OPTIMAL DESIGN
OF A SECONDARY MILLING CIRCUIT FOR TREATING CHROMITE-RICH UG-2 PLATINUM ORES

By

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PREFACE

The work in this thesis was performed at the University of KwaZulu-Natal, School of Chemical Engineering, Howard College Campus, Durban, from February 2008 to December 2010. The project was supervised by Dr. J. Pocock and Prof. B.K. Loveday.

As the candidate's supervisor I, Dr. J. Pocock agree/do not agree to the submission of this thesis.

Signed: ……………………………………….

As the candidate's co-supervisor I, Emeritus Professor B.K. Loveday agree/do not agree to the submission of this thesis.

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DECLARATION

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(iii) This thesis does not contain other persons' data, pictures, graphs or other information, unless specifically acknowledged as being sourced from other persons.

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ABSTRACT

Extraction of platinum group elements (PGE) is a major source of revenue in South Africa and the reserves represent about 75 per cent of world reserves. Most of the remaining Platinum Group Mineral (PGM) reserves are located in the UG-2 chromitite layer of the Bushveld Igneous Complex. Platinum concentrators experience significant losses of PGE in their secondary milling circuits due to insufficient liberation of platinum-bearing particles. The chromium oxide (Cr_2O_3) content in UG-2 concentrates is typically 3%, which results in operational problems in the downstream smelting process. Ways of improving the design of the secondary milling circuit were investigated, with the purpose of improving PGE recovery and reducing Cr_2O_3 entrainment in the subsequent flotation stage.

Batch-scale laboratory and pilot plant tests were carried out to investigate the optimal design of a secondary milling circuit configuration. The optimal design consisted of a conventional hydrocyclone to de-slime the feed, followed by gravity separation with a spiral concentrator circuit to separate the ore into lights (silicates-rich) and heavies (chromite-rich) fractions. Separate milling of the light and heavy fractions made it possible to grind the silicate-rich fraction finer and to avoid over-grinding of the chromite. The total milling energy was re-distributed between the silicates and chromite ball mills with 88% of the energy input to the silicates mill and 12% to the chromite mill thus reducing chromite over-grinding. The effects on the recovery of PGE, and the entrainment of Cr_2O_3 were measured in combined batch rougher flotation tests.

The results indicated a 2% improvement in the secondary rougher flotation PGE recovery for the densifier underflow sample as compared to the standard MF-2 circuit, and most significantly the Cr_2O_3 entrainment was reduced by over 30% overall.

Attritioning of the chromite-rich heavies fraction and ball milling of the silicates-rich lights fraction resulted in a 52% reduction of Cr_2O_3 in the rougher flotation concentrate and a 0.4% increase in PGE recovery (0.4%) as compared to the standard circuit. The improved reduction in chromite entrainment may be attributed to the lower fines generation with attritioning (52.8%-106μm) as compared to ball milling with a 12% energy input (83.6% -106μm). Over 50% of the chromite minerals remained in the +106μm of the attritioned heavies product as compared with

21% for the ball milled spiral heavies stream. This accounted for a significant proportion of the overall chromite reduction in the flotation concentrate and supported the motivation for the inclusion of a separate grinding circuit for the chromite and silicate particles.

Pilot plant testwork on a VHG (very high grade) spiral concentrator circuit followed by laboratory milling and rougher flotation tests confirmed the above conclusions. A 3.7% improvement in PGE recovery was noted with a 32% Cr_2O_3 reduction in the secondary rougher flotation concentrate as compared to the standard circuit.

The statistical reliability of the laboratory and pilot plant data were quantified at various stages of the testwork due to the heterogeneous nature of the feed material and representative sampling. The repeat analyses on selected flotation tests for the high grade ore revealed that the variances were below 0.5%, 4%, and 7% for the head grades, PGE and Cr_2O_3 recoveries respectively. The flotation results for the standard and significantly improved milling circuits had variances in the 4E recoveries for the low grade ore and pilot plant ore of below 5.5% and 1% respectively. Low variances (<1%) in the Cr_2O_3 recoveries were noted for the low grade and pilot plant ores.

A preliminary cost estimate was undertaken based on the pilot plant data to determine what value the proposed circuit could add for an additional 3.7% PGE recovery. An additional revenue of approximately R50 000 per day could result based upon the platinum mineral recovery only. The other precious metals, i.e. palladium and rhodium were neglected and would further increase the overall revenue. The minimum payback period for the estimated capital investment would be approximately 4 years. This confirmed the benefit of this improved secondary milling circuit design as a viable option. A closed-circuit operation of the silicates mill should offer more significant benefits compared to the open circuit option; however, this was not considered in the current testwork.

This project has confirmed the benefit of separate ball milling and the use of a spiral concentrator as an effective gravity separation device in the secondary milling circuit for a chromite-rich (>50%) UG-2 platinum ore.

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NOMENCLATURE

SYMBOL	DESCRIPTION	UNITS
ρ	Density	kg/m^3
SG	Specific gravity of ore	-
V	Volume	m^3
d	Ball diameter	m
D	Mill diameter	m
μ	Viscosity of liquid	Pa.s
g	Acceleration due to gravity	m/s^2
P	Power	kW
W	Work defined by Bond's Law	kWh/t
W_i	Work index of ore	kWh/t
N_c	Critical Speed of the mill	rpm
F	Flow rate	t/h
L	Length	m
F_{80}	Screen size at which 80% of the feed material passes	µm
P_{80}	Screen size at which 80% of the product material passes	µm

CHAPTER 1: INTRODUCTION

1.1. Background

Platinum is a major resource to the South African economy as it generates large revenues which contribute to the GDP, result in job creation, infrastructure, skills development and also address the poverty crisis in the country. Platinum group elements (PGE – platinum, palladium, rhodium, ruthenium, osmium, and iridium) have unique properties such as inertness, corrosion and oxidation-resistance, high melting temperatures, and excellent catalytic properties which have made them very valuable for a wide variety of industrial applications (Xiao and Laplante, 2004). Platinum is commonly used in catalytic converters for motor vehicles (~50%), jewellery (~30%), electrical components, petroleum catalysts, and for several other processes (Glaister *et al.*, 2010).

The main exploitable reserves of Platinum Group Minerals (PGMs) within South Africa are the UG-2 (Upper Group 2) deposits. Currently, almost all UG-2 deposits are deep level, and therefore feasibility studies are based on batch testwork of borecore samples (Hay and Roy, 2010). Since UG-2 ore has a high chromite ($FeO.Cr_2O_3$) content, the reduction of the chromium oxide (Cr_2O_3) recovery in the final flotation concentrate is crucial to the smelting process, as the smelter furnace feed can be processed effectively only at an upper limit of about 2.5% Cr_2O_3 (Hay and Roy, 2010).

Platinum concentrator plants experience significant losses in their overall PGE recoveries due to the inefficiencies of their grinding and flotation processes (Rule and Anyimadu, 2007). Currently, the main method of secondary processing is through fine open circuit grinding, which is both energy intensive and can lead to low PGE recoveries (Bryson, 2004).

The main losses of platinum minerals occurs in the flotation tailings and hence the effect of improving the design of the secondary ball milling circuit was investigated for the potential to recover more PGEs, whilst reducing Cr_2O_3 recovery to the flotation concentrate.

The common hypothesis is that liberated chromite grains and silicate-rich PGE-bearing particles can segregate inside conventional ball mills during secondary grinding with an overflow discharge arrangement, which allows for the formation of a slurry pool (Bryson, 2004; Hinde and Kalala, 2009). If there is preferential segregation of the silicate-rich particles into the slurry pool, they will bypass the grinding process. However, insufficient research has been conducted to determine if these effects could be minimized.

1.2. Objectives of the project

The main objectives of the project were:

- To determine the possibility of improving the design of a secondary grinding circuit in a typical UG-2 concentrator.
- To determine whether the grinding energy could be re-distributed to achieve PGE recoveries in excess of 90%.

Separate grinding of the chromite and silicate particles has been investigated in the past (Rule and Anyimadu, 2007) and implemented on certain Anglo Platinum UG-2 concentrators. The key contribution of this project to the body of knowledge for UG-2 processing was the investigation of the method of separation of the chromite and silicate particles in the design of secondary milling circuits.

The Anglo Platinum UG-2 secondary milling circuit incorporated a conventional hydrocyclone for separation of the silicate-rich and chromite rich streams prior to ball milling and separate flotation; however, the effect of incorporating a spiral concentrator circuit, for gravity separation of platinum-bearing silicate particles from chromite particles present in UG-2 ores was investigated in this project.

The concept of implementing a spiral circuit within a PGM flowsheet is not unfamiliar to researchers. Researchers such as Bryson (2004), Deeplaul and Bryson (2004), Rule and Anyimadu (2007), and Lidell (2009) have mentioned the potential application of spiral concentrators on UG-2 ores due to the high chromite content; however, its specific application to secondary milling circuit design were not proposed. Spiral circuits have also been implemented on a two Anglo Platinum concentrators primarily for chromite rejection and UG-2 tailing re-floats (Rule and Anyimadu, 2007).

The effect of incorporating a spiral concentrator circuit into the existing concentrator milling circuit primarly for the purpose of producing silicate-rich and chromite rich products for separate milling have not been quantified in the existing literature based on the PGE recovery and chromite reduction.

The principle of energy re-distribution for the potential to achieve an improved PGE recovery was another significant contribution of this project to the field of UG-2 processing. The existing literature does not address this option; however, the mineralogy of the UG-2 ore suggested that this was an intuitive option. The build-up of chromite in the furnace causes operational problems for downstream smelting (Hay and Roy, 2010). Energy re-distribution may prove to a beneficial method of reducing the overgrinding of chromite particles.

Lee (1996) noted that the PGM distribution in UG-2 mineralogy correlates with a textural zoning with the composite chromitite layer. This is made up of two successive chromitites without the development of intervening silicate layers, and the PGMs are found to be invariably interstitial to the chromite grains. Hay and Roy (2010) further noted that most PGMs occur in association with base metal sulphides and silicates, with the base metal sulphides occurring almost entirely with interstitial silicate, and are only rarely enclosed within chromite particles. This suggested that the implementation of separate milling of the silicate and chromite particles, or alternative technologies such as attritioning of chromite particles, may significantly improve the overall design of the secondary milling circuit for UG-2 concentrators.

Through this project it was hoped that a significant reduction in Cr_2O_3 entrainment and higher PGE recoveries were achieved. This is of significant interest to platinum producers and would constitute a novel secondary milling circuit design.

1.3. Major aspects of test work programme

In order to develop a new secondary milling circuit design for the UG-2 concentrator, three major aspects were covered in the test work programme:

Firstly, a plant audit and mineralogical evaluation of the UG-2 ore was carried out. This involved a detailed literature survey on UG-2 processing and milling circuit design, to study the behaviour of the UG-2 ore within the Bushveld Igneous Complex. This was followed by a plant audit campaign of a secondary milling and rougher flotation circuit of an industrial scale operation to benchmark the current process. Representative samples from various points in the process were collected from the plant for size-by-assay, laboratory batch milling and flotation test-work. A detailed mineralogical evaluation of the secondary rougher tailings was conducted at Mintek using a Mineral Liberation Analyzer (MLA).

The second aspect of the test work programme was the optimization of the secondary ball milling circuit through the incorporation of gravity separation. A range of secondary ball milling circuit design configurations involving separate milling of the silicates and chromite particles, energy redistribution, closed-circuit milling of the silicates, attritioning of the chromite particles and removal of a clean chromite fraction, were tested in the laboratory under batch conditions. These were used to determine whether the incorporation of a gravity separation circuit in combination with these circuit modifications could improve the secondary rougher flotation recovery of platinum minerals and result in a significant reduction in the entrainment of chromite minerals to the secondary rougher concentrate.

The final aspect of the test work programme was the commissioning and testing of the new secondary milling circuit design on an industrial scale spiral concentrator circuit. A full scale VHG (very heavy grade) spiral circuit configuration was commissioned and tested by an industrial vendor to determine the overall impact of current spiral technology for treating UG-2 platinum ores. Laboratory size-by-assay, batch milling and rougher flotation tests were conducted on the VHG spiral concentrator products to confirm the benefits of this new milling circuit design as compared to the laboratory test work campaign on the MG (medium grade) spiral concentrator.

The subsequent chapters in this thesis provide an insight into the recent literature on this subject, the laboratory and pilot equipment and experimental procedures, and the discussion of results. The proposed secondary milling circuit design is presented at the conclusion of the thesis.

CHAPTER 2: LITERATURE REVIEW

Most UG-2 milling circuits adopt a two stage milling and flotation process with a secondary grind of 80%-75µm targeted after primary flotation (Deeplaul and Bryson, 2004). The mineralization of the UG-2 ore is one of the main factors that influence liberation of the PGMs and are discussed in detail in the literature review as the characterizations of the major minerals present and predominant mineral associations are fundamental to this study.

The effect of poor classification of the chromite and silicate particles prior to secondary milling is considered to be one the major concerns in the PGM flowsheet and is covered in this chapter. This is a major contributor to the poor liberation of PGMs and the cause of overgrinding of the chromite particles. The effect of entrainment of chromite to the flotation concentrate on the smelting process is discussed in the literature and the benefit of UG-2 chromite tailings for ferrochrome production and various ferrochrome technologies are noted. The floatability of UG-2 ore is briefly discussed thereafter, as this was required to benchmark the benefit of improvements in the milling circuit design.

Anglo Platinum has implemented a separate milling circuit for the silicates and chromite particles (Rule and Anyimadu, 2007) and the benefits of this design were compared with other milling circuit configurations. Due to the large density difference between the chromite (4.5 SG) and silicate particles (~2.6 SG), gravity separation was considered as a potential option to achieve a better separation of these particles.

Spiral concentrators have been successfully used for years in the mineral sands industry. Anglo Platinum have historically used them for chromite removal and tailings re-floats on a few existing UG-2 concentrator circuits (Rule and Anyimadu, 2007). Their principle of operation and advancement in technology over the years is discussed at the latter part of the literature.

An increase in PGE recovery and a reduction in the overall chromite entrainment are major design considerations in the final proposed milling circuit and therefore a critical review of previous researchers' work in the field of UG-2 ore processing was required and are detailed in the subsequent sections that follow.

2.1. An Overview of the Platinum Ore Bodies in the Bushveld Complex

The initial section of the literature focuses on the Bushveld Igneous Complex as it is the main resource of platinum ore for South African platinum concentrator plants and therefore a thorough understanding of the mineralogy of platinum reserves in the ore body is required.

The Bushveld Igneous Complex is located in three distinct mineralized zones, namely the Merensky Reef, Upper Group 2 Chromitite (UG-2) and the Platreef (Cawthorn, 1999). Figure 2.1 shows a simplified map of the Bushveld Complex illustrating the main lineage areas.

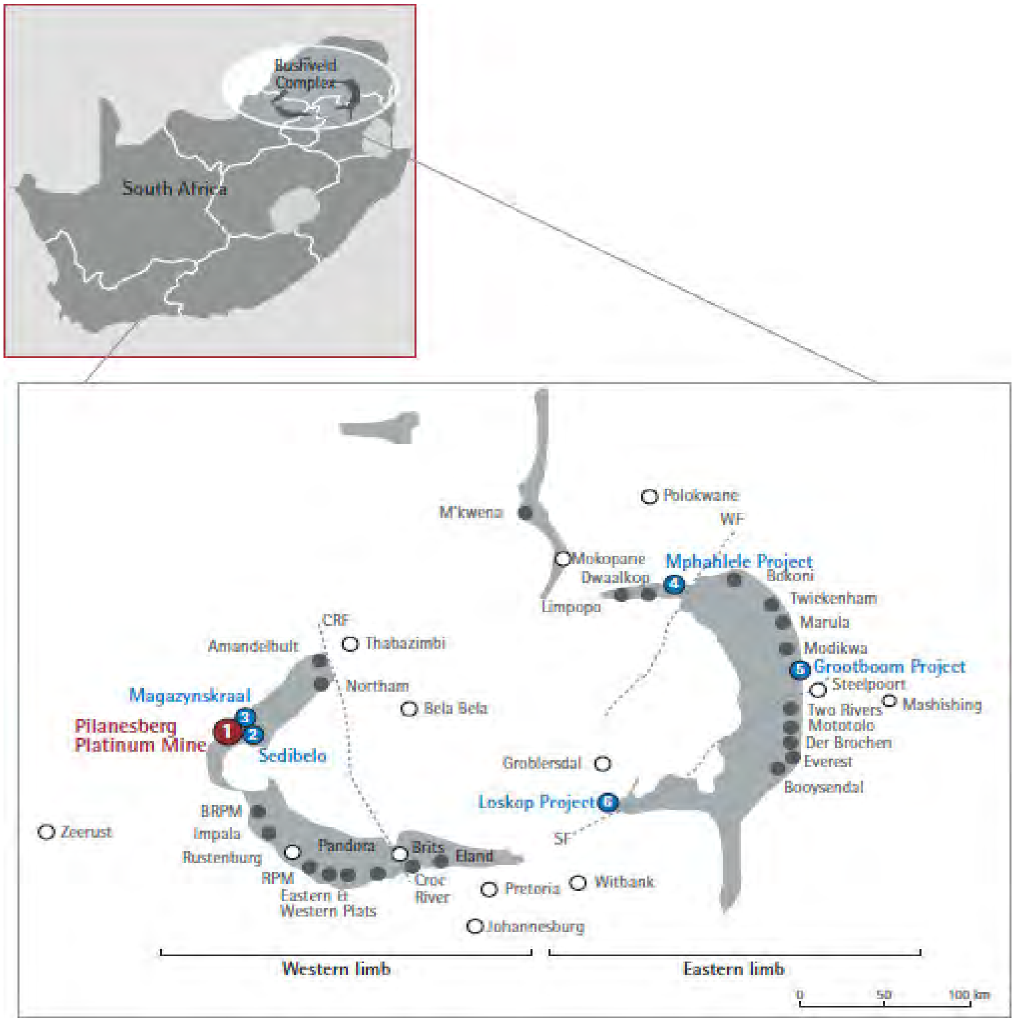


Figure 2.1: Map of the Bushveld Complex (www.platmin.com/p/p_i.asp accessed on 16 September 2010)

Lee (1996) indicated that the Bushveld Complex has a wide variety of commodities as compared to other world-wide complexes. The complex is a producer of platinum (Pt), palladium (Pd), chromium (Cr), iron (Fe) and vanadium (V) whilst gold (Au), silver (Ag), copper (Cu), nickel (Ni), cobalt (Co), sulphur (S) and titanium (Ti) also occur as by-products of platinum extraction. Platinum is by far the most important and widely used of the platinum group. It is malleable, ductile, practically infusible, unoxidisable and unattacked by all acids except aqua regia (Wagner, 1973). The PGE mineralisation contains a large proportion of platinum, which makes this complex platinum dominant as compared to the other geographical areas which tend to be more palladium dominant, e.g. Stillwater Mine in Montana, USA with a 4:1 palladium to platinum ratio. The combined platinum and palladium content in the Merensky Reef, Platreef and UG-2, are about 55% and 32%, 44% and 46%, and 46% and 30%, respectively, with the remaining 15% comprising of various other metals (Cawthorn, 1999).

The Bushveld Complex consists of three magmatic lineages, i.e. the main, critical, and lower zones (Lee, 1996). Eales and Cawthorn (1996) indicated that the Bushveld Complex is still remarkably unmetamorphosised and undeformed. The parental magma for the Lower Zone is basalt ($Be_3Al_2SiO_6$), rich in magnesium (Mg) and silica (Si). Evidence of plagioclase (composite silicates) is lacking for the Critical Zone. The Main Zone contains a pyroxenite (major igneous rock type) marker above the more evolved aluminious tholeiitic liquids (a type of basalt igneous rock), which includes the entire Upper Zone, which is a magnetite (iron oxide, Fe_3O_4) layer. The Upper zone has trace amounts of PGM.

The world’s largest platinum-bearing ore bodies, the Merensky and Upper Group 2 Chromite Layer (UG-2) are found in the critical zone with the Platreef commonly found in the main zone, which is base-metal (copper and nickel) sulphide rich with sporadic chromite ($FeO.Cr_2O_3$). Lee (1996) stated that the mineralogy has to be considered in any comparison made between deposits, and in the evaluation of deposits for potential worth. Cawthorn (1999) showed that the mineral layering precipitated as the temperature fell, resulting in a repetitive crystallization sequence into sub-horizontal mineral layers determined by the phase relations and composition of the molten magma (Figure 2.2).

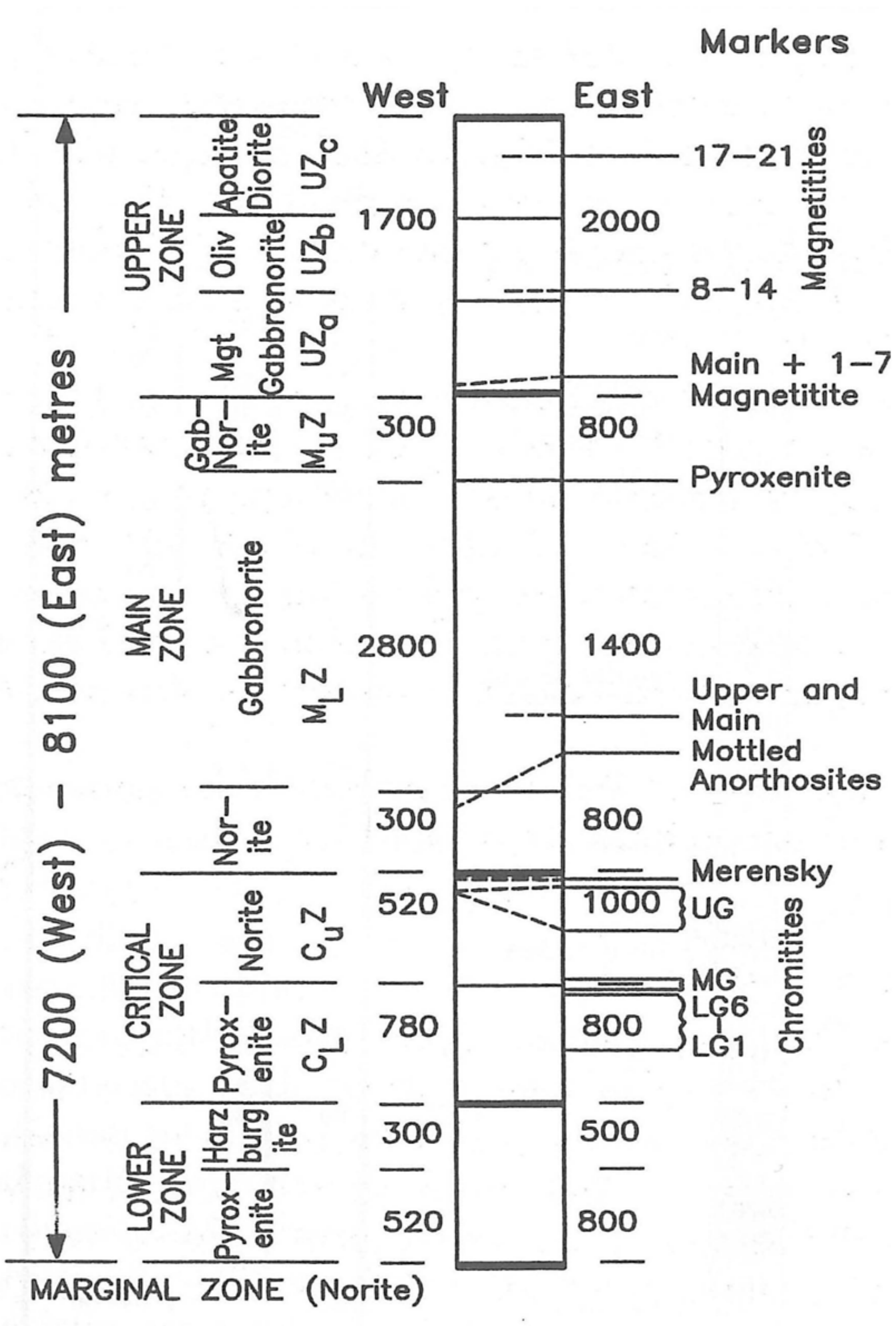


Figure 2.2: Stratigraphic section through the bushveld Complex, showing the locations of the UG-2 Chromitite and Merensky reefs (Cawthorn, 1999)

The chromitite layers are found in the Lower, Middle and Upper groups, and are numbered from the bottom upwards; where the UG-2 chromitite is located in the second layer of the upper group (Van der Merwe and Cawthorn, 2005).

PGMs occurs in well-defined layers in the Merensky Reef and UG-2; however, the Platreef in the northern limb is currently the focus for extensive exploration. The PGE grade (mass of PGE in grams per tonne of ore) is typically between 3 to 7 g/t for the Merensky and UG-2 Reefs; however, the Platreef has lower PGE grades generally below 4g/t (Kinnaird, 2005).

2.1.1. The Merensky Reef

Lee (1996) indicated that the Merensky Reef has PGEs which are dominantly Pt-Pd sulphides, and lesser and approximately equal amounts of PGE-arsenides (eg. $PtAs_2$), PGE-tellurides ($PtTe_2$) and other semi-metal phases. Ru sulphides and alloys, dominated by laurite (RuS_2) are associated with the reef chromitite layers with the PGE-alloys that have iron-rich phases regionally dominant. He noted that mica is also common in the reef and that the PGE grade varies with the width of the reef with the average thickness varying between 30 and 80cm thick. The average PGM grain size is about 236μm (Lidell *et al.*, 1986).

2.1.2. The Platreef

The Platreef does not have a varying PGE content. The PGEs are present as isoferroplatinum (Pt-Fe), sperrylite ($PtAs_2$), cooperate (PtS), merenskyite $[(Pt.Pd)(Te.Bi)_2]$ and a wide range of semi-metal alloys . The PGEs are associated with the base metal sulphides and with a higher proportion of silicate, with the mineral composition and textural relationship being related to the floor rock lithology (Kinloch, 1982). The PGE are on average coarser grained than the Merensky and UG-2 Reefs (Lee, 1996).

2.1.3. The UG-2 Reef

The UG-2 (Upper Group 2) chromitite layer in the upper Critical Zone is probably the largest PGE resource discovered on Earth (Vermaak, 1985). Chromitite is a rock containing chromite

($FeO.Cr_2O_3$) as its main constituent, consisting of 40-50% Cr_2O_3, and is usually analysed as a percentage of chromium oxide (Cr_2O_3) (Hay and Roy, 2010).

The chromitite layer occurs between 15 and 400m below the Merensky Reef with the smallest vertical separation noted in the western side and the greatest separation noted in the eastern Bushveld (Lee, 1996). The UG-2 ore consists of 0.5 –1.0 m thick chromitite rock with two to four minor (<1cm) chromitite seams occurring in the hanging wall (Lee, 1996) and Gain (1985) stated that the chromite content is 60-90%, with an average Cr/Fe ratio between 1.26-1.4 (i.e. an increase of about 43.5% Cr_2O_3). Lee (1996) stated that the PGEs are interstitial to the chromite grains and the only PGE commonly enclosed by chromite is laurite (RuS_2). The UG-2 reef has potholes present and these zonal alterations contain PGE alloys (bornite [Cu_5FeS_4], troilite [FeS]) and isoferroplatinum [$(Pt.Pd)_3(Fe.Cu)$]).

PGMs in the UG-2 are often much higher than in the Merensky and Platreef with up to 10 ppm PGE+Au (3.6 ppm Pt, 3.81 ppm Pd, 0.3 ppm Rh) (Gain, 1985). The Cu and Ni contents are low (< 0.05%) and thus the amount of accessory base metal sulphides is low (Lee, 1996). Schouwstra *et al.,* (2000) also indicated that the reef contains about 200 to 300ppm nickel occurring as nickel sulphides and less than 200ppm copper occurring as copper iron sulphides. Hay and Roy (2010) showed a general view of UG-2 ore in Figure 2.3.

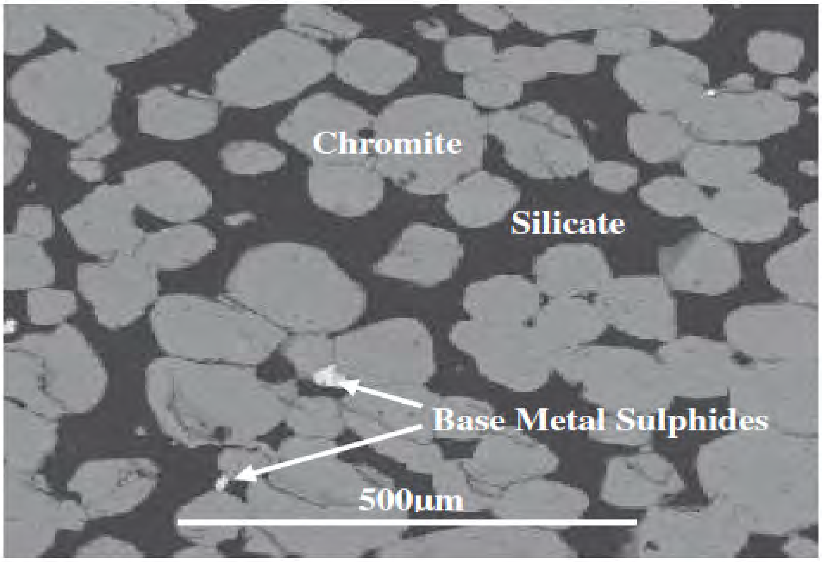


Figure 2.3: General view of UG-2 ore (Hay and Roy, 2010)

The base metal sulphides, i.e. mostly chalcopyrite, pentlandite and pyrrhotite, amount to less than 0.1% of the mass of the reef (Hay and Roy, 2010). Feng and Aldrich (1999) stated that the major base metal sulphide in UG-2 ores is pentlandite ($(Fe,Ni)_9S_8$). Although the sulphide content is low, most of the PGMs are attached to the base-metal sulphide grains (pyrite, pentlandite, chalcopyrite, and pyrrhotite) so the PGM may still be considered sulphide associated (Lee, 1996).

The major gangue minerals present are pyroxene (15-20%), feldspar (3-5%) and talc (1-3%) (Hay and Roy, 2010). Lee (1996) stated that some PGE may occur within the orthopyroxene gangue with high reserves of chromitite found within the Lower and Middle Zones.

PGMs occur predominantly as sulphides and some of these include: Laurite ($((Ru, Os, Ir)S_2)$, cooperite (PtS), braggite ((Pt, Pd)S), and malanite ($(Pt, Rh, Ir)_2CuS_4$). They may also occur as non- sulphide minerals such as: Pt-Fe alloys ((PtFe) and (Pt_3Fe)), tellurides, bismuthinides etc (Penberthy *et al.*, 2000) as shown in Figure 2.4.

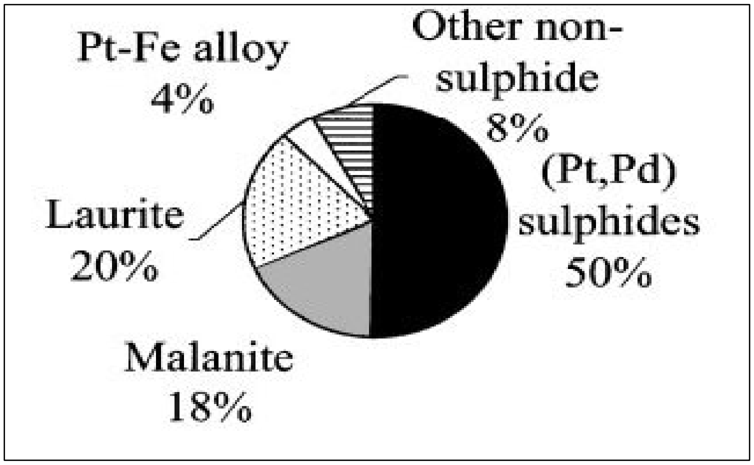


Figure 2.4: PGM compositions for a UG-2 ore sample (Penberthy et al, 2000)

Hay and Roy (2010) indicated that base metal sulphides occurred mostly with interstitial silicates, and are not commonly enclosed within chromite particles (Figure 2.5). The authors stated that about 2% of the PGMs are associated with chromite; however, this association may increase for severely altered ores (10%).

