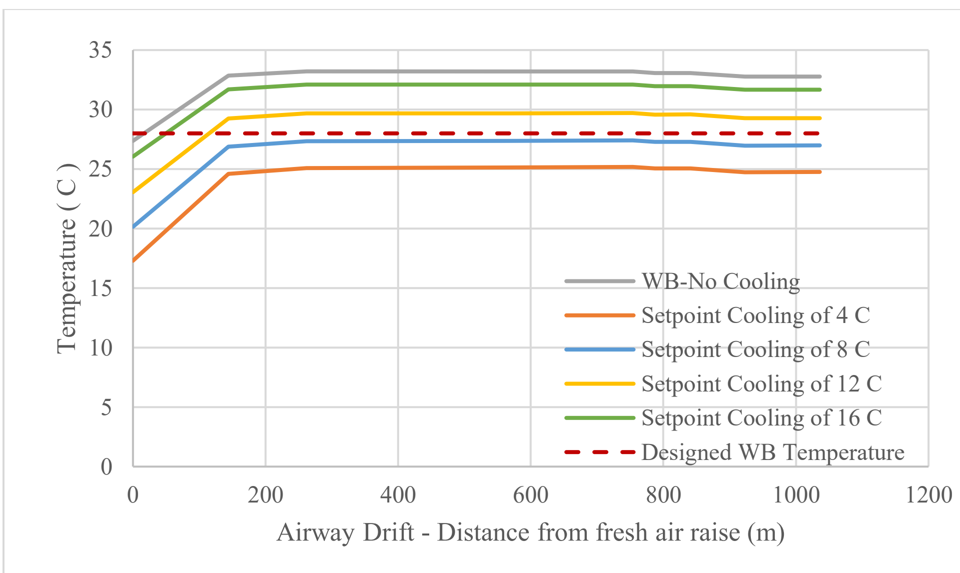

Figure 26) Wet Bulb Temperature Distribution at Zone 4- Solely Surface Cooling Plant – Diesel Scenario

Thus, the surface cooling duty for the diesel engine scenario can be estimated based on an 8℃ output wet bulb temperature for the surface cooling, with the results given in Table 18.

Diesel scenario	Surface Plant
Airflow (kcfm)	1,455.5
Air flow (m^3/s)	686.9
Density	1.187
Mass flow (kg/m^3)	815.35
Inlet Air Wet bulb (℃)	18
Outlet Air Wet bulb (℃)	8
Barometric Pressure (kPa)	101.6
Sigma Heat	
S_{in} (kJ/kg)	49.71
S_{out} (kJ/kg)	24.52
Cooling Duty = $\dot{m} \times (|S_{out} - S_{in|})$ (kW)	**20,539**

Table 18) Estimation of Surface Refrigeration Requirement for Diesel Scenario

As Table 18 shows, by applying 20.5 MW of cooling at the surface for the diesel scenario, the wet bulb temperature never exceeds the designed wet bulb temperature.

5.2.1.2 Surface Cooling for Electric Fleet Scenario

Like the diesel scenario, a sensitivity analysis has been done with determination of optimum value of output temperature in surface cooling plant for the electric engine scenario. In this analysis, the output cooling wet bulb temperatures ranged between 4℃ and 16℃, then the maximum wet bulb temperature at stopes had been simulated by ClimSIM software. Figure 27 shows a relationship between output cooling wet bulb temperature, cooling duty, and the maximum wet bulb temperature at stopes for the electric engine scenario.

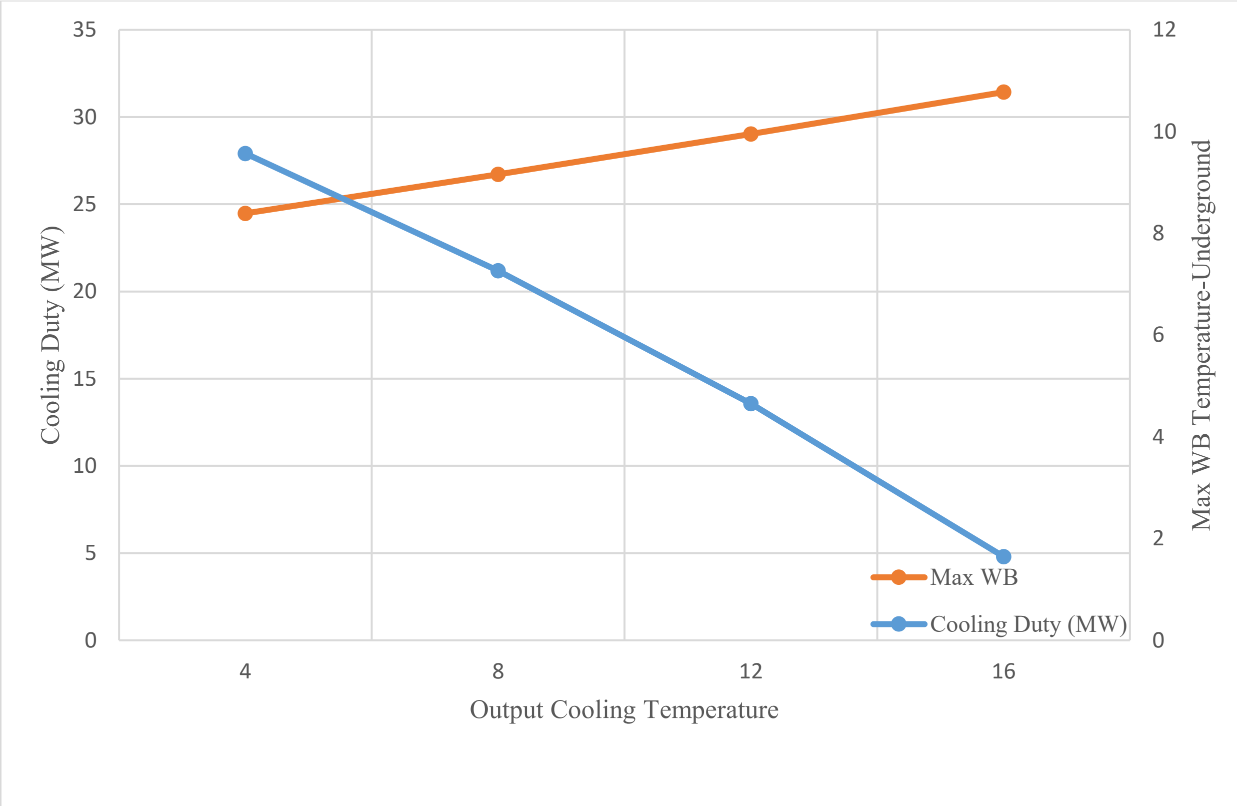

Figure 27) Different Output Temperature for Surface Cooling Plant- Electric Scenario

The wet bulb temperature distribution is simulated in the shaft for four ranges of output cooling wet bulb temperature. The results are shown in Figure 28. In this figure, the wet bulb temperature never exceeds designed wet bulb temperature in the shaft by setting different output wet bulb temperature on the surface cooling.

Figure 29 to Figure 32 shows the wet bulb temperature distribution at Zone 1 to Zone 4. As it is shown in Figure 29, all different ranges of output cooling temperature meet the designed wet bulb temperature. However, Figure 30 shows that output cooling wet bulb temperature of 16 ℃ doesn't meet the design. By looking at the results shown in Figure 31 and Figure 32, it is obvious that the best output cooling wet bulb temperature is 8℃ for surface cooling in electric scenario. In this temperature, the maximum wet bulb temperature at stopes never exceeds 28℃. Output cooling wet bulb temperature of 4℃ gives too much cooling on the surface which requires more cooling power on the surface and increases the costs. Thus, by considering 8℃ output wet bulb temperature in the surface cooling, the cooling duty for electric scenario can be estimated in Table 19.

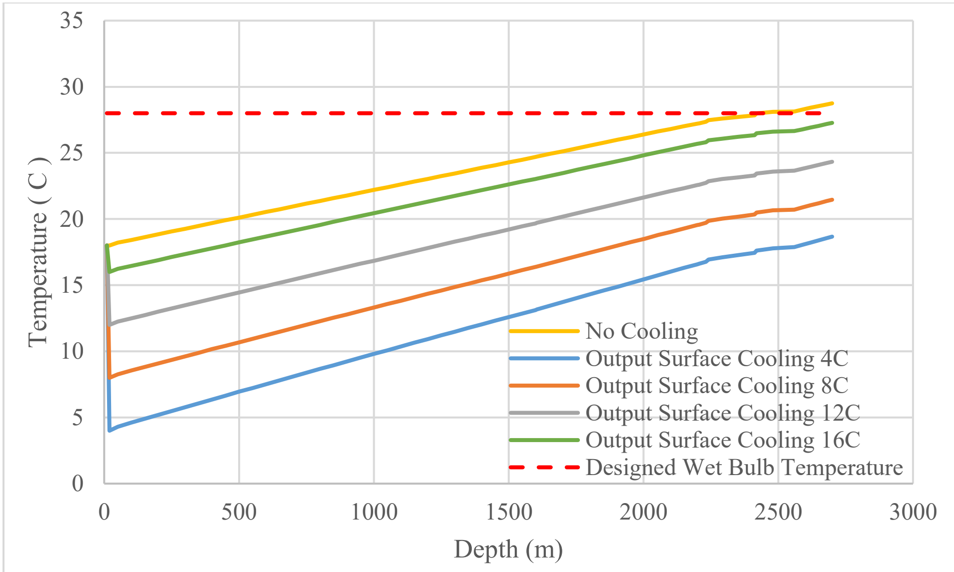

Figure 28) Temperature Distribution in Shaft for Different Ranges of Output WB Temperature - Electric Scenario

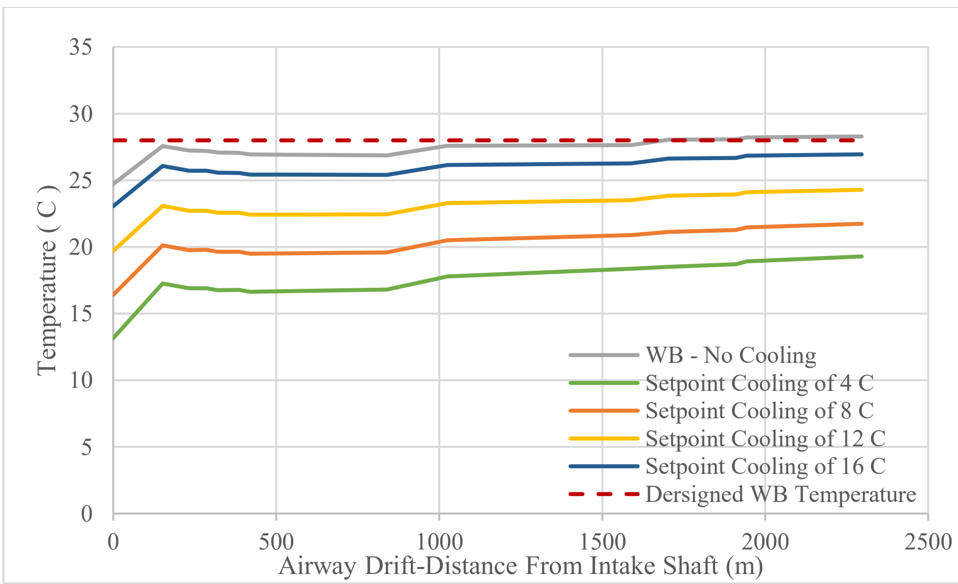

Figure 29) Wet Bulb Temperature Distribution at Zone 1- Solely Surface Cooling Plant – Electric Scenario

