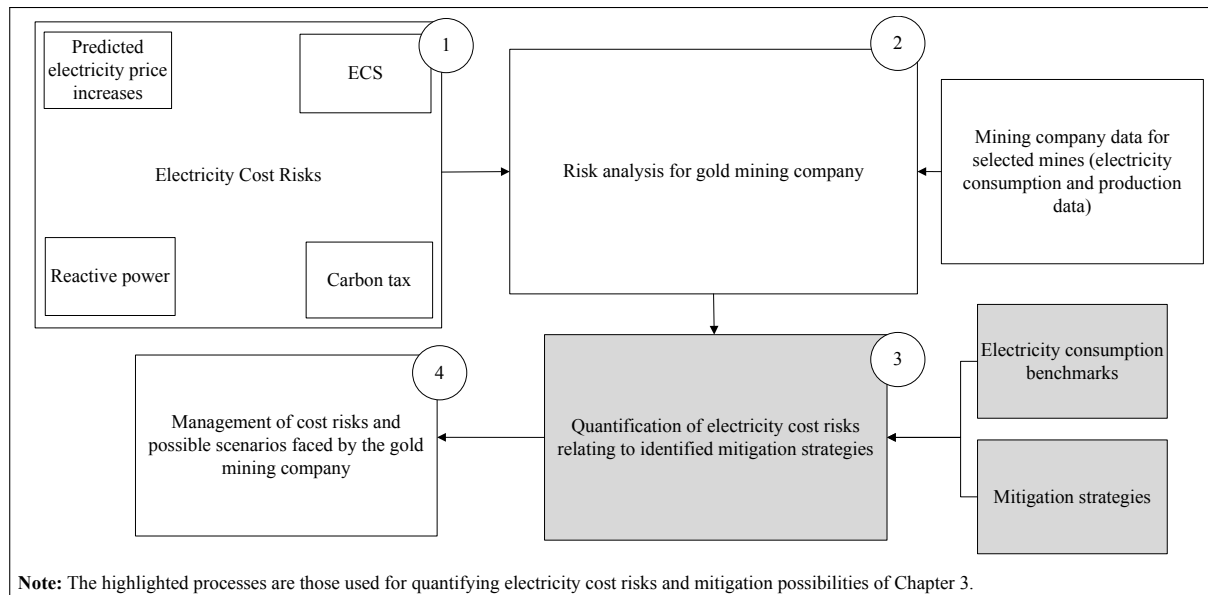


# Chapter 3 : Mitigation through benchmarking and DSM



## 3.1 Introduction

In Chapter 2 the electricity cost risks were discussed. These risks indicated a high potential price increase of up to 12% if these risks are not well managed. It is evident that the large gold mining companies in South Africa must therefore save electricity to mitigate the potential financial impacts. In this chapter, the highest consuming services of a selected mining company will be benchmarked. From the benchmarked services, the potential to develop mitigation strategies will be discussed. Technologies and proven techniques will be reviewed to quantify the possible financial impact.

## 3.2 Benchmarking electricity consumption

There are several electricity-intensive mining operations in the gold mining process. These electricity consuming mining operations include: drilling, blasting, loading, hauling and ancillary services. Every gold mine is unique and differs in terms of gold grade, depth, rock type and allocation of resources. This influences the energy intensity of the mine. Benchmarking has been used in several studies to identify misuse of electricity or to identify a relationship between electricity consumption and gold production [1–3].

The aim of this section is to set the benchmark values for the largest energy-consuming services of the selected gold mining company. The first goal was to identify mines that are low electricity consumers but high gold producers, and from this, to benchmark the mines according to electricity risk. This will provide a starting point to identify the large consumers and best mining practises in order to derive mitigation strategies.

The mining company selected has 61 electricity billing points related to the mining shafts on twelve South African mining operations. From the data set, eight mines were selected according to the following:

### **Impact representation**

The eight selected mines form part of the top ten electricity users of the company. Identifying opportunities and implementing electricity-reducing projects there will provide the biggest impact on the mining company compared to the other mines. The top ten electricity consuming mines consume 74% of all the South African operations. The top ten mines also form part of the mining company's biggest development area. The eight mines selected would thus provide a very good estimation of the possible cost risks faced by the mining company and the possibility of reducing electricity.

### **The availability of data**

Data correctness is essential in providing possible electricity cost estimates for the mining company. The mines were selected according to the availability of the data. Installed capacities, flow and pressure data were obtained from the Supervisory Control and Data Acquisition (SCADA) systems of each mine. The savings obtained from the projects in the presented case studies were verified by an independent M&V professional. Other account information received was obtained via Eskom accounts and did not deviate from the measured data by more than 2%. The incoming data for each service was obtained through electricity measurement equipment on site, used by mining personnel to verify the provided Eskom accounts. It can therefore be assumed that the data obtained and used within the study is correct.

## **3.2.1 Advantages of benchmarking**

There are several advantages to using benchmarking to quantify risk and to determine the mean values for electricity consumption services. These advantages include:

- Helping to identify electricity inefficient use.
- Improving efficiency which can then help with increased production.

- Providing a standard for comparison, and as a guideline for new mining development.
- In case of an emergency with low electricity supply, benchmarking will provide a starting point of mitigation, identifying mines with high electricity consumption and low profit contribution.
- The standards will induce a lower direct mining cost and improve profit margins.

Of the international energy management standards, only a few relate to the mining industry. These common international industrial standards include [4]:

- Energy policies and procedures training.
- Strategic plans that require measurement.
- Identified key performance indicators for each company and mining sector for specific countries to measure progress and optimise industrial systems.

From the provided international best practise standards for benchmarking, it is clear that a production entity cannot manage or understand inefficient use of electricity if the electricity cannot be measured. The approach derived for benchmarking the main electricity consuming elements and used to develop mitigation strategies is illustrated in Figure 3.1.

The first step was to collect all the needed data to create benchmarks for the selected mines. The mines were categorised to identify possible improvement with other similar but well-performing mines. The three largest electricity consuming mining services were identified and all the needed information listed. The possible mitigation for each mining service was summarised and then validated with a case study. The mitigation results and approach taken for each service will be discussed in the following sections.

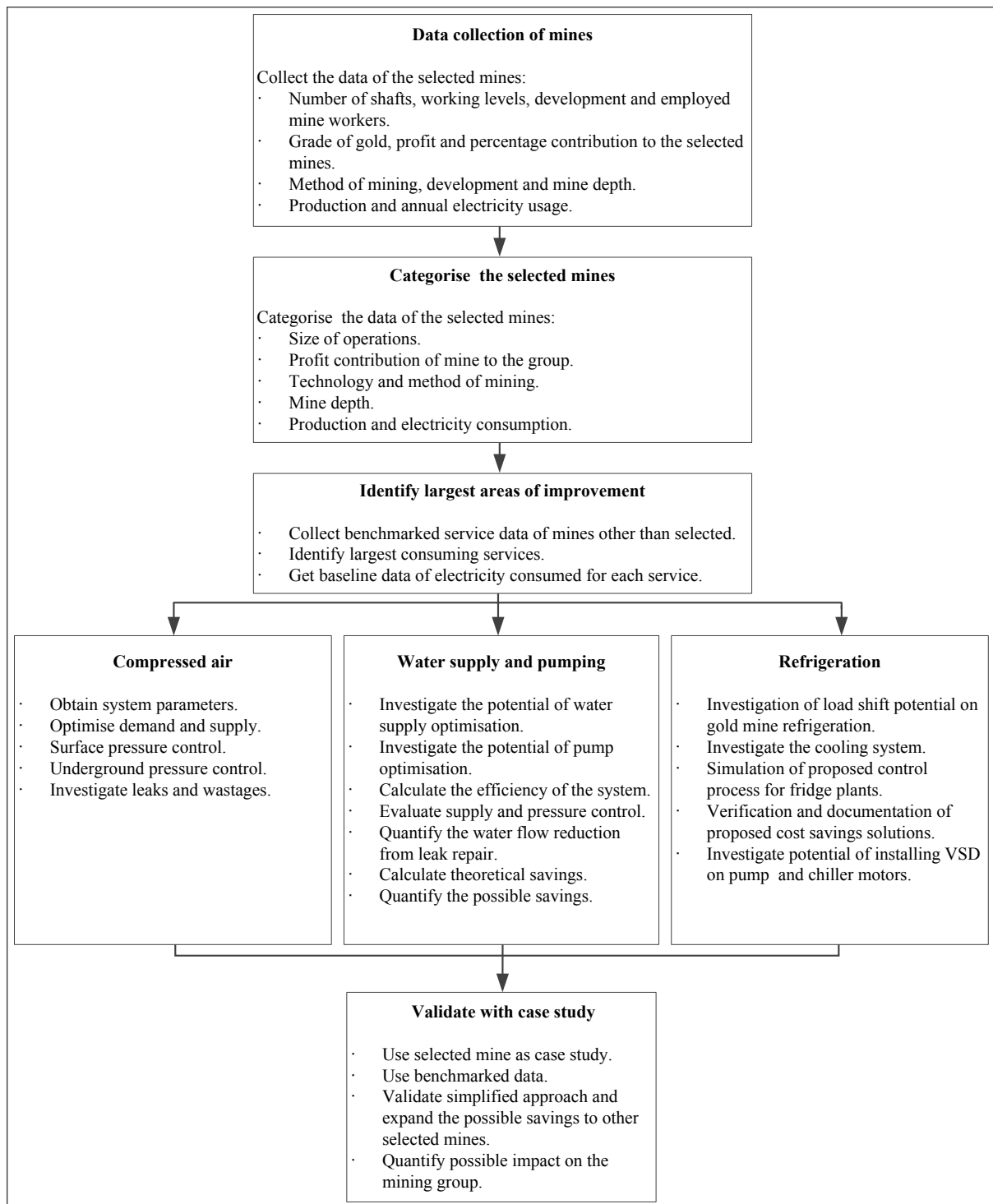


Figure 3.1: Illustration of the benchmark approach used to identify possible mitigation strategies.

### 3.2.2 Benchmarking points

Several successful international energy reducing programmes are available [5]. One of the key programme elements includes benchmarking of present energy efficiency programmes. Canada, a leading energy efficient country, has compiled the relevant costing for gold and

other underground bulk mining minerals and compared it to international standards [6]. By means of benchmarking, the electricity cost savings potentials of the different analysed mines were determined.

The previous benchmarking studies performed compared production and electricity consumption. By identifying the best electrical performance in relation to production, the possible production improvement or electricity reduction was calculated. However, this could provide a false representation or opportunity as the key electricity influential aspects of each mine were not listed. High-production mines could provide a false representation of being verified as electricity efficient. It is thus crucial that all the information is available to provide better insight into possible improvement in electricity usage.

This would provide the opportunity for comparing similar mines with other mines and providing a more accurate calculation of possible electricity usage improvements. From international best practises for energy management, the main steps for benchmarking typically include [4], [6]:

- Identify areas of interest to be monitored or that will benefit from benchmarking.
- Research and collect usable data for the abovementioned areas of interest.
- Collect and categorise the relevant data sets, selecting the best analysed sector with the best relevant energy efficiency.
- Determine and evaluate the conditions under which the best energy efficient rated scenarios can be obtained and the required actions to obtain this.
- Derive a continuous monitoring process and set up a detailed action plan and targets to obtain the determined best class practices.

### **3.2.3 Benchmarking results of selected mines**

For the benchmark analysis, eight similar gold mines of the mining company were identified and the production and key production services were tabulated. The first step in the benchmark process was to collect data from the eight mentioned mines. The data and variables selected must have an influence on or correlation with the energy intensity of the mines. The following data elements were selected to aid in providing benchmark descriptions or analysis.

## Operations size

The size of the operations will be encapsulated in the number of employees, number of shafts, and tonnes milled. The goal will be to find the relationship between the size of operations and electricity consumption. In the case where operations need to respond to load shedding or limited power supply, the determined benchmark will indicate the effect which load shedding will have on the mining group financially and labour-wise. The size of operations will be classified as either small, medium and large. The three classes will be determined first, by comparing the eight mines and dividing each mine into its own 33% percentile for the three key aspects - production, number of employees and number of shafts. See Table 3.1.

Mine	Production Levels working on	Number of shafts	Development	Employed mine worker		Production x 1000 t (metric)		Scale of operation
				2011	2010	2011	2010	
Mine A	6	1	Steady state	3 050	3 067	868	899	Medium
Mine B	5	1	Closing down	3 418	3 611	426	528	Medium
Mine C	3	1	Steady state	2 547	2 865	541	788	Medium
Mine D	3	2	Steady state	1 436	1 436	407	439	Small
Mine E	8	2	Build up	4 983	5 049	1 099	1 035	Large
Mine F	8	1	Build up	2 866	2 858	387	339	Medium
Mine G	3	1	Build up	2 811	2 676	805	777	Medium
Mine H	7	1	Steady state	4 982	4 901	1 343	1 518	Large
33 % Percentile	4			2 828	2 860	462	605	
66 % Percentile	7			3 278	3 404	844	857	

Table 3.1: Scale of operations for the eight selected mines of the selected mining company.

## Profit contribution of each mine

The operating profit is defined by the grade of the gold, the direct operating cost and the profit reported in the annual report. The profit contribution of each mine will be analysed to determine the contribution relationship between the selected mines.

The abovementioned factors will aid in selecting mines or areas where electricity cost risks may be present. Profit contribution and operational size are key indicators when developing mitigation strategies to minimise penalties during high electricity cost risk periods such as ECS. The mines will be classified as either highly profitable or fairly profitable as listed in Table 3.2. It should be noted that key individual mines (Mine A, E and H) contribute ten times more compared to the others. This is due to some mines still being developing mines with high operational costs. Typically in emergency situations, the lower-profit contributing mines will be isolated to avoid penalties.

As illustrated in Chapter 2 the mining company could face a minimum predicted price increase of 47% from the ECS by managing their reference consumption and continuing operations as normal. Due to the excess rates of the ECS the electricity component of the

direct mining cost would then increase from 13% to 19%. The additional 6% increase would result in a potential loss in profit. The goal would be to put low profitable or developing mines on hold or to reduce the electricity of the company below the penalty range. The potential electricity price increase could be absorbed if the gold price is very high. It is therefore crucial that the mining company is aware of the profit contribution of each mine in relation to its electricity consumption.

Mine	Grade gold average (g/t)		Profit (Rand-million)		Operating cost (Rand-million)		Percentage contribution of the selected mines	Profitability
	2011	2010	2011	2010	2011	2010	2011	2011
Mine A	5	5	568	567	758	710	23%	High
Mine B	7	8	66	385	855	729	3%	Fair
Mine C	4	4	103	48	614	862	4%	Fair
Mine D	5	5	137	127	387	376	6%	Fair
Mine E	5	5	504	255	1 270	1 137	20%	High
Mine F	5	4	76	57	475	318	3%	Fair
Mine G	4	4	176	203	904	675	7%	Fair
Mine H	5	4	830	710	1 177	1 113	34%	High

Table 3.2: The profitability and operational costs of the selected mines.

## Technology and method of mining

Technology and energy efficient design play an important role in the electricity consumption profile of any mine. The equipment used to monitor and for mining will aid in providing an indication of the expected electricity consumption or demand of the analysed mines.

The method of mining refers to either mechanised or conventional mining. Mechanised mining methods refer to the use of trackless mechanised mining equipment for both development and stope mining. Mechanised mining equipment is used for most mining activities including drilling for blast and support holes as well as cleaning. Mechanised mining consumes less compressed air and is less labour intensive. Mechanised mining is more efficient in hauling gold faster than the conventional way.

Conventional mining refers to mining by means of hand-held drilling machines (normally powered by compressed air) for drilling support and blast holes. Cleaning is accomplished by means of scrapers connected to winches. Mines will either be classified as conventional or mechanised mines, as listed in Table 3.3.

From mechanised mining design objectives, it can be assumed from the data obtained that mechanised mines will have lower energy usage due to less compressed air being used for hand-held compressed air machinery as well as other components. Production can be higher due to the safer environment which will induce fewer production delays due to accidents or fatalities. This results in lower labour costs and increased productivity.

## Mine depth

Deeper mines require more cooling and consume more electricity to pump out water from greater depths. The relationship between pumping energy and cooling relative to mine depth will be explained in Sections 3.3 and 3.5. Mine depth relates to high rock pressures where more energy is required to create a safe environment. The mines are classified as shallow (Depth < 2000m), deep (2000m < Depth < 3000m) and ultra deep (Depth > 3000m) [4], [7].

Mine	Method of mining	Development	Mine depth (m)	Mine type
Mine A	Conventional mining	Steady state	2300	Deep
Mine B	Conventional mining	Closing down	3264	Ultra Deep
Mine C	Conventional mining	Steady state	2362	Deep
Mine D	Conventional mining	Steady state	1400	Shallow
Mine E	Conventional mining	Build up	3600	Ultra Deep
Mine F	Conventional mining	Build up	2400	Deep
Mine G	Mechanised mining	Build up	2350	Deep
Mine H	Conventional mining	Steady state	2366	Deep

Table 3.3: Mine depth and the classification of the mining method for the eight mines selected.

## Production and electricity consumption

This is probably the most valuable benchmark to determine the relationship between electricity usage and production. The relationship between these parameters and the selected gold mines is illustrated in Figure 3.2 and Table 3.4. The mines are also labelled so that mines with similar electricity-affecting characteristics are grouped to provide a better estimation of potential electricity cost savings. The classification code given to each mine is based on the previously discussed key aspects of the mines. The code description is provided at the bottom of each table.

Mine Classification	Mine	Production (t metric x1000)	Energy used (MWh x 1000)	Intensity (kWh/t)
↓S-C-*S-ŘF-LE	Mine D	407	108	265
↓D-C-*M-ŘH-AE	Mine A	868	249	287
↓D-C-*L-ŘH-LE	Mine H	1 343	314	234
↓D-C-*M-ŘF-HE	Mine C	541	280	518
↓D-C-*M-ŘF-AE	Mine F	387	95	245
↓D-M-*M-ŘF-AE	Mine G	805	337	419
↓UD-C-*L-ŘH-HE	Mine E	1 099	663	603
↓UD-C-*M-ŘF-HE	Mine B	426	471	1 106
↓S=Shallow	C=Conventional	*S=Small scale operation	ŘF=Fair profitable	LE=Low electricity consumption
↓D=Deep	M=Mechanised	*M=Medium scale operation	ŘH=High profitable	AE=Average electricity consumption
↓U=Ultra deep		*L=Large scale operation		HE=High electricity consumption

Table 3.4: Production and annual electricity consumption values for the selected mines.

From the presented data sets, there is a vast difference in the consumption and production profiles. Mine H consumes 47% less electricity, but yields greater production than the largest

electricity consumer, Mine E. The goal of the following sections is to determine the reasons for the vast differences in production output, as well as potential electricity consumption targets for the related production.

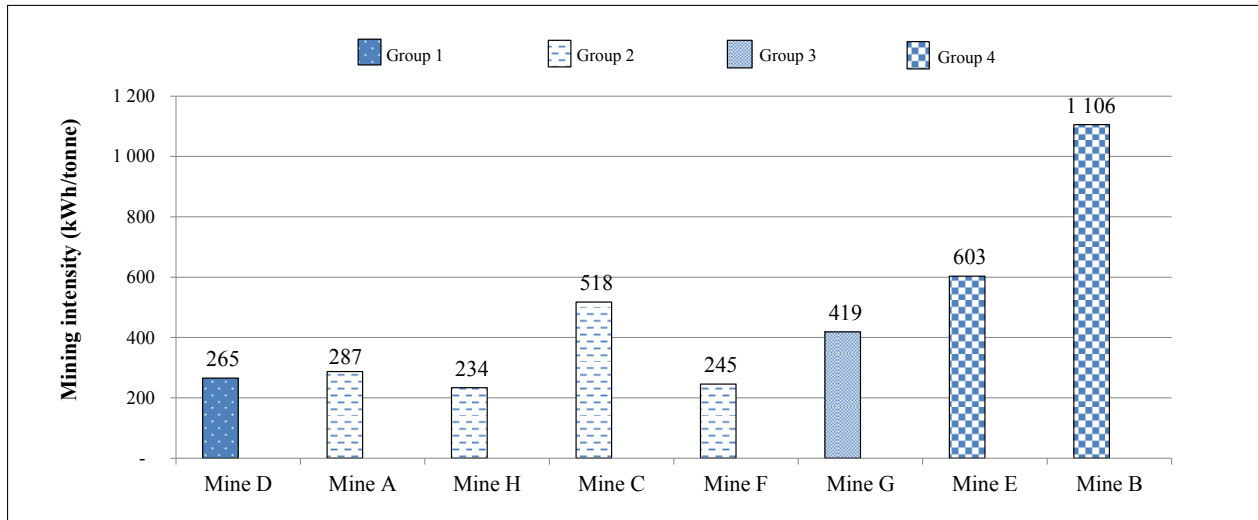


Figure 3.2: Comparison of mines for electricity consumed per tonne milled.

### 3.2.4 Electricity consumption and related installed capacities

In the previous sections each mine was categorised and benchmarked according to its descriptive elements. This is an indication of the large difference in electricity consumption and how characteristics such as mine depth, operation size and production affects this.

The next section will provide the consumption benchmarks for the largest consuming services of the gold mining industry. The intention is to focus on the largest electricity consuming services such as pumping and compressing air and by this, identifying solutions that will produce the largest possible savings for the mine. From the benchmarked services, the similar previously categorised mines can be compared to identify opportunities. The benchmarked services for South African gold mines (other than those used in this study) is illustrated in Figure 3.3. The following sections will identify areas of improvements for the large electricity consuming services.

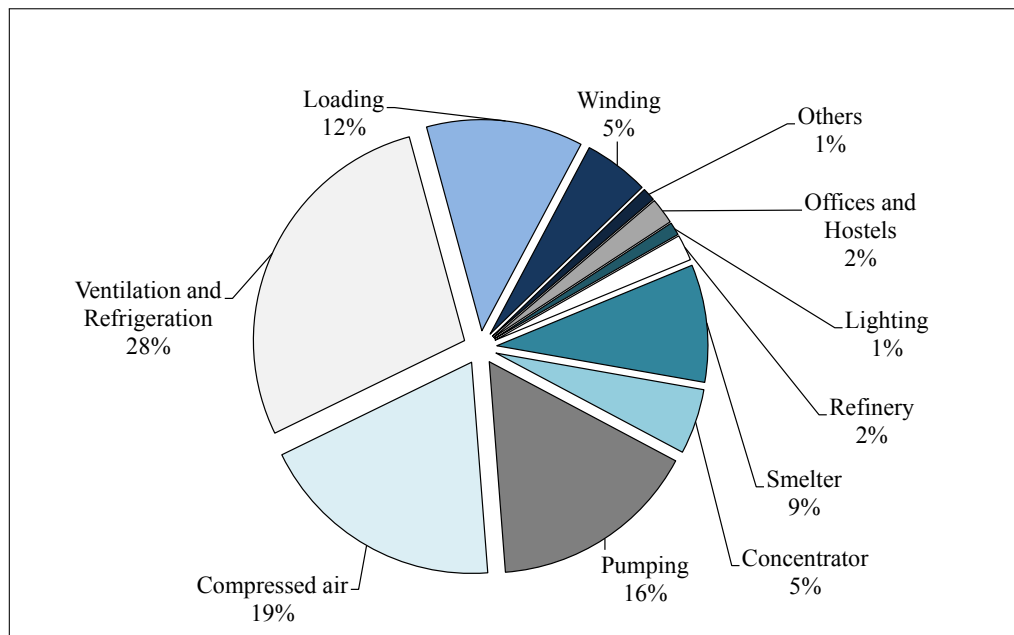


Figure 3.3: Allocation results of gold mines in South Africa other than those in the study [4].

From Figure 3.3 the largest electricity consuming sectors include ventilation and cooling systems (28%), compressed air (19%) and pumping (16%). Each of these services is essential to production and studies have been performed to relate these main services to the number of tonnes hoisted [1, 2, 6, 8].

In the sections below, the annual electricity consumption for the three identified services will be correlated to the annual electricity consumption and production. Each mine's installed capacity for these large electricity consuming services is also listed to provide insight in the electricity usage in relation to the installed capacity.

### 3.3 Pumping

Water needs to be sent down the shafts for mining activities that include drilling, cooling and sweeping [9]. Supplying and distributing water in gold mines could be a highly energy intensive and dangerous task due to the high hydraulic pressure build-up from the extreme operating depth. The pumping and cooling electricity consumption of gold mines is directly impacted by the depth of the mine [10].

Apart from mining water sent down, large amounts of daily fissure water needs to be pumped out to enable a safe working environment. In certain cases, the fissure water can be more than half of the water pumped out [8]. The average daily water pumped out from Mine E is 24 Mℓ which is almost equivalent to the volume of ten Olympic-size swimming pools. A simple method to determine the daily theoretical energy consumption of a such a large

dewatering system is to make use of the following energy calculation [11]:

$$Eps = M \times g \times h \quad (3.1)$$

where

$Eps$  = Daily energy used to extract water from the pump station (J)

$M$  = Mass of water pumped (kg)

$g$  = Gravity acceleration constant (9.81m/s<sup>2</sup>)

$h$  = Total head of the pumping station (m)

When performing the basic calculation, the static head due to the extreme depths of the mine must also be included in the theoretical electricity calculation. It is estimated that one can increase the static head by 5%. To compensate for frictional losses, an additional 5% can be added to the theoretical calculated head [12].

From Equation 3.1 one should note that the head or depth of the mine is directly proportional to the required energy to pump the water. From this, it is expected that a deep mine requires larger amounts of pumping energy than less deep mines, given that the mines are located in a similar water basin area. For the selected mines, the installed pump capacities as well as the electricity consumption were listed, as seen in Table 3.5.

Pumping electricity usage for mining group						
Mine Classification	Mine	Installed capacity (kW)	Average electricity consumed per month (kWh)	Average electricity consumed per year (kWh)	% Pump electricity consumed of total electricity	Pump electricity per tonne milled (kWh/t)
↓S-C-*S-ŘF-LE	Mine D	3 000	891 384	10 696 608	14%	26
↓D-C-*M-ŘH-AE	Mine A	11 250	3 216 001	38 592 012	15%	44
↓D-C-*L-ŘH-LE	Mine H	16 500	3 044 384	36 532 608	12%	27
↓D-C-*M-ŘF-HE	Mine C	9 000	1 534 700	18 416 400	7%	34
↓D-C-*M-ŘF-AE	Mine F	6 000	2 352 000	28 224 000	13%	73
↓D-M-*M-ŘF-AE	Mine G	7 636	1 033 664	12 403 973	4%	15
↓UD-C-*L-ŘH-HE	Mine E	26 795	7 165 767	85 989 206	13%	78
↓UD-C-*M-ŘF-HE	Mine B	23 000	7 742 057	92 904 682	20%	218
↓S=Shallow		C=Conventional	*S=Small scale operation	ŘF=Fair profitable	LE=Low electricity consumption	
↓D=Deep		M=Mechanised	*M=Medium scale operation	ŘH=High profitable	AE=Average electricity consumption	
↓U=Ultra deep			*L=Large scale operation		HE=High electricity consumption	

Table 3.5: Installed pumping capacities for the mines with related production and electricity consumption.

