A guide for managing the resource model of the copper block-cave at Palabora Mining Company

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

Phalaborwa, 24 May 2018

DECLARATION

I, Johannes Jacobus Bezuidenhout, declare that this Project Report is my unaided work. No other person's work has been used without due acknowledgement in the main text of the Project Report. The information used in the Project Report has been obtained while employed by Palabora Mining Company. It is submitted in partial fulfilment of the requirements for the degree of Master of Science at the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination at any other university.

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(Signature of Candidate)

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ABSTRACT:

The renowned PMC's copper open pit operation transitioned from surface copper operations to an underground operation in 2002. An exploration shaft from within the open pit, having an exploration tunnel below the open pit bottom served ideally for downward resource exploration drilling. Palabora Underground Mining Project was a first to cave in very competent lithology rock types which utilised the crinkle cut method at its undercut level. Unfortunately, the inadequate underground exploration drilling limited the resource classification and confidence levels, having inadequate drilling to represent the vast footprint block cave area. Consequently, the head grade and the modelled grade required annual revisions. The head grade and modelled grades diverted from each other more than once, despite all the numerous studies with minor and significant model improvements. The block model refinements included adjustments made to the block sizes, draw column alignments with blocks and additions such as the dolerite dilution representation within blocks. The resource model revision pointed firstly to the grade change between the mill grades and predicted modelled grade, and secondly to the identified geometric change and rectification thereof. Significant technical studies refined the resource model to satisfactory levels of confidence. However, the elusive cave behaviour encouraged more studies and refinements as new information became available over time. The copper open pit's north wall failure occurred in 2004, and this failure material contributed to significant resource losses. The importance of the many approaches and models which predicted or assumed the possible block cave's life after the failure characterises the PMC block cave uniquely. Some of the significant studies over time, encapsulated in this project report sketch a realistic timeline of the copper block cave at Palabora Mining Company. The initial resource losses became somewhat redeemed during the 2015 study where some of the copper gains were within the failure's glacial flow, and not from the likely toppling effect which injected some additional years to the life of mine.

Dedicated to:

Janco, Luan, & Caleb Bezuidenhout

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The opportunity and permission to use portions of material contained in this Project Report are gratefully acknowledged. The opinions expressed are those of the author and may not necessarily represent the policies of the companies mentioned. While recognising the valuable contributions of the preceding people, the author alone is responsible for any errors, omissions, and ambiguities remaining in this project report.

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NOMENCLATURE

AM	Autogenous Mill
BCF	Block Cave Fragmentation
BCB	Banded carbonatite
CFZ	Central Fault Zone
Cu	Copper
CP	Competent Person
CPR	Competent Person's Report
3DEC	A three-dimensional numerical modelling code
DBM	Draw block Model
DCM	Draw column Model
DFN	Discrete Fracture Network
FS	Feasibility Study
$FLAC^{3D}$	Fast Lagrangian Analysis of Continua - Modelling software code using flags by specific algorithms
GEMCOM	Geology mine Software package (GEMS) by GEOVIA Incorporated & Dassault Systèmes
GEOVIA	Dassault Systèmes Geovia Inc.
HIZ	Height of Interaction Zone
HOD	Height of Draw
HP UNIX	Hewlett Packard Unix operating system based on UNIX System V
ICS	International Caving Study
IRMR	Intact Rock Mass Rating
ITASCA	Itasca Consulting Group International
JORC	Australian Joint Ore Reserves Committee
JSE	Johannesburg Stock Exchange
km	kilometre
KNP	Kruger National Park
LHD	Load Haul Dumper
Lift 1	Lift one – First lift mine below the open pit (noun)
Lift 2	Lift two – Second lift mine below the first lift mine (noun)
LOM	Life Of Mine
LO	Longitude of Origin
mm	Millimetre
MineCad	Mining Computer-aided draughting software
MIK	Multiple Indicator Kriging
MISK	Multiple Indicator Simple Kriging
MIOK	Multiple Indicator Ordinary Kriging
MPa	Megapascal
MRMR	Mining Rock Mass Rating

NNWSO	North-northwest Structural Orientation
OK	Ordinary Kriging
PC	Personal Computer
PCBC	Geovia Personal Computer Block-Cave software supplied by Dassault Systèmes
PFC^{3D}	Particle Flow Code - 3 Dimensional
PMC	Palabora Mining Company Pty. (Ltd).
PUMP	Palabora Underground Mining Project
QAQC	Quality Assurance, Quality Control
REBOP	Rapid Emulator Based on PFC3D
ROM	Run of Mine
S.A.	South Africa
SAMREC	South African Code for the reporting of exploration results, Mineral Resources and Mineral Reserves
SRK	Consultancy company
SRM	Synthetic Rock Mass
Snowden	Snowden Mining Industry Consultants
SWFZ	Southwest Fault Zone
TCB	Transgressive carbonatite
Tpd	Tonnes per day
Tpa	Tonnes per annum
TDR	Time Domain Reflectometry
UCS	Unconfirmed Compressive Strength
UJRM	Ubiquitous Joint Rock Mass

1 INTRODUCTION

1.1 Overview

This section gives an overview of the purpose of the study, the background and geological setting, and justification of the project report. This section lays the foundation for what one can expect in the report, what will be determined, investigated, not attempted, and what the project report would analyse and ultimately achieve.

1.2 Purpose of the study

The project report serves as personal gaining knowledge of block cave mining in general and at Palabora Mining Company (Pty) Limited, (PMC).

The PCBC (Personal Computer Block-cave software) resource model grade indicated a slightly higher-grade value than the head grade until June 2005. From June 2005 until June 2008, the PCBC and the head-grade followed each other within reasonable limits. After June 2008, the PCBC grade indicated higher values than the head-grade, which resulted from periodic resource reviews and the actual mined draw points over time. The higher resource grade value resulted from inadequate drilling information since the FS and caused an overestimation of the resource grades. Within the Palabora block cave, the material flow remained unknown, since Lift 1 only had 'Time Domain Reflectometry' (TDR) cables for cave monitoring devices. When the head grade and the sampling grade separated, required investigations attempted to solve the reasons for such occurrences. PMC did not expect the grades to separate and the lack of drilling information contributed to the elusiveness of the block cave and its grade behaviour. Little knowledge existed on the change in PCBC grade or head-grade and especially the inverse of these two grades regarding high and low-grade occurrences. PMC resorted to the best current known facts at the time and attempted resolutions in the form of amendments within the PCBC flow parameters and resource cave-back specifications to equalise the modelled grade and head-grade.

The purpose of the study is to determine whether the copper resources at PMC's block cave is managed adequately, despite the elusive grade behaviour noticed over time. The project report will review past grade behaviours from previous reports and studies at Palabora Mine. This study will furthermore investigate possible trends in the model's grade behaviour and discuss the more recent hypothesis of a glacier slide at the slope failure.

The study aims to seek insight into the lower planned grade and unexpected higher sampled grades at the mature stage of the block cave mine nearing its end of life. The previous modelling of the resources, the block sizes, block spacing, kriging from drill samples, and grab sampling methods received in-depth audits and endured detailed studies.

However, the study will not aim to re-invent the sampling methodology or repeat the work already done on the resource model. Significant studies and information are available on these findings and justify such adjustments made over the cave's first lift (Lift 1) historical timeline since production started. Alternatively, the project report will review the historical trends and possible errors occurring, interpret previous reports and identify any significant constraints experienced. Apart from all the previously conducted studies, this project report will elaborate on additional conducted work during the expected life of mine ending phase, or rather the ending of the Lift 1 block cave.

This project report examines the significant historical grade differences between reporting ores at draw points, while the resource model estimates indicated critical anomalies. The project report summarises previous applicable studies conducted between 2003 and 2012, at the complicated copper block cave mine of PMC. The study will examine additional work done during 2015, where another grade anomaly occurred in a challenging economic setting.

This project report would merge old and new knowledge of PMC regarding the management of its resource grades and grade predictions. The PMC copper resource model endured several refined alterations from the elusive behaviour within the unknown block cave environment. This project report forms an ongoing study as more information becomes systematically available over time. The project report will ultimately be a forward-looking summary for the second lift's (Lift 2) block cave mine. It is likely that similar trends between Lift 1 and Lift 2 may occur and PMC may need to respond pro-actively to anomalies between the actual ore grades and the resource grade. This report summarises historical work, studies, events, and findings in a compact guide, especially available for PMC.

1.3 Research background

Palabora Mining Company (PMC) is located approximately 500 km from Johannesburg, in the Limpopo province. The mine is in operation since 1966, and most employees reside in the town of Phalaborwa and surrounding villages, Namakgale, Lulekani, Makushane and, Mashishimale. All of these villages form part of and fall within the broader Bha-Phalaborwa municipality. PMC historically mined 30 000 tonnes per day and produced 62 000 tonnes of copper metal per annum. Continuous improvements and modifications resulted in production rates to increase to 82 000 tonnes per day and 135 000 tonnes of copper product. The open pit operations ceased in April 2002, during which the development of the underground block cave mine was progressing (Snowden, 2010). In Figure 1 below, the mine borders the Kruger National Park (KNP), and hosts various wildlife animals, who can freely roam between the operational areas and the KNP.

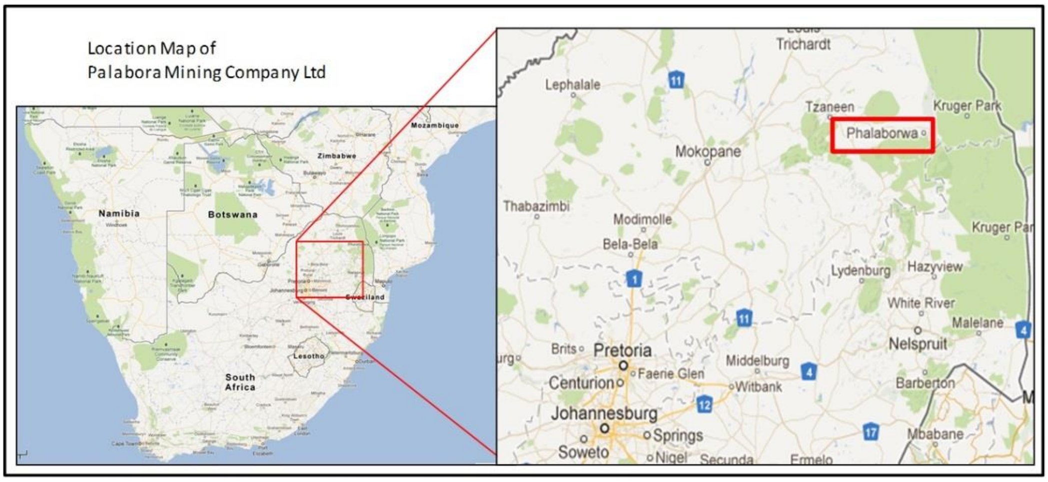


Figure 1: Phalaborwa locality plan
Source: (PMC, 2016)

1.3.1 Geological setting

Within the Phalaborwa area, an underlain Archaean granite occurs containing remnants of profoundly altered sedimentary, basic rock, and schistose rock of the “Basement Complex”. Twenty kilometres west of the copper deposit, a younger porphyritic granite intrusion occurs which is known as the Palabora Granite massif. The Palabora Alkali Complex is intrusive into the Archaean granite, and the alkali phase consists of numerous “plug like” intrusions of syenite. Occurring pyroxenite consists of pale green diopside and varying amounts of phlogopite, biotite, and apatite surrounded by a rim of felspathic pyroxenite, but remain separated from the enveloping Archaean granite via an irregular fenite zone. Two subsidiary pipes; serpentine-phlogopite-pegmatoid and the copper-bearing deposit known as the Loolekop orebody occur nearer to the centre of the major pipe (Kuschke & Tonking, 1971).

Historical records and studies indicate that an active alkaline volcano dating back 2047 million years formed the kidney-shaped Palabora igneous complex. The dimension of the complex is 6.5 km from north to south and 3.2 km from east to west (Snowden, 2010).

The elliptically shaped orebody is vertical, and elongated in an east-west direction, with physical dimensions of 650 m x 300 m. Three central fault zones are running through the ore body in E-W, NW-SE and N-S orientations. It is a composite intrusion with the age sequence from foskorite, banded carbonatite, and transgressive carbonatite, which all intruded in a concentric inter-banded structure. Karoo age dolerite dykes cut through the orebody in an SW-NE direction, illustrated in Figure 2 (Snowden, 2010).

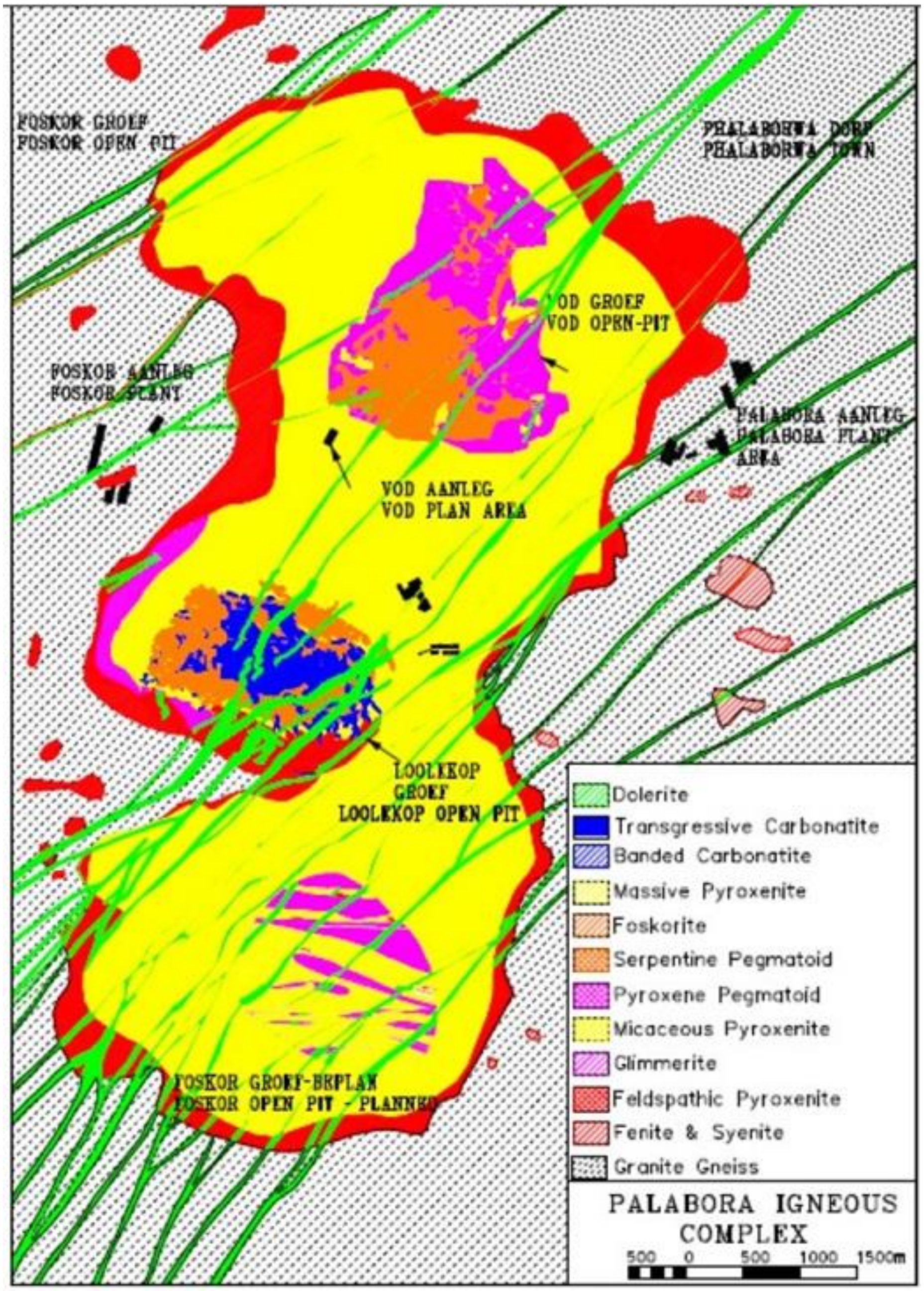


Figure 2: Palabora igneous complex
Source: (Snowden, 2010)

Within the core of the complex, a vertical composite intrusion known as the Loolekop pipe formed in a concentric arrangement of foskorite around it and with a core of banded carbonatite. The complex consists mainly of phlogopite and apatite rich pyroxenite (Snowden, 2010).

Historically PMC experienced a substantial lower head grade from the planned resource model grade (Howson, 2003). Howson (2003) listed several reasons for the grade shortfall and produced a required sampling campaign in 2003. One fundamental problem listed was too little drilling information for the Lift 1 block cave, resulting in a resource model with low confidence and errors. A more practical solution presented within Dassault Systemés GEOVIA software, which provided geological modelling, mining engineering, and survey application packages. This software formerly known as GEMCOM used to be an international geology platform in the mining industry. Howson (2004) proposed to have the PMC's geological data, resource and reserve models revised. Howson's proposal entailed the importing of existing models and data from Datamine into GEMCOM software (Howson, 2004a).

Apart from the initial adoption of the GEMCOM software by PMC, Dr Tony Diering contributed his programming skills to assist in writing software algorithms for PCBC. PCBC was a standard industrial block cave scheduling system, which motivated draw compliance and scheduling improvements at PMC. Howson (2004) also recommended an intensive sampling program to ensure the correlation between results and estimates maintain their current position. The PCBC results indicated similar overestimated resource model results, like the previous model's indication, even after the introduction of the new software. At the time, the current sampling methods identified by Howson (2004) could also improve the grade management going forward. Several studies conducted at PMC and abroad by many consultants reviewed the anomalies found over the LOM, of which some discussions follow in this project report.

The reconciliation of the resource block model changed from very low-grade values in failure material to actual sampling values from the open pit bottom benches. The analysis and revision of the failed material with actual bench samples resulted in a practical dilemma to get the data imported into GEMCOM (Howson, 2006). The revised results were not acceptable yet, and alternative approaches to the mixing parameters in PCBC required further analysis. This analysis and investigation produced acceptable errors but required additional refinement. However, the resource of the block cave remained a dilemma for PMC but became manageable with more data over time (Howson, 2006).

1.4 Problem statement

The Lift 1 block cave of PMC had very little drilling data available to ensure confidence in the resource model. The modelled resource grade and the head grade separated during 2015 while the cave consequently yielded a higher sampled ore grade, and a contrary expected lower resource model grade. PMC expected a lower grade from the resource model's indicative LOM, while cave mining was nearing the end of its life. This separation in grades required further analysis since the actual sampled grade reflected higher than the expected values (Howson, 2006).

The continuous revision of the resource grade model, conducted through continual additional information becoming available over time, should indicate the importance of understanding and correct interpretation of the PMC block cave's grade.

The required analysis and understanding of the unexpectedly higher head grade became significant in the currently depressed copper market. The situation influenced the progress of the Lift 2 project to deepen the PMC block cave mine. This unexpected higher head grade meant that a couple of years of life would add more production to the anticipated "2015" end of Lift 1 production. It remains uncertain on when the cave's grade will finally drop below the economic viability of mining the Lift 1 ore. Thus it was critical to demonstrate confidence in the improved head grade for financial reasons as this could decrease the “copper gap” period the mine expected. The “copper gap” is the time between a stoppage in copper production while the Lift 2 block cave remained in the development blasting phase without current possible copper production. The copper mining operations mined copper during the simultaneous development expansion of the future Lift 2 mine. The resource model indicates the projected LOM and when the drop in copper grades beyond the economic pay limit would indicate a stop in production. Hence, the reviewing of the resource model remained critical to PMC’s planned production, and the sustainability of the mine’s vision to produce copper beyond 2030.

Accurate information would play a significant role in the mining of Lift 2 since the refinement of the Lift 1’s resource model would ultimately affect Lift 2 situated directly below Lift 1.

The study will seek to prove whether the Lift 1 resource model of PMC’s copper block cave followed an adequate management and control strategy to counter the variation between the head grade and resource grade over its life.

1.5 Research objectives

The annual revision of the resource model and each previous study refined the resource model to a certain degree only. An optimal developed resource model or final resource model was not practically achievable, but lessons learned from this study will be beneficial to the sustainability of the mine beyond 2030. The study aims to build shareholder confidence from work conducted by the technical teams.

The project report focusses on the anomalies and historical variations in grade at PMC’s copper block cave operation. The report considers previous investigations into grade variations, appropriate adjustments and recommendations concerning the resource model and resource grade.

The study will analyse and investigate any significant PCBC parameter adjustments required to correlate the yielding sampled ore grade versus the lower predicted resource model grade PMC experienced. These parameters include the average annual volume of surface ingress material into the Lift 1 block cave. This study will investigate essential omitted factors which previously compromised the "fully comprehensive" Howson (2006) resource model, and identify future improvements from the findings obtained.

The study will investigate the following questions:

- What were the leading historical indicators for the grade shortfall at PMC's block cave mine?
- Which constraints did the software and dolerite modelling impose on resource model at PMC's block cave mine?
- What was the correlating grade variations and contributing factors during the LOM?
- What influenced the head grade to sustain beyond the predicted LOM of Lift 1?

1.6 Summary of section 1 and structure of the report

By mining the PMC block cave to the optimal grade through strict planning and an efficient draw control strategy, produced copper head grades entirely different from what the mine expected over time. PMC annually reviews the resource grade model producing an optimal plan for draw point copper grades to match the annual budget plan. A particular review was required in 2015 to optimise the draw compliance for the expected last months of the first Lift's life, nearing its end of production capacity. PMC consistently consulted with Dr Tony Diering, having expert knowledge on the PMC block cave and PCBC software. This project report will cover some of the analysis and practical application of Diering's 2015 findings and PMC visit.

Structure of the Project Report

This section has justified the research work and defined the problem statement. Section 1 provides a background of the report and includes the objectives of the report.

Section 2 provides literature for block caving mining terminology and principles and reviews papers written for Palabora's block cave mine.

Section 3 discusses the physical and geological tenure of Palabora Mine.

Section 4 discusses the timeline of the conducted studies for Palabora Mine's resources between 2002 and 2003.

Section 5 describes the data retrieval process from old computer systems and the transferral of this data into more recent and reliable systems.

Section 6 discusses the grade reconciliation study conducted between 2004 and 2005.

Section 7 focuses on the required re-estimation of the copper grades during 2005.

Section 8 reviews the 2010 PMC head grade audit as was conducted by the SRK consultancy group.

Section 9 discusses the grade diversion and resource model in 2015 and describes different approaches taken on the north wall failure. The sustaining head-grade indicated higher-grade values than what the resource model reflected.

2 LITERATURE REVIEW

2.1 Introduction

The literature review provides an overview of relevant block caving aspects, which are relevant to the PMC copper block cave having a slope failure in the pit overlying the caving operation. This section will discuss block cave mining principles and will give a good general indication of terminology used in block caving, enabling the reader to understand the block caving complexity from a novice perspective. This section also includes a review of other papers written especially for PMC and applicable mines despite PMC's uniqueness.

2.2 Block caving principles

This section of block caving principles is aimed at readers not familiar with block caving, block caving terminology, and elaborates in more detail on the topic.

The extraction level refers to the production level, where ore from draw points is withdrawn and feeds from the undercut level vertically above it. The crinkle cut design was a world first in undercutting and implemented at Palabora's Lift 1 block cave (Moss et al., 2006). In Figure 3 below, the illustration indicates the crinkle cut design for the undercut level at Palabora Mine from a side view perspective.

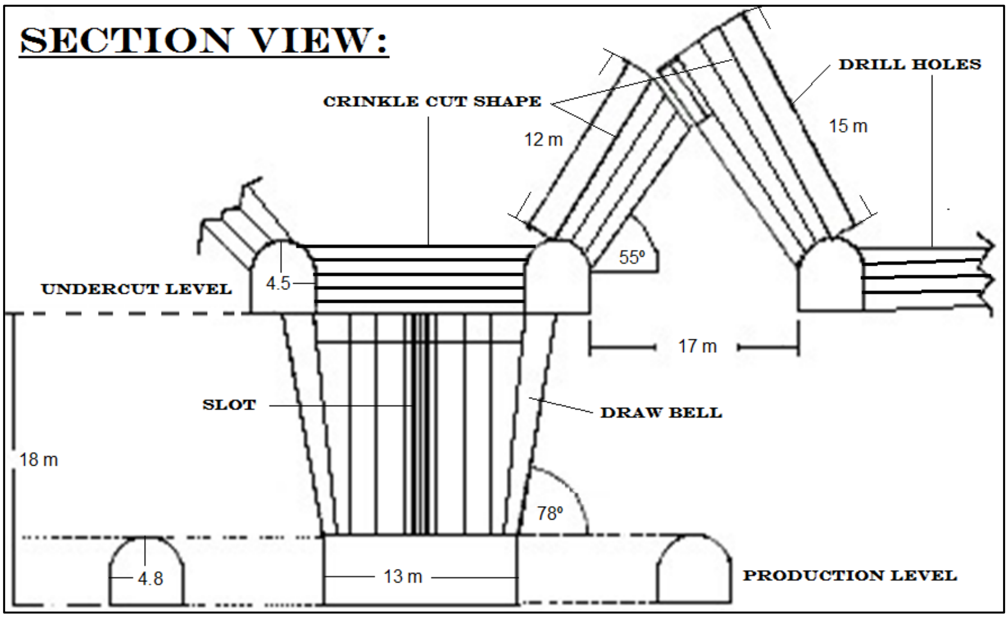


Figure 3: Palabora Mine undercut design for Lift 1
Source: after (Calder et al., 2000)

In Figure 4, fourteen collage diagrams indicate block-caving development stages and portray details regarding the extraction level, undercut level, draw bells, cave propagation, and the crown pillar.

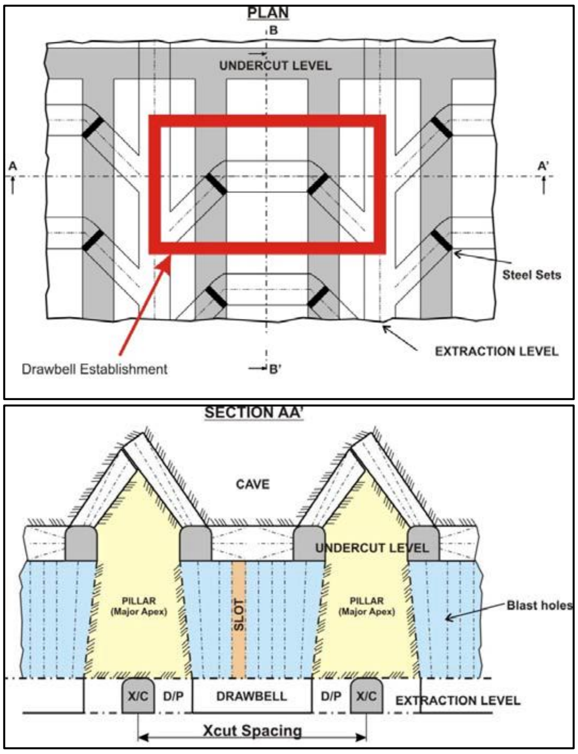
The collage diagram 1 (Figure 4), shows a drill rig drilling the development face at the extraction level. Collage diagram 2 indicates how development crosscuts are penetrating the orebody from the one side of the vertical ore body, to the other side, ultimately linked by perimeter tunnels. Approximately fifteen to eighteen metres above the extraction level as per diagram 3, the undercut level development comprises of two adjacent crosscuts for each extraction level crosscut. In collage diagram 4, 5 and 6, the undercut is ring blasted to form the continuous collapsing zone. The undercut corresponds to the caving initiation through the continuous creation of a broken rock zone at the base of the block cave. The undercut generates instability through gravity and stress allowances, acting in the rock mass, which sustains the next stage of cave propagation (Fernandez et al., 2010). Continuous mucking of the swell formed by the collapsed material is part of the undercutting process. The undercut is staggered to create panels in a sawtooth shape. This shape of the undercut is a precise scheduled blasting process and used to manage underground stress levels via the leading and lagging panels.

In the collage diagram, 7 – 10, the draw-bell drilling in the extraction level, the charging of holes with explosives and blasting thereof forms a bell-shaped cavity. The draw bell bleeds rock from the undercut level into the draw points of the extraction level through gravity, where ore mucking by load and haul machines muck material as per diagram 11 and 12. Diagram 13 indicates the “footprint” of the establishing extraction level with various draw bells and draw points following behind the development of the tunnels, while the cave propagates upwards from undercutting. During the undercutting and development phases, continuous mucking and hauling of the swell, known as production to the underground crushers takes place. The crushed size ore, tips onto conveyor belts to reach the surface. The conveyor belts at some mines tip ore in bulk silos near the shafts, where hoisting via skips in the shafts takes the material to the surface (Studio 522 Productions, Inc., 2016).

Figure 4: Block caving stages and principles
Source: (Studio 522 Productions, Inc., 2016)

The block cave-optimisation paper by Steward et al. (2010) discussed the optimal position of the footprint in a block, the draw-bell establishment, and caving parameters (Steward et al., 2010). Some of these parameters include production rate, cut-off grades, and the extraction level position.

The extraction level layouts found commonly in block-cave mines are the El Teniente (straight through), the Henderson (Z-shaped), or Herringbone shapes. The illustrations in Figure 5 show the latter Herringbone shaped extraction level design (Chitombo, 2010). This illustration indicates a plan view of the extraction level, draw point spacing with a superimposed undercut level and a typical draw bell establishment, with applicable sections A-A' and B-B' through it. PMC implemented both the Herringbone extraction level shape and the crinkle-cut shape for undercutting within Lift 1, as per the illustration's section A-A' below.



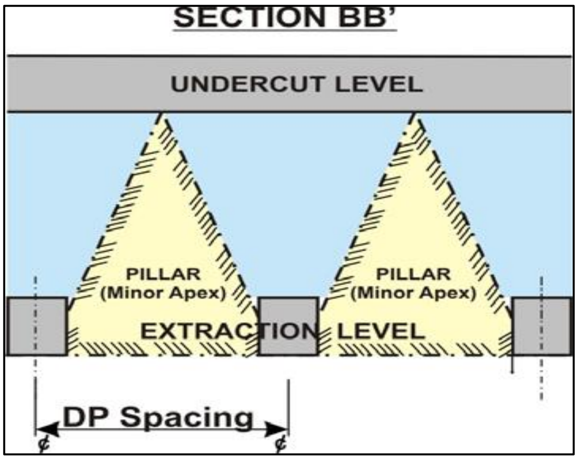


Figure 5: Drawbell establishment
Source: (Steward et al., 2010)

Figure 6 indicates the Ridgeway Deeps project with its advanced undercutting strategy layout, having a spaced draw bell dimension of 30 m x 18 m, and 18 m between the extraction and undercut level (Dunstan & Popa, 2012). The PMC Lift 1 block cave has similar dimensions, and the same cave layout, which Ridgeway Deeps project adopted after PMC.

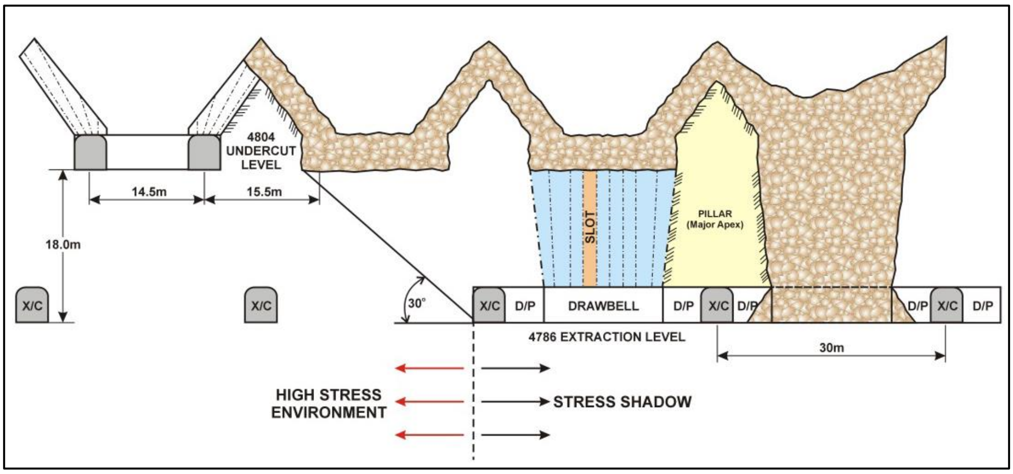


Figure 6: Advanced undercut establishment section
Source: (Dunstan & Popa, 2012)

The height of interaction zone (HIZ) of each draw bell indicated in Figure 7 represents the shape of the individual draw zones and incremental sections of ore recovery above it.

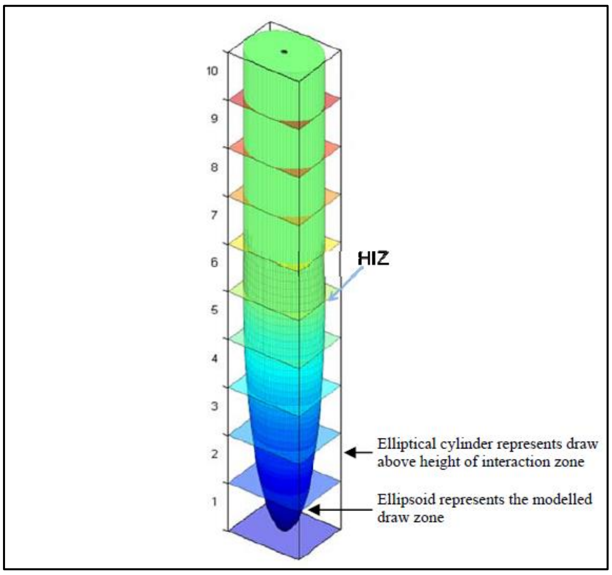


Figure 7: Representation of the cave draw ellipsoid
Source: (Steward et al., 2010)

2.3 Conceptual and numerical modelling, with empirical predictions

Prediction models are assisting tools to analyse different mining situations and to optimise the mining process, and these models produce such predictable results (Chitombo, 2010). Models have their advantages and disadvantages, which require a thorough assessment before utilisation during the analysis of real mine conditions (Gustafsson, 1998).

According to Gustafsson (1998), the most comprehensive approach to predictive modelling divides prediction models into five model-types, namely:

1. Empirical methods and models,
2. Void diffusion approach models,
3. Kinematic models,
4. Cellular automata,
5. Numerical models.

The methodology to simulate the caving process and modelled input material properties are more important than the numerical program itself. The International Caving Study (ICS) research program started in 1997 to improve the understanding of cavability, fragmentation, and gravity flow, draw control, and cave mine designs (Cundall, 2008). During this international study, the Itasca consulting group developed their numerical model PFC^{3D} (Particle flow code in 3D). Meanwhile, physical modelling studies executed by Halim (2004) and Castro succeeded with the development of a new code namely REBOP (Rapid Emulator Based on PFC^{3D}) (Cundall, 2008). REBOP simulated the drawing of fragmented rock from panel, sublevel and block caves. Figure 8 shows the PFC^{3D} sublevel caving ring and REBOP model for block caving, by simulating the predicted cave drawdown, ore recovery, and dilution (Cundall, 2008).

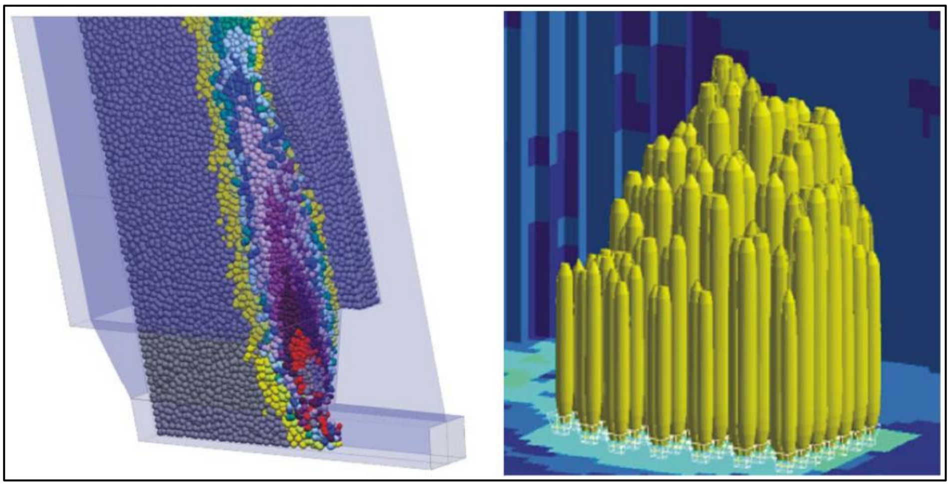


Figure 8: PFC^{3D} sublevel caving and the REBOP block caving simulations
Source: (Cundall, 2008)

The developed algorithm by Itasca Consulting Group, Inc. simulated the caving process within the macro language FISH, provided with the FLAC^{3D} (Fast Lagrangian Analysis of Continua – 3D) and 3DEC (Three-dimensional numerical modelling code) programs. These models are hence applicable to PMC and relevant to this research study. PMC applied this caving algorithm to its cave growth back-analysis, during swell pulling from the lower draw points on the production level (Board & Pierce, 2009).

A self-sustained cave develops during continuous withdrawal of broken ore, ultimately known as the stage where the undercut reached its critical dimension and forms the hydraulic radius. Duplancic and Brady (1999) described a conceptual model of the developing cave, having four central behavioural regions, as Figure 9 indicates.

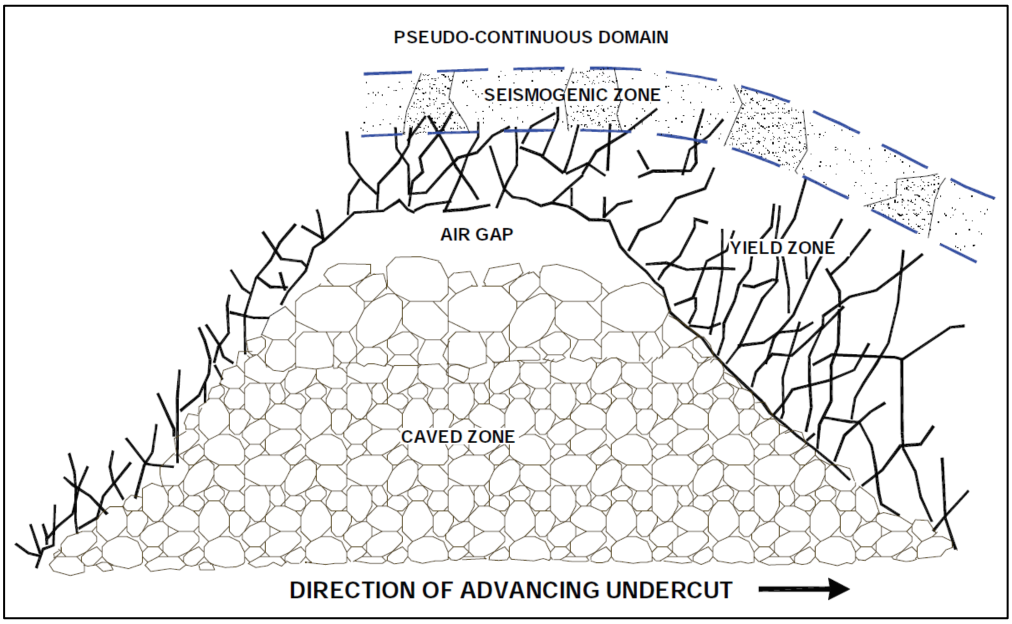


Figure 9: Conceptual model of caving
Source: **(Duplancic & Brady, 1999)**

The pseudo-continuous domain defines the host rock mass around the caving region which behaves elastically. Duplancic & Brady (1999) reported that the rock mass properties and behaviour are likely to be undisturbed. The seismogenic zone is a micro-seismic discontinuity zone, having newly initiated fractures. The yielded zone is a fractured zone where some or all-cohesive rock strengths lose all its support relative to the overlying rock mass. The caved zone or mobilised zone comprises of detached rock blocks from the rock mass. The detached rock blocks move towards the draw points due to drawing or mucking of ore (Duplancic & Brady, 1999). Figure 10 illustrates the conceptual cave model and numerical model of caving with specific zones applicable to PMC's block caving process.

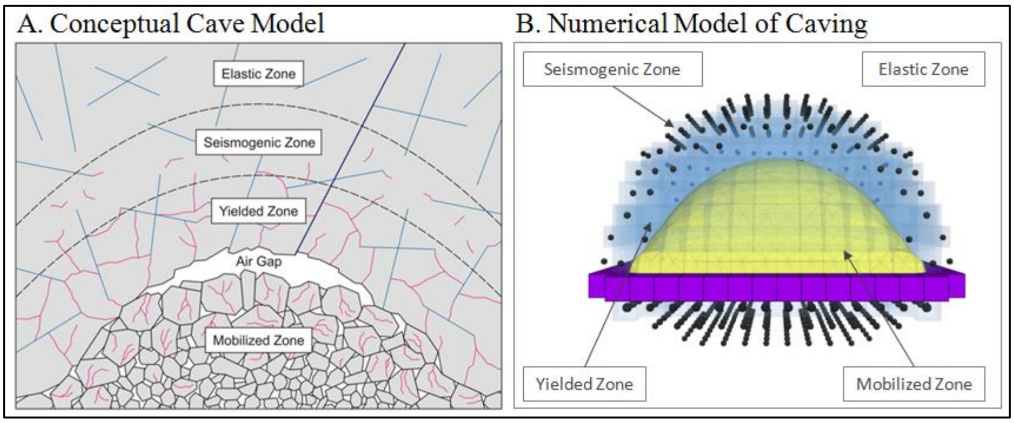


Figure 10: Main behavioural regions of a propagating cave
Source: (Sainsbury, 2010)

Board & Pierce (2009) described an alternative more rigorous approach to the material defining properties and constitutive responses of the rock masses, as per synthetic rock mass (SRM) method (Board & Pierce, 2009). The SRM methodology uses a particle flow code (PFC^{3D}) where a discrete fracture network (DFN), embeds the PFC^{3D} with intact rock matrixes and resultantly represents the outcome. These flow codes form an integral part of the overall programming in block cave software applications used by PMC.

