CHAPTER 1 INTRODUCTION

1.1 OVERVIEW

This chapter highlights the importance of the mining industry to South Africa and introduces the challenges facing the industry with respect to the energy it consumes. Once these parameters have been defined, the problem statement and purpose of the study are given, after which the methodology used is presented. Lastly, an outline of the study in terms of the chapter content is given.

1.2 RELEVANCE OF THE MINING INDUSTRY

The mining industry is one of the mainstays of the South African economy. This is borne out by the figures quoted by the Chamber of Mines of South Africa's Annual Report (2009–2010) (Chamber of Mines, 2010), which contains the latest data available. The information quoted below illustrates the loss to the South African economy if the mining sector were to be 'removed' from the economy.

If mining were temporarily removed from the South African economy, the economy would lose:

- 19% of GDP (8.8% directly)
- In excess of 50% of merchandise exports (primary and beneficiated exports)
- 1 million jobs
- 18% of gross investment (10% of direct investment)
- Approximately 30% of capital inflows into the economy (via the financial account of the balance of payments)
- 33% of the market capitalisation of the Johannesburg Stock Exchange
- 93% of the country's electrical generating capacity
- 30% of the country's liquid fuel supply
- 20% of direct corporate tax receipts

South Africa has a wide assortment of minerals available for exploitation. These resources have been effectively exploited and this has given rise to the dominant position in which mining finds itself in the South African economy. This exploitation has been facilitated by the relative accessibility of these minerals.

Historically, one of the most important commodities mined in South Africa has been gold. This has recently been matched and exceeded by the resurgence of platinum and the majority of more recent capital investments have been in this, the platinum sector. The importance of these two sectors is shown by the figures published by Statistics South Africa in their report of 2011 (2011a). These figures have been summarised in Figure 1-1.

It can be seen from these figures that the Platinum Group Metals (PGMs) earn the most revenue. The next highest revenue-generating mineral is coal, closely followed by gold. Coal, however, is not evaluated here as these mines tend to be very shallow by comparison and employ mining methods that are substantially different from the hard rock mining methods use in the platinum and gold mines.

The data represented in Figure 1-1 summarise the total value of mineral sales according to mining divisions, mineral groups and minerals. These data are collated by the Minerals Bureau and cover all mining establishments operating in the South African economy (Statistics South Africa, 2011a).

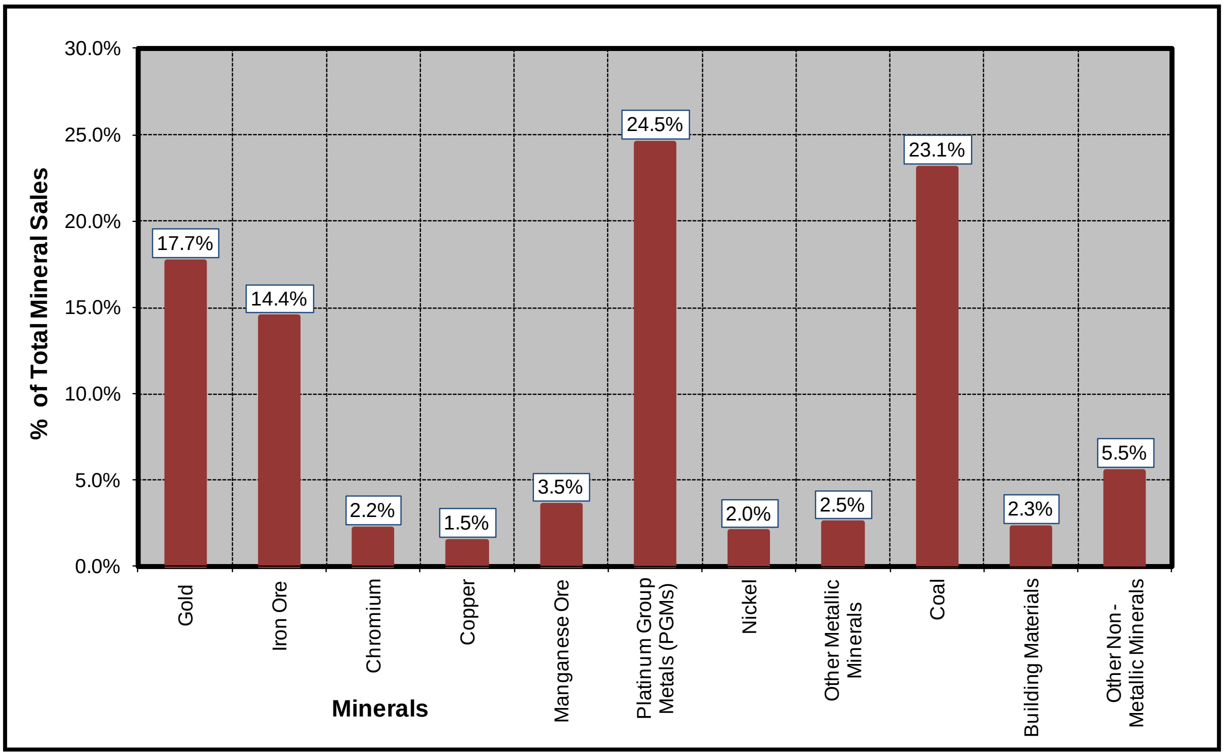

Figure 1-1: Summary of overall mineral sales contribution in South Africa (Statistics South Africa, 2011a)

As a result of the figures shown in Figure 1-1, the emphasis in this thesis is placed on relatively deep mines as are found in the platinum and gold industries. Both of these minerals are generally mined from depth, with both the platinum and gold mining industries having to go deeper in order to access additional reserves. It should, however, be noted that, generally speaking, the mines developed for gold mining are deeper (maximum depth is in excess of 4 000 m) than those for platinum mining (maximum depth in excess of 2 000 m). Both of these industries face similar challenges primarily as a result of the differences in the geothermal gradients found in the regions in which these minerals are mined.

Relatively deep-level platinum mines in South Africa are found in the Bushveld Igneous Complex (BIC). This region has some unique geophysical properties, one of which is that the temperature gradient increase with depth is approximately twice as severe as in the West Wits Basin (WWB), where the deep-level gold mines are generally situated. The geothermal gradient of platinum mines in the BIC is typically 2.2°C per 100 m (Hustrulid and Bullock, 2001), which is more than double the typical gradient of 1.1 °C per 100 m (Hartman et al., 1997) found in gold mines in the WWB.

This poses some challenges with respect to heat dissipation within mines: the virgin rock temperature (VRT) of 40 °C is realised at a depth of 650 m in the BIC, as opposed to a depth of 1 800 m in the WWB (Biffi et al., 2006). This difference must be borne in mind when discussing or comparing deep-level platinum mines with their gold counterparts.

1.3 ELECTRICAL ENERGY SUPPLY

The mining industry in South Africa faces particular challenges at present, primarily in the area of electrical energy consumption. During 2007, South Africans faced increasingly stringent load shedding of the electrical supply. In January 2008 Eskom took the unprecedented step of informing its key industry consumers (KICs) that it could no longer guarantee its supply to them. This announcement resulted in the temporary closure of all the deep-level mines associated with the large mining houses (i.e. Anglo American, Gold Fields, etc.) as a result of safety concerns if the power did indeed fail.

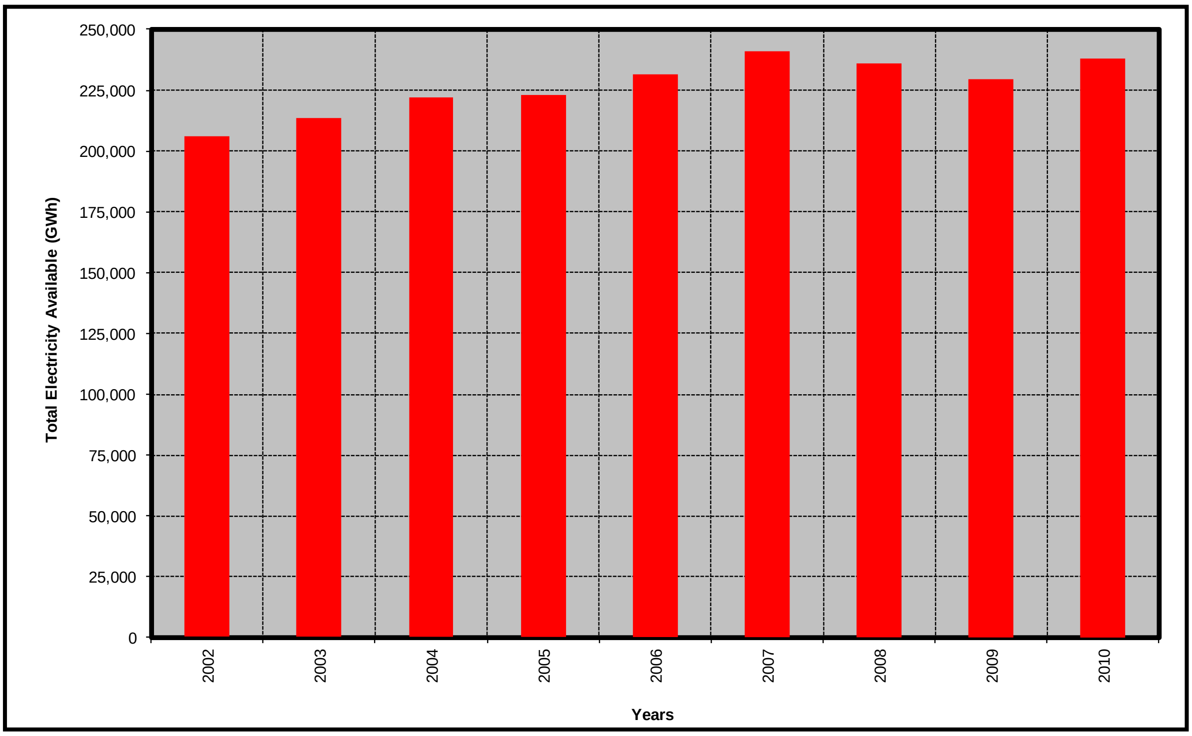

Figure 1-2: Total quantity of electricity available for distribution in South Africa (Statistics South Africa, 2011b)

Figure 1-2 shows the electricity that has been generated by Eskom over the past number of years. This graph shows that the overall generation capacity for the last five years has been reasonably consistent. This has resulted in a number of years of cheap and consistent electrical power supply. There has been a small increase in the total indicated capacity supplied by Eskom since 2002 of an average of 1.6% per annum (Statistics South Africa, 2011b). During the same period, however, the South African economy grew on average 3.7% per annum (Statistics South Africa, 2011c). It is obvious from these statistics that the economic growth of the country has not been matched by a commensurate increase in Eskom's capacity.

