AN INVESTIGATION INTO THE PROCESS MINERALOGY OF THE MERENSKY REEF AT NORTHAM PLATINUM LIMITED

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MEarthSc (Oxon)

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"Now I know in part; then I shall know fully, even as I am fully known."

- 1 Corinthians 13:12b

Declaration

I declare that this thesis, submitted for the degree of Master of Science in Engineering at the University of Cape Town, is my own work and has not been submitted prior to this for any degree or publication, at this university or any other institution.

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Christopher Paul Brough

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SYNOPSIS

This dissertation has investigated the links between the mineralogical characteristics of the Normal, NP2 and P2 Merensky reefs at Northam Platinum Ltd and their flotation performances. Flotation performance was measured in terms of the grade and recovery of the base metal sulphides (chalcopyrite, pentlandite, pyrrhotite and total sulphide) as well as through investigation of mass-water recoveries.

The Merensky reef shows considerable lithological variability both between and within mines. Northam Platinum Ltd is an example showing extensive reef variability within its own mining lease. Three common reef types were selected for investigation in this thesis: Normal reef, representing standard elevation Merensky reef, the P2 or pothole reef representing the full pothole reef and the NP2 reef or shallow pothole reef which is a transitional reef development located stratigraphically between Normal and P2 reefs.

The Merensky reef is a compound deposit consisting of two chronologically separate packages; namely the Merensky Cyclic Unit (MCU) which is the same across the three reefs and a pre-MCU footwall which varies between the three reef types. This variation in footwall mineralogy resulted in different footwall silicate textures as well as different base metal sulphide developments between the three reefs. The NP2 reef with an aluminosilicate pre-MCU footwall showed no silicate grain coarsening and only two sulphide textures. The Normal and P2 reefs with ferromagnesian footwalls showed silicate grain coarsening into a distinct pegmatitic texture, and partial sulphide re-mobilisation, with four sulphide textures observed, including fine and very fine grained sulphides.

For the NP2 reef, the presence of relatively simple sulphide textures led to optimum sulphide liberation and recovery of chalcopyrite, pentlandite, pyrrhotite and total sulphides. Furthermore, it was found that for a given particle size distribution the NP2 reef reported the shortest laboratory milling time, which was probably related to

its high plagioclase content. Coupled with this short milling time was the achievement of optimum liberation (>90% sulphide recovery) at a standard grind. In contrast, the Normal and P2 reefs both required a finer grind to improve recovery. The implication of these laboratory milling times is that the expected industrial throughput of NP2 ore relative to the Normal and P2 ore should be much higher.

The presence of greater amounts of ferromagnesian minerals (particularly orthopyroxene) and associated alteration minerals (e.g. talc, serpentine) within the Normal and P2 reefs has resulted in higher mass recoveries. This is due to inadvertent flotation of gangue through mechanisms such as the association with hydrophobic alteration minerals, as well as increased recovery through entrainment due to the froth stabilising effect of minerals such as talc. In contrast the largely unaltered NP2 reef was found to have much lower mass recoveries, despite being the only reef type to show copper activation of gangue minerals.

Overall, it was found that the NP2 reef was the best reef to process, producing optimum chalcopyrite, pentlandite, pyrrhotite and base metal sulphide recoveries and highest grades. The Normal reef is considered the most problematic to process, due to a more complex sulphide development, and higher degrees of alteration whereas the performance of the P2 reef is slightly improved relative to the Normal reef, primarily due to higher head grades.

It is recommended that further characterisation of PGM type, location and association to ascertain the similarities or differences between the three reefs, as well as evaluating the differences in flotation performance work be performed. The extension of process mineralogy investigations on the Merensky reef, to look at the differing processing performances, in terms of grinding and flotation, of the Merensky contact reef, thin pegmatitic reef, and thick pegmatitic reef, as they are developed within the Rustenburg facies is also likely to be of great benefit.

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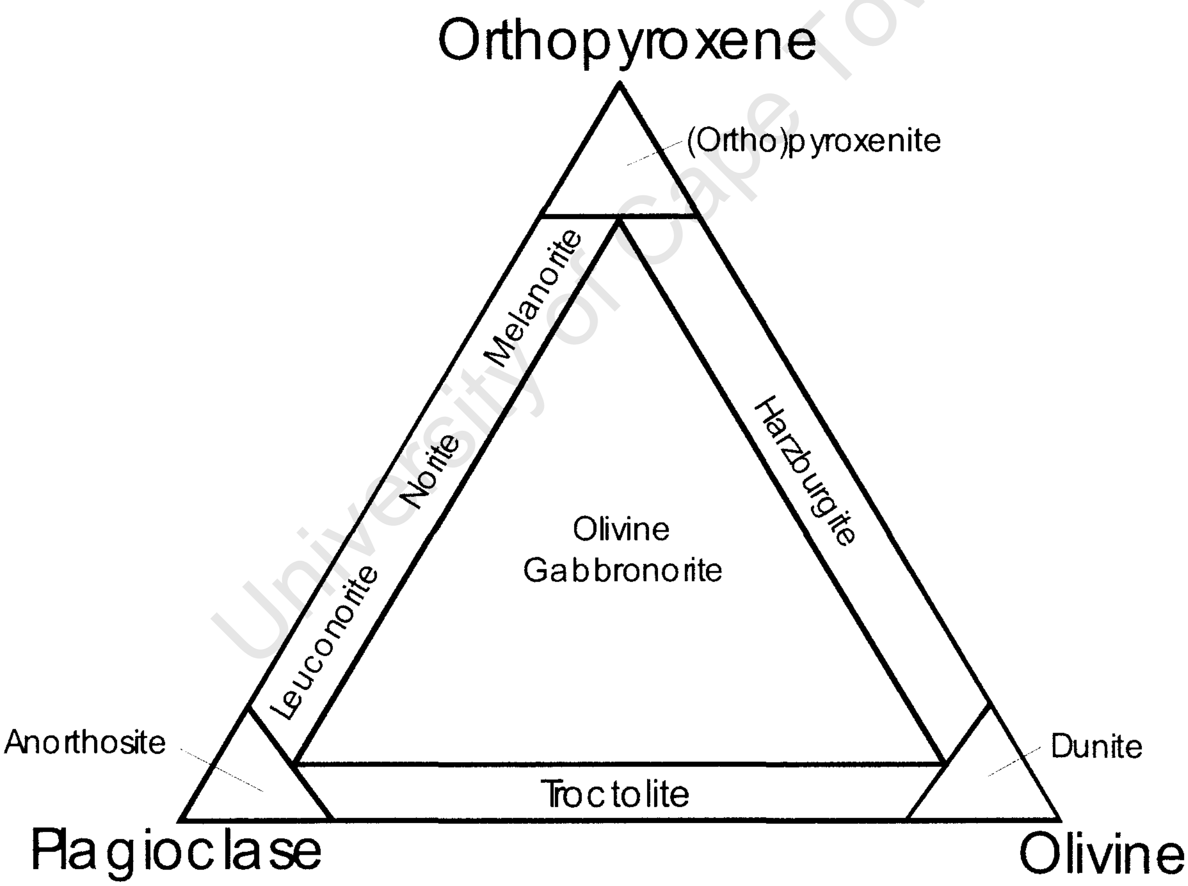
IUGS Lithological Classification

The diagram below shows a schematic representation of the lithological units encountered within the Merensky Reef and their mineralogical constituents. Each of the three points of the triangle represents a 100% concentration of that mineral. From this diagram it is therefore apparent that a melanorite predominantly consists of orthopyroxene with significant amounts of plagioclase.



Glossary - List of Abbreviations and Acronyms

AAS	- Atomic Absorption Spectrometry
BMA	- Bulk Mineralogical Analyses
BMS	- Base Metal Sulphide
BSE	- Back Scattered Electrons
CMC	- Carboxymethylcellulose
CZ	- Critical Zone
EDX	- Energy Dispersive X-Rays
FWP2	- Footwall to the Full Pothole Reef
ICP-OES	- Inductively Coupled Plasma – Optical Emission Spectrometry
LCZ	- Lower Critical Zone
MCU	- Merensky Cyclic Unit
MLA	- Mineral Liberation Analyzer
NR	- Normal Reef
NP2	- Normal-Pothole Transitional Reef
P2	- Full Pothole Reef
PGE	- Platinum Group Elements
PGM	- Platinum Group Minerals
PMA	- Particle Mineralogical Analyses
QEMSCAN	- Quantitative Evaluation of Minerals by Scanning Electron Microscope
RLS	- Rustenburg Layered Suite
SAG	- Semi-Autogenous Mill
SIBX	- Sodium Isobutyl Xanthate
UG2	- Upper Group Chromitite
UCZ	- Upper Critical Zone

1 Introduction

The Bushveld Igneous Complex within South Africa boasts the world's foremost deposits of platinum group minerals as well as economically recoverable amounts of copper, nickel, chromium and vanadium. Platinum group elements are particularly well concentrated within the Merensky and UG2 reefs, which are mined comprehensively along the majority of available outcrops and subcrops. Mineralogically, the Merensky reef is an heterogeneous package consisting of at least one chromitite stringer bounded by melanorites, leuconorites, harzburgites, dunites or anorthosites. The metallurgical process of extracting and marketing platinum group elements from the Merensky reef is extremely diverse within South Africa, nevertheless a general process flow can be summarized as mining, comminution, concentration, smelting and refining.

The Merensky reef at Northam Platinum Ltd. shows a good deal of mineralogical variability. This is due to considerable potholing during formation, which led to six geologically significant and distinctive reefs being defined of which three are selected here for investigation (Normal, NP2 and P2).

The process of transforming blasted run-of-mine ore to a suitable particle size for concentration is called comminution. This process is split into two parts, an initial crushing stage, whereby run-of-mine ore is reduced down to manageable particle size and a final grinding stage to liberate the valuable minerals from the unwanted gangue minerals. The effective liberation of sulphide minerals is the greatest challenge of comminution, as without it, the flotation of these minerals will be greatly reduced.

The typical concentration step in the processing of platinum bearing ores is froth flotation. In this process ore is upgraded from ~5 g/t to >100 g/t in several stages before transferral to the smelter and then finally the refinery. The challenge in froth flotation is to maximise this valuable mineral recovery, whilst minimising the inclusion of unwanted silicate gangue into the concentrate.

2 Literature Review

2.1 Geological Overview.

2.1.1 Bushveld Complex Geology.

The Bushveld Complex in South Africa has an outcrop and subcrop area of approximately 66 000km^2 (Von Gruenewaldt, 1977) with a thickness range of 7-9 km. The complex outcrops in four discrete limbs with a fifth hidden under younger sediments (Figure 2.1).

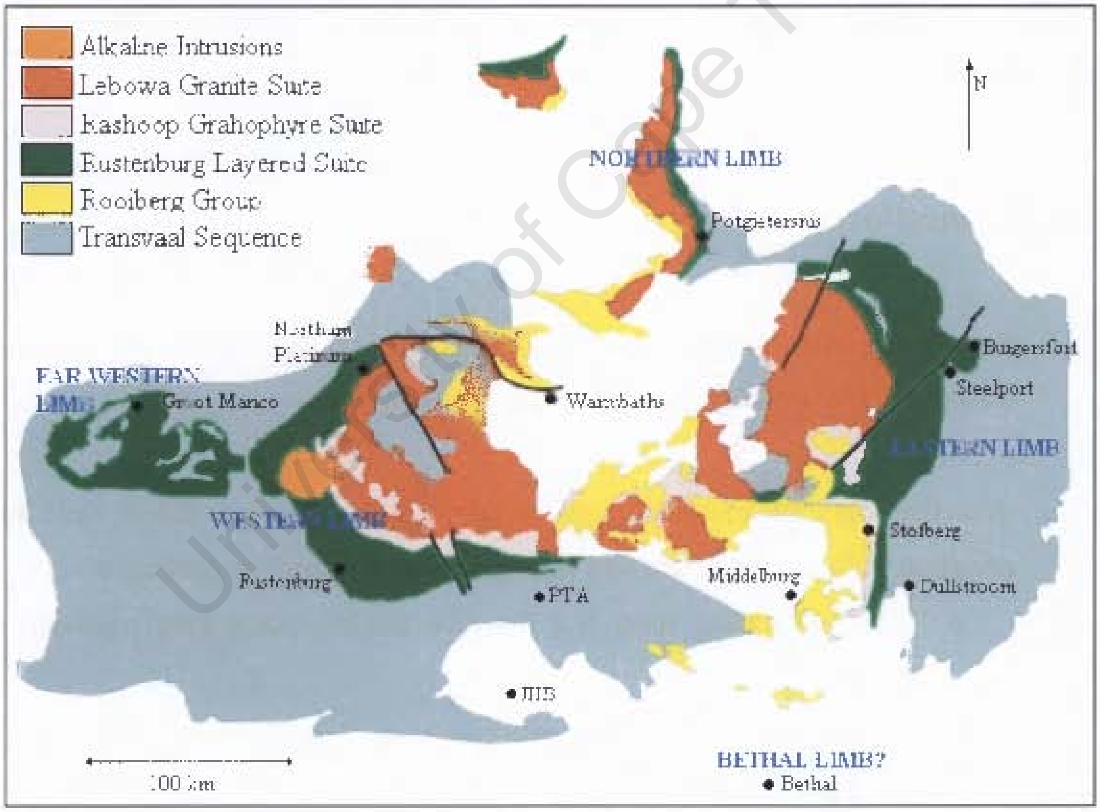


Figure 2.1: *A geological map of the Bushveld Complex, showing the four exposed limbs of the Complex, the location of Northam Platinum and the possible positioning of the buried Bethal Limb. Adapted from www.wits.ac.za.*

The four major outcrop regions are known as the Eastern limb, the Far Western limb, the Northern (or Potgietersrus) limb and the Western limb, the last of which contains Northam Platinum mine (Figure 2.2). The fifth limb, hidden under younger sediments to the southeast, is called the Bethal limb (Eales & Cawthorn, 1996). The

The discipline of process mineralogy grew out of the potential to relate beneficiation performance to the ore mineralogy, and in particular to tie variations in froth flotation performance to variations in the ore. Although many established plant operations implicitly use process mineralogy techniques to optimise production, there is a growing need to formalise these procedures and qualify the links between mineralogy and mineral processing.

At Northam Platinum Limited the mining and processing plant are run separately, and all the material from mining is collected into a single run-of-mine stockpile before being processed as a blend. This project anticipates different flotation behaviour for the three selected Merensky reef types and seeks to characterise and understand the relationship between the mineralogical variability of these reef types and their flotation performance.

1.1 Key Questions

1. How is the mineralogy of the three reef types at Northam Platinum Limited (Normal, NP2 and P2) quantitatively different?

2. Will the distinct mineralogical characteristics of the three reef types produce different mineral processing performances? In particular -
    a. How will milling time and flotation performance vary between reef types?
    b. How is the flotation performance of each reef affected by change in grind and copper addition?

3. If there is a difference in mineral processing performance, can it be explained in terms of some, or all of the following?
    a. Variations in sulphide textures and/or liberation.
    b. Variations in gangue mineral type and proportions.
    c. Variations in gangue alteration or textural development.

lithostratigraphic similarity between the Western and Eastern limbs suggests that they share a common magmatic history (Cawthorn & Webb, 2001).

As well as these five major limbs there are several satellite bodies, coeval with the Bushveld Complex, which further increase the size of the magmatic province. These include the Uitkomst intrusion, the Molopo Farms Complex, the Losberg intrusion and the Moloto intrusion (Eales & Cawthorn, 1996). The age of the Bushveld Complex and surrounding satellite bodies is placed at between 2.05 and 2.06 billion years (Walraven *et al.*, 1990).

Stratigraphically the Bushveld Complex is split into three suites (Rustenburg Layered suite, Rashoop Granophyre suite and the Lebowa Granite suite) (SACS, 1980). The Rustenburg Layered Suite (RLS) is the oldest and contains the economically mineable platinum group ore deposits such as the Merensky reef and the UG2 reef. Using the informal nomenclature of the South African Committee for Stratigraphy (SACS) the RLS is subdivided into five main zones. These subdivisions are based on the appearance or disappearance of cumulus phases, the ratio of cumulus minerals and the grain size of the cumulus minerals (SACS, 1980). This method of subdivision has been contested, notably by Kruger (1990) who proposed that defining the boundaries based on chemical and petrological variation, especially at disconformable and/or unconformable boundaries, was a more geologically robust method of zonation. However, the proposed informal stratigraphy of SACS has remained largely accepted in literature and is considered sufficient for this project. On the Western limb the five zones of the RLS are all developed, though sometimes incompletely (Figure 2.2). The total thickness of the five zones on the Western limb is estimated at 7200m (Eales & Cawthorn, 1996).

These five zones are the Marginal Zone, the Lower Zone, The Critical Zone, The Main Zone and the Upper Zone, (Figure 2.2). The Critical Zone (CZ) splits into two distinct sub zones, the Lower Critical Zone (LCZ) and the Upper Critical Zone (UCZ). The Merensky reef is located in the upper portion of the UCZ along with the last two Middle Group Chromitites (MG3 – MG4), the Upper Group Chromitites (UG1 – UG2) and the Bastard reef (Eales & Cawthorn, 1996; Viring & Cowell, 1999).

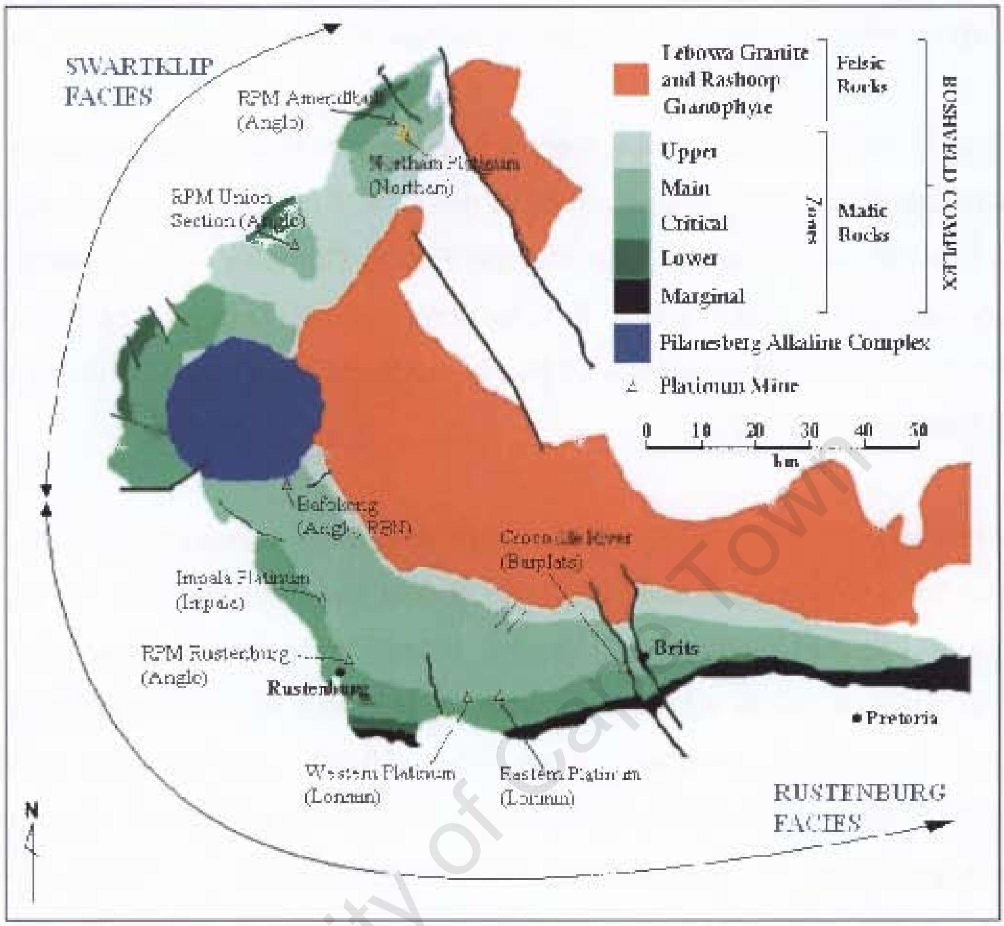


Figure 2.2: *A geological map of the western limb of the Bushveld Complex, showing the platinum mines, the position of Northam Platinum mine and the facies distribution. Adapted from www.wits.ac.za.*

The Merensky reef, which was discovered in 1924 by Dr Hans Merensky, together with the UG2 chromitite, forms the worlds' premium resource of platinum and platinum group elements (Lee, 1996). It was defined by Viring and Cowell (1999) as, 'the economically mineable zone of mineralization straddling the Merensky Unconformity and consisting of the mineralized lower portion of the Merensky Cyclic Unit (MCU) and the mineralized upper portion of the footwall.' On the Western limb the Merensky reef was subdivided into three facies (Wagner, 1929): the Kroondal facies, the Doornspruit facies and the Swartklip facies. Subsequently the Kroondal facies and the Doornspruit facies were grouped together and termed the Rustenburg facies (Figure 2.2). The greater abundance of olivine bearing rocks and reduced Merensky reef - UG2 reef separation distinguishes the Swartklip facies (Eales & Cawthorn, 1996; Wagner, 1929; Maier & Eales, 1997).

The Rustenburg facies lies to the south of the Pilanesberg Complex and strikes east-south-east. The Swartklip facies, which lies to the north of the Pilanesberg Complex strikes north-east (Figure 2.2). Viljoen (1994) further subdivided the Swartklip facies into two subfacies and the Rustenburg facies into five subfacies. The two subfacies of the Swartklip facies are the Normal reef subfacies and the Regional Pothole subfacies.

2.1.2 Northam Platinum Geology

Northam Platinum Limited is located on the Western limb of the Bushveld Complex (Figure 2.2) within the district of Thabazimbi, 15 km northeast of the town of Northam. Exploration began in 1981, and with the sinking of the No.1 shaft in 1986. Official commissioning of the concentrator begin in January 1992 (Snodgrass *et al.* 1994), with full operations beginning in 1993 (Northam Platinum Ltd. Full Annual Report, 2006). The mining authorization covers 6854 hectares and is spread over 8 farms (Viring & Cowell, 1999). Current mining operations exploit the Merensky and UG2 reefs from twin shafts down to a maximum depth of 2200 m using deep level hydropowered mining methods (Northam Platinum Ltd. Full Annual Report, 2006). The mining area also hosts two concentrator plants (one for the Merensky reef and one for the UG2 reef), a smelter and a base metals removal plant (see Section 2.3.3.5 for flowsheet). Annually Northam Platinum recovers approximately 360 000 oz of platinum (Pt), palladium (Pd), rhodium (Rh) and gold (Au) (3PGE + Au) as well as by-products of nickel (Ni), copper (Cu), ruthenium (Ru), osmium (Os) and iridium (Ir) (Northam Platinum Ltd. Full Annual Report, 2006).

Geologically the mine lies in the north-western sector of the Swartklip facies and exploits the Merensky reef from both the Normal reef subfacies and the Regional Pothole subfacies. Within these two subfacies six reef types have been identified of which only four are mined (Normal, NP2, P2 and FWP2), the last of which only on rare occasions. Unfortunately, as for individual lithological layers, nomenclature varies from mine to mine, but this project has followed that of Northam and focuses on the Normal, NP2 and P2 reefs. Their stratigraphical relationships are shown in Figure 2.3.

Potholing at Northam mine is thought to be a themomechanical erosion process (Roberts *et al.*, 2007; Smith & Basson, 2006) and is defined where the MCU transgresses the lower chromitite stringer of the Normal reef (Roberts *et al.*, 2007; see Section 2.1.2.1). Occasionally potholes within the Normal reef are comprised of all pothole reef types (Viring & Cowell, 1999).

Northam Platinum mine is the deepest platinum mine in the world, and hence has a high geothermal gradient, meaning virgin rock temperatures range between 45°C and 72°C. These high virgin rock temperatures instigated the evaluation of hydropowered equipment, which is now used throughout the mine for drilling and cleaning purposes. The magnitude of the depth also leads to high virgin rock stresses (up to 70 MPa) as well as the intersection of several difficult water bearing fissures, one of which has only been overcome recently, the so called 20-line fissure (Modern Mining, 2006). All these factors make mining the Merensky and UG2 reefs a complex prospect at Northam Platinum.

The Merensky reef and the UG2 reefs dip south-east at roughly 22 degrees, this relatively shallow dip facilitates the use of standard breast stoping to recover the ore. Breast stoping begins with a northward trending crosscut from the shaft to an intersection with the reef at that level (Bonel, 1995). Once the reef is intersected, footwall drives are cut to the east and west. From these footwall drives crosscuts are driven south every 200m to intersect with the Merensky reef (Bonel, 1995). Subsequent to intersection, raises are driven up dip to the higher level. Once this level is reached stoping begins to the east and west and the broken ore is scrapped into box holes, which lead to the underlying crosscut. From here the broken ore is transported to the shaft and subsequently up to the surface for processing (Bonel, 1995). Since breast stoping is used to recover the ores, sections with difficult or undulating surfaces (i.e. FWP2) are left in the ground.

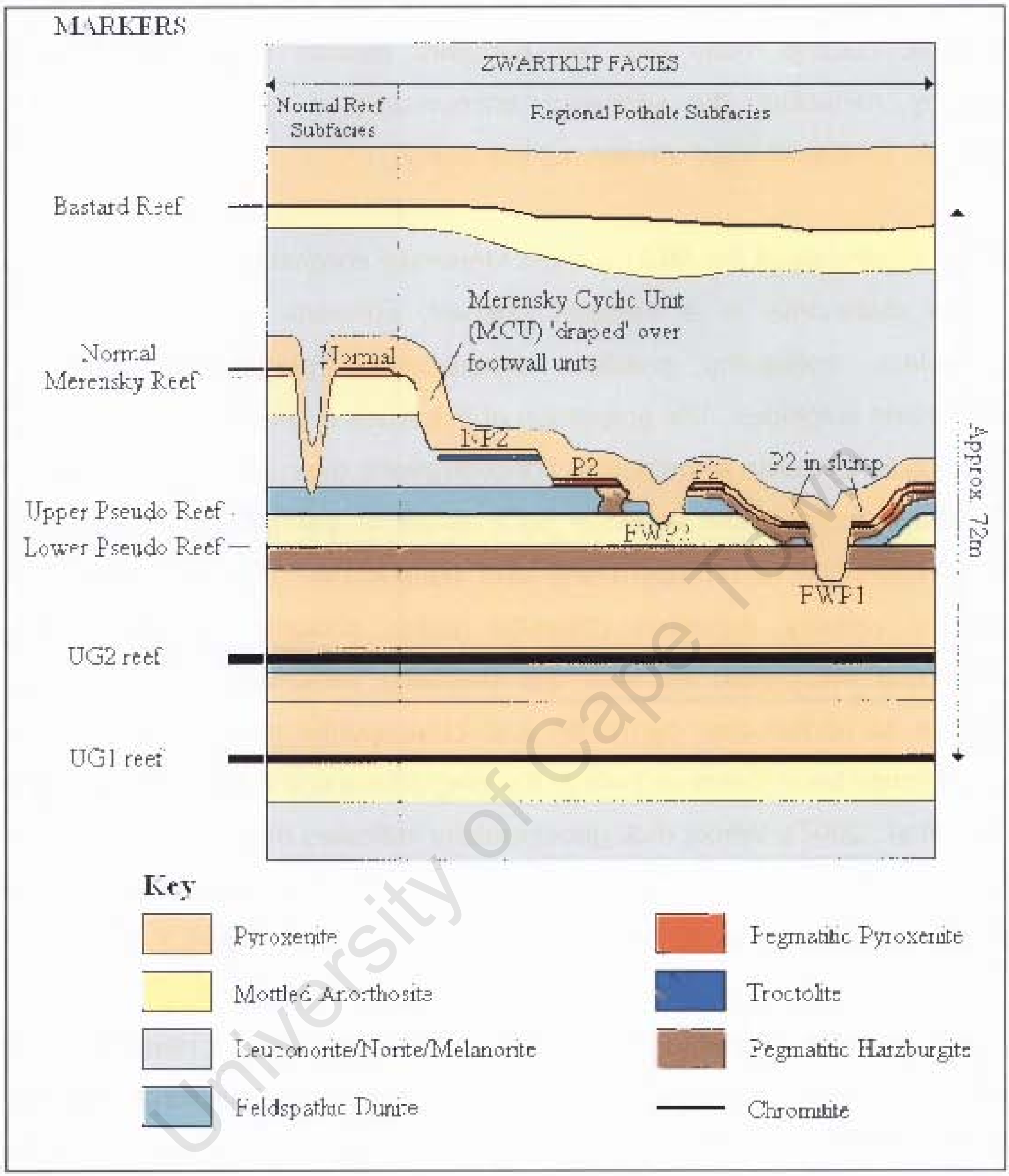


Figure 2.3: *A schematic cross-section through the upper section of the Upper Critical Zone, showing the hanging wall 'drape' and footwall components to the Merensky reefs at Northam Platinum Mine, South Africa. Adapted from Smith et al., (2003).*

2.1.2.1 Normal Reef

The Normal reef as illustrated in Figure 2.4 occurs at the highest stratigraphic position within the UCZ (Viring & Cowell, 1999). It consists of a pegmatitic melanorite bounded by two chromitite stringers overlain by the base of the MCU, and underlain by mottled anorthosite (Roberts *et al.*, 2007; Smith *et al.*, 2003). Northam reef nomenclature is shown on Figure 2.4 by letters and numbers to the right of the stratigraphic column. Note that the numbers ascend downwards through the stratigraphy, mimicking the sequence encountered in a surface drill core (see Appendix A, for the list used throughout this study).

The basal lithologies of the MCU are the Merensky chromitite (4A) and a melanorite (3A). The melanorite is a medium grained, euhedral to subhedral pyroxene orthocumulate, containing poikilitic clinopyroxene, plagioclase, minor biotite, amphibole and sulphides. The proportion of sulphides is greatest at the base (5-8%) and decreases upwards to negligible concentrations over 30-50 cm (Roberts *et al.*, 2007). The sulphides are anhedral composites of pentlandite, chalcopyrite and pyrrhotite interstitial to orthopyroxene and plagioclase. The Merensky chromitite consists of variably annealed chromite grains enclosed by plagioclase and orthopyroxene oikocrysts. As with the overlying melanorite, the sulphides are composites of pentlandite, pyrrhotite and chalcopyrite, which occur here along silicate-chromite boundaries and within the plagioclase and orthopyroxene oikocrysts (Roberts et al., 2007). Whole rock geochemistry indicates that this package is similar for the P2 and NP2 reefs, thus showing that the base of the MCU is an homogenous 'drape' over a variably eroded footwall (Roberts *et al.*, 2007, Figure 2.3).

The inter-chromitite pegmatite (5A-5H) has a variable thickness reaching up to 350 cm and decreasing systematically towards the edge of the Regional Pothole Reef sub-facies (Roberts *et al.*, 2007; Smith *et al.*, 2003). It consists of a medium to coarse-grained feldspathic melanorite or harzburgite, with minor clinopyroxene, chromite, biotite and sulphides (Roberts *et al.*, 2007; Smith *et al.*, 2003). There is some minor layering with a downwards transition from predominantly melanoritic to harzburgitic or dunitic. Plagioclase is variably present (10-35%) within the inter-chromitite pegmatite, possibly reflecting local melt segregation within a compacting cumulate pile (Roberts *et al.*, 2007). The proportion of sulphides is greatest nearest the Merensky chromitite (5-10%) and decreases downwards to <1% within 60 cm. When approaching the basal chromitite the sulphide proportion climbs again to a second peak at or near the basal chromitite (Roberts *et al.*, 2007; Smith *et al.*, 2003). These dual peaks merge together during reef thinning, forming one single sulphide peak. Mineralogically the sulphides are the same as for the basal MCU, developed as either composites interstitial to silicate minerals or as minor sulphides trapped in late stage silicate crystallization (Roberts *et al.*, 2007)

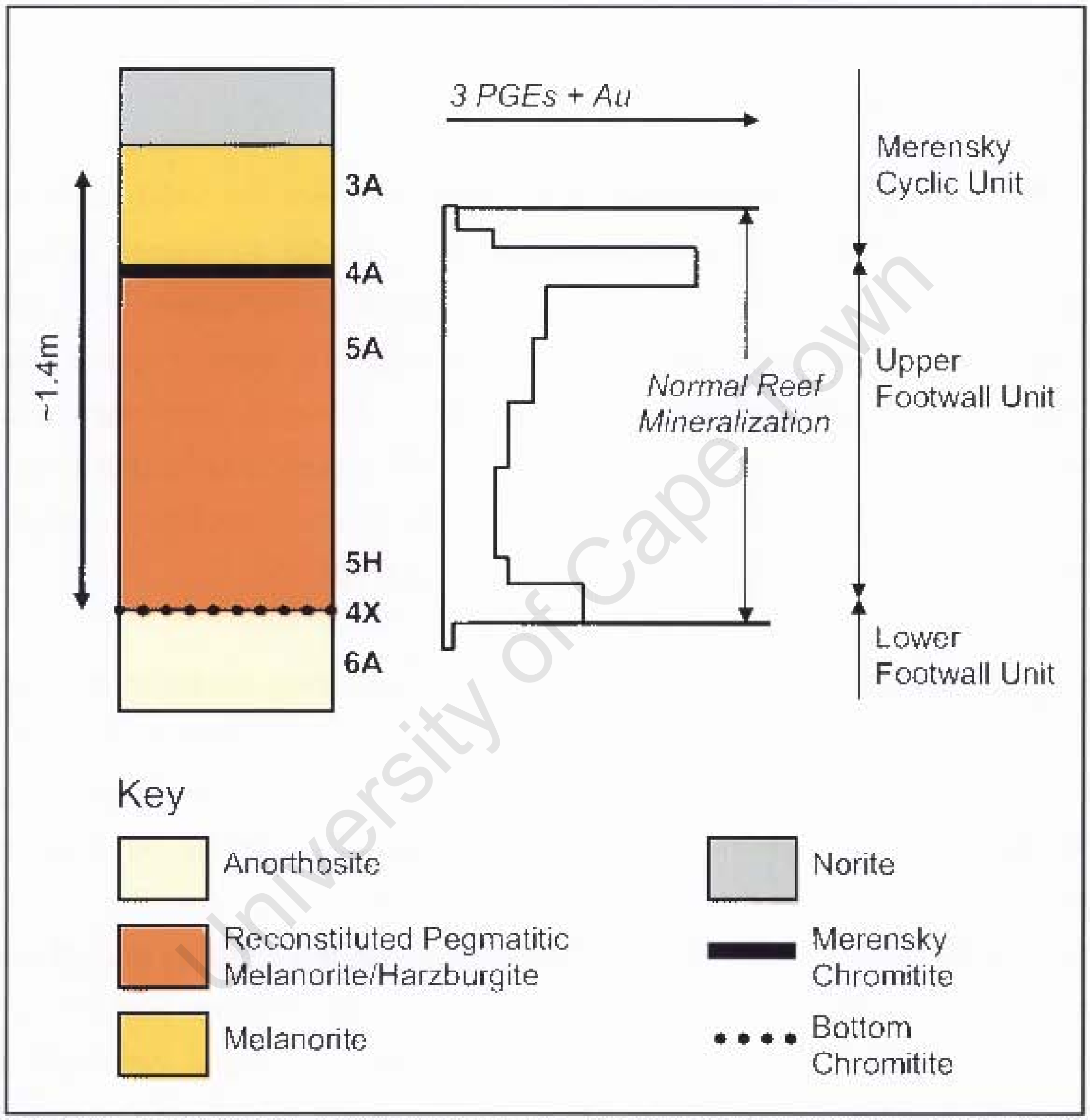


Figure 2.4: *A schematic profile through the Normal reef at Northam Platinum Mine. Also shown is the grade profile, which is indicative of the 3PGE + Au value distribution (Viring & Cowell, 1999). The short hand nomenclature used at Northam mine is shown by the letters and numbers to the right hand side of the profile.*

The basal chromitite (4X) is an undulating granular chromite orthocumulate, rarely thicker than 1 cm, and underlain by a mottled anorthosite (6A) (Smith *et al.*, 2003).

Alteration within the Normal reef, and in particular the basal MCU, reflect particular formation processes. Within the basal MCU there are several microstructures that show the plastic deformation of a cumulate pile including undulose orthopyroxene extinction, bent clinopyroxene exsolution and deformation twins within plagioclase (Roberts *et al.*, 2007). In contrast biotite grains are strain-free suggesting some post compaction crystallization. Similar strain features to the basal MCU are present in plagioclase poor inter-chromitite settings and thus similar processes are inferred.

2.1.2.2 NP2 reef

The NP2 reef as illustrated in Figure 2.5 is defined where the MCU conformably overlies the lower footwall unit, which is located roughly 12 metres below the stratigraphic height of the Normal reef (Viring & Cowell, 1999; Roberts *et al.*, 2007). It consists of a variably troctolized footwall leuconorite unit overlain by the base of the MCU. In the Normal reef the lower footwall unit is defined by small scale cyclicity between norites, leuconorites and anorthosites. The absence of troctolites at this level below the Normal reef suggests that the NP2 troctolites are a secondary metasomatized or reconstituted lithology (Roberts *et al.*, 2007).

The basal MCU lithologies are mineralogically and texturally similar to the Normal reef MCU basal lithologies (see Section 2.1.2.1). The footwall (8A/T to 9A/T) to the MCU usually consists of 5-10 cm of anorthosite underlain by 20-60 cm of troctolite, which grades into leuconorites or norites. Occasionally there is no intervening anorthosite layer between the Merensky chromitite and the troctolite (Roberts *et al.*, 2007). Routine stope mapping of the NP2 troctolites suggest that they are not due to small changes in footwall composition or localized metasomatism (Roberts *et al.*, 2007). The troctolite layer consists primarily of plagioclase, olivine, orthopyroxene, sulphide composites and minor chromite with the plagioclase occurring as cumulus, inequigranular subhedral laths (Roberts *et al.*, 2007). Olivine occurs as poikilitic, coarse (up to 12 mm) anhedral grains, enclosing pre-existing plagioclase cumulates. Orthopyroxene often occurs as anhedral grains rimming olivine and interstitial to plagioclase, with some minor anhedral chromite included (Roberts *et al.*, 2007).

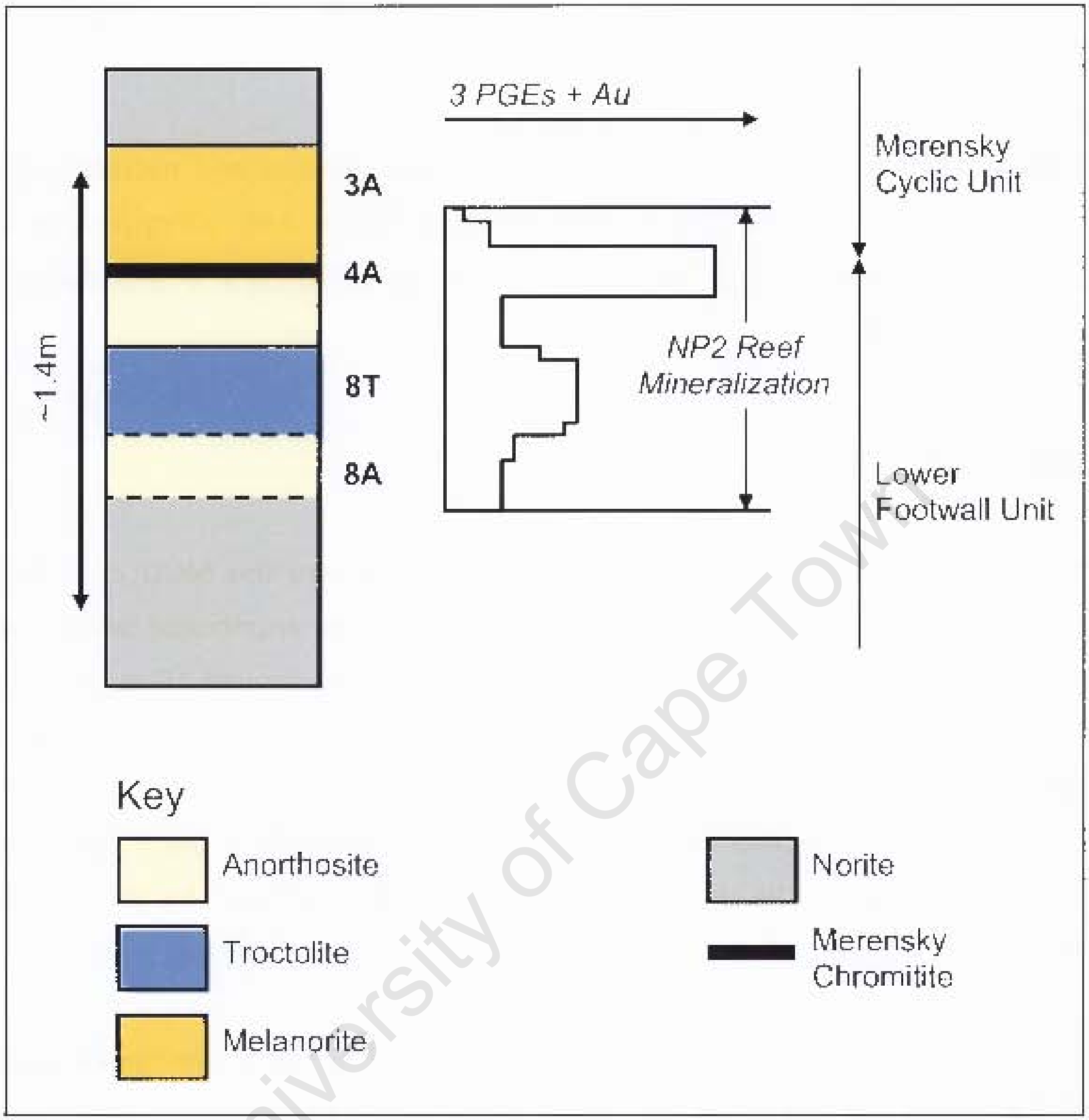


Figure 2.5: *A schematic profile through the NP2 reef at Northam Platinum Mine, South Africa showing in particular the troctolized zone just below the Merensky Chromitite. The 3PGE + Au profile is indicative of the value distribution for this reef type (Viring & Cowell, 1999). The short hand nomenclature used at Northam mine is shown by the letters and numbers to the right hand side of the profile.*

Sulphides are developed as anhedral composites of pentlandite, chalcopyrite and pyrrhotite, occurring as disseminations interstitial to plagioclase and orthopyroxene or as inclusions within orthopyroxene grains (Roberts *et al.*, 2007). Many of the sulphides have cuspate grain boundaries indicating that they were being forced into fractures that had developed during cooling. Disseminated sulphide concentrations (5- 10 vol%) are greatest near the base of the Merensky chromitite, and decrease down to 1-3 vol% by the base of the troctolite. The disappearance of olivine and sulphide occurs almost simultaneously and the lack of sulphides below the troctolite layer implies that sulphide mineralization and troctolization were coupled processes (Roberts *et al.*, 2007).

Alteration within the NP2 reef displays similar characteristics and microtextures to the Normal reef with undulose extinction within olivine and orthopyroxene and deformation twins within plagioclase indicating plastic deformation of a cumulate pile (Roberts *et al.*, 2007).

2.1.2.3 P2 reef

The P2 reef as illustrated in Figure 2.6 is defined where the MCU conformably overlies the P2 cyclic unit, and in particular a thin (1-2 cm) anorthosite, which marks the top of the P2 cyclic unit (Roberts *et al.*, 2007). This is around 16 metres below the stratigraphic level of the Normal reef. It consists of a heterogeneous pegmatitic melanorite and harzburgite bounded by two chromitite stringers, the uppermost of which is the Merensky chromitite and the lowermost of which is the P2 chromitite. This packet is overlain by the base of the MCU and underlain by the Tarentaal dunite.

The P2 reef is the lowest pothole reef mined to any volume at Northam Platinum, since the FWP2 reef is avoided due to the technical difficulty of using breast stoping to mine a rapidly undulating surface.

Basal MCU lithologies are mineralogically and texturally similar to those of the Normal reef MCU (see Section 2.1.2.1). The inter-chromitite pegmatite (5A) consists of an upper thin (4 cm) pegmatitic melanorite unit and a lower thin (10 cm) pegmatitic harzburgite (Roberts *et al.*, 2007). The inter-chromitite separation is variable and increases towards the transition to the NP2 reef. The magnitude of the chromitite separation controls the extent of pegmatitic textures, with large separations contributing to reduced grain sizes and small separations often inducing reconstitution of the Tarentaal unit to a more pegmatitic nature (Roberts *et al.*, 2007).

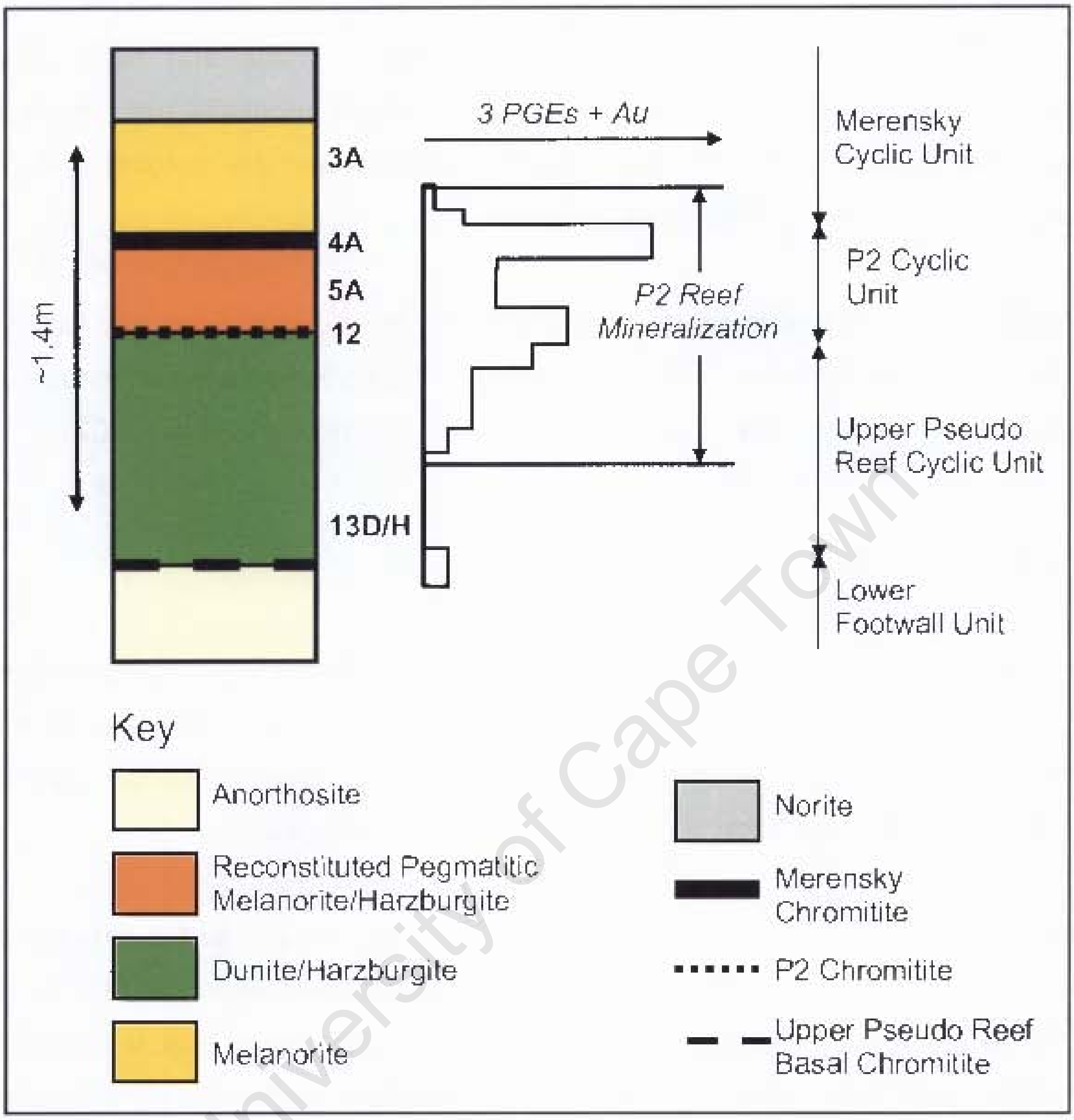


Figure 2.6: *A Schematic profile through the P2 reef at Northam Platinum Mine. The 3PGE + Au profile is indicative of the value distribution for this reef type (Viring & Cowell, 1999). The short hand nomenclature used at Northam mine is shown by the letters and numbers to the right hand side of the column.*

Orthopyroxene within the pegmatitic melanorite is developed as medium (5-10 mm) grained subhedral to euhedral crystals, or as coarse and very coarse anhedral masses. It often includes chromite and rounded anhedral olivines (Roberts *et al.*, 2007). The pegmatitic harzburgites contain rounded anhedral olivines, usually enclosed by orthopyroxene, though in the dunites olivine is more subhedral and free from orthopyroxene rims (Roberts *et al.*, 2007). As with the Normal reef, plagioclase is locally variable within the harzburgites possibly signifying local melt segregation. The inter-chromitite pegmatites of the Normal reef and the P2 reef are very similar reflecting the likelihood of similar pre-existing footwall units being reconstituted by the same hanging wall MCU 'drape'. The sulphide profile and mineralogy is essentially the same as the Normal reef inter-chromitite pegmatite with the highest concentrations associated with the chromitite stringers, and the merging of the two peaks with reef thinning (Roberts *et al.*, 2007).

