# Optimising energy recovery on mine dewatering systems

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# Abstract

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Mines in South Africa face many challenges. Chief among these are rising production costs. This coupled with labour unrests and investor uncertainty means that mines are under significant strain. An excellent method to save costs on mines is through improved energy management, leading to better profitability.

Mines are therefore continuously looking for innovative ways to reduce energy usage. Hydraulic energy recovery devices such as turbine pumps and three-chamber pipe feeder systems are often used to save energy. These devices use chilled water required for mining activities and underground cooling to aid in the resulting dewatering process.

Hydraulic energy recovery devices do not entirely replace traditional pumps. It is therefore necessary to consider cost-saving initiatives on traditional pumping systems when controlling energy recovery devices. Previous studies show that typical cost-saving initiatives on these pumping systems are load management and water supply optimisation.

Previous studies investigated the optimal integration of energy recovery devices into dewatering systems. However, these studies overlooked certain difficulties associated with controlling these systems. It was also found that certain technologies used in older studies were outdated.

The study identified a need to develop an optimisation methodology to ensure maximum energy reduction through hydraulic energy recovery systems. The methodology must allow for additional cost savings through conventional load management and water supply optimisation.

A methodology was proposed to optimally integrate hydraulic energy recovery devices into a mine dewatering system. The new process was verified through the simulation of a case study. The applicable methods were tested on the simulated case study and proved to be effective.

The methodology was also tested on an alternative practical case study after being verified. The proposed methodology was used to develop a control strategy for the case study. The aim of the control strategy was to enhance the load management performance of the mine.

It was shown that a load of 1.5 MW could be shifted from the Eskom evening peak and 2 MW from the morning peak. The result of these initiatives is a potential R1.7 million cost saving p.a. on the dewatering system of the practical case study if all the equipment remains available. The impact on the system electricity costs shows the effectiveness of the methodology.

**Keywords:** Energy recovery, Energy efficiency, Turbines, Three-chamber pipe feeder system, Demand side management, Load management, Mine water reticulation.

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# Nomenclature

## Acronyms

| | |
|---|---|
| 3CPFS | Three-chamber pipe feeder system |
| DSM | Demand side management |
| ERD | Energy recovery device |
| ESCo | Energy service company |
| ISO | International organisation for standardisation |
| L | Mining level |
| M&V | Measurement and verification |
| OPC | Open platform communication |
| PLC | Programmable logic controller |
| $R^2$ | Coefficient of determination |
| REMS | Real time energy management system |
| RMSE | Root mean squared error |
| SCADA | Supervisory control and data acquisition system |
| TOU | Time of use |
| VSD | Variable speed drive |
| WB | Wet bulb |
| WSO | Water supply optimisation |

## Units of measurement

| | |
|---|---|
| % | Percentage |
| bar | Pressure in bar |
| ˚C | Degrees Celsius |
| GW | Gigawatt |
| kg | Kilogram |
| $kg/m^3$ | Kilogram per cubic meter |
| kg/s | Kilogram per second |
| $kg^2/(s \cdot m^3 \cdot kPa)$ | Mechanical flow admittance |
| kJ/kg | Kilojoule per kilogram |
| km | Kilometre |
| kPa | Kilopascal |
| kW | Kilowatt |
| kWh | Kilowatt-hour |
| ℓ | Litre |
| m | Meter |
| $m^3$ | Cubic meter |
| Mℓ | Mega litre |
| MW | Megawatt |
| rpm | Revolutions per minute |
| R | Rand (South African currency) |
| s | Seconds |

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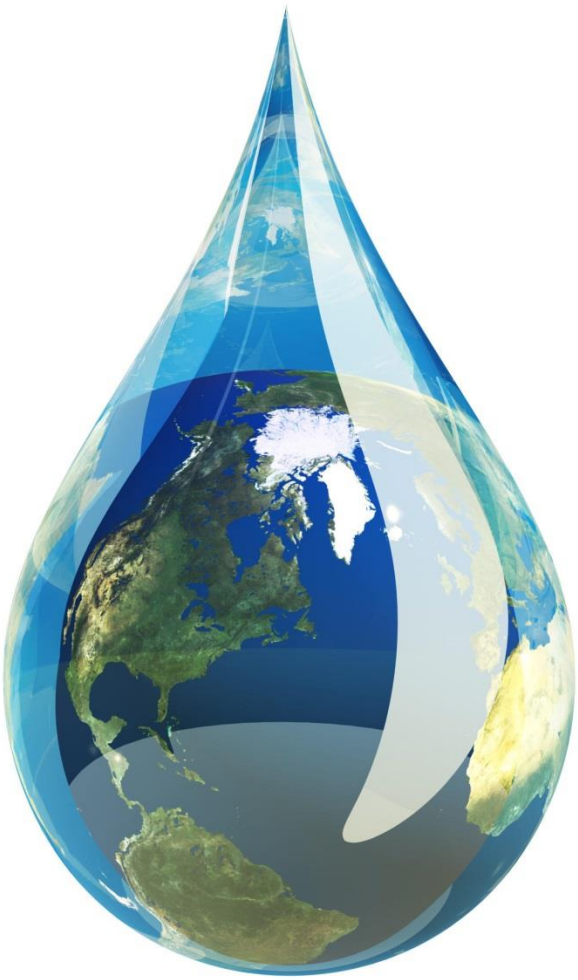
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# Chapter 1: Introduction and background



[1]

This chapter focuses on the background, identification and formulation of the research problem. A brief overview of the rest of the study is also provided.

[1] Olearys, "Planeta Agua," 27 Sep 2008. [Online]. Available: https://www.flickr.com/photos/97513256@N06/9044197344. [Accessed 19 Nov 2017].

## 1.1 Challenges faced by the South African gold mining industry

South African gold mines face many challenges. These challenges include declining ore grades, labour unrest, reduced productivity and fluctuating gold prices. As mines get older, these challenges are aggravated. The many challenges faced in South African mining is leading to investor uncertainty which puts even more strain on an already struggling industry [1].

In addition to the problems mentioned above, production costs of mines are also rising [1]. Significant contributors to this fact are rising electricity prices and labour costs. Electricity prices in South Africa have risen by 448% since 2007 [1]. The South African labour costs increased by above 10% per annum from 2011 to 2016 [2].

Since gold mining is a significant contributor to the economy of South Africa, this could lead to potential social-political problems for the country as a whole [1]. Figure 1 shows the typical expenditure of a South African gold mine.

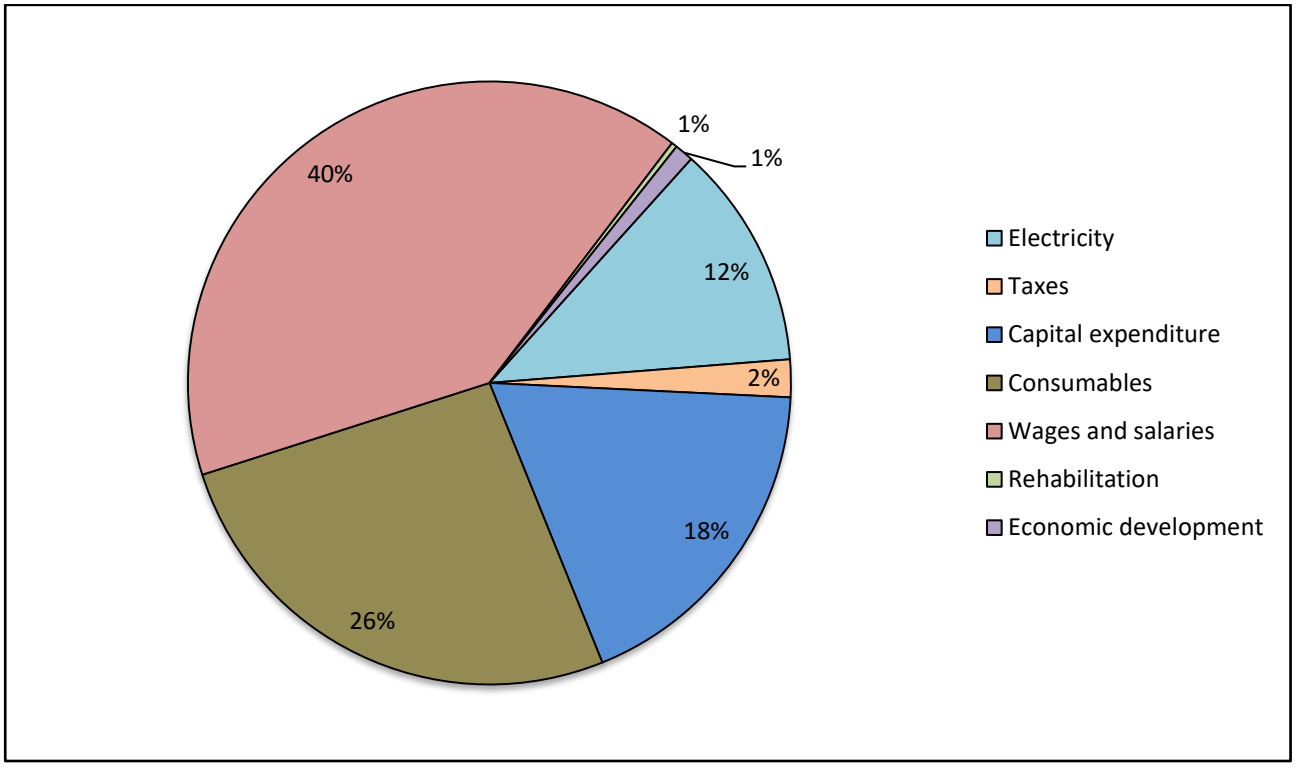



**Figure 1: Typical gold mining expenditure (adapted from [3])**

From Figure 1 it is clear that wages, salaries and capital spending forms the bulk of the mine's expenses. Electricity costs are also a significant expense. Exploring any of these avenues for potential cost savings could therefore be beneficial to the mines.

However, South African mines are amongst the most energy-intensive in the world [4, 5]. South African industries benefited from historically low electricity prices. Low electricity prices discouraged awareness of effective energy management in the past [5]. In addition to this, gold mines are getting deeper which raises the threat of energy intensity escalating further, since the energy intensity of mines typically increases with depth [6].

The high-energy intensity of South African mines is of particular concern given the recent rise in electricity prices. The electricity price increases could therefore be detrimental to the sustainability of the gold mining industry in South Africa [3].

## 1.2 Energy cost saving opportunities in South Africa

Research into the optimisation of energy-intensive systems on deep level gold mines has been shown to be justified. However, a basic understanding of the energy landscape in South Africa, as well as the mining industry, is required before the investigation of specific opportunities can commence.

The power utility, Eskom, is the primary supplier of electricity to mines in South Africa. Eskom presently has a total generation capacity of approximately 43 GW. Large coal-fired power stations provide 85% of Eskom's generation capacity. A nuclear plant, emergency gas turbines and various renewable energy sources supply the remainder of the capacity [7].

During the economic recession of 2008, South Africa started experiencing electricity supply shortages [8]. Eskom approved two new coal-fired power stations to combat this shortage. Eskom plans to commission the Medupi and Kusile power stations by 2020 and 2022 respectively. These plants will add approximately 9500 MW of power to the national grid. The commissioning of the first unit of the Medupi power station occurred in August 2015 [7].

Eskom implemented various other initiatives to help combat electricity shortages. Among these initiatives is a time of use (TOU) tariff structure and Eskom's demand side management (DSM) program. TOU tariffs and the DSM program were implemented in 1992 and 2004 respectively. These initiatives aim to lower the total generation capacity needed by Eskom [9].

TOU tariffs are used because the daily demand profile of the national grid tends to peak during certain times of the day, especially in winter months. The peaks in the national power demand can be a potential problem since the bulk of Eskom's generation capacity is provided by large coal-fired power plants that cannot be stopped regularly or deliver variable outputs [9].

It is also not possible to store electricity efficiently on a national scale and the peak power usage therefore determines the base generation capacity needed for the country. Although measures such as gas turbines and hydro storage facilities can combat this problem, these technologies are expensive [9].

TOU tariffs aim to reduce this peak demand by billing clients more for power used during peak times. The higher electricity rates in peak time encourage reduced energy usage, thus reducing the base load power supply needed on a national scale [9].

TOU is implemented in South African industries in the form of Eskom Megaflex tariffs. Eskom usually supplies large industrial consumers of electricity, like mines, directly [10]. Appendix A provides a detailed breakdown of Eskom's Megaflex structure.

Eskom's DSM program provides funding for the implementation of three main types of projects. These are energy efficiency, peak clipping and peak load management (also known as load shifting) [11]. Figure 2 shows the typical influence of the three primary types of DSM projects on the daily power profile of the given system. The impact of each is measured relative to a system reference or baseline which depicts the operation of the system before the implementation of the initiative.

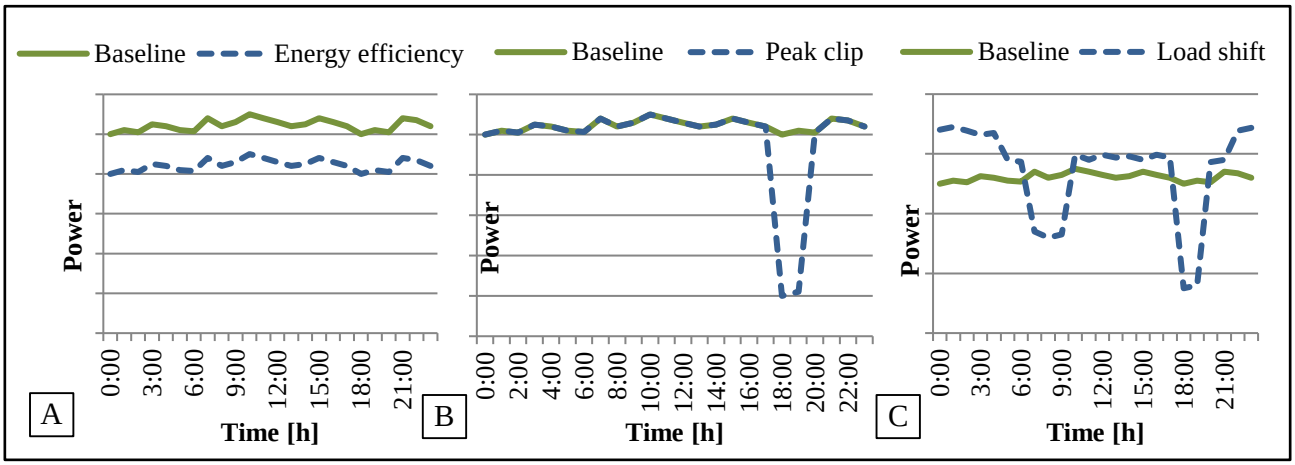



**Figure 2: DSM initiatives**

Energy efficiency projects (depicted in Figure 2-A) aim to reduce the amount of energy required for a specific task. Energy efficiency projects typically involve making control alterations or installing equipment that enables a more efficient use of the available energy [9].

Peak clipping (depicted in Figure 2-B), in turn, is an energy efficiency initiative that focuses on reducing power usage exclusively during peak times [9]. The effect of the initiative on the system’s power usage is therefore isolated to peak hours.

Load management (depicted in Figure 2-C) involves decreasing energy usage during peak times by increasing the usage during off-peak hours. The total energy usage should remain unchanged when load management is implemented, however, the peak demand reduces. The energy demand is therefore spread more evenly through the day on a national scale. Load management is therefore a cost-saving initiative since the electricity use of the system is shifted to periods with reduced rates [9].

ESCOs have implemented DSM projects on all of the significant electricity consumers of gold mines [12]. Figure 3 shows the energy consumers on a deep level gold mine. The figure also indicates the percentage contribution of each subsystem to the total power usage of a typical gold mine.

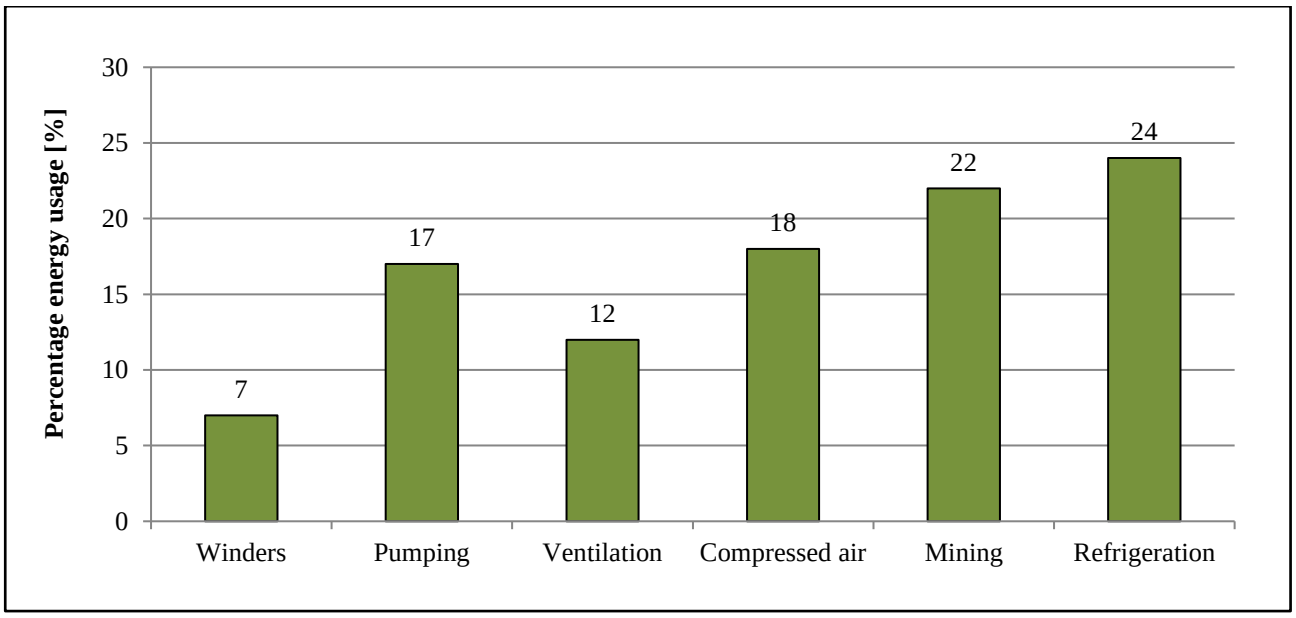



**Figure 3: Mine energy user distribution (adapted from [13])**

Refrigeration and the resulting pumping process forms the water reticulation system of deep level gold mines that consume approximately 41% of the total mine energy [13]. This study will focus on optimising initiatives on these water systems. The pumping or dewatering system of a mine, in turn, consumes up to 17% of the total energy usage of a mine. Investigating means of reducing this specific consumption is therefore worthwhile.

## 1.3 Dewatering of mines

Figure 4 shows the basic operation of a mine water reticulation system.

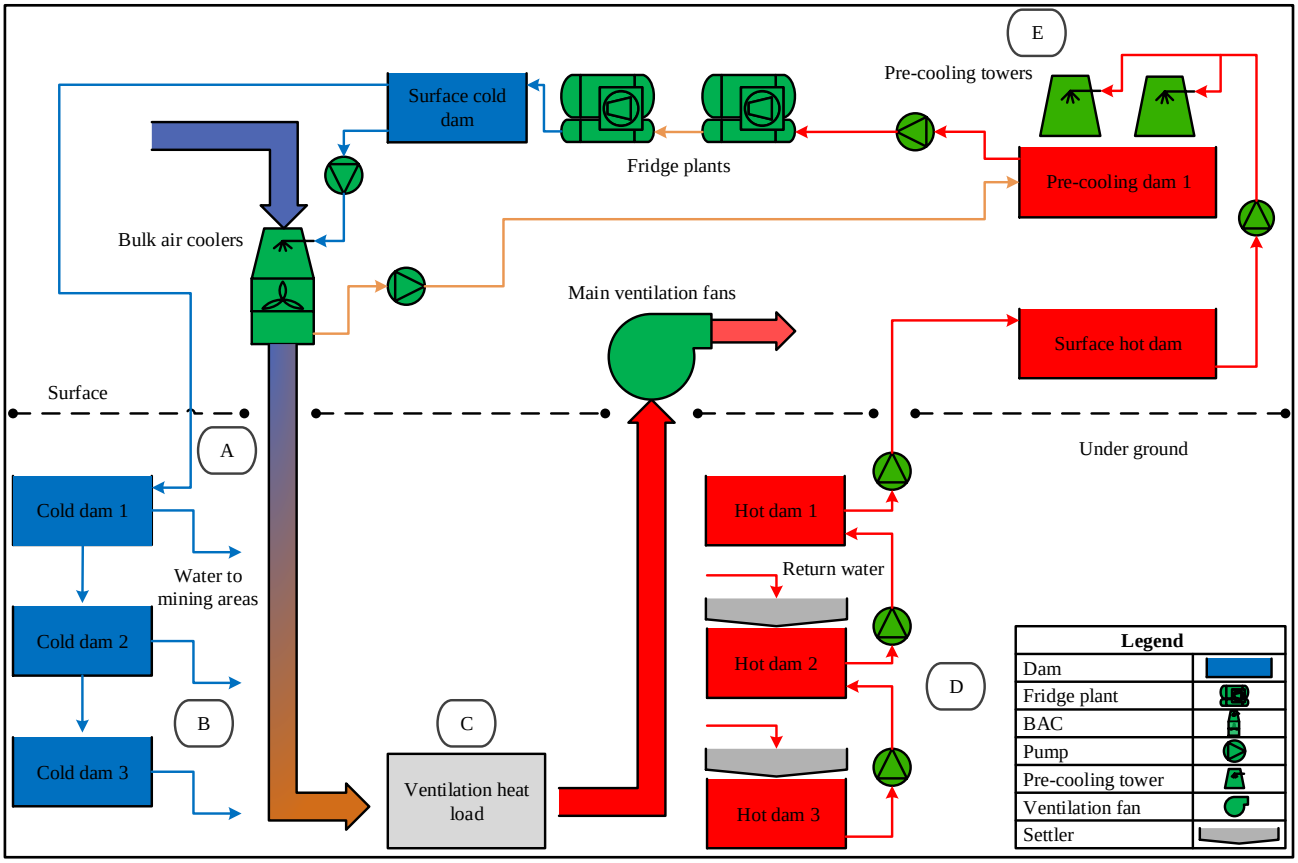



**Figure 4: Basic mine refrigeration operation**

Deep level gold mining operations in South Africa requires high volumes of water, this water is typically sent underground in a series of cascading dams as shown at point A in Figure 4. A cascading configuration minimises pressure build-up in the pipes [14]. The pressure head in the water column is reduced before it enters dams on lower mining levels [15].

Underground chilled water uses include drill cooling, cleaning and sweeping, and dust suppression among others, during the mining process. The drilling process causes a lot of friction which in turn generates heat. The blasting process generates a lot of dust which has to be suppressed for health and safety reasons [15]. Water for such uses is sent into the mining areas from the cold water dams as shown at point B.

Mines also use water for underground ventilation air cooling. The preferred work area temperature of mines in South Africa is typically below 29.5°C wet bulb [16]. Virgin rock temperatures in deep level gold mines can reach up to 70°C. Additionally, auto compression of the ventilation air causes the ventilation air temperatures to rise by approximately 0.3°C/300m of depth [17].

Extensive cooling of the ventilation air is therefore necessary to maintain legal conditions underground. Chilled water is ideally suited to provide cooling for the ventilation air since it has a relatively large thermal heat capacity and is required for other mining operations [14].

The dewatering system of deep level gold mines forms part of a larger integrated cooling system [18, 15]. Underground refrigeration occurs in many ways depending on the depths and needs of the mine, shown collectively in point C of Figure 4. Cooling the water on the surface and circulating it underground is preferable. As the mining depths increase, underground cooling or the use of ice might become necessary [19].

After the water has been used underground, it is chanelled to a series of dams as shown at point D. The water is then typically pumped out of the mine in a cascading fashion. On surface the water is refrigerated again before being sent back underground, as hown at point E in Figure 4 [18].

It is clear that these systems consists of many components. More information on the important components will now be provided.

**Clear water dams**

Clear water dams in mines are reservoirs used for the storage of water. As mentioned earlier the dams are typically scattered in a cascading fashion throughout the mine. Multiple dams are usually located on designated mining levels. These dams are either used for the storage of cold service water that still needs to be sent to the appropriate mining areas or hot used water before it is pumped out of the mine [15].

These dams can have volumes of up to 5 Mℓ [15]. The locations of these dams need to be restricted to geologically stable areas underground for safety reasons [20]. A proven tool for analysing the accuracy of flows in and out of dams is the continuity equation, displayed below [21].

$$\sum \dot{m}_i\, t + \dot{m}_i = \sum \dot{m}_e\, t + \dot{m}_e \qquad \text{Equation 1}$$

Where:

$\sum \dot{m}_i$ = the total mass flow into the dam [kg/s]

$\dot{m}_i$ = the initial mass of water within the dam [kg]

$\dot{m}_e$ = the mass of water in the dam after a given time period [kg]

$\sum \dot{m}_e$ = the mass flow out of the dam [kg/s]

t = time [s]

**Settlers**

Settlers are used in mines to clarify used service water before it is sent to clear water dams. Mud and other impurities are caught up in the bottom of settlers, separated from the clear water and pumped out of the mine using mud pumps. The clear water, in turn, overflows out of the settler into a clear water hot dam before being pumped out of the mine [18].

Note that settlers never remove all the impurities form the water, some of it therefore still settles in the bottom of dams. This leads to minimum dam level requirements to prevent the impurities from passing through pumps in large quantities. The effect of this on the control will be discussed in detail later in the study.

Note that various types of settlers exist [15]. The detailed operations of the different types of settlers will not be discussed in this study.

**Pumps**

Traditionally large multi-stage centrifugal pumps are used to dewater deep mines. These pumps pump the water between the cascading dams, and eventually out of the mine. Such pumps are energy intensive due to the significant flows delivered (up to 200 ℓ/s) and pressure heads (up to 1000m) that need to be overcome in deep mines [18].

Typically, multiple pumps are located on any given mining level to form a pumping station. These pumps are usually configured in a parallel configuration. This means that each pump can overcome the pressure head required. The flow is usually controlled by switching pumps on and off as required [18].

Equation 2 can be used to calculate the power required in a dewatering pump [22].

$$P = \frac{\rho g Q h}{\eta} \qquad \text{Equation 2}$$

Where:

$P$ = The pump power [kW]

$\rho$ = The fluid density [kg/m$^3$]

$g$ = Gravitational constant (assumed to be 9.8 [m/s$^2$])

$Q$ = The volumetric flow of the water [m$^3$/s]

$h$ = The pressure head required in the stream [m]

$\eta$ = The hydraulic efficiency of the pump

From Equation 2 it is clear that the pumps power requirements will increase if the flow or pressure head required increases.

**VSDs on pumps**

Variable speed drives (VSDs) are devices that is used to alter the rotational speed of electrical motors. By altering the speed of the motor, the pump impeller speed can be adjusted and thereby the flow of the pumps.

The speed of the pump motors and the pump itself can be adjusted in two main ways. The first is by altering the number of poles in the motor. VSDs however, alters the frequency of the current in the motor, which also alters the speed [23].

By altering the speed, the motor current and therefore power sent to the pump is reduced. It carries certain risks, however. These risks include temperature rises in the motor itself, pitting of bearings and problems with the harmonics of the VSD. However, the means of managing these problems exist [23].

VSDs are best suited to scenarios with low-pressure heads and high friction [23]. It can and has therefore been widely implemented as an energy savings measure on auxiliary pumps of refrigeration systems [24].

The influence of a change in rotational speed on pump performance can be determined by the following equations [23].

$$\frac{Q_1}{Q_2} = \frac{N_1}{N_2} \qquad \text{Equation 3}$$

$$\frac{H_1}{H_2} = \left(\frac{N_1}{N_2}\right)^2 \qquad \text{Equation 4}$$

$$\frac{P_1}{P_2} = \left(\frac{N_1}{N_2}\right)^3 \qquad \text{Equation 5}$$

Where:

Q = the volumetric flow rate of the water [ℓ/s]

N = rotational speed of the pump [rpm]

H = the pressure head delivered by the pump [m]

P = power usage of the pump [kW]

The relationship between the pump speed and flow, pressure head delivered and power usage is therefore linear, quadratic and cubic respectively.

**Valves**

Valves are used for many purposes underground. Four main types of valves will be discussed as part of this study.

***Butterfly valves***

Butterfly valves consist of a disc and a seat within an enclosure. The disc is connected to a shaft and located within the body. This section is typically connected to an inlet and outlet pipe. If the disc is rotated 90 degrees, water can flow freely over the disc. If the disc is rotated 180°, it seals against the seat, and the flow is shut off completely [18].

***Globe valves***

Globe valves have a somewhat more complex design compared to that of a butterfly valve. The enclosure of a globe valve follows an indirect path over a plug. By adjusting the position of the plug within the enclosure, the pressure drop across the valve can be regulated. By regulating the pressure drop, the flow through the valve can be controlled as required [18].

### *Pressure reducing valves*

Pressure-reducing valves (PRVs) are typically located in the inlets of the mining sections and dams. The pressure builds up in mining columns can be high. PRVs are valves with deliberately high-pressure drops. The water flowing through PRVs are therefore reduced to safe, usable pressures [25]. Note that pressure-reducing valves on the inlet of dams without ERDs present are commonly referred to as dissipators [25].

It is also important to note that both globe valves and PRVs makes use of a process called throttling. This process reduces the pressure and therefore potential flow of the water by introducing a large mechanical flow resistance to the stream. This can potentially increase the water temperature. This effect will be explored in more detail later in the study.

## Instrumentation

### *Flow meters*

Multiple flow metering techniques are available. The most commonly used in modern mines are electromagnetic flow meters. These flow meters create a magnetic field in coils around the water stream. The stream produces a voltage when it passes through the magnetic field. This voltage is directly proportional to the flow [26].

Other technologies include orifice type flow meters, ultrasonic flow meters and vortex flow meters. The details of these flow meters will not be discussed. However, it is important to note that portable ultrasonic type flow meters are frequently used in surveys and energy audits [27].

### *Pressure transducers*

Pressure sensors convert the displacement of a mechanical element due to the water pressure to an electrical impulse. This pressure is typically relative to the atmospheric pressure. If the pressure sensor is connected to control apparatus such as programmable logic controllers (PLCs), it is known as a pressure transducer [28].

### *Dam level sensors*

Dam level sensors are typically pressure transducers located in the bottom of the dam. By measuring the static pressure at the bottom of the dam, the vertical height of the water in the dam can be calculated [29].

### *Power meters*

Power meters are typically installed on pumps to measure the energy and power used by the equipment. Pumps in mines are almost always driven by three phase motors. The current and voltage of each phase is therefore measured. From these parameters, the total power usage of the pump is calculated [30].

### Energy recovery devices

An improvement on the use of PRVs or dissepators to reduce the pressure in cold water columns that was discussed is to recover the available energy and utilise it wherever possible. Energy recovery devices (ERDs) use the energy available in the cold-water stream sent down the mine to aid in the dewatering process, generate electricity or to produce shaft power for other applications [31].

ERDs can therefore, reduce the energy usage of a mine significantly. However, increasing the overall efficiency by utilising the energy available in the chilled water sent down the mine is not the only benefit of ERDs [32].

ERDs can also be used to improve the temperature of the water in underground dams. The traditional configuration of reducing the pressure head of the water before it enters the chilled water dams can increase the water temperatures by as much as 2.33°C/1000m [33].

By utilising the pressure head, most of the available energy in the water sent down the mine is converted to valuable shaft power instead of heat. The typical increase in water temperature is therefore only 0.833°C/1000m for an efficient turbine [33].

Figure 5 shows some of the main types of ERDs used in deep level gold mines' water reticulation systems.

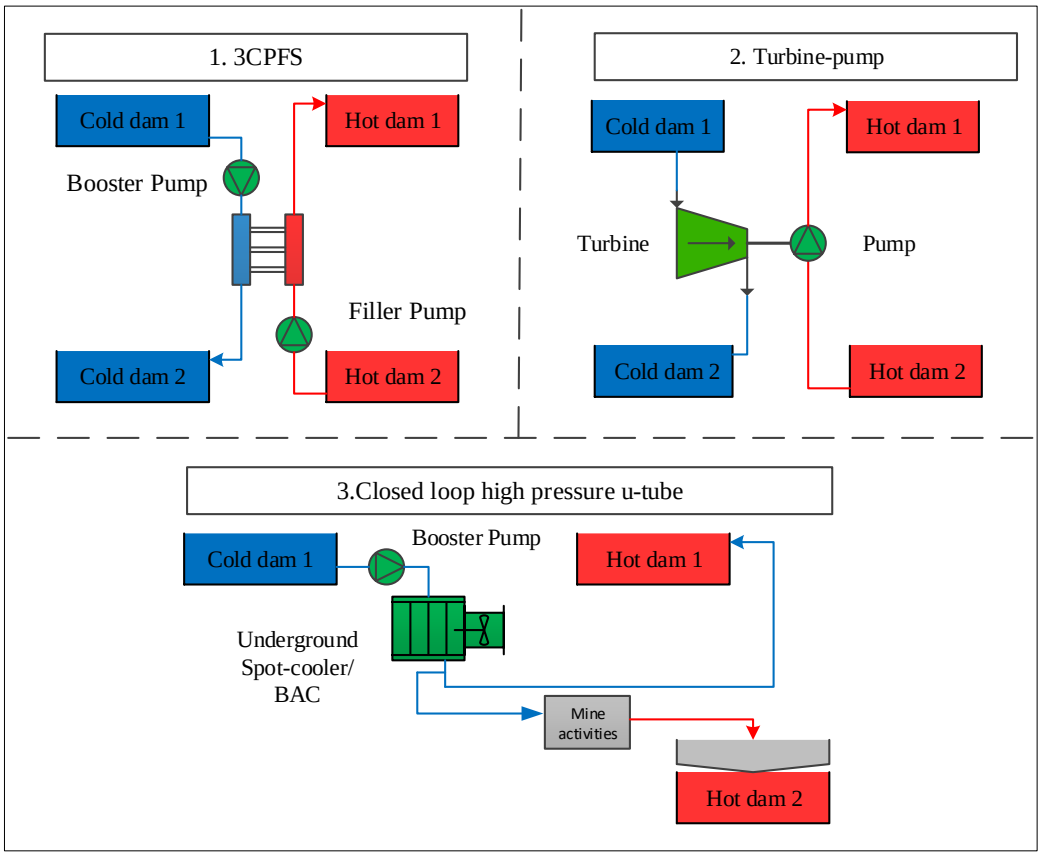



**Figure 5: Co-generative energy recovery devices**

These ERDs are three-chamber pipe feeder systems (3CPFSs) [34], turbine-pumps [34] and closed-loop high-pressure u-tube systems [25]. Note that other means of energy recovery on deep level mines do exist. Examples of these are turbines [33], pumps that double as turbines [35] and turbines that drive air compressors [32].

The first ERD shown in Figure 5.1 shows a 3CPFS. A 3CPFSs works on a u-tube principle and consists of a return and a feed column. In addition to this, three chambers, as well as a valve system is utilised [36].

Figure 6 displays the detailed working of a 3CPFS.

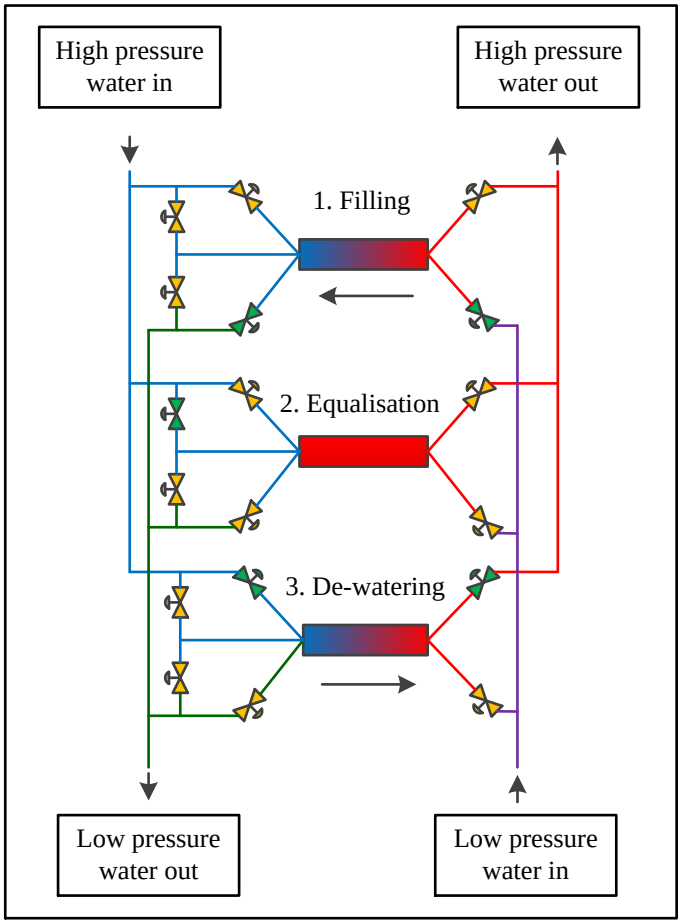



**Figure 6: 3CPFS workings (Adapted from [36])**

A height difference between the dams or auxiliary pumps known as filler and booster pumps are required for the operation of a 3CPFS. The 3CPFS will use the height difference or auxiliary pumps to induce flows and overcome mechanical losses in the system.

Since the system works on a u-tube principal, these pumps do not overcome pressure heights. The use of VSDs are therefore possible on these systems. This means that 3CPFSs can produce variable flow to meet demand [36]. This is noteworthy and has not been considered by previous studies.

Each chamber can fulfil one of three roles in an alternating fashion. These roles are filling, equalisation and dewatering [36]. The first role involves the filling of the chamber with hot water. This phase starts with the chamber full of cold water (the process is repetitive and the cold water is a residual of the last phase). Water from a hot dam is pumped into the specific chamber and displaces the cold water. Some contamination of the cold water occurs during this phase, but the chamber is filled as fast as possible to minimise this [36].

The second role involves pressure equalisation. During this phase, the pressure available in the cold water column is gradually introduced to the hot water within the chamber. If this phase does not occur, the sudden pressure change can lead to an effect known as water hammer that causes severe vibrations and could potentially damage the system [36].

The third role involves the dewatering of the chamber. During this phase, the full pressure in the cold water column is introduced into the chamber. The booster pump or height difference induces a flow through the u-tube that is formed in the system, thereby dewatering the hot water and filling the chamber with cold water [36].

Each chamber can fulfil all of the described roles. The 3CPFS alternates these processes between the three chambers. Therefore, while the first chamber is filling, the second will be equalising and the third dewatering. The 3CPFS pumps a continuous stream of water out of the mine [36].

The amount of hot water a 3CPFS system pumps out of the mine is typically equal to 90% of the cold water sent down the shaft. Mines often also have groundwater that seeps through into the underground dams. In addition to this, water that enters the mining levels cannot be used for energy recovery. A 3CPFSs do not entirely replace traditional pumps [34].

The 3CPFS do not replace traditional pumps completely. Backup pumps are still required when maintenance is required or if the system fails. It therefore raises the installation costs of the total dewatering system significantly [37].

The next ERD that will be considered is turbine-pumps as illustrated in Figure 5.2. To understand the turbine pump as a whole some background on turbines needs to be provided. Turbines used in mines can be classified into two main sub-sections. These are turbines that produces a set amount of back pressure and turbines that don’t [37].

An example of a turbine that provides no back-pressure is a Pelton wheel turbine. A Pelton wheel converts the pressure available in the cold water columns to kinetic energy through the use of water jets. These water jets in turn drives a rotating wheel. Mechanical shaft power is therefore generated and the water is reduced to ambient pressure [37].

If a set amount of backpressure is required for whatever reason, a Francis turbine can be used instead. A set of guide vanes partially converts the available pressure into kinetic energy. The pressure is therefore not reduced to ambient pressure and could be used for mining activities should it be required. The exact reduction in pressure can be controlled by altering the configuration of the guide vanes, configuration of the impeller and the rotational speed [37].

These type of turbines are in fact often pumps that double as turbines as mentioned earlier. An alternate use for such turbines is in a pressure reducing capacity in stead of PRVs before the water enters mining levels. In such a scenario, the flow through the turbine will not necessarily be constant. If this is the case, the generators are usually coupled with frequency inverters to match the rest of the electrical network it is feeding into.

Turbines are also sometimes coupled directly to pump to form components known as Turbine-pumps. The turbine on these systems is typically a Pelton wheel turbine. The pump, in turn, is usually centrifugal [32]. The system will also typically be designed to operate at a set design flow for both the turbine and pump.

Due to mechanical inefficiencies, turbine-pumps can typically only dewater a maximum of 68% of the cold water sent through the turbine. As with the 3CPFS, turbine-pumps do not entirely replace traditional pumps [32].

It is important to note that for this configuration to work with a Pelton wheel the hot and cold dams need to be in close proximity to each other, since the pressure of the cold water stream is reduced to ambient pressure.

Using a Francis turbine could also be feasible if this is not possible, but could potentially lead to reduced pumping potential if all of the pressure is not fully utilised. However, compared to the 3CPFS the system is relatively simple with less mechanical components which could be beneficial from a maintenance perspective.

The last ERD to be discussed is a closed-loop high-pressure u-tube system shown in Figure 5.3. Closed-loop high-pressure u-tube systems utilise the pressure head available in the chilled water to dewater the water required for underground ventilation cooling [25].

A closed-loop high-pressure u-tube system sends the cold water through underground coolers under pressure. The water required for the mining activities is bled off from this stream after the water has been used for cooling. The mining water therefore still needs to be pumped to the surface [25]. These systems are therefore typically used exclusively on deep mining sections where underground cooling is required.

When comparing the different ERDs it is important to consider some additional information. Due to the age of gold mines in South Africa, the majority of the actual mining activities occurs at great depths (typically 2500m to 3000m) [37]. The roles of the different ERDs discussed so far therefore varies.

The 3CPFSs, Turbines (pelton wheels in particular) and turbine pumps are used in the main columns that supplies water to the lower levels and not in the supply systems that carries the water to the specific mining levels. The closed loop high pressure u-tube system and Francis turbine technology in turn is used exclusively for this purpose [37].

