Rib Cutting Resue Stoping, improvement on stoping rates and reduction in waste dilution compared with other known resue stoping methods on a Free State gold mine

Alwyn Scholtz

(Student number: 751434)

School of Mining

University of the Witwatersrand

Johannesburg, South Africa.

Supervisor: C. Birch

Abstract:

Mining of the Basal Reef at Jeanette Mine, is typically complicated due to an overlaying Khaki Shale (shale) that has unfavourable rock engineering properties. Shale has always been either left underground or mined as part of the orebody. The first approach can only be applied in areas where the quartzite beam (directly above the Basal Reef and below the shale) is of sufficient thickness to support the shale in the hanging wall. This method is known as undercutting. Alternatively, open stoping can be applied in areas where the shale and the Basal Reef is extracted concurrently and sent to the mill as diluted ore.

Alternatively, a resue stoping method can be considered in areas where undercutting cannot be done, due to a thin quartzite middling. Resue stoping involves stowing or packing of the shale into the mined-out area and is not included as part of the hoisted rock. In the past, resue stoping was done by hand packing, which is unsuitable for a modern mine. As such, two mechanised resue stoping methods can be considered, namely; Longhole Resue Stoping and Rib Cutting Resue Stoping.

Rib Cutting Resue Stoping utilises a continuous miner ("CM") to remove the shale in a first pass, extract the reef during a second pass and backfilling the mined-out rib with shale. The use of a CM will significantly improve the extraction/mining rate, sidewall stability, backfill placement, dilution and overall safety.

Longhole Resue Stoping utilises strike drives from where longholes are drilled into the shale and the reef in an up-dip direction moving on retreat. The shale is blasted with sufficient force into the mined-out area behind it, compacting it. The reef will be loaded by Load Haul Dumper ("LHD") or dozer.

It was determined that Rib Cutting Resue Stoping is more effective than Longhole Resue Stoping due to a higher extraction rate, lower dilution, reef loss reduction and improved shale sidewall stability. The operating angles and equipment height limits the application to only 51% of the available reef at Jeanette mine with favourable dip and thickness. Longhole Stoping can navigate hard rock, shale and increased dip angles; it can be applied to 91% of the available ore deposit.

Longhole Resue Stoping and Rib Cutting Resue Stoping should both be considered as suitable stoping methods for Jeanette.

Declaration:

I declare that this report is my own, unaided work. I have read the University Policy on Plagiarism and hereby confirm that no Plagiarism exists in this report. I also confirm that there is no copying nor is there any copyright infringement. I willingly submit to any investigation in this regard by the School of Mining Engineering and I undertake to abide by the decision of any such investigation.

____

Signature (August 2018)

TABLE OF CONTENTS

FIGURES

TABLES

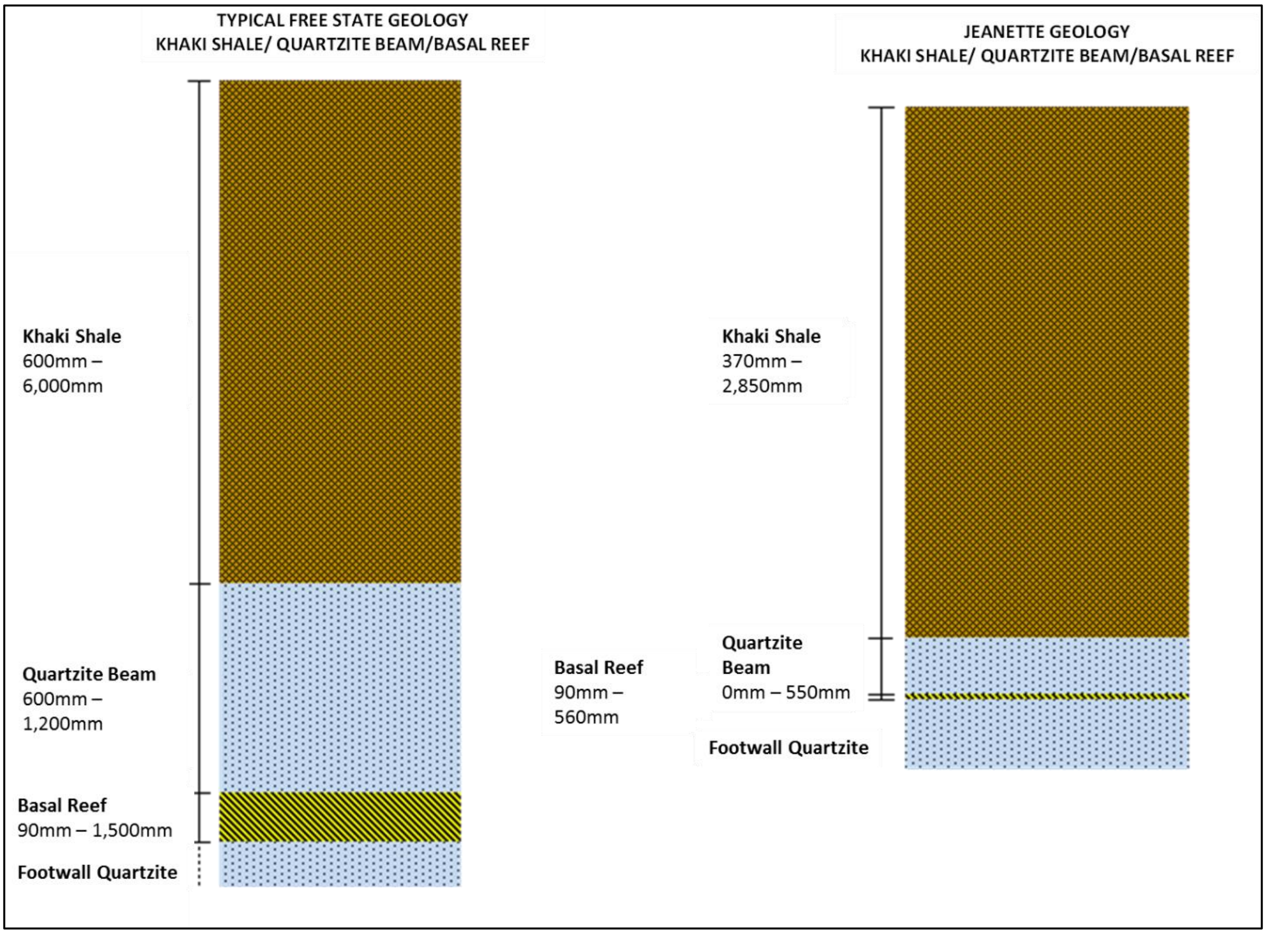
1 INTRODUCTION

Following the completion of a Concept Study ("CS") on The Jeanette Project (or "the Project") by the Khaki Shale Committee in 2012, a Pre-Feasibility Study ("PFS") was conducted by Minxcon Projects to investigate various mining methods and options for the Project. This study commenced in March 2013, concluded in August 2014 and covered the mine design, optimal mining method and the economic viability of extracting the Basal Reef deposit. Of all the different mining methods investigated, a resue stoping method was selected namely, Double Blast Longhole Resue Stoping ("Longhole Resue Stoping"). Although this was regarded as a suitable mechanised resue stoping method, other possible methods were not considered, namely Rib Cutting Resue Stoping.

A shale layer known as Khaki Shale overlays the Basal Reef and varies in thickness between 0.7 m and 2.7 m with an average of 1.7 m. The shale is geotechnically unstable and exhibits typical sedimentological lamina and distinct parting planes due to its internal mineralogical clay structure being predominantly pyrophyllite. Once the shale is exposed, it lends itself to self-mining and, typically, other mines in the area either undercut the shale or extract it along with the reef.

In this particular Project area, the shale layer is too thick to be economically extracted along with the Basal Reef, as determined as part of a concept study. Undercutting is also not possible due to the thinner overlaying quartzite providing insufficient geotechnical support during mining activities. Figure 1 illustrates the lithological relationship between typical layer thickness in the Free State Goldfields and Jeanette.

Figure 1: Lithology comparison between typical Free State gold mines and Jeanette Mine



A resue stoping method will be required when mining the Basal Reef at Jeanette; the shale will be separated from the reef during mining. It has also been determined that the shale should be packed, placed and/or stored in mined-out areas instead of being hoisted, to improve the mine's economic viability.

Effective dilution control for Jeanette is essential to the success of the Project. Effective separation between the overlaying shale and the Basal Reef will result in a high shaft head grade. If most of the mined-out shale can be repurposed as support, the tonnes that must be hoisted can be reduced and the cost per recovered ounce can be lowered. Rib Cutting Resue Stoping uses coal extraction technology to remove the shale, hence the shale can be removed at a higher rate, resulting in increased production and decreased shale dilution. The remaining reef will be a high-grade, thin (30-50 cm thick) deposit that can be extracted by footwall ripping or footwall drilling and blasting.

1.1 PROJECT SCOPE

Rib Cutting Resue Stoping is a stoping method, the required systems of which will be described from the on-reef working face to the decline ore passes. The areas in which the method can be applied at the operation will be investigated in terms of operating angles, as well as minimum and maximum shale thicknesses. The determining factor for these operating ranges will be the selection of available equipment that is proven and operational in the industry.

The shale's susceptibility to cutting will be investigated and cutting rates will be estimated. The shale has a lower Uniaxial Compressive Strength ("UCS") than the surrounding quartzite material and it has already been established that the material can be cut by roadheaders or continuous miners (CMs). The use of shale material as backfill was investigated along with the mining system that will be required to transport, mix with cement and placing thereof in worked-out areas. The benefits of this system will be compared with other known resue stoping methods. The comparison will assess:

- Operator and equipment safety
- Stoping rates and production volumes
- Dilution factors, reef-to-waste loss, shale-to-reef dilution, footwall over break, etc.
- Rock engineering conditions
- Ventilation requirements

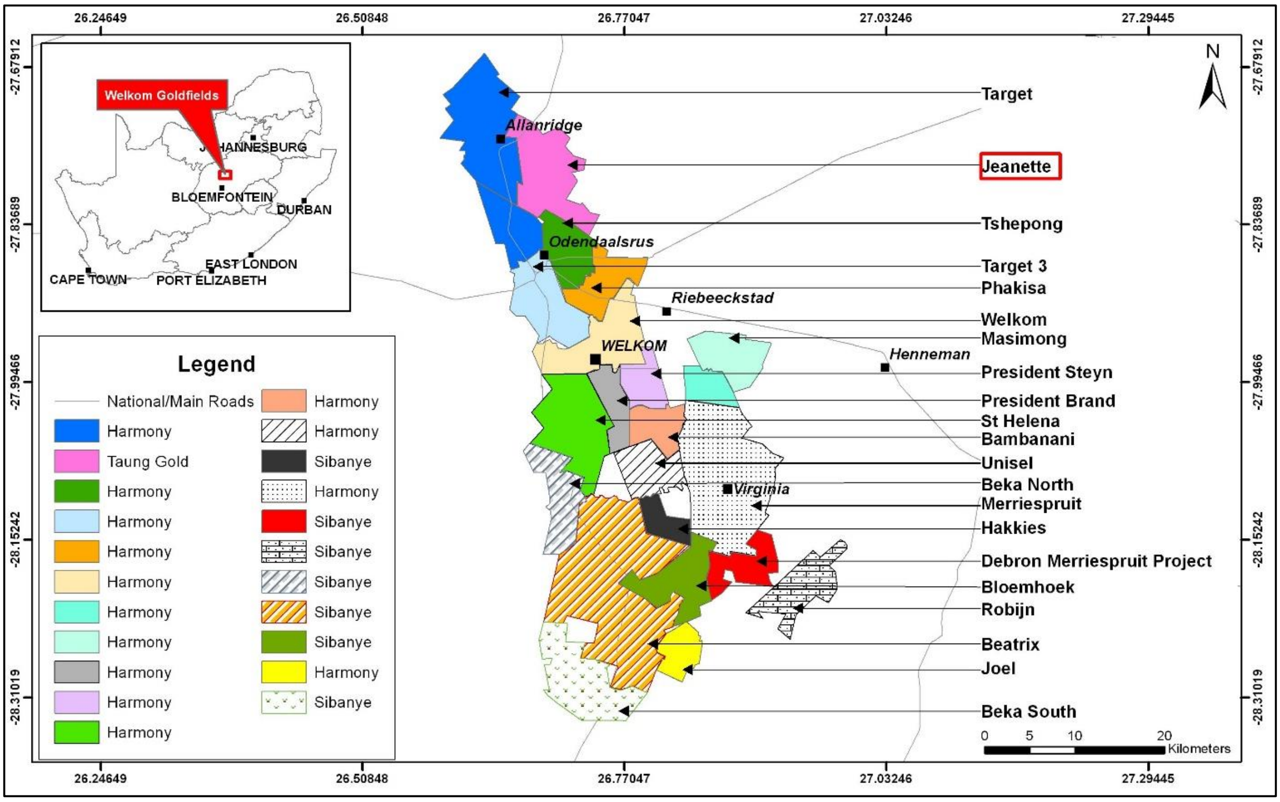
The advantages of this method over others will be presented, together with future work that will be required to implement it as a stoping method at this operation. Rock engineering aspects of the method will be addressed, but no detailed numerical modelling will be done at this stage. The study will describe a stoping method that can be applied at a Greenfield project, hence the stoping method will not be constrained by shaft capacity or services, such as water, air and electricity supply.

1.2 Location of Jeanette Mine

Jeanette Mine is situated approximately 20 km northwest of the town of Welkom in the Free State Province of South Africa and occupies an area of 5,700.84 ha in the northern Free State Goldfields. The Project Area is accessible via an un-tarred secondary road which intersects the R30 national road between the towns of Odendaalsrus and Bothaville, located west of the Project Area. The area has well-developed infrastructure, including a rail link, the R30 national road and Eskom power lines (part of a major grid), which are situated to the immediate east (central to the Project Area).

The western boundary of the Project Area is shared with the defunct Lorraine Mine, while the southern boundary is shared with Tshepong Mine, where exploitation of the Basal Reef occurs. The conventional stoping method has been tailored at this mine to successfully undercut and support the shale in the hanging wall. Figure 2 depicts Jeanette's location relative to other projects and mines.

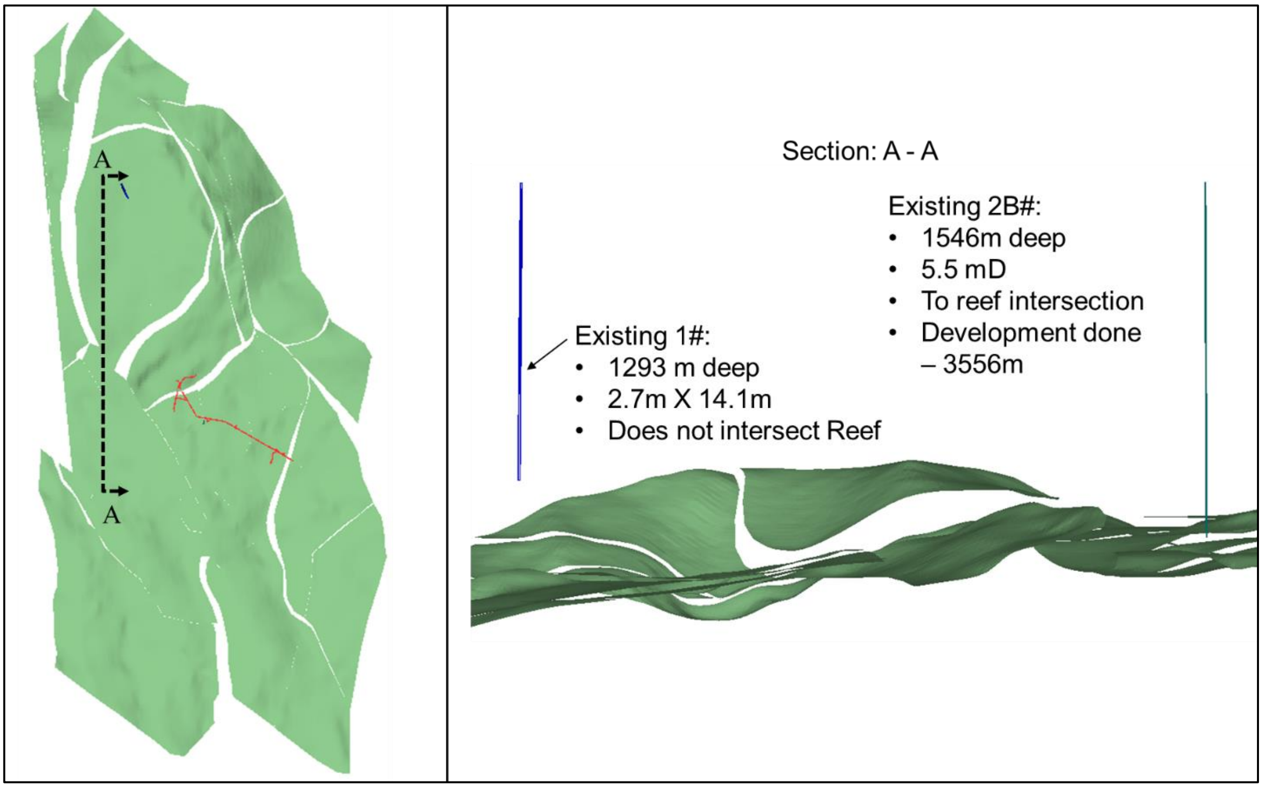
Figure 2: Jeanette Mine location in relation to other mining operations



1.3 EXISTING INFRASTRUCTURE

Service infrastructure in the area is well-established and includes roads, railway lines, water and power supplies. Existing underground infrastructure was developed in the early 1950s and consists of two vertical shafts and limited underground horizontal access development. (TaungGold, n.d.). The present mining infrastructure at the Project consists of the North 1 Shaft (rectangular at 2.7 m x 14.1 m) with a depth of 1,293 m (which was not sunk to Basal Reef elevation) and the South 2B Shaft, with a diameter of 5.5 m and a depth of 1,546 m (Figure 3).

Figure 3: Present mining infrastructure



1.4 MINING ZONE

A mining zone will function as a building block for the operation and will be used to compare the different mining methods. Each mining zone will have the same stoping and development layouts, which will function as a dedicated ventilation district, and will have the same infrastructure to provide the working areas with services like electricity and water. Each mining zone will have the same personnel, material and rock flow system and will be connected to the declines by access drives developed along strike. Declines will connect the different mining zones to the vertical shaft. This study will compare the differences between the stoping methods per mining zone.

2 LITERATURE REVIEW

2.1 INTRODUCTION

An investigation is required to document the methods that was applied in the past to deal with similar mining conditions. It has been determined that mining at Loraine mine is regarded the most relevant. This is due to similar shale thicknesses, and hangingwall conditions. The search was broadened to other operations in the Nigel area and the rest of the Free State gold fields, where resue stoping methods was practiced. This review was done on conventional resue stoping methods as well as mechanised resue methods.

Mechanical excavation methods and the benefits thereof was investigated to determine the potential advantages and disadvantages it has over drill and blast methods.

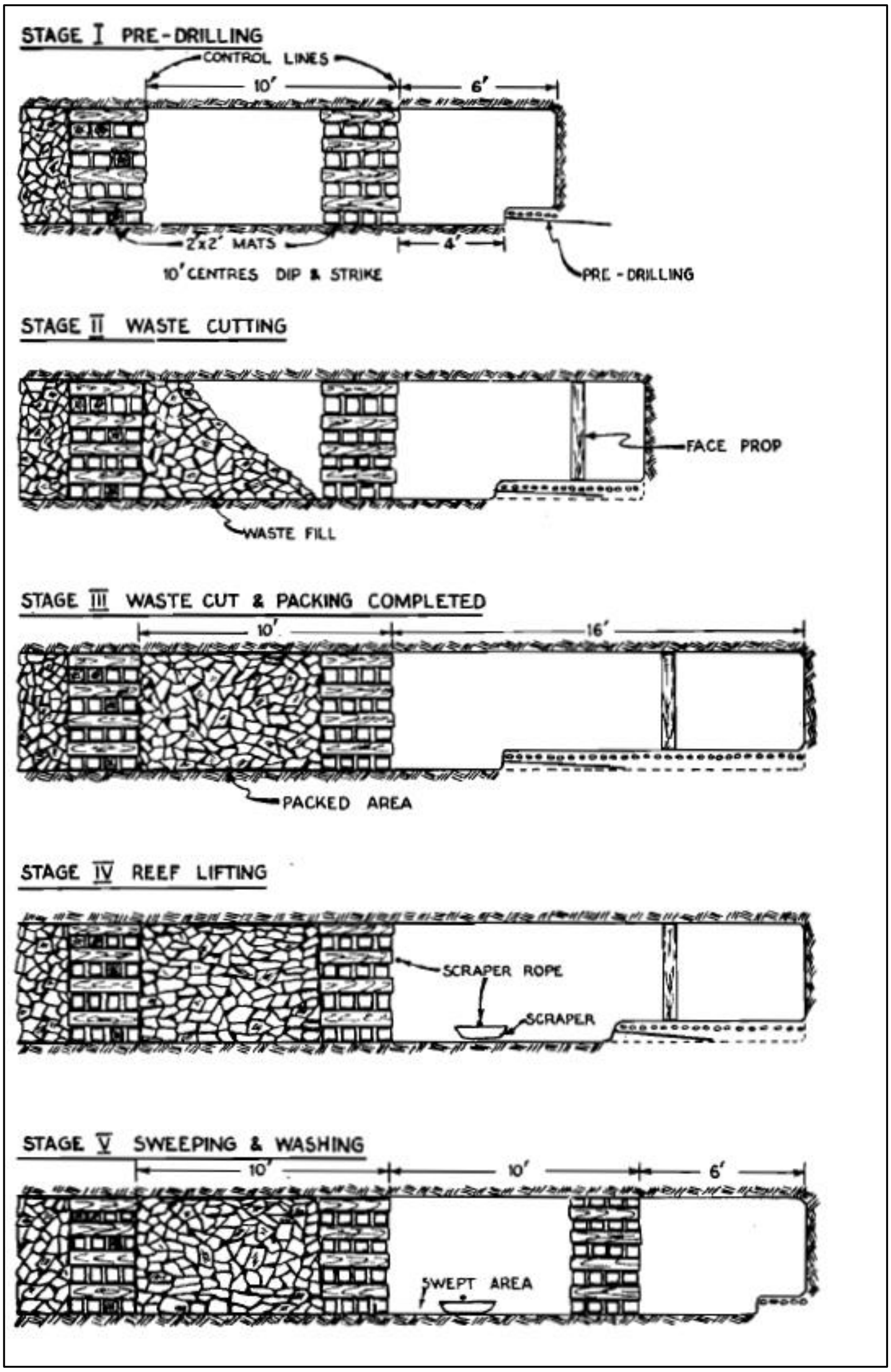
Lastly the tests results on shale is summarised under this chapter.

