# Demand Reduction

# and

# Responsive Strategies

# for

# Underground Mining

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I certify that all material in this thesis which is not my own work has been identified and that no material has previously been submitted and approved for the award of a degree by this or any other University.

Signature: ……………………………………………………………..

# Abstract

This thesis presents a demand reduction and responsive strategy for underground mining operations. The thesis starts with a literature review and background research on global energy, coal mining and the energy related issues that the mining industry face everyday. The thesis then goes on to discuss underground mine electrical power systems, data acquisition, load profiling, priority ranking, load shedding and demand side management in mining. Other areas presented in this thesis are existing energy reduction techniques, including: high efficiency motors, motor speed reduction and low energy lighting.

During the thesis a data acquisition system was designed and installed at a UK Coal colliery and integrated into the mines existing supervisory control and data acquisition (SCADA) system. Design and installation problems were overcome with the construction of a test meter and lab installation and testing. A detailed explanation of the system design and installation along with the data analysis of the data from the installed system. A comprehensive load profile and load characterisation system was developed by the author. The load profiling system is comprehensive allows the definition of any type of load profile. These load profiles are fixed, variable and transient load types. The loads output and electrical demand are all taken into consideration.

The load characterisation system developed is also very comprehensive. The LC MATRIX is used with the load profiles and the load characteristics to define off-line schedules. A set of unique real-time decision algorithms are also developed by the author to operate the off-line schedules within the desired objective function. MATLAB Simulation is used to developed and test the systems. Results from these test are presented. Application of the developed load profiling and scheduling systems are applied to the data collected from the mine, the results of this and the cost savings are also presented.

# Acknowledgements

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## ABBREVIATIONS

# Abbreviations

| | |
|---|---|
| **A** | Amperes |
| **ABS** | Accounting & business systems |
| **AC** | Alternating current |
| **ACT** | Accelerated Technology |
| **AFC** | armoured face conveyor |
| **AMI** | Advanced metering infrastructure |
| **AMR** | Automatic meter reading |
| **C&LM** | Energy conservation and load management |
| **CCM** | Central control module |
| **CCS** | Carbon capture and storage |
| **CFL** | Compact fluorescent lamp |
| **CFM** | Cubic feet per minute |
| **CHP** | Combined heat and power units |
| **CIS** | Customer information systems |
| **CMV** | Coal Mine Velenja |
| **COM** | Component Object Model |
| **CPP** | Coal preparation plant |
| **CT** | Current transformers |
| **DC** | Direct current |
| **DCM** | demand control module |
| **DCOM** | Distributed Component Object Model |
| **DER** | Distributed energy resources |

## ABBREVIATIONS

| | |
|---|---|
| **DLC** | Direct load control |
| **DNO** | Distribution network operators |
| **DR** | Demand response |
| **DRP** | Demand reduction pyramid |
| **DSM** | Demand side management |
| **DSR** | Demand side response |
| **EIA** | Energy Information Administration |
| **EISA** | Energy Independence and Security Act |
| **ERP** | Enterprise resource planning |
| **GIS** | Geographic information systems |
| **GPM** | Gallons per minute |
| **HAN** | Home area networks |
| **HV** | High voltage |
| **HVAC** | Heating, ventilation, and air conditioning |
| **HVDC** | High voltage direct current |
| **I** | Current |
| **IEA** | International Energy Agency |
| **IED** | Intelligent electronic devices |
| **IHD** | In-home display |
| **ILS** | Intelligent load shedding |
| **IP** | Internet protocol |
| **ISO** | International Standards Organisation |
| **kV** | Kilovolts |
| **kW** | Kilowatts |
| **kWh** | Kilowatt hours |
| **L/s** | Litres per second |
| **LBM** | Load block matrix |
| **LC Matrix** | Load Classifications MATRIX |

| | |
|---|---|
| **LCM** | Loading condition matrix |
| **LCP** | Least cost planning |
| **LDM** | Load demand matrix |
| **LED** | light-emitting diode |
| **LM** | Load management |
| **LOM** | Load output matrix |
| **LPM** | Load parameter matrix |
| **LS** | Load shedding |
| **LSM** | Load speed matrix |
| **LTE** | Long-term evolution |
| **LTM** | Load time matrix |
| **MINOS** | Mine Operating System |
| **MVA** | Mega volt amperes |
| **MW** | Megawatts |
| **NALM** | Non-intrusive appliance load monitoring |
| **NCB** | National Coal Board |
| **NEMA** | National Electrical Manufacturers Association |
| **NILM** | Non-intrusive load monitoring |
| **NIST** | National Institute of Standards and Technology |
| **OFFER** | The Office of Electricity Regulation |
| **OLE** | Object linking and embedding |
| **OPC** | OLE for Process Control |
| **OSI** | Open Systems Interconnection |
| **P** | Power |
| **PHEV** | Plug-in hybrid electric vehicle |
| **PLC** | Programmable logic controllers |
| **PQ** | Power quality |
| **PRE** | Priority ranking engine |

## ABBREVIATIONS

| | |
|---|---|
| **PT** | Potential transformers |
| **PV** | Photovoltaic |
| **R&D** | Research and development |
| **RDD&D** | Research, development, demonstration and deployment |
| **ROM** | Coal handling and preparation plant |
| **ROM** | Run-of-mine |
| **rpm** | Revolutions per minute |
| **RTL** | Real-time loop |
| **RTP** | Real-time pricing |
| **SAP** | Simple Asynchronous Protocol |
| **SCADA** | Supervisory control and data acquisition |
| **SSH** | Secure shell |
| **SVD** | Singular value decomposition |
| **T&D** | Transmission and distribution |
| **tcf** | Trillion cubic feet |
| **TCP** | Transmission control protocol |
| **TLP** | Typical load profile |
| **TOU** | Time-of-use |
| **UFLS** | Under frequency load shedding |
| **USC** | Ultra-supercritical |
| **UVLS** | Under voltage load shedding |
| **V** | Volts |
| **VA** | Volt amperes |
| **VAM** | Ventilation air methane |
| **VAR** | Volt amperes reactive |
| **VFD** | Variable frequency drive |
| **VOD** | Ventilation on demand |
| **VSD** | Variable speed drive |

**W** Watts

**WEC** World Energy Council

**WiMAX** Worldwide Interoperability for Microwave Access

## ABBREVIATIONS

# 1

# Introduction

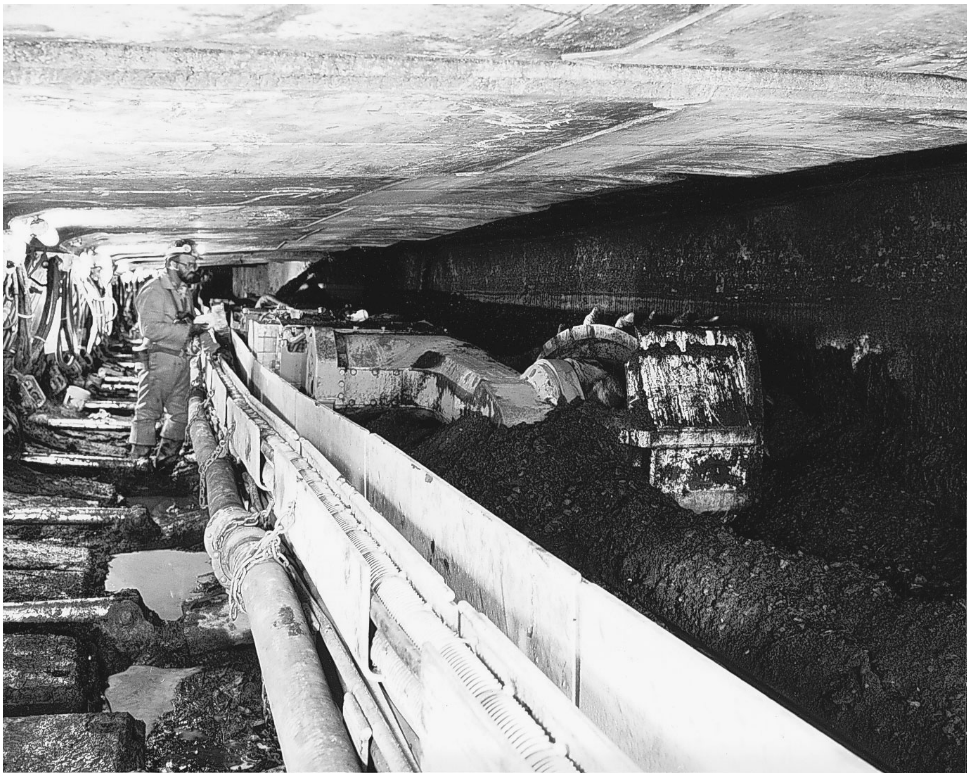


**Figure 1.1: A Longwall Shearer** (Joy Technologies Inc.)

Modern underground mining methods around the globe use state of the art machinery that perform mining operations with great capacity and precision. Their efficiency produces commodities at an alarming rate in order to meet the ever increasing demand of the developing world. Figure 1.1 shows one such machine - a Longwall shearer. It is not only the efficient operation of the mining machines that require attention, it is also the depleting energy resources that they consume. Mining industries alone are responsible for nearly 1% of world energy consumption (IEA, 2008) and are a major contributor to $CO_2$ emissions. Reducing energy consumption and thus also reducing the carbon footprint will play a key part in meeting global environmental targets such as the 2020 EU goal of 20% reduction in greenhouse gas emissions (EU, 2009).

More efficient use of energy via electricity demand reduction and response strategies would also ensure that the environmental impact of the mining is reduced, as well as providing financial cost savings to the industry as a whole. This would also have a positive impact on mine operations in areas where the energy supply is a limited resource, for example in off-grid locations. The location of coal mines tend to be in or near accessible areas and are generally grid supplied. This poses less power supply problems than for hard rock mines. Hard rock mines can be located anywhere and as the search for minerals becomes harder and harder, the locations are becoming even more remote. These remote mine locations pose many logistical problems, with energy supply being at the top of the list. The location is not the only problem that hard rock mines face in their energy supply, existing mines are going even deeper to recover more precious minerals and extend the mine life. This is increasing the strain on existing underground power distribution. As the mines get older, the ore to waste ratio tends to become higher, due to the depleting resource. This has a knock on effect of increasing the energy required for processing. The ore must be crushed to a higher degree to extract the reduced mineral content, increasing running costs and causing commodity prices to rise.

Electricity users have traditionally been enticed by the utility companies to shift their load to off-peak periods by providing peak and off-peak pricing. However, this approach cannot be used to balance short-term intermittent electrical demand and supply and is impractical for 24 hour industrial operations such as mining. There is a wealth of published research material that has been provided by the electrical industry, including Smart Grid, electrical management, power system stability, load control, load profiling, demand side management (DSM) and demand side response (DSR). However, the mining industry has published very little material and has yet to introduce many modern DSM techniques. This lack of research and implementation within the mining industry provides a significant opportunity for study. It is the overall aim of the work presented in this thesis to develop various demand reduction and management strategies for underground mining operations in order to better enable further collaboration between the mining industry and the electric utility companies. Furthermore, estimating the financial benefits, for both the electricity and the mining sector, is also a key part of the research.

Gaining knowledge of system electrical power consumption is of key importance when considering improving efficiency. Load profiling is used by utility companies to define class types of customers, allowing the utility to forecast, schedule, define tariffs and estimate transmission obligations. Power utilities calculate and offer residential, commercial, large commercial, industrial, agricultural, street lighting and standby load profiles to their customers. A utility with access to this information can manage resources and balance loads accordingly to increase the security of the supply, reducing down time whilst improving efficiency.

Forecasting demand also gives a significant advantage when large industrial customers are negotiating a better tariff structure and price with the utility. The electricity supply tariffs are changing, automatic meter reading (AMR) and advanced metering infrastructure (AMI) systems are being installed across nations globally. AMR & AMI systems provide the functionality to offer dynamic pricing and automatic load control. This pricing and load control enhancement is just a small part of the overall optimisation of the utility power network or grid started in the US, Europe, China and the Far East and is now known as Smart Grid. The way power is transmitted and distributed is becoming smarter.

The Smart Grid is self-healing, has improved power quality, is resistant to attacks and offers security of supply. Localized control, integrating renewable energy and distributed energy resources (DER) are other main objectives for the modern Smart Grid. Renewable energy is moderately predictable, although the power produced can vary from day to day, and often even more frequently. The introduction of dynamic pricing and real time interactive DSR will provide electricity users with financial incentives to shift their load on short notice. Therefore helping customers change their attitude from a demand-focus to a supply (or availability) focus, shifting their loads to reduce cost and coincide with the availability of power.

## 1.1 Research Aims and Objectives

The project will focus on developing novel demand reduction and management strategies, capable of reducing electrical power demand, specifically, when the mine is operating during peak conditions. This approach will avoid potentially large inefficiencies of the overall mine electrical load, thus improving the efficiency of mine operations. This will correlate the mines own peak demand and the utilities peak demand to ensure that the demand control in the mines are utilized more effectively and optimise the benefits of electricity cost reduction. Three core areas have been defined from the primary research into the mining and electrical power distribution industry which are as follows:

- System Monitoring and Data Acquisition
- Load Profiling and Load Characterisation
- Load Scheduling and Demand Reduction

### 1.1.1 System Monitoring and Data Acquisition

The case study for the project is a UK Coal colliery, one of the two operating deep underground coal mines still owned by UK Coal PLC. It is the largest remaining deep mine in the UK. Production began at the colliery in April 1965. The mine has two main shafts that are almost 800 metres deep. The first shaft is used for transporting personnel and materials and the second being the up-cast shaft for the mineral, being transported at the rate of up to 900 tonnes an hour. All coal production (except a small amount of house coal) from UK Coal collieries supply local power stations.

Initially a questionnaire was compiled to gain information on mining systems. However, the response was very limited and there was not enough information to continue with the thesis. After contacting UK coal, a data acquisition system was designed by the author and installed at a UK colliery. The system has 22 power meters and they were installed in two of the surface substations at the UK

Coal colliery. The system was integrated into the existing supervisory control and data acquisition (SCADA) and the mines power consumption data was logged to a database. This power consumption data, along with the mines existing SCADA data was used to develop a load profile database of the mining operation at the UK Coal Colliery. Chapter 6 outlines the data analysis methodology undertaken and Chapter 7 discusses in more detail the data acquisition system and installation.

### 1.1.2 Load Profiling and Load Characterisation

A comprehensive load profile and load characterisation system was developed by the author. The load profiling system is comprehensive allows the definition of any type of load profile. These load profiles are fixed, variable and transient load types. The loads output and electrical demand are all taken into consideration.

The load characterisation system developed is also very comprehensive. The LC MATRIX uses load profiles and the load characteristics to define the off-line schedules. The production characteristics include safety critical systems and constraints, amounts of active equipment, equipment conditions, maintenance constraints, allowable on/off periods and process limitations.

### 1.1.3 Load Scheduling and Demand Reduction

A major part of the project is the demand reduction which has required a considerable amount of analysis. The task of reducing electrical demand could be considered in many ways, with the simplest procedure of disconnecting a load, or moving some operations to off peak periods. Each service has been examined individually, the ventilation, the compressed air system, mine hoist, underground water pumping, refrigeration/cooling system. Each of the individual systems has scope for research into improving the electrical efficiency and reducing demand. However, it was also concluded that only optimizing one system would have no real effect on the overall mine efficiency and would not give any real indication or enough live production data to perform any form of load prediction or forecasting.

The LC MATRIX system is used to optimise off-line load schedules, with the minimum demand for a given production. In addition to the off-line scheduling, real-time scheduling modules have been developed by the author that perform decisions to keep the off-line schedule operating with the minimum demand, or any other option the optimisation time-line has selected, such as, maximum output, minimum demand, maximum production with minimum transportation, maximum transportation with minimum production.

To fulfil the requirements of the project all of the mines systems are included in the optimization and that complete electrical network optimization is the research focus. Forecasting would also have a major influence in reducing demand and optimising power systems. However, forecasting is only discussed as further work.

## 1.2 Thesis Layout

Below is a brief description of each chapter and its contents.

**Chapter 1** - *Introduction*: An introduction to the thesis presenting the aims and objectives.

**Chapter 2** - *Energy and Mining*: This chapter covers energy, globally and within the mining sector. How the energy of today is produced, its distribution of use and fuel type. Also discussed is the possible energy use forecasting for the world and the mining industry. The chapter also includes a brief technical overview of underground coal mining, including the coal preparation process.

**Chapter 4** - *Mine Electrical Systems and Three Phase Electricity*: A technical description of underground mine electrical systems and three phase electrical systems; apparent, active and reactive power.

**Chapter 3** - *Demand Side Management and Smart Grid*: A literature review examining demand side management and Smart Grid technology.

**Chapter 5** - *Demand Side Management for Underground Mining*: This chapter outlines demand side management in mining, common practices and a summary of the energy audit at a UK Colliery

**Chapter 6** - *Development of Electrical Demand Reduction and Responsive Strategies*: This chapter outlines the project, its methodology, procedure; including data acquisition; load profiling; priority ranking; and load shedding.

**Chapter 7** - *SCADA and Data Acquisition*: An overview of monitoring and SCADA systems and the communication protocols. Also a full design and installation description of the power meter and data acquisition system at the UK Coal colliery.

**Chapter 8** - *Project Data*: Presentation of the data collected, with the further development of the load profiling process and demand side response strategies and cost savings.

**Chapter 9** - *Conclusions and Recommendations*: Recommendation, conclusions and further work.

# 1. INTRODUCTION

# 2

# Energy and Mining

In 2011, the world's population reached 7 billion and continues to grow at a rate of 80 million a year (Worldometers, 2013). Energy and resources are under ever increasing strain as developing countries such as India and China grow their appetite for fuel. Commodities are also feeling the strain and are continuing to be in short supply. This increased demand must be met by an increase in overall production in mining which in turn is placing a further demand on global energy. This chapter discusses global energy and the mining industry, highlighting the significant problems that need to be overcome and addressed if this demand is to be ever at a sustainable level whilst at the same time still allowing population growth.

## 2.1 Global Energy

Pressure remains on countries to set carbon targets and agree to lower emissions levels to half the global greenhouse gas emissions by 2050 and keep global temperature increase to below two degrees (WEC, 2013). In addition to sustainability and environmental issues there are other political pressures to consider which are keeping Energy supply at the top of Government agenda's. A combination of factors including supply uncertainty, price volatility, US shale gas production, the global recession, increased demand from the East and trade disputes with China are also problems associated with energy resource pressure. Figure 2.1 shows

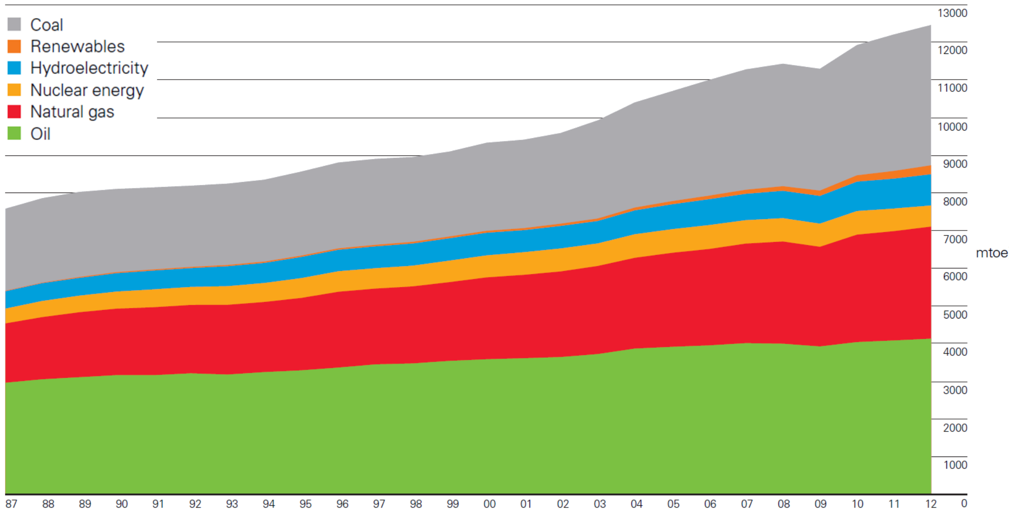



**Figure 2.1: World Primary Energy Consumption 1987-2012** (BP, 2013a)

the world energy consumption from 1987 to 2012 and its various sources. It can be seen that fossil fuel still dominate the energy profile and will continue to do so for the foreseeable future, this is further demonstrated by the fact that in the year 2012 fossil fuels counted for 87% of global energy consumption. (BP, 2013a).

It is widely understood that world economic growth has a direct relationship with energy demand. Nezhad (2009) shows clearly how demand for goods, services and energy increases as population income increases. With current growth levels there will certainly be unsustainable pressure on natural resources unless energy demand is decoupled from economic growth and more renewable resources are exploited.

### 2.1.1 Energy Prediction

A number of energy scenarios for 2050 are suggested by several organizations, including the International Energy Agency (IEA), the United States Energy Information Administration (EIA), and the World Energy Council (WEC). All scenarios foresee a substantial increase in global energy demand by 2050. In his report Nezhad (2009) analyses the IEA energy forecast scenarios, the Accelerated Technology (ACT) Scenario and the BLUE Map Scenario (IEA, 2008). These scenarios are present alongside a baseline or existing trend - if no change is made energy consumption will double by 2050 and $CO_2$ emissions will continue to rise to two-and-a-half times the current levels, even after taking into consideration energy efficiency gains and expected technological progress. In contrast, in the ACT Scenario, energy consumption will be at about 77% of the Baseline Scenario and energy-related $CO_2$ emissions will be reduced to their 2005 levels by 2050. This model also shows that by 2050, energy efficiency measures could reduce electricity demand by a third below the Baseline levels.

In the BLUE Map Scenario, energy consumption will be about 67% of the Baseline prediction and the $CO_2$ emissions will be 50% lower than the 2005 levels. This model explores the least-cost solutions to achieve this goal and limit the risks of severe climate change. To meet the objectives of the BLUE Scenario, we must develop and implement far-reaching new policies too substantially 'de-carbonize' power generation. Both the ACT and BLUE Scenarios require extensive use of renewable energy resources which would provide 35% and 46% of the total power generation requirements. The IEA report (IEA, 2008) states that these scenarios are not predictions; rather they are the "least-cost pathways that may be available to meet energy policy objectives, given a certain set of optimistic technology assumptions."

Although in both the ACT and BLUE scenarios, there is extensive use of renewable energy, fossil fuels (with carbon capture and storage (CCS) to reduce the $CO_2$ emissions) still play a major part in the global energy profile of 2050, with 45% and 30% retrospectively for the ACT and BLUE scenarios and coal-CCS still having 15% share of power generation (EIA, 2013).

What has previously been described are only possible futures and not 'what will happen'. Since 2008 many events have happened, global recession and economic turmoil leaving leaders harder decisions to make with investment and social economic support. The Fukushima disaster in 2011 has also had a major impact - changing the world's outlook on nuclear power and promoting Germany's decision to fully exit out of nuclear power by 2022. The solar photovoltaic (PV) industry has seen a price collapsed since 2008 from over 4.5 $/Wp to as low as 0.6 $/Wp. This is largely driven by low-cost production in China but has been accentuated by the fact that in 2012 the demand did not kept up with expectations and absorbed just under half of the global manufacturing capacity of about 100 GWp (WEC, 2013).

One certain trend observed is an increase in global energy demand, 36% by 2030 (BP, 2013c). However, what is not certain is how we will overcome this. The global energy sector will need to invest half of current world GDP over the next two decades in order to address these challenges and expand, transform and adapt the energy infrastructure (WEC, 2013). Governments can be reluctant and slow and in some countries corruption is also a problem. Decisions must be made, not just to hit policy targets, but to achieve significant change in energy culture and demand. In the absence of global agreements and regulations on energy or climate policy that would guide such a transition, the main policy decisions remain in the hands of national and sub-national policy makers who are mainly based in the richer, developed countries such as North America and Europe.

### 2.1.2 Coal, Oil and Shale Gas

At present, consumption levels continue as predicted with fossil fuel dominating demand. However the types of fossil fuels consumed is changing in proportion. Coal is growing in popularity whilst oil is losing its market share for the twelfth consecutive year (BP, 2013a). In the United States coals dominate grip on power-generation is starting to fade- from a high of 57% in 1985 to 42% in 2011. In 2012 coal was providing only 37% of US electricity (Katusa, 2013). This is due to the rise in US shale gas production, which has increased from less than two trillion cubic feet (tcf) in 2004 to 7,8 tcf in 2011 and is projected by the EIA to reach 16,7 tcf in 2040 (Preuss, 2013). The move to gas in the US has caused coal producers to find other markets and coal exports from the United States in March 2013 totalled 13.6 million short tons, nearly 0.9 million short tons above the previous monthly export peak in June 2012. In 2013 coal exports are projected to be for the third straight year of more than 100 million short tons, following annual exports in 2011 of 107.3 million short tons and record annual exports in 2012 of 125.7 million short tons, compared to the annual average of 70 million short tons in previous years. Most of the increase in export has been absorbed by the increased demand in Asia.

How long this 'dash for gas' will last is debatable as it is dependent on natural gas remaining competitive with coal. Utilities only switched to gas because it was cheaper but the economic edge is wearing thin. In fact, in April 2012 the two fuels are almost equivalent in terms of energy economics. In Europe, shale gas has not yet been exploited as much as in the US but this may soon change. Cheap US coal exports have had a negative impact on the European coal industry. However, It is unlikely that the UK coal mining will see any significant investment in the near future. This was recently evidenced by UK Coal's decision made last year to not re-open Harworth Colliery in Nottinghamshire which has been mothballed since 2006. Even more recently, after a large underground fire in February 2013 UK Coal had to closed Daw Mill colliery in Warwickshire. This closure goes further to reduce the once thriving UK coal industry.

### 2.1.3 The Power Industry

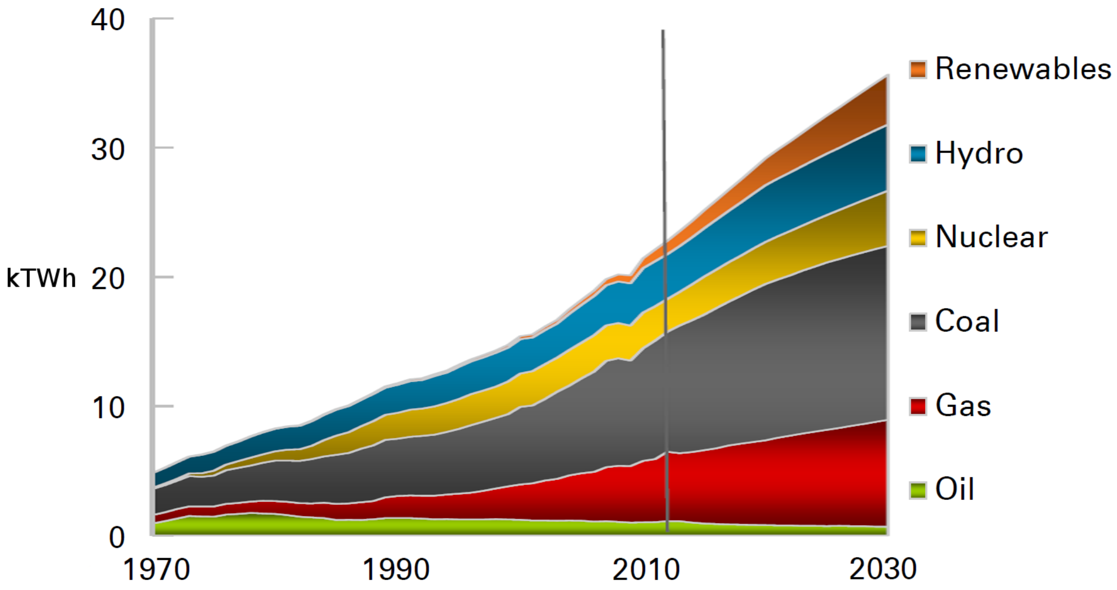



**Figure 2.2: World Power Generation 1970-2030** (BP, 2013a)

Electricity is a very versatile energy source, capable of powering all manner of machinery and equipment, it is what has shaped our modern world. Defined as a secondary energy source, as it is generated using a primary energy source. Electricity can be generated from virtually any primary energy source, using coal, gas or a nuclear reaction to produce heat to create steam for steam turbines, fuel oil for generators, or renewable energy; wind generators and photovoltaic panels converting energy from the sun.

Power generation is the only sector where all these primary fuels compete for use and in 2010, power generation was responsible for 20.2% of consumed global energy, 32% of total global fossil fuel use and 41% of energy related $CO_2$ emissions (EIA, 2011). Figure 2.2 shows the world power generation from 1970 to 2030 with the different primary fuel spread. The energy used for power generation is predicted to grow by 49% 2011-30, and accounts for 57% of global primary energy growth (BP, 2013c). Overall in 2011, 41% of world's electricity came from coal and 22% from gas, Table 2.1 shows the coal powered generation share for the top eight countries.

| Country | Coal Used |
|---|---|
| **South Africa and Poland** | 90%+ |
| **China and Australia** | 80% |
| **India** | 66% |
| **UK** | 44% |
| **US and Japan** | 20% |

**Table 2.1: Share of coal in power generation (IEA, 2013b)**

### 2.1.4 Power Generation Efficiency

Improving energy efficiency of electricity production therefore offers a significant opportunity to reduce the world's dependence on fossil fuels and in so doing helps to combat climate change and improve energy security. The age of a country's power plants, will be an important factor, as the current efficiency of most coal-fired power plants is well below state-of-the-art. "The average coal fired power plant efficiency is about 34% compared with 46% for the most efficient ones currently operating in Spain. If all world regions had the same performance as the best one in Spain, 770 Mtoe of fuel would have been saved in 2006, avoiding 2.4 Gt of $CO_2$ emissions" (Nezhad, 2009). Other countries such as Japan, Korea, some countries in Europe, including Italy and Germany and more recently China are heavily investing in modern coal fired power stations, where ultra-supercritical (USC) power plants (which can achieve efficiencies of up to 45%) are already in commercial operation (IEA, 2013a). Improving conversion efficiency in power generation means that the total fuel inputs to generate power could grow less rapidly than electricity demand. Far more electricity is generated than used, in 2009 transmission and distribution (T&D) losses and direct use in power plants amounts to 15% of global power generation (IEA, 2009). Increasing efficiency of the power plant and reducing transmission losses will provide the electrical demand at a reduced energy cost. For example, the continued implementation of Smart Grid technologies can, as some studies have shown (WEC, 2012), reduce energy consumption and transmission losses by 2-5%.

Renewable energy is to play an important part in the demand for electricity in the future. However, renewable energy requires significant infrastructure, such as roadways, transport and electrical distribution, for what is a low density energy source. Fossil fuels are needed to produce the steel, cement, copper cable etc. for the renewable energy. We will have to pay to clean up fossil fuels; however, we will also have to pay for renewable energy. Moving over to a renewable energy efficient economy, with fossil fuels minimised seems a far distant future. At the end of 2012 non-hydro renewable energy produced a total 5.2% of the world's electricity, with the UK even lower at 3.8% (REN21, 2013). If the targets, such as the UK's 15% of renewable energy by 2020 are to be achieved, a large amount of energy will be required to get there, and this energy must come from existing non-renewable sources, such as coal. Coal has such a hold on world power generation and will continue to do so. The efficiency and emissions of coal-fired power plants must be improved.

Replacing the existing fleet of power plants over the next 10-20 years with new, higher efficiency supercritical and ultra-supercritical plants will help. A one percentage point improvement in the efficiency of a conventional coal-fired power plant results in a 2-3% reduction of the plants $CO_2$ emissions (WCA, 2013). Although efficiency improvements will reduce $CO_2$ emissions, the major reduction will be the large scale introduction of CCS. The high $CO_2$ emissions of coal fired power stations means politically that CCS is the key to coals future success as without CCS coal-fired power plants do not have a long term future. CCS is, at present in slow progress and proving to be uneconomical for large scale coal plants (Grid, 2011).

Determining exactly which direction the global energy demand will go is difficult to define. BP (2013c) offer a realistic outline based on existing trends and is one possible direction. However, IEA (2009) and Grid (2011) have outlined sound feasible scenarios that are the basis for technology road-maps that can help to establish more detailed action plans and international technology co-operation to develop a more efficient energy future. All of them are in agreement that there must be huge investment in research, development, demonstration and deployment (RDD&D) of energy technologies; this also includes the increase in energy efficiency and the need for demand reduction and smart technology to implement this with seamless automation.

## 2.2 Energy in Mining

In 2005, mining used 2% of industrial energy worldwide and as industry used 35% of the total energy worldwide, mining used 0.7% of total global energy demand (IEA, 2008). Although this does not seem excessive, if it is considered in context, 0.7% of total world energy is the same as the total energy consumption by Finland and Norway combined in the same year (BP, 2013a). Mining operations are spread across the world. However, there are geological hotspots where large mineral reserves are located. In these countries the energy requirements for mining is high and are large contributors to the overall mining energy requirements. Table 2.2 shows the major mining countries and their mining energy consumption for 2005. Although these are not recent figures, they are the most recent reliable figures for mining energy consumption that could be sourced by the author.

Table 2.2 also shows the countries industrial energy use and the contribution mining has to the countries industry. Canada is the largest contributor to world mine energy using over 20%, they are closely followed by China with nearly 19%. However, in Canada this represents over 15% of the countries industrial energy use where as China's industrial energy use is nearly ten times that of Canada, this

| Country | Total Industrial (Mtoe) | Mining Energy (Mtoe) | Mining % of Industry | % World Mining Energy | % World Energy |
|---|---|---|---|---|---|
| **Canada** | 71 | 11 | 15.5 | 20.8 | 0.1420 |
| **China** | 596 | 10 | 1.7 | 18.9 | 0.1291 |
| **Russia** | 160 | 6 | 3.8 | 11.3 | 0.0774 |
| **South Africa** | 25 | 5 | 20.0 | 9.4 | 0.0645 |
| **Australia*** | 38 | 4 | 10.5 | 7.5 | 0.0516 |
| **Brazil** | 82 | 3 | 3.7 | 5.7 | 0.0387 |
| **United States** | 397 | 2 | 0.5 | 3.8 | 0.0258 |
| **Mexico** | 35 | 2 | 5.7 | 3.8 | 0.0258 |
| **Africa (Ex SA)** | 75 | 1 | 1.3 | 1.9 | 0.0129 |
| **Germany** | 85 | 1 | 1.2 | 1.9 | 0.0129 |
| **India** | 131 | 1 | 0.8 | 1.9 | 0.0129 |
| **Rest of World** | 1068 | 7 | 0.7 | 13.2 | 0.0903 |

**Table 2.2: Mining Energy by Country 2005** - (IEA, 2008) *(ABARE, 2005)

highlights China's large manufacturing industry. Mining mainly uses diesel and electricity as its energy source, the majority of diesel is either used for transport or generating electricity for the mining operation. Coal mines also use drained methane to run generators. Diesel is used more widely for underground ore transportation in hard rock mining, but this is changing as more and more mines are turning to electric or battery powered transport due to the volatile fuel energy markets of recent years.

### 2.2.1 Energy Cost

Energy cost has an impact everywhere and mining is no exception, the cost of energy is a serious proportion of the overall cost of extracting minerals for a mining company. In recent years, energy prices have soared, particularly fossil fuels with the most prominate being crude oil. Figure 2.3 shows crude oil prices from 1972-2012, and it can be seen that in six years (2002-2008) the price rose by $74/barrel, an increase of 385%. In 2008, the price then took a dramatic decline (due to the global financial crisis), only to recover again to a price of over $110 by 2011. This dramatic increase and fluctuation in price is unpredictable making it difficult for any industry, let alone mining. As a large consumer of energy, the impact on the mining industry is that even a small increase in energy can have significant implications, such as the loss of jobs and even possible mine closure.

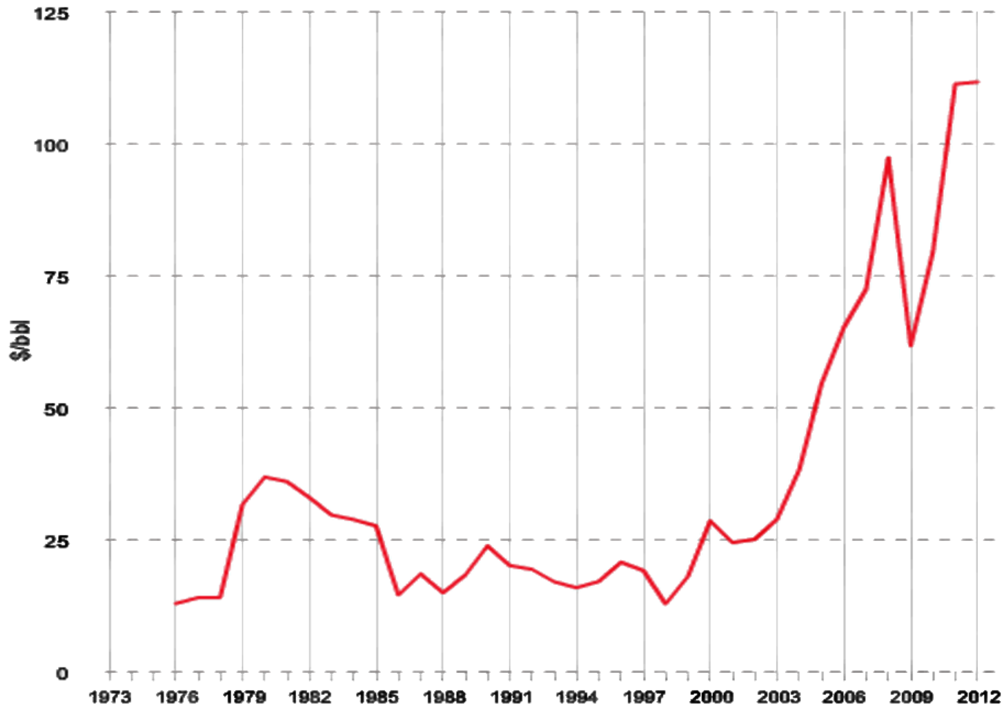



**Figure 2.3: Crude Oil Price ($/Barrel) 1972-2012** (BP, 2013b)

### 2.2.2 Energy Availability and Remote Locations

It is not only the energy price that can be a problem for mining, it is also the availability of an energy source. A mine has to be developed where the resources are and this can be located anywhere geographically. As the search for minerals gets harder, more often than not a mine is in a remote location. Remote locations not only pose a problem for the development and delivery of materials etc, they are usually without, or only have limited grid connection, forcing the mine to use on-site generation. Historically, the most popular choice has been diesel generators; however as the price of diesel rises and has become more volatile this is becoming a very expensive and riskier option. Diavik diamond mine is a good example of a remote mine, in Lac de Gras, Northwest Territories, Canada and is only accessible by air or ice road for 6 weeks during the winter.

Diavik is 100% diesel driven, with no grid connection or any possibility of one. The 24MW demand of Diavik mine is provided by diesel generators at a cost of \$0.31/kWh. When production began in 2003 the cost would have been approximately \$0.12/kWh, that is a power cost increase of 264% in less than ten years. This increasing energy cost is forcing mining companies to look at alternative sources of energy, such as renewable energy. In September 2012 Diavik diamond mine installed a 9.2MW wind farm started generating power and is projected to lower the mines annual power-related diesel fuel requirement by 10 per cent and reduce its carbon footprint by six per cent. In addition, it will reduce Diavik's seasonal winter road fuel haul by approximately 100 loads. The wind farm will produce 17GWh a year with an estimated payback period of eight years (Wyk, 2013). Some remote mines in Western Australia have a grid connection but this is limited and excess demand must be met with diesel generation. Solar is now cheaper than diesel generation (Pearson, 2012) and in the right location could provide the demand for a mine during daylight hours.

Although power generation for remote locations is expensive, they do not have the power supply security threat that mines in countries with limited utility generation have. In these countries the utilities have aggressive demand management policies and have controlled power cuts without notice. This is necessary just to keep the lights on in the towns and cities, and effect the mining industry on a daily basis. South Africa and India are prime examples of this. Eskom, South Africa's only utility has a demand response (DR) program (Eskom, 2013). The program offers large customers to reduce their demand when requested in return for financial rewards. This allows Eskom to maintain grid stability and to continue to supply with minimal interruption. Xstrata, is just one participant in the DR program, taking their chrome furnaces off-line when there isn't sufficient capacity. Xstrata uses this opportunity not only to get paid by Eskom, but to do maintenance on the chrome smelters.

The rest of Africa is in a similar situation also with demand exceeding generation during peak periods. Africa is not the only location with supply issues; in India in 2012, over 700 million people were without power for two days due to excessive demand. India's power structure is often unable to meet peak power demands. Prime Minister Manmohan Singh has called for a \$400 billion investment to help increase the capacity and reliability of India's power grid, but the country's efforts to build "ultra-mega" coal plants have hit economic snags, while its effort to build the world's largest nuclear power plant faces protests and delays (Lavelle et al., 2012).

### 2.2.3 Mine-Mouth Power Facilities

A growing trend that has been seen in the coal industry is for companies to invest in mine-mouth power facilities. This is where the mine will establish a power station on site in order to both fuel the site and generate power for the local industry. A good example of this is Coal Mine Velenja (CMV), in Slovenija. This lignite coal mine supplies all its coal to the power station next door. The power station is so close that the coal is transported straight out of the mine to the power station by conveyor belt. The lignite coal needs very little processing as the coal seam is 90m thick, so the coal contains little impurities and after crushing can go straight to the burners. With the volatile export market, particularly as seen in the coal industry over the past five years, companies are looking to protect themselves through diversification. Having a power station on site means that there are minimal transport costs, as conveyors can run from the face directly to the power station. In some cases this has resulted in reductions of 30-50% on the cost of coal for power producers in northern China (Duffield and Toime, 2013). Removing the usual risks of the supply chain, benefits the mine site and the power plant also benefits as it has a guaranteed local supply at a lower cost giving them a competitive advantage. These mine-mouth power projects are becoming increasingly popular in countries with emerging economies such as sub-Saharan Africa, China, Indonesia and Mongolia where extra power capacity is required for the expansion of industry.

Furthermore, it is well publicised that the rising global greenhouse gas emissions are having detrimental impacts across the globe and governments are looking to do all that they can to reduce the consumption of non-renewable energy sources. As a result of these campaigns, companies have seen tax increases not only on the fuels themselves but also taxes on the amount of emissions they are producing. In Australia, a new tax was introduced under the Clean Energy Act on 1st

July 2012 which means companies emitting over 25,000 tonnes $CO_2$ equivalent must pay A$23 for every tonne emitted. This fixed price will rise by 2.5% annually for the next 3 years. There are an estimated 300 companies affected, particularly the mining industry, steel production, airlines and energy sector (News, 2012). The impact of this tax on the mining industry in particular has been highly criticised. Two of the largest corporate mining companies operating in Australia, BHP Billiton and Xstrata announced the loss of 900 jobs in September 2012, which is reportedly as a result of increased taxes, rising costs and a drop in coal price all compounding in a business which is no longer as profitable (McKenna, 2012).

The mining industry will continue to use a large proportion of the world's energy resource; cost, availability, security and even taxes will continue to pose problems for the foreseeable future, not only for the mining, but all industry and the world as a whole. Energy demand will increase as the population increases and the third world develops. However, providing more energy is only part of the solution, reducing demand and increasing energy efficiency is important for world energy stability.

## 2.3 Underground Coal Mining

In this next section underground coal mining processes are explained, the machinery used and the coal preparation process is are presented.

### 2.3.1 Coal Mining

Coal has been mined for hundreds of years, with evidence of flint axes being found embedded in layers of coal in excavations dating back to Stone Age Britain. Its predominant use has always been for producing heat energy and throughout the industrial revolution Britain was at the forefront in harnessing its energy for steam engine power and furnaces. 'Coal gas', gasification of coal for lighting, producing coke, along with other uses, saw it grow into a booming industry during the 19th century. Modern mining method improvements, new technology developments, and increased mechanisation have allowed collieries to mine much deeper and more efficiently (Riley, 2011).

Since the mid-20th century, coal has yielded its place to petroleum and natural gas as the principal energy supplier of the world. Over the last 30-40 years the UK has seen a significant decline in coal production, with only a fraction of the 1,647 coal mines nationalised in 1947 (Bailey, 2007) still in operation today. However, coal mining is a very large industry worldwide, and is still very much a part of our modern society. In 2011, the total UK coal production was 18Mt from deep and surface mines combined, although consumed over 50Mt (Pro, 2012). Coal-fired power plants use the largest proportion of the total consumption at approximately 80% to generate around a third of the UK's electricity. Coal is still a considerably valuable mineral in our modern society.

Collieries vary from mine to mine, each with different problems for the mining engineers to tackle. (Coal, 1989) discusses that coal seams in the UK vary from one to around seven meters in thickness, and they can range in depth from the surface between 1350 metres and 30 metres. In most cases, a coal seam continues at the same depth and do not vary significantly in thickness. Although, they have been found at an angle of up to 1 in 2 and unpredicted geological disturbances create problems for miners. However, the methods and machinery used in modern coal mining are very similar from mine to mine.

### 2.3.2 Underground Coal Mining Methods

The two predominant modern mining techniques adopted globally are 'Room-and-Pillar' and 'Longwall', with the Longwall technique being the chosen method in Europe and Room-and-Pillar being the dominating choice for the US underground coal mining. Figure 2.4 shows a coal mining schematic with surface facilities, access shaft and the Room-and-Pillar and Longwall mining methods. In underground mining, access from the surface to the coal seam is gained by either shafts or drifts. Two shafts are constructed, an up-cast shaft and a downcast shaft, the downcast shaft provides airflow into the mine and the up-cast allows the circulated air out of the mine. The ventilation air is vital for an underground coal mine

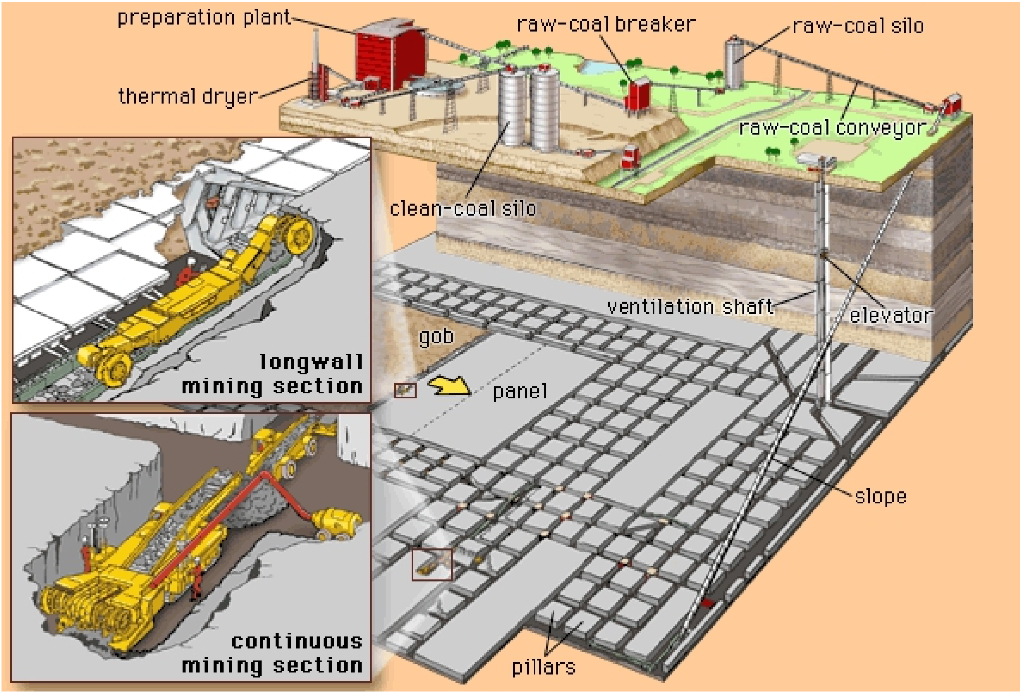



**Figure 2.4: Coal Mining Schematic** (CONSOL Coal Group)

to control the environmental conditions. After the shafts are in place, a network of roadways are driven in the seam then the installation of service infrastructure for essential activities, including human and material transport, ventilation, water handling and drainage, and power. This initial phase is termed 'mine development'.

After development the next phase or extraction is then begun. In Room-and-Pillar, entries are driven into the coal seam and at intervals along the entries. They are connected at right angles by wider entries, called rooms. The resulting grid formation of pillars supports the overhead strata. The exposed roof is supported, usually with rock bolts. Once the development of the pillars is completed the pillars themselves are mined, usually one row at a time, leaving the mined-out area, or gob, free to collapse. Room-and-pillar is not only used in coal mining, it is also ideally suited in the production of potash, sodium chloride, trona, limestone and any metallic deposits that occur in horizontal seams.

This thesis uses data from a Longwall mine, therefore this section will focus on the machinery used with a Longwall operation. In Longwall mining large blocks

**Figure 2.5: Longwall Shearer at Dawmill UK** (UK Coal)

of coal, usually 100 to 350 metres wide and 1,000 to 3,000 metres long, are developed for complete extraction. In the development stage two headings are driven from the main roadway, the width of the block to be mined, these are known as the main-gate and the tailgate. Once the headings are at the required length, another tunnel is developed to join the two; along this tunnel is where the exposed face of coal to be mined is situated. The roof is supported by roof supports, which form a protective steel canopy under which the face conveyor, workers, and shearer operate (Figure 2.5). The 'Shearer' cuts coal back and forth horizontally along the Longwall face. The roof supports move into the face after each cut allowing the roof to collapse behind, the armoured face conveyor (AFC) then removes the coal from the face ready for the next cut. From the AFC the coal passes through a primary crusher. After the crusher the stage loader transport the coal onto the mines conveyor system, then to the pit bottom for transport to the surface. In combination with roof supports and conveyors, Longwall shearers create a truly continuous mining system with a huge production capacity. Record productions exceeding 20,000 tons per day, 400,000 tons per month, and 3.5 million tons per year have been reported from a single U.S. Longwall shearer face (Britannica, 2011). In 2008, Daw Mill Colliery in Warwickshire, UK, mined over 3.25 million tonnes from its 5 metre face (Macalister, 2008) and in Oaky North, Australia the Oaky North Colliery won 6.6 million tonnes in 2011 (Cram and Mische, 2012).

Room-and-pillar and Longwall mining machinery is virtually all electric with a small number of diesel or battery powered vehicles for transporting machinery and equipment around the mine, generally for development. Room-and-Pillar may also employ loaders or shuttle cars for transporting the coal to the continuous haulage or conveyor system. Compressed air may be also used mainly for the development machinery, such as roof bolter's and drilling rigs. Table 2.3 shows the average power demand for Longwall mining machinery.

| Machine | Power Demand (kW) |
|---|---|
| **Amoured Face Conveyor (AFC)** | 800-1600 |
| **Shearer** | 400-800 |
| **Stage Loader** | 150-300 |
| **Crusher** | 100 |

**Table 2.3: Longwall Machinery Power Demand (Joy and CAT)**

## 2.3.3 Coal Preparation

Whatever the mining method used for the extraction of the coal, once it reaches the surface it is sent to the coal preparation plant (CPP) also called coal handling and preparation plant (CHPP). Mixed with the run-of-mine (ROM) coal, is shale, rock, dirt, bits of wood and metal, what are removed at the CPP. The CPP is a building or collection of buildings linked with conveyors transporting the coal through process and usually located near the mine shaft or outbound conveyor (Figure 2.6).

The CPP is an automated process that cleans and grades the coal, taking it from a raw commodity to a saleable product. The coal is graded, sized, blended and processed to fulfil precise requirements and is subjected to strict quality control. It is not only the size of the fuel that is important, but also how it burns and the ash

**Figure 2.6: Coal Preparation Plant** (UK Coal)

that is left behind. For example - power plant boilers are designed for bituminous coal with low to medium moisture, medium to high volatile matter and, within a reasonable calorific value range 'compliant coal'. If the coal does not meet these specifications, reduction of performance and excessive ash build up can occur within the boiler, requiring premature maintenance and down time.

Firstly the ROM coal is sent to a stockpile, beneath the stockpile is a reclaim tunnel. A loader pushes coal through a grill into the reclaim tunnel, the coal is then transported to a raw coal silo, and from the silo the coal is feed to the CPP at a constant rate. The first stage is crushing and screening, the crushed coal goes through screening, for coarse (sizes 2- 10 mesh), middling (10 mesh - 60 mesh) and fine (60 mesh and smaller).

The CPP can also contain, coal mills, pulverizers to reduce the coal size for the specification of the final product. The separation stage usually uses gravity process equipment, heavy media is the most popular method for cleaning coarse coal. Heavy media uses a suspension of dense powder such as magnetite or ferro-silicon in water to act as an artificial 'heavy liquid' with a specific gravity in accordance to the mineral being separated (1.45 for coal). The separation principle is quite simple in the fact that anything with a specific gravity less than that of the liquid will float, and anything with greater will sink.

Jigging separation is probably the second most popular method for coarse coal cleaning, this is where a mineral bed has a stream of water pulsed through it in backward and forward. The result is that the mineral bed separates into layers, the heaviest at the bottom and lightest on the top. Spirals are generally used for middling sizes (10 mesh to 60 mesh). Spiral separators use gravity and centrifugal force in the separation process, the coal is feed down a spiral chute, the heavier particles move out further to the edge as they travel down the chute. At the bottom of the chute are separate collection points for the different zones of the chute.

Froth flotation is generally used for separation of the -28 mesh size. Froth flotation is the separation of minerals with little difference in density, but which have a larger difference in wettability by surface-active agents that stabilize a froth formed on the surface. The froth is agitated by air bubbles through the liquid. The froth containing the coal is drained off leaving the refuge behind at the bottom of the liquid. Spirals and heavy media cyclones have shown success down to 100 mesh coal and heavy media separation is being used more often for fine grade separation. These process methods are not unique to coal and can be found in all types of mines where mineral processing is needed. They are just tailored to the properties of the individual minerals under separation.

Once screened, the coal is then sent to the next stage which consists of a series of thermal dryers to reduce the moisture content and thereby raising the BTU value of the coal. The final stage is to transport the dry coal to a clean coal silo, from where it is loaded, in most cases, on to a train for transportation to a power station or steel mill.

## 2.4 Summary

Energy and Mining are the two major themes of this thesis, this chapter has given a broad look at the world energy resource and highlighted the prominent hold coal still has on world power generation. The amount of the world's energy that is used by the mining industry and the major locations it is used in has also been presented. The problem mining as an industry faces when it comes to its power requirement and how some of these can be overcome has also been discussed.

Throughout the discussion, it has been identified that energy efficiency and demand reduction are processes that will contribute to meeting global environmental and energy targets. The next chapter discusses the history of demand management from the beginning to the modern day, how it has progressed and the problems faced when moving to a modern power grid.

# 3

# Demand Side Management and Smart Grid

Demand side management (DSM) is an approach or system that a utility or power generation company would utilise in order to change the power demand profile of its customers. To change their behaviour in a way that is more of an advantage to the utility and the availability of excess power generation. In other words:

> *"Demand side management is the planning and implementation of those electric utility activities designed to influence in ways that will produce desired customer uses of electricity changes in the utilities load shape." (Gellings, 1985)*

## 3.1 The History of Demand Side Management

Load management entered the scene in the 1960's and 1970's with early activities in Europe and New Zealand, but it was not until the late 1970's that the United States saw the emergence of DSM. The 1973 Oil Embargo, and the Iranian Revolution of 1979, sparked awareness for energy conservation. The first wave of DSM started in California and mainly concentrated on the design and implementation of 'energy conservation and load management' (C&LM) programs. Unlike later DSM incentives, price was (due to political reasons) taken as given and was not recognised as reflecting the new marginal costs. Instead of price changing, other ways had to be found to give customers the incentive to reduce demand. This energy crisis resulted in DSM initiatives that were designed to produce quick results, together with the increasing public opposition and unpredictable timings in the construction of new power stations, programs concentrated on load reduction. There was very little investment of time or money, and DSM mainly depended upon 'soft' efforts such as audits and reporting. Large commercial and industrial customers saw the introduction of time-varying pricing and in the U.S. time-of-use (TOU) pricing schemes were piloted on residential customers.

By the early 1980's the DSM focus in the United States had accelerated and concentrated on achieving several different load shape objectives. These included load management, energy efficiency and electrification. It was considered that DSM would play a major role in utilities resource planning, and in response, alternative methods of meeting demand were needed. This brought the introduction of 'least cost planning' (LCP). "LCP consistently assesses various demand and supply resources to meet customer energy-service needs at the lowest economic and social costs" (Goldman et al., 1989).

Toward the end of the 1980's, concerns surfaced that expenditure on the energy efficiency and conservation programs was resulting in loss of revenue for utilities. These concerns were justified -- the DSM spending was reducing sales. However, utilities have large fixed costs that have to be met regardless of how much power was sold. Recognising the large fixed costs meant that several 'decoupling mechanisms' would have to be implemented to ensure that these costs were covered, regardless of sales. A small increase in electricity rates were made to cover the revenue deficit due to lowering sales.