Another distinction is the role and impact of each of these systems on the system as a whole. The use of turbines coupled directly to generators and the Francis type turbine simply replaces the Dissipaters or PRVs. Since this is the case, it does not affect the control of the system at all. It fulfils exactly the same role in the system as older technologies, with the added advantage of increased efficiency.

Turbine pumps and 3CPFSs plays a direct role in the dewatering process. However, these components require a need and availability of cold service water to fulfil this role. This greatly complicates the control of the systems. The closed loop high pressure u-tube system also aids in the dewatering process. For this reason, these three systems can be classified as co-generative ERDs.

The use of ERDs, offers significant improvements over traditional dewatering and water distribution systems in mines. The improvement can be seen in both the potential improvement of the cold-water temperatures and efficiency.

The biggest disadvantage is increased capital costs that is required and from an operational point of view, the increase in complexity and especially the control of the dewatering system.

Table 1 summarises the advantages and disadvantages of each of the discussed co-generative ERDs.

**Table 1: Co-generative ERD comparison summary**

| ERD | Advantages | Disadvantages |
|---|---|---|
| Pump-turbine | • Improves overall system efficiency<br>• Service water temperature improvement<br>• Simple configuration compared to 3CPFS | • 68% overall efficiency<br>• The underground hot and cold dams need to be appropriately positioned.<br>• Backup pumps required<br>• Increased total dewatering system installation costs<br>• Complicates dewatering system control.<br>• Only offers benefit if water demand exists. |
| 3CPFS | • Improves overall system efficiency<br>• Dewatered flow ≈ supply flow | • Contamination of chilled water with hot water.<br>• Complicates dewatering system control.<br>• Only offers benefit if water demand exists.<br>• Complicated configuration of valves that can lead to increased system downtime.<br>• Increased maintenance costs Increased total dewatering system installation costs. |
| Closed-loop high-pressure u-tube | • Improves overall efficiency<br>• Underground cooling performance increase | • Used mining water cannot be introduced back into the system.<br>• Can only be used to supply water to the levels.<br>• Increased total dewatering system installation costs.<br>• Increased potential for leaks due to high pressure water supply into levels.<br>• Potential safety concerns with high pressure water into levels.<br>• Can complicate the distribution of water between underground cooling equipment. |

Although ERDs have been widely implemented on mines in the past, a limited amount of prior studies investigated the optimal integration of such systems into the rest of the water reticulation system of mines.

Notable studies conducted in the field of ERD control optimisation were conducted by Vosloo [38] and Janse van Vuuren [39]. However, the studies are relatively outdated. Recent developments in the field of energy management are therefore not included in these studies.

Due to the lack of research on the optimisation of mine dewatering systems that make use of ERDs, state of the art methodologies to optimise mine dewatering systems that utilise ERDs are not available. More detail on the previous studies conducted in the field as well as developments in the field of energy management are provided in Chapter 2 of this study.

## 1.4 Need and objectives for this study

The mining industry is under economic strain. In recent times, labour and electricity costs have been increasing. The rise in production costs decrease the profitability of mines. Mines therefore, need to reduce costs wherever possible. The literature showed that South African mines are some of the most energy-intensive in the world.

The inefficiency of South African mines means that the rising electricity costs are of particular concern to the industry. DSM projects were identified as an effective means of reducing electricity costs on deep-level mines. In the current South African energy climate, implementing similar projects outside of the DSM program can therefore also be beneficial to mines.

One of the significant energy consumers of deep-level gold mines in South Africa is the dewatering system. ERDs enhance the efficiency of the dewatering systems on mines. To minimise costs on systems that already utilise ERDs optimal integration within the rest of the dewatering system has to be ensured. Optimisation studies have been conducted on mine dewatering systems that have utilised ERDs in the past.

However, shortcomings in the literature have been identified. Important factors were not included due to the age of the available studies or need to be elaborated. New energy management tools and strategies could therefore offer opportunities for additional improvements if incorporated into existing processes.

A comprehensive methodology for optimising mine dewatering systems that utilise ERDs has to be developed. This methodology needs to be relevant within the current energy landscape of the South African mining sector.

The methodology will aim to:

- provide an evaluation process that allows for identification of possible initiatives to be implemented and effectively prioritised;
- offer an optimisation procedure for minimum energy costs; and
- deliver strategies to ensure successful implementation of the proposed initiatives.

Note that this study will not consider the addition of ERDs into the dewatering systems. The study will focus specifically on systems that already makes use of ERDs and especially co-generative ERDs as classified earlier.

## 1.5 Overview of dissertation

### Chapter 1

In Chapter 1, the background of the identified problem was provided. The background focused on the broader energy environment of South Africa and specifically the deep level mining industry. Dewatering systems were identified as a significant consumer of energy on such mines. It was shown that ERDs offer significant opportunities to save energy on such systems. Lastly, it was shown that a need for an optimisation strategy for dewatering that incorporates such devices exists.

### Chapter 2

In Chapter 2, the literature review is discussed. The literature review consists of three main sections. The first looks into evaluation techniques used on energy systems and mine dewatering systems in particular. The second consists of optimisation techniques and particularly simulation as an optimisation tool. The third investigates practical implementation strategies for energy projects on mine dewatering systems, with a particular emphasis on control of such systems.

### Chapter 3

In Chapter 3, the optimisation methodology is presented. The methodology was derived from the available literature. After the methodology was presented, it was verified. The verification process consisted of simulation of an already optimised mine. System restrictions were added in the simulation, after which the proposed strategies were used to overcome these restrictions within the simulation.

### Chapter 4

In Chapter 4, the proposed methodology was implemented on an actual mine. A complete evaluation of the system was conducted. After this, a simulation model was constructed, and an optimal control strategy was derived. The proposed strategies were validated by testing the proposed control on the mine. The results of these steps are shown and discussed in this section.

### Chapter 5

In Chapter 5, the study is summarised and the necessary conclusions were drawn. Recommendations for future work were also made in this section.

# Chapter 2: Review of mine dewatering system optimisation



[2]

This chapter provides an overview of the present state of the art on mine dewatering and analysis of energy systems. This literature review serves as the basis for the methodology and implementation phases that follow in the subsequent chapters.

[2] Highlands mining supplies, "Multistage pumps," Highlands mining supplies, 2017. [Online]. Available: http://www.highlandssupplies.com/pump-solutions/multistage-pumps. [Accessed 19 Nov 2017].

## 2.1 Preamble

In this chapter, literature relevant to the objectives of the study is discussed. The literature review is broken up into three main sections. Each section investigates means of addressing the identified objectives. The first objective of this study is to provide an effective evaluation strategy for mine dewatering systems that make use of ERDs.

The first section of the literature study therefore investigates data gathering techniques. Energy audits and the outcomes of energy audits such as system baselines and preliminary identification of potential energy cost-saving initiatives are also discussed in detail.

Secondly, optimisation strategies are considered. The study focuses on simulation as an optimisation tool in particular. Simulation strategies used on water and more specifically dewatering systems that utilise ERDs was reviewed. Simulation criteria and strategies used in previous studies are discussed in detail.

Lastly, implementation strategies were researched. The appropriate procedures to follow when implementing the developed solutions in general and particularly on dewatering systems was identified and discussed. Typical control requirements for systems that utilise ERDs were also investigated.

Note that certain studies discussed in the literature study include aspects that are relevant in more than one of the sections mentioned above. The relevant parts of such studies were therefore discussed and summarised in the applicable subsections.

## 2.2 Evaluation of cost-saving opportunities

### 2.2.1 Introduction

The first objective of this study is to develop an effective means of evaluation mine dewatering systems. In order to evaluate a thermal system, a specific set of information is required. Therefore, a means of obtaining such information is required.

This section of the literature review will start by investigating general energy analysis procedures. The required outcomes of the processes will then be identified. After this, studies that investigated and evaluated mine dewatering systems and energy recovery systems in particular will be discussed.

### 2.2.2 Energy audits

The first phase of any energy system investigation is typically an energy audit. The ISO 50001 standard is an internationally recognised standard for energy management [3] and provides guidelines for energy audits [27]. The process proposed for energy audits by the ISO 50001 standard consists of the following five steps [40]:

- Pre-audit questionnaire
- Analysis of the process
- Further auditing directions
- Implementation
- Audit report

A study conducted by Kluczek & Olszewski [27] showed the results of implementing the procedure presented in the ISO 50001 energy management standard on six case studies across various industries. The case studies considered by Kluczek & Olszewski [27] led to energy efficiency improvements on the systems considered ranging from 10% to 70%.

This shows the effectiveness of the process. However, the process presented in the ISO 50001 standard is generic. The IS0 50001 process will therefore, have to be customised and adjusted to suit the needs presented in this study.

Beggs [41] also presented energy audits as an approach to obtaining information on and understanding of energy systems. Beggs [41] identified the following critical outputs of energy audits within the context of this study:

- baselines of the typical energy usage and energy streams of the applicable system;
- general energy practices of the site including layout drawings and operational schedules;
- the conditions of the site equipment; and
- opportunities for energy management that leads to reduced costs.

Each of the identified outcomes of the system energy auditing phase relative to the study will therefore have to be addressed. Determining the state and condition of the equipment is essential. However, detailed investigations into the state of equipment can be cumbersome and do not fall within the scope of this study.

The general control practices, drawings and layouts are also important but should be obtainable from interviews with site personnel. Strategies to obtain these will therefore not be discussed in detail. However, system baselines and the identification of energy management opportunities falls well within the scope of this study. Specific steps for these phases of the evaluation will have to be investigated.

### 2.2.3 Mine dewatering equipment and process parameters

Mine dewatering systems consist of many different pieces of equipment. As shown by Kluczek & Olszewski, fundamental engineering knowledge of these components is essential [27]. The equipment and system components vary depending on the specifics of the mine.

A typical dewatering system includes dams, pumps, settlers, valves, spot coolers and ERDs. Another piece of equipment relative to this study is variable speed drives (VSDs). Typical instrumentation found on mine dewatering equipment includes power meters, flow meters, pressure transducers, dam level sensors [15].

An overview of mine dewatering and detailed explanations of the different ERDs relevant to the study was provided in Chapter 1. This study focuses on system optimisation and not the individual components.

The measurements obtained from the sensors discussed are used for multiple purposes. These can include condition monitoring, performance monitoring and control [15]. As will be shown in this literature review, the measurements obtained from such instruments will serve as the basis for any analysis done on the system.

However, not all of the measurements taken by mines are necessarily relevant to the study. To save time, it is therefore important to consider only the relevant data. A study conducted by Maré [13] proposed lists of specific parameters required for accurate predictions on a mine cooling system. The study focused mainly on developing procedures to analyse any water system on mines [13] and will be discussed in more detail later in the study.

However, the specific parameters are relevant to the study and more specifically energy audits of dewatering systems. These parameters will therefore be shown in Chapter 3 where the full methodology is finalised.

The parameters identified by Maré are extremely important for the purposes of this study and can be obtained in many ways. Kluczek & Olszewski recommended the following tools and sources for obtaining and analysing the available data [27]:

- site measurements and data acquisition systems;
- fundamentals of thermodynamics; and
- relevant software.

Note that mines do not always have instrumentation available. Inaccurate measurements are also common. The main reasons for the inaccurate measurements are lack of maintenance and harsh underground conditions [15].

It should also be noted that even though the analysis of the data logically occurs after obtaining it through measurement, the analysis might highlight inaccurate measurements. This means that the process can become repetitive. If faulty measurements exist, additional verification of the available data accuracy might be required [41].

Manual spot checks are an effective means of data verification. The recommended measuring equipment for the auditing process by Kluczek & Olszewski includes the following [27]:

- pressure and temperature loggers;
- power loggers; and
- ultrasonic clamp flow meters.

### 2.2.4 Baselining

After the required data has been obtained, it is typically processed into system references known as baselines [42]. Since the baselines make use of available data, it has to be constructed after the data is obtained.

Energy or power baselines have been extensively used as part of industrial DSM projects [43]. Vermeulen et al. [43] differentiate between two main types of baselines. These are profiled energy or power baselines and single variable baselines. Profiled baselines indicate the cumulative energy or power usage for each relevant TOU period. Single variable baselines, in turn, is merely the sum or average power or energy usage over the baseline period [43].

The best practices involved in setting up such baselines are however, still required. Typically a data set of at least three months is recommended. If repetitive behaviour such as variance in ambient conditions occurs on the system, the baseline period should be extended to accommodate these changes [44].

Additionally, baseline data should be representative of the system, and it is therefore essential to filter out any outliers or data points that could lead to misrepresented baselines [42]. Tools used in the analysis of such datasets include [42]:

- regression modelling; and
- statistical variance analyses.

Booysen [42] discussed regression modelling as a part of baseline development. Key parameters identified as part of typical measurement and verification (M&V) practice is the coefficient of determination ($R^2$), and the root means squared error (RMSE) [42]. Note that most of the baseline development techniques discussed so far are primarily used for energy and power baselines. These baselines serve as a reference point against which project performance is measured.

Analysis of the variance of the data is another effective means to analyse data. This method is a common tool in statistical analysis and has been used as part of metering and verification procedures in the past [45].

Booysen [42] developed an easy to use, practical processes to develop baselines for industrial DSM projects. The two most basic baseline models as described by Booysen is the constant baseline and the energy-neutral baseline [42].

The constant baseline never changes and therefore assumes consistent behaviour of the system. The energy-neutral baseline assumes that system changes affect the average energy or power usage of the system and scales the constant energy baseline according to a constant factor. The factor is typically the ration between the assessed period's average energy usage and that of the constant baseline [42].

However, understanding how to use the different types of baselines is of utmost importance. Smith et al. [46], Janse van Vuuren [39] and Oberholzer [29] used energy neutral baselines for performance measurement on mine dewatering load management projects. The control on these studies did not affect the energy usage of the system, making energy neutral scaling of baselines appropriate.

Schoeman [47] and Botha [48] used an alternative form of baseline scaling. These studies focused on water consumption reduction during low demand periods. The baselines in these projects were therefore scaled to the usage during high demand periods when the flows were not affected by the changes to the control.

It is therefore clear that if the initiaitive does not influence the average energy usage of the system, energy neutral scaling is applicable. It is also clear that for energy efficiecny initiaitves, fixed baselines or baselines scaled to periods where certain parameters are not affected is approriate.

The aim of this study is to integrate existing efficiency strategies in the form of ERDs with other potential initiatives. Since various types of initiaitves will be investigated, a completely generic scaling methodology cannot be recommended for this study. The engineer will therefore have to select an appropriate scaling technique which considers the above mentioned criteria.

The baselines discussed thus far focuses on the energy or power usage of the systems. However, logic dictates that similar baselines can be set up to use as a reference for other critical parameters, such as water flow, that applies to this study.

Therefore, even though these studies did not focus on such parameters per se, the techniques and principles discussed can be applied to any set of data. After the data has been collected and processed into baselines, it can be used to determine if any cost-saving potential exists on the system.

Fundamental equation to determine the various statistical parameters required for the baselines can be found in Appendix B.

### 2.2.5 Load management on mine dewatering systems

Both load management and energy efficiency initiatives have been implemented on mine dewatering systems that make use of ERDs in the past. These initiatives are therefore important considerations for any optimisation study on mine dewatering systems.

Load management is an electricity cost savings initiative that has been widely applied to dewatering systems [49, 50, 51]. As explained in Chapter 1, load management entails shifting energy usage out of peak demand periods as billed in a TOU tariff structure to off-peak periods, thereby saving electricity costs.

Equation 6 can be used as a preliminary check to determine if any load management potential exists on a given system [18].

$$R_{LS} = \frac{\bar{P}_{LS}}{\bar{P}_{ave}} \qquad \text{Equation 6}$$

Where:

$R_{LS}$ = The load shift ratio

$\bar{P}_{LS}$ = The average peak period power usage of the dewatering system

$\bar{P}_{ave}$ = The average power usage of the assessed period

A relatively high load shift ratio (value greater than one) would indicate that significant potential for load management exists on the analysed system. A load shift ratio of zero, in turn, would indicate that no potential for load management exists on the analysed system [18].

However, the possibility of load management does not necessarily mean that it is feasible. From the available literature, it is evident that two main criteria exist for load management on mine dewatering systems. These are:

1) Spare dam capacity [18]; and
2) Spare pumping capacity [51].

Sufficient spare capacity needs to be available in the dewatering dams of the mine to allow the dams to fill up during the load management period gradually. The maximum pumping capacity, in turn, needs to be sufficient to make the spare capacity available before the load management period [18].

Studies investigating load management on mine dewatering systems have been conducted in the past. De Jager [15] developed a methodology for quantifying the effect of pump unavailability on load management performance. However, although ERDs are briefly discussed, the study focused mostly on conventional pumping systems.

Stols [52] investigated the effects of various system constraints on the performance of mine dewatering systems. These constraints included lack of spare dam capacity, lack of spare pumping capacity and excessive water demands. All of these factors were found to have an adverse impact on the load management performance of the system. However, the relationship of these constraints to systems that utilise ERDs were not investigated.

Groenewald et al. [53] investigated means of improving the sustainability of load management DSM projects. A mine that utilises a 3CPFS was used as a case study. Groenewald et al. [53] found that the unavailability of the 3CPFS at certain times led to reduced load management performance. The reason cited for this was that it reduced the total pumping capacity and therefore the spare pumping capacity [53].

However, the study conducted by Groenewald et al. [53] did not specifically focus on the 3CPFS. Although it was clear that the 3CPFS had a positive impact on the potential for load management and therefore the potential for additional energy cost savings, the exact relationship was unfortunately not adequately discussed [53].

Vosloo [38] and Janse van Vuuren [39] conducted studies that investigated load management on dewatering systems that utilise ERDs. The studies showed that load management on systems that make use of ERDs are possible. However, these studies did not discuss the potential constraints that ERDs can incur.

All of the above-mentioned studies relied on real time scheduling of the dewatering equipment. Considering such strategies is therefore a proven technology and should be incorporated into any optimisation strategy.

Tang et al. [50] investigated the effect of incorporating variable flow strategies as part of a conventional dewatering system for improved load management and efficiency. However, incorporating such strategies into systems that utilise ERDs was not discussed [49].

Botha [36] and Janse van Vuuren [39] both mentioned that VSDs are sometimes installed on the auxiliary equipment of 3CPFSs. However, Botha's study mainly focused on the challenges associated of commissioning the system. He therefore did not consider optimising the system [36].

However, given the nature of the systems, VSDs can be applied to vary the flow to aid in load management initiatives on ERDs such as 3CPFSs and closed-loop high-pressure u-tube systems. Such strategies should therefore also be considered for cost optimisation through load management.

### 2.2.6 Energy savings initiatives on dewatering systems

ERDs have been discussed as a means of increasing the efficiency of mine dewatering systems. However, this study focuses on systems with ERDs which are already installed. Additional ways of achieving energy efficiency on mine dewatering systems also exist and will be discussed.

An effective means of reducing the energy demand of mine dewatering systems is improved water management [54]. In deep-level gold mines, projects that focus on reducing the water demand, and thereby the total energy usage and costs, are known as water supply optimisation (WSO) projects [47].

Typical initiatives to reduce the water demand on mines include [48]:

- leak reduction;
- stope-isolation; and
- pressure control.

Leak reduction involves identifying and fixing any leaks that may exist on the mine. The reduction of leaks decreases the amount of water that needs to be sent underground and therefore pumped out of the mine [48].

To understand stope isolation and pressure control into mining levels, it is necessary to understand normal mining schedules. For this study, it is essential to take note of two specific periods in a typical mining schedule.

Figure 7 shows the typical flow profiles into a mining section that results from the mining schedule described.

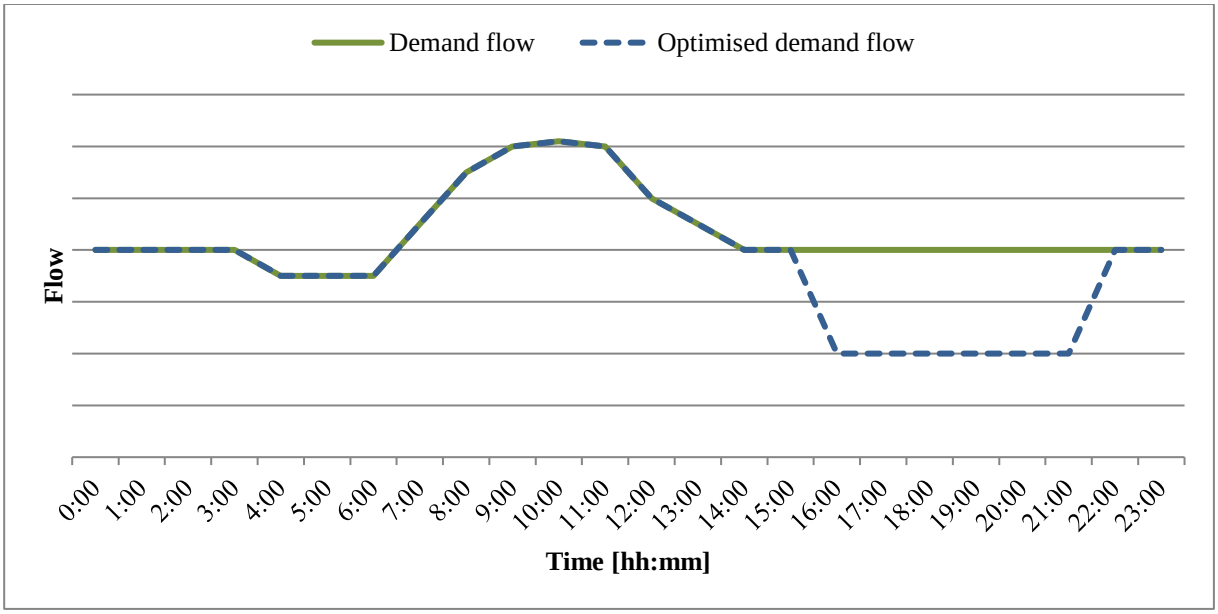



**Figure 7: Typical mine demand flow (Adapted from [47])**

The first applicable mining period in a typical schedule is the drilling shift. During the drilling shift, the rock faces in the work areas (commonly referred to as stoping areas) are prepared for blasting. This is usually accompanied by underground activity and forms the maximum demand period [47].

The second relevant period is the blasting shift. During blasting, the flow demand in the working areas is low. The reason for this is that the stoping areas have to be evacuated during the blasting period. This means that there is minimal activity in the mining areas during this period [48].

A WSO initiative aims to control the water strictly to the demand of the mine. This leads to minimum wastage. Leak reduction reduces the demand throughout the day. However, the focus of stope isolation and pressure control into the mining levels are typically isolated to the blasting shift which is the lowest demand period [47].

Stope isolation entails installing shut-off valves at the inlet of individual stopes during blasting shift. The water flow is therefore reduced since any wastage of water during such periods is minimised [48].

Pressure control into mining areas also involves reducing the flow into mining areas during the low demand period. However, the means of the control differs. Pressure control typically entails installing a by-pass valve on the primary pipe network into a mining section. The by-pass valve

will typically have control capacity. In addition to this, the pipe sections leading to and from it is smaller than the main pipeline [47].

During low demand periods, the main pipeline feeding into the mining level is shut off and the by-pass valve used instead. The valve in this by-pass pipeline is then used to reduce the water pressure into the mining section. By reducing the pressure, the flow into the working areas during low demand periods are reduced. This means that any leaks or unattended equipment would waste less water during low demand periods [47].

A quick preliminary check to identify such opportunities is the ratio between the peak demand flow and the flow during low demand periods. Equation 7 can be used to calculate this.

$$R_{WSO} = \frac{F_{low}}{F_{peak}} \qquad \text{Equation 7}$$

Where:

$R_{WSO}$ = The ratio between the low demand period and peak demand period

$F_{low}$ = The flow during the low demand period

$F_{peak}$ = The flow during the peak demand period

A ratio of one or larger would indicate definite wastage. A ratio of zero, in turn, would suggest that no wastage exists on the system.

Schoeman [47] investigated the integrated effect of pressure control into the mining levels as a means of energy and therefore energy cost savings. He found that significant reductions in the energy usage of both the refrigeration and dewatering systems of a mine could be obtained. However, his case studies did not include ERDs [47].

In a different study conducted by Maré et al. [55], the integrated effect of variable flow strategies and pressure control into mining areas were investigated. A combined average power saving of approximately 3.3 MW was found to be achievable [55]. Flow reduction into the mining areas is therefore clearly an effective means of reducing the total energy usage of mines.

Pressure control through the use of valves is therefore clearly appropriate for conventional supply systems. However, ERDs such as 3CPFSs and closed-loop high-pressure u-tube systems are also involved in the water supply side. Unfortunately, studies that focused on the optimisation of mine water supply systems through the use of VSDs could not be found.

However, the technology has been proven for pressure control in residential areas [56]. The uses in residential areas include minimising pumping energy requirements, regulating pressure for preventing and reducing leakages, and controlling water quality [56].

Since the control within the mining water supply system is not for potable water, the water quality is not of primary concern. Additionally, given the nature of both the closed-loop systems and 3CPFSs, pressure control through the use of VSDs seems infeasible. The ERDs are designed to make use of the available pressure head [34, 36] and the pressure reduction capacity will therefore be limited.

However, electrical saving on the pumps itself through reduced flows could still be feasible. This electrical saving could likely be achieved by the supply pump itself and similar to traditional WSO projects, lead to integrated electrical energy reduction due to reduced flows.

The available literature shows that WSO projects on conventional water supply systems have a positive effect on the power usage of the entire mine refrigeration system. The available studies indicate that it is an effective means of increasing the efficiency of dewatering systems. It also seems apparent that newer technologies such as VSDs could be incorporated into the supply optimisation strategies in mines to achieve similar end goals.

### 2.2.7 Integration of efficiency and cost-saving initiatives on mine dewatering systems

Janse van Vuuren [39] investigated a means of optimising the savings potential of a new 3CPFS. However, only load management was considered. Additionally, Janse van Vuuren mentioned VSDs on ERDs, but the use of such technologies was not discussed sufficiently to be included in any methodology [39].

Vosloo [38] proposed a methodology for optimising a dewatering system for minimum cost. The integration of WSO initiatives and load management was considered. It was found that by minimising the demand flow, a minimum cost model for mines could be obtained. Both case studies used for validation of Vosloo's method had ERDs installed as part of the system. However, the use of variable flow strategies were not considered by Vosloo.

It should be noted that the studies conducted by Vosloo [38] and Janse van Vuuren [39] are both relatively old. No newer studies that focused explicitly on the optimal use of ERDs in mines could be found.

Recent trends in the field of energy efficiency show increased use of variable flow strategies as a means for improved energy efficiency and energy management [50, 24, 57, 56]. The studies discussed either did not include or inadequately explained utilisation and the role of such technologies as part of the optimisation possibilities of dewatering systems that incorporate ERDs.

The previous studies focused mainly on scheduling the existing mine dewatering equipment. In certain studies such as that conducted by Vosloo, ERDs were also scheduled. These technologies will therefore be considered as part of the newly developed strategies proposed by this study.

### 2.2.8 Summary

A means of evaluating energy systems and more specifically dewatering systems on mines was researched. The basic procedures and outcomes of energy audits were discussed and evaluated. Load management and WSO were identified as potential initiatives for reducing electricity costs on mine dewatering systems. Integration of such initiatives into systems that utilise ERDs were also considered.

It was shown that existing strategies such as scheduling, pressure and flow control could be beneficial to systems that utilises ERDs. Scheduling of equipment including ERDs within these systems has been shown to be feasible. However, it can be concluded that newer technologies such as VSDs should also be considered for optimisation of dewatering systems that utilises ERDs.

## 2.3 Modelling and optimisation

### 2.3.1 Introduction

It has already been shown that mine dewatering systems that make use of ERDs consist of many components. These components will typically have a dynamic effect on each other. Quantifying the effect of identified initiatives on such systems can be cumbersome. However, tools to analyse such systems in detail have already been developed and shown to be effective in past studies.

The available literature indicates that a particularly powerful tool in the analyses of mine dewatering systems is simulation [55, 47, 38, 39]. However, the field of simulation is vast. Therefore, some basic background of simulation will be provided. Studies that focused on the simulation of mine dewatering process and particularly studies that incorporated the simulation of ERDs on such systems will also be discussed.

### 2.3.2 Simulation model purpose

Simulation is a process that aims to predict the behaviour of a system by creating an approximate model of it. These models are usually logical or mathematical. A thermal hydraulic simulation, in turn, simulates variables such as temperature, pressure and flow [58].

An important concept in evaluating systems, as found on mines, is that of an integrated system. In an integrated system, a specific system component affects or potentially affects all of the other elements in the specified system. This effect can be either delayed or immediate [58].

Another important concept is that of dynamic simulation. In a dynamic simulation as opposed to steady-state simulation, system variables and inputs change relative to time. Such simulations can therefore, deliver a very accurate and realistic description of real-world systems. Such simulations are of particular help when considering control on systems and the effect of changes in control of system components [58].

### 2.3.3 Simulation tool criteria

Multiple studies on mine dewatering systems with an active element of simulation have been conducted in the past [34, 59, 13, 38, 39]. Various simulation tools that have been proven to be useful in mine cooling systems exist [13]. Studies in other fields such as water distribution system optimisation and pump station optimisation also exist [57].

This study aims to provide techniques to integrate energy management strategies to existing efficiency strategies optimally. Load management and pressure control have been identified as additional strategies. These strategies involve control changes of the system. The constructed models will therefore have to consider the integrated and dynamic effect of the dewatering system.

However, before previous studies can be investigated, existing models for dewatering equipment and turbomachinery need to be investigated. Kalaiselvan et al. [57] considered multiple aspects of pump station simulation. Multiple studies were considered and the following basic strategies to simulate centrifugal pumps were considered.

1) Curve fitting as proposed by Chang et al. [60] and Izquierdo et al. [61].
2) Numerical interpolation proposed by Ahonen [62].
3) Pump approximation proposed by Olszewski [63].
4) Dynamic pump model proposed by Campana et al. [64], Bouwer [58] and Maré [13].
5) General pump model proposed by Yang & Borsting [65].

The curve fitting method requires multiple inputs. Among these are samples of the pressure head, flow rates as well as a full set of pump performance curves. Additionally, the rotational speed of the pumps is required to simulate the pumps in such a manner [57].

The numerical interpolation method sets up a flow vs pressure head curve. The values for the flow and applicable pressure head delivered by the pumps is derived from the Bernoulli equation and the manufacturer pump curves. After this, pump affinity laws are used to estimate the pump performance curves at various speeds [57].

Pump approximation determines the power or flow outputs of the pumps using the characteristic curves as inputs. However, detailed tests of the motor frequency and other pump coefficients have to be measured during the pump start-up [57].

The dynamic pump models also make use of characteristic pump curves. However, the curves are based on power, head and flow obtained during field tests. These type of models typically also use the pump affinity laws for variable flow applications [57].

Lastly, the general pump model characterises the pump outputs as a function of the pump speed. However, specific details of the dynamic nature of this type of model is not provided [57].

The purpose of the model types within the study is of utmost importance. The study will investigate the effects of pumps and other turbomachinery such as turbines within an integrated dewatering and water distribution system. The aim of the study is not to optimise individual pieces of equipment. Simplified models of the individual components should therefore suffice.

Existing systems will be modelled in this study. Therefore, models based on field tests should be best suited. This means that the curve fitting method proposed by Chang et al. [60] and Izquierdo et al. [61], as well as the numerical interpolation [62] is not applicable since these methods characterise the equipment to manufacturer specifications.

The pump approximation and general pump model is somewhat more complex than the dynamic pump models proposed by Campana et al. [64], Bouwer [58] and Maré [13]. Such dynamic models require only measured pressure, flow and pump power. Minimal further investigation is therefore required for such models which makes it applicable to this study.

The dynamic capacity of the pump models is important since variable flow strategies will be considered. The studies considered thus far only simulated pumps. However, it is clear that the use of dynamic models that requires minimal measureable inputs are therefore suitable for other components as well.

The models should also enable the dynamic integration of turbomachinery such as pumps and turbines into the rest of the water system. Studies that modelled entire mine water systems will therefore have to be considered.

### 2.3.4 Integrated simulation of mine dewatering systems

Du Plessis et al. [66] and Bornman et al. [67] constructed integrated dynamic simulations of mine cooling systems. However, the simulations focused on the surface refrigeration system. Optimisation of the dewatering system was not considered for these studies.

De Jager [15] proposed a simulation procedure to quantify the effect of pump unavailability on dewatering system load management performance. However, although the simulations were integrated, the pump models used were not dynamic since the application did not require it.

Van Antwerpen et al. [68] constructed an integrated dynamic simulation of the world’s deepest mine. The results showed that simulating systems of the scale found in the world’s deepest mine is indeed possible. However, it was mentioned that the study of the system was time-consuming and tedious.

Bouwer [58] and Maré [13] also proposed integrated simulation strategies for mine water systems. Given the time consuming and computational requirements of these simulations, the models used in these studies used only explicit component models [13]. These studies indicated that through the use of such models, the dynamic integrated effect required by the study can still be simulated but with less simulation and resource time [13].

Additionally, Maré [13] proposed models for various ERDs as well as water distribution and dewatering systems. The models were shown to be valid by Maré. However it was not incorporated as part of dewatering optimisation strategies [13].

Given the dynamic nature of the systems considered, integrated and dynamic models as proposed by Maré [13] and van Antwerpen et al. [68] should be used. Such models allows for realistic simulation of all of the equipment.

It seems apparent that simulation models that can be used to achieve the objectives of this study exist. However, before specific outcomes of the study can be investigated, it is vital to consider simulation procedures.

### 2.3.5 Simulation procedure

Bouwer identified the following basic simulation procedure [58]:

1) Draw a process flow diagram of the appropriate system that indicates all relevant configurations and measuring points.
2) Obtain measurements of all the relevant process parameters (flow, pressure and power etc).
3) Manually measure lacking information.
4) Select a typical operational day for simulating.
5) Consider periodic changes such as changes in the weather.
6) Set-up the simulation.
7) Compare the obtained data with the simulation outputs.
8) Iterate the process until an acceptable degree of accuracy is obtained.

The simulation accuracy and reliability is therefore verified by comparison to the actual data obtained. This is an important step in any simulation initiative. Such a simulation is also commonly referred to as a calibrated simulation [13].

Note that more detailed processes to obtain the required data (steps one to five) have already been discussed in Chapter 2.2. The rest of this section will therefore focus primarily on detailed procedures to set up the simulations. The use of the simulations as part of an optimisation strategy will also be discussed.

Maré [13] proposed a procedure to simulate any given system dynamically. The proposed procedure is presented in Figure 8.

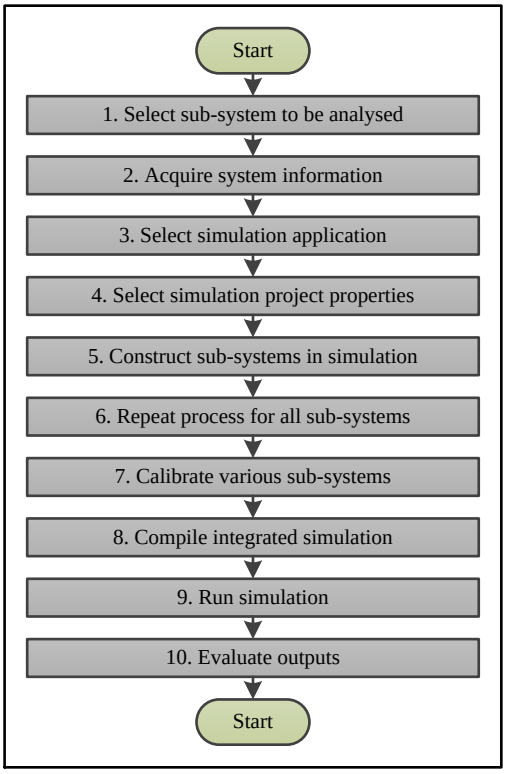



**Figure 8: Integrated simulation set-up procedure (Adapted from [13])**

For this study, the subsystems that need to be included is set. These subsystems are:

- the pumping system;
- the general water reticulation system; and
- water into the mining areas.

The calibration procedure proposed by Maré [12] also entails comparing the outputs to the actual data. Maré proposed calibrating the simulations to within an accuracy of 10% [13]. The applications as identified by Maré relevant to this study include identification of initiatives and control emulation [13].

The simulation strategies proposed by Maré [13] and Bouwer [58] are clearly applicable to the study. However, in order to use the simulations, specific outcomes of the simulation need to be set.

### 2.3.6 Optimisation of ERDs on dewatering systems

Studies that focused on the optimisation of dewatering systems that contains ERDs, in particular, are scarce. However, the literature that could be obtained will now be discussed.

Rautenbach [59] conducted a study in which he developed a 3CPFS simulation component for the dewatering system of a mine was analysed and optimised for maximum cost savings on the system. However, the possibility of additional efficiency improvements and variable flow strategies were not considered.

Janse van Vuuren [39] conducted a similar study to that of Rautenbach [59]. In this study, a real time energy management (REMS) simulation was used to optimise the saving potential of a new 3CPFS. He found that the maximum saving potential of a new 3CPFS could be obtained on mines with a high flow demand. As with Rautenbach's [59] study, additional efficiency improvements were not considered.

Vosloo [38] conducted a study into the optimisation of mine water reticulation systems. In his study, integrated simulations of the system was set up in REMS. However, careful analysis of the methodologies used indicates that the simulations were not genuinely dynamic.

The studies conducted by Vosloo [38] and Janse van Vuuren [39] are the most relevant to this study. Different optimisation criteria were used in the studies. Janse van Vuuren [39] optimised his system for maximum electricity cost savings. Vosloo [38] proposed using minimum cost as the optimisation criteria.

Note that at first glance, these criteria may seem similar, Janse van Vuurens [39] study focussed only on load management strategies and used scaled baselines. This study will consider efficiency and load management. If maximum savings is used in unison with energy neutral scaled baselines, the biggest savings potential will exist when the system is using maximum energy, which will not necessarily lead to minimum costs.

Since the efficiency of the system is of paramount importance to the study, Vosloo's [38] criteria of minimum cost, with a fixed baseline is appropriate. The dewatering systems will therefore be optimised for minimum electricity cost in this study. Note that if results of load management initiatives that do not influence the efficiency of the system is compared, energy neutral scaling is still appropriate. But, such scaling should not be used when optimising the system.

### 2.3.7 Summary

Simulation models used in the analysis of mine dewatering systems were investigated. Analyses of the available models showed that dynamic models that utilises measurable inputs are applicable to this study. It was also shown that these models should be combined into an integrated simulation in order to simulate the effect that the various components have on each other.

Additionally, studies that aimed to optimise mine dewatering systems that incorporate ERDs were investigated. Optimising the system for minimum cost was identified as the best practice when analysing mine dewatering systems.

## 2.4 Control and implementation strategies

### 2.4.1 Introduction

Means for evaluating and optimising mine dewatering systems have been investigated. However, appropriate ways of implementing initiatives still need to be researched. The rest of this chapter will focus on such strategies.

WSO and load management initiatives on mine dewatering systems can be implemented in many different ways. As was identified in the previous sections of this literature study, the aim of such initiatives should be to derive a state of operation leading to minimum costs for the mine. To ensure this, the system changes should be sustainable and offer long-term benefits for the mines [51, 12].

Therefore, measures such as automation, reporting and appropriate implementation procedures will be discussed in detail in this section of the literature review.

### 2.4.2 Pump system automation

To provide safe underground operations, proper control of the dewatering system of any mine has to be ensured. To control the dewatering system components on deep-level gold mines, extensive communication networks are required. In modern mines, the base communication between mine system components and instrumentation occurs through programmable logic controllers (PLCs) and extensive computer networks.

The dewatering instruments and equipment are typically connected directly to PLCs. The PLCs, in turn, communicates with a supervisory control and data acquisition (SCADA) system through an open platform communication (OPC) connection. This type of connection allows for reading and writing commands to be communicated between the system components [24].

Using such commands, the dewatering systems of modern mines can be controlled from a central point. It also allows for a holistic analysis of the entire system. SCADA systems typically also have data logging capacities which allow historical trends of the various measured process parameters to be set up.

Note that a critical aspect investigated in previous studies is the importance of control on such projects. Groenewald differentiated between automated control and semi-automated control [51]. Another study conducted by Smith [46] compared the sustainability of automated DSM initiatives and manual initiatives. It was found that the automation of dewatering systems led to the most sustainable long-term benefits to the mine.

Velleman et al. [69] also investigated means of pump automation on mine dewatering systems. The effects of using a consequent optimal control strategy were illustrated. It was found that the most frequent concerns from mining personnel are:

- risk of flooding mining levels; and
- short interval stopping and starting (cycling) of pumps.

The study found that an automated control strategy could reduce the cycling of the pumps if appropriately implemented. The use of appropriate alarms was also recommended to ensure that control operators can implement appropriate procedures if dam levels exceed the allowed limits [69].

Cilliers [18] investigated the effect of using profiled control parameters. It was found that additional load could be shifted out of standard Eskom TOU tariffs without hindering the load management initiatives that focus on peak times for additional cost savings [18].

These studies indicate that automated control on mine dewatering systems is indeed feasible. The methodology should clearly strive towards an automated control strategy. However, none of these studies considered ERDs as part of the mine dewatering system.

### 2.4.3 Scheduling of ERDs

The control on 3CPFSs and turbines coupled directly to pumps is somewhat more complicated. The reason for this is that at least four dams need to be considered when controlling these devices. This type of control is required since the device controls the supply water and dewatering process simultaneously. At least four dams will therefore, have to be considered simultaneously to ensure the device performs as it must [38].

However, ERDs complicate this control somewhat. The reason for this is that the cold water sent down the mine is fed through the ERD. Therefore, if the demand during peak times is relatively high, co-generative ERDs pumps the water while supplying it. It is therefore not always possible to switch the ERD off in peak hours [39].

The conventional approach of emptying the dam as much as possible before peak hours will therefore, have to be altered. Two solutions to this problem can be implemented [39]:

1) Set a higher minimum dam level
2) Reduce the flow through the ERD

Setting optimal dam levels can typically be achieved without any capital influx. Additionally, previous studies have explored this possibility in detail [18, 52, 46]. The control mechanisms as discussed so far, as well as the following control mechanisms will all aim to set optimal dam levels for load management.

Vosloo [38] used two case studies in his study to derive a minimum cost model for mine water systems. The first case study considered a system that contains turbine-pumps while the second considered a system that makes use of a 3CPFS. Through the use of strategies as listed above, effective load management was integrated with WSO strategies on the analysed mines.