The relationship between production and pump electricity consumed is illustrated in Figure 3.4.

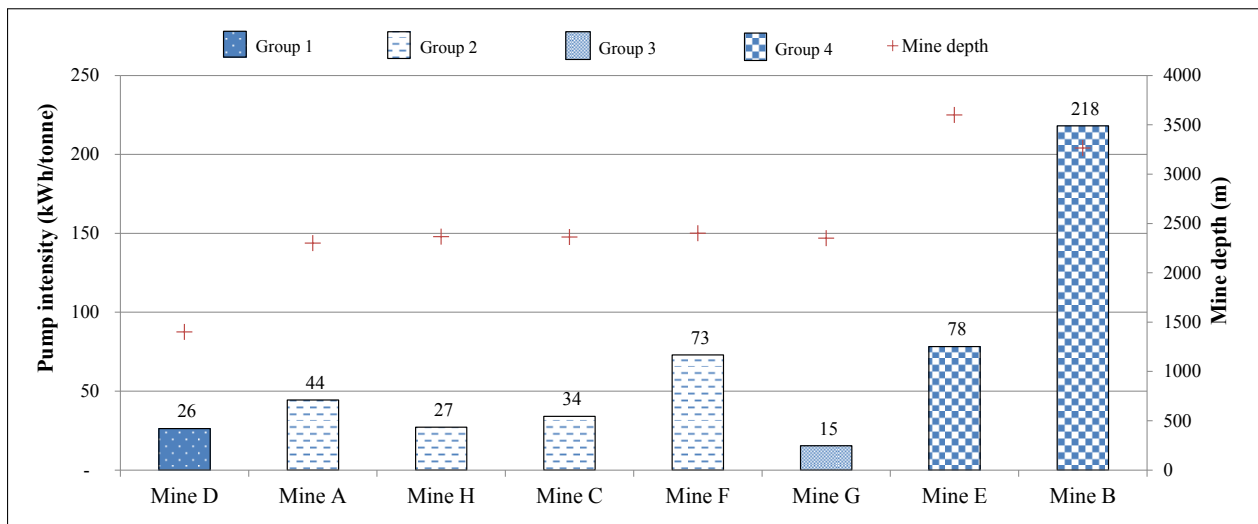


Figure 3.4: Amount of pumping electricity consumed related to production.

From Table 3.5 it should be noted that the ultra deep mines (Mine E and Mine B) consume the most pump electricity compared to the other six mines. The installed capacity of these two mines is also substantially larger than the rest. The percentage of pump electricity consumed as a percentage of the total electricity consumed for the eight mines is in the same range as the initial predicted 16% consumed pump electricity of Figure 3.3.

The consumed pump electricity of Mine G is much lower than the other mines. This is due to Mine G having a Three Chamber Pipe Feeder System (3CPFS), which recovers energy from the incoming water and then reapplies it to pump water to surface dams. A simplified layout of a dewatering system of a deep level gold mine in South Africa is illustrated in Figure 3.5 and can be described by the following numbered items:

1. Water is sent down the shaft for mining purposes.
2. After being used, the water is gravity-fed into channels back down the shaft to the bottom or nearest pumping station.
3. To filter the used mining water, the water is first passed through settlers to separate the heavier material particles from the water.
4. The water is pumped up to the next pumping station.
5. The water is pumped to the surface. More than one pumping station may be required, depending on mine depth.

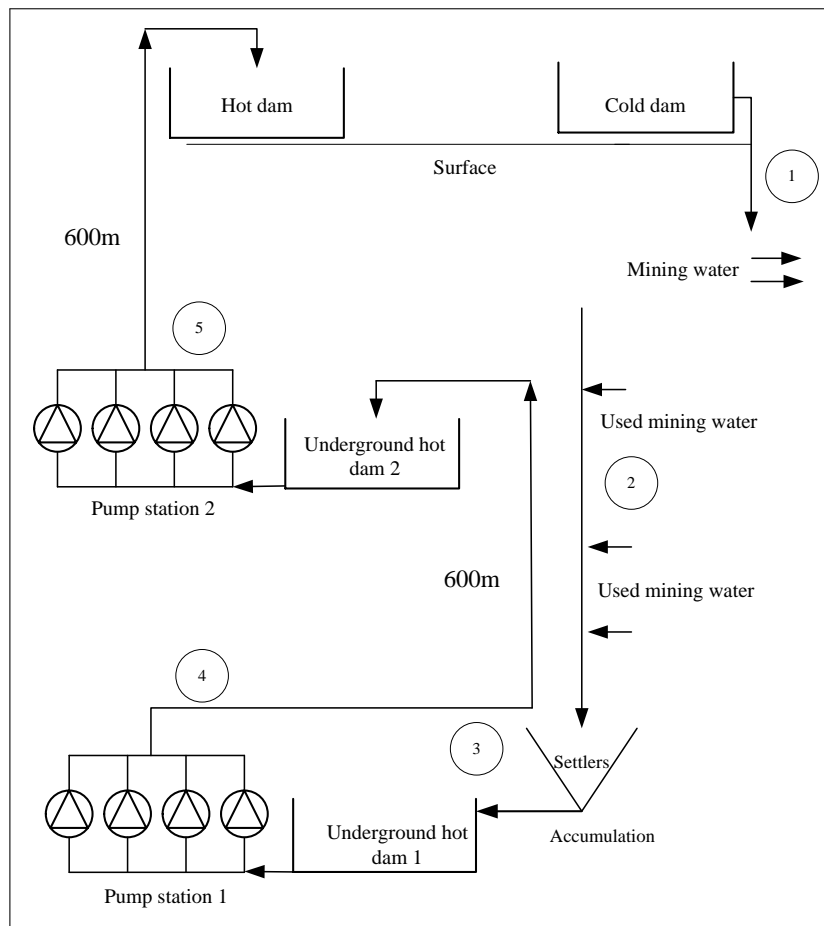


Figure 3.5: Simplified dewatering layout of a typical deep level gold mine in South Africa.

At each pump station there are several pumps used to empty the dams during high water supply periods. The combination of pumps that are operated at certain instances determines the flow in relation to the head as illustrated in Figure 3.6. The pump delivery flow is dependent on the water line diameter and number of pumps running.

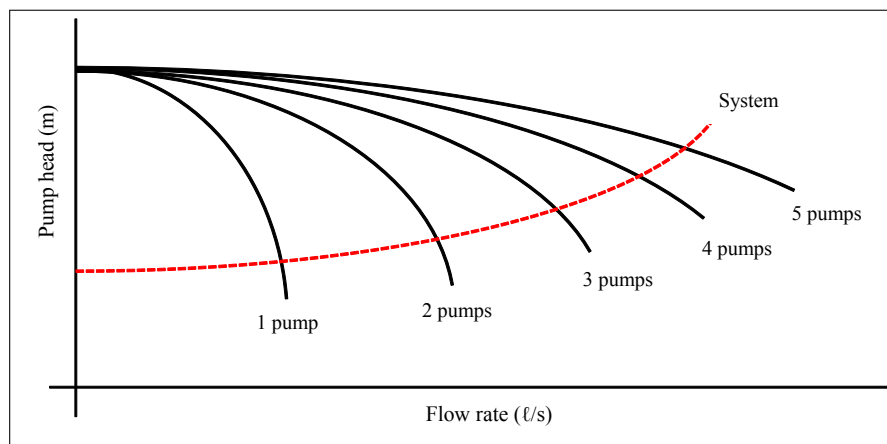


Figure 3.6: System pump curves illustrating the relationship between the number of pumps and head [8].

Two available known solutions in reducing the electricity demand for pumping systems are reducing the water consumption and/or improving the pump efficiencies [8], [13]. Other available solutions include the installation of turbines or 3-CPFS system. These systems could take up to three years to install and the payback periods of these projects can take up to two and a half years [14], [15]. Due to the cost and implementation period, these systems will not be investigated but may be beneficial for a mine that is interested in a long-term operating efficiency solution.

### Pump efficiencies

Pump efficiencies can be improved by selecting the combination of pumps with respect to the common delivery manifold. The flow can only be increased to a point where the pump curve intersects the system curve, as illustrated in Figure 3.6. The operated flow rate for a system not being well monitored could result in the electricity efficient operating point not being optimal [16], [17]. It is thus recommended for the mine to always run a combination of pumps that produce the most efficient flow in relation to electricity consumption whilst keeping safety and dam levels in mind.

Regular maintenance on large pumps could be performed to improve efficiency. Discharge flow is used to reduce thrust loading on the pump bearing by discharging the flow against a balance disk. These balancing disks on large mining pumps should regularly be repaired or replaced once wear has been detected [8]. Balancing disks leakoff flow could be as high as 12  $\ell/s$ . Assuming a head of 1000 metres and a replacement cost of R15 000, the replacement of a damaged balancing disk could be easily motivated with a payback of less than 230 hours [8].

### Leak repair

The water usage on a gold mine consists of production and non-production related usage. Due to the large amount of pressure and harsh working environment, non-production usage or leaks are common among the water reticulation systems of a deep level gold mine. Water wastage can be calculated with Equation 3.2 which is based on Bernoulli's theorem [18].

$$Q = \alpha A_t \sqrt{\frac{2(P_{inside} - P_{outside})}{\rho}} \quad (3.2)$$

where

$$Q = \text{Volumetric flow (m}^3/\text{s)}$$

$\alpha$  = Flow coefficient (-)

$P_{inside}$  = Pressure inside the pipe (Pa)

$P_{outside}$  = Pressure outside the pipe (Pa)

$\rho$  = Fluid density (kg/m<sup>3</sup>)

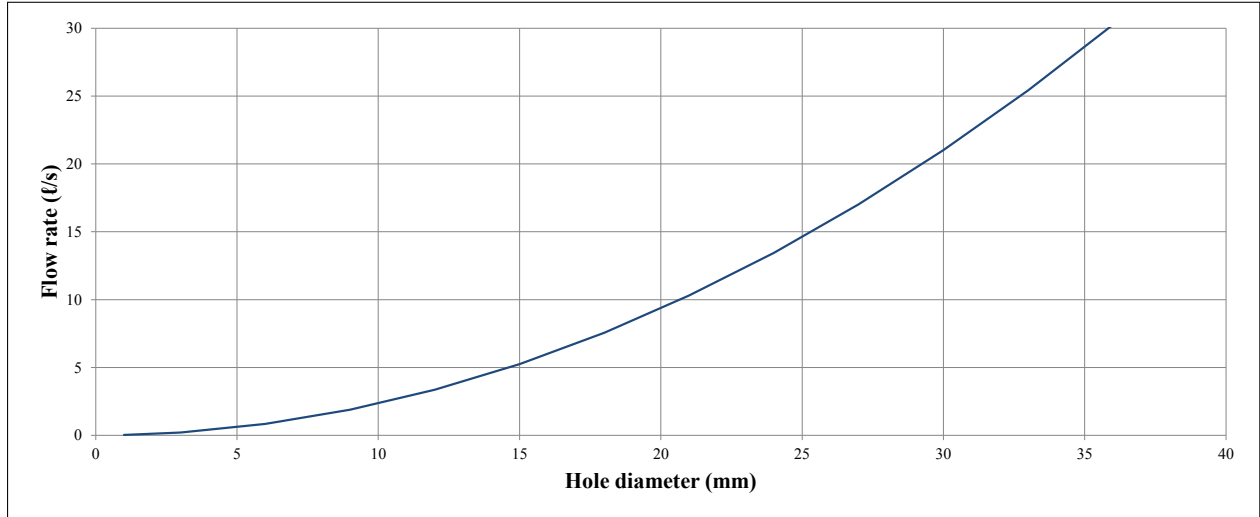


Figure 3.7: Relationship between the leak size and flow rate.

The relationship between leak size and flow rate is illustrated in Figure 3.7. To ensure that water is not wasted, the most efficient way would be to isolate the water supply line before it reaches the working areas. Manual isolation valves are used by mines, but pose the problem that the working areas are not always isolated by mine personnel after the work has been completed.

Automated isolation valves have been used to ensure automated stope isolation. The isolated valves can be controlled with a timer or by the centralised blasting system. Isolating the water according to the blasting schedule will ensure that no water will be required for production during the time of isolation [13].

Flow rate related to leaks can also be reduced by pressure control. A reduction in pressure will result in a reduction in flow (Equation 3.2). The relationship between flow rate and pressure is also illustrated in Figure 3.8 where the demand was assumed to be constant.

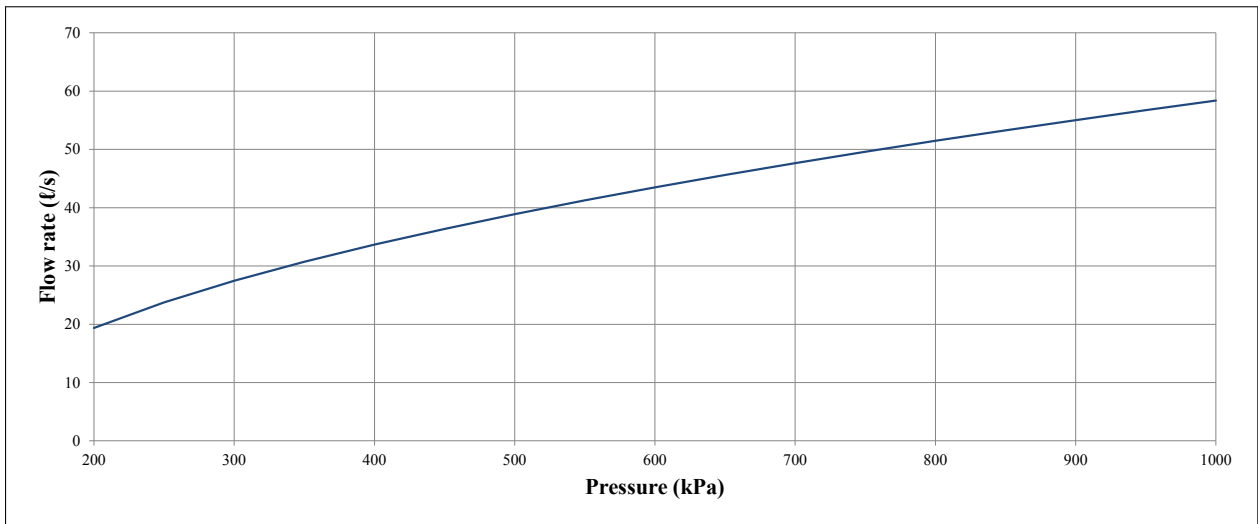


Figure 3.8: Relationship between pressure and flow rate according to Equation 3.2.

The repair of a leak can be motivated by assuming a cost of 55 c/kWh tariff and line pressure of 800 kPa. The daily cost of a leak can be calculated using Equations 3.1 and 3.2. The leak size in relationship to the cost to pump out the wasted water is illustrated in Figure 3.9 assuming a head of 1000 m. The actual cost can be assumed greater due to water being wasted and of a possible greater head.

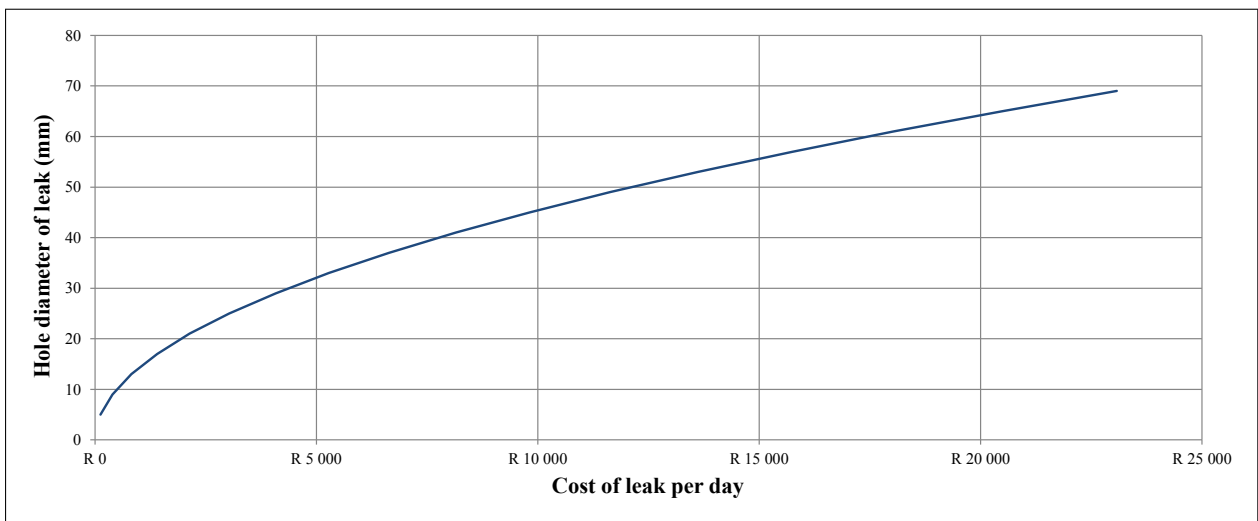


Figure 3.9: Relationship between leak diameter and daily cost to pump out the leaked water.

### Service water control

A practical method that has been verified to control the pressure is to reduce the flow by means of control valves. Figure 3.10 illustrates the actual measurement of the flow and pressure relationship for a valve control test performed at a mine. The test results are similar to that of Figure 3.8 but shows a greater reduction in flow relationship. This is assumed to be due to the flow reduction caused by the valve itself and not directly from the pressure

reduction. Each flow and pressure drop is different to the specific valve used and great care should be taken when implementing control valves on high pressure dewatering systems.

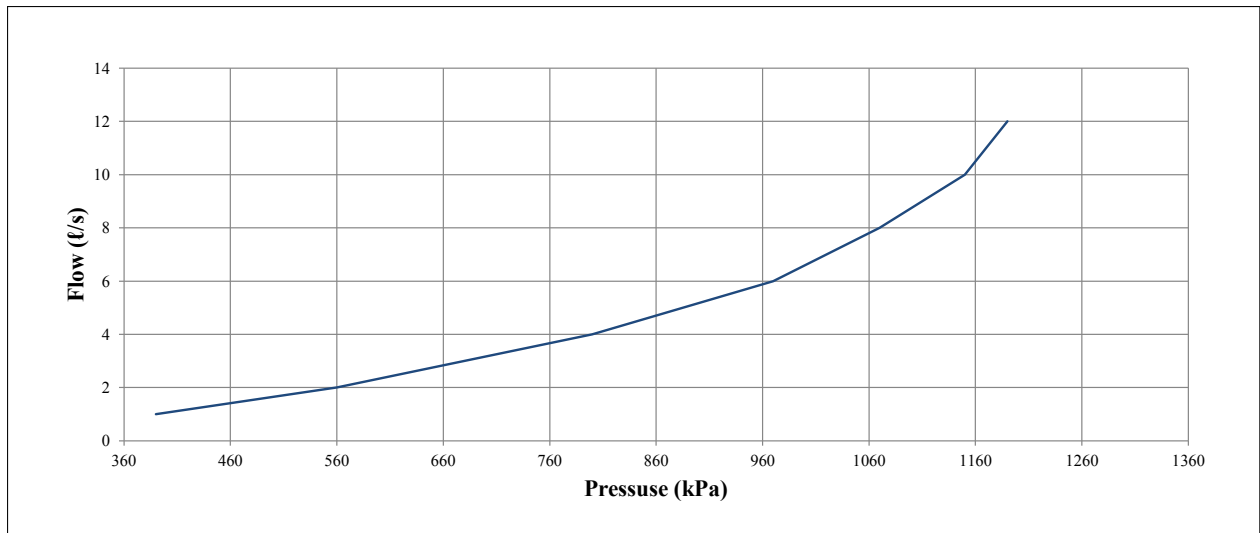


Figure 3.10: Measured relationship between pressure and flow rate for a specific valve [13].

Pressure Reducing Valves (PRVs) are used to control the pressure of the supplied water and control the flow supply to the working areas or stopes. Care should be taken not to reduce the flow below the minimum flow required for the cooling cars to provide the needed cooling in the selected mining areas.

The most efficient way to reduce pumping energy is to reduce the amount of water that needs to be pumped. As discussed this can be achieved by either reducing the pressure of the supplied water during non-drilling periods, by isolating supplied areas during non-drilling periods or by repairing leaks.

### Pumping system efficiency

Simplified methodologies have been derived, which can be used to estimate and identify possible electrical savings on a mine dewatering system [13], [8]. The efficiency of a dewatering system can be calculated with Equation 3.3.

$$\varepsilon = \frac{E_{\text{Theoretical}}}{E_{\text{Actual}}} \quad (3.3)$$

where

$\varepsilon$  = Pumping system efficiency (-)

$E_{\text{Theoretical}}$  = Calculated pumping energy required, excluding losses (kWh)

$E_{\text{Actual}}$  = The actual pump electrical energy consumed (kWh)

The actual electrical power of a pumping system can be calculated by using Equation 3.4 with the single phase current and line voltage readings supplied by measure equipment [19].

$$E_{\text{Actual}} = \sqrt{3}V_L I_L \cos \varphi \times 24 \quad (3.4)$$

where

$E_{\text{Actual}}$  = Actual power consumed by the pump system over a day (Wh)

$V_L$  = Line voltage (V)

$I_L$  = Line current (A)

$\cos \varphi$  = Load power factor (-)

The theoretical power consumption that is expected from the pumping system, related to the delivered flow and to the static head, can be calculated with Equation 3.5 (Derived from 3.1).

$$E_{\text{Theoretical}} = \frac{Q\rho gh}{3.6 \times 10^6} \quad (3.5)$$

where

$E_{\text{Theoretical}}$  = Theoretical power required by the pump system over a day (Wh)

$\rho$  = Density of the liquid (kg/m<sup>3</sup>)

$Q$  = Average flow rate (m<sup>3</sup>/h)

$g$  = Gravity acceleration constant (9.81 m/s<sup>2</sup>)

$h$  = Total head of the pumping station (m)

The actual and theoretical calculated results can be used to provide the system efficiency. The efficiency can then be used to provide a good estimation of what potential electrical savings can be obtained by reducing the water consumption or usage.

## Optimisation approach

In the section above, a simplified approach to evaluating the potential of optimising a dewatering system on a gold mine was developed. The simplified approach is illustrated in Figure 3.12. Using the equations in the simplified approach, a conservative theoretical saving of 4 MWh could be obtained by reducing the flow by 10% for the two four hour periods (mine head of 2000 m). Using an estimated system efficiency of 75% the electrical savings could be up to 3 MWh.

Several optimising strategies have been discussed where the supplied water can be controlled during certain periods of a day. A daily profile of a mine is illustrated in Figure 3.11. The consumption profile differs for certain times of the day and the drilling and blasting shift can clearly be seen in the water consumption profile.

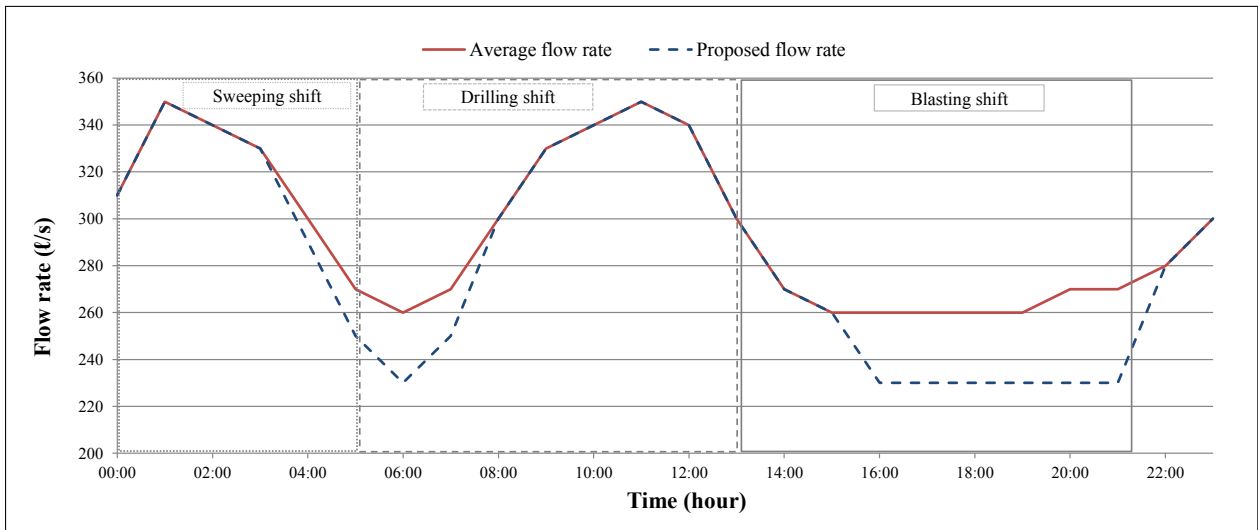


Figure 3.11: Daily flow profile of a deep level gold mine of the selected mining group.

