School of Mining Engineering

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JOHANNESBURG

COAL LOSS AND DILUTION REDUCTION IN OPENCAST PILLAR MINING METHOD

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A research report submitted to the Faculty of Engineering and the Built Environment, University of the Witwatersrand, Johannesburg, in partial fulfilment of the requirements for the degree of Master of Science in Engineering.

Johannesburg, 2022

DECLARATION

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ABSTRACT

The main objective of this research study is to identify factors that cause coal losses and dilution when underground coal pillars are exposed through opencast mining methods, and to find ways to reduce the negative impacts of these factors. The opencast mining methods used in the exploitation of underground coal pillars involve the blasting of pillars with overburden and collapse the bords before exposure by draglines, trucks, and shovel fleets.

This research study also aims to answer the main research question relating to the current causes of both dilution and coal losses, and interrogate if the existing bonus incentive scheme promotes the motivation for employees to send more waste material with the run of mine coal into the crushing plant.

To achieve the main purpose and answer the research questions, research methods such as the application of multi-criteria decision methods, simulations using software such Science RS2 and 3DDig, and application of empirical formulae to calculate pillar scaling were extensively used in this study. By using these methods, it was found that coal losses were mainly caused by the following factors:

- Roof coal losses take place due to blast shear and exposure equipment such as draglines and shovels scalp coal during exposure. The scalping happens because of inaccurate coal pillar profile sent to the monitoring system designed to assist operators with visibility of the plan to be followed;
- The other coal loss happens on the floor due to accumulation of water which the mine struggles to control in the coaling face and hard coal left on the floor because of exposure equipment cycling the pit at a faster rate and catching up with the coal extraction process before all the coal can be extracted, and
- Geological and geotechnical factors were also found to contribute immensely towards coal losses in the mine.

In terms of dilution causes, strip orientation and the current incentive scheme were found not to cause dilution in the mine. The main causes of dilution were found to be a combination of pillar scaling and current blasting practice, mode of failure when bords are collapsed and methods used to blast pillars before exposure. These causes are not currently considered because the mine assumes that the pillar shape and size have not changed since the creation of pillars some years ago.

When the coal pillar methods were reviewed with multi-criteria decision methods, it was found that bord collapse with the cleaning of waste between pillars was found to be the suitable method. However, if pillar implosion is to be used, the review found that pillar implosion with coal pillar loss of 26.5% was found to be an appropriate method.

The main recommendation for this research study is to adopt the bord collapse method with cleaning between pillars across all the pits. This method can only be successful if a detailed sequence is developed with the operators and introduce a monitoring system where a coal pillar plan can be loaded into the coal loading equipment. However, if the mine wants to continue to use pillar implosion in some of the pits, the current bench setup and method should be changed to allow coal blending from the pit. The required blending ratio to reach plant yields of above 60% is two blocks of high quality upper seam coal which are not been mined previously and one block of imploded coal pillar. The imploded coal pillar must be cleaned up to the top of the predicted profile leaving waste on the lowest points in the profile.

DEDICATION

I would like to dedicate this research study to my wife, Cecilia Mabuza for the support given during the long nights spent with me while researching this topic and taking care of the family during my absence. I also dedicate this research study to my kids, Milanie, Angela, and Angulo and my late uncle, Timhaka Zola Mabuza.

ACKNOWLEDGMENTS

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LIST OF SYMBOLS AND ACRONYMS

%	: Percent
λ_{max}	: Eigenvalue constant
AHP	: Analytical Hierarchy Process
ATCOM	: Arthur Taylor Colliery Opencast Mine
BECSA	: BHP Billiton Energy Coal South Africa
BCM	: Bulk Cubic Meter
CH_4	: Methane
CM	: Centimeter
CO	: Carbon monoxide
CO_2	: Carbon dioxide
CV	: Calorific Value
DTH	: Down the hole
FY	: Financial Year
g/cm^3	: gram per cubic centimeter

IRR	: Internal Rate of Return
K	: Kelvin
Kg/m^3	: Kilogram per cubic meter
Km	: Kilometer
kJ/mole	: Kilo Joule per mole
M	: Meter
MJ/kg	: Mega Joules per kilogram
MMS	: Middelburg Mine Services
MN	: Mega Newton
$molek^{-1}$	: Mole per Kelvin
$molg^{-1}s^{-1}$	: Mole per gram per second
MPa	: Mega Pascals
m/s^2	: Meters per second square
Mt	: Million tonnes
n	: Number of criteria

NPV	: Nett Present Value
Ppm	: Parts per million
ROM	: Run of mine
RD	: Relative Density
SAEC	: South Africa Energy Coal
s-1	: Per second
t/m^3	: Tonnes per cubic meter
US OSHA	: United States Occupational Safety and Health Administration
VDDN	: Van Dyk's Drift North
VIKOR	: Vise Kriterijumska Optimizacija I Kompromisno Resenje

1 INTRODUCTION

1.1 Purpose of the Study and Location

The main purpose of this research project is to identify the contributing factors to coal losses and dilution in the opencast pillar mining method and find ways to reduce the negative impact of these factors. This research is important because virgin coal in South Africa is becoming depleted and most of the coal reserves that are left are in the form of coal pillars and most of the virgin coal resource is in the Waterberg Coalfield (Hartnady, 2010).

The research project was done at one of the coal mines currently owned by Seriti Power, which was previously owned by South32's South Africa Energy Coal (SAEC), called Middelburg Mine Services (MMS). Figure 1-1 shows the location of Middelburg Mines Services.

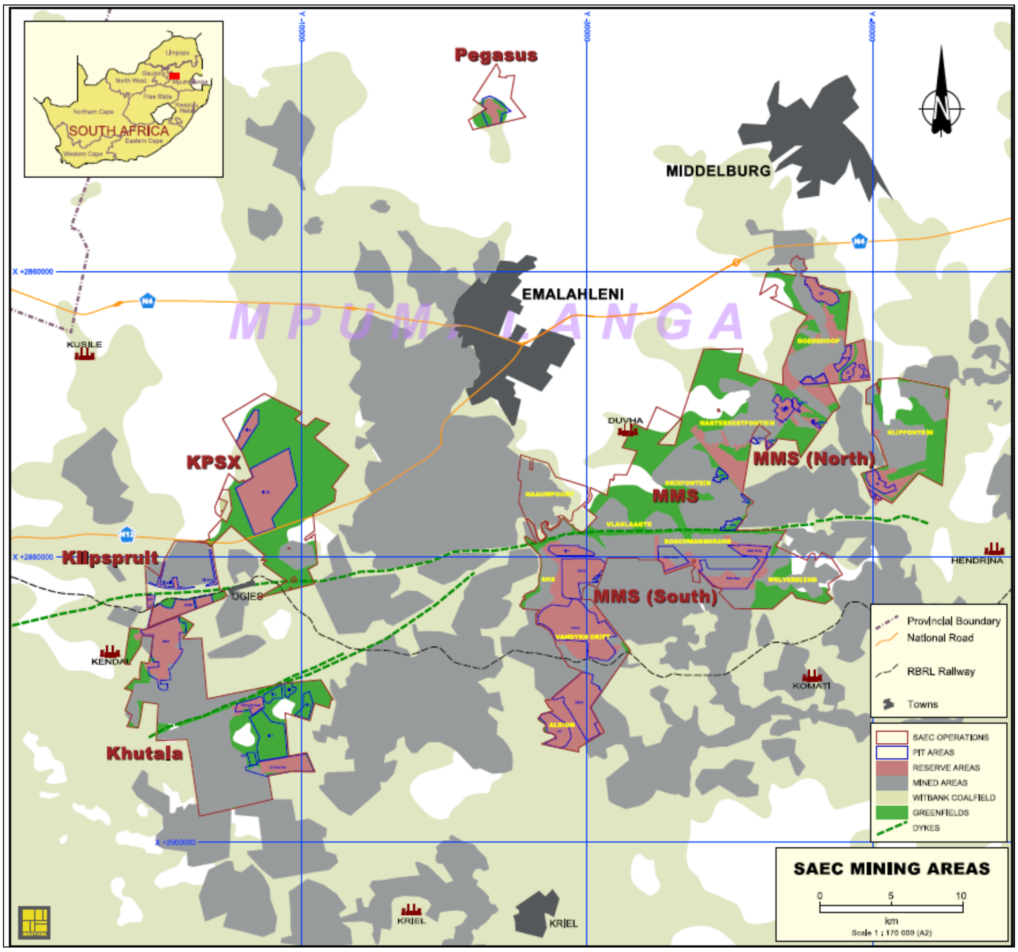

Figure 1-1: Location of Middelburg Mine Services (South Africa Energy Coal, 2021)

MMS, as shown in Figure 1-1, is situated 30km South East of Emalahleni town, formerly known as Witbank and South of Middelburg town. MMS is divided into two sections, namely, North Pits which produce coal supplied to Eskom, and South Pits which produce coal which is exported. North Pits are mainly mining greenfield areas whereas the South Pits are mainly mining underground coal pillars with draglines and truck and shovel mining methods. This research study was only focused on the South Pits because coal pillars are currently being exposed by using opencast mining methods.

MMS is a complex multi-seam opencast mine located in the Witbank Coalfields of Mpumalanga Province of South Africa. MMS started operations in the early 1980s and produces coal for both the export and domestic markets (South Africa Energy Coal, 2021). Figure 1-2 shows the stratigraphic column of South pits at MMS.

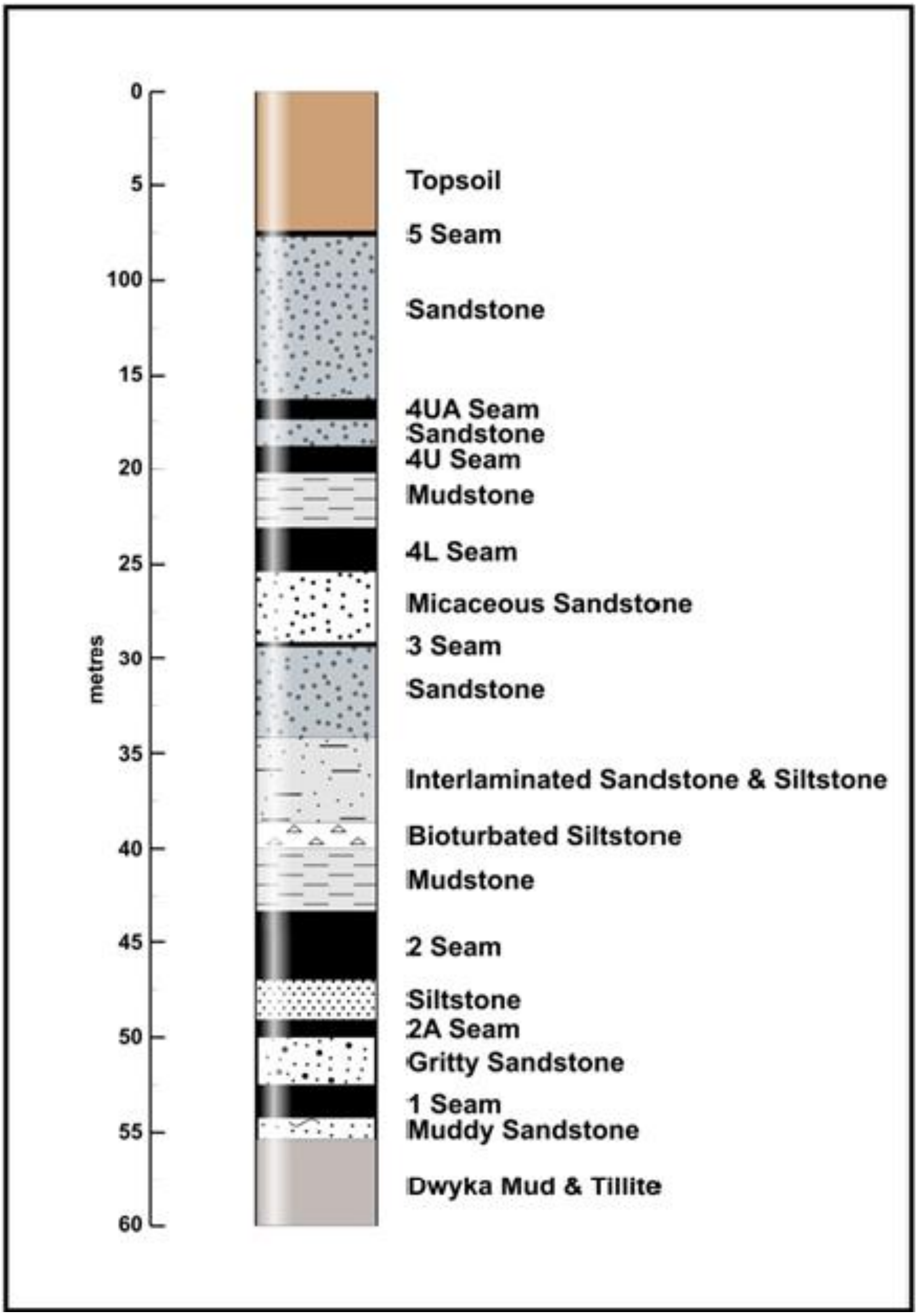

Figure 1-2: MMS South Pits Stratigraphic Column (Middelburg Mines Services, 2021)

MMS South pits reserves consist of the following coal seams:

- No. 4Upper A (4UA) seam, which makes 16.8% of the total coal reserves and it has not been mined. The average raw calorific value (CV) and ash content for this seam are 20.79MJ/kg and 32.35% respectively;
- No. 4Lower (4L) seam, which makes 19.1% of the total reserve and it was mined previously through underground bord and pillar method. The 4L seam has the following average raw qualities, that is, the CV is at 23.8 MJ/kg and the ash content is at an average of 24.97%;
- No. 2 seam coal, which makes 39.7% of the total reserve, was previously mined through underground bord and pillar method. This seam has the following average raw qualities, that is, the CV of 25.6 MJ/kg and the ash content at 21.4%; and
- No. 1 seam, which makes 24.4% of the total reserve, was partially mined through underground bord and pillar method. No. 1 seam coal has the following raw qualities, that is, the average CV of 23.04 MJ/kg and the ash content of 27.5%.

To exploit these seams, multi-pass method is used. Multi pass mining method set-up is shown in Figure 1-3.

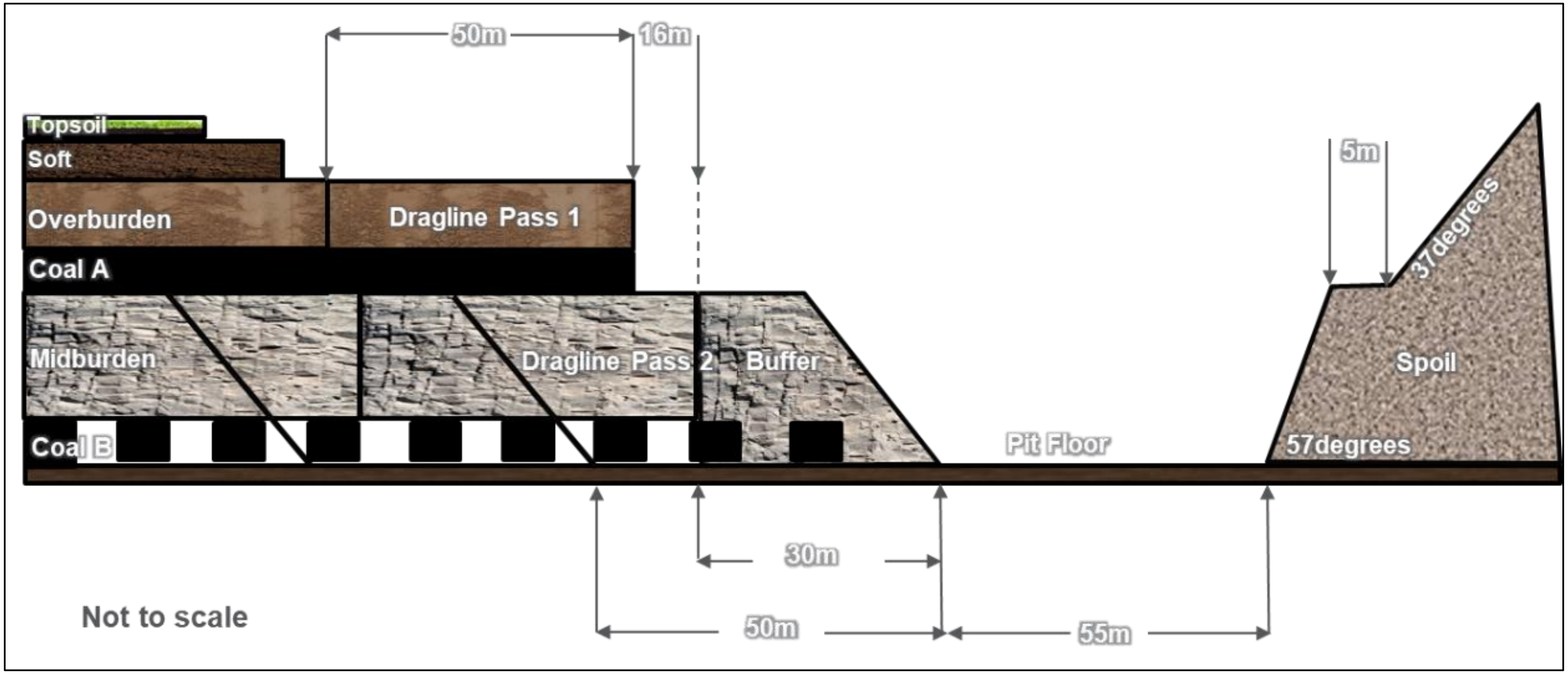

Figure 1-3: Multi-Pass Mining Method (South Africa Energy Coal, 2018)

The first step of this method is to expose Coal A by using draglines and the coal is extracted as the dragline continues to expose. The first step of the multi-pass method is called Dragline Pass 1. The second step, called Dragline Pass 2, is to expose Coal B once Coal A is fully extracted and the burden below Coal A has been drilled and blasted. However, coal blending from the same pit proves to be a challenge because this method allows for single seam to be exposed and extracted at a time.

1.2 Project Statement

Pillar mining through the opencast mining method is associated with spontaneous combustion, a high dilution percentage, and more coal losses. Spontaneous combustion is currently being reduced by collapsing either pillars or waste material between the bords, and this results in high dilution and coal losses. Therefore, coal mining companies that have coal reserves in the form of pillars, need to maximise the recovery of coal and hence minimising the operational costs. It is, however, imperative to investigate alternative mining techniques that will promote fewer coal losses and dilution, resulting in an increased plant yield.

MMS reserves that are left are mainly underground pillars which were left when underground reserves at Douglas Colliery were depleted. Douglas Colliery underground reserves were depleted towards the end of 2008 (Van der Merwe, 2008). To sustain the future of SAEC, a decision was taken by SAEC lead team to expose underground pillars through opencast mining methods. Through these methods and the age of the pillars, MMS is unsustainable due to high cost, as a result of the following factors:

- Low yield or recovery due to high diluted Run of Mine (ROM) being fed into the processing plant;
- High coal losses due to blasting techniques, types of equipment that are used for exposure and high volumes of water accumulations; and
- Monthly bonus incentives that are paid after a ROM target has been met.

Figure 1-4 shows planned coal losses and dilution that will be incurred by MMS in a 5-year period, starting from July 2021 until the end of June 2026.

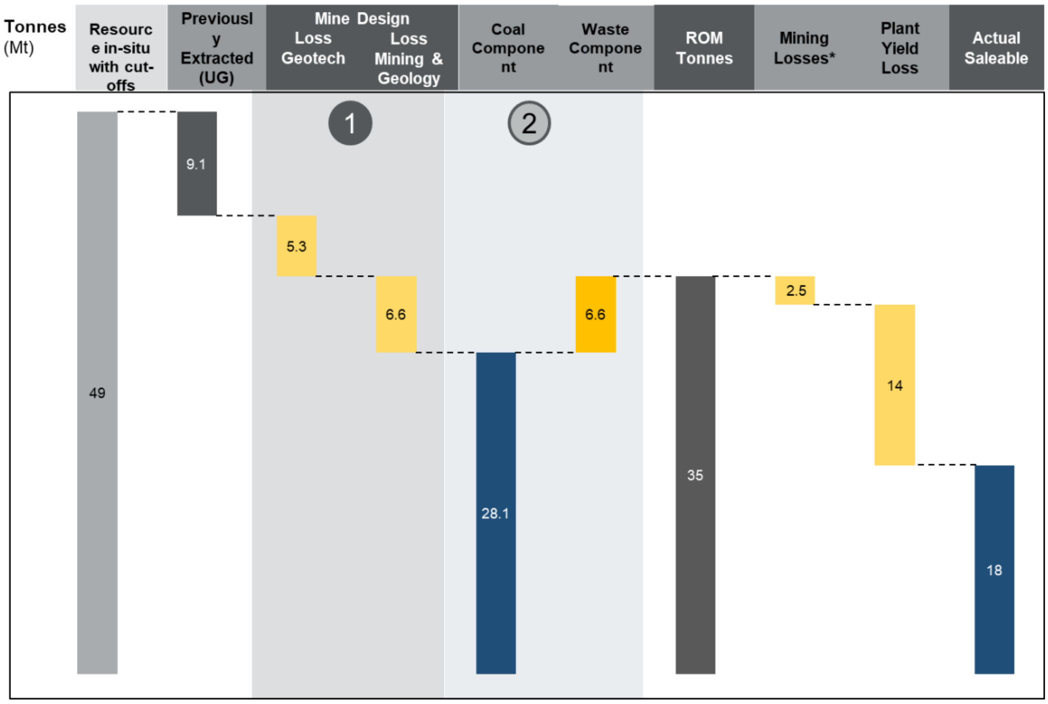

Figure 1-4: 5 -Year Planned Coal Losses and Dilution at MMS South Pits (Middelburg Mine Services, 2020)

Figure 1-4 also shows that MMS is planned to lose 11.9 Mt of insitu coal after underground coal reserves were removed and these losses are due to geotechnical, geology and mining factors. After the coal reconciliation was conducted, it was found that the planned coal losses were understated by 2.5 Mt. The total coal losses at MMS are equivalent to 36% of insitu coal tonnes after underground coal tonnes have been removed.

Another reason for this research study is to understand how dilution in the pillar mining sections can be reduced. Dilution is planned to be 6.6 Mt of waste in the 5-year plan, which is equivalent to 23.5% of the ROM tonnes fed into the processing plant. The impact of dilution in the mine is the yield loss that is planned to be 14 Mt in the 5-year plan and this is equivalent to 43% of ROM tonnes after mining losses of 2.5 Mt were removed. Figure 1-5 shows the comparison of coal ash contents when different methods to treat pillars are used.

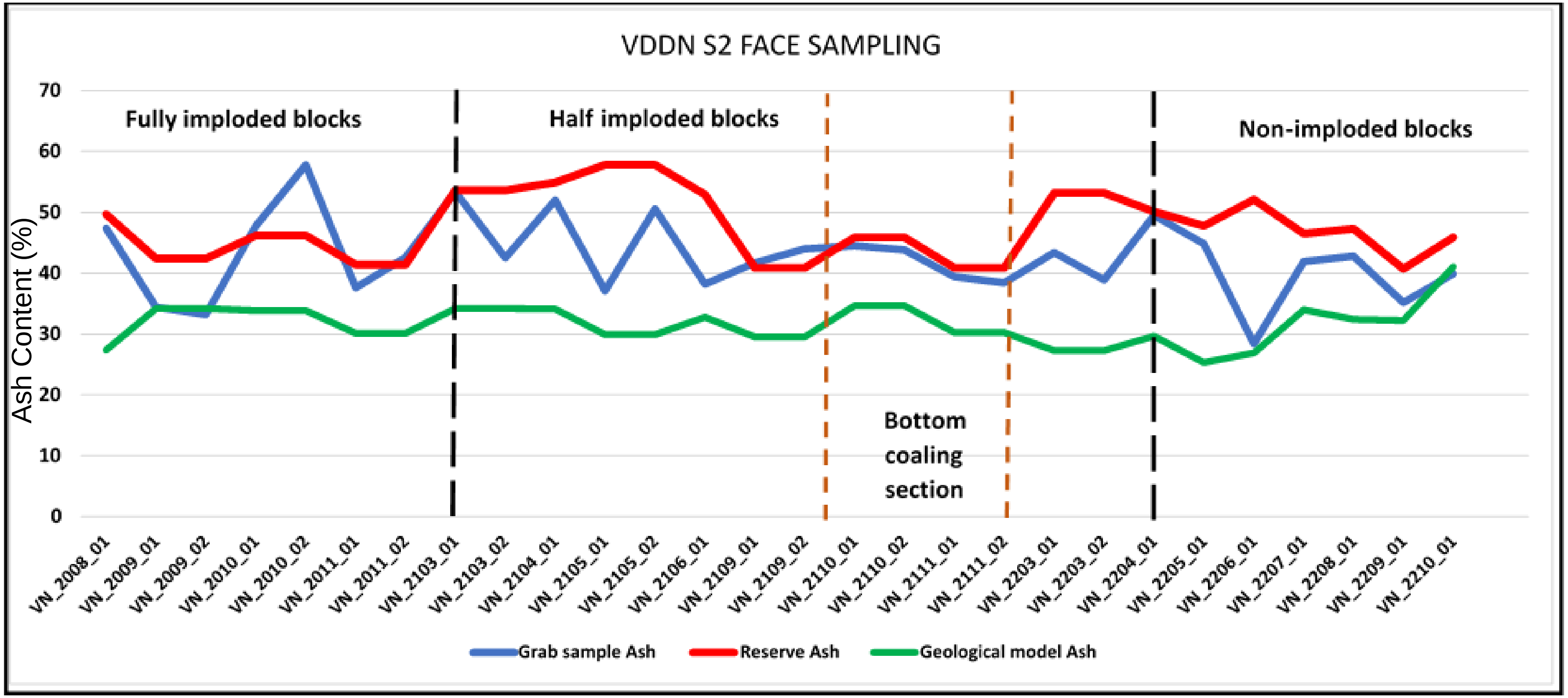

Figure 1-5: Ash content comparison for fully, half and non-imploded blocks (Middelburg Mine Services, 2020).

Figure 1-5 also shows an increase of ash content of the coal being fed to the processing plant. The high ash content is due to methods used to blast underground pillars and these methods are pillar implosion or collapse and bord collapse. High ash content will lead to drop in the overall plant yield during processing of the ROM coal, and this coal cannot be sent to a powerstation or exported without being washed in the processing plant.

1.3 Justification for Research

This research study is worth doing because of the amount of coal that the mine is currently losing and the high dilution in the pillar sections. The other motivation for this study is the following benefits that the mine can realise:

- Optimisation of revenue by minimising coal losses and
- Reduce operational costs due to a reduction in the amount of diluted coal sent to the processing plant.

1.4 Objectives of the Research Study

The objectives of this study are to:

- Review current mining practices at Middelburg Mine Services (MMS);
- Review blasting techniques that are currently being applied at MMS;
- Review current technique that is applied in the combat against spontaneous combustion;
- Review current causes of coal losses in opencast pillar mining method;
- Review current causes of dilution;
- Review planned coal losses and dilution assumptions; and
- Development of optimisation methods that can be applied to coal losses and dilution reduction.

1.5 Previous Work

An earlier study into pillar extraction by using opencast mining methods was done by Clough and Morris (1985) who identified six pillar extraction mining methods. The six methods are as follows:

- Collapsing of bords;
- Variation of the above method (Scalping);
- Preservation of the underground workings;
- Collapsing of the bords and excavation of the unblasted coal;
- Preservation of the workings and excavation of unblasted coal; and
- Collapsing of pillars.

From the analysis done by Clough and Morris (1985), it was recommended that the scalping method was the preferred method due to the following reasons:

- Increases overall yield due to removal of one metre of top coal;
- Dozer not required, but dragline bucket elevation control is required;
- Dilution reduction; and
- Safer method due to the fact that equipment will be working on solid and compacted material.

However, a recent study that was done by Ngwenyama *et al* (2017) has shown that when bords are collapsed, they do not behave in the manner that was initially predicted by Clough and Morris (1985). This recent study showed two ways in which bords collapse, that is, total collapse and partial collapse. MMS currently applies bord collapse and underground workings preservation, but the mine is still experiencing high dilution and coal losses. Due to these issues, MMS pits were put into care and maintenance in the beginning of September 2020. Therefore, a detailed research study is required to understand the causes of dilution and coal losses, and how these can be reduced or eliminated by introducing a different system or method of dealing with pillar exposure and extraction by opencast mining methods.

1.6 Research Methods

The research study was carried out using the following methods:

- Literature review: Existing studies related to pillar mining through opencast mining methods was reviewed to identify gaps that might exist in the mine's application of the opencast pillar mining method;
- Benchmarking with other operations: Benchmarking studies were carried out with other mines that are currently exposing underground coal pillars

through opencast mining methods. Benchmarking studies were aimed at comparing practices from other operations with MMS's existing practices, and if these mines have best practices then their methods might be adopted;

- Site data collection: The data collected in the mine include actual drill hole depth and position, coal scalping thickness by exposure equipment and area of water accumulation in the coaling face;
- Pit observations: The observations were done on both blasting and exposure crews. Observations on blasting crew was solely focused on charging of blast holes with explosives and on whether exposure equipment was in compliance with the stripping plan;
- Determine an optimal width for the buffer blasts because current buffer blast width at MMS is 50 meters, which is too high when compared to other mines that are currently exposing underground coal pillars through opencast mining methods in the Witbank coalfields;
- Determination of rate of pillar scaling using empirical formulas. This was done because pillars at MMS have previously failed;
- Simulations of pillar failure modes using rock engineering software called Science RS2; and
- Application of Multi-Criteria-Decision Analysis methods to review and select a suitable exposure and pillar blasting methods.

1.7 Structure of the Research Report

To achieve the objectives of this research study, the report for this study will be structured as follows:

- Chapter 1: This chapter will introduce the problem that is being investigated and gives the brief location of where the study was carried out. It will also give summary of the main objectives of the study, summary of the research methods that are used to carry the study and motivation to carry out this study;

- Chapter 2: This chapter will present literature that is related to the factors that cause both coal losses and dilution, when coal pillars are exposed using opencast mining methods. This will also go into detail on benchmarking with other coal mining operations that expose coal pillars through opencast mining methods. Various methods dealing with blasting of the coal pillars and selection of these methods with multi criteria decision analysis will also be reviewed in this chapter;
- Chapter 3: Research methodology followed in the study will be discussed and the results from these methods will also be summarised in this chapter;
- Chapter 4: This chapter will summarise the findings and cover the analysis of the results; and
- Chapter 5: This chapter will conclude and recommend on the way forward.

2 LITERATURE REVIEW

2.1 Introduction

Underground and opencast coal mining in the Witbank coalfields started in 1889 (Govender, 2017, cited King and Min, 1980). King and Min (1980) as cited by Govender (2017) said that coal mining was predominately mined through underground bord and pillar mining methods due to lack of suitable technology. In 1971, a massive opencast mine called Optimum Colliery was opened, and it introduced big equipment such as draglines, trucks, and shovels to expose and extract coal (Govender, 2017). From 1971, more opencast mines were developed, and mining virgin coal by using truck, shovels and draglines. Due to the introduction of these large equipment, greenfield and easily accessible coal in the Witbank coalfields were depleted at a faster rate, and this has led coal mining companies to start investigating ways to expose and extract coal pillars left by previous underground bord and pillar mining method.

Clough and Morris (1985) developed six basic methods of exposing coal pillars through opencast mining methods and most mines developed their methods from these basic methods. However, these methods are associated with high dilution and coal losses that reduce the coal quality and the coal reserve, and hence affecting the nett present value (NPV) of a mining operation. The six methods, effects of dilution, and ways of how coal losses are incurred will be discussed in this chapter. Except for these issues, coal pillar mining through opencast mining is also associated with the high risk of spontaneous combustions that must also be addressed when coal pillars are exposed. Spontaneous combustion causes, impacts, and mitigation methods will also be discussed in this chapter.

As it has been mentioned that there are six basic methods to deal with coal pillar exposure through opencast mining methods, selection criteria should be developed to select an appropriate method that will be condition based. Multi criteria decision analysis will be reviewed to determine a suitable selection method for this study.

Except for the technical aspects that contribute to high dilution and coal losses, human behaviour needs to be understood and determine how it contributes towards high dilution and coal losses. Human behaviour is usually influenced by bonus schemes that mines have introduced with an aim of improving productivities. The relationship between human behaviour and bonus scheme will also be addressed in this chapter.

2.2 Dilution and its Implications to Mining

Dilution in mining can be defined as the amount of waste material that is mined with ore and cannot be separated from the ore (Anoush Ebrahimi, 2013). This material is fed to the processing plant together with the ore to produce ore with specific qualities. Dilution can be expressed mathematically by the following equation (Anoush Ebrahimi, 2013):

$$Dilution = \frac{\text{Waste tonnes}}{(\text{Ore tonnes}+\text{Waste tonnes})} x\ 100 \tag{1}$$

2.2.1 Impact of Dilution in the Mining Industry

Dilution has a negative impact on the sustainability of any mining operation, if not controlled, because it can increase the operating costs and hence reduces the profitability of the operation. Table 2-1 shows the impact of dilution on coal qualities and plant yield at Klipspruit Colliery.

Table 2-1: Effects of Dilution at Klipspruit Colliery (Klipspruit Colliery, 2021a)

Description	**Units**	**4L Seam**		**4Upper Seam**		**5 Seam**		**2 Seam**	
		Uncontaminated	Contaminated	Uncontaminated	Contaminated	Uncontaminated	Contaminated	Uncontaminated	Contaminated
Material type used for contamination		None	0.61 m of sandstone parting	None	0.40 m of brownish-grey shale		0.41 m shale floor containing thin bright coal beds		0.50 m shale roof
Coal thickness	m	2.97	2.97	2.38	2.38	1.49	1.49	5.24	5.24
Raw Qualities									
Relative Density	t/m^3	1.57	1.69	1.65	1.70	1.62	1.70	1.50	1.60
Ash Content	%	25.93	36.31	32.31	36.99	31.82	39.38	21.90	29.06
CV	MJ/kg	22.40	18.48	19.56	17.66	20.73	17.90	24.42	21.70
Products									
Primary Yield	%	39.05	28.99	0	0	34.64	24.03	51.09	43.63
Secondary Yield	%	34.96	26.68	74.19	57.90	31.91	23.00	30.07	25.82
Overall Yield	%	74.01	55.67	74.19	57.90	66.55	47.03	81.16	69.45

The effects of dilution on a coal mine, as shown in Table 2-1 are summarised as follows:

- When No.4L coal seam is contaminated by 0.61m of sandstone burden, total plant yield drops by 18.34% to 55.67%. The primary and secondary yields decrease by 10.06% and 8.28% respectively. The drop in the total yield leads to an increase in discard material. In terms of raw feed qualities such as ash content and CV, the ash content increased by 10.38% and CV decreased by 3.92 MJ/kg.
- No. 4U coal has uncontaminated raw ash content and CV of 32.31% and 19.56 MJ/kg respectively. If the No. 4U coal is contaminated with 0.40m of brownish-grey shale, the total plant yield drops by 14.25%. The raw feed qualities become poorer due to brownish-grey being less carbonaceous. The raw ash content increased to 36.99% and the CV decreased to 17.66%;
- No. 5 seam and No. 2 seam coal follows the similar trend as both No. 4L and 4U seams. The biggest impact is when the coal seam thickness is thinner and has inferior raw qualities; and
- If coal seams with inferior qualities are contaminated with waste with similar or above seam thickness, the ROM coal will not make the cut-off qualities during reserving. Table 2-2 shows the MMS cut-off qualities that are applied during reserve model creation.

Table 2-2: Cut-off qualities applied in the 5-year plan (Middelburg Mine Services, 2020)

Middelburg Mine Services					
Key Modifying Factors	**5#**	**4UA#**	**4L#**	**2P#**	**1#**
Thickness Cut-off	1m	1m	1m	1m	1m
Calorific Value Cut-off	>12MJ	>12MJ	>12MJ	>12MJ	>12MJ
Volatile Matter Cut-off	>17.9%	>17.9%	>17.9%	>17.9%	>17.9%
Ash Cut-off	<45%	<45%	<45%	<45%	<45%

The 4U and 5 seam coal do not make the cut-off qualities when the thickness contamination is above 0.7m and 0.45m respectively. These seams make the CV cut-off, but get rejected on the ash content. However, coal seams with better qualities such as No. 4L and 2 seam will require more waste material to achieve rejection.

Anoush Ebrahimi (2013) has also identified the following impacts that dilution has in a mining operation, especially in terms of monetary value:

- Reduction of the overall value of the project because it prolongs the life of mine by reducing crusher's effective capacity. An extended life of mine due to dilution, reduces the project's or mine's NPV and internal rate of returns (IRR);
- Reduction of plant feed quality; and
- Operating costs for both the crusher and processing plant increase directly by the amount of dilution factor. If dilution is determined to be 15% and the processing cost is estimated to be R70/t, R10.50 is spent processing every ton of waste.

2.2.2 Causes of Dilution

Dilution in a mining operation happens in two ways (Leite et al, 2014) and Figure 2-1 shows the two types of dilution.