Groenewald et al. [53] observed that planned shut-offs of an ERD in the form of a 3CPFS would have to be incorporated if the flow is sufficiently low. However, such an initiative was not incorporated as part of the study's control strategy [53]. Vosloo [38], in turn, set schedules for turbine pumps. However, the scheduling of 3CPFSs was not considered.

It is therefore clearly possible to integrate load management strategies into energy efficiency strategies through proper scheduling of dewatering equipment. However, the improved simulation strategies and newer technology in the form of VSDs could also provide benefits to the mines and will therefore have to be incorporated into existing strategies.

The last objective of the study is to provide implementation strategies. Striving for automated control is clearly the best practice. However, the literature also highlighted clear strategies that need to be followed when testing and implementing such control strategies.

### 2.4.4 Implementation phases

Joubert [12] developed risk management processes for South African energy service companies (ESCos). The processes developed by Joubert covers the entire project lifecycle of typical industrial DSM projects. Joubert developed the processes after considering the implementation details of 129 DSM projects.

The process proposed by Joubert assumes that the investigation and evaluation phase is already in place and that control strategies have already been developed. The process proposed by Joubert therefore consists of the following steps:

1) Install required hardware.
2) Configure any additional software.
3) Test control.
4) Soft commissioning.
5) Hot commissioning.

The third step entails the installation of the appropriate hardware. The fourth step entails configuring any software that will be used. After the required hardware and software is in place, testing of the developed control can commence. This phase is of particular importance to the study [12].

The processes proposed by Joubert is proven to be effective during the implementation phase of industrial DSM projects and can, therefore, be integrated into the required methodology. Joubert identified project maintenance as an additional requirement for the sustainability of such projects [12].

### 2.4.5 Sustainability of initiatives

Grobbelaar [70] investigated means of improving the sustainability of pumping DSM projects. The study focused primarily on the control and instrumentation parameters of such systems. Three main aspects of maintenance were considered in Grobbelaar's study. These are [70]:

1) Data system maintenance.
2) Mechanical maintenance.
3) Control maintenance.

After the implementation of the initiatives, effective reporting of the impact has been identified as an important project phase [51]. To create awareness of the initiatives, it is important to continually report on the progress and cost reductions from the implemented initiatives. The M&V process also requires this step as part of DSM projects [42].

### 2.4.6 Summary

Automation strategies on dewatering systems were discussed. The particular challenges faced when ERDs are present were investigated. The lessons learned from previous studies can therefore be incorporated into the required strategies.

Procedures for implementing initiatives on mine dewatering systems were investigated. Detailed project implementation steps were identified. Lastly, the importance of project maintenance and reporting was shown.

## 2.5 Conclusion

In this chapter, the available literature was reviewed with the aim of addressing the identified objectives. The literature review focused on energy audits, thermal hydraulic simulations and general implementation strategies.

Energy audits were shown to be an effective starting point for investigations of dewatering systems. Critical outcomes of such audits were identified. Additionally, considerations during audits for systems that utilise ERDs as part of the dewatering system was identified. The required outcomes and strategies were summarised.

Studies that investigated optimisation of mine dewatering systems in the past were identified. The methods and optimisation criteria used in these studies were evaluated. Simulation was identified as an effective tool in the detailed analysis of mine dewatering systems. Simulation model criteria were identified. Additionally it was shown that optimisation for minimum system cost is appropriate for this study.

Lastly, effective implementation strategies were investigated. Implementation procedures on industrial DSM projects were considered. The importance of automation, maintenance of initiatives as well as reporting of initiatives was shown.

# Chapter 3: Optimising energy recovery on mine water systems



[3]

In this chapter, a methodology to address the identified research problem is proposed. Critical aspects of the methodology is also verified within a simulation of a case study.

[3] Microhydro, "Types of Waterwheels and Turbines," [Online]. Available: http://www.microhydro.ie/section/TypesofTurbines. [Accessed 19 Nov 2017].

## 3.1 Preamble

Chapter 1 identified a need for a complete optimisation methodology on mine dewatering systems that makes use of ERDs. The main goal of the method is to minimise costs of mine dewatering systems.

Chapter 2 evaluated various factors required for mine dewatering optimisation. These included:

- electricity cost-saving strategies on mine dewatering systems;
- simulation models and practices; and
- automation of dewatering systems.

This chapter provides a complete optimisation methodology for deep-level gold mines which incorporates ERDs. This methodology makes use of an approach that integrates both energy efficiency strategies and load management initiatives.

The research conducted and summarised in Chapter 2 forms the basis of this methodology. The lessons learned and the process followed by other authors were incorporated to form a complete methodology to address the problem at hand fully.

The method includes summaries of the specific steps where applicable. Naturally, recognition of other authors' work is indicated. Where the methodologies found in the literature were lacking, new processes were developed.

Figure 9 shows the full breakdown of the methodology.

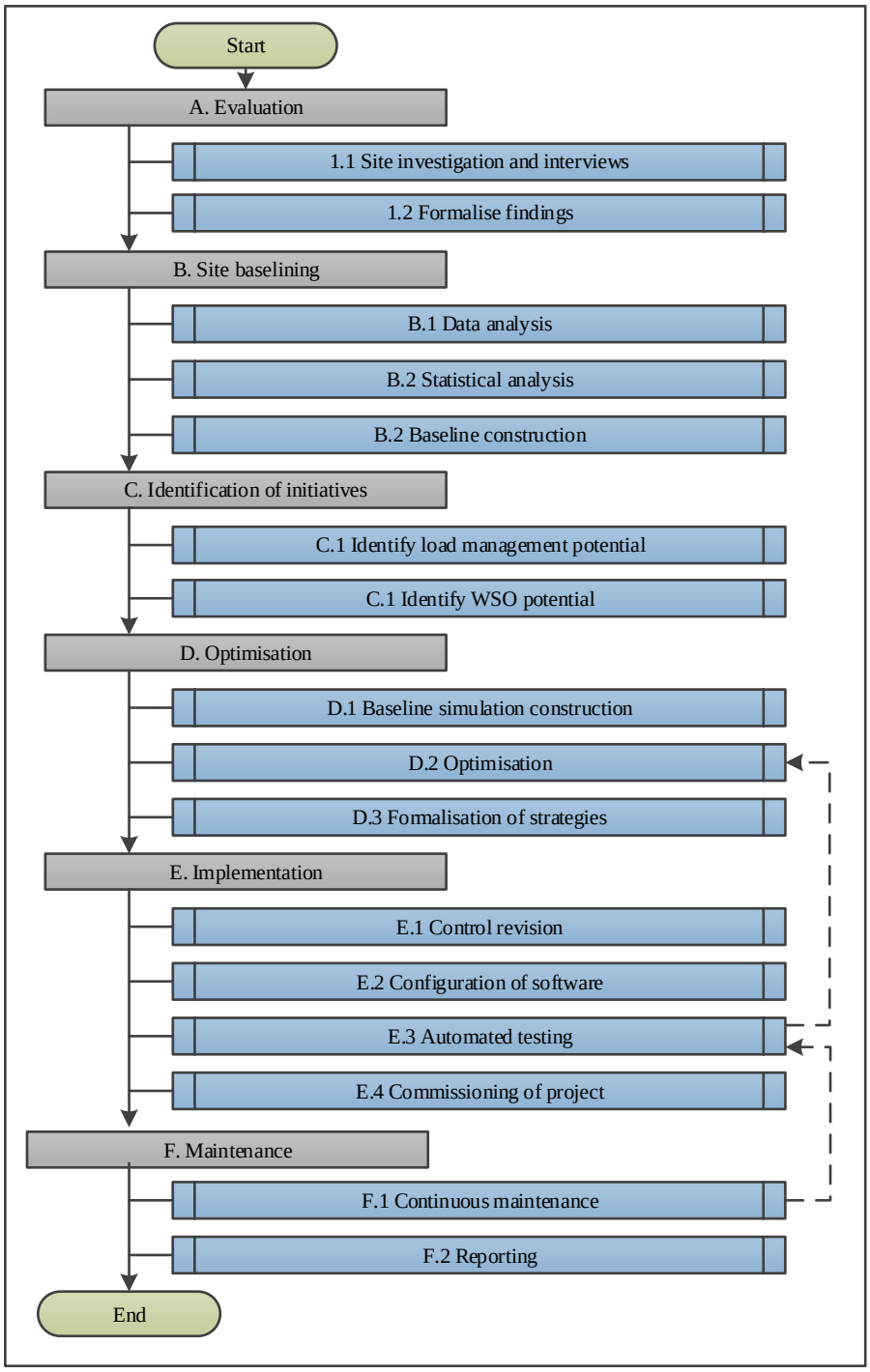



**Figure 9: Mine dewatering system optimisation methodology**

The subsequent sections of this chapter will explain the process depicted in Figure 9. Phase A - C addresses the objective of developing an evaluation procedure. Phase D describes the proposed optimisation procedure that was required, while phase E and F addresses the objectives for an effective implementation strategy.

## 3.2 Optimisation methodology

### 3.2.1 Site investigation

The literature showed the first step of optimising a mine dewatering system is a site investigation (Process A.1). The goal of the investigation is to obtain an understanding of the function of all the system components, as well as the interaction between the components.

Process A.1 will typically involve:

- a site walkthrough;
- detailed questioning of site personnel; and
- ensuring continuous access to the required data on site.

The critical outputs of this phase are:

- detailed process flow diagrams (Process A.2.1);
- equipment specifications (Process A.2.2);
- summarised site control practices (Process A.2.3); and
- historical data of critical system parameters (Process A.2.4).

Process A.2.1 entails the construction of a layout of the specific site. The available literature showed that a process flow diagram is an effective means of summarising the data obtained from this process. This diagram should include the relevant components and illustrate the specific interactions between the components.

Process A.2.2 and A.2.3 involves formalising the equipment specifications and present site control practices. The equipment specifications should be available from the relevant site personnel and should be verified during site audits.

Typical information required from Process A.2.3 includes:

- scheduling information (is the equipment allowed to be switched on and off regularly?);
- variable flow capability; and
- remote control capability.

Table 2 provides specifications required for the optimisation phase (Process A.2.3).

**Table 2: Critical equipment design specifications [13]**

| Equipment | Parameter | Mathematical representation | Unit |
|---|---|---|---|
| Pumps | Power | $P_{rat}$ | [kW] |
| | Flow | $Q_{rat}$ | [ℓ/s] |
| | Elevation | H | [m] |
| | Pump efficiency | $\eta$ | [-] |
| Dams | Volume | V | [Mℓ] |
| | Maximum level | $L_{max}$ | [%] |
| | Minimum level | $L_{min}$ | [%] |
| | Top elevation | $H_{top}$ | [m] |
| | Bottom elevation | $H_{low}$ | [m] |
| 3CPFSs | Elevation | H | [m] |
| | Flow rating | $Q_{rat}$ | [ℓ/s] |
| Turbine-pumps | Elevation | H | [m] |
| | Pump flow rating | $Q_{p,rat}$ | [ℓ/s] |
| | Turbine flow rating | $Q_{t,rat}$ | [ℓ/s] |
| High-pressure u-tubes | Flow rating | $Q_{rat}$ | [ℓ/s] |
| Process | Fissure water flow | $Q_{fiss}$ | [ℓ/s] |

After the design specifications, process flow diagrams and understanding of the systems are in place, historical performance data of the relevant system components need to be obtained (Process A.2.4). If a SCADA system is in place on the mine, all of the relevant data should be obtained for as large a period as is possible. The data should be linked to an external database to ensure continuous access to the data.

Modern mines often measure many additional parameters. The parameters listed in Table 3 shows the critical information required for this study. The parameters listed in Table 3 will be used as process variable baselines as well as simulation inputs.

**Table 3: Critical historical system parameters [13]**

| Equipment | Parameter | Mathematical representation | Unit |
|---|---|---|---|
| Pumps | Power | $P_{act}$ | [kW] |
| | Flow | $Q_{act}$ | [ℓ/s] |
| | Running status | Stat | [-] |
| Dams | Level | L | [%] |
| 3CPFSs | Downcast flow | $Q_{D,rat}$ | [ℓ/s] |
| | Upcast flow | $Q_{U,rat}$ | [ℓ/s] |
| | Running status | - | [-] |
| Turbine-pumps | Turbine flow | $Q_{T,rat}$ | [ℓ/s] |
| | Pump flow | $Q_{p,rat}$ | [ℓ/s] |
| | Running status | - | [-] |
| High-pressure u-tubes | Total flow | $Q_{rat}$ | [ℓ/s] |
| | Mining/Return flow | $Q_{min}/Q_{ret}$ | [ℓ/s] |
| Process | Fissure water flow | $Q_{fiss}$ | [ℓ/s] |

### 3.2.2 Site baselining

After the required data has been obtained, it needs to be processed into information. System baselines serve as a reference for all system changes as well as inputs to the simulations. The literature showed that setting up accurate baselines that is representative of the actual system performance is of utmost importance.

Table 3 showed the critical parameters of mine dewatering systems. Figure 10 shows the proposed process to use when constructing the required baselines.

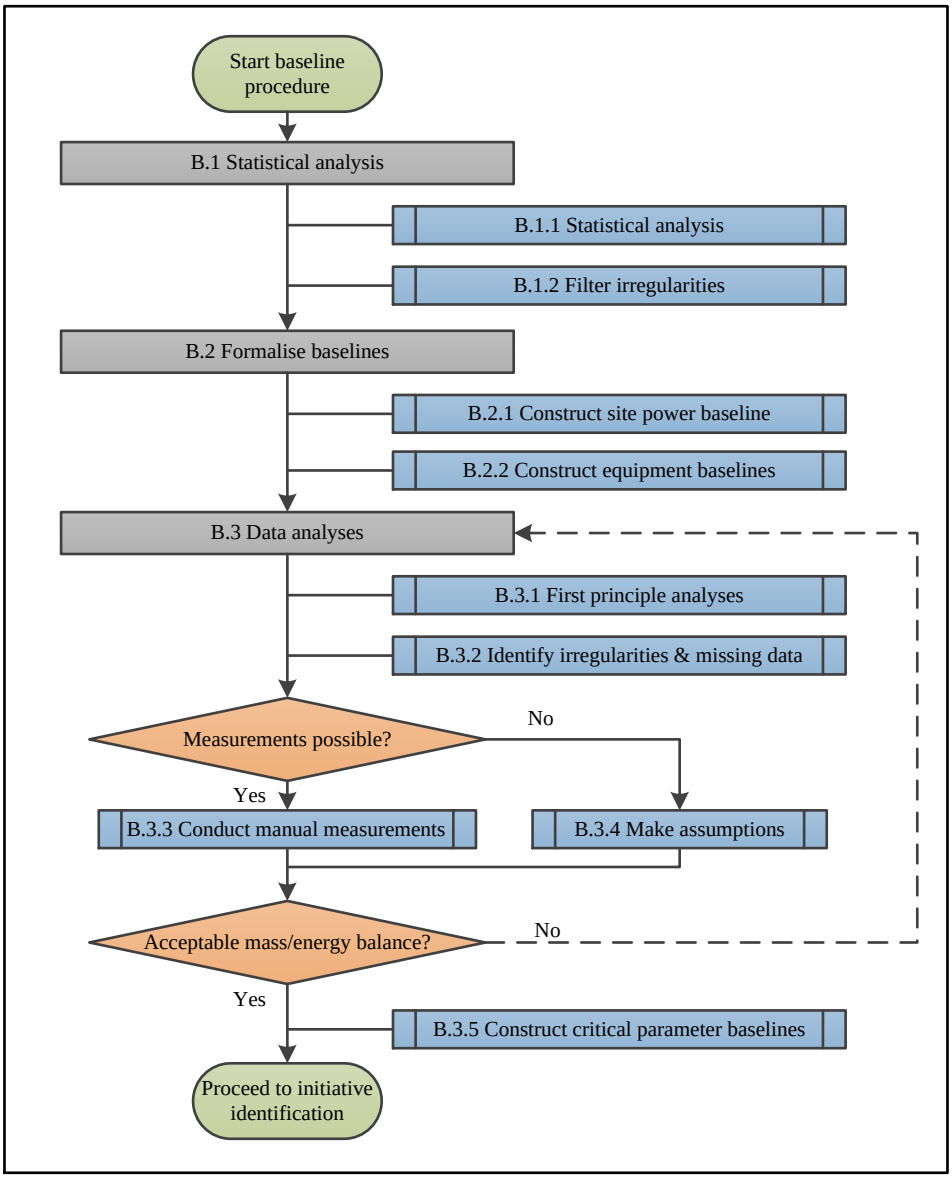



**Figure 10: Baseline procedure**

A statistical analysis of the dataset should be incorporated to filter any irregular data (Process B.1.1). The variance analysis method as proposed in the literature study [71] and summarised in Appendix B is recommended. Data that falls outside a set standard deviation of the mean can be considered as irregular. It is recommended to couple such a statistical analysis with visual inspections as proposed by Booysen [42] when considering the system data.

Any additional known irregular data should be filtered out of the dataset (Process B.1.2). If load management is investigated, this will mean that data from weekends and public holidays should be disregarded since TOU tariff structures differ during these periods. Furthermore, data from known days, where the system was controlled irregularly, should also be disregarded. Such days can include unplanned breakdowns, strikes or other events that influence the regular operation of the mine.

After an appropriate data set is decided upon, an average power profiled baseline can be set up (Process B.2.1). This baseline needs to consist of the average power usage of the entire system for each hour of the day. All of the data that was not eliminated during the statistical analysis of the appropriate period should be included in this baseline. This baseline will form the main reference for the tested initiatives and should consist of data for at least three months.

Process B.2.2 entails constructing averaged single value baselines for the individual components. These will be used for comparative purposes during the simulation calibration process discussed in section 3.2.4. The averaged baselines for the individual components should consist of data when the equipment was running. If different configurations, for example multiple pumps in parallel, are present; this should also be taken into consideration by setting up baselines for each configuration.

Process B.3 entails verifying the data accuracy of the measured process parameters. As a first check (Process B.3.1), a mass balance can be used to test the accuracy of the measuring equipment. For dams as found in mines, the continuity equation as formulated in Equation 8 holds true. The average measured flows in and out of each dam should match over time since dams have a limited capacity. If this is not the case, leaks or inaccurate measurements might be present in the system.

A simple mass or volumetric balance will highlight such anomalies. It is therefore, an effective strategy to verify the accuracy of the simulation inputs.

If a constant density is assumed, the continuity equation presented in Equation 1 in Chapter 1.3 can be re-written as:

$$\sum \dot{V}_i\, t + \dot{V}_i = \sum \dot{V}_e\, t + \dot{V}_e \qquad \text{Equation 8}$$

Where:

$\sum \dot{V}_i$ = the total volumetric flow into the dam [ℓ/s]

$\dot{V}_i$ = the initial volume of water within the dam [ℓ]

$\dot{V}_e$ = the volume of water in the dam after a given time period [ℓ]

$\sum \dot{V}_e$ = the volumetric flow out of the dam [ℓ/s]

t = time [s]

The continuity equation as applied to dams makes it clear that the average flow into the dam will converge to the average flow out of the dam over time. This limits the control on such dams.

If any inconsistencies are identified, Process B.3.3 or B.3.4 can commence. Process B.3.3 involves taking manual measurements of the irregular or missing data. Process B.3.4 involves making acceptable assumptions and should only be used if manual measurements are not possible (as proposed by Maré [13]).

Process B.3.5 entails constructing reference baselines for the critical system parameters. These baselines will be used for simulation purposes. Using averaged profiles for extended periods in this phase is not required. However, measures should be introduced to ensure that the data is representative.

The baselines that are constructed in phase B will be used for:

- performance quantification;
- simulation calibration; and
- simulation inputs.

### 3.2.3 Initiative identification

#### Load management potential

After the system data has been processed, it needs to be analysed. The first phase of the analysis proposed in this study is the preliminary initiative identification. During this phase, the potential for WSO and load management is investigated.

Process C.1 entails the preliminary investigation for load management potential. The load management ratio as formulated in Equation 6 can be used to determine if load management potential is available. Average values of the applicable energy or power baseline as constructed in Process B.1 should be used as inputs for Equation 6.

A high ratio (experience shows above 0.5) would indicate that it could be worthwhile to investigate this type of initiative further. A more detailed analysis of additional constraints relevant to dewatering systems that makes use of ERDs forms part of the optimisation processes.

#### WSO potential identification

Process C.2 involves identifying WSO potential. The literature showed that the ratio between the water flow demand during the blasting shift and that of the peak drilling period as formulated in Equation 7 could be used to identify possible waste of water.

As with the load management, this ratio provides an indication of potential (as with the load management experience shows that a ratio of above 0.5 could be worth investigating). A wide variety of possible wastage exists. Detailed audits of the working areas should therefore, be conducted to determine what the effect of the reduction would be.

In many mines, chilled water is used for cooling. The base water demand during non-production periods is therefore directly proportional to the amount of cooling required. If this is the case, the potential for WSO might be reduced, even if the WSO ratio indicates that potential for improvement is available.

Reducing the flow can therefore potentially affect the underground air temperatures. There is thus significant risk involved in reducing the flow. Investigating the full effect of the possible temperature increases due to reduced flow is therefore of utmost importance.

Note that the exact means of achieving WSO potential is not fixed and will vary on different mines. If the potential for WSO exists on the mine, detailed investigations of the specific mine’s working areas need to be conducted before recommendations are made.

This investigation should include, but is not limited to:

- investigations into the exact supply method;
- amount of underground cooling delivered; and
- identification of leaks.

### 3.2.4 Optimisation

After the system evaluation phase, baselines for all of the process parameters of the system should be in place. The engineer should also have a thorough understanding of the system. This methodology proposed simulations of the entire dewatering system to optimise the control of the dewatering system.

The process as proposed by Maré [13] was simplified for this study because many of the steps required are already included in the first three phases of the optimisation process. Additionally, the focus will be on the dewatering system and not the entire mine cooling system. The hot and cold dams located on the surface of the given mine should therefore, be selected as the simulation boundary limits.

Figure 11 shows Process D.1 which should be used to construct a calibrated baseline simulation.

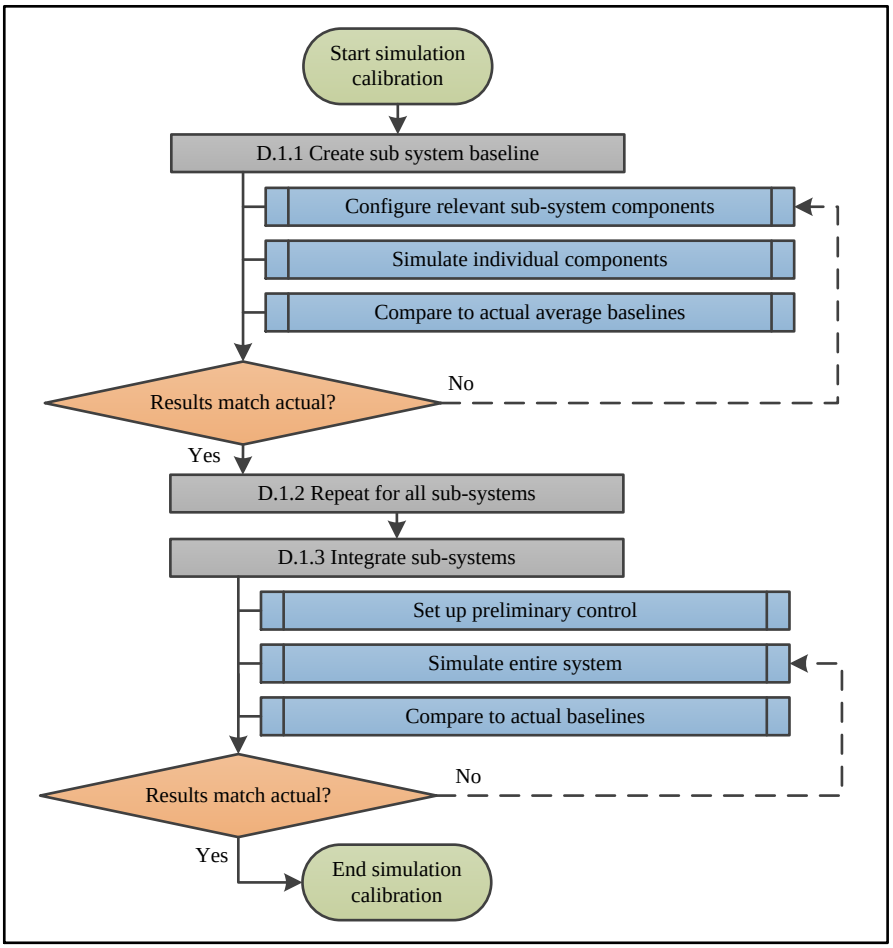



**Figure 11: Simulation calibration**

The first step of this sub procedure is Process D.1.1 which is used to construct baselines of the various subsystems. During this phase, the individual components are configured, simulated and compared to the averaged baselines of the components as constructed in Process B.2. Process D.1.1 should be repeated until the simulation components match the actual values to within 10%.

Process D.1.2 is the repetition of process D.1.1 for all the other subsystems. Process D.1.2 ensures that all of the pump stations and water supply systems are in place and matches the actual measured data. Note that unlike the individual pumps, the water supply system needs to be matched to the profiled actual flows measured into the mining section, as baselined in Process B.3.

During Process D.1.3, the various subsystem baseline simulations are integrated to form a system baseline simulation. During this phase, a preliminary control strategy is programmed into the characterised subsystems. The simulation will then deliver output profiles for all of the relevant parameters.

If the mine is already automated, the profiles and average outputs of the simulation should match the actual outputs to within 10% [13]. However, if manual control is in place on the mine, it is unlikely that the profiles will match during this phase. Therefore, only the averaged outputs of the entire system should be compared to the averaged simulation outputs, if the mine is not automated.

After the baseline simulation has been proven to be accurate; it can be used in the optimisation procedure. Process D.2 is the optimisation phase. Figure 12 shows the optimisation process proposed by this study.

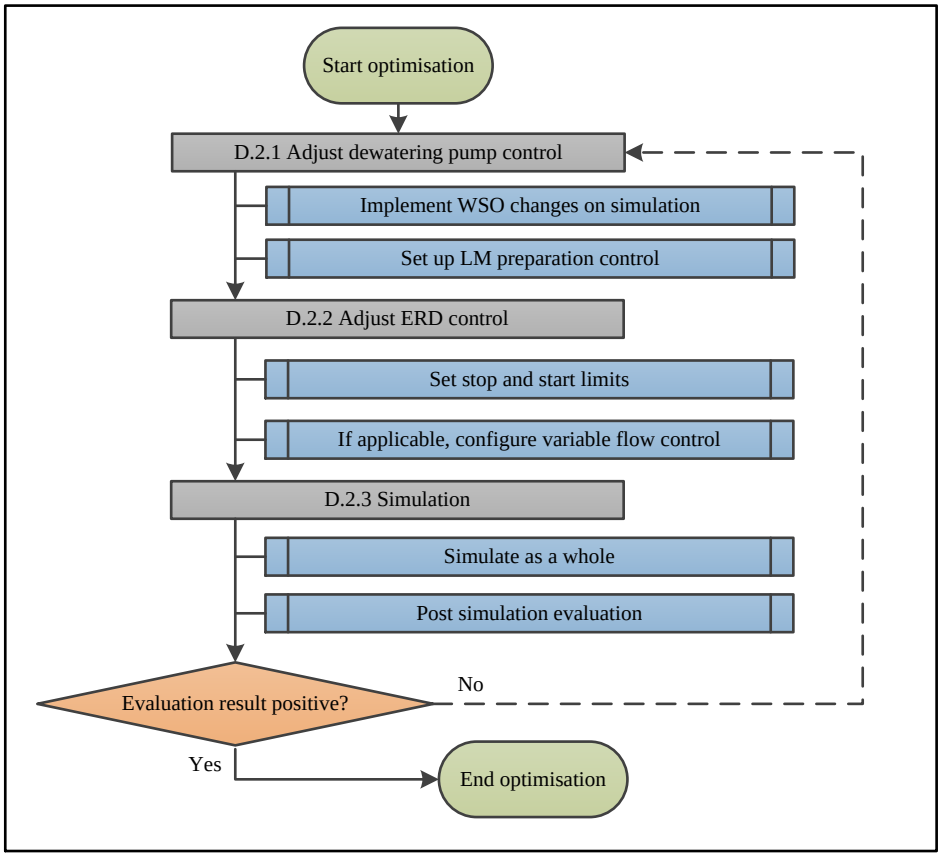



**Figure 12: Optimisation procedure**

Process D.2 entails the optimisation of the system. The first subprocedure in this phase is process D.2.1 which entails implementing the appropriate WSO adjustments in the baseline simulation and setting up load management control. The WSO changes are exclusive in the supply and only one iteration is required if the changes in the simulation are made correctly.

ERDs create unique challenges for the dewatering control. Previous studies showed that if co-generative ERDs are present, the minimum dam levels set in the control should be higher than systems without ERDs. This will allow the ERD, which is not as dependant on electrical energy, to continue dewatering the dam during peak times without emptying the dam completely.

If no ERDs are present on the specific pumping station, the minimum dam level control parameter should be as low as possible to ensure that the spare capacity available in the dam can be used for load management. The rest of the control parameters can be set up as for traditional dewatering systems.

After the changes to the dewatering pump control are set up, Process D.2.2 which entails creating the ERD control can commence. The first step of this process entails creating the ERD stop and start limits. The primary control parameter should be the downstream cold-water dam level.

If the mine does not allow the ERDs to be switched off completely, variable flow strategies can be considered. The primary control variable should also be the downstream cold-water dam level in this case. The output to the VSDs or the control valves will be a flow set point or pump speed.

Note that since co-generative ERDs can limit effective load management and WSO if not controlled properly, other dams should also be considered. The inputs from other dam levels should not be prioritised above that of the downstream dam level. However, since co-generative ERDs influences all of these parameters dynamically, optimal control will require a holistic view of the system.

After the changes to the control have been made in the simulation, procedure D.2.3 can commence. During this process, the system is simulated as a whole. After the system is simulated, checks need to be conducted to ensure that the changes made have led to system improvements.
To ensure that the control programmed into the simulation is optimal and applicable in a real-world scenario, the following questions need to be asked between optimisation iterations:

1) Does the proposed control lead to any pump cycling?
2) Do any dam levels exceed the limitations set by the mine?
3) Do the annual costs reflect an improvement on the baseline value or previous iteration?

If any pump cycling or control limits are exceeded during the simulation phase, the control needs to be revised and simulated again. This process needs to be repeated until the control is acceptable. After the control is deemed acceptable, the annual electricity costs of the obtained power or energy profile need to be calculated. This process is explained in Appendix A after the TOU tariffs used on the applicable mines are provided.

The methodology aims to derive control strategies that ensure minimum cost for the mines. The system energy cost, therefore, has to be lower than that of the baseline. The first iteration of the process will compare the cost to that of the baseline costs. All of the iterations after this first one needs to be compared to the baseline and the previous iteration.

If a change in the energy cost relative to the baseline is observed, an additional control change can be considered. If an optimal or close to an optimal solution is achieved, the difference between the optimised solution and the baseline energy cost should converge to a set value.

To measure the impact of the initiative on the system, the total predicted annual cost is therefore compared to that of the baseline and previous iterations. Appropriate control changes can therefore, be made after such comparisons.

If the checks listed above are all acceptable, the optimisation procedure can conclude. Note that if the effects of load management or WSO needs to be viewed in isolation, the control changes required for the specific initiative can simply be excluded from the procedure.

### 3.2.5 Control strategy formalisation

Process D.3 entails formalising the control strategy. After the investigations have been conducted, a formal control strategy needs to be set up. This control strategy needs to stipulate the following clearly:

- the outputs from Process A;
- the control strategy before implementation of new initiatives;
- communication between equipment, PLCs, the SCADA system and other possible third-party software;
- control boundaries of all the equipment; and
- the control strategy for each piece of equipment as constructed during Process D.2.

This document needs to be approved and signed off by all parties involved. Once the control boundaries have been set up, it can be tested.

### 3.2.6 Implementation

After a detailed control strategy has been established, the implementation of the control can commence. Joubert [12] presented a sequence to be followed that is simplified for this study.

The sequence that needs to occur is shown below.

1) Control revision (Process E.1).
2) Software configuration (Process E.2).
3) Automated testing (Process E.3).
4) Project commissioning (Process E.4).

Process E.1 entails a revision of the control. The revision serves as a final step before the rest of the sequence can commence. After all of the parties agree on the control strategy, the automation testing can start.

Process E.2 entails the configuration of the automation software. This study proposed the use of a REMS for automation. During this phase, the revised control should be programmed into the applicable REMS controllers. These controllers can then be used to control the dewatering equipment through an OPC connection with the SCADA system.

Process E.3 entails the automation testing. The automation testing should be conducted in two phases. The first is a soft commissioning phase. This phase involves giving the control software all of the required inputs. The control software displays the required outputs, but the operators manually mimic the control. Mimicking the control minimises the risk of the testing phase.

This first phase should be implemented over a minimum period of 24 hours. If any errors arise, the control strategy should be revised accordingly. After the necessary changes have been made and approved, the second phase of the automation testing can commence.

The second phase entails running the system in automatic control. This is therefore the first true test of the developed control. The full automation test should be conducted over a period of at least 24 hours.

Process E.3 entails the commissioning of the project. Depending on the results of the tests, the client may approve or reject the automated control strategy. If it is approved, the system can run in an automated state.

If the automated control strategy is not approved, the control can be reviewed again, or a manual strategy can be implemented. Although manual control can be effective, the literature showed that such projects are less likely to succeed. However, the final decision on how to implement the initiatives lies with the mine personnel.

### 3.2.7 Reporting and maintenance

Process F.1 entails the maintenance of the project. Since deep level gold mines are constantly expanding, system changes tend to occur. In addition to this, equipment failures and other unforeseen events could potentially cause significant operational alterations.

The control strategy therefore, has to be constantly updated. Continuous control updates will ensure the long-term success of the projects. Where necessary, simulations can be conducted again to ensure the success of changes made to the system. In addition to this, if the automated control strategy is insufficient in any regard, the specific incidents should be logged and reported to the responsible engineer. Appropriate action can then be taken.

Process F.2 entails reporting of the project status to raise awareness. This step is necessary to ensure that the impact of the project is made clear. In addition to this, reporting is also a requirement of the M&V process as was mentioned in Chapter 2.

### 3.2.8 Summary

A complete methodology for the optimisation of mine dewatering systems that makes use of ERDs has been proposed. This method covers the entire project lifecycle. The steps proposed have been shown to be effective in the consulted literature and for the evaluation and implementation phase, this too has been shown to be effective from literature.

Methodologies for the optimisation of the dewatering system were also developed. The optimisation methods are verified in the next section of this chapter.

## 3.3 Evaluation and simulation of mine A

### 3.3.1 Introduction

A methodology to optimise mine-dewatering systems that make use of ERDs has been proposed. Before the method is applied to a case study, the effectiveness of the method and the proposals made in the method need to be verified. The verification will be done by applying the method to an already optimised dewatering system found on Mine A.

Mine A was evaluated as the developed methodology proposes. After this evaluation, a baseline simulation was constructed. To verify the developed optimisation strategies, control limitations were applied within the simulation. The developed strategies were used to overcome the introduced limitations within the simulations. The details of this process are provided in Chapter 3.4.

Note that Mine A makes use of a REMS. A REMS was also used on Mine B. The specific control mechanisms of this software is therefore also discussed. The software used for the simulation in this study is Process toolbox.

Process toolbox makes use of generic components and has the capacity to dynamically simulate integrated thermal hydraulic systems as was identified as a requirement in the literature review. Details on the simulation components and models are provided in Appendix C.

### 3.3.2 Mine A – underground dewatering system

Figure 13 shows a layout of the dewatering system of Mine A (Process A.2.1). On Mine A, a surface refrigeration system provides chilled water to the surface cold dam. Water from the surface cold dam is supplied to the 39L cold dam through two turbine-pump sets.

Water from the 39L cold dam is supplied to the mining sections. A WSO initiative is in place on the mine, the water into the mining sections is controlled to a set pressure set-point, thereby minimising the water usage of the mine. After being used in the mining sections, the water is channelled through the mine into settlers on 75L. The settlers overflow into the 75L hot dam.

Mine A has two main pumping stations. These are located on 75L and 38L. Four multi-stage dewatering pumps are located on 75L. These pumps pump the used mining water into the 38L hot dam. Three multi-stage dewatering pumps and two turbine pumps are located on 38L. These pumps pump water from the 38L hot dam to the surface hot dam.

The refrigeration system cools the water down again before it is sent down the mine. A closed-loop water reticulation system is therefore formed. Since the system is automated, all of the mentioned components can be remotely stopped and started.

The relevant design parameters of the dewatering equipment are summarised in Table 4 (Process A.2.2).

**Table 4: Mine A – Pump specifications**

| Mine A dewatering equipment | | | |
|---|---|---|---|
| **Equipment piece** | **Flow [ℓ/s]** | **Power [kW]** | **Elevation [m]** |
| 75L pumps | 120 | 2000 | 1126 |
| 38L pumps | 120 | 2000 | 1160 |
| **Turbine pumps** | **Turbine flow [ℓ/s]** | **Pump flow [ℓ/s]** | **Elevation [m]** |
| Turbine | 170 | - | 1160 |
| Pump | - | 115 | 1160 |

The specifications of the dams are summarised in Table 5 (Process A.2.2).

**Table 5: Mine A – Dam specifications**

| Mine A underground dams | | | |
|---|---|---|---|
| **Equipment piece** | **Volume [Mℓ]** | **Max level [%]** | **Min level [%]** |
| 39L cold dam | 4412 | 80 | 35 |
| 75L hot dam | 5412 | 80 | 25 |
| 39L hot dam | 4412 | 80 | 25 |

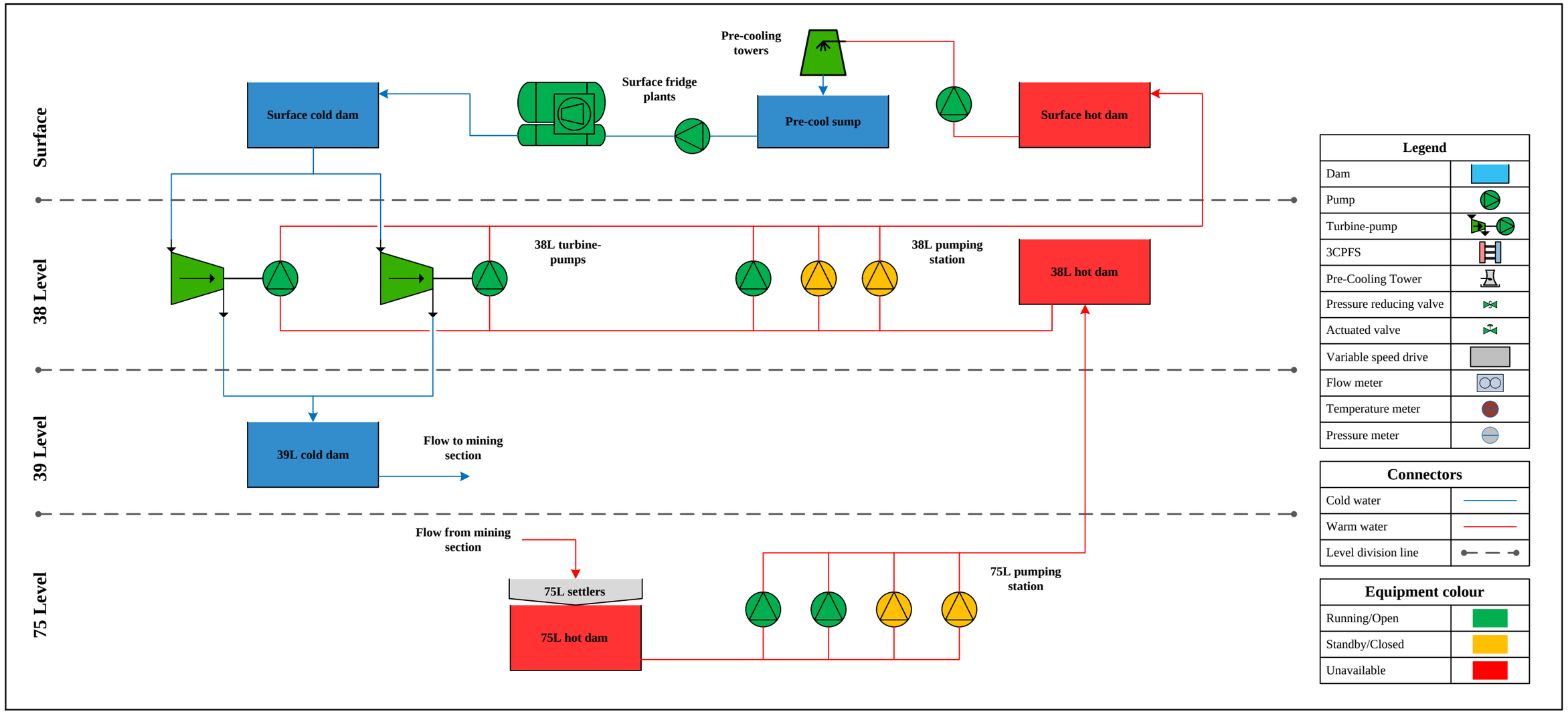



**Figure 13: Mine A – Simplified process flow of the water reticulation system of**

The dam volumes displayed in Table 5 are the actual dam levels obtained from the mine personnel. The minimum and maximum dam levels indicated are the actual limits and should not be confused with the REMS control parameters.

A REMS is installed on Mine A. The relevant system parameters on Mine A has been logged for an extended period. These parameters are therefore readily available to be used for the baselining of the system as the methodology requires.

A REMS can be used for many purposes. These include real-time pump scheduling, logging of process data and conducting steady state simulations to name but a few. The technology has also been proven to be effective in many other studies [38, 18, 15, 52].

The REMS typically communicates with the SCADA through an OPC connection. A REMS system has many built-in control mechanisms that can be used to automate mine dewatering systems [18]. A brief overview of the various control mechanism will now be provided.

The primary control mechanism of the REMS is the real-time dam levels. An overview of mine dewatering system control using dam levels are provided. Either upstream or downstream dam level control can be used to control dewatering systems. Figure 14 explains the concept of upstream and downstream dams as applicable for co-generative ERDs.

