Improving air distribution in deep-level mine ventilation systems

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ABSTRACT

Improving underground mine conditions results in fewer ventilation-related fatalities and increases productivity. The objective of a subsurface ventilation system is to ensure sufficient quantity and quality of airflow in the working areas of a mine. Deep-level mines are typically overventilated with poor volumetric efficiencies due to old working leakage, deteriorated stopping leakage, air being recirculated and high fan pressures. Consequently, the low volumetric efficiencies are a direct result of poor air distribution. Additional air is distributed through these mines to compensate for the air used wrongfully. This practice consumes an unnecessary amount of electrical energy since more air than required is supplied to the system.

Expensive electrical tariffs are the most significant contributor to mining expenses. Therefore, the management and sustainability of energy is the central focus of today's mines. Any reduction in expenses increases the lifetime and production outcome of mines. Depending on the type of mine, underground mine ventilation systems can contribute up to 40% of the total electrical cost. The contribution of the quantity control devices can range from 20% to 70% whereas the contribution of quality control devices can range from 0% to 60%.

The increasing depths, complexity, size and mechanisation of mines increase the ventilation demand, which influences the rising operational costs directly. At great depths, the ventilation cost and requirements will eventually be impossible to sustain. Mining expansion and high electrical tariffs are forcing the mining sector to reduce its operational costs while maintaining legal limits. However, the study confirmed a cubic relation between the power required to obtain a specific quantity airflow and the quantity itself ($Power \approx (Flow)^3$); therefore, a small reduction in airflow quantity can result in a large reduction of power.

Literature was reviewed about the airflow quantity control (air distribution) of a mine ventilation system. A variety of strategies were considered in which improving the ventilation system allowed for the system to be more energy efficient. However, deep-level mine ventilation systems have a few constraints that young developing mines do not have. Old deep-level mines usually lack the newest and advanced monitoring and control devices presently available. The expected lifetimes of these mines are reducing rapidly, which limits the payback period to make long-term efficient system investments viable. Making advance modern improvement is, therefore, highly unlikely on deep-level mines due to the large changes required and the expenses involved. Consequently, deep-level mines usually have lower volumetric efficiencies than young developing mines. Thus, deep-level mines have a higher potential for reducing energy usage by improving the air distribution of the mine ventilation system.

The objective of this dissertation was to create a feasible method for improving the air distribution of a deep-level mine ventilation system. The mine was considered as an integrated system in which small practical changes were made over the entire ventilation system. The ventilation changes had to be as cost-efficient as possible and remain within the mine's operational standards. The study aimed to show that combining these small changes would have a large effect on the overall air distribution of the ventilation system. The improved system could then be considered for possible energy reductions and savings.

A simulation-driven method was proposed since a simulation model is the most feasible way of considering a mine ventilation system as an integrated system. The method focused on creating, preparing and verifying a ventilation simulation model. Improvement predictions were implemented on the simulation model in a strategic order using four identification cycles. The first three identification cycles improved the air distribution from the surface to the working areas. The fourth identification cycle investigated the potential return improvements in the ventilation system. The simulation predictions enabled the researcher to analyse and investigate integrated behavioural changes on the overall ventilation system.

A deep-level gold mine (Mine A), located near Carletonville, South Africa, was used as a case study for the methodology. The simulation model was calibrated to an average deviation of 9.27 kg/s. The model's predictability was within an average deviation of 4 kg/s, which was considered acceptable for predicting improvement initiatives. The four identification cycle improvements were applied to the system.

The primary ventilation system's air distribution improved by 11%, which resulted in an overall volumetric efficiency of 78.3%. The average airflow of the working areas improved with a margin of 5 kg/s from the required airflow. This improvement was achieved despite an energy reduction initiative that resulted in the primary ventilation system being 8.4% underventilated. The improvement process reduced fan power by 1.4 MW, resulting in a R14.9 million saving within two years. Further improvement processes can still be investigated, and an unknown simulation prediction possibility can be considered for future studies.

Keywords: Mine ventilation system, deep-level, air distribution, volumetric efficiency, simulation, integrated and air flow improvement.

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NOMENCLATURE

Symbol	Description	Unit
m	Distance	Kilometres
%	Value out of a hundred	Percentage
kPa	Pressure	Kilopascal
m^3/s	Quantity	Cubes per second
m/s	Velocity	Meters per second
$°C$	Temperature	Degree Celsius
Amps/A	Electric current	Ampere
ppm	Amount of pollution	Parts per million
P	Pressure	Pascal
kg/s	Mass flow	Kilograms per second
MW	Power	Megawatt
kW	Power	Kilowatt
GWh	Power output	Gigawatt hours
R	South African currency	Rand

LIST OF ABBREVIATIONS

Abbreviation	Definition
BAC	Bulk air cooler
BF	Booster fan
CAD	Computer-aided design
CPI	Consumer price index (inflation)
DSM	Demand-side management
DSR	Demand-side response
DB	Dry bulb
E	East
IAW	Intake airway
IGV	Inlet guide vane
L	Level
MF	Main fan
MVS	Mine ventilation system
PTB	Process Toolbox
PVS	Primary ventilation system
RAW	Return airway
RBH	Raise borehole
S	Stope
SCADA	Supervisory control and data acquisition
SVS	Secondary ventilation system
TOU	Time of use
VOD	Ventilation on demand
VSD	Variable speed drive
W	West
WB	Wet bulb
XC	Crosscut

GLOSSARY

Crosscut:	A tunnel connecting the primary mine tunnels to the workings. Also referred to as an XC.
Downcast:	Air moving downwards via any tunnel.
End flows:	Primary return airflow at the end of a level.
Face:	End of a developing tunnel.
Half level:	This study uses the term "half level" where the main tunnel of a level splits to the western and eastern side.
Haulage:	Underground tunnels used for ore removal, travelling and ventilation.
Level:	A section of the mine that branches off the mineshafts.
Production block:	Represents the combination of all the levels where ore is actively being mined and removed.
Refrigeration:	Process of reducing the temperature of water and air.
Recirculation:	Air that circulates continuously from the intake to the return and back to the intake air.
Return block:	Represents all the levels used to return contaminated air.
Service block:	Represents all the levels used for services to assist the mining and ore-removing process.
Shaft:	Perpendicular to the surface and the primary connection between surface and the sections of the mine.
Short-circuit ventilation:	Airflow is obtained through a level by simply opening a connection between the intake and return.
Stope line:	The tunnel parallel to the ore reef. This tunnel, when active, is where ore mining and drilling occur. Mined-out stope lines are usually sealed or used for ventilation returns.
Upcast:	Air moving upwards via any tunnel.
Workings:	Entire system of openings in a mine used for exploitation.

CHAPTER 1: INTRODUCTION

1.1 Preamble

An underground chamber or subsurface tunnel system is an open underground space with limited openings to the atmosphere. Mines that mine ore bodies, which are considerably deep below the earth's surface, are forced to use underground mining techniques. The required amount of unwanted waste material to be excavated through open-pit mining is not feasible to implement [1]. Mines all over the world are becoming deeper due to the decreasing ore deposits near the earth's surface and the high mineral demand [2]. The world's deepest underground mine is mining at depths of approximately 3.8 km below the surface [3].

Each underground mine has a unique mining method, which is highly dependent on the ore deposit shape, geology, grade and volume. Therefore, underground mines can vary in size and depth, depending on their mining methods. The variation of mining methods used is too broad to discuss in this thesis. Hamrin summarised ten typical underground mining methods based on the corresponding significant characteristics of the methods [4]. Regardless of the type of mining method used, all underground mining operations require rock breakage, blasting and the use of heavy machinery, which contribute to the contamination of the air in mines.

The conditions in mines can become quite hazardous due to the contamination of air. The degree of air contamination depends on the nature of the mine (depth, geology, surface climate, rock properties, age of mine etc.) and the mining operations of the mine. Typical underground mine contaminants include heat, diesel emissions, fires, explosions, radiation and dust (detailed description of these contaminants are given in APPENDIX A).

The contaminants influence the air quality of the mine. This makes underground air potentially dangerous to breathe, and extremely high temperatures may also lead to heatstroke. Not only are mining personnel's lives at risk, but production is also highly reliant on these conditions. Contamination occurs in open-pit as well as underground mines. The pressing concern regarding underground mining conditions is the concentration of contamination and the limited air supply.

1.2 Subsurface mine ventilation

The death of miners in the late eighteenth and nineteenth century led to an increase in investigative studies regarding the underground conditions in mines. These studies concluded that an improvement in mine ventilation would result in fewer ventilation-related fatalities and increase the productivity of mine personnel [5].

The objective of a subsurface ventilation system is to ensure a sufficient quantity and quality of airflow in the working areas of a mine. Working areas are defined as any active mining area, developing area, waiting place, travel way or workshop that is occupied by mine personnel. Airflow in a deep-level mine must be sufficient to dilute contaminants from the working areas. Each country has specific legislative requirements regarding the quantity and quality of the air flowing in an underground mine [6].

An adequate ventilation system is essential for a mine to ensure safe working conditions for mine personnel and to adhere to legislation. Mines do not have control over surface temperatures, ore bodies, amount of fissure water or rock properties. However, factors such as the method of mining, use of machines or engines, tunnel layouts and rock breakage rates are within a mine's control [7].

These design factors are typically determined based on production requirements [8]. Mines are designed according to the most feasible and profitable mining method. As previously mentioned, the mining method depends on the ore body, which is determined by the geology of the mine. Therefore, mine ventilation systems (MVSs) should be implemented on an existing tunnel network based on production requirements [9].

MVSs are implemented with the intent to maintain and improve production rates without causing any interferences. Mining methods require machinery, additional ore tunnels, travel ways, water pipes, compressed air pipes and electrical wiring, etc. These requirements affect the tunnel sizes, lengths and locations. This increases the potential of ventilation pressure losses, leaks and recirculation [10].

1.2.1 Overview of a mine ventilation system

A mine tunnel network consists of different sections; each containing a certain number of active and developing working areas. Each section has an intake airway (IAW) and a return airway (RAW). The IAWs of all the sections are connected via a downcast shaft, and the RAWs are connected via an upcast shaft. All the air that is distributed through the mine enters the ventilation system through the downcast shaft and exits through the upcast shaft [11]. These shafts also connect all the sections of the mine with one another and the surface. Different sections of the mine can further be connected to other sections through stope lines or additional tunnels, depending on the mining technique [4].

The total amount of air entering the MVS needs to be distributed to all the sections of the mine. Thereafter, the total amount of air to be distributed to a section must be distributed between every active and developing working area on that section. Therefore, an MVS requires control devices to ensure that an adequate quantity and quality of air reaches each working area of the mine.

Ventilation control devices used in mines

The air pressure drastically decreases as air flows through tunnels. The pressure loss occurs due to resistance, which is dependent on the air velocity, airway surface roughness, tunnel length and area [6]. Discontinuities such as airway obstructions, expansions or contractions cause shock losses that lead to pressure losses. Air will always follow the route of least resistance. Therefore, ventilation control devices are necessary for controlling or producing airflow through an underground mine network [12]. Production airflow is usually controlled with active regulators while distribution airflow is usually controlled with passive regulators.

Production airflow control devices

Energy needs to be added to a ventilation system to generate pressure. The generated pressure should overcome the system's pressure losses to establish airflow. Mines commonly use fans to generate this pressure [6]. A fan is a motor-driven machine that transfers air from a low pressure to a high pressure. Two types of fans are used in mines, namely centrifugal and axial fans [13].

Axial flow fans are used in high-volume, low-pressure applications. They typically generate a pressure of 2 kPa. Axial flow fans are usually applied underground as bulk air coolers (BACs), booster fans and auxiliary fans. Centrifugal fans are used in high-pressure applications. Centrifugal fans typically generate a high pressure of 2–8 kPa and are normally used as the surface main fans of mines [13], [14].

Distribution airflow control devices

An MVS consists of a network of tunnels that split off one from another. The airflow quantity should be controlled at each split based on the ventilation demand of the section. The airflow is controlled by varying the resistance of each branch [12]. Fans are used if a higher pressure is required (active control). However, passive control takes place when a system is regulated without adding additional energy. The most common distribution airflow controllers are:

- Sealing or stoppings
- Doors (airlock systems)
- Airflow regulator
- Air crossings
- Raise borehole (RBH)

A detailed discussion of these distribution airflow controllers can be found in APPENDIX A.

1.3 Mine ventilation systems

As stated in Section 1.2, the objective of a subsurface ventilation system is to ensure a sufficient quantity and quality of airflow in the working areas of a mine. Therefore, an MVS is divided into two subsystems, namely a primary ventilation system (PVS) and a secondary ventilation system (SVS). The PVS supplies air of sufficient quantity and quality to the sections of a mine (Figure 1, marked A–G and J). The SVS distributes air to the actual working and developing areas of a mine (Figure 1, marked I and J) [10], [15].

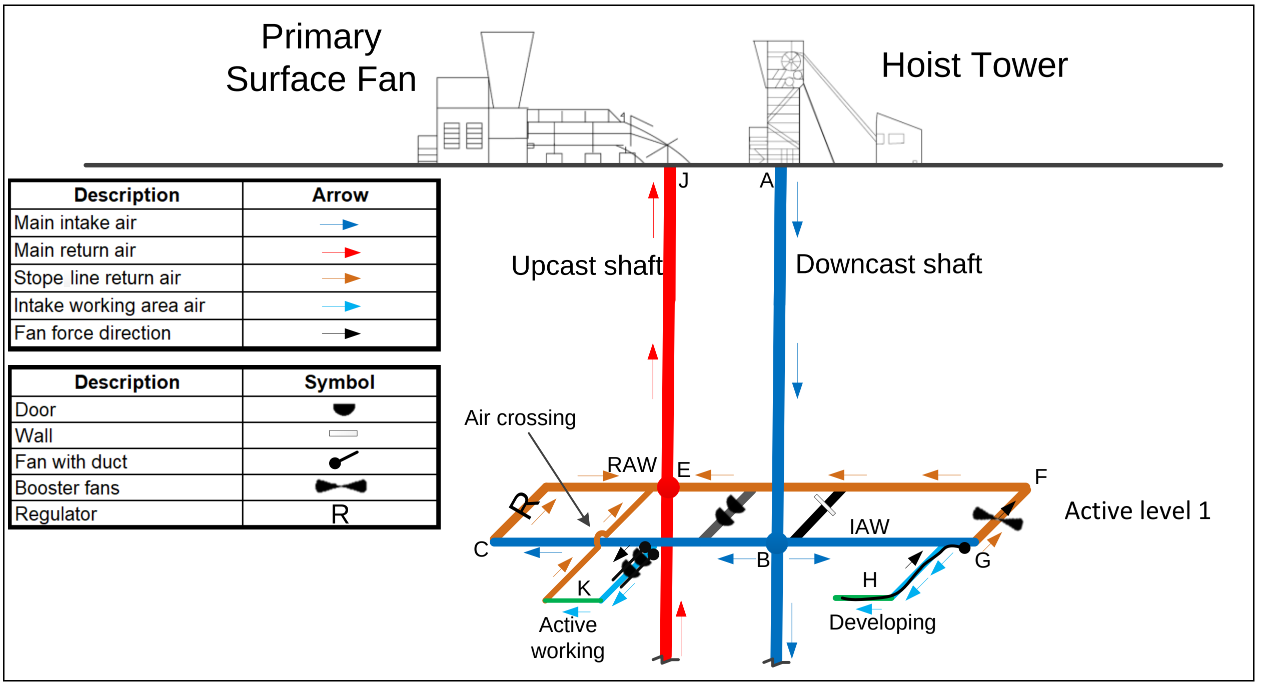


Figure 1: Overview of a general MVS (based on [10])

1.3.1 Primary ventilation system

Atmospheric air or refrigerated atmospheric air (depending on the depth and location of the mine) enters the underground mine via the downcast shaft as seen in Figure 1. The downcast shaft is connected to each section's IAW. The total amount of air entering the mine should first be distributed between the respective sections. Thereafter, the IAW is used to distribute the air throughout the section. The SVS uses the air supplied by the PVS to ventilate the working areas of the mine, which causes the air to become contaminated. The RAW removes the contaminated air from the section by moving the air to the upcast shaft where the air is removed from the system. This route is commonly known as the primary route [6], [10], [15].

The PVS is a fixed system that is established during the initial phases of a mine. As previously mentioned, the objective of a PVS is to ensure that each section of the mine has a sufficient quantity and quality of airflow. This air should be adequate to be used by the SVS to ventilate the working areas of the mine. A simple example of a primary path of a section in a mine is illustrated in Figure 1 (marked A–G and J) [15], [16]. Table 1 represents the typical quantity and quality control strategies used in a PVS (detailed discussion in APPENDIX A).

Table 1: PVS control strategies

PVS	
Quantity control	**Quality control**
Surface fans (SFs) – surface	BACs – surface
Booster fans – underground	BACs – underground

1.3.2 Secondary ventilation system

The PVS distributes the atmospheric air into the ventilation system, whereafter the air is distributed among the different sections of the mine. As previously mentioned, the PVS is a relatively fixed system that circulates air from the surface through the mine and then back again to the surface. The SVS is used to tap air off the primary path to ventilate in-stope working areas, developing areas or service areas of a mine. The objective of the SVS is to remove contamination, which is released through mining operations, and to supply fresh air to the working areas [10], [15], [17].

The SVS is a more dynamic system since continuous development frequently causes a change in air requirements [7]. Consequently, the SVS airflow control also has a variety of applications depending on the working area. These applications are discussed in detail in APPENDIX A. The SVS is highly dependent on the mining method and differs from mine to mine [15], [17]. A simple example of secondary ventilation is shown in Figure 1 (marked I and J).

Table 2 represents the typical quantity and quality control strategies used in an SVS (detailed discussion in APPENDIX A).

Table 2: SVS control strategies

SVS	
Quantity control	**Quality control**
Auxiliary fans – developing working area	Spot coolers (cooling cars)
District fans – active working area	Spray chambers

1.3.3 Collaboration of systems

An MVS is a complex integrated network of tunnels. The PVS and SVS each has an important role to ensure that ventilation is sufficient. The SVS will not be able to supply air to the working areas when the air quality or quantity from the PVS's side is poor. The PVS will not be able to provide air to the working areas without the SVS distributing air from the primary path to the localised areas. Both systems collaborate to ensure that the working areas are ventilated sufficiently [15].

1.4 Mine ventilation limitations and considerations

Atmospheric air is a composition of dry air, which consists of oxygen (20.95%), nitrogen (78.09%), carbon dioxide (0.03%) and argon (0.93%) gas. This gas composition may vary across the earth with a margin of less than 0.01%. Water vapour particles also form part of the air composition. The number of water vapour particles in the air varies depending on three parameters, namely air pressure, air temperature and amount of water-air contact [6]. These three parameters fluctuate constantly in the air moving through a deep-level mine. To determine the composition of dry air and water vapour (mine air conditions), one should measure the wet-bulb temperature, dry-bulb temperature and barometric pressure of the air. These conditional parameters, together with the air velocity and cross-section area can further be used to determine the quantity and mass flow of air [6].

The thermal comfort of mine personnel affects their work performance. The thermal comfort is influenced by the wet-bulb temperature and velocity of the air. The wet-bulb temperature can only be controlled by implementing additional cooling such as BACs, spot coolers or spray chambers. The velocity of the air is controlled by distributing the air efficiently through the mine tunnel network [6], [8].

To improve thermal comfort, one should either improve the velocity or the temperature of the air. Increasing the velocity enables evaporative and convective heat transfer between the air and the human body. Decreasing the air temperature increases the temperature difference between the human body and air. In turn, this increases the heat transfer from the body (high temperature) to the air (low temperature) [2], [6], [18].

However, the safety limits for both these two parameters are controlled based on legislation and safety considerations. The velocity of air in the working areas of a mine is recommended to be between 1 m/s and 3 m/s [2]. The minimum air velocity to dilute contaminants is 0.25 m/s and increases as the concentration of contaminants increases [6]. The recommended maximum airflow velocity in the working areas is 4 m/s. Air velocities above 4 m/s can cause dust dispersion and discomfort for personnel in the working areas [2].

The wet-bulb temperature of air in the working areas of a mine is recommended to be between 27.5°C and 28.5°C [6]. The average resting human body temperature is 37°C; any deviation greater than 3.5°C from this temperature can lead to illness and fatalities [19]. Safety regulations forbid work at a wet-bulb temperature higher than 32.5°C or a dry-bulb temperature higher than 37°C. Temperatures above 37°C reverse the heat transfer from the air to the human body, which increases the body temperature of the human body significantly [6], [13], [20].

Table 3 lists some of the regulations as given by the Mine Health and Safety Act of South Africa that are relative to this study [20].

Table 3: Mine Health and Safety Act of South Africa regulations [13], [20], [21]

Parameter	Standards
Wet-bulb temperature	$\leq 32.5\ °C$
Dry-bulb temperature	$\leq 37.0\ °C$
Pollutants in air	$\leq 1\ mg/m^3$
Working area velocities	$0.5 - 1\ m/s$

Parameter	Standards
Developing area quantities	$0.3 - 0.5\ m/s\ per\ m^2\ of\ face$
Oxygen in general air	$\geq 19\%$
Flammable gases in general air	$\leq\ 0.5\%$

1.5 Deep-level mine ventilation systems

Deep-level mines are becoming more common all over the world since near-surface deposits are decreasing [1]. The term 'ultra-deep-level' or 'deep-level' mining typically refers to mines that have been operating for more than 40 years [22].

Mine activities and working areas are constantly changing status from development, to active, to no-mining as the mine develops. The SVS control application constantly changes with the varying status of the working area. The PVS is adjusted occasionally when major SVS changes are implemented, such as the hauling of a new district working area [21], [23].

Mines should frequently consider the air distribution of the ventilation system to ensure a safe environment for personnel. However, mines tend to be overcautious and overventilate due to the historical lack of air distribution control [17], [21]. The volumetric efficiency equation (Equation 1) is used to calculate the efficiency of air distribution in an MVS. The calculation compares the airflow used with the airflow supply [24].

Equation 1: Volumetric efficiency [24]

$$Volumetric\ efficiency\ (\%) = \frac{Airflow\ in\ workings\ \left(\frac{kg}{s}\, or\, \frac{m^3}{s}\right)}{Airflow\ supplied\ to\ system\ \left(\frac{kg}{s}\, or\, \frac{m^3}{s}\right)} \times 100$$

A study conducted on coal mines in the United States concluded that the average mine's volumetric efficiency is 38% (see Figure 2). This means that the additional air (62%) is leaking, recirculating or being used to ventilate unoccupied workings. The volumetric efficiency of old mines (deep-level) is usually lower than that of young developing mines. The main reasons include the number of old workings, deteriorated stoppings, recirculation possibilities and fan pressures [24].

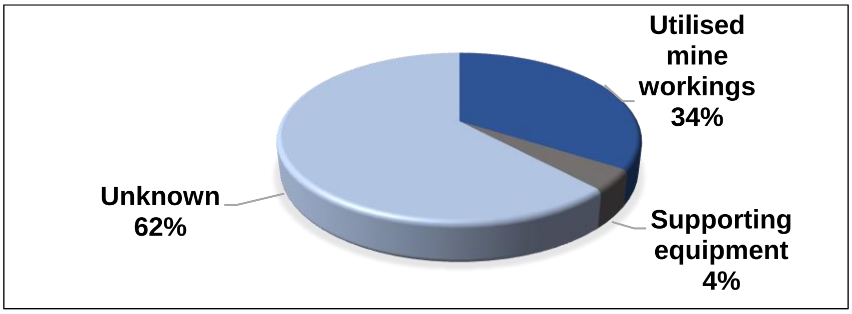


Figure 2: Average ventilation air distribution on coal mines in the United States [24]

Old mines have numerous old working areas that are sealed off. This raises the potential of short circuits due to air leaking through poor or deteriorated seals. Short-circuiting occurs when air leaks out of the IAW to the RAW before passing through the active occupied working areas [10]. High fan pressures increase the possibility of leakage and weaken the volumetric efficiency of a mine. Deep-level mines have long tunnels and more obstructions than other mines. Therefore, more ventilation controllers such as fans are required to overcome these pressure losses. A high total fan pressure increases the possibility of leaks and, therefore, decreases the volumetric efficiency of mines [24]. This phenomenon can be seen in Figure 3.

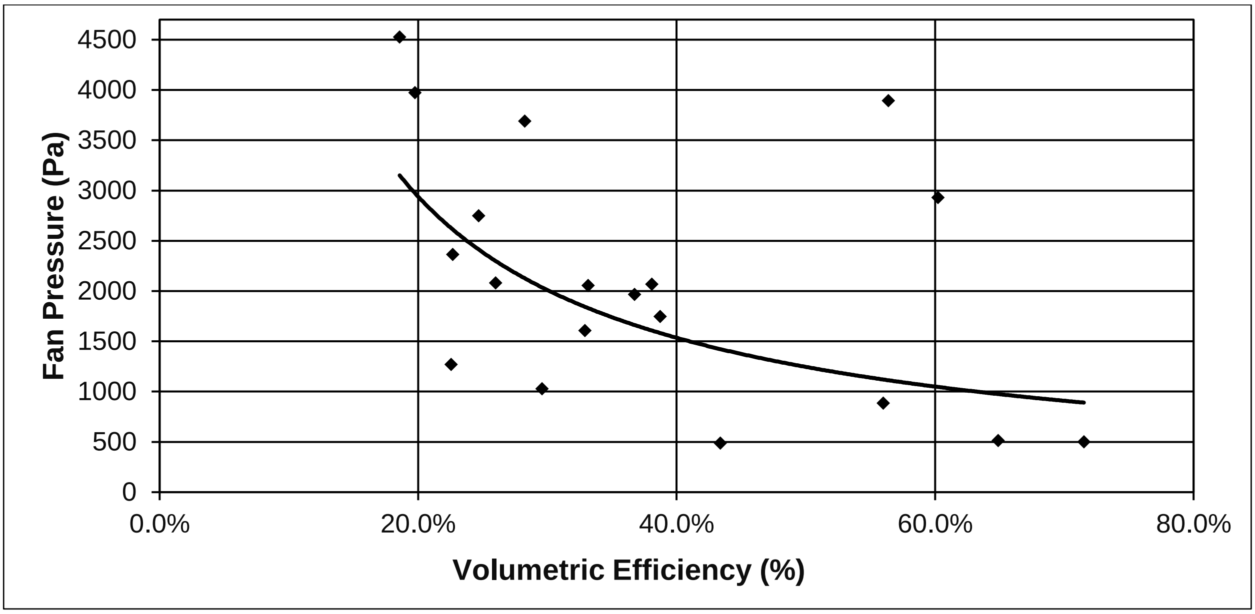


Figure 3: Volumetric efficiency compared with total fan pressure [24]

It can be concluded that a low volumetric efficiency is solemnly due to poor air distribution. This is very common in deep-level mines since more air is supplied to the mine than what is distributed to the working areas. Consequently, an unnecessary amount of energy is used to compensate for the poor air distribution and airflow losses [24].

1.6 Mine operational cost

The energy demand of the world is continuously rising due to industrial and populational growth. The world's energy demand increases with approximately 5.3% per annum [25], [26]. Electrical tariffs are constantly rising due to the struggle of supplying the rising demand [27]. The mining and industrial sectors are some of the highest consumers of energy [28]. Both South African and international increases in electricity tariffs are the mining industry's most significant contributor to expenditures [29]. As a result, the operational costs of mines are highly dependent on the electrical tariffs of their respective countries, with South Africa being a typical example [30].

The South African gold mining industry used to be the leading gold producer in the world. Over the past few years, the increase rate of Eskom's electrical tariffs has been more than the increase rate of the consumer price index (CPI). Today, less than 20% of South African gold mines are still operating profitably (detailed discussion in APPENDIX B) [31], [32].

Historically low electrical tariffs made mine personnel unaware of excessive energy use. The management and sustainability of energy is, however, the central focus of today's mines. The aim of the industry is to use energy efficiently as long as possible. This decreases the mine's operational cost and therefore extends the operational lifetime of the mine [33].

1.6.1 Mine ventilation operational cost

All sectors that have high electricity use, such as refrigeration, ventilation and processing, can benefit from being energy efficient. An underground MVS, depending on the type of mine, can contribute up to 40% of the total electrical cost of the mine [34]. The system includes all PVS and SVS air quality and quantity control components [16].

The air distribution (quantity) control parameters of the PVS usually include a small number of large fans. The SVS, on the other hand, has a large number of small fans [16]. The PVS fans of a mine must run 24 hours a day for the entire year [35]. The SVS consists of temporarily installed fans that operate when a working area is being used [16]. The relative cost contribution of the air quantity control components in a metal mine's PVS and SVS can range from 20% to 70% [15], [16].

Air-cooling (quality) control parameters include condenser cooling towers, storage dams, chillers, precooling towers, BACs (PVS) and spot coolers (SVS). The number of parameters installed in the system depends on the mine's application [33]. The relative cost contribution of the air quality control components in a metal mine's PVS and SVS can range from 0% to 60% [16].

Mines are expanding in depth, complexity, size and mechanisation. Expansion increases the ventilation demand, which directly increases the mine's operational cost. At great depths, the ventilation cost and requirements will eventually be impossible to sustain [8]. The mining expansion and electrical tariffs increases are forcing mining sectors to reduce their operational costs while maintaining legal operational limits [36].

1.7 Ventilation improvement strategies

1.7.1 Background

A ventilation system, as previously mentioned in Section 1.2, focuses on the quality and quantity of airflow at a specific working area. The PVS and SVS have specific air quality and quantity control devices and systems to ensure an adequate working environment in the working areas. Improving the control of air quality and quantity can reduce the operational cost of a mine. It was, however, realised that there is a cubic relation between the power required to obtain a specific quantity of airflow and the quantity itself ($Power \approx (Flow)^3$) [8]. In other words, a small reduction in airflow quantity can result in a large reduction of power. Therefore, this literature study focused on the airflow quantity control (air distribution) of an MVS and the potential initiatives for improvement.

A wide variety of mine ventilation air distribution improvement initiatives exists all over the world. The purpose of these initiatives is to reduce or manage the airflow quantity distributed through the PVS or SVS to reduce the operational cost. It is, however, compulsory that the improvement method does not compromise the health and safety of the working environment in the mine [29], [37].

A typical hard rock mining cycle consists of an eight-hour mining period, eight-hour ore-removing period and an eight-hour blasting period. During the mining period, workers are actively working in the working area. This includes drilling and charging explosives. During the ore-removing period, the ore is removed from the active workings and transported to surface. During the blasting period, blasting takes place and dust and fumes are cleared [29].

The conventional approach of mines is to ventilate at full capacity for 24 hours a day, 365 days a year [35]. This approach neglects the fact that the ventilation demand varies depending on the day and the mining period. For example, the actual working areas of the mine are only occupied for 16 hours a day since workers must be evacuated during the blasting period. The unoccupied working areas only require an adequate quantity of airflow to remove the fumes without extending the re-entry period (time after blasting before the working areas can be entered). Consequently, the mining cycle allows for significant savings when managed appropriately [8], [13], [29], [35].

1.7.2 Demand-side management

Demand-side management (DSM) is generally used in the mine environment to keep the energy supplied to a system within a specific margin from the system's demand side. DSM is a short-term solution for obtaining a cost-efficient system. The key focus of this method is to manipulate the electrical usage (supply) pattern of the system so that the supply is just adequate for the demand [7], [13], [15], [17], [38]. Typical load management techniques used for DSM include valley filling, peak clipping, load shifting, strategic load growth and strategic conservation (detailed discussion in APPENDIX A).

Modern DSM programmes consider strategic load growth and valley filling to be unacceptable due to the growing concern of energy security and environmental degradation. A modern definition of DSM is a technique used to reduce or manage energy consumption to lower the operational cost or accomplish policy objectives. Therefore, DSM programmes can be divided into two categories, namely energy efficiency and demand-side response (DSR) strategies [7], [17].

Energy efficiency reduces the overall energy demand of a system by either conserving energy or improving the operational efficiency. Energy can be conserved when behavioural and operational patterns are changed to reduce energy consumption over a long period. Typical examples include switching off appliances when they are not being used [7], [17]. Improving the operational efficiency of a system requires that the useful output energy is improved from the total input energy. Typical examples include replacing devices with more efficient devices or adjusting devices according to the need [7], [17].

DSR entails using equipment according to time-of-use (TOU) tariffs. This technique does not focus on the energy but rather on the energy pricing and the ideal TOU to reduce the energy consumption costs. DSR methods have been proven to reduce the operational cost of systems significantly in numerous industrial and mining applications. Studies of DSR strategy on a steel plant batch process showed an electrical cost saving of 5.7% [39], a purification plant pump station showed a saving of 32% [40], and a colliery conveyor belt system showed a saving of 49% [41]. Although DSR strategies can reduce operational costs significantly, they do not reduce the system's energy usage considerably because the technique mainly focuses on shifting load from peak periods while the amount of load remains the same [7], [17].

DSR strategies are commonly used in ventilation quality control systems. Energy savings of between 8.3% and 15% have been achieved by applying load shifting to deep-level mine cooling systems [42]. However, the strategy is not that common in ventilation quantity control systems. The assuming reason is that the ventilation airflow is too dependent on the working cycle of the mine. DSR strategies can be applied to the ventilation quantity control systems, but the mining cycle requires change according to the TOU tariffs.

Chatterjee did a study in which the ventilation airflow quantity was controlled with an energy efficiency and DSR strategy [7]. The DSR strategy required the mining cycle to start at an optimum time of the day to obtain maximum cost saving. The DSR and energy efficiency method obtained a total cost saving of 74% on weekdays, with 16% of the savings being due to the DSR strategy [7]. Studies, as discussed by Smit, focused on load clipping the ventilation system during peak periods, which is only applicable when the ventilation demand is low [17]. The limiting factor in this scenario is the ventilation demand and not the peak periods. Therefore, the ideal system regulates the ventilation based on the demand (such as an energy efficiency strategy).

Considering Chatterjee's study results, one can see that energy efficiency strategies contribute more to cost savings than DSR strategies [7]. DSR strategies depend on the TOU tariffs and therefore require significant changes to the mining cycle. Energy efficiency strategies have been proven to be the most feasible strategies to implement on an MVS [29]. However, to obtain an energy efficient MVS, one should adjust the ventilation airflow quantity according to demand, replace ventilation control devices with more efficient devices, or consider configuration improvements.

1.7.3 Ventilation on demand

The demand of an active section is higher than the demand of a non-active section. Ventilation on demand (VOD) is used to vary the air quantity that is supplied to the overall mine, section or district according to the demand instead of ventilating it at full capacity [7], [17], [43]. For example, reducing the secondary airflow in a non-active section reduces the energy consumption of district fans and/or auxiliary fans and increases the downstream primary airflow. This improves the working conditions and optimises the operational cost of the mine. An overall decrease in working activities, for example over weekends, could even reduce primary airflow by reducing main fan and/or booster fan energy consumption [43].

The general process of a VOD system entails the following [43]:

- Obtain the ventilation requirements of workers, machines and equipment.
- Determine the ventilation requirements per working district, development and section.
- Adjust the ventilation control devices according to the required demand while maintaining a dynamic airflow through the entire mine.
- Ensure that the overall ventilation of the system is distributed dynamically through the system.

VOD is the most popular method for improving the efficiency of an existing MVS. The aim is to operate the ventilation system as close to the demand limit as possible. Although VOD eliminates the overcompensational margin, it does reduce operational costs significantly. The ventilation requirements for workers, machines and equipment are determined by the mine’s performance needs and legislation requirements. The ventilation requirements for each working district, development and section are subject to the contaminants (discussed in Section 1.1) and limited by the minimum required airflow per period [17], [43].

Ventilation control devices can be adjusted manually according to the need when entering or exiting a working area. The ideal is, however, to have an automated or centralised system [43]. A good VOD system requires a sophisticated control system, constant underground airflow as well as condition monitoring. Modern simulation models and software packages are usually equipped to support VOD initiatives [17], [44].

Adjusting ventilation control devices in both the PVS and SVS is essential to the VOD strategy. The most common devices are the main fans and booster fans for the PVS and the auxiliary fan or district fan for the SVS. The most basic implementation of VOD is to deactivate fans when they are not required. Some mines apply this method to underground booster fans during peak periods to reduce operational cost [15], [43].

Fans usually operate at the same operating point for 24 hours a day [45]. Airflow is essential for the entire 24-hour cycle; however, the operating point (duty) of the fan can be adjusted according to demand using dampers, variable speed drives (VSDs) or/and adjustable guide vane controllers [43], [44].

Fan dampers

A fan damper acts as a throttle device that increases the system pressure but reduces the delivery air quantity. Although the device is easy and cheap to install, a large reduction in quantity can only results in a small reduction power. For example, using a damper control on a centrifugal fan, with its operational point on a positive power slope, to reduce the airflow quantity by 40% could result in a 12% power reduction. Therefore, dampers are not commonly used as control devices in practice due to the massive airflow compromise for little cost savings [13].

Variable speed drive on fans

A VSD is a device that varies the delivered rotational speed of a motor. The motor can be varied either by a mechanical or electric VSDs. Mechanical VSDs are easy and relatively cost-effective to install. A mechanical VSD adjusts the mechanical coupling ratio while the motor operates at a constant speed. The motor, however, still operates at its full power rating. The modern method is to use an electric VSD to vary the electrical frequency, which is used to drive the alternative current induction motor. The motor requires less power since the motor speed is reduced, which in turn decreases energy consumption [38], [46], [47].

VSD applications in mines vary from grinding mills, hoists, electric motors, compressors to fans [46]. VSD is used in ventilation systems to vary the speed of the airflow controllers according to the ventilation load of the system. A VSD connected to a fan is the most efficient way of controlling the airflow quantity of the fan [13]. The VSD reduces the fan’s motor speed until the airflow quantity is just adequate for the demand during a specific period. The motor operating at a lower speed requires less energy to operate, thereby reducing the fan’s operational cost [30].

VSDs can be applied to both SVS and PVS fans. The feasibility of a VSD is, however, dependent on the system's loading factor, payback period and planned operational lifetime [30], [47].

Constant changes in the development working areas require a constant duct length adjustment to the auxiliary fan duct system (as discussed in Section 1.2) [48]. The typical application of such a system is to run the auxiliary fan at full capacity to compensate for the worst-case scenario, namely the duct resistance will be at maximum at the maximum legal duct length. Mulder, Fourie and Stanton considered using a VSD to control the airflow of the auxiliary fan according to the development progress [21]. This system investigated in the study consisted of five auxiliary fan duct systems that ventilated four faces. Mulder *et al.* simulated an 83% reduction in energy consumption over nine months by simply replacing the ducts with less resistant ducts, changing the five-duct fan system into one-duct fan system, and using a VSD to control the fan speed. The VSD was controlled to overcome the increasing duct resistance but to still maintain the maximum allowable wet-bulb temperature and minimum allowable airflow at the face [21].

Not all district working areas in a mine are actively occupied or mined at the same time. VSDs are commonly used to control the speed of district fans when a working area is unoccupied, which reduces the airflow to the unoccupied districts. Since the SVS branches off the PVS, more air is available to the active occupied districts. Some mines have a review meeting at the beginning of each week to discuss the manual operation of VSD. During the meeting, it is determined which district fans should be active and which district fans should be switched off [7].

VSDs can further be controlled using monitoring devices to determine real-time ventilation demand. Malmberget mine uses carbon monoxide sensors and a mobile transducer to control the airflow quantity of a district working area [49]. This application allowed the mine to reduce operational cost by 29% from their previous practice of running fans at full capacity [49]. Creighton mine programmed their system according to the four operating periods of the mine's working cycle [43]. A Swedish mine implemented a computer system on the ventilation system that runs the primary fans according to the demand and the secondary fans at 50% during the start of a shift. Transmitters detect when machinery start and run the secondary fans at 100%, and adjust the primary fans according to the demand [43].

Fans with inlet guide vanes

Inlet guide vanes (IGVs) are vanes installed upstream directly in front of a fan's impeller. These IGVs, if partially closed, create a swirl of air in the direction of the impeller rotation. The swirl reduces the pressure and volume flow (quantity) and moves the operating point of the fan. As a result, the power required by the impeller reduces, which in turn reduces the energy consumed by the fan [6], [13].

It is highly unlikely that the PVS airflow quantity requirements will remain the same during the lifetime of a mine. Therefore, it is a requirement that the operating point of main fans can vary. Even though VSDs are the most efficient way of varying the operating point of a fan, it is also quite expensive to retrofit a VSD to a fan. Adjustable IGVs are therefore used to change the duty point of main fans as the mine develops through its lifetime [6], [13].

IGVs can be implemented on axial and centrifugal fans. The IGV angle of axial fans can only be changed when the fan is switched off. To adjust the angle while a fan is running is significantly more expensive than making adjustments while a fan is off. Centrifugal fans with IGVs, in contrast, can change their angles while the fans are running. Centrifugal fans usually have an IGV for start-up or shutdown purposes. The IGV is fully closed or isolated before the fan is started. This keeps the fan motor current low and gives the motor time to accumulate speed [6], [13]. The IGV controller is extremely efficient between 80% and 100% airflow. The efficiency of the system does, however, reduce if the airflow is required to be below 80% [13].

Du Plessis, Marx and Nell stated that most South African deep-level mines implement load clipping methods by using IGVs to reduce the power consumption of fans [29]. A study conducted by Venter discovered that a platinum mine in South Africa was overventilated [13]. Possibilities for reducing the energy use of the system were investigated. IGVs were used to adjust the operating point of the fan, which reduced power consumption by 13.4% [13]. Poope *et al.* achieved savings between 27% and 29% on three projects that used the IGVs of the primary fans to reduce the airflow quantity by 10% during peak periods [50].

Conclusion

VOD is the ideal method for improving the efficiency of an existing ventilation system. The limitations of VOD include the cost, feasibility and impact of the implementation. The feasibility of the system depends on the cost of the instrumentation and the payback period of the initial system. Considering the feasibility of VOD for old deep-level mines, one should understand that these mines were designed before the latest efficient technologies and monitoring systems were developed. Most fans do not have VSDs, and the only fans with IGVs are usually larger primary fans.

Retrofitting an VOD system to an integrated ventilation system require environmental instrumentation in each working area of the mine to monitor the ventilation demand. The studies investigated in this section focus on localised airflow control. Integrated VOD air flow control require a bulk amount of monitoring devices.

The lack of monitoring devices increases the expenses of the initial VOD implementation. The lifetime of a deep-level mine is not that significant when compared with a relatively young mine. The payback period of the VOD initiatives on deep-level mines is, therefore, limited due to the significant initial expenses and shorter expected lifetime of the mine.

1.7.4 Efficiency of ventilation control devices and configuration of these devices

As discussed in Section 1.2, airflow through a tunnel experiences resistance, which leads to pressure losses. Ventilation control devices are divided into two types, namely production airflow devices and distribution airflow devices. Production airflow devices (active regulators) overcome resistance, while distribution airflow devices (passive regulators) create resistance. Changing the efficiency or configuration of these ventilation control devices can lead to an optimal quantitative and qualitative air distribution, which is more energy efficient [51], [52].

Improving the efficiency of ventilation control devices

Each ventilation control device has a specific airflow control efficiency based on how well the control device fulfils its purpose. For example, how effective a door is at preventing airflow (distribution device) or how efficient a fan is at using its input power to produce airflow (production device). Improving the overall efficiency of these ventilation control devices requires changes to the control devices themselves. Improving the efficiency of airflow distribution control devices is logical and easy to implement. For example, if a ventilation seal is inefficient, it has to be resealed with ventilation foam to restrict airflow leakage. However, improving the efficiency of a production airflow control device requires more complex considerations. Fans are the most common devices used in mines to produce airflow. Improving the efficiency of an existing fan requires changes to the fan blade angle or impeller itself. Consequently, although replacing an inefficient fan with a more efficient fan improves efficiency, it is more expensive [29], [35].

The current practice of mine fan selection involves choosing a fan with a maximum pressure-quantity specification to compensate for all mining periods. Therefore, selecting a fan based on its characteristic curve, blade angle, and working area requirements for each period can lead to significant savings [16]. Consider the scenario discussed in Section 1.7.3 under “Variable speed drive on fans” in which auxiliary fans were run at full capacity to compensate for the maximum resistance of the developing working area. As discussed in Section 1.7.3, Mulder *et al.* used VOD with VSDs to improve this system [21].

Acuña *et al.*, however, used a mixed integral programming model to select the most efficient auxiliary fan with its corresponding operational settings per period to improve a similar system [16]. The type of auxiliary fan determines the characteristic curve of the fan, whereas the blade angle determines the operating point of the curve, which depends on the resistance of the system. The case study of Acuña *et al.* resulted in a maximum predicted energy saving of 33.2% [16].

Changing the configuration of ventilation control devices

Production and distribution airflow control devices are usually placed in specific locations to serve a specific purpose for a specific period. Locating these devices strategically can, however, improve air distribution and energy efficiency. The location of distribution airflow control devices is usually based on the required airflow per tunnel in the mine. These passive control devices are evident in both the PVS and the SVS, and their positions depend solely on the tunnel’s required airflow.

PVS production airflow control devices, such as main fans and booster fans, are used to control the airflow of a section or sections. The main fan position, however, is constrained to the mineshaft’s position, which is considered unchangeable. Consequently, the booster fan’s position is more flexible since the only criteria are that the fan should be in series with the main fan and that it should be connected to the required section or sections.

Lowndes and Yang used a generic algorithm to determine the ideal position for new booster fans and the optimum location and size of regulators in an MVS [51]. A case study conducted on Peak gold mine investigated the possibility of installing additional booster fans on the existing MVS.

The optimum location, number and duties of the booster fan applications with suggested main fan optimum duties were analysed. The predicted outcome of the algorithm allowed the overall operational costs of the mine fans to reduce with almost half a million rand [51].

Lowndes and Yang conducted a second case study on the El Indio mine where the expansion of the mine resulted in a shortage of airflow [51]. They realised that additional booster fans were required on various levels to assist the main fan. The implementation of one to five booster fans was investigated at 16 potential locations. The algorithm predicted that three booster fans at three different locations at specific operating points would be sufficient for ventilating the workings as required [51].

SVS production airflow control devices (auxiliary fans and district fans) are used to control the airflow of the working areas. The auxiliary fan position is, however, constrained to the development faces and is considered unchangeable. Consequently, the district fan’s position is more flexible since the only criterion is that the fan should maintain airflow through the district working area.

Kozyrev and Osintseva used an automated design planning algorithm to improve the air distribution of two mines with poor SVS air distribution [52]. The planning algorithm kept the number of main fans constant and used district fans and brattices to control the air distribution in the underground tunnel network. The planning algorithm suggested different configurations for ensuring sufficient airflow in all districts. The most feasible and efficient option required the user to either add or remove brattices or fans from the system [52].

Conclusion

Improving the efficiency or configuration of ventilation control devices can improve the energy efficiency of an MVS [51]. The strategies discussed in this section can be used to improve an existing ventilation system, but it would be quite expensive to replace all old fans or to change the entire configuration of a deep-level mine. Ideally, these strategies should be considered during the design or planning phase of an MVS.

It is also worth mentioning that it not always possible to implement the ideal predicted control device or configuration of a system. Acuña *et al.* mentioned that one should understand the complexity of moving devices underground and how the movement constrains fan selection [16].

For example, the implementation of underground fans can be limited by hoisting capacity, underground transport or available duct sizes [16].

1.7.5 Integrated ventilation simulation models

The PVS and SVS of an MVS have integrated influences on each other (as discussed in Section 1.3). The previous section's improvement strategies have all been based on the specific ventilation control device or localised improvements.

Integrated optimisation of a complex ventilation system can only be analysed using a simulation model [53]. Using a simulation model, a digital twin of an MVS can created, which represents the actual mine in appearance and behaviour [17]. The simulation model, therefore, allows for integrated predictions of control device changes, airflow investigations and problem-solving [8]. Using a simulation model to make these integrated predictions will require that the airflow and thermodynamic behaviour correspond with the mine. Simulation models used for these purposes are calibrated – meaning that the airflow of the model corresponds with the airflow of the actual mine. The calibrated simulation model is used to predict ventilation system behaviour when changes are implemented [29].

A simulation model's reliability and accuracy depend on the input values of the model [54]. The input values can either be measured manually or by using underground environmental monitoring sensors that are monitored on the surface. Real-time (live) version of simulation software can also be used if underground monitoring equipment is available. The real-time model simply updates and resimulates at short intervals using the sensor measurement data as constant inputs [17].