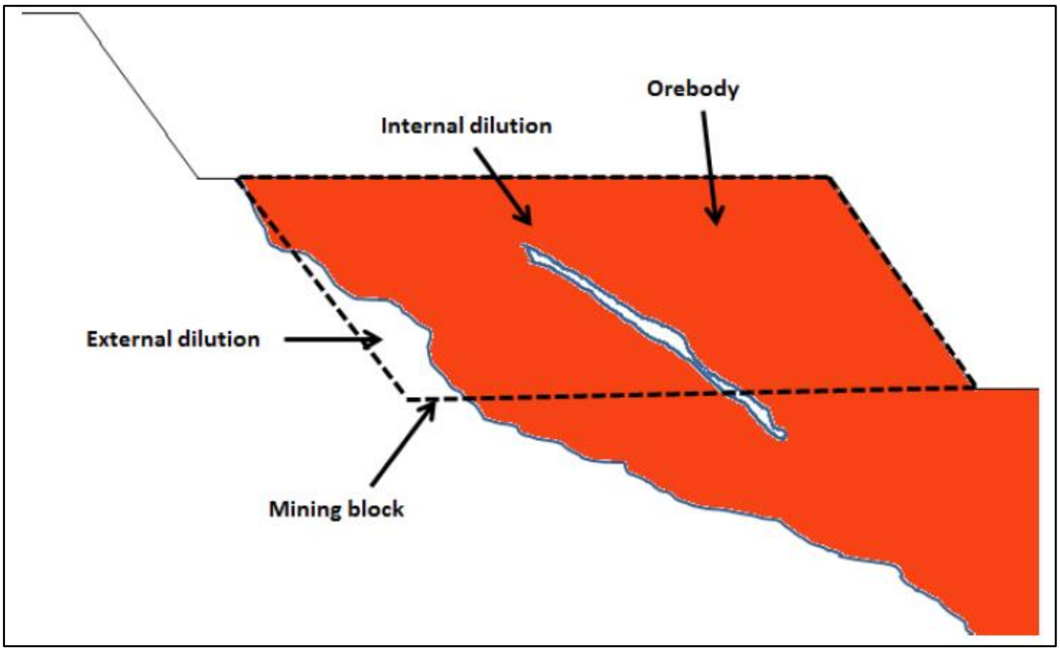

Figure 2-1: Two ways dilution happens in a mining operation (Leite et al, 2014)

The two ways dilution happens is through internal and external ways as illustrated in Figure 2-1.

Internal dilution is due to the material that is inherent inside the coal seam or ore (Leite et al, 2014). External dilution is due to the unfavourable movement of the boundary between two types of materials, that is, waste and ore. External dilution is also caused by the mining method and the equipment type being used in an operation. The smaller the equipment the less the dilution because small equipment can easily do selective mining. Except for equipment and mining methods, blasting also causes external dilution because of high powder factors causing coal pulverisation where coal and waste make contact (Leite et al, 2014).

Human factors also play a role in external dilution because of tonnage based incentives. Most mines pay incentives based on how much ore or coal is mined and crushed. Table 2-3 shows the key performance indicators that SAEC mines are using to incentivise the workforce and Table 2-4 shows how mines not owned by SAEC incentivise their workforce.

Table 2-3: SAEC opencast mines key performance indicator for incentive payments (BECSA,2014)

Production Bonus Measure	Klipspruit	Wolvekrans	Middelburg Colliery
Crushed Tons (Weighting = 50%)	✓	✓	✓
ROM Tons (Weighting = 25%)	✓	✓	✓
GM Specified Target (Weighting = 25%)	✓	✓	✓

Table 2-4: Other coal mines key performance indicator for incentive payments

Production Bonus Measure	Mine A	Mine B	Mine C	Mine D
Volume moved	✓			✓
Safety	✓	✓	✓	✓
Saleable tons produced	✓		✓	✓
Drilling Meters	✓			
Dragline BCM's	✓			✓
Plant Feed	✓			
ROM Tonnes	✓	✓	✓	
Qualities				✓

The incentive schemes for most mines are similar because they pay bonuses based on safety, ROM, crushed tonnes and saleable tonnes produced. MMS also pay employees bonuses using ROM and crushed tonnes as key performance indicators (KPI's). These KPI's for mines exposing underground pillars, where coal is mined with waste will encourage employees to mine more waste with coal and send it to the crushing plant.

2.2.3 Dilution Control

Dilution can be controlled in a mining operation by establishing certain processes. The processes that can be set-up to deal with dilution either external or internal are summarised as follows (School of Mining Engineering, 2020):

- Improve resource modelling by drilling more exploration holes and mapping of geological structures;
- Review mining methods periodically to ensure that it is still the suitable method for the geology;
- Refine powder factors by adjusting drilling and blasting parameters;
- Increase the frequency of grade control;
- Review tonnage-based incentives to be in line with the mine's dilution control strategy;
- Ensure company performance incentives are not countering to required dilution control; and
- Training and awareness of employees.

2.3 Basic Mining Methods used in the Mining of Coal Pillars

Clough and Morris (1985) have identified six basic methods that can be used to mine underground coal pillars through opencast mining methods and these methods are summarised as follows:

- Method 1: Collapsing of bords;
- Method 2: Variations of method 1;
- Method 3: Preservation of workings;
- Method 4: Collapsing of the bords and excavation of the unblasted coal;
- Method 5: Preservation of the workings and excavation of unblasted coal; and
- Method 6: Collapsing of the pillars.

2.3.1 Method 1: Collapsing of Bords

This method involves the collapse of the bords through blasting of burden overlying the coal seam. Figure 2-2 shows the bords that have been filled with either roof coal or waste material or both.

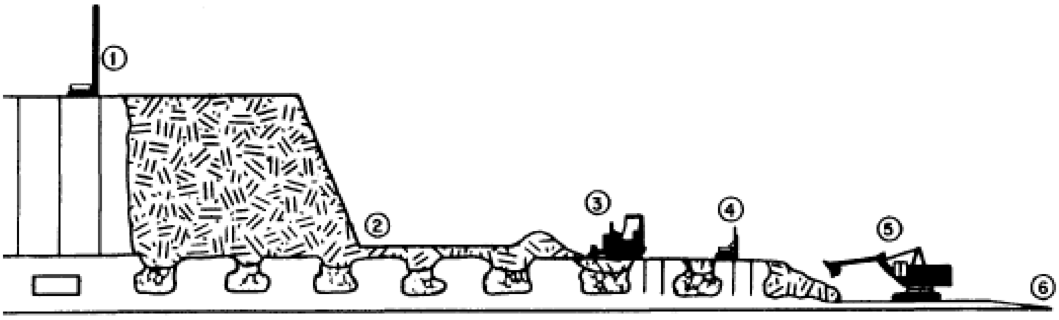

Figure 2-2: Collapsing of bords method (Clough and Morris, 1985)

Figure 2-2 also shows the numbered list of activities that take place in this method. The activities include drilling of the burden material to top of coal (1), dragline removing burden material and leave some waste on top of coal (2), dozer cleans waste on top of coal (3), drilling of the coal pillars (4) and loading of the blasted coal with the mixture of coal and waste in the bords (5). Dilution for this method is estimated to be 20% (by volume) of the coal, provided that the activities take place as described (Clough & Morris, 1985). According to Clough and Morris (1984), this method can only work if all the bords can collapse during burden blasting. This collapse is thought to take place due to dead load of the blasted burden above the coal. Clough and Morris (1985) further stated that the toe damage at the bottom of the blast holes and cratering would assist in the collapse of the top coal into the workings.

However, Ngwenyama *et al* (2017) have identified gaps on how the underground bords fail and have identified two ways or modes that underground bords fail. Ngwenyama *et al* (2017) could not pinpoint how the voids are filled after the overburden blast has taken place. Figure 2-3 shows the possible modes of failure for underground bords.

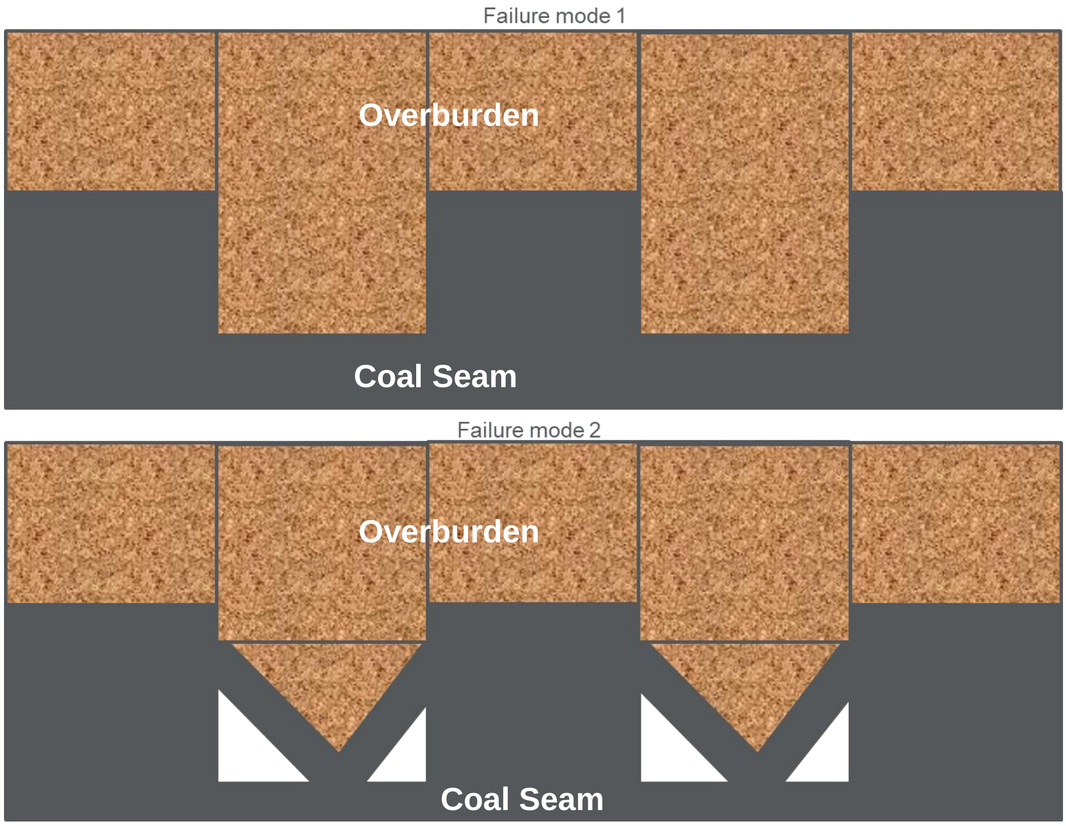

Figure 2-3: Modes of bord failures after overburden blast (Ngwenyama *et al,* 2017)

Failure Mode 1:

Failure Mode 1, as shown in Figure 2-3 assumes that the overburden material fully collapses the roof coal, allowing the roof coal to fall directly into the void and the rest of the void is filled by the overburden material.

Failure Mode 2:

The roof coal above the bords is semi-collapsed as the overburden is blasted, leaving some unfilled voids as shown in Figure 2-3. The bords are always collapsed, but in some cases, the roof coal above the bords may not fully or perfectly collapse. The main concern regarding this failure mode is the possibility of spontaneous combustion due to air penetrating the underground workings through the gap between the pillar and the semi-collapsed roof (Ngwenyama *et al,* 2017).

2.3.2 Method 2: Variation of Method 1

This method is like method 1, in terms of activities that need to be carried out. However, the only difference between method 1 and 2 is that at least a meter of coal is removed during exposure by either the dragline or truck and shovel (Clough and Morris, 1985). Figure 2-4 shows the activities that are carried out in method 2.

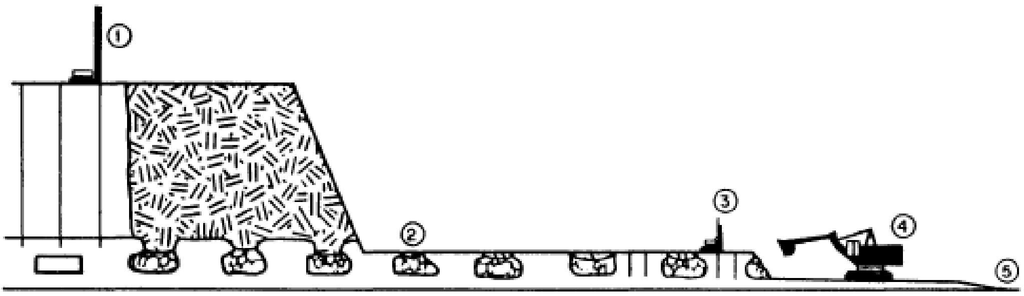

Figure 2-4: Sequence of activities in method 2 (Clough & Morris, 1985).

The activities, as shown in Figure 2-4, of this method are drilling of overburden ahead of mining to create a buffer (1), dragline exposes coal and scalps one meter of coal (2), coal drilling and blasting takes place (3), coaling fleet mines the blasted coal with waste inside the bord (4) and dozer rips the hard coal on the floor (5).

The scalping of coal by the dragline reduces amount of dilution but increases coal losses. Less dilution translates to higher plant yields and an improved NPV and IRR. The success of this method depends on the dragline operator's competency and skill because excess coal can be scalped, which will result in more coal losses. Therefore, dragline operators must be closely monitored so that over digging into coal can be prevented.

2.3.3 Method 3: Preservation of Workings

Underground bords and pillars are left undisturbed by leaving unblasted material above the coa (Clough and Morris, 1985). Figure 2-5 shows the preservation of the workings and the activities that are done to expose and extract coal.

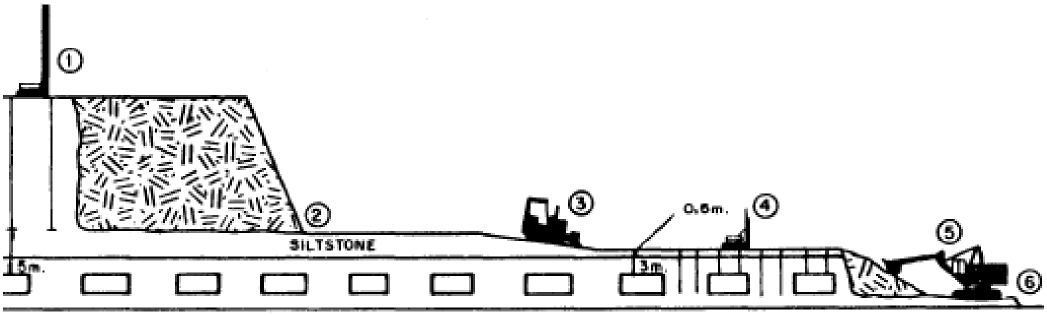

Figure 2-5: Sequence of activities for method 3 (Clough and Morris, 1985)

The underground workings are preserved by leaving a beam of 3-meter thickness above the bord and coal (Clough and Morris, 1985). Clough and Morris (1985) further stated that the 3-meter thickness will be enough to support the dead weight of the blasted burden. Figure 2-6 shows the actual underground workings that have been preserved.

Figure 2-6: Preserved underground workings with beam on top of coal (Maphanga, 2008).

Clough and Morris (1985) noted the following advantages and disadvantages for method 3.

Advantages:

- Minimal coal dilution and it is estimated to be at least 10.8%; and
- Minimal coal losses due to exposure equipment such as draglines not removing waste up to top of coal.

Disadvantages:

- Burden drill holes must be drilled at an exact depth. Compliance to drilling plan is paramount;
- Dilution will exceed 10.8% if the beam material is mainly made up of waste material;
- Equipment such as drills, dozers, excavators and trucks can break into the bord and get bogged if the beam thickness is compromised. Figure 2-7 shows an equipment bogged in the underground workings due to insufficient beam thickness.

Figure 2-7: Face shovel falling into a void (Govender, 2018)

2.3.4 Collapsing of the Bords and Excavations of the Unblasted Coal

This method is like method 1, but the only difference between this method and method 1 is that the coal is mined with larger equipment such as shovels and the coal is not blasted. Figure 2-8 shows the sequence of activities that take place in method 4.

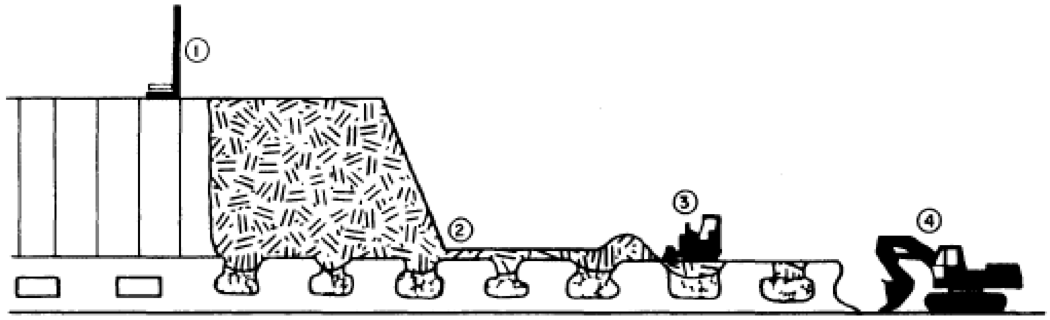

Figure 2-8: Method 4 sequence of activities (Clough and Morris, 1985)

The first activity in this method is the drilling and blasting of the buffer ahead of dragline exposure and then after the dragline removes the waste material and leaves at least a meter of waste material above the coal (Clough and Morris, 1985). A dozer pushes waste material towards the dragline for final placement. Once the dozer completes the cleaning of coal, a larger equipment such as shovels mine the unblasted coal and the mixture of coal and waste in the bords. However, this method's advantage when compared with method 1 is the reduction of coal extraction duration due to non-drilling and blasting of coal because the shovels breaks the coal without any blasting (Clough and Morris, 1985).

2.3.5 Preservation of Underground Workings and Excavation of Unblasted Coal

The underground workings are preserved by leaving a beam of at least 3 meters on top of the bord which is made up of coal and waste (Clough and Morris, 1985). Clough and Morris (1985) further stated that the beam is left intact to prevent the dead load of the overlying burden from causing failure into underground workings. Figure 2-9 shows the sequence of activities for method 5.

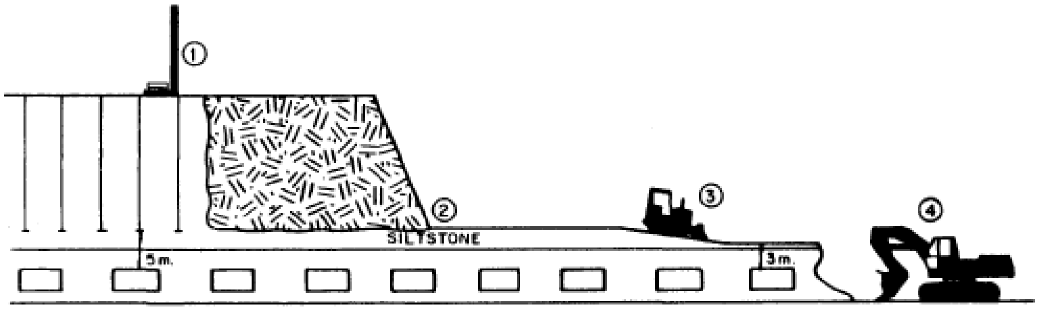

Figure 2-9: Sequence of activities for method 5 (Clough and Morris, 1985)

Coal is loaded unblasted by a shovel that has a break-out force that is more than the strength of the coal seam. This method reduces coal dilution because the dozer can push the waste material to the coal loading unit's face, but the disadvantage of this method is the high risk of spontaneous combustion (Clough and Morris, 1985). Spontaneous combustion can easily take place due to exposed bords and these bords will allow oxygen to enter the old underground workings easily. Spontaneous combustion affects coal qualities and operator visibility because of fumes from the burning coal.

2.3.6 Collapsing of Pillars

Underground coal pillars are collapsed during interburden blasting, which would result in subsidence free bench for the dragline operation (Clough and Morris, 1985). Figure 2-10 shows collapsed coal pillars that are mixed with waste material.

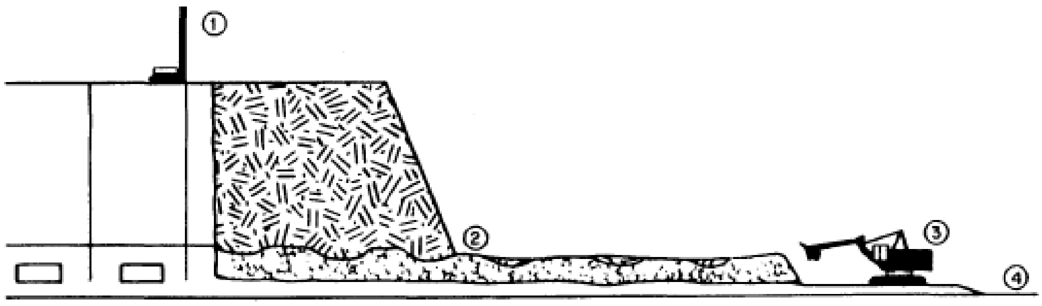
Figure 2-10: Method 6 with sequence of activities (Clough and Morris, 1985)

Clough and Morris (1985) has identified the following disadvantages that are related to this method:

- This method requires the position of the coal pillars to be accurate. The accuracy of the coal pillar positions is challenging due to some previously mined out areas have no accurate plans in a soft copy format. The plans are digitized which might prove challenging;
- Blasted coal pillars would produce a large undesirable amount of fines, which would increase the amount of coal losses and dilution percentage; and
- Doubt on how coal will be displaced due to required high powder factors to break strong overburden material.

The other factors that contributes to difficulty in getting this method right is the coal pillar sizes and shapes that are assumed not to have changed over time. An example of the pillar plan used for opencast mining is shown in Figure 2-11 and this plan is used in the blasting of the underground coal pillars.

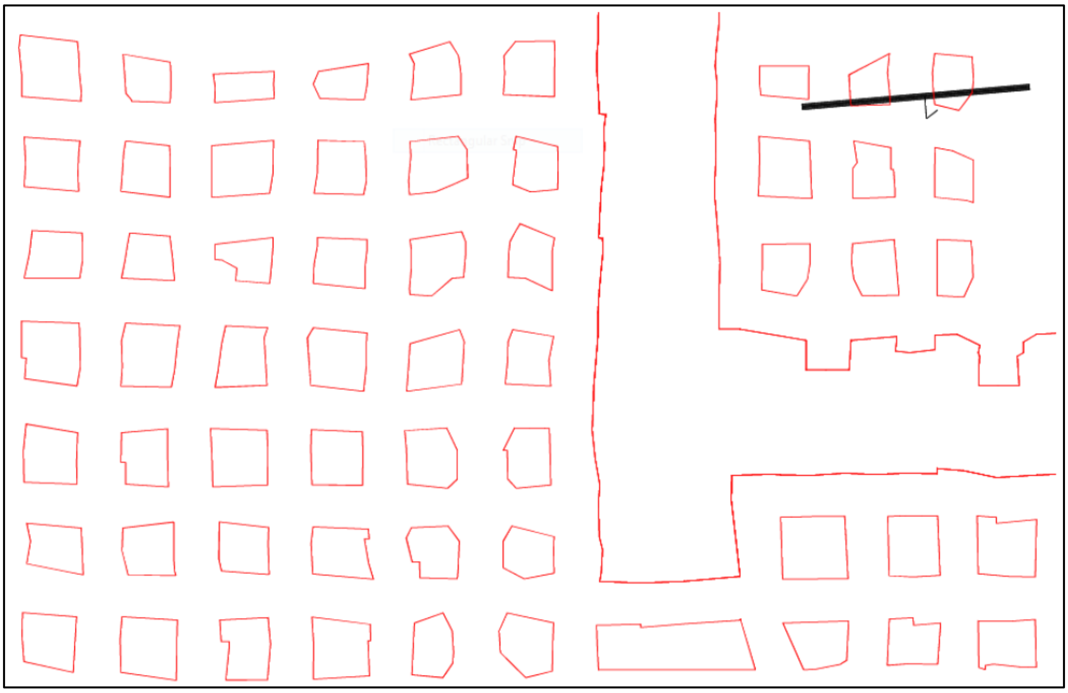

Figure 2-11: Underground pillar plan used for opencast pillar exposure and extraction (Rorke *et al*, 2016)

It is evident that the current mine plans do not consider that the pillars might have changed in size and this can be seen in Figure 2-11. Van der Merwe (2012) stated that pillars were known to have scaled over time and as a result, actual pillar size is much smaller when compared to as-mined dimensions.

Pillars, in most mines in South Africa, have failed due to the continuous scaling. Table 2-5 shows a database of pillars that have failed in some of the South African mines. Some of the failed pillars in Table 2-5 were at MMS, formerly known as Wolvekrans Colliery.

Table 2-5: Data collected of the failed pillars in different Collieries in South Africa (Van der Merwe, 2003)

Case no.	Colliery	Seam	Coal Field	Depth	Pillar width	Bord width	Mining height
s12	Coronation	W1	Witbank	25.9	3.7	8.5	3.1
s16	M Steam	W2	Witbank	21.3	4.0	8.2	4.6
s17	Wolvekrans	W2	Witbank	29.6	5.2	7.0	5.5
s40	Wolvekrans	W2	Witbank	33.5	6.1	6.7	5.5
s41	Crown Douglas	W2	Witbank	30.5	4.6	7.6	3.7
s57	Koornfontein	W2	Witbank	88.4	7.2	6.6	4.9
s116	Waterpan	W2	Witbank	61.0	6.1	6.1	4.6
s118	Waterpan	W2	Witbank	57.9	6.1	7.6	4.0
m149	Koornfontein	W2	Witbank	90.0	7.5	6.0	4.8
m151	Tweefontein	W2	Witbank	62.0	7.5	6.4	4.0
m162	Tweefontein	W2	Witbank	62.0	7.3	6.2	4.0
m166	Tweefontein	W2	Witbank	62.0	6.1	6.1	4.0
m167	Tweefontein	W2	Witbank	62.0	6.1	6.1	4.0
n171	Wolvekrans	W2	Witbank	41.0	6.4	6.4	6.2
n172	Wolvekrans	W2	Witbank	41.0	6.4	6.4	6.2
n173	Wolvekrans	W2	Witbank	41.0	6.4	6.4	6.2
n200	New Largo	W4	Witbank	43.0	4.8	6.2	2.8
s9	New Largo	W4	Witbank	50.5	5.4	6.4	2.6
s64	South Witbank	W4	Witbank	61.0	4.7	6.9	3.5
s119	W. Consolidated	W4	Witbank	41.1	4.3	6.4	3.1
m148	New Largo	W4	Witbank	28.5	3.8	5.8	2.7
m148a	New Largo	W4	Witbank	34.0	3.5	6.7	2.7
m148b	New Largo	W4	Witbank	34.0	3.5	6.7	2.7
m163	South Witbank	W4	Witbank	56.0	5.1	6.5	3.3
n198	New Largo	W4	Witbank	32.0	3.3	6.4	2.3
n199	New Largo	W4	Witbank	32.5	3.2	6.5	2.1
s42	South Witbank	W5	Witbank	53.3	5.2	6.4	3.7
s55	Blesbok	W5	Witbank	68.6	3.4	5.8	1.5
s58	South Witbank	W5	Witbank	57.9	5.2	6.4	5.5
m165	Springbok	W5	Witbank	22.0	3.5	6.5	1.6
n174	Matla	W5	Witbank	35.5	5.5	5.5	2.2
n185	Umgala	Alfred	Utrecht	101.0	9.0	6.0	3.8
n186	Umgala	Alfred	Utrecht	100.0	8.5	6.5	3.3
n187	Umgala	Alfred	Utrecht	97.0	9.0	6.6	3.7
n188	Umgala	Alfred	Utrecht	51.5	6.0	6.0	3.9
s66	Springfield	Main	South Rand	193.2	15.9	5.5	5.5
s67	Springfield	Main	South Rand	184.7	15.9	5.5	5.5
s122	Springfield	Main	South Rand	167.6	15.9	5.5	5.5
m168	Springfield	Main	South Rand	165.7	15.0	5.0	5.9
m169	Springfield	Main	South Rand	195.0	17.0	6.0	4.9
m170	Springfield	Main	South Rand	205.0	17.0	6.0	5.9
s126	Vierfontein	Main	Free State	87.8	6.1	6.1	2.0
n175	Springlake	Bottom	Klip River	70.0	7.5	5.0	1.8
n176	Springlake	Bottom	Klip River	63.5	7.5	5.0	2.1
s120	Cornelia	OFS1	Vaal Basin	128.0	9.8	5.5	3.7
m157	Sigma	OFS2	Vaal Basin	112.0	10.6	6.5	2.8
m159	Sigma	OFS2	Vaal Basin	108.0	10.6	6.5	3.2
n180	Sigma	OFS3	Vaal Basin	82.0	10.0	5.0	2.8
n181	Sigma	OFS3	Vaal Basin	96.0	12.0	6.0	2.9
n184	Sigma	OFS2a	Vaal Basin	112.0	11.5	5.5	2.9
n196	Sigma	OFS2a	Vaal Basin	104.0	12.0	6.0	3.0

The rate of scaling is determined using empirical methods that are proposed by Van der Merwe (2003). The rate of pillar scaling for Witbank Coalfields will be determined using equations 2 and 3 which are developed by Van der Merwe (2003):

$$R = 0.1624 \left[\frac{h}{T}\right]^{0.8135} \qquad \textbf{\textit{(2)}}$$

Where: R = rate of pillar scaling

h = mining height and

T= time since mining ceased in years.

The predicted amount of scaling, d_p, is then simply:

$$d_P = RT \text{ or } 0.1624h^{0.8135}\,T^{0.1865} \qquad \textbf{\textit{(3)}}$$

2.4 Impacts of Spontaneous Combustion in Opencast Mining

Pone (2008) simply defined spontaneous combustion of coal as the burning or smouldering of coal seam, coal stockpile or coal waste pile. The process of spontaneous combustion requires three factors to happen, that is, heat, oxygen and fuel. Figure 2-12 shows spontaneous combustion in a coal mine.

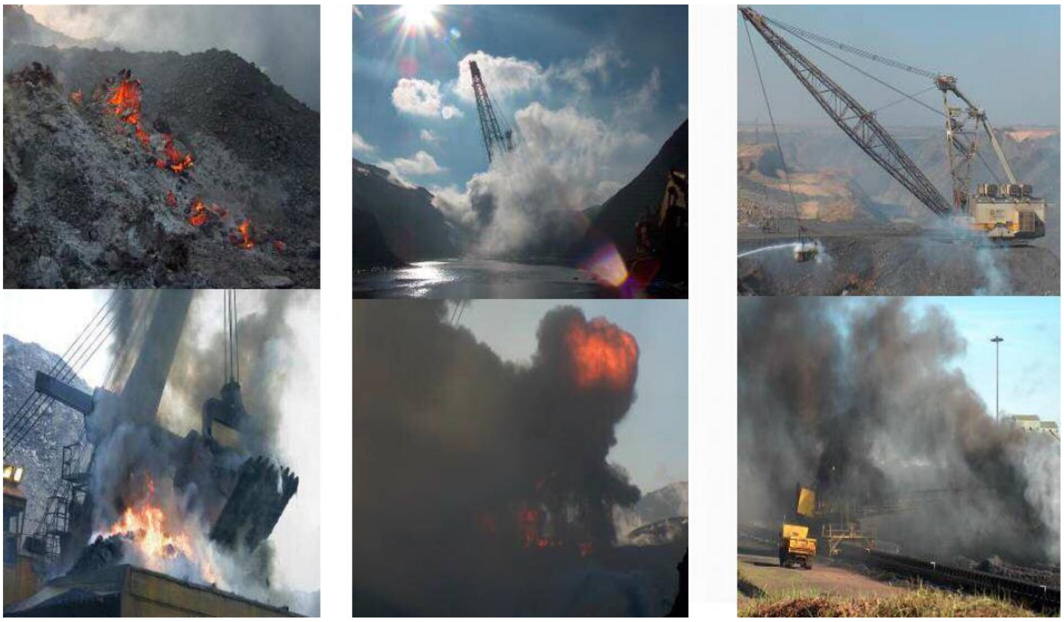

Figure 2-12: Spontaneous combustion on a coal mine in the Witbank coalfield (Miller, 2008)

The process that leads to spontaneous combustion in a coal mine is summarised as follows (coaltech, 2011):

- Oxidation occurs when oxygen reacts with the fuel, that is, coal;
- The oxidation process produces heat which is required to enable the spontaneous combustion process; and
- At higher temperatures, the oxidation reaction proceeds at higher rate. The higher the heat, the temperature raises exponentially. The rise in temperature is best explained by Arrhenius law, which states that the reaction rate increases exponentially and equation 4 gives a mathematical explanation of the reaction rate:

$$\nu = C_r C_0 A e^{\frac{E_a}{RT}} \quad \textbf{\textit{(4)}}$$

Where,

ν = Reaction rate in $molg^{-1}s^{-1}$

C_r = Combustible concentration (kg/m^3)

C_o = Oxygen concentration

A = Arrhenius frequency factor (s^{-1} or $s^{-1}c^{1-n}$)

E_a = Activation energy (kJ/mole)

R = Universal gas constant (8.314 $Jmolek^{-1}$)

T = Temperature (K)

2.4.1 Causes of Spontaneous Combustion in Underground Pillar Coal Mining through Opencast Mining Methods

Spontaneous combustion in coal mining, especially in mines that expose underground coal pillars through opencast mining methods is caused by the exposure of pillars to the factors required to make fire. The opencast mining methods let oxygen into the underground workings through cracks or unsealed over drilled overburden holes or workings not sealed on the highwall (Miller, 2008). Figure 2-13 shows an open crack on a mine that is exposing underground pillars.

Figure 2-13: Open crack on the burden above underground workings (Miller, 2008)

The cracks as shown in Figure 2-13, are generated by blasting of burden holes without presplitting and hence the explosive energy over-breaks beyond the desired mining dimensions (Miller, 2008). Presplits are usually drilled in areas where greenfield coal is mined and the process that is followed in greenfield operations is summarised as follows (Klipspruit Colliery, 2021):

- Wirelines are drilled ahead of mining. The wirelines are used to determine the top of coal contact, coal thickness and to check presence of in-seam parting;
- Presplit holes are then drilled and blasted before the commencement of infill drilling. This is done to create a safe highwall and assists in reducing crack developments due to back-breaks between the current strip and the next one. Presplit reduces back breaks because it allows explosive energy to escape without damaging the highwall; and
- Infill burden holes are drilled and blasted. The last two lines from the presplit are given slower delays so that enough time is given for material to move.

However, MMS drilling and blasting procedure for areas where underground coal pillars are exposed and mined is summarised as follows (Middelburg Mine Services, 2021a):

- Wirelines and presplits are not drilled as per greenfield coal operation procedure because presplit holes and a crack, as a result of presplit blasting will allow air into underground workings. Since wirelines are not drilled, the accuracy of drilling plans is compromised because exploration holes data are used in the creation of drilling plans. The exploration holes data makes the drilling plans inaccurate because the holes were drilled when the mine started and at a wider grid spacing;
- Infill holes are drilled and blasted. The infill holes are top primed; and
- The blasting of the infill holes without presplit creates cracks because there is no place where explosive energy can escape.

2.4.2 Impacts of Spontaneous Combustion in Coal Mines and Surrounding Communities

Miller (2008) and Dlamini (2008) have summarised the impacts of spontaneous combustion in coal mines that uses opencast methods to expose underground coal pillars and the surrounding communities as follows:

- Spontaneous combustion, together with external dilution factors decrease coal qualities such as calorific value, ash content, volatile matter and plant yield. Figure 2-14 shows the impact of spontaneous combustion on coal qualities in coal mines.

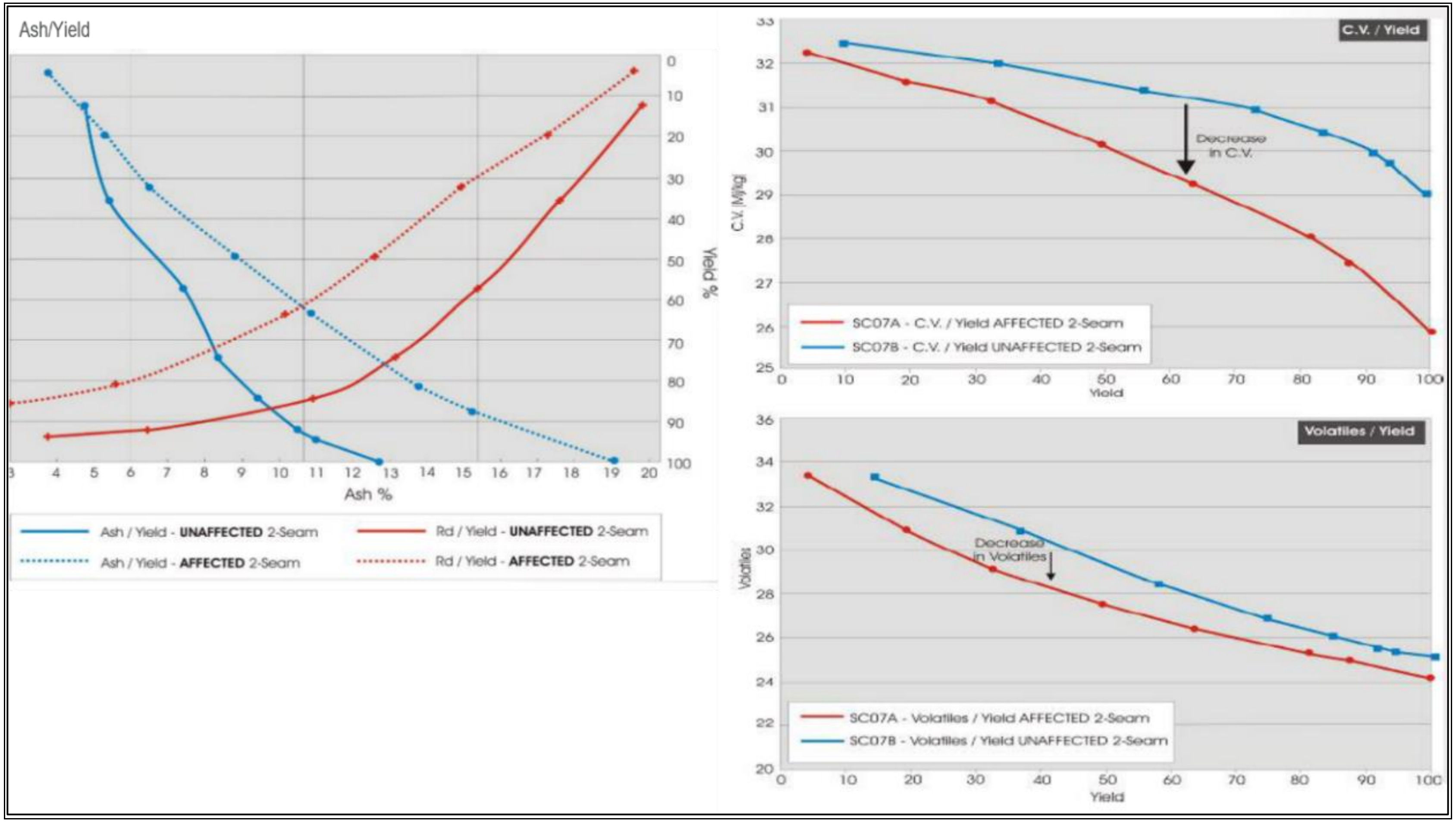

Figure 2-14: Impacts of spontaneous combustion on coal qualities (Dlamini, 2008).