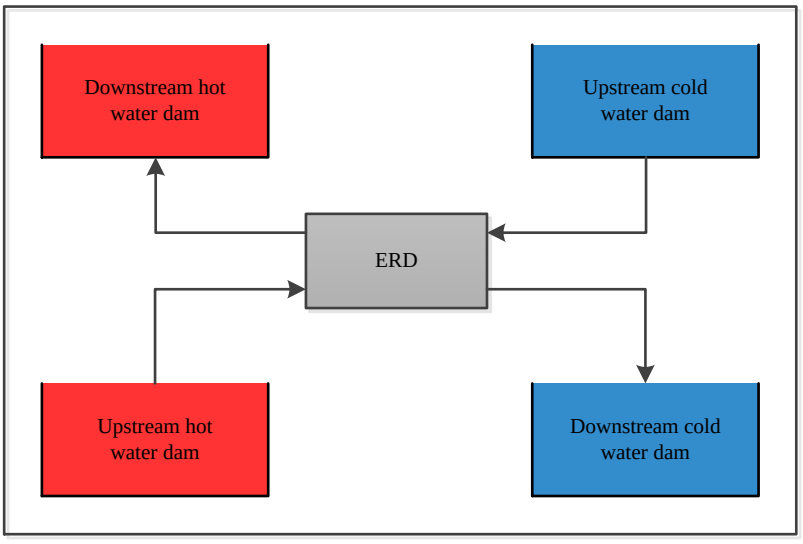



**Figure 14: Upstream and downstream dam illustration**

When traditional pumps are used for dewatering, only the hot dams indicated in red in Figure 14 are considered. The upstream dam therefore merely refers to the source of the water being transferred. The downstream dam, in turn, is the destination of the stream of water. Traditional dewatering system control will typically focus on either the upstream or downstream dam level.

The primary control of the REMS system is the REMS III pump controller. An example of the operations of an upstream REMS III controller is displayed in Figure 15. Five main control parameters are applicable. These are summarised in Table 6 along with values of the specific parameters used in the example.

**Table 6: REMS pump controller III control parameters**

| Parameter | Value |
|---|---|
| Maximum dam level [%] | 80 |
| Minimum dam level [%] | 30 |
| Control range [%] | 10 |
| Top offset [%] | 5 |
| Bottom offset [%] | 5 |

In addition to the five control parameters, the following has to be provided to the controllers:

- minimum and maximum number of pumps;
- upstream or downstream control; and
- start-up delay.

The working of the REMS III control will be explained through an example. The example makes use of the control parameters summarised in Table 6. Upstream control with a maximum of two pumps is used in the example. The example is illustrated in Figure 15 and explained afterwards.

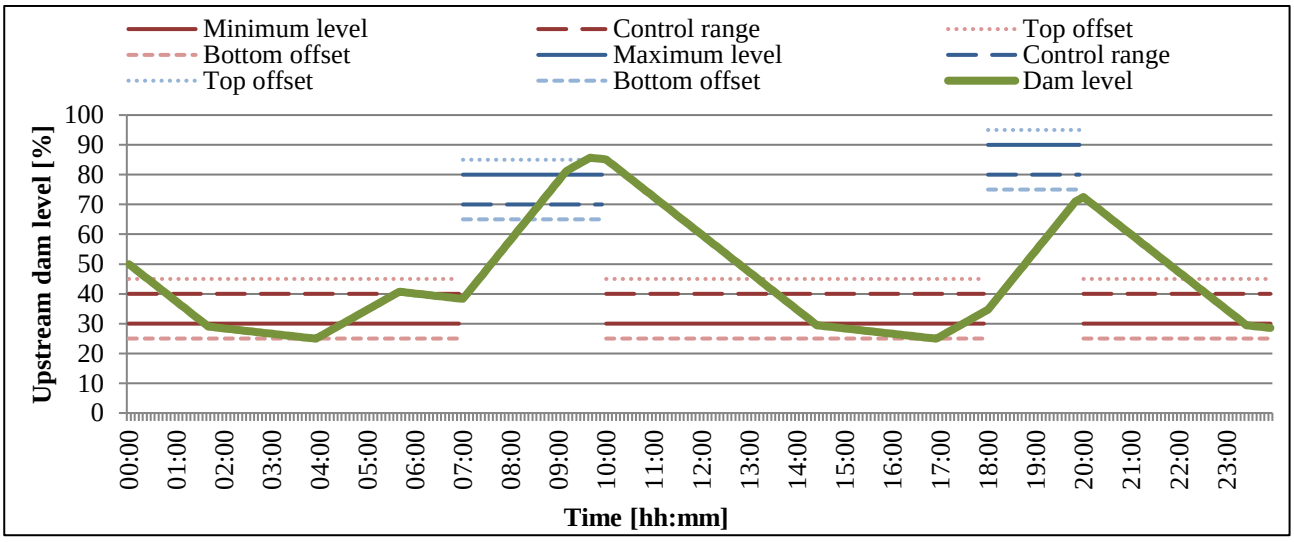



**Figure 15: Upstream dam level controller**

In the example, the dam level starts at 50%. At 50%, two pumps will be scheduled to start up automatically. During non-peak times, the controller will control to the minimum dam level specified.

The two pumps started up initially will continue to run until the dam level reaches the minimum level of 30%. The first of the two running pumps will then stop. If the level continues to decrease, the second pump will stop when the dam level reaches the difference between the minimum level and the bottom offset. The minimum dam level is therefore, a control parameter and not the absolute minimum level set by the mine.

If both pumps are switched off, it will remain off until the dam level reaches the sum of the control range and the minimum dam level. If this condition is met, a pump will start up again. If the level continues to rise, a second pump will start up when the dam level reaches the sum of a minimum level, control range and top offset. The running pump will stop again if the minimum level is reached.

In the example, neither of these cases is realised. The reason for this is that during peak times the level is controlled according to the maximum dam level. In the example, all of the running pumps are therefore switched off as soon as the peak period begins.

The pumps will remain off until the maximum dam level is reached. If the maximum dam level is reached during peak hours, one pump starts up. If the dam level was to reach the sum of the maximum level and the top offset, a second pump would start up.

A pump will be stopped again if the dam level decreases to the difference between the maximum level and the control range during peak times. A second pump will stop if the dam level continues to decline and the difference between the minimum level, control range and bottom offset is reached.

After peak hours, the pumps will be controlled to the minimum dam level once more. The control will then continue as previously described. The maximum number of pumps will therefore be started up again in the example, and the system will be prepared for the next evening peak.

The REMS III controller can control traditional dewatering pumps. However, customised control strategies will have to be developed for the ERDs. The REMS also allows for custom controllers to be programmed. These custom controllers can be used to control the ERDs as required. Custom parameters can also be added to the control of the traditional pumps if required.

When using custom control logic, the use of alternative control parameters can become necessary. These alternative control parameters typically help to avoid pump cycling. One such alternative control parameter is the use of stable dam levels. Figure 16 shows an example of a top stable dam level value.

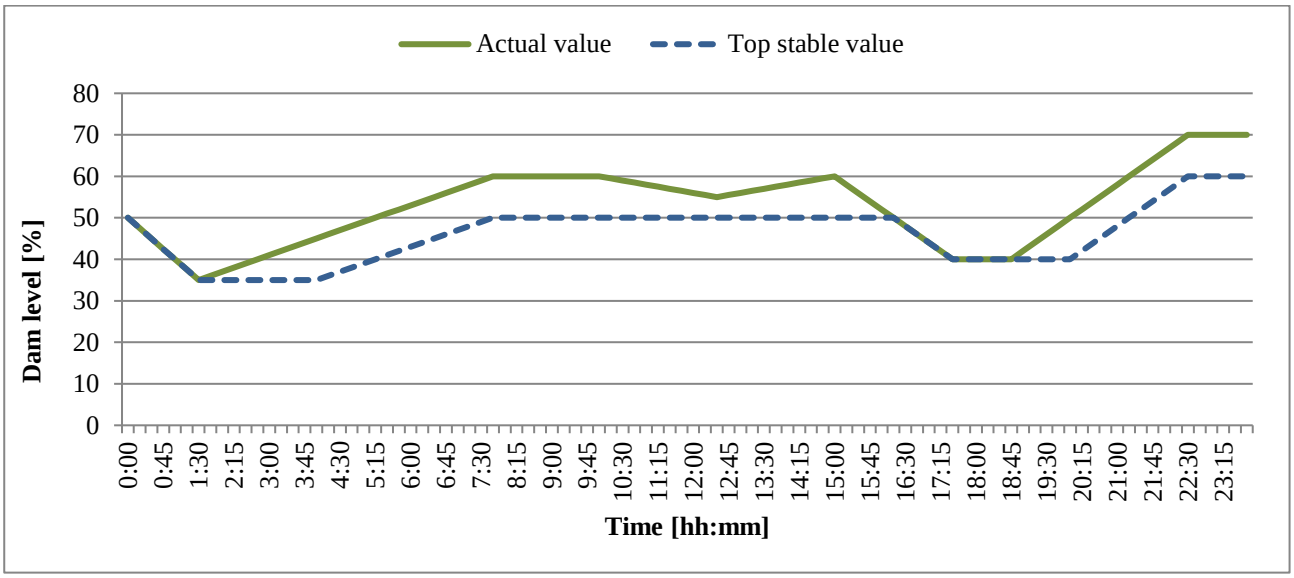



**Figure 16: Top stable value**

Four primary variables are applicable when using a stable value. These are:

- actual dam level;
- stable dam value;
- top stable value; and
- bottom stable value.

The stable dam value is the output. The actual dam value, top and bottom stable values, are used to determine the stable value.

In the example, as shown in Figure 16, a top stable value of 10% is selected and a bottom stable value of zero. The dam level starts at 50% and starts to decline steadily. At this stage, the stable dam level value will be equal to the actual dam level since the bottom stable value offset is zero.

At 01:30, the actual dam level starts to increase. The stable value will remain constant until the difference between the actual dam level and stable value is 10% (top stable value). The stable value will start to increase and maintain the 10% offset as long as the dam value is increasing.

If the actual value starts decreasing, the stable dam level will remain constant until the stable dam level is equal to the actual dam level. Since the bottom stable value is zero in this example, the stable dam level and actual dam level will be equal again after this occurs and the level continues to decline.

Using stable values allows for a time offset between the conditions. This, in turn, avoids cycling of the pumps. The use of stable values or other means of avoiding pump cycling is essential when setting up custom control on mine dewatering systems.

### 3.3.3 Mine A dewatering control

The automated control strategy of Mine A is also available. Mine A makes use of upstream control. Table 7, Table 8 and Table 9 shows the REMS control parameters (Process A.2.3).

**Table 7: Mine A – 38L pumps control**

| Parameter | Value |
|---|---|
| Control range [%] | 5 |
| Top offset [%] | 3 |
| Bottom offset [%] | 3 |
| Maximum level [%] | 80 |
| Minimum level [%] | 50 |

The 38L hot dam is controlled between 50% and 85%. Note that this is required to ensure that the turbine pumps can operate in peak hours if necessary. The control range and offset keep the dam levels close to the required levels for effective load management.

**Table 8: Mine A – 75L pumps control**

| Parameter | Value |
|---|---|
| Control range [%] | 5 |
| Top offset [%] | 5 |
| Bottom offset [%] | 5 |
| Maximum level [%] | 65 |
| Minimum level [%] | 35 |

The 75L pumps are controlled between 35% and 65%. Since no co-generative ERDs are present on this level, lower dam levels during standard and off-peak times lead to optimal load management performance.

**Table 9: Mine A – Turbine-pump control parameters**

| Level - 39 dam level [%] | Number of turbines [-] |
|---|---|
| < 55 | Start a turbine |
| < 60 | Start a turbine |
| > 90 | Stop a turbine |
| > 95 | Stop a turbine |

The control of the turbine-pumps on Mine A is relatively straightforward. The turbine pumps are controlled to the downstream cold water dam levels. The control is therefore strictly to the demand.

Note that on Mine A additional control parameters in the form of minimum and maximum number of pumps as well as turbine-pumps are utilised. The detailed control logic used for these parameters are summarised in Appendix D.

The historical data required by Process A.2.4 was readily available from the REMS server installed at Mine A. The processes discussed thus far, therefore, concludes phase A.1 as required by the methodology.

### 3.3.4 Mine A data analyses and baselining

Since the changes made for the verification procedure was artificial, a formal system reference baseline as required by procedure B.1 will not be constructed for Mine A. In addition to this, the power usage of the dewatering pumps on Mine A is not actively measured.

However, the pumps running statuses and rated power usage is available. In addition to this, the pumps pump against a constant pressure head. Therefore, the pumps were characterised during the original optimisation project, and it was assumed that the pumps use a fixed power when running.

The mine personnel agreed upon this practice. Since the time was not available to measure these values, the assumption was made for this study as well. The individual pumps and turbines were characterised to match the flow values depicted in Table 4 at the assumed power usage of each pump.

Actual demand flows into the mining areas were obtained from the on-site REMS system installed at the mine. The total flow into the surface hot dam from the underground working areas and the flow out of the surface cold dam to the underground sections were also obtained.

A three-month period with good load management performance was selected for analysis. Manual measurement of parameters was not possible during this phase. Therefore appropriate assumptions were made to ensure a balance on the system is in place. Three consecutive days during which all of the flows in the mine were within one standard deviation of the average values of the entire period were identified. Table 10 shows these values (output of Process B.1.2).

**Table 10: Mine A – Variance analyses**

| Parameter | Flow to UG [ℓ/s] | Flow to surface [ℓ/s] | Mining water [ℓ/s] | Drinking water [ℓ/s] | Total used water [ℓ/s] |
|---|---|---|---|---|---|
| Average | 136 | 152 | 107 | 9 | 115 |
| Day 1 | 141 | 147 | 113 | 9 | 121 |
| Day 2 | 153 | 169 | 118 | 9 | 127 |
| Day 3 | 129 | 142 | 121 | 9 | 130 |
| Std. deviation | 28 | 29 | 33 | 0 | 34 |

Since the reference period used for the verification is short, no additional irregularities had to be filtered from the data set (Process B.1.2). The same three-day period that was identified from the data analysis was used as a power reference. The reference baselines that was used for the rest of the verification procedure is summarised below.

Using the running statuses of the pumps and the assumed power usage, a profiled power baseline of the dewatering system on Mine A was set up. Figure 17 shows this baseline for the identified period, which serves as the equivalent to the profiled power baseline for verification purposes. Note that the individual component baselines (Process B.2.2) are shown in Table 12, where it is also compared to the simulation values.

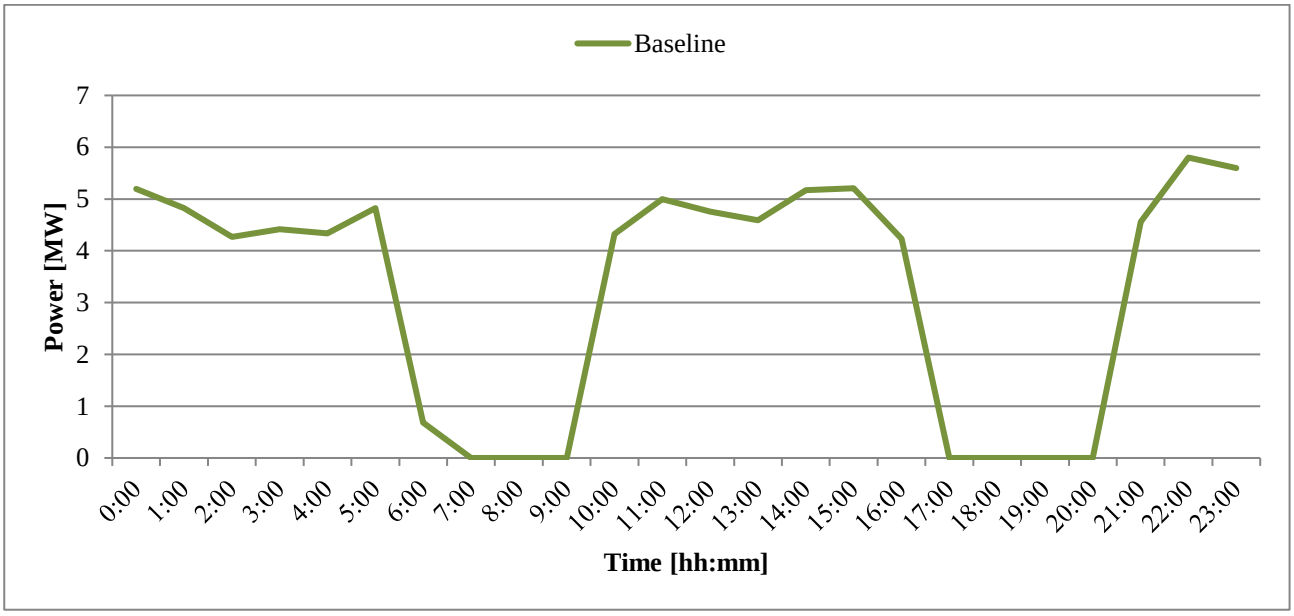



**Figure 17: Mine A – Power baseline**

To verify the accuracy of the underground water measurements, a mass balance over the entire system was conducted. The averages over the identified period is summarised in Table 11 (Process B.3.1).

**Table 11: Mine A – Critical process flow balance**

| Parameter | Flow [ℓ/s] |
|---|---|
| Total water flow to underground | 143 |
| Flow to surface | 147 |
| Drinking water used | 9 |
| Mining water used | 117 |
| Unmeasured mining water | 17 |
| Assumed fissure water | 5 |

Manual measurements (Process B.3.3) could not be conducted as part of the verification procedure. Appropriate assumptions were therefore made to calculate the unmeasured flows in the mine. These flows include unmeasured flows into mining areas and fissure water.

The fissure water was assumed to be the difference between the total water sent underground over the week and the water pumped back to surface. The unmeasured mining flows were assumed to be the difference between the total water flow to underground, the drinking water and the measured water into the mining sections (Process B.3.4). Figure 18 shows the mining water flows over the applicable three day period.

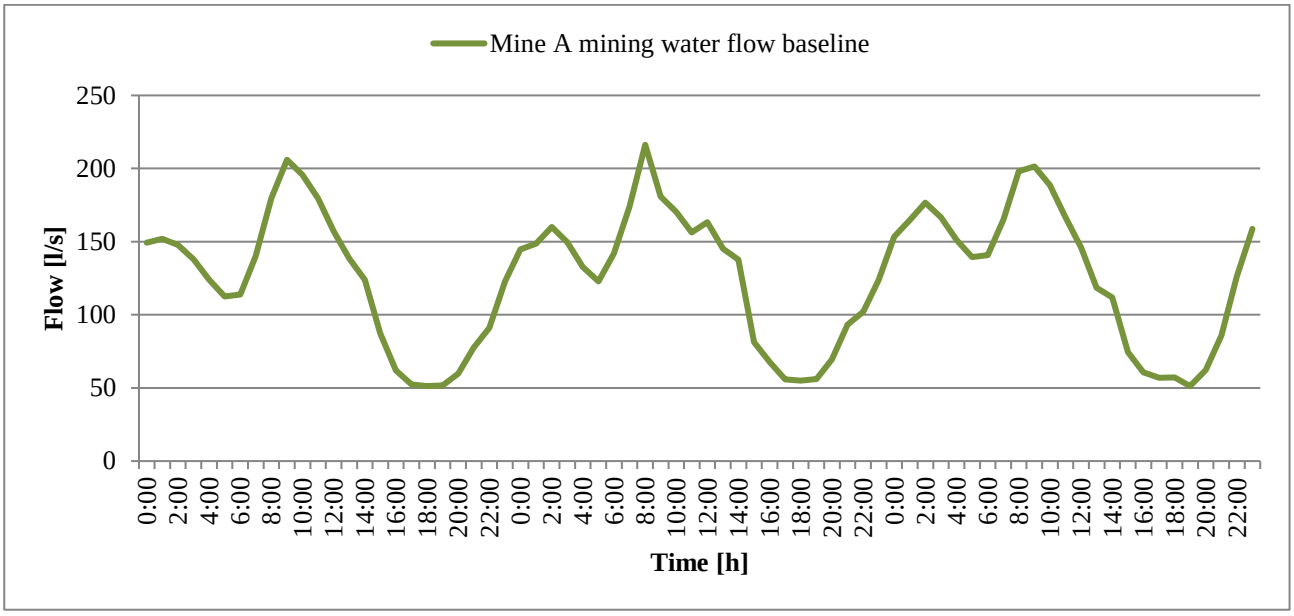



**Figure 18: Mine A – Mining water flow baseline**

These outputs conclude Process B. Process C.1 and C.2 were not applicable to the study since Mine A is already optimised.

### 3.3.5 Baseline simulation development

As the optimisation methodology suggests, the surface hot and cold dams were selected as the simulation boundaries. The generic pump components available in Process toolbox were characterised to match the design power at the specific flows measured for each pump. The pump simulation components were connected with the appropriate dams and valves to form a simulation of the entire dewatering system.

Table 12 summarises the simulated flows for the various subsystems (Process D.1.1 and D.1.2).

**Table 12: Mine A – Simulated vs actual pump flows**

| Parameter | Actual flow [ℓ/s] | Simulated flow [ℓ/s] | Difference [%] |
|---|---|---|---|
| 38L 1 turbine-pump (pump) | 106 | 115 | 9 |
| 38L 2 turbine-pumps (pumps) | 222 | 230 | 4 |
| 38L 1 turbine-pump (turbine) | 171 | 170 | 1 |
| 38L 2 turbine-pumps (turbines) | 397 | 340 | 14 |
| 38L 1 pump | 104 | 115 | 10 |
| 38L 2 pumps | 224 | 230 | 3 |
| 75L 1 pump | 117 | 120 | 2 |
| 75L 2 pumps | 228 | 240 | 5 |
| 75L 3 pumps | 327 | 360 | 10 |

Note that the power is not included in Table 12 since it is not measured. However, the simulation matches the assumed power values as summarised in Table 4. Additionally, the flow if two pumps are running does not seem accurate, since it increases when multiple pumps are running and only one column is present.

Unfortunately, additional verification of these measurements was not possible. The simulation was therefore set up as though each pump has its own column. Although this might not reflect the actual system 100%, it is sufficient for this phase of the study and should not have a significant effect on the outcomes and conclusions.

The flow profile as displayed in Figure 18 was set as the demand flow. It was assumed that this demand flows directly into the settlers on 75L. The automated control strategy that is already in place on Mine A was programmed into Process toolbox through the use of the generic controllers available in the software. This control strategy was therefore used to control the characterised components in the simulation model according to the actual demand profiles.

Figure 19 compares the actual power profile baseline over the period to the simulated power profile (Process D.1.3).

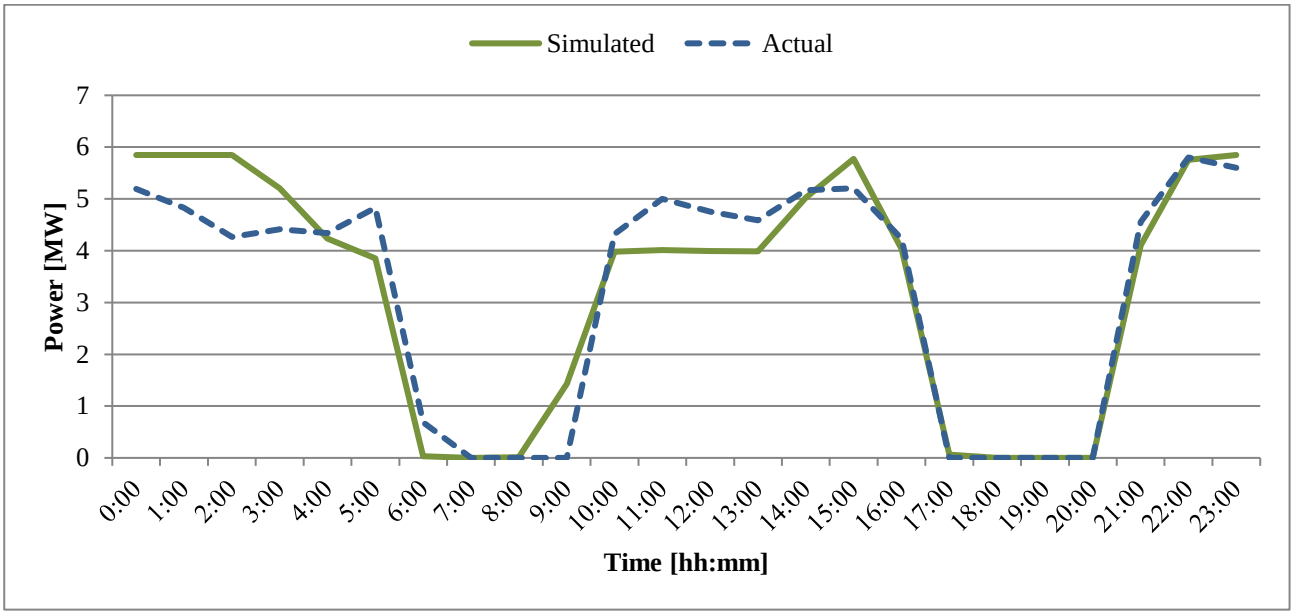



**Figure 19: Simulated and actual baseline of mine A**

A visual inspection of the simulated and actual power profiles of the dewatering system of Mine A shows slight discrepancies between the profiles. There are multiple possible reasons for this. Firstly, the system was simulated as a closed-loop system. This means that the used mining water flows directly into settlers on 75L.

In reality, this flow is buffered, and the water that flows into this dam does not necessarily match the demand flow at that specific time. It is however, almost impossible to measure this flow. Additionally, the unmeasured demand flow and fissure water were assumed to flow into the hot dam on 75L at a constant rate throughout the day.

The simulated dam levels did not match the actual dam levels for each time interval. Since these dam levels are the main control parameters, it can lead to minor differences between the simulated and actual schedules and therefore the different power profiles.

However, the average difference in power usage between the actual profile and simulated profile is only 1.4%. A correlation of 95.7% between the two profiles also exists. The simulation model can therefore be considered as calibrated.

## 3.4 Verification of newly developed optimisation strategies

### 3.4.1 Verification procedure

The dewatering system of Mine A is already optimised. The system is completely automated. In addition to the ERDs, load management and WSO initiatives are in place on the system to ensure minimum energy costs.

Since the aim of the methodology is to optimise the control of any dewatering process that makes use of ERDs, the turbine-pumps were replaced by 3CPFSs and a closed-loop high-pressure u-tube system in the simulation.

After the turbine pumps were replaced by these other ERDs within the simulation, limitations were added to the control. The limitations placed on the system did not allow for the scheduling of the ERDs thereby forcing the ERDs to remain running regardless of the demand. Therefore, the effect of this limitation on each simulated system was shown.

After the effect of the control limitations were discussed, newly developed procedures in the form of variable flow strategies were applied within the simulations to optimise the simulated systems. The evaluation tested the effect on the annual system electricity costs.

This test is included in the study to test the effectiveness of the strategies before implementation.

### 3.4.2 Simulation model alterations

The only ERDs that are present on Mine A are turbine pumps. Therefore, the turbine pumps were replaced with alternate ERDs in the simulation model to test the developed methodology on such systems. In addition to this, a baseline simulation with no ERDs installed was constructed.

Figure 20 and Figure 21 shows the power profiles for the scenario with a closed-loop high-pressure u-tube system and a 3CPFS respectively. Each was compared to the profile of the turbine pumps. The control in the altered baseline simulations was set up to ensure optimal profiles for the reference simulations are obtained. The detailed control strategies used in this simulation can be found in Appendix D.

It was assumed that the 3CPFSs could be scheduled in the same way as the turbine-pumps in the actual system. However, a minimum demand for cold water exists underground. The closed-loop high-pressure u-tube system supplies this water and the same assumption could therefore not be made for this system.

Figure 20 shows the simulated profile of Mine A with no turbine pumps and a closed-loop high-pressure u-tube in operation.

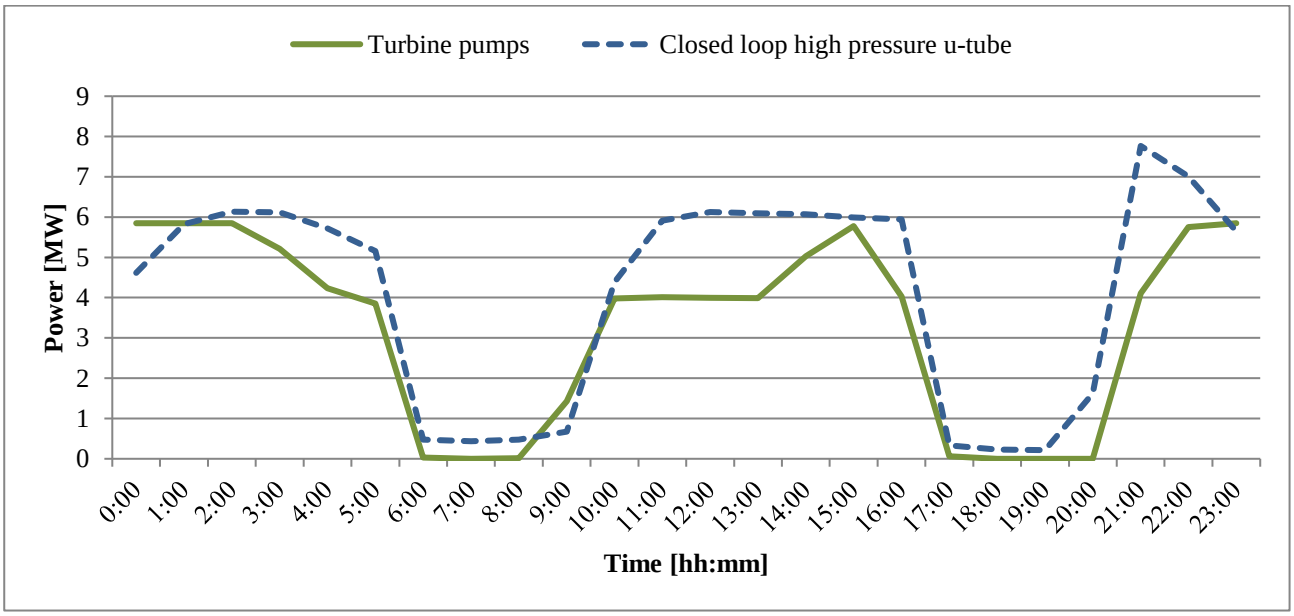



**Figure 20: Mine A – Closed-loop high-pressure u-tube baseline simulation**

The closed-loop high-pressure u-tube system is shown to use slightly more power than the turbine pumps. Note that since the booster pump that accompanies the closed-loop high-pressure u-tube system is never switched off completely, a baseload power usage is applicable. However, significant potential for load management can still clearly be seen.

Figure 21 shows the simulated profile of Mine A with the turbine pumps replaced with 3CPFSs.

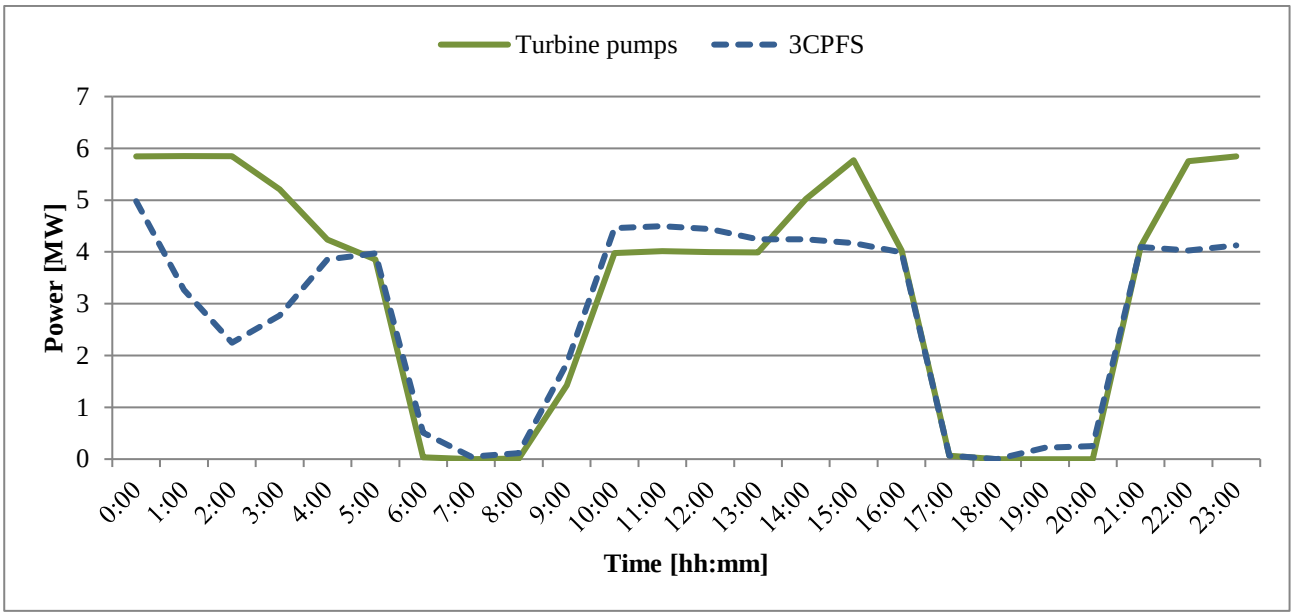



**Figure 21: Mine A – 3CPFS baseline simulation**

A clear reduction of the power usage can be seen if 3CPFSs replaces the turbine-pumps. A minor decrease in the load management potential for mine A is observed if a 3CPFS is in place. The reason for this is the increased flow delivered by the 3CPFS. On some of the simulated days, the 3CPFS dewatered the 38L hot dam during peak hours. To prevent the 38L hot dam from going below the minimum level, the simulated controllers started dewatering pumps on 75L in peak time.

Note that on Mine A, the turbine pumps can be switched on and off as required. This allows the mine to control the chilled water sent down the mine strictly according to the demand. This indicates that this is a viable option for mines with turbine pumps in place. It is also clear that if other ERDs replace the turbine pumps, this still holds true.

Table 13 compares the annual costs of each type of ERD.

**Table 13: Mine A – ERD performance comparison**

| ERD configuration | Average daily power [kW] | Average peak power [kW] | Annual electricity cost [R] |
|---|---|---|---|
| Turbine pumps | 3290 | 190 | 14.8 million |
| Closed-loop high-pressure u-tube | 4120 | 555 | 19.1 million |
| 3CPFS | 2770 | 380 | 13.1 million |

All of the ERDs clearly offers potential for load management within the simulation. The use of load management initiatives on systems that incorporates ERDs is therefore clearly justified. However, to ensure that the optimisation methodology developed is applicable, limitations will be added to the developed simulation models. After these limitations are implemented within the simulations, the optimisation methodology will be used to optimise the simulated systems.

The simulations with the limitations applied will therefore serve as the baseline simulations required by Process D.1. These limitations will then be overcome within the simulation models using Procedure D.2. Procedure D.3 will not be applied to the simulation since the tests are theoretical and will therefore not be implemented.

### 3.4.3 Introduction of limitations within the simulation

Since the simulated closed-loop high-pressure u-tube system was not stopped in the baseline simulation, only turbine pumps and 3CPFSs will be simulated in this phase of the verification.

The effects of keeping the ERD running will be shown. After the effects are shown, the use of variable flow strategies will be tested to illustrate the possibility of flow reduction and improved load management. The effect of using this strategy will then be compared to the cases where the ERDs were not stopped and delivered a constant flow.

Figure 22 shows the power profile for Mine A if one of the turbine pumps is forced to remain running compared to the simulated baseline profile.

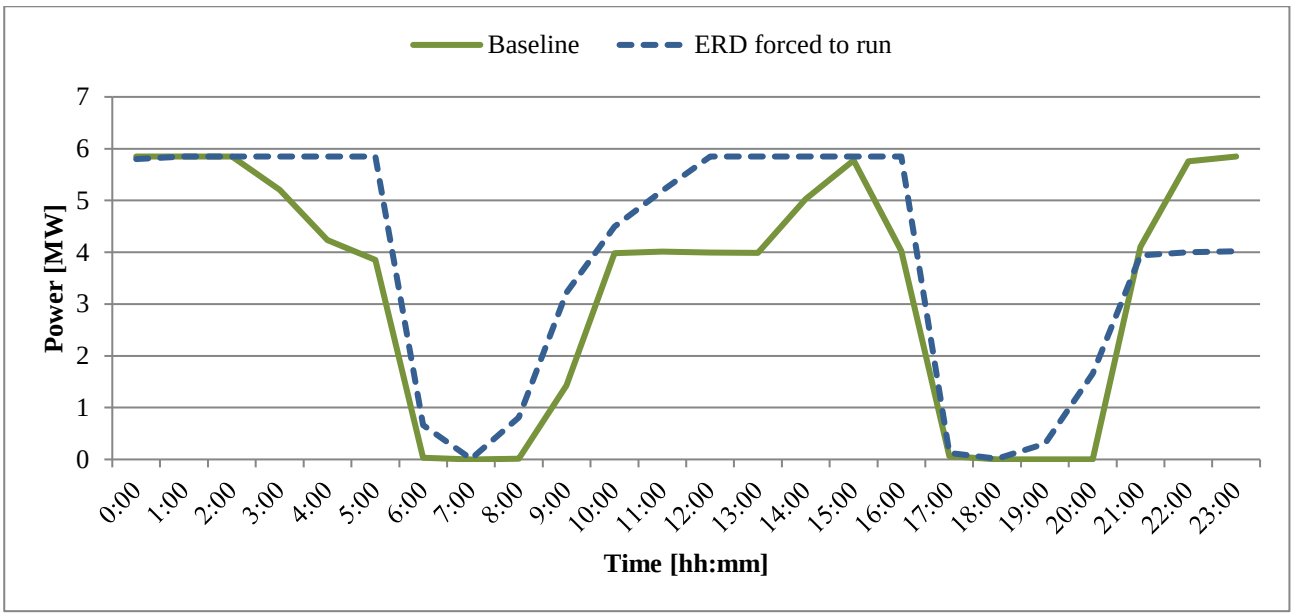



**Figure 22: Mine A – Turbine pump forced to run power profile**

A definite increase in power usage and reduction in load management performance can be seen in the profile. This is due to an increase in the flow sent underground due to the specific configuration. The effect on the 38L as with an increased flow demand leads to reduced load management potential.

The same limitations were applied for the simulated 3CPFS on Mine A. Figure 23 shows the simulated power profiles if scheduling of the 3CPFS is not allowed (forcing the 3CPFS to remain running regardless of the demand) within the simulation.

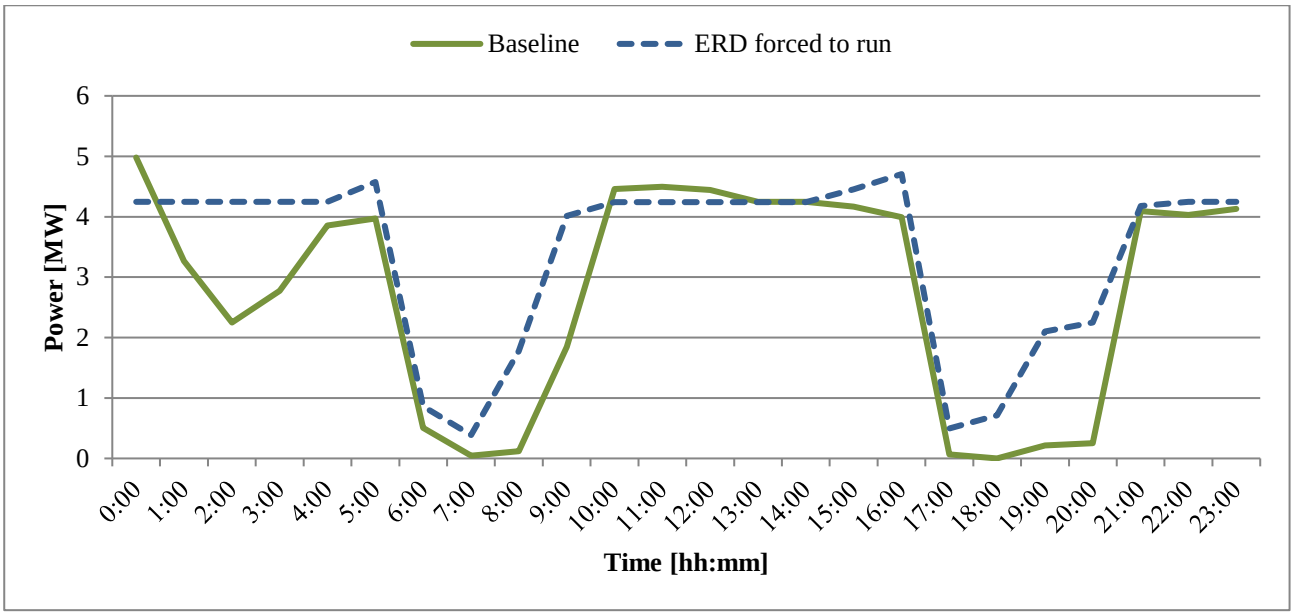



**Figure 23: Mine A – 3CPFS control inhibited power profile**

A clear increase in the overall energy usage of the system can be seen. Additionally, the load management performance also decreased significantly. Since the actual demand flow, in this case, is known, potential for improvement on the system was shown to be achievable.

The integrated effect of this decrease in performance leads to higher costs for the mines (summarised in Table 15). The methodology proposes variable flow strategies to help negate the effect of the control limitations.

### 3.4.4 Simulated introduction of variable flow strategies

The following assumptions were made to test the variable flow strategy on the turbine pumps:

- the flow through the turbine can be adjusted;
- the ratio of pump flow to turbine flow remains constant at 0.68:1 if the flow is varied;
- the turbine flow can be reduced by up to 40%; and
- the fissure water flow remains unchanged.

Variable flow control was incorporated in the simulation to maintain a dam level of 80% during standard and off-peak period. During peak hours, the dam level set-point was reduced to 40%. The control of the rest of the dewatering equipment remained unchanged. These changes allowed the turbine-pump to use minimum flow during peak times. Figure 24 shows the effect of applying a variable flow strategy on the turbine pumps to the dewatering system.