Alteration in the P2 reef is similar to the NP2 and the Normal, especially within the draped MCU unit. Alteration within the inter-chromitite pegmatite is concentrated on olivine grains which are often pervasively serpentinized (Roberts *et al.*, 2007).

2.1.2.4 Terminology

The terms 'hanging wall' and 'footwall' are common to geologists, mining engineers and metallurgists. However they have very different meanings, depending on which profession's viewpoint is taken. This can be a cause for great confusion, especially as communication lines open up between these three occupations.

With respect to the Normal Merensky reef, the geologist would use the term hanging wall to denote the basal lithologies of the MCU, namely the melanorite (3A) and the Merensky chromitite (4A). Every lithology, or rock type, below this is termed the footwall (Figure 2.7a). This is because the hanging wall represents a new pulse of magma into the magma chamber, which, as it cools, settles and begins to crystallize on the previously crystallized footwall. The hanging wall, footwall divide therefore demarcates a chronological break – the footwall is older, and there is a time gap between footwall and hanging wall deposition.

The deposition of the hanging wall gives rise to reconstitution of the footwall and the pegmatitic textures associated therewith. The PGE mineralization associated with this deposition is a separate feature, plotted alongside a stratigraphical log (Figure 2.7a). For a geologist the main PGE mineralization region is associated with the Merensky chromitite in the hanging wall, and the bottom chromitite in the footwall.

For the Mining Engineer the terms hanging wall and footwall refer to rock left behind after the stope cut has been made (Figure 2.7b), equivalent terms would be 'ceiling' and 'floor'. Thus the mining engineer speaks of supporting the hanging wall and 'cleaning', or 'brushing' the footwall to recover the remnant PGM left behind in placar pockets.

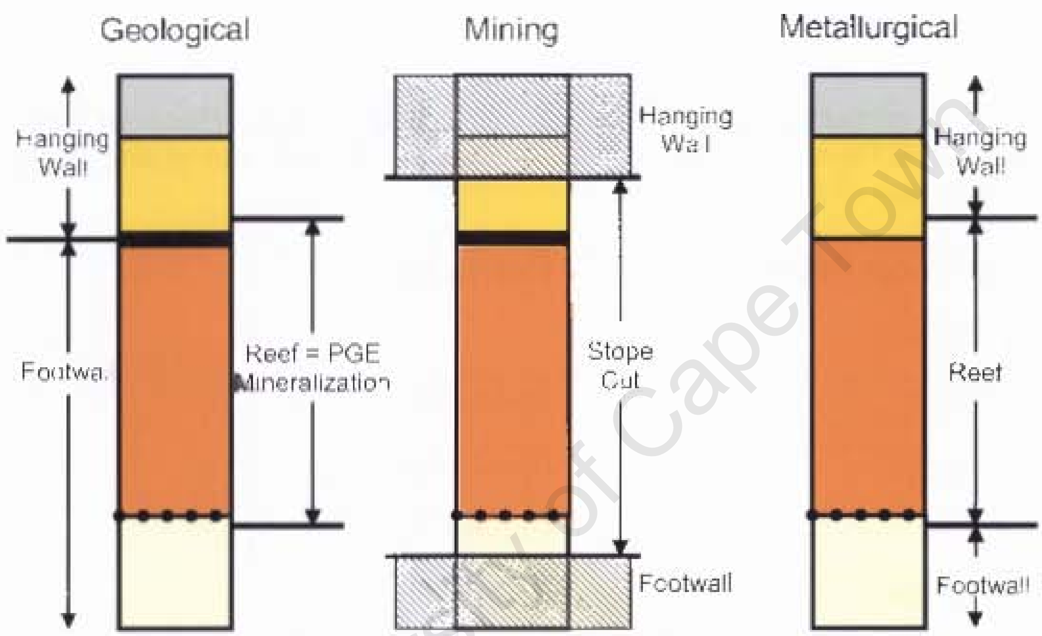


Figure 2.7: *Three diagrams representing different understandings of the terms 'hanging wall' and 'footwall'. a. geological, b. metallurgical and c. mining.*

For the metallurgist, the hanging wall and footwall are terms describing the barren sections of rock above and below the valuable reef (Figure 2.7c). The reef is defined as the area of PGE mineralization associated with the Merensky lithologies. Therefore, the terms hanging wall, reef and footwall are grade concepts with no chronological connotation. For the metallurgist it is important to minimise the recovery of hanging wall and footwall rocks.

For this study the nomenclature used throughout is geological (Figure 2.7a). This is particularly important for Chapter 4, which is the mineralogical characterization.

2.1.3 Platinum Group Minerals

Platinum group elements (PGE) and Platinum group minerals (PGM) in the Merensky reef are closely associated with the base metal sulphides (BMS). This association is seen either as the discrete PGM, or as solid solution (Ballhaus & Sylvester, 2000; Cawthorn *et al.*, 2002; Schouwstra *et al.*, 2000). Table 2.1 shows the major PGM found at Northam Platinum Ltd.

Table 2.1: *A table showing the common PGM found at Northam Platinum Limited based on Kinloch & Peyerl, (1982); Schouwstra et al., (2000); Viring and Cowell, (1999) and Snodgrass et al. (1994).*

Common PGM at Northam	Formula
Braggite	Pt>PdNiS
Cooperite	Pt>>PdNiS
Laurite	$Ru(OsIr)S_2$
Vysotskite	Pd>PtNiS
Ferroplatinum	Pt_3Fe
Sperrylite	$PtAs_2$
Moncheite	$PtTe_2Bi$
Bismuthotellurides	Pt(Pd)BiTe

Research into PGM found within the Merensky reef has been extensive (Cawthorn *et al.*, 2002; Schouwstra *et al.*, 2000; Lee, 1996; Kinloch & Peyerl, 1990). The cumulative research highlights a wide range of minerals as well as high variability from one section of the Bushveld to another. Kinloch and Peyerl (1990) observed that the Amandelbult section, (which Northam Platinum Ltd. is located within) had similar PGM assemblages to the Rustenburg section and was dominated by braggite, cooperite and occasionally laurite. Their research into potholes in the Union section and southern Amandelbult section showed a dominance of Pt-Fe alloys, although Normal elevation reefs within the same vicinity were also dominated by Pt-

Fe alloys. Schouwstra *et al.* (2000) summarized the findings for the Amandelbult section as being ~50% Pt-Fe alloys, 21% platinum-palladium sulphides, 13% platinum-palladium tellurides and 10% laurite.

Snodgrass *et al.* (1994) reported on initial findings from the exploration boreholes at Northam, which contained the following PGM constituents: ferroplatinum, cooperite, braggite, vysotskite, sperrylite, laurite, bismuthotellurides and Pd alloys. Viring and Cowell (1999) confirmed this PGM assemblage, as well as identifying a particular bismuthotelluride (moncheite). However, neither of the above two studies report quantitative PGM abundances. The most recent study into PGM occurrences at Northam was undertaken by Roberts *et al.* (unpublished) who investigated PGM variations from reef type to reef type (Figure 2.8).

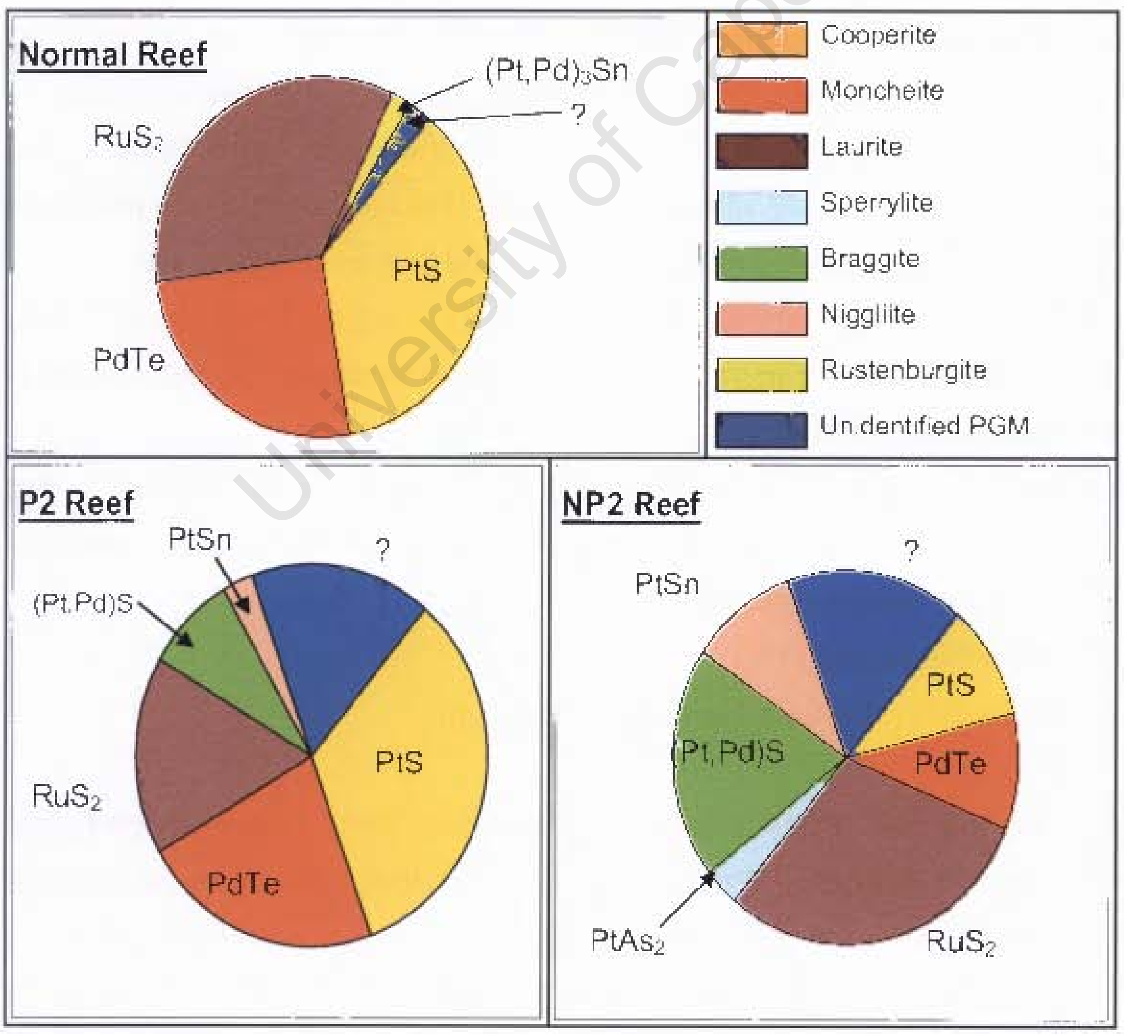


Figure 2.8: *A Series of pie charts showing the relative abundances of different PGM within the Normal, NP2 and P2 reefs (Roberts et al., unpublished).*

There is a close correlation between the Normal reef and the P2 reef and an apparent difference in modal abundances of the main minerals in the NP2 reef. However, the main PGM are the same, furthermore the lack of a distinctive PGM signature (along with the data from whole rock geochemistry and reef petrography) supports the theory that potholing at Northam is a largely thermomechanical process, and that deposition of the MCU occurred as an homogenous 'drape' over a previously eroded footwall (Roberts *et al.*, 2007). However, the sample size was statistically small (only 30 PGM in the case of the NP2 reef) and a significant number of PGM in both the NP2 and P2 reefs were unidentified.

2.1.4 PGM Mode of Occurence at Northam

This work is summarised from unpublished research by Roberts *et al.* (2004). PGM across the three reef types consistently show up to six different modes of occurrence. The two most common of these are: inclusions within BMS (Figure 2.9a,b); or hosted along BMS/silicate boundaries (Figure 2.9c,d). Together these make up 79% of all documented PGM. The remaining four associations are:

- Hosted along BMS/chromite boundaries (8%) – mostly within the Merensky chromitite, and similar in style to BMS/silicate boundaries, i.e. predominantly enclosed within the BMS.
- Locked within chromite (4%) (Figure 2.9e). In each case, the PGM is first partially or completely enclosed within a sulphide, which itself is enclosed by a secondary annealed chromite grain.
- Locked within silicate (1%)
- Located within microfractures (8%) (Figure 2.9f).

Mineralogically the main PGM observed are cooperite (PtS), Moncheite ($PtTe_2Bi$), Laurite (RuS_2) and Braggite (PtPdS). Aside from moncheite, these are all sulphide phases, which can be expected to have a close association with BMS. Of the 23 moncheite grains located, 19 were either in BMS inclusions or, BMS/silicate boundaries, and the remaining 4 were in microfractures associated with BMS.

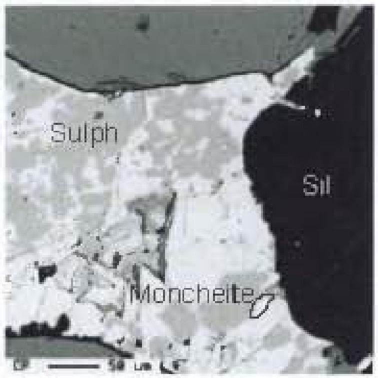


Figure 4.8a: Moncheite grain (outlined), P2 reef.

Figure 4.8b: Moncheite grain (outlined), Normal Reef.

Figure 4.8c: Laurite (outlined) occurring on the margins of three sulphide composites, Np2 reef.

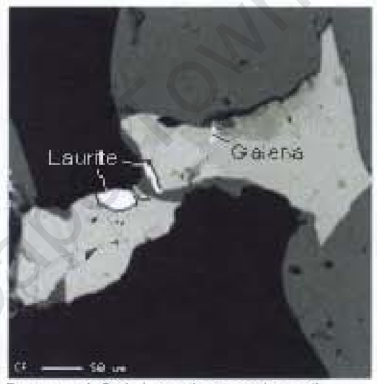


Figure 4.8d: Laurite (outlined) occurring on the sulphide margins, Normal Reef

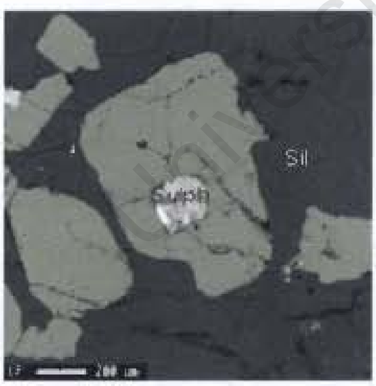


Figure 4.8e: Cooperite with BMS, in annealed chromite, P2 reef.

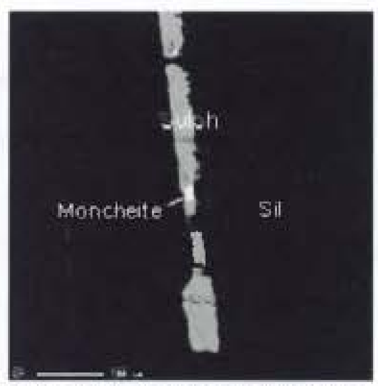


Figure 4.8f: Moncheite with BMS, within a microfracture, Np2 reef.

Figure 2.9: *PGM Mode of Occurrence. All figures are back-scattered electron images. Sil = silicate, Sulph = sulphide. (Roberts et al., unpublished)*

Although this represents six separate PGM textures, the primary association of PGM with BMS is apparent over 95% of the time. From these observations of PGM it is evident that there is an intimate spatial relationship between PGM and BMS within the original ore at Northam.

2.2 Metallurgical Overview

Platinum mining within South Africa is very varied, with small open cast, or shallow depth inclined shafts all the way down to deep, large operations using vertical shafts. In the context of process mineralogy one of the most important features is the mine-call-factor (Merkle & Mckenzie, 2002), which refers to the actual PGE grade delivered to the mill compared with the grade within the unmined reef. This factor depends mostly on mining practice, although variations in ore mineralogy may also contribute to a reduced mine-call-factor.

Mineral beneficiation has, traditionally, used comminution and flotation to liberate and then concentrate the valuable platinum group minerals (PGM). These early beneficiation stages are when the greatest PGM losses occur, which is essentially due to the wide range of PGM textures within the ore (Merkle & Mckenzie, 2002). Within comminution there needs to be a fine balance between sufficient grinding time and PGM liberation. Over-grinding is unfavorable because, not only does it waste energy, but it produces excessive fine material, which is harder to recover through flotation (Merkle & Mckenzie, 2002), whereas under-grinding reduces liberation.

Once the concentrate has been separated out the mill tailings are sent for disposal. Due to the low grades associated with platinum mining (~5-7 g/t), these mill tailings are very large, and very fine. This represents a significant environmental problem, with unsightly visual effects, and water pollution arising from contamination by heavy metals, mill reagents and sulphide compounds (Wills, 1997).

In extracting the valuable platinum group elements (PGE) from Merensky and UG2 ores most South African mineral processing operations use procedures similar to those illustrated in Figure 2.10. Design and efficient management of such complex recovery circuits have become more dependent on ore mineralogy and textures, which are now enhanced by developments in quantitative mineral measurement systems (see Section 2.4).

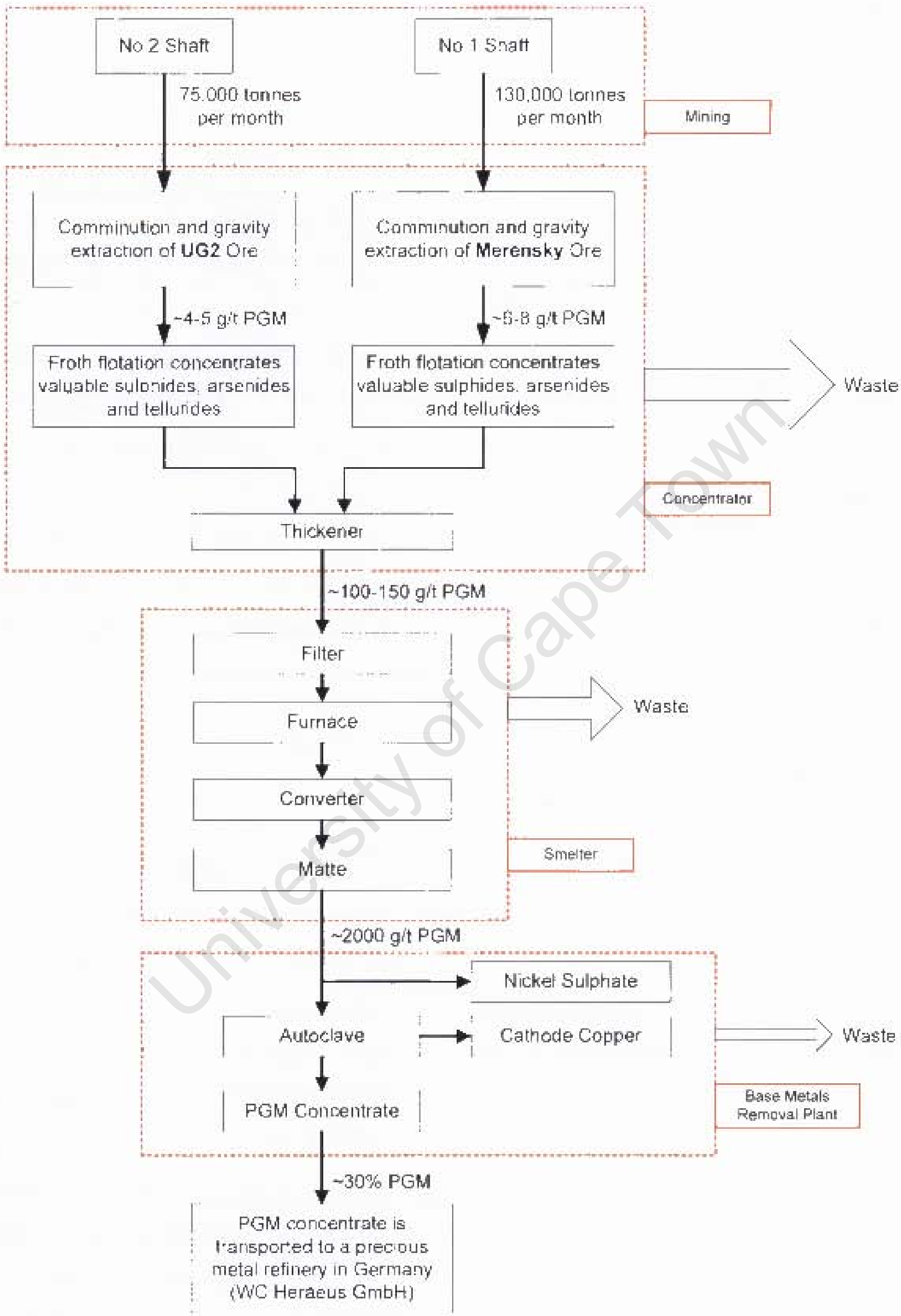


Figure 2.10: *A flowsheet showing the principle operations in extracting precious metals from the Merensky and UG2 platinum ore, with figures specific to Northam Platinum Limited. (Adapted from Merkle & Mckenzie, (2002), Vermaak, (1995) and the Northam platinum website).*

2.3 Mineral Processing Overview

2.3.1 Comminution

The first phase in mineral processing is comminution, which can be considered in two stages, crushing and grinding. After blasting, freshly excavated ore is transported via scrapers, conveyers and ore carriers to the mineral processing plant. Crushing makes transportation of this ore easier, and produces material of a controlled particle size (Wills, 1997). Crushing is a dry process, accomplished by the compression of the ore against rigid surfaces (Wills, 1997). Once the particle size of the run-of-mine ore has been sufficiently reduced, grinding is carried out to liberate the valuable minerals from the unwanted gangue minerals. Grinding is a wet process, accomplished by abrasion of the ore by free moving stainless steel media such as rods, balls or pebbles (Wills, 1997). Generally, due to the strong association of PGM with base metal sulphides (BMS), the target valuable minerals for Merensky reef froth flotation are the BMS, and a bulk sulphide concentrate is sought.

2.3.2 Liberation

Liberation is defined as the degree to which a valuable mineral is exposed from the gangue on the basis of area (Barbary, 1991). After grinding the valuable minerals can be considered as occurring in four main states of liberation, which depends upon the percentage of surface area exposed. The breakdown used in this investigation is: locked (<30%), low grade middlings (30-60%), high grade middlings (60-90%) and liberated (>90%). The degree of liberation of the valuable minerals themselves can be considered in two parts. The first is as liberated composites, separated from silicate gangue, the second is as liberated individual sulphide minerals separated from other sulphides as well as the silicate gangue. If considered as liberated composites the degree of liberation will be higher. The geological overview of the various reef types at Northam has highlighted the presence of two main types of sulphide development: medium grained, anhedral composites which would be expected to liberate and float readily; but also fine grained, monomineralic inclusions, which would be expected to remain locked during initial milling.

2.3.3 Froth Flotation

The flotation process contains two distinct phases, the pulp phase in which mineral recovery occurs, and the froth phase from which concentrated valuable minerals are separated from the bulk. Recovery of minerals to the froth phase may occur through attachment to through flowing bubbles (true flotation) or by entrainment in water passing from the pulp phase to the froth phase. Collection by the former process is selective (on the basis of surface properties), whilst the latter is non-selective and results in unwanted gangue minerals reporting to the concentrate and thereby lowering the grade. Flotation performance is a complex physico-chemical separation process affected by roughly 25 parameters, which are more fully described by 100 variables (Crozier, 1992). Klimpel (1984) divided the major variables into three main groups (Figure 2.11), and careful planning is required to analyse any particular parameter. Variables highlighted in red on Figure 2.11 are chosen for investigation.

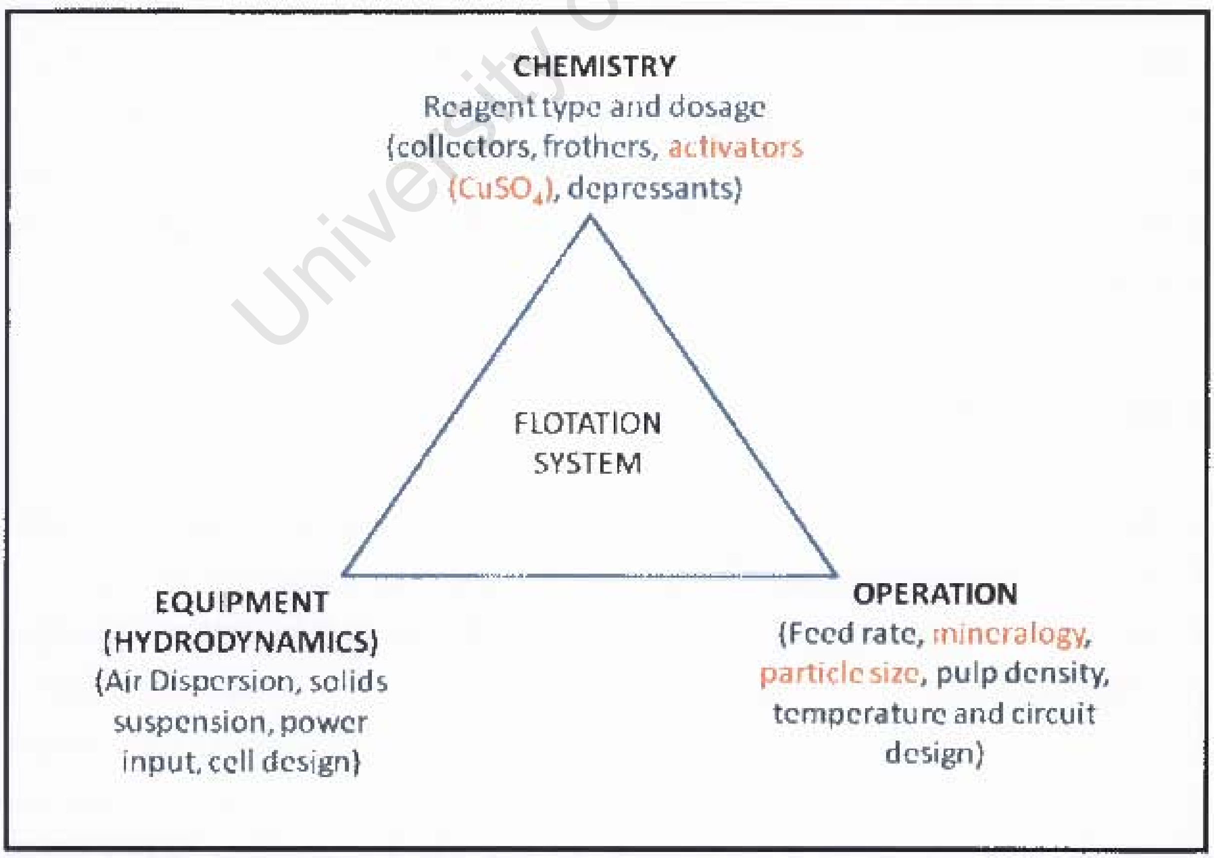


Figure 2.11: *Summary of the flotation system adapted from Klimpel, 1984. Variables shown in red are the focus of this investigation.*

For every mineral particle there is a flotation domain relating the percentage recovery to the particle size in the pulp phase. There is an optimum particle size for flotation, with upper and lower limits. For fine particle sizes (<20 μm) the collision efficiency is too low for effective flotation. For large particle sizes detachment from the bubbles becomes more common and effective flotation is also reduced.

2.3.4 Flotation Reagents

Flotation reagents are added to the pulp to manipulate mineral surface chemistry and thereby enhance differences in mineral hydrophobicity, facilitating separation of valuable minerals from gangue minerals (Wiese *et al.*, 2006). Flotation reagents are added to the slurry and dispersed by an impeller. A typical reagent suite consists of collectors, frothers, depressants and occasionally activators. Collectors work to render the surface of valuable minerals hydrophobic, depressants suppress the flotation of naturally hydrophobic gangue material and frothers help in bubble formation as well as stabilising the froth. Activators are sometimes needed to further encourage the flotation of the desired mineral, an example being the addition of copper sulphate to some Merensky ores to assist in the flotation of pyrrhotite (Wiese *et al.*, 2005b). Once the reagents have been added there is usually a suitable conditioning period, during which the reagents and the slurry are thoroughly mixed using the impeller. After this conditioning period, air is added and dispersed by the impeller.

2.3.4.1 Collector

Collectors are used in froth flotation to render the surfaces of target minerals hydrophobic, thus encouraging their true flotation and their preferential concentration during the flotation process. Northam Platinum Ltd. use Sodium Isobutyl Xanthate (SIBX) along with proprietary reagents Sascol 61 and Sascol 105 as their collectors. Sascol 61 and Sascol 105 are caustic alkali liquids. SIBX, along with sodium isopropyl xanthate (SIPX) are the most commonly used collectors in the mineral processing industry. SIBX is an example of a sulphydryl anionic collector (King, 1982; Wills, 1997).

The non polar outer coating of hydrocarbon chains in the collector renders the overall mineral hydrophobic and therefore floatable. Collectors are generally used in small amounts, as it is believed increased concentration contributes to the true flotation of some gangue minerals, thus reducing selectivity (Wills, 1997).

2.3.4.2 Activator

An activator is sometimes used to supplement the role of the collector in rendering the surfaces of the target minerals hydrophobic. Activators work in tandem with collectors by altering the surface of the target mineral such that the collector more readily binds to it. The activator used in this work, and the most common activator used in the flotation of Merensky ores, is copper (II) sulphate pentahydrate ($CuSO_4.5H_2O$). Copper sulphate is used widely in the mineral processing industry as an activator for pyrrhotite.

Of particular interest when considering copper sulphate addition is the inadvertent activation of gangue minerals rendering them floatable, which may occur when copper hydroxide species adsorb onto the surface of silicates and oxides, especially in the pH range 7 – 10 (Fornasiero & Ralston, 2005; Martinovic *et al.*, 2005).

2.3.4.3 Frother

One of the key stages in froth flotation is the formation of a stable froth, which can retain the valuable minerals for further upgrading (King, 1982). To this end a frothing agent is often added, which in addition to improving froth formation will also increase air dispersion within the pulp, reduce coalescence and reduce the speed of bubble ascent. All of these factors help increase the residence time of bubbles within the pulp, and thereby increase the possibility of particle-bubble contact (King, 1982). An ideal frother will react entirely in the liquid phase and not influence the state of the mineral surface; however in practice this is not observed (Wills, 1997). A good frother needs to have negligible collecting power.

Frothers are heteropolar organic reagents that absorb on the air-water interface, with the non-polar hydrocarbon group within the bubble. This ability reduces the surface tension and thus stabilises the bubble (Wills, 1997). The frother used in the batch flotation experiments was the proprietary reagent Sasfroth 200.

2.3.4.4 Depressant

A depressant is sometimes used to coat the unwanted hydrophobic gangue minerals, rendering them hydrophilic. Industrial organic depressants fall into two main categories; Carboxymethylcellulose (CMC), and Guar Gum. This study has worked solely with CMCs, which are anionic polysaccharides (Burdukova, 2007) with very high molecular weight.

The mechanism for CMC adsorption is now considered to be through acid/ base interactions between the depressant and the mineral surface, though for sometime the preferred mechanism was hydrophobic, hydrogen bonding (Burdukova, 2007; Laskowski *et al.*, 2007).

2.3.5 Flotation of Merensky Ores

There has been extensive work into the flotation of Merensky ores (e.g. Wiese *et al.*, 2005b, 2006; Bradshaw *et al.*, 2006; Vos, 2006), and more specifically to Northam processing (Snodgrass *et al.*, 1994).

The bulk of PGE and thus PGM are contained within, or closely associated with the base metal sulphides (BMS) within the Merensky reef (Section 2.1.3). This close association means that recovery of the BMS is crucial and the aim of the flotation processes is to maximize BMS recovery. Base metal sulphides make up roughly 1% of the Merensky run of mine ore, consisting mainly of chalcopyrite, pentlandite and pyrrhotite with occasional pyrite (Ballhaus & Sylvester, 2000).

Research has shown that chemical environment is crucial to the success of the flotation process (Bradshaw *et al.*, 2006) and in particular to the flotation of the BMS. Flotation rates of different sulphides vary considerably with chalcopyrite being the quickest, pentlandite being intermediate and pyrrhotite being the slowest (Wiese *et al.*, 2006; Buswell & Nicol, 2002), with their responses to specific reagents also being diverse. The effects of particular reagents on the flotation process are not simplistic and broad generalizations are sometimes deceptive.

2.3.5.1 Processing at Northam

The final milling and flotation flowsheet is shown in Figure 2.12. The milling stage consists of a semi-autogenous (SAG) mill, followed by a ball mill, whereby the discharge is fed to the primary cyclones. The underflow is then processed in a flash flotation cell, which recovers around 60% of the PGE within the final concentrate (Cole & Ferron, 2002). No allowance is given for specialised processing of different domains within the ore body, (i.e. Normal reef, NP2 reef and P2 reef).

The concentrate from the first cell in the rougher bank may be fed into the final concentrate stream, but the rest of the rougher concentrates are fed into the cleaner circuit. The concentrate from the scavenger circuit can be routed to the cleaner circuit or back to the head of the rougher circuit. There is no regrinding at any point within the rougher, scavenger or cleaner circuits. The cleaner circuit consists of three column cells aligned in series, with stage recovery at each column as low as 30% (Cole & Ferron, 2002)

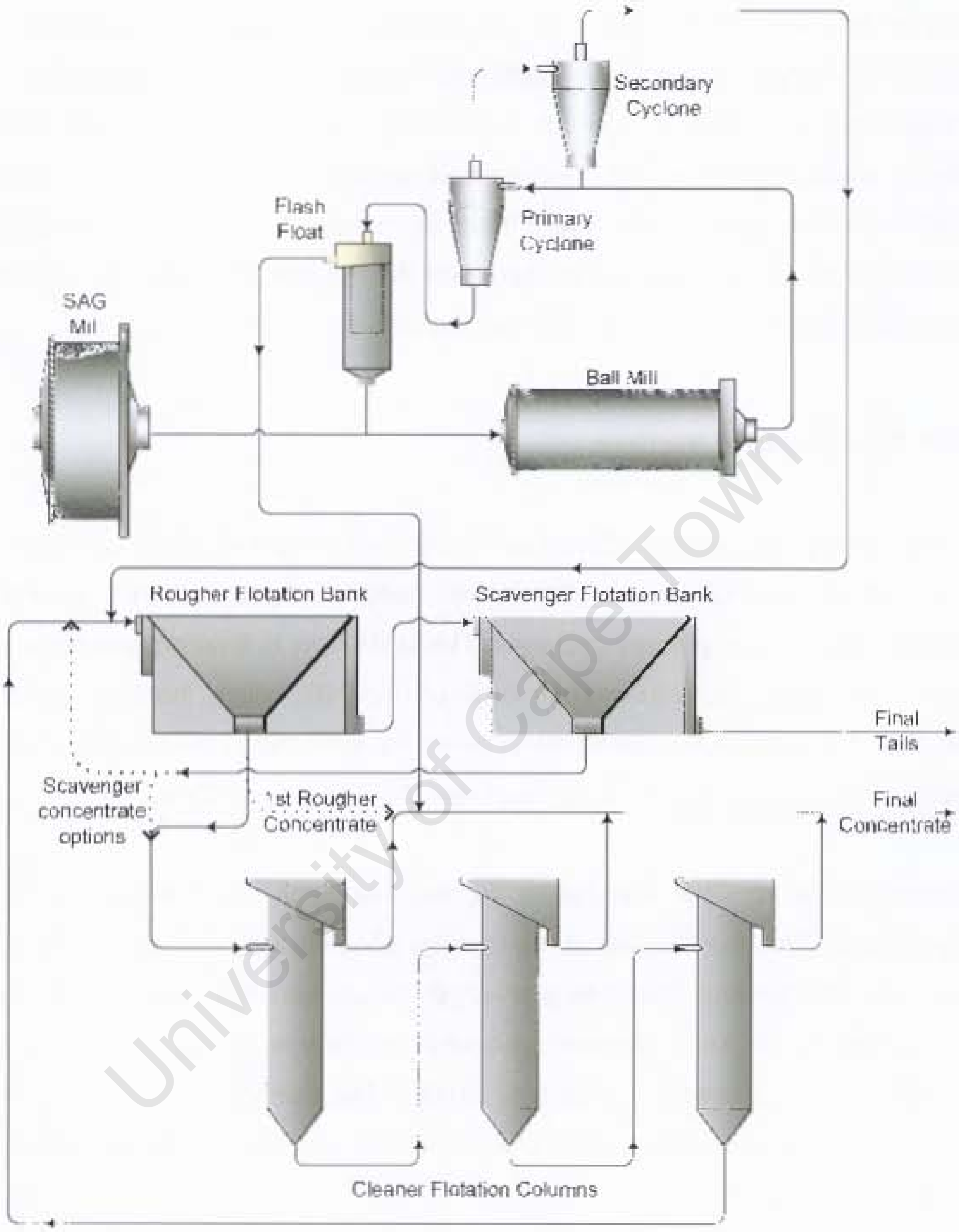


Figure 2.12: *Metallurgical flowsheet for Northam Platinum Limited, showing the major comminution and flotation processes within the concentrator (Adapted from Snodgrass et al., (1994) and Cole and Ferron, (2002)).*

2.4 Process Mineralogy

Process mineralogy, a discipline which combines mineralogy and mineral processing (Henley, 1983; Baum *et al.*, 2004; Xiao & Laplante, 2004), is a relatively new field of research reviewed in 1983 by Henley and only once subsequently (Baum *et al.*, 2004). Within the recently re-introduced term GEOMET (Haol, 2008), process mineralogy functions as an integral stage. GEOMET, an abbreviation of geometallurgy, is an ambitious and extensive overview of the life of mine, aiming to holistically integrate information from reconnaissance geophysics to reclamation, as well as make that information easily transferable between each stage (Haol, 2008). Within a GEOMET flowsheet the aim of the process mineralogist is to provide information on specific aspects of the ore mineralogy and mill products and, in so doing, to help the chief metallurgist optimize treatment flowsheets (Henley, 1983). As such, the process mineralogist needs an extensive knowledge, not only of ore mineralogy but also of the relevance of mineralogical characteristics to beneficiation processes (Henley, 1983). Figure 2.13 shows the major inputs from a process mineralogist during the setup and implementation of a new plant. The final stage is ongoing, with a constant need to monitor concentrates and tailings (Henley, 1983).

This final stage, or optimization process, has been further developed since 1983 (Fragomeni *et al.*, 2005; Baum et al., 2004; Xiao & Laplante, 2004) and this is especially shown by an example of work carried out at Raglan in Canada, where process mineralogy techniques have been implemented since 1998 (Fragomeni *et al.*, 2005). At Raglan the orebody has been classified into three distinct end members (disseminated, net-textured and massive sulphides). The term 'end member' is appropriate for the Raglan orebody as there is a continuum between these three ore types. However, at Northam there is no continuum between the three reef types so the term 'domain' is preferred. At Raglan each end member was sampled and characterised producing a robust prediction of recovery at various grind sizes. These were then mathematically combined in the current volume percentage for run of mine to determine the impact of end member distribution on metallurgical performance as a function of grind size (Fragomeni *et al.*, 2005).

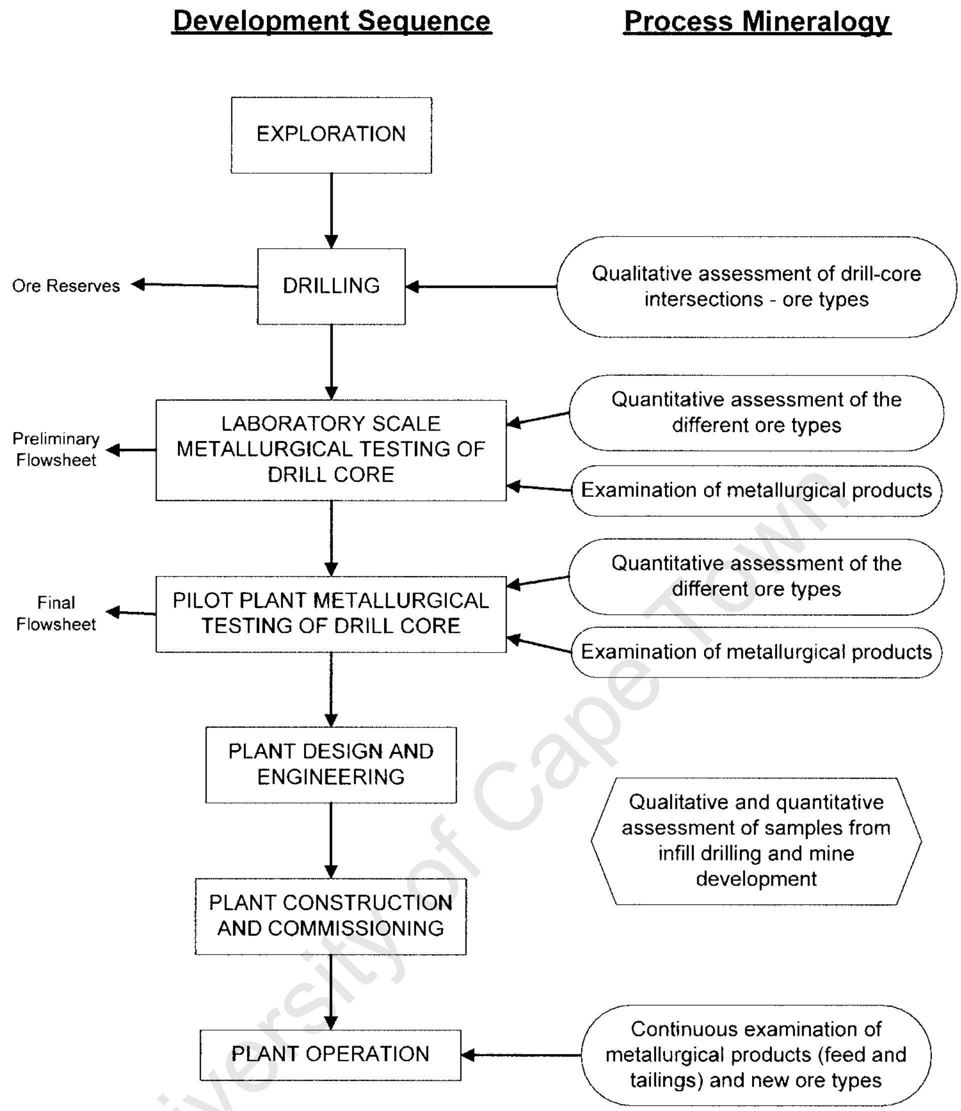


Figure 2.13: *Summary of the relationship between process mineralogy and the development and optimization of a metallurgical treatment plant. (Adapted from Henley, 1983).*

Since 1983 two key problem areas have been identified (Baum *et al.*, 2004). The first is the problem of representative sampling within a concentrator survey (and likewise geologically in a drill-core survey). The second is the problem of attaining a quantitative mineral measurement system. The paymetal content (assays) of the samples is not necessarily dependent upon the behaviour of the minerals and it is the mineral content, size and textures which determine liberation characteristics (Baum *et al.*, 2004) and thus very relevant to this study.

2.4.1 Quantitative Mineral Measurement Systems

The task of quantitatively characterising the PGM has in the past been a difficult and time consuming process. PGM are not only extremely fine-grained, but also their presence in only trace amounts means that traditional methods such are ore microscopy and X-Ray diffraction are not suitable (Merkle & McKenzie, 2002).

In the last three decades, but particularly in the 1990s two quantitative mineral measurement technologies have been developed. The first of which is QEMSCAN (Quantitative Evaluation of Minerals by SCANing electron microscope) developed by CSIRO, Australia, and the MLA (Mineral Liberation Analyser), by the Julius Krutschnitt Research Centre, Australia (Baum *et al.*, 2004).

While QEMSCAN and MLA are both examples of automated SEM based image analysis, they emphasise slightly different strategies. MLA works by gathering back scattered electron (BSE) images in the initial spatial survey, after which the minerals are further characterised using X-ray elemental analysis (Gu, 2003). MLA is particularly good for PGM based liberation analysis (Frandrich *et al.*, 2007; Lastra, 2006). QEMSCAN dispenses with the preliminary BSE survey and rather gathers X-ray elemental maps (Gottlieb *et al.*, 2000). The chief advantage of MLA is the greater efficiency and speed, whereas the QEMSCAN technique is more thorough, avoiding ambiguities in similar BSE grey scale values for different minerals.

QEMSCAN and MLA both have the ability to return an overall mineral assemblage and provide mineral liberation data for predetermined size fractions (Baum *et al.*, 2004). Despite their revolutionary effect on quantitative process mineralogy, the QEMSCAN and MLA techniques are still dependent on representative sampling (Gottlieb *et al.* 2000; Baum *et al.*, 2004; Lotter *et al.*, 2003)

An example where QEMSCAN techniques have been used effectively is within the Sudbury mining district of Ontario, Canada. The Ni-Cu-PGE sulphide ores are mined by Falconbridge Limited. The feed to their Strathcona mill consists of three distinct end members, which they labelled AB, B and C (Lotter *et al.,* 2003). Initial flotation testing showed that ores B and C performed similarly to the Strathcona baseline curve, but that ore AB has a far inferior Ni grade and recovery. The Strathcona flowsheet consists of a rodmill, from which the discharge heads into primary rougher, secondary rougher and scavenger flotation series. QEMSCAN analysis of the three ores showed that pentlandite liberation was essentially the same for all three ore types, but that liberated pentlandite was floating within the primary rougher for ores B and C, yet was being delayed in ore AB. Having ruled out liberation issues as being the problem, it was decided that poor metallurgical performance must be related to pulp chemistry (Lotter *et al.*, 2003). QEMSCAN information of ferromagnesian mineral content revealed that ore AB contained 2.6 times that in ore B and 5.0 times that in ore C. The exact reasons for the strong correlation between high ferromagnesian content and poor pentlandite recovery is not entirely understood, it may be related to alteration products, such as talc, or slime coatings covering the pentlandite surfaces.

2.4.2 Representative Sampling

In order to be certain that the plant or mine data generated from a QEMSCAN or MLA study is valid, the plant or the orebody must have been representatively sampled. This problem of representative sampling is the subject of current research (Fragomeni *et al.*, 2005; Baum *et al.*, 2004; Lotter *et al.*, 2003), and with respect to the sampling of an active stope two key points can be delimited.

1. The laboratory scale flotation tests must be reproducible.
2. The flotation sample must be representative of the ore

These twin problems were addressed by Lotter in (1995) who showed that a fundamental approach to ore sampling and preparation, as well as replicated flotation tests with quantitative diagnostics, reduced the effect of ore variance and increased the reproducibility of flotation tests.

Process mineralogy has had some notable successes, aside from the Raglan orebody and the Lihir gold mine. Process mineralogy techniques have also been successfully implemented at the El-Indio gold-silver-copper operation in Chile (Baum *et al.* 1989) as well as being partly implemented to identify a problem of fluorine contamination in copper flotation (Pangum *et al.* 2001) and in phosphate mine planning (Sant'Agostino *et al.* 2001), which testifies as to its wide applicability. Its complete implementation at the El-Indio gold-silver-copper operation led to improvements in precious metal recovery and equipment performance as well as a finalised process flow-sheet that was custom fitted to the ore (Baum *et al.* 2001).

2.5 Key Questions

Within the context of previous work reviewed here this investigation focuses on the three main questions outlined in the Introduction.

1. How is the mineralogy of the three reef types at Northam Platinum Limited (Normal, NP2 and P2) quantitatively different?

2. Will the distinct mineralogical characteristics of the three reef types produce different mineral processing performances? In particular -
   a. How will milling time and flotation performance vary between reef types?
   b. How is the flotation performance of each reef affected by change in grind and copper addition?

3. If there is a difference in mineral processing performance, can it be explained in terms of some, or all of the following?
   a. Variations in sulphide textures and/or liberation.
   b. Variations in gangue mineral type and proportions.
   c. Variations in gangue alteration or textural development.

3 Experimental Methods

3.1 Ore Preparation

Bulk samples of ore were collected at Northam Platinum mine under the supervision of chief geologist Damian Smith and freighted to the University of Cape Town. Three reef types were investigated, namely the Normal 12/39, P2 3/19 and the NP2 11/47, where the numbering refers to the level/cross cut locations.

Reef samples were unpacked and air-dried separately (Figure 3.1a). During the drying, grab samples from each reef, consisting of the larger chips, were removed to prepare thin sections, polished thin sections and ore mounts (Figure 3.1b). The amount removed was carefully chosen to be representative of the whole reef as established by in-situ mapping of the stope panels.

