AN ASSESSMENT OF SUPPORT SYSTEMS TO MINE ISOLATED BLOCKS OF GROUND AT THUTHUKANI SHAFT, KDC

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DECLARATION

I declare that this research report is my own, unaided work. It is being submitted for the Degree of Master of Science to the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination in any other University.

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(Signature of candidate)

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ABSTRACT

As most of the South African gold mines near their end of life, with regards to the original reef horizons and ore bodies planned, a large portion of the gold bearing ore lie in isolated blocks of ground which were left behind for various reasons. These blocks of ground are now being investigated with the intention of extraction together with other low grade ore bodies or reef bands. Although, the cost of gold mining has risen substantially, the price received for the finished product has also risen considerably over the past decade and it is still one of the most stable forms of long term investment *(World Gold Council, 2013)*. Thuthukani Shaft, KDC has been mining isolated blocks of ground over the past decade with improving success.

This research report details investigations into the continuously improving support systems that have been used over time with regards to the mining of isolated blocks of ground and seismic activity. The investigation reflect an improving work environment with regards to health and safety and indicates that the support system used at present is capable of preventing damage to workplaces with fairly high seismic activity.

The report contains an analysis of seismic activity with regards to injury and damage to workplaces in relation to the improvements made to the support system over time at Thuthukani Shaft, KDC. The report also contains detail regarding the improvements made to the support system over the past seven years and includes some production statistics in relation to seismic activity at Thuthukani Shaft, KDC.

In memory of all employees
that have fallen victim
to seismic related activity
at Thuthukani Shaft,
KDC

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Chapter 1

1. INTRODUCTION

As most of the South African gold mines near their end of life, a large portion of the high grade ore bodies are mined out with isolated blocks of ground remaining. Many mines are exploring their secondary reef bands which are generally of a lower and often inconsistent grade. Many of these secondary reef bands are marginal and cannot be mined profitably as the operational cost of mining increases. It also becomes costly to mine these isolated blocks of high grade reef as the volumes, in terms of tonnage extracted, are very small and cannot justify the development cost.

A combination of mining the secondary reefs, to create volume, together with the isolated blocks of high grade reef to boost the grade, in many cases, makes the mining business profitable, and extends the life of mine *(Singh and Kataka, 2006*). In many cases the capital cost of establishing the necessary infrastructure are kept to a minimum as most of the infrastructure is already established or requires refurbishment.

A large amount of uncertainty exists with regards to the mining of isolated blocks of ground relating to seismic activity and decisions will have to be made as to whether the extraction of isolated blocks of ground continue or whether mines will have to close down due to the mining business not being sustainable.

After several years of experience, the development of methods is often reviewed to evaluate whether progress has been made and whether mining can continue and is economically viable. This research report considers and assesses the improvements made to the support standard at Thuthukani Shaft, KDC in relation to the mining of isolated blocks of ground and its ability to withstand large magnitude seismic events. The term “**IBOG**” will be used to represent “**isolated blocks of ground**” in this research report.

1.1 Mine background

Located in the Witwatersrand Basin, in the magisterial district of Westonaria, some 60 km south-west of Johannesburg, Thuthukani Shaft which is one of the Kloof Driefontein Complex (KDC) shafts is actually the old Kloof Main Shaft Complex that belonged to Kloof Gold Mine, a division of Gold Fields Limited. ***Figure 1.1.1***, below, indicates the operating mines of Gold Fields International.

Kloof Gold Mine and Driefontein Gold Mine (formerly East and West Driefontein) were two separate entities under Gold Fields Limited. In 2010, a strategic move was made to consolidate these operations into one unit under the Kloof Driefontein Complex (KDC) with five operating Business Units.

KDC East comprises of the old Kloof Gold Mine Shafts and KDC West comprises of the old Driefontein Gold Mine Shafts which are grouped to form Business Units (BU) 1, 2 and 3. KDC East is made up of Business Unit 4 and 5.

Each Business Unit is responsible for ensuring that it is profitable and sustainable as a unit. ***Figure 1.1.2***, below, indicates the basic structure of the Kloof Driefontein Complex. The Kloof Driefontein Complex is one of three South African operations within the Gold Fields Group, the other two being the South Deep and Beatrix operations. The Company is in the process of separating the Beatrix and KDC operations from South Deep by creating a new mining company, Sibanye Gold which was listed on the Johannesburg Stock Exchange (JSE) in February 2013.

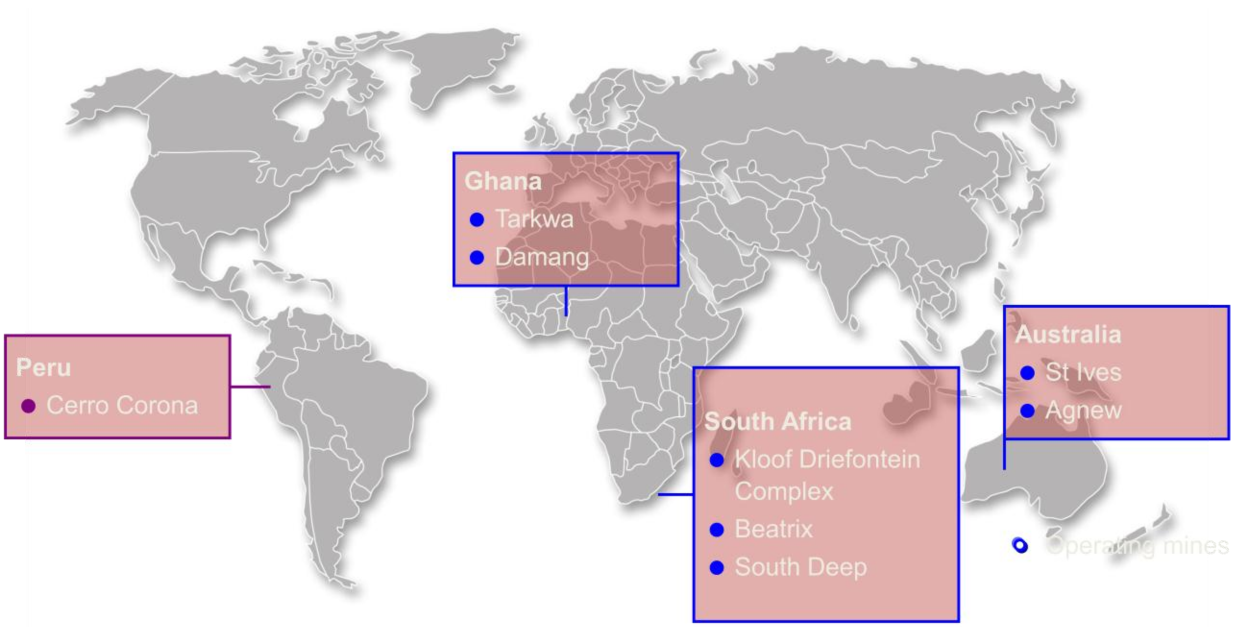


Figure 1.1.1 **Gold Fields International operating mines.**

(Source: Gold Fields Limited)

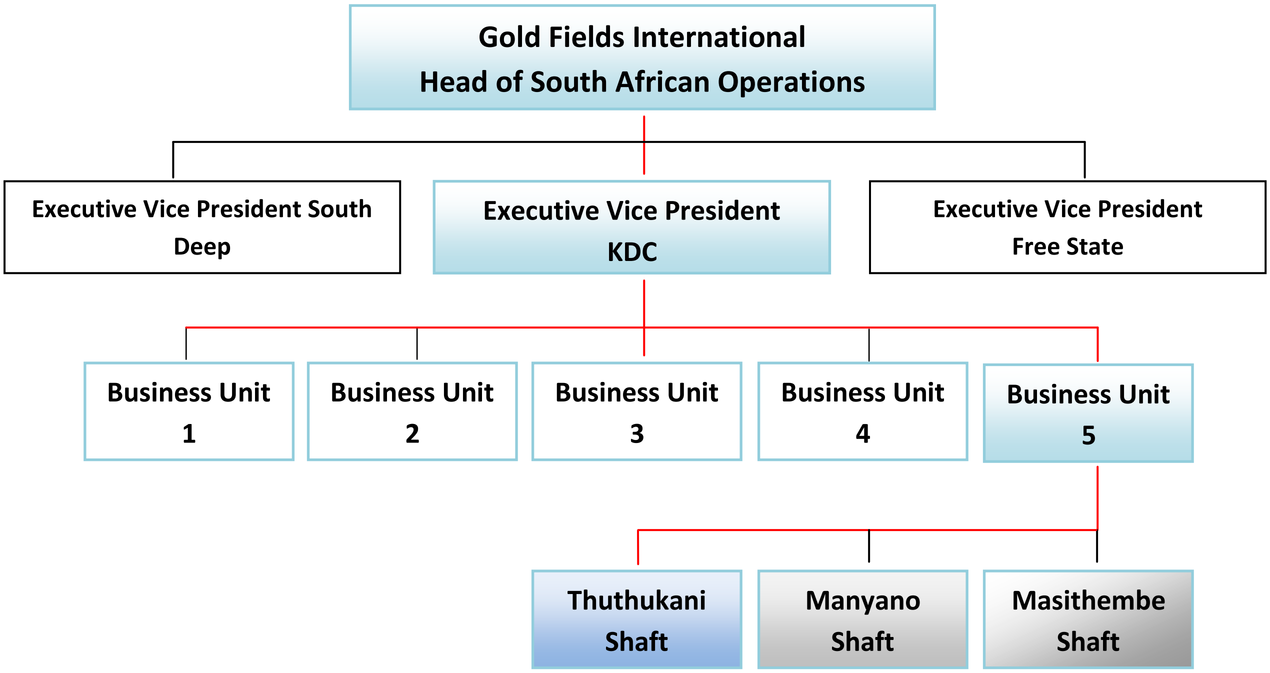


Figure 1.1.2 **Basic Structure of Kloof Driefontein Complex (KDC).**
(Source: Gold Fields Limited)

1.2 History

In the 1950's interest was focused on the area to the south-east of Libanon Gold mine, where the down-dip extensions of the Ventersdorp Contact Reef (VCR) were expected. Several boreholes, which intersected high value ore, were drilled and it was decided that this area would support another mine. In November 1964, work commenced on Kloof's main twin-shaft system.

Almost simultaneously, development from the neighbouring sister mine, Libanon, was initiated to exploit the Kloof lease area. This operation, together with the completion in 1967 of Kloof's first ventilation shaft, which was adapted temporarily for rock hoisting, enabled a considerable amount of reef to be stockpiled on surface.

In January 1968, six months ahead of schedule, the reduction plant came into operation with a milling capacity of 51 000 tons per month. The milling rates had then risen to 190 000 tons per month. Subsequently, downsizing and restructuring of the operations over the years have led to Libanon becoming part of Kloof and Kloof becoming part of Kloof Driefontein Complex (*Internal source, 1986*).

Thuthukani shaft has been in existence for over forty five years and hosts two sub shafts. Most of the high grade VCR has been mined out with isolated blocks of high grade ground remaining, together with the shaft pillar. Mining at Thuthukani Shaft occurs at a depth ranging from 2000m down to a depth of 3500m. Pillars in the 1 sub-shaft zone range from 2000m to 2500m in depth, while those in the 2 sub-shaft zone range from 2500m to 3500m in depth (*Van der Merwe, 2005*).

Figure 1.2.1 below indicates the location of KDC East (Kloof Gold Mine) in relation to Johannesburg.

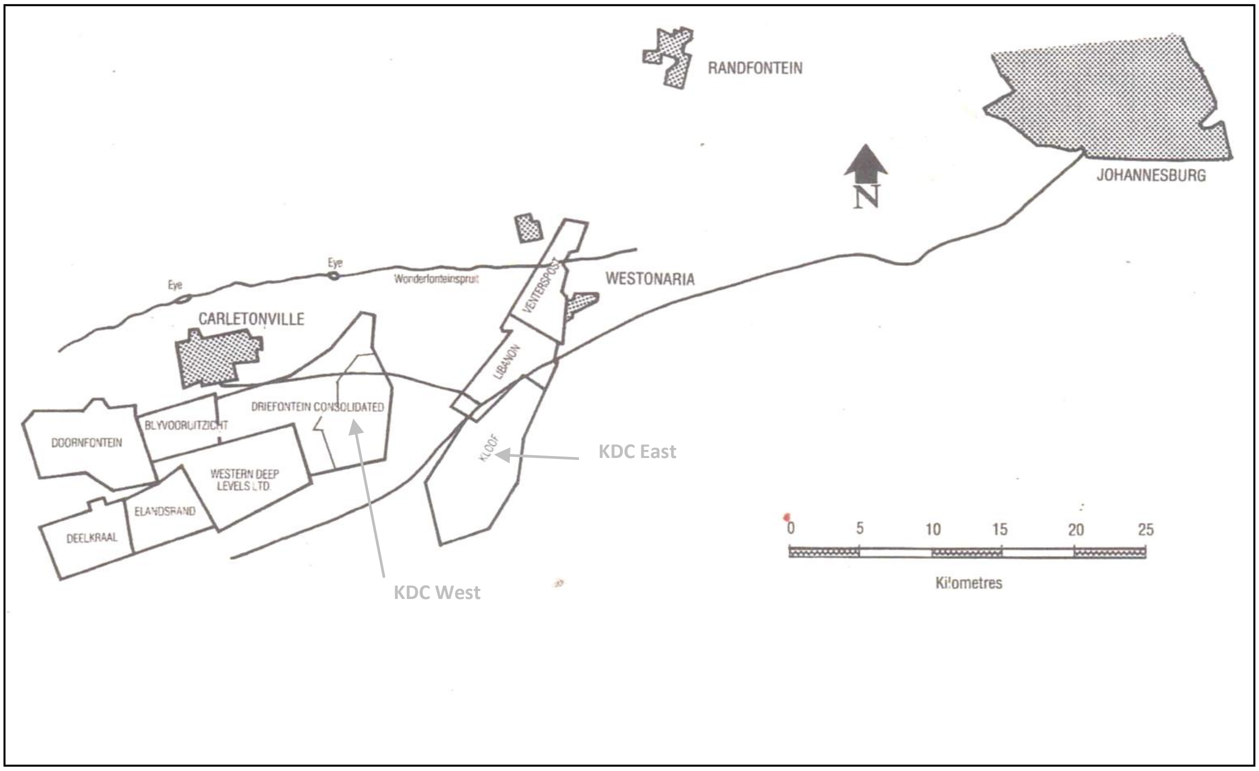


***Figure1.2.1* Location of KDC East.**
(Source: KDC East, Internal Collection)

1.3 The Geology of KDC East (Kloof Gold Mine)

The West Wits Line, also known as the Carletonville gold field, lies on the northwestern edge of the Witwatersrand basin, 35 km west of Johannesburg. Detailed prospecting took place in the area during the 1930s, leading to the eventual development of 10 mines which are amongst the biggest and richest in the entire basin. These include Deelkraal, Elandsrand, Doornfontein, Blyvooruitzicht, Western Deep Levels, Driefontein Consolidated, which all occur on the west of the Bank fault and Kloof, Libanon and Venterspost which occur east of the break. Libanon and Venterspost now fall under the umbrella of Kloof. In addition to gold, uranium and pyrite represent important by-products of the region. The gold field can be divided into two sections along geological lines; the section east of the Bank fault represents a natural sedimentological extension of the West Rand gold field and comprises of the Kloof mine operations centered around the town of Westonaria whereas the western section is sedimentologically distinct and lies between the Carletonville and Fochville. It is the western section which is more correctly referred to as the Carletonville gold field, whilst the entire region is referred to as the West Wits Line *(Robb and Robb, 1998).*

The Transvaal Sequence and Ventersdorp and Witwatersrand Supergroups are present at KDC East. The Central Rand Group prevails upwards to the Elsburg Quartzite Formation, before wedging out against the Ventersdorp Conglomerate Formation. Jeppestown shales of the West Rand Group are partially exposed in the deepest underground development of the mine.

Transvaal Sequence sediments and volcanic comprising the Timeball Hill Formation and Hekpoort Andesite Formation cover the entire lease area. The average dip is 15° to the south. Outcrops of post Transvaal diabase sills which have intruded into the Timeball Hill quartzites and shales are prominent.

Some alluvium is also present. The Venterspost Dyke is found on surface as are numerous faults.

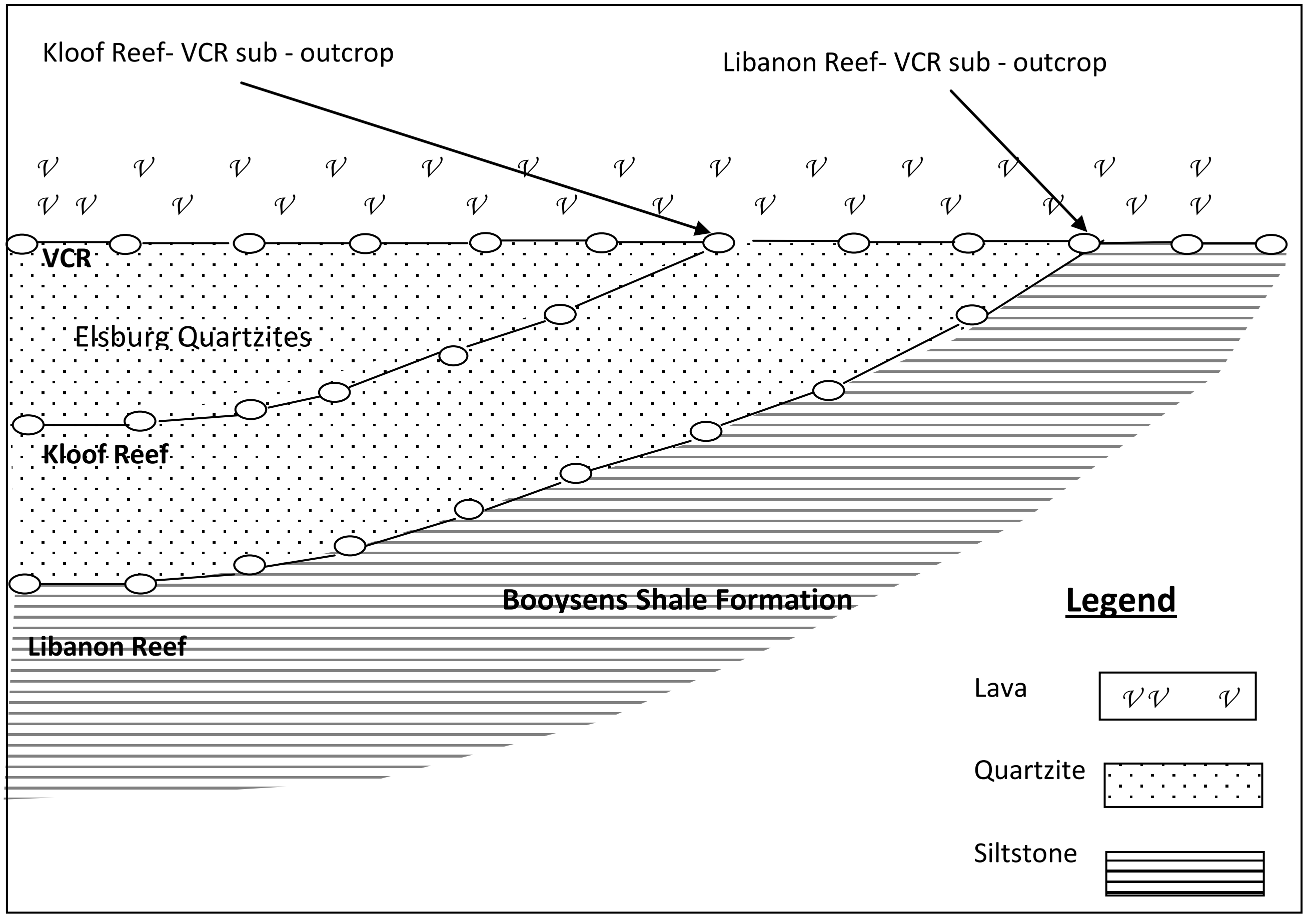


Figure 1.3.1 **Unconformable relationship between VCR and underlying strata.** (Source: KDC East, Internal Collection)

Witwatersrand Supergroup

The beds of the Central Rand Group dip to the south-east at 35°-40°. They are truncated by the VCR, also dipping to the south-east at 30°-35° as indicated in the section view, ***Figure 1.3.1***, above, which is not to scale. However, due to the general synformal structure of the Central Rand Group in this area, an arc shaped pattern is described by the sub-outcrop traces of the Kloof and Libanon Reefs. ***Figure 1.3.2*** below, indicates a plan of the sub-outcrop of the different reef bands.

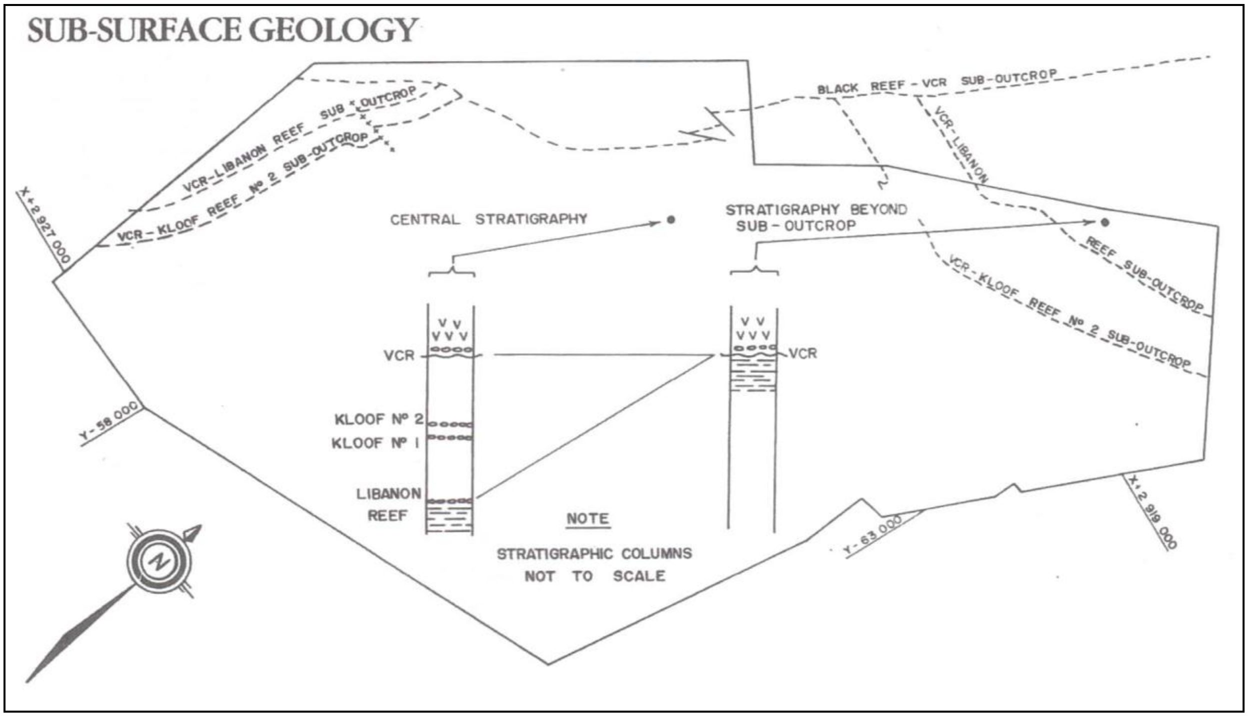


***Figure 1.3.2* Sub-surface geology.**
(Source: KDC East, Internal Collection)

An important secondary reef mined at KDC East is the No. 2 Kloof Reef of the Elsburg Quartzite Formation. It forms part of the Kloof Reefs which constitute a zone of conglomerates and quartzites which have been subdivided into the No. 1 Kloof Reef, towards the base overlain by the No.2 Kloof Reef. There are a number of marker horizons which, although not always distinct, aid locating the zone. The Footwall marker, a relatively thin grit to small-pebble conglomerate, marks the bottom of the zone of Kloof Reef bands. The Footwall marker is overlain by the No. 1 and No. 2 Kloof Reefs, respectively.

The No. 2 Kloof Reef, in turn, is overlain by the Hanging wall marker, a band of quartzites with scattered pebbles, and with a conglomerate lag at the bottom and top of the bed. Above the Hanging wall marker are the scattered pebble and glassy quartzite markers with distances of 12 and 17 meters respectively above the No.2 Kloof Reef. ***Figure 1.3.3*** is a photograph of the No. 2 Kloof Reef.

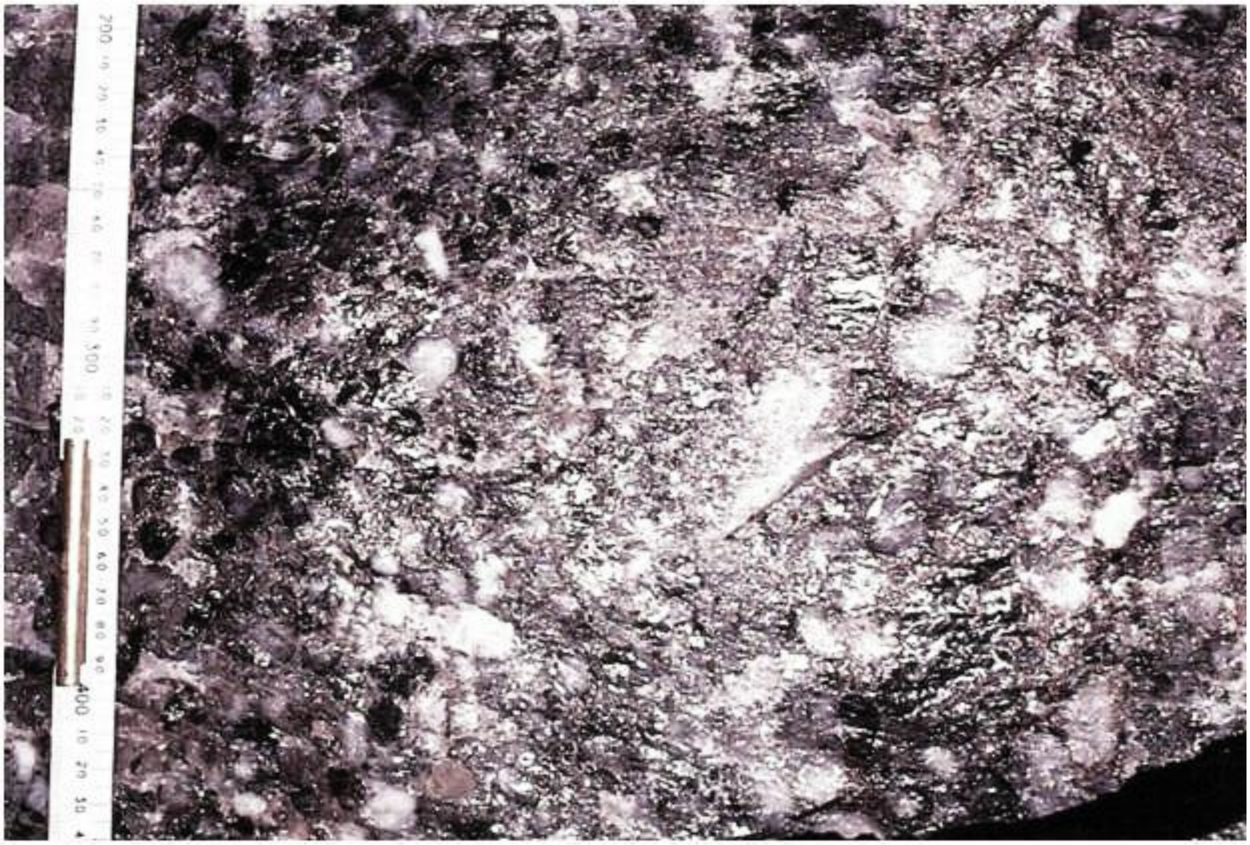


Figure 1.3.3 **No. 2 Kloof Reef.**
(Source: KDC East, Internal Collection)

The Libanon Reef of the Kimberly Conglomerate Formation lies unconformably on the Booysens Shale Formation in the mine area. It is a highly variable reef zone up to 1.8 meters thick, consisting of alternating bands of matrix- and clast-supported conglomerates and quartzite's. The pebbles are on average medium to large in size (8-32millimeters diameter) and consist mostly of quartz with lesser chert, quartzite and shale. The matrix of the conglomerates is composed mainly of very coarse quartz grains and clay-sized material, a large amount of small round carbon grains (fly speck) being apparent, with a greater concentration on bedding planes.

Figure 1.3.4 illustrates the Libanon Reef.

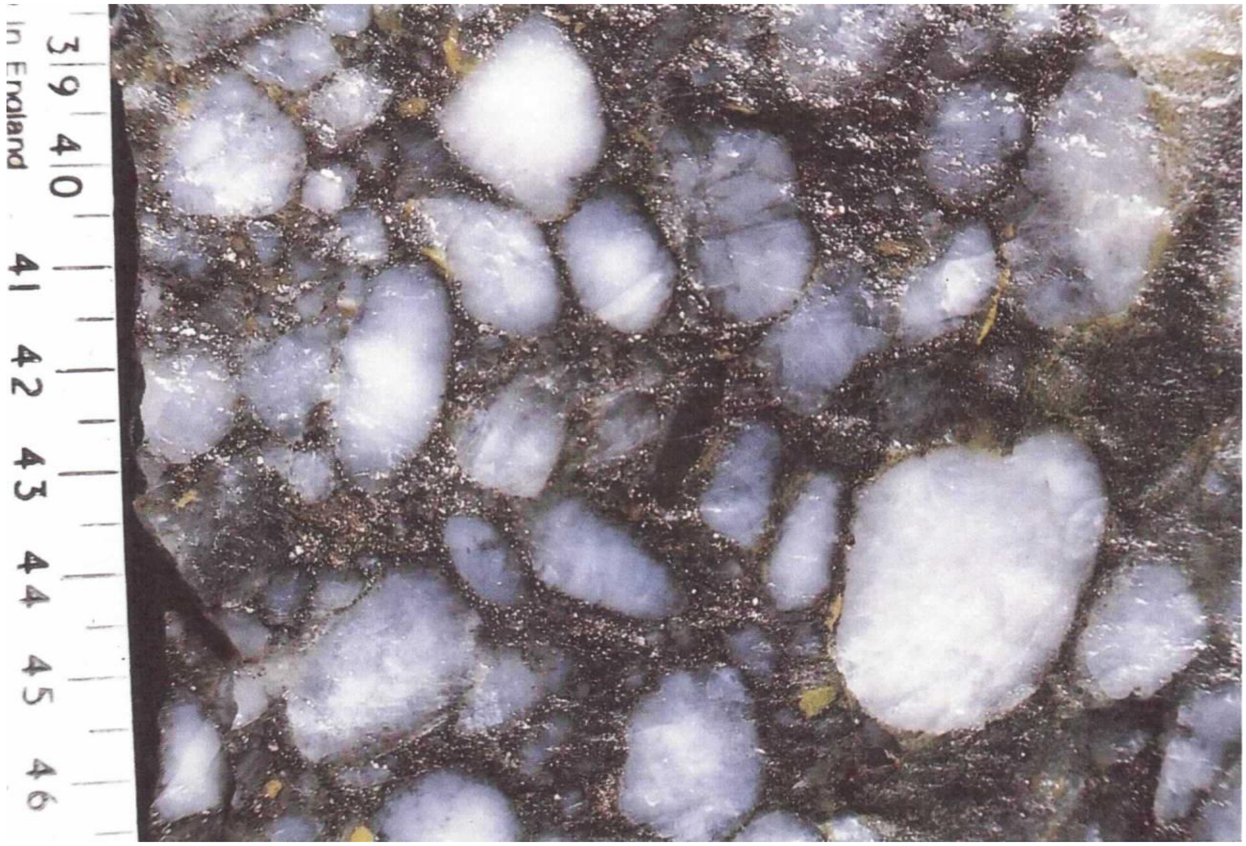


Figure 1.3.4 **Libanon Reef.**
(Source: KDC East, Internal Collection)

Ventersdorp Supergroup

The VCR is conformably overlain by the Ventersdorp Supergroup lava, which in turn is unconformably overlain by Transvaal Sequence at the base of which the Black Reef. This results in the VCR sub-outcropping against the Black Reef Formation.

<u>Venterspost Conglomerate Formation</u>

The VCR varies in thickness between 0 and 7.5 meters, but on average is usually 1.0 meter thick. It consists of conglomerates with interlayered shales and quartzites. The conglomerate consists primarily of white, milky, and smoky quartz, with occasional quartzite, shale, and agate pebbles being present. The matrix is quartzite, sorting is poor, and clast sizes range from medium to cobble size (8-256 millimeters diameter).

Due to the variety of types of VCR observed, various subdivisions have been made. The *Contact Reef* type consists of a single pebble layer, or just a contact between the hanging wall and footwall rocks.

The *Single Reef* type varies in thickness from 0.1 to 3.0 meters, composed of a single clast- or matrix- supported, generally oligomictic conglomerate. A large variation in pebble size (30 – 120 millimeters diameter) and degree of sorting is observed. The basal contact is sharp and irregular, due to channeling and scouring.

This type of reef is illustrated in ***Figure 1.3.5***.

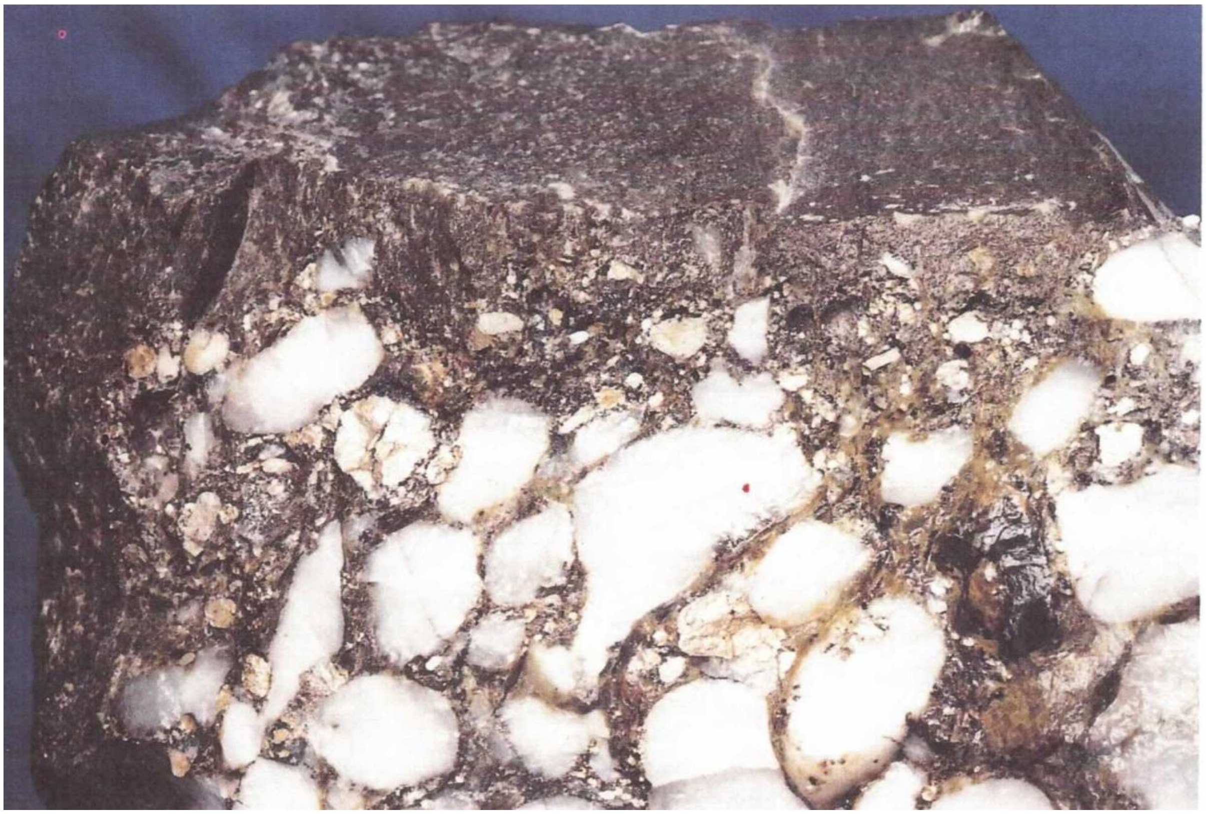


Figure 1.3.5 **VCR single conglomerate.**
(Source: KDC East, Internal Collection)

The *Black Shale* type occurs in one small area west of Thuthukani Shaft, and wedges out against the Black Reef. It is characterised by the presence of angular blocks of shale and altered lava up to 1 meter in diameter. These occur together with rounded clasts of quartz in a quartzitic matrix. The reef can be up to 3 meters in thickness and can consist of two or more distinct reef bands, sometimes separated from each other by internal quartzite bands.

The *Inter-reef* Lava type generally consists of a dark matrix, with clast-supported conglomerate (bottom band). The internal lava can reach thicknesses of 3-5 meters. The upper band is less well developed, comprising of planar bedded quartzite, often pyroclastic, and sometimes a thin conglomerate band may be present.

This type of reef is illustrated in ***Figure 1.3.6***.