By the early 1990's, new DSM program regulations, including decoupling mechanisms, cost recovery and incentives to shareholders for investing in energy efficient programmes, made DSM attractive to utilities. In the UK there was great controversy about the lack of incentives to improve energy efficiency, and scepticism grew over the effectiveness of the DSM implementations in the UK. The privatised energy utilities regulatory framework had a strong supply-led orientation, left over from the nationalised era and provided little or no incentive to the utilities to invest in energy efficiency operations - instead they maximised consumption to increase profit. In the US DSM continued to grow and in 1993, United States utilities spent $3.2 billion on DSM programs with 447 utilities having programmes in place, resulting in energy savings of 44,300 GWh, 1.6% of retail sales (Hadley and Hirst, 1995).

The main focus for DSM programmes was energy efficiency for customers, the utilities would provide subsidies for energy saving measures such as efficient heating systems, appliances, lighting and insulation. DSM was not only restricted to the US, environmental and cost driven DSM programmes were also implemented in Canada and a number of European countries, including the UK. The Office of Electricity Regulation (OFFER) re-evaluated its position on energy efficiency, producing a series of policies and technical papers on energy efficiency. Then in November 1992 OFFER announced that it intended to take energy-efficiency considerations fully into account during its 1994-1995 price control reviews. The electric distribution companies and British Gas also participated together in a joint Energy Saving Trust (Woolf and Lutz, 1993).

By the mid 1990's the utilities were under serious threat from independent power suppliers, using natural gas-fired, modular combustion turbines. Utilities introduced a wide range of cost cutting measures. Programs that imposed an increase in the rates were eliminated or reduced in size. DSM fitted into this category and as a result the spending on DSM fell dramatically as utilities prepared for competition. In the late 1990's regulators became aware of the decline in DSM expenditure and introduced a 'public goods charge' to cover the costs of DSM implementations. This was charged by the distribution utilities and had to be paid regardless of the actual power provider.

The current wave of DSM was triggered in 2000 by price spikes in the wholesale power markets and received a boost in 2001 when California experienced a serious power crisis caused by market manipulations, illegal shutdowns of pipelines by the Texas energy consortium Enron (Gonzales, 2004), and capped retail electricity prices (Sweeney, 2002). This crisis quickly spread across all Western States in the US. Pricing reform was receiving growing attention as opposed to traditional DSM programs, with dynamic pricing attracting most of the attention - being a form of time-varying pricing that goes beyond static TOU pricing. With dynamic pricing, either the rate at a specific time or the time of a specific rate is unknown. Industrial and commercial customers have had a form of this available for years as real-time pricing (RTP), RTP is retail pricing that varies (generally half hour by half hour) with the wholesale electricity price. Today's technological advancements have made it possible for dynamic pricing to enter the mass market with the introduction of smart meters to all customers domestic, commercial and industrial alike.

DSM has continued to grow across the globe. In many countries, utilities are still actively promoting traditional DSM techniques for example; Eskom's 'Energy-Efficient Motor Programme' (2007-2009) subsidising the replacement of standard efficiency Eff3 electric motors with high efficiency Eff1 motors (Parker, 2008). In the US, the market is dominated by high efficiency motors, and since 2011, the Energy Independence and Security Act(EISA) legislation imposes to sell exclusively the most efficiency National Electrical Manufacturers Association (NEMA) Premium motors, equivalents to IEC IE3.(Benhaddadi et al. (2012)).

## 3.2 Smart Grid

Although DSM continues to evolve, the electrical utility power systems today looks much like it did more than seventy five years ago. Generally they are still planned, constructed, and operated with the same theories, topologies, technologies and tactics. In fact, some of the key technologies are more than one hundred years old; the electromechanical or moving coil meter is one example that is still in use today. The existing electricity grid is unidirectional in nature. It converts only one-third of fuel energy into electricity, without recovering the waste heat. Almost 8% of its output is lost along its transmission lines, while 20% of its generation capacity exists to meet peak demand only (i.e. it is in use only 5% of the time). In addition to that, due to the hierarchical topology of its assets, the existing electricity grid suffers from domino effect failures (Farhangi, 2010).

The next section covers the existing and developing trends and technologies, explaining the basic operation, advantages and possible hurdles that need to be overcome to realize the 'Smart Grid'.

### 3.2.1 What is a smart grid?

Almost all of the papers reviewed to date by the author open with a Smart Grid description, which is more speculation than definition, "As the problem with the Smart Grid, nobody really knows what it is. Pretty much everybody agrees that we need it for varying reasons and to various degrees" (Collier, 2009). Smart Grid is emerging technology, Tai and Hogain (2009) state "The Challenge here is that there is yet to be a fully deployed, demonstrable Smart Grid in the industry." Although the Smart Grid is still undergoing major development, Figure 3.1 shows a schematic of a proposed development and Table 3.1 outlines the principle characteristics.

The Smart Grid will be self-healing and tolerant of attack, manage decentralized generation, incorporate renewable energy sources and facilitate storage options. McDonald (2008), Tai and Hogain (2009), Farhangi (2010) all discuss technology and centre around advance metering infrastructure (AMI) with robust 2-way communication being at the core of Smart Grid functions. Providing the utility with the ability to modify customers service plans, they can also meet the basic targets for load & volt/var control and asset protection. Load control and demand balancing are one of the main objectives of the modern smart grid. Plug-in hybrid electric vehicles (PHEV) feature in most of the Smart Grid definitions.

Tai and Hogain (2009) highlight the expansion of AMI to include home area networks (HAN) and that they are the 'new frontier for innovation and investment. The HAN value chain is increasing with activity ranging across all components: the gateway; in-home display (IHD); smart thermostats, switches, appliances, and energy interaction software; and DSR program management. A wide variety of investors are actively participating, including venture capital companies, AMI and grid application vendors, appliance manufacturers, and systems integrators. Standards are beginning to emerge, with ZigBee (a wireless mesh network protocol specifications) establishing significant momentum, followed by HomePlug (a power line communications protocol specifications). They also indicate grid side applications are gaining pace over the past two years, enhanced grid applications, the core of which are automation, volt/var applications, and asset monitoring-have begun to take a significant role in utility and vendor thinking about

# 3. DEMAND SIDE MANAGEMENT AND SMART GRID


**Figure 3.1: The Smart Grid** (Renewable Energy Focus 2009)

| Principal Characteristic | Smart Grid |
|---|---|
| **Self-heals** | Automatically detects and responds to actual and emerging transmission and distribution problems. Focus is on prevention. Minimizes consumer impact. |
| **Motivates & includes the consumer** | Informed, involved and active consumers. Broad penetration of DSR. |
| **Resists attack** | Resilient to attack and natural disasters with rapid restoration capabilities. |
| **Provides power quality (PQ) for 21st century needs** | Quality of power meets industry standards and consumer needs. PQ issues identified and resolved prior to manifestation. Various levels of PQ at various prices. |
| **Accommodates all generation and storage options** | Very large numbers of diverse distributed generation and storage devices deployed to complement the large generating plants. "Plug-and-play" convenience. Significantly more focus on and access to renewables. |
| **Enables markets** | Mature wholesale market operations in place; well integrated nationwide and integrated with reliability coordinators. Retail markets flourishing where appropriate. Minimal transmission congestion and constraints. |
| **Optimizes assets and operates efficiently** | Greatly expanded sensing and measurement of grid conditions. Grid technologies deeply integrated with asset management processes to most effectively manage assets and costs. Condition based maintenance. |

**Table 3.1: Smart Grid Characteristic -** (NETL, 2007).

smart-grid applications. In the near future, the focus will shift from HAN to functions that the Smart Grid enables. Specifically, these functions include the integration of renewable energy sources, co-generation, and PHEV, grid-to-vehicle and vehicle-to-grid, the latter being at least '5-10 years away' (GTM, 2009), as well as the necessary forms of distributed storage that these alternative energy technologies bring with them.

### 3.2.2 Digital Communication

The front-end communication technology will get very complicated. Two-way digital communication for remote monitoring and control devices is key to the Smart Grid success. Since the Smart Grid is about real-time data and active grid management, fast, digital, two-way communications will be required throughout the smart grid. Collier (2009) discusses the structure of the Smart Grid, and the difficulty in having common communication protocols What standard should be used? Today, the question is more complicated with options spanning power-line, broadband, and radio frequency spectrum-based wireless technologies as well as emerging high speed WiMAX (Worldwide Interoperability for Microwave Access), and the established mobile data protocols 3G and 4G or LTE (long-term evolution). While communication technologies are certainly evolving, utilities should feel encouraged that open standards, often IP based, are fast becoming the expectation for emerging technologies (McDonald, 2008).

Collier (2009) outlines the Smart Grid software system framework, with digital communication, decision and control software, implemented in a decentralized and back office manner. The huge number of devices and amount of data will prevent centralized data collection and computation. Instead, the devices will increasingly be intelligent electronic devices (IED) that can collect, organize and analyse data as well as perform computations to determine what data should be communicated where and what local control actions may be necessary. Some of the devices will have firmware built in, and some will be programmable settings and functions. An example would be a smart meter that would likely have firmware to monitor and record certain variables like voltage, current, power factor and to compute KW and KWh. It might also have programmable capabilities to determine if and when to report variables outside certain ranges. The back office software is already in use by many utilities, systems such as Enterprise Resource Planning (ERP), Accounting & Business Systems (ABS), Customer Information Systems (CIS) and Geographic Information Systems (GIS).

### 3.2.3 Smart Grid Standards

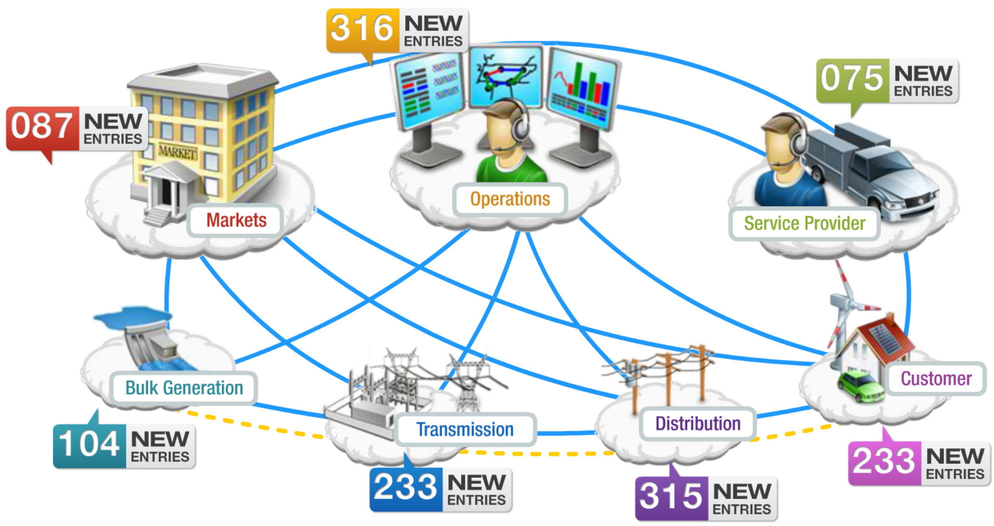



**Figure 3.2: IEEE Smart Grid Publications** (IEEE, 2013)

As with any emerging technology, initially there are no or very few fixed standards; environmental constraints and locations often depicted the system to use. However, as research and development (R&D) accelerates this rapidly changes. In Figure 3.2 are the total number of IEEE Smart Grid related publication submissions from 2010 to 2013. IEEE is working closely with the National Institute of Standards and Technology (NIST), which is developing a standards roadmap and conformance testing/certification framework for the Smart Grid, The NIST report describes a high-level reference model for the Smart Grid and identifies nearly 80 existing standards that can be used now to support Smart Grid development. The report also identifies high priority gaps for which new or revised standards are needed. IEEE currently has over 100 (including over 20 named in the NIST Framework) Smart Grid approved standards and standards-in-development, addressing interoperability of electric power, IT, and communications systems;

power-line communications; regenerative interconnection; automotive interconnection; storage; and specifications for integrating renewable energy.

### 3.2.4 Smart Grid Security

A major concern about the Smart Grid is 'security of supply', not only the physical, but with full integration and network based monitoring and control, the threat of a cyber-attack on our electricity supply is very real. The addition of millions of sensors and smart meters dramatically increases the number of points that could be targeted and become potentially vulnerable to cyber-attack. However, while these concerns should not obstruct the implementation and the deployment of Smart Grid technologies, they do need to be adequately addressed by governments, utilities and companies providing grid hardware and software. Geater (2009) the director of technical strategy, Information Systems Security at information security and encryption specialist Thales, states "Many of the challenges with smart grids are very similar to the traditional challenges that we have seen in the information security market for quite a long time. You have to identify where the points in your communication chain are, you need to know how to trust them, and you need to secure the information that flows between them."

It is not very likely that someone would hack into the consumer smart energy system and turn off a whole power station, but obviously, the more features you put in, the more entry points there is for an attack or accident. Most smart meter units, have the capability of embedding an 'encryption key' or other form of identity. However, only securing the meters themselves is not enough, keeping track of the information and having security on each link and encrypting data sent between households and utilities is a must. Geater (2009) also points out that the data could also be used for criminals and law enforcement surveillance, people's work/home, and lifestyle patterns. Putting security in place at the same time as the infrastructure is being put together - it is much easier to design a secure system than to secure an existing one.

### 3.2.5 Environmental Impact

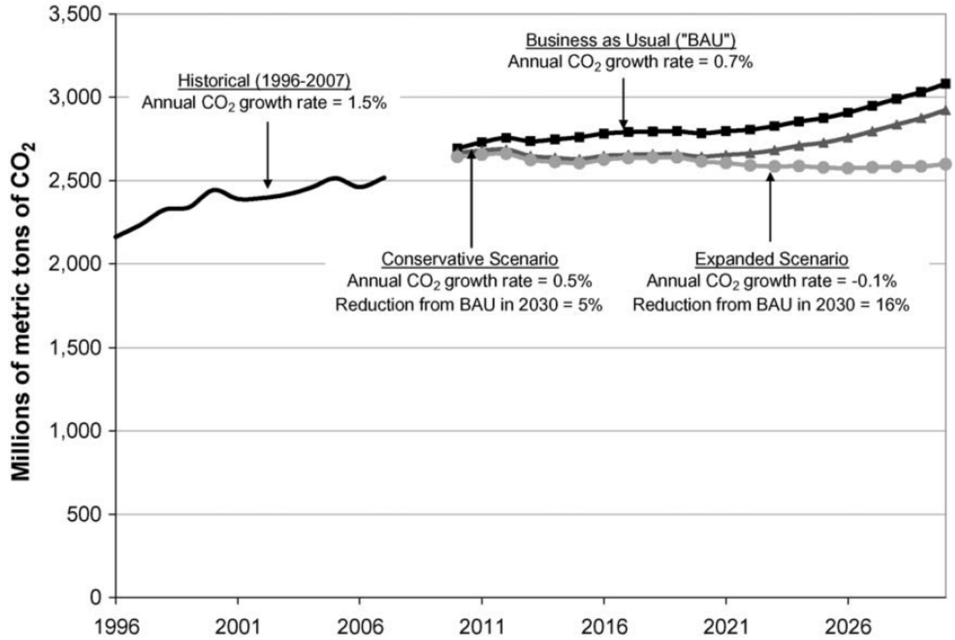



**Figure 3.3: U.S. Smart Grid $CO_2$ Emissions Projections 2030** (Hledik, 2009)

Running a national grid to optimum efficiency compared with today's inefficient operation will save carbon, shifting peak loads off peak will reduce generation, transmission and distribution losses and the general consensus is that modernizing the grid will reduce our carbon foot print. Hledik (2009) evaluates two scenarios, the conservative scenario, using only existing technology (AMI, dynamic pricing, automated technology and IHD's) and the expanded scenario, using the conservative scenario plus the inclusion of 'Smart Distribution' (higher renewable energy generation and distributed storage). PHEV's were considered for the scenario, but were left out because of the lack of development in the 'vehicle-to-grid' areas. Figure 3.3 shows the projected $CO_2$ reductions for the U.S. power sector for two different scenarios these are, 5.1% for the conservative scenario and almost 16% for the expanded scenario. Better integration of renewable energy is one of the key characteristics of the Smart Grid, Potter et al. (2009) concentrates, using meteorological information for better prediction of renewable generation, this will allow for better forecasting for load control and reduce non-renewable power requirements.

### 3.2.6 Smart Grid and Modern DSM

The traditional way of creating DSM incentives is through pricing, i.e. by lowering tariffs when below average demand is expected, so to encourage customer's flexible load shifting to the cheaper periods. Lack of automation and technology has restricted the consumer pricing tariffs to static day-rate and night-rate. DSM was designed to encourage customer behaviour on a long term basis. The Smart Grid opens up a whole new world for DSM, with the increase in DER. These include renewable resources and combined heat and power units (CHP) connected to the distribution or transmission network.

The fluctuating nature of renewable resources will make strict time-based tariffs inefficient. The modern Smart Grid will require dynamic, reactive pricing mechanisms to allow for the real-time availability of a partly unpredictable power supply. As stated by Saffre and Gedge (2010) the energy price charged to customers can vary significantly according to the time and location of the electricity consumed. This effectively means that the cost of power increases as a step function of the known aggregated load on every future time-step. It is easy to understand intuitively that this reactive pricing mechanism tends to prevent the clustering of loads in time as, the higher the demand, the higher the unit-cost and the higher the probability that new loads will enter stand-by mode, so as to avoid 'crowded periods'. When a price shift is initiated due to a new load starting, a notification broadcast is sent to customers notifying the tariff change.

DSM in Smart Grid combined with smart meters and AMI can deliver energy prices that accurately reflect the supply chain of delivering electricity. Customers are in control of their power consumption, influenced by the penalty and incentive regime. However, the main objective of DSM in the Smart Grid is to fully utilize the existing electrical network to promote the overall system in efficiency, security and sustainability and the integration of low carbon technology. Tianyu et al. (2010) state "DSM is an effective solution, which contributes to future distributed networks in various aspects, i.e. Smart Grid and smart devices, electricity market,

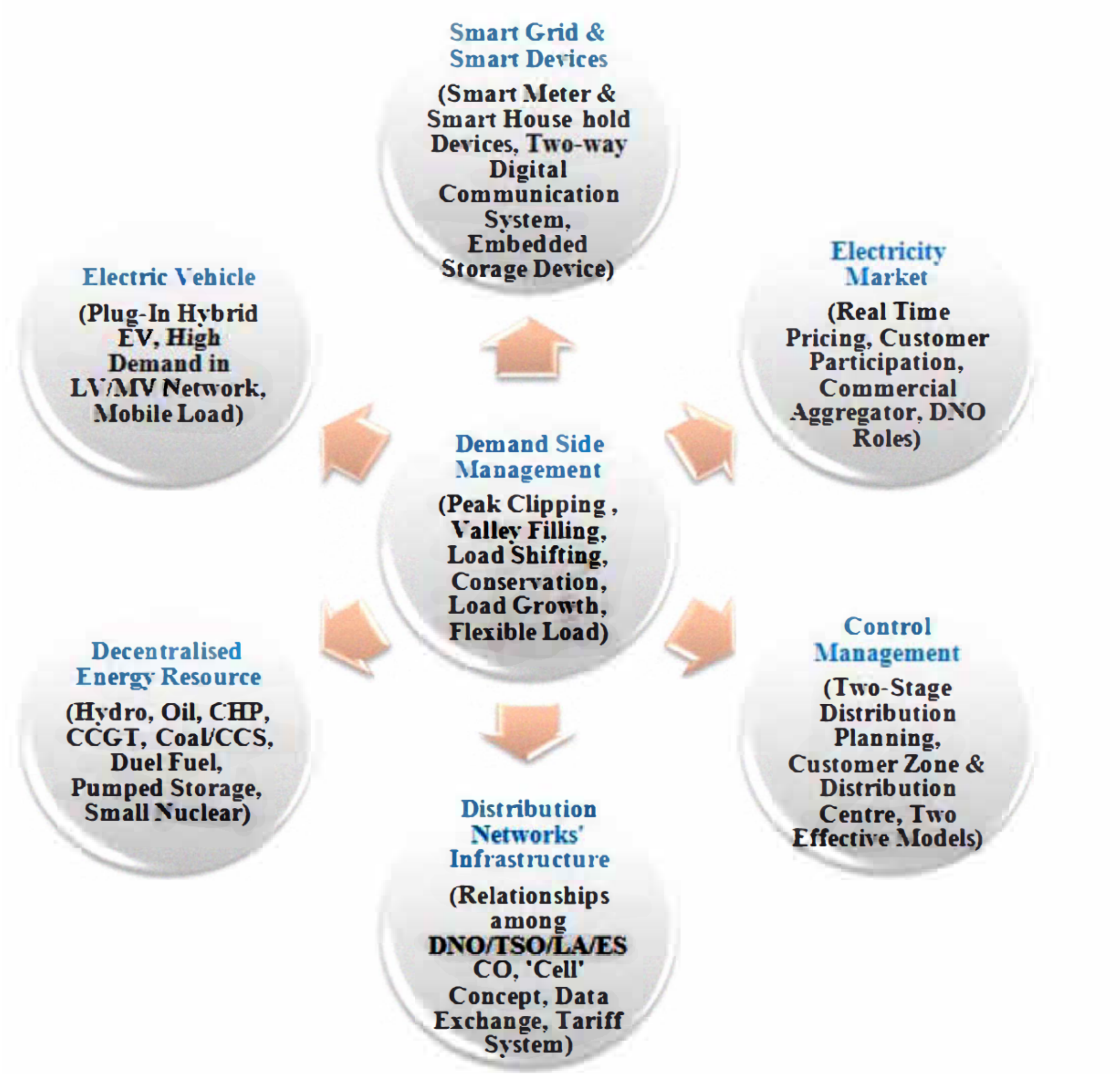



**Figure 3.4: Demand Side Management Contributions** (Tianyu et al., 2010)

control management, infrastructure construction and distribution network operators (DNO), and introduction of decentralized energy resources and electric vehicles". The relationship of these key DSM contributions are shown in Figure 3.4, the details of these individual components are also listed.

### 3.2.7 Demand Side Response (DSR) and Direct Load Control (DLC)

Modern DSM techniques are better described as DSR techniques, as this is in actual terms the final goal - to 'respond' to a market shift. This dynamic price change should reflect the cost and availability of the power, with decentralised generation systems can varying due to geographic location. Either the response is to increase or decrease demand depending on which way the price has changed. Another significant development that has come from the Smart Grid revolution is advancement in direct load control (DLC). Electrical loads such as air conditioners and water heaters are interrupted, remotely shut-down or cycled. Intelligent thermostats can be programmed to respond to a given tariff rate, to invoke the DSR and implement short term DSM.

Although DSR and direct load control (DLC) (sometime referred to as load management (LM)) are nothing new, DLC is the oldest form of dispatchable DSM. DLC programs began full-scale operation in the early 1970's using a basic, usually one-way radio communication technology signalling the operation of load control receivers, allowing utilities to directly manage participating loads. Many DLC systems that were launched as far back as the late 1970's still remain in use throughout the United States (Weller, 2011). DSR programs and DLC generally offer financial incentives for the customer, DLC is none voluntary and used for grid stability and to reduce the requirement for additional power plants or bringing on-line peaking power plants (expensive power-plants only used to provide power during peak periods).

DSR programs will also offer voluntary load control where the load control switch is in the control of the customer, where they may chose how they respond to the demand response request from the utility. DLC can be considered a 'state of the art' Smart Grid DSR technique. However, as opposed to traditional DSR, the Smart Grid DSR will have two way intelligent communications, that will carry all information about the loads size, state, requirements, etc. with better selection of loads for DLC and better timing to provide even better grid stability.

### 3.2.8 Smart Grid Development and Electric Vehicles

Smart grids are the future, they are not a replacement of the existing grid, but a modernization of the existing infrastructure. Along with the installation of communication and monitoring equipment giving a complete picture of the grid stability. Plug in hybrid electric vehicles (PHEV), at first glance, seem to be the answer to low carbon travel. However, if everyone replaced their cars with PHEV's then the demand on the grid could double. How would this increase in demand be supplied? Possibly from renewable sources or off peak charging, PHEV's could be considered as a storage device, charging up overnight/off peak periods, driven to work and reconnected to the charging point/grid. Smart batteries could supply AMI information to the grid, so during peak demand the stored energy in the batteries could then be used for local peak demand management. The problem with this is that the PHEV's would require an increase in battery size and redundancy, if the vehicle is to be of any practical use after supplying peak demand through the day. So by construction and cost, at present it seem that this is not a viable idea as customers would have to be prepared to foot the bill. Also this does not consider the continuing wear and tear on the batteries themselves. Although in theory this seem like an ideal solution, at present electricity generation is far from green.

PHEV are a step in the right direction; however, there is still a long way to go in development. As standards and data protocols are slowly confirmed, individual manufactures must adopt these instead of their own. Control software that communicates with other grid equipment, such as meters, motors, lights, heating, electrical consumer goods (e.g. LCD back lights) must do so with a common language. Standardized definitions for device types that state what information the device can provide and the commands it understands are necessary. A robust, secure standard of communication is needed to overcome the possible conflicts in vendor to vendor compatibility and security issues. Cross compatibility is a must for the interconnecting of individual micro-grids to function as a farm of micro-grids, mini-grids. Each having its own local generation, interconnected and communicating with each other and the central utility. To manage the demand, the ideal situation would be to supply a constant load via transmission and the local generation/storage would supply the peak demands. The grid's across the globe collectively are extensive and to 'smarten' them up will take many years. Ever progressing technology and backward compatibility will also become quite an issue, constantly having to upgrade software and even hardware could be costly. It will take time, but once in place the Smart Grid is the future of electricity generation, transmission and distribution.

### 3.2.9 Load Shedding

Although not defined as part of the Smart Grid, load shedding (LS) will play a large part in the integration of the smart grid, balancing local generation with local demand and integrating with DSR systems to ensure a balanced and stable network. In industrial countries across the world blackouts are becoming more frequent, these blackouts are caused by increasing load and network deficiencies. Load curtailment is one possible solution for black out prevention. Both DSM and LS have been used to provide stable power network operation. Although DSM is generally devoted to peak demand reduction and the encouragement of efficient energy use, it uses particular tariff conditions and operates in a preventive way. LS is still a methodology used worldwide to prevent power system degradation to black outs and it acts in a repressive way.

When DSM and LS are used for the management of transmission network deficiencies and network stability, it is generally associated with small differences between the absorbed power including the net losses (superior value) and the generated power (inferior value). This difference is often less than 1% of the total absorbed power (Faranda et al., 2007), as during the blackout in August 1996 in the Western North American grid. On 10 August 1996, faults at the Keeler-Allston 500 kilovolts (kV) line and the Ross–Lexington 230 kV line in Oregon resulted in excess load, which led to the tripping of the generators at the McNary Dam, causing 500 MW oscillations, which led to the separation of the North–South Pacific inter-tie near the California–Oregon border. This led to islanding and blackouts in eleven U.S. states and two Canadian provinces. It was estimated to cost from $1.5 to $2 billion and included all aspects of the interconnected infrastructures and even the environment. Among several studies that followed, some researchers have shown that a dropping (shedding) of about 0.4% of the total network load for 30 minutes would have prevented the cascading effects of this blackout (Amin, 2001).

The basic operation of all load shedding techniques is fundamentally the same, when a trigger condition is met a load is dropped. The system condition trigger may be an under frequency, under voltage, trip situation or a combination. When a trigger threshold is exceeded, commonly due to additional load coming on line, less important loads are simply dropped. The dropped load must be of necessary magnitude to restore the system threshold and network stability. The duration of the load shedding period is usually short, minimising interruption. This in some cases it maybe a combination of multiple loads that are equal or greater than the new load. Exactly what loads could be considered for load shedding would require some analysis to prevent critical loads from being dropped; a customer's loads would be split into interruptible and non interruptible parts. It also may be possible that a load could be shed in only certain conditions, such as when another load is only on or off line. Listed below are the conventional and modern load shedding techniques:

- Breaker Interlock Scheme.
- Under Frequency Load Shedding (UFLS).
- Under Voltage Load Shedding (UVLS).
- Intelligent Load Shedding (ILS).

#### 3.2.9.1 Breaker Interlock Scheme

The simplest form of load shedding, supply breakers are interlocked with the load breakers, by either hard-wired or remote signalling. The load breakers are pre-selected depending on the loss of supply. This method of load shedding is very fast since there is no processing required and the decision of which load is to be dropped was made long before the faults occurred. For example, a load is supplied by a generator and utility combination, instability outside the installation causes the main breaker to trip and open. The isolation of the installation from the power grid would then cause the load to be solely supplied by the generator. The opening of the utility supply breaker would signal the opening of the interlocked load breakers almost instantaneously. This pre-selected breaker interlock list is typically determined without any knowledge of system transient response and is often too conservative, resulting in unnecessary load shedding (Shokooh et al., 2005).

Other disadvantages of a breaker interlock scheme load shedding are that the load priority is difficult to change due the hard-wiring and the load shedding is calculated for the worst-case scenario. The load shedding operation will most likely result in the shut-down of an entire industrial facility. This unplanned outage could cause equipment damage, reduce equipment lifetime and delay facility restarting due to the necessary shut-down of remote facilities that rely on the main facility. Allen and Lee (2006) introduce the intelligent high speed cross-point switches, fast acting like the interlocked breaker, but with the added benefit of an intelligent control and load-shed tables that carry load details, enabling the switch to drop different loads depending on the shedding requirements at the time.

#### 3.2.9.2 Under Frequency Load Shedding

UFLS schemes are based on system wide dynamic performance, frequency relays are used to react to a disturbance. They detect either a rapid change in frequency and or a gradual deterioration. Once a change is detected, after a time delay to prevent nuisance tripping, an initial stage of shedding is started by tripping a set of load breakers. The frequency is then allowed to recover, if the frequency does not recover, then the next stage of load shedding is employed. Additional stages of load shedding is continued until the frequency returns to normal. After an under frequency load shedding event, frequency relays can be utilized to automatically restore or supervise the restoration of load to a power system. Sufficient time delay should be employed to assure that the power system is stable prior to initiating load restoration. IEEE (2007), Maliszewski et al. (1971) and Horowitz et al. (1971) give a full description of under frequency load shedding and restoration. Although widely used Shokooh et al. (2005) highlights some inherent drawbacks including, slow response and incorrect load shedding causing undesirable blackouts.

#### 3.2.9.3 Under Voltage Load Shedding

UVLS is believed an economical or partial solution to overcome voltage stability problems faced by electric utilities, analogous to the use of under frequency load shedding in other circumstances. Following the 1965 North-East blackout, application of under frequency load shedding became accepted utility practice (Taylor, 1992). As power systems have matured, however, with a large generation load imbalance islanding is less likely than voltage problems. As with under frequency load shedding, under voltage load shedding provides protection for unusual disturbances outside planning and operating criteria. Under voltage load shedding may also be very desirable because of potential delay of planned facilities. In general, uncontrollable system-wide voltage collapse is triggered by voltage instability in one or several heavily loaded regions weakened by transmission outages, or subject to reactive shortages. Therefore, applying UVLS in these regions is most appropriate to mitigate voltage collapse (Jianfeng et al., 2005).

#### 3.2.9.4 Intelligent Load Shedding

Conventional methods of system load shedding are too slow and do not effectively calculate the correct amount of load to be shed. The result is excessive or insufficient load shedding. Due to these inherent drawbacks of existing load shedding methods, an ILS system is necessary to make a fast, optimum, and reliable load shedding decision, accurately predict the system frequency decay and improve the response time. Shokooh et al. (2005) promotes the need for ILS and describes a working example of ILS system was recently installed at PT Newmont Batu Hijau, a mining plant in Indonesia. Replacing the existing frequency relay load shedding scheme, this, due to its slow response, would often drop too much load resulting in significant impact on production costing an average of £200,000 per day. ILS combines system on-line data, equipment ratings, user-defined control parameters; a knowledge base obtained from off-line system simulations, system dependencies, and continually updated dynamic load shed tables. From the initial network disturbance the ILS can perform load shedding in less than 100 milliseconds.

## 3.3 Summary

This chapter has introduced DSM, DSR, DLC, LS and Smart Grid, the hurdles that need to be overcome when such new technology is to be adopted by so many different parts of one industry. With this in mind, it is somewhat overwhelming to note** how integrated the final Smart Grid must be for the vision to become a reality. The importance of standards has been highlighted and just how many are required to cover all aspects and to ensure companies do not waste development time reinventing the wheel or become obsolete before they even get off the ground. DSM, DSR, DLC and Smart Grid, are now understood in terms of the definition and characteristics, with a breakdown of the Smart Gird components, automatic meter reading; digital communication; and security. The Smart Grid has the potential to reduce the environmental impact of electrical energy, the introduction of smart appliances and dynamic pricing will help this happen. The next chapter presents DSM within the mining industry and the current research. Also presented is common energy saving practices and how they can be applied in the mining industry.

# 4

# Mine Electrical Systems and Three Phase Electricity

## 4.1 Underground Mine Electrical Systems

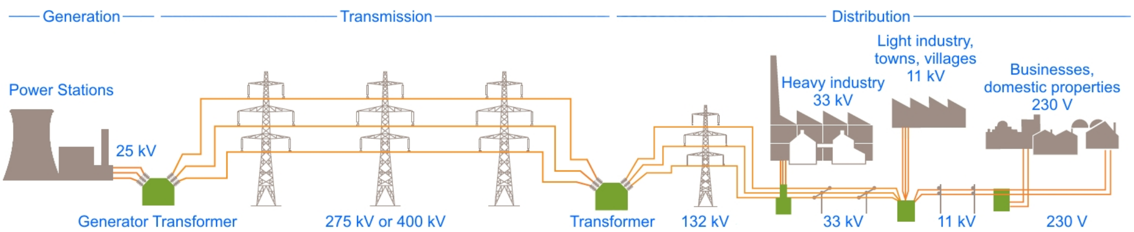



**Figure 4.1: Typical Electricity Supply Network** (National Grid)

Large underground mines, such as underground coal mines require a vast amount of energy to operate the large mining machinery and distribute it over considerable distances. Electrical power (P) is the product of voltage (V) and current (I) ($P = VI$). Distribution cables have an internal resistance that is proportional to its cross sectional area, this internal resistance causes a voltage drop ($V_{drop} = IR$) and a loss of power ($P_{losses} = I^2R$) down the length of the cable, the higher the current the higher the losses. This loss of power is converted to heat and causes internal heating of the cable and subsequently this increases the internal resistance and voltage drop, for this reason a cable has a maximum current carrying capacity proportional to it cross sectional area. To compensate for this internal resistance and to reduce the voltage drop to an acceptable level, it is necessary to distribute power at high voltages in order to reduce the current, for a given power. In the UK national transmission voltages can be as high as 400kV (higher in special circumstances) and distributed to the end user at 33kV for large industrial, 11kV for medium/light factories and 415/230V for commercial/residential customers, Figure 4.1 shows a typical supply network. High voltage direct current (HVDC) is also used for bulk transmission of electrical power over long distances, such as submarine cable links or from renewable sources such as hydro and wind. Today nearly all utility generated electricity is three-phase alternating current (AC). All three-phases are distributed to the customer and are loaded evenly with balanced loads and or even distribution. The exception to this is in areas of low power requirement, such as residential use, where the three-phases are separated and connected to an equal amount of houses to maintain balanced conditions.

As a large industrial user, a typical underground coal mine would have a 33kV grid connection, then transformed down to 11kV at the site entrance and distributed to the surface substations at this voltage. Some large surface equipment, such as the ventilations fans and winders will be powered at 11kV or 6.6kV depending on the size. Other machinery voltages can be 3.3kV, 1.1kV, 600V, 550V, 415V and 230V (single-phase), the surface and underground distribution must have transformers to supply all the required different voltages. Currently underground power is distributed to underground substations at 6.6kV or 11kV, although 6.6kV is still more common. The underground machinery voltages are 3.3kV, 1.1kV, 550V, and 415V, although some 6.6kV is more frequently being used as installations are becoming larger. For example, UK Coal collieries, the conveyor belt system is 1.1kV and the face voltage for the Joy AFC and Shearer is 3.3kV. Not all mine distribution systems are the same and can consist of ring or radial circuits. Feeders from the surface follow the shaft to the pit-bottom and to the underground substations. The distribution voltage is kept constant around the underground network and transformed to the required voltage near the machinery installation.

Electricity can be either direct current (DC) or AC. Although DC power is still used, it is generally only in special circumstances; part of a control system or old equipment. AC current is the main source of power used for underground mining machinery. There are many reasons for this, but the two main reasons are that it is safer and unlike DC, it can be transformed to different voltages, providing versatility for distribution over long distances. The majority of the machinery within an underground mine is powered by three-phase induction motors. Three phase induction motors operate under balanced load conditions and for the purpose of this thesis will be the only load type under analysis. Inductive loads also have reactive power requirements, this must be compensated with capacitive loads to correct the power factor. If the power factor is not corrected the systems apparent power will be higher than required, causing distribution losses and generation capacity to be larger than necessary. For a smooth and stable mine power system, reactive power and power factor correction is a necessity.

## 4.2 Balanced Three-Phase Circuits

Three-phase circuits can be highly complicated to analyse. However, three-phase systems are purposely designed to be balanced and every effort is made to maintain a balanced operation and if actual differences between phases can be neglected, the analysis of three-phase circuits can be as simple as the analysis of single-phase circuits. Balanced three-phase power consists of three generated voltages, each of equal magnitude and frequency but separated by $120°$ in phase. When these voltages are applied to three balanced impedances, the result is the flow of three balanced currents. A balance three-phase system can be split into three separate parts and any voltage or current in one part has a counterpart in another part, but $120°$ out of phase (Figure 4.2).

The major advantage of three-phase systems over single or two-phase systems is that they provide a more uniform power delivery. The $120°$ phase difference mean that the power waves in each phase are never at zero simultaneously, and that total instantaneous power remains constant. As a result three-phase motors provide constant torque with low vibration and easy starting. With this in mind, it could be considered that increasing the number of phases, would result in greater improvement. However, three-phase systems provide the most economical benefit, due to the fact that only a slight increase in efficiency and further complications occur using additional phases (Morley, 1990).

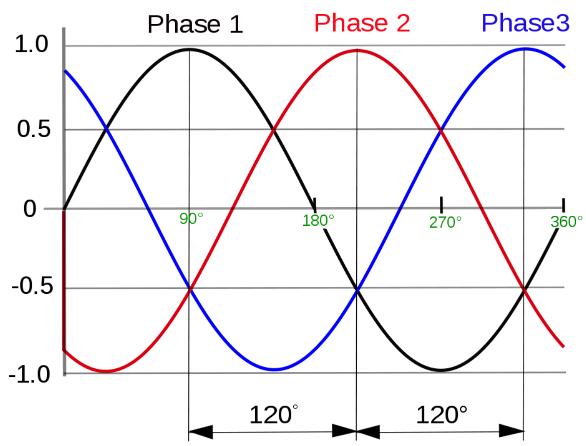



**Figure 4.2: 3 Phase AC Waveforms** (http://wikipedia.org)

To analysis three-phase circuits, it is acceptable to use single phase circuit theory. As it is an AC circuit, we have to consider the complex impedance, opposed to pure resistance in DC circuits. Impedance contains two components, resistance $(R)$ and complex component reactance $(jX)$, so the complex impedance is:

$$Z = |Z|(\cos\theta + \sin\theta) = R + jX.$$

Applying Ohms law for the current:

$$I = \frac{V}{Z}.$$

Apparent power $|S|$ (VA) is the product of the voltage and current magnitudes:

$$|S| = |V||I|.$$

Active power P (W) is the apparent power multiplied by the power factor $(pf)$ (See Equation 4.3):

$$P = |S|(pf). \tag{4.1}$$

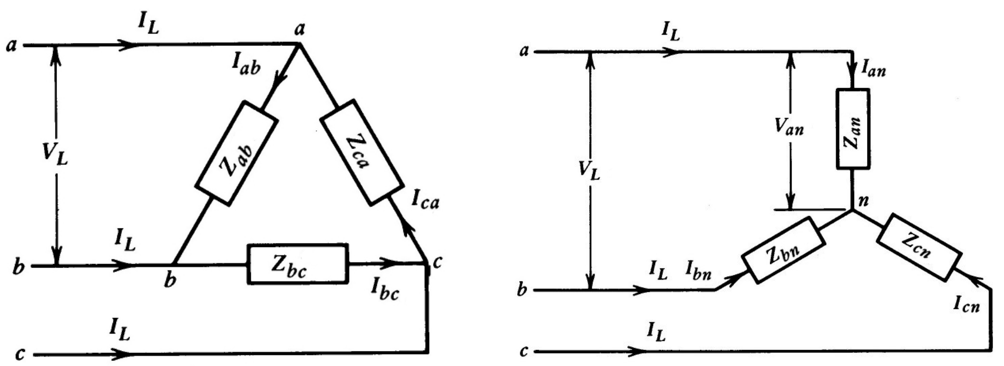
So the real part of the apparent power S(VA) is the active power P(W) and the imaginary part is known as the reactive power Q(VAR):

$$
\begin{aligned}
S &= P + jQ, \\
P &= |S| \cos\theta, \\
Q &= |S| \sin\theta,
\end{aligned}
\tag{4.2}
$$

and from Equation 4.1 and 4.2 the power factor is defined as follows:

$$pf = \cos\theta. \tag{4.3}$$

Three-phase loads can be connected either 'Y' or $\Delta$ (delta), 'Y' is sometimes




(a) Delta Connected Load (b) Wye Connected Load

**Figure 4.3:** Three-Phase Balanced Loads (McPherson, 1981)

referred to as 'wye' or 'star'. Figure 4.3 show the typical connections. As the voltage and current are the same in any of the three impedances of either the $\Delta$ or 'Y' balanced load, the average load consumed by one impedance is one third the total power consumed by the load. For a $\Delta$ load $V_{ab}$ is the line and the phase voltage (V), hence:

$$V_{ab} = V_L, \tag{4.4}$$

and in a Y load, the line current is through each load and the voltage is line-to-neutral, so:

$$
\begin{aligned}
V_{ab} &= V_{an} + V_{nb} = V_{an} - V_{bn}, \\
V_{bc} &= V_{bn} + V_{nc} = V_{bn} - V_{cn}, \\
V_{ca} &= V_{cn} + V_{na} = V_{cn} - V_{an},
\end{aligned}
$$

and

$$V_{an} = \frac{V_L}{\sqrt{3}}. \tag{4.5}$$

From Equation 4.4 we can see that for a $\Delta$ load the line voltage is equal to the phase voltage. The magnitude of the phase current (A) is:

$$I_{ab} = \frac{V_L}{|Z|},$$

and the line current is greater than the phase current, this relationship is:

$$I_L = \sqrt{3} I_{ab}.$$

The power (W) in one phase of a $\Delta$ load is given by:

$$P_{ab\Delta} = V_{ab} I_{ab} \cos\theta = V_L \frac{I_L}{\sqrt{3}} \cos\theta,$$

where $\theta$ is the angle of the phase impedance and $\cos\theta$ = power factor of the load. The volt ampers (VA) per phase is:

$$|S|_{ab\Delta} = V_{ab} I_{ab} = V_L \frac{I_L}{\sqrt{3}},$$

and the reactive volt amperes (VAR) is:

$$Q_{ab\Delta} = V_{ab} I_{ab} \sin\theta = V_L \frac{I_L}{\sqrt{3}} \sin\theta,$$

and the total power components for the complete $\Delta$ load are:

$$P_{T\Delta} = 3P_{ab} = 3V_{ab} I_{ab} \cos\theta = \sqrt{3} V_L I_L \cos\theta, \tag{4.6}$$

$$|S|_{T\Delta} = \sqrt{P_{T\Delta}^2 + Q_{T\Delta}^2} = 3V_{ab} I_{ab} = \sqrt{3} V_L I_L,$$

$$Q_{T\Delta} = 3Q_{ab\Delta} = 3V_{ab} I_{ab} \sin\theta = \sqrt{3} V_L I_L \sin\theta.$$

When considering a Y connected load, there is also the neutral terminal to consider. As the line current is equal to the phase current $(I_L = I_{an})$ and that all three line currents are of equal magnitude, but $120°$ out of phase. For a balanced load, the sum of these three currents is always zero, so the neutral current is also zero and not required to be connected. From Equation 4.5 we can see that for a Y load the line voltage is greater than the phase voltage. The magnitude of the phase current (A) is:

$$I_L = I_{an} = \frac{V_{an}}{|Z|} = \frac{V_L}{\sqrt{3}|Z|}.$$

The power (W) in one phase of a Y load is given by:

$$P_{anY} = V_{an} I_{an} \cos\theta = \frac{V_L}{\sqrt{3}} I_L \cos\theta.$$

The volt ampers (VA) per phase is:

$$|S|_{anY} = V_{an} I_{an} = V_L \frac{I_L}{\sqrt{3}},$$

the volt amperes reactive (VAR) is:

$$Q_{anY} = V_{an} I_{an} \sin\theta = \frac{V_L I_L}{\sqrt{3}} \sin\theta,$$

and the total power components for the complete Y load are:

$$P_{TY} = 3P_{abY} = \sqrt{3} V_L I_L \cos\theta, \tag{4.7}$$

$$|S|_{TY} = \sqrt{3} V_L I_L,$$

$$Q_{TY} = \sqrt{3} V_L I_L \sin\theta.$$

This is identical to the power components $P$,$|S|$ & $Q$ for the $\Delta$ load, which is what would be expected, since the line power should be determined by the voltages and currents, regardless of whether the load is Y or $\Delta$ connected. Note that the power factor is the cosine of the angle between phase voltage and phase current and not the line voltage and line current and is determined by the load impedance whether the impedance is Y or $\Delta$ connected (McPherson, 1981).

## 4.3 Summary

This chapter has given a basic outline of an underground distribution system and the common voltages used. The section on three phase electricity is important as to understand the components of power, apparent, active and reactive power. Managing the reactive power does not particularly save energy, although it does have a impact on system stability, possible demand charges.

# 5

# Demand Side Management for Underground Mining

Until recently energy use and its conservation has not been a top priority for the mining industry. That is not to say that power consumption was not of concern, just to say that the concern has been a purely financial one; the priority being to negotiate the best deal for the required energy and then see little change during an operation, 'it takes what it takes'. For example, if a new development required an additional power, this may be calculated from the installed power of the new equipment and the possible maximum demand. This may only be for a brief moment and in reality may never even exist.

However, with energy prices historically low and predictable, the financial impact of this mind set was insignificant in relation to other mining costs, such as labour. This is no longer the case, the increasing cost of installation and with energy prices constantly rising and becoming far from predictable and even energy shortages (due to increased population and load building) becoming more likely, taking measures to monitor, optimise and reduce the electrical demand of mining operations is the only viable route forward for the future global mining sector.

## 5.1 Electrical Demand

Electrical demand is defined as the instantaneous amount of power required for a system to operate at that time. Demand is commonly measured in Watts (W), kilowatts (kW) or megawatts (MW) depending on the size of the demand. Mining machinery has a large demand and generally require hundreds of kW's and more often MW's of power. Below is a list of common loads that can be found in an underground mine and their electrical demand.

- Longwall Set-up 5,000 - 20,000 kW
- Main Ventilation 500 - 5000 kW
- Compressors 500 kW each
- Main Hoist 1,000 - 15,000 kW

Electrical demand reduction is the process of reducing the power demand of a system without affecting the performance or output of the system or operation. Reducing the demand can be as simple as slowing the whole production down, but the output of the operation would be reduced and thus demand over output would not be reduced and therefore not an effective demand reduction strategy. A simple example of this would be a lamp in an office. Switch off the lamp and the demand is reduced, but quite obviously the production would slow down or cease if there is no light in the office. Replace the lamp with a low energy lamp and the electrical demand is reduced, but the production of the office is maintained.

Demand reduction should not be confused with energy saving. A production process can be rearranged to reduce the demand, but overall could still use the same amount of energy. That is not to say that energy saving practices do not reduce demand 'per se'. Just to say that the aim of a demand reduction strategy is not to save energy but to reduce the maximum required instantaneous power. In reality energy saving practices reduce demand and demand reduction saves energy. However, these are two distinct practices that can be easily confused with each other.

## 5.2 Energy Saving

Although the work of this project is demand reduction, the energy saving approach should not be over looked. What follows are some common energy saving practices that could be employed at an underground mine that would not only save energy but would also reduce demand.

- High Efficiency Motors
- Variable Speed Drives (VSD)
- More Efficient Lighting

### 5.2.1 High Efficiency Motors

Electric motors use 65% of all electricity in industry (Ruddell, 2003), and the U.S. Department of Energy (DOE) estimates that American industry could reduce by 11% to 18% the amount of electricity its countless motors require by switching to more-efficient models (Sergelen et al., 2008). Mine installations are full of motors and consume a large proportion of the mines energy demand. As motors age their efficiency also declines, this is mainly due to increased friction caused by wear and tear on the bearings. This bearing wear can also cause bad alignment, which adds to the reduced efficiency. The options available are to repair or replacement the motor. Repair, in many cases can mostly revive a motors efficiency back to a new level. Unless catastrophic failure has occurred, repair is usually the cheapest option when compared to motor replacement. But, when considering the energy cost of running the motor it may pay in the long run to replace the motor with a high efficiency motor(specification NEMA Premium motor or IEC IE3). High efficiency motors are on average 3% more efficient than standard motors and over a motors life time this can have a significant impact on the overall running cost of a motor. Figure 5.1 shows a table of a repair or replace case study. The motor is a 1000kW ventilation fan drive that operates for 8500hrs per year (10 days shut-down per year) where the user is paying £0.04p per unit for electricity.

| OLD MOTOR | | REPLACEMENT MOTOR | |
|---|---|---|---|
| Efficiency (prior to rewind) | 94.0% | Efficiency | 97.0% |
| Overhaul / rewind cost | £19,500 (Note 1) | Purchase price new | £30,000 |
| Efficiency (following rewind) | 93.0% | - | - |
| First year capital cost | £19,500 | First year capital cost | £30,000 |
| Additional energy cost in first year (due to reduced efficiency) | £3,889 | Energy **SAVING** in first year | £15,076 |
| **Total cost first year** | **£23,389** | **Total cost first year** | **£14,924** |
| Additional annual energy cost | £3,889 | Annual energy **SAVING** | £15,076 |

Note 1: Figures based on information published by the United States Government, Department of Energy.

**Figure 5.1: Repair or Replace Motor Case Study** (Morley 2011)

The rewound machine would then continue to cost an additional £3,889 per annum for the remainder of its life. The old motor will also naturally have a shorter life expectancy and be less reliable than a new replacement motor. The new replacement motor on the other hand will continue to save £15,076 in energy costs every year. On this basis the new motor would "pay for itself" before the end of the second year and over its operating life of say twenty years will save £301,520 in energy costs. In the above typical example, even if the rewinding process could retain the original design efficiency, the new machine still remains the most economical option in the first and subsequent years.

However, when considering a motor upgrade, direct replacements must be examined carefully. The existing motor may be running at part load or not actually be suitable for the application. Agamloh (2009) & Hsu JS (1995) discuss the replacement of existing motors and the issues arising from partial loaded motors and the considerations needed to ensure the efficiency gain of a high efficiency motor is realised. If these observations are considered, upgrading to high efficiency motors has the potential to offer good savings and reduce demand.

### 5.2.2 Motor Speed Reduction

Traditionally an AC induction motor is driven at a fixed speed. Flow control for pumps and fans is then achieved mechanically by restricting the flow with a damper or valve. This is a very inefficient way of control and leads to energy wastage as the motor operates outside its optimum zone. Economic benefits can be obtained by replacing these mechanical flow controls with variable speed drives (VSD), Figure 5.2 shows the basic principle behind this practice. VSD or variable frequency drives (VFD) (VFD refers to AC drives only and a VSD refers to either AC Drives or DC Drives). VFD's vary the speed of an AC motor by varying the frequency to the motor. VSD's referring to DC motors vary the speed by varying the voltage to the motor.

Most applications for VSD systems involve variable torque loads, such as fans and centrifugal pumps and air compressors, where the power consumption is proportional to the cube of motor speed. So even a small reduction in rotational speed can have a large impact of the energy consumption of the motor. To further understand the principle and the potential for huge energy savings we must look at the so called affinity laws. The hydraulics and heating, ventilation, and air conditioning (HVAC) industry use the affinity laws to indicate the influence on volume capacity, head (pressure) and/or power consumption of a pump or fan due to change in speed or impeller/fan diameter. They apply to both centrifugal and axial pumps, fans, and hydraulic turbines.

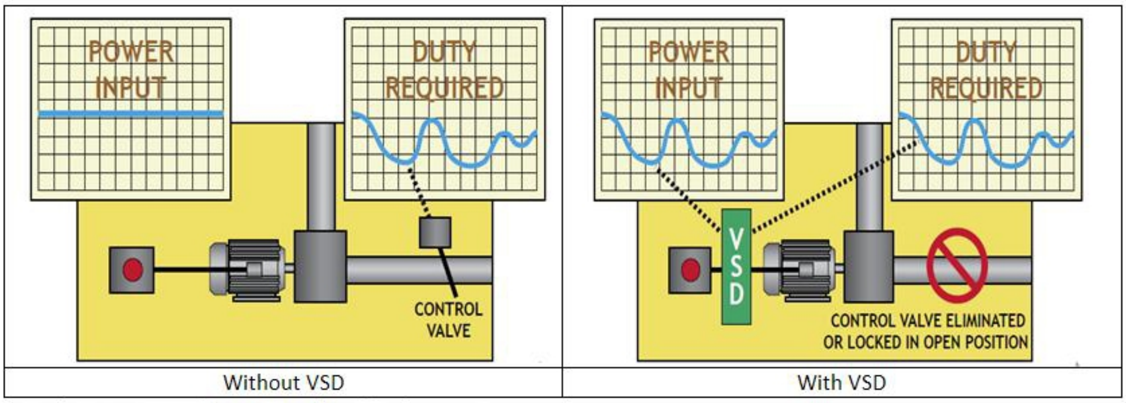



**Figure 5.2: A Pumping System without and with a VSD** (Morley 2011)

There are two sets of affinity laws, one for when the pump or fan design is constant and the rotational speed is changed and the other for when the pump impeller or fan diameter is changed, here only the former is presented for the purpose of illustrating the power and speed relationship.

$$\frac{Q_1}{Q_2} = (\frac{N_1}{N_2}),$$
$$\frac{H_1}{H_2} = (\frac{N_1}{N_2})^2,$$
$$\frac{P_1}{P_2} = (\frac{N_1}{N_2})^3. \tag{5.1}$$

Where:

- $Q$ is the volumetric flow rate (e.g. CFM, GPM or L/s).
- $N$ is the shaft rotational speed (e.g. rpm).
- $H$ is the pressure or head developed by the fan/pump (e.g. ft or m).
- $P$ is the shaft power (e.g. W).

Now let's consider the statement made earlier that a small reduction in rotational speed can have a significant savings on energy. For example, if the rotational speed $(N_1)$ of an a pump is reduced by 10%, then:

$$N_2 = 0.9N_1. \tag{5.2}$$

Substituting equation 5.2 into equation 5.1 we get:

$$\frac{P_1}{P_2} = (\frac{N_1}{0.9N_1})^3,$$

and transposed we get:

$$P_2 = 0.9^3 P_1,$$

so:

$$P_2 = 0.73P_1.$$

As we can see, a 10% reduction in rotational speed has a 27% reduction in required power. Even a 1% reduction would save 3% in power, in an industrial situation like mines where the motors are of significant size, 500kW+ for ventilation for example. 3% power savings could still amount to a worthwhile saving.

In real terms, the savings for the surface ventilation fan of the case study colliery requires an average 850kW, 24 hours a day. 3% would be 25.5kWh per hour and at today's energy prices would save £16,750 per year. 1% is used here in the example as it is possible that a ventilation fan speed could be reduced by 1% without any significant impact on the ventilation, although this is not a given as there is many issues that need to be considered before any reduction in mine ventilation is carried out.

As with the installation of high efficiency motors careful consideration must be made, as the introduction of a VSD into an existing system may cause problems. The motor design must have certain characteristics to withstand the possible voltage spikes that a VSD can cause. Sauer and Brady (2009) discuss the problems that can occur, such as stator winding failure or bearing damage from VSD voltage spikes, if the motor lacks certain design features. Most modern VSD's are fitted with filtering to reduce stator failure and motor bearings are insulated to prevent the bearing damage.