2.2 RESUE MINING METHODS

2.2.1 CONVENTIONAL RESUE STOPING

Resue stoping was widely practiced at the sub-Nigel operations from 1927 to 1930 (Bryson, 1958). It was also used as a mining method at Lorraine and the President Steyn shaft between 1955 and 1965 in the Free State Goldfields. The process involves advancing the face into the waste portion only for several metres above or below the reef. Typically waste material was hand packed in the back area, hence reliant on a large labour force. The reef is then extracted in a separate process by ensuring that blasting is done only by fracturing the reef and not to use an excessive amount of explosives that can displace the reef. Conventional resue stoping is explained with the aid of Figure 4.

Figure 4: Conventional resue stoping (Bryson, 1958)



According to Bryson, the disadvantages of this method are the following:

- Intermittent face advance – Since the reef is extracted in a second phase, face advance is stopped;
- Hanging wall control – During the waste mining cycle only temporary support can be used;
- Fluctuations in reef supply – the amount of reef hoisted or provided to the mill can vary significantly depending on the amount of panels working on the reef mining cycle across the mine;
- Productivity of method – The hand pacing of the waste material requires a large labour force and the rate of mining is directly related to the number of labourers available to do the hand packing;

- Waste dilution and gold loss – strict supervision is required to ensure that waste material is not transported as reef or to ensure that before the following waste mining cycle is started, that all the reef is removed diligently.

2.2.2 CAST BLASTING/THROW BLASTING

Since the development of millisecond delay detonators such as shock tubes and/or electronic detonators, the opportunity to selectively blast waste and reef was made possible. Effective throw blasting of waste, separating it from the reef material at the face, could replace the handpacking process that was found to be risky and at a high cost, hence throw blasting is essential for an effective resue stoping method. This was further investigated by Dr Immo Bock and according to his article, "Selective blast mining in gold mines", waste and reef can effectively be separated in a single blast by applying suitable detonator delays. Waste material was throw blasted into mined-out areas while the reef was fractured only on the face. The waste material can function as immediate roof support when cast blasted with sufficient heave (Bock, 1996). Results based on selective blast mining indicated that cast blasting is possible and that the waste material is relatively well compacted in the back area.

In a follow-up paper, additional tests were done on a platinum mine where the blasted waste rock was earmarked to function as support, however due to a solid pyroxinite hanging wall there was little to no convergence in the stope and blasted waste rock could not function as effective support (Bock, et al., 1998). This remark is particularly relevant to mining at Jeanette, the overlaying thick (8 – 12m) quartzite beam is expected to illustrate very little convergence and hence blasted shale may not function and effective support.

In addition to the tests done by Dr Immo Bock, cast blasting was tested at a platinum mine by using longhole drilling and blasting methods. In these trials the reef material was throw-blasted into a nearby Advance Strike Drive ("ASD") by drilling and blasting longholes (van Dorssen, et al., 2002). These trials were done to successfully throw blast the reef up to 25m from the face.

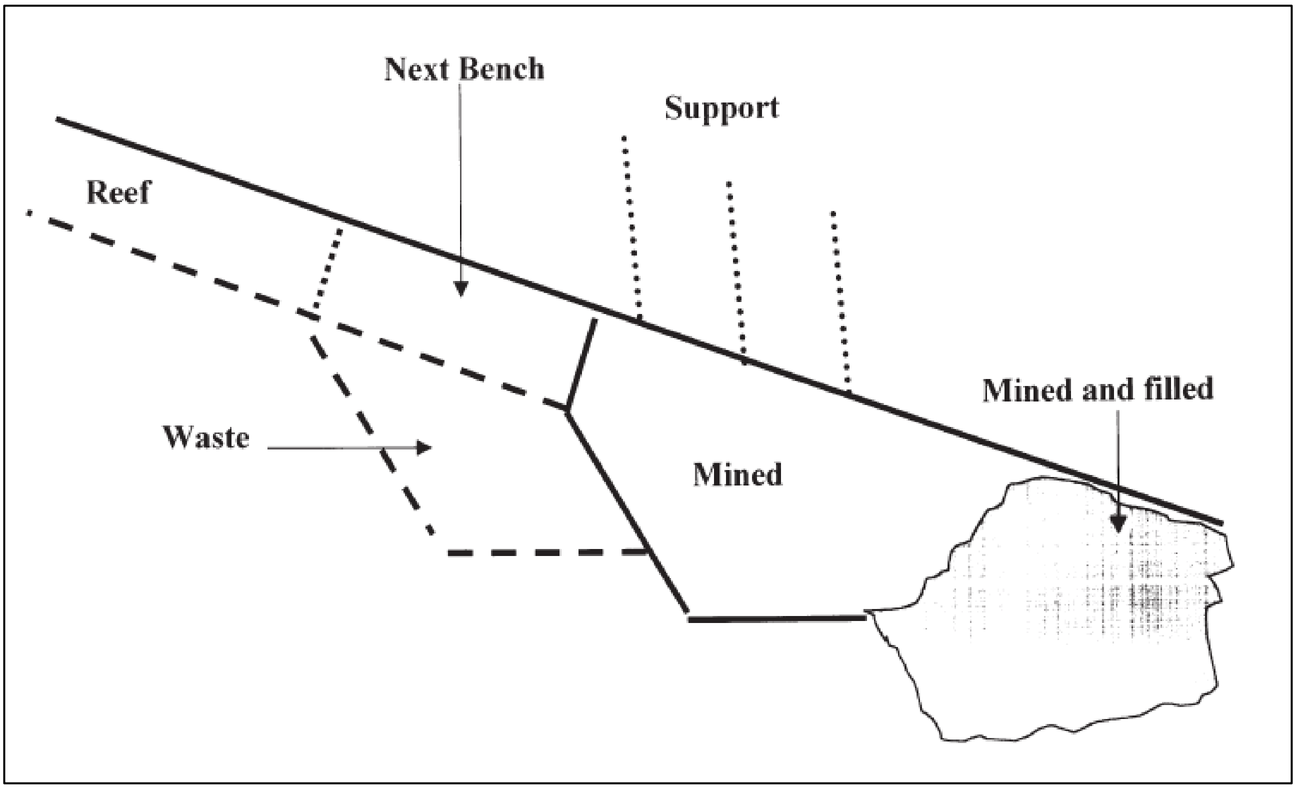
2.2.3 MECHANIZED RESUE STOPING AT TARGET GOLD MINE

Mechanised resue stoping has been regarded as a possible solution for mining a deep level deposit at Target Gold mine (Lourens, 1999). The article indicates that the proposed method was about to enter a trial phase, however no literature was obtained describing the results of the trials.

This method was developed between Sandvik and Target Gold mine and a Journal paper was published on the method proposed. According to the paper, a mechanized resue stoping method was identified as a solution to, improve safety, improve productivities and decrease mining costs. The use of mechanised equipment will require larger excavations resulting in more dilution, a resue mining method will introduce the principle of mining wide tramming narrow (Pickering, et al., 2000)

The mining method comprises 60m long on-reef strike drives that will be advanced in an up-dip direction. The drive will be blasted in 30m segments advancing 2m per blast. Selective blasting techniques will be applied to throw blast the waste material into the down-dip side of the gully. The reef will be dropped keeping it in as large as possible fractures to minimise gold losses. A section view of the mining sequence is illustrated in Figure 5.

Figure 5: Mechanized resue stoping - Target Gold mine, section view showing sequence (Pickering, et al., 2000)



The blast design for this method is crucial to the success thereof and hence electronic detonators was considered. The waste will be compacted into the bottom gully by throw blasting the material. The reef will be dropped by firing a single line of blast holes, designed not to displace the reef and to ensure a coarse fragmentation. The amount of waste material to be cast blasted was a function of equipment operating heights and the correct amount of blasted waste material required to fill the bottom gully.

2.2.4 MECHANISED LONGHOLE STOPING METHOD

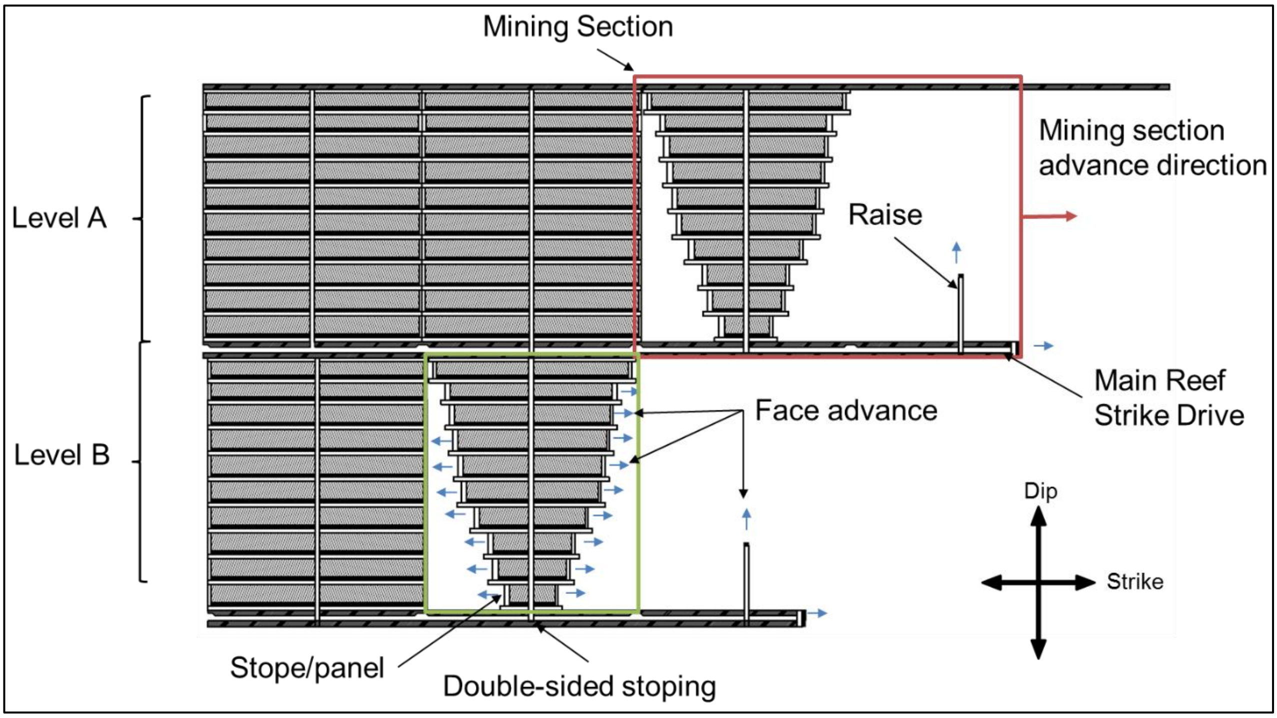
Longhole stoping was considered as a viable mechanized resue stoping method for Jeanette. There were some trials done on a platinum mine where the reef material was throw-blasted into a nearby Advance Strike

Drive (“ASD”) by drilling and blasting longholes. According to the tests done at the Platinum mine the following results were achieved (van Dorssen, et al., 2002):

- Drilling accuracy was well within the required ranges;
- To point the drill feed in the correct direction was seen as the biggest reason why holes were not drilled accurately, however this was resolved by installing an angle indicator on the feed;
- Throw blasting of ore into the strike drive was achieved;
- Final stope cleaning was done by water jetting; and
- The drilling of longholes reduced the stope height and improved grade.

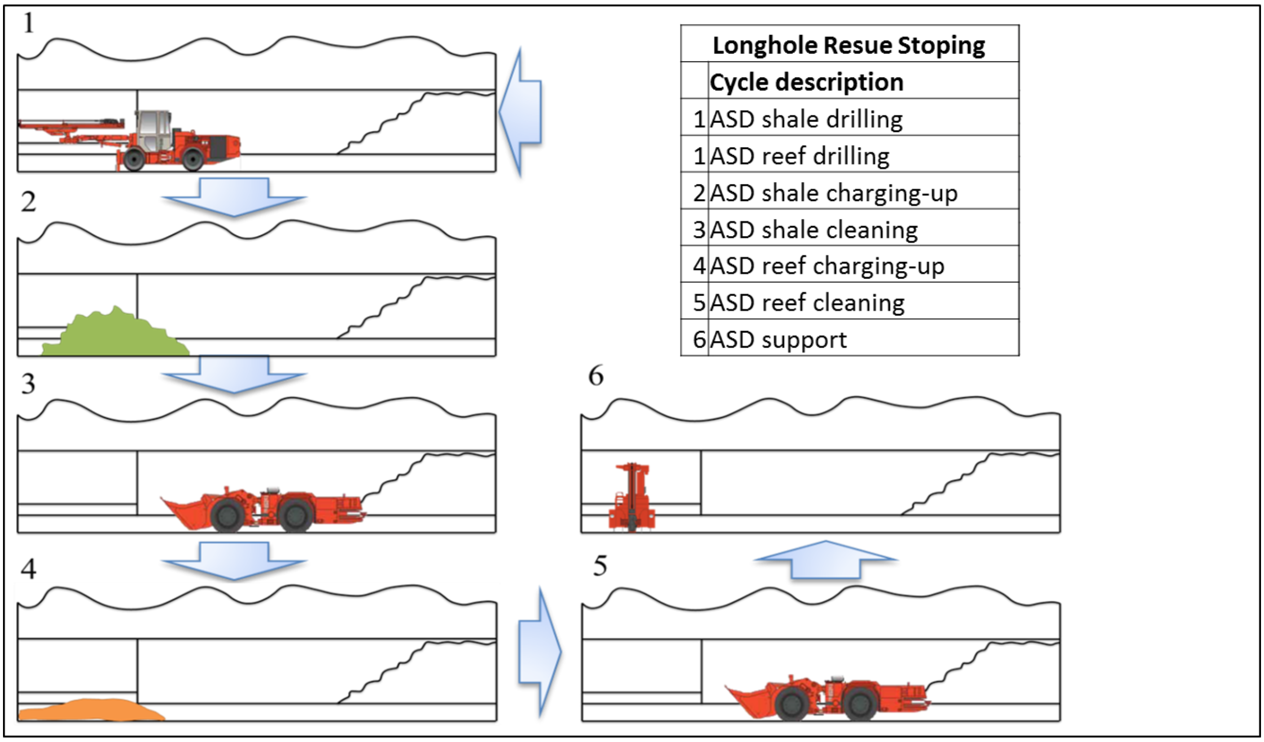
Longhole Resue stoping utilises a longhole drill rig to drill long holes into the overlaying shale and reef. The shale will be blasted and cleaned before the reef is blasted. The shale will be throw blasted into the mined-out area. The velocity of the blast will compact the shale into the worked-out area behind the advancing face. The reef holes will be decoupled to reduce the amount of explosives per blast hole. The aim is to fracture the reef into larger pieces to minimise gold loss in fine material, hence applying all the priciples discussed above. An overview of the Longhole Resue Stoping method is illustrated in Figure 6 (TaungGold, 2014).

Figure 6: Longhole Resue Stoping method layout – Plan view (TaungGold, 2014)



Most of the development will be done on-reef with a hanging wall positioned waste drive. The on-reef development mining cycle is illustrated in Figure 7 (TaungGold, 2014).

Figure 7: On-reef development – mining cycle (TaungGold, 2014)

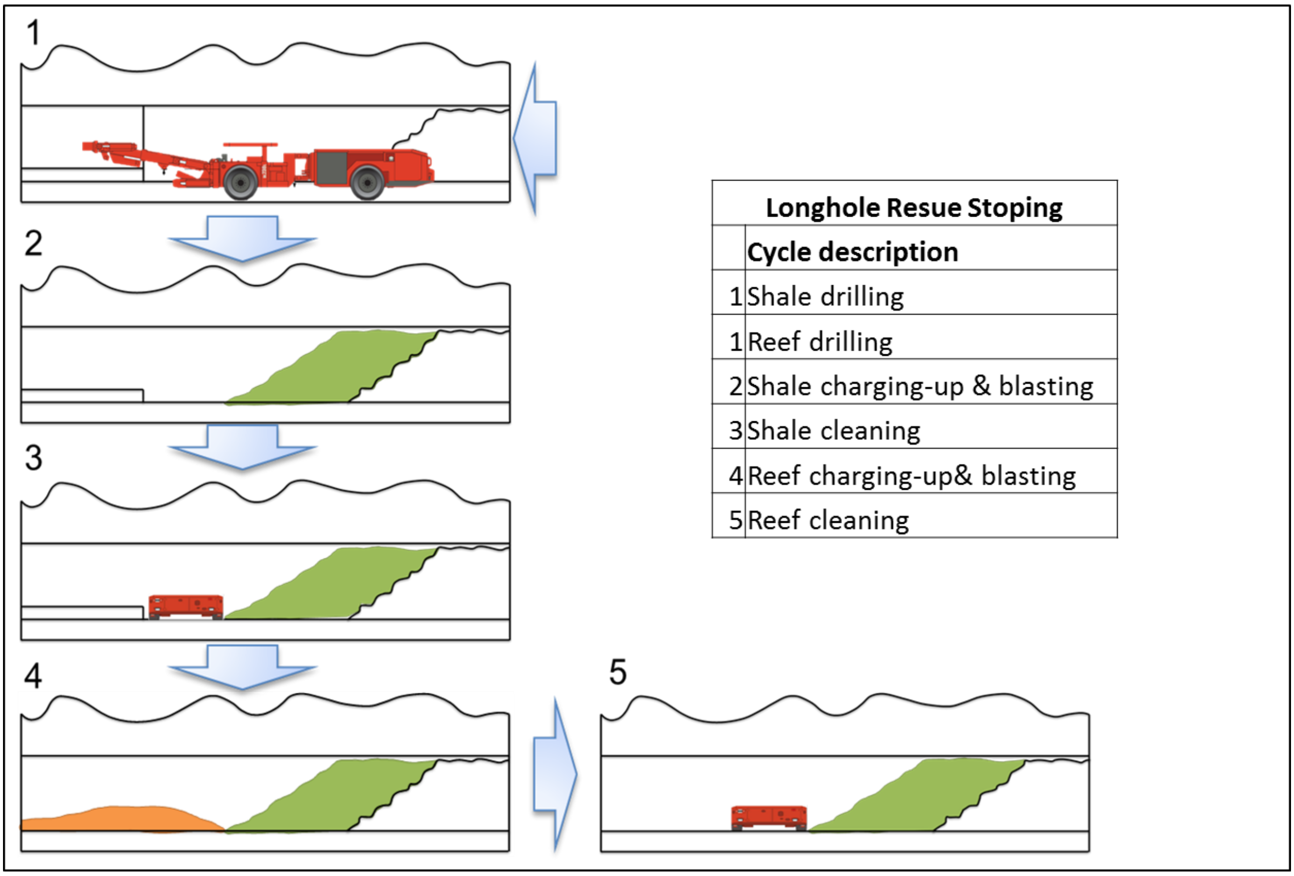


The mining cycle can be described as follows:

1. Drilling of shale and reef: the gully will be drilled with a low-profile drill rig in the direction of mining.
2. Charging and blasting of shale: the shale will be blasted first.
3. Cleaning of shale: all the shale will be loaded by a LHD and removed from the stope.
4. Charge and blast of reef: the reef will be fractured into large chunks at the face.
5. Cleaning of reef: the fractured reef will be loaded by LHD.
6. After cleaning, support will be installed.