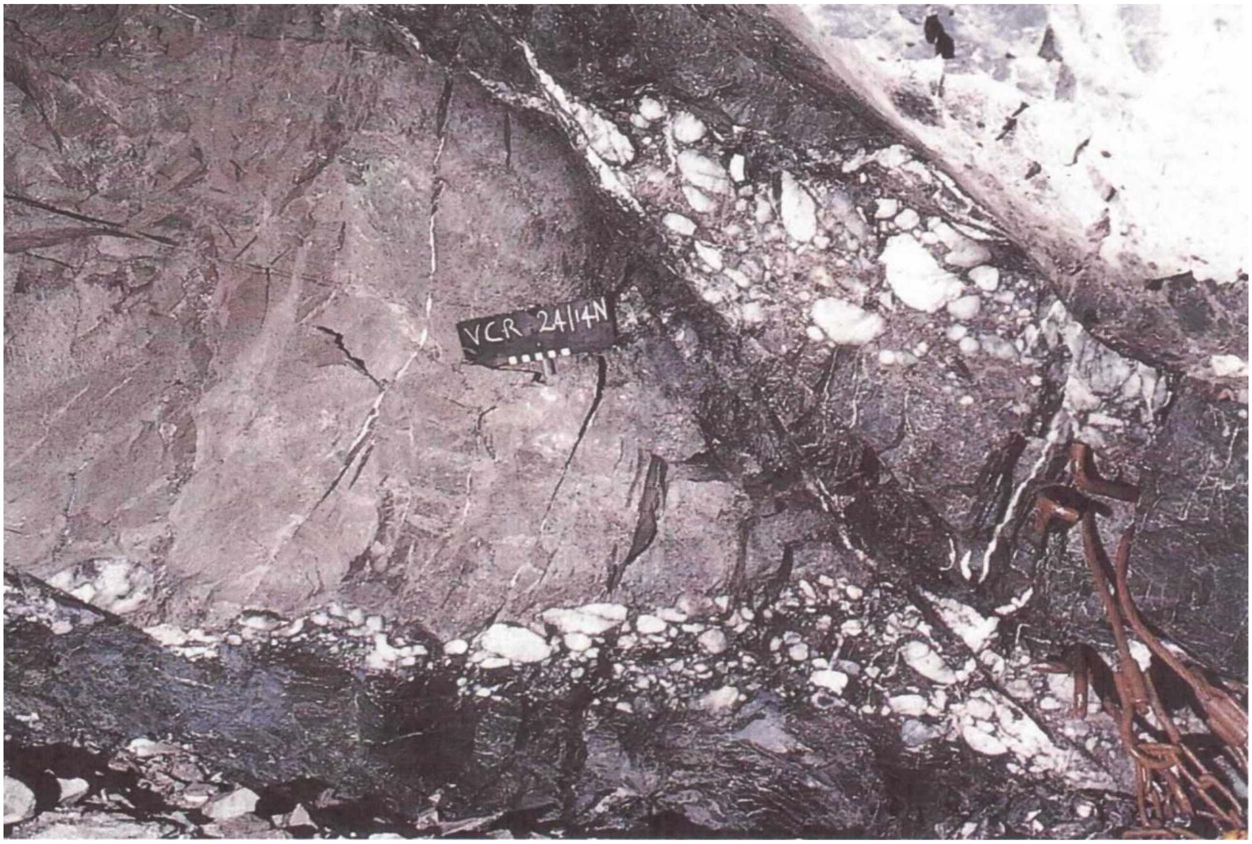


Figure 1.3.6 **VCR Inter-reef lava.**
(Source: KDC East, Internal Collection)

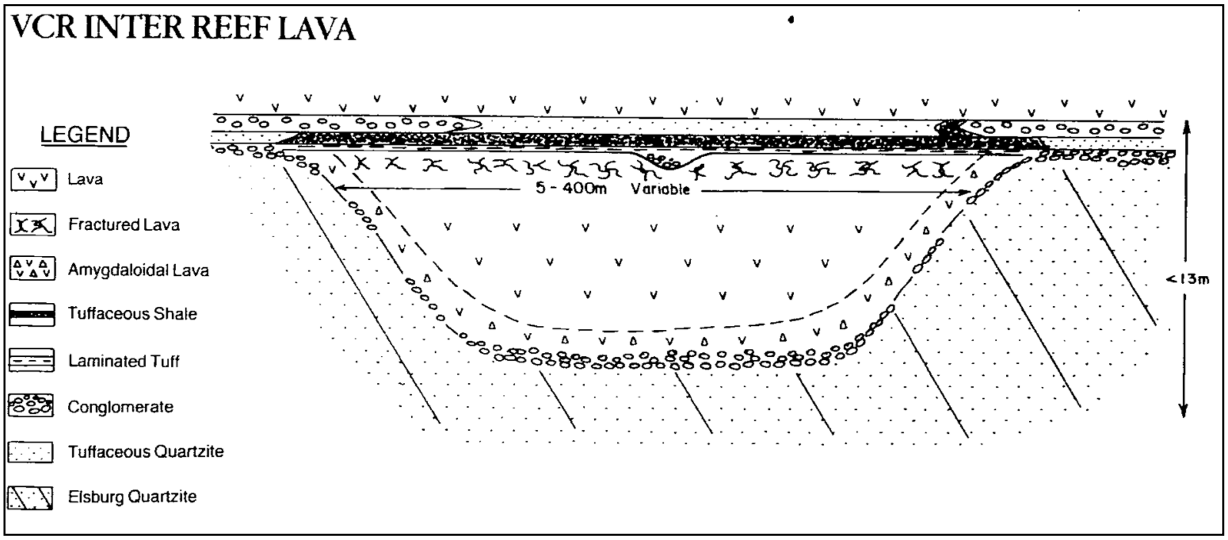


Figure 1.3.7 **VCR Inter-reef lava.**
(Source: KDC East, Internal Collection)

The *Composite Reef* type is characterized by multiple conglomerate bands separated by quartzite and shale horizons, reaching a thickness of up to 2 meters. The conglomerates are clast-supported and contain poorly sorted pebbles of white quartz. The quartzites can be planar bedded, trough cross-bedded or planar cross-bedded.

The *Quartzitic Reef* type is characterized by the absence of a conglomerate band and consists of a quartzite generally planar or trough cross-bedded. It differs vertically in colour in that it is dark grey in the upper region and grades into a light grey to buff coloured quartzite at the base. At times a thin shale band may occur above or within the upper zone. The basal contact is sharp and characterized by a thin brown zone resulting from the oxidation of sulphides. A thickness up to 3 meters is possible, and gold content is erratic and low (*Internal source, 1986*).

Figure 1.3.8 indicates the West Wits Line generalized stratigraphy.

WEST WITS LINE GENERALISED STRATIGRAPHY

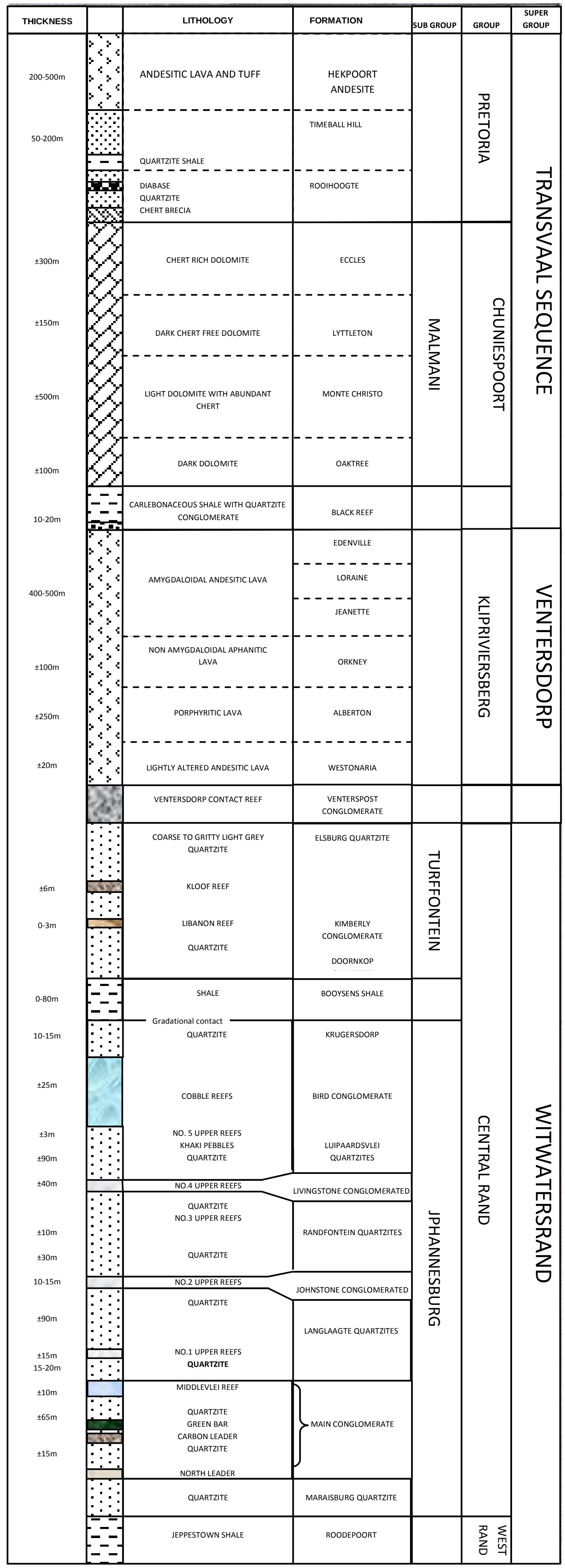


Figure 1.3.8(Source: KDC East, Internal Collection)

Structure

Two major tectonic trends are present at KDC East. The first trend corresponds to major dykes striking in a North-South direction. These include the Venterspost, the Libanon, and the “Running” dykes. The second trend, which is a parallel system of sinistral tear faults with associated dyke intrusions, strikes North East- South West.

The most important faulting on the mine are the sinistral tear faults. Due to the strike of these faults being oblique to the strike of the VCR, a resultant down-throw to the North West exists. In the south-western corner of the mine a highly faulted area is related to small scale folding. It exhibits an open fold pattern, the axes striking North West- South East, with a gentle plunge to the South East.

There are four ages of dykes on the mine. The first, the *Glenharvie dykes*, are the oldest, and are of doleritic composition. These dykes strike North West – South East, exhibiting no displacement. The *Running dyke*, as well as the *Black dyke*, are the next oldest, and are melanocratic syenites, or shonkinites, no displacement being observed. Thirdly, the *Libanon* and *Venterspost dykes*, of Pilanesberg age, are composite dykes, i.e. the margins of these dykes consist of doleritic material enclosing the leucocratic syenitic core. No displacement occurs across these dykes. The youngest dykes, or *tear fault dykes*, intruded along the fault zones, are of doleritic composition, and cut all previous dykes.

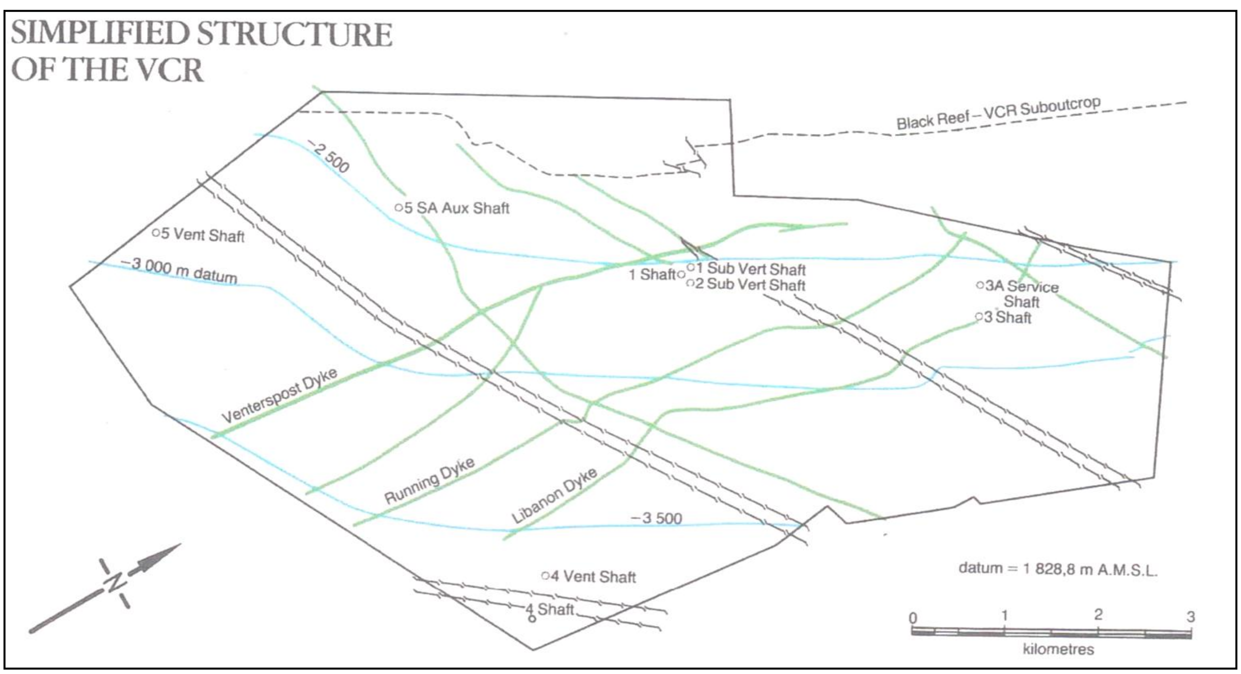


Figure 1.3.9 **Simplified Structure of the VCR.**
(Source: KDC East, Internal Collection)

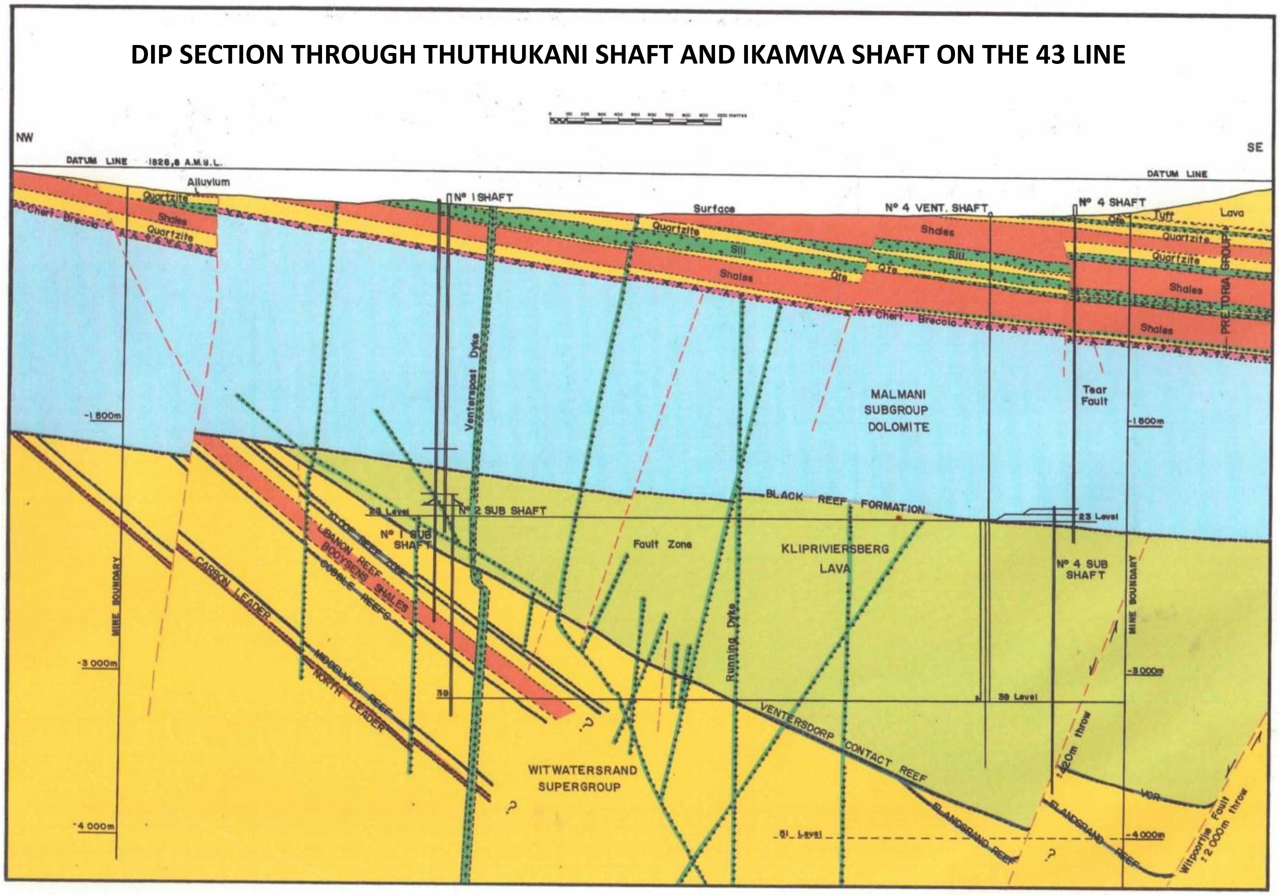


Figure 1.3.10 **Dip section through Thuthukani and Ikamva Shafts on the 43 Line.**
(Source: KDC East, Internal Collection)

A North West- South East dip section through the mine, in ***Figure 1.3.10***, illustrates most of the most important geological features described above. Kloof No. 1 and 4 Shafts are now called Thuthukani Shaft and Ikamva Shaft, respectively.

Water and Gas

Problems related to the presence of Malmani dolomite, and its associated solution cavities are not present on the surface at KDC East. This can be attributed to the solid capping of Pretoria Group rocks on the dolomite in this area, forming a firm “foundation” for surface infrastructure, varying between 200 and 800 meters in thickness, indicated in ***Figure 1.3.10***.

Underground water is occasionally encountered in development ends and diamond drill holes. It is associated with dykes and fault zones. It originates in the dolomite and uses the tectonic features to penetrate into the Witwatersrand sediments. However, these fissures are cemented so continual pumping is not necessary.

During shaft sinking through the dolomite, cover drilling invariably intersects varying amounts of water, but to date, no problem has been encountered that could not be solved by cementation.

Occasionally, small quantities of methane are encountered, 95 percent of all occurrences being associated with intrusions. The concentration of methane is generally below 10 percent and this is diluted and exhausted by the ventilation system before development continues. ***Figure 1.3.11*** indicates the different water compartments found in the KDC East lease area.

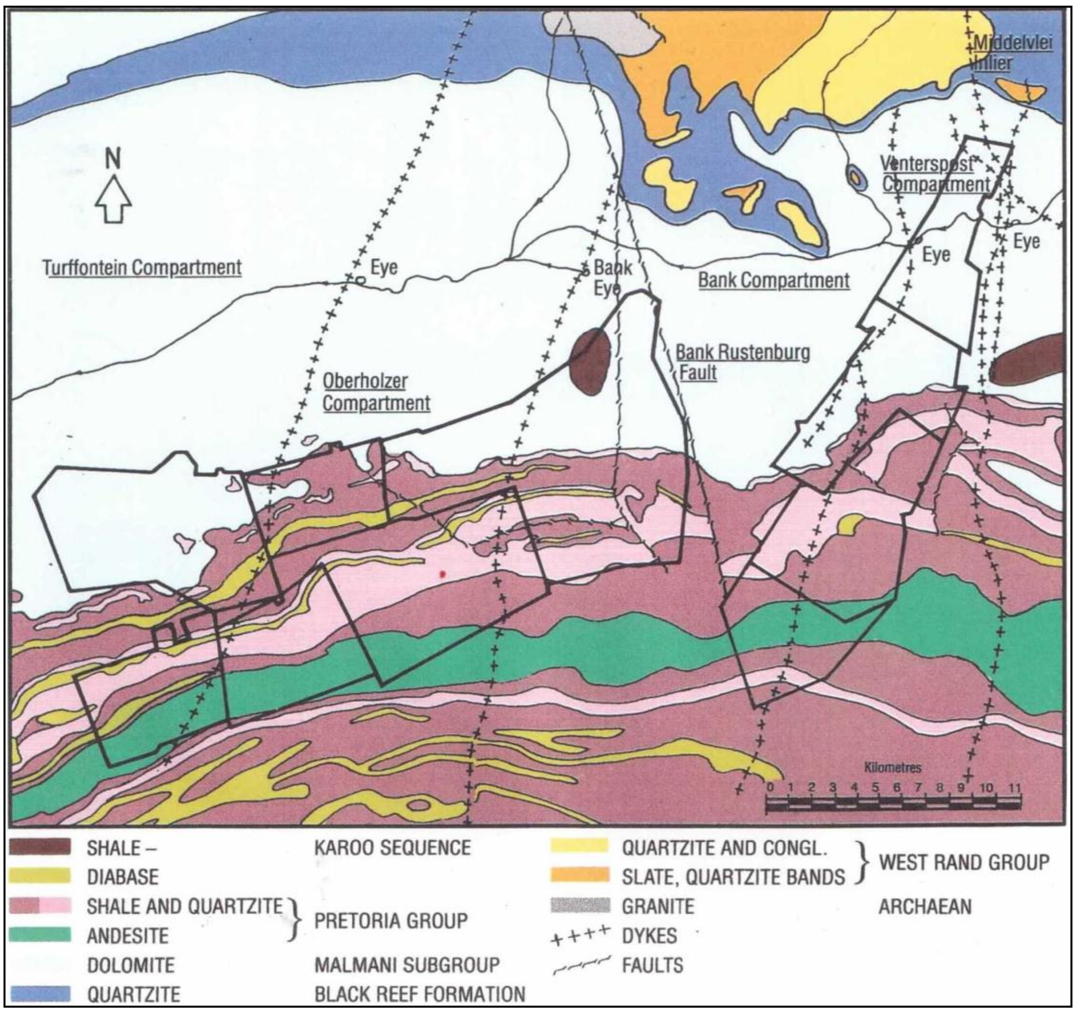


Figure 1.3.11 **Different water compartments found in KDC East lease area.**
(Source: KDC East, Internal Collection)

1.4 Research Objective

It can be noted from the section above that a significant amount of geological features and structures exist at KDC East. These geological features and structures have significant impact with regards to seismic activity at KDC East. The primary targeted reef band, being the VCR, was also described. This reef band has been extensively mined over the years, and in the past decade, only isolated blocks of VCR remaining at Thuthukani Shaft, KDC have been exploited.

Table 1.4 .1 **Production Plan for C2013**
(Source: Thuthukani Shaft, 2013)

	Production (m^2)		Gold Broken (Kg)	
Reef	Total	Monthly	Total	Monthly
VCR (IBOG)	30195	2516	3280	273.33
KL Reef	3777	314	98.22	8.19
MR	37816	3151	1161	96.75
Total Thuthukani	71788	5982	4539	378.3

Table 1.4.1 indicates the production plan for Thuthukani Shaft, KDC for calendar year 2013. It can be noted that 42% of the total production square meters and 72% of the gold required comes from the VCR IBOG.

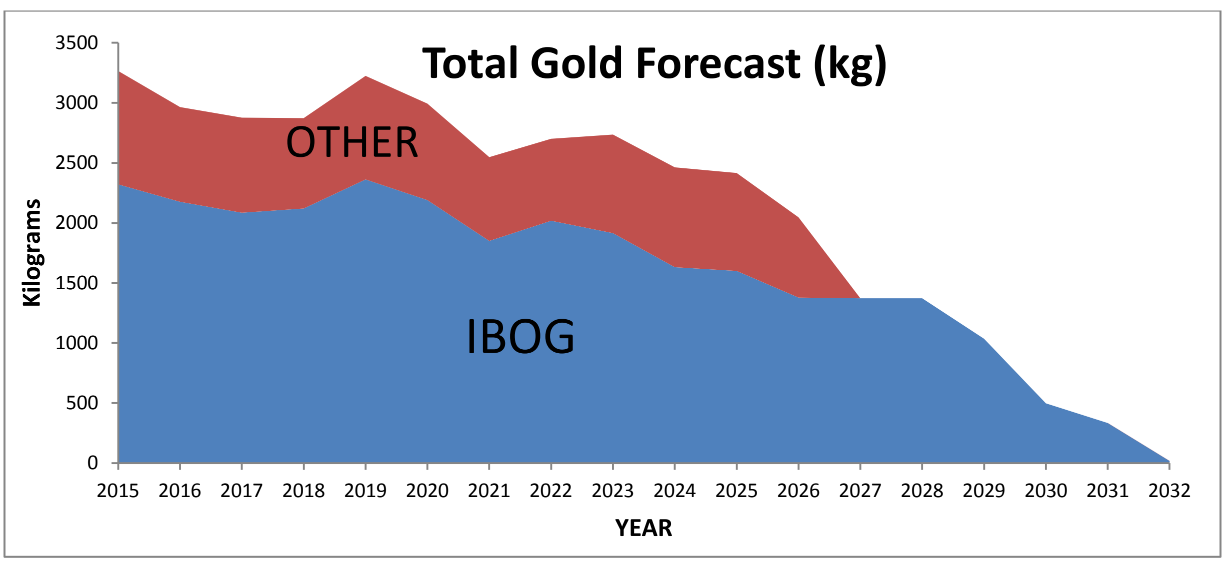


Figure 1.4.1 **Total Gold Forecast for Thuthukani Shaft, KDC.**
(Source: Thuthukani Shaft, 2013)

Figure 1.4.1 indicates the total gold forecast for Thuthukani Shaft, KDC for the life of mine plan. A large quantity of the gold comes from the mining of IBOG and from 2027 the mining of IBOG will be the only contributor to gold production extending the life of mine by five years.

In May 2008, after a fatal shaft accident at the South Deep Gold Mine, Gold Fields Limited Chief Executive Officer, Nick Holland said, *"If we cannot mine safely, we won't mine at all*"*(Oelofse, 2008).* Although a huge amount of focus and effort had been placed into improving safety at the mines, this statement prompted management at Thuthukani Shaft, KDC to place even more effort into improving the working conditions at the mine. Although improvement to the support standards were being investigated prior to this public statement, drive and focus created by this statement, led to a number of changes to the support standard that took place in the following years.

The intention of this research report is to analyse the seismic, safety and production data captured over the past seven years in order to determine if the support system employed at Thuthukani Shaft, KDC is capable of withstanding large magnitude seismic events and to analyze and assess the changes made to the support system over the past seven years that have led to the support system possibly possessing this capability when mining IBOG.

This research will add to the current knowledge base regarding the mining of IBOG. It is critical that this research be carried out at this point in time for two reasons, firstly, due to a lack of large areas with high grade ore bodies; there is a tendency for mines to start exploiting IBOG which are under high stresses. Secondly, with the considerably increased focus on safety, it is of utmost importance that these IBOG are mined safely. This study will help understand how to plan and exploit these IBOG, safely, in order to extend the life of mines and thus sustain the mining business and employment, created by the existence of the mine, for a longer period.

The industrial application of the outcomes of this study will be a well structured document that will assist the various deep level gold mines that are considering the mining of IBOG that were left behind for whatever reason. The lessons learned and best practice applied will be elaborated on to ensure that IBOG are extracted in the safest manner possible.

Chapter One of this research report describes the background, history and geology of the mine and one can take note of the complex geological structure of the Witwatersrand Basin and the types of reef found at KDC East. This chapter also describes the intention of this research report.

Chapter Two describes some of the history and work that is required when intending to mine IBOG and one is made aware of the quantity of work and the risks that are involved prior to the extraction process.

Chapter Three contains the analysis conducted from seismic, safety and production data captured over the past seven years and a major change in the statistics can be noted from the year 2008. This is, consequently, also the year that the Chief Executive Officer of Gold Fields, Nick Holland, made the public and mind-changing statement "*If we cannot mine safely, we will not mine at all*". The data analysed refer specifically to the mining of isolated blocks of VCR at Thuthukani Shaft, KDC.

Chapter Four describes the support systems employed over the period investigated and highlights the changes made. Improvements with regards to area coverage and support density are discussed and these improvements relate to the statistics analysed in chapter three.

Chapter's Five to Seven discuss pre-conditioning, in-stope netting and in-stope bolting in detail. These were introduced sequentially, from 2008, to the support standard in an effort to improve the performance of the support standard and consequently improve safety in underground workings. An immense amount of work has been conducted by various people and organisations like The Chamber of Mines, the CSIR, and Safety in Mine Research Advisory Committee (SIMRAC) over the years regarding pre-conditioning, in-stope netting and in-stope bolting. Some of the detail and best practices are discussed in these chapters.

Chapter Eight contains a conclusion and recommendation for future work with regards to improving the support standard and the mining of IBOG.

Chapter 2

2. PREPARATION FOR MINING OF IBOG AT THUTHUKANI SHAFT, KDC

2.1 History

Thuthukani Shaft, historically, mined a large portion of VCR with limited secondary reefs. The secondary reefs available are the Main Reef (MR) and the Kloof Reef (KR) ore bodies which can be mined profitably, provided that the remaining isolated blocks of VCR are mined, concurrently, to boost the overall plant grade. The mining of these IBOG must be carefully planned and executed as these IBOG are highly stressed and pose a serious health and safety risk from a seismicity point of view *(Klokow, 2006).*

IBOG were left behind for various reasons including the following:

- Ventilation

  Blocks of ground were initially mined according to the ventilation districts. As new blocks of ground were prepared, from a ventilation point of view, mining of the new blocks commenced with more and more of the mining crews being moved into the new blocks of ground. When there were only a handful of crews left in the old block, the block was abandoned and all crews were moved into the new block of ground. All equipment was stripped, reclaimed and ventilation flow cut off. Abandoning of high grades in the pillars that were left behind did not matter as most of the VCR ground comprised of high grade ore, and the focus was to keep the metallurgical plant full of ore, so there was a drive to maximize volume output from the underground workings.

- Fire

  Some blocks of ground were abandoned due to underground fires, when blocks of ground were affected by fire breakouts; they were sealed off and the production crews were moved to other parts of the mine that had been prepared for mining operations. Some of these fires took long periods of time to extinguish and the mine management, at that time, did not want to waste time in re-establishing these panels while there was plenty of good ground to mine.

  Firewalls were also planned, in the older days, to separate sections of the mine such that in the case of fires only certain sections would be affected, isolating the fire and areas affected by it. These firewalls were anything in excess of ten meters in width, on strike. In some cases, measurements in excess of thirty meters, in width, on strike, have been noted.

- Falls of Ground (FOG)

  Workplaces were also abandoned when there were falls of ground or when areas were affected by seismic related damage. Due to high grade ground being readily available and the time and cost related to re-establishing of workplaces coupled with the fear of going back into a seismically active area, workplaces were abandoned.

- Geological Features

  Some workplaces were abandoned when they intersected geological features, or when nearing a known geological feature. The task of negotiating the feature was associated with additional cost and reduced returns due to the loss created by the feature. Mining rates were also reduced when negotiating geological features, and in a highly incentivized production operation, this was seen negatively. The simpler option was to abandon the workplaces nearing geological features and move the production crew into newly prepared ground where production rates would be maintained.

- Stabilizing Pillars

  Strike and dip stabilizing pillars were also planned as part of the mine design and regional support system. The strike stabilizing pillars found at Thuthukani Shaft, KDC are in excess of thirty meters in thickness and continue for hundreds of meters on strike. They contain a huge amount of high grade VCR ore.

These workplaces remained abandoned for long periods of time and are therefore highly stressed as mining around these blocks of ground continued transferring the majority of stresses onto these remaining blocks of ground.

2.2 Safety and Production

The focus on safety has increased significantly over the past few years and seismic related injuries are just not acceptable any more. It becomes fundamentally important that for mines to remain operational in the future, where IBOG are required to be mined, that the extraction of these blocks of ground be executed in the safest manner economically possible.

Over the past six years, various interventions with regards to making the extraction of IBOG safer have been introduced with varying degrees of success. Some workplaces have experienced seismic events in excess of magnitude 1.2 where results of no damage have been recorded.

The mining of IBOG is a specialist area and the entire line management involved must be highly skilled in the extraction of IBOG. The mining of IBOG is fundamentally different from other types of mining as it is support intensive and involves strict planning and execution. One of the most important factors with regards to the mining of IBOG that make it fundamentally different from other types of mining is that extraction processes have to be consistent and regular. The mining faces cannot stand for long periods of time as stresses transferred causes the fracturing process and bed separation to build up creating unsafe mining conditions. Safety related stoppages, where the mine or sections of the mine have been stopped can have a very negative impact on the mining of IBOG. One of the special instructions issued after a lengthy stoppage period is for additional support units to be installed in between the support units that have already been installed to mine standard, closest to the face, before any blasting operations continue. Mine design and sequence of extraction is of the utmost importance and adhering to fundamental mining principles and guidelines with regards to the Mine Health and Safety Act is critical for success *(Mine Health and Safety Act, 1996)*.

Auditing of the various processes is labour intensive and involves a large amount of paperwork and data capturing. External auditing with regards to mine design and sequence of extraction is carried out regularly to ensure that compliance is ensured. This audit process is normally carried out by an external rock engineering consulting firm. Auditing of working faces takes place more often than the norm as these IBOG are planned with the expectation of high grades and frequent auditing is carried out to ensure that the plan is met with regards to sampling values, stope width control and dilution. Strata control observers are tasked with auditing workplaces to ensure that mining dimensions and rates of extraction are maintained within the plan. This will include an inspection of the support installation; face advance and lead-and-lag measurements. This information is then compared to the plan and recommendations are then made with regards to correcting abnormal occurrences or deviations from the plan, which is then fed to line management.

Thuthukani Shaft is highly dependent on the mining of IBOG, as noted in ***Figure 1.4.1***,which are normally mined at low production volumes and high grades. More than forty percent of the total production, in terms of square meters mined, comes from the mining of isolated blocks of VCR ground at Thuthukani Shaft, KDC. The entire block of ground is planned with the expectation of relatively high grades and if lower than expected grades are achieved, a negative effect on the overall business plan can be experienced. More than seventy percent of the total gold output, at Thuthukani Shaft, KDC, comes from the mining of isolated blocks of VCR ground (**APPENDIX H** indicates the typical IBOG found at Thuthukani shaft, KDC).

Loss of working places due to seismic damage can be costly if sufficient spare working panels are not readily available, spare work panels must also match the grade profile of the panel lost in order for production plans to be achieved. Time is also required to move production crews to new work places and to re-established damaged work places.

2.3 Opportunity

The main objective of all mining systems is *safe production* which is a term intended to include the philosophy that occupational health and safety, and production should not be separate issues. Whatever mining method is utilized, it must ensure an acceptable quality of life for those working in the industry and an acceptable return on investment for those who finance the venture. The concept of safe production unites the sub-objectives into a common goal *(Pickering, 1996).*

A wide range of opportunities also exist with regards to the mining of IBOG. The testing and implementation of new technology to reduce work time, remove persons from the face area where face ejections occur, to reduce the amount of labour required and to improve the overall safety conditions of the workplaces are some of the opportunities. Another is the development of specialist working teams, teams that understand the dynamics of mining IBOG and can react and respond appropriately to the various conditions that exist. There is an opportunity for the development of one’s experience and research in the mining of IBOG with regards to deep level gold mining, and for the development of new ideas, new thinking and new ways of doing things in a dynamic mining industry.

One of the interventions at Thuthukani Shaft was the introduction of the “BROKK” machines. These machines were initially developed in Sweden for the cleaning of large industry kilns. Due to the high temperatures, experienced in these furnaces, the“BROKK” machine was used to do the cleaning, remotely.

Figure 2.3.1 **The BROKK 400.**
(Source: www.brokk.com)

The machine was then adapted to be used in the underground environment and is capable of conducting most of the Opening-up processes, remotely, with the operator situated between ten to twenty meters, in a safe area, behind the machine.

Figure 2.3.2 **The BROKK machine at Thuthukani Shaft, KDC.**
(Source: Personal Collection)

Figure 2.3.2 depicts a BROKK machine in an Opening-up environment at Thuthukani Shaft, KDC. The BROKK is a trackless system and is capable of conducting work in areas where tracks are seriously damaged or worn out. It is also capable of being driven over muck piles and work in a retreat manner, barring, loading and moving rubble to a more stable area where the rubble is loaded onto a hopper and transported out of the workings by locomotive.

2.4 Opening- up and Reclamation

Opening- up is a process that is followed prior to the accessing of IBOG. It is the Opening-up of old haulages and workings in order to gain access to the targeted block of ground. Reclamation is the reclaiming of all materials including gold bearing ore which have financial value. Old material can be re-used or refurbished for use, and the “old gold” that is obtained during the process of opening- up adds value to the gold output of the mine, and in many cases caters for the cost of opening-up. The opening-up and reclamation process requires proper planning which requires investigation and research into the history of the workplace is required and the materiality and reliability of the information available is often questioned. This is a lengthy process.

Together with the compilation of all the relevant historical information and documentation, a reconnaissance of the workplace is required and this requires the expertise of highly skilled persons who follow a process of exploring old workings to ultimately access the IBOG. During the reconnaissance visit information is captured with regards to the physical and environmental conditions, in the form of measurements, notes on plans and photographs. Care is taken not to disturb any of the existing conditions as this can be an unsafe environment if the necessary care is not taken. The team progress to as far as it is physically possible. Sometimes, total closure, excessive temperatures and unknown depths of water prevent them from progressing further.

Reconnaissance visits are planned and organized according to a standard. All the necessary equipment and material, like plans and first aid kits, are provided and checked prior to entering the old workings. The mine manager and control room operators are informed of the visit, the industry numbers and details of persons that will embark on the visit together with the approximate time of entry and exit into the old workings are conveyed to the operations manager and control room operators. The control room operators will carefully monitor progress of the reconnaissance visit with members of the reconnaissance team making contact with the control room regularly. Prolonged loss of contact or a report of injury will prompt the control room to dispatch a rescue team or the necessary medical assistance that is required.