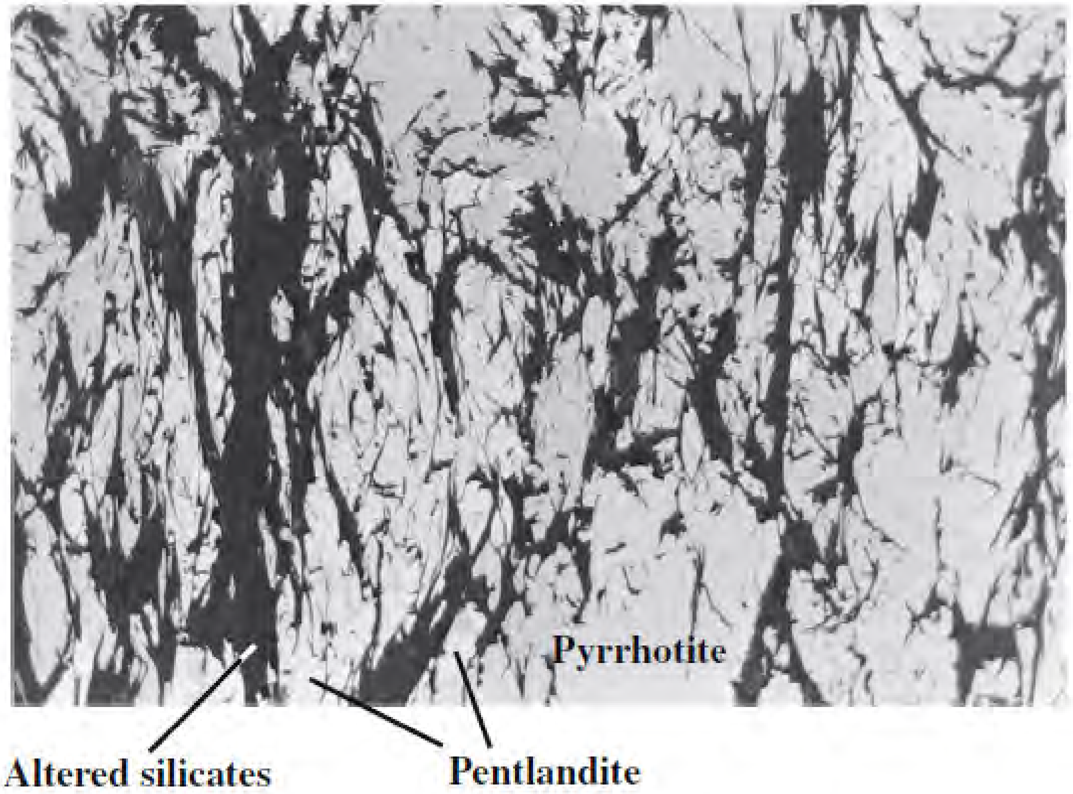


Figure 2.5: Alteration of base metal sulphides (Hay and Roy, 2010)

The UG-2 chromitite layer has two peaks in the PGM distribution with the precious metals concentrated towards the base of the layer, and also at the middle or towards the upper-contact (McLaren and de Villiers, 1982). The authors indicated that this break in the PGE distribution may correlate with the textural change noted in the chromite-silicate gangue from a granular silicate-chromite to a poikilitic (crystalline) silicate-chromite texture, best displayed on slightly weathered exposures.

McLaren and de Villiers (1982) relate the changes in the PGM composition and Pt/Pd ratio to the PGM abundance changes that occur vertically up the layer. Lee (1996) speculates that this textural zoning and bimodal PGM distributions indicate that the UG-2 reef is a composite chromitite layer made up of two successive chromites without the development of intervening silicate layers.

The PGMs are invariably interstitial to the chromite grains, and the only PGM commonly enclosed by chromite is laurite throughout the UG-2. Kinloch (1982) stated that the Ru is commonly associated with the chromitites in the rest of the Bushveld Complex and that the average PGM grain sizes are found to be 9.3µm in diameter. This indicates that UG-2 requires a much finer grinding than the Merensky ore to liberate the PGMs. This was also mentioned by Lidell *et al.,* (1986) who noted that the majority of the platinum minerals (70% by mass) in the Merensky ore were found to be between 150 and 38µm and therefore could be liberated with a much coarser grind.

Section 2.2 highlights the impact of the high chromite content in the UG-2 concentrate and describes how this affects the downstream smelting process.

2.2. Smelting of UG-2 ore and the effect of chromite

The operational problems faced when smelting UG-2 ores are summarised in this section together with the benefits of supplying UG-2 chromite-rich tailings to the ferrochrome industry. This section confirms that the UG-2 tailings stream is a major a resource for ferrochrome production rather than a low value waste stream. The advancement in ferrochrome technologies over the years and the process specifications of the UG-2 ore are summarised. This section undoubtedly provides an appreciation for the potential contribution that an improved milling circuit design with separate milling and gravity concentration may have on other large industrial sectors in South Africa.

Smelting of PGM concentrates in South Africa date back to the 1930s when blast furnaces were installed at Rustenburg (Nell, 2004). The percentage Cr_2O_3 in the final UG-2 flotation concentrate is about 3% with standard practice involving the mixing of these concentrates with high sulphide Merensky concentrate to reduce %Cr_2O_3 to below 1.5% in the smelter feed (Hay and Roy, 2010). The authors noted that although the gap in the percentage Cr_2O_3 between the furnace practical limit and the UG-2 concentrator product is being closed, certain concentrators can produce concentrates with less than 2.5% due to the difference in the mineralogy of the ore.

“Spinel” chromite in UG-2 smelting furnaces refers to certain trivalent Fe and Cr oxides with a high density, melting point, and viscosity, with a low solubility in the silicate slags and the matte, therefore at smelting temperatures; it forms a partial liquid and settles to the bottom of the furnace resulting in a reduced furnace capacity (Barnes and Newall, 2006). Hay and Roy (2010) and Barnes and Newall (2006) noted that chromite spinel causes operational problems such as difficulties in obtaining clean separation of matte and slag, high operating temperatures, a shortened furnace life due to a reduced capacity, and high maintenance and operational costs with lengthy repair periods.

A long-term solution to the problem of the ever increasing levels of chromium in the concentrate would be to abandon the current matte-smelting process in favour of an alloy-based process which would offer total flexibility with respect to the amount of chromium that can be tolerated in the feed (Nell, 2004).

Jones (2004) stated that the high chromite content in UG-2 concentrates has gained more importance in recent years as the growth in UG-2 mining is further increasing as compared to the Merensky ore. Jones and Curr (2006) stated that the ConRoast process uses a DC-arc furnace for alloy smelting (from oxide) instead of matte smelting (sulphide smelting) therefore the sulphur is removed at the beginning of the process (dead-roasting of platinum concentrate) and sent to an acid plant in the form of SO_2 (98% sulphur removal at that stage). The PGMs are collected in the iron-based alloy which is more efficient than a matte process for the collection of PGMs. The authors indicated that this process can handle a wide variety of feeds up to 100% UG-2 with low Cu, Ni or sulphur in the ore. The authors mentioned that the main advantage is that it can allow chromite in excess of 5% feed grade in the smelter without causing a problem of spinel precipitation. The iron is removed by converting, (or a few modifications to,) the base metal refinery. Geldenhuys and Jones (2008) indicated that Mintek facilities have processed over 37000 tons of PGM-containing feed materials in the last four years and have therefore demonstrated that this process is sustainable and efficient.

The market for chromite ($FeO.Cr_2O_3$) is predominantly driven by the demand for ferrochrome used in stainless steel production (Cramer *et al.*, 2004). The processing of UG-2 chromites yields significant savings in the total energy and electrical power required, both per unit of ferrochrome produced and per chromium content (Barnes *et al.*, 1983). Cramer *et al.*, (2004) investigated the impact of platinum production from UG-2 ore on ferrochrome production in South Africa. The authors noted that the Bushveld Igneous Complex (BIC) is a relatively low grade chromitite seam with a Cr to Fe ratio of 1.7 as compared with a ratio of 3 for other chromitite seams in the world. Lee (1996) and Gain (1985) noted that the Cr to Fe ratio of the UG-2 reef was lower than other chromitite seams in the BIC, with a ratio of 1.2 to 1.4 (Section 2.1.3). The authors noted that the viable seams are the LG6 (Lower Group 6), MG1/2 (Middle Group 1 and 2) and the UG-2 seam. Cramer *et al*., (2004) stated that it is relatively cheap and easy to recover chromite from a tailings stream of a platinum concentrator with at a grade of between 40 to 42% Cr_2O_3; however, this has only being exploited as a source of ferrochrome in recent years.

Two ferrochrome technologies exist to process UG-2 chromite, namely DC-arc furnace smelting and pelletizing (Cramer *et al.*, 2004). The author describes pelletizing where the ore is milled, pelletized and sintered to produce a hard, competent pellet that is a suitable replacement for lumpy ore. This is then sent to a submerged arc furnace (SAF) for smelting. A DC-arc furnace is used to supply the conditions required to produce an iron alloy which can effectively collect the PGMs, with very low residual quantities in the slag (Jones and Kotze, 2004). An extremely high temperature is achieved in the arc region of the DC-arc furnace process. This enhances the reaction kinetics and thermodynamics and ensures that the levels of chromium (Cr) conversion are higher than the SAF process. However, a higher energy input is required due to the increased heat losses. Normally the UG-2 ore Cr/Fe ratio is expected to be below 1.4 and therefore the ore is blended with another high ratio ore to produce ferrochrome with the desired specifications. DC-arc furnace smelting is preferable as the process yields an alloy of lower silica and titanium content (<1.5% Si and Ti). Cramer *et al*., (2004) indicated that chromite from UG-2 sources will be used more readily in the future for the production of ferrochrome. The authors noted that the ferrochrome process will be adjusted to accommodate the low Cr/Fe ratio of the ore and the fineness of the ore as this process is motivated by the low cost base for the conversion of UG-2 tailings to usable chromite.

Since this project focused on developing an improved secondary ball milling circuit design, the impact on the platinum minerals and Cr_2O_3 flotation recovery were monitored in response to circuit changes. A background on flotation of UG-2 ores is covered in Section 2.3.

2.3. Floatability of UG-2 ore

The floatability of UG-2 ore differs according to location due to its gangue mineral composition. The literature below provides a review of the reagents required, circuit design methodology and how methods have developed over the years in response to changing ore behaviour.

Chromite and talc are the two major gangue minerals in platinum-bearing UG-2 ore processed from the Bushveld Complex in South Africa (Mailula *et al.*, 2003). The flotation process for UG-2 ores is a bulk sulphide mineral flotation process for the recovery of platinum-bearing base-metal sulphides (Lidell *et al.*, 1986). Lidell *et al.*, (1986) noted that the UG-2 flotation occurs at a natural pH of 7.5 to 9.0, with a xanthate collector such as isobutyl xanthate.

The authors noted that the flotation circuit for UG-2 ores is similar in general principles to that for processing a Merensky ore, but since the main gangue mineral in the UG-2 ore is chromite as compared to talc for Merensky ores, the circuit design does differ in a number of aspects. The typical UG-2 circuit consists of one rougher stage followed by three cleaning stages to reduce the Cr_2O_3 content in the final cleaner concentrate (less than 3%).

Wesseldijk *et al.*, (1999) noted that the reagent suite for UG2-2 flotation is selected to maximize PGE recovery and includes the addition of an activator (copper sulphate), two thiol collectors (xanthate and dithiophosphate), a frother and a polysaccharide depressant. Modified carboxymethyl celluloses (CMCs) or silicate depressants may be used if talc is present in large proportions in the ore (Lidell *et al.*, 1986).

Greet (2008) monitored the flotation effect on UG-2 ores using a Magotteaux Mill to determine whether there were any differences in the pulp chemistry when using different grinding media. Forged steel, ceramics and varying compositions of chrome in steel media were tested. The author found that the 15% chrome alloy to be the best grinding media to enhance flotation of UG-2 ores. The flotation rates were higher for the high chrome steel as compared to conventional forged steel. Greet (2008) mentioned that these findings agreed with previous research, which showed the effect of the iron content on pulp chemistry as high iron concentrations in the flotation pulp depresses the floatable sulphide minerals. Greet (2008) showed that high chrome media reduced this effect and improved the floatability of UG-2 ore.

Valenta (2007) indicated that the selection and optimisation of the reagent suite for UG-2 reef flotation is a challenge. The author showed the significance of the effect of entrainment in the recovery of both valuable and gangue species and suggested that a greater understanding of this non-selective process is required in relation to the overall circuit performance. In addition, Wiese *et al.*, (2007) noted that although depressants are added to reduce the natural floatability of the gangue minerals present in the ore, the sulphide mineral recovery has been shown to be affected by the negative effects of guar on reducing the pyrrhotite recovery.

Mailula *et al.*, (2003) investigated the effect of copper sulphate addition on the recovery of chromite. The authors found that with no copper sulphate added, the lowest recovery of chromite was achieved. Since this was a linear relationship with the water recovered, the chromite was assumed to be entrained into the concentrate. No chromite activation was noted at low copper sulphate additions (<30g/t); however, at a dosage of 90g/t, higher chromite recoveries were obtained as compared with the standard tests. This demonstrated the inadvertent activation of chromite when copper sulphate was used as an activator during flotation.

Although the Merensky and UG-2 reefs are found in close proximity, there are differences between these ores as far as gangue mineralization and grain size of PGMs and sulphides are concerned. Feng and Aldrich (1999) investigated the effect of particle size on the flotation performance of complex sulphide ores. The authors noted that small bubbles were formed in the UG-2 flotation froth phase and showed greater rates of flotation as compared to the Merensky froths. Feng and Aldrich (1999) found that the optimum feed size for flotation was 80% -75µm and also noted that medium sized particles exhibited smaller and more stable bubbles, which resulted in more efficient flotation in this size regime.

Rule and Anyimadu (2007) noted that for typical UG-2 ores, the finer natural PGM grain size in the in-situ ore, and the relative flotation performance indicated that a relatively fine size where flotation drops off significantly. The authors investigated the effective of particle size on flotation recovery and indicated that finer particles have a better chance of recovery than coarser particles (+53µm). This study indicated that PGMs in the coarser fractions (+53µm) of the tails are not sufficiently liberated whereas PGMs contained in the ultra-fines (-10µm) are well liberated. The authors noted that this was not surprising as PGMs are typically less than 10 microns in size. The economics of additional capital equipment and the associated operating costs need to show a real business case to justify the pursuit of optimum liberation and hence recovery of metals (Rule and Anyimadu, 2007). The authors stated that grinding finer has its limitations, not only for the flotation performance, but it also affects the design of the downstream processing steps, such as, filtration, thickeners and tailings depositions. The authors referred to the ore mineralogy of Mount Isa mines, which recovers lead and zinc. In this case ultra-fine flotation was the only option to produce a saleable concentrate product (i.e. to separate the minerals for cleaning). This lead to the development of ultra-fine grinding technology (Isa Mill Technology) by McArthur River and George Fisher of Mt Isa mines to float the fines that were generated. This was based on a scaled up version of their existing technology. This technology advancement is regarded as the reason that both the two lead and zinc plants at Mt Isa mine are in operation today (Rule and Anyimadu, 2007). The authors indicated that IsaMilling technology presented some useful lessons to be applied in the PGM industry. The authors indicated that at present sub 10µm particles are floating reasonably well and ongoing tests are being conducted to improve the ultra-fine flotation. The authors believe that flotation technology will solve this problem eventually.

Pease *et al.*, (2006) noted that the conventional view was that fines “do not float” as shown in Figure 2.6. The authors stated that generally fine particles require more reagents due to a higher surface area per unit mass, have less momentum, and therefore follow water molecules more easily than coarse particles (i.e. tendency for entrainment). Flotation rates are slower and therefore lower cleaning densities, or froth washing may be required to counter entrainment. The authors stated that the rate kinetics are slower, and therefore fine particles would be similar to composite particles with flotation rate increasing with small bubbles in the froth. It was further stated that fine particles are affected by water chemistry and ions in solution and since they have a high surface area to volume, design of the thickeners and filtering is difficult. Pease *et al.*, (2006) stated that the design basis for flotation cells should account for sufficient residence time and lip length, utilize froth washing, and implement the cell design which was of the lowest cost and that meets the criteria.

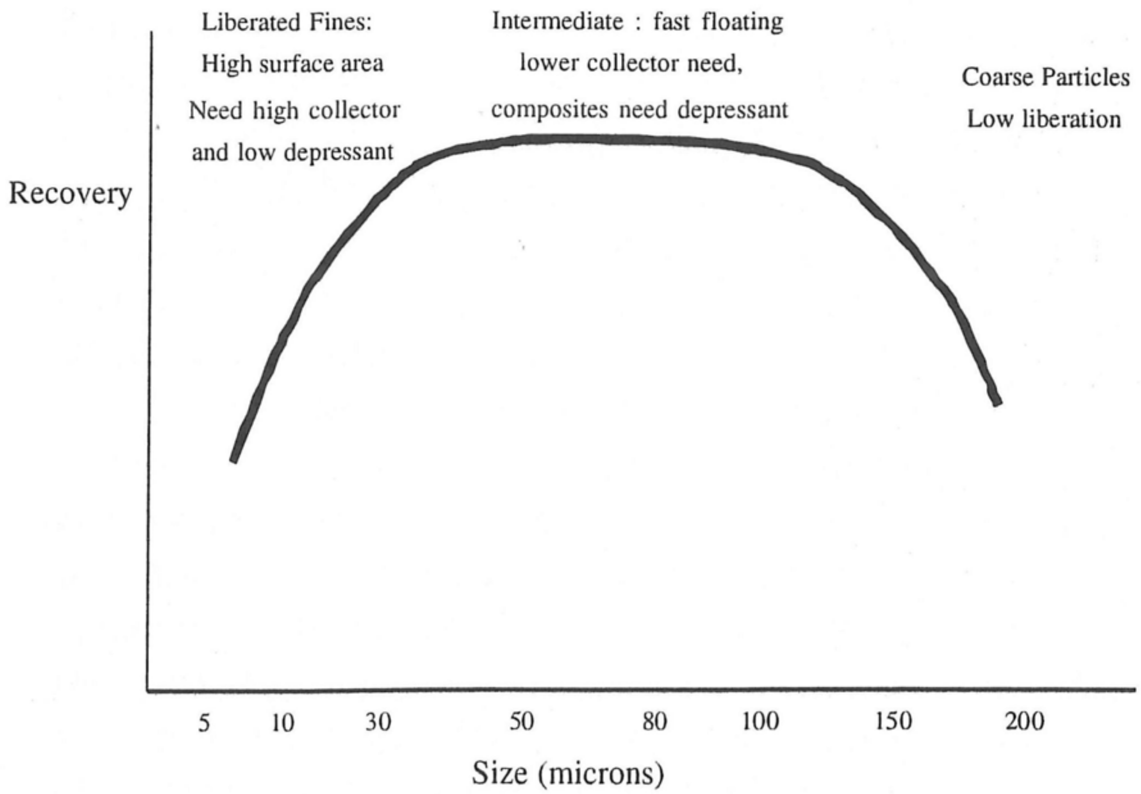


Figure 2.6: The conventional view "fines do not float" (Pease et al., 2006)

Pease *et al*., (2006) also mentioned that some design principles can be followed to ensure good fines flotation. The authors noted that the correct mineral liberation must be achieved to target grade and recovery, an efficient milling circuit should be implemented, and the cleaned mineral surfaces resulting from the grinding process must be followed by immediate flotation before the surfaces become oxidized again. The authors stated that flotation of minerals should occur in a narrow size distribution with a minimum circulating load, i.e. open circuit where possible and rather grind the cleaner feed than cleaner tails. The design of thickeners, filters, launders and pumps should account for froths containing a large proportion of fine particles.

The authors noted that a simple conceptual solution to fines flotation is to tailor the float conditions to a wide range of size distributions so as to achieve a high recovery across a broad size range, as illustrated in Figure 2.7.

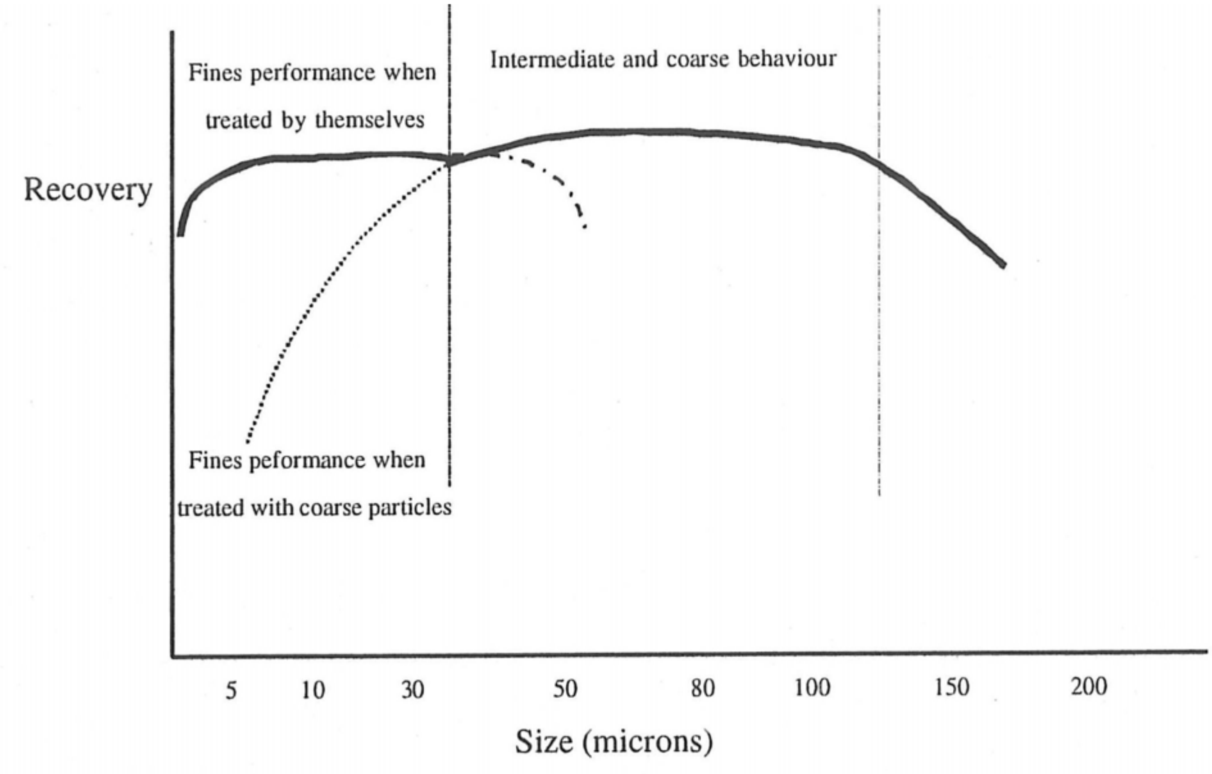


Figure 2.7: Conceptual staged grind-float circuit performance (Pease et al., 2006)

Hay and Schroeder (2005) used the SUPASIM flotation model to optimize Impala's UG-2 circuit as shown in Figure 2.8. This proprietary flotation simulation program was developed by Eurus Mineral Consultants (EMC). It can also be used for empirical mill sizing and design of flotation circuits due to its large database of plant, laboratory and pilot-plant operations data available. The actual data were kept confidential; however, the relative increase from the base recovery was mentioned.

The authors noted that in an operating plant it would be expected to achieve a relatively high recovery, a strong relationship between recovery and residence time, and a distinct change with the addition of depressant to target a high grade with a marginal loss in PGE recovery. Extensive laboratory tests, pilot plant runs and hot float tests (laboratory flotation of pilot plant samples) were performed in conjunction with the simulations on SUPASIM.

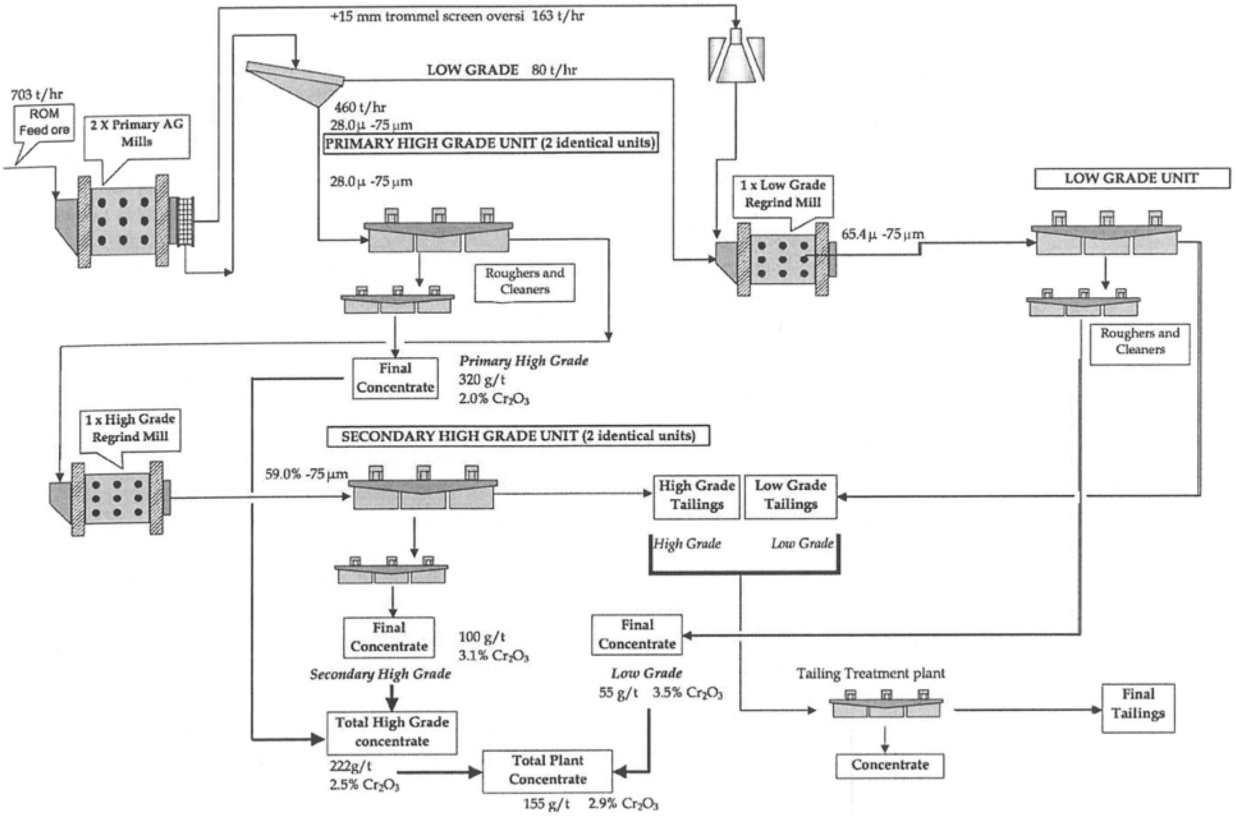


Figure 2.8: Impala MF2 Circuit (Hay and Schroeder, 2005)

Hay and Schroeder (2005) reported that recovery could be improved by 2.1%, with an increase in the secondary grind in the high grade unit from 59 to 65% -75µm. They also reported that doubling of both the rougher and cleaner residence times to 55 and 50min respectively, improved the recovery by 2 to 4%.

Ekmekci *et al.*, (2003) investigated the effect of the flotation behavior with different frother types and at various froth heights for a chromite UG-2 ore. The authors found that concentrates with lower Cr_2O_3 contents (< 3%) could be achieved by increasing the froth height. These mechanisms affect the entrainment of gangue and may also aid coarse composite particle floatability.

Hay and Roy (2010) indicated that the entrainment of chromite to the flotation concentrate is entirely due to the water recovery from the pulp/froth phase, as chromite is an oxide mineral and naturally hydrophilic, so its natural floatability is low. However, Hay (2010) noted that the success of rejecting chromite on a plant is best measured by the ratio of Cr_2O_3 to mass recovery so as to achieve good process control.

Jaseniak and Smart (2010) studied the surface chemical mechanisms for the recovery of chromite minerals in UG-2 ore flotation. The authors compared the coarse chromite in the concentrate with the tailings. The results indicated that a Cr_2O_3 recovery of more than 3% is achieved together with talc and pyroxene in single stage flotation. SEM images of the first concentrate revealed that the average particle size were above 100µm indicating true flotation and not entrainment. The author indicated that most of chromite recovered in the later stages was in fractions coarser than 35µm. The authors summarized the differences in chromite particles identified were correlated to the flotation response using a statistical method. This analysis revealed that coarse chromite particles that are recovered to the concentrate are due to their exposure to magnesium and silicon. The tailings samples show almost no sign of exposure to hydrophobic talc-like patches. The authors related the exposure to silicon to be a reliable expression as the chromite surfaces in UG-2 ore may be covered with altered silicates. The authors noted that chromite UG-2 ore can develop floatable properties by chemical reactions with the sulphide minerals in the flotation system, and through geochemical processes to form naturally floatable gangue minerals.

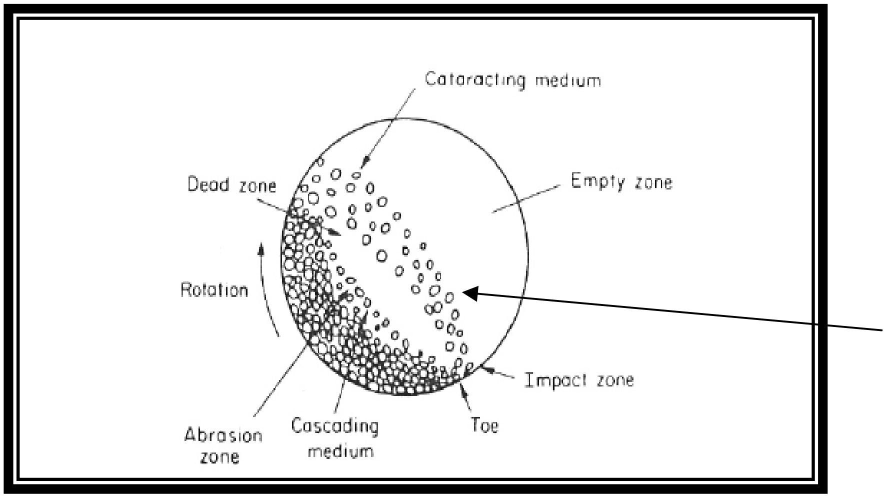
Mendez *et al*., (2009) found that the current milling models are more focused on either energy only, or population balance data, and do not incorporate liberation and water consumption together with the energy relationship. The authors indicated that simple models are required to intergrate models for design of the flotation circuits. The authors also noted that environmental issues are of major concern and should also be incorporated when designing flotation circuits.