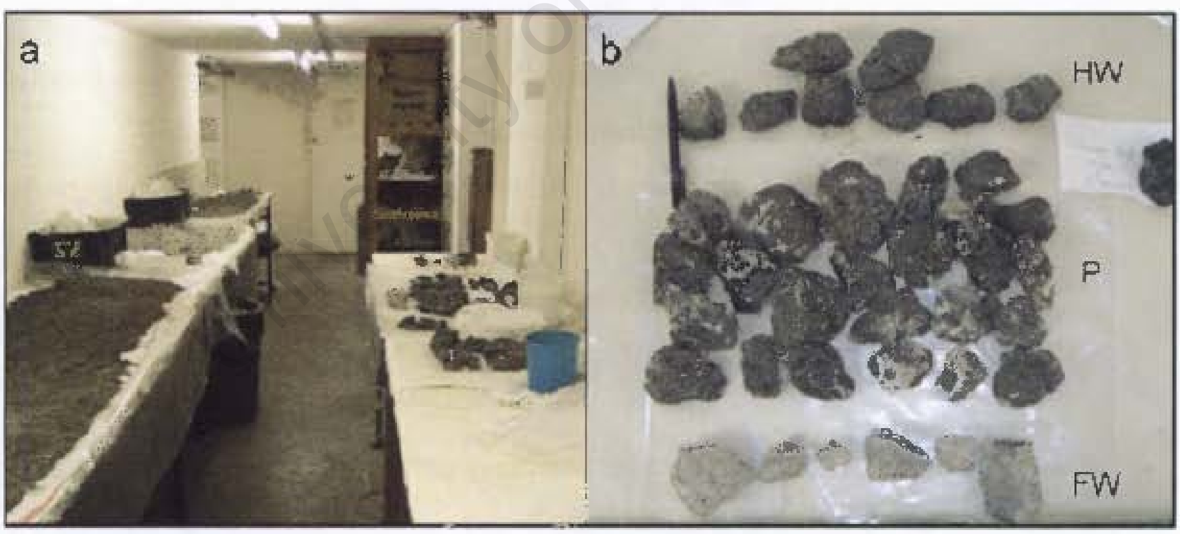


Figure 3.1: *Photos of ore preparation: a Drying of bulk samples on plastic sheeting. b Grab samples from the bulk sample of the Normal reef, organised here in stratigraphic order with the hanging wall at the top. HW – Hanging Wall, P – Pegmatite, FW – Footwall.*

After drying, the remainder of each bulk sample was screened through a 16 000 aperture stainless steel filter. The filtrand was subsequently broken down through the jaw crusher in the Department of Geological Sciences before recombining with the rest of the respective bulk sample.

Each reef was then screened through a 2 cm aperture stainless steel screen to remove fines and then fed through the cone crusher in the Chemical Engineering Department to ensure there were no chippings left in the bulk sample larger than two centimetres. The filtered fines and the sample produced from the cone crusher were recombined to produce the final bulk sample for each reef.

Once the bulk samples were prepared they were then thoroughly mixed on a steel mat, before being split into four homogenised samples, each sample being approximately 10 kg. Each of the four samples was then fed through the rotary splitter, which divided the sample down into 1 kg proportions. Once the feed was exhausted, these 1 kg samples were bagged and labelled.

3.2 Representative Sampling

It is possible that the sampling technique employed by the mine provided skewed proportions of the stope face. Therefore it is prudent to establish whether the samples received are representative of the bulk. In order to do this, two techniques were used. Both of these techniques make use of analytical data for individual components of the Normal reef (Roberts *et al.*, 2007). The first technique provides an estimate for the bulk composition assuming observed elemental proportions of the various components (hanging wall (L1), pegmatite (L2), footwall (L3)) multiplied by the quantities (Q_1, Q_2, Q_3) present at the stope face (Equation 1).

$$L_1Q_1 + L_2Q_2 + L_3Q_3 = B_{calc} \qquad (1)$$

These calculated elemental proportions were then compared with the measured bulk analyses obtained in this study by ICP-OES (see Section 3.6). Results are shown in Figure 3.2 where it can be seen that the major oxides compare closely, indicating the absence of any anomalous representation of reef components.

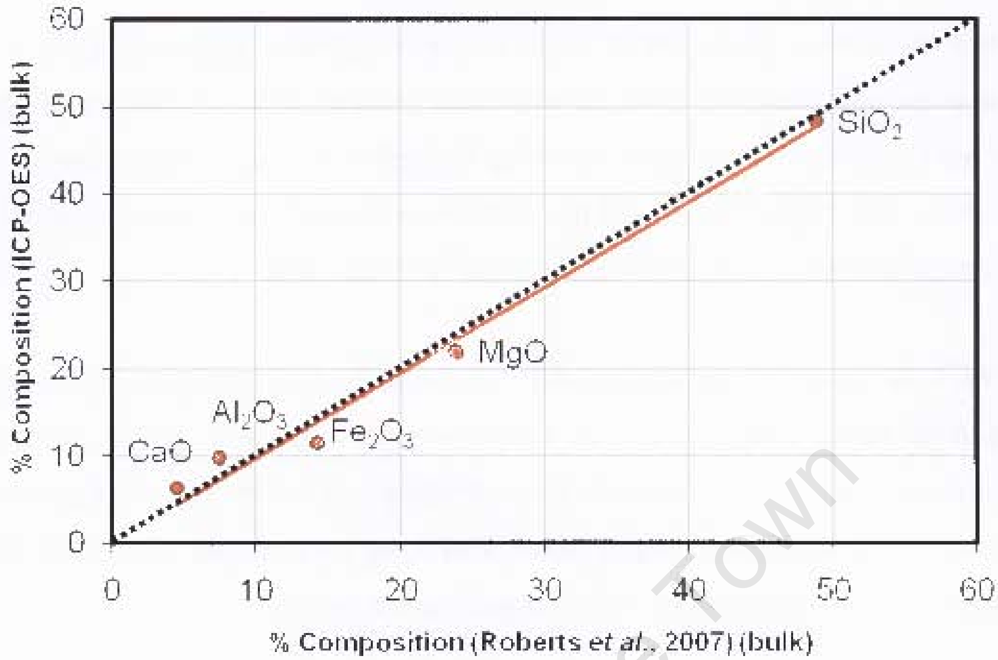


Figure 3.2: *A graph showing a comparison between the calculated major element oxide compositions (red) and the measured major elemental oxide compositions. The red line represents a trendline and the dashed black line represents the perfect 1:1 line.*

An alternative approach involving multiple linear regression allows the estimation of individual reef components using their major oxide proportions. This reverse method may be more revealing in the sense that the results provide a direct measure of reef components in the mill feed. The second technique models Equation (1) using a multiple linear regression.

With appropriate constraints (e.g. $P_1 + P_2 + P_3 = 1$) this model predicts a slight weighting in favour of the hanging wall, namely; 10% hanging wall, 86% pegmatite and 4% footwall, whilst the actual proportions are 5% hanging wall, 90% pegmatite, and 5% footwall. The results derived from the model are shown in Table 3.1 alongside those measured by Roberts *et al.*, (2007).

Table 3.1: *Table showing the comparison between the measured major element proportions of the reef and the linear regression model from this work.*

Element	Measured (Roberts et al., 2007)	Linear Regression Model
SiO_2	48.8	48.5
Al_20_3	7.4	8.6
MgO	23.8	23.0
CaO	4.5	5.3
Fe_2O_3	14.2	11.6

As can be seen from Table 3.1, SiO and MgO values are predicted very accurately by the linear regression model, with good predictions for Al_2O_3 and CaO, but a poor prediction for Fe_2O_3. However, it is important to specify that the data available for the major elemental oxides, within the individual lithologies, is too sparse to provide a definitive conclusion. For example, the calculated values for CaO and Al_2O_3 were sensitive to the inclusion or exclusion of certain sample sets.

From these two techniques, and considering the anticipated range in reef component proportions, with the likelihood of variable weighting occurring during the mining activity these results are considered acceptable for the subsequent experiments.

3.3 Mineralogical Characterisation

Mineralogical characterisation of both silicate and sulphide mineral phases were performed using optical microscopy on a Zeiss Petrographic microscope (Appendix B1). Individual sample descriptions are recorded in Appendix C.

3.4 Electron Microprobe

Analysis of olivine to determine nickel content was carried out on a JEOL Microprobe (Appendix B2) using polished thin sections created from grab samples from the original bulk samples (Figure 3.1.b). The nickel content was probed in order to determine how much of the nickel reported by chemical assay was present in gangue minerals such as olivine.

Spot analysis selection was determined in order to maximise areal and compositional coverage. The first olivine grain was covered systematically to observe compositional variations within a grain (zoning). Depending on the outcome of this exercise the rest of the olivine grains within the slide were then investigated. If the first olivine showed a significant variation in composition with spot point, then each subsequent olivine was investigated the same way. If the first olivine showed no variation with spot point then the remaining olivine grains could be investigated using just one spot point. In practise variation of composition between olivine grains (up to 10%) within a slide was greater than internal variation within an olivine grain.

3.5 Quantitative Evaluation Of Minerals By Scanning Electron Microscopy (QEMSCAN)

Feed samples for batch flotation tests were milled to the required grind, split (using a FRITSCH Rotary Samples Divider) into three size fractions and sent for QEMSCAN analysis to Intellection Pty Ltd, Brisbane, Australia. Samples from the same split were sent for data validation to Mintek in Johannesburg for ICP-OES (see appendix B3). Eighteen sized fractions were sent consisting of six from each reef in three size fractions (+75 μm, -75/+38 μm, -38 μm) and two grind sizes; standard (60% < 75 μm), and fine (80% < 75 μm). At Intellection the feed samples were representatively split into a sub aliquot and permanently mounted in an epoxy resin. The use of three size fractions minimised the effects of stereological variations.

3.5.1 Data Validation

Comparison of the chemical analysis from ICP-OES and the QEMSCAN analysis reveals excellent correlations between the major elements (Figure 3.3), particularly magnesium, aluminium, calcium and iron. Silicon values show a slight departure, with either the chemical analyses slightly overestimating, or the QEMSCAN slightly underestimating, the values.

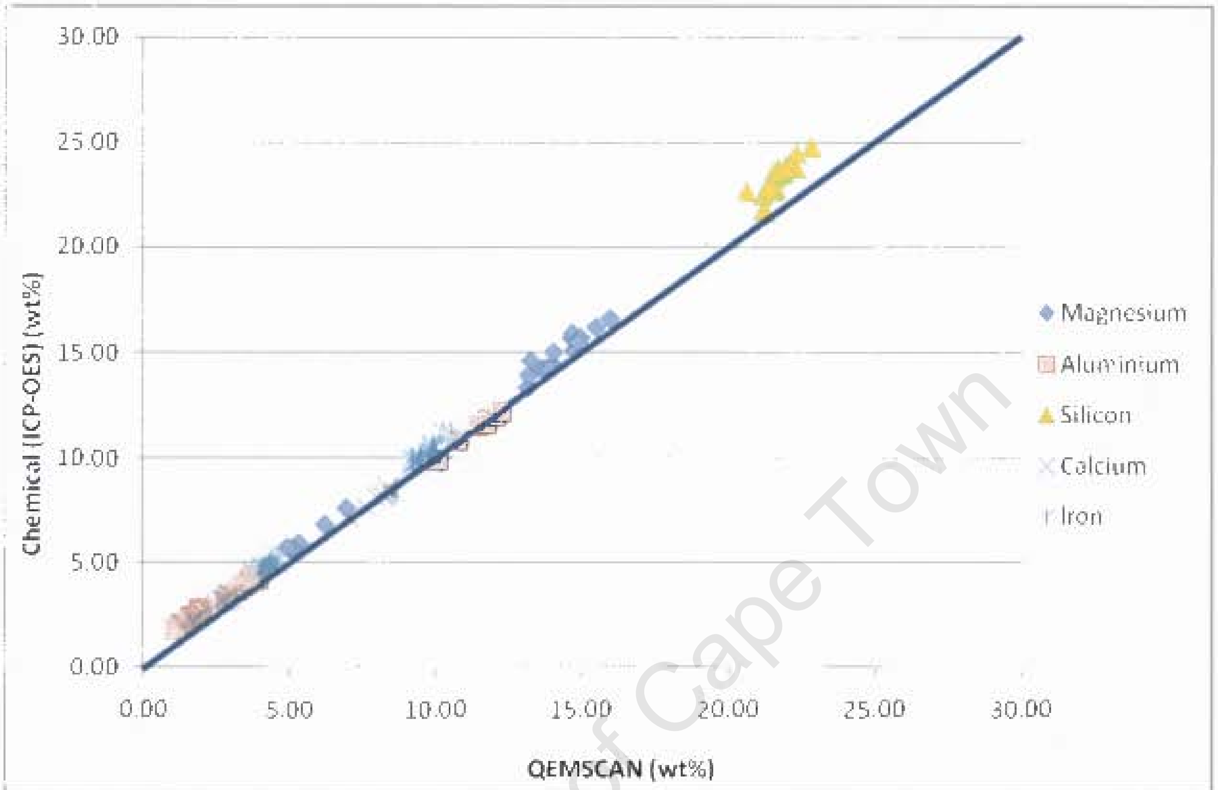


Figure 3.3: *Graph showing the correlation between the chemical assay and the QEMSCAN assay for the major elements. The line demarcates a 1:1 relationship.*

Investigation of the sulphide elements (Cu, Ni and S) reveals a slightly less perfect correlation (Figure 3.4a, b). In particular there are two obvious anomalies circled in red in Figure 3.4a. The furthest right with a QEMSCAN copper value of 3.06 is clearly anomalous, and refers to the P2 reef -38 μm size fraction at a standard grind. The second ringed value is a combined value of all the size fractions for the P2 reef at a standard grind and therefore incorporates the first value during the calculation. This incorporation forces the observed anomalous result. Figure 3.4b shows that sulphur values are consistently below the 1:1 measurement line. This is probably due to overestimation by the QEMSCAN analysis, but is still considered acceptable.

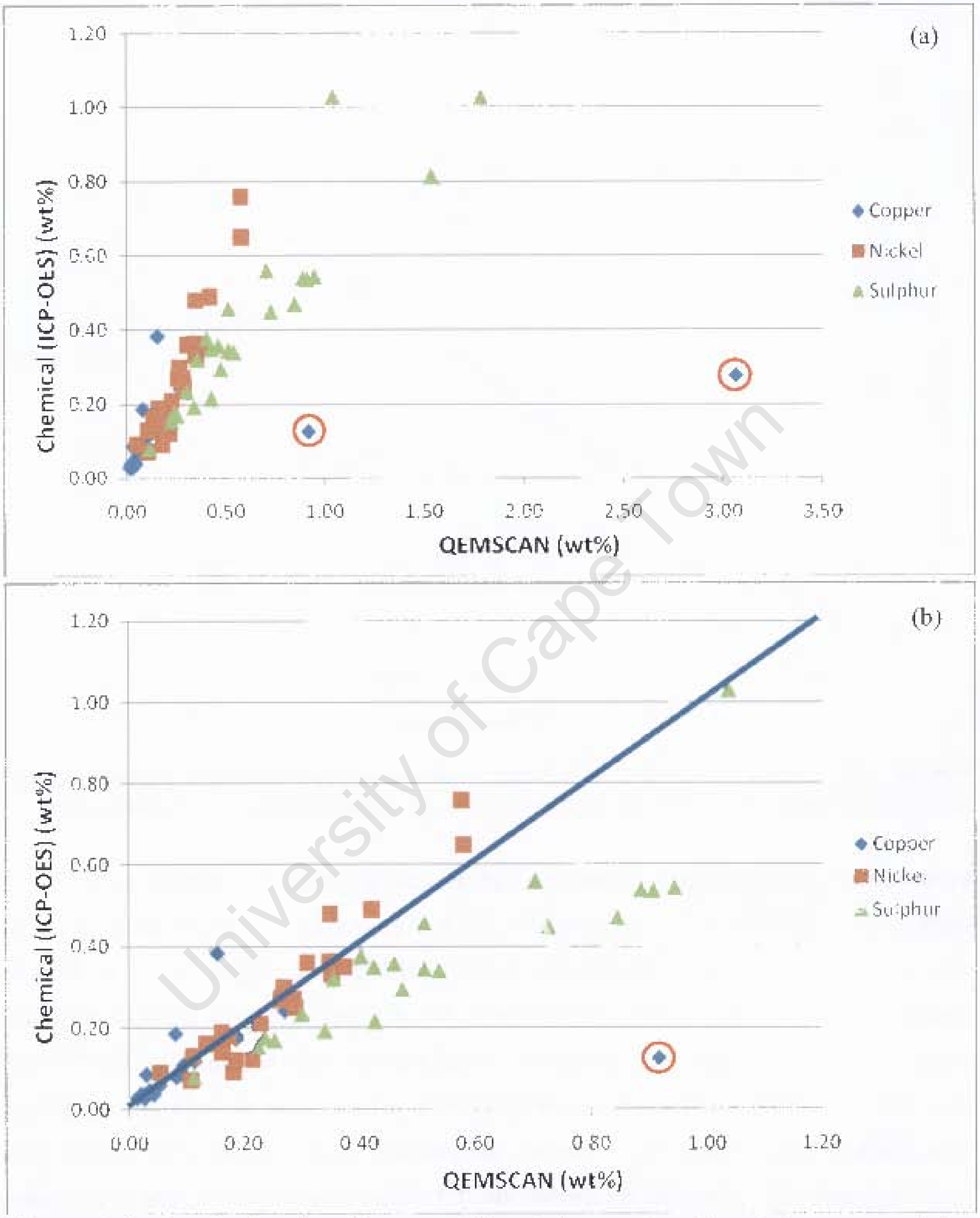


Figure 3.4: *Graph showing the correlation between the chemical assay and the QEMSCAN assay for trace elements Cu, Ni and S. Figure 3.4a shows all the data points, including two anomalous values circled in red. Figure 3.4b focuses in on the lower left corner with the solid blue line demarcating a 1:1 relationship.*

3.5.2 Analysis Methods

Two analysis methods were used to gather information from the samples sent. The first was Bulk Mineralogical Analyses (BMA). This is a linear analysis method where each mounted float feed sample is scanned in the x-direction with the y-direction line spacing pre set to ensure that each particle is intersected once (Figure 3.5). The entire block is scanned producing a high statistical population and providing information on size-by-size bulk mineralogy.

Figure 3.5: *Image showing a section of BMA analysis. Each colour represents a different mineral.*

The second method was Particle Mineralogical Analyses (PMA). This is a particle by particle analysis method which allows mineral associations to be visually represented for interpretation (Figure 3.6). Sulphide liberation was the focus of this study so only sulphide composites were preferentially analysed. The mounted flotation samples were used and a minimum of 3000 particles are analysed for each PMA analysis. This is an excellent tool for visually assessing liberation and locking information of the BMS as well as looking at secondary alteration characteristics.

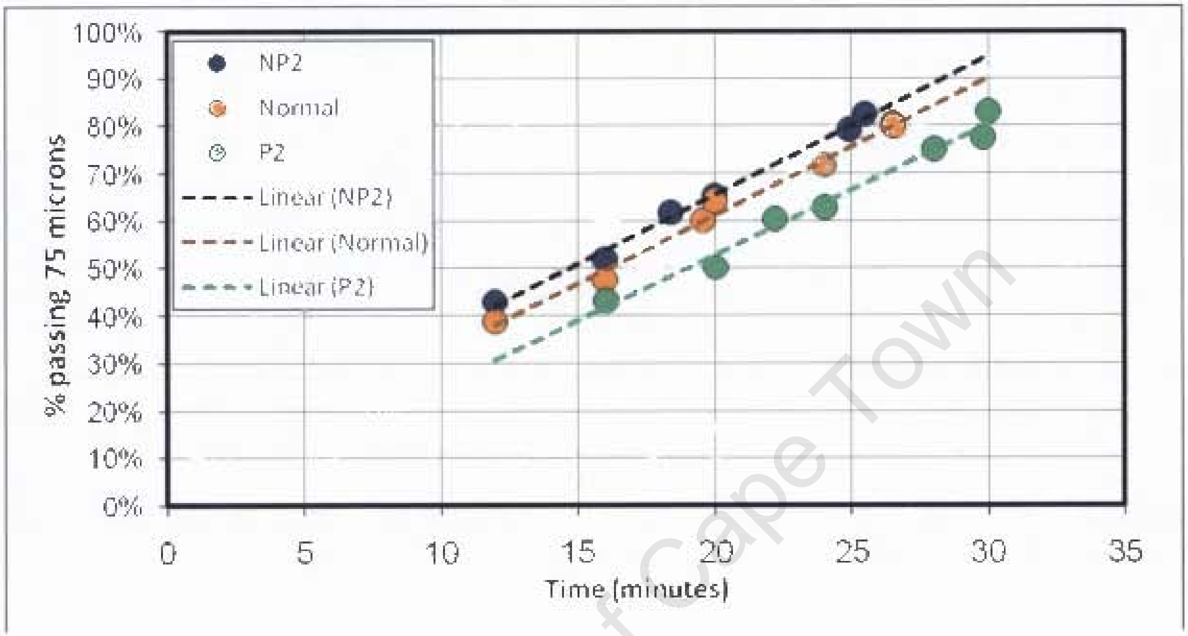
Figure 3.6: *Image showing PMA analysis of various particle grains. Each colour represents a different mineral with associations readily discernible.*

3.6 Whole Rock Chemical Analysis

Samples were sent for chemical analysis to Mintek in Johannesburg by Inductively-Coupled Plasma Optical Emission Spectrometry (ICP-OES) (see Appendix B3). Eighteen size fractions were sent consisting of six from each reef in three size fractions (+75 μm, +38 μm/ -75 μm, -38 μm) and two grind sizes. The samples were prepared from 1 kg feed fractions and split using the QUANTACHROME Rotary Micro Splitter. The samples sent to Mintek were analysed for major elements (Mg, Al, Si, Ca, Ti, V, Cr, Mn, Fe, Co, Ni, Cu, Zn and Pb) using 2 g samples from each size fraction. The determination limits for each element was 0.05%.

3.7 Grinding Curves

Grinding curves were calculated using at least four 1 kg samples for each reef. The mill used was an Erietz laboratory stainless steel rod mill with an inner chamber diameter of 258 mm. 20 stainless steel rods were used in three different sizes; 6 x 22 mm, 8 x 18 mm, 6 x 14 mm. 500 ml of tap water was added to 1 kg of ore and the mill rotated at a fixed speed of 90 rpm. The four samples for each reef were run for different times, (e.g. 12, 16, 20 and 24 mins). At the end of each run the pulp was removed and filtered to produce a 'cake'. An eighth of this cake was removed (one sixteenth from each side) and screened through 75 µm filter The percentage passing 75 µm obtained for each time was then measured so that the time required to mill to standard and fine grinds could be calculated (Figure 3.7).



Grind	% Passing 75 µm	Normal (mill time)	NP2 (mill time)	P2 (mill time)
Standard	60	19.5	18.0	23.0
Fine	80	26.5	25.0	30.0

Figure 3.7: *A graph showing the grinding curves for the three main reef types of Northam Platinum Ltd. With a table showing the time taken to mill to standard and fine grinds for each of the three reef types*

A test run was then performed to check whether the predicted times for the standard and fine grinds were correct. From the grinding curves it can be seen that the three reef types required distinctly different milling times to achieve the required particle size distribution (PSD) within the pulp, demonstrating different hardness.

A Malvern microsizer was used to check the PSD of the three reefs. Figure 3.8 shows that for standard and fine grinds, each reef type has essentially the same PSD.

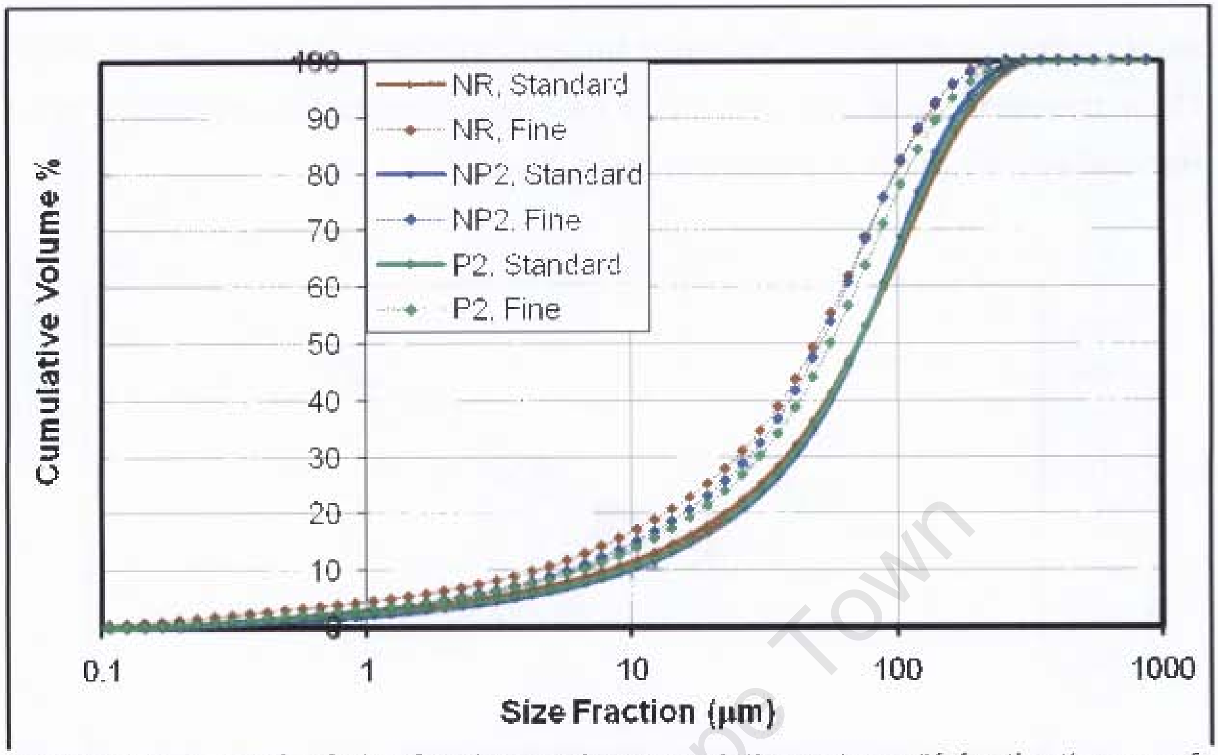


Figure 3.8: *A graph of size fraction against cumulative volume % for the three reefs at two different grind sizes. The finer grind is shown with a dashed line.*

3.8 Laboratory Scale Flotation Tests

For the batch flotation experiments each reef type was tested using two reagent suites in order to see how the different reef types responded to copper sulphate addition and changes in grind.

3.8.1 Reagents

Reagents used for the batch flotation experiments are based on those used in the beneficiation of Merensky ore at Northam Platinum Limited. The frother used for all conditions was Sasfroth 200. The depressant used for all conditions was KU-47 at 32 g/t. The collector mixture of sodium isobutyl xanthate (SIBX), and proprietary reagents Sascol 61 and Sascol 105 were used for all flotation experiments at a fixed dosage of 154 g/t, 67 g/t and 17g/t respectively. They were always mixed before addition to the pulp. The activator was copper sulphate. Activator was only used for one of the two reagent suites, and when used had a concentration of 55 g/t.

3.8.2 Water

All batch flotation experiments were performed using synthetic plant water (Table 3.2). This was obtained by adding various chemical salts to distilled water. Exact ionic concentrations for plant water will vary across all concentrators within the Bushveld. However, it can be expected that these ions will be present at all Merensky concentrators since they are derived from minerals in the ore, which are similar across the reef types.

Table 3.2: *A Table showing the concentration of ions present within the synthetic plant water (Wiese et al., 2005).*

Ion	Ca^{2+}	Mg^{2+}	Na^{+}	Cl^{-}	SO_4^{2-}	NO_3^{-}	NO_2^{-}	CO_3^{2-}	*TDS*
Concentration (ppm)	*80*	*70*	*153*	*287*	*240*	*176*	-	*17*	*1023*

3.8.3 Flotation Procedure

The procedure followed was as described by Wiese *et al.* (2005) in their work on Merensky ores. After milling the slurry was transferred to a modified 3 L Leeds flotation cell and the volume made up using synthetic plant water to produce 35% solids. At this point a feed sample was taken. An impeller, maintained at a fixed speed of 1200 rpm, was used to thoroughly mix and suspend the pulp. Reagents were then added and allowed to condition. Copper sulphate was conditioned for 5 minutes. The collector blend; SIBX, Sascol 61 and Sascol 105 were simultaneously added and conditioned for 2 minutes. KU-47 was conditioned for 2 minutes and Sasfroth 200 was conditioned for 1 minute. The order of reagent addition was always activator, collectors, depressant and frother. After conditioning, air was pumped through at 7 L/min and a sample taken every 15 seconds by scraping the froth into a collecting pan. The froth height was manually maintained at a constant depth of 2 cm. Four collecting pans were used to collect concentrates at 2, 6, 12 and 20 min. At the end of the procedure two tails samples were taken. The feeds, concentrates and tails samples were then filtered, dried and weighed before analysis. All batch flotation experiments were performed in duplicate to check reproducibility.

3.9 Analysis of Flotation Performance

In order to obtain copper and nickel data the finely milled feeds, concentrates and tails were dissolved in aqua regia, made up to a known volume, filtered and analysed on an atomic absorption spectrophotometer (AAS) using a Varian SpectraAA (Appendix B4). In order to obtain sulphur data the finely milled feeds, concentrates and tails were left in powder form before being analysed on the LECO 423 sulphur analyser (Appendix B5).

3.9.1 Reagents and Sample Preparation for AAS

The reagents used to make the aqua regia were hydroflouric acid (HF) (~40%), nitric acid (HNO_3) (~60%), perchloric acid ($HClO_4$) (concentrated) and hydrochloric acid (HCl) (~30%). The hydrochloric acid and the hydrofluoric acid were mixed in the ratio of 4:1. The milled samples were usually fine enough for immediate digestion. However the first concentrate, which was the most important for accurate grade-recovery curves, needed to be pulverised and digested twice to ensure the reliability of the nickel and copper readings.

Aliquots from each feed, concentrate and tail according to type were weighed out into 250 ml wide-mouthed Erlenmeyer flasks, which were pre-cleaned between samples using warm water, soap and distilled water. Concentrate samples required ~0.1 g of material, whereas feeds and tails required ~0.5 g. 10 ml of the HCl/HF mixture was added to the sample and brought to the boil. HNO_3 was then added and the sample boiled until the colour changed from deep brown to ochre. Finally 5 ml of $HClO_4$ was added and the sample left to boil until a stable white cloud formed within the beaker indicating completion of the reaction. Once complete the sample is transferred to a 100 ml flask and made up to 100 ml using distilled water. The sample was finally filtered through Whatman No1 into a sample bottle. This filtrate was then analysed using the AAS.

3.9.2 Calculation of Mineral Recovery

Mineral grade-recoveries (chalcopyrite, pentlandite, and total sulphide) were calculated by converting copper, nickel and sulphur values (Appendix D) to chalcopyrite ($CuFeS_2$), pentlandite ($(Fe,Ni)_9S_8$) and total sulphide values. Pyrrhotite ($Fe_{1-x}S$) grade-recoveries were calculated by mass balance.

Copper within the Merensky reef is exclusively associated with BMS such as chalcopyrite and cubanite. Copper is known to have a negligible association with the silicate gangue minerals, this, coupled with the very low quantities of cubanite, means the conversion to chalcopyrite values was a single multiplication factor.

Nickel within the Merensky reef can be associated with olivine, orthopyroxene and pyrrhotite as well as pentlandite. Since nickel has this association with various minerals a direct conversion of nickel to pentlandite would produce artificially low recoveries. Silicate nickel contents were therefore quantified and removed from the total nickel 'budget' for each float, allowing accurate determination of pentlandite recoveries. For pentlandite grades the assumption was made that all nickel reported within concentrates was pentlandite nickel.

Sulphur within the Merensky reef is mostly deported as the three main sulphides, pyrrhotite, pentlandite and chalcopyrite, with some accessory pyrite and other sulphides (e.g. cubanite). The main PGM are all associated with these sulphides. Sulphur has a negligible association with silicate gangue minerals so the conversion to total sulphide values was a single multiplication factor. The multiplication factor (0.36) was based on the ratio of the three main sulphides observed within the recalculated feeds.

Calculated pyrrhotite grades and recoveries may be subject to large errors due to their reliance on several assumptions. In particular, recoveries are sensitive to variations in the mineral stoichiometry of the copper and nickel bearing sulphides as well as to fluctuations in the monosulphide (e.g. pyrrhotite) to disulphide (e.g. pyrite) ratio.

The focus of this investigation was these major minerals, with the assumption that PGM and PGE are intimately associated with these base metal sulphides. No further PGE or PGM work was performed as part of this investigation.

4 Geological Characterisation

4.1 Petrological Characterisation

While several visits to Northam Platinum and reference to published accounts (Viring & Cowell, 1999; Roberts *et al.*, 2007) have provided generalised reef information (see Chapter 2), the bulk sampling localities were detailed by analyzing logs recorded at the time of stope blasting; it was therefore possible to estimate the proportions of each lithological component. These are shown here using the same diagrams from chapter 2 to show any likely losses of PGE paymetal content and any likely excessive inclusions of gangue material (Figures 4.1, 4.3, 4.5).

By looking at main mineral abundances and grain size distribution it was possible to ascertain the main petrological differences between the reef types. Aside from broad petrological differences careful attention was made to factors that may affect liberation or processing, such as the degree of alteration of silicate minerals, sulphide textures and sulphide grain size.

4.1.1 Normal Reef

The Normal reef bulk sample was characterised by a relatively thick inter-chromitite pegmatite ~150 cm, bounded by a thin (8cm) hangingwall melanorite and a thin (8cm) footwall anorthosite (Figure 4.1). Consequently one can be confident that the limit s of PGE mineralization have been encompassed satisfactorily. In other words the particular stope blast met the mine requirements, with limited oversampling of barren sections of the hanging wall and footwall.

4.1.1.1 Hanging Wall Melanorite (3A)

Inspection of the hanging wall bulk samples revealed that it closely resembled generalised descriptions of this layer (Viring & Cowell, 1999; Roberts *et al.*, 2007) in comprising a medium grained (1-3 mm) orthopyroxene mesocumulate with clinopyroxene oikocrysts (up to 2 cm), plagioclase, biotite, chromite and base metal sulphides (BMS) (Figure 4.2a). Individual chips in the grab sample showed variation in plagioclase content, ranging from minor interstitial patches (orthopyroxenite adcumulate) to greater proportions (melanorite orthocumulate). There were several features indicative of plastic deformation during compaction – namely, undulose extinction, deformation twins within plagioclase and indented faces between pyroxene grains suggesting grain boundary migration. However, plagioclase was occasionally, along with biotite, strain free, implying there was some post compaction crystallization. Some post compaction plagioclase was oikocrystic reaching ~4-5mm.

Secondary minerals were revealed by sub-solidus clinopyroxene exsolution from orthopyroxene, orthopyroxene alteration to biotite, amphibole, chlorite and fine grained silicates (probably talc) and minor alteration of plagioclase through sausseritisation (formation of minerals through the alteration of sodic feldspar, e.g. paragonite). Sub-solidus clinopyroxene exsolution displayed a variety of habits depending on the ambient temperature at the time. Coarse blebs indicated exsolution occurred at a high temperature, where as long, pointed needles indicated low-temperature exsolution

BMS occurred as medium grained (2-3 mm) composites of pyrrhotite, granular and minor flame pentlandite, chalcopyrite and minor pyrite, interstitial to silicate grains, and often associated with chromite and biotite grains. Individual sulphides within each composite showed a variety of grain sizes. The formation of flame pentlandite is crystallographically controlled secondary exsolution from pyrrhotite and was always a very minor component and very fine grained. BMS also occurred as fine grained monominerals interstitial to, and included within silicates. Chalcopyrite is the dominant fine-grained monomineral and often occurred as in-fill within micro-fractures, a feature which strongly suggests mobility during brittle deformation.

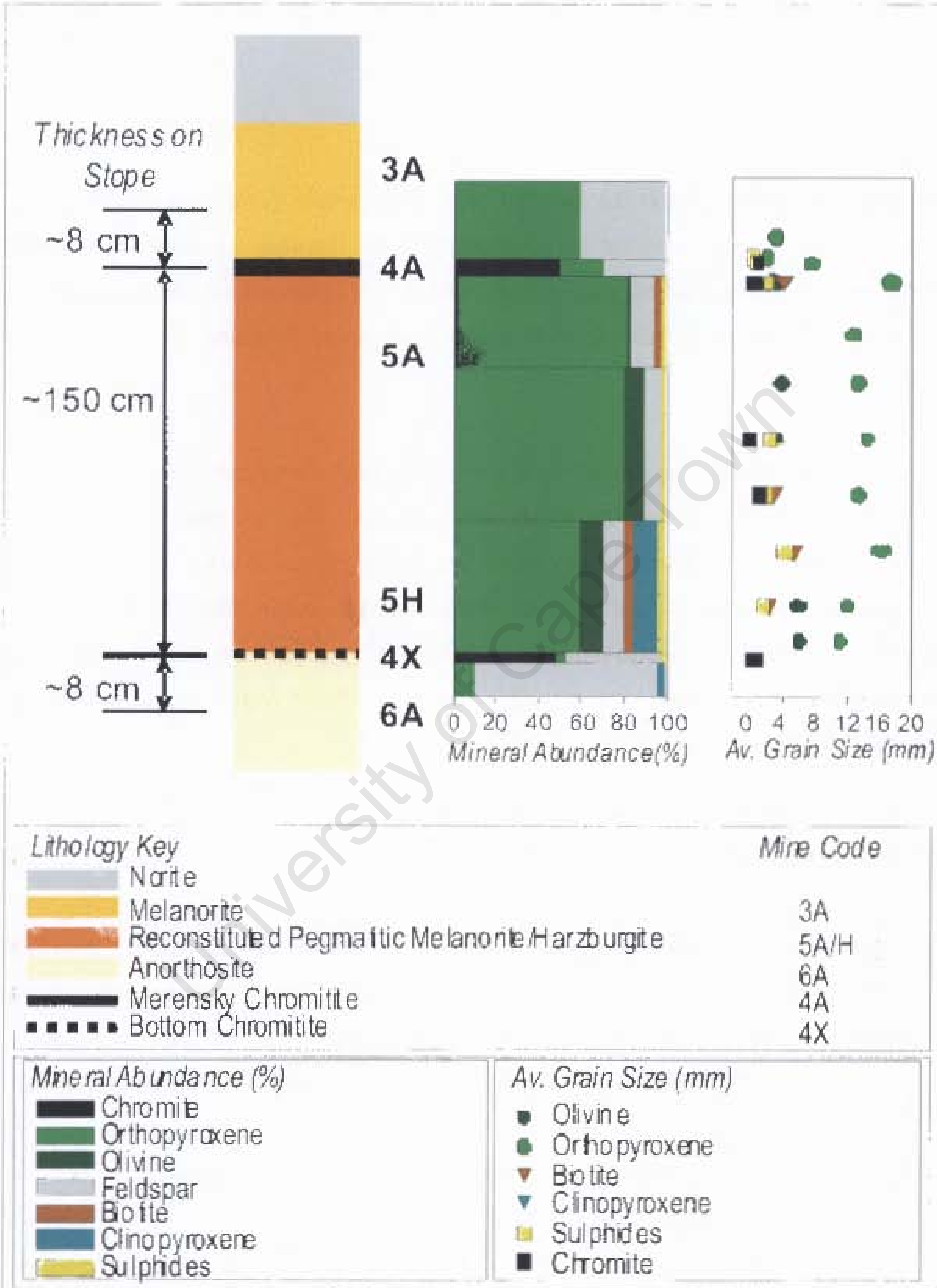


Figure 4.1: *A figure showing the stratigraphy of the Normal reef and the relative thicknesses sent in the bulk sample. Grain size and mineral abundance data is adapted from unpublished work by Roberts et al., (2004).*

4.1.1.2 Merensky Chromitite (4A)

The Merensky chromitite in the Normal reef bulk sample contained chromite grains that were small and euhedral or medium-grained discrete to coarsely annealed. Interstitial minerals included plagioclase, with minor amounts of orthopyroxene and biotite. Low temperature alteration is confined to small domains of sausseritisation within plagioclase.

BMS were mineralogically similar to the hanging wall melanorite (3A) with sulphide composites occurring interstitial to chromite grains, often at triple point junctions. However, apart from a few coarse grains, the composite size was generally smaller. Fine-grained monomineralic inclusions were common within silicates and individual chromites. Pentlandite predominantly occured as granular aggregates with minor exsolved flame pentlandite located within pyrrhotite. PGM were also observed within the chromitite (e.g. Figure 4.2b).

4.1.1.3 Pegmatitic Horizon (5A)

Investigation of the pegmatitic bulk samples revealed that it closely resembled generalised descriptions of this layer (Viring & Cowell, 1999; Roberts *et al.*, 2007) in comprising a very coarse-grained ultramafic rock, with variable amounts of orthopyroxene, plagioclase and olivine as well as minor biotite, chromite and BMS.

Orthopyroxene grains occurred in a variety of textural developments.

1) Medium grained (2-3 mm) subhedral grains.
2) Large anhedral oikocrysts (up to 3cm), hosting a wide variety of minerals.
3) Rarely as partial rims on olivine grains.
4) Occasionally included within olivine grains

These four categories of development represent different magmatic stages. (1) and (4) represent orthomagmatic (primary) grains, which settled out of a the magma column to form a cumulate pile, the oikocrystic development of orthopyroxene in (2) and olivine around orthopyroxene in (4) indicate late magmatic reconstitution, related to the deposition of the MCU hanging wall. The partial rims are also late magmatic overgrowths, which may be related to the reconstitution.

Low temperature secondary minerals derived from orthopyroxene include discrete clinopyroxene, sub-solidus clinopyroxene exsolution, amphibole and talc. Talc did not extensively rim orthopyroxene, but was rather found within orthopyroxene grains, sometimes along internal fractures (Figure 4.2c). It was also found within plagioclase, which considering its elemental constituents (magnesium rich, where as plagioclase is magnesium poor) strongly suggests an open fluid system on a crystal grain scale. Biotite has formed in a complex multistage process, since it sometimes rimmed orthopyroxene, yet was also enclosed as medium grained decossate aggregates (Figure 4.2d).

Olivine was usually included within orthopyroxene, or as discrete medium to coarse grained anhedral grains with inclusions of orthopyroxene, biotite, clinopyroxene and BMS. It also showed moderate to pervasive alteration to serpentine and dustings of fine grained magnetite, which often followed relict fractures in the host mineral.

Variable proportions of plagioclase occur as minor fine grained anhedral inclusions within orthopyroxene and olivine or as coarse grained anhedral interstitial masses. There was minor, though where present, pervasive sausseritisation (Figure 4.2e). Extensive sausseritisation, coupled with minerals such as talc or epidote suggest open fluid systems on at least a crystal grain scale.

Chromites and BMS formed a considerable subset of the mineralogy with chromite grains occurring as either small, subhedral, scattered grains included within silicates or as large extremely anhedral, annealed and often skeletal masses. BMS were medium to large (up to 5mm), anhedral composites of pentlandite (mostly granular but occasionally flame), pyrrhotite and chalcopyrite, interstitial to silicate grains

(Figure 4.2f). Composites were also chalcopyrite or pyrrhotite dominant. BMS also occurred as fine grained monomineralic grains included within silicates, as in-fill within micro-fractures (Figure 4.2g) and as very fine-grained sulphides within secondary alteration minerals such as serpentine.

4.1.1.4 Bottom Chromitite (4X)

The bottom chromitite was finer grained than the Merensky chromitite (4A) with less annealing and more modal plagioclase. There were some BMS associated with the bottom chromitite and where found, were often composite (pyrrhotite, pentlandite and chalcopyrite) or singular monomineralic grains. Texturally, the coarser composite BMS were found interstitial to chromite of plagioclase grains, with the fine-grained BMS either included or interstitial to chromite and plagioclase.

4.1.1.5 Anorthosite (6A)

The basal lithology of the geological footwall was a mottled anorthosite containing adcumulate plagioclase with minor intercumulus orthopyroxene. This mottled anorthosite contained pervasive macro-scale (2-3cm) low temperature alteration. Plagioclase showed extensive sausseritisation and orthopyroxene was replaced, often with complete pseudomorphing, to serpentine and minor talc. This heavy alteration has certain ramifications for the expected flotation behaviour, and will be discussed further in Chapter 6.

Within the mottled anorthosite the BMS were monomineralic and dominated by chalcopyrite with occasional pyrrhotite. They were found, along with chromite grains, within the heavily sausseritized sections of the anorthosite (Figure 4.2h). The BMS grain size was fine, to very fine grained. This BMS association within alteration meant that either, the BMS had been dissolved in late stage, low temperature fluids and transported before re-precipitation, or that late stage fluids eroded and entrained sulphide fragments. In either scenario sulphides have been remobilized and possibly represent minor transfer of sulphides into, or out of this Normal reef system.

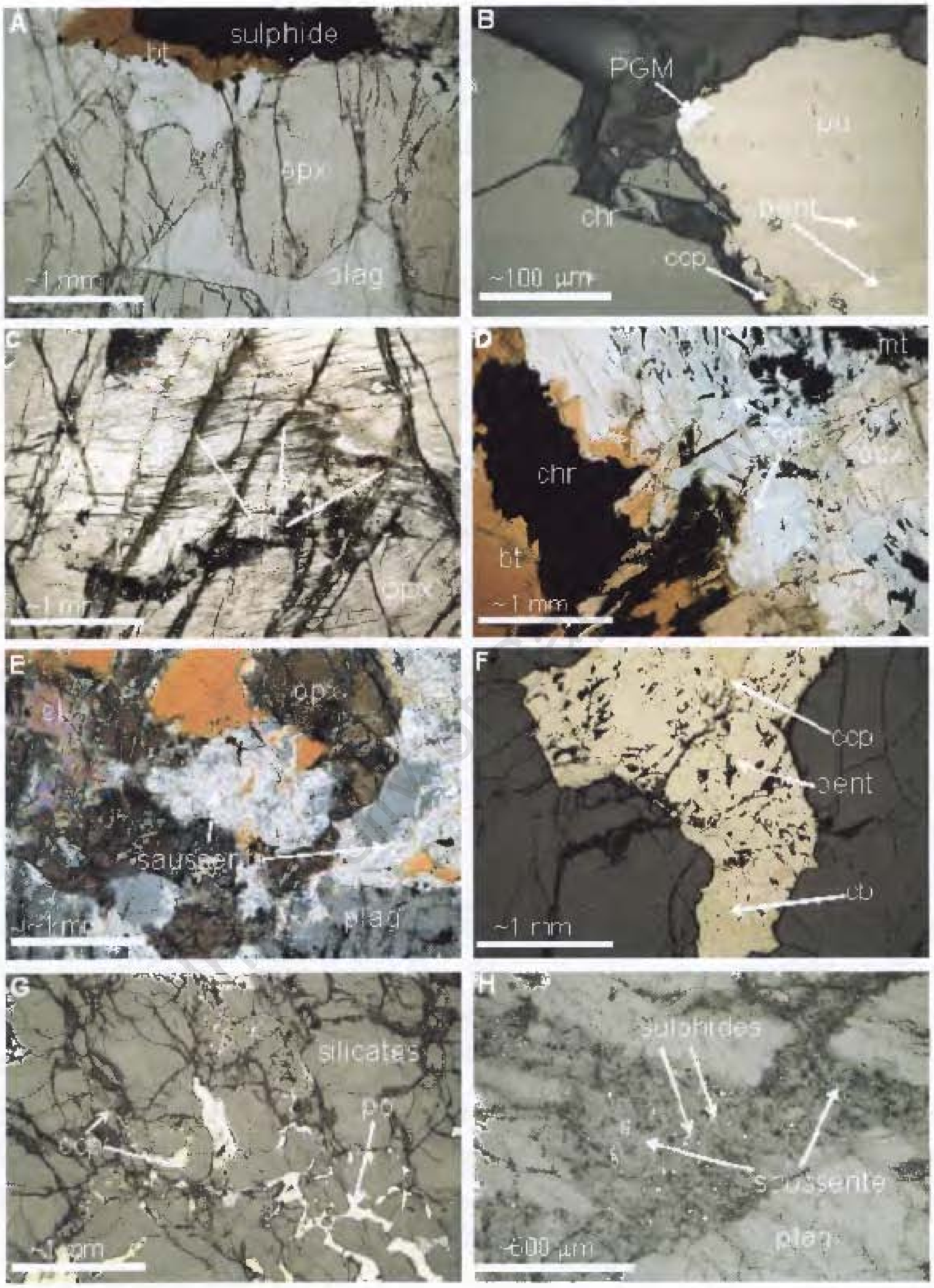


Figure 4.2: *Photomicrographs of the Normal reef. a. Hanging wall melanorite. b. Possible PGM within the Merensky chromitite. c-g (Inter–chromitite pegmatite) c. Talc lining internal fractures within orthopyroxene. d. Decossate biotite laths e. Sausserite within plagioclase f. Typical composite BMS bleb g. Fine grained sulphides. h. Fine grained sulphides within heavily sausseritized mottled anorthosite. bt - biotite, chr - chromite, opx – orthopyroxene, po – pyrrhotite, pent – pentlandite, ccp – chalcopyrite, plag – plagioclase, cb – cubanite, serp – serpentine, mt – magnetite.*

4.1.2 NP2 reef

The PGE bearing interval in the NP2 reef is much thinner than that in the Normal, so a constant stope width results in a significant proportion of barren footwall. The NP2 reef under investigation did contain the typical troctolized zone just below the 4A Merensky chromitite (Figure 4.3). From the stope cut one can be confident that the limits of PGE mineralization have been recovered into the run-of-mine.