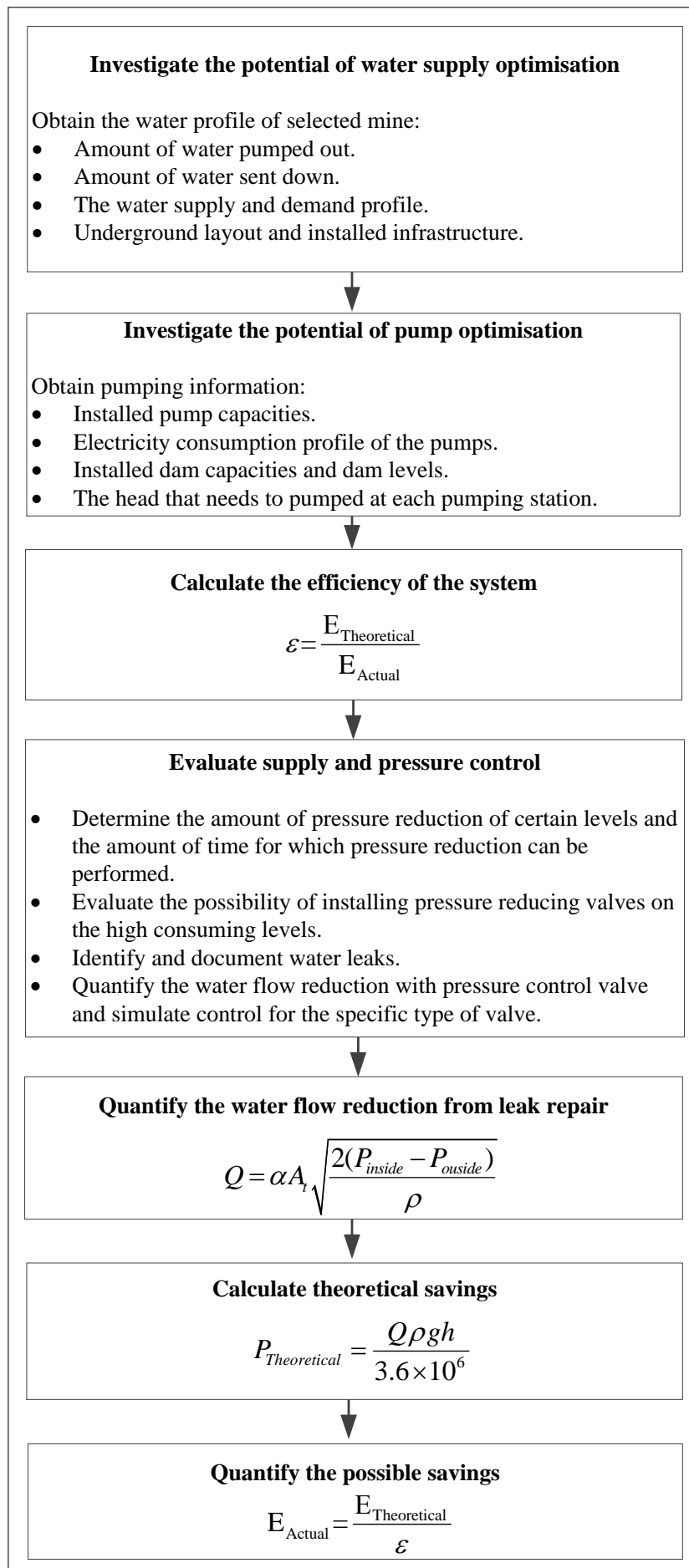


Figure 3.12: Simplified approach taken to identify the potential of a dewatering system of a gold mine.

## Load shift potential

As discussed in Chapter 2, the Megaflex TOU structure imposed by Eskom for gold mines poses a financial risk. This can be noted in the Megaflex TOU structures (Appendix C) where the high demand charge in peak season is seven times more than that of the off-peak period.

To illustrate this cost risk involved, a 1 MW constant load cost was compared to that of an optimal electrical load shift profile using the same total amount of electricity for the day. This example will also be used to illustrate the financial benefits of a load shift DSM project. Several studies have proven the successful implementation of DSM load-shifting projects on South African mine dewatering systems and mines [20], [21].

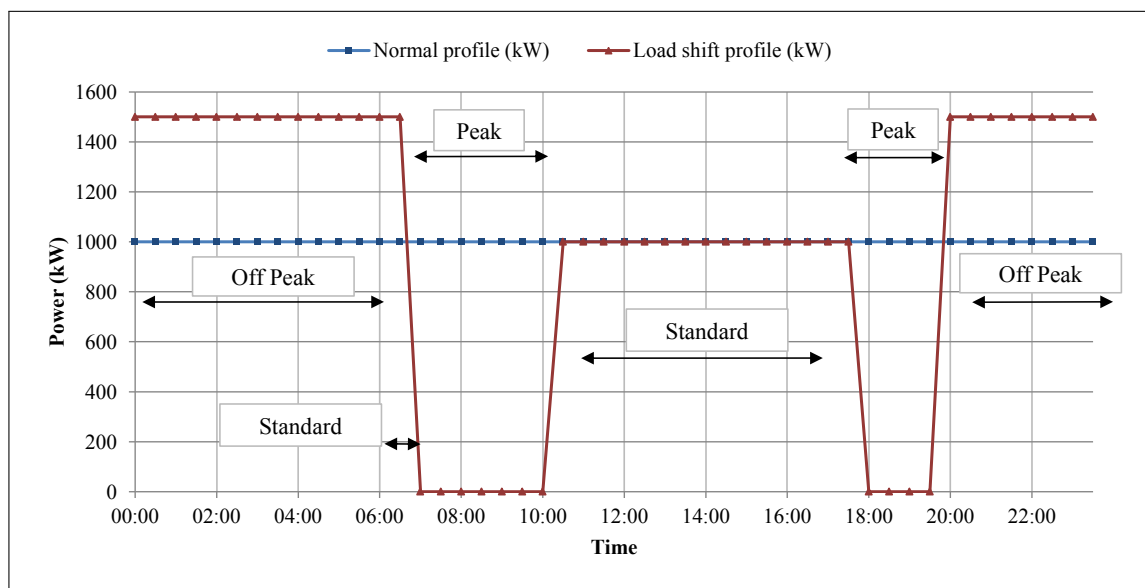


Figure 3.13: Constant 1 MW load with an optimal load shift profile comparison.

The electricity profile of the 1 MW constant load is illustrated in Figure 3.13. The different tariff structures related to the time of day are also shown with the optimal load being zero in the peak periods.

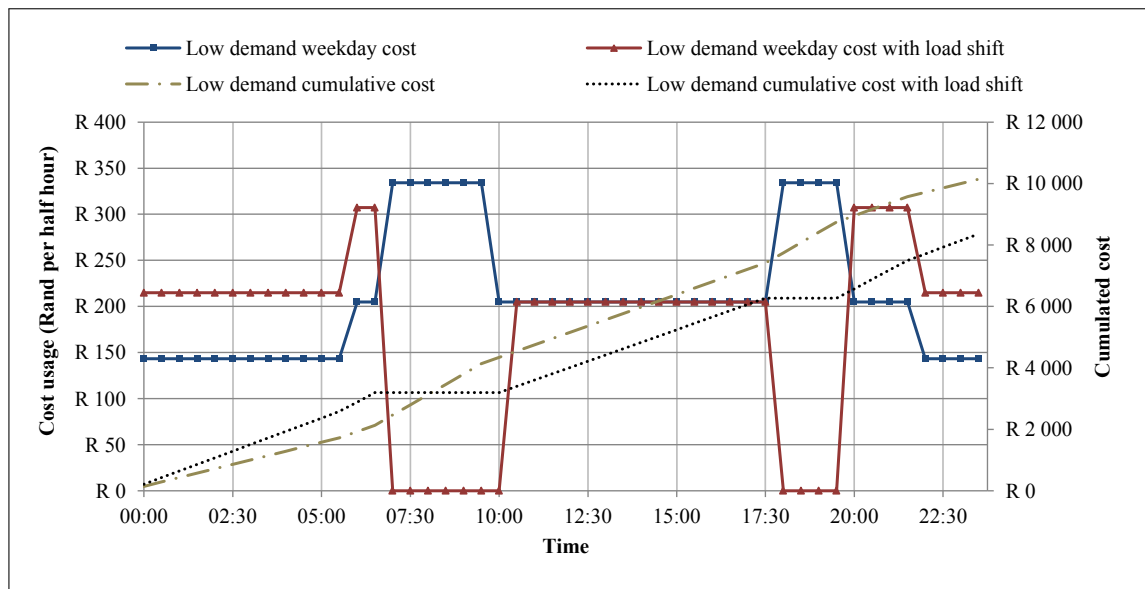


Figure 3.14: Load shift cost illustration of a 1 MW load in the low demand season.

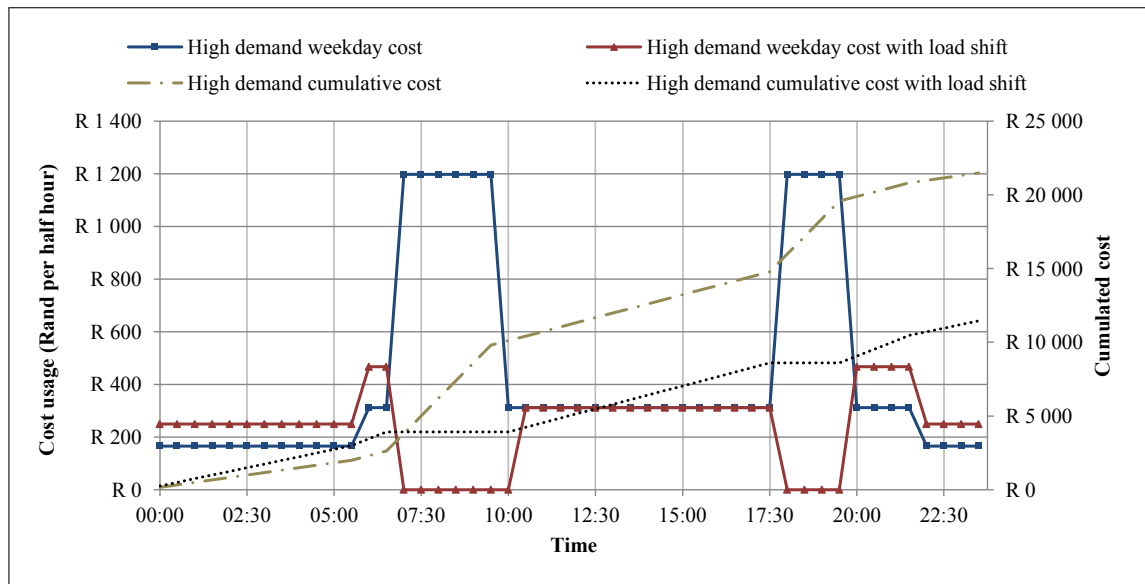


Figure 3.15: Load shift cost illustration of a 1 MW load in the high demand season.

The cost relationship for the constant and shifted 1 MW load is illustrated in Figure 3.14 for low demand season and Figure 3.15 for the high demand season.

Table 3.6 illustrates the percentage cost reduction from shifting electrical load from peak periods to non-peak periods. An estimated saving of 18% can be achieved in electrical costs for the low demand seasons and 47% in the high demand seasons.

Power profile (1 MW)	Total daily costs		Total monthly costs		Annual costs
	Low demand	High demand	Low demand	High demand	
No load shift	R 10 143	R 21 481	R 202 856	R 429 626	R 3 114 582
Load shift	R 8 356	R 11 460	R 167 112	R 229 200	R 2 191 608
% Cost savings	18%	47%	18%	47%	30%

Megaflex 2012/2013 tariffs	Low demand season (R/kWh)	High demand season (R/kWh)	% Increase
Peak	0.67	2.39	258%
Standard	0.41	0.62	52%
Off peak	0.29	0.33	16%

Table 3.6: Cost comparison of 1 MW load shift and constant profile.

From the load shift scenarios illustrated above, the relationship for performing electrical load shift and the typical price reduction that could be expected was derived. This relationship between price reduction and load shift is illustrated in Figure 3.16. The relationship was calculated and the assumption is that day shifts remain energy neutral and that the shifted load is evenly distributed in the morning before peak period and in the late evening after the peak period. This resembles the typical load shift schedule of a mine dewatering system. The electrical load shift would also be performed in the morning as well as the evening peak period.

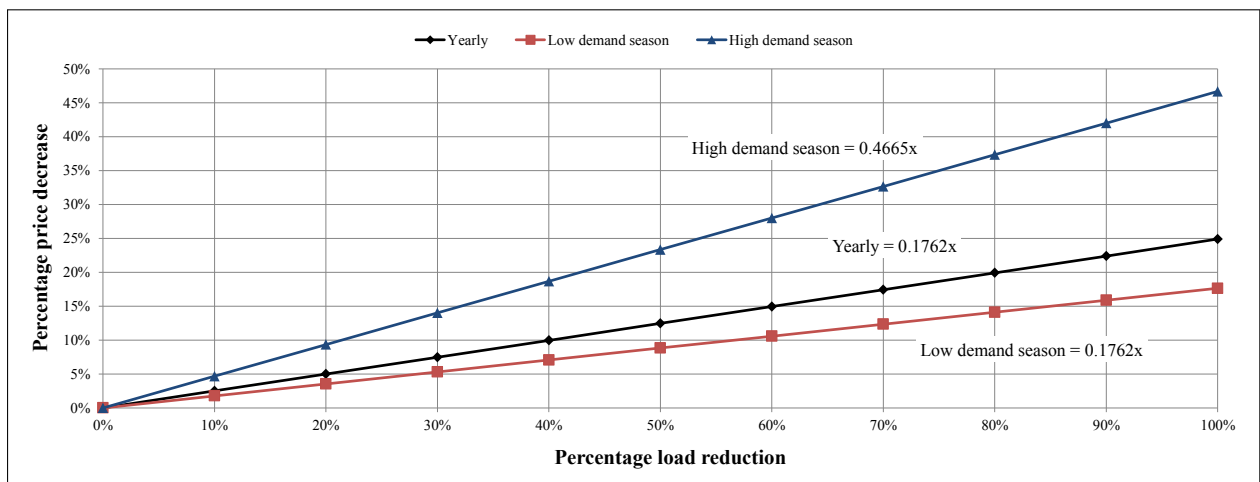


Figure 3.16: Percentage load reduction in relation to price decrease.

Control parameters that must be considered when implementing a pump electrical load shift project include:

- Sizes and number of available pumping columns.
- Storage dam capacities.

- Pump ratings and availabilities.
- Dam control levels and storage capabilities.
- Water inlet and outlet flow to dam.

The dam capacity can be modelled with Equation 3.6, where the outflow rate is related to pump power and the inflow rate can be calculated by the dam level change rate [22].

$$Contents = \sum \int InflowRate - \sum \int OutflowRate + InitialContents \quad (3.6)$$

From the calculations, it is clear that there is a financial benefit from performing load shift or consuming less electricity in the peak periods. Load management has been applied on several main services of South African industries resulting in electricity cost savings. These strategies, however, do not always reduce electricity usage overall, and cost risks such as the electricity price increase and carbon tax still remains a risk.

### 3.3.1 Case study: Pumping

#### Background and layout

Mine E is an ultra deep mine and has one of the largest installed pumping capacities of the mining group. Mining tasks are operated at a depth of 3.6 km and with a total installed pump capacity of 27 MW. The simplified layout of the dewatering system of Mine E is illustrated in Figure 3.17 and can be described by the following numbered items:

1. Chilled water is gravity-fed from the surface cooling system to the underground levels. Due to the pressure build-up, the water is passed through energy dissipaters or water turbines. The water turbines supply the generated electricity to the mine's network with an installed capacity of 2 MW.
2. Water is recirculated between the lower mining levels (level 102-113) and level 71. The water is cooled and then sent back, to ensure that minimal water is pumped out from the shaft bottom. Pumping 1 Mℓ of water throughout the day from shaft bottom to the surface takes 9.2 MWh of power theoretically. Almost half of the required power can be saved by circulating the water underground.
3. The supplied flow can be divided into two levels of supply, namely lower deepening (level 88-98) and upper production (level 71-88). A total of 24 Mℓ is supplied underground with an estimated fissure total of 2 Mℓ per day.

4. The pressure supplied to the deepening levels are lower to that of the upper production levels. The pressure is regulated by the PRVs on each system and the related supply pressure and line pressure after the PRV is shown in Table 3.8. The installed PRV does not provide the capability of controlling the pressure dynamically throughout the day.
5. Water is pumped between the different pump stations depending on the dam levels. The maximum controlled dam level is 76% and the installed pumping capacity of each pumping station and level is listed in Table 3.7.

The efficiency of the system with the total water pumped out per day is 0.75 calculated using Equation 3.3. The measured pump electricity consumption baseline is illustrated in Figure 3.18. The pump baselines illustrates two periods of pumping power reduction. This is due to a pump load shift DSM project previously implemented.

Pump No	29 Level		52 Level		75 Level		100 Level		115 Level	
	Installed capacity (kW)	Rated flow (ℓ/s)	Installed capacity (kW)	Rated flow (ℓ/s)	Installed capacity (kW)	Rated flow (ℓ/s)	Installed capacity (kW)	Rated flow (ℓ/s)	Installed capacity (kW)	Rated flow (ℓ/s)
Pump 1	1 200	150	1 200	150	1 275	150	1 600	240	2 500	265
Pump 2	1 200	150	1 200	150	1 275	150	-	-	2 500	265
Pump 3	1 200	150	1 200	150	1 115	150	1 400	240	2 500	265
Pump 4	1 200	150	1 200	150	1 100	150	1 600	240	2 500	265
Pump 5	1 200	150	1 200	150	1 115	150	1 600	240	-	-
Pump 6	-	-	-	-	1 115	150	-	-	-	-

Table 3.7: Installed pump capacity of the underground pumping stations of Mine E.

	Level	Supply pressure in main line (kPa)	PRV reduced pressure (kPa)
Upper production levels	88	5000	1000
	92	6000	1000
	65	7000	1000
	98	8000	1000
Lower reduction levels	102	2000	1200
	105	3000	1200
	109	4000	1200
	1130	5000	1200

Table 3.8: Supply line pressure and the line pressure after the PRV on the mining levels of Mine E.

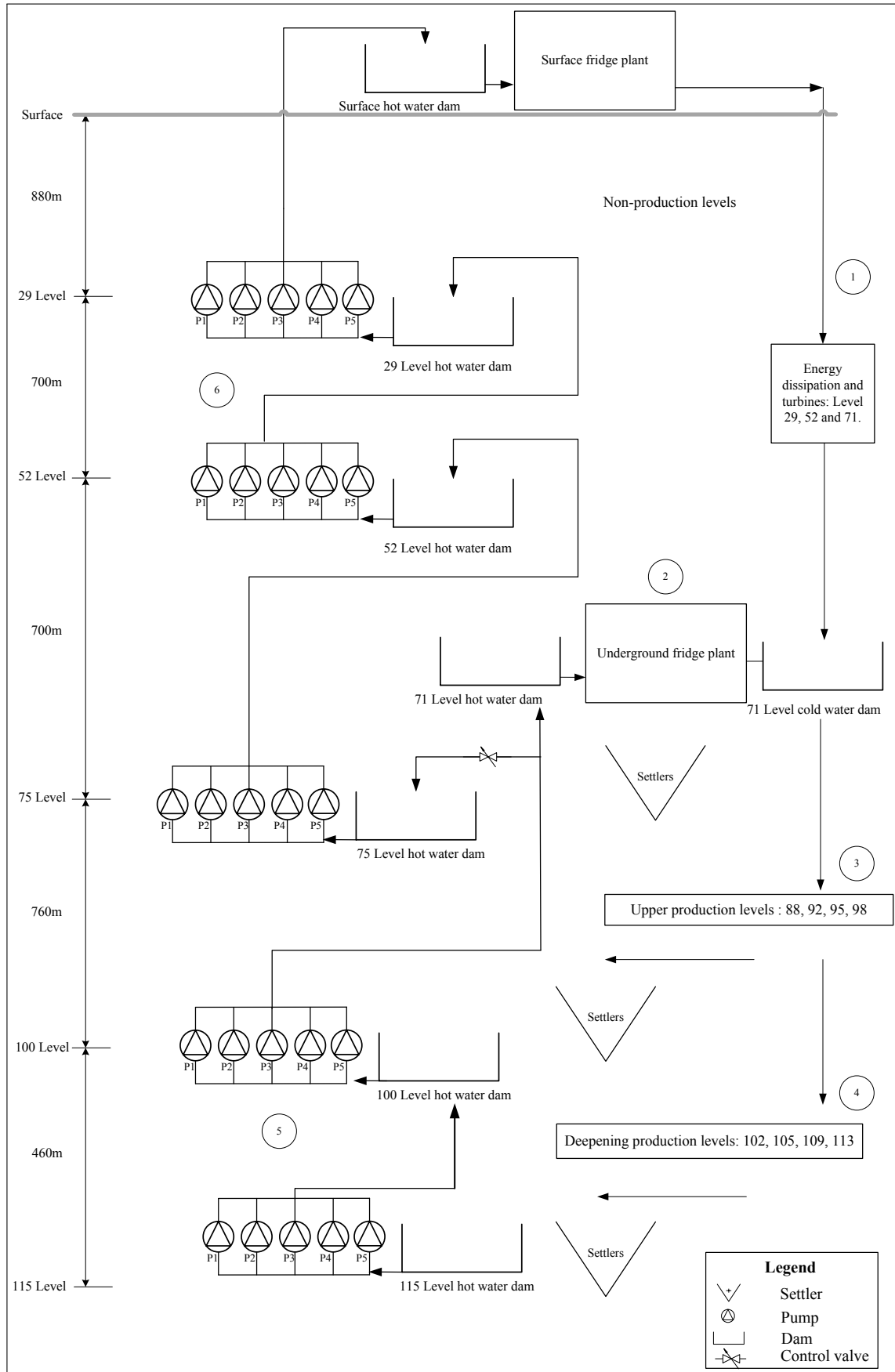


Figure 3.17: Simplified layout of the dewatering system of Mine E.

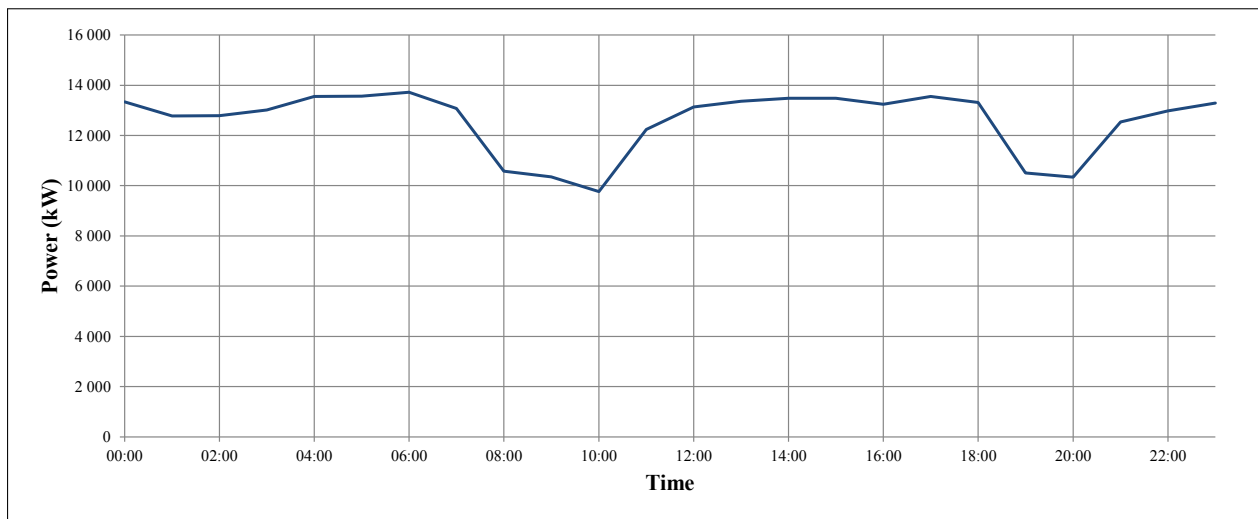


Figure 3.18: Electrical pump baseline of Mine E.

### Proposed control philosophies and installations

The supplied water will be reduced by 350 kPa but still providing a sufficient flow of 8  $\ell/s$  for the Bulk Air Coolers (BACs). The goal will be to reduce the non-productive consumption by reducing the supplied pressure during non-drilling periods resulting in a lower supplied flow. From the calculated efficiency and the measured electricity consumption baseline, the potential electrical saving was calculated. By controlling the supplied water above the minimum flow of 8  $\ell/s$  during non-drilling periods, a total of 1.3 M $\ell$  can be saved, resulting in an estimated electrical saving of 550 kW.

Stope isolation valves will be installed on the main production sections. The 30 installed valves are closed by a centralised blasting system as the section is cleared and made safe for blasting. By installing stope isolation valves and assuming 2  $\ell/s$  is supplied to each stope, isolating of 30% of the valves at 14:00 and the next 30% every hour, a total of 0.8 M $\ell$  can be saved daily, resulting in a electrical saving of 6480 kWh.

On the remaining levels, as indicated in Figure 3.19, the flow could be reduced by means of a proposed bypass control valve and isolation valve system. The isolation valve is a butterfly valve where the control valve is globe valve, providing better control and protection against cavitation. Other benefits include that the butterfly isolation valve can be used to isolate a section and the bypass section used to protect the water supply line against water hammer.

Due to the high cost related to the valves and installation thereof, as well as the high pressure columns on the deepening mine levels, the control valves were not installed with the other valves installations. The control philosophy implemented on the pressure reducing valves is illustrated in Table 3.9.

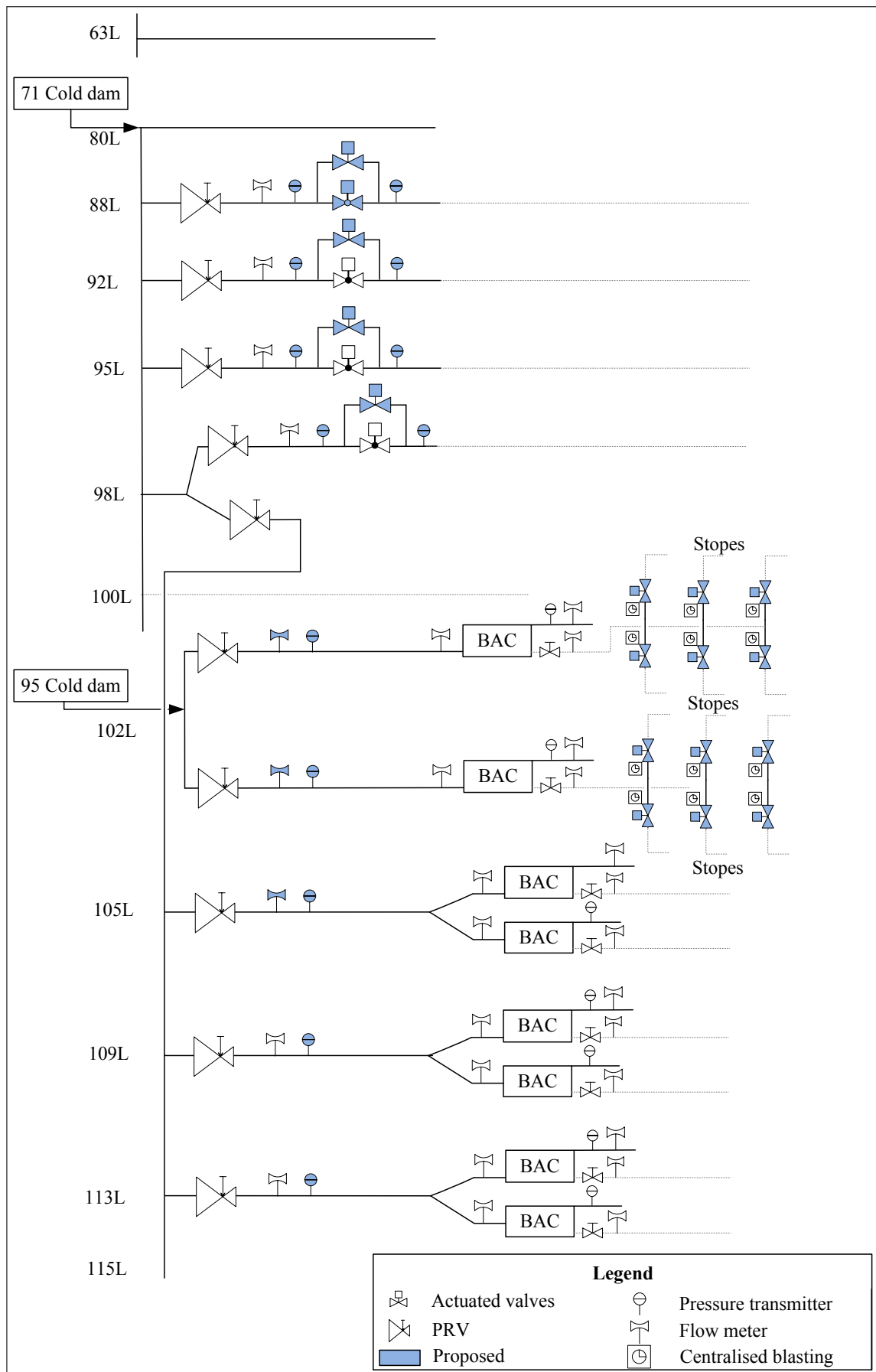


Figure 3.19: Proposed installation of control valves and instrumentation.