The mining cycle for Longhole Stoping is illustrated in Figure 8 (TaungGold, 2014).

Figure 8: Longhole Resue Stoping – mining cycle (TaungGold, 2014)



Longhole Resue Stoping	
	Cycle description
1	Shale drilling
1	Reef drilling
2	Shale charging-up & blasting
3	Shale cleaning
4	Reef charging-up& blasting
5	Reef cleaning

The mining cycle can be described as follows:

1. Drilling of shale and reef: a longhole drill rig will drill blast holes into the shale and reef in an up-dip direction. Drilling will be conducted perpendicular to the direction of mining.
2. Charging and blasting of shale: the shale will be blasted first by charging-up the longhole from the top gully. Cast blasting of the shale will be done into the stoped-out area.
3. Cleaning of shale: the extra shale will be pushed out of the panel with a remote-controlled dozer.
4. Charge and blast of reef: the reef will be fractured into large chunks at the face. The blast holes drilled into the reef will be decoupled to reduce the amount of explosives used for the blast.
5. Cleaning of reef: the reef will be pushed into the ASD by the remote-controlled dozer and loaded by an LHD.

The longhole drill rigs will access a stoping panel on apparent dip. Drilling will be done in an up-dip direction and no trackless equipment other than the XLP dozer will enter the panel. For this reason, the dip application of the mining method can be increased to 25°.

2.3 Mechanical excavation

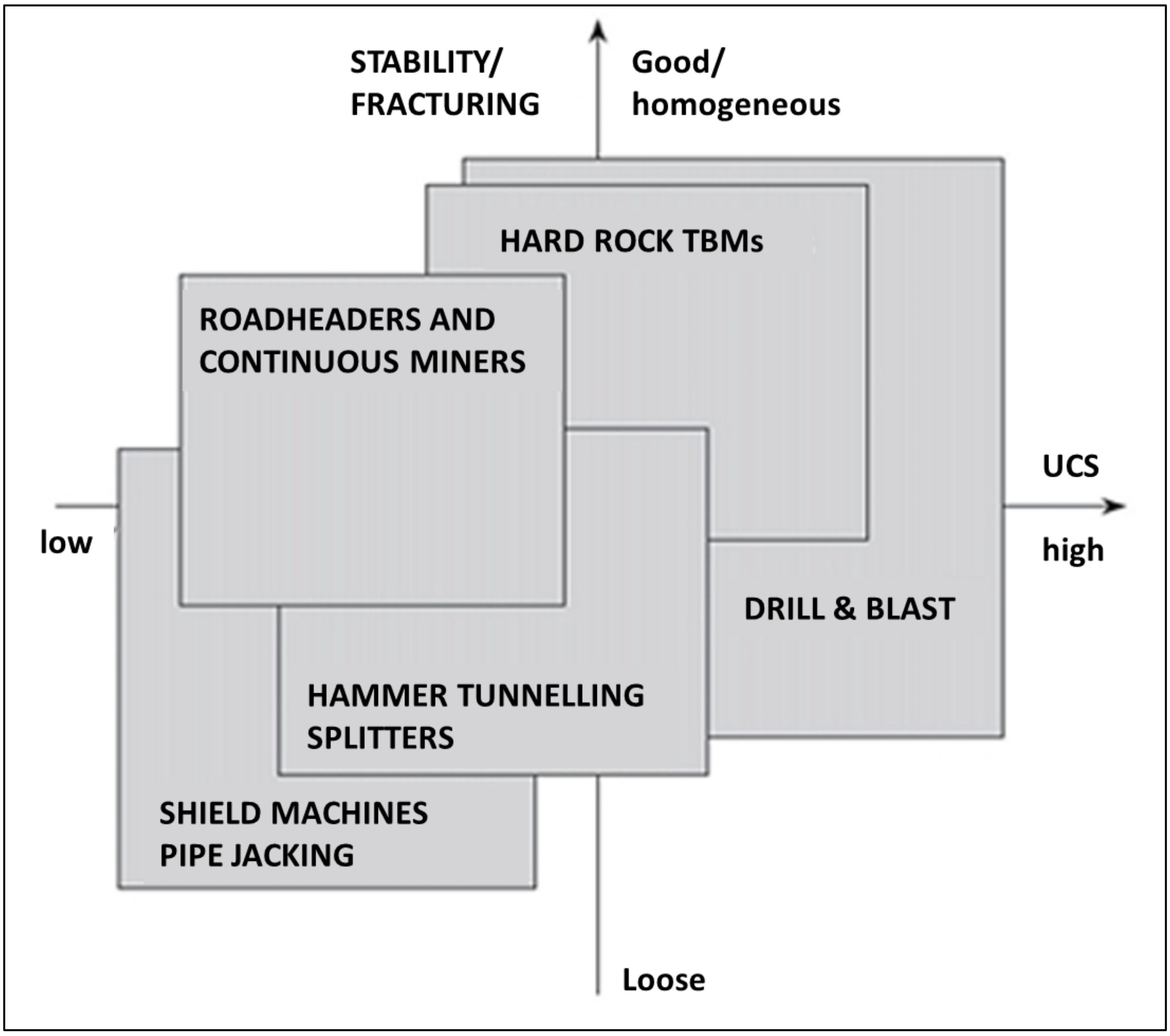
This chapter summarises the results obtained from an investigation into mechanical excavation methods to determine the potential advantages thereof for mining operations in the Jeanette Mine. The benefits that mechanical excavation or mechanical cutting can provide is grouped in the following three categories (Pickering, 2013):

- Speed – advance rates and/or development rates in metres per month;
- Excavation accuracy – control of dilution;
- Stability of excavations; and
- Ventilation.

2.3.1 An overview of Mechanical excavation equipment and their application

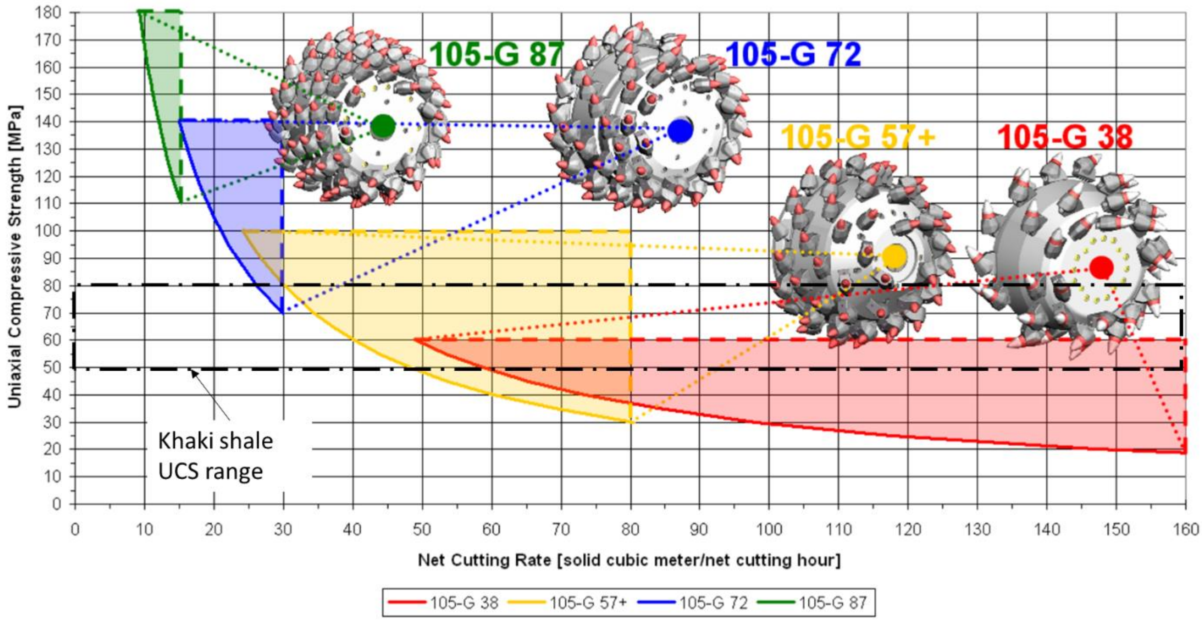
To determine the cut ability of different rock types, there are various metrics in use, of which Uniaxial Compressive Strength (UCS) and Rock stability is the most commonly used to indicate the best suitable mining method (Pickering, 2015).

Figure 9: Tunnelling/mining method based on UCS and rock stability (Priyadarshi, 2015)



Knowing that shale lays within the lower range of UCS the use of Roadheaders and/or continuous miners can be considered as suitable cutting equipment. Continuous miners and roadheaders comprise of picks that are dragged along the rock face. In harder rock conditions different cutterhead configurations are required, harder rock required more currents spaced closer together (Gehring, n.d.). The roadheader cutterhead configuration that is best suited for rock at different UCS values are illustrated in Figure 10.

Figure 10: Approximate cutting rate associated with different cutter head designs



Based on UCS testing of the Khaki shale at the project, the average UCS is 60Mpa which is well within the capabilities of roadheaders and/or CMs with favourable nett cutting rates.

2.3.2 MECHANICAL EXCAVATION METHOD ADVANCE RATES

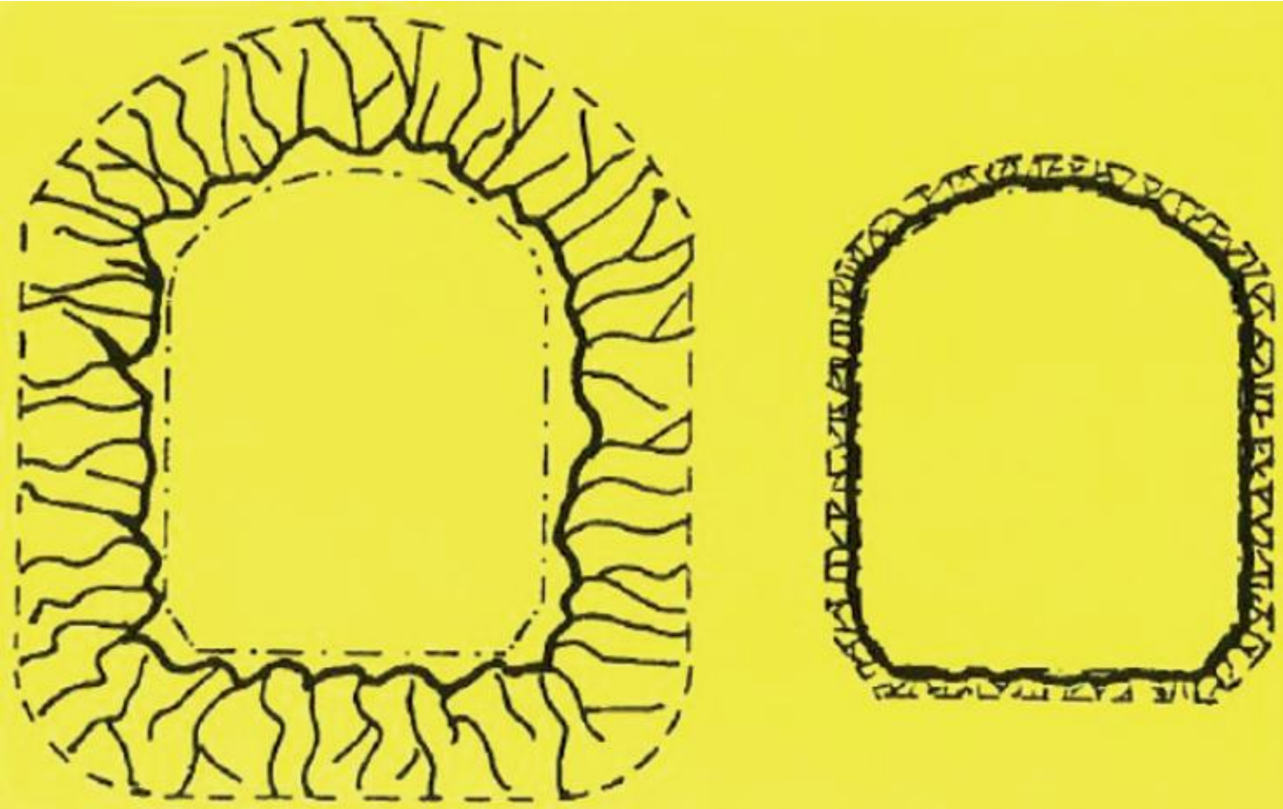
The South African coal mining industry is regarded as a benchmark for the introduction of mechanised cutters, also known as continuous miners. The South African coal industry reflected significant production increase since the early 1970's (Mohring, 1997). Since the mid 1970's there has been a steady increase in the number of continuous miners used in the South African coal mines which has a significant contribution to coal production increases (Hardman & Oberholzer, 1987). This direct correlation between the introduction of coal cutting machines and improved productivities since the 1970's can be seen as to support the significance of introducing mechanical cutting, instead of drill and blast.

According to a study that was done to compare drill and blast advance rates with mechanical excavation methods, it was found that mechanical excavating can exceed drill and blast advance rates on average by 3 times (Knights, et al., n.d.) In this same study an average advance rate for mechanised mining was determined based on information from several operations, the rate obtained was 6.8m/day. Based on Figure 10 above in comparison a roadheader can provide up to 20m/day advance in shale material.

2.3.3 ROCK ENGINEERING – ROCK CUTTING VS. BLASTING BENEFITS

The benefit of ground stability when using a CM that cuts the rock rather than blasting is well understood. Blasting causes stress fractures deep in the surrounding rock when the blast is initiated, while the cutting action ensure that no or very little fractures are created into the rock as illustrated in Figure 11.

Figure 11: Blasted rock fractures compared with mechanical cutting (Restner, et al., 2013)



Blasting is estimated to cause fracture up to 1.5m deep into the rock, which requires additional support.

The benefit of cutting in terms of ground stability can further be explained by referring to two examples (Pickering, 2013, p. 6). In a case study, two tunnels were developed in the early 1980s, one a drill and blast and the second developed by a Tunnel Boring Machine ("TBM"). Both tunnels are 1,600 m below surface and only a few metres apart. A comparison between these excavations after a seismic event is illustrated in Figure 12.

Figure 12: Damage on excavations after a seismic event (Pickering, 2013)

The TBM developed tunnel indicated some fracturing along the sides only, but is still in a good condition. The drilled and blasted tunnel indicated a significant amount of deterioration. All the support should be re-installed in the affected area where large falls of ground occurred.

The second case study refers to the local platinum mines where rock cutting methods were tested. Approximately 7,000 m^2 of platinum was extracted in the industry to date, using cutting equipment. Across this entire area there was not a single rock fall recorded. The following figure illustrates the reef cutter at work.

Figure 13: Reef cutter (Pickering, 2013)

2.3.4 VENTILATION BENEFITS

Drill and blast operations requires diesel powered LHDs and trucks to haul blasted material from the face. Electrical LHDs and trucks are available, however not typically used due to limited flexibility. The main advantage between mechanical cutting and drill and blast is the use of electrical equipment instead of the diesel units.

Compared to diesel equipment, electrical equipment is not as demanding on the ventilation and refrigeration system. Electric equipment has no diesel emissions and the heat generated from electrical motors are less than diesel engines. The design criteria used for diesel equipment and for electrical equipment are compared in Table 1.

Table 1: Ventilation design criteria comparison, between Diesel and Electric equipment (Jacobs, 2013)

		Diesel Equipment	Electrical equipment
Heat load	Factor to equipment rated power	2.5	1
Ventilation flow factor	m3/s/kW	0.063	0.04

Per kW rating, electric engine requires 20% less ventilation, only required for heat control. Electric motors do not emit any fumes or Diesel Particulate Matter. (Halim & Kerai, 2013)

2.4 SHALE PROPERTIES

This chapter consolidates the results of tests that were conducted to understand the behaviour of Khaki Shale at various applications. Shale was sourced from drill cores and obtained from other areas. Due to the limited amount of shale samples available for testing, an alternative source of shale was identified. This material is referred to as Wonderstone shale and was sourced from a quarry with a shale outcrop.

Based on a series of UCS tests the following averages were obtained for the Khaki Shale and Woderstone shale:

	UCS (Mpa)	E (GPa)	Comment
Wonderstone Shale	77	40.0	Average of all tests
Khaki Shale	60	22.3	Average of all tests

2.4.1 SHALE PROPERTIES - STUDY OBJECTIVES

The shale properties study was aimed at addressing the following key challenges regarding the potential perception of shale behaviour in mining practices:

- It has poor load-bearing properties as uncemented backfill
- It deteriorates rapidly when in touch with water
- It constrains ore passes flow
- *In situ*, under stress, the shale will heave like "toothpaste squeezed from a tube"
- Shale dust creates a severe mud issue

Laboratory tests were carried out by Tundra Bulk Solids Africa under supervision of Dr August Lamos at the University of the Witwatersrand to gain a better understanding of shale behaviour.

The following tests were conducted (Lamos, 2013):

- The load-displacement behaviour of Wonderstone Shale was tested: this was done to compare the load-bearing capacity of uncemented crushed shale with quartzite.
- Khaki Shale flow properties were tested at different quartzite/shale ratios: this was done to gain a better understanding of the flow properties through an ore pass.
- Shear strength comparisons of Khaki Shale submerged in water for up to one month: this was done to quantify the time-dependent effects of water on crushed Khaki Shale.
- Mineralogical analyses of <2 μm fraction of Khaki Shale dust compared to bulk shale material.

Where possible, standard testing procedures were followed. Indicative testing was also conducted, with the results compared to those of quartzite.

2.4.1.1 Loading behaviour of crushed shale

A resue mining method, which entails blasting the shale into the mined-out areas, is being considered for the Project. The blasted material will serve as uncemented backfill. The purpose of the PFS was to provide indicative values for the load-bearing properties of the uncemented shale backfill, to allow predictive numerical modelling of the behaviour of the material as mine support at the Project. This goal was achieved by crushing the shale to a given size distribution and compressing the loose bulk material in a press, thereby producing load-displacement (stress-strain) graphs of crushed shale.

The primary factors determining the load-bearing behaviour of the backfill are thought to be the inherent material properties of the shale, the particle size distribution ("PSD") of the comminuted material, as well as the packing density achieved in placing the backfill. The direct and derived results of this study were primarily used as input parameters in the numerical modelling of the load-bearing properties of uncemented shale backfill at the Project. The results could not, however, be used to infer the loading behaviour of *in situ* Khaki Shale, even though it may be heavily fractured. The results were also not predictive of the loading behaviour of cemented slurry backfill, whether cured or uncured.

Wonderstone Shale

To simulate a scaled Run of Mine (“RoM”) size distribution with the appropriate fractions of fines and ultra-fines, it was decided to crush the entire sample to a size of <20 mm. The PSD of this initial sample was measured and tests were conducted on different size distributions.

Quartzite

A sample size of 120 kg was crushed in the jaw crusher to the same PSD (<20 mm) as the shale.

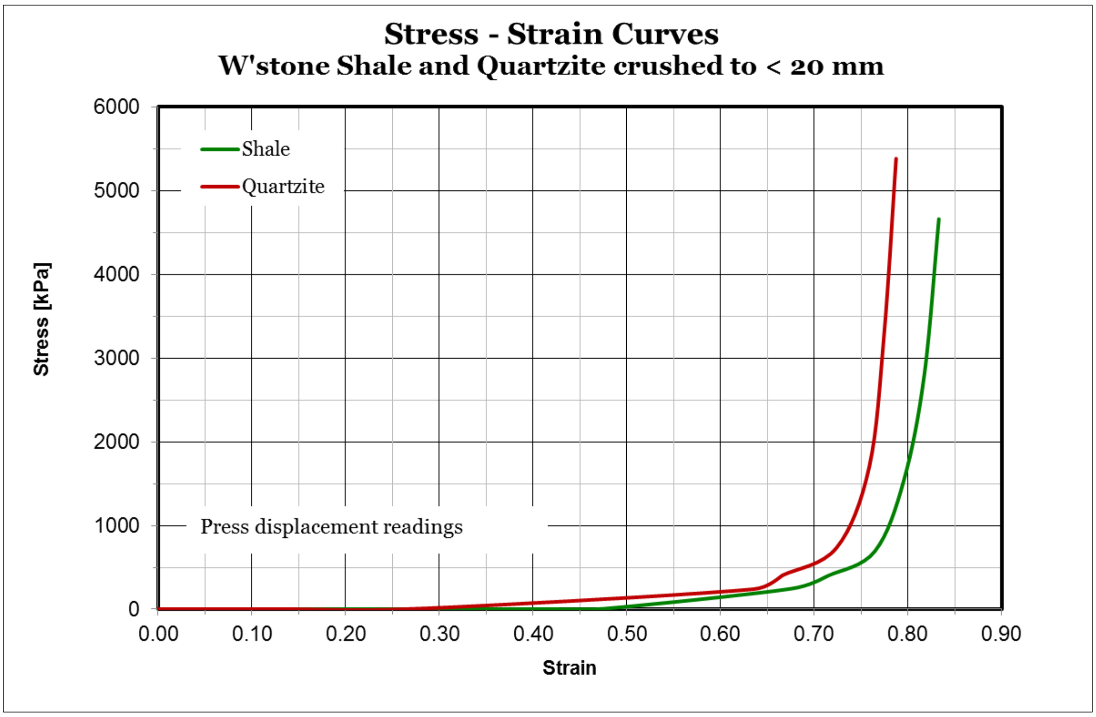
2.4.1.1.1 Compression tests

The load displacement tests were conducted in the Mohr & Federhaff 10 MN press at the Council for Scientific and Industrial Research’s (“CSIR’s”) Cottesloe laboratories. This press is large enough to test 100 kg of loose crushed samples. Displacement was measured at each platen corner and an average was taken at each planned stop with the shale placed in the centre of the platen. The geometry of the samples was measured before and after the compression cycle and relative densities were taken from the crushed samples.