The ventilation system of a mine is implemented on a system that has been designed using the production requirements (as discussed in Section 1.2). The flexibility of the ventilation system is not the main focus during the design of the mine tunnel networks. Production changes during the operational life of a poorly designed mine can lead to inadequate airflow. Reconstructing such a system has significant cost implications [8]. Kocsis, Hall and Hardcastle stated that it is of utmost importance to integrate the production objective and overall MVS during the design phase of a mine. They accomplished this integration using a simulation model, thereby increasing PVS and SVS efficiency, combining production objectives and ventilation requirements, facilitating VOD initiatives or any new technologies and, finally, reducing power consumption [8].

Simulation models allow for planning and are used to predict changes in the MVS. Some localised studies discussed in Section 1.7.2, Section 1.7.3 and Section 1.7.4 used simulation software to analyse and predict the study outcomes [7], [16], [21], [43], [50], [51], [52]. Branny and Filipek used a simulation to predict an overlay ventilation system's airflow as well as the dust and methane concentration in drifts successfully [55]. Roghanchi, Kocsis and Sunkpal applied a simulation model to predict the comfort levels of mine personnel successfully based on maximum skin wetness and the sweat rate required to achieve it [2].

Increasing the air distribution of an existing MVS requires that air leaks are reduced and that airflow quantity control is improved in the working areas. A case study conducted on a deep-level mine predicted potential savings of 10.4 GWh per annum by improving an existing MVS with a simulation model [29]. Another typical implementation of integrated mine ventilation simulation models occur when a mine is expanding and large changes need to be investigated. Mines expanding beyond their expected lifetime require ventilation adjustments to accommodate the increase in ventilation demand. Simulation models are used to investigate cost-effective solutions for the problem before changing anything on the actual system [56], [57].

Conclusion

Simulation models are used for most mine ventilation-orientated investigations due to their integration capabilities. Ventilation simulation modules were used in the previous sections to investigate changes before implementing the changes on actual mines. Using a simulation model is an affordable and risk-free investigation method. However, creating and calibrating the simulation model may be time-consuming.

1.8 Problem statement and objective

The objective of an SVS is to ensure a sufficient quantity and quality of airflow in the working areas of a mine. Cool air needs to be distributed from the surface to the underground working areas to ensure a comfortable working environment for mine personnel. Contaminants influence the air quality of a mine. This makes underground air potentially dangerous to breathe, and extremely high temperatures may also lead to heatstroke. Not only are mining personnel's lives at risk, but production is also highly reliant on these conditions. Poorly distributed air in an MVS results in an oversupply of air to the ventilation system. The additional air is distributed through the MVS to compensate for the air wrongfully distributed. This practice consumes an unnecessary amount of electrical energy since more air than required is supplied to the system.

Electrical tariffs are the mining industry's most significant contributor to expenses. The management and sustainability of energy is the central focus of today's mines to reduce operational expenses. A small reduction ventilation airflow quantity has also proven to have a significant influence on the energy consumption of an MVS. The variety of strategies investigated improved air distribution with the result of energy efficiency improvement. However, deep-level MVSs have a few constraints that young developing mines do not have. The deep-level (old) mines usually lack the newest and advanced monitoring and control devices that exist today. The expected lifetime of these mines are rapidly reducing, and the payback period of long-term efficient system investments is limited.

Advanced modern improvement is, therefore, highly unlikely on deep-level mines since the entire infrastructure of the ventilation system needs to be changed. Consequently, deep-level mines usually have lower volumetric efficiencies than young developing mines (discussed in Section 1.5). This means that the air is poorly distributed throughout the ventilation system. Therefore, deep-level mines may have a high potential for reducing energy usage by improving the air distribution of the MVS.

The objective of this thesis is to create a feasible method for improving the air distribution of a deep-level MVS.

- The mine should be considered as an integrated system in which small practical changes is made over the entire ventilation system.
- The ventilation changes should be as cost-effective as possible and should remain within the operational standards of the mine.
- The combination of all these small changes should have a large effect on the overall air distribution of the MVS. Thereafter, the improved system should be considered for possible energy reduction and savings.

1.9 Thesis layout

Chapter 1

Background was given as to why subsurface mines require ventilation and how these ventilation systems work. The volumetric efficiency of mines was analysed with specific focus on deep-level mines. The operational cost of the mining industry and, more specifically, operational ventilation expenses were discussed.

A comprehensive literature study was conducted on existing ventilation improvement strategies. The problem was stated that the investment payback period for energy efficient systems of deep-level mines is limited due to the expected lifetime of these mines. The objective is, therefore, to create a feasible integrated air distribution improvement strategy that will apply ventilation changes as feasibly and cost-efficiently as possible.

Chapter 2

Chapter 2 discusses a general methodology for improving the air distribution of a deep-level MVS. The method requires a calibrated ventilation simulation model to predict improvement. The chapter is written in general to allow the reader to apply the method to any deep-level mine. The methodology describes how the simulation model is prepared and used to improve the air distribution of the mine. Recommendations are made throughout the chapter since not all MVSs are the same.

Chapter 3

The improvement methodology described in Chapter 2 is applied to a mine located near Carletonville, South Africa. The improvement possibilities are analysed for each section of the mine. The overall improvement and cost implication of the system changes are discussed.

Chapter 4

The thesis concludes by summarising the outcomes of Chapter 3. The feasibility of the improvement method is discussed with some recommendations.

CHAPTER 2: METHODOLOGY

2.1 Preamble

The ventilation network and air distribution of deep-level mines can be quite complex and difficult to comprehend. Improvement methods are usually localised or expensive to implement. Deep-level mines have a shorter possible payback period than young developing mines, which limits their improvement possibilities. As deep-level mines usually have low volumetric efficiencies, they may have a higher potential of reducing energy usage by improving the air distribution of the MVS [24].

A ventilation simulation model gives an overview of the layout of such a complex ventilation system. This study used a simulation model to consider the overall integrated MVS and the potential improvements thereof [53]. Improving the air distribution of the ventilation system improves the volumetric efficiency, thereby potentially reducing the operational cost [24]. An airflow calibrated model imitates the actual air distribution of the mine and has the capability to predict integrated airflow behaviours [8]. Comparing the current mine air distribution with required air distribution allows problematic areas to be identified. Applying basic ventilation knowledge and implementing integrated changes to a ventilation system improve air distribution over the entire mine.

2.2 Overview of improvement methodology

The ventilation terms discussed in Chapter 1 are used in the methodology to generalise the method for all deep-level mines. Figure 4 gives an overview of the methodology used to improve the air distribution of a deep-level MVS. The methodology allows one to understand the MVS before improving the actual system. The method consists of five key sections:

- Collecting the current mine’s ventilation information.
- Setting up and calibrating the simulation model.
- Verifying the simulation model.
- Comparing the required air distribution with the simulated air distribution to identify improvement possibilities.
- Representing and implementing the improvement plan.

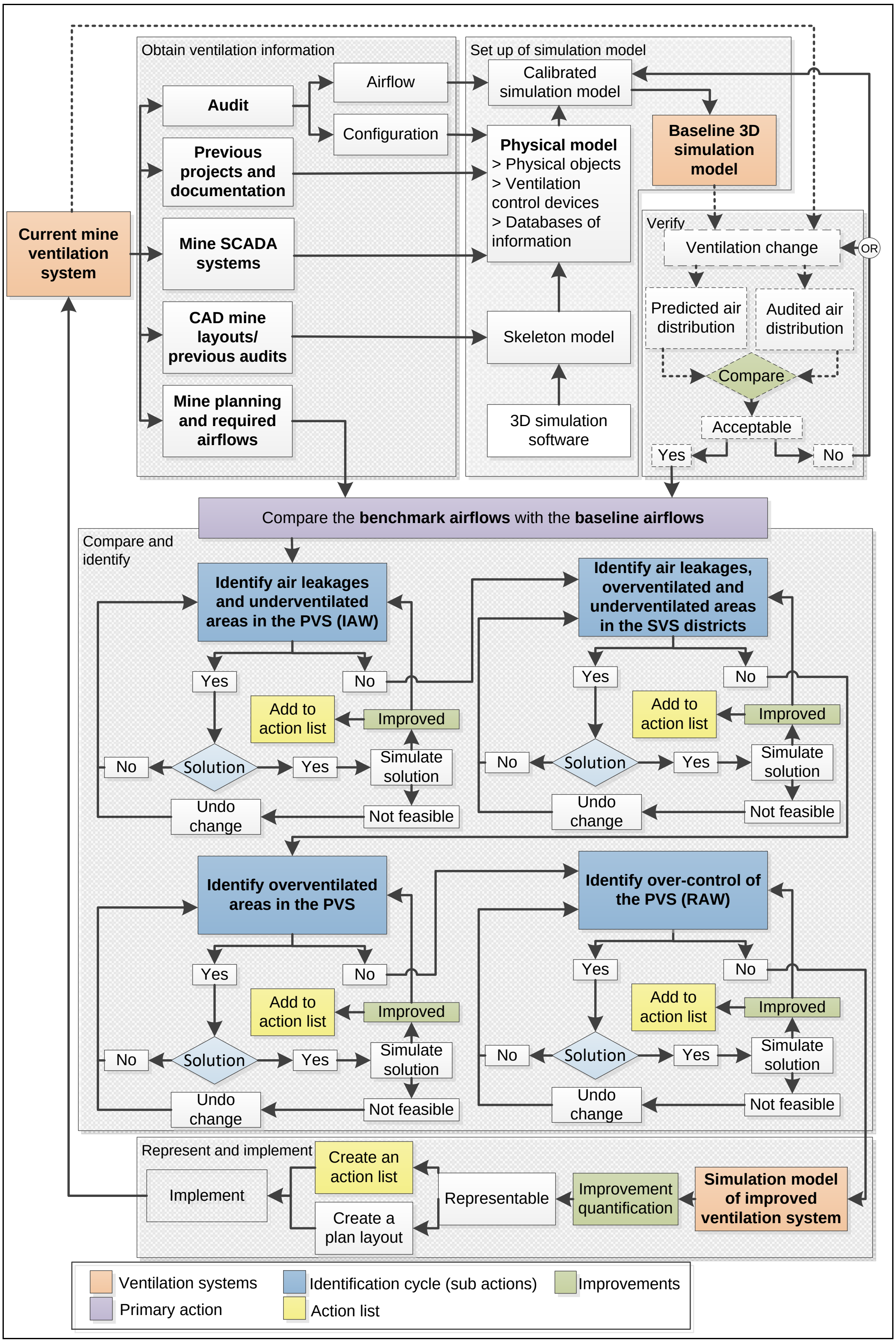


Figure 4: Methodology overview

CAD: computer-aided design; SCADA: supervisory control and data acquisition

The following sections discuss the improvement method in detail. General recommendations and considerations are also given considering that all mines differ regarding their information availability, configurational layouts, mining methods and ventilation systems.

2.3 Deep-level mine ventilation system information

Improving the air distribution of a large integrated complex deep-level MVS requires a comprehensive understanding of the system's appearance and behaviour. As much as possible information relating to the MVS needs to be assembled. The information required is divided into:

- Configurational information to create the skeleton of the simulation model.
- Airflow measurements to calibrate the simulation model.

The techniques used for obtaining this information are shown in Figure 5 and are discussed in subsequent sections.

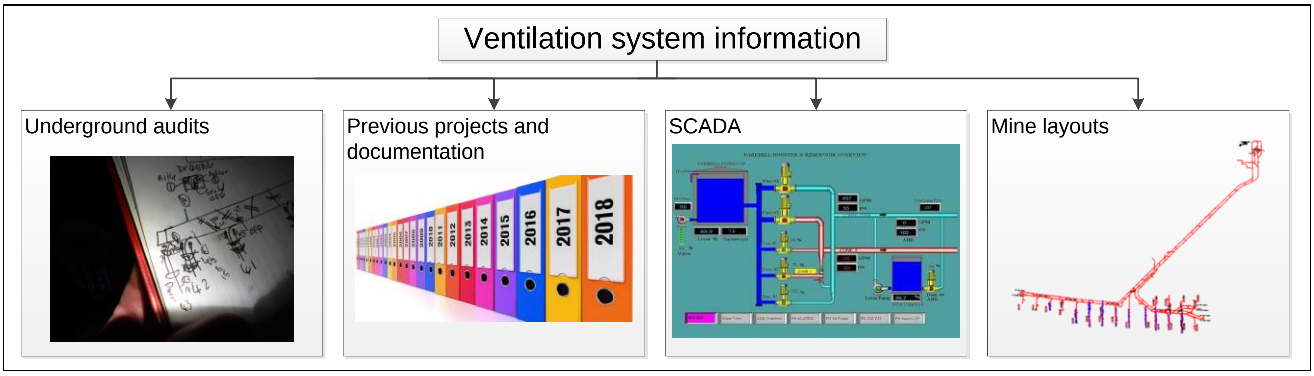


Figure 5: Techniques for obtaining ventilation information

2.3.1 Underground ventilation audits

New underground audits may be the most valuable and reliable technique for obtaining ventilation information. Consequently, auditing is also the most work-intensive and time-consuming technique. Depending on the availability of mine information, the information gathered using the three other techniques shown in Figure 5 may be limited or outdated. Both configurational information and airflow measurements can be obtained during underground audits. Although these audits are explained separately, it is more feasible to conduct these audits simultaneously.

Configurational audits

During a configurational audit, all physical objects in the underground tunnel system are mapped and defined on a two-dimensional drawing (example shown in Figure 6). The ventilation control devices (described in Section 1.2) are the essential objects to consider during these audits. These devices have a direct influence on the air distribution of the system and are essential for the simulation model to imitate the system's behaviour.

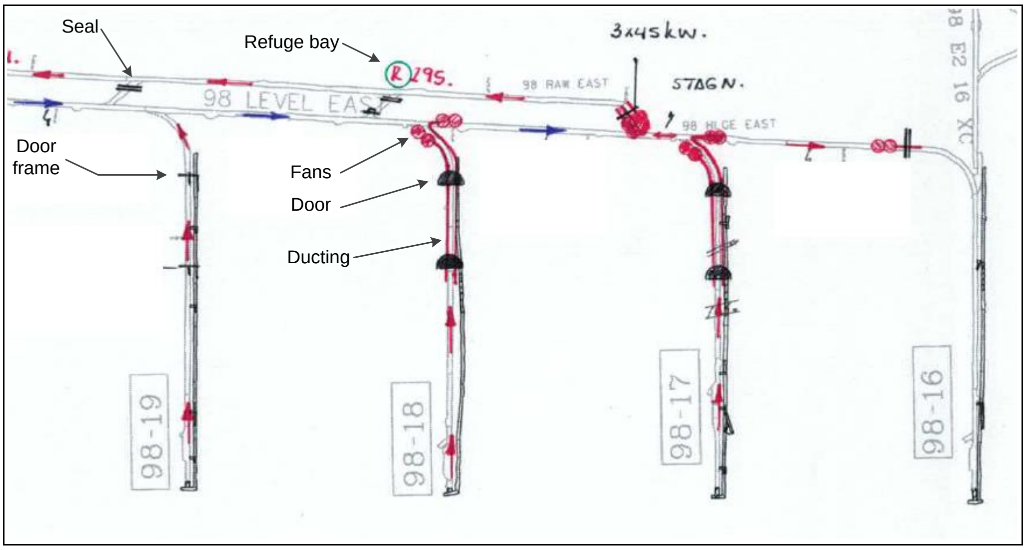


Figure 6: Typical example of a configurational audit

A typical list of objects audited during these audits is given in APPENDIX B. However, special emphasis is placed on the following ventilation control devices discussed in Section 1.2:

- Location and condition of fans, doors, stoppings, regulators and seals.
- Size and operational status of fans and ducts.
- Size and operational status of BACs and their fans.

It is recommended that additional information such as stores, refuge bays, workshops and resting places are also noted. Although the additional information does not affect the air distribution of the mine directly, it may influence the problem-solving process. The additional information should act as an informative database for the mine. For example, consider a scenario where a tunnel that is sealed off at multiple connections needs to be opened. Removing a connection with a single wall is more feasible than removing a connection with an old refuge bay.

Airflow audits

Mass flow is the main parameter analysed when improving the air distribution of an MVS. Mass flow is defined as the total amount of air in kilograms that passes a fixed point per second. Airflow (mass flow) at a fixed point depends on the cross-sectional area, air moisture content, air velocity, and air conditions, such as pressure and temperature, at that point. Consequently, airflow cannot be measured using a single measuring device. The following measuring devices are required to obtain the relevant parameters at each measuring point underground:

- Barometer – barometric pressure (Pa)
- Anemometer – air velocity (m/s)
- Whirling hygrometer – wet-bulb temperature ($°C$)
- Whirling hygrometer – dry-bulb temperature ($°C$)
- Laser distance meter – haulage (tunnel) width (m)
- Laster distance meter – haulage (tunnel) height (m)

Thereafter, psychrometric calculations should be used to calculate the mass flow of dry air at each measuring point. Measuring points should be selected based on an audit area that acts as a black box. The audit area (black box) has connections where air either enters or exits the area. The sum of the air entering the black box should equal the sum of the air exiting the black box (see APPENDIX C for more details). The black box method simplifies the audit process and allows one to verify the accuracy of the audit measurements. Figure 7 shows the principle of an audit area, with n-amount of measurement points, and how the measurements are verified.

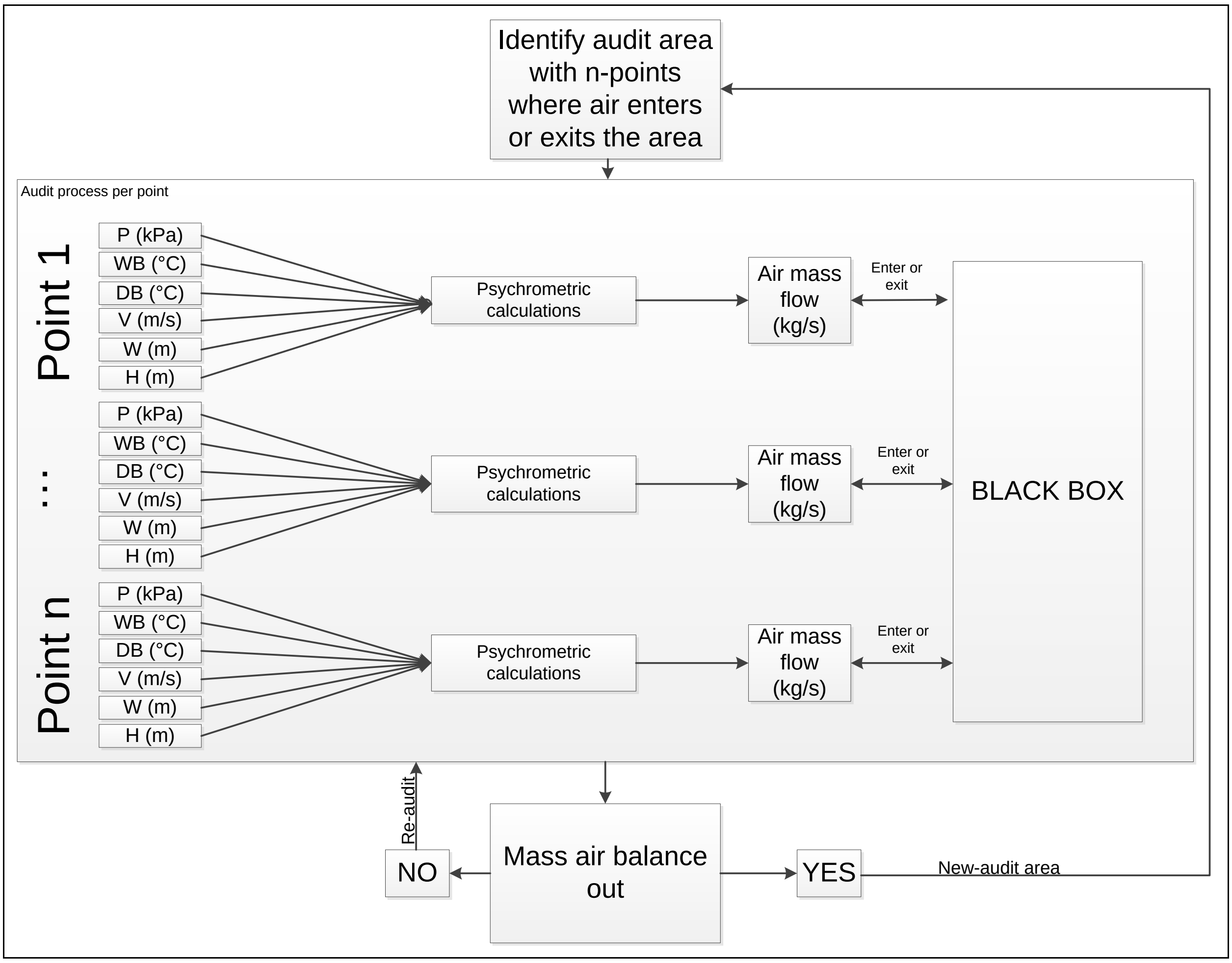


Figure 7: Measurement points of an audit area that acts as a black box

P: pressure; WB: wet-bulb temperature; DB: dry bulb temperature; V: velocity; W: width; H: height

An example of a practical application of an audit area (black box) and its corresponding measuring points are shown in Figure 8.

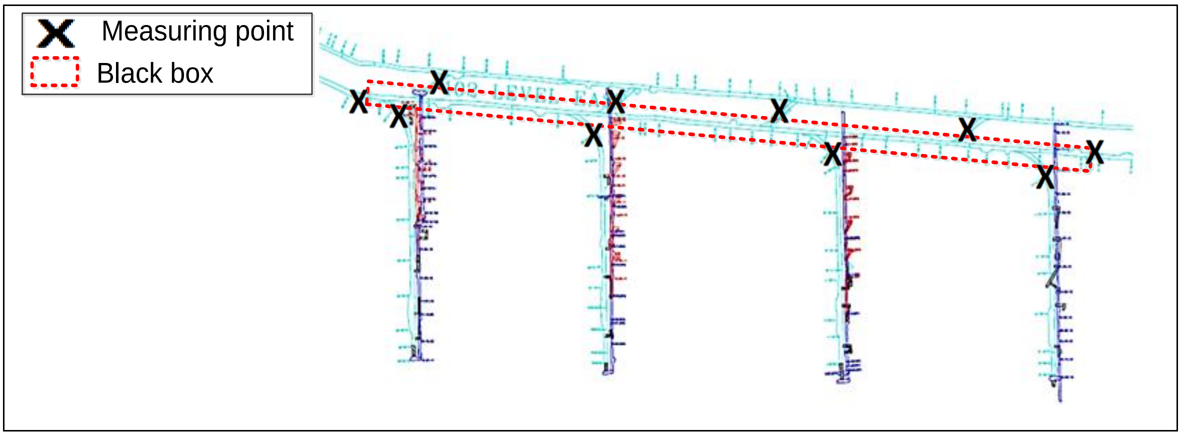


Figure 8: Practical example of a black box and measuring point selection

Each audit area is connected to an adjacent area or areas. These audit areas should correspond with one another when combined as an integrated system. For example, considering a mine with three sections with n-amount of workings each, the typical breakdown (neglecting any exceptions) of the audit areas would appear as shown in Figure 9.

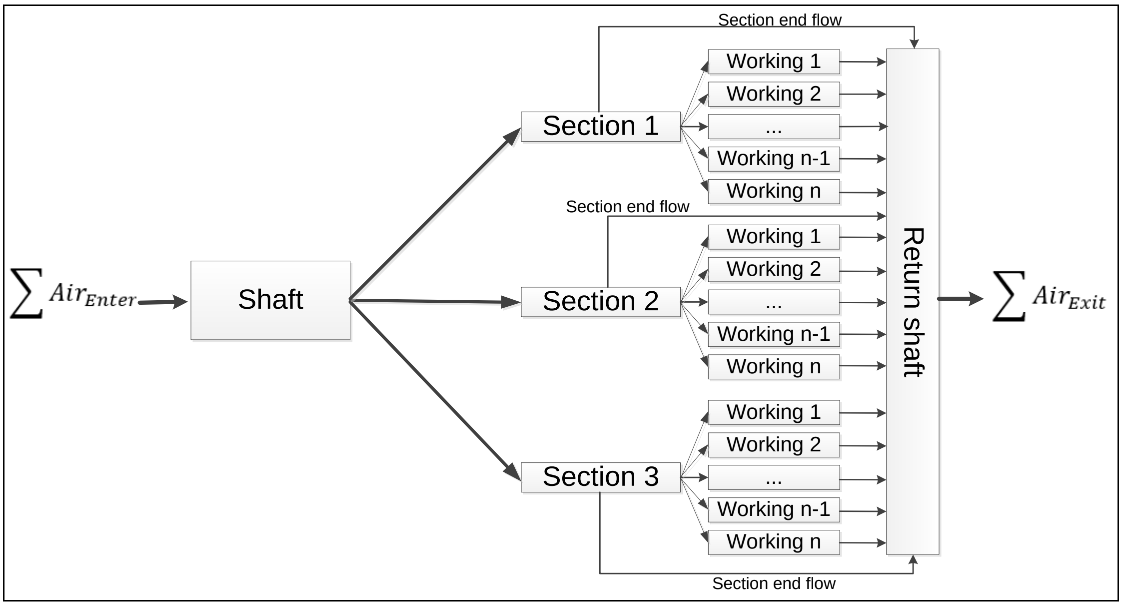


Figure 9: Ventilation system black box breakdown

Each square in Figure 9 represents an audit area black box, and each of these black boxes can be represented in Figure 7. The black box process is the first step of simplifying and verifying the audit data before it is used in the simulation model. It is essential to ensure that the data is reliable before the simulation model is calibrated since the recalibration process can be time-consuming.

Whenever the audit data black boxes do not balance, investigations, re-audits or assumptions are required.

2.3.2 Previous projects and documentation

Previous sealing or large ventilation projects typically require ventilation-related information. These projects usually trigger localised detailed ventilation investigations that refer to limitations and findings not documented anywhere else. Although these documents provide limited information, they can still contribute additional information to consider when solving distribution problems.

2.3.3 Supervisory control and data acquisition systems

The supervisory control and data acquisition system, commonly known as the SCADA system, is a software system used to monitor and control processes [58]. SCADA systems are used by underground mines to monitor and control pumps, turbines, valves, fans, fridge plants etc. SCADA systems require a connection from each monitoring point to the central control room. The SCADA systems of modern mines are sophisticated and have a variety of monitoring points. However, they can be limited on old deep-level mines to specific data due to historical old systems.

The number of monitoring points of a SCADA system will differ from mine to mine. Using a SCADA is an easy swift method for obtaining information. The system allows for an integrated view of the overall mine at a specific point in time. Using information gathered through the SCADA system is highly recommended if the information is reliable. The reliability of the data at specific points can be verified by comparing the SCADA data with logger data or audit data over a period. However, SCADA ventilation monitoring, such as airflow, is not that common on old, deep-level mines. The SCADA system in this study was used to obtain water flow rates, as well as inlet and outlet temperatures of the BACs (which are commonly monitored with SCADA systems in all types of mines).

2.3.4 Mine layouts

The mining industry uses mine layout drawings to keep track of developing working areas and to plan future mining sections. These layouts are created with computer-aided design (CAD) software and should be accurate to ensure that mining commences at the right geological area. Layout are usually three-dimensional (3D) and consist of a scaled tunnel network with coordinates. Layout drawings contain essential information that can be used for the simulation model skeleton since they present the most accurate visual representation of the current mine.

2.4 Setup of the simulation model

A 3D simulation model of a ventilation system allows for a simplified analysis and integrated system investigation. This improvement methodology requires an accurate and reliable 3D simulation software that can appear and act as the current mine. The setup of the simulation model starts with a 3D skeleton of the mine tunnel network. Underground tunnels act as a transportation medium for air and are the most important parameter in the simulation model.

The skeleton of the model should be created with the help of the CAD mine layout drawings (discussed in Section 2.3.4). This skeleton acts as the foundation to which both the visual and behavioural parameters are added in the simulation. All information collected are added to the skeleton model to create a digital twin of the mine. Figure 10 shows a typical example of a mine skeleton that was created using the CAD drawing as basis.

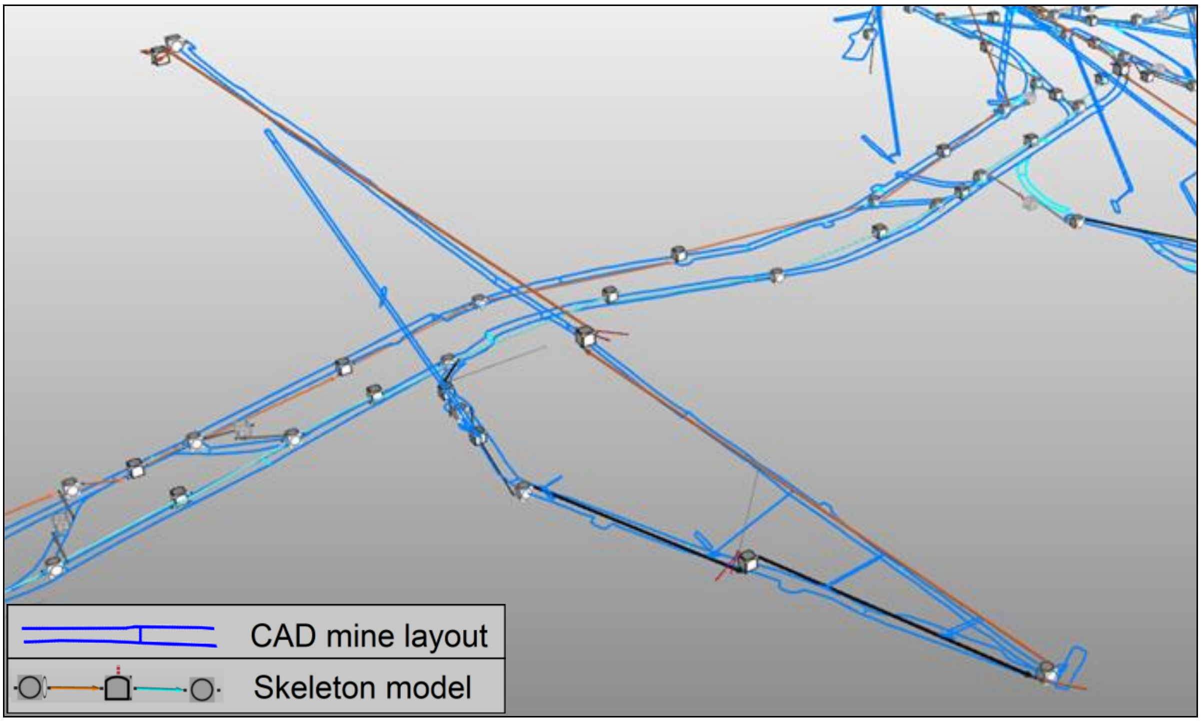


Figure 10: Skeleton model created from CAD mine layouts

The completed simulation skeleton represents the mine with all tunnels open and with no restrictions. Adding all the configurational and physical information allows the simulation to represent the mine physically as it is. The control devices represent the mine as it was during the audit. For example, if a ventilation door was audited as being open, it should be open in the model. At this point, the simulation model should represent the mine physically as it was during the audit.

2.5 Calibration of the simulation model

The simulation setup phase should now be concluded, and the model should represent the actual physical mine. However, this does not mean that the air will behave the same. Additional parameters such as tunnel sizes, obstructions, corners and cross-sectional area influence the airflow resistance of the actual tunnels. Air always follows the route of least resistance. Therefore, the simulation model needs to be calibrated according to the actual airflows of the MVS to allow the model to behave like the actual mine [29].

The air boundaries (surface intake and return of the system) of the simulation model are constant values and should be set to the actual values measured. It is essential that the ventilation control devices should have the correct specifications and operational status. The water boundaries of the BACs (if applicable) should be set to the actual measured values. Thereafter, the resistance of each branch in the model is varied to achieve the corresponding airflow audited in that branch. The air distribution of the model is therefore calibrated to correspond with the air distribution of the actual mine. Not much attention is given to the calibration of the temperatures of the mine since the objective of this study is to improve air distribution.

The calibration of the ventilation system has five critical parts to consider:

1. PVS intakes:
    a. The total amount of air entering the mine.
    b. Downcast shaft air distributed per section.
    c. Intake air distribution of each section.
2. PVS returns:
    a. The total amount of air exhausted out of the mine.
    b. Air returned to the upcast shaft per section.
    c. Return air distribution of each section.
3. SVS branches off the PVS:
    a. District intake air distribution per section.
4. SVS returns to the PVS:

  a. District return air distribution per section.

5. BACs:
   a. Water inlet and outlet temperature.
   b. Airflow through BACs.

The simulation model should be calibrated at steady-state conditions since the ventilation information has been collected over an extended period. The fully calibrated simulation model can, therefore, be considered as an average of the air distribution of the MVS at a single point in time.

2.6 Verification of the simulation model

The best stable calibrated version of the simulation model should be verified. The verification process quantifies the calibration accuracy and predictive ability of the simulation model. The air distribution of a well-calibrated simulation model corresponds with the actual air distribution of the MVS if the control devices and configuration of both systems correspond [29]. Therefore, the simulation model is verified by applying ventilation changes on both the actual and simulated system and comparing their airflow behaviours.

The degree of correspondence between the actual and the simulated system is dependent on the reliability and period over which the ventilation information was obtained. The simulation model is calibrated at steady-state conditions based on information collected over an extended period. Airflow fluctuations during both the calibration and verification audits will affect the accuracy of the simulation model airflow behaviour. An unacceptable correspondence between the two systems will require recalibration or reverification.

Verifying the simulation model with multiple or large consequential changes allows for a more accurate verification process. Verifying the effect of a small change does not have an integrated effect on the system and will consequently not be a reliable verification of the model's overall accuracy. The following recommendations are made when selecting a verification change to apply to the system:

- The change should have a noticeable effect on the air distribution.
- The change should preferably not affect the production area (return airflow changes are recommended).

- The change should have large consequences over more than one section (to verify the integrated system).

From this point on, the verified calibrated simulation model is considered the baseline of the study. This baseline model represents the current MVS. All deviations from this system are considered as improvement or analytical scenarios.

2.7 Mine planning and airflow requirements

The quantitative objective of an MVS is to ensure an adequate airflow quantity in the working areas of the mine (discussed in Section 1.2). Each working area in a mine has an ideal required airflow for maintaining good environmental conditions. Therefore, the required airflow of the SVS working area is identified as the primary benchmark for the overall air distribution requirements of the mine. The mine's ventilation personnel can provide a list of airflow demands for each working area of the mine. The airflow demand per working area is part of the SVS (as discussed in Section 1.3). The airflow availability of the SVS depends on the PVS (discussed in Section 1.3). Therefore, all ventilation system airflow requirements can be calculated from the working area requirements.

Consider Figure 11, as an example, which represents the mine as four sections. The purpose of the ventilation system of each section is to supply a sufficient quantity and quality air to the SVS (district and development) workings. The required airflow for the SVS is then obtained, and backwards mass balances are used to determine the PVS requirements.

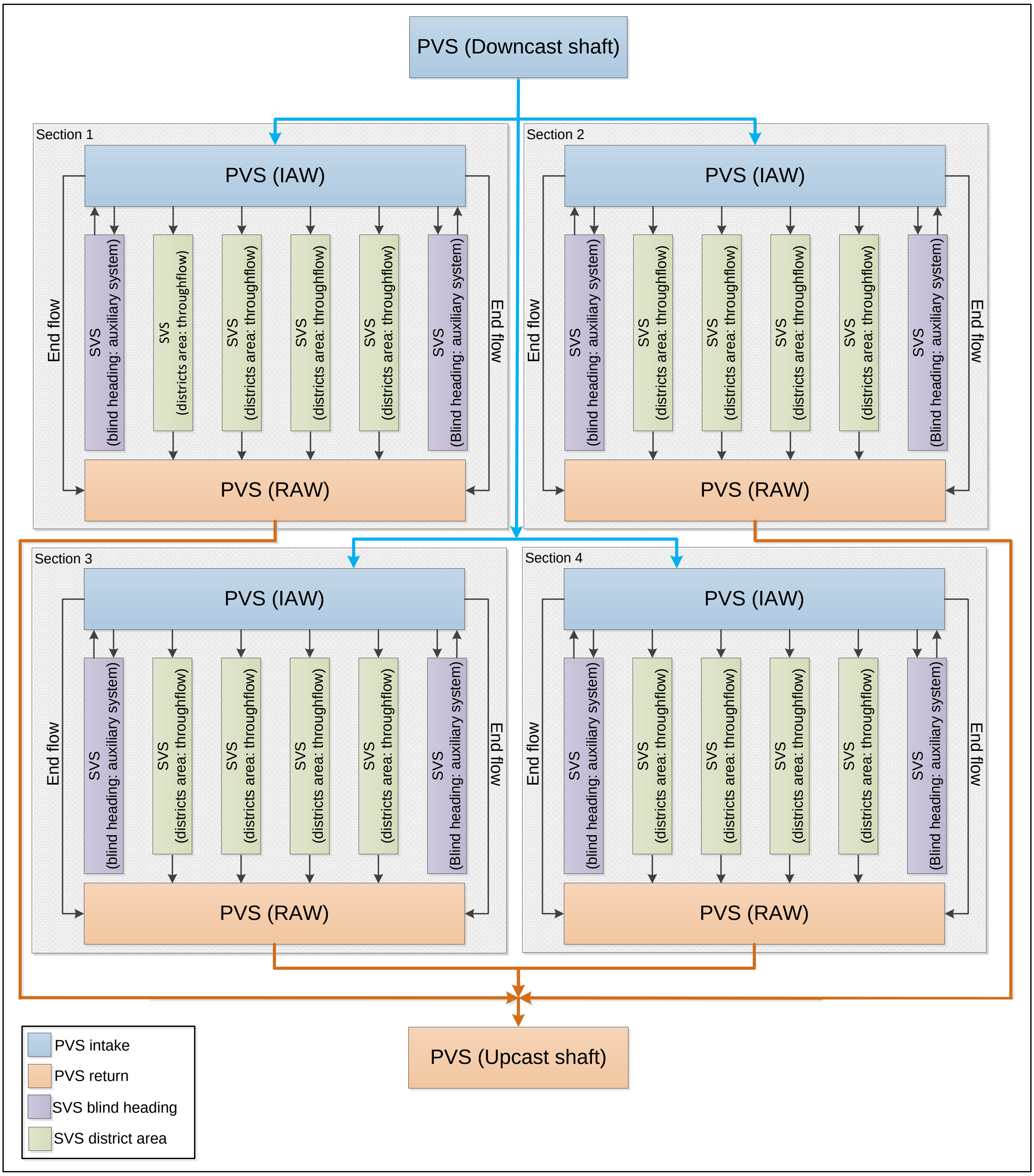


Figure 11: Ventilation systems collaboration and requirements

In general, consider a mine with:

- **i**-amount of sections;
- **n**-amount of district mining areas per section;

- **h**-amount of end flows per section; and
- **p**-number of blind developing headings per section.

The airflow requirement for the two working areas (district and developing) are obtained from the mine ventilation personnel. These airflow requirements are considered as the constants, namely $C_{DA_{k,j}}$ and $C_{BH_{k,g,t}}$

$$\begin{aligned}&\textit{where section } \boldsymbol{k} = \mathbf{1} \textit{ to } \boldsymbol{i},\\ &\textit{district areas } (DA)\ \boldsymbol{j} = \mathbf{1} \textit{ to } \boldsymbol{n} \textit{ for each section},\\ &\textit{blind headings } (BH)\ \boldsymbol{t} = \mathbf{1} \textit{ to } \boldsymbol{p} \textit{ for each section},\\ &\textit{and end flows } (EF)\ \boldsymbol{g} = \mathbf{1} \textit{ to } \boldsymbol{h} \textit{ for each section}.\end{aligned}$$

The airflows requirements of the development areas can be used to determine the end flows per section:

Equation 2: End flow requirements

$$EF_{k,g} = \sum_{t=1}^{p} C_{BH_{k,g,t}}$$

The required district airflows together with the end flows can be used to determine the PVS (IAW) requirements:

Equation 3: PVS (IAW) requirements

$$PVS\ (IAW)_{Section\ \boldsymbol{k}} = \sum_{j=1}^{n} DA_{k,j} + \sum_{g=1}^{h} EF_{k,g}$$

Thereafter, the requirements for the PVS (IAW) can be used to determine the overall requirements of the system:

Equation 4: MVS overall requirements

$$PVS_{Downcast\ shaft} = PVS_{Upcast\ shaft} = \sum_{k=1}^{i} PVS\ (IAW)_{Section\ \boldsymbol{k}} = \sum_{k=1}^{i} PVS\ (RAW)_{Section\ \boldsymbol{k}}$$

The required (ideal) air distribution of the MVS is now considered the benchmark of the study.

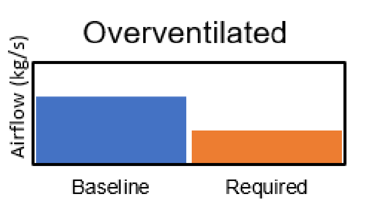
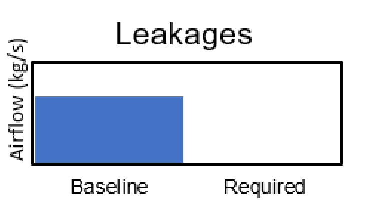
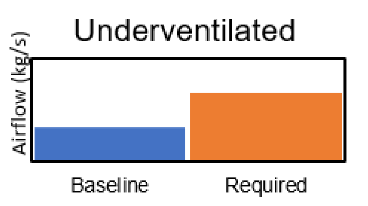
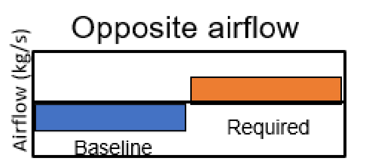
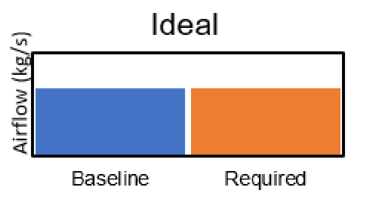
2.8 Comparison, identification and improvement

2.8.1 Preamble

Comparing the baseline air distribution (simulation model) with the benchmark (required) air distribution enables one to identify areas with overventilation, underventilation, opposite airflow, leaks or restrictions. A column chart is the most appropriate way of comparing the baseline with the benchmark airflows of a specific branch. Table 4 represents the six typical airflow comparison scenarios when comparing airflows.

The first five comparisons in Table 4 represent the five problematic scenarios one can identify in an MVS. A basic ventilation knowledge of ventilation control devices is required to resolve these problematic areas. The baseline airflow can be improved to correlate with the required airflow by adding, removing or adjusting the settings of ventilation control devices. Each problematic scenario in Table 4 also has a list of possible solutions to resolve the problem. However, these are just common recommendations since the most feasible solution must consider the mine's practice and the availability of the type of control devices, and must further investigate the working area and future plans therefor.

Table 4: Six typical airflow comparisons

Airflow comparisons			
	**Possible solutions**		**Possible solutions**
	Close or install door		Close or install door
	Install brattice		Install stoppings
	Install regulator		Install seals
	Switch fan off		Close open manhole
	**Possible solutions**		**Possible solutions**
	Remove or open door		Open or remove door
	Remove brattice		Remove stoppings
	Remove regulator		Remove seals
	Switch fan off		Open manhole
			Switch fan on
	**Possible solutions**		**Possible solutions**
	Close or install door		None required
	Install fan		
	Switch fan on		
	Install DF system		

DK: district fan

The MVS is a complex integrated system. Improving problematic areas at random may require reimprovements and reconsiderations throughout the process. Improving the system in the correct order of dependency will allow for integrated changes with few reimprovements and reconsiderations. The order in which the comparisons and improvements are done is essential for an integrated system since ventilation changes will result in a chain reaction. Four cyclical identification processes are used to improve the system in the correct order of dependency. By comparing the baseline with the benchmark, the cycles are used to identify a specific problem in a certain area. Possible solutions are then simulated, and the action list is updated with the most feasible improvement solutions. The cycles are repeated for each system until no more problems are identified.

2.8.2 Air leakage and underventilated areas in the PVS (IAW)

The air that is casted down into a mine is split between the IAWs of the section. The downcast air distribution of the baseline is compared with the benchmark air distribution. This identification cycle considers the decrease in airflow quantity as the air is casted down to the production block.

Old deep-level mines have several services and unoccupied mined-out sections above the production block. These IAW sections should either be regulated or sealed off to prevent large short circuits. Reducing the air losses in the downcast shaft before the production block increases the quantity supplied to the production block of the mine.

The volumetric efficiency of the downcast air distribution between IAWs differs from mine to mine. The number of sections and connections between the IAW and the RAW, above the production block, as well as the total fan pressure of the mine, influence the number of air leaks.

2.8.3 Over- and underventilated areas in the SVS

After implementing the changes discussed in Section 2.8.2, the air supplied by the PVS should now be sufficient to ventilate the various SVSs. This identification cycle compares the air distribution of the baseline SVS with the benchmark air distribution. Air leakage and over- and underventilated district areas are identified and improved according to the required airflow. This increases the air distribution between the PVS and the SVS. It is important to note that this only applies to throughflow (district) working areas. The development working areas are not considered in this cycle because the air used in development workings is returned to the PVS IAW (as shown in Figure 1).

The district workings, which are ventilated by the SVS, remove air from the PVS IAW to supply the workings. The airflow to these workings is considered critical for the health and safety of mine personnel. An inefficient quantity of airflow in the district working areas can halt production. Although an overventilated district working area is considered more acceptable than an under-ventilated working area, an excessively overventilated working area is not desirable either since the excess air could have been used more efficiently.

All overventilated areas have to be identified and improved before underventilated areas are resolved because decreasing the airflow of one area may increase the airflow in another. The most common solution for under ventilated areas is installing fans. The focus is on improving the system as much as possible with airflow distribution control devices (passive regulators) before installing additional production airflow control devices (active regulators).

2.8.4 Overventilated areas in the PVS (IAW)

The development end is the PVS IAW area after the last district working area (see Figure 1, C–D and G–F). This area is usually still in the development phase and only consists of a single through-flow to the RAW (called the end flow). The end flow can either be regulated passively or actively depending on the required airflow. Overventilated end flows should be avoided as much as possible since overventilation can waste useful air. This cycle identifies overventilated end flows by comparing the baseline end flow with the benchmark end flow.

2.8.5 Overcontrol of the PVS (RAW)

The distribution of the air from the surface to the working areas must now be optimised according to the best possible scenario. The simulation model in this identification cycle is not used to obtain the required airflows in specific areas (like the previous sections). However, changes should be applied to the simulation model to investigate return improvement initiatives, which should focus on lowering the resistance of the return system and identifying improvement initiatives.

The return systems of mines (especially deep-level mines) can overcontrol airflow. Tunnel and level statuses change as mines develop. Previously regulated tunnels can now be used as return tunnels. Air regulation in return tunnels wastes energy since fans need to overcome pressure losses. It is highly recommended that the potential of reducing the airflow in this cycle be investigated. Deep-level mines are often overventilated, which can go unnoticed due to the inefficient distribution. Therefore, this cycle is implemented last since the other cycles have already improved air distribution.

2.9 Improvement impact on the overall ventilation system

The simulation model should now reflect the desired air distribution of the MVS. However, the impact of the improved system needs to be quantified. The quantified impact is usually required to motivate the project potential to mine personnel. The quantification is based on the improved air distribution and the cost implications thereof.

The overall air distribution improvement is quantified by considering whether the airflow improvement is within a margin of 5 kg/s of the required airflow. The extent of the air distribution improvement is identified by evaluating the change in overall PVS air distribution efficiency. An improvement in air distribution efficiency will have a cost implication on the MVS (as discussed in section 1.6 and 1.7). The cost implications of the system improvement are quantified by comparing the baseline simulation fan power cost with the improved simulation fan power cost. Fan power costs are determined by using the relevant power utility tariffs. Initial project expenses to improve the MVS is also considered to determine the payback period.

2.10 Representation and implementation

The simulation model should now reflect the desired air distribution of the MVS. All the changes applied in the simulation model need to be implemented on the actual MVS. These changes should be simplified and listed in an understandable and representable form. Layouts are a common resource for planning and investigations. Create a mine ventilation layout with an improvement plan to supply the mine.

2.11 Conclusion

Preparation is required to use simulation software to predict the improvement of air distribution in a deep-level mine. The preparation done in this methodology entailed collecting data, creating a skeleton simulation model, and calibrating and verifying the model. The fully calibrated simulation model was used to predict the integrated air distribution improvement in the MVS.