Time	Mining activity	Isolation valve (% Open)	Control valve (% Open)	Downstream pressure (kPa)
00:00-16:00	Drilling and Sweeping	100%	100%	1 000
16:00-22:00	Blasting	0%	Control enabled	650
22:00-00:00	Sweeping	100%	100%	1 000

Table 3.9: Pressure control ranges and time for the control valves.

## Implementation and results

The isolation and control valves were installed, and in parallel with the installation, leak investigation and leak repair were performed. More than 10 leaks were identified with an average diameter of 10 mm resulting in a total water reduction of 7 Mℓ per day. The original water consumption profile with the controls implemented and the measured water profile are illustrated in Figure 3.20.

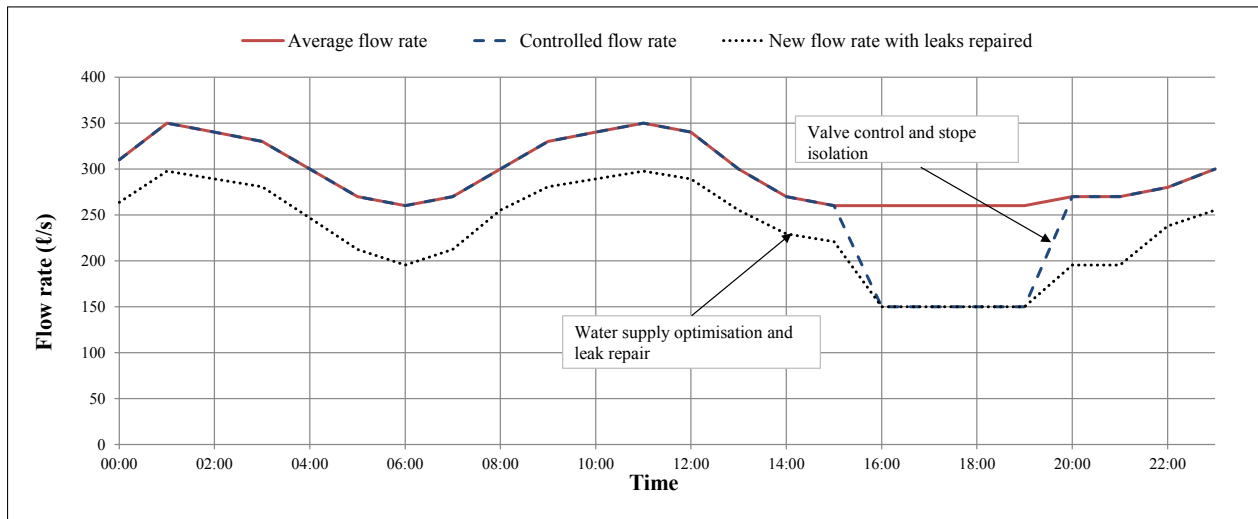


Figure 3.20: Controlled water flow and baseline flow of Mine E.

The combined efforts of supply optimisation and leak reduction resulted in daily demand reduction of 3.83 MW. The estimated savings calculated using Equation 3.3 in relation to daily demand reduction was 3.18 MW. The 17% error in calculation is expected to be due to:

- The water loss due to leaks was much greater than calculated from the estimated leak sizes.
- The flow reduction from the stope isolation valves worked better than expected.
- The reduction in water resulted in the optimal pump schedule, improving the efficiency of the pumps station on each level.
- Less fissure water was pumped out during performance assessment of the project.

## Further expansions on mines

The proven approach was used to quantify and identify possible electricity cost savings projects on the dewatering systems of the selected mines. The load shift potential and energy efficiency project were identified and the potential projects listed in Table 3.10. The identified projects are projects that have been thoroughly investigated and the proposed scope has been reviewed by mine personnel. Potential project savings are estimated purely on available data received from the mine and quantified by the model. The targets of potential projects listed have been conservatively estimated with a 30% safety margin.

<b>Pumping</b>			
<b>Mine</b>	<b>Target (MW)</b>	<b>Mechanism*</b>	<b>Status</b>
Mine F	2.80	LS	Identified
Mine H	3.30	EE	Identified
Mine A	3.90	LS	Implemented
Mine B	5.80	LS	Implemented
Mine C	4.00	LS	Implemented
Mine C	2.40	LS	Implemented
Mine E	1.60	EE	Implemented
Mine E	3.00	LS	Implemented
Mine G	2.35	LS	Implemented
Mine H	3.10	LS	Implemented
Mine D	0.30	EE	Potential
Mine A	0.70	EE	Potential
Mine C	0.45	EE	Potential
Mine F	0.40	EE	Potential
Mine G	0.40	EE	Potential
Mine B	0.80	EE	Potential
Total implemented peak load reduction (MW):			24.55
Total future peak load reduction (MW):			27.35
Total implemented pumping profile reduction (MW):			1.60
Total future pumping profile reduction (MW):			6.35
* <b>LS</b> : Load shift, <b>EE</b> : Electricity Efficiency			

Table 3.10: Identified electricity cost savings on the dewatering service of the selected mines.

From the presented data in Table 3.10, it can be seen that mostly load shift projects have been implemented on the dewatering systems of the selected mines. It is mostly due to the large load potential which results in larger DSM funding for infrastructure. The required infrastructure and engineering cost related to the load shift projects of less than 1 MW are difficult to motivate to the client, as funding loads of less than 1 MW are not included in the ESCo funding model. Projects producing less than 1 MW could be combined into performance contracting, but pose the risks of managing several projects to obtain one larger saving.

The potential savings related to the pumping electricity usage relates to 51 GWh efficiency and 27 MW load shift. The electricity efficiency projects relate to an additional 12% reduction of the average yearly pumping electricity consumed by the selected mines.

### 3.4 Compressed air

The main use of compressed air in gold mines is for drilling and loading. Compressed air is also sometimes used for ventilation in emergencies, providing a fresh air supply in refuge chambers or through open ended pipes. The relationship between electrical power and compressed air mass flow delivery can be calculated with Equation 3.7 [23].

$$P_{electrical} = \frac{\dot{m}_{air} w_{comp,in}}{\eta_{motor}} \quad (3.7)$$

where

$P_{electrical}$  = Electrical power (kW)

$\dot{m}_{air}$  = Compressed air mass flow rate (kg/s)

$w_{comp,in}$  = Energy required to compress a unit mass of air (kJ/kg)

$\eta_{motor}$  = Efficiency of the electrical motor (-)

From Equation 3.7 the electrical power consumption is directly related to the compressed air mass flow rate. Decreasing the required compressed air mass flow will result in lower electrical energy. Likewise, improving the compressor motor efficiency will reduce the required electrical energy. The energy required to compress a unit mass of air can be calculated with Equation 3.8.

$$w_{comp,in} = \frac{nRT_{inlet}}{\eta_{comp}(n-1)} \left[ \left( \frac{p_2}{p_1} \right)^{(n-1)/n} - 1 \right] \quad (3.8)$$

where

$w_{comp,in}$  = Energy required per unit mass of air (kJ/kg)

$n$  = Polytropic compression exponent (-)

$R$  = Gas constant (287 J/kg.K)

$T_{inlet}$  = Line temperature (K)

$\eta_{comp}$  = Compressor efficiency (-)

$p_2$  = Compressor discharge pressure (absolute pressure) (kPa)

$p_1$  = Compressor inlet pressure (kPa)

With the compressed air mass flow rate provided by:

$$\dot{m}_{air} = C_{discharge} \left( \frac{2}{k+1} \right)^{\frac{1}{k-1}} \frac{p_{line}}{RT_{line}} A \sqrt{kR * 1000 \left( \frac{2}{k+1} \right) T_{line}} \quad (3.9)$$

where

$\dot{m}_{air}$  = Compressed air mass flow rate (kg/s)

$C_{discharge}$  = Discharge coefficient (-)

$k$  = Specific heat ratio (-)

$p_{line}$  = Line pressure (kPa)

$R$  = Gas constant (287 J/kg.K)

$T_{line}$  = Line temperature (K)

$A$  = Minimum cross-sectional area (m<sup>2</sup>)

The polytropic compression exponent varies between 1 and 1.4 for intercooled compressors. The most common type of compressor used with deep level mines are intercooled centrifugal compressors. For improved energy efficiency, the aim would be to get the polytropic compression close to one. While the the polytropic compression is determined by the design [23], proper maintenance also influences the operation and polytropic compression exponent.

Although the efficiency of the machine is also determined by the design, this electricity savings approach for compressors will not focus on changing the design of the compressor but rather demand-side improvement, that can still be performed with an optimal machine.

### 3.4.1 Compressed air optimisation

Some initiatives for optimising compressed air electricity consumption include the following:

**Make use of cooler intake air:** Adjusting the intake manifold to use cooler air rather than hot air from other areas results in better compression and electricity savings. The results of these savings produce payback periods of between five months and two years [24].

**Install and/or upgrade compressor controls:** By upgrading the compressor controls, savings of between 0.8% and 10% can be achieved on typical industrial systems, resulting in payback periods of ten months [24].

It is evident from Equations 3.7 and 3.8 that large electricity savings are possible by operating the compressor at a low discharge pressure set point. This can be achieved with guide vane controls or by switching the compressor on and off, for a compressed air network with multiple compressors [25]. It is also evident that the delivery pressure must be as low as possible to prevent unnecessary over supply or consumption of air. The relationship between system pressure and compressor power consumption is illustrated in Figure 3.21.

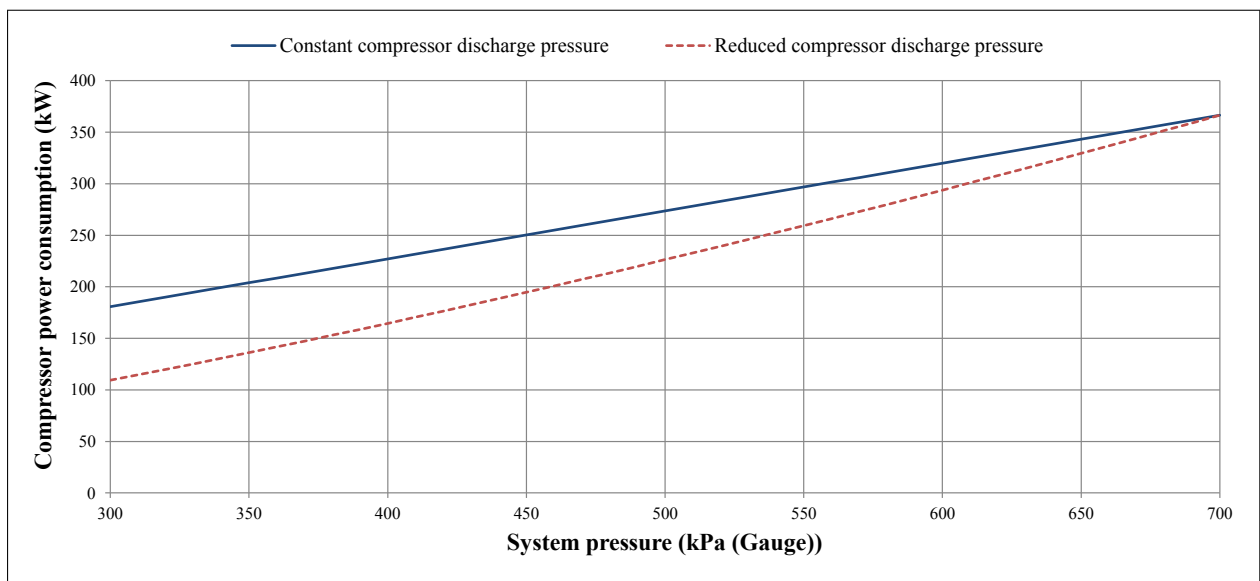


Figure 3.21: Relationship between system pressure and compressor power consumption [25].

Both the scenarios illustrated in Figure 3.21 indicate the compressor power for a fixed leak or supply size. The difference in compressor power consumption can clearly be noted, where reduced system pressure is obtained through control valves. The lower power consumption profile is obtained by varying the discharge pressure between 700 kPa and 300 kPa according to the reduced system pressure.

From the analysis it was derived that a general rule of thumb can be used to estimate the compressor savings resulting from reduced pressure. The rule of thumb states that a 14 kPa reduction will result in a 1% reduction in compressor power [25]; this only applies to pressures ranging between 700 kPa and 300 kPa.

**Reduce pressure:** It has been reported that pressure reducing projects resulted in energy reductions of between 0.5% and 1% with project paybacks of four months. There are two main strategies for reducing pressure on gold mines: either by implementing surface- or underground pressure control.

The underground and surface pressure of a gold mine is controlled according to the different mining shifts and the equipment with the highest pressure requirement. For a gold mine, the required operating pressure for the plants will determine the control pressure set point [26] [27]. This is due to the the required operating pressure of the shaft being lower during non-drilling periods or mining shifts. Equation 3.10 was derived for surface pressure control and electrical compressor consumption [25].

$$P_{electrical} = F_{line} \cdot p_{line} \quad (3.10)$$

where

$P_{electrical}$  = Electrical power (kW)

$F_{line}$  = Power to line pressure ratio (kW/kPa)

$p_{line}$  = Line pressure (kPa)

From Equation 3.10 it is evident that the same rule of thumb can be applied in calculating the expected compressed air power reduction in relation to the surface line pressure reduction. Reduction in pressure results from the reduction in mass flow from valves reducing the amount of air flow to the production elements or leaks.

It was also proven that the same rule of thumb can be applied to underground valve controls [25]. For underground control, the minimum pressure will be determined by the loading boxes where the typical pressure required by loading boxes is 450 kPa. Other influences such as auto compression as well as ring feeds must be taken into consideration with underground pressure control.

**Eliminate or reduce components dependent on compressed air:** Compressed air tools and equipment can sometimes be replaced with electric equipment. Case studies show that the replacement of this equipment resulted in a saving of more than 0.5% with an average project payback of six months [24].

In 1996 AngloGold Ashanti prompted feasibility studies to identify the possibility of using electric drills rather than pneumatic drills. Tau Tona Mine was chosen as a case study due to low operating pressures at certain areas of the mine. The manufacturer claimed an estimated 38% reduction in operating costs from using electric drills rather than pneumatic drills. In 2010 the prototype was rolled out for testing but was declared inadequate for underground mining [28].

Cost comparisons in 2003 shows that the electric drills manufactured by Hilti was on average 5.5 times more costly to operate than traditional pneumatic drills [28]. A large part of the

costs involved with implementing electric drills consists of the capital required for installing electrical reticulation. The capital investment and running overheads for the electric drill and pneumatic drills were then compared. It was found that the operating cost of the electric system from the second year onwards would be R21.5-million more than the pneumatic system [29].

Other pneumatic equipment, such as loaders used to clear blasted rock, requires compressed air. Alternatives to the loaders have been developed which are hydraulic loaders; they cost an estimated 40% more than the standard pneumatic loaders [30]. The replacement of pneumatic loaders with electric loaders has a payback period of two years [25].

**Repair air leaks:** Leaks on compressed air system can vary greatly from small leaks (0.5% usage) to large leaks resulting in losses of up to 30%. Repairing leaks is always seen as a good investment, repairing small leaks with 0.5% savings could produce paybacks in only three months [24].

On average it was found that leaks waste between 10% and 30% of the total supplied compressed air [31]. The effect of a leak hole with a 10 mm diameter is estimated to result in the loss of 14 kW electrical power, making up less than 0.1% of a typical large gold mine compressed air system. It is suggested that repairing a small leak would be uneconomical due to loss in production from isolating the line. Rather, it must rather be reported but immediate focus must be placed on finding large leaks.

From Equation 3.8 and 3.9 the compressed air electricity usage related to a leak size can be calculated. The relationship between leak size and compressed air electricity wastage is illustrated in Figure 3.22. The following parameter values are assumed:

- Gauge pressure at 500 kPa.
- Atmospheric pressure at 87 kPa.
- Air temperature at 25 °C.
- Line temperature at 28 °C.
- A compressor efficiency of 80%.
- Coefficient of leak taken as 0.65 [23].
- Motor efficiency taken as 0.98 (synchronous motors).
- Specific heat ratio 1.4 (air).
- Polytropic constant taken as 1.4.

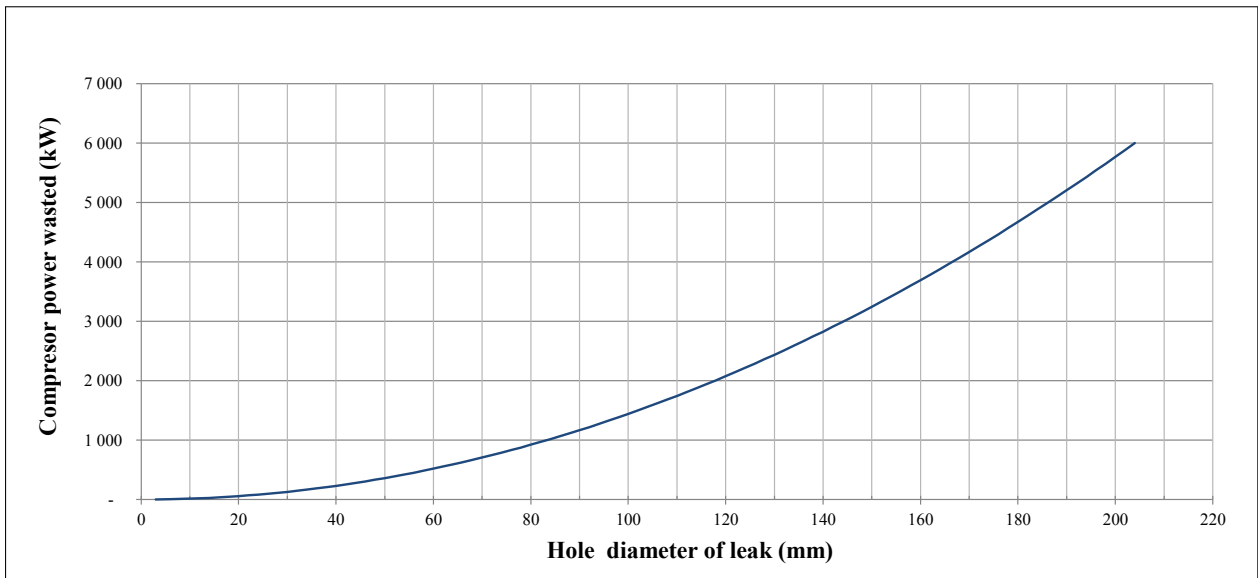


Figure 3.22: Compressed air electricity wastage related to leak size.

To motivate the need to repair leak sizes, the electricity cost occurred from leaks is illustrated in Figure 3.23, assuming an average electricity cost of 0.55 R/kWh.

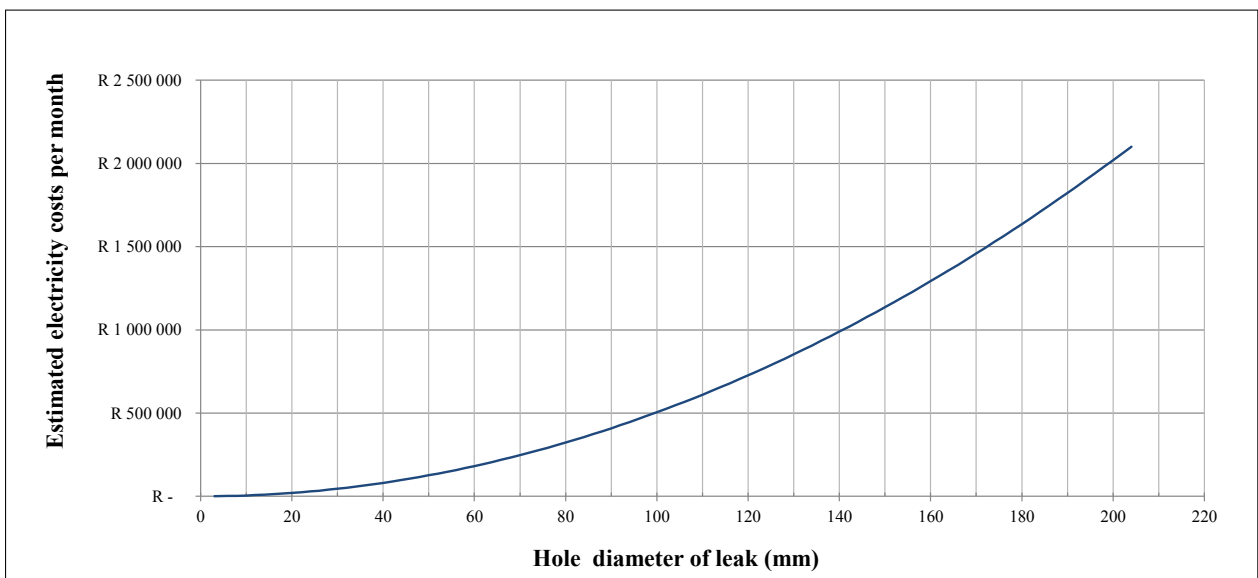


Figure 3.23: Compressed air electricity cost related to leak size.

**Recover wasted heat:** Waste heat from compressors can be applied to other processes producing savings of 2% with paybacks of ten months [24].

**Maintain filters and coolers:** Compressed air travelling through the machine is greatly affected by dirty filters, resulting in poor efficiency. By performing regular and proper maintenance, efficient operation of the compressor will be maintained [24].

**Implement a combination of the abovementioned strategies:** It would be beneficial to the complete operation of the machine if all of the abovementioned methods are applied.

Results of up to 50% reductions have been claimed by implementing a combination of the strategies [24].

Studies revealed that some mines only implemented certain strategies on a mine as illustrated in Table 3.11.

Implemented: ✓ Limited: +	Energy efficiency measures						
	Efficient compressor selection	Compressor control	Surface distribution control	Underground distribution control	Replace pneumatic equipment	Reduced system pressure	Leak management
Mine 1					+		
Mine 2		✓				✓	
Mine 3			✓	+			
Mine 4			✓				
Mine 5		✓	✓				
Mine 6			✓		+		
Mine 7			✓	✓			
Mine 8		✓		✓			
Mine 9							
Mine 10						✓	
Mine 11						+	+

Table 3.11: Illustration of implemented compressed air mitigation strategies on South African mines [25].

Some of the abovementioned initiatives provide better advantages than others, resulting in better savings results. The complexity of the mine compressed air systems could lead to the misinterpretation of possible energy savings initiatives [25]. To fully understand a compressed air system the following information must be obtained:

- The design supply characteristics and values for the installed compressors. This will include information such as supply flow, pressure and the installed power rating of motor.
- The compressor control and capacity range, by identifying the electricity usage of the compressor related to the compressed air supply. This information will be provided either from the SCADA system or from flow meters and power loggers.
- The power consumption profile and related pressure profile. This data can also be obtained from the SCADA or log books with system pressure and compressor statuses. The data will show the availability the compressor, the frequency of stopping and starting the compressors, and the level of automation will also be indicated.

A simplified approach was developed to illustrate the effect of the different techniques used to improve compressed air electricity usage [25]. From the study, it was found that a simplified approach can be used to predict energy savings opportunities rather than using available simulation software. The advantages of the simplified approach include:

- Less required information such as pipe diameters, flow and pressure for the complete layout of compressed air system.
- The approach starts with the mitigation strategies providing the largest savings.
- The required data can be logged with portable monitoring equipment, if not available on the SCADA.
- Simplified estimation formulas were derived to provide a good estimate of the possible electricity savings with the related mitigation strategies.

Savings potential on a compressed air mining system can be identified with limited resources. The simplified approach is more cost effective as well as time effective.

From these studies discussed, a simplified approach was derived from Marais [25] as a starting point to identify possible savings with minimal information and to quantify the savings with related mitigation strategies. The compressed air energy savings strategy is summarised in Figure 3.24.