4.1.2.1 Hanging Wall Melanorite (3A)

From the bulk samples, this lithology was very similar to the Hanging Wall Melanorite described for the Normal reef (See Figure 4.2a) with only the degree of strain-free grains different. In particular although some of the biotite grains were strain free, there were fewer plagioclase grains and no poikilocrystic grains which were strain free. This indicates increased compaction which is consistent with the increase in stratigraphic depth.

4.1.2.2 Merensky Chromitite (4A)

Compared to the same horizon in the Normal reef there was far less annealing and a smaller average grain size. Discrete chromite grains were subhedral to anhedral (Figure 4.4a, b), with occasional sections of chromite annealing. Chromite grains often contained globular inclusions of biotite, but also included plagioclase, orthopyroxene and BMS. These globular inclusions may be primary – that is, magma that was enclosed during or prior to chromite settling, or they may be secondary – that is apparent inclusions generated by annealing chromite grains that trap residual melt within the cumulate pile.

Interstitial minerals included poikilitic plagioclase and orthopyroxene as well as minor biotite (Figure 4.4a, b). Low-temperature alteration within the silicates consisted of variable sausseritization of plagioclase and minor replacement of orthopyroxene by fine grained silicates (probably talc).

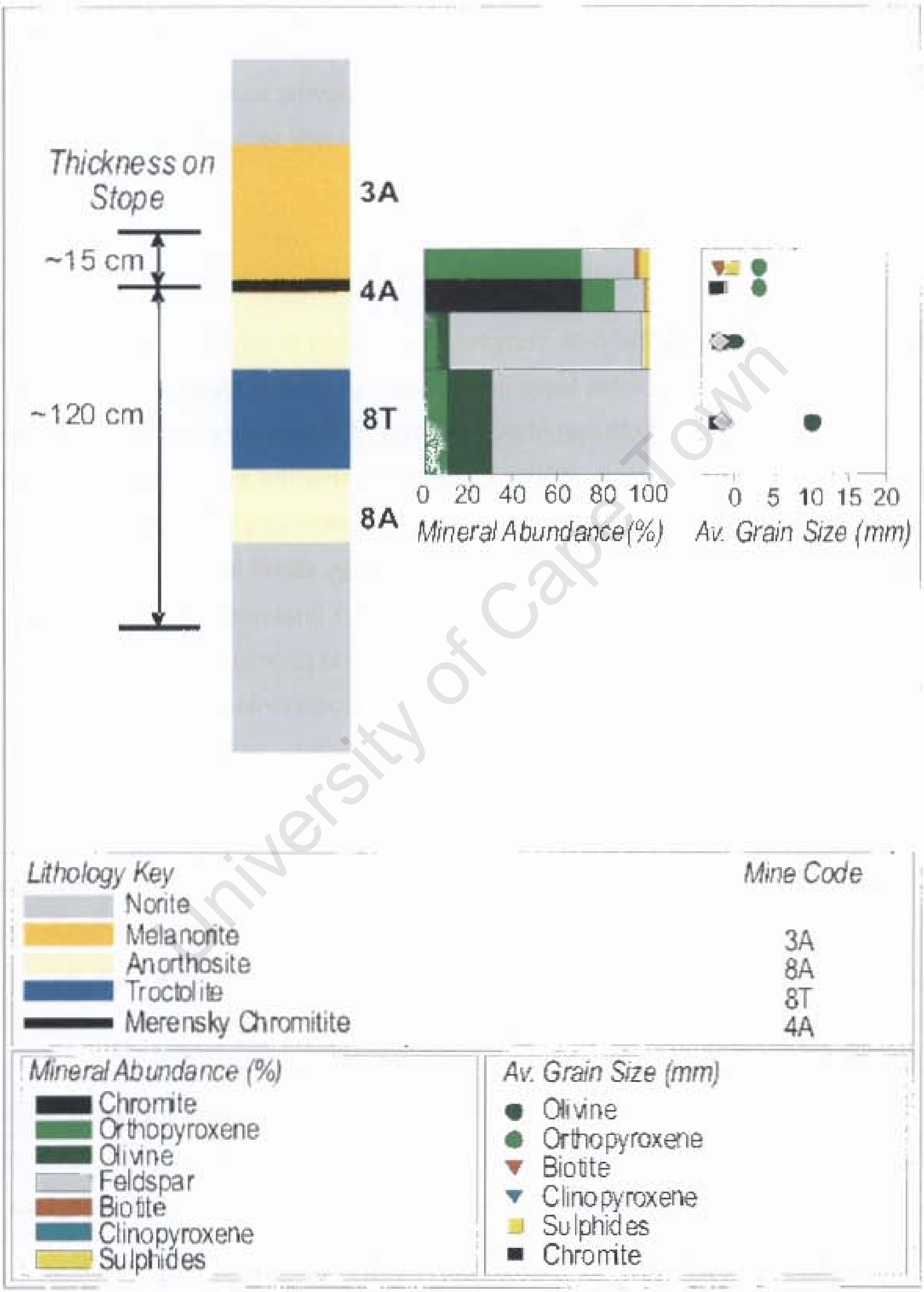


Figure 4.3: *A figure showing the stratigraphy of the NP2 reef and the relative thicknesses sent in the bulk sample. Grain size and mineral abundance data is adapted from unpublished work by Roberts et al., 2004.*

BMS were mineralogically similar to the Melanorite (3A) with medium grained composites interstitial to chromite and silicate grains (Figure 4.4b). However, as with the Normal reef, the composite size was generally smaller. BMS also occurred as fine-grained, mainly monomineralic, but rarely composite, inclusions within chromite and silicate grains. PGM were also found within this layer (e.g. Figure 4.4c).

4.1.2.3 Footwall (8A/T)

From the stope logs and the bulk samples the footwall to the NP2 reef consisted of an anorthosite layer, a troctolite layer and a norite layer. It is impossible to tell, either from the logs or from the bulk samples, the order of the layering but using the model provided by Viring and Cowell, (1999) it is thought that the anorthosite layer occurs nearest the Merensky chromitite. This agrees well with chips showing a chromitite intersecting a 'sugary' anorthosite. The next lithology down should be the troctolite followed by another thin anorthosite layer before the final norite layer. From logs of the footwall well below the Normal reef it is known that prior to reconstitution the NP2 footwall consisted of medium grained norites and leuconorites. This means that the troctolite layer represents the limit of reconstitution produced by downward percolating fluids related to the MCU mineralization.

The anorthosite layers were plagioclase adcumulates with minor to moderate interstitial orthopyroxene (Figure 4.4d), as well as minor biotite, clinopyroxene, chromite and BMS. Plagioclase showed minor sausseritization, and orthopyroxene showed negligible alteration to talc, or sub-solidus exsolution to clinopyroxene. BMS within this layer were generally fine grained, monomineralic chalcopyrite, found included and interstitial to the silicates. However, closer to the Merensky chromitite a number of medium (~1mm) composites of pyrrhotite, pentlandite (granular and minor flame), chalcopyrite and pyrite were found (figure 4.4e). Pyrite was always located within pentlandite as skeletal developments.

The troctolite layer was composed of olivine and plagioclase, with lesser amounts of orthopyroxene, biotite, sulphides and chromite. From the amount of troctolite chips within the bulk sample, this layer was relatively thin compared to the total thickness of the footwall. Olivine occurred as widely spaced medium to coarse grained anhedral poikilitic masses, often enclosing plagioclase. Olivine was frequently partially or completely rimmed by anhedral, orthopyroxene, which itself may partially enclose plagioclase grains (Figure 4.4f). Plagioclase grains were euhedral to subhedral forming an orthocumulate groundmass, often with deformation twins. All the minerals showed weak alteration with sausseritization of plagioclase and replacement of olivine by serpentine and magnetite.

Chromite grains within the troctolite were often large, anhedral and displaying dendritic textures. They were also small, extremely anhedral with skeletal habits, and included within orthopyroxene and olivine. BMS were pyrrhotite, pentlandite, chalcopyrite and pyrite (Figure 4.4g) occurring as highly anhedral composites interstitial to plagioclase and orthopyroxene as well as fine monomineralic grains included within silicates. Chalcopyrite is often associated with biotite or included within micro-fractures. As with the anorthosite layer, the pyrite is extremely anhedral and uniquely associated with pentlandite. As pyrite is a secondary mineral, its abundance within the NP2 footwall suggests a post-magmatic sulfidisation event has contributed to an overprint on the NP2 footwall.

Norite was the lowermost layer of the footwall. Judging from the bulk samples, a highly significant amount of norite was present in the footwall of the stope face and would therefore have been incorporated into the run of mine. The main minerals are orthopyroxene and plagioclase, with minor clinopyroxene, biotite, chromite and BMS. This was a plagioclase orthocumulate with interstitial poikilitic orthopyroxene (Figure 4.4h). The orthopyroxene showed variable exsolution of clinopyroxene and very minor internal alteration (probably to talc). The plagioclase showed minimal alteration to sausserite.

BMS consisted of fine grained, monomineralic chalcopyrite, located interstitial to, and included within silicates, as well as some minor pyrrhotite. There were some occasional composites of chalcopyrite, pentlandite and pyrrhotite, but no pyrite was found in this layer. Throughout the footwall (8A/T) no very fine grained sulphide textures were observed, which is in contrast to the Normal reef pegmatite.

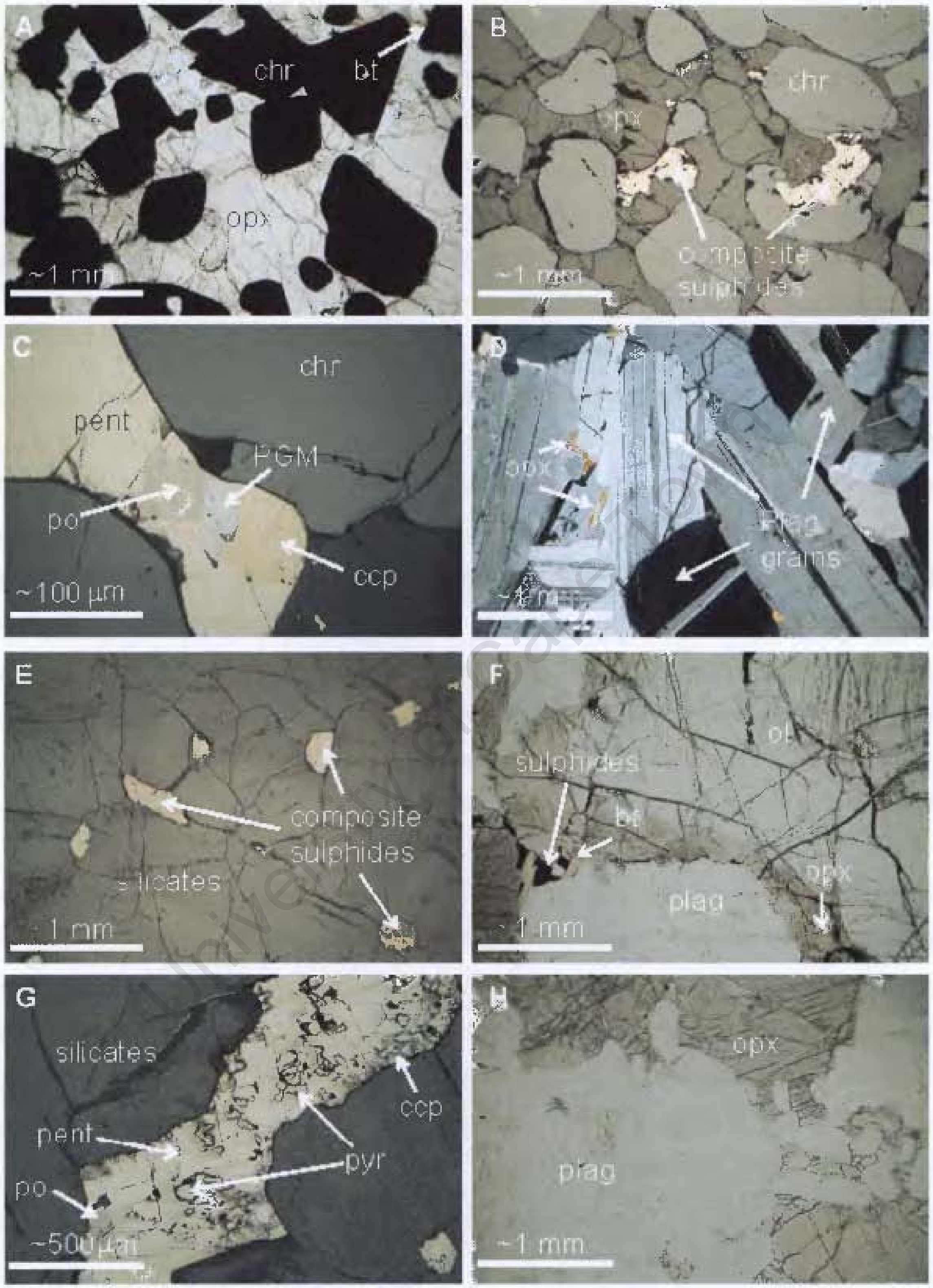


Figure 4.4: *Photomicrographs of the NP2 reef. a. Merensky chromitite. b. composite sulphides within the Merensky chromitite c. Possible PGM within the Merensky chromitite. d Plagioclase adcumulate in footwall e. Medium to fine grained, composite sulphides within plagioclase adcumulate f. Oikocrystic olivine within the troctolite g. Medium grained composite sulphides containing exsolved pyrite. h. Poikilitic orthopyroxene surrounding plagioclase. Nomenclature as for Normal reef photomicrographs.*

4.1.3 P2 Reef

The P2 reef bulk sample was characterised by a thin (~38 cm) inter-chromitite pegmatitc, bounded by the basal MCU units above, and a harzburgite layer below. The basal MCU units were ~34 cm thick, and this incorporates a significant amount of barren rock as PGE mineralization has only a limited extension into the hanging wall. Consequently one can be confident that the limits of PGE mineralization have been encompassed satisfactorily, but with some minor over-sampling of barren hanging wall sections.

4.1.3.1 Hanging Wall Melanorite (3A)

From the bulk samples, this lithology was very similar to the hanging wall melanorite described for the Normal reef, and the NP2 reef (Figure 4.6a). The main comparative difference was the further increase in strain related features. In particular there was little indication that any of the biotite or plagioclase grains were strain-free and there were some well formed deformation twins within the plagioclase. This, together with the NP2 melanorite, may suggest a gradual increase with pothole depth, of the degree of compaction.

4.1.3.2 Merensky Chromitite (4A)

Although there are several chromitite chips within the bulk sample, only a few could be positively identified as belonging to the Merensky chromitite, principally through their association with the medium grained melanorite (3A). From these samples it could be seen that this was a chromite orthocumulate, very similar in texture and development to the NP2 Merensky chromitite, with chromite grains including biotite as well as orthopyroxene, plagioclase and sulphides. In contrast to the NP2 reef there is a smaller modal proportion of orthopyroxene, however this may reflect lateral variability within the Merensky chromitite rather than a distinct difference between the two reefs.

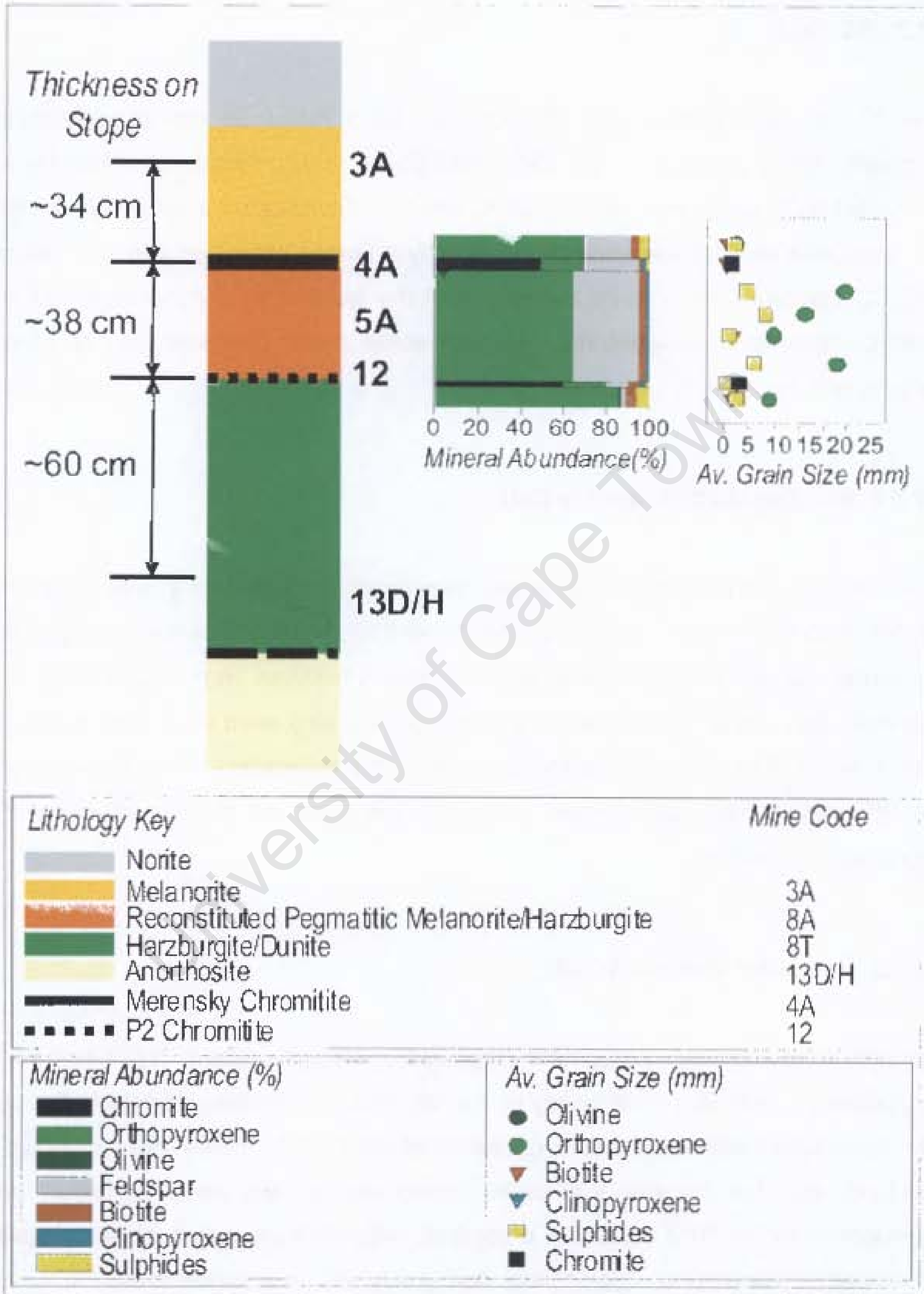


Figure 4.5: *A figure showing the stratigraphy of the P2 reef and the relative thicknesses sent in the bulk sample. Grain size and mineral abundance data is adapted from unpublished work by Roberts et al., 2004.*

BMS were mineralogically and texturally similar to the Melanorite (3A) as well as to BMS developed in the Merensky chromitites within the Normal and NP2 reefs (Figure 4.6b). As with the Normal and NP2 reefs, PGM were found in this layer (Figure 4.6c).

4.1.3.3 Pegmatitic Horizon (5A)

From the stope logs and the bulk samples it could be seen that the pegmatitic horizon was relatively thin (~38cm), much thinner than the ~157cm observed for the Normal reef. Investigation of the pegmatitic bulk samples revealed, as with the Normal, that it closely resembled generalised descriptions of this layer (Viring & Cowell, 1999; Roberts *et al.*, 2007) in comprising a very coarse-grained ultramafic rock, with variable amounts of orthopyroxene, plagioclase and olivine. From logs of the footwall well below the Normal reef it is known that prior to reconstitution the P2 footwall consisted of harzburgites, norites and a thin chromitite seam carrying 1-2 ppm PGE mineralization.

Aside from total thickness, this lithology closely resembled petrological descriptions given for the Normal reef pegmatite (see Section 4.1.1.3). However, the volumetric extent of low-temperature alteration was less pronounced, and there was no definitive talc within the plagioclase. This suggests that fluid disruption of the P2 pegmatitic horizon did not lead to an open system on the crystal grain scale. Silicate textures observed within this pegmatite indicated both primary and secondary origins for several minerals (e.g. biotite, as shown in Figures 4.6d and 4.6e)

The P2 reef had slightly different modal proportions than the Normal reef, containing more orthopyroxene, but less olivine. BMS textures were essentially the same as the Normal reef pegmatite, containing four separate textures, (medium grained composite blebs, Figure 4.6f, fine grained monominerals included within silicate minerals, Figure 4.6g, sulphide in-fill within microfractures, and very-fine grained sulphide located within low-temperature replacement minerals). This is in contrast to the NP2 footwall which contained only two sulphide textures.

4.1.3.4 P2 Chromitite (12)

The P2 chromitite was logged in the stope logs. However, it was unidentifiable within the bulk samples and it is very likely that due to its very thin nature it disintegrated during transit from Northam Platinum Ltd to the University of Cape Town.

4.1.3.5 Harzburgite (13H)

From the bulk samples this lithology consisted of orthopyroxene oikocrysts, enclosing medium grained, anhedral olivine. This represents two stages in the magmatic history, with olivine representing orthomagmatic crystal settling, and the poikilitic orthopyroxene representing late magmatic reconstitution. These oikocrystic orthopyroxene grains are evidence of the pegmatitic lithology infiltrating through the P2 chromitite as indicated above.

There was moderate to pervasive alteration of olivine to serpentine with dustings of magnetite, occasionally forming continuous veins cross-cutting, and interlinking several grains (Figure 4.6h). The presence of very minor sulphide inclusions within these veins represents grain-scale to macro-scale remobilization of sulphide within, or into this footwall lithology. However the quantities are much smaller than observed within the Normal reef anorthosite, and would likely represent an insignificant transfer of sulphides.

Sulphide proportions within this harzburgite were very low, being contained within fine grained composites of pyrrhotite, chalcopyrite and pentlandite or as monomineralic chalcopyrite included or interstitial to the major silicate phases.

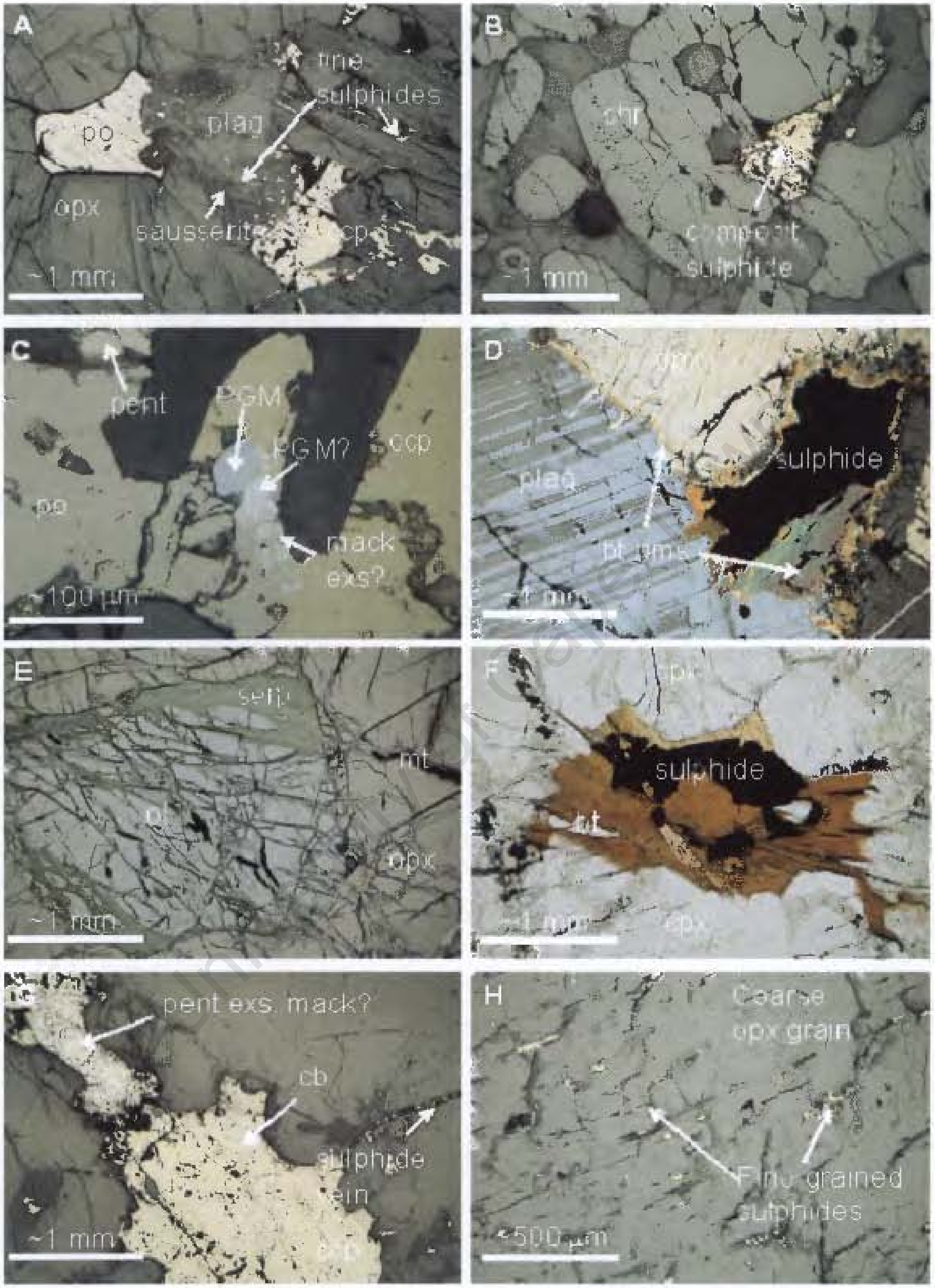


Figure 4.6: *Photomicrographs of the P2 reef. a. Medium and fine grained sulphides in the hanging wall melanorite. b. Composite sulphides within the Merensky chromitite c. Possible PGM within the Merensky chromitite. d Biotite rims on orthopyroxene e. Alteration of olivine to serpentine f. Biotite laths interstitial to coarse silicates g. Abundant fine grained silicates included within coarse orthopyroxene h. Coarse, composite sulphide, interstitial to coarse silicate grains. cpx – clinopyroxene, mack – mackinawite, exs. – exsolved.*

4.2 Mineralogical Characterisation

Quantitative mineralogical values, as calculated by QEMSCAN (Table 4.1, Figure 4.7, Figure 4.10), show distinct differences between the milled feeds of the three reef types, (see Appendix D for size by size data set).

Table 4.1: *Percentage abundance of minerals in weight% within each reef as calculated by QEMSCAN. Total alteration is the summative value of amphibole, serpentine, talc, chlorite, other silicates and magnetite.*

Mineral	Normal Reef (wt%)	NP2 Reef (wt%)	P2 Reef (wt%)
Orthopyroxene	51.2	25.4	68.6
Clinopyroxene	4.41	3.22	5.43
Plagioclase	16.7	62.6	7.16
Olivine	17.3	1.73	8.66
Mica	1.80	1.06	0.90
Quartz	0.68	0.36	0.39
Amphibole	0.45	0.26	0.55
Serpentine	0.33	0.22	0.38
Talc	0.27	0.12	0.32
Chlorite	0.57	0.08	0.13
Other Silicates	0.46	0.17	0.09
Chromite	3.34	2.72	3.87
Magnetite	0.55	0.28	0.34
Other	0.32	0.35	0.51
Pentlandite	0.55	0.48	0.88
Pyrrhotite	0.48	0.48	0.72
Chalcopyrite	0.22	0.23	0.28
Pyrite	0.09	0.11	0.15
Other Sulphides	0.16	0.13	0.26
Total Sulphides	1.71	0.82	1.14
Total Alteration Minerals	2.64	1.13	1.80

4.2.1 Silicate and Oxide Mineralogy

The NP2 reef is plagioclase rich (62.6 wt%), where as the Normal and P2 reef are relatively plagioclase poor (16.7 wt% and 7.16 wt%, respectively). In contrast the Normal and P2 reefs are orthopyroxene rich (51.2 wt% and 68.6 wt%, respectively) compared with 25.4 wt% in the NP2 reef.

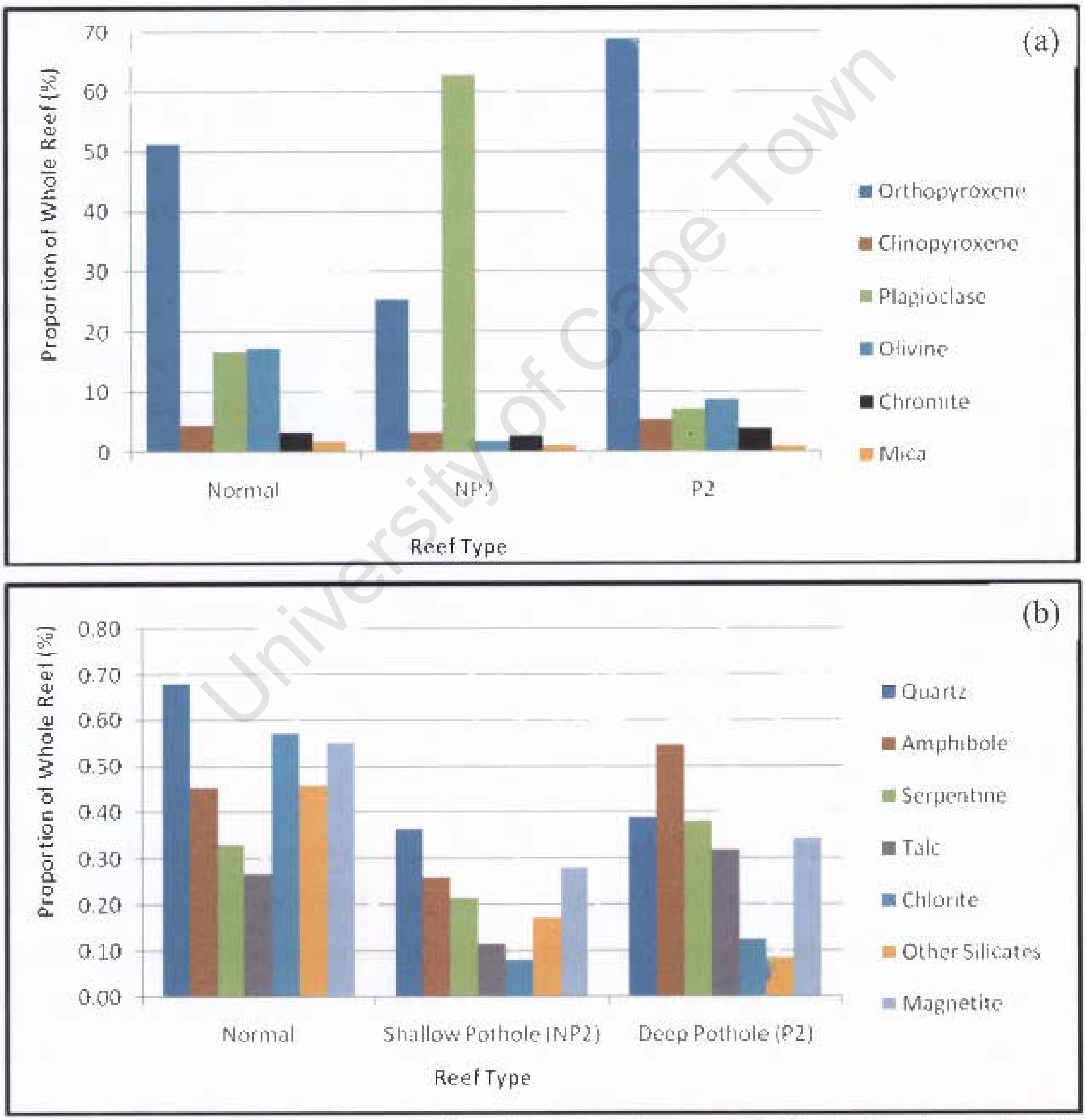


Figure 4.7: *Modal mineralogy of the three reef types. (a) Bulk silicate and oxide minerals, (b) Alteration and replacement minerals.*

The olivine content of the NP2 reef was the lowest (~1.73 wt%) of the three reefs, reflecting the thin width and wide spacing of olivine in the troctolite layer. In comparison the Normal reef (17.3 wt%) and the P2 reef (8.66 wt%) contained more olivine, due to the presence of variable harzburgitic developments within the pegmatite (Normal and P2 reef) or below the pegmatite (P2 reef).

Clinopyroxene is roughly constant between the three reefs, varying from 3.22 wt% to 5.43 wt%. Chromite is also roughly constant, though as is expected the NP2 reef contains the lowest value (2.72 wt%) since it contains just one chromitite stringer.

From Figure 4.7b and Table 4.1 it is apparent that the Normal reef contains the greatest amount of alteration minerals (2.64 wt%), followed by the P2 reef (1.80 wt%) and lastly the NP2 reef (1.13 wt%). Quartz, which may be a product of the downward migration of fluids subsequent to the emplacement of the MCU is also highest in the Normal reef (0.68 wt%, compared with 0.36 wt% for the NP2 and 0.39 wt% for the P2 reef). Furthermore, mica (mostly biotite), which may be primary or secondary is also greatest within the Normal reef (1.80 wt%), compared with 0.90 wt% in the P2 reef and 1.06% in the NP2 reef. Low-temperature minerals such as talc and chlorite may induce inadvertent gangue flotation so the higher total values present in the Normal and P2 reefs are important for explaining different flotation behaviours. The degree of alteration is also crucial to the ore hardness, and will affect milling throughput.

4.2.2 Nickel Deportment

Nickel is often used as a proxy for pentlandite ($(Fe,Ni)_9S_8$) recovery in flotation owing to its presence within the crystal lattice. However, this is misleading as nickel may also be associated with silicates (olivine and orthopyroxene) as well as solid solution traces in pyrrhotite. To improve the conversion of nickel values, obtained from feeds, concentrates and tails, to pentlandite values, allowance must be made for the presence of nickel in these three minerals.

Nickel abundance data obtained by electron microprobe analysis of olivine for each reef type was constant and high, with some variations in magnesium/iron ratios (forsterite content) (Figure 4.8). Along with the investigations into olivine, the literature values obtained for the nickel content of orthopyroxene (Cawthorn, 1999), and the nickel content of pyrrhotite (Becker, personal communication 2007) are recorded in Table 4.2.

Table 4.2: *Table showing wt% NiO within silicate gangue minerals and pyrrhotite.*

Mineral	Olivine	Orthopyroxene	Pyrrhotite
wt% NiO	0.47	0.089	0.55
Std Dev	0.03	-	0.29

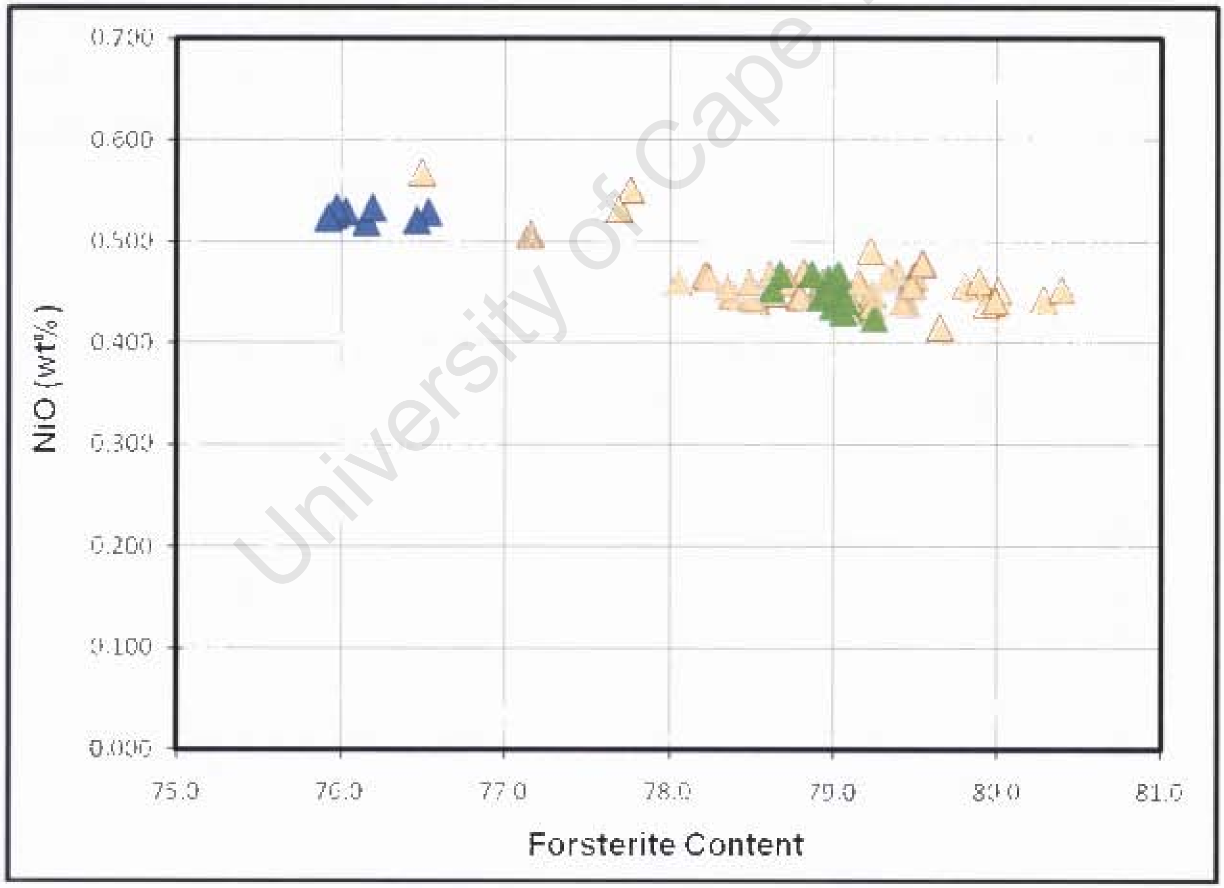


Figure 4.8: *Graph of NiO (~nickel concentration) against forsterite number. Forsterite content is a measure of the iron to magnesium ratio within olivine. A value of '0' refers to the 100% iron end-member (Fayalite) and a value of '100' refers to the 100% magnesium end-member (Forsterite). Normal reef = orange, NP2 reef = blue, P2 reef = green.*

The relatively constant nickel values across olivine grains, meant that converting nickel values to pentlandite values for the laboratory scale flotation experiments was a case of subtracting nickel locked within olivine, orthopyroxene and pyrrhotite before converting this new nickel value to pentlandite, through a single multiplication factor. The iron to nickel ratio within pentlandite is widely acknowledged to be roughly 1:1.

Based on these partitions, and taking the values recorded in Table 4.1, QEMSCAN investigations were able to discern the complete deportment of nickel between different minerals within the different reefs (Figure 4.9).

From the combined nickel deportment it can be seen that the amount of nickel contained within pentlandite varies between each reef type; 62% for the Normal reef, 85% for the NP2 reef and 77% for the P2 reef. These values show how potentially misleading grade-recovery data could be when derived from nickel values alone, since relative differences would change on conversion of nickel values to pentlandite values.

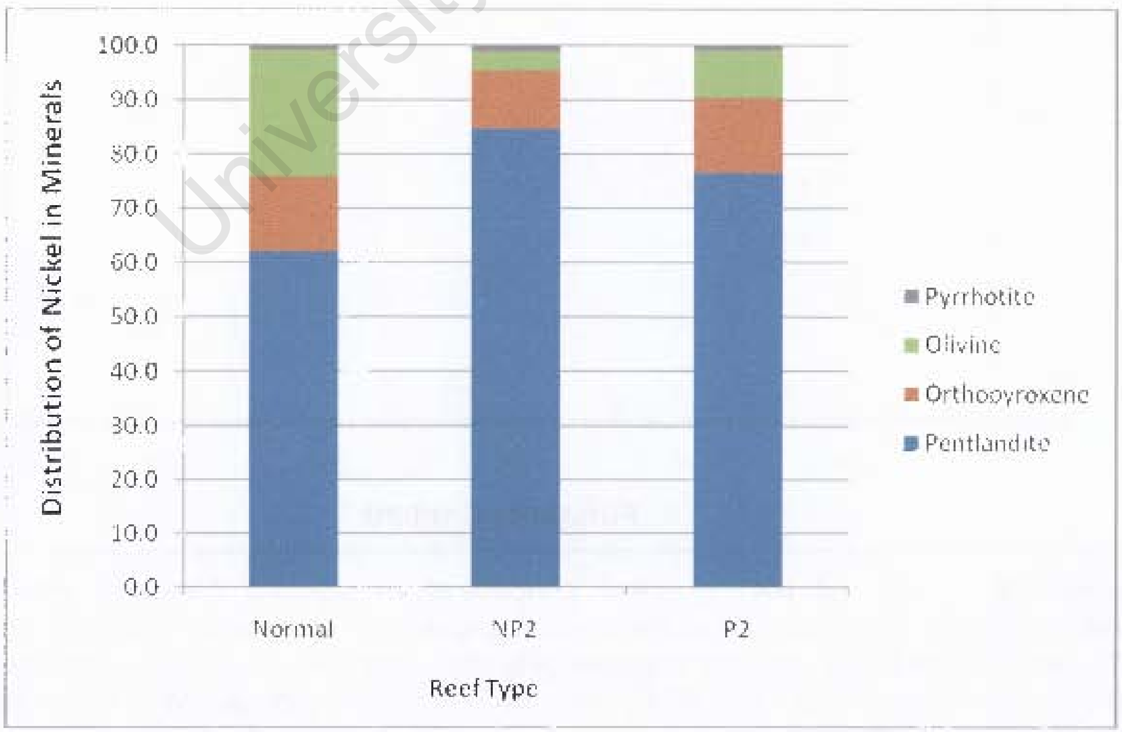


Figure 4.9: *Bar chart showing nickel deportment within the minerals orthopyroxene, olivine, pentlandite and pyrrhotite for each of the three reef types.*

4.2.3 Sulphide Mineralogy

Figure 4.10 and Table 4.1 show that the BMS inventory was roughly equivalent for the Normal and NP2 reefs (1.50 wt% and 1.43 wt% respectively), but significantly higher within the P2 reef (2.29 wt%).

For all three reef types pentlandite was more abundant than pyrrhotite, which was more abundant than chalcopyrite. Pentlandite formed ~35% of the total sulphide budget, with pyrrhoite forming ~30%, chalcopyrite forming 15-20% and other sulphides (e.g. pyrite) formed between 15-20%. These ratios were broadly consistent for all three reef types.

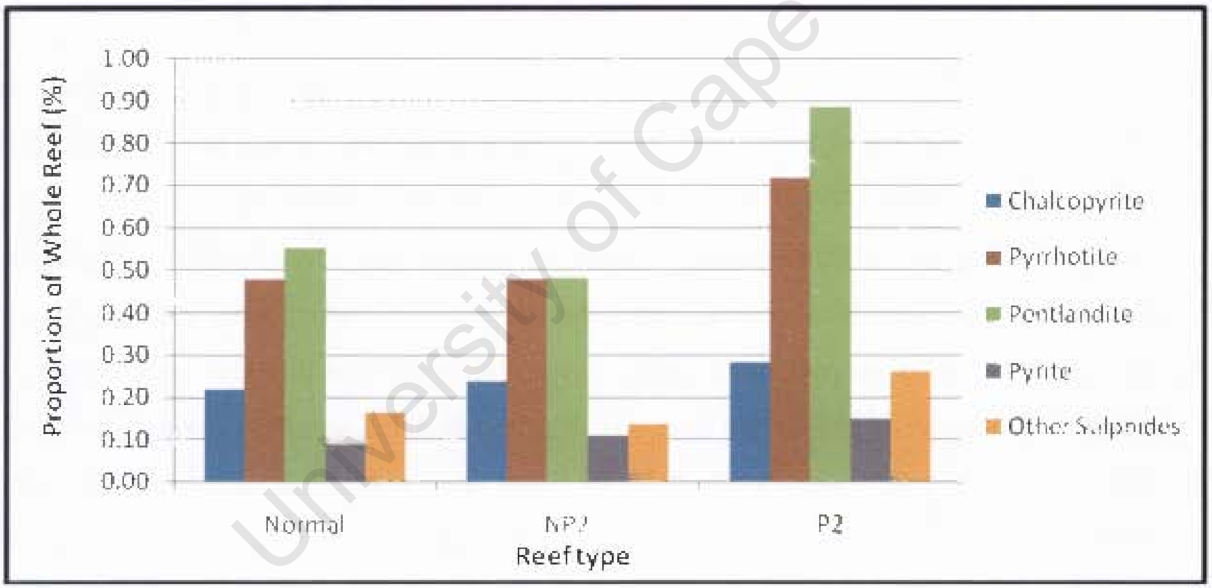


Figure 4.10: *Sulphide modal mineralogy of the three reef types.*

4.2.3.1 Sulphide Deportment

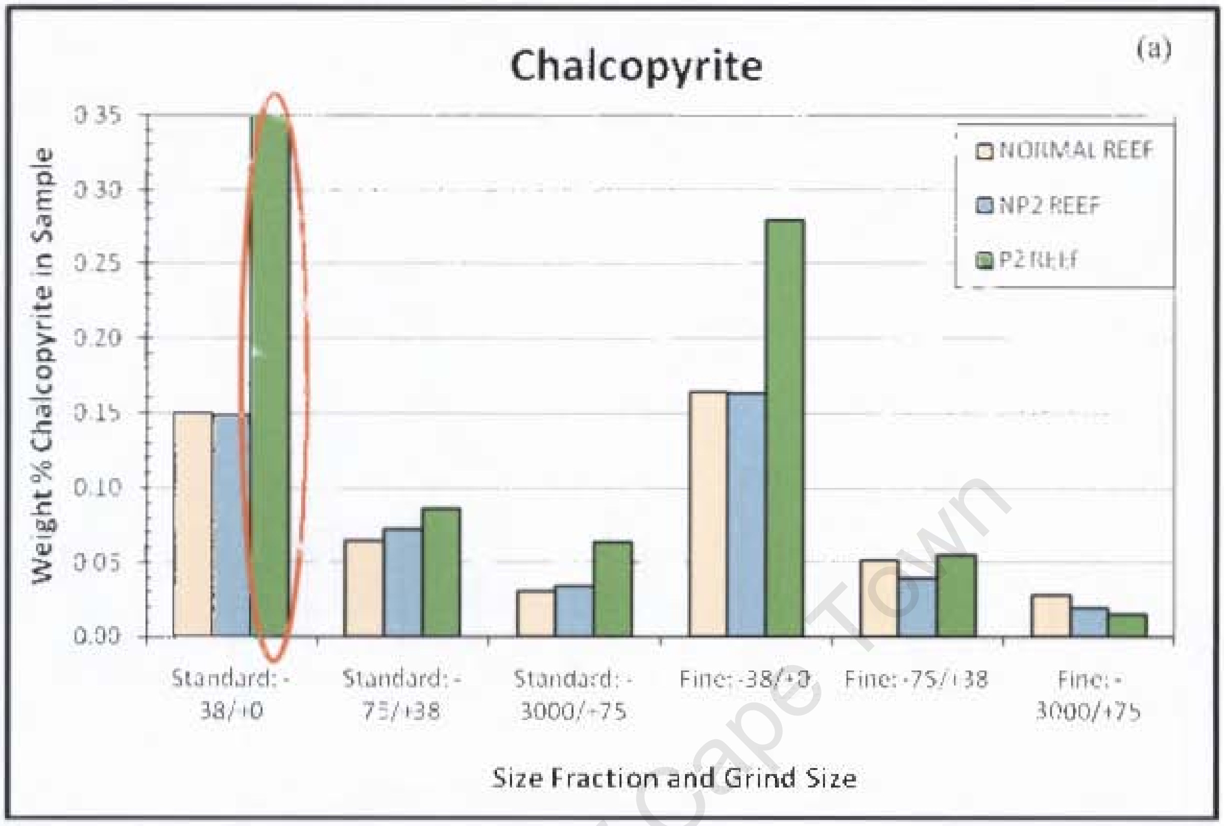
Analysis of the recalculated milled feeds with 95% confidence limits (Table 4.3), shows that two of the observed differences calculated from chemical data are clearly significant. The first is the difference in pentlandite content between the P2 reef (0.72 wt%) and the Normal and NP2 reefs (0.48 wt% and 0.45 wt% respectively). the second is the difference in total sulphide content between the P2 reef and the Normal reef. For pyrrhotite the differences are just within error and the QEMSCAN data (Figure 4.10) strongly suggest that there is a significant difference between the

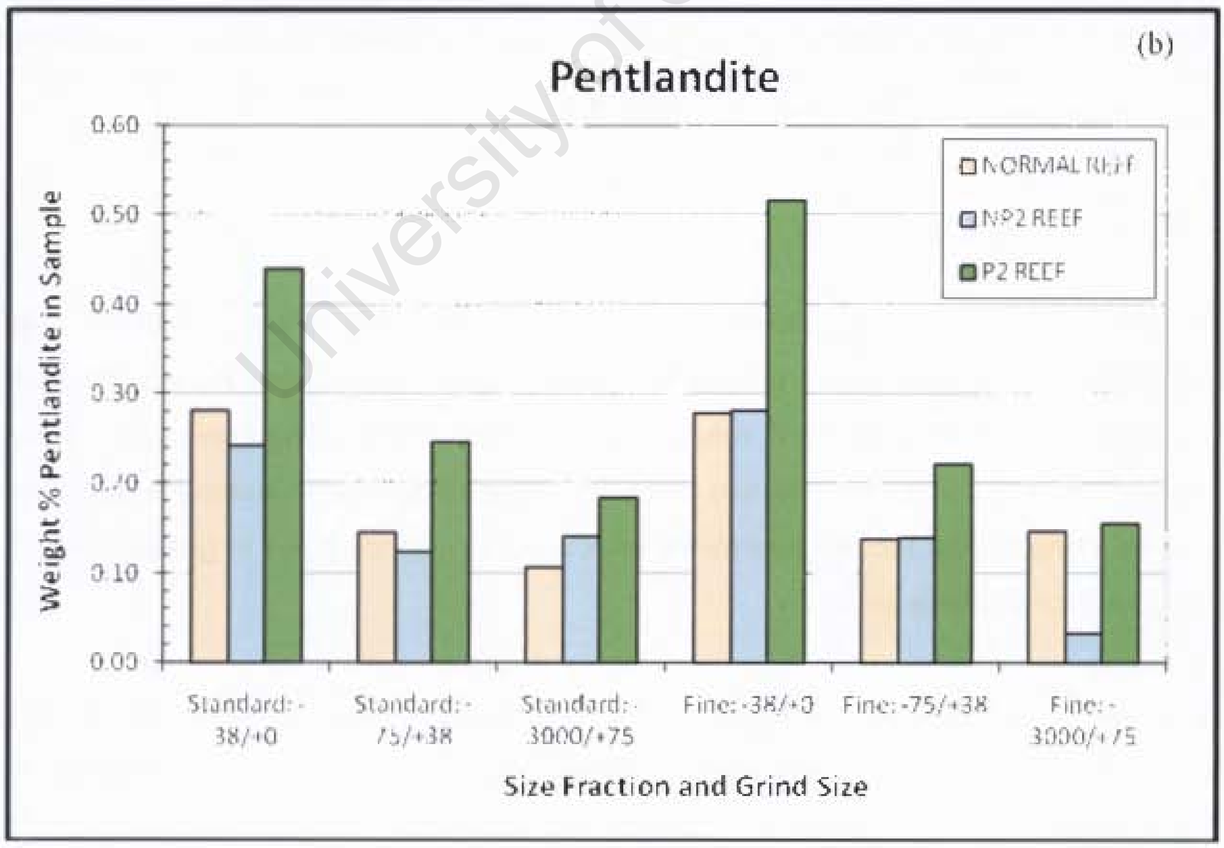
P2 reef and the other two reef types. Since the chemical data for pyrrhotite is based on a mass balance estimate, the QEMSCAN data is considered more reliable. For chalcopyrite the differences are well within error, and there is therefore no statistically significant difference between the three reef types.