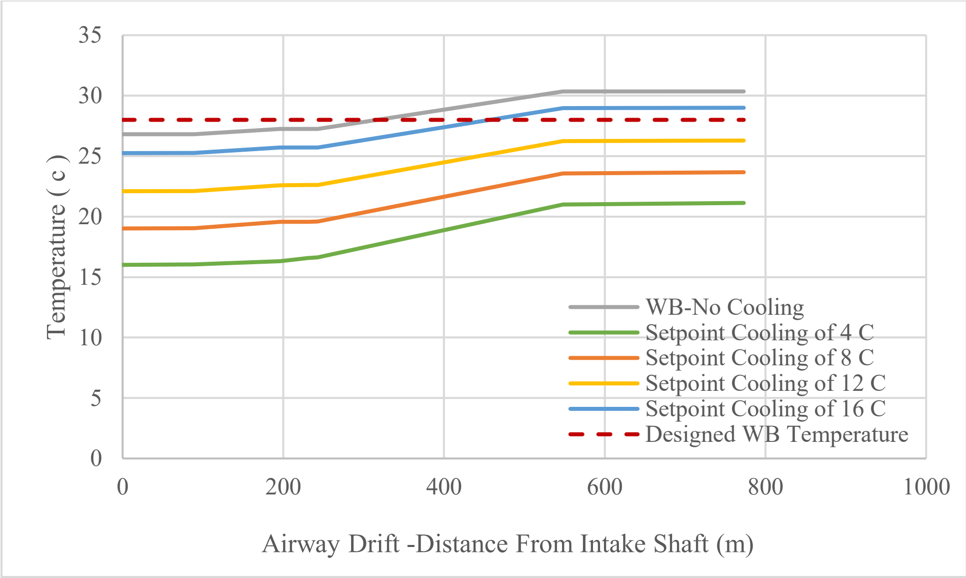

Figure 30) Wet Bulb Temperature Distribution at Zone 2- Solely Surface Cooling Plant – Electric Scenario

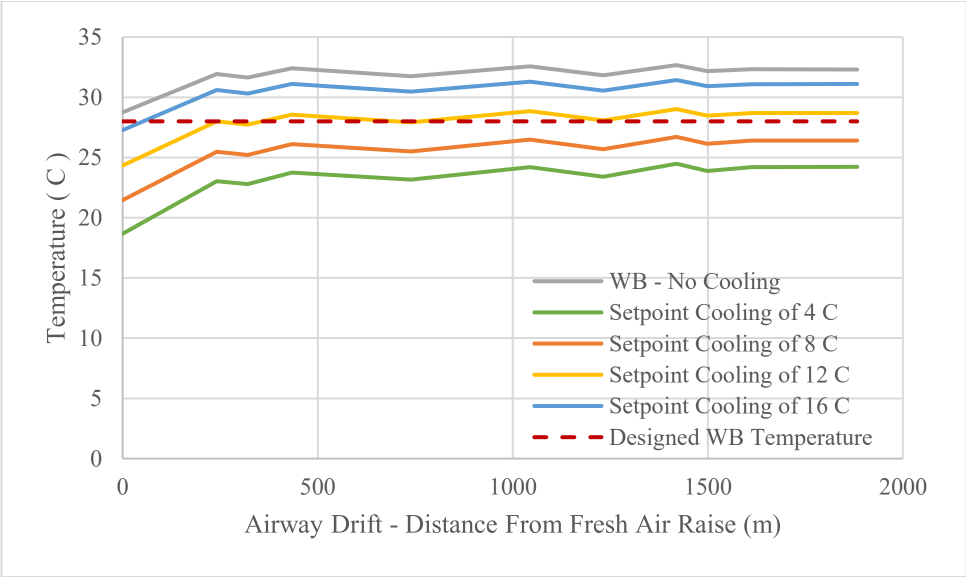

Figure 31) Wet Bulb Temperature Distribution at Zone 3- Solely Surface Cooling Plant – Electric Scenario

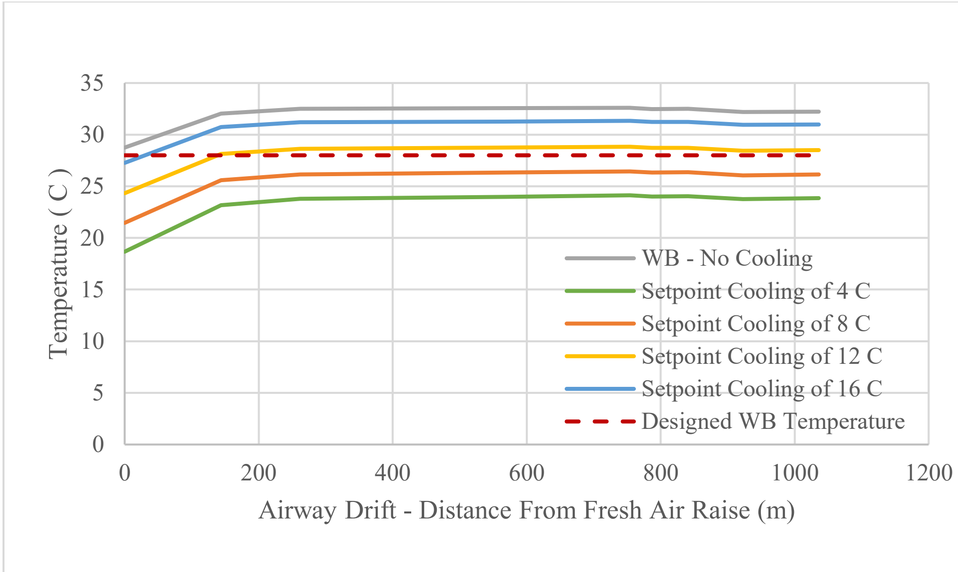

Figure 32) Wet Bulb Temperature Distribution at Zone 4- Solely Surface Cooling Plant – Electric Scenario

Electric scenario	Surface Plant
Airflow (kcfm)	514.9
Airflow (m^3/s)	243
Density (kg/m^3)	1.187
Mass flow (kg/m^3)	288.4
Inlet Air Wet bulb (℃)	18
Outlet Air Wet bulb (℃)	8
Barometric Pressure (kPa)	101.6
Sigma Heat	
S_{in} (kJ/kg)	49.71
S_{out} (kJ/kg)	24.52
Cooling Duty $= \dot{m} \times (\lvert S_{out} - S_{in} \rvert)\ \ (kW)$	**7,265**

Table 19) Estimation of Surface Refrigeration Requirement for Electric Scenario

As Table 19 shows, a 7.3 MW sole surface cooling plant is required to keep the wet bulb temperature below 28℃ at stopes for the electric scenario.

5.2.2 Solely Underground Cooling Scenario

5.2.2.1 Underground Cooling for Diesel Fleet Scenario

With underground cooling, the first two zones will be ventilated without cooling. In this situation, the wet bulb temperature in Zone 1 reaches 30℃ and at Zone 2 exceeds 28℃. The heat distribution graphs for Zones 1 and 2 which are shown in Figure 33 and Figure 34, justify that underground cooling is not feasible for diesel scenario in our conceptual mine.

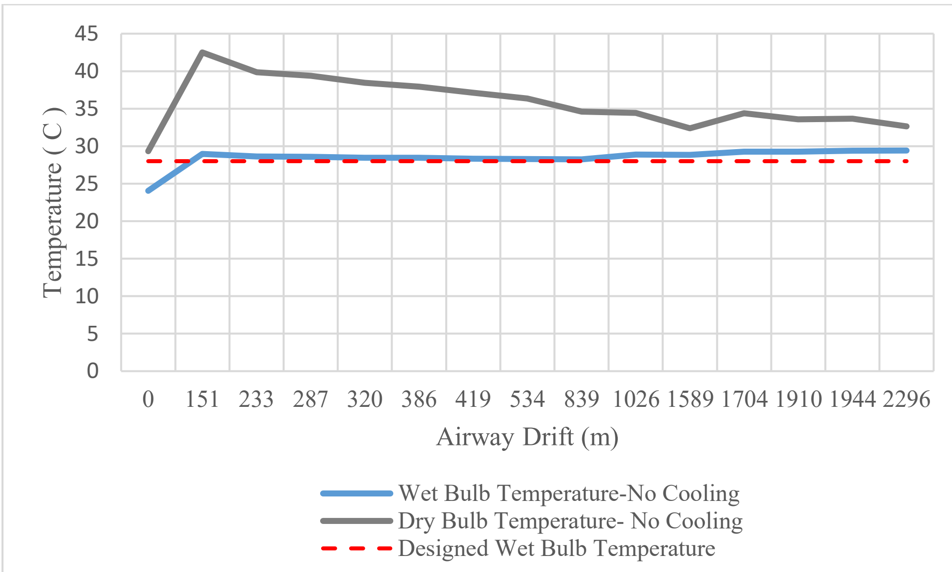

Figure 33) Wet Bulb Temperature Distribution at Zone 1- No Cooling- Diesel Scenario

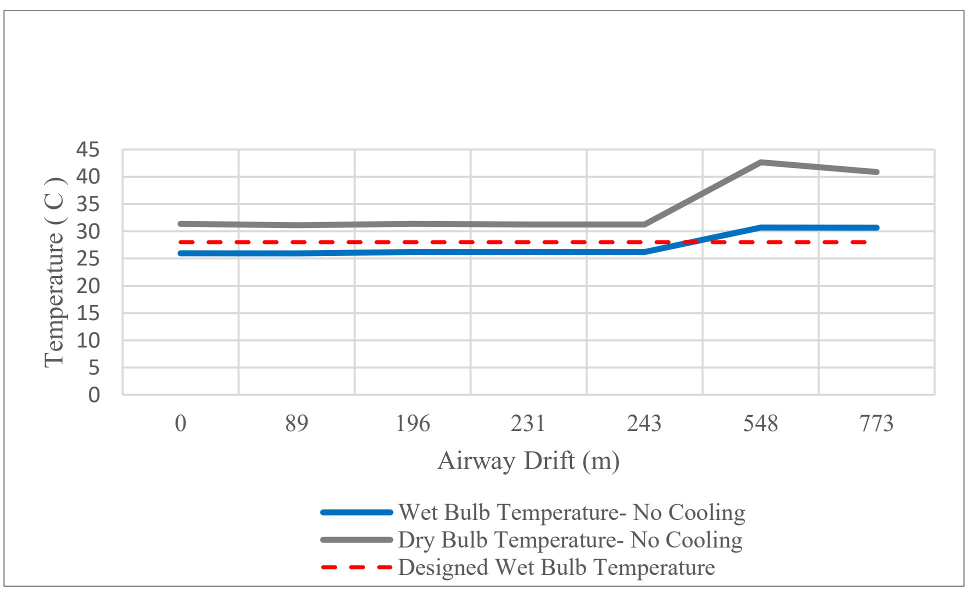

Figure 34) Wet Bulb Temperature Distribution at Zone 2- No Cooling- Diesel Scenario

5.2.2.2 Underground Cooling for Electric Fleet Scenario

Like the underground cooling in the diesel scenario (Section 5.2.2.1), the underground cooling option is not feasible for the electric scenario as well. With no surface cooling, the wet bulb temperature exceeds 28℃ at Zone 1 and Zone 2 (c.f. Figure 35 and Figure 36).

