INCLINED CAVING AS A MASSIVE MINING METHOD

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1 Executive Summary

Finsch Mine is a Kimberlite diamond mine located at Lime Acres in the Northern Cape of South Africa. The mine was founded in 1961 and started surface mining in 1964. Underground production commenced in 1990 using a Modified Blast Hole Open Stoping method for the mining of Blocks 1, 2 and 3. Block 4 is currently being mined as a Block Cave.

The process of identifying and optimising a mining method to mine the Block 5 ore body started in 1991 and in 2006 Inclined Caving was identified as being a technically feasible method. This report aims to document the process employed in developing this method by the Block 5 Pre-Feasibility Team as well as discussing the technical challenges encountered during this process.

The report commences with a history of Finsch Diamond Mine and highlights the complex geology and threat of sidewall failure which prompted the decision to use Block Caving as the mining method for Block 4. A literature study of mines which implemented mining methods upon which the Inclined Cave was conceptualised is then presented. These practices were then used to form the basis of the designs on which the initial geotechnical modelling was done and built upon through an iterative process of modelling and design changes. The ventilation of the mining area, initial productivity simulation results and the applicability of automation and communition processes in the Inclined Cave are also presented.

The report concludes through an investigation into some of the challenges of the mining method that Inclined Caving is a technical option available for further investigation in determining the optimal mining method to be employed at Block 5, Finsch Diamond Mine.

2 Introduction

The Block 5 Conceptual Study was started in 1991 with a high level study and concluded that a Block Cave mining method would be suit the ore body. This study was restarted in 2000 by a confirmation study and in 2002 a full conceptual team was established. The results of this study indicated that a Block Cave design should be taken into the Pre-Feasibility study as the preferred mining method based on Net Present Value (NPV) pending the results of the proposed Geotechnical and Geological Drilling Program.

This proposal was taken to the Finsch Underground Geotechnical Study (FUGS) team, a panel of Cave mining experts, comprising of:

- Dr. Alan Guest (Lead)
- Dr. Dennis Laubscher
- Dr. Loren Lorig
- Jarec Jakubeck

The proposed mining method was dismissed as being impractical as a result of geotechnical and brow wear concerns and it was suggested that an "Inclined Cave" Mining concept be investigated. This method has never been implemented and the concept was based on the practices at King Section at Gath's Mine in Zimbabwe and Cassiar Mine in Canada that used a "False Footwall" mining method with some degree of success.

The Inclined Cave mining method for use at Finsch Diamond Mine was first documented by Paucar and Mthombeni in a paper presented at MassMinn 2004. This report aims to discuss the Inclined Cave Mining method as proposed for Finsch Block 5 by detailing the findings of modelling and design work.

3 History of Finsch Diamond Mine

Finsch Diamond Mine is located 160km west of Kimberley in the Northern Cape of South Africa and first started mining kimberlite in 1964 after the discovery of the pipe in 1960. Mining was started using open cast methods until an elevation of 430 m below surface was reached in September 1990 where after Modified Blast Hole Open Stoping (BHOS) was used to mine Block 1, 2 and 3. (Preece, 1998) Two shafts were sunk, namely Main Shaft, a 9m diameter circular shaft which is used for men, material and ore handling, and Waste Shaft which initially was used as a ventilation intake and to hoist but has since been decommissioned and is now used as a ventilation exhaust shaft

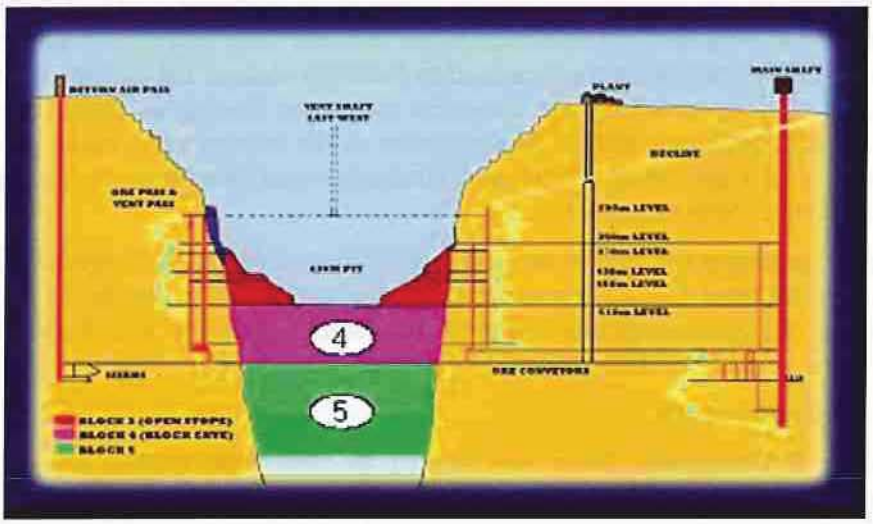

Figure 1: Cross section of the Finsch Diamond mine (Murray et al, 2003)

Block Caving was selected as the mining method for Finsch Block 4 with the main driver being the risk of sidewall failure in the pit if the Blast Hole Open Stoping method had been continued which would have diluted the ore being mined. Block Caving has been used in De Beers for a number of years at the Kimberly Mines and Cullinan Mine (Previously Premier Mine), and has gained favour because of the relatively low operating cost albeit with a high Capital cost associated with the mining method.