Figures 2-14 shows that the coal qualities have dropped immensely when compared to the before mining coal qualities due to;

- Increased production costs due to machine damages. The machines, as shown in Figure 2-12, work in hot areas which are not primarily designed to work in such conditions;
- Increased top of coal contamination due to poor visibility from dragline operators;
- Decreased production rates because of poor visibility. The lesser production rates increase the life of mine, which will have a negative impact on NPV and IRR;
- Increased safety risks for the workforce due to charging and blasting of hot holes;
- Impact on the saleable produced due to impacts on yields and coal qualities;
- The processing plant is affected by the wet coal because cooling of the hot or burning coal might be required during coal load and haul. The wet coal in the processing plant blocks chutes, slippage on conveyor inclines above 15 degrees, tail blockages and build-up on take-up pulleys.
- On the surrounding communities, spontaneous combustion has health impacts because of the gases that coal procedures when it burns. The gases that are produced, with actual and required standard quantities are summarised in Table 2-6.

Table 2-6: Type of gases produced with actual and required quantities (Pone, 2008)

Compounds	Concentration (ppm)	US OSHA Standards (ppm)
	Witbank	
Benzene	7 ppm	
Toluene	5 ppm	200 ppm
Xylenes	1 ppm	100 ppm
CO	700 ppm	50 ppm
CH_4	2 000 ppm	0 ppm
CO_2	32 000 ppm	5 000 ppm

These gases cause mainly asthma and lung related illness. The health statistics recorded at clinics in and around Witbank show high rate of lung rated illnesses (Pone, 2008).

2.4.3 Ways to Reduce and Combat Spontaneous Combustion

Most mines that are currently exposing underground coal pillars through opencast mining methods control and combat spontaneous combustion through the following methods:

- Mining Methods: The mining methods that are commonly used in the exposure of underground pillars are the bord collapse and pillar collapse. These methods have been discussed in section 2.3 of this chapter. A buffer blast is blasted ahead of mining. A buffer is created by blasting one strip ahead of the current strip to be exposed. MMS currently leaves a buffer width of 30m and at an angle of respose of 55 degrees. However, some mines in the Witbank coalfields recommend that the buffer width must never be less than a third of the cut width (coaltech, 2011). If the cut width is 60m, then the minimum width is 20m. Coaltech (2011) further stated that ATCOM mine suggested that the buffer width should not be a function of cut width, but it should be a proportion of the height of burden above the seam;
- Cladding with softs: Highwall is covered with soft material after the exposure of the highwall. This material forms a layer which is made up of

particles that are closely spaced from each other. Figure 2-15 shows a highwall being cladded with soft material.

Figure 2-15: Cladding of an exposed highwall with soft material (Miller, 2008)

The effectiveness of cladding can be seen from Figure 2-15 because the smoke that was coming out from uncladded highwall was stopped with soft material.

- Sealing of cracks above underground workings: Cracks that are created by infill burden blasting are sealed immediately with soft material. Figure 2-16 shows a crack being covered or sealed with soft material.

Figure 2-16: Exposed crack being sealed with soft material (Miller, 2008)

- Sealing of the over drilled holes: The over drilled burden holes have a potential of letting oxygen into the underground workings due to pressure differences. The holes are plugged with steel plates. Figure 2-17 shows over drilled holes being sealed.

Figure 2-17: Sealing of over-drilled holes with a steel plates (Nel, 2008).

2.5 Factors that Cause Coal Losses

Coal losses, whether in mines that have greenfield reserves or underground coal pillars, are inevitable because of the mining methods and equipment being used to expose the coal. Coal losses are caused by some of the following factors and are briefly described in this section of the chapter:

- Blast shear;
- Coal scalping by exposure equipment; and
- Coal loss at the coal mixed zone.

2.5.1 Blast Shear

Blast shear is one of the factors that causes coal losses due to blasting. Figure 2-18 shows how blast shear causes coal losses during a blasting process.

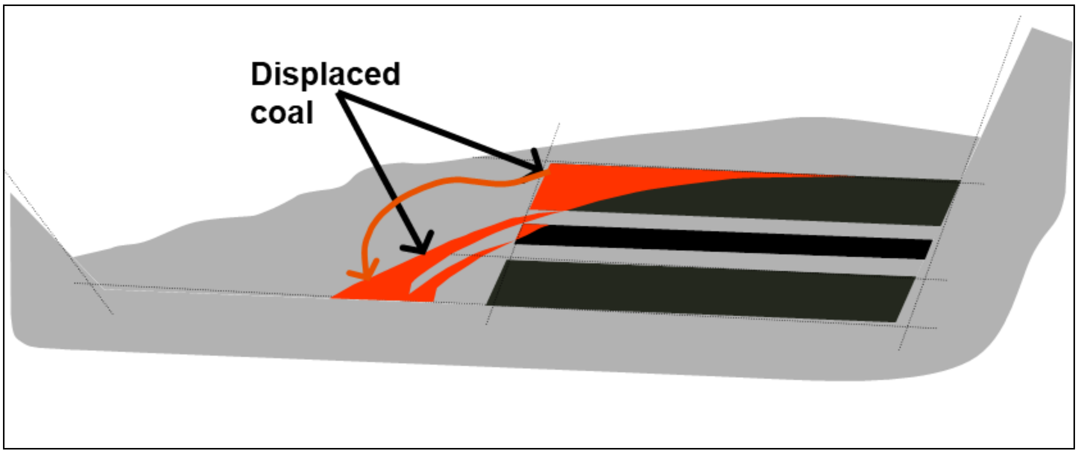

Figure 2-18: Blast shear process (BHP Billiton Mitsubishi Alliance (BMA), 2007)

Blast shear shatters the coal during blasting because of explosive shock energy and rock movement as shown in Figure 2-18. The main causes of this are the drilling of overburden holes to exactly the top of the coal and how the blast is charged and initiated. These two factors cause losses if the direction of the blast is moved towards the crest of the blast. Blast shear is mostly seen during cast blasting with the presence of a free face, but it also happens when the blast is initiated against buffer material.

2.5.2 Coal Scalping by Exposure Equipment

Coal scalping is usually caused by exposure equipment such as draglines, shovels and excavators digging into coal and Figure 2-19 shows exposure equipment digging into coal.

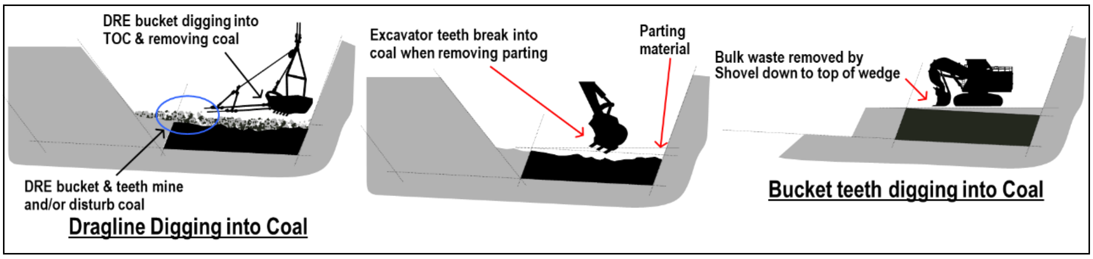

Figure 2-19: Exposure equipment digging into coal (BMA, 2007)

According to BMA (2007), equipment digging into coal is due to the following reasons:

- Non-compliance to the digging or stripping plan;

- Incorrect stripping plan sent to equipment; and
- Coal that is being exposed generally soft.

It is important to understand these factors because roof coal losses are mainly as a result of the above reasons.

2.5.3 Coal Loss at the Mixed Zone

A mixed zone is a zone where overburden material and coal get mixed, and it makes it nearly impossible for exposure equipment to separate coal and waste. The mixed zone can be seen in Figure 2-20.

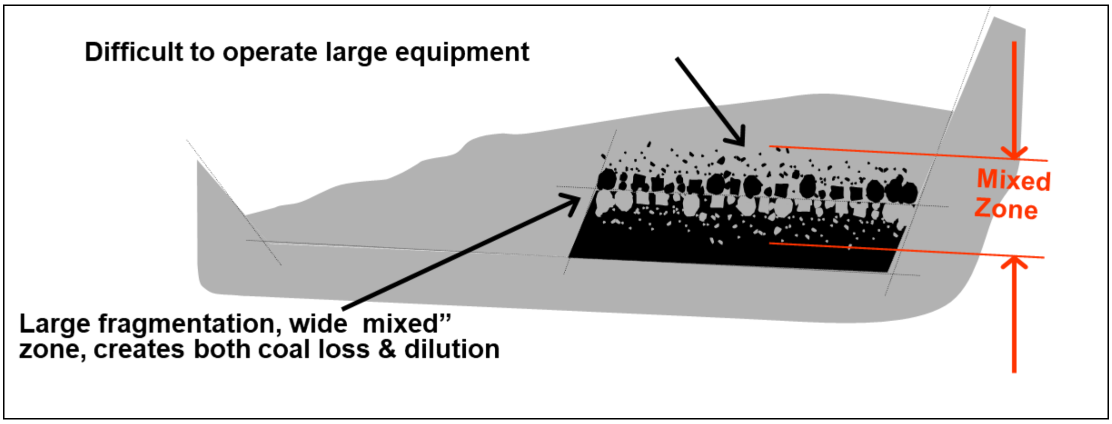

Figure 2-20: Mixed zones between overburden material and coal (BMA, 2007)

The coal losses in the mixed zone are due to following reasons:

- Over-drilling of the overburden holes;
- Incorrect backfill material used for overburden holes;
- Explosive energy escaping into coal due to inherent cracks in the overlying strata; and
- Coal being blasted with the overburden.

This is very important especially when pillars are imploded or when bords are collapsed because, in either of these processes, coal will have to be blasted with the overburden. The challenge is, how will the waste that is mixed with coal be cleaned before extraction or, is the coal in the mixed zone extracted without cleaning.

2.6 Benchmarking with other Coal Operations Mining Underground Coal Pillars through Opencast Mining Methods

This section discusses the benchmarking of MMS's existing pillar mining method with other mines that have or are currently exposing underground coal pillars through opencast mining methods. The first thing will be to briefly discuss the existing methods at MMS South Pits and then give details on the mines that are compared with MMS South Pits. The mines that will be discussed in this section are Tweefontein Colliery's Boschmans North (BN) pit and Glisa Colliery, owned by Eyesizwe Coal (Pty).

2.6.1 MMS South Pits Coal Pillar Methods

MMS South has four pits that expose underground coal pillars through opencast mining methods. These pits use draglines and truck and shovel to expose some virgin coal and underground coal pillars. The four pits treat coal pillars differently. Two pits on the eastern side of MMS south, that is Boschmanskrans (BMK) West and East apply the bord collapse method. The bords are collapsed because the eastern area is considered hot area and the coal pillars are mined with waste (Rorke *et al*, 2016). Figure 2-21 shows how burden holes are drilled and initiated.

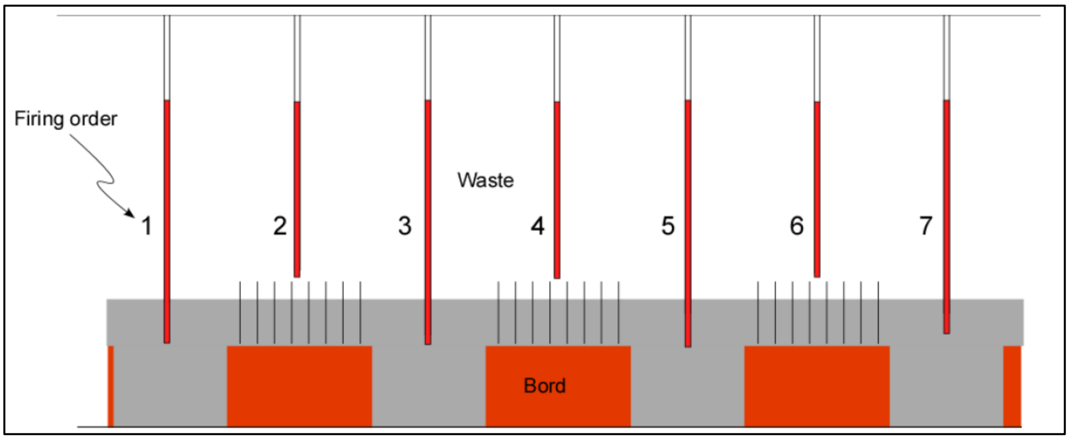

Figure 2-21: BMK West and East planned drilled depth and sequence of firing burden holes (Rorke *et al*, 2016)

According to Rorke *et al* (2016), burden holes do not penetrate the pillars, and pillars may or may not fragment during blast depending on the coal strength, pillar height and the amount of collapse in the bords caused by scaling. As shown in Figure 2-21, the holes are fired sequentially. The bords are expected to collapse due to the dead weight of the overlying strata. Figure 2-22 shows the estimated distribution of coal and waste after the blast.

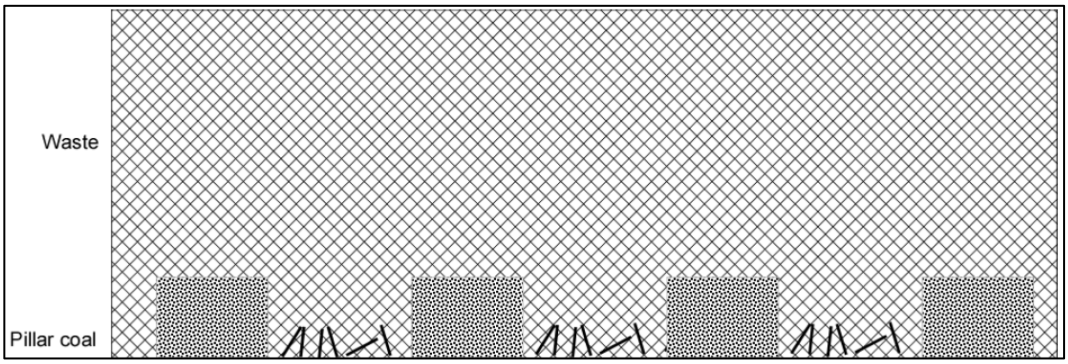

Figure 2-22: Estimated distribution of coal and waste after the blast (Rorke *et al*, 2016)

This method will require selective mining of the coal to minimise dilution and may require post blasting of the coal pillars (Rorke *et al*, 2016). However, selective mining in both pits is not done, the coal pillars are mined with the waste and fed to the processing plant.

The pits on the south eastern side of MMS South apply the pillar collapse method. Figure 2-23 shows fully collapsed pillars with shale contamination at the mining face.

Figure 2-23: Collapsed coal pillar with shale contamination

Figure 2-23 also shows that coal is completely mixed with waste which makes it difficult to selectively mine coal to the processing plant, and waste moved to the waste dump. The shale contamination makes it difficult to even separate with a rotary breaker because it's relative density is almost like coal relative density of between 1.52 and 1.69 t/m^3.

2.6.2 Tweefontein Colliery

Tweefontein Colliery is a coal mine which is owned by Glencore Plc and it is located in the Witbank coalfields. Tweefontein exposes underground pillars through opencast mining methods in one of their pits called Boschmans North (BN) (Tweefontein Colliery, 2013). Figure 2-24 shows the plan of underground pillars at BN pit.

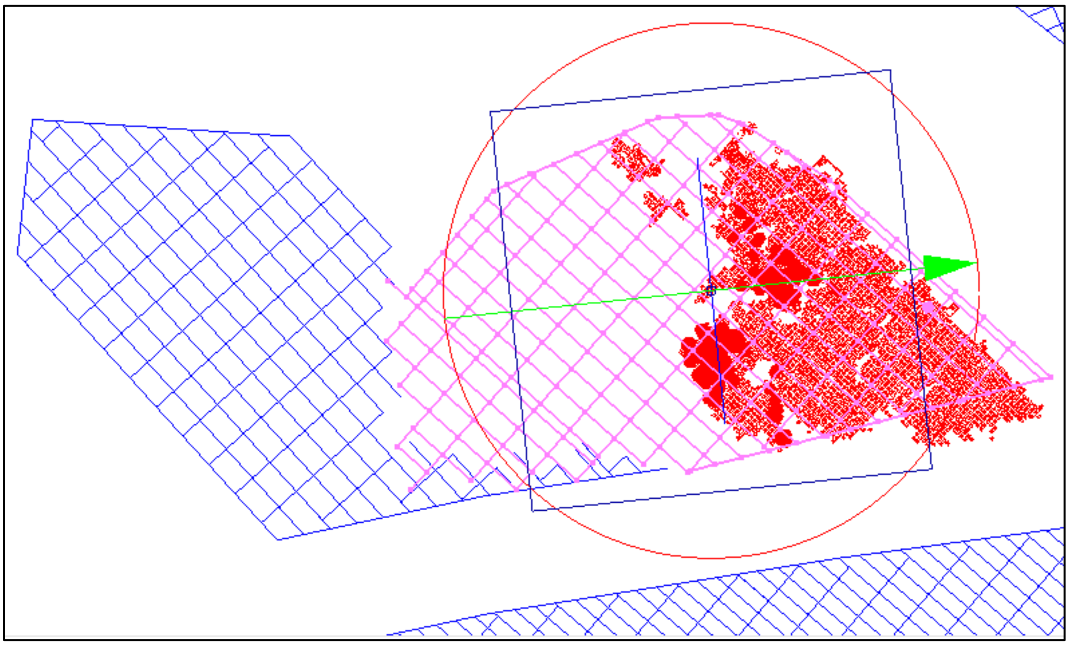

Figure 2-24: BN pit underground workings (Tweefontein Colliery, 2013)

Tweefontein Colliery trialled four different pillar methods in the BN pit and these methods were trialled out one at a time to see if the trial was a success. The four methods trialled at BN pit are described as follows:

Method 1:

In method 1, a face shovel or excavator mines soft material and exposes the hard burden. Figure 2-25 shows the face shovel prestripping soft material up to hard burden.

Figure 2-25: Face shovel prestripping soft material up to hard burden (Tweefontein Colliery, 2013)

Once the soft material is removed, the hard burden is drilled to top of coal and blasted. Figure 2-26 shows the drilling of the hard burden up to top of coal.

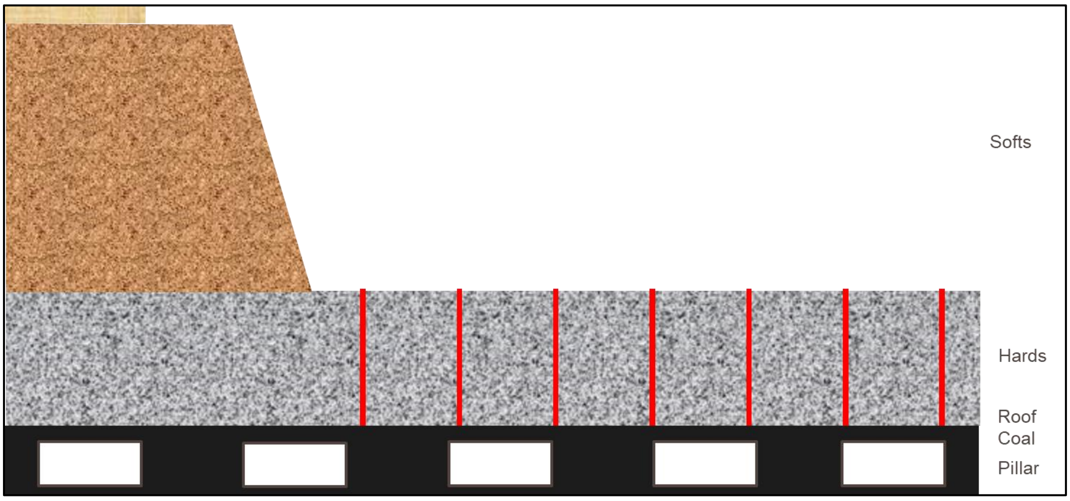

Figure 2-26: Drilling of hard burden to top of coal (Tweefontein Colliery, 2013)

The bords are expected to fail after the blast due to the dead weight of the overlying blasted burden material. The bords did not fail as planned because one of the face shovel's tramming tracks subsided into the bord of the underground workings and this is shown in Figure 2-27. The sinkhole dimensions were found to be 4 meters by 3 meters. The failure of the hard burden took place while the face shovel was loading the soft material.

Figure 2-27: Face shovel tracks subsided into the underground workings (Tweefontein Colliery, 2013)

Tweefontein Colliery investigated the incident shown in Figure 2-27 and the following factors were found to be the main contributing factors (Tweefontein Colliery, 2013):

- A minimum beam that will be able to support the load of face shovel was not determined, instead an assumption was made that the hard burden would be able to support the weight of the equipment;
- The weight and ground bearing pressure were found to be higher than the strength of the hard burden;
- The hard burden thickness was found to be 4.3 m thick and the expected burden thickness was estimated to be in a range of 5m to 10m; and
- Suspected pillar scaling might have taken place because the coal pillars were mined in the late 1940’s and early 1950’s using conventional drill and blast methods.

Method 2:

After the investigation of the incident due to Method 1, the mine modified Method 1 and called it Method 2. Method 2’s plan was to drill and blast softs and hard burden together and planned drilling strategy for method 2 is shown in Figure 2-28 (Tweefontein Colliery, 2013).

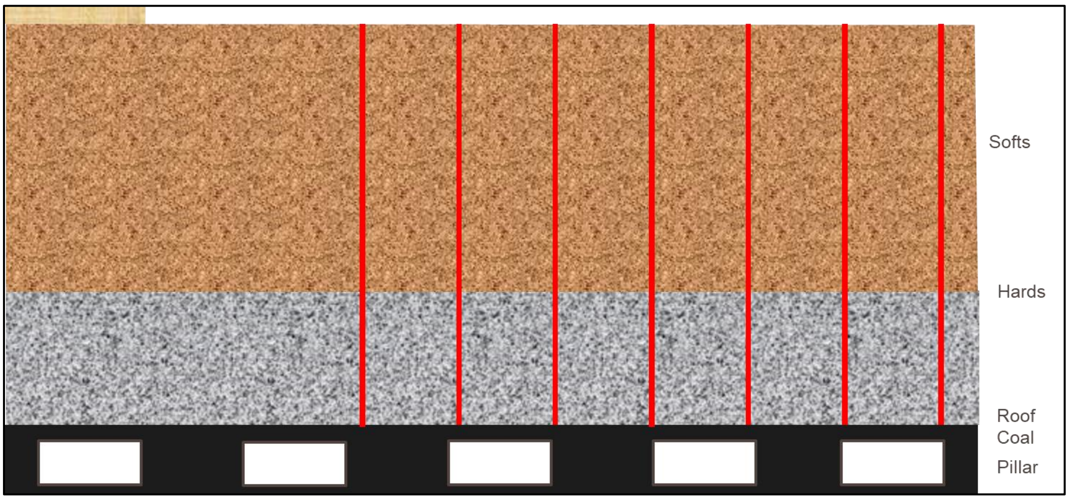

Figure 2-28: Method 2 drilling and blasting strategy (Tweefontein Colliery, 2013)

The assumption in this method was that the bords will collapse due to the dead weight of the blasted burden (Tweefontein Colliery, 2013). Like Method 1, Method 2 was unsuccessful because a truck and face shovel got bogged into the bord of underground workings. The two incidents that happened due to method 2 application are shown in Figure 2-29.

Figure 2-29: Face shovel and truck incident due to method 2 (Tweefontein Colliery, 2013).

After the investigation of method 2 incident, the mine came up with the following remedial actions (Tweefontein Colliery, 2013):

- Change mining method and drill holes into the pillars only; and
- Digitising the cloth plans since these plans were done in the 1950's.

Method 3:

In Method 3, soft, hard burden and coal pillars are drilled and blasted together. Figure 2-30 shows the drilling process in method 3.

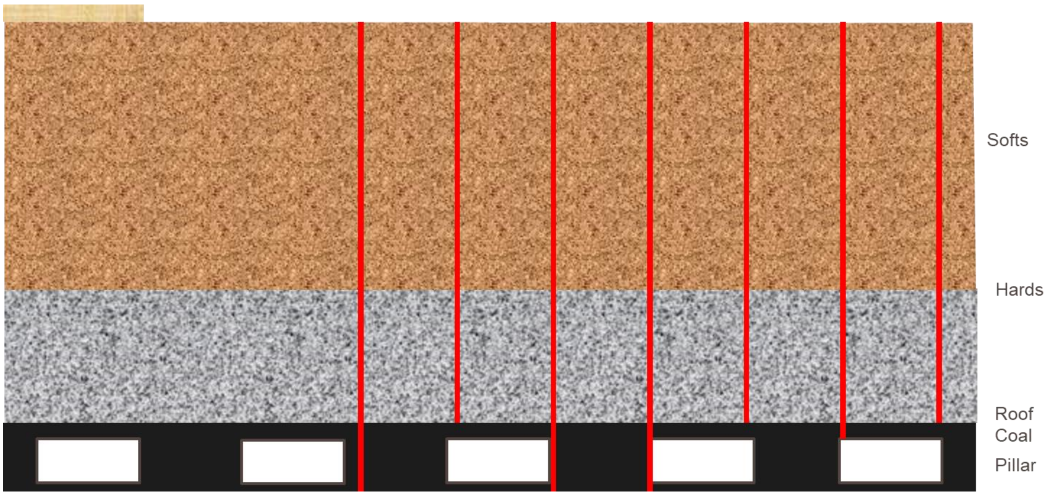

Figure 2-30: Method 3 drilling and blasting process (Tweefontein Colliery, 2013)

Again, in this method, the bords were assumed to have failed due to dead weight of the blasted overburden material. Like with the previous methods, that is Methods 1 and 2, the face shovel track subsided into the bord of underground workings. The mine investigated the incident and recommended another method change. This method would become Method 4.

Method 4:

In Method 4, soft, burden, coal pillars and bords are drilled and blasted together in at the same time. Figure 2-31 shows the drilling process for method 4.

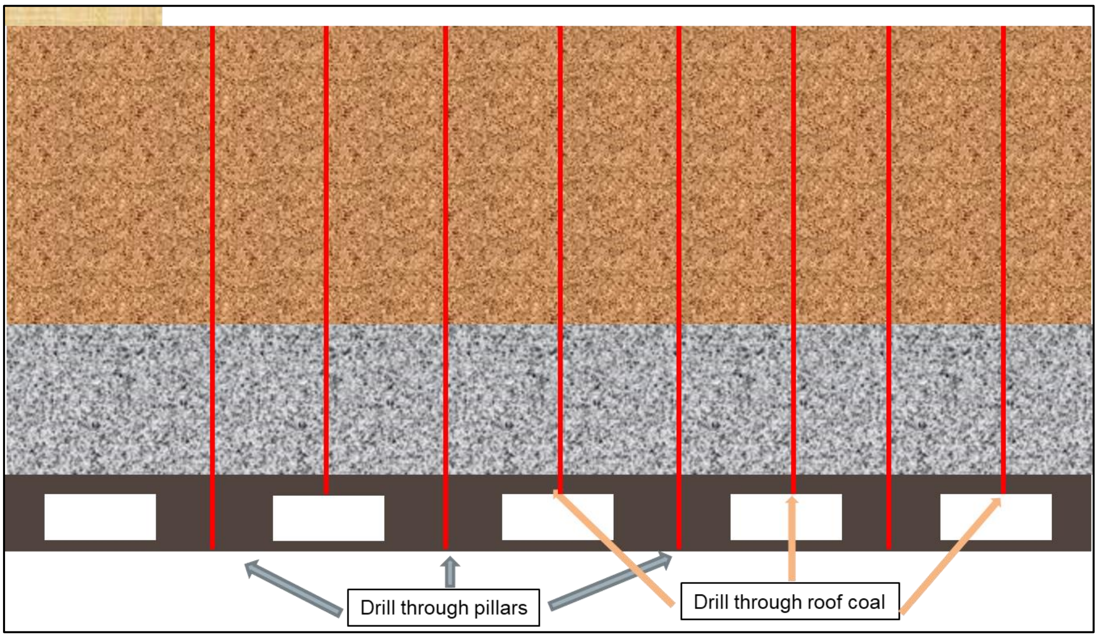

Figure 2-31: Method 4 drilling strategy (Tweefontein Colliery, 2013)

The bords are collapsed by drilling holes into the roof of the bords and then charge with explosives. This method does not rely on the dead weight of the overburden material to collapse the bord. The exposed coal for this method is shown in Figure 2-32.


Figure 2-32: Coal profile after loading out waste material (Tweefontein Colliery, 2013)

2.6.3 Glisa Colliery

Glisa Colliery is one of the mines owned by Eyesizwe Coal (Pty) and located in the Belfast coalfields, in Mpumalanga. Glisa Colliery exposes No. 2 seam coal pillars through opencast mining methods. The method that is used is the preservation of underground workings by leaving a beam made up of coal and waste material. Figure 2-33 shows the method used at Glisa Colliery and mining sequence that is followed.

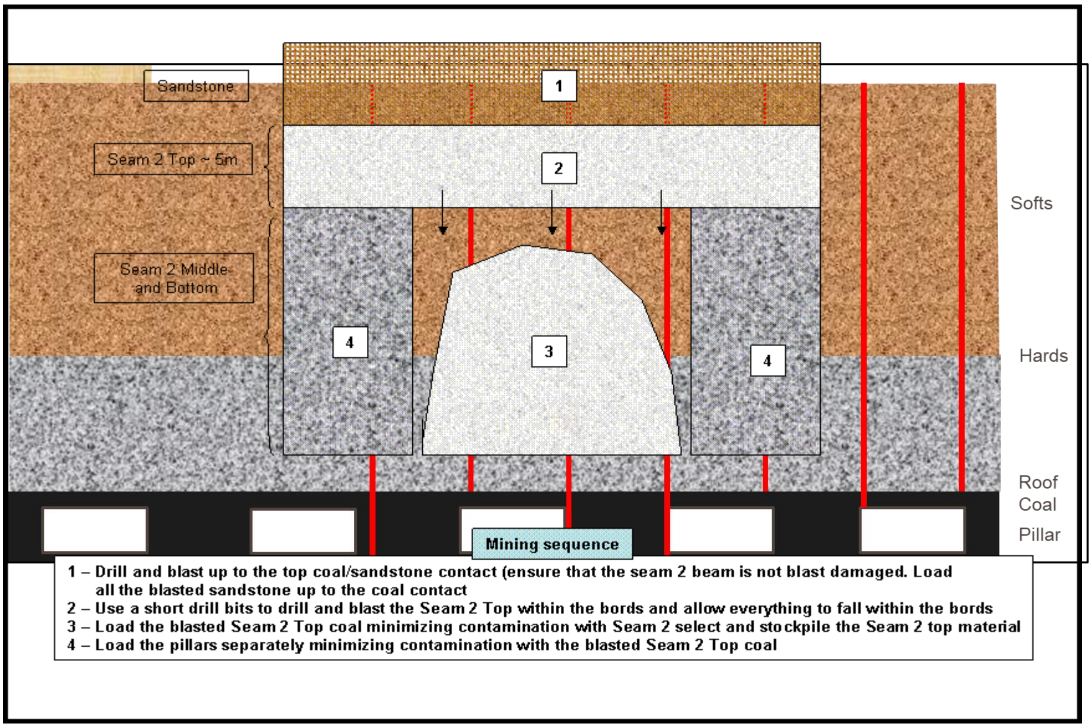

Figure 2-33: Glisa mining method and mining sequence (Maphanga, 2008)

The beam, as shown in Figure 2-33, was determined to be 3.75m (Maphanga, 2008). The beam at Glisa Colliery was determined using equation 5 (Maphanga, 2008):

$$t = \frac{qgL^2+(F_{Truck}/L)}{2\sigma_{tp}} \qquad \textbf{\textit{(5)}}$$

Where,

t = Beam thickness (m)

q = Density of parting material (kg/m^3)

g = Gravitational acceleration (9.81 m/s^2)

L = Span (m)

F_{Truck} = Surcharge load (MN)

σ_{tp} = Tensile strength of parting material (MPa)

2.7 Multi Criteria Decision Analysis (MCDA): Review and Select Underground Coal Pillar Mining Methods

Multi criteria decision analysis (MCDA) techniques are a class of multi-criteria decision making (MCDM) (Mahase, 2017). MCDM is a process that is used to solve problems with more than one objective (Mahase, 2017 cited Pohekar & Ramachanran, 2003). Mahase (2017) has found that there are 246 case studies that were done using MCDA's and 8 of the MCDA are mostly used in most case studies in the mining industry. The MCDA that are commonly used in mining industry are as follows (Mahase, 2017):

- Analytical Hierarchy Process (AHP);
- Elimination and Choice Expressing Reality (ELECTRE);
- Multi-Attribute Utility Theory (MAUT);
- Preference Ranking Organisation Method for Enrichment Evaluation (PROMETHEE);
- Simple Additive Weighting Method (SAW);
- Technique for Order Preference by Similarity to Ideal Solutions (TOPSIS); and
- Vise Kriterijumska Optimizacija I Kompromisno Resenje (VIKOR)

This chapter does not aim to go into detail on all the listed MCDA methods but will only discuss two preferred methods because they are easy to execute and they have been extensively used in mining industry to solve complex problems. The two MCDA methods that will be used are AHP and VIKOR.

2.7.1 VIKOR

VIKOR was developed in 1998 by Serafim Opricovic (Gelvez and Aldana, 2014). It was developed for multi criteria optimisation of complex systems. This method determines the compromise ranking list, the compromise solution and weight stability intervals for preference stability of the compromise solution. VIKOR procedure is summarised as follows in a sequential form (Gelvez and Aldana, 2014):

- Determine the alternatives to be compared. The alternative in this case will be methods that will be used in dealing with underground coal pillars;
- Set-out criteria to be used to compare the alternatives;
- Determine the best and worst values of all criterion function i = 1; 2; n;
- Compute the values S_j and R_j, j = 1; 2…. j by using the equations 6 and 7:

$$S_j = \sum_{i=1}^{n} w_i (f_i^* - f_{ij}) / (f_i^* - f_i^-) \qquad \textit{(6)}$$

Where,

w_i = weighting;

f^*_i = best alternative;

f_{ij} = alternative value for each criterion; and

f^-_i = worst alternative

$$R_j = max_i \left[w_i (f_i^* - f_{ij}) / (f_i^* - f_i^-) \right] \quad \textbf{\textit{(7)}}$$

- Compute Q_j values by using equation 8:

$$Q_j = \frac{v(S_j - S^*)}{(S^- - S^*)} + (1 - v)\frac{(R_j - R^*)}{(R^- - R^*)} \quad \textbf{\textit{(8)}}$$

Where v is a constant and the best value was determined to be 0.5.

- Rank the alternatives, sorting by the values of S, R and Q in decreasing order; and
- The compromise solution, A(1) is equal to the Q(min) if the acceptable advantage Q(A2) – Q(A1) ≥ DQ, where DQ is equal to 1/(j-1) and A(2) is the second alternative in the Q ranking and the acceptable stability in decision making where A(1) must be the best also in S or R.

2.7.2 Analytical Hierarchy Process (AHP)

AHP was developed in 1990 by Saaty (Abdalla *et al*, 2013). AHP is a tool that is used to combine qualitative and quantitative factor in the selection of a process. Abdalla *et al* (2013) cited Musingwini and Minnit (2008) that AHP uses mathematical frameworks which start with pairwise comparison of the relative weight of each criterion over another. The comparison is made with the use of scaling numbers which indicate the weight and dominancy of an element over another element with respect to the criterion to which they are compared. The scaling numbers and their descriptions are shown in Table 2-7.

Table 2-7: Saaty's scale (Atanasova-Pachemeska *et al*, 2014)

Importance	Definition	Explanation
1	Equally important	Both elements have equal contribution in the objective
3	Moderately important	Moderate advantage of the one element compared to the other
5	Strongly important	Strong favouring of one element compared to the other
7	Very strong and proven importance	One element is strongly favoured and has domination in practice, compared to the other element
9	Extreme importance	One element is favoured in comparison with the other, based on strongly proved evidences and facts
2,4,6,8	Inter values	

In this research study, AHP will be used to determine the weighting that is required in VIKOR. It is used to determine the weighting because the ratings of each criterion are done by personnel who have relevant experience in the field being reviewed. The other reason for AHP preference is because of its ability to determine if judgements are reasonably consistent and unbiased (Musingwini, 2010). To check if the judgements are consistent, a Consistency Ratio (CR) is calculated by assessing the calculated Consistency Index (CI) against judgements that are made completely at random (Musingwini, 2010). The CI is calculated using equation 9 (Musingwini, 2010):

$$\mathrm{CI} = \frac{\lambda\mathrm{max} - n}{\mathrm{n} - 1} \qquad \textbf{\textit{(9)}}$$

Where,

λmax = Eigenvalue constant; and n = number of criteria.

Once CI is calculated, a random indices (RI) is determined using random consistency index. Table 2-8 shows random consistency index as determined by Saaty.

Table 2-8: Random index (RI) for n-ordered matrix (Musingwini, 2010)

Matrix Order	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
RI	0.00	0.00	0.58	0.90	1.12	1.24	1.32	1.41	1.45	1.49	1.51	1.48	1.56	1.57	1.59

To check if the judgements are consistent, the calculated CR must be less than or equal to 0.1 and if it is greater than 0.1, the rating is inconsistent, and it means that the rating of each criterion must be redone (Atanasova-Pachemeska *et al*, 2014).

Musingwini (2010) summarised the step that are followed when using AHP:

- Map objectives into criteria and then normalise the criteria. Normalising of the criteria removes different units;
- Assign a weight (w_j) to each criterion (C_j) such that $\sum_{j=1}^{n} wj = 1$;
- Aggregate weights and normalise outcomes (O_{ij}) values to get value function $V(A_i)$ for each alternative (A_i); and
- Execute decision by selecting alternative with highest value function $V(A_i)$, that is, AHP priority score.