A benchmark (required) air distribution was used as the target for system improvement. This target was based on the required airflow per working area of the mine as stipulated by ventilation personnel. Four identification cycles were followed in a strategic order to improve the air distribution of the ventilation system. Each of these cycles focused on a specific area of the mine and compared that area’s simulated and required airflow.

Basic knowledge of ventilation control devices was used to resolve problematic areas. Devices were added, removed, or settings were adjusted to enable the simulation airflow to correlate with the required airflow. The final identification cycle further focused on investigating potential airflow reduction initiatives.

CHAPTER 3: CASE STUDY

[1]

3.1 Preamble

This section investigates the air distribution improvements of a deep-level gold mine (hereafter Mine A), which is located near Carletonville, South Africa, and mines at a depth of 3.4 km underground. Mine A has been operating since 1978 and uses conventional mining methods in a sequential grid layout. The mine has two vertical shafts as well as two subvertical shafts. The mine mainly operated between 73 Level (L) and 98L at depths of 1.6–2.8 km below the surface. However, after a feasibility study was conducted in 1990, a deepening project was approved to exploit the western high-grade ore body. Both subvertical shafts were extended by approximately 500 m and four new production levels were established to develop the ore reef.

3.2 Overview of Mine A’s ventilation system

Figure 12 shows an overview of Mine A’s current ventilation system. All mining on the levels above 98L was ceased in 2014. The levels were either sealed off or used as service levels. Table 5 gives an overview of the mine’s current level classifications.

[1] Obtained from: Mining.com [Online]. Available: https://www.mining.com/

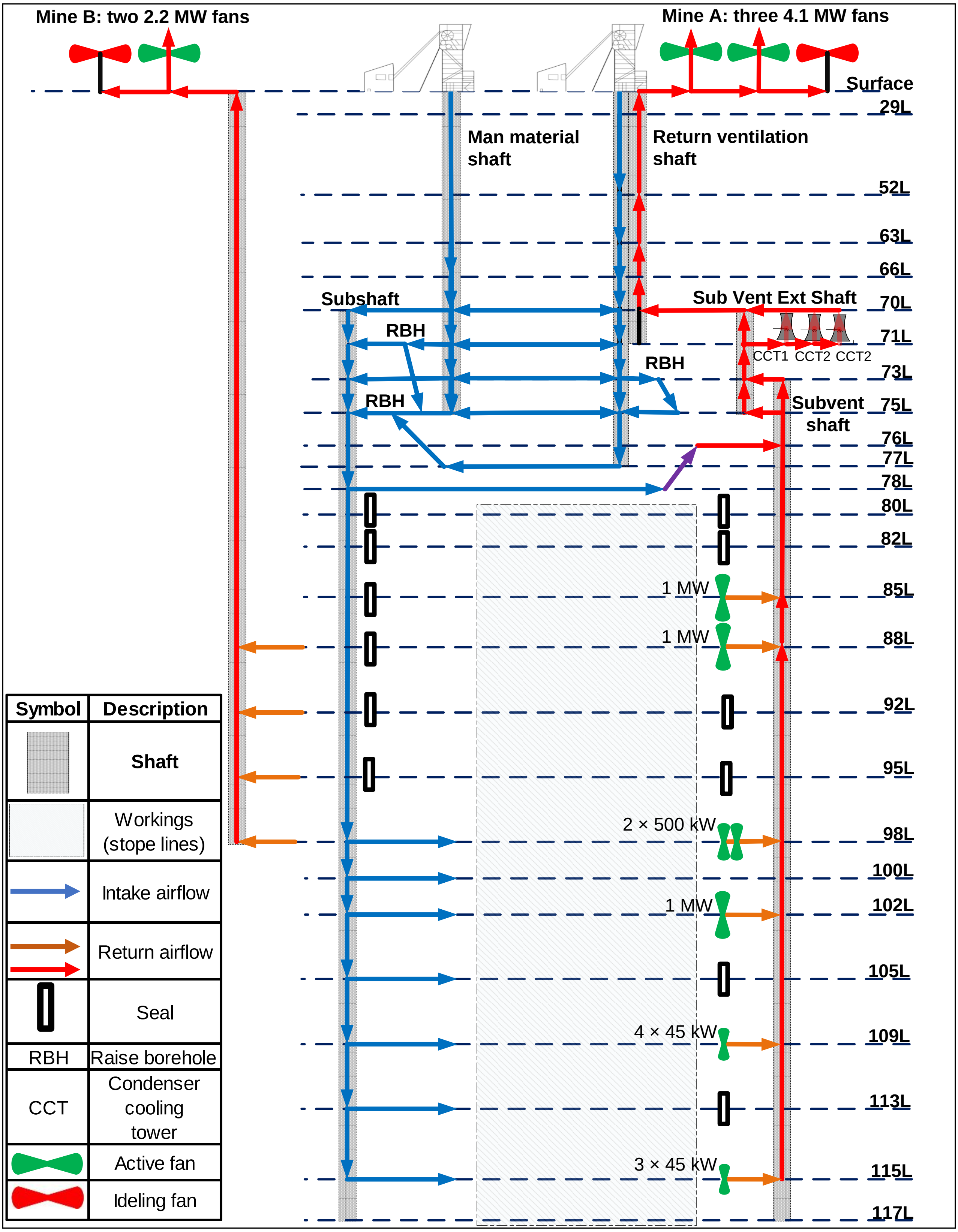


Figure 12: Overview of Mine A ventilation system

Table 5: Mine A level classifications

General classification	Level	Status	Description
Pumping	29L	No mining	Pump station
	52L	No mining	Pump station
Old mine	63L	No mining	Abandoned
	66L	No mining	Abandoned
Services	70L	No mining	Subshaft winders
	71L	No mining	Refrigeration
	73L	No mining	Hoisting
	75L	No mining	Pumping and BAC
	77L	No mining	Conveyors
Training	76L	No mining	Return air from 78L
	78L	Mining	Training centre
Old mine	80L	No mining	Abandoned
	82L	No mining	Abandoned
Return Block	85L	No mining	Return air from production block
	88L	No mining	Return air from production block
	92L	No mining	Return air from production block
	95L	No mining	Return air from production block
Production Block	98L	Travel way	Secondary escape for 102L
	100L	No mining	Refrigeration and BAC
	102L	Mining	Full production
	105L	Mining	Full production
	109L	Mining	Full production
	113L	Mining	Full production
Pumping	115L	No mining	Pumping station
	117L	No mining	Shaft bottom

Mine A casts a total of $\pm$1 070 kg/s air down by means of the man/material and return ventilation shaft. The return ventilation shaft is a dual shaft with the one half of the shaft downcasting and the other half upcasting. Mine A has three 4.1 MW surface extraction fans, of which two is actively operating with the third on standby. The fans are connected to the upcast half of the return ventilation shaft. However, the development of the new mine on the western side led to an additional return connection to Mine B. Mine B is an abandoned shaft 5 km west of Mine A with two 2.2 MW surface extraction fans: one fan is actively operating and the other fan is on standby (see Figure 12). Mine A uses additional underground booster fans to assist with the air extraction process. Table 6 indicates the primary fans used on Mine A (fans can also be seen in Figure 12).

Table 6: Mine A booster fans

Level	Description	Size (kW)	Contribution	Status
Surface	Mine A fan 1	4 100	Mine A return ventilation shaft	Essential
Surface	Mine A fan 2	4 100		Backup
Surface	Mine A fan 3	4 100		Essential
Surface	Mine B fan 1	2 200	Mine B return ventilation shaft	Essential
Surface	Mine B fan 2	2 200		Backup
76L	76L Backup booster fan – **not maintained**	2 × 520	Stope line airflow return to the ventilation shaft	Backup
85L	85L Booster fan	1 000		Essential
88L	88L Booster fan	1 000		Essential
92L	92L Booster fan – **not operational**	2 × 160		Not operational
95L	95L Booster fan – **not operational**	1 × 160		Not operational
98L	98L Booster fan	2 × 500 – double stage		Essential
102L	102L Booster fan	1 000	Level end flow return to the ventilation shaft	Essential
109L	109L Booster fan	4 × 45		Essential
115L	115L Booster fan	3 × 45		Essential

The natural rock temperatures of Mine A can reach up to 60°C and the heat load due to auto-compression alone can be up to 30 900 kW. South Africa's average summer ambient temperature can reach up to 24°C. Working conditions in the mining block are unbearable without additional air-cooling systems. Mine A has three surface BACs (man material 1, man material 2 and return ventilation), two underground downcast BACs (on 75L and 100L) and eight production block BACs (two on 102L, two on 105L, two on 109L and two on 113L) that maintain the temperature of the working areas within legal limits.

The CAD layout of Mine A show in APPENDIX D, which also describes the naming convention and terms used on this specific mine. The current air distribution of Mine A is not discussed in detail. However, it should be noted that there are two general return airflow paths, namely the end flow return system and the stope line return system (described in APPENDIX D).

3.3 Calibrated simulation model of Mine A

Process Toolbox (PTB) was used to create a 3D simulation model of Mine A's ventilation system, illustrated in Figure 13. Configurational data was used to make the model physically identical to the actual mine.

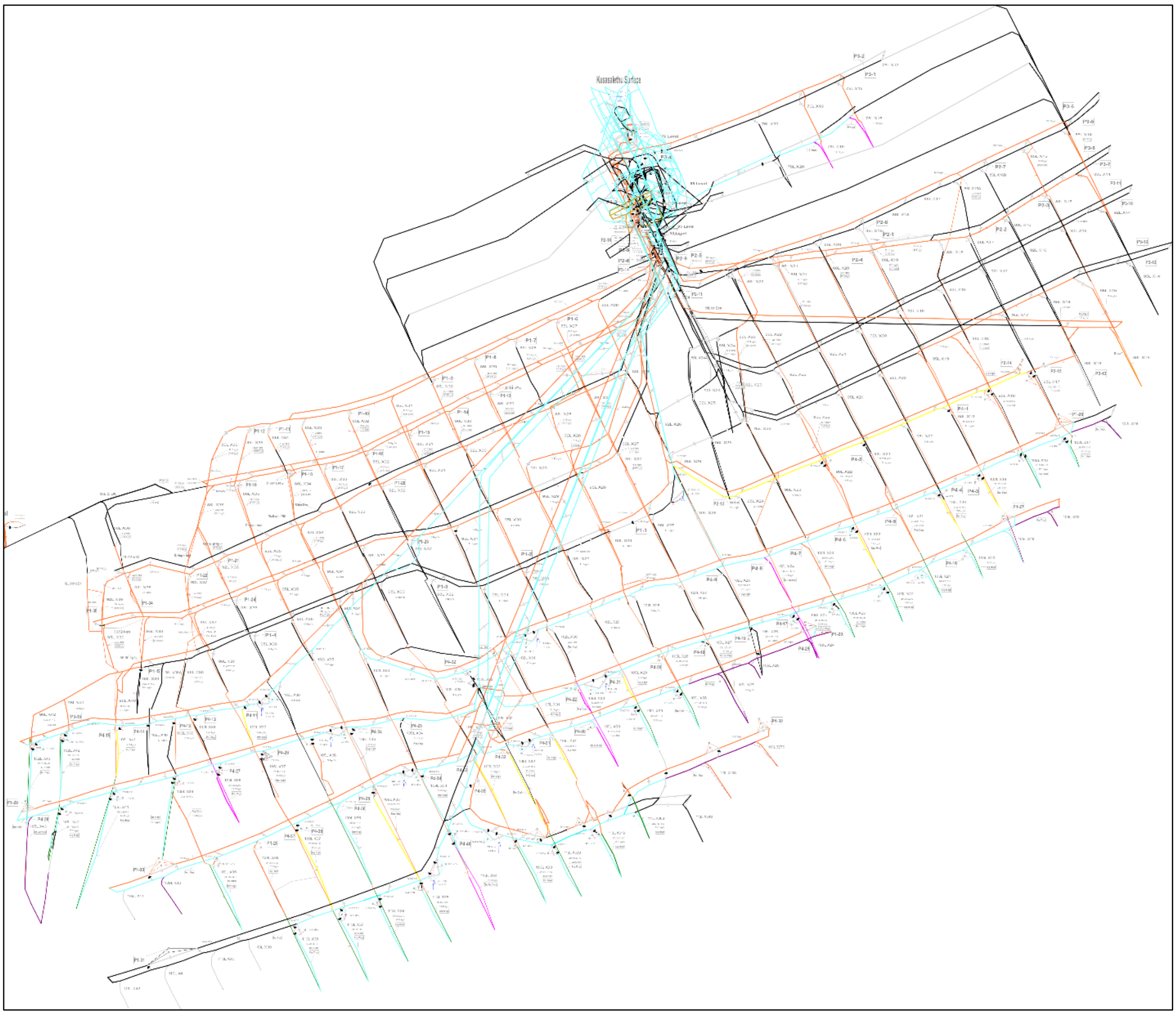


Figure 13: PTB simulation of Mine A connected to Mine B

The simulation model consists of 184 fans, 148 doors, 141 ventilation ducts, and 474 walls. The calibration of the model was based on 350 airflow data points audited over six months. The integrated analysis of these airflow data points showed that the PVS can fluctuate with $\pm$50 kg/s while the SVS can fluctuate with $\pm$20 kg/s. These fluctuations occur due to the dynamic behaviour and integrated dependency of the mine ventilation airflow over six months. The model was, therefore, calibrated at steady-state conditions according to the average of the 350 airflow data points.

The model calibration was considered on five critical integrated parts of the ventilation system, discussed in Section 2.5. The calibrated model may exhibit large deviations from the audit data (reference data) when considering the parts individually. However, the calibrated parts should be considered as one integrated system. An example of the critical parts integrated dependency is explained by using the calibrated 105L. The total level intake had an absolute deviation of 2.81 kg/s from the audit data (see Section 3.3.1, Figure 14). The PVS half-level intake had an absolute deviation of 20 kg/s (see Section 3.3.1, Figure 16). Consequently, the PVS returns of 105L have an absolute deviation of 2.66 kg/s while the SVS district areas have an average absolute deviation of 1.66 kg/s (see Figure 17 and Table 9).

The 20 kg/s deviation on the PVS half-level intake may appear large when only considering that part of the level. Consequently, the other parts of the level are calibrated to within 3 kg/s. Therefore, a large deviation in individual parts will not necessarily mean the system is not calibrated. The model was calibrated considering the integrated dependencies of the airflow due to the fluctuating audit data. Consequently, these deviations are relatively small since the mine distributes over 1 000 kg/s through its tunnel network.

It is important to notice that the airflow direction is also considered in this study. Therefore, air entering the district working (crosscut), or section (level intakes), is positive. Air exiting a crosscut, or level, is negative.

3.3.1 PVS intake calibration

The downcast airflow measured was 1 037 kg/s and the simulation calibrated airflow was 1 060.47 kg/s. Therefore, the total intake of the mine was calibrated to an accuracy of 98%. The air distribution of the downcast air was calibrated according to the intake airflows of each level from 70L to 115L (see Figure 14). The level intakes were calibrated within an average airflow of 6.26 kg/s.

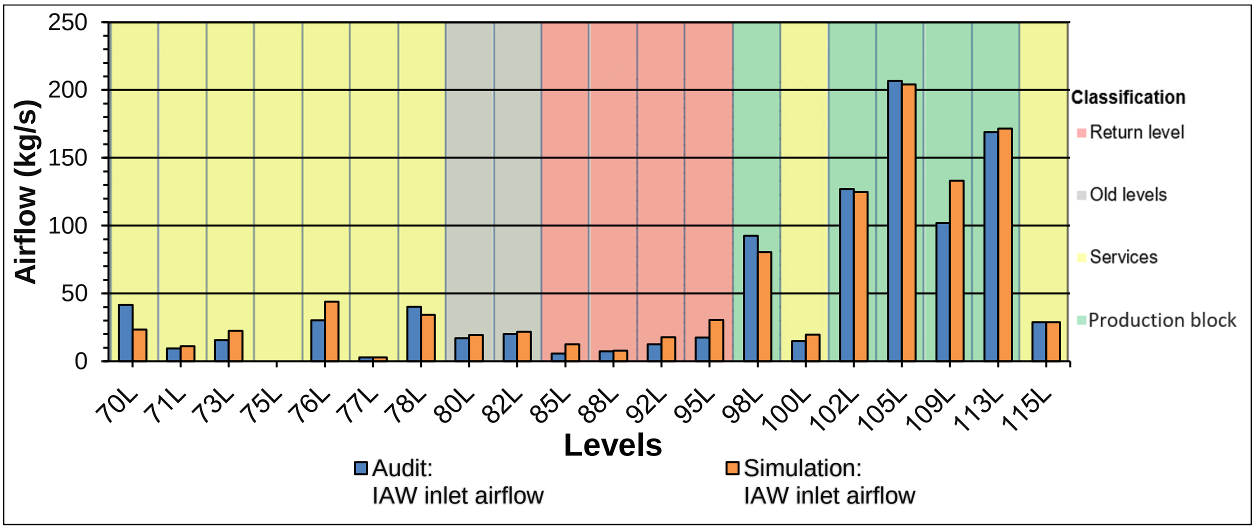


Figure 14: Calibrated level intakes air distribution

The air utilisation of the downcast air was analysed by creating a black box mass balance of the shaft (see Figure 15).

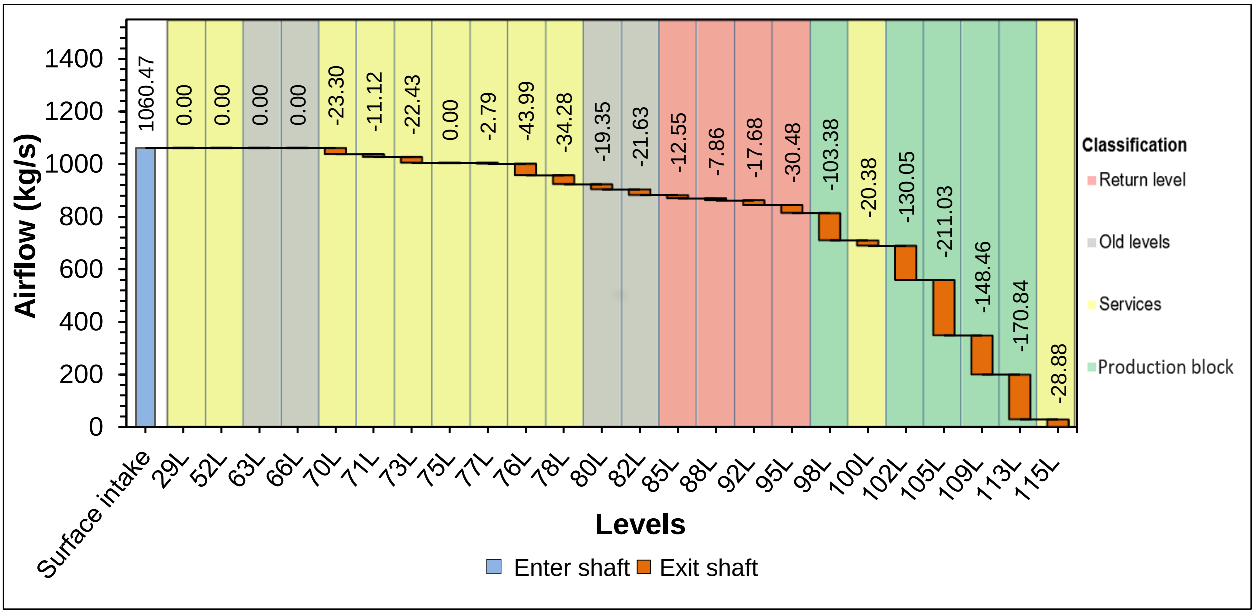


Figure 15: Calibrated PVS downcast shaft air distribution

The air used on each level above 98L was either due to leaking or service levels' ventilation air, and no further calibration was required there. The air distributed to the production levels, however, would either be used on the eastern or western block of the mine. This production block's primary air distribution was calibrated to an average airflow of 8.25 kg/s, see Figure 16.

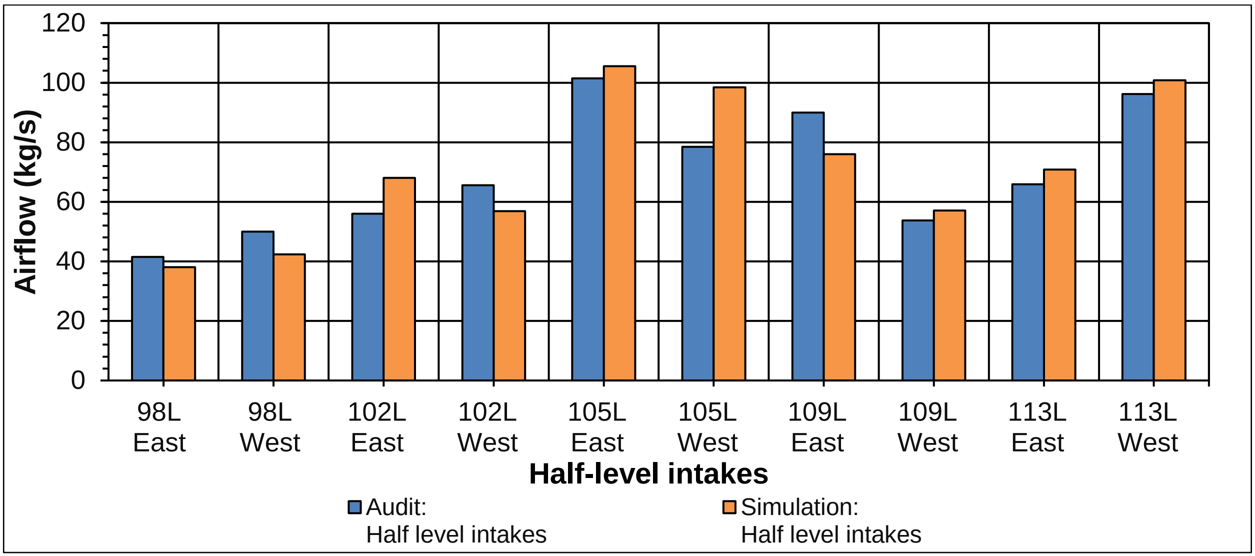


Figure 16: Calibrated half-level IAW intakes

Table 7 summarises the accuracy of the PVS intake calibration. A detailed PVS intake airflow calibration accuracy table can be found in APPENDIX E, Table 28.

Table 7: Summary of the PVS intake calibration

Description	Average absolute deviation (kg/s)
Total mine intake airflow	22.61
All level intakes	6.26
Production levels east and west (half-level) intakes	8.25

The PVS is the system with the most integrated dependencies (see Figure 11). Therefore, the PVS is also the part were the calibrated model will show the largest deviations from the audit data. The actual airflow of the PVS can fluctuate by up to 50 kg/s over six months. These deviations are, however, small when considering that the complex integrated MVS distributes over 1 000 kg/s of air through the mine ventilation network. Therefore, the calibrated production block crosscut deviations are considered acceptable.

3.3.2 PVS returns calibration

The PVS IAW air that has not been used by the SVS district areas (crosscuts) returns to the PVS RAW at the end of the level. These end flows were calibrated to an average airflow of 2.3 kg/s, see Figure 17.

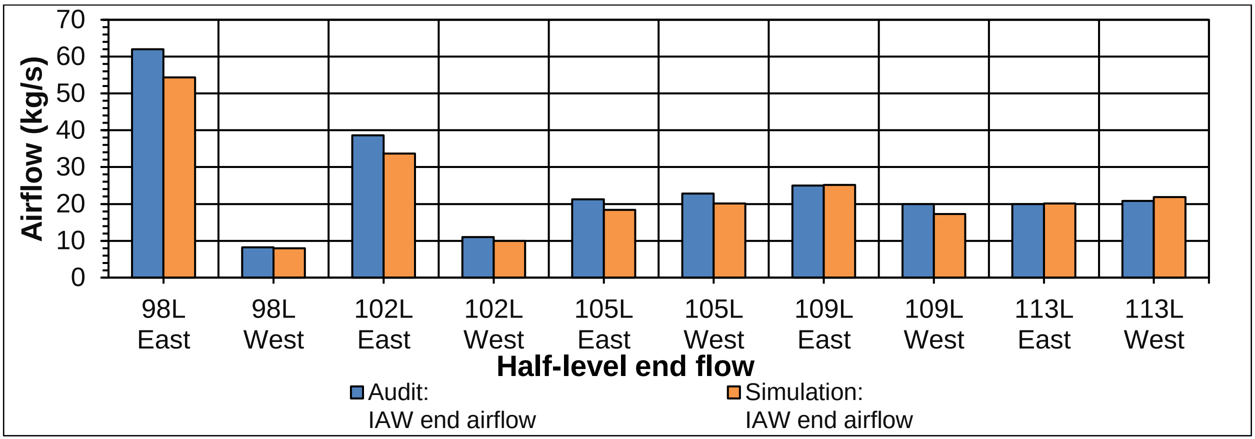


Figure 17: Calibrated half-level IAW returns

The PVS RAW (returning end flows) of each level is connected via RBHs to dedicated levels (98L, 102L and 109L) with booster fans that return the air to the upcast shaft. The air used in the SVS is also upcasted via the stope lines to dedicated return levels (95L to 85L). These return levels stope lines are connected to Mine B on the western side and have booster fans for returning air to Mine A's upcast shaft. The active fans were audited, and the model was calibrated to an airflow accuracy of 12.95 kg/s, a return pressure accuracy of 1.34 kPa, and a delta pressure accuracy of 0.36 kPa.

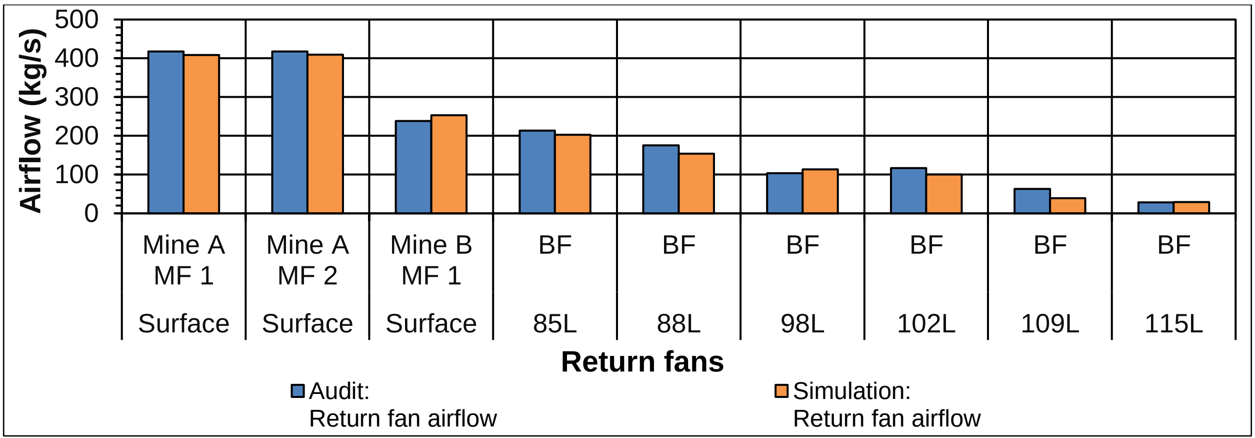


Figure 18: Calibrated primary return fans

MF: Main fan; BF: Booster fan

A black box mass balance of the total upcast shaft (Mine A and Mine B combined) was used to analyse the air distribution of the primary return system (see APPENDIX E, Figure 47). Table 8 summarises the PVS return calibration accuracy. A detailed PVS return airflow calibration accuracy table can be found in APPENDIX E, Table 29.

Table 8: Summary of the PVS return calibration

Description	Average absolute deviation (kg/s)
Half-level end flows	2.26
Return fans	12.95

As discussed earlier, the PVS is the system with the most integrated dependencies and will, therefore, show the largest deviations from audit data. The calibrated PVS return deviations are acceptable considering the model was calibrated at steady-state conditions based on the average airflow of the mine over six months.

3.3.3 SVS branches off the PVS calibration

The air that is used by the SVS districts and throughflow crosscuts branches off the PVS IAW. The distribution of air into each crosscut was calibrated for each level. The black box mass balance of each level was used to analyse the air distribution of the primary air to the crosscuts (secondary air). The calibration graphs for all production levels can be found in APPENDIX E.

Table 9 summarises the crosscut airflows for the calibrated production block. The deviation values were calculated based on the difference between the airflows of the calibrated model and the audited airflows. The average crosscut airflow was calibrated to an absolute deviation of 1.54 kg/s per crosscut.

Table 9: Production block calibration summary

Detailed sections	Description	Average absolute deviation (kg/s)
Production block	98L crosscut airflow	2.10
	102L crosscut airflow	1.42
	105L crosscut airflow	1.66
	109L crosscut airflow	0.85
	113L crosscut airflow	1.66
	Production block average	**1.54**

The SVS has fewer dependencies than the PVS. The SVS deviations are relatively small when considering that the mine's actual airflow can fluctuate by up to 20 kg/s in the SVS over six months. Therefore, the calibrated production block crosscut deviations are considered acceptable.

3.3.4 SVS returns to the PVS calibration

The air that is used by the SVS crosscuts returns to the PVS RAW on dedicated levels. The air distribution of the SVS air to the PVS RAW was calibrated. The black box mass balance of each return level was used to analyse the air distribution of the secondary air to the primary return levels. These detailed calibration graphs can be found in APPENDIX E.

Table 10 summarises the return block airflows. The deviation values were calculated based on the difference between the airflows of the calibrated model and the audited airflows. The calibrated average absolute deviation of the crosscut airflow was 2.1 kg/s per crosscut. The detailed calibration graphs of all the other levels are shown in APPENDIX E.

Table 10: Return block calibration

Detailed sections	Description	Average absolute deviation (kg/s)
Return block	85L crosscut return airflow	1.10
	88L crosscut return airflow	1.91
	92L crosscut return airflow	2.33
	95L crosscut return airflow	3.08
	Return block average	**2.10**

As discussed earlier, the SVS has fewer dependencies than the PVS. The deviations of this system were considered small since the system can fluctuate by up to 20 kg/s in the SVS over six months. Therefore, the calibrated return block crosscut deviations are considered acceptable.

3.3.5 BAC calibration

The BACs of the mine were calibrated according to the average SCADA data obtained for the period when the production level airflow audit was conducted. The audit on Mine A was conducted during the winter season of 2018. Surface BACs were not active during the winter season because the ambient air temperature was sufficient for downcasting. The underground production block BACs, however, operated 24 hours a day, 360 days a year and, as such, were considered essential elements in the system. These BACs were calibrated within an average airflow of

18.5 kg/s. The water inlet temperature, water outlet temperature and water flow rate of the BACs were also calibrated. These detailed calibration graphs can be found in APPENDIX E.

Table 11 summarises the return block airflows. The deviations were calculated based on the difference between the calibrated model's airflows and the audited airflows. The calibrated deviation of the absolute average BAC airflow was 18.15 kg/s per crosscut.

Table 11: Production block BAC calibration summary

Detail sections	Description	Average absolute deviation (kg/s)
Production block BACs	102L East	23.03
	102L West	16.65
	105L East	15.32
	105L West	17.50
	109L East	14.20
	109L West	22.97
	113L East	24.05
	113L West	11.50
Average production block BAC airflow		**18.15**

The calibrated production block BACs are critical with the largest deviations from the audit data. The airflow through the BACs is highly dependent on fan running statuses and doors of the production block. Each BAC has 90 kW axial fans with only two ventilation doors between the intake and exhaust of these fans. These doors are also the only access points to the half-level production workings. Therefore, the calibration of the BACs was more focused on the total airflow required to satisfy the mass balance in the working areas after the BACs.

3.3.6 Summary

The simulation model was calibrated according to the airflow audits conducted on the mine. The simulation model represents a large, complex and integrated mine with over 1 000 kg/s of air distribution through the network. Auditing this mine took six months to complete and airflow fluctuations were evident during the audit period. Therefore, the simulation model was integratable calibrated at steady-state conditions according to the 350 airflow data points audited over the six months.

Table 12 summarises some of the calibrated sections of the MVS. All detailed graphs are shown in APPENDIX E.

Table 12: Summary of the overall calibration of Mine A

Description	Average absolute deviation (kg/s)
Total mine intake airflow	22.61
Level intake average	6.26
East and west split average	8.25
Average end flows	2.26
Average return fans	12.95
Production block average	1.54
Return block average	2.10
Average production block BAC airflow	18.15
Overall average	**9.27**

The total mine intake, average return fans, and production block BAC airflows exhibited the largest airflow deviations. The reason for this is that this PVS is dependent on the SVS deviations. For example, a level's intake airflow is dependent on the half-level airflow and a half-level airflow is dependent on the half-level working area's airflow. Therefore, considering the complexity and size of this mine, the simulation model can now be classified as a calibrated simulation model. From this point, the calibrated simulation is considered as the baseline of the improvement process.

3.4 Verification of Mine A simulation model

At this stage, the calibrated simulation model should behave the same as the actual mine. The predictive ability of the model was verified by comparing the airflow after making configurational changes with the airflows of both the actual mine and the simulation model. After the calibration audits were conducted, the ventilation system on the western side of the mine changed over the preceding six months. The calibrated simulation model (baseline) was adjusted to the configuration of the current mine, and the model's predicted airflows were analysed. Six western levels of the current mine were also audited during this period and compared with the simulation predictions.

The changes that were applied to both systems are listed in APPENDIX F, Table 30. Simplified layouts of the western side configuration before and after making are shown in APPENDIX F, Figure 68 and Figure 69, respectively. The PVS IAW behavioural changes of the simulation model and the audit data were compared. Line charts of the PVS airflow in the western IAW haulage [from Crosscut 36 (XC36) to XC43] are shown for 102L, 105L and 109L (see Figure 19, Figure 20 and Figure 21, respectively).

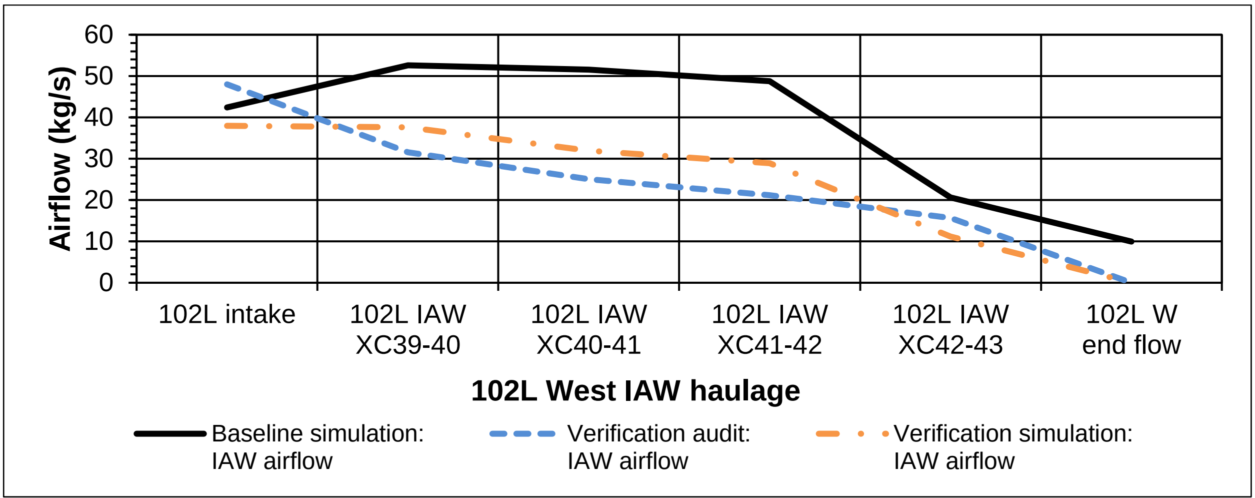


Figure 19: 102L West IAW verification

The audited airflow on the western side of 102L decreased from the baseline airflow. The predicted airflow also decreased from the audited airflow with an average of 5.87 kg/s.

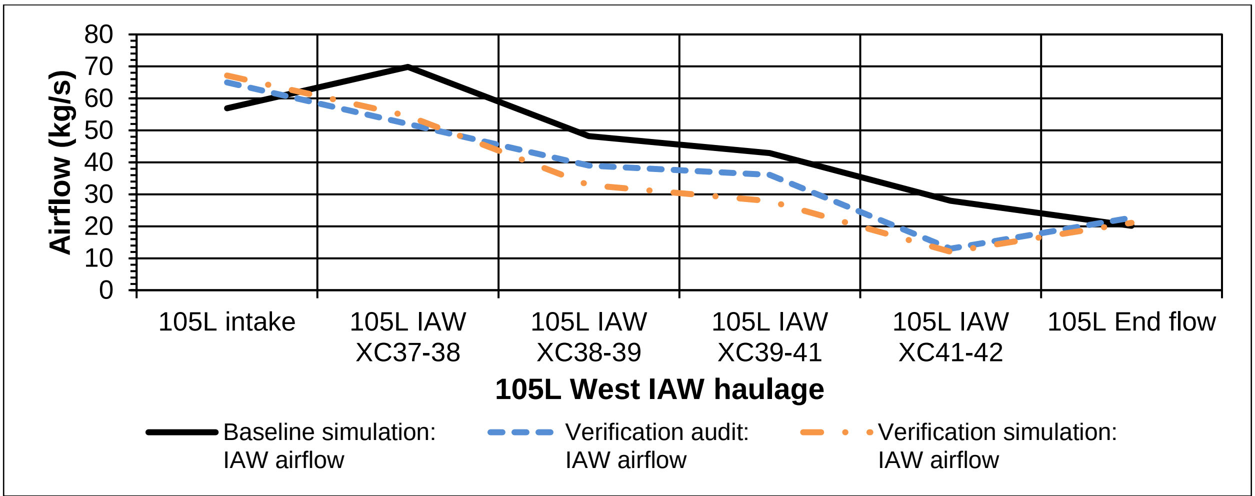


Figure 20: 105L West IAW verification

The audited airflow on the western side of 105L decreased from the baseline airflow. The predicted airflow also decreased from the audited airflow with an average of 3.59 kg/s.

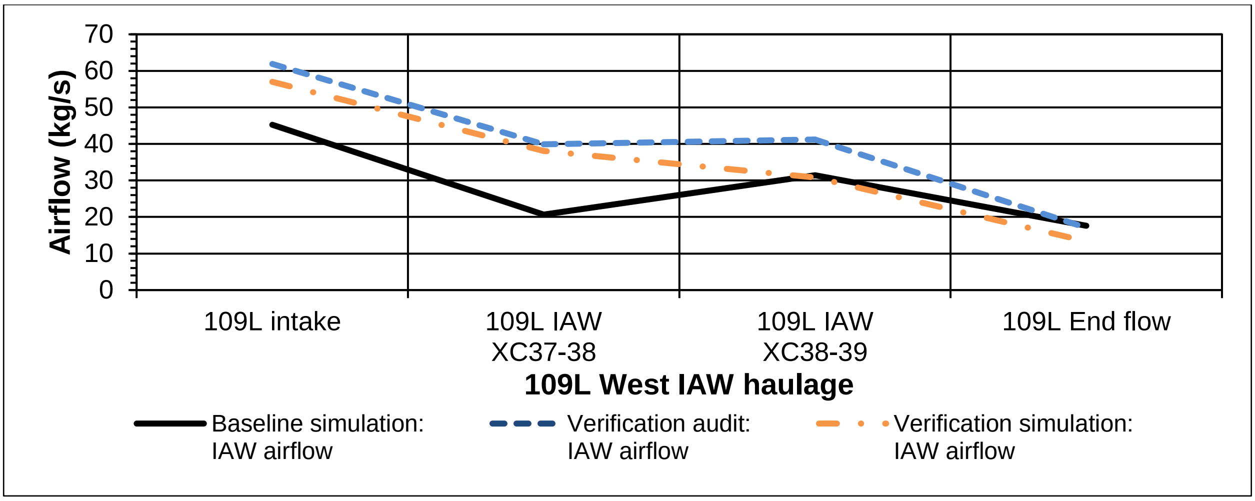


Figure 21: 109L West IAW verification

The audited airflow on the western side of 109L increased from the baseline airflow. The predicted airflow also increased from the audited airflow with an absolute deviation of 5.2 kg/s. It is clear from the above figures that there is a definite correlation between the predicted and the audited airflows of the IAW haulage (PVS).

Figure 22 compares the SVS working areas (crosscuts). The airflows of specific crosscuts were measured and compared with the simulation predictions.

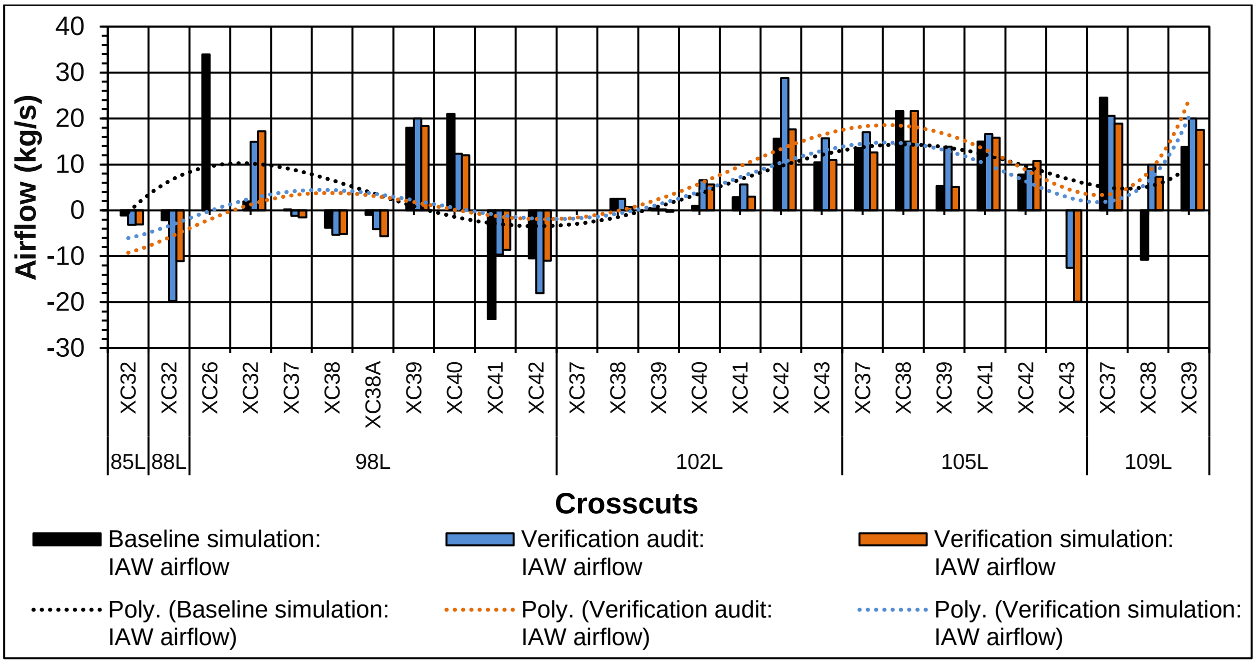


Figure 22: Western crosscut verification

Figure 22 shows that there is a definite correlation between the verification audit data and the simulation prediction (see trendlines). A few noticeable abilities and shortcomings are discussed briefly:

- The airflow direction change on 109L XC38 was predicted with an accuracy of 2.71 kg/s. This airflow change occurred due to a new tunnel connection between stope (S) 38 and S37. The new tunnel is uncalibrated since it was created after the simulation was calibrated. Therefore, a larger deviation could be expected since a calibrated resistance has not yet been applied to the tunnel.
- The airflow quantity predicted for the new crosscut on 105L XC43 was predicted with an accuracy of 7.37 kg/s. The new tunnel is uncalibrated since it was created after the simulation was calibrated. Therefore, a larger deviation could be expected since a calibrated resistance has not yet been applied to the tunnel.
- The airflow reductions on 98L XC41 due to the open doors on 102L XC43 were predicted with a deviation of 1.03 kg/s. This is a calibrated crosscut with a fan component that is calibrated to the exact fan specifications (fan curve). A more accurate and reliable prediction is expected from this crosscut.

- The airflow increases on 109L XC39 due to an additional active fan were predicted with a deviation of 2.51 kg/s. This is a calibrated crosscut with a fan component that is calibrated to the exact fan specifications (fan curve). A more accurate and reliable prediction is expected from this crosscut.

A summary of the overall predictions is given in Table 13.

Table 13: Summary of verification accuracy

System	Description	Average absolute deviation (kg/s)
PVS	102L west IAW	5.87
	105L west IAW	3.59
	109L west IAW	5.20
SVS	85L crosscuts	0.01
	88L crosscuts	8.6
	98L crosscuts	1.61
	102L crosscuts	3.10
	105L crosscuts	4.94
	109L crosscuts	2.31
	Average PVS	**4.89**
	Average SVS	**3.43**
	Average overall	**3.92**

Figure 19, Figure 20 and Figure 21 show that the model predicts the behaviour of the mine when changes are implemented. There are however deviations between the simulation and the verification audit data. These deviations are relatively small for a complex integrated MVS distributing over a 1 000 kg/s of air through its ventilation network. The simulation model was therefore considered acceptable for predicting changes in the overall MVS. The free flow and new tunnel prediction accuracies, however, do require additional investigations and audits before predicting such a section. However, it was proven that active districts (crosscut with the airlock system) are accurate for predictions.

3.5 Improving the air distribution of Mine A

3.5.1 Preamble

During the verification phase (Section 3.4), it was proven that the predictive ability of the simulation model was sufficient. The simulation model could be used to analyse improvement possibilities. Over a hundred ventilation airflow requirements for each production level of the mine were obtained from on-site ventilation personnel (see APPENDIX G for the full crosscut planning, configuration and required airflows). An overview analysis of the system was conducted to understand the current air distribution and its potential areas of improvement. The volumetric efficiency calculation (discussed in Section 1.5) was used to determine the distribution efficiencies of each specific part of the ventilation system. Table 14 summarises Mine A's air distribution before the improvements were implemented.

Table 14: Summary of Mine A's air distribution before improvements

Summary of Mine A air distribution	
Location	**Baseline airflow (kg/s)**
Total downcast	1 060.47
Air utilised on service levels	117.96
Air utilised on production levels	763.76
Air losses (PVS downcast shaft leaks)	178.75
Production levels IAW (at split)	714.10
Air losses on a production level (PVS level leaks)	49.66
Sum of crosscut and developing intakes	835.67

The PVS air distribution efficiency of Mine A was 67.34%, which meant that only 714.1 kg/s of the 1 060.47 kg/s air was used efficiently on the production block. The outstanding 346.37 kg/s of air was either leaking or being used to ventilate service levels (by means of short-circuit ventilation). The required airflow for the production block was 716 kg/s, which was calculated based on the crosscut and development working area requirements. Consequently, the crosscuts used 835.67 kg/s of air. This was impossible based on a simple mass balance since the PVS only supplied 763.76 kg/s of air to the SVS. A more in-depth investigation of the model found that the additional 121.57 kg/s used in the crosscuts and development areas was air that was being reused. Return air either leaked out of the PVS RAW or, by means of opposite flow, out of crosscuts (more specifically travel way crosscuts). In other words, the PVS did not supply enough air according to the SVS demand; thus, the system adjusted itself and utilised return air to ventilate the SVS. Consequently, the return air contaminated the intake air, which influenced the conditions of the working areas negatively. Therefore, the air distribution of both PVS and SVS of Mine A was considered for improvement.

The processes as described in Section 2.8 was implemented on Mine A to improve the air distribution of the ventilation system. The four cyclical identification processes were used to identify problematic areas on the baseline model and test different solutions for improving the air distribution of the ventilation system. Thereafter, the most feasible solution was selected and added to the action list. The following sections discuss the improvement process implemented on Mine A.

3.5.2 Identify air leakage in PVS of Mine A

Of the air distributed down the PVS downcast shaft, 78.46% was distributed to useful sections of the mine. The outstanding 21.5% of air was lost on unoccupied, inactive levels. Non-production levels accounted for 300 kg/s of the air used; however, according to the level requirements, only 130 kg/s was required. All the PVS downcast air leaks were identified by means of a line graph (see Figure 23) depicting the distribution of the upcast and downcast air distribution of the entire ventilation system. The gap between the upcast and downcast represents the air moving between levels via RBHs or stope lines. If the upcast and downcast change simultaneously without a gap between the lines (for example 71L in Figure 23), there are leaks or short-circuit ventilation on that level.