Figure 1-3: Comparison of annual percentage change in available electricity and GDP (Statistics South Africa, 2011c)

Eskom noted in its 2010 Annual Report that the current maximum generating capacity was approximately 40 870 MW, as opposed to 40 503 MW in 2009 and up from an estimated 38 747 MW in 2008. It was, however, noted in the 2010 Annual Report that demand had returned to the 2007 level by March 2010. However, it was also noted that if energy efficiency measures were not put in place by the winter of 2010, the power system would be under pressure beyond the winter of 2010 and in 2011 the risk of interruption would increase.

Eskom has improved its reserve margin since 2008 when it was 5.6 to 10.6% in 2009 and 16.4% in 2010. The reserve margin is the difference between the net system capability and the maximum load requirement. The 2009 margin is in line with internationally required margins which are generally in the region of 10 to 15% and the 2010 margin is in excess of this requirement.

This margin was in part achieved as a result of the small growth in sales that Eskom achieved in 2008, namely 2.9% (4.9% was forecast), and the negative growth in sales it experienced in 2009, namely -4.3%, and again a small growth in 2010 of 1.3% (Eskom, 2008, 2009, 2010). Nevertheless, Eskom has embarked on a substantial programme to ensure that the load shedding experienced in 2007 and 2008 does not happen again. In addition, the future needs of the country need to be taken into account.

To provide an example of the state of South Africa's electrical generating capacity, a comparison is made between the growth in South Africa's Gross Domestic Product (GDP) and the increase in available electricity over the same period. These figures are shown in Figure 1-3. It should be noted that the period used for this comparison is from 2003 to 2010. During this period the cumulative growth of the GDP was above 32%, while the increase in the country's electrical generating capacity was 15.6%. This means that South Africa is not increasing its electrical supply capacity concomitantly with the rate at which it is growing.

1.3.1 Increases in Energy Costs

In December 2007, the National Energy Regulator of South Africa (NERSA) awarded Eskom a revised price increase of 14,2% for the 2008/2009 evaluation period ending in March 2009. Although this increase was well in excess of the country's CPI, it was well below the 60% that Eskom requested. This request by Eskom resulted in the urgent call for Nedlac to convene a 'national energy summit'. The meeting was in response to a call by the African National Congress (ANC) and other stakeholders for further consultation and explanation of the request by Eskom for the tariff increase. The summit raised the point that electricity tariffs should ensure the sustainable development of industry, but that it must avoid imposing unacceptable costs to the poor and an excessive shock to the economy.

In response to this request, NERSA announced on 18 June 2008 an additional increase in the electricity tariff of 13,3% for the year ending March 2009 which resulted in a 27.5% average increase year-on-year. NERSA also ruled that the price increase to 'poor' residential customers should be limited to 14,2%.

In the following year, Eskom also requested an increase of 34% for the 2009/2010 period. NERSA granted a tariff increase of 31.3% for the last nine-month portion of this period (ending on 31 March 2010). This increase did, however, include a new levy from the government on generating electricity from non-renewable resources. Thus, the 31.3% increase resulted in a real increase of 24.1%, which was again substantially below the requested increase.

Once again in 2010, Eskom requested an increase in tariffs of 35%. This was again refused by NERSA, which allowed an annual increase of only 24.8%.

The future increases determined by NERSA were defined in its 2010 Annual Report (NERSA, 2010), as follows:

1 Year 2010–2011: 24.8%

2 Year 2011–2012: 25.8%

3 Year 2012–2013: 25.9%

1.3.1 General Discussion

As the requirement for additional energy in South Africa drives the development of additional sources of electrical energy, the cost of this energy will increase. Eskom has already requested increases in the amount it charges for electricity well above inflation, primarily to allow it to source the capital required for the building of additional power stations. To demonstrate the effect that these increases could have on the operating costs of a mine, a number of alternatives were evaluated. The results of this analysis are shown in Figure 1-4. The baseline cost of 10% of the monthly production costs was increased by the indicated amount as indicative of the energy costs. The production costs themselves were increased by an estimated 6% to account for inflation. The overall increase in the percentage of the monthly costs was calculated, and this is the percentage increase shown in the graph.

The best-case scenario shows that the energy costs will increase to become a total of 20% of the monthly cost by 2015; the worst case shows this to be 40% of the monthly cost by 2015.

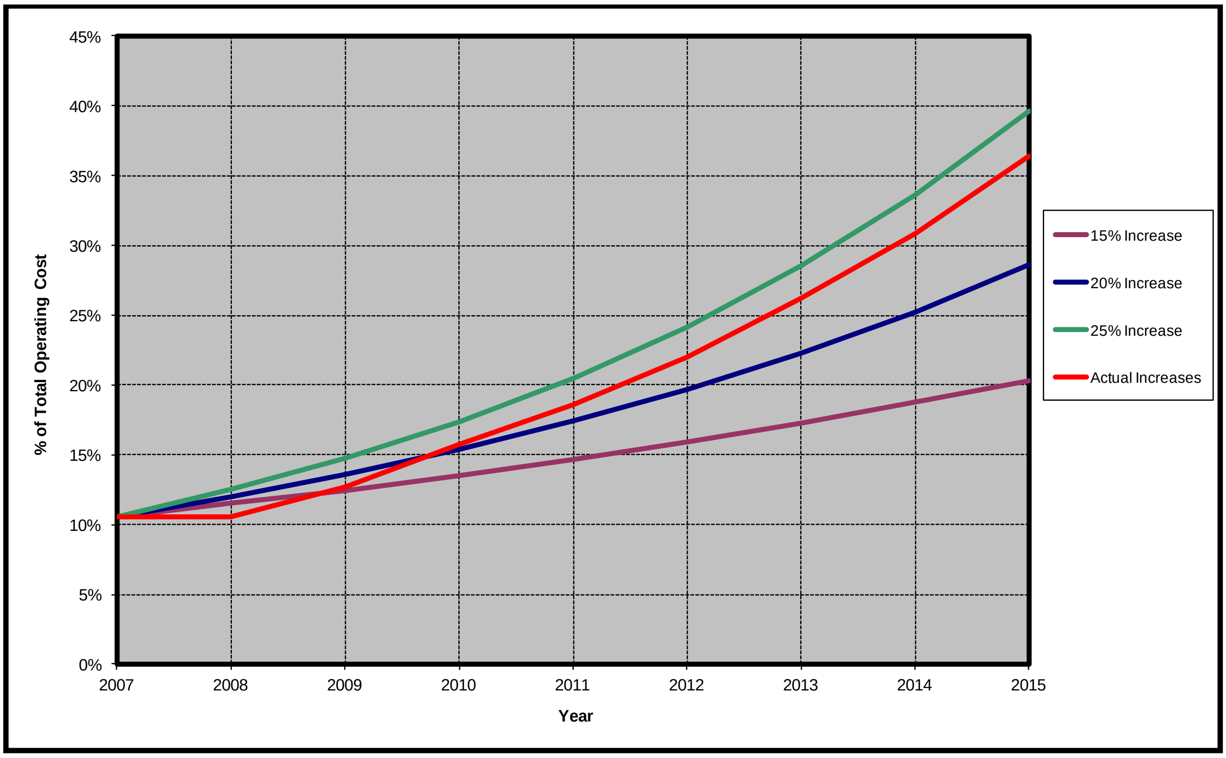

Figure 1-4: Consequence of increases with respect to operating costs

In order to ensure that they are able to operate effectively in this environment of increasing energy costs and reduced energy availability, the mining industry must optimise the manner in which it uses the available energy. This evaluation and optimisation provide the primary thrust of the work presented in this thesis and therefore the data presented in the graph are quantified more fully as the work progresses.

1.4 EVALUATION OF ENERGY CONSUMPTION IN DEEP-LEVEL MINES

The purpose of the work presented in this section is to quantify the actual electrical energy used in a typical mine and to highlight the largest consumers. This analysis will allow us to determine the area of the mine that will benefit most from additional analysis.

1.4.1 General Parameters

To ensure a meaningful analysis of the electrical energy consumed by a typical deep-level mine, the basic parameters of the study must be quantified. In this instance only underground deep-level mining operations are considered, i.e. mines that are serviced using vertical shafts and declines as part of an access strategy.

1.4.2 Initial Evaluation

To evaluate the electrical energy consumption of deep-level mines, the various primary electrical energy consumers in the mine must be understood. Calculations were done for the design of an underground platinum mining operation. The results of these calculations show the installed electrical energy consumption capacity for the various areas in a standard underground mining operation. These results are shown in Figure 1-5. The ‘model’ mine was designed with a shaft depth of approximately 1 000 m and a decline system originating at the bottom of the shaft was used to access the ore body. Figure 1-5 shows the total use energy calculation for each of the appropriate sections, Figure 1-6 shows these figures as a percentage of the total expected energy use. The general parameters for the mine analysed are:

i	Total design tonnage	:	± 185 kilotonnes per month (ktpm)
ii	Main shaft diameter	:	8.5 m
iii	Main shaft depth	:	1 048 m
iv	Ventilation shaft diameter	:	6.5 m
v	Anticipated airflow	:	± 650 kg/s
vi	Mining method	:	Scattered breast
vii	Panel length	:	30 m