2.8 Chapter Summary

The aim of this chapter was to review and identify factors that would lead to high coal dilution and losses when opencast mining methods are applied. It has been found that dilution happens in two ways, that is, internal and external. However, the main contributing factors are those that cause external dilution such as mining methods, human behaviours due to tonnage-based incentive schemes and the type of equipment used to expose underground coal pillars. It was also found that external dilution can be reduced by doing selective mining and using fit for purpose mining methods and equipment, but internal dilution was found to be the main challenging one because it is inherent in a coal seam. Dilution and spontaneous combustion were found to have detrimental impact on the coal mine because it lowers the coal qualities, increases operating costs due to high maintenance costs of equipment, causes health issues to employees and surrounding communities.

In this chapter, it has also been found that studies were previously done to determine various methods that deal with dilution and spontaneous combustion. Six methods were developed by Clough and Morris in 1985 and each method has its challenges that must be resolved. However, no evidence was found that was used to select these method for a mine that was planning to expose underground coal pillars through opencast mining methods. Some mines used trial and error methods to determine a mining method and through trial and error, mines had safety incidents where machines got bogged into underground bords. It is for this reason that multi criteria decision methods were discussed. Their aim is to review the existing methods with appropriate criteria and then compare the method used with other alternative methods.

In terms of coal losses, mines whether mining green-field coal or underground bords and pillars, lose coal in a similar fashion. The main ways that coal mines lose coal were found to be due to blast shear, equipment type used, and coal loss due mixed zone caused by over-drilling and blasting of coal pillars with the waste.

3 RESEARCH METHODOLOGY

3.1 Introduction

This chapter gives an overview of the methodology that was used to achieve results in this research study and presents the results from these methods. It also aims to give details on how each methodology that was used, has been set up. The methods that will be discussed in this research study are as follows:

- Gathering of information from previous reports;
- MCDA techniques were selected to review and select appropriate methods for MMS,
- Model set-up for the analysis of pillar behavior when overburden blasting takes place;
- Coal pillar profile simulation model set-up; and
- How samples were taken to determine the impact of current practice on coal pillar implosion.

3.2 Gathering of Information from Previous Reports

Some of the information was collected from previous survey reports and charging instructions supplied to the execution team by the drilling and blasting planners. The information collected from the reports includes the following:

- Coal losses;
- Historic production statistics;
- Actual plant yields;
- Saleable coal produced; and
- Production budget numbers

3.3 MCDA techniques were used to review and select an appropriate method for MMS

As stated in section 2.7 of Chapter 2 of this research study, MCDA techniques will be used to review and select appropriate methods in dealing with underground pillars when opencast mining methods are used for exposure. Mahase (2017) identified eight commonly used MCDA techniques in mine planning and two of the eight methods were preferred for this review and selection. The two methods are preferred because they require ratings of the criteria by people who are experienced in the field that is being investigated in this research study and due to ease of application of the two methods. The selected methods, as discussed in the literature review, are AHP and VIKOR.

VIKOR and AHP require the following steps to effectively work in this research study. The steps of how each method will be used are summarised as follows:

Step 1: Determine alternative methods

Step 1 is to select appropriate alternative methods to be compared against each other. The alternative methods are selected as described in section 2.7 of Chapter 2.

Step 2: Identify criteria to be used to compare alternative methods

The criteria are selected based on the importance and impacts it has on the mining of underground coal pillars when opencast methods are used. The criteria for this research study will be the ones that will have a greater impact on the profitability and operability of the mine. The main factors are summarised as follows:

- Calorific Value impact: This is a measure of the heat content or energy value of coal (Falcon and Ham, 1988). This is the most important commercial parameter for coal and it is expressed in mega Joules per kilogram of coal;
- Ash content impacts: Ash content is the residue that remains after the complete combustion of coal (Falcon and Ham, 1988). This is important because ash content will impact the amount of ash that will need to be placed in the dump and has the potential to increase discard material. It is much better for the end-users of coal because less ash content will reduce environmental liability;
- Overall plant yield: This criterion is imperative because it is an indicator of how much saleable product will be produced when underground coal pillars with waste material are sent to the processing plant. The lesser the saleable product, the returns on margins will also be lower;
- Dilution: This criterion is aimed at determining the dilution percentage for each alternative mining method;
- Coal losses: the Coal loss percentage for each method is calculated;
- Required technical skills on blasting and exposure workforce: This is having to do with the required skill of the personnel that will plan and execute the preferred alternative

mining method. This criterion will be determined with a rating and Table 3-1 shows the rating matrix that will be used.

Table 3-1: Scale Matrix

Scale	**Meaning**
1	Not preferred
2	Mildly preferred
3	Moderately preferred
4	Greatly preferred
5	Always preferred

- Application of method on hot areas: One would want to apply a method that will be easy and practical in all areas in the mine. This is important because it will reduce skill sets that are required to execute the method;
- Amount of coal fines: This is to determine how much each method will generate fines if it is selected.

Step 3: Quantify the values of each alternative

To carry out the review and selection of an appropriate method to treat pillars, the values of each alternative, based on the criterion are determined by using different quantification methods.

Step 4: Rank and calculate the weighting score of each criterion

The selected criteria are ranked by experts to determine the importance of each criterion by using a score from 1 – 10, with 1 carrying the least weighting and 10 carrying the highest weighting. The weightings are then analysed using AHP to determine the biasedness of the expected weightings. If the weightings are found to be biased, a new weighting must be done.

Step 5: Rank the alternatives

VIKOR is then used to rank and select the appropriate method to be applied at MMS.

3.4 Model set-up for the Analysis of Pillar Behavior when Overburden Blasting takes place

The model was set up to determine how bord failures occur during overburden blasting and to understand the impact of pillar scaling when coal pillars are imploded. The model was set up using science RS2 software. The input parameters that were used in the set-up of the model are as follows:

- Pillar centers are 15 m;
- Depth from surface to no.2 seam floor is 25 m;
- Pillar size used: 8 m wide and 8 m length;
- Coal thickness: 6 m;
- Mining height: 4 m; and
- Bord width: 7 m.

3.5 Coal Pillar Profile Simulation Model Set-Up

The profile of the coal pillar when blasted is simulated using 3DDig software and the model is set up as follows:

- The software requires the following inputs for simulations: current pit topography, top, and bottom of the coal seams, the relative density of each seam, and survey pillar plan in a dxf format;
- The next step is to remove the overburden material to expose the coal seam;
- Then, the coal pillars are created using a cut tool in 3DDig;
- Once pillars are created, the pillars are then blasted;
- The blasted overburden is then placed on top of the blasted coal pillars and
- The blasted coal pillars are then exposed using either draglines or shovels.

3.6 Sampling Done to Determine the Impacts of Pillar Implosion

Coal samples were taken to determine the impacts of pillar implosion. Samples were taken in blocks where pillars were fully imploded, half imploded pillars, and non-imploded pillars. Figure 3-1 is a diagram illustrating the points in blue with a capital letter "T" and a number where grab samples were collected along strip 20, 21, and part of strip 22 in the various blocks as the coal faces advance

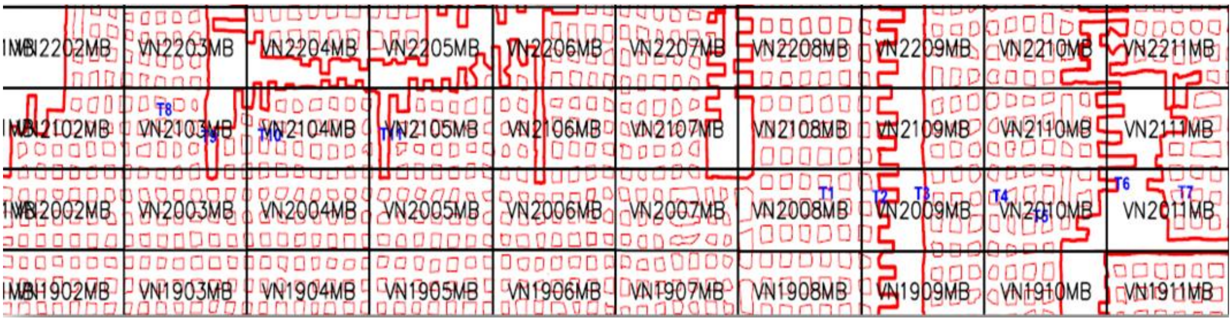

Figure 3-1: The sampling points for various blocks at Van Dyk's Drift North (VDDN) pit

Each sample consists of 3 to 4 bags with a combined weight of approximately 45 kg to 60 kg. Samples are taken across the entire mining face and height. This is to ensure that the sample is representative of the entire mining face.

Figure 3-2 shows the No. 2 seam mid-burden mining layout, the blocks in the yellow rectangle are the fully imploded blocks, the blocks in the green rectangle are the half-imploded blocks (transition from imploded pillars to non-imploded pillars), while the blocks in the blue rectangle are the non-imploded blocks.

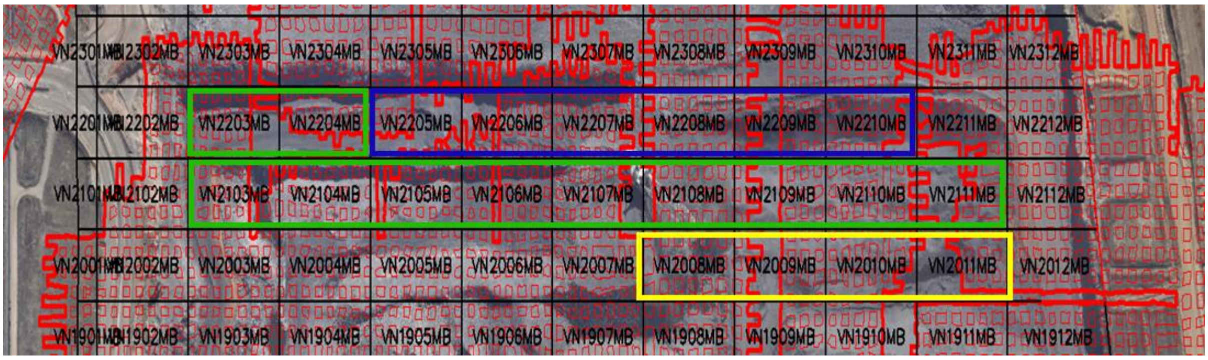

Figure 3-2: Samples taken in fully, half and non-imploded pillars

If during the sampling of the mining face, excessive contamination is identified, the production team was notified to ensure that the material does not end up at the crushing plant.

The plant laboratory collects the samples on the same day. The laboratory will crush the entire grab sample to 25 mm and split the sample into a representative 6 kg samples using a rotary splitter. The proximate and washability analysis are then done on the representative sample. The proximate analysis consists of calorific value, inherent moisture, ash content, volatile matter, fixed carbon, and total sulphur. The washability analysis is a process whereby coal samples are separated into float and sink fractions in liquids of increasing density.

There is a 24-hour turnaround time for laboratory results. This is quite important because the plant might still be receiving this material from the coal face. The plant can use the laboratory washability results to make an informed decision on what relative density to wash the VDDN S2 material to achieve a plant yield greater than 40%.

3.7 Presentation of Results

The results obtained using various methods are presented in Appendices A to M. The description of what is contained in each Appendix is summarised in Table 3-2.

Table 3-2: List of Appendices

Appendix	Information contained in the Appendix
A	Coal Loss results from the Financial year 2018 to the Financial year 2020
B	Financial Years 18 and 19 Production physicals (Budget versus Actual)
C	Charging instructions for VDDN and BMK East pits
D	Job profile and qualifications for a surface blaster
E	Job profile and qualifications for a blasting assistant
F	Strip Orientation results
G	Results of the coal samples taken on imploded blocks, half imploded blocks, and non-imploded blocks
H	Bonus scheme amendment memorandum
I	Criteria results for various coal pillar methods
J	AHP Results
K	Calculated Results using VIKOR
L	Margin calculation for a mine with a strip ratio of 5.5 BCM/ton
M	Amount of coal loss and dilution

3.8 Chapter Summary

This chapter explained the methodology used to obtain the results in this research study and presented these results. The methods used include MCDA, information gathering from previous reports such as survey and budget reports; set-up of models used in the simulation of bord collapse and pillar implosion. The obtained results were summarised in the Appendices.

4 ANALYSIS OF THE RESULTS

4.1 Introduction

This chapter presents the analysis of the results obtained and the results of this research study have been summarised in Appendices A to M. The analysis will focus on coal losses, dilution causes, review of methods used to treat pillars, and other topics that are related to coal pillar mining such as buffer width required to prevent spontaneous combustion and technology currently used to do spot checks on the location of the pillars.

4.2 Coal Loss Analysis

The coal loss results in Appendix A show that MMS has lost coal due to the following reasons:

- Coal left on the floor;
- Coal loss through the roof of coal;
- Coal not exposed;
- Coal exposed but not mined; and
- Geotechnical and geology factors.

Figure 4-1 shows the summary of coal lost from financial year 18 to 20 due to the above reasons.

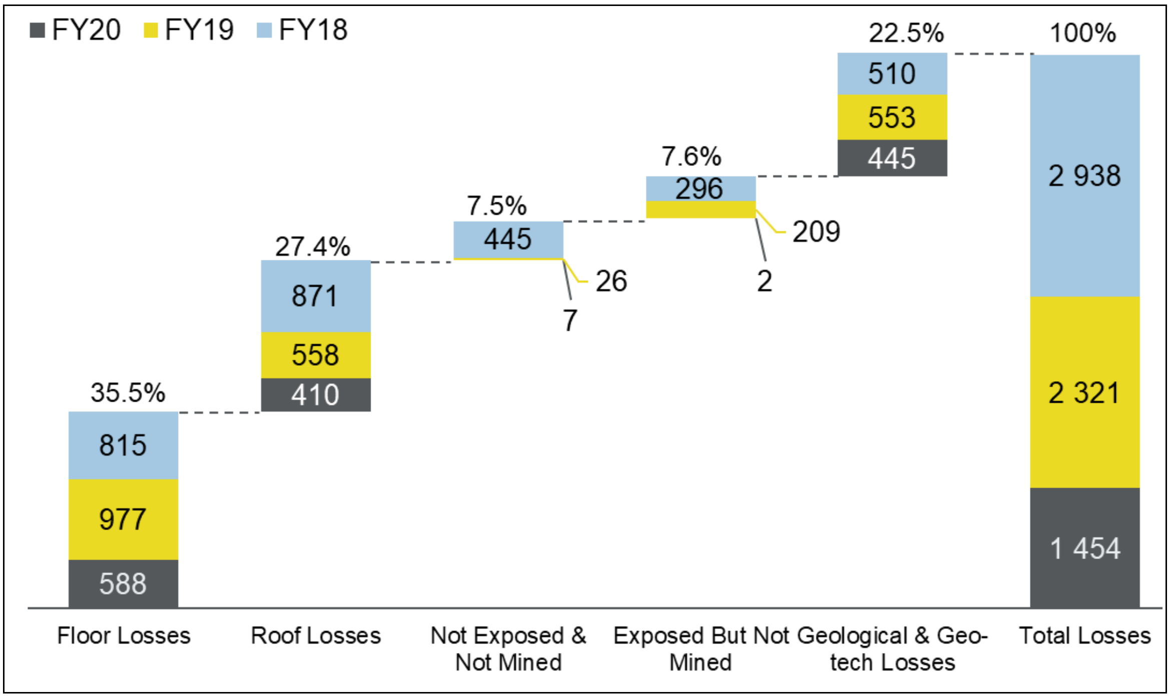

Figure 4-1: MMS's actual coal losses from financial year 18 to 20

The main areas where coal losses happened at MMS were between floor, roof, geotechnical, and geology factors. These areas and factors contributed 85.4% of the total coal losses in the previous financial years, with floor coal being the major contributor, followed by roof coal losses and then by the combination of both geotechnical and geology factors. However, there was an improvement on year on year losses because overall losses dropped by an average of 43.95% per year. The decrease in coal losses, especially in FY20 was due to mining more coal in a pit with less underground coal pillars and water challenges. In this pit alone, the overall ROM mined exceeded the budgeted ROM tons by 30.63%. This section will discuss how coal in these areas was lost.

4.2.1 Roof Coal Loses

As it has been discussed in Chapter 2.3, the roof coal losses are caused by blast shear and exposure equipment such as draglines and shovels digging into coal because coal has been blasted with waste in some cases. The blasting of coal with waste makes it extremely difficult for draglines to separate coal from waste because coal has mixed with waste.

However, the least cause of coal losses at MMS is blast shear because the burden blasts of both upper and lower burdens have been confined by the buffer from the prior blast. The roof coal loss at MMS is highly likely to be caused by equipment digging into coal because of inaccurate burden drilling plans sent to the burden drills. The inaccuracy is caused by the geological data that is used to create drilling plans. The geological data that is used is coming from the exploration holes that were drilled when underground mining first started, and these holes were drilled at a wider grid spacing. The grid spacing used for most of the holes was at 100m x 100m. Due to inaccuracy in drilling, coal is drilled and blasted with burden in areas where bords were supposed to be collapsed without drilling into the top of the coal.

However, in areas where pillars are imploded, the roof coal losses happen because coal is mixed with waste, and exposure equipment scalps some of the coal while exposing the coal. This happens because the stripping plans that are sent to the exposure equipment are inaccurate, as these plans have been developed from the same data as the drilling plans. The other contributing factor to equipment scalping coal is the incorrect blasted coal pillar profile prediction used. The mine's desired coal pillar profile assumes that the blasted coal pillar fills the bords and waste lies on top of the blasted coal pillar. The expected coal pillar profile is shown in Figure 4-2.

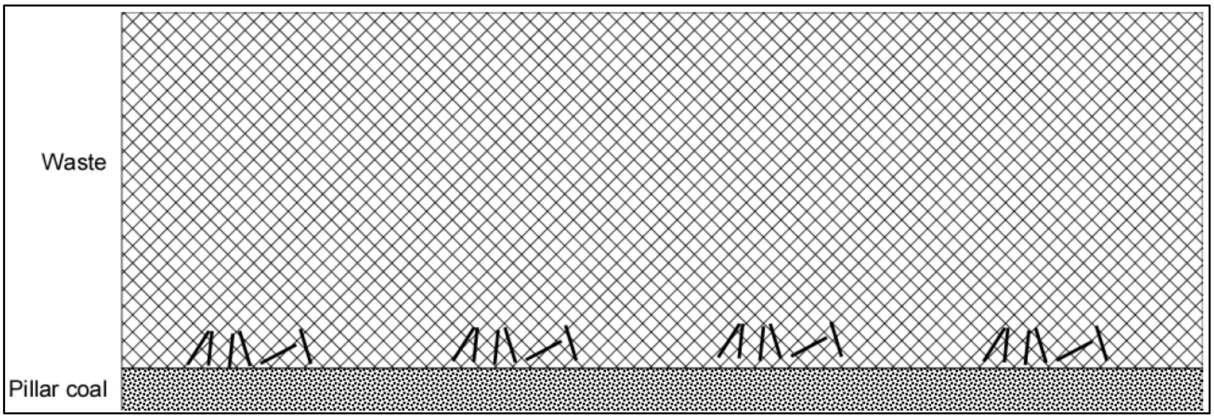

Figure 4-2: Expected coal pillar profile after the blast (Rorke et al, 2016)

The desired coal pillar profile is not achieved because holes are charged with explosives as a single column and fired at the same time. The initial studies regarding coal pillar blast required coal pillars and waste to be charged and fired separately. The sequence of events that should happen to achieve the Figure 4-2 profile is shown in Figure 4-3.

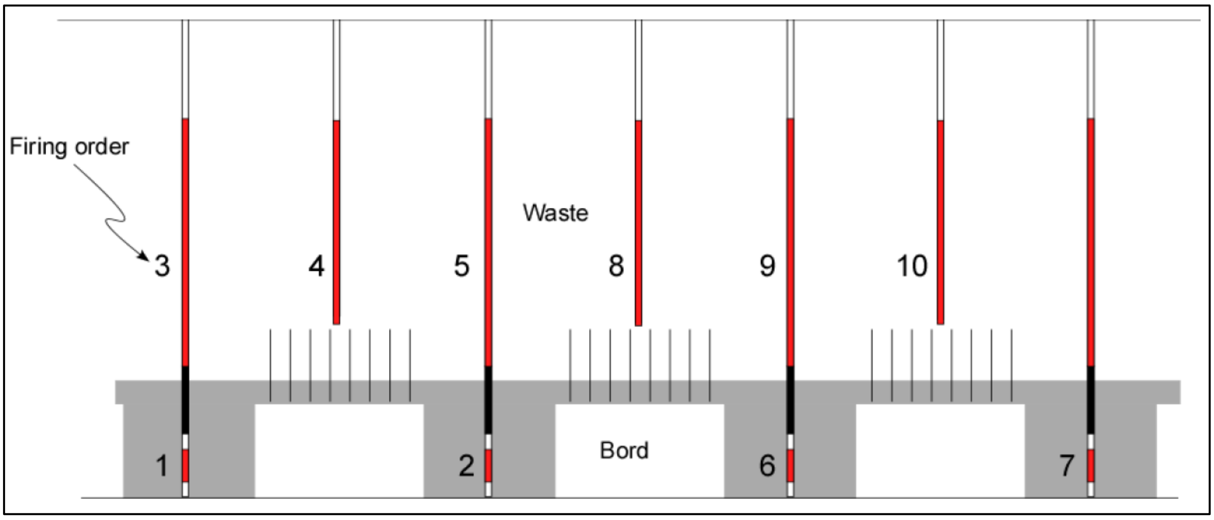

Figure 4-3: Sequence of events to achieve desired coal pillar profile (Rorke et al, 2016)

The coal pillar and waste have their independent detonators which are separated by at least 2 meters of stemming material. The two detonators allow for both coal and waste to be fired separately. The coal is fired first and then followed by the waste material. This type of blasting practice requires skilled blasting personnel and operating discipline. Through my observations and blasting personnel skill audit, MMS does not have qualified blasting personnel because all were hired with just a matric certificate and mathematics as a prerequisite. The profile for blaster and blasting assistant roles are shown in Appendices D and E.

4.2.2 Floor Coal Losses

The floor losses took place due to the following reasons:

- Water accumulation on the floor of the coal

The coal is left on the floor because of high water levels in the active coaling face. This is due to water seeping from old underground workings which might be as a result of broken underground seals and dam walls. The underground seals such as ventilation brick walls and underground dam walls might have been broken by surface blasts that took place since the beginning of the opencast mining at MMS.

According to Fourie and Green (1993), surface blasts that produce peak particle velocity (PPV) that exceeds 110mm/s will cause damage to the underground structures and cause roof coal to collapse. Fourie and Green (1993) further stated that PPV's of greater than 390mm/s will cause severe damage to the underground structures and roof collapse. A sample of two blasts was analysed to determine what impact will they have on the underground workings. The blasts analysed are from VDDN and BMK East pits and they are shown in Appendix C. Figures 4-4 and 4-5 show the vibration results for VDDN and BMK blasts respectively.

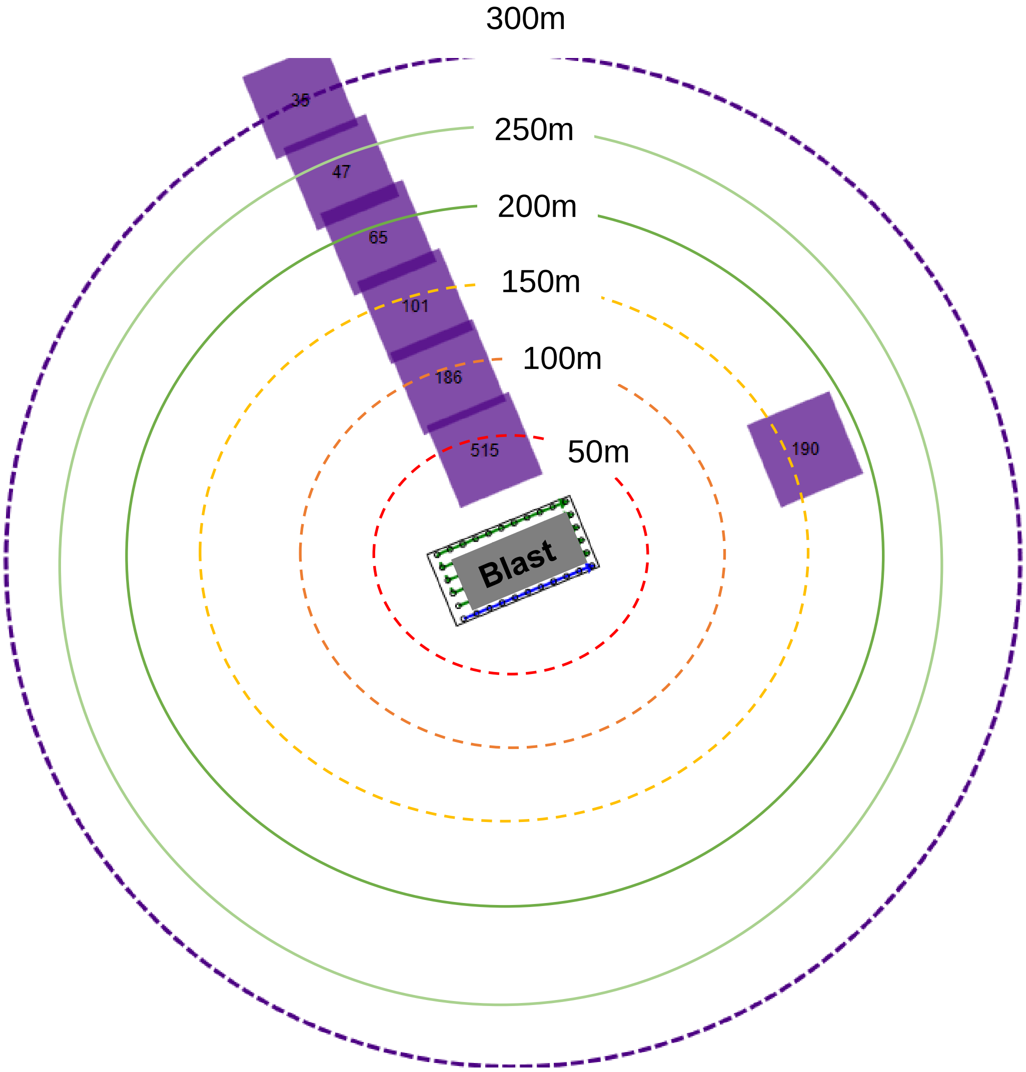

Figure 4-4: Vibration results for BMK blast (not to scale)

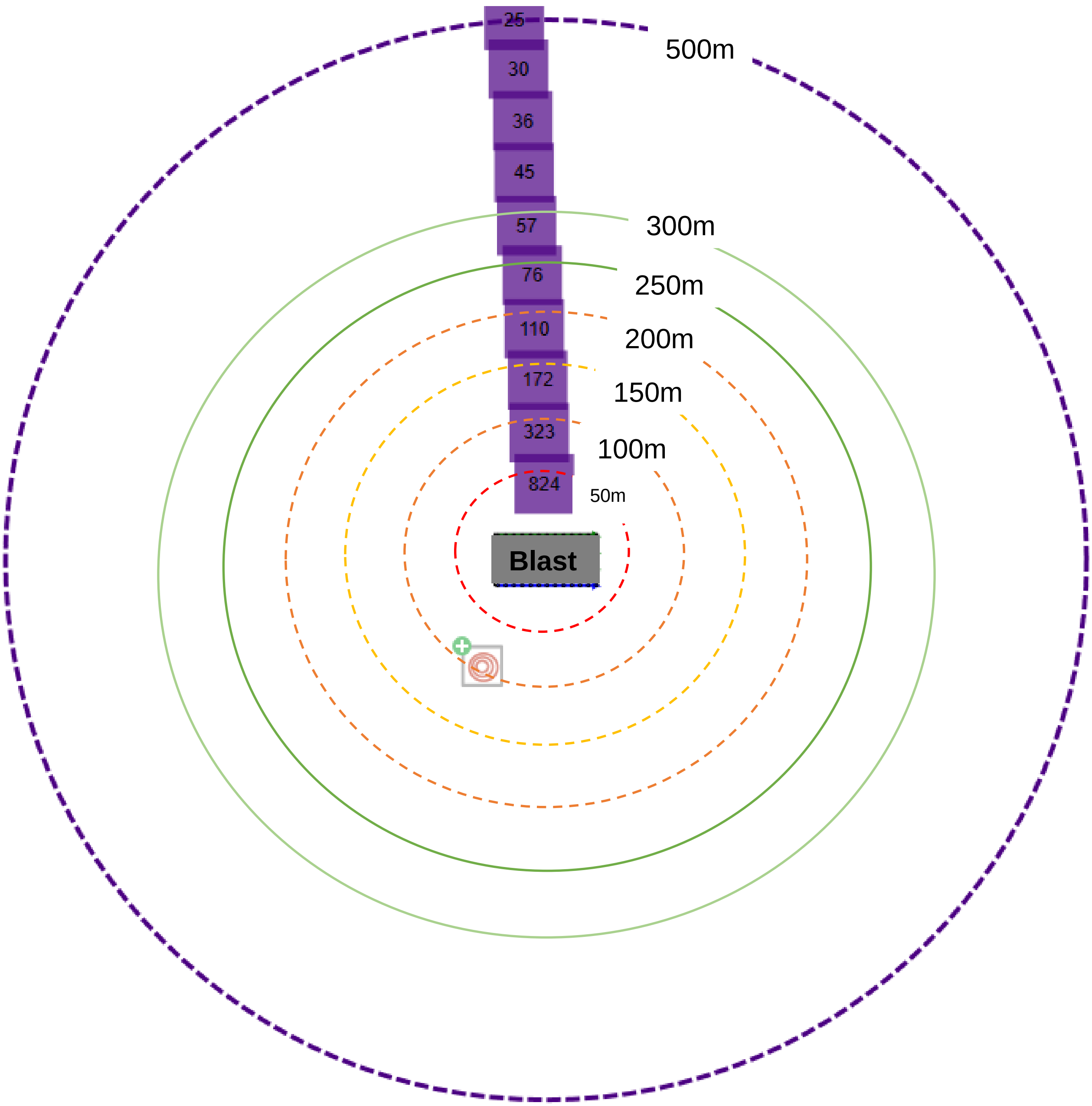

Figure 4-5: Vibration results for VDDN blast (not to scale)

BMK East blast, as shown in Figure 4-4 is likely to have caused damages to underground infrastructure and roof collapse at a distance of between 50 m and 100 m. However, at a distance greater than 150 m, no damages are unlikely to happen due to surface blasts. VDDN blast vibration results show that severe damages to underground infrastructure and roof collapse are likely to happen at distances which are between 0m and 125 m, and minor damages will take place at distances of up to 250 m from the blast. The increase in damage radius for VDDN is due to an increase in charge per delay and the explosive charges for the two blasts are shown in Appendix C. Most of the water that is flowing into the working face might have been partly caused by the surface blasts that took place over time and the results were the damage to the roof coal and breaking ventilation seals and dam walls.

- Hard coal left on the floor

The hard coal is left on the floor because the coal is too hard to dig and exposure equipment, predominately draglines, catching up with the coal extraction processes. The mine prioritises the draglines and writes off coal that was supposed to be ripped by dozers or drilled and blasted. The draglines are being prioritised because mining personnel claims that it costs more money to idle the draglines than mining coal left on the floor. The margin calculation, as shown in Appendix L shows that the mine will lose R546.78/saleable ton if the mine decides to write-off coal seam with a primary and secondary yield of 34% and 38% respectively. The qualities of primary and secondary products are assumed to be 6000 kcal/kg and 4800 kcal/kg respectively. The impact of idling the dragline is an increase in the life of the mine, which in turn decreases the NPV and IRR of the operation.

4.2.3 Geotechnical and Geology Losses

- Geology Losses

Geology losses are due to differences in the actual roof and floor elevations when compared to the resource model. The main geology losses come from the floor of the coal where the actual resource model elevation is different from the actual floor.

- Geotechnical Losses

Geotechnical coal losses at MMS were due to spoil sloughing on the void and covering the coal edge. Figure 4-6 shows spoil failure covering the coal edge.

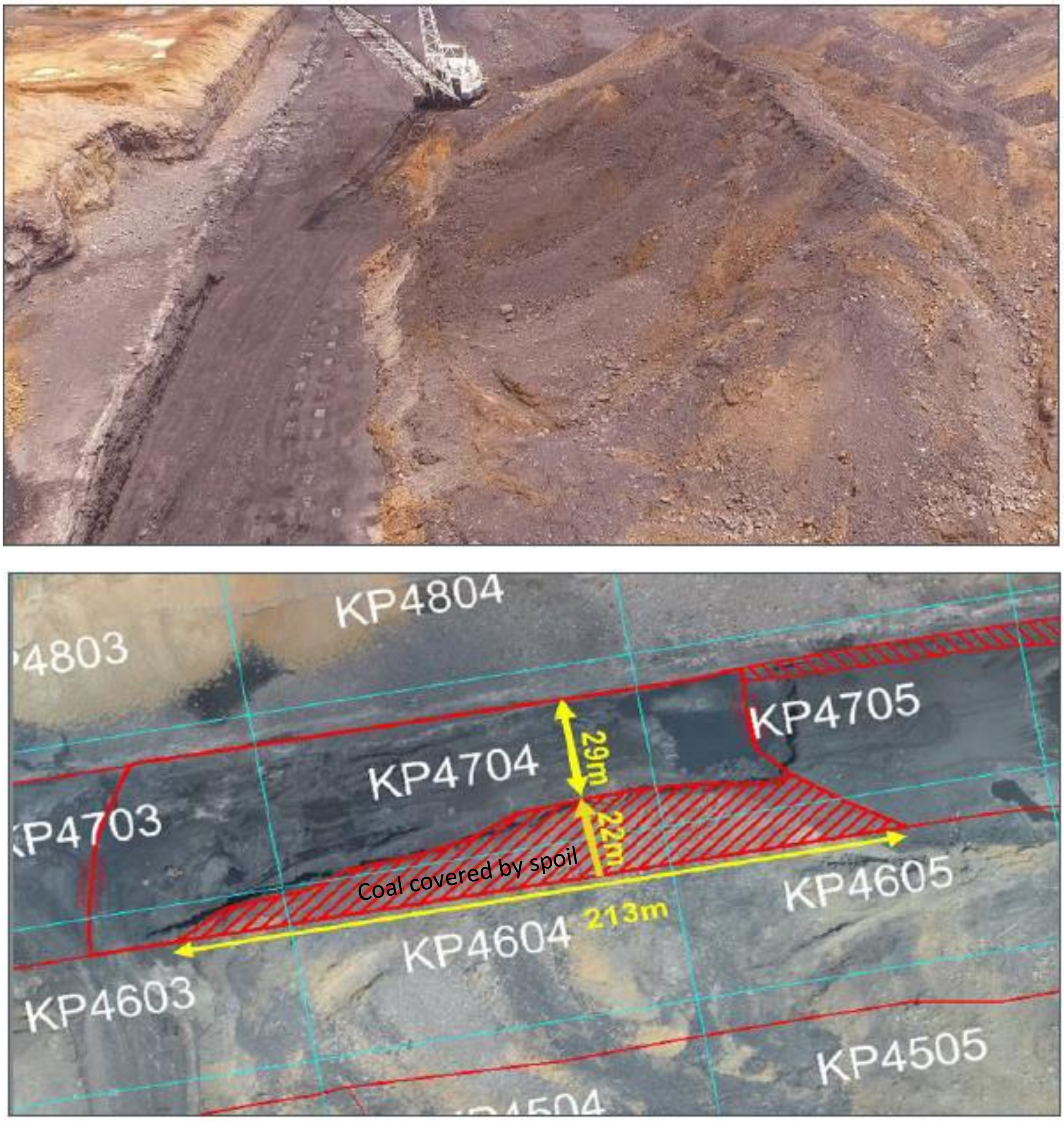

Figure 4-6: Spoil pile covering 22m of coal edge

The failure of the spoils on the coal edge was due to tight spoiling by the dragline. The cause of the tight spoiling was due to the dragline sequence. The sequence for the dragline starts on the shallow side of the pit and advances to the deeper section of the pit. Figure 4-7 shows a cross-section of where the dragline starts exposing coal, advance direction, and sections across the pit.

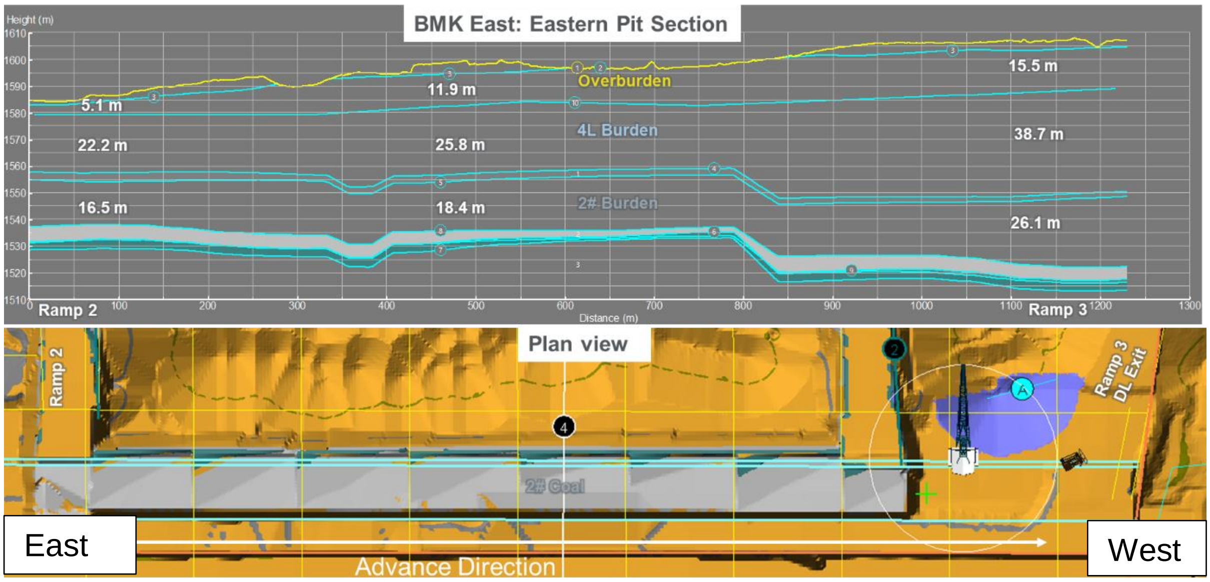

Figure 4-7: Cross section showing dragline advance direction in relation to the burden thickness

The spoils are tight because there is no geotechnical berm left. An example of a geotechnical berm that is required is shown in Figure 4-8.