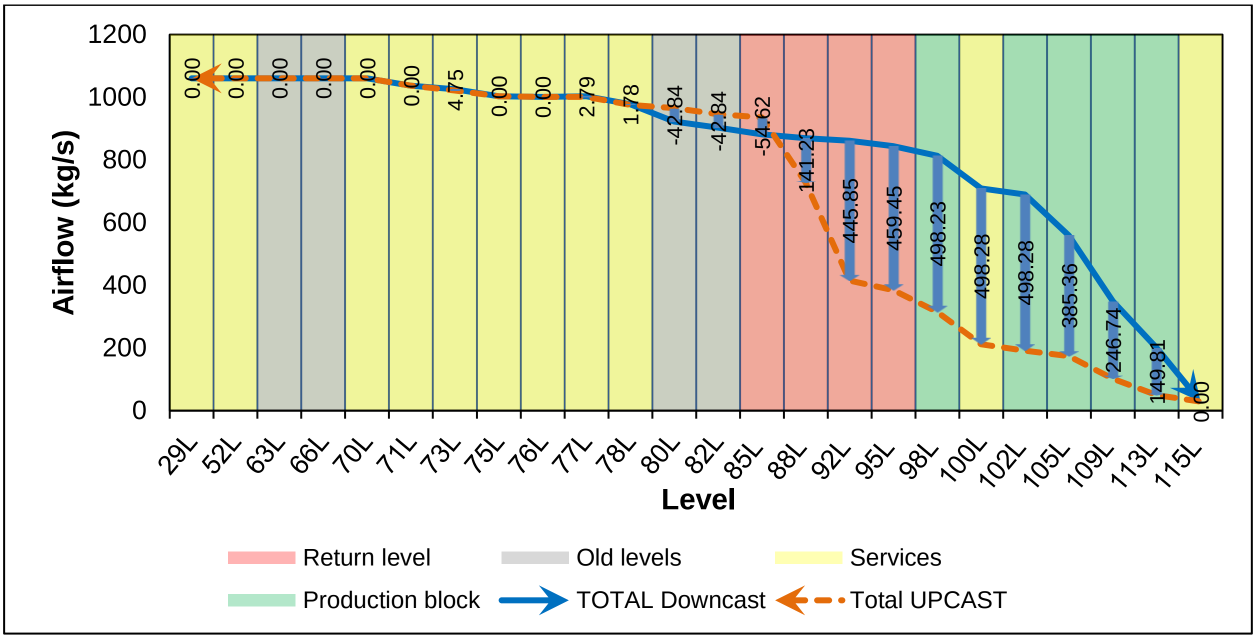


Figure 23: Upcast and downcast air distribution

All identifiable leaks were sealed on the simulation model, and the following level intake predictions were achieved (see Figure 24):

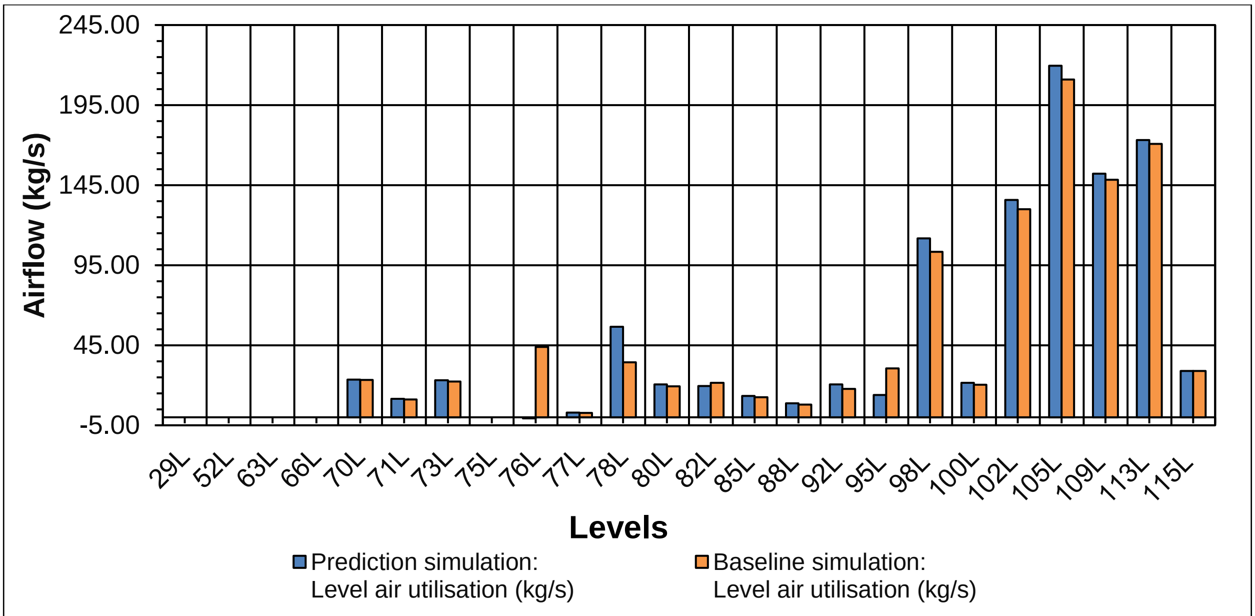


Figure 24: Level intake predictions after leaks were sealed in PVS

It should be noted that the baseline simulation model was calibrated by considering small (closed door or wall crack leaks) as well as large leaks (open door or broken wall). All large leaks were identified and reduced. Although a single small leak does not have a large effect on the ventilation system, together small leaks contribute to a large air loss in the system. All leaks not physically identified in the mine during the audit phase were considered as small leaks and non-reducible. In total, these leaks were found to be a quite significant amount of 166 kg/s.

It is evident from Figure 25 that the production level intakes increased as a result of the decrease in air losses in the PVS downcast system. As a result of the 60 kg/s reduction in downcast leaks, the PVS air distribution improved with 5% (83% downcast efficiency). However, the small leaks above the production block still used an unaccounted 135 kg/s of air after the identifiable leaks were removed. This leakage was considered as unchangeable since the leaks included several small leaks over a long distance.

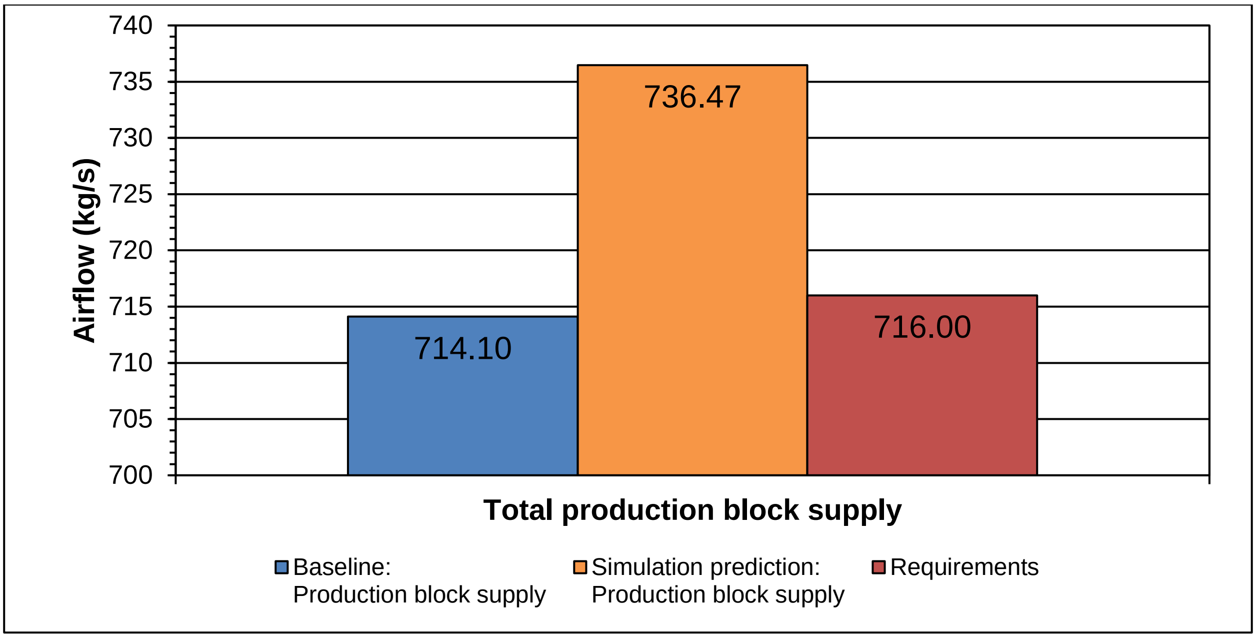


Figure 25: Total production block improvement after PVS leaks were removed

Although the PVS downcast was improved as much as possible, improvement was limited due to uncontrollable small leaks. Therefore, the efficiency of the downcast system could not be improved above a maximum efficiency of 83%. However, the air supplied to the production block was now sufficient for supporting the crosscut and developing the working demand without reusing air.

3.5.3 Identify over- and underventilated working areas in Mine A's SVS

The total airflow supplied to the production block was improved as described in Section 3.5.2 (see Figure 25) to supply the production block with enough airflow to accommodate each crosscut requirement without reusing air. However, this air was distributed poorly between the half levels, thereby resulting in some over- and underventilated sections (see Figure 26).

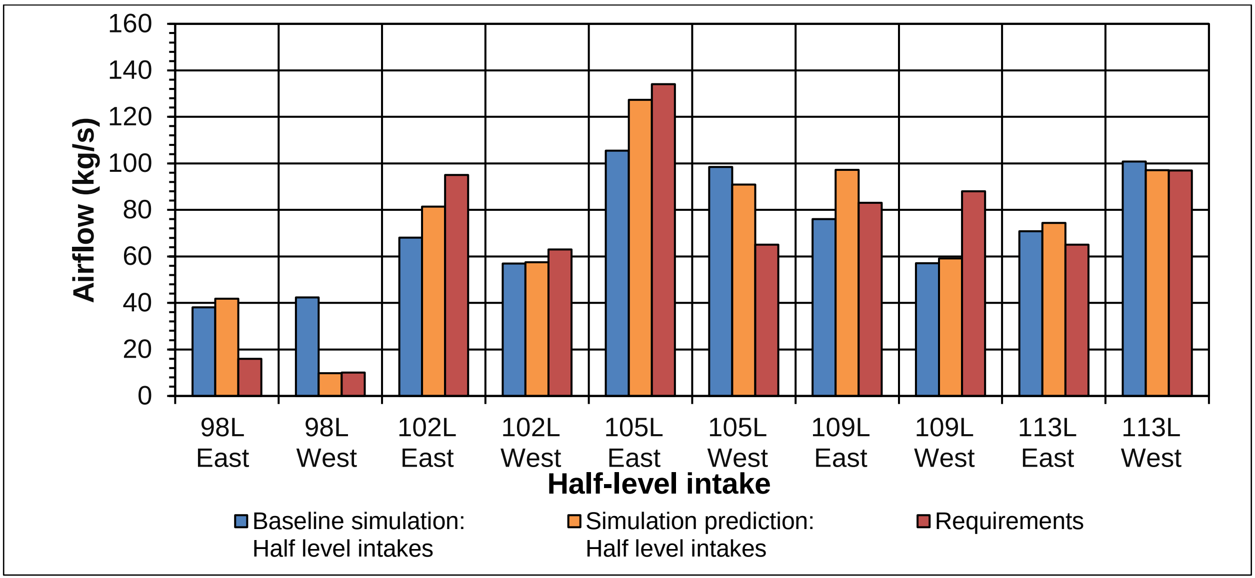


Figure 26: Half-level intake airflow after PVS leaks were removed

The required airflow for the half level (compared with the prediction airflow in Figure 26) was calculated based on the crosscut and development requirements of that half level. Therefore, to improve the air distribution per half level, one needs to consider the active crosscuts and the air distribution of the development workings (SVS). The crosscut (part of the SVS) was first optimised by identifying and improving all overventilated crosscuts using passive controllers. Thereafter, active controllers were used to identify and improve all the underventilated crosscuts. The methods discussed in Section 2.8 were used to identify and apply changes to the workings.

All production level crosscuts were improved according to the airflow requirements. The baseline simulation crosscut airflow was compared with the crosscut airflow requirements. Possible solutions were implemented on the simulation model until the best scenario was achieved (the cycle). All production level crosscut improvement graphs are shown in APPENDIX G.

The distribution efficiency of the SVS useful air utilisation (occupied working areas) of the PVS IAW air was improved. In other words, the air supplied by the PVS was now utilised more efficiently in the working areas. Consider production 102L as example in Table 15.

Table 15: 102L PVS IAW distribute to SVS air distribution efficiency

Half level	Simulation: Baseline (%)	Simulation: Prediction (%)
102L East	136.13*	88.16
102L West	76.94	93.02

*over 100% due to reuse of return air

The 136.13% air utilisation on 102L east half level is a typical example of air being reused in the system. More air was used by the SVS than what was supplied by the PVS IAW. Return air either reverses flow or leaks out of the crosscuts and RAWs before being reused in the SVS occupied crosscuts. The airflow distribution of occupied crosscuts also improved according to the required airflows. The quantification of this distribution improvement is calculated by determining the offset value between the simulated airflow and the required airflow.

Table 16: 102L crosscut (SVS) distribution offset (based on required airflow)

Description	Simulation: Baseline	Simulation: Prediction
Average percentage offset (%)	119.70*	28.68
Average absolute offset (kg/s)	9.16	3.19

*over 100% due to reuse of return air

It is evident from Table 16 that the airflow in each crosscut improved, meaning that it was closer to the required airflow. The air distribution of the crosscuts improved with an average margin of 5 kg/s of the required airflows (see APPENDIX G, Figure 71). The reuse of air was also no longer possible since all travel ways and no-mining working areas were either sealed off or regulated. Consequently, the improvement of the SVS over- and underventilated working areas resulted in a more even PVS half-level air distribution (see Figure 71).

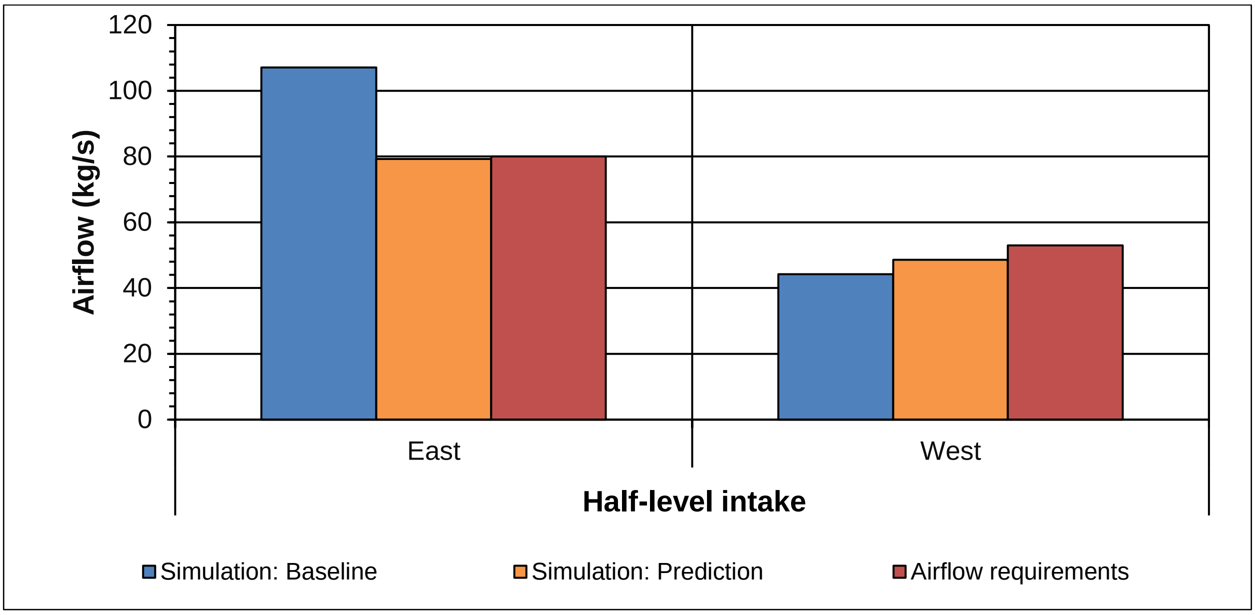


Figure 27: 102L half-level intakes after SVS was improved (figure include end flows)

3.5.4 Identify overventilated areas in Mine A's PVS

The previous sections showed the improvements of the intake air distribution from the surface to the crosscut and development working areas. However, the PVS is still overventilated in some areas, which influences the overall distribution of the production block. These overventilated areas were identified and the airflows were reduced to meet the requirements. Figure 28 shows the improvements made within the overventilated PVS. The airflow comparison method, discussed in Section 2.8, was used to identify the problematic areas. Improvement initiatives were applied to each of these areas to reduce the airflow in the PVS. Table 34 summarises the improvements to the overventilated PVS areas. The compare, identify and improve method discussed in Section 2.8 was used to identify and apply changes to the sections.

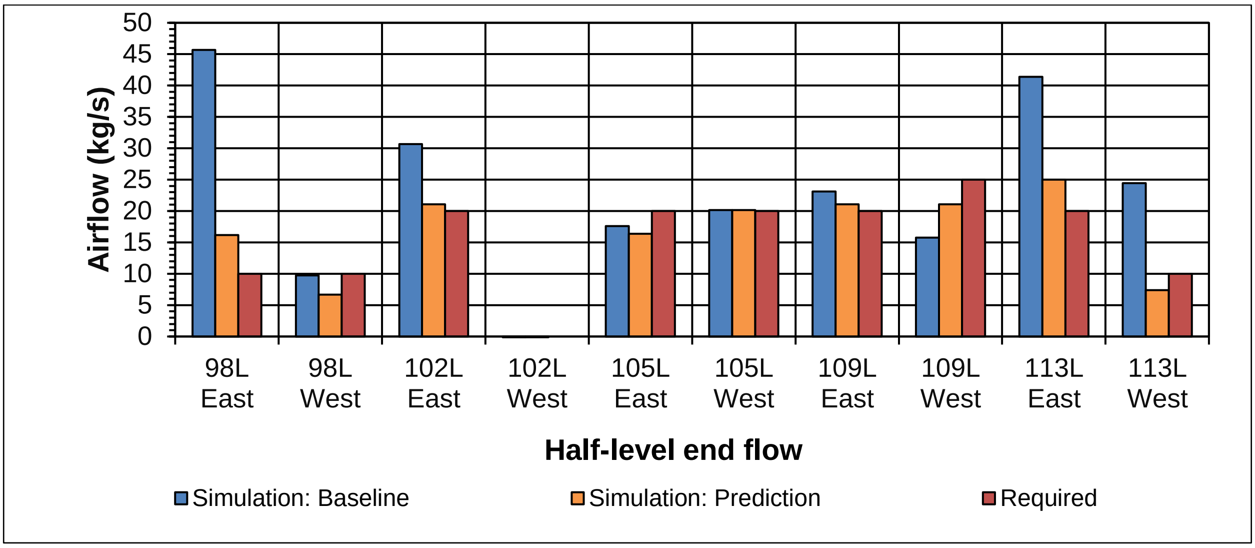


Figure 28: Mine A half-level end flows

The air distribution of the end flows, which form part of PVS, was improved with an average margin of 5 kg/s from the required airflows (see APPENDIX G, Figure 72). The distribution improvement between the end flows was quantified by calculating the offset value between the simulated airflow and the required airflow (see Table 17).

Table 17: End flow distribution offset (based on required airflow)

Description	Simulation: Baseline	Simulation: Prediction
Average percentage offset (%)	72.93	19.14
Average absolute offset (kg/s)	9.7	2.71

It is evident from Table 17 that the airflows at each PVS end were improved. The overventilated end flows were decreased according to the required airflows. The average end flows only deviated with an average of 19.14% from the required airflow, thereby availing more air for the crosscuts.

3.5.5 Identify overcontrol return air in Mine A's PVS

The previous sections discussed the optimisation of the distribution of the air from the surface to the working areas. This section describes the simulation model used to investigate potential return improvements on the MVS. Two overcontrol return investigations were identified and investigated on Mine A.

Return manifold investigation

The first investigation focused on removing all large unnecessary restrictions in the PVS RAW. Regulators were found on all the return crosscuts on 85L, 88L, 92L and 95L. These regulators were used previously to regulate the return air per working stope line. However, the mine no longer required these regulators and they were removed to reduce the return resistance of the overall system.

Working area stope lines return air to dedicated return fans, which was considered a risky operation since a fan failure could lead to a ventilation catastrophe in the dependent working areas. Therefore, a return manifold was implemented on the simulation model and all unnecessary regulators and seals were removed from the return system. The removal of these ventilation control devices reduced the resistance of the system, thereby allowing the airflow to increase through the return fans.

Figure 29 shows an example of the airflow increase on 88L XC14–XC38 due to the regulator removal.

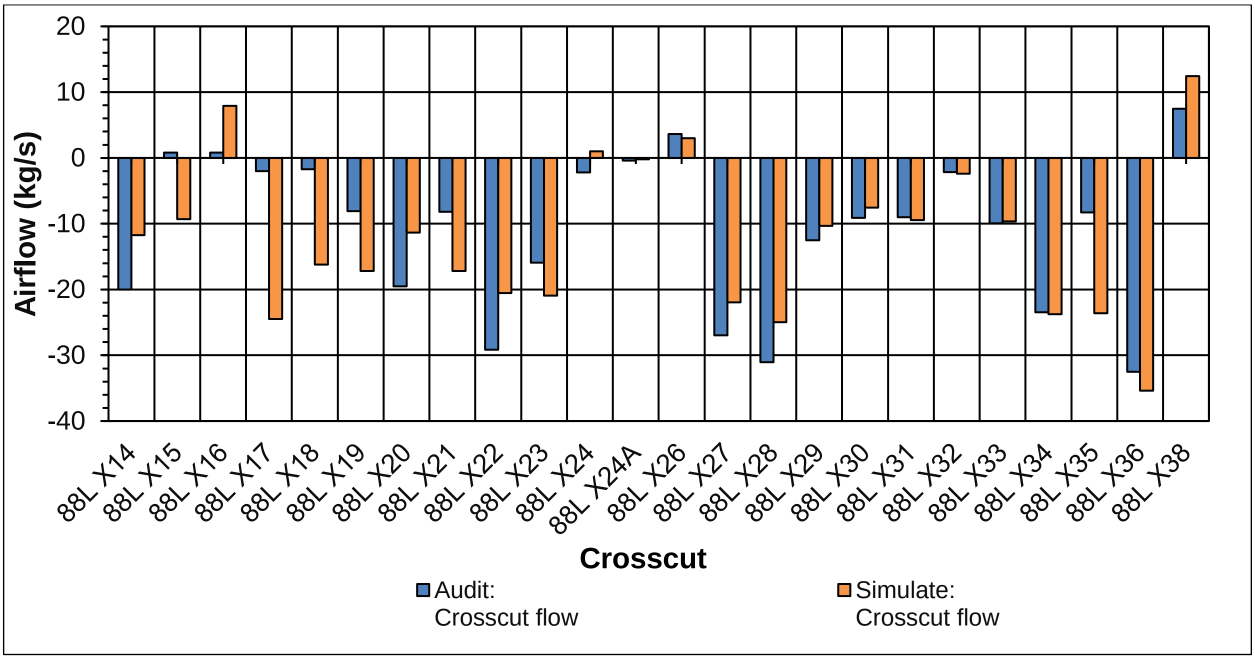


Figure 29: Removal or PVS return air regulation

The production block had an overall air distribution efficiency increase of 1% due to the regulator removal. The active working districts (active crosscuts) had an average increase of 1 kg/s. The development end flows had an average small decrease of 0.15 kg/s (see Table 18). Consequently, the airflow into the crosscuts that are part of the SVS increased with an unnoticeable small change in the PVS end flows.

Table 18: Average production block crosscut airflow change after PVS return system improvement

Location		Average crosscuts airflow (kg/s)		
		Simulation: Baseline	Simulation: Prediction	Airflow change
98L	East	16.1	10.4	−5.68
	West	6.7	12.5	5.75
102L	East	15.3	16.7	1.36
	West	11.5	15.2	3.66
105L	East	15.9	16.0	0.10
	West	14.7	15.5	0.79
109L	East	16.7	16.9	0.14
	West	11.8	12.1	0.34
113L	East	18.4	18.7	0.23
	West	13.2	13.5	0.39
		Overall crosscut average		0.95
		Overall end flow average		−0.15

The reduction in resistance allowed more air to return through the stope lines to the booster fans on 85L and 88L. The reduction in return resistance had an effect on the overall MVS. The resistance of the airflow route was lowered through the stope lines. The airflow was increased through the working areas by simply removing unnecessary ventilation controllers.

Table 19 shows how the airflow increased by 12.9 kg/s through the stope line return fans and only reduced by 4.57 kg/s in the end flow return fans. The reduction in return resistance had an effect on the overall MVS. The resistance of the airflow route was lowered through the stope lines. The airflow was increased through the working areas by simply removing unnecessary ventilation controllers.

Table 19: Return fan predictions after PVS return system improvement

Level	Fan	Size available (kW)	Simulation: Baseline (kg/s)	Simulation: Prediction (kg/s)	Airflow change (kg/s)
Surface	Mine A main fan 1	4 100	407.49	408.11	0.62
Surface	Mine A main fan 2	4 100	407.59	408.21	0.62
Surface	Mine B main fan 1	2 200	246.98	246.8	−0.18
85L	Booster fan	1 000	201.50	207.19	5.69
88L	Booster fan	1 000	150.20	157.41	7.21
98L	Booster fan	2 × 500 kW – double stage	106.27	104.92	−1.35
102L	Booster fan	1 000	89.09	86.92	−2.17
109L	Booster fan	4 × 45	36.80	35.94	−0.86
115L	Booster fan	4 × 45	29.37	29.18	−0.19
			Total fan		**1.04**
			Total surface return		**1.06**
			Total stope line return		**12.9**
			Total end flow return		**−4.57**

Return airflow reduction investigation

The second investigation was based on the necessity of Mine B’s 2.2 MW surface main fan. Section 3.5.2 (see Figure 25) discussed the 736.47 kg/s (2.86% overventilated) improvement of the supplied airflow of the production block. However, removing the main fan located at Mine B would result in a 250 kg/s return airflow reduction on the western side of the mine. A new booster fan installation was investigated in collaboration with the main fan removal to compensate for the significant change on the western side of the return system.

The simulation model was, therefore, updated with a new booster fan on 92L and the main fan was deactivated on Mine B. A few additional configurational actions were identified to allow air to distribute efficiently to the new booster fan location. Thereafter, the simulation predicted a total Mine A downcast airflow reduction of 222.82 kg/s. As a result, the supplied airflow to the production block decreased to 655.99 kg/s, which meant that the production block was underventilated with 8.38% (see Table 20).

Table 20: Summary of overall air distribution efficiency and percentage offsets

	Baseline	Prediction before Mine B fan	Predictions after Mine B fan
Total downcast (kg/s)	1 060.47	1 056.66	837.65
Production levels IAW at split (kg/s)	714.10	736.47	655.99
PVS air distribution efficiency (%)	67	70	78
Percentage offset based on required flow (%)	−0.27	2.86	−8.38

The main concern, however, was the effect this airflow compromise would have on the working areas of the mine. The airflow of the PVS production block only reduced by 68 kg/s to 655.99 kg/s. This reduction was distributed over the five production levels with an average airflow reduction of 6.21 kg/s per half level (see Table 21).

Table 21: Half-level production block intake airflow reduction (Mine B main fan removed and 92L booster fan installed)

Half level	Simulation prediction (kg/s)		Change in airflow (kg/s)
	Before Mine B main fan removed	Mine B main fan removed; 92L booster fan installed	
98L East	10.43	8.18	−2.25
98L West	12.45	10.70	−1.75
102L East	93.35	88.82	−4.54
102L West	58.22	49.80	−8.43
105L East	140.73	130.33	−10.40
105L West	68.79	60.68	−8.11
109L East	80.04	77.12	−2.92
109L West	78.49	69.54	−8.95
113L East	98.04	88.96	−9.08
113L West	77.59	71.87	−5.72
		Average	**−6.21**
		Total	**−68.36**

This airflow reduction per half level was distributed between the working areas of that half level. The average reduction of the working areas was predicted to be 1 kg/s. Table 22 shows that the largest changes would take place in the vamping and reclaiming working areas. The most occupied areas, namely the active and travelling working areas, had small reductions of under 1 kg/s.

Table 22: Working area reduction (Mine B main fan removed and 92L booster fan installed)

Working area status	The average reduction in airflow (kg/s)
Active	−0.94
End flow	−0.75
No mining	−0.18
Travel way	−0.46
Vamping or reclaiming	−2.73
Average	**−1.01**
Lower extreme	**−4.56**

This average compromise of 1 kg/s in the working areas was considered acceptable since the advantages of removing a main fan and installing a new booster fan to the system would:

- Reduce maintenance cost.
- Reduce operational cost since a fan capacity of 1.1 MW is removed from the system.
- Improve safety (Mine B shaft is abandoned and unfenced with frequent illegal activities).
- Improve system efficiency.
- Reduce heat due to auto compression.

The new return configuration improved the air distribution of the MVS. The downcast air distribution to the production block improved with 8% (see Table 20). The downcasted air is now being utilised more efficiently. Figure 30 shows the downcast shaft air distribution where the large reduction in airflow was noticed on the levels above the production block. This shows that despite the large reduction in supply airflow, the system was still capable of maintaining the required airflow in the production block of the mine.

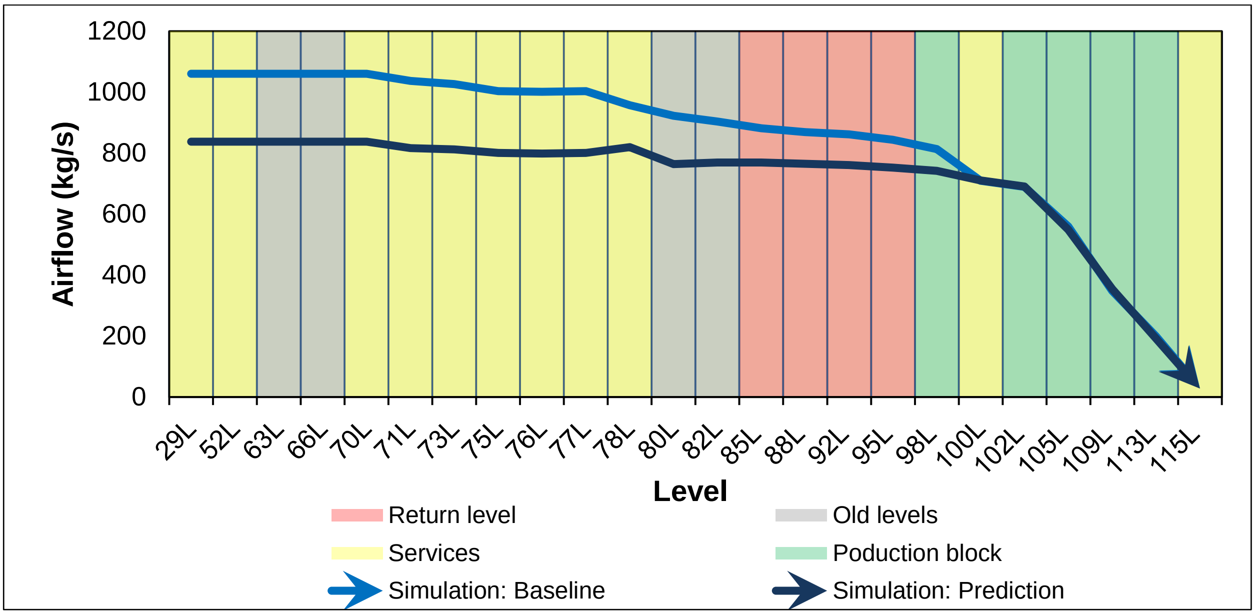


Figure 30: Mine A downcast airflow (Mine B main fan removed and 92L booster fan installed)

3.6 The improved ventilation system of Mine A

The four cyclical identification processes were used to improve the ventilation system of Mine A. Each cycle that has been implemented in the previous sections had a different influence on the ventilation system's performance. The first three identification cycles focused on improving the air distribution of the air from the surface to the working areas of the mine. The fourth identification cycle investigated the potential return improvements on the MVS.

This section briefly discusses the overall MVS improvements and the cost implications the improvement process will have for the mine.

3.6.1 Air distribution improvement

Table 23 summarises the air distribution after all the improvements have been implemented on the simulation model.

Table 23: Summary of Mine A's air distribution after improvements

Location	Baseline airflow (kg/s)	Prediction airflow after implementation (kg/s)
Total downcast	1 060.47	837.65
Air used on service levels	117.96	128.60
Air used on production levels	763.76	694.05
Air losses (PVS downcast shaft leaks)	178.75	15.01
Production levels IAW (at split)	714.10	655.99
Air losses on the production level (PVS level leaks)	49.66	38.05
Sum of crosscut and developing intakes	835.67	674.75

The overall PVS air distribution efficiency of the mine improved with 11% (78.3% PVS air distribution efficiency), meaning that 656 kg/s of the 837.649 kg/s air was now used efficiently on the production block. Table 20 summarised the improvement and fan removal effect on the mine's air distribution.

Figure 31 is a graphical representation of the overall improvement of the entire MVS. The graph shows the difference between the simulated airflow and the required airflow at specific points in the PVS and SVS (0 kg/s is considered as ideal).

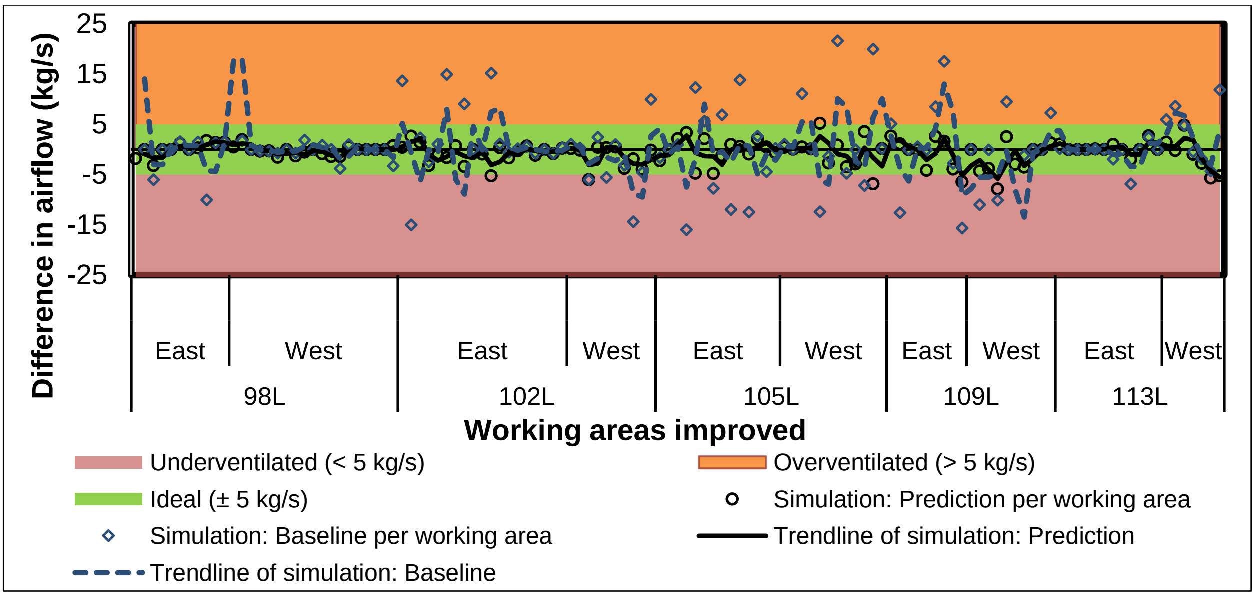


Figure 31: Overall difference between simulation and required airflow (0 kg/s was considered ideal)

It is evident from Figure 31 that despite the PVS being underventilated by 8.4%, the overall average airflow of the system improved within an average margin of 5 kg/s from the required airflows (see APPENDIX G, Table 36 for more detail). These results therefore validate the study objectives aimed at improved underground ventilation.

3.6.2 Cost implications

The improvement methodology that was implemented in Section 3.5 reduced the total fan power requirements by 1.38 MW over the entire MVS (see Table 24). This power reduction is due to all the fans being removed from the MVS.

Table 24: Mine A fan input power

Section	Simulation: Baseline fan power input (kW)	Simulation: Prediction fan power input (kW)	Change in input power (kW)
Mine A fan 1	3 897.19	3 923.34	26.15
Mine A fan 2	3 897.50	3 923.57	26.08
Mine B fan 1	1 850.18	0.00	−1 850.18
85L booster fan	848.57	776.63	−71.94
88L booster fan	599.70	551.67	−48.03
92L booster fan – New	0.00	508.19	508.19
98L booster fan	135.91	183.96	48.05
102L booster fan	312.98	310.91	−2.06
109L booster fan	101.20	97.12	−4.08
115L booster fan	93.16	91.74	−1.43
Surface BAC fans	391.24	350.43	−40.81
Underground downcast BAC fans	206.24	203.73	−2.51
Production block fans (BAC, end fans and crosscut fans)	2 064.58	2 096.99	32.41
Total fan power input	**14 398.45**	**13 018.28**	**1 380.16**
Operational cost p.a. (Rand)	**R120 649 602.50**	**R109 084 726.73**	

The initial installation and action implementation expenses required to improve Mine A's ventilation system are summarised in Table 25. It should be noted that the cost of the new booster fan on 92L cost was not included since the mining group of Mine A had a 1 MW backup that could be installed.

Table 25: Improvement action implementation expenses

Implementations	Quantity	Cost per unit	Total cost
Install ventilation brattices	13	R8 000.00	R104 000.00
Remove wall (blasting)	49	R5 000.00	R245 000.00
Construct a ventilation seal or brick wall	28	R4 000.00	R112 000.00
Seal leaks (with ventilation spray)	6	R10 000.00	R60 000.00
Install a ventilation door	9	R30 000.00	R270 000.00
Seal a 760 mm column (seal plate)	30	R5 000.00	R150 000.00
Install a 45 kW fan	14	R50 000.00	R700 000.00
Install additional booster fan (installation cost)	1	R7 567 222.00	R7 567 222.00
	Overall total initial cost		**R9 208 222.00**

Table 26 indicates the simulated annual cost savings by comparing the baseline operational cost with the improved ventilation system initial implementation and operational cost. Savings are based on Eskom tariffs of 2019–2020 with an 8.3% inflation rate.

Table 26: Simulated fan operational savings over 2 years

	Year 1	Year 2	Total
Baseline operational cost (excl. maintenance cost)	R120 649 602.50	R130 663 519.51	R251 313 122.00
Improved system operational cost (excl. maintenance cost)	R109 084 726.73	R118 138 759.05	R227 223 485.77
Annual operational savings	**R11 564 875.77**	**R12 524 760.46**	**R24 089 636.23**
Improved system initial project implementation cost	R9 208 222.00	R -	R9 208 222.00
Net annual cost savings	**R2 356 653.77**	**R12 524 760.46**	**R14 881 414.23**

The cost related to the actions and implementations required to improve the ventilation system has a payback period of less than one year and a saving of R2 356 653.77. A total saving of R14.9 million is expected in a period of two years.

3.7 Representation and implementation of Mine A improvement plan

Section 3.5 discussed the changes and implementations applied to the simulation model of Mine A and the effect thereof in terms of predicted air distribution and cost improvement on the ventilation system. A detailed drawing and list of actions should be presented to mine management to allow them to apply these changes to the actual ventilation system.

The actions implemented in Section 3.5 were divided into four projects. Each of these projects has a different motive for improving the ventilation system. Each project has an action list with a layout that gives mine personnel an overview of the project objective. The actions in the action list have reference numbers that refer to the layout of that specific project. The action list and layouts of the four projects can be found in APPENDIX H.

CHAPTER 4: CONCLUSION AND RECOMMENDATIONS

4.1 Conclusion

Contaminants influence the air quality of a mine. This makes underground air potentially dangerous to breathe, and extremely high temperatures may also lead to heatstroke. Not only are mining personnel's lives at risk, but production is also highly reliant on these conditions. Poorly distributed air in an MVS results in an oversupply of air to the ventilation system. The additional air is distributed through the MVS to compensate for the air wrongfully distributed. This practice consumes an unnecessary amount of electrical energy since more air than required is supplied to the system. The objective of this thesis is to create a feasible method for improving the air distribution of a deep-level MVS.

It was found in literature that a simulation model is the best way of considering a deep-level mine as an integrated system. An air distribution improvement methodology was, therefore, assembled that required a simulation model of the entire MVS. Creating the simulation model required extensive preparational work, information gathering and behavioural calibrations. The digital twin (calibrated simulation model) was used as the baseline ventilation system of the study. This model enabled the user to analyse and investigate the integrated behavioural changes of the overall MVS. The four identification cycles were followed in a strategic order to improve the air distribution of the MVS. The first three identification cycles improved the air distribution from the surface to the working areas. The fourth identification cycle investigated the potential return improvements on the MVS.

A deep-level gold mine (Mine A), which is located near Carletonville, South Africa, was identified as a case study for the methodology. Mine A mines at depths of 3.4 km underground. PTB was used to create a 3D simulation model of Mine A's ventilation system. Configuration audit data and mine documentation were used to make the model physically identical to the actual mine. The model was integrable calibrated at steady-state conditions according to the 350 airflow data points audited over six months. The average calibration airflow deviation of all the major ventilation sections of the mine was 9.27 kg/s. These deviations were considered small since the simulation model was representing a complex MVS that distributes over a 1 000 kg/s of air through the mine. The audit data was also fluctuating due to the dynamic behaviour of the ventilation system. This calibrated model was used as the baseline and representation of the current mine. All deviations from this baseline were considered as improvement possibilities. The simulation model predictability was investigated during the verification phase. The model predicted within an average absolute deviation of 4 kg/s. The simulation model was considered acceptable for predicting improvement initiatives.

The ventilation airflow requirements for each production level of the mine were obtained from on-site ventilation personnel. The airflow requirements were considered as the benchmark for the improvement process. Localised small changes were made to the ventilation system by means of four identification cycles. The first three identification cycles:

- Improved the downcast to PVS air distribution with 5%.
- Eliminated the reuse of air in the SVS of the mine.
- Improved the air distribution per working area within a margin of 5 kg/s of the required airflow.
- Improved the air distribution per half-level end flow within a margin of 5 kg/s of the required airflow.

The fourth identification cycle was used to investigate overcontrol return and potential improvement initiatives that would allow for energy savings. The first investigation focused on removing all large unnecessary restrictions in the PVS RAW. This increased the average airflow by 1 kg/s per crosscut working area. The second investigation was based on the necessity of Mine B's 2.2 MW surface main fan. The possibility of installing a new booster fan was investigated in collaboration with removing the main fan to compensate for the significant change on the western side of the return system. However, this change compromised the system by 68 kg/s, resulting in a production block that was 8.38% underventilated. The main concern, however, was the effect that this airflow compromise had on the working areas of the mine. Consequently, the simulation predicted an average working area reduction of 1 kg/s. This small reduction was acceptable considering the significant advantages the deactivation of this fan predicted.

The overall improvement process had 11% PVS air distribution efficiency improvement. The overall average airflow of the working areas of the mine was improved with an average margin of 5 kg/s despite the 8.4% underventilated PVS. These results, therefore, validate the study objectives. The improvement process allowed for a fan power reduction of 1.4 MW, resulting in a R14.9 million saving within two years. This study can be implemented in all types of underground mines within the industry. The improved air distribution not only allows for a safer working environment, but also shows potential for reducing expenses.

4.2 Limitations and recommendations

The study had some limitations and constraints that requires further investigation. Underground audit data was the most important information and basis of the entire study. The availability of underground audit information of a deep-level mine is limited due to the following reasons:

- Airflow can change daily in an MVS.
- Sealed-off areas that have been unoccupied for years may be unsafe to enter.
- Flooded or collapsed areas are impossible to reach.
- RAWs can become extremely hot and can result in a health issue for the auditor.

Ventilation airflow is constantly changing due to the dynamic behaviour of the ventilation system. Ventilation audit data will, therefore, have fluctuating results. Simultaneous airflow information will allow a simulation model to be calibrated more accurately for a specific point in time. It is recommended that future studies should focus on methods to obtain airflow data on deep-level mines over a short period. The simulation model calibrated accuracy of audit data obtained in a short period should then be compared to the simulation model calibrated based on audit data obtained over a long period. The investigation should consider whether it is better to calibrate a simulation model using the average airflow fluctuations, or with fixed airflow data at a specific point in time.

The ideal calibrated simulation model would be an air quantity and quality calibrated model. The primary focus in this study was to improve the air distribution of the MVS. The air quantity was calibrated with little calibration of the air quality. Further studies are recommended for improving the air distribution of an MVS with a simulation model that is calibrated for both air quality and quantity. The investigation should consider the effect the calibrated airflow quality will have on the accuracy of airflow predictions and whether the additional calibrations are worth additional effort and time.

A simulation model should be calibrated to a point where the model can make predictions. A tunnel that is not calibrated will not reflect the mine’s actual airflow. Predictions in this study were limited to existing tunnels. Predicting new tunnels resulted in inaccurate airflows due to the unknown resistance the new tunnel will have. A further study is recommended were a new tunnel’s resistance is estimated based on the existing calibrated tunnels. This will allow the mine to investigate new developing tunnels and the ventilation implications on the MVS.

After calibration, the simulation model should be applied as a prediction tool for any future initiatives. A study is recommended during which the sustainability and maintainability of a calibrated simulation model is monitored as the mine develops.

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APPENDIX A. Background

This section provides additional information one can consider when reading this dissertation.

Typical underground mine contaminants include* [5]*:

- Heat

  Heat contamination occurs when heat energy is transferred to the air from a source. Heat is generated from friction between falling rocks, running machinery, strata heat, auto compression, explosion detonations, water influx, human metabolism, oxidation and pipelines [2], [6].

  Strata heat is energy that is transferred from the surface of the rock to the surrounding air. The heat transfer rate depends on the virgin rock temperature, which depends on the depth of the rock, as well as the geophysical properties of the rock [36].

  Auto-compression is the largest contributor of heat in a deep-level mine. Auto-compression occurs due to the transformation of the potential energy of the downward-moving air in a mine shaft into enthalpy. Auto compression, depending on the site, can contribute up to 50% of the total heat transferred to the air [6].

  The addition of heat to the surrounding air will lead to an increase in temperature and a decrease in air quality. High temperatures in a mine can cause the human body temperature to rise and this may lead to a heat stroke. During a heat stroke, the central nervous system of the body starts to dysfunction and this may result in a coma, convulsions or delirium [59]. Additionally, high temperatures in a mine can lead to a decrease in miners' alertness, reaction time, coordination and dexterity [17].

- Diesel emissions

  Diesel engines are frequently used in a mine since they are easy to maintain, durable and reliable. These engines release dangerous gases such as carbon monoxide (CO), nitric oxide (NO), carbon dioxide (CO_2), nitrogen oxide (NO_x), hydrocarbons and fine diesel particles into the air. Chronic overexposure to these emissions will affect human health in numerous ways [60].

  Exposure over a long period at a concentration as low as 400 ppm can lead to death since the absorption of oxygen to the red blood cells are blocked. The emissions become explosive when reaching a concentration between 13% to 74% in air. At lower concentrations the excess gas leads to an increase of respiratory- and heart rates. Concentrations greater than 20% are extremely toxic for the human body and can result in rapid death. The gas produces nitric acid (HNO_3) in the human respiratory system that leads to pulmonary oedema. The gas can be considered as toxic at concentrations of 1 ppm. A concentration of 90 ppm can lead to death within 30 minutes [61].

- Fires and explosions

  A pressing concern of underground fires and explosions are not the blast or fire itself, but the contamination of the toxic gasses (like CO) released into the air. Inhalation of toxic gases leads to death, and this makes it one of the most hazardous occurrences in a mine. Additionally, visibility becomes extremely poor when an underground mine is filled with smoke, and this makes it impossible for miners to navigate the mine. The gases can only exit the underground system using the limited connections to the surface [62].

- Radiation

  Uranium can be found anywhere in the earth's crust. Therefore, the elements radium and radon can be found in all rock, water and soil in underground mines [63]. The radium and its progeny emanation occur between the mineral crystals and the rock structure. The gas then escapes to the surrounding atmosphere through rock interstices or fractures [64].

  These gases are odourless and invisible, making it impossible to detect with human senses. The human body also gives no immediate indication of exposure to these gases [64]. Human exposure to the radium progeny is a massive danger to the lungs and increases the chance of developing lung cancer [63].

- Dust

  The atmospheric air that we breathe on the surface is not solely made of gases but also consists of solid and liquid particles. The solid and liquid particles are known as aerosols and are invisible to the naked eye. Aerosols become part of our atmospheric air by means of natural and industrial sources.