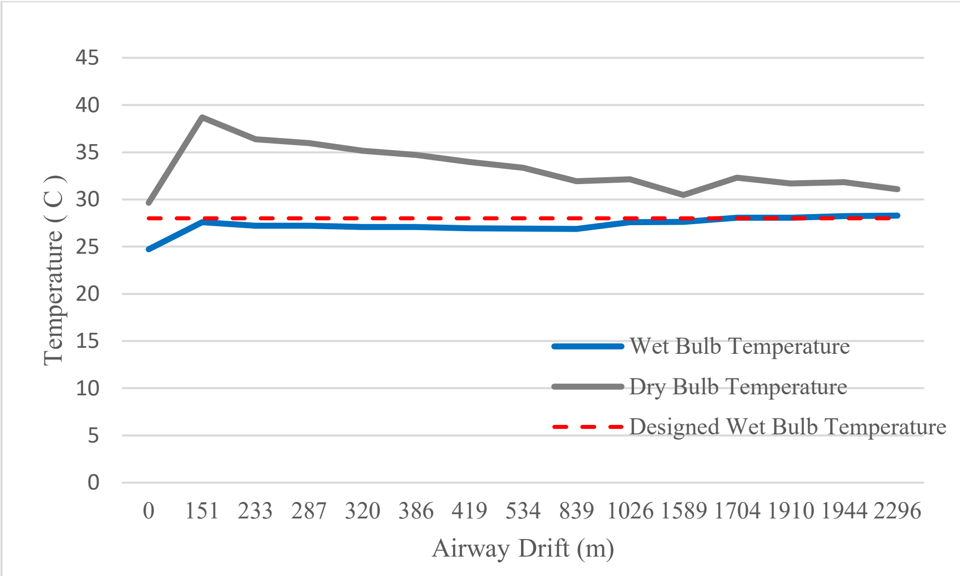

Figure 35) Wet Bulb Temperature Distribution at Zone 1- No Cooling- Electric Scenario

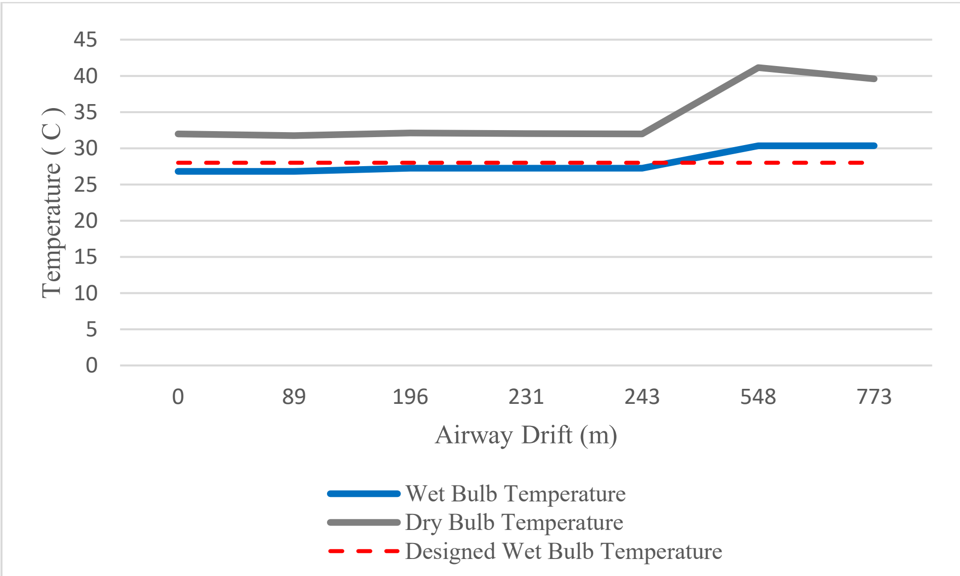

Figure 36) Wet Bulb Temperature Distribution at Zone 2- No Cooling- Electric Scenario

5.2.3 Combination of Surface and Underground Cooling Plant Scenario

The combination of surface and underground cooling is the tricky part here as the output wet bulb temperature of the surface cooling plant has an effect on the input wet bulb temperature of the underground cooling plant. To do this, a range of output wet bulb temperatures from the surface cooling plant is simulated to get the input wet bulb temperature for the underground cooling plant. Then, a range of output wet bulb temperatures for the underground system is calculated. The range for the surface cooling plant varies between 4℃ and 16℃ , and between 10℃ and 20℃ for the underground cooling plant. For each output wet bulb temperature on the surface cooling plant, the ranges of output wet bulb temperature are simulated for the underground cooling plant. Then, a basic optimization has been done and the results will be discussed. The simulation is done with ClimSIM software and the figures have been plotted with Matlab software.

5.2.3.1 Surface and Underground Cooling for Diesel Machines Scenario

In order to evaluate the required combination of surface and underground cooling plant power in the conceptual mine for the diesel engine scenario, one should simulate the maximum wet bulb temperature at working stopes by utilizing the combined surface and underground cooling output wet bulb temperature. Figure 37 shows the maximum wet bulb temperature at work stopes by varying the surface and underground output cooling wet bulb temperature.

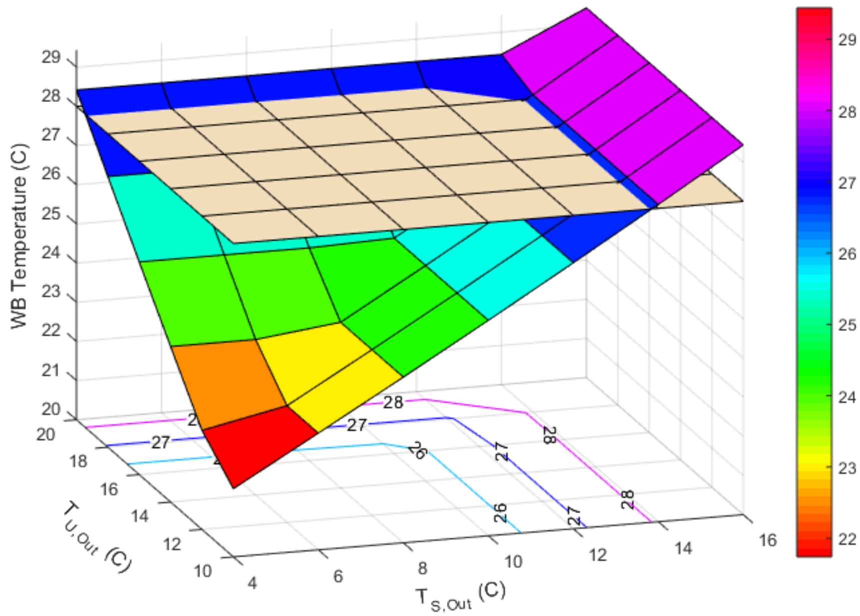

Figure 37) Maximum Underground Wet Bulb Temperature by Utilizing Different Surface and Underground Output Temperatures-Diesel Scenario

In Figure 37, the wet bulb temperature of 28 ℃ is indicated by a surface in cream color. The contour lines in Figure 37 show that by utilizing an output wet bulb temperature above 14℃ on the surface cooling and above 19 ℃ in underground plant, the maximum wet bulb temperature exceeds 28 ℃ at stope. Thus, output wet bulb temperature of 14℃ and above are not feasible and should be excluded from the study.

In the case of the surface cooling plant, Figure 38 indicates the cooling duty by varying the surface and underground cooling output temperature. As Figure 38 shows, the cooling duty changes from 10 to 25 MW for different output wet bulb temperature.

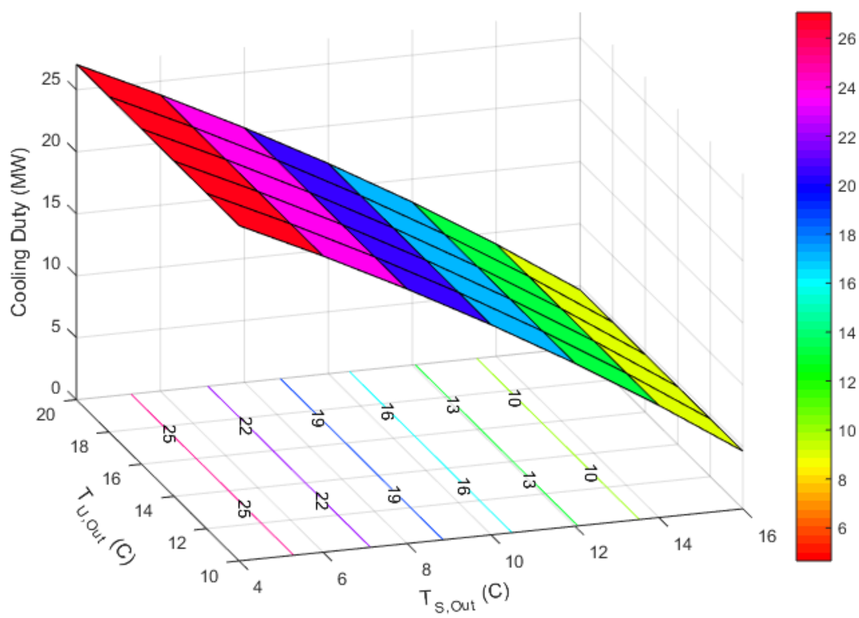

Figure 38) Surface Cooling Duty with Variation of Surface and Underground Output Temperature- Diesel Scenario

In the case of an underground cooling plant, Figure 39 shows a 3D view of the underground cooling duty, output wet bulb temperature on the surface, and output wet bulb temperature underground.

In Figure 39, the red zone shows zero cooling from the underground plant. It means that if a surface cooling with output wet bulb temperature of 4°C to 8°C is utilized, the input temperature of the underground cooling plant is cooler than the output setpoint of 18°C to 20°C. So, the underground cooling plant is useless in this range.

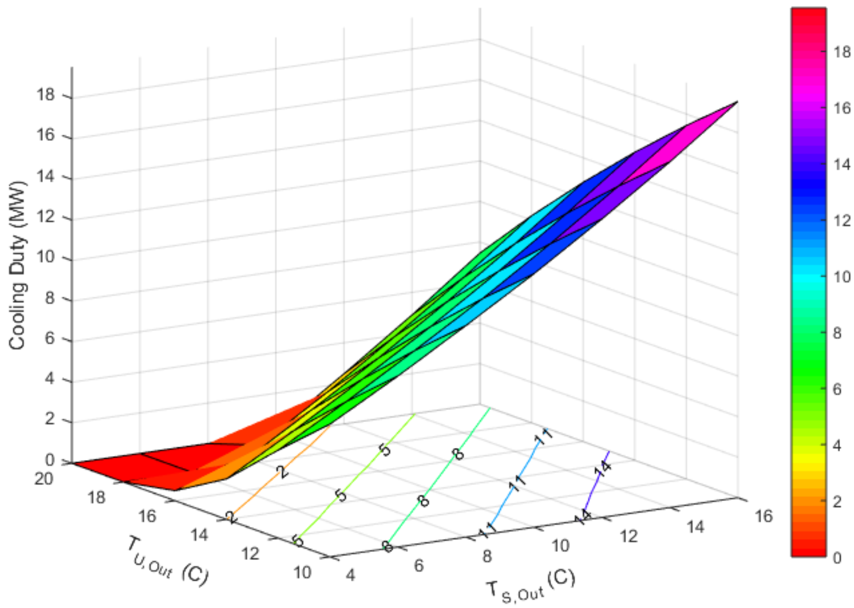

Figure 39) Underground Cooling Duty with Variation of Surface and Underground Output Temperature- Diesel Scenario

Finally, Figure 40 shows the combination of both the surface and the underground cooling plants. As it is justified in Figure 37, a surface output wet bulb temperature above 14°C and above 19°C for underground cooling should be neglected. So, surface output wet bulb temperatures between 4°C and 14°C and underground output wet bulb temperatures between 10°C and 19°C should be considered. In these ranges, the optimum total cooling duty, which should be the minimum in Figure 40, is 16 MW. But if one needs to be more conservative, the total cooling duty between 16 and 19 MW should be considered. So, the surface wet bulb cooling temperature is between 12°C and 14°C and for underground should be in the range of 17°C and 18°C. As Figure 40 indicates, the optimum point for combined surface and underground cooling must be 17.6 MW for surface and underground output wet bulb temperature of 13°C and 18°C , respectively.

These optimum values meet the designed wet bulb temperature, which is plotted in Figure 37.

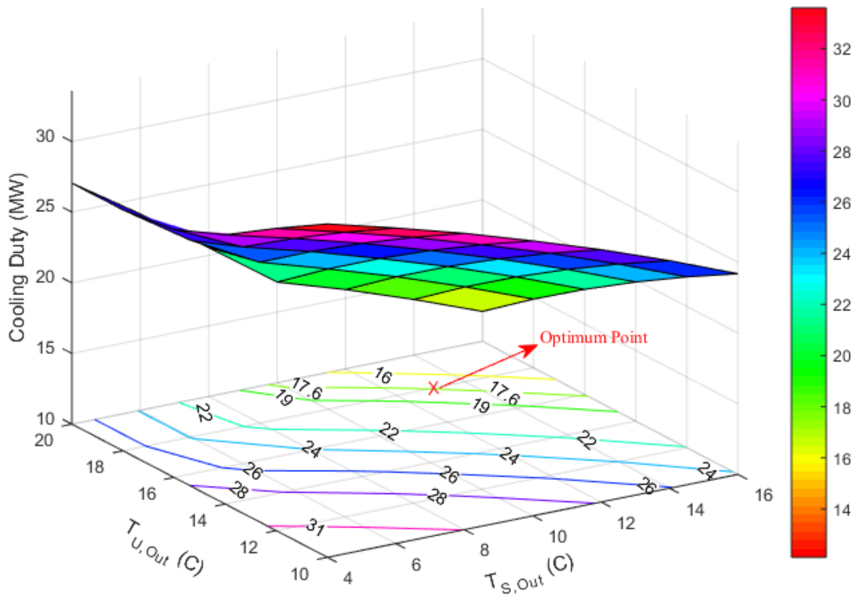

Figure 40) Combination of Surface and Underground Cooling with Variation of Surface and Underground Output Temperature-Diesel Scenario

Diesel scenario	Surface Plant	U/G Plant
Airflow (kcfm)	1461.82	767
Air flow (m^3/s)	689.9	362
Local Density (kg/m^3)	1.187	1.379
Mass flow (kg/m^3)	818.9	499.2
Inlet Air Wet bulb (℃)	18	22.45
Outlet Air Wet bulb (℃)	13	18
Barometric Pressure (kPa)	101.6	119.755
Sigma Heat		
S_{in} (kJ/kg)	49.71	57.82
S_{out} (kJ/kg)	36.06	44.91
Cooling Duty $= \dot{m} \times (\lvert S_{out} - S_{in} \rvert)$ (kW)	**11,178**	**6,444.7**
Total Cooling Duty (kW)	**17,622.7**	

Table 20) Estimation of Surface and Underground Refrigeration Requirement for Diesel Scenario

As Table 20 shows, the total cooling duty is estimated to be 17.6 MW.