Many VSD's are not installed for speed control, but to reduce starting power and the associated voltage drops. From a energy saving point of view, this my not have the opposite effect, VSD losses increase the whole system losses by 40% compared to grid operation. These additional losses must be considered carefully, as they can exceed any energy savings gained by upgrading to high efficiency motors (Benhaddadi et al., 2012). Benhaddadi et al. (2012) also specify the right loading points for high efficiency motors and drive systems to benefit from an energy saving point of view. Most application of VSD save energy with an average of 30% per drive unit (Benhaddadi et al., 2012). Therefore we can deduce that the saving potential which can be achieved by using VFD is significant. For example, Sergelen et al. (2008) presents a case study where a major pumping station operated by the Cooperation of Mongolia is working with about 50% less energy. This is the result of replacing the synchronous motor in the pump station with an AC induction motor and VSD.

Whether installing a high efficiency motor, a VSD or both it is clear that energy savings, and hence demand reduction, can be made. However, it also has been highlighted that these installation require careful thought and system analysis to ensure that the theoretical savings are to be realised and not lost in an inefficient or miss matched system. Section 5.2.4 discusses further examples of uses of VSD in the mining industry.

### 5.2.3 More Efficient Lighting

An underground mine could have thousands of light units and reducing the energy consumption by just a fraction would still give a worthwhile reduction in their energy cost. Typically, changing ordinary halo-phosphor fluorescent tubes to new tri-phosphor tubes could save around 10% on lighting energy costs. Tri-phosphor tubes also have extended lamp life and emit a more natural light. This is not the only lighting improvements. The compact fluorescent lamp (CFL) is used all over Europe since the incandescent lamp began to be phased out in 2009, saving almost 80% of energy when comparing the two lamp types (Smith, 2010). Recently the most significant savings in lighting are with light-emitting diode (LED) technology, both in the mining lamp and face lighting. Victor Products Ltd has developed a ATEX LED face lighting unit the ML102 (Figure 5.3) and has equipped a face at Kellingley colliery with 100 units (Schwartz, 2011). This light unit emits more light than a 60W incandescent lamp at a maximum of 15W. Lighting technology is constantly progressing and although constantly keeping up may not prove cost effective, upgrading a ten year old lighting system will most likely have its benefits.

**Figure 5.3: The Victor ML102** (LED Face Lighting)

### 5.2.4 Energy Efficiency in Mining

Although there are publications on mining and energy, most of the papers discuss the specific efficiency of individual machines or services. Ventilation optimization dominates the research, including optimising the design of the ventilation system and air flow, as discussed by Pritchard (2008) and Keskimaki (1989). A case study of controlling the main fan and reducing its load during mine evacuation/blasting periods is presented by du Plessis (2008) and gives significant savings of nearly $700,000 a year.

### 5.2.5 Ventilation on Demand (VOD)

Hardcastle et al. (1997), Kocsis et al. (2003), Allen (2008) and O'Connor (2008) are the prominent researchers in ventilation on demand (VOD). VOD is where air flow, quality monitors and asset tags attached to personnel and machinery trigger ventilation booster fans as they progress through different areas of the mine. Although VOD has had a patent awarded (Sudbury Mining, 2010), VOD is not yet wide spread, but has the potential to have a significant impact on reducing energy consumption of hard rock mine ventilation systems. Due to the methane gas, VOD is not really viable for coal mine ventilation systems.

### 5.2.6 Methane Drainage and Ventilation Air Methane (VAM)

Methane drainage is used across the underground coal mining industry, where methane from the uncut coal reserve is pumped to the surface and used to generator electricity. Research that is still in the very early stages is that of Johnson et al. (1998) and Srivastava and Harpalani (2005), where they consider the possibility of using ventilation air methane (VAM) to power generating plans. Unlike the drained methane, VAM has very low methane content and is difficult to manage. Cluff et al. (2013) have developed a novel way to capture the energy in the VAM and reduce methane emissions with a specialist VAM burner.

### 5.2.7 Conveyor Belts Systems)

Other areas of significant research related to mine energy is the conveyor belt system. Kitts and Stees (2000) introduce the addition of a VFD to a coal mine Longwall system. The VFD improves the start-up and running of the conveyor, the belt is started at a slow speed and then run at this speed to stabilize the belt before full speed is initialized. This reduces the in rush current to the motor, mechanical shock and wear and tear on the belt. One VFD that is widely used on coal mine conveyor systems is the Breuer Drive Its technical specifications and application are detailed by Helbing (2000). Zhifu Yu (2010) describes the application and power savings of VFD's to a coal mine hoist, air compressor and ventilation fan. Marx and Calmeyer (2004) and Yingling et al. (1997) have also researched mine conveyor systems, Marx and Calmeyer (2004) produced a model to research into the speed control and regeneration from belts running downhill to save energy and Yingling et al. (1997) controlling the belt speed and the discharge of a continuous miner and a Longwall operation. This is to enable a better loaded conveyor system to improve the overall efficiency and reduce peak loads. Ristic and Jeftenic (2012) present a fuzzy logic controller for the system of belt conveyors in an open-pit mine with adjustable speed drives based on the principle of optimum energy consumption.

### 5.2.8 Power System Stability and Regeneration)

Natan et al. (2009) presents the benefits and systems for dynamic VAR compensation (or power factor correction) for a mine hoist at LKAB in Sweden. Not only was the power system stability and operation improved, but the mine production was increased due to the reduced required apparent power (VA) (Section 4.1). Regeneration is the recovery of wasted energy, as in the paper by Mazumdar (2009), where the fall of the drag-line excavator bucket regenerates into an UltraCapacitor energy storage to provide control power for the on-board medium voltage gas insulated switchgear. These are only a handful of published research in the mining world, more and more improvements and savings emerge as the energy focus continues to increase.

### 5.2.9 Commercial Energy Audit - UK Coal Colliery

Energy audits are when a full assessment of an installation or production process is carried out to evaluate all possible areas for improvement, energy and cost reduction. During the work carried out at the UK Coal colliery (Chapter 7), UK Coal contracted Siemens to carry out an energy audit of the whole of the surface operations at the colliery. The audit was carried out with an energy saving approach from a cost point of view. The overall conclusion was that the whole plan had to be implemented to achieve the best cost benefit as energy prices would out cost the install investment and the payback period would extend beyond the mine life.

| Executive Summary | Energy Saving (kWh/yr) | Saving (£/yr) | Capital Costs (£) | Pay Back (yrs) |
|---|---|---|---|---|
| **Targeted Energy Management & Automated Monitoring** | 3,025,920 | £145,929 | £191,000 | 1.3 |
| **Electrical System Improvements** | 212,399 | £310,528 | £193,000 | 0.6 |
| **Motors & Drives (Alignment & VSD)** | 8,360,175 | £614,284 | £824,430 | 1.3 |
| **Compressed Air Improvements** | 894,356 | £58,661 | £67,378 | 1.1 |
| **Lighting Improvement** | 798,137 | £58,645 | £146,563 | 2.5 |
| **Equipment** | 134,740 | £9,822 | £14,000 | 1.4 |
| **Totals** | **13,425,727** | **£1,197,869** | **£1,436,371** | **1.2** |

**Table 5.1: Siemens Energy Audit Summary**

The main results of the audit are summarised in Table 5.1. It can be seen that the total payback time for the recommendations is only 1.2 years. This is a very short period and would worth implementing the changes. However, examining the individual changes separately, we can see that the motors & drives improvement has the most impact. This improvement relies on the ability of the systems to actual allow the introduction of the VSD and the speed reduction the cost savings will require. As already discussed, a mine has many critical systems, the ventilation and environment being the most important. Any changes to the airflow or pressure could have a negative effect on the stability of this critical system and would not be considered by mine management. Further more, due to the uncertainty of a UK colliery today, any changes that have a payback period more than a year and a half probably would not get considered either. This just highlights how energy audits can give the impression that large amount of savings can be made, although in real terms savings are a lot less.

## 5.3 Summary

The energy saving approach definitely helps to reduce demand and energy costs, but there is an initial cost implication whether it is new motor or VSD, as there is usually a large capital cost outlay and this must be paid for before the savings are realised. A demand reduction strategy is looking at how efficiently machines are used together, when and for how long. The next chapter explores the technical details of DSM and the influences they have had on the project and how they have shaped the methodology. The concept defined is a demand reduction strategy that is generic in approach, taking each individual load and considering its operation in relation to the other loads that are present in the system.

# 6

# Development of Electrical Demand Reduction and Responsive Strategies

So far this thesis has focused on the development of energy related research in the mining and power industries and the common practices that are used today to increase efficiency and reduce demand. This chapter presents the unique work that has been further developed by the author from existing research. This is an overall network optimisation of mine power systems and the integration into the Smart Grid and DSR. The full test and modelling results are not given here (as they are presented in Chapter 8, Project Data), although the complete methodology, how it was derived and the conceptual operation is presented in this chapter.

## 6.1 Demand Reduction Methodology

Electrical power is the main energy source of underground mining today. Coal mines (and increasingly - hard rock mines) rely on electrical power for most underground operations. An underground mine consists of many systems, including ventilation; hoisting; transportation; pumping; compressed air; cooling; heating and lighting. As reviewed in the previous Section 5.2.4, all these systems can be optimised independently. However, without a central control, these systems will still demand power regardless of other system demands. The optimisation includes multiple systems within the electrical network, considering them all to produce a smoother demand curve. The optimisation has been considered in two parts. Firstly, how can the demand be instantly reduced without effecting the production or overall working of the mine, and secondly, a more aggressive approach that changes production and operations.

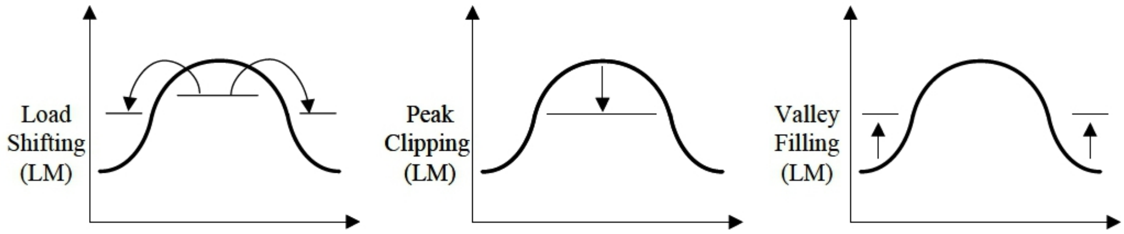



**Figure 6.1: DSM Techniques**

The methodology for the design of the demand reduction and responsive strategies is inspired by traditional DSM techniques (as described in Section 3.1) and utilities' approach to electrical grid management. As electricity can not be stored in large quantities, it must be managed on a minute-by-minute basis to ensure the supply meets the demand. If we consider this in opposite, the demand must meet the supply. This is the basic approach for the methodology of the proposed demand reduction and responsive strategies. The traditional DSM techniques used for the demand reduction strategy are, peak clipping; valley filling and load shifting (Figure 6.1).

Load shedding is to be included in the real-time scheduling to increase network stability and security of supply. Techniques such as replacing machinery for modern high efficiency machinery is to be ignored as it is well known and common practice. Increasing the efficiency of an individual system can always be performed and would not have any significant impact on network management strategy.

The Smart Grid also plays a major part in the development with the key focus on DSR and RTP. Mine power systems can also be in remote locations without grid connection. When this is the case the DSR would not be due to pricing as the electricity (usually from diesel generators) would not fluctuate in price as with RTP. Instead the DSR would be from the actual availability, the maximum available demand due to generation limitations. In some cases the DSR would be due to both limitations, price and availability.

Utilities have to constantly manage their resources to keep up with the constantly changing demand. Power stations must be brought on-line and others on-standby to respond instantly when requested by grid control. An excellent example of this is the so called 'TV Pick-up' of the UK. This is a specific UK only phenomenon where at the break or end of a popular TV program or sporting event between 1.5 and 1.75 million kettles switch on. To accommodate the peak in demand the National Grid bring on-line 3GW of instantaneous power within a time frame of about five minutes (BBC, 2008). Much of this demand is supplied by hydro-electric power, as these type of power stations can supply the demand instantly at the push of a button. This is very expensive and some of these power stations only exist for the purpose of supplying peak demand. This highlights further the need for wide spread DSR especially for large consumers like the mining industry.

When a peak demand event occurs the grid balancing engineers know how much power is required, not by guess work, but from data collected from past events. This time domain data are known as load profiles. An electrical demand curve or profile for a given period of time. To have an understanding of the operational events in a mine, it was clear that load profiles would be needed. It is load profiles that are the basis of this research and DSR strategy development. The development of the network management strategy has been highly influenced by El-Metwally et al. (2006) and Mohagheghi and Raji (2012). El-Metwally et al. (2006) presents the use of priority ranking rules that are used for peak looping and load shifting. These rules are applied off-line to provide a reduced demand schedule. Mohagheghi and Raji (2012) uses workstation groups within the industrial plant and also applies an off-line ranking model with the addition of a real-time ranking modification module and with a DR feedback. All these processes are included in the intelligent demand reduction system detailed in this chapter.

Once the loads were profiled, their characteristics can then also be defined. This will highlight how much or little a load could be manipulated within the mine pro-

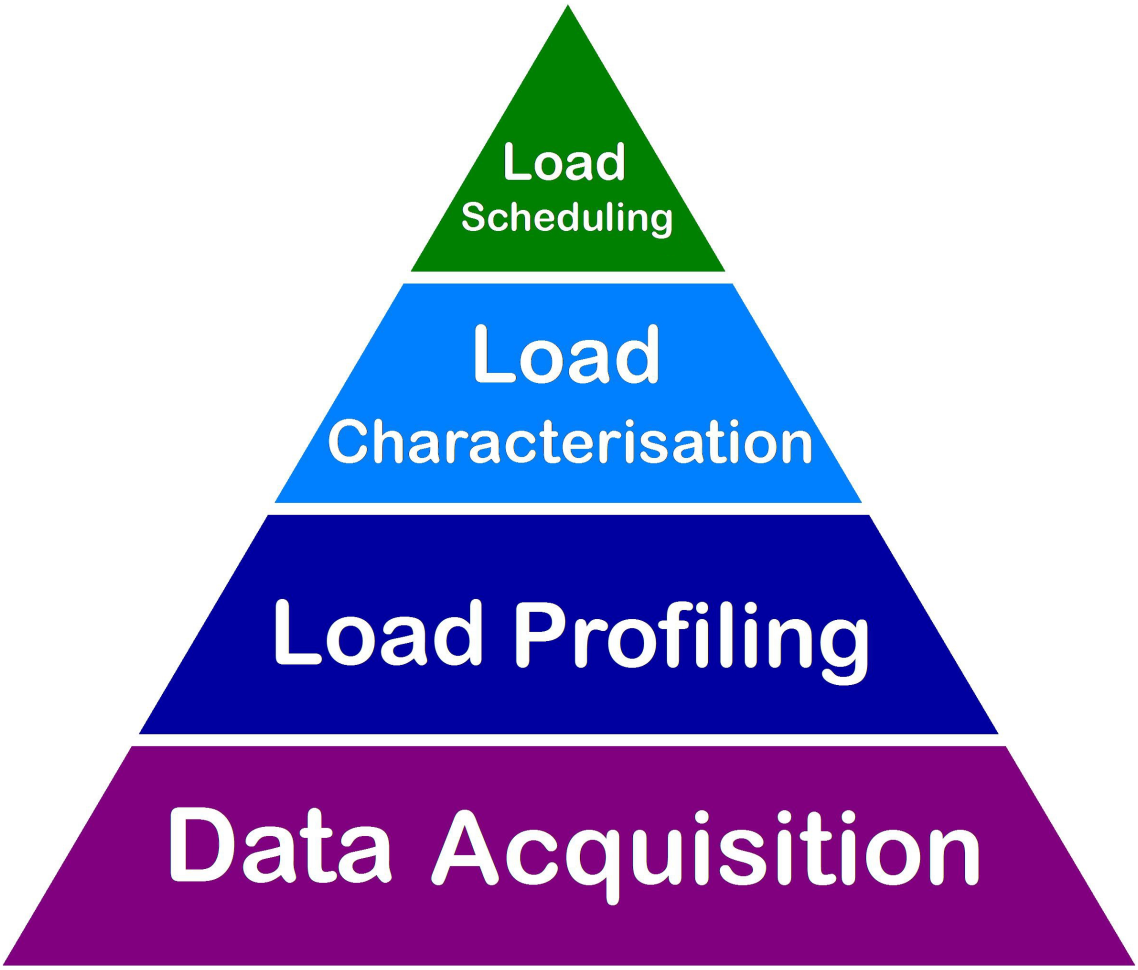



Figure 6.2: Demand Reduction Pyramid (DRP)

duction cycle. The final stage is then to optimise the running of the loads with reference to the load profiles, characteristics and response triggers from either electrical price or availability. The management system must consider available demand, system stability, constant valley filling and peak lopping, and load shedding in response to peak demand or electricity price.

Figure 6.2 shows the demand reduction pyramid (DRP). The DRP gives the procedure in its logical hierarchy form. Each level is proportional to how important each stage is on the overall success of the final demand reduction strategy. Next each stage is described, its influences, how it was derived and current state.

## 6.2 Stage 1: Data Acquisition

From the start it was obvious that some relevant data would be required on the current electrical demand in underground mining. It had to consider how and where this information would come from. The possible areas where the data could be sourced are, contacting mining companies direct or manufacturers of equipment. The most desirable data for the project would be real-time power consumption data from a working mine. However, with no mine available to source this data from it was clear that this would not be easy to obtain. Furthermore, real data is inherently difficult to collect/obtain, requiring extensive installation of monitoring/recording equipment and the co-operation of a willing mining company. For this reason another data source for the project was considered first choice.

Equipment manufacturers produce data sheets for their individual pieces of equipment. However, in general the only electrical information that is available is the maximum rated power of the machine or motor. The maximum rated power does not provide any information on when, how long and under what load conditions the machine are being used. A motors rated power is very different to its actual power consumption. For example a water-pump motor will be rated for the maximum pumping head and flow rate. However, it may only be installed in a situation where the head is half the maximum. Then the real power consumption will only be a fraction of the pumps rated power. Data sheets are a source of relevant information. However, additional information is required to build a better picture of mining electrical demand. (Some manufactures have been contacted directly for more detailed data on there machinery including, motor load/power curves etc. These are to aid in the analysis of the real-time data collected.)

### 6.2.1 Questionnaire

As previously mentioned, the author did not have a source of live production data. Due to this a questionnaire was complied. The questionnaire focused on production and electrical energy consumption and was set-up on-line for easy completion. It was developed using an existing on-line questionnaire application to suit the requirements of the thesis and advertised via the CSM Association newsletter.

#### 6.2.1.1 Questionnaire design

Initially the questionnaire was designed with a large number of questions and sections. The sections covered every aspect of an underground coal mine. These sections included, mine characteristics and method, production, stock piling, operation costs, power, and energy management. Each section had a number of questions which required very detailed answers. This questionnaire was distributed to the world mining industry in the hope of getting good response. This was not the case and in fact very little response was received, in fact, only two mines fully completed it. It was felt that this was due to the confidentiality of the mining industry and the lengthy response required to complete the questionnaire.

The main propose of the questionnaire was to gain knowledge of the coal mining power usage. The questionnaire was redesigned to only focus on power usage. This reduced the amount of questions and the detail required. This redesigned questionnaire was much simpler, focused and quicker to complete. The redesigned questionnaire was complied and distributed to the contact the author had made during the thesis that had coal mine connections for them to complete. The questionnaire was compiled to gain knowledge of the European coal mines, and the main focus was the size of the mines and their electrical demand.

## 6.3 Questionnaire Results

The results of the questionnaire are presented next. The results have been analysed and summarised here for presentation. However, the complete questionnaire results are in Appendix B. The locations and the number of mines that data has been contributed via the questionnaire is outlined in below:

- Spain - 11 Mines
- Poland - 3 Mines
- Slovenija - 1 Mine
- UK - 1 Mine

The questionnaire was divided into sections. These section are mine basics, production, development, transportation, pumping, ventilation and power consumption. It can be seen that Spain has contributed data for the most amount of mines. Although this has no reflection on the coal production many of these mines are very small and have non mechanised mining methods. Only mines using the Longwall method will be included in the summarised results. This only includes four Spanish mines, the remaining seven mines only accounts for 8% of the total production of the questionnaire results. Table 6.1 shows the summarised results from the questionnaire. The results contain the mine production, the mine depth, conveyor system length, ventilation demand and energy consumption.

From the results, it can be seen there is a relationship between the size of depth of the mine and the ventilation demand. This is what would be expected. There is not a direct relationship between any of the other values, the mine depth does have a loose relationship with the energy demand and production. However, it is not as clearly defined as the depth and ventilation demand.

| Mine | Production (Tonnes/day) | Depth (m) | Conveyor Length (m) | Ventilation Demand (kW) | Energy (MWh/Year) |
|---|---|---|---|---|---|
| **Poland 1** | 7,500 | 936 | 14,030 | 4,100 | 232,426 |
| **Poland 2** | 15,700 | 896 | 23,350 | 5,270 | 204,975 |
| **Poland 3** | 17,500 | 701 | 18,362 | 2,000 | 158,968 |
| **Spain 1** | 1,100 | 300 | 3,000 | 850 | 24,000 |
| **Spain 2** | 630 | 250 | 1,500 | 300 | - |
| **Spain 3** | 700 | 973 | 3,021 | 380 | 20,572 |
| **Spain 4** | 1,000 | 400 | 10,000 | 320 | 31,164 |
| **Slovenija** | 11,000 | 458 | 5,880 | 925 | 51,957 |
| **UK** | 8,000 | 800 | 7,000 | 2,950 | 61,007 |

**Table 6.1: Questionnaire Summarised Results**

### 6.3.1 Live Production Data

With very limited responses to the questionnaire there was not enough information provided for the development of any DSR strategies. Even with statistical data from the questionnaire combined with manufacturers machinery data sheets, there would only be enough information to give a basic understanding of a mine power system. However, to develop a model with mining process conditions would still be unknown, as data of real life working patterns of a mine and its machinery would still be unavailable. It is the opinion of the author that a basic model would not be enough to develop or prove any electrical demand reduction strategies. For any research to be considered feasible and have any impact within the mining industry a model developed needed real life mining data as its reference.

The author made direct contact with UK Coal with the request for live production data for one of their collieries. The data was not available as UK Coal did not monitor or record this information. The author made the proposal to UK Coal that the University of Exeter would supply and install the equipment required to monitoring and record this data. In return, UK Coal would give access to the power and SCADA system data. This provided the author the exact data required to continue the research. Chapter 7 describes the development and installation of the data acquisition system at the UK Coal Colliery.

## 6.4 Stage 2: Load Profiling

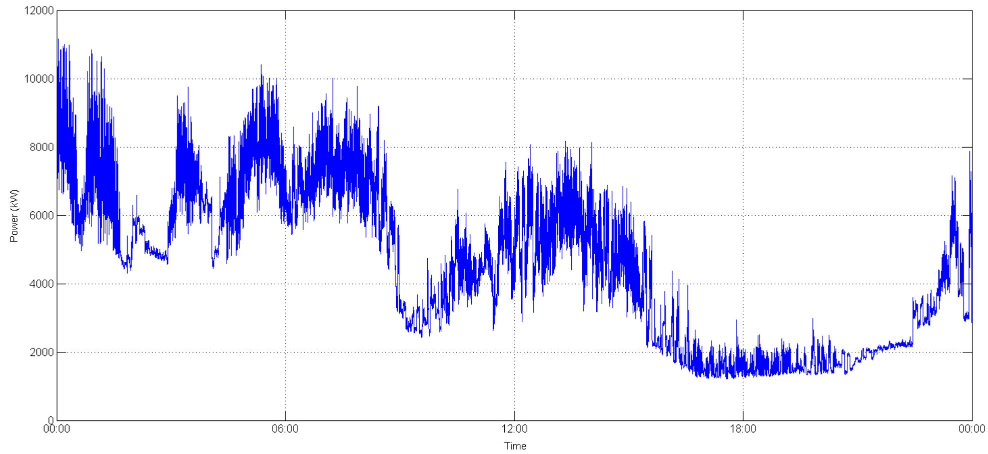



**Figure 6.3: Typical Load Profile (TLP)** (24 hour mine site TLP)

Load profiling started in the utility sector and is used to define class types of the customers, allowing the utility to forecast, schedule, raise settlement, reconcile and estimate transmission obligations. This is typically based on hourly accumulated energy consumption and applied to the individual hours of a billing period, so that customers without half hourly meters are able to participate in the electric retail market (Dumbrava et al., 2008). Power utilities calculate and offer residential, commercial, large commercial, industrial, agricultural, street lighting and standby load profiles to their customers. These profiles are known as typical load profiles (TLP) (Figure 6.3). Traditional methods for developing of TLPs from live production data are:

- Dynamic profiling - updated daily using the days data for a days profile.
- Static load profiles - based on 1-3 years of live production data.
- Proxy-Day load profiles - based on a profile 'like' the given day.
- Dynamic models – hourly models fitted to live production data.

The TLP are defined from metering in substations, large commercial and industrial sites. The type of load profiling method used will depend on the data availability, equipment availability, accuracy requirements, regulatory requirements and cost considerations. In addition to the traditional load profiling methods, other more sophisticated methods also exist. For example, Gavrilas et al. (2008), Lo et al. (2003), Gerbec et al. (2005) and Zakaria et al. (2006) with their use of fuzzy logic and Pitt and Kitschen (1999) with the use of data mining techniques. What all these load profiling papers have in common is that the research is all based on clustering of load profiles. The data sets are a large collection of customer profiles and the different techniques to cluster the profiles into similar load profile groups. These groups are then used to define the TLP and assigned the appropriate tariff, schedule and balance the transmission system. Comprehensive studies of the most popular load profile clustering techniques used today are presented by Prahastono et al. (2007) and Kim et al. (2011).

Load profiling is a key component of the Smart Grid and modern DSM. With the introduction of AMR and AMI more detailed load profiling will become normal - as will higher resolution for individual switched loads, with shorted time intervals of minutes, or even seconds. It will not only be the utilities that use load profiling techniques - the customers (especially large industrial users) will come to depend on these profiles in order to control and synchronize their loads to achieve the lowest demand possible.

### 6.4.1 Load Profiling: Research Requirements

The load profiling methods described so far are for the utilities to define the usage of their customers. Network optimisation requires more detailed profiles, not only for the complete site, but for each individual load. Individual loads are the components that make up the overall or system profile. Currently there is no published load profiling research that exactly fits these. What follows is an outline of the requirements and the method derived to fulfil them.

The load profiles that are wanted for this project are the power requirement over time for a given load in its production cycle. The profile will be the pattern that is repeated by the load in its normal operation. For example, the profile of a water pump, pumping at normal flow rate, or a conveyor belt at normal speed with normal load. Collecting all the profiles together will then allow the optimising of the overall or system load profile. The TLP that are used by utilities are usually large datasets. However, they are only referenced occasionally so this does not pose a problem for computational time or resources. A real-time DSR and scheduling control algorithm however, may reference the data sets of individual loads or collection of loads many times a minute. Accessing large datasets this frequently would slow down any control system and is impractical. A more efficient way of load profiling a load is needed. One possible way of efficient load profiling is explained below.

Once a real-life situation of a load has been examined in detail the load profile matrices are available for use for an optimisation algorithm to schedule in real time. If the loads were still defined with large datasets or complex equations this would not be possible without a large amount of computing power such as a supercomputer. It is not practical or cost effective to install supercomputers within a mine installation and would serve no practical benefit. The alternative method described here is known as model skeletonisation. There are two types of loads: fixed and variable.

### 6.4.2 Fixed load: load profiles

A fixed load load profile is the simplest of the load profiles to define, it is simply the power demand of a load during its normal operation, such as its kW's or MW's. A fixed load is defined as a load that runs at a constant speed, constant load and constant output. There maybe additional parameters if the load has a specific start-up or shut-down profile. However, these are defined with the additional load parameters, see section 6.4.4. The fixed load load profile is simply defined with a power requirement, as follows:

$$\text{Load Power Requirements} = x \text{ kW}. \tag{6.1}$$

where: $x$ is the constant power in kW required by the load.

### 6.4.3 Variable load: load profiles

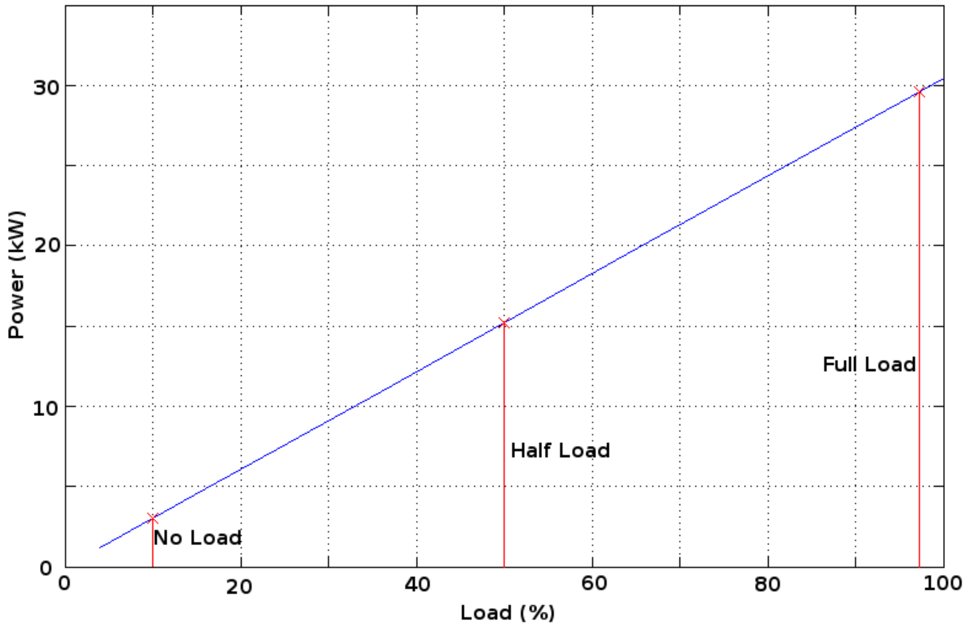



**Figure 6.4: Variable Load Graph** (Load (%) x Power (kW))

As we have seen from the previous section (6.4.2), it is a relatively simple process to define the load profile of a fixed load. However, when it is a variable load under examination the process is far less simple. There are more parameters to consider, the load, speed and the efficiency all have an effect on the power demand. First consider a single speed motor. As the motors load is increased the power demand will increase to keep the motor at a constant speed (Figure 6.4). Looking at the graph we can see that there is multiple positions along the line, this depends on how much load is applied to the motor. Although in reality, the motor may pass through the whole operating range, it is only likely to operate for any significant amount of time at certain points or ranges along the line. For example, no load or idle, half load and full load (Figure 6.4). Now instead of a large dataset or complex equation to define the load profile, we have three discrete values. These are defined in the Loading Condition Matrix (LCM) as follows:

$$\text{Loading Condition Matrix (LCM)} = (no\ load,\ half\ load,\ full\ load),$$
$$\text{Load } n = (C_1,\ C_2,\ C_3), \qquad (6.2)$$

where: $n$ is the load number and $C_1$, $C_2$, and $C_3$ are the constant power in kW's required by the load at the different loading conditions of the load.

To further increase the complexity the motor may also be controlled by a VSD. The electrical demand of the motor could be effected by the varying speed of the motor as well as the varying load. As a mine may have many VSD's a similar approach to the single speed variable load is needed for the load profiling of these machines. During operation a VSD may move through a range of speeds. However, it will generally not constantly change speed up and down (although there are special cases, such as a variable speed compressors), it will operate at one speed for a given period of time and then change to another.

For example to save energy a conveyor belt may run at a slow speed when it is empty and speed up to another speed when transporting material. The conveyor may also be loaded with different amounts of material, giving a three dimensional operating parameters (Figure 6.5). As with the single speed variable load, the load profile approach is to to define discrete values for the possible operating speeds (Figure 6.5). These operating points will relate to the real-life situation of the load.

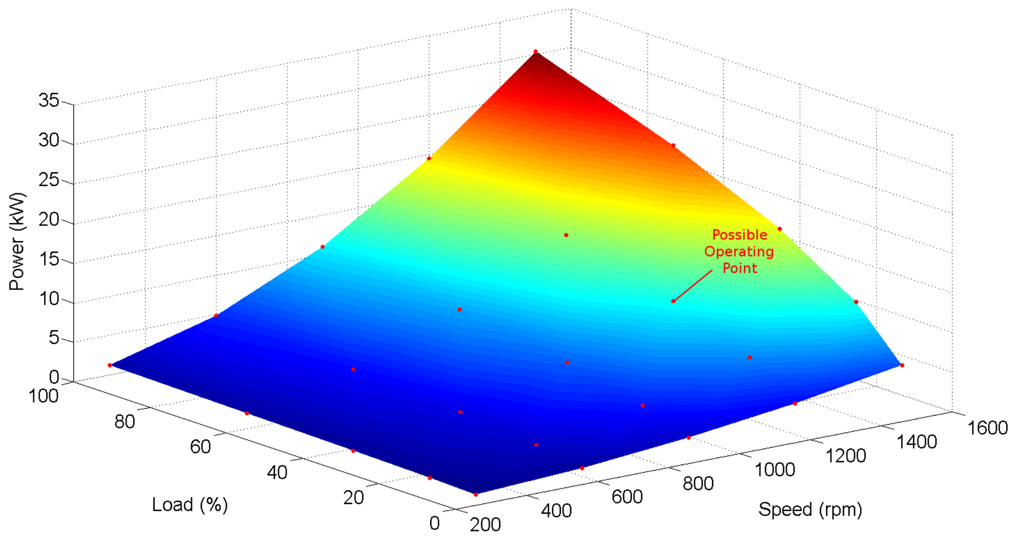



**Figure 6.5: Variable Load & Speed Graph**
(Load (%) x Speed (rpm) x Power (kW))

For example the speed possibilities could be slow, medium or fast. So, from LCM we have different load conditions and now we define different speeds points with the Load Speed Matrix (LSM) as follows:

$$\text{Load Speed Matrix (LSM)} = \begin{pmatrix} Slow \\ Medium \\ Fast \end{pmatrix},$$

$$\text{Load} \ = \begin{pmatrix} S_1 \\ S_2 \\ S_3 \end{pmatrix}, \tag{6.3}$$

where: $n$ is the load number ID and $S_1$, $S2$, and $S_3$ are the constant power in kW's required by the load at different speeds of the load. If we combine the two we have different load points at different speeds. So from a very complex load profile situation with three dimensions we now have a set of discrete speeds with discrete load conditions that relate the load in a real-life situation. So, using the LCM and the LSM as we can define the Load Demand Matrix (LDM) as follows:

$$\text{Load Demand Matrix (LDM)} = \begin{pmatrix} P_{C_1S_1} & P_{C_2S_1} & P_{C_3S_1} \\ P_{C_1S_2} & P_{C_2S_2} & P_{C_3S_2} \\ P_{C_1S_3} & P_{C_2S_3} & P_{C_3S_3} \end{pmatrix}, \tag{6.4}$$

where: $P_{C_1S_1}$ is the constant power required by the load at loading condition 1 and speed condition 1, $P_{C_2S_1}$ is the constant power required by the load at loading condition 2 and speed condition 1, and so on.

The LDM (Equation 6.4) is defined from the three value LCM (Equation 6.2) and a three value LSM (Equation 6.3). However, this is just an example and the LCM and LSM can contain as little or as many values as required from the analysis of the load in question. The LDM will then have the number of rows and columns accordingly. For example, a single speed load with five load conditions will have a LCM with five values, a LSM one value and the relating LDM will have one row and five columns. Or a load with one loading condition and four operating speeds would have a LCM with two values, a LSM with four values and a LDM with four rows and two columns. The fixed load load profile in the previous section (6.4.2) is presented for completeness. However, the variable load load profile method will be used to define the fixed load profile. The load will simply be defined with a single speed and loading condition.

So far, only the load demand has been profiled. The load's demand has no relevance for an optimisation model if the actual output or production of the load is not defined. The LOM has the same dimensions as the LDM. However, the LOM does not contain demand values but the loads output, for example a water pumps litres per hour or a conveyor belts tonnes per hour. The Load Output Matrix (LOM) is defined as follows:

$$\text{Load Output Matrix (LOM)} = \begin{pmatrix} O_{C_1S_1} & O_{C_2S_1} & O_{C_3S_1} \\ O_{C_1S_2} & O_{C_2S_2} & O_{C_3S_2} \\ O_{C_1S_3} & O_{C_2S_3} & O_{C_3S_3} \end{pmatrix}.$$

Where: $O_{C_1S_1}$ is the constant production output of the load at loading condition 1 and speed condition 1, $O_{C_2S_1}$ is theconstant production output of the load at loading condition 2 and speed condition 1, and so on.

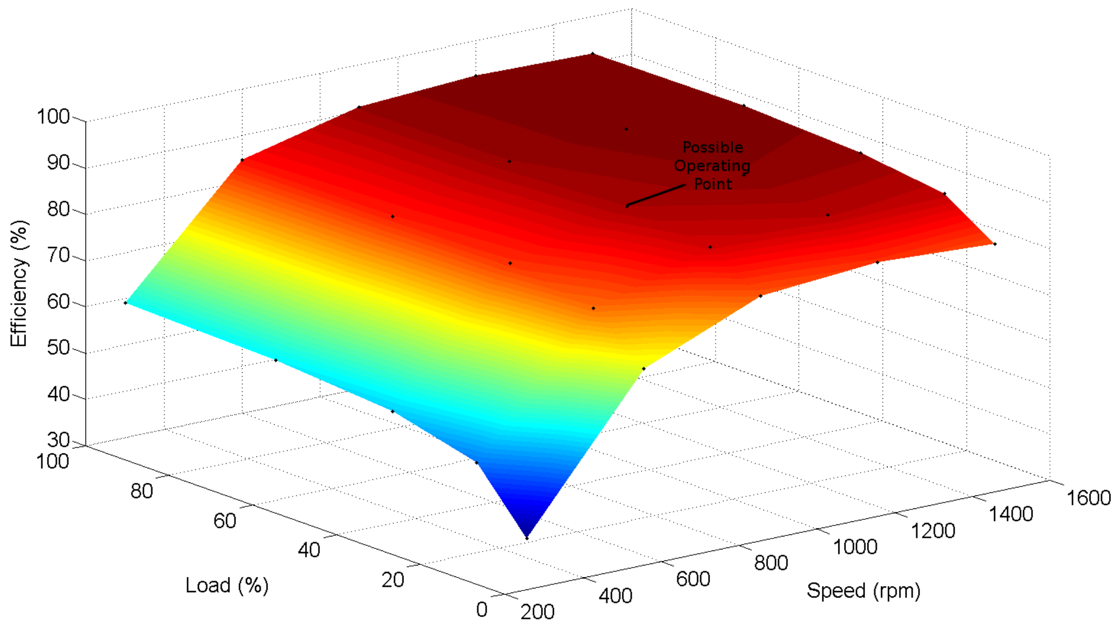



**Figure 6.6: Variable Load Efficiency Graph**
(Load (%) x Speed (rpm) x Efficiency (%))

Another thing to consider is how efficient a specific operating point maybe. Figure 6.6 shows the efficiency graph of the demand graph from Figure 6.5. The operational point efficiency of a load is very important, especially when operating the load in real-life. As we can see from the graph, some points are far from efficient and would not be used in a real-life situation. A load has only one point of maximum efficiency, anywhere else is not as efficient. A load will have to operate outside its maximum efficiency most of the time. Exactly how much must be defined within an efficiency threshold and will be load dependant. The threshold may also be related to the energy cost at the time. For example it may not be electrically efficient to increase the speed of a conveyor or pump. However, if the electricity price is low enough then it may be cost effective and efficient for the overall system operation at that time.

### 6.4.4 Additional Demand Parameters: Load Profiles

The additional demand parameters are to define values such as transient start-up demand, the initial peak of demand as a load starts up before settling into it's steady-state. The parameter is defined with a time value and a demand value. The time value is the time period in seconds from start-up to the end of the transient peak demand. The demand value is the peak demand during the time period. The parameter is defined in the Load Parameter Matrix (LPM) which is defined as follows:

$$\text{Load Parameter Matrix (LPM)} = (t,\ P_{2x})$$

where $t$ is the time period in seconds for how long the additional demand is required, from the starting point to the point of steady-state demand and $P$ is the peak demand during that period.

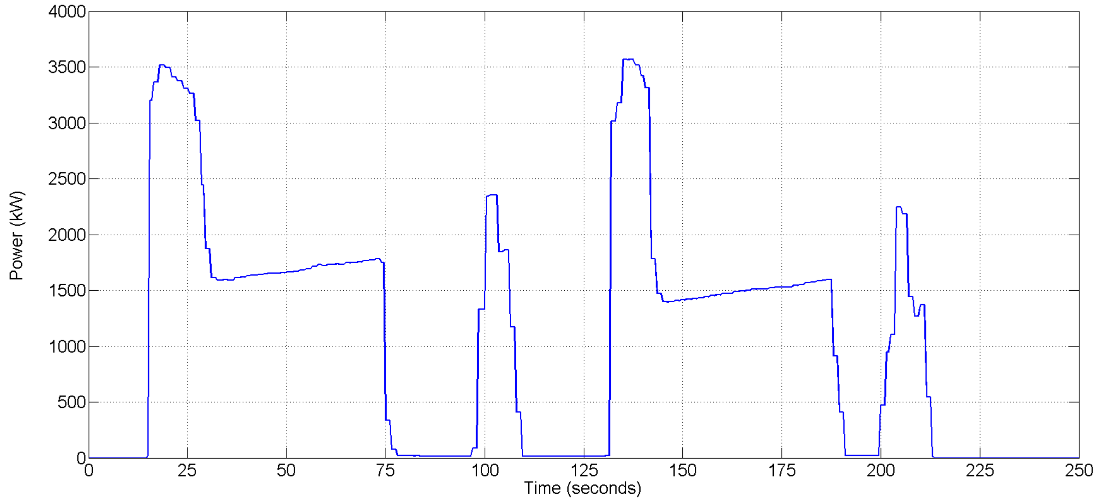



**Figure 6.7: TLP for Mine Hoist** (One complete operational cycle)

### 6.4.5 Non-Steady-State: Load Profiles

The load profiling discussed so far only considers steady state conditions. The load may be variable, but between fixed periods of steady-state load or speed conditions. However, not every load runs under a steady state condition. A load profiling methods must be devised for none steady-state loads as well. Non-steady-state loads are loads that perform a repetitive operation with a non-constant power demand. A perfect example of a non-steady-state load is a mine hoist. Figure 6.7 shows a load TLP of a mine hoist cycle. A large amount of power is required when the hoist starts moving and then drops off when the hoist is in transit (See Figure **??** for a full explination of the cycle).

The principle is much the same as in the previous sections with the operational demand held in a matrix. However, the load profile is split into logical demand blocks (Figure 6.8). Two matrices are defined for the non-steady-state load profile. The Load Time Matrix (LTM) as follows:

$$\text{Load Time Matrix (LTM)} = (t_1,\ t_2,\ t_3),$$

where $t_1$, $t_2$ and $t_3$ are times in seconds for the non-steady-state load demand

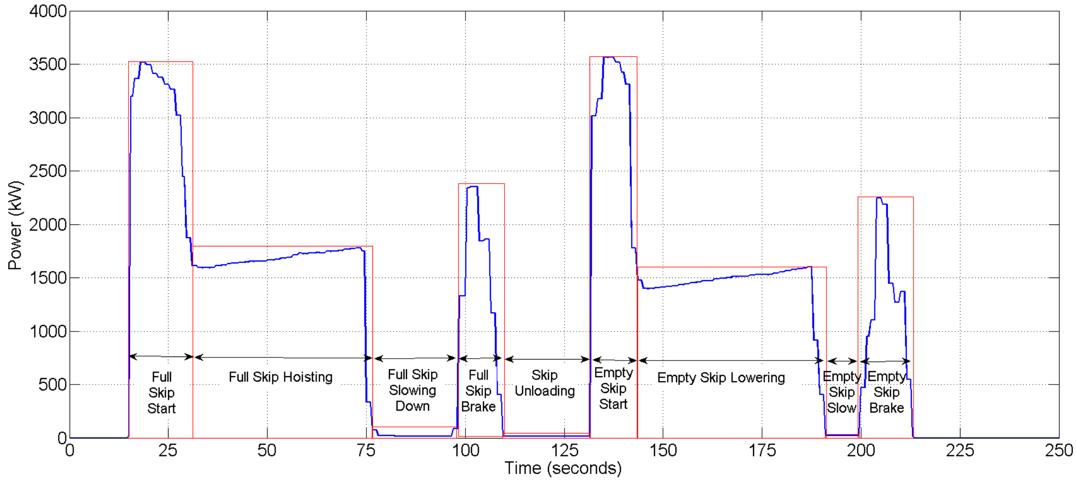



**Figure 6.8: Mine Hoist Demand Blocks** (One complete operational cycle)

blocks, and the Load Block Matrix (LBM) is defined as follows:

$$\text{Load Block Matrix (LBM)} = (B_1,\ B_2,\ B_3),$$

where $B_1$, $B_2$ and $B_3$ are the peak demand values in kW's for the defined non-steady-state load demand blocks. The LTM has the time in seconds for duration of the load block and the LBM has the maximum demand for the LTM. The dimensions of the LTM and the LBM are the same (a single row with the number of columns equalling the number of load blocks used to define the non-steady-state load). The index of the LTM locates the demand for that block and vice versa. The example in the Figure **??** would have a LTM and LBM as follows:

$$\text{Load Time Matrix (LTM)} = (15,\ 45,\ 20,\ 10,\ 12,\ 40,\ 8,\ 12),$$

$$\text{Load Time Matrix (LBM)} = (3500,\ 1800,\ 100,\ 2400,\ 50,\ 3600,\ 1600,\ 2200),$$

There is no LOM or LPM for the non-steady-state load. As the nature of the load is that all the transient power is defined in the LBM. The load has a set operation cycle and does not have a varying output.

### 6.4.6 Reactive Power Load Profiles

The load profiling in this section only considers active power. However, it is proposed to also have a LDM for reactive power. This will allow for the control of the reactive power demand and reduce the need for reactive power demand from the grid or on site generation from capacitor banks. To include both active and reactive power will increase the complexity of the off-line scheduling as two demand values will have to be considered before the best time-block, this has not yet been tested and will not be part of this project.

### 6.4.7 Aggregated Data Load Profiles

From the live production data a load profile database can be compiled, this requires analysis of the data. The ideal solution for the data acquisition system is to have complete distribution of power meters across the mine electrical installation. This would make the task of load profiling the individual loads simpler as each load would have its own power meter and dataset. In reality (including the installation discussed later in Chapter 7) this usually is not the case. Firstly, it is impractical to install hundreds of meters across a mine electrical distribution system and secondly, the cost implications would, in many cases, out justify the means. Data interfaces of modern equipment sometimes include power or energy data that can be interfaced into the SCADA system. This data can be added to the power meter data to improve the detail of the base data. The terminology used for incomplete data is sparse or aggregated data. The loads that have dedicated power meters or communications output are easy to profile as the data is already available. It is not only the loads that have their own dataset that are needed. The individual load data that are aggregated together in system or sub system load profile are also needed. These individual load profiles are difficult to extract into separate load profiles.

#### 6.4.7.1 Non-intrusive Appliance Load Monitoring (NALM)

The process of disaggregating is also know as non-intrusive appliance load monitoring (NALM). NALM started back in the 1980s when Hart (1992) tried first to match a-priori known domestic appliance signatures to an overall power signal by using real and reactive power measurements at a rate of 1Hz. Field tests proved to be successful, especially with larger loads. Norford and Leeb (1996) follow-up research introduced transient event detection at high sampling rates to disaggregate devices with similar power consumption. The majority of the NALM research concentrates on residential data including recent research by Streubel and Bin (2012) and Weiss et al. (2012). Domestic appliances are more distinct in their operation and exhibit a more on/off profile, thus are easier to detect. Wichakool et al. (2009) have developed a method for modelling and estimating current harmonics of variable loads. These harmonics could be used for Non-intrusive Load Monitoring (NILM). NILM moves away from domestic only loads and is more applicable to mine power systems. Akbar and Khan (2007) and Laughman et al. (2003) both use harmonics for NALM. However this is still focusing on household appliances, Lei et al. (2013) present a review of different power system analysis techniques for NILM including harmonic signatures, although they also state that Zeifman and Roth (2011) and Lei et al. (2011) review the current techniques for NIALM (the modern term for NALM) and NILM. Both state that there is no complete NILM solution that is suitable for all types of household appliances. This is also true for commercial and industrial loads. Lei et al. (2011) also states "One solution to solve this problem is combine solutions from different methodologies together to provide the most reliable results. We foresee that the development of algorithms and improvement of software are the future of NILM."

Following the methodology suggested by Lei et al., a solution that could be applied directly to aggregated power data for the disaggregation and definition of the load profile matrices required for this research can be developed. Hart (1992) introduces the concept of appliance signature and defined it as a measurable parameter of the total load that gives information about the nature or operating state of an individual end use in the load. There are two types of appliance signature: non-intrusive and intrusive, we are only interested in the non-intrusive signature. "A non-intrusive signature is one that can be measured by passively observing the operation of a load" (Hart, 1992). There are two types of non-intrusive signature: steady state and transient. Initially only the steady state signatures are considered for the development of the load profiling from aggregated data process. This is not to complicate the process and allow the the process to be built up in stages.

The steady state signature is derived from the difference between the steady state properties of an appliance's operating states. For example, the steady state power signature of a heater is the power difference between its off state and on state. What follows is the procedure to disaggregate steady state aggregated power meter data into individual steady state load vales.

Firstly, we know the total instantaneous power is made of the combined power of all the loads:

$$P_{total} = \sum_{k=1}^{n} P_k, \tag{6.5}$$

and from either Equation 4.6 and 4.7 the total power of each load is defined as:

$$P_{load} = \sqrt{3} V_{load} I_{load} \cos \theta_{load}. \tag{6.6}$$

Substituting equation 6.6 into equation 6.5 we get:

$$P_{total} = \sum_{k=1}^{n} \sqrt{3} V_k I_k \cos \theta_k. \tag{6.7}$$

However, as the loads are not the same, more information is needed to disaggregating the $P_{total}$. The disadvantage of aggregated data is that the start and finish of the load signature is unknown. NIALM and NILM research aims to find these start and finish locations of the load signature solely with analysis of the aggregated load profile. The advantage of industrial (and some commercial) installations is that they generally have SCADA systems. The SCADA systems usually monitor and record the start and stop operations of the loads across the plant or in this case the mine. The start and stop information removes the guess work in finding the start and finish locations of a load signature. Correlating the SCADA and aggregated power data provides a data set that can be analysed with initial level of detail that is not always available.

It can be assumed that not all loads are running all of the time and different loads are running simultaneously at different times. What follows is a procedure to define steady state load profiles from correlated SCADA and aggregated power data. The detail we know is what loads are running and their total power demand. Table 6.2 give a example of what how the data set could look for five loads over six time slots.

| Time Slot | Load 1 on/off | Load 2 on/off | Load 3 on/off | Load 4 on/off | Load 5 on/off | Total Power |
|---|---|---|---|---|---|---|
| **1** | 1 | 1 | 1 | 0 | 0 | $P_{T1}$ |
| **2** | 1 | 1 | 0 | 1 | 0 | $P_{T2}$ |
| **3** | 0 | 0 | 0 | 1 | 1 | $P_{T3}$ |
| **4** | 0 | 1 | 0 | 1 | 0 | $P_{T4}$ |
| **5** | 0 | 0 | 0 | 1 | 0 | $P_{T5}$ |

**Table 6.2: Possible SCADA/Power Data Set**

As the load data we are dealing with is steady state the power demand for an individual load will be the same (or virtually the same, within a very close tolerance) every time the load is running. The data can be considered and solved as simultaneous linear equations (See Equations 6.8).

$$
\begin{aligned}
P_{L1} + P_{L2} + P_{L3} &= P_{T1}, \\
P_{L1} + P_{L2} + P_{L4} &= P_{T2}, \\
P_{L4} + P_{L5} &= P_{T3}, \\
P_{L2} + P_{L4} &= P_{T4}, \\
P_{L4} &= P_{T5}.
\end{aligned}
\tag{6.8}
$$

These can be solve with matrix multiplication:

$$
\underbrace{\begin{pmatrix} L_1 & L_2 & L_3 & \cdots & L_n \\ L_1 & L_2 & L_3 & \cdots & L_n \\ L_1 & L_2 & L_3 & \cdots & L_n \\ \vdots & \vdots & \vdots & & \vdots \\ L_1 & L_2 & L_3 & \cdots & L_n \end{pmatrix}}_{=A} \cdot \underbrace{\begin{pmatrix} P_{L1} \\ P_{L2} \\ P_{L3} \\ \vdots \\ P_{Ln} \end{pmatrix}}_{=x} = \underbrace{\begin{pmatrix} P_{T1} \\ P_{T2} \\ P_{T3} \\ \vdots \\ P_{Tn} \end{pmatrix}}_{=b}
\tag{6.9}
$$

The matrix $x$ contains the unknown demand values for each of the loads. So, if

$Ax = b$, then $b/A = X$. However, in matrices we cannot divide, although we can multiply by an Inverse Matrix, which achieves the same results. So we now have $AxA^{-1} = bA^{-1}$, we know that $AA^{-1} = 1$, we end up with $1x = bA^{-1}$ or $A^{-1}b = x$.

Here is an example using the data from Table 6.2 and some example power values. We from the SCADA on/off signals we know the coefficients matrix ($A$) and from the total power we know the column matrix ($b$):

$$A = \begin{pmatrix} 1 & 1 & 1 & 0 & 0 \\ 1 & 1 & 0 & 1 & 0 \\ 0 & 0 & 0 & 1 & 1 \\ 0 & 1 & 0 & 1 & 0 \\ 0 & 0 & 0 & 1 & 0 \end{pmatrix}, \quad b = \begin{pmatrix} 200 \\ 150 \\ 80 \\ 135 \\ 35 \end{pmatrix}, \tag{6.10}$$

next we need the inverse of $A$:

$$A^{-1} = \begin{pmatrix} 0 & 1 & 0 & -1 & 0 \\ 0 & 0 & 0 & 1 & -1 \\ 1 & -1 & 0 & 0 & 1 \\ 0 & 0 & 0 & 0 & 1 \\ 0 & 0 & 1 & 0 & -1 \end{pmatrix}. \tag{6.11}$$

We can then solve $A^{-1}b = x$:

$$\underbrace{\begin{pmatrix} 0 & 1 & 0 & -1 & 0 \\ 0 & 0 & 0 & 1 & -1 \\ 1 & -1 & 0 & 0 & 1 \\ 0 & 0 & 0 & 0 & 1 \\ 0 & 0 & 1 & 0 & -1 \end{pmatrix}}_{=A^{-1}} . \underbrace{\begin{pmatrix} 200 \\ 150 \\ 80 \\ 135 \\ 35 \end{pmatrix}}_{=b} = \underbrace{\begin{pmatrix} 15 \\ 100 \\ 85 \\ 35 \\ 45 \end{pmatrix}}_{=x}. \tag{6.12}$$

So the power demand values for the five loads are:

$$\begin{aligned} P_{L1} &= 15kW, \\ P_{L2} &= 100kW, \\ P_{L3} &= 85kW, \\ P_{L4} &= 35kW, \\ P_{L5} &= 45kW. \end{aligned} \tag{6.13}$$

The solving for the load demand a values here is relatively simple and is fine for a small number of loads. The matrix multiplication will also only work when the coefficients matrix ($A$) is a square matrix, as an inverse can only be found for square matrices. In reality there may be more than five loads, although the results will be more accurate if small amounts of loads are disaggregated at a time. Due to the time period required of a dataset, the amount of time samples will nearly always be more than the number of loads. This will make the coefficients matrix ($A$) non-square. The non-square matrix can be dealt with using the a pseudo inverse matrix and calculated by singular value decomposition (SVD). The disaggregation will not be done by hand and will be solved using MATLAB. MATLAB has functions that can create the pseudo inverse matrix and multiply matrices. In MATLAB the matrices can also be divided, directly solving for the values matrix ($x$). The matrices are not divided as this is not possible. However, MATLAB takes care of the inversion and multiplication of the matrices.

Next is was necessary to develop a method of defining the load profiles from automatic analysis of a data set using MATLAB. It was considered that using live data from the outset would prove to be too complicated. The reason for this is that the real data can contain noise and the demand of a real load may not be constant and can fluctuate around an average demand. Real live data will also require filtering and preparing before any attempt is made to disaggregate it into separate load profiles. The development of the live data processing is presented in Chapter 8. It was decided that a basic system would be created first using steady state loads.