  The respiratory system of all air-breathing creatures evolved to handle most aerosols that arise naturally. However, with the increase of aerosols, due to industrial operations, the concentration of contamination may exceed the ability of the respiration system to remove it on-time.

The term “dust” is used to describe the solid aerosols in the air. Mineral dust is formed in an underground mine when rocks are crushed, blasted, ground or broken. The dust concentration levels in underground mines may be hazardous since there is only limited air in an underground system [65]. The most common harmful airborne particles known in the mining industry is silica dust and asbestos fibres. These particles cause diseases like silicosis in the long term [66].

Distribution airflow controllers (passive regulators)

Seals or stoppings are used to seal old tunnels or working areas that are no longer required. They are also used to seal all connections made between the IAW and the RAW during the development of a section. The purpose of a seal or a stopping is to completely prevent air from following a specific path [10], [12].

Doors are used when the airflow through a branch is not required but should be accessible if required. An airlock, which is double doors that are used to restrict the airflow through a single branch, is used when regular access is required. The purpose of these doors is to prevent air from short-circuiting when someone tries to access the area. The double doors are necessary since the frequent opening of the doors may disrupt the ventilation conditions of a section. The doors are also commonly used in areas with a high-pressure difference, such as booster fan chambers [10], [12].

A regulator is an artificial resistance implemented in a tunnel or branch [67]. The passive regulator consists of a door or a wall with an adjustable orifice. It is typically implemented as a rigid ventilation column through an airlock. Conversely, an active regulator is a duct with a fan that is placed through an airlock [10]. Brattice cloth nailed to the door frames can also be used as passive ventilation controllers. The airflow can then be controlled by adjusting the size of the openings between the strips [12].

An air-crossing is defined as a tunnel that is built over or under another tunnel to avoid a connection between the two tunnels. Air-crossings are used to avoid fresh air from mixing with contaminated air. There are two types of air-crossings, namely overcast and undercast. Overcast (crossover) is the most commonly used application, while the undercast is a simple and alternative application that can be used when an overcast is not possible [14]. A raise borehole, on the other hand, is an additional, vertical tunnel made between levels to allow air to follow an additional route to either enter or exit a section [68].

Primary ventilation system quantity control strategies

The PVS uses fans along the primary path to ensure an adequate amount of airflow through each section of the mine. The surface main fans determine the total amount of air distributed through the mine. All the air that moves through the mine should, at some point, pass through the main fans. Surface main fans are essential, and some mines have backup surface main fans in case of failures or maintenance [10], [12].

Three kinds of surface main fan configurations are used on a mine, namely a forcing system, push-pull system, or an exhausting system, as seen in Figure 32 [10], [69].

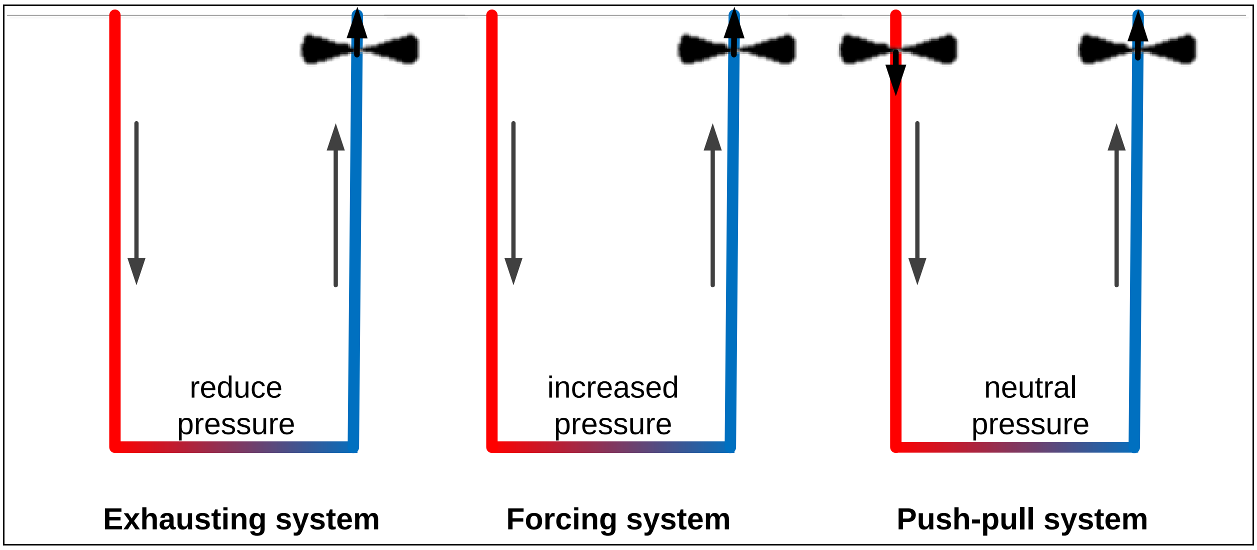

Figure 32: Surface fan configurations (based on [10])

The most common application in a deep-level mine is the exhausting system. Exposure of mine personnel to contaminated return air is avoided at all costs. Consequently, a mine uses the downcast shaft for hoisting. The forcing and push-pull systems both have fans at the downcast shaft side. The constant activities occurring in a shaft increases the likelihood of circulation or leakages [10].

A booster fan is an underground fan, located in series with the surface fan(s), that boosts air passing through by increasing the pressure. Booster fans are used in both PVSs and SVSs. A booster fan is generally added to a fast-developing mine's system especially when the mine develops to such an extent that the surface fans do not have the pressure capability to overcome the pressure losses.

Booster fans are commonly used in PVSs to return air to the upcast shaft. The booster fan can also be used to augment ventilation in deeper, more distant sections of a mine [6], [12], [70].

Primary ventilation system quality control strategies

Un-cooled air at depths below 2 km acts as a heat load instead of a heat sink. Conditions worsen as the quantity of un-cooled air that is supplied to a system increases. Consequently, the heat generated at these depths needs to be removed through refrigeration. The PVS uses bulk air coolers (BACs) to lower the temperature of the air. BACs are located either on the surface, underground or at both locations depending on the layout of the mine [33].

A BAC uses chilled water, provided by a refrigeration system, to cool the air that passes through it. This is achieved by either spraying the water or using a finned tube heat exchanger. An axial fan is used to force air through the sprayed water or coils to remove the heat from the air, thus improving the air quality. A surface BAC is usually located near the shaft with ventilation fans to force ambient air through it. BACs are commonly used on deep-level mines due to the tremendous heat pickup of auto compression [6], [33].

Underground BACs are used when cooling closer to the district areas of the mine is required. An underground BAC is then used to cool the air before it enters a section [6], [15], [33].

Secondary ventilation system quantity control strategies

An SVS supplies air to the working areas that require ventilation. As previously mentioned, the SVS taps off from the PVS to ventilate the working areas. This includes ventilating a blind heading (dead-end tunnel) or a district area [17].

Auxiliary fans

An unfinished tunnel under development is called a blind heading. It is a tunnel that stops with a dead end. A blind heading has no airflow and this causes the stagnant air to become contaminated by the virgin rock temperature and mining operations [71], [72].

An auxiliary fan that is connected to a duct creates a pressure difference which results in airflow through the duct [6]. Line brattices can be used instead of ducts, however significant resistance is then added to the system. The duct length will be extended regularly to remain near the developing face. There are generally four types of configurations regarding auxiliary ventilation.

Each configuration has its own set of advantages and disadvantages. The configuration used is highly dependent on the greatest contributor to air contamination (which is dependent on mining methods and the geological aspects of the mine) [10].

1. Exhausting system

   An exhausting system consists of a fan extracting air through a duct from the working face of the blind heading. The contaminated air at the face of the heading is drawn into the duct and removed from the working area. Fresh air, from the main intake air (PVS), then flows along the full length of the heading to reach the face. A simple representation of the system is shown in Figure 33. This system is also a more detailed representation of the development marked H in Figure 1 [10], [73].

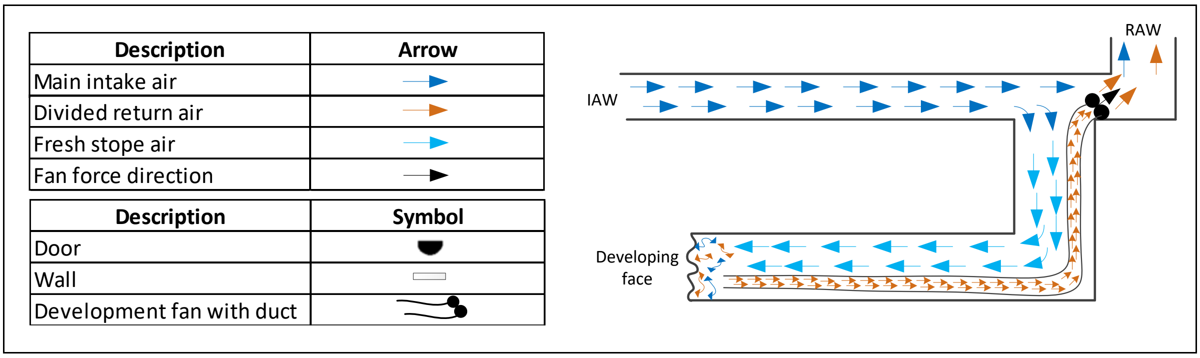

Figure 33: Exhausting auxiliary fan system (based on [10])

   The exhausting system is the preferred system when the dust is the largest contributor to air contamination. The main reason for this is the safe removal of contaminated air through a concealed duct. The duct of this system should preferably be near the face to allow sufficient mixing of air with contaminants. Additional studies showed that the face ventilation effectiveness of this auxiliary system is 10% [10], [73].

2. Forcing system

   A forcing system consists of a fan forcing air through a duct to the working face of the blind heading. Fresh air from the main intake air (PVS) is forced into the heading and blows against the face. The contaminated air at the face of the heading then flows out the full length of the heading back into the PVS. A simple representation of the system is shown in Figure 34 [10], [48], [73].

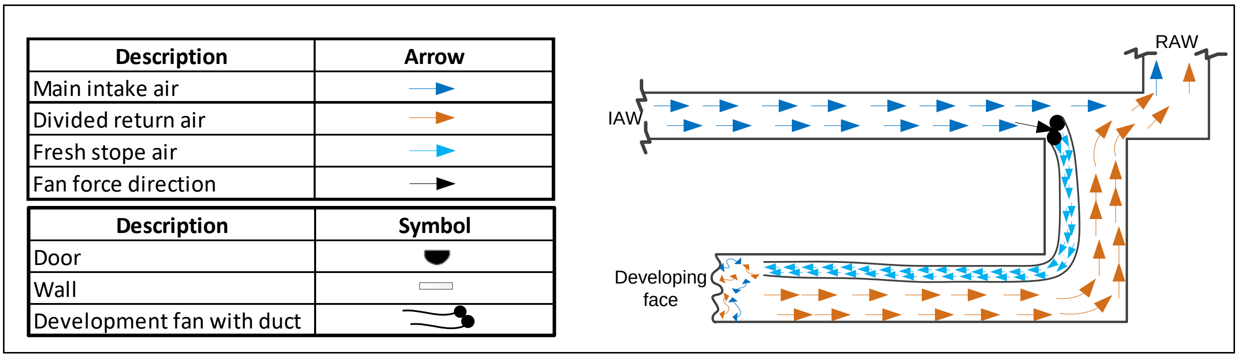

Figure 34: Forcing auxiliary fan system (based on [10])

This system is preferably used when heat or methane is the largest contributor to air contamination. In this system, the air is forced with a high velocity against the face. This causes a turbulent mixing effect of the air and contaminants before the air exits the heading. Additional studies showed that the face ventilation effectiveness of this auxiliary system is 39.9%. The major disadvantage of this system is that the contaminated air exiting the heading affects the entire heading [10], [73].

3. Forcing system with exhaust overlap

This system consists of a force fan with a duct and an overlapped exhaust fan with a duct. The force fan blows air against the face, mixing the air with the contaminants. An extraction fan then draws out the air through a duct and removes it from the working area [10], [55], [74].

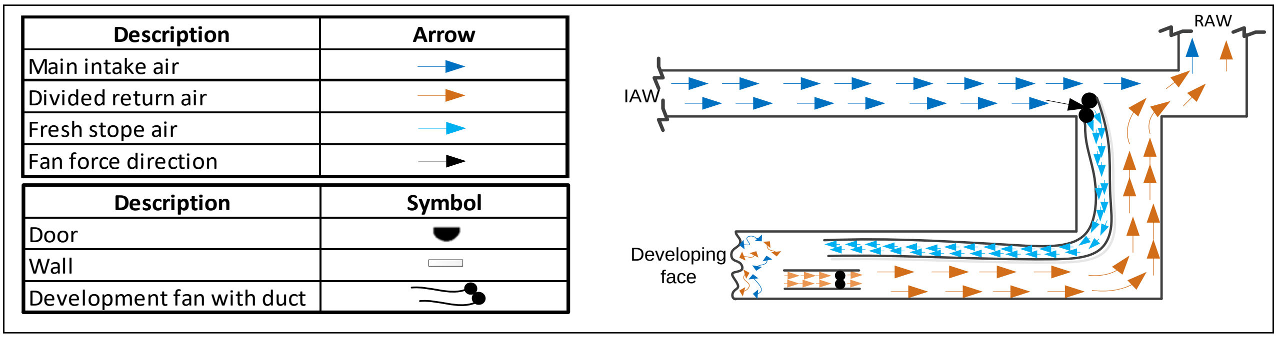

Figure 35: Forcing system with exhaust overlap (based on [10])

The advantage of this system is that the force fan ensures sufficient mixing between the air and the contaminants. However, the entire heading is affected by the contaminated air exiting the heading [74].

4. Exhausting with force overlap

   This system consists of an exhaust fan with a duct and an overlapped force fan with a duct. The extraction fan draws out the air through a duct and removes it from the working area. The force fan also blows air against the face mixing the air with the contaminants. [10], [55], [74].

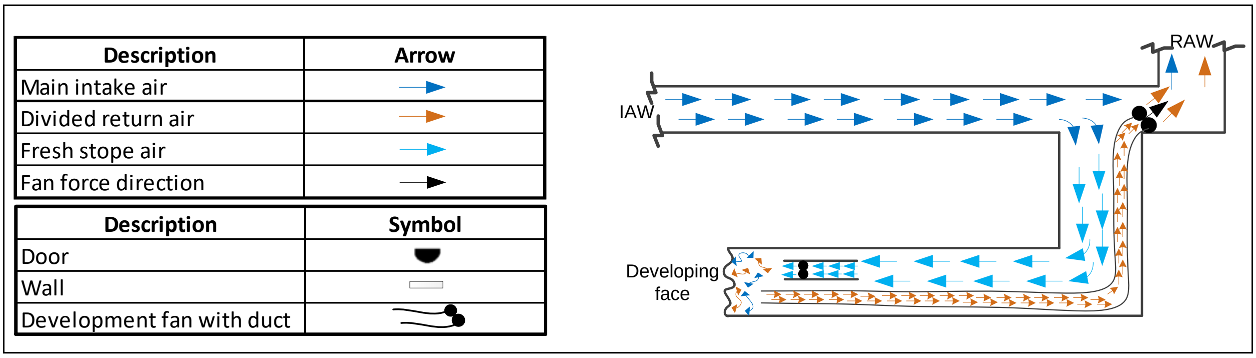

Figure 36: Exhausting with force overlap (based on [10])

   An advantage of this system is that the force fan ensures a proper mixture occurs between the contamination and the air before it is removed by the exhaust fan in the concealed duct. Therefore, the contaminated air does not flow through the full length of the heading [10], [74].

Non-developing dead ends, namely workshops, worker lunchrooms, battery bays, etc. also need to be ventilated. These working areas are usually located in a connection between the IAW and the RAW. These working areas can be ventilated by means of an auxiliary fan or by opening a hole in the stopping and regulating the air leaking between the IAW and RAW [75].

District fans

Through flow, ventilation is implemented on workings when the developing phase is completed. The tunnel has an entrance and an exit through which the air can flow [43], [76]. A district can, therefore, be classified as a mining area (stope line) [77].

The district fans or circuit fans are used to regulate the amount of air directed to a specific district. The implementation of the district fans depends on the site where it is implemented. The common district control method is implemented using an axial fan, airlock and corrugated spiral-ducting. Air is forced by means of the fans through the ducts that bypass the airlock. Then the pressure on the inside of the airlock is increased to allow the air to flow through the entire working to the return [77], [78].

A simple representation of the system is shown in Figure 37. This system is also a more detailed representation of the active working marked *K* in Figure 1 [10], [73].

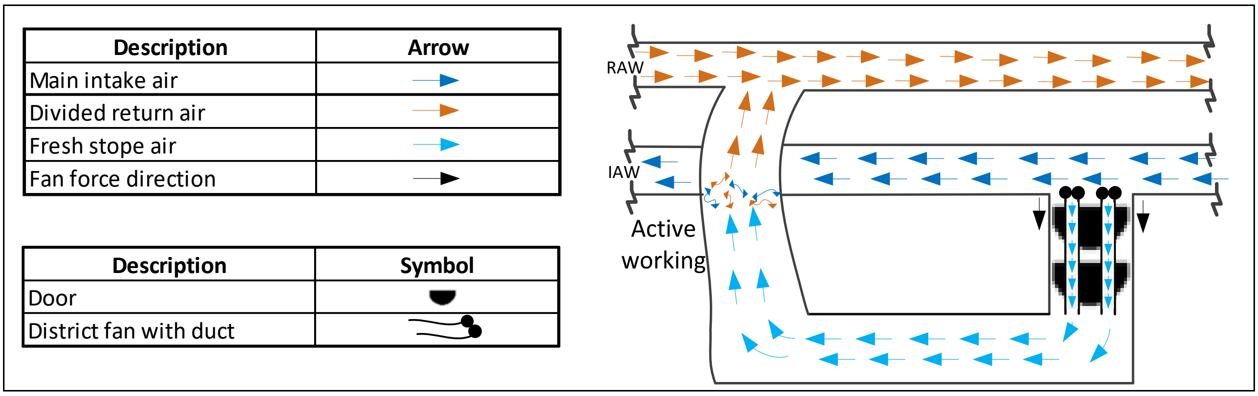

Figure 37: District fan system (based on [77]**)**

Secondary ventilation system quality control strategies

Additional cooling methods are used when the PVS's air quality is not deemed adequate to ventilate the working areas. Spot coolers (cooling cars) or spray chambers are usually used in a district or a development working area to reduce the temperature of the air [17], [46].

The SVS quality control methods are based on the same principals as the PVS's BAC. The spot cooler (cooling car) consists of an axial fan and a finned tubed heat exchanger. Chilled water flows through the tubes of the heat exchanger, while air is forced over the tubes by means of a fan. Heat transfers from the air to the chilled water, thus improving the temperature of the air [17], [46]. Figure 38 illustrates a simple layout of a spot cooler.

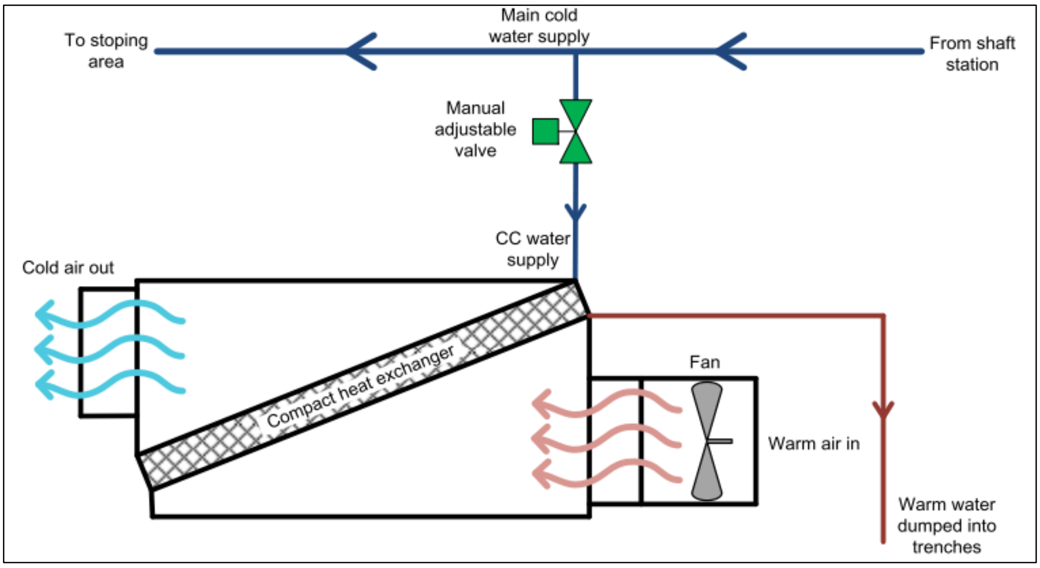

Figure 38: Spot cooler (cooling car) [46]

Spray chambers are based on the same principle as spot coolers. Chilled water is sprayed in a tunnel while air flows through it. Heat is transferred from the air to the water, thus reducing the temperature of the air. With this method the heat transfer is more effective since the water is in direct contact with the air [46]. Figure 39 illustrates a simple layout of a spray chamber.

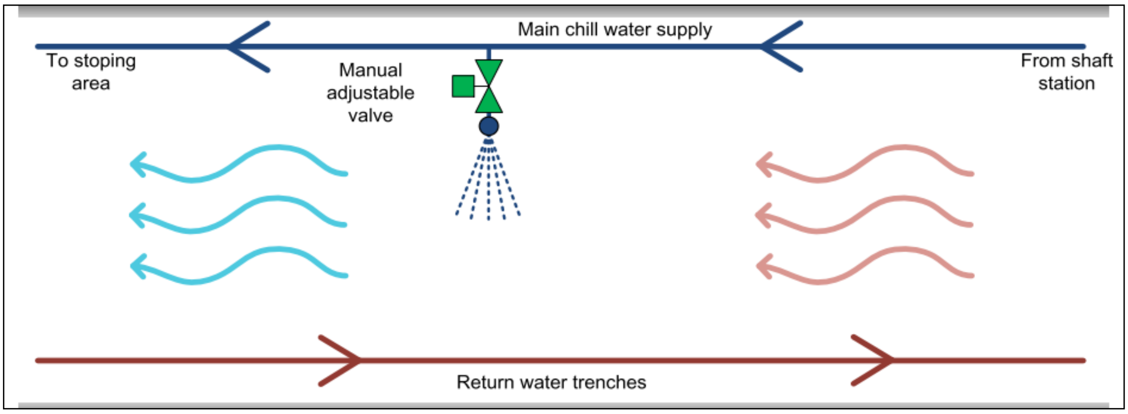

Figure 39: Spray chambers (secondary ventilation) [46]

South Africa gold mining operational cost

The mining and industrial sector in South Africa consumes almost 40% of the energy supplied by the country [28]. Approximately 95% of the electricity supplied by the country comes from a company called Eskom [30]. The power utility's electrical tariffs-increase rate rose above the consumer price index (CPI) rate in 2003 [79], shown in Figure 40.

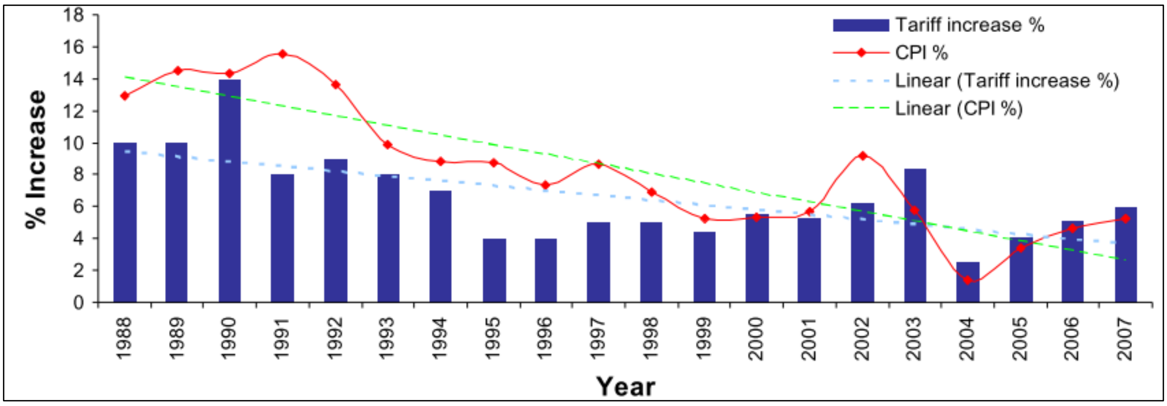

Figure 40: Eskom tariff increase compared to CPI [79]

This meant that South African mine expenses were continually rising without any corresponding rise in production outcome. The South African gold mining industry was the leading gold producer in the world. Today, however, less than 20% of the gold mines in South Africa are still operating profitably [31], [32]. Mines are profitable when the production outcome of the mine is more than their operational cost. The operational costs of gold mines in South Africa are mainly increased due to the electrical tariffs increase. Considering the cost increase of Sibanye Gold (South African Gold mine) for the year 2007, 2012 and predicted for 2017, one can see the influence the electrical expenses has on the other sectors of the mine [30], shown in Figure 41.

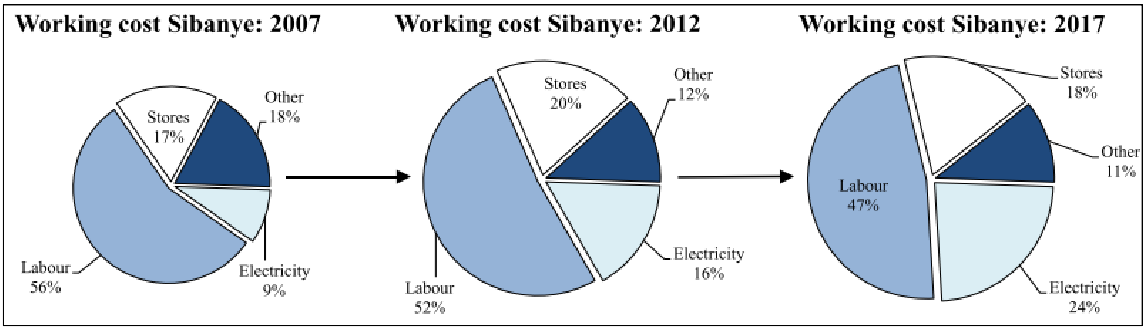

Figure 41: Sibanye Gold operational cost [30]

Internationally, increases in electricity tariffs are the mining industry's largest contributor to expenditures [29]. Historically low electrical tariffs made mine personnel unaware of the excessive use of energy. The management and sustainability of energy is the central focus of today's mines. The aim is to use energy efficiently for as long as possible. This will lead to a decrease in the mine's operational costs and therefore extend the operational lifetime of the mine [33].

Demand-side management (DSM)

1. Valley filling

   Valley filling is when the user is encouraged to increase the system load during off-peak periods. This increases the energy usage during the off-peak periods so that an improved load factor is achieved. This technique may be desirable when the average energy price is higher than the long-run incremental cost. The method, however, is not commonly used since it does not reduce the average power consumption [7], [13], [17].

2. Peak clipping

   Peak clipping is when the system load is clipped during the peak-periods. This enables the user to reduce expensive electricity use. This is usually implemented when blasting takes place and services like compressed air is not necessary during the period. This technique can lead to production losses if not managed correctly. Peak clipping is a crucial technique when a power utility cannot maintain the load during peak-periods [7], [13], [17].

3. Load shifting

   Load shifting is a combination of valley filling and peak clipping. It entails shifting the load from peak periods to off-peak periods. Benefits of both methods are obtained by this method. The overall consumption, however, does not necessarily change. The reduction in operational cost becomes apparent by reducing electrical consumption during the expensive periods (peak periods). This method is used in refrigeration and pumping systems where the system can prepare for a peak period shift by operating at an off-peak period shift [7], [13], [17].

4. Strategic load growth

   Strategic load growth is when the user is encouraged to increase energy consumption regardless of the period. This will typically happen when the excess capacity of electricity is available for a specific duration [7].

5. Strategic conservation

   Strategic conservation is the opposite of strategic load growth where energy consumption should decrease regardless of the period. This will typically happen when there is a shortage of capacity [7].

APPENDIX B.Configurational audit (object list)

The following list is an example of the typical objects to be audited and mapped during a configurational audit:

- Seal
- Fan
- Wall
- Door (open or closed)
- Doorframe (without door)
- Regulator
- BAC
- Cooling car
- Refuge bay
- Workshop
- Resting places
- Battery bay
- Hopper bay
- Ducting
- Blockages
- Barricades
- No-entry areas
- Flooded areas
- Fissure water leaks
- Fridge plants
- Settler dams
- Pumps
- Shaft (downcast and upcast)
- Stores

APPENDIX C. How a black box works

A black box works on the principle that air (mass/matter) cannot be created or destroyed. Therefore, the total air entering a black box must also exit the box (illustration shown in Figure 42).

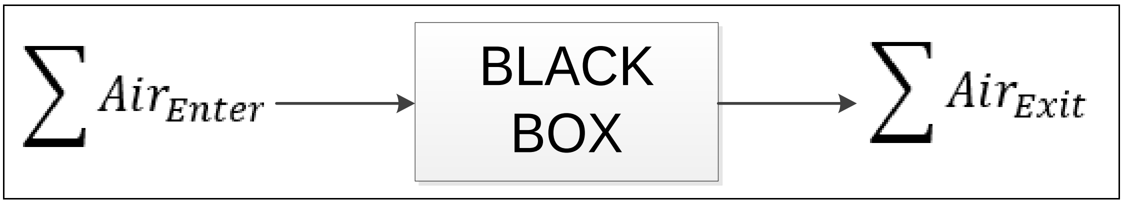

Figure 42: Black box

Equation 5: Black box principle

$$\sum Air_{Enter} = \sum Air_{Exit}$$

APPENDIX D. Additional information of Mine A (connected to Mine B)

CAD drawings

Figure 43 shows a top view of Mine A and B with a connection between the stope lines of the mine.

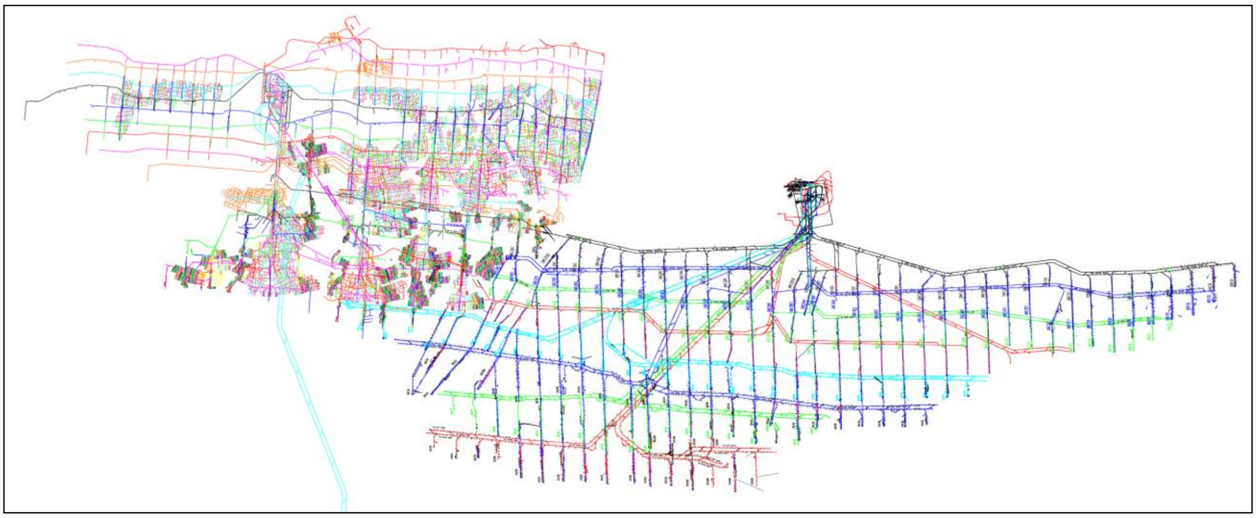

Figure 43: Top view of Mine A (right) connected to Mine B (left)

Figure 44 shows an example of the working area (crosscut) numbering of the mine.

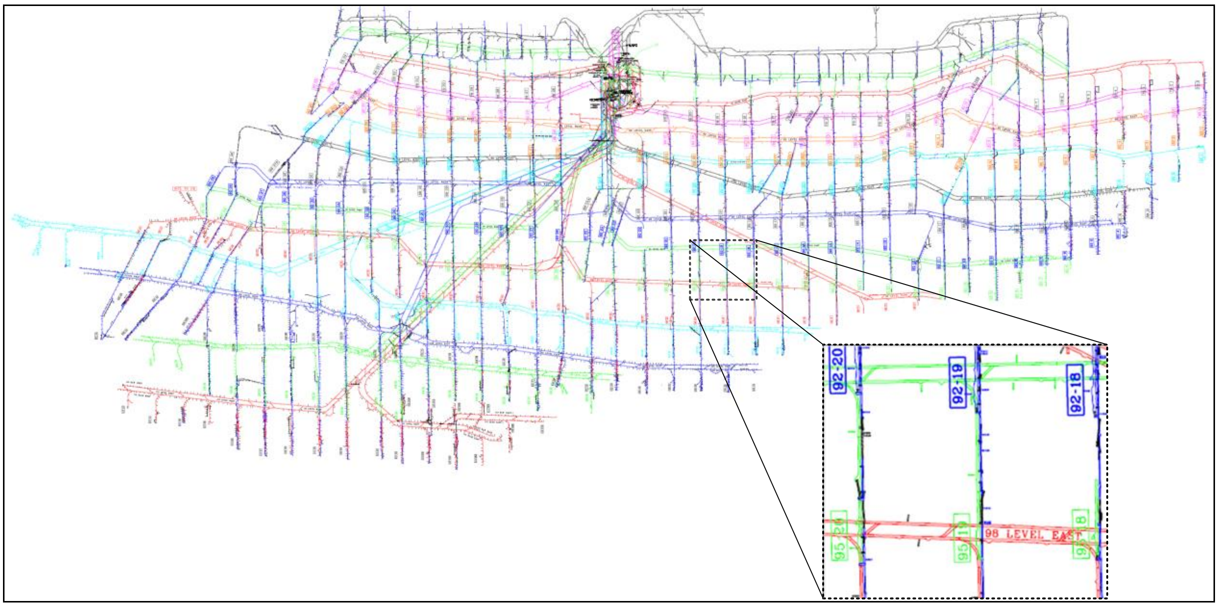

Figure 44: Mine A layout and working area numbering

Figure 45 and Table 27 represent the standard terms used to describe areas in Mine A.

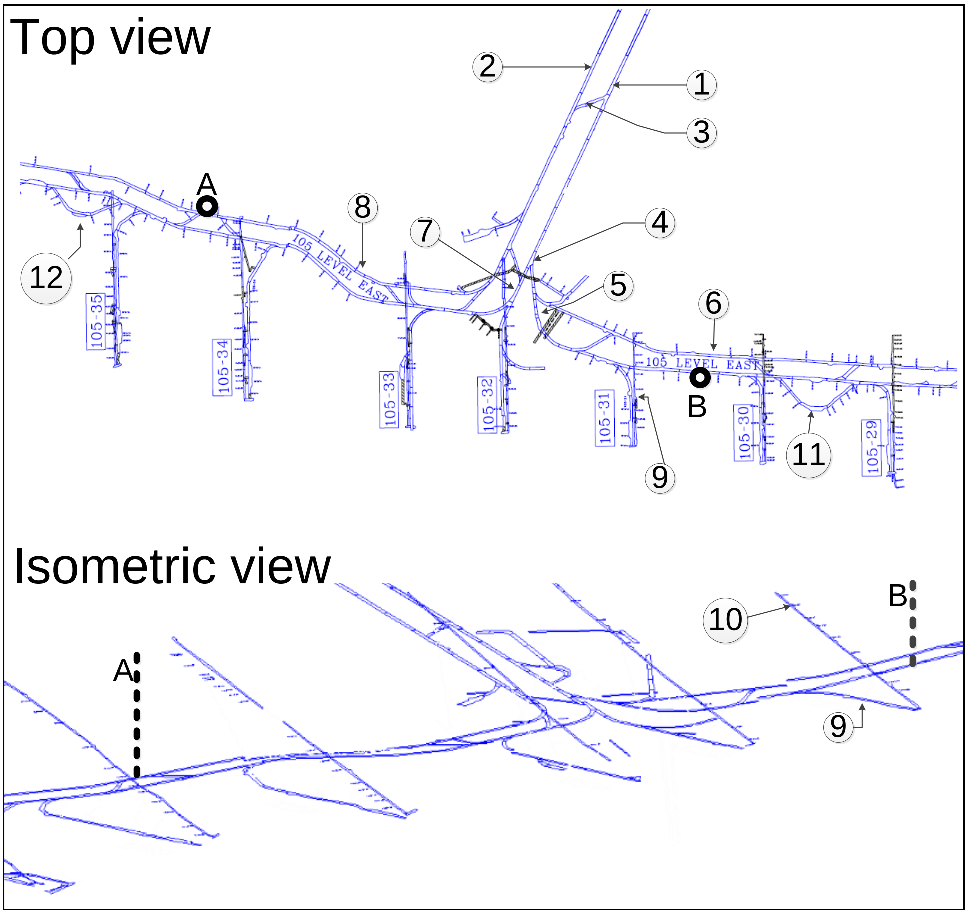

Figure 45: Mine A descriptive layout

Table 27: Mine A descriptive terms

Reference	Description
1	Main IAW haulage
2	Main RAW haulage
3	Connecting tunnel
4	IAW split
5	East IAW haulage
6	East RAW haulage
7	West IAW haulage
8	West RAW haulage
9	Crosscut
10	Stope line
11	East BAC
12	West BAC

Return airflow systems

1. End flow return system

The PVS air not utilised by the SVS will turn via end flows. The airflow path is as follows:

- PVS (IAW): Air downcast and split between Active level 1 and 2.
- PVS (IAW): The air distributed per level then splits either east or west.
- PVS (IAW) to SVS (RAW): The air not utilised in the crosscuts returns via the level's RAW and RBHs to the dedicated return level (in this case Active level1's RAW).

2. Stope line return system

The PVS air utilised by the SVS will turn via stope lines towards dedicated return levels. The airflow path is as follow:

- PVS (IAW): Air downcast and split between Active level 1 and 2.
- PVS (IAW): The air distributed per level then splits either east or west.
- PVS (IAW) to SVS (districts): The air utilised in the crosscuts splits off the PVS (IAW).
- PVS (IAW) to SVS (districts): The air utilised in the crosscuts splits off the PVS (IAW) and upcast via stope lines.
- SVS (district lines) to PVS (RAW): The air returns on inactive levels used as return levels.

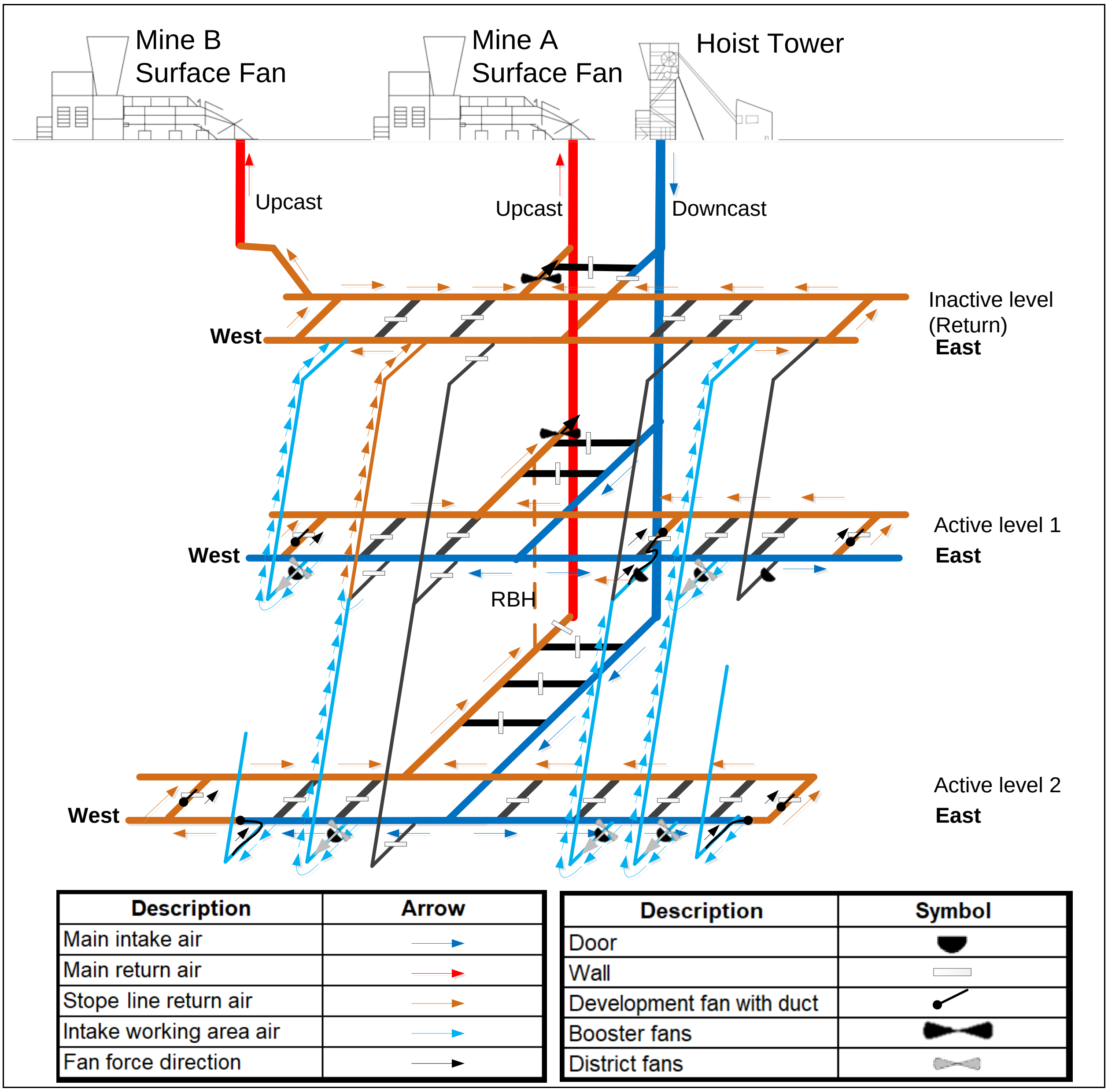

Figure 46: Mine A's simplified ventilation system

APPENDIX E. Calibrated simulation of Mine A graphs

PVS intake calibration

Table 28: Calibrated half level IAW intakes

Detail sections	Description	Average absolute offset (kg/s)	Average percentage offset (%)
Total mine intake	**Total mine intake airflow**	**22.61**	**2.13**
Downcast shaft air distribution	70L	2.00	7.91
	71L	0.92	9.06
	73L	0.87	3.74
	75L	0.00	0.00
	76L	13.69	45.17
	77L	0.00	0.10
	78L	5.92	14.73
	80L	2.35	13.80
	82L	2.63	13.87
	85L	4.95	65.10
	88L	0.56	7.66
	92L	5.18	41.46
	95L	7.98	35.46
	98L	12.03	13.01
	100L	4.53	30.22
	102L	2.09	1.65
	105L	2.81	1.36
	109L	31.11	30.50
	113L	2.72	1.61
	115L	0.18	0.64
	Level intake average	**6.26**	**16.15**
East and West distribution	98L East	3.41	8.21
	98L West	7.62	15.24
	102L East	12.03	21.48
	102L West	8.68	13.24
	105L East	4.00	3.94
	105L West	19.99	25.46
	109L East	13.89	15.45
	109L West	3.31	6.16
	113L East	4.94	7.50
	113L West	4.57	4.75
	East and West split average	**8.25**	**12.14**

PVS return calibration

A black box mass balance of the total upcast shaft (Mine A and Mine B combined) was used to analyse the air distribution of the primary return system (see Figure 47).