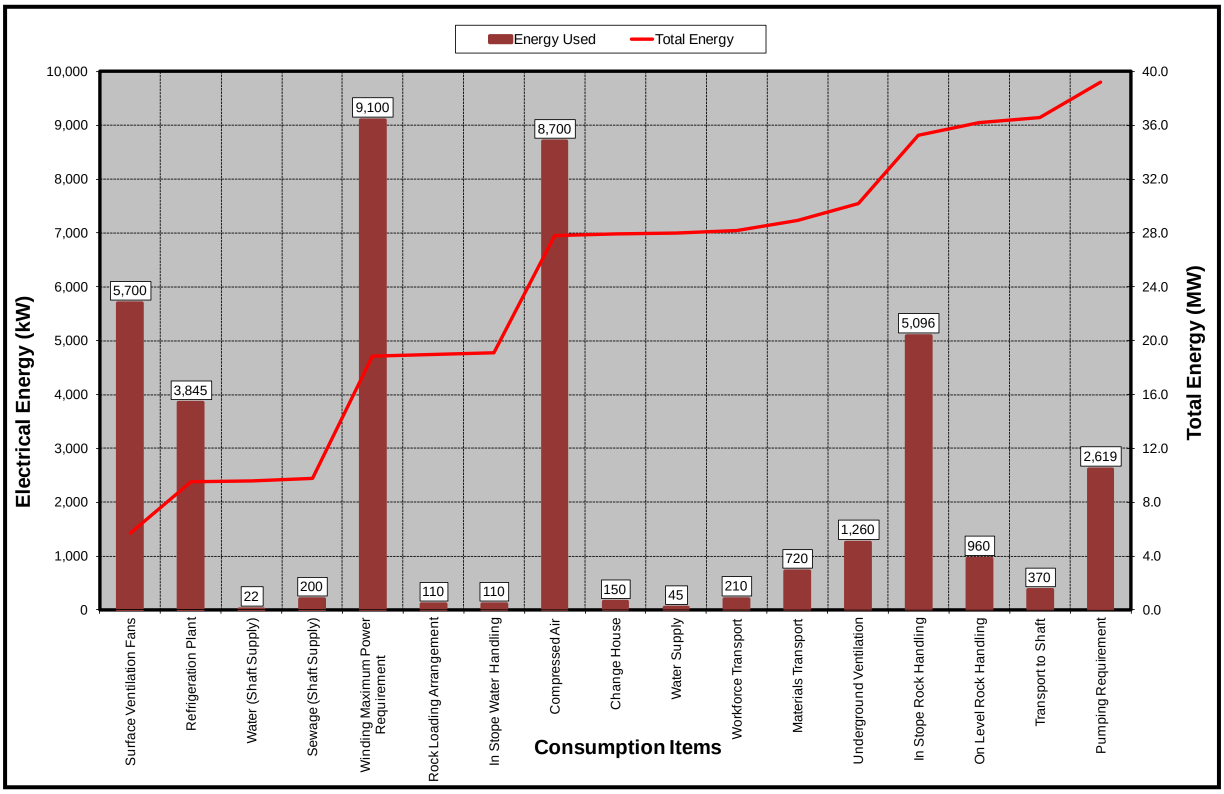

Figure 1-5: Summary of mine installed electrical energy consumption requirements

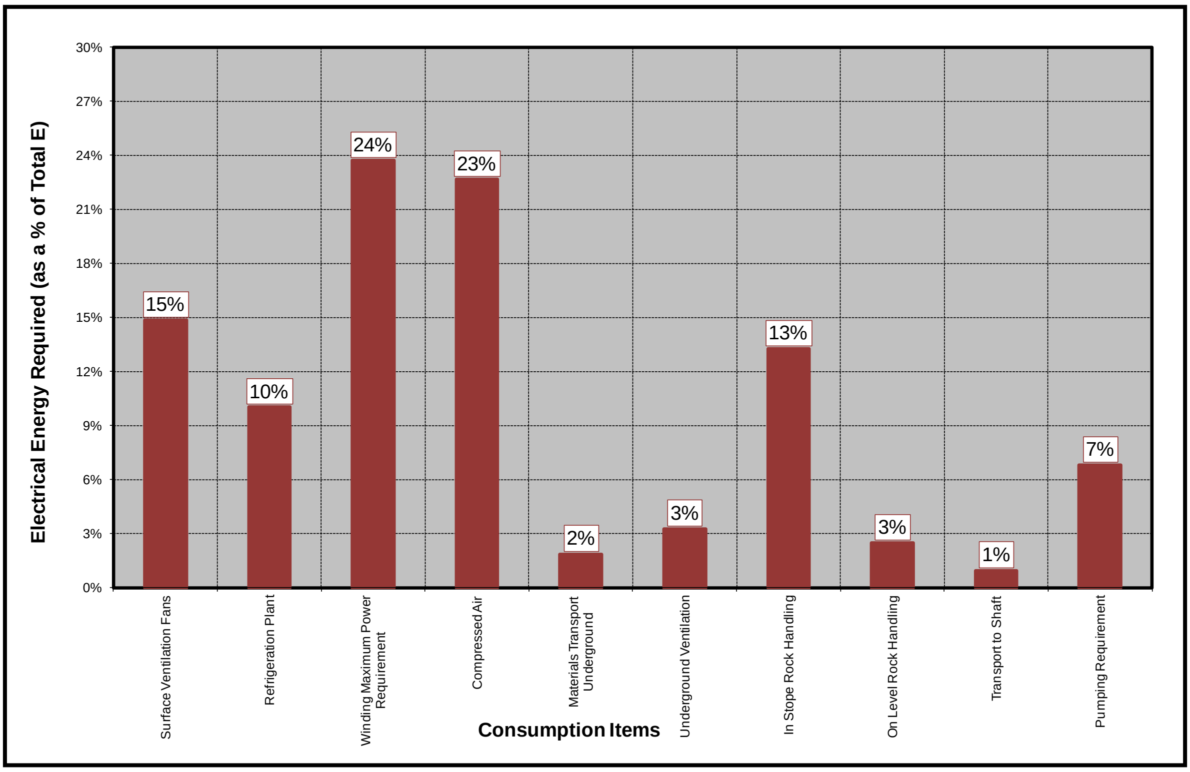

Figure 1-6: Summary of mine installed electrical energy consumption requirements (as % total installed consumption capacity)

The results of this analysis are useful in that they allow us to see explicitly which of the electrical energy users in the mine potentially consume the most energy. These primary users and their total contribution to the energy consumed are listed below:

1 Winding maximum power requirement (24%)

2 Compressed air (23%)

3 Surface ventilation fans (15%)

4 In-stope rock handling (13%)

5 Refrigeration plant (10%)

6 Pumping requirements (7%)

These are the users that consume up to 92% of the total energy required for effective operation of the mine. It must be borne in mind during this analysis, that the total installed capacity does not necessarily mean that a particular item consumes the most electrical energy. Comment is added in the various sections discussed below as to whether this is the peak or the average installed electrical consumption capacity

An overview of the winding requirement, compressed air requirement, in-stope rock handling and pumping requirements is presented here. The remaining ventilation and refrigeration requirements provide the primary thrust for this research and are discussed in more detail in Section 1.4.3.

1.4.2.1 Winding maximum power requirement

The winding requirement of the shaft is the highest peak consumer of electrical energy for the shaft system analysed. It consumes 24% of the total energy requirement, a total of 9.1 MW. The motor size and therefore peak electrical power required for winding is based on the power needed to accelerate the mass to the travelling velocity and cooling requirement of the motor. The motor is therefore sized to accommodate the peak power requirement and does not represent a steady-state consumption. The actual electrical power requirement is considerably less, especially when it is considered that winding systems are generally balanced, either with a counterweight or an empty skip counterbalancing the transporting cage or skip respectively.

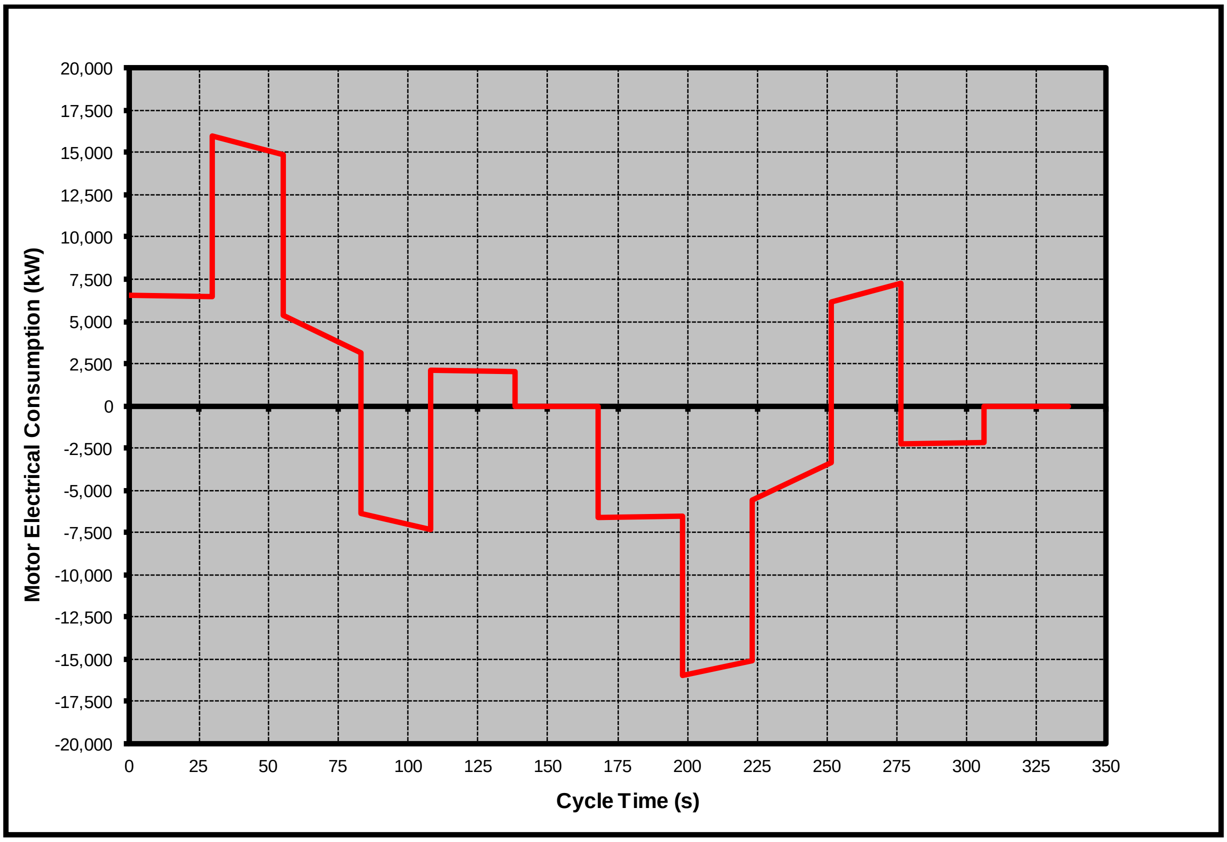

Figure 1-7: Typical winder cycle

Figure 1-7 represents a typical winder cycle. The total electrical power required peaks as the load is accelerated and quickly drops as the conveyance reaches its constant transport velocity. This requirement is then reversed for the deceleration portion of the wind and the cycle is then repeated.