Mintek (Deeplaul and Bryson, 2004) in conjunction with Lonmin developed a Mintek MF2 process which utilises a separate high grade and low grade cleaning circuit to isolate the fast floating PGMs from the slow floating PGMs and gangue minerals. The recovery of the PGMs from the gangue minerals was achieved through controlled depressant addition. This data also forms the basis of further research in order to optimise the flotation circuit configuration and to account for the slow-floating composite particles.

The flotation process for the UG-2 ore is very sensitive to the ore mineralogy, gangue mineral composition and degree of sulphide mineral content. Since loss in platinum flotation recovery could be due to several factors, i.e. composite particles that require a high residence time, poor liberation of PGE-bearing particles, a poor cleaning circuit design, etc, there is significant scope for optimization. The focus of this study is on methods of reducing entrainment of chromite by use of selective grinding of silicate minerals.

2.4. Comminution Theory

This section summarizes the fundamental milling theory required to aid in the development of a robust secondary ball milling circuit design. The cascading process in a ball mill is illustrated by the following diagram.



Cascading medium

Figure 2.9: Cascading process within a ball mill (Wills, 2007)

The critical speed of the mill is found from a force balance between the gravity force on a single milling ball and the centrifugal force acting (Wills, 1997). At this point everything centrifuges:

$$N_c = \frac{42.3}{\sqrt{(D-d)}} \tag{1}$$

Where N_c is the critical speed of the mill in revs per minute, D is the mill diameter (m) and d is the ball diameter (m).

For the effective grinding of particles the mill should operate between 65 to 80% of its critical speed (Wills, 1997). Literature does not indicate any effects of ore density on the critical speed equation and therefore further laboratory investigations would need to be conducted to determine whether grinding of the lower density particles are favoured at lower speeds, and vice versa.

The Bond Equation (Wills, 1997) is defined as the work done (kWh/t) as follows:

$$W = 10W_i\left(\frac{1}{\sqrt{P_{80}}} - \frac{1}{\sqrt{F_{80}}}\right) \tag{2}$$

Where W_i is the work index of the ore (a measure of its hardness) usually between 10 and 20kWh/t, P_{80} is the screen size at which 80% of the milled product passes through and F_{80} is the screen size at which 80% of the feed material passes through.

The Power draw of a mill (kW) is given by the following equation (Wills, 1997):

$$Power(kW) = Work(kWh/t) \times Flowrate(tph) \tag{3}$$

The above equations are used to determine the following mill design parameters:

- the optimal speed the mill in rpm
- the power being utilised during milling (using Bond's Law and flowrate)
- the power needed according to the mill filling
- the control of the process through density and power usage

A typical laboratory or industrial scale mill has the following design parameters that should be considered (Wills, 1997)

- Type of media required i.e. Steel or pebbles
- Batch (lab) or continuous (industrial)
- Mill speed such that a cascading action occurs (65-80% of the critical speed that causes centrifuging)
- % volumetric loading of the mill charge (media+slurry) should be less than 50% of the total mill volume

- Ratio of media to slurry by volume to achieve maximum grinding efficiency (ratio of balls to slurry is typically 50:50 or 60:40 respectively)
- 60-80% solids content of slurry depending on the fines content and type of ore.

Partyka and Yan (2007) noted that fine grinding was not utilized in the past as there were many perceived disadvantages. The mill speed had to be lower than the critical speed otherwise a portion of the charge would centrifuge. The authors noted that this upper limit fixed the maximum electrical power draw of the mill, and the size of the inactive zones increased with a drop in ball size. Gao and Weller (1994) noted with fine grinding a large proportion of the mill volume was kept empty to facilitate the cascading action of the mill charge. Zheng *et al.*, (1994) and Yan *et al.*, (1995) noted that fine grinding was also perceived to have poor energy efficiencies.

Currently fine grinding using stirred media mills with ceramic media are common industrially; however, modifications to the ball mill operating conditions have not been sufficiently investigated to confirm their limitations. Grinding finer (far smaller than liberation size) in turn results in some disadvantages besides the high power consumed; the production of very fine particles or slimes reduce the efficiency of recovery by flotation (Radhakrishnan, 1999). The author noted that slimes also negatively impact on pollution control and increases the cost of final tailings discharge.

Bond (1961) noted that the selection of the grinding media is based on the media size that will just break the largest feed particles. Equation 4 was developed to determine the largest size ball required.

$$B = 25.4\sqrt{\frac{F}{K}}\sqrt{\frac{SG \times W_i}{\%C_s\sqrt{3.281 \times D}}} \tag{4}$$

Where B is the ball diameter (mm), F is the 80% passing size of the feed (mm); K, an empirical constant = 350 for wet grinding and 335 for dry grinding, SG is the specific gravity of the material being milled, W_i is the Bond Ball Work Index of the ore, %Cs, fraction of the critical speed, and D is the diameter of the mill inside liners (m).

Austin *et al.*, (1984) showed how the size of the ball media should be selected based on the size of the particles being milled. The authors conducted dry grinding experiments in a laboratory mill and recorded the results as effect of ball size on rate of breakage at each particle size (Figure 2.10).

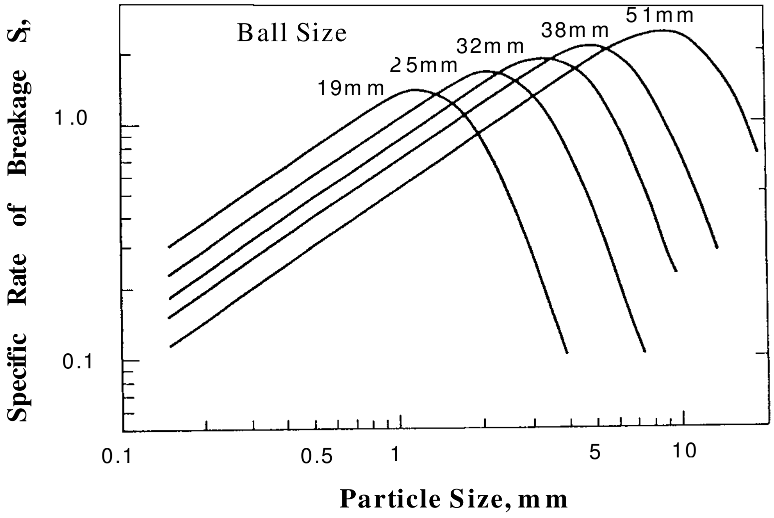


Figure 2.10: The effect of ball size on the rate of breakage of particles in a (dry) laboratory ball-mill (a redrawn illustration by Loveday (2010) based on the findings of Austin et al., 1984)

This figure clearly shows that the rate of grinding of particles less than 500 microns can be achieved by using a larger proportion of small media (< 19mm) as compared to the large media sizes (e.g. 51mm). Loveday (2010) stated that Bond's equation can be used to show that more than half of the energy required for typical ball-milling operations is required to reduce size from about 500 microns to finished product and hence more than half of the mill charge should consist of balls of about 15mm in diameter or smaller. Loveday (2010) also noted that the steady-state addition of two sizes is common, but the use of small balls (20mm or less) is not economical in most mineral processing operations, due to the increased cost of small balls and a less than proportional ball life.

The comminution theory introduces the major aspects required to understand the main operating parameters that affect milling and the following section covered the effect of material transport in ball mills.

2.5. Material Transport in ball mills

Material transport is one of the major factors that contribute to density segregation patterns in the mill. The different mill configurations do not always result in perfectly mixed pulp and thus result in poor grinding efficiencies. The effect of two typical types of mill discharge mechanisms and their effect on grinding is discussed in the following section.

Fuerstenau *et al*., (1986) studied the material transport in ball mills by considering material flow through two types of discharge arrangements, namely a grate-or diaphragm discharge (open end) versus an overflow discharge (constricted-end) ball mill as shown in Figure 2.11.

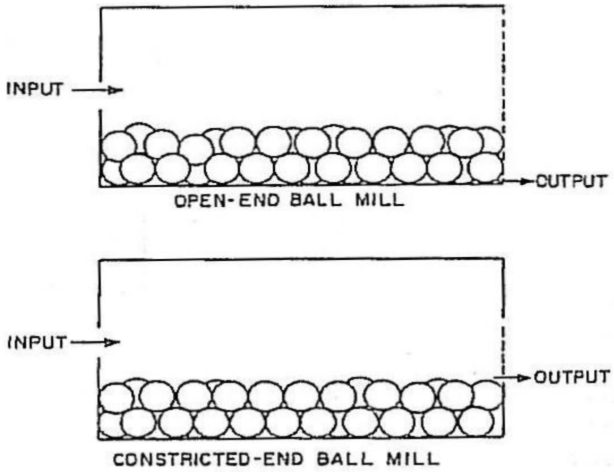


Figure 2.11: Material flow in an Open-End (Grate) and Constricted-End (Overflow) Ball Mill (Fuerstenau et a.l, 1986)

They performed investigations on the effect of feed rate, mill rotational speed, and ball filling. The authors noted that the dependent variables were the material hold-up and residence time distribution. These were characterized by the Peclet Number (Pe), which measures the amount of dispersion of pulp flowing through the mill. The Peclet Number is defined below.

$$P_e = \frac{uL}{D} \tag{5}$$

Where u is the axial convective velocity of material in the mill (m/s), L is the length of the mill (m) and D is the axial dispersion coefficient of material in the mill (m^2/s) determined from tracer tests.

The authors stated that grate discharge mills are favoured where a fairly coarse product is required and when it is necessary to avoid extreme fines in downstream processes. The overflow discharge arrangement has the advantage of a straightforward design and is normally used for finer grinds.

The authors stated that the difference in performance characteristics of these two types of mill has been attributed to steeper hold-up profiles in grate discharge mills resulting in lower retention in this type of mills. The authors note that besides this difference in arrangement, very little other information is known relating to its effects on mill design, using breakage kinetics and transport models.

2.6. Typical UG-2 Secondary Ball Milling Grinding Circuits

Currently, most circuits treating UG-2 platinum ores involve two stages of grinding and flotation. Primary grinding usually involves a run of mine (ROM) ball mill, followed by flotation, secondary ball-milling of the flotation tails, and secondary flotation. The tailings from the primary flotation are thickened and separated in a hydrocyclone, and the hydrocyclone underflow is fed to the secondary open circuit ball mill. The feed to the secondary ball mill usually comprises of large amounts of liberated chromite and silicate-rich particles containing the bulk of the PGMs in a locked state. In order to minimise PGE losses and energy expenditure, there is considerable incentive to find ways of milling the silicate-rich particles selectively. In South Africa, UG-2 plants commonly consist of an MF2 (Mill-Float-Mill-Float) where primary grind is in the region of 30-40% -75µm followed by a finer secondary grind of 80% -75µm (Hay and Schroeder, 2005). Each grind stage is followed by its own flotation circuit.

Lidell (2009) conducted a review of 25 years of UG-2 concentrator design and highlighted all the developments over the years including future research required. Mixtures of Merensky and UG-2 ore were used initially and noted that the first UG-2 only concentrator was commissioned in 1983 at Western Platinum Mine. The concentrator adopted an MF1 circuit (Figure 2.13) and achieved a 73% PGE recovery with 3% chromite content in the concentrate. The author noted that mining techniques had changed to reduce fines generations and that the cleaner residence times were increased and distributed between the primary and secondary rougher concentrates. This resulted in about a 10% improvement in the PGE recovery with a reduction in the chromite content of about 10%. Mineralogical work conducted in 1984 on the MF1 circuit flotation tailings revealed that about 45% of the PGMs locked in the silicate occurred in the +75µm fraction at a grade of 2.25g/t. The chromite was partly liberated in the -75+11µm fractions; however, this was rimmed (lined) by talc and base metal sulphides. The slimes (-11µm) accounted for about 20% of the PGE losses with a high degree of slow floating material present. Attritioners were incorporated into the Kroondal flowsheet as it had difficulties in finding a market for its concentrates, and thus used attrition scrubbers to enhance product quality.

Lidell (2009) noted that the liberation analyses led to a modified mill design to a two stage process with the first stage (primary) to liberate clean chromite from the base metals/PGMS and silicates whilst keeping the PGMs locked to the base metals. The secondary stage was included to liberate the base metals/PGMs from the silicates.

Classification is equally important in a milling circuit design as hydrocyclones tend to operate less efficiently when processing fine chromite, wasting energy and increasing chromite entrainment whilst not effectively allowing for the grinding of silicates to liberate the PGMs and base metals (Lidell, 2009). The author mentioned the importance of removing silicate waste as it has a higher energy consumption for crushing and milling and can be detrimental to the flotation process (if the chromite seam was narrow). However, since PGMs are present near the chromite content, dense media separation (DMS) was used for this application to minimise PGM locked particles in the low density products.

Lidell (2009) noted that Anglo Platinum's Kroondal mine adopted an MF2 configuration in 1997 with a primary grind of 40% -75µm and a secondary ball mill grind of 80% -75µm.

DMS was used to remove waste from wide reef mining and rod mills were used to liberate clean chromite with minimal sliming. A spirals circuit was incorporated on the primary rougher tails stream to recover the clean chromite which provided a revenue stream and lower mass for tails disposal.

Lidell (2009) stated that the present design of UG-2 concentrators still employs MF2 circuit with a move by some mines such as Anglo Platinum, Pilanesburg Platinum and Platinum Mile towards two-stage tertiary milling for fine grinding using horizontal and vertical stirred mills. The ball mills are run open circuit and are favoured over rod mills due to lower capital expenditure, although rod mills improve liberation of PGMs. The classification is still done using a hydrocyclone which results in overgrinding of chromite; however, this unit is extensively used in industry as no potential solution to this classification problem has been found. It was noted from X-ray analysis of concentrates used for process control that the chromite content of secondary concentrates is not properly dealt with and would increase as fine milling becomes more common. It was also stated that the Pilansberg Platinum mine commissioned its newest mine in March 2009 which treats UG-2, Merensky and Pseudo reefs. The open pit mining of UG-2 reduced waste dilution (less than 8% waste). The milling circuit design incorporated an MF2 circuit with a fine grinding tertiary Isa mill from the secondary mill discharge cyclone underflow stream to achieve a grind of 95% -75µm.

Lidell (2009) noted that high pressure grinding rolls (HPGR) was the next breakthrough in UG-2 processing. Fine milling technology also evolved with Anglo Platinum leading the field with its tertiary and regrind stages. Northam Platinum was the first to use an HPGR as a primary liberation tool for UG-2 ores. Lidell (2009) also mentioned that classification has not progressed over the last 25 years beyond hydrocyclones since fine screening technology such as Pansep and Microscreens failed due to mechanical challenges. However, recent developments in fine screening by the Derrick Corporation of the USA will be covered in Section 2.10. Column cell technology as a final cleaning stage was advanced by Northam Platinum and is covered in Section 2.8. Lidell (2009) indicated that attrition scrubbing of rougher concentrates is an economical option to improve the concentrate grade and reduce the chromite entrainment; however, it is underutilized on plants.

Lidell (2009) concluded that future developments in UG-2 concentrator design may focus on the use of DMS if the run of mine gangue minerals are in excess of 15%. The author noted that an HPGR may be implemented after the DMS as this will result in a lower power intensity in the primary ball mill. It was further noted that primary flotation may result in high PGE grades and lower chromite in the smelter feed. Lidell (2009) also mentioned that the recovery of chromite from the primary rougher tails may be conducted as UG-2 is presently an under-utilized chromite resource. The secondary and tertiary milling of the chromite recovery tailings and silicate-rich stream after potential clean chromite removal may reach 100% -75µm and the potential for fine cleaning with columns.

Lidell (2009) provided an excellent review of how the design of UG-2 concentrator plants have developed over the past 25 years and how various new technologies have impacted on the industry. It is clearly evident from Lidell's review that classification using a hydrocylone is one of the major concerns of the UG-2 concentrator plant design as it seems that the high chromite entrainment to the flotation concentrate emanates from this unit operation. Proper separation is required at this stage, i.e. prior to secondary grinding. The author favoured chromite rejection and fine milling of the silicates stream which contains the majority of the PGMs. It was suggested that spiral concentrators definitely has a future application in UG-2 concentrators as Anglo Platinum do use this technology at Kroondal, primarly for waste rejection.

Since the chromite constitutes about 50 to 60% of the UG-2 ore, a chromite rejection stream of may be 30-40% of the chromite may reduce processing costs for plants. This obviously depends on the PGM losses in the clean chromite stream. One of the objectives of this project is to quantify the platinum deportment in a chromite cleaner spiral circuit to justify such a design change to existing UG-2 concentrators. It will also be interesting to note the secondary grind that can be achieved after such a separation or even through a separate milling configuration has been implemented by Anglo Platinum. This will also determine whether tertiary mills are really required, or whether conventional ball mills are able to deliver a relatively fine grind through separate milling, if a separation is achieved prior to secondary ball milling.

Hay and Schroeder (2005) further noted that the two grinds firstly aim to liberate the PGMs from the coarse chromitite and secondly target the fine PGMs locked in silicates. The secondary milling process has inefficiencies in classification as the heavier chromite concentrates in the mill recycle stream (circulating load) and results in preferential milling of the denser minerals.

Nel *et al*., (2005) stated that pilot plant test work at Mintek established a unique ore separation flowsheet (Figure 2.12) which would allow for an increased mill throughput (25%) and an increase in the PGE recovery.

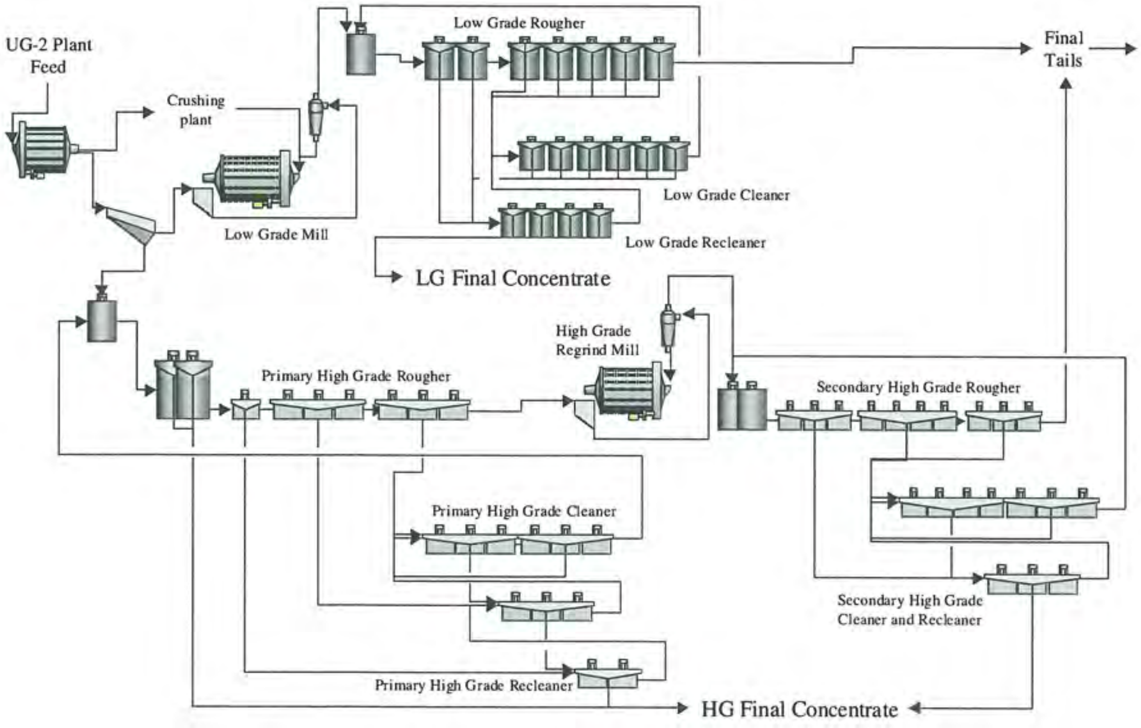


Figure 2.12: Unique ore separation circuit (Nel et al., 2005)

This circuit converted the closed circuit primary semi-autogenous (SAG) mill into a fully autogenous (FAG) open circuit mill. The classification process consisted of screens as a substitute for cyclones. The screen undersize contained chromite and was PGM rich. This was treated in a two-stage milling and flotation circuit. The screen oversize was low in PGM and chromite and was also treated separately in the same manner. This circuit incorporated a closed circuit regrinding mill with the hydrocyclone overflow fed directly to the secondary flotation feed. This circuit was commissioned and after a period of one year the design throughput was achieved, but the PGE recovery increases never materialized. This prompted Impala Platinum to conduct further optimization tests.

Nel *et al*., (2005) stated that after further optimization it emerged that an open-circuit secondary regrinding mill should be adopted instead of the closed circuit option. Pilot plant results showed a 3.7% PGE recovery benefit and therefore this configuration was implemented at the Impala Platinum Limited UG-2 plant. The results were encouraging as open circuit grinding has the problem of short-circuiting; however, this option in Figure 2.13 incorporated two hydrocyclones, with the second hydrocyclone included to reduce short-circuiting of coarse silicates to the hydrocyclone overflow as these particles normally contain platinum particles which require further liberation. Mineralogical data of a mill feed and product were analysed using QEMSCAN technology for this circuit modification.

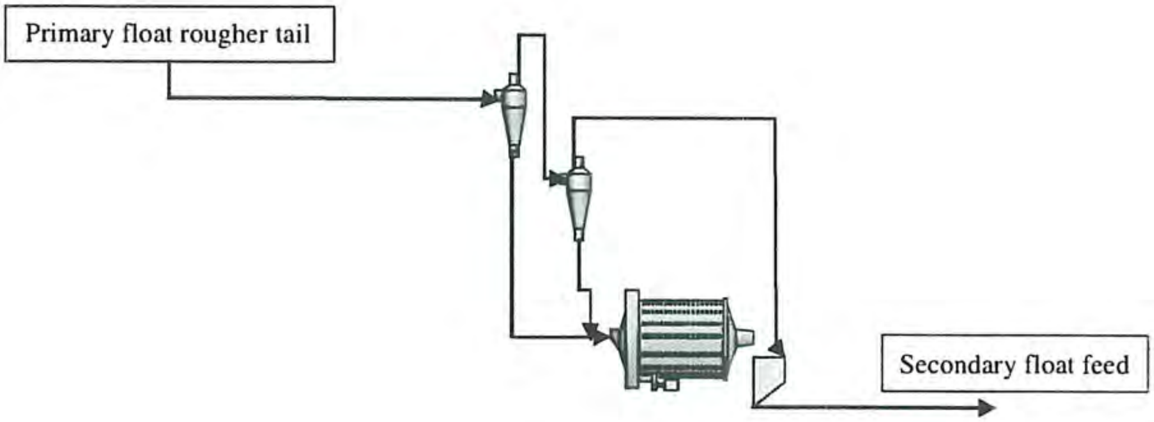


Figure 2.13: Open circuit regrinding at Impala Platinum (Nel et al., 2005)

The authors stated that most predictions relating to the benefits of open circuiting regrind milling were valid. As the circulation rate of the dense chromite particles was lowered, there was a decrease in the entrainment of chromite slimes into the flotation concentrate and a reduction in the amount of coarse silicates was also noted. This resulted in an increased the floatability of the ore. This shows that there may be a density segregation effect occurring in the regrind mill which results in a varying grind of chromite and silicate particles thus affecting the liberation of platinum-bearing particles and thereby lowering the overall PGE flotation recoveries.

Nel *et al*., (2005) showed the benefit of this process with the use of PGM deportment charts for both open circuit and closed milling in Figure 2.14 and 2.15 respectively.

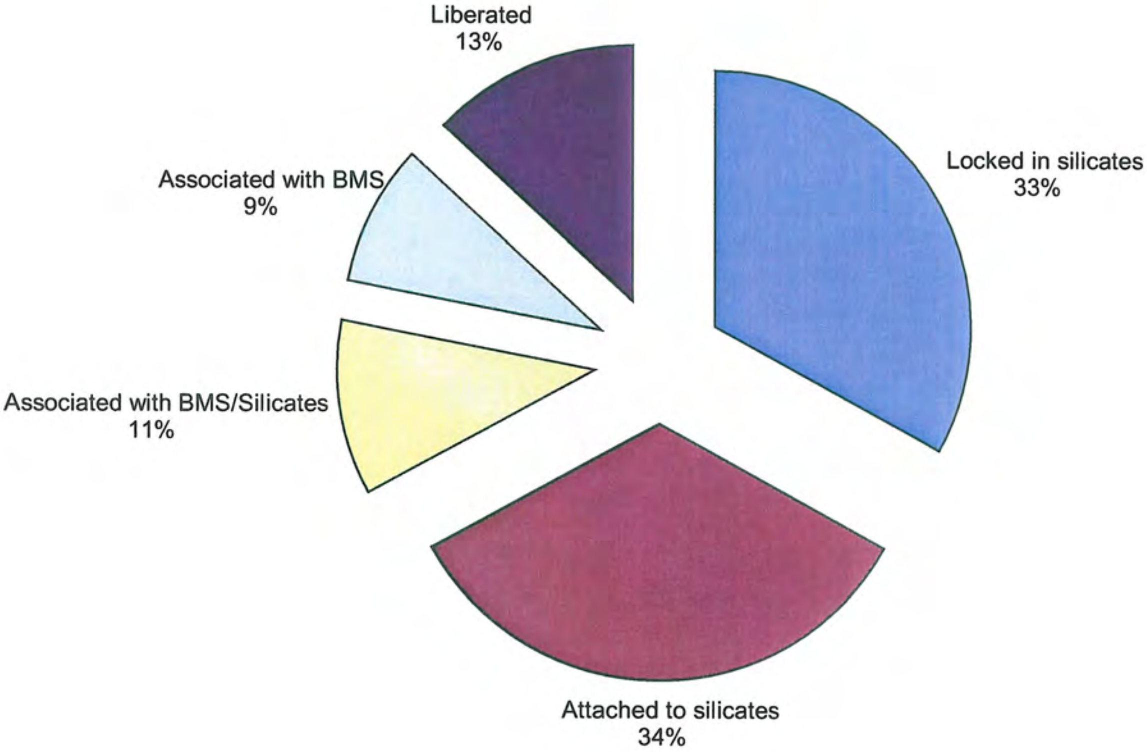


Figure 2.14: PGM deportment for closed- circuit regrinding (Nel et al., 2005)

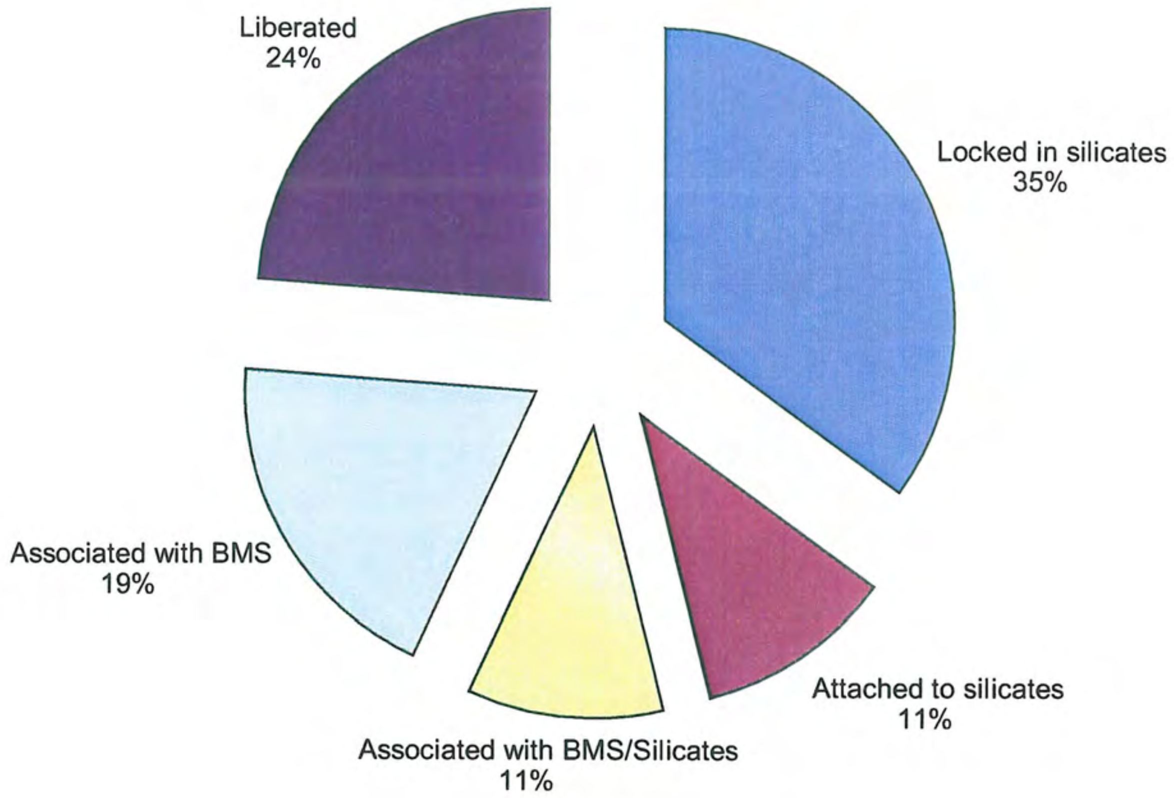


Figure 2.15: PGM deportment for open- circuit regrinding (Nel et al., 2005)

The liberated PGMs increased from 13 to 24% and the PGMs attached to the silicates decreased to 11% with the change from closed circuit to open-circuit regrinding. The authors noted that open circuit regrinding increased the ease with which a particle floats ("float rank") and it is more conducive to efficient attritioning as well.

Most industrial plants favour open circuit secondary grinding of UG-2 ore to reduce the overgrinding of the chromite minerals; however, if the chromite and silicate minerals can be separated by density, then closed circuit grinding should be the better design option for the silicate minerals. The section below gives an overview of some of these milling circuit configurations.

Mintek have assisted Lonmin with the implementation of the MF2 (mill-float-mill-float) circuit and thereafter with the conversion of UG-2 milling circuits to open circuit to reduce the high circulation of chromite to the secondary ball mill and to prevent the silicates from being displaced (Deeplaul and Bryson, 2004). The Mintek secondary milling open-circuit design is shown in Figure 2.16. This circuit includes a thickener unit which is fed with primary rougher tailings and hydrocyclone overflow material and thus reduces the amount of coarse silicates that may short circuit through the hydrocyclone overflow and bypass the secondary milling process. This configuration was shown to make significant inroads in reducing the presence of coarse unliberated silicates which contain the majority of the platinum-bearing minerals.