Simulated testing obtains synthetic materials' properties such as strength anisotropy and brittleness, where practical techniques currently cannot derive these properties (Sainsbury et al., 2008). Cundall (2008) indicated the particular interest to have the ability to acquire rock-mass scale effects and then be able to predict such effects. Figure 11 indicates the dominant vertical oriented joint set within the rock mass which is observable if samples occur in various orthogonal directions during uploading (Cundall, 2008). The conceptual model and numerical models apply to PMC, and the Cundal (2008) carbonatite stress and strain response tests apply to Palabora Mine's lithology. The CP reports indicate that the application of Rebop and PCBC determined the effect of waste on the production plan (PMC, 2016).

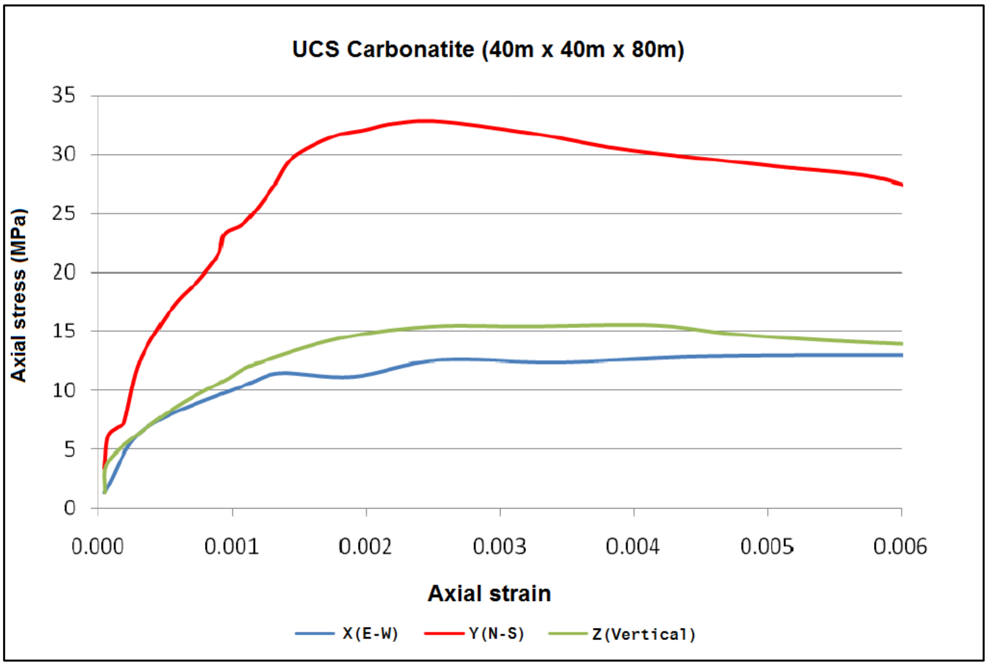


Figure 11: Palabora carbonatite stress-strain response tests
Source: (Cundall, 2008)

However, Sainsbury (2012) found a correlation between the FLAC^{3D} numerical model and the physical evidence of the magnitude regarding the north failure of the Palabora Mine site. The numerical modelling studies of caving are critical to PMC's block caving, where modelling inputs and outputs resemble the best fit to the real cave behaviour.

2.4 Material flow and the modelling thereof

Extensive studies over the past five decades exist for ore flows of sublevel caving. Studies and simulations of material flow models, calibrated against mill production and mine values or optimisation through applied physical and numerical modelling techniques improved over time (Shekhar et al., 2016). The increase of ore recovery is dependent on the monitoring of the grade at draw points which are crucial in controlling dilution. A constant monitoring system is required to monitor effective draw control that loads criteria to reduce dilution and improve recovery (Shekhar et al., 2016). Summarising the past research on material flow and draw control indicates factors such as production planning and production targets, cut-off and shut-off grades, dilution entry and total dilution. It also includes mine design and ring design, mine layouts, ore geology and geometry, the performance of proximate draw points, and the nature of material flow (Shekhar et al., 2016). This research knowledge contributes to the material flow and drawing compliance at PMC.

Diering (2000) described details of the PCBC program and those components of the modelled process of block caving. The origin of the software development happened in 1998 at Premier Diamond Mine in South Africa. Diering (2010), Robertson, and Kirsten worked in close cooperation with Laubscher (2000) during the development of the software (Diering, 2010). PCBC does not model material in a geotechnical or geomechanical way but applies empirical rules to mix material, improving grade predictions, and enhancing effective long and short range planning and design strategies. Components of the process include a sufficient description of the geometric and grade characteristics of the geological environment. Constructed PCBC block models comprise of grades, density, rock codes, and percentage of fines, fracture frequency rating, and draw point locations (Diering, 2010).

A slice file represents the draw column above the draw point before any initial extraction commences. This slice file is calculated by the software from a draw cone's perimeters while it constructs a draw point "reserve" (Diering, 2010). Each column can comprise of a number of slices, and the vertical dimension of each slice corresponds to the block model. Different assigned categorical components identify the slices such as shared or unique, coarse or fine and ore or dilution (Diering, 2010). The different components of the slice file shown in Figure 12 define the computed tons for each slice.

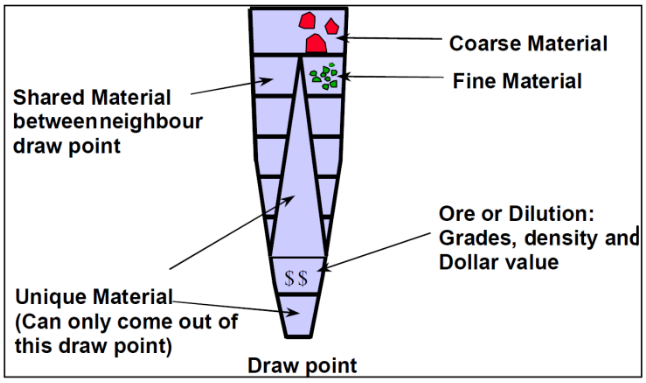


Figure 12: Different components of the slice file
Source: (Diering, 2010)

Diering (2007) described in detail the various flow mechanisms such as vertical mixing, horizontal mixing, toppling, rilling, erosion, compaction, major surface movements and inclined flow along a contact. The details of these flow mechanisms are beyond the scope of this study. Various flow mechanisms shown in Figure 13, indicates the terminology used and occurrence of mixing variables found in block caving.

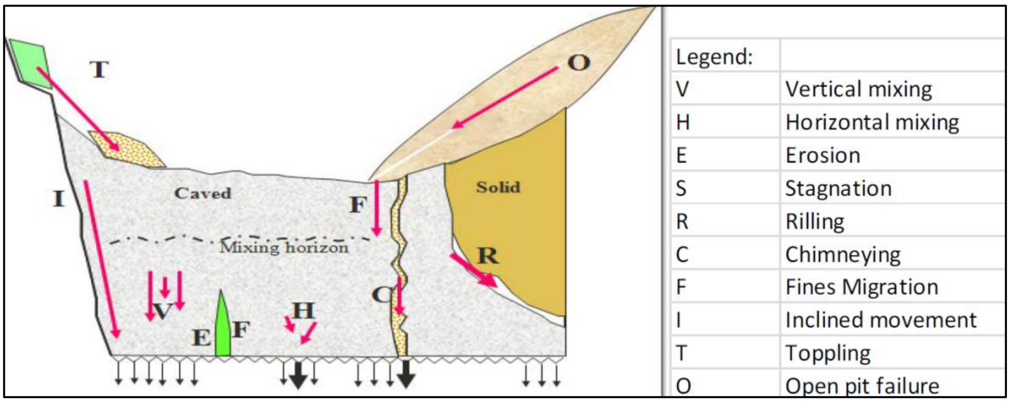


Figure 13: Typical flow mechanisms in a block cave
Source: (Diering, 2007)

At PMC, the various parameters captured in PCBC, include the external source of waste dilution, which formulates the draw control plan. The PCBC software accounts these factors to establish the required draw rates and draw zones for each production shift. The PCBC tool is an integral part of the resource management at PMC and applies to this project study.

2.5 The impact of poor fragmentation on cave management

Palabora experienced poor fragmentation on the western side of the cave during the ramp-up stage. This western side indicated the highest draw columns with its youngest draw points, having a much coarser fragmentation than the rest of the block cave (Ngidi & Pretorius, 2010). The need to review the long-term production scheduling arose with effective production rates but required acceptable limits. The comparison between the FS predicted fragmentation data and actual data highlighted the required adjustment of the production plans, and operational processes accordingly. In the first three years of production, eighty percent of the total Lift 1 production, resulted from the mining of oversized material. Secondary blasting treats hang-up material and oversized material occurring within the draw points (Ngidi & Pretorius, 2010).

In Figure 14 below, Atkins (2013) compiled an oversize exercise to determine the average size of the material. The oversized material analysis was broken down into the rock lithologies, which had a direct correlation with the copper grade in the PMC block cave (Atkins, 2013).

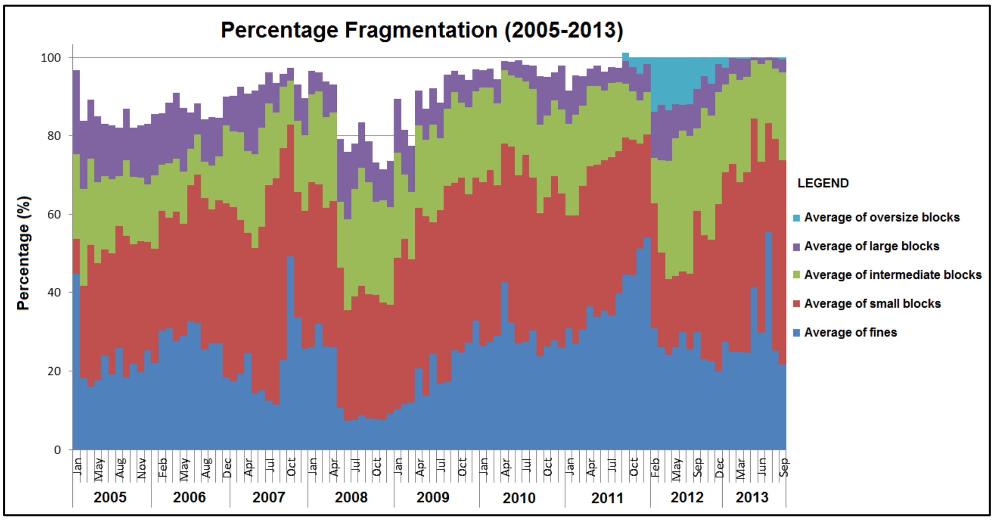


Figure 14: PMC Lift 1 fragmentation composition
Source: (Atkins, 2013)

In Figure 15, the lithology classification of sampled material in the production level plots according to their representative percentages.

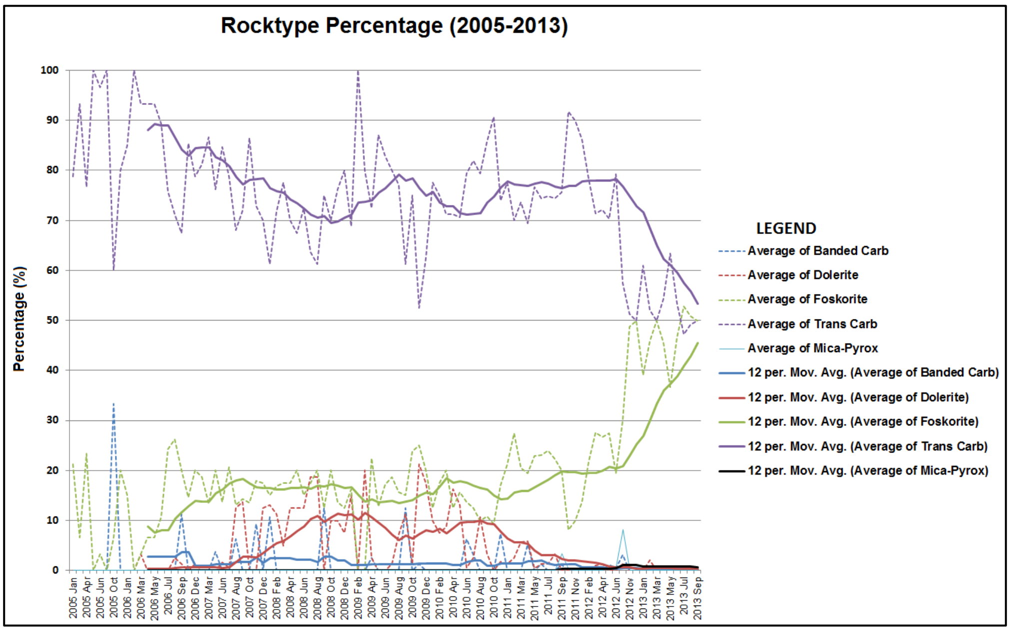


Figure 15: PMC Lift 1 percentage rock types
Source: (Atkins, 2013)

When Atkins (2013) plotted the representative fragmentation data in the cave, the geographical display in Figure 16 below indicates the fragmented distribution within the Lift 1 footprint.

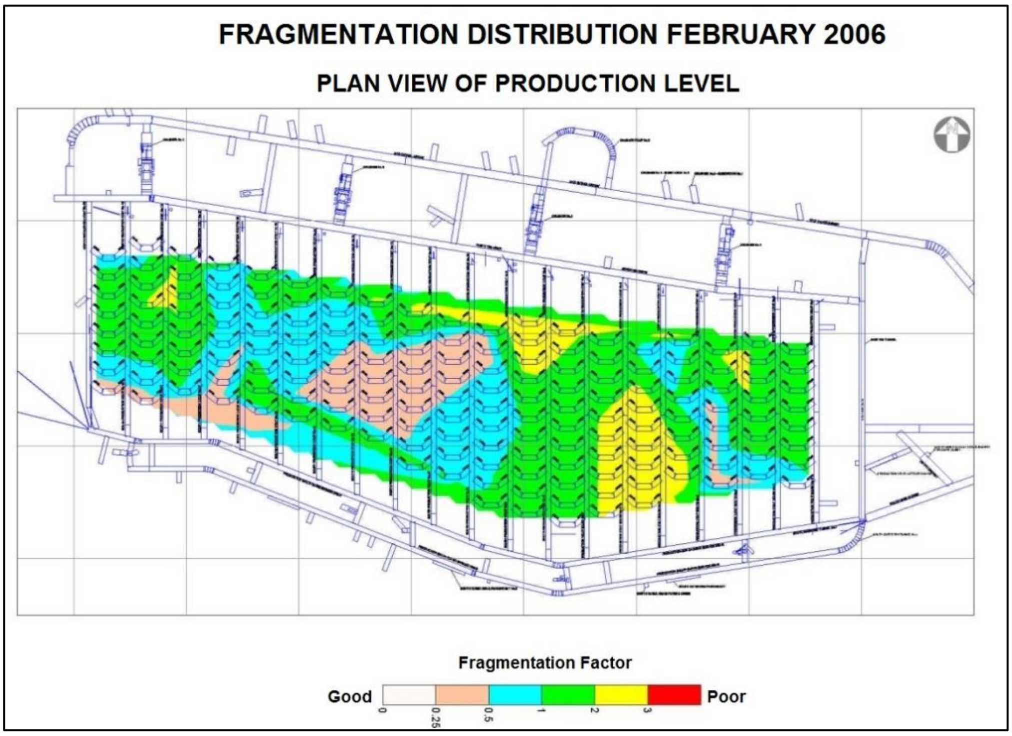


Figure 16: PMC Lift 1 fragmentation analysis 2006
Source: (Atkins, 2013)

Massive fragmentation can cause draw points to hang up, and large boulders consequently influence the draw control and flow of material. The loading and pulling of ore in the draw points affect ore recovery, hence draw control optimisation maximises the ore recovery (Shekhar et al., 2016).

These studies contributed to the draw control strategy of PMC, which aimed to delay the dilution entry and opted for draw optimisation. The optimised draw stretched from the poorly fragmented western section of the production level (Shekhar et al., 2016).

2.6 Early physical model studies and computer software

Extensively used studies on gravity flow mechanisms use physical models to determine draw control strategies. Draw rates affect the intermixing of ore and waste, which requires drawing control strategies to optimise the operation (De La Comisión et al., 2012). This knowledge supplements the PCBC software mixing variables and draw rates of PMC, with specific reference to the Diering (2015) ore and waste mixing parameter review.

The ellipsoid theory by Kvapil (1992) and renowned studies conducted via physical models, sand models, and draw spacing studies by Heslop (1983), led to Laubscher's research (Halim, 2004). Halim (2004) recorded that the REBOP (Rapid Emulator Based on PFC^{3D}) model by Carlson et al. (2004) resulted in response to the associated difficulties from run times in particle flow codes. Carlson et al. (2004) modelled REBOP in the form of encoded algorithms. The significance of this research contributed towards a better understanding of the block cave mine at PMC. The Carlson et al. (2004) research form a part of high-level background programming in the existing software used at PMC (Halim, 2004).

Brummer et al. (2006) reviewed the adoption of the 3DEC numerical models to predicted the impact of underground mining on an open pit slope at PMC in 2006 (Xu et al., 2006). The research by Brummer et al.

(2006) covers extensive modelling for slope failures in various ways. The simulations and calculations cover aspects such as slope failure predictions in using the numerical model. The Brummer et al. (2006) paper additionally cover slicing, mining, filling simulations, landslide simulations and prediction of landslides. These simulations are primarily utilised when mines transition from open pit mining to underground mining (Xu et al., 2006). This knowledge applies to the later approaches PMC followed since the slope failure review and during the revision of the geological resource model.

2.7 Palabora Mine, a unique block cave mine

There is an abundance of literature and studies for block cave mines which deals extensively with the segregation of grades and the expected behaviour of fragmented rock and flow patterns.

Du Plessis & Martin (1991) assessed the deformation behaviour of the pit wall and used the explicit finite difference code “FLAC” during the modelling of the PMC open pit. Their model’s results had difficulty in making any final predictions (Ahmed, 2009). Their research work could not predict that a slope failure was to occur at Palabora, but notably, this work emphasises the uniqueness of each mine (Ahmed, 2009).

Laubscher (1994) focussed in on block caving methods, and he wrote a revised paper seven years later. The Laubscher (1994) paper became the most widely used reference for cave mine designs (Chitombo, 2010). Laubscher (2000) elaborated extensively in the international caving study manual on block caving, but most mines are unique and treated individually for having different caving probabilities. Due to the uniqueness, a component of elusive behaviour in every cave exists which require caves to be continuously monitored and measured to form reasonable conclusions (Snowden, 2010). Apart from all the written reports and conducted studies, the PMC block cave and resource grade has predominantly been a conundrum for experts.

2.8 Crown pillar failure of the PMC open pit in 2004

In April 2004, the cave broke through into the open pit, without evidence of any significant air gap forming above the caved rock mass (Glazer & Hepworth, 2006). Figure 17 indicates the crown pillar between the remainder of the propagating cave and the bottom of the open pit.

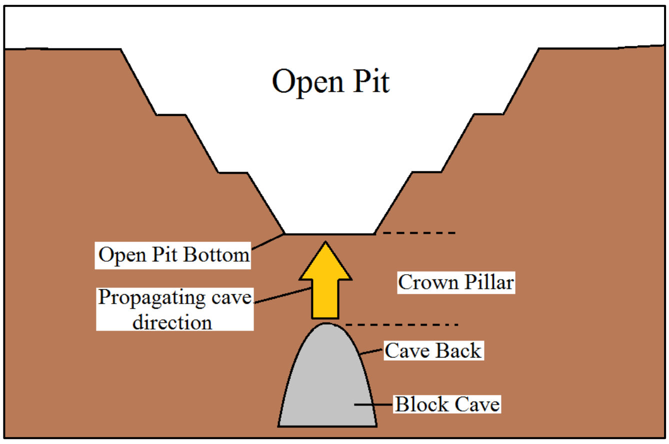


Figure 17: The crown pillar between the open pit and cave

Existing research concludes that the deformation zones such as the macro or microzones, and the direction of the break back at slope failures occur as a structural break back from the glory hole (Butcher & Jenkins, 2006). This phenomenon applies to PMC where the pit failure extends in the northwest of the copper open pit due to weaker zones.

Sainsbury (2012) reported that the mobilised zone connected from the cave into the open pit floor during the first quarter of 2004. The study indicated that mobilisation within the numerical model occurs initially along pre-existing fault traces, as Palabora experienced from the northern slope failure (Sainsbury, 2012). Figure 18 indicates the numerical simulation of the cave breakthrough at the PMC open pit.

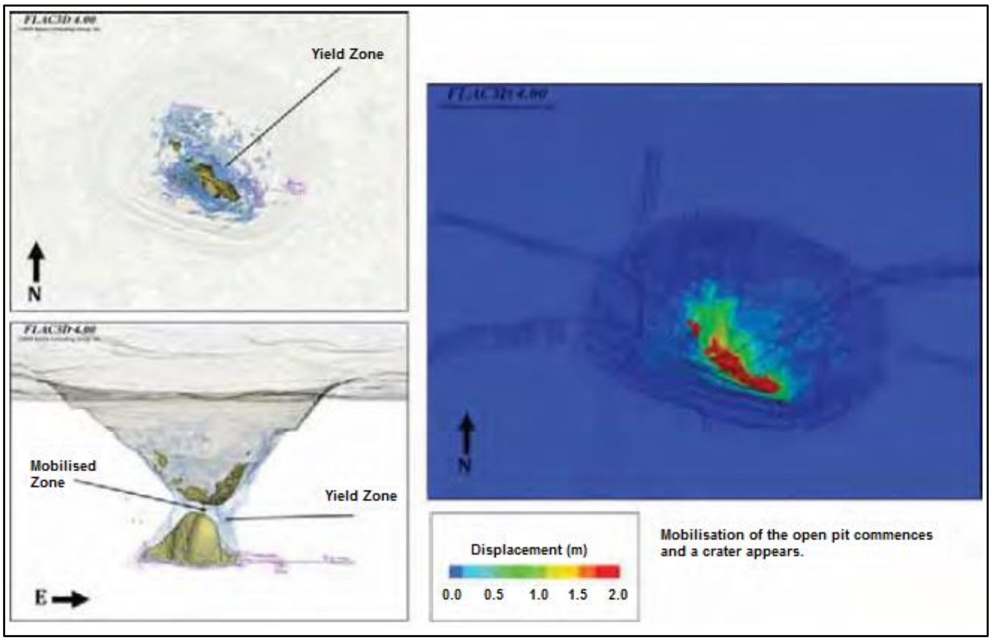


Figure 18: A numerical simulation of the cave breakthrough at PMC
Source: (Sainsbury, 2012)

2.9 The influence of major fault zones at Palabora Mine

In 2003, shortly after the crown pillar became de-stressed a series of complex slope movements occurred (Severin et al., 2010). The northwest wall failed gradually, and its slow development continued over a period of 18 months. By 2008, the central block of the failed mass appeared massively deteriorated (Severin et al., 2010). Five vital structures exist within the Palabora orebody: CFZ (Central Fault Zone), SWFZ (Southwest Fault Zone), Tree Fault Zone, Mica Fault Zone, and the so-called NNWSO (North-northwest Structural Orientation). This NNWSO comprises of a group of structures with its orientation confined in the open pit (Sainsbury et al., 2016).

Figure 19 below indicates the spatial orientation of the Mica fault, Central fault and other significant faults at Palabora Mine in a 3D geological model.

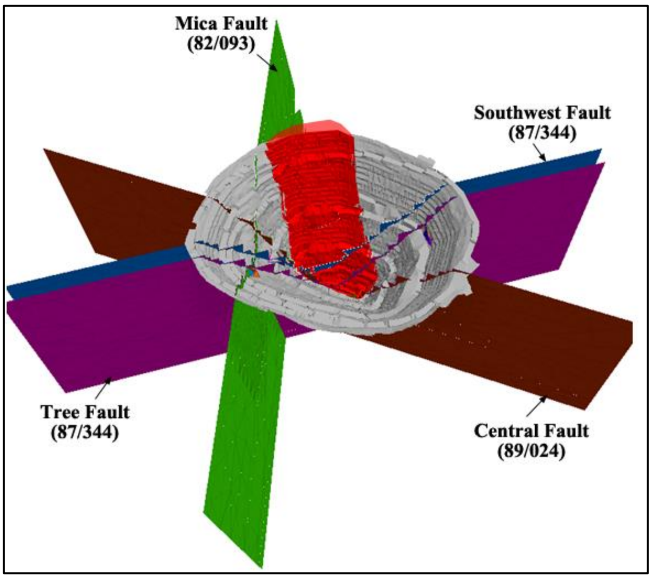


Figure 19: Major fault zones at PMC
Source: (Severin et al., 2010)

Severin & Eberhardt (2012) indicated that horizontal stresses could rotate and concentrate around the pit bottom and toe of the slope, which led to higher stresses. The possibility of a concentrating effect of major geological features existed especially around faults on shear stresses within the northern pit wall (Severin & Eberhardt, 2012). The anticipated cave back position in Figure 20 illustrates the ore reserve dilution-envelope where the shoulders stabilised and remained intact. The assumed intact shoulders uphold due to the continuous influx of surface material (Diering, 2015).

The 3D illustration of the Mica fault in Figure 20 at Palabora Mine indicates the spatial relationship between the underground operations and the open pit.

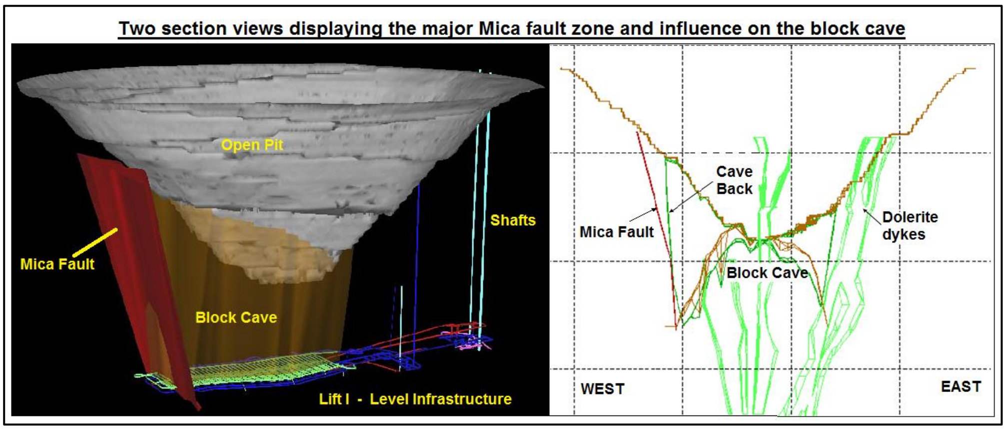


Figure 20: Mica fault in 3D
Source: after (Severin et al., 2010)

The fault zones transecting the open pit had a significant influence on the failure. Severin et al. (2010) reasoned that the Mica fault restricted the western wall to induced strains when the breakthrough occurred. However, the cave back migrates along the dominant joint set in the northern wall and appears not to migrate up and along one of the major faults (Severin et al., 2010). This knowledge base importantly indicates the possible structures through the resource body. Locked up ore at the cave back will consequently incur resource losses, as the shoulders still did not cave as was anticipated. These shoulder losses were not included in the FS since the failure contributed to the shoulders forming in the cave back.

2.10 Back analysis of the Palabora Mine’s caving behaviour in 2008

Sainsbury et al. (2008) reported that a conducted back-analysis of the caving behaviour of Palabora Mine verified the SRM-UJRM (Synthetic Rock Mass - Ubiquitous Joint Rock Mass) technique, which represented Palabora’s jointed rock mass. Sainsbury et al. (2008) noted from recorded indications that the transition of the surface to underground block cave mine happened in 2000. On the other hand, the open pit design’s overall slope angle increased from 37º to 58º in the more competent rock at the pit bottom. Hence, some numerical studies on the open pit design presumed instability potentials, while the open pit’s steep slopes remained intact throughout decommissioning of the pit operations (Ahmed et al., 2014). Sainsbury et al. (2008) indicated that the open pit failure led to potential sterilisation and dilution of the ore reserve. Figure 21 below indicates the reproduced pit slope-failure mechanism in the numerical model, which compared well with the onsite observations (Sainsbury et al., 2008).

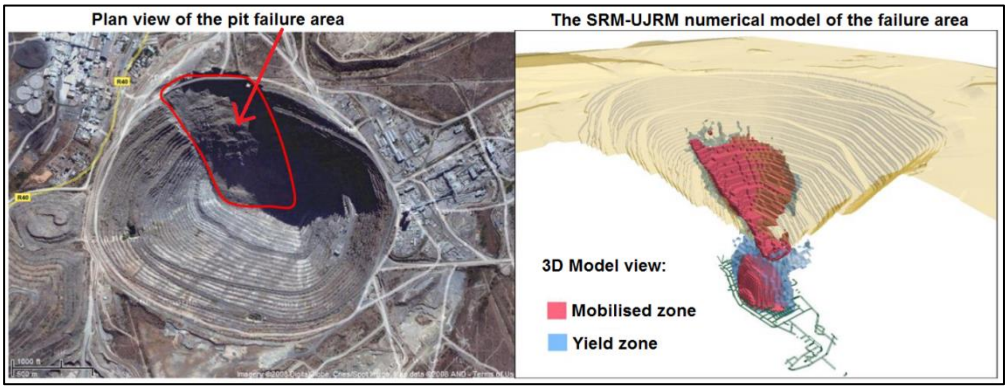


Figure 21: The Palabora pit failure mechanism reproduced by the SRM-UJRM
Source: (Sainsbury et al., 2008)

Sainsbury et al. (2008) indicated that production increased from the western section of the mine, causing a rapid vertical advance of the yield zone. Sainsbury et al. (2008) also found that the breakthrough of the cave volume mobilised sub-vertical joints in the pit slope above the yield zone. Figure 22 indicates the development of the pit slope-failure mechanism over time.

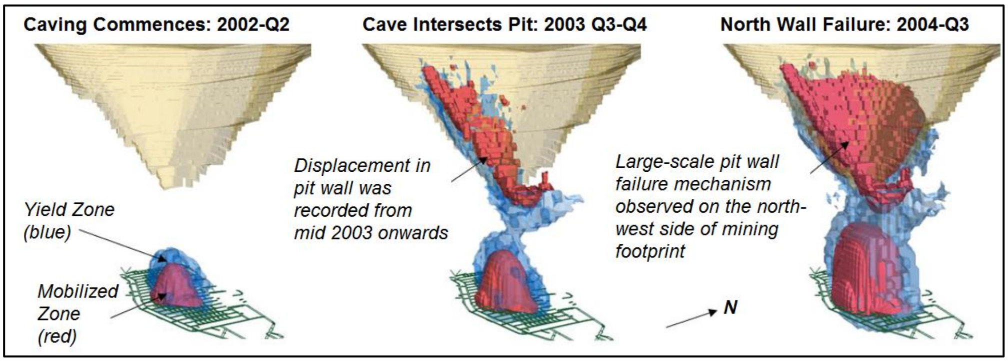


Figure 22: Development of the pit slope-failure mechanism over production time
Source: (Sainsbury et al., 2008)

The paper by Brummer et al. (2006) described the numerical 3DEC models developed by Itasca Consulting Canada Incorporated for the Palabora open pit walls and floor in 2006. The study investigated mechanisms of slope deformation, failure of the pit walls, and sought clarity on the future long-term pit wall instability. The 3DEC models indicated the possibility of the north wall failure and that its stability depended on the actual direct caving control (Brummer et al., 2006). Sainsbury (2012) reported that used criteria for assessing the cave behavioural regions from a caving and subsidence outline are valid at the Palabora Mine site. Figure 23 illustrates the production drawn from a simulated model and respectively the drawn height progression.

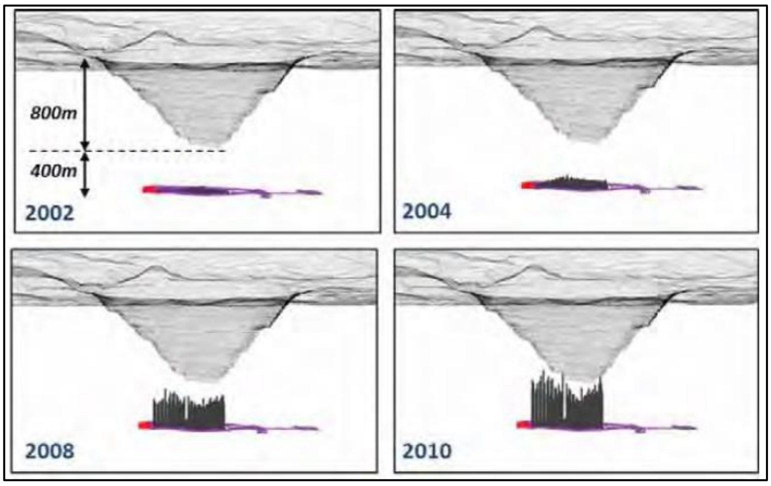


Figure 23: Historical mining record at the Palabora block cave mine
Source: (Sainsbury, 2012)

The accuracies of these studies play an important role when the input parameters of the influx of surface material into the block cave affect the overall resource model of PMC.

2.11 Micro-seismic analysis between the block cave and the pit slopes

Studies by Glazer & Hepworth (2006) indicated that the micro-seismic activity followed the approximate undercut progression pattern and became evident in the back-analysis of the Palabora Mine. This study indicated that the dominant fracturing mechanism induced in the cave volume resulted from shearing (Glazer & Hepworth, 2006). The loss of cohesion below the north wall triggered the final failure of the open pit volume, resulting in high damage indicated by the clustered index. The clustered index is a damage parameter, which combines source locations and useful event sizes that characterise the degree of clustering and interaction between fractures acting as sources of micro-seismic events (Reyes-Montes et al., 2010). Figure 24 shows an isometric perspective of micro-seismic events recorded at PMC.

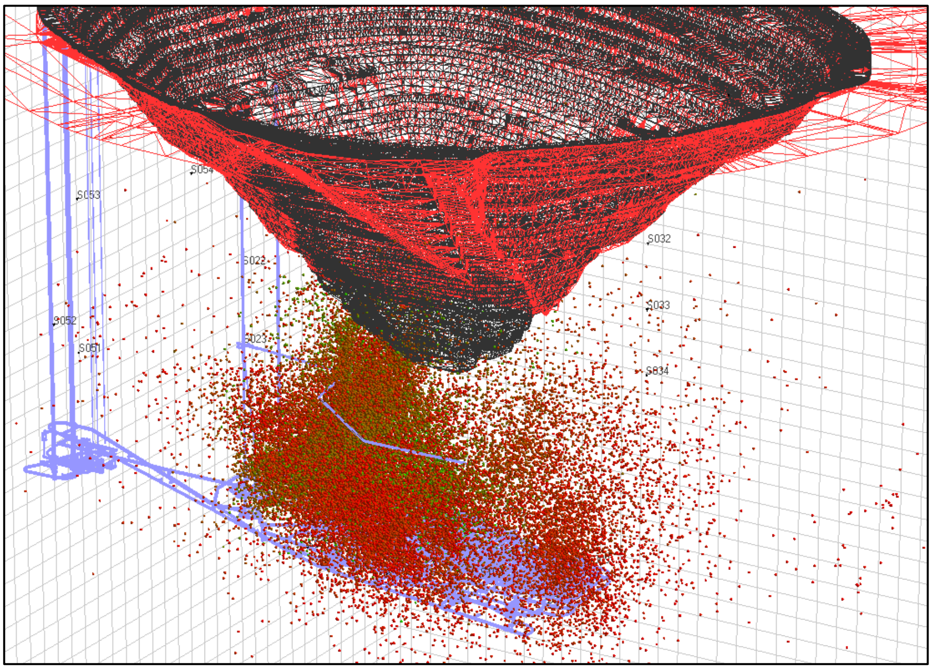


Figure 24: Micro-seismic events recorded at PMC
Source: (Reyes-Montes et al., 2010)

Ultimately, the evolution of seismic damage and behaviour observed at the lower open pit resultantly interprets as a precursor to the potential induced seismicity at the slope failure level (Reyes-Montes et al., 2010).

With seismic data analysis studies, reliable and successful monitoring of the caving process assists with caving draw compliance. At Palabora, the seismic activity rates dropped with production stoppages of several consecutive days between 2002 and 2008 (Glazer & Townsend, 2010). For these short periods of stoppage, where no material pulling from the cave occurred, the cave expansion stopped since the cave had no place for the caving material. Palabora was always pulling at a lower rate than the natural cave progression rate, indicating a minimal or no expansion void at the top of the cave (Glazer & Townsend, 2010).

These studies and knowledge base is equally important for the resource reporting, and additionally, similar mine modelling can be improved with sufficient subsidence, fragmentation and failure predictions, potentially affecting the resource grades.

2.12 Fracture banding in caving mines

The Cumming-Potvin et al. (2016) paper describes alternative criteria which question the Duplancic & Brady (1999) model. This paper indicates that the Duplancic & Brady (1999) conceptual model did not resemble the anticipated conceptual caving. Previously the Duplancic & Brady (1999) model has been widely accepted by the mining industry (Cumming-Potvin et al., 2016).

In the latest discovery, a series of fractures develop parallel to but ahead of the cave front and the cave periphery. By investigating the direction of movement and lack of damage to the asperities along the fractures, authors judged these fractures to be extensional (Cumming-Potvin et al., 2016). Alternatively, the cave progressed in what seemed to be several leaps to successive parallel fractures. A resultant discontinuous damage profile ahead of the cave back develops in contradiction with previous interpretations of continuous damage profiles (Cumming-Potvin et al., 2016). This parallel fracturing which occurs ahead of the cave back is known as fracture banding. Cumming-Potvin et al. (2016) indicated that several authors found that the tensile fractures manifested in what appears to be a series of discontinuities parallel to the cave front. This indication explains that these observations already existed in previous studies (Cumming-Potvin et al., 2016).

The phenomenon of fracture banding by indication occurs in real cave mines, and field observations are essential to support fracture banding. However, Cumming-Potvin et al. (2016) indicated that fracture banding in the cave evolution needs more recognition as an essential mechanism of cave propagation. Importantly additional research needs to determine the condition under which fracture banding occurs. The significance of fracture banding in caving mines still has to be determined. Cave design and cave management could determine leading indicators for the importance, and role fracture banding plays in cave propagation (Cumming-Potvin et al., 2016).

This study applies to PMC where the cave back and the slope failure outlines show indicative signs of possible fracture banding. Fracture banding possibly affected the outline of the PMC cave back as per Diering's (2015) indicated resource model.

2.13 The PMC open pit slope failure and its impact on the resources

Across the industry, whether in sublevel caving operations or block caving, dilution and recovery issues existed because of the constant mixing of ore and caved material (Shekhar et al., 2016). Table 1 indicates the effect of dilution and ore loss in mine economics, affecting the mine profits and return on investment. Dilution has a particular amplified effect, which requires adequate control (Shekhar et al., 2016).

Table 1: Effect of dilution and ore loss on mine economics - Source: (Steffen & Kuiper, 2011)

Hustrulid (2001) indicated that:	**Elbronds (1994) indicated that:**
A 5% decrease in ore grade would result in a 15% loss in profits	A mine's Net Present Value (NPV) is halved if either of the following scenarios occur: ** Ore losses become 20% higher than expected, or* ** Ore grade becomes 10% lower than expected (e.g. via dilution)*
A 5% increase in ore grade yields a greater improvement to profit than reducing mining costs by 5%	The above scenarios could increase production costs by 75%

Ngidi and Pretorius (2008) reported that after the cave propagated through the crown pillar into the open pit bottom, the toe supporting the pit walls lost its footing. The pit walls unravelled at the weaker lithology contacts and fault lines, which caused the north pit wall to slide into the pit bottom (Ngidi & Pretorius, 2008). Ngidi and Pretorius (2010) estimated that 150 million tonnes of waste from the slope failure of the open copper pit failed into the block cave of PMC. Ngidi and Pretorius (2010) calculated resultant losses of up to 30% from the original ore reserve. This project report identified that Ngidi & Pretorius (2008) were on the money as per say when they correctly mentioned the sliding of the failure to the pit bottom. Unfortunately, Ngidi & Pretorius (2008) focussed in large on the reserve losses and not on the sliding effect of the failure while Diering (2015) re-identified the sliding through tests in 2015. Ngidi and Pretorius (2010) reported that critical grade variations at PMC called for continuous reserve reconciliations which became inevitable. The Ngidi & Pretorius studies influenced the importance of annual flyover surveys of the open pit. With flyover data, calculations of waste influx material flowing into the block cave from the failure enable more accurate future predictions of the minable resource. These studies bear valuable information and contribute to the resource management at PMC.

Additional losses form at the shoulders of the cave and remain problematic to the cave unless pre-conditioning via drilling from the surface becomes possible. Figure 25 indicates probable losses of reserve areas at the cave’s top.

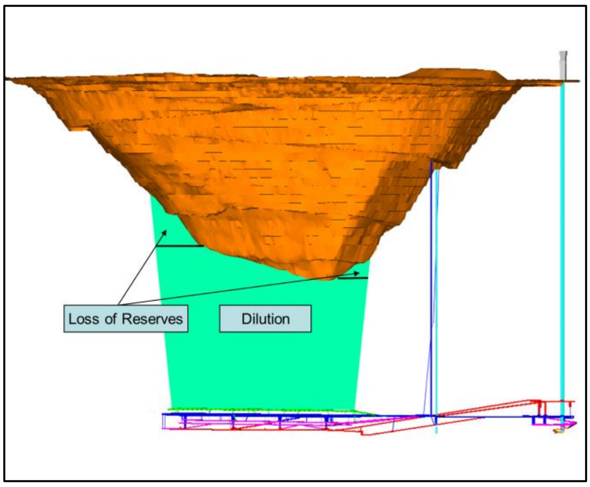


Figure 25: Reserve losses indicated as remnant shoulders
Source: (PMC, 2007)

Studies over time created a cave mine database for empirical analysis and characterisation of caving-induced surface subsidence. The populated database formed from more than a hundred cave mining operations, including ceased mines and operational mines across the world (Woo et al., 2013). PMC incurred a 30% ore reserve loss from an 86 to 88-degree caving angle and slope failure which primarily contributes to the dilemma the company faced (Woo et al., 2013). This research by Woo et al. (2013) supplements the Ngidi & Pretorius (2010) report which states that the low-grade ingress material posed an adverse effect on the LOM of the block cave.