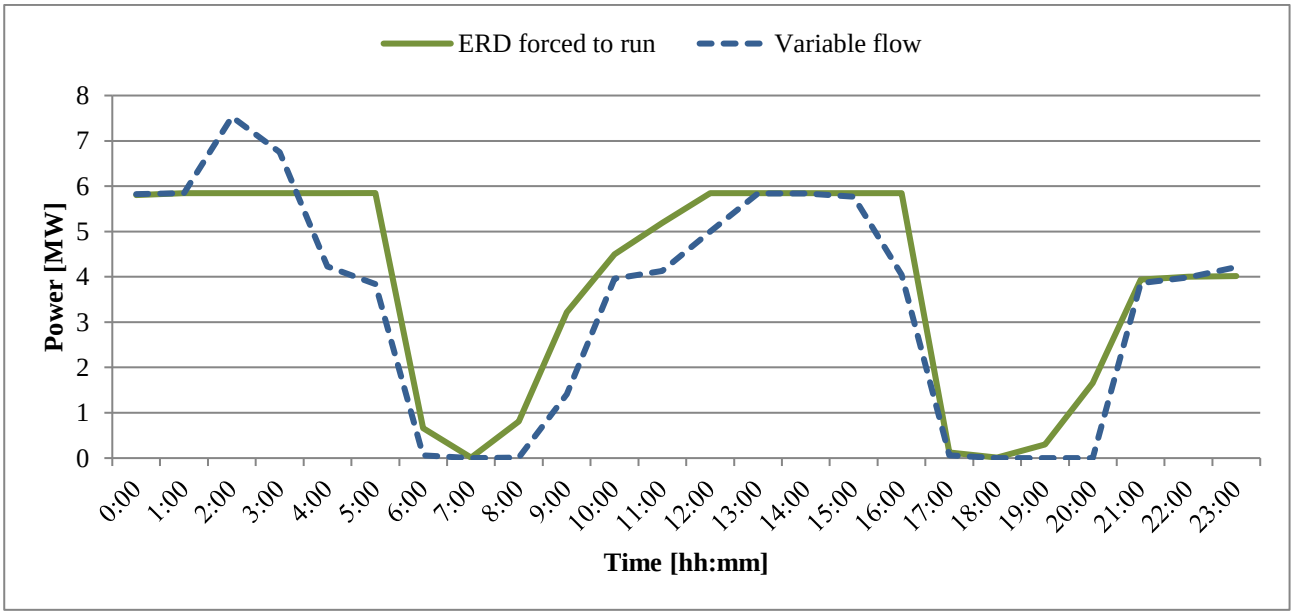



**Figure 24: Mine A – Turbine pump variable flow power profile**

Table 14 shows the minimum and maximum dam levels for the simulation.

**Table 14: Mine A – Simulation dam level analysis**

| Period | Minimum level | Maximum level |
|---|---|---|
| 38L hot dam | 45 | 80 |
| 39L cold dam | 40 | 85 |
| 75L hot dam | 35 | 85 |

Table 15 summarises the improvements seen.

**Table 15: Mine A – Turbine pump variable flow comparison**

| Parameter | Baseline simulation | ERD forced to run | Variable flow |
|---|---|---|---|
| Daily energy use [kWh] | 78900 | 92700 | 82200 |
| Peak power [kW] | 290 | 870 | 285 |
| Average flow out of mine [ℓ/s] | 145 | 172 | 145 |
| Standard period power [kW] | 3100 | 4080 | 3460 |
| Off-peak power [kW] | 5305 | 5380 | 5280 |
| Annual electricity costs [R] | 14.8-million | 18.4-million | 15.5-million |

A clear improvement can be seen from the scenario where the turbine pumps were forced to run. The overall system power usage can be seen to decrease while the load management performance increases. The variable flow strategy allows for control according to demand. Additionally, it is clear that the variable flow strategy can be used to incorporate load management initiatives into the control practices of the site.

Variable flow strategies were also tested for the simulated 3CPFS. In this case, the following assumptions were made:

- the 3CPFS has VSDs installed; and
- the speed of the pumps can be reduced by up to 40%.

Variable flow control was simulated on the 3CPFSs. The input variable for the controller was the downstream cold water dam, in this case, the 39L dam. Appropriate control parameters were selected. A profile was set for the set point of the controller. The standard and the off-peak set point were set as 80%. The peak time set point, in turn, was set as 40%.

The change in control allowed the 3CPFS to deliver minimum flow during peak times. During the rest of the day, no more flow than the demand was delivered. This in turn allowed for optimal load management performance.

Figure 25 shows the improved profile obtained from the variable flow strategy implemented.

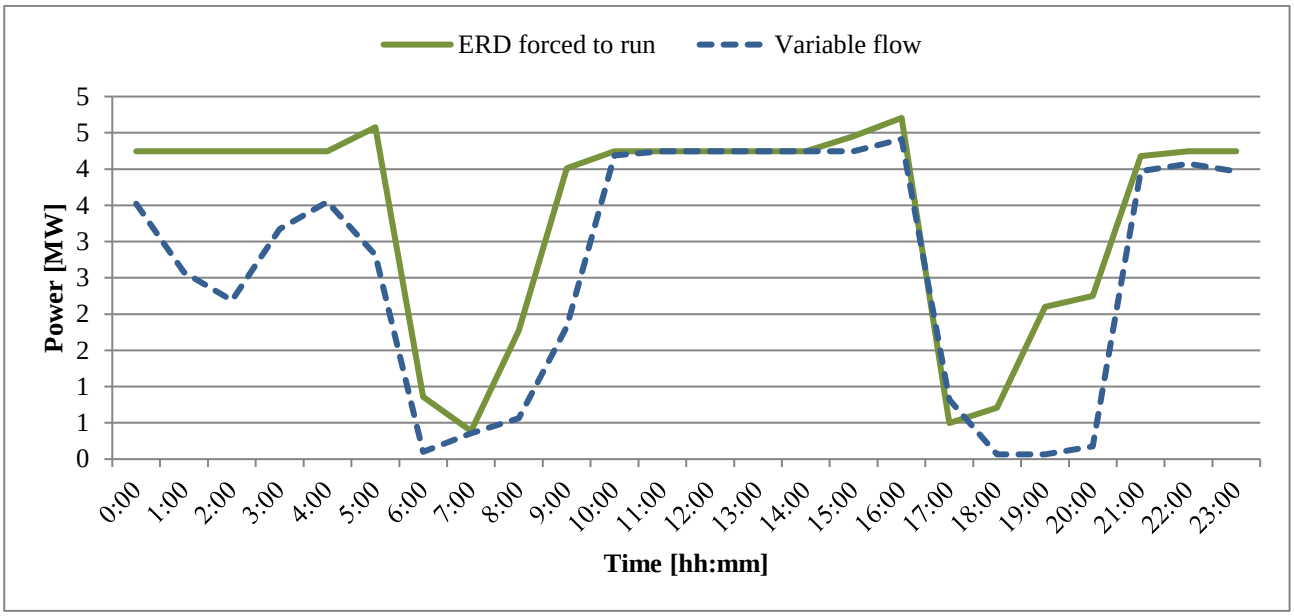



**Figure 25: Mine A – 3CPFS variable power profile**

As with the turbine pumps, a decrease in the overall energy usage of the system can be seen if variable flow strategies are incorporated into the control. Additionally, the load management performance also increased significantly. Table 16 shows the exact improvements obtained.

**Table 16: Mine A – 3CPFS variable flow comparison**

| Parameter | Baseline | ERD forced to run | Variable flow |
|---|---|---|---|
| Energy use [kWh] | 66450 | 81430 | 63630 |
| Peak power [kW] | 445 | 1795 | 575 |
| Average flow out of mine [ℓ/s] | 150 | 179 | 150 |
| Standard rate power [kW] | 3500 | 3730 | 3410 |
| Off-peak power [kW] | 3655 | 4285 | 3230 |
| Annual electricity costs [R] | 13.1 million | 17.4 million | 12.9 million |

Variable flow strategies on ERDs therefore, offer significant potential for minimising cost. The cost reduction is brought on since the ERDs can be controlled to the demand flow and incorporated into load management strategies.

## 3.5 Summary

A comprehensive methodology for the optimisation of mine dewatering systems that makes use of ERDs was presented. The methodology offered procedures to investigate and baseline any given site. Procedures to identify and integrate energy cost saving initiatives into a comprehensive control strategy were also presented.

Calibrated simulations of an optimised mine were created. The simulation accuracy was shown to be > 90% accurate. Various constraints and system alterations were made in the simulation to allow for the testing of the procedures presented in the methodology. All of the relevant procedures were verified through the use of the calibrated simulation.

This methodology can therefore, be used to optimise any mine dewatering system that incorporates ERDs. It is therefore applicable for any co-generative ERD that is directly involved in the dewatering process.

# Chapter 4: Implementation of optimisation methodology



[4]

This chapter shows the effect of the implemented solutions on actual mines. This serves as the validation of the methodology.

[4] Siemag Tecberg, "P.E.S. in mine cooling systems," 2017. [Online]. Available: http://www.siemag-tecberg.com/infocentre/technical-information/ti_14-pes-mine-cooling-systems.html. [Accessed 19 Nov 2017].

## 4.1 Preamble

In Chapter 3, a comprehensive methodology for the optimisation of dewatering systems that utilises ERDs was presented. The verification procedure implemented indicated that it could be used to optimise any dewatering system on deep level mines that makes use of co-generative ERDs. This method will now be applied to a practical case study.

## 4.2 Evaluation of Mine B

### 4.2.1 Introduction

The evaluation phase is the first step of the method. During this phase, the mine used as a case study was analysed. All of the steps presented in the methodology were implemented.

Figure 26 shows the complete underground dewatering system of Mine B. The locations of all the measuring equipment and major system components are shown (Process A.2.1). The summarised control practice is provided below with the layouts as a reference (Process A.2.3).

### 4.2.2 Mine B dewatering system

On Mine B, chilled water is supplied to the underground work areas from the surface chilled dam. This water is fed into two columns on the surface. Each column has two booster pumps at the inlet. These booster pumps provide the pressure required to overcome the losses in the two 3CPFSs located on 1200L. After exchanging pressure with the hot water through the 3CPFSs, the water is fed into the 1200L chilled dam.

From the 1200L chilled dam, the cold water is sent through the second set of booster pumps. At this point of the evaluation, it becomes necessary to differentiate between the setup of the two 3CPFSs on 1200L and 77L. The 3CPFS indicated on the left of Figure 26 will be referred to as the west 3CPFS and the one on the right-hand side as the east. This holds true for both 1200L and 77L.

The east and west 3CPFSs on 77L receive water from separate columns. Both columns have two dedicated booster and filler pumps. One backup pump is included with each of the pump sets that can pump into either column in case of a breakdown.

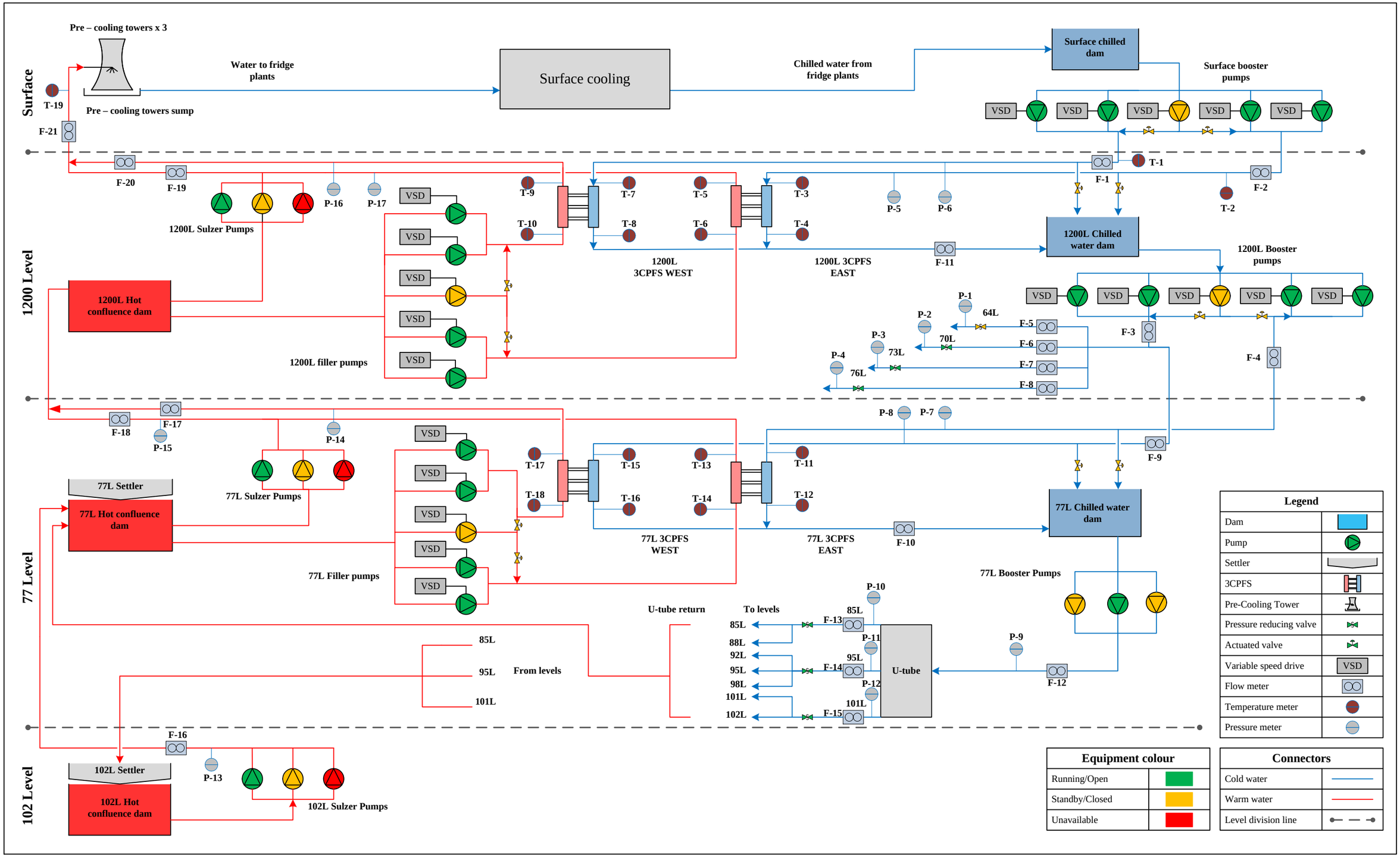



**Figure 26: Mine B – Underground water reticulation system**

The west column that feeds from 1200L provides chilled water for the mining levels between 1200L and 77L before entering the west 3CPFS on 77L. No water is bled off from the east column. The water from this column enters the 77L east 3CPFS directly. After exchanging pressure with the hot water from the 77L hot dam, both columns feed water into the 77L chilled dam.

The water is sent to the mining sections through a closed-loop high-pressure u-tube system from the 77L chilled dam. A booster pump supplies the necessary pressure to induce a flow and overcome the mechanical losses of the u-tube. Cold water is sent through underground spot coolers in the various mining sections under pressure. After passing through the air coolers, the water required for mining is bled off. The excess water is fed into a return column.

The return water is fed into hot dams on 77L. The mining water is channelled into settlers above the 102L hot dam. The settlers, in turn, overflows into the 102L hot dam. Hot water is pumped from the 102L hot dam to the 77L hot dam. This happens through the use of three large multi-stage pumps. These pumps all pump into a single column.

The 77L hot dam, therefore, receives return water from the high-pressure u-tube system as well as the water pumped from the 102L hot dam. In addition to these two sources of water, the mining water from the mining levels between 1200L and 77L feed into settlers above the 77L hot dam. These settlers overflow into the 77L hot dam.

The hot water is pumped from the 77L hot dam to the 77L east and west 3CPFS as well as the 1200L hot dam. Three multi-stage pumps pump water to the 1200L hot dam as well as the two 3CPFSs on 77L. It is important to note that the east 3CPFS water flows into the same column as the pumps. The west 3CPFS has its dedicated column.

From the 1200L hot dam, the water is pumped to the surface and the 1200L 3CPFSs. Three large multi-stage pumps and the two 3CPFSs on the 1200L pump the water directly into pre-cooling towers on the surface. Similarly to the 77L pumping station, the east 3CPFS on 1200L water flows into the same column as the 1200L pumps. The west 3CPFS on 1200L has its dedicated column.

After flowing through the pre-cooling towers, the water enters the pre-cooling towers sump. From the pre-cooling sump, the water is cooled again. After the water is cooled, the cycle is re-initiated.

Table 17 summarises the critical design specifications of the various pumps discussed (Process A.2.2).

**Table 17: Mine B – Dewatering pumps design specifications**

| Level | Pump role | Rated power [kW] | Design flow [ℓ/s] | # Pumps |
|---|---|---|---|---|
| Surface | 3CPFS booster | 315 | 160 | 5 |
| 1200L | Dewatering | 2800 | 180 | 3 |
| | 3CPFS booster | 275 | 160 | 5 |
| | 3CPFS filler | 90 | 160 | 5 |
| 77L | Dewatering | 2800 | 180 | 3 |
| | U-tube booster | 900 | 400 | 3 |
| | 3CPFS filler | 160 | 160 | 5 |
| 102L | Dewatering | 2000 | 180 | 3 |

Table 18 provides the information on the relevant dams in the system (Process A.2.2).

**Table 18: Mine B – Dam specifications**

| Level | Dam | # Dams | Total capacity [Mℓ] | Max level [%] | Min level [%] |
|---|---|---|---|---|---|
| Surface | Pre-cool sump | 1 | 9.00 | 95 | 45 |
| 1200L | Hot confluence | 2 | 7.00 | 85 | 35 |
| | Cold confluence | 1 | 5.00 | 85 | 35 |
| 77L | Hot confluence | 4 | 2.86 | 85 | 35 |
| | Cold confluence | 1 | 1.50 | 85 | 35 |
| 102L | Hot confluence | 2 | 6.74 | 85 | 25 |

A basic understanding of the surface refrigeration system is an important consideration for this study. During normal operations, the chilled water demand can be readily supplied, if there is sufficient water in the pre-cooling sump and all of the equipment is available. However, a load management initiative in the form of a thermal ice storage dam is implemented on the refrigeration system of Mine B during the evening peak period.

The thermal storage capacity of this dam is utilised during evening peak times. During the evening peak times, approximately 400 ℓ/s of chilled water is pumped through the thermal ice storage dam instead of the fridge plants. The cooling capacity during peak times is therefore reduced.

The implications of this on the dewatering system control are that the surface chilled dam needs to be as high as possible to prepare for the load management initiative and can be expected to decline during the evening peak period. This is an essential consideration for the dewatering control since the 3CPFSs on 1200L obtains water from the surface chilled dam. Additional detail on the surface refrigeration system is supplied in Appendix E.

Up to this point, all of the processes and interaction between the critical system components have been described. A basic understanding of the interaction between the REMS, the SCADA and PLCs will also be required.

Mine B has a SCADA system installed in its control room. All the dewatering equipment can be remotely stopped or started from the SCADA system. Three operators who work in eight-hour shifts manually implement control of all the equipment through the SCADA system.

The SCADA is connected to PLCs which in turn provides commands to the equipment. The traditional pumps can therefore, be stopped and started as required from the control room. Similarly, all of the flow rates of the 3CPFSs can also be controlled from the SCADA. The 3CPFSs on Mine B have a low, medium and high flow rate setting.

In addition to this, the outputs of the measuring equipment of Mine B is also available through the SCADA system and stored for a three-month period. A REMS was installed on Mine B when the detailed project investigation started. This platform can read and write values out to the SCADA through an OPC connection. The REMS can be used to control the required equipment through the SCADA and PLCs.

Mine personnel set up additional SCADA functionalities to enable effective communication between the REMS and the SCADA. A REMS enabled/disabled button was added to the SCADA. Through this controller, the operator can disable the write permission of the REMS and thereby revert to manual control if needed.

A heartbeat signal was also incorporated to ensure that the control automatically reverts to manual if any problems on the REMS server occurs. The REMS platform was also used to log critical parameters. The required parameter was logged in two-minute intervals on Mine B. This allowed the required data to be stored for periods longer than the three months that is available on the SCADA.

In addition to the mine instrumentation, third-party energy measurements of the dewatering pumps were available. The energy was measured in kilowatt-hour format. The energy data was converted to an average power over each hour to make it comparable with the on-site data.

Table 19 shows a summary of the critical parameters that were logged as part of the project (Process A.2.4).

**Table 19: Mine B – Measured parameters**

| Process variable description | Unit |
|---|---|
| Dewatering pump running status | [-] |
| 3CPFS operational status | [-] |
| Closed-loop high-pressure u-tube booster pump running status | [-] |
| 3CPFS flow set point | [ℓ/s] |
| 3CPFS flows | [ℓ/s] |
| Pump station flows | [ℓ/s] |
| Dam levels | [%] |
| Chilled water consumption per level | [ℓ/s] |
| Total chilled water into the mine | [ℓ/s] |
| Total water pumped out of the mine | [ℓ/s] |
| Total energy usage of system (excluding 3CPFS auxiliary pumps) | [kWh] |
| Power usage of 3CPFS auxiliary pumps | [kW] |

The flows and power usage of the individual pumps were not available. However, the flows of each pumping station were measured.

The running statuses of the individual pumps, the power usage of the entire system and flows of specific sets of equipment were available. The individual pumps could be characterised reasonably well by isolating periods where certain pumps ran alone.

Note that the energy usage of the 3CPFS auxiliary pumps was not included in the total energy usage. Appropriate assumptions were therefore made during the characterisation process.

### 4.2.3 Baselining of Mine A

Data for the baselining was obtained from two primary sources. The first is data logged on the REMS system installed on site. The second source is third part energy data. Historical energy data from the third party is readily available. However, data stored on the REMS server is only available from the period when it was installed.

Table 20 shows the average monthly power, evening peak power, morning peak power and peak to average ratio for 17 consecutive months. The variance of the data is also summarised in the table (Process B.1.1).

**Table 20: Mine B – Monthly power usage**

| Month | Average [kW] | Evening peak average [kW] | Morning peak average [kW] | Peak to average ratio [-] |
|---|---|---|---|---|
| Month 1 | 5340 | 2075 | 2255 | 0.41 |
| Month 2 | 5725 | 2510 | 2545 | 0.44 |
| Month 3 | 5795 | 2550 | 2605 | 0.44 |
| Month 4 | 5420 | 2080 | 2655 | 0.44 |
| Month 5 | 5670 | 2215 | 3725 | 0.52 |
| Month 6 | 5925 | 2485 | 4370 | 0.58 |
| Month 7 | 6045 | 3120 | 4745 | 0.65 |
| Month 8 | 7185 | 5675 | 6330 | 0.84 |
| Month 9 | 5960 | 3420 | 5925 | 0.78 |
| Month 10 | 6150 | 2690 | 4455 | 0.58 |
| Month 11 | 5515 | 1365 | 2250 | 0.33 |
| Month 12 | 6030 | 2160 | 3510 | 0.47 |
| Month 13 | 5300 | 1230 | 2500 | 0.35 |
| Month 14 | 4930 | 1085 | 2050 | 0.32 |
| Month 15 | 4605 | 965 | 2290 | 0.35 |
| Month 16 | 4855 | 1240 | 2280 | 0.36 |
| Month 17 | 5525 | 1760 | 2715 | 0.41 |
| **Average** | 5645 | 2270 | 3365 | 0.49 |
| **Median** | 5670 | 2160 | 2660 | 0.44 |
| **Std. deviation** | 595 | 1130 | 1350 | 0.15 |

Manual optimisation studies on Mine B started at the beginning of month 10. During this time, the manual load management that was already in place was improved upon. The project baseline had to be chosen from data pre-dating this period.

Only values that fall within a confidence interval of 1 of the average value was considered for a baseline period. Additionally, month 7 was also disregarded since the mine closed for an extended period of this month.

Month 5, 6 and 9 were therefore selected as the baseline period for this study since these are the most recent months before the investigations started and fell within one standard deviation of the mean.

The power profile of this period is depicted in Figure 27 (Process B.2.1).

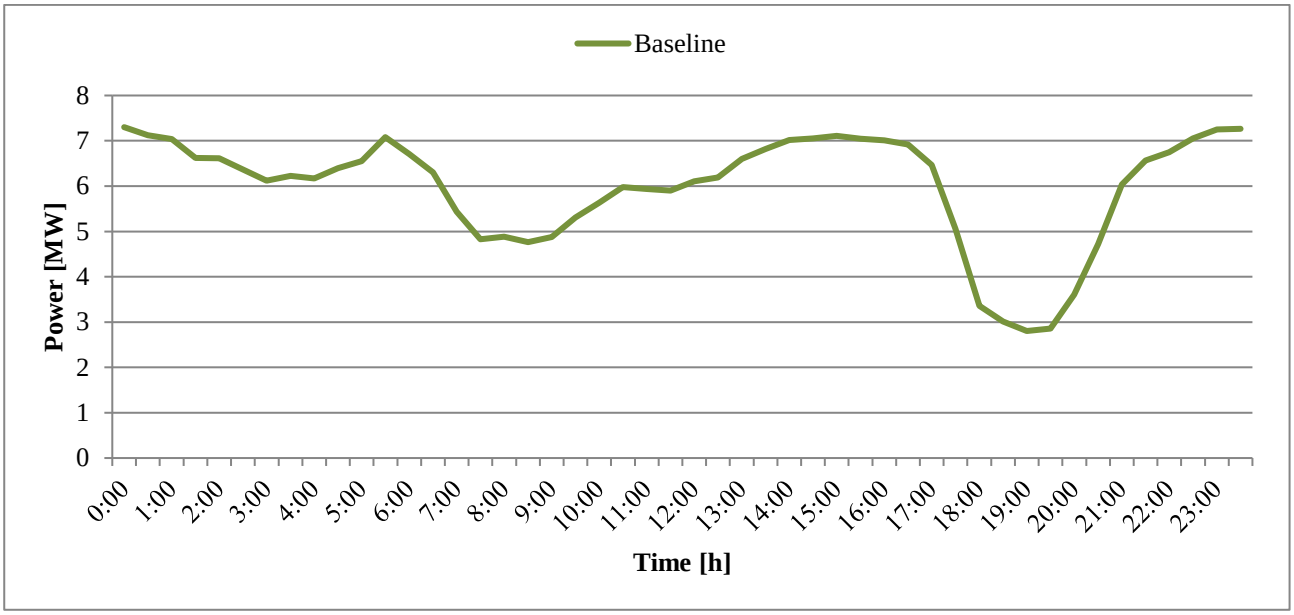



**Figure 27: Mine B – Baseline power profile**

The average power usage of the period, as well as the two peak periods and the peak to average ratio of the period is shown in Table 21.

**Table 21: Mine B – Baseline power comparison**

| Parameter | Value |
|---|---|
| Average power [kW] | 5975 |
| Average evening peak power [kW] | 3010 |
| Average morning peak power [kW] | 5015 |
| Peak to average ratio [-] | 0.70 |

The third party energy data was used to calculate the power values of the individual pieces of equipment. Since a single tag value included all of the pumps in the system, the power data for individual pumps were averaged when the applicable pump or pump combination was running alone.

If the pump never ran alone in the available data set, the power usage of the pump in combination with specific pumps with known averages was determined. The difference between the two averages was calculated to determine the average power of the specific pump.

Table 22 shows these values (Process B.2.2).

**Table 22: Mine B – Actual pump flow and power**

| Pump | Pump flow [ℓ/s] | Pump power [kW] |
|---|---|---|
| 77L Booster pump 3 | 330 | 757 |
| 77L Booster pump 2 | 370 | 826 |
| 102L 1 dewatering pump | 180 | 1843 |
| 102L 2 dewatering pumps | 340 | 3703 |
| 77L 1 dewatering pump | 180 | 2726 |
| 77L 2 dewatering pumps | 340 | 5122 |
| 1200L 1 dewatering pump | 180 | 2882 |
| 1200L 3CPFS auxiliaries | 250 | 746 |
| 77L 3CPFS auxiliaries | 260 | 879 |

Note that the active measurements of the 3CPFSs power were not reliably available due to PLC communication errors. However, measurements of the power usage of the 3CPFSs at the high set points could be obtained. It was therefore assumed that the auxiliary pumps of the 3CPFs would adhere to Equation 5 (found in Chapter 1.3) regarding power usage and flow for the other set points.

However, because the 3CPFS power usage could not be adequately verified in the simulation, the power usage of the 3CPFSs as found on Mine B will not be compared throughout the rest of this study. Therefore, the effect of the 3CPFS on the other components in the dewatering system will be shown, but the power usage of the 3CPFS itself will not be included in any system comparisons or cost calculations.

Although manual load management improvements started immediately after the investigations, an optimal load management strategy still had to be developed. Since flow data of the mine was not available over the baseline period, the average flows over a three-month available period were analysed to see what the variance in the flow typically is.

Table 23 summarises the findings.

**Table 23: Mine B – Analyses of the variance in flow data of Mine B**

| Parameter | Cold water to underground [ℓ/s] | Hot water out of mine [ℓ/s] | Energy usage [kWh] |
|---|---|---|---|
| Average | 436.7 | 458.5 | 124150 |
| Std. deviation | 26.6 | 32.9 | 20530 |
| Std. deviation percentage of average | 6.0 | 7.0 | 17.0 |

The variance in the flow data is relatively small. The standard deviation over the analysed period is only 6% for the flow to underground and 7% for the hot water out of the mine. This indicates that the flow usage for the mining activities stays relatively constant.

The variance of the energy usage of the system is significant compared to that of the flow. The literature and simulations conducted in Appendix D indicated that a linear relationship between flow and power usage exists on dewatering systems with no load management.

A linear regression model was compiled to determine the correlation between the mine flow demand and typical energy usage on Mine B.

Figure 28 provides a visual representation of the model.

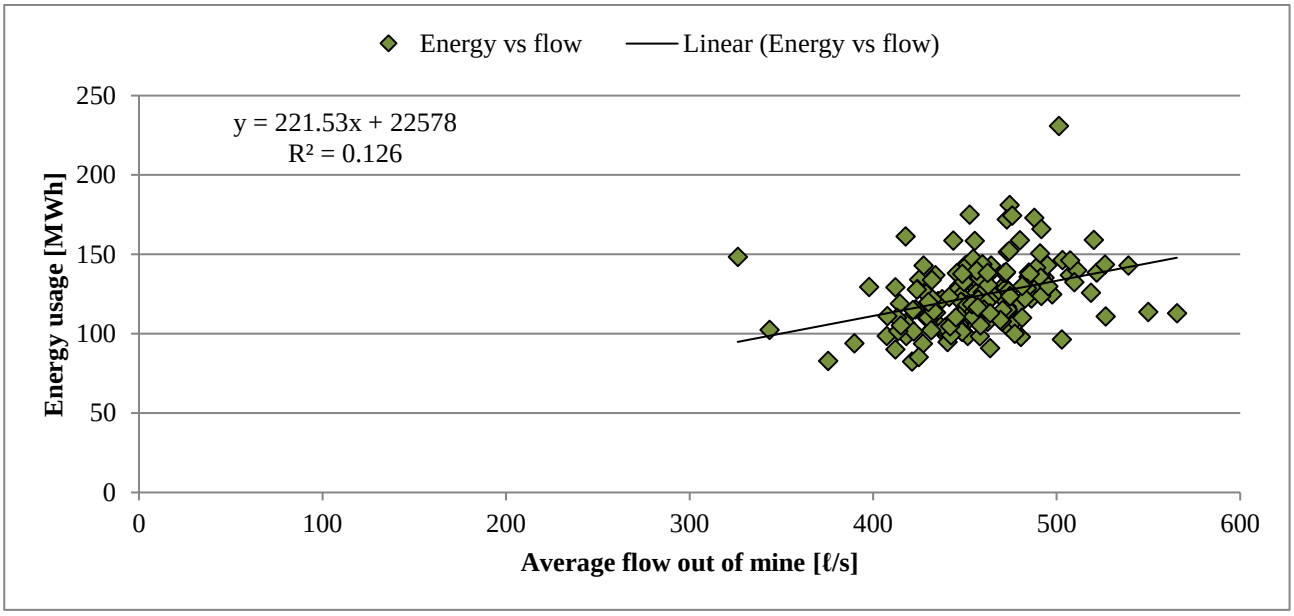



**Figure 28: Mine B – Relation between flow demand and energy usage on Mine B**

$R^2$ value of the regression line is shown to be approximately 0.126. This indicates that almost no correlation exists between the flow and power usage for the available data on Mine B. Given the small variance in the flow compared to that of the energy usage and the weak correlation between flow and energy usage, other likely factors have a significant influence on the energy usage of the system.

Such influences could include:

- sporadic unavailability of the 3CPFSs; or
- changes in efficiency due to different running configurations.

It is clear that the flow demand and therefore the total water sent to underground and dewatered from Mine B varies relatively little. Therefore, flow data from periods in the available datasets will be used in the baseline simulations even though it does not correspond to the primary baseline period.

The reference period used for the optimisation process was therefore selected from days where the ESCO was already involved and not from the same period as the profiled power baseline. Three suitable days were identified.

Before the data from the suitable reference days were used as inputs into the simulations, a careful mass balance of the system was conducted to identify any inconsistencies.

Table 24 summarises the average flows calculated from reliable data (Process B.3.1).

**Table 24: Mine B – Baseline demand flows**

| Parameter | Flow [ℓ/s] |
|---|---|
| Flow to underground | 435 |
| Flow from underground | 444 |
| Water to middle mine | 80 |
| Water to main mining sections | 353 |
| 102L settler inflow | 182 |
| U-tube return to 77L hot dam | 171 |
| Fissure water to 102L settler | 9 |

It was found that the flows measured in and out of the mine matched the flows into and out of the surface pre-cooling dam and chilled dams relatively well. These flows were therefore deemed accurate.

However, the flows out of the 1200L chilled dam and the 77L chilled dam were found to be inconsistent. The average measured water flow out of the 77L cold dam differed by 25% to that measured into the dam. It was assumed that the measured flows into the mining areas between 1200L and 77L were accurate since no manual measurements of these flows were possible. Using the dam levels, dam volumes and the available accurate flows, hourly average flows between the levels could be calculated using Equation 8.

The total flow into the main mining area below 77L was therefore, a calculated value. The total measured flow out of the 77L cold dam was scaled with a constant value. The data from the three identified days were processed into hourly profiles, ready to be used as simulation inputs (Process B.3.4).

Note that the entire middle mine flow profile was assumed to end up in the 77L settlers. The flow to the lower mine split up into mining water and high-pressure u-tube return water. The u-tube return water was assumed to remain fixed at 171 ℓ/s, and the remainder of the water therefore, returned to the 102L hot dam.

Figure 29 shows the flow demands into the relevant mining areas that served as the simulation inputs (Process B.3.5).

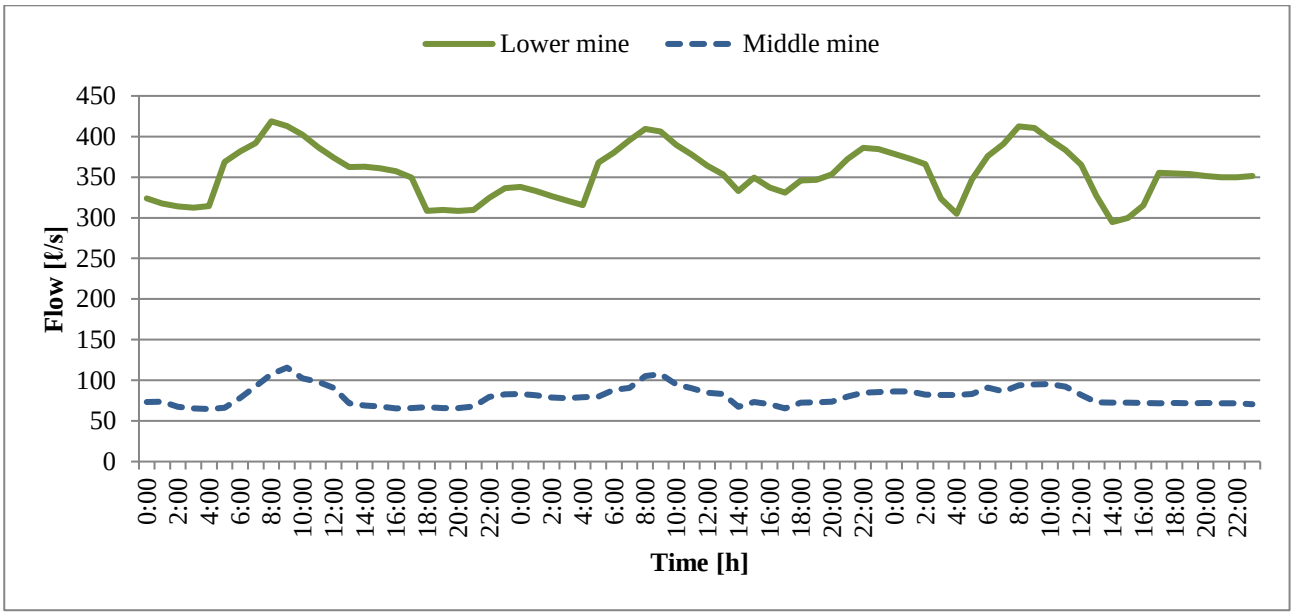



**Figure 29: Mine B – Actual demand flows - Mine B**

### 4.2.4 Preliminary initiative identification

Table 25 shows the preliminary load management and WSO ratios (Process C.1 and C.2).

**Table 25: Mine B – Load management and WSO preliminary tests**

| Parameter | Value |
|---|---|
| $R_{LM}$ | 0.65 |
| $R_{WSO}$ | 0.84 |

The load management ratio indicates that investigations into additional load management might be justified. However, the WSO possibilities required a more detailed investigation.

Detailed audits of the underground spot coolers on Mine B were therefore conducted to see if any potential for reduced flow exists. Measurements of critical spot coolers were conducted to quantify the typical effectiveness of the spot coolers on Mine B.

The studies focus on the spot coolers located in the lower mine section since this was the most significant consumer of chilled water on Mine B.

Table 26 summarises the findings of the audits (continuation of Process C.2).

**Table 26: Mine B – Measured spot cooler efficiencies**

| Spot cooler | Air inlet temperature [°C] | Air inlet RH [%] | Air outlet temperature [°C] | Water inlet flow [ℓ/s] | Water inlet temperature [°C] | Water outlet temperature [°C] | Spot cooler efficiency |
|---|---|---|---|---|---|---|---|
| 1 | 29.07 | 98.81 | 24.50 | 15.24 | 20.00 | - | - |
| 2 | 29.07 | 98.85 | 24.80 | 15.24 | 20.00 | - | - |
| 3 | 28.32 | 100.00 | 17.90 | 18.08 | 11.50 | 17.90 | 38.00 |
| 4 | 28.33 | 100.00 | 17.90 | 18.08 | 11.50 | 17.90 | 38.00 |
| 5 | 29.28 | 100.00 | 23.95 | 17.29 | 14.31 | 16.57 | 15.00 |
| 6 | 23.91 | 100.00 | 17.89 | 15.65 | 13.48 | 16.60 | 30.00 |
| 7 | 32.72 | 100.00 | 25.42 | 9.16 | - | - | |
| 8 | 32.12 | 100.00 | 25.71 | 9.33 | 13.63 | 20.08 | 35.00 |
| 9 | 33.10 | 100.00 | - | 9.33 | - | - | |

Spot cooler 1 and 2, as listed in Table 26, were disconnected from the u-tube return column due to blockages in the return column. This leads to inefficiencies since the surplus water flooded the immediate surroundings of the spot coolers. This water had to be drained to the 102L hot dam and was not returned to the 77L hot dam.

In addition to this, Mine B makes use of centralised blasting. This means that all of the working areas are evacuated during the blasting shift. This in turn leads to a reduced ventilation cooling demand during this period. It can therefore be assumed that the flow to the mining areas can be reduced in this period.

Two ways of reducing the flow to the mining sections exist on Mine B. The first is switching off the 77L booster pump that feeds the high-pressure u-tube system. This is a drastic measure and could lead to significant increases in underground air temperatures. More thorough investigations and tests would have to occur before such measures can be recommended and was not considered for this study due to time and resource constraints.

However, investigations showed that a second possibility exists. Mine B utilises one of three booster pumps on 77L at any given time. Booster pump 1 serves mainly as a backup and is rarely used. Booster pump 2 is used for the majority of the time. Booster pump 3, in turn, has a permanently throttled discharge valve. This reduces the average flow through this pump by approximately 40 ℓ/s.

Presently booster pump 3 is used at the discretion of the operators and therefore primarily used when the chilled water dam on 77L is at risk of running empty. The flow into the mining areas is therefore not controlled to the demand but to a fixed rate.

However, the mine personnel required additional studies into the potential impact of such initiatives. Detailed simulation of the ventilation systems does not fall within the scope of this study. However, the potential impact of the identified initiatives was simulated.

### 4.2.5 Baseline simulation development

After the evaluation and preliminary investigations were concluded, a simulation model of Mine B was constructed. The subsystems in the simulation were characterised to match the information shown in Table 22.

Table 27 compares the simulated flows values to the actual values (Process D.1.1 and D.1.2).

**Table 27: Mine B – Simulated vs actual pump flows**

| Pump | Actual flows flow [ℓ/s] | Simulated flows [ℓ/s] | Difference [%] |
|---|---|---|---|
| 77L Booster pump 3 | 330 | - | - |
| 77L Booster pump 2 | 370 | - | - |
| 102L 1 dewatering pump | 180 | 181 | 1 |
| 102L 2 dewatering pumps | 340 | 360 | 6 |
| 77L 1 dewatering pump | 180 | 183 | 1 |
| 77L 2 dewatering pumps | 340 | 360 | 6 |
| 1200L 1 dewatering pump | 180 | 180 | 1 |
| 1200L 3CPFS auxiliaries | 250 | 250 | 1 |
| 77L 3CPFS auxiliaries | 260 | 260 | 1 |

Table 28 compares the simulated power to the actual power of the pumps (Process D.1.1 and D.1.2).