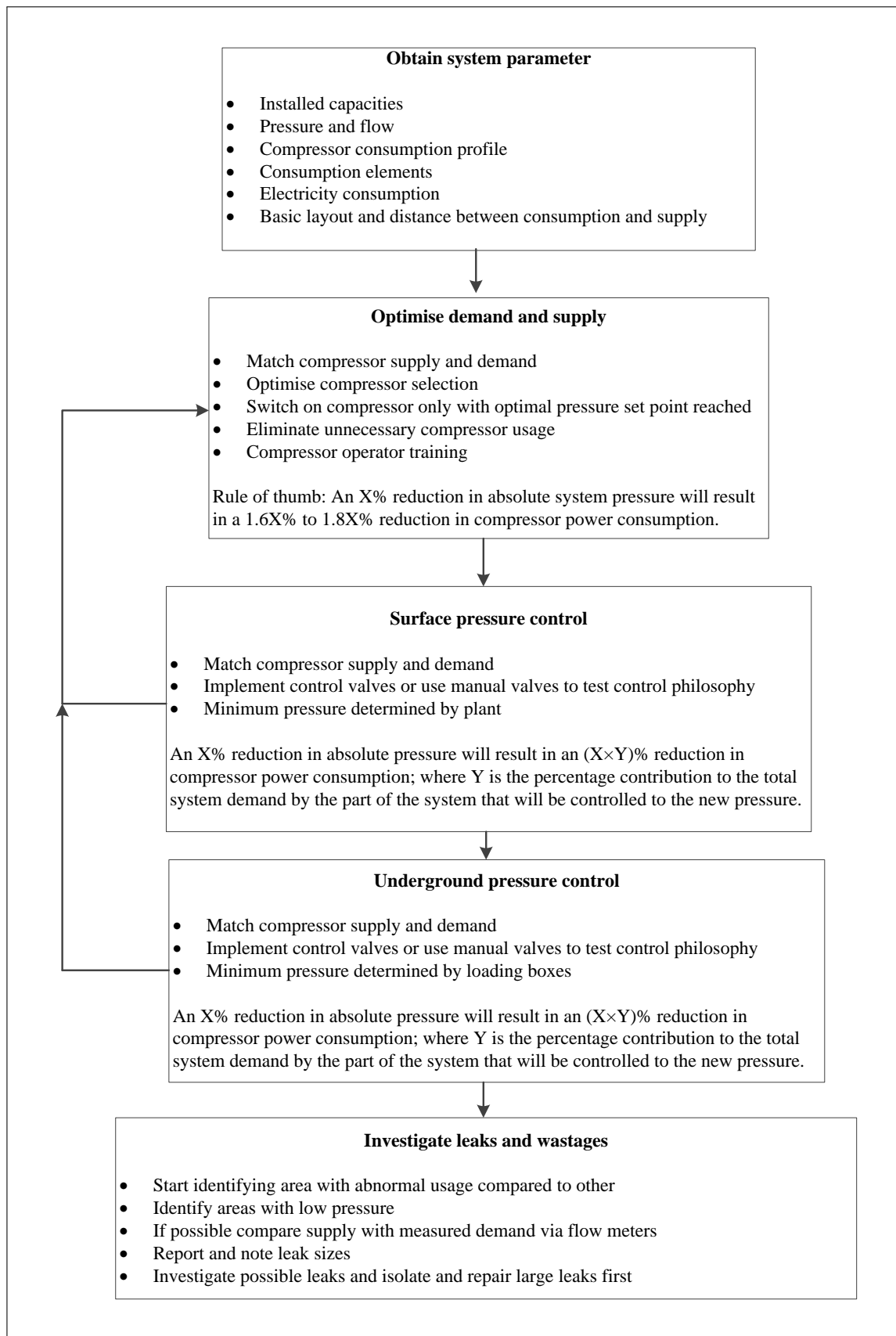


Figure 3.24: Compressed air savings strategy.

The derived compressed air benchmarks are illustrated in Table 3.12 and Figure 3.25. From Table 3.12 it should be noted that Mine G's compressed air consumption of the total

consumed electricity is very low compared to other mines, where Mine D's is the highest. Mine G's low compressed air consumption is due having mechanised drilling operations. In terms of electricity usage per month, Mine A and Mine E are the highest. Mine E is a larger mine which indicated that there is scope of improvement on Mine A. Mine A was reviewed first, as it consumes 2.7 more compressed air electricity than Mine G. From the benchmark table, the advantage can be seen that having electricity influencing information provides a better identification of electricity usage improvement strategies.

Compressed air electricity usage for mining group						
Mine Classification	Mine	Installed capacity (kW)	Average electricity consumed per month (kWh)	Average electricity consumed per year (kWh)	% Compressed air electricity consumed of total electricity	Compressed air electricity per tonne milled (kWh/t)
↓S-C-*S-ŘF-LE	Mine D	8 375	2 673 181	32 078 170	41%	79
↓D-C-*M-ŘH-AE	Mine A	17530	7 481 692	89 780 304	36%	103
↓D-C-*L-ŘH-LE	Mine H	19 200	5 136 425	61 637 095	20%	46
↓D-C-*M-ŘF-HE	Mine C	11 600	6 439 840	77 278 075	28%	143
↓D-C-*M-ŘF-AE	Mine F	17 200	4 102 668	49 232 016	22%	127
↓D-M-*M-ŘF-AE	Mine G	6 500	2 150 400	25 804 800	8%	32
↓UD-C-*L-ŘH-HE	Mine E	21 200	8 544 619	102 535 433	15%	93
↓UD-C-*M-ŘF-HE	Mine B	11 750	4 956 509	59 478 113	13%	140

↓S=Shallow      C=Conventional      \*S=Small scale operation      ŘF=Fair profitable      LE=Low electricity consumption  
 ↓D=Deep      M=Mechanised      \*M=Medium scale operation      ŘH=High profitable      AE=Average electricity consumption  
 ↓U=Ultra deep      \*L=Large scale operation      HE=High electricity consumption

Table 3.12: Installed compressor capacities for the mines with related production and electricity consumption.

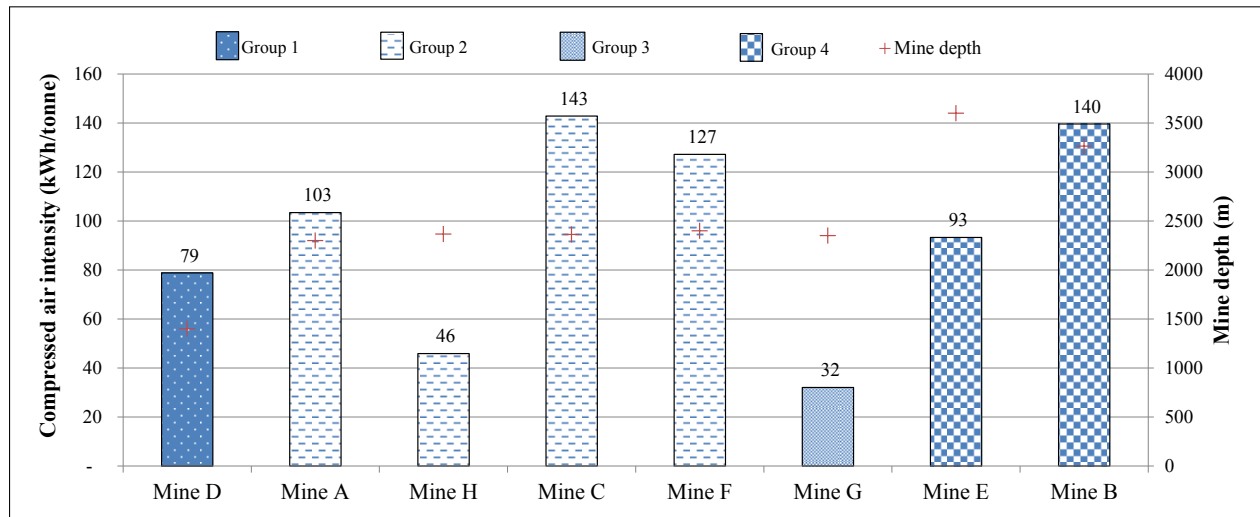


Figure 3.25: Amount of compressor electricity consumed in relation to production.

### 3.4.2 Case study: Compressed air

#### Compressed air optimisation on Mine A

Possible improvement by reviewing the compressed air benchmark data and installed capacities was identified. Mine A was selected due to having the second largest compressed air

consumption and not being highly profitable. From the benchmark data, Mine A has a potential electrical consumption improvement of 32% compared to the monthly electricity usage of Mine H. Mine A has a potential improvement of 23% regarding compressed air electricity consumption per tonne milled. The optimisation and reviewed solutions on the compressed air network of Mine A, with the achieved results is discussed in the sections below.

## Background and layout

Mine A consists of three shafts which all are interlinked with one common compressed air line. Shaft A-1# is the main mining shaft where Shaft A-2# and A-3# are the decommissioned shaft used for ventilation and a secondary escape route respectively. The layout with the installed capacities of the compressors is indicated in Figure 3.26.

The compressor baseline electricity usage is illustrated in Figure 3.27 with the related surface pressure profile of Shaft A-1#. Mine A uses different mining shifts which influences the compressed air demand, being higher during drilling periods and lower during non-drilling periods. One would expect the compressed air electricity usage to follow the demand, consuming more compressed air electricity when the demand is needed and less during non-drilling periods.

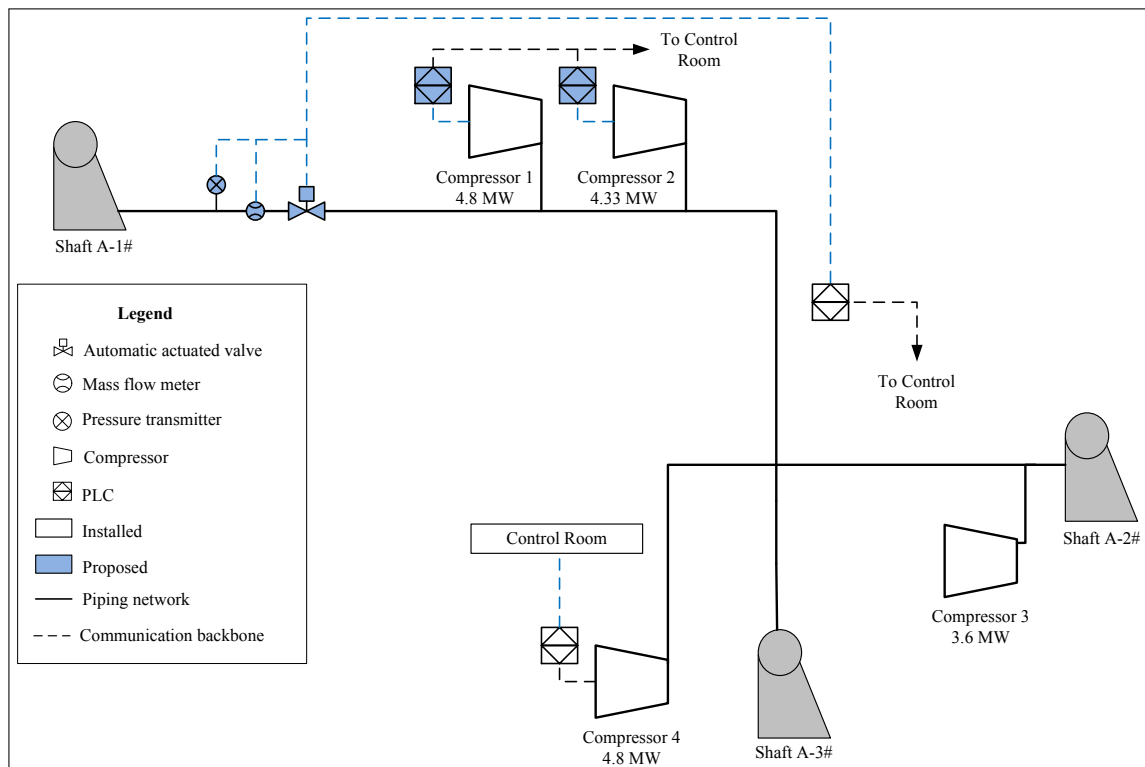


Figure 3.26: Simplified layout of the compressed air network of Mine A.

A DSM project was investigated to reduce the compressed air electricity consumption profile

in non-drilling periods by shutting off the compressors or by controlling the delivery pressure and supply. The system requirements for non-drilling periods and drilling periods were investigated and are listed in Table 3.13.

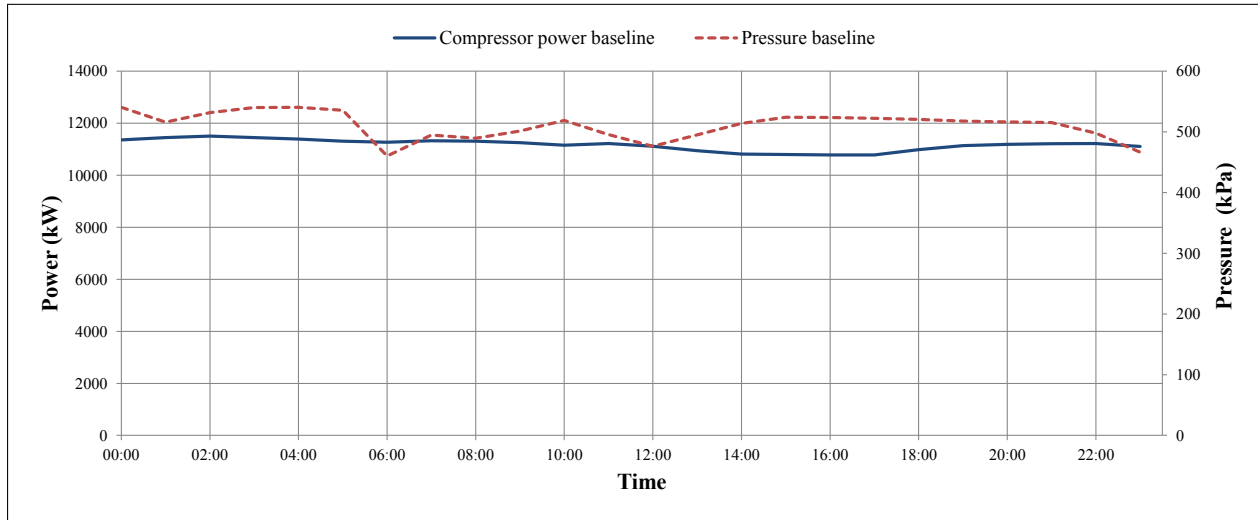


Figure 3.27: Pressure of shaft A-1# and power baseline of Mine A.

Pressure schedule (time period of set points)	Minimum value surface (kPa)	Minimum value underground (kPa)
Peak drilling shift (00:00 – 18:00) & (21:00 – 00:00)	480	480
Non drilling (18:00 - 21:00)	350	350

Table 3.13: Minimum operating pressures related to the mining periods of Mine A shaft A-1#.

### Proposed control philosophies and solutions

Three different compressed air optimisation solutions were investigated and are discussed below.

#### Solution 1:

The first proposed solution was to switch off one compressor during non-drilling periods. The compressor selected was Compressor 3 due to its remote location and pipe friction losses from the supply shaft to the main operating shaft.

The compressed air ring requires that at least two compressors run at the same time, it was confirmed with a test shown in Figure 3.28 where Compressor 3 at A-2# was switched off between non-drilling periods with Compressor 1 and 2 running. The pressure at the

main production shaft dropped below 400 kPa. This proved that in non-drilling periods, it is only necessary for two compressors to run. However, due to security concerns and previous attempts on cable theft of Compressor 3, it was rather requested by mine personnel to investigate other compressor combination solutions.

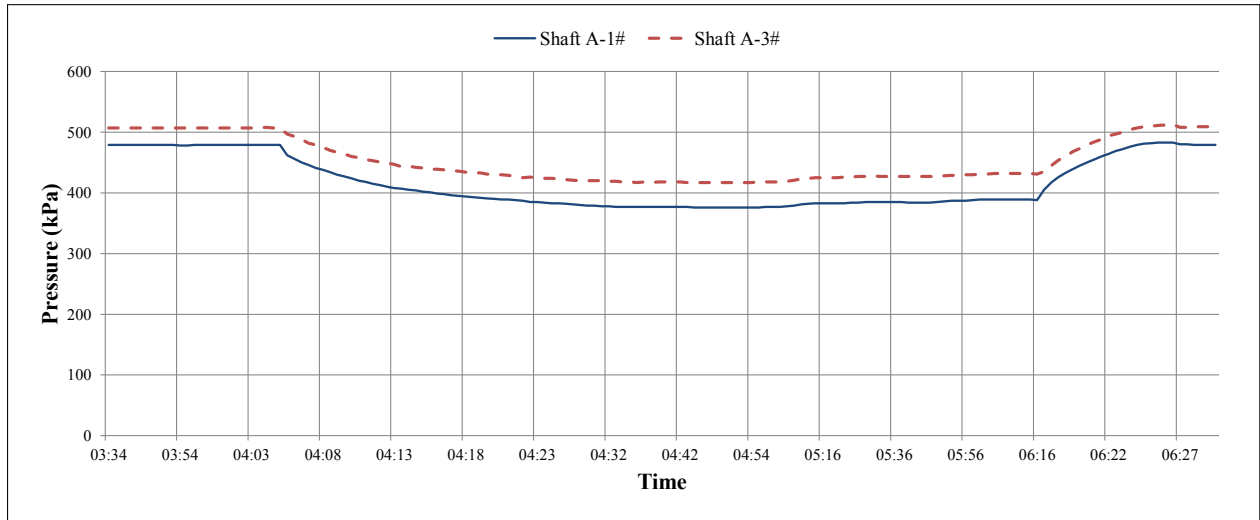


Figure 3.28: System pressure control test by switching off Compressor 3.

### Solution 2:

A second solution investigated was to relocate Compressor 3 to the main mining shaft A-1#. A simulation was performed with commercial package KYPipe<sup>1</sup> with the compressor supply flow and supply pressures. It was concluded that if Compressor 3 at Shaft A-2# was moved to shaft A-1# theoretically only 189 kW of compressed air would be saved. The small saving is due to the compressor not having power-reducing guide vane control and not having an automated stop and start ability. From a service delivery point of view, it would make sense to relocate the compressor. An increased compressed air pressure delivery at A-1# from 480 kPa to 540 kPa is possible.

The time, cost and resultant savings related to relocating a compressor does not justify the project. The time and costs of relocating the compressor is estimated to be R10-million.

### Solution 3:

The third solution was to upgrade and automate the compressors at the main shaft. This provides the compressor operator with the ability of remotely controlling the pressure set point proportionally to the demand, and remotely stopping and starting the compressors. Underground air valves will also be installed to provide the operator with the ability to control the pressure on each level automatically, according to a specified set point.

Mining levels can be isolated when drilling is complete and where other levels still require

<sup>1</sup>KYPipe: Hydraulic calculation and simulation software package, available at <http://kypipe.com/kypipe>

compressed air. The pressure will be reduced with the combination of the proposed control valves and compressor delivery set point control. The proposed underground compressed air control valves, measuring and control equipment is illustrated in Figure 3.29.

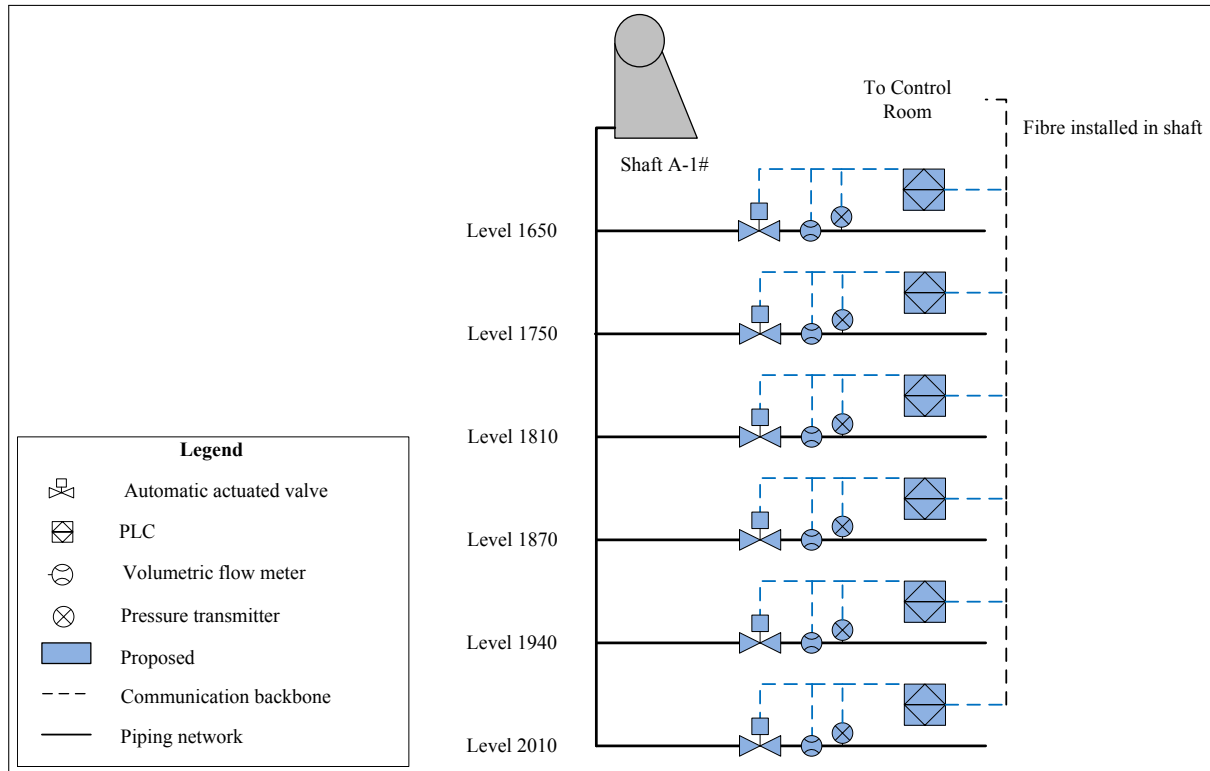


Figure 3.29: Underground layout of proposed pneumatic control valves.

An alternative of installing Variable Speed Drives (VSD) on the compressor's electrical motor rather than stopping and starting the compressors was investigated. Due to the high costs involved in installing a VSD on a typical mining compressor (R6.2-million [32]) and the limited information or research on the VSD solution, it was decided to best implement Solution 3.

### Control philosophy and results

The simplified compressed-air savings approach (Figure 3.24) was used, where the estimated electrical efficiency saving was calculated as 1.45 MW. The savings and calculations are illustrated in Figure 3.30. The simplified approach was used due to the limited information of the compressed air network and the timely/costly effort in investigating all the pipe lengths, dimensions and measuring of the applicable flows and pressures for each level via a portable pressure and flow meter.

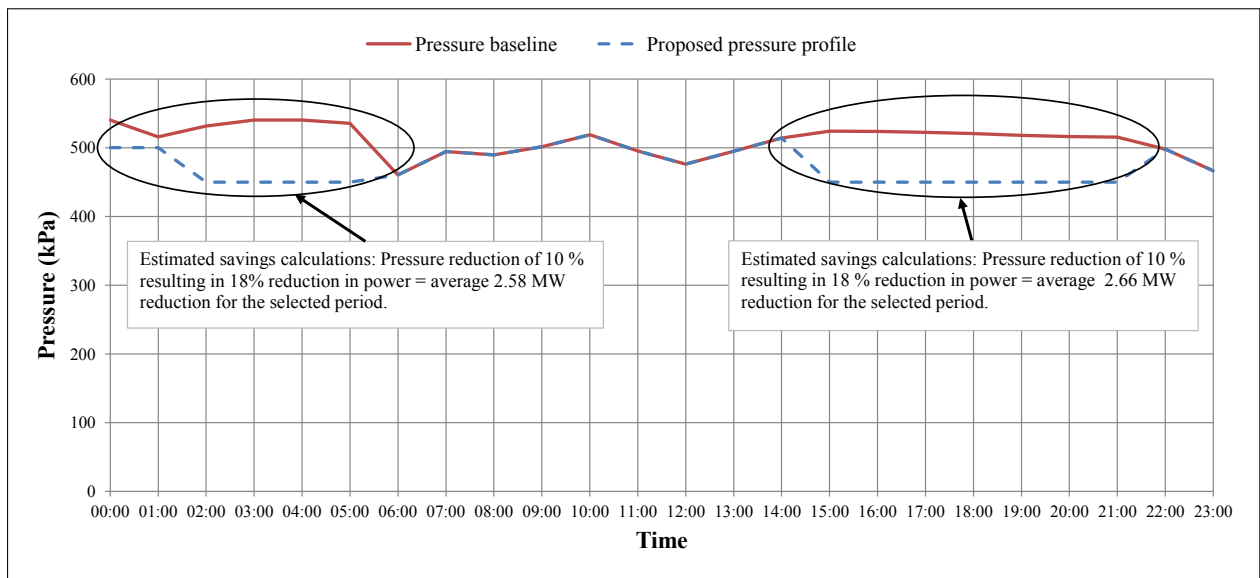


Figure 3.30: Estimated compressed air savings by incorporating pressure set point control and control valves.

The DSM project was implemented and the obtained compressed air surface pressure and compressor set point control is illustrated in Figure 3.31.

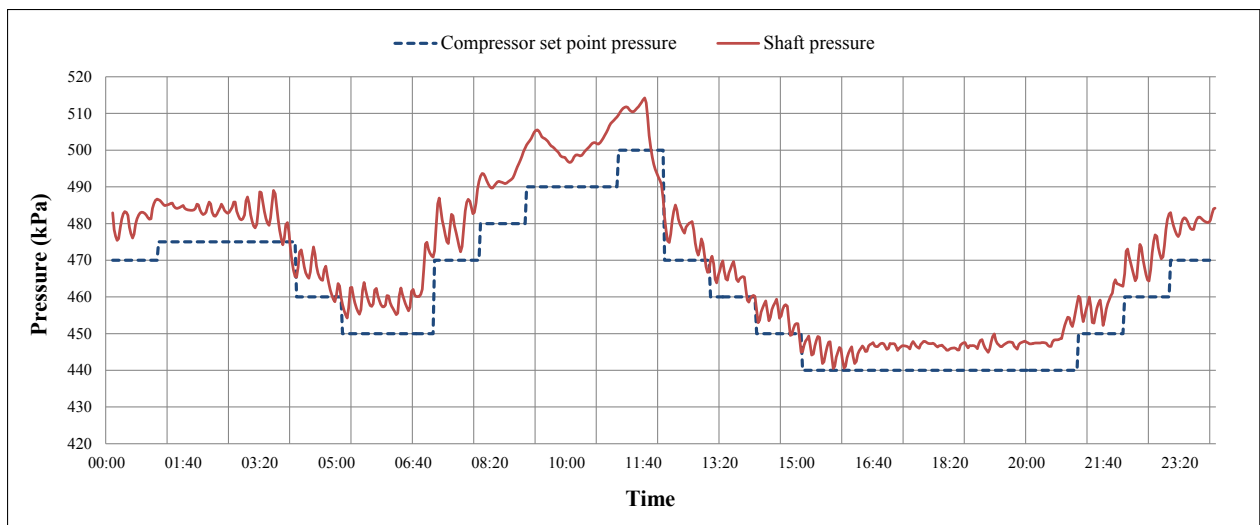


Figure 3.31: Pressure set point control of the supply pressure at shaft A-1#.

The new compressed power consumption profile is illustrated in Figure 3.32. The project resulted in an increased compressed air electricity saving of 1.7 MW for the illustrated average monthly profile. The DSM project resulted in an average baseline reduction of 15% of compressed air power. More savings can be realised with optimal compressor placement and underground leak repair.