Table 4.3: *Table of mean sulphide mineral values and their 95% confidence limits, as calculated from the chemical assays of the batch flotation tests of the three different reef types*

Reef Type	Chalcopyrite		Pentlandite		Pyrrhotite		BMS	
	Mean	95% Limit	Mean	95% Limit	Mean	95% Limit	Mean	95% Limit
Normal	0.21%	0.02%	0.48%	0.06%	0.33%	0.02%	1.02%	0.08%
NP2	0.22%	0.06%	0.45%	0.05%	0.35%	0.05%	1.06%	0.14%
P2	0.26%	0.04%	0.72%	0.07%	0.43%	0.07%	1.39%	0.24%

By analysing the deportment using QEMSCAN data it was also possible to determine how sulphide deportment varies by size. Figure 4.11 a,b and c show that for chalcopyrite, pentlandite and pyrrhotite the main deportment is in the fine size fraction (-38 μm), with progressively smaller concentrations towards the coarse size fraction (+75 μm). This relationship holds for both the standard and the fine grind. An exception is found for pyrrhotite under a standard grind. In this scenario the main pyrrhotite deportment is still in the fine size fraction but the smallest concentration is found in the intermediate size fraction for all three reef types. It is also observed that for all three reef types the +75 mm size fraction drops down for the fine grind for all the sulphides, which is consistent with the increase in grinding.


Chalcopyrite
(a)
Weight % Chalcopyrite in Sample
0.35
0.30
0.25
0.20
0.15
0.10
0.05
0.00
NORMAL REEF
NP2 REEF
P2 REEF
Standard: -38/+0
Standard: -75/+38
Standard: -3000/+75
Fine: -38/+0
Fine: -75/+38
Fine: -3000/+75
Size Fraction and Grind Size


Pentlandite
(b)
Weight % Pentlandite in Sample
0.60
0.50
0.40
0.30
0.20
0.10
0.00
NORMAL REEF
NP2 REEF
P2 REEF
Standard: -38/+0
Standard: -75/+38
Standard: -3000/+75
Fine: -38/+0
Fine: -75/+38
Fine: -3000/+75
Size Fraction and Grind Size

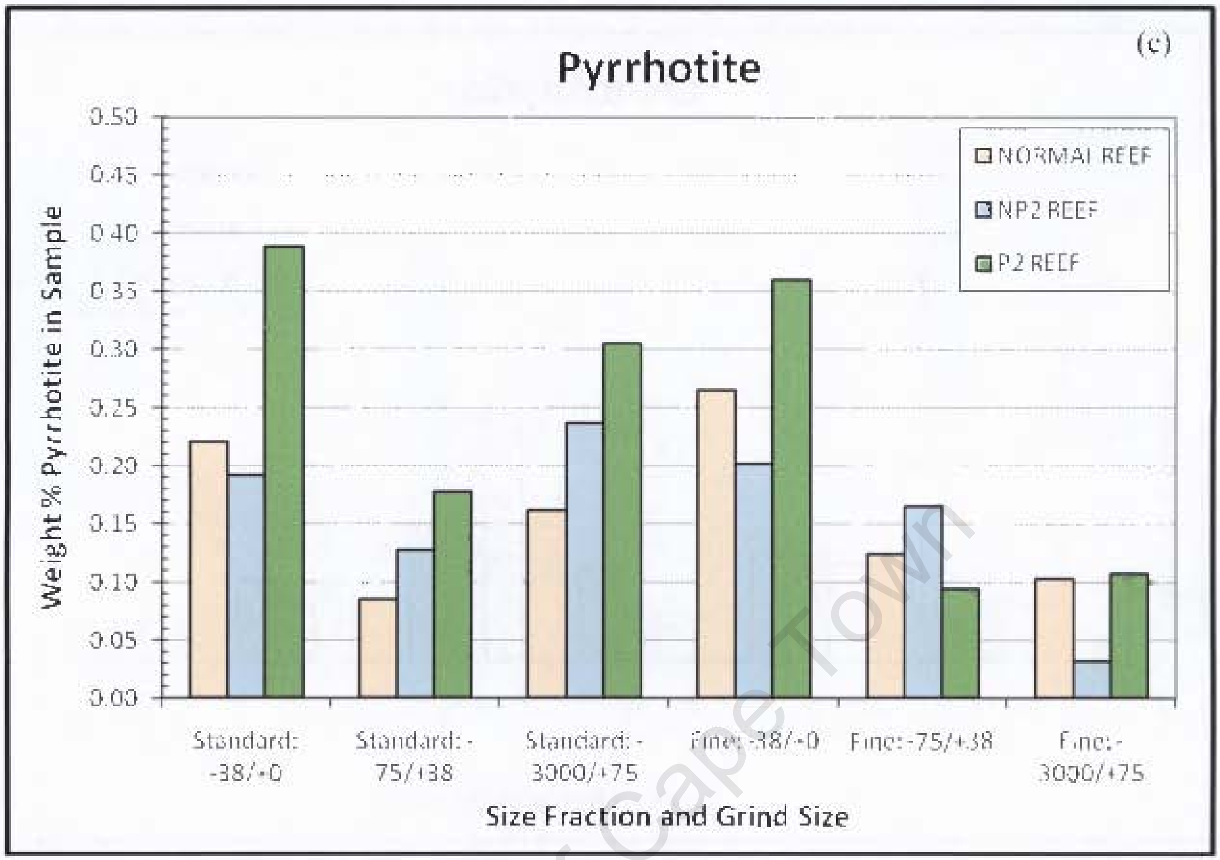
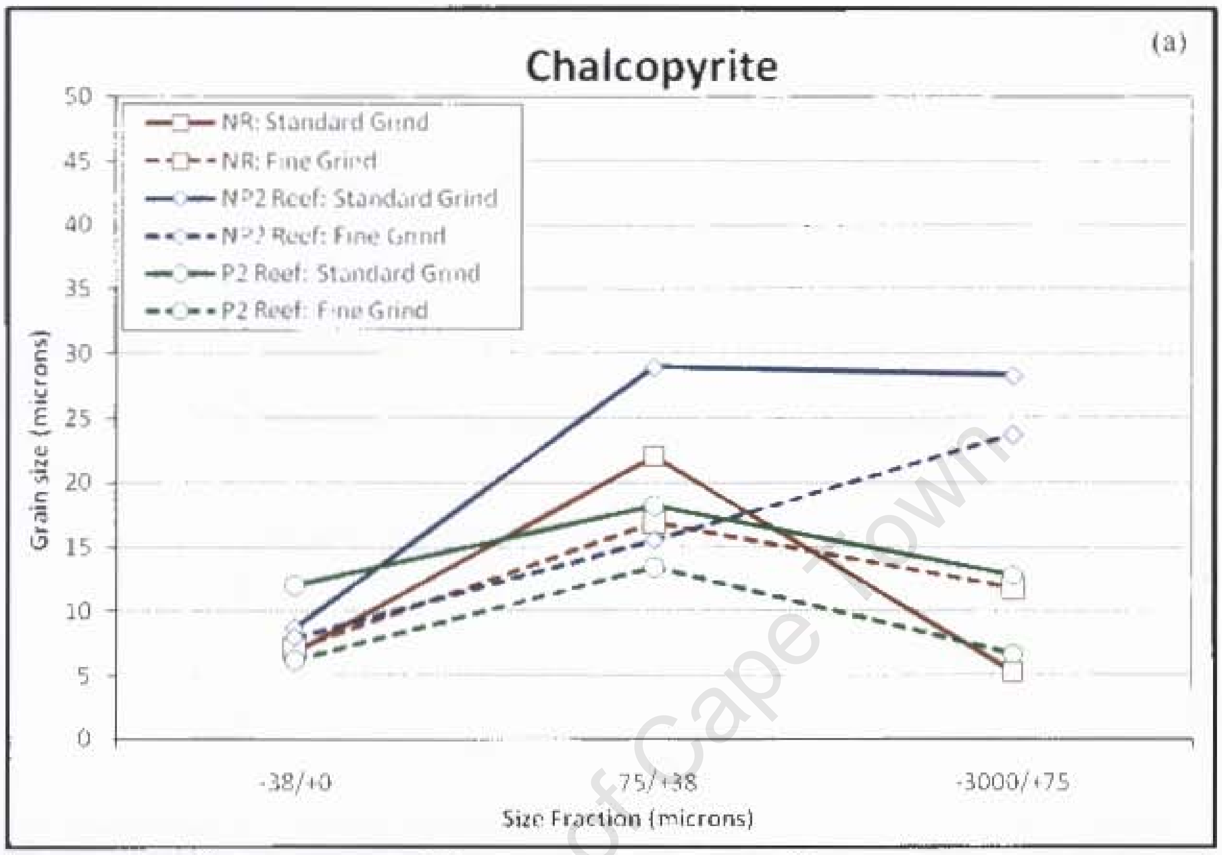


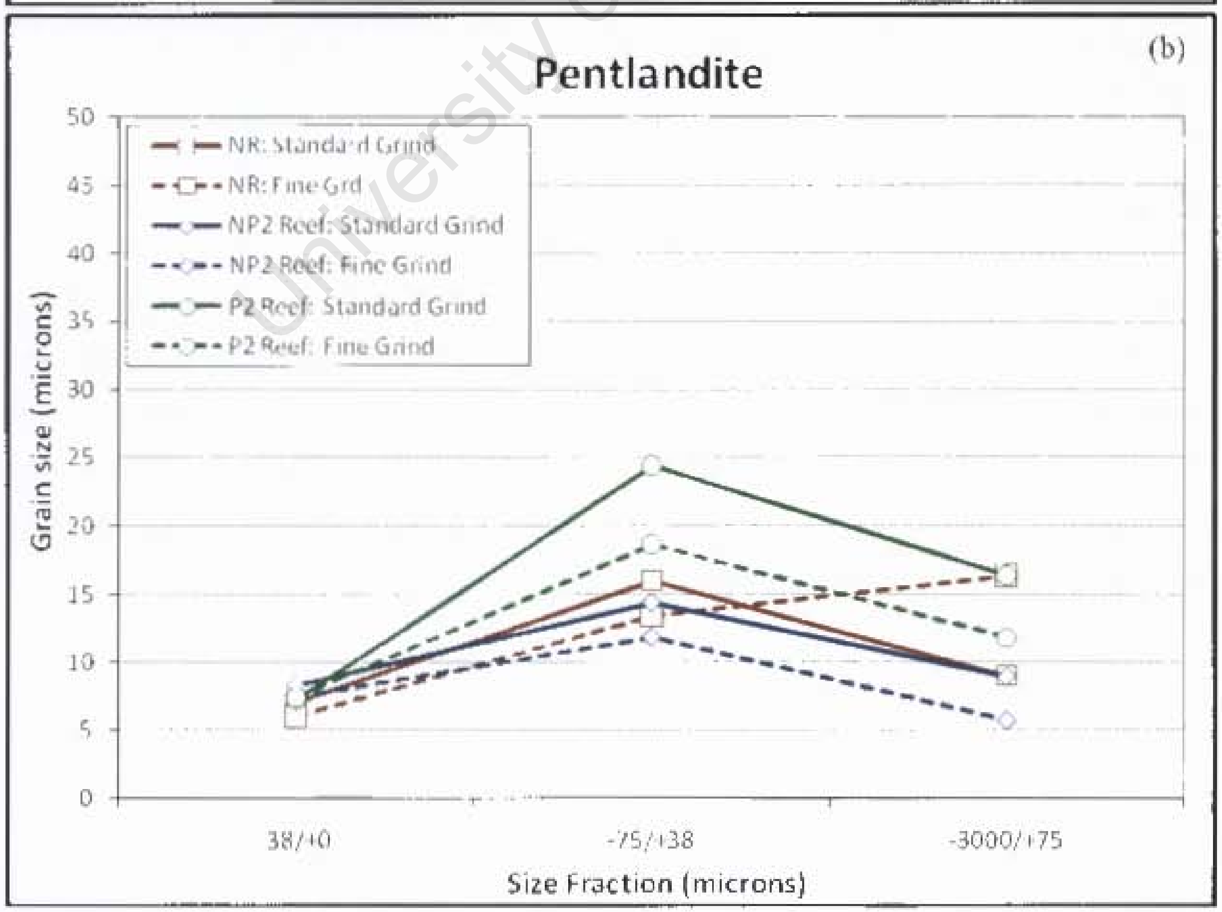
Figure 4.11. *Sulphide deportment by size fraction. (a) chalcopyrite, (b) pentlandite and (c) pyrrhotite for the three reef types. The value for chalcopyrite circled in red in (a) is anomalous as pointed out in section 3.5.1.*

4.2.3.2 Sulphide Grain Size

Sulphide grain size was estimated from QEMSCAN data from the sized feed fractions. Since all the BMS grains were smaller than mean particle size for each size fraction it is assumed that values are reflective of pre-milling grain size. These values are only considered relative to one another as absolute values are uncertain due to stereological factors and minor effects due to grinding, as in general values are lower for the finer grind.

It is apparent that the NP2 reef contains coarser grains for chalcopyrite and pyrrhotite, but that the P2 reef contains coarser grains for pentlandite. The Normal has intermediate to low grain sizes for all three sulphides. Figure 4.12. also shows that pyrrhotite is the coarsest of the three sulphides, whilst chalcopyrite and pentlandite have very similar grain sizes.


(a)
Chalcopyrite
NR: Standard Grind
NR: Fine Grind
NP2 Reef: Standard Grind
NP2 Reef: Fine Grind
P2 Reef: Standard Grind
P2 Reef: Fine Grind
Grain size (microns)
50
45
40
35
30
25
20
15
10
5
0
-38/+0
-75/+38
-3000/+75
Size Fraction (microns)


(b)
Pentlandite
NR: Standard Grind
NR: Fine Grd
NP2 Reef: Standard Grind
NP2 Reef: Fine Grind
P2 Reef: Standard Grind
P2 Reef: Fine Grind
Grain size (microns)
50
45
40
35
30
25
20
15
10
5
0
38/+0
-75/+38
-3000/+75
Size Fraction (microns)

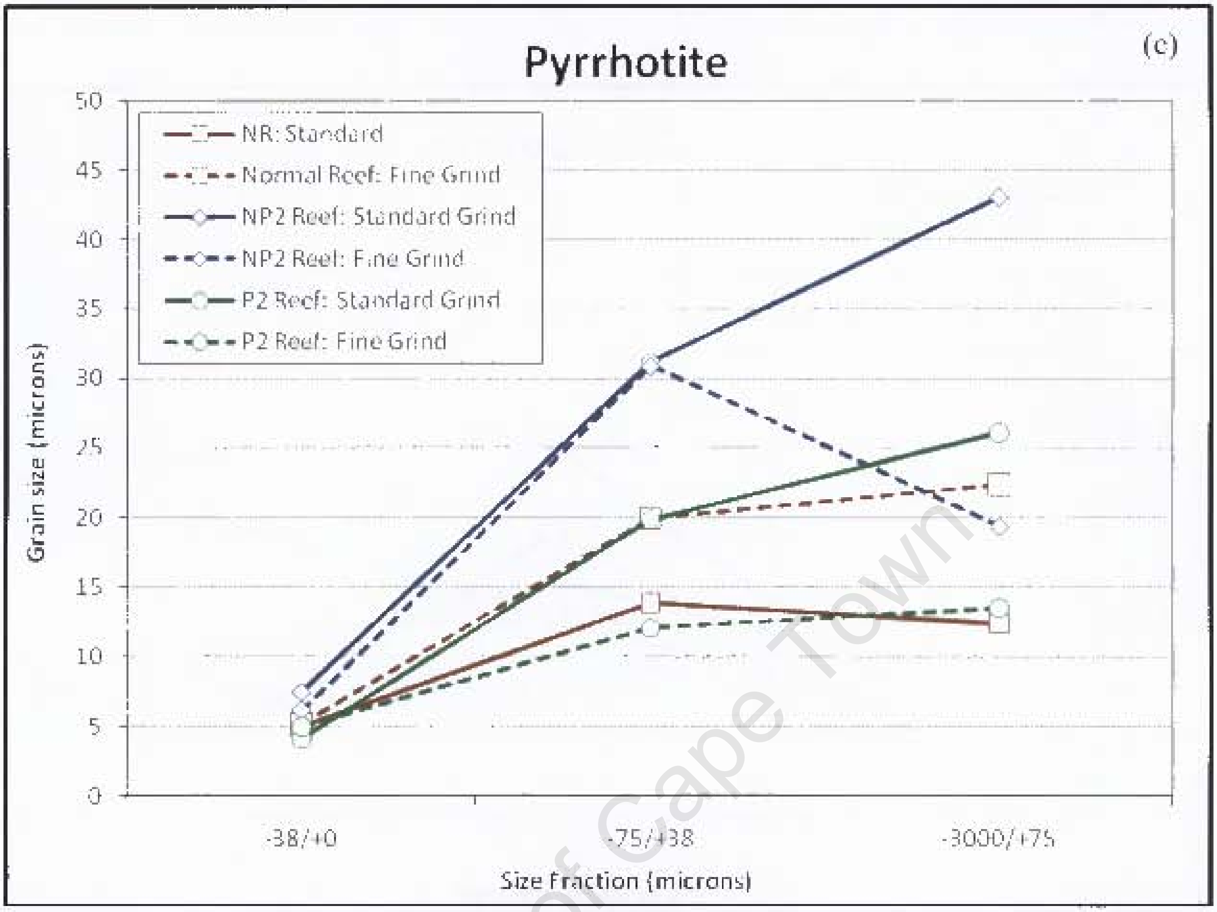
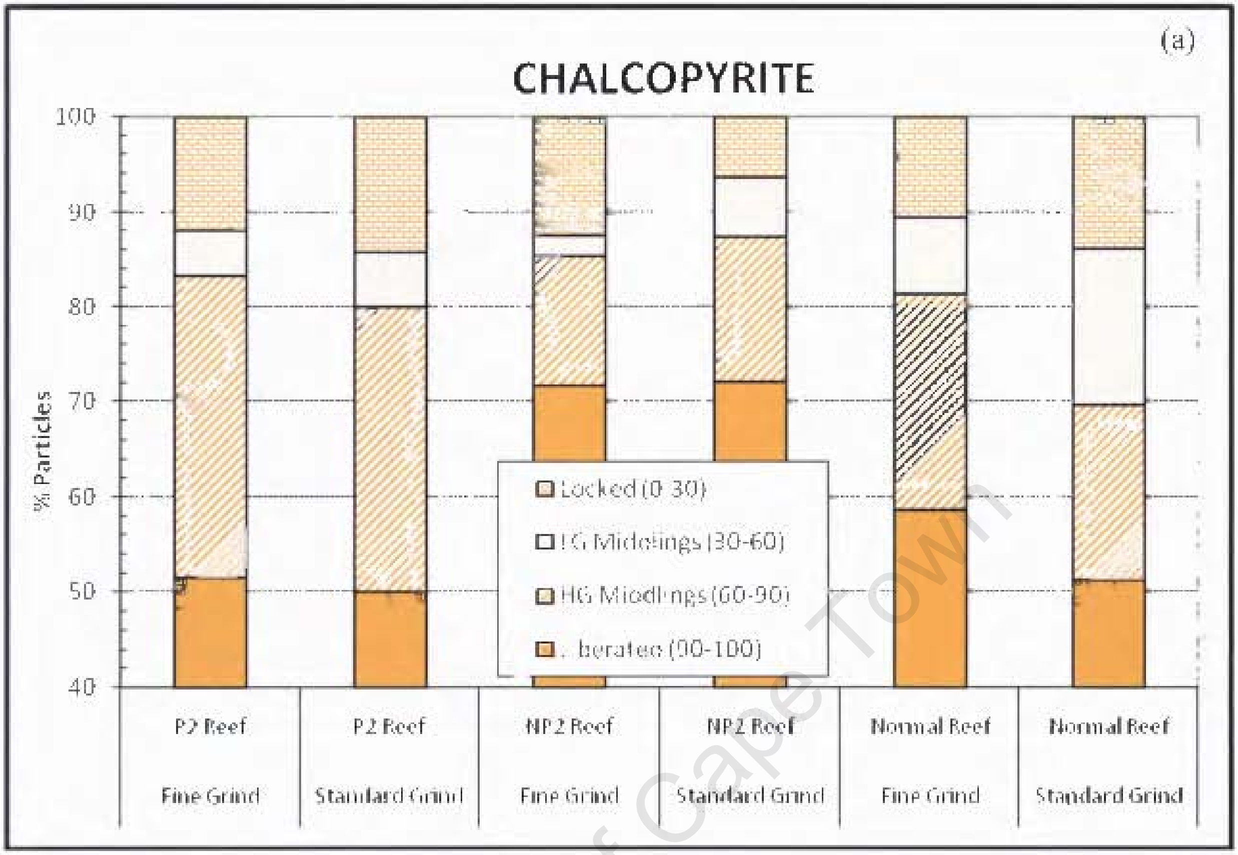
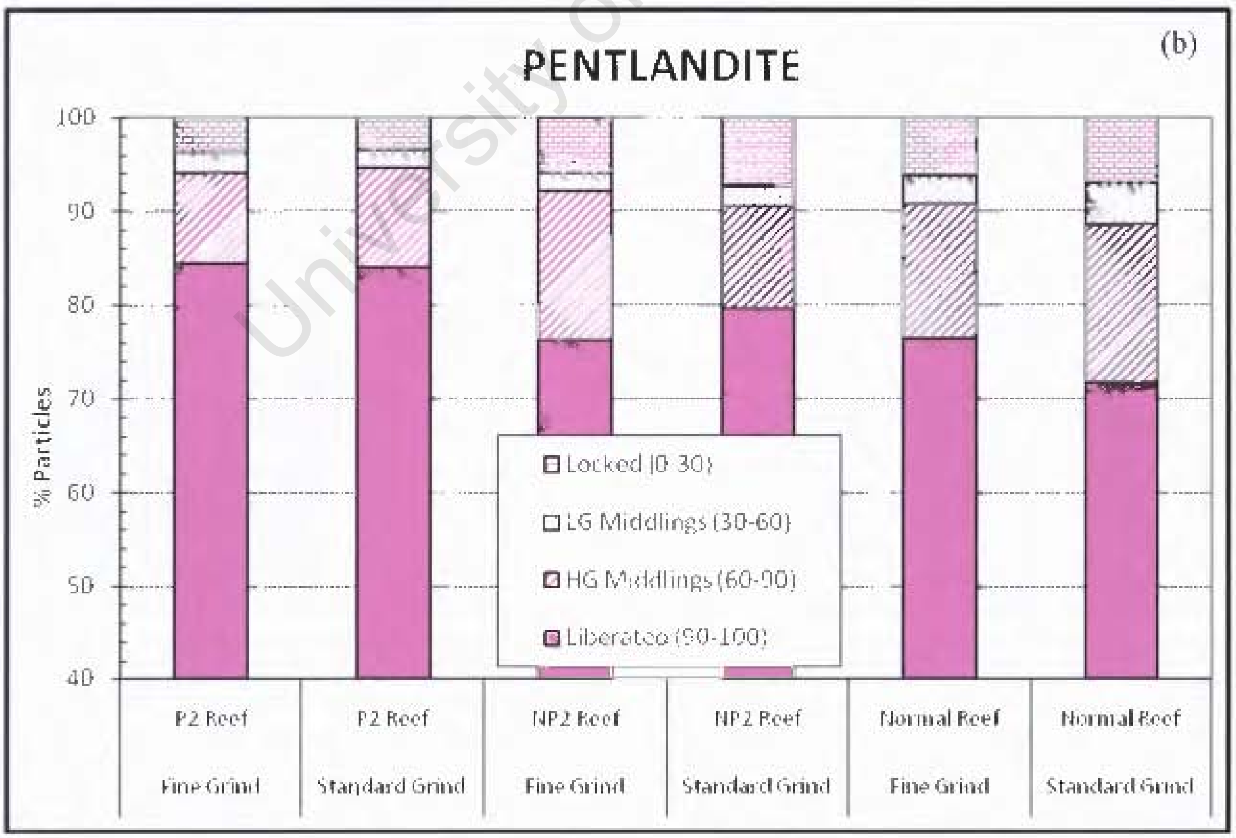


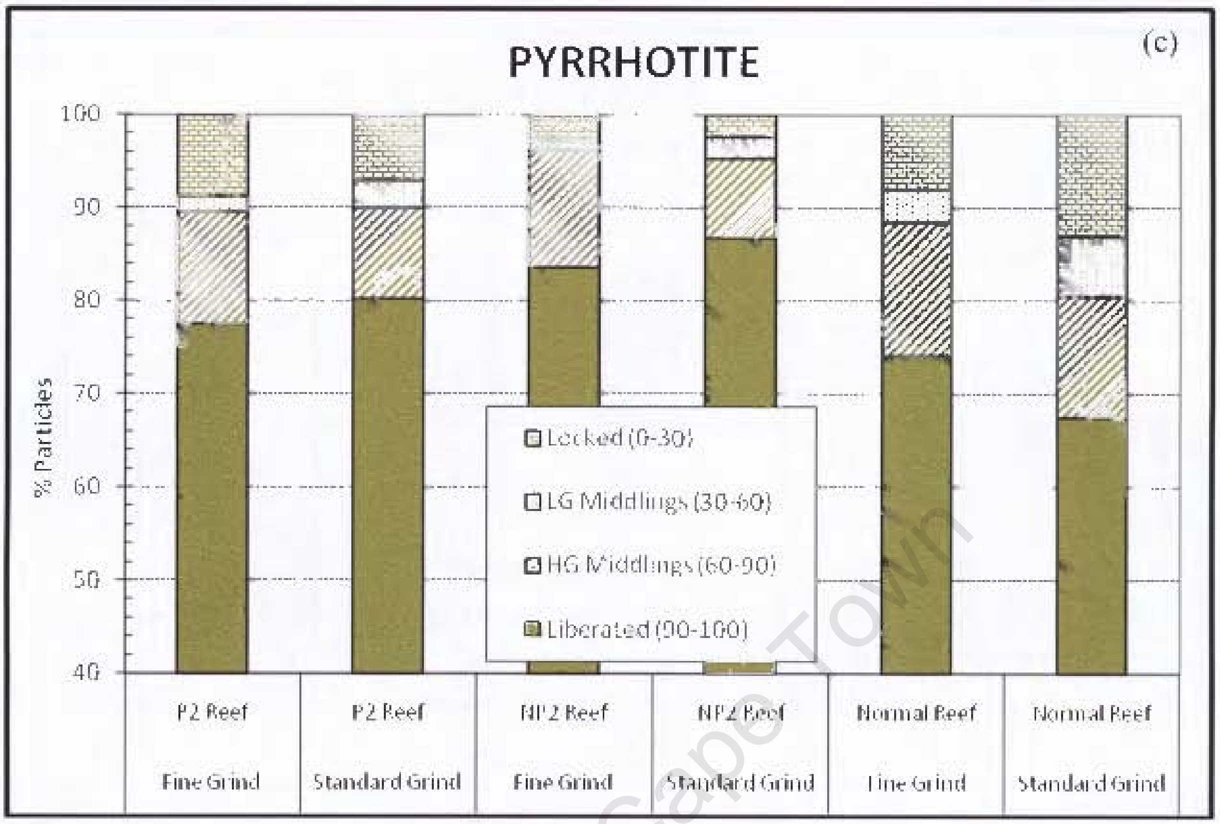
Figure 4.12: *Grain size values for the three main sulphides. (a) chalcopyrite, (b) pentlandite and (c) pentlandite within the three different size fractions for the three reef types.*

4.2.3.3 Sulphide Liberation

There are two aspects to sulphide liberation, firstly, there is liberation of the individual sulphides from one another (e.g. pyrrhotite from chalcopyrite), and secondly, there is liberation of sulphide composites from gangue minerals. In this investigation a fully liberated particle is defined as more than 90% of its surface area exposed (see Section 2.3.2). Figure 4.13 shows the degree of liberation for the three main sulphides, as well as composite sulphides.


(a)
CHALCOPYRITE
% Particles
100
90
80
70
60
50
40
Locked (0-30)
LG Middlings (30-60)
HG Middlings (60-90)
Liberated (90-100)
P2 Reef
Fine Grind
P2 Reef
Standard Grind
NP2 Reef
Fine Grind
NP2 Reef
Standard Grind
Normal Reef
Fine Grind
Normal Reef
Standard Grind


(b)
PENTLANDITE
% Particles
100
90
80
70
60
50
40
Locked (0-30)
LG Middlings (30-60)
HG Middlings (60-90)
Liberated (90-100)
P2 Reef
Fine Grind
P2 Reef
Standard Grind
NP2 Reef
Fine Grind
NP2 Reef
Standard Grind
Normal Reef
Fine Grind
Normal Reef
Standard Grind



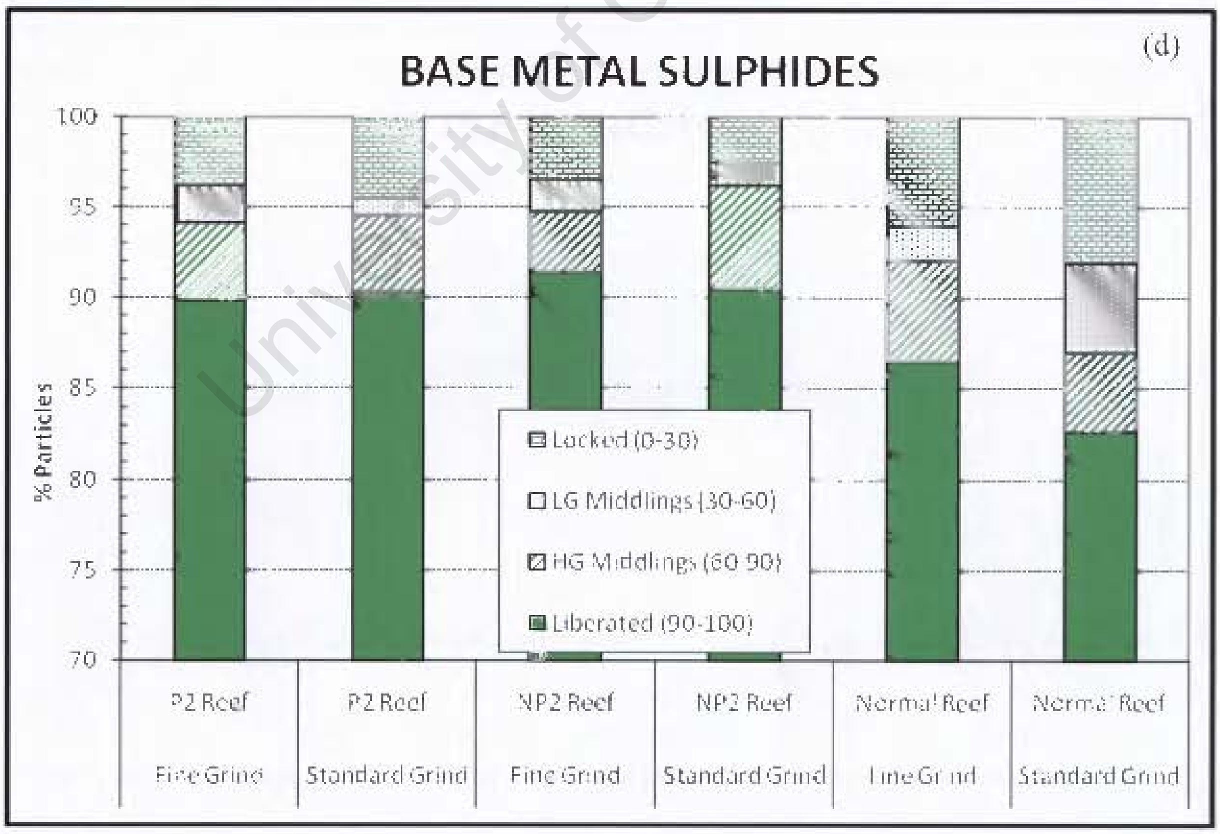


Figure 4.13: *Liberation of the three main sulphides and composite sulphides. (a) chalcopyrite, (b) pentlandite, (c) pyrrhotite, (d) base metal sulphides for the three reef types.*

For composite sulphides, which are very likely to float, it is evident that BMS are over 80% liberated for both grind conditions (Figure 4.13). For individual sulphides the degree of liberation varies from 50 to 87%, with pentlandite generally being the most liberated (72-84%) and chalcopyrite the least liberated (50-72%) (Figure 4.13). The NP2 reef shows optimum liberation for BMS (91%), chalcopyrite (72%), and pyrrhotite (85%), with intermediate liberation for pentlandite (76-79%). This favourable liberation correlates well with the observed higher grain sizes. The P2 reef shows the greatest pentlandite liberation (84%), but with the worst chalcopyrite liberation (52%). The Normal reef shows the worst pyrrhotite liberation (~70%). The Normal reef is also the only reef to be significantly affected (i.e. >5% change in liberation) by a change in grind size (Figure 4.13).

For composite sulphides the degree of locking is lowest for the NP2 reef (3-4%), closely followed by the P2 reef (4-5%) with the Normal reef highest at 6-8%. For the individual sulphides the NP2 reef has the lowest degree of locking for pyrrhotite (~4%) and chalcopyrite (6-12%), with intermediate locking for pentlandite (6-8%). The P2 reef shows the lowest degree of locking for pentlandite (~4%), but the highest degree of locking for chalcopyrite (~12-15%). The Normal reef shows the highest degree of locking for pyrrhotite (8-12%) with similar pentlandite locking to the NP2 reef and similar chalcopyrite locking to the P2 reef.

4.3 Summary

From the petrological and mineralogical investigations, some key findings can be recognized.

- All reef types contain the base of the MCU as a ‘drape’ forming the hanging wall, with the Normal and P2 reefs having very similar footwalls, reflecting their similar pre-MCU emplacement compositions. The NP2 reef has a very different footwall composition, reflecting the presence of a highly leucocratic pre-MCU emplacement footwall.

- The main sulphides within the three reefs are pyrrhotite, pentlandite and chalcopyrite, with a significant proportion of other sulphides (~15% of the total sulphide). Sulphides are developed as one dominant texture, with three subsidiary textures. The main texture is medium grained (0.5 mm – 2 cm) composites. The three subsidiary textures are fine grained monomineralic sulphides, fine grained sulphide veins and very fine-grained sulphides enclosed within late stage secondary alteration minerals. Pyrrhotite grain size is slightly coarser than chalcopyrite and pentlandite. It is observed that the P2 reef contains the highest sulphide head grades.

- The main silicate minerals are orthopyroxene, olivine and plagioclase with the Normal and P2 reefs dominated by orthopyroxene. However, the NP2 reef is distinct, principally containing plagioclase, with only moderate orthopyroxene.

- Nickel is deported between four minerals: orthopyroxene, olivine, pyrrhotite and pentlandite, with the majority of that deportment shared between olivine and pentlandite.

- Alteration of primary magmatic minerals to secondary assemblages is greatest within the Normal reef, then the P2 reef and finally the NP2 reef, which is largely unaltered.

- The pegmatitic sections of the Normal and P2 reefs signify the presence of partially open fluid systems, which have allowed for increased sulphide remobilisation and complexity relative to the NP2 reef. As such the NP2 reef contains just two sulphide textures, with a smaller abundance of sulphides deported as fine grained sulphides. Furthermore the NP2 reef does not contain the very fine grained sulphides locked within alteration minerals.

- Liberation of sulphide composites is greater than 80% for all three reef types, but is best for the NP2 reef with liberation in excess of 90%. For the Individual sulphides the NP2 reef shows the greatest liberation, except for pentlandite, which shows optimum liberation within the P2 reef. The Normal reef shows the worst liberation, and is the only reef significantly affected by grind.

5 Processing Results

5.1 Results Overview

Table 5.1 summarises the flotation performance results, showing final cumulative mineral grades and recoveries as well as total mass and total water recoveries (see Appendix D for full results). These results were subject to an analysis of variance to quantitatively assess the statistical significance of changing flotation parameters. Table 5.2 shows which results were statistically significantly different (see Appendix F for details and calculations), and which results and interactions may be considered statistically insignificantly different. In cases where the experimental error was high, differences were shown to be statistically insignificant, but were practically significant. Conversely when the error was very small it showed statistical significance but was practically insignificant.

It is apparent from Table 5.1 that the three reef types have performed differently under the same flotation conditions. For example, evaluation of the results of the standard grind with copper sulphate addition tests show that, cumulative base metal sulphide (BMS) recoveries are; 71.6% for the Normal reef, 90.1% for the NP2 reef and 81.3% for the P2 reef. Similarly, comparing chalcopyrite, pentlandite, pyrrhotite, total mass and total water recoveries at each of the flotation conditions reveals differences between the three reef types. It is particularly noticeable from Table 5.1 that the Normal reef has the lowest mineral grades recoveries of the three reef types making it the most problematic of the three reefs to mine and process.

Tables 5.1 and 5.2 further show that the three reef types respond differently to changes in the flotation conditions. For example, the Normal reef is shown to be significantly affected by copper sulphate addition and grind for pentlandite, pyrrhotite, BMS and total mass recoveries. In contrast, for the P2 reef the effect of grind is far less pronounced and in the NP2 reef grind size has no effect on flotation performance and only total mass is affected by copper sulphate addition, indicating gangue activation.

Table 5.1: *Summary of the main results for the laboratory scale flotation test for each reef type, comparing tests using standard reagent suites, reagents suites without copper sulphate and different grind sizes. Ccp – chalcopyrite, Pn – pentlandite, Po – pyrrhotite, BMS – base metal sulphides. The two values ringed in orange show more than 100% recovery for pyrrhotite within the NP2 reef. Unlike chalcopyrite, pentlandite and total sulphides these pyrrhotite values are calculated by mass balance. They are therefore subject to errors in sulphide stoichiometry and ratios, which may produce inaccurate recoveries.*

Reef Type	$CuSO_4$ Addition	Grind Size	Total Mass Recovery (g)	Total Water Recovery (g)	Ccp Grade	Ccp Recovery	Pn Grade	Pn Recovery	Po Grade	Po Recovery	BMS Grade	BMS Recovery
Normal Reef	✔	Standard	57.6	660	3.19%	83.5%	7.22%	84.2%	3.06%	49.9%	16.2%	81.6%
		Standard	54.4	632	2.43%	85.2%	5.46%	74.5%	2.41%	40.8%	13.8%	71.6%
	✔	Fine	68.9	854	2.94%	85.4%	5.39%	74.3%	2.00%	71.4%	13.8%	84.9%
		Fine	66.1	789	1.81%	84.9%	4.26%	70.7%	3.58%	44.5%	10.9%	74.6%
NP2 Reef	✔	Standard	34.6	498	3.63%	90.0%	7.73%	83.9%	11.02%	100.8%	29.9%	90.1%
		Standard	29.8	516	3.67%	90.1%	7.31%	78.5%	12.41%	106.5%	32.3%	91.6%
	✔	Fine	32.7	420	4.77%	87.7%	10.09%	78.5%	10.68%	98.1%	30.6%	92.8%
		Fine	30.4	526	4.01%	90.7%	9.07%	78.3%	11.60%	94.8%	33.7%	90.9%
P2 Reef	✔	Standard	41.4	355	6.21%	81.4%	12.62%	73.8%	5.77%	50.2%	30.0%	81.3%
		Standard	44.7	446	3.84%	86.2%	10.82%	70.5%	3.77%	38.1%	24.5%	77.1%
	✔	Fine	46.5	401	5.23%	85.4%	11.24%	70.1%	4.82%	50.6%	26.0%	84.4%
		Fine	45.1	450	3.71%	86.7%	9.76%	66.4%	2.54%	29.0%	21.9%	76.9%

Table 5.2: *Analysis of variance of flotation experiments, showing which flotation parameters (copper sulphate addition, grind size and reproducibility) had statistically significant effects on the results. The table on the left shows the calculated ANOVA percentages with the table on the right showing which results were statistically significant. *** is >95% significant; ** is 90-95% significant; * is statistically insignificant but practically significant; ~ is statistically significant but practically insignificant and blank boxes are both practically and statistically insignificant. Repeatability is a statistically insignificant effect for all the results.*

Ore Type		Std Dev	CuSO₄	Grind	Rpt.	Interactions			
			A	B	C	AB	AC	BC	ABC
Normal Reef	Ccp	1.2	43.9	59.1	36.8	72.6	56.4	64.2	76.1
	Pn	2.5	98.7	97.0	3.2	91.3	40.4	43.8	85.5
	Po	3.1	99.8	99.6	73.0	98.5	22.3	72.2	55.0
	Total BMS	1.6	99.9	95.4	79.0	10.1	51.5	51.5	51.5
	Total Mass	2.8	79.2	99.6	41.0	11.4	79.2	36.2	65.5
NP2 Reef	Ccp	0.7	96.8	84.8	7.8	96.4	82.8	54.8	49.3
	Pn	2.6	77.6	79.7	33.7	79.7	18.4	79.1	71.0
	Po	4.5	27.4	91.4	52.2	77.3	64.6	79.4	10.5
	Total BMS	1.8	14.8	52.8	46.3	75.2	74.0	75.2	5.9
	Total Mass	0.6	99.8	70.8	84.9	92.0	45.0	7.0	45.0
P2 Reef	Ccp	1.3	97.5	69.3	30.0	83.3	82.2	6.2	65.8
	Pn	1.2	98.5	98.9	70.0	16.9	70.4	42.7	56.8
	Po	6.5	97.8	60.2	26.7	63.4	86.7	27.1	30.7
	Total BMS	1.1	99.9	87.7	15.0	92.2	85.6	24.6	50.0
	Total Mass	2.9	33.9	75.0	20.6	67.7	40.1	35.5	85.9
	Average	2.8							

Ore Type		Std Dev	CuSO₄	Grind	Rpl.	interactions			
			A	B	C	AB	AC	BC	ABC
Normal Reef	Ccp	1.2							
	Pn	2.5	***	***		**			
	Po	3.1	***	***		***			
	Total BMS	1.6	***	***					
	Total Mass	2.8	*	***					
NP2 Reef	Ccp	0.7	~			~			
	Pn	2.6							
	Po	4.5		**					
	Total BMS	1.8							
	Total Mass	0.6	***			**			
P2 Reef	Ccp	1.3	~						
	Pn	1.2	***	***					
	Po	6.5	***						
	Total BMS	1.1	***			**			
	Total Mass	2.9							
	Average	2.8							

5.2 Mass-Water Recoveries

Tables 5.1 and 5.2 shows the final cumulative values for mass and water recovery, along with their statistical significance. Figure 5.1 compares the effects of copper sulphate addition and grind size on mass-water recovery curves with flotation time. From Figure 5.1 it is clear that the NP2 reef has a reduced mass to water ratio, compared with either the Normal or P2 reefs. Furthermore, the NP2 reef shows the effect of copper sulphate activation of gangue minerals, a feature that is absent from either the Normal or P2 reefs. The Normal reef is affected by grind with greater mass recoveries, probably due to increased froth stabilisation and recovery by entrainment.