4 Geology

The Finsch Kimberlite pipe is located in a host rock of dolomites and limestone overlain with banded ironstone. The kimberlite has been classified into a series of 8 different kimberlites, namely F1 to F8 as depicted in Figure 2 .

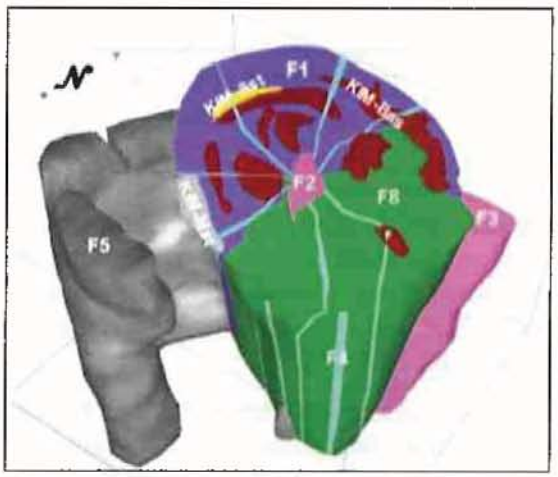

Figure 2: Top section of the Finsch Pipe showing internal facies and intrusions (Ekkerd, 2005)

The Rock Mass Ratings (RMR) of these kimberlites peak in the F8 kimberlites, these being the strongest, and are weakest in the F1 kimberlites. The following two broad zones can be distinguished and are depicted in Figure 3.

1) Southern half of the pipe – predominately F8 – Fair RMR (41 – 60 in brown in Figure 3)
2) Northern half of the pipe – predominately F1 – Poor RMR (21 – 40 in yellow in Figure 3)

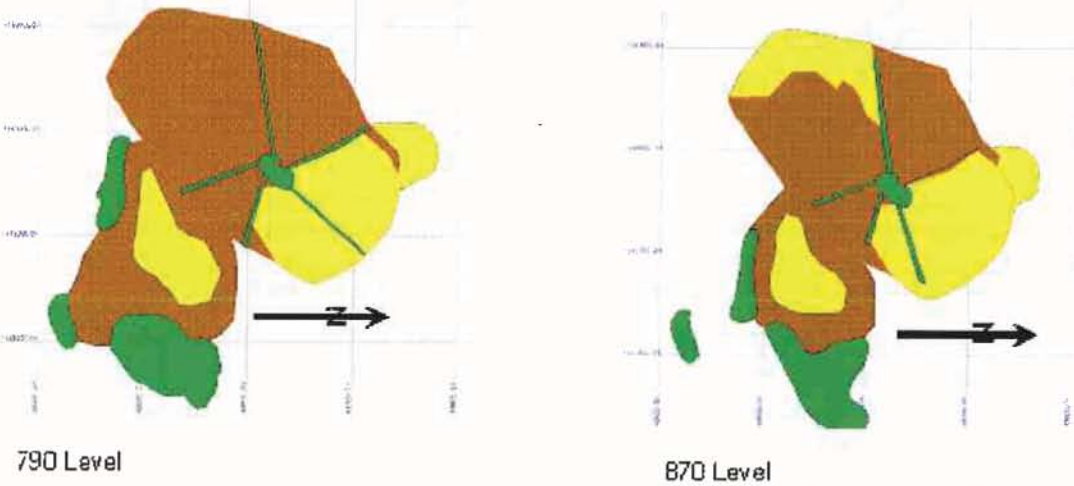

Figure 3: Data from Preliminary GEMCOM RMR Model with RMR rating 21-40 brown, 41-60 in yellow and 61- 80 in green. (Legast, 2006)

The host rock consists mainly of a series of horizontal layered dolomites and limestones overlain by a banded ironstone formation. From geotechnical drilling the ratings have been calculated as shown in Table 1:

Rock Type	UCS MPa	Min UCS MPa	Max UCS MPa	Elastic Modulus GPa	Tensile Strength MPa	Shear Strength MPa	Poisson Ratio
F1 Kimberlite	54	20	88	20	4	9	0.199
F8 Kimberlite	72	40	178	23	7	8	0.223
Dolomite all types	276	108	419	89	12	34	0.278

Table 1: Average strength for the main kimberlite type F1 and F8 and Dolomite (Lorig, 2005)

5 Block 4

The development of the infrastructure for the Block 4 Block Cave commenced before 1997 with slot cutting occurring in 2005. Although the tunnel spacing of 30m on the production level and 15m on the undercut was slightly greater than has been the case at some of the other mines in the group, a more stringent support regime has been created where high density roof bolting and cable bolting is the norm. This is complimented by shotcrete, straps and wire mesh.

Drawpoints are laid out in a single sided Herringbone or Henderson layout with semi-autonomous Load Haul Dump Machines (LHDs) planned to load directly into autonomous 50t Dump trucks. These trucks then transport the ore to the gyratory crusher, from where it is transported to the shaft by conveyor and hoisted to surface. These layouts can be seen in Figure 4.