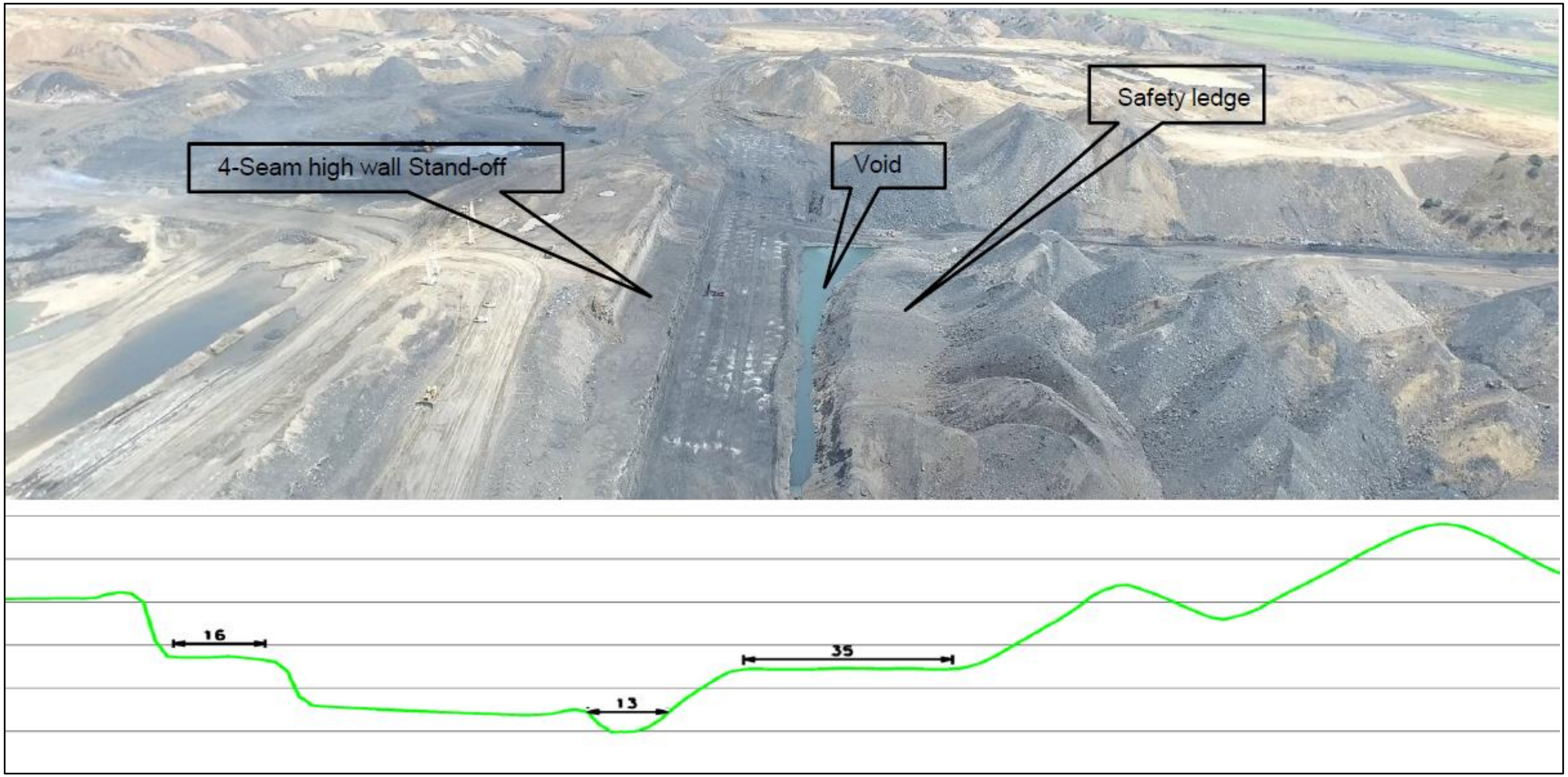

Figure 4-8: An example of a geotechnical berm or safety ledge

The geotechnical berm reduces the impact of surcharge loading on the edge of the spoil and hence increases the factor of safety for the spoils. The other cause of geotechnical losses at MMS is the buffer materials that are left on the highwalls of burdens with no underground workings on the end cut highwalls. The buffer material on these areas was left due to the incident that the mine had in 2018, where material on the highwall fell on the shovel. This happened while the shovel was cleaning the highwall. Figure 4-9 shows the shovel incident that happened at MMS South.

Figure 4-9: Highwall material fell on the shovel.

This incident changed how highwalls are dealt with at MMS. To ensure the employees remain safe, the decision to leave buffer material on the highwalls of areas with no evidence of underground workings. This was left in place to prevent equipment from working next to high highwalls. Figure 4-10 shows the buffer left on the highwall.

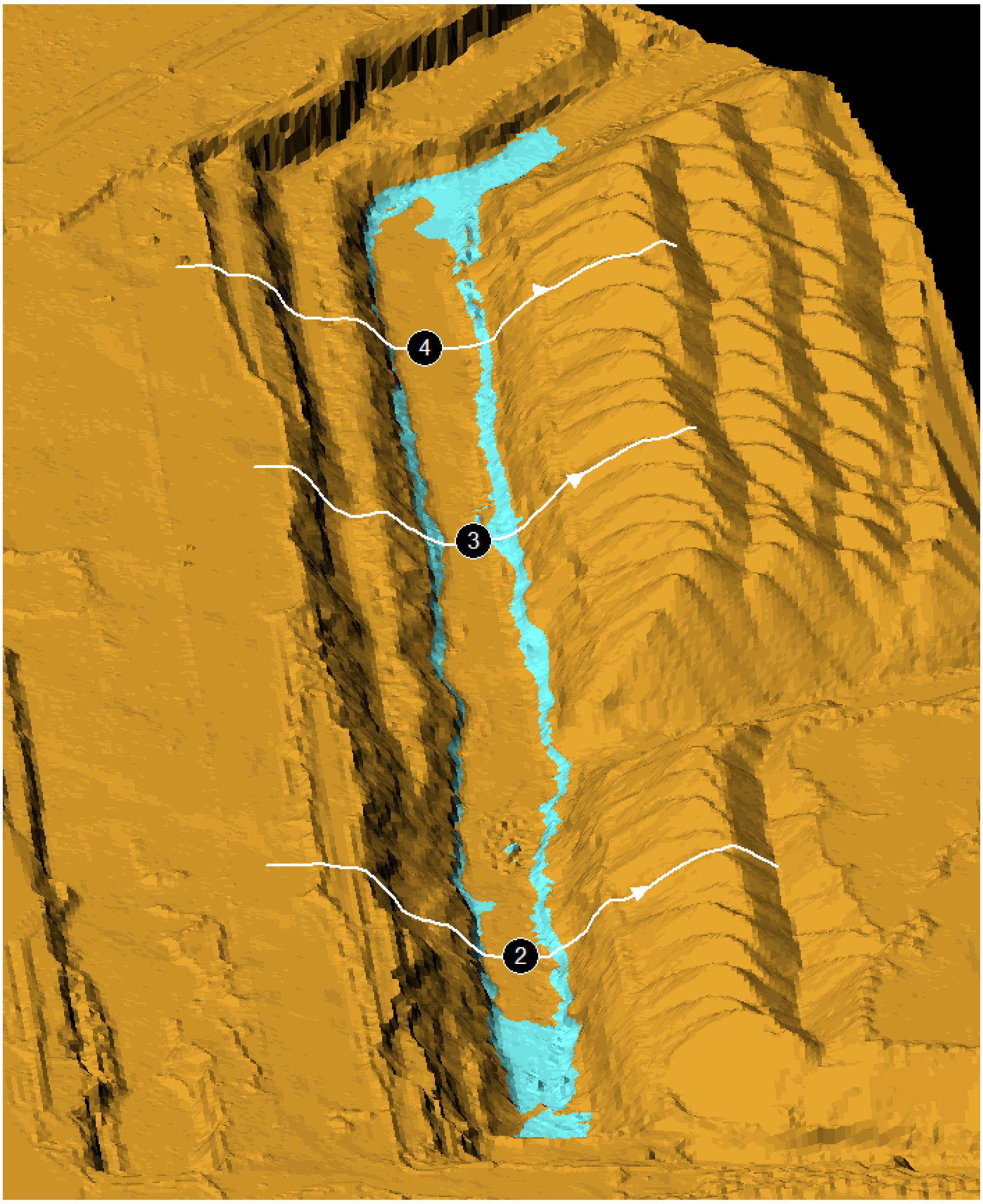

Figure 4-10: Buffer left on the upper bench highwall and no geotechnical berm left

The cross section lines 2 to 4 in Figure 4-10 shows that the highwall in the direction of advance is not vertical because of blasted material left against the highwall. The highwall design where there is no presence of underground coal pillars requires highwall to be vertical and no material left on the highwall. However, MMS deviated from the original design due to an incident where frozen material fell on the shovel while trying to remove the frozen material. Figure 4-10 also shows that there is no evidence of a safety stand-off on the spoil side. This is caused by the thicker upper burden and also incorrect placement of ramps in the pit. This pit has two ramps which are 150 meters apart and also in the deeper part of the pit.

4.3 Coal Dilution Analysis

The amount of dilution is not currently measured by surveyors at MMS because of the difficulty in distinguishing between waste and coal. The amount of dilution that is currently being used in the mine is the planned percentage of 30 % where pillars are imploded and 14% in sections where bords are collapsed. If the measured ROM in a specific block of coal has been calculated to be more than the reserve tons, the extra tons are considered as waste, and if it's less, is declared as a loss. However, the main indicators of the amount of waste being fed into the plant are the extra loss on plant yield and reject rock removed during the crushing stage. Figure 4-11 shows the plant yield comparison between plan and actual.

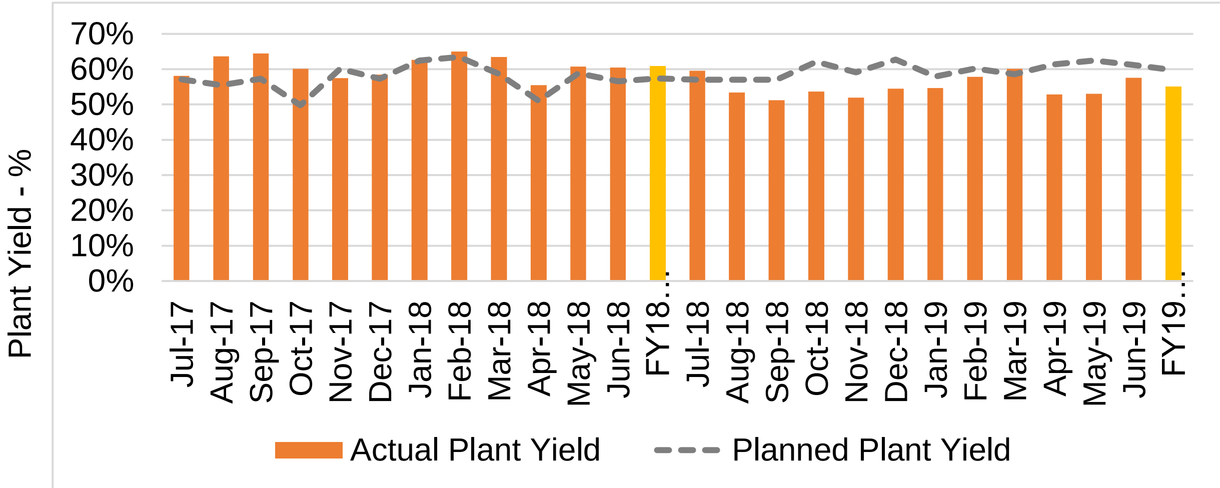

Figure 4-11: Financial Year (FY) 18 to 19 plant yield comparison

In FY18, MMS reached the planned plant yield, and this can be attributed to mining clean and mining a majority of virgin coal. A pit that only mined virgin coal contributed 23% of the total coal fed into the processing plant in FY18 and the pit got depleted in FY19. This can be seen through the yield results for FY19 because the impact of dilution started to show. MMS only reached the target twice in FY19. The plant yield drop between the two financial years is 6%, which represents a great loss in revenue for the mine.

To pinpoint the exact causes of coal dilution at MMS that lead to excessive yield losses, the following factors were analysed to determine if they play any role:

- Strip orientation to the underground roadways;
- Pillar scaling;
- Mode of bord failure;
- Pillar mining methods; and
- Influence of monthly incentives paid to the workforce in the coal face.

4.3.1 Strip Orientation

Six different strip orientations were analysed and these orientations are listed as follows:

- Zero degrees;
- 30 degrees;
- 45 degrees;
- 60 degrees;
- 75 degrees; and
- 90 degrees

The results of the strip orientation in Appendix F show that there are no significant differences in the coal volume of the pillars and the void percentages. This implies that strip orientation does not cause dilution in opencast pillar mining methods where overburden material between pillars is not cleaned by any exposure equipment. However, if a decision is taken to reduce dilution by cleaning the waste material between pillars with exposure equipment such as draglines, it will be difficult to do so in orientations above zero degrees and less than 90 degrees because it will be difficult to control the bucket. The dragline works efficiently when the strip direction and underground bords are either perpendicular or parallel to each other.

Strip orientation does not cause dilution, but might cause coal losses due to some of the following reasons:

- If the strip orientation is in the direction of water flow. This will result in floor coal loss due to high levels of water that will accumulate and be challenging to pump out of the pit; and
- Cleaning between pillars by exposure equipment might lead to both edge and roof losses. This has been analysed in detail in section 4.2.

4.3.2 Pillar Scaling

Van der Merwe (2003) found that pillars at MMS South (formerly Wolvekrans Colliery) have previously failed which might be caused by pillar scaling. The amount of pillar scaling was estimated using equation [2], $R=0.1624[h/T]^{0.8135}$ where h is the mining height and T, is the time since mining ceased in years. The assumptions used for the estimation are as follows:

- Pillar centres: 15 m;
- Depth from surface to No. 2 seam floor: 25 m;
- Pillar size: 8 m x 8 m; and
- Average mining height: 4m.

Figure 4-12 shows the results of calculated pillar scaling for three different mining heights.

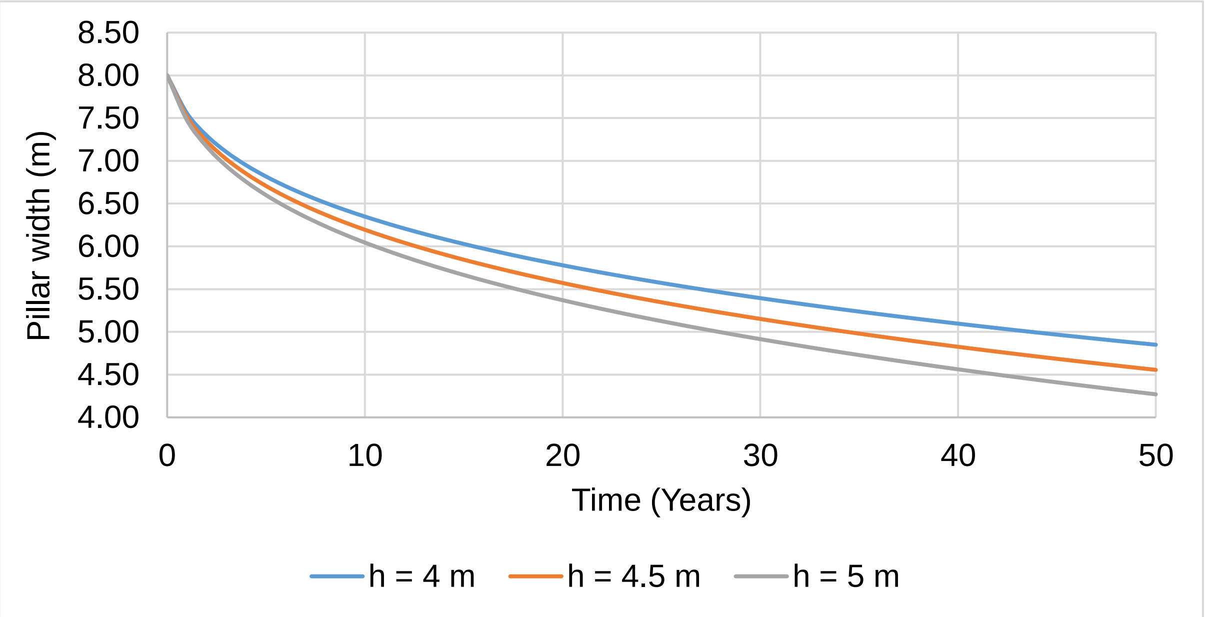

Figure 4-12: Results of calculated pillar scaling at MMS South pits

The rate of scaling varies per year and starts aggressively when pillars have just been created. The rate, however, then flattens out over time and when the mining height is increased, the rate of pillar scaling also increases. A comparison of scaled and unscaled pillar shapes is shown in Figure 4-13.

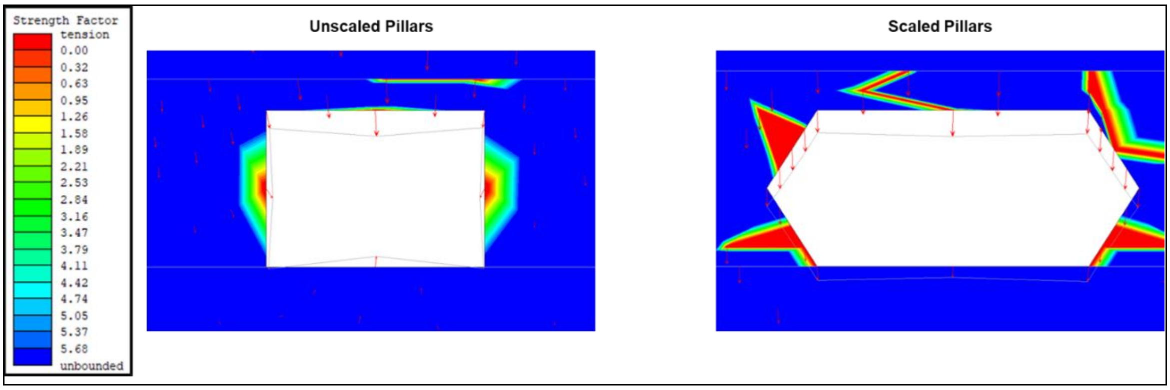

Figure 4-13: Scaled pillar versus unscaled pillar

The scaled pillar takes a shape that is like an hourglass shape. The weakest point on the bord roof is at the middle of the roof span (shown in red) and it is where these pillars would likely fail. The failure of the pillars will unlikely cause the waste beam to fail as well and this means that dilution will be unlikely to happen when the pillars scale. Dilution in scaled pillars can happen when blasting of the pillars with waste is introduced. Figure 4-14 shows the effects of explosives when scaled and unscaled pillars are loaded with explosives.

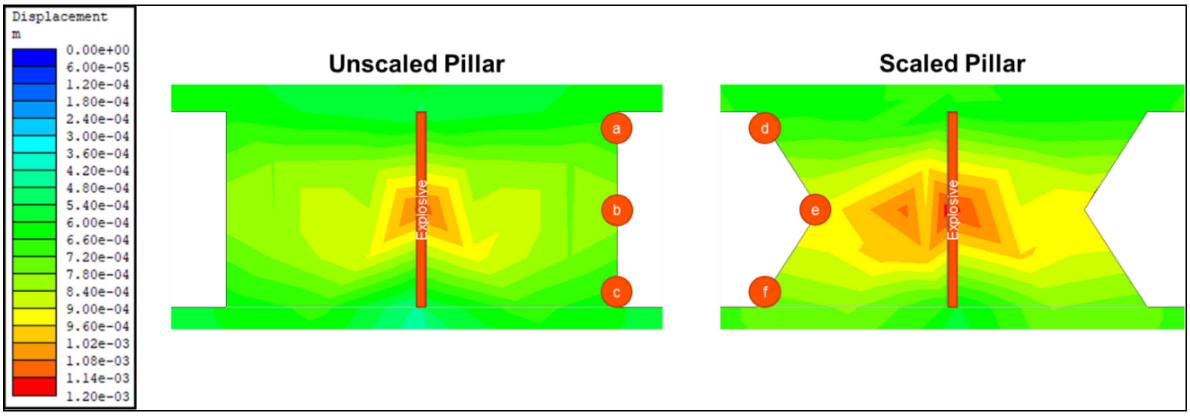

Figure 4-14: Effects of explosives on coal pillars

When an unscaled pillar is loaded with explosives, points a, b and c are almost equidistant from the explosive column, thus under ideal conditions represents an even resistance for explosive energy. Explosives are known to exert force equally in all directions on the adjacent rock and break where the rock offers the least resistance (Blast Movement Technologies, 2018). Midpoint b in unscaled pillar experiences 30% more initial displacement relative to corners a and c, and it is likely to break first at point b. Unscaled pillar is likely to break evenly during pillar implosion.

With regards to the scaled pillar, point e offers a path of least resistance for explosives, and explosives gases are likely to escape through point e. Midpoint e experiences 50% more initial displacement relative to corners d and f. However, point f will offer more resistance due to lose scaled material confining the pillar bottom. The current blasting practice of blasting pillar coal and waste at the same time might cause dilution due to the uncontrolled movement of both waste and coal.

4.3.3 Mode of Bord Failure

An analysis was done to determine how bords collapse when pillars have scaled or not, and to establish if there is a relationship between dilution and how bords collapse during blasting. Figure 4-15 shows the total displacement of the bord when the overlying roof has been charged with explosives and initiated.

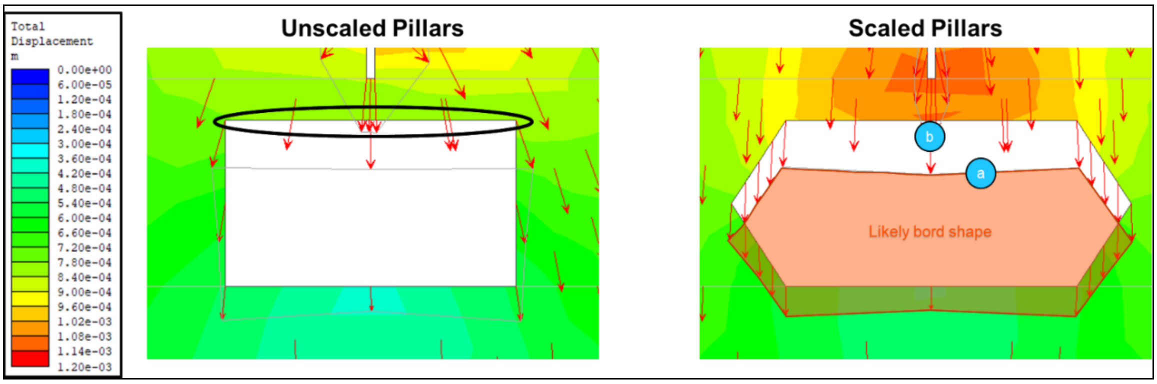

Figure 4-15: Total displacement of the bord on scaled and unscaled pillars

When unscaled pillar bord is loaded with explosives and detonated, the initial displacement of the bord roof (circled area) is more uniform and the colour remains in the green region across the span. The unscaled pillar bord will collapse as a unit and this will make it easy to distinguish between waste and coal while mining coal because the waste will be exactly on top of roof coal. This will assist mining operators to do selective mining if a need arises and hence reducing dilution.

Point b on the scaled pillar bord, as shown in Figure 4-15, experiences more initial displacement relative to the other points on the bord roof and the colour ranges from yellow to red. The roof of bord between scaled pillars is more likely to break in the middle when an explosive column is detonated. The fragmentation results of the material above the bord are likely to be poor and hence resulting in high levels of coal contamination.

The high percent of contamination might be due to wider voids between the coal fragments which will be filled by the waste material.

4.3.4 Pillar Mining Method

The methods that are currently used in the pillar sections at MMS are pillar implosion and bord collapse. Pillar implosion is applied at the VDDN pit and bord collapse is applied at both BMK East and West pits.

- Pillar implosion – current state

In this method, coal pillars are blasted with overlying waste material. This method requires drill holes to be drilled from the top of waste up to the bottom of the coal pillar. The waste and coal pillar is charged separately, that is, the coal charge is placed at the bottom of the hole and then stemmed with at least 2 m of waste material and then, the waste charges are placed on top of coal stemmed material. The two charges in the hole are fired separately, where coal is fired first and followed by waste. This is done to reduce dilution.

However, the current practice is different from what's been described. The current practice is that coal pillars and overlying waste are drilled and charged with explosives as a single hole, and then fired at the same time in a single shot. This practice which is different from the design introduces high percentages of contamination. Figure 4-16 shows contaminated coal where pillar implosion was carried out.

Figure 4-16: Coal mixed with waste in a fully imploded pillar

Coal samples were taken to determine the impact of pillar implosion in one of the pits currently using this method. The sample results confirm the negative impacts of the current practice in pillar implosion pits because the relative density that the plant should wash fully imploded pillars is an average of 1.85 g/cm^3 which is greater than the maximum acceptable RD of 1.65 g/cm^3. Figure 4-17 is a combined graph illustrating the relative density, calorific, and ash values from the samples collected from the fully imploded, half imploded, and non-imploded coal pillars.

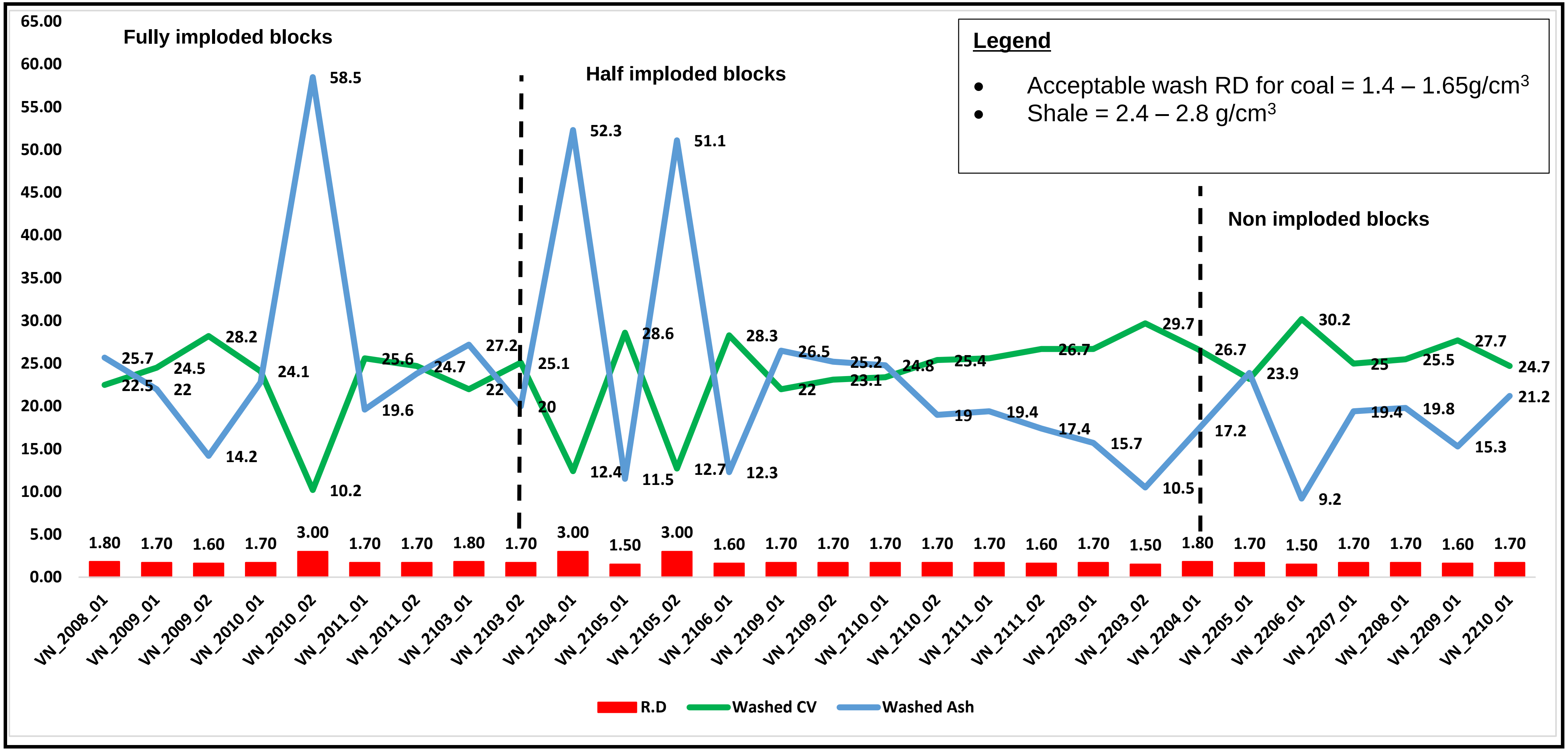

Figure 4-17: Coal qualities of samples collected from fully, half and non-imploded blocks

The relative density (RD) in the fully imploded blocks ranges from 1.60 g/cm^3 to 1.80 g/cm^3 with calorific values ranging from 22.5 MJ/kg to 28.2 MJ/kg and ash values of between 14.2% to 25.7%. The majority of the samples are outside the acceptable wash RD for coal of between 1.40 g/cm^3 to 1.65 g/cm^3 due to contamination. There is a sample within the fully imploded blocks that has a relative density of 3.00 g/cm^3. This sample has a calorific value of 10.2 MJ/kg and an ash value of 58.5%. This is way beyond the MMS cut-off ash of 45%. It would be impossible for the plant to wash at a relative density of 3.00 g/cm^3 to get a yield greater than 40%. The maximum density at which the processing plant can wash material from the pits is approximately 1.80 g/cm^3.

In half-imploded blocks, four samples are within the acceptable wash RD for coal. Their relative densities range from 1.50 g/cm^3 to 1.60 g/cm^3, with calorific values ranging between 26.7 MJ/kg to 29.7 MJ/kg and ash of between 10.5% to 17.4%. Two samples have a relative density of 3.00 g/cm^3 in block VN 2104/01 and VN 2105/02. The calorific values of these two samples are 12.4 MJ/kg and 12.7MJ/kg while their ash values are 51.1% and 52.3% respectively.

In non-imploded blocks, the relative density of the grab samples ranges from 1.50 g/cm^3 to 1.70 g/cm^3. The samples are producing much better and more predictable results as compared to the fully and half-imploded blocks with calorific values ranging from 24.7 MJ/kg to 30.2 MJ/kg, while their ash values are between 9.2% to 21.2%.

In examining the overall relative density graph, one should consider the grab sample material that is sampled in the field. This is before the material goes through the rotary breaker and crusher to separate the waste material from the coal. To achieve an average plant yield of greater than 40%, the imploded pillars should be washed at an average relative density of 1.85 g/cm^3. This is way beyond the acceptable wash RD for coal at 1.40 g/cm^3 to 1.65 g/cm^3. This can only suggest that waste material was being sent to the plant. Samples taken in the fully and half imploded blocks have an average RD of 1.85g/cm^3 compared to 1.64g/cm^3 in the non-imploded blocks. The reduction in the relative density is an indication of the decreasing added contamination from the fully imploded blocks to the non-imploded blocks.

To further understand the impact of the current way of doing pillar implosion, one needs to review if the predicted blasted coal pillar profile is as per both Clough and Morris (1985) and Rorke (2016) prediction. Both Clough and Morris (1985) and Rorke (2016) predictions assumed that imploded coal pillars fill the bords and their predicted profiles have been shown in Figures 2-10 and 4-2. Simulations using 3-DDig software were then conducted to determine how imploded coal pillars behave when blasted separately. Figure 4-18 shows a plan view of pillars before implosion and pillars after the implosion.

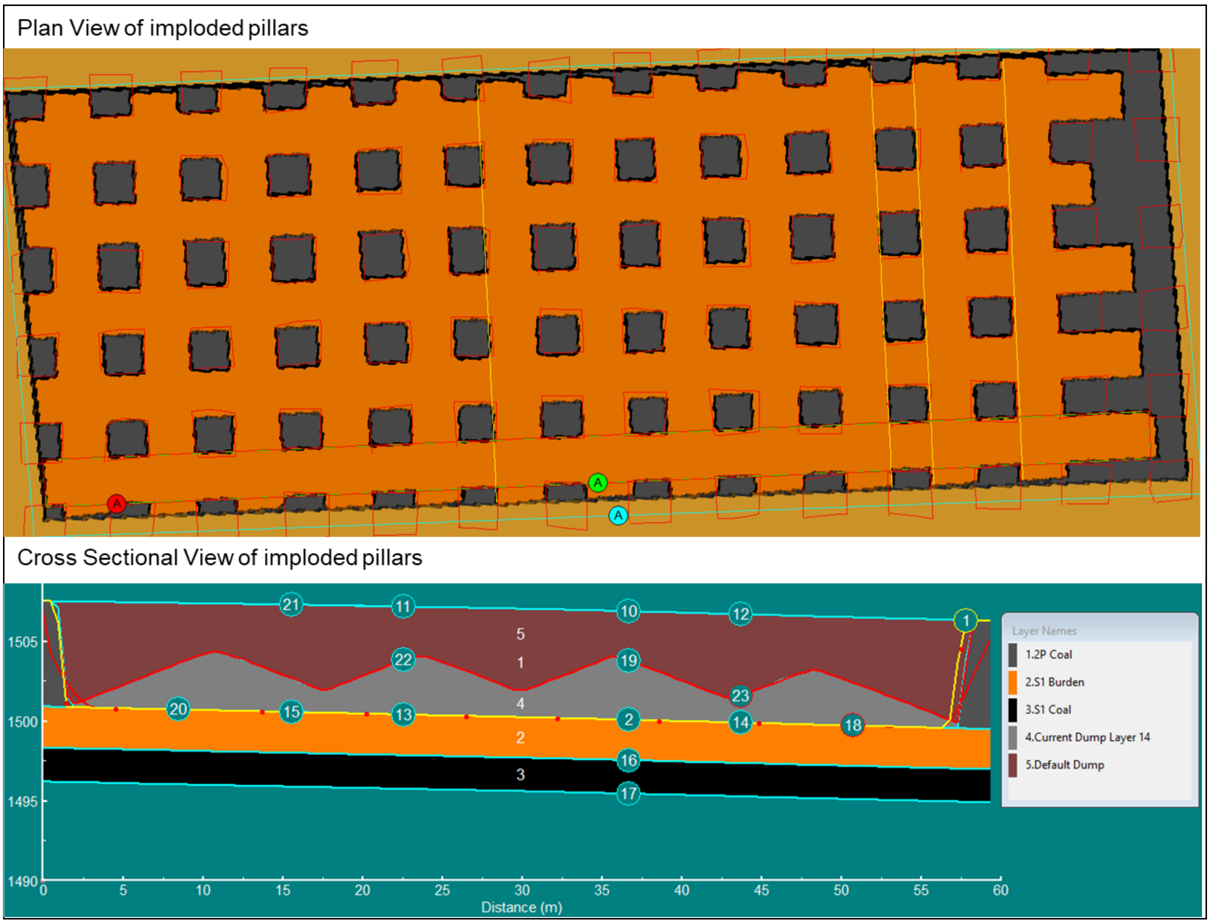

Figure 4-18: Plan view of pillars before implosion and a section view post implosion

The simulated profile using 3-DDig shows that the coal pillars do not fill voids as the previous estimation by Clough and Morris (1985) and Rorke (2016). The imploded coal pillars form a profile that is like a wave ripples profile. This type of profile proves to be difficult to separate coal and waste because of the high volumes of waste where the blasted pillars meet. Therefore, dilution in this method is caused by the wave ripple profile because exposure equipment such as draglines will struggle to follow the coal without scalping some of the coal. If the mine decides to follow the coal profile, a high percentage of coal losses can be incurred. The amount of coal losses and dilution that can be incurred if a certain depth is taken when exposing imploded coal pillars is shown in Appendix M.

If a decision is taken to leave at least 0.5m of waste material above the coal, overall plant yield will drop by more than 50% when compared to the uncontaminated coal seam. As the waste material above coal is reduced, the dilution percentage decreases, but roof coal losses will increase from 0% at 0.5m above coal to 59.25% at 3m into coal. The mining height that gives compromise results, especially in terms of dilution reduction and minimal roof coal losses is mining 2m into the estimated imploded coal pillar profile.

- Bord Collapse – current practice

As stated in the previous chapters, bord collapse at MMS South is applied in areas considered to have hot underground workings. The bord collapse method is being practiced at both BMK East and West. The currently planned dilution percentage for these two pits is 14%. The planned dilution in these two pits is too little because the simulation using 3-DDig software has found that the expected dilution for this type of method is more than 14%. The expected dilution as per 3-DDig software simulations is shown in Figure 4-19.

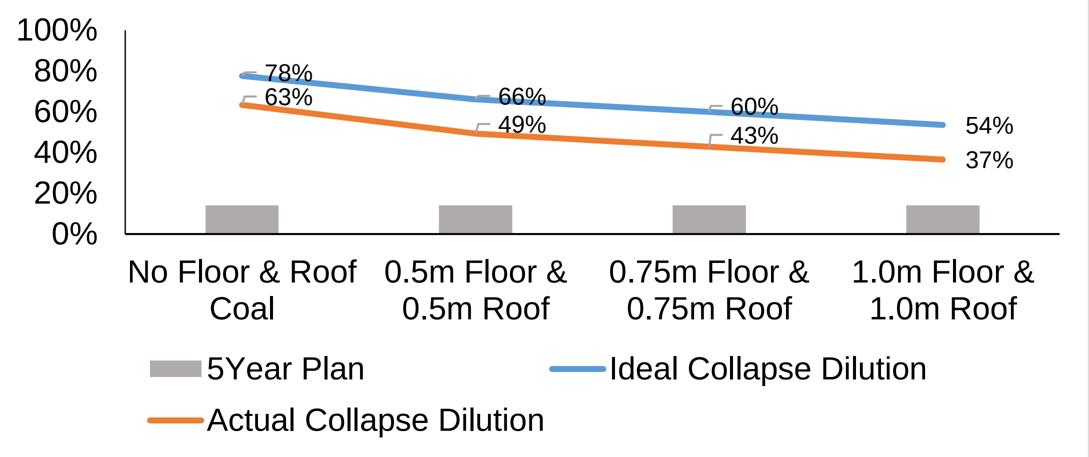

Figure 4-19: Simulated dilution percentages for bord collapse method.