A simulation model was built by the author in SimPowerSystems™, a MATLAB® Simulink® module that provides component libraries and analysis tools for mod-

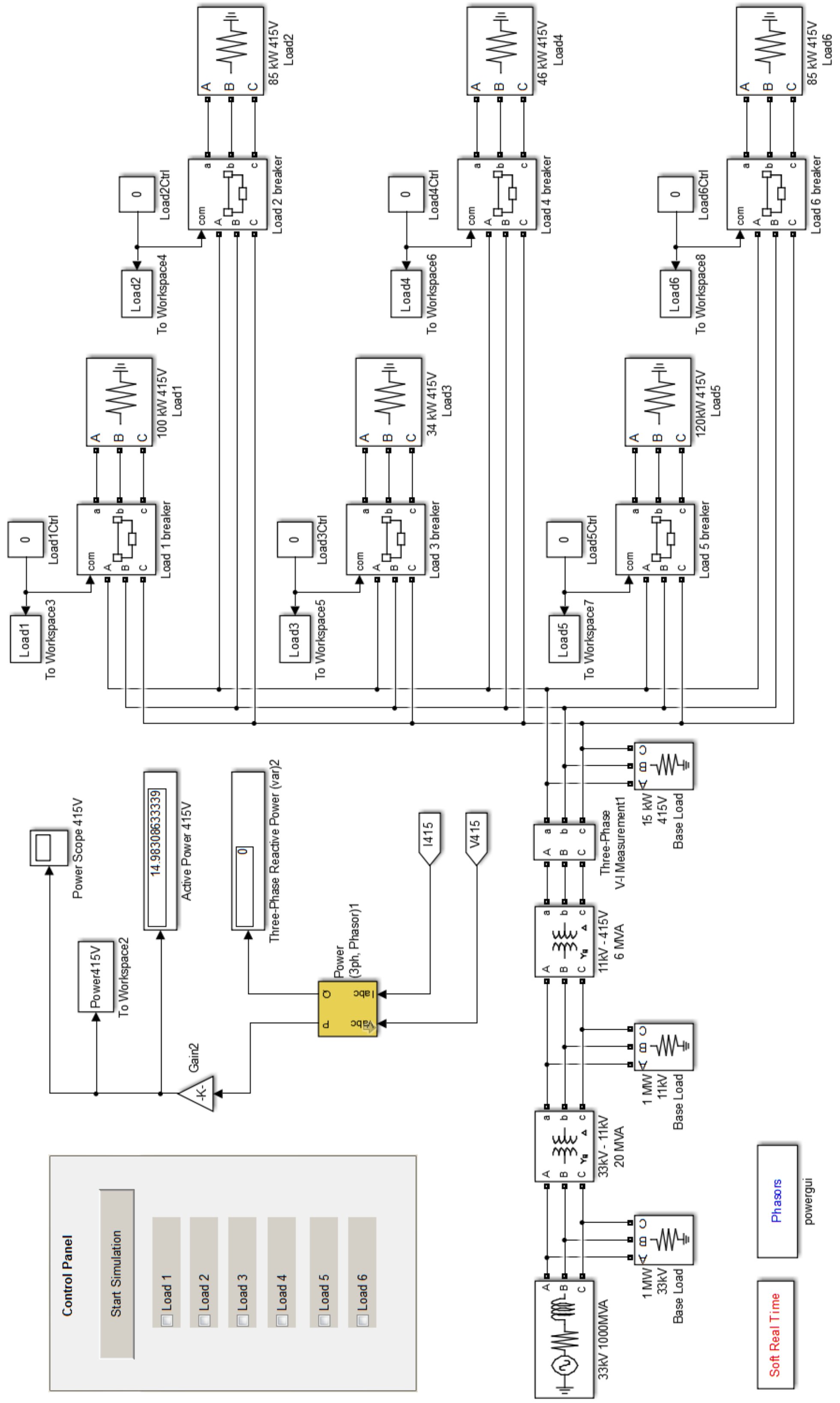



**Figure 6.9: SimPowerSystem Model and Control Panel** (Six resistive loads for load profiling)

elling and simulating electrical power systems. The model was built with six resistive loads, Figure 6.9 shows a screen shot from Simulink. For the initial system developmentResistive loads were selected for their linear characteristics and steady state power demand. The model was built with a control panel that can switch on and off the loads manually during simulation (Figure 6.9). The simulation was run for a set period of 300 seconds, an additional third party Simulink block 'Soft Real Time' (Rouleau, 2008) to reduce the speed of a Simulink model to real time. This was important so that the load control panel could be used to generate the output data. A coefficient matrix was defined from the simulation and control panel operation. A row entry was made with a sampling rate of 500 milliseconds. During the simulation the loads were switched on and off giving a data set of five minutes long and a coefficient matrix with 600 rows and six columns (one for each load). MATLAB code was developed to disaggregate the aggregated data created by Simulink model.

During the operation of the simulation with no loads switch on there was a power demand overhead. This overhead was from the inclusion of the base-load modules that are required by Simulink transformer modules to operate properly. This can be considered to be the simulation of real-life system base-load. To compensate for this in the model and the MATLAB code, an additional column was added to the coefficient matrix for the base-load. This column consisted only of 1's as the base-load is always present during system operation and this had the same affect as running a load all the time.

| | Simulation Run | | | | | |
|---|---|---|---|---|---|---|
| **Load** | **1** | **2** | **3** | **4** | **Average** | **Accuracy** |
| **Base (15 kW)** | 15.037 | 15.036 | 15.051 | 15.208 | 15.071 | 0.005 |
| **1 (100 kW)** | 99.238 | 99.229 | 99.235 | 99.186 | 99.220 | 0.008 |
| **2 (85 kW)** | 84.433 | 84.429 | 84.415 | 84.380 | 84.414 | 0.007 |
| **3 (34 kW)** | 33.876 | 33.873 | 33.873 | 33.847 | 33.863 | 0.004 |
| **4 (46 kW)** | 45.798 | 45.791 | 45.790 | 45.767 | 45.788 | 0.005 |
| **5 (120 kW)** | 118.930 | 118.935 | 118.923 | 118.886 | 118.928 | 0.009 |
| **6 (85 kW)** | 84.431 | 84.434 | 84.435 | 84.393 | 84.427 | 0.007 |

**Table 6.3: SimPowerSystem Model Simulation Results**

The simulation was run four times and the code was used each time to solve the different dataset. Table 6.3 show a snapshot of results from four simulation runs.

The table also shows the average of these runs and the accuracy of the results compared to the original load size. The results were conclusive in the fact that the accuracy of the results is more than 99.9%. This level of accuracy is far more than is required for a real life situation.

This analysis was be carried out on all of the SCADA data to complete the load profile database; this database will then provide the building blocks for the demand reduction strategy algorithm. The accuracy and detail of the load profile database has a great reflection on the load management results.

## 6.5 Stage 3: Load Characterisation

This section explains stage three of the demand reduction and DSR system, load characterisation. At first it could be considered that load characterisation is the same as load profiling. However, it is not. Load profiling is only concerned with the power demand of a given load. Load characterisation is the character of the load within the system and is defined as follows: How the load is operated? Is it reliant or dependent on the other loads in the network? Can the load operate with or without other load? Or is the load in a group of loads that operate together or in sequence? The characterisation of a load is not straight forward and must be done in steps. These steps are listed below:

**Step 1** Main Categories.

**Step 2** Allocate Load Group.

**Step 3** Define Load Characteristics.

### 6.5.1 Step 1: Main Categories

The first step is to define the main categories into which the loads fits. Three main categories have been defined, these are as follows:

- Constant Loads.
- Free Loads.
- Cyclic Production Loads.

#### 6.5.1.1 Constant or Fixed Loads

Constant loads are loads that exist in the system, although their operation can not be changed or removed. The load does not need to be running constantly. However, it must have a fixed operation cycle that can not be changed, for example due to safety or production/operational constraints. Main ventilation fans are a good example of a constant or fixed load. Constant or fixed loads are not included as part of the scheduling optimisation in the fact that they exist as part of the base-load demand. The free and cyclic loads are scheduled around them as their power demand is a constraint. Fixed loads are still free to have any energy efficiency or modernisation techniques applied to reduce demand and save costs.

#### 6.5.1.2 Free Loads

Free loads are loads that can operate without safety or production/operational constraint. The load can have constraints on how often it must operate; however, these time constraints can be managed so the load can operate at different times. Scheduling of free loads mainly focuses on the valley filling and price advantages. Examples of free loads are, pumps, development process and the hoist.

#### 6.5.1.3 Cyclic Production Loads

Cyclic production loads are loads that the mine operation and production rely on and can not continue without. These loads maybe in groups or individual; however, most likely have constraints that they need other loads or groups of loads to operate before, during or after their operation. Due to the nature of the relatively fixed mine production cycle, power demand for a given production output is optimised. However, different levels of production output can be scheduled to fluctuate depending on price or maximum demand. Examples of the cyclic production loads are the face machinery or the conveyor system.

The rest of this chapter contains flow diagrams. These diagram are to illustrate a system in a logical way. There are different symbols to show different types of operations, Figure 6.10 show these different symbols and their meanings.

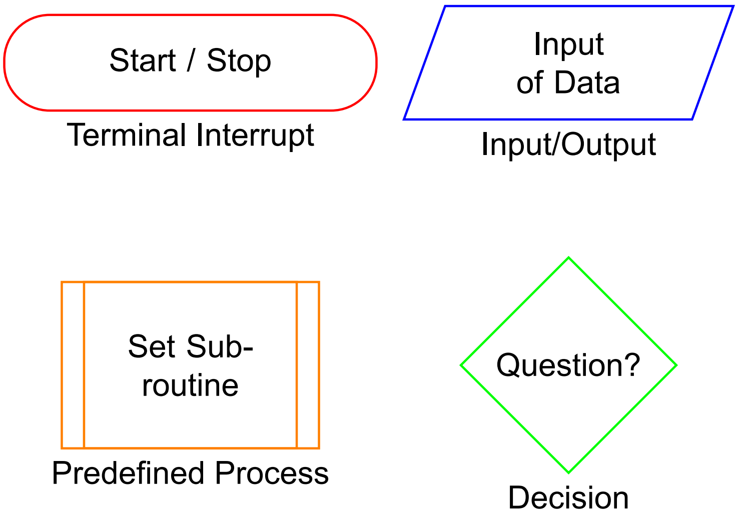



**Figure 6.10: Flow Diagram Key**

Figure 6.11 shows the flow diagram of the sequence required to define the main category of a load. All the loads in the mine are put into a category and as previously mentioned, the fixed loads are left to operate as they are and the cyclic production and free loads then go through the next step, allocating load groups.

### 6.5.2 Step 2: Allocate Load Group

The production of an underground coal mine is pretty much set and follows a constant routine of extraction, underground transportation, and finally transportation to surface. This is either by hoisting or conveyor, depending on the depth of the mine. With Longwall mining there is also the development of the next extraction face of the mine. Room-and-Pillar does not have the development stage as the nature of the process is like one large development. Figure 6.12 shows a block

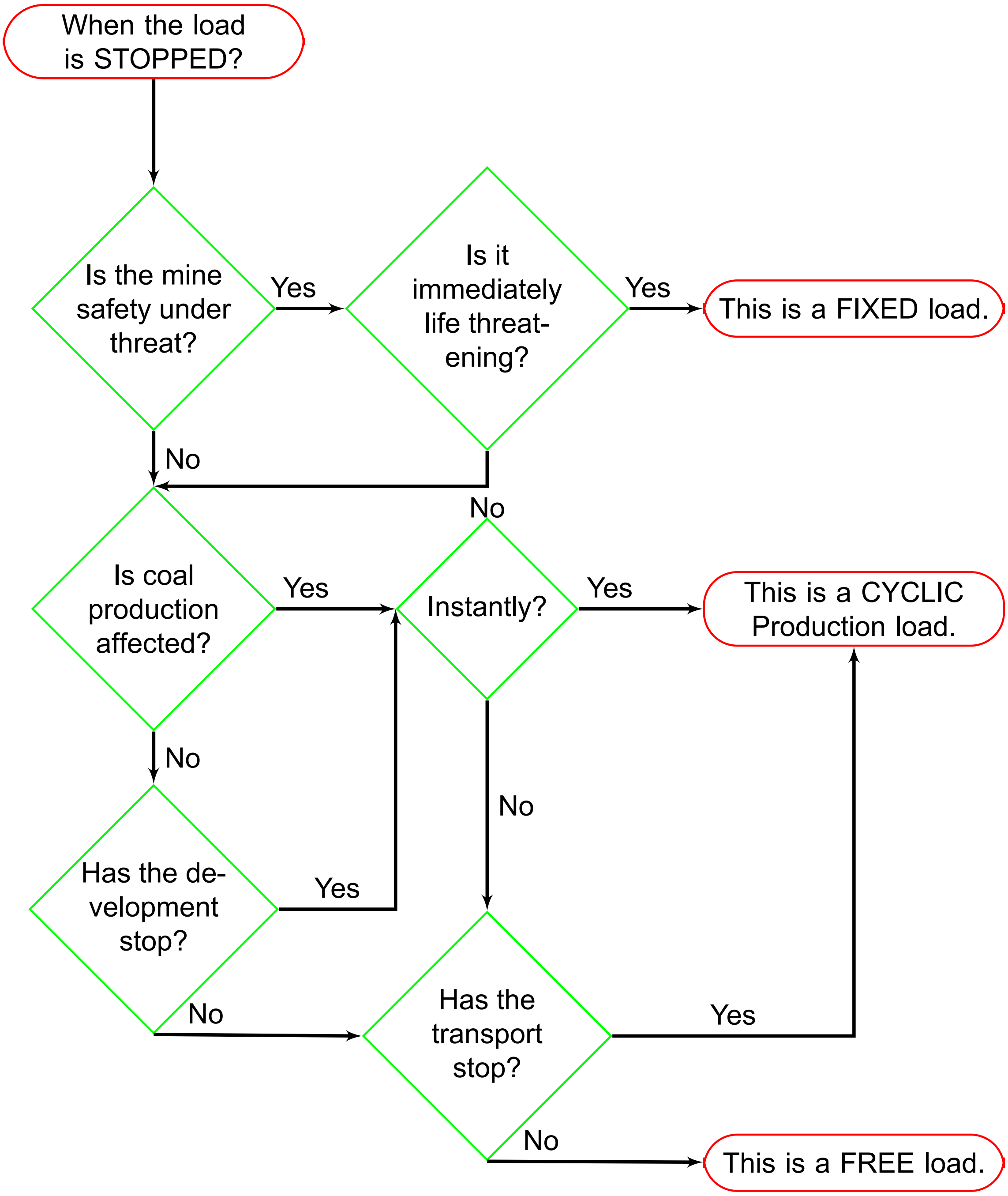



**Figure 6.11: Main Category Flow Diagram**

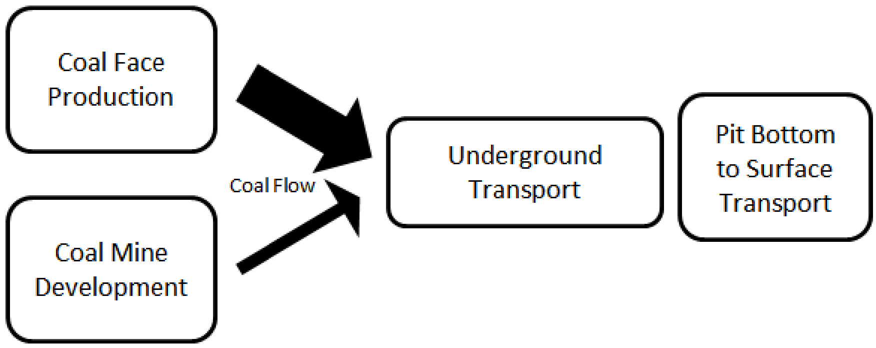



**Figure 6.12: Longwall Mine Block Diagram** (Coal flow and mine layout.)

diagram of the Longwall mine layout and the different areas or sections of the mine. These areas relate to the load groups. Only the cyclic production and the free loads are put into groups. The free loads go straight into the free load group. It is the cyclic production loads that require more processing to allocate them into their correct groups. Figure 6.13 shows the flow diagram of the sequence required to define the load group for the cyclic production loads. After the loads have been put into their load groups they can then go through the final step of load characterisation.

### 6.5.3 Step 3: Define Load Characteristics

The loads have now been categorised and grouped. The last step is to define the characteristics of these grouped loads. The load characteristics define the way that the load exists with the network and its group. The following list outlines possible load characteristics:

- Load Running Time.
- Load Running Interval.
- Regular or Random Load.
- Any Linked Loads or Patterns.
- Overlap Running of Linked Loads.

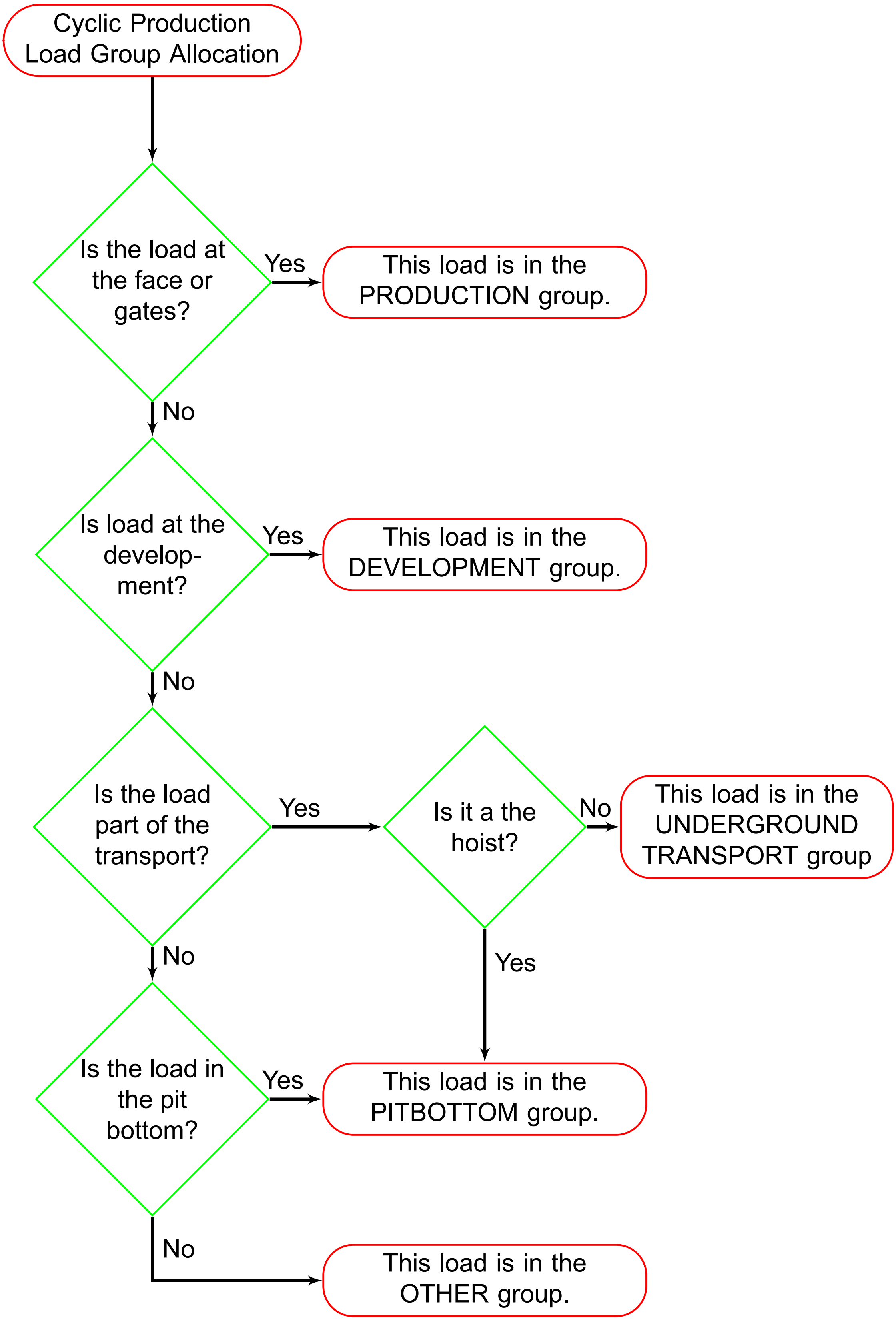



**Figure 6.13: Main Category Flow Diagram**

The characteristics are defined by analysing the SCADA data, the data that provides the operational periods of each load. There is no power data required for this step, all the power data analysis is done in stage 2 (Stage 2: Load Profiling). The list is not definitive as new loads may create new characteristics and not all characteristics will be relevant to every load. An example of this is the free loads. By nature they are free and will only have one characteristics, the load running time. The analysis process is not detailed here as it requires real data to present properly. Chapter 8 presents the analysis process and results from the live data collected from the installation work at the UK Coal Colliery.

### 6.5.4 Load Classifications MATRIX

As we have seen, the three steps of load classifications involves manual input to complete the process. A fully automated system is the ultimate goal. If a system could take a data set and produce the optimised load schedule with minimal manual input this would have a enormous impact on all industrial processes, not only mining. To increase the possibility of automating the entire network optimisation system from data analysis to scheduling has inspired the development of the Load Classifications MATRIX or LC Matrix by the authour.

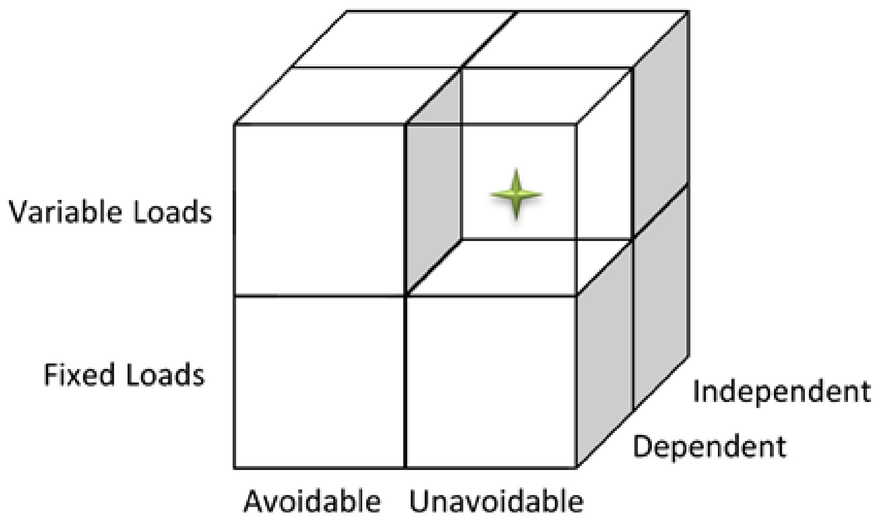



**Figure 6.14: Load Classifications MATRIX** (LC MATRIX)

| Load | Speed | Avoid-ability | Dependency |
|---|---|---|---|
| **1** | 1 | 1 | 0 |
| **2** | 0.5 | 0.8 | 1 |
| **3** | 0.8 | 0.6 | 0 |
| **4** | 1 | 1 | 0.8 |
| **5** | 0.7 | 0 | 0 |
| **6** | 1 | 0 | 0 |

**Speed** (0 = Stopped - 1 = Fixed)

**Avoid-ability** (0 = Avoidable - 1 = Unavoidable)

**Dependency** (0 = Independent - 1 = Dependent)

**Table 6.4: LC Matrix Values Table**

Figure 6.14 shows a graphical representation of the Load Classifications MATRIX (LC MATRIX) which has been devised to group the load profiles, using this matrix, loads can be easily defined into their categories by allocating their constraint characteristics. The matrix represented here only has three dimensions as this is the maximum number of dimensions that can be visualised. The point defined on the matrix in Figure 6.14 is an unavoidable, dependent, variable load. This is only one load. In reality there would be tens or even hundreds of loads and the LC MATRIX could have many dimensions. Although, as the dimensions increase the impact on the load profile would be smaller and smaller. This approach to load profiling allows greater definition without continuously defining new terms. The number of categories is defined by $2^n$ where $n$ is the number of terms, the three dimensions allows for eight categories ($2^3$).

The LC MATRIX allows for the definition of the load profile constraints. However, the demand data may have to be extracted from aggregated power data and the load profiles as in stage 2, will still have to be defined. This an early stage development into automated definition of a load constraint characteristics. The concept uses a numerical matrix to define the load characteristics. Table 6.4 contains example data for six loads as they would be defined using the LC Matrix method. The table key shows the values. It can be seen that the values of the table are proportionally representative of how much a load is avoidable and dependent (load speed is also defined, although this value is yet to be fully utilised). zero being completely avoidable i.e. free and one being completely unavoidable i.e. fixed.

The dependency value is the same, zero being completely independent and one being completely dependent. The dependency is different to the avoid-ability value, as it is to do with the load groups and if a load must run together with or without another load. The reason for not defining the characteristics with a straight 0 or 1 is that there is no distinction between loads. It was considered that the value can be used to rank the load in an order. In Chapter 8 it is tested on the real mine data. However, in the next section, how the LC MARTIX feeds into the load scheduling is described and tested using simulated data from the model used in Stage 2: Load Profiling.

## 6.6 Stage 4: Load Scheduling

The last stage of the network optimisation is load scheduling, this involves ordering the load start-up and running to give the maximum production with minimum electrical demand. There are two points to this scheduling and load operation, off line scheduling and real-time operation.

### 6.6.1 Off-Line Scheduling

Optimising a systems operation is done by the operation of the loads to get the maximum output from the minimum input. Initially this can be managed off-line to give the theoretical best performance. The load profiles created in stage 2 are put in their groups and fitted together like building blocks, the load characteristics then release or restrict the time-line to define the load operating point and duration. The optimisation of the time-line has many options, such as, maximum output, minimum demand, maximum production with minimum transportation, maximum transportation with minimum production. The options are depicted by the response that is required, firstly by the DSR signal and by the previous operation. For example, the last DSR signal may have reduced production, but not transportation, so next time demand is maybe optimised to increase transportation in order for the mine to gain momentum and get back on track for daily production targets. It may not be an external DSR signal that is responsible for the existing operation mode, and emergency of breakdown could be the reason.

The load characteristic and development of the LC MATRIX have also been extended into off-line scheduling. So far this is only at a basic level, although it is at a point when it can be used for off-line scheduling. The list that follows outlines the definitions that have been made for four possible combinations of load characteristics types:

**A Loads** Unavoidable and Dependent.

Ranking order of avoid-ability.
No adjustment to profile position is made.

**B Loads** Unavoidable and Independent.

Ranking order of avoid-ability.
Whole operational profile is shifted to minimum demand.

**C Loads** Avoidable and Dependent.

Ranking order of dependency.
Each operational block is shifted to minimum demand.

**D Loads** Avoidable and Independent.

No ranking.
Load is used for valley filling.

Figure 6.15 shows how these different load types are scheduled for minimum demand. These definitions are not fully defined and are not presented as final. As with the LC MATRIX they are in early the stages of research and will also evolve as they are tested and developed. The real mine data in Chapter 8 will help in the development of these definitions. Other dimensions and rule definitions may be added as the data requirements presents themselves. The A,B, and C Loads are kept in the original operation blocks, just shifted forward or backward along the time-line for the lowest demand (Figure 6.15). How the off-line scheduling module makes its decisions for each load type is as follows;

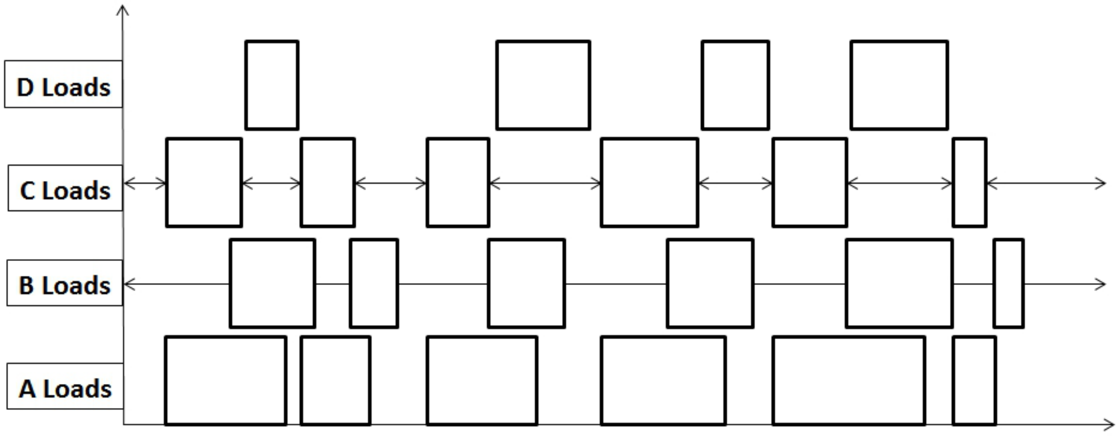



**Figure 6.15: LC MATRIX** (Load Type Off-Line Scheduling)

A Loads are fixed and have no adjustment made to their profile, time-period of the off-line scheduling operation is defined and the A Load on/off operation is left. From the load profile the model knows the demand of the each load and a demand curve for all the A Loads is then calculated.

Next is the B Loads. These are kept in the operational pattern; however, the whole time-period profile of the load is shifted along the time-line and positioned where the demand is the lowest. The start of the B Load profile (this is the beginning of the first operation of the load) is positioned at the beginning of the time-period along with the A Load profile and then the demand is calculated along the whole time-line for both A and B Loads and stored. The whole B Load profile is then shifted one time frame (500 millisecond in this case) and the demand is calculated again, this step is repeated until the end B Load operational block has reached the end of the time-period. The demand for each time frame shift has been stored and the B Load profile is scheduled at the time frame with the lowest overall demand. This is repeated for all the B Loads and then the final A and B Load profile is stored ready for the C Loads.

The C Loads are similar to the B Loads, except that each operational block is shifted individually, they are kept in order and shifted through the time frame within its time slot. Each operational block has a time slot, this time slot size is calculated as follows:

$$\text{Time Block Size} = \frac{\text{(Time Period-Total Load Operational Time)}}{\text{Number of Operational Blocks}}.$$

The first C Load block is shifted from the start to the end of its time slot and scheduled at the point of the lowest demand, the same is then done to the next block. However, any remaining time slot left after the previous block is added to the beginning of the current time slot. This is continued until all the blocks have been scheduled and repeated for all the C Loads.

Finally the D-Loads are scheduled, the D-Loads being used for valley filling. All the operational blocks are allocated into one block and the optimised A, B, and C load profile schedule is scanned and the lowest demand valley is filled with the amount of D load operation that covers the low demand figure. The valley filling process is repeated until there is no more D Load operation time left unscheduled.

There has been no indication of how long a time-period is and this is not a set period. The time-period would be unique for the given system and analysis would be needed to determine its optimum length. However, the time-period that would probably be used would be one hour or 60 minutes. The reason for this is that most production figures are commonly given in amounts per hour. It must also be noted that, if a time-period was too long it would cause difficulties with scheduling, as loads may become bunched up or too spread out.

At first it was thought that the LC MATRIX would be used to schedule a complete system at once. As the development progressed, it was soon apparent that dealing with the whole system at once would, firstly be very complicated and difficult to manage and secondly would introduce inaccuracy in scheduling producing schedules that would not work in real life. To reduce the complications and prevent impossible schedules, the LC MATRIX will be used to schedule individual groups of load first. Once the groups have been scheduled, then the groups can be scheduled together. This will produce a complete system schedule built in layers and will ensure that the schedules will work in real life.

The Simulink model used in Section 6.4.7 (Figure 6.9) was also used to simulate a data set for use with the LC MATRIX off-line scheduling. At this point very little attempt was made to create simulation data representative to real mine data. As previously mentioned in Section 6.4.7, the simulation data to develop and test the concept and then in Chapter 8 real data will be used to continue the research. The simulation was run, the different loads were switched on and off in a vague pattern and sometimes randomly to simulate different loads. This data was stored and then an LC MATRIX was defined to give one of each of the load characteristic types. MATLAB code has been developed to perform the functions of the ranking and load shifting. As seen in Figure 6.15 the blocks were shifted in turn to produce a new load operation schedule with a reduced power demand.

#### 6.6.1.1 Selecting the Lowest Demand

Initially, it was thought that the lowest demand time-slot was the one where the load block had the lowest value for a given time frame. The time-period of the load block is split into time-slots and the demand for each time-slot is calculated giving a list of values. As the load block is shifted there is a new list of demand values for the new load block position and every other position that the load block is tested in. The demand list with the lowest value does not necessarily represent the load block location with the lowest demand. As this position may also have the highest demand (a peak and a valley due to the other loads on the time-line). The mean is not used as this would increase the inaccuracy of the peak and trough demand selection.

To ensure that the lowest demand position has been selected, at each time-slot the lowest demand and the number of occurrences was recorded. After the load block has been shifted through the time-block the demand values are analysed and the block position with the maximum number of occurrences of the lowest demand was selected. This is the mode of the lowest demand. However, this could still contain a high demand peak. To ensure that the time-slot has the lowest demand instead of the lowest demand point, the standard deviation of the demand for each time-slot is used. The first occurrence of the lowest standard deviation is then used to select the time-slot for the load block.

Figure 6.16 is a graph of the demand curve for a simulation run. The graph shows

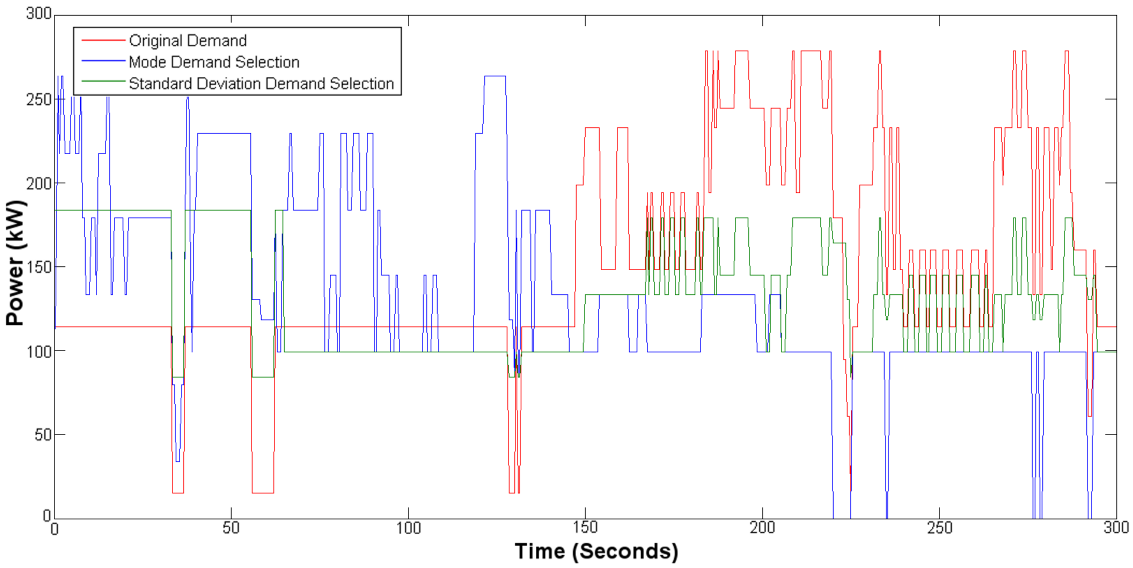



**Figure 6.16: LC Matrix** (Demand Curve Comparison)

three curves; the original; optimised with the mode average demand selection; and the standard deviation demand selection. Table 6.5 shows the basic values from the graph. It can be seen that not only using the standard deviation method to select the time-slot for the load operation block gives the lowest demand, it also give a smoother demand curve that has less peaks and troughs and comparing the three standard deviations values in Table 6.5 highlights how efficient the LC MATRIX is at reducing demand for the off line scheduling.

| Demand Curve | Max (kW) | Min (kW) | Std (kW) |
|---|---|---|---|
| **Original** | 278.39 | 14.98 | 61.99 |
| **Mode** | 263.40 | 0 | 56.49 |
| **Standard Deviation** | 183.72 | 84.44 | 35.14 |

**Table 6.5: LC Matrix Demand Curve Comparison**

## 6.6.2 Real-Time Scheduling

Once the off-line scheduling has been completed, this schedule is then put into operation. This is an optimum operation of the industrial process, underground coal mining in this case. The schedule does not take into consideration the real life situation and expects things to run smoothly. This is almost always not the case, once the schedule is running or on-line, loads may breakdown, fail to start, run for longer etc. This all has an effect on the on-line schedule and the production output of the mine. As events happen decisions must be made to recover the on-line schedule to the optimum operation if the optimum electrical demand and production output is to be achieved.

If there is a breakdown then this reduction in demand can be capitalised by another load or group of loads until the interruption is rectified. This will allow for a production recovery later in the schedule as the capitalising load may not need its later time-slot. Furthermore, if a load overruns its scheduled time-period this will also have an effect on demand and production and may stop additional loads from starting, thus missing their time slot.

The constant changing nature of the real-time production schedule posses the problem of how to manage the production process in real-time. Repeated off-line scheduling with reference to the historical operation of the mine, i.e. have the loads fulfilled their operation time-slots or do they still need to run? Has the load missed their time-slot or have they operated earlier? In addition to this, what is the current priority ranking for the load? Are any loads at critical ranking and must be operated regardless?

What follows, is the detailed explanation of the real-time decision making, central control module (CCM), the priority ranking engine (PRE) and the demand control module (DCM) that make up the intelligent network optimisation system. It is mainly presented with the use of flow diagrams, starting with Figure 6.17, this is the flow diagram for the real-time CCM. The load profile data for the load is sent to the CCM with the request to start. The network state is analysed by the PRE and DCM and then the CCM then responds with a signal to start or wait depending on the outcome from the PRE and the DCM.

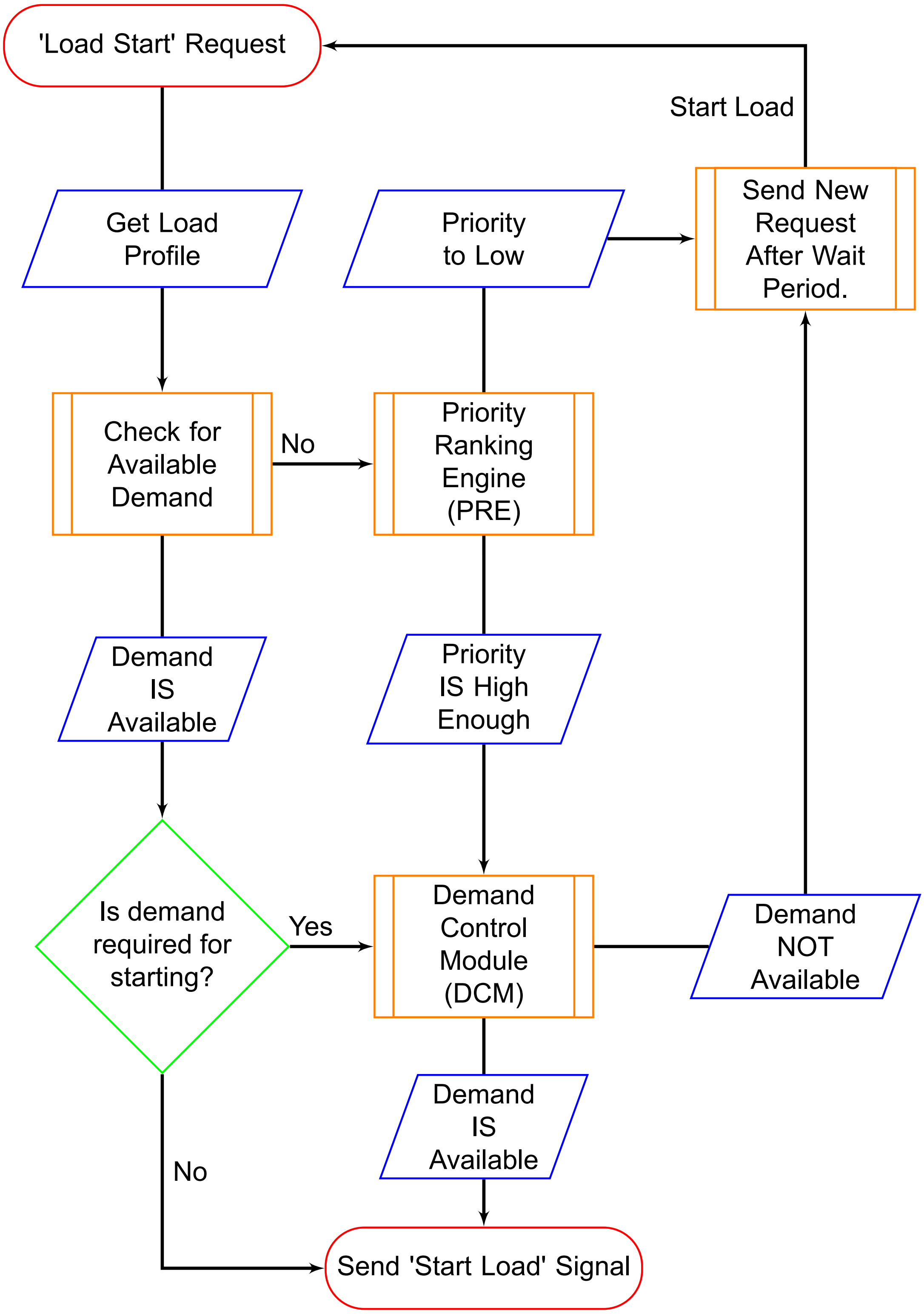



**Figure 6.17: Central Control Module (CCM)**

#### 6.6.2.1 Real-Time Demand Availability Check

Before the PRE and DCM analyse the system, there is a simple demand availability check, this is a simple logical calculation and is as follows:

IF $(Maximum\ Demand\ -\ CurrentDemand) > (Required\ Demand)$

THEN

$The\ Demand\ Available.$

ELSE

$Demand\ NOT\ Available.$

END.

After this if there is enough demand and there is no additional demand required as defined by the LAM (Equation 6.5), the CCM will respond with a signal to start the load. If the available demand is not enough for the load to run of then the PRE will consider if the load has priority to start.

#### 6.6.2.2 Priority ranking engine (PRE)

The flow diagram in Figure 6.18 shows the operation and decision making of the PRE. If the PRE makes the decision to request the demand or there is enough demand to run the load, but not to start it then the DCM will manage the network operation to provide enough demand to allow starting of the load. The PRE may not decide to request the demand and respond with a 'Priority to Low' signal, the load is then placed into cue and a new request to run will be sent. Depending on the priority the load my wait for automatic restarting request or drop until a manual or scheduled start request reoccurs. The length of time the load is kept waiting will also depend on its priority and characteristics.

The PRE requires the 'Time Priority' to be calculated, the time priority considers the loads that must run for a set amount of time for every time-period. These are generally free loads, such as pumping that must pump a certain amount e.g. pump for 35 minutes every hour. However, the cyclic loads also have a time priority, this for the amount of running time required for a specific production output. If the load

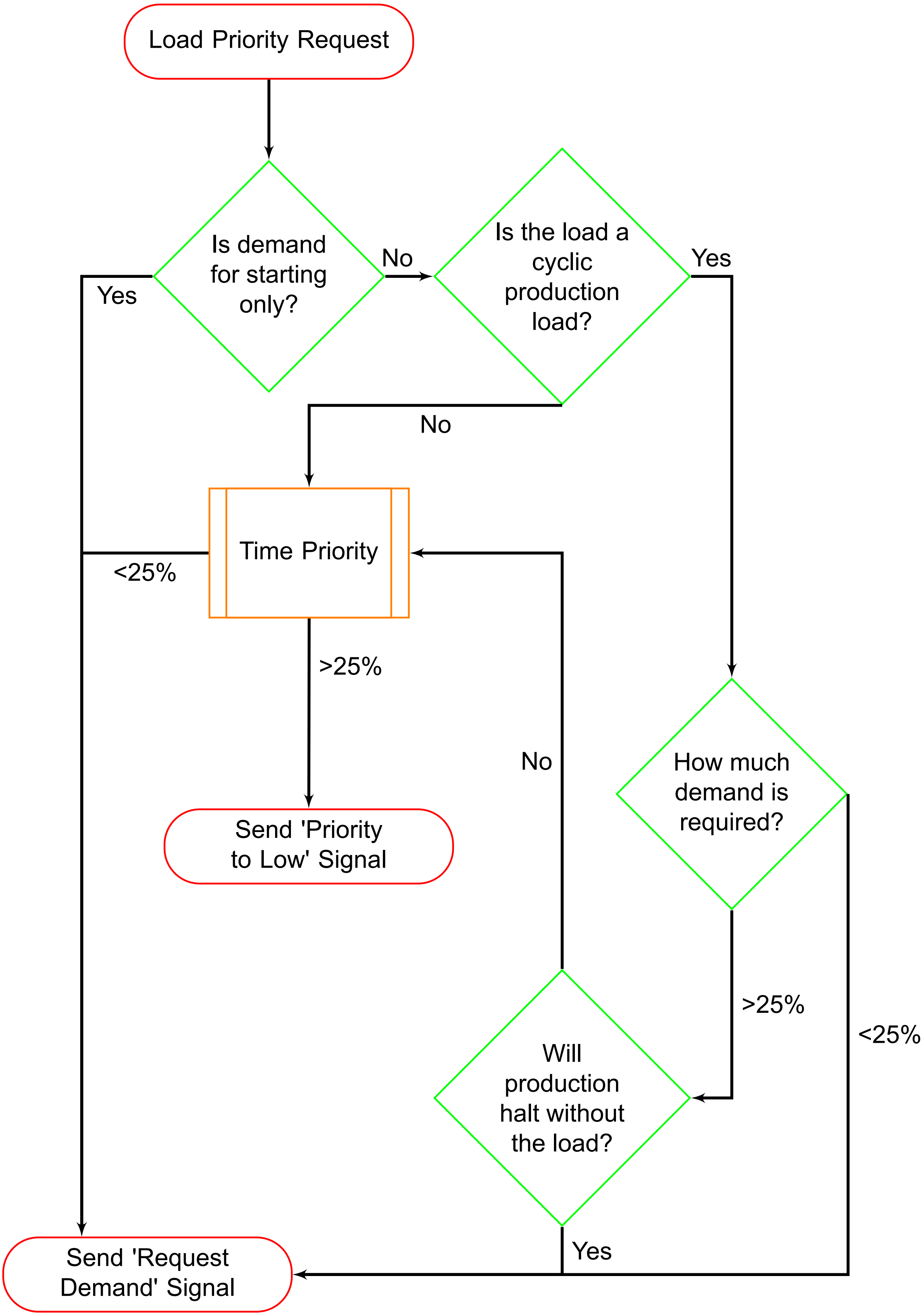



**Figure 6.18: Priority Ranking Module (PRE)**

has not run its allocated time yet then the priority becomes high.

Before the cyclic load is passed to the 'Time Priority' process, there are two additional checks when compared to the free loads. These checks are to prioritises the cyclic loads over free loads and will request demand without a 'Time Priority' check, if the set thresholds are met. The checks are 'Will the production halt without the load?' if the answer to this question is yes, then the 'Time Priority' is skipped and the demand is requested and 'How much demand is required?', this is a simple calculation similar to the demand availability check and is as follows:

$$Required\ Demand\ (\%) = \left(\frac{Required\ Load\ Demand - Available\ Demand}{Required\ Load\ Demand} * 100\right).$$

The time priority is calculated as follows:

IF $\left(\frac{Remaining\ Load\ Operation\ for\ Time\ Period}{Remaining\ Time\ Period} * 100\right) < (25\%)$

THEN

$Request\ Demand.$

ELSE

$Priority\ to\ Low.$

END.

It can be seen from Figure 6.18 that the output signals from the 'Time Priority' check have defined values. These values are thresholds to define the load priority. The load either starts now or not, the values of less than 25% for starting and greater than 75% for waiting are not fixed. They would be defined depending on the system under control, the maximum demand, the size of loads (in the system), the number of loads etc. This is the same for the values for the 'How much demand is required?' check, they would also be defined in the same way, depending on the system characteristics.

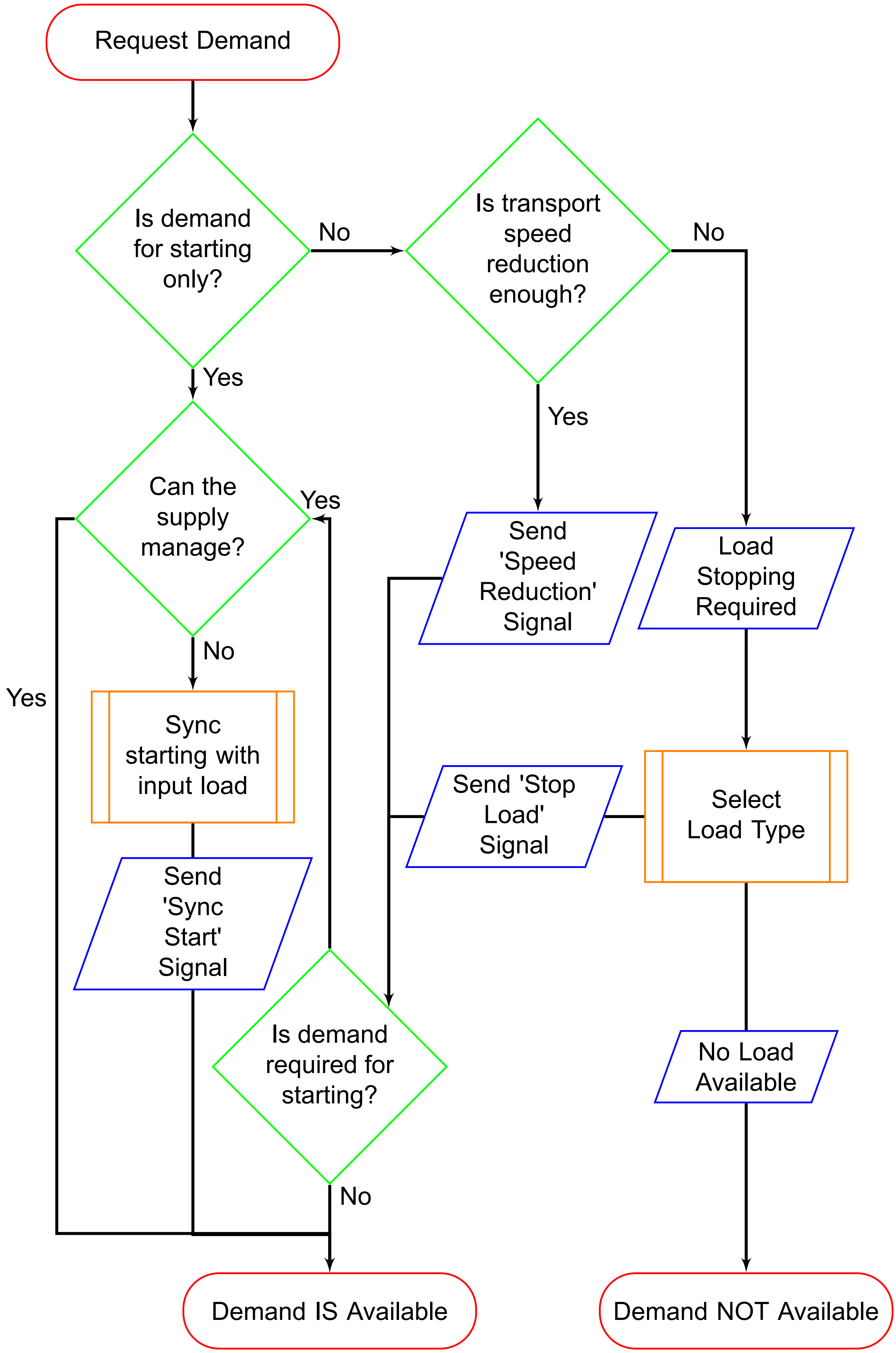



**Figure 6.19: Demand Control Module (DCM)**

#### 6.6.2.3 Demand Control Module (DCM)

The next module in the real-time scheduling system is the DCM This will make the final decision to whether or not the 'Demand Request' request is granted or the load is placed into the queue of waiting loads, see Figure 6.19 for the flow diagram of its operation. There are additional decisions and processes in the DCM, these are; 'Is transport speed reduction enough?' and the 'Select Load Type' sub-routine. The transport speed reduction depends on the underground transport rate i.e. tonnes per hour excluding the pit to surface transport. The current speed depends on the transport system state and the minimum speed is the speed that equals the maximum rate of the pit to surface transport, this minimum speed ensures that the mine production is at maximum for the pit to surface transport bottle neck. The decision is considered as follows:

IF $\left(\frac{Demand\ for\ Current\ Speed - Demand\ for\ Minimum\ Speed}{Requested\ Demand}\right) < 1$

THEN

$Send\ 'Speed\ Reduction'\ Signal.$

ELSE

$Load\ Stopping\ Required.$

END,

and the 'Select Load Type' sub-routine functions as follows:

IF $(Number\ of\ FREE\ loads\ with\ a\ time\ priority\ > 25\%) \geq 1$

THEN

$Select\ FREE\ load\ with\ the\ highest\ priority.$

ELSEIF $(Number\ of\ CYCLIC\ loads\ with\ priority\ < 25\%) \geq 1$

THEN

$Select\ lowest\ 'Time\ Priority'\ CYCLIC\ load\ without\ halting\ production.$

ELSE

$No\ Load\ Available.$

END.

If there is additional demand required for starting, the system considers the type of supply that it is connected to. If there is a grid connection or spare generator capacity on-line, then it is considered that no special measures are needed for the starting of the load. However, if the supply is from a renewable energy source or other limited output supply then there must be consideration for this, Figure 6.20 is a flow diagram of the starting demand supply decision.

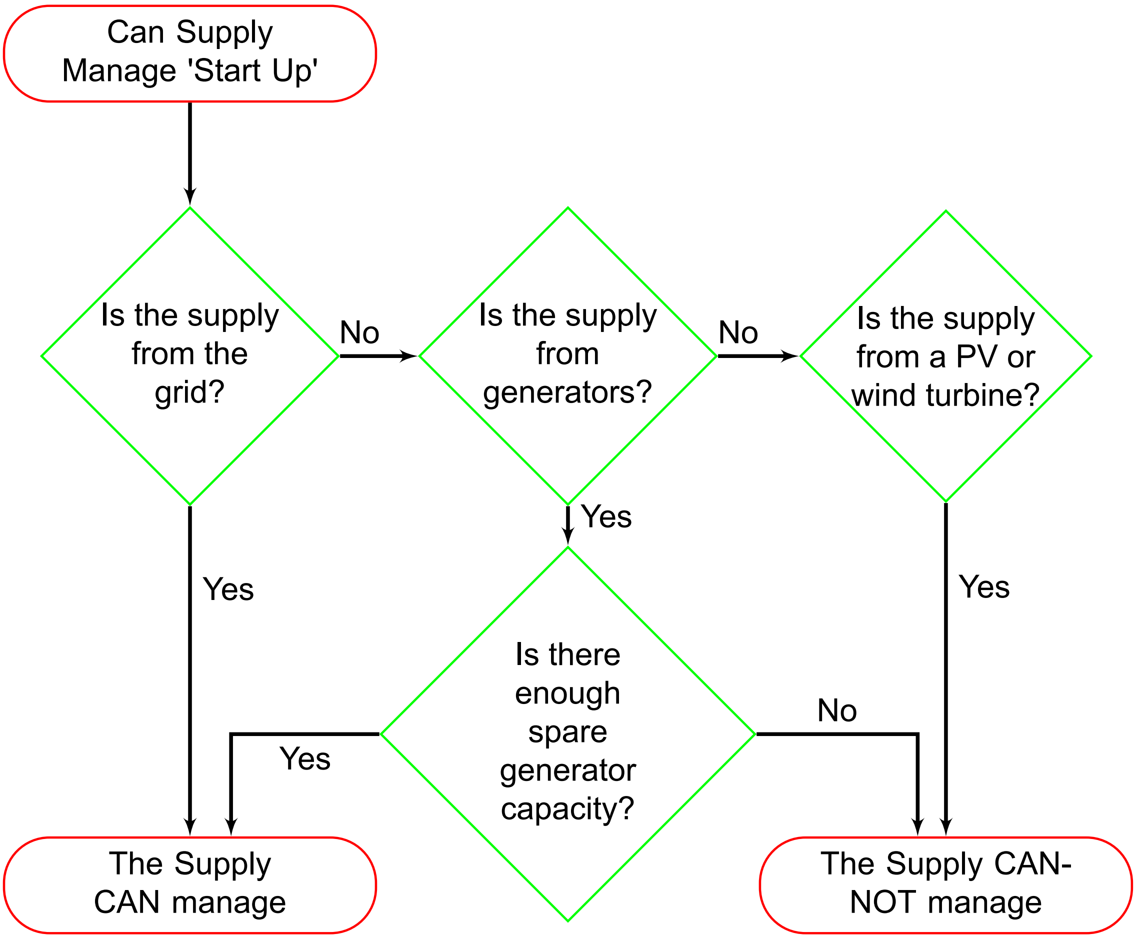



**Figure 6.20: Can Supply Manage? Flow Diagram**

#### 6.6.2.4 The Real-Time Loop

The algorithm outlined above are the operational decisions for when a load start request is made from either the off-line schedule or a manual request for a small load that has not been scheduled. For the optimisation in real-time to work, there

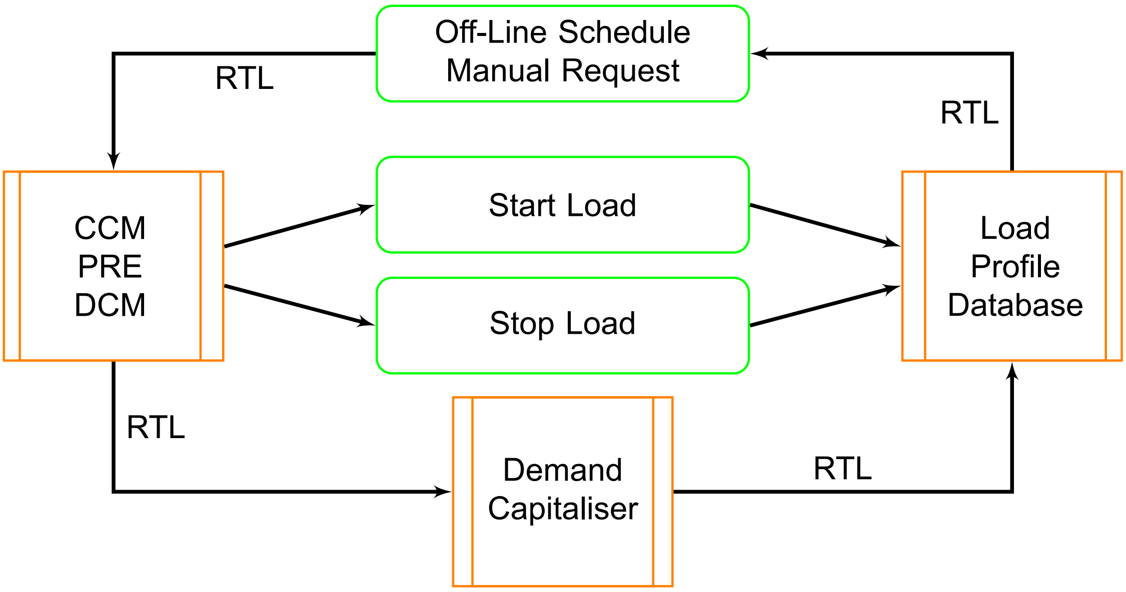



**Figure 6.21: Real-Time Loop (RTL) Flow Diagram**

must be constant real-time monitoring. This is carried out with the real-time loop (RTL), see Figure 6.21. The loop constantly refers to the off-line schedule and runs it like a program, as the starting points of loads occur along the time-line the start request is sent to the CCM. The CCM is in constant reference to the load profile database and updates it as a load is started or stopped. This way the state of the system is always known and can be acted on. There is also an additional module to the RTL is a demand capitaliser, this module constantly monitors for a dip in demand and initiates a request to capitalise on the available demand, for example increasing transport speed or recover production from a breakdown or load stop signal.