The working of winders and the associated men and material transport is well understood and are therefore not considered in detail here. It should, however, be noted that there could be a small decrease in the total electrical energy consumed here as a result of changes to the shaft and cage configuration; these are included in the final analysis.

1.4.2.2 Compressed air

One of the primary consumers of electrical energy on the mine is the compressed air required for powering the rockdrills. In a conventional compressed air system as was used for the analysis depicted above, the compressed air generation equipment accounts for a total of 23% of the total energy requirement. This amounts to a total of 8.7 MW (Figure 1-6). The use of these rockdrills is generally cyclical and therefore the electrical energy is not consumed 24 hours a day. However these rockdrills are generally required to be used during the "peak" electrical supply period and as such are expensive to operate. In addition, a certain "base load" of compressed air is required to maintain the compressed air pressure in the system when the rockdrills are not being use.

There are a number of alternatives to the pneumatic rockdrills considered above, specifically hydraulic drills using high-pressure water (hydropower), as well as electric rockdrills. All of these systems have varying efficiencies and operational flexibilities. They have been examined in detail in other studies and are not considered further in this work.

Additional information on hydropower and electric rockdrill systems can be obtained in the papers by Wills (2008) and Petit (2006).

The various drilling systems and their advantages, disadvantages and efficiencies are well understood and therefore do not form part of the scope of the work reported in this thesis.

1.4.2.3 In-stope rock handling

This refers to the transport of the broken rock at the panels in the stopes to the box holes such that it is ready to be taken to the central ore passes before it is removed from the mine. The technique used in the analysis above is based on using face scrapers in the face and scrapers in the raise line to transfer the broken rock to the box holes. The calculation from the example shows a use of 13% of the total electrical energy or 5.0 MW (Figure 1-6). None of these scrapers requires a significant quantity of electrical energy. However, the mine-wide face-cleaning requirement results in a number of these systems being used in parallel for the cleaning shift of the mine, leading to a significant cumulative use of electrical energy.

There are a number of alternatives to traditional face scraping as used in the example above, primarily using water jet-assisted techniques (i.e. another application for hydropower). Additional information on hydropower equipment alternatives for the stopes can be obtained in the paper by Du Plessis et al. (1989).

The various stoping equipment suites and the techniques used for cleaning the panels and stopes are well understood and have been researched and reported on in numerous other papers. They therefore do not form part of the scope of the work reported in this thesis.

1.4.2.4 Pumping requirement

The pumping requirement for various mines can differ significantly, depending on the fissure water within the mine and on the mining method. The analysis completed in the example used for the initial evaluation came from a mine with very little fissure water. There are some mines that have a significant quantity of fissure water to contend with and the overall electrical energy required to dewater these mines will be larger than the total electrical energy calculated here. This analysis showed that the pumping will consume approximately 6% of the mine's total electrical energy requirement, or 2.6 MW during the pumping shift.

There are a number of alternatives to the traditional pumping systems used in the example, primarily using energy-regeneration techniques such as the chamber pump system. Additional information on more energy-efficient pump systems can be obtained from the paper by Fraser and Le Roux (2007).

The evaluation, control and removal of water from mines has been well researched and reported on. This topic is therefore well understood and the associated systems have been examined in detail in other studies. It is not considered further in this work.

1.4.3 Evaluation of Ventilation and Refrigeration

The ventilation and refrigeration requirements for a mine must always be considered together. The primary requirement of the ventilation in a stoping arrangement is the provision of adequate air velocities at the face. This quantity of air can, however, be affected by the cooling requirement. It is common practice to cool the air before it is taken into the mine and to use it as the cooling medium underground. This can, however, have an effect on the total air quantity required if the heat requiring dissipation in an area results in higher airflow rates through that area than are required by ventilation concerns alone. There are a number of areas in a typical ventilation analysis that could benefit from a more detailed evaluation. These are discussed in more detail here.

The ventilation component of the analysis used for this comparison is some 15% of the total electrical energy requirement in the mine or 5.7 MW (Figure 1-6). This is required for the main ventilation fans. While it is possible to “turn the fans down” during periods where the mine has fewer personnel underground, it is never recommended that the ventilation flow to the mine be stopped altogether. The refrigeration component of the analysis used for the comparison of the electrical energy consumption in mines is some 10% of the total electrical energy requirement in the mine or 3.8 MW (Figure 1-6). Together these two considerations total 25% of the total electrical energy consumed. The objective of this discussion is to examine these areas and determine whether any of these areas could benefit from additional investigation. It must be noted again here that the refrigeration requirement is not considered directly in this report but, as discussed above, must be considered in conjunction with the ventilation of the mine. The refrigeration requirement will therefore be affected by changes in the efficiency with which the ventilation air can be delivered to the mine’s workings.

A number of areas have been investigated with respect to the demand side management (DSM) of ventilation and cooling systems. These are discussed in more detail in the next section. It must be noted though that DSM differs from energy efficiency in that it deals with the use of the currently installed systems to specifically reduce the cost of operating them through the optimisation of the different times of use. The emphasis of the work reported here is on the reduction of the total electrical energy requirement of the mine at the design stage, thus increasing the overall energy efficiency of the system while it is being constructed.

1.4.3.1 Demand side management

Marx et al. (2008) deal with the DSM of ventilation and cooling systems on mines. In this paper they evaluate two areas, namely fan control and thermal storage and present a summary of the DSM projects currently under way in South Africa.

The fan power required to move air through a mine is proportional to the cube of the volume of flow for that fan. Thus reducing the airflow by, for example, 15% will, in theory, reduce the fan power by approximately 38%. This theoretical power reduction is possible, but in reality there are a number of factors that change and reduce this benefit, e.g. the potentially lower power factor, the motor efficiency and the aerodynamic changes of the fan (Du Plessis and Marx, 2007). This paper reviews the three basic methods for the control of mine ventilation fans, i.e. the outlet dampers, fan speed and inlet guide vanes. In addition they emphasise the need for an efficient and robust communication, monitoring and control infrastructure. These systems are useful for retrofitting on existing mines and indeed have been successfully implemented in some cases.

Thermal storage systems are not new techniques. This system is based on generating ice or cold water at times during the day when the cost of electrical energy is at its cheapest and preserving this cooling capacity until it is required during the mine's busiest period of the day. Deep-level gold mines have used large chilled water dams located on surface to provide thermal storage for mine cooling systems in the past. This technique is also commonly used in air-conditioning systems in North America and Europe (Wilson et al., 2003).

The technologies discussed above, as well as others not discussed here, which can provide significant electrical energy savings in South African mines, are available and their application is well understood. These technologies are based on the efficient use of the systems in place and control of the ventilation and cooling systems such that the times when the electrical energy is used is optimised with respect to the overall load on South Africa's power grid, and that the load is timed to limit the overall consumption during periods of peak electrical demand.

1.4.3.2 General remarks

In addition to the generation and optimisation systems described in the preceding sections, there is also the question of the distribution of the required cooling capacity. The usual technique is to cool the air used for the ventilation of the mine and to distribute it throughout the mine. This method has the advantage that the air that is required to provide a safe working mine can also be used for cooling. However, in deep-level mines, the driving factor is not the distribution of sufficient air, but rather the distribution of the cooling medium. This can result in an increase in the air quantities being distributed to the workings as a result of the need to distribute cooling, rather than meeting the actual ventilation requirement, with a commensurate increase in the power required for the distribution of this ventilation and cooling.

It should also be noted at this point that although there is a drive to use the concept of 'cooling power' as a more adequate indication of the working environment's ability to provide the necessary cooling capacity, this has not been fully accepted as yet (Biffi et al., 2006). The adoption of this standard could result in the reduction of the overall quantity of air being required.

In some instances where significant cooling is required, it becomes inefficient and, in specific cases, impossible to transport the required cooling using the ventilation air. The "process of gravitational compression, or autocompression in downcast shafts produces an increase in the temperature of the air. This is independent of any frictional effects and will be superimposed upon the influence of any heat transfer with the surrounding strata that may occur across the shaft walls." (McPherson, 1993). If the shaft is sufficiently deep, this can result in the air becoming a heat source instead of a cooling medium.

These factors must be considered in association with the general ventilation and cooling requirement of the mine. It was noted by Biffi et al. (2006) that one of the most important methods available to reduce the energy consumption of a mine is the integration of the ventilation system design as a fully fledged component of the overall mine design.

The importance of integrating the ventilation design with the mine design and the design of the various mine components cannot be over emphasised. The technique of designing mines in modular form, with extensions being managed as additions to these modules, is also important. This integration must also be seen in the context of the drive to make mines more operationally flexible. If this flexibility is met with increasingly costly ventilation and cooling costs, the benefit of having a flexible system will be offset by these costs.

This mine flexibility requirement can come in two forms. In the first instance it is the operating flexibility to deal with changes in the economy and the price being offered on the market for the commodity. Macfarlane (2005) emphasises the importance of flexibility in the mine plan by arguing that "where flexibility to deal with changing economic cycles has not been created, reactive planning has to be undertaken, which is value destroying".

In the second instance, if operating flexibility is regarded as the ability to move the mining operation swiftly to different production faces when issues of grade control or unpredicted geological structures require it, then the mine's resource can be managed effectively to allow it to control the production of its resource while ensuring that the maximum portion of ore reserve is exploited

(Musingwini et al., 2006).

1.4.4 Calculation and Verification Data

To highlight the areas of the ventilation system that require the most electrical energy, the example mine discussed in Section 1.4.2 needs to be shown in more detail. A schematic of the arrangement is shown in Figure 1-8. The refrigeration requirement was not analysed separately as in this example the cooling capacity required by the mine is distributed using the bulk air cooler on surface to chill the air. No additional cooling was required. The quantities of air noted in Section 1.4.2 (page 9) of this thesis provide the cooling requirement needed by the mine.