Figure 4-19 also shows the dilution percentage for the two-mode bords collapse with different conditions regarding coal left on both the floor and roof. The dilution percentage for scaled pillar bord ranges from 63% to 37% depending on how much coal was left on both floor and roof of the bord, and for as formed conditions dilution percentages range from 78% on where no coal was either left on both the floor and roof to 54% on where at least a 1m of coal was left on both the floor and the roof of the coal.

4.4 Influence of Monthly Incentives Paid to the Workforce in the Coalface

The bonus scheme discussed in Chapter 2 was changed in September 2018 to incorporate crushed tonnes, waste moved per pit, and saleable produced. However, the bonus is only payable when 91% of the forecasted saleable product has been achieved. A detailed memorandum with the changes is summarised in Appendix H. Figures 4-20 and 4-21 show bonus targets versus actual achieved from FY18 to FY19.

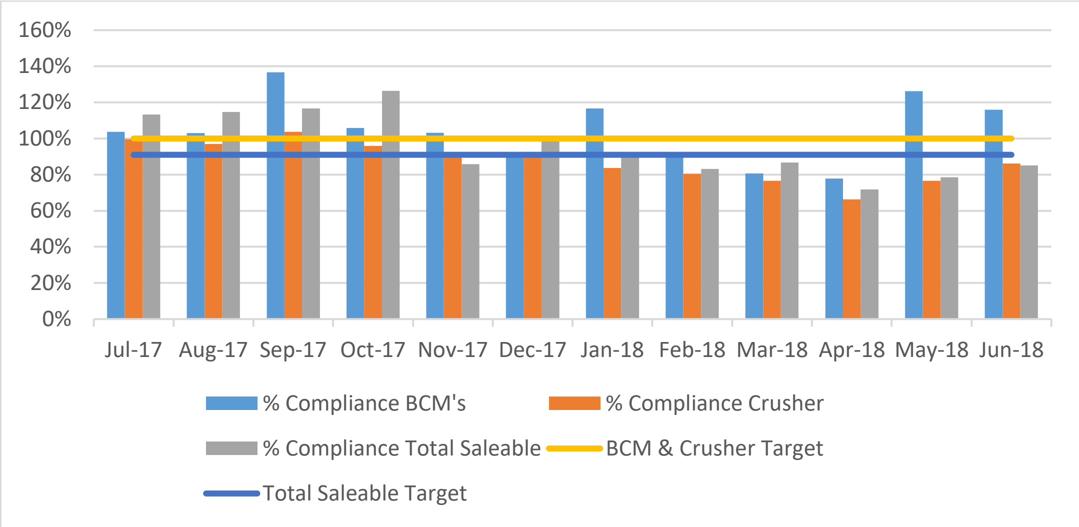

Figure 4-20: FY18 budget and actual bonus KPI’s.

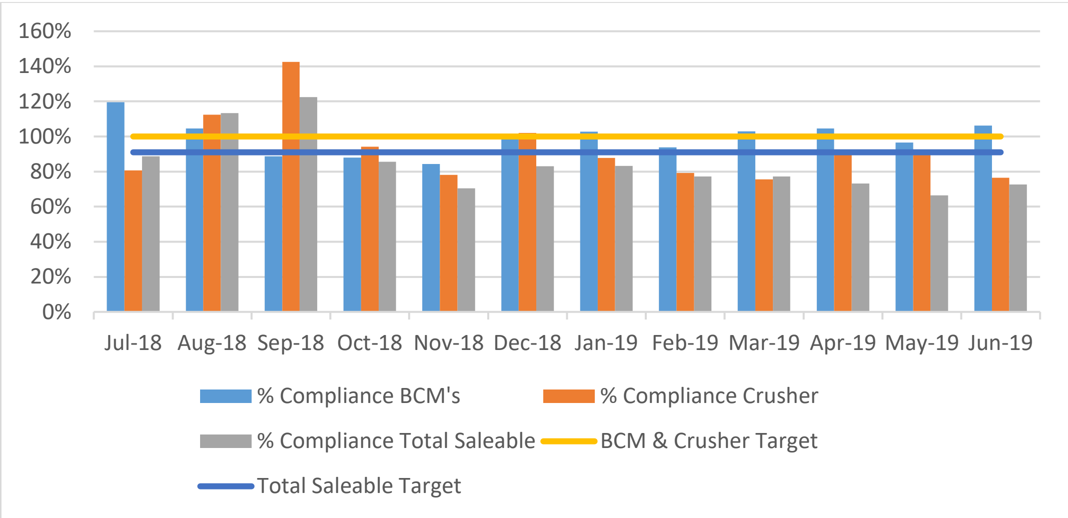

Figure 4-21: FY19 budget and actual bonus KPI’s.

In FY18, the production bonus was only paid six times, and this was due to the higher plant yield achieved in FY18. However, in FY19, the production bonus was only paid twice. The updated production bonus KPI's does not influence mining personnel to mine more waste with a run of mine coal because the mine did not pay a bonus every month.

4.5 Other Topics Related to Coal Pillar Mining through Opencast Mining Methods

These topics have no influence on the causes of both dilution and coal losses when coal pillars are exposed through opencast mining methods. The topics that will be discussed include buffer widths required to minimise the risk of spontaneous combustion and the method currently used to confirm the position of coal pillars if they are the same as in the survey working plan.

4.5.1 Buffer Widths

A buffer is created to assist in preventing spontaneous combustion from taking place in the old underground workings. The minimum buffer width that is currently left at MMS South pits is 30m and at an angle of repose of 55 degrees. Most mines in the Witbank coalfields that are exposing coal pillars through opencast mining methods use a rule of thumb to determine the minimum buffer width. The rule of thumb, as discussed in Chapter 2, states that the minimum buffer width should not be less than one-third of strip width. If this rule of thumb is applied at MMS South pits, the buffer width that should be left on a 50m pit width is 16.7m, which can be rounded off to 20m. Wider buffer widths do assist in reducing spontaneous combustion, but they introduce other challenges such as no blast gain on a stacked bench set-up and hence increases exposure costs. The material that was supposed to be moved by the blast, which is the cheapest form of moving material, ends up being moved by either draglines or a combination of truck and shovel fleet. The cost comparison between different modes of moving waste material is shown in Figure 4-22.

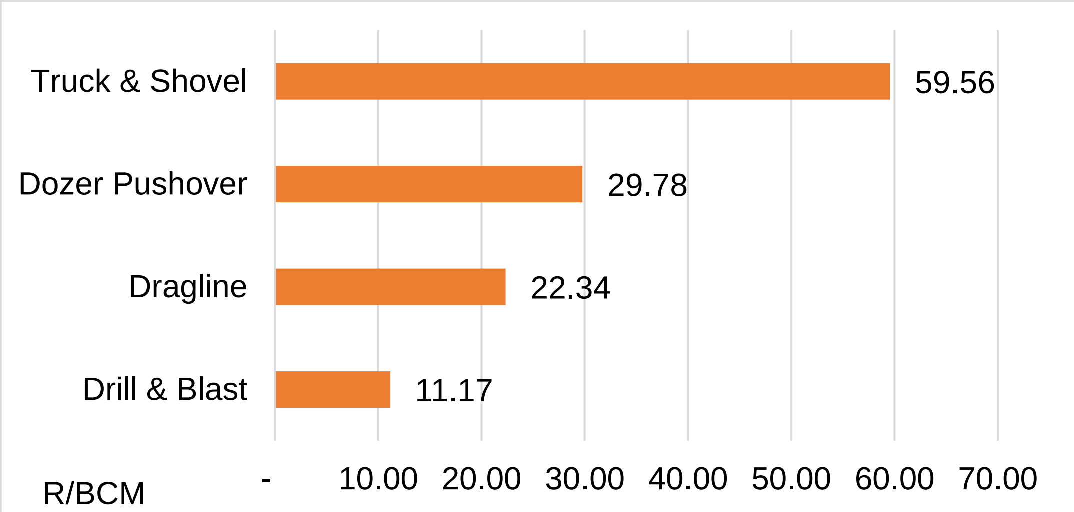

Figure 4-22: Cost comparison between different modes of moving waste material (MEC Mining, 2019)

The other factor that affects blast gain is the mandatory geotechnical stand-offs between benches. The current geotechnical stand-off on a stacked bench set up is 16m. When this stand-off is combined with the 30m buffer width the total stand-off between the stacked benches is 46m. This combination leads to all waste material being removed by mechanical means. To check whether the geotechnical stand-off of 16m is too excessive for MMS, a rockfall simulation was conducted. The results of the rockfall simulation for stacked bench set-up are shown in Figure 4-23.

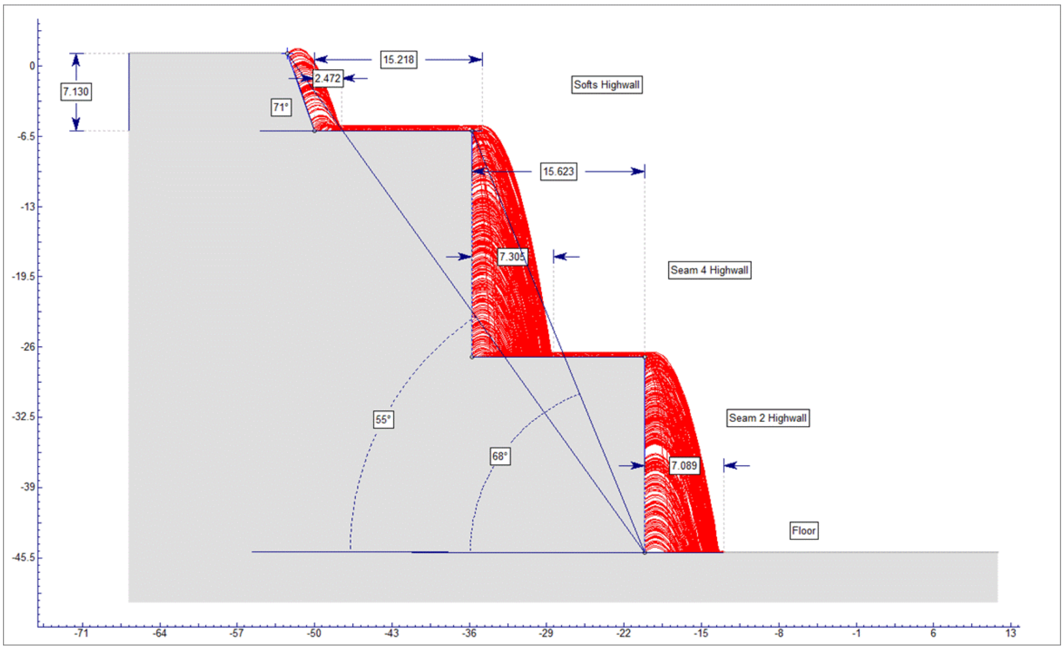

Figure 4-23: Rockfall simulation results done on stacked bench set-up.

The simulation results show that the maximum distance a dislodged rock from the crest of the highwall can bounce up to a horizontal distance of 7.305 m. These results from the rockfall simulation prove that the current geotechnical stand-off on a stacked bench set-up is excessive.

4.5.2 Coal Pillar Position System

The current system that is used to check if the position of the coal pillars is similar to the digitised underground plan is called Down-the-hole (DTH) scan. DTH scan requires drill holes to be drilled through to the underground workings in advance. Once the hole is drilled, the DTH scan is then lowered into the hole and scans the workings. Figure 4-24 shows the scan done at MMS South Pits.

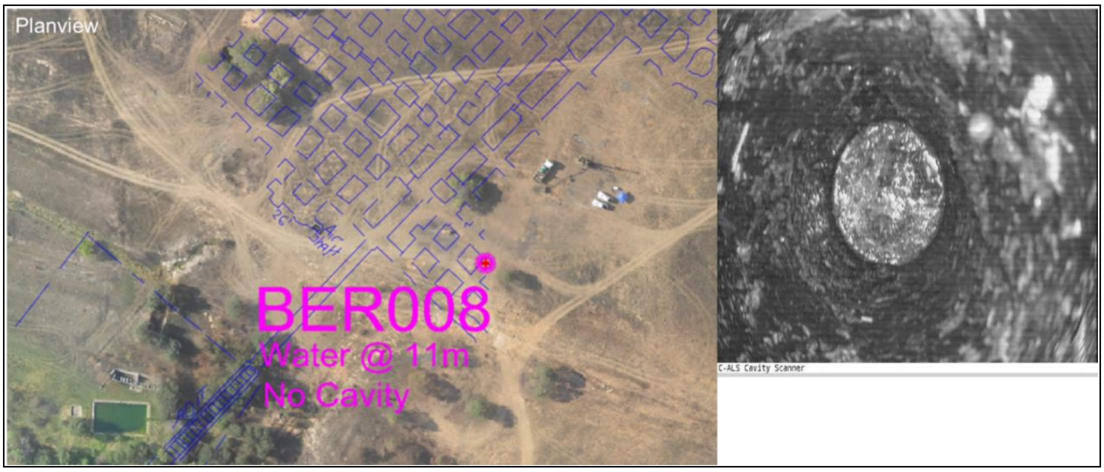

Figure 4-24: DTH scan done at MMS

In this area where a scan was done, a bord was not intersected as it has been predicted by the old survey plan showing underground workings. However, this technology has a limitation especially in areas considered to be hot. The scan cannot be lowered in areas such as in BMK East and West. The other issue with this technology is that drill holes must be drilled into the underground workings and holes into the workings increase the risk of spontaneous combustion by allowing oxygen to enter the workings.

4.6 Review of the Current Coal Pillar Methods

This section aims to review if the current methods that are being used to deal with underground coal pillars before exposure through opencast mining methods are appropriate for MMS South pits conditions. The review of these methods was done using MCDA methods called AHP and VIKOR. The coal pillar methods that were reviewed are summarised as follows:

- Pillar Implosion where 0.5m waste is left on the roof of the predicted coal pillar profile;
- Pillar Implosion where waste on the roof of the predicted profile is removed without scalping any coal on the roof peaks;
- Pillar Implosion where 0.5m of coal is removed with waste. This is done to reduce contamination on the lowest points on the wave-like profile;
- Pillar Implosion - 1.0m of coal removed with waste;

- Pillar Implosion - 1.5m of coal removed with waste;
- Pillar Implosion - 2.0m of coal removed with waste;
- Pillar Implosion - 2.5m of coal removed with waste;
- Pillar Implosion - 3.0m of coal removed with waste;
- Bord Collapse with no cleaning between pillars; and
- Bord Collapse with cleaning between pillars.

The exposure heights of the imploded pillars were measured from the original height of the pillar. Figure 4-25 shows the original line where the exposure height is measured from, when imploded coal pillars are exposed.

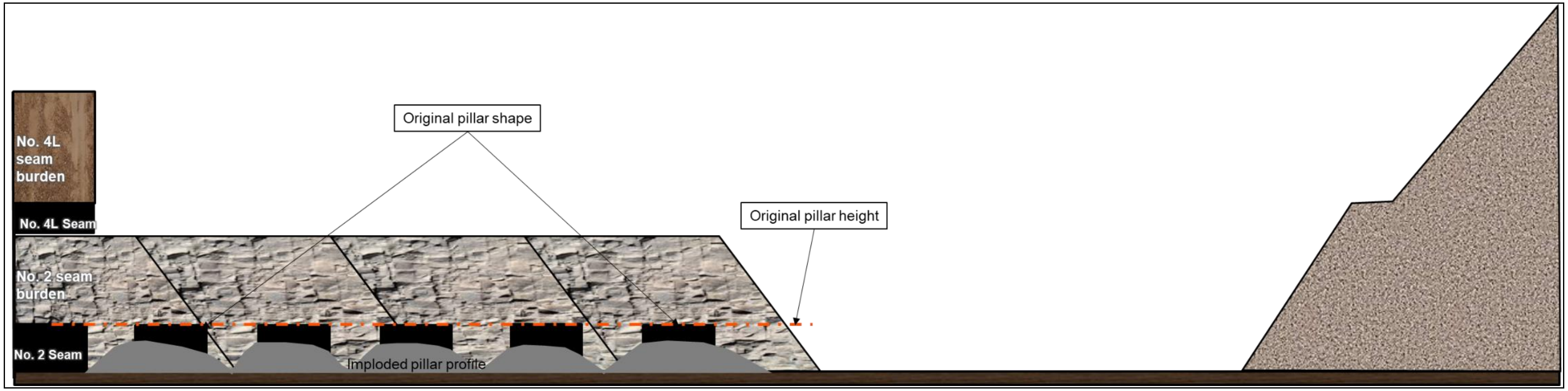

Figure 4-25: Imploded coal exposure height measured from the original coal pillar height.

The results as summarised in Appendices J and K show that the most suitable method to deal with pillars before exposure with opencast methods, whether in areas prone to spontaneous combustion or not, is bord collapse with the cleaning of waste between pillars. If the mine wants to implode pillars in some sections, pillar implosion with 2.5 m of waste and coal is removed during exposure. This is done to reduce dilution where the cast from different pillars meets and ultimately improves overall plant yield. The only issue with cleaning 2.5m of waste with coal is 26.5% of the pillar coal will be lost due to scalping.

4.7 Chapter Summary

Coal dilution in underground coal pillar mining through opencast mining methods is caused by various factors, but one cannot pinpoint the exact cause of dilution. To determine the exact causes of dilution, five possible causes were analysed. The five possible causes are strip orientation, effects of pillar scaling, effects of how bords collapse, current methods applied in dealing with pillars before exposure, and the influence of the current bonus incentive scheme. Strip orientation and bonus incentive scheme were found not to cause dilution, but the other three were found to contribute towards dilution in coal pillar mining.

In terms of coal losses, three areas were identified to be the major contributors to coal losses at MMS. The main contributors were found to be roof loss due to scalping by exposure equipment, floor coal losses caused by uncontrollable water accumulation and hard coal left on the floor because of the exposure equipment catching up with the coal extraction process, and the combination of both geological and geotechnical factors.

MCDA methods, AHP and VIKOR, were used in the review of the current methods that are applied in the treatment of pillars before the exposure by draglines and shovels. Seven methods were compared against each other and five of these methods were derived from two core methods, namely: pillar implosion and bord collapse. The outcome of the review was bord collapse with cleaning between pillars was the preferred method and followed by pillar implosion where 26.5% of coal pillar is lost while exposing.

5 Conclusion and Recommendations

5.1 Conclusion

This research study found that coal losses were indeed caused by the way pillars are imploded during blasting activities. The coal pillars are currently blasted with waste in a single charge of explosives and initiated using the same surface tie-up. This current blasting practice mixes the coal and waste uncontrollably. The uncontrolled mixture of waste and coal makes it difficult for exposure equipment to separate waste from coal. Except for the current blasting practices, high levels of water accumulation on the coaling face and hard coal on the floor leads to floor coal losses because water cannot be managed effectively, and exposure equipment catching up with the coal extraction process while trying to rip the hard coal on the floor. The combination of both geology and geotechnical factors was found to be the third major cause of coal losses due to inaccurate geological coal elevations and spoil failure.

Dilution at MMS is likely to be caused by pillar methods used to treat pillars before exposure by draglines and shovels. It was also found that the mode of bord collapse and effects of pillar scaling do contribute to high dilution when coal pillars are exposed through opencast mining methods. However, strip orientation and bonus incentive schemes were found to have minor to no contribution towards dilution on the mine.

The review of coal pillar methods using MCDA established that the most suitable method to treat pillars is bord collapse with waste cleaning between pillars. If pillar implosion is to be used in the mine, MCDA review found that pillar implosion with 2.5m of cleaning from the peak of the coal pillar profile was ideal. The issues with this method are that it can only be applied in areas that are not prone to spontaneous combustion and 26.5% of coal pillar loss can be incurred, which is unacceptable because it negatively affects the profitability of a mining operation.

5.2 Recommendations

This section presents the recommendations of the research study to address the findings that were identified during the study. The recommendations are listed as follows:

- When pillars are imploded, explosive deck charging must be applied. This will allow coal and waste to be blasted separately whereby the coal is fired first and then followed by the burden. This practice will reduced uncontrolled mixture of coal and waste and it will also reduce coal fines caused by the excessive explosive energy on the coal pillar zone;

- Improve the utilisation of the monitoring system installed in the machines to assist with plan compliance. The system such as Pegasys and Argus are installed in the draglines and shovels respectively. When using these systems, the operators must put them on dual-mode, where both the plan and section can be viewed simultaneously. The section view will show the bucket position in relation to the target elevation and the plan view will show the boundary of the plan. The proper utilisation of the monitoring systems can improve the bench accuracy to within 40 cm of the plan and this can eliminate roof coal losses due to exposure equipment digging into coal (Modular mining, 2021). Figure 5-1 shows an example of the view the operators should use while operating draglines and shovels.

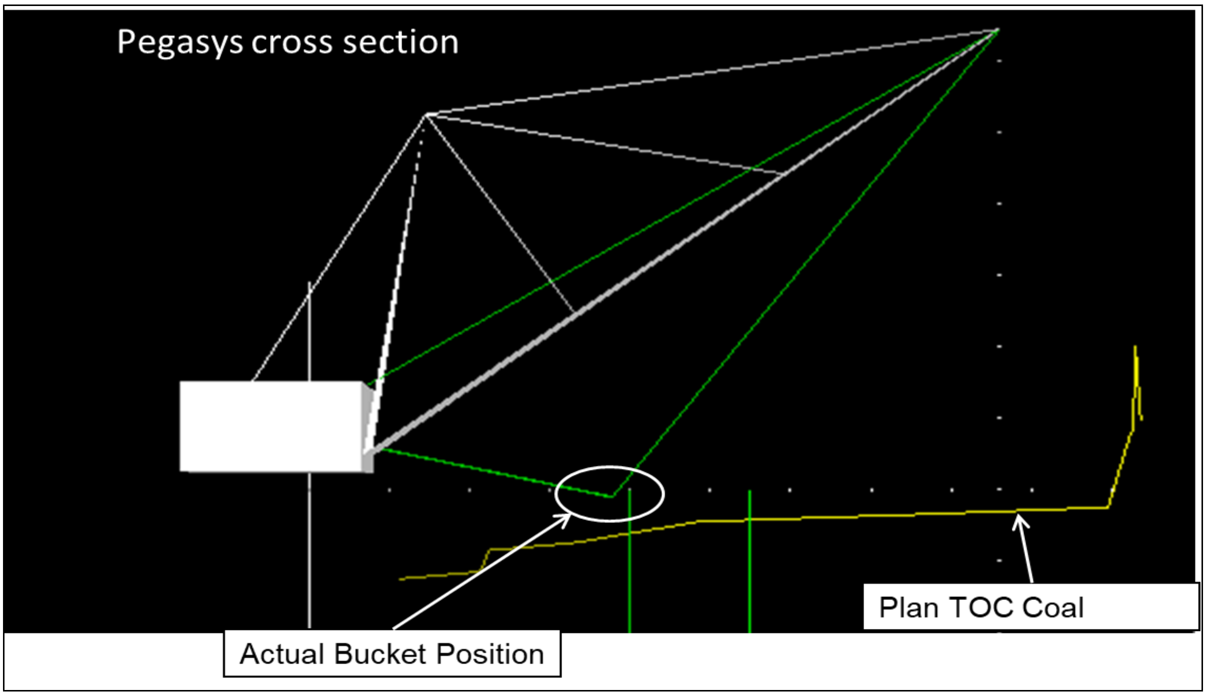

Figure 5-1: Section view of the operator screen in Pegasys

- Blend coal from different seams. This can be achieved either by sequencing the pits that send coal to the same crushing plant to be in different seams. If BMK East is mining no.2 seam coal pillars, BMK West must be on no. 4L coal. The blend of no. 2 seam pillars and virgin no. 4L coal is shown in Table 5-1.

Table 5-1: No. 2 seam pillar coal blended with virgin no.4L coal

Description	Pillar Implosion							
	"+0.5m"	"0m"	"-0.5m"	"-1m"	"-1.5m"	"-2m"	"-2.5m"	"-3m"
Blending with one block of S4L								
Ash	45.55	42.95	39.96	36.57	32.96	29.39	26.31	24.48
CV	15.23	16.20	17.33	18.59	19.94	21.26	22.37	22.92
RD	1.84	1.81	1.76	1.71	1.66	1.61	1.56	1.53
Overall Yield	49.30	53.28	57.89	63.10	68.65	74.12	78.82	81.51
Blending with two blocks of S4L								
Ash	38.47	36.31	33.94	31.40	28.83	26.44	24.49	23.35
CV	17.78	18.59	19.47	20.41	21.36	22.24	22.94	23.28
RD	1.75	1.72	1.68	1.64	1.61	1.57	1.54	1.53
Overall Yield	61.16	64.54	68.24	72.21	76.23	79.98	83.05	84.88

The blending shows an improvement in the yield across all pillar methods. The ratio that gives the best results is two blocks of no. 4L seam and one block of no. 2 seam pillars. If this cannot be constantly achieved, change the bench set-up from multi-pass to double bench. A double bench allows two benches to be exposed at the same time. The disadvantage with double bench is that there will be no blast gain due to buffers on the bottom bench and dragline productivity drops due to an increase in the amount of repositioning on the pad;

- Apply the current bord collapse method with some modifications. The addition to the current method is the cleaning of waste in between the pillars. The modification will improve overall plant yield and hence reduce operational costs because of less reject and discard being handled by the trucks. If waste can be removed in between the pillars, contamination can be reduced to between 0% and 10%. Figure 5.2 shows how contamination is cleaned in between the pillars.

Figure 5-2: Cleaning of waste material between coal pillars (New Hope Group, 2021)

However, for this method to properly work, a detailed sequence should be developed. The proposed steps to make this method successful is as follows:

Step 1: Clean waste around pillars 1 and 2 with an excavator. Figure 5-3 shows the bord collapse before cleaning between pillars and Figure 5-4 shows the cleaning of waste between pillars 1 to 3.

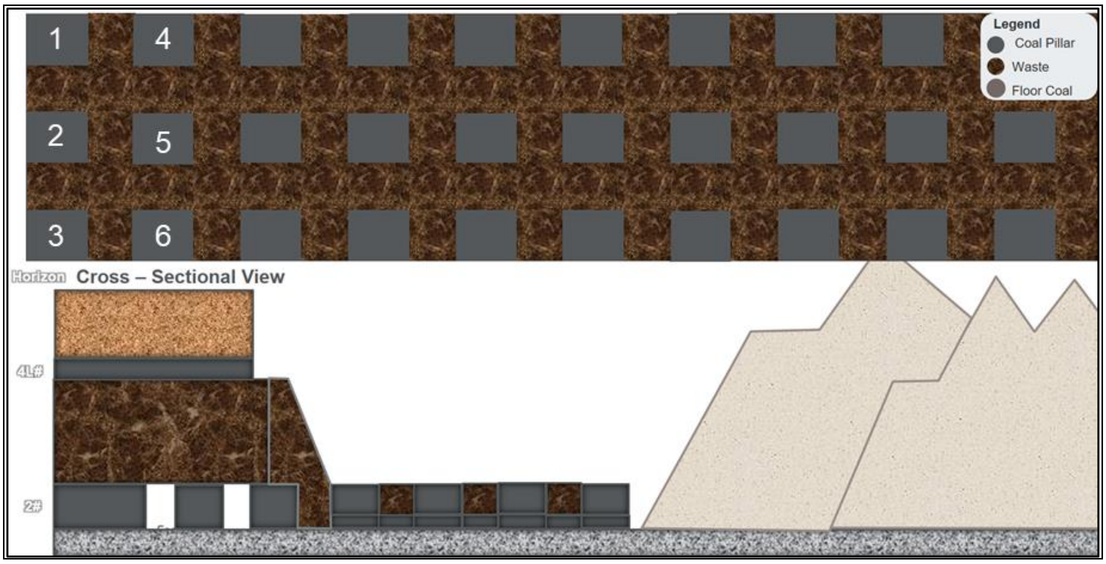

Figure 5-3: Bord collapse before cleaning of waste in between pillars.

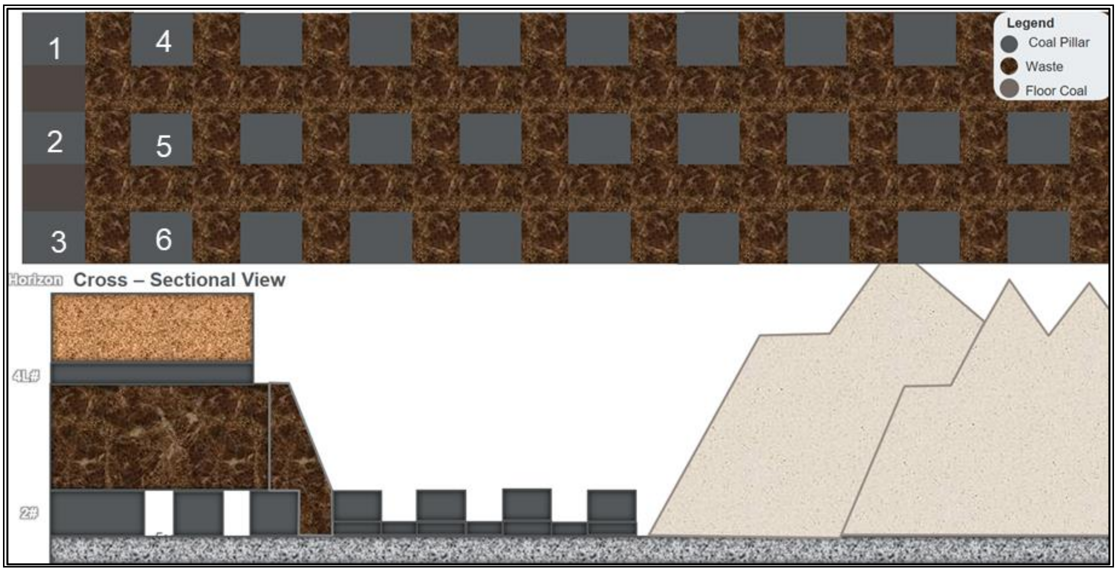

Figure 5-4: Bord collapse after cleaning waste between the pillars

Step 2: Once cleaning of waste between pillars is done, mine the coal pillars 1, 2, and 3 with loose coal, leaving hard coal on the floor. The extraction of coal pillars 1, 2, and 3 will expose the next waste. The waste must be cleaned until coal pillars 4, 5, and 6 are exposed.

Step 3: Extract coal pillar 5. The mining of coal pillar 5 is shown in Figure 5-5.

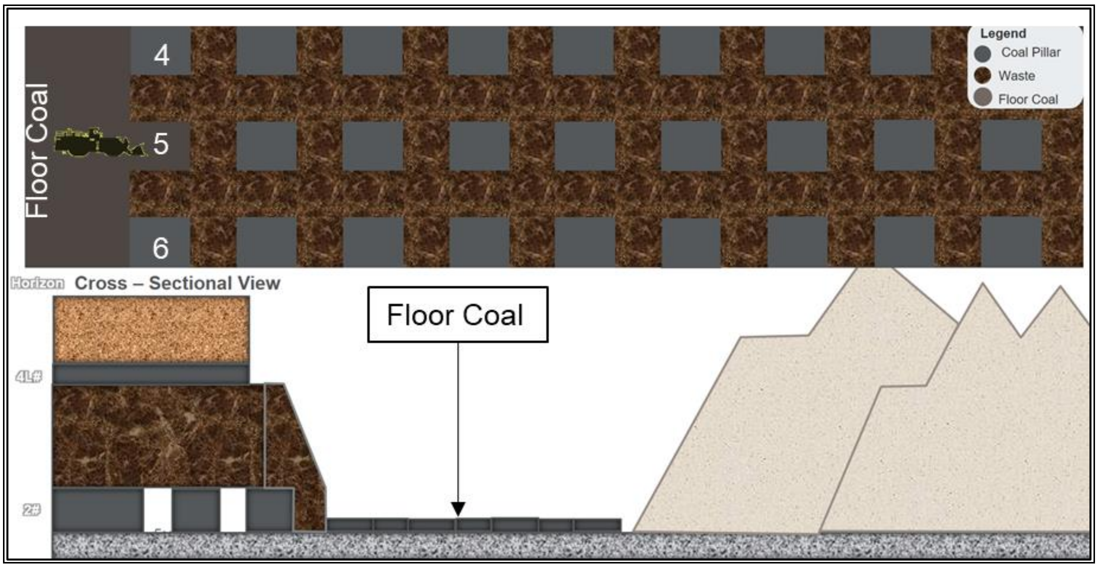

Figure 5-5: Loader mining pillar no. 5.

The mining of pillar no.5 will expose waste that is linking pillars 4 and 6, which must be removed until the sides of both pillars 4 and 6 and the pillar in front of pillar 5 is exposed. This sequence will continue until all the coal pillars with loose coal have been extracted.

Step 4: Rip the floor coal and if too hard, drill and blast the hard floor coal.

- Clean the upper burden highwall with an excavator before the dragline starts exposing the upper coal seam. This can be achieved by creating a dragline stand pad with dozers rather than letting the dragline create its own stand. The excavators clean the highwall as the dozers drop the blasted overburden. The pad height that the dozers must leave for the dragline to stand must be equal to or less than the maximum reach of the excavator used on the highwalls. This will eliminate frozen material falling on the shovels and an increase in blast gain percentage;
- To achieve more blast gain, change the buffer on the bottom bench and geotechnical stand-off between the upper and lower bench. The buffer can be reduced by applying a rule of thumb that was discussed in Chapter 2 (section 2.4.3). The recommended buffer width and geotechnical stand-off for a 50 m pit width are 20 m and 10 m respectively. These changes have the potential to improve the blast gain to 8% from 0%; and
- Review the current technology used to confirm coal pillar position because the current one will introduce the risk of spontaneous combustion in the underground workings.