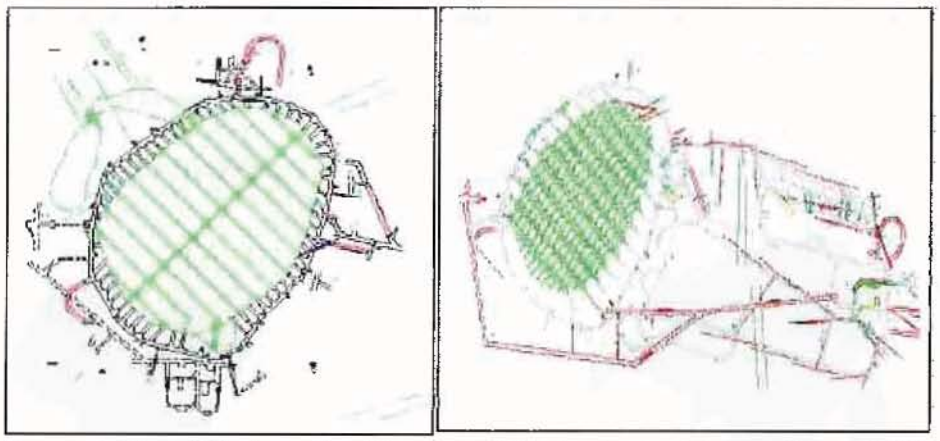

Figure 4: Block 4 Undercut Level Layout amd Production Level Layout (Jarvis, 2002)

6 Block 5 Conceptual Study (Murray et al, 2003)

The Block 5 Conceptual Study was started in 1991 focussing on the use of a Block Cave mining method. This study was reinitiated in 2000 by a confirmation study and in 2002 a full conceptual team was established and the study reinitiated. (Murray et al, 2003)

This Conceptual Study Team investigated three mining methods to determine which was the most applicable to the Finsch Block 5 ore body. These being:

- Sub level Caving (SLC)
- Block Caving (BC) and
- Front Caving (FC)

As part of this study Dr. Robin Kear was requested to conduct an economic desktop study and it was from his report that the mining block height and peak production rates were determined. (Murray et al, 2003) The main conclusions were as follows:

- The Cave should have at least 250m block height if the precursor were excluded and not mined.
- Mining should start in the high grade portion of the ore body and move to the lower grade portion so as to take maximum advantage of the early revenue and bolster NPV
- Mining the ore body at rates higher than 5.8 Mtpa showed little advantage and so a cap of 5.8Mt should be in place in future studies.

It was however soon realised that the 10 m thick zone containing intensely bedded secondary cryptagal dolomite with shale layers, intersected the country rock just above 880 meter level, very close to the minimum level recommended by Dr. Kear. This zone was found, through drilling, to be significantly weaker than the rest of the country rock mass and any infrastructure placed in this region would be subject to major support installations.

Above this zone, a previous exploratory drilling program in the kimberlite had drilled clusters of Large Diameter Drill Holes (LDD Holes). These holes were in the order of 300mm in diameter and once completed were never backfilled or grouted as a result of the poor ground conditions experienced. Over time these holes have filled with water which has spread to the surrounding kimberlite resulting in large zones of weathered material. These LDD holes are know to have stopped around the same elevation at which the shale layer occurs since this was felt to be the most likely level for the construction of a Block Cave.

The combination of these two factors has ensured that any cave mining operations have to be conducted below the level of the shale and therefore no shallower than 888m Level.

Based on these results the three proposed mining options were traded off against each other.

7 The Pre-Feasibility Block Cave

Post the October 2004 FUGS meeting the Pre-Feasibility Study Team initiated the design of an Incline Cave based on the recommendations of the above meeting (Lorig, 2004). Soon after however the scope of the project was set such that the project team not abandon the Block Cave mining method and an instruction was given that two variations of the Block Cave concept were to be designed in parallel with that of the Inclined Cave.

8 Geotechnical Investigations

The number of levels in an Inclined Cave is in direct proportion to the IDZ, tunnel spacing and footprint area of the ore body to be mined. This is best explained as follows:

- Each drawpoint can be assigned a tributary area dependant on the IDZ of the ore body
- Pillar widths are determined based on rock strength and the stress they are expected to be subjected to

- The greater the pillar width, the greater the distance between drawpoints
- The vertical spacing is determined as a function of the strength of the rock mass and height of draw

According to the IDZ theory at the time, the draw cones of adjacent drawpoints should be positioned such that they interact so as to promote interactive draw with the spacing of these drawpoints being determined by two dimensions, namely tunnel spacing and level spacing.

8.1 Tunnel and Level Spacing

Two different layouts can be created for the same ore body , namely a physically strong layout, which relies on geotechnical considerations, and secondly a layout that focuses on the ideal spacing of drawpoints so as to achieve interactive draw. As can be expected these two objectives are not complimentary and layouts are designed with a lower overall strength so as to obtain better draw or vice versa. It is seldom possible to create a balance between the two.

In the Finsch Block 5 design it was realised at an early stage in the project that the ore body being considered for Block 5 would be subjected to larger stresses than had been experienced in Block 4, mainly because of the depth of mining. Itasca had therefore been commissioned to perform modelling throughout the Pre-Feasibility stage of the project using both $FLAC^{3D}$ and $FLAC^{2D}$.