Figure 41 shows the wet bulb temperature before and after combined cooling in the shaft.

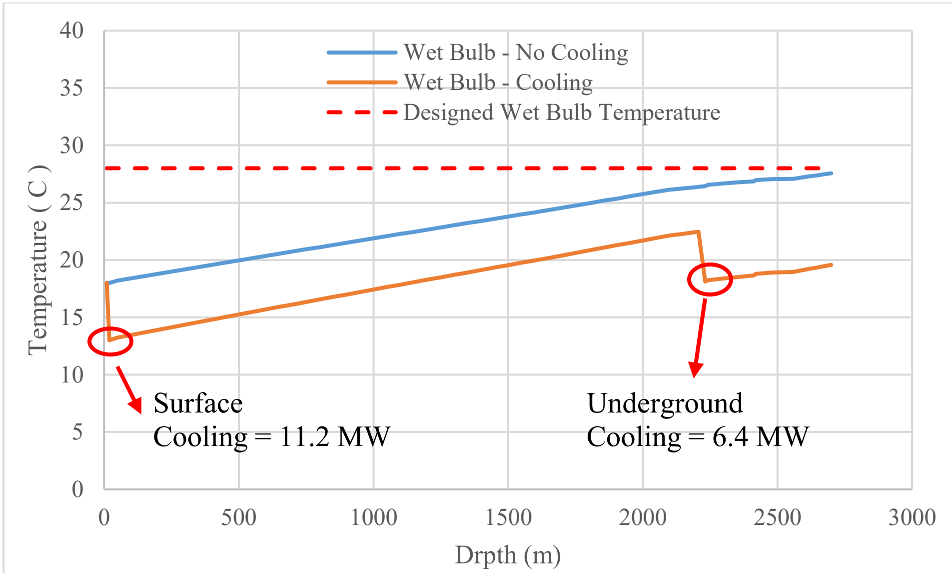

Figure 41) Temperature Distribution in the intake Shaft- Combined Surface and Underground Cooling Plants -Diesel Scenario

Figure 42 to Figure 45 indicates that the wet bulb temperature never exceeds after combined surface and underground cooling plants in the power of 11.2 MW and 6.4 MW, respectively.

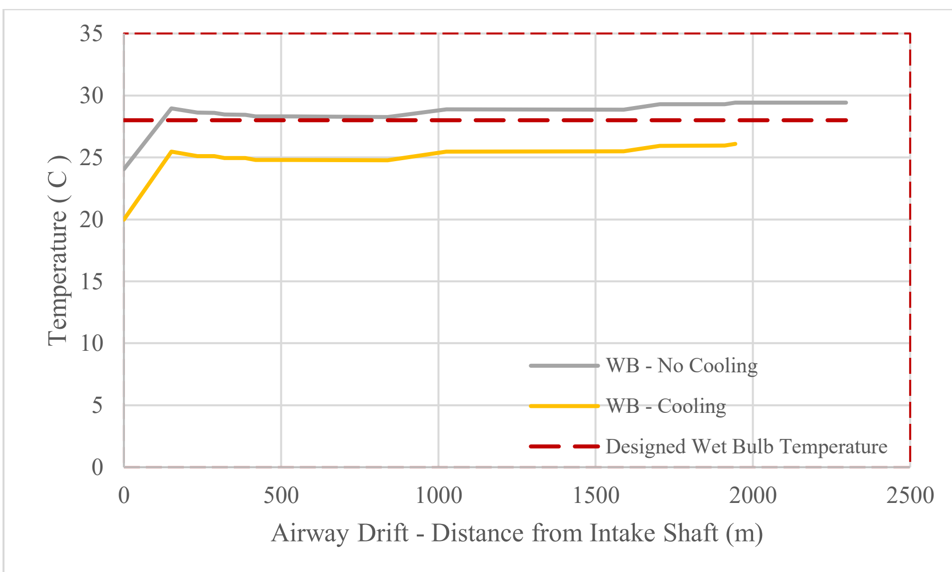

Figure 42) Wet Bulb Temperature Distribution at Zone 1- Combined Surface and Underground Cooling Plants- Diesel Scenario

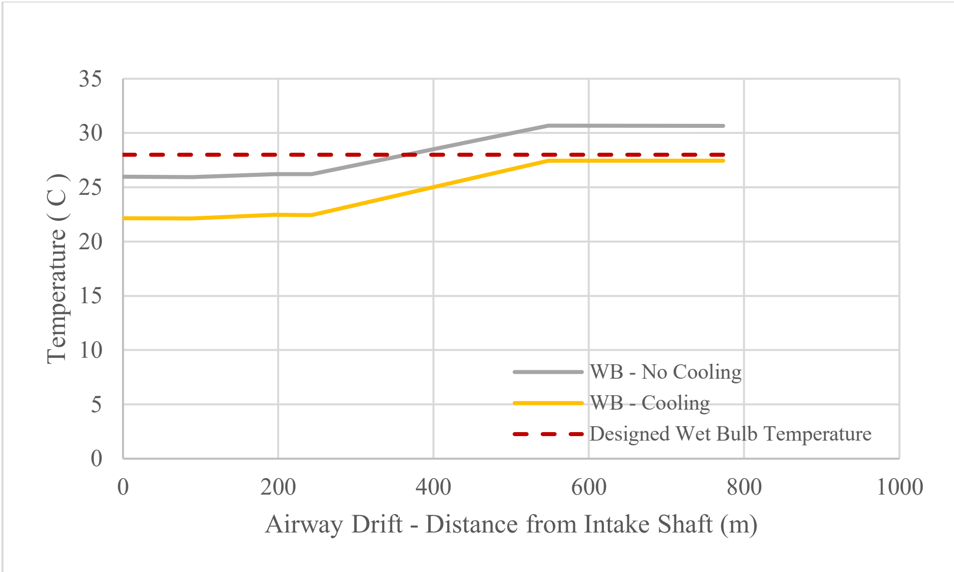

Figure 43) Wet Bulb Temperature Distribution at Zone 2- Combined Surface and Underground Cooling Plants- Diesel Scenario

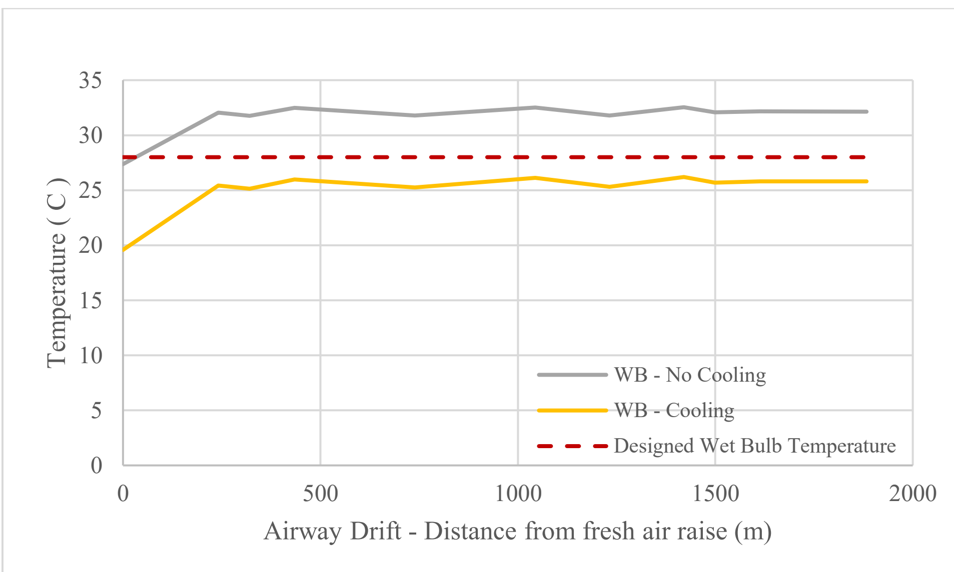

Figure 44) Wet Bulb Temperature Distribution at Zone 3- Combined Surface and Underground Cooling Plants- Diesel Scenario

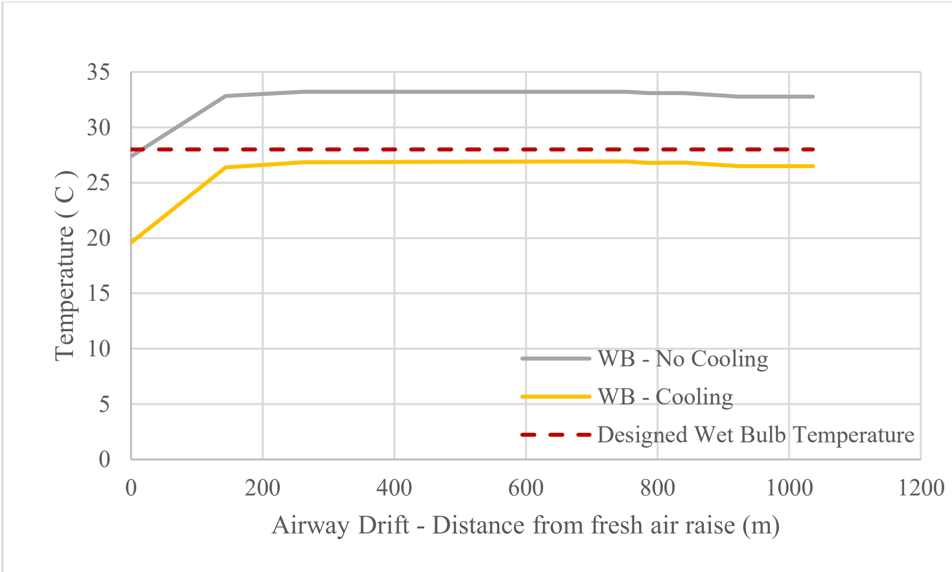

Figure 45) Wet Bulb Temperature Distribution at Zone 4- Combined Surface and Underground Cooling Plants- Diesel Scenario

5.2.3.2 Surface and Underground Cooling for Electric Scenario

The methodology to estimate the surface and underground cooling demand for the electric scenario is the same as the diesel scenario discussed in 5.2.3.1.

As it is shown in Figure 46, the maximum wet bulb temperature at a stope is simulated by the ClimSIM software for different output wet bulb temperatures on the surface and underground.