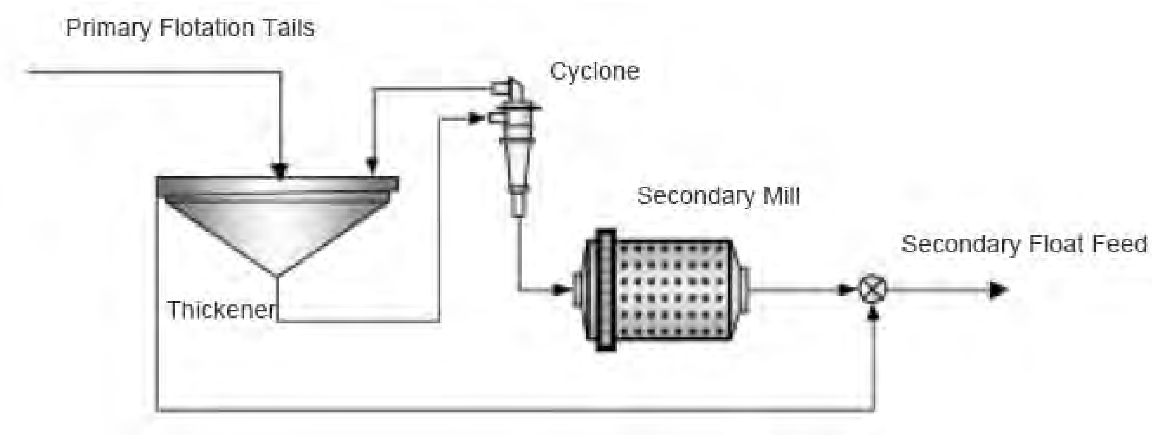


Figure 2.16: Mintek secondary milling circuit –open circuit operation for UG2-ore (Deeplaul and Bryson, 2004)

Hay (2010) stated that since the development of the first MF2 UG-2 plant in 1980s, chromite rejection has evolved from an art to a science and currently to a process control engineering problem. Hay (2010) indicated that the Mintek circuit enjoyed limited success as its specific PGM ores had a high degree of liberated PGMs, gangue and chromite; however, this is not true for all PGM ores. It was noted that operation of this circuit requires a lot of operator skill and process control. A separate low grade cleaner scavenger circuit is also required in the flotation circuit. Mines such as Northam Platinum and Sedibelo have adopted the use of column-cell flotation technology to produce a means of rejecting chromite more effectively with a greater degree of flexibility which does not require a significant amount of operator control and skill (Hay, 2010).

Anglo Platinum has also addressed the chromite entrainment problem and poor liberation of the coarse silicates (Rule *et al.*, 2007). Certain plants employ separate grinding and flotation for their silicate and chromite streams. Conventional hydrocyclones are used to separate these streams prior to milling. A few concentrators (e.g. Ivan and Mortimer) also have spiral circuits installed between the primary and secondary milling circuits for the removal and discard of chromite and also for tailings re-floats.

A diagram of the Anglo Platinum secondary milling circuit for their UG-2 ore concentrator plant is shown in Figure 2.17.

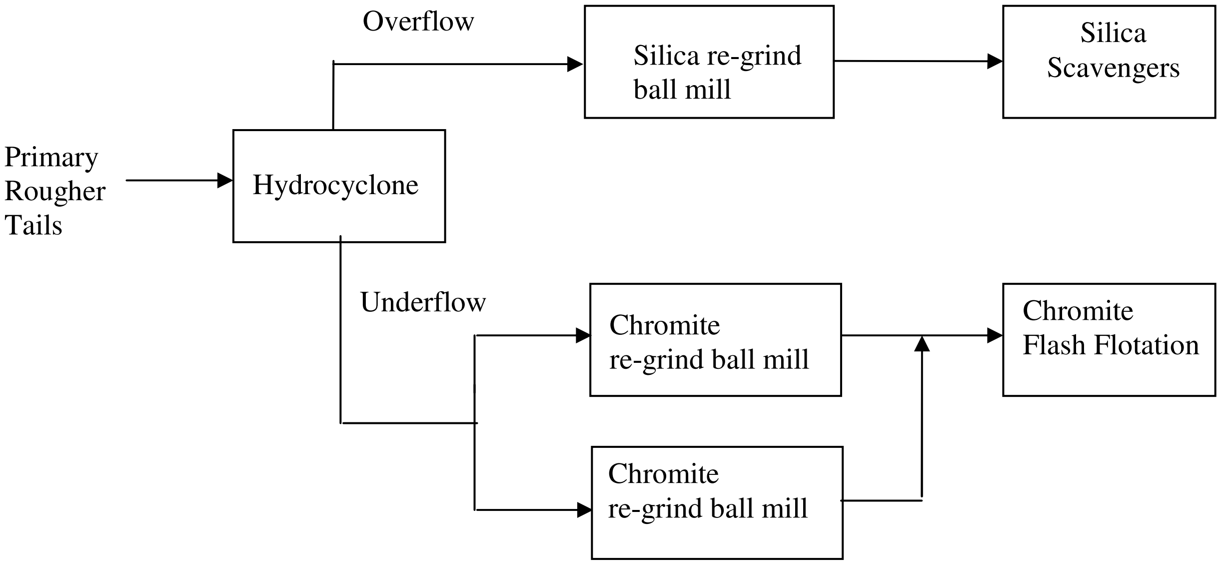


Figure 2.17: Overview of a UG-2 Operation at Anglo Platinum (Rule and Anyimadu, 2007)

This circuit design indicated that separate milling of the silicates and chromite is necessary to overcome the problem. A 1.67MW ball mill was used for the silicates re-grind and a combined 1.25MW power draw for the chromite re-grinds. This was based on mineralogy and assay by size data and showed that a larger proportion of the energy should be distributed to the silicates re-grind to liberate the locked platinum bearing particles. Separate flotation was also used as the flotation feed particle size and surface chemistry for silicates and chromite differs.

A hydrocyclone was used as the separating device; however, due to the large density difference between the chromite particles (4.5 SG) and silicate particles (~2.6 SG) with associated PGE's, the hydrocylone may provide a poor split with the coarse silicate reporting to the overflow and the fine chromite reporting to the underflow for flotation.

The consequences of such a split are that if the cyclone overflow stream (flotation feed) contains coarse silicates with locked PGMs, a loss in valuable mineral recovery will be noted (Mainza *et al.*, 2004). The fine chromite present in the underflow stream is sent back to the grinding circuit resulting in over-grinding of barren chromite (Becker *et al.*, 2008). This can result in a build-up of a high circulating load and loss in fresh feed capacity due to the mill inefficiency and therefore an efficient separating device is required to produce a relatively pure chromite and silicate products for separate milling. Another option of reducing the high re-circulating load was investigated by Mackinnon *et al.*, (2003) on a gold ore with the use of a flash flotation cell which can process coarse particles and reduce the tendency to over-grind particles.

Various modeling tools incorporating computational fluid dynamics (CFD) have been used to predict the dense flow classification of conventional hydrocyclones in an attempt to modify improve their operational performance; however, Schubert (2010) found that most of these models do not reflect the separation effect caused by the hydrocyclone flow and are not validated using experimental data or integrated into a separation model because of their complexity. This has lead to researchers looking at modified hydrocyclone configurations such as the three product cyclone to overcome shortcomings in conventional hydrocyclone classification.

Bryson (2004) discussed the mineralogical control of minerals processing circuit design. He noted that for an effective running of a PGM processing circuit, the mining plan and concentrator design must take into account specific mineralogical features of the ore. The author stated that within the chromitite layer of the UG-2 reef, the mineral chromite does not carry a significant PGM content and can be separated, and discarded after the primary milling and flotation stage. Some operations believed that if the chromite was removed and discarded at this point, it would cut back on the size and cost of secondary milling and flotation. This may also gain a secondary source of revenue from the sale of the chromite. The separation is currently achieved using spiral circuits which exploits the density differences between chromite and silicate minerals; however, a decision to include such a chromite removal stage into a UG-2 plant design is based on whether a significant amount of mass can be removed by this route without incurring a loss of PGMs. It was concluded that this process depends on the mineralogy of the ore treated and may not be easy in all cases.

Bryson’s comments are justified and further confirm the effect of UG-2 ore mineralogy and how different ore samples impact on concentrator plant design.

2.7. Classification with a three product cyclone

In most mineral processing circuits, classification is an important operation to ensure that values are liberated before being directed to the next unit operation, and as a method of improving on energy utilization (Bryson, 2004).

The use of a conventional hydrocyclone for the classification of UG-2 ores has resulted in many disadvantages especially when classifying the dense chromite particles found in the ore (Mahmoud *et al.*, 2009).

A hydrocyclone separates particles in a fluid stream on the basis of settling velocity, which is a function of particle size, shape and density. Becker *et al.*, (2008) quantified the influence of classification with a 3 product cyclone on liberation and recovery of PGEs in UG-2 ore as shown in Figure 2.18. The 3 product cyclone has an additional vortex finder concentric in the hydrocyclone resulting in two overflows, i.e. an inner and outer overflow, with one of the overflows comprising of middlings and other fines.

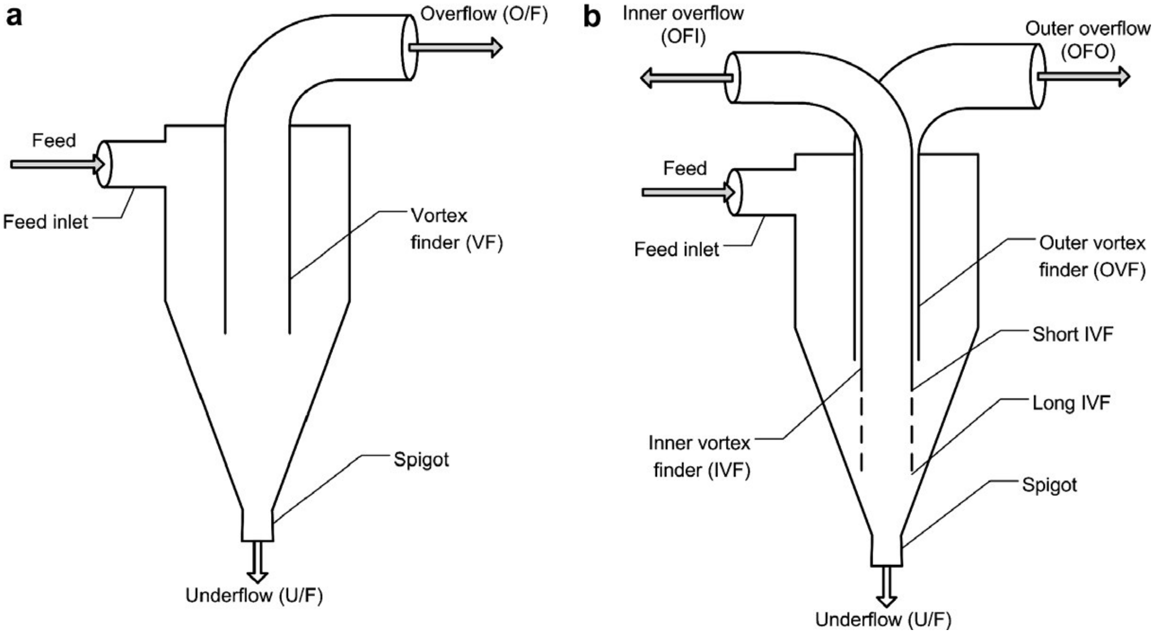


Figure 2.18: Illustration of the design (a) the conventional cyclone and (b) the 3 product cyclone (Becker et al., 2008)

Mainza *et al.*, (2004) noted that the middlings product of this unit can be screened as it contained 24% and 19% of the +100µm silica, which can be recovered through standard classification (Mainza *et al.*, 2004).

The outer overflow contains coarse silicates and the fine chromite fraction, which are screened at 100µm with a Pansep screen. The coarse silicates report to the screen oversize and the fine chromite reports to the screen undersize where it is combined with the fine overflow and sent to the flotation circuit.

Pilot plant trials were performed on a Lonmin Platinum Concentrator UG-2 platinum ore sample by Mainza *et al.*, (2004) based on the following circuit shown in Figure 2.19.

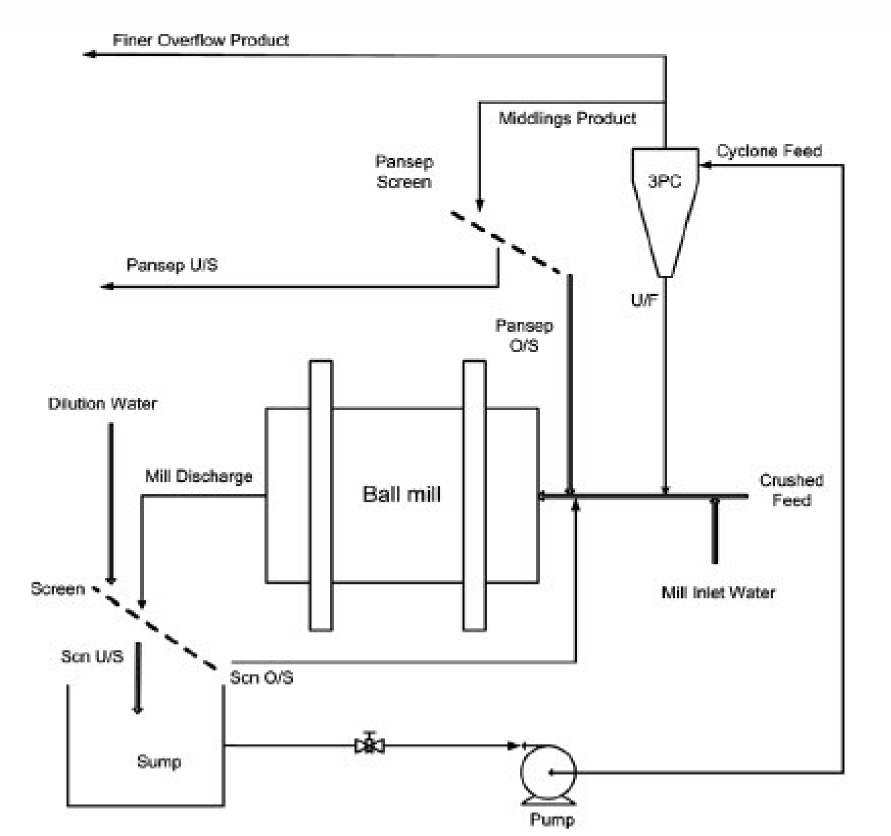


Figure 2.19: Pilot plant test rig set up for 3 product cyclone (Becker et al., 2008)

The authors showed that the 3 product cyclone reduced the overall chromite content in the flotation feed stream, (55.1wt% vs. 42.7%) with less fine chromite and an increased liberation of PGMs (21 to 34 grains in a 100 grain mineralogical sample size) with the locked PGMs also decreased from 46 grains to 27 grains.

The efficiency of separation of the 3 product cyclone may be improved through the use of finer screening technology below 75µm. The investigations carried out by Mainza *et al.*, (2004) and Becker *et al.*, (2008) suggested that alternative gravity separation methods should be investigated to determine if there is potential to achieve a better split of chromite and silicate particles.

2.8. The effects of new technolgies for processing UG-2 platinum ores

High Pressure Grinding Rolls (HPGR) are presently seen as available technology to be considered in feasibility studies to be compared with more conventional crushing and grinding processes such as SAG milling or multi-stage crushing for primary grinding (Meer, 2010). Achievements of high energy efficiency, high unit capacity and a low overall operating cost led to increasing numbers of units being installed in a wide range of application in the mineral processing sector in the past few years (Meer, 2010).

Solomon *et al.*, (2010) compared the effects of high pressure grinding rolls (HPGR) with conventional ball mills for platinum bearing ores. The authors indicated that this technology has shown potential in recent years to preferentially liberate minerals in the ore. The unit was developed in the 1980s and consists of two counter rotating rolls; one fixed while the other roll is movable. The floating roll is connected to a hydropneumatic spring system which applies a force on the roll which in turn applies a force on the particle bed (Solomon *et al.*, 2010). Hinde and Kalala (2009) have indicated that the breakage in an HPGR is different as compared to a tumbling or stirred media mill, as the grind size and specific energy have a nonlinear relationship. The authors attributed this to the greater grinding friction in HPGRs as the void volume in the particulate bed decreases, and therefore the energy component that results in the breakage is also reduced.

Solomon *et al.*, (2010) performed their tests on a Krupp Polysius HPGR unit on three platinum ore samples (Merensky, UG-2 and Platreef) at Mintek and compared their results to conventional ball milling technology. The flotation results for the UG-2 sample indicated that ball mill still resulted in better grades and recoveries as compared to the HPGR. The ball mill was also found to have a smaller fines product and the authors attribute this to the improved grades and recoveries in the ball milling circuit.

The authors found that the HPGR was useful in the primary circuit as it has the potential for significant increases in capacity for multiple stage circuits. This study indicates that conventional ball mills are still able to maintain high grades and recoveries as compared to alternative milling technologies and therefore further research should rather focus on optimising the separation process for high chromite UG-2 ores.

Palm *et al*., (2010) investigated the effect of using different comminution procedures for the flotation of platinum group minerals. The authors noted the recent benefits noted in industry of using HPGR technology especially in the iron ore and diamond industry. It was stated that few investigations to date have focussed on PGMs. The authors investigated the use of an HPGR and conventional crushing in combination with wet or dry rod milling and noted the batch flotation response. The author noted that the flotation grade and recoveries showed worse results when compared to conventional crushing and wet milling. Palm *et al*., (2010) indicated that the HPGR increased the platinum liberation; however, with no improvement in the platinum flotation due to a high degree of fines below 10µm that were produced, dry grinding resulted in a highly oxidising environment (high pulp potential) which negatively impacted on the flotation process.

Liu and Luo (2010) conducted a simulation study on two-stage grinding with high pressure roller mills on an iron ore material. The authors indicated that a two-stage process is more effective and recycling of the coarse product to the first HPGR results in a closer sized product with less ultra fines, a lower specific energy and a low circulating load. The authors indicated that this configuration could replace two or more one-stage grinding circuits that are run in parallel. The flotation response was not quantified despite the fact that this is often the main factor which determines the viability of particular circuit. The production of less ultra-fine material would normally reduce sliming during flotation and reduce reagent requirements, so it is definitely a good development in current grinding circuit design principles and should be investigated on a platinum mineral sample, with batch flotation tests on the final product.

Hinde and Kalala (2009) conducted optimisation tests on secondary milling circuits for UG-2 ore and found that through the use of fine screening technology, it was evident that the chromite was effectively liberated and virtually barren of PGM. The majority of the PGMs were associated with and locked in the silicate minerals (Hinde and Kalala, 2009). The authors also noted that the liberated chromite has a much higher density than the silicates and reports preferentially to the hydrocyclone underflow whereas the lighter silicates containing the locked PGM have a tendency to report to the hydrocyclone overflow and therefore do not undergo secondary milling. This clearly highlights the problem of the inefficiencies in the hydrocyclone circuit. This does indicate that separate milling would be the only solution if the hydrocyclone circuit is maintained in the existing plants.

The authors conducted several simulation studies of alternative circuit design options incorporating open and closed circuit configurations depending on the quantity of waste rock present. Several attempts have been made over the years to replace hydrocyclones with vibrating screens in UG-2 concentrators (Hinde and Kalala, 2009). The authors indicated that although the energy saving benefit was noted, the changes were not practical based on current technology. It was noted that a significant breakthrough was achieved with the introduction of new fine screening technology with the introduction of the Stacksizer by Derrick Corporation in the USA. These screens were non-blinding with high open areas and with a high life span as compared with conventional wire mesh screens. The authors also indicated that Mintek have recently conducted tests using a Derrick test rig for UG-2 applications.

Anyimadu *et al.*, (2006) investigated ultra-fine grinding with an IsaMill for Anglo Platinum Western Limb Tailings Retreatment (WLTR) plant in Rustenburg.

The circuit is shown in Figure 2.20.

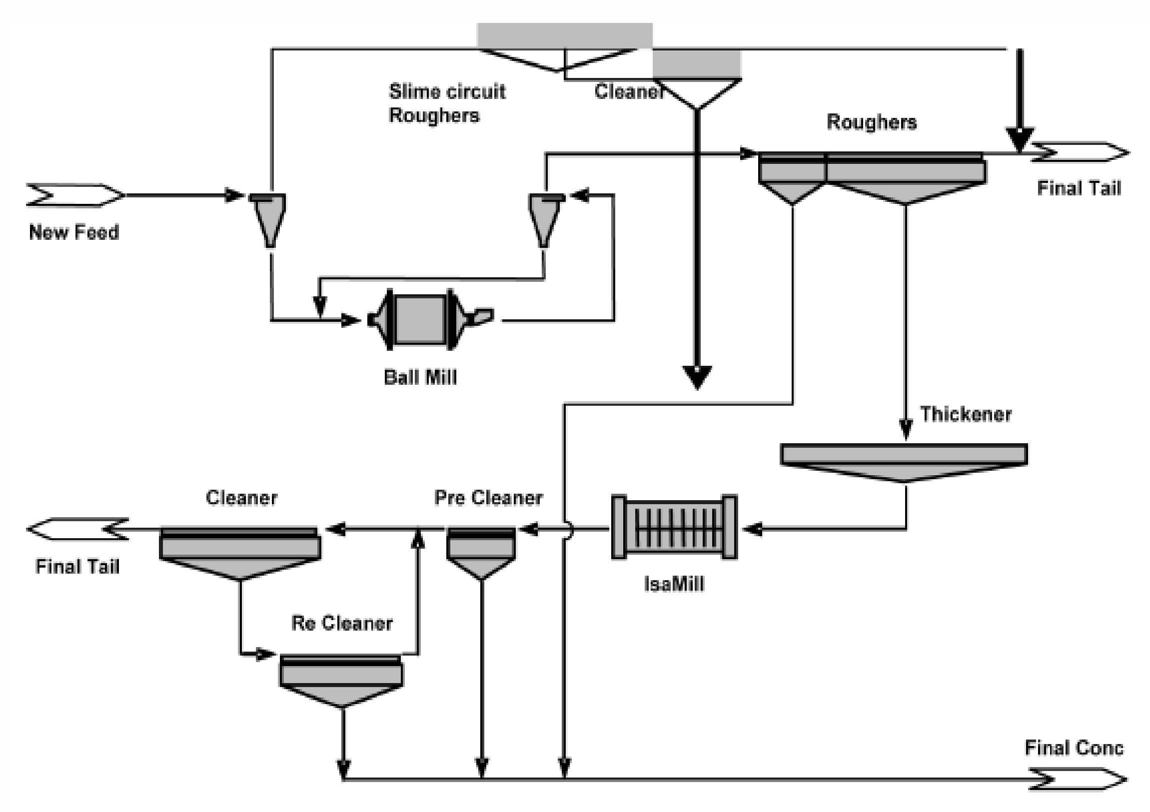


Figure 2.20: Simplified WLTR Flow sheet (Anyimadu et al., 2006)

A 10 000 litre 2.6 MW IsaMill was jointly developed by Anglo Platinum, Xstrata and Netszch for the WLTR project. The unit was run in open circuit on the tailings retreatment plant. The authors stated that the mineralogical and fractional assay data indicated that ultra-fine grinding (UFG) technology would improve the extraction efficiencies. Anyimadu *et al.*, (2006) indicated that the IsaMill technology achieved finer grinding (90% -51 µm) and better PGM flotation recovery (~3% 3PGE). The authors concluded that IsaMill technology operating costs were lower than expected and the operating experience at their other plants would determine the progress of implementation of this technology.

Rule *et al.*, (2008) conducted ultra-fine grinding tests on intermediate flotation concentrates at the Pt Mile plant on Anglo Platinum tailings. The authors investigated the use of 355kW Stirred Media Detriator (SMD) unit designed by Metso and column flotation cells. The diagram below shows the concentrator plant circuits for the Merensky and UG-2 ore.

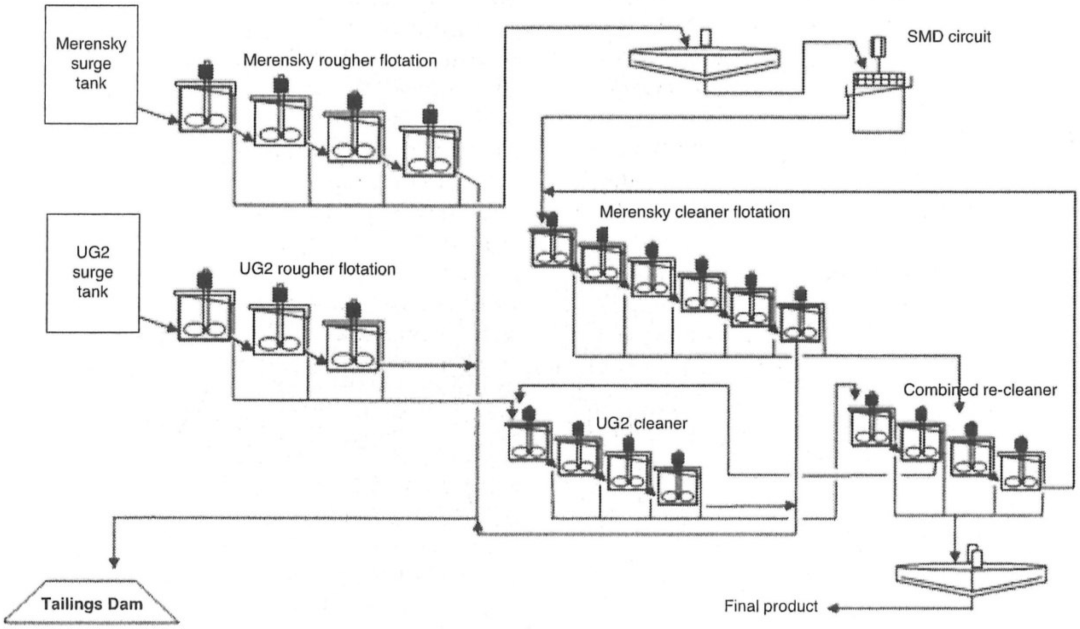


Figure 2.21: Application of the SMD^{TM} circuit to the Platinum mile plant circuits (Rule et al., 2008)

The SMD was able to operate at a throughput of 35tonnes per hour with about 6 tonnes of media. Anglo Platinum were also able to compare SMD technology with other UFG technology available on the market. The SMD exceeded its initial design specifications and confirmed its capability as a fine grinding unit. It was found to significantly improve the final concentrate grade as compared to historical data. With the application of a UFG feed thickener with the SMD they found a major improvement in their grades and recoveries. The recoveries more than doubled and the grades increased by 170%; however, one side effect of the UFG unit was that in the rougher concentrates a high entrainment of chromite was noted. The authors noted that the SMD was successfully implemented and the Pt Mile operation would also commission Deswik 2000 litre vertical stirred mills equipped with twin 350kW fluid drives. It was noted that larger milling capacity would improve the selective fine grinding of coarser fractions in the main feed stream. Column flotation technology was also successfully implemented to reduce the ever increasing chromite entrainment problem which increases with ultra fine grinding.

Northam Platinum has been successful at implementing new technology to reduce the chromite entrainment during flotation. The plant also employs an industry standard MF2 circuit for their UG-2 concentrator circuit. Substantial research was conducted on chromite entrainment and it was found that UG-2 chromite affects three areas, namely the overall platinum recovery, the upgrade of minerals and the amount of chromite in the final concentrate (Lanham, 2010). Lanham (2010) stated that if an attempt is made to recover 100% PGEs, the level of chromite entrained in the froth phase to the concentrate will be high, and if an attempt is made to eliminate chromite in the concentrate then the recovery of PGEs will be unacceptably low. Their flotation circuit optimisation and new technology development was led by Minnaar. Northam's original plant design estimated that an 85% PGE recovery could be achieved while limiting the chromite entrainment to an acceptable level of 2%. Until 2006 Northam Platinum achieved a 79% PGE recovery from their UG-2 concentrator with a 4.5% chromite entrainment.

It was therefore essential to blend this concentrate with the Merensky reef concentrate in a ratio of 4:1 (Merensky/UG-2) to maintain the smelter levels below 1.5%. Northam’s breakthrough in the chromite reduction technology emanated from industry discarded sparger column flotation cells with internal sparger designs as compared to conventional flotation tank cells. Northam designed and patented an external sparger arrangement to treat UG-2 as shown in Figure 2.22.

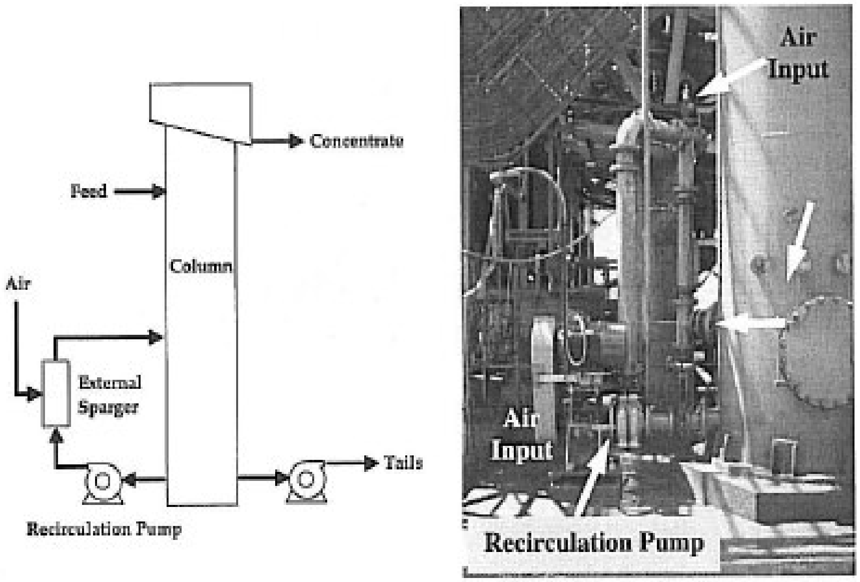


Figure 2.22: Northam column cells with external spargers (Hay, 2010)

This technology was installed in February 2004 and their chromite entrainment reduced by 44.4% from 4.5% to 2.5% and the PGE upgrade ratio improved by more than 9 times. A second column was added in 2009 running this in series mode with the other column cell. The overall UG-2 concentrate recovery improved in excess of 85% 3PGE+Au and final chromite entrainment was reduced below 2%.

Northam faced a further challenge of needing to maintain their overall production of the mine by producing more UG-2 as the amount of Merensky reef that was being mined was decreasing (Lanham, 2010). A rod milling process (Figure 2.23) was investigated and it posed a constraint as in comparison to conventional ball mills, the particles produced were of a narrow size range ;however, the capacity was low and had to be shutdown to replace rods at regular intervals (Lanham, 2010). The rod mill was a major defence against high chrome levels as it prevents over-grinding of the chromite fraction, which once finer, will be more easily entrained to the concentrate.

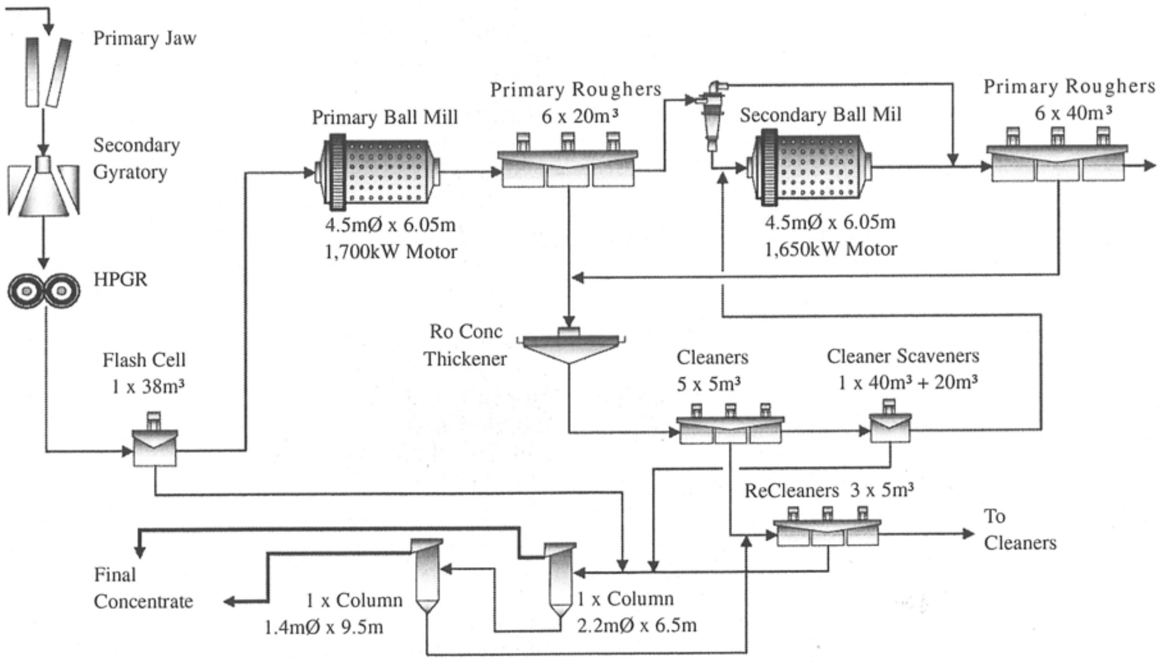


Figure 2.23: Northam UG-2 process flowsheet (Hay, 2010)

Northam therefore modified their milling circuit to incorporate the new technology in the form of a high pressure rolls crusher (HPRC) in series with a ball mill in their primary circuit with an open-circuit ball mill for the secondary circuit (Lanham, 2010). The primary ball mill was run with a lower ball load to limit grinding in the primary circuit. This new milling circuit design resulted in higher PGE recovery, a higher milling throughput (from 100 to 160 tonnes per hour) and the power usage was also decreased. PGE recoveries in excess of 85% were achieved and the chromite entrainment was below 2%. Northam are now able to blend UG-2 and Merensky concentrate in a 1:1 ratio and the research team are looking at eventually running UG-2 on its own in the future as the Merensky reserves deplete. At present the Northam UG-2 concentrator chromite in concentrate runs between 1.8 and 2% with a PGE recovery of 85%. This is the best in the western limb industry and has been predicted to reduce below 1.5% in the future (Lanham, 2010).