**Table 28: Mine B – Simulated vs actual pump power**

| Pump | Actual power [kW] | Simulated power [kW] | Difference [%] |
|---|---|---|---|
| 77L Booster pump 3 | 757 | - | - |
| 77L Booster pump 2 | 826 | 860 | 4 |
| 102L 1 dewatering pump | 1843 | 1855 | 1 |
| 102L 2 dewatering pumps | 3703 | 3710 | 1 |
| 77L 1 dewatering pump | 2726 | 2714 | 1 |
| 77L 2 dewatering pumps | 5122 | 5336 | 4 |
| 1200L 1 dewatering pump | 2882 | 2800 | 3 |
| 1200L 3CPFS auxiliaries | 746 | 740 | 1 |
| 77L 3CPFS auxiliaries | 879 | 860 | 2 |

Note that the available power data for the 3CPFS auxiliary pumps was questionable. However, the measured power data of the 3CPFS auxiliary pumps correlates reasonably well to the design specifications when running at maximum flow. These conditions therefore characterised the pumps.

The dams, pipes and valves found in the dewatering system of Mine B were also included in the simulation. The actual dam volumes as summarised in Table 18 were programmed into the generic Process toolbox components. The resistance of the pipe networks was empirically characterised in unison with the pumps to ensure that the power and flow of each pump match the actual values.

The actual demand flows into the various mining sections as calculated earlier was provided as simulation inputs. A preliminary control strategy was set up. This control strategy controlled all of the components within the allowed limits. Note that this preliminary control will not add value to the study and will not be provided.

Table 29 compares the average critical parameters of the baseline simulation to the actual profile over the three-day reference period (Process D.1.3).

**Table 29: Mine B – Baseline simulation accuracy verification**

| Parameter | Actual | Simulated baseline | Percentage difference [%] |
|---|---|---|---|
| Pump power usage [kW] | 4785 | 4870 | 1.9 |
| 3CPFS power usage [kW] | - | - | - |
| Flow out of mine [l/s] | 445.0 | 450.0 | 1.1 |

The average power usage of the system and flow out of the mine are both within a 2% error margin. The peak power usage differs significantly since the power profile does not match the actual profile as was already explained. The accuracy of the average simulated power usage and flow out of the mine shows that the simulation is calibrated.

### 4.2.6 Summary

The system has been evaluated and analysed. A representative system baseline has been created from available data. An appropriate simulation reference period has been identified. The relevant data from this period has been processed as required by the methodology and used as inputs for a baseline simulation.

All of the components in the dewatering system of Mine B was characterised. A baseline simulation was set up that consists of all the characterised components of Mine B. The processed data from the identified reference period was set as demand flow inputs. A preliminary control strategy was applied on the characterised components. The baseline simulation was shown to reflect the actual average power usage accurately and flows of the system.

## 4.3 System optimisation

### 4.3.1 Introduction

Two main initiatives were shown to be feasible on Mine B. The first is improved load management. The second is a WSO initiative aiming to improve the control into the mining areas. The load management initiative required improved control of the entire dewatering system. The WSO initiative in turn focused mainly on the control of the booster pumps into the mining areas.

Therefore, two separate simulations were constructed. The first utilised the optimisation methodology to derive an optimal control strategy for the dewatering equipment. In the second simulation, the control of the booster pumps into the mining areas was altered to show the effect of the possible WSO initiative.

### 4.3.2 Load management control strategy

To optimise the control, procedure D.2.1 was therefore applied on Mine B, with the additional preliminary assumption that there are not WSO opportunities available on the mine. Therefore, no alterations were made to the demand flow. The aim was therefore to create an optimal control strategy for the dewatering pumps and 3CPFSs.

Note that the 77L booster pumps that feed the closed-loop high-pressure u-tube system on Mine B were not controlled. In addition to this, switching off the 3CPFSs was not allowed, however, variable flow control on the 3CPFSs could be applied since all the auxiliary pumps had VSDs installed. In addition to the flow set points, all of the dewatering equipment could be controlled.

The following critical system considerations were identified as essential to consider for the control development during the evaluation phase (considerations for Process D.2.1):

- sufficient capacity had to be available in the 102L hot dam to allow the 102L Sulzer pumps to be shut off during peak times;
- the 77L hot dam needed enough water in it to allow the 77L 3CPF systems to keep on running through peak times;

- the 1200L hot dam needed enough water in it to allow the 1200L 3CPF systems to keep on running through peak times;
- the level of the pre-cooling tower sump has to be low enough to allow for a nett influx of water during peak times;
- the level of the surface chilled dam should be high enough to allow for a nett decrease in water during peak times;
- the underground chilled dams should always have enough water in it for the mine operations to perform as required; and
- the flow through the east 3CPFS on both 1200L and 77L is limited if pumps on the same level run since it pumps into a common manifold.

### Simulation results

Table 30 shows the critical simulated results (Process D.2.3). Note that the actual baseline was scaled energy neutrally to match that obtained in the baseline simulation since only load management strategies were considered in this phase. Each of the measured power values over the baseline period was therefore scaled with a constant value. The average power usage of the baseline matches the average power usage of the baseline simulation as a result of this scaling.

**Table 30: Mine B – Simulated optimised profile**

| Parameter | Baseline | Simulated optimised | Difference |
|---|---|---|---|
| Pump power usage [kW] | 4870 | 5000 | 130 |
| Morning peak power usage [kW] | 4195 | 920 | 3275 |
| Evening peak power usage [kW] | 2515 | 855 | 1660 |
| Peak to average ratio [-] | 0.7 | 0.2 | 0.5 |
| Flow out of mine [l/s] | 445 | 450 | 5 |
| Annual electricity costs [R] | 26.0 million | 23.9 million | 2.1 million |

The optimisation procedure delivered a R2.1 million annual cost improvement through load management.

Figure 30 shows the simulated optimised profile that was obtained (Process D.2.3).

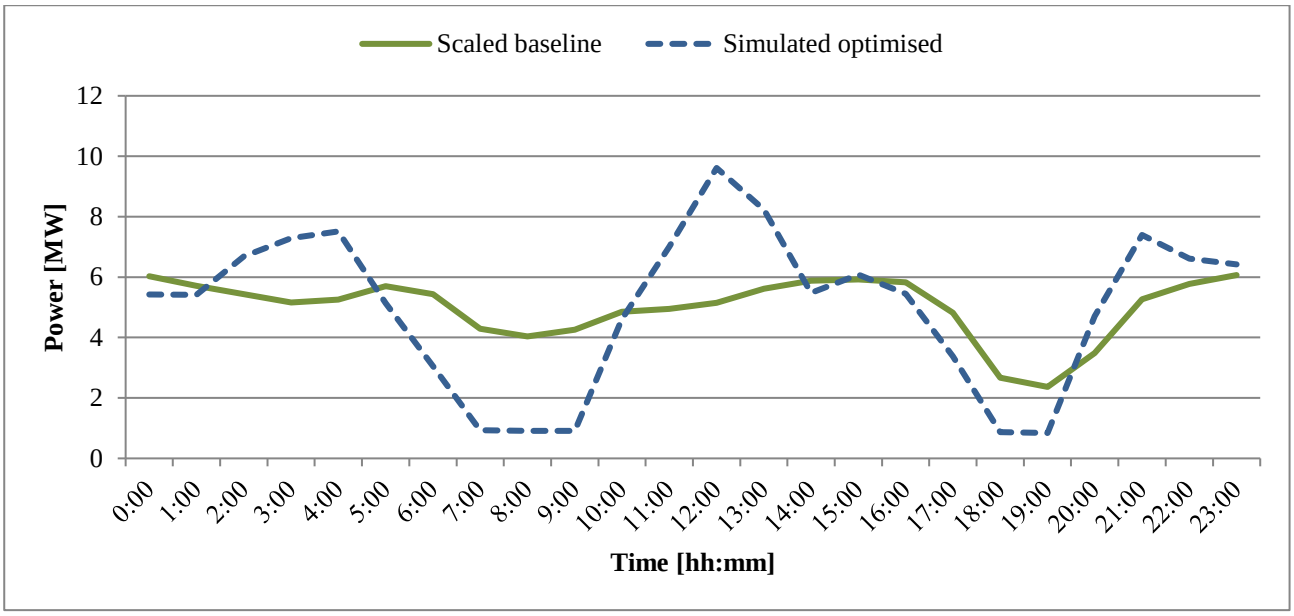



**Figure 30: Mine B – Simulated optimised profile**

It is clear that a full load shift was not achievable. The 77L booster pumps are included in the power profile. This pump feeds the high-pressure u-tube and is never switched off. The power usage of the booster pumps therefore formed baseload power usage of the system.

Figure 31 shows the pump schedules over the applicable days (Process D.2.3).

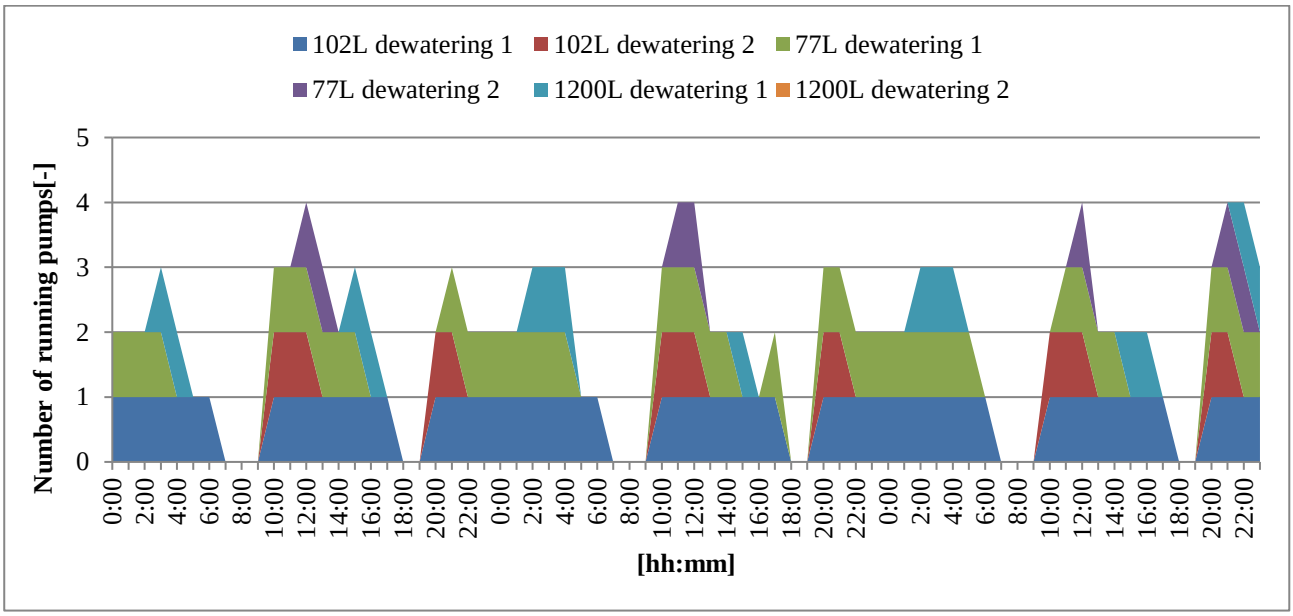



**Figure 31: Mine B – Simulated pump schedules**

Note that at times, some of the pumps ran for relatively short periods. However, the time scale on the x-axis of Figure 31 is compact, and none of the pumps ever ran for a period shorter than half an hour. The pumps never stopped and started irregularly during the simulation period.

Table 31 summarises the minimum and maximum dam levels obtained over the simulated period (Process D.2.3).

**Table 31: Mine B – Simulated minimum and maximum dam levels**

| Dam | Maximum level [%] | Minimum level [%] |
|---|---|---|
| 1200L cold water dam | 84.7 | 55.4 |
| 1200L hot water dam | 80.0 | 57.0 |
| 77L cold water dam | 69.0 | 54.7 |
| 77L hot water dam | 84.9 | 67.4 |
| 102L hot water dam | 60.3 | 25.5 |
| Pre-cooling tower sump | 54.0 | 30.8 |

It is clear that all of the dam levels remained within the allowable levels in the simulated period. The optimised control strategy was therefore deemed sufficient. An overview of the developed control will now be provided.

Upstream control was implemented on the dewatering pumps of all the pumping stations. The minimum and maximum number of pumps on the levels served as an additional control measure. The control logic for the 3CPFSs considered multiple dam levels simultaneously before a flow set point is set.

The following control mechanisms were quantified (final iteration of Process D.2.2):

- REMS controller parameters for each dewatering pump station;
- maximum number of pumps allowed to run on a specific level;
- minimum number of pumps allowed to run on a specific level; and
- 3CPFS flow set point control logic.

### 102L dewatering pump control

Table 32 shows the control logic for determining the maximum number of pumps on 102L.

**Table 32: Mine B – 102L maximum number of dewatering pumps**

| 77L Hot dam level | 102L Hot dam level | Maximum no. of pumps |
|---|---|---|
| ≥ 88 | ≥ 90 | 1 |
| | ≥ 80 | 0 |
| | < 80 | 0 |
| ≥ 85 | ≥ 90 | 1 |
| | ≥ 80 | 1 |
| | < 80 | 0 |
| ≥ 40 | ≥ 90 | 2 |
| | ≥ 80 | 2 |
| | < 80 | 2 |
| < 40 | ≥ 90 | 3 |
| | ≥ 80 | 2 |
| | < 80 | 2 |

The maximum number of pumps on 102L is a failsafe to ensure that the number of pumps running on the level at any given time remains within the allowed limits. If the control strategy works as it is designed to do, the maximum number of pumps on 102L should remain at 2.

Table 33 shows the control logic for the minimum number of pumps on 102L.

**Table 33: Mine B – 102L minimum number of dewatering pumps**

| 77L Hot dam level | 102L Hot dam level | Minimum no. of pumps |
|---|---|---|
| < 40 | ≥ 35 | 1 |

The minimum number of pumps ensures that the level of the 77L hot dam remains within the allowed limits and is required since the 3CPFSs will never be switched off. Therefore, if all of the pumps are switched off during peak time and 77L 3CPFSs dewaters the 77L hot dam too fast, available water in the 102L dam will be pumped up.

The minimum and maximum dam level logic make use of top stable value inputs of the dams, with a stable offset of 10%. This is to avoid cycling. Therefore, if the 77L hot dam dips below the control parameter of 40%, a pump will remain active on 102L until the level reaches 50%. The REMS controllers will make use of the actual dam levels as the control parameters.

Table 34 shows the REMS controller logic for the pumps on 102L.

**Table 34: Mine B – 102L dewatering pumps REMS controller control logic**

| Parameter | Value |
|---|---|
| Control range [%] | 10 |
| Top offset [%] | 5 |
| Bottom offset [%] | 5 |
| Maximum level [%] | 80 |
| Minimum level [%] | 30 |

Note that the maximum and minimum levels are control parameters as described in Chapter 2. The controller therefore, aims to control the level of the dam between 25% and 45% during standard and off-peak periods. In peak periods, it is controlled between 65% and 85 %. This leads to the load being shifted out of the peak periods as required.

### 77L dewatering pump control

Table 35 shows the control logic for determining the maximum number of pumps on 77L.

**Table 35: Mine B – 77L maximum number of dewatering pumps**

| 1200L Hot dam level | 77L Hot dam level | Maximum no. of pumps |
|---|---|---|
| ≥ 92 | ≥ 80 | 1 |
| | ≥ 70 | 0 |
| | < 70 | 0 |
| ≥ 86 | ≥ 80 | 2 |
| | ≥ 70 | 1 |
| | < 70 | 0 |
| ≥ 40 | ≥ 80 | 2 |
| | ≥ 70 | 2 |
| | < 70 | 1 |
| < 40 | ≥ 80 | 3 |
| | ≥ 70 | 2 |
| | <7 0 | 2 |

Unlike the 102L control parameters, the maximum number of pumps on 77L is not exclusively in place to ensure for emergency scenarios and is therefore actively incorporated into the rest of the control strategy.

Table 36 shows the control logic for the minimum number of pumps on 77L.

**Table 36: Mine B – 77L minimum number of dewatering pumps**

| 1200L Hot dam level | 77L Hot dam level | Minimum no. of pumps |
|---|---|---|
| < 40 | ≥ 35 | 1 |

The minimum number of pumps limit is in place on 77L for similar reasons as the 102L control limits. The minimum number of pumps ensures that the 1200L hot dam level does not exceed the allowed limits. As with the 102L controllers, top stable values were used for the minimum and a maximum number of pumps control logic, and actual dam levels for the REMS controller.

Table 37 shows the REMS controller logic for the pumps on 77L.

**Table 37: Mine B – 77L dewatering pumps REMS controller control logic**

| Parameter | Value |
|---|---|
| Control range | 20 |
| Top offset | 10 |
| Bottom offset | 5 |
| Maximum level | 85 |
| Minimum level | 55 |

Note that the minimum level that was set on 77L is significantly higher than that set on 102L. The 3CPFSs is the reason for this. The preparation for a dam with a co-generative ERD differs to regular control in this regard. The dam needs to remain full to ensure that it does not empty completely during peak times.

### 1200L dewatering pump control

Table 38 shows the REMS controller logic for the pumps on 1200L.

**Table 38: Mine B – 1200L dewatering pumps REMS controller control logic**

| Parameter | Value |
|---|---|
| Control range | 10 |
| Top offset | 5 |
| Bottom offset | 15 |
| Maximum level | 80 |
| Minimum level | 60 |
| Minimum number of pumps | 0 |

Note that the minimum number of pumps on 1200L is fixed at 0 since the pumps feed into the pre-cooling dams which are not connected to a 3CPFS. The reasoning for the rest of the control parameters is similar to that on 77L.

Table 39 shows the control logic for determining the maximum number of pumps on 1200L.

**Table 39: Mine B – 1200L maximum number of dewatering pumps**

| Pre-cool sump level | 1200L Hot dam level | Maximum no. of pumps |
|---|---|---|
| ≥ 92 | ≥ 80 | 1 |
| | ≥ 70 | 0 |
| | < 70 | 0 |
| ≥ 86 | ≥ 80 | 1 |
| | ≥ 70 | 1 |
| | < 70 | 0 |
| ≥ 40 | ≥ 80 | 2 |
| | ≥ 70 | 2 |
| | < 70 | 1 |
| < 40 | ≥ 80 | 3 |
| | ≥ 70 | 2 |
| | < 70 | 2 |

### 77L 3CPFS control

Table 40 shows the control logic for the 77L 3CPFSs.

**Table 40: Mine B – 77L 3CPFS control logic**

| Downstream CW level | Upstream CW level | Upstream HW level | Downstream HW level | 3CPFS East Flow SP | 3CPFS West Flow SP |
|---|---|---|---|---|---|
| > 70 | ≥ 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | | < 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | < 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | | <40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| ≥ 55 | ≥ 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | Medium | Medium |
| | | < 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | < 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | | < 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| < 55 | ≥ 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | High | High |
| | | <40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | < 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | | < 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |

The sequence in which the dams are evaluated is clearly shown in Table 40. The sequence prioritises the control to the downstream cold water dam level and the demand. Note that all of the input parameters in the control logic are top stable dam level values with a stable offset of 10%. As with the minimum and a maximum number of pumps, this will ensure that cycling between flow set points does not occur.

The amount of conditions might seem excessive. However, it was set up in such a way as to quickly make changes and add additional conditions if changes in the system do occur or if it is deemed necessary at a later stage during the project progress. The control was set up to control strictly to the demand. High and medium set points are therefore only considered as an option if the dam levels are below the levels described in Table 41.

### 1200L 3CPFS control

Table 41 shows the control logic for the 1200L 3CPFSs.

**Table 41: Mine B – 1200L 3CPFS control logic**

| Downstream CW level | Upstream CW level | Upstream HW level | Downstream HW level | 3CPFS East flow SP | 3CPFS West flow SP |
|---|---|---|---|---|---|
| > 70 | ≥ 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | | < 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | < 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | | < 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| ≥ 55 | ≥ 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | Medium | Medium |
| | | < 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | < 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | | < 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| < 55 | ≥ 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | High | High |
| | | < 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | < 40 | ≥ 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |
| | | < 40 | ≥ 85 | Low | Low |
| | | | < 85 | Low | Low |

Top stable control with a stable offset was implemented as with the control on the 77L 3CPFSs.

**Start-up delay**

A delay sequence was programmed between the pumps installed on the same levels. If pumps installed on the same level start up simultaneously, it can result in a power failure. The delay is set to 5 minutes per electrical feeder. This start-up delay condition was not included in the simulation but is a necessary control parameter nevertheless.

### 4.3.3 Simulation of WSO initiative

To quantify the potential impact of a WSO initiative on Mine B, the optimised control strategy developed in Chapter 4.3.1 was used as a reference. Based on the detailed WSO investigations, the following assumptions were made:

- The flow into the 102L hot dam can be reduced by 15 ℓ/s (cooling cars return columns were disconnected);
- The total flow through the high-pressure u-tube system can be reduced by 40 ℓ/s during the blasting shift of the mine.

The reduced flow to the 102L dam involved altering the distribution of water within the simulation. The 40 ℓ/s flow reduction in the blasting shift, in turn, entailed switching to the 3rd booster pump within the simulation during the allotted times.

Table 42 shows the simulation results (Process D.2.1). Note that the simulation of Mine B with the optimised control serves as the baseline for this comparison.

**Table 42: Mine B – WSO simulation results**

| Parameter | Baseline | Simulated WSO | Difference |
|---|---|---|---|
| Pump power usage [kW] | 5000 | 4700 | 300 |
| Morning peak power usage [kW] | 920 | 935 | 15 |
| Evening peak power usage [kW] | 855 | 795 | 60 |
| Peak to average ratio [-] | 0.18 | 0.19 | 0.1 |
| Flow out of mine [l/s] | 467 | 452 | 15 |
| Annual electricity costs [R] | 23.9 million | 22.9 million | 1 million |

It is clear that an additional R1 million annual savings can be obtained through the implementation of the initiatives. Note that this saving excludes saving possibilities on the surface refrigeration system due to the reduced flow and is therefore conservative.

Figure 32 shows the obtained power profiles (continuation of Process D.2.3).

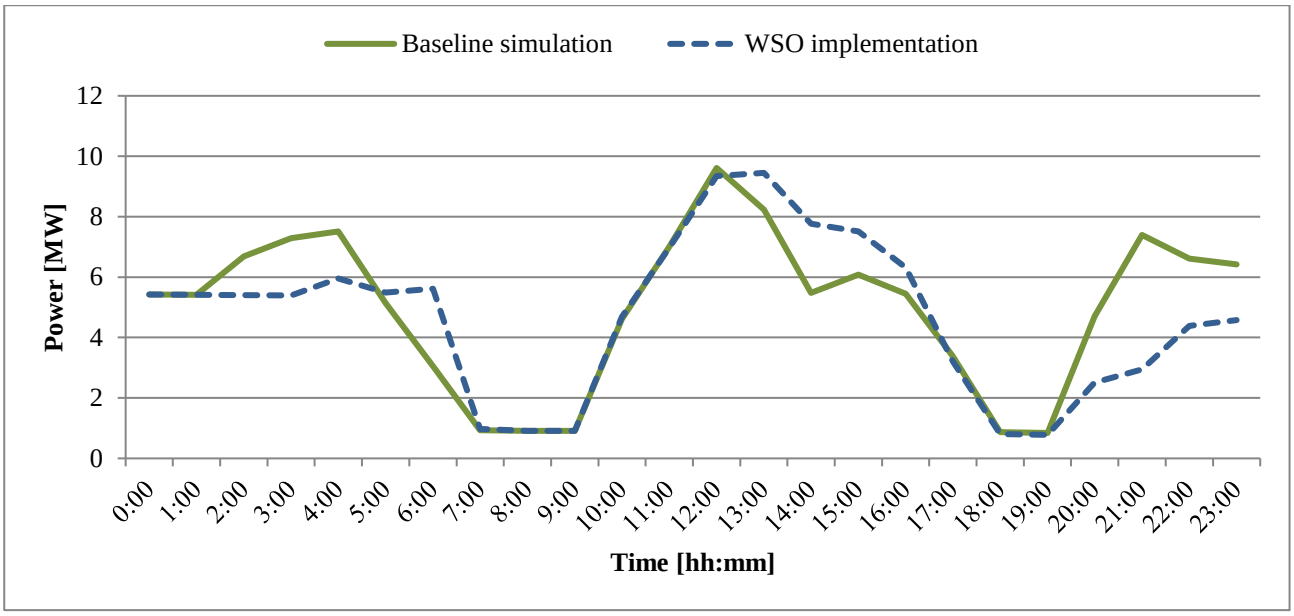



**Figure 32: Mine B – WSO simulation results**

Figure 32 shows the expected changes in the power profile compared to that of the simulated optimised profile. It is clear that the comeback load after the evening peak period is reduced if the WSO initiative is implemented. This can be attributed to the reduced flow during this period. The correlation between the two profiles during the rest of the day is similar.

### 4.3.4 Summary

The optimisation methodology developed was implemented on Mine B. The entire dewatering system was characterised and simulated. The simulations showed that a R2.2 million cost reduction is achievable through the implementation of load management initiatives. An additional cost reduction through water supply reductions of R1 million p.a. was also shown to be achievable.

A detailed control strategy was developed and optimised using the developed simulation. The control strategy stipulated control parameters for all of the dewatering equipment. The required outcomes of procedure D was therefore completed and could be implemented on Mine B.

## 4.4 Validation of methodology

The evaluation and optimisation procedures have been discussed for Mine B. Note that because reliable data for the 3CPFS auxiliary pumps were not available, only the dewatering pumps and 77L booster pumps will be considered for comparative purposes.

### 4.4.1 Implementation results

The simulated control was programmed fully into the on-site REMS system (Process E.2). After a final control revision (Process E.1), it was tested on Mine B over a three-day period. During the first testing period, the developed REMS control was mimicked by the control room operators. The resulting power profile is shown in Figure 33 (Process E.3).

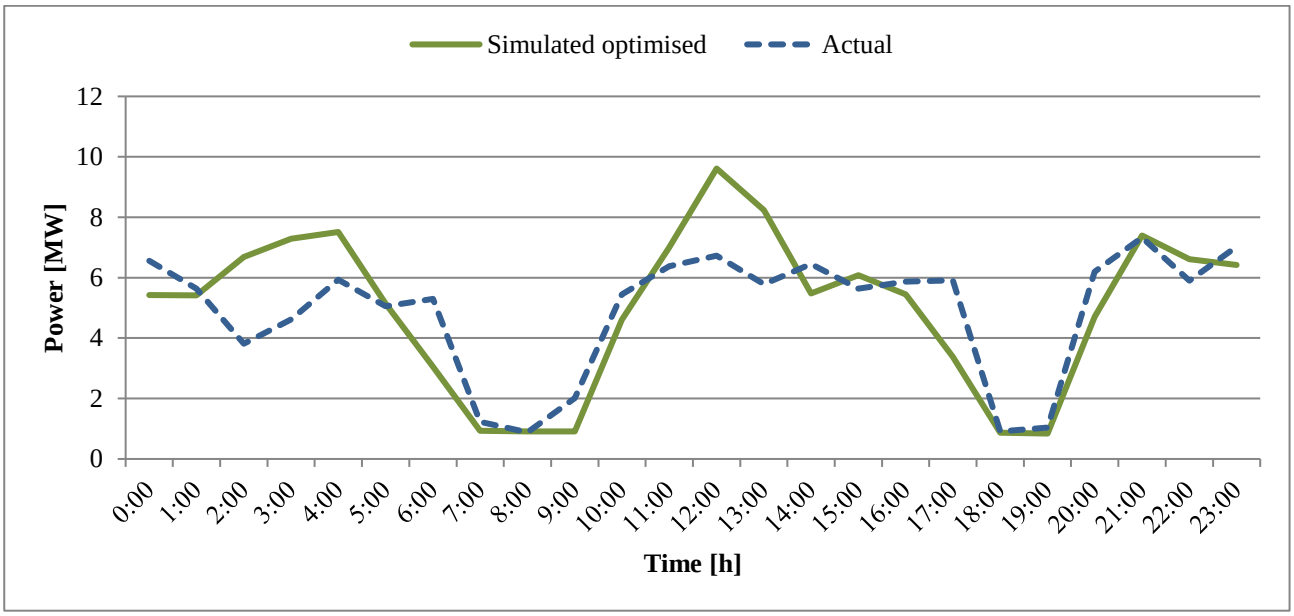



**Figure 33: Mine B – Simulated vs actual power profile**

Figure 33 shows apparent differences between the actual profile and the simulated profile. Table 43 summarises these differences.

**Table 43: Mine B – Comparison of simulated and test 1 results**

| Parameter | Simulated optimised | Implementation results | Difference |
|---|---|---|---|
| Average power [kW] | 5000 | 4900 | 100 |
| Evening peak power [kW] | 855 | 975 | 125 |
| Morning peak power [kW] | 920 | 1375 | 455 |
| Standard period power [kW] | 5910 | 6095 | 185 |
| Annual electricity cost [R] | 23.9 million | 24.1 million | 0.2 million |

During the testing period, deviations between the simulated and actual profiles were observed. The demand flow profiles and dam levels which form the main control parameters were obviously different over the two periods.

In addition to this, the control strategy was not accurately followed. Certain issues were experienced and led to situations where the control had to be reset. These issues are discussed in Chapter 4.4.2. Additionally, the booster pumps into the mining sections were not controlled according to a set control philosophy.

During the tests, a slight reduction in the power usage can be observed. Slightly poorer load management performance was also observed during the test compared to the simulation values. The result is that the annual savings achievable for the implementation period power profile differs with 8.3% from the predicted profile.

Note that since only load management was applied, energy neutral scaling was applied in all of the baselines used throughout the rest of the document. Figure 34 compares the obtained profile to the scaled baseline.

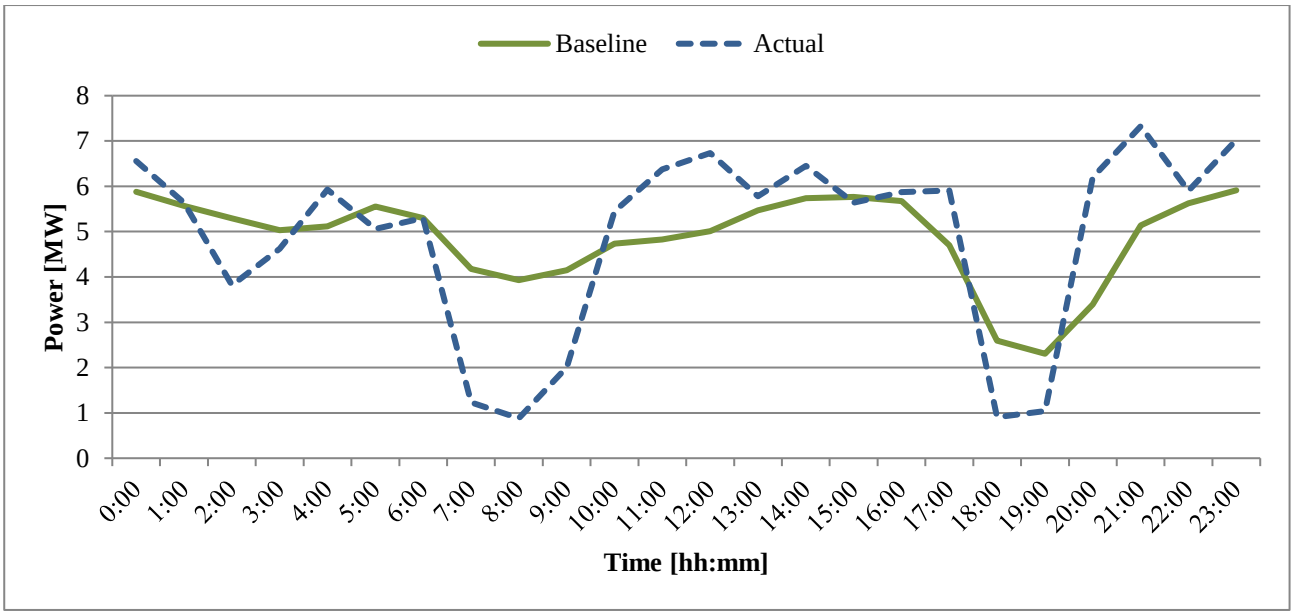



**Figure 34: Mine B – Scaled baseline compared to optimised profile**

Table 44 compares the critical parameters for the scaled baseline and manual testing period.

**Table 44: Mine B – Soft commissioning result comparison**

| Parameter | Baseline | Testing period | Difference |
|---|---|---|---|
| Average power [kW] | 4900 | | - |
| Evening peak power [kW] | 2470 | 975 | 1495 |
| Morning peak power [kW] | 4115 | 1375 | 2740 |
| Standard period power [kW] | 5100 | 6095 | 995 |
| Annual electricity cost [R] | 26.0 million | 24.1 million | 1.9 million |

It is clear that a significant improvement on the scaled baseline was observed from during the testing period. Problems were experienced during this initial testing period. However, the nature of the problems were such that the simulations did not highlight it. The REMS control had to be reset multiple times during the testing period; this led to differences between the simulation and actual profile.

### 4.4.2 Soft commissioning problems

#### Dam level fluctuations

A significant problem that was picked up during the soft commissioning phase was inaccuracies in the dam level sensors directly after the start-up of the pumps. The dam level sensor malfunctioned for approximately 30 seconds. This was due to the transient effect of the pump start and stops on the sensors. During these 30 seconds, the dam sensor reading dropped by as much as 15%. After this period, the dam level settled to a slightly reduced value (typically 3% lower than the value before the start-up).

This problem could have caused potential cycling of pumps had the system been automatically controlled during this testing phase. Since upstream control was used, the sudden dip in the real-time control parameter caused a stop command to be sent to the pump directly after the pump started up. A significant risk of pump cycling was therefore present on the mine.

This phenomenon was not clear in the logged data since the two-minute logging resolution is larger than the 30 seconds that the problem typically lasted. Additional control measures were therefore required. The real-time dam levels were switched to a control parameter known as an oscillation limited value. These tag values filtered temporary fluctuations and provided the required control stability.

Table 45 shows the oscillation limited values that were added to the control.

**Table 45: Mine B – Oscillation limited control values**

| Parameter | Value |
|---|---|
| Upper boundary | 5 |
| Lower boundary | 2 |
| Input tag | Stable dam level |
| Time interval | 1 minute |

By using oscillation limited values as control parameters, the transient effect of the pumps on the dam level sensors could be filtered out. This allowed for stable control without pump cycling.

### 3CPFS control

The 3CPFS control performed as expected. Even though the dam limits stayed within the allowed parameters during both peak periods, it reached close to the minimum levels during the morning peak period. The mine requested that alterations be made to the 3CPFS control.

The reasoning for the more conservative approach was that if a 3CPFS were to fail when the dam level is close to the minimum level, it might require sending water underground through dissipaters. This can lead to potential inefficiencies. Additional control parameters were therefore set up for the 3CPFS control when the chilled dam levels are close to the minimum level. These control parameters are shown in Appendix G.

## 4.4.3 Automation test

After the changes to the control strategy were brought on, the full automation strategy was implemented (second phase of Process E.3). The tests were planned for three days. However, after 36 hours of testing, a 3CPFS malfunctioned and became unavailable. The test was therefore stopped after this occurred.

Figure 35 shows the resulting power profile of this test.

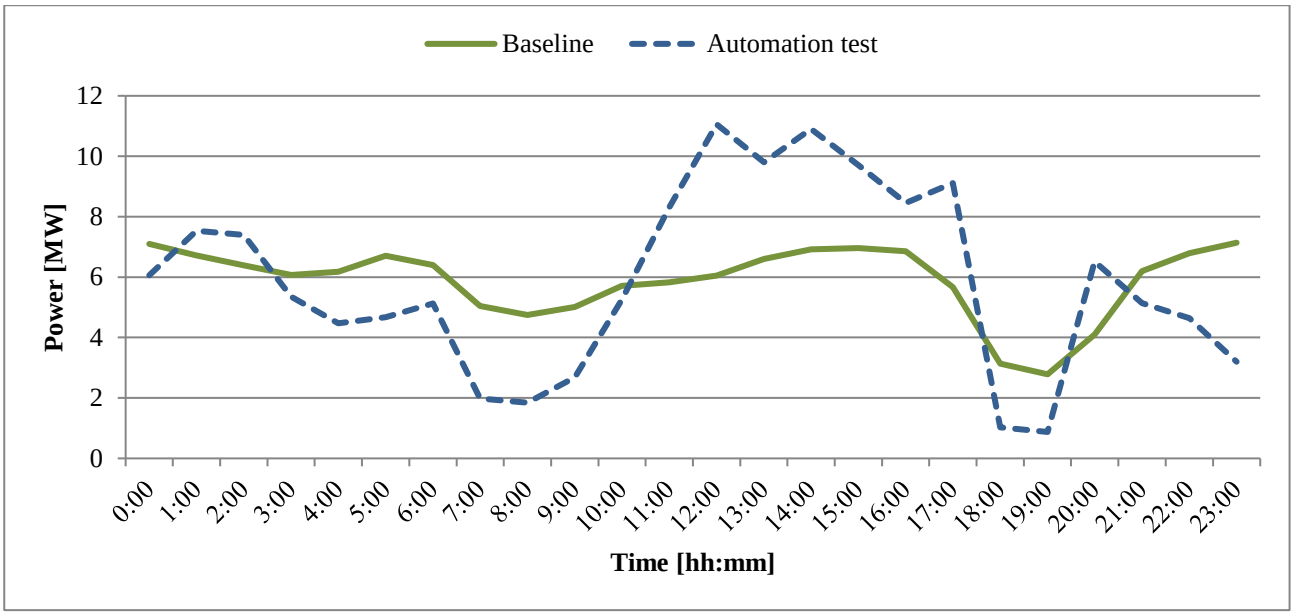



**Figure 35: Mine B – Automation testing results**

Table 46 summarises the impact of the automated control.

**Table 46: Mine B – Automation testing period result comparison**

| Parameter | Baseline | Automated testing | Difference |
|---|---|---|---|
| Average power [kW] | 5880 | | - |
| Average evening peak power [kW] | 2960 | 950 | 2010 |
| Average morning peak power [kW] | 4935 | 2170 | 2765 |
| Standard period power [kW] | 6120 | 8125 | 33 |
| Annual electricity costs [R] | 31.4 million | 29.8 million | 1.6 million |

After this testing period, the mine started experiencing more frequent breakdowns of the 3CPFSs. Additional testing did not commence within the timeframe of this study. However, the conducted test indicates that an automated control strategy is feasible for 3CPFSs that utilise variable flow strategies.

It is important to note that the results from the soft commissioning were more favourable than that of the automation testing. A more aggressive comeback load after the morning peak is observed for the automation test period than that of the soft commissioning. This comeback load is the result of the more conservative control changes requested by the mine.

In addition to this, it is important to note that the automation test period was concise. The output power profile can be expected to replicate the simulated values more closely if it is implemented for a more extended period and the system stabilises to the control.

However, the results indicate that an automated control strategy is indeed feasible. The automated control strategy still required minor adjustments to be considered for it to be optimised. Since these adjustments were not implemented or tested, it will not be presented in this study.

### 4.4.4 Improved manual control

After the testing period, the mine continued to implement manual load management strategies. A period of three months was evaluated. During this period, only thirty days with an average 3CPFS availability of above 90% occurred.

It is important to note that data loss occurred and therefore 15 days could not be considered for evaluation. The result of this is that over the 90 days analysed, 45 days had a known 3CPFS availability of less than 90%.

Figure 36 shows the average achieved profile for days with a 3CPFS availability of above 90% (Process E.4 validation).

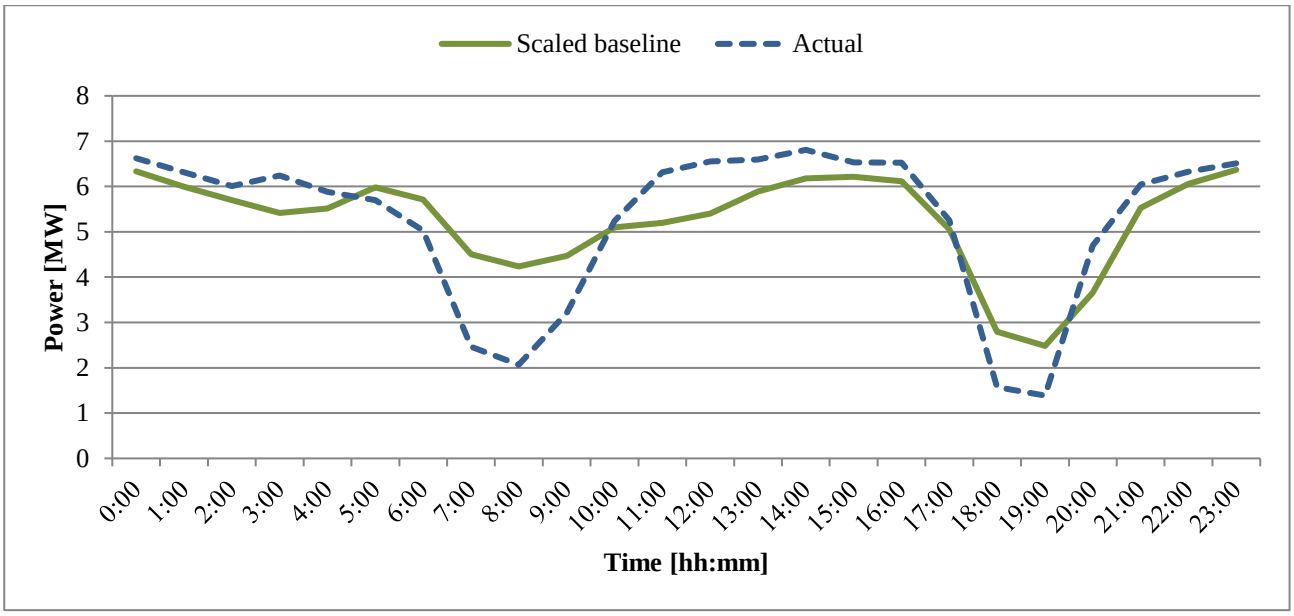



**Figure 36: Mine B – Manual performance with high 3CPFS availability**

Table 47 shows the summarised results over the 30 days with a 3CPFS availability of above 90%.

**Table 47: Mine B – Manual load management performance with high 3CPFS availability**

| Parameter | Baseline | High 3CPFS availability | Difference [%] |
|---|---|---|---|
| Average power [kW] | 5245 | | - |
| Average evening peak power [kW] | 2640 | 1480 | 1160 |
| Average morning peak power [kW] | 4400 | 2580 | 1820 |
| Standard period power [kW] | 5460 | 5960 | 500 |
| Annual electricity costs [R] | 28.0 million | 26.4 million | 1.7 million |

A definite improvement in the load management performance can be observed under the conditions compared to that of the baseline period. An annual cost decrease of R1.7 million is achievable if these conditions could be extended through the rest of the year. This validates the use of variable flow strategies on ERDs as part of load management initiatives on mine dewatering systems.