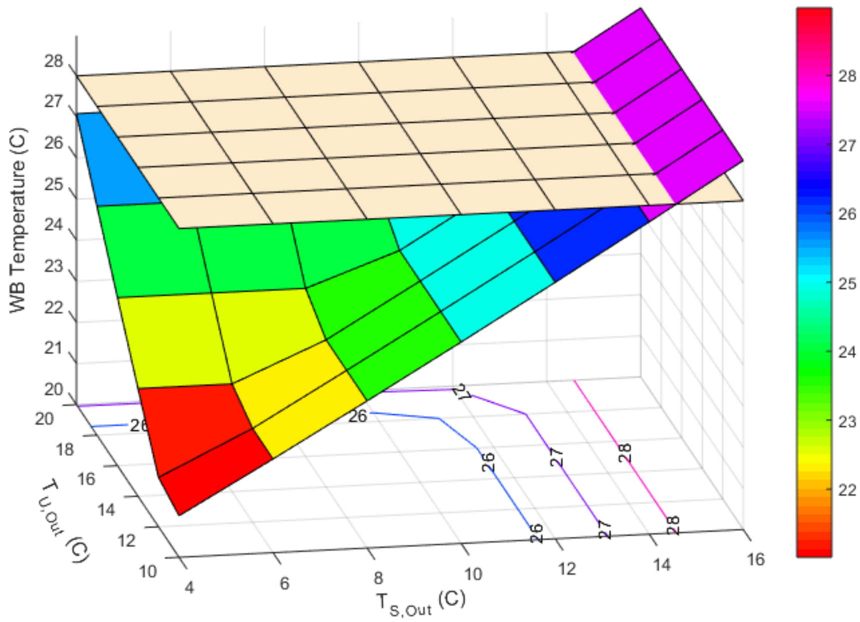

Figure 46) Maximum Underground Wet Bulb Temperature by Utilizing Different Surface and Underground Output Temperature-Electric Scenario

It is obvious that a surface cooling plant output wet bulb temperature above 14.5 °C is not feasible due to exceeding designed wet bulb temperature.

Evaluation of the surface cooling plant for the electric scenario is shown Figure 47. As it is shown, the cooling duty should be considered between 4 to 9 MW for the output temperature below 14.5°C.

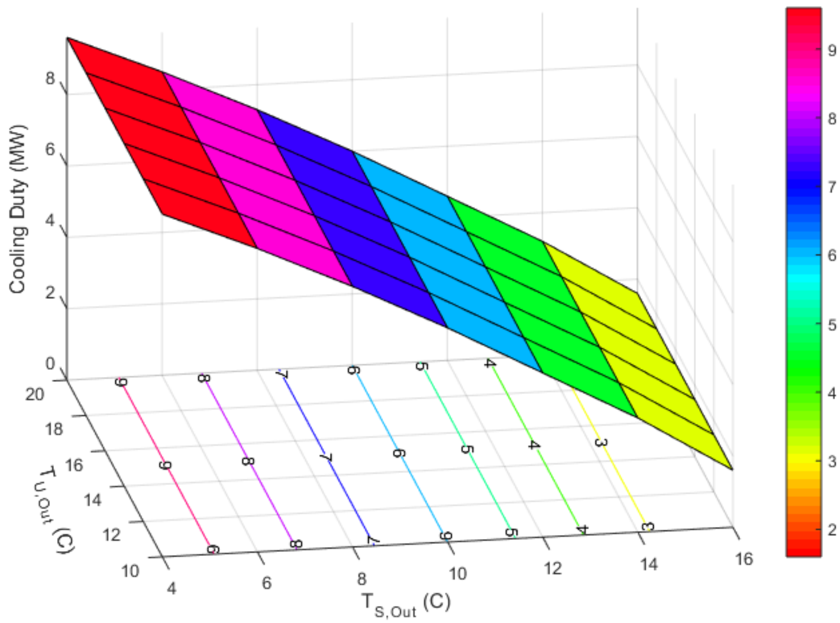

Figure 47) Surface Cooling Duty with Variation of Surface and Underground Output Temperature- Electric Scenario

Figure 48 shows the variation of underground cooling duty is estimated to be between 1 and 6 MW. In Figure 48, if the surface output wet bulb temperature is between 4℃ and 8℃, the underground cooling is useless because the input temperature of the underground cooling system is cooler than the output wet bulb temperature of 18℃ to 20℃ (red zone in Figure 48).

Figure 49 shows the possibility of employing surface and underground cooling plants for electric scenario. In Figure 49, the lowest total cooling capacity of the surface and underground cooling plants is simulated as 6 MW. For 6 MW of cooling duty, the output wet bulb temperature for the surface cooling plant must be set between 12℃ and 14℃, and between 19℃ and 20℃ for the underground cooling plant. Referring to Figure 46, the maximum wet bulb temperature for the total 6 MW cooling duties of surface and underground cooling plants will be around 27℃. By referring to Figure 47 and Figure 48, an underground output temperature of 19℃ and surface output temperature of 13℃ would be recommended. The reason is that more can be saved on capital costs with lower underground cooling power requirements, as underground cooling plants are more expensive than surface cooling plants.

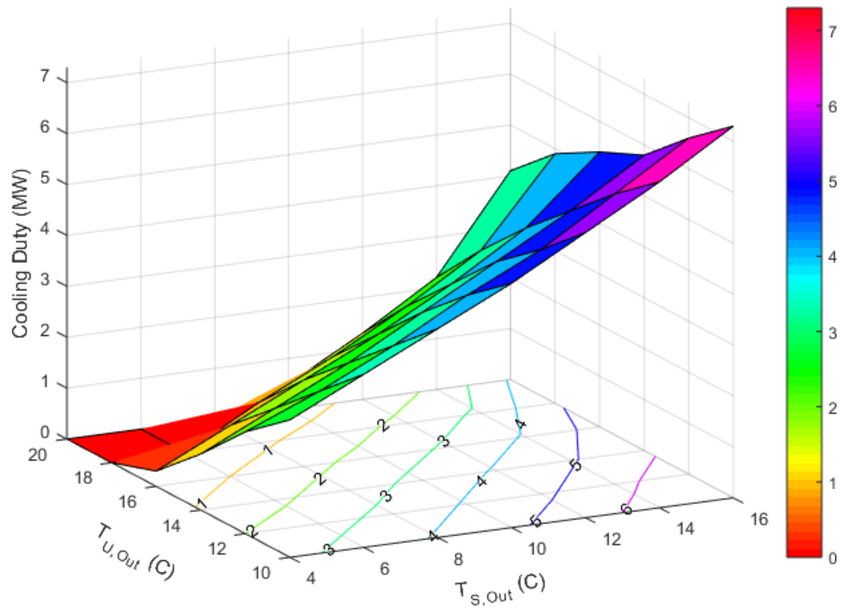

Figure 48) Underground Cooling Duty with Variation of Surface and Underground Output Temperature- Electric Scenario

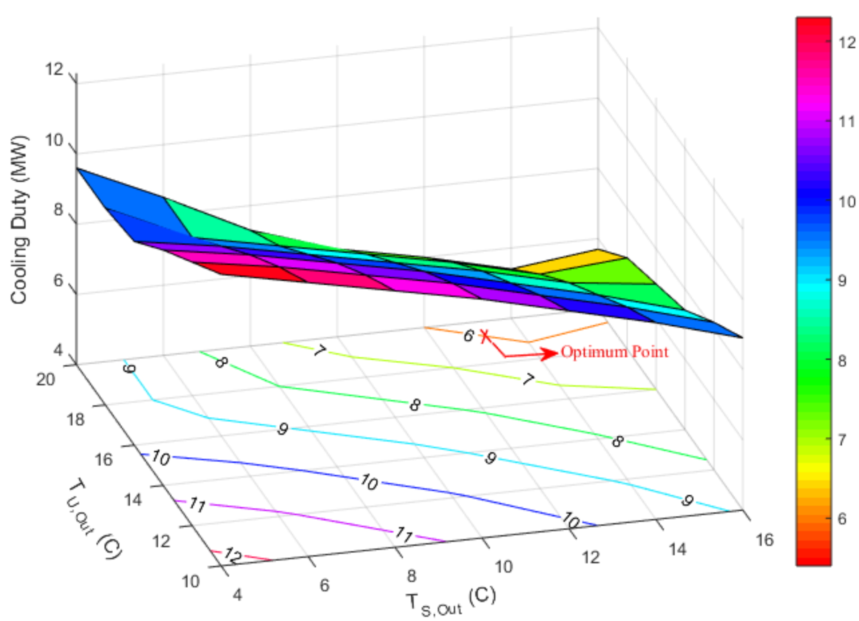

Figure 49) Combination of Surface and Underground Cooling with Variation of Surface and Underground Output Temperature- Electric Scenario

Table 21 indicates the same estimated cooling duty on the surface and underground as is shown in Figure 47, Figure 48, and Figure 49.

Electric scenario	Surface Plant	U/G Plant
Airflow (kCfm)	514.9	257.7
Air flow (m^3/s)	243	121.64
Local Density (kg/m^3)	1.187	1.484
Mass flow (kg/m^3)	288.44	180.5
Inlet Air Wet bulb (℃)	18	20.34
Outlet Air Wet bulb (℃)	13	19
Barometric Pressure (kPa)	101.6	129.127
Sigma Heat		
S_{in} (kJ/kg)	49.71	58.03
S_{out} (kJ/kg)	36.06	45.54
Cooling Duty $= \dot{m} \times (\lvert S_{out} - S_{in\rvert})$ (kW)	**3,937.2**	**2,254.4**
Total Cooling Duty (kW)	**6,191.6**	

Table 21) Estimation of Surface and Underground Refrigeration Requirement for Electric Scenario

Figure 50 shows that the wet bulb temperature never exceeds the designed wet bulb temperature in the intake shaft after combined surface and underground cooling plants in the power of 3.9 MW and 2.25 MW, respectively. Also, Figure 51 to Figure 54 indicate that the maximum wet bulb temperature at the stopes never reaches the designed temperature of 28℃.

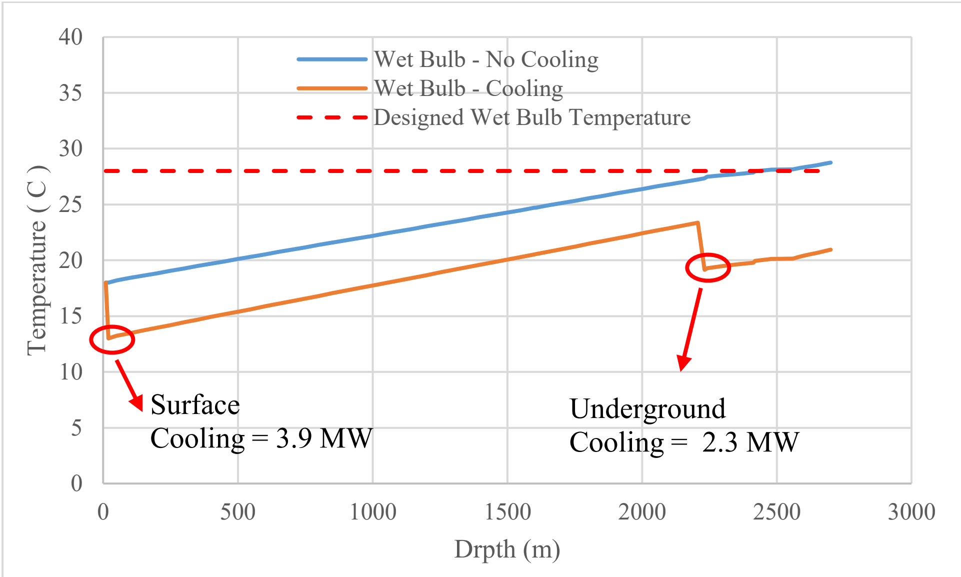

Figure 50) Temperature Distribution in the intake Shaft- Combined Surface and Underground Cooling Plants -Electric Scenario

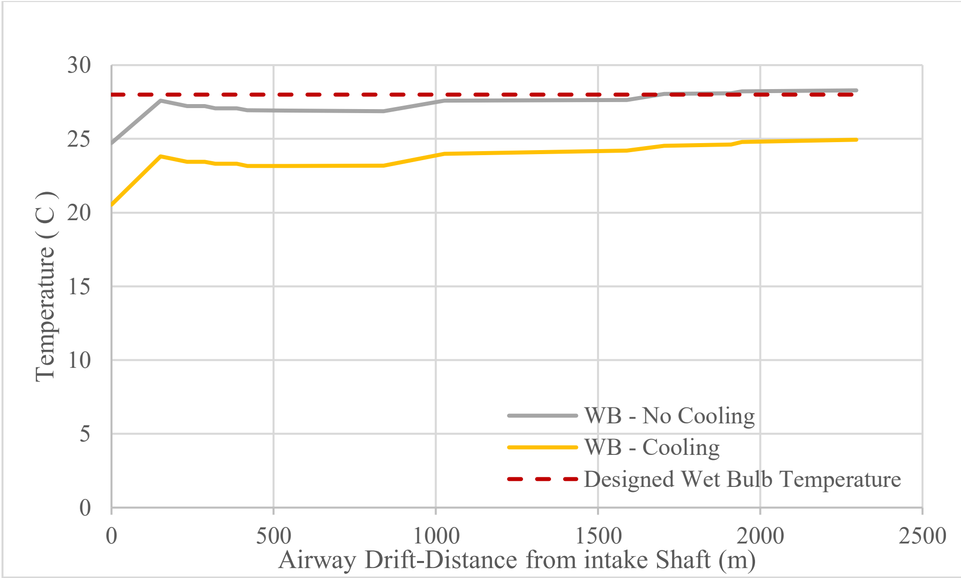

Figure 51) Wet Bulb Temperature Distribution at Zone 1-Combined Surface and Underground Cooling Plants- Electric Scenario