Figure 2.4.1 **Opening-up reconnaissance visit.**
(Source: Personal Collection)

Opening-up is categorized into four groups ranging from 25% opening-up, up to 100% opening-up which is total closure of a tunnel or access way. ***Figure 2.4.1*** depicts a member of a reconnaissance team inspecting a fall of ground on a reconnaissance visit. The reconnaissance visit member is looking at a typically 50% opening-up situation leading from a 25% opening-up situation. It is important to understand that once barring of the side walls and hanging walls take place, the muck pile being inspected will increase in volume. One can also notice the extent of deterioration of the track work and the reddish-brown colour of the stagnant water caused by oxidation of the infrastructure.

Figure 2.4.2 depicts a 100% opening-up scenario. Ring-set support and timber cribbing that was damaged over time will have to be removed and replaced as part of the opening- up procedure. Care musts be exercised in the removal process as one does not know the extent of damage that lies behind the damaged support units. Ten linear meters per month is typically planned for this type of opening-up and in many cases this rate is not achieved.

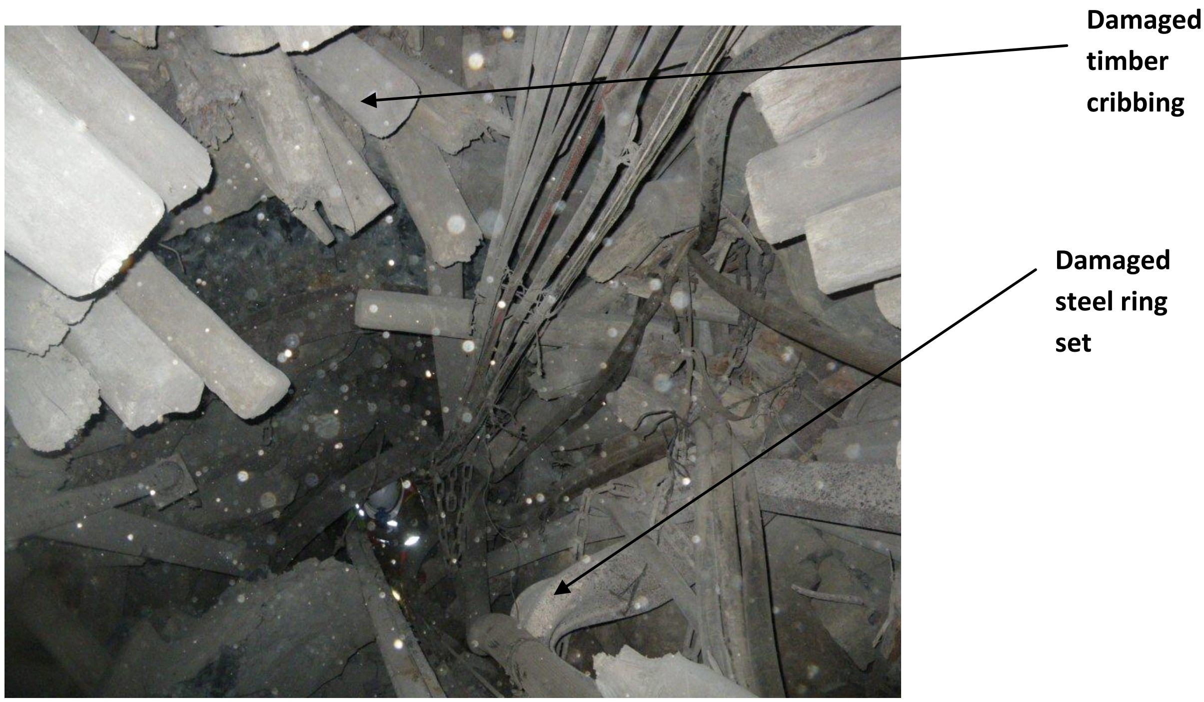


Figure 2.4.2 **Typical 100% opening-up scenario.**
(Source: Personal Collection)

The information captured on the reconnaissance visit is then used to plan the Opening-up of old workings with regards to support and equipping, and a time frame is also determined. The information available is used to assume the conditions of the workings that are further than the last point of access. Grab samples taken on the reconnaissance visit are used to give an indication of the gold grades expected from the targeted block of ground, this is also compared to the historical sampling data in order to improve the level of confidence with regards to the value expected from the targeted block of ground.

The Opening-up and reclamation process must be able to link into the production planning for the mine. If this cannot be done with a certain amount of confidence, the entire mining plan is at risk of failure due to the fact that workplaces may not be accessed and equipped in time for mining as indicated on the production plan for that specific year. It therefore requires special expertise whereby persons are able to make good judgments with regards to setting up the Opening-up, equipping and production plan for that specific workplace. The process also has to be ‘project managed’ with the necessary persons being accountable for their input and output.

Further to this, a highly skilled work team is required to actually conduct the Opening-up and equipping of these workplaces which are perceived to have relatively unsafe conditions. The process of Opening-up old workings is a laborious task as the team has to be very aware of what is happening to the environment around them as they proceed with Opening-up towards the IBOG targeted. A continuous risk assessment with regards to safety is conducted which can be time consuming as the environment changes so often. The team has to be skilled in identifying change that could affect their health and safety and address the issues of concern while at the same time trying to progress with their Opening-up process.

2.5 Conclusion

The mining of IBOG requires a large amount of investigative work which is a combination of desk top and practical work with regards to planning and execution, as noted above. It is of utmost importance that the support system employed is capable of protecting the persons that work in these areas during the extraction process. The hard work, effort and financial input that go into the planning and execution are wasted if persons are injured, and production operations have to cease. Furthermore, the loss of production from high grade IBOG jeopardizes the monthly production plan and ultimately the business plan of the mine with regards to gold output, placing more pressure on other workings to over produce in order to accommodate for the loss.

The following chapter contains an analysis of seismic, safety and production data, which was captured over a seven year period, from 2005 to 2011, in order to determine if the changes made to the support standard over this period did add value with regards to reducing injury and damage to workplaces where seismic events in excess of magnitude 1 were experienced.

Chapter 3

3. ANALYSIS OF SEISMIC, SAFETY AND PRODUCTION DATA RELATING TO SEISMICITY AT THUTHUKANI SHAFT, KDC

3.1 Introduction

A ***seismic event*** is a transient earth motion caused by a sudden failure of the earth's crust. The resulting emission and radiation of kinetic energy in the form of ground vibrations causes a sensible 'shock' or tremor. Depending on many factors, this energy may or may not result in damage to underground or surface structures. The term is applied to events ranging from largest earth quake to the smallest of events which result in a barely-detectable tremor close to the source- a range of ten orders of magnitude. The magnitude of vibration at the source is commonly described in terms of Richter magnitude (*Jager and Ryder, 1999).*

In the early 1900s it became apparent that blocks of ground left along geological structures or as a result of the mining configuration needed to be defined, classified and declared as it was found that there was an increase in the number of rockbursts and mining problems associated with these blocks of ground. A committee, the "Witwatersrand Rockburst Committee", was appointed in 1924 to report on the occurrence of rockbursts and the control thereof. The Witwatersrand Rockburst Committee published a report that was presented to Parliament in 1924. In this report a rock burst was defined as phenomenon known as "Pressure Burst," "Air Blast," "Strain Burst," "Rock Thrust," "Blistering," "Spitting," "Sudden Flaking," "Bumps," "Crumps" and "Quakes," all of which referred to the sudden rupturing of rock in situ not caused intentionally by tools or explosives. These phenomena would occur in the solid, newly opened stopes, stope faces, stope pillars, dykes, sides of a level, drive pillars, working shafts, remnants and or faults (*Le Roux, 2008).*

A ***rock burst*** is the sudden and violent disruption of rock or disturbances of excavation walls in mines, which is caused by, or accompanied by, a 'shock' or tremor (seismic event) of sufficient magnitude to cause obvious damage to excavations and support, or widespread simultaneous falls of rock; a rock burst is a consequence of mining activity *(Jager and Ryder, 1999).*

Bearing the above definitions in mind, data captured by the seismic, safety and survey departments with regards to mining of IBOG at Thuthukani Shaft, KDC over the past seven years were retrieved and analysed. Seismic data and injury statistics were compared to identify and correlate events, location, injury and damage. The analyses indicate clear trends that were established in relation to the improvements made to the support standard. Production data was limited but some important findings were determined from the data that was available. It is important to note that damage caused to workings with regards to the definition of ‘rock burst’ above was the main focus as this leads to injury. The last part of the definition states that ‘a rock burst is a consequence of mining activity’ and this was highlighted from the analysis of the production results where periods of higher production resulted in more seismic activity. The analyses and trends established are discussed in the following sub-sections.

3.2 Seismic Data Analysis

Seismic data captured for all significant events at Thuthukani Shaft with regards to the mining of IBOG in Section 22 was analyzed from 2005 to 2011. Section 22 consists of mining of the Ventersdorp Contact Reef band and a relatively small portion of the Kloof Reef band. Seismic data is captured daily and is filled in a weekly report. Data consists of capturing the origin of the seismic event by different geophones situated in various parts of the mine. These geophones have an error margin of up to thirty meters in radius. By using the information captured by three or more geophones, the origin of the seismic event can be determined with a fair amount of accuracy. The data consists of an X, Y and Z co-ordinate with regards to location of the event. This location will result in the name of the workplace where the event originated. The magnitude of the event is also captured together with the time of the event. Further to the data captured immediately for the weekly report will be the number of persons injured and the damage caused to the working places. This information will contain the number of panels affected, the face length of the panels and number of square meters and tonnage lost due to damage and rehabilitation of the workplaces affected.

Seismic data analysis commenced with firstly identifying all events in excess of magnitude 1 as recorded by the seismic system at Thuthukani Shaft over a seven year period from the beginning of 2005 to the end of 2011. These events were separated from events with magnitude less than 1. A few seismic events that occurred in the Main Reef Section were also eliminated as they did not involve the mining of IBOG. The analyses of the seismic events are described in the following sub-sections of this report.

3.2.1 Number of Seismic Events

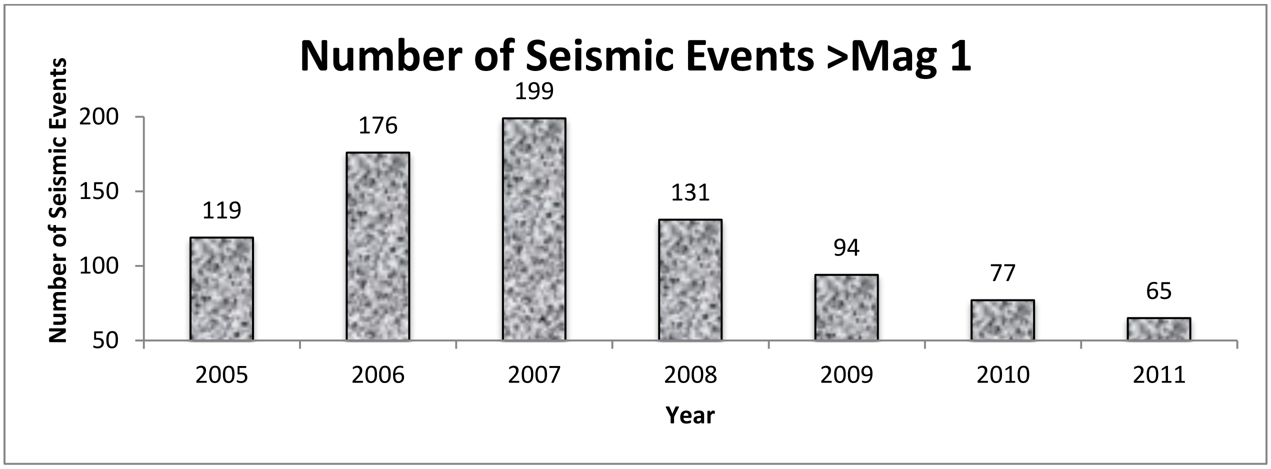


Figure3.2.1.1 **Number of seismic events with magnitude greater than 1.**
(Source: KDC Seismology Department database, 2005-2011)

Figure 3.2.1.1 indicates the number of seismic events per year with magnitude greater than 1 for Thuthukani Shaft, Section 22 (section mining isolated blocks of VCR ground). There was an increase in the number of events greater than magnitude 1 from 2005 up to 2007 with 199 events recorded in 2007. This may be attributed to the increase in production of approximately one thousand square meters experienced during this period which is discussed further in this report. The following four years clearly indicate a downward trend year on year regarding seismic events with magnitude greater than 1 with the year 2011 producing only 65 events with magnitude greater than 1. This represents a reduction of 67% as compared to the number of events in 2007. There is a subsequent decrease in production experienced during this period.

3.2.2 Number of Seismic Events with Injury and Damage to Workings

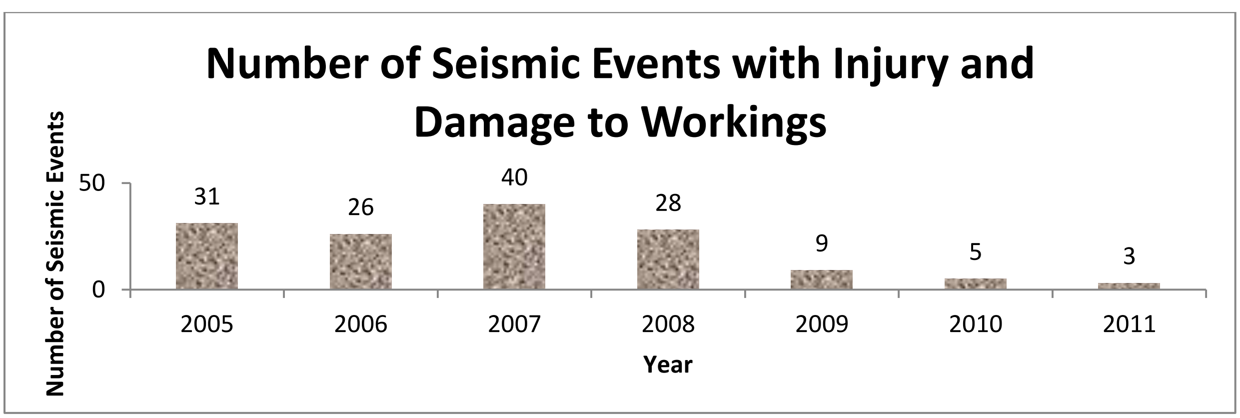


Figure 3.2.2.1 **Number of seismic events with injury and damage to workings.**
(Source: KDC Seismology Department database, 2005-2011)

Figure 3.2.2.1 indicates the number of seismic events with magnitude greater that 1 which have resulted in either injury or damage to workings, or both injury and damage to workings. The trend line, indicated on the graph after the year 2007, shows a distinct downward trend for the following four years with only three seismic events in excess of magnitude 1 resulting in injury and/or damage to the workings. This is a major improvement when comparing these results to the results obtained in 2007.

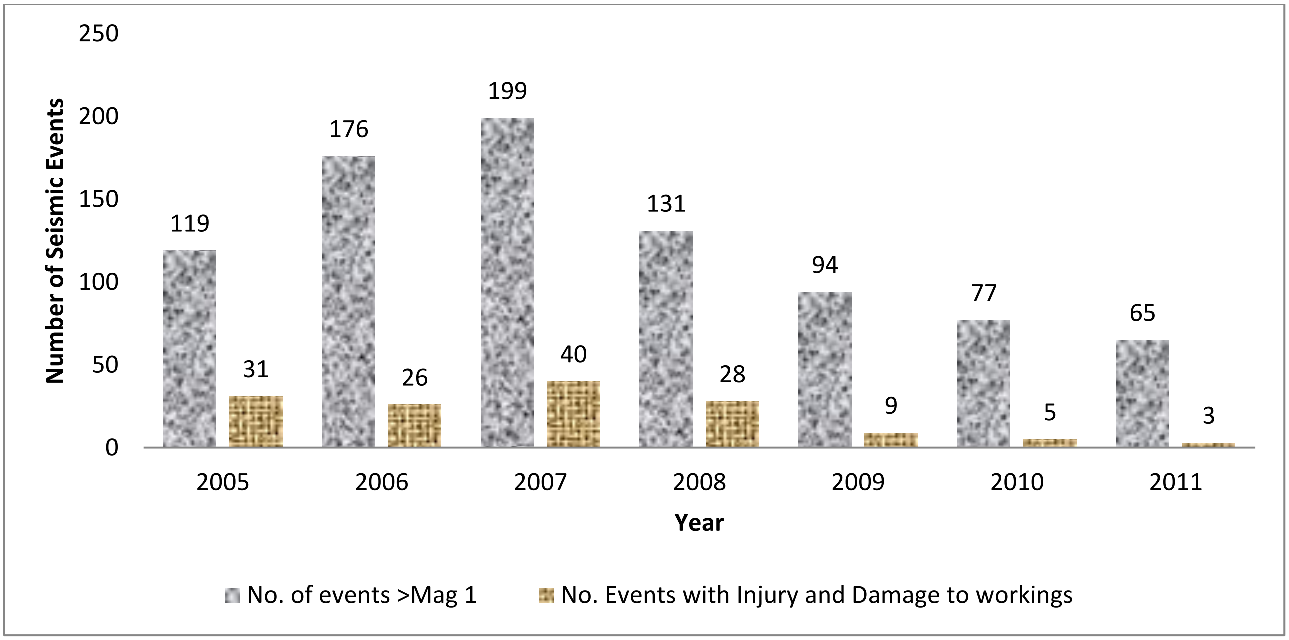


***Figure 3.2.2.2* Number of seismic events *with injury and damage* in relation to total number of seismic events with magnitude greater than 1.**
(Source: KDC Seismology Department database, 2005-2011)

Figure 3.2.2.2 indicates the number of seismic events with magnitude greater than 1 in relation to the number of seismic events with magnitude greater than 1 which have resulted in injury and damage to the workings. A common downward trend is noted where there is a decrease in the total number of events with magnitude greater than 1 from 2007 and there is also a downward trend from 2007 with regards to events with magnitude greater than 1 which have resulted in injury and damage to workings.

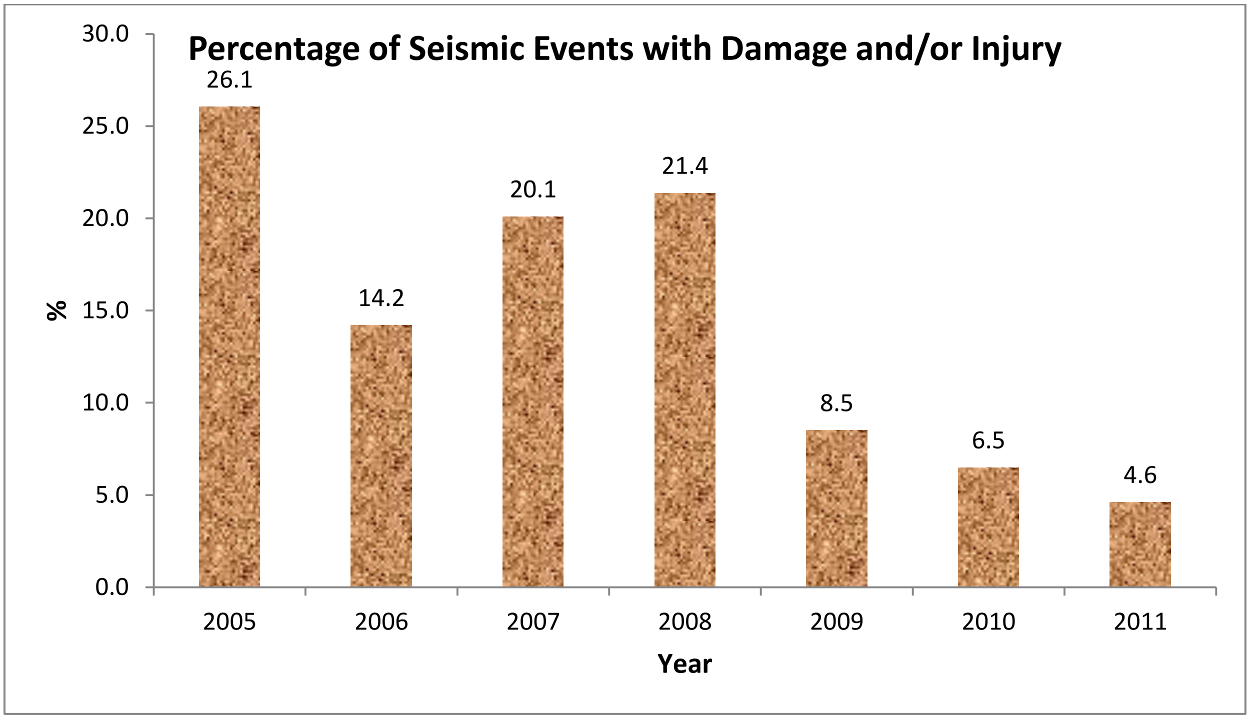


***Figure 3.2.2.3* Total number of seismic events with injury and/or damage with magnitude greater than 1, as a percentage of the total number of seismic events with magnitude greater than 1.**
(Source: KDC Seismology Department database, 2005-2011)

Figure 3.2.2.3 indicates the number of seismic events with magnitude greater than 1 which have resulted in either injury or damage to the workings, or both injury and damage to the workings, expressed as a percentage of the total number of seismic events with magnitude greater than 1 recorded for each year. A distinct downward trend can be noticed from 2008 to 2011, with only 4.6% of events in 2011 resulting in injury and/or damage to the workings. This figure was over 25% in 2005. This turning point in 2008 correlates to the statement made by the Chief Executive Officer of Gold Fields Limited that if we could not mine safely, we would not mine at all, and the subsequent improvement in the performance of the support during this period.

3.2.3 Number of Seismic Events with no Injury and Damage to Workings

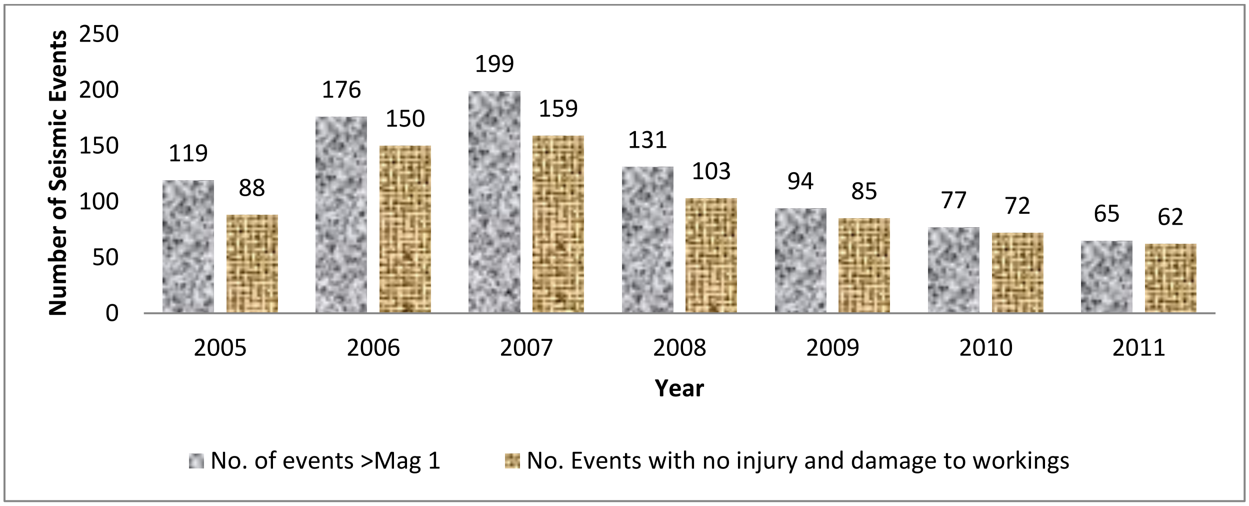


Figure 3.2.3.1 **Number of seismic events with *no injury and damage* in relation to the total number of seismic events with magnitude greater than 1.**
(Source: KDC Seismology Department database, 2005-2011)

Figure 3.2.3.1 indicates the number of seismic events with magnitude greater than 1 in relation to the number of seismic events with magnitude greater than 1 which have resulted in no injury and damage to the workings. It can be noted that from 2009, a large amount of seismic events in excess of magnitude 1 have resulted in no injury and damage to the workings.

3.2.4 Injury Analysis

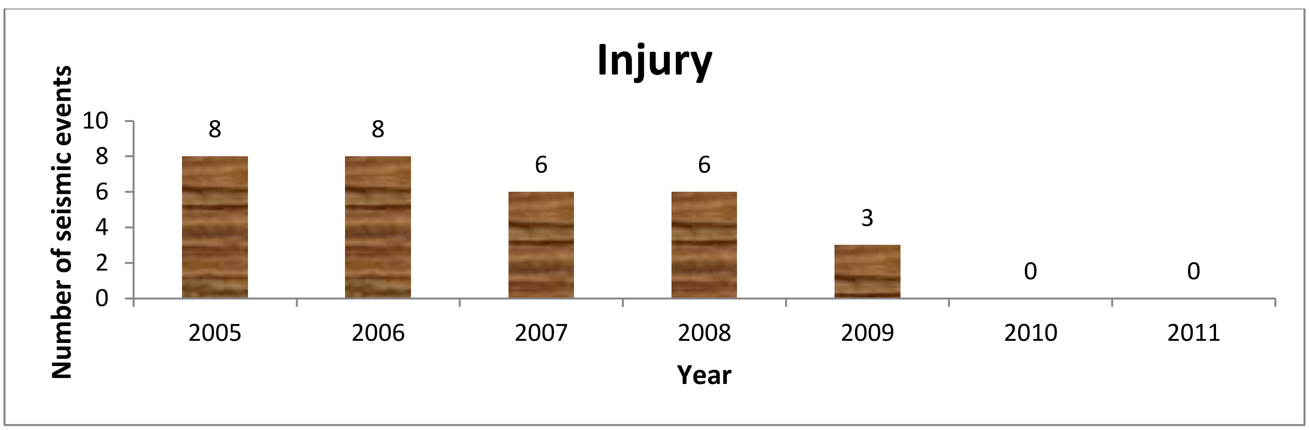


Figure 3.2.4.1 **Number of seismic events with magnitude greater than 1 that resulted in injury.**
(Source: KDC Safety and Seismic Department Database, 2005-2011)

Figure 3.2.4.1 indicates the number of seismic events with magnitude greater than 1 that resulted in injury. There is a definite downward trend noted from 2005 to 2011 with no events with magnitude greater than 1 resulting in injuries reported in 2010 and 2011.

This is very encouraging statistics, because there were 77 and 65 events with magnitude greater than 1 during 2010 and 2011, respectively which, although, is a smaller number of events compared to the previous years, have resulted in no injury to persons.

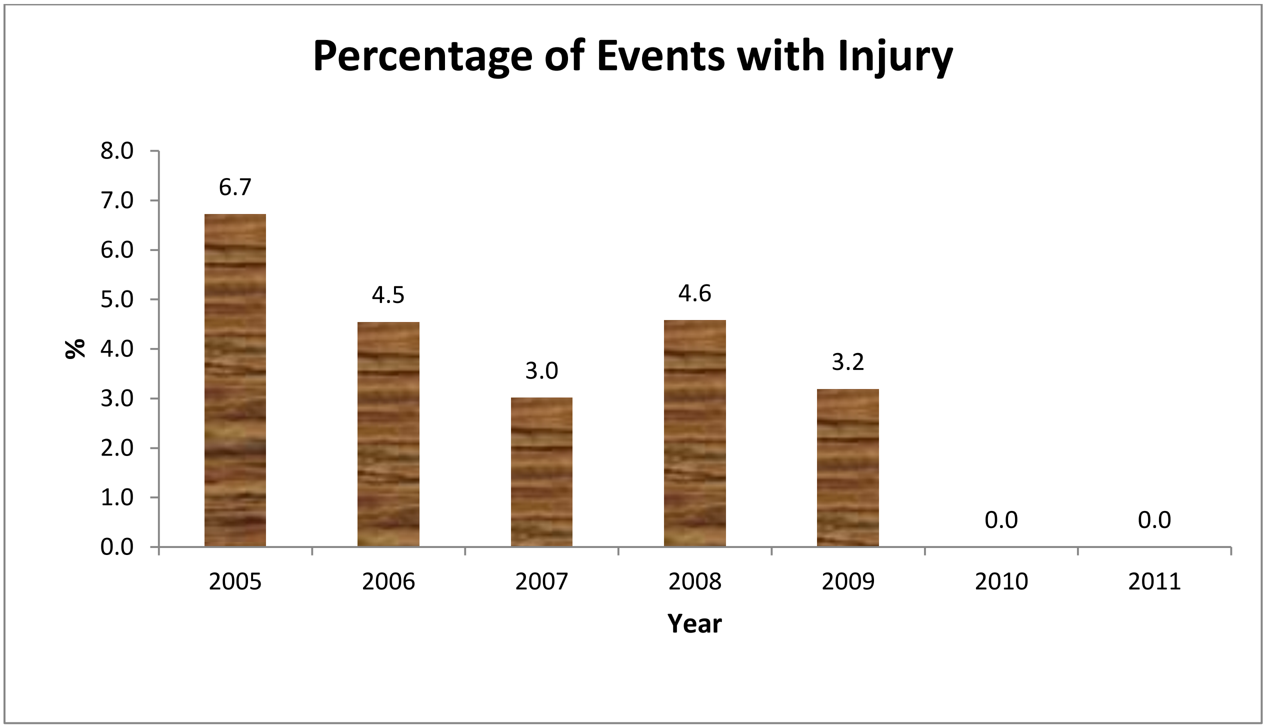


Figure 3.2.4.2 **Percentage of seismic events with injury against total number of seismic events.**
(Source: KDC Safety and Seismic Department Database, 2005-2011)

Figure 3.2.4.2 indicates the number of seismic events with magnitude greater than 1 that resulted in injury expressed as a percentage of the total number of seismic events recorded, with magnitude greater than 1. A downward trend can be noted from 2005 to 2011 with the period between 2008 and 2011 showing a distinct downward trend.

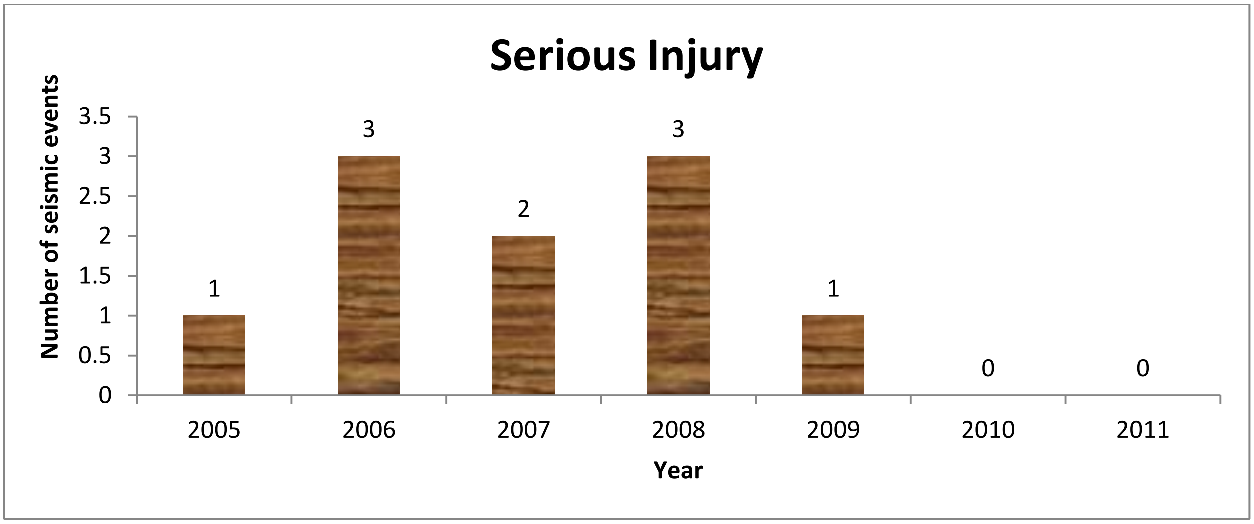


Figure 3.2.4.3 **Number of seismic events with serious injury.**
(Source: KDC Safety and Seismic Department Database, 2005-2011)

Figure 3.2.4.3 indicates the number of seismic events with magnitude greater than 1 which have resulted in serious injury. A downward trend can be noted from 2008 with no events with magnitude greater than 1 resulting in serious injury in 2010 and 2011.

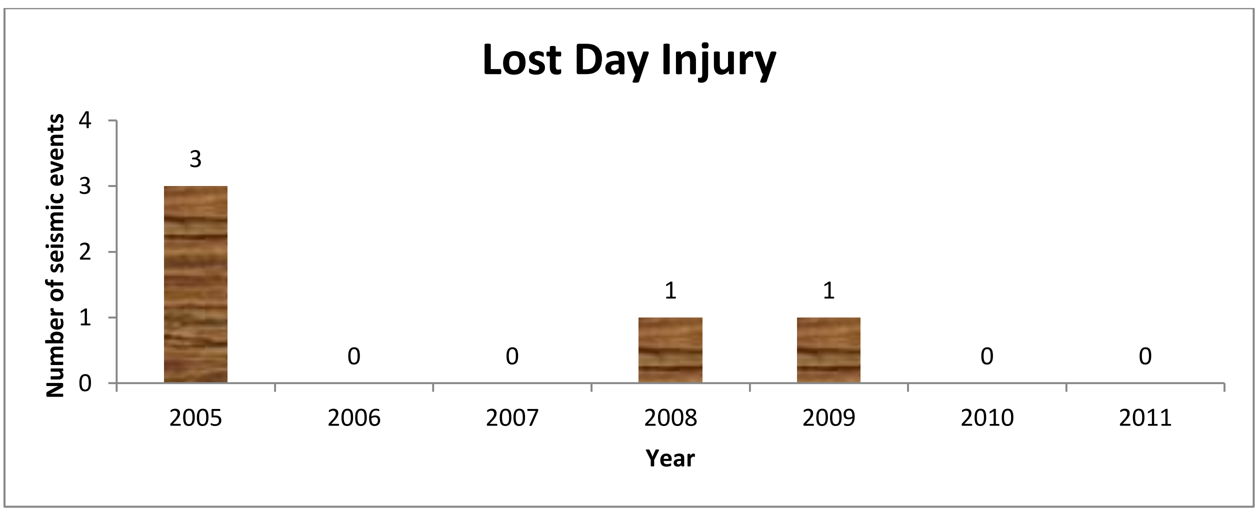


Figure 3.2.4.4 **Number of seismic events with lost day injury.**
(Source: KDC Safety and Seismic Department Database, 2005-2011)

Figure 3.2.4.4 indicates the number of seismic events with magnitude greater than 1 that have resulted in lost day injuries, from 2005 to 2011.

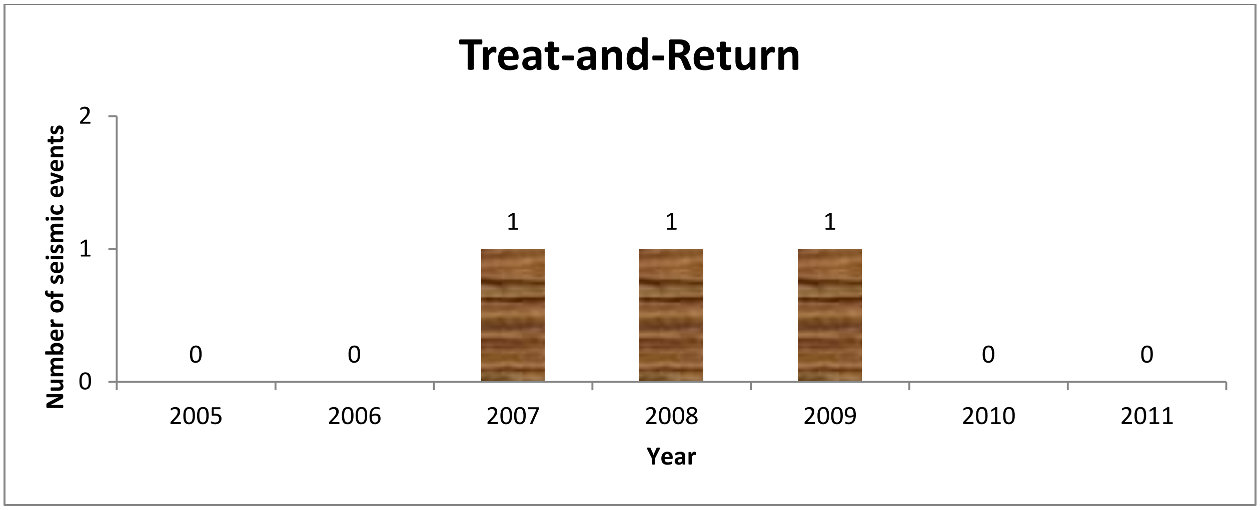


Figure 3.2.4.5 **Number of seismic events with treat -and -return injury.**
(Source: KDC Safety and Seismic Department Database, 2005-2011)

Figure 3.2.4.5 indicates the number of seismic events with magnitude greater than 1 that have resulted in treat and return injuries, from 2005 to 2011.