Figure 14: Crushed shale compression test

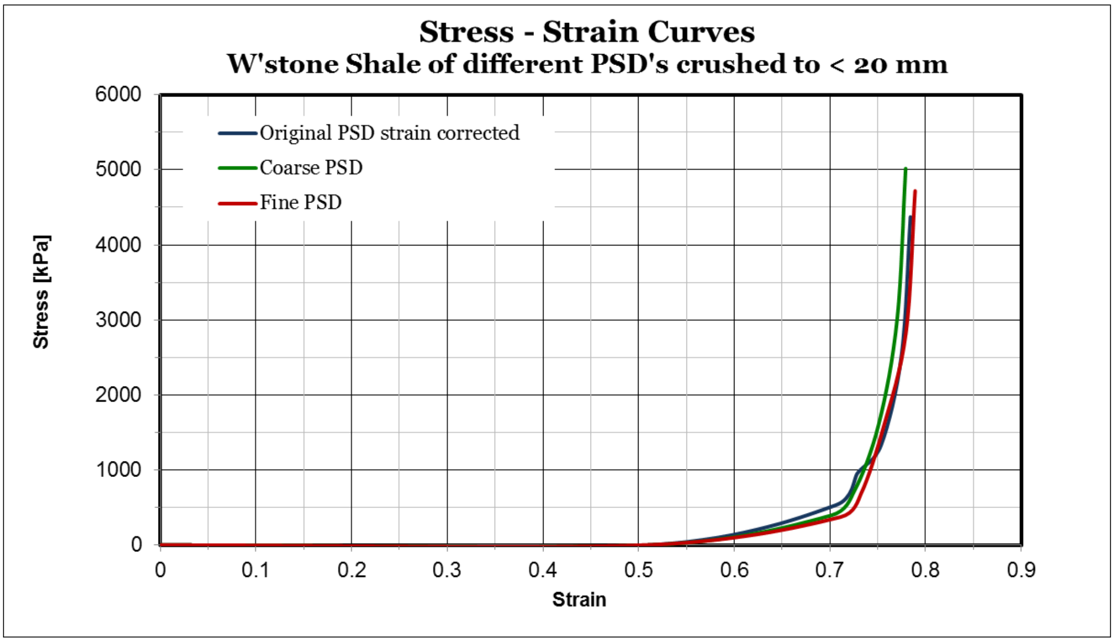
The displacement of the shale was measured and compared to that of quartzite (see Figure 15).

Figure 15: Stress-strain comparison: Wonderstone Shale-Quartzite



The shale endured more deformation than the quartzite before taking load. The difference (± 7% variance) is considered minor. The effect that different PSDs have on the shale's load-bearing properties was also tested. Figure 16 illustrates the results of the different PSDs.

Figure 16: Stress-strain relationship of crushed shale at different particle size distributions



No significant difference in load-bearing characteristics exist between the coarse, original and fine PSDs. The bulk densities of the material were measured during the test (see Table 2).

Table 2: Compressed crushed shale bulk densities

Sample	Density (t/m^3)
Loose bulk prior to compression	1.55
Centre of compressed area, test 3	2.37
Centre of compressed area, test 4	2.39
Off centre of compressed area, test 4	1.8
5 kN loading (triple cone tests)	1.62
10 kN loading (triple cone tests)	1.64
20 kN loading (triple cone tests)	1.75
50 kN TBSA laboratory bulk density	1.85

The following conclusions were drawn after the compression tests were conducted:

- Khaki Shale and quartzite show similar compression behaviour. Both materials have self-confining and strain-hardening properties, as well as similar angles of repose.
- Khaki Shale would behave in a similar fashion to quartzite tailings, if used as uncemented backfill.
- The shale material is insensitive to PSD within the range tested. Mechanically-cut shale, as well as blasted shale can be used as uncemented backfill.
- The results indicate that the shale, as uncemented backfill, would begin to take load at a width/height ratio of about 4:1, and at a packing relative density of about 1.6.

2.4.1.2 Flow properties

A standard testing procedure (developed by TUNRA bulk solids) used to determine the shear strength of material, was used to test shale flow properties. The shear strength of the following samples was tested:

- 100% Khaki Shale
- A blend of 75% Khaki Shale and 25% quartzite
- A blend of 25% Khaki Shale and 75% quartzite

The different shale to reef ratios with the associated flowability indexes are illustrated in Table 3.

Table 3: Comparison of pure shale and blends

	Material		
	100% Shale	75% Shale 25% Quartz	25% Shale 75% Quartz
Unconfined yield strength at 20 kPa major consolidation stress	6.0 kPa	7.2 kPa	9.5 kPa
Flow-ability index	0.31	0.41	0.57
Handling	easy	easy-moderate	moderate

The following conclusions were drawn after the flow properties tests were conducted on Khaki Shale:

- Khaki Shale and blends of Khaki Shale and quartzite show consistent yield strengths, dependent on the blend ratios.
- Khaki Shale and blends of Khaki Shale and quartzite show easy to moderate handling properties.

2.4.1.3 Effect of water on shale

A Khaki Shale sample was crushed to <4 mm particle size and split into 3 parts. The three samples were designated to undergo different tests. A shear strength test was conducted on the first sample using the Jenike shear cell. The test was performed directly after the shale was immersed in water to indicate the condition of the shale at "time zero" since placed under water. The other two samples were immersed in water for 2 and 4 weeks respectively. The same test was performed on these samples that delivered the results indicated in Table 4 (Lamos, 2013).

Table 4: Shale shear strength properties after being placed in water

Immersion period	Shear strength kPa
Instantaneous @50% mc	20.33
2 weeks	20.45
4 weeks	24.24

It can be concluded that Khaki Shale does not deteriorate noticeably when coming into contact with water; there is no reduction in shear strength after it has been immersed in water for a month. No weathering on sharp edges of the shale was observed after immersion in water for this period.

2.4.2 SHALE AS CEMENTED BACKFILL STUDY

This study comprises two parts. The first part aims to determine the properties of cemented shale to act as support. The second part aims to design an underground backfill plant that could use shale underground to create a cemented backfill with adequate strength for application directly into mined-out areas.

2.4.2.1 Cemented backfill properties

One of the economic advantages of shale is that it can be used as cemented backfill. The Department of Civil Engineering at the University of Pretoria was tasked with formulating a backfill using a Khaki Shale sample. The backfill had to adhere to the following requirements:

- Shale must be crushed to less than 20 mm in size for use in backfill.
- The mix must be pumpable for at least 3 hours.
- The mix must have a strength of at least 1 to 2 MPa at 28 days' drying time.
- The mix must be able to stand alone within 7 days.
- Water must not leach out of the mix.
- The mix must be durable for at least 3-6 months.

Tests were conducted on various cement types in different quantities. Strength tests were conducted after 4, 7 and 28-day drying periods. The following conclusions were drawn after the tests were conducted:

- Shale can be used as mine backfill.
- Minor degradation takes place during the first 6 months.
- Less than 100 kg/m^3 cement is needed to make backfill with a strength of 2 MPa.
- A water-reducing agent and aeration are essential.
- No additional fines are needed; PSD is favourable after crushing.

These conclusions guided the second phase of the study, namely the Backfill Plant Design.

2.4.2.2 Cemented shale backfill plant design

A fit-for-purpose backfill plant will be required to produce the cemented shale backfill. This plant will be situated within a mining zone and the plant will be relocated as mining progresses. The plant was designed with the CM method in mind and the following were the set design criteria:

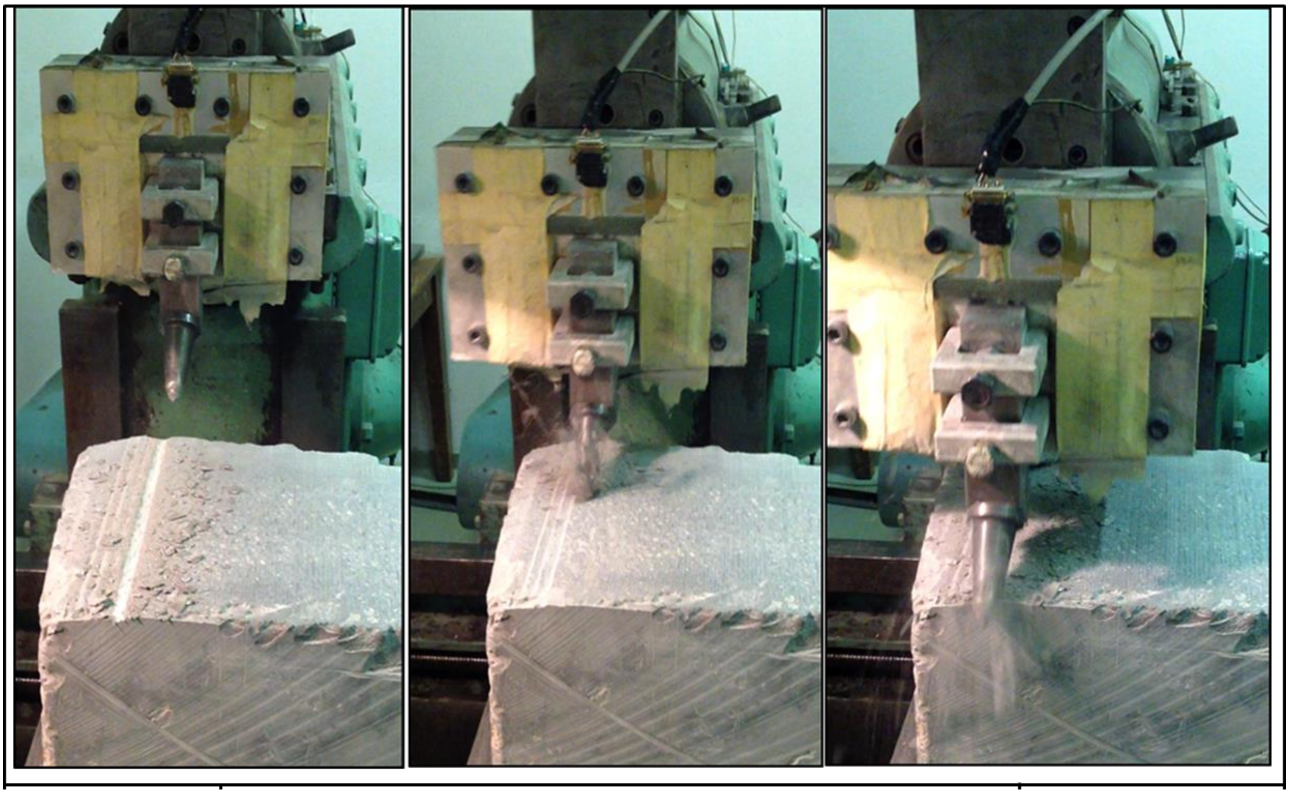
- Modular comminution and backfill preparation plants must be located on different levels and in different mining zones.
- Low-profile skid-mounted mobile equipment must be utilised to make the plant more mobile.
- Plants must be horizontal in design for location in development ends or excavations, about 4-5 m wide and 4-5 m high so that the plant can fit into a typical cross-cut or haulage.
- A 50-65 t/hr solids throughput must be ensured to match the CM method.
- Crushed product with 100% by-mass passing 10-20 mm (maximum size) in PSD.
- Stiff fill containing about 75% by-mass solids.
- Cement dosage containing 4,5-6% by-mass (UCS 0,6 to 2.0 MPa).
- For matched mining and backfilling, and un-comminuted Khaki Shale by-passing, the backfill preparation facility per shift will be 15,32% by-mass of the Khaki Shale tonnage mined.
- Excess Khaki Shale will ultimately require hoisting to surface.

The design, capital cost and operating cost for the plant are not described in more detail in this study.

2.4.3 SHALE CUTTING

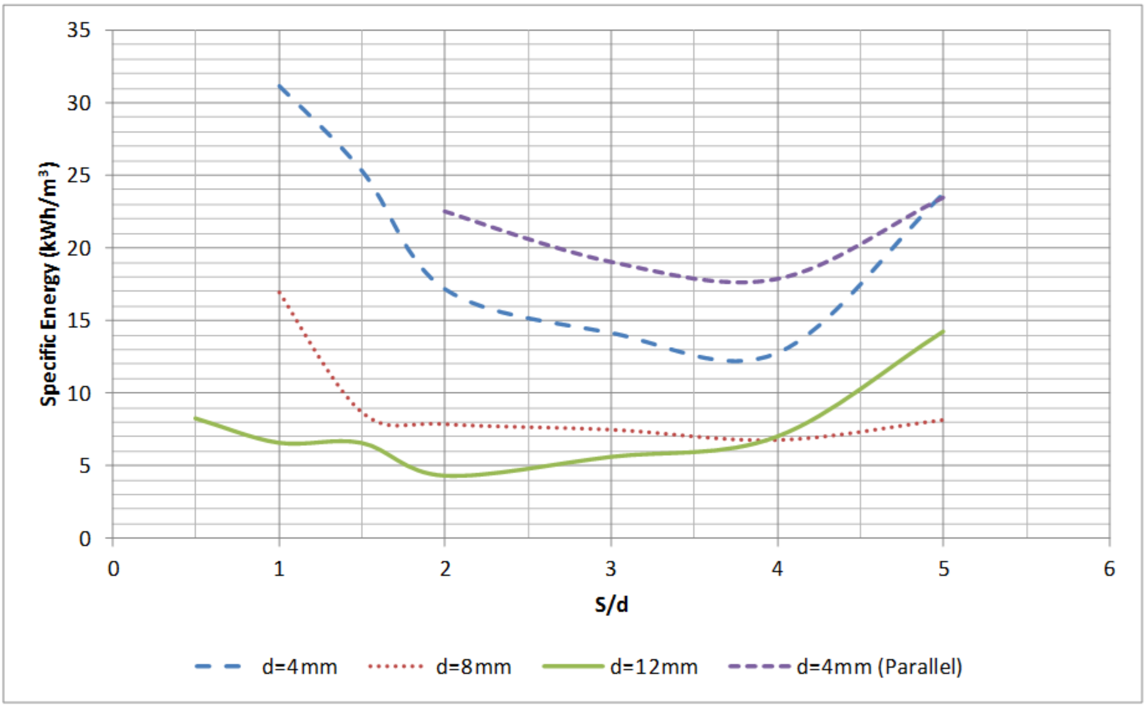
An investigation into the applicability of mechanical excavation techniques was proposed to seek methods to overcome the instability caused by the fracturing of Jeanette Khaki Shale during blasting. If the weaker shale formation above the reef is excavated mechanically, a less fractured, more stable sidewall condition should be created. To assess whether the Khaki Shale can be excavated, a similar, more readily-available rock type, the Wonderstone Pyrophyllite, was used in a suit of linear cutting tests. To conduct the linear cutting test, a single, intact block was required (only fragments of Khaki Shale were available). The linear cutting rig is illustrated in Figure 17.

Figure 17: Linear cutting test rig



The cutting of Wonderstone Pyrophyllite was intended to alleviate two main challenges in the mechanical excavation applicability assessment of Jeanette's Khaki Shale. The first challenge was related to the effective utilisation of a mechanical cutting system in the target area. To address this challenge, cutting efficiency parameters were derived from the tests. These are the specific energy values which are calculated for the optimum pick spacing to cutting depth ratio (S/d), which is partially determined and is a proposed ratio of 2. Specific energy value for a cutting depth of 12 mm and a spacing of 24 mm provided an opportunity for an efficient cutting design (see Figure 18).

Figure 18: Specific energy vs. spacing ratio



The lower the specific energy of the d=12 mm, the more efficient the cutting configuration. This did, however, lead to the second challenge, which is related to the Project productivity requirements and the technical capability requirements of the ground conditions. These challenges can only be partially addressed based on the results of the linear cutting tests. Parameters related to this matter develop from the relationship between the cutting depths and pick forces.

The force required to excavate the Khaki Shale is approximately twice that required for typical coal cutting at any given cutting depth. However, a suitable mechanical excavation system can be designed and operated to excavate at shallower penetration depths (12 mm) with lower force requirements. This option should be further investigated with the manufacturer's specifications and the envisaged production rates.

2.5 Conclusions

Conventional resue stoping methods are not suitable for the Jeanette Project, these methods are labour intensive with a high exposure to risk. The methods were abandoned in the past for more productive methods such as open stoping or undercutting.

Longhole mechanised resue stoping is regarded as a viable alternative method. This method has been proven to effectively throw blast material and narrow vein extraction on flat to moderately dipping ore deposits in the platinum industry.

One important aspect about the Jeanette Project is the fact that the shale is a thick soft overlying layer of rock that has a suitable hardness to be extracted by continuous miners or roadheaders. This has the advantage of high rate, continuous extraction rather than a cyclic method of drill, blast, clean and support.

Based on the outcomes of the test work done on the shale, the following conclusions can be made:

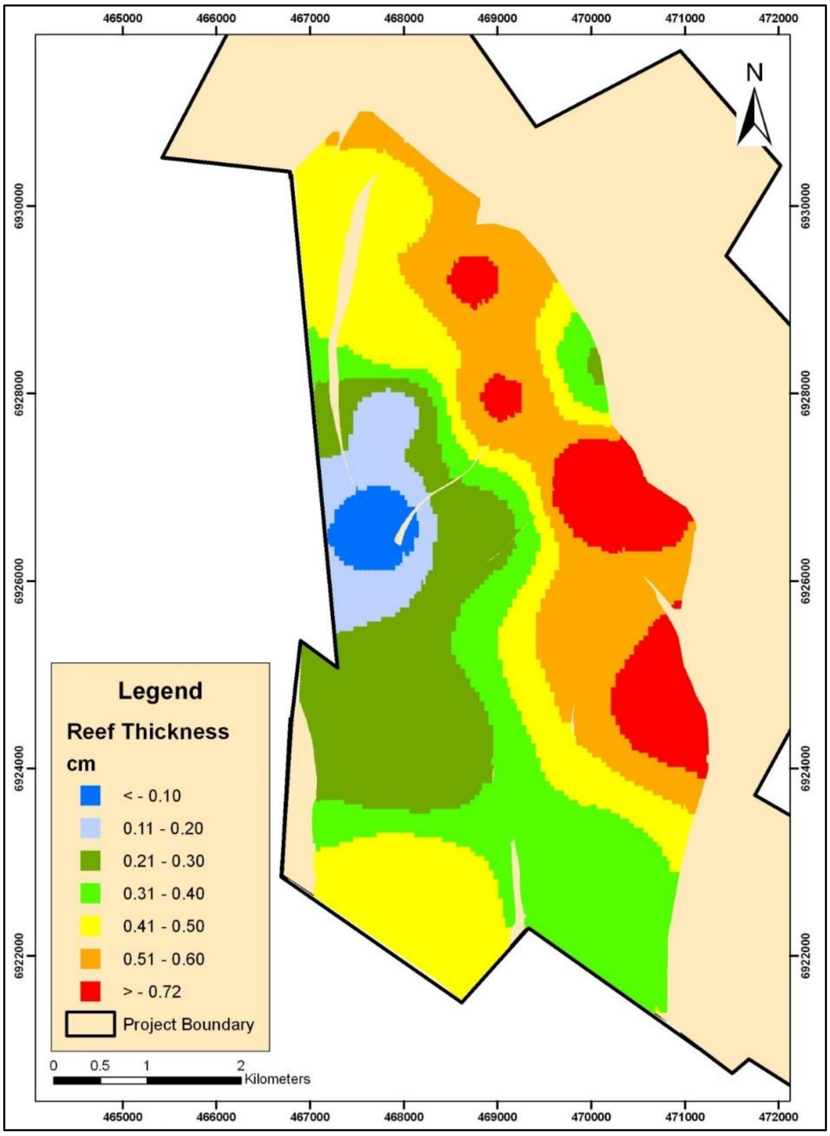
- The shale can be used as cemented backfill.
- The technology to convert “as mined” shale into cemented backfill exists and can be located close to the mining area in a relatively small excavation of 20-25 m^2.
- As uncemented backfill, shale will perform in a similar manner to uncemented mine tailings.
- The shale can be cut by conventional pick cutting machines used in soft rock mining. The pick life will be reasonable and cutting will cause less damage to the remaining shale.
- The technology of longhole drilling in narrow reefs has been demonstrated in South Africa. The practicalities of drilling at depth in stressed shale have not been evaluated.
- Throw or cast-blasting of shale will be possible and good results can be expected

3 JEANETTE OREBODY PROPERTIES

3.1 BASAL REEF AND MIDDLING QUARTZITE

The Basal Reef is developed at the Project Area region as a thin bitumen-rich and pyrite-rich pebble conglomerate, ranging from 1 cm up to 70 cm in thickness (Figure 19). The Basal Reef is overlain by a quartzite layer, known as the Middling Quartzite. The Middling Quartzite varies in thickness from a few cm up to 70 cm. This unit is, in turn, overlain by the pyrophyllite-rich Khaki Shale (Young, 2016).