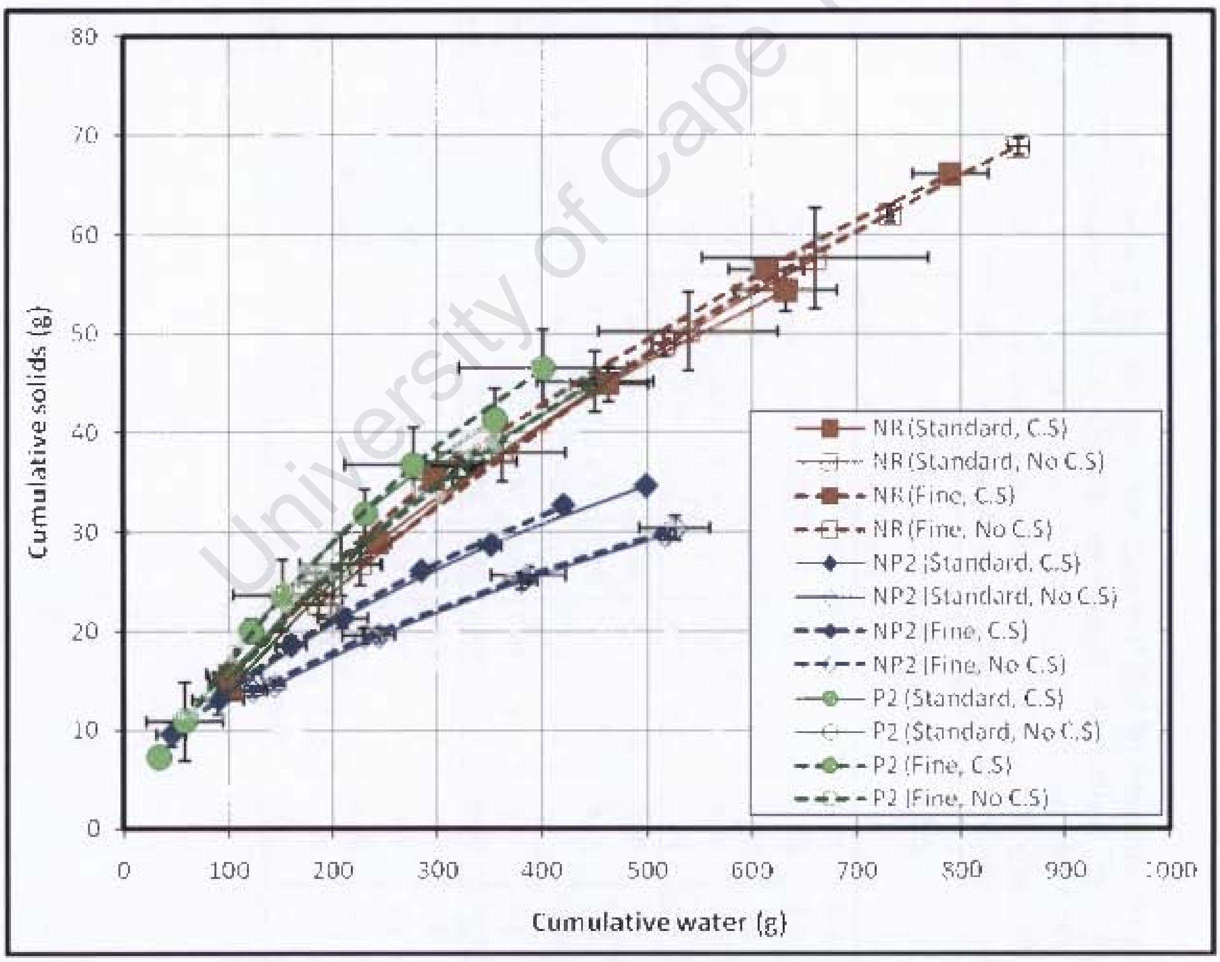


Figure 5.1: *Graph showing mass-water recoveries for the standard and fine grinds, with and without copper sulphate for the three reef types. C.S. refers to copper sulphate.*

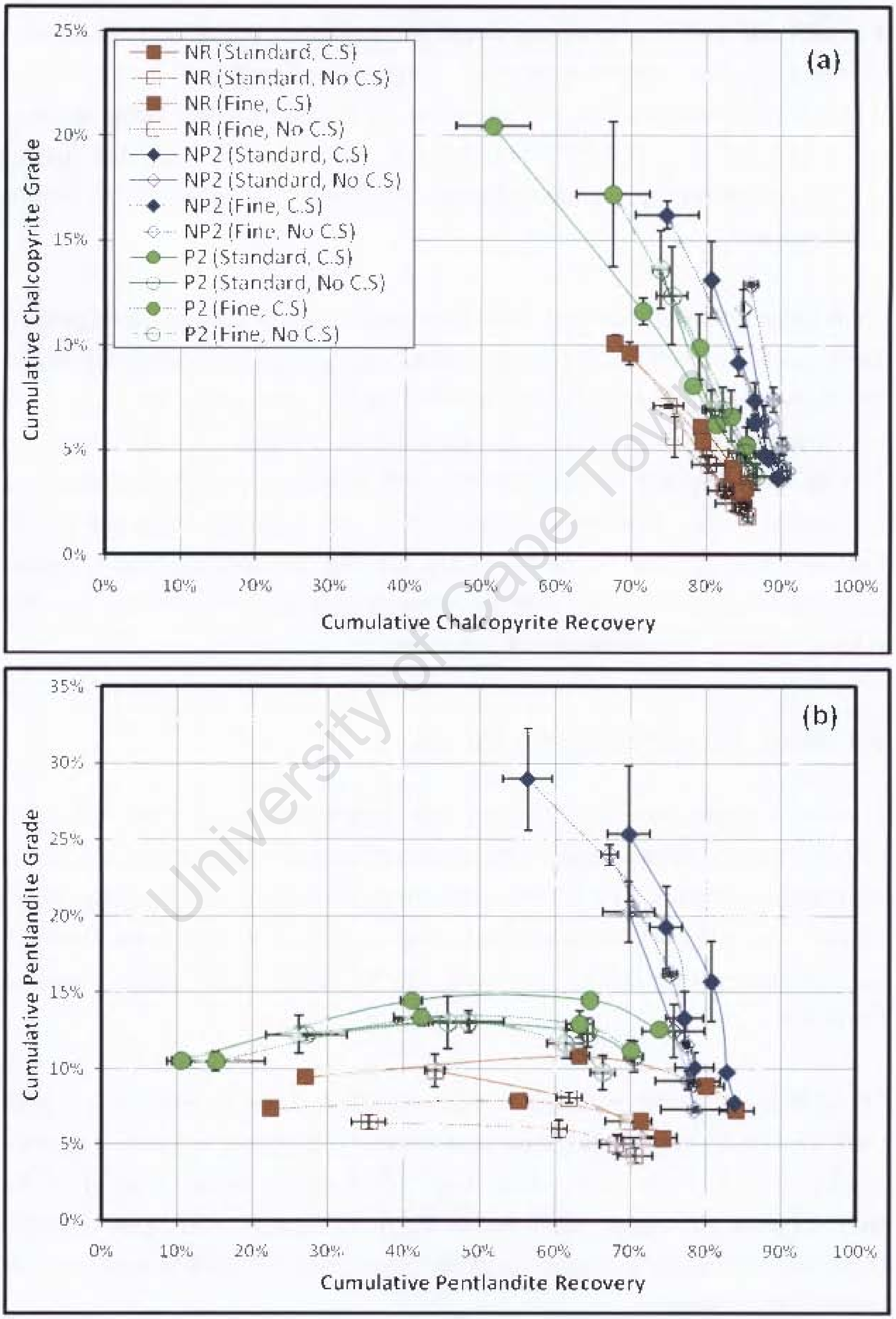
For each reef type the mass-water curves are non-linear. This reflects the recovery of solids by two separate processes, namely entrainment and flotation. Since entrainment is non-selective, and dependent on particle size and water recovery, some gangue minerals will always be recovered. However floatable gangue represents a potentially treatable side effect of the flotation process, with the addition of depressant.

If it is assumed that the amount of gangue recovered by entrainment is roughly the same for all three reef types, then the higher mass to water ratios within the Normal and P2 reefs strongly suggest more floatable gangue within these two reefs. From the cumulative values the magnitude of the increased floatable gangue component may be up to 35g and no less than 10g. This represents considerable dilution of concentrate grade, especially considering that the maximum mass pull for the Normal reef is 70g and for the P2 reef is just 46g. The NP2 reef shows copper activation of gangue minerals, whereas neither the Normal nor the P2 reefs do. This is likely to be due to the activation of plagioclase.

5.3 Mineral Grade-Recovery Curves

Cumulative chalcopyrite recoveries are high (81.4-90.7%) for all three reef types, reflecting its excellent floatability performance. Within this range the NP2 reef reports the highest recoveries (87.7-90.7%), with comparable cumulative final grades to the P2 reef (~5%). The Normal reef reports similar recoveries to the P2 reef (80-85%), but with lower grades (2-3%), correlating with the higher mass recovery shown in Figure 5.1.

Cumulative pentlandite recoveries show considerable variation between the three reefs (66.4-84.2%) as well as a clear dependence on grind size and copper sulphate addition. The NP2 and Normal reef report the optimum recovery (~84%) for the standard grind with copper sulphate. For the Normal and P2 reef types increased grind lowers grades and recoveries, whilst copper sulphate addition improves final grades and recoveries.


(a)
NR (Standard, C.S)
NR (Standard, No C.S)
NR (Fine, C.S)
NR (Fine, No C.S)
NP2 (Standard, C.S)
NP2 (Standard, No C.S)
NP2 (Fine, C.S)
NP2 (Fine, No C.S)
P2 (Standard, C.S)
P2 (Standard, No C.S)
P2 (Fine, C.S)
P2 (Fine, No C.S)
Cumulative Chalcopyrite Grade
Cumulative Chalcopyrite Recovery
0% 5% 10% 15% 20% 25%
0% 10% 20% 30% 40% 50% 60% 70% 80% 90% 100%
(b)
Cumulative Pentlandite Grade
Cumulative Pentlandite Recovery
0% 5% 10% 15% 20% 25% 30% 35%
0% 10% 20% 30% 40% 50% 60% 70% 80% 90% 100%

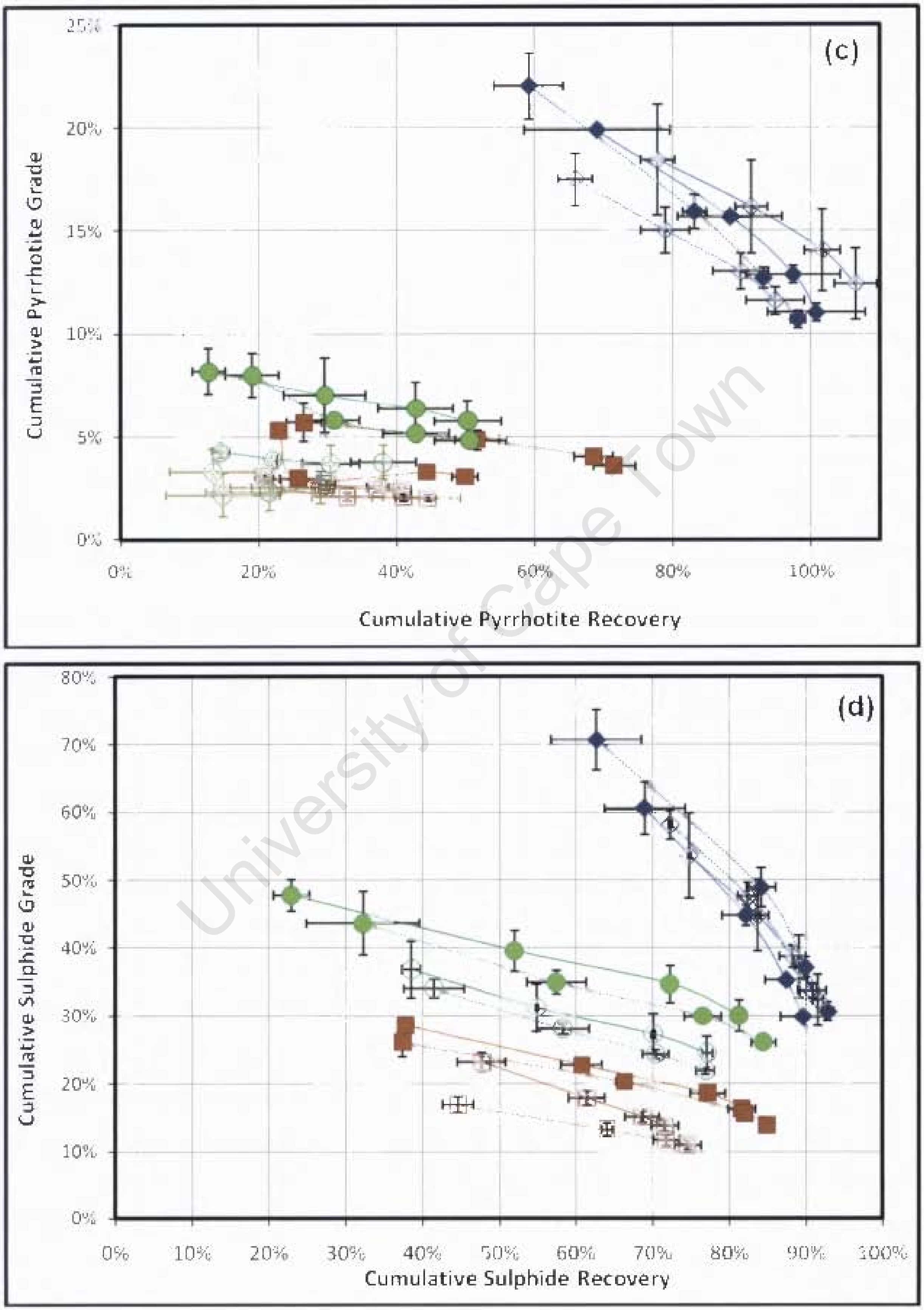


Figure 5.2: *Mineral Grade-Recovery Curves. a.Chalcopyrite, b. Pentlandite, c. Pyrrhotite, d. Total BMS. Note that the vertical scales are different in each case. The key for all four graphs is as displayed on graph (a). C.S. refers to copper sulphate.*

Pentlandite recovery is reduced for all cases with increased grind, even though liberation is improved (see Section 4.2).

Cumulative pyrrhotite recoveries were calculated, not measured, and problems associated with this approach are displayed in Figure 5.2c. Total recoveries are displaced from their true values as shown by greater than 100% recoveries for the NP2 reef and occasional concentrates reporting nearly no pyrrhotite recoveries (these appear as 'kinks' in the graph, e.g. for the Normal reef standard grind with copper sulphate). These difficulties arise from assumptions inherent to mass balance calculations, namely that the stoichiometry of copper and nickel minerals is fixed, and the monosulphide (pyrrhotite) to disulphide (pyrite) ratio within the sample remains constant. For the NP2 reef the pyrrhotite recoveries have been overestimated, but for the Normal and P2 reefs the recoveries have probably been underestimated.

Despite these problems there are some definite patterns that can be identified. Firstly, copper sulphate addition improves pyrrhotite grades and recoveries for both the Normal and P2 reefs, but has no effect on the NP2 reef. The effect of increasing grind also appears to improve Normal reef recovery, but reduce NP2 reef recovery. Furthermore, despite the displacement in recovery, it seems likely that the NP2 reef reports the highest pyrrhotite recoveries, which is in accordance with its higher BMS recoveries as well as its better pyrrhotite liberation (see Section 4.2).

Cumulative BMS recoveries show considerable variation between the three reefs (71.6-92.8%), and are variably affected by copper sulphate addition and grind. The Normal and P2 reefs show increases in grade and recovery with copper sulphate addition. However the effect of grind is to lower grades, whilst improving recoveries. The NP2 reef reports the optimum recoveries (90.1-92.8%), which is ~5% greater than either the Normal and P2 reefs. NP2 BMS recoveries are unaffected by either grind or copper sulphate.

5.4 Discussion Of Effects

5.4.1 Effect Of Copper Sulphate Addition

Tables 5.1 and 5.2 show that copper sulphate addition has statistically significant, but variable effects over the three reef types. For the Normal reef and P2 reefs, pentlandite, pyrrhotite and total sulphides recovery show increases, with total mass and chalcopyrite the variables unaffected by copper sulphate addition

For the NP2 reef, only total mass pull shows a significant increase with copper sulphate addition. This distinct increase within the NP2 reef reflects inadvertent activation of gangue minerals. Since the Normal and P2 reefs do not show any such increase, inadvertent copper activation of gangue minerals was not indicated.

Figure 5.2 shows that grades are also affected by copper sulphate addition. For the Normal and P2 reefs, chalcopyrite, pentlandite, pyrrhotite and total sulphide grades all improve. However, although grades and recoveries improve, the initial recoveries are in fact lower. This is because the initial BMS recovery is lower but initial mass pull is much lower showing BMS make up a greater proportion of a much smaller mass.

For the NP2 reef, copper sulphate addition leads to the flotation of more gangue (Figure 5.1). However, although total mass pull is higher, for the initial concentrate (Figure 5.1) the value is lower. This feature means that for BMS the initial grades are higher (Figure 5.2). As expected however the increased flotation of gangue with time, means that final cumulative grades end up lower (or equivalent in the case of pentlandite) than for experiments without copper sulphate.

5.4.2 Effect Of Grind Size

Tables 5.1 and 5.2 show that, with the exception of the Normal reef, grind size is less statistically significant that copper sulphate in affecting BMS or mass recovery. For the Normal reef, pentlandite shows a significant reduction in final recoveries, but pyrrhotite and total sulphide showed increases in recovery. Total mass also shows a significant increase. For the NP2 reef there are no significant changes for any of the parameters, whilst for the P2 reef only pentlandite total recovery shows a significant reduction. As with copper sulphate addition chalcopyrite total recovery shows no significant change in any of the reef types.

Figure 5.1 shows the increase in mass pull for the Normal reef with a fine grind size. This increase reflects increased grinding of gangue minerals, exposing low temperature alteration minerals, thereby encouraging true flotation of more gangue minerals. These low temperature alteration minerals (e.g. talc) also stabilise the froth encouraging recovery by entrainment. That the P2 reef isn't likewise affected suggests that minerals such as chlorite play a more prominent role in gangue flotation than expected.

Figure 5.2 shows that grades are also affected by grind size, though not as extensively as with copper sulphate addition. For the Normal and P2 reefs, pentlandite and total sulphides show a reduction in grade with pyrrhotite showing a minor increase.

For pentlandite, the increase in grind (from standard to fine) is followed by a reduction in recovery, despite improved liberation. This reduction is statistically significant in the Normal and P2 reefs, and also occurs for the NP2 reef with copper sulphate addition. This is most likel related to pulp chemistry effects such as collector starvation or increased surface oxidation related to increased exposed surface area. Pyrrhotite isn't likewise affected due to competitive adsorption, whilst chalcopyrite will always float due to its good natural floatability.

5.5 Summary of Flotation Results

From the flotation results, some key findings can be recognized.

- The NP2 reef shows the best recovery for chalcopyrite, pentlandite, pyrrhotite and BMS, following by the P2 reef and lastly by the Normal reef.

- For the NP2 reef all sulphide grades and recoveries are unaffected by copper sulphate addition or grind size. However the NP2 reef does show copper activation of gangue minerals (most likely plagioclase).

- Grade-recovery curves and analysis of variance show that overall the Normal reef responds to copper sulphate addition and changes in grinds size. In contrast the P2 reef responds to copper sulphate addition but is only moderately affected by changes in grind.

- Chalcopyrite recoveries are unaffected by copper sulphate addition. However, initial grades are improved within the Normal and P2 reefs. Chalcopyrite recovery is unaffected by changes in grind size.

- For all reef types pentlandite grade increased with copper sulphate addition. However, only the Normal and P2 reefs show improved recoveries. A fine grind size reduces pentlandite grades and recovery within all three reefs.

- For the Normal and P2 reefs, pyrrhotite grades and recoveries improve with copper sulphate addition. With increased grind grades remain roughly the same, but recoveries improve reflecting the dependence of pyrrhotite flotation on liberation. For the NP2 reef there was no statistically significant effect on recovery or grade.

- Grade-recovery curves for BMS for the Normal and P2 reefs show improvements in grade and recovery with copper sulphate addition. With an increased grind, grades are reduced but recoveries are improved. For the NP2 reef, BMS grades and recoveries are unaffected by copper sulphate addition or grind.

6 Discussion

6.1 The Effect of Mineralogy on Flotation Performance.

6.1.1 Milling Time

Milling times for the three reefs suggest that the NP2 reef is the softest, relative to the Normal and P2 reefs, with the P2 reef the hardest. This difference in hardnesss can be attributed to mineralogy, texture and degree of alteration. It implies that plagioclase rich rocks (i.e. the NP2 reef) are softer than orthopyroxene rich rocks (i.e. the Normal and P2 reefs). Furthermore, since the mineralogy of the Normal and P2 reefs are similar the role of alteration and texture are also critical. The Normal reef is slightly more altered than the P2 reef, and contains a much thicker inter-chromitite pegmatite (~150 cm, compared with ~40 cm). This suggests that increased alteration and pegmatitic character serve to decrease the milling time. The major implication of these milling times is the expected ore throughput time, with the NP2 reef expected to process quicker than either the Normal or P2 reefs.

6.1.2 Floatable Gangue

The basic gangue minerals are the same for each reef, namely orthopyroxene, plagioclase, olivine, clinopyroxene, chromite and secondary alteration minerals such as biotite, serpentine, talc and chlorite. Relative to the NP2 reef, the Normal and P2 reefs contain a much higher abundance of primary ferromagnesian minerals, orthopyroxene, olivine and associated altered serpentine, talc and chlorite.

Talc is a highly floatable gangue mineral, capable of not only entering the concentrate through true flotation, but also of carrying other gangue minerals such as orthopyroxene into the concentrate through association (Becker *et al.*, 2006). Furthermore, talc has a froth stabilising effect promoting increased water recovery, and therefore mass recovery by entrainment.

The higher quantities of floatable gangue within the Normal and P2 reefs are due to two main processes, namely; inadvertent flotation of gangue minerals through association with hydrophobic gangue minerals (figure 6.1), and increased entrainment through increased froth stability. The main gangue mineral in each process is orthopyroxene.

Figure 6.1: *Orthopyroxene (dark green) and talc (red) particles from the P2 reef. Talc is clearly associated with the rims of the particles.*

This increased quantity of floatable gangue has two main effects. The first is to lower grades, this is seen for the Normal and P2 reefs, where initial grades are lower than the NP2 reef, particularly for the total sulphide values (cf. Figure 5.2d). The second is to slow the rate of recovery of BMS particles, and thereby reduce total recovery. That the Normal reef has a larger mass recovery than the P2 reef suggests that alteration minerals such as chlorite play a more prominent role in inadvertent gangue flotation than previously thought.

Relative to the P2 reef the Normal reef has lower chalcopyrite, pentlandite, pyrrhotite and BMS concentrate grades. This feature is related to initial head grade (see Section 6.1.4), but is also related to increased water recovery, and therefore mass recovery by entrainment due to increased froth stability. Since the talc values for the Normal and P2 reefs are similar, this increased froth stability is most likely related to the overall abundance of alteration minerals (2.64 wt% compared with 1.80 wt%), and the modal values for minerals such as chlorite (0.57 wt% compared with 0.13 wt%).

6.1.3 Sulphide Liberation

Across the three reef types there is one main sulphide texture and three subsidiary textures; the main sulphide texture is fine to medium grained (0.5-5mm), composites (Figure 6.2a), predominantly of pyrrhotite, pentlandite and chalcopyrite, but also with minor amounts of pyrite, cubanite and other sulphides. The three subsidiary textures are: fine grained (<0.2mm), largely monomineralic inclusions in the major silicate phases (Figure 6.2b) and chromite; sulphides concentrated within microfractures that occurred during brittle deformation (Figure 6.2c); and very fine unidentifiable sulphides located within secondary silicate minerals such as serpentine (Figure 6.2d).

The Normal and P2 reefs contain all four sulphide textures, whereas the NP2 reef contains only composite sulphides (Figure 6.2a) and fine grained sulphides (Figure 6.2b). This lack of very fine sulphide development and sulphide veining suggests that sulphides within the NP2 reef will be the easiest to liberate as is the case for chalcopyrite, pyrrhotite and BMS.

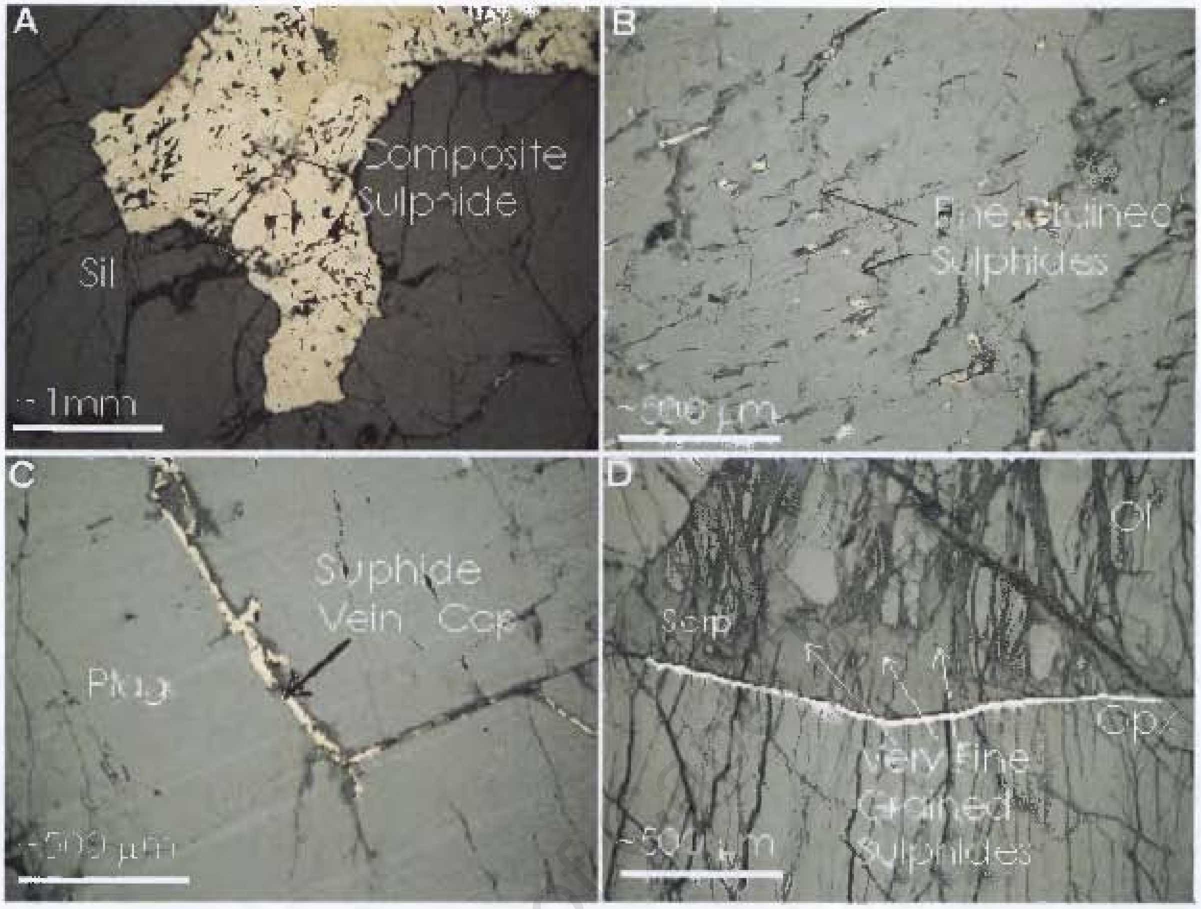


Figure 6.2: *Sulphide textures. a. composite sulphide about 2mm across. b. Fine grained sulphides locked in an orthopyroxene megacryst. c. sulphide vein within plagioclase grain. d. Very fine grained sulphides enclosed in serpentine.*

The reason for this difference between the NP2 reef and the Normal and P2 reefs is the pegmatitic nature of the footwall, which is related to the initial 'pre-MCU event' footwall. Since the footwall to the Normal and P2 reef was melanocratic, the subsequent deposition of a new hot magma pulse (the MCU), led to reconstitution and grain coarsening. This encouraged the development of coarse composites as well as complex secondary remobilisation of sulphides, generating three subsidiary textures. In contrast the footwall to the NP2 reef was leucocratic, and responded very differently to the deposition of the MCU. Rather than grain coarsening, the magma infiltrated between the plagioclase grains occasionally reacting with rare orthopyroxene grains. The result was largely undisturbed anorthositic or leucocratic layers within which is a thin troctolized layer representing the extent of downwards magma infiltration.

The variations in sulphide development described above can be linked in with observed variations in liberation and flotation performance. Firstly, the dominance within all reef types of medium grained composite sulphides (Figure 6.1a) explains the excellent liberation (>80%) observed for all BMS at both grind sizes. Since PGM are invariably associated with sulphides, any such PGM associated with the major composite sulphides can also be expected to be recovered.

Secondly, the presence of fine-grained BMS locked within primary orthomagmatic (e.g. orthopyroxene) minerals explains why sulphide recovery is not optimised for each of the three reefs. This is because after grinding the fine-grained sulphides will be locked or only partially liberated, and being trapped within hydrophilic minerals will be retained in the pulp. The lower chalcopyrite, pyrrhotite and BMS recoveries observed within the Normal and P2 reefs values is a function of the greater quantity of fine grained sulphides minerals present.

The presence of very-fine grained sulphides within the Normal and P2 reefs, deported within secondary replacement minerals (serpentine and paragonite), represents a further, though minor, decrease in liberation. Furthermore, these replacement minerals display minor mobility, often being traced for the length of a thin section (~5 cm). In more altered reef areas the degree of mobility will probably be much higher making the transportation of BMS away from the reef region highly likely. This is a critical point, since as already noted the PGM are intimately associated with the BMS and loss of paymetal content from the reef region is clearly detrimental.

6.1.4 Head Grade

The P2 reef contains the highest head grades of the three reefs, with significantly more pentlandite and pyrrhotite and total sulphide relative to the Normal and NP2 reef types (Figure 4.10). This is due to sulphide mineralization occurring in association with the P2 chromitite prior to the deposition of the MCU. The higher head grades correlate with higher concentrate grades and recoveries relative to the Normal reef for chalcopyrite, pyrrhotite, pentlandite and BMS. However, the NP2 reef still contains higher pentlandite, pyrrhotite and BMS grades than the P2 reef as well higher recoveries. This feature does not follow the general trend (Napier-Munn, 1998) and confirms that this relationship is dependent on ore type.

6.2 Metallurgical Factors Affecting Flotation

6.2.1 Copper Sulphate

The addition of copper sulphate has been shown to be statistically significant (Table 5.2) in improving recoveries for pentlandite, pyrrhotite and BMS for the Normal and P2 reefs. For the NP2 reef only total mass is significantly (Table 5.2) affected by copper sulphate addition, and this is related to the inadvertent copper activation of gangue minerals. Martinovic et al, (2005) showed that the addition of copper sulphate resulted in the inadvertent activation of all the major Merensky reef gangue minerals: namely orthopyroxene, feldspar and chromite. However, they also showed that the hydrophobicity of talc is simultaneously reduced. Since orthopyroxene and talc are mineralogically associated, this simultaneous inadvertent activation of orthopyroxene, but reduced hydrophobicity of talc may act to nullify each other resulting in negligible increased orthopyroxene flotation. It is therefore inferred that the copper activation observed within the NP2 reef is related to the inadvertent activation of feldspar minerals, which being in far greater abundance within this reef, leads to increased mass recovery. It is also noted that all reef types showed improved initial mineral grades, however in the NP2 reef this is not carried through to the final concentrate.

6.2.2 Grind Size

The three reef types were variably affected by increasing the grind (from standard to fine). The Normal reef shows a significant improvement (>5%) in liberation for all the sulphides, though this can only be definitively correlated with BMS recovery. A positive correlation is expected with pyrrhotite recovery but due to the limitations of the mass balance calculation this cannot be verified. The use of quantitative mineral analysis on the concentrates is recommended to resolve the effects of grind on pyrrhotite recovery. Chalcopyrite recovery is unaffected by improvement in liberation (51 to 59%) due to its easy floatability and an increase in pentlandite liberation (71 to 76%) is inversely related to a reduction in recovery. This inverse relationship is not due to decreased chalcopyrite association (see Appendix D for data), rather it is interpreted as a function of collector starvation due to the increased surface area of the pentlandite, or possibly increased surface oxidation.

Due to the lower amounts of fine grained sulphides and the lack of very fine grained disseminated sulphides within the NP2 reef, liberation and mineral recovery are unaffected by an increase in grind (Tables 5.1, Figure 5.2). This means that effective liberation and recovery of BMS (>90%) is achieved at the standard grind. However, a reduction in recovery is noted for pentlandite with increased grind. As with the Normal reef, this feature is regarded as due to pulp chemistry effects.

For the P2 reef there was no significant change in liberation with change in grind. Considering that the sulphide development was very similar to the Normal reef, this feature is most likely related to the increased hardness of the ore and points to undergrinding of the reef type, even with 80% passing 75 μm. Reduced alteration, relative to the Normal reef, may also contribute to the increased hardness. A reduction in recovery for pentlandite is the only statistically significant effect of increased grind (Table 5.2), though increased grades are noted. As with the Normal and NP2 reefs pulp chemistry effects are interpreted as the reason for this.

6.3 Summary

6.3.1 Normal Reef

The Normal reef consists of a thick inter-chromitite pegmatite (~150 cm) bounded by two chromitite stringers, a melanorite and an anorthosite. The formation of this pegmatitic zone led to grain-scale re-mobilisation of base metal sulphides, increasing their complexity and allowing the formation of four sulphide textures. Of these textures, the increase in abundance of fine-grained sulphides, and very fine grained sulphides has led to the worst liberation characteristics of the three reef types (the Normal reef was the only reef to show improved liberation with increased grinding). Moreover, the Normal reef also contains the largest abundance of alteration minerals (talc, serpentine, chlorite), which are thought to increase the inadvertent flotation of gangue minerals, as well as stabilise the froth allowing increased recovery by entrainment. The complex sulphide development, together with increased alteration makes the Normal reef the most problematic of the three reefs to mine and process.

6.3.2 NP2 Reef

The NP2 reef consists of a single chromitite stringer underlain by medium grained anorthosites, troctolites and norites and overlain by a melanorite. The formation of the troctolite through interaction with the MCU has resulted in simple sulphide textures relative to the Normal and P2 reefs. That is medium grained composite sulphides and smaller amounts of fine grained monomineralic sulphides. In addition, the degree of alteration is the lowest for the three reef types, and the time taken to mill to the required grind is the shortest. This shorter laboratory milling time suggests higher throughput of ore on an industrial scale will be possible, ensuring a greater turnover of ore to paymetal content. The simpler sulphide developments and lower degree of alteration has resulted in the best base metal sulphides of the three reef types, as well as the lowest floatable gangue recovery. These sulphide developments together with favourable alteration and modal mineralogy make this reef type the easiest ore to process relative to the Normal and P2 reefs. Copper activation of gangue minerals is unique to this reef type, a feature related to the high plagioclase content of the reef.

6.3.3 P2 Reef

The P2 reef is mineralogically similar to the Normal reef but is considered less problematic to process due to several slight differences. Firstly, the P2 reef consists of a thin (~40cm) inter-chromitite pegmatite bounded by a harzburgite and a melanorite. As with the Normal reef, the formation of this pegmatite led to grain-scale re-mobilsation of base metal sulphides, increasing their complexity and allowing the formation of four sulphide textures. However, since this pegmatite was just 40 cm thick the detrimental effect on liberation and base metal sulphide was not as pronounced as in the Normal reef. Relative to the Normal reef the P2 reef contains higher sulphide feed grades and lower degrees of alteration. The higher sulphide feed grades improve chalcopyrite, pentlandite, pyrrhotite and BMS grades and recoveries, whilst the lower degrees of alteration also achieve the same by reducing gangue recovered by flotation and entrainment. However, the P2 reef was the hardest reef to mine taking 23 mins to grind to 60% passing 75 μm, compared with 18 mins for the NP2 reef, and 19 mins for the Normal reef. This extra hardness is probably related to its increased orthopyroxene content, and reduced alteration relative to the Normal reef. Unlike the Normal reef, increased grinding did not improve liberation, and this is probably related to the increased hardness of the ore.

7 Conclusions and Recommendations

This project investigated the unique mineralogy and processing requirements of three Merensky reef types (Normal, NP2 and P2) at Northam Platinum Ltd, with the aim of linking observed mineralogical differences to differences in the beneficiation of the base metal sulphides through froth flotation.

The overall conclusion from this investigation is that the NP2 reef is the easiest of the three reefs to process, producing optimum chalcopyrite, pentlandite, pyrrhotite and base metal sulphide recoveries and highest grades. The Normal reef is considered the most problematic, with the P2 reef slightly improved relative to the Normal reef.

7.1 Conclusions

From this investigation the following conclusions can be drawn with respect to answering the initial key questions outlined in Chapters 1 and 2.

- The NP2 reef is mineralogically and texturally distinct and is dominated by plagioclase (62.6 wt%). It has a medium grained texture throughout and is largely unaltered (1.13 wt% alteration minerals). In contrast the Normal and P2 reefs are very similar, being dominated by orthopyroxene (51.2 wt% and 68.6 wt%, respectively), having a pegmatitic lithology (~150 cm in the Normal Reef and ~40 cm in the P2 reef) and containing higher levels of alteration (2.64 wt% and 1.80 wt% alteration minerals, respectively).

- The NP2 reef had the shortest milling time to achieve the same particle size distribution, followed by the Normal reef and lastly by the P2 reef. This difference is associated with higher plagioclase content of the NP2 reef and lower orthopyroxene content. The Normal reef has a reduced time compared with the P2 reef due to slightly greater alteration and a greater proportion of pegmatitic textures. In addition to this the NP2 reef showed no improvement in grade or recovery with increased grind, whilst the Normal and P2 reefs reported improvements in pyrrhotite and BMS recovery. This suggests that the NP2 reef need not be milled for any longer than required to produce a standard size distribution to achieve BMS recoveries of >90% but that the Normal and P2 reefs would undergo re-grinds to optimize recovery. This suggests a dramatically improved ore throughput is possible for the NP2 reef relative to the Normal and P2 reefs.

- The NP2 reef reports the best sulphide liberation, consistent with largest grain size, followed by the P2 reef and lastly the Normal reef. This is due to the lack of a pegmatitic reef section, and hence the lack of complex sulphide developments (e.g. very fine grained sulphides). The Normal reef was the only reef type significantly affected, in terms of liberation, by changes in grind and this reflects the high proportion of fine and very fine grained sulphides within the reef.

- The NP2 reef reports the lowest amount of floatable gangue, followed by the P2 reef and lastly the Normal reef. This is due to the lowest degree of alteration and orthopyroxene content, resulting in reduced inadvertent flotation of gangue and reduced recovery of gangue by entrainment.

- Copper sulphate addition improves pentlandite, pyrrhotite and BMS recoveries for the Normal and P2 reefs, but has no significant affect on the NP2 reef. The NP2 reef does show copper activation of gangue, related to its high plagioclase content.

- Increased grind reduces pentlandite, pyrrhotite and BMS grades for the Normal and P2 reefs, whilst increasing pyrrhotite and BMS recoveries. The NP2 reef is largely unaffected by changes in grind.

- For all reef types an increase in grind improves pentlandite liberation, but this is negatively correlated with a reduction in recovery. The reasons for this have not been confirmed but are interpreted as resulting from collector starvation due to increased surface area exposure for the same collector dosage, or increased surface oxidation.

7.2 Recommendations

In light of the investigations carried out within this project a number of recommendations can be made for further work.

- Characterisation of PGM type, location and association to ascertain the similarities or differences between the three reefs, as well as evaluating the differences in flotation performance.

- Confirmation of pyrrhotite flotation performance with quantitative mineralogical analysis (QEMSCAN or MLA) of concentrates and tails, with the aim of more reliably linking results to feed characteristics. More detailed information on chalcopyrite and pentlandite recovery could be ascertained through the same analysis.

- A more fundamental analysis of pentlandite behaviour using surface analytical techniques (e.g. TOF-SIMS) in order to ascertain the reasons for the consistent reduction in pentlandite recovery with increased grind and liberation.

- The inference of copper activation of feldspar within the NP2 reef could be further confirmed by surface analytical techniques (e.g. TOF-SIMS) to determine whether copper activation was the sole cause for the increased reporting of gangue material to the concentrate.

- A detailed investigation into the hydrophobicity of other gangue alteration minerals such as biotite, serpentine and chlorite. Especially with reference to the significant proportions of these minerals within the Normal and P2 reefs, and their possible effects on gangue activation, and froth stabilisation.

- Further investigations into other reef types at Northam Platinum Limited, in particular NP2 reefs with far thicker, or absent troctolite layers, and the FWP2 reef.

- The extension of process mineralogy investigations on the Merensky reef, to look at the differing processing performances, in terms of grinding and flotation performance, of the Merensky contact reef, thin pegmatitic reef, and thick pegmatitic reef, as they are developed within the Rustenburg facies.

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Appendix A – Borehole and Sampling Logging Codes and Descriptions

A1 Normal Reef

3A	Merensky Pyroxenite (Melanorite)
4A	Merensky Chromitite
5A	Merensky Pegmatite
5H	Merensky Pegmatite
4X	Bottom Chromitite
6A	Footwall Anorthosite

A2 NP2 Reef

3A	Merensky Pyroxenite (Melanorite)
4A	Merensky Chromitite
8T	Footwall Marker Troctolite
8A	Footwall Marker Anorthosite
8N	Footwall Marker Norite

A2 P2 Reef

3A	Merensky Pyroxenite (Melanorite)
4A	Merensky Chromitite
5A	Merensky Pegmatite
12	P2 Chromitite
13D	P2 Dunite
13H	P2 Harzburgite

Appendix B - Analytical Methods

B1 Zeiss Petrographic Microscope

The Zeiss petrographic microscope has both a substage transmitting light source for analyzing translucent minerals such as orthopyroxene and an incident light source for analyzing the 'opaque' ore minerals such as the base metal sulphides. This dual light source capability means that sulphide textures and associations within a polished thin section can be readily discerned. The microscope was housed in the Department of Chemical Engineering at the University of Cape Town.

B2 JEOL Electron Microprobe (EMP)

Concentrations of nickel within olivine, orthopyroxene and plagioclase were determined on an electron microprobe. Analyses were run on a JEOL JXA-8100 Electron Probe Microanalyzer housed in the Department of Geological Sciences at the University of Cape Town. The current was 20nA, with an accelerating voltage of 25kV and beam diameter of 10μm. Prior to analysis, polished thin sections were carbon coated. This was done so that during bombardment by the electron beam there wasn't a build up of charge on the slide. Synthetic and natural standards were used to calibrate the machine prior to analysis (Table B.1).

Calibration Mineral	Element Calibrated For
Rhodochrosite	Manganese
Potassium Feldspar	Aluminium
Diopside	Calcium
Rutile	Titanium
NiSi	Nickel
Chromite	Chromium
M-Ol	Magnesium, Silicon, Iron
K-H	Sodium, Potassium

Table B.1: Calibration minerals used for particular elements.

B3 Inductively Coupled Plasma - Optical Emission Spectrometry (ICP-OES)

Calibration standards were prepared from certified salts, diluted and matrix matched. The reference standards used were 51/ 71 (a Cu-Ni concentrate) and SARM71 (UG2 reef medium grade ore).

Feed samples were analysed for major elements (Mg, Al, Si, Ca, Ti, V, Cr, Mn, Fe, Co, Ni, Cu, Zn and Pb) using an ICP-OES housed in Mintek, Johannesburg. 2g samples were used from each size fraction with the determination limits for each element as 0.05%. Samples were fused with sodium peroxide and the molten is leached in either hydrochloric acid or nitiric acid before ICP-OES analysis.

B4 Atomic Absorption Spectrometry (AAS)

Copper and nickel analysis of the feeds, concentrates and tails were performed using the Varian AAS110 (automatic) and Varian AAS30 (manual) flame spectrophotometers. The gas used was an air/acetylene mixture with an air flow rate of 3.5 l/min and acetylene flow rate of 1.5 l/min. Copper method is a 100 mg/l calibration standard using wavelength 327.4 with a slit width of 0.5 nm. Nickel method is a 100 mg/l calibration standard using wavelength 352.5 with a slit width of 0.2 nm. Calibration was performed using a new rational algorithm with QC standards used routinely (limits 95% - 105%). The measurement mode was integration with an average of three readings for each sample.

B5 Sulphur Analyser

Sulphur analyses were performed on all feed, concentrate and tails samples using the LECO 423 sulphur analyser with automatic loader, housed within the Department of Chemical Engineering at the University of Cape Town. For standard operating conditions the sample is first weighed out (between 0.1g and 0.5g) before being placed into a furnace and burned in the presence of excess medical oxygen. The flowrate for the medical oxygen was 3.25 l/min and the temperature was 1350 ^{o}C. Analysis is by infrared detection. Linear calibration was performed using four standards with different sulphide concentrations (0.56%, 5.05%, 14.0%, 29.87%). Each standard was analysed using its own method, setting integration times using either low or high infrared cells.