According to the article “Block Caving: Mining Specialisation ” which Walker (2014) wrote, Professor Yves Potvin highlighted in 2011 that several block-caving risks existed. One main risk is the sterilisation of a significant quantity of broken ore, requiring draw control management (Walker, 2014). Rio Tinto indicated that the failure resulted in less future copper revenue, since possible additional costs to move facilities away from the failure’s path may be inevitable. The failure significantly affected the resource picture, and the reserve loss was nearly 30%. The slope failure resulted in lower confidence in PMC’s ability to predict behaviour (Calder, 2006). Table 2 below indicates the status of Lift 1 mineable tonnes and losses as per the specified reports.

Table 2: Mineable tonnes status per year

Year (end)	Estimated Resources Mt (CP Reports)	CPR Grade % Cu	Cave Management Reported Mined tons (Mt)	Estimated Balance of Resources	Cumulative Residual Tons (Estimated - mined)	Variance (Mt)	Competent Person Report & other reports	Comments
Pre-2000	245	0.69		244	244		FS Report & Calder (2006)	The Pre FS indicated 245 Mt
2000	228	0.69	0	243	220	-23	Reserve Statements	Marginal Ore on surface removed UG total, Calder (2006) mentioned 220 Mt
2001	225	0.69	0	228	220	-8	Reserve Statements	Annual Reconciliation
2002	216	0.69	3	225	217	-8	Reserve Statements	Additional 3 Mt mined from UG Reserve in the open pit
2003	216	0.69	7	213	210	-3	Reserve Statements	Additional Ramp scavenging in open pit depletes UG Reserve
2004	201	0.65	9	209	202	-8	Howson (2006) SLC Workshop	The Northern slope failure occurred
2005	138	0.65	10	138	121	-17	2009 CPR	54 Mt written off (+- 30%) from 185 Mt
2006	120	0.63	11	120	110	-10	2009 CPR	Howson(2006) revised resource model
2007	104	0.62	12	104	98	-6	2009 CPR	Annual Reconciliation
2008	91	0.62	12	91	86	-5	2009 CPR	Annual Reconciliation
2009	75	0.61	12	75	74	-1	2009 CPR	Annual Reconciliation
2010	62	0.60	11	62	63	1	2010 CPR	Annual Reconciliation
2011	49	0.57	11	49	52	3	2011 CPR	Annual Reconciliation
2012	35	0.54	9	35	43	7	2012 CPR	Annual Reconciliation
2013	26	0.57	10	26	33	7	Cave Management Records	Annual Reconciliation
2014	15	0.57	12	15	21	7	Cave Management Records	Annual Reconciliation
2015	40	0.55	10	40	11	-29	2016 CPR	Reconciliation of Failure material added +-29 Mt , Diering(2015)

In 2010, Snowden (2010) indicated only 75 Mt available as feasible tonnes to be mined. Notably, in 2011, the indicated remaining tonnes was 49 Mt and caving was expected to end at the end of 2015 (Snowden, 2010). However, Diering (2015) reconciled the failure's material with applicable grades after four years of mining since the 2011 CP report. Diering (2015) indicated a similar remaining total mineable tons in 2015 as was predicted to be the remaining total back in 2011. From the table above, there is a clear indication that the initial calculations reported a massive loss in reserves, while only much later the failure material added a couple of more years to the LOM beyond 2015.

Walker (2014) referred to the author of the monograph namely, Dr Tony Diering, the vice president of Geovia's caving business unit. Walker (2014) reported from Diering's indication that ore reserve estimations are more complicated due to "peculiarities" of block cave mining. These "peculiarities" include complications such as mining geometry, material fragmentation, material flow, or dilution from various sources (Walker, 2014).

Diering's follow-up PMC visit in 2015, resulted from continuous work in progress and a different grade diversion experience (Diering, 2015). This acknowledged work applies to the management of the PMC copper resource model.

The SRK Consulting (2011) audit report for PMC indicated that the north wall collapse of the pit sidewall incurred a 130 Mt of waste material deposited on top of the Lift 1 reserve. This waste is expected to filter into and through to reporting draw points situated on the extraction level of Lift 1 (SRK Consulting, 2011). SRK Consulting (2011) forecasted that mining would become uneconomic by 2015 despite the uncertainty to quantify the waste with a positive number. Hence the original 254 Mt deposit reduced to a merely 117 Mt as a result of the 2004 failure. The overlying waste will undoubtedly result in waste ingress at an earlier stage than initially planned, and SRK expected these results to show in 2010 (SRK Consulting, 2011).

Figure 26 shows the evolution of the north wall failure when looking north-west.

Figure 26: Evolution of the north wall failure (north-west view)
Source: (Sainsbury et al., 2016)

2.14 Toppling theory at Palabora Mine and the dilution envelope

The copper block cave of PMC came near to its end of life as was predicted to end in 2015 (SRK Consulting, 2011). During the expected final period for Lift 1's LOM, additional studies on the copper resource and draw control refinement were required. Importantly PMC had to keep the mine producing for as long as possible. The complicated toppling failures of rocks slopes result from combinations of continuous and discontinuous deformation. Multiple sources exist with research on simulating slope deformations and shear strength reduction methods that were adopted by industry. A case study of a slope at the Fushun open pit mine in China accurately indicated the depicted rock deformation (Li et al., 2015). The toppling failures of slopes are one aspect at PMC where the resource modelled grade and draw point's sampling grade indicated an inverse parting during 2015 (Diering, 2015).

In Figure 27 below, diagram (a) indicates the dilution envelope, (b) indicates the waste material from the pit failure, (c) indicates the waste material mixing with the cave material and (d) indicates the dilution envelope where Calder (2006) expected toppling. The toppling of material can also occur at the top of the open pit, where sidings fracture and material topples to the bottom of the open pit from higher areas above (Diering, 2015).

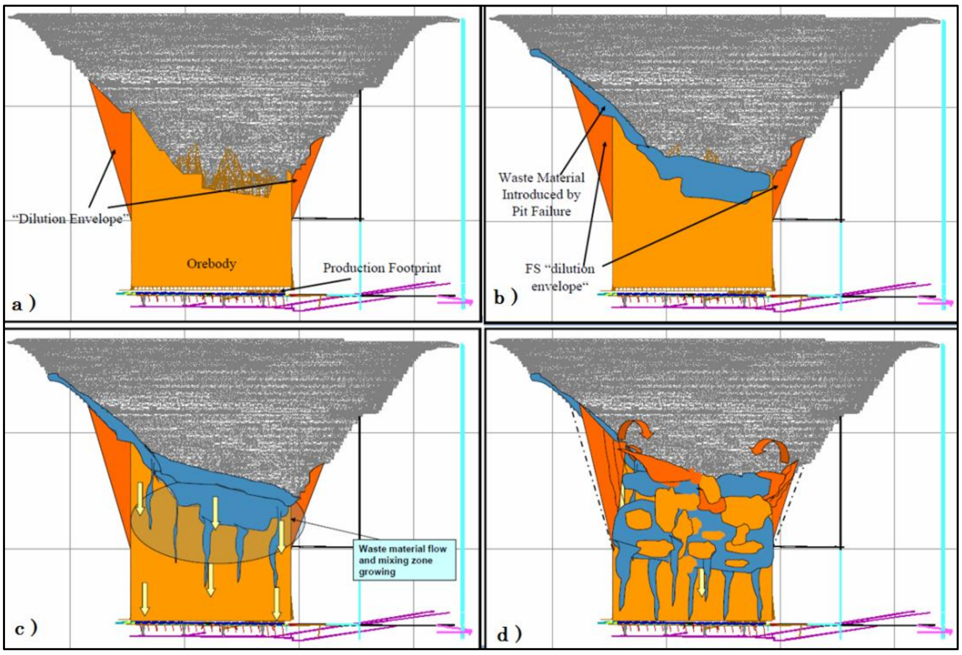


Figure 27: Toppling effect at Palabora Mine
Source: (Calder, 2006)

Failure mechanisms driven by the interaction of geology with the mining operations such as plain, wedge, circular, toppling failures and step-path failures cause rock falls (Barnet, 2003). These failures and knowledge base apply to PMC where observations indicated minimal toppling during the PMC slope failure. A more detailed discussion on toppling follows in Section 9.

Draw control within the Dassault Systemes Geovia's PCSLC (Personal Computer Sub-Level-Caving) modelling software has extensive technical detail on material flow and how this modelling works through established algorithms (Shekhar et al., 2016). This project report acknowledges programmed algorithms from conducted studies and work, but it will not attempt to elaborate on the mixing science. This report will reference researched mixing detail only since it influences the resource management at PMC.

Diering (2015) frequently consults at PMC and contributes valuable insight and expertise regarding the understanding of PMC's block cave. The Diering (2015) resource analysis came at a critical time when the mine experienced grade discrepancies. Section 9 elaborates in more detail on the work Diering (2015) conducted regarding the Palabora block cave behaviour and resource reconciliation.

2.15 Conclusion

This section covered block caving in simple terminology and described what block caving is all about. The literature review focused on conceptual and numerical modelling, which predicts material flow and cave behaviour. Poor fragmentation results in challenging drawing control, which alternatively affects the resource grade. The crown pillar failure of PMC entailed several studies over time, including a micro-seismic analysis between the cave and pit slopes. These studies contribute to efforts of understanding block caves and to predict block cave behaviour more accurately.

Notably, the vast extent of material, studies, and content on block caves could never narrate all the relevant criteria for PMC's copper block cave behaviour. Thus, the need for further research exists in the specific case of PMC's block cave, having higher than expected grades at its end of life, despite the inflow of external material from the slope failure.

The required additional research could include fracture banding in caving mines, and possibly determine the role it played during the LOM and the effect it had on the resource grade.

3 PMC HISTORY AND THE MINERALISATION OF PMC

3.1 Introduction

Kuschke & Tonkin (1971) reported that Carl Mauch was the first European to record copper occurring at Loolekop during 1868 and 1871. The first mining around Loolekop was for phosphates in 1930 and vermiculite in 1936. Kuschke & Tonkin (1971) also reported that Mr C.H. Cleveland, a pioneer prospector of the area noted large books of "rotten mica" exfoliated during a grass fire, and consequently discovered the vermiculite. "The late Dr Hans Merensky played an active part in the prospecting for phosphates and vermiculite during and following the Second World War." (Kuschke & Tonking, 1971). Prospecting for copper dates back to 1952 when the logical Unit of the Atomic Energy Board discovered uranothorianite in the carbonatite from Loolekop. However, the prospecting programme established that the concentration of radioactive minerals had no economic significance, but the copper rendered the deposit to be of value (Kuschke & Tonking, 1971). The drilling programme confirmed over 300 million tons of ore amenable to opencast extraction via an open pit size of 1 500 m x 900 m, with a final depth of 370 m at a finished pit slope of 45º (Kuschke & Tonking, 1971). Shaft sinking started in October 1960, and development blasting of 1 610 m on the 122-metre-level commenced during June 1961 to March 1962. Bulk sampled rock from this development ran 100 tonnes per day into a pilot plant. The results formed a reliable basis to plan full-scale mining, milling, concentrating, and smelting operations, on a scale previously unknown in South Africa (Kuschke & Tonking, 1971).

3.2 Lithology densities and empirical stability relationship

The carbonatite uniaxial strength is about 120 MPa with variations between 90 MPa and 160 MPa, depending on the mineralogy. The dolerite is a stronger brittle rock with a uniaxial strength of 320 MPa. Dolerites adjacent to the primary faults are locally weathered and less secure, its reduced strength is approximately 80 MPa (Ngidi & Pretorius, 2010). Table 3 and Table 4 below indicate the rock strengths, rock mass and caved-rock properties of the Palabora Mine deposit. Table 3 indicates that the rock strengths are very competent rock and whether the ore body would propagate, posed as an initial uncertainty but modelling tests indicated the cavability of PMC's cave (Moss et al., 2006).

Table 3: Rock Strengths - Source: (PMC, 2007)

Rock Type	UCS Strength (MPa)	RMR (Laub)
Carbonatite	120	66
Dolerite	330	66
Foskorite	63	56
Micaceous Pyroxenite	80	54

Table 4: Rock mass and caved rock properties - Source: (Severin & Eberhardt, 2012)

Model	Density	Cohesion	Friction Angle	Bulk Modulus	Shear Modulus	Tension
Units	(kg/m³)	(MPa)	(deg)	(GPa)	(GPa)	(kPa)
Rockmass	2700	2.9	45	1.79	1.32	100
Caved Material	2300	0.0	30	0.42	0.19	100
Faulted Rock	2500	1.0	35	0.85	0.59	0

The FS predicted caving to initiate when the undercut area reached an area of 140 m x 140 m. Despite the lack of knowledge and substantial uncertainty regarding the unpredicted caving possibility, the overall footprint dimensions were sufficient to counter the uncertainty (Moss et al., 2006).

Moss et al. (2006) reported that the required dimensions to initiate caving were approximately 30% greater than the predicted dimension, while the hydraulic radius at which caving occurs proved well within the footprint dimensions. The hydraulic radius of 45 m formed during April 2002 when caving initiated (Moss et al., 2006). Moss et al. (2006) described Palabora having the strongest rock mass for any block cave operation to date. The unadjusted intact rock mass rating (IRMR) values plotted on Laubscher's (2000) stability chart, indicated the hydraulic radius for Lift 1 is 42 metres. In Figure 28 the red line represents the initiated continuous caving specifics for Lift 1 (Rio Tinto Technical Team, 2011).

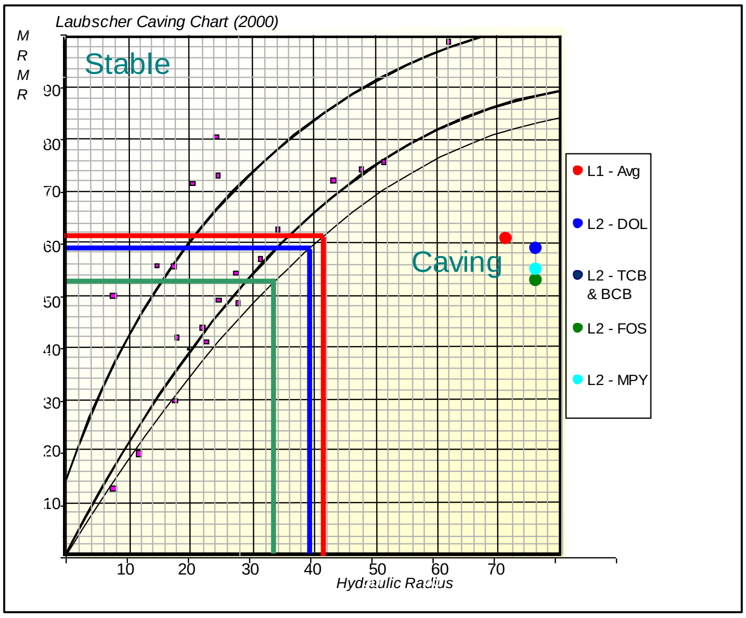


Figure 28: PMC's Lift 1 hydraulic radius shown in red
Source: (Rio Tinto Technical Team, 2011)

The empirical stability relationship established by Laubscher (1995) predicted a hydraulic radius for caving of 35 m where the MRMR of 60 transpired for the Palabora orebody (Calder et al., 2000). Itasca conducted the detailed caving potential investigation for the Palabora Underground Mine by using FLAC^{3D} and 3DEC numerical codes. The investigation predicted a slightly more significant hydraulic radius than Laubscher's empirical relationship (Rio Tinto Technical Team, 2011).

3.3 Mineralisation

The Phalaborwa complex covers an area of about 1950 hectares, which consist mainly of phlogopite and apatite rich pyroxenite. Pyroxenites incurred successive intrusions by a series of more differentiated rocks, such as foskorite, olivine, magnetite, apatite, phlogopite rock, and a central intrusion of sövite (transgressive carbonatite) (Snowden, 2010). This sövite intrusion shows an intimate relationship with the foskorite and covers about 50 hectares at the surface. The intrusion is composed of calcite and magnetite having minor quantities of dolomite, apatite, chalcopyrite, bornite and various silicates. The carbonatite is the mined copper-bearing rocks with magnetite, and minute amounts of platinum and gold as by-products (Snowden, 2010).

3.4 Mineral resources and reserves background

Rio Tinto South Africa (RTSA) and Palabora developed the mineral resource model for the Palabora underground mine. The SUMP suite programme, based on traditional 3-dimensional block modelling approaches is a multi-matrix model. The estimation of grades performed by a kriging estimator estimated the grades and other parameters of the blocks (Calder et al., 2000). The delineated footprint's boundary of the vertical draw columns and the mineable resource caving angles ranged between 82º and 88º. These supplied delimiters by RTSA defined the Palabora block cave mine. The initial feasibility results and evaluation forecasted minable reserves of 244.6 M tonnes, with a 0.69% copper grade, producing 1.6 M tonnes of copper (Calder et al., 2000). The initial calculation of the Palabora LOM resulted in approximately 20 years of block cave mining.

For Palabora Mine to fulfil the requirements and readmission to the official listed Johannesburg Stock Exchange (JSE), the May 2010 competent persons' (CP) report complied with the JORC code (Snowden, 2010). The 2010 CP report indicated the total remaining proved mineral reserves was

75.33 million tonnes of ore at 0.61% copper content. Palabora Mine indicated a LOM of approximately six years of future caving since 2010. During 2010, several options to extend the LOM beyond 2016, became an active investigation (Snowden, 2010).

3.5 Mining method

The current mining method entailed a mechanised block cave, which exploits the ore below the final open pit area. The block cave comprises of an undercut and production level, of which the undercut is the uppermost level sitting at 1183 m below surface. This level is about 460 m below the last worked pit bottom on the surface (Snowden, 2010). At a merely 18 m below the undercut level, the production level comprises of draw points and other production infrastructures such as four underground crushers and the conveyor infrastructure to hoist material vertically to the surface (Snowden, 2010). Figure 29 shows an isometric perspective of the Palabora block-cave mine layout.

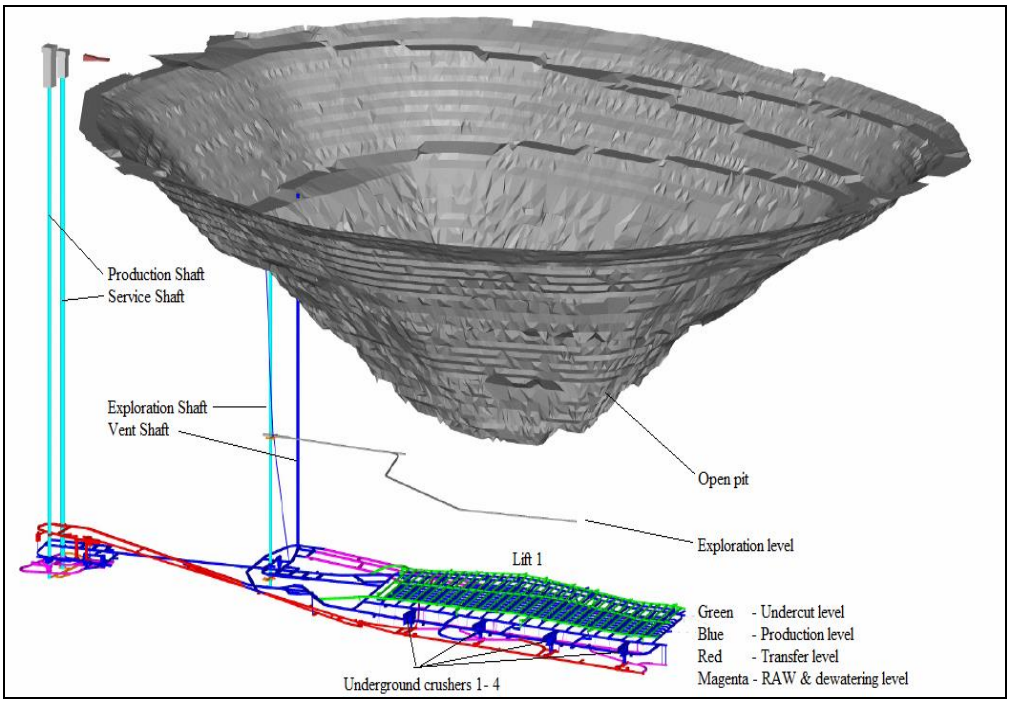


Figure 29: Palabora block cave mining layout
Source: after (Ngidi & Pretorius, 2010)

A diesel-powered fleet of twenty LHD (load and haul dumper) machines with a 12-tonne payload are utilised to muck material from the draw points into the jaw crushers, located at the northern periphery of the cave. The average one-way hauling length is about 175 m, where the crushers discharge crushed material into the sacrificial conveyors (Snowden, 2010). The sacrificial conveyors feed onto a single horizontal section inclined conveyor (conveyor 5), which handles 1 800 tonnes per hour. Conveyor 5 delivers the material into the two 6 000 tonne production shaft silos. The ore is ultimately hoisted to the surface using four 32 tonne payload skips driven by a Koepé winder, where one skip counterweights the other (Snowden, 2010).

Two main shafts are in operation, a 10 m diameter service shaft and a 7.4 m production shaft, both sunk to a final depth of 1 280 m below the surface collar. The service shaft serves for men or material conveyancing and is the primary ventilation intake infrastructure. The maximum hoisting capacity of the production shaft's skips can load about 33 000 tonnes per day, and it serves as a supplementary ventilation intake (Snowden, 2010).

3.6 Underground loading and crushing

A Modular dispatch system monitors and directs the LHD loading activities for the underground fleet from a central underground dispatch room. A generated production schedule from a cave management database then links to the LHD communication devices and directs the LHD fleet underground to specific draw points and crusher tips. The four jaw crushers ensure that fragmentation of material is suitable for loading onto the conveyor belts where the crushed ore is less than 220 millimetres in respective sizes (Snowden, 2010).

3.7 Underground sampling in Lift 1 at PMC

In the assessment of the underground sampling procedure followed at PMC Howson (2006) and Snowden (2010) questioned the representative shortfall of the sample sizes. Where 30 000 tonnes per day is mucked from underground, the original monthly 20 kilograms from a draw point grab-sample is too infrequent (Snowden, 2010). The grab samples are particularly non-repeatable rank guesses of what the draw point constitutes of, at a specific time. Snowden (2010) considered these samples biased towards fines. Moisture and running water affect fines mostly. Hence Snowden (2010) deemed this sampling as poor practice. Palabora Mine uses these sampling values as an internal indicative data measure for draw control. However, the more the number of grab-samples taken in draw points increases, the more the confidence levels of sampling results improve (Cook, 2011). Six to eight small shovelled scooped up samples forms a sample, bagged by samplers at the face of the draw points. The shovelled sampling targets material smaller than 200 mm. Figure 30 shows in a) how to bag a composited sample underground with 6 to 8 small shovels of material and b) represents approximately 15 – 20 kg of material.

Figure 30: PMC samplers bag a 20 kg grab-sample in a draw point
Source: (Cook, 2011)

The draw points vary in fragmentation size, from fines to blocky material, and significant different material sizes occur where it is moist (Cook, 2011). The impractical exercise to move and crush bulk representative samples from draw points remains problematic for PMC and most caving operations. The resultant grab sampling aims at practical and achievable sampling for internal indicative measures. Cook (2011) indicated that the sampling aims towards practical and achievable sampling where potential tracked bias factors back into the predictive modelling. PMC created a tonne per sample tool where the past five samples in each drawing point enable tracking of the sample coverage over the entire footprint as per indicated hot and cold map in Figure 31. Figure 31 additionally indicates the average tonnes between the last five samples taken across the footprint and highlights those draw points requiring urgent sampling in red or warmer colours (Cook, 2011).

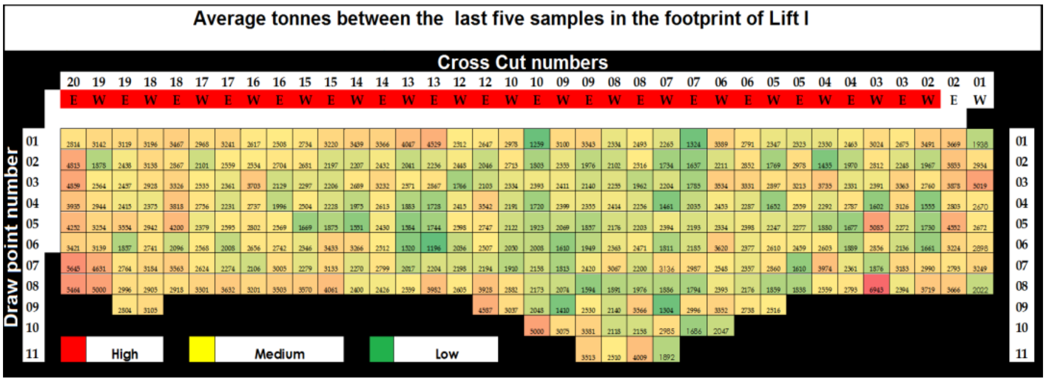


Figure 31: Average tonnes between the last five samples
Source: (Cook, 2011)

3.8 Geological results and exploration drilling at PMC

Initial drilling commenced from the open pit, and an exploration shaft sunk from bench 28, allowed for deeper drilling via the exploration level. The exploration level tunnel allowed for drilling into the orebody. From the Lift 1 production level, downward drilling into a possible Lift 2-mine commenced as per indicated magenta drill holes in Figure 32 below (Calder, 2006). The blue lines are holes drilled from the surface in the area mainly covering the open pit, the red lines are holes drilled from the indicated red exploration level, and the holes drilled from the production level in Lift 1 shows in magenta.

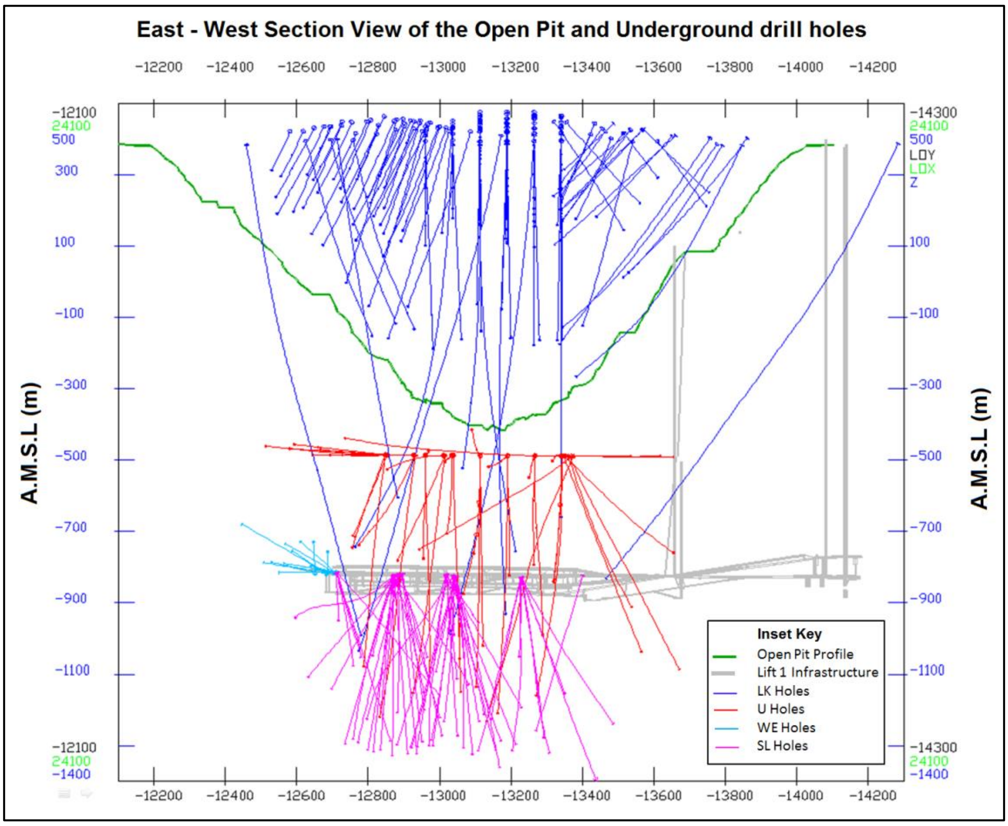


Figure 32: West – East profile showing relative positions of drill-hole types
Source: (Calder, 2006)

Exploration drilling at the Loolekop hill where the deposit outcropped occurred approximately during 1964 to 1976 to explore the open pit volume (PMC, 2016). Drill holes from the exploration level approximately -492 m elevation drilled horizontally and downwards occurred during 1991 and 1993. The drilling of Lift 2 exploration holes from the Lift 1 level underground occurred during 2007 and 2013 (PMC, 2016).

3.9 Plant and head grade sampling description

The plant comprises two autogenous mills (AM) and a separate stream for secondary crushing processes. Dolomite, which affects the floatation process, is not forming part of the immediate throughput but tapped off in an approximate proportion of 5 - 10 percent. The tapped-off material according to Snowden (2010) was immaterial to the overall mass balance, which does not incur many copper losses since reprocessing of this material forms part of the processing over time (Snowden, 2010).

The current head grade sampling point is at the AM cyclone overflow while the previous position was on the secondary ball-mill cyclone overflow. The repositioning of the sampler occurred during the latter half of 2007, measuring daily head grades since the beginning of 2008 (Snowden, 2010). According to Snowden (2010), a study conducted on the repositioning of the new sampling points indicated that unbiased results were statistically consistent with the previous location points. Snowden (2010) concluded that the initial positioning of the Multotec sampling units had the same unbiased consistency irrespective of the relocation of the units. In the milling process, 100% of the material drawn from the surface stockpiles report to the cyclone overflows (Snowden, 2010).

The sampling of the plant head-grade results from a cyclone overflow using two Multotec double-stage sampling units, installed at the cyclone overflow pipe. Figure 33 illustrates the Multotec two-stage sampling units installed at the cyclone overflow pipe, measuring the head grade (Snowden, 2010). Four stainless steel cutters subsample the initial sample at the secondary sample cutter. These samples are composited per shift, filtered at the plant and dispatched to the laboratory for final analysis. Sample collection occurs through the pipe column to the sample cutters during a full stream primary cut (Snowden, 2010). In addition to the primary sample, a sub-sample, taken by a secondary sample cutter, comprises of four stainless steel cutters.

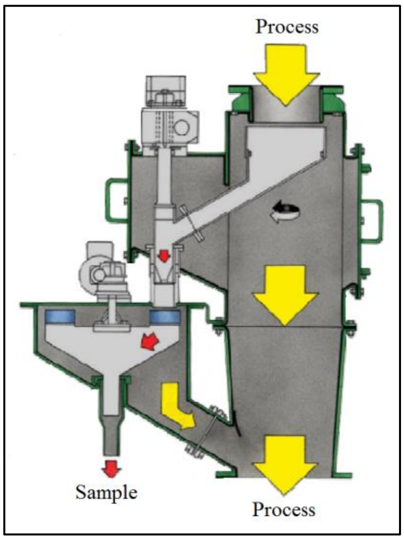


Figure 33: A Multotec sampler used for sampling the head grade at PMC
Source: (Snowden, 2010)

After dispatching composited and filtered samples from the plant to the laboratory, the analysis of samples follows. Snowden (2010) found the equipment and sampling procedures at PMC acceptable.

Figure 34 illustrates the flow sheet of the underground material stream with the sample cutter locations.

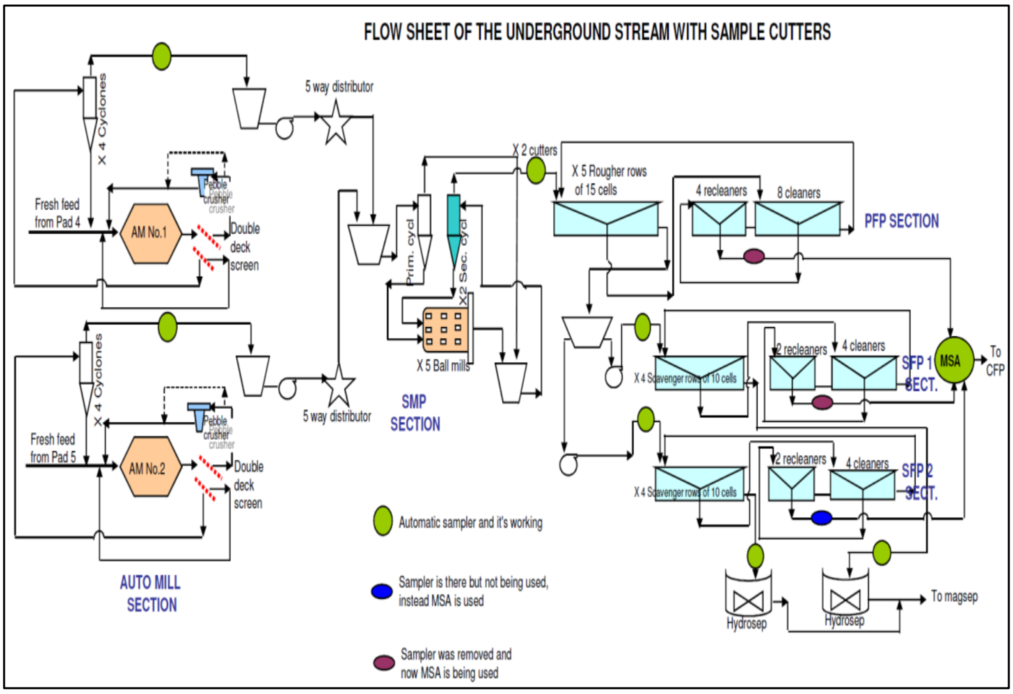


Figure 34: Flow sheet of the underground stream with sample cutters
Source: (Snowden, 2010)

Current assaying of individual samples involves an acid decomposition followed by atomic absorption spectrophotometry for copper and acid decomposition followed by inductively coupled plasma spectrometry for iron (PMC, 2016). The composite samples analysis for Cu entailed two methods; Cu-AA62a and ME-O62. The PMC laboratory follows atomic absorption procedures (PMC, 2016).

3.10 Conclusion

In section 3, the number of drill holes for the FS study of Lift 1 was not adequate to be representative of the cave and resource model. This shortcoming attributed to the lower confidence in grade from the modelled resource. PMC worked with the available information and consistently found discrepancies between the head-grade and the modelled grade of the resource. Continuous refinement and adjustments to the parameters within PCBC followed as more information became available through time. PMC started on the back foot with the Lift 1 resource calculations in that Lift 1 had inadequate drilling information. Both the methodology to model the resource and the sampling bias from poor representative sample sizes featured throughout the LOM. Consequently, PMC remained in an inevitable position to ensure that resource revisions occurred annually. However, the actual sampling results which remained not acceptable to good sampling practices, representation and biased provided valuable information to understand the block cave behaviour despite its indicative measure.

4 CREATING A TIMELINE FOR THE PMC RESERVE STUDIES

4.1 Introduction

This section looks back in recorded historical reports of previous onsite studies, forming part of methods to refine and improve the resource grade of Lift 1's block cave. The section will cover the conducted studies between 2002 and 2006. The section also reviews the significant challenges recorded over time regarding the status quo of the resource model for PMC's first block cave.

4.2 Resource grade studies and reports on PMC's block cave mine.

Historical written reports for PMC, covered in this section will provide a summarised understanding of previous grade anomalies between the head-grade and reserve model grade.

Reports dating from 2003 by Howson (2003) identified requirements for the ore grade scrutiny and revision. Later in 2010, an audit and report conducted by Snowden Consultants made recommendations for the PMC block model and resource classifications (Snowden, 2010). This research will discuss critical details, and anomalies found, but will not attempt to expand the current knowledge base thereof.

4.3 The basis of initial SRK studies in 2002 and Howson in 2003

The elliptically shaped, and vertically dipping copper deposit of the PMC mine comprises of a low-grade ore body where the open pit mine started with production in 1956 (Glazer & Hepworth, 2006). The PMC copper open pit's last production blast occurred in April month during 2002. From the approximately 903 million tonnes of ore mined from the open pit, PMC produced about 4.8 million tonnes of refined copper. A total number of 2.1 million tonnes of ore and waste removed from the surface mining open pit operation, the surface mining operation initiated possibilities to transition underground by establishing a block cave mining operation below the open pit. The transitioning to an underground operation indicated a possible injection to the PMC's life of mine (Glazer & Hepworth, 2006).

PMC started with the shaft sinking and development project of the underground mine in 1995 while the actual caving initiation process happened somewhere in April 2002. About eight months after the caving started, the crown pillar between the upward propagating block cave and the bottom of the open pit excavation presumably holed into one another.

The estimated crown pillar failure occurred approximately by the end of 2002 (Glazer & Hepworth, 2006). Late in 2002 after the block cave started with production, the consulting company SRK reviewed the operation since PMC produced low copper production grades (Howson, 2006). The SRK report recommended additional required sampling of the orebody, and SRK believed that the resource model was overestimating the in-situ copper grades. Howson (2003), the principal consultant and geologist of the technical services team from the Rio Tinto company investigated this first resource grade anomalies which PMC experienced. Rio Tinto tasked Howson (2003) to undertake additional studies at PMC which started in December 2003 and onwards via the scheduled visits to the mine site. During the studies of Howson (2003), an identified compliance shortcoming at PMC required urgent attention as Howson (2006) described and detailed it in his 2006 report. Howson (2006) described the objective of the study in the documented report to support PMC with the declaration of its copper resources and reserves while meeting the Australasian “JORC” code compliance standards. These extensive studies developing the reserve classification led to the deployment of a technical tasked team from PMC’s largest shareholder company - Rio Tinto (Howson, 2006).

Howson (2003) reported that a starting point for the required work evolved from two other previous PMC reports namely the “Palabora Underground Mining Project” (PUMP) and the Feasibility Study (FS). The first “Feasibility Report – March 1994, Volume 2 – Ore Reserves” report, needed substantial reviewing before it was at an acceptable standard. The other report, “Feasibility Report – January 1996” included the classification of “Mineable Reserves” which was accepted by PMC as the basis for developing the block cave mine. These two available reports, with its information and the interpretation thereof at the time, were presumably satisfactory at first. However, an objective arose to demonstrate that the occurring dolerite in the run of mine (ROM) ore was controllable to maintain a satisfactory ROM copper grade (Howson, 2006). The technical team decided to conduct a reconciliation study to compare production grades with those grades estimated in the model. The findings from their study gained momentum where conclusions indicated exceeding biased copper grade estimations in the FS resource model in the early stages of the production's actual grades (Howson, 2006).

Further deepening the PMC economic crisis was to re-capitalise the Lift 1 project, because the project ran over schedule and created a capital shortage. Hence Howson (2004b) indicated in April 2004 that PMC might expect to be re-capitalised towards the third quarter of 2004. Howson (2004b) also indicated the need to continue with the resource reconciliation work, and where PMC ensured ongoing comparisons, between the predicted resource-model grades and the actual production grades. PMC also needed to re-develop a LOM production schedule, including ore fragmentation predictions which will take place during scheduling, since it would assist to forecast future equipment requirements (Howson, 2004b). Howson (2004b) also indicated that the available geotechnical information required a full review and verification. This review should include the collation thereof into a database to assemble the geotechnical model (Howson, 2004b).

4.4 Analysing and updating Palabora's geological databases

Following the 2002 SRK report, Rio Tinto tasked their technical team to conduct further analysis of the ROM and resource grade comparisons (Howson, 2003). Howson (2003), analysed the geological data, in an attempt to understand why the copper grade to that date was substantially lower than the planned head grade. Howson (2003) visited PMC during 8-17 December 2003 and worked with the PMC's technical team on site to initialise the grade discrepancy investigation. On the 19th of December 2003, Howson summarised their findings in a written memo labelled as "Project GBG046". In summary, Howson (2003) indicated that the geological model showed an increase in dolerite content at the undercut level but upwards the content remained the same. Howson (2003) also stated that the production data revealed higher frequencies of pulling at lower-grade draw bells than at higher-grade draw bells. Furthermore, the remaining grade shortfall concludes to over pulling of dolerite draw bells and an overestimation of the reserve grade in the blocks mined. Howson strongly proposed the implementation of new software, hardware and systems to utilise the current equipment on site (Howson, 2003).

4.5 Identified software limitations (2003) and required upgrades

The software and hardware in use during the original feasibility study analysis were very outdated according to Howson, and thus recommended new systems that will improve PMC's analysis and reports on grade control (Howson, 2003).

The resource model and associated data from the old (circa 1995) Hewlett Packard (HP) "UNIX" computer operating system, required a transferral to modern computer equipment and software for improved processing and reconciliation (Howson, 2006).

The early study criticised the initial "Datamine" database system, which dated back to the mid-nineteen-nineties as was driven by old computers. Anne-Marie van den Heever from PMC assisted in exporting the resource block model's files from Datamine into Microsoft Access, better known as the "Access Block Model" (ABM) (Howson, 2003). This new ABM further enabled additional processing, and quality checking (Howson, 2003).

Howson's associate Dave Frost-Barnes produced the new "Dolerite Model" (NDM) and installed it into the "Access Block Model Database" during the middle of 2003 (Howson, 2003). The model produced an updated dolerite dilution and copper grades model (Howson, 2003). When Howson (2003) and the team compared this model versus the 1995 FS model, more dolerite than initial predictions showed, with indications of a corresponding decrease in copper grade at the undercut level. However, this new model's results were similar to the 1995 expectation, which was higher above the undercut level (Howson, 2003).

Valuable and irreplaceable information was at risk if the old HP system failed. The old Datamine database comprised of drilling, assay, logging, structural, zone modelling, and block-models (Howson, 2006). The dolerite solid modelling, faults and all drill-hole data, strings and wireframes as described in the FS grade zones, were safely transferred and retrieved from the Datamine systems (Howson, 2006). Howson (2006) and the team structured a well-documented summary of all the transferred files, as is discussed in section 5. The exercise involved timeous efforts to ensure that no or very little data was lost or omitted during the transfer process.

Reconciliation also involved the revision of the FS resource model format, into a new model format namely the "Draw Block Model" (DBM). Dolerite dykes from more recent underground mapping in the vicinity and updated block dolerite values with extended dimensions formed part of the reconciliation (Howson, 2006).

The Howson (2003) report indicated that a successful transfer of the NDM data into GEMCOM software realised on the 15th of December 2003, in the United Kingdom town Bristol. This model's transfer enabled the PCBC program to produce block cave scheduling results, undertaken by Dr Tony Diering who assisted with all the background parameter programming within PCBC (Howson, 2003).