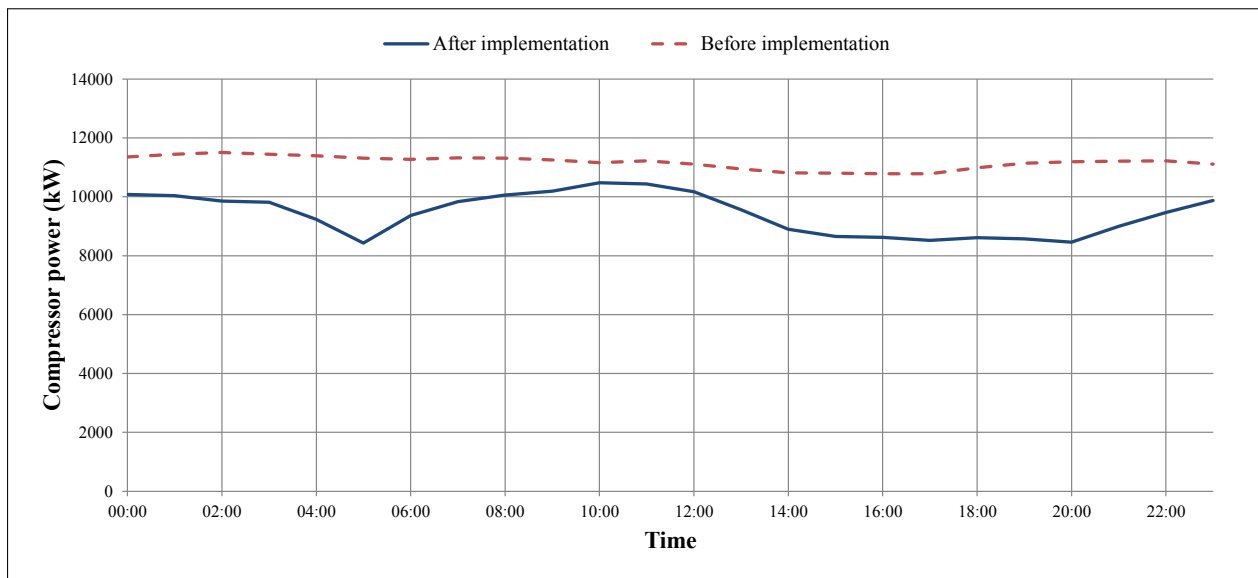


Figure 3.32: Compressed air electricity consumption profile after project implementation.

### Further expansion on other mines

The proven approach was used to quantify and identify possible electricity cost savings projects on the compressed air systems of the selected mines. The peak clip potential and energy efficiency projects were identified and the potential projects listed in Table 3.14.

Compressed air			
Mine	Target (MW)	Mechanism*	Status
Mine A	1.50	EE	Implemented
Mine B	1.96	EE	Implemented
Mine C	1.52	EE	Implemented
Mine D	2.50	PC	Implemented
Mine E	2.80	PC	Implemented
Mine F	2.40	PC	Implemented
Mine F	1.50	EE	Implemented
Mine H	1.44	EE	Implemented
Mine G	0.30	EE	Potential
Mine E	1.20	EE	Potential
Mine D	0.40	EE	Potential
Total implemented peak load reduction (MW):			7.70
Total future peak load reduction (MW):			0.00
Total implemented compressed air reduction (MW):			3.46
Total future compressed air reduction (MW):			9.82
* <b>LS</b> : Load shift, <b>EE</b> : Electricity Efficiency			

Table 3.14: Identified electricity cost savings on the compressed air service of the selected mines.

For the implemented electricity efficiency projects, the loads were reduced throughout the day. The lower loads in the evening made it difficult to motivate a additional peak clip

project by lowering the required pressure even further or stopping a compressor for only two hours.

The potential savings related to the compressed air electricity usage relates to 79 GWh efficiency and 7.7 MW load shift. The electricity efficiency projects relate to an additional 10% reduction of the average yearly compressed air electricity consumed by the selected mines.

It is recommended that for further investigations the compressed air electricity profiles of selected mines must be analysed. Where a compressed air electricity profile does not clearly indicate the production schedules, it indicates the opportunity for compressed air usage optimisation.

## 3.5 Cooling

Underground Virgin Rock Temperatures (VRTs) for deep level gold mines in South Africa can reach up to 42°C-57°C [33]. The deeper the mine, the more heat needs to be dissipated from the work environment to make it safe. The heat is dissipated with cooled ventilated air as well as the use of cold water spray in the working environment.

The rate at which the temperature typically increases per kilometre depth is 12°C. Formal heat stress administrative and management actions need to be taken when Wet Bulb (WB) temperatures exceed 28°C. Routine work must not be permitted when the Dry Bulb (DB) temperature exceeds 37°C or 32°C WB [34]. The depth of the mine thus has a direct relationship with the required cooling energy for safe working temperatures.

The amount of virgin rock to the relative mining depth is illustrated in Figure 3.33 for certain parts in South Africa. The amount of cooling needed to cool down a 150 m long underground working environment with VRTs of 43°C to 27°C DB is 127 kW, as illustrated in [33]. The number of working levels in the mines has a direct relationship to the extent of cooling needed. It is thus essential that cooling is provided for only the needed sections, to ensure optimal cooling system electricity use.

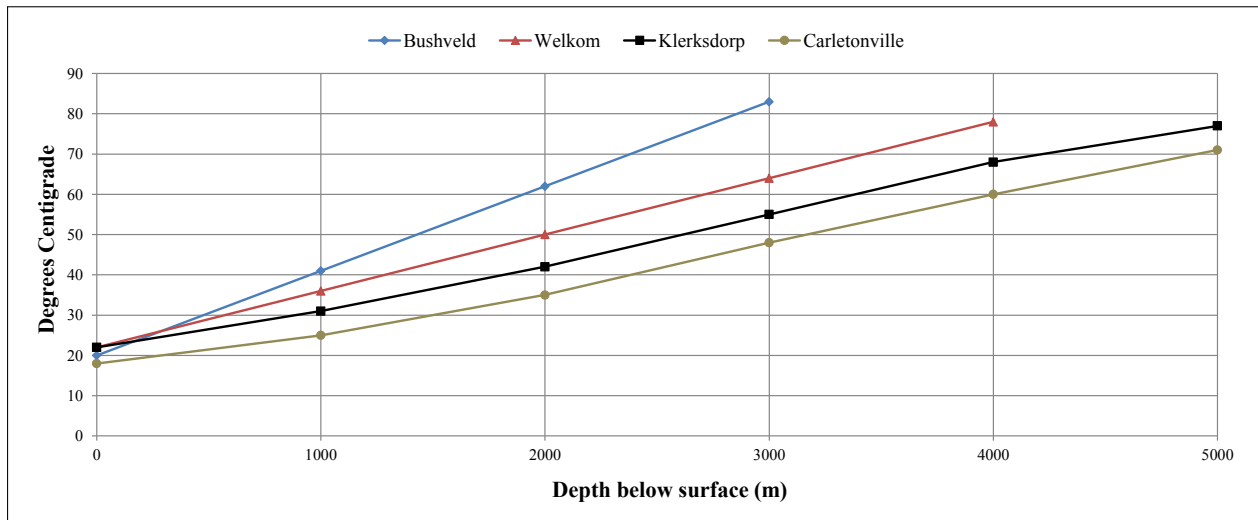


Figure 3.33: Virgin rock temperatures below surface [35].

For the eight selected mines the installed cooling system capacities and electricity consumption is listed in Table 3.15. A cooling system on a gold mine consists mainly of chillers, pumps and fans. From Table 3.15 the relationship between mine depth and installed cooling capacity can be noted, where the largest cooling system electricity consuming mines are the two ultra deep mines Mine E and Mine B.

From Figure 3.34 it can be seen that there is a large difference between the relationship of cooling consumption per tonne milled for Mine E and Mine B. This is due to Mine E almost producing double the quantities of gold compared to Mine B as shown in Table 3.1. This indicates the possibility of optimisation where possible non-production areas or levels are cooled. The following section will provide insight into the system parameters that must be taken into account when deriving energy efficiently measures on the cooling system.

Cooling electricity usage for mining group						
Mine Classification	Mine	Installed capacity (kW)	Average electricity consumed per month (kWh)	Average electricity consumed per year (kWh)	% Cooling electricity consumed of total electricity	Cooling electricity per tonne milled (kWh/t)
↓S-C-*S-ŘF-LE	Mine D	5 640	3 360 000	40 320 000	51%	99
↓D-C-*M-ŘH-AE	Mine A	10 500	5 376 000	64 512 000	26%	74
↓D-C-*L-ŘH-LE	Mine H	7 700	4 569 600	54 835 200	17%	41
↓D-C-*M-ŘF-HE	Mine C	5 400	3 360 000	40 320 000	14%	75
↓D-C-*M-ŘF-AE	Mine F	20 000	6 048 000	72 576 000	33%	188
↓D-M-*M-ŘF-AE	Mine G	16 000	9 565 920	114 791 040	34%	143
↓UD-C-*L-ŘH-HE	Mine E	19 000	11 424 000	137 088 000	21%	125
↓UD-C-*M-ŘF-HE	Mine B	18 000	10 080 000	120 960 000	26%	284
↓S=Shallow		C=Conventional	*S=Small scale operation	ŘF=Fair profitable	LE=Low electricity consumption	
↓D=Deep		M=Mechanised	*M=Medium scale operation	ŘH=High profitable	AE=Average electricity consumption	
↓U=Ultra deep			*L=Large scale operation		HE=High electricity consumption	

Table 3.15: Installed cooling system capacities with related production and electricity consumption.

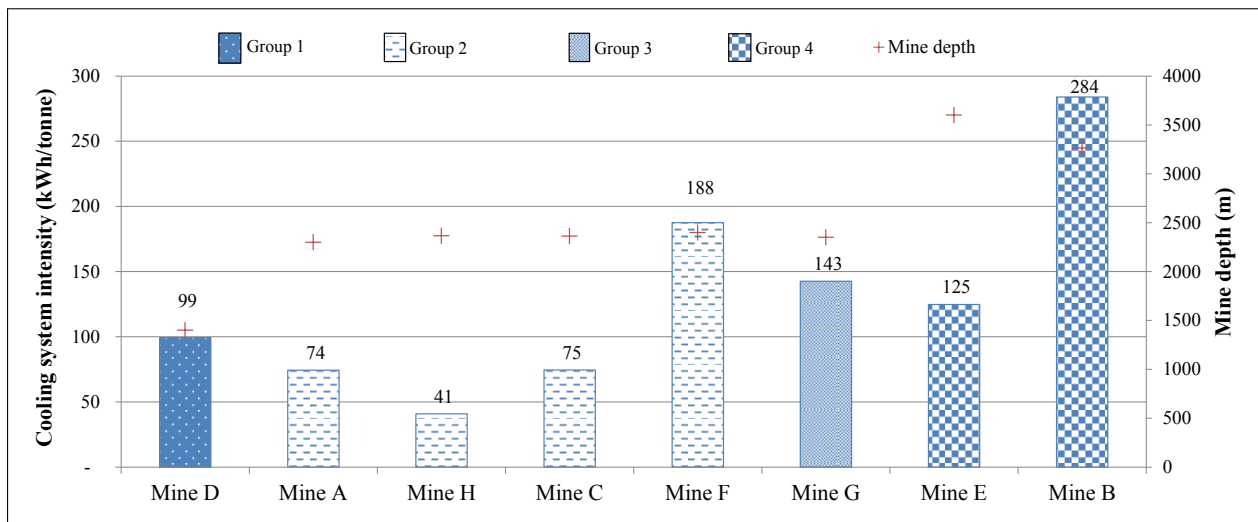


Figure 3.34: Amount of cooling electricity consumed in relation to production.

As discussed in Section 3.3, service water and cooling are interlinked. Chilled water from the cooling system is recirculated through the bulk air coolers and then supplied to the underground works or instope coolers. The amount of cooling required is also related to the large amounts of water that needs to be pumped out, having a direct impact on the pumping energy required.

A solution that has been implemented is freezing the service water and sending it down as ice. The advantage of sending down ice is that a larger heat load can be absorbed reducing the amount of cooling water that needs to be pumped out by almost a quarter, although typical ice plants are expensive to install and maintain [36]. A simplified layout of a typical South African gold mine cooling system is illustrated in Figure 3.35 and described by the numbered items below.

1. Cold water is sent down after being cooled by the cooling system.
2. To ensure constant supply for varying demand, it is stored in a cold dam.
3. The cold water is also sent to the BAC to cool the ventilation air sent down.
4. Hot water is pumped out of the mine and stored in a hot water dam.
5. The hot water is then passed through a precooling tower before being passed through the evaporator side of the cooling system.

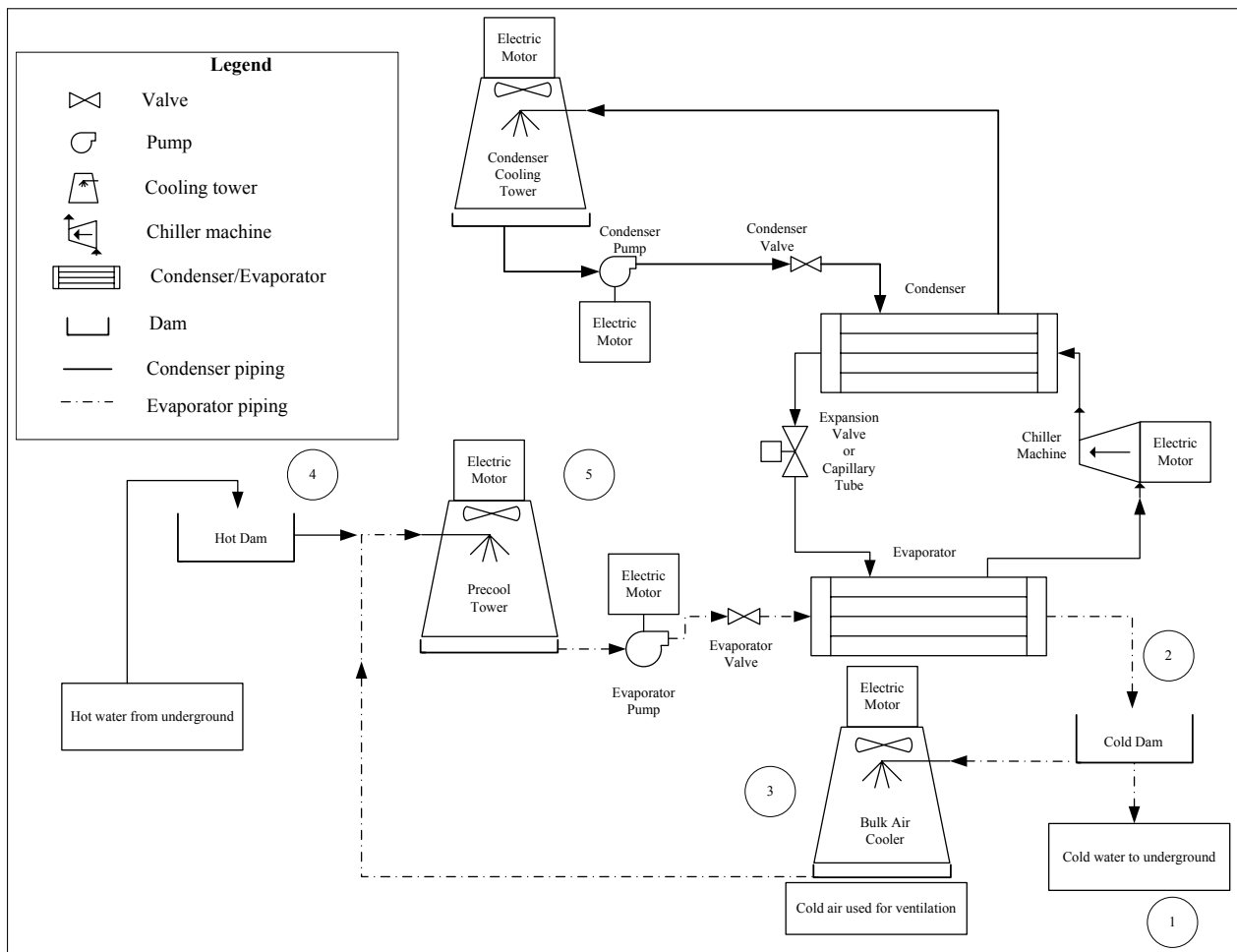


Figure 3.35: Simplified layout of a typical cooling system on a gold mine.

### Load control according to TOU tariffs

To save electrical costs load shift strategies have been developed from cooling systems on the mines [36]. These strategies have been proven to successfully shift load of up to 4.2 MW in the evening [37].

A method for identifying possible load shift projects was developed from Van der Bijl [36] by investigating the load shift potential on twelve different mine cooling systems in South Africa. From the derived method, the load shift potential is primarily determined by the chiller machine utilisation and the installed dam capacity.

The chiller machine utilisation was investigated with daily electricity consumption profiles. The average electrical load and the installed capacity were compared according to Equation 3.11.

$$Utilisation\% = \left(1 - \frac{Q_{average}}{Q_{installed}}\right) \times 100 \quad (3.11)$$

where

$Q_{average}$  = Electrical load of the plant (kW)

$Q_{installed}$  = Installed capacity of the plant (kW)

It was concluded that if the utilisation percentage was greater than 10%, a larger load shift potential was possible for the site [36]. This primarily applies to a summer baseline of electrical consumption. If there is enough chiller capacity, then the cold dam capacity must be verified. The ideal load shift period required is two hours. The cold dam must be large enough to provide cold water for two hours according to the mine demand for cold water underground. The time required to reduce the cold water to the minimum control level specified by the mine must be at least two hours. The goal would be to manage existing equipment with control systems and not replace existing infrastructure or upgrade supply with additional infrastructure.

Cooling system load shift strategies are dependent of the tariff structure, plant operation, climate and storage capacity. A simplified control approach was described in Van der Bijl [36] consisting of a preparation and control period, according to the TOU structure. To effectively calculate and determine the possible savings from a mine cooling system, several constraints must be evaluated. The following sections will provide insight into determining the possible electrical cost savings on a typical cooling system for South African mines.

To perform a electrical load shift on a cooling system, the following steps must be taken [36]:

### **Preparation**

- Decrease the water inflow to the hot dam to ensure that it does not overflow in the Eskom peak period.
- Increase the chilled water level in the cold dam for the stopping of machines in the Eskom peak period.
- Increase the flow of the cold water and cool the water to as cold as possible to induce thermal storage.
- Back-pass the cold water from the cold dam to either the hot dam or the chiller machine to lower the temperature or to reduce the chiller machine's compressor power.

### **Control and load shift**

- Stop the fridge plant according to the correct constraints and procedures.

- Increase the hot water dam level to maximum before starting fridge plants.
- Decrease the cold water dam level to minimum before starting a chiller machine.
- Back-pass the cold water from the cold dam to either the hot dam or the chiller machine to lower the temperature or to reduce the chiller machine's compressor power.

The dam levels according to the supply and demand flow can be calculated with Equation 3.12 [37].

$$DL_2 = \frac{\left[\left(\frac{DL_1}{100}\right) \times Dv + (\dot{m}_{in} - \dot{m}_{out}) \times 3.6 \times t\right] \times 100}{Dv} \quad (3.12)$$

where

$$\begin{aligned} DL_2 &= \text{New dam level (\%)} \\ DL_1 &= \text{Previous dam level (\%)} \\ \dot{m}_{in} &= \text{Inflow (\ell/s)} \\ \dot{m}_{out} &= \text{Outflow (\ell/s)} \\ Dv &= \text{Dam volume (m}^3\text{)} \\ t &= \text{Time (hour)} \end{aligned}$$

The temperatures related to the dam can be calculated with Equation 3.13 [37].

$$DT_2 = \frac{\left[\left(\frac{DT_1}{100}\right) \times Dv \times DT_1 + (\dot{m}_{in} \times T_{in} - \dot{m}_{out} \times T_{out}) \times 3.6 \times t\right]}{\left(\frac{DL_1}{100}\right) \times Dv + \dot{m}_{in} + \dot{m}_{out}} \quad (3.13)$$

where

$$\begin{aligned} DT_2 &= \text{New dam temperature (}^\circ\text{C)} \\ DT_1 &= \text{Previous dam temperature (}^\circ\text{C)} \\ DL_1 &= \text{Previous dam level (\%)} \\ T_{in} &= \text{Temperature of inflow (}^\circ\text{C)} \\ T_{out} &= \text{Temperature of outflow (}^\circ\text{C)} \\ \dot{m}_{in} &= \text{Inflow (\ell/s)} \end{aligned}$$

$\dot{m}_{out}$  = Outflow ( $\ell/s$ )

$Dv$  = Dam volume ( $m^3$ )

$t$  = Time (hour)

The steps involved in investigating and determining the potential load shift are listed in [36]. The two processes were reviewed and simplified into one process, as illustrated in Figure 3.36.

During the investigation, many parameters may be unknown but can be calculated by two cooling system fundamental equations. The thermal energy absorbed by the cooling system through the cold water is calculated with Equation 3.14.

$$Q = \dot{m}C_p\Delta T \quad (3.14)$$

where

$Q$  = Thermal energy (J)

$\dot{m}$  = Mass flow (kg/s)

$C_p$  = Specific heat constant (-)

$\Delta T$  = Change in temperature ( $^{\circ}C$ )

The chiller machine's Coefficient Of Performance (COP) can be calculated with Equation 3.15. If the compressor's power is unknown, the amount of electrical energy can also be calculated by using the design COP.

$$COP = \frac{Q}{E_{Comp}} \quad (3.15)$$

where

$Q$  = Thermal energy (kW)

$E_{Comp}$  = Compressor electrical energy (kW)

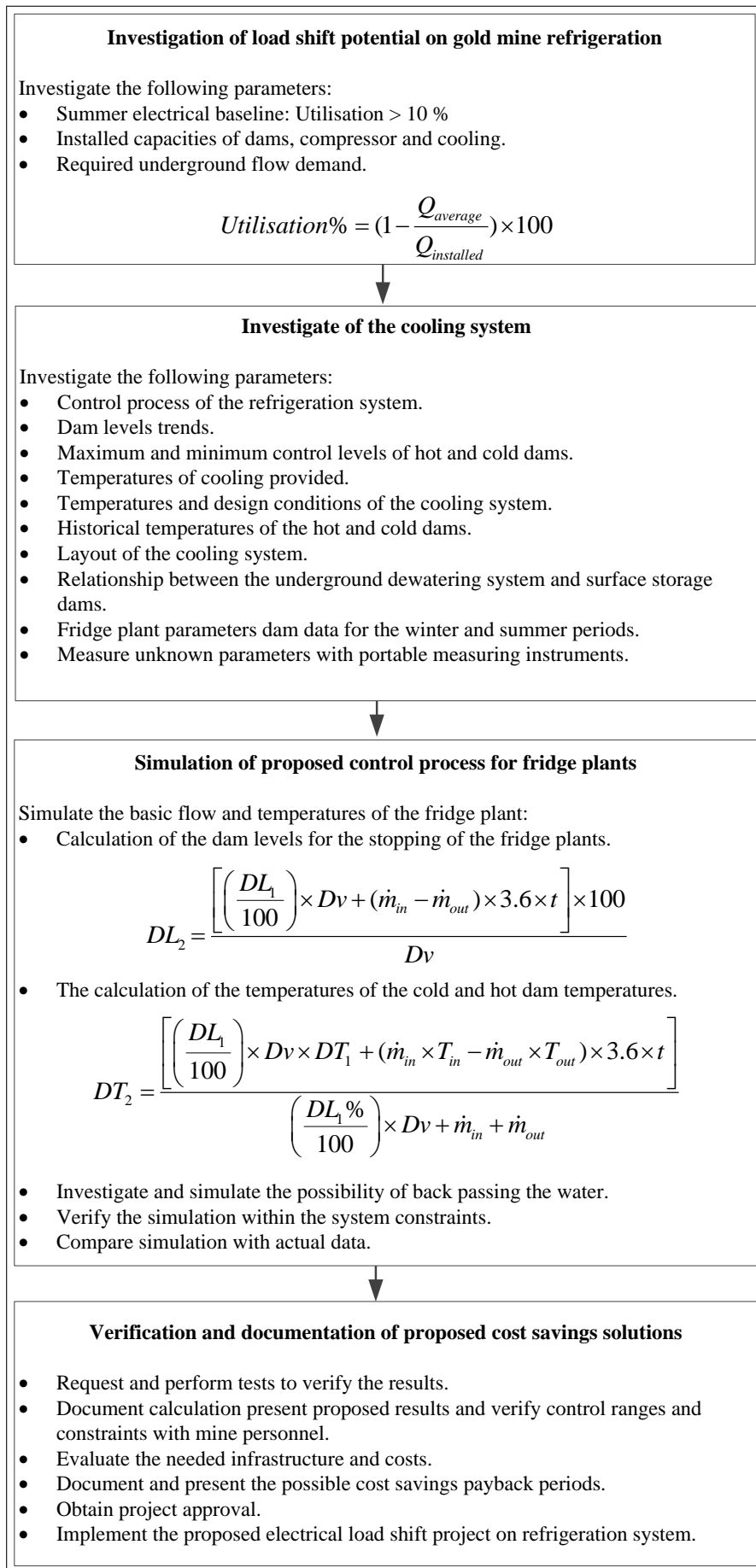


Figure 3.36: Typical steps needed to derive and calculate the potential load shift possibility on a cooling system.

## Matching cooling supply with demand

It was found from several studies that the supply of coolant (air and water) is oversupplied without taking energy efficiency and optimal operations into consideration [8], [38]. These studies used basic performance indicators that can be used to calculate and assess energy efficiency interventions. A good solution would be to match supply and demand by adjusting the supplied coolant temperature of the cooling system accordingly [39].

Typical inefficient operation refers to the installed motors being oversized and operating at low load factors. Other factors include slow operational changes due to ambient changes and consistent water wastages. The installation of VSDs on typical cooling system motors has been investigated by reviewing inefficient operations on several mines [40]. This prompted the review of the selected mining company to identify the possibility of installing VSDs on the cooling systems of the identified mines. Figure 3.37 illustrates a scenario where the flow was throttled back with a valve to match the cooling requirements. The valves were fully opened and a VSD was implemented to control the required flow resulting in electrical savings.

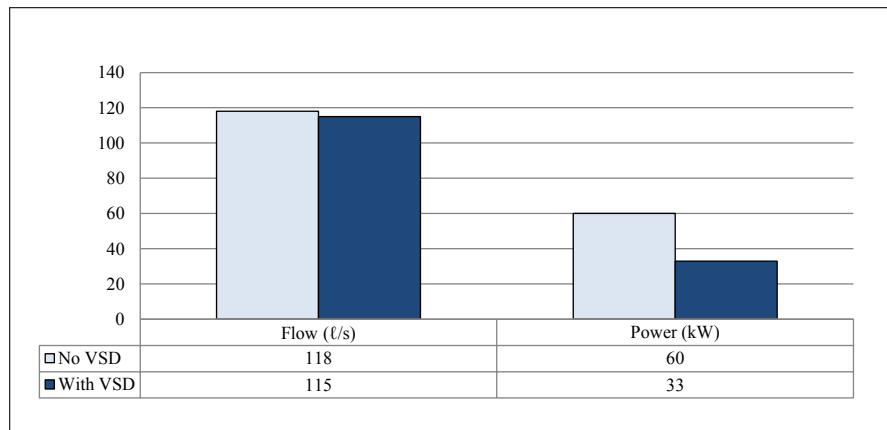


Figure 3.37: Evaporator flow rate comparison of VSD-controlled flow and valve restricted flow.