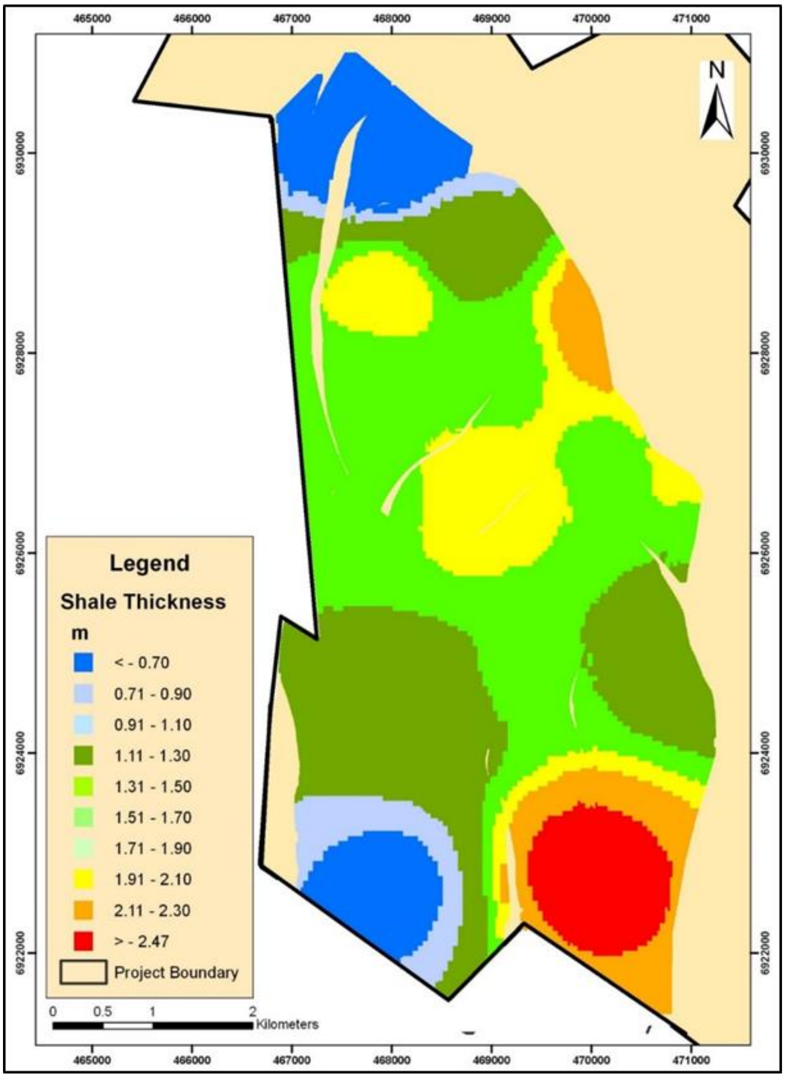
Figure 19: Basal Reef thickness at Jeanette Mine



3.2 SHALE THICKNESS

Figure 20 represents the Project Area's Khaki Shale isopach. The argillaceous shale overlies the Basal Reef Middling Quartzite and varies in thickness from less than 70 cm in the north, to more than 270 cm in the south. A west-east thickening of the Khaki Shale is also evident, ranging in thickness from approximately 100 cm in the west, to more than 240 cm in the east. The shale is geotechnically potentially unstable and exhibits typical sedimentological lamina and distinct parting planes due to its internal mineralogical clay structure being predominantly pyrophyllite, a phyllosilicate mineral composed of aluminium silicate hydroxide: $Al_2Si_4O_{10}(OH)_2$ (TaungGold, n.d.).

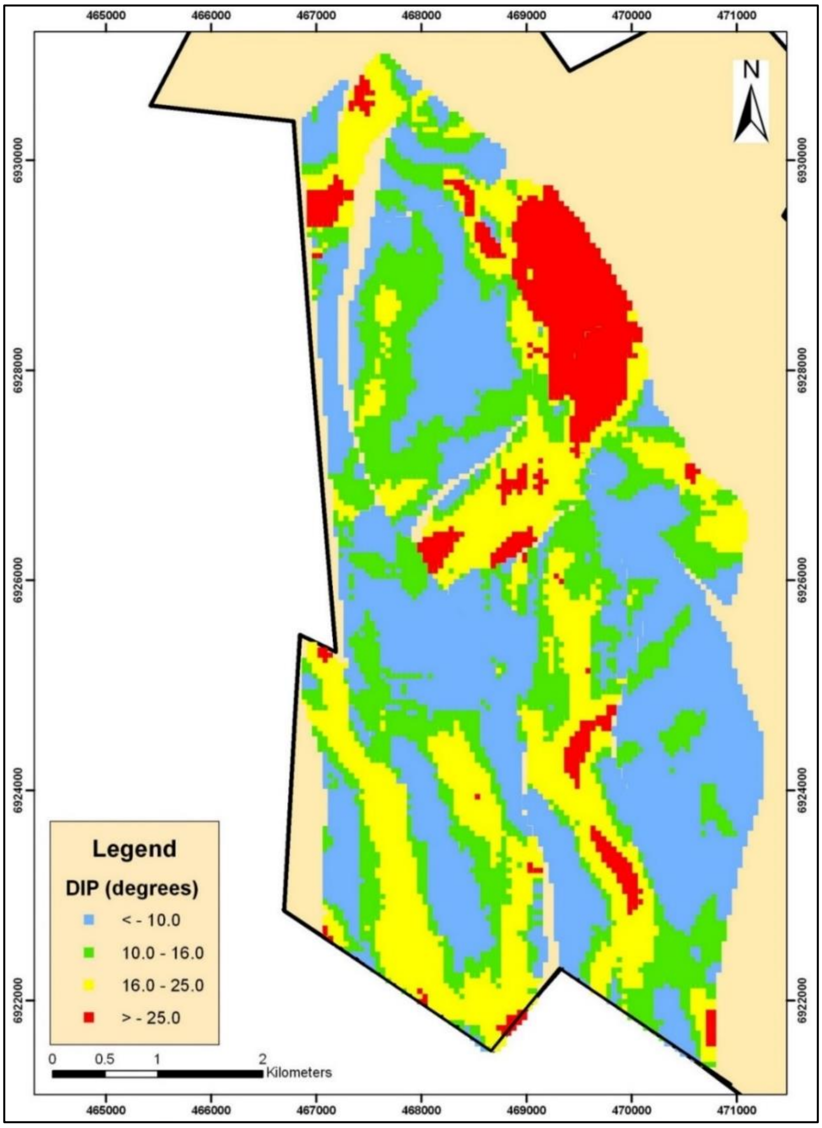
Figure 20: Khaki Shale thickness at Jeanette



3.3 DIP

The results of a 3D seismic survey indicate that the Basal Reef dip within the Project Area ranges from less than 10º in the central section of the study area to more than 25º on the eastern flanks (Figure 21). The area delineated for mine design in the PFS is situated to the west of the steeper reef block areas with dips exceeding 10º.

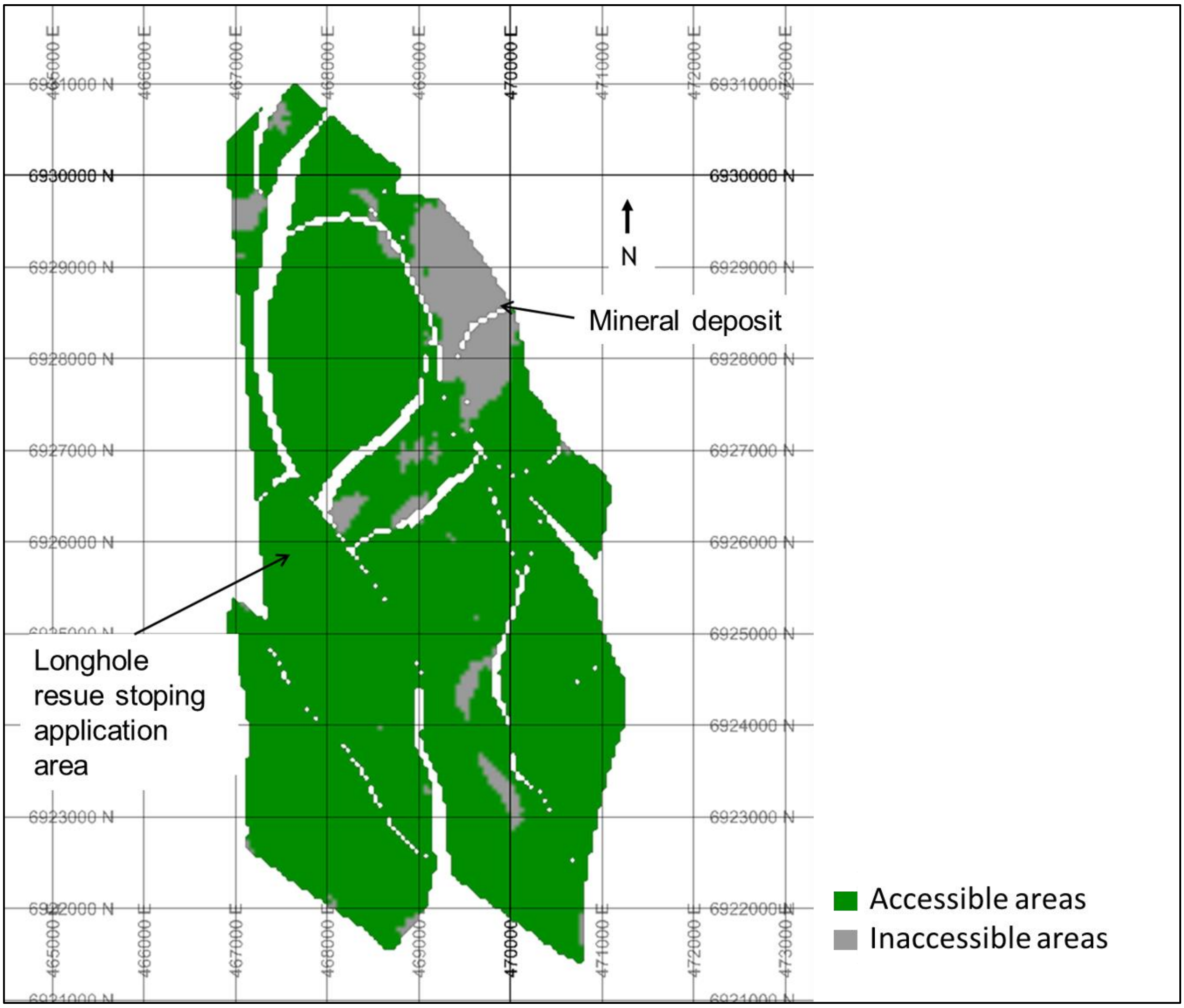
Figure 21: Reef dip (slope) distribution at Jeanette



3.4 APPLICATION AREA OF MECHANISED RESUE STOPING METHODS

Placing ore drives on apparent dip for machine travel and executing longhole stoping on true dip by drilling longholes up-dip, the moderately steeply dipping parts of the deposit can be accessed up to maximum dip of 25 degrees. These operating ranges result in approximately the whole target area being mined with this method (see Figure 22).

Figure 22: Longhole Resue Stoping application area



The Longhole Resue Stoping method can be applied to approximately 91% of the target area.

4 RIB CUTTING RESUE STOPING – DETAIL DESIGN

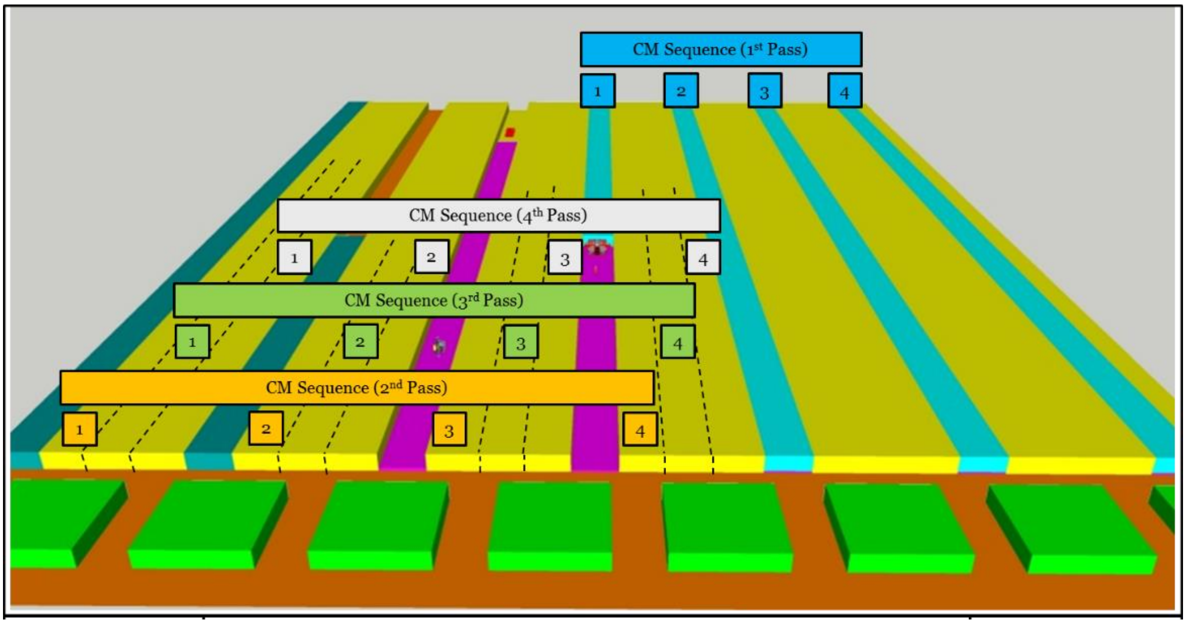
To develop Rib Cutting Resue Stoping as a mining method the following process was followed:

- Selection of mining fleet based on required operating parameters;
- Calculate the mining cycle for various shale thicknesses and back lengths (rib lengths); and
- Determine the optimal stope design based on the best advance rate.

4.1 MINING METHOD OVERVIEW

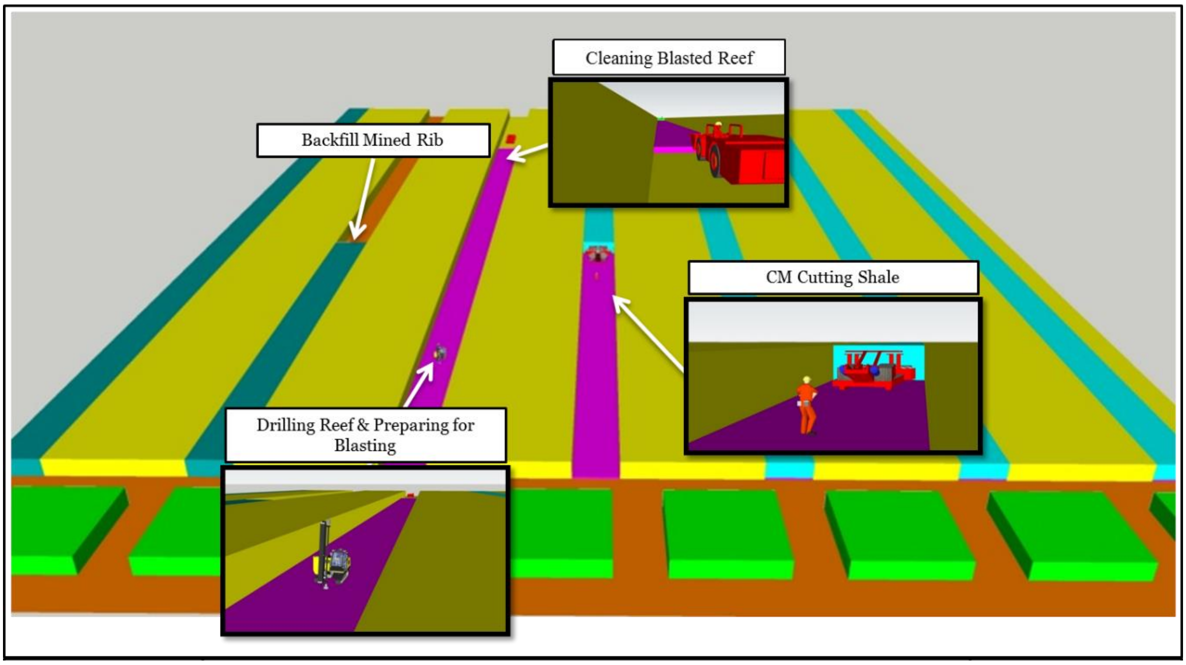
Rib Cutting Resue Stoping removes the shale in a first pass by mechanically cutting the shale using CMs or Roadheaders. The reef is extracted in a separate process behind the cutting machines. These cutting machines are incapable of cutting through the very hard hanging wall quartzites; they can only cut shale. This process is illustrated in Figure 23.

Figure 23: Rib Cutting Resue Stoping – cutting sequence



After cutting away the shale, the reef will be lifted by drilling and blasting. Once the reef is removed, the rib will be backfilled with a cemented shale backfill. The second pass will be done after the backfill has cured for at least 7 days. The different activities for such a mining method are illustrated in Figure 24.

Figure 24: Rib Cutting Resue Stoping – activities



In the following chapters, Rib Cutting Resue Stoping will be developed as a stoping method and discussed in more detail.

4.2 EQUIPMENT SELECTION

Equipment selection includes the fleet required to mine the reef and a fleet to mine the shale. The final fleet selection will apply to a typical mining zone.

4.2.1 SELECTION OF SHALE CUTTER

4.2.1.1 Approach

The equipment considered for this mining method must be based on existing proven technologies. The application of CMs in South African coal mines is widely practiced. Supporting equipment will comprise either shuttle cars, battery haulers or conveyor belts that will transport the shale material to a batch plant. The operating heights and angles of each CM are considered to determine the size of the application area in the mine design. The operating angles of the other mobile equipment must also be considered.

4.2.1.2 Equipment operating ranges

Various types of available cutting equipment are considered for shale cutting. Tests indicate that the amount of energy required to cut the shale is approximately twice that required for coal. No equipment has, however, been ruled out. Equipment modifications are expected to be limited to cutter head design. The ultimate cutting rate will be less than that of cutting coal.

Four equipment suppliers were approached to submit information regarding their mechanized miners; only two companies had equipment that could be used in the specific operating conditions. The two suppliers are regarded as reputable CM or roadheader suppliers globally. They manufacture equipment for different applications, including coal, salt, limestone, gypsum, potash and shale.

Typical cat track grouser bars are placed across the track and positioned to drive the machine in a forward or reverse direction, with little ability to withstand sideways traction. This will be compounded by cutting the shale down to the quartzite contact, where the low friction coefficient between steel and quartzite will exacerbate the sideways sliding. The machines will, therefore, only be used to mine in the dip direction.

An alternative to the track-mounted rock cutter is the rock cutting machine that is staked between the hanging-wall and the footwall. Cutting forces are no longer influenced by the mass of the machine and the footwall contact; they react through the staking system. Machine movement is achieved using a “walking” system where part of the machine is staked and the rest is moved (this process is repeated). The disadvantage of walking systems is that the movement is slow compared to track-mounted machines. Fortunately, in this specific operation, the advance speed is less critical than in coal mining. The major advantage is that they can operate on virtually any dip configuration.

An additional consideration when selecting a rock cutting machine for mining shale is the ability to determine the shape of the excavation. When cutting directly up-dip or down-dip, the excavation shape will be rectangular, assuming a uniform hanging wall and footwall. Any undulations will require a cutting head that will follow the strata. It has been determined that the shale is suitable for the CM, however, the top contact with the shale is quartzite that is not unsuitable for cutting. This rules out the use of a drum-type CM should the hanging wall contact with the shale be undulated. Based on experience at surrounding mines, the shale hanging-wall contact is expected to be uniform. The assumption is that a CM is suitable to create a clean cut against the hanging wall quartzite.

All the cutting equipment considered for application at Jeanette are illustrated in Table 5 and ranked according to percentage application area.