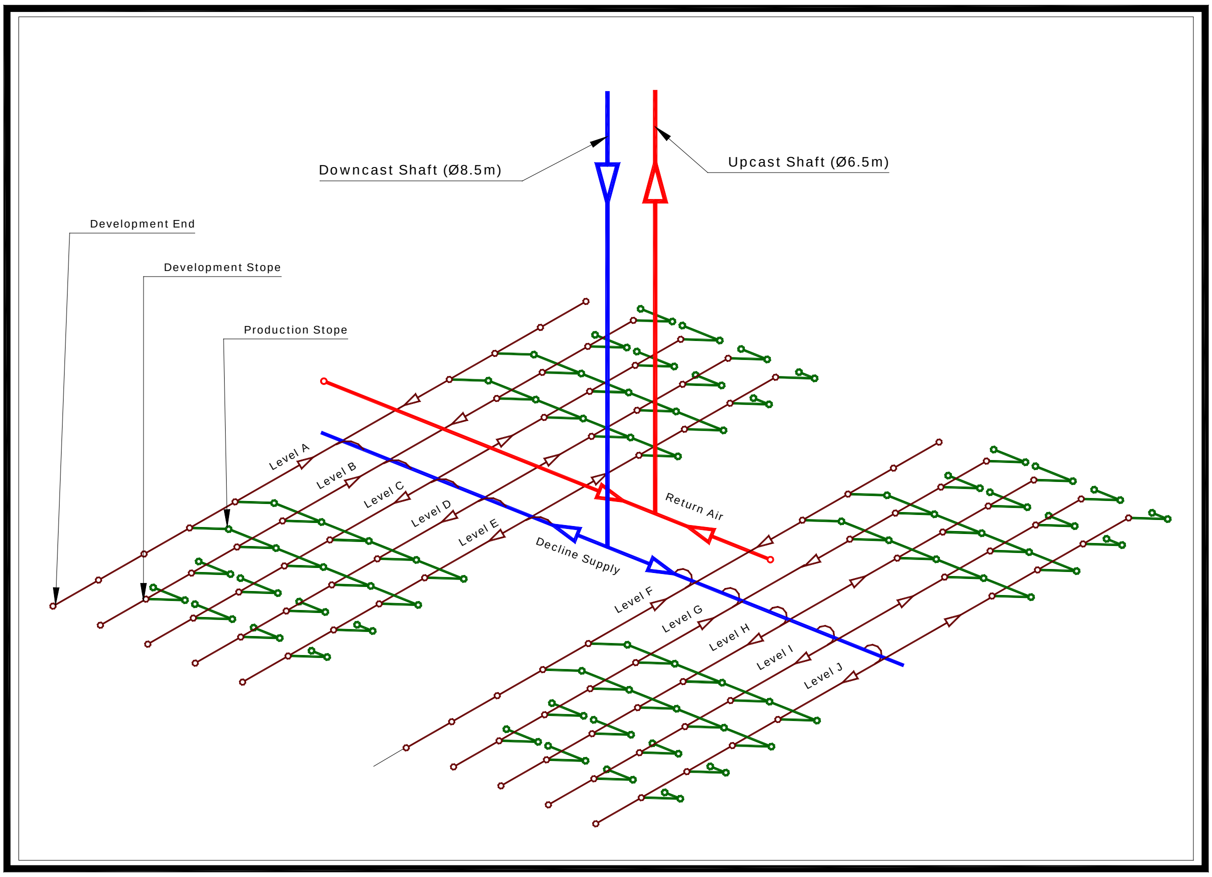

Figure 1-8: Schematic of ventilation layout

The electrical energy required to move air around the mine is based on the quantity of air in a section as well as the pressure required to move this air to that section. In this regard the main ventilation airways and the resistance to the airflow quantities needed must be carefully considered.

For the 'model' mine under examination here, the sections shown in Table 1-1 have been identified and shown in more detail to highlight the areas which require the most energy to move the required air through them. The details of this calculation can be found in Appendix A.

Table 1-1: Summary of ventilation power requirements for mine analysed

Item	Description	ΔP (Pa)		Q (kg/s)	Power (kW)	
1	Primary intake configuration (downcast)	1,793	35%	703	1,261	54%
2	Main intake systems	645	13%	150	97	4%
3	Haulage intakes	634	12%	70	44	2%
4	Stope configuration	90	2%	70	6	0%
5	Stope return configuration	379	7%	70	26	1%
6	Main return airway	656	13%	340	223	10%
7	Return shaft (exhaust)	951	18%	703	669	29%
Note: The flows and changes in pressure shown here are of the of mine ventilation system and are a selection of the worst-case scenarios in certain areas to allow evaluation of the efficiency of the overall system and to provide a guide as to the areas that will benefit most from an optimisation exercise.						

It follows from this analysis that the two areas resulting in the high pressure changes and therefore requiring the most power and from the main ventilation fans during operation are the downcast and upcast shafts. These two areas consume 83% of the total power supplied by the fans in this configuration. This power requirement translates directly to an increased requirement for electrical energy.

This high percentage indicates that this could be an area of worthy of additional analysis. To achieve this, the ventilation manager at Impala Platinum Mines was approached. The specific request to this department was to allow access to a shaft to do measurements that would confirm the high pressure drops in the shaft and potentially to optimise the shaft design to optimise these pressure drops, resulting in an overall reduction in the energy required to operate mines.

1.4.4.1 Verification of the shaft pressure losses identified

Impala Platinum Mines Management was approached with the request to allow testing on their various shafts in their mining group, primarily as there are a number of shafts available for testing with easy reach of each other.

Impala Platinum Mines agreed to allow the initial shaft pressure loss tests to be conducted on one of their shafts. Impala No. 14 shaft was chosen as this shaft consists of two downcast and two upcast shafts, all of which were accessible. A schematic of the shaft configuration is shown in Figure 1-9.

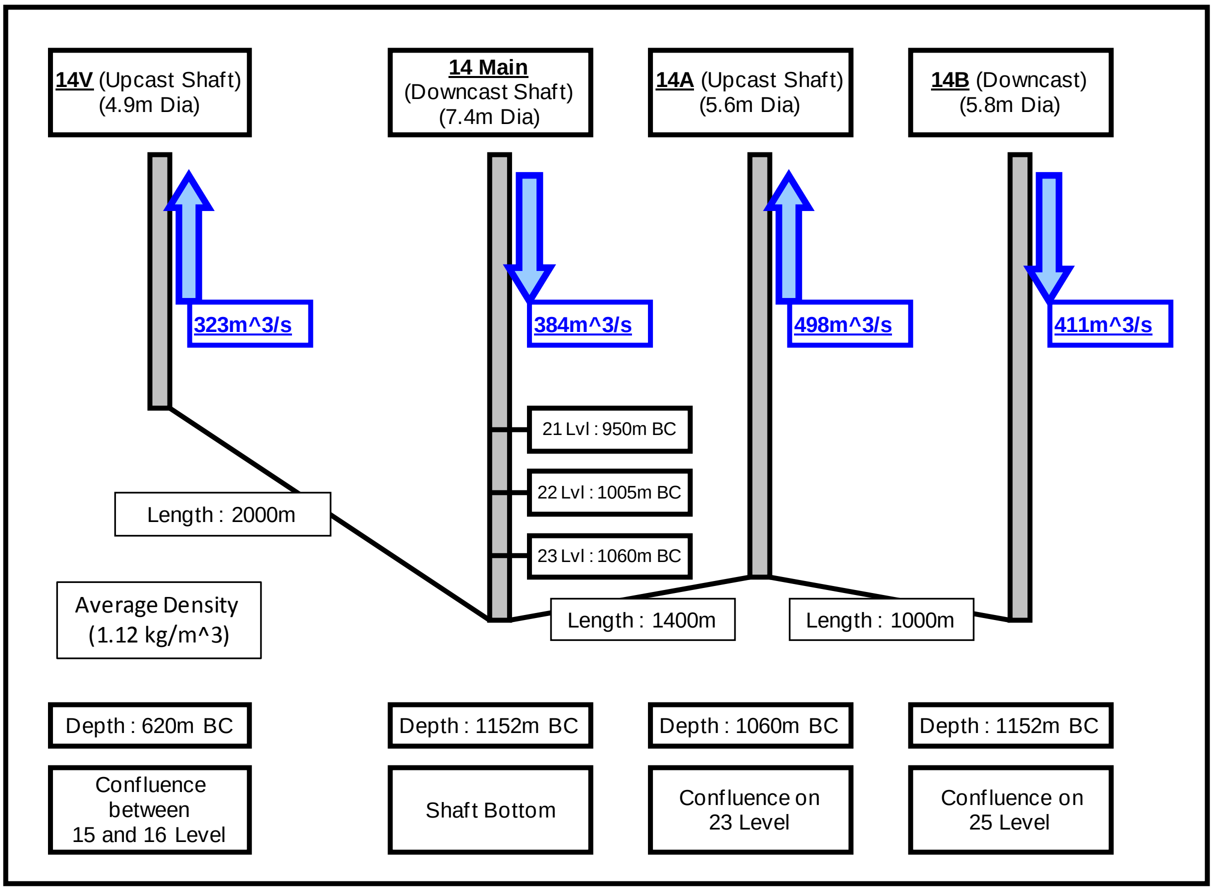

Figure 1-9: Schematic of mine shafts tested

It should be noted from the schematic depicted in Figure 1-9 that there is a discrepancy between the total downcast quantity of air and the total upcast quantity of air measured (795 m^3/s as opposed to 843 m^3/s, i.e. some 6% difference). This difference is attributed to the experimental error primarily as a result of the complex shapes shown at the bottom of the upcast shafts. In addition, the mine does experience some additional ventilation inflow from neighbouring shafts, the exact quantity of which is not known. The details of these data and the subsequent calculation can be found in Appendix B of this thesis.

The initial tests were designed to ascertain, on a broad perspective, the pressure losses in the shaft systems as a whole. In the equipped downcast shaft, it was possible to take measurements at various points in the shaft. This was not possible for the unequipped ventilation shafts, so on these shafts measurements were taken as close to the shaft as possible try to limit the interference from shock entrance and exit losses. The indirect pressure survey method used for these calculations is the barometric survey method laid out in the Mine Ventilation Society of South Africa's Handbook (1989). Sufficient data were required to allow the pressure drop of the shaft to be calculated in accordance with this method. In this regard the following measurements were taken:

1 Wet bulb temperature

2 Dry bulb temperature

3 Static pressure

4 Velocity pressure (or dynamic pressure)

The specifics of the test methodology can be found in Appendix B. The results of this analysis are listed in Table 1-2.