Figure 37 shows the performance curves of Mine B with manual load management strategies implemented and low 3CPFS availability.

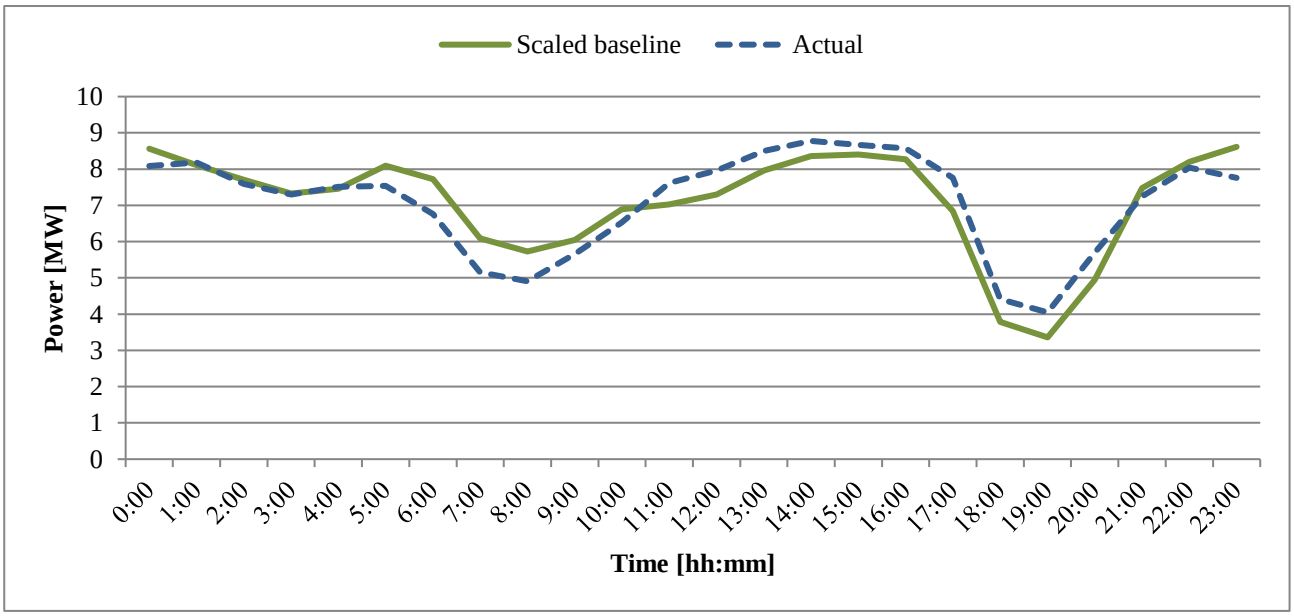



**Figure 37: Mine B – Manual load management performance with low 3CPFS availability**

Table 48 shows the load management performance of Mine B with low 3CPFS availability.

**Table 48: Mine B – Manual load management performance with low 3CPFS availability**

| Parameter | Baseline | Low 3CPFS availability | Difference [%] |
|---|---|---|---|
| Average power [kW] | 7095 | | - |
| Average evening peak power [kW] | 3570 | 4230 | 660 |
| Average morning peak power [kW] | 5955 | 5240 | 715 |
| Standard period power [kW] | 7380 | 7645 | 265 |
| Annual electricity costs [R] | 37.9 million | 37.9 million | - |

It is clear that a significantly reduced load management performance is observed during the days with a low 3CPFS availability. The evening peak power usage increased. The annual cost difference between the two profiles is negligible.

This highlights the importance of proper maintenance on the ERDs. The unavailability of the 3CPFSs leads to reduced pumping capacity. This unavailability has an integrated effect on the performance of the system. The inefficiency leads to reduced load management capacity which in turn leads to an additional increase of the total system energy cost.

### 4.4.5 Reporting and maintenance

A reporting system was set up to ensure the performance of the manual load management. The report was sent out daily and contained the performance of the previous day (Process F.2). Examples of the daily report are included in Appendix H.

Since the system was not fully automated, active maintenance was conducted on the server to ensure continual access to the relevant data.

### 4.4.6 Summary

From the practical tests, it is clear that the processes identified from literature are effective. The soft commissioning yielded favourable results. Additionally, measurement problems that were not apparent in the data analysis were identified and appropriate control adjustments were made.

Full automation tests were conducted after the soft commissioning concluded. The test showed that automatically controlling a dewatering system that makes use of multiple ERDs is feasible. However, the developed control strategy was designed for a 100% availability of the ERDs.

Sporadic unavailability of the ERDs hindered the control. Mine personnel therefore implemented manual control strategies. The results from this manual control indicated that if the ERDs could be made available indefinitely, annual savings of R1.7 million can be achieved. However, the average impact of the initiative on days with an average 3CPFS availability below 90% yielded a negligible benefit compared to the baseline period.

## 4.5 Conclusion

It was shown that the developed optimisation methodology could be applied to an actual mine dewatering system that makes use of ERDs. The mine considered for a case study has a complex dewatering system with four 3CPFSs and a closed-loop high-pressure u-tube system.

A full evaluation of the mine was presented. A representative baseline was developed to measure the performance of the project. In addition to this, an appropriate three day reference period was identified. The data over this period was analysed and processed.

All of the relevant system components were characterised and programmed into a baseline Process toolbox simulation. The optimisation methodology was applied to the simulation. The simulation results indicated that approximately 1.5 MW load could be shifted from the evening peak and 4 MW from the morning peak, with a potential annual cost reduction of R2.1 million.

In addition to the improved load management, investigations into potential WSO initiatives showed that a flow reduction of approximately 40 ℓ/s during the blasting shift is possible. Additionally, 15 ℓ/s could be redirected from the 102L hot dam to the 77L hot through the closed-loop high-pressure u-tube system.

Simulations showed that these initiatives would lead to an average reduction of 300kW. The improvement in efficiency would lead to an additional annual cost saving of R1.0 million. This saving excluded the potential benefit of reduced flow on the surface refrigeration system and is therefore extremely conservative. Due to concerns regarding the temperature increases, the identified WSO initiatives were not implemented.

The control strategy that was developed during the simulation of the system was tested. The first phase of testing consisted of a soft commissioning during which the control room operators mimicked the automated strategy. During this phase, it was shown that an annual cost reduction of approximately R1.9 million is feasible. This corresponded reasonably well to the simulated value.

During the soft commissioning, it became apparent that the dam level sensors provided false values during transient periods of the start-up of pumps. Since the problems only arose in short bursts, the control could be adjusted to accommodate the changes.

After the control was finalised, a full automaton test was conducted. However, due to mechanical issues on the 3CPFSs, the automation test could only be conducted for a 36 hour period. The mechanical issues persisted after this and the mine therefore, implemented improved manual load management after the testing period.

The available days in the manual load management period where the 3CPFS availability was above 90% was analysed. It was apparent that under these conditions, an annual cost reduction of approximately R1.7 million is possible. However, an analysis of days with low 3CPFS availability showed that the manual load management strategy yielded no additional benefit compared to the baseline period.

Regardless of this, the practical test indicates that the strategies presented in the methodology can be used to optimise a dewatering system that utilises ERDs. This therefore validates the methodology.

# Chapter 5: Conclusion and recommendations



5

In this chapter, the study is summarised. The entire process followed is concluded in this summary. The specific problems addressed in the study are also highlighted. Lastly, recommendations for future work are made.

[5] P. Duddu, "Mining technology," 11 Sep 2013. [Online]. Available: http://www.mining-technology.com/features/feature-top-ten-deepest-mines-world-south-africa/. [Accessed 19 Nov 2017].

## 5.1 Summary

The mining industry in South Africa and in particular the gold mining industry is presently experiencing economic hardships. There are plenty of contributing factors to this problem. The contributing factors include reduced productivity, increased labour costs and rising electricity prices.

South African mines have to save costs to remain competitive. It was shown that the rising electricity prices in South Africa are of particular concern since South African mines are amongst the most energy-intensive in the world.

Industrial DSM projects were identified as an effective way to reduce the energy costs of mines. On deep South African mines, the dewatering system can consume up to 17% of the total electricity on the mine. Plenty of past DSM projects have been implemented on such systems.

ERDs were identified as an attractive means of reducing the energy consumption and enhancing the performance of mine dewatering systems. However, it was shown that these devices could potentially complicate control strategies.

It was therefore clear that even though ERDs offer significant energy saving opportunities in mine dewatering systems, optimal integration of ERDs into dewatering systems is required to ensure minimum costs. It was clear from the literature that although previous studies investigated such integration strategies, a complete optimisation strategy for ERDs on mine dewatering does not exist.

Additionally, most of the studies that discussed the integration of ERDs on mine dewatering systems were found to be relatively old. These studies did not consider recent developments in the field of energy management. The recent developments include new simulation tools and an increase in the use of variable flow strategies.

A methodology to optimise mine dewatering systems that utilise ERDs was therefore proposed after a thorough literature survey was conducted. To ensure that the methodology was truly comprehensive, research was conducted into energy audit strategies, optimisation through simulation, as well as energy project implementation strategies.

The developed strategy proposed optimising for minimum annual energy costs. The literature showed that this could be achieved by controlling the demand flow and by implementing load management strategies to minimise electricity usage in peak times. The literature also showed that appropriate scheduling or the use of variable flow strategies could be used to achieve these objectives.

To verify the developed optimisation methodology, Mine A was analysed. Mine A is a deep-level gold mine that is fully automated and has successfully implemented energy efficiency and has load management initiatives in place. Additionally, Mine A makes use of ERDs in the form of two turbine-pumps. To verify the effectiveness of the strategy, a calibrated simulation of Mine A was created.

Restrictions were placed on the control that is in place on Mine A within the simulation. These restrictions were overcome within the simulation using the strategies proposed in the methodology, thereby verifying the effectiveness of the strategies. To show that the strategy applies to other ERDs, the turbine pumps were replaced with 3CPFSs and closed-loop high-pressure u-tube systems within the simulation. The same restrictions were simulated and overcome within the simulation.

The simulations indicated that the developed strategies could be used for any dewatering system that makes use of ERDs. Therefore, the developed strategy was applied to a suitable case study: Mine B. Mine B has a complex dewatering system that consists of four 3CPFSs, multiple large dewatering pumps, as well as a closed-loop high-pressure u-tube system.

A full evaluation of Mine B was conducted. All of the required information for optimisation was obtained. System reference baselines were constructed. Preliminary investigations were conducted and indicated that potential for both load management and WSO initiatives existed on Mine B.

All of the components of the dewatering system of Mine B were characterised and simulated. The optimisation methodology was applied to the baseline simulation, and the simulations indicated that an increased evening peak load shift of 1.5 MW and morning peak shift of 2.4 MW was feasible. This increased load management performance and would lead to a R2.1 million annual cost reduction.

Simulations also showed that the WSO initiatives that were identified could lead to an average power reduction of approximately 300kW. The energy cost reduction leading from this efficiency increase could lead to an additional R1 million annual cost reduction.

The WSO initiative was achievable through a combination of leak repairs and supply flow control alterations. The load management initiatives improvement was shown to be achievable through improved control of the dewatering pumps in unison with the VSDs on the 3CPFS auxiliary pumps. However, due to concerns of raising underground temperatures, these measures could not be implemented.

The developed control strategy for improved load management was tested. The first phase of testing consisted of a soft commissioning. During this phase of testing, the developed control was mimicked by the control room operators. The results showed that a 1.5 MW evening peak reduction and 2.75 MW morning peak reduction is feasible.

However, specific problems were experienced during the testing period. The control was therefore adjusted to compensate for the problems experienced. Additional control adjustments were made at the request of the mine personnel. The control strategy was tested over the second period. During this phase, the dewatering system of Mine B was automatically controlled for approximately 36 hours.

The control could not be implemented for a period longer than this given the fact that some of the 3CPFSs became unavailable during the period due to mechanical problems. The test therefore only lasted approximately 36 hours. However, the result of the test indicates that automation of such systems is feasible.

Consequently, the mine proceeded with an improved manual load management strategy. However, during the proceeding three-month period, only 30 days with a 3CPFS availability of above 90% occurred. During these 30 days, an improved evening peak load shift of 2 MW was achieved and an improved morning peak period load shift of 2.75 MW was achieved.

However, during the rest of the period, no improvement in load management performance from the baseline period was observed. Nevertheless, the manual load management strategies implemented indicated that if a 3CPFS availability of above 90% is in place on Mine B, an annual cost saving increase of approximately R1.6 million is achievable through manual load management strategies. This saving would be in addition to improved efficiency.

## 5.2 Problems addressed

The research problem stated that a complete optimisation strategy for ERDs in mine dewatering systems had to be developed. The specific outcomes of the methodology were to develop an evaluation strategy, optimisation methodology and finally effective implementation strategies.

Evaluation strategies in the form of energy audits were identified from the literature. Specific outcomes of the audits were identified and summarised processes to develop system reference baselines were also investigated and presented. It was clear from the literature that the identified processes have in the past been effectively applied to multiple other industries as well as mine dewatering systems.

As part of the evaluation phase, investigation requirements to identify WSO and load management strategies on mine dewatering systems was discussed. The critical requirements for these initiatives were investigated and summarised together with preliminary checks identified in previous studies.

Integrated dynamic simulations were identified as a useful tool to conduct more detailed investigations and optimisation studies on mine dewatering systems. Criteria for suitable simulations were identified and discussed.

After this, an optimisation strategy was presented. The strategy entails repetitively simulating the system with alterations to the control between iterations. The optimisation criteria identified from literature is minimum system costs. The process could therefore, be used to quantify the impact of the identified initiatives and develop the necessary control simultaneously.

General guidelines for the strategy and checks to be conducted between optimisation iterations were also presented. These include checks for pump cycling, dam limits and calculating and comparing the optimisation criterion, which is minimum cost. Guidelines found in the literature with regards to specific limits for optimal load management was also presented.

Additionally, the use of proper scheduling of ERDs was investigated and presented through the use of Mine A as a case study. The use of variable flow strategies were also investigated and presented as an alternative if scheduling is not possible. The considered case studies validated the use of both of these strategies as a means to improve energy management on mine dewatering systems that make use of ERDs.

Lastly, it was shown that it is feasible to automate a complex dewatering system that makes use of ERDs optimally if all of the ERDs remain available. However, it is important to note that this was rarely the case in the considered case study. It can be concluded that the stated research problem was addressed and all of the listed objectives were met.

## 5.3 Recommendations

The study assumed that all of the equipment on the relevant dewatering systems are available. However, it became evident during the testing of the developed methods that ERDs and in particular 3CPFSs can be subject to sporadic unavailability.

Due to the complexity of the system and the limited time and resources that were available, the development of methodologies that aim to overcome this problem was not considered. It could be argued that the methodology could be used to develop control strategies for each possible configuration. However, this would be time and resource intensive. It would also be difficult to test such strategies given the unpredictable nature of the equipment failures. However, if a successful strategy is developed, the results could be worthwhile.

In addition to this, the WSO initiatives identified were not implementable due to concerns that the reduced cooling would adversely affect underground temperatures. It should therefore, be considered to develop detailed simulations of the underground ventilation networks of Mine B to further investigate the possibility since clear potential for efficiency improvements exists on the system.

However, no simulation software could be found that can dynamically integrate the water and ventilation system of mine cooling systems. Therefore, the development of such tools is recommended to aid in future studies.

The focus of this study was on the underground dewatering system. However, ERDs have an integrated effect on the entire cooling system of mines. The recently developed simulation strategies can easily be used to quantify such effects. The integrated effect of the temperature improvements that ERDs such as turbines offer can therefore, be combined with the energy and load management potential increase investigated in this study.

Lastly, the effects of the reduced flow through the 3CPFSs on the cold water sent down the shaft should be investigated. By reducing the flow, the contact time of the hot and cold water will increase which could have potential negative effects.

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# **Appendix A:** Eskom TOU tariff cost calculations

This study aimed to optimise dewatering systems to achieve minimum electricity cost. A thorough understanding of the tariffs used in the applicable mines is therefore required. Table 49 summarises the tariff structure applicable on the considered mines.

**Table 49: Eskom TOU tariff structure [10]**

| Period | Summer weekday [c/kWh] | Summer Saturday [c/kWh] | Winter weekday [c/kWh] | Winter Saturday [c/kWh] |
|---|---|---|---|---|
| 00:00 | 40.09 | 40.09 | 46.24 | 46.24 |
| 01:00 | 40.09 | 40.09 | 46.24 | 46.24 |
| 02:00 | 40.09 | 40.09 | 46.24 | 46.24 |
| 03:00 | 40.09 | 40.09 | 46.24 | 46.24 |
| 04:00 | 40.09 | 40.09 | 46.24 | 46.24 |
| 05:00 | 40.09 | 40.09 | 46.24 | 46.24 |
| 06:00 | 62.89 | 40.09 | 278.33 | 46.24 |
| 07:00 | 91.14 | 62.89 | 278.33 | 84.68 |
| 08:00 | 91.14 | 62.89 | 278.33 | 84.68 |
| 09:00 | 91.14 | 62.89 | 84.68 | 84.68 |
| 10:00 | 62.89 | 40.09 | 84.68 | 46.24 |
| 11:00 | 62.89 | 40.09 | 84.68 | 46.24 |
| 12:00 | 62.89 | 40.09 | 84.68 | 46.24 |
| 13:00 | 62.89 | 40.09 | 84.68 | 46.24 |
| 14:00 | 62.89 | 40.09 | 84.68 | 46.24 |
| 15:00 | 62.89 | 40.09 | 84.68 | 46.24 |
| 16:00 | 62.89 | 40.09 | 84.68 | 46.24 |
| 17:00 | 62.89 | 40.09 | 278.33 | 46.24 |
| 18:00 | 91.14 | 62.89 | 278.33 | 84.68 |
| 19:00 | 91.14 | 62.89 | 84.68 | 84.68 |
| 20:00 | 62.89 | 40.09 | 84.68 | 46.24 |
| 21:00 | 62.89 | 40.09 | 84.68 | 46.24 |
| 22:00 | 40.09 | 40.09 | 46.24 | 46.24 |
| 23:00 | 40.09 | 40.09 | 46.24 | 46.24 |

Note that Sundays are charged at a fixed rate. This rate is the low demand rate indicated in the green sections in Table 49. Public holidays in turn is billed at the same rates as Saturdays.

All of the simulations outputs and measured data used in this study were compiled into hourly profiles. The annual electricity cost was calculated through multiplying the power output of each hourly interval as summarised in Table 49. Since a kilowatt power unit was used for all of the calculations, the power usage over the hour and energy unit in kWh was equal in all instances and could be used interchangeably.

The winter and summer tariffs were calculated separately. Additionally, the calculations considered public holidays and weekend rates as well. All of the applicable periods in a given year was therefore considered when calculating the annual electricity cost.

# **Appendix B:** Fundamental equations

**Statistical analysis tools**

The following equations can be used to calculate the required parameters [71].

$$R^2 = 1 - \frac{SSResid}{SSTo} \quad \text{Equation 9}$$

Where:

$SSResid$ = The residual sum of squares

$SSTo$ = Total sum of squares

These two variables, in turn, can be calculated by the following equations [71]:

$$SSTo = \sum_{i=1}^{n} (y_i - \dot{y}_i)^2 \quad \text{Equation 10}$$

$$SSResid = \sum_{i=1}^{n} (y_i - \bar{y})^2 \quad \text{Equation 11}$$

Where:

$y_i$ = The i$^{th}$ value to be predicted

$\dot{y}_i$ = The predicted value of $y_i$

$\bar{y}$ = The mean value

$n$ = The number of values

The mean value of any parameter over a period can be calculated by [71]:

$$\bar{y} = \frac{y_1 + y_2 + y_3 + \cdots + y_n}{n} \qquad \text{Equation 12}$$

The RMSE value can be calculated using the following equation [71]:

$$RMSE = \sqrt{\frac{\sum_{i=1}^{n}(y_i - \dot{y}_i)^2}{n}} \qquad \text{Equation 13}$$

The standard deviation of a sample indicates the typical variance of the system and can be calculated using [71].

$$s = \sqrt{\frac{SSResid}{n-1}}$$

# **Appendix C:** Simulation model details

## Software overview

The simulation software used for this study is Process Toolbox. The software is not commercially available and the basic equations and logic used in the simulations will therefore be provided. Process toolbox is the intellectual property of Enermanage (Pty) Ltd.

Note that all of the information provided in this appendix is the sole property of Enermanage (Pty) Ltd and should not be used without the consent of the owner. A representative can be contacted using the details below:

**Tell:** +27 (0)12 809 0412

**Postal:** Postnet Suite 420, Private Bag X37, Lynnwood Ridge, 0040

Process toolbox consists of a list of generic components that can be used to construct integrated simulations of thermal-hydraulic systems. The simulation is therefore component based and can be constructed through the use of a graphical user interface. Figure 38 shows a screenshot of a basic pumps station.

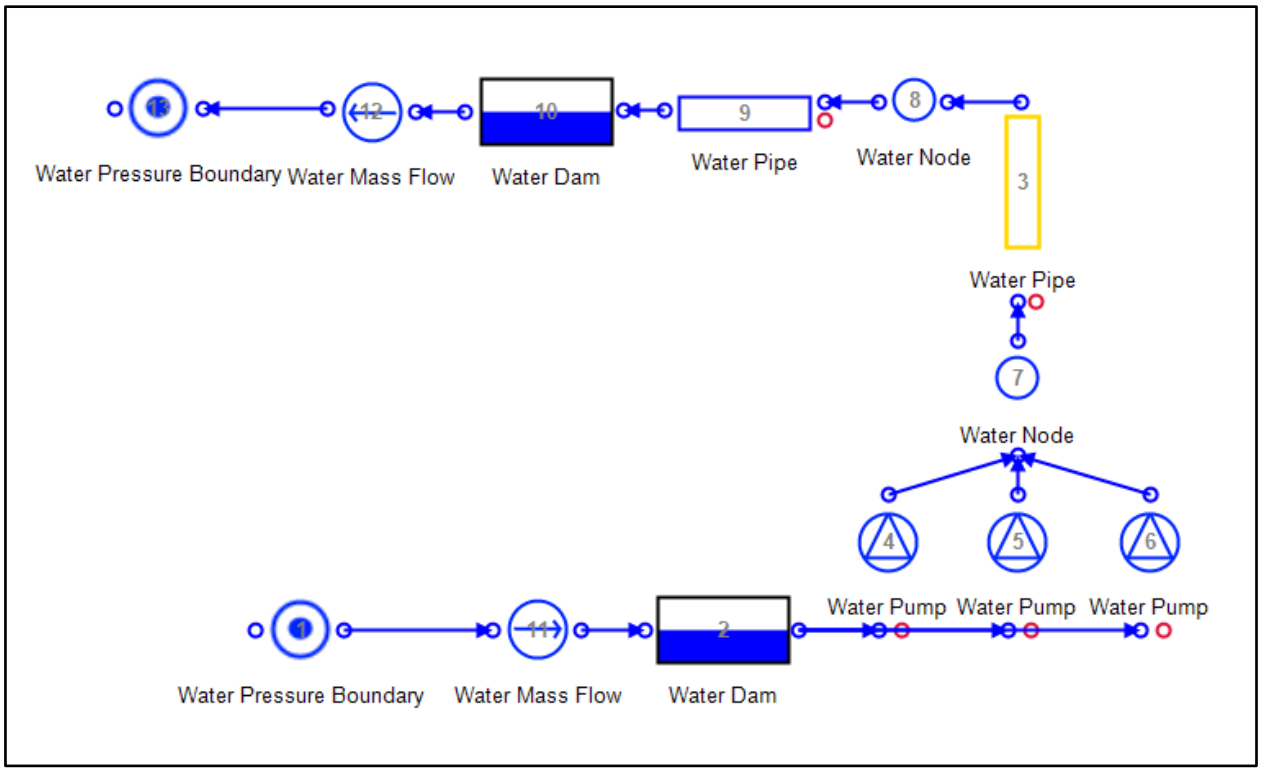



**Figure 38 – Process toolbox basic pump station**

In addition to the graphical interface, a set of standard project properties can be set. These include the number of time steps to be simulated, the size of the time steps simulated and the amount of sub time sets within these periods.

Figure 39 shows a screenshot of the project property menu.

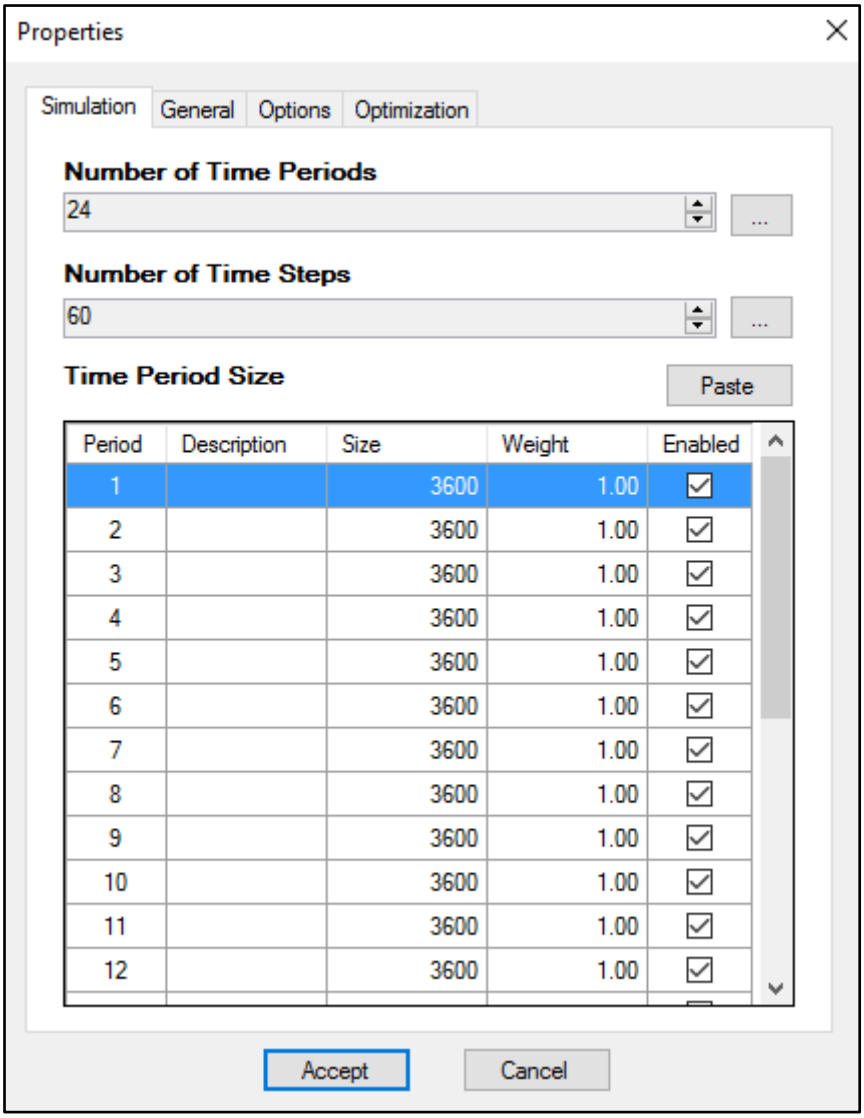



**Figure 39 – Process toolbox project properties screenshot**

The software was developed to be easily integrated with Microsoft Excel ©. Therefore, all of the inputs was processed in excel and copied to the software. The calibration process also made use of Microsoft Excel ©. Each of the time periods within the simulation can therefore be compared to actual data.

Process toolbox makes use of four main types of components these are:

- system boundaries;
- component connectors (pipes and nodes);
- general components (heat exchangers, pumps, ERDs and other relevant components); and
- controllers.

All of the components used in the simulation will be explained. The simulation icon, basic mathematical models and assumptions will be listed. All of the models used in process toolbox are explicit. This means that initial values for all of the components has to be set. The required inputs and initial guess value parameter will therefore also be provided.

## Process toolbox models

### Water pressure boundaries

The first component used in the simulations is a water pressure boundary. The pressure boundary has a fixed pressure and temperature for each time period. It is used as a system boundary and can deliver or absorb an infinite amount of mass or energy.

**Table 50: Water pressure boundary component details**

| Component icon | Inputs | Interface inputs | Outputs |
|---|---|---|---|
|  | • None | • Water pressure<br>• Water temperature<br>• Elevation | • Water enthalpy<br>• Water density<br>• Water pressure<br>• Water temperature |

The only assumption made for the water pressure boundary is that all of the conditions remain constant for each time period.

The relevant equations used for the water pressure boundary is shown below.

$$h_{wo} = (-0.0000023832 + 0.0009843694T_w)P_w + 4.18537\,Tw + 0.14399 \quad \text{Equation 15}$$

$$T_{wo} = T_w \quad \text{Equation 16}$$

$$P_{wo} = P_w \quad \text{Equation 17}$$

$$\rho_w = 1000 \quad \text{Equation 18}$$

Where:

$P_w$ = Water pressure [kPa]

$T_w$ = Water temperature [°C]

$h_{wo}$ = Water outlet enthalpy [kJ/kg]

$P_{wo}$ = Water outlet pressure [kPa]

$T_{wo}$ = Water outlet temperature [°C]

$\rho_w$ = Water density [kg/m$^3$]

### Water node

Water nodes serves as thermal links between the components. Each node has a set volume. The properties of the water, which serves as the fluid is calculated within each node.

**Table 51: Water node component detail**

| Component icon | Inputs | Interface inputs | Outputs |
|---|---|---|---|
|  Water Node | • Inlet link enthalpy<br>• Inlet link mass flow<br>• Outlet link mass flow<br>• Water pressure | • Water volume<br>• Elevation<br>• Initial pressure<br>• Initial temperature | • Water enthalpy<br>• Water density<br>• Water temperature |

The assumptions made for the water node is:

- 100% mixing and no stratification of the water within the node occurs; and
- no heat transfer with the environment.

Process toolbox uses the following fundamental equations in this component:

$$Q_s = Q_{scomp} \quad \text{Equation 19}$$

$$h_{w0} = \frac{\sum_{j=1}^{n}(m_{wi}^{j} h_{wi}^{j} dt) + M_w^0 h_{w0} + Q_s}{M_w + \sum_{j=1}^{n}(m_{w0}^{j} dt)} \quad \text{Equation 20}$$

$$\rho_w = 1000 \quad \text{Equation 21}$$

$$T_w = \frac{h_{w0} - 0.14399 - 0.0009843694 \times P_{w0}}{4.18537 - 0.0000023832 \times P_{w0}} \quad \text{Equation 22}$$

$$M_w = \rho_w V_w \quad \text{Equation 23}$$

Where:

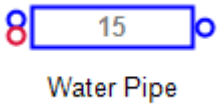
| | | |
|---|---|---|
| $Q_s$ | = | Sensible heat in the node [kW] |
| $Q_{scomp}$ | = | Heat generated by upstream components [kW] |
| $h_{wo}$ | = | Water outlet enthalpy [kJ/kg] |
| $m_{wi}^{j}$ | = | Inlet water mass flow from the link at the specific time interval [kg/s] |
| $h_{wi}^{j}$ | = | Inlet water enthalpy flow from the link at the specific time interval [kJ/kg] |
| $M_w$ | = | Node water mass [kg] |
| $M_w^0$ | = | Node water mass in the previous time step [kg] |
| $m_{w0}^{j}$ | = | Water mass flow out of the node [kg/s] |
| $h_{w0}$ | = | Enthalpy out of the node [kJ/kg] |
| $P_{w0}$ | = | Water pressure [kPa] |
| $T_{w0}$ | = | Water outlet temperature [°C] |
| $\rho_w$ | = | Water density [kg/m$^3$] |
| $V_w$ | = | Node volume [m$^3$] |

## Water pipe

The water pipe component is the second connector used in the simulation. The mechanical losses of the system is determined and quantified in the pipe component. The pipe component also has a valve fraction parameter and can therefore be used for flow or pressure control within the simulation. The pipe component has no volume.

The assumption made for this component is that the fluid properties stays constant throughout the length of the pipe.

**Table 52: Water pipe component detail**

| Component icon | Inputs | Interface inputs | Outputs |
|---|---|---|---|
|  | • Inlet node pressure<br>• Outlet node pressure<br>• Inlet node temperature<br>• Outlet node temperature | • Initial inlet pressure<br>• Initial outlet pressure<br>• Initial mass flow<br>• Valve fraction | • Mass flow |

Process toolbox uses the following fundamental equations in this component:

$$\rho_w = 1000 \qquad \text{Equation 24}$$

$$k_w = \frac{m^2}{\rho_w (P_{inlet} - P_{outlet}) k_v} \qquad \text{Equation 25}$$

$$m_w = (k_w \rho_w (P_{up} - P_{down}))^{0.5} \qquad \text{Equation 26}$$

Where:

$\rho_w$ = Water density [kg/m$^3$]

$k_w$ = Water flow admittance [kg$^2$/(s·m$^3$·kPa)]

$m$ = Water mass flow rate (measured) [kg/s]

$P_{inlet}$ = Inlet pressure [kPa]

$P_{outlet}$ = Outlet pressure [kPa]

$k_v$ = Valve fraction [-]

$m_w$ = Water mass flow rate (calculated) [kg/s]

$P_{up}$ = Upstream node pressure [kPa]

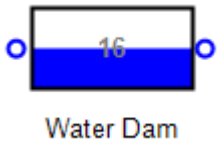
$P_{down}$ = Downstream node pressure [kPa]

### Water dam

The water dam also serves as a thermal link. However, the inlet and outlet flow is not assumed to be equal and the absolute volume of the water within the dam therefore varies. The following assumptions are made for the water dam component:

- 100% mixing of the water occurs;
- a constant density of 1000 kg/m$^3$ throughout the dam; and
- The cross sectional area of the dam remains constant.

**Table 53: Water dam component detail**

| Component icon | Inputs | Interface inputs | Outputs |
|---|---|---|---|
|  | • Inlet node pressure<br>• Outlet node pressure<br>• Inlet node temperature<br>• Outlet node temperature | • Water pressure<br>• Water temperature<br>• Volume<br>• Top elevation<br>• Bottom elevation<br>• Level | • Mass flow |

Process toolbox uses the following fundamental equations in this component:

$$Q_s = Q_{scomp} \qquad \text{Equation 27}$$

$$h_{w0} = \frac{\sum_{j=1}^{n}(m_{wi}^{j}\, h_{wi}^{j} dt) + M_w^0 h_{w0} + Q_s}{M_w + \sum_{j=1}^{n}(m_{w0}^{j}\, dt)} \qquad \text{Equation 28}$$

$$T_w = \frac{h_{w0} - 0.14399 - 0.0009843694 \times P_{w0}}{4.18537 - 0.0000023832 \times P_{w0}} \qquad \text{Equation 29}$$

$$\rho_w = 1000 \qquad \text{Equation 30}$$

$$M_w = M_w^o + \sum_{j=1}^{n}(m_{wi}^{j}\, dt) - \sum_{j=1}^{n}(m_{wo}^{j}\, dt) \qquad \text{Equation 31}$$

$$L = \frac{M_w}{V_w \rho_w \times 100} \qquad \text{Equation 32}$$

Where:

$Q_s$ = Sensible heat in the node [kW]

$Q_{scomp}$ = Heat generated by upstream components [kW]

$h_{wo}$ = Water outlet enthalpy [kJ/kg]

$m_{wi}^{j}$ = Inlet water mass flow from the link at the specific time interval [kg/s]

$h_{wi}^{j}$ = Inlet water enthalpy flow from the link at the specific time interval [kJ/kg]

$M_w$ = Node water mass [kg]

$M_w^0$ = Node water mass in the previous time step [kg]

$m_{w0}^{j}$ = Water mass flow out of the node [kg/s]

$h_{w0}$ = Enthalpy out of the node [kJ/kg]

$P_{w0}$ = Water pressure [kPa]

$T_{w0}$ = Water outlet temperature [°C]

$\rho_w$ = Water density [kg/m$^3$]

$V_w$ = Node volume [m$^3$]

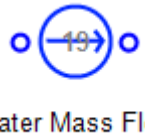
L = Dam level [%]

### Water mass flow

The mass flow component forces a constant system flow regardless of the system properties. This component does not perform any work and therefore uses no energy. The following assumptions are applicable:

- No work (which implies constant enthalpy); and
- component cannot be simulated in a series configuration.

**Table 54: Water mass flow component detail**

| Component icon | Inputs | Interface inputs | Outputs |
|---|---|---|---|
|   | • None | • Mass flow | • Mass flow |

### Water pump

The pump component uses power and generates a pressure differential within the stream. Additionally, heat is generated due to inefficiencies. The following equations are applicable:

- A constant efficiency is assumed regardless of the load; and
- there is no heat transfer with the environment.

**Table 55: Water pump component detail**

| Component icon | Inputs | Interface inputs | Outputs |
|---|---|---|---|
|   | • Mass flow | • Pressure curve parameters<br>• Efficiency curve parameters<br>• Speed fraction<br>• Water density<br>• Motor efficiency | • Pump motor power |

Process toolbox uses the following fundamental equations in this component:

$$dP = (a_2 {m_w}^2 + a_1 m_w + a_0)(N^2) \quad \text{Equation 33}$$

$$E_p = b_2 \left(\frac{m_w}{N}\right)^2 + b_1 \left(\frac{m_w}{N}\right) + b_0 \quad \text{Equation 34}$$

$$Pwr = \frac{m_w dP}{\rho_w E_p E_m} \quad \text{Equation 35}$$

Where:

$dP$ = Pressure differential [kPa]

$a$ = Pressure curve coefficients [-]

$m_w$ = Water mass flow rate [kg/s]

$N$ = Pump speed fraction [-]

$E_p$ = Pump efficiency [-]

$E_m$ = Pump motor efficiency [-]

$b$ = Efficiency curve coefficients [-]

$\rho_w$ = Water density [kg/m$^3$]

$Pwr$ = Pump power [kW]

### Water turbine

The water turbine is a simplified model of turbines. It converts pressure energy into electrical energy The following equations are applicable:

- A constant efficiency is assumed regardless of the load and flow; and
- there is no heat transfer with the environment.

**Table 56: Water turbine component detail**

| Component icon | Inputs | Interface inputs | Outputs |
|---|---|---|---|
|  | • Mass flow | • Pressure curve parameters<br>• Efficiency curve parameters<br>• Speed fraction<br>• Water density<br>• Motor efficiency | • Pump motor power |

Process toolbox uses the following fundamental equations in this component:

$$K_t = \frac{{m_{wr}}^2}{\rho_w dP_r} \quad \text{Equation 36}$$

$$dP = \frac{{m_w}^2}{\rho_w K_t} \quad \text{Equation 37}$$

$$Pwr_f = \frac{m_w}{1000 dP} \quad \text{Equation 38}$$

$$Pwr = Pwr_f E_t E_g \quad \text{Equation 39}$$

Where:

$K_t$ = Turbine admittance [kg$^2$/(s·m$^3$·kPa)]

$m_{wr}$ = Water mass flow rating [kg/s]

$\rho_w$ = Water density [kg/m$^3$]

$dP_r$ = Pressure difference rating [kPa]

$dP$ = Pressure difference [kPa]

$m_w$ = Water mass flow [kg/s]

$Pwr_f$ = Fluid power in the turbine [kW]

$Pwr$ = Turbine generation power [kW]

$E_t$ = Turbine efficiency [-]

$E_g$ = Generator efficiency [-]

### Water 3CPFS

The 3CPFS component exchanges the pressure of one stream of water to another. The following equations are applicable:

- Inlet flow = outlet flow; and
- there is no heat transfer with the environment.

**Table 57: Water 3CPFS component detail**

| Component icon | Inputs | Interface inputs | Outputs |
|---|---|---|---|
|  | • Upstream pressure<br>• Downstream pressure | • Water inlet pressure<br>• Water outlet pressure<br>• Water mass flow | • Mass flow |

Process toolbox uses the following fundamental equations in this component:

$$\rho_w = 1000 \qquad \text{Equation 40}$$

$$k_w = \frac{m^2}{\rho_w (P_{inlet} - P_{outlet}) k_v} \qquad \text{Equation 41}$$

$$m_w = \left(k_w \rho_w \left(P_{up} - P_{down}\right)\right)^{0.5} \qquad \text{Equation 42}$$

Where:

$\rho_w$ = Water density [kg/m$^3$]

$k_w$ = Flow admittance [kg$^2$/(s·m$^3$·kPa)]

$P_{inlet}$ = Measured inlet pressure [kPa]

$P_{outlet}$ = Measured outlet pressure [kPa]

$k_v$ = Valve fraction [-]

$P_{up}$ = Upstream node pressure [kPa]

$P_{down}$ = Downstream node pressure [kPa]

### Step controller

The step controller allows for stop and start control logic to be programmed into the simulation. No assumptions are made for this controller. Note that only the controller functionalities used in this study is explained in this text.

**Table 58: Step controller component detail**

| Component icon | Inputs | Interface inputs | Outputs |
|---|---|---|---|
|  Step Control | • Control inputs | • Number of steps<br>• Slope<br>• Start limit<br>• Stop limit | • Control output |

Process toolbox uses the following fundamental equations in this component if upstream control is used:

| | | |
|---|---|---|
| | $If\ C_i > L_{start}\ then\ C_o = 1$ | Equation 43 |
| | $If\ C_i < L_{stop}\ then\ C_o = 0$ | Equation 44 |

Where:

$C_i$ = The control input

$C_o$ = The control output

$L_{start}$ = The level start condition

$L_{stop}$ = The level stop condition

## PI controller

The PI controller makes use of proportional and integral control. The output signal is a fraction between zero and one. Two input signals can be provided to the controller. The second input is subtracted from the first and a value of zero is assumed if none is provided.