The block model was created from drilling data in GEMCOM and included the mined out pit and all subsequent mining including Blocks 1,2,3 and 4 resulting in the following mine scale model.

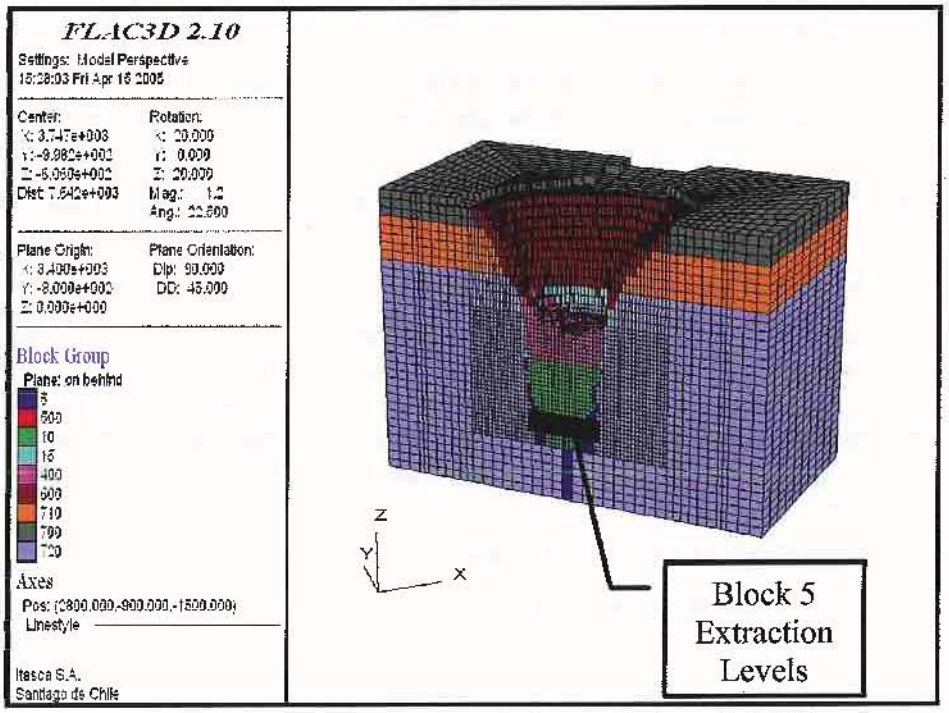

Figure 5: Mine scale model - view looking north (LORIG, 20005)

8.2 ITASCA Modelling – (Lorig, June 2005)

The first modelling was aimed at determining the need for an over cut in the Inclined Cave Layout. Two models were run with results indicating that although only a small benefit was achieved, an over cut did reduce the stress on subsequent levels and that an undercut would be essential to ensure caving up until at least four levels were in production.

8.3 ITASCA Modelling – (Lorig, Feb 2006)

The February 2006 modelling aimed to investigate the effects of mining activities on the mining block and determine which of the two proposed layouts, namely the Block Cave and the Inclined Cave presented the more competent layout. To do this both the Inclined Cave and Block mining layouts were modelled in 6 monthly increments, showing the progression of the development and then the undercutting of the ore body. The GEMCOM block model was again used but enhanced to show the facies differences.

From these results of the Rock Mass Strength / Stress analysis the following was concluded:

- Abutment stresses resulted in a strength to stress ratio of 0.05 to 0.15 in F1 kimberlite on 888 Level
- In the Block Cave, strength/stress ratios of 0.15 to 0.25 are anticipated just behind the undercut on 888 Level
- In the Inclined Cave, strength / stress ratios in the order of 0.2 to 0.25 in F1 kimberlite is anticipated ahead of the advancing undercut.
- Both options suggested that serious instabilities were possible and careful design of support systems would be required.

Overall, although the differences were small, the inclined layout showed an advantage over the Block Cave.

8.3.1 Uncaved Remnants

A further concern was that remnants remaining unbroken in the Inclined Cave would form arches across the ore body. This was also modelled and found to not be a risk as shown in Figure 6.

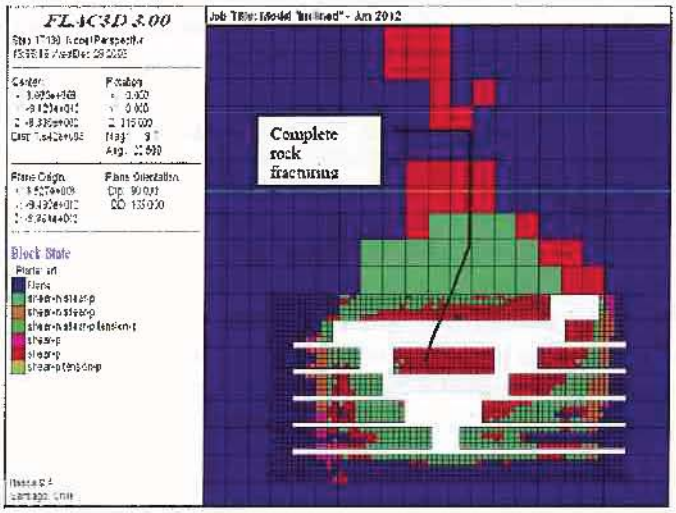

Figure 6: Vertical section indicating that complete failure will occur around the undercut (LORIG, Feb 2006)