Table 1-2: Summary of ventilation pressure losses

Impala No. 14 shaft (downcast) (384 m^3/s)		
918 Pa	Measured	19% Difference
748 Pa	Calculated	
Impala No. 14V shaft (upcast) (323 m^3/s)		
1 302 Pa	Measured	31% Difference
902 Pa	Calculated	
Impala No. 14A shaft (upcast) (520 m^3/s)		
694 Pa	Measured	7% Difference
642 Pa	Calculated	
Impala 14B No. shaft (downcast) (411 m^3/s)		
1 021 Pa	Measured	15% Difference
1 175 Pa	Calculated	

Total pressure for fan 1 (14A Upcast Shaft)	
4 088	Pa
Total pressure for fan 2 (14V Upcast Shaft)	
3 202	Pa

54% of the total fan pressure available from the main fans is used to overcome the pressure drop experienced as a result of air moving through the shafts.

Notes:

i The percentage difference noted in the measurements of the No. 14 Downcast and the No. 14A upcast shaft are consistent with the percentage difference noted in the overall quantity of air measured in the shaft and presented in Figure 1-9.

ii The percentage difference noted in the measurements of the No. 14V upcast and the No. 14C downcast shafts is significantly higher. This increase in the experimental error is a result of the increasingly complex ventilation distribution shape at the bottom of these shafts. In addition, the mine has an unknown quantity of air being entrained from neighbouring shafts.

iii The detailed calculation and schematics used to calculated the above figures can be found in Appendix A.

The calculations from the initial shaft configuration are borne out by the measurements. These indicate that the pressure losses in both of the vertical shaft complexes absorb most of the pressure supplied by the ventilation fans.

A comparison was also made between the measurements taken and the current theory available. These showed some correlation in some areas. It should be noted, though, that when the shaft configuration became more complicated (i.e. when station take-off occurred or multiple entry ways were noted in the ventilation configuration), the correlation between the measured and calculated data was not in close agreement. This is particularly apparent in the No. 14B downcast shaft configuration shown in Figure 1-10. To obtain the final flow through this shaft, the velocity in each of the four openings was measured and the total quantity of air that would be entrained in this shaft was summed. The normal experimental error in this configuration would thus also be summed. This was mitigated by ensuring that the testing was done in strict accordance with good experimental procedure.

The real-life complexity of the systems being measured and the associated differences between the calculated data and the measured data can give rise to discrepancies. However, another reason for these discrepancies could be the inadequate nature of the basic calculations used to verify the measured data. Although this is undoubtedly not true for the classically derived formulae (i.e. the pipe friction flow model), there is perhaps room for improvement in the empirically derived data, particularly for the calculation of 'shock' losses and losses associated with partially obstructed airways.

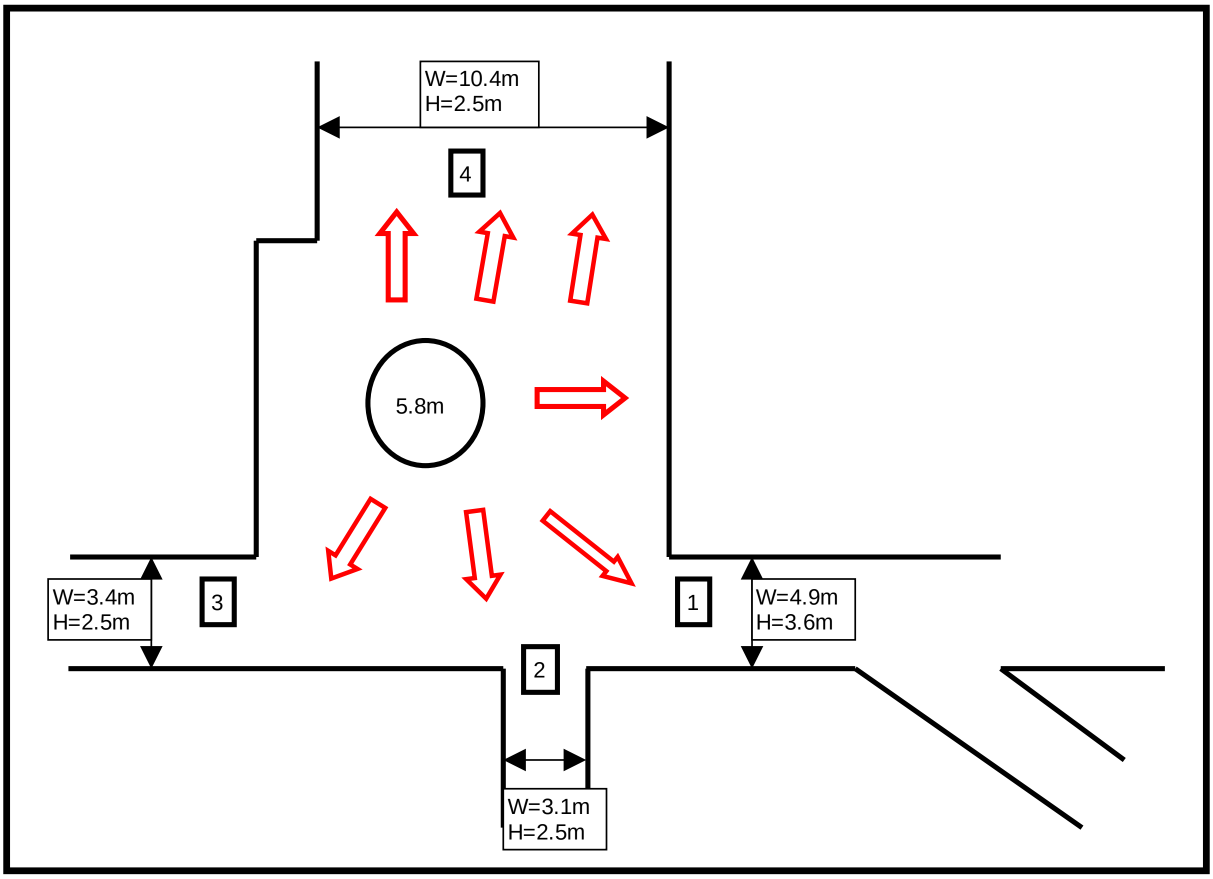

Figure 1-10: Configuration of No. 14B downcast shaft bottom

Despite the above errors, the quality of the data collected and the subsequent analysis allows us to draw some conclusions. These are:

1 Ventilation fans must supply a significant amount of energy to the ventilation air to overcome the frictional resistance in shaft systems.

2 The theory associated with the prediction of these pressure losses potentially requires additional investigation.

The primary reason for this recommendation is that the relationship between the power required by a fan (and therefore the electrical energy) to move air against a resistance which would result in pressure loss is directly proportional. This is governed by the relationship of the power required by a fan equals the product of the pressure required and the flow rate (if compressibility effects are ignored).

It should also be noted that the measurements discussed here were taken in an equipped shaft from a stationary conveyance. The shaft resistance and therefore the pressure drop will increase if the main cage or any of the cages and/or skips in the shaft are moving against the airflow of the shaft.

1.5 PROJECT BACKGROUND

This section describes the process undertaken to complete a shaft design, the process of costing these shafts and the difficulties encountered in this costing. Finally, the status of the current theory used for the analysis of shaft systems is evaluated and potential areas of investigation highlighted.

1.5.1 Evaluation of Costs Associated with the Sinking, Equipping and Operation of Shafts

To evaluate the potential costs and savings from modifications to the shaft systems, it was necessary to identify the parameters of the resistance within the shaft systems that could influence the pressure losses associated with it. It must be noted that while this analysis is normally done in conjunction with the overall mine, in this instance our interest is specifically in the shaft itself.

As the size of an airway increases, its resistance, and therefore operating costs, will decrease for a given airflow. However, the capital costs of excavating the airway increase with its size. The total combination of capital and operating costs is the total cost of owning and ventilating the airway. The most economical or optimum size of the airway and its associated fitting occurs when this total cost is a minimum (McPherson, 1993).

Determining accurate costs for the sinking of shafts is difficult when they are used as a basis for comparison. However, for this exercise, to provide a consistent basis for comparison, it has been assumed that the shaft-sinking company will supply all the temporary works and will sink the shaft. An example of the increasing costs associated with sinking shafts of various diameters is shown in Figure 1-11.

Generally, it is noted that a significant long-term reduction in costs will be required in order to justify the immediate increase in the capital costs. The increases in energy costs noted in Section 1.3.1 make the potential savings more significant and, potentially, where the economic saving would not have been significant in the recent past, these savings could now provide the basis for re-evaluation of the shaft-sinking costs.

A financial model was introduced by Barenbrug (1961) to evaluate the effectiveness of changes in shaft design. This model is still used today in various guises and was used by McPherson (1993) to demonstrate the evaluation required to calculate the total cost for a shaft. An example of this model is shown in Figure 1-12.