Table 5: Available cutting equipment and their application coverage

Machine type	Height operating range (m)		Cut width (m)	Traction method	Angle capability on strike	Angle capability up dip & down dip	Ability to cut irregular excavations	Application area	Rank
	Min	Max							
Sandvik Continuous Miner MC350	1.2	3.1	3,5	Tracks	8°	14°	No	52%	1
Sandvik Bolter Miner MB230	1.2	2.35	4,7 – 5,2	Track	8°	14°	No	52%	2
Joy Continuous Miner 14HM15	1.3	2.5	2 x 3,6	Tracks	8°	14°	No	49%	3
Joy Continuous Miner 14HM27	1.3	2.2	2 x 3,6	Tracks	8°	14°	No	48%	4
Sandvik Bolter Miner MB250	1.37	2.7	4,7 – 5,2	Track	8°	14°	No	47%	5
Sandvik Bolter Miner MB270	1.5	3.4	4,7 – 5,2	Track	8°	14°	No	43%	6
Sandvik Bolter Miner MB630	1.6	3.2	4,2 – 6,0	Tracks	8°	14°	No	36%	7
Joy Roadheader EBZ55	1.8	3.9	2,5 – 4,9	Tracks	8°	18°	Yes	28%	8
Shanghei Kemei Electromechanical EBZ75	1.8	3.5	<5,2	Tracks	8°	16°	Yes	26%	9
Joy Continuous Miner 12HM31-AAA	1.8	3.2	2 x 3,6	Tracks	8°	14°	No	23%	10
Sandvik Continuous Miner MC250	1.8	3.6	3,5	Tracks	8°	14°	No	23%	11
Joy Roadheader EBZ100E	1.9	4.6	3,2 – 5,3	Tracks	8°	18°	Yes	22%	12
Sandvik Reef Miner MN220	1.1	1.3	4,33	Walking	25°	25°	No	7%	13
Sandvik Bolter Miner MB290	2.1	3.7	4,7 – 5,2	Track	8°	14°	No	5%	14
Sandvik Bolter Miner MB650	2.2	3.6	4,2 – 6,0	Tracks	8°	14°	No	2%	15
Sandvik Roadheader MR340	2.4	4.9	7,4	Tracks	8°	18°	Yes	1%	16
Joy Roadheader EBH200	2.5	4.5	4,0 - 6,4	Tracks	8°	18°	Yes	1%	17
Sandvik Continuous Miner MC470	2.4	5	3,5	Tracks	8°	14°	No	0%	18
Sandvik Bolter Miner MB670	3	4.5	4,2 – 6,0	Tracks	8°	14°	No	0%	19

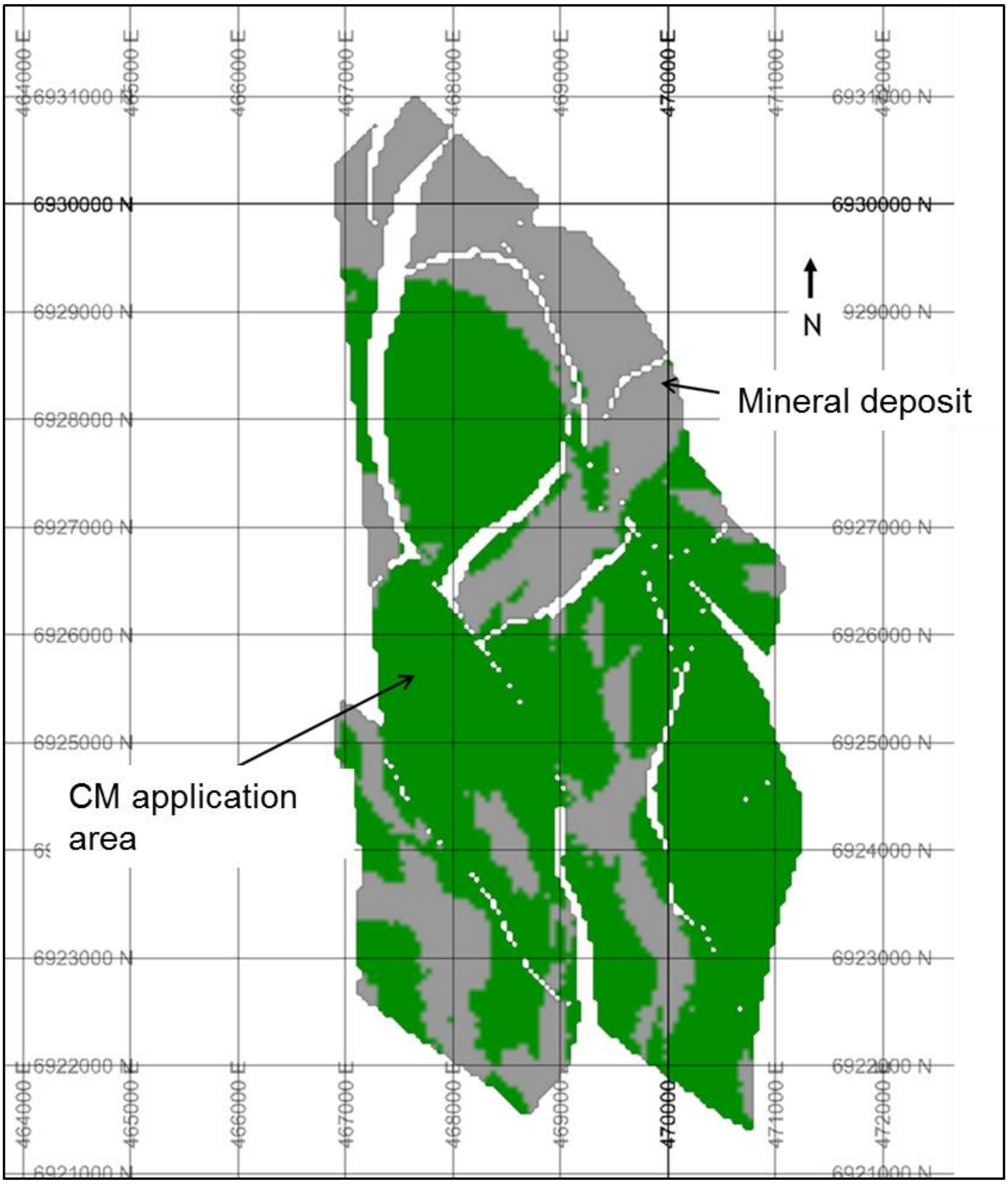
The Sandvik CM and Sandvik Bolter Miner has the highest % application (52%). Combining two types of machines should be considered to increase the overall application area (see Table 6).

Table 6: Cutting machine pairing

Machine type/name	Nr.	% of mineral deposit that is amenable to the specific combination																		
Sandvik Continuous Miner MC350	1																			
Sandvik Bolter Miner MB230	2	52%																		
Joy Continuous Miner 14HM15	3	52%	52%																	
Joy Continuous Miner 14HM27	4	52%	52%	49%																
Sandvik Bolter Miner MB250	5	52%	52%	49%	49%															
Sandvik Bolter Miner MB270	6	52%	52%	50%	50%	47%														
Sandvik Bolter Miner MB630	7	52%	52%	50%	50%	47%	43%													
Joy Roadheader EBZ55	8	57%	57%	54%	54%	52%	47%	41%												
Shanghei Kemei Electromechanical EBZ75	9	55%	55%	52%	52%	50%	45%	39%	28%											
Joy Continuous Miner 12HM31-AAA	10	52%	52%	50%	50%	47%	43%	36%	28%	26%										
Sandvik Continuous Miner MC250	11	52%	52%	50%	50%	47%	43%	36%	28%	26%	23%									
Joy Roadheader EBZ100E	12	56%	56%	53%	53%	51%	46%	40%	28%	28%	27%	27%								
Sandvik Reef Miner MN220	13	57%	56%	56%	55%	55%	50%	44%	36%	33%	31%	31%	29%							
Sandvik Bolter Miner MB290	14	52%	52%	50%	50%	47%	43%	36%	28%	26%	23%	23%	22%	13%						
Sandvik Bolter Miner MB650	15	52%	52%	50%	50%	47%	43%	36%	28%	26%	23%	23%	22%	9%	5%					
Sandvik Roadheader MR340	16	53%	53%	50%	49%	48%	43%	37%	28%	26%	24%	24%	22%	8%	6%	2%				
Joy Roadheader EBH200	17	53%	53%	50%	49%	48%	43%	37%	28%	26%	24%	24%	22%	8%	6%	2%	1%			
Sandvik Continuous Miner MC470	18	52%	52%	50%	48%	47%	43%	36%	28%	26%	23%	23%	22%	8%	5%	2%	1%	1%		
Sandvik Bolter Miner MB670	19	52%	52%	49%	48%	47%	43%	36%	28%	26%	23%	23%	22%	7%	5%	2%	1%	1%	0%	
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
		Sandvik Continuous Miner MC350	Sandvik Bolter Miner MB230	Joy Continuous Miner 14HM15	Joy Continuous Miner 14HM27	Sandvik Bolter Miner MB250	Sandvik Bolter Miner MB270	Sandvik Bolter Miner MB630	Joy Roadheader EBZ55	Shanghei Kemei Electromechanical EBZ75	Joy Continuous Miner 12HM31-AAA	Sandvik Continuous Miner MC250	Joy Roadheader EBZ100E	Sandvik Reef Miner MN220	Sandvik Bolter Miner MB290	Sandvik Bolter Miner MB650	Sandvik Roadheader MR340	Joy Roadheader EBH200	Sandvik Continuous Miner MC470	Sandvik Bolter Miner MB670

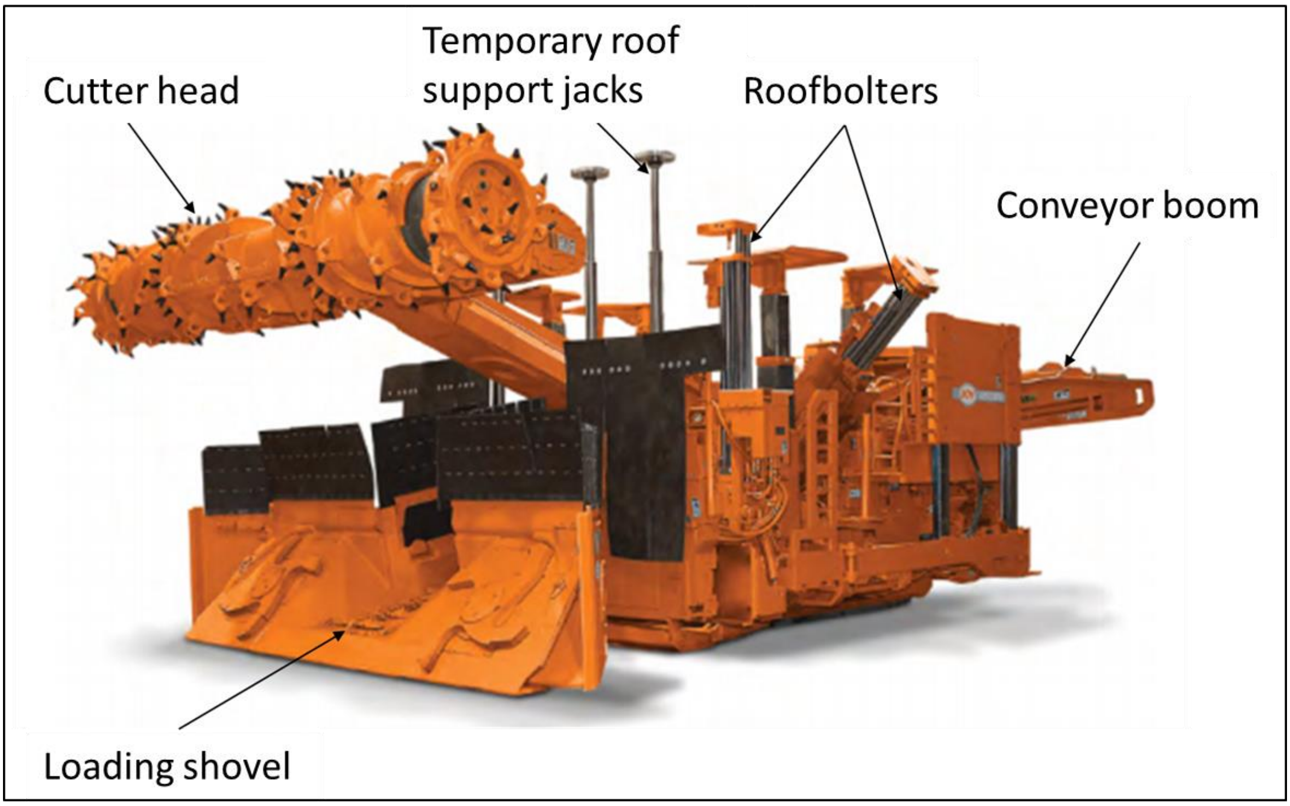
An additional 4.8% of the total mineral deposit can be extracted when implementing a second machine. This is only a marginal improvement and the logistical complications of moving the machine between work areas will be cumbersome. Using the Sandvik Bolter Miner MB230 will provide an adequate extraction rate and the bolting capabilities of the machine is considered an added benefit. The application area for this machine is illustrated across the mineral deposit in Figure 25.

Figure 25: CM operating ranges across the mineral deposit (52%)



The basic components of the bolter miner are illustrated in Figure 26.

Figure 26: Basic components of a Bolter Miner – Source: (Joy Global, 2016)



4.2.2 SHALE HAULING

Shale hauling from the CM to the backfill plant can be done with either a conveyor train, shuttle car or battery hauler. The only conveyor train currently available on the market is the Joy 4FCT. According to the specifications provided by the equipment supplier, the minimum operating height for the conveyor is 1.4 m. In addition to higher required operating height, the economic production rate for the conveyor train is 300 ktpm or more, approximately 10x more than the anticipated production rate per mining zone. Electric shuttle cars and battery haulers were the only other types of equipment considered suitable for the operation. The available units on the market are summarised in Table 7.

Table 7: Available hauling equipment

	Sandvik LS195	Joy 21SC04	Joy BH-10	Comment
Operating dip (Deg)	12	12	12	In direction of travel
Min height (m)	1.7	1.2	1.2	According to supplier
Load capacity (t)	10	8	11.4	Supplier info
Drive	Diesel	Electric cable	Battery	

The two Joy alternatives are within the same operating heights of the bolter miner. The shuttle car is connected to a gully box with an electric power cable. Although the cable length is not specified by Joy, cable lengths are typically up to 300 m. The length is limited by the space available on the cable reel. If the mine can supply a higher voltage to the shuttle car, a thinner cable can be used that will use less space on the cable reel. The shuttle car is preferred over the battery hauler in the coal industry mainly due to its lower operating cost and purchase price (Holtzhausen, 2014). However, the battery hauler is more flexible.

4.2.3 FINAL FLEET SELECTION

Based on operating heights and to achieve the calculated rate, the required fleet per mining zone is illustrated in Table 8.

Table 8: Fleet selection summary

Shale cutting	Specifications	Drive	Nr per mining zone
Bolter miner	1.6 m operating height	496kW installed power, 1,000V	1
Shuttle car	1.8 m operating height, 11.4 t payload	150kW electric motor	2
Reef extraction			
Development drill rig	Single boom, 1.8 m operating height	40kW diesel engine, 45kW electric motor	1
LHD	1.8 m operating height, 9.6 t payload	170kW diesel engine	1
Water jet		37kW electric motor	1
Waste development			
Development drill rig	Single boom, 1.8 m operating height	40kW diesel engine, 45kW electric motor	1
LHD	1.8 m operating height, 9.6 t payload	170kW diesel engine	1
Bolter	Single boom, 1.8 m operating height	75kW diesel engine, 30kW electric motor	1
Dump truck	20 t payload, 2.3 m operating height	240kW diesel engine	2
Charging unit	Scissor lift for emulsion charging		1

4.3 MINING CYCLE

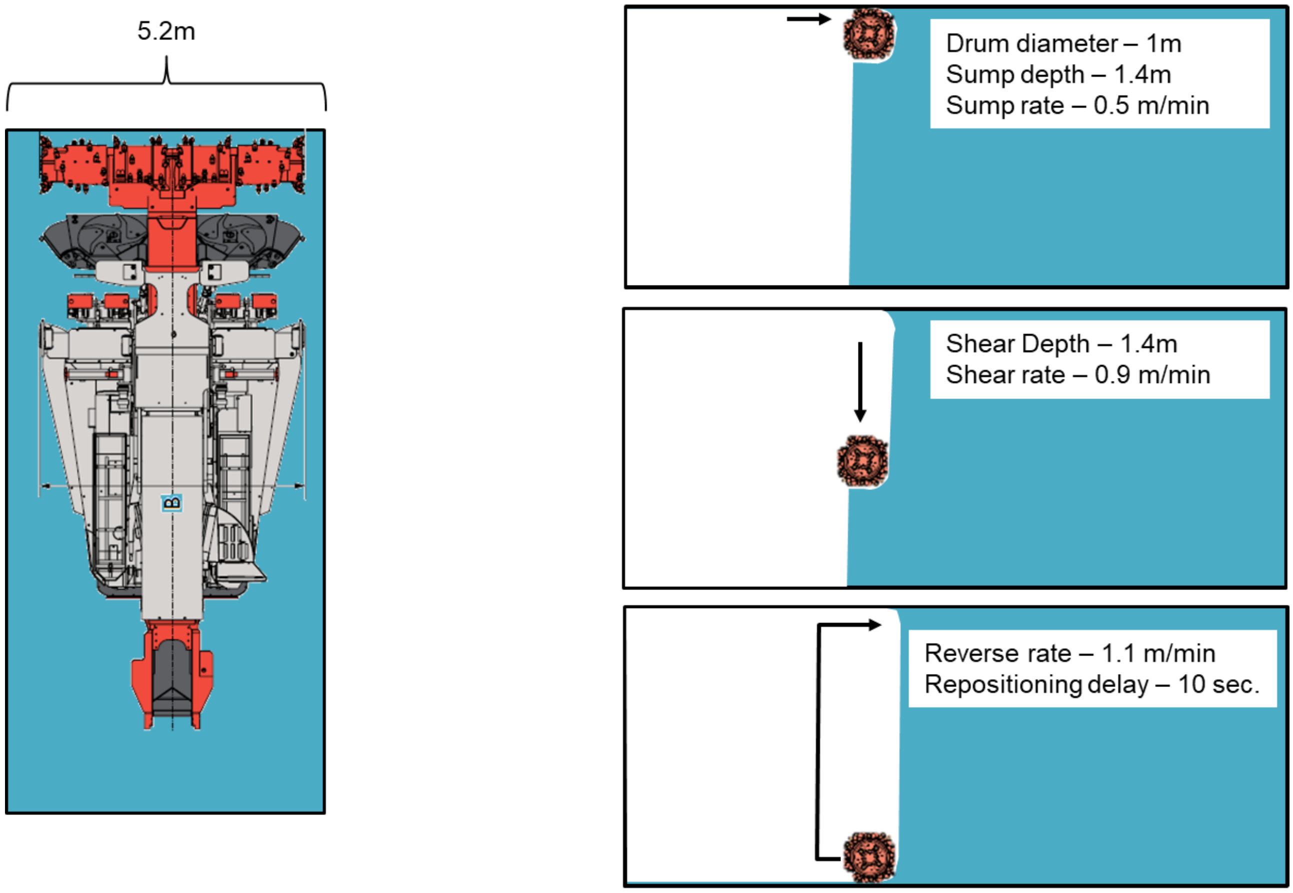
To calculate the production from a mining zone, each activity in the mining cycle should be analysed to estimate the impact it will have on the mining rate. Rib Cutting Resue Stoping consists of a separate shale cutting and reef extraction cycle that will occur in separate locations, resulting in largely independent activities. The shale extraction cycle was selected as the rate driver for the mining zone. This implies that the reef extraction should match or exceed the shale cycle extraction rate.