A VSD provides a variable voltage, current and frequency to the motor by means of Pulse With Modulation (PWM). The change in power (current and voltage) is directly related to the speed and torque of the motor [41]. This provides the ability of changing the speed according to the demand. The relationship between speed and power reduction of an electric motor is illustrated in Figure 3.38.

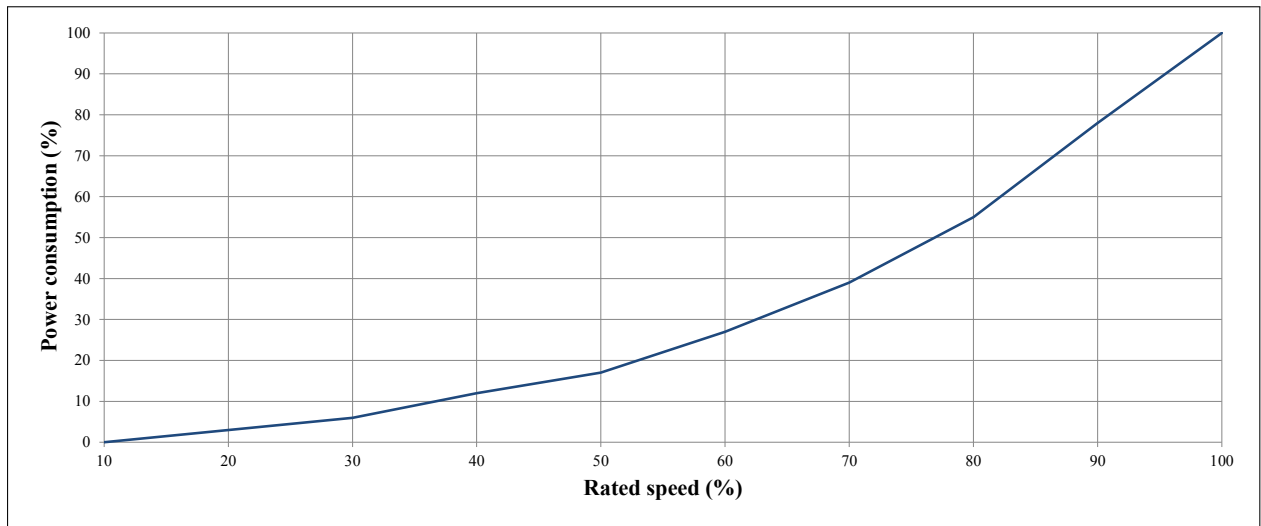


Figure 3.38: Power reduction and VSD motor speed relationship [42].

Pumps on a mine cooling system consist of chill water, a condenser, evaporator, transfer and BAC with ranges of 50 kW to 500 kW. Axial fans include a precooling tower, condensers cooling tower and BAC with ranges of 40 kW to 400 kW, with the average total installed capacity of 10.8 MW [40].

It has been proven that of the total cooling system electricity consumed, the chiller consumes 66%, pumps 27% and the fans 7% [38]. A study was performed where an estimated energy saving on each of the systems was estimated with VSD implementation. From various studies, it has been proven that with VSD implementation and controlling the motor speed, electricity savings can be realised by controlling the speed to match the given load [43] [44].

Methods have been derived to estimate the potential savings on the different motors of the cooling system of a mine [40]. The method is based on the simplified approach developed by Saidur et al. [45]. The electrical consumption on chillers and pumps can respectively be calculated by Equations 3.16 [46] and 3.17 [47].

$$EC_{chiller} = (OH)(\dot{Q}_c)(LF)(COP^{-1}) \quad (3.16)$$

and

$$EC_{pump, fan} = (OH)(\dot{W}_{rated})(LF) \quad (3.17)$$

where

$$EC_{chiller} = \text{Chiller electrical energy consumption (MWh)}$$

$\dot{Q}_c$  = Chiller rated cooling capacity (MW)

$OH$  = Operating hours (h)

$LF$  = Load factor (-)

$COP$  = Coefficient of performance (-)

$EC_{pump,fan}$  = Pump or fan electrical energy consumption (MWh)

$\dot{W}_{rated}$  = Pump or fan power rating (MW)

The mines of the selected mining group were analysed and it was found that on the cooling system little or no VSDs have been installed. This can be accounted to the lack of knowledge of mine personnel or the new technologies that resulted in VSD being much cheaper than it was in the past. With the lower electricity tariffs in the past, new technology initiatives such as VSDs were not pursued due to the expensive costs involved [40].

By using the relationship between speed or load reduction, the electricity savings of chillers operating between 60% and 100% were calculated and are shown in Figure 3.39 for a typical 5 MW chiller machine [40]. These savings can be realised by the installation of a VSD and by operating the chiller as normal.

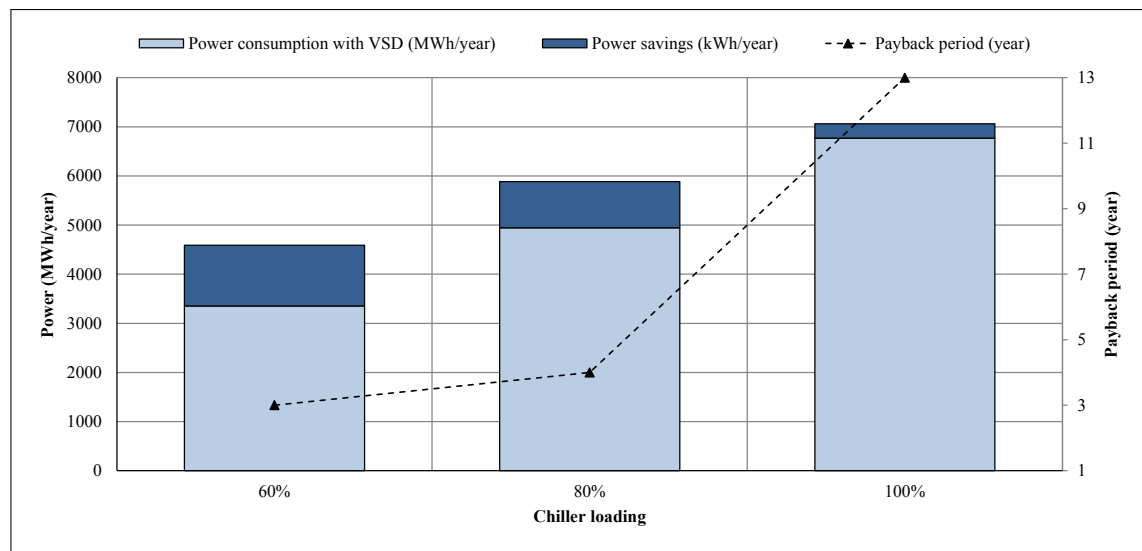


Figure 3.39: Chiller machine cost savings with VSD installation payback scenarios [40].

The higher the loading factor, the lower the feasibility of installing a VSD on a chiller machine. As shown in Figure 3.39, the payback period doubles as the loading factor is increased by 20 % [40].

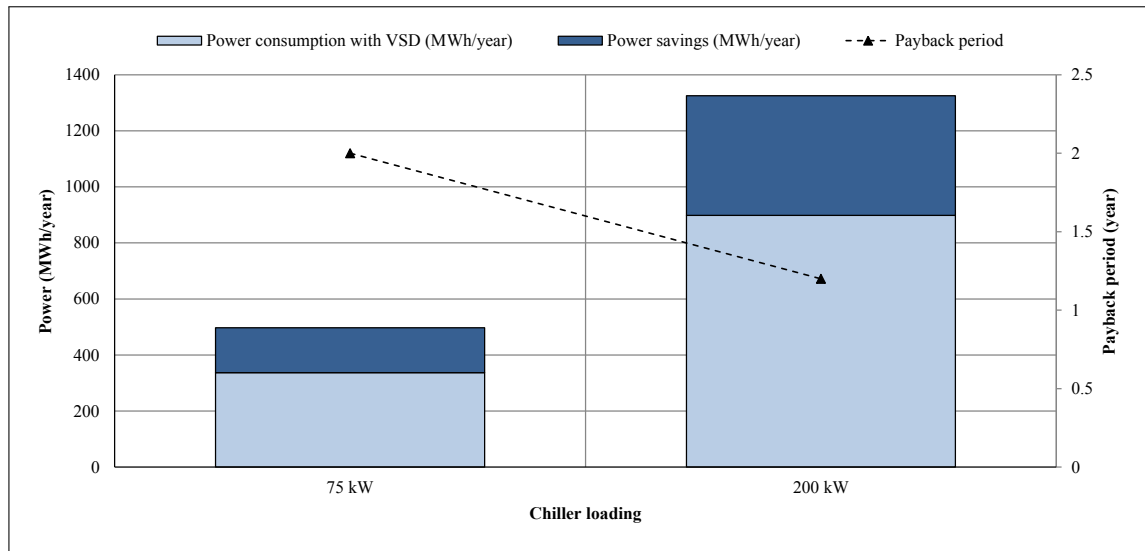


Figure 3.40: Pump and fan cost savings with VSD installation payback scenarios.

VSD installation and electricity savings scenarios were also investigated on pumps and fans of cooling systems. Realistic savings can be realised by operating the pumps at reduced daily load profiles. The payback periods of the pumps related to the possible savings of 32% is much lower than that of chiller VSD installations [40]. This is mainly accredited to the lower cost of low voltage VSDs. The cost relationship per kW rating of medium and low voltage VSDs is illustrated in Figures 3.41 and 3.42. Compared to chillers, pumps have a better electricity savings return in relation to VSD costs. This is mainly accredited to the high load factor of a chiller due to constant high cooling requirement of the mines.

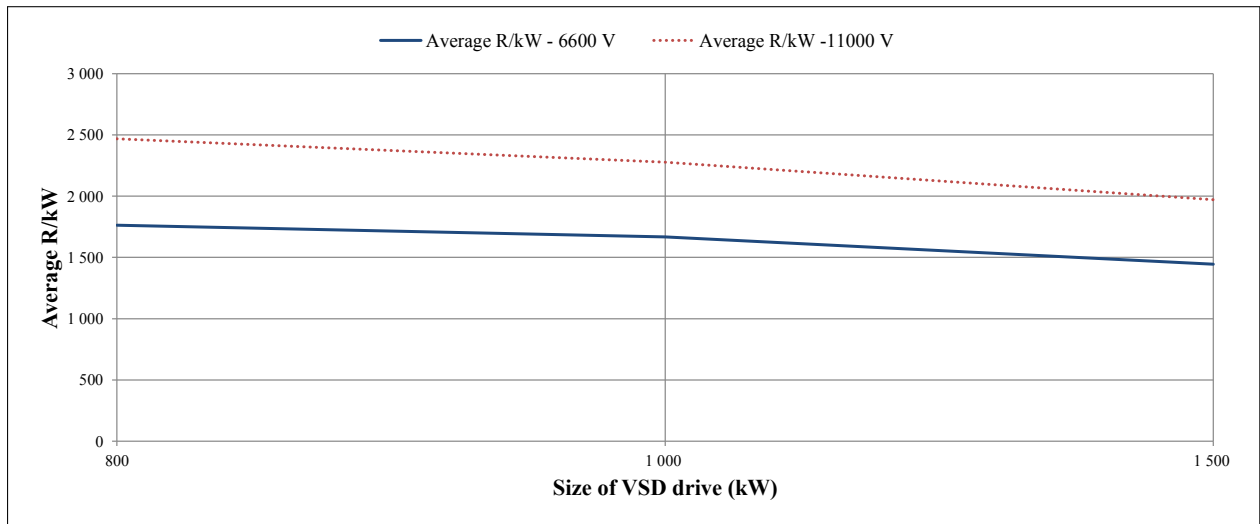


Figure 3.41: Cost comparison of average R/kW for different size chiller VSDs.

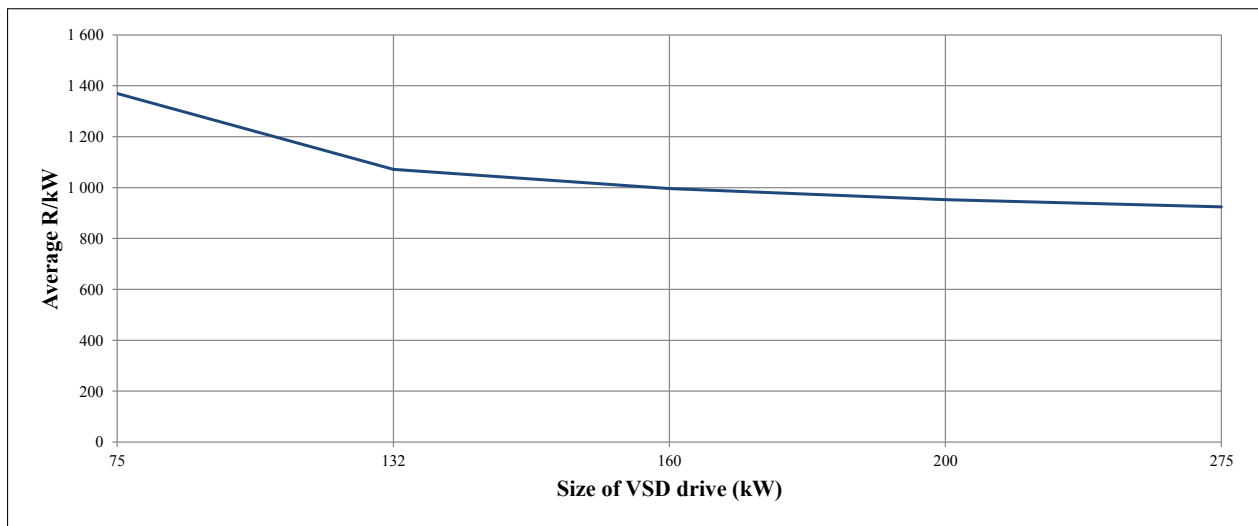


Figure 3.42: Cost comparison of average R/kW for different size pump and fan VSDs.

It has also been shown that variable flow strategies do not adversely affect performance or mine service delivery [48]. Savings of 60% have been proven with 30% flow reductions. Flow reduction of more than 30% it is not recommended, as reduced flow rates lower than 30% of the designed flow results in COP degrading [49]. The controlled VSD flow and valve regulated flow for a gold mine case study were compared and is illustrated in Figure 3.37 [40]. From the abovementioned section and case study it have been proven that by installing VSDs on the cooling system of a gold mine could result in savings of up to 29.9% of the installed capacity of the cooling system [40].

It is apparent from the abovementioned sections that the whole cooling system must be reviewed once installing VSDs on the pumps. The capability provided by VSDs to control the flow provides the opportunity of reducing electricity consumption or improving the service delivery of the dependent subsystems. The additional advantages of variable flow control resulting from VSD installations has been proven on several South African gold mines [38]. The following section will describe the control related to each system with regards to energy efficiency and flow control.

### **Bulk Air Cooler flow control**

The Bulk Air Coolers (BAC) found on the mines of the selected mining group were designed for a required supply air temperature of  $\pm 8^{\circ}\text{C}$  WB for a specific summer's day ambient condition. Optimal control would include controlling the BAC at a constant temperature for a hot summer's day throughout the year. The lower ambient WB temperatures in the winter months or cooler days will thus require that less cold water be supplied to the BAC to cool the air down to the required temperature. The lower flow required by the BAC will result in savings on the pumps controlled by VSDs and on the compressor of the chiller machine.

### Cooling system evaporator flow control

The evaporator water flow can be reduced by 30% of the design without having a measurable negative effect on the heat transfer of the evaporator [48]. Most of the mines for the investigated mining group use cold storage dams to supply underground works with cold water. When there is an over supply of cold water, the cold water either gets dumped or overflows to the hot dam.

By controlling the evaporator flow, the cold dam levels can be maintained at lower levels, reducing the flow in the pumps and the amount of cold water that needs to be cooled. This results in the chiller machine compressor savings due to the reduced cooling demand as well as VSD savings on the pumps.

### Cooling system condenser flow control

Condenser flows on mine cooling systems can be reduced by 40% from the designed minimal flow without resulting in a loss in heat transfer. The reduced flow of 60% will result in the same COP with full load conditions [50]. In scenarios where the condenser flow is lower than the minimum flow for partial loads, the COP of the cooling system will be effected. The loss of COP is outweighed by the electricity savings on the condenser pumps.

These control energy savings strategies will be achieved mainly by VSD installations and optimal control. Another optimising strategy for the cooling system includes reviewing and ensuring that precooling towers for the cooling system is operating efficiently and is cleaned regularly. Well designed and maintained precooling towers can operate up to a COP of 30, where an efficient mine chiller machine operates at a COP of 6. From the greater COP performance of the precooling tower, it is beneficial to ensure correct operation of the precooling tower [50].

### Simplified approach for VSD pump savings

The condenser and evaporator flow will very rarely be reduced below 30% of the design flow. An equation was developed to provide a conservative estimation of the potential VSD savings from the flow reduction. Equation 3.18 was derived from Figure 3.43 where the flow reduction is between 0% and 30%. The estimation is illustrated on the graph, assuming an intersection at zero.

$$VSD_{savings} = \left(1 - \frac{m_{reduced}}{m_{measured}}\right) P_{pump} \times 2 \quad (3.18)$$

where

$$VSD_{savings} = \text{Savings resulting VSD flow reduction (kW)}$$

$m_{measured}$  = Measured flow before VSD implementation ( $\ell/s$ )

$m_{reduced}$  = Reduced flow with VSD implementation ( $\ell/s$ )

$P_{pump}$  = Measured power usage of pump with measured flow (kW)

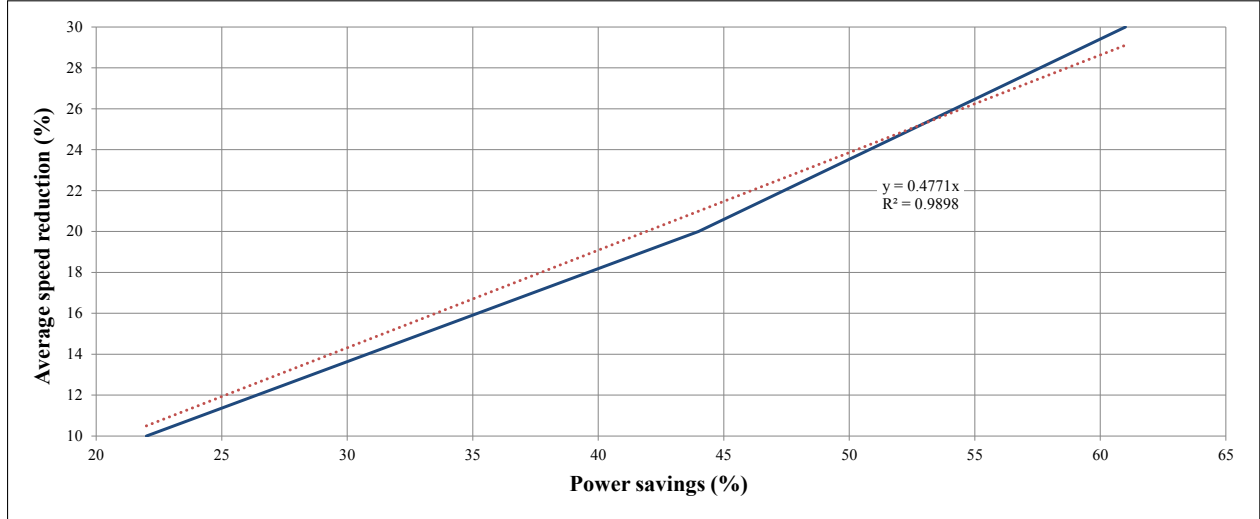


Figure 3.43: VSD power savings related to speed reduction [42].

A simplified approach for investigating the possible implementation of VSDs on the cooling system of a gold mine is illustrated in Figure 3.44.

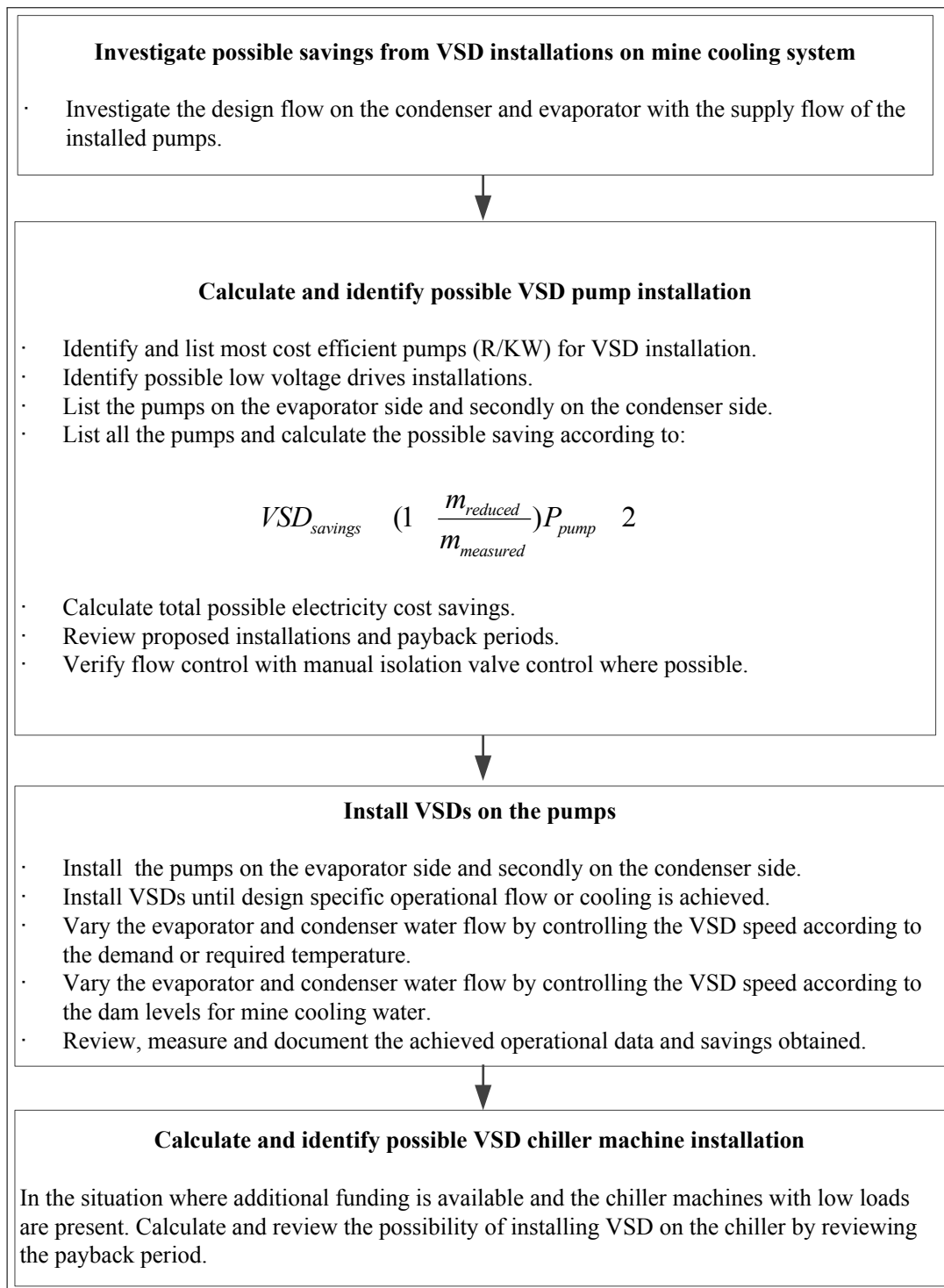


Figure 3.44: A simplified approach for investigating the possible implementing of VSDs on a cooling system.

### 3.5.1 Case study: Cooling

#### Cooling auxiliaries and fridge plant optimisation of Mine E

The electricity cost mitigation strategies regarding cooling systems investigated in this study include performing electrical load shift by utilising the cold dam as storage for the underground water supply as a first solution. The second solution is to install VSDs on the pumps or chiller machines and vary the flow according to the temperature and water demand underground. The load shift strategies will provide electricity cost savings but will not result in overall energy reduction or carbon emission reductions.

To minimise possible electricity cost risks not being TOU related, the focus of this section is to identify, validate control and review implemented strategies to reduce average electricity usage throughout the day. This case study will then discuss the second solution of installing VSDs on the cooling system pumps and regulating the flow for each subsystem.

#### Background and layout

For this case study, Mine E was selected due to the large cooling capacity and being one of the deepest mines with the highest cooling system electricity consumption of the selected mines. Mine E has an installed surface cooling system capacity of 13 MW with a total cooling system capacity of 19 MW including the underground cooling system.

The simplified layout of the cooling system of Mine E is illustrated in Figure 3.45, with the system specifications listed in Tables 3.16, 3.17 and 3.18. The cooling process and cooling system can be described by the following numbered items as indicated in Figure 3.45.

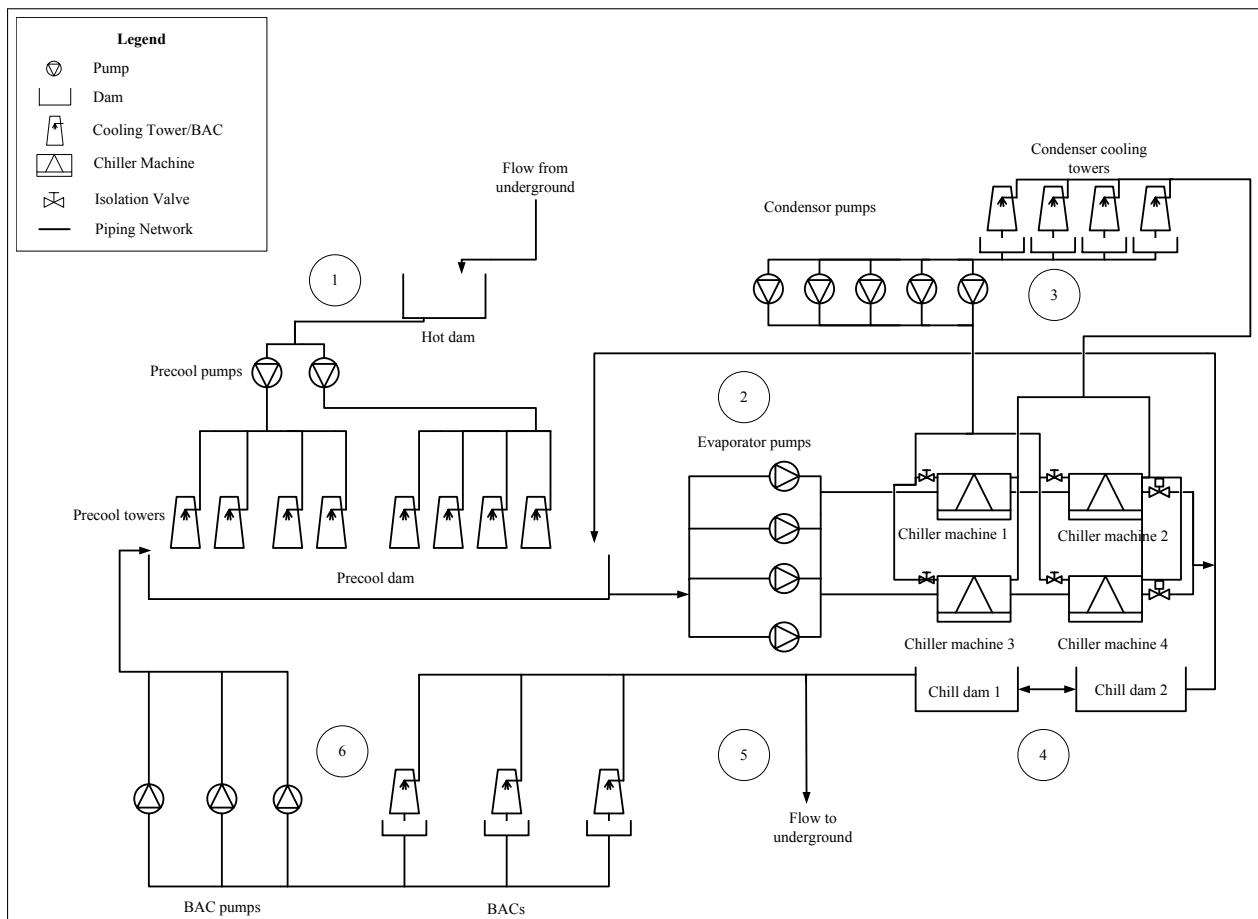


Figure 3.45: Basic layout of Mine E's surface cooling system.