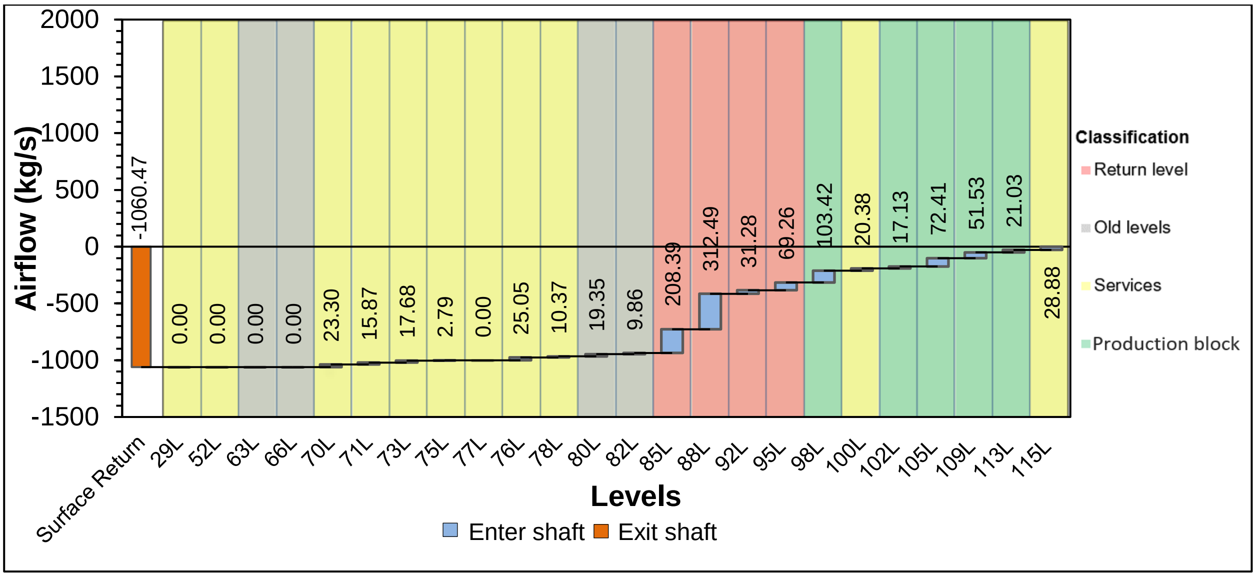

Figure 47: Calibrated PVS upcast shaft air distribution

Table 29: Calibrated PVS return system (end flows and return fans)

Detail sections	Description	Average absolute offset (kg/s)	Average percentage offset (%)
End flows	98L East	7.61	12.27
	98L West	0.26	3.17
	102L East	4.96	12.85
	102L West	1.04	9.45
	105L East	2.92	13.72
	105L West	2.66	11.65
	109L East	0.14	0.55
	109L West	1.82	4.11
	113L East	0.17	0.84
	113L West	1.07	5.16
	Average end flows	**2.26**	**7.38**
Primary return fans	Surface Mine A MF 1	9.09	2.17
	Surface Mine A MF 2	8.99	2.15
	Surface Mine B MF 1	15.07	6.33
	85L BF	10.76	5.03
	88L BF	21.42	12.20

Detail sections	Description	Average absolute offset (kg/s)	Average percentage offset (%)
	98L BF	9.96	9.63
	102L BF	16.91	14.43
	109L BF	24.19	38.15
	115L BF	0.18	0.64
	Average return fans	**12.95**	**10.08**

SVS branches off the PVS additional calibration graphs

98L crosscut airflows

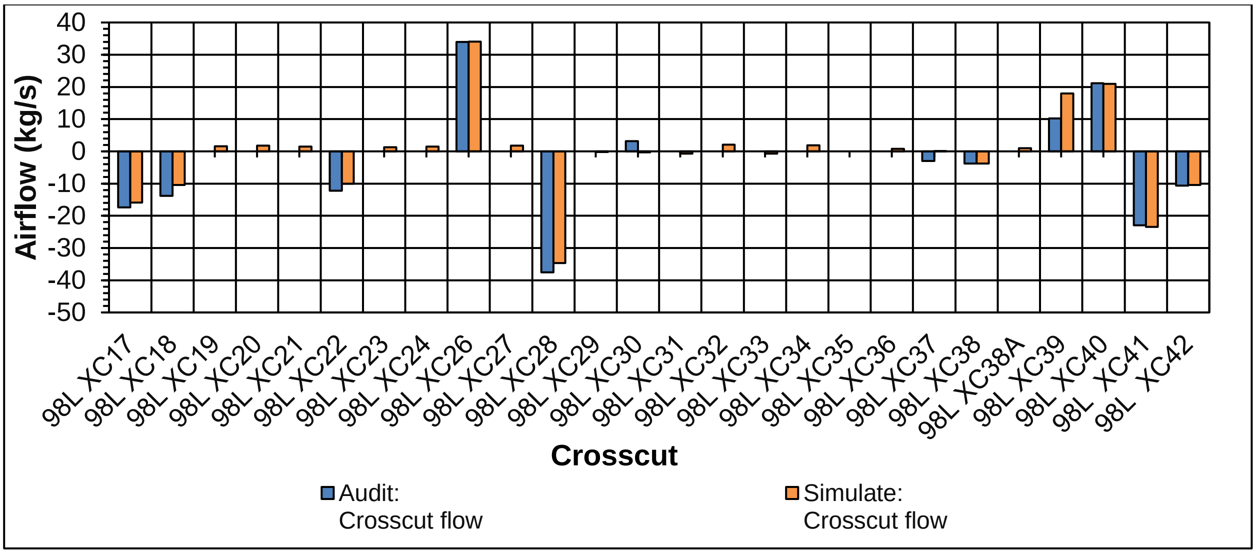

Figure 48: 98L calibrated crosscut (part of SVS) airflow

98L level mass balance

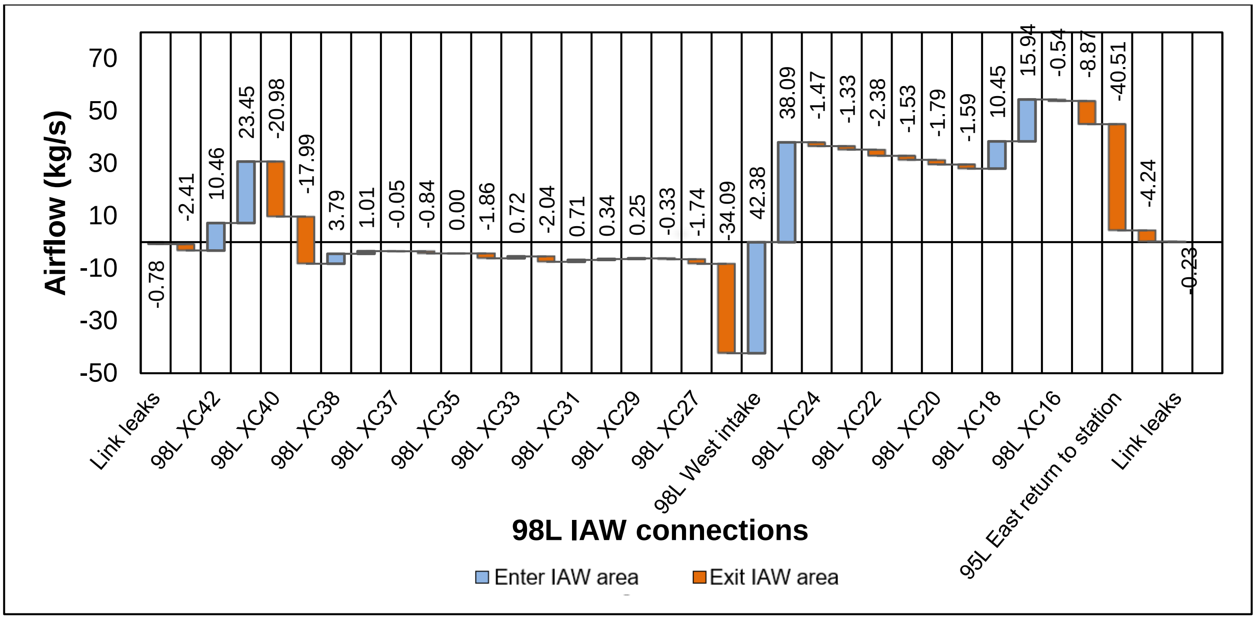

Figure 49: 98L calibrated level mass balance

102L crosscut airflows

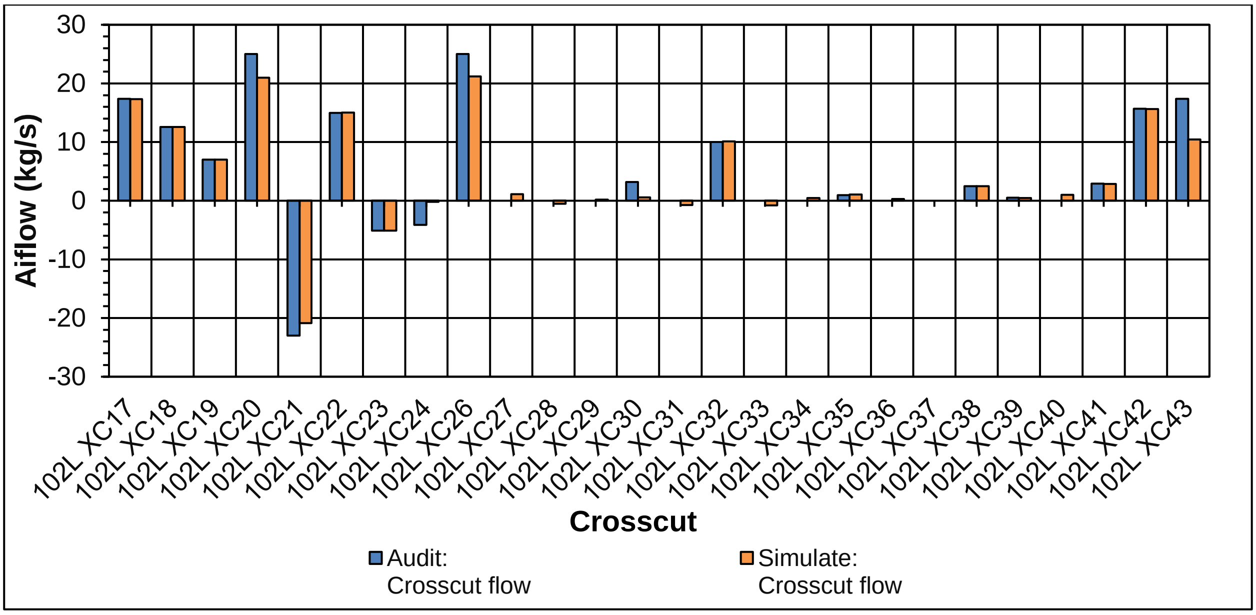

Figure 50: 102L calibrated crosscut (part of SVS) airflow

102L level mass balance

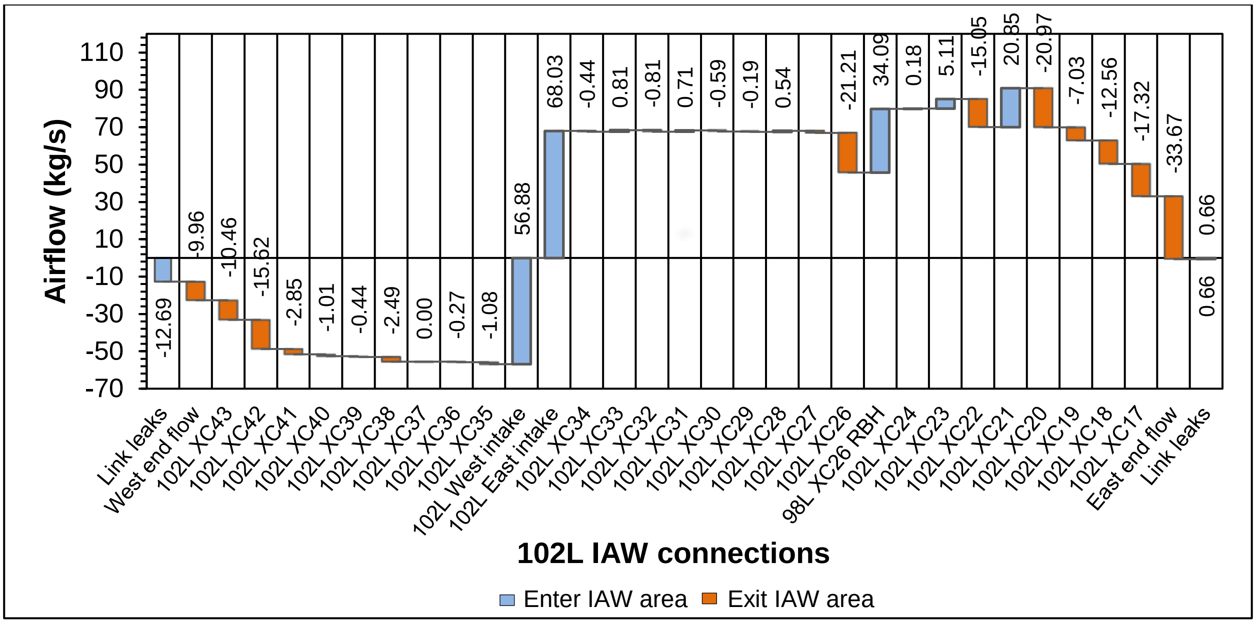

Figure 51: 102L calibrated level mass balance

105L crosscut airflows

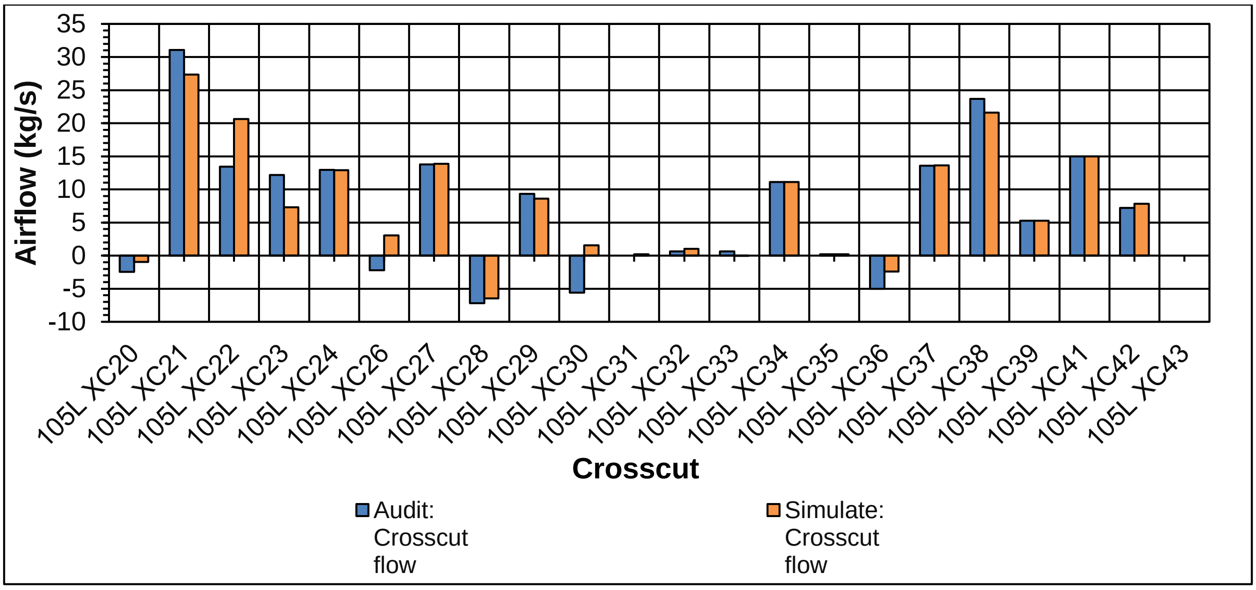

Figure 52: 105L calibrated crosscut (part of SVS) airflow

105L level mass balance

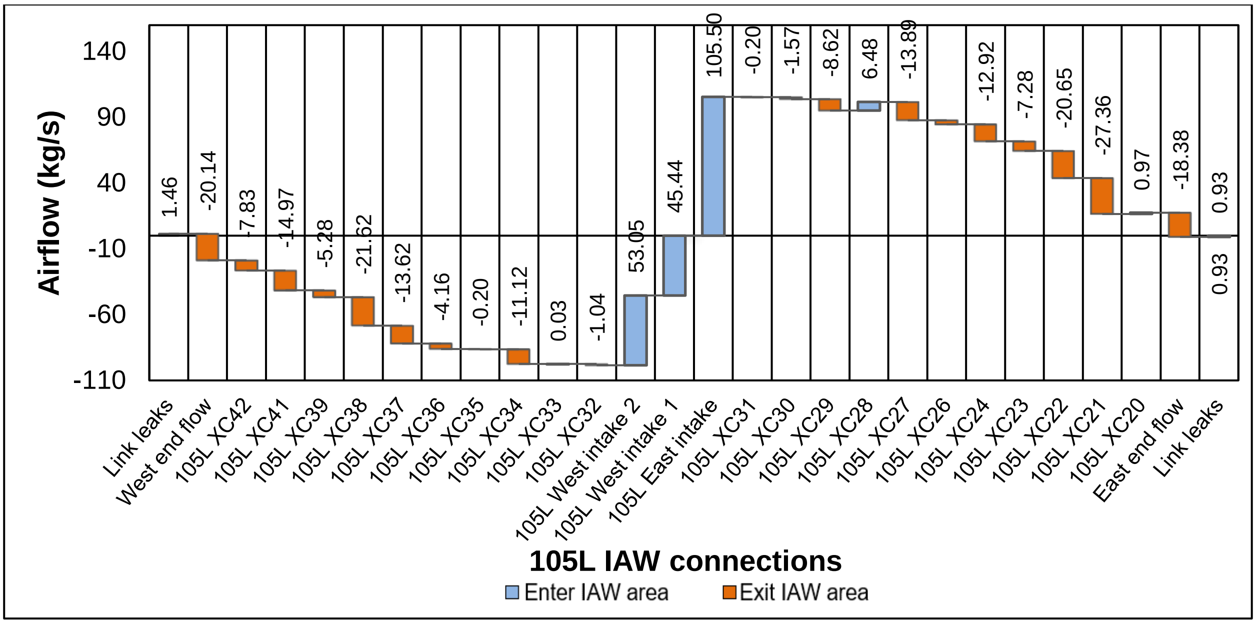

Figure 53: 105L calibrated level mass balance

109L crosscut airflows

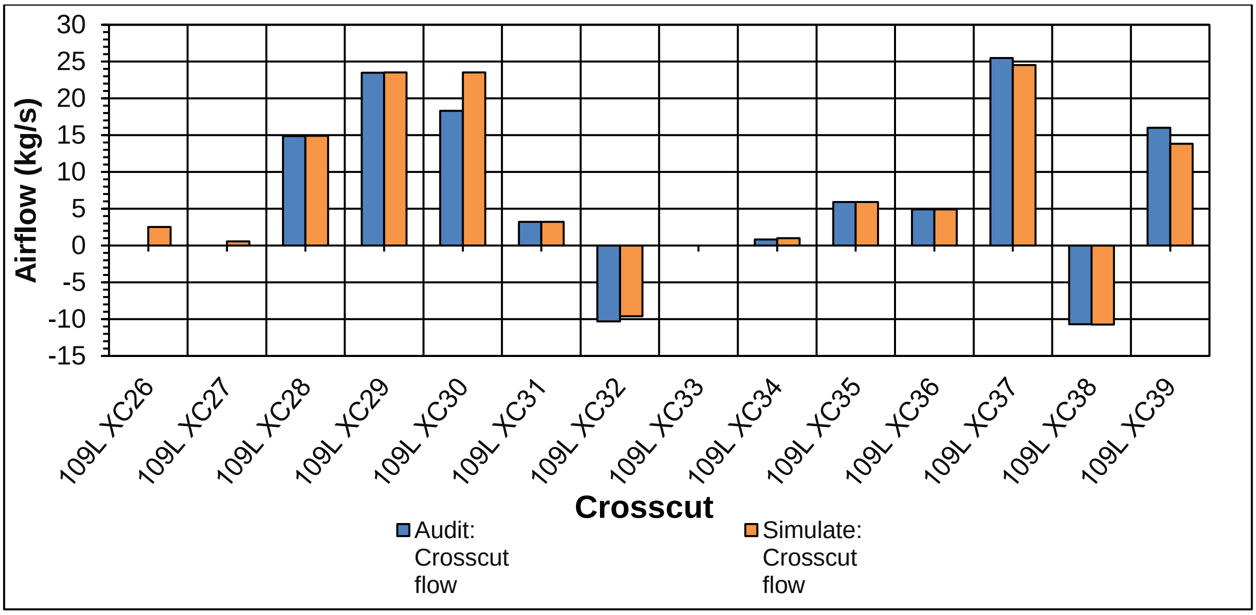

Figure 54: 109L calibrated crosscut (SVS) airflow

109L mass balance

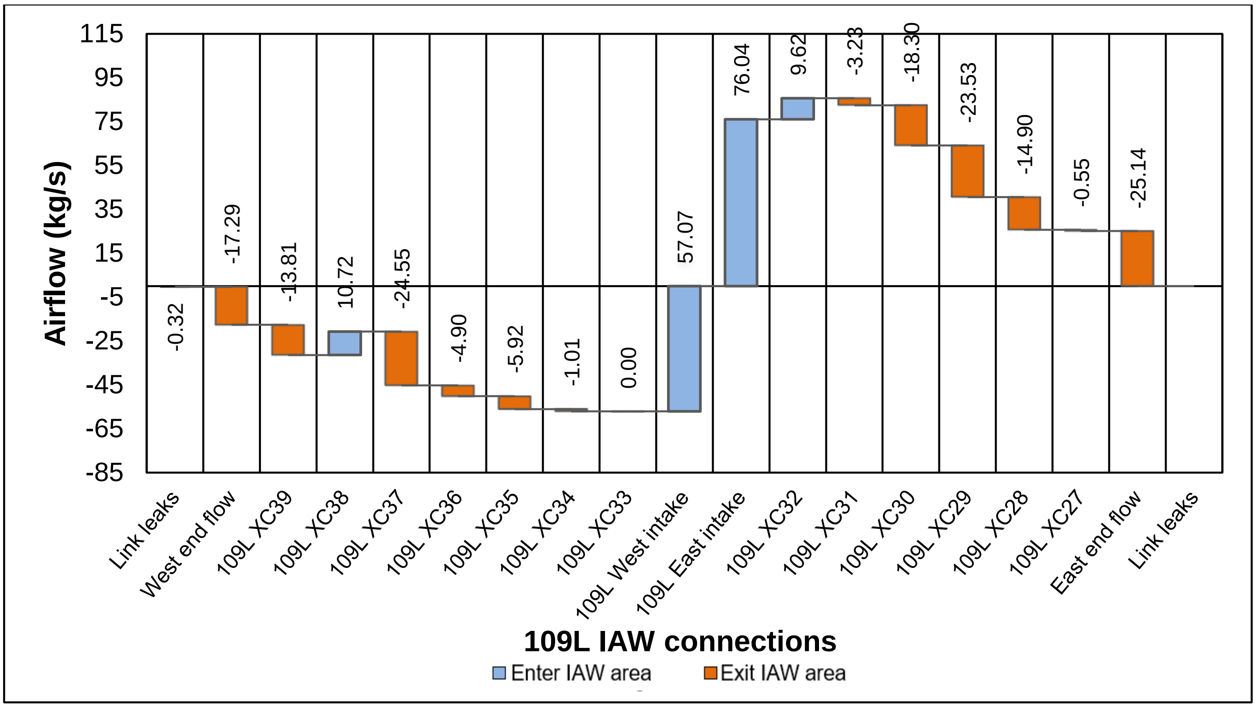

Figure 55: 109L calibrated mass balance

113L crosscut airflows

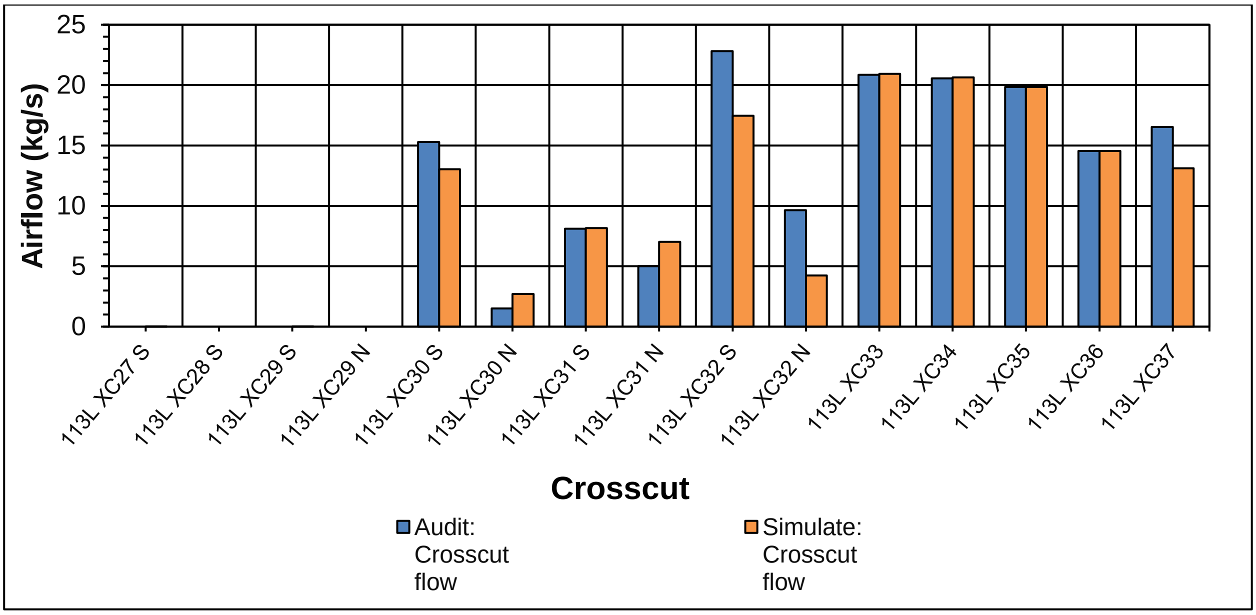

Figure 56: 113L calibrated crosscut (SVS) airflow

113L mass balance

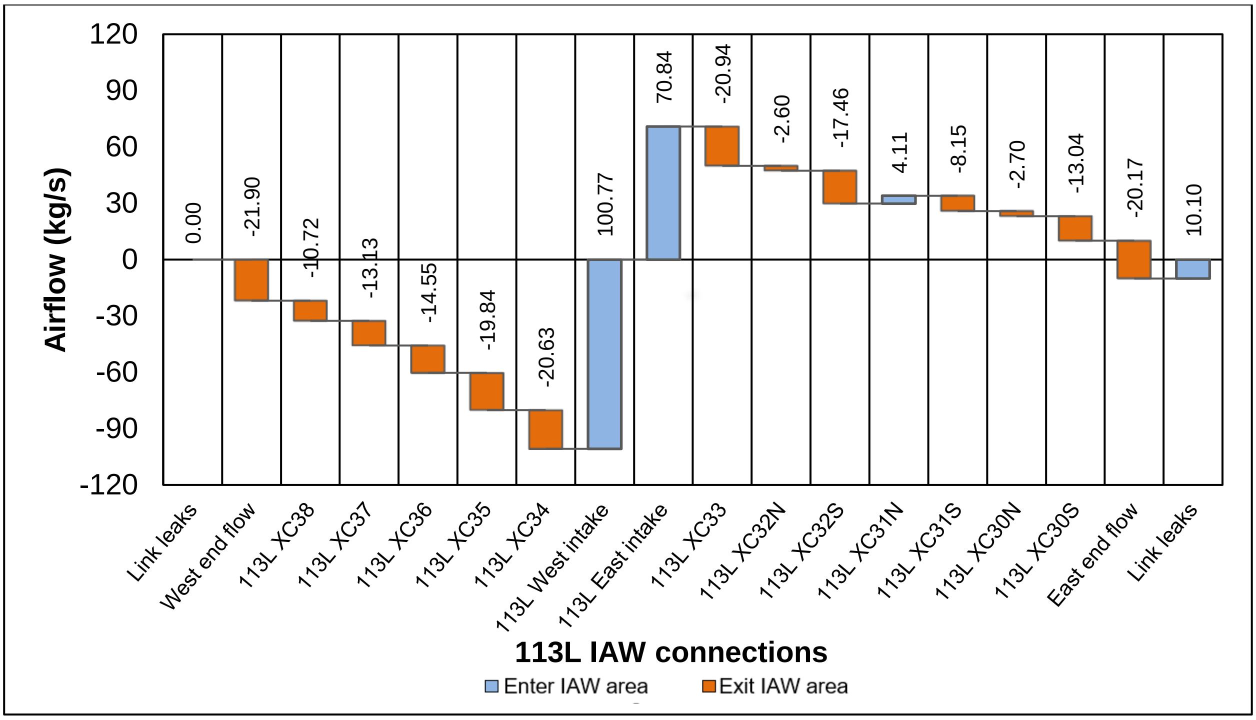

Figure 57: 113L calibrated level mass balance

SVS returns to the PVS additional calibration graphs

85L crosscut airflows

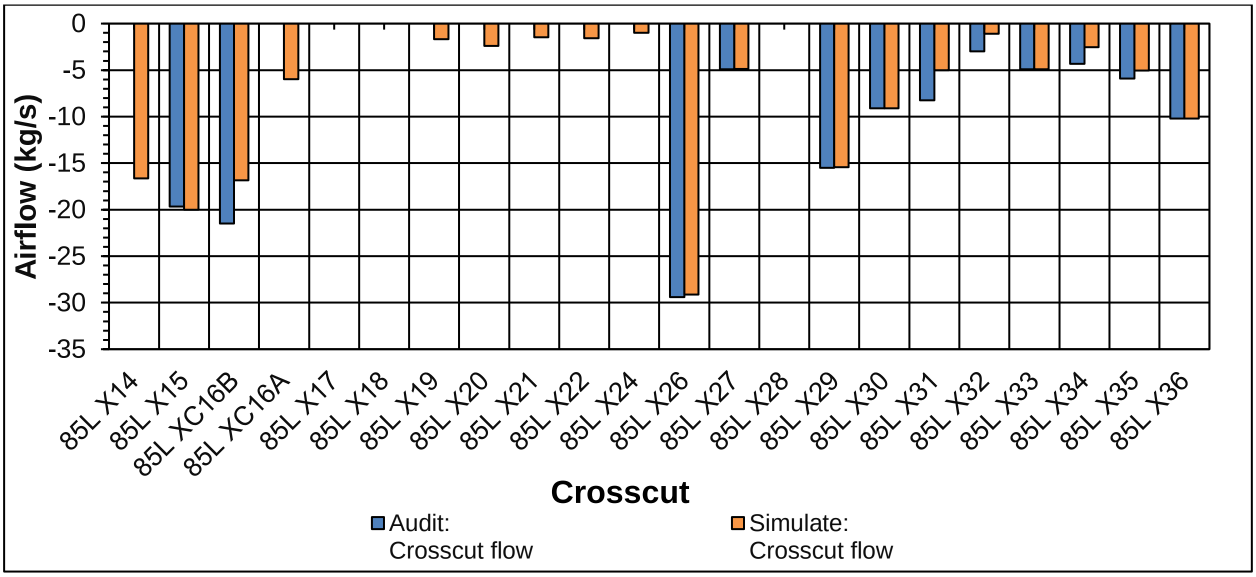

Figure 58: 85L calibrated crosscut (SVS) airflow

85L mass balance

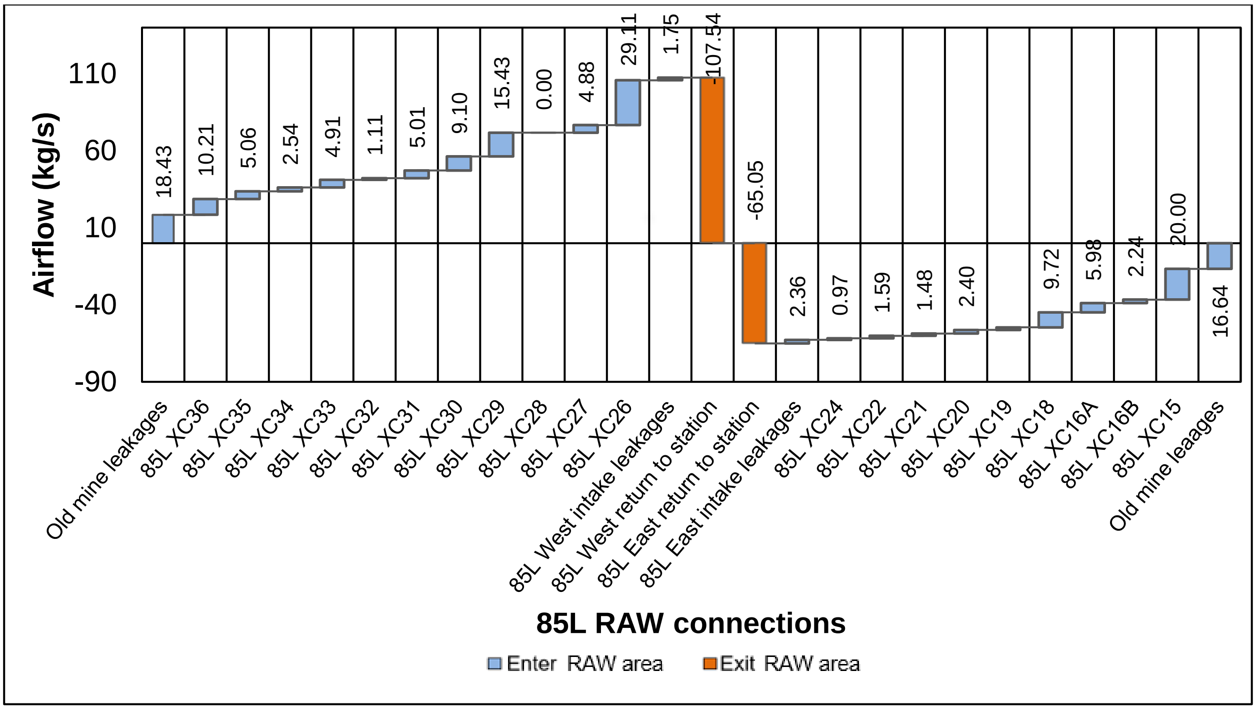

Figure 59: 85L calibrated level mass balance

88L crosscut airflows

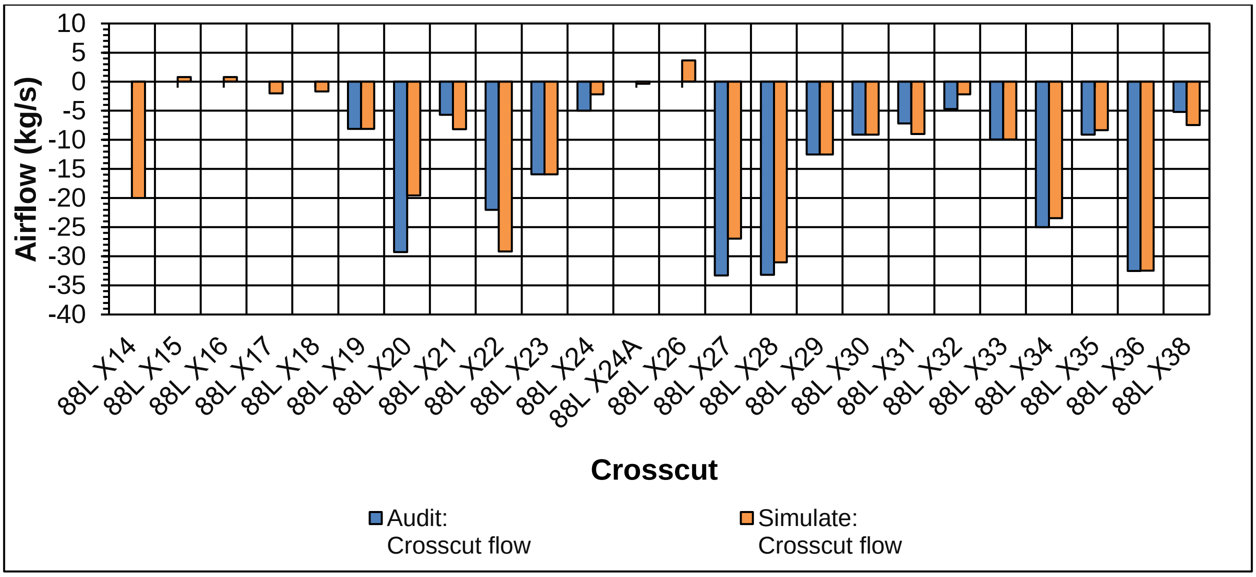

Figure 60: 88L calibrated crosscut (SVS) airflow

88L mass balance

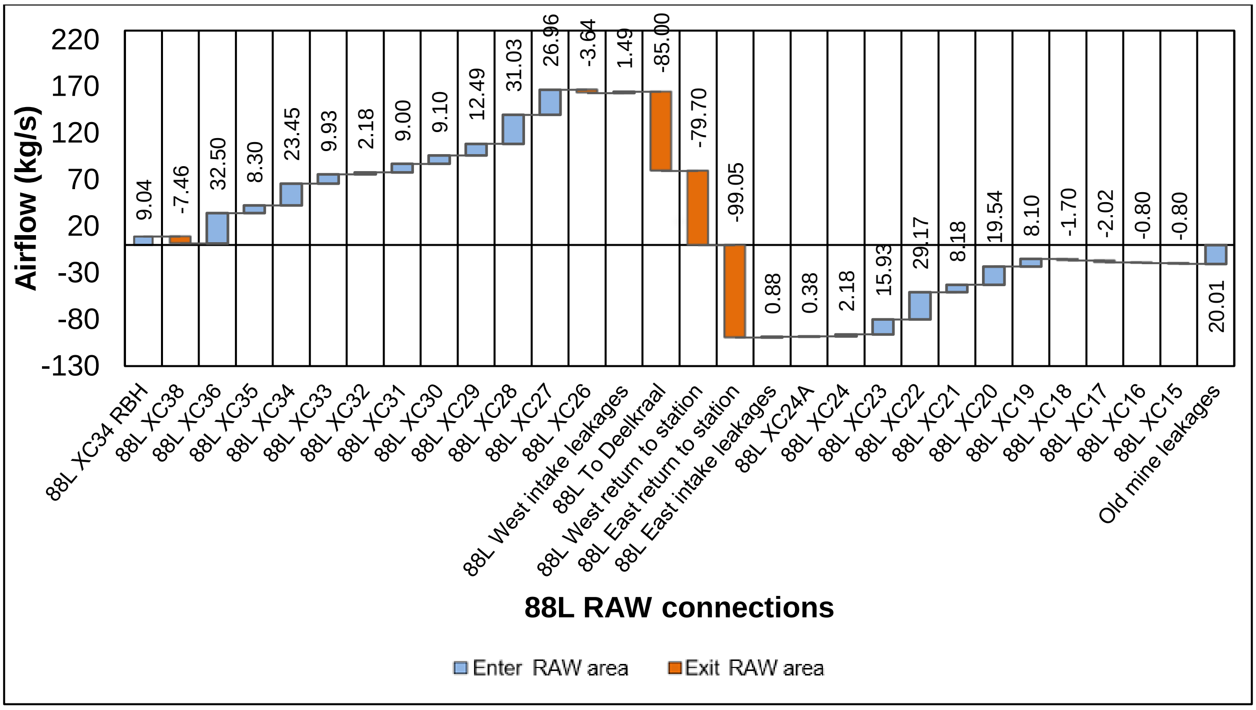

Figure 61: 88L calibrated level mass balance

92L crosscut air flows

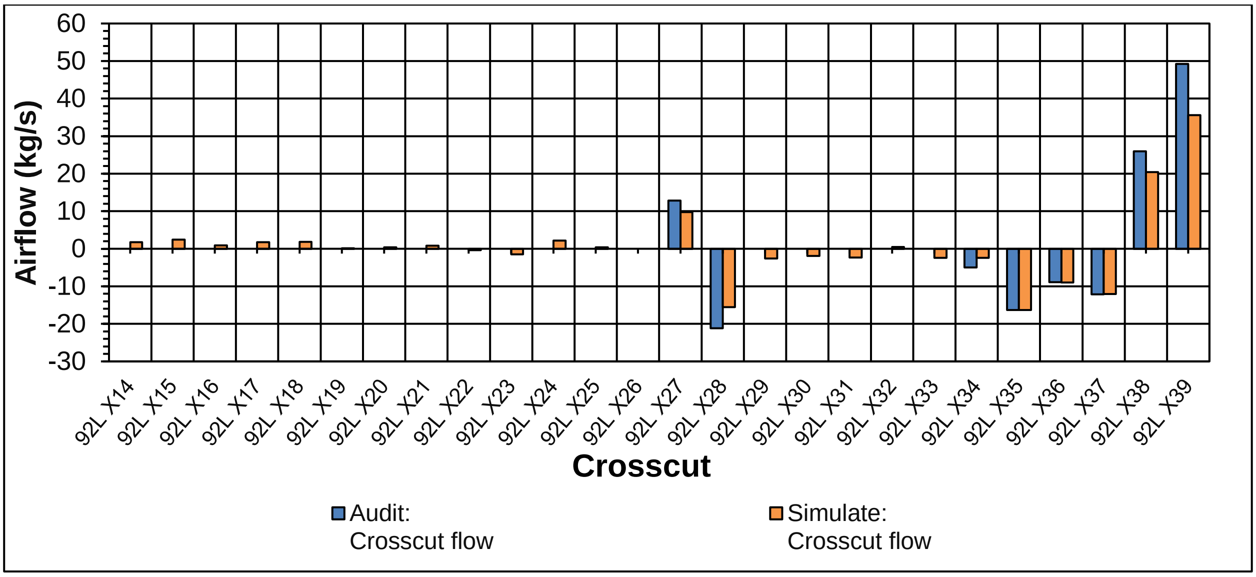

Figure 62: 92L calibrated crosscut (SVS) airflow

92L mass balance

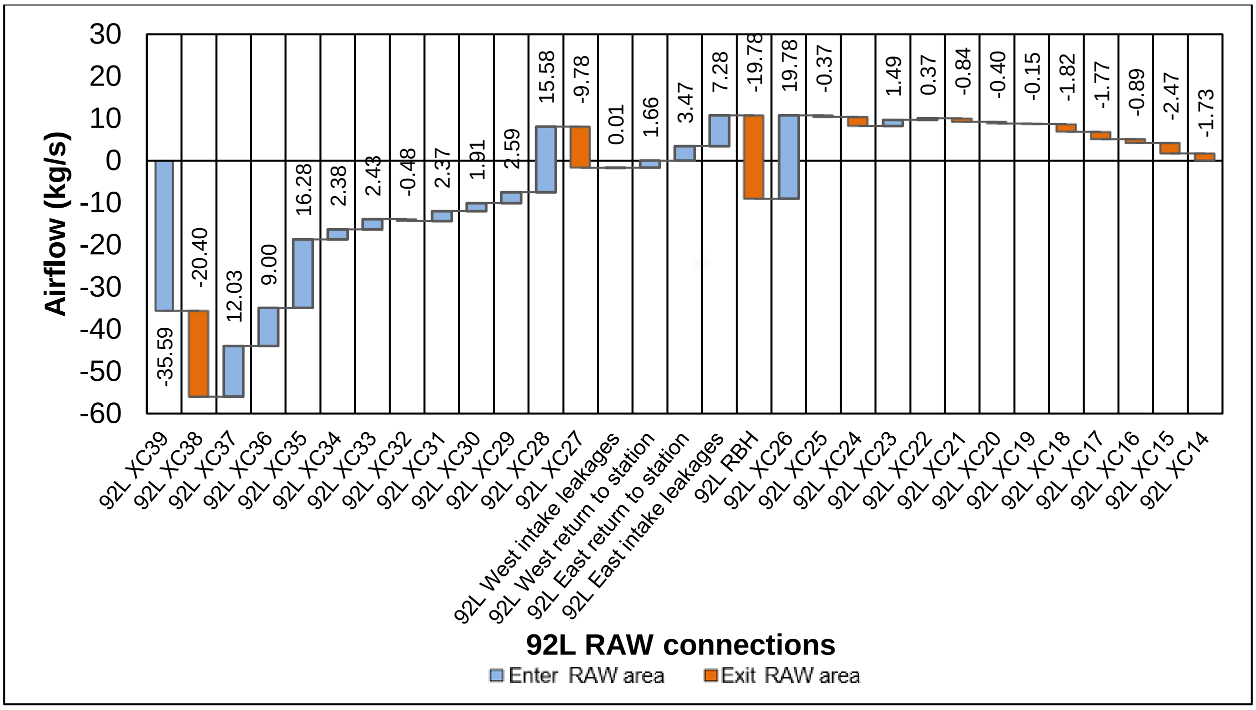

Figure 63: 92L calibrated level mass balance

95L crosscut air flows

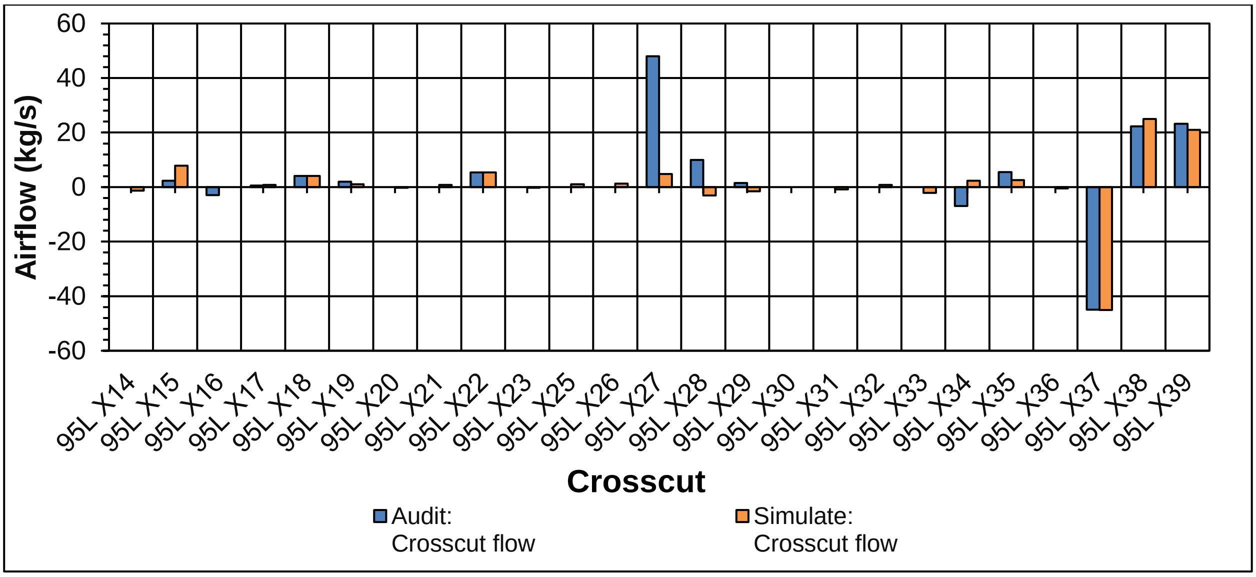

Figure 64: 95L calibrated crosscut (SVS) airflow

95L mass balance

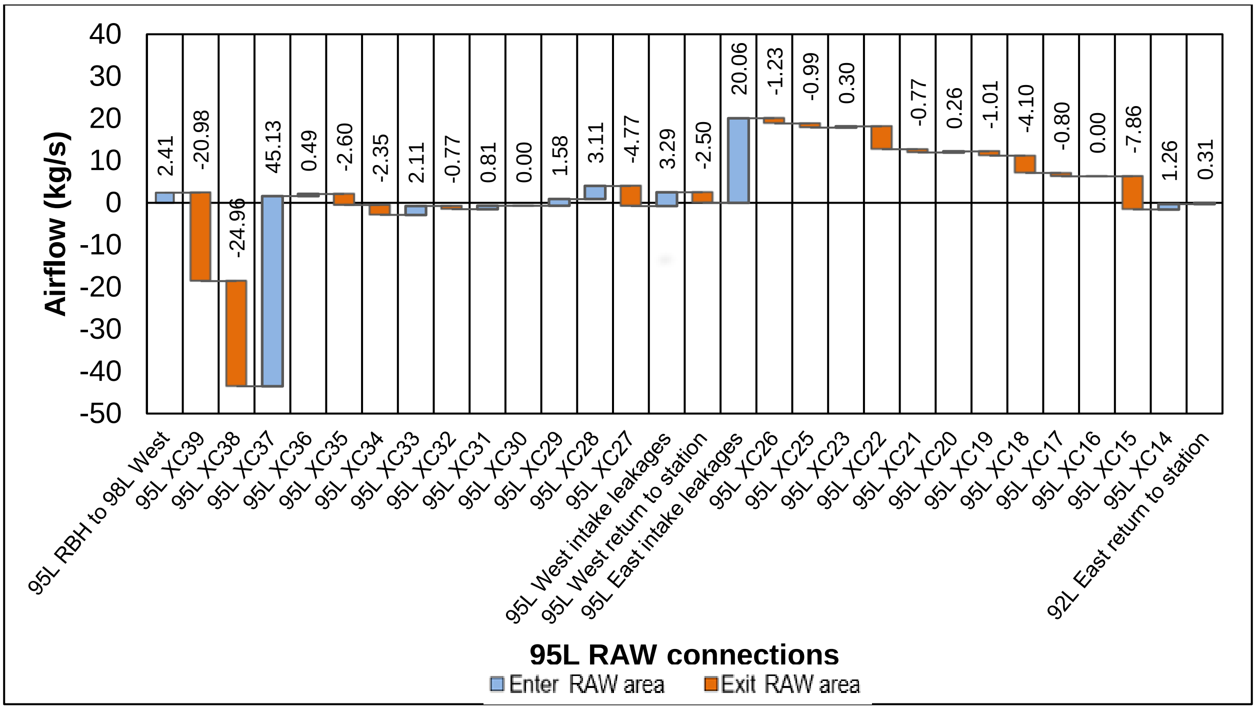

Figure 65: 95L calibrated level mass balance

Bulk air cooler calibration graphs

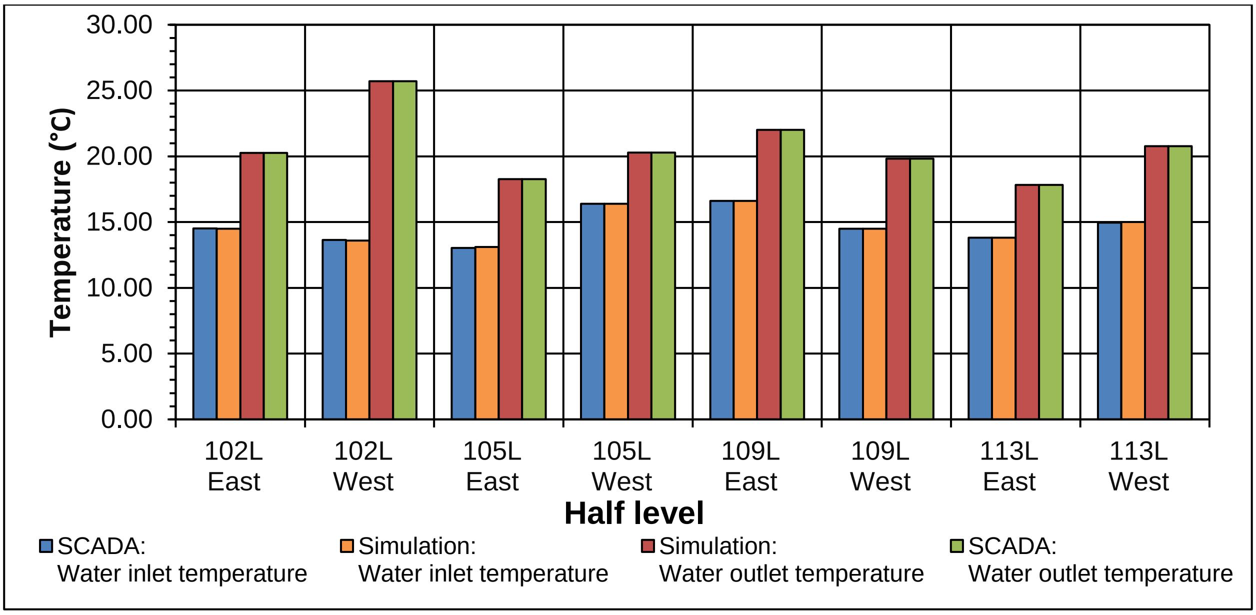

Figure 66: Production block BAC calibrated water boundary conditions

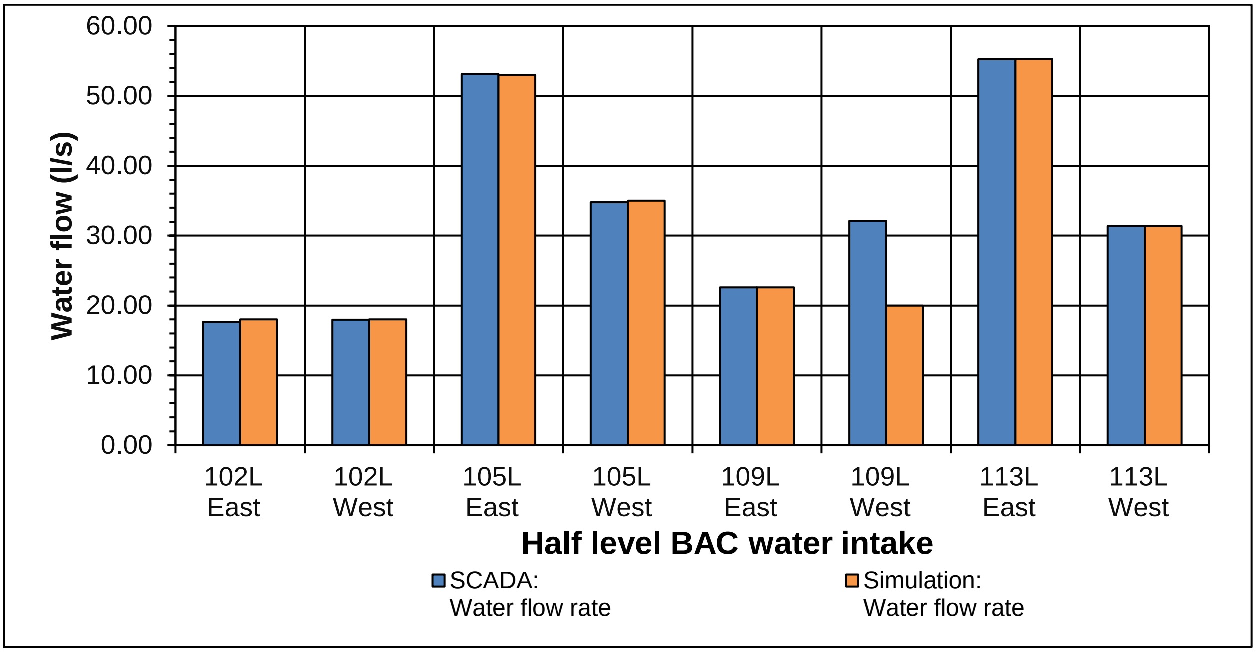

Figure 67: Production block BAC calibrated water boundary flow

APPENDIX F. Verification of Mine A simulation model

A specific area of Mine A was used during the verification phase. A simplified drawing of this area's configuration during the calibration phase is shown in Figure 68.