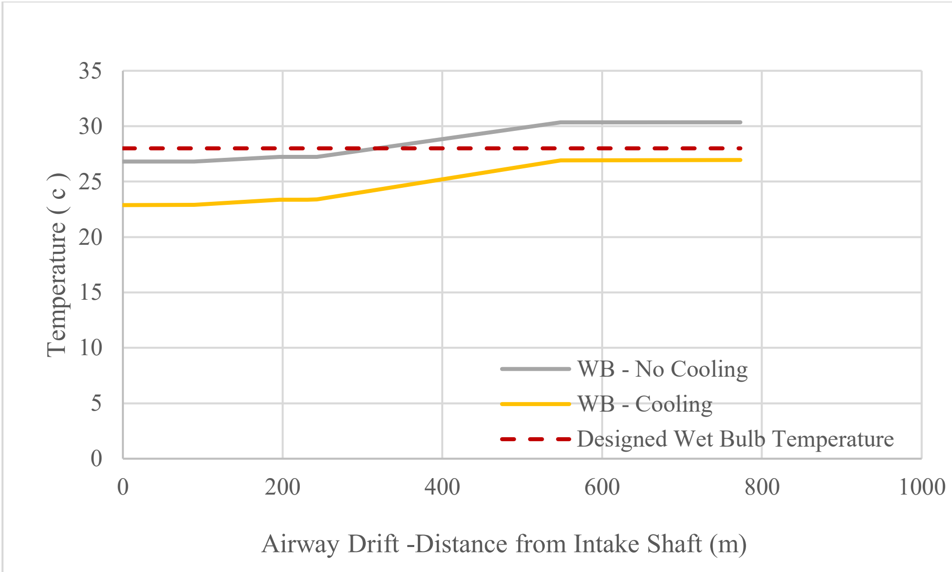

Figure 52) Wet Bulb Temperature Distribution at Zone 2-Combined Surface and Underground Cooling Plants- Electric Scenario

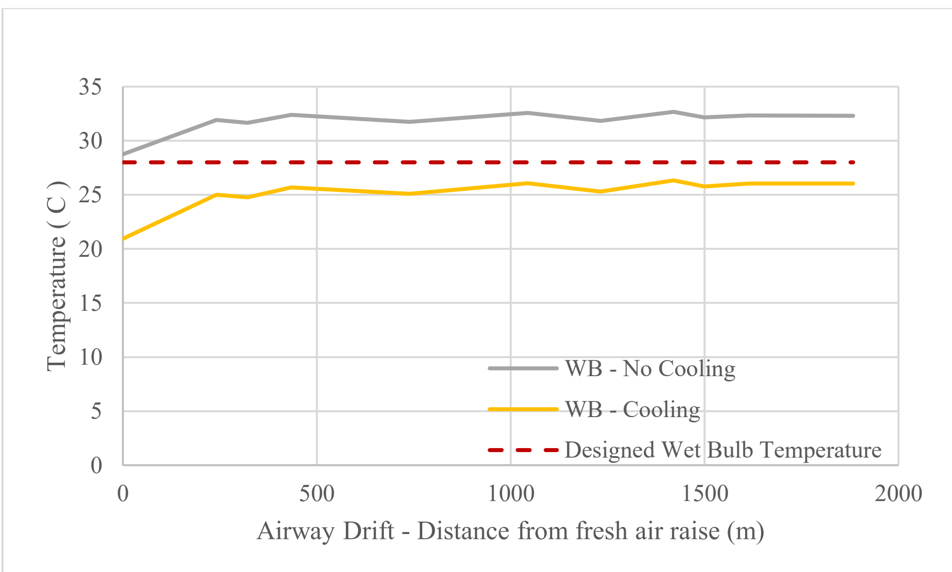

Figure 53) Wet Bulb Temperature Distribution at Zone 3-Combined Surface and Underground Cooling Plants- Electric Scenario

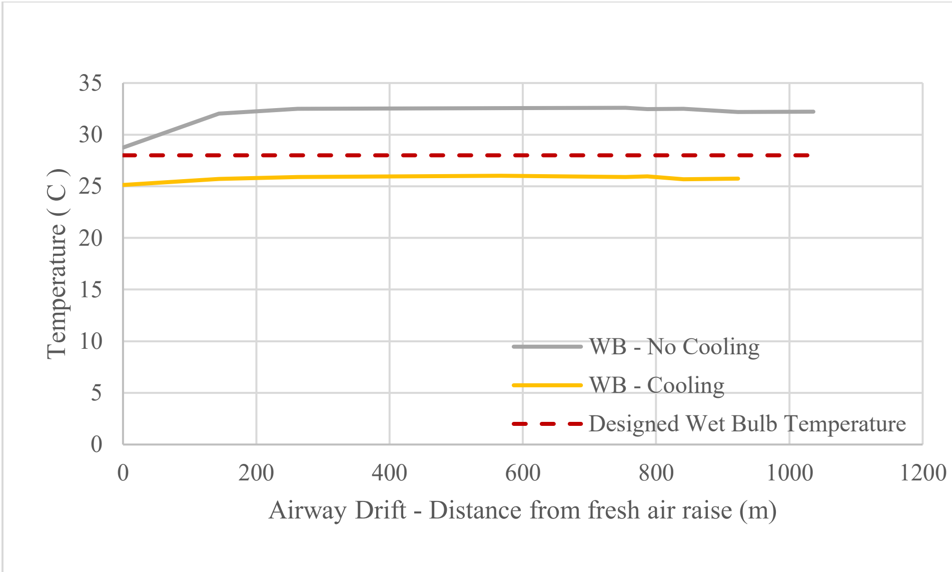

Figure 54) Wet Bulb Temperature Distribution at Zone 4-Combined Surface and Underground Cooling Plants- Electric Scenario

5.3 Conclusion on Sizing the Cooling Plant

The different possibilities of cooling design scenarios are: solely surface, solely underground, and combination of surface and underground. Two different scenarios of diesel and electric equipment were discussed in sections 5.2.1, 5.2.2, and 5.2.3. As it has been concluded in section 5.2.2, the sole underground cooling plant is not feasible for both diesel and electric equipment scenarios. As it is estimated in Table 18, the total surface cooling duty is estimated to be 20.5 MW and 7.3 MW for the diesel and electric equipment scenarios, respectively. For a combination of surface and underground cooling, the total cooling duty is estimated 17.6 MW and 6.2 MW for diesel and electric scenarios, respectively (c.f. Table 20 and Table 21). It is assumed that vapour compression for surface cooling costs $664/kW and $1245/kW for an underground cooling plant as Capital Expenditure (CAPEX) (Trapani, Romero and Millar 2016). Table 22 gives a compared estimation of surface and combined cooling Operating Expenditure (OPEX) for the diesel scenario, by considering a cooling operation of 20 hours a day (2 shift x 10 hours) for 340 days a year.

The electrical cost in Canada is evaluated at 9 Canadian cents for each kWh, which is used in Table 15. Also, the interest rate in Canada is considered 4.5% annually based on bank of Canada's website. In this thesis, the discount rate is considered to estimate the net present value. By definition, the discount rate is a factor applied to a projected income stream in order to discount the value of future benefits and costs to its present value (Collins 2013). In mining industry, the discount rate ranges between 5% and 15%. By assuming 8% discount rate for the mining companies, Table 22 can give an estimation for a designed cooling system over 15 years. In this study, labour, maintenance and renovation costs are assumed as of sustaining costs, which are considered the same for both surface and underground cooling plants (Trapani, Romero and Millar 2016). In Table 22, OPEX over 15 years is calculated from Equation 46 (McPherson 1993).

$$P_o = \frac{S_o}{i}[1 - \frac{1}{(1+i)^n}] \qquad \text{(Equation 46)}$$

Where,

P_o is the present value of OPEX

S_o is the future sum of OPEX in a specific number of years

i is the discount rate

n is the number of years

<table>
<tr><th colspan="2" rowspan="2">Cooling Scenario</th><th rowspan="2">Power (MW)</th><th rowspan="2">Annual OPEX (M$/Year)</th><th colspan="4">Present Values</th></tr>
<tr><th>CAPEX (M$)</th><th>OPEX Over 15 Years (M$)</th><th colspan="2">Total Costs(M$) = OPEX Over 15 Years + CAPEX</th></tr>
<tr><td>Single Plant</td><td>Solely Surface</td><td>20.5</td><td>12.55</td><td>13.60</td><td>107.4</td><td colspan="2">121</td></tr>
<tr><td rowspan="2">Combined Plants</td><td>Surface</td><td>11.2</td><td>6.85</td><td>7.44</td><td>58.6</td><td>66.04</td><td rowspan="2">107.6</td></tr>
<tr><td>Under-ground</td><td>6.4</td><td>3.92</td><td>7.97</td><td>33.6</td><td>41.6</td></tr>
</table>

Table 22) Cost Estimation for Cooling Design - Diesel Scenario

As it is shown in Table 22, the combined surface and underground cooling plants is more feasible than a surface cooling plant for the conceptual mine. A sole surface cooling plant costs 13.4 M$ more than a combined surface and underground cooling plants over 15 years. Moreover, combined surface and underground cooling can provide better underground environmental conditions and it is more flexible for future developments in the mine design.

For the electric equipment scenario, Table 23 compares an estimation of surface and combined cooling plants. The CAPEX and OPEX are estimated over 15 years and are assumed to be the same as the diesel scenario per kW of cooling power.

<table>
<tr><th colspan="2" rowspan="2">Cooling Scenario</th><th rowspan="2">Power (MW)</th><th rowspan="2">Annual OPEX (M$/Year)</th><th colspan="4">Present Values</th></tr>
<tr><th>CAPEX (M$)</th><th>OPEX Over 15 Years (M$)</th><th colspan="2">Total Costs(M$) = OPEX Over 15 Years + CAPEX</th></tr>
<tr><td>Single Plant</td><td>Solely Surface</td><td>7.30</td><td>4.47</td><td>4.85</td><td>38.3</td><td colspan="2">43.2</td></tr>
<tr><td rowspan="2">Combined Plants</td><td>Surface</td><td>3.94</td><td>2.41</td><td>2.62</td><td>20.6</td><td>23.22</td><td rowspan="2">37.8</td></tr>
<tr><td>Under-ground</td><td>2.25</td><td>1.38</td><td>2.8</td><td>11.8</td><td>14.6</td></tr>
</table>

Table 23) Cost Estimation for Cooling Design - Electric Scenario

As Table 23 shows, the sole surface cooling plant costs 5.4 M$ more than a combined surface and underground cooling plant over 15 years for the electric equipment scenario.

Regarding Table 22 and Table 23, combined surface and underground cooling plants is more cost-effective than sole surface cooling plants for both the diesel and electric equipment scenarios within the studied conceptual mine.

Chapter 6
Conclusion

As mentioned, the scope of this thesis is to compare the ventilation and cooling requirement of the two different scenarios: diesel and electric underground mining equipment fleet. Ventilation costs are mostly dependent on the electric power of the main fans. Although the auxiliary and booster fans have an effect on ventilation costs, only the main fan power has been considered in this study. The annual ventilation operating costs are given in Table 24.

	Diesel Scenario	Electric Scenario
Fan Total Pressure (kPa)	35.6	3.85
Fan Shaft Power (MW)	33.93	1.25
Fan Electrical Power, 85% Efficiency (MW)	35.7	1.31
OPEX (M$/year)	**28.15**	**1.03**
OPEX Over 15 Years (M$)	**240.9**	**8.8**

Table 24) Comparison on Ventilation Operating Costs

Moreover, as Table 24 shows, the electrical fan power in diesel engines is 27 times more than electric scenario. As it is discussed in section 2.5.2, the electric engines do not produce any DPM however, as section 2.4.4 mentioned, the diesel engines produce the DPM gases. Thus, regarding to section 2.3, more airflow needs to be coursed in diesel engine scenario to dilute the DPM emissions. That is why the fan electrical power in diesel scenario is higher than electric scenario.