#### 6.6.2.5 Pricing Response

The whole system described above has only considered the maximum demand and managing the network of loads to operate the system within a demand threshold. To benefit from RTP, additional control will have to introduced to also manage this. However, it would be considered that this would be minimal, as the system already has all load management in place. How to manage pricing would be to alter the maximum demand threshold depending on price. This would automatically introduce RTP response. To get the most benefit from RTP multiple off-line schedules would be defined and then as the price changed the system would automatically or manually select the defined schedule to fit the pricing. The selection would be made with reference to system analysis results that would be created during the initial demand reduction system set-up. The off-line schedules could be defined for different levels of production; no development; slower transport; no pit to surface transport; etc. These schedules would depend on the mine operators preferences and are not restricted in any way.

## 6.7 Power Quality and Network Stability

During the continuous production of an underground coal mine, the systems are in constant use. The ventilation fans constantly force air through the tunnels to remove the heat and gases and keep the environmental conditions safe and acceptable. Transportation and haulage of the coal is performed as quickly as the machinery will allow, in order to transport the coal to the surface and the coal preparation plant. Followed by transportation to the power stations. The Longwall face machinery cutting coal without concern to the demand requirements of the other mines services, to prevent any delays that cause loss in production. Dewatering pumps run on demand, whenever there is enough water to pump. This simultaneous use of heavy machinery, without prior warning can cause a voltage drop of the underground power supply and thus, an excessive rise in current. This rise in current can cause an overload breaker supplying a piece of machinery, to trip. This type of overload is not due to machinery malfunction, but a coincidental tripping caused by excessive peak power consumption and voltage collapse phenomenon. This unmanaged use of machinery across the mine can put unnecessary strain on a power system and not only cause nuisance tripping, but premature failure of control gear and transformers. Available power is also affected by the unmanaged use of machinery, for instance, to restart a large booster fan or a heavily loaded conveyor that has tripped out may not be possible, as the required start-up power is more than is available and would require the shut-down of multiple systems to increase the available power, just to restart the fan. During the fan restart process, the mine production that has been halted will take additional lost production time to regain full operation.

This is an all too common scenario and affects the mining industry on a daily basis. Although the CCM will make every effort to control the demand, the addition of a load shedding system would help prevent this nuisance tripping and increase network stability, it is proposed to include an intelligent load shedding function, such as the one discussed by Shokooh et al. (2005). Although load shedding is predominantly a utility function for network stability and asset protection, it can be used for power system stability on a small scale within large industrial plants such as mines. This way, if the network starts to become unstable, the lesser priority loads can be dropped to prevent such tripping. When a large amount of power is required, the load profile database can be consulted and only the necessary amount of load will be reduced to fulfil the required power. This will not only help in the overall stability of the mine power system, but will be of significant advantage to mine power systems in remote places where available power is limited. In addition to enabling a stable power system this approach can be used to reduce the load-flow worse-case scenario and extend the life of the existing power system. In some circumstances, this will save the mine £100,000's.

## 6.8 Summary

This chapter has explained in detail the methodology and concept of the demand reduction strategies and network optimization developed by the author. This includes data acquisition, load profiling, load characteristics, and load scheduling off-line and real-time. The authors research into NALM and NILM is also included. The load characterization and LC MATRIX scheduling method development by the author are all fully explained. A MATLAB Simulink simulation model and the further development of LC MAtrix is presented with results. The LC MATRIX and its benefit in the off-line scheduling with test results from using the simulation model is also presented. Finally the concept of RTP introduction and load shedding to increase network stability are introduced. The concepts and ideas in this chapter are to be tested more thoroughly in Chapter 8 using the real-live data collected during the work explained in the next chapter.

# 6. DEVELOPMENT OF ELECTRICAL DEMAND REDUCTION AND RESPONSIVE STRATEGIES

# 7

# SCADA and Data Acquisition

To understand the operation of a system, information on that system is required. This information can be collected in many ways, from basic manual observation and note taking, to a completed SCADA system. A SCADA system can have as little or as much detail as desired for an individual system or components of the system. This chapter introduces SCADA systems and the popular communication protocols used within these systems. There is also the detailed description of the power meter installation and SCADA integration of the power meter systems installed by the author at the UK Coal colliery.

## 7.1 Monitoring and Control

For the control of any system it is first necessary to monitor; the extent of the monitoring and control required will be completely dependent on the type of system and industry involved. Control and monitoring within the mining industry includes control/automation of machinery and the monitoring of the environment/safety monitoring: gas, temperature, dust level monitoring. Hind (1999) gives a concise review of control and monitoring systems in the mining industry. It discusses how the technology has changed over the decades and presents the general trends for the future. Mines have become increasingly larger and more mechanised as technology has progressed; control and monitoring systems have become more widely used. Hind (1999) states that there are generally four elements to a control and monitoring system:

- Sensors to detect information.
- Outstations to condition the sensor signals (e.g. Programmable Logic Controllers).
- A data transmission link to transfer information.
- A central station to provide supervisory control and monitoring.

One of the earliest control and monitoring systems to be introduced into mines was MINOS (or Mine Operating System), which was developed in the UK by the National Coal Board (NCB) following a research project during the 1970's. MINOS provides a complete control and monitoring system for a mine using computer technology. It interfaced to equipment, such as ventilation fans, AFC etc., through sensors and programmable logic controllers (PLC). Amongst its few other aims, the main objectives of the MINOS system were reduction of repair and maintenance costs; improvement in output quality; improvement in monitoring and information; elimination of human error, and increase in safety levels and improvement of working conditions (Burns et al., 1985). Further development of MINOS saw the introduction of the 'Simple Asynchronous Protocol' or SAP, to overcome the problem of incompatibility between various MINOS equipment supplied by different companies.

MINOS is now almost obsolete and has been superseded by SCADA. SCADA systems allow constant monitoring systems and include events and alarm notifications. UK Coal Mining PLC now utilises a modern SCADA network that has completely replaced MINOS and SAP (except for a few legacy devices) (Ford, 2006). Optical fibre with repeaters at least every 2km provide a 100BaseT Ethernet network. This system provides UK Coal's collieries with a large bandwidth SCADA network, thus providing opportunity for a broad range of applications. A significant aspect of this development is that the SCADA network refers to the International Standards Organisation (ISO) Open Systems Interconnection (OSI) seven-layer model, see Figure 7.1. In contrast to MINOS, this ensures interoperability of components from different manufacturers and provides the flexibility needed for future developments.

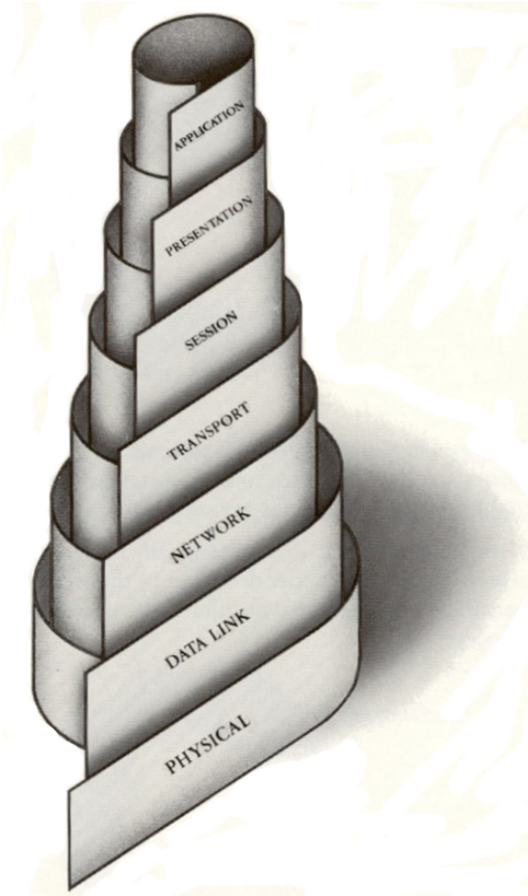



**Figure 7.1: ISO Open System Interconnection (OSI) reference model** (UK Coal)

It is not only the network and sensors that defines a SCADA system, but also the software that provides the window into the mines operation. Figure 7.2 is a screen shot from UK Coal's SCADA system of the methane pumping plant. This is what is know as a human–machine interface or HMI. The HMI presents the process data to a human operator and through which the human operator controls the process. The HMI is usually linked into the SCADA database and software applications, to provide trending, diagnostic data, and management information from a particular sensor or machine. The HMI allows the operator to see a schematic representation of the plant being controlled. For example in Figure 7.2 the valves of the methane pumping plant and their positions can be clearly seen, allowing the operator instant decisions based on the plants operation.

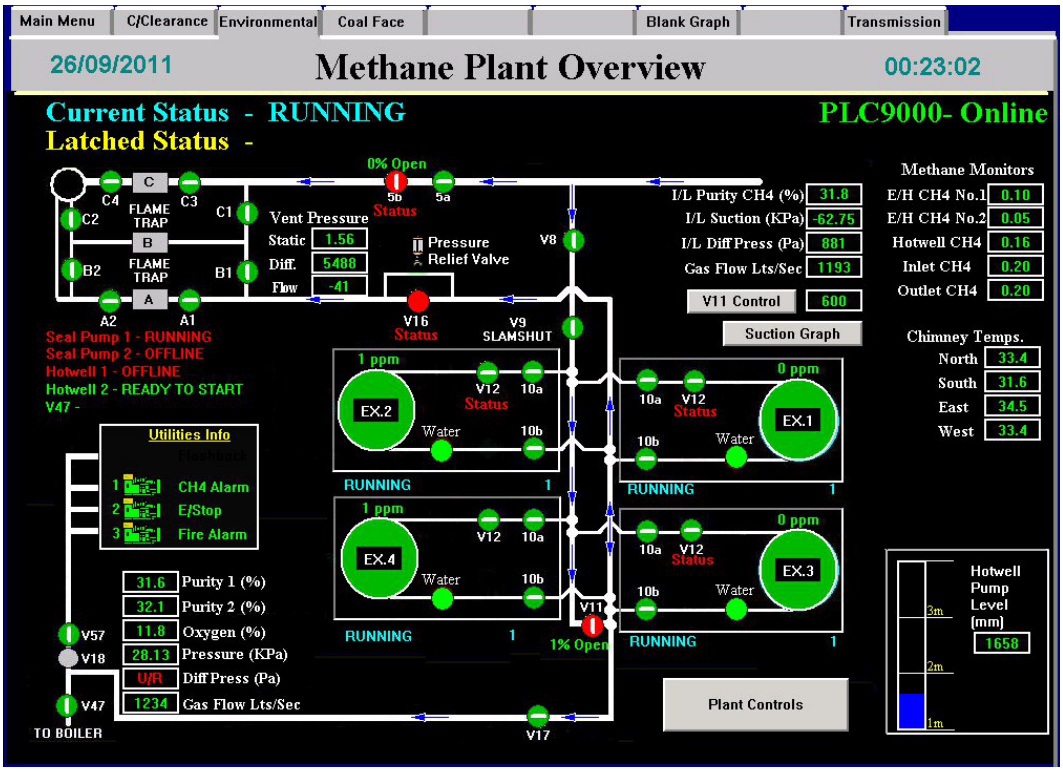



**Figure 7.2: SCADA Methane Plant Overview** (UK Coal)

### 7.1.1 Object Linking and Embedding (OLE) for Process Control (OPC)

There are many different devices and machinery in industrial installations, each having a hardware interface that requires a software driver for connection to the software application that sends the control signals. The control software, such as SCADA applications require drivers for every type of hardware interface, this can cause conflicts or restrictions in access as only one application can access the hardware at one time. Until 1996, when the original object linking and embedding (OLE) for process control (OPC) specification (version 1.0) was released, this was a common problem. However, OPC solved this problem by introducing an open interface over which PC-based software components are able to exchange data. It is based on the Microsoft COM (Component Object Model) and DCOM

(Distributed COM) technologies (Li and Nakagawa, 2002). OPC therefore offers an ideal basis for connecting industrial applications and office programs with the field equipment typically in the automation level, as seen in Figure 7.3. Support for OPC by automation suppliers reached critical mass in late 1998, propelling it from a specification to a de-facto industry standard.

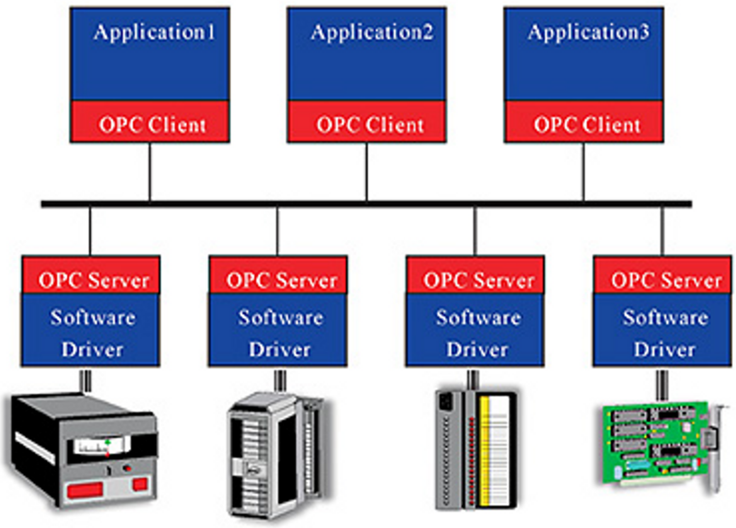



**Figure 7.3: OPC Connectivity** (ADLINK Technology, Inc.)

OPC eliminates the issue of software companies developing different drivers for all the hardware they want their applications to support, or hardware manufactures having to develop drivers for different software application. Instead the hardware manufactures can develop their own drivers with the standard OPC connectivity. The basic operation of OPC is based on tags, this is where the hardware level driver reads or writes to OPC tags from which the software application interacts. For example, to control a pipe valve, the valve reads its open state from the valve tag with a value 0-100 for its position (0 is fully open and 100 is fully closed). The software writes the integer value to the valve tag representing the position selected by the operator, the valve reads the new value and responds accordingly.

### 7.1.2 MODBUS

One particularly popular device level interface protocol is MODBUS. MODBUS is a serial communication protocol originally developed in 1979 by the PLC manufacturer Modicon (now Schneider Electric) and has since been adopted as the de-facto standard for communication for commercial electrics. The MODBUS protocol is based on a master/slave structure with communication between MODBUS devices via messages. A standard MODBUS network consists of one master (although in some instances there can be multiple masters) and up to 247 slaves, each slave has a unique address from 1 to 247. Originally the MODBUS communication ran on RS-232 interface, however most modern implementations use the RS-485 standard as it allows for high data rates over longer distances. Further development of the MODBUS protocol has extended the standard to transmission control protocol/internet protocol (TCP/IP) networks (wired and wireless). However, the MODBUS protocol running over the serial and TCP/IP networks is very different, so integration is only possible with the use of a communication gateway. What has made the MODBUS protocol so popular is that it is flexible and easy to implement, not only intelligent devices like micro-controllers, PLCs, HMIs etc. can communicate using the MODBUS protocol, also many intelligent sensors have MODBUS interfaces with the ability to send their data to host systems.

#### 7.1.2.1 MODBUS Message Structure

The MODBUS message structure is independent of the physical interface used, from plain RS-232 to MODBUS/TCP over Ethernet the messages are the same. The gives MODBUS protocol standard great versatility and a very long life, regardless of the connection or interface type. This allows for hardware structure of an industrial network to be upgraded without major changes to the software that operates over the network. A device can communicate with several MODBUS nodes on a network regardless of the interface type. Different types of interface can also co-exist on the same network without the need for different protocols for the each connection. On serial interfaces like RS-232 or RS-485, MODBUS messages are sent in plain form over the network as the network is dedicated to

MODBUS. When using more versatile networks such as TCP/IP over Ethernet, the MODBUS messages are wrapped in packets that meet the format of this network type. This allows MODBUS to co-exist with other traffic for different types of connections. Although the main MODBUS message structure is peer-to-peer, MODBUS is able to function on both point-to-point and multi-drop networks.

The structure is the same for every MODBUS message, consisting of four basic elements. To make it easy to parse the message, the elements follow the same sequence for every message. The master in the MODBUS network always initiates the conversation. The master sends a message to a slave and depending on the message content the slave will take action and respond to the message. Each slave has a unique address, a valid slave address is in the range of 1-247. The message contains an address in order for the correct slave to respond to the message, all the other slaves will ignore the message. If the message address is 0 (this is known as the broadcast address) all slaves will respond. A slave always responds to a MODBUS message, the response contains the slaves address. This way the master can verify that the MODBUS device has actually responded to the message. Data is stored through the use of coils and registers. Coils store simple binary values while registers store numerical values. Both coil and register values are stored in tables having specific addresses relating to the stored values. This relationship allows for the proper message to be framed under the MODBUS protocol.

## 7.2 The Data Acquisition System Installation

The main part of the agreement made between the University of Exeter and UK Coal was that the author would carry out all the work required to complete the installation of the data acquisition system at the designated UK Coal colliery. This included, the system design; equipment selection and purchasing; software selection and set-up; communication cables and server set-up and overall project management. Some help was provided by UK Coal in the way of labour for installation of the communication cables as these had to be routed internally through the high voltage (HV) switches and proper authorisation was required for this part of the installation. The costs for the data acquisition system equipment were also to be covered by this project.

Located in Yorkshire the UK Coal colliery, is one of the last few remaining operating deep underground coal mines left in the UK. It is one of two owned by UK Coal PLC. In the 1950s it was established that there was up to seven workable coal seams (Coal, 2011) and in April 1965 the colliery began production. There are two main shafts one for the workers and materials and the other as an up-cast shaft for the coal. This up-cast shaft can transfer the coal at a rate of up to 960 tonnes an hour. At the time of writing the colliery is extracting from the Beeston Seam and is currently in development of its fourth district in this seam, and now in the salvage process at the second face and producing at the third district.

Prior to the Beeston Seam, UK Coal was mining in the Silkstone seam, at 650m deep. The seam is estimated to have reserves and resources of 21 million tonnes, although this seam was not as favourable, physically or environmentally as the current Beeston seam. As the mining in the Silkstone seam finished toward the end of 2009, the Beeston Seam, laying 50m deeper and is estimated to have similar reserves and resources, became accessible due to a £55 million investment programme. This investment has expected to extend the life of the mine until 2015. Further reserves are also expected to be accessible in the Silkstone Seam thereafter, which will extend the life of mine to 2019 (Coal, 2011).

The mines electrical power is fed from the national grid with two 33kV overhead transmission lines to the site substation. Before entering the site these are transformed to 11kV and distributed to the main substation via four 11kV underground distribution cables. The main substation contains two bus sections, one providing supply to the second substation and the other distributing the 11kV to the surface equipment and the coal preparation plant. This bus section also provides supply to the medium and low voltages transformers for other surface equipment and buildings. The second substation contains one main bus section that supplies all the underground power. This bus section is supplied from the first bus section of the main substation via five transformers, three 4MVA and two 10MVA transformers one for each of the five shaft feeders.

Once they reach the underground, the shaft feeders then distribute via the underground substations at various voltages, 6.6kV, 1.1kV, 500V and 415V. This outlines the main distribution layout of the colliery. There is also the methane pumping plant supplied from the 6.6kV substation and the methane gas driven generators that supply the main 11kV surface bus section.

### 7.2.1 Power Meter Selection

The next stage in the process was the system design and equipment selection that was installed at the mine. There were four main points to be considered when selecting the equipment; how much data was required, the level of detail, how it would be recorded and cost. The level of detail had two considerations; how much detail did the mine require and how much did the author require for the research. The mines' requirements were relatively low with their main requirement being, energy consumption, power, and power factor. The frequency of the data sampling was not a high priority for the mine. The author's requirements were different and it was their opinion that all power system variables were required at the highest sampling frequency possible. The cost was obviously going to the main restricting factor in the selection of the equipment.

Firstly, the power meters that were to be installed needed to be selected. They had to be cost-effective and practical. The main factor that affected cost was the size of the display on the front panel of the meter. The display was not important as the meters would be interfaced to the SCADA system and the values displayed via a computer screen. Another cost factor was the communications interface, the two choices were serial or Ethernet. Ethernet is easiest to set-up and configure and also offers full duplex communications, but at a high cost. Serial communications are cheaper, but require more complex configuration to set-up, although this is a once only operation once the installation is complete. The serial communication would also require additional equipment for the interfacing to the computer/server. The cost of the Ethernet interfacing would double the cost of the whole installation, even when taking into account the additional equipment required for the serial communication set-up.

With all these considerations in mind, a decision was made to use power meters from Northern Designs, a UK company based in Bradford. The model of meter is the Cube 400, a three phase digital power meter with a three line front panel display for instant reading of the supply state, and a MODBUS 485 serial communications interface that is used to receive the data from the power meter. The power meter can monitor all aspects of the supply, including volts, amps, power (active and reactive), frequency, power factor and harmonic distortion (harmonic distortion is an optional extra that was selected, although not required for demand reduction research, it will prove useful when analysing the power system for power quality, interruptions for machinery and NILM.) As well as the instantaneous values, the meter also has energy counters for import and export of the energy amount from the power system.

### 7.2.2 Switch Selection

**Figure 7.4: 11kV Surface Substation** (Three Phase, Triple Pole Switches)

The surface substations have bus sections that are made up of a row of interlinked incoming and outgoing switches, Figure 7.4 shows the row in the 11kV substation. All these switches are three phase, triple pole. Since their installation, these switches have had analogue meters installed. The meters only register the amount of energy used and require manual reading and logging to gain any actual energy profile. This is impractical and would not provide enough data for the research, therefore the meters needed replacing. Extensive assessment of the distribution layout was carried out by the author, determining where power meters needed to be installed to get the energy profile with the minimum number of installed meters to provide enough detail. The assessment concluded that 24 power meters were required, but unfortunately only 22 meters were installed. Only switches with an existing energy meter could have a power meter installed, as the physical installation costs and power disruption needed to install meters in switches without an existing meter was not feasible or cost affective for the remaining two.

### 7.2.3 Switch Monitoring Details

The installation in the 11kV substation was as planned, but in the 6.6kV there had to be some alterations. Although it was disappointing that the installation was not exactly as initially designed, it was not a critical change and alterations had to be accepted. All the underground feeders were monitored, although the capacitor bank, compressor house and the methane pumping station did not have dedicated metering. With the case of the methane pumping station, this could be calculated using the result of the bus incoming meters minus the shaft feeders.

Initially it was thought that the capacitor bank monitoring was required, however, this data was not required for the project as reactive power, power factor and power factor correction was to be assumed because it was a separate issue and already managed carefully by UK Coal and did not play part of the demand reduction strategies.

The surface compressor house unfortunately could not be separately monitored and was combined with the 4MVA shaft feeders. However, after the installation of the power meters, the surface compressors were upgraded and a new supply was installed from the 415v supply and although this was still combined with other 415v loads, it was easier to determine than it was when it was part of the underground demand. Chapter 8 discusses in detail the data collected from the installation.

Below is a list of the switches where Cube 400 power meters were installed:

**11kV Substation**

- No.3 Y.E.B. Incomer
- No.4 Y.E.B. Incomer
- No.1 Feeder Fan House
- No.1 500kVA Trans 11kV/550V
- No.1 Tower No.1 Feed
- No.2 Tower No.1 Feed
- C.P.P. No.1 Feed
- No.1 Y.E.B. Incomer
- Methane Generators 11kV/415V
- No.2 Y.E.B. Incomer
- C.P.P. No 2 Feed
- No.2 Tower No 2 Feed
- No.1 Tower No 2 Feed
- No.2 500kVA Trans 11kV/550V
- No.2 Feeder Fan House

**6.6kV Substation**

- Incomer No.1 Transformer 4MVA
- Incomer No.2 Transformer 4MVA
- Incomer No.3 Transformer 4MVA
- No.4 Shaft Feeder Cable
- No.4 Incomer 10MVA
- No.5 Incomer 10MVA
- No.5 Shaft Feeder Cable

### 7.2.4 Replacement of the Power Meters

The installation of the meters replaced the existing analogue meters that were already installed. The high voltages and currents meant that the power meters are not connected directly to the supply, but through current transformers (CT) and potential transformers (PT). These step-down the working current and voltage to a much lower output for the monitoring equipment, in this case the power meters. The CT and PT have a fixed ratio depending on the current and voltages present, for example a CT with the ratio of 400:5, means at 400 amps the current flow at the output of the CT would be 5 amps. For a PT the ratio may be 3300:110 volts. The Cube 400's were installed and used the existing CT and PT's that were original installed for the analogue meters. The existing analogue meters were mounted in a cartridge format. These meters were removed and the new power meter were installed in their place (Figures 7.5(a) and 7.5(b)). The cartridges have metal contacts on the base. The meter were wired to these contacts. When the cartridge was inserted to the location on the switch, contact was made and the circuits for the CT and PT's was complete and the power meter was then on-line.

(a) Analogue Energy Meter

(b) Digital Cube 400 Power Meter

**Figure 7.5:** Power Meters

### 7.2.5 Auxiliary Equipment and Communication

The communication cables were independent to the main meter wiring and required separate new wiring to each individual meter. In addition to the 22 power meters, the data acquisition system required additional hardware. The equipment consisted of two servers (one in the substation and one at Penryn Campus), two Moxa MODBUS gateways. The system also required various software packages including, OPC Top Server, OPC Logger, SQL Server, SSH server and tunnel software (for secure connect between servers at the mine and the University servers). The equipment layout, software installation, and the overall system design is shown in Figure 7.6.

The meters' MODBUS interfaces were originally going to be all connected together in one daisy chain; this would prove to be unsatisfactory as it would limit

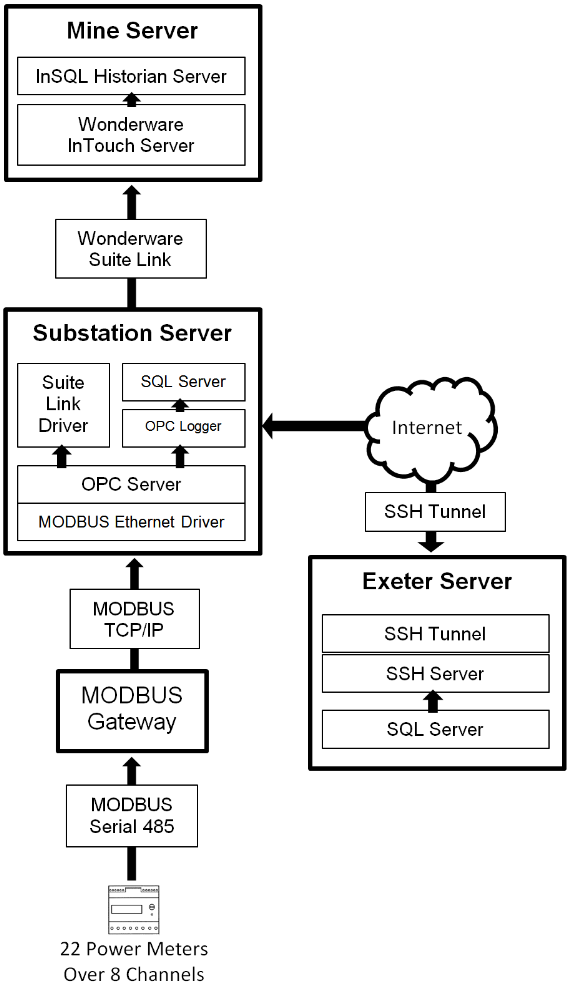



**Figure 7.6: Data Acquisition System** (Equipment and Software Integration)

the amount of data that could be received from the meters. The MODBUS interface is a RS-485 Serial 2 wire half-duplex and therefore only one meter could communicate to the computer at a time. Only either a request for data or the corresponding reply can exist on the interface at one time. This meant that 22 meters interrogating one at a time would take up to 5 seconds before the first meter could log data again. This was not acceptable as it was considered that the resolution was much too low and a better approach needed to be taken.

The meters were separated into eight channels, three meters per channel and two channels with only two meters. This allows eight meters to be interrogated at once and all 22 within 1 second. Figure 7.7 shows a diagram of the final installation of

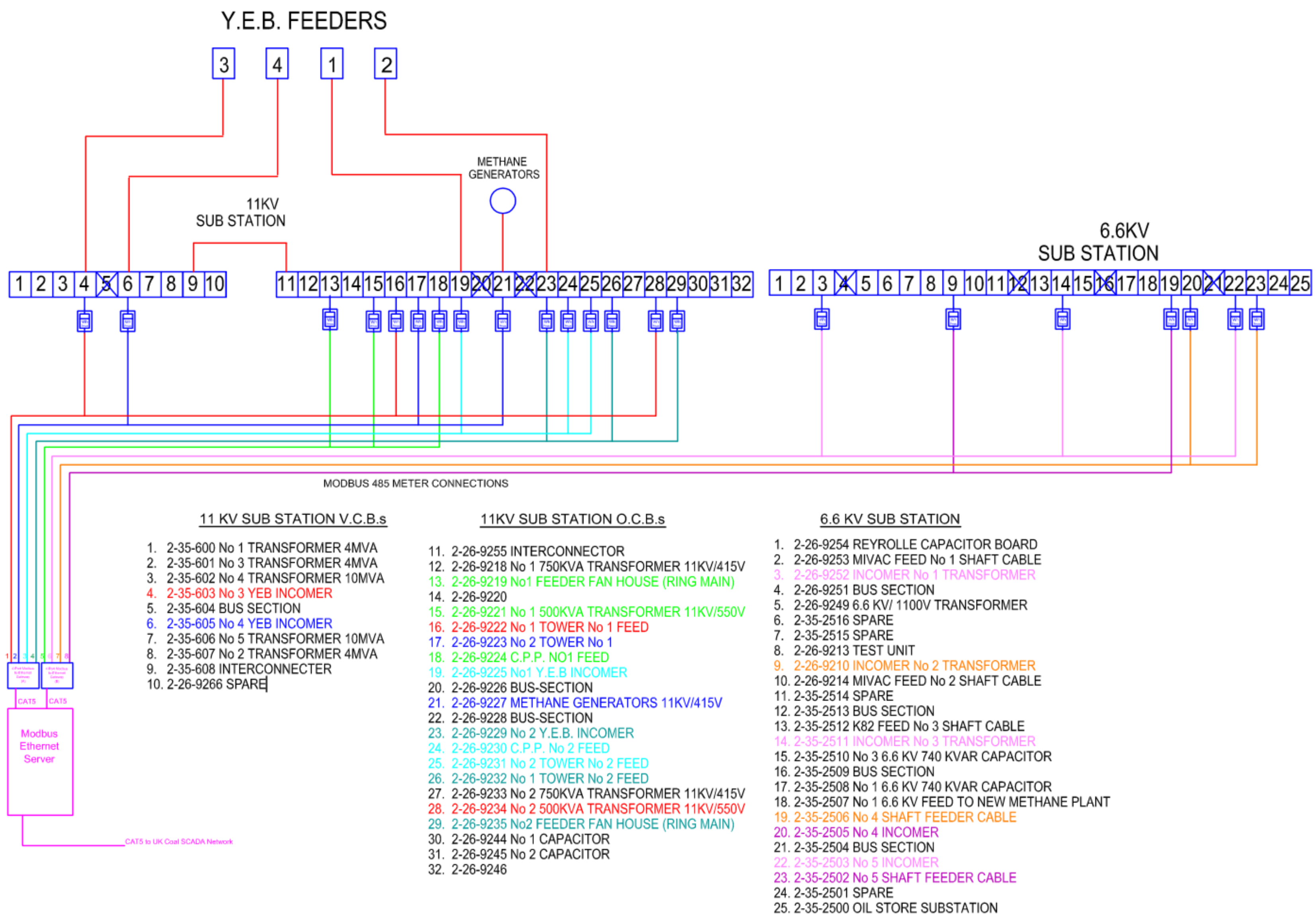



**Figure 7.7: MODBUS Wiring Diagram** (Individual Channel Wiring)

the communication wiring and the individual channels. The MODBUS gateways needed for the communication set-up are; two MOXA MB3480 Serial Gateways with four serial ports each, they connect to a substation server via Ethernet, see Figure 7.8.


**Figure 7.8: Moxa MODBUS Gateways** (The 11kV Substation Server)

Along with the meters and the MODBUS gateways, two Dell Power Edge servers were purchased, one of the servers is located in the substation; this server stored all the output data from the meters. Hardware is not the only requirement for a power monitoring system; software was required to integrate all the hardware together into a complete system. To select the correct software, firstly its requirements had to be assessed; one particular requirement for the system was that the existing SCADA system must be able to connect to the power meters and retrieve the mine power consumption data, without affecting the ability to log the data to the server.

The SCADA system at the UK Coal colliery is Wonderware InTouch, a very sophisticated system that already has facilities to talk to MODBUS equipment via its DAServer. It was considered that the DAServer could be used to connect to the power meters. However, the DAServer can only work with full duplex communications so it could not be used. Wonderware does not support half-duplex MODBUS equipment, but recommends a third part solution developed by Software Toolbox.

The solution provided by Software Toolbox is their OPC DA Server product 'TOP Server' (Figure 7.9). The software acts as a layer between the hardware and the OPC client software, in this case UK Coal's Wonderware InTouch or an OPC data logger for the author's purposes.

In addition to the MODBUS suite, necessary for the installation at the UK Coal colliery, the OPC TOP Server has many drivers for different hardware and protocols providing the standard OPC master availability. This has the added flexibility of being able to integrate different types of hardware into a standardised OPC SCADA system. Remote OPC connections over a network can be very difficult to set-up due to security within the DCOM setup of the Microsoft Windows operating system. For this reason Wonderware InTouch uses its own protocol SuiteLink, TOP Server can provide the data over SuiteLink and is certified by Wonderware.

Serial communication ports have the limitation that they only allow a single connection at any one time. The TOP Server overcomes this limitation, by connecting to the serial port as the single connection and then accepting multiple connections. This gives multiple clients access to the same hardware and data. In reality, the MOXA gateways have already removed the single connection limitation, as the gateway handles multiple request/connections. This is because the connection to the gateway is via Ethernet (TCP/IP) which is a full duplex, high bandwidth protocol. Although having multiple masters is possible, this would reduce the frequency each master would receive data and for that reason, the TOP Server has been used to act as the only master on the system. When a client connects to the TOP Server and requests a value, the TOP Server requests the value from the meter and returns it to the client. If another client then connects to the TOP Server and requests the same value, the TOP Server does not send another request, but returns the most up to date value and continues to update this value with one request from the slave. As there is more than one meter per channel and that only one request can exist on that channel at a time, the TOP Server rotates through each device on a channel in a 'round robin' approach. The data request order of the devices is the same order as the devices were added to the TOP Server configuration. The TOP Server can have a maximum of 128 channels and can communicate with each of them simultaneously.

As previously mentioned, the InTouch SCADA at UK Coal is connecting to the TOP

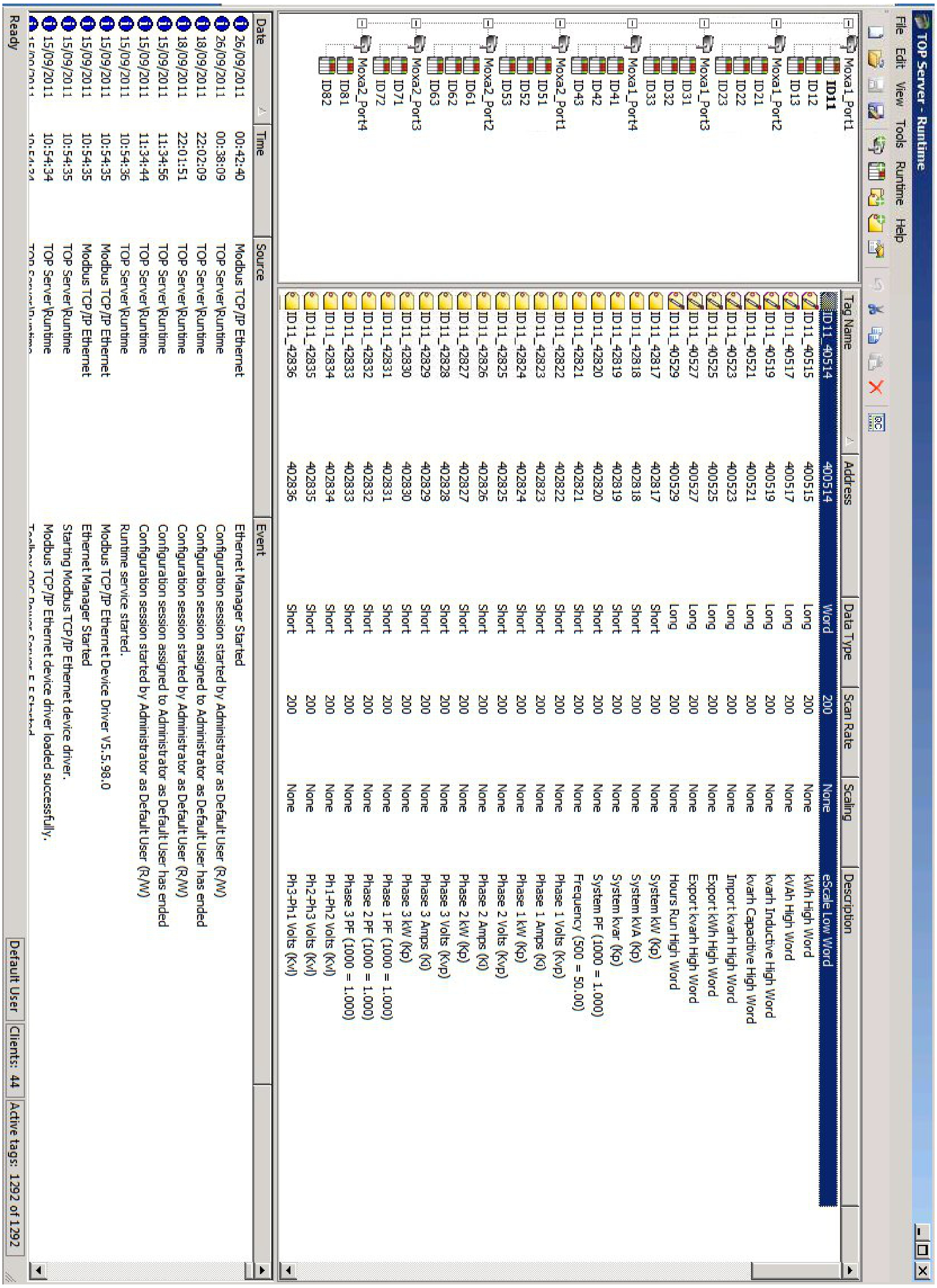



Figure 7.9: Software Toolbox - TOP Server

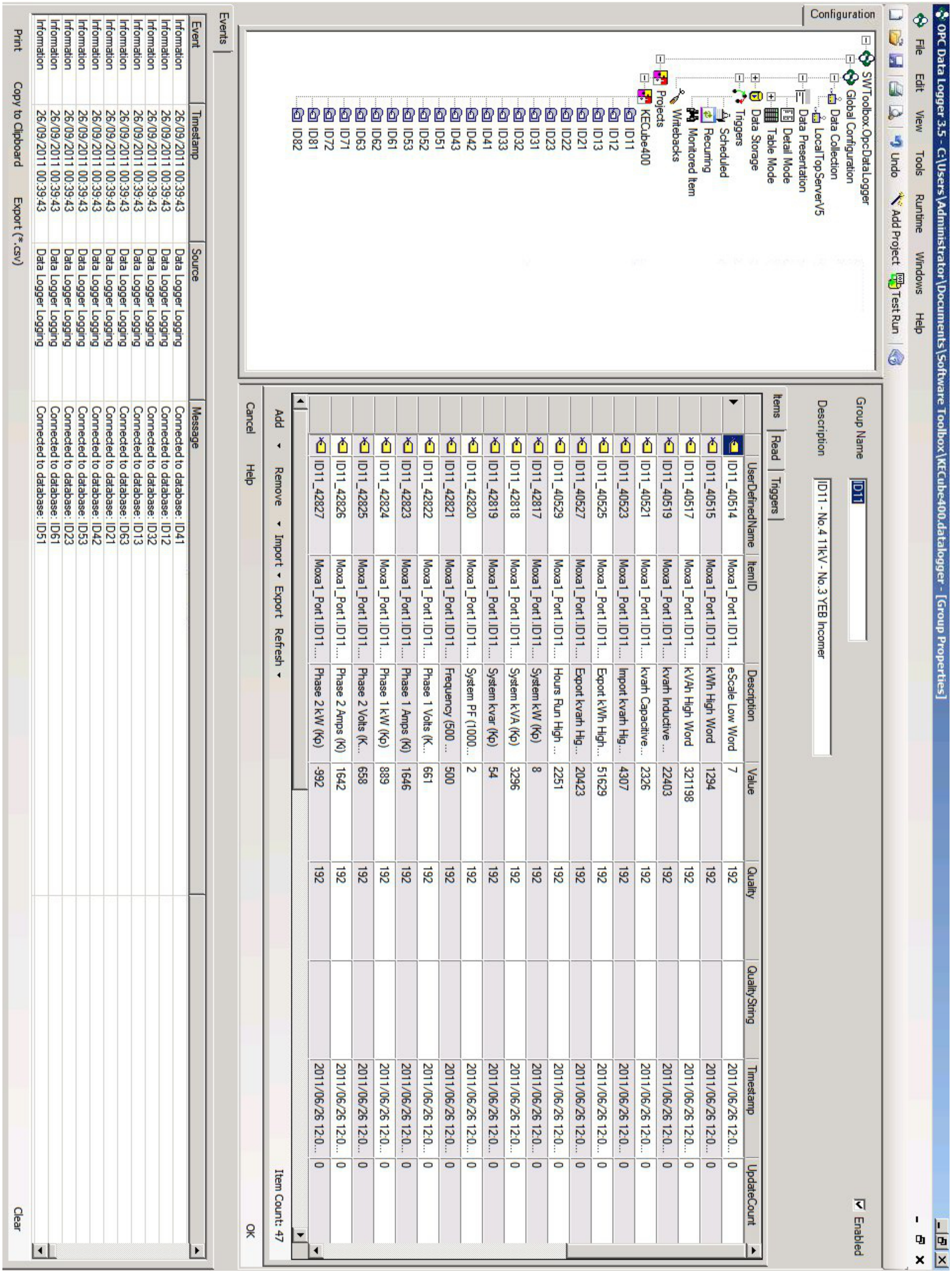



Figure 7.10: Software Toolbox - OPC DataLogger

Server via SuiteLink over the colliery's fibre optic TCP/IP network. As the author's requirements were to log at a higher frequency and detail than the mine, additional OPC logging software was required. The Software Toolbox product, 'OPC Data Logger' (Figure 7.10) fulfilled this requirement. Due to the educational nature of the project, Software Toolbox kindly provided a free licence for the duration of the project. The data logger logged to a Microsoft SQL Server database. The OPC Data Logger will connect to any OPC server, but works exceptionally well with the TOP Server and provided exactly what was required.

Each power meter logged its data to a separate database table and was set-up in the Data Logger as a separate group. This way each meter could have a different data request rate. Different request rates were useful for logging at lower intervals for the power meters that monitor more of a constant load, for example the ventilation fan or methane generator plant. The data was retrieved from the database over a secure connection between the substation server and the second server situated in the lab at University of Exeter, Penryn.

A Secure Shell (SSH) secure tunnel was set-up from within the UK Coal network to the lab server. Local ports on the substation server are forwarded though the secure tunnel to local ports of the lab server. This allowed secure database replication. In addition to the power meter data, as shown in the SCADA Power Usage screen-shot in Figure 7.11, the author also had access to the the colliery SCADA data; which is continually logged by UK Coal. SQL scripts (see Appendix 1) were written to allow events and time period snapshots to be queried from the substation server and stored for secure transfer to the lab server for analysis together with the power meter data.

### 7.2.6 System Fault Finding

Once the power meters were installed and the software was set-up, the data acquisition system was complete. Unfortunately, from the outset not all the meters were reading the correct values. Some of the CT's of the existing installation were not installed correctly to work with a digital power meter; with the original moving coil meters the polarity was not so critical. The CT configuration on the 3 phase

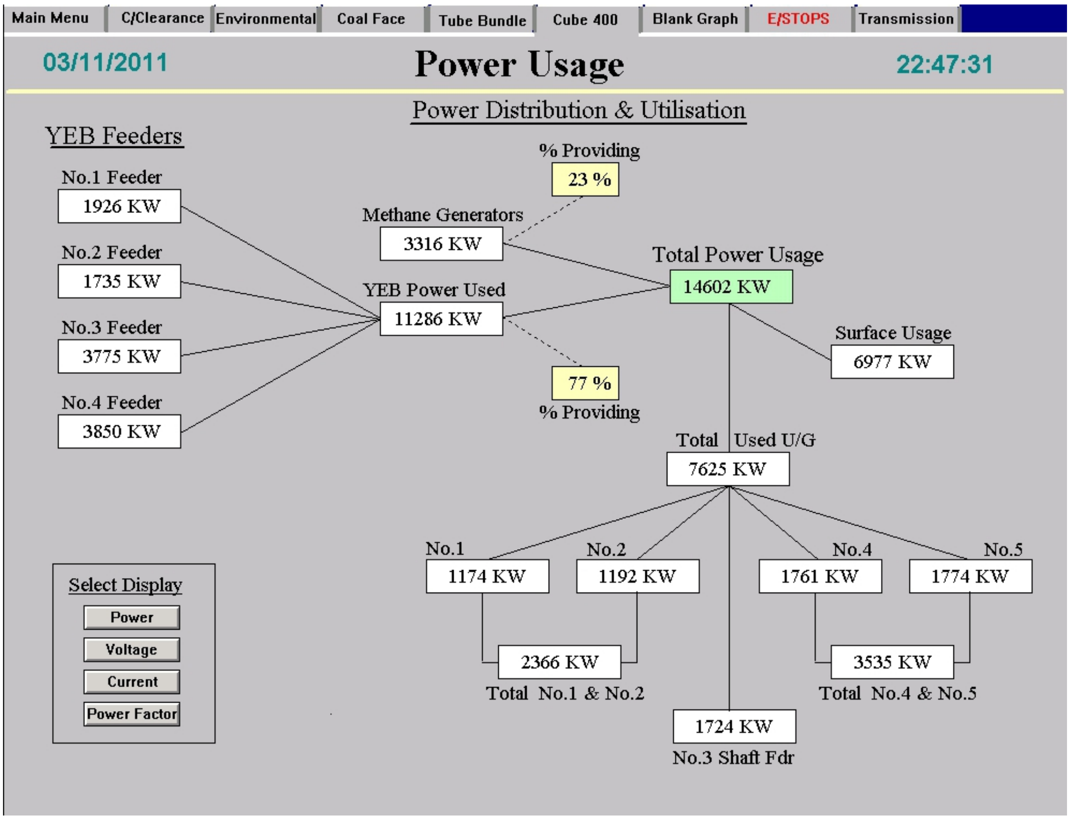



**Figure 7.11: Mine Power Consumption** (The UK Coal colliery SCADA System)

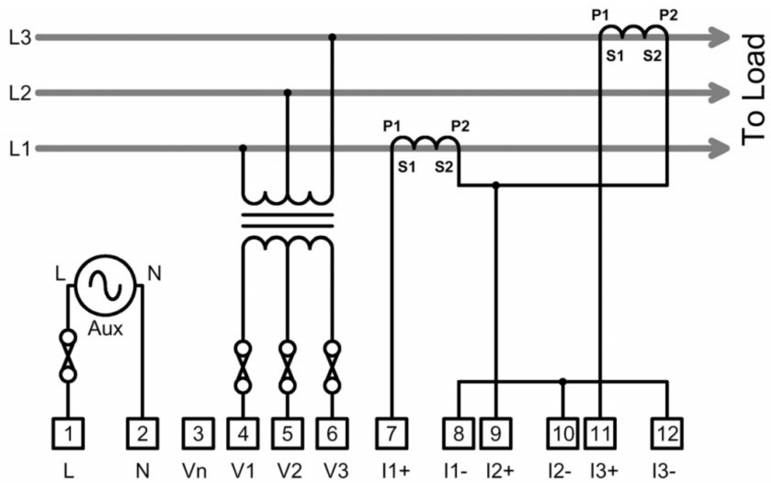



**Figure 7.12: Cube 400 Wiring Diagram** (HV 2 CT Configuration)

switches is a two CT set-up. One CT on each line one and line three. These are connected to the meter with the negatives connected through the line two CT terminals of the meter, see Figure 7.12 for the circuit diagram for meter connection. The original moving coil meters would still function correctly, even if the polarity or direction of one of the CT's is not correct, as they only monitor an actual current and not its direction. However, this is not the case with the digital power meters. The polarity and direction are critical as must be correct for the internal firmware to calculate the exact values.

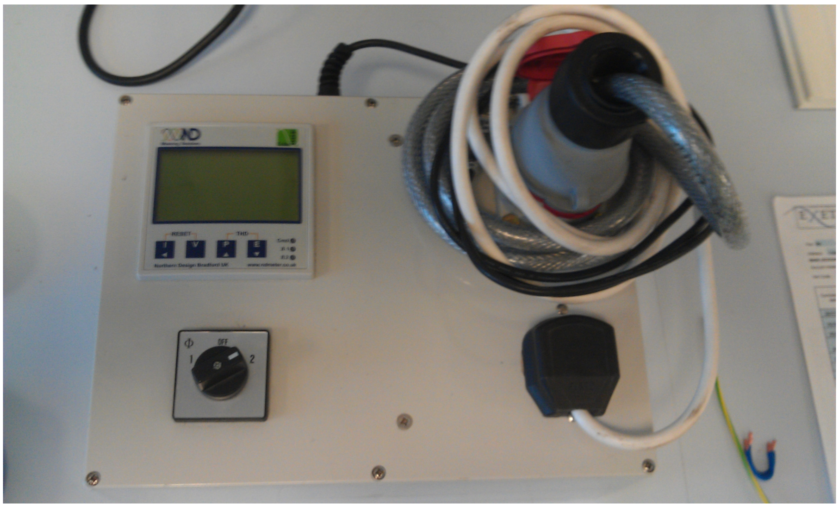


Figure 7.13: Test Meter: Front Panel

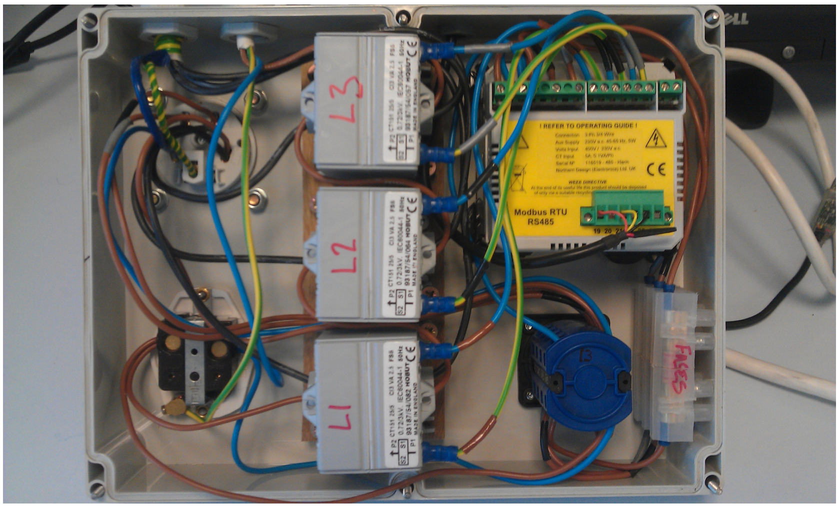



Figure 7.14: Test Meter: Internal Wiring

During the initial system design and configuration a meter was built into a enclosure with CT's connected to selectable single and three phase sockets for portable power metering of single or three phase loads. The front panel and interior wiring of the test meter can be seen in Figure 7.13 and 7.14. The test meter was also used to set-up and preconfigure the server software and test the system in the lab before the final installation at the UK Coal colliery, firstly to save time, but also to eliminate any issues that may cause difficulties on-site.

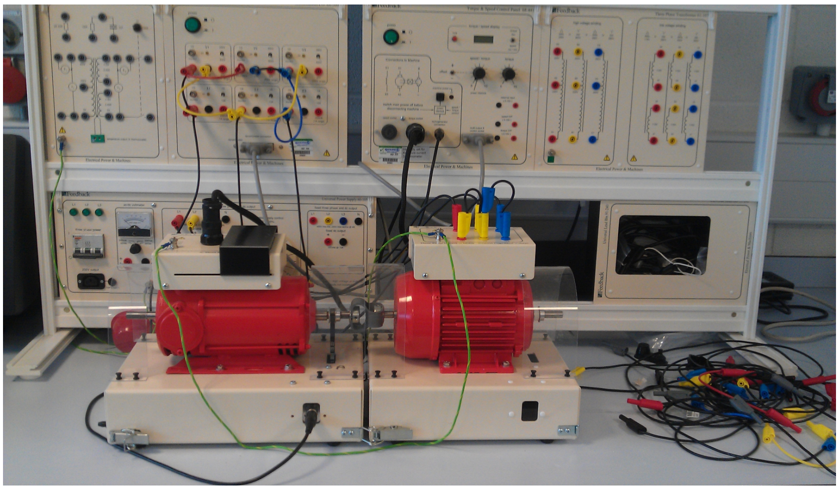


**Figure 7.15: Feedback Test Bench** (Two CT Configuration)

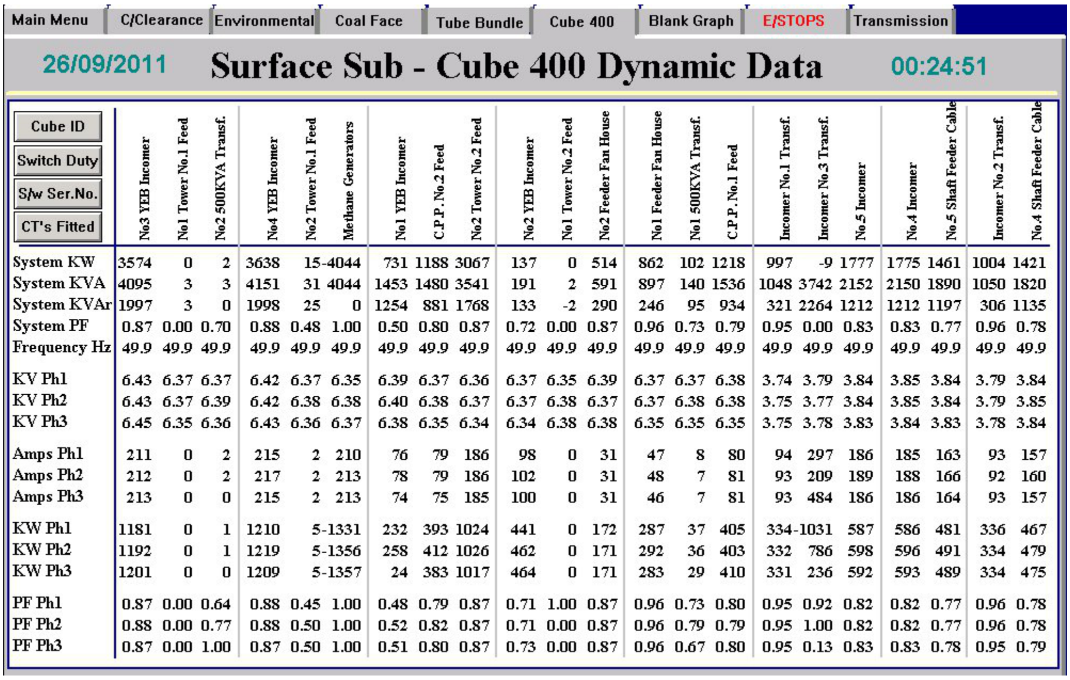
Before any further work was carried out on the meter installation, the test meter was rewired into a two CT configuration and connected to the Feedback test bench (Figure 7.15) and using the test induction motor, different wiring configurations were tested in order to replicate the incorrect meter readings of mine substation power meter installation. As a result of these tests, it was possible not only to conclude that the CT polarity previously mentioned was not a problem and would not need rewiring, but also to compile a procedure to follow to ensure each power meter was wired correctly and producing reliable data. The procedure for the meter wiring is outlined below:

**Cube 400 Wiring Procedure**

(See Figure 7.12 for Terminal Identification)

- Connect all wiring as the diagram.
- Check the current, if unbalanced then swap reverse the CT S1 and S2 on each CT one at a time and both until the current is balanced.