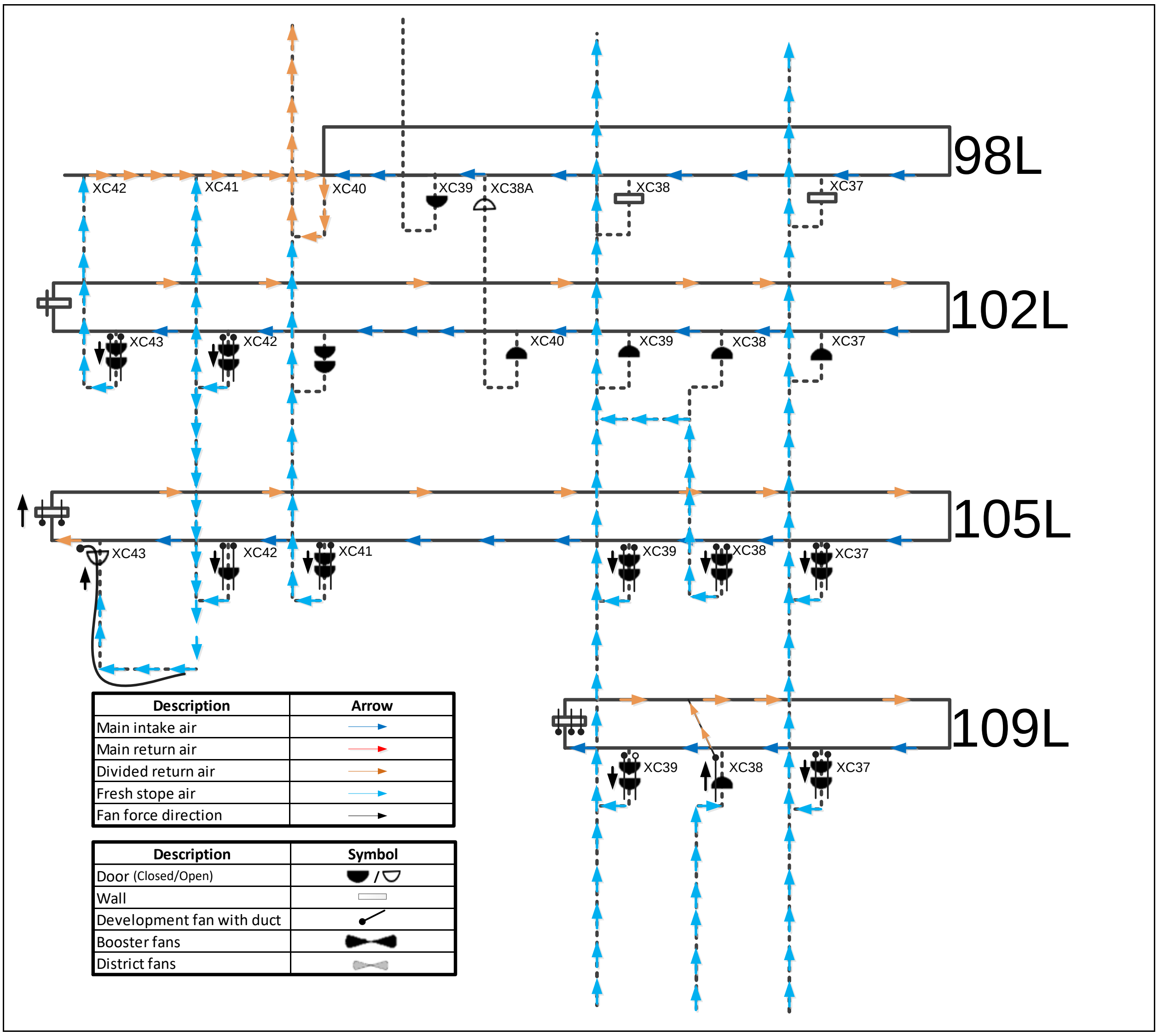

Figure 68: Mine A West (during calibration audit)

The following changes were applied to both the actual MVS and the simulation model.

Table 30: Verification implemented actions on Mine A

Item	Level	Actions implemented	Status
1	88L	Remove wall and regulator: XC32	Completed
2	98L	Seal RBH (downcast to 102L East IAW): XC26	Completed
3	98L	Remove wall and install a door with 2 x 760 mm columns	Completed
4	98L	Open XC door	Completed
5	102L	Door half open: XC40	Completed
6	102L	Stop one crosscut district fan: XC42	Completed
7	102L	Stop crosscut district fans: XC43	Completed
8	102L	Open crosscut doors: XC43	Completed
9	102L	Downcast air to 105L XC43: S42	Completed
10	105L	Connect S43 with S42	Completed
11	105L	Install a 45 kW and 15 kW return fans with doors	Completed
12	105L	Run 2 x 45 kW end fans: West End	Completed
13	109L	Run 1 x 45 kW crosscut fan: XC38	Completed
14	109L	Close doors: XC38	Completed
15	109L	Connect S38 to S37	Completed
16	109L	Run both 45 kW and 15 kW fans	Completed
17	109L	Run 2 x 45 kW end fans: XC39	Completed

A simplified drawing of the area’s configuration during the verification phase is shown in Figure 69.

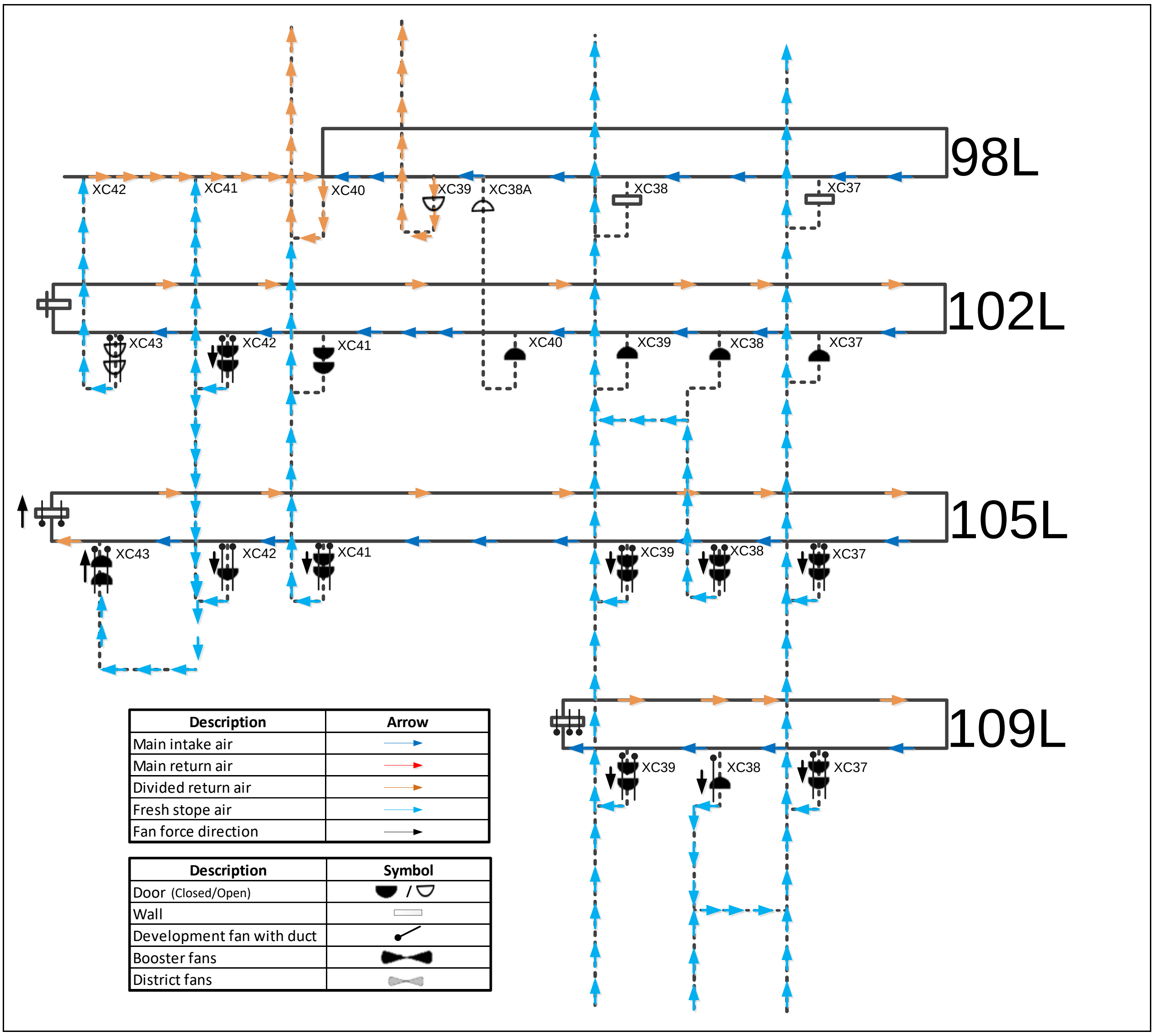

Figure 69: Mine A West (verification changes implemented)

APPENDIX G.Air distribution improvement of Mine A

Airflow requirements

Table 31: Overall airflow requirements of Mine A

Location			Working area status	Current implementation	Airflow requirement (kg/s)
Half level		Working area			
85L	East	14	Return	Sealed XC with an open crossover	N/A
85L	East	15	Return	Sealed XC with an open crossover	N/A
85L	East	16B	Return	Sealed XC with sealed crossover	N/A
85L	East	16A	Return	Sealed XC with an open crossover	N/A
85L	East	17	Return	Sealed XC	N/A
85L	East	18	Return	Sealed XC	N/A
85L	East	19	Return	Sealed XC with sealed crossover	N/A
85L	East	20	Return	Sealed XC with sealed crossover	N/A
85L	East	21	Return	Sealed XC with sealed crossover	N/A
85L	East	22	Return	Sealed XC	N/A
85L	East	24	Return	Sealed XC	N/A
85L	West	26	Return	Open XC	N/A
85L	West	27	Return	Sealed XC with open 760 mm column	N/A
85L	West	28	Return	Sealed XC with sealed 760 mm column	N/A
85L	West	29	Return	Sealed XC with open 760 mm column	N/A
85L	West	30	Return	Sealed XC with open 760 mm column	N/A
85L	West	31	Return	Sealed with small open door	N/A
85L	West	32	Return	Sealed XC with open 760 mm column	N/A
85L	West	33	Return	Sealed XC with open 760 mm column	N/A
85L	West	34	Return	Sealed XC with open 760 mm column	N/A
85L	West	35	Return	Sealed XC with open 760 mm column	N/A
85L	West	36	Return	Sealed XC with open 760 mm column	N/A
88L	East	14	Return	Sealed XC with open 760 mm column	N/A
88L	East	15	Return	Sealed XC with open 760 mm column	N/A
88L	East	16	Return	Sealed XC with sealed 760 mm column	N/A
88L	East	17	Return	Sealed XC with sealed 760 mm column	N/A
88L	East	18	Return	Sealed XC with sealed 760 mm column	N/A
88L	East	19	Return	Sealed XC with an open crossover	N/A
88L	East	20	Return	Sealed XC with an open crossover	N/A
88L	East	21	Return	Sealed XC with an open crossover	N/A
88L	East	22	Return	Sealed XC with an open crossover	N/A
88L	East	23	Return	Sealed XC with an open crossover	N/A
88L	East	24	Return	Sealed XC	N/A

Location			Working area status	Current implementation	Airflow requirement (kg/s)
Half level		Working area			
88L	East	24A	Return	Sealed XC	N/A
88L	West	26	Return	Sealed XC	N/A
88L	West	27	Return	Open XC	N/A
88L	West	28	Return	Open XC with open crossover	N/A
88L	West	29	Return	Open XC with open 760 mm column and sealed crossover	N/A
88L	West	30	Return	Sealed XC with open 760 mm column	N/A
88L	West	31	Return	Sealed XC with open 760 mm column and open crossover	N/A
88L	West	32	Return	Sealed XC with open 760 mm column	N/A
88L	West	33	Return	Sealed XC with open 760 mm column	N/A
88L	West	34	Return	Sealed XC with open 760 mm column and open crossover and RBH	N/A
88L	West	35	Return	Sealed XC with open 760 mm column	N/A
88L	West	36	Return	Open XC with large restriction	N/A
88L	West	38	Return	Sealed XC with open 760 mm column	N/A
92L	East	14	Return	Sealed XC with open 760 mm column	N/A
92L	East	15	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	16	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	17	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	18	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	19	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	20	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	21	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	22	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	23	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	24	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	25	Return	Sealed XC with sealed 760 mm column	N/A
92L	East	26	Return	Sealed XC with sealed 760 mm column	N/A
92L	West	27	Return	Open XC	N/A
92L	West	28	Return	Open XC	N/A
92L	West	29	Return	Sealed XC with sealed 760 mm column	N/A
92L	West	30	Return	Sealed XC with sealed 760 mm column	N/A
92L	West	31	Return	Sealed XC with sealed 760 mm column	N/A
92L	West	32	Return	Sealed XC with sealed 760 mm column	N/A
92L	West	33	Return	Sealed XC with sealed 2 x 760 mm columns	N/A
92L	West	34	Return	Open XC	N/A
92L	West	35	Return	Open XC	N/A

Location			Working area status	Current implementation	Airflow requirement (kg/s)
Half level		Working area			
92L	West	36	Return	Sealed XC with open 760 mm column	N/A
92L	West	37	Return	Sealed XC with open 760 mm column	N/A
92L	West	38	Return	Open XC	N/A
92L	West	39	Return	Open XC	N/A
95L	East	14	Return	Sealed XC	N/A
95L	East	15	Return	Sealed XC with open 760 mm column	N/A
95L	East	16	Return	Sealed XC with open 760 mm column	N/A
95L	East	17	Return	Sealed XC with sealed 3 x 760 mm columns	N/A
95L	East	18	Return	Open XC	N/A
95L	East	19	Return	Open XC	N/A
95L	East	20	Return	Sealed XC with sealed 760 mm column	N/A
95L	East	21	Return	Sealed XC with sealed 2 x 760 mm column	N/A
95L	East	22	Return	Sealed XC with open 760 mm column	N/A
95L	East	23	Return	Sealed XC with sealed 3 x 760 mm column	N/A
95L	East	25	Return	Sealed XC with sealed 760 mm column	N/A
95L	East	26	Return	Sealed XC	N/A
95L	West	27	Return	Sealed XC with open 760 mm column	N/A
95L	West	28	Return	Open XC with an open crossover	N/A
95L	West	29	Return	Open XC	N/A
95L	West	30	Return	Open XC	N/A
95L	West	31	Return	Sealed XC	N/A
95L	West	32	Return	Sealed XC	N/A
95L	West	33	Return	Sealed XC	N/A
95L	West	34	Return	Open XC	N/A
95L	West	35	Return	Open XC	N/A
95L	West	36	Return	Sealed XC with sealed 760 mm column	N/A
95L	West	37	Return	Open XC	N/A
95L	West	38	Return	Open XC	N/A
95L	West	39	Return	Open XC	N/A
98L	East	End	End flow	4 x 45 kW fans	10
98L	East	15	Return	Open XC (some restrictions)	20
98L	East	16	Travel way	Sealed XC	6
98L	East	17	Travel way	Doors with 2 x45 kW fans	N/A
98L	East	18	Travel way	Doors with 2 x45 kW fans	N/A
98L	East	19	No mining	Door	6
98L	East	20	No mining	Wall	N/A

Location			Working area status	Current implementation	Airflow requirement (kg/s)
Half level		Working area			
98L	East	21	No mining	Door to access bypass fans	0
98L	East	22	No mining	Door with 2 x 45 kW return fans	0
98L	East	23	No mining	Wall	0
98L	East	24	No mining	Wall	0
98L	West	26	No mining	RBH	0
98L	West	27	No mining	Wall	0
98L	West	28	Return	Wall	0
98L	West	29	No mining	Wall	0
98L	West	30	No mining	Wall	0
98L	West	31	No mining	Wall	0
98L	West	32	Return	Wall	N/A
98L	West	33	No mining	Wall	0
98L	West	34	No mining	Wall	0
98L	West	35	Return	Wall	N/A
98L	West	36	No mining	Wall	0
98L	West	37	No mining	Wall	0
98L	West	38	No mining	Wall	0
98L	West	38A	No mining	Doors (wrong way)	0
98L	West	39	Return	Door (chained to the wall)	N/A
98L	West	40	Return	Open	N/A
98L	West	41	Travel way	Open	N/A
98L	West	42	Travel way	Open	N/A
98L	West	End	End flow		10
102L	East	End	End flow		20
102L	East	16	Active	Open (AF system)	15
102L	East	17	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
102L	East	18	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
102L	East	19	Travel way	Doors with a 15 kW & 45 kW fan (airlock)	6
102L	East	20	Travel way	Doors with a 15 kW & 45 kW fan (airlock)	6
102L	East	21	Travel way	Open	6
102L	East	22	Travel way	Doors with a 15 kW & 45 kW fan (airlock)	6
102L	East	23	Travel way	Open	6
102L	East	24	No mining	Doors	0
102L	East	26	Travel way	Doors	6
102L	East	27	No mining	Wall	0

Location			Working area status	Current implementation	Airflow requirement (kg/s)
Half level		Working area			
102L	East	28	No mining	Wall	0
102L	East	29	No mining	Wall	0
102L	East	30	No mining	Wall	0
102L	East	31	No mining	Wall	0
102L	East	32	Return	Wall with 2 x 45 kW return fans	N/A
102L	East	33	No mining	Wall	0
102L	East	34	No mining	Wall	0
102L	West	35	No mining	Wall	0
102L	West	36	No mining	Wall	0
102L	West	37	Travel way	Doors	6
102L	West	38	No mining	Doors	0
102L	West	39	Travel way	Doors	6
102L	West	40	No mining	Doors	0
102L	West	41	Travel way	Doors	6
102L	West	42	Active	Doors with a 15 kW & 45 kW fan (airlock)	35
102L	West	43	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
102L	West	End	End flow	Wall with open 760 mm column	0
105L	East	End	End flow	2 x 45 kW return fans	20
105L	East	18	Developing	Open (AF system)	N/A
105L	East	19	Active	Open (AF system)	15
105L	East	20	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	East	21	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	East	22	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	East	23	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	East	24	Active	Doors with a 15 kW & 45 kW fan (airlock)	6
105L	East	26	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	East	27	No mining	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	East	28	Travel way	Doors with a 15 kW & 45 kW fan (airlock)	6
105L	East	29	Travel way	Doors	6
105L	East	30	Vamping or reclaiming	Doors	6
105L	East	31	No mining	Wall	0

Location			Working area status	Current implementation	Airflow requirement (kg/s)
Half level		Working area			
105L	West	32	No mining	Wall	0
105L	West	33	No mining	Wall	0
105L	West	34	No mining	Doors	0
105L	West	35	No mining	Doors	0
105L	West	36	Travel way	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	West	37	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	West	38	No mining	Doors with a 15 kW & 45 kW fan (airlock)	12
105L	West	39	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	West	41	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	West	42	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
105L	West	43	Return	Open (auxiliary fan system)	N/A
105L	West	End	End flow	3 x 45 kW return fans (2 active)	15
109L	East	End	End flow	3 x 45 kW return fans	30
109L	East	24	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
109L	East	26	Developing	Open (auxiliary fan system)	N/A
109L	East	27	Developing	Open (auxiliary fan system)	N/A
109L	East	28	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
109L	East	29	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
109L	East	30	Travel way	Doors with a 15 kW & 45 kW fan (airlock)	12
109L	East	31	Travel way	Door	6
109L	East	32	Travel way	Door	6
109L	West	33	Travel way	Door	6
109L	West	34	Vamping or reclaiming	Door	15
109L	West	35	Travel way	Door	6
109L	West	36	Active	Door with 45 kW fan	15
109L	West	37	Active	Doors with a 15 kW & 45 kW fan (airlock)	6
109L	West	38	Active	Doors with a 15 kW & 45 kW fan (airlock)	6
109L	West	39	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
109L	West	40	Developing	Open (auxiliary fan system)	15

Location			Working area status	Current implementation	Airflow requirement (kg/s)
Half level		Working area			
109L	West	41	No mining	Open	0
109L	West	End	End flow	3 x 45 kW return fans	15
113L	East	End	End flow	2 x 45 kW return fans	20
113L	East	26	Developing	Open (auxiliary fan system)	N/A
113L	East	27 S	Developing	Open (auxiliary fan system)	N/A
113L	East	28 S	Developing	Open (auxiliary fan system)	N/A
113L	East	29 S	No mining	Wall	8
113L	East	29 N	No mining	Wall	0
113L	East	30 S	Developing	Open	10
113L	East	30 N	Travel way	Door 15 kW fan (inside 1 x 45 kW return fan)	12
113L	East	31 S	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
113L	East	31 N	Travel way	Door	6
113L	East	32 S	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
113L	East	32 N	Travel way	Doors with a 15 kW & 45 kW fan (airlock)	6
113L	West	33	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
113L	West	34	Vamping or reclaiming	Doors with a 15 kW & 45 kW fan (airlock)	12
113L	West	35	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
113L	West	36	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
113L	West	37	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
113L	West	38	Active	Doors with a 15 kW & 45 kW fan (airlock)	15
113L	West	End	End flow	3 x 45 kW return fans	10

Air distribution improvement of shafts

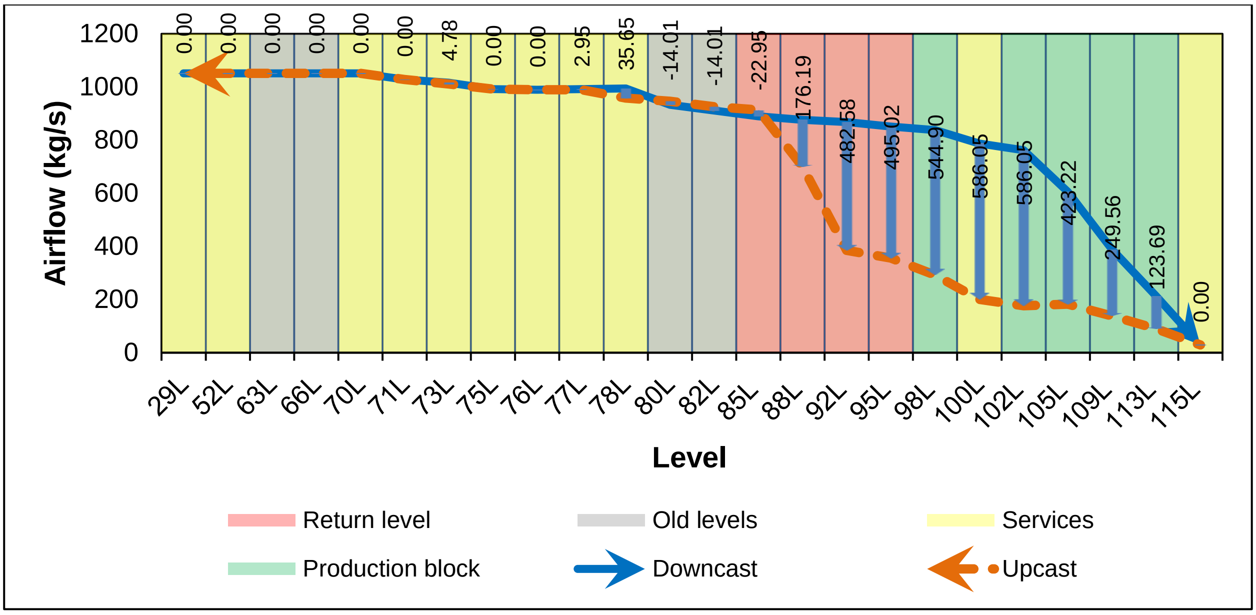

Figure 70: Overall upcast and downcast air distribution after improvement

Table 32: Mine A downcast airflow (Mine B main fan removed and 92L booster fan installed)

	Simulated shaft airflow (kg/s)		Change in airflow (kg/s)
Level	**Baseline**	**Prediction**	
29L	1060.47	837.65	-222.82
52L	1060.47	837.65	-222.82
63L	1060.47	837.65	-222.82
66L	1060.47	837.65	-222.82
70L	1060.47	837.65	-222.82
71L	1037.17	816.03	-221.14
73L	1026.05	812.14	-213.91
75L	1003.62	800.50	-203.12
76L	1000.83	798.90	-201.94
77L	1003.62	800.50	-203.12
78L	956.85	819.39	-137.46
80L	922.57	764.17	-158.40
82L	903.22	769.59	-133.63
85L	881.59	769.03	-112.55
88L	869.04	764.98	-104.06
92L	861.18	760.51	-100.67
95L	843.50	752.97	-90.53
98L	813.02	741.91	-71.11

	Simulated shaft airflow (kg/s)		Change in airflow (kg/s)
Level	**Baseline**	**Prediction**	
100L	709.64	710.55	0.91
102L	689.27	690.46	1.20
105L	559.21	549.34	-9.87
109L	348.18	354.32	6.13
113L	199.72	192.43	-7.29
115L	28.88	27.78	-1.10

Over- and underventilated working areas in Mine A's SVS improvements example for 102L

Table 33: 102L SVS (crosscut) airflow comparisons and improvement

Half level	Crosscut	Crosscut status	Simulation: Baseline (kg/s)	Comparison (Baseline & requirements)	Solution implemented	Simulation: Predicted (kg/s)	Airflow change
East	XC16	Active	N/A	Underventilated: -15 kg/s under	Install doors with a 15 kW & 45 kW fan (Mine procedure)	15	Improve: 15 kg/s to requirements
East	XC17	Active	17.32	Overventilated: 2.32 kg/s over	No action (acceptable difference)	15.54	Improve: 1.78 kg/s to requirements
East	XC18	Active	12.56	Underventilated: -2.44 kg/s under	No action (acceptable difference)	11.23	Deviate: 1.33 kg/s from requirements
East	XC19	Travel way	7.03	Overventilated: 1.03 kg/s over	Seal the idle fan (column)	5.21	Improve: 0.24 kg/s to requirements
East	XC20	Travel way	20.97	Overventilated: 14.97 kg/s over	Remove fans and seal columns (hang brattices) door wrong way	4.88	Improve: 13.86 kg/s to requirements
East	XC21	Travel way	-20.85	Underventilated: -26.85 kg/s under	Seal open column and hang brattices	7.24	Improve: 25.61 kg/s to requirements
East	XC22	Travel way	15.05	Overventilated: 9.05 kg/s over	Brattices	3.23	Improve: 6.28 kg/s to requirements
East	XC24	No mining	-0.18	Underventilated: -0.18 kg/s under	Seal XC	-0.8	Deviate: 0.62 kg/s

Half level	Crosscut	Crosscut status	Simulation: Baseline (kg/s)	Comparison (Baseline & requirements)	Solution implemented	Simulation: Predicted (kg/s)	Airflow change
							from requirements
East	XC26	Travel way	21.21	Overventilated: 15.21 kg/s over	Install double layer ventilation brattices	2.23	Improve: 11.45 kg/s to requirements
East	XC27	No mining	1.11	Overventilated: 1.11 kg/s over	No action	0.43	Improve: 0.69 kg/s to requirements
East	XC28	No mining	-0.54	Underventilated: -0.54 kg/s under	No action	-1.65	Deviate: 1.1 kg/s from requirements
East	XC29	No mining	0.19	Overventilated: 0.19 kg/s over	No action	0.37	Deviate: 0.18 kg/s from requirements
East	XC30	No mining	0.59	Overventilated: 0.59 kg/s over	No action	0.98	Deviate: 0.39 kg/s from requirements
East	XC31	No mining	-0.71	Underventilated: -0.71 kg/s under	No action	-1.03	Deviate: 0.32 kg/s from requirements
East	XC33	No mining	-0.81	Underventilated: -0.81 kg/s under	No action	0.26	Improve: 0.54 kg/s to requirements
East	XC34	No mining	0.44	Overventilated: 0.44 kg/s over	No action	0.47	Deviate: 0.03 kg/s from requirements
West	XC35	No mining	1.08	Overventilated: 1.08 kg/s over	Seal open column and chain door	0.58	Improve: 0.5 kg/s to requirements
West	XC36	No mining	0.27	Overventilated: 0.27 kg/s over	No action	-0.31	Deviate: 0.03 kg/s from requirements
West	XC37	Travel way	0	Underventilated: -6 kg/s under	No action	0	Deviate: 0 kg/s from requirements
West	XC38	No mining	2.49	Overventilated: 2.49 kg/s over	Seal XC (open old fan holes)	1.44	Improve: 1.05 kg/s to requirements
West	XC39	Travel way	0.44	Underventilated: -5.56 kg/s under	Change direction of	6.35	Improve: 5.2 kg/s to requirements

Half level	Crosscut	Crosscut status	Simulation: Baseline (kg/s)	Comparison (Baseline & requirements)	Solution implemented	Simulation: Predicted (kg/s)	Airflow change
					door and run fan		
West	XC40	No mining	1.01	Overventilated: 1.01 kg/s over	Seal XC	0.42	Improve: 0.59 kg/s to requirements
West	XC41	Travel way	2.85	Underventilated: -3.15 kg/s under	Seal open columns and hang brattices	4.33	Improve: 1.48 kg/s to requirements
West	XC42	Active	15.62	Underventilated: -14.38 kg/s under	Switch off all fans & remove doors	26.26	Improve: 10.63 kg/s to requirements
West	XC43	Active	10.46	Underventilated: -4.54 kg/s under	Switch off fans & remove doors	9.67	Deviate: 0.79 kg/s from requirements

Figure 71 is a graphical representation of the overall improvement of the SVS on 102L. The graph shows the difference between the simulated airflow and the required airflow at a specific crosscut (the desired value is zero).

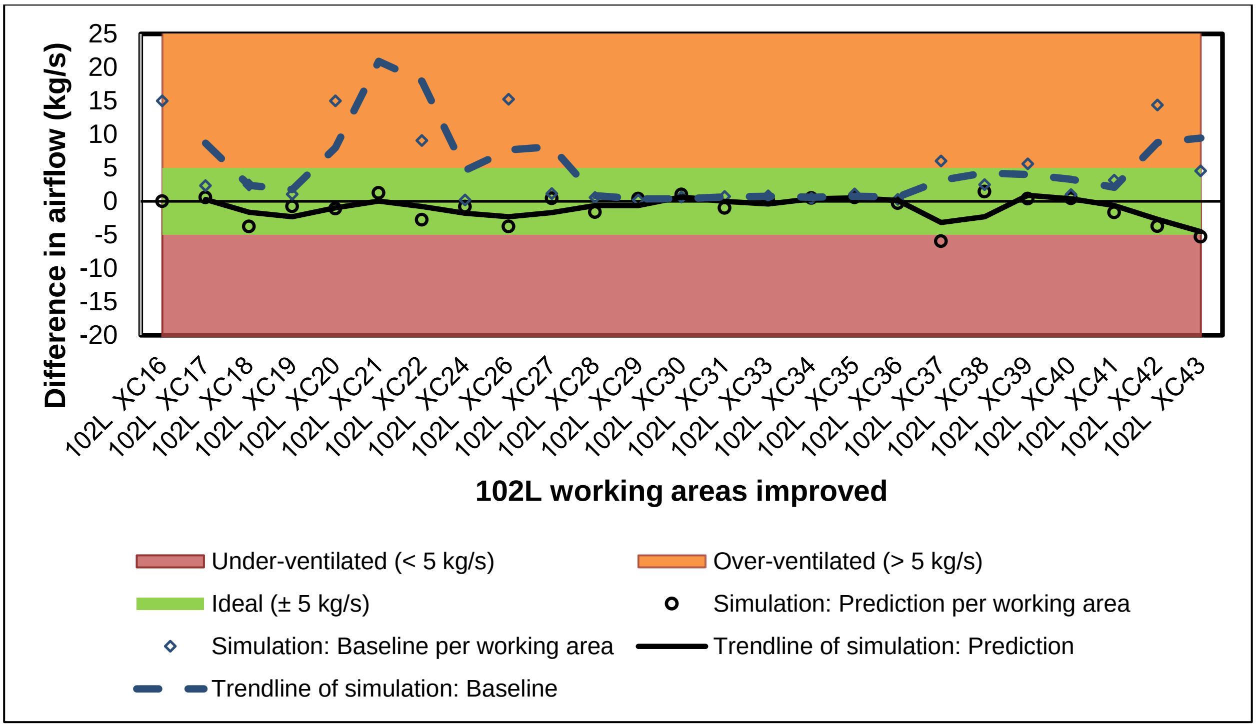

Figure 71: 102L SVS difference between simulation and required airflow (0 kg/s was considered ideal)

Overventilated areas in Mine A's PVS improvements

Table 34: PVS overventilated comparisons and improvement

Location		Baseline: Simulation		Comparison (Baseline & requirements)	Simulation: Prediction		Airflow change (kg/s)
		Implementation	Airflow (kg/s)		Implementation	Airflow (kg/s)	
98L	East	4 x 45 kW return fans	45.7	Overventilated: 35.7 kg/s over	Install ventilation brattices and doors	16.17	-29.53
	West	Open	9.8	Underventilated: -0.25 kg/s under	Install ventilation brattices and doors	6.70	-3.05
102L	East	3 x 45 kW return fans	30.7	Overventilated: 10.69 kg/s over	Run 2 x 45 kW return fans (seal idling fans)	21.08	-9.61
	West	No fans	-0.1	Underventilated: -0.12 kg/s under	No action	-0.12	0
105L	East	2 x 45 kW return fans	17.6	Underventilated: -2.38 kg/s under	No action	16.36	-1.26
	West	3 x 45 kW return fans	20.1	Overventilated: 0.14 kg/s over	Run 2 x 45kW return fans (seal idling fan)	20.14	0
109L	East	2 x 45 kW return fans	23.1	Overventilated: 3.1 kg/s over	No action	21.08	-2.01
	West	2 x 45 kW return fans	15.8	Underventilated: -9.24 kg/s under	Run 1 x 45 kW return fan (Seal idling fans)	21.08	5.32
113L	East	3 x 45 kW return fans	41.4	Overventilated: 21.38 kg/s over	Run 2 x 45 kW return fans (seal idling fans)	25.01	-16.37
	West	2 x 45 kW return fans	24.4	Overventilated: 14.44 kg/s over	Run 1 x 45 kW return fan (seal idling fan)	7.37	-17.07

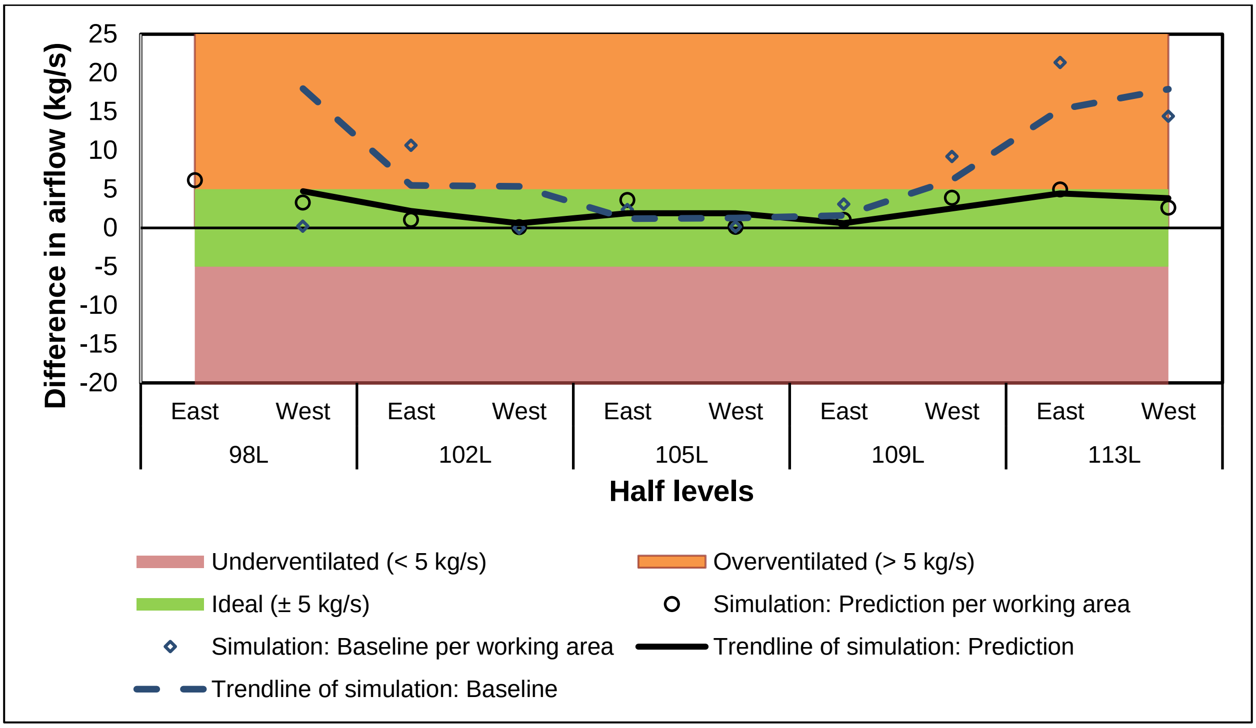

Figure 72: End flows difference between simulation and required airflow (0 kg/s was considered ideal)

Return manifold investigation additional data

Table 35: Production block predictions after PVS return system improvement

Location		Working area	Simulation: Baseline	Simulation: Prediction	Airflow change (kg/s)
98L	East	End	16.11	10.43	-5.68
	West	End	6.70	12.45	5.75
102L	East	End	21.10	20.20	-0.90
		XC16	4.96	18.61	13.65
		XC17	20.36	16.24	-4.12
		XC18	14.89	11.69	-3.21
	West	XC42	25.78	31.77	5.99
		XC43	8.85	13.88	5.04
		End	-0.11	-0.16	-0.05
105L	East	End	16.46	16.37	-0.09
		XC19	18.93	18.59	-0.35
		XC20	19.95	20.99	1.04
		XC21	10.42	10.59	0.17
		XC22	17.71	17.84	0.13
		XC23	10.47	10.46	-0.01
		XC26	17.69	17.49	-0.20
	West	XC37	12.93	13.22	0.29
		XC39	6.90	7.15	0.25
		XC41	16.66	16.65	-0.02
		XC42	16.69	20.09	3.41
		End	20.14	20.14	0.00
109L	East	End	21.21	21.35	0.14
		XC24	16.56	16.64	0.08
		XC28	11.11	11.27	0.16
		XC29	18.08	18.25	0.17
	West	XC36	6.05	6.27	0.22
		XC37	17.49	18.02	0.53
		XC38	12.04	12.67	0.62
		XC39	11.51	12.58	1.07
		End	11.85	11.11	-0.74
113L	East	End	24.66	25.21	0.55
		XC31 S	13.18	13.34	0.16
		XC32 S	17.48	17.46	-0.02
	West	XC33	17.11	18.00	0.89
		XC35	19.85	19.84	0.00
		XC36	12.80	13.23	0.42
		XC37	12.52	13.10	0.58
		XC38	9.51	9.98	0.47
		End	7.16	7.12	-0.04
				Average XC	0.95
				Average end flow	-0.15

Overall working areas

Table 36: Overall working area improvement

Location	Simulation		Required airflow (kg/s)	Simulation: Baseline		Simulation: Predicted	
	Baseline	Prediction		Absolute offset (%)	Percentage offset (%)	Absolute offset (%)	Percentage offset (%)
98L E End	38.09	8.18	10	28.09	281	1.82	18
98L W End	6.7	10.7	10	3.3	33	0.7	7
102L XC16	0	17.66	15	15	100	2.66	18
102L XC17	17.32	16.38	15	2.32	15	1.38	9
102L XC18	12.56	11.8	15	2.44	16	3.2	21
102L XC19	7.03	4.82	6	1.03	17	1.18	20
102L XC20	20.97	4.35	6	14.97	250	1.65	28
102L XC21	-20.85	6.76	6	26.85	447	0.76	13
102L XC22	15.05	2.58	6	9.05	151	3.42	57
102L XC39	0.44	6.38	6	5.56	93	0.38	6
102L XC41	2.85	2.24	6	3.15	52	3.76	63
102L XC42	15.62	28.11	30	14.38	48	1.89	6
102L XC43	10.46	11	15	4.54	30	4	27
102L E End	33.67	20.41	20	13.67	68	0.41	2
105L XC19	15.97	17.16	15	0.97	6	2.16	14
105L XC20	-0.97	18.37	15	15.97	106	3.37	22
105L XC21	27.36	10.24	15	12.36	82	4.76	32
105L XC22	20.65	17.15	15	5.65	38	2.15	14
105L XC23	7.28	10.18	15	7.72	51	4.82	32
105L XC24	12.92	4.77	6	6.92	115	1.23	21
105L XC26	3.03	16.02	15	11.97	80	1.02	7
105L XC28	-6.48	5.06	6	12.48	208	0.94	16
105L XC29	8.62	8.07	6	2.62	44	2.07	35
105L XC30	1.57	6.48	6	4.43	74	0.48	8
105L XC36	-2.41	15.18	10	12.41	124	5.18	52
105L XC37	13.62	12.29	15	1.38	9	2.71	18
105L XC39	5.28	6.5	10	4.72	47	3.5	35
105L XC41	14.97	12.08	15	0.03	0	2.92	19
105L XC42	7.83	18.57	15	7.17	48	3.57	24
105L E End	18.38	17.71	20	1.62	8	2.29	11
105L W End	20.14	20.14	20	0.14	1	0.14	1
109L XC24	2.44	16.13	15	12.56	84	1.13	8
109L XC28	14.9	10.86	15	0.1	1	4.14	28
109L XC29	23.53	17.6	15	8.53	57	2.6	17
109L XC30	23.53	7.6	6	17.53	292	1.6	27

Location	Simulation		Required airflow (kg/s)	Simulation: Baseline		Simulation: Predicted	
	Baseline	Prediction		Absolute offset (%)	Percentage offset (%)	Absolute offset (%)	Percentage offset (%)
109L XC31	3.23	2.09	6	2.77	46	3.91	65
109L XC34	1.01	7.69	12	10.99	92	4.31	36
109L XC35	5.92	2.24	6	0.08	1	3.76	63
109L XC36	4.9	7.13	15	10.1	67	7.87	52
109L XC37	24.55	17.5	15	9.55	64	2.5	17
109L XC38	-10.72	12.04	15	25.72	171	2.96	20
109L XC39	13.81	11.44	15	1.19	8	3.56	24
109L E End	25.14	22.63	20	5.14	26	2.63	13
109L W End	17.29	11.29	10	7.29	73	1.29	13
113L XC31 S	8.15	12.99	15	6.85	46	2.01	13
113L XC32 S	17.46	17.71	15	2.46	16	2.71	18
113L XC33	20.94	16.44	15	5.94	40	1.44	10
113L XC34	20.63	11.78	12	8.63	72	0.22	2
113L XC35	19.84	19.8	15	4.84	32	4.8	32
113L XC36	14.55	14.01	15	0.45	3	0.99	7
113L XC37	13.13	12.25	15	1.87	12	2.75	18
113L XC38	10.72	9.27	15	4.28	29	5.73	38
113L E End	20.17	21.08	20	0.17	1	1.08	5
113L W End	21.9	4.75	10	11.9	119	5.25	52
Average				**7.73**	**78**	**2.65**	**7.73**