The comparison of the two mining methods showed little evidence that one method was superior to the other, yet the differences did favour the Inclined Cave layout with "Serious instabilities" being identified in the contact zone around the pipe and in areas of lower Rock Mass Rating (RMR<40) and Uniaxial Compressive Strength (UCS≤55) where careful support designs would need to be done. (Lorig, Feb 2006)

8.3.2 Conclusions of the February 2006 Report

From the information provided above it was recommended that the following be considered to increase the strength of the Inclined Cave layout:

- Increase the size of the pillars between tunnels
- Since the tunnels were orientated perpendicular to the principle horizontal stress, rotating them would reduce the amount of damage anticipated. This would be impracticable due to the geometry of the pipe and the increase in tramming distances.
- Reducing the tunnel size in the "blue"
- Use a more circular tunnel geometry than currently is the practice
- Adjust undercut lead/lags

9 General Layouts

In the Finsch Block 5 Inclined Cave layouts, the ore body would be exploited from below the shale layer with the first full level being located on 888 Level. From this point initial designs have levels spaced at 18 meter intervals. Unlike other mining methods where the number of levels is pre-determined based on production requirements, the Inclined Cave relies on the spacing of tunnels, orientation of the tunnels on different levels relative to each other to determine the number of levels. These principles would now be discussed.

9.1 Vertical Alignment

It was decided early on in the project that tunnels of successive levels should be stacked as opposed to being arranged in a staggered layout so that a stronger layout would be formed with a single, continuous pillar being formed between the columns of successive tunnels. Had the tunnels been staggered, as is the practice on modern SLCs (Bull & Page, 2000) and the Koffiefontein Front Cave (Rabe & Hannweg, 2003), the constraining forces would have been less. Support required to ensure the stability of the brow would also have been increased.

9.2 Tunnel Orientation

The Finsch pipe has been found to be slightly elliptical (Legast, 2006) with the long axis lying on an East to West plane as can be seen below in Figure 7 with a precursor, of hypabysal kimberlitic material and more competent than the rest of the ore body, extending to the south. This precursor is not planned to be mined as part of the Block 5 Inclined Cave and it was preferred to locate all infrastructure and as many tunnels out of this zone as possible.

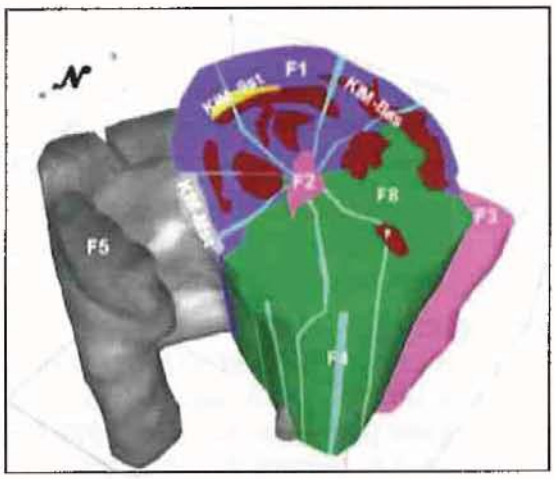

Figure 7: Top section of the Finsch Pipe showing internal facies and intrusions (Ekkerd, 2005)

The resulting decision was that tunnel orientation would be at a north / south orientation and would ensure that the distance that an LHD would need to travel within the pipe would be minimised. Most of the internal dykes would also be intersect perpendicularly reducing the risk of tunnel collapse in these areas.

9.3 Tunnel Dimensions and Spacings

Since RMR does not account for the effects of blast induced damage whilst MRMR does through the application of an appropriate factor which varies between 80% for poor blasting to 100% to boring operations. In the application by ITASCA, the 0.5m zone immediately surrounding the tunnel was reduced by 50%, far more than the MRMR factor while the subsequent 0.5m zone was reduced by 25% of the original RMR.

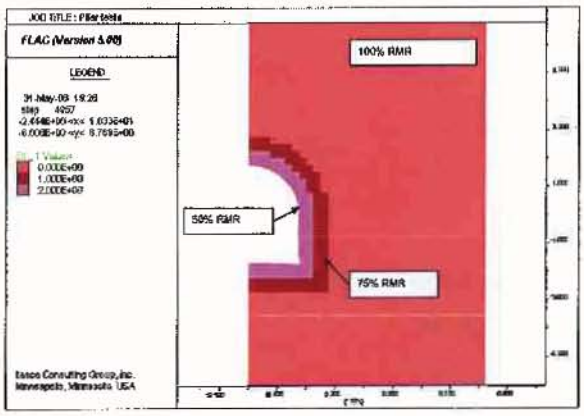

Figure 8: Assumed distribution of reduced RMR around mined tunnel (3.5m x 3.5m) (Lorig, June 2006)

9.4 Support

The primary support in kimberlite tunnels would begin with the controlled blasting of the rock to minimise over break. Thereafter a layer sealant woudl be applied to reduce the effects of weathering as the virgin ground is subjected to atmospheric conditions. Shotcrete would be applied through which grouted roof bolts of 3.0m length would be installed at 0.7m spacing. In the ITASCA modelling of June 2006 (Lorig, June 2006) it

had been found that roof bolts would also be required in some areas in the footwall to minimise footwall heave.