The analysis of contoured production data between May and November 2003, such as tonnes drawn per draw bell per month for the footprint area comprised of superimposing these on plans of the grades at the undercut level. The resultant indication was greater pull volumes of lower grade material in the west of the cave than those volumes of higher grades found in the east (Howson, 2003).

Additional plans evolved to transfer the NDM data into the Block-Caving-Scheduling-system. This block cave scheduling-system, introduced and produced by the consultant Robin Kear, attempted to compare the new dolerite effects with those used in the FS. Howson (2003) reported that the study of Kear showed that the MineCAD-based mining model data needed a revision. Howson (2003) noted Kear's argument that the model presumably originated from the copper open pit data, which has not been in use for several years (Howson, 2003).

Howson (2003) additionally highlighted that where the draw column and dilution envelope designs misaligned from old MineCAD designs, such required a re-design, as a result of changes since the FS. Howson (2003) recommended that PMC should not attempt to make further progress with this scheduling system, but instead follow the use of the GEMCOM's PCBC software for block cave scheduling. The recommendation to utilise PCBC came as an industry standard for block cave scheduling, which was already in use by several other operations, especially the Rio Tinto mines (Howson, 2003).

4.6 Critical early revision of Lift 1's resource model

A revision of the resource model concluded that the 20 x 40 x 60 metres block sizes within the model to be unsatisfactory for reconciling the progress of the block cave production and for scheduling in PCBC. Howson (2003) suggested revising the block sizes with 20 x 20 x 20-metre block sizes.

The resource required further revision to report lashed or loaded tonnages extracted from each drawing point with readily assigned grades from the DCM on a daily basis. Thus, daily grade estimates of grades from the model became comparable with actual production grades, enabling further investigation regarding the veracity of the models (Howson, 2006).

Figure 35 indicates the summarised revision process of the resource model.

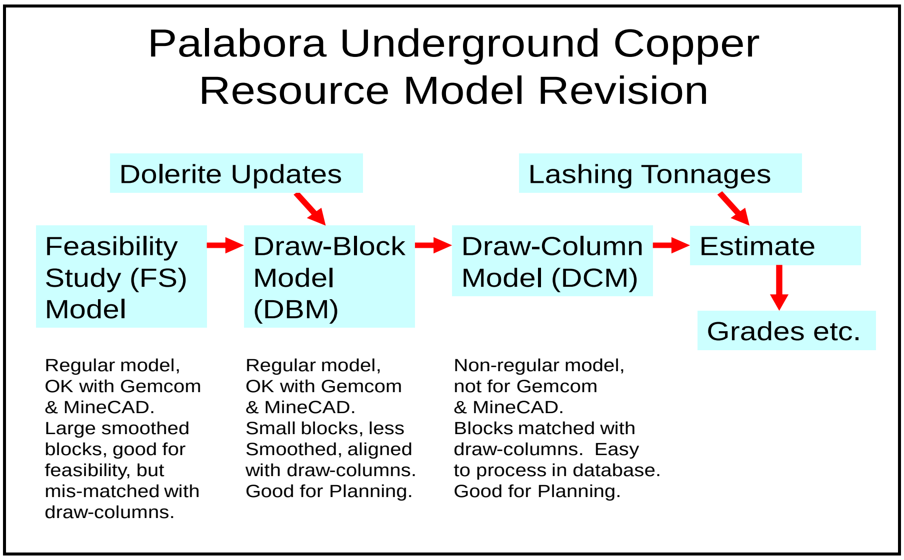


Figure 35 - Summary of the resource model revision
Source: (Howson, 2006)

The prematurely created 1995 FS resource model, emerged before the economic and practical area of the block cave's footprint was determined. This first resource model caused a mismatch of the FS model grid and that of the draw points. The created "Draw Block-Model" (DBM) from the reconciliation process, coincided with the draw point intervals as was required (Howson, 2006).

4.7 Initial grade shortfall indicators

Possible reasons for the shortfall in grade, from the initial analysis and underground visit of Howson and the team during December 2003 found:

- More dolerite at the undercut level and a shorter distance above it than was initially expected,
- Ore has been pulled at higher rates in the western areas of moderate grade than from the higher eastern areas,
- Excessive draw quantities from dolerite yielding draw points which were mainly free flowing and not reported accurately,

- Partially blocked draw points by large boulders of carbonatite comprising of both carbonatite and dolerite, resultantly had dolerite in smaller fragments running past the more massive carbonatite boulders holding dolerites back.

Although no specific evidence existed, the original grade model may have overestimated the grade (Howson, 2003).

The best test to the shortfall in grade would be through a comprehensive sampling campaign at the production level, resulting in constant disruptions to production (Howson, 2003). However, this recommendation to conduct further sampling of the in-situ rock entailed a highly undesirable procedure according to PMC. PMC decided to conduct a reconciliation study instead of drilling, which aimed to compare production grades with those estimated in the model (Howson, 2006).

4.8 Conclusion

In this section, it seems that the fixing of one concern led to several other arising challenges. Systematically the best practice possibilities enabled avenues to improve the resource model, but it was not immediately possible. Notably, two main drivers contributed to the specific revision of the PMC block cave's resource model. The first driver was the grade change between the mill grades and predicted modelled grade, and the second driver was the identified geometric change and rectification thereof. Section 5 elaborates more on the geometric details under point 5.6.2. An additional concern evolved when the introduction of PCBC also indicated that the resource grade reflected higher-grade values than those recorded from actual mining. Howson (2006) indicated that the interpolation of the applied PCBC algorithm required refined adjustments. Howson (2006) explained that the conducted and executed tests from using different kriging methods sought ways to establish the correct interpolation thereof. Howson (2006) managed to identify the initial grade shortfall areas, which addressed the draw compliance.

5 NEW MILLENNIUM SOFTWARE AND SYSTEM CHALLENGES

5.1 Introduction

The revision of Palabora's underground copper geology and resources data project successfully retrieved and transferred FS data from the HP UNIX Datamine system to a personal computer (PC). This section covers a brief description of the process that followed to ensure best possible accuracies while maintaining the data's credibility (Howson, 2006).

5.2 Converting Datamine folders and models

A preceding February 2004 report covered the in-depth descriptive details of folders and files retrieved and copied across to a more reliable, faster and better PC. Howson (2006) indicated that the extracted files and folders had the same names as the old Datamine databases, which included:

- **Block models**, with all the FS block model data,
- **Dolerite**, with solid modelling or perimeters describing the FS dolerite dyke envelopes as interpreted from previous drill holes,
- **Faults**, with solid modelling or perimeters describing the FS faults as interpreted from drill holes,
- **Holes**, which included all drill hole data,
- **Strings**, with solid modelling or perimeters describing the FS grade zones as interpreted from drill holes,
- **Wire Frames**, with solid modelling describing the FS grade zones as interpreted from drill holes.

Upon investigation, the above details in the preceding 2004 report became an annexure in the March 2006 report. After the files transfer, the study indicated that the exported CSV files and direct data importation into Microsoft (MS) Excel or MS Access were successful (Howson, 2006). The exporting process indicated several challenges in handling the data between files. Datamine exports output records with a maximum length of

240 characters only. Any Datamine table exceeding 30 fields risked a truncation of the output records in MS Access. Therefore, if more than 30 fields existed, the output of fields occurred in concatenated groups of 20 each in the Access database (Howson, 2006).

5.3 The 2004 resource model revision

Howson (2004c) reported that the revision of the resource model started during January and February 2004. This work was complementary to work previously conducted as per section 4.6 above. Howson (2004c) and the technical team made certain assumptions from the data in the model to process information. The technical team assumed dolerite had a tonnage factor value of 0.325 m^3 per tonne, and consequently assigned zero values for all applicable grades. The average tonnage factor calculated at 0.329 m^3 per tonne for all rock between 800 and 720 m below mean sea level, in the footprint area of the cave (Howson, 2004c). Howson (2004c) indicated that the cross-sectional draw column area assumingly ranged in the size of 17 m x 17 m covering a 289 m^2 area. Thus, a downwards distance of rock movement in a draw column for 1000 tonnes drawn, resultantly descends vertically by 1.14 metres at each drawing point. Howson (2004c) assumed the base of the caving was towards the floor of the production level, which was 18 m (Howson, 2004c). The delimiters above were perimeters for Howson’s calculations within the resource review.

5.4 Transformation of coordinate systems

Howson (2004c) found two sets of coordinates in use during the feasibility study of Lift 1. These comprised of the initial geographical coordinates in the geological modelling and differed from the mine planning Longitude of Origin (LO) coordinate system. The geographical coordinates increased east on the X-axis and north on its Y-axis. The LO system increases its X to the south and the Y to the west and requires a conversion of coordinates to match the locality and geological models. The known term for this conversion of coordinate systems is the transformation of coordinates. With the introduction of the GEMCOM software to PMC, GEMCOM generated the third set of coordinates, which are the reversed LO system coordinates. The z values in elevation between the data sets remained the same throughout the process of data synchronisation as was imported (Howson, 2004a).

The applied transformation process of the required co-ordinate conversions for Datamine's Geo-models implicates a rectification formulae as is shown below:

LOX = 26 000 – GeoY	GeoX = 0 – 11 000 – LOY,
LOY = 0 – 11 000 – GeoX	GeoY = 26 000 – LOX.

Note that the omitted constant of 2 630 000 in the LOX value is for simplicity reasons at PMC.

A third 'GEMCOM' system evolved and resulted in accordance with the following transformation formulae:

GemcomX = GeoX + 11 000	GeoX = GemcomX – 11 000,
GemcomY = GeoY – 26 000	GeoY = GemcomY + 26 000,
GemcomX = 0 – LOY,	LOX = 0 – GemcomY,
GemcomY = 0 – LOX,	LOY = 0 – GemcomX.

The reports of Howson (2006) and the team's documents described detailed processes and steps followed in the entire amendment process.

5.5 Misaligned draw points and draw columns

Howson (2004b) highlighted the volume of significance before describing the geological information. In Figure 36 the production level in grey and draw points in blue defines the footprint outline.

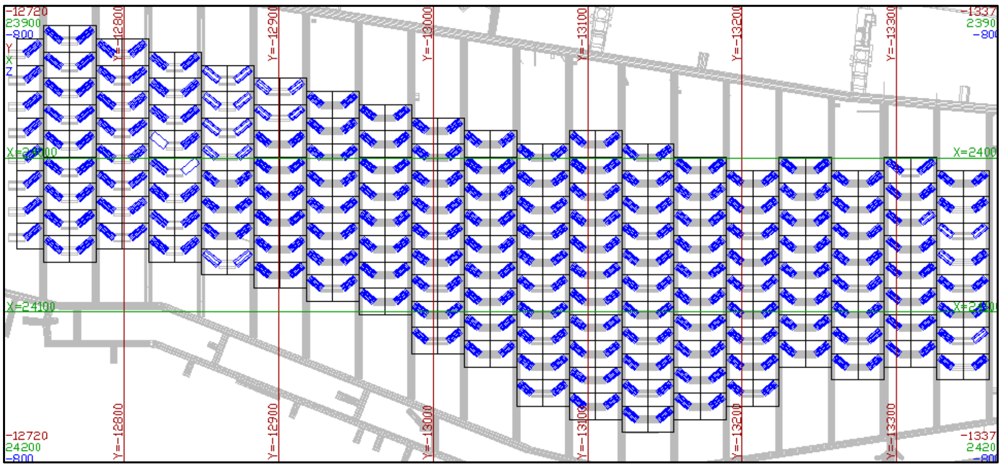


Figure 36: Footprint design indicating draw points
Source: (Howson, 2004b)

5.6 A summary of the different created models

This section and the list below discusses the different created models briefly, and elaborates in more detail how they serially improved the previously created model:

- The FS Model
- The FS Grade Model
- The FS Dolerite Model and Dolerite Dilution
- The New Dolerite Model (NDM), 2003
- The Draw Block Model (DBM)
- The Draw Column Model (DCM)

5.6.1 The FS model

In March 1994, the FS report Volume 2 and titled "Ore Reserves" describes the geology and geological modelling (Howson, 2004c). The FS model represented the block cave as a regular block model comprised of blocks with a 40 m East-West by 20 m North-South, and a 60 m vertical size. This model reflected the emplacement sequence of intrusive rocks in two parts namely: "Non-dolerite grade model" and "dolerite model" (Howson, 2004c). The separation of these two main parts in the FS model resultantly came as a requirement to update each part independently.

5.6.2 The FS grade model

The FS grade model comprised of the tonnage factor, copper, triuranium octoxide, magnetite, phosphate, and other grade percentages (Howson, 2004c). These estimates occurred in non-dolerite rocks, namely banded and transgressive carbonatite, micaceous pyroxenite and foskorites. The copper percentage has the most interest in the FS grade model. The block grade values and its applied geostatistical methods never anticipated for the inclusion of the dolerite dykes (Howson, 2004c).

The plot in Figure 37 below indicates the miss matching of the FS model's blocks with the draw columns in black, and the values represent percentages without the influence of the existing dolerites.

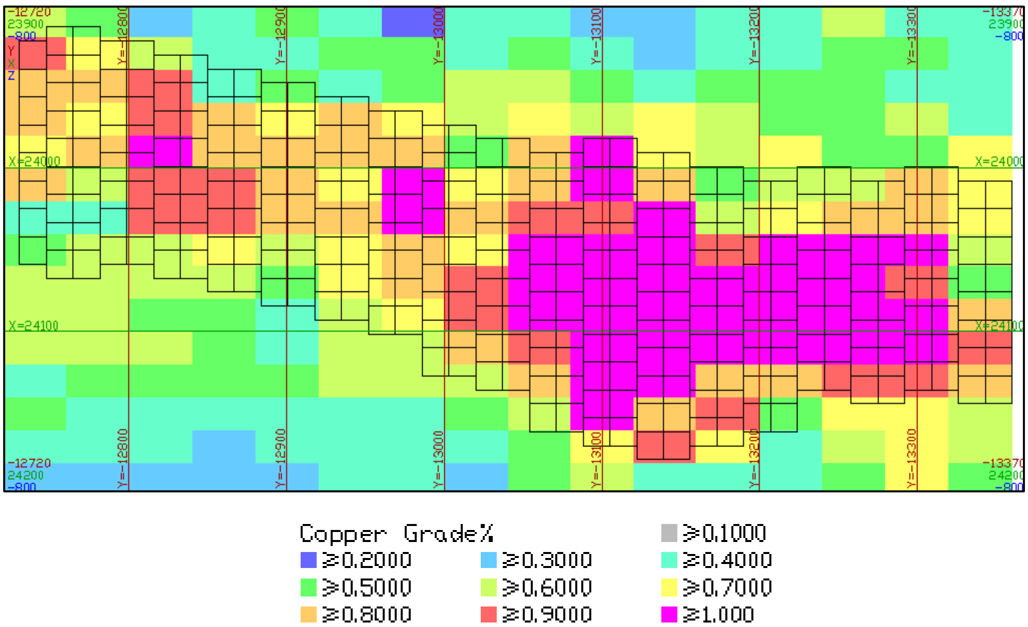


Figure 37: The FS grade model and Cu grade values
Source: (Howson, 2004c)

The non-dolerite rock's tonnage factor values in the FS grade model, was estimated mean values assigned for each rock type (Howson, 2004c).

5.6.3 The FS dolerite model and dolerite dilution

After mineralisation, the dolerite dykes intruded the ore body with negligible copper. Each block model had a percentage of dolerite in the dolerite model. During the FS, Datamine software modelled the dykes as perimeters on successive levels. By determining the proportion of each block lying within a dyke's perimeter produce an assigned percentage of dolerite by volume allocated to each block (Howson, 2004c). Through the determining of "dolerite-diluted" copper grade (Gdd) values, enabled grade predictions made from the block model. According to the Howson (2004c) report the applied formulae to calculate the mass-weighted mean, while assuming that the copper grade of dolerite is zero is: Gdd = Gc x (100 – Dol) / (Td + 100 – Dol).

The average tonnage factor of 0.325 seemed to be a constant since there was minimal variation in the dolerite. The adjustments to the FS model resulted in the FS dolerite model as is shown in Figure 38.

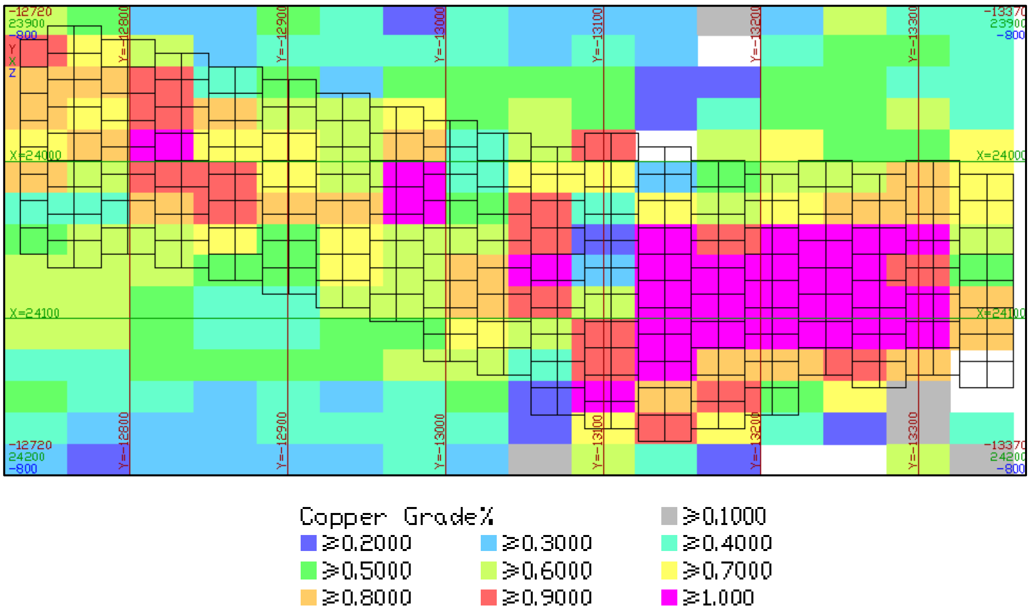


Figure 38: The FS dolerite model and resulting copper grades at – 800 m elevation.
Source: (Howson, 2004c)

The blocks indicated as white areas had copper grades of less than 0.1% Cu and were essentially dolerite. The newly calculated grades in the FS dolerite model indicated in Figure 38 compares reasonably well with those in Figure 37 for higher than 0,7 % Cu (Howson, 2004c).

5.6.4 The “New Dolerite Model” (NDM), 2003

During a software training exercise with the GEMCOM system in 2001, the modelling of dykes via a “solid-modelling” technique created TIN envelopes. A small number of 3D triangulated network of dolerites and non-dolerite rock contacts in drill holes formed envelopes by manual interpretation. This construction formed dolerite dykes in form and volume, where proportions of each block within dyke envelopes determined the dolerite percentage assigned to its volume (Howson, 2004c). This NDM was more representative than its former Datamine-based FS model. However, the mapping conducted in the Eastern end of the footprint area indicated inaccuracies in the “New Dolerite Model”. Howson (2004c) modified this area precisely as the rest of the footprint, still had to be developed. Figure 39 below illustrates the dolerite dilution of the FS copper grades at – 800 m elevation (Howson, 2004c).

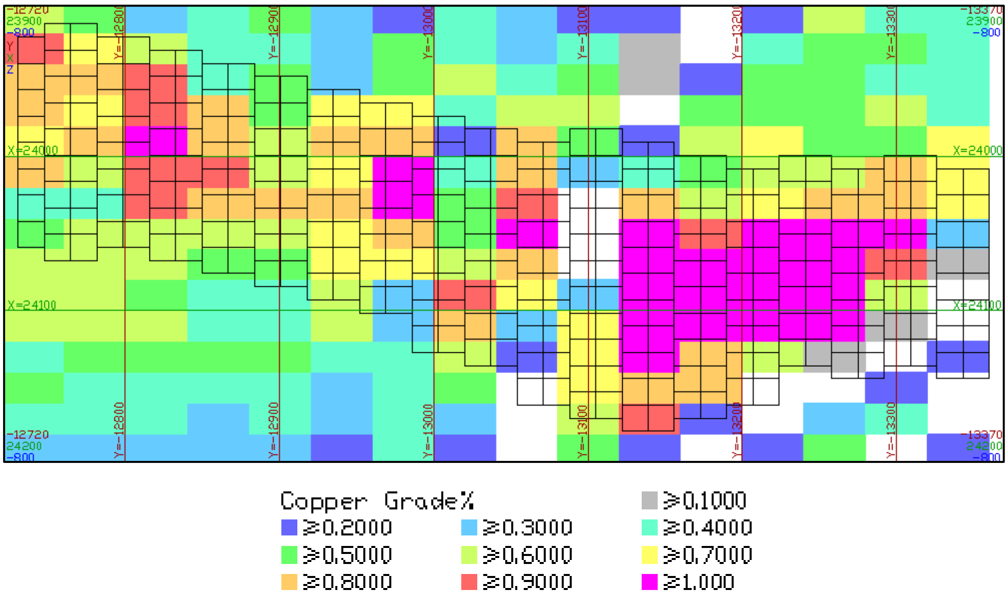


Figure 39: The dolerite dilution of the FS copper grades
Source: (Howson, 2004c)

Howson (2004c) reported that an M.Sc. student Joerg Neff completed the mapping model, but there was no full updated version conducted due to personnel movements. In the plotted Figure 40 below, a comparison shows the undercut level with mapped dolerite dyke contacts in blue dots corresponded partially with the green intersections of the NDM TIN dolerite envelopes (Howson, 2004c).

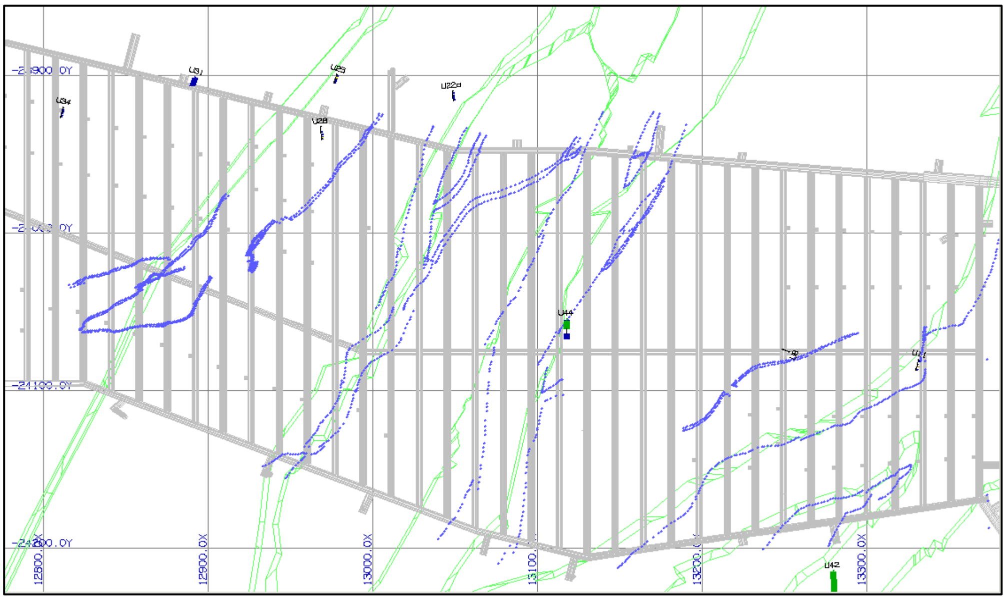


Figure 40: Mapped dolerite dykes versus the NDM tin envelopes
Source: (Howson, 2004c)

The above comparison indicated inaccuracies in all the dykes, and some dykes like the far left dyke showed narrowing or irregular stringers, which are complicated to present in the model. The dolerite dilutes the grades sampled from ROM ore, and for comparison, the FS model grades required the same dilution by the modelled dolerite (Howson, 2004c). In all simplicity, most updated geological information is a requirement in any current applied model. Howson (2004) stated that the created dolerite model achieved only a partial completion up to the -720 m elevation in the GEMCOM system. This partial model was according to the mapping and thoroughly revised. The revision enabled the assignment of the dolerite percentage value per block from these envelopes and Howson (2004a) titled this model as the “Draw Block Model”.

5.6.5 The “Draw Block Model” (DBM)

The FS block model configuration indicated it was not ideal for comparison with production figures (Howson, 2004c). In facilitation of the reconciliation, a revised model was required from the FS model data, having blocks coinciding with the draw point intervals, namely the “Draw Block Model” (Howson, 2006). To determine the dolerite quantities within each large block, Howson (2006) involved the smoothing of information. The second stage of smoothing was required to determine the dolerite in the draw columns from the miss-matched blocks (Howson, 2004c). Similar to the updating from mapping data, the block configuration changes involved coinciding blocks in the model with the draw columns’ positions. This change formed part of the geographical change PMC’s model required.

The DBM parameters and configuration entailed the following:

	X (Easting)	Y (Northing)	Z (Elevation)
Number of blocks	44	35	36
Block sizes	17 m	8.5 m	10 m
Gemcom Min Coordinates	12646	-24195	- 840
Gemcom Max Coordinates	13394	-23897.5	- 480

The copper and all other grade updates followed by intersecting each DBM block with its appropriate FS model block. Consequently, this enabled the determined grades in the DBM blocks through the application of the weighted mass means of grades in the intersecting volumes. This calculation itself involved the smoothing of the FS grades (Howson, 2004c). However, Howson (2004c) stated that the latter smoothing process would replicate the smoothing of draw column grades determined from the FS model’s blocks at an insignificant level. In Figure 41 below, the plot indicates the dolerite diluted copper grades in the DBM at -800 m with more significant improvements noticeable than the plot in Figure 39.

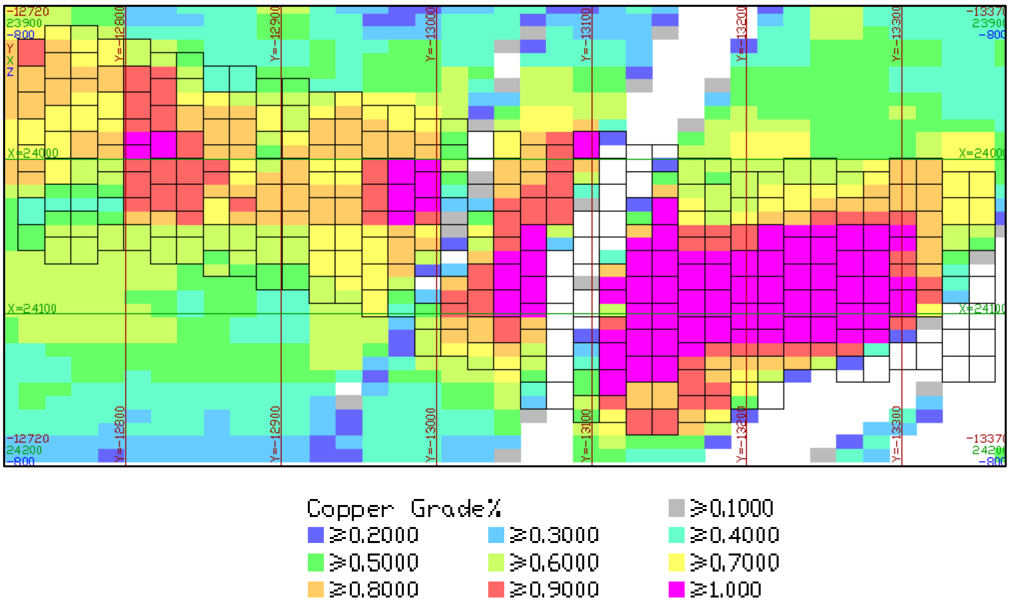


Figure 41: The dolerite diluted copper grades in the DBM model
Source: (Howson, 2004c)

The comparison of the DBM model plot, with relation to the other plots, illustrated the draw columns coinciding with two model blocks, including the position and grade influence the main dykes had. Therefore, the second stage of smoothing was not required, and the reduction of the block size caused a further lateral and vertical smoothing reduction in the first stage's dolerite determination (Howson, 2004c).

5.6.6 The "Draw Column Model" (DCM)

The DCM followed from the DBM by a simple combination of values in pairs of adjacent blocks, which fell within the same draw column (Howson, 2006). The resultant array of blocks in plan-view, plot blocks at an offset to the North-South columns. The DCM in the database provided grades for reconciliation purposes. The offset North-South columns rendered the DCM inappropriate for use in mine planning programs, where the DBM was more suitable for such use (Howson, 2006).

When comparing Figure 42, Figure 43, and Figure 44, the DCM indicated a variation of copper and dolerite grades with depth.

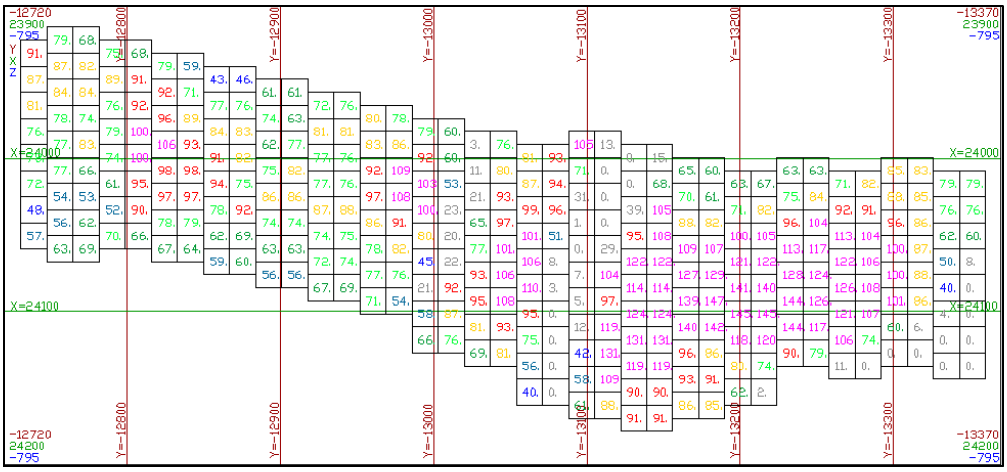


Figure 42: Copper percentage x 100 grades from the DCM at level –795m
Source: (Howson, 2004c)

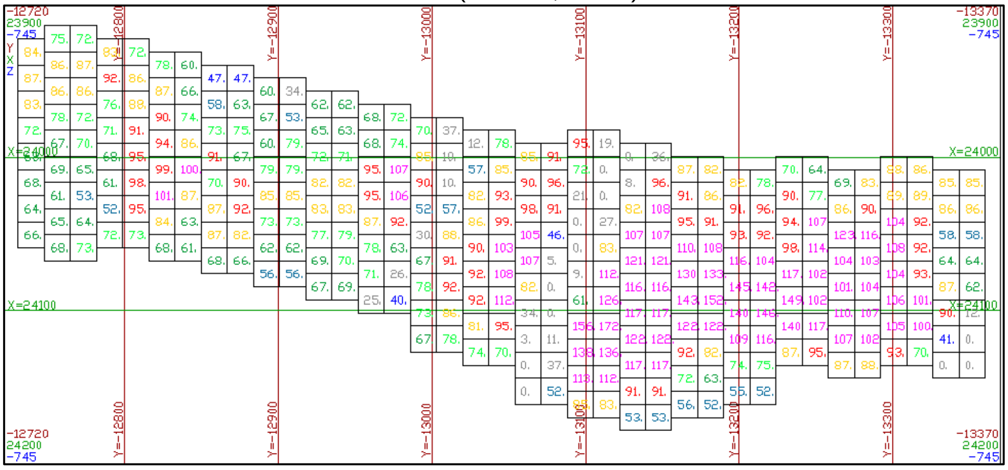


Figure 43: Copper percentage x 100 grades from the DCM at level –745m
Source: (Howson, 2004c)

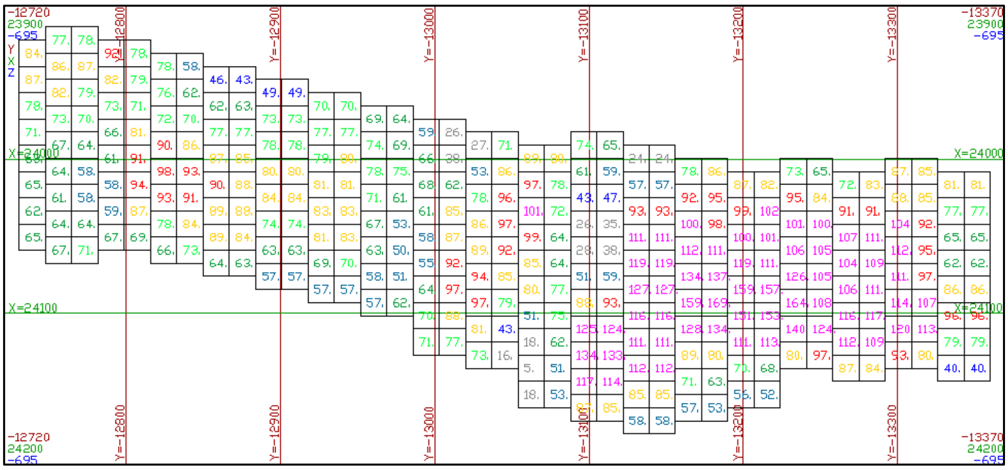


Figure 44: Copper percentage x 100 grades from the DCM at level –695m
Source: (Howson, 2004c)

An investigation of the above plots resulted in an additionally created graph as per Figure 45 which indicates the variation in the dolerite and copper grades with their respective elevations for all draw columns.

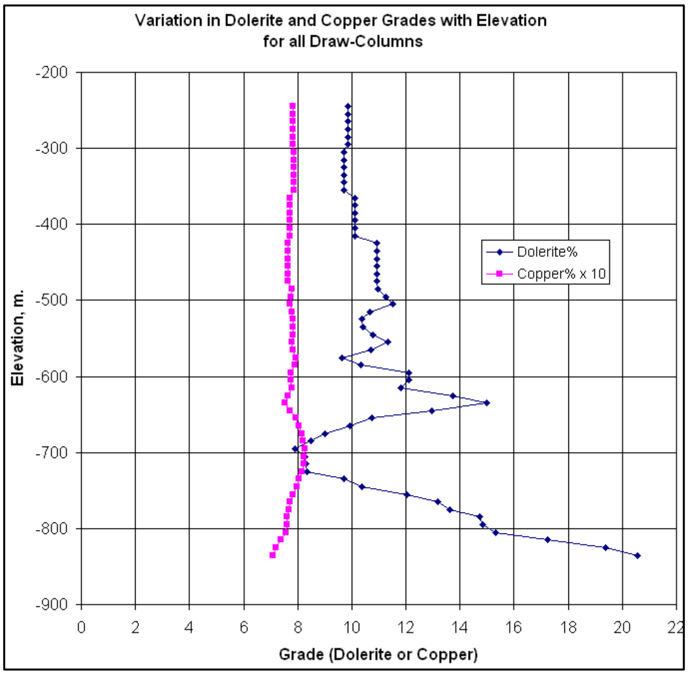


Figure 45: Variation in dolerite and copper grades with elevation for all draw columns
Source: (Howson, 2004c)

Although the undercut base is at about -800 m and the bottom of the open pit above lies at about -300 m, Howson (2004c) advised in his report that the interpretation of the graph above was problematic. It may appear that an expected grade at the start of production was well represented and indicated by the “Copper % X 10” trace just above the -800 m elevation. This interpretation of the graph was however not correct since the historical reports indicated that more mining of material occurred at the centre of the block cave (Howson, 2004a). At the centre, the grade was usually higher in the earlier years, and that mining reports should have indicated significantly higher grades mined. In later years, Howson (2004c) expected a decrease in grade, when more ore drawing from the periphery during production would follow.

5.7 Draw point production figures

The captured Excel production records from each of the 324 block-caving draw points and the number of LHD buckets counted per shift for each drawing point was on the PMC network. Howson (2003) and the team indicated that from their understanding during the early production phase, LHD operators reported their numbers from memory recollection after shifts until mid-December 2003. A new dispatch system resolved the inaccuracies from such memory recall process (Howson, 2006).

The LHD bucket factor to convert buckets into tonnage in 2003 rose from the initial 8.6 tonnes per bucket to 9.7 tonnes in 2004. During the study conducted from 13 to 30 January 2004, the backdated number of kilotons drawn from each drawing point from the start of caving was 9300 kt more until 12 January 2004 (Howson, 2004c). The updated and plotted production figures for the PMC cave shown below in Figure 46 is as per colour-coded ranges between 10 kilotons.

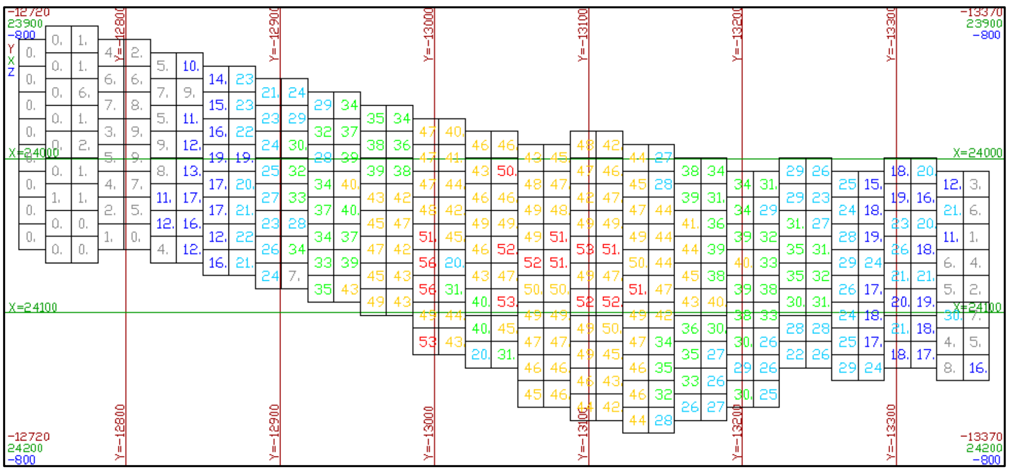


Figure 46: Production figures in kilotons from the start of caving until 12 January 2004
Source: (Howson, 2004c)

In a review of the above plot, with the previous DBM dolerite diluted copper grades, it is apparent that the highest tonnes produced were from areas with high dolerite. This indication indicated that more production from higher dolerite areas substantially accounted for the low grades achieved in 2003 (Howson, 2004c).

An additional plot from a trail of tonnes drawn with specific results between 13 and 30 January 2004 shown in Figure 47 below indicated some detected inaccuracies per kiloton. These numbers are not from the dispatch system but came from the operators' handwritten lashed sheets (Howson, 2004c).

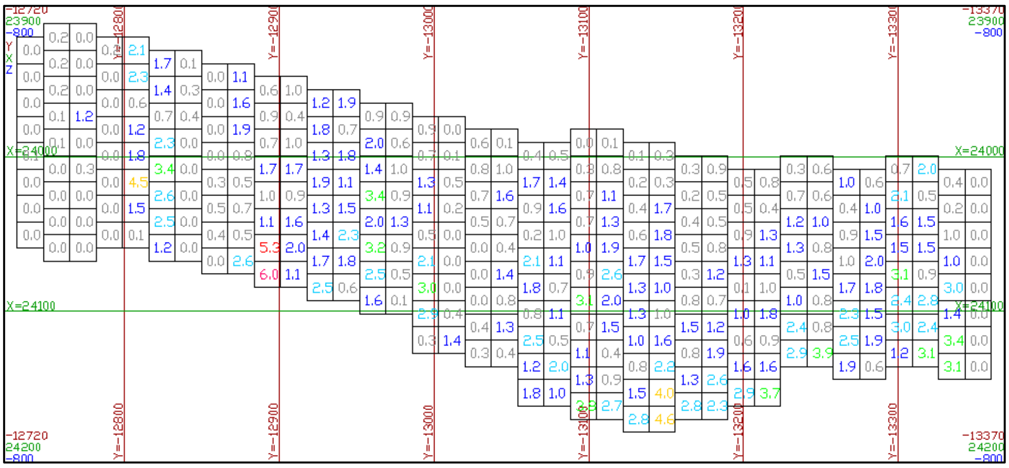


Figure 47: 1 Kilotonne LHD operator lashing inputs from 13 to 30 January 2004
Source: (Howson, 2004c)

The former inputs from the trial study are to be very accurate in excess bearing dolerite crosscuts, due to dolerite control during the indicated trail (Howson, 2004c). A noticeable inaccuracy such as 1.2 kt reported from the wrong draw point on the west side of the mine reflected in the third column from the left which was concerning. An average calculated tonnage factor for all rock in every draw column between the - 800 and – 720 m elevation was 0.328 m^3/t (Howson, 2004c). Each draw column has an area of 17 m x 17 m, and hence a metre of the column contained on average 878.56 tonnes. Thus, a thousand tonnes drawn from a draw point represented 1.14 m of the draw column height as an average figure (Howson, 2004c).

It was concurrently and theoretically possible to determine how far up in meter units each draw column of the original in-situ rock the quantity reports from. The comparison between the given quantity taken from a draw point and the previously known tonnages lashed at the same draw point was analysed (Howson, 2004c).

The referred exercise above took a variable base elevation of 18 m above the draw point floor and calculated the elevation of the original in-situ rock for each draw column. The conducted exercise followed for the dates between 13 January start of shift up until 30 January end of the shift. The results produced a defined volume in each column, from which an extracted grade value reported from the block model (Howson, 2004c).

The tonnages drawn from 13 to 30 January 2004 indicated that the mass weighed mean grades for the total mined volume was 12.5% dolerite by volume based on the DBM. The copper grade resulted in 0.78% for the same period of mining (Howson, 2004c). Daily tonnes drawn during this trial period did not vary significantly with a mean of 20.1 kt while the minimum and maximum values calculated to be between 17.1 kt and 22.7 kt respectively (Howson, 2004c). The determined grades for these specific tonnages and resultantly from the DBM as per the graph in Figure 48 below indicated a relative constant copper production grade.