4.3.1 SHALE EXTRACTION CYCLE

4.3.1.1 Cutting cycle

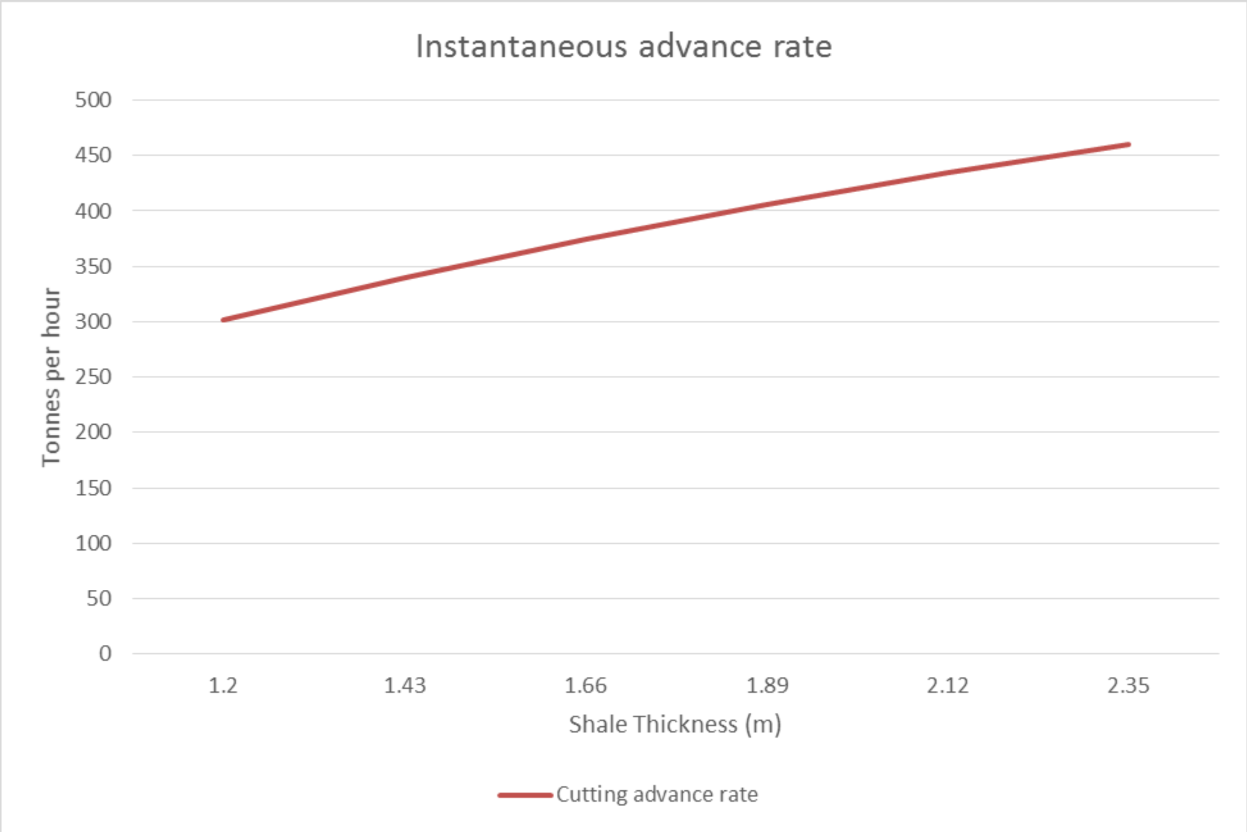
During the cutting cycle (see Figure 27), the CM will cut away the overlaying Khaki Shale and deliver the broken material to the waiting shuttle car behind the machine. The cutting rate was estimated by applying a downward shearing motion to account for the hardness of the shale compared to coal. The rates used in the illustration were obtained from original equipment manufacturers (“OEMs”).

Figure 27: Cutting cycle parameters – shale



The instantaneous advance rate was calculated for different shale thicknesses (see Figure 28). The rate calculated represents the cutting rate of the CM and excludes waiting time for support installation, shuttle car movement of the machine between ribs or any downtime. The sump rate used is half of that normally used in coal cutting rate calculations.

Figure 28: CM instantaneous advance rate



4.3.1.2 Support cycle

The support cycle will occur concurrently with the cutting cycle due to the use of a bolter miner. The support cycle was calculated to determine if it will delay the cutting cycle. The required support installation was determined by Open House Management Systems ("OHMS") and is as follows:

- 2.4 m long tendons
- 1.5 m x 1.5 m pattern
- Four bolts

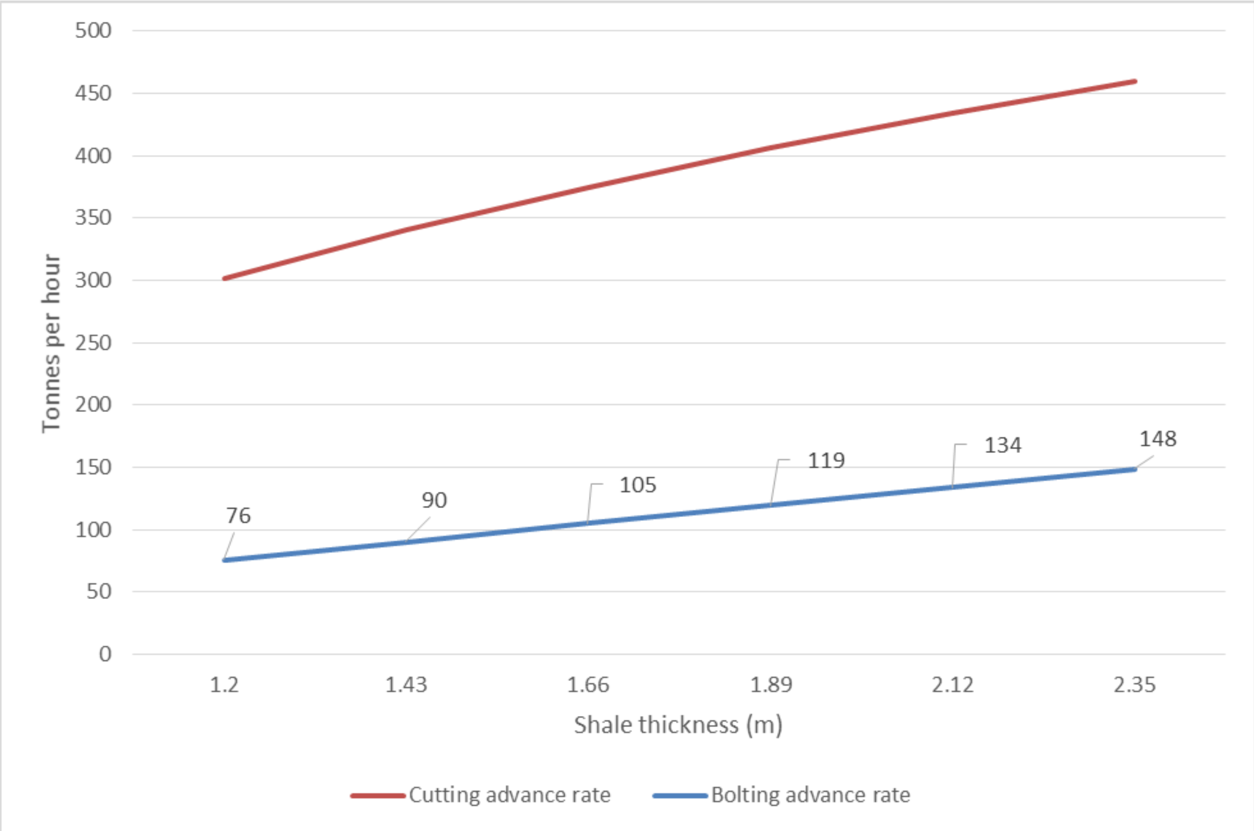
Based on the above criteria, a minimum of 7 bolts will be installed at every 1.5 m face advance and 4.6 per linear metre advance. The summary of the calculation is illustrated in Table 9.

Table 9: Support cycle installation rate calculation

Support cycle time			Comment
Hole depth	m	2.40	Required bolt length
Number of holes per face	holes	8	
Setup at face	min	0.00	CM positioned at face
Movement to next hole	min	0.75	
Penetration rate - per drill	m/min	2.40	
Single/double boom	No.	2.00	4 Booms on machine
Double boom factor	%	100%	
Penetration rate - per drill	m/min	2.40	Typical penetration rate into Quartzite
Install roof bolt (spin - hold - tension)	min	3.25	
Time per bolt	min	5.00	
Bolts installation rate across 2 booms	Bolts/min	0.40	
Installed bolts per hour	bolts/hr	24.00	Typically, 11 bolts per hr for single boom
Installation rate (per metre advance)	m/hr	4.50	Effective advance rate for bolting cycle

The tonnes per hour rate at various shale thicknesses is illustrated in Figure 29.

Figure 29: Instantaneous advance rate, bolting cycle and cutting cycle



The bolt installation rate across the different shale thicknesses will be consistent due to the same number of bolts that must be installed per linear metre. The required rock support will decrease the effective advance rate of the CM.

4.3.1.3 Shale hauling cycle

The selected shuttle car has the following operating parameters based on manufacturer specifications:

- Machine width – 3.2 m
- Machine length – 8.3 m

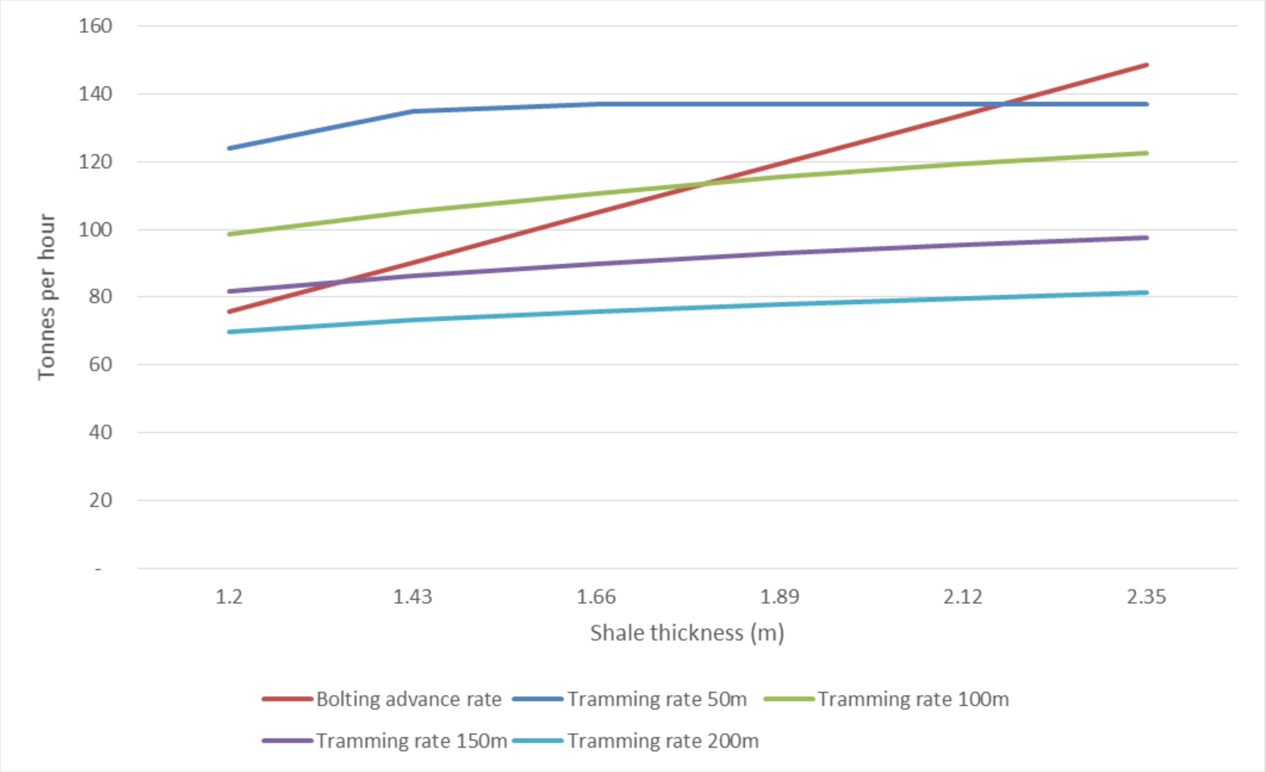
At least two shuttle cars must be in operation per mining zone to improve CM utilisation. The CM will cut a rib 5.2 m wide, allowing only one shuttle car to travel at a time. One car will have to wait at the entrance of the rib for the second car to load and transport shale out of the rib. The cycle calculation is illustrated in Table 10.

Table 10: Cleaning cycle at short rib tramming distances (50 m)

Cleaning cycle time			**Comment**
Number of cars per rib	**Nr.**	1.00	Limited space in rib restricts movement to one car only
Tramming capacity	**t**	8.00	Rated load capacity
Tramming speed	km/hr	6.00	According to Joy Global
Tramming speed	m/min	100	
Tramming distance	m	50	This is the route from the CM face to the strike drive at the bottom only and back. Will vary as rib becomes longer.
Cutting rate	t/hr	263	The CM will be able to install support whilst waiting for the shuttle car. A 50% rate split was assumed between the instantaneous cutting cycle and the bolting cycle rates.
Loading time of shuttle car while CM is cutting	min	1.83	Including bolting delay
Shuttle car positioning time at CM	sec	20	
Shuttle car loading duration	min	2.16	
Tramming delay from CM to bottom of rib	min	1.00	Allow for tramming time in and out of rib only
Duration of tramming and loading cycle down rib	**min**	**3.16**	
Tramming distance to tip	m	100	The distance to the backfill plant, 200 m max from the furthest active rib
Tipping delay	min	1.50	
Tramming time to tip and back	**min**	**3.50**	
Maximum cycle time between two shuttle cars	**min**	**3.50**	
Effective tramming rate	**t/hr**	**137**	
CM idle time	**min**	**2.68**	**CM will wait for empty shuttle car**
CM utilisation	**%**	**38%**	

The effective rate of mining the rib at different shale thickness and shuttle car tramming distances is illustrated in Figure 30.

Figure 30: Effective CM system advance rate



4.3.2 BACKFILL

The required backfill system was developed to a conceptual phase. The system is required to be a continuous one that follows the CM. The importance of the backfill system is twofold: to reduce the amount of Khaki Shale to be hoisted and provide support in the mined-out areas. The type and strength of backfill that can be created by using the Khaki Shale was tested at the University of Pretoria's civil labs. These tests provided input into the backfill plant design. The backfill plant will have no impact on the mining cycle, except for system downtime that was assumed at 90%.

The mined shale will be tipped by the shuttle cars into a feeding bin at the front end of the backfill plant. The material will be crushed and mixed with cement and water; this mixture will be pumped into mined-out ribs from the up-dip side. A wall will be constructed at the top and bottom of each rib to contain the backfill. The backfill will have a 28-day curing time before the rib directly adjacent can be extracted.

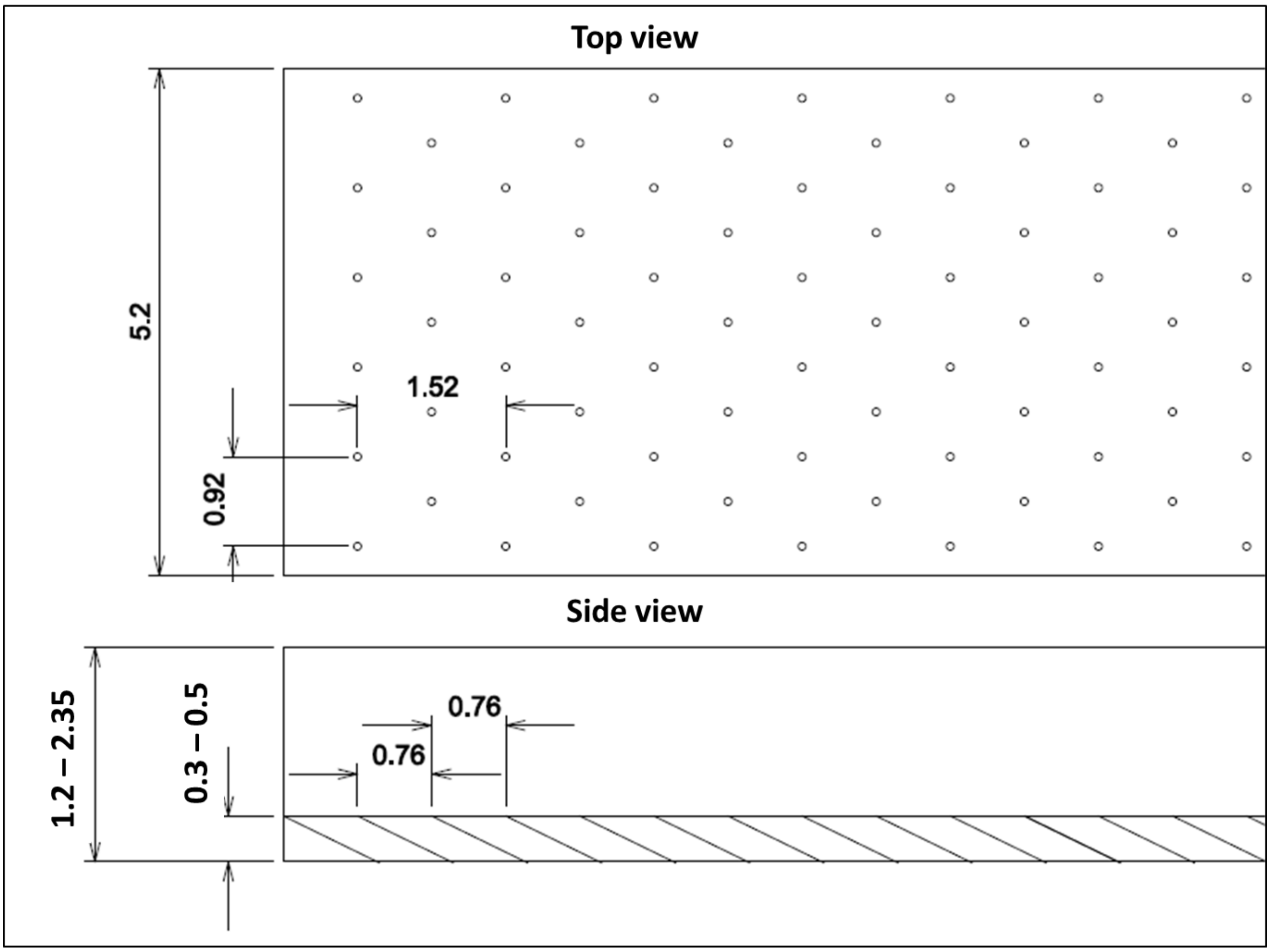
4.3.3 REEF EXTRACTION

Reef extraction includes the removal of the quartzite beam and the Basal Reef. The preferred system should achieve the same advance rates as the CM. The CM is designed to cut rock with a UCS of up to 70 MPa. The quartzite and Basal Reef typically fall out of this range with UCS values of more than 120 MPa. Providing a second machine, such as a Roadheader, to extract the reef is unsuitable, due to the operating angles and the abrasiveness of the quartzite. As such, drilling and blasting is proposed.

4.3.3.1 Drilling

Drilling will be done from the top down at an angle with a drill rig. The drilling pattern is illustrated in Figure 31. Drilling the reef from the top will have a production advantage over drilling the reef from the side. Drilling can be completed for the whole rib at once and blasting can follow directly afterwards. The mining rate for reef extraction will not be determined as it is safe to say that it will not constrain production rates. Should this become a constraint, a second or even a third drill rig can be added to speed up the drilling cycle.

Figure 31: Footwall ripping – blast design



4.3.3.2 Cleaning

Once blasted, cleaning can be done with a combination of water jets and LHD. LHD loading and water jet finish will have a perfectly clean footwall as a result with minimal gold losses. Once water jetting is complete, backfill can be done.

4.4 MINING METHOD LAYOUT

4.4.1 BACK LENGTH

A mining zone will progressively move along the strike of the orebody to allow for near-flat access by mobile equipment. The back length represents the distance along each mining zone's dip. A longer back length will reduce the time spent by equipment moving between ribs; more time will be spent mining each rib, resulting in a higher production rate. However, the production rate will be negatively influenced by excessive tramming distances. A shorter back length will result in a higher advance along strike and will require more frequent backfill plant movements. The optimal back length will have the best balance between tramming time and backfill plant movements.

The cycles calculated in Section 4.3.1 do not allow for delays due to the movement of the bolter miner between ribs and the movement of the backfill plant. Movement of the backfill plant will result in stoppage of the shale mining process. The parameters used for the delay between ribs are illustrated in Table 11.

Table 11: Backfill plant movement parameters and CM tramming speed

Backfill plant		
Backfill plant movement delay	days	5
Backfill plant movement frequency	m	100
Effective delay per rib	**hr/rib**	4.23
CM travelling speed	km/hr	0.50
CM rigging down	min	20
CM set-up at face	min	20

The CM travelling duration was calculated based on different back lengths; the effective delay per rib is illustrated in Table 12. The duration includes the movement along strike between the ribs.

Table 12: CM travelling duration

Back length	Min
50.00	48
100.00	54
150.00	60
200.00	66
250.00	72
300.00	78

The t/hr rate as calculated for the CM was calculated over different back lengths and shale widths. This rate was converted to an hr/rib rate excluding the CM movement and backfill plant movement delays. In the third table the hr/rib was calculated, including the CM and backfill plant movement. The resultant t/hr were calculated for the different shale thickness and back lengths in the last table.