The primary support is then followed by cable anchors (which are intended to stitch the pillars together), wire mesh and vibro mesh as an aerial support medium and tendon cable straps. A final layer of shotcrete would then be applied over this installation

At drawpoints and dolomite / kimberlite contacts it is anticipated that stiffer support, in the form of steel arches would be required. This methodology has been used at Finsch and Cullinan in the past with good success rates.

9.5 Production Layouts

The following diagrams represent plan views of all levels from 870 Level to 960 Level. As can be seen, an undercut level has been designed on 888 Level with the aim of reducing stress for all subsequent production levels. On 924 Level another undercut is done, the mid cut, which is used to destress the levels below these. The method to be employed and scheduling of the undercut would be discussed under the heading Undercutting.

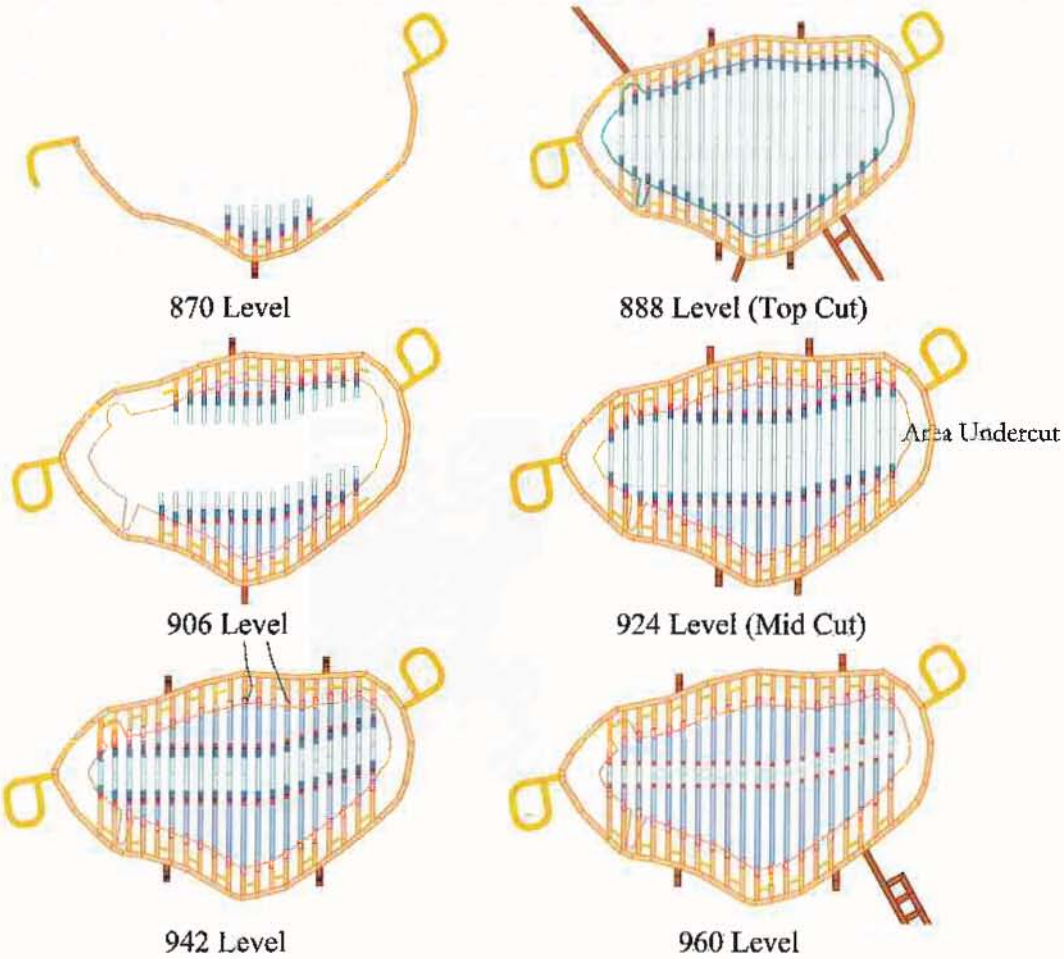

Figure 9: Inclined Cave layouts, (Not to scale) (Herselman, 2006)

The design of the above layouts was achieved by commencing the design on the bottom most level, situating drawpoints in the middle of the ore body. Subsequent level's

troughs were then placed so that they interacted with the lower drawpoints, ensuring an equal coverage of the entire ore body. Once the uppermost levels were reached any remaining footprint that had not been assigned coverage had a drawpoint associated.

10 Undercutting Methodology

10.1 The Need for an Undercut(s)

The effects of the inclusion of an undercut were modelled in terms of Rock mass Strength / Stress diagrams (Lorig, 2005). The large zones of reduced stress, indicate that less intense support systems would be required up to the fourth production level where after the benefit gained is no longer significant and a second undercut or "mid cut" would need to be included in the design. (Lorig, 2005)

10.2 Direction of Undercut

Block 4 would be undermined through the undercutting of Block 5 and as such mining in the same direction would be adventitious and allow the lag between the two blocks to be unconstrained.