- When the current is balance, check the PF and swap the order of the V1,V2 & V3 until the PF is balanced.
- Now check the PF reading, rotate the position of V1, V2 & V3 until the PF is correct
  i.e. not leading or lagging very low.
- Now check the kW's for the correct polarity
  i.e. positive for a load and negative for generation.
- Correct the polarity of the kW's by swapping all the CT wires
  i.e. i+'s to i-'s and vice versa.
- The power meter should now read correctly.




| | No3 YEB Incomer | No1 Tower No.1 Feed | No2 500KVA Transf. | No4 YEB Incomer | No2 Tower No.1 Feed | Methane Generators | No1 YEB Incomer | C.P.P. No.2 Feed | No2 Tower No.2 Feed | No2 YEB Incomer | No1 Tower No.2 Feed |
|---|---|---|---|---|---|---|---|---|---|---|---|
| System KW | 3574 | 0 | 2 | 3638 | 15 | -4044 | 731 | 1188 | 3067 | 137 | 0 |
| System KVA | 4095 | 3 | 3 | 4151 | 31 | 4044 | 1453 | 1480 | 3541 | 191 | 2 |
| System KVAr | 1997 | 3 | 0 | 1998 | 25 | 0 | 1254 | 881 | 1768 | 133 | -2 |
| System PF | 0.87 | 0.00 | 0.70 | 0.88 | 0.48 | 1.00 | 0.50 | 0.80 | 0.87 | 0.72 | 0.00 |
| Frequency Hz | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 |
| KV Ph1 | 6.43 | 6.37 | 6.37 | 6.42 | 6.37 | 6.35 | 6.39 | 6.37 | 6.36 | 6.37 | 6.35 |
| KV Ph2 | 6.43 | 6.37 | 6.39 | 6.42 | 6.38 | 6.38 | 6.40 | 6.38 | 6.37 | 6.37 | 6.38 |
| KV Ph3 | 6.45 | 6.35 | 6.36 | 6.43 | 6.36 | 6.37 | 6.38 | 6.35 | 6.34 | 6.34 | 6.38 |
| Amps Ph1 | 211 | 0 | 2 | 215 | 2 | 210 | 76 | 79 | 186 | 98 | 0 |
| Amps Ph2 | 212 | 0 | 2 | 217 | 2 | 213 | 78 | 79 | 186 | 102 | 0 |
| Amps Ph3 | 213 | 0 | 0 | 215 | 2 | 213 | 74 | 75 | 185 | 100 | 0 |
| KW Ph1 | 1181 | 0 | 1 | 1210 | 5 | -1331 | 232 | 393 | 1024 | 441 | 0 |
| KW Ph2 | 1192 | 0 | 1 | 1219 | 5 | -1356 | 258 | 412 | 1026 | 462 | 0 |
| KW Ph3 | 1201 | 0 | 0 | 1209 | 5 | -1357 | 24 | 383 | 1017 | 464 | 0 |
| PF Ph1 | 0.87 | 0.00 | 0.64 | 0.88 | 0.45 | 1.00 | 0.48 | 0.79 | 0.87 | 0.71 | 1.00 |
| PF Ph2 | 0.88 | 0.00 | 0.77 | 0.88 | 0.50 | 1.00 | 0.52 | 0.82 | 0.87 | 0.71 | 0.00 |
| PF Ph3 | 0.87 | 0.00 | 1.00 | 0.87 | 0.50 | 1.00 | 0.51 | 0.80 | 0.87 | 0.73 | 0.00 |

| | No2 Feeder Fan House | No1 Feeder Fan House | No1 500KVA Transf. | C.P.P. No.1 Feed | Incomer No.1 Transf. | Incomer No.3 Transf. | No.5 Incomer | No.4 Incomer | No.5 Shaft Feeder Cable | Incomer No.2 Transf. | No.4 Shaft Feeder Cable |
|---|---|---|---|---|---|---|---|---|---|---|---|
| System KW | 514 | 862 | 102 | 1218 | 997 | -9 | 1777 | 1775 | 1461 | 1004 | 1421 |
| System KVA | 591 | 897 | 140 | 1536 | 1048 | 3742 | 2152 | 2150 | 1890 | 1050 | 1820 |
| System KVAr | 290 | 246 | 95 | 934 | 321 | 2264 | 1212 | 1212 | 1197 | 306 | 1135 |
| System PF | 0.87 | 0.96 | 0.73 | 0.79 | 0.95 | 0.00 | 0.83 | 0.83 | 0.77 | 0.96 | 0.78 |
| Frequency Hz | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 | 49.9 |
| KV Ph1 | 6.39 | 6.37 | 6.37 | 6.38 | 3.74 | 3.79 | 3.84 | 3.85 | 3.84 | 3.79 | 3.84 |
| KV Ph2 | 6.37 | 6.37 | 6.38 | 6.38 | 3.75 | 3.77 | 3.84 | 3.85 | 3.84 | 3.79 | 3.85 |
| KV Ph3 | 6.38 | 6.35 | 6.35 | 6.35 | 3.75 | 3.78 | 3.83 | 3.84 | 3.83 | 3.78 | 3.84 |
| Amps Ph1 | 31 | 47 | 8 | 80 | 94 | 297 | 186 | 185 | 163 | 93 | 157 |
| Amps Ph2 | 31 | 48 | 7 | 81 | 93 | 209 | 189 | 188 | 166 | 92 | 160 |
| Amps Ph3 | 31 | 46 | 7 | 81 | 93 | 484 | 186 | 186 | 164 | 93 | 157 |
| KW Ph1 | 172 | 287 | 37 | 405 | 334 | -1031 | 587 | 586 | 481 | 336 | 467 |
| KW Ph2 | 171 | 292 | 36 | 403 | 332 | 786 | 598 | 596 | 491 | 334 | 479 |
| KW Ph3 | 171 | 283 | 29 | 410 | 331 | 236 | 592 | 593 | 489 | 334 | 475 |
| PF Ph1 | 0.87 | 0.96 | 0.73 | 0.80 | 0.95 | 0.92 | 0.82 | 0.82 | 0.77 | 0.96 | 0.78 |
| PF Ph2 | 0.87 | 0.96 | 0.79 | 0.79 | 0.95 | 1.00 | 0.82 | 0.82 | 0.77 | 0.96 | 0.78 |
| PF Ph3 | 0.87 | 0.96 | 0.67 | 0.80 | 0.95 | 0.13 | 0.83 | 0.83 | 0.78 | 0.95 | 0.79 |

**Figure 7.16: SCADA Data Power Meter Check**

A SCADA screen was set-up to check the data, this screen can be seen in Figure 7.16. The data was checked along side the meter front panel, the OPC Top Server and the SCADA screen. Every meter was checked and the procedure followed to conclude that all the meters were monitoring their switch current and voltage properly. All meters except Incomer No.3 Transformer 4MVA were working correctly, the wiring procedure was followed. However, the meter never gave the correct readings and it was concluded that one of the CT was faulty. The meter only supplied the booster fans and was a constant load. Due to this it was considered that the meter could be wired with one CT into all inputs to give a reading. This was not successful as the meter did not respond as expected. Instead, the Incomer No.3 Transformer 4MVA data is calculated as follows:

Incomer No.3 Transformer 4MVA $=$
(No.3 Y.E.B. Incomer + No.4 Y.E.B. Incomer) $-$
(Incomer No.1 + Incomer No.2 + Incomer No.4 + Incomer No.5).

### 7.2.7 System Accuracy Check

**Comparison of Power Usage at Colliery**

**Colliery - Electricity Usage** *(Taken from Cube 400 monitoring on InSQL)*

| | ID 1_1 - N0.3 YEB Incomer | | | ID 2_1 - No.4 YEB Incomer | | | ID 3_1 - No.1 YEB Incomer | | | ID 4_1 - No.2 YEB Incomer | | | YEB Incomers |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | *Cube400_MWh_11* | | | *Cube400_MWh_21* | | | *Cube400_MWh_31* | | | *Cube400_MWh_41* | | | |
| Time Period | Start Val | End Val | Used (MW) | Start Val | End Val | Used (MW) | Start Val | End Val | Used (MW) | Start Val | End Val | Used (MW) | Total Supplied (MW) |
| 1/5/12 to 31/5/12 | 13056 | 14973 | 1917 | 13268 | 15216 | 1948 | 5285 | 6148 | 863 | 5401 | 6266 | 865 | **5593** |
| 1/6/12 to 30/6/12 | 14973 | 16697 | 1724 | 15216 | 16964 | 1748 | 6148 | 6719 | 571 | 6266 | 6831 | 565 | **4608** |

**Colliery - Electricity Usage** *(Taken from Y.E.B. Meter Readings)*

| | ID 1_1 - N0.3 YEB Incomer | | | ID 2_1 - No.4 YEB Incomer | | | ID 3_1 - No.1 YEB Incomer | | | ID 4_1 - No.2 YEB Incomer | | | YEB Incomers |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | *Cube400_MWh_11* | | | *Cube400_MWh_21* | | | *Cube400_MWh_31* | | | *Cube400_MWh_41* | | | |
| Time Period | Start Val | End Val | Used (MW) | Start Val | End Val | Used (MW) | Start Val | End Val | Used (MW) | Start Val | End Val | Used (MW) | Total Supplied (MW) |
| 1/5/12 to 31/5/12 | 74960500 | 76891100 | 1930600 | 76329300 | 78288000 | 1958700 | 80895500 | 81767300 | 871800 | 21849600 | 22723700 | 874100 | **5635.2** |
| 1/6/12 to 30/6/12 | 76891100 | 78645100 | 1754000 | 78288000 | 80064900 | 1776900 | 81767300 | 82331900 | 564600 | 22723700 | 23281700 | 558000 | **4653.5** |

**Figure 7.17: Power Meter Accuracy Check**

Additional checks were also carried out against the energy bills and the energy registers of the incoming meters (Figure 7.17). Using the total highlighted in Figure 7.17 it was proven that the power meter installation was accurate within 1%, the calculations were as follows:

$$5593/5635.2 = 0.993 \text{ and } 4608/4653.5 = 0.99.$$

## 7.3 Summary

The chapter has summarised the data collection system that has been installed at the UK Coal colliery. Details of hardware, software and SCADA integration are all presented. The problems with the CT and PT's encountered when installing the new digital meters are described. How these problems were overcome, by the building of a test meter. This test meter was then used with a test load in the lab to develop a power meter wiring procedure. As well as the SCADA and simultaneous data logging problems that different protocols caused and how they were overcome. These problems were overcome with the use of a data page on the SCADA system. This page was then used to check the data with the front panel of each meter. if the meter readings were incorrect, the wiring procedure was followed. This produced a robust power meter installation that was accurate to within 1%. The next chapter presents the data collected. The data is also applied to the demand reduction systems presented in Chapter 6.

# 8

# Project Data

The following chapter presents the data collected during the project, firstly, a detailed profile of the UK Coal colliery's operation, production output and its energy use. Secondly, the concept of the DRP is applied to the data. The load profiles for the different services are defined. The disaggregation of power data is also presented with test data and results. The LC MATRIX is used to create an optimised scheduling from load profile data and compared with the original profiles.

## 8.1 The Colliery Profile

By the end of 2012, the data acquisition had been logging for over one year. However, as explained in Chapter 7, there was some initial teething problems with the accuracy of the data. After correcting the accuracy issues and stabilising the communications the system had logged nearly eight and a half months (252 days) of continuous data. The dates of the reliable data are from the 26 February to the 5 November 2012. The colliery coal production data is not available on the SCADA system, as there is no coal flow monitoring equipment installed at the mine. The coal production data was provided separately by the colliery management, the dates for the production data are from the 26 February to the 2 November 2012. As both the production data and the energy data are required to provide analysis, the date range of the data used throughout this chapter is from the 26 February to the 2 November 2012.

### 8.1.1 Production and Cost Overview

The production data and the electrical energy consumption of the mine is presented first. This shows the overall energy cost for the mine production and gives a good understanding of how energy intensive a coal mine is. The average saleable coal production for the date period is 5340 tonnes per day, with an average energy consumption of 40 kWh per tonne. These values are only the active power consumption, there is also reactive power consumption to consider. The average reactive power consumption of the mine is 3.46 kvarh per tonnes.

For the period of the data set, the colliery was on a fixed price tariff with different prices for day and night, with the night period being from midnight to 7am. The price the colliery was paying for February and March of 2012 was 7.439p per kWh day and 5.957p kWh night. The energy prices quoted here are the only values available and are used throughout this chapter. The mine operates 24 hours a day, so if it is considered that the energy use is spread evenly across 24 hours, this gives an average of 6.7p per kWh and a coal energy price of £2.71 per tonne.

As mentioned above the mine requires reactive power in addition to active power. The amount of reactive power can be managed with power factor correction. A properly managed electrical distribution system should have enough power factor correction to keep the power factor at above or at least 0.8. The inductive loads of a mine require active control of the capacitive loads to ensure this value is maintained. It is not only the network stability that benefits. A large consumer will be charged by the utility to supply reactive power, although this charge compared to the active power is relatively low, at 0.123p per kvarh. The average power factor of the colliery is a little low at 0.75. However, the annual reactive power charge for the mine is only approximately £5.5k, which is fairly insignificant compared to the actual power charges. Another charge that is also included within an industrial consumer tariff is the availability charge. This is a significant charge at £0.6893 per KVA and with careful management can be reduced to a minimum. The colliery has a fixed availability of 20566 KVA and was paying £14176.14 per month. The actual maximum demand of the mine operations is 17750 KVA, this is 2816 KVA less than the charged availability. A saving of £1941.06 per month or over £23k per year could be made, just by renegotiating the availability charge with the utility.

### 8.1.2 Power Demand

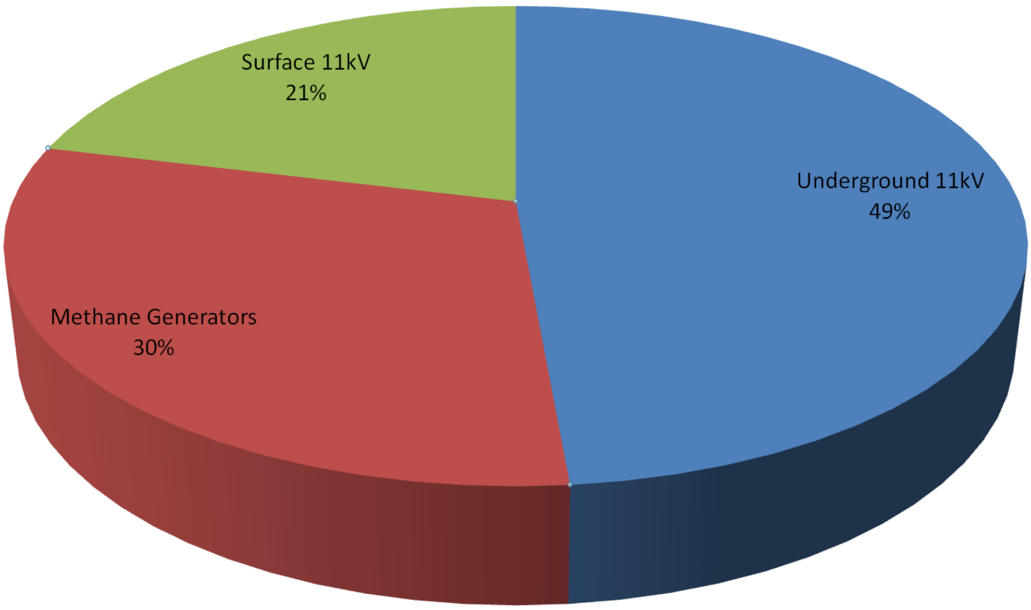



**Figure 8.1: Average Power Supply**

The mine is supplied from the national grid and on-site methane powered generators, which are supplied by the mine with pumped methane. Figure 8.1 shows the average distribution of supply, the surface and underground is grid supplied and the methane generators are connected to the surface distribution. This gives a supply requirement of 49% for the underground and 51% for the surface operations.

Figure 8.2 show the total average demand of the main areas of the mine. It can be seen that the highest demand is for the coal preparation plant at 19%, although this is many individual load collected together. This is the same for the other high demand areas, like the 6.6kV 4MVA and 10MVA underground feeder supplies. These supplies supply power to the underground operations, the breakdown of

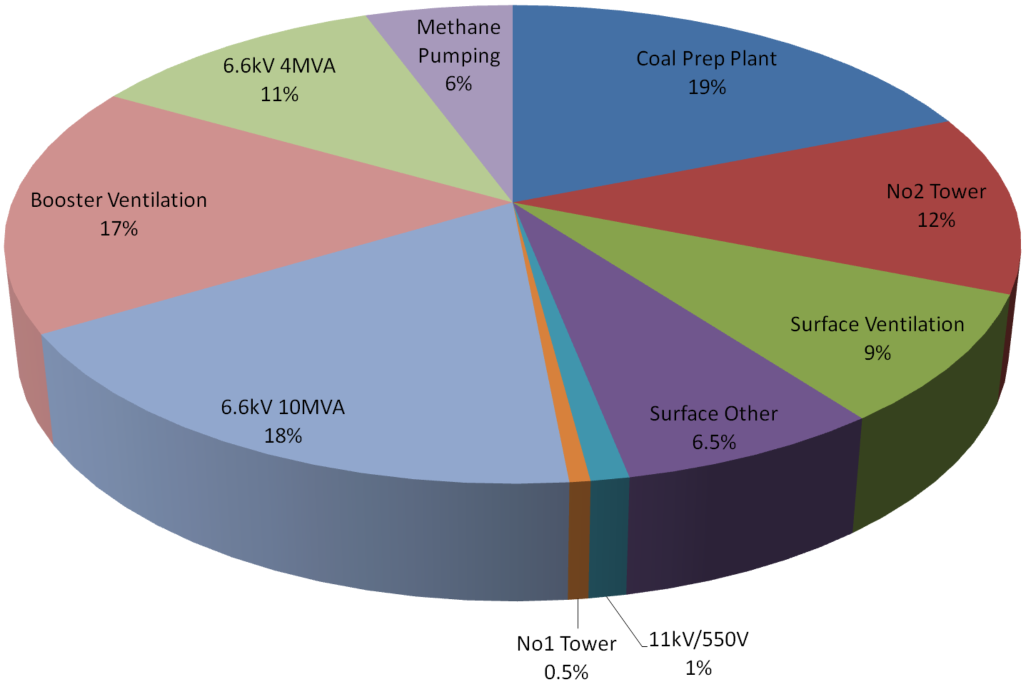



**Figure 8.2: Total Average Demand**

the individual loads is listed in Section 8.2.1. If we consider the surface fan and underground booster ventilation as one service, it is by far the most demanding service, with 26% of the overall demand. No2 Tower, the coal hoisting tower (uses over 10%.)

To get a better picture of the surface and underground demand, Figures 8.3 and 8.4 give the same total average demand separated into two charts for the surface and underground average demand. It can be seen in Figure 8.4 there is a demand for methane pumping, this is a surface service supplied from the 6.6kV distribution. The methane pumping is the net cost to supply the methane to the methane generators. If we consider this demand as a surface demand, this changes the surface and underground requirements to 57% for the surface and 43% for the underground operations. However, we must also consider that methane pumping is required to supply some of the actual power for the surface in the first place. Figure 8.5 shows the adjusted chart without the methane pumping demand, giving the adjusted figures of 52% for the underground and 48% for the surface operations. Figure 8.6 shows the adjusted total average demand and Figure 8.7 shows the adjusted underground demand. The surface demand requires no adjustment

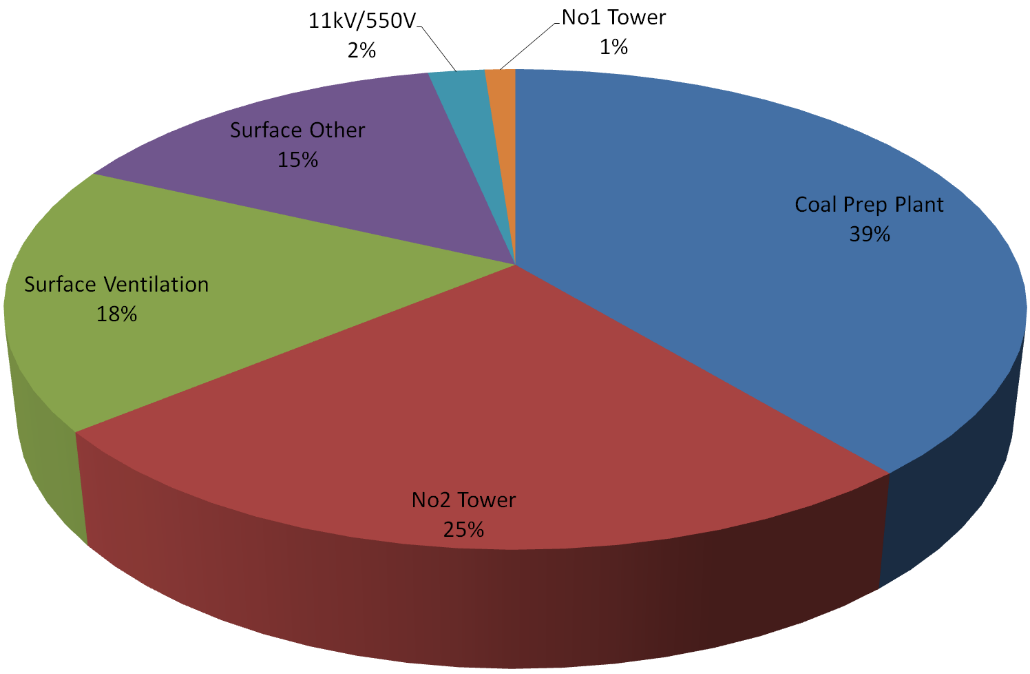



**Figure 8.3: Surface Average Demand**

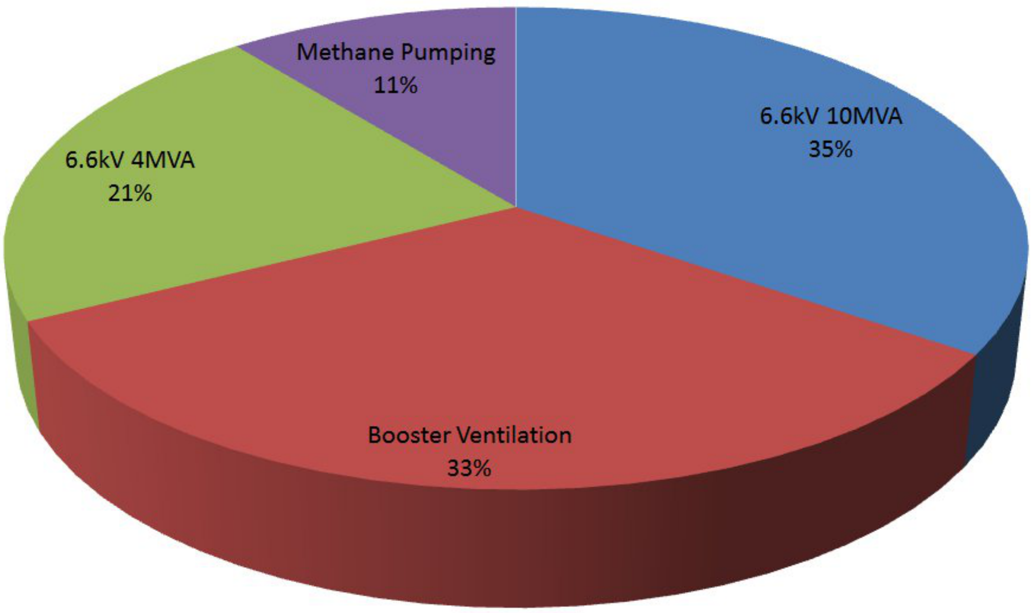



**Figure 8.4: Underground Average Demand**

as it is not affected by the methane pumping. However, the ventilation demand has increased slightly to 27% of total energy use. The ventilation operates constantly and is the most expensive service (apart from labour) for any underground coal mine.

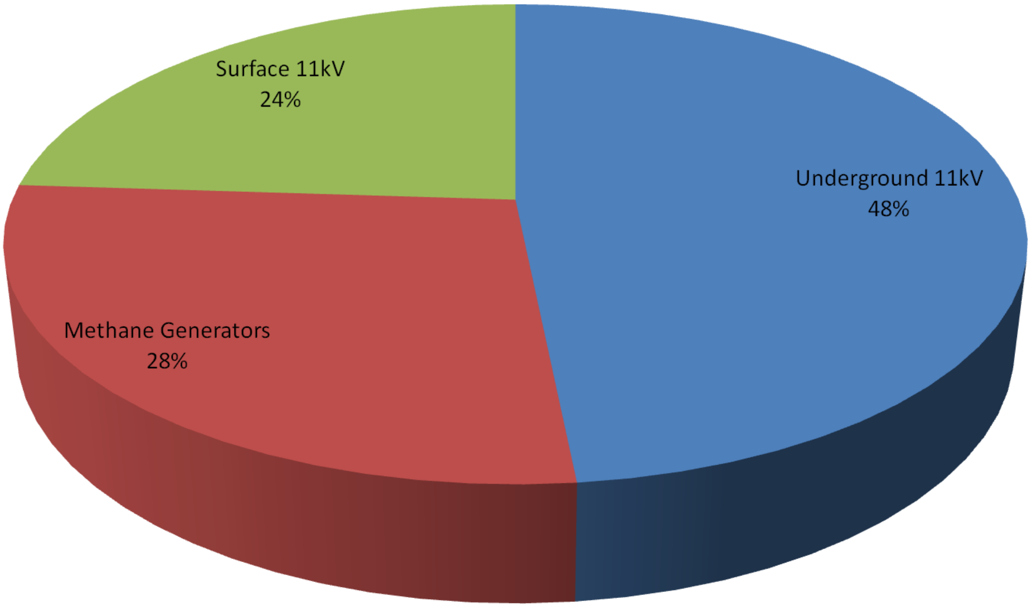



**Figure 8.5: Average Power Supply: No Methane Pumping**

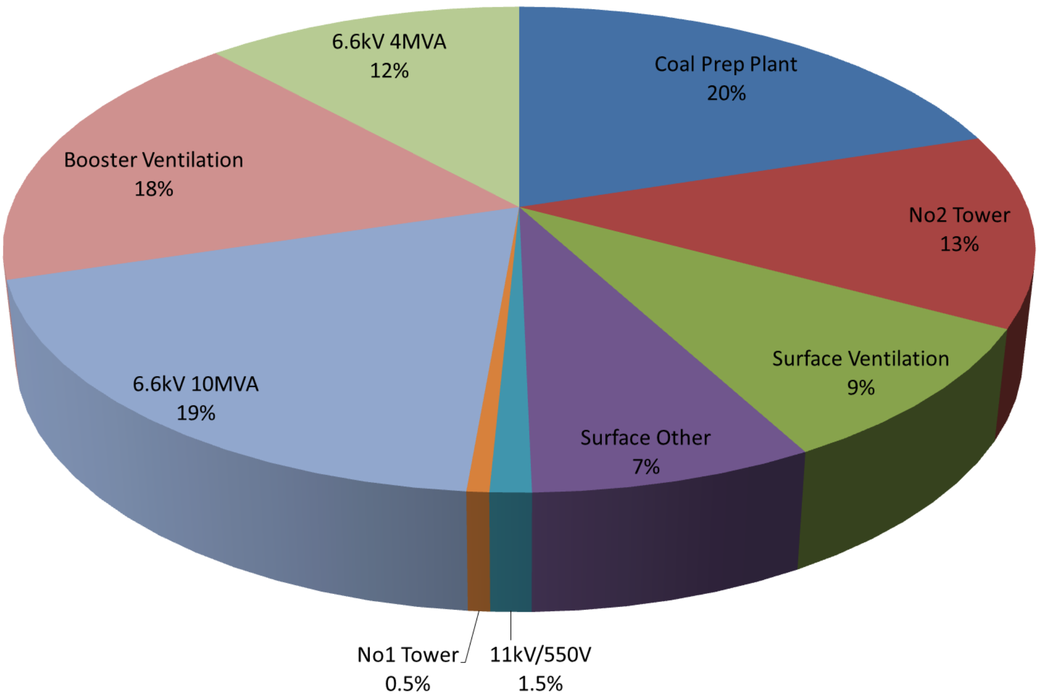



**Figure 8.6: Total Average Demand: No Methane Pumping**

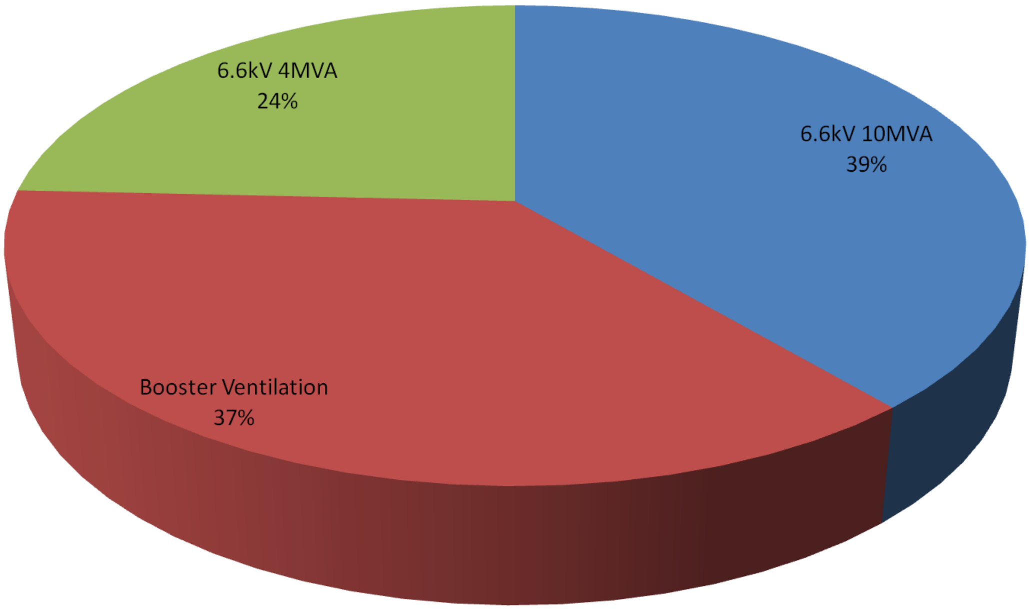



**Figure 8.7: Underground Average Demand: No Methane Pumping**

It is very unlikely that the assumption made earlier, that the energy use is spread evenly across the 24 hour period is correct and further analysis of the energy consumption shows that the average spread of energy use during the day and night is 59% and 41% respectively. With the ventilation requiring so much, it would seem that most of the production takes place during the day period. Shifting demand to the night period would make considerable energy savings. However, labour costs are generally high on night shifts and may out weigh the cost savings. Not enough information is available on the labour cost of the colliery to comment any further. Although, cost savings for night demand will be presented without considering possible additional night shift labour costs.

### 8.1.3 Production Detail

So far only a single average value of 5340 tonnes per day has been quoted for the mine production. This value does not give any detail to the production over time, and is a low daily value for a mine of this size. There are other considerations that go to make this value low. The is no production on Sundays, so considering this alone increases the daily rate to 6230 tonnes. Other factors are holidays, breakdowns, geological issues preventing production, mine safety, to name but a few.

Figure 8.8 shows a graph of the mines weekly production with the energy usage for that week, it can be seen that the production can vary considerably. It can be noted that the week ending 31st of August, is the first week that the production had started at a new face and there was no production for the first two days of that week. After that, the production is low, this maybe due to the bedding in of the machinery to the new face, as after a couple of week production picks up to an acceptable level. The energy use, as would be expected, follows the level of production. However, it can be seen that when the production levels are very low, the energy use does not drop right down, this is due to the base-load of the mine. The base-load is the constant power required to keep the mine operating before any production. Whether the mine is in production or not, there is a large demand just to keep the mine open, i.e. ventilation, surface buildings load, and lighting, these are described as the base-load, they can also be considered as the fixed loads. It is already known that the ventilation is 26% of the mines demand, this alone will have a significant contribution to the mines base-load demand.

Further analysis of the mine machinery operations and the production will highlight any differences and variations in operations. For example, how long the shearer was in operation per tonne, this could be an indication of geological issues or increased contamination in the coal. Figure 8.9 shows the amount of energy for the different areas of the mine for the same weeks in Figure 8.8. Much of the energy for the main areas of the mine remained constant such as the Surface Ventilation, 550V and No.1 Tower. The booster ventilation remained constant until the week ending the 17 August after that the booster ventilation uses significantly less. The booster ventilation system has recently been upgraded, the actual date is not known, although from Figure 8.9 it seems that it could of been around this period. During the week ending 28 September it looks like the booster ventilation was barely operated at all. No.2 Tower and C.P.P. are fairly consistent in following the production trend. This is what should be expected as the more coal the more hoisting and processing is required. The C.P.P. has a high demand during the low production period of the weeks of the 7 and 14 September. It can only be assumed that some of the mine output for this period is from the mine stock pile,

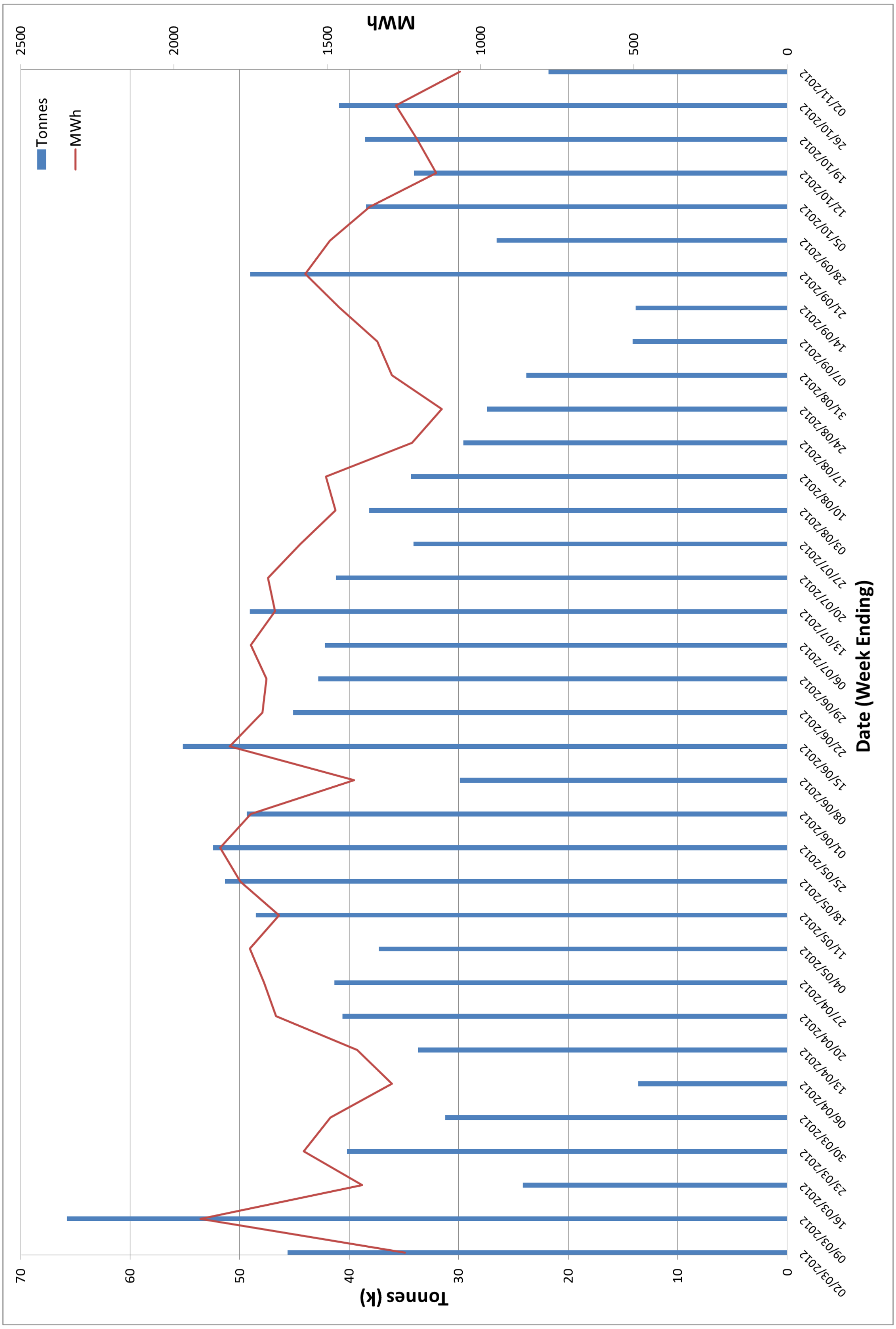



**Figure 8.8: Weekly Production and Energy: 26 Feb. to 2 Nov. 2012**

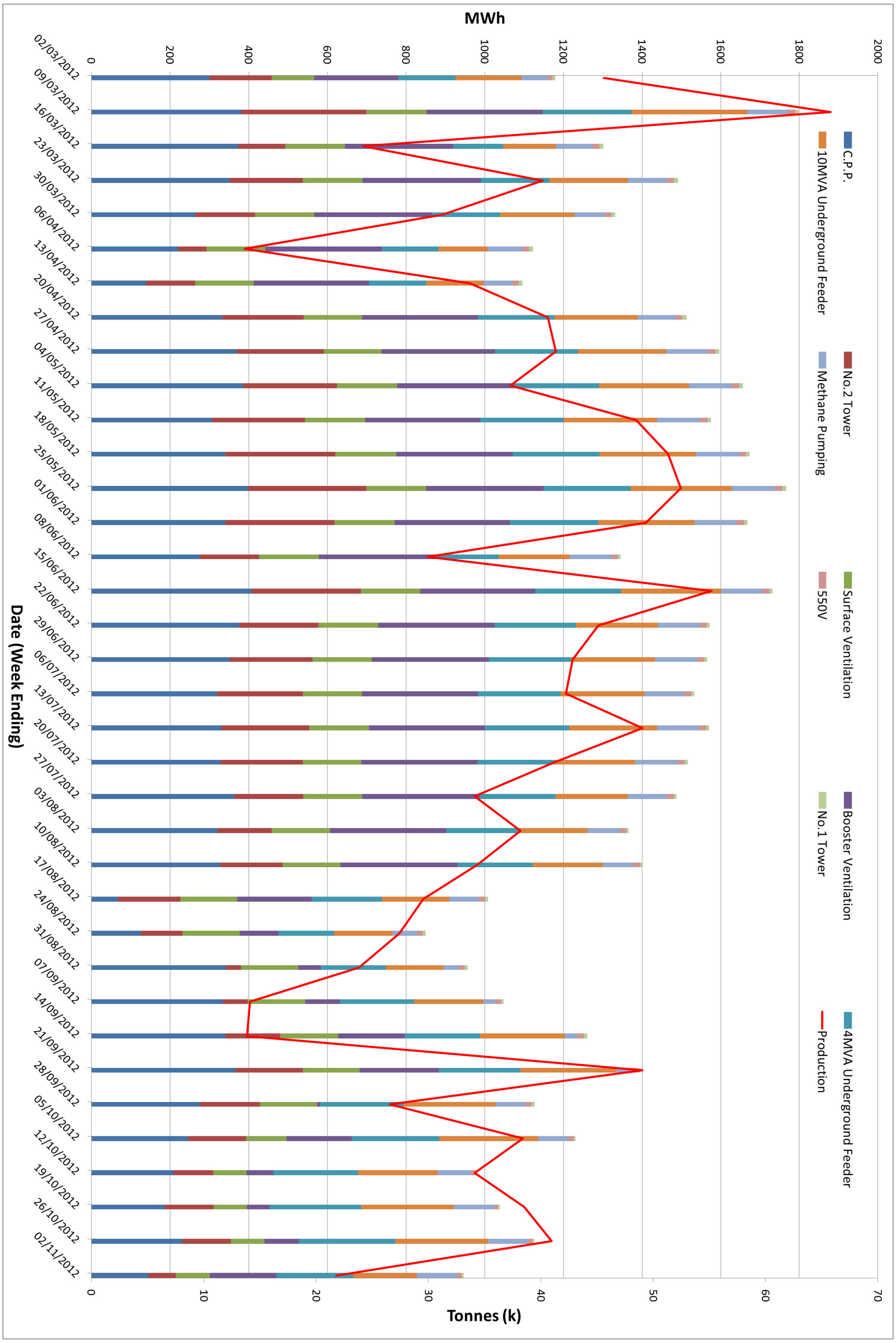



**Figure 8.9: Weekly Energy Distribution: 26 Feb. to 2 Nov. 2012**

as the hoisting is relatively low.

The 4MVA and 10MVA underground feeders vary and are consistent with the mine production. This to be expected as these are the supplies for the underground operations. The 4MVA supplies the pit bottom, pumping, underground compressors and the conveyor system nearest the shaft. The 10MVA supplies the further away conveyor systems, the face machinery and the development machinery. Detailed analysis of the machine operations supplied from the 4MVA and 10MVA are undertaken later in this chapter.

## 8.2 Analysing Live Production Data

This section presents the process and results of the application of the intelligent demand reduction system to the live production data that has been acquired using the data acquisition system described in Chapter 7. The four stages of the DRP are applied to the data. These are as follows:

- Stage 1: Data Acquisition
- Stage 2: Load Profiling
- Stage 3: Load Characterisation
- Stage 4: Load Scheduling

### 8.2.1 Stage 1: Data Acquisition

Chapter 7 presented the actual system installed at the mine. However, the system produced a vast amount of data, which needs selecting for analysis. This section presents the selection process of this data.

In the previous section, the analysis of the mine energy and production data showed that the hoist (No.2 Tower) varies according to the production. The hoist is on the surface and has its own power meter and requires minimal analysis. It is considered that the hoist is maybe the best indication of how much the mine has produced. Analysis of the production data and hoist data to find a collation between the hoist and production may be possible. This way the mine production could be estimated from the power meter data, without the need to source sensitive production data from the mine management.

It was also seen that the power demand of the 4MVA and 10MVA underground feeders also vary according to the mine production. Any equipment and its enclosures for the use underground must compile to ATEX flame proof regulations. This would increase the costs and practical requirements to a level that it was only possible to install power meters within the surface substations. The underground supply is provided by five 6.6kV feeders, three 4MVA and two 10MVA. There are power meters installed on all five feeders, but the two 10MVA and two of the 4MVA feeders are connected as two ring circuits, thus the readings from these circuit meters must be combined, so effectively there are only three meters covering all the underground power consumption. The remaining 4MVA feeder only provides power for the underground booster fan, which only gives two meters for all of the other underground services. The 10MVA feeders supply the face and some of the conveyor train and the 4MVA feeders supply the rest. An overall figure for the underground power consumption is not enough detail to give any considerations to how and where the demand could be reduced. Following is a list of the loads that are supplied by the 4MVA and 10MVA feeders:

**4MVA Feeder**

- Pit Bottom: Huwood Crusher
- Pit Bottom: Skip Vibrator Bed
- Conveyor: Tippler
- Conveyor: Huwood
- Conveyor: Hoist Skip Feed 1
- Conveyor: Hoist Skip Feed 2
- Conveyor: Strata
- Conveyor: Ranger
- Conveyor: Major
- Conveyor: Kellington
- Conveyor: Silkstone Access
- Pump: Beeston Intake
- Pump: 503 Face T/G

**10MVA Feeder**

- Conveyor: 502's L/G
- Conveyor: 503's L/G
- Conveyor: 503's L/G Tripper
- Conveyor: 504's L/G X-Slit
- Conveyor: 504's L/G
- Conveyor: 504's T/G Return
- Conveyor: 504's T/G

- Conveyor: Whitley 60"
- Conveyor: Beeston Intake
- Conveyor: Beeston Intake 2
- Face: Stage Loader
- Face: Crusher
- Face: AFC M/G
- Face: AFC T/G
- Face: Shearer
- Pump: 504 L/G
- Pump: D10s Flow

The list of loads above is the loads that are monitored with the SCADA system. This means there is data about their operation. There are also other loads in the mine that are fed from the 4MVA and 10MVA feeders; some are production loads and free loads. Unfortunately, there is not a definitive list of loads that are not monitored by the SCADA system. However, some of these loads are known. They are the mine lighting, loco chargers, and the main 250 kW pit bottom pump. The pump is an important load to profile, although the demand should not be difficult to identify in the data. The loco charges and lighting are also important and could benefit from demand management. However, as no information is known about them, i.e. power requirements and when they are being operated, there demand will be much harder to identify, for this reason they will be ignored.

The instantaneous values of the data acquisition system have a sampling rate of nearly twice a second. Studying 22 power meters for 256 days has resulted in a data set of over 100GB, with over 44 million values per each meter, per variable. A data set this large is impractical to analyse and a method of selecting the data for analysis had to be devised. It was decided to use snapshots of short periods of SCADA and power meter data. These snapshots are 24 hours long. As the mine had no special seasonal power requirements, such as heating during the winter months and from the analysis of the energy data, it was decided that the time of year of has no influence on demand. Events in the mine operation were used to select the snapshots. Three snapshots were used for the analysis of the 4MVA and 10MVA data, the event in operation were as follows:

- No Production.
- Peak Production.
- Average Production. management

The data logged into the SQL Server database is not equally spaced in time. This poses a problem for any data analysis. To compare data from different power meters or any load on/off SCADA data, the data must be at the same time intervals so that at any event the data is known at that time. The data could be interpolated when needed. However, this would pose the problem of needing to search for logged values either side of an event then interpolate to get the given value. This is awkward to do correctly and could easily introduce errors. It was considered that the table data would be interpolated first so that all the tables had their data with equal time spacing, so if a query was made for power data at a SCADA event, the event time could be used directly. A SQL script was written to perform the table interpolation. However, the process was very slow taking 24-48 hours to complete one weeks data from one power meter. With 22 power meters it was obvious that another approach was needed. The script was streamlined and only the required values were selected, this sped up the processing. However, this still was not acceptable, as every time non-interpolated data was needed, a long period was needed to process the data. Although the mine SCADA was not logging as much detail from the power meters as the OPC logger, it was considered that it was logging enough detail for this part of the analysis and scripts were written to query not only the SCADA event data, but also the SCADA power meter data as well.

A new database was created with the instantaneous power values from the CUBE 400 meters and the relevant SCADA data every 500 milliseconds. The SCADA data contained information about the loads in the list above, including the load contactor or switch state, the data contains discreet values of 0 and 1, 0 for off and 1 for on. The SCADA data also contains; some power information on the face machinery, conveyor belts, belt speeds, belt tear shut off and bunker level. The snapshots were then taken from the new database; the events were chosen as follows: CSM visited the colliery on the 12 March 2012, the purpose of the visit was to update some of the SCADA integration and check the system was operating correctly. However, on arrival the mine was not in production, due to the methane gas levels. The booster fan had been shut-down over the weekend for maintenance and after they had been restarted the ventilation and methane levels had not returned to normal levels.

It was a couple of days before the mine was back in production, this gave an excellent 'No Production' snapshot. Firstly without the booster fans and minimal underground activity and secondly with the booster fan and minimal activity. The 'No Production' snapshot is one of the most important snapshots, as the mine is in constant operation the base-load demand is difficult to define. Once defined, the base-load can be subtracted from the data set leaving only the data for the production or cyclic loads. The other two snapshots, average and peak production are defined using the production data. The details of all the snapshots are as follows:

- No Production: (10/03/2012 06:45:00 - 11/03/2012 06:45:00)
  Production 0 tonnes
  Development 0m.

- Peak Production: (18/10/2012 06:45:00 - 19/10/2012 06:45:00)
  Production 11000 tonnes
  (8.8 Face Strips, 6.65m Advance)
  Development 21.6m.

- Average Production: (24/07/2012 06:45:00 - 25/07/2012 06:45:00)
  Production 5816 tonnes
  (6.3 Face Strips, 4.3m Advance)
  Development 10.8m.

### 8.2.2 Stage 2: Load Profiling

The next stage is the load profiling of the loads. Some loads have their own dedicated meter and are easy to profile. However, most of the production loads that are to be included in the optimisation are underground and aggregated together in the 4MVA or 10MVA data. The data needs disaggregating into the individual load profiles. The process that was followed to do this is outlined below:

- Define 4MVA and 10MVA base-loads power and subtract
  this from the data sets.

- Process the limited SCADA power data and subtract this from the data set.
- Apply the disaggregation method described in Section 6.4.7 to define the remaining load profiles.

Although the snapshots are 24 hours, this is too large to analyse or visualise properly and difficult to tell if the data contains errors or not. For this reason, small chunks from the snapshots were also used for analysis.

#### 8.2.2.1 Define the base-load

To define the base-load the data 'No Production' snapshot is used. During this snapshot, there is no booster fan running and the methane levels were too high to operate the mine. This also meant that the labour force underground was at a minimum. Figure 8.10 shows the demand profile for the 24 hour no production snapshot of the 4MVA and 10MVA supplies. It can be clearly seen, at about 15:30 the power demand remains constant on both the 4MVA and the 10MVA. This continues for most of the profile. Following this a small section of the profile is examined, Figure 8.11 shows a 1 hour segment (17:00-18:00) from the same profile. It can be clearly seen that the demand is not constant, although the demand is changing at a constant rate, for both the 4MVA and the 10MVA.

The 4MVA has a minimum of 383kW and a maximum of 549kW. However, this maximum is caused by a combination of two loads operating a regular intervals. The first load has a demand of 75kW peaks that occur approximately every 2 minutes for about 15 seconds, and the other has roughly the same demand, although the intervals are different at about every 4 minutes for 5 minutes of operation. As it can be seen this is a very regular pattern. The SCADA data for the same period was analysed to see if the load causing the fluctuations is monitored by the SCADA system. None of the load monitored by the SCADA system were switching. The loads that are not monitored by the SCADA include the compressors. If there is no production then there is no compressed air use, although there will be some leaks from the system and these would use air at a constant rate, so it was considered that the two loads are air compressors. These are first load, although

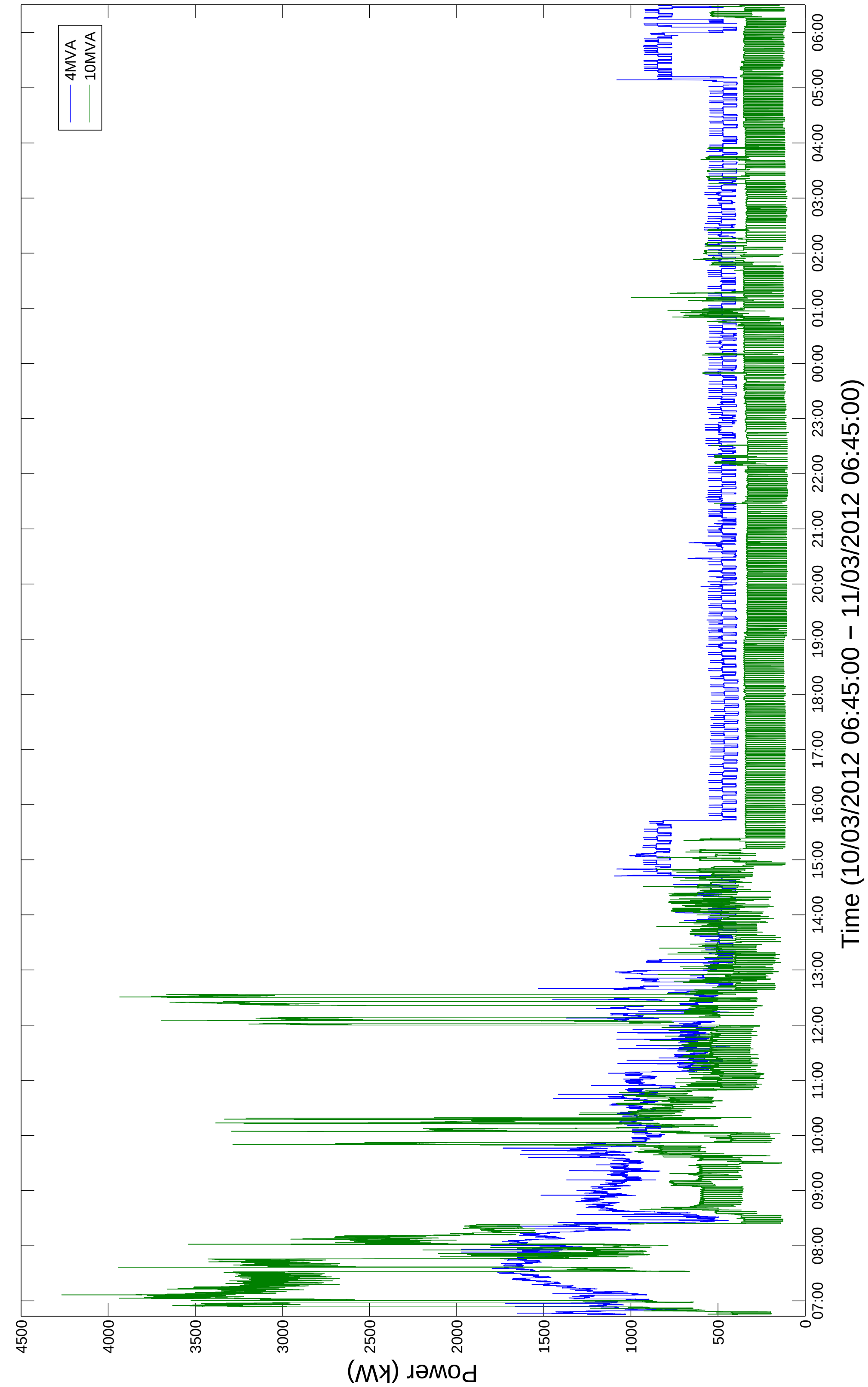



**Figure 8.10: No Production Snapshot: 4MVA and 10MVA Demand Profile**

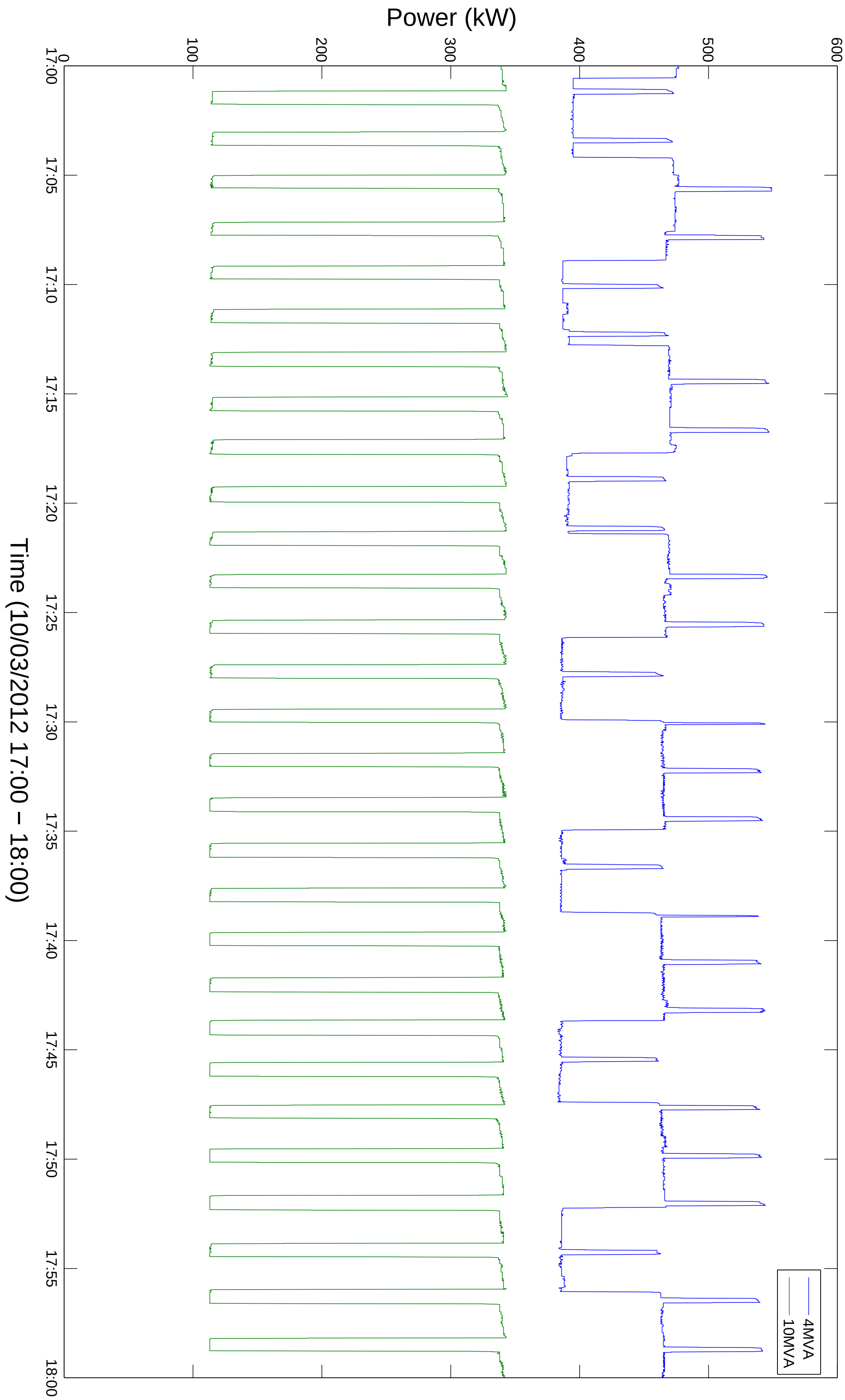



**Figure 8.11: No Production Snapshot: 4MVA and 10MVA 1 Hour Demand Profile**

using demand it is only 15 seconds every two minutes, this is one minute every eight minutes and seven and a half minutes every hour, and a total of 3 hours every 24. This combined with the second load that runs for over 19 hours every 24, adds up to a energy use of over 1.6MWh everyday and a cost of £3.5k per month, fixing the leak in the air system would be cost effective and reduce the underground demand. If these two loads are not considered then the base-load of the 4MVA supply is 380kW.