The research and development undertaken by the platinum industry into new technologies for UG-2 processing is clearly focussed on reducing chromite entrainment and improving the overall PGE recovery in the final concentrates. All the projects are guided by the smelter requirements and penalties faced for high chromite concentrates. New milling technology, fine screening below 75µm and flotation circuit optimisation are key areas. Each concentrator has to adopt technology that is specific to their UG-2 ore as the final grades and recoveries are governed by the origin of the ore.

Bryson (2004) stated that a significant effort is being made by the industry to understand the factors which control the design and scale up of a PGM milling and flotation circuit. The author indicated that to design a flotation circuit it is important to understand the mineralogical features that dictate the potential grades and recoveries that are achievable from a specific ore. Difficulties relating to processing different ores prompted the installation of new technologies in the platinum industry. Milling technologies such the Isamill (horizontal bead mill) were developed to treat poorly liberated base metal concentrates and larger mills are being installed to treat cleaner tails streams and possibly in the future even plant streams. Other technologies such as the Stirred Media Detritor and Pelorex mill are also being evaluated by the platinum mines. Plant process control strategies and flotation circuit optimisation are currently being investigated and that most process improvements are based on the amount of chromite in the platinum concentrate that the smelter can tolerate. New technologies in smelting such as the Conroast process claim to be more environmentally friendly and to tolerate high chromite levels; however, some adjustments to the base metal refineries are required. Future work may involve pre-concentration, effective classification, and evaluation of hydrometallurgical routes.

It is evident from literature that although the research into new milling technology is ongoing with mines such as Northam Platinum achieving excellent results through their flotation and milling circuit design, the conventional ball mill is still the favoured option as the industry standard for primary and secondary milling circuits (MF2) for majority of the platinum producers. HPGR technology has definitely shown a benefit of improved liberation and throughput in the primary UG-2 circuit; however various options still exist for the design of the secondary milling circuit for treating UG-2 ores with high chromite content.

Density separation of the chromite and silicates fraction of the UG-2 ore prior to secondary grinding and flotation seems to be the obvious choice for treating these chromite rich ores as many researchers have confirmed that conventional hydrocyclones are inefficient and result in a loss of platinum-bearing silicate particles to the secondary milling process. Current literature definitely indicates that platinum producers are targeting higher PGE recoveries in excess of 90% with low chromite entrainment (below 1%) through ongoing research and plant optimisation studies.

2.9. Spiral Concentrator Theory

This section provides the background theory to provide ann understanding of how spiral concentrators operate and their potential application to a PGM flowsheet.

Gravity concentration has typically been a low cost and environmentally friendly process for separating minerals (Richards *et al.*, 2000). The separation of two or more minerals by gravity is due to their relative movement in response to gravity, and one or several other forces (Burt, 1999). Recent studies suggest that many industrial processing plants that make use of density separators have the potential to significantly improve metallurgical performance through proper application of existing process engineering expertise (Lutrell, 2003). Onal and Guney (1990) noted that if gravity separation methods are used at chromite concentrators were the feed material is below 1mm in size then the material would tend to pass to the fine fractions according to their mineralogical structures and the ore-genesis. Shaking tables and spiral concentrators have been generally used after the crushing and grinding process at the concentrators, resulting in fine size chromite particles (-38μm) that are removed as tailings (Guney *et al.*, 2001).

A spiral concentrator is an example of a flowing film device where a thin layer of particles in water flows down a slight incline and is subsequently washed with a gentle flow of water where particles distribute themselves depending on density and size (Fuerstenau and Han, 2003). Spiral concentrators are simple units used to separate particles based on size, density and shape (Glass *et al.*, 1999).

The five main forces that act on the particles are gravity, centrifugal, drag, lift and friction forces (Das *et al.*, 2007). These forces determine the segregation of the particles and are shown in Figure 2.24.

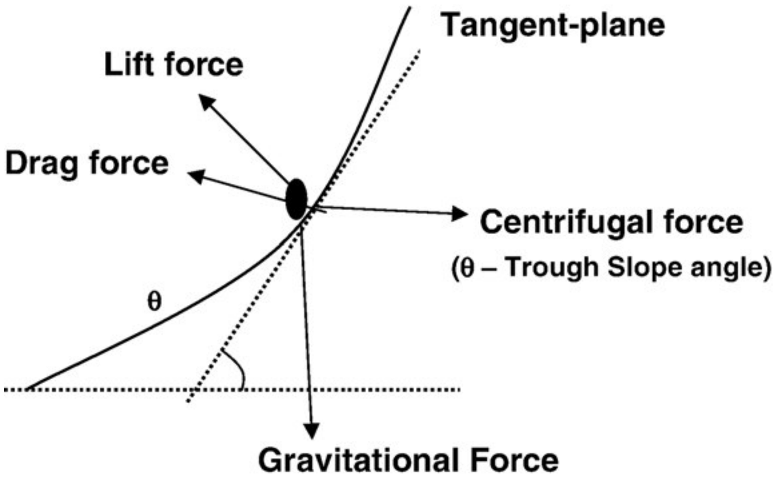


Figure 2.24: The different forces acting on a particle as it flows down a spiral. (Das et al., 2007)

The high density material moves to the inner edge of the spiral trough whereas the low density material moves to the outer wall of the spiral trough. Particle transport in a spiral concentrator trough occurs in two main ways: suspended-load motion, where the particles remain suspended in the upper layers of the fluid with minimal interaction, whereas in bed-load motion, inter-particle interaction is increased significantly (Atasoy and Spottiswood, 1995). Primary or down-trough flow results in the downward particle transport and secondary or cross-trough flow results in the cross-sectional rotation of particles (Atasoy and Spottiswood, 1995).

Kapur and Meloy (1998) stated that spiral separators exhibit the most complex flow regime of all gravity separators available. The fluid characteristics are governed by the Reynolds-averaged turbulent Navier–Stokes equations. The fluid flow is assumed to be Newtonian and to have constant physical properties (Das *et al.*, 2007).

Das *et al.*, (2007) indicated that the performance of a spiral concentrator is governed by the design parameters diameter, height, number of turns, pitch, slope, shape of the trough and its dimension as illustrated in Figure 2.25.

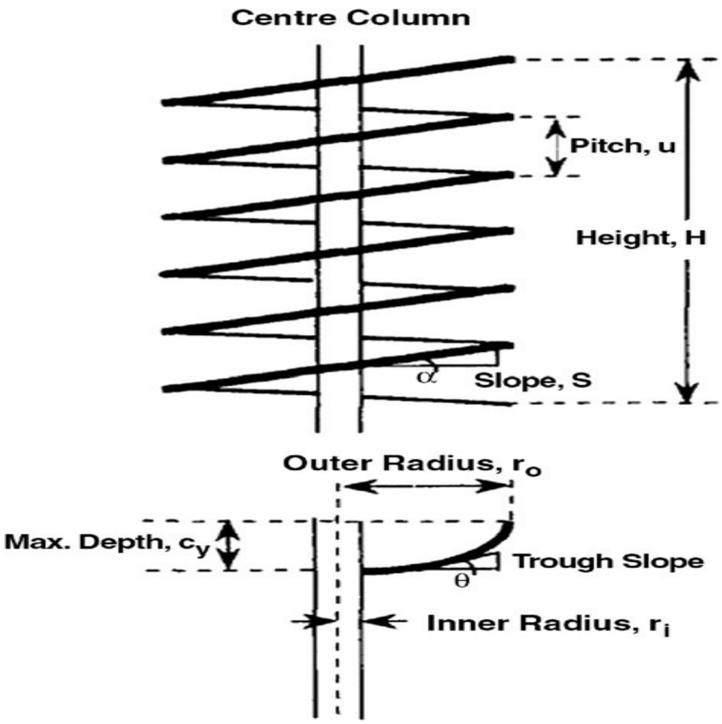


Figure 2.25: Design parameters of spiral concentrators. (Das et al., 2007)

Gulsoy and Kademli (2006) performed tests on the operational parameters of a Reichert spiral (model HG7) using a mica-feldspar ore (avg. SG=3.4). The authors investigated feed rate, percentage solids content, and splitter opening and specified the following optimised parameters for the Reichert spiral:

- Slurry Flow - 0.5 – 1.5 l/s
- Solids content - 15 %
- Splitter opening - Cut Point at 1/3 of splitter opening (~6cm)

Atasoy and Spottiswood (1995) indicated that Bagnold forces, which cause the particles to be dispersed and separate from each other, arise at high pulp densities. The authors noted that recent studies with spiral concentrators have indicated that Bagnold forces affect the particle transport in the inner region of the trough when the bed-load and pulp density is over 50%. The authors noted from their investigations on a coal sample that a one turn spiral is sufficient time for the separation of coarse particles (+1700μm) whereas five or six turns are required for dense mineral removal at small sizes (-850μm+300μm). The research showed that medium sized particles are normally contained along the cross section of the trough and that shorter length spirals are more effective when separating coarse particles and that the splitter position has a more pronounced effect than spiral length. However, it was emphasised that these observations may vary for different materials.

Holland-Batt (1995) studied the effect of design methods for spiral concentrators and showed that the fundamental fluid dynamics play a significant role in the selection of pitch and trough shape. It was noted that deflectors restore fluidity to the material where excessive accumulation of solids arises at high slurry densities and that the designs of the feed box, splitters and collection facilities affect operation and quality of separation. The flow pattern as indicated in Figure 2.26 is deduced from basic considerations of the hydrodynamics. The primary flow is downwards whereas the secondary flow is due to the greater frictional retardation on the lowermost layers of the primary flow with an inward and outward flow from the spiral centre column in the lower region and upper regions respectively.

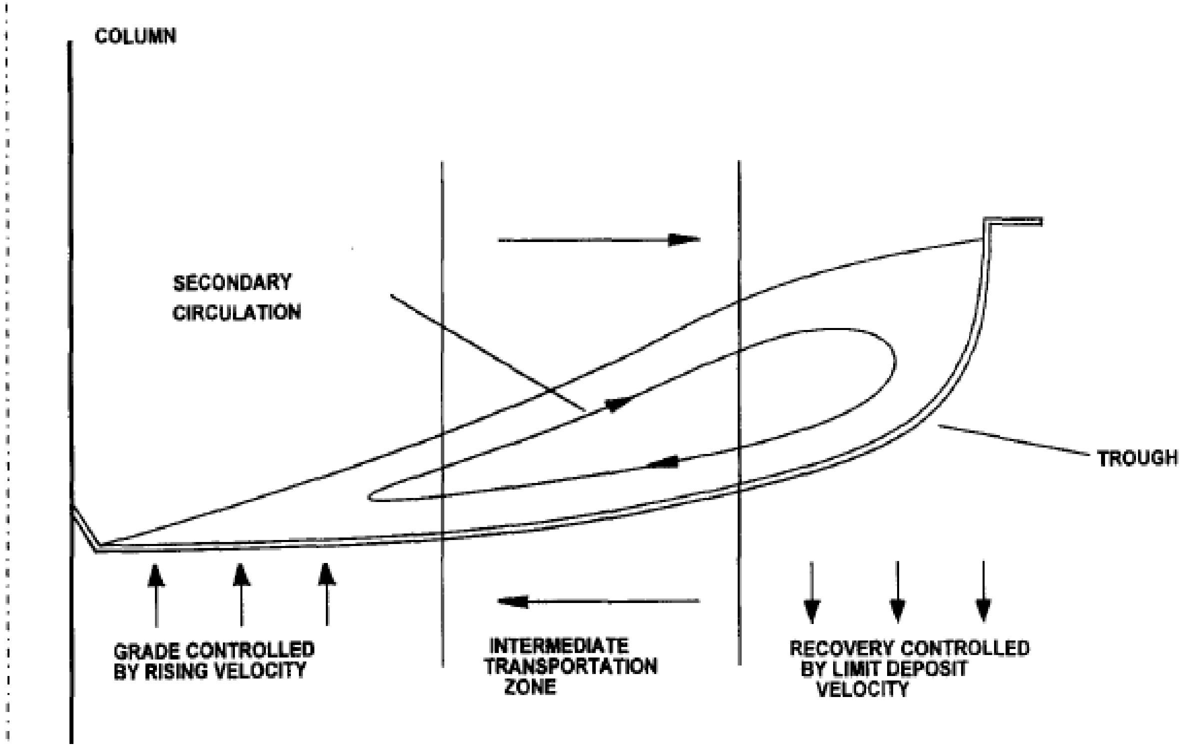


Figure 2.26: The separation mechanism (Holland-Batt, 1995)

Conventional spiral concentrators have a limit of around 2mm for coarse particles and a limit of 75µm for fine particles (Richards *et al.*, 2000). If fine particles are present, classification of these particles can occur and hence fine high density particles can be found with the low density particles. New spiral technology has been developed that is specifically designed to treat material below 0.1mm (Richards *et al.*, 2000).

Hearn and Sadowski (2003) stated that the separation efficiencies in spirals can be improved through the use of hydraulic classifiers for the feed or re-treatment of the first pass and in a second pass unit. This indicated that a hydrocyclone underflow stream in a platinum concentrator plant would be a suitable feed for efficient separation on a spiral circuit.

Holland-Batt (2008) stated that gravity separation of particles is affected by a combination of the hydrodynamics of the flowing slurry and the low inertia of the fine particles when the settling velocity is less than the rising effect of the slurry velocity. It was noted that the lower limit is about 75µm for heavy minerals and increases to about 125µm for coal.

PGMs formed with sulphides, arsenides, and alloys can be generally recovered by gravity separation (Xiao *et al.*, 2009). Xiao *et al.*, (2009) noted that PGM sulphide minerals, i.e. pentlandite, and chalcopyrite are readily recoverable by gravity; however, pyrrhotite is not readily recoverable. Although these investigations were conducted on Canadian platinum ore samples, it does provide some notable conclusions. The authors noted that the PGM loss is an important factor when justifying the use of a gravity circuit and that fine particles may not be gravity recoverable. This indicates that gravity processes may be viable in the primary stage of a PGM flowsheet when the grind is about 40% -75µm and that difficulties may be encountered if a spiral is utilised after a regrind stage (~80% -75µm).

The following section discusses recent advances in spiral technology which has made it more competitive on an industrial scale over the years. The various spiral designs and their industrial applications are noted.

2.10 Advances in spiral technology

Advances in spiral technology have improved the metallurgical performance, modernised the materials of construction, increased unit capacities and simplified operation to ensure that spirals continue to play a major role amidst other advancing and competing technologies (Palmer and Vadeikis, 2010).

Palmer and Vadeikis (2010) studied the new developments in spiral and spiral plant operations and found many advantages of spiral technology. The authors stated that besides mechanical stability, spirals are low cost in terms of capital and operation. Spiral banks cost less than the ancillary equipment such as feed sump, pumps and associated motors and electrical fittings. These units are environmentally attractive with a low energy consumption, use no reagents, require little training to operate, are visibly easy to monitor, and most importantly are able to tolerate varying feed conditions. Unlike other devices feed conditions on a spiral can change significantly with little impact on metallurgical performance (Palmer and Vadeikis, 2010). The authors noted that their separation efficiencies relative to processing costs are high making spiral separation the chosen processing option in many modern applications.

The feed capacity of these units have increased immensely over the years with the first spiral having a capacity of 1 to 1.5tonnes/h with a footprint are of 0.36m^2 whereas the latest technology can treat between 20 to 30 tonnes/h in a footprint area of 0.72m^2. This indicates that the new units are capable of handling the large feed tonnages experienced on UG-2 concentrators. This would also result in a cost saving as excessive spiral banks will not be required as in the past.

The efficiency of a spiral concentrator is best when the relative density difference between the materials to be separated is greater than one (Wills, 1997). Honaker *et al*., (2007) noted that the separation performance is referred to by the feed characteristics and operating conditions where the feed characteristics include the particle size by size weight distribution, density distribution, volumetric flow rate and the solid concentration. The authors conducted pilot and in-plant studies and found the spiral separator to be an effective technology for cleaning coal as fine as 44µm. the optimum percent solids was about 12% with a separation density of 1800kg/m^3, with a mass flow of about 0.6 tonnes per hour.

Palmer and Vadeikis (2010) stated that the current metallurgical practice is to base the separation efficiency of the spiral separator unit on the product grade, recovery and yield in relation to the feed grade.

It is defined as follows:

$$E_s = \frac{(R - Y)}{(1 - \frac{f}{ma})} \tag{14}$$

Where: E_s is the separation efficiency (%), R is the recovery of the concentrate (%), Y is the mass yield to concentrate (%), f is the feed grade (%) and ma is the maximum possible assay for concentrate (%).

Spiral designs vary according to feed characteristics and a major design consideration is the average specific gravity of the particles (Palmer and Vadeikis, 2010). Spiral concentrators have been mainly used in the mineral sands treatment industry. Mineral Technologies have designed several new spiral separators to suit their feed categories, such as LHM (light heavy mineral), VHG (very high grade), and HC33 (high capacity). The VHG and HC33 spiral are some of the latest developments in spiral technology. Both spiral separators are able to treat high density feeds. This VHG spiral is able to create a more vigorous flow to keep the particle bed mobile enough for the separating mechanisms and also disrupts the influence of turbulence and mixing (Palmer and Vadeikis, 2010). The HC33 spiral is a high capacity wash water spiral for chromite, iron ore and final cleaning stages.

Richards and Palmer (1997) have summarised some applications of some of the high capacity spiral concentrators, namely the high capacity coal spiral concentrators and the high capacity mineral spiral concentrators. The authors stated that significant advances of spiral technology have taken place since their origin in the 1950's. These improvements have focussed on metallurgical performance or separation efficiency.

High capacity coal spirals:

These units have wide diameters, high capacity, wash waterless fibreglass/urethane spiral for coal cleaning and special mineral applications involving shape separation. Splitters at different points down the unit give this spiral concentrator a high capacity to remove waste ore and siliceous contaminants from coal or mica.

High Capacity Mineral Spirals:

The MG2 model spiral was developed for high capacities, primarily for roughing. The authors indicated that it had the following advantages:

- A large feed capacity based on a specific plant area resulting in a lower capital expense
- A simple feed distribution and product launder
- More operator control over the process

The MG2 occupies over twice the area of a conventional low capacity spiral such as the MG4 to equivalent separation efficiency.

The developments in spiral separator design are a testament to the ongoing use of spiral technology in current minerals processing applications. These units have been modified to compete with the advanced gravity separators whilst still maintaining their low cost, easy maintenance and environmentally friendly status. In addition, these units have been designed to handle varying feed compositions, high density application and most importantly higher plant capacities to ensure that they are able to meet the current production demands. Current spiral technology suggests that these units may be incorporated into PGM flowsheets without any major plant modifications. These units may provide a low maintenance solution to separating the chromite and silicate particles prior to secondary grinding.

2.11. Conclusion

The previous literature has provided significant knowledge into current strategies for processing chromite rich UG-2 ores.

The ore mineralization does indicate that PGMs are predominantly associated with the base metal sulphide minerals. PGMs were found to be interstitial with the chromite minerals, which indicated that separate grinding and attritioning may be beneficial design options. There was also evidence that indicated that some PGMs are locked within the chromite minerals (laurite);

however, this was not quantified. The average PGM grain size in UG-2 ore was found to be about 9µm therefore the option of achieving finer grinding was targeted through energy redistribution to the silicates to liberate any locked PGMs.

Since the UG-2 secondary ore sample was predominantly -500mm microns, it was decided to incorporate the equation derived by Bond (1961) in Section 2.4 to determine a ball size choice for the laboratory test work campaign. This equation was used to predict the maximum ball size required to break the largest UG-2 particles. These parameters were selected for the grinding tests and remained constant for the duration of the project.

The impact of a high chromite content in the flotation concentrate on smelting (Section 2.2) was quantified for each milling circuit design option through batch rougher flotation tests. The optimum flotation conditions for the sample of UG-2 ore were supplied by Lonmin Platinum based on their standard laboratory flotation method. This reagent suite consisted of a xanthate collector, a copper sulphate activator, a talc depressant and a frother. The activator and depressant were omitted from the reagent suite to quantify the amount of PGEs and Cr_2O_3 recovered for each configuration.

The optimised spiral concentrator parameters of Gulsoy and Kademli (2006) in Section 2.9 were used as base case conditions when optimising the separation of chromite and silicates for the UG-2 platinum ore sample as it also had an average SG between 3.2-3.4.

The following section describes the laboratory and pilot plant experimental test work for developing a robust secondary ball milling circuit design.

CHAPTER 3: LABORATORY AND PILOT PLANT TEST WORK CAMPAIGN

This chapter provides an overview of the laboratory test work campaign and the rationale for each secondary milling circuit configuration that was investigated in this project. The experimental procedure and specifications for the confirmation test work on the pilot plant, spiral circuit run is also covered in this chapter. Detailed equipment specifications and experimental procedures are shown in Appendix A.

The standard UG-2 concentrator MF2 (Mill-Float-Mill-Float) circuit is shown in Figure 3.1. The MF2 circuit targets a primary grind of about 40% -75µm and a secondary grind of about 80% -75µm. There are two stages of flotation with the primary flotation treating most of the high grade, fast floating platinum minerals that were liberated in the primary ball mill.

An approximate distribution of the 4E (Pt, Pd, Rh, Au) and Cr_2O_3 minerals is given in Figure 3.1. About 65% of the 4E are recovered in the primary float with about 1% entrainment of Cr_2O_3 as the grind is quite coarse. About 35% of the 4E is contained in the primary rougher tailings (PRT) which is then densified, with the densifier underflow stream sent for regrinding to liberate more platinum minerals. The densifier overflow contains ultra-fines and some short-circuited silicate and chromite minerals. This stream is usually recombined with the secondary mill discharge to form a secondary flotation feed stream (SFF).

This secondary ball mill is run in an open circuit configuration to reduce over-grinding of chromite minerals; however, since a fine grind is targeted, a fair amount of chromite minerals are liberated. About 55 to 60% of the platinum minerals in the secondary flotation feed are recovered in the secondary rougher concentrate. Both low and high grade rougher concentrates are combined and further concentrated in a cleaner flotation circuit to produce a final concentrate of about 85% 4E recovery and with about 3% Cr_2O_3 entrained.

Figure 3.2 shows the standard UG-2 secondary circuit configuration. This was used to benchmark the current process for each sample of ore received from Lonmin Platinum and the results were compared with all the new secondary milling circuit designs that were investigated.

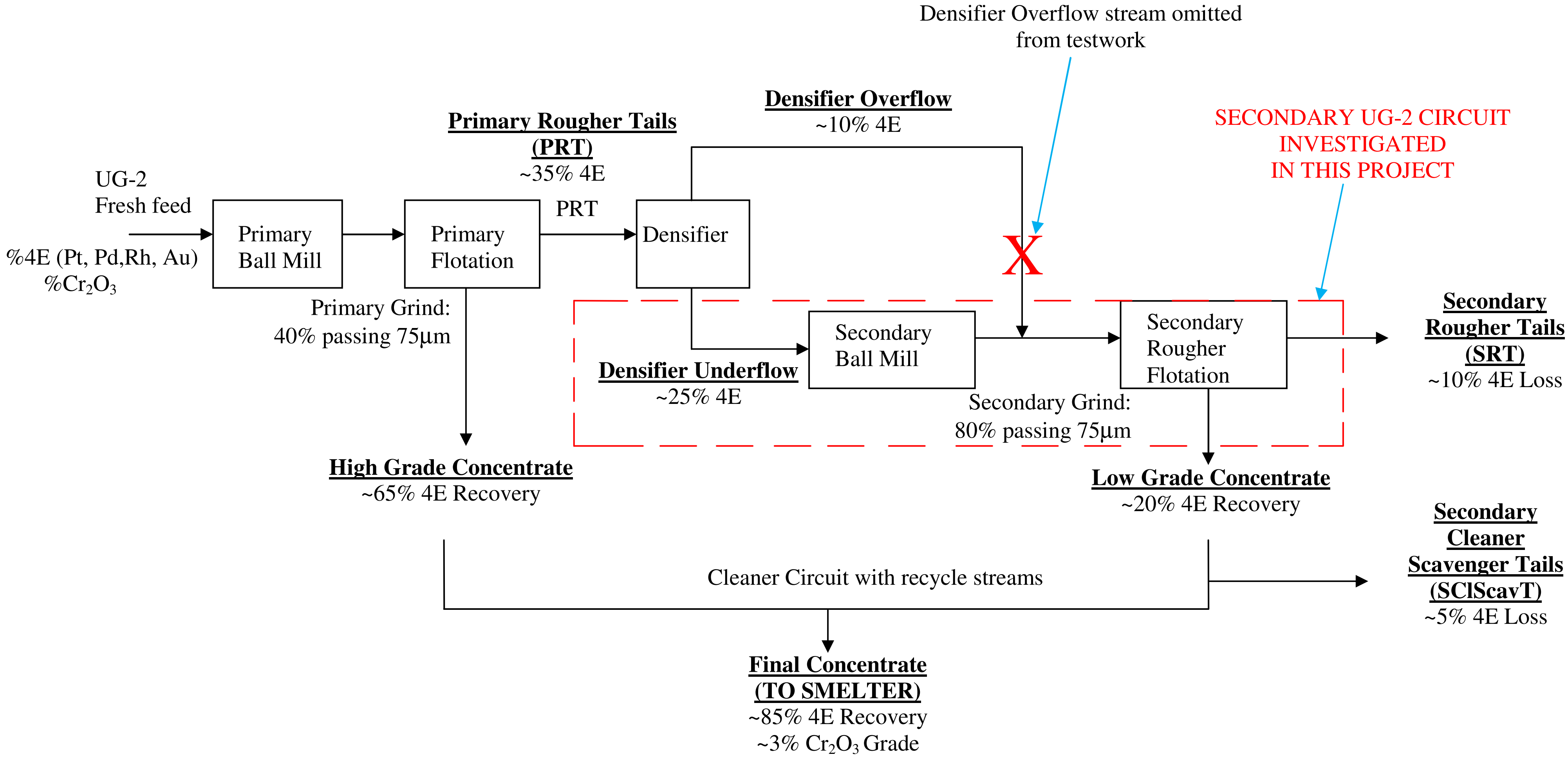


Figure 3.1: Standard UG-2concentrator MF2 flowsheet

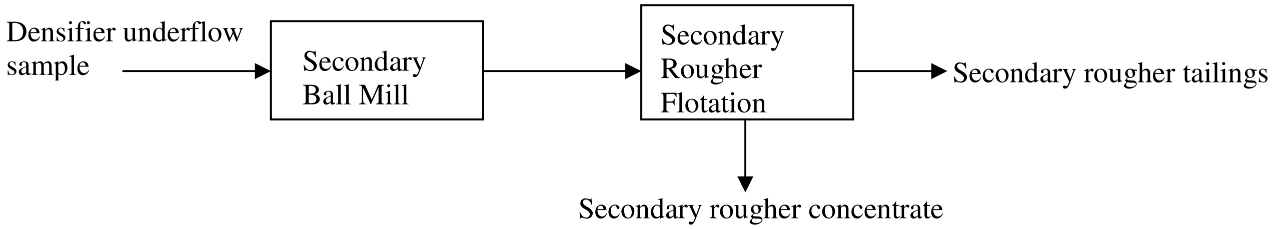


Figure 3.2: Standard secondary circuit

This project was focussed on the secondary milling and rougher flotation circuit only of the PGM concentrator flowsheet as the main source of chromite entrainment is after secondary grinding. A densifier underflow sample was used for the laboratory and pilot plant test work campaign Since the densifier overflow stream, which contained roughly about 10% of PGEs, was not considered in this study, it must be noted that secondary circuit only represented about 25% of the overall PGEs that reports to the secondary milling and flotation circuit. The densifier underflow is the feed to the secondary ball mill and therefore it was decided to base the optimisation test work on this stream. This study was focussed on improving the recovery of the PGEs and reducing the recovery of the Cr_2O_3 minerals present in the densifier underflow stream.

Standard rougher flotation tests were conducted to determine the effect of each milling circuit design, and comparisons were based on the improvement in the secondary rougher recoveries of 4E and Cr_2O_3. It must be noted that secondary cleaner and re-cleaner flotation tests were not considered in this project. This would require a more complex flotation circuit arrangement with the use of depressants and cleaner tailings recycle stages to determine a true reflection of the recovery in the final concentrate. Locked cycle tests would have been required and this would have increased the complexity of the experimental work and repeat analyses. In addition, several cycles would be required to reach steady state.

The following section gives a brief description of the plant audit of a secondary UG-2 concentrator.

3.1. Plant Audit – Platinum Concentrator Plant

3.1.1. Sampling Campaign

A plant visit was undertaken on 10th April 2008 to a platinum concentrator at Lonmin Platinum to survey the plant, determine the relevant sampling points and also to prepare a sampling schedule for the secondary milling and rougher flotation circuit.

A five day campaign was thereafter conducted from 21th to 25th July 2008 to collect samples of the secondary milling and flotation circuits. These samples were required for detailed size by assay analyses for the secondary circuit of the plant. A bulk sample of about 250kg of the densifier underflow stream, which is the feed to the secondary ball mill, was also collected as this was required for all the laboratory milling and flotation tests for the project.

The flotation concentrate streams could not be sampled as a bulk stream, since the mixing point was in a large launder which flowed directly into the concentrate tank, and this was then pumped directly to the cleaner circuit for further mineral upgrade. This was combined and treated as a bulk sample.

A diagram of the secondary milling and rougher flotation circuit at Lonmin Platinum is shown in Figure 3.3. The secondary rougher concentrates had 10 banks of cells, with a total of 16 sampling points present; however, the Secondary Rougher Concentrate 8 (SRC 8) was offline during the campaign thus 15 stream samples were taken.