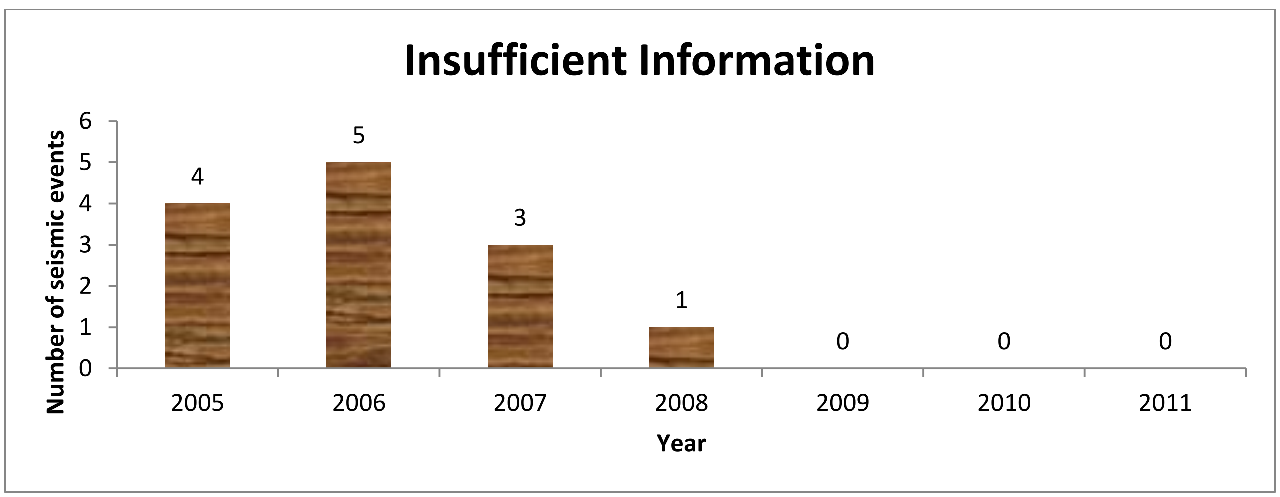


Figure 3.2.4.6 **Number of seismic events with injury - insufficient information.**
(Source: KDC Safety and Seismic Department Database, 2005-2011)

Figure 3.2.4.6 indicates the number of seismic events with magnitude greater than 1 where details relating to the event have not been captured properly.

It can be noted, from the injury graphs above, that from 2008 there is a vast improvement with regards to injury and the reporting structure. There is a marked reduction in the number of events resulting in injury and all information reported to the safety department correlates to the information presented by the seismic department.

The year 2008 was a turning point for proactively improving the support standard and reducing injury and damage to workplaces. If the statistics, which are presented in ***Figure 3.2.4.6***, were to be added to those that were analysed earlier, it would just make the period from 2005 to 2007 look even worse and show a greater improvement from the year 2008 to 2011. It would not affect the implication that an improvement with regards to injury and damage to workplaces has been noted during the period 2005 to 2011at Thuthukani Shaft, KDC.

3.2.5 Damage Analysis

A major improvement occurs from 2008, with 28 events with damage to workings, to 2009, with only 7 events resulting in damage to workings. The following years maintain this low number.

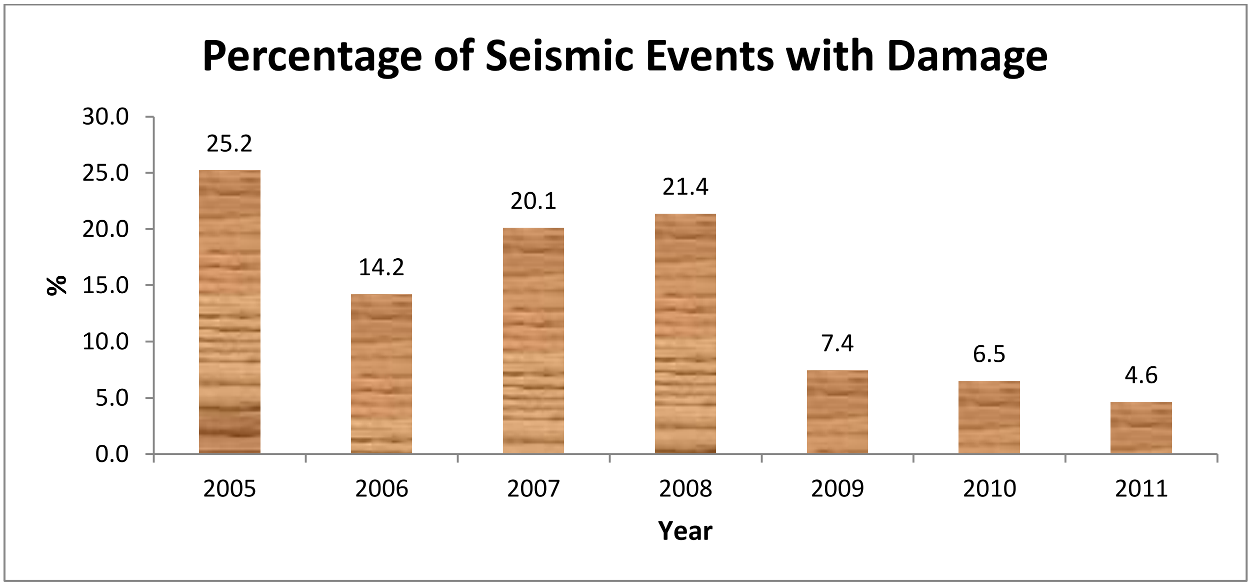


Figure 3.2.5.1 **Number of seismic events with damage to workings as a percentage of the total number of seismic events**.
(Source: KDC Seismology Department database, 2005-2011)

Figure 3.2.5.1 indicates the number of seismic events, with magnitude greater than 1, that have resulted in damage to the workings, expressed as a percentage of the total number of seismic events with magnitude greater than 1 recorded for each year. A marked improvement can be noted from 2008 to 2011 with only 4.6% of the total number of events in 2011 resulting in damage to the workings. This graph indicates a distinct turning point in 2008 and a constant downward trend from this point forward.

Figure 3.2.5.2 indicates the number of seismic events with magnitude greater than 1 which have resulted in no damage to the workings in relation to the total number of seismic events with magnitude greater than 1 and ***Figure 3.2.5.3*** indicates the number of seismic events with magnitude greater than 1 which have resulted in no damage to the workings expressed as a percentage of the total number of seismic events with magnitude greater than 1 recorded for each year.

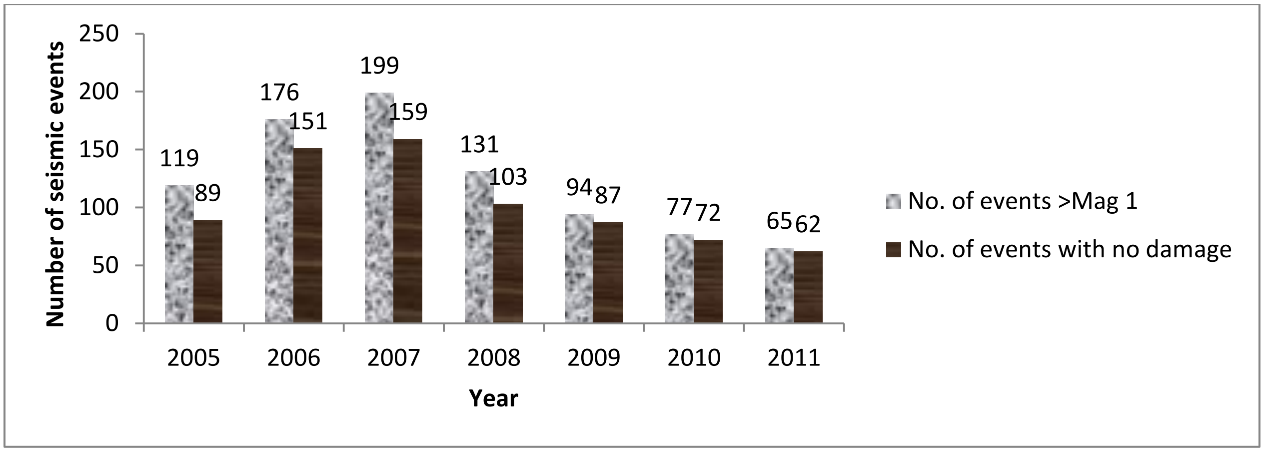


Figure 3.2.5.2 **Number of seismic events with no damage in relation to total number of events.**
(Source: KDC Seismology Department database, 2005-2011)

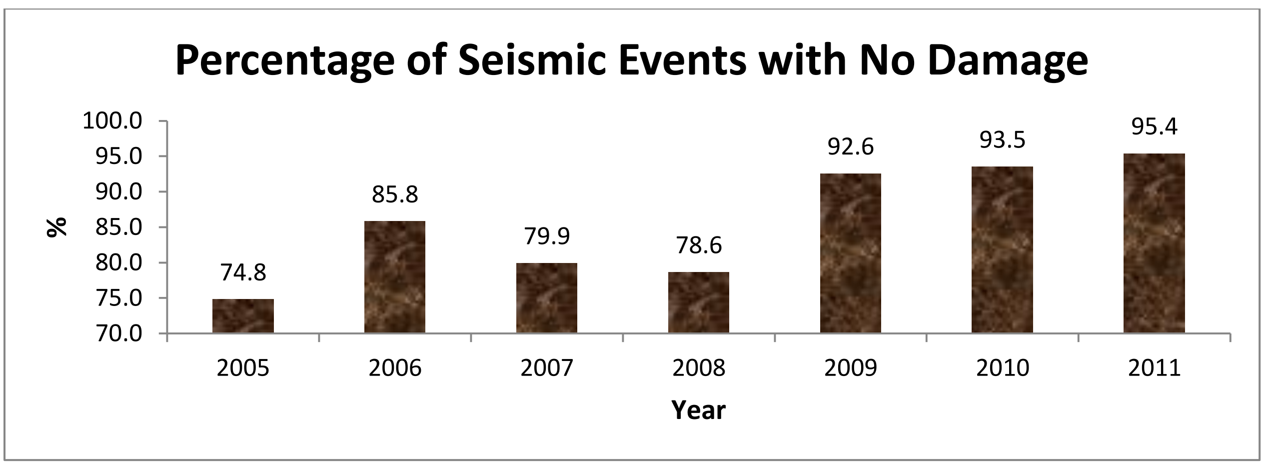


Figure 3.2.5.3 **Number of seismic events with no damage as a percentage of total number of events.**
(Source: KDC Seismology Department database, 2005-2011)

It can be noted that the there is a marked improvement in 2009, 2010 and 2011 with regards to working places not being damaged from seismic events with magnitude greater than 1 with 2011 recording 95.4 % of seismic events with magnitude greater than 1 resulting in no damage to workings. This is almost a 20% improvement, jumping from a 78.6% in 2008.

3.2.6 Criteria used for damage ranking

The information contained in the weekly seismic reports indicated the square meters and tonnage that was lost due to damage caused by seismic activity. This loss accounts for the time taken to re-establish the workplace, in which time the blasting operations in the panel ceases. This information was used in conjunction with the criteria below to determine the number of blasts lost, which is dependent on the length of the panel. The number of blasts lost is then used to calculate an approximate number of days lost which is the number of days that was required to rehabilitate the workplace and re-establish the working face. The criteria that were used are as follows:

1. Working places with face lengths from zero meters (0m) to twenty meters (20m) were assumed to be blasted thrice a week.
2. Working places with face lengths greater than twenty meters (20m) were assumed to be blasted twice a week.
3. A face advance per blast was assumed to be one meter (1m)
4. Square meters lost due to seismic damage, as recorded in the seismic reports, were divided by the face length to determine the face advance and the number of blasts lost.

Damage categorization was calculated as follows:

a. Zero (0) to two(2) days lost – Minimum Damage
b. Three (3) to five(5) days lost – Average Damage
c. More than five (5) days lost – Major Damage

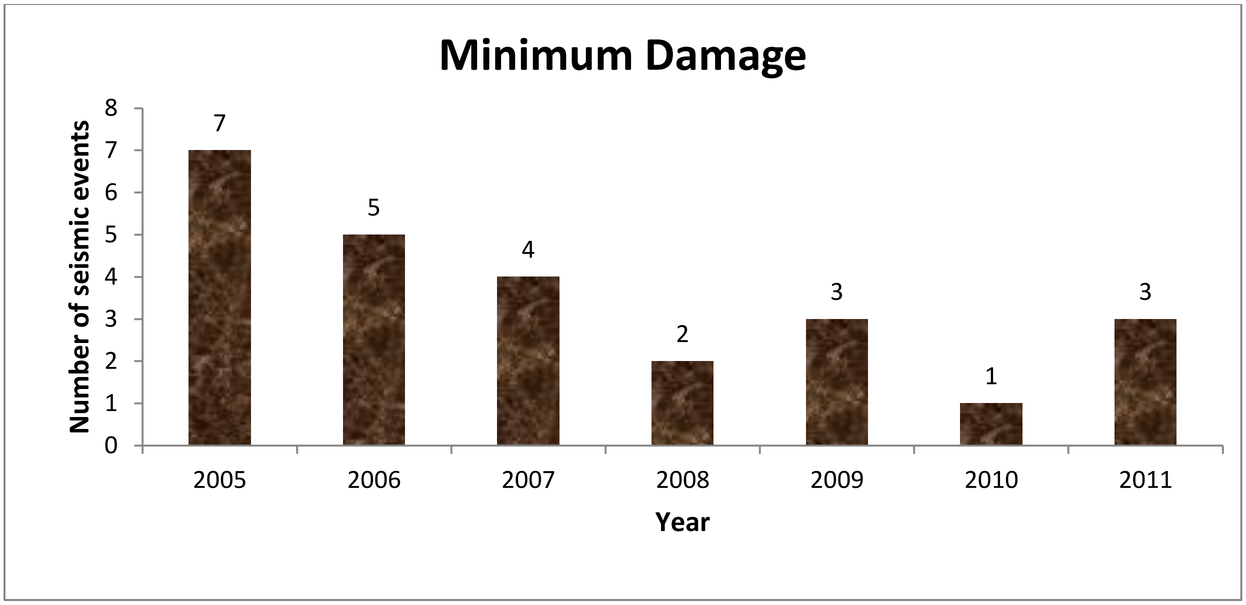


Figure 3.2.6.1 **Number of events with minimum damage to workings.**
(Source: KDC Seismology Department database, 2005-2011)

Figure 3.2.6.1 indicates the number of seismic events with magnitude greater than 1 which have resulted in minimum damage to workings. A clear downward trend can be noted from 2005 to 2008 and a more or less static situation from 2008 to 2011, although the number of events are quite low, between 1 and 3 events resulting in minimum damage to workings.

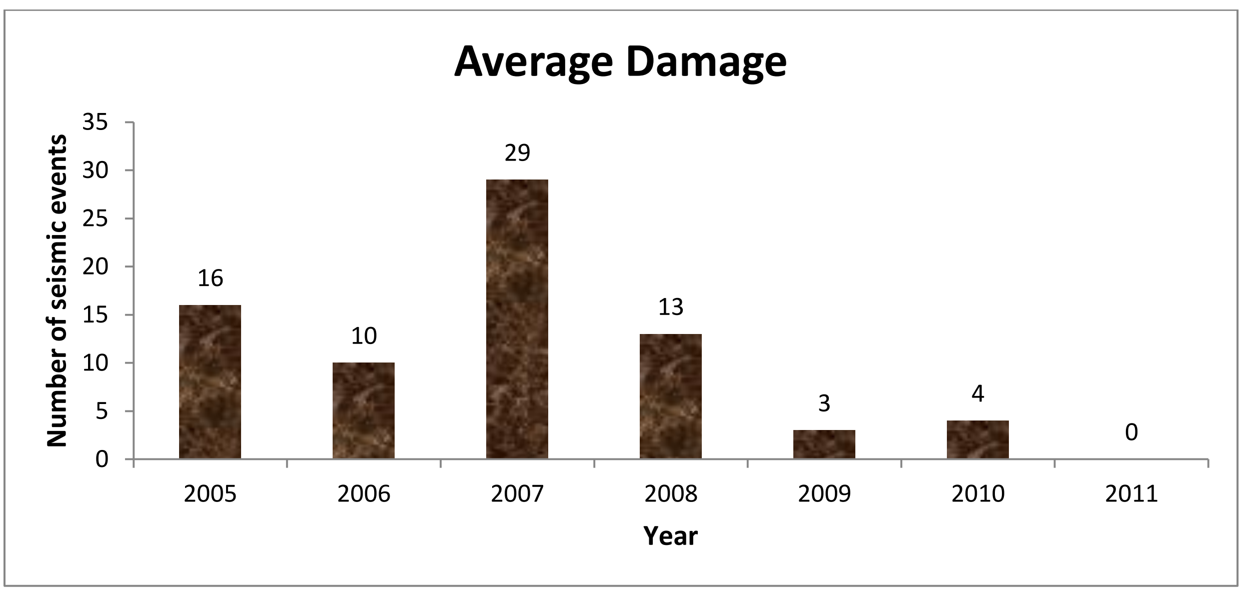


Figure 3.2.6.2 **Number of events with average damage to workings.**
(Source: KDC Seismology Department database, 2005-2011)

Figure 3.2.6.2 indicates the number of seismic events with magnitude greater than 1 which have resulted in average damage to workings. A downward trend is also noted in this graph with the year 2011 producing no events with average damage to workings. A major improvement can be noted from the year 2008.

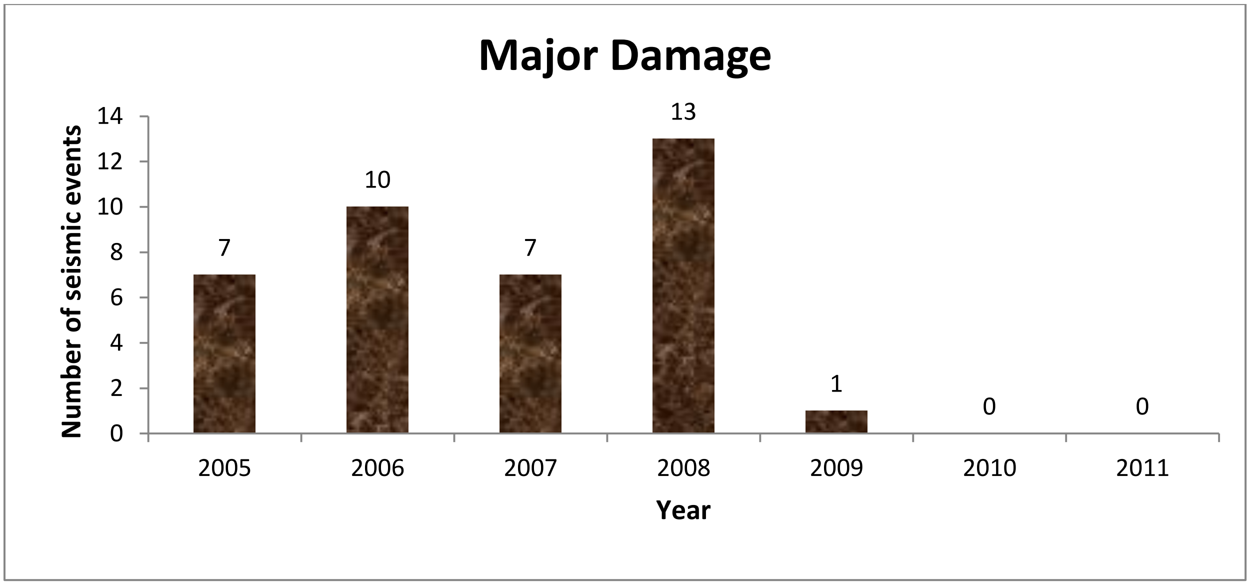


Figure 3.2.6.3 **Number of events with serious damage to workings.**
(Source: KDC Seismology Department database, 2005-2011)

Figure 3.2.6.3 indicates the number of seismic events with magnitude greater than 1 which have resulted in serious damage to workings. A marked improvement can be noted in the preceding 3 years with one event with serious damage in 2009 and no serious damage in 2010 and 2011. An upward trend in serious damage can be noted from 2005 to 2008, with 2008 peaking with 13 events with serious damage to workings. 2008 was also the turning point and a distinct improvement from that point forward is noted. This can be attributed to the major focus and drive that was injected into improving the support standard, preventing fall of ground accidents and creating safer working environments.

3.3 Production Analysis

Monthly production results for Thuthukani shaft was retrieved from 2005 to 2011, however, workplace specific results were not retrievable. Monthly total results for section 22 (section mining isolated blocks of VCR) were retrieved and analysed as follows.

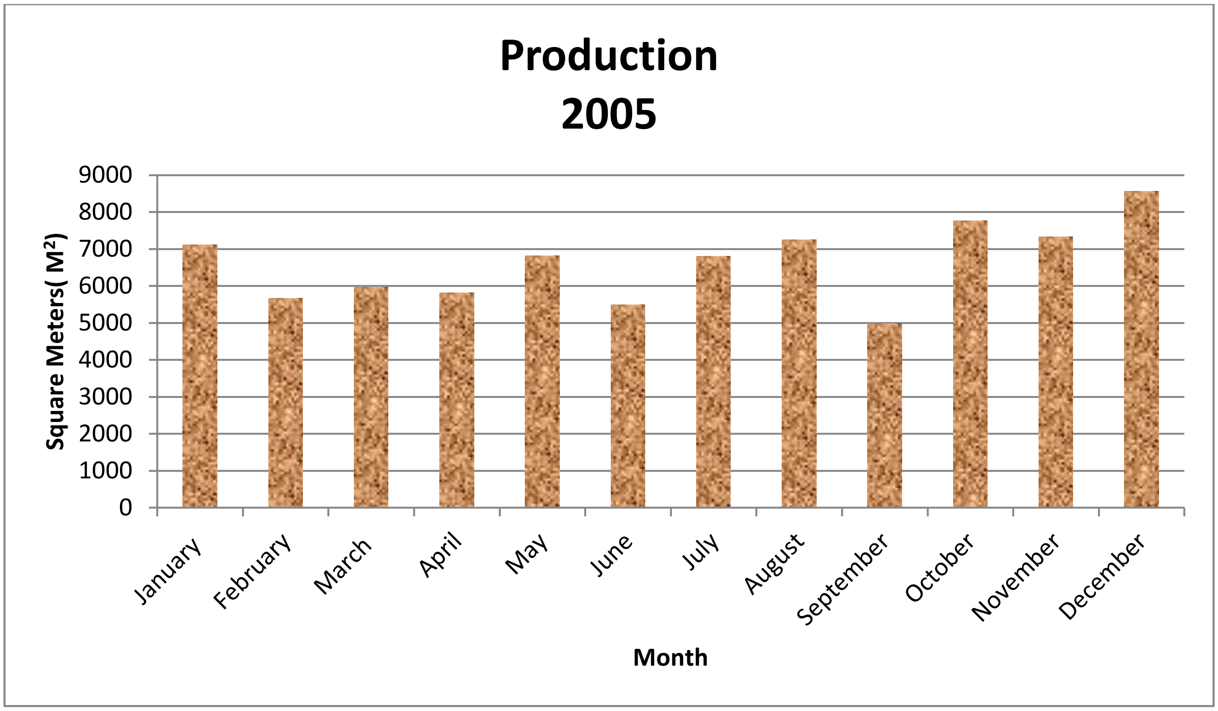


Figure 3.3.1 **Monthly production results for 2005.**
(Source: KDC, BU5 Survey Department Database)

The monthly average production result for 2005 was approximately 6500 square meters.

Production output was hampered in September 2005 due to the halting of blasting operations in certain sections of the mine due to unsafe environmental conditions. These conditions had to be rectified and audited prior to production operations continuing.

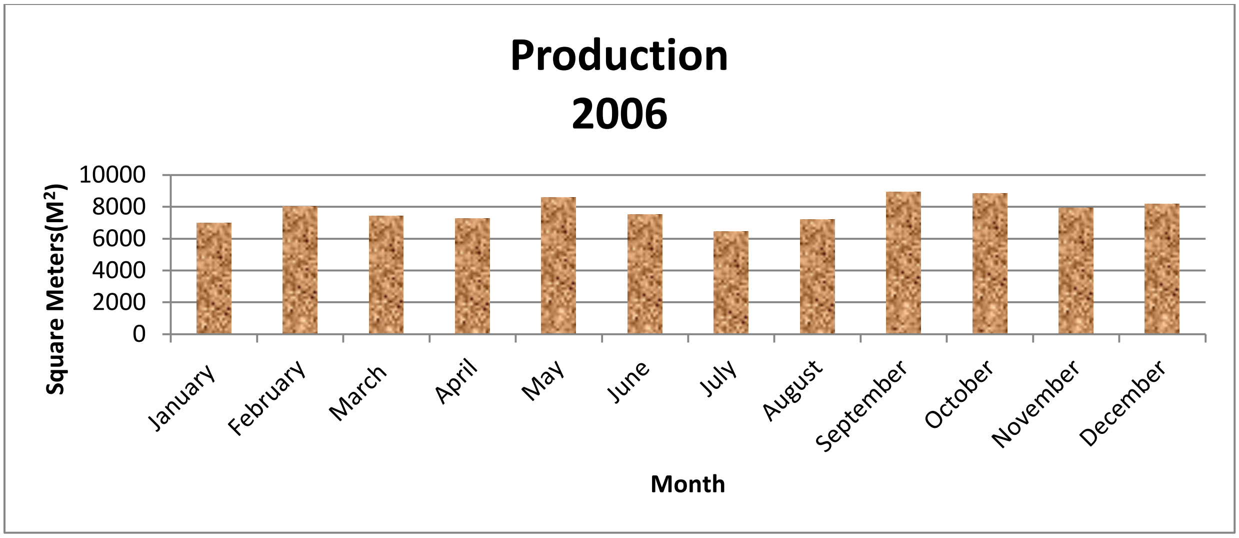


Figure 3.3.2 **Monthly production results for 2006.**
(Source: KDC, BU5 Survey Department Database)

The monthly average production result for 2006 was approximately 7500 square meters. Production output was hampered in July 2006 as a result of two notices for the suspension of work operations issued by the Department of Minerals and Energy (DME) in the form of a Section 54, for sub-standard work conditions. Conditions had to be first rectified, audited and presented to the DME prior to production operations continuing in the affected workplaces.

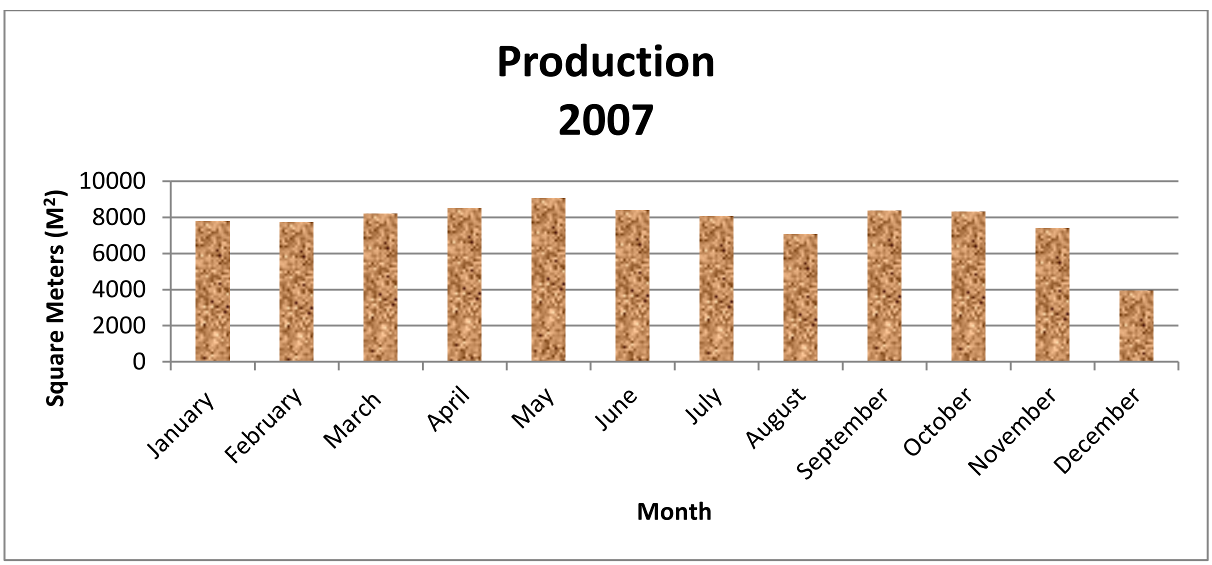


Figure 3.3.3 **Monthly production results for 2007.**
(Source: KDC, BU5 Survey Department Database)

The monthly average production result for 2007 was approximately 7500 square meters. Production was affected negatively in December 2007 due to production being halted in order to rectify sub-standard work conditions in certain parts of the mine as identified by the inspector of mines from the Department of Mineral Resources.

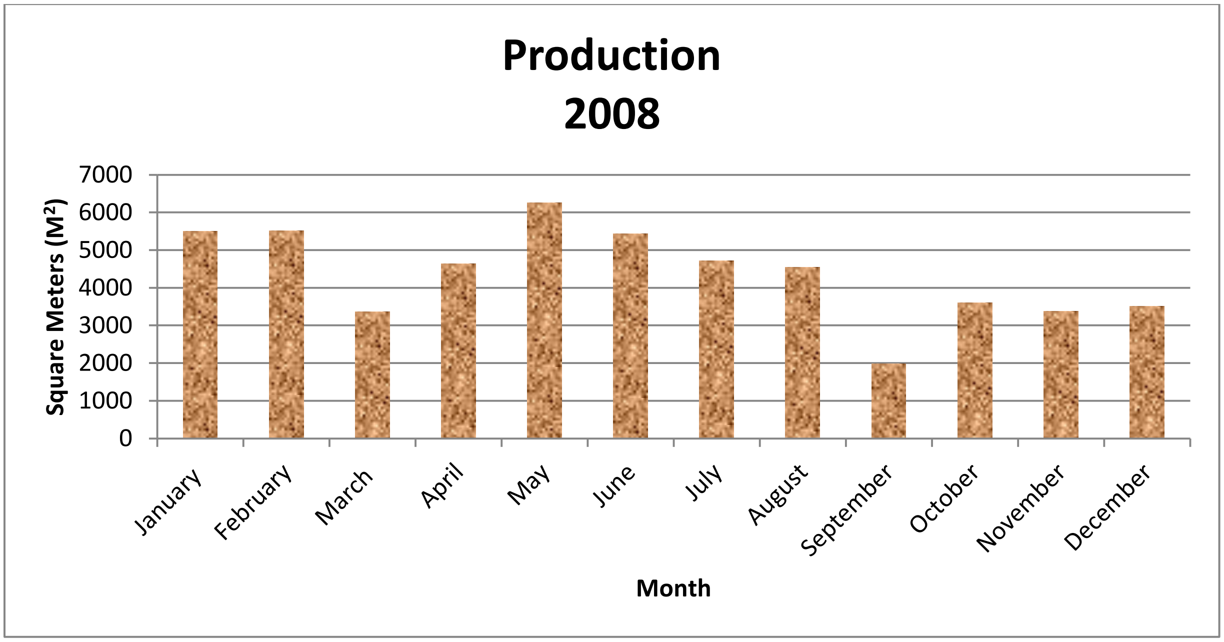


Figure 3.3.4 **Monthly production results for 2008.**
(Source: KDC, BU5 Survey Department Database)

The monthly average production result for 2008 was approximately 4500 square meters. Production output was hampered in March 2008 due the suspension of work operations until environmental conditions were rectified to within legal limits and track work brought up to mine standard in certain parts of the mine as identified by the inspector of mines from the Department of Mineral Resources(DMR) and recorded on a Section 54 document. Production was also affected negatively, in September 2008 due to a fall of ground fatal that occurred at Kloof 3 Shaft. A section 54 document was issued by the Department of Mineral Resources halting operations until an investigation into the fatal accident had been conducted and presented to the DMR. During the halting of all blasting operations at the mine, workplaces had to be inspected, all sub-standard work conditions had to be rectified and audited prior to work commencing.

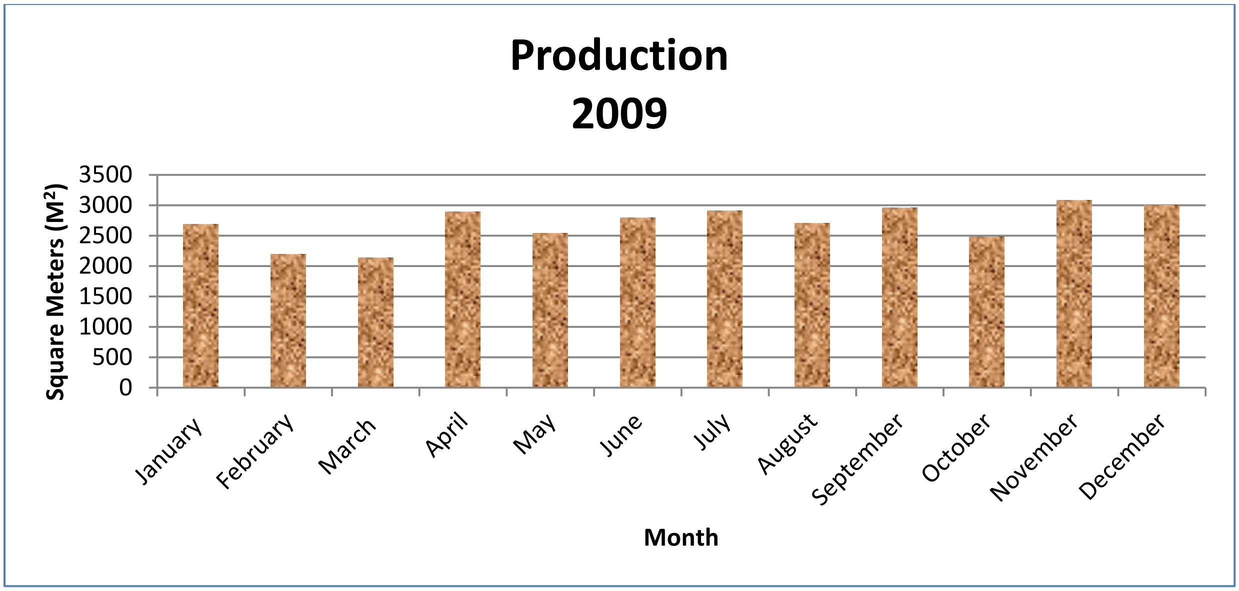


Figure 3.3.5 **Monthly production results for 2009**.
(Source: KDC, BU5 Survey Department Database)

The monthly average production result for 2009 was approximately 2500 square meters. Production output was affected negatively in February 2009 due to a fatal accident at Kloof 3 Shaft where a seismic event of magnitude 2.5 occurred. Production was halted by the Department of Mineral Resources until investigations and findings were conducted and presented to the Department of Mineral Resources.

Production was further halted in March 2009 due to a fatal accident that occurred at Thuthukani Shaft where an employee had fallen off the headgear. All shaft operations were halted until the investigation into the accident was conducted and presented to the Department of Mineral Resources. Although production operations were not stopped, material and equipment required for mining operations were not allowed to be transported, forcing mining operations to stop.

Production was affected negatively in October 2009 due to the stopping of all tramming operations resulting from a tramming related accident that took place at Kloof 8 Shaft. Investigations had to be conducted and corrective action implemented to prevent a similar type of accident.

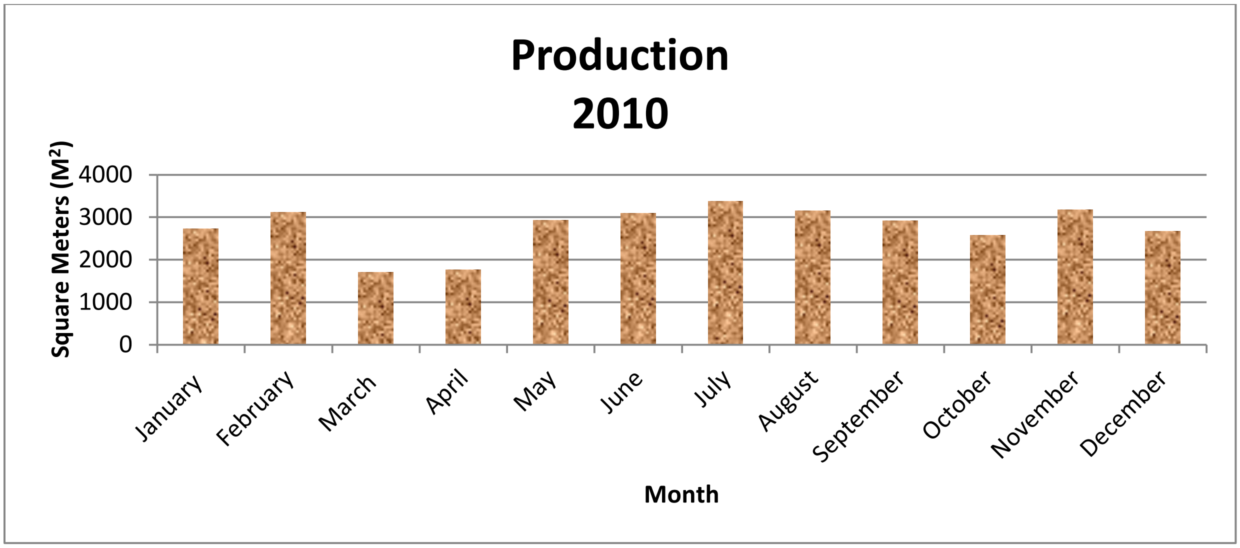


***Figure 3.3.6* Monthly production results for 2010**.
(Source: KDC, BU5 Survey Department Database)

The monthly average production result for 2010 was approximately 2500 square meters. Production was affected negatively in March and April 2010 due to shaft repair work that was required in March when a collapsed pump column had to be replaced. A fall of ground fatal accident at Kloof 4 Shaft and a separate accident at Kloof 7 Shaft resulted in operations being halted in April resulting in poor production delivery during this month.