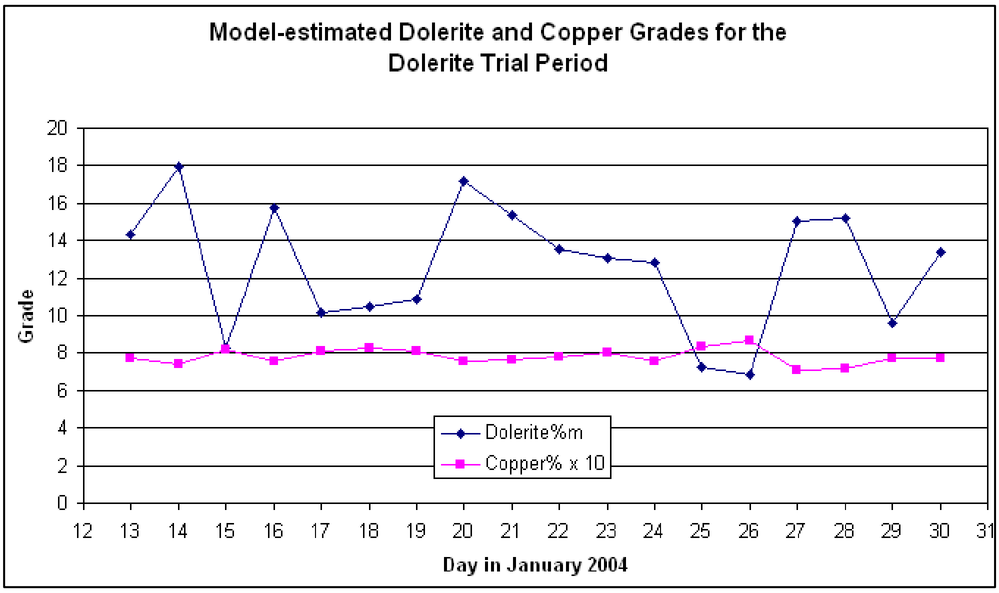


Figure 48: Model-estimated dolerite and copper grades for the dolerite trail period
Source: (Howson, 2004c)

5.8 Conclusion

During the resource model grade and head grade investigation, Howson (2004c) indicated that the computer systems in use by PMC posed a stability risk regarding data losses. If the old PC encountered any system or hardware issues, the available data could be irrecoverable. Alternative acquired systems ensured that data transfers realised with exceptional care, and maintained their accuracies and integrity. The transformation of coordinate systems, and different created models for dolerite, grades, and draw blocks formed part of the conversion. The data conversion exercise was successful according to Howson (2004c).

6 THE GRADE RECONCILIATION STUDY (2004-2005)

6.1 Introduction

This section analysed the reconciliation study, conducted by the technical teams and Howson (2006). The grade reconciliation study aimed to determine how well the production grades corresponded with the extracted grades from the resource model. A good correspondence would conclude an adequate representation of the material that was minable in future. Ultimately, such representation will indicate reliable predictors for future production from the resource and reserve estimates (Howson, 2006).

6.2 Mill production Cu grades versus mucked Cu grades from the model

Graphs indicated the comparison between production tonnages and grade figures, from supplied PMC concentrator monthly spreadsheets. The mining section supplied figures for mucked tonnes from each drawing point per day. These draw point positions determined the location of the grade in the model in the horizontal plane (Howson, 2006). The total tonnes mucked from each drawing point determined the material located in the vertical direction. In the early stage of the block cave, the presumably drawn material situated above the draw point in its 17 m x 17 m draw column will arrive and report at the draw point in an orderly sequence during drawing (Howson, 2006).

A database query determines how far up the respective draw column rock reports as mucked tonnes, including daily mucked tonnes from its draw point as per its original in-situ status. For each tonne pulled from a draw point, the 17 m x 17 m draw column descended on average about one millimetre (Howson, 2006). The column heights placed above the draw column base's elevation determined the elevation of the original in-situ rock. These elevations, with their horizontal coordinates of the draw point, defines its spacial point. A software query determines from the blocks in the DCM, which block encloses this point in space. The stored grades in this block consider the value of the resource model's estimated grade of the in-situ rock (Howson, 2006). The mean grades are daily determined and assembled with the mucked tonnes as per compiled spreadsheet. In conclusion, a value comparison between the production tonnes, the corrected copper, and dolerite grades computes within the spreadsheet (Howson, 2006).

6.3 Grades through time

The comparison of copper mill production grades with the same grades estimated from the resource model, and according to the tonnes mucked from the daily-recorded draw points, reflected differences as is illustrated in Figure 49 below (Howson, 2006). Note that the graph included smoothing by applying a five-day tonnage weighted average. The graph's starting date was 1 December 2003 but included some previous ore obtained from the open pit through ramp scavenging. This additional ore consequently eliminated a clear comparison (Howson, 2006).

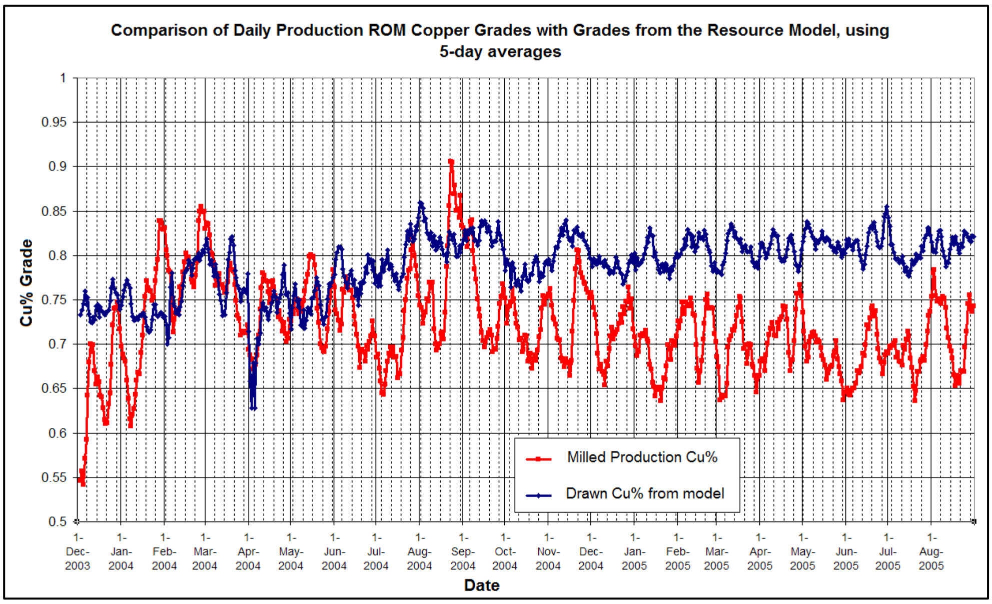


Figure 49: Smoothed ROM vs estimated copper grades through time
Source: (Howson, 2006)

In the above comparison, the mucked copper percentage from the model indicated a higher copper percentage grade than the mill production copper percentage. The strong cyclicity in ROM grade shows an amplified representation in the mill production and Howson (2006) ascribed this to bias in the mill feed assays while the mucked copper grade seemed overestimated. Similar to the produced graph above, an additional constructed graph followed for dolerite grade percentages, based on silica grade percentages (Howson, 2006). Note that the milled production dolerite percentage only started in January 2004. At the time, routine analysis for silica in the mill feed started, enabling the determination of dolerite in the mill feed as per illustrated graph in Figure 50 below (Howson, 2006).

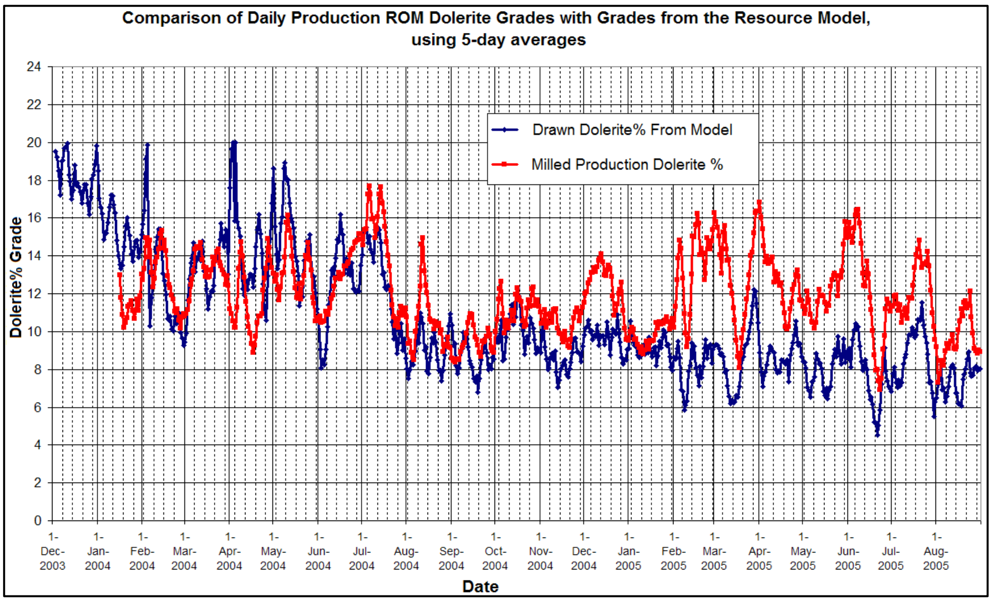


Figure 50: Smoothed ROM vs estimated dolerite grades through time
Source: (Howson, 2006)

In Figure 49, from the middle of 2004, it was apparent that the copper mill production grades were significantly less than the corresponding grades, as was estimated from the FS resource model. In Figure 50, the mill production dolerite grade- estimates from silica grades were more than the corresponding grade estimates from the dolerite model (Howson, 2006). From the above, Howson (2006) indicated a copper grade shortfall and suggested additional testing to establish whether it resulted from dolerite dilution as was expected. In Figure 51 below, the compared copper grades with the influence of dolerite removed reflected that the mill production undiluted Cu percentage was much lower than the mucked model having undiluted Cu percentages. The aforementioned compared grades, which Howson (2006) termed “un-diluted copper grades”, represented the copper grade of non-dolerite rock for both the mill and the FS model.

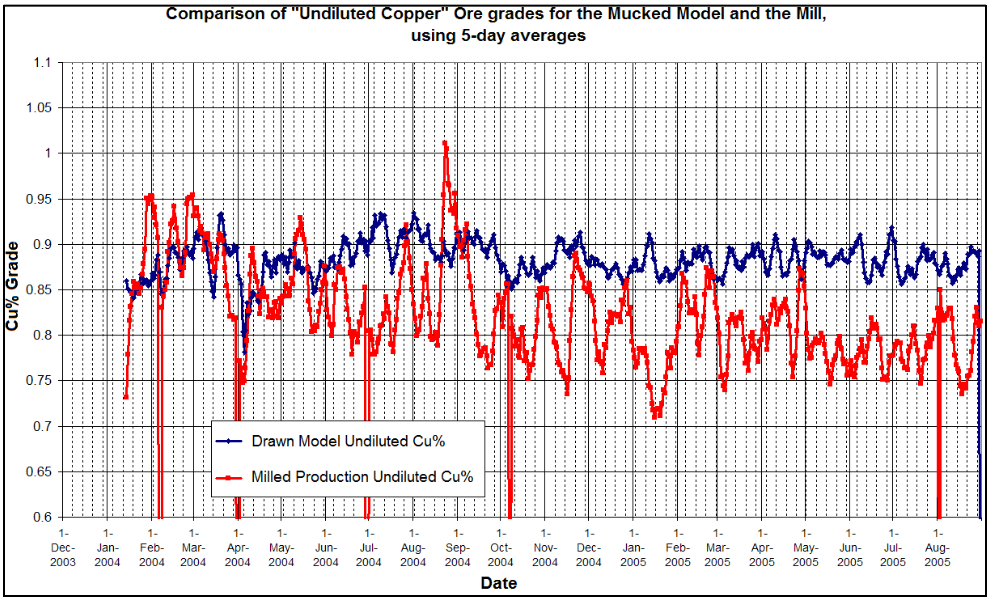


Figure 51: Smoothed ROM vs estimated undiluted copper grades through time
Source: (Howson, 2006)

This grade discrepancy deepened as per indication in Figure 51 above where the mucked and milled undiluted copper percentages parted from each other. The second half of 2004 and the entire 2005 indicated the un-diluted-dolerite copper grade of the ROM ore at 10% less than the equivalent grade prediction in the model (Howson, 2006). An intensive investigation done on the shortfall only eliminated one of the fundamental reasons such as biased assaying. More detail on the discrepancy and some possible hypotheses from the investigation follows in section 6.5.

6.4 Reserve estimations and reserve classification (2004-2005)

In July and August month 2004 Howson (2006) and the technical team compiled a section on the reserve estimates as was updated by PMC. It served as a record of conducted work, required for a possible repeat, once a significant re-estimation of the mineral resource model becomes necessary again (Howson, 2006). The reserve re-estimation looked at the mine reserves during its operations and reflected on changes that took place since the earlier FS estimation of reserves.

Howson (2006) indicated that the changes that took place, which affected the ore reserves since the FS, were:

- The deepening of the open pit beyond the planned FS depth and ramp scavenging,
- The resource model revision with the re-interpretation of dolerite from underground mapping,
- The draw bell layout revision, and loss of row “W” at the western end of the footprint, the addition of “A1” and changing the initially planned number of draw bells from 172 to 166,
- The ore depletion from the block cave production until the end of 2003,
- Depletion from the block cave production since January 2004.

6.4.1 Reserve volumes

In Figure 52 below, the 3-D image illustrates the volumes modelled during the FS, to determine the block cave reserves. The illustration indicates the volume in green for the planned final bench perimeters of the open pit, not forming part of the block-cave reserves (Howson, 2006).

Figure 52 illustrates the block cave reserve as summarised blocks from one to thirteen. The planned 172 draw bells shown in yellow are from block number one. Block number two comprises of the undercut, which is a sub-horizontal 4 m high layer in purple and just visible above the draw bells. The draw columns in deep blue above the undercut are the number three to number eight blocks (Howson, 2006). The turquoise nested volumes which surrounds the draw columns shows a truncation above by the open pit benches and forms part of blocks nine to thirteen (Howson, 2006).

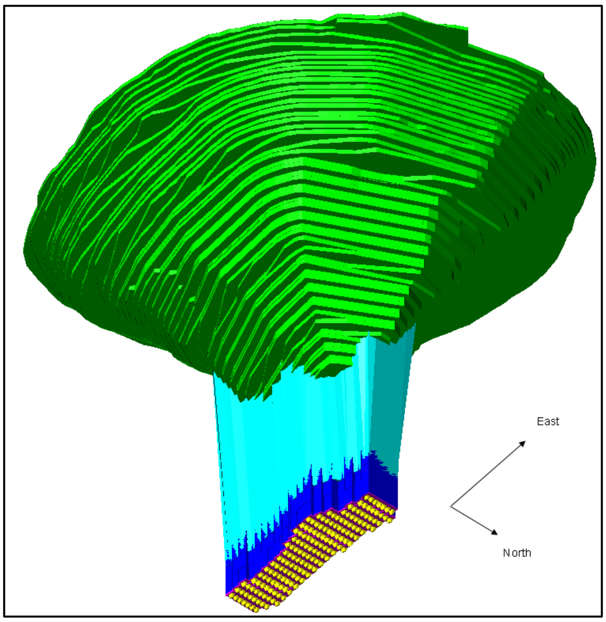


Figure 52: A 3-D image of the block caving reserve volume
Source: (Howson, 2006)

6.4.2 Reserve classification

PMC required that its ore reserves be compliant to the JORC (Australian Joint Ore Reserves Committee) code classification. However, the FS did not include the introduction of the code. The ore reserves were termed "Minable Reserves" and classified the vertical block numbers from one to eight as 98.56% "Proven" and the dilution envelopes from nine to thirteen as 82,84% "Proven" (Howson, 2006). The FS classified the percentages used as an unrealistic impression of the precision that was possible and could not state the geological uncertainty or the risk of these estimates. The FS also classified only reflected geological risks including transferrals from the resources as measured and indicated. The FS never included the mining risk (Howson, 2006).

In the JORC code, the measured resources transferred to the proven reserves, indicated resources as transferred probable reserves. In cases where a low level of geological risk exists, with a relatively high mining risk, the code allowed for transferrals of measured resources to probable reserves (Howson, 2006). Thus, the FS reflected correctly, but a reclassification by the JORC principals was required. With a low geological risk in the FS, the material in these blocks reflected almost an entire classification within the “proven” category. The acceptable reconciliation results indicated confidence in the estimation of this material with consistency in the “measured resources” classification (Howson, 2006). The given mine plan included drawing of all the draw points, beyond their vertical extent, and it was thus very likely that virtually all material above the draw points will eventually report to the draw points. Adverse “chimneying” effects could delay some material and dilute or replace material from the surrounding dilution envelopes in a negligent manner. Hence, the mining risk remained low (Howson, 2006).

The dilution envelopes of blocks nine to thirteen needed reclassification to report as probable reserves. An overall level of geological risk at the FS for this material was higher than for the vertical blocks. In considering the geological risk alone, this material still needed their classification as measured resources and required a transfer to proven reserves (Howson, 2006). The resource estimation as a whole implies some reduction in the geological risk from the acceptable reconciliation results. However, a significant mining risk exists in the extraction of the dilution envelopes. An expectancy existed from the plan that material will travel towards the draw points. During rock slope failures, these vertical blocks’ material flow subjects to a series of design caving angles (Howson, 2006). On the other hand, failures of the material may not happen as per planned predictions, and may not arrive at the draw points at all. In some areas, the material may fail at much lower predicted design angles. Failures can result in much lower grade material or as waste from the periphery of the open pit moving towards the draw points, ahead of the dilution envelope material (Howson, 2006). Lower grade material could preclude the extraction of higher-grade material to follow. Howson (2006) hence indicated that the uncertainty in the way that the dilution envelopes will fail implied a significant mining risk, which required a reclassification to be probable and not proven reserves. Only with greater certainty on the models of failure, upgrading of some of the reserves to proven reserves was possible (Howson, 2006).

The JORC code required resources to have “reasonable prospects for eventual economic extraction”. At the Palabora Mine block cave, any mineralised material, which lies outside the existing reserve envelope, may have such prospects; else, they would be within the reserves. Any additional resources to the reserves were concurrently not reported (Howson, 2006).

6.5 Reasons for lower than expected copper grades

During 2004 and 2005, the measured production grades at the mill were lower than the estimated FS grades. The “Grade Reconciliation Study” indicated that the mill feed contained about 10% less copper that may be predicted from the FS model and as per the DBM’s indication (Howson, 2006). Further clarity was required since very little evidence was conclusive. However, Howson (2006) indicated that discussions and investigations transpired through their work. Notably, possible reasons for the discrepancy such as bias in the mill feed assays and appropriate corrections to calculations improved the discrepancy to a certain degree (Howson, 2006).

6.6 Summary for the grade discrepancy

Howson (2006) stated that there was no specific reason which contributed to the grade discrepancy Palabora experienced. However, several reasons which were rather significant in combination contributed to the grade discrepancy. The phasing of the FS caused a severe inadvertent problem in the use of the mineral resource model, and Hoswon (2006) considered the overriding reason for the discrepancy in the following ways:

- During phase 3 of the FS, a massive planned footprint area led to the drilling, modelling, and geological reviews from that assumption. The phase 3 mine plan's rejection followed by a reduction in the draw points (Howson, 2006).

- During phase 4, no further geological involvement occurred as the phase 3 model assumingly would suffice. The much smaller planned footprint focussed only on the higher-grade core of the deposit (Howson, 2006).
- The coarse variability of the deposit and local inaccuracies became much more significant via the adjustment to a smaller and more particular volume (Howson, 2006).
- The substantial reduction in the level of drilling indicated much of the phase 3 drilling resided outside the footprint. With the relative low horizontal grade continuity, the formerly excluded drilling was ineffective, to control the estimation of the minable grade (Howson, 2006).

6.7 Hypotheses concerning the FS copper estimation

The probability existed that the FS model overestimated the copper grade globally or locally. This overestimation concerned the volume mined during 2004 and 2005, which lies between the elevations -790 m, and -590 m within the footprint area. Howson (2006) noted at this stage that overestimation does not necessarily apply to the entire model. The following hypotheses explain the possibilities in short.

6.7.1 Drilling data density hypothesis

Drilling data densities indicated that the drill hole sampling of the smaller phase 4 footprint mining volume was inadequate in number and included the core diameter of holes. The drill-hole size is relatively small in the coarse nature of the mineralisation, leading to no small estimation variance (Howson, 2006). In Figure 53 below, the indexing finger points at Palabora's carbonatite copper mineralisation found at an oversized boulder in a draw point. The pale groundmass is carbonate, and the darker grey patches are magnetite. The copper mineral is bornite and occurs in purplish streaks at the finger and the two cap lamp beams. By sampling a 3.6 cm drill-hole from this rock, the sample is undoubtedly unlikely to be representative of the whole rock mass (Howson, 2006).

Figure 53: The mineralisation is of a very coarse nature
Source: (Howson, 2006)

6.7.2 Biased drilling locations hypothesis

Biased drilling locations found in the smaller footprint with manually targeted drilling locations have resulted in biased results due to irregular drill grid spacings. Howson (2006) noted that higher grades occurred from the central area when he projected such higher grades into intermediate grade areas of the deposit, and where there was no drilling.

6.7.3 Variable grade projection distance hypothesis

Variable grade projection distances indicated higher projected grades beyond the allowable limits in the estimation process. The FS estimation assumed that all grades in any one-grade zone possibly projected equally by a single anisotropic variogram model for that zone (Howson, 2006). Each zone included both high and low grades at a sampling scale and even smaller scale. The expectancy was that the variogram range should vary with grade. The assumed possibility existed that some of the highest grades had limited projection distances and one should avoid any possible projection and instead consider it as pure nugget mineralisation (Howson, 2006). The FS did not address that the estimation took variable ranges of high and low grades into account, and where the highest grades could only be projected a short distance if at all (Howson, 2006).

6.7.4 Grade zoning interpretation hypothesis

Grade zoning areas never identified risks inherent in construction via grade zones with limited data. It is not clear what the proportion of the discrepancy was due to grade zoning. A re-calculation of the resource model without the grade zoning may eliminate such undesirable effects, but it would remain subjective to the issue (Howson, 2006).

6.7.5 Non-consideration of density in grade estimation hypothesis

Howson (2006) noted that the density in grade estimation excluded the grade and density relationships during the estimation. On the other hand, variable density means that samples have different support regarding the sample mass. Despite the applied kriging to the grades, expressed as the mass of copper per unit mass of rock, the assumption was that all samples had equal support. However, the density still had a negative relationship with the biased copper grade estimation. In Figure 54 below, the scatter plot indicates the relationship between the density and copper grade, but without a clear correlation.

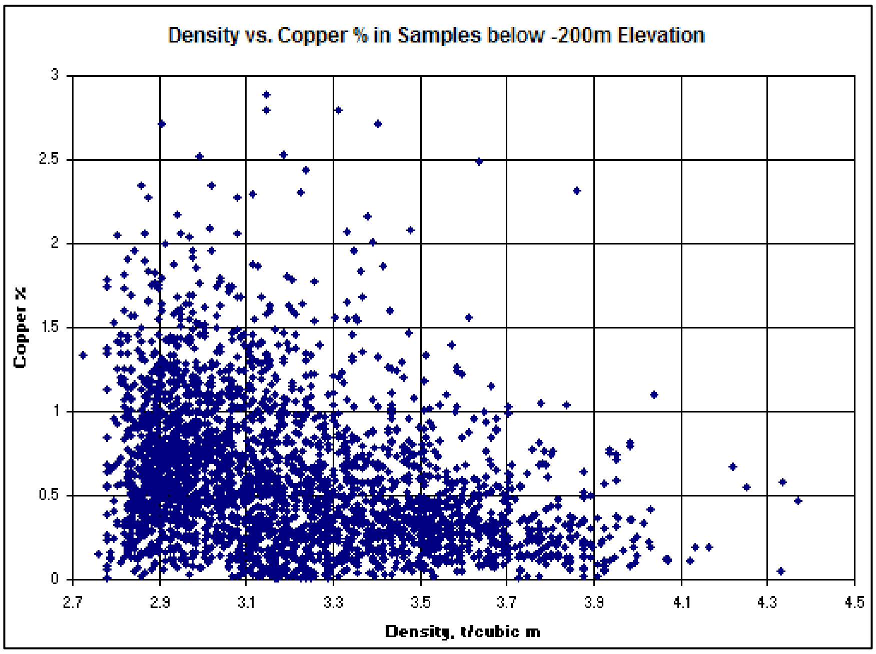


Figure 54: Density versus copper % in samples below -200 m elevation
Source: (Howson, 2006)

Howson (2006) concluded this aspect of the FS estimation positively identifies to be responsible for overestimation, although it remained a small percentage and it was far short of the 10% discrepancy.

6.7.6 Conclusion

The dolerite dilution reflected as being inadequate since more dolerite existed than what was predicted (Howson, 2006).

The above hypotheses relate to the estimation of the resource and the FS planning. The main lesson learnt is that any mineral resource estimation and project review in isolation to the mining or extraction plan is not advisable. It remains crucial that the mining plan consider the risks and potential effects of any possible inadequacies of the geological data it resulted from. The impact of a change in mining volume on overall resource quality required the earliest identification thereof (Howson, 2006).

6.8 Hypotheses not concerning the FS copper estimation

6.8.1 Dolerite dilution hypothesis

The copper grade was lower since dolerite was present in the Mill feed in higher proportions than planned in the FS. Howson (2006) reasoned that the dolerite proportion in the FS model was under-estimated, or dolerites preferentially gravitated towards the draw points. The higher than expected dolerite levels had a minor influence on the copper grade and was not accounting for most of the discrepancy (Howson, 2006).

6.8.2 Surface dilution Hypothesis

Early dilution from the open pit collapse reported at the draw points where low-grade material from the open pit walls, migrated downwards through the broken rock in the block cave to reduce the grade. The fast-moving fine-grained surface material appeared in more massive proportions than expected at the draw points. At the time of the investigation, there was no clear evidence from geologists to conclude this hypothesis yet (Howson, 2006).

6.8.3 Preferential secondary fragmentation of low-grade hypothesis

As secondary fragmentation occurred in the cave column, lower grade material being less hard, crush easier than the stronger higher-grade rock. Thus, the lower grade material in smaller particle sizes descended faster to report at the draw points much sooner. As there was no evidence on this yet, a recommendation by Howson (2006) followed to investigate more into this expectancy.

6.8.4 Variable bucket factor hypothesis

Tonnages drawn from draw points by LHD loaded buckets, varied in factor especially in higher than average at lower-grade draw points, and lower than average in higher-grade draw points. Where bucket factors varied between eight and eleven tonnes per bucket, the bucket factor seemed unlikely to be more than a minor contributor to the discrepancy (Howson, 2006).

6.8.5 Incorrect in-situ height hypothesis

The estimates of the FS model assumed a rate of descent of broken rock, which was constant for each tonne pulled across the 17 m x 34 m area attributed to each draw bell. The previous tonnes pulled from a draw bell determined how far up were the ore's in-situ location. When less than the full draw bell area descended, then the exact in-situ location was much higher up, causing a decline in grade which led to the discrepancy (Howson, 2006).

6.8.6 Draw control hypothesis

The dispatch system did not work adequately, in that lower-grade material was preferentially mined relative to the recorded data. It seemed that this material was more accessible to obtain, and resulted in "good" production rates from lower-grade draw points. At the time Howson (2006) indicated that this hypothesis seemed unlikely.

6.9 Conclusion

The grade reconciliation study involved several hypothesis studies, which investigated improvements of the resource model. The reserve estimates served as a framework during any required repeat or re-estimation. Since the FS most of the changes affecting the ore reserves formed part of the grade reconciliation and is accounted for during the Howson (2006) investigation. The correct resource classification by the JORC principals had to be implemented correctly and reviewed. The drilling data had unrepresentative errors due to the fragmentation and spatial distribution of mineralisation. Little correlation existed between the density and copper grades, and ultimately this contributed to overestimations in Cu grades.

7 THE RE-ESTIMATION OF COPPER GRADES

7.1 Introduction

In December 2005, a decision followed to re-estimate the copper grade model, from existing data in an attempt to explore and address some reasons for the discrepancy. The previous section discussed some details, which leads up to this section. The development of a model for block caving simulation which would resultantly produce more accurate copper grades, i.e. more closely correspond with grades obtained by mining remained the primary objective (Howson, 2006). The required model could then replace the FS model as the basis for reserve reporting and scheduling up to the end of the mine's life. For consistency and compatibility with previous work, the same model grid as for the DBM was used (Howson, 2006). A similar required procedure as per the FS had to address the un-diluted copper grade, ignoring the dolerite dykes. The applied existing dolerite model would dilute the copper grades in turn. The use of two groups of holes for the estimation included the surface drill holes labelled "LK" for "Loolekop" and "U" holes for underground (Howson, 2006). The block model dimensions of 17 m, 8.5 m, and 20 m, were approximately the simple multiples of two, one, and two times the composite length. Howson (2006) noted that this length is convenient for kriging discretisation.

7.2 Analysis of length versus grade

Figure 55 below shows a scatter plot of composite length vs copper percentage for all composites below -200 m and indicates a general decline in grade with increasing length.

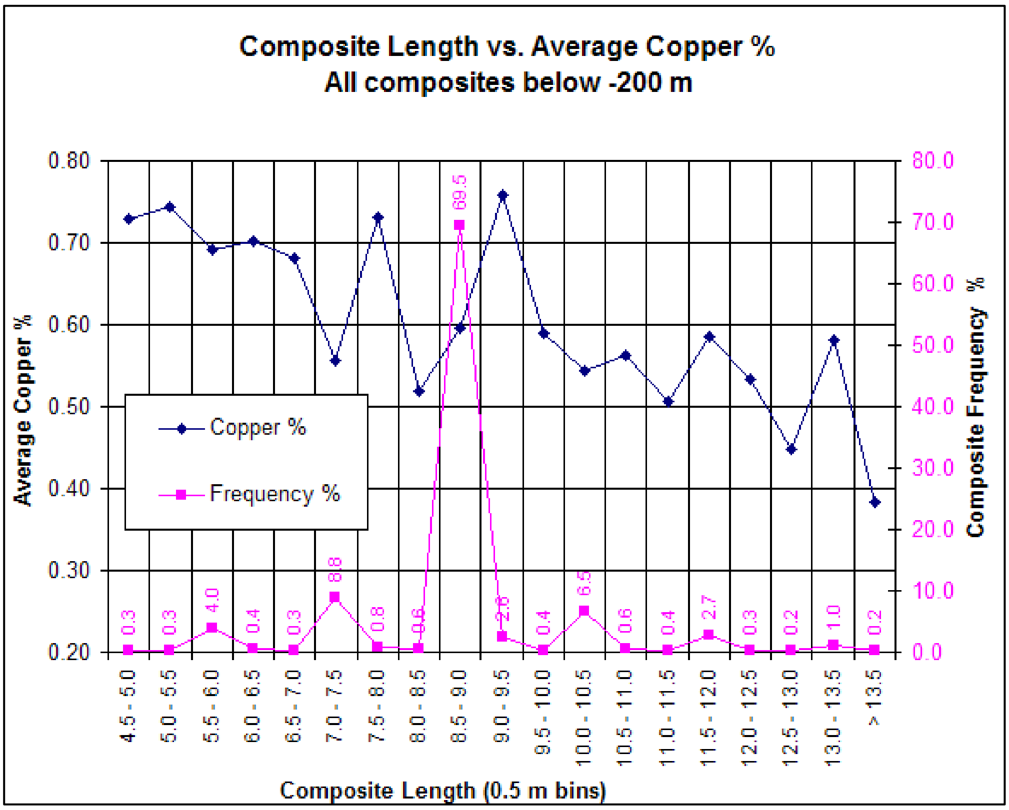


Figure 55: Composite length vs average copper %, for all composites below -200 m
Source: (Howson, 2006)

The above plot indicates a general decline in grade with increasing length, which upon interpretation indicated that the dataset was positively biased. The bias required testing to find the length-weighted mean of all the copper grades, which was 0.594% and compared with the simple mean copper grade of 0.597% (Howson, 2006). Howson (2006) indicated that the representation of 0.5% bias of the total contained copper, being probably too insignificant to warrant it further attention.

7.3 Distribution analysis

The three figure-plots of Figure 56, Figure 57, and Figure 58 are log-probability cumulative frequency plots of copper grades, copper x density, and density values (Howson, 2006).

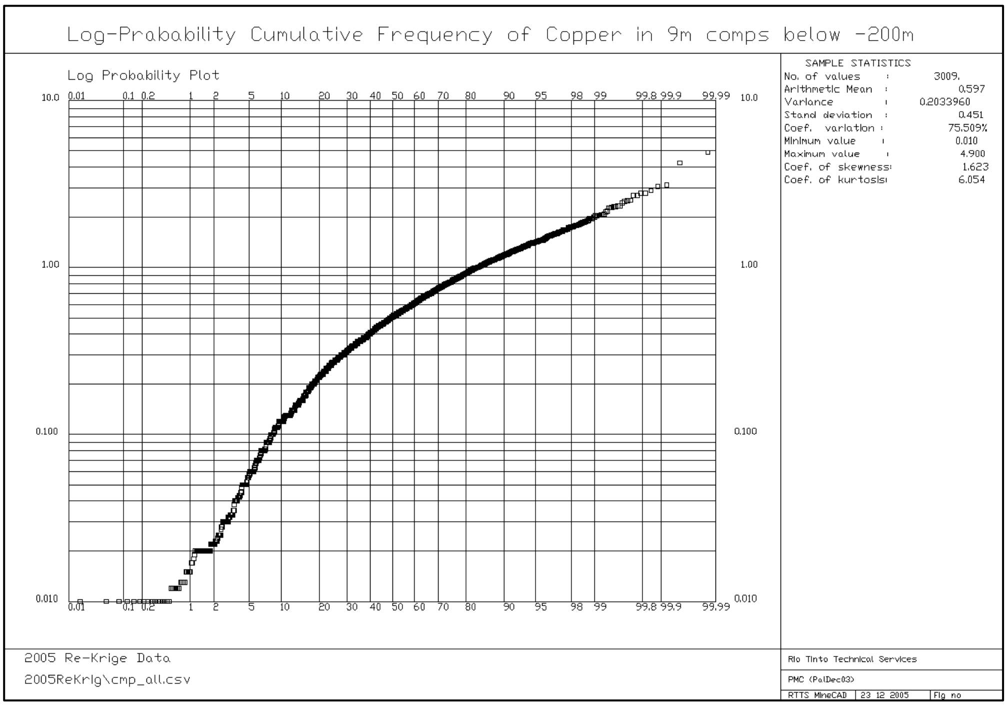


Figure 56: Log-probability cumulative frequency of Cu in 9 m comps below -200 m
Source: (Howson, 2006)

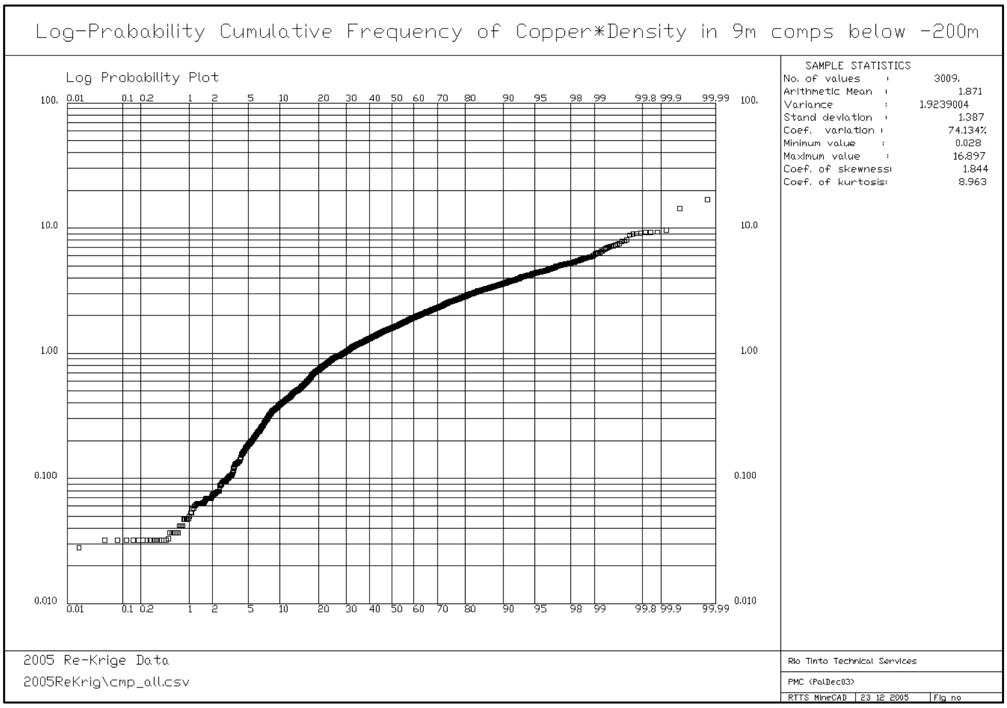


Figure 57: Log-probability cumulative frequency of Cu in 9 m comps below -200 m
Source: (Howson, 2006)

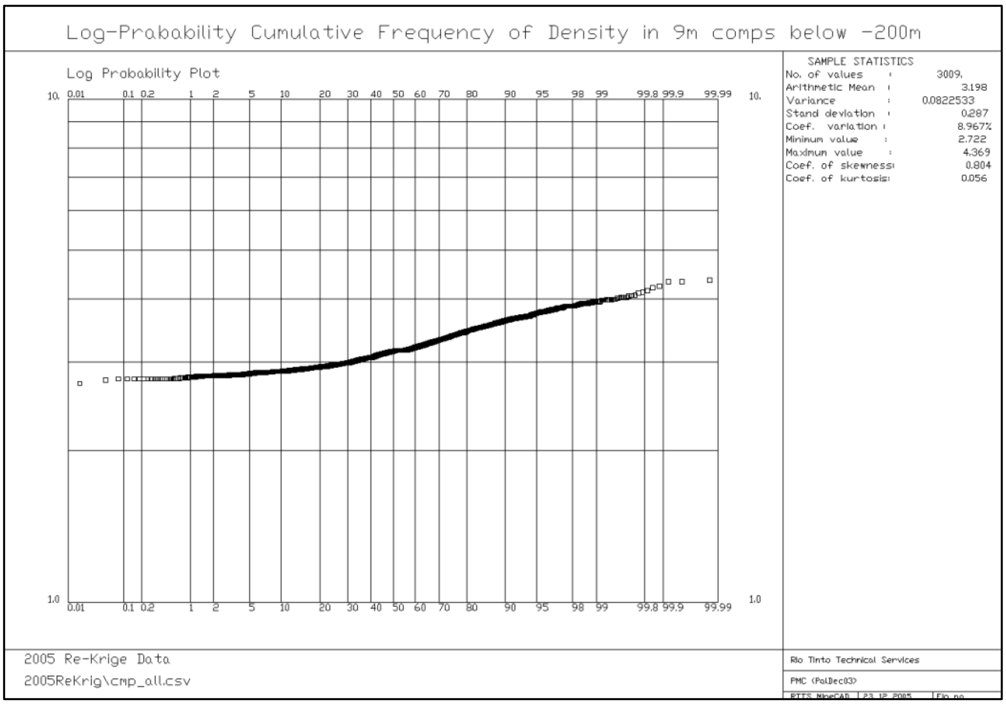


Figure 58: Log-probability cumulative frequency of Cu in 9 m comps below -200 m
Source: (Howson, 2006)

Howson (2006) used the log-probability cumulative frequency plots to analyse the component populations in the dataset. In general, a population of values that has a normal or lognormal distribution plot generally in a straight line (Howson, 2006). The changes in line direction indicated more than one population and reflected possible phases of mineralisation. All three plots have curves and tend to suggest that mixed population plots from the dataset (Howson, 2006). Two main populations of copper grades appeared with a break at 0.2% to 0.4% copper. The occurrence may be due to the remobilisation identified and was indistinct since the two populations would be requiring sampling at a much shorter interval than the 9 m. There was hence no suggestion that straightforward kriging of copper metal accumulation in a single domain would be inappropriate (Howson, 2006).

7.4 The 2005 ordinary kriging without grade zoning

The 2005 ordinary kriging (OK) model did not use grade zoning, to produce a straightforward model. The model estimated copper, as metal accumulation values, which calculated copper times density. This calculated estimation enjoyed preference above the grades. Howson (2006) firstly computed the separately estimated density, before applying it to produce a copper percentage model from the metal accumulation model.

The 2005 OK model gave very similar results in comparison with the FS copper grade model. Lower copper grades by about 1% in the volume reflected for volumes already mined and suggested that the grade zoning and grade-density reasons for the discrepancy were of minor significance. What the model did not address was issues such as data density, biased locations, or variable grade projection distances (Howson, 2006).

7.5 Re-estimation using multiple indicator methods

Within the first quarter of 2006, two applied variants of the Multiple Indicator Kriging (MIK) technique derived the Cu density which was copper multiplied by the density. In obtaining the copper grade estimates, the Cu density estimates required division by density as estimated previously without any need to apply MIK to density (Howson, 2006). Two variants of MIK were applicable namely: Multiple Indicator Ordinary Kriging (MIOK) and Multiple Indicator Simple Kriging (MISK). As a result, two commonly applied variants of kriging within the multiple indicator frameworks of MIK, MIOK, and MISK followed. MIK is a form of estimation in which different grade level estimates calculate independently. Indicators estimated for each model block were proportions of the block above a series of threshold values, preselected to give adequate coverage to the distribution of grades in the deposit (Howson, 2006).

The series of proportions, in turn, formed a cumulative frequency distribution curve for grades within the block. The cumulative frequency determines the overall combined grade for the block, constituting of the block estimate for this kriging method (Howson, 2006). In MIK, kriging of the series of indicators are independent, and indicators for each threshold can have their separate geostatistical parameters. Through this method, the accommodation of variation in variogram range was previously identified (Howson, 2006).

The graph of Figure 59 illustrates the change in variogram range with Cu density grades. The graph shows Cu density threshold values on the horizontal axis and variogram range on the vertical axis. The traces for the three-variogram axes are separately coloured.