**Table 59: PI controller component detail**

| Component icon | Inputs | Interface inputs | Outputs |
|---|---|---|---|
|  | • Control inputs | • Control limit 1<br>• Control limit 2<br>• Integral gain<br>• Minimum control output<br>• Set point | • Control output |

Process toolbox uses the following fundamental equations in this component if upstream control is used:

$$K_p = Abs\left(\frac{1}{CL_2 - CL_1}\right) \quad \text{Equation 45}$$

$$If\ CL_2 < CL_1\ then\ E_p = ST - (C_{i1} - C_{i2}) \quad \text{Equation 46}$$

$$E_i = E_i + E_p dt \quad \text{Equation 47}$$

$$C_o = K_p E_p + K_i E_i \quad \text{Equation 48}$$

$$If\ C_o > 1\ Then\ C_o = 1 \quad \text{Equation 49}$$

$$If\ C_o < C_{om}\ Then\ C_o = C_{om} \quad \text{Equation 50}$$

Where:

| | | |
|---|---|---|
| $K_p$ | = | Proportional gain |
| $CL_2$ | = | Control limit 2 |
| $CL_1$ | = | Control limit 1 |
| $C_{i1}$ | = | Control input 1 |
| $C_{i2}$ | = | Control input 2 |
| $E_i$ | = | Integral error |
| $E_p$ | = | Proportional error |
| $K_i$ | = | Integral gain |
| $C_o$ | = | Control output |
| $C_{om}$ | = | Minimum output |

# **Appendix D:** Mine A simulation specifics and additional constraints

**Basic reasoning**

In chapter 3.3 the methodology was applied to an already optimised system. The system was evaluated and simulated following the processes developed in the method. It was shown that the optimisation strategy could be successfully applied to both turbine pumps and 3CPFSs.

However, it should be noted that Mine A is a marginal mine. Therefore, the system does not run at its full capacity. Therefore, additional constraints were added to the system to verify the effectiveness of the proposed strategies if the implementation conditions are not ideal. The first constraint tested was reduced dam capacities. This typically occurs in older mines that have dirty dams filled with mud.

The second constraint was increased demand flow. This will typically happen in newer mines or mines that are actively expanding. The effectiveness of the strategies on systems with increased flows was therefore also tested.

Note that all of the steps of the methodology will not be repeated for this phase. Only the results will be summarised. It is also important to note that only the component inputs as summarised in Table 60 and Table 61 was adjusted in the simulations. The system control, therefore, remained unchanged.

In addition to the additional simulations, a detailed simulation model is provided. Figure 40 shows the simulation layout.

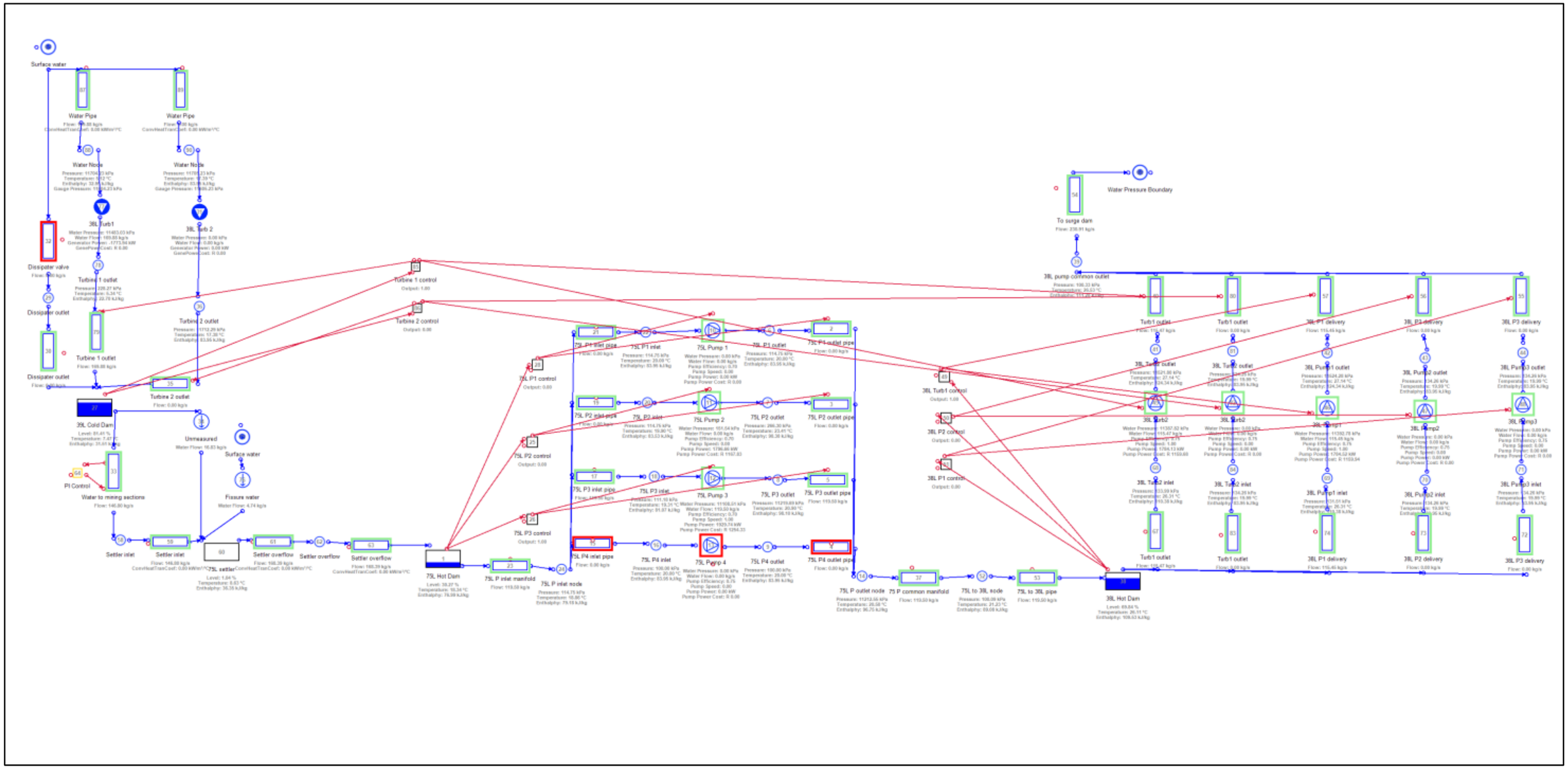



**Figure 40: Mine A – simulation model layout**

## Dam capacity reduction

The capacities of all of the dewatering dams on Mine A were reduced in the simulation as indicated in Table 60, for each simulation.

**Table 60: Mine A – Simulated dam capacity reduction**

| Percentage reduction [%] | 75L hot dam volume [Mℓ] | 38L hot dam volume [Mℓ] |
|---|---|---|
| 0 | 5412 | 4412 |
| 10 | 4870 | 3970 |
| 20 | 4329 | 3529 |
| 30 | 3788 | 3088 |
| 40 | 3247 | 2647 |
| 50 | 2706 | 2206 |
| 60 | 2164 | 1764 |
| 70 | 1623 | 1323 |
| 80 | 1082 | 882 |

Figure 41 shows the effect of reducing the dam capacity on the overall system power usage, peak power usage and annual system energy costs for a system with no ERDs.

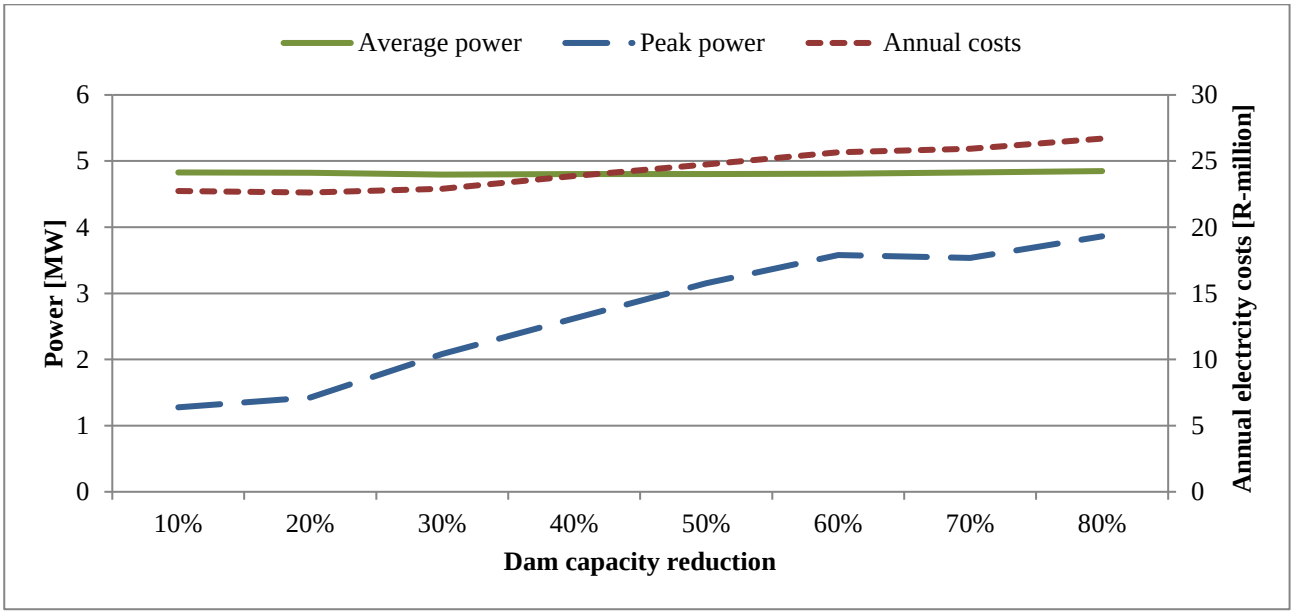



**Figure 41: Mine A – No ERDs - power and cost comparison**

Since it is a load management initiative that was tested, the power usage of the system remained constant. As expected, the load management potential of the system decreased. The load management effectiveness decrease led to a steady increase in the systems electricity costs as the dam capacity reduced.

Figure 42 shows the effect of reducing the dam capacity on the overall system power usage, peak power usage and annual system energy costs for a system with a closed loop high-pressure u-tube system installed.

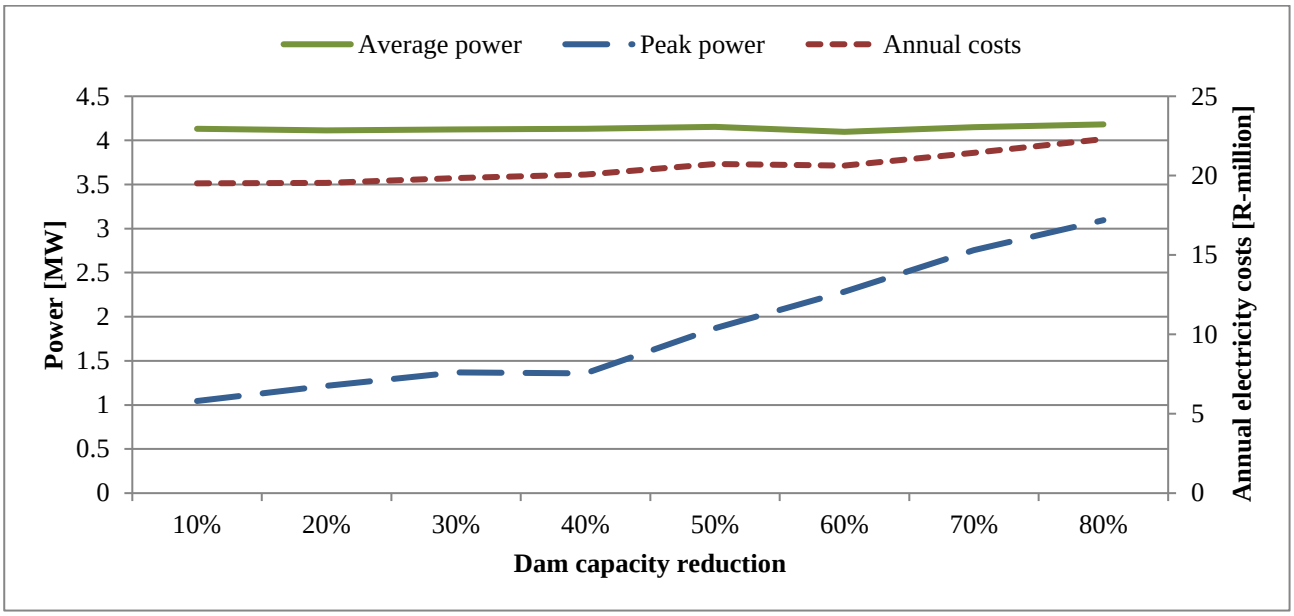



**Figure 42: Mine A – Closed-loop high-pressure u-tube power and cost comparison**

A similar effect on the load management potential of the system is noted on the closed loop high-pressure u-tube system. The load management potential decreased as the dam capacity decreased. Interestingly, the effect was small until the capacity reduction increased beyond 40%. At this point, the load management performance deteriorated at a faster rate.

The likely reason for this is that the reduction in dam capacity did not affect the 75L hot dam as severely for this scenario. The reason for this is that a portion of the demand flow goes through the u-tube directly to the 38L hot dam when a high-pressure u-tube is in place. The load management capacity of the 75L pumps is therefore only significantly affected after the dam capacity reduction reaches 40%.

Figure 43 shows the effect of reducing the dam capacity on the overall system power usage, peak power usage and annual system energy costs for a system with a turbine-pump system installed.

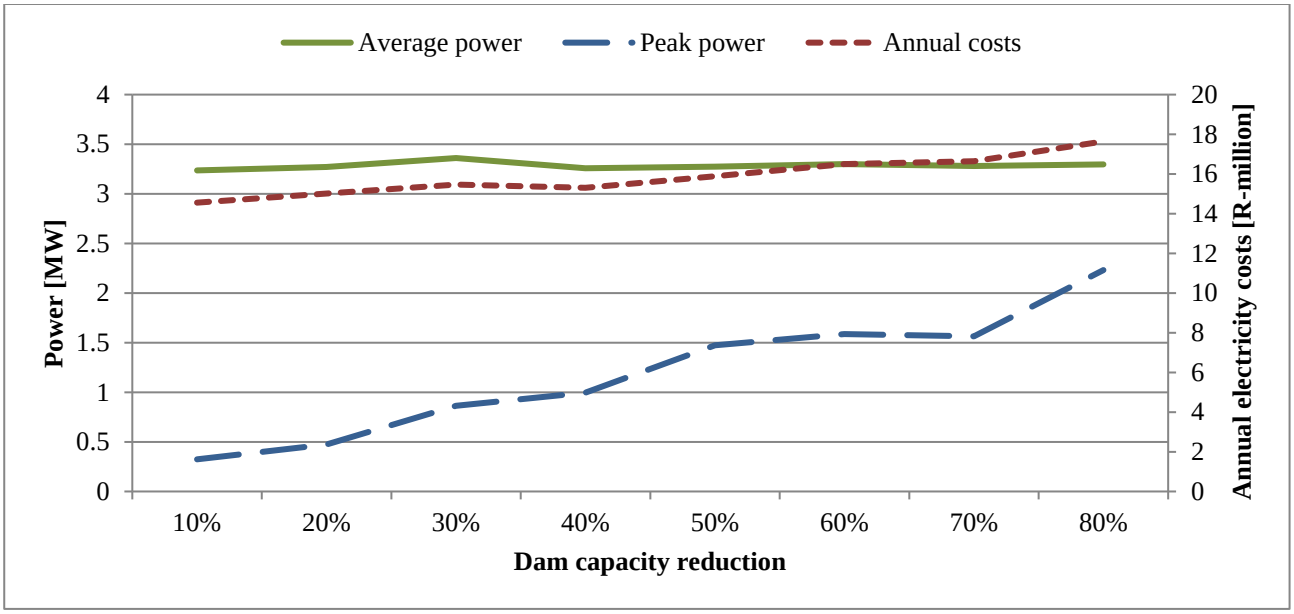



**Figure 43: Mine A – Turbine pump power and cost comparison**

A steady decrease in the load management effectiveness of the system is also evident if turbine pumps are in place. As in the other cases, the average power usage remains constant, and the systems electricity costs increases as the dam capacity reduce.

Figure 44 shows the effect of reducing the dam capacity on the overall system power usage, peak power usage and annual system energy costs for a system with a 3CPFS installed.

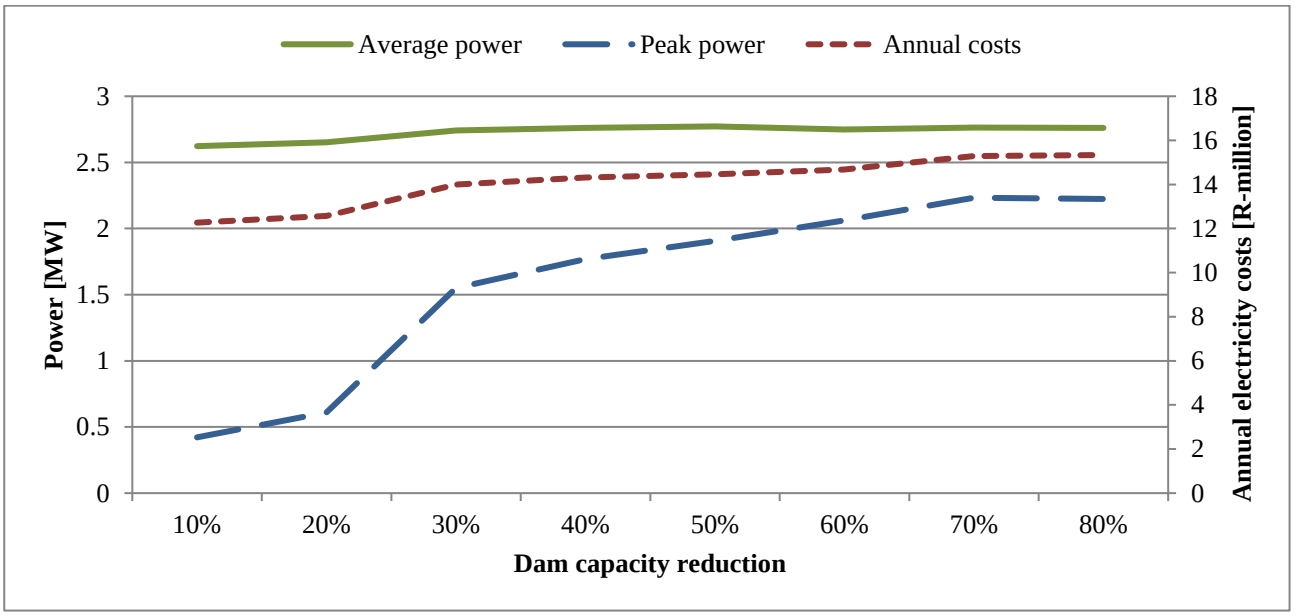



**Figure 44: Mine A – 3CPFS dewatering power and cost comparison**

In this scenario, the load management performance decreases drastically as the dam capacity is reduced. Co-generative ERDs dewaters the dams during peak times. Therefore, if sufficient capacity is not available, pumps on lower levels might have to be forced to run to ensure that the dams do not run empty.

Since the dewatering capacity of a 3CPFS, is higher than that of turbine pumps, the effect of the reduced dam capacity is more pronounced.

### Increased demand flows

The second system constraint that was tested is an increased demand flow. The demand flow will be increased by a constant factor for simulations of each type of ERD as well as the system with no ERDs. The effect on the system power usage, peak power usage and annual costs will be shown.

Table 61 shows the demand flows that were set as inputs for the simulation.

**Table 61: Mine A – Simulated demand flow increase**

| Percentage increase [%] | Demand flow [ℓ/s] |
|---|---|
| 0 | 125 |
| 10 | 138 |
| 20 | 150 |
| 30 | 163 |
| 40 | 175 |
| 50 | 188 |
| 60 | 201 |
| 70 | 213 |
| 80 | 226 |

Figure 45 shows the effect of increasing the flow demand on the overall system power usage, peak power usage and annual system energy costs for a system with now ERDs installed.

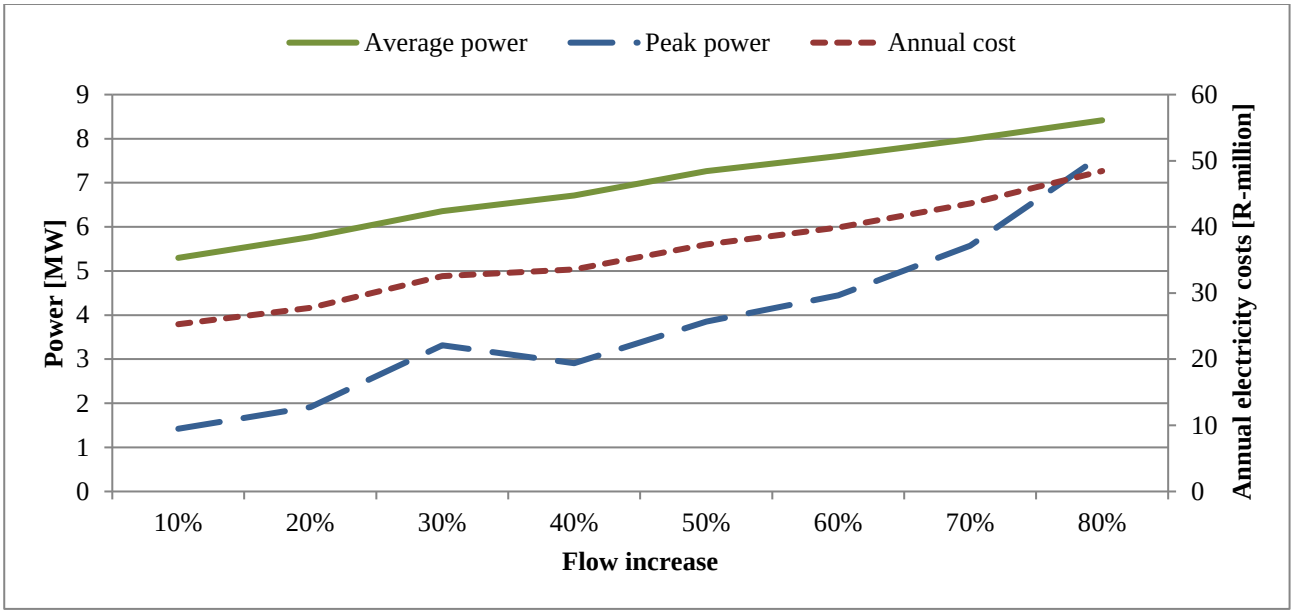



**Figure 45: Mine A – Increased flow demand, no ERDs**

On mine A, the peak power usage is shown to converge to that of the average daily power usage as the peak dam capacity reduces. This finding corresponds to research conducted by Stols [52]. This scenario is illustrative of a system with no ERDs installed.

Note that the decline in load management potential is not linear when ERDs are not present. The peak power is almost equal to the average power usage when the flow is increased by 80%, whereas with lower flows, the load management potential is still significant.

As with the reduced dam capacities, this constraint was applied to the simulated systems with ERDs present. Figure 46 shows the effect of increasing the flow demand on the overall system power usage, peak power usage and annual system energy costs for a system with a closed loop high-pressure u-tube system installed.

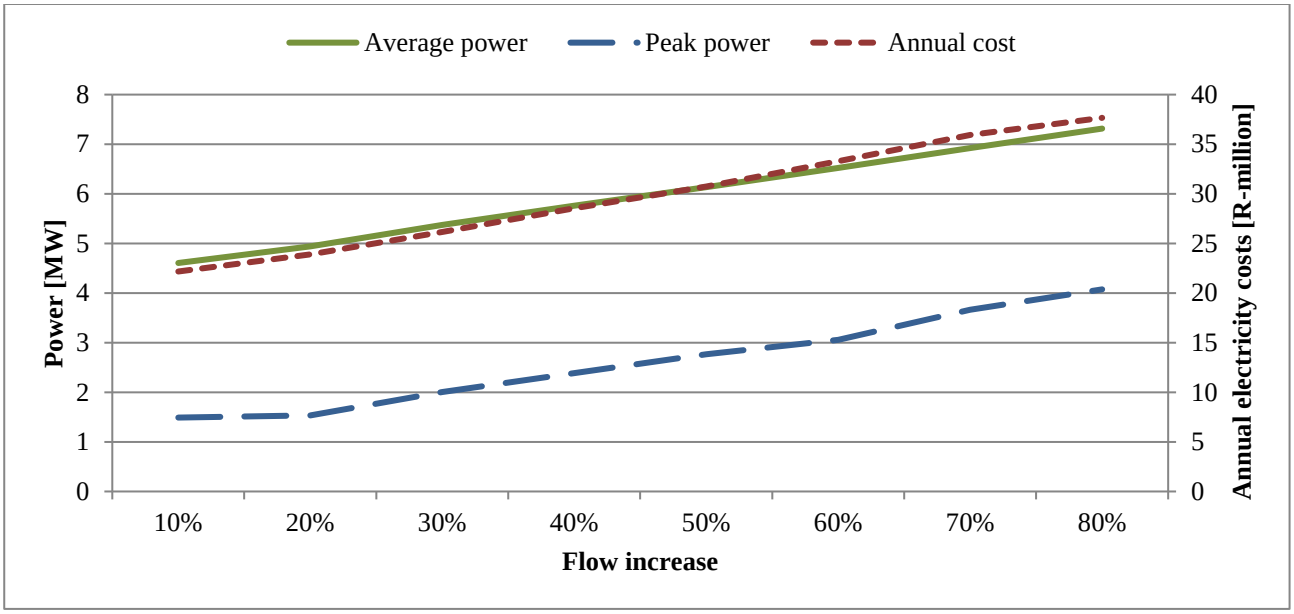



**Figure 46: Mine A – Increased flow demand, closed-loop high-pressure u-tube**

The increased flow increases the power usage of the system as expected. The load management is also negatively affected. However, the difference between the peak power usage and the average power usage remains almost constant for all of the simulated flows. Therefore, even though the load management potential is negatively affected, some potential remains for all of the simulated flows.

Figure 47 shows the effect of increasing the flow demand on the overall system power usage, peak power usage and annual system energy costs for a system with a turbine pump system installed.

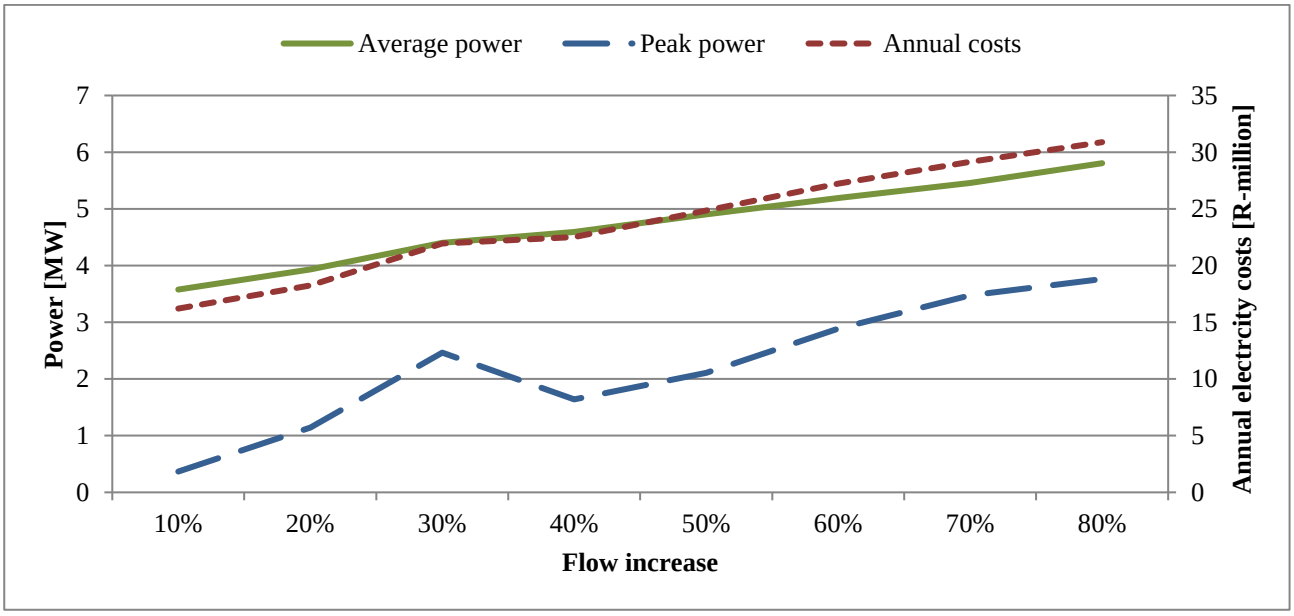



**Figure 47: Mine A – Increased flow demand, turbine pumps**

The effect of on the power usage and load management potential for a system with turbine pumps is similar to that of the system with a closed loop high-pressure u-tube system installed. However, the load management potential is somewhat more pronounced.

The reason for this is the effect that the turbine pump has on the 38L hot water dam especially during periods when the water demand is high. During such times the 38L hot dam is dewatered while the water supply is being met. If the level of the 38L hot dam starts running dangerously low, pumps on the 75L pump station have to start to avoid damaging the turbine pump with muddy water.

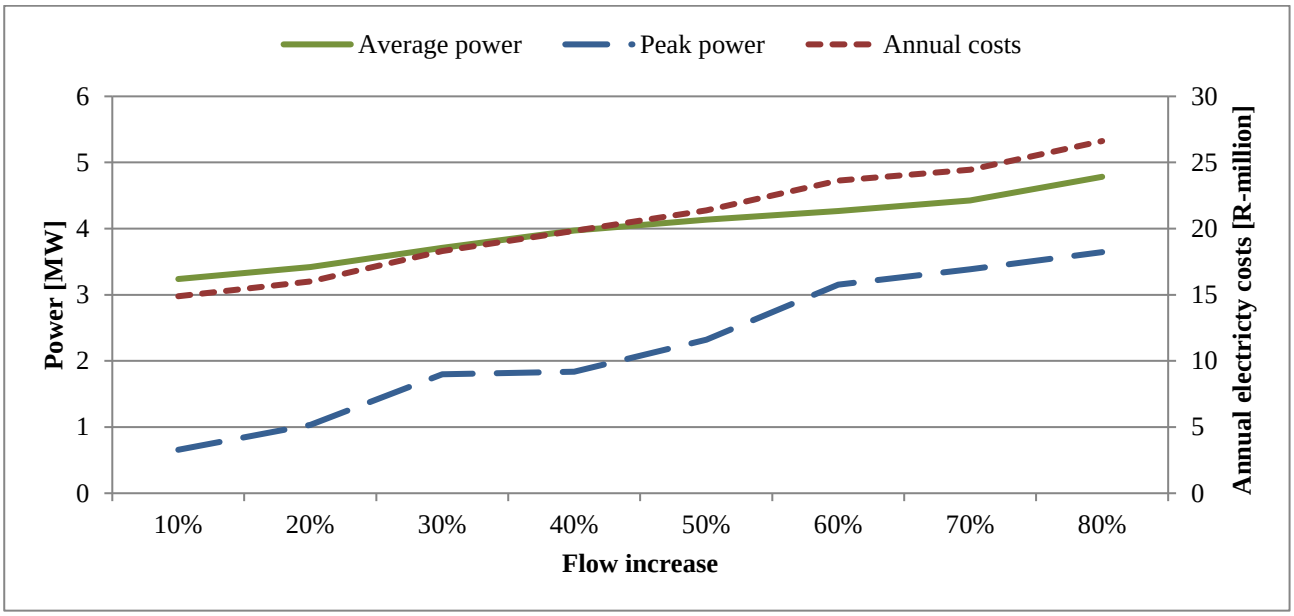



**Figure 48: Mine A – Increased flow demand, 3CPFS**

As in all the scenarios, a significant increase in power usage and a decrease in load management performance is observed. The load management potential decrease is more pronounced than that of the other ERDs. This is because the 3CPFS has a similar effect on the 38L dam than the turbine pumps, but due to the higher dewatering flows of the 3CPFS, the effect is more pronounced.

In general, the peak power is shown to rise as the flow demand rises. However, the overall power usage of the systems increases with a similar slope for the system arrangements that has ERDs installed whereas in the system with no ERDs the peak power usage converges to average power especially at high demand flows. This highlight the potential for load management on systems with ERDs installed.

A definite increase in the power usage of the systems is evident as the flows increase. This was expected since the water demand is the main energy driver on dewatering systems. It also highlights

the need to control the flow strictly to the demand and therefore the importance of water supply optimisation initiatives on any dewatering systems that.

It should be noted that even though the ERDs still offer significantly better efficiency at high flow rates compared to no ERDs, the electricity costs still rise significantly as the flow rises. This cost increase emphasises the need to control the system strictly to the demand and to continually look for ways to reduce the demand flow on such systems.

Table 62 summarises the integrated effect of the flow increase on the performance for some of the key simulated scenarios.

**Table 62: Mine A – ERD integrated WSO effect summary**

| **Baseline** | | | | |
|---|---|---|---|---|
| **Parameter** | **No ERD** | **Turbine pump** | **3CPFS** | **Closed loop high-pressure u-tube** |
| Average power usage [kW] | 4815 | 3285 | 2770 | 4120 |
| Peak power usage [kW] | 990 | 190 | 380 | 555 |
| Annual electricity cost [R-million] | 22.9 | 15.0 | 13.3 | 19.4 |
| **40% Dam capacity reduction** | | | | |
| **Parameter** | **No ERD** | **Turbine pump** | **3CPFS** | **Closed loop high-pressure u-tube** |
| Average power usage [kW] | 4805 | 3260 | 2760 | 4130 |
| Peak power usage [kW] | 2620 | 995 | 1770 | 1360 |
| Annual electricity cost [R-million] | 23.9 | 15.3 | 14.3 | 20.1 |
| Percentage cost increase [%] | 4 | 2 | 7 | 4 |
| **80% Dam capacity reduction** | | | | |
| **Parameter** | **No ERD** | **Turbine pump** | **3CPFS** | **Closed loop high-pressure u-tube** |
| Average power usage [kW] | 4850 | 3295 | 2760 | 4180 |
| Peak power usage [kW] | 3860 | 2230 | 2230 | 3095 |
| Annual electricity cost [R-million] | 26.7 | 17.6 | 15.3 | 22.3 |
| Percentage cost increase [%] | 17 | 17 | 15 | 15 |
| **40% demand flow increase** | | | | |
| **Parameter** | **No ERD** | **Turbine pump** | **3CPFS** | **Closed loop high-pressure u-tube** |
| Average power usage [kW] | 6710 | 4595 | 3975 | 5765 |
| Peak power usage [kW] | 2910 | 1640 | 1840 | 2390 |
| Annual electricity cost [R-million] | 33.6 | 22.5 | 19.8 | 28.6 |
| Percentage cost increase [%] | 47 | 50 | 49 | 47 |
| **80% demand flow increase** | | | | |
| **Parameter** | **No ERD** | **Turbine pump** | **3CPFS** | **Closed loop high-pressure u-tube** |
| Average power usage [kW] | 8415 | 5805 | 4785 | 7315 |
| Peak power usage [kW] | 7630 | 3760 | 3645 | 4075 |
| Annual electricity cost [R-million] | 48.4 | 30.9 | 26.6 | 37.7 |
| Percentage cost increase [%] | 111 | 106 | 100 | 94 |

It is clear that the percentage cost increase for all of the ERDs are similar if the dam capacity is reduced. However, the cost increase for the ERDs is slightly lower than the scenario with no ERDs if the flow is increased.

Note that even though the percentage price increase for all of the scenarios is similar when the dam capacity is reduced, the power usage in peak times remains lowest for the scenarios where ERDs is present. The power usage during peak times is also significantly lower in the cases where ERDs are present for all of the scenarios.

Turbine pump offers the best load management performance. However, with an 80% increase in flow demand the 3CPFS surpass it with regards to both load management potential and overall efficiency. The likely reason for this is the higher pumping capacity that the 3CPFS offers.

At high flow demands the percentage price increase for the scenarios with ERDs present is also slightly lower compared to that of the scenario with no ERDs present. This indicates that the spare pumping capacity that the ERD introduces to the system has a positive impact on the load management potential in general.

In addition to this, it is interesting to note that the percentage cost increase is significantly higher than the percentage flow increase for all of the simulated scenarios. This indicates that there is an integrated relationship between flow demand and load management potential. Therefore the implementation of WSO initiatives could benefit load management initiatives on dewatering systems with and without ERDs.

# **Appendix E:** Mine B simulation layout

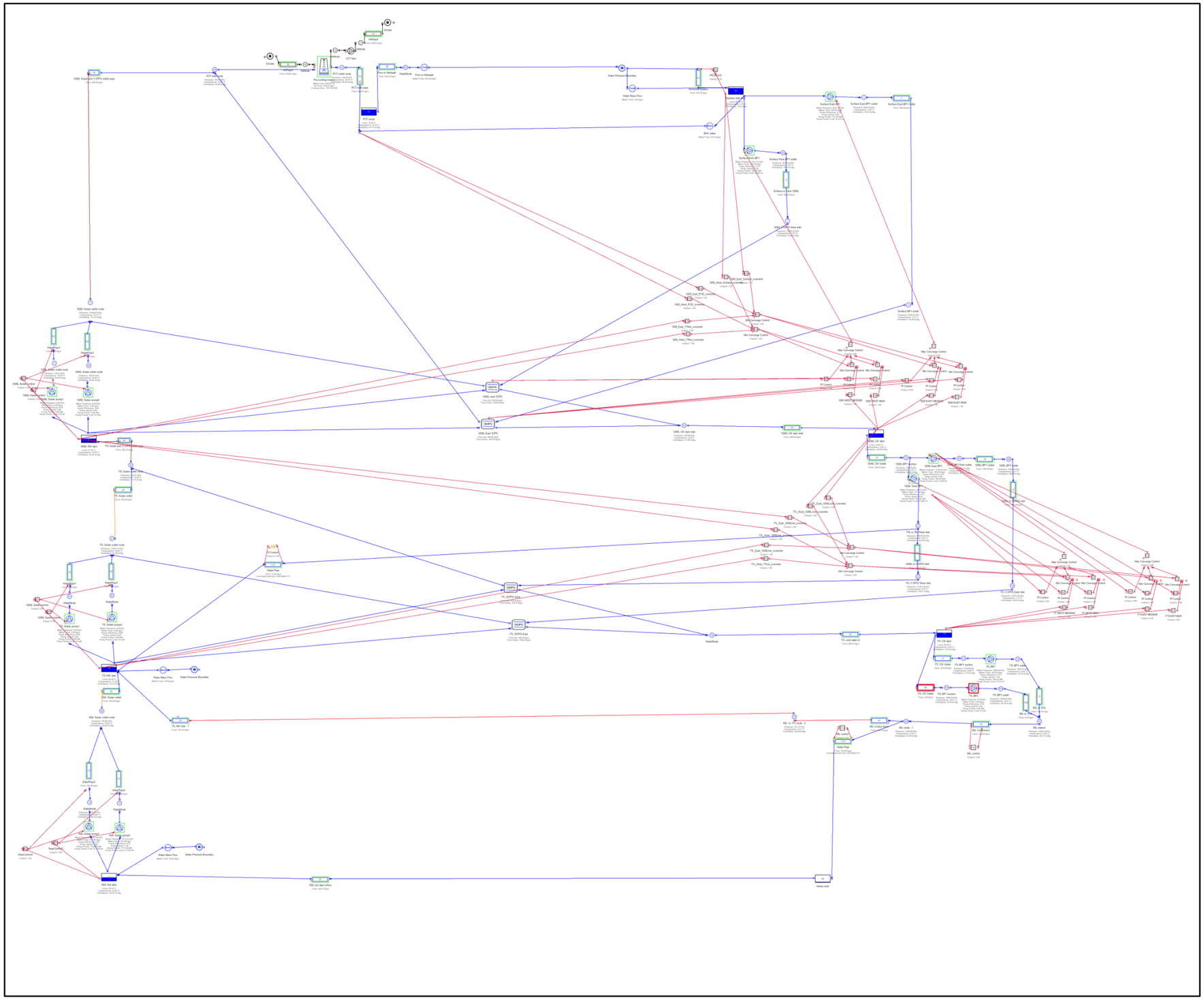


**Figure 49: Mine B – Simulation model layout**

# **Appendix F:** Mine B reporting example

[redacted]

## 1 Project information

**Project name:** [redacted]

**Project number:** [redacted]

**Tariff structure:** MEGAFLEX Cost =<300km =>500V & <66kV

**Target impact:** 1.40 MW

## 2 Performance ([redacted])

| **Performance of day:** | | **Month-to-date performance:** | |
|---|---|---|---|
| Impact: | 1.98 MW | Average impact: | 1.89 MW |
| Cost saving: | R 3 018 | Cumulative cost savings: | R 10 581 |
| Missed opportunities: | - | Cumulative missed opportunities: | - |

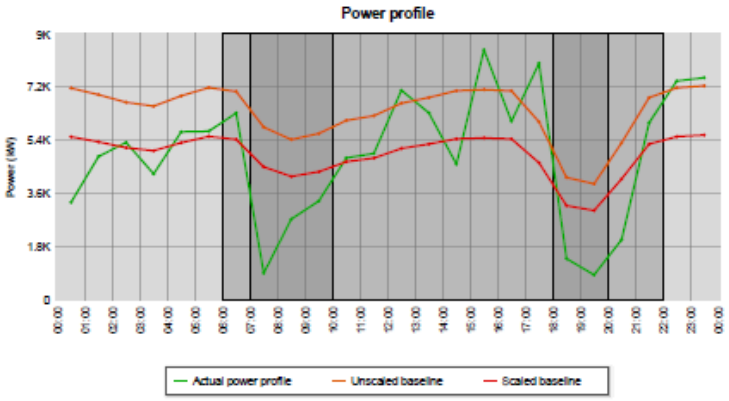



Figure 2.1: *Power profile and baseline for* [redacted]

[redacted]

[redacted]

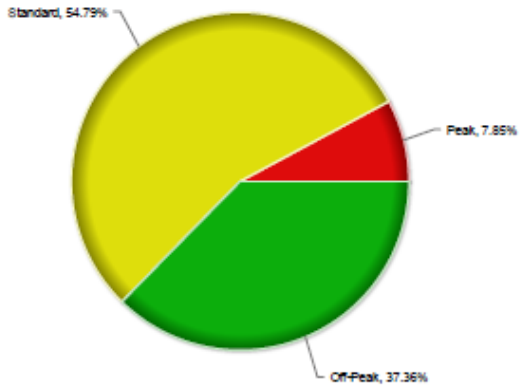



Figure 2.2: *Daily energy usage distribution -* [redacted]

[redacted]