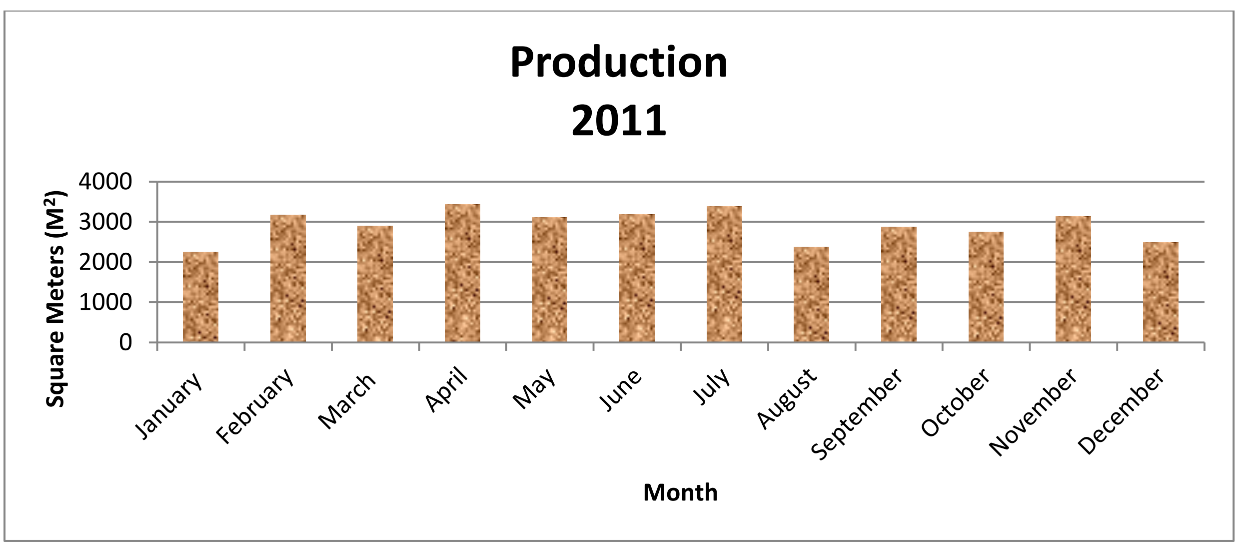


***Figure 3.3.7* Monthly production results for 2011**.
(Source: KDC, BU5 Survey Department Database)

The monthly average production result for 2011 was approximately 3000 square meters. Production was affected negatively in some months due to stoppages initiated by the Department of Mineral Resources for the entire Kloof Driefontein Complex and strike action.

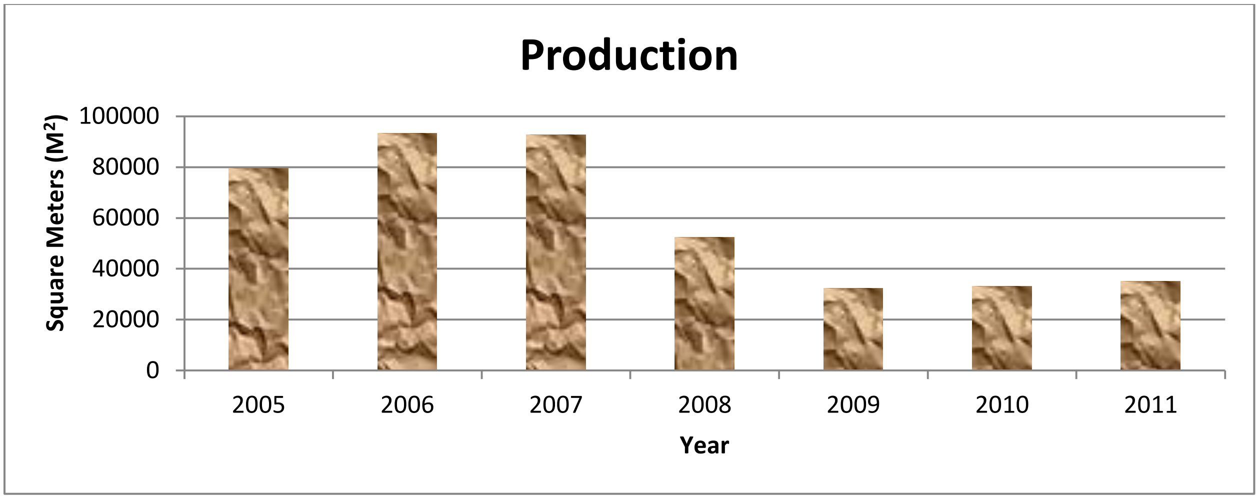


Figure 3.3.8 **Annual production results from 2005 to 2011.**
(Source: KDC, BU5 Survey Department Database)

The Annual production results graph, in ***Figure 3.3.8*** indicates a drastic decrease in production from approximately 85 000 square meters, on average, up to 2007 to approximately 35 000 square meters up to 2011. This drop in production was attributed to the shaft repair work that commenced in 2008 where the working capacity of the shaft was reduced to 50%. Access to the workings was gained via a secondary shaft and the lengthy travel and transport times led to a reduction in production. Future production is planned at approximately 35 000 square meters per annum for the life of mine for section 22 and since production levels are not planned to increase, it is expected that the total number of seismic events per annum should also not increase.

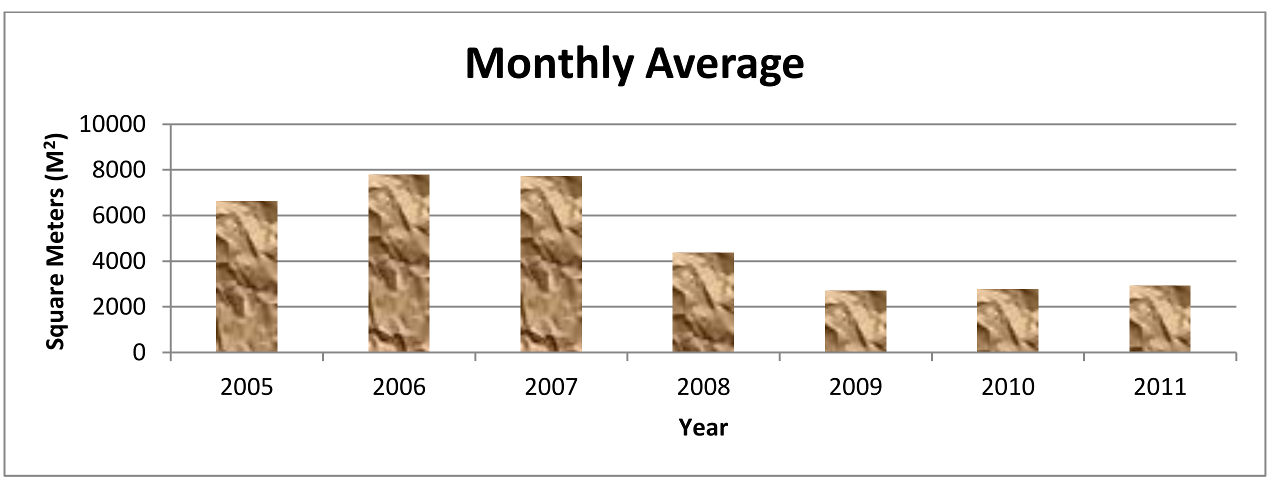


Figure 3.3.9 **Monthly average production per year**.
(Source: KDC, BU5 Survey Department Database)

Figure 3.3.9 indicates the monthly average production per year and clearly indicates the drop in production when the shaft repair program commenced.

3.4 Conclusion

Figure 3.2.1.1 indicates the number of seismic events in excess of magnitude 1 per year and when comparing the graph to the graph indicating the monthly average production per year, a relationship can be noted between production and number of seismic events. A reduction in production led to a reduction in the number of seismic events in excess of magnitude 1.

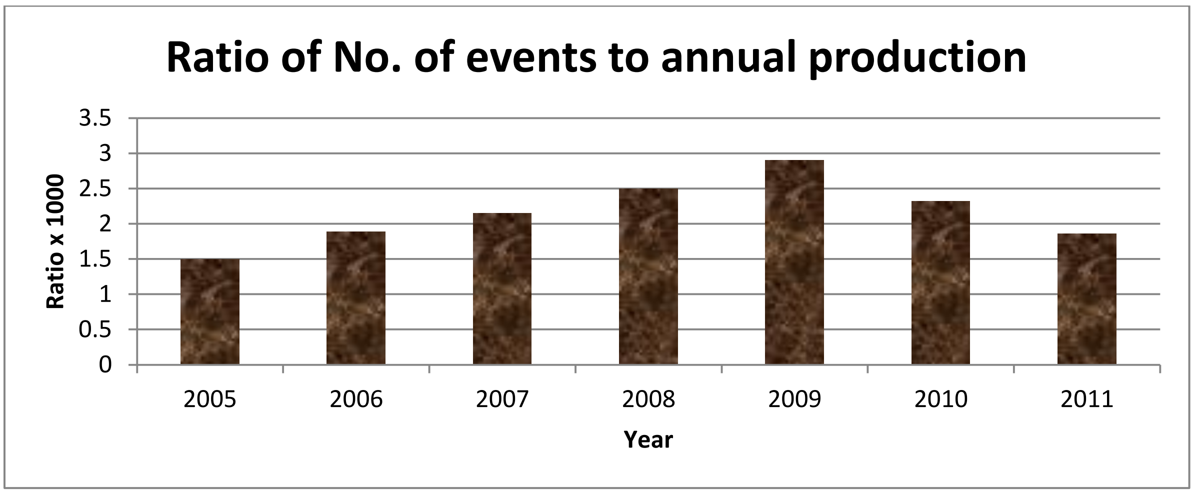


Figure 3.4.1 **Ratio of number of seismic events to annual production**
(Source: KDC, BU5 Survey and Seismic Department Database)

Figure 3.4.1 indicates the ration of the number of seismic events in excess of magnitude 1 to the annual production results for each year which is multiplied by 1000. An increase in the number of events per square meter extracted can be noted from 2005 up to 2009 after which a consistent decrease can be noted from 2009 to 2011.

Although there was a decrease in the number of seismic events in excess of magnitude 1 which appear to be related to the apparent decrease in production output over the years, large magnitude events did occur. ***Figure 3.2.5.1*** indicates clearly that a decreasing percentage of events, annually, resulted in damage to workings, with 2011 producing 4.6% of events in excess of magnitude 1, resulting in damage to workings. So, although the number of seismic events in excess of magnitude 1 had decreased over the recent years, the damage caused by these events had also decreased.

Figure 3.4.2 indicates the Major Damage Forecast which is the number of events, with magnitude greater than 1, which could have resulted in major damage from 2009 to 2011. This graph was generated by using the average production to number of events with major damage ratio from 2005 to 2008, and applying it to the production delivered for each year from 2009 to 2011. By applying a minimum factor of five days lost per event with major damage, it can be noted that fifteen to twenty days would have been lost each year. Assuming a 23 shift production month, this equates to almost losing the entire months production. Assuming that an average of twenty kilograms of gold is produced from each panel, this equates to approximately twenty kilograms of gold being lost each year. Assuming a gold price of R 400 000/kg, this equates to a loss in revenue of R 8 000 000.

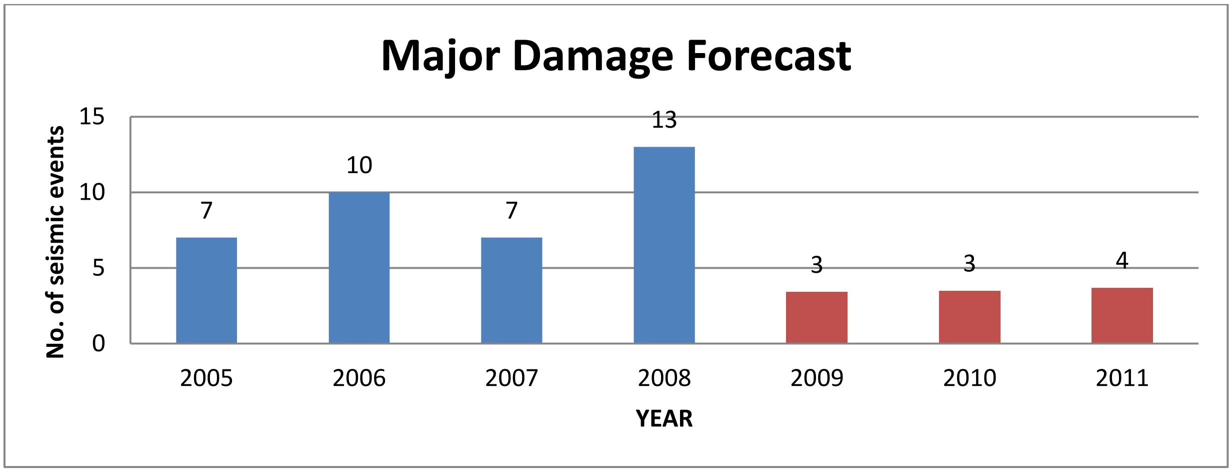


Figure 3.4.2 **Number of seismic events with major damage - forecast.**
(Source: KDC, BU5 Survey and Seismic Department Database)

However, post 2008 this trend did not occur, with only one event with major damage being recorded in 2009 and no events with major damage being recorded in 2010 and 2011.

By using the same criteria above and applying it to the 2008 results, we have 13 events with major damage. Assuming that 5 days were lost per event, a total of 65 days is lost for the year. Assuming a 23 day month, a total of 2.83 months is lost. Assuming a loss of 20kg of gold per month, a total of 56.5 kg of gold is lost for the year. Assuming a gold price of R 400 000/kg, this equates to a loss of over R 22 million for the year in direct revenue. This excludes the cost of labour and material required to re-establish the damaged panels and also excludes the cost of injury.

Chapter 4

4. SUPPORT SYSTEMS AND DESIGN IMPROVEMENTS OVER TIME

The purpose of this chapter of the research report is to investigate the support systems used in 2005 and how the design of this system has evolved over time to the one that is in current use. Improvement of the support system will be noted, sequentially, in an effort to improve the safe working conditions in relation to seismic related damage to working places. These improvements to the support standards will be discussed in relation to the analysis conducted with regards to seismic events in excess of magnitude 1 that were experienced at Thuthukani Shaft over the past seven years.

4.1 1999 Panel Support Layout (A.S.P.3.10) (APPENDIX A)

- In 1999 the Stoping and Support Method standard was revised and the support layout for panels on composite packs and elongates included the use of pack support units and pre-stressed elongate support units with no load spreader headboards.

- The elongates spacing on dip was 1.5 m and on strike an alternating sequence of 1.8 m and 1.3 m was designed, measured skin-to-skin.

- The distance from the face to the last line of elongate support was not to exceed 2.8 m

- Pack spacing was not to exceed 3 m on dip and 2 m on strike, measured skin-to-skin.

- The distance from the last line of pack support to the face was not to exceed 3.6 m before the blast.

- A line of temporary support was to be installed if the elongate to face distance exceeded 1.4 m. The dip spacing of these temporary support units was not to exceed 1.2 m.

4.2 2005 Panel Support Layout (A.S.P.2.1.30A) (APPENDIX B)

- The support standard included the use of pack support units and pre-stressed elongate support units with load spreader headboards.
- The elongate spacing was 1.5 m on dip and strike, measured skin -to- skin.
- The distance from the face to the last line of elongate support before the blast was not to exceed 2.4 m
- Pack spacing was not to exceed 2.3 m on dip and strike, measured skin- to- skin.
- The distance from the face to the lasts line of packs was not to exceed 4.8 m after the blast.
- Where stoping widths exceeded 2.2 m, double packs were to be installed with the long axis on dip.
- A line of temporary support was to be installed if the elongate to face distance exceeded 1.5 m. The dip spacing of these support units was not to exceed 1.5 m and no person was to work further than 1 m from support

4.3 2009 Panel Support Layout (APPENDIX C)

- In 2009, the support standard was changed to improve support coverage. In working places where only water jetting was used to clean, additional elongates with load spreader headboards were to be installed on dip midway between two lines of packs.
- In working a place where scraper cleaning was carried out additional elongates were to be installed midway between the rows of packs with load spreader headboards in the direction of mining.
- Pack support spacing was reduced to 1.9 m on strike and 1.5 m on dip. Umbrella packs were introduced and were to be installed at the discretion of the responsible line management depending on local ground conditions.
- Elongate spacing was also reduced to 1.5 m on strike and 1.3 m on dip.
- **Pre-conditioning** was also introduced as a standard on all working panels.

4.4 2011 Managerial Support Instruction (M.S.I.2011/01A) (APPENDIX D)

In 2011, the support standard was revised to include **in-stope netting**. Investigations into rock fall accidents concluded that rocks were falling between support units in the vicinity of the working face and injuring persons. Furthermore, in seismic related accidents, most of the damage and injury was cause by fractured rock falling from the hanging in the vicinity of the working faces. This led to the sourcing, further design and installation of nets to capture falling rock in the vicinity of the working face which will allow for persons to be protected from falling rocks and also allow some time to escape from the affected area.

The installation of nets forms part of the standard and is installed by hooking the nets on to the elongate support which have fixtures designed for this purpose. The nets are then tightened against the hanging wall. Normal work operations are carried out for the shift and the nets are removed and stored in a safe place at the end of the shift.

Nets are inspected for damage and are replaced when damaged or when the prescribed period for use is exceeded.

4.5 2012 Managerial Support Instruction (MSI 2011/01C) (APPENDIX E)

In 2012, **in-stope bolting** was added to the support standard. This addition involves the drilling and installation of tendons into the hanging wall together with a resin injection to help consolidate the hanging wall. Tendons must be installed after every blast at a dip spacing not exceeding 1.5 m and not more than 0.5 m from the advancing face, on strike for breast mining, and an alternate configuration would be employed for up-dips and wide raises.

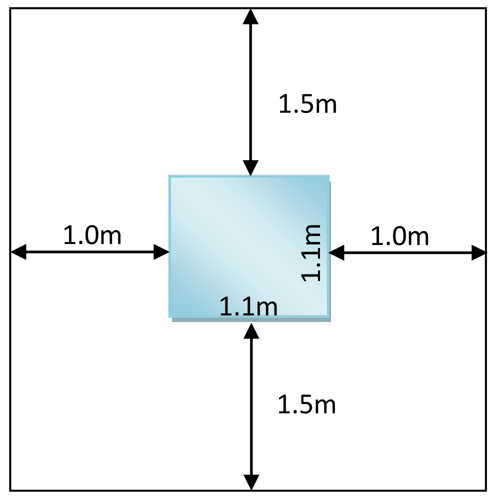
4.6 Area Coverage Improvement over the years

4.6.1. Area Coverage1999 (Not to scale)

Pack Support

Area covered by a single pack:
3.1 m x 4.1 m = $12.71\ m^2$

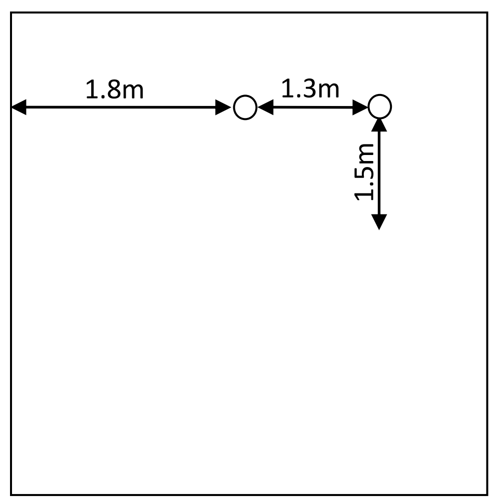
For a 20m x 20m panel:
$400\ m^2 / 12.71\ m^2$ = 32 packs



Elongate Support

Area covered by a set of elongates:
3.1 m x 1.5 = $4.65\ m^2$
Area covered by a single elongate:
4.65 m / 2 = $2.325\ m^2$

For a 20m x 20m panel:
$400\ m^2 / 2.325\ m^2$ = 172 packs



An area of 20 m on strike and 20 m on dip required 32 composite packs and 172 elongates to comply with the support standard in 1999.

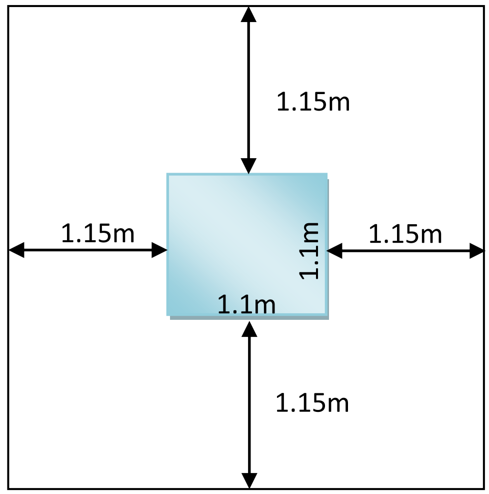
Figure 4.6.1.1 **Area coverage calculation 1999.**
(Source: personal calculation from support standard 1999)

4.6.2. Area Coverage 2005 (Not to scale)

Pack Support

Area covered by a single pack:
3.4 m x 3.4 m = 11.56 m^2

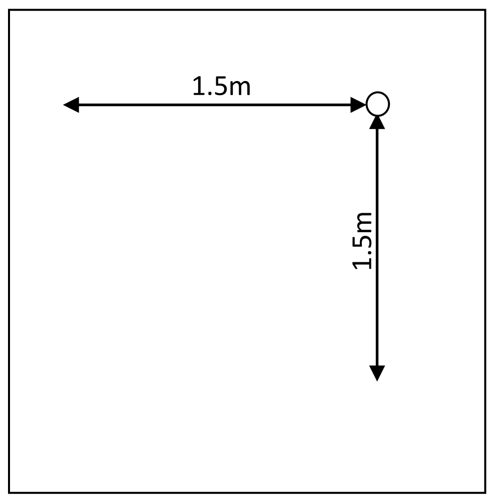
For a 20 m x 20 m panel:
400 m^2 / 11.56 m^2 = 35 packs



Elongate Support

Area covered by a single elongate:
1.5 m x 1.5 m = 2.25 m^2

For a 20m x 20m panel:
400 m^2 / 2.25 m^2 = 178 elongates



An area of 20 m on strike and 20 m on dip required 35 composite packs and 178 elongates to comply with the support standard in 2005.

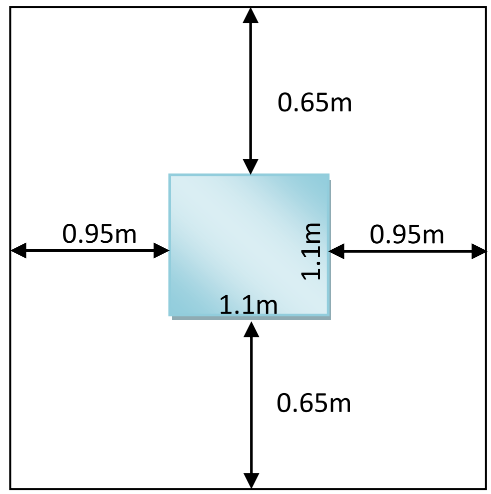
Figure 4.6.2.1 **Area coverage calculation 2005**
(Source: personal calculation from support standard 2005)

4.6.3. Area Coverage 2009 (Not to scale)

Pack Support

Area covered by a single pack:
3.0 m x 2.4 m = 7.2 m^2

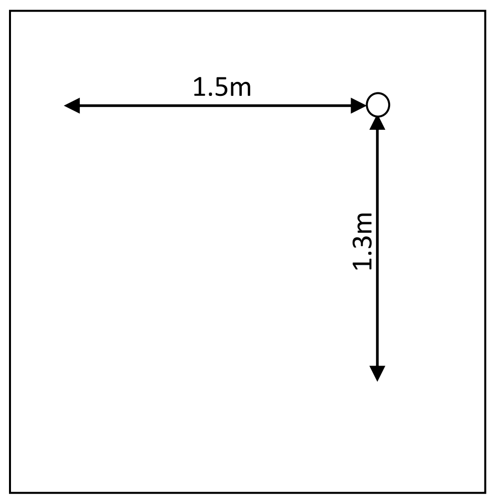
For a 20m x 20m panel:
400 m^2 / 7.2 m^2 = 56 packs



Elongate Support

Area covered by a single elongate:
1.5 m x 1.3 m = 1.95 m^2

For a 20m x 20m panel:
400 m^2 / 1.95 m^2 = 205 elongates



An area of 20 m on strike and 20 m on dip required 56 composite packs and 205 elongates to comply with the support standard in 2009.

Figure 4.6.3.1 **Area coverage calculation 2009**
(Source: personal calculation from support standard 2009)

The simple calculations above indicate how support density has improved over the years with regards to pack and elongate support. This improvement was prompted by the fact that workplaces were still experiencing gravity and seismic related rock fall damage. An area of 400 m^2 required 32 timber packs in 1999, whilst in 2009; this area required 56 timber packs. The area coverage and support density has drastically improved over this period.

These improvements, although increasing mining costs, have added value with regards to preventing damage to workplaces and ensuring the health and safety of persons employed in these workings.

Furthermore, the introduction of **pre-conditioning**, **in-stope netting** and the latest introduction of **in-stope bolting** have all added immense value with regards to improving the health and safety conditions of the working environment.

Table 4.6.1 **Summary of changes made to the support standard.**
(Source: personal compilation, 1999-2012)

SUPPORT CHANGE					
	Pack Support		**Elongate Support**		**Additional**
YEAR	**Pack Spacing (dip x strike)**	**Area supported by unit (m^2)**	**Elongate Spacing (dip x strike)**	**Area supported by unit(m^2)**	
1999	3.0 m x 2.0 m	12.71	1.0 m x 1.5 m	2.33	
2005	2.3 m x 2.3 m	11.56	1.5 m x 1.5 m	2.25	
2009	1.5 m x 1.9 m	7.2	1.3 m x 1.5 m	1.95	Pre-conditioning and additional elongates
2011	1.5 m x 1.9 m	7.2	1.3 m x 1.5 m	1.95	In-stope netting
2012	1.5 m x 1.9 m	7.2	1.3 m x 1.5 m	1.95	In-stope bolting

Table 4.6.1 indicates the changes made to the support standard from 1999 to 2012. It can be noted that there is an increase in support density from 1999 to 2009 from a resultant decrease in area supported per support unit. From 2009 the area supported by pack and elongate support has remained constant; however, pre-conditioning was introduced as a standard followed by in-stope netting in 2011 and lastly, in-stope bolting in 2012.

4.7 Conclusion

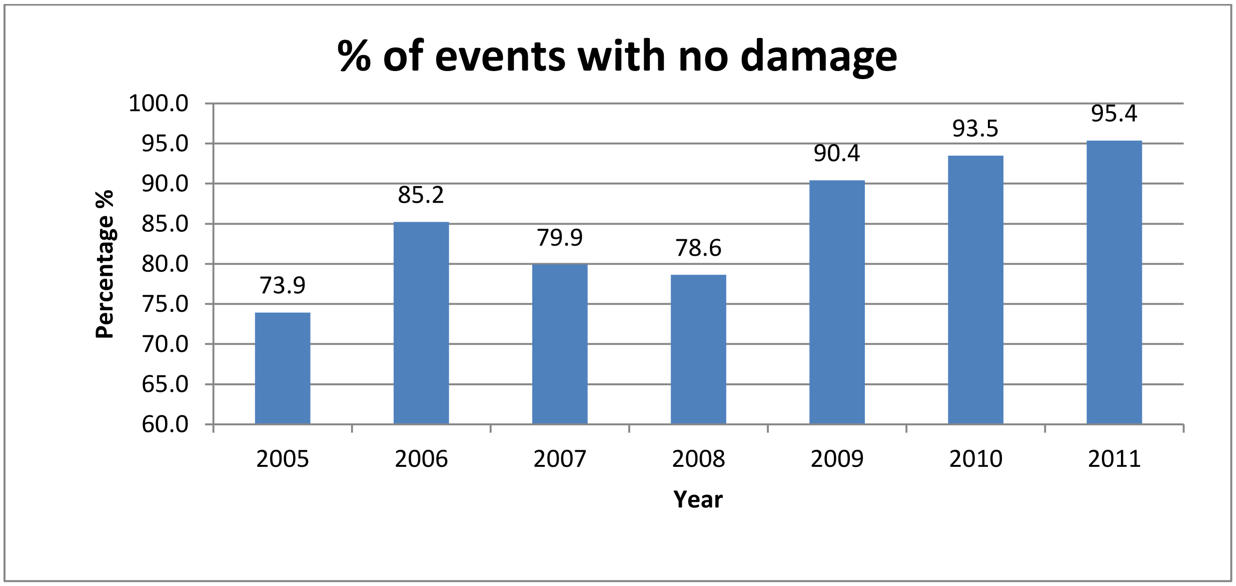


Figure 4.7.1 **Percentage of total seismic events with no damage to workings.**
(Source: KDC Seismology Department database, 2005-2011)

Figure 4.7.1 indicates the percentage of total number of seismic events with magnitude greater than 1 which have resulted in no damage to the workings from 2005 to 2011. A general improvement can be noted from 2005 to 2011. A significant improvement can be noted from 2009 to 2011. This improvement is, consequently, in line with the improvements made to the support standard from 2009 whereby the support density was significantly increased and the introduction of pre-conditioning and in-stope netting as a standard on all working panels.

By using the latest cost per timber unit of R 23.08 and R 801.50 per rock prop, a cost comparison between the different support upgrades was calculated.

Table 4.7.1 **Cost of increasing support density**.
(Source: personal compilation, 2013)

Year	No. Packs	No. Elongates	Cost of Packs (Rands)	Cost of Elongates (Rands)	Total Cost (Rands)	Cost/m^2 (Rands)	% Increase
1999	32	172	33235.2	137858	171093.2	427.73	**0**
2005	35	178	36351	142667	179018	447.55	**4.6**
2009	56	205	58161.6	164307.5	222469.1	556.17	**24.3**

Table 4.7.1 shows the cost of increasing support density using the support requirements as per standard for an area of 400m^2, 20 m on strike and 20 m on dip. The cost per square meter is also shown together with the increase in percentage cost from one standard to the next. There was a 24.3% increase in support costs from the 2005 support standard to the 2009 support standard. This information was applied to the production results for each year, as shown in ***Table 4.7.2***, indicating the total amount of capital spent on timber and elongate support.

Table 4.7.2 **Capital spent on timber and elongate support.**
(Source: personal compilation, 2013)

Year	Square meters mined (M^2)	Cost (Rands)
2005	79553	35 603 547
2006	93367	41 785 934
2007	92698	41 486 526
2008	52464	23 480 001
2009	32378	18 007 761
2010	33211	18 471 053
2011	35037	19 486 625

Table 4.7.2 indicates the capital spent on timber and elongate support for each year from 2005 to 2011. It can be noted that although the cost per square meter increased by over 24% in 2009, the total capital cost decreased from 2008 due to the decrease in production output.

If one considers the year 2008, just over R 23 million was spent on total support costs. The previous rough calculation, in chapter 3, indicated a loss of revenue of over R 22 million in gold production, excluding the cost of re-establishing workplaces and injury for the same year. Although there is an increase in support cost of 24.3% from the 2005 support standard to the 2009 support standard, there is a definite financial gain with regards to not losing production and, more importantly, a safety gain with regards to reducing injury and fatalities.

Pre-conditioning has made a significant contribution towards creating safer mining environments at Thuthukani Shaft, KDC. The following chapter discusses pre-conditioning in detail, how it works and the benefits of conducting pre-conditioning correctly.

Chapter 5

5. IMPACT OF PRE-CONDITIONING ON SAFETY

5.1 Introduction

Pre-conditioning was known as a de-stressing blast and was introduced at East Rand Proprietary Mines (ERPM) with positive results and a reduction in rock bursts *(Roux et al, 1957)*. Despite these results Pre-conditioning was not generally accepted in industry. Experimental work was conducted by the Chamber of Mines from 1987 to 2003 with the results published as SIMRAC Project Reports:

- GAP 030 - January 1996
- GAP 336 - January 1998
- GAP 811 - April 2003

This work gave an understanding of the Pre-conditioning mechanism and application methodology. The published outcomes included detailed implementation guidelines. Face perpendicular Pre-conditioning techniques were tested and it was documented and proven that in deep mines substantial benefits could be obtained by using face perpendicular Pre-conditioning and this technique is used at Thuthukani Shaft, KDC.

Thuthukani Shaft KDC has also committed to industry best practices and pre-conditioning is considered as industry best practice to reduce the effects of strain bursting and face ejection damage caused by seismic events. Pre-conditioning, when executed correctly, improves ground conditions thereby creating a safer mining environment.

Pre-conditioning had been used at Thuthukani Shaft in the past. However, it was not part of the general support standard and was recommended by the Rock engineering department during planning sessions with regards to specific workplaces where face bursting was foreseen as a hazard. In 2009, Pre-conditioning was introduced into the mining standard for all working panels at Thuthukani Shaft, KDC.

5.2 Why should we conduct pre-conditioning?

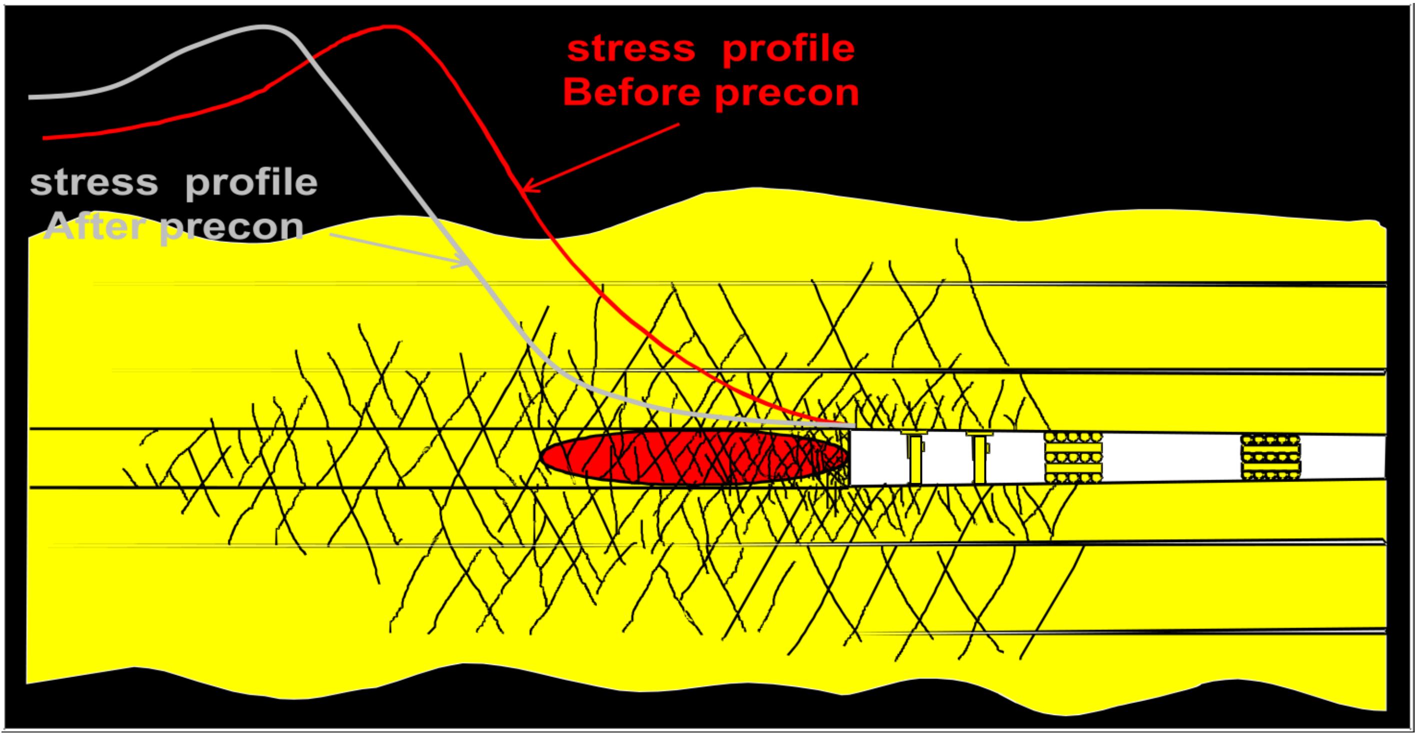


Figure 5.2.1 **Stress profile before and after Pre-conditioning.**
(Source: GAP 336, 1998)

Pre-conditioning is a process used to remobilize existing fractures ahead of the face and possibly create new fractures by blasting longer holes, as explained in sections 5.3 and 5.4 of this report (*Mahne, 2004).* The strong bonds between fracture planes ahead of the face are weakened or destroyed, therefore, strain energy cannot build up directly ahead of the working face, and it will build up further ahead of the working face. A cushion of stress relieved rock exists ahead of the face. The 'broken rock' ahead of the face acts as a buffer between the stressed rock and the work force. This cushion is able to absorb the energy from large events further away and also reduces, or eliminates, the effects of strain bursting.