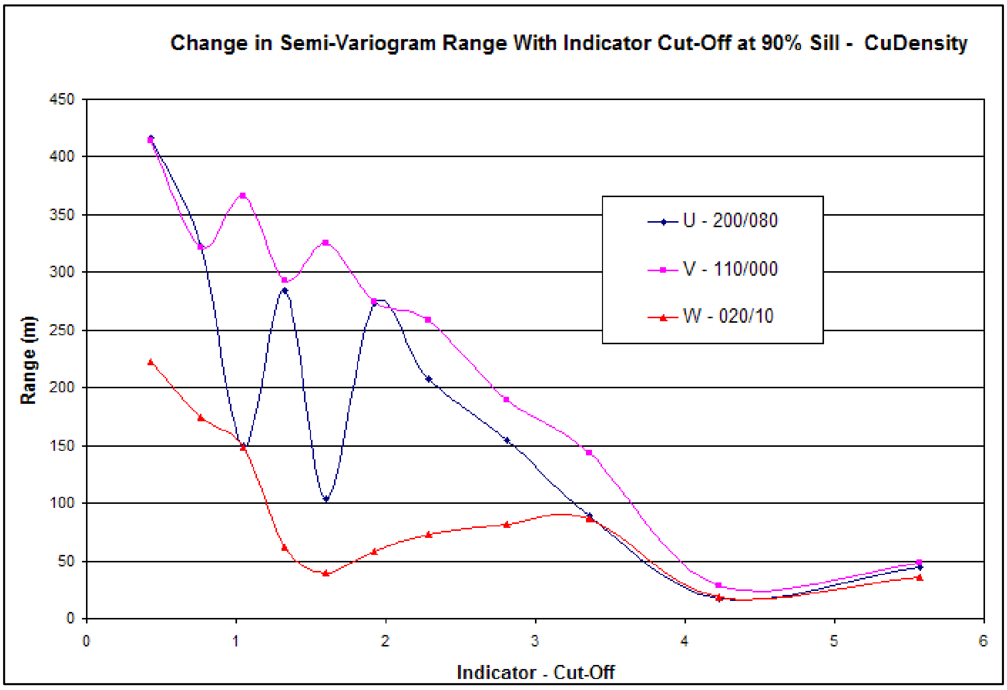


Figure 59: Change in variogram range with indicator threshold
Source: (Howson, 2006)

7.6 Comparison of estimates

7.6.1 Footprint copper grade estimates by elevation

The division of the output data from the geostatistical software 'Isatis', included Cu density values with the previously estimated density values to derive copper percentage grades. Howson (2006) compared these copper grades with adjustments in the models for dolerite dilution. The graphs in Figure 60 below compares the FS model grades from the DBM with those from the recent re-estimates. The comparison of the mean grade illustrations was from within the overall draw point perimeter, which is the footprint on a 20 m level basis (Howson, 2006).

The left graph below in Figure 60 indicates the FS model grades in dark blue, the OK grades in red, the MIOK grades in magenta, and the MISK grades in green. The right-hand graph indicates the percentage differences for OK, MIOK, and MISK in the same colours than those on the left.

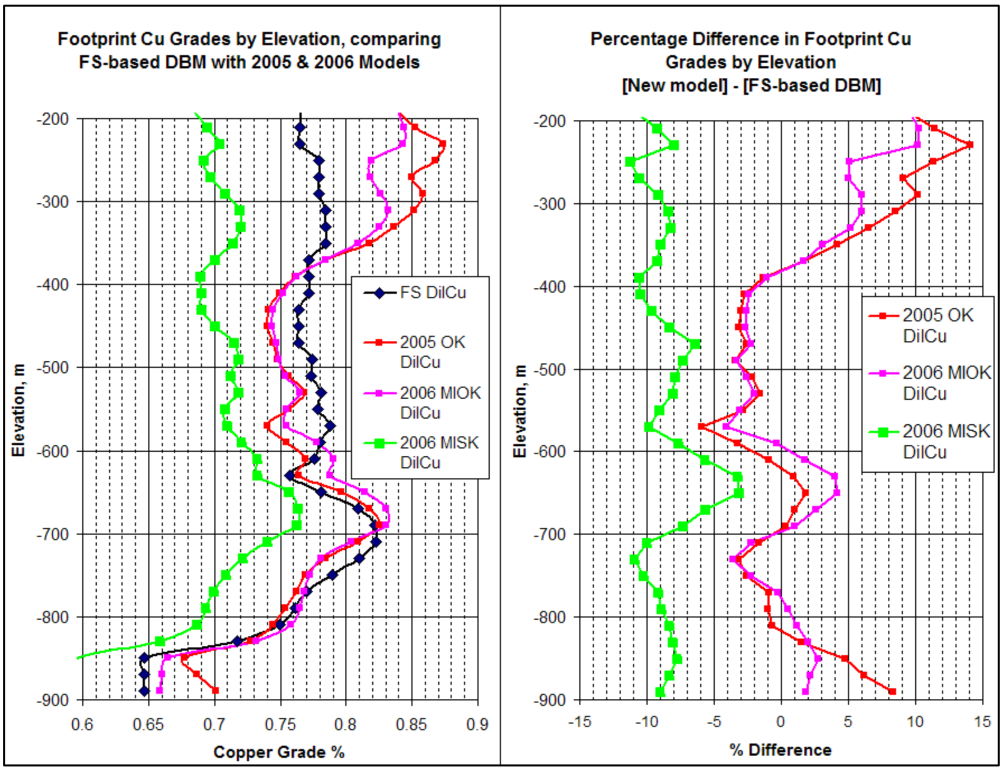


Figure 60: Comparison of various grade estimations by level
Source: (Howson, 2006)

A clear indication existed that those levels between -700 m and -800 m, the MISK estimates were about 10% less than the FS model. The OK and MIOK estimates were also lower, but only by about 2%. In conclusion, a significant reduction of the MISK relative to the FS model persisted for all the calculated levels (Howson, 2006).

7.6.2 Grade reconciliation study comparison with MISK

During the grade reconciliation study, Howson (2006) compared the exact production grades from the sampling at the mill, with possible predicted grades from the FS model. Resultantly, since mid-2004, the FS model grades overestimated the mill grades by more than 10%. When comparing the following plots in Figure 61 and Figure 62 with their direct counterparts in section 6.3, the outcome from the MISK model appeared to produce estimates, which were much closer to the production grades than that of the FS model. The FS was an OK model with grade zoning (Howson, 2006). Figure 61 indicates the compared mill copper grade in red with the grades from the MISK model in dark blue.

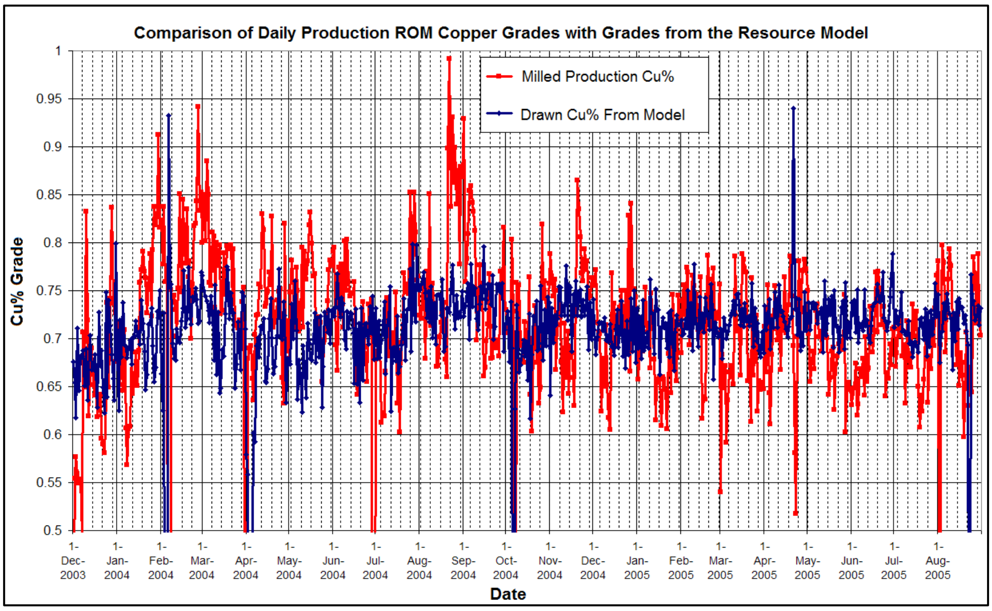


Figure 61: Daily production ROM vs estimated copper grades
Source: (Howson, 2006)

Figure 62 below shows the same data than Figure 61, where the latter used smoothed data via a 5-day tonnage-weighted average.

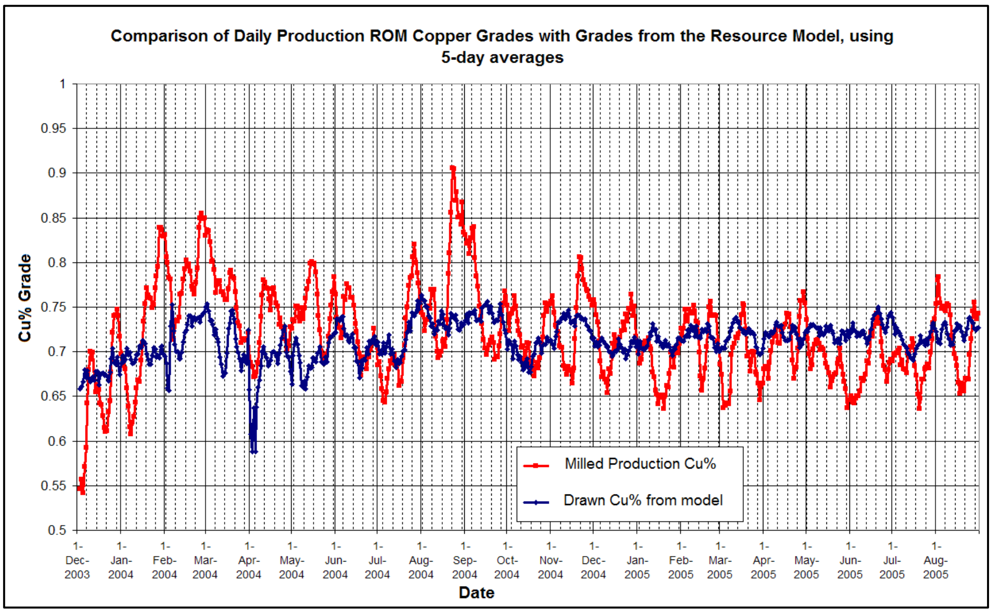


Figure 62: Daily production ROM vs estimated copper grades 5-day average
Source: (Howson, 2006)

The graph in Figure 63 below shows the variation in the mill and mucked model ratio through time in magenta. The black line is a 56-day moving average line.

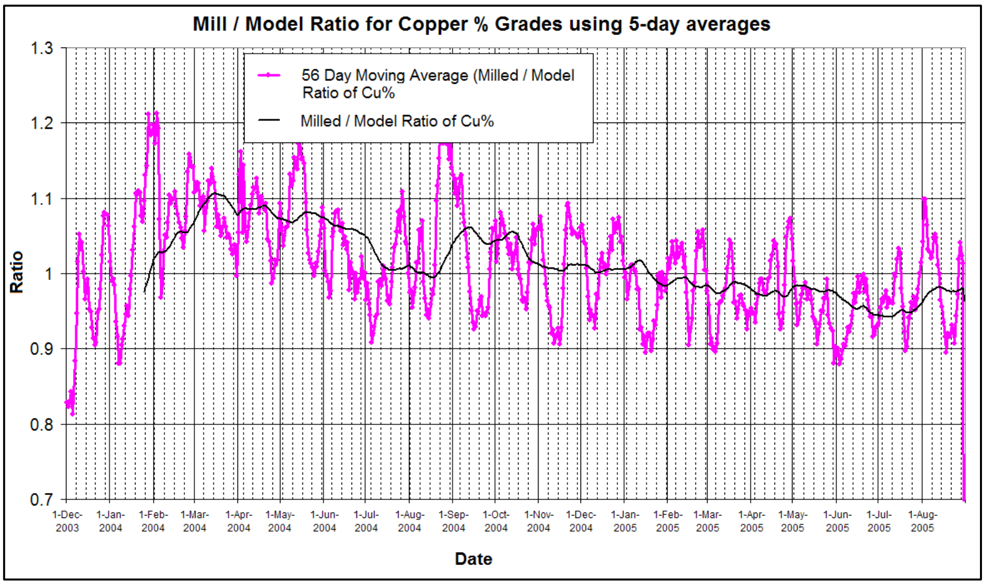


Figure 63: Mill / Model ratio for copper % grades using 5-day averages
Source: (Howson, 2006)

The ratio for the mill and mucked model reflects in the graph of Figure 64 below in magenta. The black line is a sixth-order polynomial trend curve.

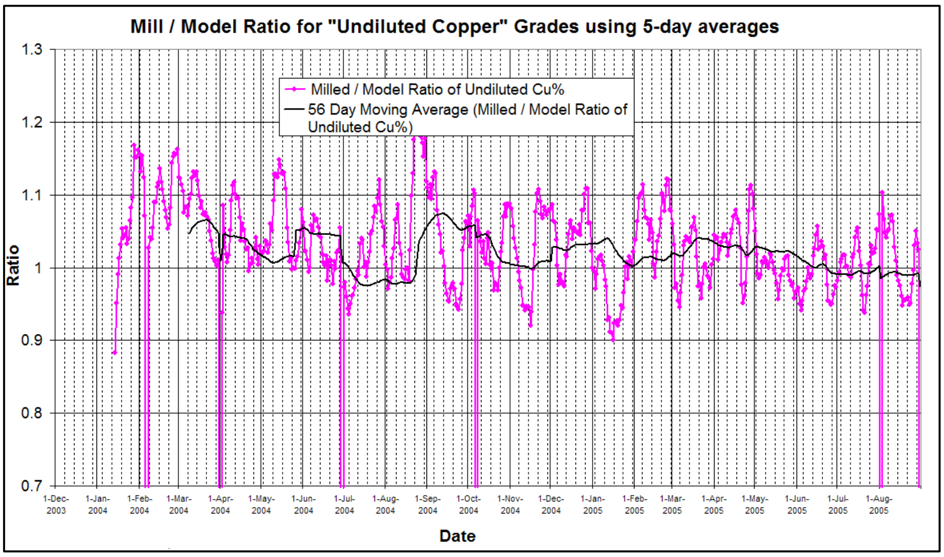


Figure 64: Mill / Model ratio for "Undiluted copper" grades using 5-day averages
Source: (Howson, 2006)

7.7 Conclusion

From the comparison of estimates, it was clear that the MISK model appeared to have produced estimates that were much closer to the production grades than either the FS model or the recently calculated OK and MIOK models (Howson, 2006).

Where only sparse data existed, OK and MIOK used techniques namely the "Lagrangian Multiplier" to extend the influence of point data such as composites beyond the variogram prescription. Within the Palabora footprint, the data was sparse, and a minor bias in the drilling occurred towards higher-grade locations (Howson, 2006). The effect magnifies by the widespread influence effect of OK to produce a model that appears positively biased by about 10% within the current mined volume. However, this applied to the FS, 2005 OK, and the MIOK models. Thus where sparse data existed, the shortfall in influence was made up by using the mean value of the deposit.

The graphs in Figure 65 compare production copper grades with corresponding grades predicted by the FS model from January 2003 to the end of August 2005.

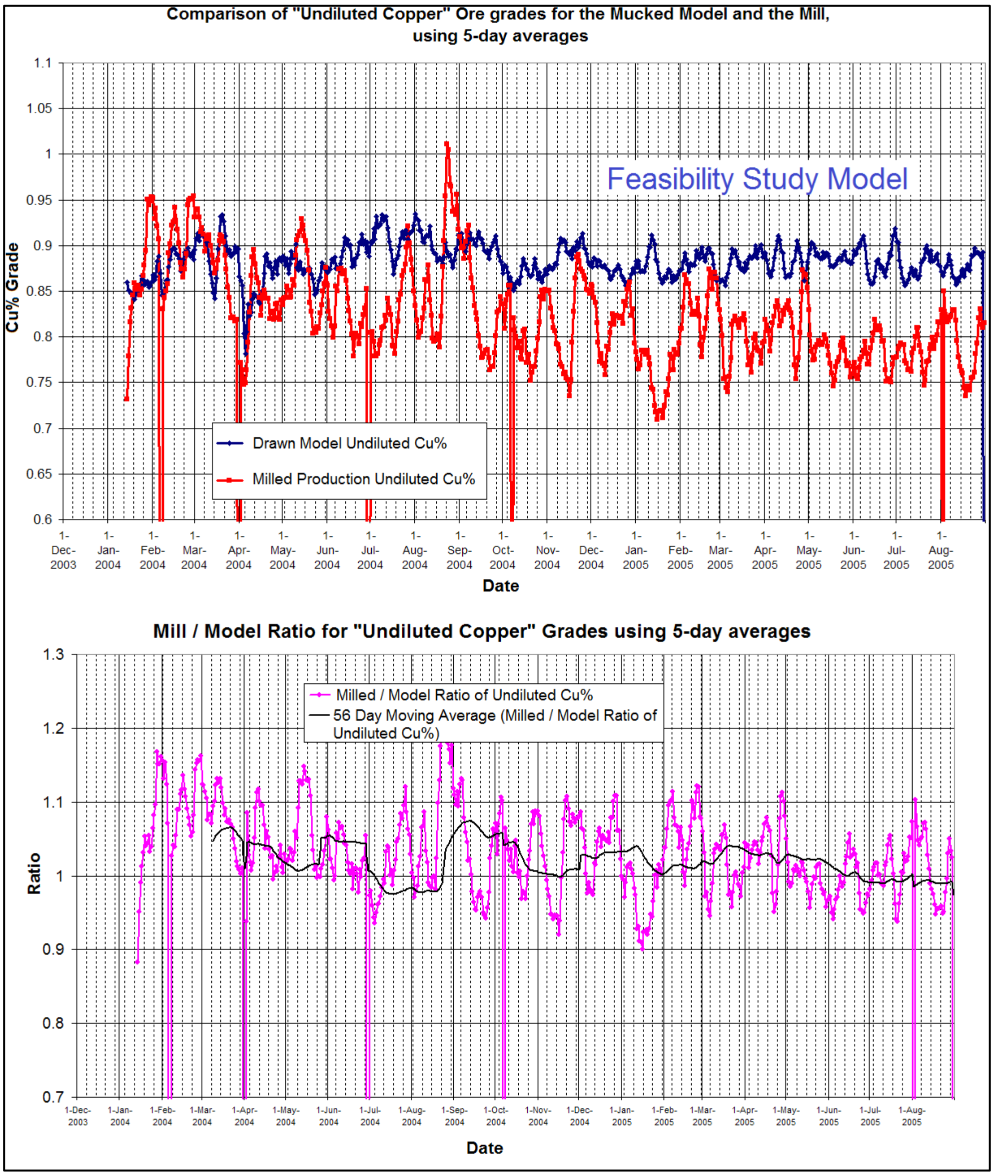


Figure 65: Comparison of production copper grades with predicted FS grades
Source: (Howson, 2006)

The magenta trace indicates the ratio of the red over the blue from the top graph, and from 2004 the ratio is below 0.9. The indication is that the model overpredicts production grades by more than 10%.

When OK methods were applied, a small positive drilling pattern bias resultantly magnifies also by a feature of OK combined with sparse data. Through the application of MISK, it seemed that the positive drilling pattern bias resulted in a counteracting negative bias in the deposit mean values for Palabora (Howson, 2006). The application of a 'Multiple Indicator' technique was supporting the change in variogram range with the grade. MISK seemed to be the most effective estimation method for Palabora copper, but the MISK model required additional testing with the PCBC software to estimate production grades. Figure 66 below indicates a very close correspondence between the mill and the model results and the ratio hovers close to one.

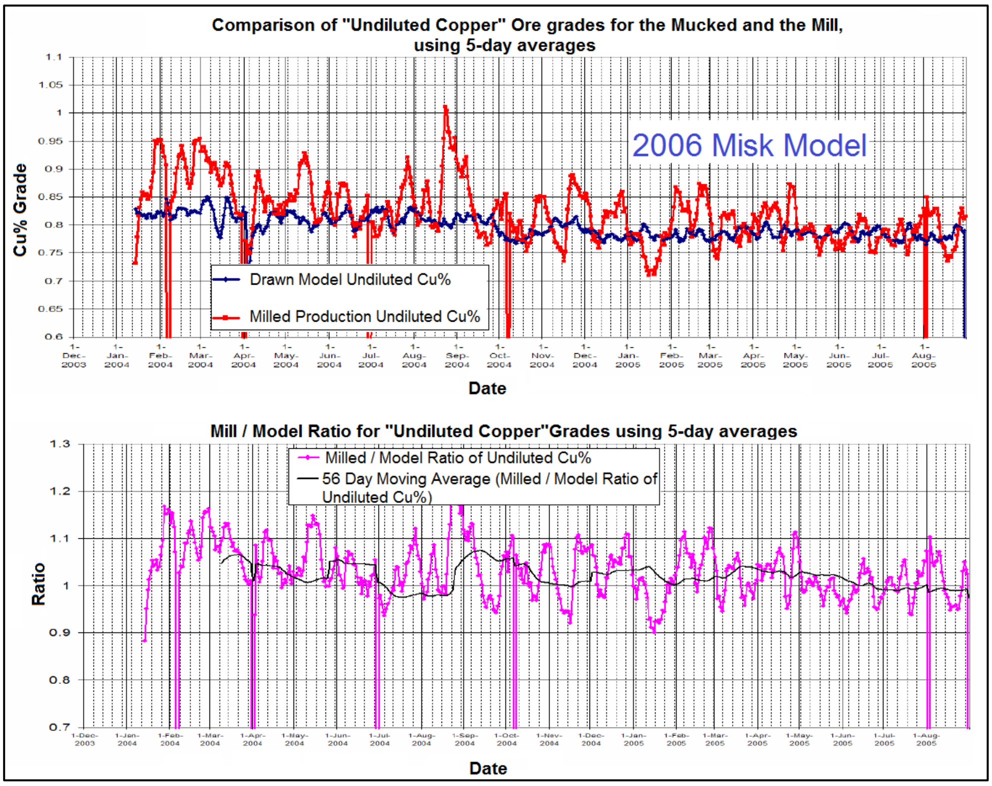


Figure 66: Comparison of production copper and mill grades using the MISK model
Source: (Howson, 2006)

The success of the MISK model and above graphs does not suggest that MISK is a great new way forward in estimation, but MIK in the form of MISK is valuable to accommodate the variation in variogram range (Howson, 2006).

The re-estimation exercise had valuable lessons for other studies and other deposits, ranging from order-of-magnitude studies, through pre-feasibility and feasibility studies to production (Howson, 2006).

8 PMC HEAD GRADE AUDIT JULY 2010-2011

8.1 Introduction

In 2010, PMC invited Snowden Mining Industry Consultants (Snowden) to investigate a new discrepancy between the measured copper head grade from the plant and the grade estimated by the resource model. The grade discrepancy has manifested itself during the past several years, and results ranged from 10% to 12%, which led to the required block model re-estimation in 2006 (Snowden, 2010). During 2010, this discrepancy continued to influence the mine with effective draw compliance in the grade and maximising the LOM of the block cave (Snowden, 2010).

The application of a variety of algorithms simulates the mixing of material, taking place as LHD machines extract material from the draw points. PCBC accepts the actual draw tonnes from respective draw points and reconciles it against the actual production results (Snowden, 2010). A comparison between the PCBC grade and the geological block model as per Figure 67, indicates the PCBC figures are lower than the in-situ block model grade.

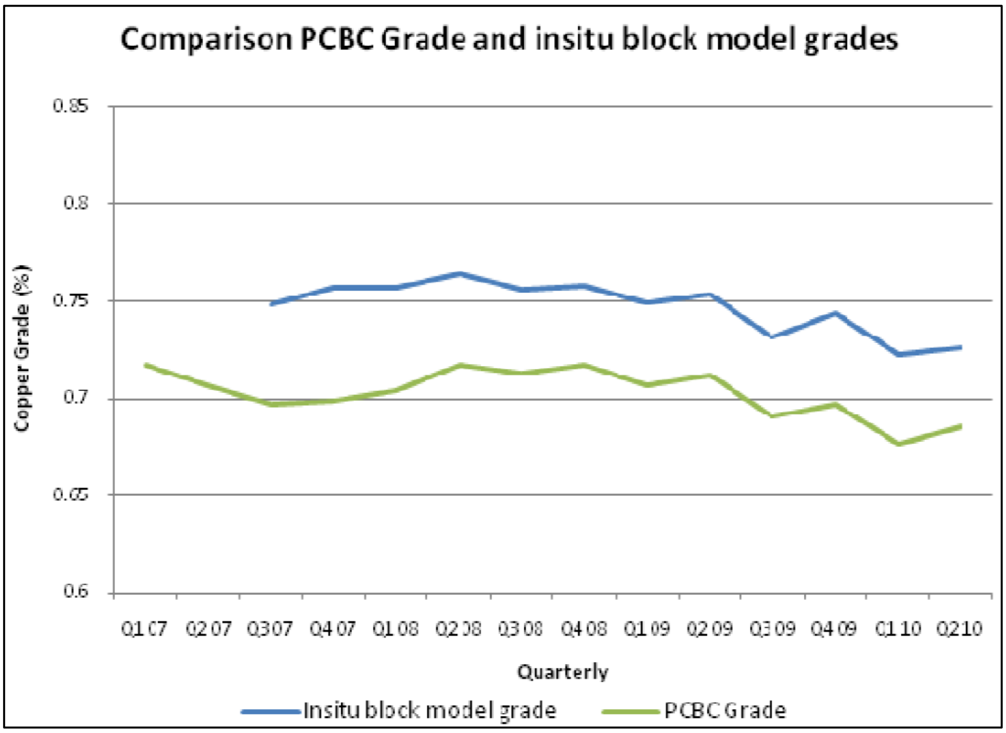


Figure 67: Grade comparison between PCBC and the geological block model
Source: (Snowden, 2010)

The actual sampled grade began to diverge in 2008, and the 2010 sampling indicated a discrepancy of between 4% and 8% (Snowden, 2010).

8.2 Methodology used

Snowden (2010) followed a methodical approach, which tested the resources, mining, and plant operation evidence for or against the tested hypothesis. The following subsections describe the details assessed during the investigation process.

8.2.1 Geological investigations

Snowden (2010) focussed their geological investigations on the resource estimates and supporting data. Factors which Snowden (2010) considered to contribute to the discrepancy between the head grade and PCBC grade included the resource estimation and geological interpretation. Snowden (2010) reasoned that evidence of poor quality or poorly distributed data would indicate a poorly estimated resource block.

A poorly estimated resource block model will result in differences between the recovered grades and the predicted PCBC grades (Snowden, 2010). Other sub-factors are affecting the resource estimation, including data distribution and quality and quality of estimates and grade continuity.

The geological interpretation investigated evidence for a change in geological composition, seemingly erroneously omitted in the initial interpretation or no recognition thereof. Snowden (2010) tested whether any geological interpretive evidence supported the drop in head grade. The geological evidence for the head grade to be lower or such geological interpretation are sub-factors affecting the geological interpretation (Snowden, 2010).

8.2.2 Mining investigations

Snowden's (2010) mining investigation considered possible causes for the overall variance between the PCBC production grade and the head grade. These causes included critical areas of the mining operation. According to Snowden (2010), the variance in head grade could exist from an incorrect setup, calibration, or use of the PCBC model. Snowden (2010) reviewed the production schedule from PCBC and compared it to the production profile. Areas which Snowden (2010) covered in their study follow in the paragraphs below.

Snowden (2010) reviewed the block cave design parameters such as draw cone dimensions, and draw point spacing to determine the suitable spacing thereof. Snowden (2010) noted that the incorrect design parameters based on overestimated fragmentation commonly resulted in narrower draw zones. These narrower draw zones advanced faster through the ore profile, resulting in lower grade ore material mined sooner than expected. Furthermore, a higher percentage of external dilution drawn into the draw point efficiently lowered the mining grade which Snowden (2010) had to review.

Internal and external dilution affecting the head grade to be lower was included in the Snowden (2010) investigation, to identify the existence of possible poor draw control. Material loaded not conforming to the mining plan, and incorrect tonnes resulted in increased dilution where waste material drawn into the cave happened sooner than anticipated. Snowden (2010) therefore investigated whether the draw control practice was of an acceptable standard across the block cave as is discussed in more detail in section 8.5.2. Snowden (2010) commented that the current draw control could improve in especially Sector 4 for draw points having longer tramming distances, but overall it seems reasonable.

8.2.3 Plant Investigation

Snowden (2010) focussed on the sampling of the head grade and the laboratory analysis of such samples used for the head grade directive. Measurements of total tonnes mined and head grade calculations fell under the magnifying glass of Snowden (2010) during the plant investigation. Every single entity during investigation and assessment included evaluation of each potential for having miss-calculations or measurement errors that might have influenced the head grade to drop (Snowden, 2010).

8.3 Geological evidence supporting the drop in head grade

Snowden (2010) reasoned that the draw point sampling was a good indicator as to whether the drop in head grade observed from April to June 2008 was authentic. Samples taken from draw points by the geological department at monthly intervals entailed five sub-samples over the draw face with a shovel in a best practical manner (Snowden, 2010). If the fragmentation was coarse, hand samples were possible but left more substantial portions behind that could not be hand loaded. In Snowden's (2010) opinion, this sampling procedure was not adequate or representative of the draw points, and the results were only good for indicative measures. Either a more extensive sample per draw point was required, or more massive rocks chipped into equal-sized particles would reduce the bias towards fines (Snowden, 2010). The frequency of monthly sampling intervals was too infrequent to provide a representative assessment but remained a known problem to the geological department.

The bias in the sampling was consistently higher than the head grade as per Snowden's (2010) investigation, which required further investigation. A possibility existed that sulphides collected preferably in less coarse material, and therefore preferential sampling, which excluded certain material types, could be reasons for the higher head grade. Dolerite diluted the ore and did not contain grade. Hence Snowden (2010) expected a reduction in the head grade where it occurred. Snowden (2010) concluded from the data that observations on dolerite content and copper content were not matching and the likely cause was bias in the current sampling methodology. Snowden (2010) indicated that if no error in the head grade sampling and analytical process existed, the drop in head grade during April 2008 to June 2008 was likely to be authentic.

8.3.1 Spatial distribution of lithology and geological interpretation

Snowden (2010) indicated that the provided block model contained only the dolerite percentage. Therefore, the geology of the block model omitted available detailed geology not provided to Howson (2006) to allow for further investigation in 2006. The lack of a detailed geological model informed Snowden (2010) to use the drill-hole file to investigate the different rock types and their distribution. Through the completion of a slicing exercise on the drill-hole data, it was possible to observe any lithological ratio changes between the primary lithological units (Snowden, 2010).

Snowden (2010) used the slicing tools within the software to slice a portion of the model between elevations and retrieve data applicable to the sliced portion only. These slices represent the model in vertical sliced portions as is indicated by Figure 68's elevation axis.

In Figure 68 the level slicing of drill-hole data, which was raw un-composited drill-hole data, reflected the lithological data and their contribution of copper to the overall grade. These main constituents were transgressive carbonatite, banded carbonatite, and foskorite (Snowden, 2010). Snowden (2010) observed a sharp grade contact between the transgressive carbonatite and the rest, but it was more gradational between banded carbonatite and foskorite.

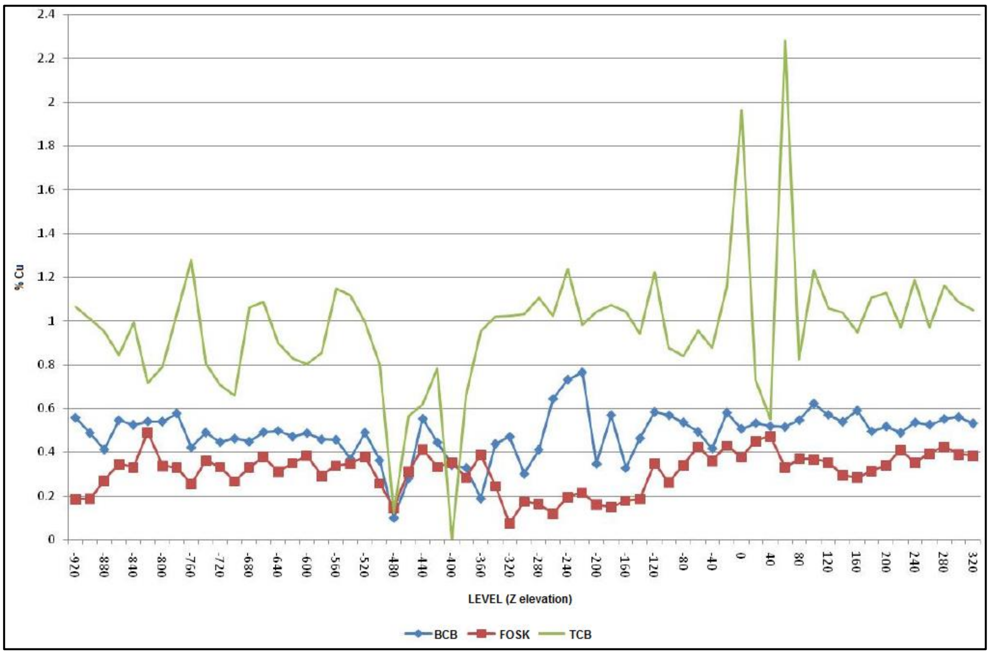


Figure 68: Level slicing of drill-hole data and copper percentage
Source: (Snowden, 2010)

8.3.2 The impact of foskorite

Snowden (2010) established from drill-hole data that the primary constituent in the cave was transgressive carbonatite (TCB) with banded carbonatite (BCB) around it, while foskorite remained on the periphery. These specific lithology types are ore material as they all contained copper grades. Micaceous pyroxenite was external dilution and is rich in mica and diopside, while it differs from the other ore materials. Since foskorite material extended beyond the mining footprint, it was more an external sidewall dilution in some circumstances (Snowden, 2010). The average foskorite grade was 0.35% Cu, and it was lower than the transgressive carbonatite or banded carbonatite. Snowden (2010) investigated the possibility whether an increased amount of foskorite ingress into the draw cones, from the sidewalls or externally, caused the reduction in head grade. Snowden (2010) additionally considered a higher expected quantity of foskorite present in the draw points and investigated data observations recorded by the on-mine geological team.

The findings indicated that no BCB was present while observing a relatively large percentage of foskorite. The lack of observed BCB indicated a possible misclassification between BCB and TCB in the broken ore underground. Snowden (2010) noticed that the crosscuts, where the expected higher grades should occur, had higher quantities of foskorite. Snowden (2010) also noted a contradiction between raw drill-hole data and data from the GEMCOM model and concluded that the 2006 GEMCOM geological data did not represent the drill-hole data. Should the observations be correct, then, substantial evidence would present that higher than expected low-grade foskorite quantities reported to draw points, affecting the head grade negatively (Snowden, 2010).

8.3.3 Findings on geological interpretation

Snowden (2010) concluded that the geological interpretation contributed to the variance in grade. Snowden (2010) found omitted lithological units and information in the current block model except for dolerite. Furthermore, the erroneous exclusion of relationships between TCB, BCB and foskorite from the resource estimate occurred. Snowden (2010) recommended a re-estimation, which included all relevant lithological units. Snowden (2010) observed higher than expected quantities of foskorite in the draw points and it resulted from the incorrect interpretation of the geological model or external dilution.

8.4 The resource estimation

The resource estimation review sought for any cause, which may have contributed to the variance in head grade and the PCBC grade. The following aspects such as the quality of the data, the quality of the estimation and grade continuity finalised the investigation Snowden (2010) conducted. PMC provided a 'rekriged09' GEMCOM block model, with historical drill-hole data from the open pit and from the underground exploration drive to Snowden (2010) for their investigation. Table 5 below lists the raw data files received from the mine.

Table 5: Raw data received from the mine - Source: (Snowden, 2010)

Data received	Date received	Filename
PCBC resource block model	23 June 2010	rekriged09.csv
All collar positions	23 June 2010	allcollars.csv
Lithological file for drillholes	23 June 2010	geology.csv
Assay file for lift one	23 June 2010	lift1assay.csv
Collars for lift one	23 June 2010	lift1collars.csv
Downhole survey for lift one	23 June 2010	lift1survey.csv
Surface drillholes	29 June 2010	surveyedholes.csv
Surface assays file for surface drillholes	1 July 2010	lkcomb.csv

The open pit contained a significant amount of drilling carried out during its operations where only a small amount of data was available between the exploration drive and the open pit (Snowden, 2010). Figure 69 presents this shortfall of drilling data, where only a couple of drill holes existed between the open pit bottom and the exploration drive.

8.4.1 Data quality and laboratory QAQC

Most of the captured drilling data and work backdate to pre-1996, making the quality control processes applied to collect data challenging to verify. Snowden (2010) however noted that the QAQC procedures for the second lift were appropriate. Figure 69 indicates the surface holes in red and the underground exploration holes in blue.

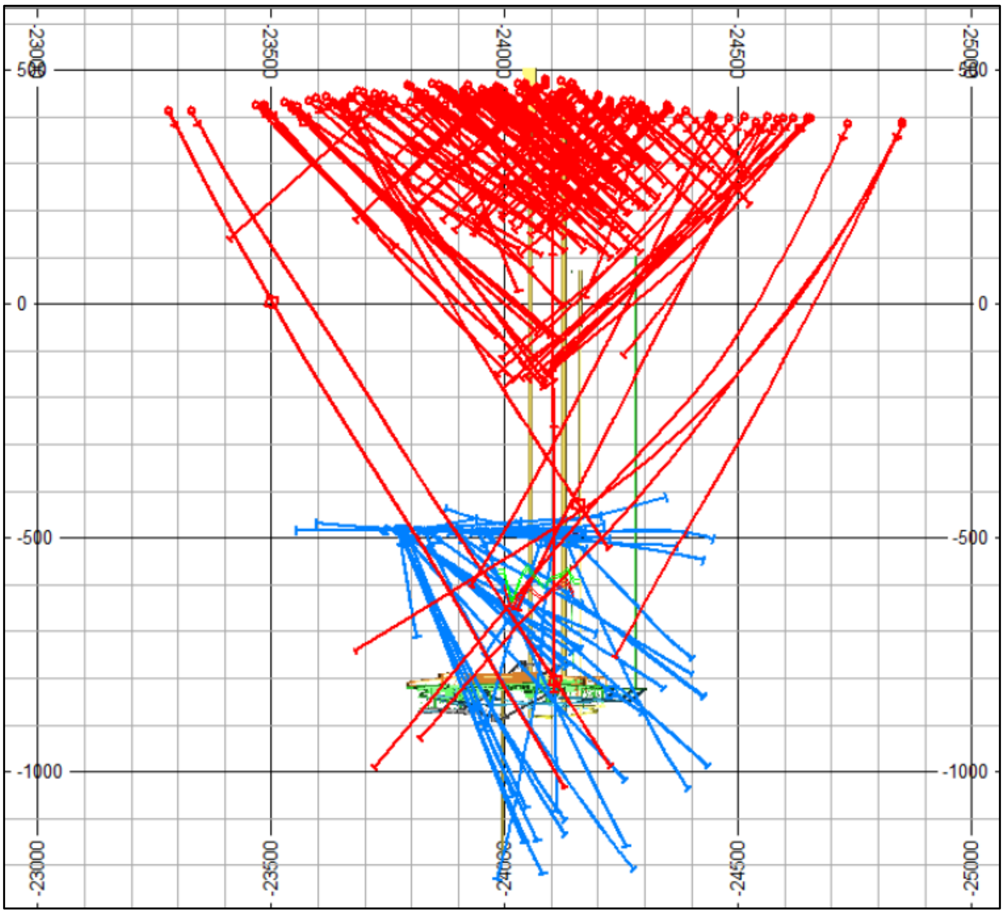


Figure 69: North-South section showing drill-holes at PMC
Source: (Snowden, 2010)

Also, Figure 70 indicates the distribution of the drill data and the scheduled position of the drawing height on the production level using PCBC plots shown below in comparison. The third quarter draw height for 2008 is red, green for 2009 and blue indicates the drawing height for 2010. Snowden (2010) indicated that the height difference between the height of draw (HOD) and the pit bottom ranged between 100 m and 200 m.

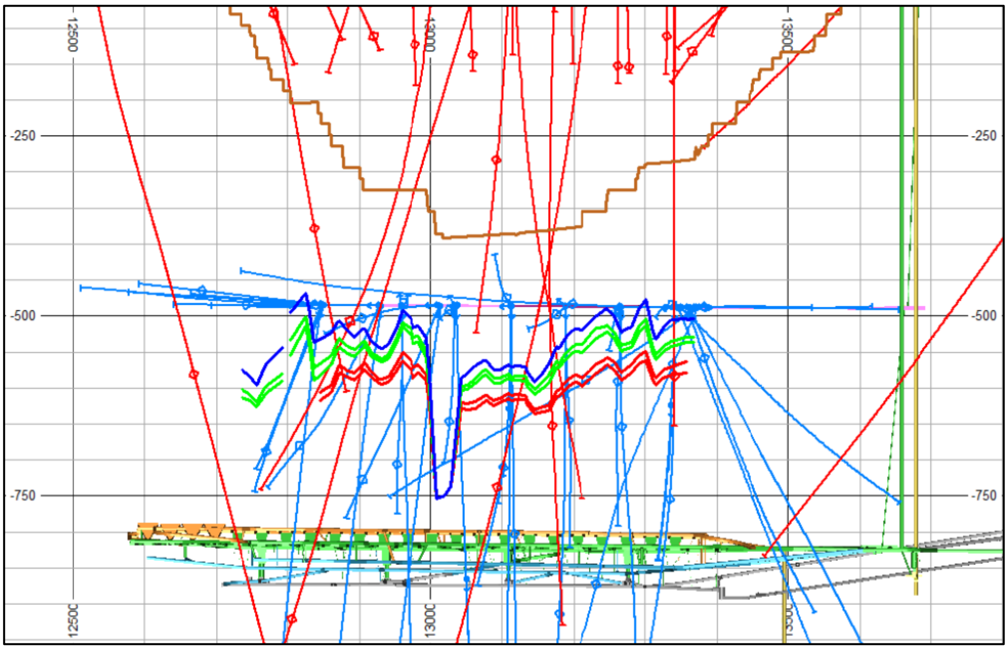


Figure 70: North-South section of the underground production level - draw height
Source: (Snowden, 2010)

The current drilling program and samples audit comprised of all internal QAQC procedures and the use of internal standards for calibration. Snowden (2010) could not provide evidence to conclude whether the negative long-term drift indicated that the head grade reduction was due to an analytical procedure. Snowden (2010) thus indicated that the procedures, monitoring, and control of the analytical processes were appropriate and that the results were reliable and unbiased.