Table 13: Back length calculation

t/hr - Effective cutting		Shale thickness (m)				
		1.2	**1.5**	**1.8**	**2.1**	**2.35**
Back length (m)	**50**	75.82	94.77	113.72	132.68	137.14
	100	75.82	94.77	113.65	119.00	122.74
	150	75.82	87.50	91.90	95.36	97.74
	200	69.82	74.01	77.13	79.56	81.21
	250	60.95	64.12	66.45	68.25	69.46
	300	54.08	56.57	58.37	59.75	60.68

Hr/rib – Effective cutting		Shale thickness (m)				
		1.2	1.5	1.8	2.1	2.35
Back length (m)	50	11.11	11.11	11.11	11.11	12.03
	100	22.22	22.22	22.24	24.78	26.88
	150	33.33	36.10	41.25	46.38	50.63
	200	48.26	56.91	65.53	74.12	81.26
	250	69.10	82.11	95.08	108.01	118.76
	300	93.46	111.69	129.89	148.04	163.13

Hr/rib – All-inclusive		Shale thickness (m)				
		1.2	1.5	1.8	2.1	2.35
Back length (m)	50	14.02	14.02	14.02	14.02	14.94
	100	25.23	25.23	25.25	27.79	29.89
	150	36.44	39.21	44.36	49.49	53.74
	200	51.47	60.12	68.74	77.33	84.47
	250	72.41	85.42	98.39	111.32	122.07
	300	96.87	115.10	133.30	151.45	166.54

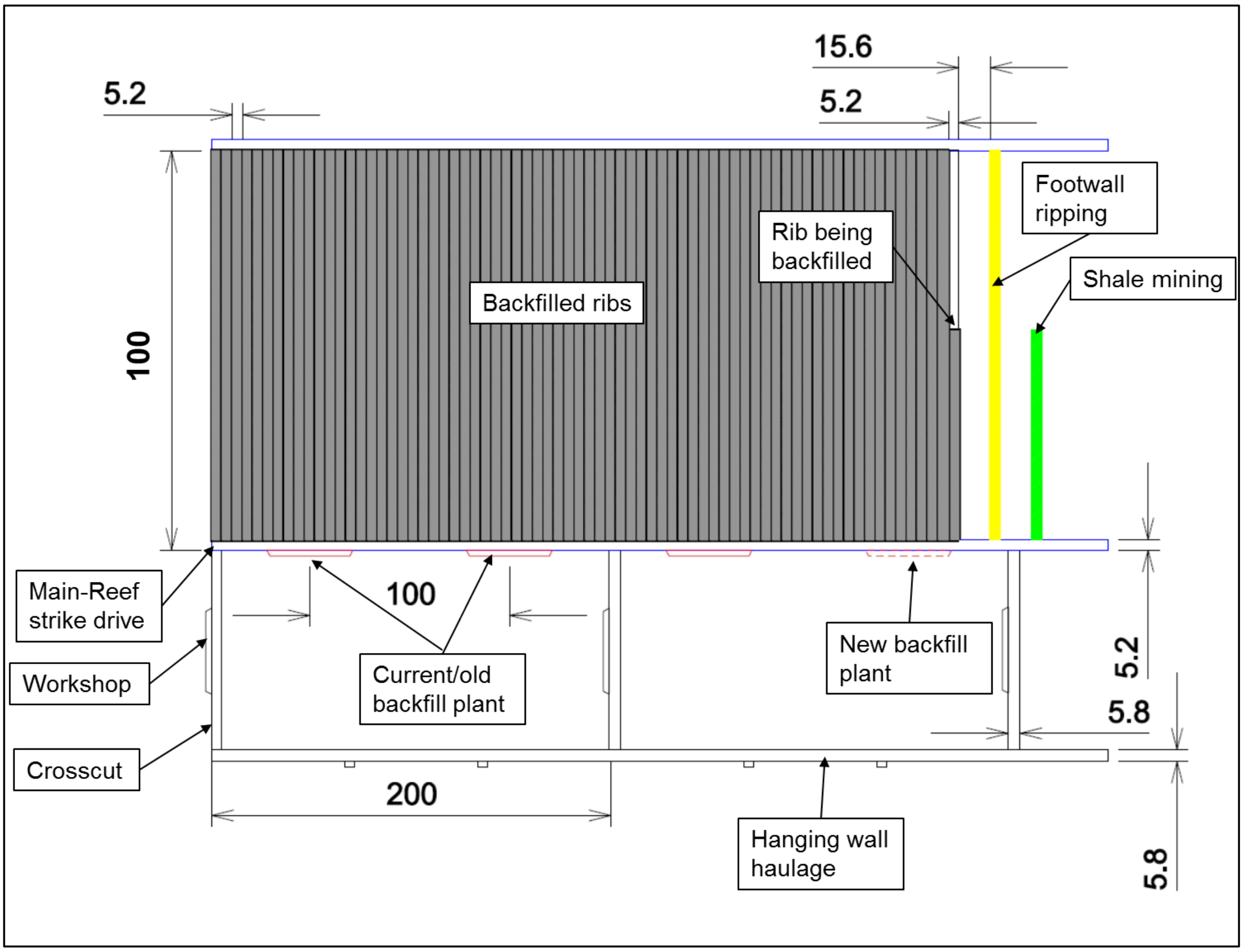
t/hr – All-inclusive		Shale thickness (m)				
		1.2	1.5	1.8	2.1	2.35
Back length (m)	50	60	75	90	105	110
	100	67	83	100	106	110
	150	69	81	85	89	92
	200	65	70	74	76	78
	250	58	62	64	66	68
	300	52	55	57	58	59

It is evident that the optimal back length is between 50 and 150 m long. A 100 m back length was selected as a good design average.

4.4.2 MINING METHOD LAYOUT

With the back length known, the rest of the mining layout was determined. Based on best practice, the distance between crosscuts was selected at 200 m. The tipping point will advance along with stoping in the on-reef drive, hence the tramming distance from the workplace to the belt can be kept to a minimum. The layout for the Rib Cutting Resue Stoping is illustrated in Figure 32.

Figure 32: Rib Cutting Resue Stoping layout



4.5 Mining Rate Estimation

4.5.1 Cutting Rate

Based on the 100 m strike distance, the cutting rate will range, depending on the shale thickness. The cutting rate for the different shale thicknesses are illustrated in Table 14.

Table 14: CM cutting rate estimation

	Shale thickness (m)				
	1.2	**1.5**	**1.8**	**2.1**	**2.35**
t/hr	67	84	100	106	110
m/hr	3.96	3.96	3.96	3.60	3.35
m/month	1,932	1,932	1,931	1,754	1,631
m^2/month	10,047	10,047	10,041	9,123	8,481

4.5.2 Navigating Faults

The CM is only capable of cutting shale. Should there be an occurrence of frequent minor faulting, the CM would have to be removed and drilling and blasting be utilised until the CM can continue. Lorraine (current target mine) is relatively free of minor faults and since it is adjacent to the Project, it is assumed that similar conditions will exist there (Bryson, 1958).

5 DILUTION AND AMOUNT OF SHALE HOISTED

Based on the mining layouts for the two mining methods, a dilution calculation was done for both (see Table 15). The fundemantal difference in dilution between the two methods can be attributed to the following:

- Expected inaccurate drilling with the longhole drill rig
- Additional footwall overbreak in strike drives and raises, vital for Longhole Stoping equipment access
- The removal of shale with the CM will be clean, due to ease of cutting shale only
- Footwall ripping can be more controlled due to short holes that are drilled

Table 15: Dilution comparison between Longhole Stoping and Rib Cutting Resue Stoping

Dilution factors			
Common area	**Unit**	**CM**	**Longhole**
Back length	m	**100**	**250**
Raise spacing	m	**200**	**200**
Area	m^2	20,000	50,000
SG	kg/m^3	2.81	2.81
Stoping width			
Shale	m	**1.56**	**1.56**
Reef	m	**0.31**	**0.31**
Combined	m	1.87	1.87
Volume			
Shale	m^3	31,200	78,000
Reef	m^3	6,200	15,500
Combined	m^3	37,400	93,500
Tonnage			
Shale	t	87,672	219,180
Reef	t	17,422	43,555
Combined	t	105,094	262,735
Tonnage dilution factors	**Unit**	**CM**	**Longhole**
ASG footwall			
ASG width	m	**0.00**	**4.50**
ASG in footwall overbreak	m	**0.00**	**0.60**
ASG overbreak volume	m^3	0	540
ASG overbreak tonnes	t	0	1,517
Face/panel length	m	**5.2**	**30**
Number of panels per back length	No.	19	7
Total ASG overbreak tonnes	t	0	10,622
Raise footwall			
Raise width	m	**0.00**	**6.50**
Raise in footwall	m	**0.00**	**0.20**
Raise volume	m^3	0	325
Raise tonnes	t	0	913
Panel footwall overbreak			
Overbreak	cm	**10.00**	**20.00**
Overbreak volume	m^3	2,000	10,000
Overbreak tonnes	t	5,620	28,100
Strike drives			
Width	m	**6.50**	**6.50**
Strike development in footwall	m	**0.10**	**0.10**
Volume	m^3	260	260
Tonnes	t	731	731
No. strike dev per area	No.	**1.00**	**1.00**
Total tonnes	t	731	731
Tonnage dilution			
Dilution tonnage	t	6,351	40,366
Area tonnes	t	105,094	262,735
Total tonnage	t	111,445	303,101
Dilution	**%**	**6.0%**	**15.4%**
Footwall dilution % (ore)	**%**	**36.5%**	**92.7%**

According to Longhole Stoping tests that were conducted, drilled holes are typically straight. However, setting-up the machine to ensure that the holes are being drilled in the correct direction is more difficult (van Dorssen, et al., 2002, p. 153). New technologies are availible that improve the accuracy of longhole drilling (Ferreira, 2003). To accurately drill the Basal Reef at a thickness of only 0.5 m to 0.3 m can still be a challenge, hence shale dilution and reef loss or overbreak is expected.

Longhole drilling stope layout consists of strike drives and raises. In steeper dipping areas, more than 11º, the raises should be developed on apparent dip. The footwall of the raises should be blasted level so that equipment can tram in the raises. The same applies to the strike drives and will result in additional footwal overbreak.

The operating parameters of the CM prevent it from cutting into the footwall of hanging wall quartzite and operation of the CM would be easy to prevent cutting into the hanging wall or footwall; limited shale dilution is expected. The CM layout does not require any footwall overbreak for gullies and drives. Footwall overbreak during reef extraction is expected to be less due to better control over the drilling of shorter holes. Water jet cleaning can be applied to both mining methods, hence removal of blasted reef would be expected to be similar. The amount of shale that is expected to be hoisted for each method is calculated; Table 16 illustrates how much shale can be stowed or packed by each method.

The fundamental difference between the two methods in terms of shale stowing can be attributed to:

- The CM delivering all the shale being cut to the backfill plant where it will be piped to the mined-out rib.
- The placing of shale backfill in the ribs to a near 100% fill to the hanging wall.
- Effective throw-blasting and dozing of the shale into the back area which will not result in a 100% fill to the hanging wall.
- Considering the layouts, the Rib Cutting method will have a greater portion available for backfill placement. The shale mined during gully and strike drive development will have to be hoisted.

Table 16: Shale stowing calculation

Shale stowing			
Common area	**Unit**	**CM**	**Longhole**
Width	m	100	200
Length	m	250	250
Area	m^2	25,000	45,000
Face length	m	5.2	20.00
Shale width	m	1.60	1.60
Shale SG	kg/m^3	2.83	2.83
Reef width	m	0.35	0.35
Reef SG	kg/m^3	2.70	2.70
Reef development			
Raise		-	-
Raise width	m	-	4.50
Raise length	m	-	360
Raise area	m^2	-	1,620
Gully			
No. faces	No.	-	4
Gully width	m	-	4.50
Gully length	m	-	180
Gully area	m^2	-	810
Total gully area	m^2	-	3,240
Strike development			
No. strike drives	No.	2	4
Strike development width	m	5.20	4.50
Strike development length	m	100	180
Strike development connections	m	70	138
Strike development area	m^2	883	1,433
Total strike development	m^2	1,766	5,732
Production area			
Development area	m^2	1,766	10,592
Stoping area	m^2	23,234	34,408
Total area	m^2	25,000	45,000
Created volume			
Development			
Shale	m^3	2,825	16,948
Reef	m^3	618	3,707
Stoping			
Shale	m^3	37,175	55,052
Reef	m^3	8,132	12,043
Shale broken volume			
Bulking factor	%	160%	160%
Stoping	m^3	59,480	88,084
Development	m^3	4,520	27,116
Shale broken SG	kg/m^3	1.8	1.8
Total shale broken volume	m^3	64,000	115,200
Shale stowing (stoping only)			
Stoping created volume	m^3	45,307	67,095
Packing factor	%	100%	95%
Backfillable volume	m^3	45,307	63,740
Shale stowed - Stoping	m^3	45,307	63,740
Shale removed - Stoping excess	m^3	14,173	24,343
Shale broken volume - Stoping	m^3	59,480	88,084
Volume removed			
Shale	m^3	18,693	51,460
Reef	m^3	8,750	15,750
Tonnage removed			
Shale	t	33,063	91,019
Reef	t	23,625	42,525
Total tonnage	**t**	**56,688**	**133,544**
% shale stowed	%	71%	55%

The gold head-grade is compared in Table 17:

Table 17: Gold head grade Comparison

Shale Stowing			
Common Area	**Unit**	**CM**	**Long Hole**
Gold Grade			
Grade	cm.g/t	950	950
Channel Width	cm	35	35
Broken Grade	g/t	27.14	27.14
Broken Gold Content	kg	641	1,154
Hoisted Grade	**g/t**	**11.31**	**8.64**

5.1 MINOR FAULTS

Minor faults are defined as faults with throws of 1 m or less. Based on discussions with individuals with experience on surrounding mines and according to literature, minor faults experienced in the Basal Reef rarely affected the quartzite hanging wall above the shale (Bryson, 1958, p. 4). The shale layer seems to absorb the impact of these minor faults, whilst forming, and prevents the breaking of the hanging wall quartzite beam. This is expected to be even more pronounced in areas with thick shale. This implies that the hanging wall quartzite is relatively smooth and continuous which will greatly assist in CM operation.

5.2 OTHER FACTORS

Remaining factors, such a fault losses, fault dilution, reef loss and mine call factor ("MCF") would affect both mining methods in the same way and although these factors must be considered a mineral reserve, they are not detailed in this study.

6 CONCLUSIONS

Conventional resue stoping methods were abandoned in the past for other more productive and viable stoping methods, such as open stoping. In cases where the shale is very thick, such as for Jeanette, open stoping is not viable, due to a large amount of dilution. This resulted in large portions of Basal Reef being left unmined, such as the Basal Reef at Jeanette.

Longhole drilling and blasting is currently perceived as the preferred stoping method for the Project. This method requires the drilling of longholes into the shale. Once the holes are drilled, a plastic tube must be inserted into the shale to prevent the hole from collapsing. The shale will be cast-blasted into the mined-out area behind the advancing face. Alternatively, a CM can be used to remove the shale above the reef. This is a continuous process where the shale is being cut by the CM, transported by shuttle cars and tipped into a backfill plant. Backfill is created and pumped back into the extracted ribs.

Longhole drilling can use mining on apparent dip angles, hence the method can be used to extract up to 91% of the available ore deposit. Available coal mining equipment can be used to extract the shale of approximately 52% of the available deposit area. Longhole Stoping has an estimated rate of approximately 3,500 m^2/month over a back length of 200 m.

The mining rate for the CM method is significantly higher at approximately 10,000 m^2/month over a back length of 100 m. The Khaki Shale is a soft, broken rock that lends itself to self-mining once exposed (Bryson, 1958). The importance of cutting the ribs rather than blasting raises is considered an important stability consideration. Rib Cutting Resue Stoping indicates a reduction in waste overbreak during stoping and improved shale stowing.

7 RECOMMENDATION

The use of a CM in the Project area based on mining conditions from the surrounding area makes the method a viable option. The absence of minor faulting along with the shale’s susceptibility to cutting will make the CM ideal to extract the shale quickly and continuously. Improved sidewall stability can be expected due to the cutting action of the CM.

Longhole Resue Stoping can be used on apparent dip by developing a raise on apparent dip so that trackless equipment can still travel up and down the raise as required. The CM can only be used to extract 52% of the mineral deposit. The use of the Longhole Resue Stoping will still be required for near full extraction of the deposit. Rib Cutting Resue Stoping should be considered for application in suitable areas due to the improved mining rates and sidewall stability.

8 BIBLIOGRAPHY

Amadeo, K., n.d. *US Economy.* [Online]
Available at: http://useconomy.about.com/od/inflationfaq/a/US-Inflation-Rate.htm
[Accessed 30 June 2014].

Bishop, A., n.d. *Investec.* [Online]
Available at: https://www.investec.co.za/research-and-insights/economy/economic-research/price-inflation/cpi-outlook.html
[Accessed 30 June 2014].

Bock, I., 1996. Selective blast mining in gold mines. *The Journal of The South African Institute of Mining and Metallurgy,* pp. 183 - 186.

Bock, I., Jagger, L. & Robinson, R., 1998. An economic model for gold and platinum mining using selective blast mining. *The Journal of The South African Institute of Mining and Metallurgy,* pp. 75 - 84.

Bryson, R., 1958. Resue Stoping at Loraine Gold Mines Ltd.. *Association of Mine Managers,* p. 15.

Bryson, R. H., 1958. Resue Stoping at Loraine Gold Mine. *SAIMM*, 23 May, pp. 401 - 504.

Crushed Wonderstone Shale Compression Behaviour (n.d.) Erhan .

De Jager, E. J., 1996. *Mine Call Factor. Unaccounted for Gold, was it there in the first instance?.* [Online]
Available at: http://www.ammsa.org.za/
[Accessed 13 November 2016].

Ferreira, P., 2003. Improved technologies in longhole blast hole drilling, applied to dropraising and longhole stoping as well as the application of a small twin boom mechanized drillrig. *SAIMM,* pp. 233 - 240.

Filen, C., 2012. *Mineweb.* [Online]
Available at: http://www.mineweb.com/archive/great-basins-burnstone-gold-mine-removing-obstacles-to-productivity-growth/
[Accessed 28 July 2016].

Gehring, K., n.d. *Roadheader,* Turin: Polytechnic University of Turin.

Global, J., 2016. *Room and Pillar Entry development.* [Online]
Available at: https://mining.komatsu/underground-mining
[Accessed 02 2016].

Halim, A. & Kerai, M., 2013. *Ventilation Requirements for 'Electric' Underground Hard Rock Mines - A Conceptual Study.* Adelaide, Australian Mine Ventilation Conference.

Hardman, D. R. & Oberholzer, J. W., 1987. The Performance of continious miners in South African coillieries. *The Journal of The South African Institute of Mining and Metallurgy,* pp. 7-10.

Holtzhausen, W. H., 2014. A comparative study between shuttle cars. *SAIMM*, April, pp. 299-303.

Jacobs, W., 2013. *Electric LHDs in Underground Hard Rock Mining: A Cost/Benefit Analysis.* Perth: University of Western Australia.

Knights, P., Ramezanzadeh, A. & Stewart, P., n.d. *Benchmark Drill and Blast and Mechanical Excavation Advance Rates for Underground Hard-Rock Mine Development,* Brisbane, Australia : The University of Queensland and CRC Mining.

Lamos, A., 2013. *Bulk Material Properties of Khaki Shale,* Johannesburg: Tunra.

Lourens, C., 1999. Resue mining re-emerges. *Mining Weekly*, 27 August.

Mohring, R. P., 1997. The South African coal industry: current position and future challenges. *The Journal of The South African Institute of Mining and Metallurgy,* pp. 205 - 208.

Pickering, R., 2013. *Benefits of Cutting & Cutting Actions.* Johannesburg, CMMS.

Pickering, R., 2013. *Benefits of Cutting & Cutting Actions.* Johannesburg, The Centre for Mechanised Mining systems.

Pickering, R., 2015. *Benefits & Fundamentals of Cutting.* Johannesburg - WITS, The Centre for Mechanised Mining Systems.

Pickering, R., Ras, D., Cockcroft, D. & Tose, S., 2000. Preliminary study into a mechanized resue mining method for deep level mining. *The Journal of The South African Institute of Mining and Metallurgy,* Volume MARCH/APRIL, pp. 83 - 88.

Preston, I. W., 1984. The Presinking of Harmony No. 4 shaft by use of the MATLA headgrear. *SAIMM,* pp. 153 - 163.

Priyadarshi, H., 2015. *Technomine.* [Online]
Available at: http://technology.infomine.com/reviews/tunneling/welcome.asp?view=full
[Accessed 15 08 2017].

Restner, U., Schimek, P. & Halbmayr, C., 2013. RAPID MINE DEVELOPMENT SYSTEMS IN UNDERGROUND HARD ROCK MINING. *Halbmayr*, 03 11, p. 1.

TaungGold, 2014. *Jeanette Pre-feasibiliuty study.* Johannesburg: s.n.

TaungGold, n.d. *Jeanette project - history.* [Online]
Available at: http://www.taunggold.com/gold-sa/jeanette-project-history
[Accessed 24 May 2016].

van Dorssen, P. et al., 2002. A long hole stoping system for mining narrow platinum reefs. *SIAMM*, April, pp. 151-154.

WILLIS, R. P., DIXON, J. R., Cox, J. A. & Pooley, A. D., 2004. A framework for the introduction of mechanized mining. *SAIMM,* pp. 117 - 124.

Young, D., 2016. *Updated Competent Persons Report on the Basal Reevaluation, Jeanette Project, Republic of South Africa,* Bryanston: The Mineral Corporation.