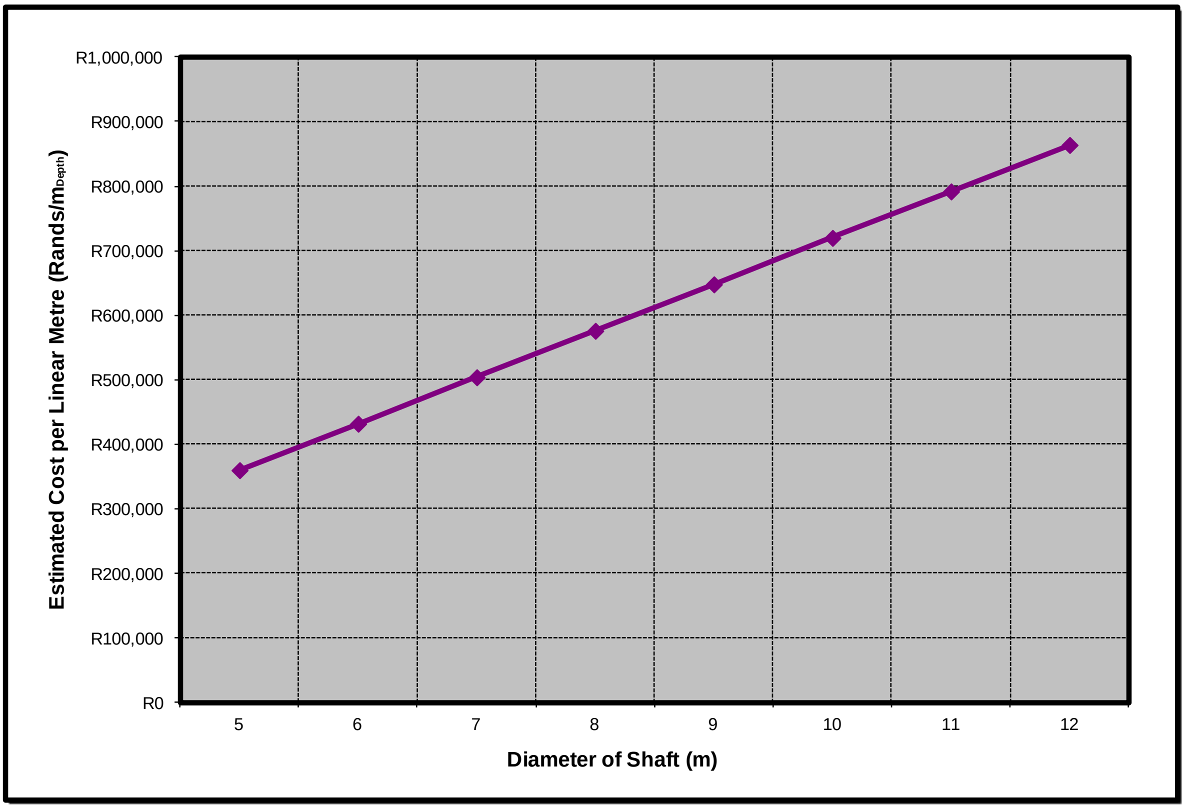

Figure 1-11: **Estimated cost of shaft sinking per linear metre**

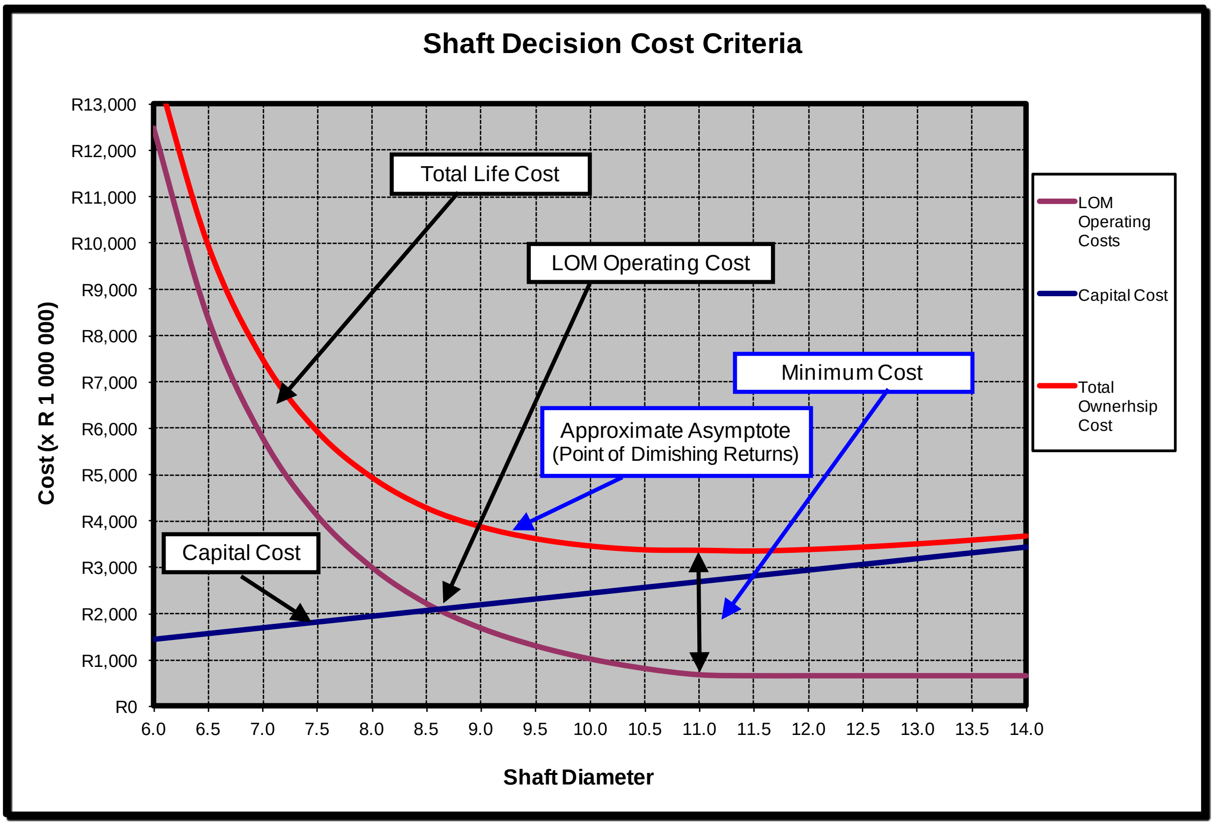

Figure 1-12: **Shaft decision cost criteria**

Figure 1-11 and Figure 1-12 provide the basis for the evaluation of the shaft diameter with respect to operating costs. These parameters are well understood and used extensively. The emphasis of the work reported on this in this thesis will therefore be on comparing the overall lifecycle cost of different equipping options and the associated capital costs in order to elicit a total cost for the life of mine (LOM). This decision was made when discussions with sinking professionals highlighted that fact that the costs for equipping and maintaining the various shaft steelwork options are sufficiently similar to make delineation very difficult.

The comparison will be based on a period of 20 years at a lower-than-anticipated electrical tariff increase of 12% per annum. In order to derive an accurate cost, the cash flows should also be discounted such that the time value of money is also taken in to account. This allows a net present value to be determined which provides a realistic cost to compare the various savings. All future savings were discounted by 7% per annum.

When evaluating the cost effectiveness of the different options, the entire capital cost of the shaft was not considered. Only the areas affected by the changes were evaluated, these included the operating costs for the fans (but no capital costs which could accrue as a result of the reduced pressure requirement). The costs for the buntons and guides were included, where appropriate, as well as other equipment associated directly with the shaft, as appropriate.

1.5.2 Components Contributing to the Present Shaft Resistance and Subsequent Pressure Losses

This section gives the background to the specific area of interest and provides some insight into the status of work being conducted in this area of research.

1.5.2.1 Parameters used for designing a shaft

To understand the parameters contributing to the shaft resistance and the subsequent pressure losses, we will first broadly define the steps (McPherson, 1993) that are generally taken to design a typical equipped downcast shaft from the perspective of the ventilation requirement of the mine:

1 Determine the duties required for

   i rock hoisting

   ii personnel transport

   iii mass, dimensions and frequency of materials and equipment

2 Analyse the above with respect to the conveyance sizes and required hoisting speeds.

3 Conduct the ventilation analysis:

   i The primary requirement for ventilation will be based on the mine design, the layout and the distribution of the workings over the LOM.

   ii Standard factors will be used for estimating the shaft resistance and other significant contributors to the overall pressure drop in the mine.

4 Assess the dimensions of proposed shaft fittings. This will include:

   i Shaft fittings (i.e. buntons and guides)

   ii Services (i.e. pipes, cables, etc.)

5 Conduct an optimisation exercise to find the size of shaft that will pass the required airflow at the minimum combination of operating costs and capital expense.

6 Review the free air velocity of the ventilation requirement to ensure that it is within accepted limits. (Generally, the velocity in an equipped downcast shaft should be approximately 10 m/s and that in the unequipped upcast shaft should not exceed 20 m/s.) (The free air velocity is calculated by taking the volumetric flow rate of the air and dividing it by the area of the shaft available to allow the air to flow (ie the total shaft area minus

the area of the pipes, bunton and other shaft fitting (excluding the conveyances)).

7 Review the coefficient of fill of the conveyances to ensure that they are within accepted limits. (If this exceeds approximately 30% for two or more conveyances or 50% for a single conveyance, then the dimension of the conveyance should be reviewed.)

8 Calculate the maximum relative velocity between the airflow and the largest conveyance. (If this exceeds approximately 30 m/s, then additional precautions should be taken to improve the stability of the conveyance.) This relative velocity should not exceed 50 m/s.

9 Assess the air velocities at all the loading/unloading stations.

10 Determine the total resistance of the shaft using ventilation network analysis techniques.

11 Complete the overall ventilation network analysis with the established shaft resistance in order to determine the final fan pressures required.

The specific area of interest is the design of the shaft system itself and the effect that the equipment within the shaft has on the overall resistance offered to the ventilation air flowing through it.

1.5.2.2 Shaft configuration and analysis techniques

Mine shafts, specifically downcast shafts, that are used as primary ventilation routes are different in their airflow characteristics from other subsurface openings because of the aerodynamic affects of the various shaft fittings, i.e.

i guide rails

ii buntons

iii pipes, cables and ropes

iv conveyances (shape, size and velocity)

McPherson (1987) noted that shafts contribute heavily to the overall resistance of a mine ventilation circuit. This was borne out by the measurements discussed in the preceding sections.

The most recent theoretical work analysing these particular areas of shaft systems was carried out initially by McPherson (1987). It was based on the work carried out by Bromilov in 1960. In turn, both of these papers relied heavily on work carried out by Stevenson (1956). These papers will be discussed in more detail in CHAPTER 2. Suffice it to note here that a large portion of the analyses presented in this thesis is based on work published in 1956, 1960 and 1987.

The accuracy of these equations was verified by Deen (1991). In this analysis, using the modified technique described by McPherson (1987), he achieved accuracies of 5 to 12% when comparing the overall measured shafts resistances with the resistances calculated theoretically.

In this regard, a survey was undertaken by Wallace and Rogers (1987) to collate the actual installations of a number of shafts. The data from a total of 37 shafts were evaluated and the various factors for each shaft were collated. The details of these results will be discussed more fully in CHAPTER 2. A summary of these findings is as follows:

i The coefficient of fill should not exceed 30% for shafts with two or more conveyances, and an upper limit of 50% is suggested for a single conveyance.

ii A free air velocity of 10 m/s is considered to be acceptable, with a maximum relative velocity of 30 m/s.

iii As shafts increase with depth, the coefficient of fill should decrease.

iv It is recommended that air bypasses or enlarged shaft stations be installed to minimise high airflow resistances.