The FS drilling QAQC was not readily available, which concerned the limited extent of assessment of PMC's resource estimation for the current mining area. The quality of the resource estimate required that the resource block model provide excellent representative drilling data with the geological interpretation thereof (Snowden, 2010). Snowden (2010) realised they had to adjust the model and drill hole data to exclude the dolerite. Snowden (2010) composited the drill hole data to 9 m lengths and he de-clustered the drill hole data to correspond with the original composite length used by PMC. Snowden (2010) additionally included the vertical slices and northing geographical orientation for the entire estimated block model with the 9 m composited drill hole data. The observation Snowden (2010) made revealed that the grade was over-smoothed when comparing it with the original data. There were limited samples numbers between the pit bottom and exploration level, while a significant reduction in the average grade of drill hole data showed. The concerning matter for Snowden (2010) was the model between the exploration drive, and the pit bottom did not reflect the grade of the drill hole data adequately. Through the application of vertical slices through the entire estimated block model, and the 9 m composited drill hole data Snowden (2010) could evaluate how the number of samples represented areas of the model. Figure 71 shows the average grade for Cu in red, composited 9 m drill holes in blue and the number of samples in green. The grade observations made indicated over-smoothing of the grade when Snowden (2010) compared it to the original data. There were also a limited number of samples taken between the bottom of the pit and the exploration level, reflecting a significant reduction in the average grade of the drill-hole data.

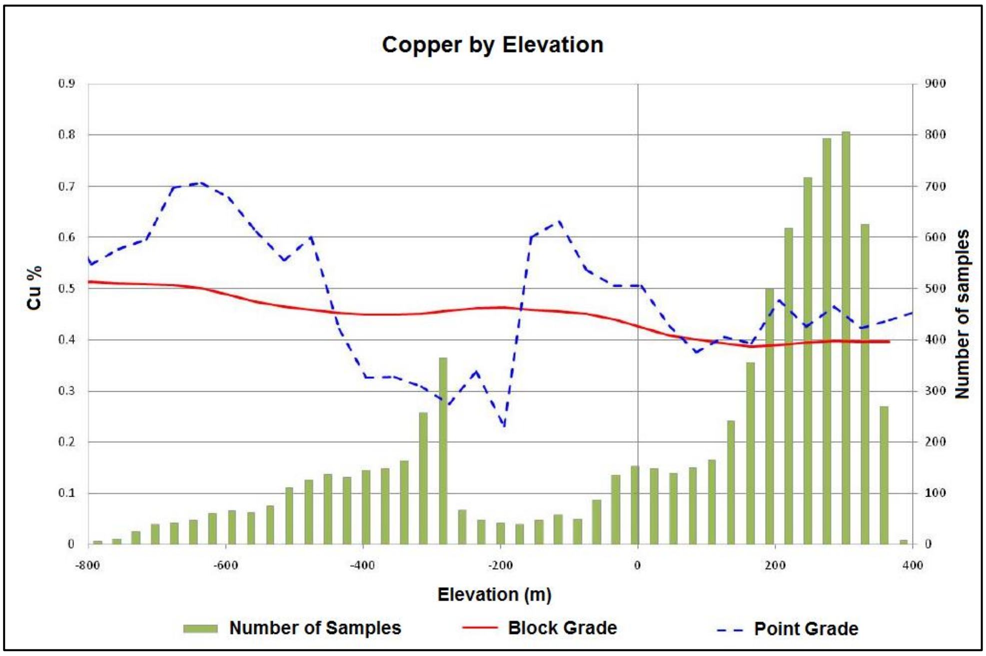


Figure 71: Level slicing of drill-hole data
Source: (Snowden, 2010)

When Snowden (2010) sliced the data in an easting orientation, the peripheral grades to the north and south reflected a grade overestimation. The over-smoothing of the data was problematic and would result in an overestimation of low grades but an underestimation of higher grades. Snowden (2010) indicated that the insufficient available data prevented to quantify the impact of the smoothing on the overall grade of the orebody. Overall, the quantity of data within the mining footprint was inadequate, and an unexpected variation in grade estimation could occur as a result. Howson (2006) supported the same concern, in that he mentioned the inadequate drilling and sampling of the mined volume (Snowden, 2010).

The block model estimates and drill hole assay grades showed an excellent visual correlation through the investigation from Snowden (2010). Smoothing via the estimation process was prominent where higher estimated grades indicated lower than the composited assay grades and conversely lower estimated grades were higher than the lower composited grades. It remained difficult for Snowden (2010) to quantify the smoothed expectancy with the limited available data. Similarly, PMC and Rio Tinto reported grade discrepancy problems historically. Grade discrepancies experienced in 2004 and 2005 resulted from a combination of factors but mainly from estimation techniques. The inadvertent phasing of the FS seemingly contributed as the overriding reason for the discrepancy. Snowden (2010) indicated that the implied classification of the PMC resource at the measured level was not justifiable and that the resource classifications required a full review.

Although Snowden (2010) used the preferred 9 m composites in his analysis, the composites changed to 10m in 2013 (PMC, 2016). During 2012 a comparison between composite lengths of 3 m and 9 m indicated that composites of 9 m had a larger smoothing effect where the swath plots plotted against the X, Y and Z-axis (Snowden, 2010).

8.4.2 Grade continuity analysis

By understanding the grade continuity, the spatial analysis of the data required an overview. Snowden (2010) removed data containing zeros in the "% Cu" field, after investigating the drill-hole data. The continuity analysis illustrated that there were a 100º strike direction and vertical dip continuity in grade and that there was no evidence that the copper was bottoming out (Snowden, 2010).

8.4.3 Findings on the resource estimation

Snowden (2010) stated that reservations about the resource estimates were the main contributor to the grade variance. The quantity and quality of data were inadequate within the mining blocks and was not verifiable, especially where limited sample data affected the quality of the estimate. Snowden (2010) found the resource classification to be optimistic despite good grade continuity observations. Estimations extrapolated outside the data area and grades for the contamination within the draw were biased from what Snowden (2010) observed.

SRK Consulting (2011) reported that the resource estimate resulted from drilling within the open pit and exploration level approximately 850 m below the surface. The affected caving area between the exploration level and the final open pit was impoverished in drilling information (SRK Consulting, 2011). The caved zone's grade estimation resulted from relatively wide spaced and distant information. The methodology for estimation was Multiple Indicator Simple Kriging for PMC's block cave. SRK Consulting (2011) also reported that the lack of segregation of drill-hole samples in the estimates affected the calculation and modelling of experimental semi-variograms.

8.5 Mining audit

The PMC mine planning process involved the combined current geological block model without draw point positions and production requirements in meeting the plan. The PCBC software ensured that no double accounting of material occurred for intersecting cones (Snowden, 2010). To simulate the actual extraction of material mixing during material loading operations was a complicated calculation. Hence, a variety of mixing algorithms within the calculations of the software applied to the simulation (Snowden, 2010). Snowden (2010) noted that PCBC functioned as a reconciliation tool when applying the ‘playback’ function, as was applied by PMC.

The period 2004 to 2008 revealed a very close trend for tonnes and grade when Snowden (2010) compared it to the actual production head grade results. However, the grade diverged in 2008, and the current sampling indicated a discrepancy of between 4% and 8%. Snowden (2010) initially compared PCBC outputs from the in-situ geological block model by using the height of draw (HOD) as the basis for determining extracted material. Snowden (2010) concluded that there were no gross set-up errors in PCBC and affirmed that the grades in the geological model were representative of the PCBC output grades (Snowden, 2010). During the Snowden (2010) investigation, it was not appropriate to benchmark the PMC block-cave against other block-caves because of the level of analysis. Snowden (2010) chose to compare the PCBC ‘playback’ outputs with the head feed over time. Although Snowden (2010) realised that a correlation in the outputs did not prove the correctness of the PCBC parameters, it would demonstrate an achieved level of calibration. The PCBC outputs with the actual head grade feed results indicated a realistic comparison. However, Snowden (2010) could not prove the calibration of PCBC since there was no real grade or geological markers within the ore body that was usable for such comparison. Studying Figure 72, the period 2004 to 2008 indicated a higher head grade between 0% and 2% from the expected grade, while the period after that was between -4% and -6% lower (Snowden, 2010).

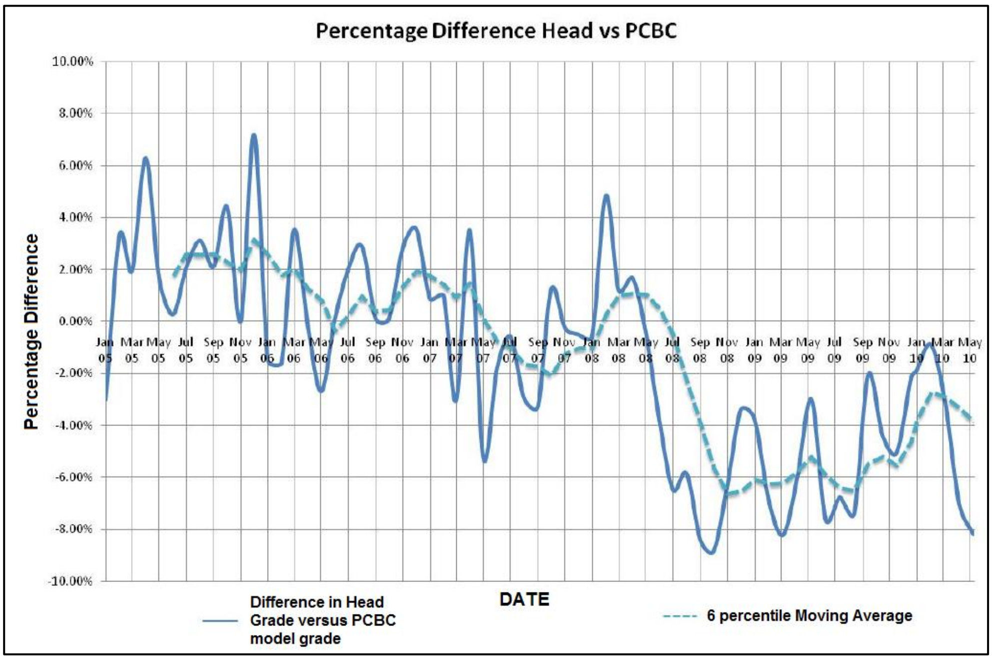


Figure 72: Percentage difference between the head grade and PCBC grade
Source: (Snowden, 2010)

Notably, Figure 73 indicated that the head grade approximated the expected grade on average over the entire period. Hence, Snowden (2010) concluded the PCBC model more suitably calibrated over the long term than the short term.

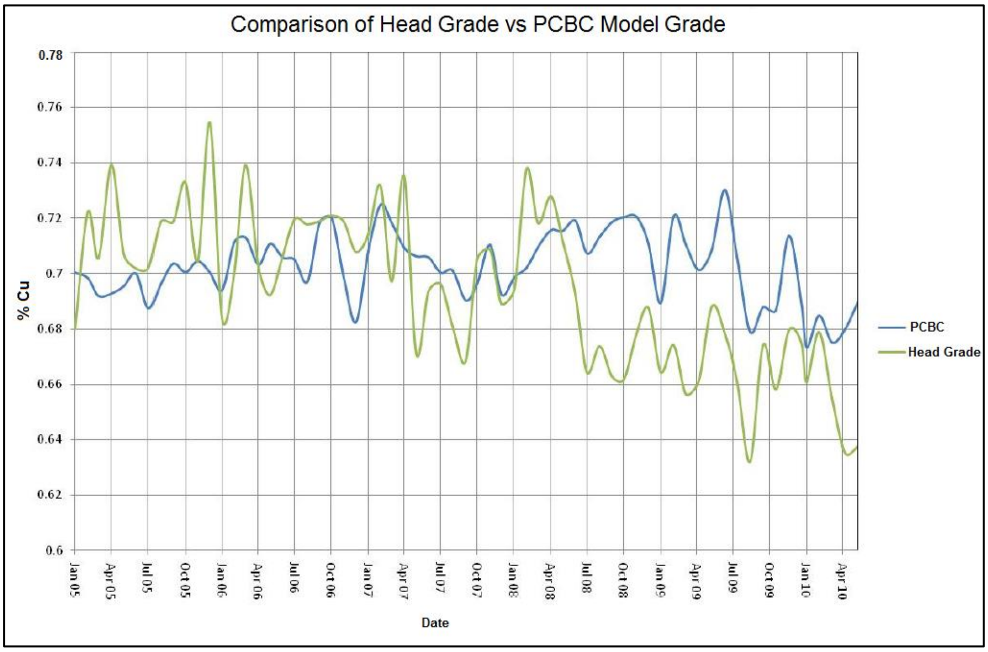


Figure 73: Head grade and PCBC grade comparison
Source: (Snowden, 2010)

Snowden (2010) noted the software supplier's intimate involvement in setting up PCBC with appropriate input parameters, which became more applicable where section 9 discuss additional conducted work done by Diering (2015). The input parameters remain subjective as these inputs are difficult to measure accurately (Snowden, 2010). When Snowden (2010) compared the PCBC output with the inputs from the in-situ geological block model, the height of draw (HOD) formed the basis for determining the material extracted. Table 6 below reflects the input parameters Snowden (2010) used.

Table 6: PCBC input parameters - Source: (Snowden, 2010)

Parameter	Unit	Value
Pre-vertical mix	Horizon	160 m
	Mix cycles	2
Sequential vertical mixing	Horizon	189 m
	Mix cycles	2
Topping slope - broken	Angle	37º
Toppling slope - intact	Angle	55º
Range for neighbours	Distance	27 m

8.5.1 Fragmentation and dilution

Snowden (2010) indicated during the feasibility study a fragmentation estimation program was developed to predict rock size distribution from the block cave. The program, namely: Block Cave Fragmentation (BCF) predicted fragmentation between 80% and 42%. Ngidi & Pretorius (2010) embarked on a paper titled: "An assessment of the impact of poor fragmentation on cave management", and they found that the number of oversize rocks (>2 m) ranged between 58% and 14%. Figure 74 indicates that an overestimation of the coarseness of the fragmentation occurred.

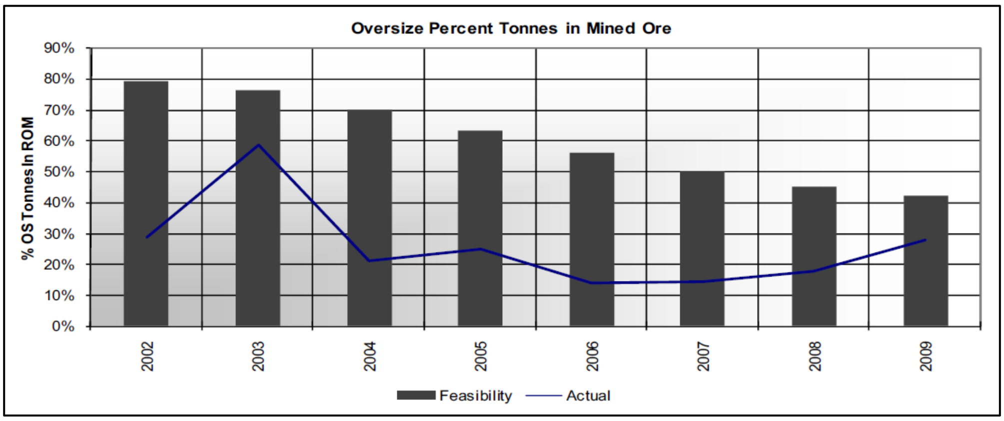


Figure 74: Percentage oversize (>2 m) tonnes for the run of mine
Source: (Ngidi & Pretorius, 2010)

Snowden (2010) also indicated that fragmentation affected the draw zone diameter, where finer material resulted in smaller draw zones. By this indication, it suggested that the draw zone's diameter seemed to be smaller than what the original design proposed. Snowden (2010) observed relatively finer fragmentation in the western side of the block cave and coarser fragmentation on the eastern side of the cave. In Figure 75 below, the production level plot indicated the fragmentation and grade in three identified zones as per the PMC production team's experience.

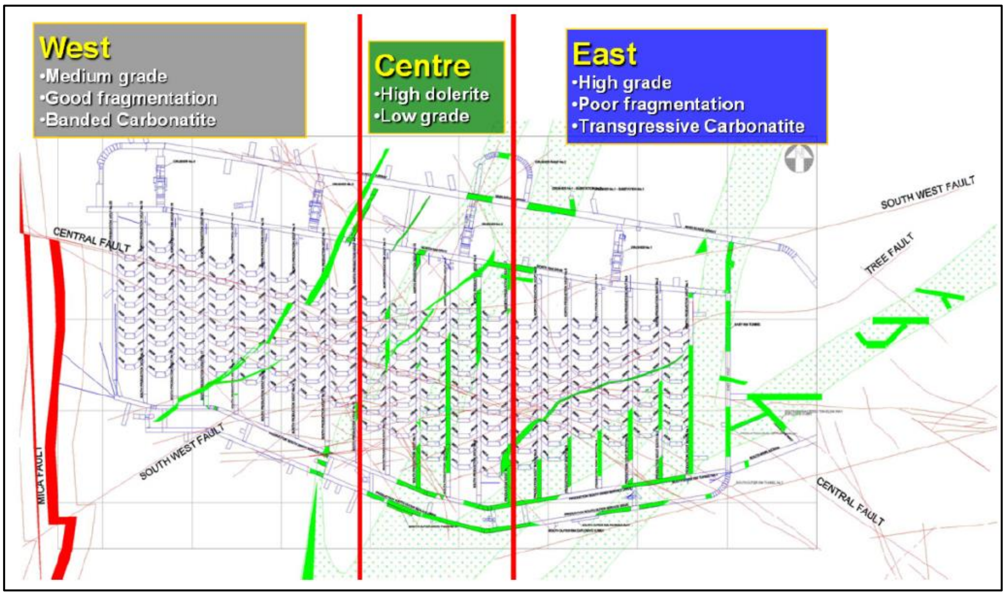


Figure 75: The fragmentation and grade as experienced by PMC production
Source: (Snowden, 2010)

Snowden (2010) reviewed the fragmentation, which indicated an underestimation in fragmentation across the block cave especially on the western side of the cave. The underestimation could result in smaller planned draw zone diameters with an excess broad draw point spacing design (Snowden, 2010). Underestimating the fragmentation could also result in the decrease in grade which Snowden (2010) observed. This decrease in grade required an assessment testing whether ore material above the current mining draw height mined, indeed mines faster than planned. If the above underestimation was correct, the expectation was an orebody grade above the current mining horizon that was significantly lower than the current mining zone. Hence, Snowden (2010) sliced the orebody up into 40 m slices from the production level underground, up to the open pit bottom. Snowden's (2010) finding indicated that the current draw elevation was at approximately -600 m and had a grade of 0.77%. Snowden (2010) concluded that no evidence was found to support the probability of having smaller planned draw zones.

The PMC block cave experienced both internal and external dilution. Internally, dolerite dykes intruded across the orebody, with reasonably well-defined positions in the block model. External dilution considered by Snowden (2010) comprised mainly of material situated above the block cave. Snowden (2010) further indicated that the dilution resulted from the 2004 sidewall failure within the pit, where lower grade material occurred around the mining footprint. Snowden (2010) expected this external dilution to increase as waste material filtered through the draw cones into the drawing points.

A consistent draw rate across the entire block cave footprint would ensure that waste material or dilution did not prematurely enter the caving ore material, by sterilising parts of the orebody or reduced grades. Snowden (2010) noted the current PMC philosophy being the drawing of higher draw cones quicker on the west and east of the block cave and ensuring minimised sterilisation of draw cones through toppling. Snowden (2010) considered the internal dolerite dilution to be well controlled and in Figure 76 below; little evidence existed to indicate that preferential loading was taking place.

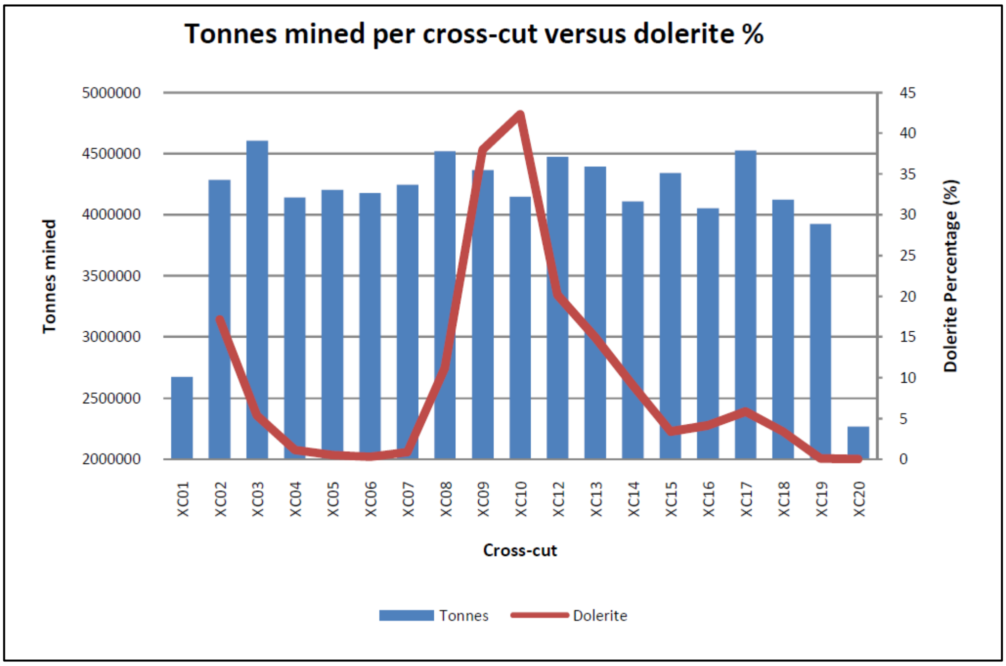


Figure 76: Tonnes mined per crosscut versus the dolerite percentage
Source: (Snowden, 2010)

Snowden (2010) indicated that higher than expected low-grade foskorite material occurred in the head feed ranging between 20% and 40%. Although the origin of this material was unknown, it was likely to be part of the poorly modelled orebody, or it resulted from external dilution.

8.5.2 Draw control

In an attempt to assess the PMC draw control management, Snowden (2010) addressed a couple of aspects namely; initial draw control practice pre-2005, draw control by sector, draw control by draw point number and preferential draw for high-grade areas. Snowden (2010) raised concerns regarding the poor draw control management during the initial years of the block cave, which resulted in more erratic draw horizons or HOD across the present cave.

PMC numbered its draw points from the north to south in the crosscuts. The northern draw points were hence automatically closer to the crushers and therefore had the lowest hauling distance. Snowden (2010) investigated the possibility of preferential drawing of draw points closest to the crushers. Snowden's investigation pointed that preferential draw occurred but considered the impact very low since these draw points made up a small portion of the overall draw (Snowden, 2010).

Snowden (2010) assessed whether PMC undertook preferential draw to target the higher-grade areas and neglected the lower grade areas in the process. This investigation considered tonnes extracted per crosscut in comparison with the average grade per crosscut, and the results indicated little evidence of high grading as per illustration in Figure 77 below (Snowden, 2010).

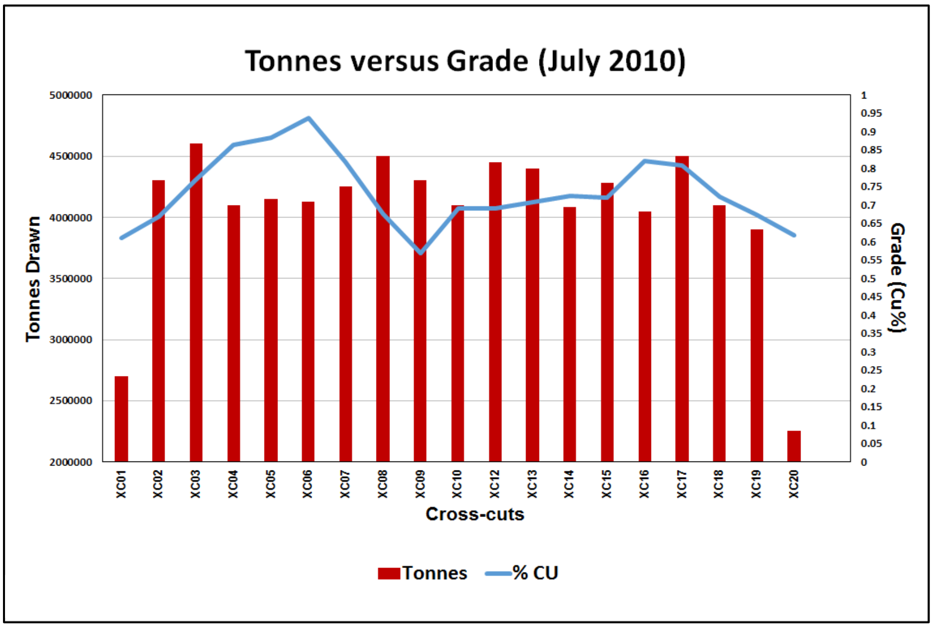


Figure 77: Tonnes drawn per crosscut versus grade per crosscut
Source: (Snowden, 2010)

Snowden (2010) concluded that the initial poorly managed draw control negatively affected the overall draw control, resulting in lower grade material drawn through the cave faster than was anticipated. An average bucket factor applied across the cave was an additional concern.

Reconciliation of a weightometer on the head feed belt that measures total mined tonnes followed by using the number of buckets to determine the bucket factor. Considering this bucket factor, it was not a proven concern regarding the head grade discrepancy, but it could adversely affect the mine in the longer term (Snowden, 2010).

8.5.3 Reported mining findings

Some critical areas of the mining operation considered by Snowden (2010) could contribute to the overall variance between the PCBC production grade and head grade. However, it was unlikely that any of these factors have contributed to the increase in grade variance. The initial poor draw control during the initial cave mining period with possible decreasing draw zone diameters may have resulted in the dilution to increase in the future. The final expectation from Snowden (2010) was that external dilution would increase with waste material filtering down through the cave. Snowden (2010) recommended increased sampling in the block cave of PMC to monitor and ensure better dilution and cave management.

8.6 Findings on the plant head grade sampling

Snowden (2010) deemed the equipment and sampling procedures followed by PMC to be acceptable.

8.6.1 Mass measurements

According to Snowden (2010), the underground ore tonnage treated in the plant, and measured via the feed belt’s scale installations were correct and adequately calibrated. Snowden (2010) reconciled the overall underground tonnes to the auto mill feed belt’s tonnes. The confidence level in tonnes recorded according to Snowden (2010) was of high level and satisfactory.

When Snowden (2010) investigated the compliance with all internal QAQC procedures and internal standards for calibration they highlighted a concern regarding the long-term negative drift which remained evident. However, if the head grade reduction were due to analytical procedures, the standard would provide evidence to make such a conclusion. In Figure 78 below, the standard drift depicted by the green line on the graph represents the cumulative difference of the standard, and the certified value of the standard (Snowden, 2010).

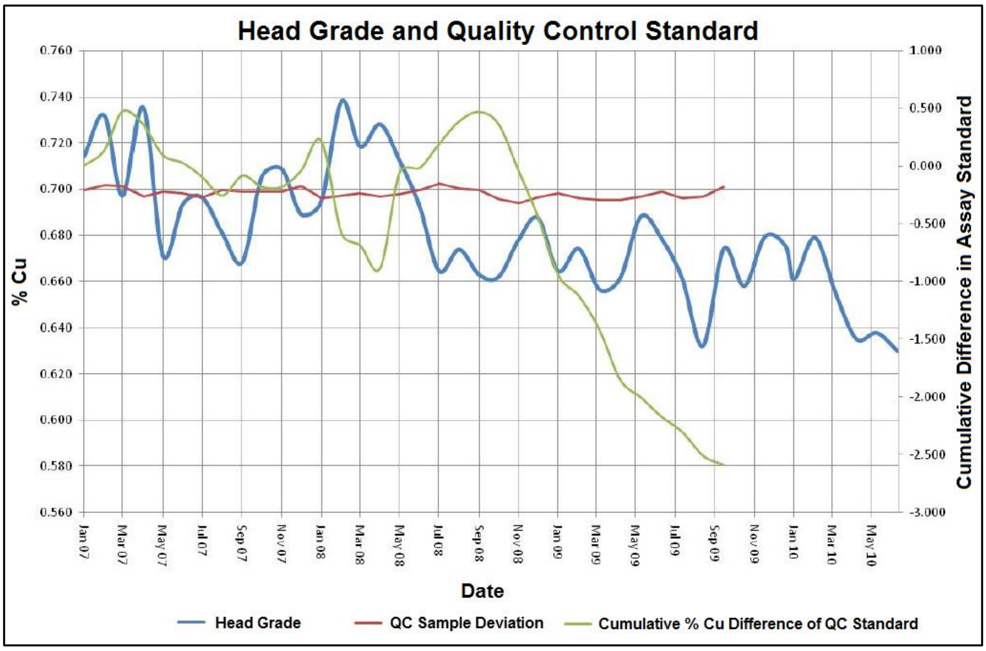


Figure 78: The cyclone overflow quality-control standard & head grade per month
Source: (Snowden, 2010)

Snowden (2010) indicated that the procedures, monitoring and control of the analytical processes investigated, was appropriate and the results were reliable and unbiased.

8.7 Conclusion

Snowden (2010) concluded that the issues relating to the resource model were most likely the critical factors for the variance between the head grade and the PCBC output grade. The possibility existed that the current fragmentation issues may have exacerbated this discrepancy.

The limited sample data coverage and unavailability of QAQC data compromised the underlying data used in the resource estimation.

Snowden (2010) considered the estimation smoothed, but alternatively resulted in an overestimation of lower grade areas and an underestimation of high-grade areas. Contributing to the limited information it resultantly created a lower confidence classification of the resource model. Since the block model excluded the lithology units except for dolerite, higher than expected quantities of foskorite observed may have influenced the external dilution interpretation of the geological model.

This project report wishes to emphasise the specifical term used by Snowden (2010) in section 8.5.1. Snowden (2010) referred to “toppling” and this term according to this project report never received the attention it required. Seemingly, the slope failure’s grade impact and actual effect were not adequately tested or correctly interpreted. This critical failure grade which affects the PMC block cave lacked adequate detail in the previous resource reviews. This project report addresses the failure’s grade in more detail in the next section.

9 REVIVING THE 2015 RESOURCE MODEL

9.1 Introduction

During 2015, PMC experienced another grade diversion between the head grade and the modelled PCBC grade. The head grade yielded way above the model, while the model indicated that the cave’s underground grade was drastically depleting. Figure 79 shows old forecasts for 2015 with a drastic depletion of the grade below 0.5% Cu from 2015 onwards (Diering, 2015).

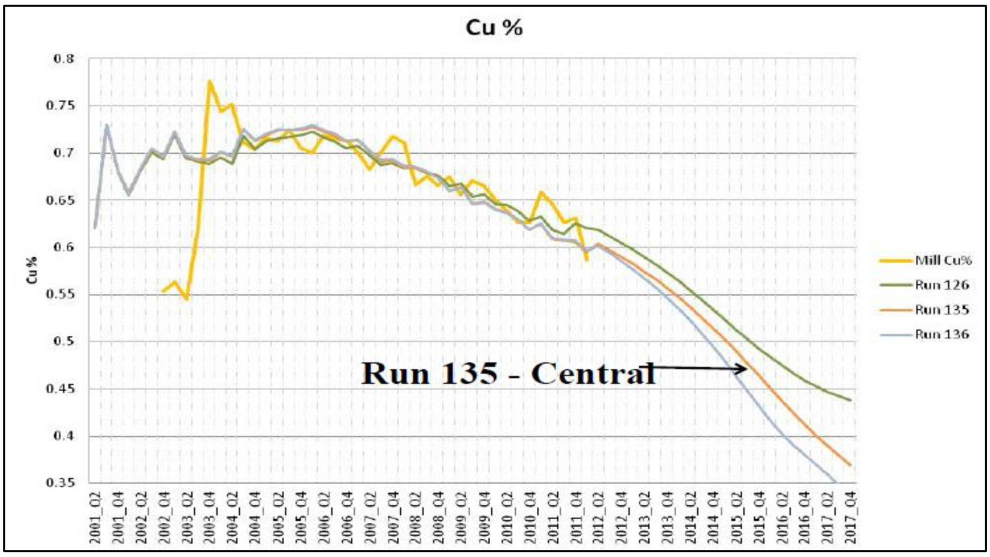


Figure 79: Quarterly indicative modelled grades from PCBC
Source: (Diering, 2015)

PMC's technical team consulted Dr Tony Diering to investigate this unanticipated scenario, in an attempt to understand what caused this grade behaviour and its interpretation thereof. The grade behaviour affected the draw control strategy and required a severe revision to anticipate the behaviour of the cave's grade by any means. The conundrum Palabora faced with higher than expected grades at the ending of Lift 1 was not expected. PMC needed to understand whether the achieved grades were short-lived or not. The higher head grade could be short-lived, and if true, the mine would ultimately face a copper-recession or copper-gap. The reason being is that PMC's Lift 2 mine progressed still in its development phase and was nowhere near any copper production yet. This section describes how PMC dealt with this grade divergence and covered the investigation and outcome.

9.2 The resource copper grade investigation

Diering (2013) described in detail key factors affecting the calculation of a mineable ore reserve and its computation, such as dilution sources, flow mechanisms, mine economics, excavation geometry, mining sequence, and residual material situations. Diering has more than 25 years of block caving experience (Diering, 2013). The resource model investigation in

2014 and 2015, created new insight into the elusive block cave behaviour of PMC's Lift 1 block cave.

Initial work on the 2015 resource model included the basis for modelling investigation. PMC revised the failure material, which flowed into the block cave from the surface (Diering, 2015). The failure material in the resource model had minimum copper values assigned to it, and the geology department investigated these assigned grades. In sections 2.12 there were indications of reserve losses from the failure, and Ngidi & Pretorius (2010) mentioned that the failure was practically waste material.

Hans-Dieter Paetzold, the 2015 geology specialist at PMC, used old open pit blast-hole data and supplemented the underground exploration drilling with these open pit information. In the process, a “de-clustering” technique combined the blast-hole and exploration data usage (Diering, 2015). The remodelling of the resource with this additional information allowed for better estimates of the upper cave portions, and failing material. The estimated metal content in the open pit failure zone was better estimated, with the modelling done for lithology, density and copper (Diering, 2015). The resource model before these additions had practically included more waste from the failure material. However, quantifying the current size of the failure zone, its interpretation, and influence would allow for the highest confidence levels possible.

9.3 Modelling of the failure zone and the estimated glory-hole position

The available annual Fly-Lidar surveys over the pit enabled the modelling of geometrical changes in failure over time. Wireframes created from the survey points enabled a failure area calculation of the annual displaced material and differences between successive years (Diering, 2015). Figure 80 shows the progressive views of the open pit failure used for calculating the failure volumes.

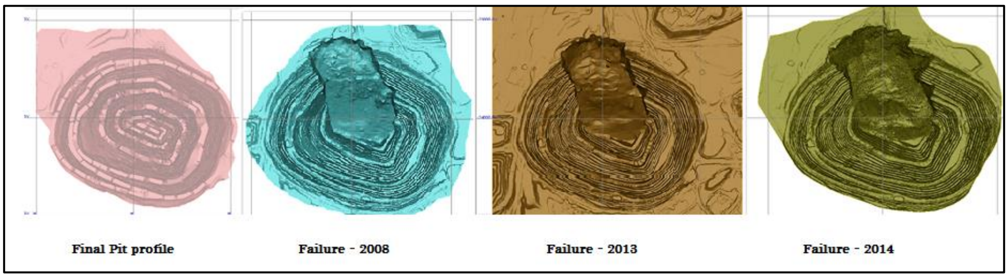


Figure 80: Successive views of the open pit failure
Source: after (Diering, 2015)

A theoretical but practical extrapolated construction of the failure base in the open pit enabled the calculation of each flyover's failure capacity. Although the real failure bottom or such surface extent of it was still unknown and covered with failure material, the joining of the original pit bottom and failure's top extent served as a logical but practical modelled floor (Diering, 2015). Diering (2015) outlined the cave's breakthrough and estimated demarcation from the failure material's shape and behaviour in the bottom of the open pit. Figure 81 illustrates the top of the cave, where the breakthrough was visible and demarcated in magenta (middle) and flooded with a blue outline (right).

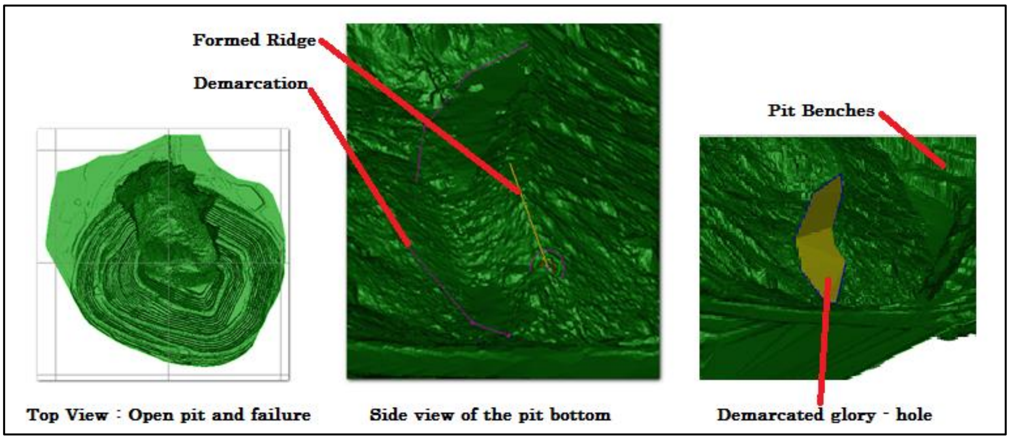


Figure 81: The top of the cave and demarcated glory-hole position
Source: after (Diering, 2015)

The estimated position of the top of the cave served as investigation regarding the influence it had on the draw points below it in the cave at the production level. By superimposing the outline on top of the cave, the affected draw points became identifiable, as Figure 82 indicates.

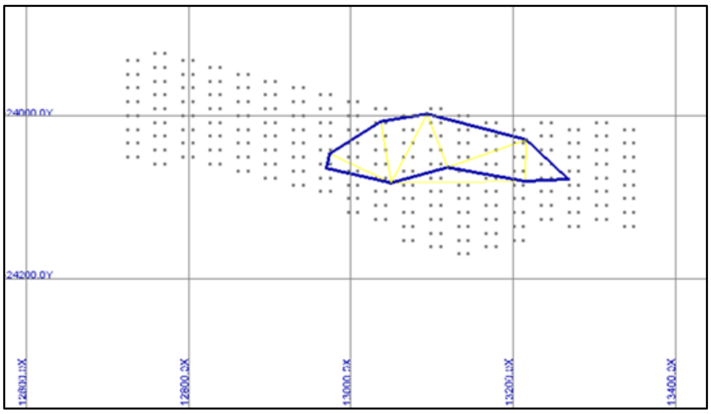


Figure 82: Top of cave superimposed over the draw points
Source: after (Diering, 2015)

The first attempted surface area of the failure floor resulted in an area which was not large enough, which Diering (2015) adjusted to extend from the failing crest to the glory hole. Figure 83 indicates the blue string lines, forming the applicable failure floor area, and where the floor and model were superimposed, the shortfall in floor surface area was visible. However, the far right picture in Figure 83 indicates the corrected purple string line floor, covering the entire applicable failure area. It is important to note that the higher grade material in the failure zone is at the base of the failure (closest to the glory hole) (Diering, 2015).

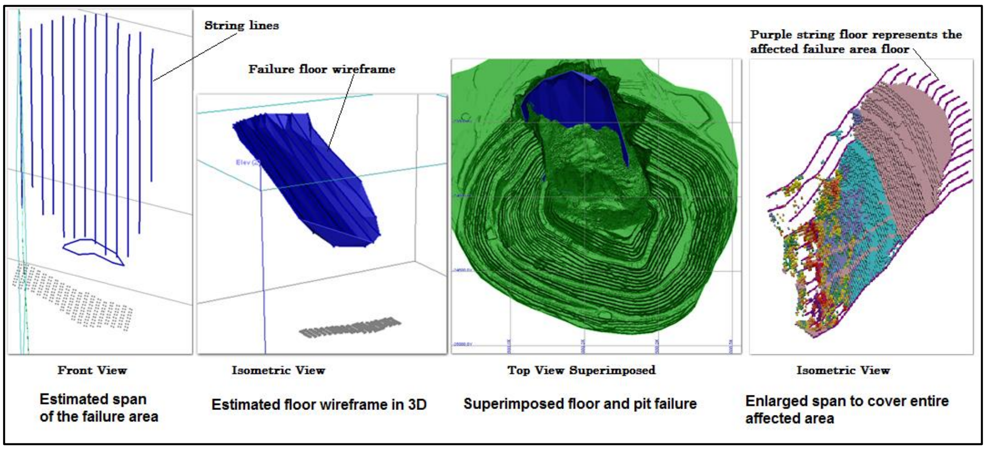


Figure 83: Views of the failure area floor
Source: after (Diering, 2015)

9.4 The concept of failure study

During the 2015 investigation, a different possible concept identified by Diering (2015) and the PMC Technical team resulted in PMC having more than one failure concept. The toppling concept showed that the 2015 grade situation and resultant grade representative dummy runs in PCBC, still matched insufficiently. The concept of failure zones led the team to demarcate the failure zone in the hope to identify better grade zones (Diering, 2015). The failure zone comprised of higher Cu grades at the base and lower Cu grades at the crest of the open pit. Diering (2015) divided the failure zone into different failure zones, ranging from top to bottom over the failure surface area (Diering, 2015). Figure 84 below indicates how this investigation studied different grade zones applicable to the specific failure material.