Appendix C – Petrological Descriptions

C1 – Normal Reef

TS = Thin Section
PTS = Polished Thin Section

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	75%	A medium-grained (1-3 mm) subhedral orthopyroxene ortho to adcumulate with interstitial anhedral plagioclase. Minor clinopyroxene, biotite, chromite and sulphides. Varaible alteration of: orthopyroxene to talc, plagioclase to sausserite and very minor alteration of biotite to chlorite.
Northam Code	3A	Plagioclase	20%	
Sample Type	TS	Talc	1-2%	
Sample Code	HWNR01	Clinopyroxene	1%	
Rock Type	Melanorite	Biotite	1%	
		Sulphides	1%	
		Sausserite	1%	
		Trace amounts of chlorite, amphibole, and chromite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	75%	Silicate mineralogy as for above. Sulphide mineralogy consists of medium grained anhedral composites of pyrrhotite, granular pentlandite and chalcopyrite; interstitial to, and included within silicates. Also as monomineralic fine grained inclusions within silicates
Northam Code	3A	Plagioclase	20%	
Sample Type	PTS	Talc	1-2%	
Sample Code	HWNR01	Clinopyroxene	1%	
Rock Type	Melanorite	Biotite	1%	
		Sulphides	1%	
		Sausserite	1%	
		Trace amounts of chlorite, amphibole, and chromite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	70%	See Sample Description for TS for HWNR01
Northam Code	3A	Plagioclase	15%	
Sample Type	TS	Clinopyroxene	5%	
Sample Code	HWNR02	Talc	3-4%	
Rock Type	Melanorite	Biotite	2-3%	
		Sulphides	2%	
		Chromite	1%	
		Trace amounts of chlorite, amphibole, and sausserite	<1%	

Appendix C

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	70%	See Sample Description for PTS for HWNR01
Northam Code	3A	Plagioclase	15%	
Sample Type	PTS	Clinopyroxene	5%	
Sample Code	HWNR02	Talc	3-4%	
Rock Type	Melanorite	Biotite	2-3%	
		Sulphides	2%	
		Chromite	1%	
		Trace amounts of chlorite, amphibole, and sausserite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	35%	A medium-grained subhedral orthopyroxene ortho to adcumulate with minor interstitial anhedral plagioclase, also containing a coarse clinopyroxene poikilocryst, which dominates ~20% of the slide. This overlies a coarse well annealed chromite orthocumulate. There are minor interstitial sulphides throughout the slide.
Northam Code	3A-4A	Clinopyroxene	25%	
Sample Type	TS	Chromite	20%	
Sample Code	CRNR01	Plagioclase	10%	
Rock Type	Chromitite + Melanorite	Sulphides	5%	
		Talc	3-4%	
		Biotite	1%	
		Trace amounts of chlorite, amphibole, and sausserite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	35%	Silicate mineralogy as for above. Sulphide mineralogy consists of medium grained (2-3 mm) anhedral composites of pentlandite, pyrrhotite and chalcopyrite interstitial to silicates. Also as monomineralic fine grained inclusions within silicates.
Northam Code	3A-4A	Clinopyroxene	25%	
Sample Type	PTS	Chromite	20%	
Sample Code	CRNR01	Plagioclase	10%	
Rock Type	Chromitite + Melanorite	Sulphides	5%	
		Talc	3-4%	
		Biotite	1%	
		Trace amounts of chlorite, amphibole, and sausserite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Chromite	80%	A heavily annealed chromite accumulate, with minor interstitial plagiocalse, orthopyroxene rims and biotite. Sulphides are medium to finegrained composites of pyrrhotite, chalcopyrite and pentlandite. Monomineralic sulphides are common at triple point junctions within zones of heavy chromite annealing.
Northam Code	4A	Plagioclase	15%	
Sample Type	PTS	Orthopyroxene	2-3%	
Sample Code	CRNR02	Sulphides	1-2%	
Rock Type	Chromitite	Biotite	1-2%	
		Trace amounts of saussurite and talc	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Olivine	45%	Coarse grained (~2cm) pegmatite, containing variably altered olivine and plagioclase, with minor orthopyroxene. Plagioclase shows alteration to saussurite and possibly minor epidote, with some moderate veining. Olivine is variably replaced by serpentine, with dustings of magnetite.
Northam Code	5A-5H	Plagioclase	25%	
Sample Type	TS	Orthopyroxene	8%	
Sample Code	PGNR01	Serpentine	6%	
Rock Type	Pegmatite (Troctolite)	Saussurite	6%	
		Chromite	4-5%	
		Biotite	3-4%	
		Talc	1-2%	
		Sulphides	1%	
		Chlorite	<0.5%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Olivine	45%	Silicate mineralogy as for above. Sulphides are fine grained monominerals, predominantly of pyrrhotite and chalcopyrite. They are normally found included within the larger, coarser silicate and chromite grains. Very fine grained sulphides are found consistently within the serpentine, and occassionally the saussurite.
Northam Code	5A-5H	Plagioclase	25%	
Sample Type	PTS	Orthopyroxene	8%	
Sample Code	PGNR01	Serpentine	6%	
Rock Type	Pegmatite	Saussurite	6%	
		Chromite	4-5%	
		Biotite	3-4%	
		Talc	1-2%	
		Sulphides	1%	
		Chlorite	<0.5%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	80%	A coarse (~4 cm) orthopyroxene enclosing anhedral olivine. Secondary mineral assemblages are minor, with orthopyroxene showing minor replacement by talc, and olivine by serpentine. Chromite is usually associated with olivine, though not exclusively so.
Northam Code	5A-5H	Olivine	10%	
Sample Type	TS	Plagioclase	3%	
Sample Code	PGNR02	Chromite	3%	
Rock Type	Pegmatite	Talc	2-3%	
		Biotite	1%	
		Sulphides	1%	
		Serpentine	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	80%	Silicate mineralogy as for above. Sulphides are fine grained, monominerals, predominantly of pyrrhotite and chaclopyrite. They are included within orthopyroxene.
Northam Code	5A-5H	Olivine	10%	
Sample Type	PTS	Plagioclase	3%	
Sample Code	PGNR02	Chromite	3%	
Rock Type	Pegmatite	Talc	2-3%	
		Biotite	1%	
		Sulphides	1%	
		Serpentine	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	35%	Coarse (~2cm) grained pegmatite, consisting of orthopyroxene and olivine, with minor plagioclase. Secondary alteration is moderate to pervasive with replacement of orthopyroxene by talc; olivine by serpentine or magnetite; and plagioclase by sausserite. There are a few medium, widely dispersed, chromite grains
Northam Code	5A-5H	Olivine	35%	
Sample Type	TS	Plagioclase	10%	
Sample Code	PGNR03	Sulphides	5%	
Rock Type	Pegmatite (Harzburgite)	Serpentine	3-4%	
		Talc	3-4%	
		Sausscrite	2-3%	
		Chromite	1-2%	
		Biotite	1-2%	
		Clinopyroxene	~1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	35%	Silicate mineralogy as for above. Sulphides are predominantly medium to fine grained composites or fine grained monominerals consisting of pyrrhotite, pentlandite, chalcopyrite and minor cubanite. There is also a significant amount of very fine-grained, finely dispersed monomineralic sulphides within secondary (e.g. serpentine) minerals.
		Olivine	35%	
Northam Code	5A-5H	Plagioclase	10%	
		Biotite	5%	
		Sulphides (po, pent, ccp)	5%	
Sample Type	PTS	Chromite	2-4%	
		Serpentine	2-4%	
Sample Code	PGNR03	Clinopyroxene	1-2%	
		Sericite	1-2%	
Rock Type	Pegmatite	talc?	~1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	40%	Medium to Coarse grained pegmatite, consisting of orthopyroxene, clinopyroxene and olivine. Olivine is anhedral, occassionally included within orthopyroxene and occassionally interstitial. There is variable alteration of orthopyroxene to talc and of olivine to serpentine, with one fracture linking olivine grains across the slide.
		Olivine	25%	
Northam Code	5A-5H	Clinopyroxene	20%	
		Plagioclase	5%	
Sample Type	TS	Serpentine	2-3%	
		Talc	2-3%	
Sample Code	PGNR04	Biotite	2-3%	
		Chromite	1%	
Rock Type	Pegmatite	Sulphides	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	40%	Silicate mineralogy as for above. Sulphides are fine grained monominerals, predominantly of chalcopyrite and pyrrhotite, with a significant amount of very fine-grained sulphides within the secondary replacement minerals, especially serpentine.
		Olivine	25%	
Northam Code	5A-5H	Clinopyroxene	20%	
		Plagioclase	5%	
Sample Type	PTS	Serpentine	2-3%	
		Talc	2-3%	
Sample Code	PGNR04	Biotite	2-3%	
		Chromite	1%	
Rock Type	Pegmatite	Sulphides	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	70%	Medium to coarse orthopyroxene, enclosing plagioclase, olivine and biotite. Variable alteration of orthopyroxene to talc and of olivine to serpentine. Sulphides are medium to fine grained composites interstitial to the silicates, or fine grained inclusions within silicates. There are also a number of very fine grained sulphides within serpentine.
Northam Code	5A-5H	Plagioclase	10%	
Sample Type	PTS	Olivine	6%	
Sample Code	PGNR05	Biotite	6%	
Rock Type	Pegmatite	Talc	6%	
		Sulphides	1-2%	
		Chromite	1-2%	
		Serpentine	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	70%	Coarse (3.5 cm) orthopyroxene enclosing olivine, plagioclase and biotite. There is variable alteration of orthopyroxene to talc and of olivine to serpentine. Olivine is medium to coarse grained (5-6mm), while biotite is medium (2mm) and plagioclase is variable. Chromite is medium grained, and sparsely dispersed.
Northam Code	5A-5H	Olivine	10%	
Sample Type	TS	Plagioclase	8-10%	
Sample Code	PGNR06	Talc	7-8%	
Rock Type	Pegmatite	Biotite	2-3%	
		Sulphides	1%	
		Chromite	1%	
		Serpentine	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Orthopyroxene	70%	Silicate mineralogy as for above. Sulphides are fine grained composites or monominerals, with a minor association of very fine grained sulphides within serpentine and narrow microfractures.
Northam Code	5A-5H	Olivine	10%	
Sample Type	PTS	Plagioclase	8-10%	
Sample Code	PGNR06	Talc	7-8%	
Rock Type	Pegmatite	Biotite	2-3%	
		Sulphides	1%	
		Chromite	1%	
		Serpentine	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Plagioclase	80%	A medium grained euhedral to subhedral plagioclase adcumulate, with minor interstitial anhedral orthopyroxene and sulphides. Overlain by a medium grained chromite orthocumulate.
Northam Code	4X-6A	Orthopyroxene	10%	
Sample Type	TS	Chromite	7%	
Sample Code	BCNR01	Sulphides	1-2%	
Rock Type	Chromitite + Anorthosite	Sausserite	1-2%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Plagioclase	80%	Silicate mineralogy as for above. Sulphides are fine grained composites or monominerals, consisting primarily of pyrrhotite and chalcopyrite.
Northam Code	4X-6A	Orthopyroxene	10%	
Sample Type	PTS	Chromite	7%	
Sample Code	BCNR01	Sulphides	1-2%	
Rock Type	Pegmatite	Sausserite	1-2%	
		Clinopyroxene	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Plagioclase	60%	A medium grained euhedral to subhedral plagioclase adcumulate, with minor interstitial orthopyroxene. There is pervasive alteration of plagioclase to sausserite and of orthopyroxene to serpentine and talc, with macro-scale veins.
Northam Code	6A	Sausserite	30%	
Sample Type	TS	Orthopyroxene	5%	
Sample Code	ANNR01	Sulphides	2-4%	
Rock Type	Mottled Anorthosite	Talc	1%	
		Serpentine	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Plagioclase	60%	See Sample Description for TS for ANNR01
Northam Code	6A	Sausserite	30%	
Sample Type	TS	Orthopyroxene	5%	
Sample Code	ANNR01b	Sulphides	2-4%	
Rock Type	Mottled Anorthosite	Talc	1%	
		Serpentine	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Plagioclase	60%	Silicate mineralogy as for above. Sulphides are fine to very fine grained monominerals of pyrrhotite and chalcopyrite. They are exclusively associated with regions of pervasive alteration, suggesting remobilization during late stage magmatic processes.
Northam Code	6A	Saussarite	30%	
Sample Type	PTS	Orthopyroxene	5%	
Sample Code	ANNR01	Sulphides	3-4%	
Rock Type	Mottled Anorthosite	Talc	1%	
		Serpentine	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Plagioclase	85%	A medium grained euhedral to subhedral plagioclase orthocumulate, with minor interstitial orthopyroxene. There is moderate alteration of plagioclase to sericite and of orthopyroxene to serpentine and talc.
Northam Code	6A	Orthopyroxene	8%	
Sample Type	TS	Saussarite	5%	
Sample Code	ANNR02	Serpentine	1%	
Rock Type	Mottled Anorthosite	Sulphides	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	Normal	Plagioclase	85%	Silicate mineralogy as for above. Sulphides are fine grained composites or monominerals, consisting predominantly of chalcopyrite with associated pyrrhotite.
Northam Code	6A	Orthopyroxene	8%	
Sample Type	PTS	Saussurite	5%	
Sample Code	ANNR02	Serpentine	1%	
Rock Type	Mottled Anorthosite	Sulphides	1%	

C2 - NP2 Reef

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Orthopyroxene	70%	A medium grained (1-2 mm) subhedral orthopyroxene ortho to accumulate with interstitial anhedral plagioclase, biotite and sulphides. Very minor alteration of orthopyroxene to talc? Plagioclase to sericite and biotite to chlorite.
Northam Code	3A	Plagioclase	20%	
Sample Type	TS	Sulphides	5%	
Sample Code	HWNP01	Biotite	1-2%	
Rock Type	Melanorite	Talc	1-2%	
		Clinopyroxene	1-2%	
		Trace amounts of chlorite, amphibole, chromite and saussurite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Orthopyroxene	70%	Silicate Mineralogy as for above. Sulphide mineralogy consists of medium grained anhedral composites of pyrrhotite, granular pentlandite and chalcopyrite; interstitial to, and included within silicates. Also as monomineralic fine grained inclusions within silicates.
Northam Code	3A	Plagioclase	20%	
Sample Type	PTS	Sulphides	5%	
Sample Code	HWNP01	Biotite	1-2%	
Rock Type	Melanorite	Talc	1-2%	
		Clinopyroxene	1-2%	
		Trace amounts of chlorite, amphibole, chromite and saussurite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Orthopyroxene	80%	See Sample Description for TS for HWNP01
Northam Code	3A	Plagioclase	12%	
Sample Type	TS	Sulphides	4-5%	
Sample Code	HWNP02	Biotite	1-2%	
Rock Type	Melanorite	Talc	1-2%	
		Clinopyroxene	1-2%	
		Trace amounts of chlorite, amphibole, chromite and saussurite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Orthopyroxene	80%	See Sample Description for PTS for HWNP01
Northam Code	3A	Plagioclase	12%	
Sample Type	PTS	Sulphides	4-5%	
Sample Code	HWNP02	Biotite	1-2%	
Rock Type	Melanorite	Talc	1-2%	
		Clinopyroxene	1-2%	
		Trace amounts of chlorite, amphibole, chromite and saussurite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Chromite	70%	A variably annealed chromite adcumulate with coarse grained, orthopyroxene and plagioclase oikocrysts. Biotite and sulphides (pentlandite, pyrrhotite, chalcopyrite) are interstitial or included within chromite grains. There is moderate though variable alteration of plagioclase and orthopyroxene.
Northam Code	4X	Orthopyroxene	20%	
Sample Type	PTS	Plagioclase	5%	
Sample Code	CRNP02	Biotite	1-2%	
Rock Type	Chromitite	Saussurite	1-2%	
		Talc	1-2%	
		Sulphides	1-2%	

Appendix C

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase Chromite Sulphides Orthopyroxene Sausserite & Talc Biotite	80% 10% 5-6% 2-3% 2% <1%	A medium grained subhedral plagioclase ortho to adcumulate with interstitial sulphides and minor interstitial orthopyroxene. There is very some minor alteration of orthopyroxene to talc, and plagioclase to sausserite. This is overlain by a chromite adcumulate, containing adcumulous plagioclase
Northam Code	4A-8A			
Sample Type	TS			
Sample Code	CRNP01			
Rock Type	Chromitite + Anorthosite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase Chromite Sulphides Orthopyroxene Biotite	80% 10% 5-8% 1-2% <1%	Silicate mineralogy as for above. Sulphides are pyrrhotite, granular and minor flame pentlandite, chalcopyrite and pyrite. Pyrite is nearly always included within pentlandite. Sulphides are developed as medium grained anhedral composites interstitial to plagioclase, or fine grained monomineralic inclusions.
Northam Code	4A-8A			
Sample Type	PTS			
Sample Code	CRNP01			
Rock Type	Chromitite + Anorthosite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase Orthopyroxene Trace amounts of biotite, clinopyroxene, chromite and sulphides	95% 5% <1%	A medium grained subhedral plagioclase adcumulate with minor interstitial orthopyroxene.
Northam Code	8A			
Sample Type	TS			
Sample Code	CBANNP01			
Rock Type	Anorthosite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	95%	Silicate mineralogy as for above. Sulphides and chromites are fine grained monomineralic inclusions within silicates or interstitial to silicates. Sulphide mineralogy is predominantly, though not exclusively, chalcopyrite
Northam Code	8A	Orthopyroxene	5%	
Sample Type	PTS	Trace amounts of biotite, clinopyroxene, chromite, sausserite and sulphides	<1%	
Sample Code	CBANNP01			
Rock Type	Anorthosite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	85%	A medium grained subhedral plagioclase adcumulate with moderate interstitial orthopyroxene. In places orthopyroxene is becoming poikilitic.
Northam Code	8A	Orthopyroxene	15%	
Sample Type	TS	Chromite	<1%	
Sample Code	ANNP02	Trace amounts of biotite, clinopyroxene, sausserite and sulphides	<1%	
Rock Type	Anorthosite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	80%	Silicate mineralogy as for above. Sulphides and chromites are fine grained monomineralic inclusions within silicates or interstitial to silicates. Sulphide mineralogy is predominantly, though not exclusively, chalcopyrite
Northam Code	8A	Orthopyroxene	20%	
Sample Type	PTS	Chromite	1%	
Sample Code	ANNP02	Trace amounts of biotite, clinopyroxene, sausserite and sulphides	<1%	
Rock Type	Anorthosite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	45%	A plagioclase orthocumulate with large (~0.6mm), interstitial, anhedral olivine grains, partially or completely rimmed by orthopyroxene. Minor chromite and Very minor amounts of alteration minerals.
Northam Code	8T	Olivine	25%	
Sample Type	TS	Orthopyroxene	20%	
Sample Code	FWNP02	Sulphides	3-4%	
Rock Type	Troctolite	Chromite	2-3%	
		Biotite	2-3%	
		Trace amounts of clinopyroxene, talc, saussurite and serpentine	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	45%	Silicate mineralogy as for above. Sulphides are very similar to CRNP01, with medium grained composites of pyrrhotite, chalcopyrite, granular and minor flame pentlandite and pyrite. Pyrite is nearly always included within pentlandite. Fine-grained monomineralic sulphide inclusions are common.
Northam Code	8T	Olivine	25%	
Sample Type	PTS	Orthopyroxene	20%	
Sample Code	FWNP02	Sulphides	3-4%	
Rock Type	Troctolite	Chromite	2-3%	
		Biotite	2-3%	
		Trace amounts of clinopyroxene, talc, saussurite and serpentine	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	55%	A plagioclase orthocumulate, with interstitial, highly anhedral orthopyroxene, occasionally rimming olivine and chromite. Medium grained sulphide composites of pyrrhotite, granular and minor flame pentlandite, chalcopyrite and pyrite are also interstitial or included within silicates.
Northam Code	8T	Orthopyroxene	25%	
Sample Type	PTS	Olivine	10%	
Sample Code	FWNP02(b)	Chromite	5%	
Rock Type	Troctolite	Sulphides	3%	
		Biotite	2-3%	
		Trace amounts of clinopyroxene, talc, saussurite and serpentine	<1%	

Appendix C

Sample Summary		Minerals	Ratio	Sample Description
Reef Type Northam Code Sample Type Sample Code Rock Type	NP2 8T PTS FWNP04 Troctolite	Plagioclase Olivine Orthopyroxene Sulphides Biotite Chromite Trace amounts of clinopyroxene, talc, sausserite and serpentine	50% 30% 12% 4% 2% 1% 1%	A plagioclase orthocumulate, with large (5-8mm), poikilitic olivine, partially or completely rimmed by orthopyroxene. Medium grained (up to 1mm) sulphide composites of pyrrhotite, granular and minor flamed pentlandite, chalcopyrite and pyrite are mainly interstitial to silicates, but occasionally included.

Sample Summary		Minerals	Ratio	Sample Description
Reef Type Northam Code Sample Type Sample Code Rock Type	NP2 - TS TRNP01 Leuconorite	Plagioclase Orthopyroxene Trace amounts of biotite, clinopyroxene, sausserite, talc and sulphides	55% 40% ~5%	A medium grained, subhedral, plagioclase adcumulate with medium grained, anhedral, interstitial orthopyroxene. Alteration is very minor with clinopyroxene exsolution, and plagioclase alteration to sausserite.

Sample Summary		Minerals	Ratio	Sample Description
Reef Type Northam Code Sample Type Sample Code Rock Type	NP2 - PTS TRNP01 Leuconorite	Plagioclase Orthopyroxene Trace amounts of biotite, clinopyroxene, sausserite, talc and sulphides	55% 40% ~5%	Silicate Mineralogy as for above. Sulphides and chromites are fine grained. Chromite grains are mostly included within silicates. Sulphides are usually monomineralic and dominated by chalcopyrite, with minor pyrrhotite and pentlandite. They can be found interstitial to, or included within, the silicate minerals.

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	58%	See Sample Description for TS for TRNP01
		Orthopyroxene	40%	
Northam Code	-	Trace amounts of biotite, clinopyroxene, talc, biotite and sausserite	3-4%	
Sample Type	TS	Sulphides	1-2%	
Sample Code	TRNP02			
Rock Type	Leuconorite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	55%	Sulphides consist of fine to medium grained composites of pyrrhotite, pentlandite and chalcopyrite. Normally interstitial, and with occassional very fine grained monominerals. Also within this slide were three PGM! If this rock belongs to the norite footwall - this is very intriguing.
		Orthopyroxene	40%	
Northam Code	-	Trace amounts of biotite, clinopyroxene, talc, biotite and sausserite	3-4%	
Sample Type	PTS	Sulphides	1-2%	
Sample Code	TRNP02			
Rock Type	Leuconorite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	55%	See Sample Description for TS for TRNP01
		Orthopyroxene	40%	
Northam Code	-	Trace amounts of biotite, clinopyroxene, talc, biotite and sausserite	~5%	
Sample Type	TS			
Sample Code	TRNP03			
Rock Type	Leuconorite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	60%	See Sample Descript on for PTS for TRNP01
Northam Code	-	Orthopyroxene	40%	
Sample Type	PTS	Trace amounts of biotite, clinopyroxene, talc, biotite and sausserite	~5%	
Sample Code	TRNP03			
Rock Type	Leuconorite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	65%	See Sample Description for TS for TRNP01
Northam Code	-	Orthopyroxene	35%	
Sample Type	TS	Trace amounts of biotite, clinopyroxene, talc, biotite and sausserite	~5%	
Sample Code	FWNP01			
Rock Type	Leuconorite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	62%	Fine grained interstitial composites or monominerals, predominantly of chalcopyrite, but with accessory pentlandite and/or pyrrhotite.
Northam Code	-	Orthopyroxene	35%	
Sample Type	PTS	Trace amounts of biotite, clinopyroxene, talc, biotite and sausserite	~3%	
Sample Code	FWNP01			
Rock Type	Leuconorite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	67%	See Sample Description for TS for TRNP01
Northam Code	-	Orthopyroxene	30%	
Sample Type	TS	Trace amounts of biotite, clinopyroxene, talc, biotite and sausserite	~3%	
Sample Code	FWNP03			
Rock Type	Leuconorite			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	NP2	Plagioclase	67%	Fine grained composites and monominerals, dominated by chalcopyrite, but with accessory pentlandite, pyrrhotite and even some pyrite. The presence of pyrite suggests this might be a sulphide and olivine poor portion of the troctolite layer.
Northam Code	-	Orthopyroxene	30%	
Sample Type	PTS	Trace amounts of biotite, clinopyroxene, talc, biotite and sausserite	~3%	
Sample Code	FWNP03			
Rock Type	Leuconorite			

C3 – P2 Reef

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene	70%	A medium grained (1-2mm), orthopyroxene ortho to adcumulate, with interstitial plagioclase, sulphides and biotite. There is internal alteration of the orthopyroxene to a fine-grained silicate (probably talc), and minor clinopyroxene exsolution.
Northam Code	3A	Plagioclase	20%	
Sample Type	TS	Sulphides	3-4%	
Sample Code	HWP201	Talc	3%	
Rock Type	Melanorite	Biotite	2%	
		Clinopyroxene	1%	
		Chromite	1%	
		Sausserite	<0.5%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene	70%	Silicate mineralogy as for above. Sulphide mineralogy consists of medium grained, highly anhedral, interstitial composites of pyrrhotite, pentlandite and chalcopyrite. Occasional monomineralic sulphides are found included to the main silicate minerals.
Northam Code	3A	Plagioclase	20%	
Sample Type	PTS	Sulphides	3-4%	
Sample Code	HWP201	Talc	3%	
Rock Type	Melanorite	Biotite	2%	
		Clinopyroxene	1%	
		Chromite	1%	
		Sausserite	<0.5%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene	60%	See Sample Description for TS for HWP201. But with a clinopyroxene poikilocryst enveloping orthopyroxene grains within the centre of the slide.
Northam Code	3A	Clinopyroxene	10%	
Sample Type	TS	Plagioclase	10%	
Sample Code	HWP202	Sulphides	3-4%	
Rock Type	Melanorite	Talc	3%	
		Biotite	2%	
		Chromite	1%	
		Sericite	<0.5%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Chromite	65%	A variably annealed chromite orthocumulate to adcumulate, with coarse, poikilitic orthopyroxene and plagioclase grains. Sulphides, biotite and orthopyroxene are often included within chromite grains.
Northam Code	4A	Plagioclase	20%	
Sample Type	TS	Orthopyroxene	10%	
Sample Code	CRP201	Biotite	1-2%	
Rock Type	Chromitite	Talc	1-2%	
		Sausserite	1-2%	
		Sulphides	1-2%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Chromite	65%	Silicate mineralogy as for above. Sulphides are generally fine grained, composites of pyrrhotite, pentlandite and chalcopyrite, found interstitial to silicate and chromite grains Though they may also be found included within chromite grains.
Northam Code	4A	Plagioclase	20%	
Sample Type	PTS	Orthopyroxene	10%	
Sample Code	CRP201	Biotite	1-2%	
Rock Type	Chromitite	Talc	1-2%	
		Saussurite	1-2%	
		Sulphides	1-2%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene	40%	Orthopyroxene ortho to adcumulate with interstitinterstitial plagioclase, sulphides and biotite. There is internal alteration of the orthopyroxene to a fine-grained silicate (probably talc), and minor biotite. A clinopyroxene oikocryst envelops part of the melanorite. Sulphides are fine to medium grained composites of ccp, pn and po, with an abundance of very fine grained monominerals.
Northam Code	4A	Chromite	30%	
Sample Type	PTS	Plagioclase	12%	
Sample Code	CRP202	Clinopyroxene	8%	
Rock Type	Melanorite + Chromitite	Sulphides	5%	
		Talc	3%	
		Biotite	2%	
		Saussurite	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene	75%	A medium grained (1-2mm), orthopyroxene ortho to adcumulate, with interstitial plagioclase, sulphides and biotite. There is internal alteration of the orthopyroxene to a fine-grained silicate (probably talc), and minor clinopyroxene exsolution. Sulphides show extensive remobilisation, with po, ccp and pn in fine to medium grained composites and monominerals
Northam Code	4A	Plagioclase	10%	
Sample Type	PTS	Sulphides	5%	
Sample Code	CRP203	Chromite	4%	
Rock Type	Melanorite	Talc	3%	
		Biotite	2%	
		Saussurite	1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene Chromite Plagioclase Olivine Sulphides Talc Biotite Serpentine Saussersite	80% 4-5% 3-4% 3-4% 3-4% 3-4% 1-2% 1% <0.5%	An orthopyroxene megacryst (~3cm) enclosing anhedral olivine, sulphides, chromites and plagioclase. This megacryst lies just below the base of the Merensky chromitite. There is internal alteration of orthopyroxene to talc, and olivine to serpentine. The top of the slide contains a thin chromitite layer
Northam Code	5A			
Sample Type	TS			
Sample Code	PGP201			
Rock Type	Pegmatite (Pyroxenite)			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene Chromite Plagioclase Olivine Sulphides Talc Biotite Sericite Saussersite	80% 4-5% 3-4% 3-4% 3-4% 3-4% 1-2% 1% 1%	Silicate mineralogy as for above. Sulphide mineralogy consists of medium grained composites of pyrrhotite, granular and minor flamed pentlandite, and chalcopyrite. There is also abundant fine grained monomineralic inclusions within the orthopyroxene megacryst.
Northam Code	5A			
Sample Type	PTS			
Sample Code	PGP201			
Rock Type	Pegmatite (Pyroxenite)			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Plagioclase Orthopyroxene Sulphides Clinopyroxene Talc Biotite Saussersite	55% 30% 5% 5% 3-4% 1-2% 1-2%	Coarse grained (>1cm), highly anhedral, plagioclase and orthopyroxene grains. Orthopyroxene shows alteration to clinopyroxene, both as exsolution and discrete grains. Also moderate alteration to talc and biotite rims. Sulphides are present interstitial to orthopyroxene and plagioclase.
Northam Code	5A			
Sample Type	TS			
Sample Code	PGP202			
Rock Type	Pegmatite (Norite)			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Plagioclase	55%	Silicate mineralogy as for above. Sulphides are medium grained (up to 5mm), composites of pyrrhotite, pentlandite (mostly granular) and chalcopyrite. There is minor exsolution of cubanite from chalcopyrite and possibly mackinawite from pentlandite as well as extensive infilling of microfractures by fine-grained chalcopyrite.
Northam Code	5A	Orthopyroxene	30%	
Sample Type	PTS	Sulphides	5%	
Sample Code	PGP202	Clinopyroxene	5%	
Rock Type	Pegmatite (Norite)	Talc	3-4%	
		Biotite	1-2%	
		Saussurite	1-2%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene	60%	Orthopyroxene megacrysts (~2.5cm) enclosing medium grained, anhedral olivine grains, or olivine clusters. There is minor internal alteration of orthopyroxene to talc and biotite, and olivine to serpentine. Exsolution of clinopyroxene from orthopyroxene is also common
Northam Code	5A	Olivine	25%	
Sample Type	TS	Plagioclase	5-6%	
Sample Code	PGP203	Talc	5-6%	
Rock Type	Pegmatite (Harzburgite)	Clinopyroxene	1-2%	
		Biotite	1-2%	
		Sulphides	<1%	
		Serpentine	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene	60%	Silicate mineralogy as for above. Sulphides are rare, and only present as monomineralic grains interstitial to, or included within the major silicate phases.
Northam Code	5A	Olivine	25%	
Sample Type	PTS	Plagioclase	5-6%	
Sample Code	PGP203	Talc	5-6%	
Rock Type	Pegmatite (Harzburgite)	Clinopyroxene	1-2%	
		Biotite	1-2%	
		Sulphides	<1%	
		Serpentine	<1%	
		Magnetite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
		Orthopyroxene	40%	Coarse grained (<2cm) orthopyroxene and clinopyroxene enclosing medium grained olivine, chromite, plagioclase and sulphides. There is also some interstitial medium grained orthopyroxene. There is moderate alteration of pyroxene to talc or biotite, with minor alteration of olivine to serpentine
Reef Type	P2	Clinopyroxene	40%	
Northam Code	5A	Plagioclase	5%	
		Chromite	4-5%	
Sample Type	TS	Olivine	3-4%	
		Talc	3-4%	
Sample Code	PGP204	Sulphides	3%	
		Biotite	1-2%	
Rock Type	Pegmatite (Pyroxenite)	Serpentine	<1%	

Sample Summary		Minerals	Ratio	Sample Description
		Orthopyroxene	40%	Silicate mineralogy as for above. Sulphides consist of medium to fine grained composites of pyrrhotite, pentlandite and chalcopyrite. There is also moderate amounts of fine grained monomineralic sulphides included with the main silicate phases.
Reef Type	P2	Clinopyroxene	40%	
Northam Code	5A	Plagioclase	5%	
		Chromite	4-5%	
Sample Type	PTS	Olivine	3-4%	
		Talc	3-4%	
Sample Code	PGP204	Sulphides	3%	
		Biotite	1-2%	
Rock Type	Pegmatite (Pyroxenite)	Serpentine	<1%	

Sample Summary		Minerals	Ratio	Sample Description
		Orthopyroxene	70%	Medium to coarse grained orthopyroxene, enclosing anhedral, small to medium grained olivine. There is only minor alteration of orthopyroxene to talc; and of olivine to serpentine.
Reef Type	P2	Olivine	15%	
Northam Code	5A	Plagioclase	10%	
		Sulphides	2%	
Sample Type	TS	Talc	2%	
		Biotite	1%	
Sample Code	PGP205	Serpentine	1%	
		Clinopyroxene	<1%	
Rock Type	Pegmatite (Harzburgite)			

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene	70%	Silicate Mineralogy as above. Sulphides are medium (~2mm) to fine grained composites of pyrrhotite, chalcopyrite and pentlandite. Remobilisation of sulphides into microfractures is common, especially of chalcopyrite, and fine grained monomineralic sulphides are common close to medium grained composites.
Northam Code	5A	Olivine	15%	
Sample Type	PTS	Plagioclase	10%	
Sample Code	PGP205	Sulphides	2%	
Rock Type	Pegmatite (Harzburgite)	Talc	2%	
		Biotite	1%	
		Serpentine	1%	
		Clinopyroxene	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene	65%	Coarse (~2cm) orthopyroxene grains enclosing medium grained (~2mm) anhedral olivine grains. There is some minor alteration of olivine to serpentine and magnetite, as well as minor alteration of orthopyroxene to talc and biotite.
Northam Code	5A	Olivine	30%	
Sample Type	TS	Plagioclase	1-2%	
Sample Code	PGP206	Serpentine	1-2%	
Rock Type	Pegmatite (Harzburgite)	Talc	1%	
		Magnetite	1%	
		Trace amounts of sulphides, clinopyroxene chromite and biotite	<1%	

Sample Summary		Minerals	Ratio	Sample Description
Reef Type	P2	Orthopyroxene	65%	Silicate mineralogy as for above. Sulphides are fine grained composites of pyrrhotite, pentlandite and chalcopyrite, or monomineralic chalcopyrite grains. They are usually included within the major silicate phases.
Northam Code	5A	Olivine	30%	
Sample Type	PTS	Plagioclase	1-2%	
Sample Code	PGP206	Serpentine	1-2%	
Rock Type	Pegmatite (Harzburgite)	Talc	1%	
		Trace amounts of sulphides, magneite chromite and biotite	<1%	

Appendix D - QEMSCAN Data

D1 – Normal Reef

D1.1 – Element Assay

	Normal Reef: Standard Grind				Normal Reef: Fine Grind			
Size Fraction		-38/+0	-75/+38	-3000/+75		-38/+0	-75/+38	-3000/+75
Min Size (mm)		0.0	38.0	75.0		0.0	38.0	75.0
Max Size (mm)		38.0	75.0	3000.0		38.0	75.0	3000.0
Particle Size (mm)		11.8	40.5	77.7		12.0	39.5	72.2
Mass Flow Distribution (%)		27.93	23.47	48.60		34.77	31.20	34.03
Element Chemical Assay (%)	Combined	-38/+0	-75/+38	-3000/+75	Combined	-38/+0	-75/+38	-3000/+75
O	44.20	43.02	44.08	44.94	44.19	43.06	44.36	45.18
Na	0.28	0.29	0.31	0.26	0.28	0.28	0.33	0.23
Mg	14.07	13.16	13.69	14.77	13.94	13.21	13.89	14.75
Al	3.56	3.77	3.95	3.24	3.44	3.67	3.81	2.87
Si	21.48	21.16	21.13	21.84	21.57	21.22	21.45	22.04
S	0.51	1.03	0.46	0.23	0.51	0.90	0.35	0.25
K	0.12	0.13	0.13	0.10	0.12	0.12	0.14	0.10
Ca	2.77	3.02	3.02	2.50	2.69	2.97	2.88	2.23
Ti	0.11	0.10	0.12	0.10	0.09	0.10	0.09	0.09
Cr	1.50	1.70	1.84	1.23	1.59	1.83	1.65	1.30
Mn	0.01	0.01	0.00	0.01	0.01	0.01	0.01	0.01
Fe	9.44	9.86	9.40	9.21	9.48	9.91	9.23	9.27
Co	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ni	0.26	0.42	0.28	0.16	0.26	0.34	0.22	0.21
Cu	0.08	0.19	0.09	0.02	0.08	0.17	0.05	0.03

D1.2 – Mineral Mass in Size Fraction

	Normal Reef: Standard Grind				Normal Reef: Fine Grind			
Mineral Mass % in Fraction	Combined	-38/-0	-75/+38	-3000/-75	Combined	-38/+0	-75/-38	-3000/+75
Orthopyroxene	50.16	46.17	46.75	54.11	52.33	47.02	49.02	60.81
Clinopyroxene	4.40	5.77	4.01	3.80	4.42	5.86	3.73	3.59
Plagioclase	17.04	17.01	19.55	15.83	16.42	16.62	19.02	13.83
Olivine	18.39	16.33	19.46	19.07	16.28	15.97	18.60	14.47
Mica	1.76	2.50	1.80	1.32	1.84	2.29	1.81	1.40
Quartz	0.68	0.85	0.46	0.68	0.68	0.84	0.59	0.61
Amphibole	0.38	1.07	0.19	0.07	0.53	1.06	0.36	0.14
Serpentine	0.29	0.59	0.38	0.08	0.37	0.65	0.41	0.06
Talc	0.25	0.46	0.25	0.13	0.28	0.52	0.22	0.10
Chlorite	0.61	0.76	0.43	0.61	0.53	0.68	0.42	0.48
Other Silicates	0.49	0.52	0.41	0.52	0.43	0.53	0.35	0.39
Chromite	3.25	3.66	3.97	2.67	3.43	3.93	3.57	2.79
Magnetite	0.49	0.94	0.54	0.20	0.61	1.05	0.44	0.33
Pentlandite	0.54	1.01	0.62	0.22	0.56	0.80	0.44	0.43
Pyrrhotite	0.47	0.79	0.36	0.33	0.49	0.76	0.40	0.30
Chalcopyrite	0.24	0.54	0.27	0.06	0.24	0.47	0.16	0.08
Pyrite	0.10	0.21	0.04	0.06	0.08	0.19	0.04	0.00
Other Sulphides	0.16	0.31	0.13	0.08	0.12	0.27	0.06	0.03
Other	0.31	0.52	0.37	0.15	0.34	0.50	0.37	0.16

D1.3 – Grain and Particle Size

	Normal Reef: Standard Grind				Normal Reef: Fine Grind			
Mineral Particle Size (µm)	Combined	-38/+0	-75/+38	-3000/+75	Combined	-38/+0	-75/+38	-3000/+75
Particle size	28.12	11.83	40.53	77.67	24.09	11.99	39.45	72.21
Orthopyroxene	20.37	8.03	30.33	53.20	18.18	8.18	30.26	55.32
Clinopyroxene	10.30	5.67	17.05	20.99	8.62	5.29	17.84	19.41
Plagioclase	21.86	9.05	36.52	59.84	17.93	8.74	37.31	49.26
Olivine	16.45	6.67	25.71	35.86	13.48	6.61	27.14	31.72
Mica	4.56	2.82	7.94	7.74	4.27	2.68	8.40	7.50
Quartz	10.31	5.31	17.36	22.23	8.88	5.04	21.79	19.56
Amphibole	1.98	1.77	3.63	3.36	2.09	1.78	3.35	3.84
Serpentine	2.21	1.84	3.33	2.35	2.20	1.83	3.32	2.43
Talc	2.42	1.90	3.65	3.22	2.34	1.94	3.68	3.51
Chlorite	4.46	2.75	5.96	6.95	4.04	2.77	6.43	6.38
Other Silicates	5.90	3.14	8.54	8.91	4.26	3.01	6.47	5.92
Chromite	16.56	7.27	30.15	52.27	13.70	7.23	32.68	34.90
Magnetite	4.13	2.97	9.90	5.75	4.08	2.94	9.31	9.35
Pentlandite	8.71	7.09	16.01	8.97	8.44	6.04	13.50	16.39
Pyrrhotite	7.32	4.95	13.84	12.40	7.84	5.12	19.95	22.38
Chalcopyrite	7.93	6.77	22.03	5.27	8.50	7.08	16.91	11.76
Pyrite	2.71	2.26	3.86	3.95	2.49	2.30	4.30	2.94
Other Sulphides	2.62	1.97	4.99	3.56	2.19	1.88	3.81	3.71
Other	3.19	2.48	4.45	4.23	3.06	2.40	4.82	3.44

D2 – NP2 Reef

D2.1 - Element Assay

	NP2 Reef: Standard Grind				NP2 Reef: Fine Grind			
Size Fraction		-38/+0	-75/+38	-3000/+75		-38/+0	-75/+38	-3000/+75
Min Size (mm)		0.0	38.0	75.0		0.0	38.0	75.0
Max Size (mm)		38.0	75.0	3000.0		38.0	75.0	3000.0
Particle Size (mm)		12.2	44.6	74.6		11.4	42.7	70.9
Mass Flow Distribution (%)		28.51	30.25	41.24		38.94	40.42	20.64
Element Chemical Assay (%)	Combined	-38/+0	-75/+38	-3000/+75	Combined	-38/+0	-75/+38	-3000/+75
O	45.14	44.46	45.22	45.56	45.35	44.72	45.57	46.10
Na	0.86	0.93	0.90	0.79	0.96	0.94	1.02	0.87
Mg	5.35	4.37	5.06	6.24	5.13	4.36	4.94	6.96
Al	11.46	12.14	11.76	10.76	11.71	12.25	12.04	10.04
Si	21.55	21.34	21.51	21.71	21.79	21.40	21.91	22.27
S	0.53	0.93	0.41	0.33	0.47	0.83	0.29	0.11
K	0.03	0.04	0.03	0.02	0.03	0.04	0.03	0.02
Ca	8.06	8.37	8.31	7.66	8.12	8.40	8.38	7.05
Ti	0.08	0.08	0.10	0.08	0.07	0.06	0.07	0.07
Cr	1.37	1.32	1.41	1.37	1.16	1.31	1.03	1.12
Mn	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Fe	4.40	4.34	4.18	4.61	4.06	4.14	3.73	4.57
Co	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ni	0.17	0.31	0.13	0.10	0.16	0.26	0.11	0.05
Cu	0.09	0.19	0.08	0.03	0.08	0.15	0.04	0.03

D2.2 - Mineral Mass in Size Fraction

	NP2 Reef: Standard Grind				NP2 Reef: Fine Grind			
Mineral Mass % in Fraction	Combined	-38/+0	-75/+38	-3000/+75	Combined	-38/+0	-75/+38	-3000/+75
Orthopyroxene	25.95	19.61	24.51	31.39	24.80	19.67	23.81	36.39
Clinopyroxene	3.33	3.50	3.38	3.19	3.10	3.22	3.01	3.07
Plagioclase	61.81	64.31	63.94	58.51	63.45	64.98	66.19	55.18
Olivine	1.80	1.73	1.77	1.88	1.65	1.69	1.80	1.29
Mica	0.94	2.13	0.61	0.36	1.18	2.23	0.56	0.41
Quartz	0.33	0.66	0.29	0.14	0.40	0.60	0.32	0.17
Amphibole	0.22	0.66	0.06	0.02	0.31	0.59	0.14	0.10
Serpentine	0.17	0.36	0.13	0.07	0.26	0.43	0.20	0.08
Talc	0.09	0.21	0.06	0.03	0.14	0.24	0.09	0.05
Chlorite	0.08	0.18	0.08	0.02	0.08	0.14	0.05	0.02
Other Silicates	0.18	0.27	0.17	0.14	0.16	0.24	0.11	0.11
Chromite	2.95	2.84	3.04	2.95	2.50	2.82	2.23	2.41
Magnetite	0.27	0.54	0.28	0.07	0.29	0.52	0.17	0.10
Pentlandite	0.51	0.85	0.41	0.34	0.45	0.72	0.35	0.16
Pyrrhotite	0.55	0.67	0.42	0.57	0.40	0.52	0.41	0.15
Chalcopyrite	0.25	0.52	0.24	0.08	0.22	0.42	0.10	0.09
Pyrite	0.10	0.20	0.10	0.04	0.11	0.23	0.03	0.02
Other Sulphides	0.10	0.27	0.07	0.01	0.16	0.33	0.08	0.01
Other	0.35	0.48	0.43	0.20	0.36	0.44	0.37	0.17

D2.3 - Grain and Particle Size

	NP2 Reef: Standard Grind				NP2 Reef: Fine Grind			
Mineral Particle Size (μm)	Combined	-38/+0	-75/+38	-3000/+75	Combined	-38/+0	-75/+38	-3000/+75
Particle size	27.93	12.19	44.56	74.61	21.54	11.45	42.74	70.93
Orthopyroxene	26.78	9.83	38.51	62.21	21.03	9.53	38.06	61.11
Clinopyroxene	15.97	7.36	27.74	36.31	13.52	7.51	27.85	33.06
Plagioclase	24.92	10.80	43.59	71.61	19.49	10.31	42.15	67.26
Olivine	10.30	4.86	14.18	21.19	8.87	4.68	21.85	20.45
Mica	2.49	1.99	5.28	4.42	2.35	2.01	4.63	4.89
Quartz	6.80	4.72	21.04	11.69	6.76	4.77	17.72	13.84
Amphibole	1.90	1.81	3.12	2.47	1.97	1.75	3.44	2.64
Serpentine	2.17	1.88	3.13	2.46	2.21	1.87	3.41	2.57
Talc	2.10	1.82	3.26	2.59	2.15	1.85	3.36	2.58
Chlorite	3.14	2.49	5.85	3.97	2.81	2.37	5.59	2.98
Other Silicates	5.93	3.76	10.72	8.57	4.96	4.01	7.60	6.45
Chromite	14.46	6.85	21.90	27.98	11.49	6.68	24.37	31.21
Magnetite	3.23	2.34	8.24	4.00	2.58	2.03	6.97	5.19
Pentlandite	9.36	8.24	14.42	8.93	8.14	7.39	11.86	5.68
Pyrrhotite	15.76	7.48	31.27	43.04	9.97	6.11	30.91	19.40
Chalcopyrite	12.23	8.66	28.93	28.30	9.21	7.90	15.53	23.70
Pyrite	4.88	3.68	11.97	4.92	4.16	4.05	5.29	3.97
Other Sulphides	2.14	1.87	3.83	2.21	2.24	1.86	7.45	3.86
Other	2.85	2.29	3.85	2.84	2.86	2.12	4.33	3.45

D3 - P2 Reef

D3.1 Element Assay

	P2 Reef: Standard Grind				P2 Reef: Fine Grind			
Size Fraction		-38/-0	-75/+38	-3000/+75		-38/+0	-75/+38	-3000/+75
Min Size (mm)		0.0	38.0	75.0		0.0	38.0	75.0
Max Size (mm)		38.0	75.0	3000.0		38.0	75.0	3000.0
Particle Size (mm)		14.2	40.0	74.5		11.6	42.7	71.8
Mass Flow Distribution (%)		28.44	26.77	44.79		33.91	31.92	34.17
Element Chemical Assay (%)	Combined	-38/+0	-75/+38	-3000/+75	Combined	-38/+0	-75/+38	-3000/+75
O	43.82	40.92	44.32	45.36	44.58	43.03	44.76	45.95
Na	0.16	0.17	0.19	0.13	0.15	0.17	0.18	0.12
Mg	14.65	13.28	14.70	15.50	14.97	14.04	14.87	15.97
Al	1.66	1.92	1.94	1.33	1.63	1.95	1.85	1.10
Si	21.64	20.57	21.72	22.27	22.02	21.30	21.95	22.79
S	0.88	1.76	0.70	0.42	0.72	1.52	0.40	0.22
K	0.08	0.10	0.06	0.08	0.05	0.08	0.05	0.02
Ca	1.71	2.03	1.95	1.37	1.70	1.97	1.85	1.29
Ti	0.10	0.12	0.12	0.09	0.10	0.11	0.12	0.09
Cr	1.84	1.70	2.11	1.76	1.76	2.08	2.12	1.10
Mn	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Fe	9.96	10.27	9.92	9.78	9.94	10.58	9.68	9.53
Co	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Ni	0.34	0.57	0.37	0.18	0.35	0.58	0.29	0.18
Cu	0.92	3.07	0.10	0.04	0.12	0.27	0.05	0.02

D3.2 – Mineral Mass in Size Fraction

	P2 Reef: Standard Grind				P2 Reef: Fine Grind			
Mineral Mass % in Fraction	Combined	-38/+0	-75/+38	-3000/+75	Combined	-38/+0	-75/+38	-3000/+75
Orthopyroxene	67.78	58.14	65.54	75.23	69.40	61.58	67.49	78.96
Clinopyroxene	5.47	6.81	5.75	4.45	5.39	6.00	5.42	4.76
Plagioclase	7.17	7.55	8.88	5.91	7.15	8.19	8.57	4.80
Olivine	8.56	8.48	10.37	7.52	8.76	9.44	9.82	7.12
Mica	1.05	1.54	0.74	0.93	0.74	1.34	0.66	0.22
Quartz	0.45	1.02	0.37	0.13	0.33	0.61	0.28	0.10
Amphibole	0.51	1.21	0.35	0.16	0.58	1.15	0.38	0.19
Serpentine	0.31	0.63	0.34	0.10	0.45	0.73	0.47	0.15
Talc	0.26	0.49	0.25	0.11	0.38	0.66	0.33	0.14
Chlorite	0.14	0.28	0.14	0.06	0.11	0.22	0.09	0.03
Other Silicates	0.10	0.20	0.10	0.04	0.07	0.12	0.07	0.02
Chromite	3.96	3.67	4.54	3.79	3.79	4.48	4.57	2.37
Magnetite	0.30	0.72	0.21	0.08	0.39	0.75	0.27	0.14
Pentlandite	0.87	1.55	0.92	0.41	0.89	1.52	0.70	0.45
Pyrrhotite	0.87	1.37	0.66	0.68	0.56	1.06	0.29	0.31
Chalcopyrite	1.19	3.67	0.32	0.14	0.35	0.82	0.17	0.04
Pyrite	0.17	0.47	0.08	0.03	0.13	0.31	0.05	0.02
Other Sulphides	0.18	0.52	0.06	0.02	0.18	0.48	0.04	0.01
Other	0.67	1.67	0.38	0.20	0.35	0.54	0.34	0.16

D3.3 – Grain and Particle Size

	P2 Reef: Standard Grind				P2 Reef: Fine Grind			
Mineral Particle Size (µm)	Combined	-38/+0	-75/+38	-3000/+75	Combined	-38/+0	-75/+38	-3000/+75
Particle size	30.71	14.23	39.99	74.46	24.25	11.63	42.72	71.75
Orthopyroxene	24.75	9.92	33.81	60.84	21.11	9.38	36.39	57.64
Clinopyroxene	10.73	5.70	20.27	21.16	12.29	7.32	21.41	20.42
Plagioclase	19.80	8.49	36.06	60.85	17.39	9.03	37.76	52.02
Olivine	14.05	6.05	24.58	33.98	12.19	6.25	25.06	29.27
Mica	4.94	2.88	8.06	11.64	3.76	2.87	7.98	6.66
Quartz	6.12	4.75	16.81	10.03	5.94	4.52	15.34	8.93
Amphibole	2.13	1.84	3.30	2.96	2.11	1.84	3.35	2.66
Serpentine	2.16	1.83	3.15	2.41	2.18	1.78	3.25	2.46
Talc	2.34	1.90	3.54	2.83	2.33	1.90	3.68	3.04
Chlorite	2.86	2.14	6.27	3.91	2.72	2.32	4.67	3.11
Other Silicates	4.10	3.04	8.78	5.73	3.81	2.98	6.81	4.28
Chromite	15.66	6.72	27.71	31.65	12.29	6.76	28.20	25.39
Magnetite	3.48	2.90	6.46	5.66	3.13	2.48	7.06	4.92
Pentlandite	10.33	7.20	24.41	16.35	9.28	7.42	18.65	11.82
Pyrrhotite	7.53	4.09	19.93	26.13	6.36	4.97	12.07	13.51
Chalcopyrite	12.30	11.96	18.23	12.74	6.72	6.13	13.41	6.62
Pyrite	2.53	2.27	5.85	3.36	2.43	2.21	4.57	3.39
Other Sulphides	2.13	2.02	3.45	2.48	1.98	1.87	3.98	2.84
Other	3.30	3.15	4.53	3.14	2.66	2.17	3.99	2.91

Appendix E – Liberation Data

	Chalcopyrite				Pyrrhotite			
	Liberated (>90%)	High Grade Middlings (60-90%)	Low Grade Middlings (30-60%)	Locked (<30%)	Liberated (>90%)	High Grade Middlings (60-90%)	Low Grade Middlings (30-60%)	Locked (<30%)
P2 Reef Fine: -3000/+75	0.35	1.13	0.47	3.12	21.46	2.54	0.05	1.35
P2 Reef Fine: -75/+38	23.06	10.10	1.65	6.96	29.00	5.38	0.51	2.76
P2 Reef Fine: -38/+0	28.17	20.48	2.60	1.91	26.97	4.20	1.15	4.62
Cumulative	51.59	31.70	4.72	11.99	77.43	12.12	1.71	8.74
P2 Reef Standard: -3000/+75	5.47	3.70	1.71	9.55	30.96	3.17	0.54	1.79
P2 Reef Standard: -75/+38	25.01	10.60	2.18	3.30	28.97	3.01	1.27	2.04
P2 Reef Standard: -38/+0	19.54	15.68	1.83	1.43	20.32	3.60	1.13	3.19
Cumulative	50.02	29.98	5.71	14.29	80.25	9.78	2.94	7.03
NP2 Reef Fine: -3000/+75	6.89	0.07	0.00	2.83	7.79	0.25	0.00	0.77
NP2 Reef Fine: -75/+38	27.43	1.03	0.00	7.93	59.64	3.29	0.00	1.32
NP2 Reef Fine: -38/+0	37.39	12.44	2.21	1.79	16.18	9.07	0.95	0.74
Cumulative	71.71	13.54	2.21	12.55	83.61	12.61	0.95	2.83
NP2 Reef Standard: -3000/+75	19.95	0.00	1.06	0.92	40.36	1.15	0.00	0.44
NP2 Reef Standard: -75/+38	29.23	6.81	3.99	4.49	33.65	2.77	1.83	1.16
NP2 Reef Standard: -38/+0	22.98	8.42	1.12	1.03	12.74	4.55	0.56	0.78
Cumulative	72.16	15.23	6.17	6.44	86.76	8.47	2.39	2.37
NR Reef Fine: -3000/+75	5.76	0.43	0.03	3.26	17.26	2.45	0.15	2.31
NR Reef Fine: -75/+38	21.57	5.92	5.37	4.98	28.85	5.70	2.08	2.25
NR Reef Fine: -38/+0	31.25	16.39	2.63	2.42	28.09	6.17	1.13	3.55
Cumulative	58.59	22.74	8.02	10.66	74.21	14.32	3.36	8.11
Normal Reef Standard: -3000/+75	5.32	0.00	5.24	7.36	28.61	4.23	1.73	5.06
Normal Reef Standard: -75/+38	16.64	3.68	5.40	3.85	15.92	5.05	1.59	3.11
Normal Reef Standard: -38/+0	29.33	14.64	5.81	2.71	22.96	3.76	3.14	4.85
Cumulative	51.30	18.33	16.45	13.92	67.49	13.05	6.45	13.01