Evaluation of the overall calculation technique highlights the significant contribution that the buntons make to the overall resistance of the shaft. The data obtained from the Impala shaft indicate that 80.9% of the total resistance factor is a result of the buntons. The remaining 19.1% comes from the roughness of the shaft walls and the contribution of the shaft fittings to reducing the overall free area of the shaft.

The buntons at Impala No. 14 shaft are oval in shape and therefore have a low coefficient of drag, which is used in calculating the resistance offered by the buntons. In spite of this, they still offer a significant resistance when compared with the remainder of the shaft.

Regarding the question of the bunton shape, scale shafts have been tested by a number of researchers and these experimental data have been used in the design of specific shaft systems. Most of this work is also circa 1960, and it was used by Bromilov (1960) to verify the results he obtained from the theoretical analysis. These papers and their results will be discussed in more detail in Chapter 2. Suffice it to note here that the primary purpose of these papers was to analyse the effect on the shaft resistances of circular shafts vs. rectangular shafts, as well as to evaluate the shape of the buntons being installed in shafts. The overall conclusion of all the tests was that the more streamlined the bunton section, the less its contribution to the overall shaft resistance. This work also reviewed shafts that were being built at that time and does not examine what the best shaft configuration would be for a particular shaft system.

1.5.3 Justification of Additional Work

The general analysis of shaft systems and the techniques currently being used have been discussed. The validity of these analysis techniques has been verified both by testing in the field and by research work describing the specific verification of the technique (Bromilov, 1960).

The primary aim of the work discussed here was to determine what can be done to reduce the operating costs in shafts significantly as these systems still consume the largest portion of pressure supplied by the main ventilation fans.

As was noted in Section 1.4.4, the most significant portion of the pressure drop occurs as a result of the buntons themselves. However, the one parameter that was not included in the initial analysis was the movement of the cages themselves. The analysis technique does provide guidance as to how these should be treated and a detailed evaluation of this is provided in CHAPTER 2. It should be noted here that the theory shows that this movement can increase the overall resistance of the shaft by more than 30%, depending on its configuration and the hoisting speed.

The following specific areas were investigated:

1 *Buntons and guides*: This analysis includes evaluation of the buntons' resistance based on the obstruction they place in the path of the airflow. No work has previously been done on optimising the actual configuration of the buntons in their situation when compared with the obstruction offered by the sidewall or the support of shaft services.

2 *Cages*: This analysis includes evaluation of the cages, once again based on the obstruction they place in the path of the airflow. No work has previously been done on the potential for streamlining these conveyances specifically in conjunction with the bunton configuration. In addition, the potential for reducing the resistance by having various conveyances moving at different intervals has not been quantified.

3 *Services (pipes, cables, etc.)*: This analysis includes evaluation of the obstruction offered by these services based on the amount they reduce the overall shaft cross-section. McPherson (1987) also noted that these services may indeed reduce the overall resistance of the shaft based on the obstruction they offer and the resultant potential reduction in the swirl of the ventilation flow. No work has previously been done on the preferred placement of these services and the specific configuration that would optimise this effect.

1.6 PROBLEM STATEMENT AND PURPOSE OF STUDY

The current theory for the definition of shaft resistances allows us to evaluate shafts and to predict the overall resistance they offer to the ventilation air flowing through them. This theory, however, does not allow us to optimise the shaft cross section layout specifically to reduce this resistance by the moving of conveyance travel zones and varying the placement of buntons, guiders and services within the shaft. The deficiencies in the theory are highlighted in the thesis. The purpose of this study is to minimise the shaft resistance by varying the actual layout of the shaft and the placement of the shaft buntons, guides and services.

1.7 OBJECTIVES

Several objectives were set for this study, and these are detailed in the following sections.

1.7.1 Literature Study

A detailed literature review was undertaken with the following specific objectives:

- Identify all the aspects pertaining to the resistance that shaft systems offer to the ventilation air passing through them. This includes the fittings in the shaft, the stations and associated steelwork, as well as the entrance and exit effects experienced by the ventilation air.
- Identify modelling work completed on shaft airway analysis with the specific purpose of understanding and collating the appropriate parameters to ensure that any modelling work undertaken for the current analysis will be accurate.

1.7.2 Evaluation of Current Shaft Configurations

During this phase of the work, the resistances of different shafts were measured. The specific objective of this phase was to ensure that the current shaft design trends are understood, as well as the resistance these configurations offer to the ventilation air flowing past them.

The data obtained from these measurements are subjected to a theoretical analysis based on the current techniques available in order to ensure that the strengths and limitations offered by these techniques are understood and incorporated into the next phase of the work.

The following specific parameters are noted:

1 Shaft

- i Diameter
- ii Lining

2 Shaft fittings

- i Buntons
- ii Guides
- iii Pipes

3 Conveyances

i Type and Number of conveyances

ii Speed of conveyance

1.7.3 Detailed Analysis

During this phase of the work, the results of the evaluation described in Section 1.7.2 were subjected to a theoretical analysis. This analysis consists of two parts:

1 Analysis of the shaft systems evaluated with respect to the current available theory.

2 Analysis of the shaft systems evaluated with the use of a computational fluid dynamics (CFD) model.

1.7.4 Installation, Maintenance and Costing

The output of the detailed analysis provides parameters for the design of shaft systems in a manner that will limit the resistance of the shaft. These design parameters must, however, be evaluated against the following specific requirements:

1 *Installation*: The system must be able to be installed with ease. In this instance preference was given to the completion of upfront work to limit the actual time taken for the installation as far as is practically possible.

2 *Maintenance*: The proposed system must be maintenance friendly. The specific parameter against which this was evaluated is that it must be possible to complete all maintenance work effectively during the weekly shaft inspection shift.

3 *Cost*: The overall cost of any proposed design must be such that the any capital cost requirement is mitigated by the reduction in operating costs. These costs will be evaluated in Section 3.7.

1.7.5 Conclusions

The specific objective of all the preceding work is to define the design requirements and systems of deep-level shaft systems such that these shafts offer as little resistance as possible to the ventilation flow around them.

1.8 METHODOLOGY

1.8.1 Literature Study

A review of all the work undertaken with respect to the analysis of shaft resistances was undertaken. Each of the particular areas discussed in Section 1.7 was subjected to the following procedure:

1. *Assemble information*: Various search tools and library facilities were used to find as much information as possible on each of the subjects in question.
2. *Review information*: All the assembled information was reviewed, analysed and collated with respect to pertinence to the current research and its applicability.
3. *Presentation*: CHAPTER 2 presents the results of the literature study.

1.8.2 Evaluation of Current Shaft Configurations

During this phase of the work as many shafts as possible were examined and the pressure drops over the shaft were measured. This was done in conjunction with Impala Platinum Mines. The one requirement Impala placed on this work was that it must in no way affect the production of any shaft. Therefore all the measurements were taken in periods when the shafts were being maintained or not being used at all.

The shaft measurements were taken at a time when the conveyances in the shaft were limited in their movement in order to obtain results that are not overtly affected by these conveyances. Once the initial measurements had been completed, the winders and the various levels within the shaft were measured over a number of shifts to ascertain whether there were any pressure spikes in the system when the shaft is operating at a steady state.

The above data were evaluated against the current theory such that any strengths and weaknesses of the theory could be confirmed and used in the subsequent analysis. This evaluation also provided an initial check to ensure the quality of the measured data. The analysis was completed immediately after each test so that if additional measurements were required, they could be taken immediately.

1.8.3 Detailed Analysis

During this phase of the work, the results of the evaluation in Section 1.7.2 were subjected to a theoretical analysis. This analysis consisted of two parts:

1. Analysis of the shaft systems evaluated with respect to the current available theory
2. Analysis of the shaft systems evaluated by means of a computational fluid dynamics (CFD) model

1.8.4 Conclusions

The conclusions on the specific design considerations are drawn in a manner that will allow any future design work to include the recommendations from this work.

In addition, potential areas of difficulty are highlighted and the area that will make the most significant difference to the shaft resistance is defined.

Finally, recommendations are made as to what direction future work in this regard should take.

1.8.5 Outline of the Study

This chapter has described the background to the work and its specific objectives. Chapter 2 provides an overview of the literature available on this subject, without commenting on work not directly associated with the stated objectives.

CHAPTER 3 presents the experimental procedure and the specific analysis techniques used in the evaluation. The results of this evaluation and analysis are in CHAPTER 4 and CHAPTER 5. The final chapters give the conclusions drawn from the analysis and comment on the economic evaluation of the shafts.

1.9 SUMMARY AND CONCLUSIONS – CHAPTER 1

1.9.1 Summary

As a result of the rising electrical energy costs in South Africa, a method was being sought to reduce the overall electrical consumption of typical mine shaft systems. To achieve this, the first step was to analyse a typical shaft system and to determine what areas required the most energy to operate.

A typical shaft configuration was analysed and the primary energy consumers were identified. The ventilation fans for this system were found to consume 15% of the total energy of the shaft system. More than 50% of the energy supplied to an equipped downcast shaft to move the ventilation air through the shaft was calculated to be consumed by the shaft itself, more specifically by the pressure losses that occur in the shaft as the ventilation air passes through it. This area was deemed worthy of additional evaluation.

This evaluation took the form of completing a series of static tests on the shaft to determine the actual pressure losses of the systems and comparing these losses with those predicted by the current available theory. No. 14 shaft at Impala Platinum was used for these tests.

No. 14 shaft consists of two downcast and two upcast shafts. The pressure losses in each of these were measured and compared against the theory. It was found that there was a difference of approximately 20% between the theory and the measured results. More importantly, it was noted that 54% of the electrical energy consumed by the ventilation fans was used to overcome the pressure losses within the equipped downcast shaft. The remainder of the work presented here was conducted in order to reduce this percentage by as much as possible.

To complete the analysis, the objectives of the work were defined and the manner in which these were achieved was laid out. The following steps were identified as being required:

1 Literature study

2 Definition of the objectives

3 Evaluation of current shaft configurations

4 Detailed analysis

5 CFD simulation of the shaft system

6 Economic evaluation

1.9.2 Conclusions

The following conclusions can be drawn from this chapter:

1 The increase in electrical energy costs in South Africa requires that the electrical energy consumed by shaft systems be reduced by as much as is practically possible.

2 One of the primary electrical consumers is the ventilation fans, most of whose electrical energy is required to overcome the pressure drops experienced in transporting air down the equipped shaft.