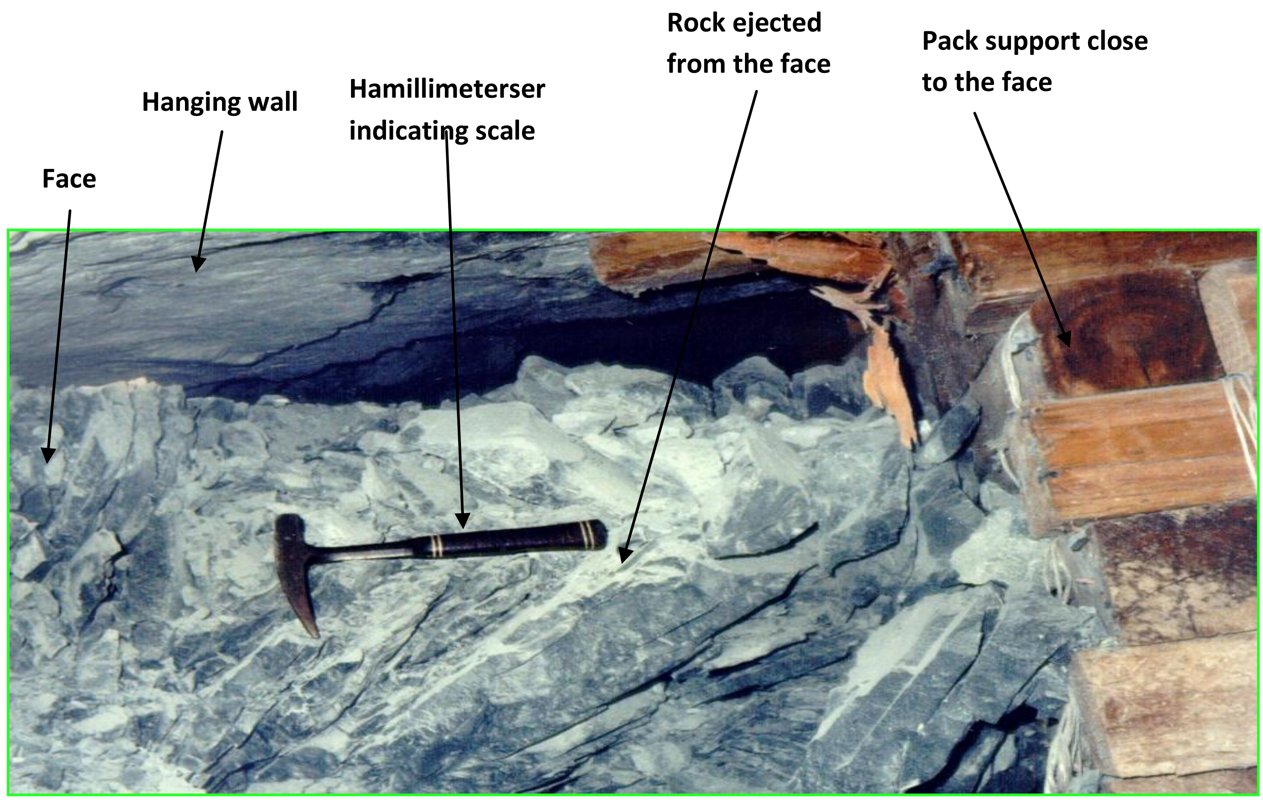


Figure 5.2.2 **Face ejection caused by a large event over 1500m away.**
(Source: Durrheim, 1996)

Figure 5.2.2 indicates face ejection of rock caused by a seismic event which was located 1.5 km away from the working place depicted in the picture. This is an indication of the fact that if a mining face is highly stressed, a seismic event which is located some distance away can still trigger face ejection of rock. One can notice that the pack support is fairly close to the face; however, this has no effect in preventing the face ejection. Pre-conditioning of the face would have, most likely, prevented this face ejection.

The mechanism of Pre-conditioning is local, that is it affects the area directly ahead of the face and it is temporary, that is it is effective for one blast only. The zone influenced around the Pre-conditioning hole has a radius of approximately 1.5m as indicated in ***Figures 5.2.3*** and ***5.2.4***.

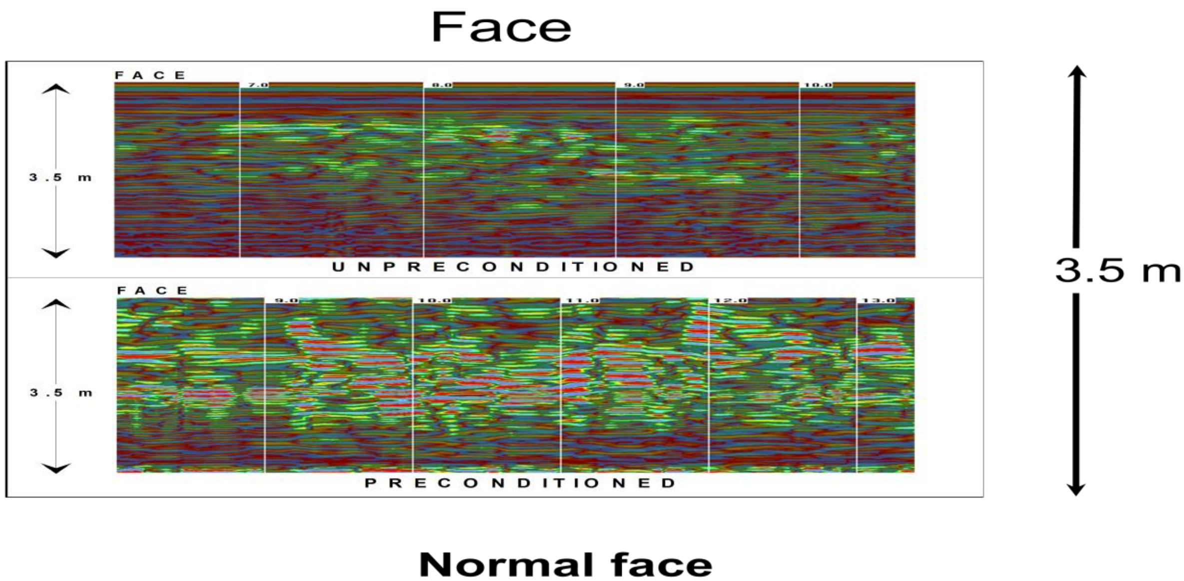


Figure 5.2.3 **Face fracture mapping using ground penetrating radar for a normal face at a depth of 3.5m.**
(Source: GAP 336, 1998)

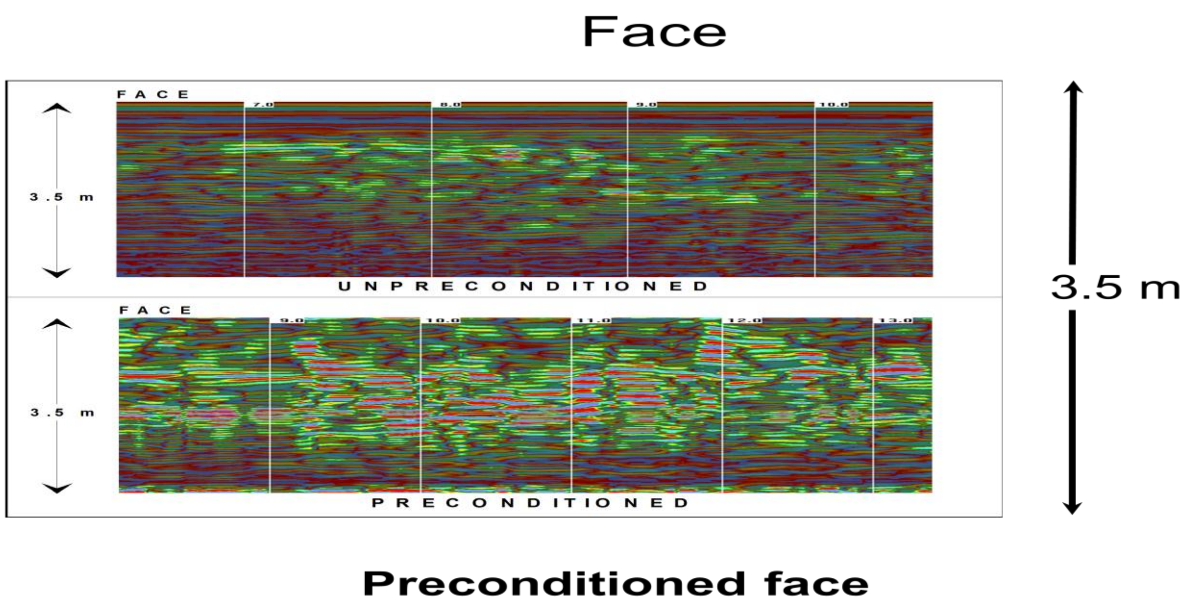


Figure 5.2.4 **Face fracture mapping using ground penetrating radar for a pre-conditioned face at a depth of 3.5m.**
(Source: GAP 336, 1998)

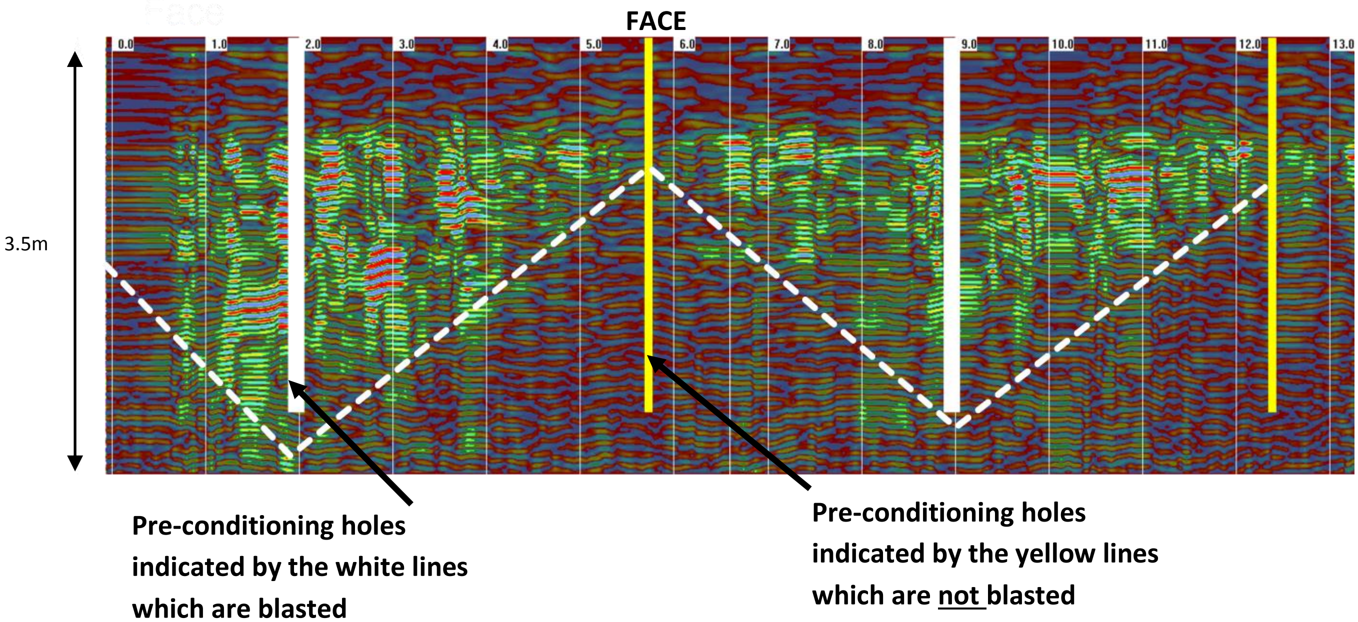


Figure 5.2.5 **Fracture mapping – Zone of influence.**
(Source: GAP 336, 1998)

Figure 5.2.5 above, indicates the fracture mapping, enclosed by the broken white lines and one can notice that if the blasted Pre-conditioning holes are spaced more than 6m apart there is almost no overlap in fracturing and the Pre-conditioning holes that were not blasted, indicated in yellow, results in stress concentration on the face.

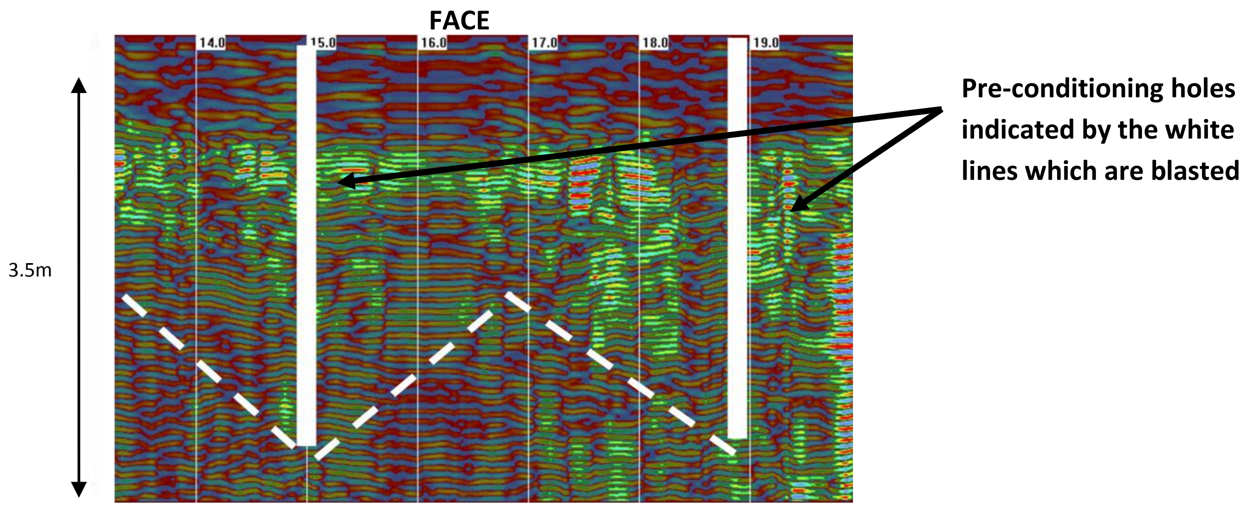


Figure 5.2.6 **Fracture mapping – Zone of influence.**
(Source: GAP 336, 1998)

Figure 5.2.6 depicts pre-conditioning holes that are blasted at closer spacing indicated by the solid white lines. The area enclosed by the broken white lines indicates the fracture zone and one can notice the overlap of the fracture zone when Pre-conditioning holes, indicated by the yellow lines in ***Figure 5.2.5*** , are blasted, resulting in reduced stress concentration on the face.

The process of Pre-conditioning involves the drilling of blast -holes at a determined spacing on the face which are, a determined length, longer than the normal blast -hole length. At the beginning and end of the panel, these holes are drilled at a 30° angle into the sidewalls.

These holes are then charged up with explosives and blasted together with the normal blast- holes, resulting in a zone of rock ahead of the face where the fractures are remobilized due to the blast energy emitted from these charged up Pre-conditioning holes.

5.3 Marking and Drilling

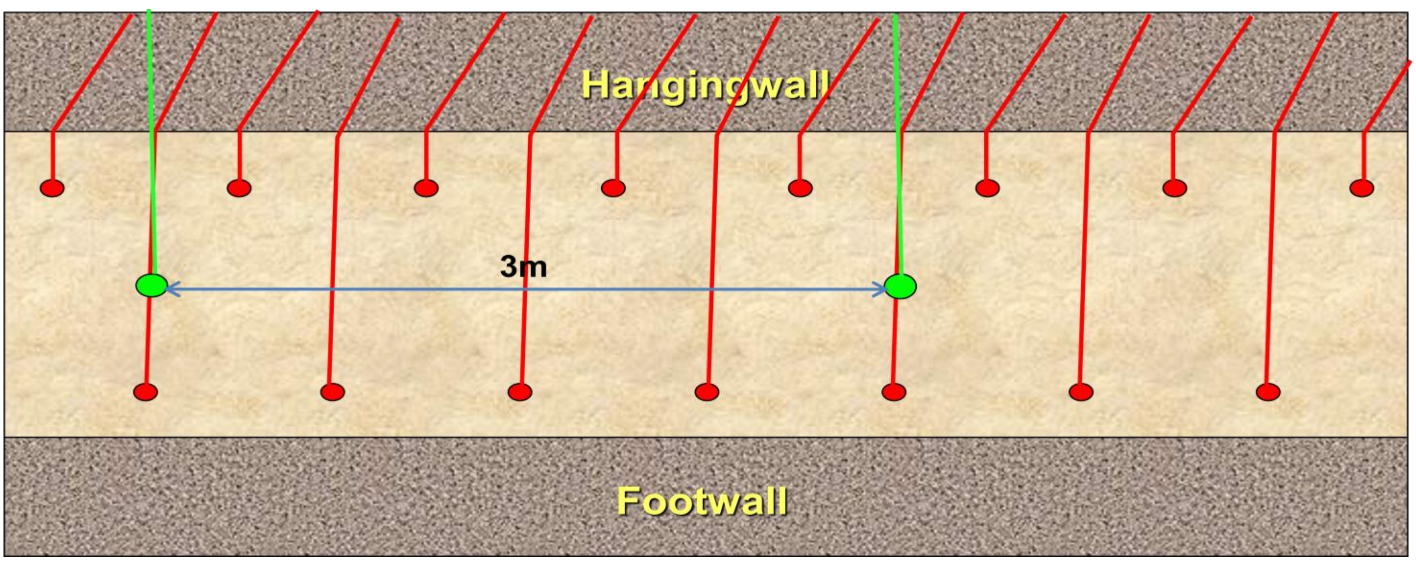


Figure 5.3.1 **Marking of Pre-conditioning holes.**
(Source: Gold Fields, 2008)

Figure 5.3.1 indicates the marking of Pre-conditioning holes, 3m apart (in green) and blast holes (in red). The marking of Pre-conditioning holes in the correct positions is vitally important as studies have shown that incorrect marking can lead to adverse effects with regards to stress distribution and stress build up on the face. The zone of influence around a Pre-conditioning hole extends to a radius of 1.5m and a spacing of 3m between any two consecutive holes is ideal.

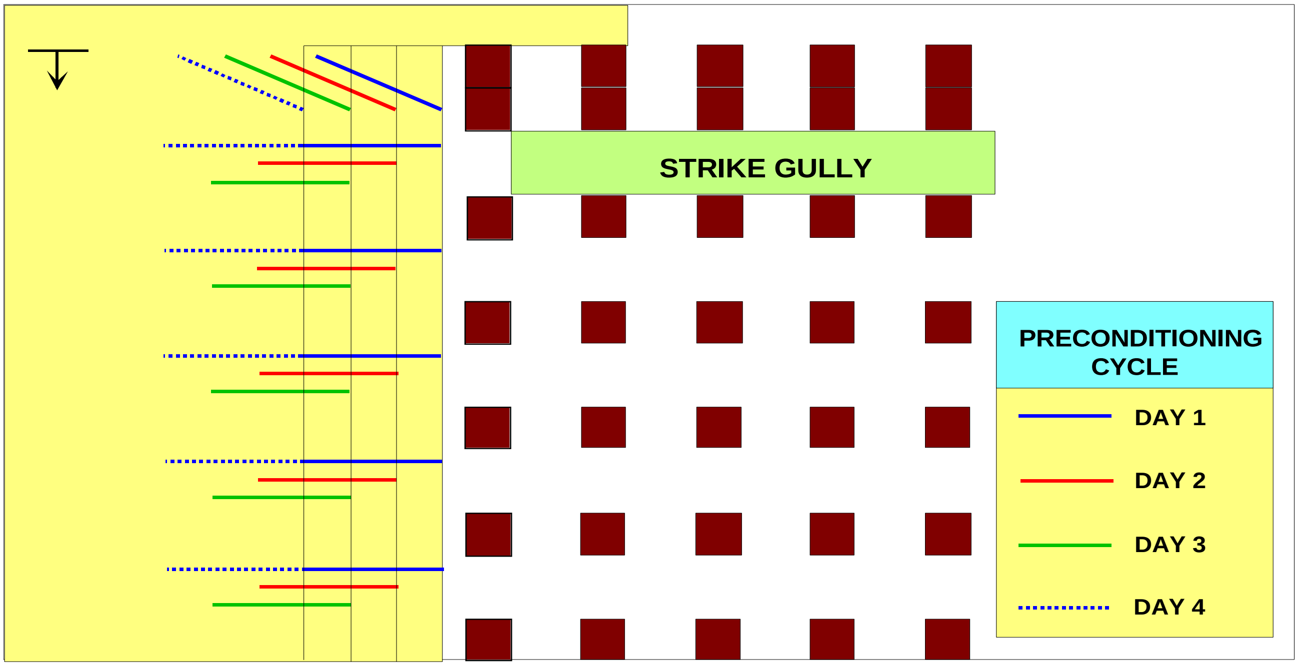


Figure 5.3.2 **Plan view of the marking and drilling of Pre-conditioning holes.** (Source: GAP 336, 1998)

Figure 5.3.2 indicates the drilling cycle of the Pre-conditioning holes. Holes are drilled 15 cm away from the previous blasted hole in a 4 day cycle. This ensures that compliance to mining law with regards to the drilling of shot holes next to sockets is upheld and that consistency with regards to the spacing of pre-conditioning holes is maintained. The four day cycle also ensures that the vestiges of the Pre-conditioning hole drilled on the first day are totally eliminated by the third day.

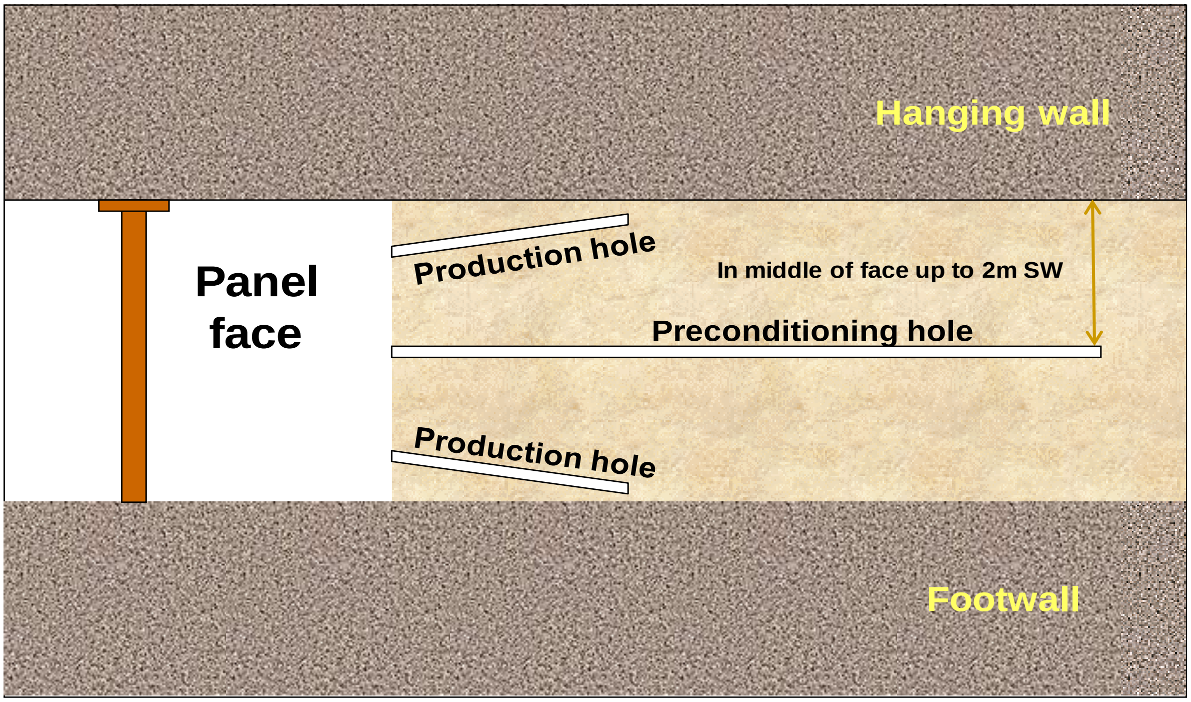


Figure 5.3.3 **Section view of drilling of a Pre-conditioning hole.**
(Source: GAP 811, 2003)

Pre-conditioning holes are drilled in the middle of the face and must be twice the length of a normal blast hole *(Mahne, 2004).*

5.4 Charging up and Timing

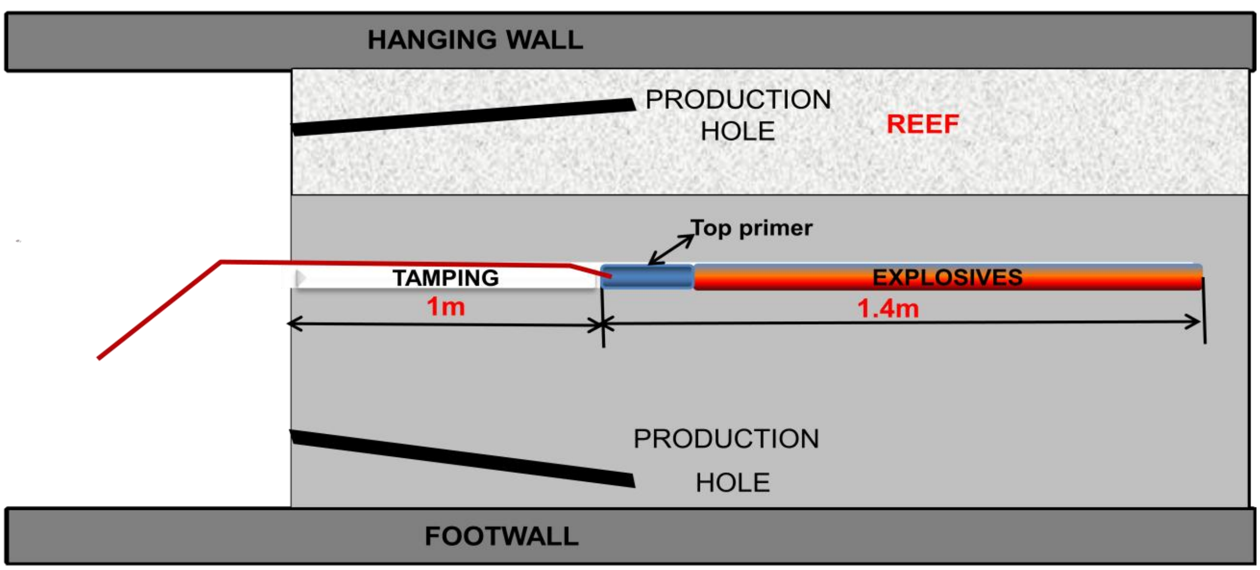


Figure 5.4.1 **Charging up of Pre-conditioning hole.**
(Source: GAP 811, 2003)

Figure 5.4.1 indicates the charging up of the Pre-conditioning hole. The hole is charged up with explosives, a top primer and tamping whereas the production hole is charged up with a bottom primer, explosives and tamping. This ensures that the primers in both the production hole and Pre-conditioning hole are more or less the same depth in the hole, this is important as it makes the retrieval of explosives in the case of a misfire more manageable.

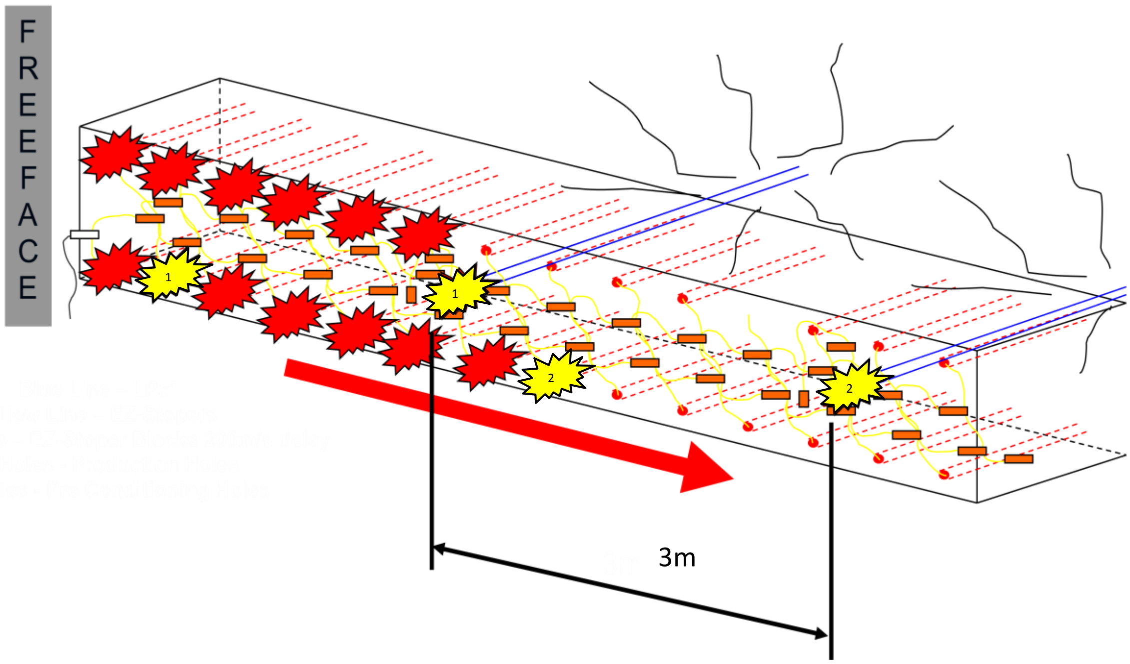


Figure 5.4.2 **Timing of Pre-conditioning holes in relation to the blast holes.**
(Source: Gold Fields, 2008)

Figure 5.4.2 is an illustration of the timing of Pre-conditioning holes in relation to the blast holes. The Pre-conditioning holes are represented by the blue solid lines while the blast holes are represented by the red broken lines. Pre-conditioning holes should be timed in such a way that the Pre-conditioning holes are blasted ahead of the blast holes. The recommendation is that it should be 1m ahead of the blast holes as the purpose of the Pre-conditioning hole is not to blast the face but to mobilize existing fractures.

5.5 Advantages of Pre-conditioning

A decrease in the localized damage caused by strain burst has been noticed as indicated in ***Figure 5.5.1***. A marked improvement can be noted in 2008 and 2009 when Pre-conditioning was introduced into the support standard for the mine.

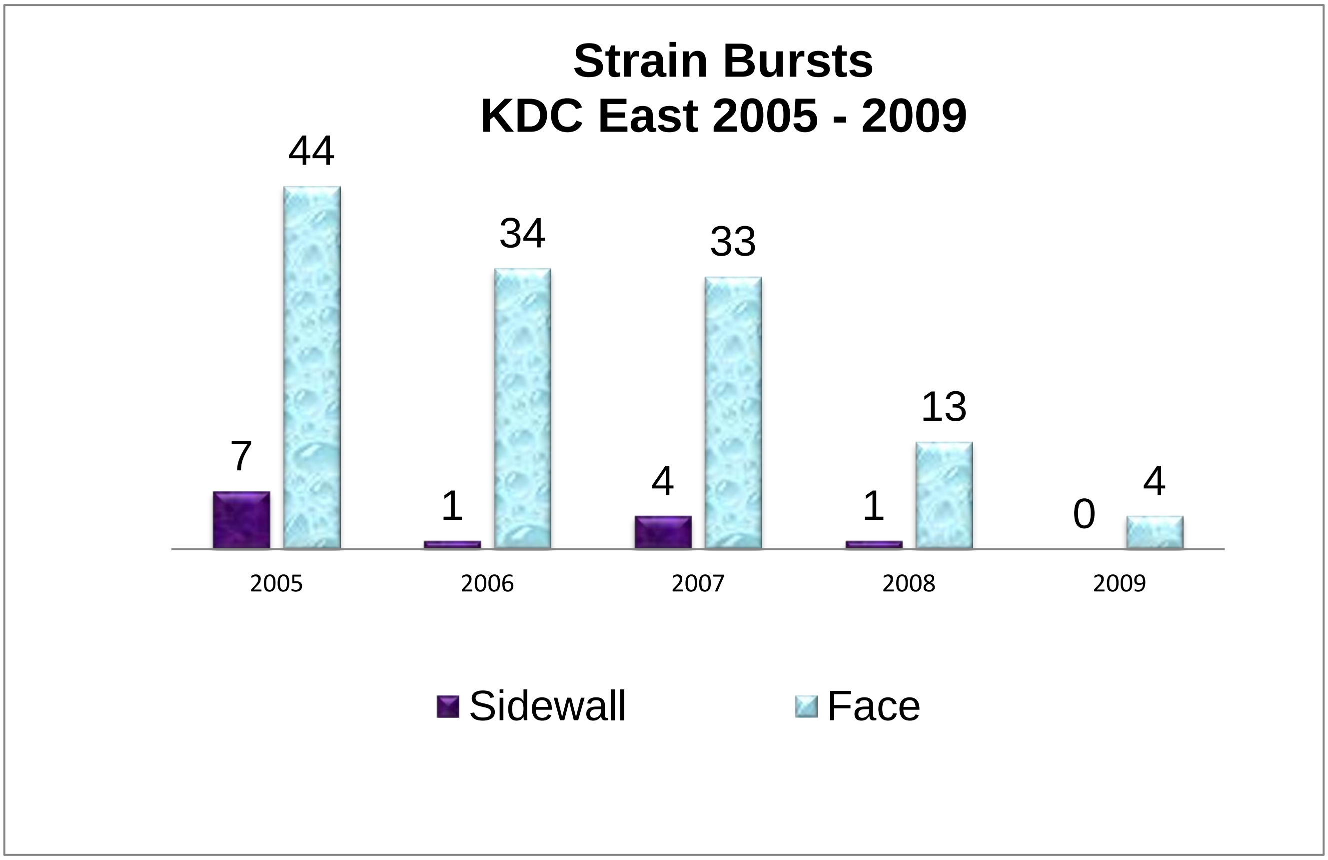


Figure 5.5.1 **Number of Strain Bursts from 2005 to 2009 for KDC East.**
(Source: Gold Fields, 2010)

Improved hanging wall conditions, face advance and fragmentation have been noticed. ***Figure 5.5.2*** and ***5.5.3*** indicate panels without Pre-conditioning and with Pre-conditioning respectively. A significant difference in the hanging wall, face and fragmentation condition can be noticed.

Figure 5.5.2 **Panel with no Pre-conditioning conducted.**
(Source: Toper, 2003)

Figure 5.5.3 **Panel with Pre-conditioning conducted.**
(Source: Toper, 2003)

It is vitally important to ensure the correct application of Pre-conditioning as failure to apply the correct method could result in undesired effects to the extent of worsening the situation rather than alleviating face bursts *(GAP 811, 2003).* Isolated or incorrectly positioned Pre-conditioning holes can transfer stress to adjacent areas and mining panels and if positioned too far ahead of the face, they can transfer stress back towards the face *(GAP 030, 1996).*

Pre-conditioning is effective only in a small zone, less than 4m into a mining face. As confining stresses increase further into the rock mass ahead of the face the strength of the rock mass increases, thus the potential for fracture generation is limited *(GAP 030, 1996)*.

Figure 5.5.3 indicates much better hanging wall conditions as compared to ***Figure 5.5.2***. This improvement of the hanging wall conditions caused by implementing pre-conditioning of the face and side walls helps in maintaining a consistent and more solid beam of rock in the hanging wall. This results in more stable hanging wall conditions which are more robust when a seismic event occurs. The hanging wall in ***Figure 5.5.2*** is more prone to a shake down and localized falls of ground due to its blocky and fractured nature.

Prior to Pre-conditioning being introduced as a widely used standard in 2009 at KDC East, it was normally recommended by the rock engineering department as part of special instructions issued when considering or planning the mining of remnant areas. After receiving instructions for the wide application of Pre-conditioning in 2008 and the formal introduction of Pre-conditioning to the mining standard in 2009, at KDC East, a significant drop in the number of face bursts was noticed and is depicted in ***Figure 5.5.4***.

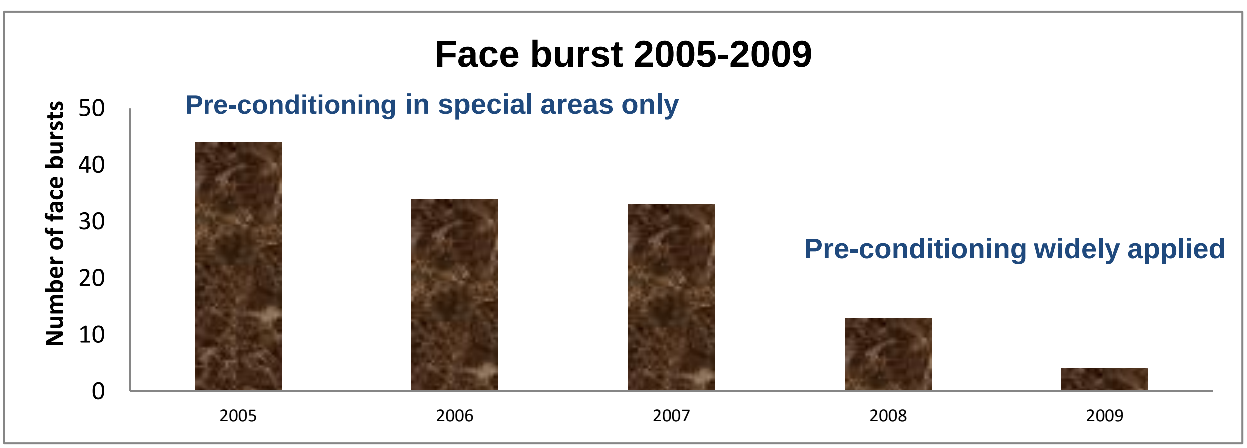


Figure 5.5.4 **Number of face bursts experienced at KDC East from 2005 to 2009.** (Source: Gold Fields, 2010)

5.6 Conclusion

Pre-conditioning increases the cost of mining as more explosives and accessories, longer drilling steel and time to complete the Pre-conditioning work is required which could also result in an increase in labour requirements. However, the benefits of pre-conditioning, which are noted above, are multiple if one has to consider the cost of rehabilitation of damaged workplaces, and more importantly the cost of injury and fatalities. Pre-conditioning improves the safety conditions of the underground workplace which is a fundamental requirement by law in that the mine is required to ensure that the health and safety conditions in underground workings are conducive for human employment.

Pre-conditioning requires skills to be developed with regards to the process as incorrect installation of Pre-conditioning holes can have an adverse effect on the work environment. It is therefore fundamentally important that an understanding of pre-conditioning is developed and communicated to the workforce to emphasize the positive effects when conducted correctly and the negative effects when conducted incorrectly.