10.3 Shape of the Undercut

A flat undercut design, moving from F8 to F1 was considered more practical for the following reasons:

- The undercut approaches the opposite contact at a more perpendicular angle
- In the F1 to F8 design, a long, thin pillar would be created as the face approaches the contact where as the F8 to F1 design ends in a more rounded pillar exposing less of the contact to the abutment stress.
- Moving the face from the south to the north would result in both options approaching the contact parallel throwing too much stress onto it.

When considering the chevron undercut the following can be seen:

- The tonnage profile is enhanced since tons can be loaded from both retreating faces, as more tunnels are available.
- In moving from F1 to F8 the undercut face would advance parallel to the opposite contacts inducing stress along the entire face.
- In moving from F8 to F1 the undercut approaches the contact at an acceptable angle which can be modified by changing the central point of the ore body.

11 Infrastructure

11.1 Development

Finsch mine is currently exploiting the Block 4 ore body which is located above Block 5 with production only expecting to start decreasing in 2011 (Finsch Mine, 2006). With only one hoisting facility currently installed on the mine, Waste Shaft was earmarked to be re-equipped to serve as a primary waste handling facility for Block 5. Waste shaft was originally installed to handle waste from underground operations and was decommissioned and is currently only used a return air way. The new loading station would be constructed on 888 Level with tips into which 20 ton dump trucks can discharge development waste rock.

Main shaft would be deepened and the hoisting speed increased to allow it to hoist 5.4Mt per annum. During the deepening, a spare compartment in the shaft would be equipped with a 7 ton skip and winding arrangements which would allow waste from

shaft deepening and development to be hoisted to surface. During shaft deepening a sub shaft assembly would be installed that would hoist waste from the lowest level of the mine on 1020 Level using a kibble on a multi-drum winder to the existing 65 Level.. This would then empty the ore into a 70°, 5 meter diameter pass which would feed into a loading flask and ultimately the 7t skip. On completion of Main shaft deepening and equipping this 7ton skip could be decommissioned although the availability might prove useful in later expansion projects.

11.2 Steady State

In the Finsch Block 5 Pre-Feasibility Inclined Cave layout diesel LHDs would tip into tips which would be equipped with a rock breaker and link into a 6 meter diameter ore pass located on either side of the pipe. These would each be equipped with a hydrostroke feeder on the haulage level and would feed into 50 ton dump trucks. The ore would then be transported to the tip in one of the two tramming loops. The simultaneous tipping of three trucks would be incorporated into the design of the tip before the ore is fed into the crushing assembly.

The choice of communition method would be decided in the feasibility study phase of the project with the following three options in contention:

- Mineral sizer
- Gyratory crusher
- Jaw crusher.

From the crusher the -300mm ore would be transported on a conveyor belt to the main shaft loading box and hoisted to surface.

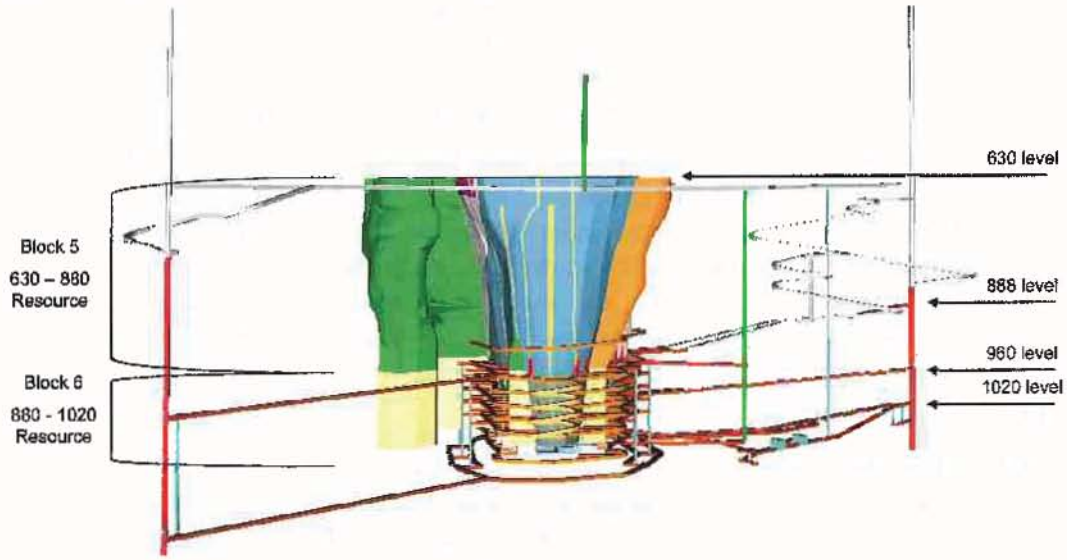

Figure 10: Graphical representation of the Block 5 Inclined Cave design (Herselman, 2006a)

12 Ventilation

All tunnels in the Inclined Cave layout would form "dead ends" on all levels at some stage in their lives, with the top and mid cuts starting off as holed tunnels before they are undercut and ending up as "dead ends", and would need to be ventilated with an exhaust system using ventilation ducting with two tunnels being serviced by a single

raise bored ventilation hole leading to the ventilation and water handling level. The cost of such a system would be expensive in terms of capital and operating costs.