OPEX and CAPEX were estimated for diesel and electric equipment scenarios in Table 22 and Table 23 for a surface cooling plant and a combination of surface and underground cooling plants scenarios. As these two tables show, the cooling power in diesel scenario is higher than electric scenario. As it is mentioned in section 3.3.2, diesel engines not only produce sensible heat, but they also create latent heat with the maximum motor efficiency of 33%. It means that only $1/3^{rd}$ of the energy extracted from fuel combustion is converted to work against gravity, and the remaining $2/3^{rd}$ is released in the form of heat to the underground environment. Whilst, electric engines only produce sensible heat and its motor efficiency is more than 90%. Thus, diesel engines produce more heat in compared to electric engines with the same motor power rate. That is why cooling power demand for diesel engine scenario is higher than the electric engine scenario which are given in Table 22 and Table 23, respectively.

Since the mid of 2000's the inflation of crude oil on the world market has been increasing from 40$ per barrel to 80$ per barrel where it was seen peaks of 140$ per barrel (Varaschin, J. and E.D. Souza n.d.).Table 25 and Table 26 show an estimation in fuel costs and electricity prices for diesel and electric scenario, respectively.

In Table 25 and Table 26, two 10 hour-shifts per day are considered. It is assumed that the mine working days are 340 days a year.

Table 25 shows the estimated fuel consumption in different zones based on an adjusted diesel power rate which work in each zone. In this table, the fuel consumption is assumed 0.3 litres for each kW.h of diesel engines.

Zones	Adjusted Diesel Power rate (kW)	Fuel Consumption (0.3L/kW.h)	Fuel Costs($0.9/L) Annually($M)	OPEX Over 15 years (M$)
Zone 1	1259	377.7	2.31	30.82
Zone 2	1501	450.3	2.76	36.83
Zone 3	1570	471	2.88	38.43
Zone 4	1570	471	2.88	38.43
Total	5901	1770	10.8	**92.4**

Table 25) Diesel Fuel Operating- Diesel Scenario

To calculate the electrical power for charging batteries, Table 26 can give an estimation of electrical equipment work in each mine zone.

Zones	Adjusted Electric Power rate (kW)	Electric Costs ($0.09/kW.h) Annually	OPEX Over 15 Years (M$)
Zone 1	881.43	0.54	5.8
Zone 2	1050	0.64	6.9
Zone 3	1099	0.67	7.2
Zone 4	1099	0.67	7.2
Total	4131	2.5	**21.4**

Table 26) Electric Machines Operating Costs- Electric Scenario

To estimate CO2 emission, regarding Occupational Health and Safety (OH&S, Mines And Mining Plants, Regulation 854 2017) regulations, each litre of diesel fuel emits 0.00269 tonnes of CO_2 and each MW hour of electricity emits 0.133 tonnes of CO_2. Indeed, each tonne of carbon emission costs 27 Canadian dollars (Julian Varaschin, Euler De Souza n.d.). Table 27 shows the costs of carbon emission for the diesel and electric equipment scenarios. In Table 27, in order to calculate fuel use, the effective utilization factor was given for each piece of equipment from Table 30 to Table 33 for the length of time the diesel equipment operators (2 shifts x 10 hours). Then, the power rate was adjusted based on the quantity of equipment and utilization factor. Assuming fuel consumption of 0.3 L/ kW.hr (Stinnette 2013), the fuel consumption can be determined in litres per hour by multiplying this number with total the adjusted diesel power rate.

Table 27 estimates the fuel consumption of diesel equipment over 340 days a year. For electricity consumption, the total adjusted electric power rate is multiplied by 20 work hours a day, 340 days a year. Then, the carbon emission costs are estimated. The results are given in Table 27.

Scenarios	Total Adjusted Power Rate (kW)	Fuel/Electric Consumption	CO_2 Emission (tonne)	Carbon Emission Costs Annually ($ k)	OPEX Over 15 Years ($ k)
Diesel Scenario	5901	12,038,040 (litres)	32,382.3	0.87	**7.45**
Electric Scenario	4131	28,090,800 (kW)	3,736.1	0.1	**0.86**

Table 27) CO2 Emission Costs for Fuel and Electricity Used

Also, the carbon emission should be calculated for combined surface and underground cooling plants and ventilating operation as well. It should be noticed that the ventilation runs 365 days a year, 24 hours a day. However, the cooling plants run only during shift hours which are 340 days a year, 20 hours a day.

Scenarios	Ventilating Operation (MW)	Cooling Operation (MW)	CO_2 Emission (tonne)	Carbon Emission Costs Annually ($ k)	OPEX Over 15 Years ($ k)
Diesel Scenario	35.7	17.6	57,510.8	15.53	**132.9**
Electric Scenario	1.31	6.19	6,914.8	1.87	**16.0**

Table 28) CO2 Emission Costs for Cooling and Ventilating Operation

In conclusion, a summary of the costs involving the ventilation main fans (OPEX), cooling plants (OPEX and CAPEX), fuel and CO_2 emission costs is summarized in Table 29 for both scenarios. The OPEX is estimated over 15 years with 8% discount rate in Canada. Note that the cooling system mentioned in the Table 29 is the combined surface and underground cooling plants.

The CO_2 emission in the Table 29 is the summation of Table 27 and Table 28.

Costs	Diesel Scenario	Electric Scenario	Savings (M$)
Ventilation (OPEX)	240.9	8.8	232.1
Cooling (OPEX and CAPEX)	107.6	37.8	69.8
Fuel	92.4	21.4	71
CO2 Emission	0.14	0.02	0.12
Total	441.04	68.02	**373**

Table 29) A Comparison on Potential Sources of Cost Savings Over 15 Years

One can see the obvious cost reduction of a fully electrical mining equipment fleet over a fully diesel fleet over 15 years of mine life. Not only does it bring cost reduction, it reduces the exposure of underground workers to harmful diesel fumes.

Chapter 7

Recommendations

Further research and considerations are needed to make the conversion of a fully diesel fleet to an electric fleet viable:

- Capital cost comparison between electrical equipment and diesel equipment
- Study of hybrid vehicles, allowing the same capacity of diesel with the partial benefit of electrical equipment
- Optimisation of the electrical engine's battery life time, maintenance costs, and environmental impact should be considered
- Ventilation saving by implementing VOD on electric and diesel equipment scenarios
- Future development of underground battery charging methods, slow charging, fast charging and battery swap station and their effects on materials handling, ventilation and cooling system

Chapter 8

Appendix A

(Airflow requirement in each zones)

Activity	Equipment	Quantity	Power (hp)	Power (kW)	Utilization Factor	Adjusted Total Power (kW)	Air flow Required ($0.075\ m^3/s/kW$)	Heat Load Production (kW)
Production	6yd LHD	1	279	208	100%	208	15.6	588.8
	Emulsion Loader	1	138	103	40%	41	3.1	116.5
	Production Drill	1	160	119	40%	48	3.6	135.1
	Toyota Jeep	1	128	95	40%	33	2.9	108.0
	Subtotal						25.1	948.4
	15% Leakage Added to Air Flow						28.9	
Development	jumbo	0	160	119	40%	0	0	0
	Toyota Jeep	0	128	95	40%	0	0	0
	Bolter	0	154	115	40%	0	0	0
	6yd LHD	0	279	208	100%	0	0	0
	Backhoe	0	46	34	40%	0	0	0
	Subtotal						0	
Ramp	36t Truck	2	400	298	100%	597	44.7	1688.3
	Boom Truck	1	201	150	40%	60	4.5	169.7
	Scissor Truck	1	147	110	40%	44	3.3	124.1
	Mobile Drill	0	160	119	40%	0	0.0	0.0
	Fork Lift	2	46	34	40%	27	2.1	77.7
	Grader	1	146	109	40%	44	3.3	123.2
	Toyota Jeep	4	128	95	40%	153	11.5	432.2
	Subtotal						69.3	2615.1
Summary	Production	1			100%		28.9	948
	Development	0			100%		0.0	0.0
	Ramp	1			100%		69.3	2615.1
	Garage	1			100%		13.21	
	Subtotal						111.4	
	Primary Leakage	15%					16.7	
	Design Allowance	15%					16.7	
Total in Zone 1 (Underground)							**144.9**	**3564**
Recommended Volume Flow on the surface (Based on $\rho_{Local} = 1.27\ kg/m^3$)							**154.6**	

Table 30) Air Flow Required and Heat Load Production for First Zone (Diesel Scenario)

Activity	Equipment	Quantity	Power (hp)	Power (kW)	Utilization Factor	Adjusted Total Power (kW)	Air flow Required ($0.075\ m^3/s/kW$)	Heat Load Production (kW)
Production	6yd LHD	1	279	208	100%	208	15.6	588.8
	Emulsion Loader	1	138	103	40%	41	3.1	116.5
	Production Drill	1	160	119	40%	48	3.6	135.1
	Toyota Jeep	1	128	95	40%	38	2.9	108.0
	Subtotal						25.1	948.4
	15% Leakage Added to Air Flow						28.9	
Development	jumbo	1	160	119	40%	48	3.6	135.1
	Toyota Jeep	1	128	95	40%	38	2.9	108.0
	Bolter	2	154	115	40%	92	6.9	260.0
	6yd LHD	1	279	208	100%	208	15.6	588.8
	Backhoe	1	46	34	40%	14	1.0	38.8
	Subtotal						30.0	1130.7
	15% Leakage Added to Air Flow						34.5	
Ramp	36t Truck	2	400	298	100%	597	44.7	1688.3
	Boom Truck	1	201	150	40%	60	4.5	169.7
	Fork Lift	2	46	34	40%	27	2.1	77.7
	Grader	0	146	109	40%	0	0.0	0.0
	Toyota Jeep	1	128	95	40%	38	2.9	108.0
	Scissor Truck	1	147	110	40%	44	3.3	124.1
	Subtotal						57.4	2167.7
Summary	Production	1			100%		28.9	948.4
	Development	1			100%		34.5	1130.7
	Ramp	1			100%		57.4	2167.7
	Garage	0			100%			
	Subtotal						120.8	
	Primary Leakage	15%					18.1	
	Design Allowance	15%					18.1	
Total in Zone 2 (Underground)							**157.1**	**4246.8**
Recommended Volume Flow on the surface (Based on $\rho_{Local} = 1.29\ kg/m^3$)							**170.3**	

Table 31) Air Flow Required and Heat Load Production for Second Zone (Diesel Scenario)

Activity	Equipment	Quantity	Power (hp)	Power (kW)	Utilization Factor	Adjusted Total Power (kW)	Air flow Required ($0.075\, m^3/s/kW$)	Heat Load Production (kW)
Production	Jumbo	1	160	119	100%	48	3.6	135.1
	6yd LHD	1	279	208	40%	208	15.6	588.8
	Emulsion Loader	1	138	103	40%	41	3.1	116.5
	Toyota Jeep	1	128	95	40%	38	2.9	108.0
	Subtotal Air						25.1	948.4
	15% Leakage Added to Air Flow						28.9	
Development	jumbo	1	160	119	40%	48	3.6	135.1
	Toyota Jeep	1	128	95	40%	38	2.9	108.0
	Bolter	1	154	115	40%	46	3.5	130.0
	6yd LHD	1	279	208	100%	208	15.6	588.8
	Backhoe	1	46	34	40%	14	1.03	38.8
	Subtotal						26.5	1001
	15% Leakage Added to Air Flow						30.5	
Ramp	36t Truck	2	400	298	100%	597	44.7	1688.3
	Boom Truck	1	201	150	40%	60	4.5	169.7
	Fork Lift	1	46	34	40%	14	1.03	38.8
	Grader	1	146	109	40%	44	3.3	123.2
	Toyota Jeep	2	128	95	40%	76	5.7	216.1
	Scissor Truck	1	147	110	40%	44	3.3	124.1
	Mobile Drill	1	160	119	40%	48	3.6	135.1
	Subtotal Air Flow Required						66.1	2495.3
Summary	Production	2			100%		28.9	948.4
	Development	2			100%		30.5	1000.7
	Ramp	1			100%		66.1	2495.3
	Garage	0			100%		0	
	Subtotal						125.5	
	Primary Leakage	15%					18.8	
	Design Allowance	15%					18.8	
Total in Zone 3 (Underground)							**163.2**	**4444.4**
Recommended Volume Flow on the surface (Based on $\rho_{Local} = 1.32\, kg/m^3$)							**181.0**	