Regular audits conducted by rock engineering personnel at Thuthukani Shaft, KDC include the inspection of Pre-conditioning holes, coaching and training of the workforce and disciplinary action taken against supervision if standard processes are not being adhered to. **Appendix F** indicates a typical Pre-conditioning audit check list. The careful inspection of Pre-conditioning holes include the inspection of holes drilled on the day of the audit and also an inspection of the sockets of the previously drilled Pre-conditioning holes to ensure that Pre-conditioning is being conducted with each blast and according to standard.

Figure 5.6.1 **Checking of a Pre-conditioning hole at Thuthukani Shaft, KDC.**
(Source: Personal Collection)

Figure 5.6.1 shows two pictures where a pinch bar is used to check the depth of the remaining socket of a previously blasted Pre-conditioning hole, after it had been cleared of all remaining explosives and washed out.

In many cases where face bursting had occurred after the implementation of the standard for Pre-conditioning, it was found, upon investigation that the standard process for Pre-conditioning had not been adhered to and a program for re-training of the production crew then had to be followed.

In some areas where the hanging wall conditions were extremely poor, localized falls of ground continued to occur. These falls of ground occurred in areas where the hanging wall lava was extremely weak. This lava is known as the Westonaria Formation (WAF) and tends to rapidly deteriorate when exposed to air and water. Means to arrest these falling rocks and prevent injury had to be sought and this led to the exploration of in-stope netting. The following chapter discusses in-stope netting which was considered as best practice and was introduced to the support standard for the entire mine.

Chapter 6

6. IN-STOPE NETTING AS ADDITIONAL PROTECTION

6.1 Introduction

In 2009, falls of ground were still being recorded even in areas where the standard support requirements and density had been exceeded. Localized falls of ground were noted in the area closest to the advancing face, between support units. A commitment was made to increase the areal support density at the stoping faces. Two techniques were readily available. The one was to install in-stope nets which would essentially provide additional protection to the day-shift drilling and charging crews, performing these operations. The second technique was the installation of roof bolts in the hanging wall. This installation has the advantage that the unsupported span after the blast can be reduced to less than 1.5m without interfering with cleaning operations.

There are a variety of geo-nets, geo-grids and geo-mesh available on the market. The stiffer geo-mesh has proved not be suitable for rockburst conditions. It also tends to be unmanageable in the stoping environment. Recently, a galvanized wire rope mesh had become available on the market. These nets had not been tested for rockburst conditions. Underground tests indicated that the rope tended to unravel and injuries to eyes and hands were seen as hazards. They were also seen to be heavy and not easily manageable in the underground workings.

Geo-nets were developed specifically to overcome the disadvantages of the geo-mesh and geo-grids. Energy is transferred from the rock impacting on the net panel to the net perimeter and from there onto the attachment points. Diagonal reinforcement ropes transfer load directly to the net attachment points, they strengthen the net substantially. Geo-nets have the potential to reduce and eliminate rockfall incidents in stopes and tunnels. The choices of safety nets, as well as their specific installation procedures, are dependent on site specific requirements.

In 2011, in-stope netting was introduced into the support standard for all panels at Thuthukani Shaft with the intention of increasing areal support and reducing rock fall injuries.

6.2 Mine Safety Net Development and Application

Generally, mine safety nets are used to protect persons from falling rock in temporarily supported development ends, tunnels, advance strike gullies and stopes. These diverse applications require specific net designs, resulting in a wide product range. As a rule, however, mine safety nets need to carry specified static or dynamic loads with the least possible deformation. Because of the application-specific characteristics of mine safety nets, there is no South African national standard for them. Consequently, neither the South African Bureau of Standards (SABS), nor the CSIR, performs standard tests on these components. In 2007, the Safety in Mine Research Advisory Committee (SIMRAC) made available their Savuka facility for the testing of mine safety nets. The Savuka drop test became the accepted industry test standard for mine safety nets. From 2007 to mid-2009, Norsenet, in collaboration with SRK Consulting, the designated operators of the site conducted approximately 200 tests, which formed the core of Norsenet's database of safety net performance under various loading conditions *(Skarbovig et al, 2010).*

Some research into "Mine safety net development and applications" was conducted by *Skarbovig et al*, (2010) and presented at the South African Institute for Mining and Metallurgy Conference, Second Hardrock Safe-Safety Conference, 2010. Some of the information presented is indicated below.

Figure 6.2.1 **Safety net test at the Savuka testing facility.**
(Source: Skarbovig, 2010)

In-stope netting is widely used in the gold and platinum industry nowadays. ***Figure 6.2.1*** above, indicates a safety net test at Savuka, showing the 450 kg twin drop mass arrested by the safety net. The diagonal and perimeter net reinforcement ropes attached to the Camlok prop hooks can be noted. ***Figure 6.2.2*** indicates a typical installation of in-stope netting.

Figure 6.2.2 **Typical installation of in-stope netting.**
(Source: Skarbovig, 2010)

6.3 Conclusion

In-stope netting has added immense value with regards to improving the support system used at KDC from its introduction in 2011. Its practical functionality of arresting falling rocks which are either gravity or seismic related does not only improve safety statistics, but also gives the working crews a sense of added safety. People have a sense of added safety when entering a stope face where in-stope netting has been installed.

Figure 6.3.1 **In-stope netting at Thuthukani Shaft, KDC.**
(Source: Personal Collection)

In practical examples where large amounts of rocks have been ejected from the hanging wall, it gave the workforce “a bit of time” to escape. ***Figure 6.3.1*** indicates the installation of in-stope netting in one of the stopes at Thuthukani Shaft, KDC. The nets are installed between the last line of packs and the face. Additional nets are installed on the face to arrest the ejection of rock from the face in the event of a face burst. Blast holes are drilled through the apertures in the net.

Figure 6.3.2 indicates an incident at Thuthukani Shaft, KDC, where in-stope netting installed on the face arrested the rock ejected from the face in a face burst event. If the in-stope netting was not installed on the face, the ejected rock would have been ejected into the back areas and possibly injured persons working there. The installation of in-stope netting on the face is not standard practice; it is carried out as an additional safety precaution and is becoming a more common practice when mining isolated blocks of ground at Thuthukani Shaft, KDC.

Figure 6.3.2 **In-stope netting used on the face at Thuthukani Shaft, KDC.**
(Source: Personal Collection)

In-stope netting, as a standard, was designed in order to protect persons from fall of grounds from the hanging wall. However, in-stope netting is only effective if installed properly. Ineffective installation will allow too much of sag rendering the net ineffective and may cause an obstruction when recovering trapped persons. It is also critically important that the nets are attached to secure support units, as support units that are unstable will also render the net ineffective as it will not be able to arrest falling rocks with minimum deformation to the net and/or prevent injury to persons.

Figure 6.3.3 depicts a production supervisor inspecting the arresting of a fall of ground by the in-stope netting at Thuthukani Shaft, KDC. It is clear that the rock arrested by the net would have caused serious injury to persons that were working underneath the net had the net not been installed.

Figure 6.3.3 **In-stope netting in action at Thuthukani Shaft, KDC.**
(Source: Personal Collection)

There are a wide range of applications of safety nets with different performance criteria. It is important for the rock engineer of a specific mine to ensure that the correct specifications of nets are used in conjunction with the support design planned for a specific workplace to ensure that the nets being used are capable of arresting falling rocks and preventing injury.

Constant auditing and training of the workforce is required to ensure that the in-stope net is installed according to standard for it to be effective and that the nets are removed and stored in a safe place, to prevent damage, at the end of the shift.

This addition and improvement to the support standard is not only effective from an arresting of falling rocks point of view, but it also improves the sense of safety felt by the workforce and all others entering the stope face as the implication is that for it to be properly installed, the support units, supporting the net, are properly installed.

The addition of safety nets to the support standard comes at a cost; it requires additional work and time and the net itself comes at a cost, however, the added safety benefit outweigh this additional cost and enables the workforce to be “more at ease” when conducting their operational tasks and allows them to focus on the work at hand.

As mentioned in the introduction of this chapter that the second technique was to implement a system of in-stope bolting. In-stope bolting is a more permanent system of protection. Once the roof bolts are installed into the hanging wall, they cannot be removed, whereas, in the case of in-stope netting, the nets have to be removed before the blast and stored in a safe place to prevent damage. The risk of being injured, in the removal of the net process, does exist. The following chapter discusses in-stope bolting which was the last addition to the support standard at Thuthukani Shaft, KDC at this point in time.

Chapter 7

7. IN-STOPE BOLTING AS TEMPORARY AND PERMANENT SUPPORT

7.1 Introduction

In-stope bolting is one of the many measures implemented at KDC aimed at eliminating falls of ground in the stope face area. In-stope bolting assists in this regard by increasing the stability of the immediate hanging wall beam thereby increasing its support resistance. It also provides face area support during all periods of the mining cycle, it is a permanent installation providing twenty- four hour support which is not removed and increases the area coverage of the support system used.

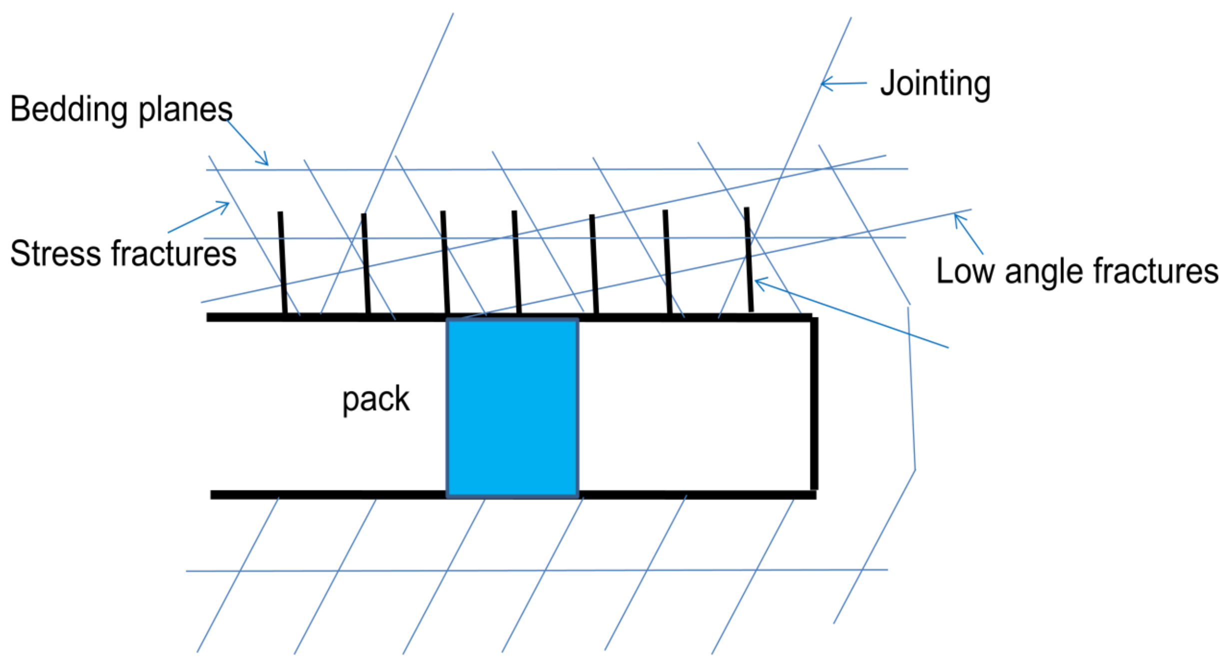


Figure 7.1.1 **Typical section of a stope panel. (Not to scale)**
(Source: Gold Fields, 2012)

Figure 7.1.1 is an indication of a typical section of a stope panel indicating the bedding planes, jointing, stress fractures and low angle fractures. In-stope bolting tends to bind some of these fractures zones in the hanging wall, thus reinforcing the hanging wall beam.

7.2 How does rock bolting work?

The following set of pictures demonstrates how rock bolting works and is similar to the demonstration that Tom Lang conducted in the Snowy Mountain Project *(Hoek, 2004).*

Figure 7.2.1 **Steel Box.**
(Source: Gold Fields, 2012)

Figure 7.2.1 depicts a steel box that was created to demonstrate the effects of rock bolting. The box contains four solid steel side walls which were welded together with the top and bottom of the box left open.

Figure 7.2.2 **Broken Rock.**
(Source: Gold Fields, 2012)

Figure 7.2.2 depicts a bucket of broken rock that will represent the broken and fractured rock that we normally find in the hanging wall of underground workings. This is only a representation and the rocks found underground will be much larger.

Figure 7.2.3 depicts a wooden board that was used to mark and install some bolts, nuts and washers at regular intervals. The board is placed into the bottom opening of the steel box and clamped to the sides of the steel box by use of G-clamps that are installed on the four sides of the steel box.

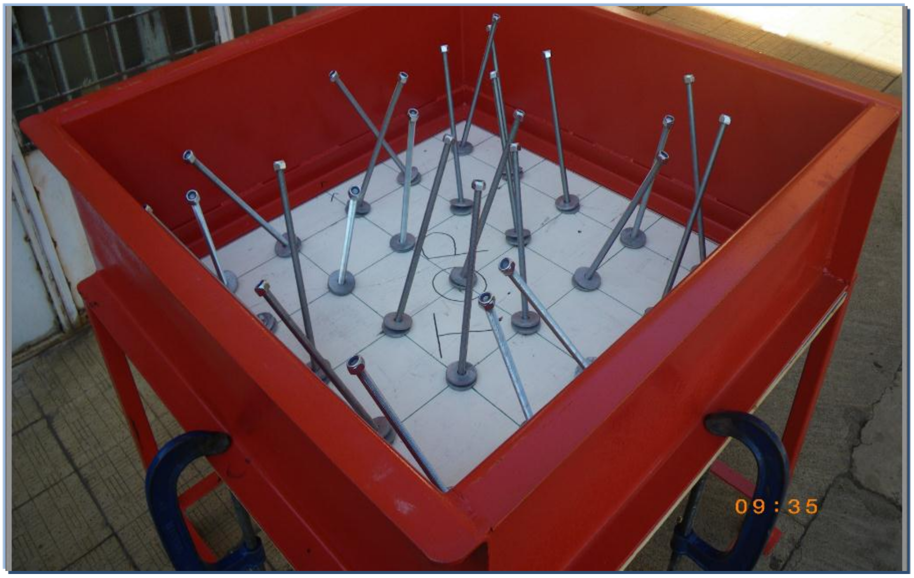


Figure 7.2.3 **Bolts and washers.**
(Source: Gold Fields, 2012)

Figure 7.2.4 depicts the broken rock, indicated in ***Figure 7.2.2***, which is poured into the steel box and onto the wooden board. The rocks are spread evenly onto the wooden board and amongst the bolts that protrude out.

Figure 7.2.4 **Broken rock between the bolts compressed.**
(Source: Gold Fields, 2012)

Figure 7.2.5 **Tightening the bolts.**
(Source: Gold Fields, 2012)

Figure 7.2.5 depicts the steel box being filled with more rocks, which is compressed into the steel box and washers and nuts being fastened onto the bolts.

Figure 7.2.6 **The bottom of the steel box showing the beam created.**
(Source: Gold Fields, 2012)

Figure 7.2.6 depicts the steel box, being viewed from underneath, where the G-clamps and wooden board have been removed. The compressed rock and tensioned bolts form a beam of rock.

Figure 7.2.7 **A rock prop is placed on top of the beam created demonstrating the strength of the beam created**.
(Source: Gold Fields, 2012)

Figure7.2.7 depicts a rock prop, which is in excess of twenty kilograms in mass, placed on the beam that was created with the bucket of rocks and bolts. This is an indication of the strength of the beam formed.

The figure below explains how rock bolts work.

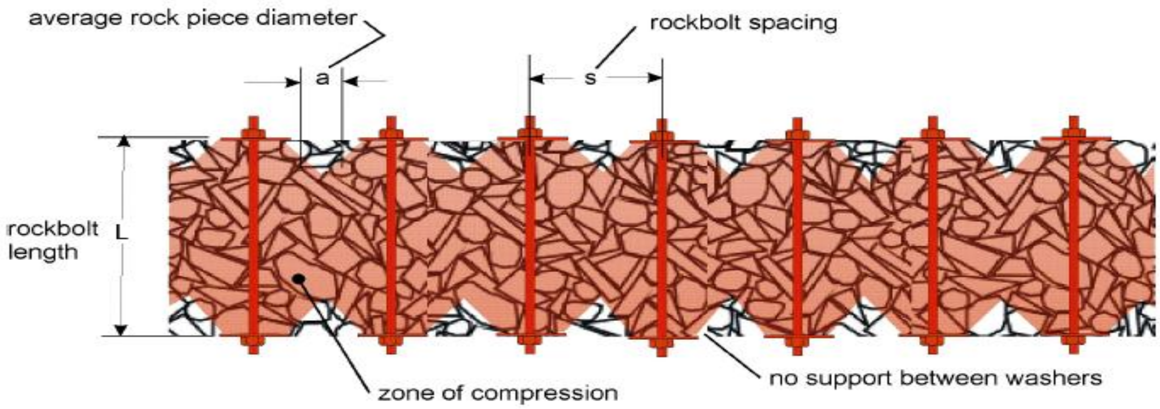


Figure 7.2.8 **How rock bolts work.**
(Source: Gold Fields, 2012)

A zone of compression is induced in the region shown in red. This zone will provide effective reinforcement to the rock mass. However, in the underground environment, it is not possible to install washers and nuts on both ends of the bolts as one has to drill into the hanging wall. Tensioning is achieved by either creating lateral forces from the bolt or by the use of a quick setting resin. This is one mechanism of stabilization by bolting; others include binding bottom roof layers onto upper ones and stabilizing jointed hanging wall by intersecting joint and fracture planes.

7.3 Types of bolts that can be used with in-stope bolting

Henning and Ferreira, (2009) have done some research into “In-stope bolting for a safer working environment” which was presented to The South African Institute of Mining and Metallurgy Hard Rock Safe, Safety Conference in 2009. They described the types of bolts that can be used with in-stope bolting. ***Figure 7.3.1*** indicates this representation. ***Table 7.3.1***, from their presentation, indicates the different characteristics of these bolts.

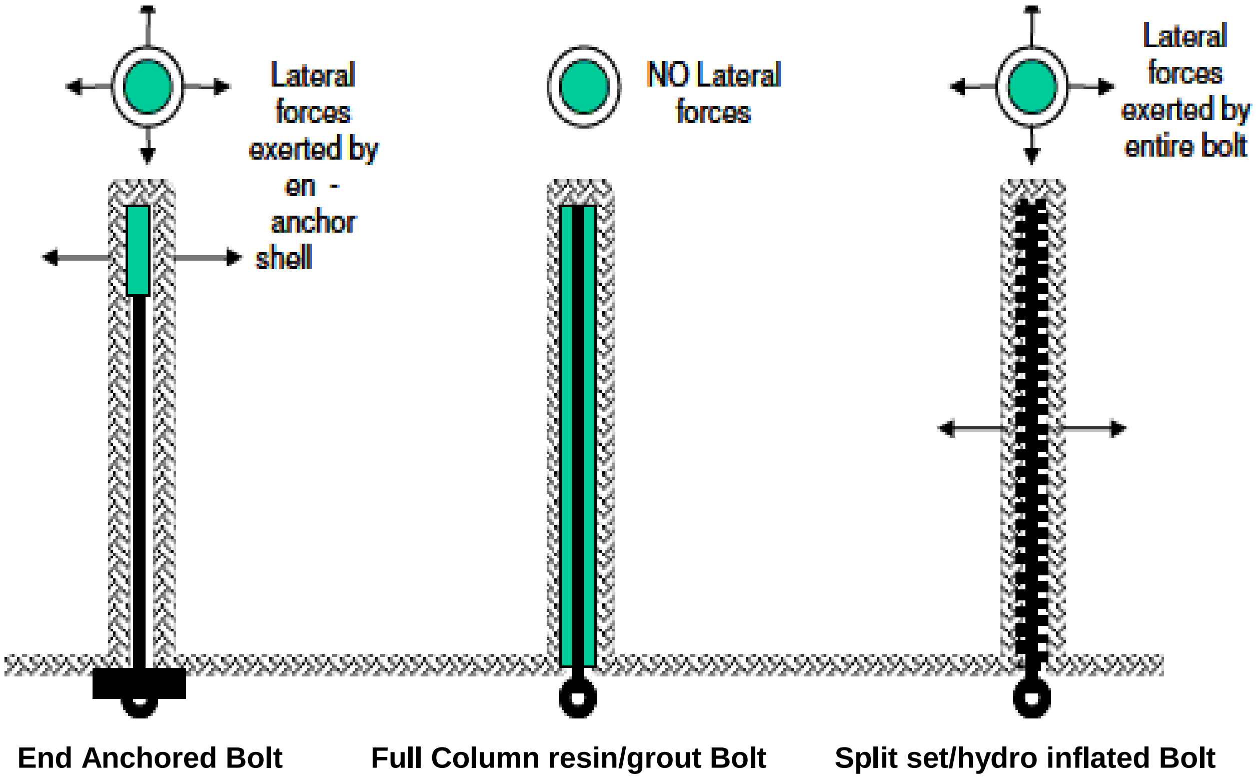


Figure 7.3.1 **Types of bolts that can be used with in-stope bolting.**
(Source: Henning and Ferreira, 2009)

All of the bolts indicated in ***Figure 7.3.1*** have been tested and used at KDC for various purposes non-systematically; however, with the introduction of in-stope bolting as a standard practice, the full column resin/grout bolt was the bolt of choice and detail regarding full column grout/ resin is discussed in the following section of this report. Standardizing of in-stope bolting material and procedures makes facilitation of training practices and auditing of workplaces more manageable.

Table 7.3.1 **Characteristics of different types of bolts used with in-stope bolting.**

Characteristic	End anchor/rock Stud	Resin/full column grouted	Hydro Bolt	Split Set
Equipment Requirements	T-spanner/Pneumatic torque wrench	Spinning adaptor/grout pump	25 Mpa pump, hoses and couplers	Pusher dolly
Installation Time	60 seconds	60 seconds	60 seconds with pump in good working order	60 seconds +
Full Column Support	No	100%	Approx 85%	Approx 85%
Active Support	Yes – if properly tensioned	Active when 2 speed resin system is used and tensioning is done before the setting time of 5/10 resin	Yes- if properly inflated	Yes-provided that hole diameter is correct
Typical Peak Load	10.5 tons	15.5 – 17 tons	10.7 tons	5 tons- slip in excess
Installed Reliability	70%	90%	80%	85% depending on proper installation
Sensitivity to undersized bore hole dia.(e.g. bit wear)	Not generally sensitive	Sensitive when oversized holes are drilled. Recommendation is to drill 28millimeters holes with 20millimeters resin bolt	Decreased performance due to partial inflation	Will increase. Difficult to install
Underground Resilience	Damaged plate, thread, if not properly tensioned	Excellent	Good, valve damage unlikely	Good
Shear Yield Ability	16-18millimeters	Low 7millimeters	16-18 millimeters	16-18millimeters
End Anchoring and Critical Bond Length	Anchor on shell only	Good with 2 speed resin and if tensioning of bolt is done	Friction anchoring over length of bolt	No end anchoring, friction only

Comments	Expensive, ineffective in not tensioned properly	Low cost, very strong and early strength when resin is used	Very expensive-corrosion	Corrosion-dependent on correctly drilled holes

(Source: Henning and Ferreira, 2009)

7.4 Resin Capsule and Rock Bolt Specifications

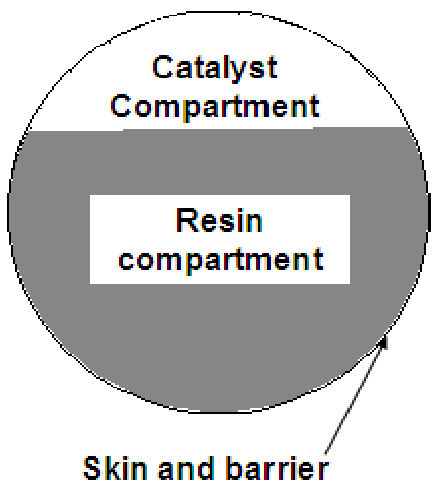


Figure 7.4.1 **Cross-section of a resin capsule. (Not to scale)**
(Source: Minova, 2011)

Figure 7.4.1 represents the cross-section of a resin capsule. A resin capsule is a two-compartment tube containing a polyester resin composition in one compartment and a catalyst composition in the other compartment. When the capsule is used the separation between the two compartments is broken and the two compositions are mixed together. The catalyst causes the resin to harden at a controlled and pre-selected speed. The hardened resin securely anchors the rock bolt into the hole, strengthening the rock.

The skin of the capsule is special polyester which is strong enough to contain and protect the contents during manufacturing, storage and handling, yet breaks up easily in the rock bolt hole, to completely release the two compositions and not interfere with the contact between the set resin and the rock or the rock bolt.

The hardened resin is a high-strength grout which completely fills the gap between the rock bolt and the rock, securely bonding the bolt into the hole. ***Figure 7.4.2*** indicates how this occurs.

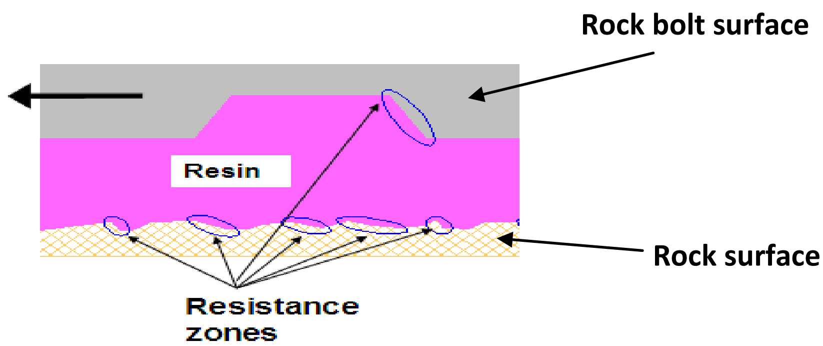


Figure 7.4.2 **Resin between the rock and rock bolt.**
(Source: Minova, 2011)

It is the rock bolt which strengthens the rock. The function of the resin is to transmit stresses between the rock and the rock bolt without allowing the rock bolt to move. The resin does this by filling all irregularities on the surface of the rock and the bolt. The two surfaces cannot slide past each other, nor can the rock close in towards the bolt. The rock is effectively locked into place by the resin and rock bolt combination. The resin is not glue, if the two surfaces are pulled away from each other, the resin will offer relatively little resistance.

The full volume between the rock bolt and the rock is filled with resin; the bolt can but does not have to be tensioned. The advantages of using the resin capsule are that the resin is stiff and has a highly effective suspension and clamping effect and it retains this effectiveness. It does not require a competent anchoring layer and the bolt is protected from corrosion.

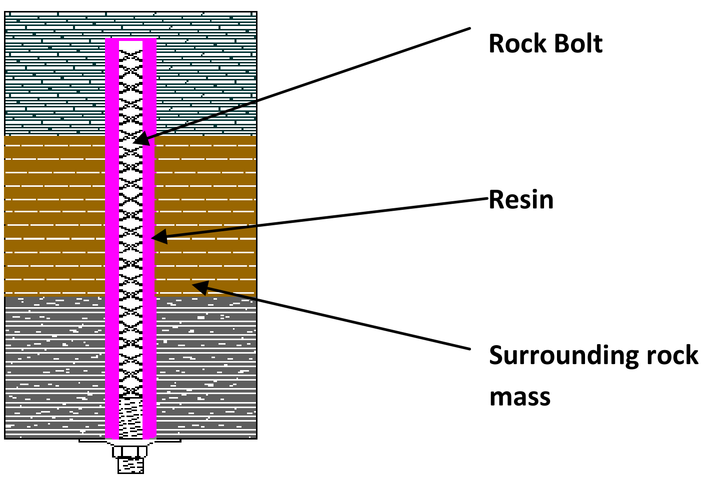


Figure 7.4.3 **Rock bolt, resin and surrounding rock mass.**
(Source: Minova, 2011)

Some of the precautions to be taken are that the correct size drill bit must be used when drilling the rock bolting holes and the correct length of hole must be drilled. Over sized holes will eliminate “full column” resin fill. ***Figure 7.4.4*** below indicate the hole depth and diameter.

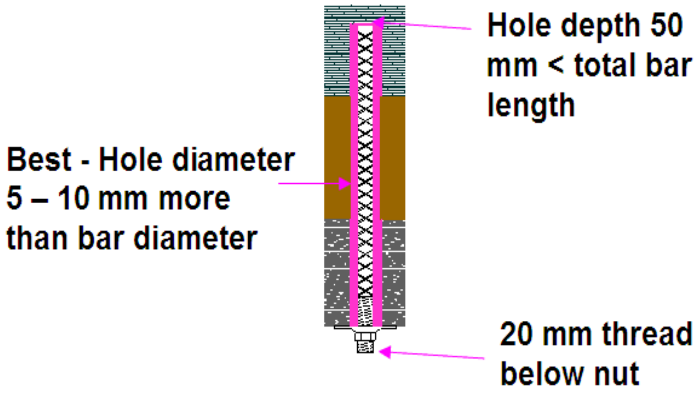


Figure 7.4.4 **Hole depth and diameter.**
(Source: Minova, 2011)

Figure 7.4.5 below indicates some important specifications when installing roof bolts.

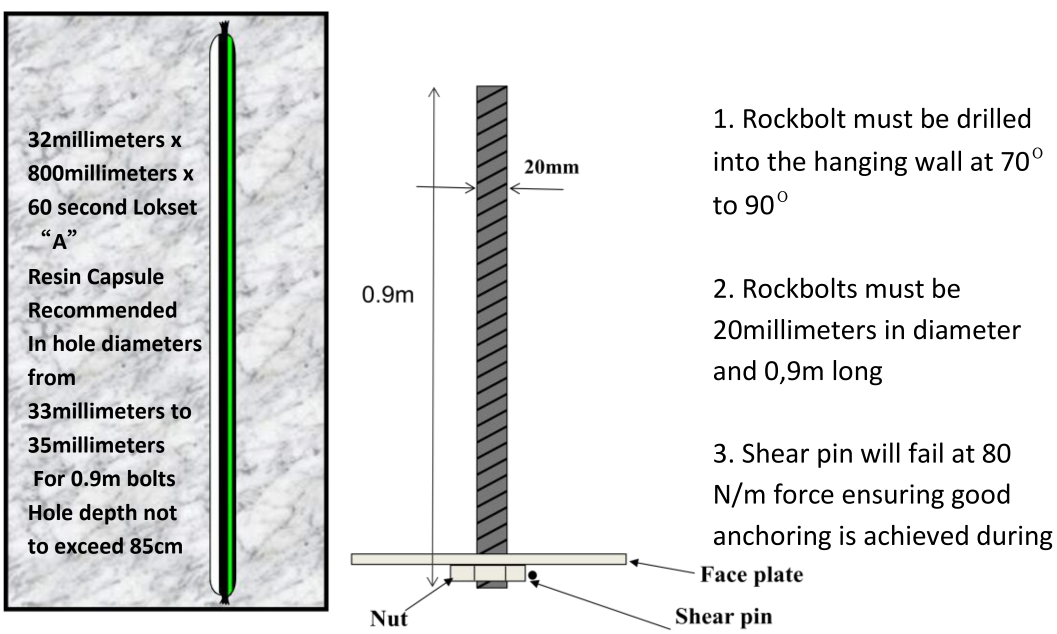


Figure 7.4.5 **Important specifications.**
(Source: Gold Fields, 2012)

7.5 Installation Procedure

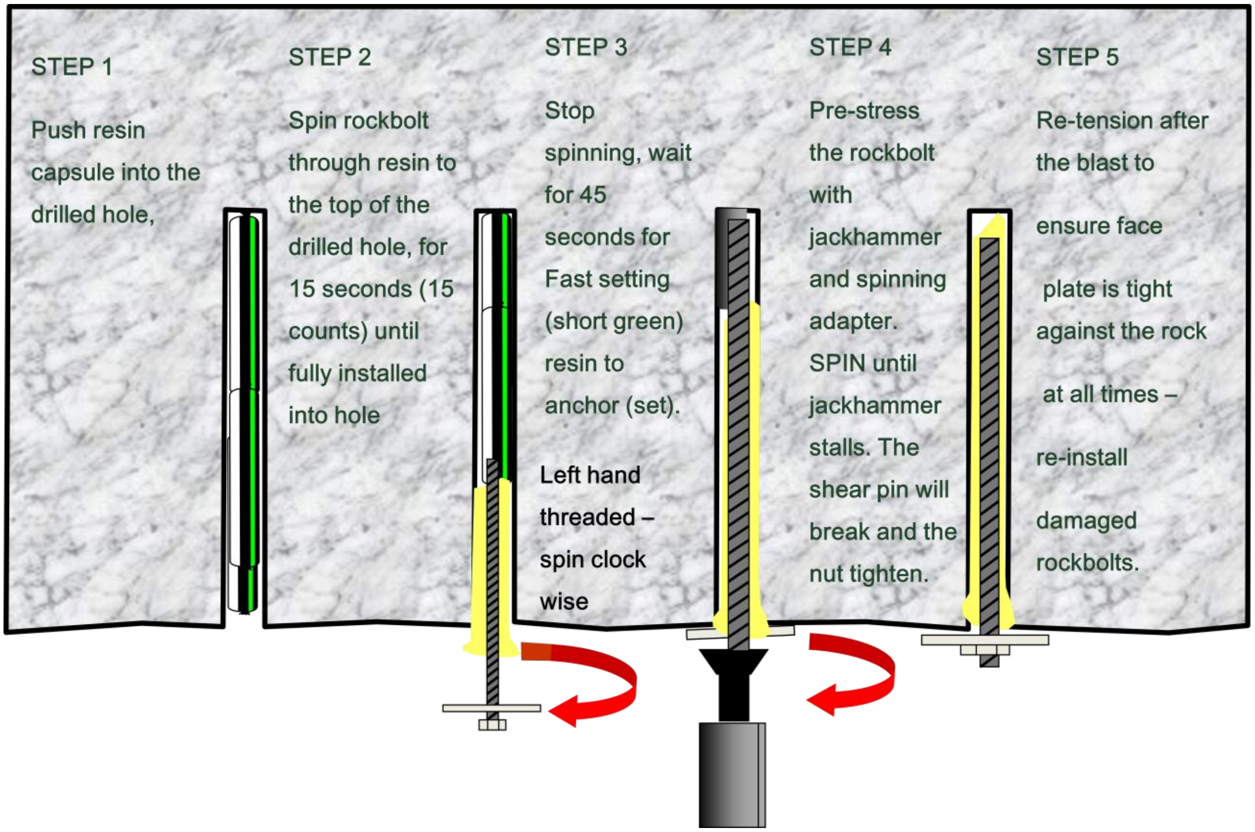


Figure 7.5.1 **Resin and rock bolt installation procedure**.
(Source: Gold Fields, 2012)

Figure 7.5.1 indicates the resin and rock bolt installation procedure that was introduced as part of the improvement made to the support standard at KDC in 2012. The procedure makes reference to spinning which is a process that breaks the skin of the capsule, releasing the resin and catalyst, mixing them together. The catalyst causes the resin to start setting. If the bolt is not spun enough, the resin and catalyst will not mix properly and the resin will not set hard rendering the bolt ineffective. If spinning is excessive, the partially-set resin is broken up and too weak to be effective. It is therefore, critically, important for the procedure to be followed accurately for in-stope bolting to be effective.

7.6 Conclusion

In-stope bolting was the last amendment made to the support standard at Thuthukani shaft and this standard had to be fully implemented during the course of 2012. Initial installation involved the drilling of roof bolts with hand held machines towards the end of 2011. This was received with resistance from the workforce as it implied additional work. However, the benefits outweigh this resistance as follows.

In-stope bolting offers a twenty- four hour permanent support. This means that employees working on different shifts benefit from this installation. The installation also strengthens the immediate hanging wall beam, unlike temporary support where pressure is applied to the beam and then released when removing temporary support. This constant application and releasing of pressure onto the hanging wall has a negative effect with regards to falls of ground.

Drilling and installation times have increased due to the added workload, however, cleaning times will be reduced due to less obstruction in the cleaning path in the form of temporary support legs and there will be less rock to handle from fall out. Dilution is also reduced when fall out is reduced during and after blasting operations.

Stope width control also improves when hanging wall conditions improve. In-stope bolting, which is a permanent installation, provides additional support in the back areas when sweepings are being conducted.

Other means of in-stope bolting in the form of rock consolidation, with spiral bolts, abutment plates, wire mesh and resin injections, conducted by private contractors e.g. WIMICO, have been conducted in the past and will continue in the future in areas where ground conditions warrant it , for example, in areas where the hanging wall lava is absolutely weak, WAF areas. This installation is an expensive procedure and is recommended by the responsible rock engineer after careful inspection of the affected workplaces and does not form part of the improvement of the general support standard for the mine.

Figure 7.6.1 **Rock consolidation at Thuthukani Shaft, KDC.**
(Source: Personal Collection)

Figure 7.6.1 shows two pictures indicating rock consolidation with abutment plates and wire mesh. The spiral bolt is used as a drill steel and is drilled into the hanging wall, the drill machine is removed and rock consolidation resin is pumped through the spiral bolt into the rock mass. This resin permeates into all the fractured zones, intersected by the spiral bolt, and consolidates the rock mass. Abutment plates with wire mesh are used to further reinforce the installation and rock consolidation.

Figure 7.6.2 **In-stope bolting at Thuthukani Shaft, KDC.**
(Source: Personal Collection)

Figure 7.6.2 shows the drilling of in-stope bolting and the installed bolt.
In-stope bolting was only introduced to the support standard in 2012 and the different methods and equipment will now be tested in an effort to make in-stope bolting effective and as user friendly as possible.