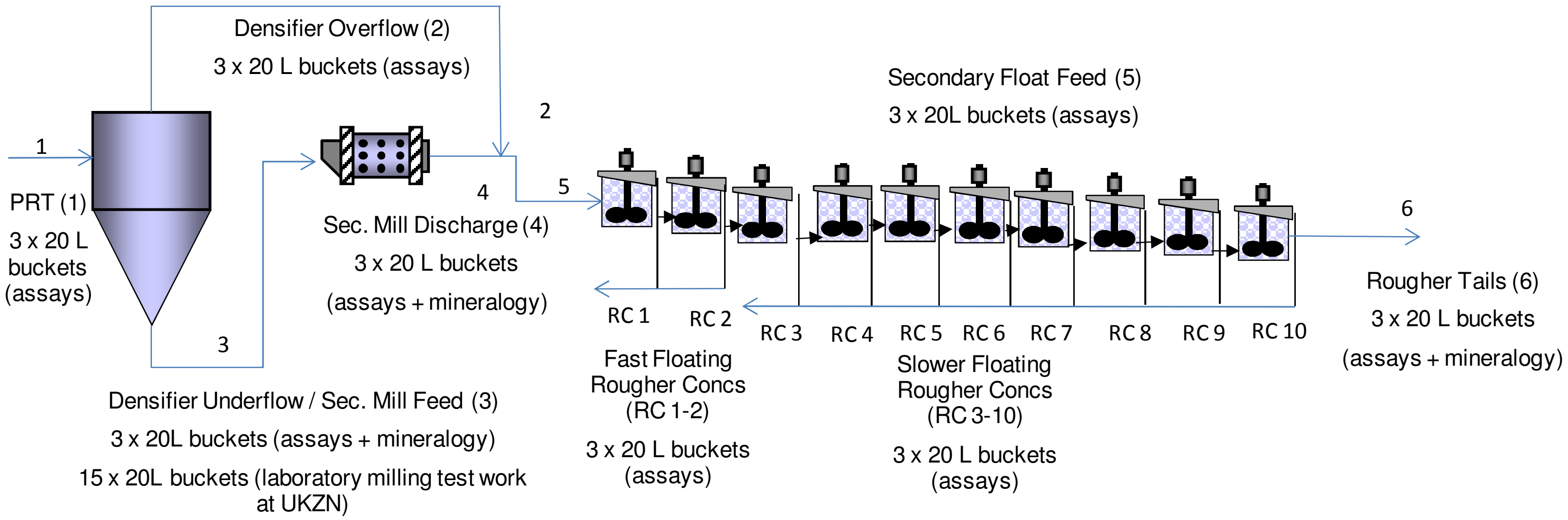


Figure 3.3: Concentrator Plant Secondary Milling and Rougher Flotation Circuit

A sampling team assisted in taking the samples for the size-by-assay analyses. The steady state operation of the plant was determined by the plant metallurgist. A sample cutter was used to collect a representative sub-sample of the stream every hour for a three hour period, to minimise inconsistencies due to any plant instabilities that could affect the metal concentrations. Twenty litre buckets were used and each bucket was large enough to accommodate three cuts. The sampling was carried out in triplicate (Sample A, B, C) for each stream, so that additional sample sets were available if errors were noted in the assay results and to quantify the variability between the samples. Untimed samples were taken as the data was not required for a metallurgical balance. Samples A and B were sent to Mintek for sizing analyses and this data is present in Appendix B3. Sample set C was sent to UKZN for sample preparation and sizing. The assays were conducted at Mintek's Analytical Services Division.

The detailed sample preparation for the plant audit circuit samples and densifier underflow bulk sample are shown in Appendix A1.1 and A1.2 respectively.

3.1.2. Detailed Mineralogical Evaluation

A detailed understanding of the mineralogy of an ore-body is significant when improving the efficiency of the concentrator plant milling process. QEMSCAN (Quantitative Evaluation of Minerals by Scanning Electron Microscopy) analysis has been shown to readily provide separator performance as a function of particle size and density (Pascoe *et al*., 2007). Mineralogical evaluations were conducted on a secondary rougher tailings (SRT) plant sample.

The SRT sample was submitted to MINTEK for a PGM search and a bulk modal analysis. The details of this method are provided in Appendix A2.

The mineralogical test work was conducted to provide the following information:

- PGM species and size distribution
- PGM liberation and floatability characteristics
- Minerals associated with the PGMs
- Bulk modal results, i.e. minerals identified and their relative proportions. Mineral association- i.e. the manner in which the various phases are associated with each other.

3.1.3. Heavy Liquid Separation (HLS)

The purpose of the HLS tests was to determine whether the high density chromite minerals contained a large proportion of platinum bearing particles. Tests were conducted on two coarse size fractions of the densifier underflow stream (-212+150µm and -150+106µm fraction). Since the ore had been through a primary milling process, it was already quite fine (40% -75µm), and therefore high density solutions in excess of 2.96g/cm^3 could not be used, as it would require long settling times (Section 4.1.2). The Physical Separation Group at Mintek therefore put together a test work procedure to treat the samples provided. A detailed description of the Mintek HLS method is shown in Appendix A3.

All sub-samples were analysed for Cr_2O_3, SiO_2 and Fe_2O_3 using ICP-AES (Inductively Coupled Plasma - Atomic Emission Spectroscopy) and for 4E (Au, Pt, Pd, Rh) in duplicate utilising the fire assay method.

3.2. Laboratory Equipment

The main laboratory equipment required for the project was as follows:

- Full scale MG-type rougher and cleaner spiral concentrators
- A 200mm diameter by 300mm length fixed speed (70rpm) ball mill
- A Denver flotation machine with 4.2 litre flotation cell
- A Vibrating screen shaker
- Screens (106, 75, 53, 38 µm)
- Pressure filter
- Sampling tools i.e. Jones Riffle

A picture of the full scale MG-type rougher and cleaner spiral concentrators are shown in Figure 3.4. Pictures of the other laboratory equipment are shown in the Appendix A4.

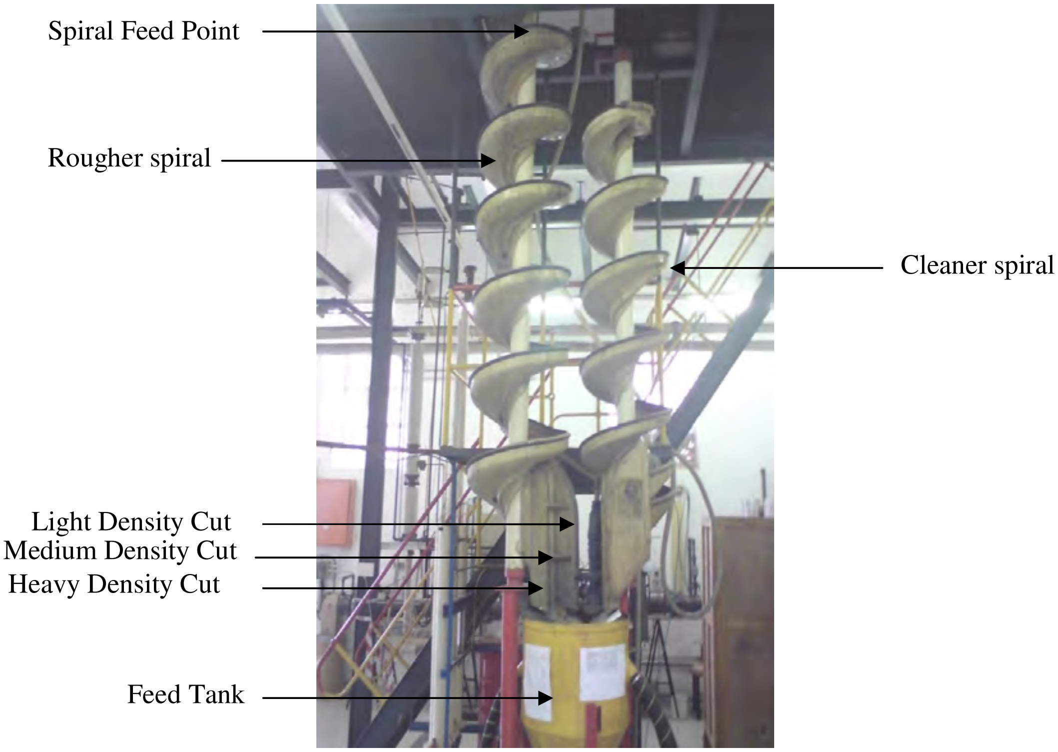


Figure 3.4: MG type rougher and cleaner spiral concentrators

3.3. Effect of varying proportions of dense minerals[1]

The first research question to be investigated in the laboratory was whether preferential grinding of silicates or chromite particles was occurring in a batch ball mill. A series of laboratory tests were conducted to investigate these effects. A sample of ilmenite derived from beach sand (supplied by Richards Bay minerals) and a sample of limestone (Kulu 200C) of a similar size range was available in the laboratory These samples were used for the laboratory investigations as the densities were similar to that of the silicates and chromite particles.

Ilmenite (ρ=4600kg/m^3) was used to simulate the chromite particles and limestone (ρ=2650kg/m^3) was chosen to simulate the silicate particles in the batch mill. It was also quite simple to determine the proportion of limestone and ilmenite in each size interval as most of the limestone (85% dissolution for this particular bag of Kulu 200C) dissolved in an excess

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[1] Maharaj,L., Pocock, J., Loveday, B.K., Hinde, A., 2009, Density segregation and selective grinding of a synthetic UG-2 ore mixture in a batch laboratory ball mill, Proceedings of the South African Chemical Engineering Congress, Cape Town, 20-23 Sept. 2009

stoichiometric amount of hydrochloric acid (HCL), and therefore, analyses were quick, and the results could be easily processed. The work indices were also between 12.8 – 13.3 kWh/t which was similar to the UG-2 ore being studied from the sample information provided by the platinum producer.

Varying proportions of these high and low density minerals were used to determine whether selective grinding of the high or low density mineral occurred, and the effect of density segregation under batch milling conditions.

A variable speed batch laboratory mill of 30cm in diameter and 30cm in length was used for the experimental work. The length to diameter (L/D) ratio was 1.

The mill specifications were fixed for all the runs and are shown in Table 3.1 below.

Table 3.1: Mill base case specifications

Mill diameter	0.30m
Mill length	0.30m
L/D ratio	1
Volumetric loading of total charge	40%
Ball size	27mm
Mill rotational speed	70rpm
Milling Time	20min

The table below represents the feed samples used for the milling tests:

Table 3.2: Data on feed samples

Limestone	Ilmenite
W_i = 12.8 kWh/t	W_i = 12.3 kWh/t
ρ = 2650kg/m^3	ρ = 4600 kg/m^3

The effects of seven volumetric feed ratios were investigated on the “as is” (un-sized) synthetic ore mixture as shown below in Table 3.3.

Table 3.3: Volumetric ratios of "as is" synthetic ore feed mixtures

Ratio	Volume % of ilmenite	Volume % of limestone
1	100	0
2	80	20
3	60	40
4	50	50
5	40	60
6	20	80
7	0	100

Table 3.4 shows the tests done on the -300µm ilmenite and limestone feed samples

Table 3.4: Volumetric ratios of "-300µm" synthetic ore feed mixtures

Ratio	Volume % of ilmenite	Volume % of limestone
1	50	50
2	80	20

The experimental procedure for the "as is" and -300µm feed samples of ilmenite and limestone was similar. In the latter tests on the screened samples, bulk feed samples of "as is" limestone and ilmenite were wet screened at 300µm, and the -300µm fraction was filtered, dried and stored for the tests. The detailed experimental procedure is shown in Appendix A5.

3.4. Slurry short-circuiting ("pooling") tests

Short-circuiting (pooling) effects of silicates particles in a UG-2 ore have been mentioned by Bryson (2004), Hay and Schroeder (2005), and Hinde and Kalala (2009). Hinde and Kalala (2009) and Bryson (2004) have hypothesised that silicate particles float above the grinding zone, bypass the grinding process and exit the mill, thereby resulting in poor liberation of the platinum bearing particles. Slurry pooling tests were therefore investigated using two separate test methods.

In method one, two laboratory mills (570mm and 300mm diameter) were fed with a UG-2 densifier underflow plant sample based on a 40% volumetric loading of media and pulp. The 570mm diameter ball mill was run for five minutes and then "crash stopped" and a sample from the top of the pulp was removed from its loading port. This sample was filtered, dried and subjected to a "sink-float" analysis using pure tetrabromoethane (TBE). A second laboratory mill (300mm diameter) as shown in Figure 3.5 was run with its water-port open i.e. in a semi-batch mode and samples collected from the mill discharge were also filtered, dried and subjected to sink float analysis.

300mm diameter mill
L/D Ratio = 1.0

200mm diameter mill
L/D Ratio = 1.05

Figure 3.5: Cylindrical mills of varying L/D ratios for experimental work

TBE has a specific gravity (SG) of 2.96 and was used to determine the proportion of low density material in the samples that were collected. The specific gravity of the silicate particles is about 2.65 with the chromite particles having an SG of 4.6. Since a standard fresh feed of perfectly mixed UG-2 ore has an SG of 3.2-3.4, if the collected samples contain a significant proportion of silicates, then they would have an SG of below 2.96 and would thus float on the TBE heavy liquid. This was used to quantify the effect of slurry pooling.

In test method two, a 200mm diameter mill (Figure 3.6) was loaded with 1kg of ilmenite and 1kg of limestone. The media and pulp loading was 40% of the total mill volume. The mill was allowed to run for 10 minutes at 70rpm and then crash stopped. The lid was opened slightly so that the most of the water in the mill could drain out slowly without displacing any solid material. Once this was completed, a thin metal sheet was used to slice the solid bed into three slices i.e. top layer, middle layer, and bottom layer. A sample of material on the left and right side-walls of the mill cylinder were also collected as separate samples as shown in Figure 3.6. All samples were dried and analysed to determine the proportion of limestone and ilmenite present, and whether a distinct density segregation profile existed in the mill.

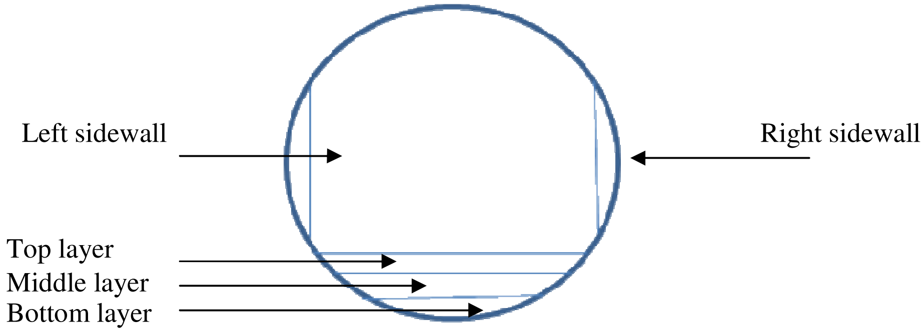

Figure 3.6: Slices of solids removed from "crash stopped" mill (The mill was rotating clockwise)

These tests may determine whether the slurry "pooling" phenomenon could be evaluated on a batch scale and whether an equal mixture of high density and low density material showed evidence of internal density segregation effects.

3.5. Laboratory commissioning tests on a rougher spiral concentrator using a synthetic UG-2 mixture

The following section focussed on determining the effect of pre-concentration with the use of a gravity separation device, i.e. a spiral concentrator to separate the UG-2 ore into a chromite-rich (heavies fraction) and a silicates-rich (lights fraction). In order to achieve a good separation it was decided to use the synthetic ore mixture of ilmenite and limestone to commission the existing full scale MG-type spiral with base case operating conditions, as determined by Gulsoy and Kademli (2006).

The detailed laboratory procedure is provided in Appendix A6. This procedure was similar for all tests on the spiral concentrator. Table 3.5 shows the commissioning tests that were conducted to determine the optimum operating condition for the synthetic ore.

Table 3.5: Effect of varying the operating parameters

Parameters	**Variations**
Feed Slurry Flow (l/s)	0.40, 0.48, 0.55, 0.58, 0.66, 0.78, and 1.20
% Solids	10, 15, 20, and 30
Splitter Opening (cm)	3, 6, and 9
Particle Size	-300 microns

Once the optimum operating conditions were determined from these commissioning tests using the ilmenite and limestone mixture, subsequent runs were conducted on the UG-2 densifier underflow sample from the concentrator plant to confirm these conditions. The product samples collected from these runs were analysed for copper and nickel using an ICP analyser. The acid digestion technique shown in Appendix A7 was developed as part of this project for the analyses.

The results were then analysed to determine the base metal recoveries (copper and nickel) which can be related to the amount of platinum minerals recovered in the spiral products, as the base metal sulphides are associated with platinum-bearing minerals. These results gave an indication on the effectiveness of the spiral concentrator for treating UG-2 ore and the following section focused on determining the effect of incorporating the spiral separator into a secondary milling circuit configuration.

3.6. The effect of the design of a secondary grinding circuit on platinum flotation from a UG-2 ore[2]

The envisaged milling circuit configuration is shown in Figure 3.7.

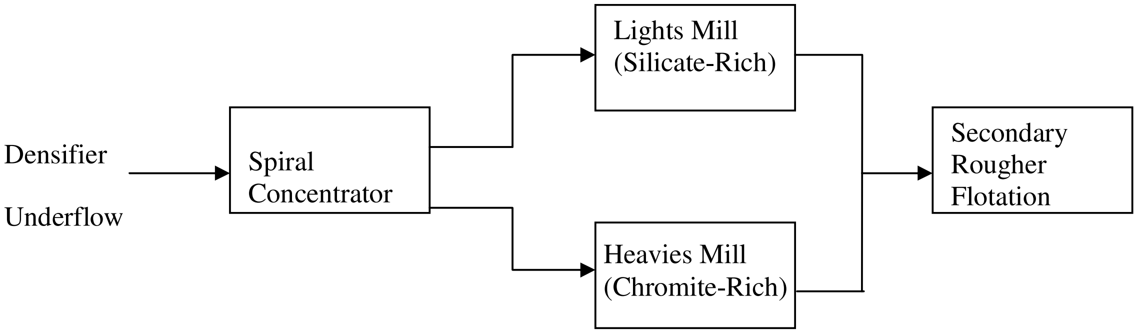

Figure 3.7: Envisaged UG-2ore secondary milling circuit configuration

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[2] Maharaj, L., et al., 2010, The effect of the design of a secondary grinding circuit on platinum flotation from a UG-2 ore. Miner. Eng., doi:10.1016/j.mineng.2010.08.007

The circuit incorporates the separate milling technology employed by Anglo Platinum (Rule and Anyimadu, 2007); however, a spiral concentrator was used as the separating device, and combined rougher flotation was conducted so as to minimize additional changes to the existing secondary flotation circuits on concentrator plants.

3.6.1 Feed sample preparation

A 20kg densifier underflow sample (high grade 4E sample) from a UG-2 platinum ore concentrator plant was separated on the spirals concentrator at optimised laboratory settings determined from a series of commissioning tests to produce a lights (silicate-rich), and a heavies (chromite-rich) fraction. The densifier underflow had a size distribution of 60% -150µm with a top size of 425µm. This appeared to be ideal for efficient use on a spiral based on criteria by Richards *et al*., (2000) (Section 2.9). The spiral had a mass split of 45% to the lights and 55% to the heavies with a platinum 4E analysis revealing that 65% of the platinum group minerals reported to the lights (silicate-rich) fraction. For simplicity it was decided to use a 50: 50 split of the spiral products when conducting the batch milling tests. These results are discussed in Chapter 4 as the 100% lights (100% L) and 100% heavies (100% H) separate milling option.

An alternative option of varying the amount of dense material in the feed composition to the mill was also investigated. The UG-2 densifier underflow spiral lights and heavies products were combined in a ratio of 80% heavies and 20% lights in one mixture and in a ratio of 80% lights and 20% heavies in a second mixture for separate milling tests. The correct mass proportions were prepared to achieve the desired volumetric ratios. These results are discussed in Chapter 4 as the 80/20 L-H and 20/80 L-H separate milling option.

3.6.2 Experimental Procedure

The detailed experimental procedure for the UG-2 spiral run is shown in Appendix A8. Separate batch ball milling tests were conducted on the spiral lights and heavies products keeping the overall energy input to the system constant. Sixteen minutes of milling time was required to mill a 500g hydrocyclone underflow feed to 80% -75µm using a single ball mill under the conventional open circuit milling conditions. The separate milling tests were therefore conducted on a fixed mill feed of 250g to each ball mill and the total energy was re-distributed between the lights and heavies mill based on the variations in the milling times as shown in Table 3.6.

Table 3.6: Milling time

Milling Energy (min)		Total Time (min)
Lights (Silicate-rich)	Heavies (Chromite-rich)	
14 (88%)	2 (12%)	16 (100%)
12 (75%)	4 (25%)	16 (100%)
10 (63%)	6 (37%)	16 (100%)
8 (50%)	8 (50%)	16 (100%)
6 (37%)	10 (63%)	16 (100%)

The milled products were then combined to form a 500g sample for the rougher flotation tests. Flotation tests were conducted in a Denver flotation machine with an impeller speed of 700rpm and conditioned for 3 minutes. A simple reagent suite consisting of a standard sulphide mineral collector (300g/t SIBX) and a frother (40g/t XP 200) were used for the rougher flotation tests at a natural pH of the ore. Analyses were carried out on the flotation concentrate and tailings for 4E (Pt, Pd, Rh, Au) by a fire assay method and Cr_2O_3 by ICP analysis. The resulting recovery data were plotted and compared. Repeat milling and flotation tests were conducted on a low grade 4E sample for the 100% L and 100% H option. The water recoveries were recorded for the low grade sample.

3.6.3 Sizing of mill feed and products

All mill feed and products of the secondary ball milling design options were sized at 106, 75, 53, and 38µm to compare their product size distributions and effect on particle breakage. Samples were first wet screened at 38µm and the +38 µm fraction was dried and screened to determine an accurate size distribution.

Selected milling samples, i.e. standard ball mill test, separate milling with equal energy input of 8 minutes to lights and heavies mill, and the best circuit configuration option were submitted for chemical analyses for 4E and Cr_2O_3 to determine the metal distributions in each size fraction.

3.7. Attritioning Tests

Literature by Lidell (2009) indicated that attritioning (Section 2.6) may have future benefits in the platinum industry, thus some scoping attritioning tests were carried out on the rougher spiral heavies samples (100% H) at two different percent solids, i.e. 30 and 40%. Each sample was attritioned using a Denver attritioning unit at 500rpm for five minutes. These samples were then filtered, dried and screened into five size fractions. The samples were sent to Mintek for 4E and Cr_2O_3 analyses to determine whether the platinum minerals could be lightly scrubbed off the chromite particle surfaces in a laboratory scenario. Duplicate flotation tests were conducted by separately milling a 250g spiral lights sample in a ball mill based on the best milling option determined in Section 3.6, and attritioning of a 250g spiral heavies fraction for 5 minutes at 40% solids followed by combined rougher flotation based on the standard laboratory method. The samples were filtered and submitted for 4E and Cr_2O_3 analyses.

The milling circuit configuration is shown in Figure 3.8.

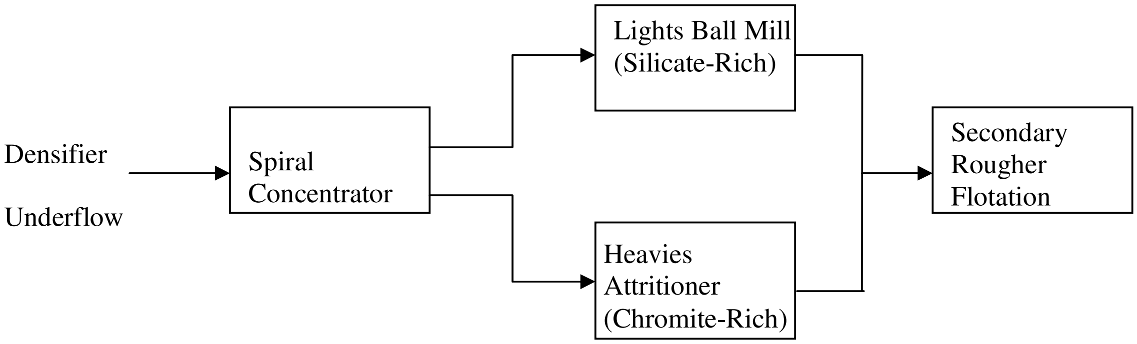

Figure 3.8: UG-2 ore secondary milling circuit configuration with ball milling and attritioning

3.8.Recyle spiral concentrator circuit with single ball mill

An alternative secondary milling circuit design configuration tested the effect of maintaining a single ball mill design in the secondary circuit, but focussed on running the silicate material in a closed circuit ball milling arrangement with a spiral concentrator and additional hydrocylone. The objective of this design was to try and liberate more platinum minerals by targeting a finer grind in the lights material, which contained the majority of the platinum material and it also aimed to reduce the slurry "pooling" effect as mentioned by Hinde and Kalala (2009) and Bryson (2004) by incorporating a recycle stream for the lights material. Figure 3.9 showed the diagram of this alternate milling circuit design. This circuit was investigated by conducting a laboratory locked cycle test.

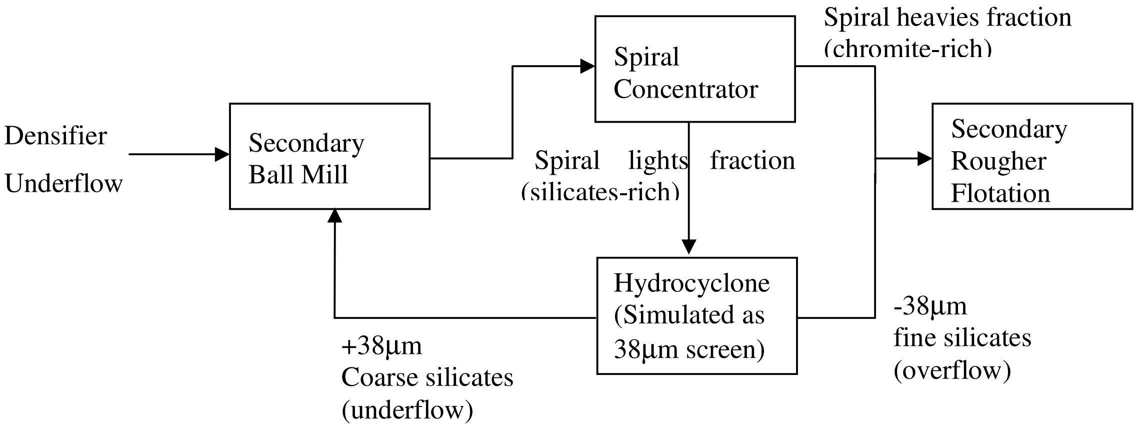

Figure 3.9: Alternative UG-2 ore secondary milling circuit configuration with a closed circuit ball mill with a spiral concentrator

The spiral concentrator material was deslimed with a 38µm screen to simulate a hydrocyclone cut point of 38µm. The +38µm screen underflow silicate material was recycled to the ball mill and the -38µm screen overflow silicates material was combined in mass proportional with the spiral heavies fraction, and floated under standard batch flotation conditions to produce a bulk rougher concentrate. The mass of the -38µm recycle silicates material was recorded on every cycle of the test to determine once steady state was reached. Flotation tests were performed on each step and the samples were submitted to Mintek for 4E and Cr_2O_3 analyses. These results were compared to the spiral circuit configuration with separate ball mills for the silicate and chromite material as it seemed to be a intuitive design choice, as it did not require the additional capital costs of purchasing a new "chromite" ball mill. This allowed the silicates material to be run in a closed circuit configuration so that it can be finely ground, which may liberate more platinum minerals.

3.9. Cleaner spiral tests

Rule and Anyimadu (2007) mentioned that Anglo Platinum have two existing plants (Ivan and Mortimer concentrator) where chromite is rejected with the use of a spiral circuit placed between the primary and secondary circuit of the concentrator. It was therefore decided to investigate the viability of producing a relatively "clean" chromite fraction (i.e. containing low platinum values) which could be discarded prior to secondary ball milling. This would reduce the mass flow to the secondary milling and flotation circuit as the clean chromite may be sent for ferrochrome production and the remaining chromite containing platinum-bearing particles could be treated with the silicates material, or through alternative milling technologies. This circuit shown in Figure 3.10 was investigated in the laboratory on the MG-type cleaner spiral concentrator unit on a low grade 4E sample.

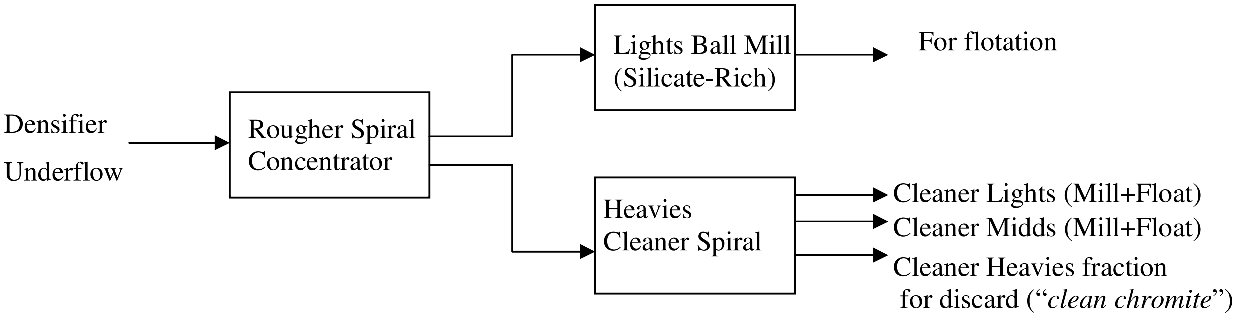

Figure 3.10: Cleaner spiral circuit for chromite rejection

Bulk rougher tests were conducted to produce sufficient mass to run on the cleaner unit. The unit was run at 0.5 litres/s with a 15% solids feed of rougher chromite material. The unit was allowed to reach a steady state and the cutters were positioned such that a "relatively clean" band of chromite could be removed as the tailings product. A lights, middlings and heavies product of the cleaner spiral was collected. These samples were wet screened at 38µm and the +38µm material was dried and screened at 106, 75, 53, 38µm. The -38µm material mass was reconciled with the mass of the wet screened sample.

All sizing samples were submitted for 4E and Cr_2O_3 analyses at Mintek to determine the metal deportment. This would give a good indication whether a reject chromite stream option could be considered in the design of the UG-2 secondary milling circuit.

3.10. Size-by-assay of rougher tailings samples

Flotation tailings samples for the standard process and the best milling configuration were sized at 106, 75, 53, 38, and -38µm. These samples were submitted for 4E and Cr_2O_3 analyses at Mintek, to determine the amount of coarse and fine platinum minerals that were liberated and floated as a result of the new milling circuit design, as compared to current process. This would indicate whether the new process was effective in reducing the amount of fine Cr_2O_3 minerals, which would impact on the Cr_2O_3 entrainment to the flotation concentrate.

3.11. Flotation Rate Tests

Flotation rate tests were conducted to compare the standard process with the new milling circuit design configuration. The standard laboratory rougher flotation method was followed; however, three concentrates were collected for each test, i.e. 0-2min, 2-5min, and 5-10min. These samples together with the tailings were submitted for 4E and Cr_2O_3 analyses to determine whether there was any improvement in the amount of liberated PGMs in the flotation concentrate by adopting the new circuit design option. The water recoveries were also recorded.

3.12. Pilot plant test work on a spiral concentrator circuit

The entire secondary milling circuit design thus far was based on an existing full scale spiral concentrator and therefore a pilot plant campaign was conducted in collaboration with Lonmin, SGS South Africa and Roche EDI Mining on the latest spiral technology available.