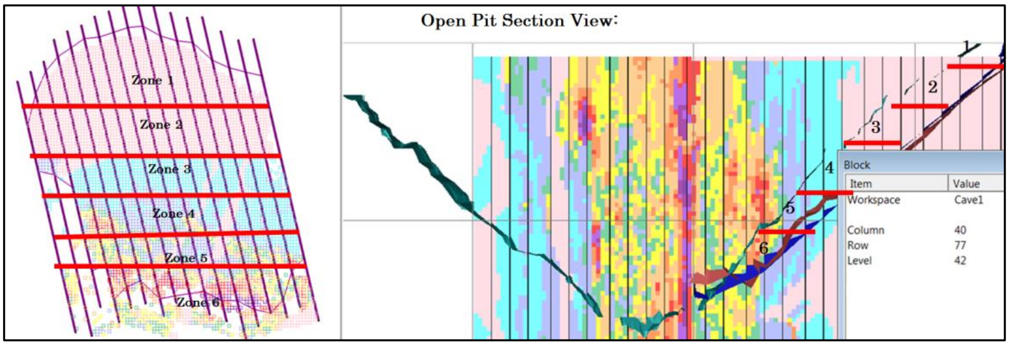


Figure 84: Concept of failure zones
Source: after (Diering, 2015)

The modelled progress shown in Figure 85 indicates the application of failure zone material to fail into the cave with a more northern overhang for its draw zone ellipsoids. The failure material before failure is coloured magenta in the picture a) where the volume shows in b) and the assigned grades with different colours is in c).

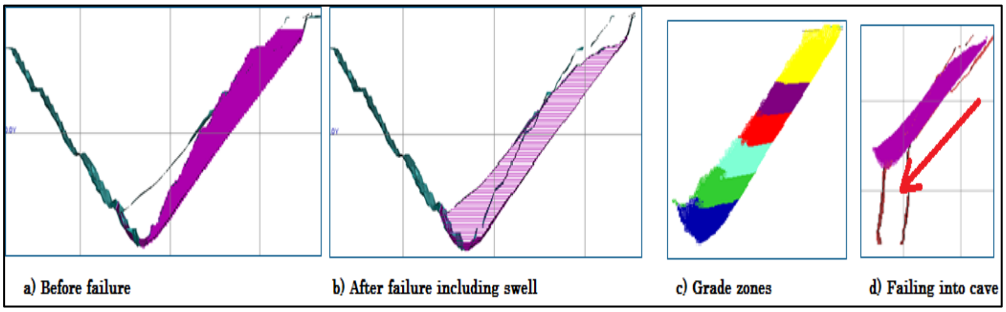


Figure 85: The before and after failure zone evaluated
Source: after (Diering, 2015)

Where the failure ultimately enters the cave or fails into the cave, the modelling represented such as per indicated right-hand picture d) above. The calculation of the swell resulted from the volume before, multiplied with the swell to equal the expected volume after (Diering, 2015). The calculated swell factor in the failure zone needed to compare with the total mined-to-date volumes. The calculated before and after volumes from the failure zone had to conserve both the tonnes and metal content (Diering, 2015). Diering (2015) evaluated the modelled failure, ran it in PCBC, and tested results against the current mined and remaining tonnes. The progressing model steadily took shape, and the investigation progressed to find a possible solution.

The evaluation of the construction indicated that with 165 Mt in the cave zone, 151 Mt of broken material was additional from the failure. The new total available tonnes of 316 Mt with 127 Mt mined over the LOM indicated 189 Mt remaining within the cave (Diering, 2015). By calculating the model in a straight-line depletion model, the results revealed that approximately 500 000 tonnes of Cu remained in the cave (Diering, 2015). The graph in Figure 86 indicates the straight-line results to deplete the entire cave, where mining would deplete the total Cu content to zero.

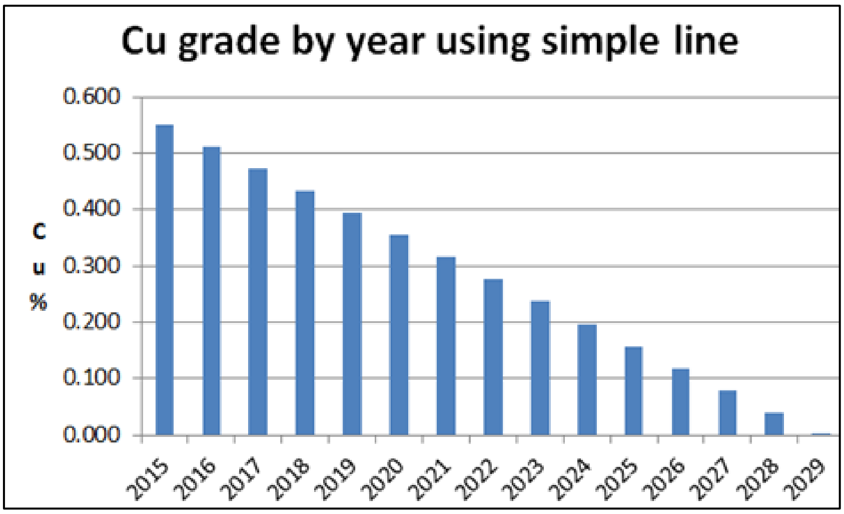
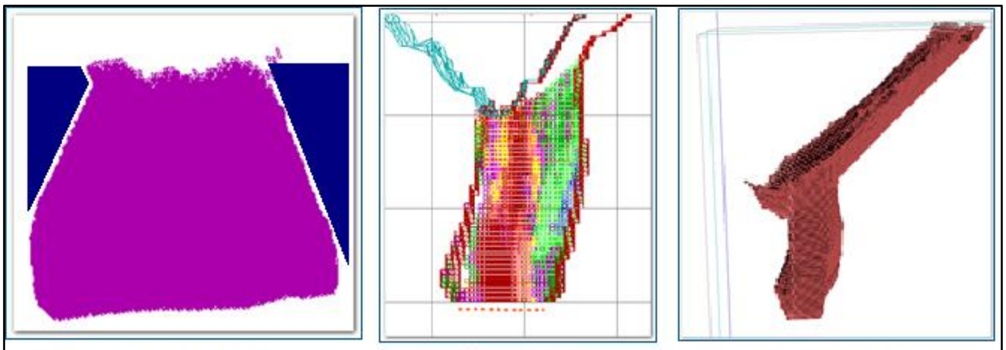


Figure 86: Straight-line depletion graph for Lift 1
Source: (Diering, 2015)

The straight-line model's metal content compared with the content and same volume within PCBC. The exclusion of blocks above the shoulders as per Figure 87 a) and representation of modelled failure in b) from c) indicates the progress of the investigation (Diering, 2015). Diering (2015) suggested that a very sudden reduction in the current grades at the time would be unlikely as there was still a significant amount of copper in the broken rock mass.



a) Blocks above shoulders excluded b) Mixed failure flow c) Progressive model

Figure 87: The failure volume modelled within PCBC
Source: (Diering, 2015)

The final process included the addition of the failure model to the overall resource model, where the constructed sliced blocks of the overall "new" cave model dummy run with the updated production history. The illustration in Figure 88 indicates the superimposed former cave profile and the amended final cave profile.

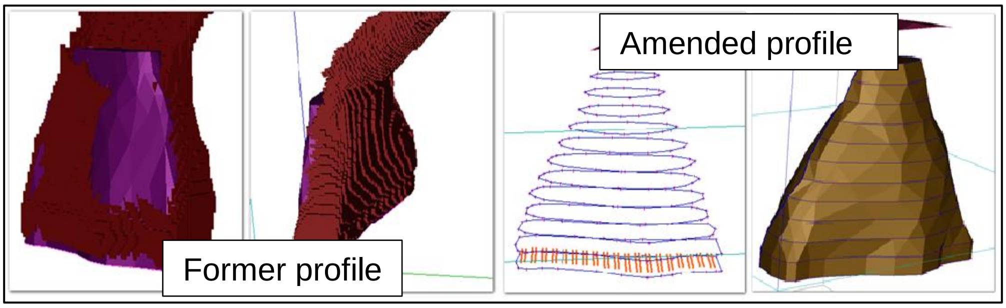


Figure 88: Reshaping the Lift 1 cave profile from indicated parameter adjustments
Source: (Diering, 2015)

The progress of the investigation was tested, and modelled in CA3D, where the Cu grade graphs produced for several PCBC runs, required more scrutiny (Diering, 2015). Figure 89 illustrates that the head grade and modelled grade improved, in that they correlated better in the mid-years of production, but they still deviated unacceptably in the later years of the LOM (Diering, 2015).

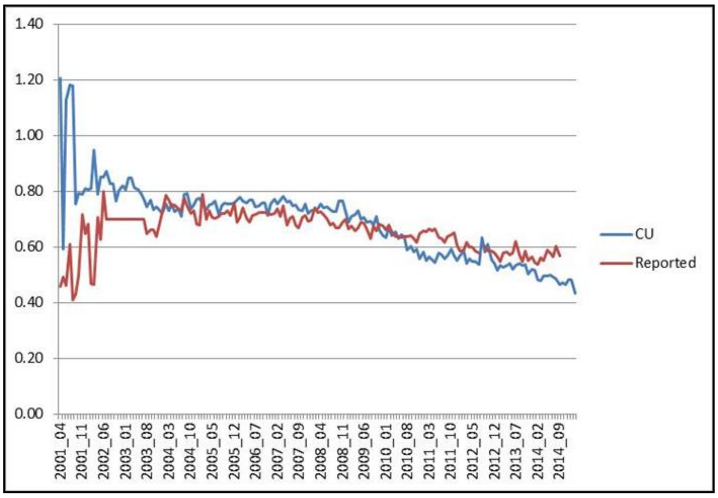


Figure 89: The modelled Cu grade versus the reported grade
Source: (Diering, 2015)

The toppling mechanism and parameter in the PCBC program were set to erode or “shave” the failure surface, which was not evident as per indication in Figure 90.

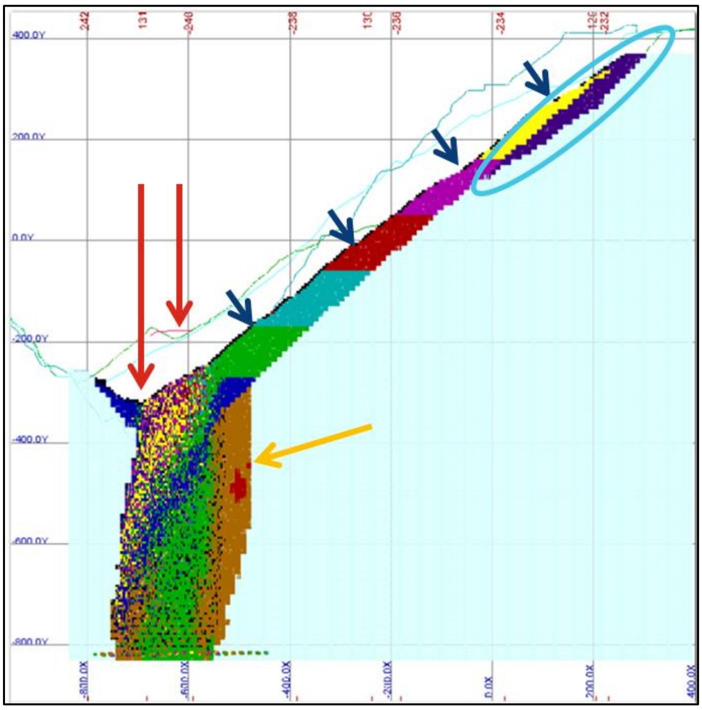


Figure 90: Deficiencies determined from the PCBC run and study
Source: (Diering, 2015)

Diering (2015) furthermore indicated that the subsidence zone appeared in the wrong place, with specific reference to the red arrows in Figure 90 above. The 2014 failure illustrated minimal movement in the cyan oval outline, and the cave back on the north side appeared poorly (yellow outline) (Diering, 2015). Shortcomings existed in this model, but the model produced valuable additional learnings.

9.5 The glacier flow concept

Diering (2015) reasoned from the shortcomings of the concept of failure study that a glacial flow concept for the failure area could have occurred. From previous glacier-forming conducted studies, snow deposits in the accumulation zone at high altitudes. As the snow transforms into “Névé” and becomes increasingly dense through time, air pores disappear to form ice. With more snow depositing at the hilltops, the upper layers become heavy, and gravity pulls the upper layers downward, to lower altitude areas (Pauly, 2014). Figure 91 shows the flow direction of a glacier, where it slides downwards on a mountain slope.

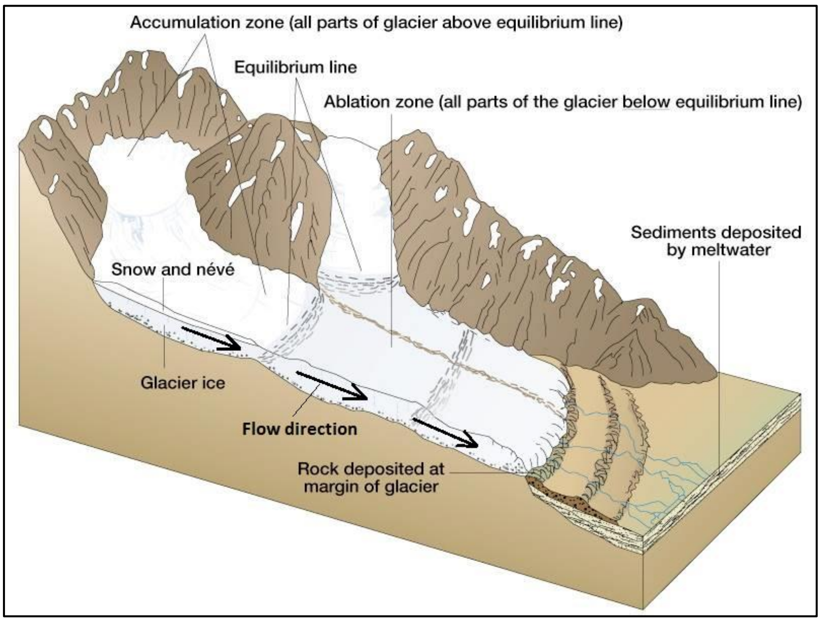


Figure 91: Glacier formation dynamics
Source: after (Pauly, 2014)

Diering (2015) indicated from the learnings in the concept of failure model how the toppling reflected insignificantly and ineffective to the actual failure zone and from its results. The glacial flow concept study required parameter changes and insertions within PCBC to simulate glaciers. A template mixing flow model enabled Zone 6 and Zone 5 in Figure 84 to “flow” into the top of the cave model (Diering, 2015).

Additionally, the cave back outlines required adjustment, and the draw cone modelling needed to match the cave back. The Diering (2015) study indicated that all “not-in-situ” material required a full removal from the block model. Diering (2015) adjusted the draw cone positions outwards to create tilted draw cones or inclined draw cones as is shown in Figure 92.

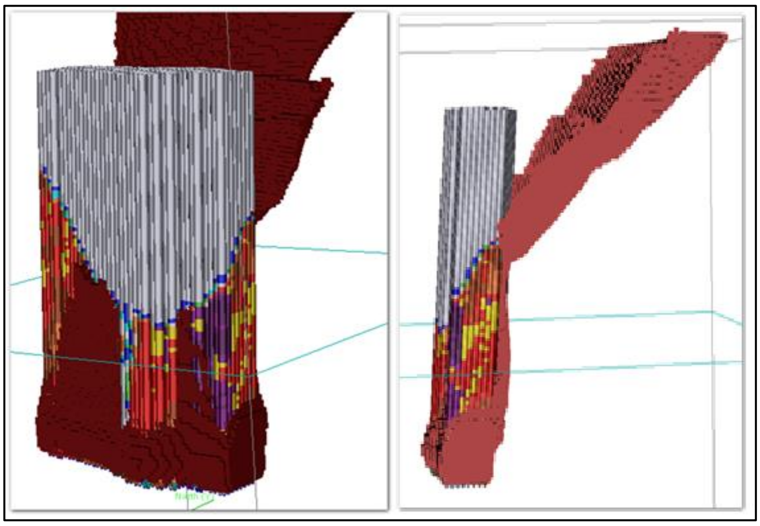


Figure 92: Inclined draw cones
Source: (Diering, 2015)

The inclined draw cones resulted from moving the top positions of the draw cones outwards, to align with the actual modelled breakthrough position. Figure 93 indicates the required and amended adjustment of the draw cone positions (Diering, 2015).

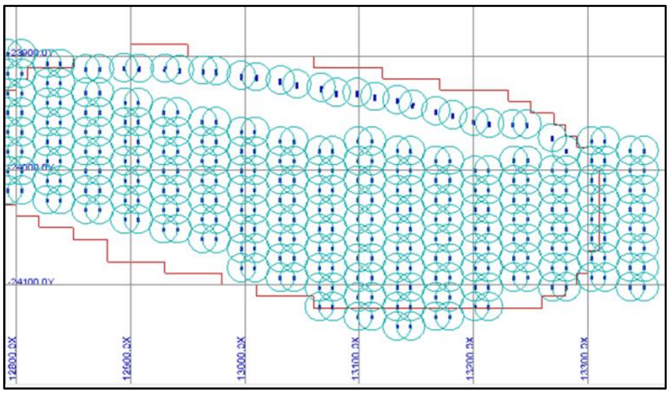


Figure 93: Adjustment of the draw cone positions at the top of the cave
Source: (Diering, 2015)

The conceptual cave back model indicated from the comparison with the broken material calculations that it required further refinement. Figure 94 a) indicates the concept model, b) indicates the cave zone ready to feed into the modelled draw zone. Figure 94 c) illustrates the variance in volume where the cave back size required refinement (Diering, 2015).

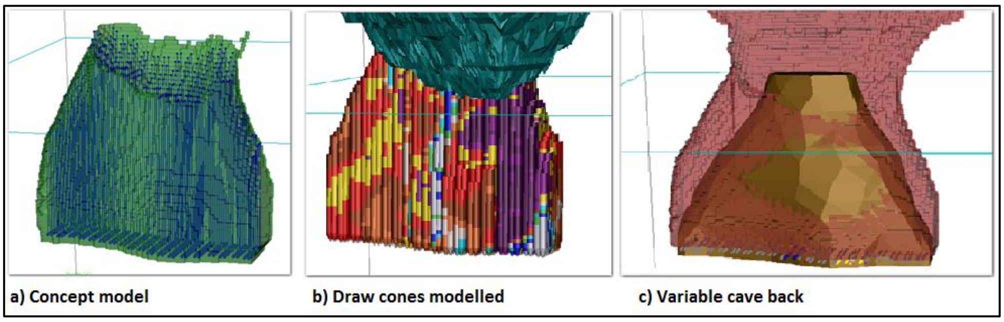


Figure 94: Concept model of modelled draw cones and variable cave back
Source: after (Diering, 2015)

The flow of material simulated in the glacial flow resource model produced encouraging grade results, in the attempt to perfect the actual scenario in the model.

Several runs with parameter refinements within PCBC have produced results ranging from not good to best-fit figures when compared against the straight-line depletion model. The outcome of the refined and glacial flow concept model forecasted best results between the two concepts and injected two more years into the life of Lift 1 (Diering, 2015). Figure 95 below indicates the resultant graph, where the reported sampling grades in the Lift 1 cave correlated best with the straight-line model and the PCBC adjusted resource model.

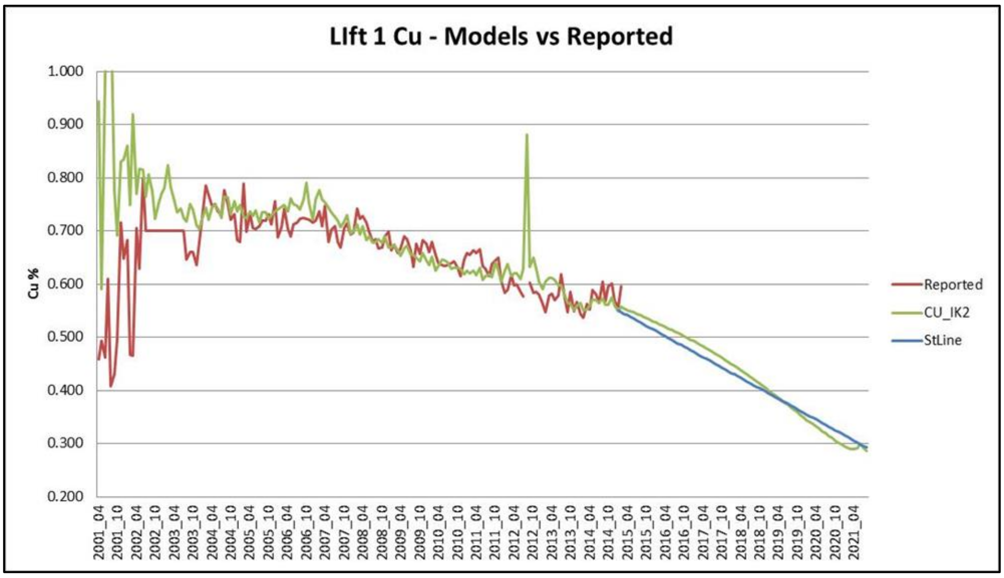


Figure 95: PCBC modelled and forecasted grades
Source: (Diering, 2015)

9.6 Conclusion

The crown pillar broke through in 2004 whereby the weak zones in the open pit lost its footing and started a slope failure. This external material bled into the current cave, and the previous block model did not include external sources (Marshall, 2012). The Diering (2015) study identified several additional factors that were not forming part of the resource model, which revealed an unexpected sustaining resource grade at the time.

Higher unexpected actual grades from sampling values in the cave versus the predicted planned lower grades from PCBC required an investigation of the cave behaviour. The technical team of PMC had to review the status of the block cave and discovered new information to refine the resource model with modifying parameters. The parameters included the refinement of the inclination of the draw cones, the area of block model coverage, blast-hole, and drilling data from the open pit and an estimated failure floor to calculate the failure volume (Diering, 2015).

With the influx of the failure zone material into the block cave, which reported at the production level, the resource model's investigation and reconciliation followed. The reconciliation followed two conceptual studies. The first conceptual study has indicated that if flow and toppling parameters were only applicable, the results were not meeting the actual and expected situation for Lift 1. Ultimately, the inclusion of the actual bench mapping and drill assay blast information into the resource model improved the model.

The second concept of a glacial flow perspective, followed only after the first concept revealed unsatisfactory variances. The previous and erroneously omitted failure's grade values, previously modelled as having lower grade material, resulted in the indicative head grade and model grade departing from one another. The departing grades fundamentally led to lower predicted grades in the resource model, which featured from the PCBC data.

The failure zone’s bottom surface, which is still undefinable, can be controversial but estimated and calculated into a best “practical floor”. This practical floor currently defines the calculated failure material volumes and open pit valuation data and geology assigned to it. The Diering (2015) investigation added at least two more years to the original expected LOM for Lift 1. This injection reduced the copper-gap in a tough economic time and where Lift 2 is still in its development phase. Further refinements to the 2015 resource model are possible with further investigation.

10 CONCLUSION

Palabora Mining Company started copper prospecting in 1952. Initially, Palabora was an open pit mine, which transitioned to underground block cave mining around 2002. The Palabora Underground Mining Project developed a world first cave mine in competent rock and used the crinkle cut design in its undercut level for caving. Despite the sampling and drilling from the exploration tunnel below the open pit, inadequate drilling for the underground resource existed. This drilling shortfall contributed to many grade diversions between the head grade and model grades during the life of mine.

Many resource studies covering the life of mine had to seek information to verify and adjust the resource model where it was applicable. The early mining and draw control was poorly managed and had manual input sheets. The LHD operators completed the shift mucking from their recollection after shifts. The underground draw control upgraded to an electronic tag reading system, where the system registered machines at draw points electronically in a database. This database is more accurate than a manual system while no system errors occur. These systems are not flawless, but provide reliable results in the draw control. This fundamental shortcoming contributed to the initial drawing inaccuracies, which affected the resource of Palabora Mine.

In 2004, the technical team of Rio Tinto investigated the grade discrepancy between the head grade and modelled grade. Software and system exchanges provided best practical means to monitor and manage the block cave. The internationally recognised block cave system PCBC allowed for back calculations and enhanced results. Data transfer from Datamine into PCBC resulted in a successful but strategic task, to ensure that no data loss occurred in the process. All the old files, such as geology faults and dykes accrued to the resource block specifics and dimensions, ensured a steady workable platform. The geological database and dykes formed a large part of the revised resource work. Each component of the previous studies highlighted problematic areas and rectifying measures with duly analysed and appropriate actions.

The geographical orientation of the FS blocks mismatched the draw column positions. The new DBM model allowed for dolerite updates and the alignment of draw columns and model blocks. In a cautious but systematic approach, the new software models corrected the resource errors to best-fit practices. Not only were the geographical errors addressed, but modelled grades were broken down and tested through various available particle flow models and multiple kriging options. The block size refinements had undergone several stages of scrutiny, where initial blocks of 20 m x 40 m x 60 m had inadequate data. The inadequate data were responsible for the poor confidence of the resource grades. This reduction of block sizes to 20 m x 20 m x 20 m produced better comparable results than the previous conflicting grades. The lashed tonnages required updates in the software to produce realistic results for comparisons. Hence, the initial grade shortfall against the modelled grade highlighted several concerns. The shortfall in grade related to the first overestimated grade in the FS model. Furthermore, the actual dolerite dilution was not practically representable within the models, which made the resemblance challenging to compare apples with apples. The DBM assigned dolerite percentages to each block, which made the dolerite dilution effect more practical with reasonable resource grades and improved draw control reporting.

The underground sampling size remained inadequate to be representative of the total mucked and lashed tonnes and proved to be biased towards fines. However, the conducted sampling within the production area allowed for draw control and internal referencing, which proved alternatively to be an excellent leading indicator. The Lift 1 Palabora block cave dealt with almost every possible confrontation, of which the most significant flaw in the mine appeared to be the sparse drilling coverage to establish a confident Lift 1 resource model.

Minor identified contributing factors such as variable bucket factors or significant factors with many hypotheses arose over Palabora's block cave, and the mine's end was even more uncertain. A massive blow to the block cave occurred as a slope failure at the open pit above the block cave. The failure's waste material contributed to excessive losses in reserves. The model grade and the head grade contrasting trends continued after 2005, despite various corrections and improvements to the model. Where the head grade departed from the PCBC grade, Snowden (2010) could not calibrate the PCBC values without adequate smart markers. Palabora Mine anticipated revising its Lift 1 resource annually due to the poor drilling coverage. The revisions incorporated studies over time made by several individuals and companies who incorporated new evidence as information became available over time.

The head grade trended slightly below the modelled grade, whereas in June 2008 and around 2015 the modelled grade dipped below the head grade. Thus, an inverse of grade anomalies occurred together with correlating grade behaviours during the LOM of Lift 1.

During the 2015 study, the head grade sustained its position above the modelled grade, and a need to investigate the grade behaviour was critical. The investigation led by Dr Diering followed different concepts of failure studies. The studies revealed that toppling has erroneously misinterpreted the failure zone. The failure zone slid downwards to the bottom of the open pit, and Diering (2015) included the actual values from the open pit mining to the resource model. Previously the failure zone material, which bled into the block cave, had very low copper grade values. With the additional values and glacial flow parameters adjusted within PCBC, the refined resource model indicated results that are more acceptable and was more practical with regards to the interaction of the cave and the failure. The adjustment within the resource allowed Palabora Mine to extend the LOM of Lift 1 with two additional years. This injection of more copper life came at a very welcoming time for Palabora Mine, where the new Lift 2 mine was still in its development phase.

PMC anticipated extending its copper production beyond 2030. Over the years, the Palabora Mine succeeded with limited knowledge to adequately maintain, manage, and control its copper resources as an ongoing study.

11 RECOMMENDATION

Identified frozen material at the pit base not moving or indicating to report into the current cave could be refined in the model. Some of the expected higher-grade estimates locked up on this bottom surface lie dormant since the failure happened. The pit failure zone and the perceived failed floor could receive more strict attention, and its model can improve as new information becomes available.

The cave back can be refined more accurately where modelling comprises of the updated height of draw indications and the surface indication of the northern propagation direction. The block model can improve via alternative interpretations in PCBC to refine the grades between the LOM estimate and actual head grade. Additional studies conducted will play a critical role at the end of the Lift 2's LOM when Lift 1 and the surface material reports into the second block cave of PMC. The Lift 2 environment should consist of adequate smart markers, which will produce tangible results for the flow of material and where the Lift 2 resource would eliminate the shortfalls of the Lift 1 cave.

Mines such as the Grasberg block cave, Finsch Mine, Andina Mine, and Venetia Mine have similar interactions between the open pit and block cave operations. There is currently little-written knowledge on the economic effects of failures and external material exercises on the grades within these block caves. Ultimately, more information on how several mines deal with their slope failures and how the influence of failures affect their resource grades will be beneficial for future studies. The lessons learned will form excellent topics for future students doing research.

This project report also mentioned the phenomenon of fracture banding in caving mines, which requires further studies and investigation. The additional investigations can contribute towards the effect fracture banding had on the slope failure, cave management, and resource grade. Fracture banding could assist in refining the cave back and the block cave model optimally regarding its resource management and grades.

This project report hence invites other candidates in the field to extend the knowledge base of this report, where more resource work after 2015 at PMC may focus on the future Lift 2 operations.

12 REFERENCES

Ahmed, H.M., 2009. *Influence of block cave mining on pit slope deformation mechanisms*. Vancouver: The University of British Columbia The University of British Columbia.

Ahmed, H.M., Eberhardt, E. & Dunbar, W.S., 2014. Interaction between block caving and rock slope deformation kinematics. *Mining Technology*, 123(3), pp.3-13.

Atkins, R., 2013. *Lift 1 Fragmentation Study*. [Power Point slide show] Phalaborwa: Rio Tinto.

Barnet, W.P., 2003. Geological control on slope failure mechanisms in the open pit at Venetia Mine. *South African Journal of Geology*, 106(2-3), pp.149-164.

Board, M. & Pierce, M.E., 2009. A Review of Recent Experience in Modelling of Caving. In *Proceedings of the International Workshop on Numerical Modeling for Underground Mine Excavation Design*. Atlanta, USA, 2009. NIOSH.

Brummer, R.K., Li, H., Moss, A. & Casten, T., 2006. The transition from open pit to underground mining: an unusual slope failure mechanism at Palabora. In *In Proceedings of International Symposium on Stability of Rock Slopes in Open Pit Mining and Civil Engineering*., 2006. The South African Institute of Mining and Metallurgy.

Butcher, R.J. & Jenkins, P.A., 2006. Subsidence Effects Associated With the Block and Sub Level Caving of Massive Orebodies. In *2nd International Seminar: Strategic versus tactical approaches in mining*. Perth, 2006. Australian Centre for Geomechanics.

Calder, K., 2006. *Rio Tinto Copper Projects Block Cave Design Workshop*. [Online] Salt Lake City: Rio Tinto Technical Services.

Calder, K., Townsend, P. & Russell, F., 2000. Palabora Underground Mine Project. In *Proceedings of the MassMin 2000 Conference*. Brisbane, 2000. The Australian Institute of Mining and Metallurgy.

Carlson, G., Tyler, D., DeWolfe, C. & Lorig, L., 2004. Understanding gravity flow for mix and dilution modelling at Henderson Mine. In Karzulovic, A. & Alvaro, M., eds. *Proceedings MassMin 2004*. Santiago, 2004. Instituto de Ingenieros de Chile.

Chitombo, G.P., 2010. Cave mining: 16 years after Laubscher's 1994 paper 'Cave mining - state of the art'. *Mining Technology - Australian Centre for Geomechanics*, 119(No 3), pp.132-141.

Cook, A., 2011. *Draw Point Sampling Observations - Palabora*. Phalaborwa: PMC internal Memo.

Cumming-Potvin, D., Wesseloo, J., Jacobsz, S.W. & Kearsley, E., 2016. Fracture banding in caving mines. *The Journal of the Southern African Institute of Mining and Metallurgy*, 116(August), pp.753-761.

Cundall, P., 2008. *Recent Advances in Numerical Modelling for Large-Scale Mining Projects*. Minneapolis: Australian Centre for Geomechanics Newsletter.

Dassault Systemes, 2013. *Leading Block Cave Planning and Scheduling Solution*. [Online] Available at: http://www.3ds.com\Geovia\PCBC [Accessed 26 June 2015].

De La Comisión, M., Massa, J.V., Pierce, M. & Tavantzis, A.T., 2012. *Study of the gravity flow mechanisms at Goldex by means of a physical model*. Thesis. Santiago de Chile: University of Chile.

Diering, T., 2007. Template Mixing An Alternative Depletion Engine for Block Cave Scheduling. In *Proceedings APCOM*. Santiago, 2007. APCOM.

Diering, T., 2010. PC-BC: A Block Cave Design and Draw Control System. In Chitombo, G., ed. *MassMin 2000 Proceedings*. Brisbane, Australia, 2010. The Australasian Institute of Mining and Metallurgy.

Diering, T., 2013. *Reserve Estimation for Block Cave Mines Using Geovia PCBC*. White Paper. Vancouver: Dassault Systemes Geovia / PCBC.

Diering, T., 2015. *Palabora Lift 1 PCBC update*. [Powerpoint presentation].

Du Plessis, L. & Martin, D.C., 1991. Numerical Modelling Studies for Design of High Rock Slopes at Palabora Copper Mine. In *7th ISRM Congress. International Society for Rock Mechanics*., 1991.

Dunstan, G. & Popa, L., 2012. Innovative Cave Establishment Practices at Ridgeway Deeps. In *MassMin 2012*. Salsbury, 2012.

Duplancic, P. & Brady, B.H., 1999. Characterisation of caving mechanisms by analysis of seismicity and rock stress. In Vouille, G., Berest, P. & Balkema, L.E., eds. *Proceedings of the 9th ISRM Congress on Rock Mechanics*. Paris, 1999.

Fernandez, F., Evans, P. & Gelson, R., 2010. Design and implementation of a damage assessment system at Argyle Diamond's block cave project. In Potvin, Y., ed. *Caving*

2010: Second International Symposium on Block and Sublevel Caving. Perth, 2010. Australian Centre for Geomechanics.

Glazer, S.N. & Hepworth, N., 2006. Crown pillar failure mechanism - case study based on seismic data from Palabora Mine. *Mining Technology*, 115(No 2), pp.75-84.

Glazer, S.N. & Townsend, P.A., 2010. Initial and east cave breakthrough events at Palabora Mining Company. In Potvin, Y., ed. *Caving 2010, Proceedings of the Second International Symposium on Block and Sublevel Caving*. Perth, Australia, 2010. Australian Centre for Geomechanics.

Gustafsson, P., 1998. ISSN 1402-1544 *Waste rock content variations during gravity flow in sublevel caving, Analysis of full-scale experiments and numerical simulations.* Doctoral Thesis. Sweden: Lulea Tekniska Universitet Lulea Tekniska Universitet.

Halim, A.E., 2004. 3D large-scale physical modelling for studying interactive drawing and draw point spacing in Block Caving Mines. In *Julius Kruttschnitt Mineral Research Centre International Student Conference*. Queensland, 2004. University of Queensland.

Heslop, T.G., 1983. The application of interactive draw theory to draw control practice in large chrysotile asbestos mine. In *Mining and Metallurgical Operations in Zimbabwe*. Harare, Zimbabwe, 1983.

Howson, M., 2003. *Update and Analysis of Palabora's Geological Databases*. Bristol, UK: Rio Tinto Technology Technical Services.

Howson, M., 2003. *Update and Analysis of Palabora's Geological Databases*. Bristol, UK: Rio Tinto Technology Technical Services.

Howson, M., 2004a. *Proposal to PMC to Revise the Geological Data and Resource / Reserve models*. Bristol, UK: Rio Tinto Technology Technical Services.

Howson, M., 2004b. *Resources, Reserves and Geotechnical Data at Palabora Underground*. Bristol, UK: Rio Tinto Technical Services.

Howson, M., 2004c. *Revision of Palabora Underground Copper and Geology and Resources Data 2004*. Bristol, UK: Rio Tinto Technical Services.

Howson, M., 2006. *Revision of Palabora underground copper resources and reserves 2004 to 2005*. Bristol, UK: Rio Tinto Technical Services.

Kuschke, O.H. & Tonking, M.J.H., 1971. Geology and Mining operations at Palabora Mining Company Limited, Phalaborwa, NE Transvaal. *Journal of the South African Institute of Mining and Metallurgy*, 1, pp.12-23.

Kvapil, R., 1992. *Sublevel Caving*. Littleton, Colorado: SME Mining Engineering Handbook.

Laubscher, D.H., 1994. Cave mining - state of the art. *The South African Institute of Mining and Metallurgy*, 94(10), pp.279-293.

Laubscher, D.H., 2000. A Practical Manual on Block Caving. In *Block Caving Manual*. Queensland: The University of Queensland. pp.1-30.

Li, Z. et al., 2015. A case study integrating numerical simulation and GB-InSAR monitoring to analyse flexural toppling of an anti-dip slope in Fushun open pit. *Engineering Geology*, 197, pp.20-32.

Marshall, K., 2012. *Cave Induced Surface Subsidence*. Phalaborwa: PMC.

Moss, A., Diachenko, S. & Townsend, P., 2006. Interaction between the block cave and the pit slopes at Palabora Mine. *The Journal of The South African Institute of Mining and Metallurgy*, 106(7), pp.479 - 484.

Ngidi, S.N. & Pretorius, D.D., 2008. Cave management ensuring optimal life of mine at Palabora. In *MassMin 2008: In Proceedings of the 5th International Conference and Exhibition on Mass Mining*. Luleå, Sweden, 2008. Luleå University of Technology Press.

Ngidi, S.N. & Pretorius, D.D., 2010. Impact of poor fragmentation on cave management. In Potvin, Y., ed. *Second International Symposium on Block and Sublevel Caving*. Perth, 2010. Australian Centre For Geomechanics.

Pauly, M., 2014. *Canadian Cryospheric Information Network - Glacier Formation*. [Online] Available at: https://ccin.ca/home/ccw/glaciers/formation [Accessed 24 December 2017].

PMC, 2007. *Lift I modelling*. [Online] Phalaborwa: PMC Geology Department.

PMC, 2016. *Annual report for mineralised inventory, mineral resources and ore reserves*. Competent persons report. Phalaborwa: Palabora Mining Company PMC.

Reyes-Montes, J.M. et al., 2010. Microseismic tools for the analysis of the interaction between open pit and underground developments. In Potvin, Y., ed. *Caving 2010, Proceedings of the Second International Symposium on Block and Sublevel Caving*. Perth, 2010. Australian Centre for Geomechanics.

Richards, D., 2004. *Slope Failure Dilution Modelling Notes, December 2004*. Phalaborwa: Gemcom Africa (Pty) Ltd.

Rio Tinto Technical Team, 2011. *Lift II Order of Magnitude Study - Concept document Section 13 Orebody knowledge*. Phalaborwa: Rio Tinto.

Sainsbury, B., 2010. Sensitivities in the numerical assessment of cave propagation. In *Proceedings 2nd International Symposium on Block and Sublevel Caving*., 2010. Itasca Australia Pty Ltd and The University of New South Wales.

Sainsbury, B., 2012. *Phd. A model for cave propagation and subsidence assessment in jointed rock masses*. New South Wales: University of New South Wales.

Sainsbury, B., Pierce, M.E. & Mas Ivars, D., 2008. Analysis of Caving Behaviour Using a Synthetic Rock Mass - Ubiquitous Joint Rock Mass Modelling Technique. In Potvin, Y., Carter, J., Dyskin, A. & Jeffrey, R., eds. *Proceedings of the 1st Southern Hemisphere International Rock Mechanics Symposium*. Perth, Australia, 2008. Australian Centre for Geomechanics.

Sainsbury, D.P. et al., 2016. Caving-induced Subsidence Behaviour of Lift 1 at the Palabora Block Cave Mine. In *Seventh International Conference & Exhibition On Mass Mining*. Sydney, 2016. MassMin 2016.

Severin, J.M. & Eberhardt, E., 2012. Influence of stress path during the transition from open pit to block cave Mine: A Palabora example., 2012. MassMin2012.

Severin, J.M., Eberhardt, E. & Woo, K.S., 2010. Influence of major fault zones on 3D ground deformations caused by open pit block cave interactions. In Potvin, Y., ed. *Caving 2010, Proceedings of the Second International Symposium on Block and Sublevel Caving*. Perth, 2010. Australian Centre for Geomechanics.

Shekhar, G., Gustafson, A. & Schunnesson, H., 2016. *Draw Control Strategy and Resource Efficiency in Sublevel Caving*. Research Report. Sweden: Lulea University of Technology.

Snowden, 2010. *Palabora Mining Company Limited: Head Grade Audit*. Johannesburg: Snowden Johannesburg Office.

SRK Consulting, 2011. *Rio Tinto Corporate Assurance Internal Audit : PMC*. Audit report. Phalaborwa: Rio Tinto.

Steffen, S. & Kuiper, P., 2011. *Maximising ore recovery and the implication for cave monitoring and management practices - Application of the smart marker system in block and Sub-Level mines.* [Online] Available at: http://elexonmining.com/wp-content/uploads/Maximising-ore-recovery.pdf [Accessed 27 November 2017].

Steward, C.A., Allman, A. & Hall, B.E., 2010. Block cave optimisation - a value-driven approach. In Potvin, Y., ed. *Second International Symposium on Block and Sublevel Caving*. Perth, Australia, 2010. Australian Centre for Geomechanics.

Studio 522 Productions, Inc., 2016. *Freeport McMoran - Grasberg Mineral District*. [Video screen shots] Available at: http://studio522.com/freeport_grasberg/ [Accessed 26 November 2017].

Walker, S., 2014. *Engineering and Mining Journal*. [Online] Available at: http://www.e-mj.com/features/4259-block-caving-mining-specialization.html#.WlTVm66WZhE [Accessed 03 January 2017].

Woo, K.-S., Eberhardt, E., Elmo, D. & Stead, D., 2013. Empirical investigation and characterisation of surface subsidence related to block cave mining. *International Journal of Rock Mechanics & Mining Sciences*, 61, pp.31-42.

Xu, N., Zhang, Y. & Mei, G., 2006. A numerical modelling method for predicting pit slope deformation and failure caused by underground mining under an open-pit. Beijing, China, 2006. Montreal Canada: International Society for Rock Mechanics 2015.