Chapter 8

8. CONCLUSION AND RECOMMENDATION

8.1 Summary and Conclusions

The mining of IBOG has been in existence for the past decade at Thuthukani Shaft, KDC. There is a large amount of investigation, planning, preparation, effort, time and money that goes into the mining of these IBOG and some of the detail was described in chapter two of this research report. During the planning phase, a register (**Appendix G** – a page from the register drawn up for Thuthukani Shaft, KDC) is drawn up of all the IBOG present at the shaft. This register provides details which firstly indicate whether the IBOG targeted are profitable or not. This is achieved by feeding all the necessary data regarding the specific isolated block of ground into a computer generated model which then calculates, using the latest gold price, whether the isolated block of ground is profitable or not. The program can be manipulated by changing the parameters with regards to cost and future gold price received to determine whether a specific isolated block of ground may become profitable to mine in the future. Monte Carlo simulations are then conducted by using a range of values to determine whether an isolated block of ground has the potential to add value to the company. Appropriate planning for that specific block of ground can then be determined.

A number of criteria related to survey, sampling, geology and rock engineering must be satisfied in order to ensure that the safety and financial risks are minimized or eliminated. This data is also fed into the model in the form of ratings. Safety is of utmost importance, and in some cases where the model has indicated an isolated block of ground being profitable to mine, poor rock engineering ratings have led to that block of ground being abandoned.

The register created, will indicate whether an isolated block of ground is profitable to mine or not, it will also indicate whether the block of ground is safe enough to mine or not. Non-profitable and/or unsafe blocks are abandoned. The register also indicates the sequence of extracting the isolated blocks of ground as determined by the rock engineering department. If this sequence is not adhered to, it could lead to further isolated blocks of ground being abandoned due to rock engineering safety factors being exceeded.

Together with the register, a master plan is developed depicting the IBOG and some of the information contained in the register. This plan is used by the production personnel to plan for Opening-up and rehabilitation of access ways to the IBOG.

Chapter three described and discussed the analysis conducted on seismic data, injury data, damage data and some production data captured from 2005 up to 2011. The injury and damage data analysis indicated a general decrease in injury and damage over the period investigated. Most of the data presented in this report indicate a marked improvement from 2008. This correlates to the improvements made to the support systems over time with the increase in support density up to 2008, the introduction of pre-conditioning as a standard in 2009, the introduction of in-stope netting in 2011 and finally, the implementation of in-stope bolting in 2012 which, in the authors opinion, will have a major positive effect with regards to preventing fall of ground accidents in the underground workings of Thuthukani Shaft, KDC.

Chapter five, six and seven describe some of the detail with regards to pre-conditioning, in-stope netting and in-stope bolting. A description of how they function and how they add value with regards to creating a safer mining environment is specified. It is interesting to note that pre-conditioning, in-stope netting and in-stope bolting have been in existence for many years and have been used on different mines and at specific work places where the individuals that required them motivated and justified their use. It is only in the recent past, that there is a focus and commitment towards drastically improving the safety in South African mines and a major drive to prevent fall of ground accidents that have created a need for the above mentioned to be explored and implemented systematically and systemically.

Analysis of seismic data from 2005 to 2011 have indicated that the support systems at Thuthukani Shaft, KDC , when mining isolated blocks of ground, have been continuously improving to such an extent that seismic events in excess of magnitude 1 have resulted in workplaces experiencing little or no damage. The 2011 statistics in ***Figure 4.7*** indicate that 95% of the total number of seismic events, with magnitude greater than 1, resulted in no damage to workings; however, there are still 5% of seismic events, with magnitude greater than 1, which did cause damage to workings. With the recent introduction of in-stope bolting to the support standard in 2012, it is expected that workplaces will be even more robust with regards to withstanding large magnitude seismic events.

Bearing this in mind, it can be noted that the mining of IBOG can be executed safely and economically. Most of the IBOG that were investigated at Thuthukani Shaft, KDC were found to be in a position to be mined safely and economically, but there were a few that were found to be unsafe or not economically viable, for example, in APPENDIX G, the block is shown to produce a profit margin of sixty four percent, however the risk relating to rock engineering criteria were found to be high and the recommendation was that this block could not be mined. The mining of IBOG require proper investigations, planning, sequence of extraction, and effort. It is an academic process that requires time and patience. There are no quick- wins or short-cuts. Support systems will have to be developed using industry best practices, implemented and strictly adhered to in order to achieve success. This does not mean that there will be a decrease in seismic activity or that seismic activity will be eliminated (*Vieira et al*), it means that seismic activity can be managed better and that the support system used will have the capacity to withstand fairly large seismic events, preventing injury to the workforce and damage to the workings.

8.2 Recommendation for Future Work

The author was unable to retrieve detailed production data which would have been work place specific. This information would have assisted in determining whether increasing production led to certain workplaces experiencing more seismic activity, whether improving the support standard led to a decrease in production output, and whether over supporting of workplaces led to no damage caused by large magnitude seismic events where production outputs were poor. The crew size linked to specific crew performances would also indicate what the effect of the improving support standard on production output, as observed over time would be. It is recommended that this data be captured and stored for future research work.

Analysis of the production data that was retrievable for the entire section indicated a direct relationship between production and seismic activity. Periods of high production indicated a larger number of seismic events in excess of magnitude 1, as described in chapter 3 of this research report.

It was noted that data capturing and record keeping had improved in the latter years of the research period. The author was unable to retrieve consistent records of the conditions of the support in relation to the standard after a major seismic event and it is recommended that proper investigations be conducted after major seismic events, whether damage is caused or not, in order to note whether deviations from mining standards are present or not and to note what had been done correctly in cases where no damage had been experienced. In many cases where no injury and/or damage are caused by a seismic event, the investigation is limited, or not conducted at all.

The introduction of in-stope bolting at Thuthukani Shaft, KDC as a mine standard was initiated in 2012. Data relating to the effects of in-stope bolting with regards to damage caused by seismic activity or the absence thereof is required for future analysis. This information will assist in determining if the addition of in-stope bolting to the support standard at Thuthukani Shaft, KDC is capable of reducing the damaged caused by the

5% of seismic events in excess of magnitude 1 that did cause damage to workings in 2011, further.

From the analysis of data and support improvements researched over a period from 2005 to 2011, the following high level process should be followed when attempting to mine IBOG and establishing a support system for this purpose:

1. Establish a company statement that will cause people within the company to act with energy, focus and enthusiasm. It was noted in this study that a major turning point was established when the Chief Executive Officer of Gold Fields Limited, Nick Holland, stated publicly, "*If we cannot mine safely, we won't mine at all*". This caused all persons within the company to really focus on safety and creating safe working environments. Production delivery targets dropped as shaft repair programs took precedence and ultimately a distinct change could be noted in the data presented in this report with regards to support improvements relating to seismic events in excess of magnitude 1, after this statement was made.
2. A thorough investigation of the targeted block and its surrounding area is required. This includes historical data relating to mine design, geology, rock engineering and sampling data that was captured regarding the area, and site inspections in the form of reconnaissance visits must be conducted in order to assess rehabilitation requirements and obtain samples to confirm historical data.
3. An analysis of historical data concerning the area targeted must be conducted to establish trends and behavior of the ore body and targeted block of ground.
4. Opening-up, rehabilitation and development plans must be established together with a schedule for execution.
5. A mine design plan, which will include a sequence of extraction of the ore body, from the rock engineering department must be developed and verified, preferably, by an external consultant.
6. A support system which must consider the unique properties and characteristics of the targeted block of ground as well as industry best practice must be designed with the intention for this support system to withstand large magnitude seismic events which seem to be more prominent when mining IBOG.
7. A system to capture and store data relating to the changes and improvements made to the support system over time must be developed so that proper analysis can be conducted in the future to determine the impact of changes made to the support system and future requirements.

As more deep level gold mines start to explore the IBOG that remain on their shafts with the intention of extracting them in order to prolong the life of their shafts, it is important that the high level points, above, be considered to ensure that safety and production targets are met.

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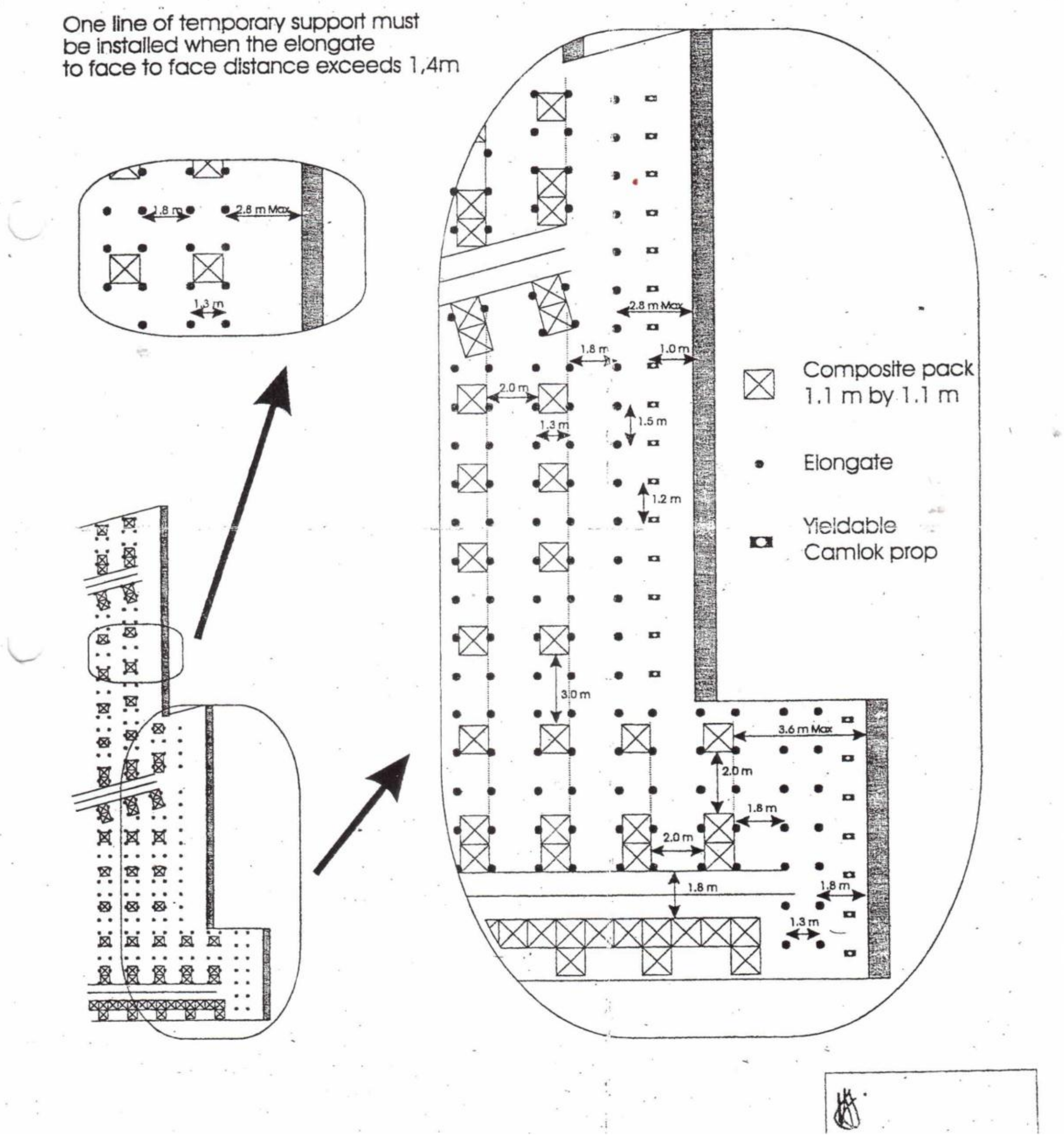
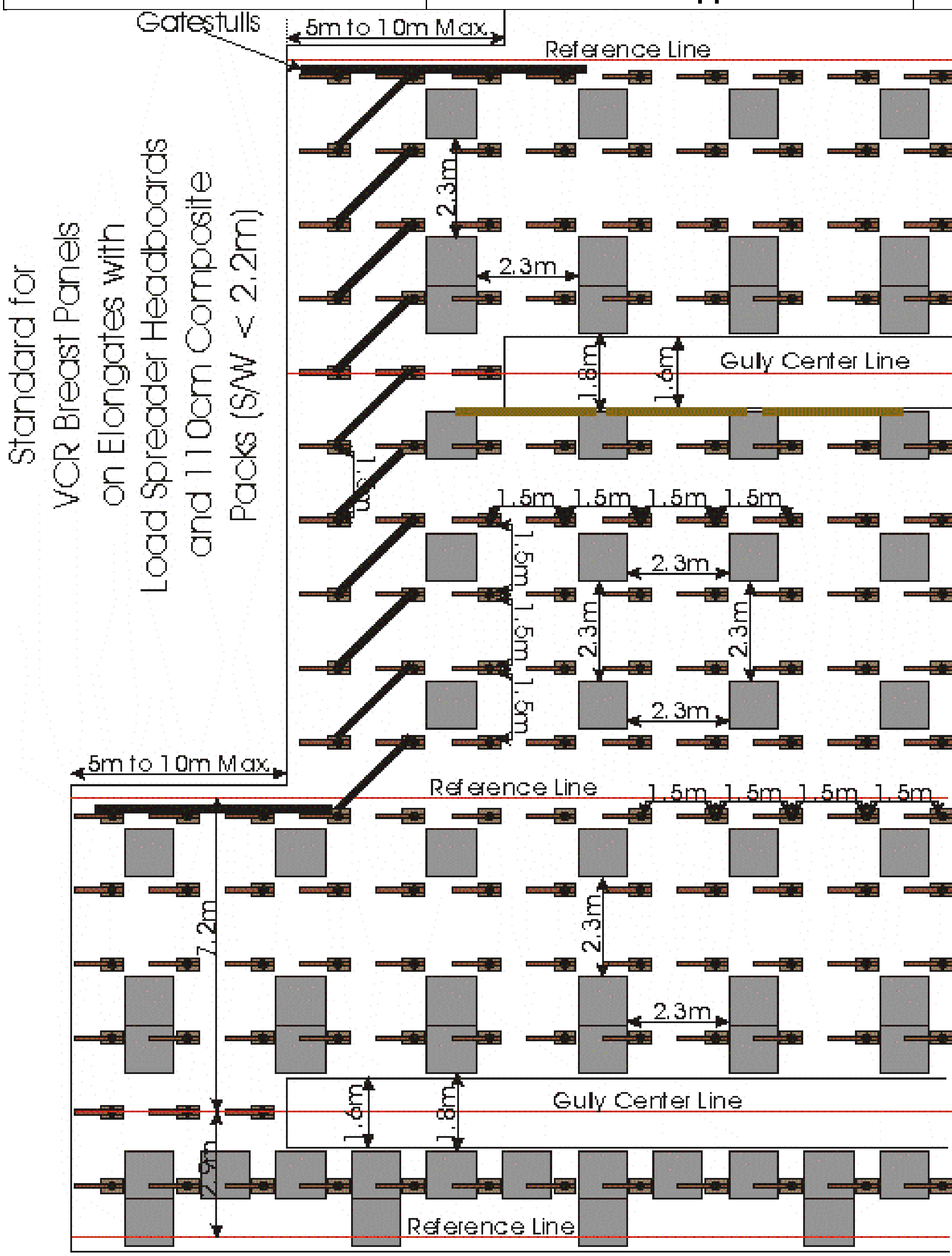
Kloof Gold Mine
Support layout for panels on
Composite packs and elongates A.S.P. 3.10



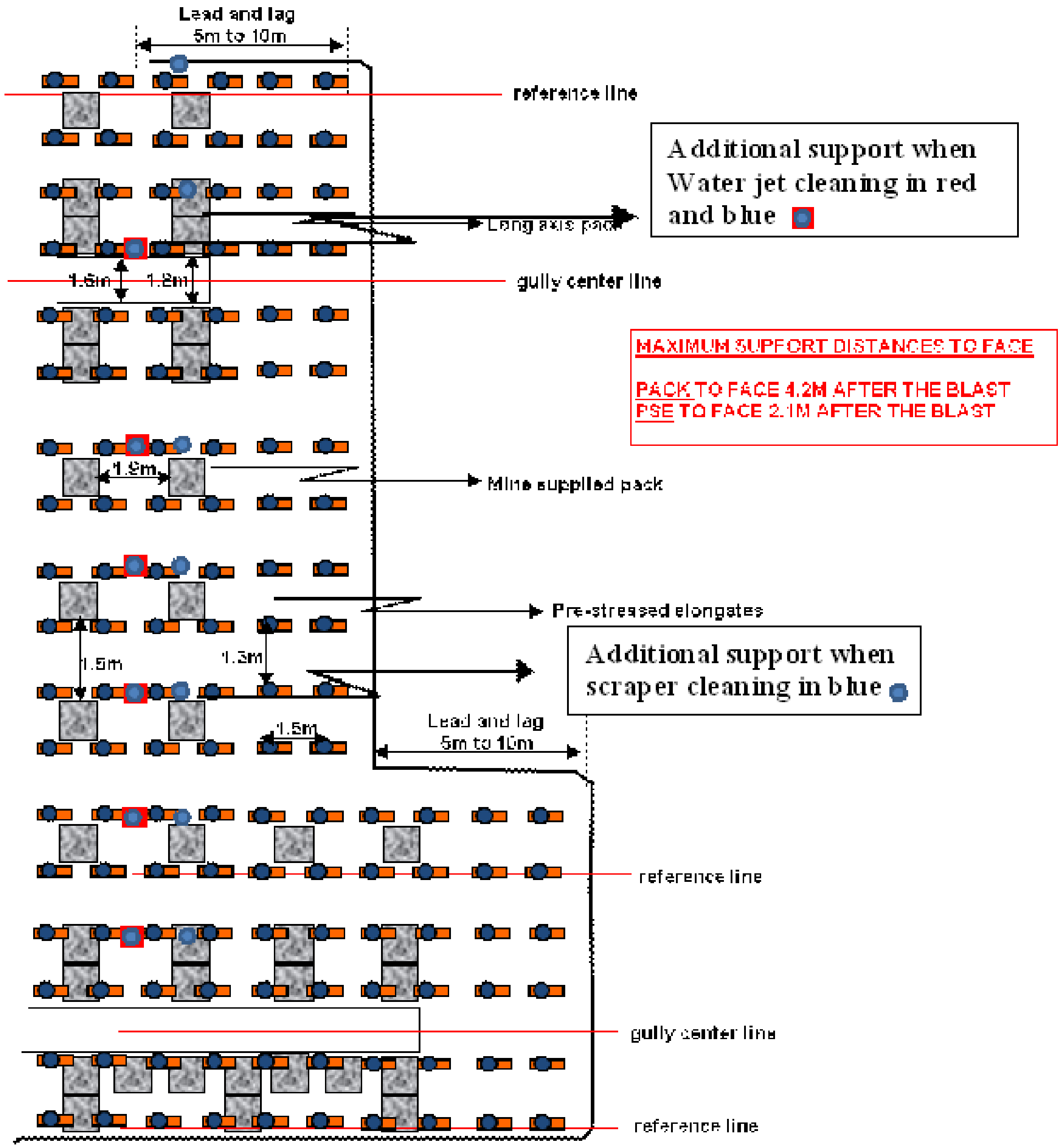
APPENDIX A

Kloof	Standard	Number: EN5
A.S.P 2.1.30A	Install Support	1.1 Revision 5 – June 2005



APPENDIX B

Kloof gold mine
Support standard for breast panels
using mine supplied packs and P.S.E's
Stoping width less than 2.2m



NOTE

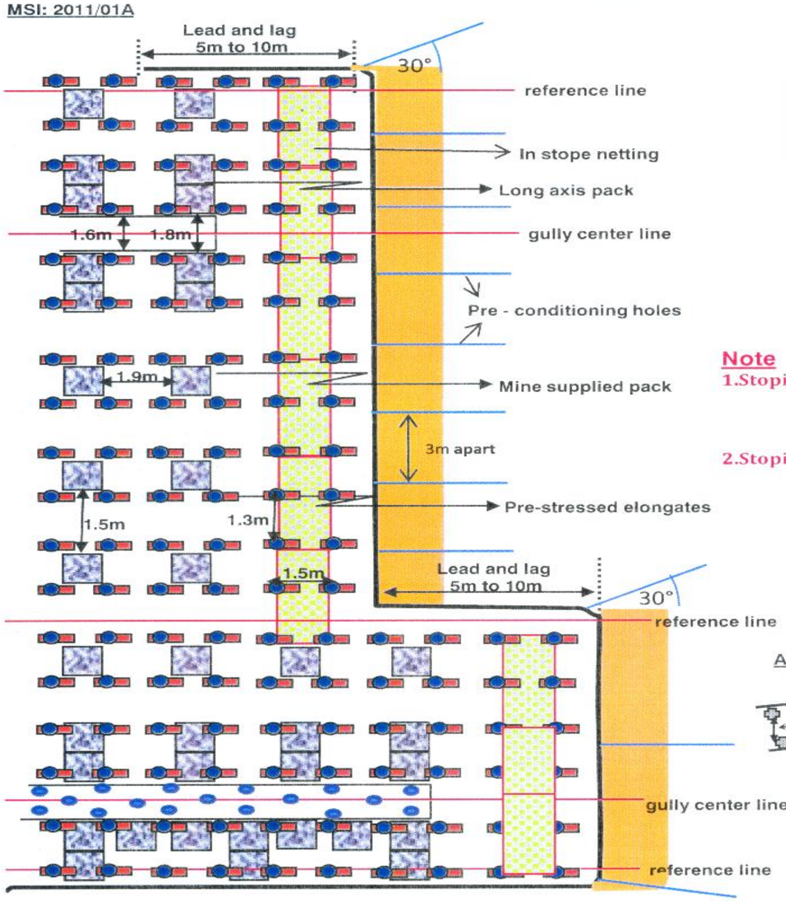
1. WHERE POSSIBLE UNSUPPORTED SPANS EXCEEDING 2M MUST BE REDUCED BY RE-INSTALLING ADDITIONAL SUPPORT.
2. NO PERSON TO WORK FURTHER THAN 1M FROM INSTALLED SUPPORT.
3. ALL ELONGATE'S THAT DISLODGED BETWEEN THE FACE AND THE SWEEPING LINE TO BE RE-INSTALLED.
4. ALL OLD AREA'S TO BE BARRICADED OFF.

KDC BU 5

Managerial Support instruction

Support standard for breast panels in VCR using mine supplied packs and P.S.E's

MSI: 2011/01A



MAXIMUM SUPPORT DISTANCES TO FACE

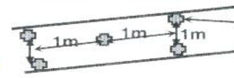
PACK TO FACE 4.2M AFTER THE BLAST
PSE TO FACE 2.5M AFTER THE BLAST

Note

1.Stoping Width less than 2.2m
- 1.1m x 2.2m Mine supplied packs on the gully
- 1.1m x 1.1m Mine supplied packs in stope

2.Stoping Width greater than 2.2m
- 1.1m x 2.2m Mine supplied packs on the gully
- 1.5m x 1.5m Mine supplied packs in the panel

ABUTMENT GULLY SUPPORT LAYOUT



NOTE
1. All dislodged or removed elongates between the face and the sweeping line must be re-installed
2. Maximum stoping that the support design can accommodate is 3m.
3. Where the gully width exceeds 1.8m additional support to be installed.
4. No person to work further than 1m from support
5. Back area's (behind sweeping line) to be barricaded off
6. No go zone std's and rules to be adhered to at all time.

10/6/2011
K Stead
SNr Manager Operations
BU 5

F.C. OLIVIER 9/6/11.
OPERATIONS MANAGER
KLOOF 1 #
B Olivier
Manager Operations
Main Shaft

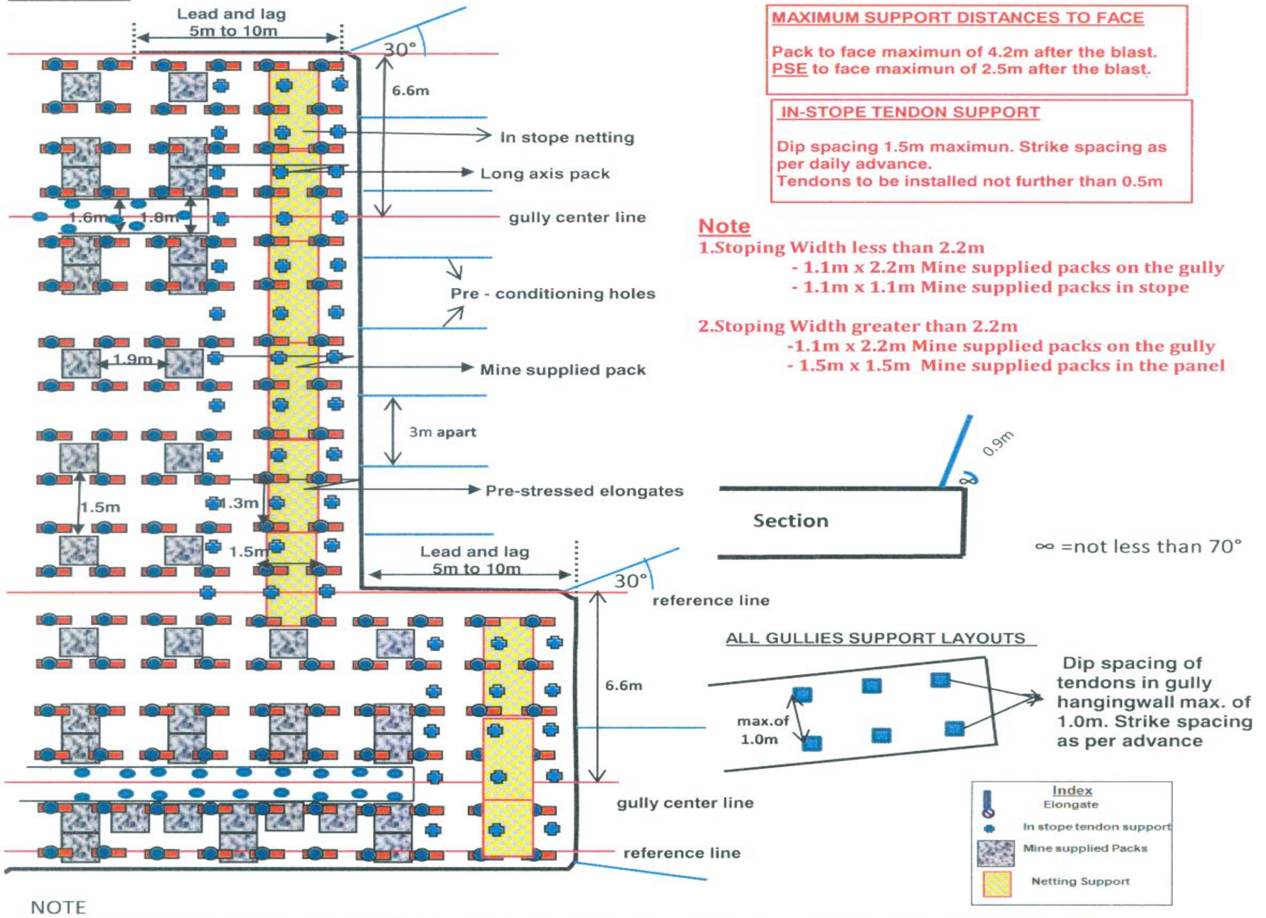
10/06/2011
S Kubheka
Manager Operations
No: 8 Shaft
Simphiwe Kubheka
Manager Operations
2011 -06- 1 0

8/6/2011
E Makinen
Unit Manager: Rock Eng.
BU 5
Erkki Makinen
Chief Rock Engineer
Kloof 1#

APPENDIX D

KDC BU 5
Managerial Support instruction
Support standard for breast panels in VCR and Kloof Reefs using mine supplied packs, P.S.E's and in-stope tendons

MSI: 2011/01C



NOTE
1. All dislodged or removed elongates between the face and the sweeping line must be re-installed
2. Maximum stoping width that the support design can accommodate is 3m.
3. Where the gully width exceeds 1.8m additional support to be installed.
4. No person to work further than 1m from support
5. Back area's (behind sweeping line) to be barricaded off
6. No go zone standards and rules to be adhered to at all times.

23/01/2012.
K Stead
SNR Manager Operations
BU 5

12/1/2012.
M Kriel
Manager Operations
Main Shaft

12/01/2012
S Kubheka
Manager Operations
No: 8 Shaft

12/1/2012.
T Thompson
Manager Operations
No:7 Shaft

E Makinen
Unit Manager: Rock Eng.
BU 5

APPENDIX E

Preconditioning checklist

Date (dd/mm/yy) 15 / 01 / 11
Shaft 2 SV
Shift boss M. MOIANE
Miner
Crew number
Workplace VC 23-43 N4A blast
Panel

Tick the appropriate box or enter a number where required

Equipment and material for preconditioning

	YES	NO	NUMBER
1. Is the correct drill-steel for preconditioning available (3 m)?	✓		
2. What is the length of the drill-steel for the preconditioning holes?			2.4 m
3. What is the length of the drill-steel for the production holes?			1.2 m
4. Is a charging stick of the correct length available?	✓		
5. Is sufficient tamping clay available for all the preconditioning holes?			

Conditions in the panel

	YES	NO	NUMBER
1. Is there an overhanging face?		✓	
2. Are there large unfractured slabs of rock in the face?	✓		
3. Is the hangingwall in a poor condition?		✓	
4. What is the face length?			
5. What is the stoping width?			

Preconditioning sockets

	YES	NO	NUMBER
1. How many preconditioning sockets can be identified in the face?			0
2. Is the number of preconditioning sockets appropriate for the face length (3 m spacing between sockets)?		✓	
3. Can the preconditioning socket in the top of the panel (angled into the abutment) be identified?		✓	
3.1 If this socket is found, what is the distance between this socket and the second one in the panel?			
4. Are the other preconditioning sockets spaced at approximately 3 m intervals on the remainder of the face?		N/A	
5. Are pairs of preconditioning sockets found (a socket from the previous day and one from the day before)?		✓	
6. Are all the preconditioning sockets of a crushed nature (good preconditioning)?			
7. Are there any of the preconditioning sockets that misfired or still contain explosives?		✓	
8. Are there any blowouts (large holes with no crushing)?		✓	
9. Are there any preconditioning sockets that show no crushing (holes not charged)?		✓	

New preconditioning holes

	YES	NO
1. Are the new preconditioning holes drilled approximately 50 cm from the existing preconditioning sockets?		✓
2. In high stoping width areas (> 1.2 m), are the preconditioning holes drilled 60 cm below the hangingwall?		✓
3. In low stoping width areas (< 1.2 m), are the preconditioning holes drilled in the centre of the face?		
4. Are the holes charged correctly (bottom 2 m filled with explosives, top 1 m filled with tamping clay)?		✓
5. Are the holes top primed?		✓
6. Is the timing of the preconditioning holes being done correctly?		✓

Feedback from miner and drilling crews

	YES	NO
1. Does the miner or drilling crews experience any problems with preconditioning?		
2. Do they see the improvements in panel conditions and the benefits of preconditioning?		

Comments from production people

Based on the information collected above, is preconditioning correctly implemented in this panel?

Yes **No**

If preconditioning is not implemented correctly, what remedial action is recommended?
It is recommended to drill the holes at 90° and the abutment holes @ 45° to the face

Designation: Supervisor Strata Control
Name: Gerard De Wee
Signature:

Note: Guidelines to assist with the completion of this checklist are available

APPENDIX F

PILLAR FEASIBILITY

Pillar Number	33/10A	Shaft	2 SV
Work Place	N33-38C	Date of Compilation	17/12/2010

Ore Reserve Information

Block No.	m²	cmg/t	Kg	SW	Mineable
KV33X34F-88	1360	4722	176.6		Yes
			0.0		
			0.0		
			0.0		
			0.0		
			0.0	ChW	
			0.0	86	
			0.0	SW	
			0.0	142	
				[illegible]	
Total	1360	4722	176.6		
Mineable	1360	4722	176.6		
Extraction	1088	4722	141.3		
Max depletion rate	120	4722	15.6		
Months production	9	4722	15.6		
Gold Recovered			124.6		
Revenue			R 35,514,117		

PARAMETERS	
Extraction	[illegible]
Block Factor	100%
MCF	90%
Rec. Factor	98%
Gold Price R/kg	285000
Dilution - cm	72
Costs	
O-Up R/m	8500
Dev. R/m	11026
Production R/ton	1628
Rails, Water, Air, Elect.	4000
Met Cost R/ton	74
Stope val g/t	33.7
Mill val g/t	22.3

Cost breakdown

	Unit	Quantity	Rands
Opening Up	metres	[illegible]	R 935,000
Construction	metres	[illegible]	R 80,000
FW Lifting	metres	[illegible]	R 110,260
Development	metres	[illegible]	R 760,794
Waste cut m³ to Waste		0	R -
Waste cut m³ to Reef		0	R -
Stope Prod + Met Cost	Tons milled	6343	R 10,795,854
Total Direct Cost			R 12,681,908
Direct Profit / loss			R 22,832,209
Profit Margin			64%

Final Recommendation.

Do not mine. RME Risk Matrix flag RED

Remarks

Initial Investigation Requirements:

MRM Evaluation:

Block values checked against the "spot a dog" value plan and the paleomorph model. Ore Reserve valuation accepted.

Name: T. Kompe Date: 05/04/11 Signature:

Geology:

Available structure correct as per plan. To be verified when on site investigation is completed.

Name: T. Kompe Date: 05/04/11 Signature:

Rock Engineering

Pillar extraction sequence: 33\7, 33\8, 33\10A

Mine as a series of up dips

Mining layout to accommodate 2nd escape route.

Name: S. Mulder Date: 14-04-11 Signature:

Recommendation: This pillar can not be mined.

Mining: Shaft Manager.

Opening up including services: portion of the FWDN +110m at 100%.

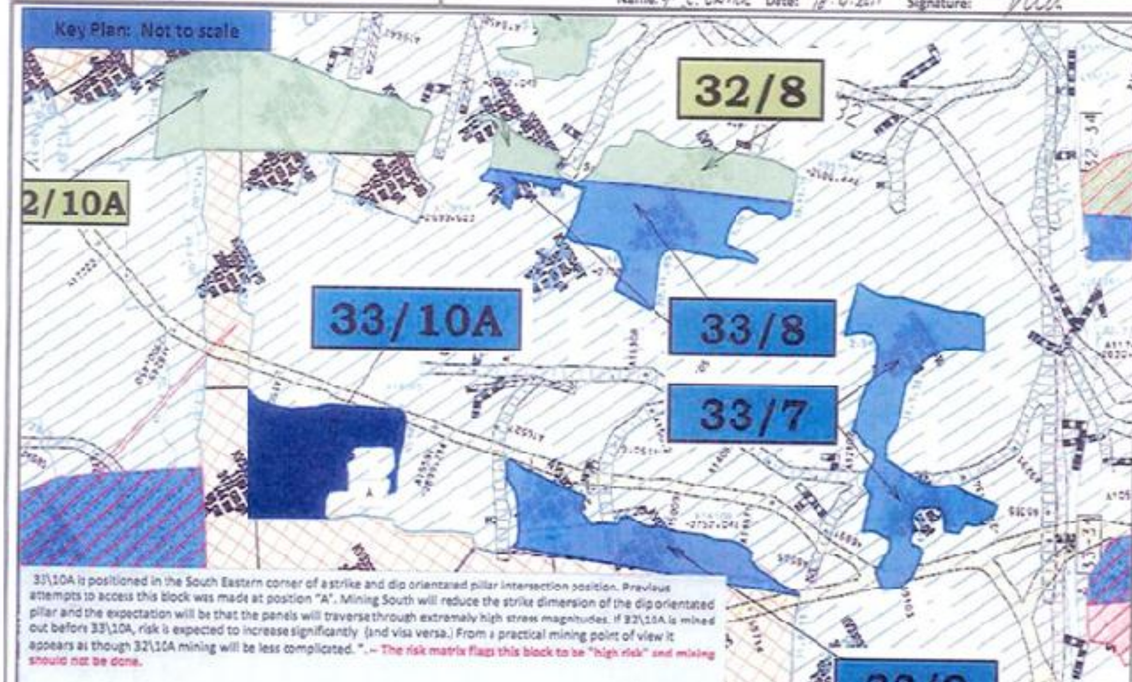
Dev: XC=45m, BH=9m, TW=15m, FWL=20m. Old TW 20m Re-equip for 2nd escape.

Richard Madhoormahed
Manager Mining

Name: Date: 13/04/11 Signature:

Operations / Senior Operations Manager

Name: F. C. Burne Date: 18-4-2011 Signature:



33\10A is positioned in the South Eastern corner of a strike and dip orientated pillar intersection position. Previous attempts to access this block was made at position "A". Mining South will reduce the strike dimension of the dip orientated pillar and the expectation will be that the panels will traverse through extremely high stress magnitudes. If 32\10A is mined out before 33\10A, risk is expected to increase significantly (and visa versa.) From a practical mining point of view it appears as though 32\10A mining will be less complicated. ". – The risk matrix flags this block to be "high risk" and mining should not be done.

Working Place	Pillar Number	Size of Pillar(m²)	Geology Structures	Geological Risks	Cluster (y/n)	Clamp Pillar (y/n)	Prev. Extract Effort	Seismic Rating	Seismic History	Stress Magnitude	Pill Shape & Position	Regional Pillar	Exp.Failure Mode	Rating	Risk
N33-38C	33\10A	<2500	Medium	Medium	No	Yes	Yes	High	High	High	Bad	Yes	Face	79	High

APPENDIX G

PLAN OF IBOG AT THUTHUKANI SHAFT, KDC