The Capital costs are affected mainly by the cost of purchasing all the required ventilation ducting and fans with the running costs being driven mainly by the cost of running fans in every tunnel. Two drivers exist to control this cost, the first being to reduce the number of fans by placing larger fans after the point where the two ventilation ducts join and secondly using a Ventilation on Demand (VoD) system to control the amount of time the fans stay on.

At its most complex this system would sense the presence of an employee or machine and turn on the fan, turning off when the person or machine exits the area.

13 Communition and Secondary Breaking

In the Inclined Cave layout, primary fragmentation would occur as a result of long hole drill and blasting at the troughs and undercut. This would, as discussed earlier be initiated by creating the free breaking face by raise boring a hole beyond the top of the undercut into which subsequent rings would be blasted. The swell would be loaded off between these blasts with an LHD. Little oversize material is expected from this as rings would be design in such a manner that rocks larger than that can be handled by an LHD would not be produced. Secondary breaking is required for rocks larger than what can be safely loaded by an LHD from a drawpoint and transported to the tip.

14 Areas of Concern

14.1 Air Blast Potential

Following on the experiences at North Parkes (Hebblewhite, 2002), a lot of emphasis has been placed on the possibility of air blasts in new cave designs.

In the Inclined Cave the same concerns over air blasts would be valid as with conventional block caves with the same principles being used to address the concerns, namely the cushion of ore left in the drawpoints, adequate monitoring of the cave back and proper draw control.

14.2 Mud Push Potential

As with any caving operation there is the possibility of mud pushes and mud rushes occurring. Where the Inclined Cave is more beneficial over methods such as SLCs and Front Caves is that because of the number of levels and footprint over which these occur, the ingress of water can be diverted between a number of drawpoints, in much the same way that water is handled in Block Caves through preferential draw of selected drawpoints. (Laubscher, 2000)

15 Conclusion

Inclined Caving, although as yet not proven through implementation, has been found in this pre-feasibility study to be a technically feasible as a mining method which can be employed in the mining of Block 5 at Finsch Mine. This was done through the appropriate use of technology, in the form of geotechnical modelling and the involvement of industry experts in the form of the FUGS team.

This report has detailed the major advantages that the Inclined Cave has as a mining method in Kimberlite ore bodies, these being:

- Ability to increase brow wear by retreating drawpoints where necessary
- Layout can be tailored to suit the ground conditions to ensure pillar strength and tunnel stability
- Drainage of groundwater is simpler than in block caves since water can be drawn over a number of levels
- Rehabilitation and secondary breakage has fewer adverse effects on production than in other mining methods due to the layout
- The development and construction schedule is felt to be manageable and achievable
- The risk of boundary wedge failure is reduced because of the placement of drawpoints on the contacts
- Higher production capability is achievable as a larger number of tunnels are available than in a SLC layout of the same size. making for better utilisation.

There are a number of factors which need further investigation in a subsequent phase of the project. These include optimising tunnel dimensions and the resulting support regimes and modelling the stresses induced on the ore body through various orientations of the undercut. The results of these studies would conclude the technical feasibility study into the use of the Inclined Cave Mining Method for use at Block 5, Finsch Mine.

16 Acknowledgments

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Donovan matriculated in 1996 from Christian Brother's College in Boksburg where after he commenced his Bachelor's degree in Mining Engineering at the University of the Witwatersrand, graduating in 2000 on a full De Beers Bursary. In 2001 he started on a graduate training program at De Beers' then Premier Mine where as part of the program obtained his underground blasting ticket and Mine Managers Certificate. Moving into the planning section at Cullinan fuelled his interest in mine design and planning where he was heavily involved in short term planning for the mine as well as a number of on-mine projects including the Cullinan BAW project, a JV with Hatch and Murray and Roberts Cementation, until its conclusion in 2005. At this point he was transferred to De Beers' head office and formed part of the Finch Block 5 project until its conclusion in April 2007. Donovan obtained a Graduate Diploma in Engineering in 2004 and converted this to a Master in Engineering in 2007 based on a research project entitled, Incline Caving as a Massive Mining Method through Wits. Following his passion for project work, he joined TWP Consulting in May 2007 as a Mining Engineer and was appointed as Lead Mining engineer on the Sheba's Ridge Bankable Feasibility Study Team soon after. Whilst on this project he was responsible for the ultimate delivery of the mine design as well as the associated mining infrastructure, including an investigation into In Pit Crushing and Conveying. In August 2008 Donovan was appointed Section Head: Mine Design and Simulation in TWP's Mining section and has continued to maintain his focus on Mine design as well as to develop the mine design capabilities in that company. He has also been involved in a number of smaller projects, both surface and underground, at various levels from Due Diligence reviews through to Bankable Feasibility Studies. Donovan has been admitted as a Professional Engineer through the Engineering Council of South Africa and written and passed the Project Management Institute's Project Management Professional Certification programme. He is also a member of the South African Institute of Mining and Metallurgy, where he serves on the organising committee of the Diamonds Source to Use Conference 2009 and an Associate of the Mine Managers Association of South Africa.