The 10MVA demand has one load of 230kW that is operating at regular intervals, approximately every minute for about two and a half minutes. The contactor state data in the SCADA database shows that there are no other loads running. When there is no production, the mine is still flooding at a constant rate. This flood water needs pumping to the surface. The main water pump is 350HP (262kW) and not monitored on the SCADA (It is monitored on the MINOS system, however the author did not have access to this system or data.) It is considered that the load seen on the 10MVA demand is the main pump controlled by a sump float switch. It is considered that the fluctuation of the base demand between 115kW and 340kW is this operating of the main water pump preventing the mine from flooding. As the water pump is always needed, it is proposed the demand is defined as part of the base-load, defining the 10MVA base-load as 340kW. A total of 750kW underground base-load demand.

#### 8.2.2.2 Define production load profiles

The main production loads are the conveyor belts, Longwall face machinery and the development continuous miner. The SCADA contains current and voltage data for the Longwall face machinery. This allows easy definition of the face load profiles. To profile the production machinery the peak and average snapshots were used, Figure 8.12 shows the demand profile for the 24 hour peak production snapshot and Figure 8.13 shows the demand profile for the 24 hour average production snapshot for the 4MVA and 10MVA supplies. The profiles were compared to a minimum of five 24 hour periods selected using peak and average energy and production data. This ensured that the profiles are representative of the peak and average demand throughout the time period.

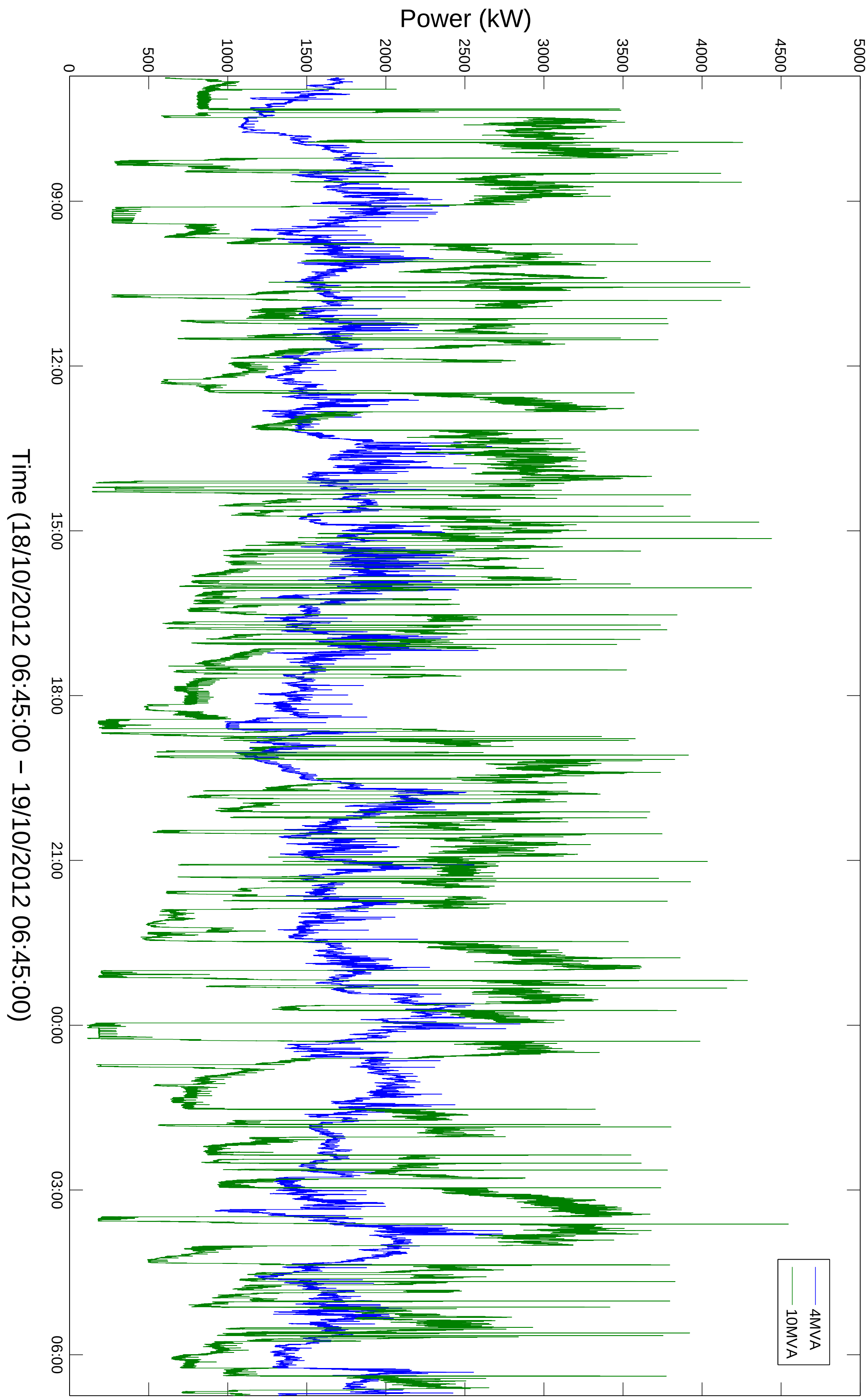



**Figure 8.12: Peak Production Snapshot: 4MVA and 10MVA Demand Profile**

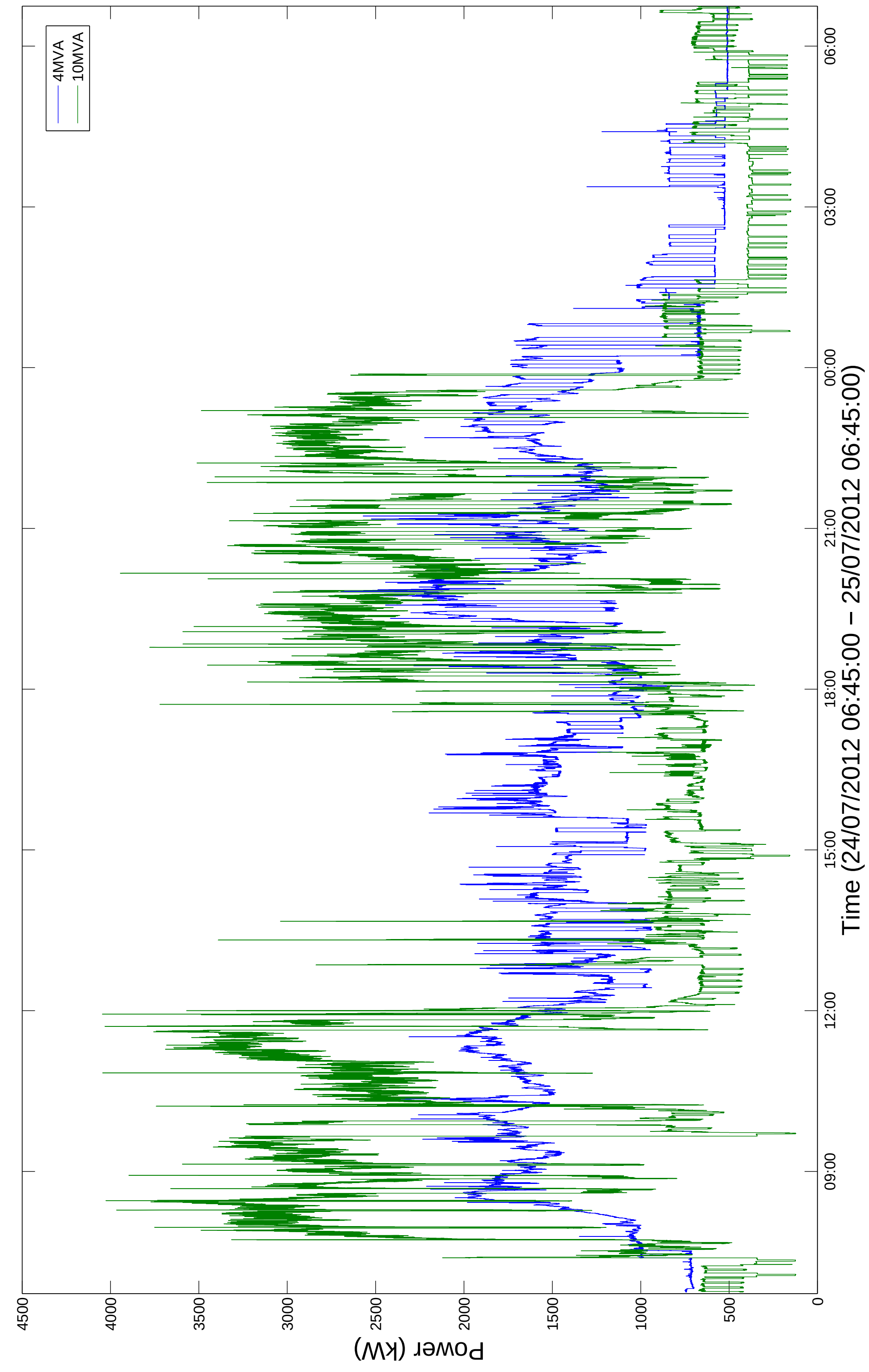



**Figure 8.13: Average Production Snapshot: 4MVA and 10MVA Demand Profile**

Figures 8.12 and 8.13 show the peak and trough demand profile of the mine production. The peak production demand is very consistent and only returns to the base-load on a few occasions. In contrast the average demand returns to the base-load level with regular intervals. It can been see that the mine operation are very different during peak production and average production. The peak production demand does not have as many distinct load signatures as the average production demand. The peak production demand profile will be used to profile the loads that have power data available as this profile data is better suited to provide steady-state load demand. Loads such as the face machinery, and the Major conveyor belt. However, the average production will be used for the disaggregation process, the distinct loads operations of this demand profile are better suited to this type of analysis. A one hour snapshot from each of the production profiles is made to do the analysis, Figures 8.14 and 8.15 show these one hour snapshots.

For the peak demand profile, it is considered that what hour was used for the analysis is unimportant as firstly, the profile has a dense demand and secondly, it is the SCADA data for the period that is to be used for the profile and not the profile data itself. This means that, it is only important that the machines are operating at steady-state peak conditions. However, the average production is a different matter, for examination of Figure 8.15 the operation of the mine can be seen to be less than the peak. The profiling of the individual loads using their power data is described using the face machinery SCADA power data, Figure 8.16 show the face power data for the same period of the one hour peak production snapshot.

The profile of all the machines can be seen, the AFC motors tail gate and main gate ends, the crusher, the stage loader, and the shearer. The large start-up peaks can also be seen, especially for the AFC. The pattern of the face machinery can also be seen. The AFC is started before the coal cutting starts, this is to reduce the possibility of the AFC being unable to start. Figure 8.17 shows 10 minutes of steady-state operation of the face machinery, their demands are as follows:

- AFC M/G - 610 kW
- AFC T/G - 565 kW

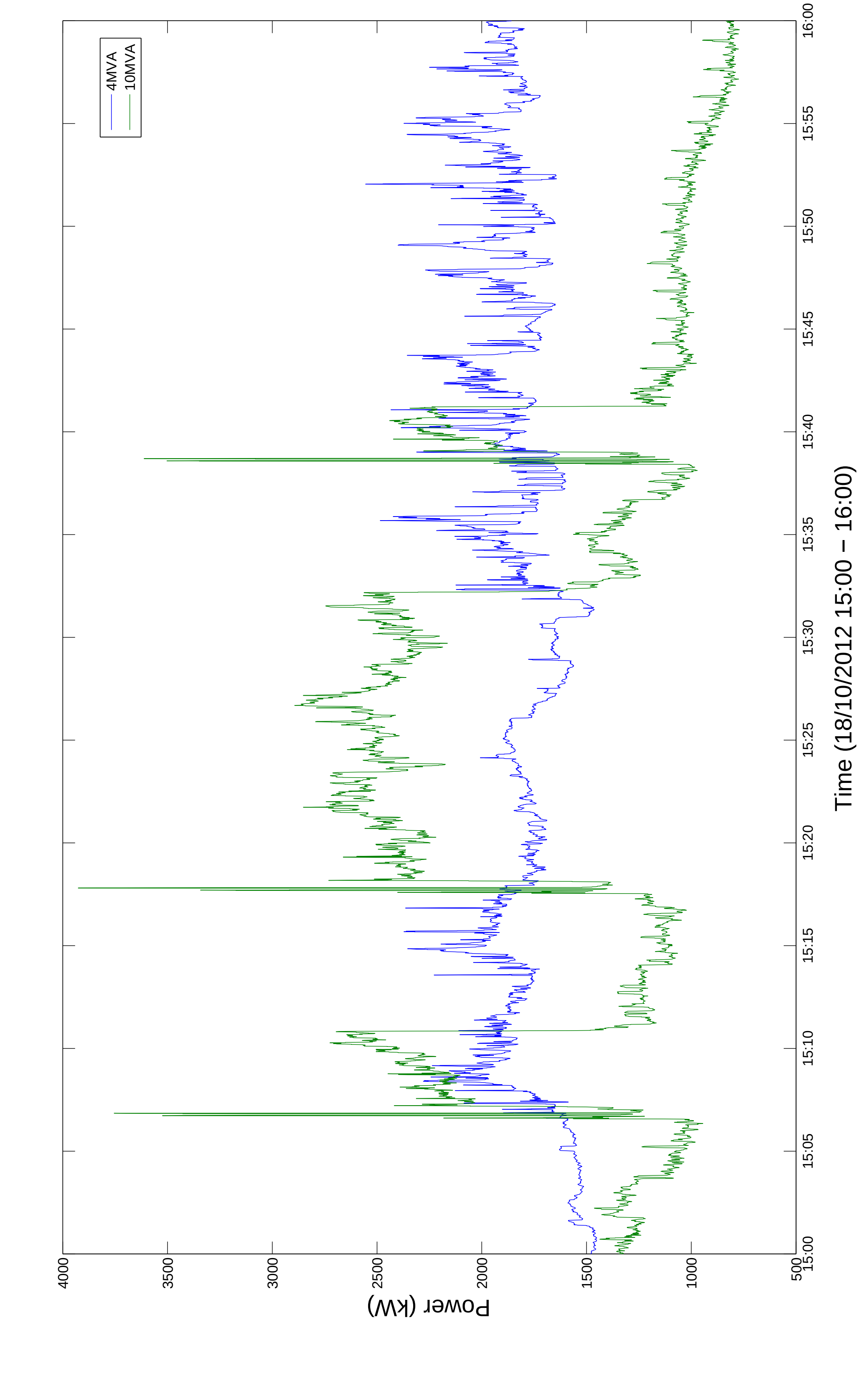



**Figure 8.14: Peak Production Snapshot: 4MVA and 10MVA 1 Hour Demand Profile**

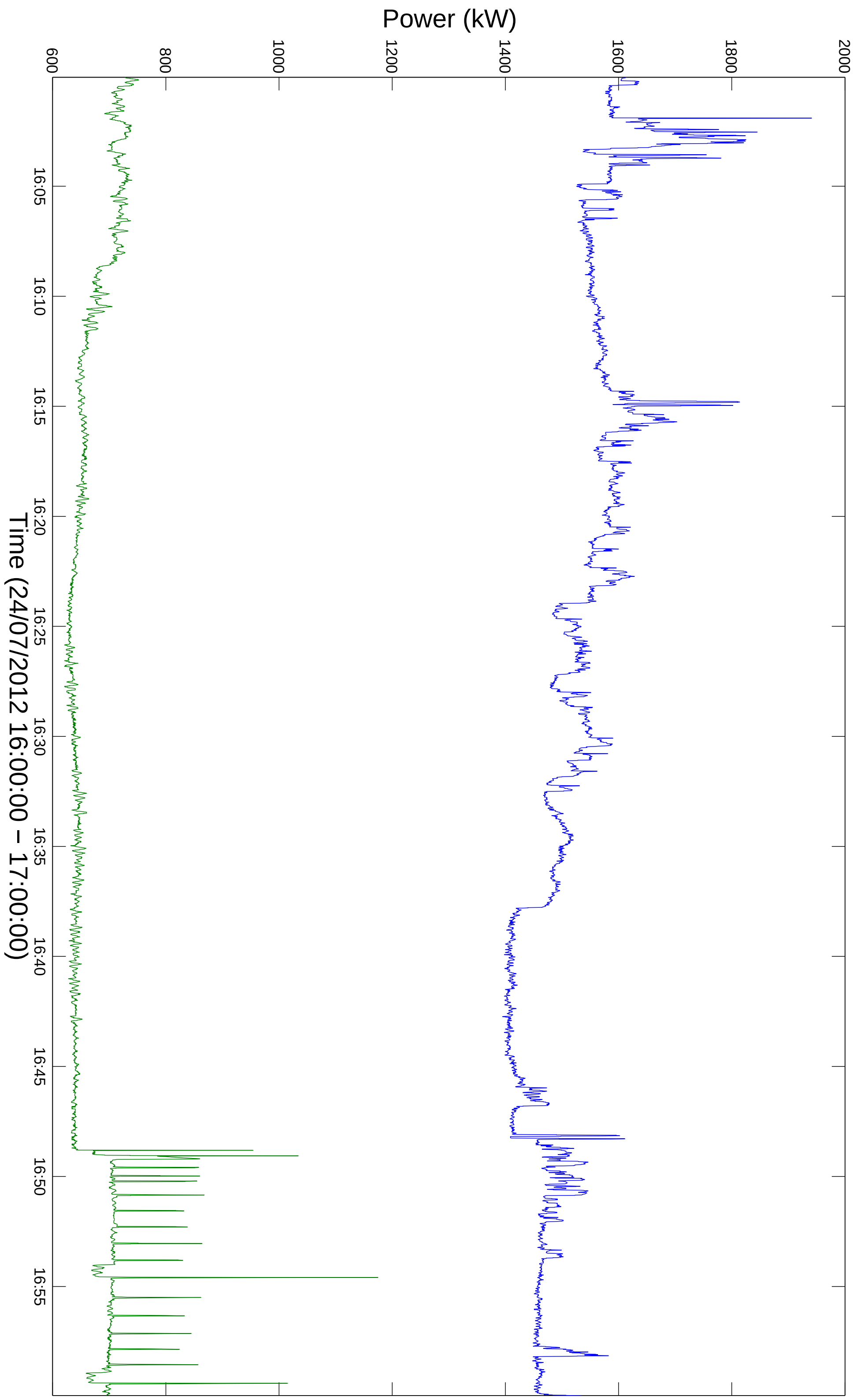



**Figure 8.15: Average Production Snapshot: 4MVA and 10MVA 1 Hour Demand Profile**

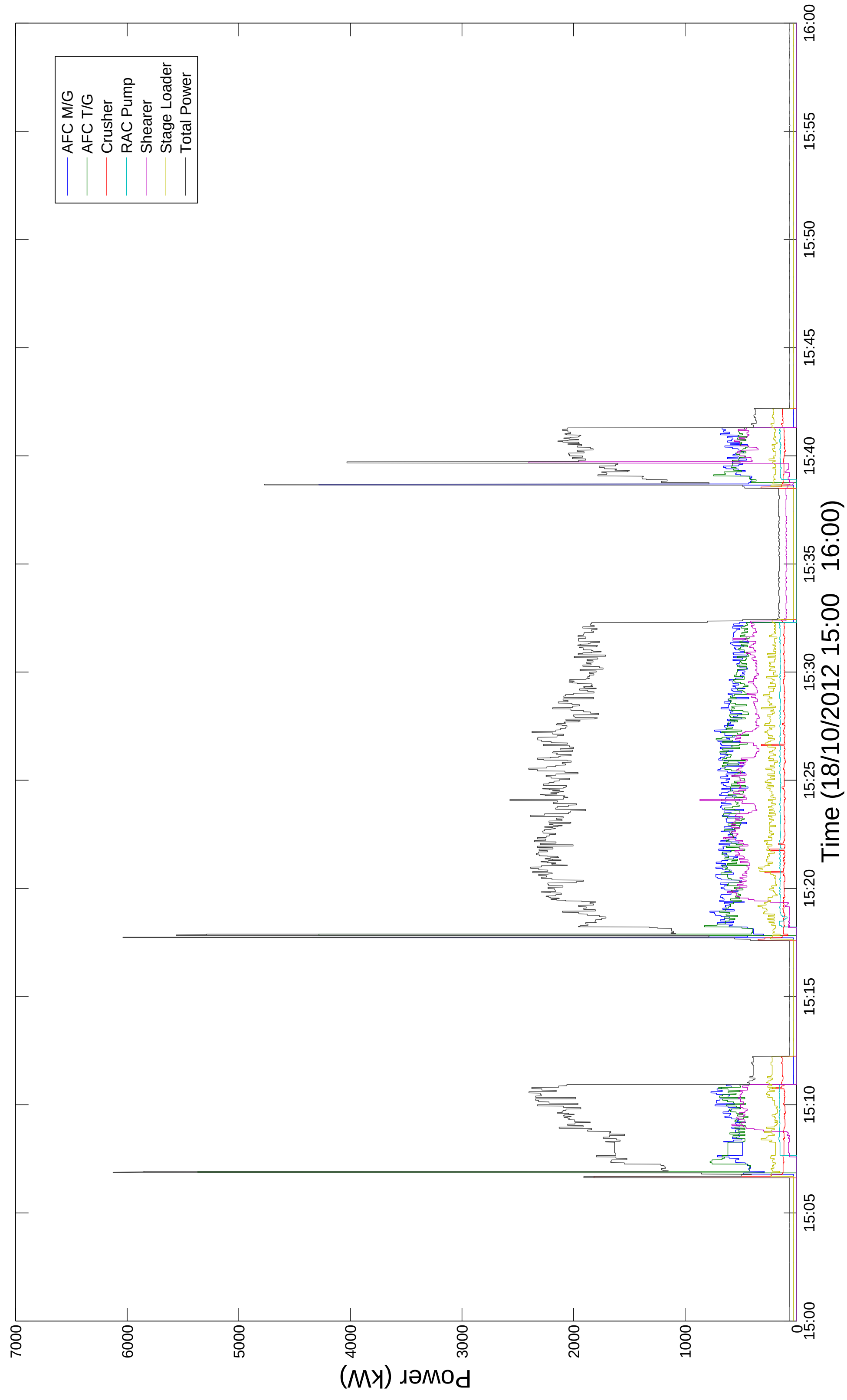



**Figure 8.16: Peak Production Snapshot: Face Machinery 1 Hour Demand Profile**

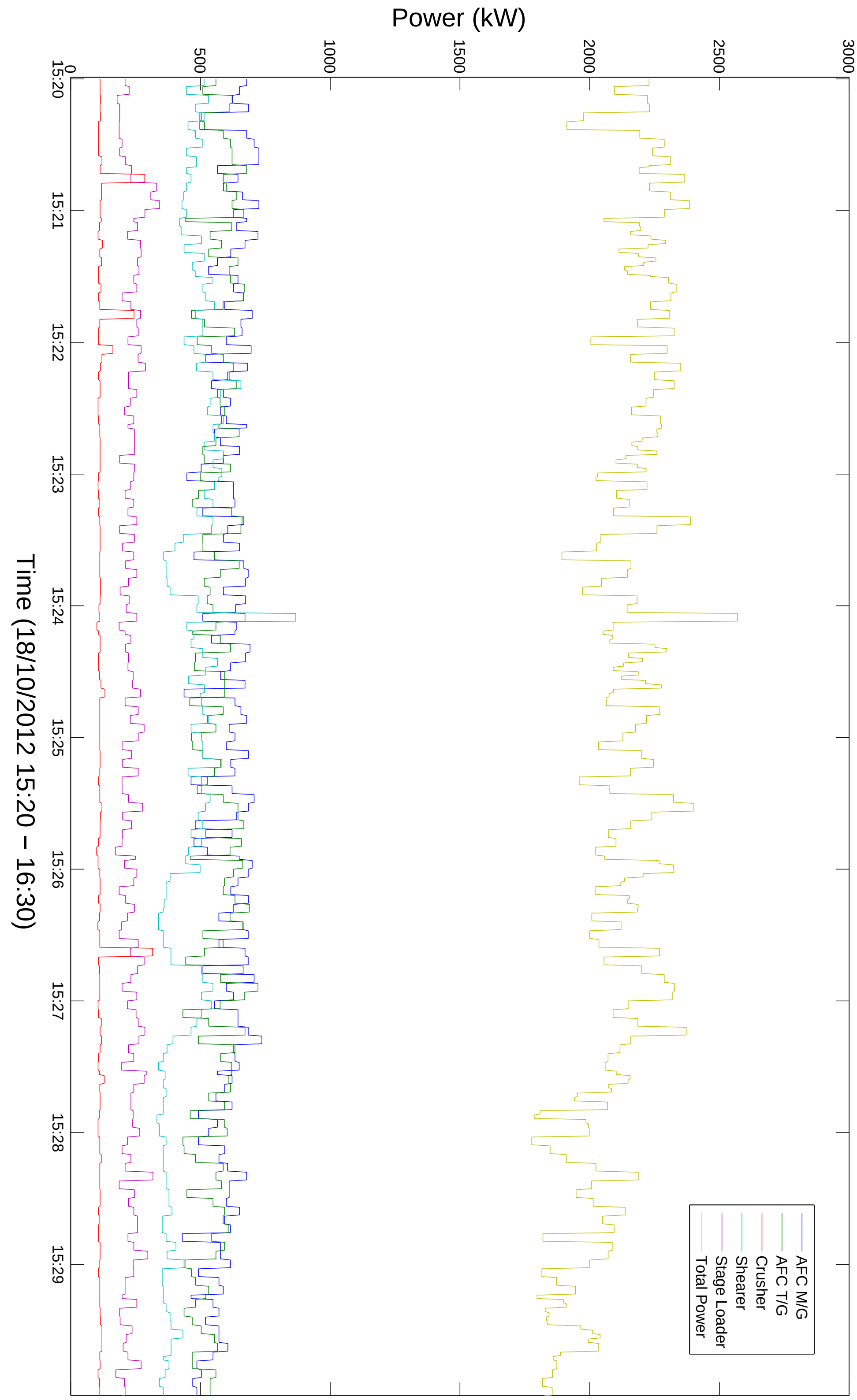



**Figure 8.17: Peak Production Snapshot: Face Machinery 10 Minute Demand Profile**

- Shearer - 453 kW
- Crusher - 115 kW
- Stage Loader - 232 kW

### 8.2.2.3 Disaggregating Power Data

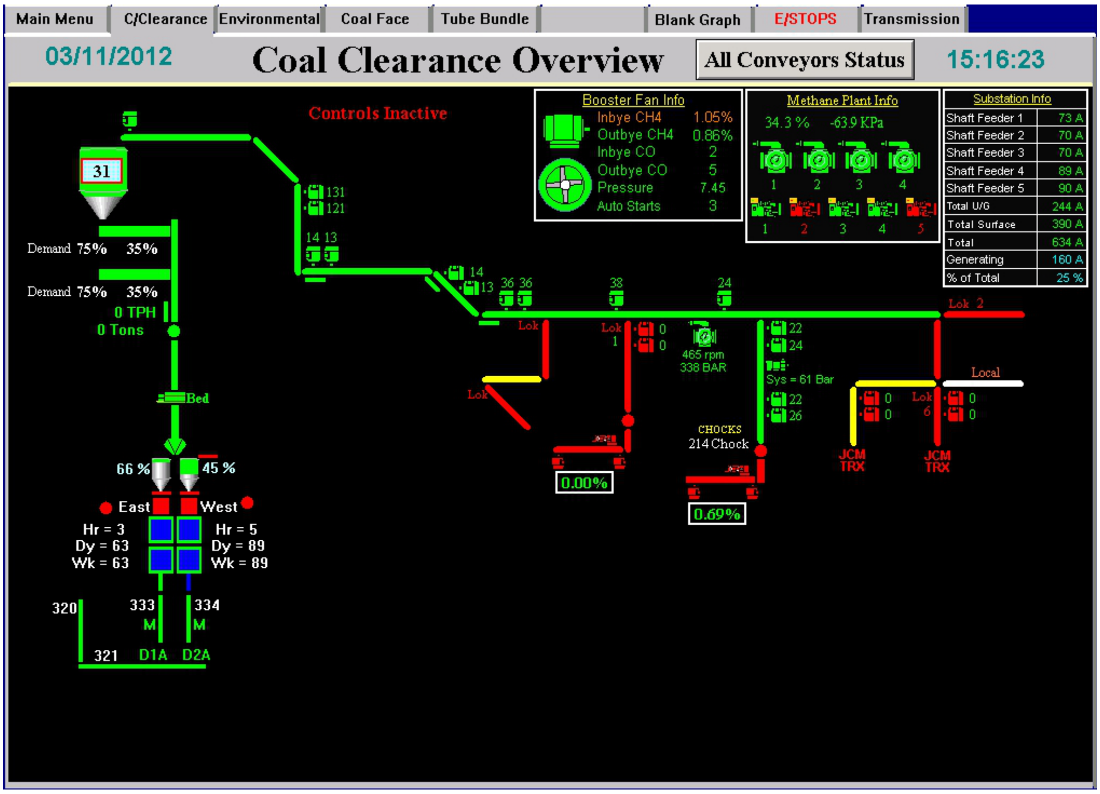



**Figure 8.18: SCADA Conveyor System Screen** (UK Coal)

The conveyor belts are not so simple, the conveyor belt state in monitored by the SCADA system. The SCADA also logs a current value for the conveyor belts. However, this value does not seem to have any relevance to the actual current demand of the conveyor drive, as it is the current of the DC side of the inverter drive and not the input of the drive. Figure 8.18 show the SCADA screen for the conveyor system. The current values for each conveyor can be seen on each drive. The belts that are running are in green. A simple calculation to check these current values, will prove that they are not directly relevant to the power demand.

Table 8.1 shows the details of the conveyor system, the last two column shows the value from Figure 8.18 and the power they would be demanding if it was correct.

| Conveyor List | No. of Motors | Volts (V) | No. of Power(kW) | SCADA Current | SCADA Power |
|---|---|---|---|---|---|
| **Pit Bottom Skip Plant** | 2 | 550 | 11 | - | - |
| **Tippler** | 1 | 1100 | 112 | - | - |
| **Huwood** | 0 | 1100 | 0 | - | - |
| **Out Feeder 1** | 1 | 550 | 30 | - | - |
| **Out Feeder 2** | 1 | 550 | 30 | - | - |
| **Strata** | 1 | 1100 | 225 | - | - |
| **Ranger** | 2 | 1100 | 400 | - | - |
| **Major** | 2 | 1100 | 600 | 131, 121 | 249, 230 |
| **Kellington 60"** | 2 | 1100 | 1000 | 14, 13 | 27, 25 |
| **Whitley 60"** | 2 | 1100 | 500 | 14, 13 | 27, 25 |
| **Beeston Int No.1** | 2 | 1100 | 1000 | 36, 36 | 69, 69 |
| **Beeston Tripper 1** | 1 | 1100 | 300 | 38 | 72 |
| **Beeston Tripper 2** | 1 | 1100 | 300 | 24 | 46 |
| **Silkstone Access No.1** | 1 | 1100 | 48.75 | - | - |
| **502's L/G** | 2 | 1100 | 300 | - | - |
| **503's L/G** | 2 | 1100 | 600 | 22, 26 | 42, 50 |
| **503's L/G Tripper** | 2 | 1100 | 600 | 24, 22 | 46, 42 |
| **504's L/G X-Slit** | 1 | 1100 | 110.25 | - | - |
| **504's T/G** | 2 | 1100 | 500 | - | - |
| **504's T/G Return** | 1 | 1100 | 110.25 | - | - |
| **504's T/G** | 2 | 1100 | 500 | - | - |
| **Beeston Ret Heading** | 1 | 1100 | 110.25 | - | - |
| **Beeston Int No.2** | 1 | 1100 | 112.5 | - | - |

**Table 8.1: Conveyor Belt System Details**

The SCADA current data for the Major conveyor is real current and as can be seen in Table 8.1 is representative to the actual power. This data can be used to profile the Major conveyor belt. The table clearly shows that the other current data is not representative of the actual current of the conveyors, so this data cannot be used to profile the conveyor system. Instead the process of Section 6.4.7 will be applied to the data, the results of this process follow.

The average production one hour snapshot for the 4MVA, was processed by firstly subtracting the base-load and any other power data from the SCADA. According to the SCADA the state of the loads for the period is:

- Pit Bottom: Huwood Crusher - **Stopped**
- Pit Bottom: Skip Vibrator Bed - **On/Off**
- Conveyor: Tippler - **Stopped**
- Conveyor: Huwood - **On/Off**
- Conveyor: Huwood Fines - **On/Off**
- Conveyor: Hoist Skip Feed 1 - **Running**
- Conveyor: Hoist Skip Feed 2 - **Stopped**
- Conveyor: Strata - **Stopped**
- Conveyor: Ranger - **Stopped**
- Conveyor: Major - **Running** (SCADA Power Subtracted)
- Conveyor: Kellington - **Running**
- Conveyor: Silkstone Access - **On/Off**
- Pump: Beeston Intake - **On/Off**
- Pump: 503 Face T/G - **Stopped**

The load coefficients matrix was then created from the SCADA data for the same period, only the loads that are operated on/off are included. There is also a column added for the running loads, as they are treated in the same way as the base-load was in Section 6.4.7. The unknown load demands are then attempted to be solved. The first attempt did not produce any convincing results. Another attempt was made with a 30 minute time period. It was considered that with less data less errors would exist. This still did not produce any convincing results, two more attempts were made with the same data at 15 and 7.5 minutes, the results for all the runs attempts are shown in Table 8.2.

In the table, it can be seen that the results contain large errors for some of the loads, although the correct values were solved for Conveyor: Silkstone Access on the 30 minute attempt and the Running loads was possibly solved correctly with

| | Load Demand | 1 Hour | 30 Mins | 15 Mins | 7.5 Mins |
|---|---|---|---|---|---|
| **Running Loads** | 1030 kW | 950.55 | 502.24 | 2.62e14 | 3.17e14 |
| **Huwood** | 187 kW | 7.82e13 | 1.74e14 | 0.00 | 0.00 |
| **Huwood Fines** | - | -7.82e13 | -1.74e14 | 175.69 | 276.72 |
| **Skip CV** | - | 45.76 | 32.42 | 60.20 | 108.53 |
| **Skip Vibrator** | - | -3.69 | -7.70 | -10.94 | -39.67 |
| **Beeston Intake** | 3.6 kW | -9.33 | 81.90 | 86.50 | -3.17e14 |
| **Silkstone Access** | 48 kW | -98.39 | 47.08 | -2.62e14 | 0.00 |

**Table 8.2: Disaggregation of Live Data Results**

the one hour and 30 minute attempts. This is not conclusive and requires further research. The data set was not selected with any selection method, to get any data that would be ideal for this type of process, the only selection was from average coal production and visually from the 24 hour load profile. Further research on this method would need better selection of the data. It is recommended that the next time this is attempted, that the data set is tailored to try to get lots of short sets with a specific load in mind, perhaps solving for one load at a time with short data set over a longer period than one hour. As each load was solved the data could be reduced, helping to solve for the next load. These are only suggestions and only further research will develop the final solution.

### 8.2.3 Stage 3: Load Characterisation

The next stage is to define the load characteristics, at this stage of the project it is felt that it is not necessary to characterise all the loads and profiles. However it is felt that it is necessary to further prove the LC MATRIX concept with real power data. This is done by using the peak production one hour data snapshot. The profiles of the face machinery and the hoisting system are used for the optimisation of the load profiles with the LC MATRIX. The hoisting system is defined as the fixed load and the face machinery is defined as a cyclic production load. The LC Matrix optimisation of the face and hoisting system is presented in the next section.

### 8.2.4 Stage 4: Load Scheduling

Figure 8.19 shows the snapshot of the real data taken directly from the power meters and SCADA, the total demand curve can also be seen. First the face machinery is optimised for the lowest demand. Figure 8.20 is the optimised schedule of the profile in Figure 8.16. Figure 8.21 shows the hoist profile for the peak production one hour snapshot and Figure 8.22 show the optimised profile for hoist. The next step is to optimise the hoist and face together to reduce the demand even further. Figure 8.23 shows the final face and hoist optimised schedule. Table 8.3 shows a summary of the result from the optimisation. The table contains the maximum demand and standard deviation for the original and optimised load profiles, allowing the demand reduction gains to be easily understood.

| | Original | Original | Optimised | Optimised |
|---|---|---|---|---|
| **Load Profile** | **Maximum (kW)** | **Std Dev (kW)** | **Maximum (kW)** | **Std Dev (kW)** |
| **Face** | 6125.8 | 959.4 | 5427.0 | 861.7 |
| **Hoist** | 4523.0 | 991.9 | 4276 | 1006 |
| **Total** | 9234.5 | 1464.9 | 6384.3 | 1391.1 |

**Table 8.3: LC MATRIX Face and Hoist Optimisation**

The LC MATRIX optimisation of the face and hoist machinery has reduced the maximum demand by 2850kW. There has been very little change in the profiles, just a small shift in the starting points of some of the loads. The demand reduction is nearly 3MW. This has a significant impact and is a constant demand reduction. A central control system with the optimised schedule to operate the load to will always see this reduction. During peak periods the mine has a peak of 18MW. This includes the unoptimised face and hoist profiles. However if the optimised peak face and and hoist profile is operated, this would be reduced to 15MW and save the mine over £2000 per month.

## 8.3 Summary

This chapter has presented the analysis of the data form the UK Coal colliery. Firstly, the profile of the mines production and energy used was presented. The

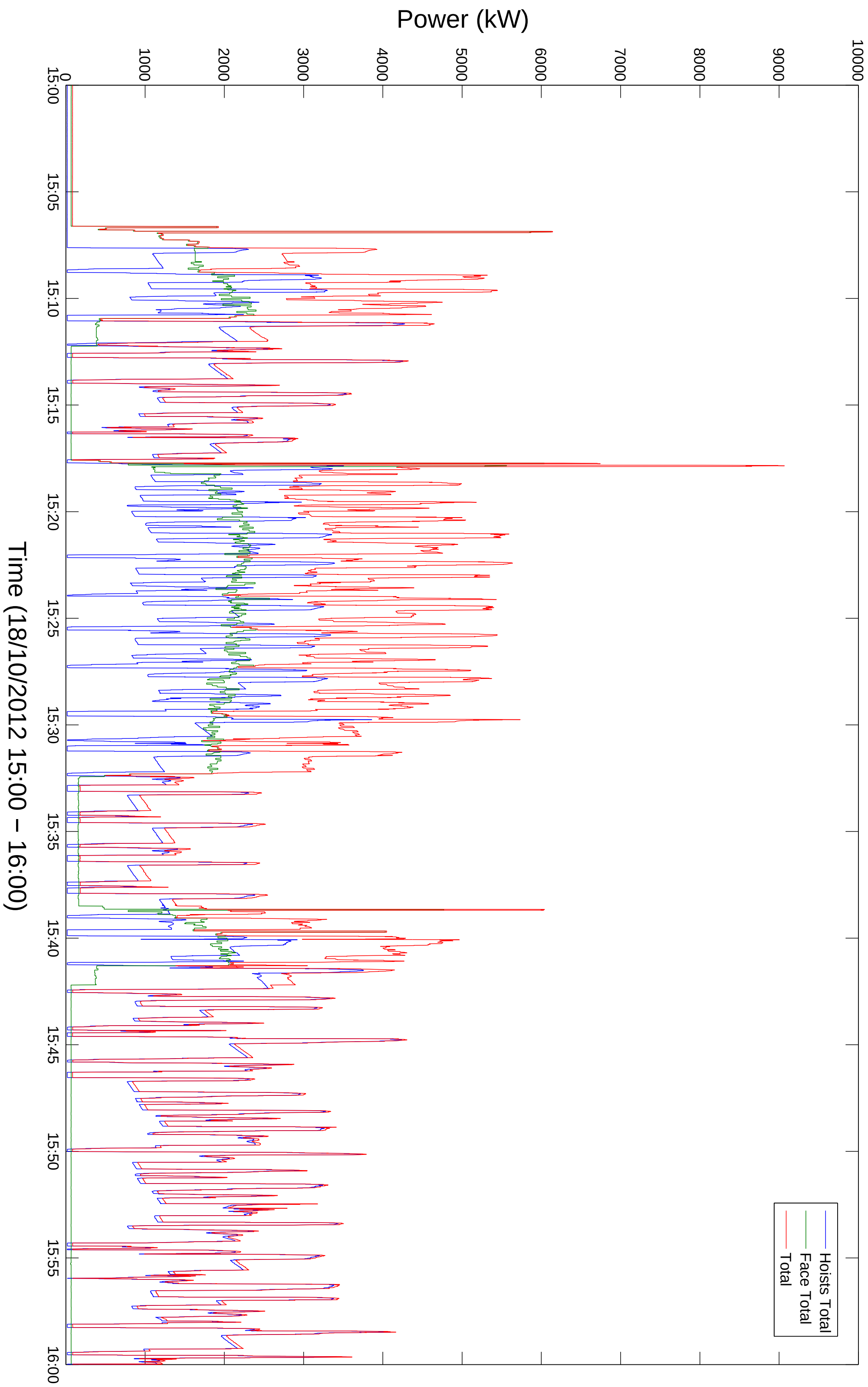



**Figure 8.19: Peak Production Snapshot: Hoists and Face Machinery 1 Hour Demand Profile**

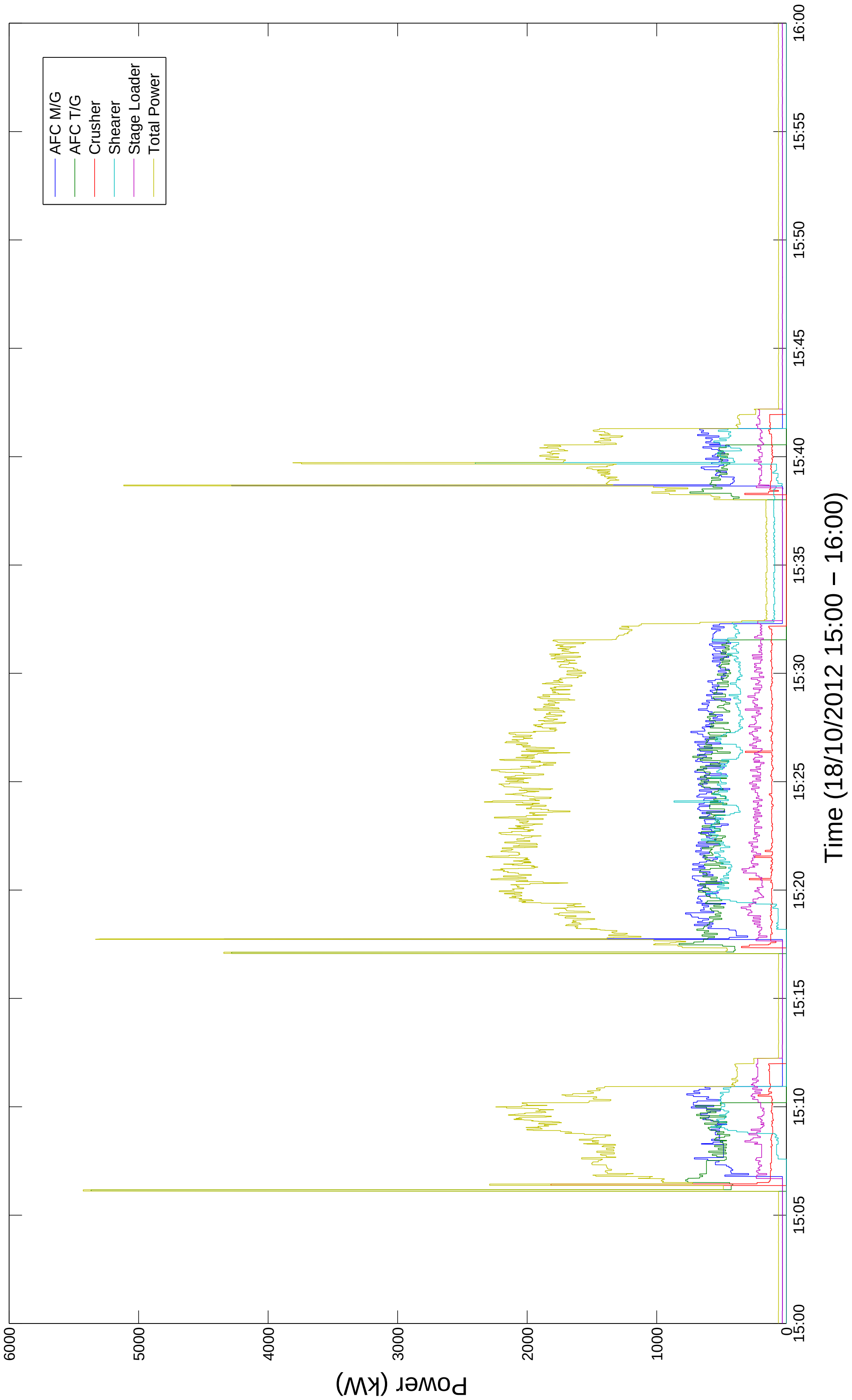



**Figure 8.20: Peak Production Snapshot: Optimised Face Machinery 1 Hour Demand Profile**

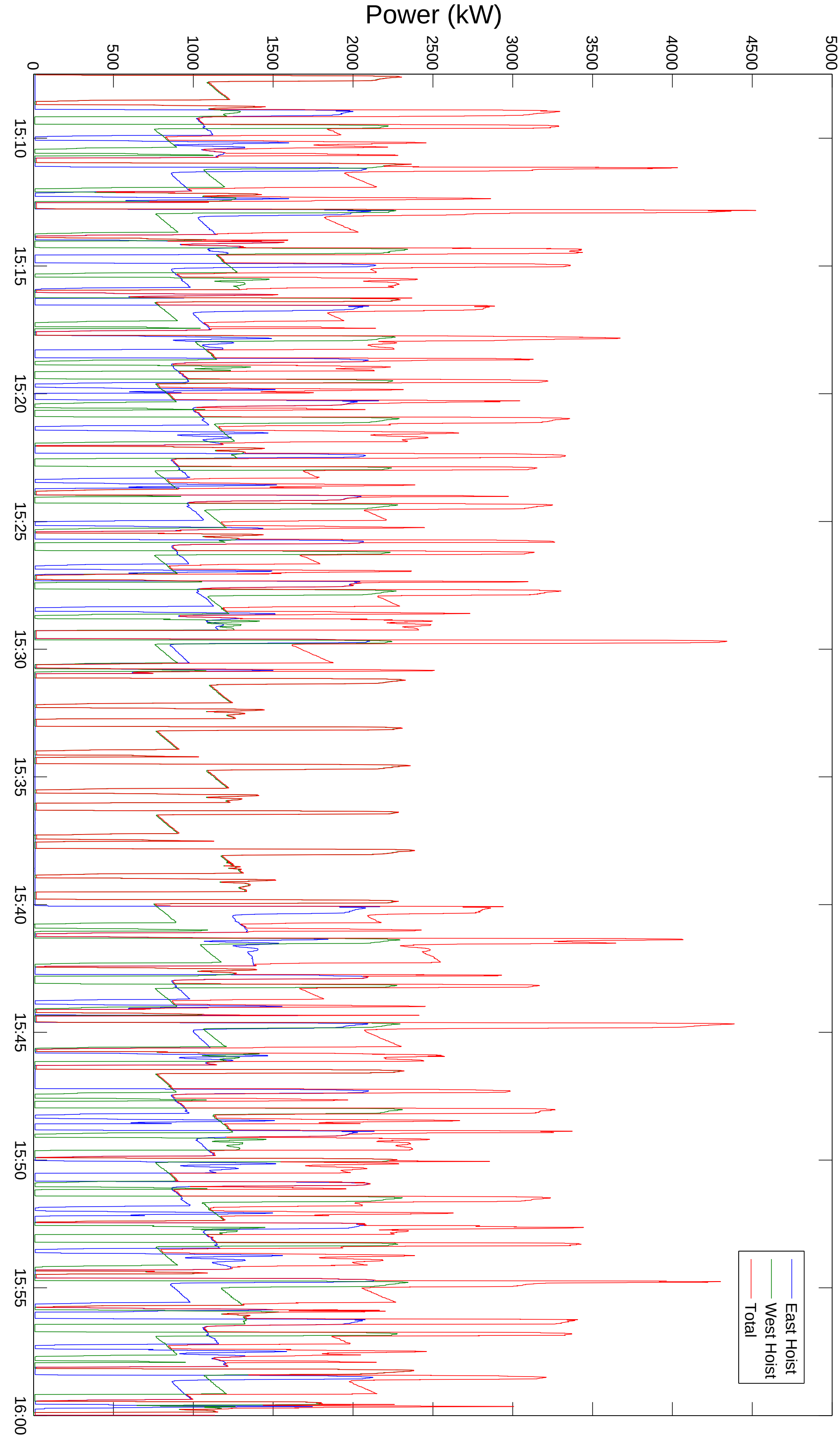



**Figure 8.21: Peak Production Snapshot: Hoist 1 Hour Demand Profile**

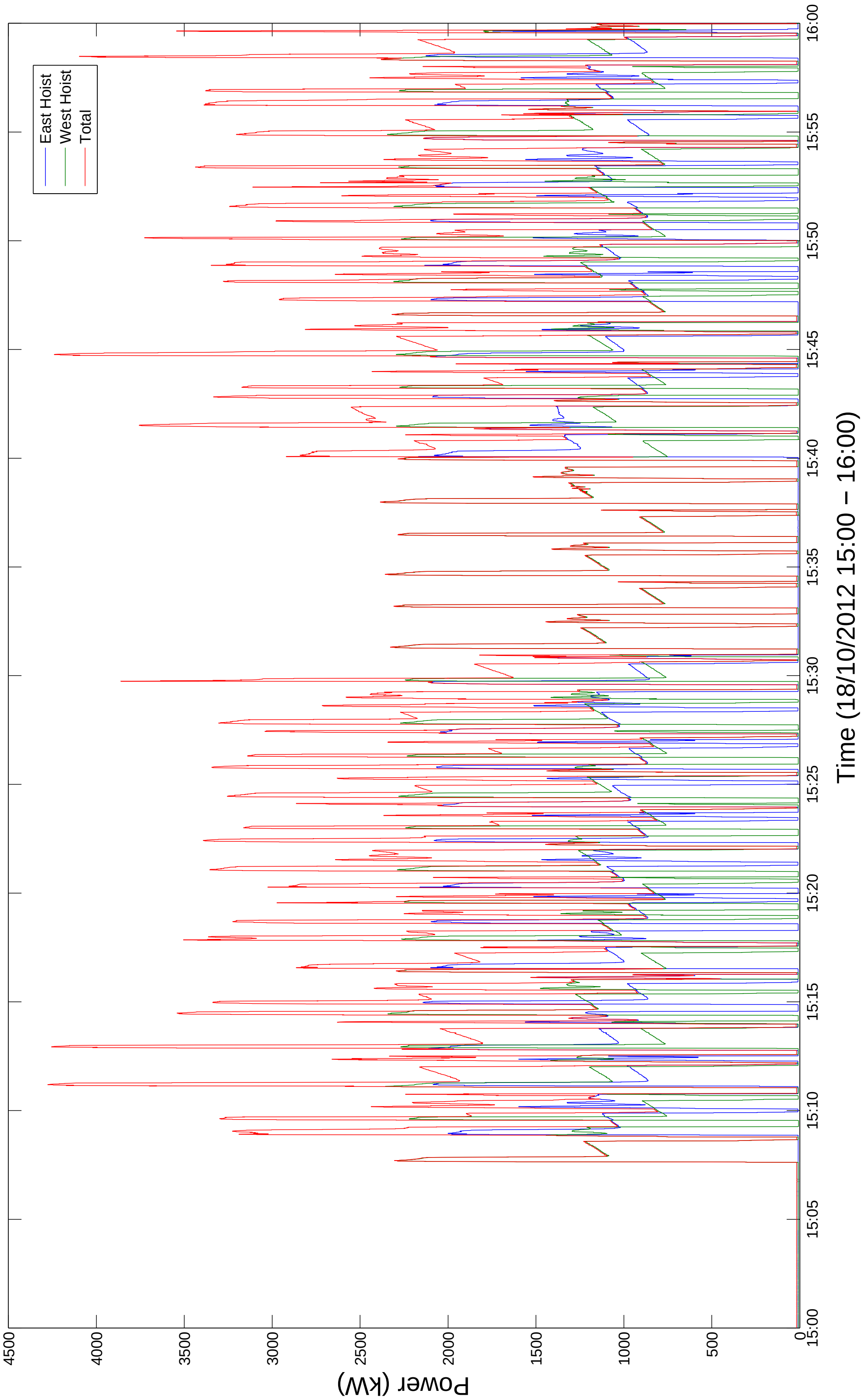



**Figure 8.22: Peak Production Snapshot: Optimised Hoists 1 Hour Demand Profile**

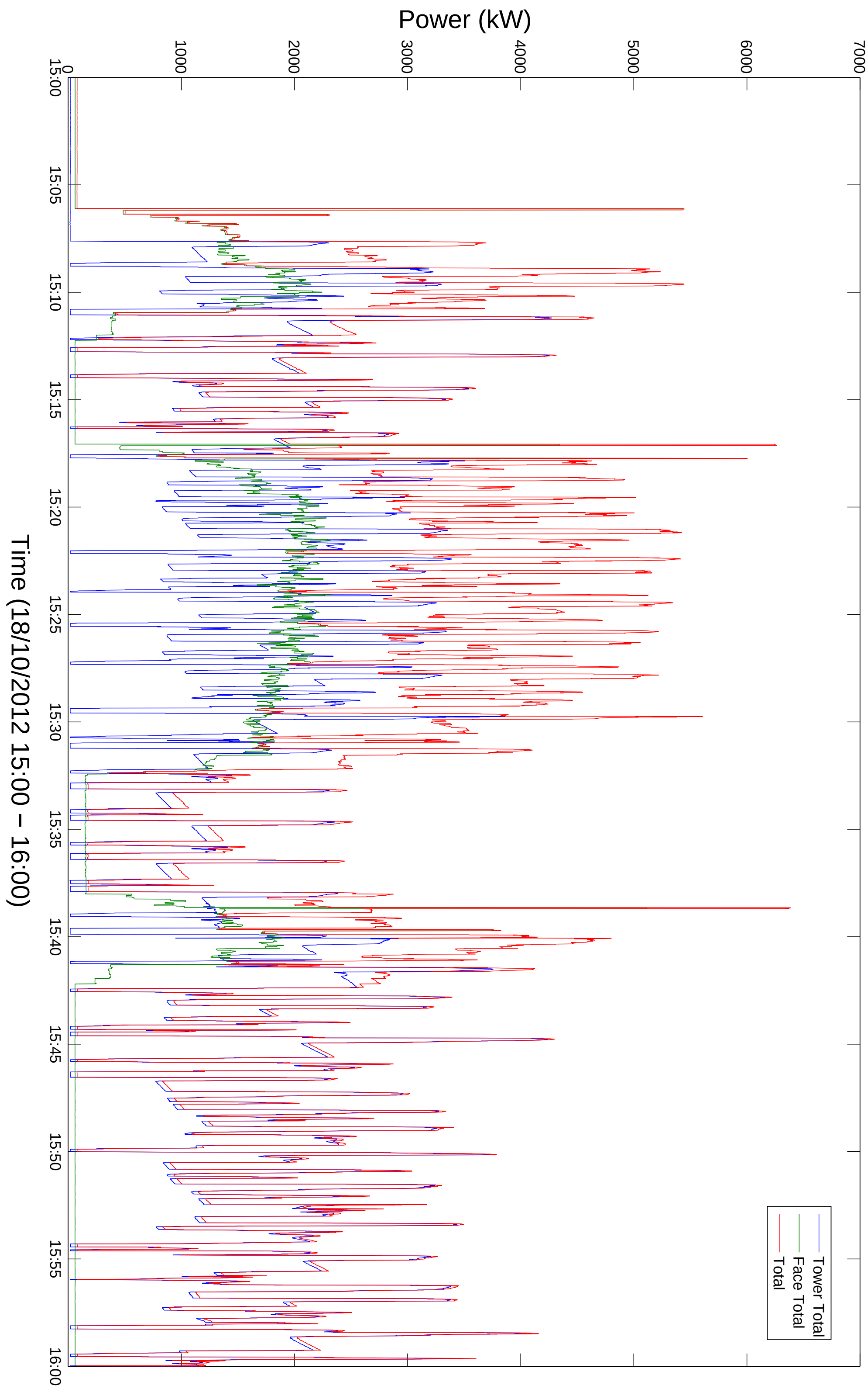



**Figure 8.23: Peak Production Snapshot: Optimised Hoists and Face 1 Hour Demand Profile**

concepts and strategies presented in Chapter 6 are then tested with the real production data. The demand reduction optimisation of the LC MATRIX has been proven to work, firstly with simulated data in Chapter 6 and in this chapter with the real mine data. The concept of disaggregation of aggregated data was also presented, although the results were not conclusive and this method requires further development. The next chapter contains the conclusions and recommendations for further work.