1. Hot mining water ( $28^{\circ}\text{C}$ ) is pumped from underground and supplied to the precooling tower via two precooling pumps.
2. The pre-cooled water ( $18^{\circ}\text{C}$ ) is then supplied to the evaporator side of the cooling system via four evaporator pumps.
3. The condenser cooling system consists of five condenser pumps and four condenser cooling towers.
4. The chilled water from the chiller machines is supplied to two chill dams as storage.
5. The cold water from the chill dams ( $5^{\circ}\text{C}$ ) is supplied underground and to the BAC.
6. The BAC chilled water is then supplied to the precooling dam depending on the sump levels of the BAC.

### Proposed control philosophies and solutions

The proposed control philosophy was to install VSDs on selected pumps, with the goal of obtaining savings from matching the supply flow with the needed control or specified flow.

Description	Surface
Number of Chiller Machines	4
Make	Hitachi
Compressor Type	Centrifugal
Refrigerant	R134a
Voltage (V)	6600
Cooling Capacity (kW)	13300
COP	6.65
Evaporator Outlet Temperature (°C)	5.9
Condenser Inlet Temperature (°C)	18.5
Evaporator Water Flow (kg/s)	300
Condenser Water Flow (kg/s)	600
Evaporator Pump Motor Rating (kW)	90
Number of Evaporator Pumps	4
Condenser Pump Motor Rating (kW)	185
Number of Condenser Pumps	5

Table 3.16: Cooling system design specifications for Mine E.

Description	Surface MM	Surface RV
Number of BACs	2	1
Water Inlet Temperature (°C)	3	3
Water Outlet Temperature (°C)	14	14
Water Flow (kg/s)	150	150
Airflow (kg/s)	225	230
Air Inlet Wet Bulb Temperature (°C)	18	18
Air Outlet Wet Bulb Temperature (°C)	7	7
Pump Motor Rating (kW)	75	75
Number of Pumps	2	1

Table 3.17: Surface BAC specifications for Mine E.

Description	Surface
Number of Cooling Towers	8
Water Inlet Temperature (°C)	28
Water Outlet Temperature (°C)	24
Water Flow (kg/s)	300
Air Inlet Wet Bulb Temperature (°C)	7
Pump Motor Rating (kW)	75
Number of Pumps	2

Table 3.18: Precooling tower specifications for Mine E.

These VSD installations and variable flow controls can provide up to 30% electrical savings on the cooling system [38].

The proposed variable flow control and improved electrical operation of the different auxiliaries of Mine E's cooling system can be described as follows:

1. Keep the present precooling tower control philosophy and only improve the cooling capabilities of the cooling towers by replacing or upgrading the cooling towers.
2. Install VSDs on three of the evaporator pumps. Only switch on the evaporator pumps when needed or vary the flow according to the cooling demand on the chill dams.
3. Install VSDs on the condenser pumps and vary the flow to obtain optimal heat transfer of 5°C over the condensers.
4. The chill dams levels will be controlled accordingly while always being above 50% and temperatures of not above 5°C.
5. The BAC is gravity-fed and control valves will be used to control the BAC underground supply temperature.
6. VSDs will be installed on the BAC pumps to ensure the maximum sump level of 90%.

The proposed VSD and control valve installations are illustrated in Figure 3.46.

### **Implemented control philosophy and results**

From the investigation and proposed equipment implementation, the following findings were presented for Mine E.

The precooling tower did not yield a sufficient cooling and the fill material needed to be replaced. The low ambient WB temperatures represented either low humidity or cool air or a combination of both [51]. The precooling towers were upgraded with new fill material yielding much better cooling.

The condenser flow of Mine E was reduced by 8% with VSD control resulting in a power reduction of 21%. Figure 3.47 illustrates the flow reduction and electricity savings obtained from the VSD flow control on the condenser pumps. Due to the relatively large installed capacity of the condenser pumps, an average electricity saving of 199 kW was realised. The second contribution to the large savings compared to other VSD installation on the cooling system are due to the condenser control being less sensitive to heat load changes than the evaporator. The heat load is determined by the evaporator flow control, making it essential to install viable flow control on the evaporator side.

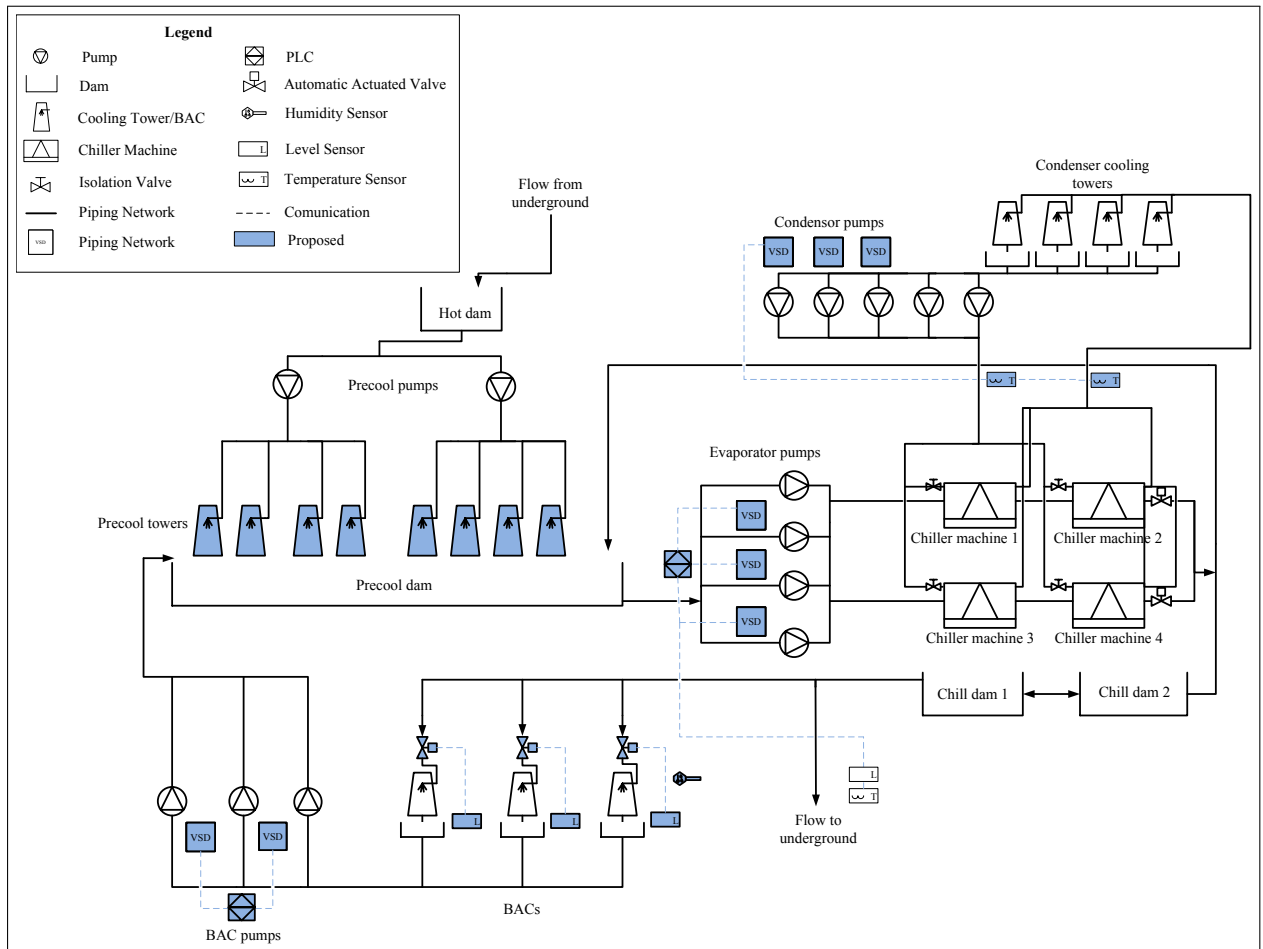


Figure 3.46: Basic layout of Mine E surface cooling system with proposed BAC valve and VSD implementation.

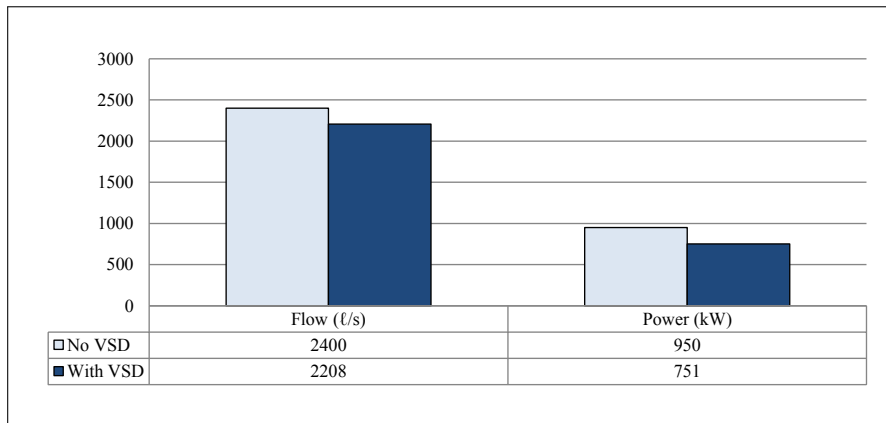


Figure 3.47: VSD condenser pumps electricity savings and flow comparisons.

Figure 3.48 illustrates the flow and reduction from VSD installation on the evaporator pumps and resultant electricity savings. The evaporator flow was reduced by 27% according to the proposed control philosophy of 30%. The average evaporator pump power decreased by 69%. The combined savings of the VSD evaporator control resulted in an average load reduction of 177 kW. The savings are realised for controlling the pumps on partial and full load in relation to the chill water demand. The savings obtained also correspond to the simplified approach of estimating VSD savings illustrated in Figure 3.44.

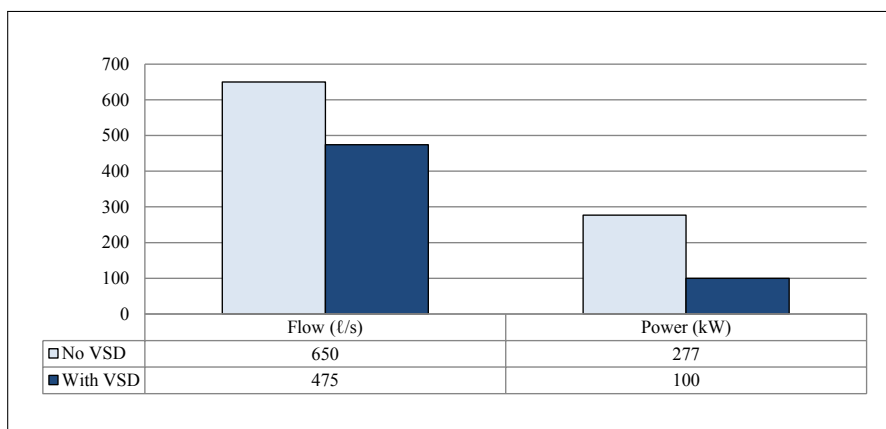


Figure 3.48: VSD evaporator pumps electricity savings and flow comparisons.

The water pumped by the BAC pumps is directly related to the water supplied to the BAC. Due to lower cooling requirements of the BAC on lower WB temperature days, less water will be pumped from BAC to the precooling dam. The reduced amount of water that is also sent to the BAC also relates to the lower amount of cooling energy consumed by the cooling system. The average flow rate reduction of the BAC pumps was 46%. The relationship between the flow reduction and electricity reduction is illustrated in Figure 3.49. The reduction of flow resulted in a pump power reduction of 66%. The electrical savings resulting from the BAC VSD installation are lower in comparison to the other VSD pump installations. However, the lower cost of the smaller BAC pump VSDs and by only

Item	Selected component improvements	Saving (MWh)	Saving %
1	Pre-Cooling Towers	6 662	19.6%
2	BAC Water Flow Control	1 198	2.6%
3	Evaporator Water Flow Control	903	6.0%
4	Condenser Water Flow Control	2 640	7.2%
<b>Total</b>		<b>11 402</b>	<b>35%</b>

Table 3.19: Savings contribution of each system.

installing two, the BAC VSD installation was approved. The average impact of the BAC VSD installation was 70 kW. The savings contribution of the related improved auxiliaries is summarised in Table 3.19.

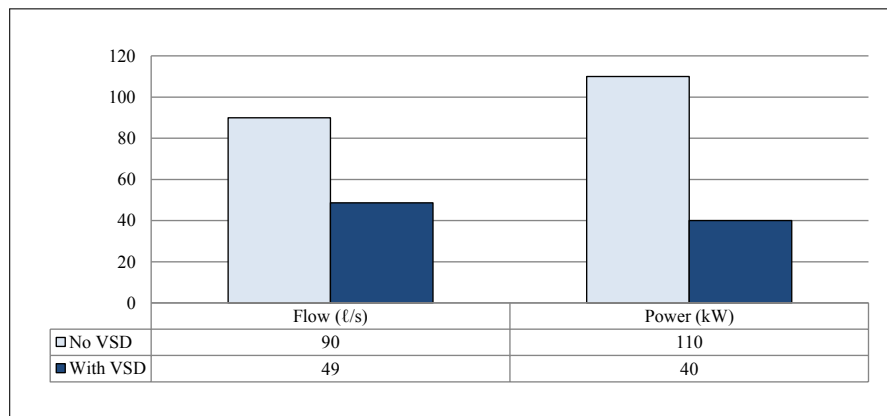


Figure 3.49: VSD BAC pumps electricity savings and flow comparisons.

From the combined initiative, the total combined electricity savings from the project resulted in 1.77 MW where the VSD pump electrical savings accounts for 447 kW. The other large electricity savings can be accounted to the large amount of water that was reduced on the chiller side. This is due to the evaporator flow control only supplying enough water to keep the chill dams full and water reduced through the BAC with valves. Cooling was no longer circulated in the precooling dams to obtain the correct water inlet temperature. The second factor was the improvement of the precooling towers which also allowed for less cold water needed to be circulated in the precooling dam.

The combined effect of the precooling towers and reduced flow resulted in optimal chill water supply with little or no over cooling.

### Further expansion on other mines

From the above-listed results, it becomes evident that the largest savings were realised from the improvement of the precooling tower's performance. The savings which resulted from the VSDs installed on the pumps accounted for 25% of the total 1.7 MW savings of the project. The savings would not have been realised without VSDs. From other studies, it

was found that the largest portion of the savings was realised on the chiller machines from the reduced flow on the evaporator side [38].

From this, it is evident that for optimal savings the system as a whole must be analysed and the supply control optimised according to the design specifications. These estimations must first be simulated and service delivery taken into consideration. To ensure production, safety remains top priority and the required temperature should always be measured to validate assumptions of proposed control.

The proven approach was used to quantify and identify possible electricity cost-savings projects on the cooling systems of the selected mines. The load shift potential and electricity efficiency projects were identified and the potential projects listed in Table 3.20.

<b>Refrigeration</b>			
<b>Mine</b>	<b>Target (MW)</b>	<b>Mechanism*</b>	<b>Status</b>
Mine A	2.68	LS	Identified
Mine B	2.80	LS	Identified
Mine E	2.90	LS	Identified
Mine G	3.10	LS	Identified
Mine H	3.30	LS	Identified
Mine H	1.50	EE	Identified
Mine E	1.50	EE	Implemented
Mine E	6.18	LS	Implemented
Mine H	1.60	LS	Implemented
Mine D	0.45	EE	Potential
Mine A	0.84	EE	Potential
Mine C	0.43	EE	Potential
Mine F	1.60	EE	Potential
Mine G	1.28	EE	Potential
Mine B	1.44	EE	Potential
Total implemented peak load reduction (MW):			7.78
Total future peak load reduction (MW):			22.56
Total implemented refrigeration profile reduction (MW):			1.50
Total future refrigeration profile reduction (MW):			7.54
* <b>LS</b> : Load shift, <b>EE</b> : Electricity Efficiency			

Table 3.20: Identified electricity cost savings on the cooling service of the selected mines.

The potential savings related to the compressed air electricity usage relates to 60 GWh efficiency and 22.5 MW load shift. The electricity efficiency projects relate to an additional 8% reduction of the average yearly cooling system electricity consumed by the selected mines.

There are smaller projects that can be implemented, but these are the largest savings identified by the ESCo. The 8% electricity reduction corresponds to that of other studies with VSD installed on cooling system auxiliaries.

### 3.6 Benchmarking and optimisation results

From the results, benchmarking can be used as a decision model during high cost-risk periods and to identify possible electricity usage improvement. For the electrical improvement of individual services, the available technologies must be reviewed. After reviewing proven mitigation strategies, a total reduction of 30% on the eight selected mines services can be achieved. From the data it should be noted that compressed air technologies have been fairly well implemented, indicating that the focus must rather be on other services, such as cooling.

The potential electricity reduction for the selected services is illustrated in Table 3.21 with the relationship of each service's electricity reduction in relation to total electricity reduction also indicated. It should be noted that a total electricity reduction of 6% could be obtained by implementing the identified and possible electricity savings projects. The 6% reduction is based on the derived quantification approaches and provides an indication of what level of mitigation can be obtained.

<b>Mining service</b>	<b>% Potential electricity reduction in main services</b>	<b>% Electricity reduction in relation to total power consumed</b>
Pumping:	12%	2%
Compressed air:	10%	2%
Refrigeration:	8%	2%

Table 3.21: Results of the possible obtainable mitigation in relation to total power consumed and each main service.

The identified load shift potential for the selected mines were quantified and is listed in Table 3.22. The potential load reduction for the main mining services was calculated as 25%. The electricity cost savings would relate to a yearly total electricity cost reduction of 1%, derived from Figure 3.16 of Section 3.3. The identified electrical load shift and the electricity efficiency projects could result in a combined electricity cost reduction of 7%.

<b>Mining service</b>	<b>% Potential electrical load reduction in main services</b>	<b>% Electricity price reduction for the main service</b>	<b>% Electrical load reduction for total electrical load</b>	<b>% Electricity price reduction for total electricity cost</b>
Pumping:	7%	1%	1%	0%
Compressed air:	0%	0%	0%	0%
Refrigeration:	18%	3%	5%	1%

Table 3.22: Results of the possible obtainable load reduction in relation to total power consumed and each main service.

To quantify the mitigation effect of the possible electricity reduction, the potential price risks scenarios were compared with the potential mitigation strategies. The potential risk impact

is illustrated in Figure 3.50. For the calculations and illustration the following assumptions were made:

**Mitigation scenario one:** The first scenario is an annual price increase of 8% and the additional 4% carbon tax increase in 2015; thereafter increasing by 12% annually. No mitigation was attempted and operations continued as before. The annual electricity costs in 2017 in comparison to 2012 are illustrated on the secondary axis as 47%.

**Mitigation scenario two:** As concluded from this chapter, a total mitigation reduction of 7% through identified projects and load management is possible. The identified projects are implemented over three years with a 2.3% reduction each year starting in 2014. The project will be paid off through the first year's savings (2013 for 2014) and with mine capital allocated for electricity reduction. Electricity costs are 7% lower than in mitigation scenario one (2017) resulting in annual electricity costs 37% higher than in 2012.

**Mitigation scenario three:** Mitigation with three years 2.3% electricity efficiency plus additional IDM funding of 0.55 R/kW is received. The additional IDM funding results in an additional electricity cost saving of 1% from 2012 and 2017. The electricity cost in 2017 will be 37% more than in 2012.

If the mine was to implement mitigation scenario two, over the five year period R675-million would be saved in electricity costs. If the IDM funding was utilised, an additional R144-million would be received through funding and would ideally be used to fund additional DSM projects.

The ECS is classified as a short-term, high-cost risk situation when the supply is heavily burdened and load shedding needs to be avoided. Thus ECS was not included for this five yearly cost risk analysis with available mitigation strategies. Reactive power factor improvement was not investigated due to the payback period being up to five years. If reactive power charges were charged throughout the year, the mitigation and possible scenarios should be investigated.

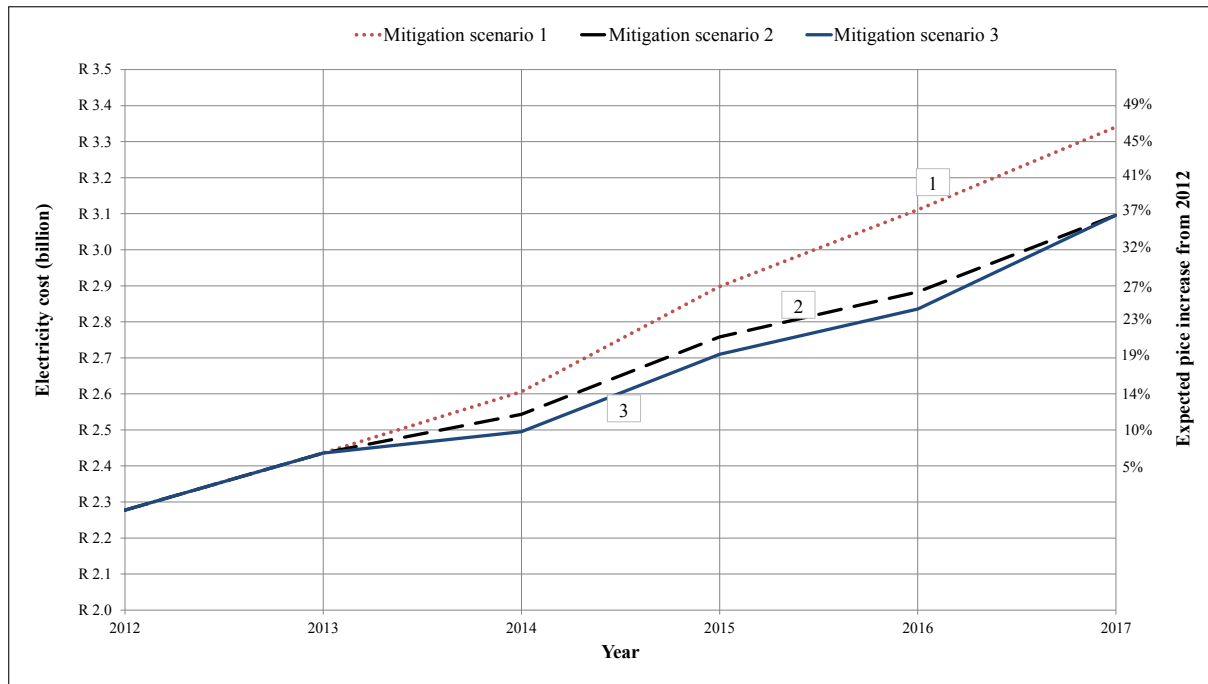


Figure 3.50: Mitigation scenarios and potential cost impact.

The identified savings with the available technologies were proven with relevant case studies. Other technologies are available and the mine should invest in investigating other technologies on other listed services. The initial mitigation approach is based on the principle that fast implementable, proven and large savings must be implemented first. The potential estimated savings were conservatively calculated and to quantify the exact savings and required infrastructure, each potential project must be thoroughly investigated first.

A simplified method to quantify the possible scenarios is by converting the infrastructure cost related to the possible electrical savings in terms of a return of investment [38]. A database of equipment cost (e.g. valves, PLCs and VSDs) and supplier lists should be compiled to calculate the project budget and review all the potential projects before implementing.

Ideally the identified projects must have a payback of less than one year to reduce the impact of the yearly price increase. To evaluate potential projects, the infrastructure costs related to obtaining the savings needs to be evaluated. The project could incorporate the financial benefits of DSM funding. By assuming a yearly average electricity cost of 55 c/kWh and 55 c/kWh for DSM funding the ideal project budget for an identified project could be calculated with Equation 3.19.

$$\text{ProjectBudget(Rand)} = E_{Savings} \times E_{Cost} \times 365 \times 24 + DSM_{funding} \times E_{Savings} \times 365 \times 24 \quad (3.19)$$

where

$E_{Savings}$  = Electricity Savings (kWh)

$E_{Cost}$  = Electricity Cost (R/kWh)

$DSM_{funding}$  = DSM funding (R/kWh)

## 3.7 Conclusion

In this chapter, the largest electricity consuming services were identified and the related mitigation strategies derived. As the first step the benchmarked data was used to identify potential mines where electricity cost savings projects can be implemented. The characteristics which influence electrical intensity in gold mining were identified and the selected mines were classified accordingly.

For each service a derived mitigation strategy was developed which can be used by mine personnel to identify and quantify potential electricity-reducing projects. To verify the developed strategy, implemented DSM projects on the mines of the selected mining company were reviewed as case studies.

After reviewing and proven mitigation strategies, a total reduction of 30% on the eight selected mines services can be achieved. The identified electrical load shift and the electricity efficiency projects could result in a combined electricity cost reduction of 7%. The combined cost risk of 12% quantified in Chapter 1 was reviewed according to the potential electricity reduction with the calculated potential DSM projects.

The multiple derived and identified DSM projects should be managed accordingly to ensure prompt implementation and sustained savings, resulting in a combined electricity cost reduction. Chapter 4 will provide insight in to the developed reporting methodology used in this study to manage and identify additional electricity savings through monitored services.

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