85L air distribution improvement

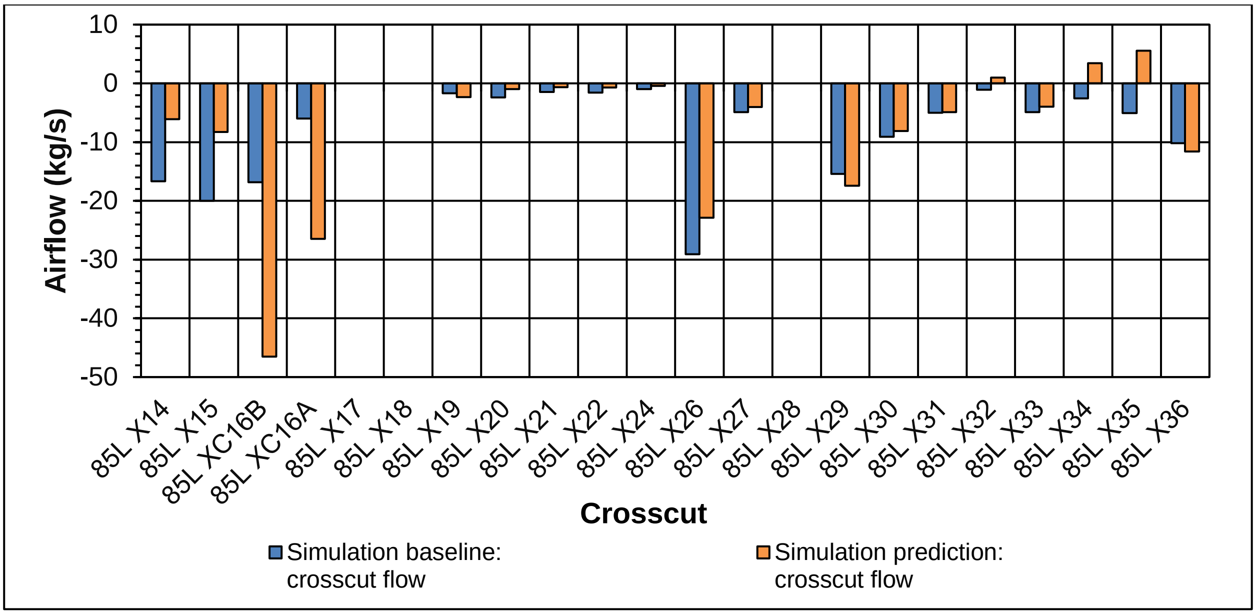

Figure 73: 85L improved return crosscut (XC) airflow

88L air distribution improvement

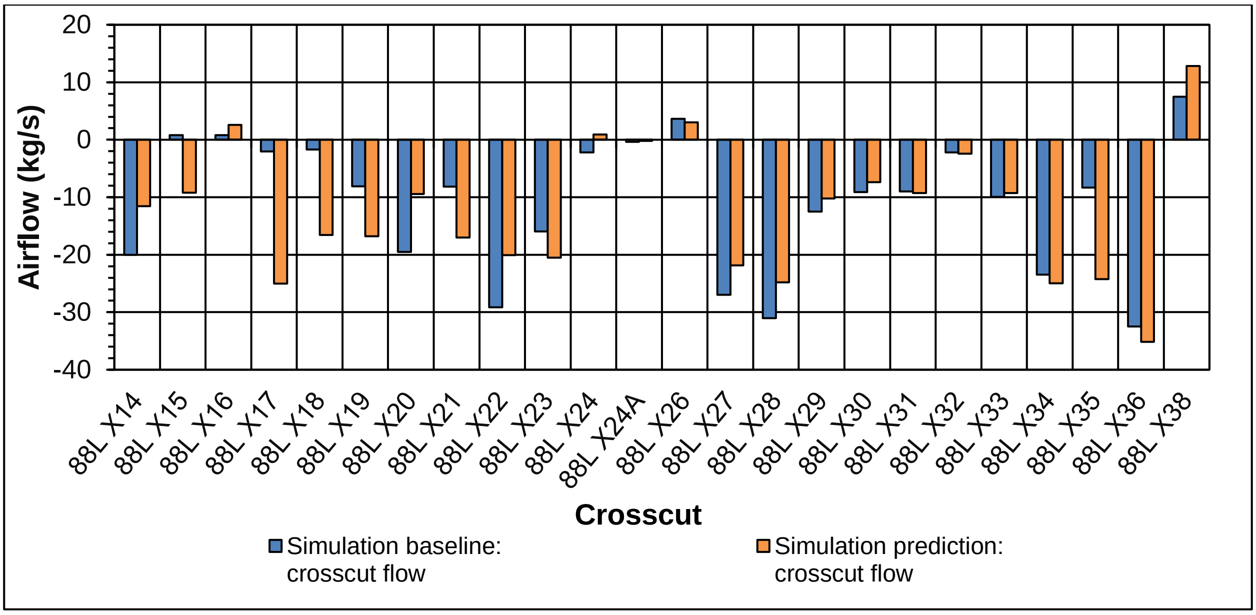

Figure 74: 88L improved return crosscut (XC) airflow

92L air distribution improvement

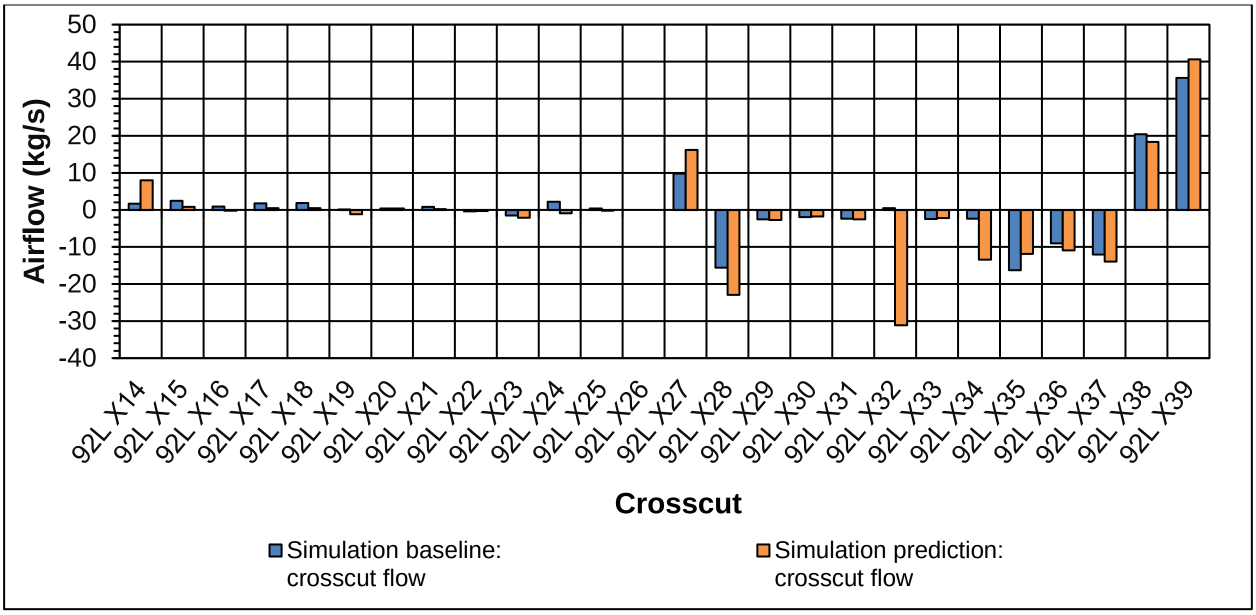

Figure 75: 92L improved return crosscut (XC) airflow

95L air distribution improvement

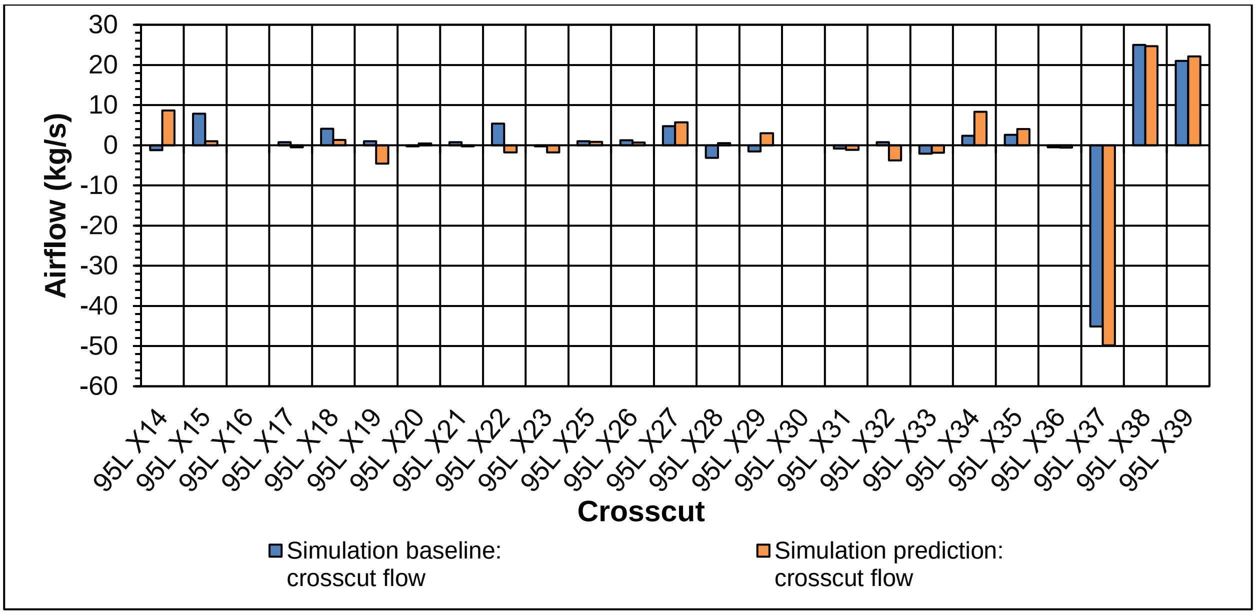

Figure 76: 95L improved return crosscut (XC) airflow

98L air distribution improvement

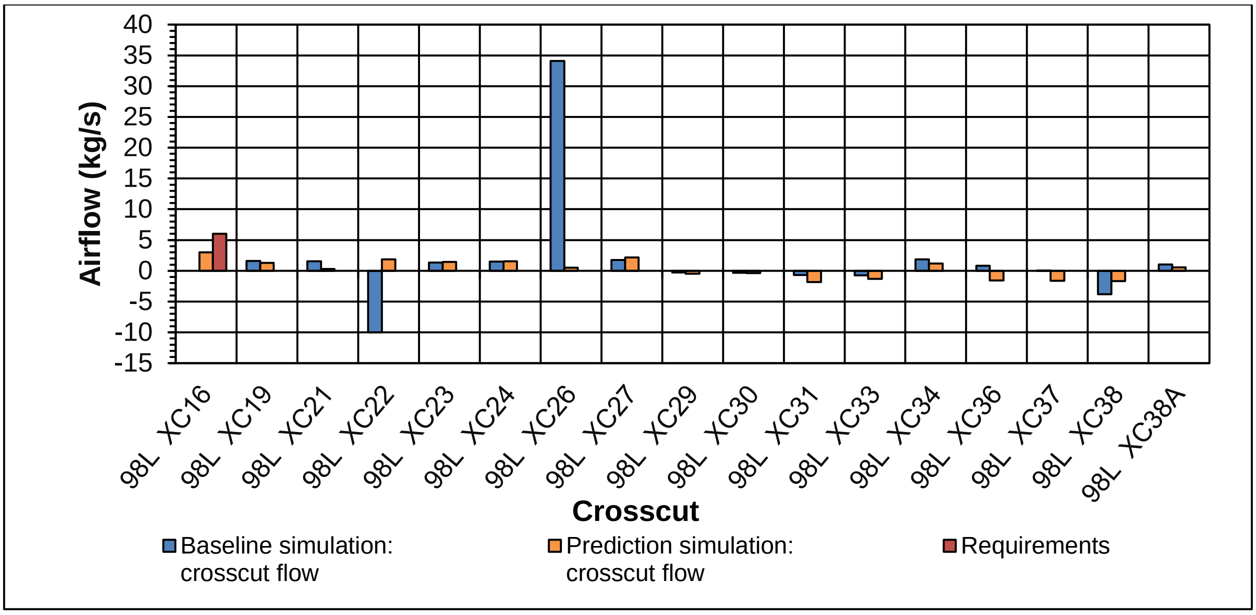

Figure 77: 98L improved crosscut (XC) airflow

102L air distribution improvement

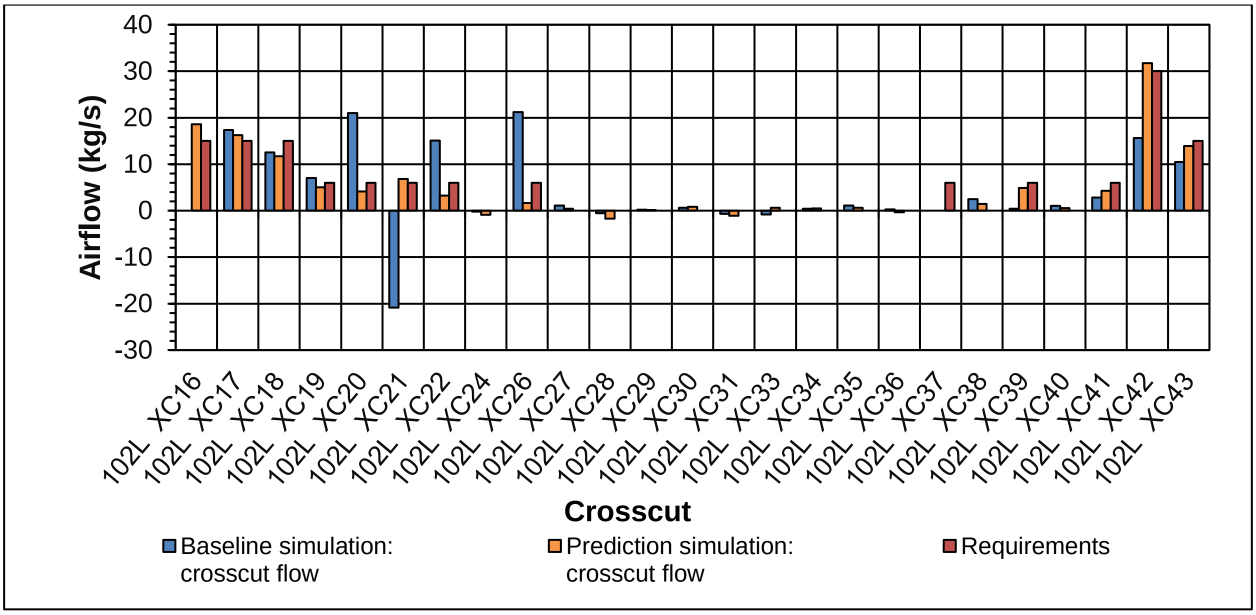

Figure 78: 102L improved crosscut (XC) airflow

105L air distribution improvement

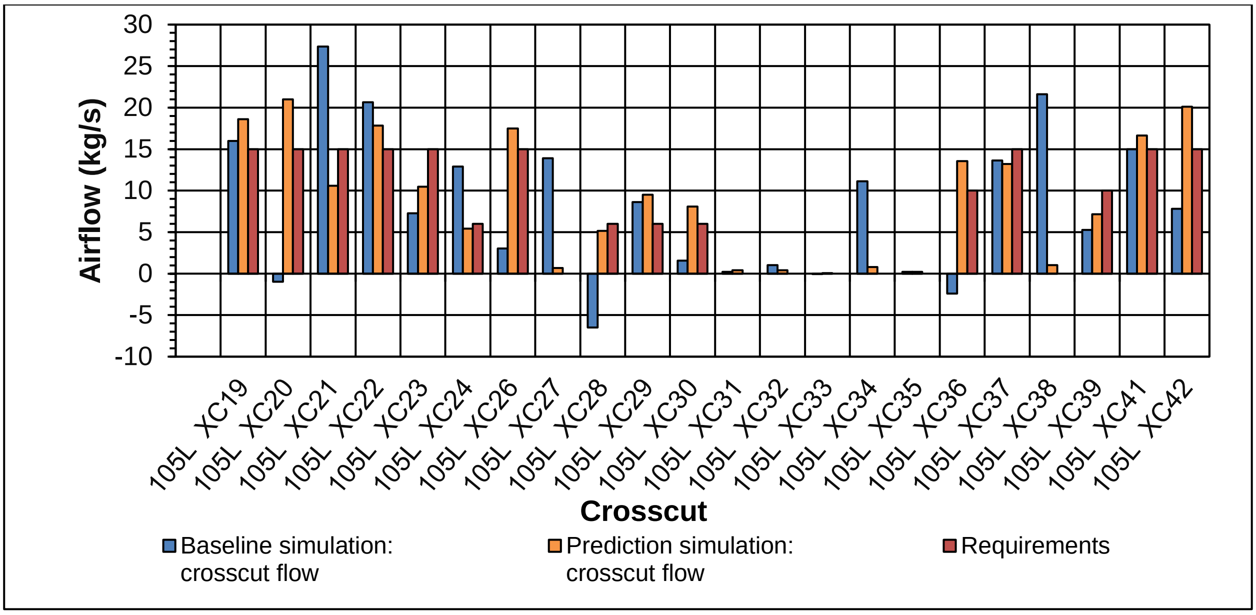

Figure 79: 105L improved crosscut (XC) airflow

109L air distribution improvement

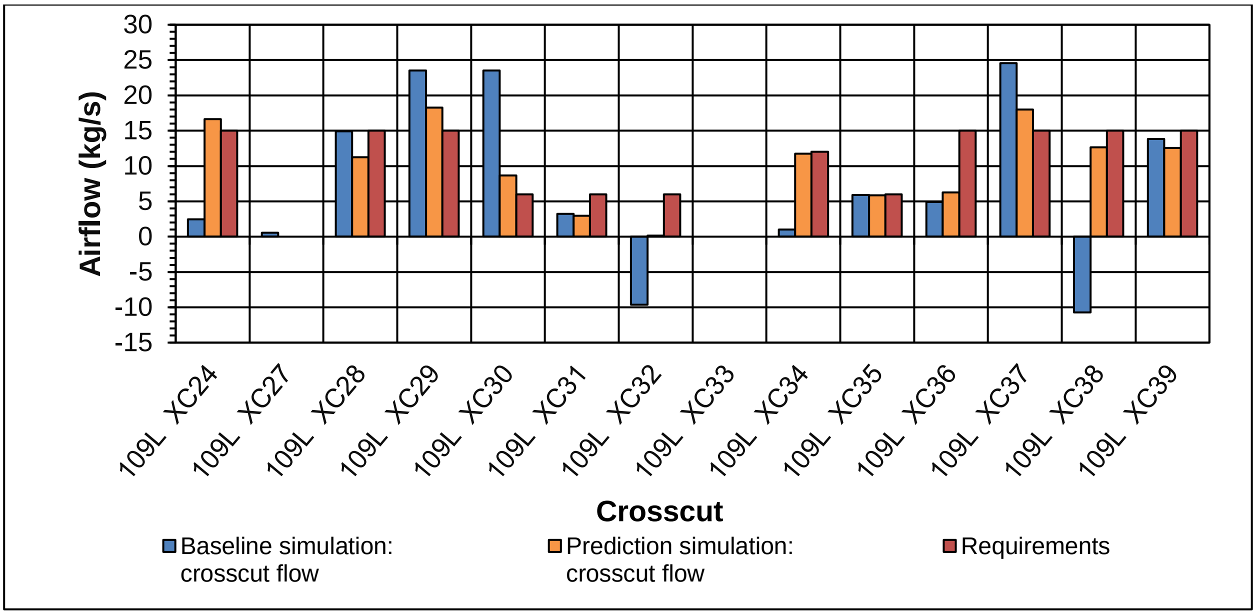

Figure 80: 109L improved crosscut (XC) airflow

113L air distribution improvement

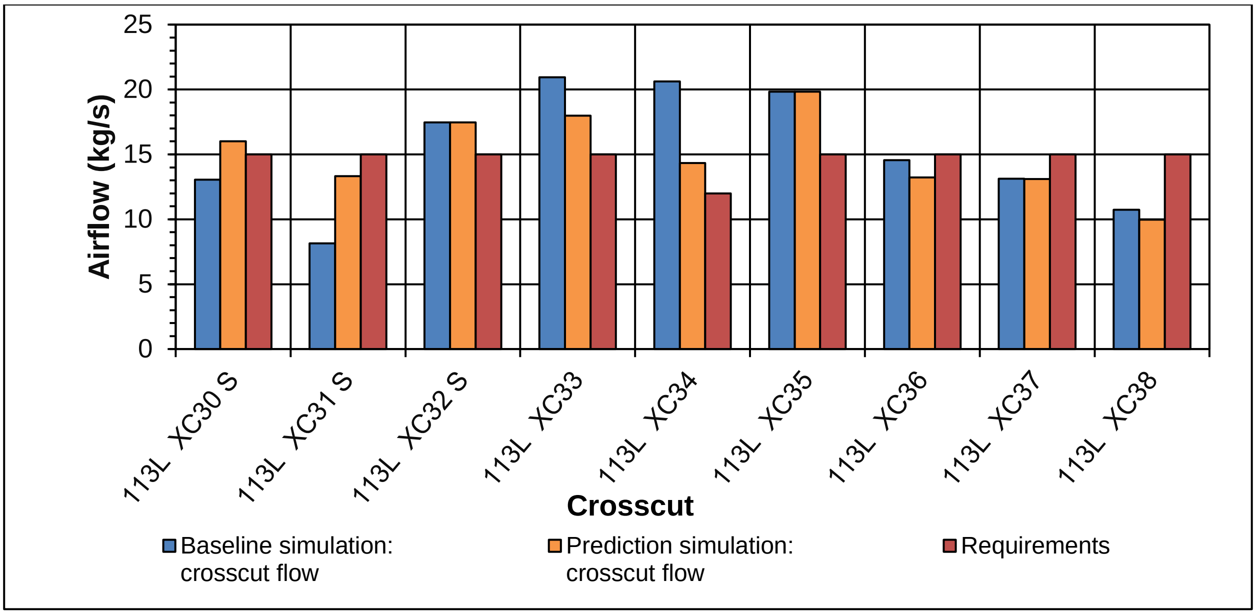

Figure 81: 113L improved crosscut (XC) airflow

End flow air distribution improvement

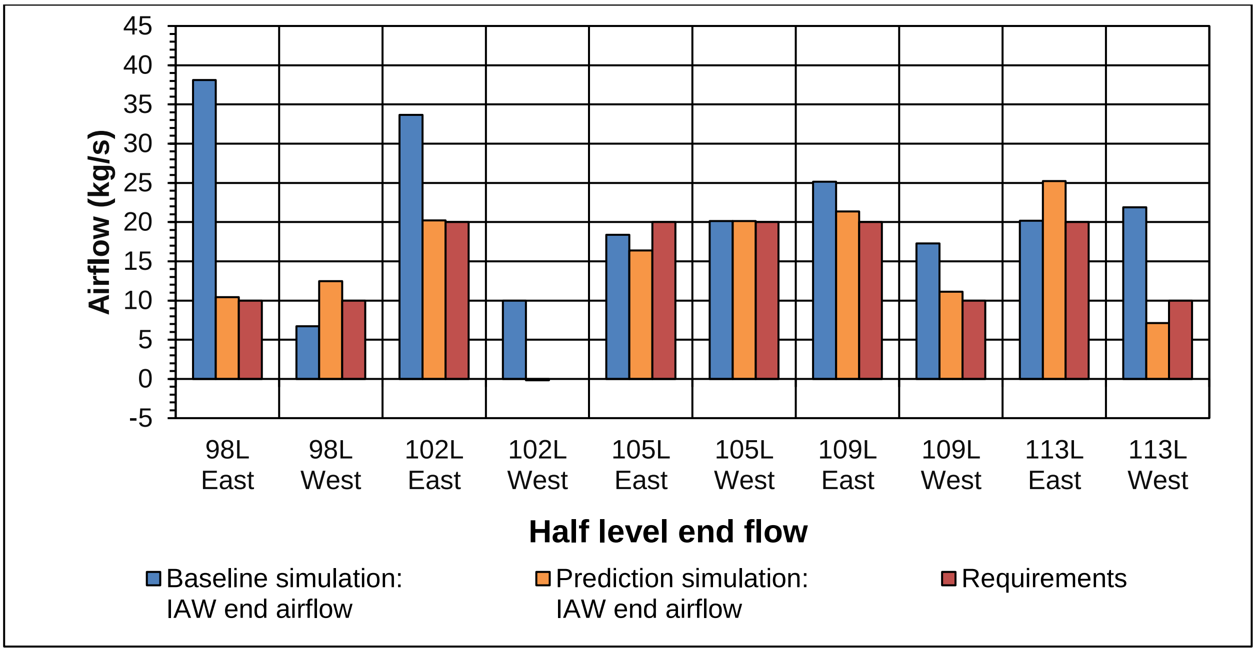

Figure 82: Production block improved end flows

APPENDIX H. Mine A air distribution improvement action plan and layouts

Project 1: Return manifold

This project was focused on actions to reduce the overall return system resistance of the mine. It includes end flow control and regulation removal in the return network (see Figure 83). The readable action plan can be found in Table 37.

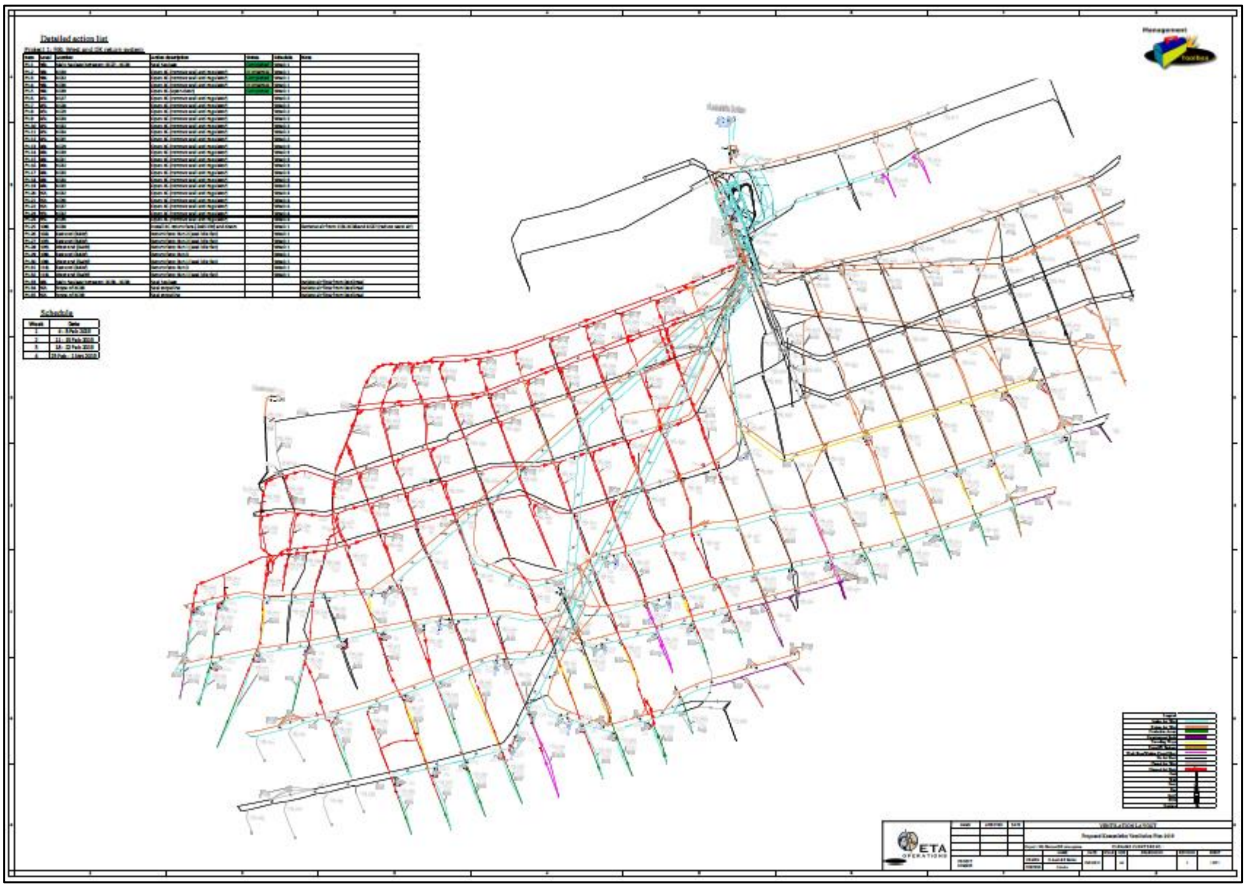

Figure 83: Project 1's schematic layout representation

Table 37: Project 1 action plan

Item	Level	Location	Action description
P1-1	98L	Main haulage between: XC27 - XC28	Close doors (chain)
P1-2	98L	XC32	Open XC (remove wall and regulator)
P1-3	98L	XC39	Open XC (open door)
P1-4	85L	XC36	Open XC (remove wall and 760 mm column)
P1-5	85L	XC35	Open XC (remove wall and 760 mm column)
P1-6	85L	XC34	Open XC (remove wall and 760 mm column)
P1-7	85L	XC31	Open XC (remove 2 x walls)
P1-8	85L	XC32	Open XC (remove wall and 760 mm column)
P1-9	85L	XC30	Open XC (remove wall and 760 mm column)
P1-10	85L	XC29	Open XC (remove wall and 760 mm column)
P1-11	85L	XC28	Open XC (remove wall and 760 mm column)
P1-12	85L	XC27	Open XC (remove wall and 760 mm column)
P1-13	85L	XC26	Seal XC with open 760 mm column
P1-14	85L	XC29-30 connecting	Completely remove wall
P1-15	85L	XC28-29	Completely remove wall
P1-16	88L	Main RAW haulage	Remove regulator (frame)
P1-17	88L	XC38	Open XC (remove wall and regulator)
P1-18	88L	XC36	Remove restriction
P1-19	88L	XC35	Open XC (remove wall and regulator)
P1-20	88L	XC34	`
P1-21	88L	XC33	Open XC (remove wall and regulator)
P1-22	88L	XC32	Open XC (remove wall and regulator)
P1-23	88L	XC30-31 connecting	Completely remove wall
P1-24	88L	XC34-35 East connecting	Completely remove wall
P1-25	88L	XC34-35 West connecting	Completely remove wall
P1-26	92L	XC37	Open XC (remove wall and regulator)
P1-27	92L	XC36	Open XC (remove wall and regulator)
P1-28	92L	XC32	Open XC (remove wall and regulator)
P1-29	92L	Main haulage between: XC29 - XC30	Close 2 x doors
P1-30	92L	Main haulage between: Before XC27	Close 2 x doors
P1-31	95L	XC32	Open XC (remove wall and regulator)

Item	Level	Location	Action description
P6-1	88L	XC24A	Open XC (remove wall and 2 x 760 mm column)
P6-2	88L	XC22, XC23, XC24 and XC24A 1st connecting	Completely remove wall (old refuge bay)
P6-3	88L	Station RBH to 92L	Open RBH (remove wall with 2 x 760 mm)
P6-4	92L	XC33-34 connecting	Open connecting (remove 3 x walls)
P6-5	92L	XC34-35 East connecting	Open connecting (remove wall and 760 mm column)
P6-6	92L	XC36-37 connecting	Open connecting (remove old refuge bay)
P6-7	92L	XC26-27 connecting	Open connecting (remove wall)
P6-8	92L	XC37-38 connecting	Open connecting (remove wall)
P6-9	92L	West RAW haulage (just after split)	Remove regulator (frame)
P6-10	92L	West RAW haulage (just after split)	Open RAW haulage (remove wall with 2 x 760 mm columns)
P6-11	92L	2nd connecting in West haulage	Seal 760 mm column
P6-12	92L	Station RAW (before vent shaft)	Install 2 x high pressure sliding doors
P6-13	92L	East RAW haulage (just after cross over)	Seal RAW
P6-14	92L	XC25	Open XC (remove wall and regulator)
P6-15	92L	XC24, XC25 and XC26 RAW	Completely remove wall

Project 2: 98L return system

98L is on the verge of becoming part of the return network (mining activities are soon concluding on the level). Large return system adjustments were made on this level due to the new tunnel availability that can improve the return airflow and distribution (see Figure 84). The readable action plan can be found in Table 38.

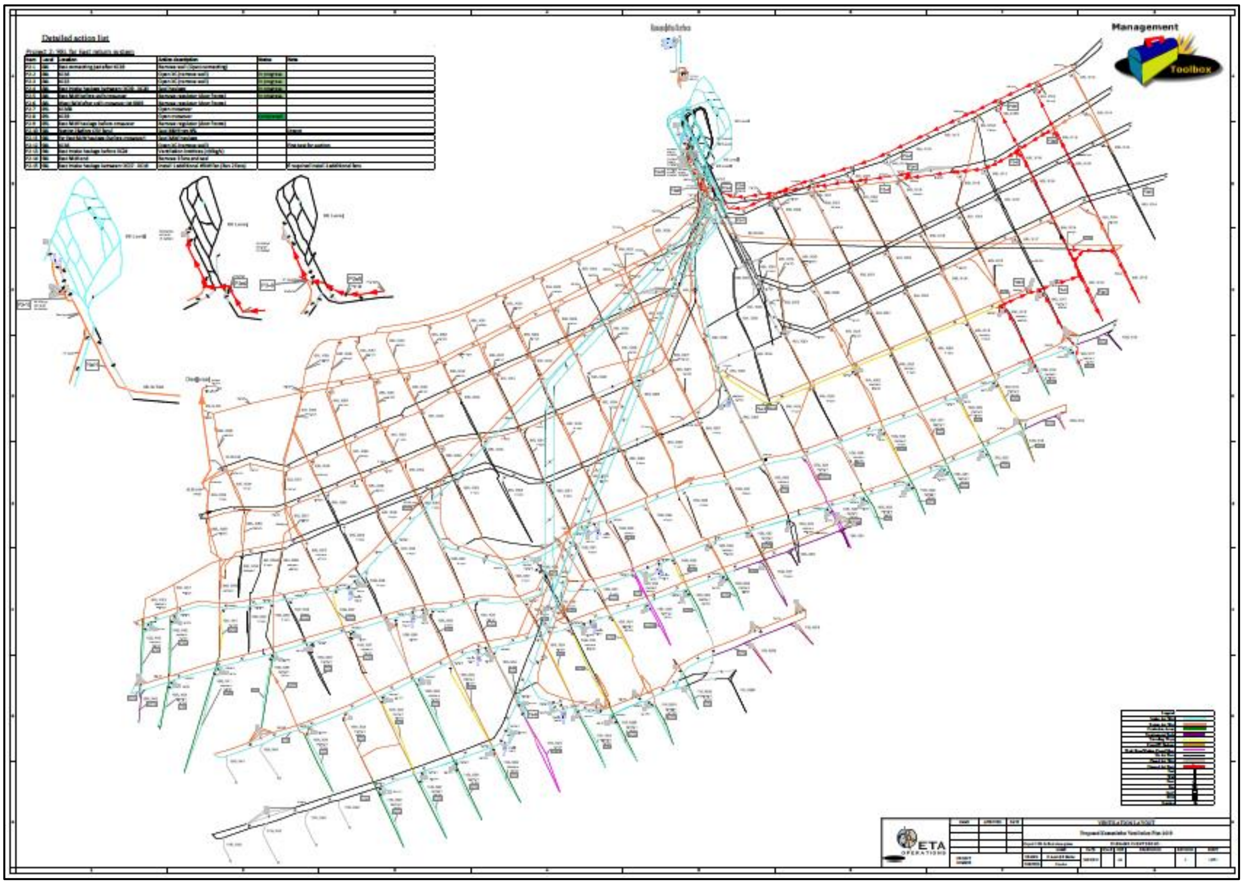

Figure 84: Project 2's schematic layout representation

Table 38: Project 2 action plan

Item	Level	Location	Action description
P2-1	88L	East connecting just after XC19	Seal open 760 mm column
P2-2	88L	XC16	Open XC (remove wall)
P2-3	88L	XC15	Open XC (remove wall)
P2-4	88L	XC17	Open XC (remove wall)
P2-5	88L	East haulage between XC18-19	Close XC door
P2-6	88L	East RAW between XC16-17	Open connecting (remove wall)
P2-7	88L	East RAW between XC18-19	Open RAW haulage (remove wall)
P2-8	88L	East RAW between XC16-17	Seal haulage
P2-9	88L	East RAW before split crossover	Remove regulator (door frame)
P2-10	88L	West RAW after split crossover (at RBH)	Remove regulator (door frame)
P2-11	85L	XC16B	Open crossover
P2-12	85L	XC19	Open crossover

Item	Level	Location	Action description
P2-13	85L	East RAW haulage before crossover	Remove regulator (door frame)
P2-14	98L	Station (before CSC fans)	Seal RBH from 95L
P2-15	98L	Far East RAW haulage (before crossover)	Seal RAW haulage
P2-16	98L	XC16	Open XC (remove wall)
P2-17	98L	East intake haulage before XC24	Ventilation brattices (±10kg/s)
P2-18	98L	East RAW end	Remove 3 fans and seal
P2-19	98L	East intake haulage between: XC17 - XC16	Install 1 additional 45 kW fan (Run 2 fans)
P2-20	98L	XC15	Remove restriction

Project 3: Sealing plan

The sealing plan project was focussed on removing all unnecessary leakages out of the PVS system. This will typically include connection leakages between the IAW and RAW as well as old level leakages (see Figure 85). The readable action plan can be found in Table 39.

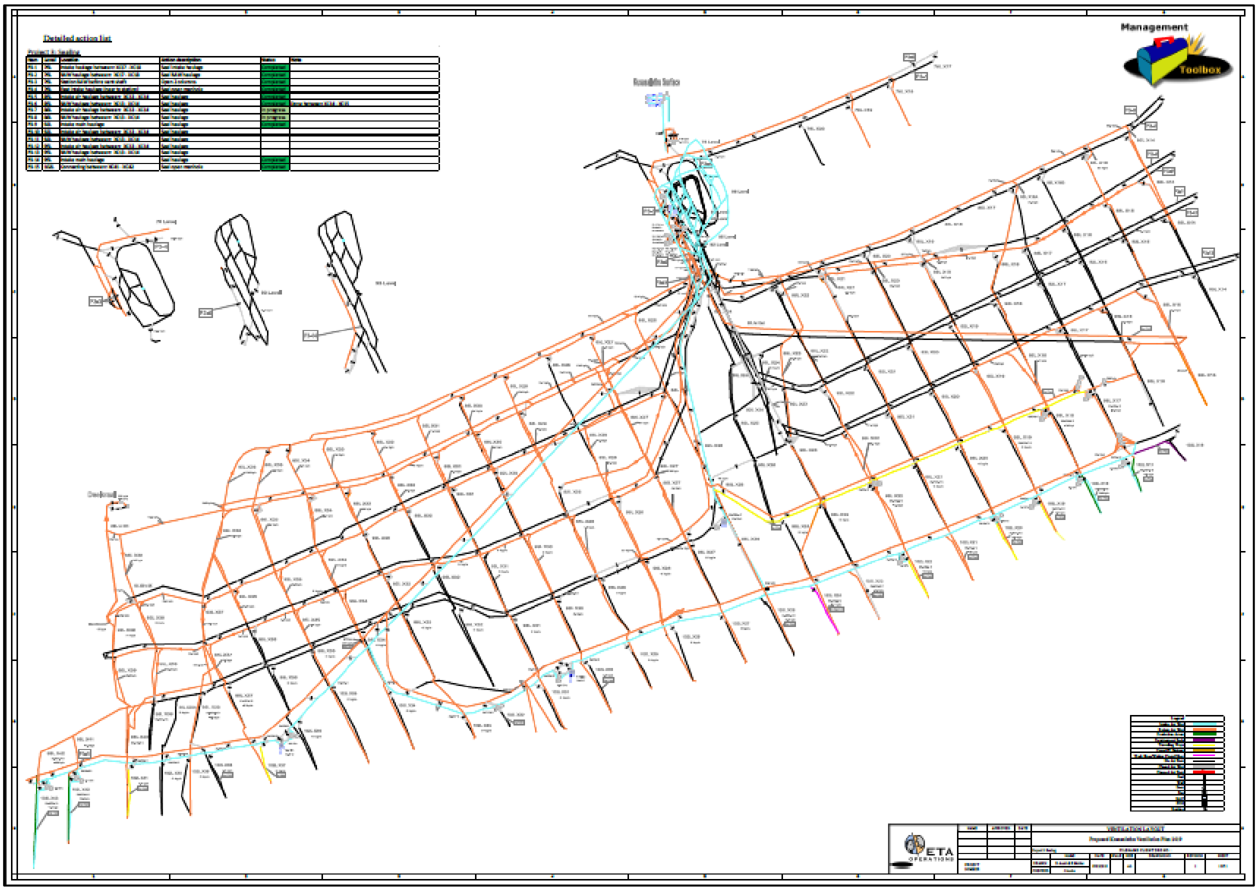

Figure 85: Project 3's schematic layout representation

Table 39: Project 3 action plan

Item	Level	Location	Action description
P3-1	76L	Intake haulage between: XC17 - XC18	Seal intake haulage
P3-2	76L	RAW haulage between: XC17 - XC18	Seal RAW haulage
P3-3	76L	Station RAW before vent shaft	Open 2 columns
P3-4	76L	East intake haulage (near to station)	Seal open manhole
P3-5	85L	Intake air haulage between: XC13 - XC14	Seal haulage
P3-6	85L	RAW haulage between: XC13 - XC14	Seal haulage
P3-7	88L	Intake air haulage between: XC13 - XC14	Seal haulage
P3-8	88L	RAW haulage between: XC13 - XC14	Seal haulage
P3-9	92L	Intake main haulage	Seal haulage
P3-10	92L	Intake air haulage between: XC13 - XC14	Seal haulage

Item	Level	Location	Action description
P3-11	92L	RAW haulage between: XC13 - XC14	Seal haulage
P3-12	95L	Intake air haulage between: XC13 - XC14	Seal haulage
P3-13	95L	RAW haulage between: XC13 - XC14	Seal haulage
P3-14	95L	Intake main haulage	Seal haulage
P3-15	98L	XC26 (RBH)	Seal RBH
P3-16	102L	Connecting between: XC41 - XC42	Seal open manhole

Project 4: Production block airflow regulation

The final project focusses on the production block airflow regulation. The production block is the most critical section of the mine. The purpose of this project is to make sure the required airflow is distributed to each crosscut of the mine (see Figure 86). The readable action plan can be found in Table 40.

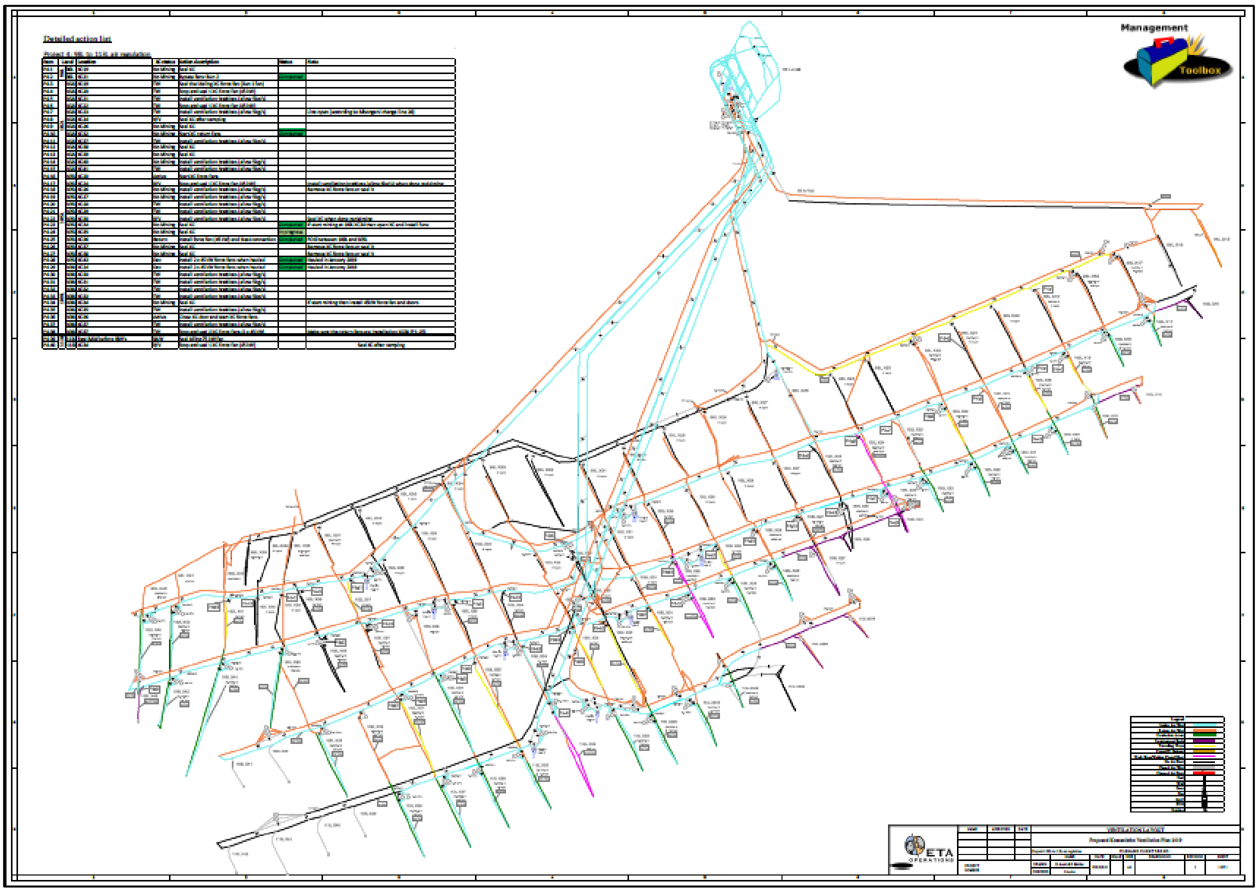

Figure 86: Project 4's schematic layout representation

Table 40: Project 4 action plan

Item	Level	Location	XC status	Action description
P4-1	98L	XC19	No mining	Seal XC
P4-2	98L	XC20	Return	Install 2 x 45 kW return fans
P4-3	98L	XC21	No mining	Bypass fans: Run 2 (and seal door)
P4-4	98L	XC22	No mining	Remove 2 x 45 kW return fans
P4-5	98L	East intake haulage	-	Install ventilation brattices (allow 10 kg/s)
P4-6	98L	West intake haulage	-	Install ventilation brattices (allow 10 kg/s)
P4-7	98L	XC29	No mining	Seal small leaks (hole in wall)
P4-8	98L	XC30	No mining	Seal small leaks (hole in wall)
P4-9	98L	XC32	Return	Install door with 2 x 45 kW return fans
P4-10	98L	XC35	Return	Install 1 x 45 kW return fan
P4-11	98L	XC37	No mining	Seal open 760 mm column
P4-12	98L	XC16	-	Install 1 x 15 kW & 45 kW force fans and doors (airlock)
P4-13	102L	XC19	Travel way	Seal the idling XC force fan (Run 1 fan) and close doors
P4-14	102L	XC20	Travel way	Stop and seal both XC force fans and Install ventilation brattices (allow 5 kg/s)
P4-15	102L	XC21	Travel way	Install 45 kW force fan and install ventilation brattices (allow 5 kg/s)
P4-16	102L	XC22	Travel way	Install ventilation brattices (regulate to 6 kg/s) and seal both 760 mm columns
P4-17	102L	XC23	Travel way/Return	Install 2 x 45 kW return fans
P4-18	102L	XC24	No mining	Seal XC and 760 mm column
P4-19	102L	XC26	Travel way	Install ventilation brattices (regulate to 6 kgs)
P4-20	102L	XC32	Return	Start 2 x 45 kW return fans
P4-21	102L	XC35	No mining	Seal man door and seal 760 mm open column
P4-22	102L	XC37	Travel way	Install ventilation brattices (allow 5 kg/s)
P4-23	102L	XC38	No mining	Seal XC and 760 mm open column
P4-24	102L	XC39	Travel way	Install 1 x 45 kW force fan
P4-25	102L	XC40	No mining	Seal XC
P4-26	102L	XC41	Travel way	Install ventilation brattices (allow 5 kg/s) and seal 2 x 760 mm open columns
P4-27	102L	XC41-42	-	Open RAW loop Install airlock doors in intake haulage Install 2 x 75 kW fans in RAW
P4-28	102L	XC42	-	Stop both fans and open XC doors
P4-29	102L	XC43	-	Stop both XC fans and open XC doors
P4-30	105L	XC19	-	Install 1 x 45 kW force fan and doors (airlock)
P4-31	105L	XC20	Active	Start 1 x 45 kW XC force fans

Item	Level	Location	XC status	Action description
P4-32	105L	XC21		Run 1 x 45 kW force fan and seal idling fan
P4-33	105L	XC24	Travel way	Run 1 x 15 kW fan and seal idling fan
P4-34	105L	XC26	Active	Run 1 x 15 kW fan and seal idling fan
P4-35	105L	XC27	No mining	Install ventilation brattices (allow 5 kg/s)
P4-36	105L	XC28	Travel way	Start 1 x 45 kW XC force fans
P4-37	105L	XC29	Travel way	Install ventilation brattices (allow 5 kg/s)
P4-38	105L	XC30	Vamping or reclaiming	Install ventilation brattices (allow 5 kg/s)
P4-39	105L	XC34	No mining	Seal XC
P4-40	105L	XC35	No mining	Seal XC
P4-41	105L	XC36	Travel way	Remove 1 x 45 kW return fan and install 1 x 45 kW force fan
P4-42	105L	XC38	No mining	Seal XC
P4-43	105L	XC43	Travel way/Return	Install 2 x 45 kW return fans
P4-44	109L	XC24	Active	Install 1 x 45 kW force fan and doors (airlock)
P4-45	109L	XC29	Active	Run 1 x 45 kW force fan and seal idling fan
P4-46	109L	XC30	Travel way	Install ventilation brattices (allow 5 kg/s) and seal 2 x 760 mm open fans
P4-47	109L	XC31	Travel way	Open doors
P4-48	109L	XC32	Travel way	Install ventilation brattices (allow 5 kg/s) and close doors
P4-49	109L	XC33	Travel way	Install ventilation brattices (allow 5 kg/s)
P4-50	109L	XC34	Vamping or reclaiming	Open XC doors
P4-51	109L	XC35	Travel way	Install ventilation brattices (allow 5 kg/s) and seal 760 mm column
P4-52	109L	XC36	Active	Install 1 x 45 kW force fan and run 2 x 45 kW force fans
P4-53	109L	XC37	Active	
P4-54	109L	XC38	Active	Install 1 x 45 kW force fan and run 2 x 45 kW force fans
P4-55	109L	XC39	Active	Run 1 x 15 kW force fan and seal idling fan
P4-56	113L	East RAW before RBHs	-	Seal Idling 75 kW fan
P4-57	113L	XC34	Vamping or reclaiming	Stop and seal 1 XC force fan (45 kW)
P4-58	113L	XC31 S	Active	Run both XC force fans
P4-59	113L	XC32 S	Active	Run 1 x 45 kW force fan and seal idling fan
P4-60	113L	XC33	Active	Run 1 x 45 kW force fan and seal idling fan
P4-61	113L	XC34	Vamping or reclaiming	Run 1 x 45 kW force fan and seal idling fan