# 9

# Conclusions and Recommendations

This thesis has presented demand reduction and responsive strategies for underground mining operations. Demand reduction can be approached in many ways, from simply switching off loads - to a complicated management system. It was considered that the simplest ways are sometimes the best and the methodology was not to find the most complicated genetic algorithm or neutral network to solve the problem, but to start with the basic approach and develop the project from there. The standard ways to save energy and increase efficiency were first researched. These included high efficiency motors, motor speed reduction, low energy lighting all which are applicable to the mining industry. However, these are not the only solution to the demand reduction problem. What was needed next was a deeper look into the mining processes themselves. This required data analysis and the analysis required data. The mining industry is not known for its openness and this was the first hurdle to overcome. A questionnaire was compiled, but the response was limited. It was concluded that this was due to the confidentiality of the mining industry and the lengthy response required to complete the questionnaire.

Live production and power data from a mine was required. It was then that the author approached UK Coal for this data and a agreement was made between the University of Exeter and UK Coal. The author then designed and installed the data acquisition system at the UK Coal colliery. Extensive research was required before the system could be installed. This included power meters, CTs and PTs, serial communication wiring, MODBUS communication, SCADA systems and software, OPC, SQL and relation databases, and database replication. The hardware/software system had to be robust and deliver secure and reliable data under demanding conditions. The data rate frequency had to be as high as possible. This required further investigation into MODBUS gateways to increase the data through put from the meters - in fact the system monitors and logs were at the highest rate possible for the hardware installed. During the design and installation problems with the power meter wiring and SCADA integration were encountered. These were overcome with the construction of a test meter and and the development of a test system in the lab. This helped the author to produce a wiring procedure for the power meters and the existing CT and PT wiring. Checks were also made to validate the data from the power meters, and was proven to be over 99% acurate. Once these problems were overcome, the system was considered a complete success and at the time of writing has been in operation without problems for over two years.

During the design and installation of the data acquisition system, the author spent over 210 hours at the colliery and had many opportunities to develop relationships and gaining knowledge within the coal mining industry. The Longwall retreat was the mining method of choice at the colliery so its operation and the processes surrounding it became the focus of the project. The Longwall mining method posed a difficult application to apply any demand reduction strategies. The reason for this is that the process is so defined. It is unlike other industrial processes such as car manufacturing. With car manufacturing, there are many different stages and these stages are independent until towards the end. For example, the engine needs building, the interior needs making, the chassis needs welding; all of these can be done independently and combined at the end. This gives plenty of scope to optimisation and synchronising.

Longwall mining is the complete opposite. Nearly every stage relies on the last - and they all rely on the flow of coal. Interrupt this and the whole process stops. This observation allowed the process to be looked at in a different way; ensuring that during optimisation the process is never stopped and to use the time or waiting period to the advantage of the system. The in build bottlenecks that restricted the system. For example the hoisting system has a maximum through put rate, mine any faster than that rate and the mine is over working. However, the mine is a machine and machines breakdown. This over working can sometimes help to compensate for these breakdowns periods.

The final system concept proposed, developed and tested during this project, took the simple approach and started with basic load control. The first problem to be considered, was the starting up of the loads. Most loads in a mine are motors and motors require high start-up demands, synchronising these loads so that they do not start simultaneously will always reduce demand. The next thing to consider was the information needed, it was considered that only basic information was needed about the load's demand. Basic information allows faster computation speeds and simpler operation algorithms. The third thing to be considered and probably one of the most important; any system must have a generic approach, not to be fixed to this one mine, process or even industry. This led the author to use a priority rule base real-time decision system, with load characteristics to group and process the loads. In this way the rules could be defined and fitted to the individual process without having to redesign the main system.

A system of load profiling, load characterisation and load scheduling was developed from scratch by the author. The load profiling was developed using the a skeletonisation approach, this is where the details of a load are reduced to only the required details. The loads were divided into load types, fixed demand, variable demand and transient demand. Matrices were then used to define the demand of these load types, matching matrices were also defined for the load output. This approach is simple and requires very little data storage of processing power to manipulate or store. Any type of load can be defined with system and is robust enough to continue operation with very little refinement.

The load characterisation system is not as far developed as the load profiling. However, the concept to define the load characteristics and load scheduling is at a working state. The load characterisation process, defines the loads into categories and groups. Firstly into their main categories, which are constant loads, free loads and cyclic production loads and secondly into there load groups. The load groups are define from the areas of the mine, for example the face machinery, the development machinery or the pit bottom machinery. At this point the system is still considered robust and requiring little or no further development.

It is the introduction of the LC MATRIX and the load scheduling that requires further work. However, that is not to say that the LC MATRIX does not work, because it does. Once the load profiles and characteristics are defined, further load constraints are applied. These depend on how the load operates within the system and are as follows: unavoidable or avoidable, dependent or independent, a load is either one or the other from each pair. For example, unavoidable and independent or avoidable and dependent or any other combination. A MATLAB Simulink model was developed to test the LC MATRIX, test data was used and the LC MATRIX performed as expected. However, it was considered that the rules could be refined or added to. The whole system including the LC MATRIX was then applied to the live production data, with success. Optimising the face and hoist load profile groups it was possible to reduce the peak demand by 3MW and the working demand by 100kW. This is a significant reduction, especially the peak demand as this could translate into a minimum saving of £2000 per month.

The LC MATRIX scheduling is off-line, although the author has also developed a set of real-time algorithms. Although these have not been tested, they are considered to be unique and should be further developed by the author.

The system in this project is not fully tested in an industrial situation. However, the work presented in this thesis has demonstrated an in depth understanding of electrical demand of mine processes. This has been highlighted with the design and installation of the data acquisition system, development of the load profiling systems, and testing of the load scheduling concept. This thesis has combined these individual tasks to provide a complete demand reduction and responsive strategy system.

## 9.1 Further Work

To increase the industrial impact of the research, the author would like to extend the research into the development of a forecasting and demand optimisation model for dynamic pricing. This would allow mine power systems to be fully integrated into the smart grid. It is considered critical to perform an analysis of all the factors affecting electricity tariffs, particularly the factors influencing the price of electricity. To identify the external factors that may affect the price of electricity, the real time electricity market and electricity supply industries would be pro-actively analysed. This would be used to predict the changes in the energy utilities and renewable energy industries, and how these changes would have an impact on the supply side costs and the existing tariff structure. The further research would forecast the electrical demand of a mine several hours and days in advanced, providing the mine operator a pro-active tool for real-time "advisory decision support".

The further research would take the initial step to build a further model. This model would be of a complete mine power system that operated in real-time; the real-time decision scheduler described in this thesis requires modelling to debug and develop it further. It is at the point where only a model of the system will allow the project to progress. The power system model will also allow the further development of the off-line scheduling that is required to realise the RTP response system.

Other further work that fits into this project would be to use Mixed Integer Programming (MIP) to develop the LC MATRIX. Christodoulos and Floudas (2005) use MIP for process scheduling and Carvalho & Millar (2012) use MIP in mine site energy supply optimisation. MIP has been widely used in production planning, sequencing processes, distribution and logistics problems, refinery planning, power plant scheduling. The nature of the model and that the loads are defined as discrete fixed operations and that the demand reduction problem is a load scheduling task. The optimization technique best suited to the demand reduction problem is MIP.

# 9. CONCLUSIONS AND RECOMMENDATIONS

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# Appendix A

# MATLAB and SQL Code

### Data Time Space Correction

```
/* EqualSpacingTimeData_Power.sql This file queries the OPC Logger database and equally
spaces the data */
USE [Underground_090312_1200_7Days]

DECLARE @tableName nvarchar(50)
DECLARE @tableIncrement int
DECLARE @firstDate dateTime
DECLARE @newDate dateTime
DECLARE @nextDate dateTime
DECLARE @lastDate dateTime
DECLARE @firstDateQuery nvarchar(max)
DECLARE @nextDateQuery nvarchar(max)
DECLARE @lastDateQuery nvarchar(max)
DECLARE @dateDiff float
DECLARE @newDateDiff float
DECLARE @diffRatio float
DECLARE @multiplyer int
DECLARE @dateIncrement int
DECLARE @numColumns int
DECLARE @columnsCount int
DECLARE @numColumnsQuery nvarchar(max)
DECLARE @columnNameQuery nvarchar(max)
DECLARE @columnName nvarchar(25)
DECLARE @firstValueQuery nvarchar(max)
DECLARE @firstValue int
DECLARE @nextValueQuery nvarchar(max)
DECLARE @nextValue int
DECLARE @newValueQuery nvarchar(max)
DECLARE @newValue int
DECLARE @newValueInsertQuery nvarchar(max)
DECLARE @newValueUpdateQuery nvarchar(max)
DECLARE @columnNameList nvarchar(max)
DECLARE @newValueList nvarchar(max)

/* column list */
SET @columnNameList =
',kWh,kVAh,kvarh_ind,kvarh_cap,imp_kvarh,exp_kWh,exp_kvarh,kW,kVA,kvar,PF,Hz,V1,A1,kW1,V2,A2,k
W2,V3,A3,kW3,PF1,PF2,PF3,V12,V23,V31,kVA1,kVA2,kVA3,kvar1,kvar2,kvar3,V1THD,V2THD,V3THD,A1THD,
A2THD,A3THD'

SET NOCOUNT ON
SET @tableIncrement = 1


WHILE @tableIncrement < 4
BEGIN
    /* SET @tableName */
    IF @tableIncrement = 1 SET @tableName ='ID11'
    IF @tableIncrement = 2 SET @tableName ='ID21'
    IF @tableIncrement = 3 SET @tableName ='ID61'
    IF @tableIncrement = 4 SET @tableName ='ID63'
    IF @tableIncrement = 4 SET @tableName ='ID71'
    IF @tableIncrement = 4 SET @tableName ='ID72'
    IF @tableIncrement = 4 SET @tableName ='ID81'
    IF @tableIncrement = 4 SET @tableName ='ID82'

    SET @multiplyer = 1
    SET @dateIncrement = 500 -- milliseconds
```

### Data Time Space Correction

```
SET @numColumns = 39

/* Get the first dateTime stamp from @tableName */
SET @firstDateQuery = N'SELECT @firstDate = MIN(dateTime) FROM '+@tableName+''
--PRINT @firstDateQuery
EXEC SP_EXECUTESQL @firstDateQuery, N'@firstDate dateTime OUTPUT', @firstDate OUTPUT

SET @newDate = DATEADD(ms, @dateIncrement, DATEADD(ms, -DATEPART(ms, @firstDate), @
firstDate))

/* make sure the @newDate is greater than the @firstDate*/
WHILE @newDate < @firstDate
BEGIN
    SET @newDate = DATEADD(ms, @dateIncrement*@multiplyer, DATEADD(ms, -DATEPART(ms, @
    firstDate), @firstDate))
    SET @multiplyer = @multiplyer + 1
END

/* Get the last dateTime stamp from @tableName */
SET @lastDateQuery = N'SELECT @lastDate = MAX(dateTime) FROM '+@tableName+''
--PRINT @firstDateQuery
EXEC SP_EXECUTESQL @lastDateQuery, N'@lastDate dateTime OUTPUT', @lastDate OUTPUT

/* create new dates all through the database date range getting the  */
WHILE @newDate < @lastDate
BEGIN
    /* Get the next dateTime greater than the next @newDate stamp from @tableName */
    SET @firstDateQuery = N'SELECT @firstDate = MAX(dateTime) FROM '+@tableName+' WHERE
    dateTime < '''+convert(nvarchar, @newDate, 21)+''''
    --PRINT @firstDateQuery
    EXEC SP_EXECUTESQL @firstDateQuery, N'@firstDate dateTime OUTPUT', @firstDate OUTPUT

    /* Get the next dateTime greater than the next @newDate stamp from @tableName */
    SET @nextDateQuery = N'SELECT @nextDate = MIN(dateTime) FROM '+@tableName+' WHERE
    dateTime > '''+convert(nvarchar, @newDate, 21)+''''
    --PRINT @firstDateQuery
    EXEC SP_EXECUTESQL @nextDateQuery, N'@nextDate dateTime OUTPUT', @nextDate OUTPUT

    /* difference between the two dates either side of the @newDate */
    SET @dateDiff = DATEDIFF(ms, @firstDate, @nextDate)

    /* difference between the @newDate and the @firstDate */
    SET @newDateDiff = DATEDIFF(ms, @firstDate, @newDate)

    --PRINT 'First Date : '+convert(nvarchar, @firstDate, 21)

    --PRINT 'New Date : '+convert(nvarchar, @newDate, 21)

    --PRINT 'New Diff ms :'+convert(nvarchar, @newDateDiff)

    --PRINT 'Diff ms :'+convert(nvarchar, @dateDiff)

    SET @diffRatio = @newDateDiff/@dateDiff

    --PRINT 'Diff Ratio : '+convert(nvarchar, @diffRatio)

    --PRINT 'Next Date : '+convert(nvarchar, @nextDate, 21)
```

```
/* next new date */
SET @newDate = DATEADD(ms, @dateIncrement, @newDate)

/* Get the number of columns in table */
--SET @numColumnsQuery = N'SELECT @numColumns = MAX(ordinal_position) FROM
information_schema.columns WHERE table_name = '''+@tableName+''''
--EXEC SP_EXECUTESQL @numColumnsQuery, N'@numColumns int OUTPUT', @numColumns OUTPUT
--PRINT @numColumns

--SELECT column_name FROM information_schema.columns WHERE table_name = @tableName
and ordinal_position = 3

SET @columnsCount = 1
--SET @columnNameList = ''
SET @newValueList = ''

WHILE @columnsCount <= @numColumns
BEGIN
    /* Get the name of column in table */
    --SET @columnNameQuery = N'SELECT @columnName = column_name FROM
    information_schema.columns WHERE table_name = '''+@tableName+''' AND
    ordinal_position = '+convert(nvarchar, @columnsCount)+''
    --PRINT @columnNameQuery
    --EXEC SP_EXECUTESQL @columnNameQuery, N'@columnName nvarchar(25) OUTPUT',
    @columnName OUTPUT
    --PRINT @columnName

    /* SET columName */
    IF @columnsCount = 1 SET @columnName = 'kWh'
    IF @columnsCount = 2 SET @columnName = 'kVAh'
    IF @columnsCount = 3 SET @columnName = 'kvarh_ind'
    IF @columnsCount = 4 SET @columnName = 'kvarh_cap'
    IF @columnsCount = 5 SET @columnName = 'imp_kvarh'
    IF @columnsCount = 6 SET @columnName = 'exp_kWh'
    IF @columnsCount = 7 SET @columnName = 'exp_kvarh'
    IF @columnsCount = 8 SET @columnName = 'kW'
    IF @columnsCount = 9 SET @columnName = 'kVA'
    IF @columnsCount = 10 SET @columnName = 'kvar'
    IF @columnsCount = 11 SET @columnName = 'PF'
    IF @columnsCount = 12 SET @columnName = 'Hz'
    IF @columnsCount = 13 SET @columnName = 'V1'
    IF @columnsCount = 14 SET @columnName = 'A1'
    IF @columnsCount = 15 SET @columnName = 'kW1'
    IF @columnsCount = 16 SET @columnName = 'V2'
    IF @columnsCount = 17 SET @columnName = 'A2'
    IF @columnsCount = 18 SET @columnName = 'kW2'
    IF @columnsCount = 19 SET @columnName = 'V3'
    IF @columnsCount = 20 SET @columnName = 'A3'
    IF @columnsCount = 21 SET @columnName = 'kW3'
    IF @columnsCount = 22 SET @columnName = 'PF1'
    IF @columnsCount = 23 SET @columnName = 'PF2'
    IF @columnsCount = 24 SET @columnName = 'PF3'
    IF @columnsCount = 25 SET @columnName = 'V12'
    IF @columnsCount = 26 SET @columnName = 'V23'
    IF @columnsCount = 27 SET @columnName = 'V31'
    IF @columnsCount = 28 SET @columnName = 'kVA1'
    IF @columnsCount = 29 SET @columnName = 'kVA2'
    IF @columnsCount = 30 SET @columnName = 'kVA3'
```

```
            IF @columnsCount = 31 SET @columnName = 'kvar1'
            IF @columnsCount = 32 SET @columnName = 'kvar2'
            IF @columnsCount = 33 SET @columnName = 'kvar3'
            IF @columnsCount = 34 SET @columnName = 'V1THD'
            IF @columnsCount = 35 SET @columnName = 'V2THD'
            IF @columnsCount = 36 SET @columnName = 'V3THD'
            IF @columnsCount = 37 SET @columnName = 'A1THD'
            IF @columnsCount = 38 SET @columnName = 'A2THD'
            IF @columnsCount = 39 SET @columnName = 'A3THD'

            /* column list */
            --SET @columnNameList = @columnNameList+','+@columnName

            /* Get the first value of column in table */
            SET @firstValueQuery = N'SELECT @firstValue = '+@columnName+' FROM '+@tableName+
            ' WHERE dateTime = '''+convert(nvarchar, @firstDate, 21)+''''
            --PRINT @firstValueQuery
            EXEC SP_EXECUTESQL @firstValueQuery, N'@firstValue int OUTPUT', @firstValue OUTPUT
            --PRINT @firstValue

            /* Get the next value of column in table */
            SET @nextValueQuery = N'SELECT @nextValue = '+@columnName+' FROM '+@tableName+'
            WHERE dateTime = '''+convert(nvarchar, @nextDate, 21)+''''
            --PRINT @nextValueQuery
            EXEC SP_EXECUTESQL @nextValueQuery, N'@nextValue int OUTPUT', @nextValue OUTPUT
            --PRINT @nextValue

            SET @newValue = ((@nextValue-@firstValue)*@diffRatio)+@firstValue
            --PRINT @newValue

            /* column list */
            SET @newValueList = @newValueList+','+convert(nvarchar, @newValue)

            /* UPDATE @newDate Row with @newValue for @coulmnName */
            --SET @newValueUpdateQuery = N'UPDATE '+@tableName+'EQ SET ['+@columnName+'] =
            '+convert(nvarchar, @newValue)+' WHERE [dateTime] = '''+convert(nvarchar,
            @newDate, 21)+''''
            --PRINT @newValueUpdateQuery
            --EXEC SP_EXECUTESQL @newValueUpdateQuery

            /* Next Column */
            SET @columnsCount = @columnsCount + 1

        END
        --PRINT @newValueList
        --PRINT @columnNameList
        /* INSERT New date into @tableName.EQ with NULL row, ready for UPDATES with
        @newValue */
        SET @newValueInsertQuery = N'INSERT INTO '+@tableName+'EQ ([dateTime]'+@
        columnNameList+') VALUES ('''+convert(nvarchar, @newDate, 21)+''''+@newValueList+')'
        --PRINT @newValueInsertQuery
        EXEC SP_EXECUTESQL @newValueInsertQuery

    END

SET @tableIncrement = @tableIncrement + 1
END
```

**MATLAB Workspace Variables**

```matlab
% AveProPower4n10MVA_SQL.m This file queries the database though MATLAB's database module
and puts the data into workspace variables %

% Set preferences with setdbprefs.
s.DataReturnFormat = 'cellarray';
s.ErrorHandling = 'store';
s.NullNumberRead = 'NaN';
s.NullNumberWrite = 'NaN';
s.NullStringRead = 'null';
s.NullStringWrite = 'null';
s.JDBCDataSourceFile = 'D:\PhD\LoadProfiling\090312_7Day\JDBC.mat';
s.UseRegistryForSources = 'yes';
s.TempDirForRegistryOutput = 'C:\Users\nick\AppData\Local\Temp';
s.DefaultRowPreFetch = '10000';
setdbprefs(s)

%date for query 24/07/2012  10.8    108.00          5816    6.3 4.3
sqlDateStart = '2012-07-24 22:00:00';
sqlDateEnd = '2012-07-24 23:00:00';
preName = 'AvePro_1hr_';


% 'Exp_23', 'Methane_Generators_11kv_415v',
meterChannels = {'61','63','71','72','81','82'};
meterNames = {'No1_Incomer_4MVA','No5_Incomer_10MVA','No4_Incomer_10MVA','No5_Shaft_Feeder',
'No2_Incomer_4MVA','No4_Shaft_Feeder'};

% Make connection to database.  Note that the password has been omitted.
% KEHistory26Feb28Days KEHistory22April28Days KEHistory01July28Days KEHistory26Sept54Days
% Using JDBC driver.
conn = database('KEHistory01July28Days','admin','adm1n',
'com.microsoft.sqlserver.jdbc.SQLServerDriver','jdbc:
sqlserver://localhost:1433;database=KEHistory01July28Days');


% Read data from database.
e = exec(conn,sprintf('SELECT dateTime FROM historyTable_Cube400_wide WHERE dateTime BETWEEN
''%s'' AND ''%s''',sqlDateStart,sqlDateEnd));
e = fetch(e);
close(e)

% Assign date data to output variable.
v = genvarname([preName 'DateNum4n10MVA']);
eval ([v '= datenum(e.Data)']);
%create xAxis 1-(number of records) makes it easy to plot the results

% Assign date data to output variable.
v = genvarname([preName 'Date4n10MVA']);
eval ([v '= e.Data']);

% Assign date data to output variable.
w = genvarname([preName 'xAxis4n10MVA']);
eval ([w '= 1:numel(eval (v))']);


for i= 1:numel(meterChannels),

    e = exec(conn,sprintf('SELECT Cube400_KW_%s FROM historyTable_Cube400_wide WHERE
```

```
    dateTime BETWEEN ''%s'' AND ''%s''',meterChannels{:,i},sqlDateStart,sqlDateEnd));
    e = fetch(e);
    close(e)

    % Assign data to output variable.
    % generate Var name with channel array
    v = genvarname([preName meterNames{:,i}]);
    eval ([v '= cell2mat(e.Data)']);
    %display(sprintf('SELECT Cube400_KW_%s FROM historyTable_Cube400_wide WHERE dateTime
    BETWEEN ''%s'' AND ''%s''',meterChannels{:,i},sqlDateStart,sqlDateEnd));

end

%4MVA_Total
% Read data from database.
e = exec(conn,sprintf('SELECT (Cube400_KW_61 + Cube400_KW_81) AS [4MVA_Total] FROM
historyTable_Cube400_wide WHERE dateTime BETWEEN ''%s'' AND ''%s''',sqlDateStart,sqlDateEnd));
e = fetch(e);
close(e)

% Assign date data to output variable.
v = genvarname([preName '4MVA_Total']);
eval ([v '= cell2mat(e.Data)']);

%10MVA_Total
% Read data from database.
e = exec(conn,sprintf('SELECT (Cube400_KW_72 + Cube400_KW_82) AS [10MVA_Total] FROM
historyTable_Cube400_wide WHERE dateTime BETWEEN ''%s'' AND ''%s''',sqlDateStart,sqlDateEnd));
e = fetch(e);
close(e)

% Assign date data to output variable.
v = genvarname([preName '10MVA_Total']);
eval ([v '= cell2mat(e.Data)']);

%Methane_Pumping
% Read data from database.
e = exec(conn,sprintf('SELECT ((Cube400_KW_63 + Cube400_KW_71)-(Cube400_KW_72 +
Cube400_KW_82)) AS Methane_Pumping FROM historyTable_Cube400_wide WHERE dateTime BETWEEN
''%s'' AND ''%s''',sqlDateStart,sqlDateEnd));
e = fetch(e);
close(e)

% Assign date data to output variable.
v = genvarname([preName 'Methane_Pumping']);
eval ([v '= cell2mat(e.Data)']);

% Close database connection.
close(conn)
```

## Load Profiling

```
% LP_Basic_Solve.m This file load profiles individual loads %

% Get variable size
numLoads = 6;
% Create time array
t = 0:0.5:((numel(Load1)/2)-0.5);

% Create a array or all ones, this is the base load as this is alway on
Load0 = ones(numel(Load1),1);

% Create loads list
loadList = 'Load0';
for i = 1:numLoads,
    % Generate load varable list for coefficients matrix
    loadList = sprintf('%s Load%d', loadList, i);
    % fprintf('%s\n', v);
    % loadList = genvarname([loadList 'Load' sprintf('%d\n', i)]);
end

loadsCoef = eval(sprintf('[%s]', loadList));

% Create coefficients matrix for simultaneous equation solving
% loadsCoef = eval ([v '= datenum(e.Data)']);
% fprintf('%d \n', loadsCoef(:,1:5));
% fprintf(repmat('%14.0f',1,numLoads), loadsCoef(:,1:5));
% Solve linear simultaneous equation
% Using Answer Matrix / Coefficients Matrix
% Unknowns = Answer Matrix / Coefficients Matrix

% Ax = b
% A inv(A) x = b inv(A)
% b inv(A) = x
% inv(loadsCoef) = LoadValues
% LoadValues = loadsCoef\Power415V;

LoadValues = loadsCoef\Power415V;
% LoadValues_LU = loadsCoef\Power415V_LU;
fprintf('%s ', loadList);
fprintf('\n');
fprintf('%d ', LoadValues);
fprintf('\n');

% Maximum demand
maxDemand = max(Power415V);
fprintf('Maxiumu Demand ');
fprintf('%d ', maxDemand);

for i = 0:numLoads,
    % Generate load variable name
    u = (['Load' sprintf('%d',i)]);
    v = genvarname(['load' sprintf('%d',i) 'Time']);
    % fprintf('%s %s\n',u,v);

    % Get the indicies when the load is on, load$Time
    evalc ([v '= 0.5*numel(find(eval(sprintf(''[%s]'', u))))']);
end
```

### LC MATRIX Demand Reduction

```
% demandReductionSTD.m This file uses data from LP_Basic and reduce the demand %

% create an array of zeros
loadProfileOP = zeros(size(loadsCoef, 1),numLoads);

%get ALoads using LC_Matrix
aLoads = ALoads(LC_Matrix, LoadValues);
aLoadsDemand = 0;
% loop through aLoads and place the operation profiles in loadProfileOP matrix
if isempty(aLoads)==0
    for i = 1:size(aLoads,1),
       % put the loadProfiles into the location of the load id
       % (+1 due to base load)
       loadProfileOP(:,aLoads(i,1)) = loadsCoef(:,aLoads(i,1)+1);
       % total demand for aLoads
       aLoadsDemand = aLoadsDemand + LoadValues(aLoads(i,1)+1);
    end
end

%get BLoads using LC_Matrix
bLoads = BLoads(LC_Matrix, LoadValues);

% loop through bLoads and place the operation profiles in loadProfileOP matrix
if isempty(bLoads)==0
    for i = 1:size(bLoads,1),
        % get the length time before the first load operation
            % (+1 due to base load)
        bfTime = find(loadsCoef(:,bLoads(i,1)+1), 1, 'first')-1;
        % get the length time after the last load operation
            % (+1 due to base load)
        lsTime = find(loadsCoef(:,bLoads(i,1)+1), 1, 'last')+1;
        % total time to shift profile
        shiftTime = size(loadsCoef,1) - lsTime + bfTime;
        % check that there is room to shift bLoad
        if shiftTime > 0
            % move the bLoad to the start in loadProfileOP
            % (+1 due to base load)
            tempLP = loadsCoef(:,bLoads(i,1)+1);
            % remove the first lot of zeros and check demand this the start @ 1
            tempLP(1:bfTime-1) = [];
            % add the same amout of zeros to the end to keep the matrix the
            % same size
            tempLP(size(tempLP,1)+1:size(loadsCoef,1)) = 0;
            % temp file for mode average demand, the lowest value is the best
            % position for the profile
            demandTemp = zeros(shiftTime,3);
            % create temp file to shift
            tempLPs = loadProfileOP;
            tempLPs(:,bLoads(i,1)) = tempLP;
            % loop through and shift the loadProfile for the bLoad for the
            % lowest demand
            for j = 1:shiftTime,
                % create temp zero var for shift demand
                shiftDemandTemp = zeros(size(loadsCoef,1),1);
                % loop through load profile and store demand for each time slot
                for k = 1:size(tempLPs,1),
                    % get ids of loads with demand for each row.
                    rowLoadIds = find(tempLPs(k,:));
```

```
                % loop through ids and add up demand
                if rowLoadIds~=0,
                    for l = 1:size(rowLoadIds,1),
                        % (+1 due to base load)
                        shiftDemandTemp(k) = shiftDemandTemp(k) + LoadValues(rowLoadIds(l
                        )+1);
                    end
                end
            end
            % store the standard deviation of the demand
            demandTemp(j,3) = std(shiftDemandTemp);
            % store the mode of the demand for the current shift and
            % the shift id so that this shift can be used
            % change 0 in nan so min picks up the none zero minimum
            shiftDemandTemp(~shiftDemandTemp) = nan;
            % store the unique min
            demandTemp(j,1) = min(unique(shiftDemandTemp));
            % and the number of occurrences, this is to get the lowest
            % demand
            demandTemp(j,2) = numel(find(shiftDemandTemp==min(unique(shiftDemandTemp))));
            % shift the tempLP to the next position
            % tempTemp var
            tempTempLP = zeros(size(tempLP,1),1);
            % loop through positions
            for m = 1:size(tempLP,1)-1,
                % shit the position by 1
                tempTempLP(m+1) = tempLP(m,1);
            end
            % copy tempTempLP to orginial tempLP now it has been
            % shifted
            tempLP = tempTempLP;

        end
        % select the minimum demand of the shifts and use that shift
        % location for the bLoad
        % all the ids that have the minimum standard deviation
        shiftIDs = find(demandTemp==min(demandTemp(:,3)));
        % get the first min standard deviation of the demand
        shiftID = shiftIDs(find(demandTemp(shiftIDs,3)==max(demandTemp(:,3)), 1, 'first'
        ));
        % get load profile again as it has been shifted we know the
        % loaction, just shift it to the selected shift
        tempLP = loadsCoef(:,bLoads(i,1)+1);
        % shift the bLoad profile to the position of lowest demand
        % remove zereos from before
        tempLP(1:bfTime-1-shiftID) = [];
        % add zeros onto the end
        tempLP(size(tempLP,1)+1:size(loadsCoef,1)) = 0;
        % and apply it to the loadProfileOP
        loadProfileOP(:,bLoads(i,1)) = tempLP;
    else
        % if there is no space to shift bLoad place in profile as is
        % (+1 due to base load)
        loadProfileOP(:,bLoads(i,1)) = loadsCoef(:,bLoads(i,1)+1);
    end
```

```
        % get valleys for bLoads, the aLoad that is the closest to the
        % bLoad demand
%          bLoadValleyIDs = min(aLoads(bLoads(i,3)<aLoads(:,3),3)));
        % postion the bLoad sequence start

    end
end

%get CLoads using LC_Matrix
cLoads = CLoads(LC_Matrix, LoadValues);

% loop through cLoads and place the operation profiles in loadProfileOP matrix
if isempty(cLoads)==0
    for i = 1:size(cLoads,1),
        % get cLoads LoadBlocks in order
        % Generate load variable name
        u = (['Load' sprintf('%d',cLoads(i,1))]);
        % debug: fprintf('%s %s\n',u,v);
        % get the blocks for the cLoad(i)
        loadBlocks = LoadBlocks(eval(u));
        % get total time for profile
        profileSize = size(eval(u),1);
        % add all the block sizes together
        totalBlock = sum(loadBlocks(:,2));
        % get the space that the block can move into
        % (total space - total block space)
        % / number of blocks = block space
        blockSpace = floor((profileSize - totalBlock)/size(loadBlocks,1));
        fprintf('%d \n',blockSpace); % debug:
        % create temp empty profile
        tempLP = zeros(size(eval(u),1),1);
        % create temp empty final profile
        tempLPFinal = zeros(size(eval(u),1),1);
        % create temp system prolfile
        tempLPs = loadProfileOP;
        % set start position
        startPosition = 1;
        % set the blocks so far
        blocksPlaced = 0;
        % loop through cLoad Blocks and position them all in a optimised
        % profile
        for j = 1:size(loadBlocks,1),
            % get the load block window
            % add all the block sizes together minus the blocks that have been
            blockTotal = sum(loadBlocks(j:size(loadBlocks,1),2));
            % get the end position of the last block
            startPosition = find(tempLP==1,1,'last')+1;
            % get the end position of the block space
            lastPosition = blocksPlaced + (blockSpace * j)+1;
            % if the start position is empty, it's the first block
            if isempty(startPosition)==1
                startPosition = 1;
            end
            fprintf('Start %d \n',startPosition); % debug:
            % calculate block positions
            blockPositions = lastPosition - startPosition;
            fprintf('%d \n',blockPositions); % debug:
```

```
        % get current block size
        blockSize = loadBlocks(j,2);
        % put the block onto the temp profile at the start of the space
        tempLP(startPosition:startPosition+blockSize-1,1) = 1;
        % add profile to temp load profiles so that this can be used to
        % calculate the demand
        tempLPs(:,cLoads(i,1)) = tempLP;
        % temp file for shift position demand min and max
        demandTemp = zeros(blockPositions,5);
        % loop through the block positions
        for k = 1:blockPositions,
            % create temp zero var for shift demand
            shiftDemandTemp = zeros(blockSize,1);
            % loop through block size and store demand for each position along the block
            for l = 1:blockSize,
                % get ids of loads with demand for each row.
                rowLoadIds = find(tempLPs(startPosition-1+k+l,:));
                % loop through ids and add up demand
                if rowLoadIds~=0,
                    for m = 1:size(rowLoadIds,2),
                        % (+1 due to base load)
                        shiftDemandTemp(l) = shiftDemandTemp(l) + LoadValues(rowLoadIds(m
                        )+1);
                    end
                end
            end
            % store the min and max demand (using function ShiftDemand)


            %ShiftDemand = zeros(1,4);
            % store the standard deviation of the demand
            demandTemp(k,5) = std(shiftDemandTemp);
            % store the mode of the demand for the current shift and
            % the shift id so that this shift can be used
            % change 0 in nan so min picks up the none zero minimum
            shiftDemandTemp(~shiftDemandTemp) = nan;
            % store the minimum shift demands
            demandTemp(k,1) = min(unique(shiftDemandTemp));
            % and the number of occurrences, this is to get the lowest demand
            demandTemp(k,2) = numel(find(shiftDemandTemp==min(unique(shiftDemandTemp))));
             % store the maximum shift demands
            demandTemp(k,3) = max(unique(shiftDemandTemp));
            % and the number of occurrences, this is to get the highest demand
            demandTemp(k,4) = numel(find(shiftDemandTemp==max(unique(shiftDemandTemp))));

            % xyzxx = ShiftDemand(shiftDemandTemp);
            %demandTemp(k,:) = ShiftDemand;
            % shift the tempLP to the next position
            % by putting a zero at the beginning and a one at the end of
            % the block
            % zero at the beginning
            tempLP(startPosition+k-1) = 0;
            % one at the end
            tempLP(startPosition+blockSize+k-1) = 1;
        end

        % add up the blocks so far
        blocksPlaced = blocksPlaced + blockSize;
```

```
            % now select the best position for the block and position it on
            % the profile, then move to the next block until all blocks
            % have been positioned

            % select the minimum demand of the shifts and use that shift
            % location for the cLoad loadBloack
            % all the ids that have the minimum demand
            shiftIDs = find(demandTemp(:,5)==min(demandTemp(:,5)));
            % get the first max occurrences
            shiftID = shiftIDs(find(demandTemp(shiftIDs,5)==max(demandTemp(:,5)), 1, 'first'
            ));
            % get load profile again as it has been shifted we know the
            % location, just shift it to the selected shift put 1's for the
            % block from best shift position to the end of the block
            tempLP(startPosition+shiftID:startPosition+shiftID+blockSize-1,1) = 1;
            % then make any locations after that 0's
            tempLP(startPosition+shiftID+blockSize:end,1) = 0;

        end
        % and apply it to the loadProfileOP
        loadProfileOP(:,cLoads(i,1)) = tempLP;
    end
end

%get CLoads using LC_Matrix
dLoads = DLoads(LC_Matrix, LoadValues);

% loop through cLoads and place the operation profiles in loadProfileOP matrix
if isempty(dLoads)==0
    for i = 1:size(dLoads,1),
        % get cLoads LoadBlocks in order
        % Generate load variable name
        u = (['Load' sprintf('%d',dLoads(i,1))]);
        % debug: fprintf('%s %s\n',u,v);
        % get the blocks for the dLoad(i)
        loadBlocks = LoadBlocks(eval(u));
        % get total time for profile
        profileSize = size(eval(u),1);
        % add all the block sizes together
        totalBlock = sum(loadBlocks(:,2));
        % create temp empty profile
        tempLP = zeros(size(eval(u),1),1);
        % create temp empty final profile
        tempLPFinal = zeros(size(eval(u),1),1);
        % create temp system prolfile
        tempLPs = loadProfileOP;
        % while loop until all the load is profiled
        blocksRemaining = totalBlock;
        % add all the load blocks to the profile, the deepest valleys first
        % then the next deepest and so on.
        while blocksRemaining > 0
            % create temp zero var for shift demand
            shiftDemandTemp = zeros(profileSize,1);
            % loop through load profile size and store demand for each position
            % along the profile, this get the profile before the dLoad is
            % applied and we then get the deepest valleys
            for j = 1:profileSize,
                % get ids of loads with demand for each row.
```

```
                rowLoadIds = find(tempLPs(j,:));
                % loop through ids and add up demand
                if rowLoadIds~=0,
                    for k = 1:size(rowLoadIds,2),
                        % (+1 due to base load)
                        shiftDemandTemp(j) = shiftDemandTemp(j) + LoadValues(rowLoadIds(k)+1);
                    end
                end
            end
            % while loop until all the load blocks have been allocated
            % get the lowest demand for the current profile
            minDemand = min(unique(shiftDemandTemp));
            % get valleys for the lowest demand
            %valleyIDs = LoadValleys(shiftDemandTemp,minDemand);
            valleyIDs = find(shiftDemandTemp==minDemand);
            % fill the valley with the load, check for the blocks remaining
            % check that there is more load blocks than valleys
            if blocksRemaining > size(valleyIDs,1)
                % fill the valleyIDs with 1's
                tempLP(valleyIDs(:),1) = 1;
                blocksRemaining = blocksRemaining - size(valleyIDs,1);
            else
                % where there is less blocks than valleys just put the load
                % in the first available valley
                % fill the valleyIDs with 1's
                tempLP(valleyIDs(1:blocksRemaining),1) = 1;
                blocksRemaining = blocksRemaining - blocksRemaining;
            end
            % change the row that have been already uesed to nan on the
            % tempLPs so the valleys do not repeat
            tempLPs(valleyIDs,:) = nan;
        end
        % add temp profile to loadProfileOP
        loadProfileOP(:,dLoads(i,1)) = tempLP;
    end
end
```

# Appendix B

# Questionnaire Data

## Mine Basics

| | | |
|---|---|---|
| Name and Location of the mine? | | Halemba-Wirek<br>Ruda Śląska |
| How deep is the mine? | m | 936 |
| How far is the furthest point form the shaft? | m | 4100 |
| What is the mining method? | | Longwall retreat |

## Production

| | | |
|---|---|---|
| What is the average coal production? | tonnes/day | 7459 |
| How many working faces does the mine have? | | 4 |
| What size are the faces? | Length (m) | 279 |
| | Height (m) | 2.66 |
| How many tonnes of coal are cut each pass? | tonnes/pass | 768 |
| On average how long does one pass take? | hours | 1 |
| What is the cost per tonne? | £ | 69 |

## Development

| | | |
|---|---|---|
| How many faces are under development? | | 2-3 |
| How many tunnels are under development at one time? | | 11-14 |
| What is the size of the tunnels? | Height: | 3.8 |
| | Width: | 4.5 |
| How far does a tunnel advance every 24 hours? | | 38 |

## Transportation

| | | |
|---|---|---|
| How long is the underground conveyor system? | m | 14030 |
| How many conveyors are there in the system? | | 37 |
| What distance does the longest belt cover? | m | |
| Is there any other transport underground, If so what, and how far for? | | Locos 4600m |
| Is the coal hoisted? | | |
| **Yes** | | |
| How many skips/winders are there? | | 4 |
| What is the capacity of each skip? | tonnes | 2x20, 2x30 |
| What is the maximum through put of the hoisting system? | tonnes/day | 22000 |
| **No** | | |
| Is the coal conveyed out? | | |
| What is the maximum through put of the conveyor system? | tonnes/hr | |

## Pumping

| | | |
|---|---|---|
| How much water is pumped from underground to surface per day? | m3/day | 15120 |
| What power rating is the underground to surface pump? | kW | |
| What is the pumping head for this pump? | m | 380, 525, 1030 |
| How many hours per day does the pump run? | hours/day | 20 |

## Ventilation

| | | |
|---|---|---|
| What is the operational ventilation power for surface fans? | kW | 4100 |
| and underground booster fans? | kW | |

## Power Consumption

| | | |
|---|---|---|
| How much power does the mine require? | MWh/year | 232426 |
| Average Peak Production Non Productive | Peak/Non % | 81.6/18.4 |
| What is the average energy consumption per day? | MWh/day | 636.7835616 |

## Heat Energy

| | | |
|---|---|---|
| Thermal enrgy fo heating shafts | GJ | 80929 |

| | | | | | | |
|---|---|---|---|---|---|---|
| **Poland** | | | | | | |
| Knurów-Szczygłowice | Ziemowit | Aragón 1 | Aragón 2 | Asturias 1 | Asturias 2 | Leon 1 |
| Knurów | Lędziny | | | | | |
| 896 | 701 | 400 | 225 | 300 | 250 | 400 |
| 9000 | 7200 | 3000 | 3500 | 5000 | 2000 | 3000 |
| Longwall retreat | Longwall retreat | Sutirage | Room-and-Pillar | ngwall retre | Longwall retreat | Sublevel caving |
| 15773 | 17483 | 1095.890411 | 246.5753425 | 1095.89 | 630.1369863 | 821.9178082 |
| 6 | 4 | 5 | 1 | 1 | 1 | 20 |
| 246 | 173 | 150 | 2000 | 180 | 90 | 22 |
| 2.5 | 2.92 | 20 | 5.5 | 2.2 | 3.5 | 12 |
| 641 | 524 | 200 | 225 | 420 | 700 | 800 |
| 1 | 1 | 24 | 24 | 12 | 24 | 48 |
| 60 | 37 | | 53.95 | | | 99.6 |
| 8-9 | 4 | 5 | 3 | 6 | 2 | 2 |
| 17-19 | 13-14 | 20 | 30 | 20 | 5 | 4 |
| 3.8 | 3.8 | 3 | 5 | 3.2 | 7 | 3.2 |
| 4.5 | 4.5 | 3.2 | 5.5 | 4.5 | 8 | 4.5 |
| 67 | 69 | 20 | 3.5 | 2.4 | 1 | 2.4 |
| 23350 | 18362 | 2000 | 7000 | 3000 | 1500 | 8000 |
| 44 | 39 | 4 | 1 | 8 | 1 | 15 |
| | | 5000 | 7500 | 1000 | 2500 | 1000 |
| Locos 46300 | Locos 18000 | No | No | No | No | No |
| 9 | 4 | | | | | 1 |
| 2x20, 2x15, 4x12.5, 1x30 | 4x30 | | | | | 20 |
| 53700 | 37600 | | | 7200 | | |
| | | Yes | Yes | Yes | Yes | |
| | | 5km | 7.5km | 5km | 2.5km | |
| 17280 | 29016 | 1000 | 0 | 3360 | | 1000 |
| | | 400 | | 4x202 | | 600 |
| 550, 650, 850, 650, 850 | 200, 500, 650 | 400 | | 260 | 250 | 420 |
| 15, 22, 22, 9-12, 9-12 | 11, 20, 24 | 6 | | 8 | | 8 |
| 5270 | 2000 | 400 | 50 | 1x750, 1x65 | 1x200, 1x90 | 500 |
| | | 22 | | 2x15 | 15 | 15 |
| 204975 | 158968 | 18300 | 1104 | 24000 | | 13200 |
| 79.8/20.2 | 80.5/19.5 | ?/70 | | | | 65/35 |
| 561.5753425 | 435.5287671 | 45 | | 65 | | 35 |
| 117425 | 29493 | | | | | |

**Spain**

| Leon 2 | Leon 3 | Leon 4 | SA Hullera Vasco Leonesa | Área Sueros-Mieres-Riosa Principado de Asturias |
|---|---|---|---|---|
| 600 | 149 | 250 | 600 | 570 |
| 4000 | 1500 | 2000 | 4000 | 3000 |
| Sublevel caving | Faces with workers | Plow system | Sublevel caving | Sublevel caving |
| 767.1232877 | 109.5890411 | 246.5753425 | 2191.780822 | |
| 15 | 7 | 20 | 15 | 7 |
| 15 | 125 | 135 | 22 | 450 |
| 12 | 0.9 | 0.9 | 12 | 25 |
| 400 | 135 | 250 | 450 | 400 |
| 24 | 48 | 12 | 24 | 12-18 months |
| 99.6 | 95.45 | 83 | | |
| 8 | 2 | 2 | 15 | 5 |
| 12 | 3 | 4 | 20 | 2 |
| 3.2 | 3 | 3.6 | 3.7 | 3.27 |
| 4.5 | 3.3 | 3.9 | 3.2 | 4.5 |
| 2.4 | 1.5 | 1.5 | 2.4 | 4 |
| 8000 | 1300 | 1000 | 5000 | 6962 |
| 18 | 2 | 5 | | 19 |
| 1000 | 1300 | 2000 | 2000 | 1231 |
| No | No | No | Train | No |
| 1 | | | 1 | 2 |
| 20 | | | | 8 |
| | | | | 2200-2800 |
| | Yes | | | |
| 5km | 130 | 180 | | |
| 1100 | 1200 | 630 | | 6432 |
| 600 | 450 | 750 | | See Comment |
| 450 | 105 | 250 | | See Comment |
| 8 | 3.6 | var | | 35 The Whole System |
| 550 | 150 | 2050 | 2x1200 | 900 |
| 15 | 15 | 15 | 35 | 15, 22, 37 |
| 14400 | 1740 | 6000 | | 14965 |
| 65/35 | 55/45 | ?/6MWh/day | | 80/20 |
| 40 | 14 | 16.5 | | 70 |

| | | | |
|---|---|---|---|
| | | | **Slovenija** |
| Carrio. Pola de Laviana | María Luisa and Sotón Pits | Pozo Santiago, Caborana | Premogovnik Velenje |
| Principado de Asturias | Modesta Area | Principado de Asturias | CMV Velenje |
| 973 | 527.55 and 583.65 | 400 | 458 |
| 3780 | 3000 | 3000 | |
| Longwall advance | Longwall advance and Sublevel caving | Longwall advance and Miner | |
| 700 | | 1000 | 10958.90411 |
| 2 | 1, 5 | 3 | 2 |
| 120 | 500, 450 | 100 | 150 |
| 1.5 | 147, 25 | 2.5 | 4H, 12V |
| 250000 | 400 | 400 | 700H, 2300V |
| 12 months | 12-18 months | 24 | 0.5H, 1.5V |
| 20.75 | | | 25.2984 |
| 2 | 3 | 3 | 1 |
| 2 | THN 450-C and THN 2UA | 3 | 6-8 |
| 3.27 | 3.27, 2.94 | 3.27 | 4.46 |
| 4.5 | 4.5, 3.6 | 4.5 | 4.86 |
| 3 | 2-5 | 2 | Total 30 |
| 3021 | 2500 Belt, 4000 Chain | 10000 | Main 3540, Level 2340 |
| 19 | 7 Belt, 25 Chain | 22 | 8, 5 |
| 295 | 500 | 1500 | 600, 800 |
| Wagons | See Comment | No | |
| | | 1 | |
| | | 19 | |
| | | 1500 | |
| Yes | 2200 litre hoppers | | |
| 1000/day | 1200/day | | 2100 |
| 5010 | | 12000 | 3200 + 1900 + 1075 = 6175 |
| See Comment | See Image to the right | | 900, 400, 400 |
| See Comment Above | | | 630, 280, 280 |
| 20 | See comment | | 8, 24, 5 |
| 240 | 420-1120 | 260 | 450 |
| 75, 45, 22 | 15, 37, 45 | 37, 22 | 2x15, 6x22, 8x24, 4x30 |
| 20572.13 | 16021, 15774 | 31164.47 | 51957 |
| 84/16 | | Peak 5.5MW | 12384kW/5891kW |
| 56.362 | 2.19, 2.175 | 125 | 180 |

**UK**
UK Coal Colliery
Yorkshire
800
8000
Longwall retreat

8000
1
350
2.8
1050
1

| | BOMBAS | | | | | | MOTORES | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | PLANTA | BOMBA nº | MARCAS | MODELO | H (m) | Q m³/h | MARCAS | HP | Rpm | voltaje |
| | 1ª | 3ª | WORTHINGTON | 2 1/2 R-2 | 65 | 75 | TIPO FX 180 MW2 | 30 | 2940 | 380 (Nº3600084) |
| | 1ª | 1-2 | WORTHINGTON | 2 R-2 | 65 | 75 | SIEMENS | 30 | 2900 | 220 |
| | 4ª | 4 | MIGUEL PRADO | AM-150 | 220 | 250 | WESTINGHOUSE | 350 | 1485 | 5000 |
| | 4ª | 2 | EMICA | ETA 125/40 | 240 | 80 | A.E.G | 150 | 1485 | 380 (Nº91-5051 83) |
| | 4ª | 3 | EMICA | ETA 125/40 | 240 | 80 | SIEMENS | 180 | 1485 | 380 |
| | 4ª | 1 | MIGUEL PRADO | AM-175 | 220 | 250 | WESTINGHOUSE | 350 | 1485 | 5000 |
| | 5ª | 1 | WORTHINGTON | 3L2 | 140 | 100 | SIEMENS | 30 | 2950 | 380 (Nº 300844) |
| | 5ª | 2 | WORTHINGTON | 3L2 | 140 | 100 | SIEMENS | 40 | 2945 | 380 (Nº LCL 923949) |
| | 5ª | 3 | FLYGT | BS-2201 | 60 | 144 | FLYGT | | 2940 | 380 |
| | 7ª | 2 | RHUOPUMPEN HIDROTECAR | LE200/470/5 | 400 | 250 | SIEMENS | 570 | 1475 | 5000 |
| | 7ª | 4 | RHUOPUMPEN HIDROTECAR | LE200/470/5 | 400 | 250 | SIEMENS | 570 | 1475 | 5000 |
| | 7ª | 3 | RHUOPUMPEN HIDROTECAR | LE200/470/5 | 400 | 250 | AR 560 | 570 | 1475 | 5000 |
| | 7ª | 1 | RHUOPUMPEN HIDROTECAR | LE200/470/5 | 400 | 250 | A.E.G | 570 | 1492 | 5000 |
| | 10ª | 1 | SULZER | HPL-3720/5 | 198 | 216 | ALCONZA | 381 | 1480 | 5000 |
| | 10ª | 3 | SULZER | HPL-3720/5 | 198 | 288 | ALCONZA | 381 | 1480 | 5000 |
| | 10ª | 5 | FLYGT | BS-2125 | 30 | 15 | FLYGT | 11 | 2800 | 380 |
| | 10ª | 6 | ABS | AFP-1032.1 | 30 | 15 | FLYGT | 11 | 2800 | 380 |
| | SUB10ª | 2 | FLYGT | BS-2400 HT | 150 | 90 | FLYGT | 66 | 2950 | 500 |
| | SUB10ª | 1 | FLYGT | BS-2400 HT | 150 | 90 | FLYGT | 66 | 2950 | 500 |
| POZO MULTIPLE | SUB10ª | 1 | FLYGT | BS-2125.69 | 40 | 15 | FLYGT | 13 | 2800 | 500 |
| POZO MULTIPLE | SUB10ª | 2 | FLYGT | BS-2125.69 | 40 | 15 | FLYGT | 9.7 | 2800 | 500 |

2
4
3.7
5.3
12

| Nº BOMBA | MARCA Y MODELO | POTENCIA | CAUDAL $m^3/h$ | ALTURA VERTIDO |
|---|---|---|---|---|
| 1 | WORTHINGTON 2 1/2 R2 | 22 kW | 70 $m^3/h$ | 60m |
| 2 | SULZER HPL 21-10 | 103 kW | 101 $m^3/h$ | 180m |
| 3 | SULZER HPL 32-17 | 160 kW | 144 $m^3/h$ | 180m |
| 4 | SULZER HPL 32-17 | 160 kW | 180 $m^3/h$ | 180m |
| 5 | WORTHINGTON GL 13 | 160 kW | 369 $m^3/h$ | 125m |
| 6 | WORTHINGTON 6UZD 15 | 510 kW | 400 $m^3/h$ | 315m |
| 7 | WORTHINGTON 6UZD 15 | 510 kW | 400 $m^3/h$ | 315m |
| 8 | SULZER HPL37/20/30-9 | 600 kW | 360 $m^3/h$ | 315m |
| 9 | WORTHINGTON 6UZD 15 | 510 kW | 400 $m^3/h$ | 315m |
| 10 | FLYGT BS-3126-181 | 7,40 kW | 60 $m^3/h$ | 3m |
| 11 | SULZER HPL37/20/30 | 132 kW | 360 $m^3/h$ | 84m |
| 12 | SULZER HPL37/20/30 | 132 kW | 360 $m^3/h$ | 84m |
| 13 | SULZER HPL37/20/30 | 132 kW | 360 $m^3/h$ | 84m |
| 14 | FLYGT BS-3126-181 | 7,40 kW | 60 $m^3/h$ | 25m |
| 15 | FLYGT BS-3126-181 | 7,40 kW | 60 $m^3/h$ | 25m |
| 16 | WORTHINGTON 3UNB-11 | 55 kW | 108 $m^3/h$ | 95m |
| 17 | WORTHINGTON 3UNB-11 | 55 kW | 108 $m^3/h$ | 95m |

7000
5
2500
Locos

2
18-21.5
40 Skips/hour

136.3827564
279.3

5

1790
1150

18.1MW/12.9MW
167.1428571