Lonmin Platinum supplied a 250kg sample of the densifier underflow (secondary mill feed sample) to SGS for the test work. Downer EDI Mining based in Richards Bay supplied two VHG spiral concentrators for the test work which was conducted by the Minerals Processing division at SGS in Booysens, Johannesburg. The VHG spiral concentrator specifications are shown in Appendix A9.

The spiral concentrator circuit tested at SGS was identical to Figure 3.10 as it was imperative to determine the mass proportions and metal deportment to all streams using current spiral technology designed for high density chromite ores. The SGS circuit was set-up and commissioned once the spiral units arrived from Richards Bay. These circuits were run with slurry to set the cutter positions and thereafter 5kg of each stream were collected. Six samples were collected in total, namely, the rougher spiral feed (densifier underflow), rougher lights, rougher heavies (cleaner spiral feed), cleaner lights, cleaner middlings and cleaner heavies samples. SGS dried and bagged the samples and sent them to the School of Chemical Engineering, UKZN for sizing. The sized samples were then submitted to Mintek for 4E and Cr_2O_3 analyses.

Duplicate flotation tests and size-by-assays of flotation tailings were conducted on these samples to confirm the best laboratory results that were achieved on the existing MG type spiral concentrator. These tests would confirm the viability of the proposed milling circuit design configuration for the secondary UG-2 concentrator.

CHAPTER 4: LABORATORY RESULTS AND DISCUSSION

In this Chapter, the results from the experimental work is discussed and provide the necessary evidence to justify the choice for the secondary milling circuit design for the UG-2 concentrator.

It is appropriate to begin the discussion with the plant audit data as this information provides an understanding of the current process and identifies some process inefficiencies which were addressed during the laboratory test work campaign. All samples were analysed for 4E (Pt, Pd, Rh, Au) and Cr_2O_3 unless otherwise stated, as these are the main minerals of focus; however, the grades (metal concentrations) are proprietary, and therefore the comparisons in results are based on recoveries for the flotation process, and metal distributions for the sizing data.

4.1. Analysis of plant audit samples

4.1.1 Size-by-assay data

The size-by-assay data for the secondary UG-2 concentrator plant audit samples are discussed in this section. This aids in benchmarking the current secondary milling and flotation concentrator circuit. The mass, platinum 4E and Cr_2O_3 distributions for the plant audit samples are shown in Tables 4.1 to 4.3.

Table 4.1: Mass distribution per size fraction for plant audit samples

	Primary Rougher Tails (PRT)	Densifier Underflow	Densifier Overflow	Mill Discharge	Secondary Float Feed (SFF)	Secondary Rougher Tails (SRT)
Screen Size (µm)	Mass Distribution (%)					
+106	34.3	49.4	4.4	11.4	9.6	8.9
-106+75	21.1	29.8	14.9	26.7	16.0	23.8
-75+53	11.1	16.7	11.1	11.5	9.4	11.3
-53+38	3.9	3.0	5.2	4.9	4.1	5.5
-38	29.6	1.1	64.4	45.5	60.9	50.5
Total	100.0	100.0	100.0	100.0	100.0	100.0

Table 4.1 represents the mass distribution of the plant audit samples across the size interval ranging from +106µm to -38µm. The primary milling circuit grind is typically 40% -75µm and this is confirmed by the PRT sample which comprises about 45% material -75µm. A 13% variance was noted between the typical grind and the current plant operation.

The most significant stream is the densifier underflow (mill feed) as this sample was used for all the optimisation tests that follow. Table 4.1 showed that 49.4% of the material occurs in the coarse (+106µm) fraction with a further 29.8% in the -106µm+75µm fraction. This sample had been deslimed with a hydrocyclone (densifier), and therefore contained a very small proportion of fines from the process (1.1%). Since the PRT is the feed stream to the densifier, these profiles indicated that majority of the coarse material (+75µm) reports to the densifier underflow and majority of the fine material (-38µm) from the separator, reports to the densifier overflow stream. The densifier overflow stream does contain a significant proportion of coarse material as noted in findings by Hinde and Kalala (2009), which mentioned the inefficiencies of these units in achieving a good classification. Over 19% of the material in the overflow stream is quite coarse (+ 75µm) and bypasses the secondary grinding circuit. These particles may contain a significant proportion of platinum minerals that require further liberation; however, the magnitude of this short circuiting effect will be quantified by the data in Table 4.2.

Another interesting stream is that of the secondary rougher tails (SRT) as it is the major proportion of the final tailings (discard) from the process. This stream contained close to 33% coarse material (+75µm) and also a large proportion of fines (51% -38µm). This will also be related to Tables 4.2 and 4.3 which will provide details on the platinum mineral and Cr_2O_3 distributions in this circuit.

Table 4.2: 4E distribution per size fraction for plant audit samples

	Primary Rougher Tails (PRT)	Densifier Underflow	Densifier Overflow	Mill Discharge	Secondary Float Feed (SFF)	Secondary Rougher Tails (SRT)
Screen Size (µm)	4E Distribution (%)					
+106	31.6	28.7	4.1	10.0	5.3	12.2
-106+75	16.0	39.4	8.6	17.7	8.8	24.9
-75+53	7.8	23.5	4.9	7.1	5.1	11.6
-53+38	2.8	6.1	1.8	3.4	2.1	5.7
-38	41.8	2.3	80.5	61.7	78.6	45.6
Total	100.0	100.0	100.0	100.0	100.0	100.0

Table 4.2 showed the 4E distribution per size fraction for the plant audit samples. The data for the densifier underflow indicated that majority of the platinum minerals occur in the +53, +75 and +106 µm (91.7% 4E) and would therefore need further grinding to liberate these platinum minerals to achieve additional flotation recoveries. Nineteen percent of the coarse material by mass that had short-circuited to the densifier overflow stream contained about 13% of the PGE. This would have a very small chance of being recovered in the flotation concentrate as these particles would be too coarse even if extra residence time was allocated in the flotation circuit. The most significant 4E distribution from these streams was shown by the SRT, where it was apparent that about 37% of PGEs in that stream occur in the coarse fraction (+75 µm). The

remaining 4E that were not recovered were largely found in the fines (-38 µm). Pease *et al.*, (2006) discussed various factors that result in the loss of fines and these recommendations could be implemented by the concentrator circuit. Since this project was focused more specifically on the liberation of PGMs from coarse particles, optimization tests did not deal with issues affecting fines recovery. A comparison of the densifier underflow (mill feed) to the mill discharge 4E distributions showed that the secondary ball mill is effective at liberating the platinum minerals present in the coarse fractions, as the proportion of 4E in the coarse particles (+75µm) was reduced by 40%. The main concern from this data was; however, that the mill discharge still contained about 28% of +75 µm PGMs which would have a smaller chance of being recovered during flotation. The platinum distributions for this specific concentrator plant did provide evidence to suggest that the secondary milling circuit design does in fact require further optimisation, to improve the liberation of PGMs from the coarse particles. These minerals may be locked in or attached to the coarse particles; however, only a detailed mineralogical analysis can provide insight into these occurrences.

The Cr_2O_3 distributions are shown in Table 4.3. They were similar to the 4E distributions, with the majority of the Cr_2O_3 present in the coarse fractions, and only 1.3% Cr_2O_3 occurring in the densifier underflow -38 µm fraction. After milling, the proportion of Cr_2O_3 in the fines increased to 43% in the mill discharge, and the SFF stream contained 62.3% Cr_2O_3 in the -38 µm fraction after being mixed with the densifier overflow stream. This was quite excessive and was a clear indication that overgrinding of Cr_2O_3 occurred in the secondary ball mill. The SRT stream showed a 51% Cr_2O_3 content, which means that around 11% of fine Cr_2O_3 minerals (-38 µm) were entrained in the flotation concentrate, as the metal distribution in the other fractions seems to proportionally increase to some extent in relation to the decrease of Cr_2O_3 in the ultra-fines fraction.

Table 4.3: Cr_2O_3 distribution per size fraction for plant audit samples

	Primary Rougher Tails (PRT)	Densifier Underflow	Densifier Overflow	Mill Discharge	Secondary Float Feed (SFF)	Secondary Rougher Tails (SRT)
Screen Size (µm)	Cr_2O_3 Distribution (%)					
+106	36.1	48.5	1.0	11.2	9.1	8.7
-106+75	22.9	29.1	6.8	28.5	15.5	23.5
-75+53	11.2	17.6	7.9	12.2	9.1	11.1
-53+38	3.5	3.5	5.2	5.0	4.0	5.4
-38	26.2	1.3	79.0	43.1	62.3	51.2
Total	100.0	100.0	100.0	100.0	100.0	100.0

It was evident from the metal distribution tables of the sizing data that there was no clear size interval at this stage of the process that exclusively contains Cr_2O_3 or PGMs which could be isolated for further liberation or waste discard. The PGMs and Cr_2O_3 present in the ball mill discharge, SFF and in the SRT stream are predominantly concentrated in the ultra-fines (-38 µm) and coarse fractions (+75µm). It can also be noted that a large proportion of particles lost in the SRT stream are due to poor liberation, or overgrinding, which created a substantial amount of slimes which were not effectively recovered in the flotation circuit. The operation of the densifier seemed to be quite inefficient, and therefore considering the fact that UG-2 ore consists of about 60% chromite and 40% silicates, density separation would need to be evaluated to determine whether it could yield an improved separation of Cr_2O_3 and PGM as opposed to the use of conventional hydrocyclones which were modified to run as densifiers.

4.1.2 HLS tests on densifier underflow sample

Heavy liquid separation tests were conducted at Mintek in duplicate on a -150+106µm fraction of the densifier underflow sample. This size range was chosen to determine whether a coarse fraction of the secondary mill feed material contained a density class which was rich in Cr_2O_3 and barren of PGMs. The sizing of the +106µm material was as follows: 35% in -150+106µm fraction, 41% in the -212+150µm fraction and 24% of the mass above 212µm. Table 4.2 revealed that the +106µm contained about 29% of 4E and 48% of the Cr_2O_3 minerals in the densifier underflow size-by-assay distributions. Since HLS laboratory tests are normally done on +1mm feed material, Mintek indicated that this test work could only be done at one density fraction as fine material settling rates were much longer, especially in the higher density ranges (above 3.0-3.6 g/cm^3). These tests were therefore conducted using a density regime of 2.96g/cm^3. The results in Table 4.4 indicated that majority of the PGEs (75% of size fraction) are found in the +2.96SG range with almost all the heavier Cr_2O_3 minerals (99.7%), based on an average of the duplicate tests. This was expected, due to the low density regime selected due to the fineness of the feed material. However, it can also be inferred that as this is a secondary mill feed sample, and most of the platinum minerals above 106µm may be locked in or attached to composite silicate particles, as well as with Cr_2O_3 mineral associations, and thus resulting in most of the 4E reporting to the higher density regime. It was therefore concluded that the HLS results for this coarse fraction of the densifier underflow sample was not sufficient proof at this stage that a gravity circuit would be beneficial for processing the UG-2 material prior to secondary ball milling. Further investigations were required to provide such evidence.

Table 4.4: HLS results on a -150+106μm sample

-150+106μm	Mass (%)	4E Recovery (%)	Cr_2O_3 Recovery (%)
+2.96SG	85.0	75.7	99.7
-2.96SG	15.0	24.3	0.3

4.1.3 Mineralogy of the secondary rougher tails (SRT) sample

A 200g un-sized sample of the SRT from the plant audit was sent to the Process Mineralogy Group at Mintek for a PGM search and a bulk modal analysis. Since this was a low grade sample of below 1g/t 4E, only 61 PGM grains were found and analysed. This may affect the statistical validity of results; however, this data still provides sufficient understanding of where the majority of losses of platinum minerals arise. In order to interpret the mineralogy data it is important to understand how PGM grains occur in plant samples. Table 4.5 defines the common terminologies used when discussing such data and Figure 4.1 illustrates how the PGM grain mode of occurrence can be interpreted when viewing the subsequent mineralogy data.

Table 4.5: Table of mineralogical terms (Mintek)

L	Liberated PGMs
SL	PGMs associated with liberated BMS (Base Metal Sulphides)
AG	PGMs attached to Silicate or Oxide gangue particles
SAG	PGMs associated with BMS attached to Silicate or Oxide gangue particles
SG	PGMs associated with BMS locked in Silicate or Oxide gangue particles
G	PGMs locked within Silicate or Oxide gangue particles

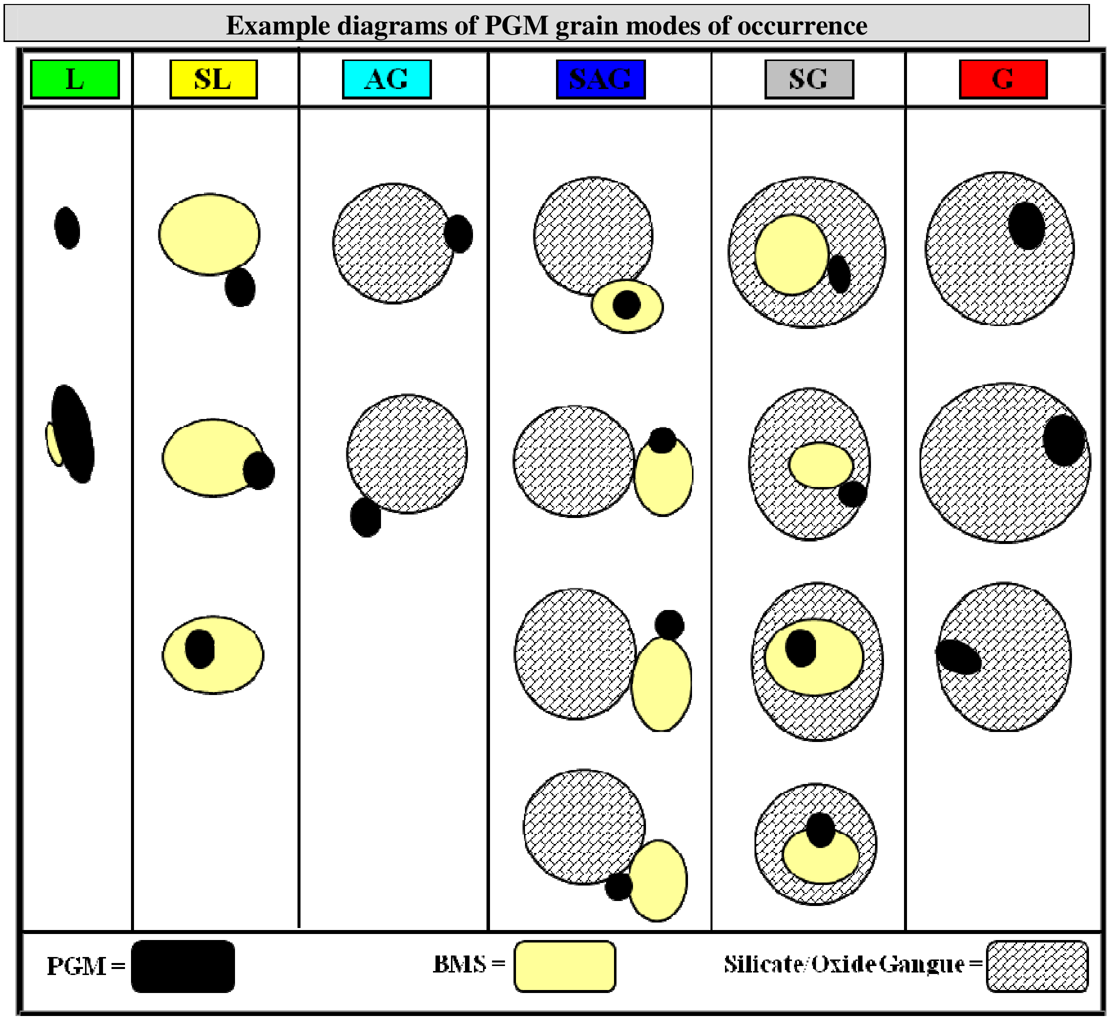

Figure 4.1: Example of PGM grain mode of occurrence (Mintek)

A modal analysis based on the 61 PGM grains revealed that the main minerals in the secondary rougher tailings (SRT) were chromite (65%), pyroxene (16%) and feldspar (14%) by mass, with all other minerals such as pyrite, pentlandite, chalcopyrite and pyrrhotite less than one percent by mass.

The PGM grain mode of occurrence is shown in the Table 4.6 and Figure 4.2 shows the volume percent of PGMs relating to the grain mode of occurrence.

Table 4.6: PGM grain mode of occurrence (Mintek Analysis)

PGM grain mode of occurrence			
	PGM Vol %	**No of PGM grains**	**No of PGM grains smaller than 3 µm**
L	13.0	9	4
SL	18.3	6	2
AG	24.0	15	8
SAG	12.8	11	6
SG	6.6	4	1
G	25.2	16	7
	100.0	**61**	**28**

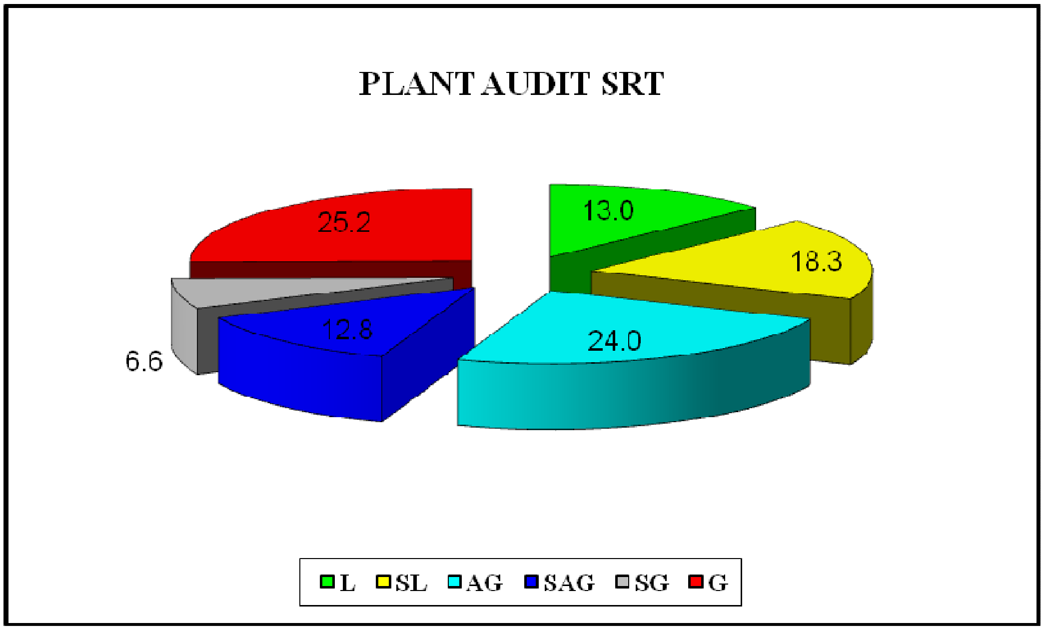

Figure 4.2: MLA analysis of SRT sample (Mintek Analysis)

It can be seen from Table 4.6 that out of the 61 PGM grains that were selected, 15 grains (24%) are attached to silicate or oxide gangue particles, 11grains (12.8%) are associated with base metal sulphides and 16 grains (25.2%) are locked within silicate or oxide gangue minerals. Most notably of the 25.2% locked PGMs, almost half of those PGM grains (7) are less than 3µm which indicates that only a very fine grind will liberate such particles. About 13% of the grains are fully liberated (9 grains) but still lost to the flotation tailings. This is also due to the fact that almost fifty percent of them were also extremely fine (<3µm) and were lost due to sliming effects or poor flotation techniques available to float ultra-fine liberated particles.

The distributions of the different PGM types are shown in Table 4.7.

Table 4.7: Distribution of PGM types in SRT sample (Mintek Analysis)

Distribution of PGM types			
PGM Type	**PGM Vol %**	**No of PGMs**	**Average ECD (µm)**
PtFe	27.3	12	4.0
PtS	21.2	17	3.0
PtRhCuS	9.5	4	3.9
PtPdS	9.2	9	2.7
Ru(Os,Ir)S	6.7	4	3.4
PtPdBiTe	5.9	1	6.7
PdS	3.4	4	2.5
PdPb	3.1	2	3.4
PtPdSb	3.1	1	4.8
PtAs	2.8	2	3.2
PtAsS	2.6	2	3.2
PtPbCuNiS	2.4	1	4.3
PdBiTe	2.2	1	4.1
AuAg	0.6	1	2.1
	100.0	**61**	

This data confirmed the results of the modal analysis which indicated that pyroxene (PtS) and feldspar (PtFe) were two of the main minerals found in the sample. These two minerals accounted for the major PGM losses as there were 17 grains of PtS, 9 grains of PtPdS and 12 grains of PtFe found in the SRT. Other PGM types are in small proportions and it may not be statistically relevant to consider them as there are less than 5 grains of these PGM types.

From the mineralogy of the SRT plant audit sample, it is evident that a large proportion of PGMs that were lost to the final flotation tailings after secondary grinding and rougher flotation are due to poor liberation, as most PGM grains were either attached or locked with silicate or oxide gangue (silicate or chromite), or with base metal sulphides. This is a definite indication that the PGM circuit does require an improvement to the secondary ball milling process. A large proportion of Cr_2O_3 minerals were noted in this sample (65% by mass), therefore gravity separation may show potential to achieve a good separation of the silicates from the chromite. It may have been beneficial to compare the mineralogical effect of the optimized milling circuit design with this plant audit, or, the current process based on laboratory tests; however, these analyses are normally quite expensive, and were therefore not considered further. In addition their statistical relevance is normally debated by many specialists especially for low grade samples where a large number of PGM grains (>100) are required to ensure the reliability of conclusions.

The current process as benchmarked by the plant audit data has provided a platform for establishing the laboratory test work plan, as the main concerns of the plant were identified in this section and therefore the following sections focus on the results of the laboratory test work campaign.

4.2. Varying proportions of dense minerals in a batch laboratory ball mill[3]

The initial laboratory milling tests were conducted using a synthetic mixture of ilmenite and limestone as discussed in Section 3.3. Table 4.8 shows the mill operating conditions for the initial tests.

Table 4.8: Mill Base Case Specifications

Mill Diameter	0.30m
Mill Length	0.30m
L/D Ratio	1
Volumetric Loading of Total Charge	40%
Balls to Slurry Ratio	50:50
Ball Size	27mm

Table 4.9 shows the grinding data for pure limestone and ilmenite that were initially sized and milled for 20 minutes in the ball mill as per the base case mill specifications shown above. The work indices of chromite, ilmenite and limestone are similar (12 to 13 kWh/t, (Wills, 2007)). Hence, artificial mixtures of limestone and ilmenite were used to determine how the presence of a dense mineral affected the grinding of a lighter mineral. Both samples were screened at 425μm to remove any large particles that may affect the overall results. The same volume of ore was fed to the mill for each of the proportions and pure samples milled.

Table 4.9: Data on Pure Ore Feed Samples

Limestone (Pure)	Ilmenite (Pure)
$W_i = 12.8$ kWh/t	$W_i = 12.3$ kWh/t
$\rho = 2600 kg/m^3$	$\rho = 4800 kg/m^3$
$F_{80} = 350$ μm	$F_{80} = 300$μm
$P_{80} = 155$ μm	$P_{80} = 210$ μm

Although the limestone was slightly coarser initially (F_{80} =350μm) it had a lower P_{80} after grinding, which shows that this limestone sample has a lower work index than the ilmenite

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[3] Maharaj,L., Pocock, J., Loveday, B.K., Hinde, A., Density segregation and selective grinding of a synthetic UG-2 ore mixture in a batch laboratory ball mill, Proceedings of South African Chemical Engineering Congress, 20-23 Sept. 2009

sample. The measured and theoretical limestone work indices differed as the laboratory sample had an 85% purity based on repeated HCL dissolutions tests (Appendix A5).

The size distributions of the limestone and ilmenite samples are shown in Appendix B6 and the F_{80} values were 350μm and 300μm respectively. Table 4.10 indicates the P_{80} values for the ilmenite and limestone samples, which were milled for 20 minutes under standard milling conditions as defined in Table 4.9. The feed sample ratios in Table 4.10 are represented in terms of volume percentage limestone added to the mill.

Table 4.10: Effect of varying ore ratios on the "as is" feed mixtures

Feed Sample Ratio	Limestone P_{80} (μm)	Ilmenite P_{80} (μm)
0% Limestone	-	210
20% Limestone	141	197
40% Limestone	144	208
50% Limestone	169	169
60% Limestone	177	177
80% Limestone	147	239
100% Limestone	155	-

Table 4.10 showed that there appeared to be some interactions in the middle of the range (50 -60 per cent limestone), where the grinding of the lighter fraction was reduced. Figure 4.3 illustrates a logarithmic plot of the size distribution data. The dashed line at 1.90 represents the point of 80% cumulative mass percent passing for a particular screen size and was used to determine which ratio of limestone to ilmenite resulted in the lowest screen size. The P_{80} for the 20%, 40% and 80% limestone are within a very narrow size range and thus these graphs are within a close proximity to each other.

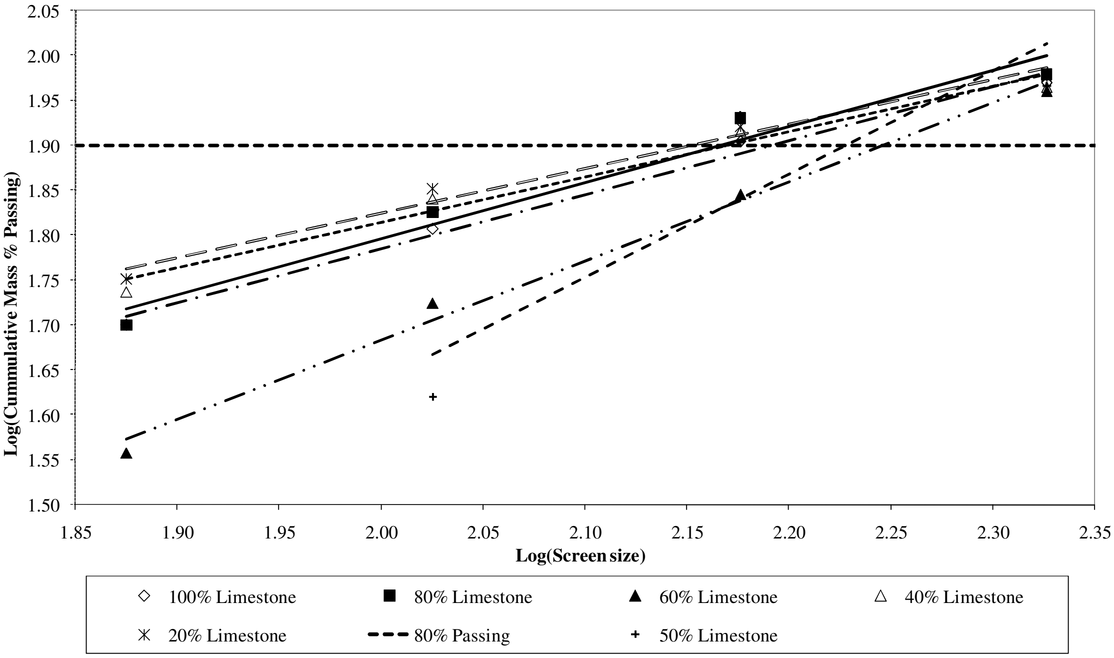

Figure 4.3: Limestone product distribution for "as is" feed mixture ratios

The test with 20% limestone (80% ilmenite) appeared to give the finest size distribution of limestone. The limestone and ilmenite P_{80}'s for this test was 141μm and 197μm respectively. However, it can be concluded that besides the 50 and 60% ratios, all other proportions produced similar results, and therefore several repeats would be required to the confirm that the 20% ratio of limestone was optimal.

Based on the understanding of the cascading process that occurs in a mill (Section 2.4) it could be inferred that some segregation occurred in the toe of the mill, exposing the lighter mineral to impacts from the cascading balls. Since the results were counter intuitive to what would be expected from basic understanding of density effects, selected tests were replicated. These replication tests were conducted using a -300μm ilmenite and limestone sample as opposed to un-sized limestone material in the first batch of tests. The aim of screening the limestone was to reduce the size range of the sample so that it would be in a similar range of a typical UG-2 ore secondary milling feed sample. Replication tests were performed on the 20% and 50% limestone mixtures to confirm the resulting trend. The comparison graphs are shown in Figure 4.4. These graphs were not included on the previous plot since they incorporated -300μm limestone.

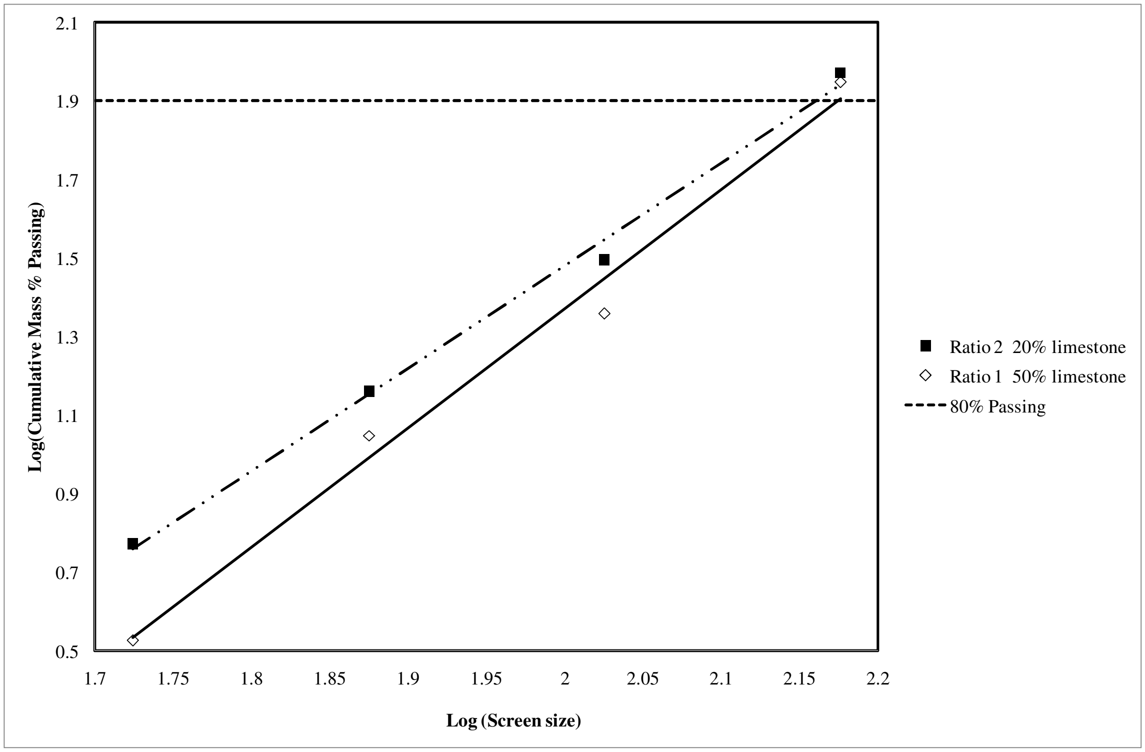

Figure 4.4: Limestone product distribution for -300μm feed mixture ratios

This curve was in agreement with the previous findings and although the change in P_{80} was marginal, there was a significant increase in the proportion of finer material. The close P_{80} values may have been due to the screening of the feed material which reduced the coarse limestone particles in the feed. Table 4.11 below shows the actual P_{80} values for these -300μm tests.