Table 32) Air Flow Required and Heat Load Production for Third Zone (Diesel Scenario)

Activity	Equipment	Quantity	Power (hp)	Power (kW)	Utilization Factor	Adjusted Total Power (kW)	Air flow Required ($0.075\ m^3/s/kW$)	Heat Load Production (kW)
Production	Jumbo	1	160	119	100%	48	3.6	135.1
	6yd LHD	1	279	208	40%	208	15.6	588.8
	Emulsion Loader	1	138	103	40%	41	3.1	116.5
	Toyota Jeep	1	128	95	40%	38	2.9	108.0
	Subtotal Air Flow Required						25.47	948.4
	15% Leakage Added to Air Flow						29.29	
Development	jumbo	1	160	119	40%	48	3.6	135.1
	Toyota Jeep	1	128	95	40%	38	2.9	108.0
	Bolter	1	154	115	40%	46	3.5	130.0
	6yd LHD	1	279	208	100%	208	15.6	588.8
	Backhoe	1	46	34	40%	14	1.03	38.8
	Subtotal Air Flow Required						26.87	1001
	15% Leakage Added to Air Flow						30.91	
Ramp	36t Truck	2	400	298	100%	597	44.7	1688.3
	Boom Truck	1	201	150	40%	60	4.5	169.7
	Fork Lift	1	46	34	40%	14	1.03	38.8
	Grader	1	146	109	40%	44	3.3	123.2
	Toyota Jeep	2	128	95	40%	76	5.7	216.1
	Scissor Truck	1	147	110	40%	44	3.3	124.1
	Mobile Drill	1	160	119	40%	48	3.6	135.1
	Subtotal						67.01	2495.3
Summary	Production	2			100%		28.9	948.4
	Development	2			100%		30.5	10001
	Ramp	1			100%		66.1	2495.3
	Garage	0			100%		0	0
	Subtotal						125.5	
	Primary Leakage	15%					18.8	
	Design Allowance	15%					18.8	
Total in Zone 4 (Underground)							**163.2**	**4444.4**
Recommended Volume Flow on the surface (Based on $\rho_{Local} = 1.32\ kg/m^3$)							**181.0**	

Table 33) Air Flow Required and Heat Load Production for Fourth Zone (Diesel Scenario)

Activity	Equipment	Quantity	Power (hp)	Power (kW)	Utilization Factor	Adjusted Total Power (kW)	Air flow Required $(0.036\, m^3/s\,/kW)$	Heat Load Production (kW)
Production	6yd LHD	1	279	208	100%	146	5.24	146
	Emulsion Loader	1	138	103	40%	29	1.04	29
	Production Drill	1	160	119	40%	33	1.2	33
	Toyota Jeep	1	128	95	40%	27	0.96	27
	Subtotal						8.44	235
	15% Leakage Added to Air Flow						9.71	
Development	jumbo	0	160	119	40%	0	0	0
	Toyota Jeep	0	128	95	40%	0	0	0
	Bolter	0	154	115	40%	0	0	0
	6yd LHD	0	279	208	100%	0	0	0
	Backhoe	0	46	34	40%	0	0	0
	Subtotal						0	0
Ramp	36t Truck	2	400	298	100%	418	15.03	418
	Boom Truck	1	201	150	40%	42	1.51	42
	Scissor Truck	1	147	110	40%	31	1.1	31
	Mobile Drill	0	160	119	40%	0	0	0
	Fork Lift	2	46	34	40%	19	0.69	19
	Grader	1	146	109	40%	30	1.1	30
	Toyota Jeep	4	128	95	40%	107	3.85	107
	Subtotal						23.29	646.9
Summary	Production	2			100%		9.71	234.6
	Development	0			100%		0	0
	Ramp	1			100%		23.29	646.9
	Garage	1			100%		13.21	
	Subtotal						46.21	
	Primary Leakage	15%					6.93	
	Design Allowance	15%					6.93	
Total in Zone 1 (Underground)							**60.08**	**881.4**
Recommended Volume Flow on the surface (Based on $\rho_{Local} = 1.27\ kg/m^3$)							**64.12**	

Table 34) Air Flow Required for First Zone (Electric Scenario)

Activity	Equipment	Quantity	Power (hp)	Power (kW)	Utilization Factor	Adjusted Total Power (kW)	Air flow Required ($0.036\, m^3/s\,/kW$)	Heat Load Production (kW)
Production	6yd LHD	1	279	208	100%	146	5.24	145.6
	Emulsion Loader	1	138	103	40%	29	1.04	28.8
	Production Drill	1	160	119	40%	33	1.20	33.4
	Toyota Jeep	1	128	95	40%	27	0.96	26.7
	Subtotal						8.44	234.6
	15% Leakage Added						9.71	
Development	jumbo	1	160	119	40%	33	1.20	33.4
	Toyota Jeep	1	128	95	40%	27	0.96	26.7
	Bolter	2	154	115	40%	64	2.32	64.3
	6yd LHD	1	279	208	100%	146	5.24	145.6
	Backhoe	1	46	34	40%	10	0.35	9.6
	Subtotal						10.07	279.7
	15% Leakage Added						11.58	
Ramp	36t Truck	2	400	298	100%	418	15.03	417.6
	Boom Truck	1	201	150	40%	42	1.51	42.0
	Fork Lift	2	46	34	40%	19	0.69	19.2
	Grader	0	146	109	40%	0	0.00	0.0
	Toyota Jeep	1	128	95	40%	27	0.96	26.7
	Scissor Truck	1	147	110	40%	31	1.10	30.7
	Subtotal						19.3	536.2
Summary	Production	1			100%		9.71	234.6
	Development	1			100%		11.58	279.7
	Ramp	1			100%		19.30	536.2
	Garage	0			100%		0	
	Subtotal						40.59	
	Primary Leakage	15%					6.09	
	Design Allowance	15%					6.09	
Total in Zone 2 (Underground)							**52.77**	**1050.5**
Recommended Volume Flow on the surface (Based on $\rho_{Local} = 1.29\ kg/m^3$)							**57.21**	

Table 35) Air Flow Required for Second Zone (Electric Scenario)

Activity	Equipment	Quantity	Power (hp)	Power (kW)	Utilization Factor	Adjusted Total Power (kW)	Air flow Required ($0.036\, m^3/s\,/kW$)	Heat Load Production (kW)
Production	Jumbo	1	160	119	100%	33	1.20	33
	6yd LHD	1	279	208	40%	146	5.24	146
	Emulsion Loader	1	138	103	40%	29	1.04	29
	Toyota Jeep	1	128	95	40%	27	0.96	27
	Subtotal						8.44	234.6
	15% Leakage Added to Air Flow						9.71	
Development	jumbo	1	160	119	40%	33	1.20	33
	Toyota Jeep	1	128	95	40%	27	0.96	27
	Bolter	1	154	115	40%	32	1.16	32
	6yd LHD	1	279	208	100%	146	5.24	146
	Backhoe	1	46	34	40%	10	0.35	10
	Subtotal						8.91	247.5
	15% Leakage Added to Air Flow						10.25	
Ramp	36t Truck	2	400	298	100%	418	15.03	418
	Boom Truck	1	201	150	40%	42	1.51	42
	Fork Lift	1	46	34	40%	10	0.35	10
	Grader	1	146	109	40%	30	1.10	30
	Toyota Jeep	2	128	95	40%	53	1.92	53
	Scissor Truck	1	147	110	40%	31	1.10	31
	Mobile Drill	1	160	119	40%	33	1.20	33
	Subtotal						22.22	617.2
Summary	Production	2			100%		9.71	234.6
	Development	2			100%		10.25	247.5
	Ramp	1			100%		22.22	617.2
	Garage	0			100%		0	
	Subtotal						42.18	
	Primary Leakage	15%					6.33	
	Design Allowance	15%					6.33	
Total in Zone 3 (Underground)							**54.83**	**1099.3**
Recommended Volume Flow on the surface (Based on $\rho_{Local} = 1.32\ kg/m^3$)							**60.82**	

Table 36) Air Flow Required for Third Zone (Electric Scenario)

Activity	Equipment	Quantity	Power (hp)	Power (kW)	Utilization Factor	Adjusted Total Power (kW)	Air flow Required $(0.036\, m^3/s\,/kW)$	Heat Load Production (kW)
Production	Jumbo	1	160	119	100%	33	1.20	33
	6yd LHD	1	279	208	40%	146	5.24	146
	Emulsion Loader	1	138	103	40%	29	1.04	29
	Toyota Jeep	1	128	95	40%	27	0.96	27
	Subtotal Air Flow Required						8.44	234.6
	15% Leakage Added						9.71	
Development	jumbo	1	160	119	40%	33	1.20	33
	Toyota Jeep	1	128	95	40%	27	0.96	27
	Bolter	1	154	115	40%	32	1.16	32
	6yd LHD	1	279	208	100%	146	5.24	146
	Backhoe	1	46	34	40%	10	0.35	10
	Subtotal Air Flow Required						8.91	247.5
	15% Leakage Added						10.25	
Ramp	36t Truck	2	400	298	100%	418	15.03	418
	Boom Truck	1	201	150	40%	42	1.51	42
	Fork Lift	1	46	34	40%	10	0.35	10
	Grader	1	146	109	40%	30	1.10	30
	Toyota Jeep	2	128	95	40%	53	1.92	53
	Scissor Truck	1	147	110	40%	31	1.10	31
	Mobile Drill	1	160	119	40%	33	1.20	33
	Subtotal Air Flow Required						22.22	617.2
Summary	Production	2			100%		9.71	234.6
	Development	2			100%		10.25	247.5
	Ramp	1			100%		22.22	617.2
	Garage	0			100%		0	
	Subtotal						42.18	
	Primary Leakage	15%					6.33	
	Design Allowance	15%					6.33	
Total in Zone 4 (Underground)							**54.83**	**1099.3**
Recommended Volume Flow on the surface (Based on $\rho_{Local} = 1.32\ kg/m^3$)							**60.82**	

Table 37) Air Flow Required for Fourth Zone (Electric Scenario)

Chapter 9

Appendix B

(Assumptions)

Groups	Parameters	Values	Unit
Surface condition	Pressure	101.6	kPa
	Wet bulb temperature	18	℃
	Dry bulb temperature	23	℃
	Geothermal gradient	1.8	$℃/100m$
	Surface rock temperature	22.5	℃
Geometry	Tunnel	5.07	Hydraulic diameter (m)
	Intake Shaft	7.32 (24)	$m\ (ft)$
	Exhaust Shaft	6.71 (22)	$m\ (ft)$
	Inclined	3.66 (12)	$m\ (ft)$
Atkinson friction	Shafts (Concrete lined, Rope guides, pipe fitting)	0.0065	kg/m^3
	Tunnel (Unlined, typical conditions, no major irregularities)	0.012	kg/m^3
	Inclined (Unlined, major irregularities removed)	0.014	kg/m^3
Shock Losses	90 degrees (Bend), $r/w = 1$	0.2	Dimensionless
	45 degrees, $r/w = 1$	0.04	Dimensionless
Rock Thermal Properties	Rock density	2700	
	Specific heat	840	$J/kg℃$
	Conductivity	2.7	$W/m℃$
	Diffusivity	1.16	$10^{-6} \times m^2/s$

Table 38) Table of Assumptions- Imported to VentSIM

Junction	Equivalent Length (m)
Tunnel to Raise (90 degree)	13.11
Tunnel to Raise (45 degree)	7.06
Tunnel to Tunnel(90 degree)	12.68
Raise to Tunnel (45 degree)	82.26
Raise to Tunnel (90 degree)	82.39
Intake Shaft to Tunnel (90 degree)	19.65
Tunnel to Exhaust shaft (90 degree)	135.88

Table 39) Calculated Equivalent Length at Junctions + Shock Losses

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