	PENTLANDITE				BASE METAL SULPHIDES			
	Liberated (>90%)	High Grade Middlings (60-90%)	Low Grade Middlings (30-60%)	Locked (<30%)	Liberated (>90%)	High Grade Middlings (60-90%)	Low Grade Middlings (30-60%)	Locked (<30%)
P2 Reef Fine: -3000/+75	5.88	0.00	0.17	0.85	11.00	0.30	0.38	1.21
P2 Reef Fine: -75/+38	42.39	2.17	0.48	1.72	39.12	1.36	0.73	1.58
P2 Reef Fine: -38/+0	36.07	7.52	1.56	1.19	39.77	2.62	0.91	1.00
Cumulative	*84.34*	*9.69*	*2.21*	*3.76*	*89.89*	*4.28*	*2.03*	*3.80*
P2 Reef Fine: -3000/+75	10.02	0.00	0.64	1.44	19.35	1.34	0.12	2.50
P2 Reef Fine: -75/+38	40.70	2.13	0.42	0.98	37.84	1.07	0.23	1.21
P2 Reef Fine: -38/+0	33.28	8.45	0.91	1.02	33.18	1.83	0.57	0.75
Cumulative	*83.99*	*10.59*	*1.97*	*3.45*	*90.37*	*4.24*	*0.92*	*4.47*
NP2 Reef Fine: -3000/+75	0.00	2.78	0.00	0.87	5.91	0.00	0.00	1.01
NP2 Reef Fine: -75/+38	44.77	4.50	1.12	2.19	52.84	0.19	1.32	1.24
NP2 Reef Fine: -38/+0	31.51	8.49	0.95	2.81	32.72	3.14	0.47	1.18
Cumulative	*76.28*	*15.77*	*2.07*	*5.88*	*91.46*	*3.33*	*1.78*	*3.42*
NP2 Reef Fine: -3000/+75	6.52	0.00	0.00	1.78	26.12	0.00	0.25	0.69
NP2 Reef Fine: -75/+38	38.56	2.81	0.97	3.84	38.85	2.99	0.60	1.22
NP2 Reef Fine: -38/+0	34.50	8.05	1.23	1.76	25.54	2.71	0.43	0.60
Cumulative	*79.58*	*10.86*	*2.19*	*7.37*	*90.51*	*5.70*	*1.28*	*2.51*
Normal Reef Fine: -3000/+75	9.88	0.00	0.13	1.41	11.94	1.13	0.06	2.55
Normal Reef Fine: -75/+38	26.86	3.89	0.86	2.41	32.66	1.25	0.80	1.79
Normal Reef Fine: -38/+0	39.64	10.48	2.02	2.41	41.93	3.14	1.06	1.67
Cumulative	*76.38*	*14.38*	*3.01*	*6.23*	*86.54*	*5.53*	*1.92*	*6.01*
Normal Reef Fine: -3000/+75	4.83	2.77	0.00	2.17	16.60	0.27	2.66	3.92
Normal Reef Fine: -75/+38	23.75	3.97	2.01	1.84	25.50	0.89	0.59	1.99
Normal Reef Fine: -38/+0	43.07	10.18	2.41	3.00	40.60	3.18	1.68	2.12
Cumulative	*71.64*	*16.93*	*4.42*	*7.01*	*82.70*	*4.34*	*4.93*	*8.02*

Appendix F – Analysis of Variance

F1 – Normal Reef

Normal Reef	Total Mass										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	54.0	109.8	244.4	493.4	-	-					
a	55.8	134.6	249.0	-11.8	-3.0	17.4	1.0	17.4	2.3	0.2	**79.2**
b	68.2	114.1	0.0	45.6	11.4	259.9	1.0	259.9	33.7	0.0	**99.6**
ab	66.4	134.9	-11.8	1.2	0.3	0.2	1.0	2.6	0.3	0.6	**41.0**
c	61.2	1.8	24.8	4.6	1.1	2.6	1.0	0.2	0.0	0.9	**11.4**
ac	52.9	-1.8	20.8	-11.8	-3.0	17.4	1.0	17.4	2.3	0.2	**79.2**
bc	69.2	-8.3	-3.6	-4.0	-1.0	2.0	1.0	2.0	0.3	0.6	**36.2**
abc	65.7	-3.5	4.8	8.4	2.1	8.8	1.0	8.8	1.1	0.3	**65.5**

Error MS	7.72
Error SD	2.78

Normal Reef	Chalcopyrite										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	81.9	167.4	338.2	678.2	-	-					
a	85.5	170.8	340.0	2.2	0.6	0.6	1.0	0.6	0.4	0.6	**43.9**
b	85.9	170.1	2.6	3.2	0.8	1.3	1.0	1.3	0.8	0.4	**59.1**
ab	84.9	169.9	-0.4	-4.4	-1.1	2.4	1.0	0.4	0.3	0.6	**36.8**
c	85.2	3.6	3.4	1.8	0.4	0.4	1.0	2.4	1.6	0.3	**72.6**
ac	84.9	-1.0	-0.2	-3.0	-0.7	1.1	1.0	1.1	0.7	0.4	**56.4**
bc	85.0	-0.3	-4.6	-3.6	-0.9	1.6	1.0	1.6	1.1	0.4	**64.2**
abc	84.9	0.1	0.2	4.8	1.2	2.9	1.0	2.9	1.9	0.2	**76.1**

Error MS	1.51
Error SD	1.23

Appendix F

Normal Reef	Pentlandite										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	69.9	155.7	301.9	603.5	-	-					
a	85.8	146.2	301.6	30.5	7.6	116.3	1.0	116.3	18.3	0.0	**98.7**
b	72.4	157.8	17.3	-23.5	-5.9	69.0	1.0	69.0	10.9	0.0	**97.0**
ab	73.8	143.8	13.2	-16.1	-4.0	32.4	1.0	0.0	0.0	1.0	**3.2**
c	75.2	15.9	-9.5	-0.3	-0.1	0.0	1.0	32.4	5.1	0.1	**91.3**
ac	82.6	1.4	-14.0	-4.1	-1.0	2.1	1.0	2.1	0.3	0.6	**40.4**
bc	69.0	7.4	-14.5	-4.5	-1.1	2.5	1.0	2.5	0.4	0.6	**43.8**
abc	74.8	5.8	-1.6	12.9	3.2	20.8	1.0	20.8	3.3	0.1	**85.5**

Error MS	6.36
Error SD	2.52

Normal Reef	Pyrrhotite										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	42.0	90.6	201.0	413.2	-	-					
a	48.6	110.5	212.2	72.0	18.0	648.7	1.0	648.7	63.5	0.0	99.9
b	41.2	90.7	34.7	50.7	12.7	321.1	1.0	321.1	33.9	0.0	99.6
ab	69.3	121.5	37.3	35.6	8.9	158.7	1.0	15.5	1.6	0.3	73.0
c	39.6	6.6	19.9	11.1	2.8	15.5	1.0	158.7	16.8	0.0	98.5
ac	51.1	28.1	30.8	2.6	0.7	0.9	1.0	0.9	0.1	0.8	22.3
bc	47.9	11.6	21.5	10.9	2.7	14.9	1.0	14.9	1.6	0.3	72.2
abc	73.6	25.8	14.2	-7.3	-1.8	6.6	1.0	6.6	0.7	0.4	55.0

Error MS	9.46
Error SD	3.08

Normal Reef	BMS										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	72.8	155.7	316.0	625.4	-	-					
A	82.9	160.3	309.4	41.0	10.3	210.1	1.0	210.1	85.9	0.0	99.9
B	75.8	150.7	18.8	12.6	3.2	19.8	1.0	19.8	8.1	0.0	95.4
ab	84.5	158.7	22.2	0.6	0.1	0.0	1.0	5.4	2.2	0.2	79.0
C	70.3	10.1	4.6	-6.6	-1.7	5.4	1.0	0.0	0.0	0.9	10.1
ac	80.4	8.7	8.0	3.4	0.9	1.4	1.0	1.4	0.6	0.5	51.5
bc	73.3	10.1	-1.4	3.4	0.8	1.4	1.0	1.4	0.6	0.5	51.5
abc	85.4	12.1	2.0	3.4	0.9	1.4	1.0	1.4	0.6	0.5	51.5

Error MS	2.45
Error SD	1.56

F2 – NP2 Reef

NP2 Reef	Total Mass										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	28.9	63.4	125.6	255.0	-	-					
A	34.5	62.2	129.4	14.6	3.7	26.6	1.0	26.6	46.3	0.0	99.8
B	29.9	65.4	8.0	-2.6	-0.7	0.8	1.0	0.8	1.5	0.3	70.8
ab	32.3	64.0	6.6	-5.0	-1.3	3.1	1.0	1.8	3.1	0.2	84.9
C	30.6	5.6	-1.2	3.8	1.0	1.8	1.0	3.1	5.4	0.1	92.0
ac	34.8	2.4	-1.4	-1.4	-0.4	0.2	1.0	0.2	0.4	0.5	45.0
bc	30.8	4.2	-3.2	-0.2	-0.1	0.0	1.0	0.0	0.0	0.9	7.0
abc	33.2	2.4	-1.8	1.4	0.4	0.2	1.0	0.2	0.4	0.5	45.0

Error MS	0.58
Error SD	0.76

Appendix F

NP2 Reef	Chalcopyrite											
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	89.6	179.6	358.3	716.8	-	-					
A	90.0	178.7	358.5	-6.2	-1.5	4.8	1.0	4.8	10.4	0.0	96.8
B	90.3	180.5	-1.5	-3.4	-0.9	1.4	1.0	1.4	3.1	0.2	84.8
ab	88.4	178.0	-4.7	-6.0	1.5	4.5	1.0	0.0	0.0	0.9	7.8
C	90.5	0.4	-0.9	0.2	0.1	0.0	1.0	4.5	9.7	0.0	96.4
ac	90.0	-1.9	-2.5	-3.2	-0.8	1.3	1.0	1.3	2.8	0.2	82.8
bc	91.1	-0.5	-2.3	-1.6	-0.4	0.3	1.0	0.3	0.7	0.5	54.8
abc	86.9	-4.2	-3.7	-1.4	-0.3	0.2	1.0	0.2	0.5	0.5	49.3

Error MS	0.46
Error SD	0.68

NP2 Reef	Pentlandite										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	83.4	158.8	317.5	638.4	-	-					
A	75.4	158.7	320.9	-10.4	-2.6	13.5	1.0	13.5	2.1	0.2	77.6
B	78.4	165.9	-6.1	-11.0	-2.8	15.1	1.0	15.1	2.3	0.2	79.7
ab	80.3	155.0	-4.3	11.0	2.8	15.1	1.0	1.4	0.2	0.7	33.7
C	84.3	-8.0	-0.1	3.4	0.8	1.4	1.0	15.1	2.3	0.2	79.7
ac	81.6	1.9	-10.9	1.8	0.5	0.4	1.0	0.4	0.1	0.8	18.4
bc	78.3	-2.7	9.9	-10.8	-2.7	14.6	1.0	14.6	2.2	0.2	79.1
abc	76.7	-1.6	1.1	-8.8	-2.2	9.7	1.0	9.7	1.5	0.3	71.0

Error MS	6.53
Error SD	2.55

Appendix F

NP2 Reef	Pyrrhotite										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	104.3	200.0	395.2	800.3	-	-					
a	95.7	195.2	405.1	-4.8	-1.2	2.8	1.0	2.8	0.1	0.7	27.4
b	97.8	214.5	-9.0	-28.7	-7.2	103.0	1.0	103.0	5.1	0.1	91.4
ab	97.4	190.6	4.3	18.1	4.5	40.8	1.0	12.2	0.6	0.5	52.2
c	108.7	-8.6	-4.8	9.9	2.5	12.2	1.0	40.8	2.0	0.2	77.3
ac	105.8	-0.4	-23.9	13.3	3.3	22.0	1.0	22.0	1.1	0.4	64.6
bc	91.8	2.8	8.1	19.1	4.8	45.5	1.0	45.5	2.3	0.2	79.4
abc	98.9	7.1	9.9	1.8	0.4	0.4	1.0	0.4	0.0	0.9	10.5

Error MS	20.1
Error SD	4.48

NP2 Reef	BMS										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	91.1	179.2	363.8	731.0		-					
A	88.1	184.6	367.2	1.0	0.3	0.1	1.0	0.1	0.0	0.9	14.8
B	97.2	184.3	-7.8	4.0	1.0	7.0	1.0	7.0	0.6	0.5	52.8
ab	92.4	182.9	3.8	6.8	1.7	5.8	1.0	1.4	0.5	0.5	46.3
C	97.1	-3.0	5.4	3.4	0.8	1.4	1.0	5.8	1.8	0.2	75.2
ac	92.2	0.2	-1.4	6.6	1.7	5.4	1.0	5.4	1.7	0.3	74.0
bc	89.6	0.1	3.2	-6.8	-1.7	5.8	1.0	5.8	1.8	0.2	75.2
abc	93.3	3.7	3.6	0.4	0.1	0.0	1.0	0.0	0.0	0.9	5.9

Error MS	3.17
Error SD	1.78

F3 – P2 Reef

P2 Reef	Total Mass										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	45.2	84.5	176.6	355.5	-	-					
A	39.3	92.1	178.9	-3.9	-1.0	1.9	1.0	1.9	0.2	0.7	33.9
B	42.9	87.7	0.4	11.1	2.8	15.4	1.0	15.4	1.8	0.2	75.0
ab	49.2	91.2	-4.3	9.3	2.3	10.8	1.0	0.7	0.1	0.8	20.6
C	44.2	-5.9	7.6	2.3	0.6	0.7	1.0	10.8	1.3	0.3	67.7
ac	43.5	6.3	3.5	-4.7	-1.2	2.8	1.0	2.8	0.3	0.6	40.1
bc	47.4	-0.7	12.2	-4.1	-1.0	2.1	1.0	2.1	0.2	0.6	35.5
abc	43.8	3.6	-2.9	-15.1	-3.8	28.5	1.0	28.5	3.4	0.1	85.9

Error MS	8.51
Error SD	2.92

P2 Reef	Chalcopyrite										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	85.8	167.4	338.4	678.3	-	-					
A	81.6	171.0	339.9	-12.7	-3.2	20.2	1.0	20.2	12.3	0.0	97.5
B	85.1	168.0	-3.4	7.5	1.9	7.0	1.0	7.0	4.3	0.1	89.3
ab	85.9	171.9	-9.3	6.1	1.5	4.7	1.0	0.3	0.2	0.7	30.0
C	86.6	-4.2	3.6	1.5	0.4	0.3	1.0	4.7	2.8	0.2	83.3
Ac	81.4	0.8	3.9	-5.9	-1.5	4.4	1.0	4.4	2.7	0.2	82.2
bc	88.0	-5.2	5.0	0.3	0.1	0.0	1.0	0.0	0.0	0.9	6.2
abc	83.9	-4.1	1.1	-3.9	-1.0	1.9	1.0	1.9	1.2	0.3	65.8

Error MS	1.64
Error SD	1.28

Appendix F

P2 Reef	Pentlandite										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	69.6	142.8	278.8	561.7	-	-					
A	73.7	136.0	282.9	14.2	3.6	25.2	1.0	25.2	16.8	0.0	98.5
B	65.2	145.9	9.2	-15.7	-3.9	30.7	1.0	30.7	20.4	0.0	98.9
ab	70.8	137.0	5.0	0.8	0.2	0.1	1.0	2.1	1.4	0.3	70.0
C	71.4	3.6	-6.8	4.1	1.0	2.1	1.0	0.1	0.1	0.8	16.9
Ac	74.5	5.6	-8.9	-4.2	-1.0	2.2	1.0	2.7	1.4	0.3	70.4
bc	67.5	3.1	1.9	-2.1	-0.5	0.6	1.0	0.6	0.4	0.6	42.7
abc	69.5	2.0	-1.1	-3.0	-0.8	1.1	1.0	1.1	0.8	0.4	56.8

Error MS	1.51
Error SD	1.23

NP2 Reef	Pyrrhotite										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	34.7	88.3	164.5	335.8	-	-					
a	53.6	76.2	171.3	67.4	16.9	568.5	1.0	568.5	13.3	0.0	97.8
b	22.0	88.3	51.2	17.5	-4.4	38.3	1.0	38.3	0.9	0.4	60.2
ab	54.2	83.0	16.3	18.8	4.7	44.4	1.0	5.7	0.1	0.7	26.7
c	41.4	18.9	-12.2	6.8	1.7	5.7	1.0	44.4	1.0	0.4	63.4
ac	46.8	32.3	-5.3	-34.9	-8.7	152.1	1.0	152.1	3.5	0.1	86.7
bc	36.0	5.4	13.4	6.9	1.7	5.9	1.0	5.9	0.1	0.7	27.1
abc	46.9	10.9	5.5	7.9	2.0	7.7	1.0	7.7	0.2	0.7	30.7

Error MS	42.9
Error SD	6.55

Appendix F

P2 Reef	BMS										
Treatment Combination	Response	(i)	(ii)	(iii)	Effects	Sum of Squares	Degrees of Freedom	Mean Square	F	p(F)	Confidence
(1)	76.6	158.2	319.8	639 0	-	-					
A	81.6	161.6	319.2	23.8	6.0	70.8	1.0	70.8	64.1	0.0	99.9
B	76.0	158.4	14.6	5.8	1.5	4.2	1.0	4.2	3.8	0.1	87.7
Ab	85 6	160.8	9.2	7.0	1.8	6.1	1.0	0.0	0.0	0.8	15.0
C	77.5	5.0	3.4	-0.6	-0.1	0.0	1.0	6.1	5.5	0.1	92.2
Ac	80.9	9.6	2.4	-5.4	-1.4	3.6	1.0	3.6	3.3	0.1	85.6
Bc	77.5	3.4	4.6	-1.0	-0.3	0.1	1.0	0.1	0.1	0.8	24.6
abc	83.3	5.8	2.4	-2.2	-0.6	0.6	1.0	0.6	0.5	0.5	50.0

Error MS	1.10
Error SD	1.05

Appendix G – Flotation Data

Float Number	Sample Type	Conc (g)	Cum Cu Grade	Cum Ccp Grade	Cum Cu Recovery	Cum Ccp Recovery	Cum Ni Grade	Cum Pn Grade	Cum Ni Recovery	Cum Pn Recovery
FLOAT 15 (NR 12/39 p60<75, without Cu Sulphate)	Feed	15.18								
	C1	21.99	2.43%	7.03%	73.5%	73.5%	4.04%	10.67%	28.0%	45.2%
	C2	13.87	1.39%	4.03%	78.6%	78.6%	3.16%	8.34%	37.6%	60.7%
	C3	11.55	0.96%	2.78%	80.9%	80.9%	2.48%	6.54%	41.9%	67.6%
	C4	6.63	0.76%	2.18%	81.9%	81.9%	2.05%	5.40%	43.4%	70.0%
	Tails 2	12.99								
	Tails 3	13.30								
FLOAT 16	Feed	14.59								
	C1	23.50	2.51%	7.16%	76.3%	76.3%	3.46%	9.12%	26.8%	43.3%
	C2	16.58	1.59%	4.59%	81.8%	81.8%	2.97%	7.85%	39.1%	63.1%
	C3	12.92	1.15%	3.33%	84.1%	84.1%	2.46%	6.50%	44.6%	71.9%
	C4	8.17	0.92%	2.67%	85.2%	85.2%	2.09%	5.51%	46.6%	75.1%
	Tails 2	14.82								
	Tails 3	15.57								
FLOAT 17 (NR 12/39 p60<75, with Cu Sulphate)	Feed	15.37								
	C1	14.29	3.59%	10.36%	68.0%	68.0%	3.50%	9.22%	16.5%	26.7%
	C2	15.31	2.14%	6.19%	79.4%	79.4%	4.02%	10.61%	39.8%	64.2%
	C3	16.52	1.42%	4.11%	83.9%	83.9%	3.27%	8.63%	50.7%	81.8%
	C4	9.69	1.10%	3.18%	85.5%	85.5%	2.65%	6.99%	53.2%	85.8%
	Tails 2	15.08								
	Tails 3	14.49								
FLOAT 18	Feed	15.54								
	C1	14.32	3.40%	9.81%	67.6%	67.6%	3.66%	9.67%	17.0%	27.4%
	C2	13.89	2.07%	5.99%	79.3%	79.3%	4.21%	11.11%	38.7%	62.5%
	C3	15.46	1.41%	4.08%	83.3%	83.3%	3.43%	9.05%	48.7%	78.5%
	C4	9.22	1.11%	3.21%	84.9%	84.9%	2.82%	7.45%	51.2%	82.6%
	Tails 2	12.72								
	Tails 3	13.24								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Po Grade	Cum Po Recovery	Cum S Grade	Cum BMS Grade	Cum S Recovery	Cum BMS Recovery	Mass Recovery (g)	Water Recovery (g)
FLOAT 15 (NR 12/39 p60<75, without Cu Sulphate)	Feed	15.18								
	C1	21.99	3.27%	22.0%	8.3%	24.10%	49.9%	49.9%	22.0	171
	C2	13.87	2.83%	31.0%	6.5%	18.66%	63.0%	63.0%	35.9	318
	C3	11.55	2.68%	38.9%	5.4%	15.71%	70.1%	70.1%	47.4	479
	C4	6.63	2.54%	42.0%	4.9%	14.31%	72.8%	72.8%	54.0	584
	Tails 2	12.99								
	Tails 3	13.30								
FLOAT 16	Feed	14.59								
	C1	23.50	2.95%	19.6%	7.7%	22.23%	45.5%	45.5%	23.5	203
	C2	16.58	2.52%	28.6%	5.9%	17.12%	59.7%	59.7%	40.1	405
	C3	12.92	2.37%	35.6%	5.0%	14.54%	67.1%	67.1%	53.0	600
	C4	8.17	2.28%	39.6%	4.6%	13.20%	70.3%	70.3%	61.2	736
	Tails 2	14.82								
	Tails 3	15.57								
FLOAT 17 (NR 12/39 p60<75, with Cu Sulphate)	Feed	15.37								
	C1	14.29	5.42%	23.6%	10.0%	28.79%	38.5%	38.5%	14.3	102
	C2	15.31	3.08%	27.8%	7.8%	22.61%	62.6%	62.6%	29.6	252
	C3	16.52	3.13%	44.0%	6.3%	18.25%	78.7%	78.7%	46.1	490
	C4	9.69	2.86%	48.6%	5.5%	15.87%	82.9%	82.9%	55.8	667
	Tails 2	15.08								
	Tails 3	14.49								
FLOAT 18	Feed	15.54								
	C1	14.32	5.24%	22.3%	9.8%	28.41%	37.2%	37.2%	14.3	101
	C2	13.89	2.85%	23.9%	7.9%	22.82%	58.8%	58.8%	28.2	233
	C3	15.46	3.44%	44.7%	6.6%	18.93%	75.6%	75.6%	43.7	438
	C4	9.22	3.25%	51.1%	5.8%	16.62%	80.3%	80.3%	52.9	597
	Tails 2	12.72								
	Tails 3	13.24								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Cu Grade	Cum Ccp Grade	Cum Cu Recovery	Cum Ccp Recovery	Cum Ni Grade	Cum Pn Grade	Cum Ni Recovery	Cum Pn Recovery
FLOAT 27 (NR 12/39 p80<75, without Cu Sulphate)	Feed	15.88								
	C1	25.22	2.20%	6.32%	75.6%	75.6%	2.59%	6.80%	21.0%	33.9%
	C2	22.85	1.21%	3.49%	82.8%	82.8%	2.44%	6.45%	38.0%	61.3%
	C3	13.42	0.88%	2.55%	85.0%	85.0%	2.08%	5.48%	43.3%	69.8%
	C4	6.75	0.69%	2.00%	85.9%	85.9%	1.75%	4.62%	44.8%	72.4%
	Tails 2	15.50								
	Tails 3	13.95								
FLOAT 28	Feed	16.56								
	C1	28.34	1.71%	4.94%	75.6%	75.6%	2.35%	6.17%	23.0%	37.1%
	C2	21.59	1.01%	2.91%	81.8%	81.8%	2.13%	5.63%	37.1%	59.9%
	C3	12.76	0.70%	2.03%	84.0%	84.0%	1.73%	4.58%	41.3%	66.7%
	C4	6.80	0.56%	1.62%	84.9%	84.9%	1.48%	3.90%	42.8%	69.0%
	Tails 2	15.16								
	Tails 3	15.27								
FLOAT 29 (NR 12/39 p80<75, with Cu Sulphate)	Feed	16.16								
	C1	16.02	3.20%	9.23%	69.6%	69.6%	2.69%	7.11%	13.9%	22.4%
	C2	19.73	1.69%	4.88%	79.6%	79.6%	2.93%	7.72%	33.7%	54.4%
	C3	21.32	1.16%	3.34%	83.5%	83.5%	2.43%	6.41%	43.9%	70.9%
	C4	9.36	0.91%	2.63%	84.9%	84.9%	2.00%	5.29%	45.8%	73.8%
	Tails 2	15.00								
	Tails 3	14.25								
FLOAT 30	Feed	15.05								
	C1	14.94	3.45%	10.01%	69.9%	69.9%	2.91%	7.64%	14.0%	22.5%
	C2	19.88	2.05%	5.92%	79.5%	79.5%	3.06%	8.07%	34.7%	56.0%
	C3	21.05	1.46%	4.21%	83.4%	83.4%	2.56%	6.76%	44.5%	71.9%
	C4	9.85	1.13%	3.26%	84.9%	84.9%	2.08%	5.49%	46.4%	74.8%
	Tails 2	14.69								
	Tails 3	15.30								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Po Grade	Cum Po Recovery	Cum S Grade	Cum BMS Grade	Cum S Recovery	Cum BMS Recovery	Mass Recovery (g)	Water Recovery (g)
FLOAT 27 (NR 12/39 p80<75, without Cu Sulphate)	Feed	15.88								
	C1	25.22	2.22%	17.6%	6.13%	17.7%	43.3%	43.3%	25.2	213
	C2	22.85	1.93%	29.3%	4.80%	13.9%	64.6%	64.6%	48.1	508
	C3	13.42	1.95%	37.8%	4.24%	12.2%	73.0%	73.0%	61.5	730
	C4	6.75	1.92%	41.2%	3.97%	11.5%	75.8%	75.8%	68.2	847
	Tails 2	15.50								
	Tails 3	13.95								
FLOAT 28	Feed	16.56								
	C1	28.34	2.83%	26.4%	5.56%	16.1%	46.1%	46.1%	28.3	240
	C2	21.59	2.22%	36.5%	4.35%	12.6%	63.6%	63.6%	49.9	524
	C3	12.76	2.14%	44.2%	3.85%	11.1%	70.5%	70.5%	62.7	735
	C4	6.80	2.09%	47.9%	3.61%	10.4%	73.3%	73.3%	69.5	862
	Tails 2	15.16								
	Tails 3	15.27								
FLOAT 29 (NR 12/39 p80<75, with Cu Sulphate)	Feed	16.16								
	C1	16.02	5.06%	24.8%	8.51%	24.6%	37.0%	37.0%	16.0	111
	C2	19.73	4.52%	49.3%	6.75%	19.5%	65.6%	65.6%	35.8	309
	C3	21.32	3.81%	66.5%	5.26%	15.2%	81.6%	81.6%	57.1	639
	C4	9.36	3.41%	69.3%	4.68%	13.5%	84.5%	84.5%	66.4	815
	Tails 2	15.00								
	Tails 3	14.25								
FLOAT 30	Feed	15.05								
	C1	14.94	6.35%	28.3%	9.55%	27.6%	37.8%	37.8%	14.94	90.43
	C2	19.88	5.19%	53.9%	7.26%	21.0%	67.0%	67.0%	34.82	282.26
	C3	21.05	4.23%	70.5%	5.56%	16.1%	82.3%	82.3%	55.87	588.64
	C4	9.85	3.76%	73.6%	4.90%	14.2%	85.4%	85.4%	65.72	764.25
	Tails 2	14.69								
	Tails 3	15.30								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Cu Grade	Cum Ccp Grade	Cum Cu Recovery	Cum Ccp Recovery	Cum Ni Grade	Cum Pn Grade	Cum Ni Recovery	Cum Pn Recovery
FLOAT 1 (NP2, 11/47, p60, with Cu Sulphate)	Feed	15.06								
	C1	12.06	4.99%	14.43%	80.72%	80.72%	9.59%	28.53%	57.38%	57.38%
	C2	8.58	2.76%	7.97%	86.70%	86.70%	6.25%	17.56%	10.59%	67.97%
	C3	7.82	1.55%	4.47%	88.60%	88.60%	3.62%	9.87%	1.78%	69.75%
	C4	6.05	1.26%	3.65%	89.99%	89.99%	2.94%	7.98%	0.86%	70.60%
	Tails 2	14.88								
	Tails 3	16.07								
FLOAT 2	Feed	14.59								
	C1	14.17	4.09%	11.81%	80.33%	80.33%	8.47%	22.26%	60.79%	60.79%
	C2	7.79	2.32%	6.72%	85.96%	85.96%	4.21%	13.88%	7.97%	68.76%
	C3	6.92	1.62%	4.68%	87.86%	87.86%	3.15%	9.70%	1.67%	70.44%
	C4	5.87	1.25%	3.62%	88.98%	88.98%	2.52%	7.49%	0.89%	71.32%
	Tails 2	3.20								
	Tails 3	3.36								
FLOAT 3 (NP2, 11/47, p60, without Cu Sulphate)	Feed	15.34								
	C1	14.27	3.83%	11.15%	84.82%	84.82%	7.03%	18.88%	57.04%	57.04%
	C2	4.80	2.01%	5.81%	87.48%	87.48%	4.22%	11.27%	4.62%	61.66%
	C3	5.44	1.55%	4.47%	88.85%	88.85%	3.30%	8.77%	1.51%	63.17%
	C4	4.38	1.26%	3.65%	89.63%	89.63%	2.70%	7.17%	0.65%	63.82%
	Tails 2	15.84								
	Tails 3	15.54								
FLOAT 4	Feed	16.56								
	C1	15.00	4.29%	12.38%	84.99%	84.99%	8.25%	21.68%	61.13%	61.13%
	C2	5.13	2.39%	6.90%	88.16%	88.16%	5.21%	13.73%	5.47%	66.60%
	C3	5.55	1.65%	4.77%	89.62%	89.62%	3.67%	9.66%	1.72%	68.32%
	C4	4.95	1.27%	3.68%	90.54%	90.54%	2.83%	7.45%	0.77%	69.08%
	Tails 2	16.18								
	Tails 3	15.45								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Po Grade	Cum Po Recovery	Cum S Grade	Cum BMS Grade	Cum S Recovery	Cum BMS Recovery	Total Mass (g)	Total Water (g)
FLOAT 1 (NP2, 11/47, p60, with Cu Sulphate)	Feed	15.06								
	C1	12.06	19.72%	61.53%	21.91%	63.33%	65.22%	65.22%	12.1	73
	C2	8.58	15.54%	82.98%	15.88%	45.89%	80.87%	80.87%	20.6	193
	C3	7.82	12.58%	92.62%	12.26%	35.43%	86.11%	86.11%	28.5	343
	C4	6.05	10.72%	95.72%	10.35%	29.90%	88.11%	88.11%	34.5	495
	Tails 2	14.88								
	Tails 3	16.07								
FLOAT 2	Feed	14.59								
	C1	14.17	20.06%	76.47%	19.99%	57.78%	72.68%	72.68%	14.2	108
	C2	7.79	15.84%	93.60%	15.12%	43.70%	85.19%	85.19%	22.0	226
	C3	6.92	13.17%	102.29%	12.15%	35.10%	90.00%	90.00%	28.9	359
	C4	5.87	11.32%	105.84%	10.34%	29.87%	92.16%	92.16%	34.8	502
	Tails 2	3.20								
	Tails 3	3.36								
FLOAT 3 (NP2, 11/47, p60, without Cu Sulphate)	Feed	15.34								
	C1	14.27	16.53%	76.04%	17.01%	49.17%	74.64%	74.64%	14.3	154
	C2	4.80	14.58%	89.66%	14.23%	41.13%	83.43%	83.43%	19.1	259
	C3	5.44	12.64%	99.84%	11.77%	34.02%	88.71%	88.71%	24.5	406
	C4	4.38	11.20%	104.30%	10.25%	29.62%	91.05%	91.05%	28.9	541
	Tails 2	15.84								
	Tails 3	15.54								
FLOAT 4	Feed	16.56								
	C1	15.00	20.35%	79.50%	20.09%	58.06%	74.84%	74.84%	15.0	135
	C2	5.13	17.75%	93.04%	16.79%	48.53%	83.96%	83.96%	20.1	230
	C3	5.55	15.48%	103.51%	14.02%	40.51%	89.40%	89.40%	25.7	355
	C4	4.95	13.62%	108.67%	12.10%	34.98%	92.09%	92.09%	30.6	493
	Tails 2	16.18								
	Tails 3	15.45								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Cu Grade	Cum Ccp Grade	Cum Cu Recovery	Cum Ccp Recovery	Cum Ni Grade	Cum Pn Grade	Cum Ni Recovery	Cum Pn Recovery
FLOAT 9 (NP2, 11/47, p80, with Cu Sulphate)	Feed	16.41								
	C1	8.73	5.76%	16.66%	71.61%	71.61%	11.19%	31.35%	45.80%	54.12%
	C2	9.32	3.34%	9.65%	84.68%	84.68%	7.80%	21.17%	64.55%	76.28%
	C3	7.82	2.28%	6.58%	87.13%	87.13%	5.41%	14.54%	66.89%	79.04%
	C4	6.38	1.69%	4.89%	88.42%	88.42%	4.04%	10.82%	67.97%	80.32%
	Tails 2	16.07								
	Tails 3	16.90								
FLOAT 10	Feed	15.49								
	C1	10.45	5.46%	15.75%	77.53%	77.53%	10.13%	26.60%	49.71%	58.74%
	C2	8.68	3.02%	8.72%	83.88%	83.88%	6.60%	17.37%	62.01%	73.27%
	C3	7.27	2.09%	6.05%	85.74%	85.74%	4.62%	12.17%	63.86%	75.46%
	C4	6.80	1.61%	4.64%	86.88%	86.88%	3.56%	9.37%	64.94%	76.73%
	Tails 2	15.40								
	Tails 3	16.47								
FLOAT 31 (NP2, 11/47, p80, without Cu Sulphate)	Feed	16.43								
	C1	13.57	4.44%	12.81%	85.23%	85.23%	9.42%	24.50%	56.15%	66.35%
	C2	5.45	3.63%	7.00%	88.41%	88.41%	6.23%	16.33%	63.56%	75.10%
	C3	6.05	3.08%	4.89%	89.70%	89.70%	4.46%	11.71%	65.53%	77.43%
	C4	4.87	2.66%	3.75%	90.34%	90.34%	3.44%	9.06%	66.34%	78.38%
	Tails 2	16.84								
	Tails 3	14.99								
FLOAT 32	Feed	15.49								
	C1	14.43	4.50%	12.99%	86.63%	86.63%	8.94%	23.52%	57.71%	68.19%
	C2	5.60	2.70%	7.82%	89.44%	89.44%	6.10%	16.09%	63.80%	75.39%
	C3	6.23	1.90%	5.48%	90.58%	90.58%	4.37%	11.53%	65.50%	77.39%
	C4	4.57	1.48%	4.28%	91.14%	91.14%	3.44%	9.08%	66.22%	78.25%
	Tails 2	16.19								
	Tails 3	12.67								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Po Grade	Cum Po Recovery	Cum S Grade	Cum BMS Grade	Cum S Recovery	Cum BMS Recovery	Total Mass (g)	Total Water (g)
FLOAT 9 (NP2, 11/47, p80, with Cu Sulphate)	Feed	16.41								
	C1	8.73	23.19%	55.59%	25.57%	73.89%	58.46%	58.46%	8.73	35.47
	C2	9.32	16.52%	81.88%	17.63%	50.94%	83.33%	83.33%	18.05	149.54
	C3	7.82	13.06%	92.75%	13.23%	38.23%	89.63%	89.63%	25.87	282.35
	C4	6.38	11.00%	97.38%	10.94%	31.62%	92.42%	92.42%	32.25	415.89
	Tails 2	16.07								
	Tails 3	16.90								
FLOAT 10	Feed	15.49								
	C1	10.45	20.88%	62.65%	23.36%	67.50%	66.84%	66.84%	10.45	55.63
	C2	8.68	15.35%	84.34%	16.20%	46.83%	84.89%	84.89%	19.13	170.74
	C3	7.27	12.33%	93.50%	12.47%	36.05%	90.18%	90.18%	26.4	287.39
	C4	6.80	10.37%	98.85%	10.26%	29.65%	93.27%	93.27%	33.2	425.5
	Tails 2	15.40								
	Tails 3	16.47								
FLOAT 31 (NP2, 11/47, p80, without Cu Sulphate)	Feed	16.43								
	C1	13.57	18.39%	67.59%	20.66%	59.72%	72.42%	72.42%	13.57	116.39
	C2	5.45	15.81%	81.42%	16.98%	49.07%	83.40%	83.40%	19.02	219.36
	C3	6.05	13.65%	92.65%	13.81%	39.92%	89.43%	89.43%	25.07	376.6
	C4	4.87	12.07%	97.82%	11.92%	34.44%	92.15%	92.15%	29.94	522.76
	Tails 2	16.84								
	Tails 3	14.99								
FLOAT 32	Feed	15.49								
	C1	14.43	16.61%	64.11%	19.61%	56.67%	72.17%	72.17%	14.43	129.68
	C2	5.60	14.25%	76.35%	15.95%	46.10%	81.51%	81.51%	20.03	240.54
	C3	6.23	12.39%	87.00%	13.00%	37.57%	87.07%	87.07%	26.26	398.07
	C4	4.57	11.13%	91.76%	11.39%	32.91%	89.55%	89.55%	30.83	530.47
	Tails 2	16.19								
	Tails 3	12.67								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Cu Grade	Cum Ccp Grade	Cum Cu Recovery	Cum Ccp Recovery	Cum Ni Grade	Cum Pn Grade	Cum Ni Recovery	Cum Pn Recovery
FLOAT 38 (P2 3/14 p60<75, with Cu Sulphate)	Feed	14.98								
	C1	7.58	7.13%	20.59%	55.05%	55.05%	4.16%	10.98%	8.75%	11.41%
	C2	11.62	3.86%	11.16%	72.16%	72.16%	5.70%	15.03%	30.75%	40.12%
	C3	11.06	2.78%	8.05%	78.56%	78.56%	5.78%	15.24%	49.00%	63.91%
	C4	8.99	2.23%	6.46%	81.55%	81.55%	5.12%	13.52%	56.12%	73.21%
	Tails 2	15.39								
	Tails 3	14.75								
FLOAT 39	Feed	14.84								
	C1	7.05	7.03%	20.32%	48.15%	48.15%	3.83%	10.10%	7.40%	9.65%
	C2	13.88	4.17%	12.07%	71.08%	71.08%	5.29%	13.96%	32.33%	42.17%
	C3	12.62	2.78%	8.05%	78.07%	78.07%	5.24%	13.82%	50.14%	65.41%
	C4	9.94	2.06%	5.96%	81.37%	81.37%	4.44%	11.73%	57.10%	74.49%
	Tails 2	15.28								
	Tails 3	14.14								
FLOAT 40 (P2 3/14 p60<75, without Cu Sulphate)	Feed	14.54								
	C1	15.61	4.26%	12.31%	73.04%	73.04%	4.32%	11.40%	20.11%	26.23%
	C2	9.92	2.16%	6.23%	79.73%	79.73%	4.50%	11.87%	33.80%	44.09%
	C3	11.68	1.51%	4.37%	84.16%	84.16%	4.44%	11.73%	48.93%	63.83%
	C4	8.00	1.20%	3.46%	85.77%	85.77%	3.83%	10.11%	53.33%	69.57%
	Tails 2	14.21								
	Tails 3	14.45								
FLOAT 41	Feed	14.75								
	C1	14.22	5.17%	14.93%	74.53%	74.53%	4.99%	13.17%	20.04%	26.14%
	C2	10.2	2.62%	7.59%	81.35%	81.35%	5.22%	14.28%	36.55%	47.68%
	C3	11.3	1.77%	5.13%	84.81%	84.81%	4.96%	13.04%	49.74%	64.89%
	C4	8.51	1.46%	4.22%	86.58%	86.58%	4.56%	11.54%	54.75%	71.42%
	Tails 2	14.64								
	Tails 3	14.85								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Po Grade	Cum Po Recovery	Cum S Grade	Cum BMS Grade	Cum S Recovery	Cum BMS Recovery	Total Mass (g)	Total Water (g)
FLOAT 38 (P2 3/14 p60<75, with Cu Sulphate)	Feed	14.98								
	C1	7.58	8.96%	14.39%	17.11%	49.44%	24.63%	24.63%	7.6	37
	C2	11.62	8.30%	33.77%	14.43%	41.71%	52.62%	52.62%	19.2	124
	C3	11.06	7.26%	46.54%	12.67%	36.62%	72.81%	72.81%	30.3	233
	C4	8.99	6.45%	53.63%	10.95%	31.64%	81.61%	81.61%	39.3	360
	Tails 2	15.39								
	Tails 3	14.75								
FLOAT 39	Feed	14.84								
	C1	7.05	7.39%	11.01%	15.95%	46.11%	21.28%	21.28%	7.1	29
	C2	13.88	5.73%	25.35%	12.96%	37.45%	51.32%	51.32%	20.9	120
	C3	12.62	5.49%	38.88%	11.31%	32.69%	71.79%	71.79%	33.6	228
	C4	9.94	5.10%	46.84%	9.84%	28.43%	80.94%	80.94%	43.5	350
	Tails 2	15.28								
	Tails 3	14.14								
FLOAT 40 (P2 3/14 p60<75, without Cu Sulphate)	Feed	14.54								
	C1	15.61	4.07%	15.29%	11.73%	33.89%	39.45%	39.45%	15.6	106
	C2	9.92	3.66%	22.50%	9.96%	28.78%	54.80%	54.80%	25.5	196
	C3	11.68	3.07%	27.48%	8.72%	25.20%	69.92%	69.92%	37.2	335
	C4	8.00	3.19%	34.72%	7.86%	22.72%	76.58%	76.58%	45.2	459
	Tails 2	14.21								
	Tails 3	14.45								
FLOAT 41	Feed	14.75								
	C1	14.22	4.46%	13.69%	13.74%	39.72%	37.78%	37.78%	14.2	86
	C2	10.2	4.10%	21.60%	11.64%	33.64%	54.95%	54.95%	24.4	172
	C3	11.3	4.33%	33.35%	10.17%	29.41%	70.25%	70.25%	35.7	302
	C4	8.51	4.34%	41.45%	9.07%	26.21%	77.54%	77.54%	44.2	433
	Tails 2	14.64								
	Tails 3	14.85								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Cu Grade	Cum Ccp Grade	Cum Cu Recovery	Cum Ccp Recovery	Cum Ni Grade	Cum Pn Grade	Cum Ni Recovery	Cum Pn Recovery
FLOAT 42 (P2 3/14 p80<75, with Cu Sulphate)	Feed	15.61								
	C1	13.66	5.12%	14.74%	70.99%	70.99%	4.16%	10.97%	15.14%	19.75%
	C2	12.54	3.05%	8.80%	80.43%	80.43%	4.93%	13.01%	34.49%	44.99%
	C3	13.17	1.99%	5.75%	84.09%	84.09%	4.69%	12.37%	49.55%	64.63%
	C4	9.86	1.61%	4.64%	85.94%	85.94%	4.07%	10.75%	54.26%	70.78%
	Tails 2	14.85								
	Tails 3	15.26								
FLOAT 43	Feed	17.35								
	C1	8.08	6.80%	19.67%	62.01%	62.01%	3.82%	10.07%	8.09%	10.56%
	C2	13.06	3.82%	11.03%	76.48%	76.48%	5.23%	13.79%	30.57%	39.88%
	C3	12.95	2.59%	7.48%	81.56%	81.56%	5.14%	13.57%	47.53%	62.01%
	C4	9.68	2.01%	5.82%	83.94%	83.94%	4.45%	11.73%	53.27%	69.48%
	Tails 2	16.51								
	Tails 3	16.53								
FLOAT 44 (P2 3/14 p80<75, without Cu Sulphate)	Feed	16.93								
	C1	13.89	4.84%	13.99%	73.67%	73.67%	4.62%	12.19%	17.91%	23.37%
	C2	10.59	2.65%	7.66%	80.36%	80.36%	5.16%	13.61%	34.68%	45.23%
	C3	10.54	1.84%	5.33%	83.49%	83.49%	4.69%	12.37%	45.78%	59.71%
	C4	7.92	1.44%	4.16%	85.05%	85.05%	4.01%	10.58%	50.00%	65.23%
	Tails 2	15.96								
	Tails 3	16.39								
FLOAT 45	Feed	16.10								
	C1	17.57	3.71%	10.72%	76.61%	76.61%	4.66%	12.30%	23.71%	30.92%
	C2	11.37	2.14%	6.19%	83.71%	83.71%	4.77%	12.59%	39.78%	51.89%
	C3	10.22	1.53%	4.41%	86.60%	86.60%	4.17%	11.01%	48.44%	63.19%
	C4	8.19	1.13%	3.27%	88.00%	88.00%	3.38%	8.93%	51.77%	67.53%
	Tails 2	16.44								
	Tails 3	15.99								

Appendix G

Float Number	Sample Type	Conc (g)	Cum Po Grade	Cum Po Recovery	Cum S Grade	Cum BMS Grade	Cum S Recovery	Cum BMS Recovery	Total Mass (g)	Total Water (g)
FLOAT 42 (P2 3/14 p80<75, with Cu Sulphate)	Feed	15.61								
	C1	13.66	7.24%	21.76%	13.94%	40.28%	37.52%	37.52%	13.7	85
	C2	12.54	5.82%	33.53%	11.65%	33.67%	60.16%	60.16%	26.2	186
	C3	13.17	5.33%	46.13%	10.08%	29.13%	78.20%	78.20%	39.4	324
	C4	9.86	5.01%	54.21%	8.82%	25.49%	85.58%	85.58%	49.2	458
	Tails 2	14.85								
	Tails 3	15.26								
FLOAT 43	Feed	17.35								
	C1	8.08	8.75%	16.33%	16.24%	46.94%	27.15%	27.15%	8.1	32
	C2	13.06	5.81%	28.36%	12.50%	36.13%	54.68%	54.68%	21.1	118
	C3	12.95	5.00%	39.35%	10.62%	30.68%	74.87%	74.87%	34.1	231
	C4	9.68	4.64%	46.93%	9.20%	26.59%	83.30%	83.30%	43.8	344
	Tails 2	16.51								
	Tails 3	16.53								
FLOAT 44 (P2 3/14 p80<75, without Cu Sulphate)	Feed	16.93								
	C1	13.89	2.51%	8.98%	12.11%	35.00%	38.74%	38.74%	13.9	87
	C2	10.59	1.43%	8.97%	9.91%	28.63%	55.85%	55.85%	24.5	180
	C3	10.54	1.67%	15.06%	8.57%	24.78%	69.14%	69.14%	35.0	302
	C4	7.92	1.99%	21.95%	7.69%	22.22%	76.03%	76.03%	42.9	412
	Tails 2	15.96								
	Tails 3	16.39								
FLOAT 45	Feed	16.10								
	C1	17.57	4.09%	17.72%	11.44%	33.06%	44.37%	44.37%	17.6	125
	C2	11.37	2.89%	20.63%	9.51%	27.48%	60.75%	60.75%	28.9	236
	C3	10.22	2.91%	28.06%	8.28%	23.92%	71.57%	71.57%	39.2	363
	C4	8.19	3.09%	36.05%	7.44%	21.49%	77.74%	77.74%	47.4	489
	Tails 2	16.44								
	Tails 3	15.99								