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APPENDICES

APPENDIX A: Coal Loss results from Financial year 2018 to Financial year 2020

Table A1: Coal Losses for MMS South Pits for Financial Years 2018 and 2019

YTD Coal Losses From:		201807	To:	201906									
Pit	Short Term Plan	Mining Floor Losses	Mining Roof Losses	Not Exposed Not Mined Losses	Exposed But Not Mined Losses	Geological& Geotech Losses	Total Lossses Model and Mining	Total Mining Lossses	Mining Floor Losses	Mining Roof Losses	Not Exposed Not Mined Losses	Exposed But Not Mine Losses	Geological&Geotech Losses
	Tons	Tons	Tons	Tons	Tons	Tons	Tons	Tons	%	%	%	%	%
BMK EAST	3 066 080	270 570	147 692	14 338	126 944	473 314	1 031 216	557 902	9%	5%	0%	4%	15%
BMK WEST	1 783 663	248 991	178 292	11 264	38 644	39 624	503 982	464 358	14%	10%	1%	2%	2%
SKS	261 618	5 803	51	-	-	40 090	43 396	3 306	2%	0%	0%	0%	15%
VDD North	1 835 071	90 595	79 400	-	28 472	-	170 650	170 650	5%	4%	0%	2%	0%
Pit 4	3 071 163	361 049	152 045	-	14 699	-	512 223	512 223	12%	5%	0%	0%	0%
WVK TOTAL	**10 017 596**	**977 009**	**557 480**	**25 603**	**208 758**	**553 028**	**2 261 467**	**1 708 439**	**10%**	**6%**	**0%**	**2%**	**6%**
WVK FY18	**8 637 147**	**815 060**	**871 318**	**445 398**	**295 953**	**510 378**	**2 634 874**	**2 124 496**	**9%**	**10%**	**5%**	**3%**	**6%**

Table A2: Coal Losses for Financial Years 2019 and 2020

YTD Coal Losses:	From:	201907	To:	202006									
Pit	Short Term Plan	Mining Floor Losses	Mining Roof Losses	Not Exposed Not Mined Losses	Exposed But Not Mined Losses	Geological& Geotech Losses	Total Lossses Model and Mining	Total Mining Lossses	Mining Floor Losses	Mining Roof Losses	Not Exposed Not Mined Losses	Exposed But Not Mine Losses	Geological&Geotech Losses
	Tons	Tons	Tons	Tons	Tons	Tons	Tons	Tons	%	%	%	%	%
BMK EAST	1 287 124	111 433	-	-	1 101	255 575	368 110	112 534	9%	0%	0%	0%	20%
BMK WEST	883 543	239 603	159 624	7 230	-	77 031	483 488	406 457	27%	18%	1%	0%	9%
SKS	82 285	258	63 540	-	-	-	63 799	63 799	0%	77%	0%	0%	0%
VDD North	1 051 582	66 625	70 090	-	-	6 056	138 892	132 836	6%	7%	0%	0%	1%
Pit 4	2 465 863	170 917	117 533	-	1 186	106 075	390 668	284 593	7%	5%	0%	0%	4%
WVK TOTAL	**5 770 397**	**588 837**	**410 788**	**7 230**	**2 287**	**444 738**	**1 444 956**	**1 000 219**	**10%**	**7%**	**0%**	**0%**	**8%**
WVK FY19	**10 017 596**	**977 009**	**557 480**	**25 603**	**208 758**	**553 028**	**2 261 467**	**1 708 439**	**10%**	**6%**	**0%**	**2%**	**6%**

APPENDIX B: Financial Years 18 and 19 Production Physicals (Budget versus Actual)

Table B1: Financial year 18 actual waste moved versus budget waste

Pit	Description	Jul-17	Aug-17	Sep-17	Oct-17	Nov-17	Dec-17	Jan-18	Feb-18	Mar-18	Apr-18	May-18	Jun-18	FY18
SKS	BCMs Budget	1 153 383	1 314 294	1 031 900	937 371	994 399	1 107 623	1 005 005	984 299	1 491 142	1 295 731	400 200	400 200	12 115 547
	BCMs Actual	1 303 097	1 050 784	1 589 578	1 258 084	601 097	799 543	878 712	225 000	596 797	484 974	853 986	589 464	10 231 118
	Variance	149 714	-263 510	557 678	320 713	-393 302	-308 080	-126 293	-759 299	-894 345	-810 757	453 786	189 264	-1 884 429
BMK West	BCMs Budget	1 338 436	1 120 817	1 168 387	1 073 738	1 116 660	1 104 496	954 390	1 155 195	945 103	1 027 124	1 043 813	1 043 024	13 091 183
	BCMs Actual	1 371 953	1 059 399	1 581 578	912 590	1 013 871	749 813	1 719 160	1 088 658	966 261	1 208 044	1 338 812	1 287 723	14 297 862
	Variance	33 517	-61 418	413 191	-161 148	-102 789	-354 683	764 770	-66 537	21 158	180 920	294 999	244 699	1 206 679
BMK East	BCMs Budget	1 542 126	1 493 409	1 388 424	1 100 957	1 251 798	1 169 643	1 287 150	1 259 996	1 352 123	1 208 668	1 340 022	1 654 666	16 048 982
	BCMs Actual	1 392 808	1 819 611	1 921 116	1 161 193	1 864 455	1 376 010	1 424 630	1 530 990	1 361 624	1 170 420	1 634 621	2 041 871	18 699 351
	Variance	-149 318	326 202	532 692	60 236	612 657	206 367	137 480	270 994	9 501	-38 248	294 599	387 205	2 650 369
VDD North	BCMs Budget	673 644	750 741	705 096	686 294	693 702	563 177	647 829	635 783	555 991	699 846	750 741	779 736	8 142 580
	BCMs Actual	786 229	676 381	818 282	662 529	614 076	465 645	513 832	608 480	625 052	545 792	603 489	833 230	7 753 016
	Variance	112 585	-74 360	113 186	-23 766	-79 627	-97 532	-133 997	-27 303	69 062	-154 054	-147 252	53 494	-389 564
PIT 4	BCMs Budget	55 400	344 970	487 160	494 590	421 760	411 480	409 560	392 720	819 160	786 320	815 358	791 920	6 230 398
	BCMs Actual	85 867	568 265	627 229	550 378	530 408	595 969	485 327	535 027	616 053	493 504	1 059 971	663 453	6 811 451
	Variance	30 467	223 295	140 069	55 788	108 648	184 489	75 767	142 307	-203 107	-292 816	244 613	-128 467	581 053
Total BCMs	**BCMs Budget**	**4 762 989**	**5 024 231**	**4 780 967**	**4 292 950**	**4 478 320**	**4 356 419**	**4 303 934**	**4 427 993**	**5 163 519**	**5 017 689**	**4 350 134**	**4 669 546**	**55 628 691**
	BCMs Actual	**4 939 955**	**5 174 441**	**6 537 782**	**4 544 775**	**4 623 907**	**3 986 981**	**5 021 660**	**3 988 155**	**4 165 788**	**3 902 735**	**5 490 879**	**5 415 741**	**57 792 798**
	Variance	**176 966**	**150 210**	**1 756 815**	**251 825**	**145 587**	**-369 439**	**717 726**	**-439 837**	**-997 731**	**-1 114 955**	**1 140 745**	**746 195**	**2 164 107**

Table B2: Financial year 18 actual versus budget run of mine coal

Pit	Description	Jul-17	Aug-17	Sep-17	Oct-17	Nov-17	Dec-17	Jan-18	Feb-18	Mar-18	Apr-18	May-18	Jun-18	FY18
SKS	ROM Mined Budget	233 547	208 535	205 946	273 330	252 295	278 563	285 016	279 447	187 791	235 603	178 166	113 871	2 732 111
	ROM Mined Actual	263 553	173 824	275 905	255 914	248 168	98 238	269 289	46 220	59 395	99 331	115 303	194 233	2 099 374
	Variance	30 006	-34 712	69 960	-17 416	-4 127	-180 325	-15 726	-233 227	-128 396	-136 272	-62 863	80 362	-632 737
BMK West	ROM Mined Budget	196 366	211 002	186 901	193 353	213 106	183 341	252 607	166 331	208 984	182 924	225 684	191 150	2 411 750
	ROM Mined Actual	224 047	99 965	154 531	167 543	173 533	129 193	135 989	179 139	117 672	198 237	198 885	199 503	1 978 238
	Variance	27 681	-111 037	-32 370	-25 810	-39 573	-54 148	-116 618	12 808	-91 312	15 313	-26 799	8 353	-433 512
BMK East	ROM Mined Budget	295 499	292 492	301 432	235 575	328 018	243 033	222 023	287 626	320 366	292 996	296 581	305 703	3 421 344
	ROM Mined Actual	161 579	345 607	249 604	326 561	350 509	265 603	336 309	319 205	346 264	267 931	257 855	319 115	3 546 142
	Variance	-133 920	53 115	-51 828	90 985	22 491	22 570	114 286	31 579	25 898	-25 065	-38 726	13 412	124 798
VDD North	ROM Mined Budget	0	0	0	125 155	61 058	97 592	106 000	92 975	143 367	113 371	151 030	185 450	1 075 997
	ROM Mined Actual	63 973	113 824	41 464	32 106	60 858	81 209	49 820	83 056	68 027	95 879	130 885	122 100	943 201
	Variance	63 973	113 824	41 464	-93 049	-200	-16 383	-56 180	-9 919	-75 340	-17 492	-20 145	-63 350	-132 796
PIT 4	ROM Mined Budget	0	0	0	0	0	0	0	0	129 519	204 501	206 402	279 111	819 533
	ROM Mined Actual	0	0	0	0	62 543	4 036	0	37 371	92 814	34 007	157 598	122 165	510 534
	Variance	0	0	0	0	62 543	4 036	0	37 371	-36 705	-170 494	-48 804	-156 946	-308 999
Total ROM	**ROM Mined BD**	**725 412**	**712 029**	**694 279**	**827 413**	**854 477**	**802 528**	**865 646**	**826 379**	**990 027**	**1 029 395**	**1 057 863**	**1 075 285**	**10 460 734**
	ROM Mined LE	**713 153**	**733 220**	**721 505**	**782 124**	**895 611**	**578 279**	**791 408**	**664 991**	**684 172**	**695 385**	**860 526**	**957 116**	**9 077 489**
	Variance	**-12 259**	**21 191**	**27 225**	**-45 289**	**41 134**	**-224 249**	**-74 238**	**-161 388**	**-305 855**	**-334 010**	**-197 337**	**-118 169**	**-1 383 245**

Table B3: Financial year 18 actual versus budget crushed coal volume

Pit	Description	Jul-17	Aug-17	Sep-17	Oct-17	Nov-17	Dec-17	Jan-18	Feb-18	Mar-18	Apr-18	May-18	Jun-18	FY18
SKS	ROM Crushed Budget	233 547	208 535	205 946	273 330	252 295	278 563	285 016	279 447	187 791	235 603	178 166	113 871	2 732 111
	ROM Crushed Actual	280 844	138 436	248 018	276 837	223 505	162 629	259 692	34 094	83 744	99 331	115 303	194 233	2 116 665
	Variance	47 296	-70 100	42 072	3 507	-28 790	-115 934	-25 324	-245 353	-104 047	-136 272	-62 863	80 362	-615 445
BMK Total	ROM Crushed Budget	491 865	503 494	488 334	428 928	541 124	426 373	474 630	453 957	529 350	475 920	522 265	496 853	5 833 094
	ROM Crushed Actual	377 446	438 249	430 977	484 640	436 454	488 110	414 199	510 198	513 918	453 420	434 476	518 618	5 500 705
	Variance	-114 419	-65 245	-57 357	55 712	-104 670	61 737	-60 431	56 241	-15 432	-22 500	-87 789	21 765	-332 389
VDD North	ROM Crushed Budget	0	0	0	125 155	61 058	97 592	106 000	92 975	143 367	113 371	151 030	185 450	1 075 997
	ROM Crushed Actual	63 973	113 824	41 464	32 106	60 858	81 209	49 820	83 056	68 027	95 879	130 885	122 100	943 202
	Variance	63 973	113 824	41 464	-93 049	-200	-16 383	-56 180	-9 919	-75 340	-17 492	-20 145	-63 350	-132 795
PIT 4	ROM Crushed Budget	0	0	0	0	0	0	0	0	129 519	204 501	206 402	279 111	819 533
	ROM Crushed Actual	0	0	0	0	62 543	4 036	0	37 371	92 814	34 007	129 729	92 165	452 665
	Variance	0	0	0	0	62 543	4 036	0	37 371	-36 705	-170 494	-76 673	-186 946	-366 868
Total ROM	**ROM Crushed Budget**	725 412	712 029	694 279	827 413	854 477	802 528	865 646	826 379	990 027	1 029 395	1 057 863	1 075 285	10 460 734
	ROM Crushed Actual	722 263	690 509	720 459	793 583	783 360	735 984	723 711	664 719	758 503	682 637	810 393	927 116	9 013 237
	Variance	-3 149	-21 520	26 180	-33 830	-71 117	-66 544	-141 935	-161 660	-231 524	-346 758	-247 470	-148 169	-1 447 497

Table B4: Financial year 18 actual versus budget saleable produced, feed to plant and plant yields.

Activity	Description	Jul-17	Aug-17	Sep-17	Oct-17	Nov-17	Dec-17	Jan-18	Feb-18	Mar-18	Apr-18	May-18	Jun-18	FY18
Feed														
SEP FTP	Export Budget	652 871	640 826	631 794	728 124	786 118	722 276	761 768	768 533	851 423	957 338	930 919	978 509	9 410 500
	Export Actual	725 703	640 686	655 235	762 055	705 751	715 528	699 344	622 735	683 219	633 257	752 318	850 000	8 445 831
	Variance	72 832	-140	23 441	33 932	-80 367	-6 748	-62 424	-145 798	-168 204	-324 081	-178 601	-128 509	-964 668
Saleable														
SEP Export	Budget	319 907	317 209	291 098	299 292	400 711	351 174	413 672	411 716	436 555	421 814	477 694	480 823	4 621 665
	Actual	396 770	377 181	384 693	422 954	381 196	394 072	407 873	379 893	424 686	345 698	424 052	471 325	4 810 393
	Variance	76 863	59 972	93 595	123 662	-19 515	42 898	-5 799	-31 824	-11 869	-76 116	-53 642	-9 498	188 728
SEP Middlings(External Saleable)	Budget	52 752	38 450	71 048	63 074	71 704	62 364	62 263	75 914	63 140	67 079	70 262	72 821	770 869
	Actual	25 177	30 598	37 522	35 325	24 551	24 391	30 236	25 150	8 834	5 664	6 375	0	253 823
	Variance	-27 575	-7 852	-33 526	-27 749	-47 153	-37 973	-32 027	-50 764	-54 306	-61 415	-63 887	-72 821	-517 046
Yield														
SEP PrimaryYield	Budget	49%	50%	46%	41%	51%	49%	54%	54%	51%	44%	51%	49%	49%
	Actual	55%	59%	59%	56%	54%	55%	58%	61%	62%	55%	56%	55%	57%
	Variance	6%	9%	13%	14%	3%	6%	4%	7%	11%	11%	5%	6%	8%
SEP SecondaryYield	Budget	8%	6%	11%	9%	9%	9%	8%	10%	7%	7%	8%	7%	8%
	Actual	3%	5%	6%	5%	3%	3%	4%	4%	1%	1%	1%	0%	3%
	Variance	-5%	-1%	-6%	-4%	-6%	-5%	-4%	-6%	-6%	-6%	-7%	-7%	-5%
Total Saleable														
Total Saleable Production (Incl Middlings)	**Budget**	372 659	355 659	362 146	362 366	472 414	413 538	475 935	487 631	499 695	488 893	547 955	553 643	5 392 534
	Actual	421 947	407 779	422 215	458 279	405 747	418 463	438 109	405 043	433 520	351 362	430 427	471 325	5 064 216
	Variance	49 288	52 120	60 069	95 913	-66 667	4 925	-37 826	-82 588	-66 175	-137 531	-117 529	-82 318	-328 318

Table B5: Financial year 19 actual waste moved versus budget waste

Pit	Description	Jul-18	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18	Jan-19	Feb-19	Mar-19	Apr-19	May-19	Jun-19	FY19
SKS	BCMs Budget	0	0	0	0	0	0	0	0	0	0	0	0	0
	BCMs Actual	112 232	0	0	0	220 836	0	0	0	0	0	0	0	333 068
	Variance	112 232	0	0	0	220 836	0	0	0	0	0	0	0	333 068
BMK West	BCMs Budget	1 124 041	973 850	918 658	1 198 986	983 051	1 156 095	1 113 533	899 339	971 400	1 057 416	1 377 575	889 294	12 663 238
	BCMs Actual	1 184 146	1 067 173	549 439	1 682 486	868 374	1 089 357	1 238 848	803 923	867 622	1 098 849	1 013 501	1 354 257	12 817 975
	Variance	60 105	93 323	-369 219	483 500	-114 677	-66 738	125 315	-95 416	-103 778	41 433	-364 074	464 963	154 737
BMK East	BCMs Budget	1 387 424	1 459 135	1 534 978	1 509 598	1 387 715	1 332 999	1 447 017	1 264 695	1 422 225	1 451 506	1 543 499	1 424 672	17 165 463
	BCMs Actual	2 674 039	1 441 918	1 618 335	1 611 578	1 400 997	2 148 062	2 392 371	1 736 192	1 440 504	1 939 861	1 357 237	1 301 031	21 062 126
	Variance	1 286 615	-17 217	83 357	101 980	13 282	815 063	945 354	471 497	18 279	488 356	-186 262	-123 641	3 896 663
VDD North	BCMs Budget	1 180 079	1 091 001	958 419	1 700 698	1 600 230	1 891 874	1 671 825	1 738 786	1 598 118	1 820 872	1 645 197	1 798 428	18 695 527
	BCMs Actual	1 110 796	1 101 841	896 695	1 064 178	1 057 315	1 263 609	1 049 074	1 039 028	1 721 751	1 681 873	2 186 330	1 688 029	15 860 518
	Variance	-69 283	10 840	-61 724	-636 520	-542 915	-628 265	-622 751	-699 758	123 633	-139 000	541 132	-110 399	-2 835 009
PIT 4	BCMs Budget	1 345 298	1 363 049	1 455 750	1 547 737	1 208 614	1 669 447	1 619 925	1 330 752	1 579 036	1 581 898	1 750 780	1 408 173	17 860 458
	BCMs Actual	935 208	1 498 271	1 249 543	879 073	816 529	1 579 765	1 336 232	1 333 937	1 708 390	1 456 122	1 539 061	1 522 288	15 854 419
	Variance	-410 090	135 222	-206 207	-668 664	-392 085	-89 682	-283 693	3 185	129 354	-125 776	-211 719	114 115	-2 006 039
Total BCMs	**BCMs Budget**	5 036 842	4 887 035	4 867 806	5 957 019	5 179 609	6 050 414	5 852 300	5 233 572	5 570 780	5 911 692	6 317 052	5 520 567	66 384 687
	BCMs Actual	6 016 421	5 109 203	4 314 012	5 237 315	4 364 051	6 072 623	6 016 525	4 913 080	5 738 267	6 176 706	6 096 129	5 865 605	65 919 937
	Variance	979 579	222 168	-553 794	-719 704	-815 558	22 209	164 225	-320 492	167 487	265 014	-220 923	345 038	-464 750

Table B6: Financial year 19 actual versus budget run of mine coal

Pit	Description	Jul-18	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18	Jan-19	Feb-19	Mar-19	Apr-19	May-19	Jun-19	FY19
SKS	ROM Mined Budget	0	0	0	0	0	0	0	0	0	0	0	0	0
	ROM Mined Actual	4 803	12 207	0	0	56 099	0	0	0	0	0	0	0	73 109
	Variance	4 803	12 207	0	0	56 099	0	0	0	0	0	0	0	73 109
BMK West	ROM Mined Budget	206 483	142 121	57 103	197 293	183 698	205 472	174 366	192 930	238 557	171 884	185 230	198 557	2 153 694
	ROM Mined Actual	175 986	119 347	74 153	189 356	173 459	190 734	100 270	181 874	188 961	136 932	149 556	191 761	1 872 388
	Variance	-30 497	-22 774	17 049	-7 937	-10 239	-14 738	-74 096	-11 056	-49 596	-34 952	-35 674	-6 796	-281 306
						48 591	-10 856	191 051	120 487	7 333	-113 799	104 678	-8 443	
BMK East	ROM Mined Budget	279 208	215 830	186 846	291 822	304 968	247 568	271 336	257 012	250 194	244 977	239 774	290 073	3 079 607
	ROM Mined Actual	271 180	271 637	407 695	274 302	257 851	350 206	355 932	298 496	278 604	505 010	357 395	251 234	3 879 542
	Variance	-8 028	55 807	220 849	-17 520	-47 117	102 638	84 596	41 484	28 410	260 033	117 621	-38 839	799 935
								132 389						
VDD North	ROM Mined Budget	132 086	243 852	183 499	241 302	270 190	247 080	253 611	242 355	241 845	247 657	249 275	241 898	2 794 651
	ROM Mined Actual	162 578	110 635	128 672	220 811	181 914	182 091	194 333	128 239	119 873	104 680	157 992	267 078	1 958 895
	Variance	30 492	-133 217	-54 827	-20 491	-88 276	-64 989	-59 278	-114 116	-121 972	-142 977	-91 284	25 180	-835 755
PIT 4	ROM Mined Budget	149 535	147 689	159 478	219 779	298 909	284 249	319 748	318 905	323 568	316 432	321 689	318 311	3 178 292
	ROM Mined Actual	116 522	239 389	179 993	223 291	105 330	312 461	297 899	149 449	222 594	198 487	189 101	210 416	2 444 932
	Variance	-33 013	91 700	20 515	3 512	-193 579	28 212	-21 849	-169 456	-100 974	-117 945	-132 588	-107 895	-733 360
Total ROM	**ROM Mined Budget**	767 312	749 492	586 926	950 196	1 057 765	984 369	1 019 061	1 011 203	1 054 164	980 949	995 968	1 048 839	11 206 244
	ROM Mined Actual	731 069	753 215	790 512	907 760	774 653	1 035 493	948 434	758 058	810 032	945 109	854 043	920 489	10 228 866
	Variance	-36 243	3 723	203 586	-42 436	-283 112	51 124	-70 627	-253 145	-244 132	-35 840	-141 925	-128 350	-977 378

Table B7: Financial year 19 actual versus budget crushed coal volume

Pit	Description	Jul-18	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18	Jan-19	Feb-19	Mar-19	Apr-19	May-19	Jun-19	FY19
SKS ROM	ROM Crushed Budget													0
	ROM Crushed Actual	4 803	12 207	0	0	56 099	0	0	0	0	0	0	0	73 109
	Variance	4 803	12 207	0	0	56 099	0	0	0	0	0	0	0	73 109
BMK ROM STOCKPILE	ROM Crushed Budget													0
	ROM Crushed Actual				0	28 585	0	0	0				13 900	42 486
	Variance	0	0	0	0	28 585	0	0	0	0	0	0	13 900	42 486
SKS ROM STOCKPILE	ROM Crushed Budget													0
	ROM Crushed Actual		121 305	0		22 347	0	0	47 385		4 885	6 462		202 384
	Variance	0	121 305	0	0	22 347	0	0	47 385	0	4 885	6 462	0	202 384
BMK Total	ROM Crushed Budget	485 691	357 951	243 949	489 115	488 666	453 040	445 702	449 942	488 751	416 861	425 004	488 630	5 233 302
	ROM Crushed Actual	434 633	358 943	511 864	473 604	431 310	522 793	453 460	476 774	457 946	582 364	537 600	442 995	5 684 286
	Variance	-51 058	993	267 915	-15 511	-57 356	69 754	7 758	26 831	-30 805	165 503	112 596	-45 635	450 985
VDD North	ROM Crushed Budget	132 086	243 852	183 499	241 302	270 190	247 080	253 611	242 355	241 845	247 657	249 275	241 898	2 794 651
	ROM Crushed Actual	62 406	110 635	190 255	209 638	181 914	182 091	194 333	128 239	127 092	104 680	157 992	135 232	1 784 507
	Variance	-69 680	-133 217	6 757	-31 664	-88 276	-64 989	-59 278	-114 116	-114 753	-142 977	-91 284	-106 666	-1 010 143
PIT 4	ROM Crushed Budget	149 535	147 689	159 478	219 779	298 909	284 249	319 748	318 905	323 568	316 432	321 689	318 311	3 178 292
	ROM Crushed Actual	116 522	239 389	134 131	212 118	105 330	298 940	246 172	149 449	211 010	198 487	189 101	210 416	2 311 064
	Variance	-33 013	91 700	-25 348	-7 661	-193 579	14 690	-73 576	-169 456	-112 558	-117 945	-132 588	-107 895	-867 228
Total ROM	ROM Crushed Budget	767 312	749 492	586 926	950 196	1 057 765	984 369	1 019 061	1 011 203	1 054 164	980 949	995 968	1 048 839	11 206 244
	ROM Crushed Actual	618 364	842 479	836 250	895 360	825 585	1 003 824	893 965	801 847	796 048	890 416	891 154	802 543	10 097 836
	Variance	-148 948	92 988	249 324	-54 836	-232 179	19 455	-125 096	-209 356	-258 116	-90 533	-104 814	-246 296	-1 108 408

Table B8: Financial year 19 actual versus budget saleable produced, feed to plant and plant yields.

Activity	Description	Jul-18	Aug-18	Sep-18	Oct-18	Nov-18	Dec-18	Jan-19	Feb-19	Mar-19	Apr-19	May-19	Jun-19	FY19
SEP FTP	Export Budget	650 000	667 047	545 841	825 101	939 559	879 484	925 226	918 685	938 645	903 011	914 226	916 931	10 023 757
	Export Actual	550 340	807 107	743 792	817 151	751 987	841 339	816 336	739 732	706 226	766 837	758 855	708 880	9 008 583
	Variance	-99 660	140 059	197 951	-7 950	-187 571	-38 145	-108 890	-178 953	-232 419	-136 174	-155 372	-208 051	-1 015 175
Saleable														
SEP Export	Budget	370 497	380 231	311 130	512 331	555 555	551 750	535 829	553 359	549 522	554 299	571 128	560 726	6 006 356
	Actual	328 764	431 209	380 994	438 614	391 469	458 418	446 302	427 529	424 323	405 538	379 247	407 801	4 920 207
	Variance	-41 733	50 978	69 864	-73 716	-164 086	-93 332	-89 527	-125 829	-125 199	-148 761	-191 881	-152 925	-1 086 149
5Seam	Budget	0	0	0	0	0	0	0	0	0	0	0	0	0
	Actual	0	0	0	21 899	43 375	25 587	34 115	10 201	0	23 708	33 484	0	192 369
	Variance	0	0	0	21 899	43 375	25 587	34 115	10 201	0	23 708	33 484	0	192 369
SEP Middlings(External Saleable)	Budget	0	0	21 103	0	22 548	0	0	0	18 120	13 521	0	0	75 292
	Actual	3 398	53 316	64 104	47 811	11 645	45 018	44 852	44 852	32 798	0	0	0	347 794
	Variance	3 398	53 316	43 001	47 811	-10 903	45 018	44 852	44 852	14 678	-13 521	0	0	272 502
Yield														
SEP PrimaryYield	Budget	57%	57%	57%	62%	59%	63%	58%	60%	59%	61%	62%	61%	60%
	Actual	60%	53%	51%	54%	52%	54%	55%	58%	60%	53%	50%	58%	55%
	Variance	3%	-4%	-6%	-8%	-7%	-8%	-3%	-2%	2%	-8%	-12%	-4%	-4.9%
SEP SecondaryYield	Budget	0%	0%	4%	0%	2%	0%	0%	0%	2%	1%	0%	0%	1%
	Actual	1%	7%	9%	6%	2%	5%	5%	6%	5%	0%	0%	0%	4%
	Variance	1%	7%	5%	6%	-1%	5%	5%	6%	3%	-1%	0%	0%	3%
5700kcal	Budget	370 497	380 231	311 130	512 331	555 555	551 750	535 829	553 359	549 522	554 299	571 128	560 726	6 006 356
	Actual	328 764	431 209	380 994	438 614	391 469	458 418	446 302	427 529	0	405 538	379 247	407 801	4 495 884
	Variance	-41 733	50 978	69 864	-73 716	-164 086	-93 332	-89 527	-125 829	-549 522	-148 761	-191 881	-152 925	-1 510 472
5500kcal	Budget	0	0	0	0	0	0	0	0	0	0	0	0	0
	Actual	0	0	0	0	0	0	0	0	424 323	0	30 543	0	454 866
	Variance	0	0	0	0	0	0	0	0	424 323	0	30 543	0	454 866
4800 Product	Budget	0	0	0	0	0	0	0	0	0	0	0	0	0
	Actual	0	0	0	0	0	0	0	6 908	0	44 849	31 274	15 310	98 341
	Variance	0	0	0	0	0	0	0	6 908	0	44 849	31 274	15 310	98 341
Total SEP Export	Budget	370 497	380 231	311 130	512 331	555 555	551 750	535 829	553 359	549 522	554 299	571 128	560 726	6 006 356
	Actual	328 764	431 209	380 994	460 513	434 844	484 005	480 417	444 638	424 323	474 095	474 548	423 111	5 241 460
	Variance	-41 733	50 978	69 864	-51 818	-120 711	-67 745	-55 412	-108 721	-125 199	-80 204	-96 580	-137 615	-764 896
Total Saleable Production (Incl Middlings)	**Budget**	370 497	380 231	332 233	512 331	578 103	551 750	535 829	553 359	567 642	567 820	571 128	560 726	6 081 648
	Actual	332 162	484 525	445 098	508 324	446 489	529 023	525 269	489 490	457 121	474 095	474 548	423 111	5 589 254
	Variance	-38 335	104 294	112 865	-4 007	-131 614	-22 727	-10 560	-63 869	-110 521	-93 725	-96 580	-137 615	-492 394

APPENDIX C: Charging Instructions for VDDN and BMK East pits

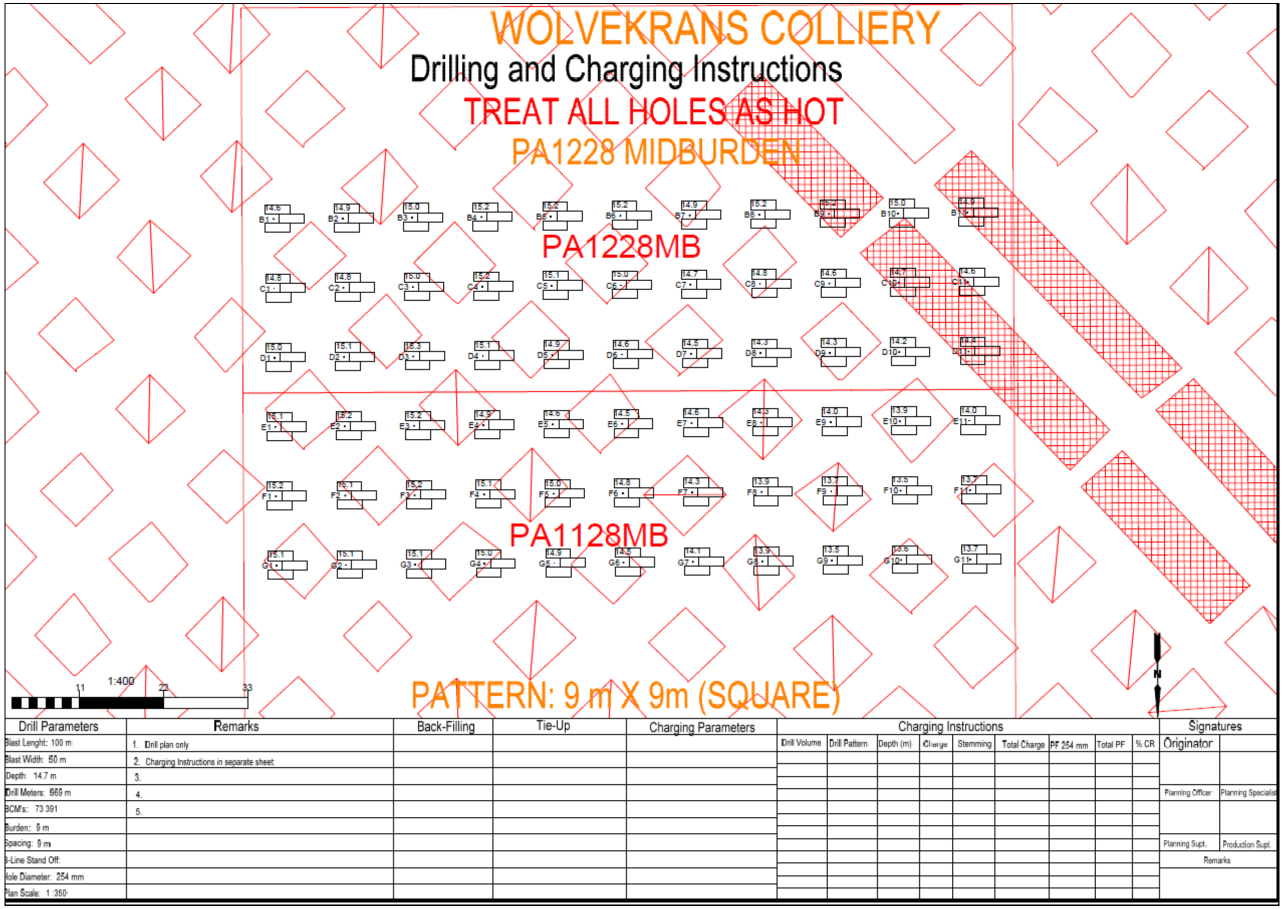

Figure C1: Drill plan for BMK East No. 2 Seam Burden The figure is not legible. Use the landscape format to increase the size

Table C1: BMK East 2 Seam burden charging instructions The letters and numbers in this table are barely legible.

PA1228 S2RP HOT HOLE AREA Charging Parameters: Hef 207: Cup Density @ 0.95g/cm3 Booster Position: Top Priming 800g Booster Holes above 45 degrees are HOT No Back Filling AVERAG DEPTH: 14.7 POWDER FACTOR: 0.61						DESWIK PLAN Blasters Feedback			
Hole Number	Hole Depth	Temp. before cooling	Temp. after cooling	Stemming	Kg's per Hole	Actual Depth	Actual Stemming	Actual KG's per Hole	Comments
B1	14.6			3	695.6				
B2	14.9			3	713.6				
B3	15			3	719.6				
B4	15.2			3	731.6				
B5	15.2			3	731.6				
B6	15.2			3	731.6				
B7	14.9			3	713.6				
B8	15.2			3	731.6				
B9	15.2			3	731.6				
B10	15			3	719.6				
B11	14.9			3	713.6				
C1	14.8			3	707.6				
C2	14.8			3	707.6				
C3	15			3	719.6				
C4	15.2			3	731.6				
C5	15.1			3	725.6				
C6	15			3	719.6				
C7	14.7			3	701.6				
C8	14.8			3	707.6				
C9	14.6			3	695.6				
C10	14.7			3	701.6				
C11	14.6			3	695.6				
D1	15			3	719.6				
D2	15.1			3	725.6				
D3	15.3			3	737.6				
D4	15.1			3	725.6				
D5	14.9			3	713.6				
D6	14.6			3	695.6				
D7	14.5			3	689.6				
D8	14.3			3	675.2				
D9	14.3			3	675.2				
D10	14.2			3	669.2				
D11	14.4			3	681.2				
E1	15.1			3	725.6				
E2	15.2			3	731.6				
E3	15.2			3	731.6				
E4	14.9			3	713.6				
E5	14.6			3	695.6				
E6	14.5			3	689.6				
E7	14.6			3	695.6				
E8	14.3			3	675.2				
E9	14			3	657.3				
E10	13.9			3	651.3				
E11	14			3	657.3				

PA1228 S2RP HOT HOLE AREA Charging Parameters: Hef 207: Cup Density @ 0.95g/cm3 Booster Position: Top Priming 800g Booster Holes above 45 degrees are HOT No Back Filling AVERAG DEPTH: 14.7 POWDER FACTOR: 0.61						DESWIK PLAN Blasters Feedback			
Hole Number	Hole Depth	Temp. before cooling	Temp. after cooling	Stemming	Kg's per Hole	Actual Depth	Actual Stemming	Actual KG's per Hole	Comments
F1	15.2			3	731.6				
F2	15.1			3	725.6				
F3	15.2			3	731.6				
F4	15.1			3	725.6				
F5	15			3	719.6				
F6	14.8			3	707.6				
F7	14.3			3	675.2				
F8	13.9			3	651.3				
F9	13.7			3	639.3				
F10	13.5			3	627.4				
F11	13.7			3	639.3				
G1	15.1			3	725.6				
G2	15.1			3	725.6				
G3	15.1			3	725.6				
G4	15.0			3	719.6				
G5	14.9			3	713.6				
G6	14.5			3	689.6				
G7	14.1			3	663.3				
G8	13.9			3	651.3				
G9	13.5			3	627.4				
G10	13.6			3	633.4				
G11	13.7			3	639.3				
Extra Holes									
Cut Width	50m	Drill Meters	969m	Burden	9m	Notes			
Blast Length	100m	BCM's	73 379	Spacing	9m	Area tested negative for a reactive ground			
AVE Depth	14.7m	Hole Diam	260mm	Drill Vol	81m	Tie-up: Full chevron with 75ms			
Total holes	66	Stemming	3m	PF	0.61				

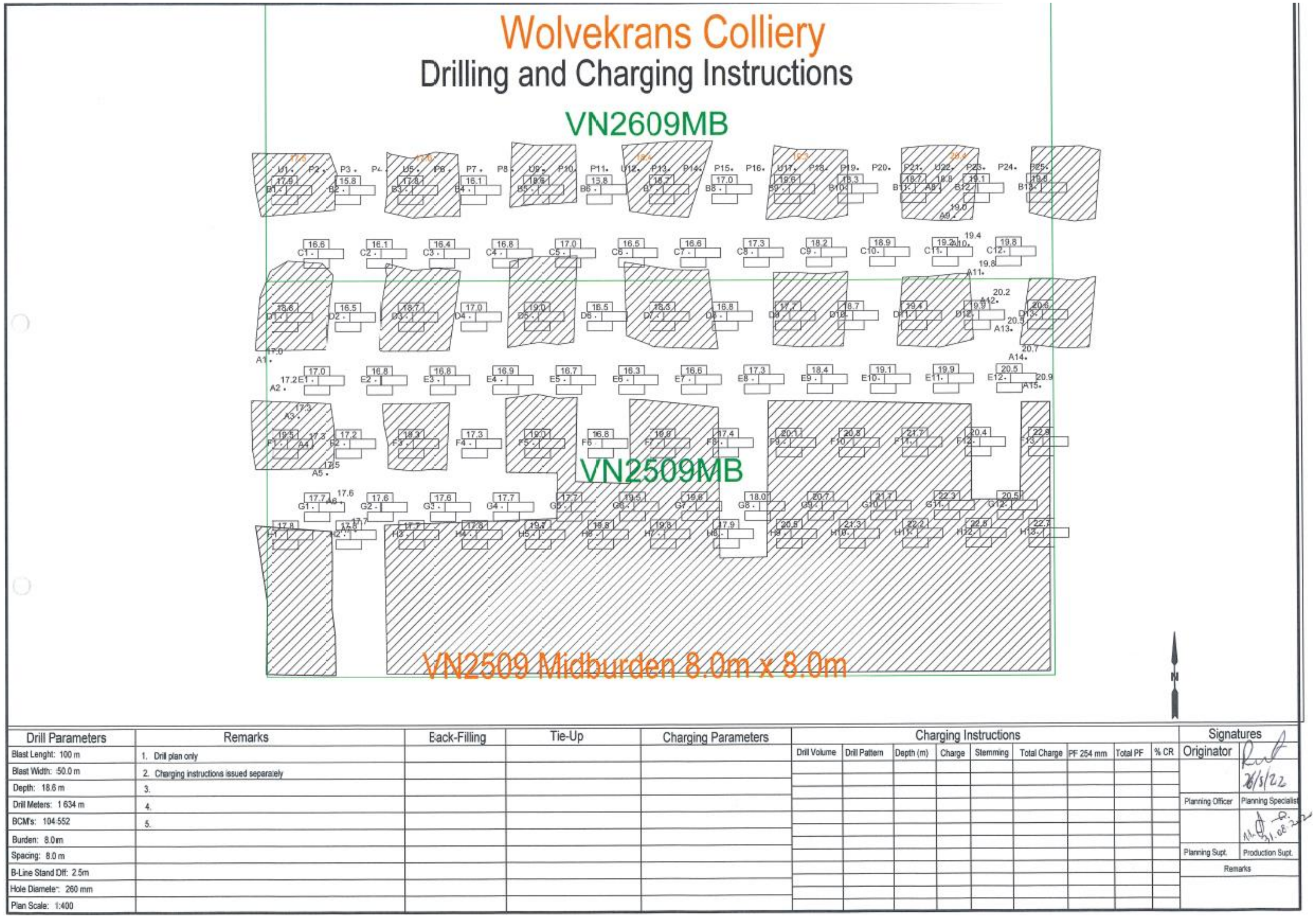

Figure C2: Drilling plan for VDDN midburden

Table C2a: VDDN Midburden Charging Instructions

VN2509_2S **HOT HOLE AREA**

Hole Diameter : 260mm
Charge mass per meter : 64.6 kg/m
Charging Parameters: Hef 20?: Cup Density @ 1.05/cm³
Production Hole:Top Booster 300g 4.0m from collar
Timing: 75ms, Full Chevron
Average Depth: 16.7 Powder Factor: 0.77

Block Number	Hole Number	Temp. before cooling	Temp. after cooling	Hole Depth	Stemming	Kg's per	Blasters Feedback: Actual Depth	Blasters Feedback: Actual KG's Per Hole	Blasters Feedback: Notes
VN2609	B1			15.6	4.0	746.1			
VN2609	B2			15.4	4.0	730.9			
VN2609	B3			16.8	4.0	824.4			
VN2609	B4			14.7	4.0	684.7			
VN2609	B5			16.6	4.0	811.6			
VN2609	B6			15.1	4.0	714.4			
VN2609	B7			17.6	4.0	874.4			
VN2609	B8			16.1	4.0	779.2			
VN2609	B9			18.5	4.0	945.9			
VN2609	B10			16.6	4.0	806.3			
VN2609	B11			18.6	4.0	939.4			
VN2609	B12			16.9	4.0	826.3			
VN2609	B13			15.7	4.0	766.7			
VN2609	C1			15.6	4.0	744.3			
VN2609	C2			15.2	4.0	722.0			
VN2609	C3			15.0	4.0	707.0			
VN2609	C4			14.9	4.0	696.9			
VN2609	C5			14.8	4.0	694.7			
VN2609	C6			15.2	4.0	719.1			
VN2609	C7			15.6	4.0	742.9			
VN2609	C8			16.2	4.0	783.8			
VN2609	C9			16.7	4.0	812.4			
VN2609	C10			16.8	4.0	823.3			
VN2609	C11			16.9	4.0	829.6			
VN2609	C12			17.3	4.0	865.3			
VN2509	D1			17.5	4.0	865.8			
VN2509	D2			15.4	4.0	734.4			
VN2509	D3			17.2	4.0	850.2			
VN2509	D4			15.3	4.0	723.6			
VN2509	D5			17.1	4.0	840.0			
VN2509	D6			15.2	4.0	721.0			
VN2509	D7			17.6	4.0	873.3			
VN2509	D8			16.0	4.0	768.0			
VN2509	D9			18.7	4.0	941.3			
VN2509	D10			17.0	4.0	832.6			
VN2509	D11			19.0	4.0	963.3			
VN2509	D12			17.3	4.0	867.5			
VN2509	D13			15.7	4.0	766.4			
VN2509	E1			15.5	4.0	741.5			
VN2509	E2			15.5	4.0	741.3			
VN2509	E3			15.5	4.0	739.4			
VN2509	E4			15.5	4.0	735.2			
VN2509	E5			15.4	4.0	730.1			
VN2509	E6			15.7	4.0	752.1			
VN2509	E7			15.9	4.0	766.4			
VN2509	E8			16.4	4.0	798.6			
VN2509	E9			17.1	4.0	841.9			
VN2509	E10			17.3	4.0	852.5			
VN2509	E11			17.3	4.0	869.7			
VN2509	E12			17.7	4.0	894.8			
VN2509	F1			17.6	4.0	889.8			
VN2509	F2			15.5	4.0	741.1			
VN2509	F3			17.9	4.0	889.3			
VN2509	F4			15.7	4.0	752.0			
VN2509	F5			17.7	4.0	877.4			
VN2509	F6			15.8	4.0	755.4			
VN2509	F7			18.2	4.0	910.3			

Table C2b: VDDN Midburden Charging Instructions

VN2509	F8			16.5	4.0	804.2			
VN2509	F9			19.2	4.0	992.9			
VN2509	F10			17.6	4.0	888.5			
VN2509	F11			19.5	4.0	1020.0			
VN2509	F12			17.7	4.0	895.1			
VN2509	F13			16.1	4.0	797.6			
VN2509	G1			15.9	4.0	764.1			
VN2509	G2			15.9	4.0	767.0			
VN2509	G3			16.1	4.0	777.4			
VN2509	G4			16.3	4.0	786.6			
VN2509	G5			16.5	4.0	800.3			
VN2509	G6			16.9	4.0	842.1			
VN2509	G7			17.1	4.0	856.0			
VN2509	G8			17.3	4.0	856.6			
VN2509	G9			17.7	4.0	895.7			
VN2509	G10			18.0	4.0	922.6			
VN2509	G11			17.9	4.0	910.8			
VN2509	G12			18.1	4.0	922.4			
VN2509	H1			15.9	4.0	766.0			
VN2509	H2			15.7	4.0	752.6			
VN2509	H3			16.0	4.0	773.3			
VN2509	H4			16.2	4.0	783.7			
VN2509	H5			16.3	4.0	804.0			
VN2509	H6			16.7	4.0	830.0			
VN2509	H7			17.0	4.0	849.3			
VN2509	H8			17.2	4.0	847.1			
VN2509	H9			17.3	4.0	871.4			
VN2509	H10			17.8	4.0	906.4			
VN2509	H11			17.6	4.0	892.4			
VN2509	H12			17.8	4.0	909.7			
VN2509	H13			19.7	4.0	1034.8			
VN2510	G1			17.8	4.0	909.5			
VN2510	H1			17.9	4.0	916.1			

Planned **Actuals**

Pattern	8 x 8	64				Pattern	8 X 8	64	
Meters	1504			Total kg	73964	Meters	0.0	0	Total kg
AVG Depth	16.7			BCM	96259	AVG Depth	#DIV/0!	0	BCM
Total holes	90.0			P/F	0.77	Total holes	0.0	#DIV/0!	P/F

Planning Specialist ______________________

Planning Supt. ______________________

Production Supt. 2020-08-31. ______________________

Note: Area tested negative for reactive ground
Cut-off holes must not be charged
Do not auger wet holes
Use booster weights on all wet holes

APPENDIX D: Job Profile and Qualifications for a Surface Blaster

JOB DETAILS	
Job Title	Miner Surface
Job Code	MIN_001
Job Grade	C2
Function	Production
Location	Operations
1 Up Manager	Supervisor Production
Purpose	To supervise a team of people and ensure safe and cost-effective production at an operational level
Key Relationships	Supervisor Production Operators Attendants Clerk Data Capture
Direct Reports	None

CORE ACCOUNTABILITIES	
Job Outputs	**Sub Outputs**
Execution	– Safely execute blasts – Blast designs to ensure safe, cost effective and desirable material fragmentation is achieved – Participation in the drill and blast planning process and ensuring compliance to plan and designs – Ensure safety of people within the blasting block and the affected areas – Supervision of the personnel working with or assisting with explosives – Manage blasting accessories on the blasting block – Compiling blasting reports – Examine the work area before entry – Reading of the drilling plans – Do construction before drilling the area – Identifying the high wall and low wall hazards – Staking of the drill pattern – Loading and offloading of the drills from a low bed – Changing the drill bits and rods, crossover sub – Changing the stabilizer – Recording hourly drilled meter – Daily maintaining of the MBA board – Reporting all hazards to the Pit Superintendent – Updating and reviewing of drilling SOP's – Time and attendance log sheets – Logging of work instructions – Maintaining high health and safety standards
Governance & Compliance	– Ensure compliance to mining parameters

GENERIC ACCOUNTABILITIES		
Job Outputs	**Sub Outputs**	
Health & Safety	–	Ensure compliance to all legislative regulations to maintain a risk free, zero harm environment
	–	Ensure adherence to policies and procedures
	–	Reporting of incidents, accidents and property damage
People Management	–	Coaching and mentoring of staff
	–	Report incidents of disciplinary / grievances issues
	–	Maintaining discipline amongst the people
	–	Supervise the achievement of budgeted volumes / tonnage per team

MINIMUM REQUIREMENTS OF POSITION			
Category	**Description**		
Qualification	–	Grade 12 with Mathematics	Required
	–	Blasting Assistant Level 2 / Competent A Level 2 / Magazine Master	
	–	Rock Breaking / Blasting Certificate	
Experience	–	Minimum of 2 years general working experience in Mining operations	Required
	–	Experience as a Surface Miner	
	–	Supervisory skills	Advantageous
Technical Skills	–	Strata Control Certificate	Required
Technical Knowledge	–	Sound knowledge and understanding of applicable legislation	Required
	–	Mining Machinery	
Behavioural Competencies	–	As per TTS define	Core

APPENDIX E: Job Profile and Qualifications for a Blasting Assistant

JOB DETAILS	
Job Title	Attendant Blasting
Job Code	ATT_002
Job Grade	B2
Function	Production
Location	Operations
1 Up Manager	Supervisor Production
Purpose	To provide general assistance to ensure safe and cost-effective mining
Key Relationships	Supervisor Production Clerk Data Capture Operator – Various Miner
Direct Reports	None

CORE ACCOUNTABILITIES	
Job Outputs	**Sub Outputs**
Execution	- Barricade blasting area as instructed by the Miner, before charging up - Measure pre-drilled holes - Place accessories into the holes - Lower explosive bags into designated holes at pre-determined depths - Cut ropes to planned length - Prime pre-split holes - Tie all lines from different holes together - Clear prepared blasting area of all equipment and people with specified radius - Close blasting area to specified radius as instructed by the Miner - Ensure compliance to mining parameters

GENERIC ACCOUNTABILITIES	
Job Outputs	**Sub Outputs**
Health & Safety	- Comply with and ensure adherence to safety regulations - Reporting of incidents, accidents and property damage

MINIMUM REQUIREMENTS OF POSITION		
Category	**Description**	
Qualification	- Grade 12 with Mathematics - Grade 10 for Internal Candidates	Required
Experience	- Coal mining industry	Advantageous

Technical Skills	- Driver's License - Operation of all relevant machines in section	Advantageous
Technical Knowledge	- Safety regulations and standards	Required
Behavioural Competencies	- As per TTS define	Core

APPENDIX F: Strip Orientation Results

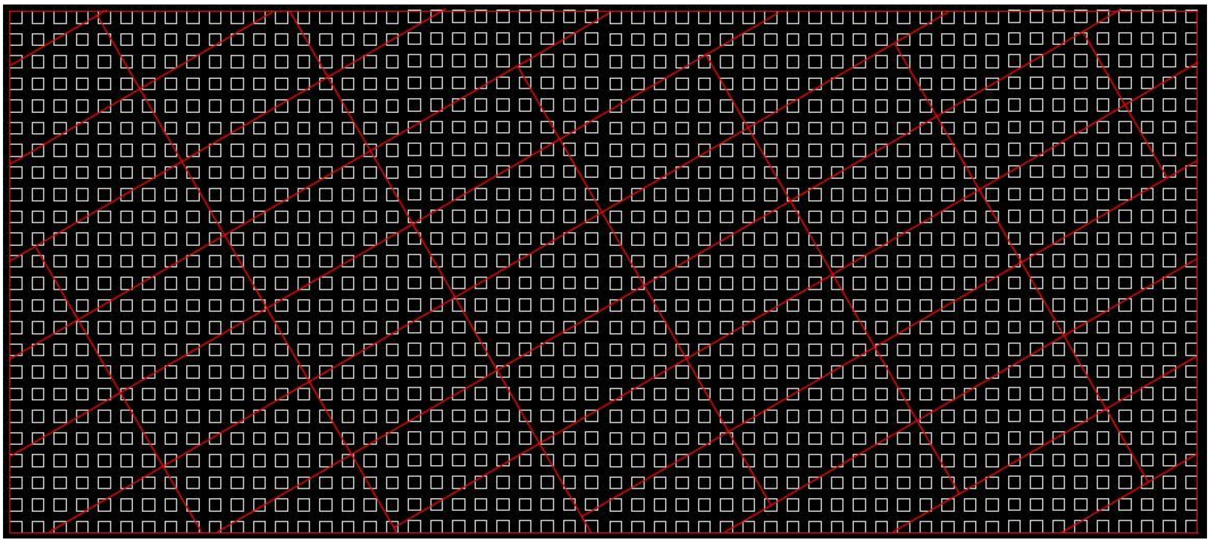

Figure F1: Mining strips orientated 30 degrees in relation to the underground roadways

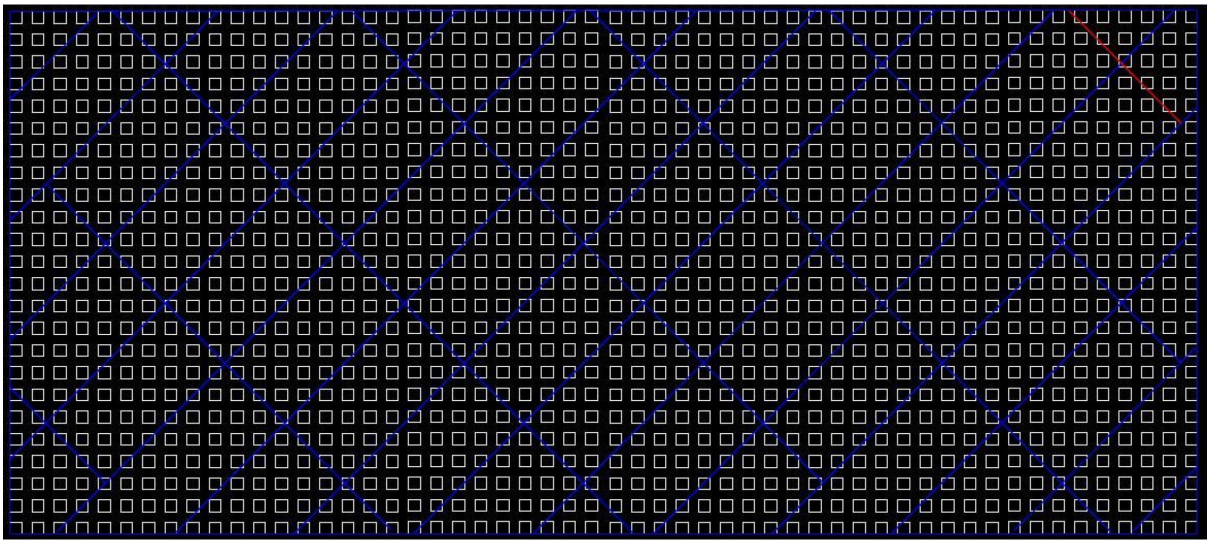

Figure F2: Mining strips orientated 45 degrees in relation to the underground roadways

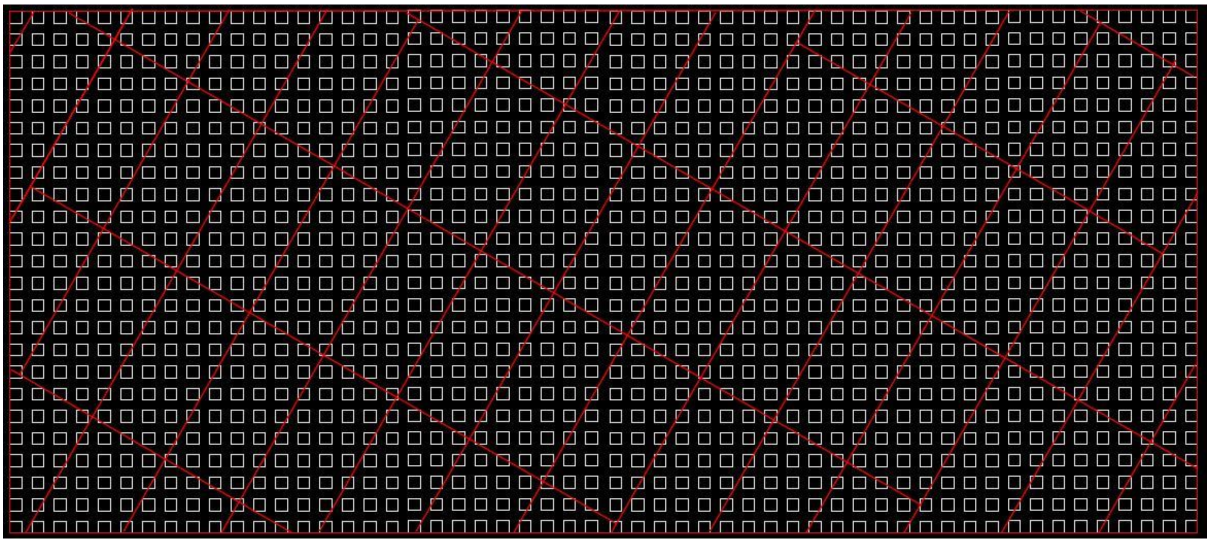

Figure F3: Mining strips orientated 60 degrees in relation to the underground roadways

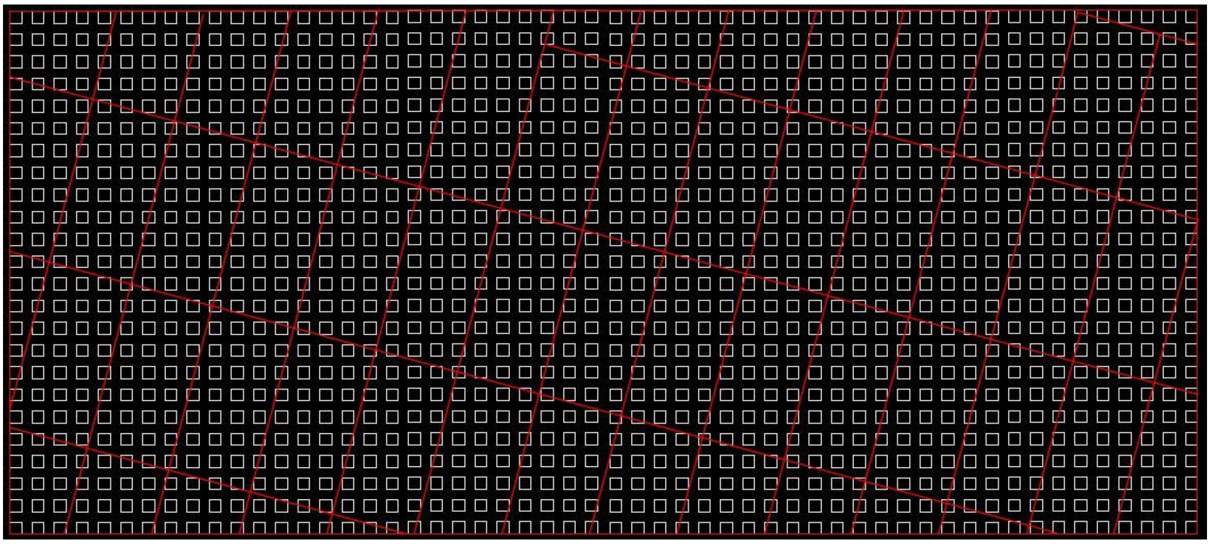

Figure F4: Mining strips orientated 75 degrees in relation to the underground roadways

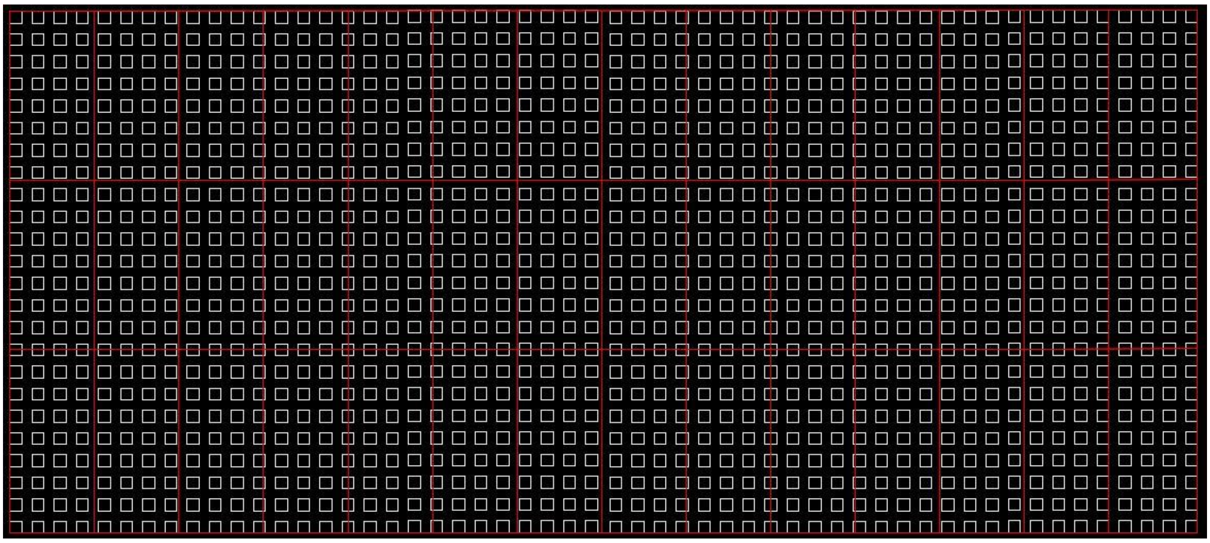

Figure F5: Mining strips orientated 90 degrees in relation to the underground roadways

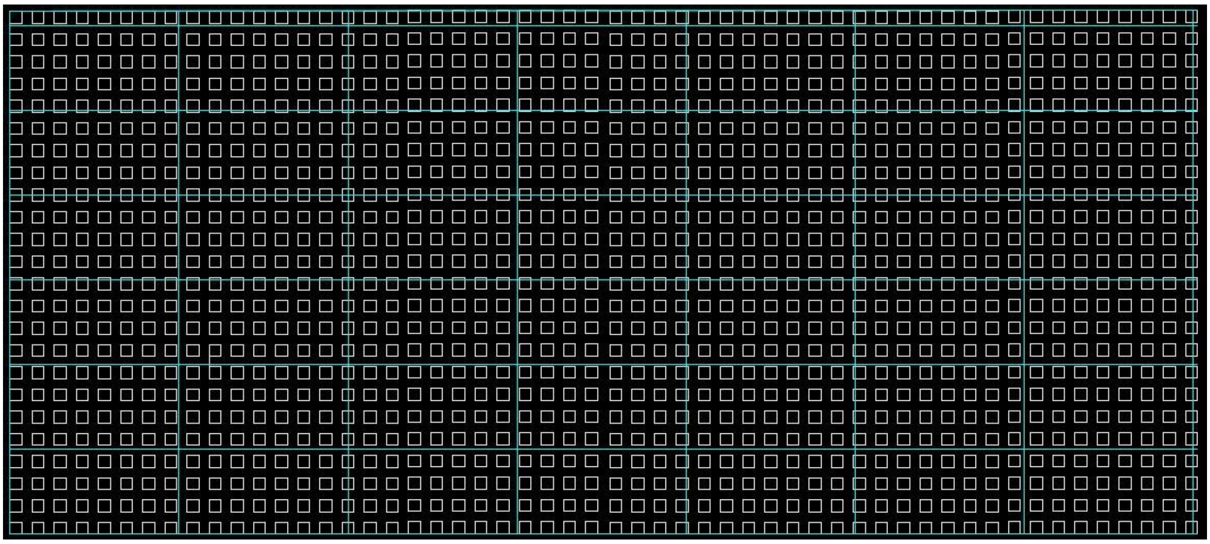

Figure F6: Mining strips orientated parallel the underground roadways

Table F1: Strip Orientation Results

Orientation	Pillar Coal percentage	Bord Percentage	Total Percentage
0 degrees	29%	71%	100%
30 degrees	29%	71%	100%
45 degrees	29%	71%	100%
60 degrees	29%	71%	100%
75 degrees	29%	71%	100%
90 degrees	30%	70%	100%

APPENDIX G: Results of the coal samples taken on imploded blocks, half imploded blocks and non-imploded blocks

Table G1: Washed results for samples taken on fully imploded, half imploded and non-imploded blocks

Implosion type	Block ID	RD	Washed CV	Washed Ash
Fully imploded blocks	VN_2008_01	1.8	22.5	25.7
	VN_2009_01	1.7	24.5	22
	VN_2009_02	1.6	28.2	14.2
	VN_2010_01	1.7	24.1	24.1
	VN_2010_02	3	10.2	58.5
	VN_2011_01	1.7	25.6	19.6
	VN_2011_02	1.7	24.7	24.7
	VN_2103_01	1.8	22	27.2
	VN_2103_02	1.7	25.1	20
Half Imploded blocks	VN_2104_01	3	12.4	52.3
	VN_2105_01	1.5	28.6	12.3
	VN_2105_02	3	12.7	51.1
	VN_2106_01	1.7	28.3	26.5
	VN_2109_01	1.7	22	25.2
	VN_2110_01	1.7	23.1	19
	VN_2110_02	1.7	25.4	19.4
	VN_2111_01	1.7	26.7	17.4
	VN_2111_02	1.6	26.7	15.7
	VN_2203_01	1.7	29.7	10.5
	VN_2203_02	1.5	26.7	17.2
Non imploded blocks	VN_2205_01	1.7	23.2	23.9
	VN_2206_01	1.5	30.2	9.2
	VN_2207_01	1.7	25	19.4
	VN_2208_01	1.7	25.5	19.8
	VN_2209_01	1.6	27.7	15.3
	VN_2210_01	1.7	24.7	21.2

APPENDIX H: Bonus Scheme Amendment Memorandum

Memorandum

Date	04 September 2018
To	Wolvekrans Employees
Copy	Wolvekrans HODs
From	Andre Roux
Our Ref	FY19 Production Bonus Scheme

South32 Coal Processing
Farm Driefontein
Broodsnyersplaas Road
eMalahleni, 1035
P O Box 1672
Middelburg, 1050
South Africa
T +27 13 689 4449
south32.net

Following Export bonus memo dated 01 July 2017, the following amendments have been made to the current production bonus scheme:

Measure	Current	New
LTI modifier	33% per 3.1 appointee per incident	33% per Department per incident
Crushed target	Separate targets for BMK and SKS	One target for WVK (sum of BMK and SKS)

Herewith summary of WVK Bonus Scheme:

All monthly targets for production bonus measures will be as per the 20th forecast. BCM's targets will be set for each production area except Total Saleable and Crushed which are common measures across all areas. Quality measure is only applicable to processing.

The production areas and bonus measures are defined as follows:

Areas	KPI's			
	BCM's 25% Weighting	Crushed 25% Weighting	Ash Compliance 25% Weighting	Total Saleable 50% Weighting
BMK East and West	X	X		X
SKS, VDDN and Pit 4	X	X		X
South Export Plant		X	X	X
Maintenance (Mining)	X	X		X
Services	X	X		X

No bonus payment if less than **91%** of Total Saleable is achieved. The amount will be prorated based on achievement. A sliding scale of R200 for every 1% will be applied from 91% and a maximum of R2000 at 100%.

Bonus for BCM's is per area (BMK or SKS) and is paid only when 100% of the targeted volumes are achieved. Services and maintenance (Mining) will receive 50% payout from BMK and SKS respectively.

The Crushed target is based on total crushed for Wolvekrans (Sum of BMK and SKS). All areas will have the same collective target and receive the same payout. Total crushed is paid only when 100% of the targeted tonnes are achieved.

Compliance to Ash is paid at 80% of the target or above and is only applicable to processing teams. The required ash range is 15.4% – 17.3% for 6000kcal product and 17.7% - 18.8% for 5700Kcal product.

Financial Measures

Primary Measures:

Measure	Target	Amount	Secondary Bonus Applicable
BCM's	100%	R 1000	No
Total Crushed	100%	R 1000	No
Quality Compliance	80%	R 1000	No
Total Saleable	91%	R 200 at 91% and R2000 at 100%	Yes
Primary bonus when all KPI's achieved		R 4000	

Secondary Measures

An additional Secondary bonus is payable based on straight line scale of R100 for every 2000 additional total saleable tonnes produced.

Team Modifiers

Lost Time Injuries

A 33% modifier will be applied per Department. Should an LTI involve a contractor, the same principle will apply.

Medical Treatment Case

A 15% modifier will be applied per Superintendent. Should an MTC involve a contractor, the same principle will apply.

Safety Incentive

A Safety incentive bonus will still be payable to all qualifying employees as prescribed in the FLD.

Yours sincerely

André Roux
Operations Manager WVK

Yours sincerely

20064441

NUM Representative

APPENDIX I: Criteria Results for Various Coal Pillar Methods

Table I- 1: Quantified results of the selected criteria used in the review of coal pillar method

Methods	Coal loss	Dilution	Impact on Ash Content	Impact on CV	Overall Plant Yield	level on blasting workforce	level on exposure workforce	Application on hot areas	Possibility of coal fines
Pillar Implosion - 0.5m waste left on the roof	0%	68%	59.37	10.31	28.28	5	5	1	25%
Pillar Implosion - 0.0m waste left on the roof	0%	64%	56.90	11.27	32.11	5	5	1	25%
Pillar Implosion - 0.5m of coal removed with waste	0%	58%	53.72	12.51	37.05	5	5	1	25%
Pillar Implosion - 1.0m of coal removed with waste	1%	51%	49.66	14.09	43.34	5	5	1	25%
Pillar Implosion - 1.5m of coal removed with waste	5%	42%	44.78	15.99	50.92	5	5	1	25%
Pillar Implosion - 2.0m of coal removed with waste	13%	33%	39.23	18.15	59.52	5	5	1	25%
Pillar Implosion - 2.5m of coal removed with waste	26%	23%	33.61	20.34	68.24	5	5	1	25%
Pillar Implosion - 3.0m of coal removed with waste	59%	19%	31.54	21.14	71.45	5	5	1	25%
Bord Collapse - No cleaning	0%	63%	56.76	11.33	20.33	1	1	5	2%
Bord Collapse Cleaning	5%	14%	28.43	22.35	64.28	1	1	5	2%

APPENDIX J: AHP Results

Table J1: Criteria rating based on employee's preference

Criteria	Coal loss	Dilution	Impact on Ash Content	Impact on CV	Overall Plant Yield	Required skill level on blasting workforce	Required skill level on exposure workforce	Application on hot areas	Possibility of coal fines	Sum
Coal loss	1.00	0.33	3.00	3.00	0.33	3.00	3.00	0.33	0.50	14.50
Dilution	3.00	1.00	3.00	3.00	0.33	4.00	4.00	0.33	3.00	21.67
Content	0.33	0.33	1.00	1.00	1.00	3.00	3.00	0.50	2.00	12.17
Impact on CV	0.33	0.33	1.00	1.00	1.00	3.00	3.00	0.50	2.00	12.17
Overall Plant Yield	3.00	3.00	1.00	1.00	1.00	4.00	4.00	2.00	3.00	22.00
level on blasting workforce	0.33	0.25	0.33	0.33	0.25	1.00	1.00	0.33	0.33	4.17
Required skill level on exposure workforce	0.33	0.25	0.33	0.33	0.25	1.00	1.00	0.33	0.33	4.17
Application on hot areas	3.00	3.00	2.00	2.00	0.50	3.00	3.00	1.00	3.00	20.50
Possibility of coal fines	2.00	0.33	0.50	0.50	0.33	3.00	3.00	0.33	1.00	11.00
Total	13.33	8.83	12.17	12.17	5.00	25.00	25.00	5.67	15.17	122.33

Table J2 : Calculation of priority vector to be used as weighting in VIKOR

Criteria	Coal loss	Dilution	Impact on Ash Content	Impact on CV	Overall Plant Yield	Required skill level on blasting workforce	Required skill level on exposure workforce	Application on hot areas	Possibility of coal fines	Priority Vector	Sum	Product
Coal loss	0.08	0.04	0.25	0.25	0.07	0.12	0.12	0.06	0.03	0.11	1.14	10.17
Dilution	0.23	0.11	0.25	0.25	0.07	0.16	0.16	0.06	0.20	0.16	1.73	10.58
Impact on Ash Content	0.03	0.04	0.08	0.08	0.20	0.12	0.12	0.09	0.13	0.10	0.95	9.60
Impact on CV	0.03	0.04	0.08	0.08	0.20	0.12	0.12	0.09	0.13	0.10	0.95	9.60
Overall Plant Yield	0.23	0.34	0.08	0.08	0.20	0.16	0.16	0.35	0.20	0.20	2.10	10.49
Required skill level on blasting workforce	0.03	0.03	0.03	0.03	0.05	0.04	0.04	0.06	0.02	0.04	0.35	9.89
Required skill level on exposure workforce	0.03	0.03	0.03	0.03	0.05	0.04	0.04	0.06	0.02	0.04	0.35	9.89
Application on hot areas	0.23	0.34	0.16	0.16	0.10	0.12	0.12	0.18	0.20	0.18	1.95	10.89
Possibility of coal fines	0.15	0.04	0.04	0.04	0.07	0.12	0.12	0.06	0.07	0.08	0.79	10.18
Total	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	10.30	10.14
			λ	10.14								
			n	9.00								
			CI	0.14								
			IR	1.45								
			CR	**0.099**								

APPENDIX K: Calculated Results using VIKOR

Table K1: Determination of best and worst values of the selected criteria

Methods	Coal loss	Dilution	Impact on Ash Content	Impact on CV	Overall Plant Yield	Required skill level on blasting workforce	Required skill level on exposure workforce	Application on hot areas	Possibility of coal fines
Pillar Implosion - 0.5m waste left on the roof	0%	68%	59.37	10.31	28.28	5	5	1	25%
Pillar Implosion - 0.0m waste left on the roof	0%	64%	56.90	11.27	32.11	5	5	1	25%
Pillar Implosion - 0.5m of coal removed with waste	0%	58%	53.72	12.51	37.05	5	5	1	25%
Pillar Implosion - 1.0m of coal removed with waste	1%	51%	49.66	14.09	43.34	5	5	1	25%
Pillar Implosion - 1.5m of coal removed with waste	5%	42%	44.78	15.99	50.92	5	5	1	25%
Pillar Implosion - 2.0m of coal removed with waste	13%	33%	39.23	18.15	59.52	5	5	1	25%
Pillar Implosion - 2.5m of coal removed with waste	26%	23%	33.61	20.34	68.24	5	5	1	25%
Pillar Implosion - 3.0m of coal removed with waste	59%	19%	31.54	21.14	71.45	5	5	1	25%
Bord Collapse - No cleaning	0%	63%	56.76	11.33	20.33	1	1	5	2%
Bord Collapse Cleaning	5%	14%	28.43	22.35	64.28	1	1	5	2%
Weights	0.11	0.16	0.10	0.10	0.20	0.04	0.04	0.18	0.08
Best f+	**0%**	**14%**	**28.43**	**22.35**	**71.45**	**1.00**	**1.00**	**5**	**2%**
Worst f-	**59%**	**68%**	**59.37**	**10.31**	**20.33**	**5**	**5**	**1**	**25%**

Table K2: S_j and R_j Calculations

Methods	Coal loss	Dilution	Impact on Ash Content	Impact on CV	Overall Plant Yield	Required skill level on blasting workforce	Required skill level on exposure workforce	Application on hot areas	Possibility of coal fines	Sj	Rj
Pillar Implosion - 0.5m waste left on the roof	-	0.16	0.10	0.10	0.17	0.04	0.04	0.18	0.08	**0.86**	**0.18**
Pillar Implosion - 0.0m waste left on the roof	0.00	0.15	0.09	0.09	0.15	0.04	0.04	0.18	0.08	**0.81**	**0.18**
Pillar Implosion - 0.5m of coal removed with waste	0.00	0.13	0.08	0.08	0.13	0.04	0.04	0.18	0.08	**0.76**	**0.18**
Pillar Implosion - 1.0m of coal removed with waste	0.00	0.11	0.07	0.07	0.11	0.04	0.04	0.18	0.08	**0.69**	**0.18**
Pillar Implosion - 1.5m of coal removed with waste	0.01	0.09	0.05	0.05	0.08	0.04	0.04	0.18	0.08	**0.61**	**0.18**
Pillar Implosion - 2.0m of coal removed with waste	0.02	0.06	0.03	0.03	0.05	0.04	0.04	0.18	0.08	**0.52**	**0.18**
Pillar Implosion - 2.5m of coal removed with waste	0.05	0.03	0.02	0.02	0.01	0.04	0.04	0.18	0.08	**0.45**	**0.18**
Pillar Implosion - 3.0m of coal removed with waste	0.11	0.02	0.01	0.01	-	0.04	0.04	0.18	0.08	**0.47**	**0.18**
Bord Collapse - No cleaning	-	0.15	0.09	0.09	0.20	-	-	-	-	**0.53**	**0.20**
Bord Collapse Cleaning	0.01	-	-	-	0.03	-	-	-	-	**0.04**	**0.03**
										0.04	**0.03**
										0.86	**0.20**

Table K3 : Q_j Calculation and Ranking of the Alternatives

Methods	Calculate Qj	Ranking
Pillar Implosion - 0.5m waste left on the roof	0.94	10
Pillar Implosion - 0.0m waste left on the roof	0.91	9
Pillar Implosion - 0.5m of coal removed with waste	0.88	8
Pillar Implosion - 1.0m of coal removed with waste	0.83	7
Pillar Implosion - 1.5m of coal removed with waste	0.79	5
Pillar Implosion - 2.0m of coal removed with waste	0.73	4
Pillar Implosion - 2.5m of coal removed with waste	0.69	2
Pillar Implosion - 3.0m of coal removed with waste	0.70	3
Bord Collapse - No cleaning	0.80	6
Bord Collapse Cleaning	-	1

Acceptance of Ranking						
C1	Acceptance of acceptance					
Q(a2)-Q(a1)>=DQ		DQ=1/(j-1) where j = no of alternatives			DQ	0.11
					Q(a2)-Q(a1)	0.69
C2	Acceptable stability in decision making					
	Alternatives must also be best ranked by either R value or S value					

APPENDIX L: Margin Ranking calculation for a mine with a strip ratio of 5.5 BCM/ton

Table L1: Margin for a mine with a strip ratio of 5.5 BCM/ton

Description	**Unit**	**Production physicals**		
Strip Ratio	BCM/t	5.5		
Waste removed	BCM	5.5		
ROM	t	1		
Description	**Unit**	**Unit Cost**	**Cost - R**	
DL Costs	R/BCM	10.20	56.10	
Drill & Blast	R/BCM	15.30	84.13	
ROM	R/t	29.64	29.64	
Plant & Crushing	R/t	73.12	73.12	
Total			242.98	
Description		**Price**	**Product**	**Revenue**
Primary Product		1 410	0.34	479.50
Secondary Product		816	0.38	310.25
Total Revenue				789.76
Margin				**546.77**

APPENDIX M: COAL LOSS AND DILUTION AMOUNTS

Table M1: Coal loss and dilution on different mining heights to the coal pillar profile.

Description	No Contamination	"+0.5m"	"0m"	"-0.5m"	"-1m"	"-1.5m"	"-2m"	"-2.5m"	"-3m"
Waste Tons	0	37 868	31 166	24 575	18 234	12 446	7 533	3 863	1 707
Coal Tons	17 871	17 871	17 844	17 803	17 615	17 001	15 634	13 207	7 282
Coal Loss (%)		**0%**	**0.15%**	**0.38%**	**1.43%**	**4.87%**	**12.52%**	**26.10%**	**59.25%**
Dilution		**68%**	**64%**	**58%**	**51%**	**42%**	**33%**	**23%**	**19%**
Ash	**20.75**	**59.37**	**56.90**	**53.72**	**49.66**	**44.78**	**39.23**	**33.61**	**31.54**
CV	**25.34**	**10.31**	**11.27**	**12.51**	**14.09**	**15.99**	**18.15**	**20.34**	**21.14**
RD	**1.51**	**2.05**	**2.02**	**1.97**	**1.91**	**1.85**	**1.77**	**1.69**	**1.66**
Overall Yield	**88.20**	**28.28**	**32.11**	**37.05**	**43.34**	**50.92**	**59.52**	**68.24**	**71.45**