Table 4.11: Effect of varying ore ratios on the -300μm feed mixtures

Feed sample ratio	Limestone (P_{80})	Ilmenite (P_{80})
20% limestone	138	140
50% limestone	141	144

These investigations indicated that selective grinding of the less dense particles was occurring to some extent as the ratio of the low density material is reduced i.e. at a 20% limestone and 80% ilmenite feed ratio. Since the difference was quite small the effect of implementing this feed ratio would need to be considered on a UG-2 platinum sample to determine its ability to substantially improve the platinum recovery, or to reduce chromite entrainment in the flotation concentrate. The subsequent test work focussed on determining a method to quantify the short-circuiting effect on a batch scale.

4.3. Slurry shortcircuiting (“pooling") and density segregation tests

Hinde and Kalala (2009) and Bryson (2004) mentioned that short-circuiting effects (pooling) of the platinum-rich silicates occurred in the conventional secondary open-circuit ball milling circuits with overflow discharge arrangement, due to a large density difference between the silicates and chromite particles, which results in the inefficient operation of the hydrocyclone (densifier) and ball mill. This section focussed on laboratory "crash stop" tests on a batch ball mill to determine if these effects could be quantified on a laboratory scale.

Two methods were used to test for short-circuiting and density segregation of silicates as described in Chapter 3.4, with method one using the UG-2 densifier underflow sample and method two using a synthetic mixture of limestone and ilmenite.

The first method involved taking several milled samples on the surface of the pulp immediately following the "crash stop". This was done after allowing the mill to run for ten minutes to ensure sufficient time for any possible density segregation effects to occur. The pulp samples were then subjected to "sink-float" analysis with TBE to determine the percentage of material in the floats (low density < 2.96SG) and in the sinks (high density > 2.96SG). Three repeat samples were taken from each of the two mills that were investigated; however, in both cases the proportion of material in the floats (< 2.96SG) was negligible (<1%). It was therefore concluded that short circuiting or "pooling" could not be adequately quantified using a laboratory "crash stop" test technique and that a continuous laboratory setup was required to provide conclusive evidence of pooling. A continuous arrangement would have been favoured; however, the time required in achieving the steady state conditions may have posed difficulties in handling and storage of the large volumes of the mill discharge stream.

It was therefore decided to use a second test method to investigate whether the laboratory ball mill was influenced by density segregation effects during its operation. The mill was run and sampled as mentioned in Section 3.4. The drain water was noted to contain a negligible amount of solids. Table 4.12 shows the results achieved from these tests.

Table 4.12: Density segregation results

Sections	Limestone (% by mass)	Ilmenite (% by mass)
Top layer	60	40
Middle layer	56	44
Bottom layer	43	57
Left sidewall	44	56
Right sidewall	46	54

Each layer was sampled in duplicate and an average variance in the limestone and ilmenite mass percentage was 5.6%. The right sidewall had a variance of 26% and this significantly contributed to the high average variance, as the other four layers had variances of below 2%. The results indicated that a slightly higher mass proportion of limestone (lighter material) occurred in the top and middle layers with the bottom layer containing marginally more ilmenite. The sidewall distributions were quite similar to each other. From these overall results it can be concluded that under batch conditions there is evidence of density segregation as the more dense material has a higher percentage in the bottom layer, but not significant enough to determine whether slurry pooling will occur industrially since segregation. The current test method was also limited since the segregation could be due to sedimentation after crash stopping the mill.

Since density segregation and pooling were difficult to quantify under laboratory conditions, it was decided that further investigations should be focussed on separating the minerals efficiently prior to milling. It was decided to investigate the potential for spiral concentrators to provide this improved density separation and these findings are discussed further in the subsequent sections.

4.4. Commissioning runs on the spiral concentrator with a synthetic UG-2 ore mixture

The initial phase of test work involved water commissioning of the full scale MG-2 type rougher spiral concentrator. This involved determining a start-up and shut down procedure. The feed flow to the spiral concentrator was manually controlled with the use of the tank recycle (bypass) valve. The recycle valve position was maintained at a constant setting for the duration of the runs and the main feed valve to the unit was adjusted until the desired flow was achieved.

The commissioning runs on the spiral used a synthetic ore mixture of 70% ilmenite to limestone mixture by mass, as the chromite content in UG-2 ore can vary between 60 to 90% for various deposits. The spiral concentrator reached steady state once a constant feed flow and slurry density were achieved. This was typically within 20 minutes of start-up.

Samples of lights, middlings and tailings were collected for each test. The effect of feed flow-rate, percent solids, splitter opening and particle size on the fractional recovery of limestone to the lights fraction were investigated. Duplicate samples were taken for each test and the results are based on the average recoveries for each run. The effect of flowrate was based on a fixed percent solids of 15% and a 1/3 splitter setting (6cm). Figure 4.5 shows the results of the effect of flowrate on the fractional recovery of limestone in lights, middlings and heavies fractions for synthetic feed, unscreened and screened at 300µm.

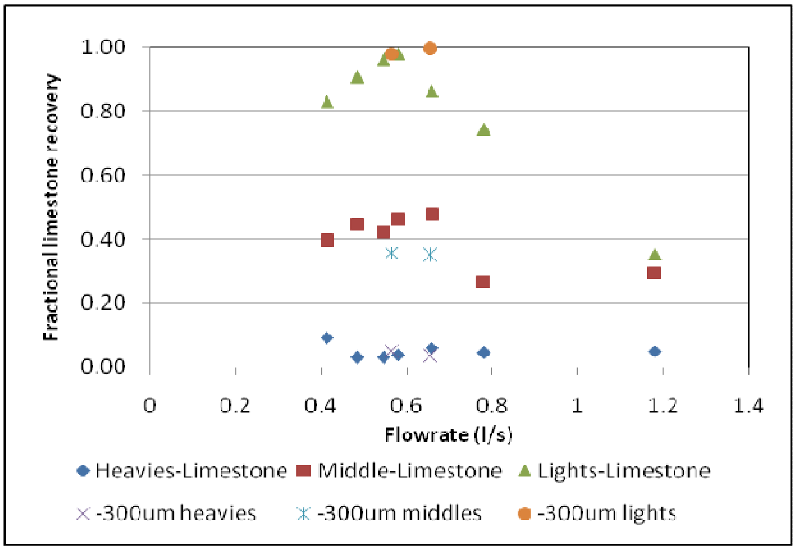

Figure 4.5: Fractional limestone recovery vs flowrate for unsized and -300µm synthetic feed mixture

Figure 4.5 showed that at a flowrate of 0.66 litres/s, the fractional recovery of limestone in the lights (purity) was 86% for the unscreened sample, with an overall recovery of 79.2% limestone as shown in Table 4.13. At higher flowrates, i.e. 0.78 and 1.18 litres/s higher overall limestone recoveries were achieved in the lights fraction; however the purity of limestone decreased quite significantly to 35% in the lights fraction. The 0.66 litres/s flowrate was therefore selected as an optimum flowrate based on a trade-off between limestone recovery and purity. After determining this optimum feed flow for the unscreened synthetic mixture, screening at 300µm was used to remove fine particles from the feed.

Tests were conducted at 0.58 and 0.7l/s to determine how the spiral concentrator would operate with a narrow size range of particles. The purity of limestone in the lights increased to 99% with a 3 % drop in the overall recovery of limestone to 76.5%. Pre-screening was beneficial in improving the overall purity of limestone in the lights; however, since the UG-2 ore sample was already processed in a hydrocyclone (densifier) primarily for fines removal further classification may not be required at this stage of the process design.

Table 4.13: Effect of feed flow on limestone and ilmenite recovery

Flow (l/s)	0.41	0.48	0.55	0.58	0.66*	0.78	1.18
Tailings Mass Fraction (High density fraction)							
Limestone	0.09	0.03	0.03	0.04	0.06	0.05	0.05
Ilmenite	0.91	0.97	0.97	0.97	0.94	0.95	0.95
Middlings Mass Fraction (Mixed density fraction)							
Limestone	0.39	0.45	0.42	0.46	0.48	0.27	0.30
Ilmenite	0.61	0.55	0.58	0.54	0.52	0.73	0.70
Light Density Mass Fraction (Low density fraction)							
Limestone	0.83	0.91	0.96	0.98	0.86	0.74	0.35
Ilmenite	0.17	0.09	0.03	0.02	0.14	0.26	0.65
Recovery of limestone in lights (%)	77.0	70.7	66.1	65.9	79.2	88.9	83.5

It was therefore concluded that the 0.66l/s feed flow was found to be most suitable for this synthetic ore and this flow rate was fixed when varying the other parameters.

The effect of percent solids on the efficiency of separation on the spiral concentrator is shown in Figure 4.6 and Table 4.14. A base case of 15% solids was used from literature for the initial tests; however, the unit was run at 10, 20, and 30% solids to determine an optimum operating point.

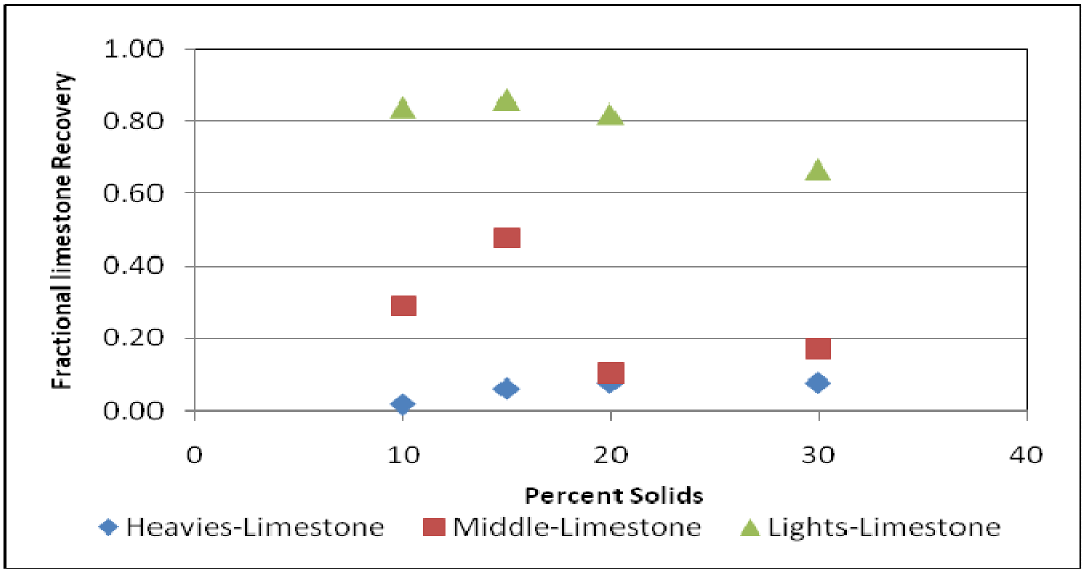

Figure 4.6: Fractional limestone recovery vs percent solids for unsized synthetic feed mixture

Table 4.14: The effect of percent solids on the recovery of limestone

% solids	10	15	20	30
Heavy Density Mass Fraction				
Limestone	0.02	0.06	0.08	0.08
Ilmenite	0.98	0.94	0.92	0.92
Middle Density Mass Fraction				
Limestone	0.29	0.48	0.11	0.17
Ilmenite	0.71	0.52	0.89	0.83
Light Density Mass Fraction				
Limestone	0.84	0.86	0.82	0.67
Ilmenite	0.16	0.14	0.18	0.33
Recovery (%)	91.0	79.2	82.4	74.2

The results indicated that a 15% solids loading by mass in the spiral concentrator resulted in the highest purity of limestone in the lights fraction (86%). Although the 10 % solids does result in the highest recovery (91%) of limestone in the lights fraction, it had a slightly lower purity in that fraction (84%). The middlings fractions had a large proportion of ilmenite (71%) which may not be suitable for the UG-2 ore if the middlings fraction required further milling, as this would result in further chromite liberation, which could result in the process being unfeasible. The run at 30% solids resulted in the lowest limestone recovery (74.2%) with high levels of ilmenite in the lights fraction (33%). Based on these results it was decided to continue operating the spiral concentrator at 15% solids as purity levels in all three fractions, i.e. lights middlings and heavies were more realistic.

The next parameter that was investigated was the splitter opening which was set at one-third open (6cm) for the base case tests. Two additional tests were conducted at a splitter opening of 3cm and 9cm respectively and the results are shown in Figure 4.7 and Table 4.15.

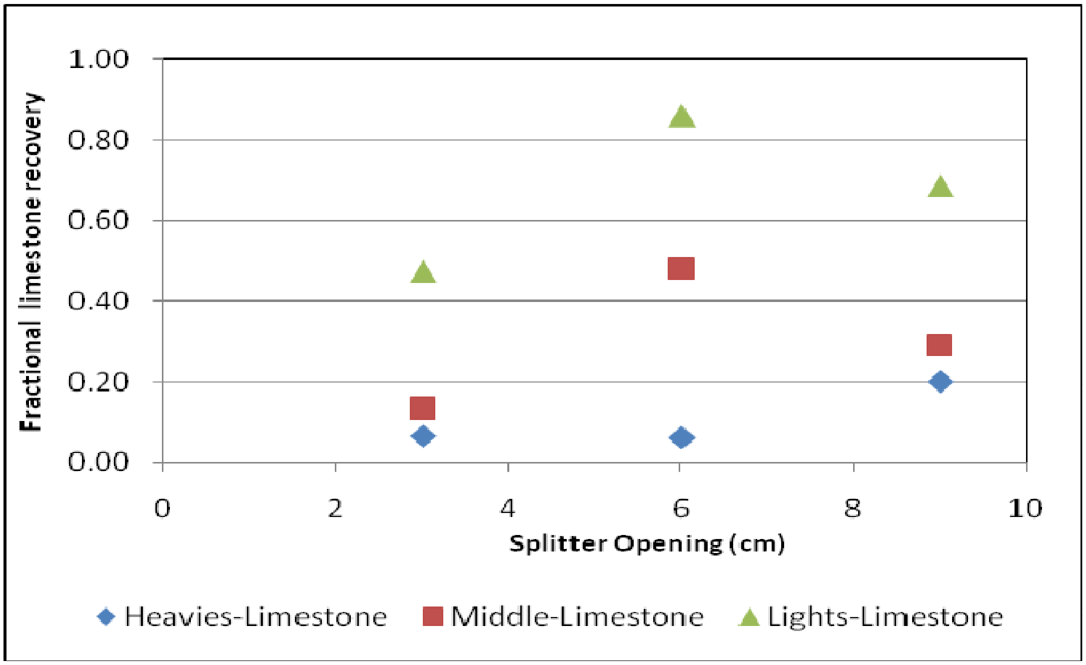

Figure 4.7: Fractional limestone recovery vs splitter opening for the unsized synthetic feed mixture

Table 4.15: The effect of splitter opening on the recovery of limestone

Opened (cm)	3	6	9
Heavy Density Mass Fraction			
Limestone	0.07	0.06	0.20
Ilmenite	0.93	0.94	0.80
Middle Density Mass Fraction			
Limestone	0.13	0.48	0.29
Ilmenite	0.87	0.52	0.71
Light Density Mass Fraction			
Limestone	0.48	0.86	0.69
Ilmenite	0.52	0.14	0.31
Recovery (%)	96.3	79.2	42.6

The base case splitter opening was also found to be an optimum value as both the 3cm and 9cm settings resulted in some inefficiency. The 3cm setting had an extremely high overall limestone recovery (96.3%); however, the limestone purity was very low (48%). The 9cm splitter opening setting resulted in a 42.6% overall limestone recovery being achieved in the lights, with a 69% purity. High levels of purity were desired in the lights fractions and therefore it was evident that the 6cm setting would be ideal.

Repeat tests were not conducted for the commissioning runs as duplicate samples were collected for every stream sampled, with the average data plotted in the above figures (4.5, 4.6, and 4.7); however, further validation of individual tests was proposed for future work. Once the optimised parameters were determined for the MG- type spiral concentrator, it was now important to run the unit with the UG-2 densifier underflow sample to determine whether these optimised parameters were suited for the separation of the platinum-rich silicate and chromite particles present in the platinum ore.

These optimised conditions were tested on the platinum sample. Two additional flowrates were chosen to determine whether the unit was in fact running at optimum conditions. A single stage (rougher) spiral run was conducted and duplicate samples were collected once the unit reached steady state. Samples were analysed for copper using an ICP method as shown in Section 3.6. Platinum minerals in UG-2 ore have been shown to occur with base metal sulphides (copper and nickel) and therefore the copper and chromium recoveries were monitored to determine whether the platinum mineral could be concentrated in the lights fraction. The acid digestion technique was not suitable for dissolving all the chromium (Cr) minerals; however, the relative difference in each of the tests could be used to interpret the effect of changing the flowrate. The results are shown in Figure 4.8. The chromium recoveries were not plotted as a result of poor dissolution; however, the recoveries are discussed.

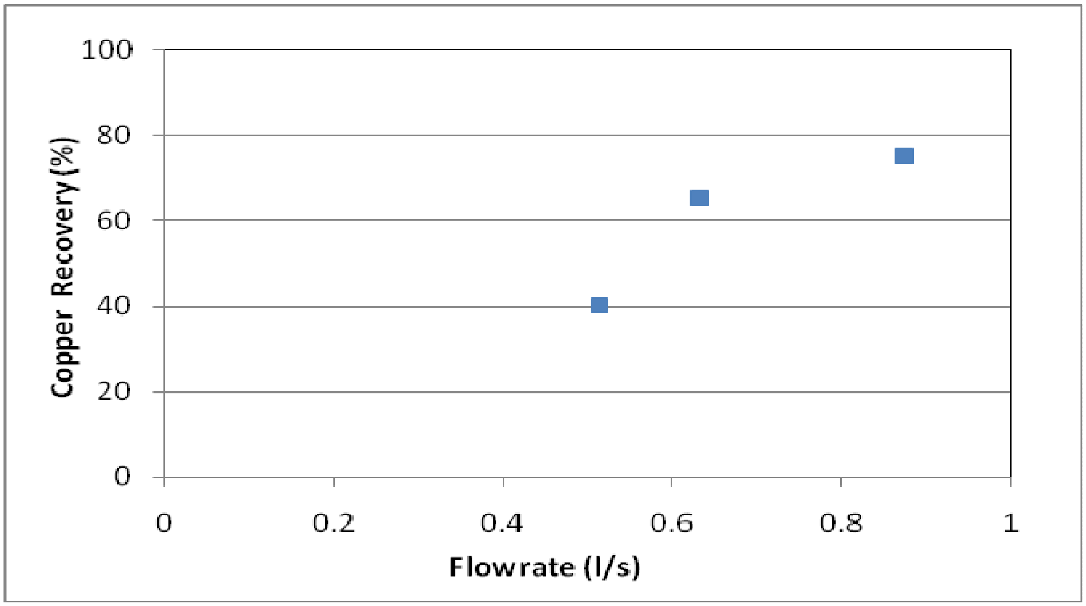

Figure 4.8: Overall copper recovery to the lights fraction vs flowrate for the UG-2 densifier underflow sample

The spiral concentrator was run at three different flowrates, i.e. 0.51, 0.63, and 0.87l/s. The results indicated that the highest copper (Cu) recovery of 75.1% was achieved at 0.87l/s; however, the Cr recovery to the lights was also quite high at 67%. The results at lower flows were as follows: 0.63l/s produced a 65% Cu recovery to the lights with a 27.7% Cr recovery, and the 0.51l/s run resulted in a 40% Cu recovery and a 41.7% Cr recovery. The raw data is available in Appendix B7. Duplicate samples of the 0.63l/s run were sent to Mintek for analysis for Total PGM+Au to determine whether there was scope for adopting this process route.

The results indicated that a 65% Total PGM +Au recovery was achieved in the lights fraction. This was very encouraging and showed that the spiral concentrator may have potential to provide a good separation of platinum-bearing silicate particles from chromite particles.

It was therefore decided to investigate the effects of incorporating a spiral concentrator, with separate milling of the heavy and light fractions. These results are discussed in the subsequent section.

4.5. The effect of a separate ball milling circuit design for UG-2 ores incorporating spiral concentrators

4.5.1 Initial testwork (High 4E grade in feed sample)[4]

The conventional process route of open circuit ball milling of the secondary densifier underflow stream (feed to secondary mill) followed by rougher flotation resulted in a 70% recovery of 4E platinum minerals and a 2.3% Cr_2O_3 recovery under batch conditions. These base case recovery data must be considered when evaluating the new circuit incorporating a spiral separator to produce a lights fraction (silicate-rich) and a heavies fraction (chromite-rich) followed by separate milling of the spiral lights and heavies using two ball mills at varied milling times. The total energy input to the milling circuit was fixed (16 minutes) as per the conventional process route. It should be noted that the two fractions (lights and heavies) were re-combined after separate milling, and the mixture was floated in a batch test under standard conditions. The 80/20 L and 20/80 H combinations were used to simulate a simple but relatively inefficient gravity separator, for example a flat-bottomed cyclone. The preliminary grinding work also showed interesting effects with this combination.

The flotation results for the effect of separate milling of the spiral product mixtures and for the 100% spiral lights and heavies products are shown in Table 16 and 17 respectively. The percentage mass that was recovered to the rougher flotation concentrate was expressed as masspull in all subsequent data tables. A sample of the data was repeated to test the reproducibility of results and the 4E recoveries were found to be within a 4% variance, with the Cr_2O_3 recoveries having a variance of between 5 to 7% (Refer to Appendix B8 for the variances and raw data).

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[4] Maharaj, L., et al. The effect of the design of a secondary grinding circuit on platinum flotation from a UG-2 ore. Miner. Eng. (2010), doi:10.1016/j.mineng.2010.08.007

Table 4.16: Flotation performance on the separate milling of the spiral product mixtures

Energy Distribution during milling (%)		Masspull (%)	Recovery (%)	
80% Lights, 20% Heavies	80% Heavies, 20% Lights	Concentrate	4E	Cr_2O_3
88	12	7.4	69.6	1.9
75	25	8.6	59.2	2.6
63	37	8.1	63.5	2.4
50	50	7.0	64.7	1.8
37	63	7.0	65.6	1.9

Table 4.17: Flotation performance on the separate milling of the 100% spiral lights and heavies products

Energy Distribution during milling (%)		Masspull (%)	Recovery (%)	
100% Lights	100% Heavies	Concentrate	4E	Cr_2O_3
100	0	8.0	67.6	1.5
88	12	7.5	69.0	1.4
75	25	7.7	68.2	1.7
63	37	8.0	68.9	2.1
50	50	8.1	68.6	2.4
37	63	7.7	67.0	2.3

The recoveries of the 4E and Cr_2O_3 have been plotted as a function of the energy input to the lights fractions as shown in Figures 4.9 and 4.10.

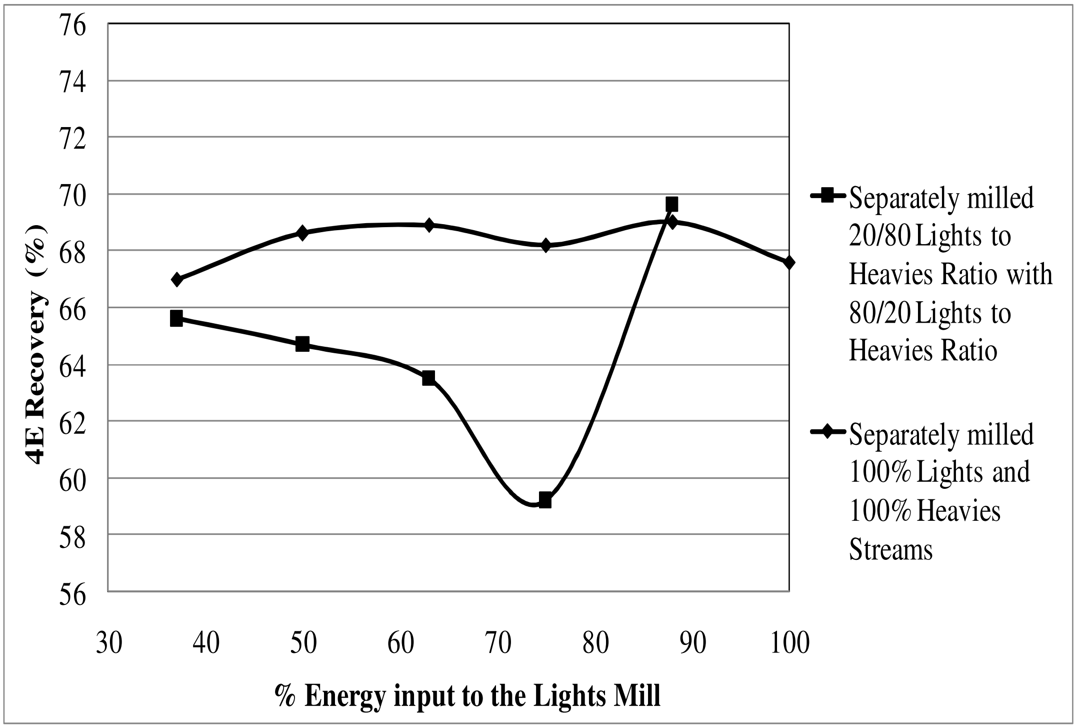


Figure 4.9: 4E recoveries as a function of the energy distribution to the lights mill

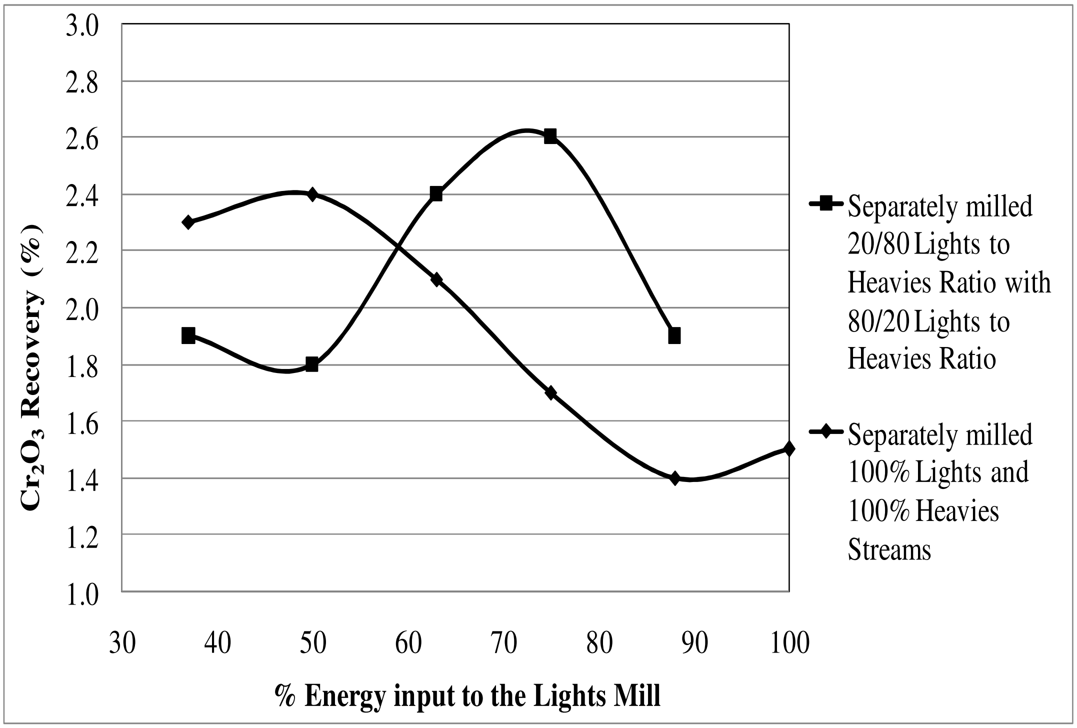


Figure 4.10: Cr_2O_3 recoveries as function of the energy distribution to the lights mill

The 4E platinum mineral recoveries in Figure 4.9 showed no improvement in the base case recovery (about 69% 4E). However, apart from a single point, it was evident that separate milling of the pure lights (100%) and heavies followed by combined flotation is a better option than combining the spiral products into varying feed ratios prior to separate grinding. The 4E recoveries are more than 2 to 3% higher for the majority of the tests with an increase in the lights mill energy input.

The Cr_2O_3 recoveries are compared in Figure 4.10, as a function of the energy input to the lights mill. Separate milling showed real benefit as the Cr_2O_3 recovery was significantly reduced from 2.4% to 1.4% as the lights energy input was increased to 88% of the total energy input. Most significantly, the 4E recoveries were quite constant at approximately 69%; however, a 40% reduction in the Cr_2O_3 entrainment was achieved.

The results have also indicated that the heavies fraction may not need a large proportion of grinding energy as the 4E recovery was 67.6% for the 100% energy input to the lights mill, as compared to 69% for the 88% energy input to the lights mill, i.e. only 12% energy addition to the heavies fraction for a further 1.4% 4E recovery, and thus alternative technologies such as attritioning may be explored in this regard.

The results also confirmed the benefit of the implementation of separating ball milling of the silicates and chromite fractions by Anglo Platinum in their secondary circuit design (Rule and Anyimadu, 2007). The most important advantage of a separate milling operation was that it allowed for the silicates mill to be run in closed circuit to maximize the platinum recovery. It may also reduce the possibility of the hypothesized slurry pooling effect (short-circuiting of silicates) noted for the overflow discharge ball mills.

The significant reduction in the Cr_2O_3 recoveries (~40%) is due to the fact that most of the total milling energy can be re-distributed to the silicates ball mill thus reducing the energy input and over-grinding on the chromite fraction. The lower Cr_2O_3 recoveries may also be attributed to the reduction of any density interactions that may occur between the high-density chromite and low-density silicate particles in the ball mill.

4.5.2 Repeat test work and size-by-assay analyses (Low 4E grade in feed sample)

A series of repeat tests were conducted on the 100% lights and heavies fractions from the rougher spiral concentrator samples as these results were very encouraging and suggested that this milling circuit design could be an improved secondary milling circuit design for UG-2 concentrator. Size-by-assay analyses were also conducted for all energy distributions that were considered in Table 4.17. The 4E and Cr_2O_3 analyses were carried out in duplicate for the sizing data.

The analysis of the feed sample (densifier underflow) used for these repeat tests were found to have a PGE feed grade of about half that of the previous sample in Section 4.5.1. This was possibly due to an instability in the plant operation, when sampling the densifier underflow. However, these 4E feed grades are more in line with the expected secondary circuit 4E feed grades. The Cr_2O_3 grades were quite comparable to each other in both the low and high 4E grade samples, and thus the final Cr_2O_3 recoveries did not show any fluctuations. Therefore, the 4E recovery trends produced from the testwork on the low grade ore can be interpreted in a similar manner to the high grade ore, except that a notable downward shift in the recoveries will be noted due to the lower PGM content in the feed.

The flotation results for the 100% lights and 100% heavies fractions that were milled separately at different energy distributions are shown in Table 4.18.

Table 4.18: Repeat flotation results on the separate milling of the 100% spiral lights and heavies products (low grade feed sample)

Energy distribution to the lights and heavies mill (%)		Masspull (%)	Recovery (%)	
Lights	Heavies	Concentrate	4E	Cr_2O_3
88	12	6.7	54.9	1.5
75	25	6.2	51.9	1.8
63	37	6.2	51.2	2.1
50	50	7.3	51.1	2.5
37	63	6.2	48.0	2.2

The repeat results confirmed that the 88% energy input to the lights ball mill and 12% energy input to the heavies ball mill resulted in an average 4E recovery of 54.9% and a 1.5% Cr_2O_3 entrainment to the secondary rougher concentrate. A standard test repeated on this low grade densifier underflow sample revealed that the average recovery for the conventional single milling process was 51.3 % 4E and 2.3% Cr_2O_3. This indicated that the separate milling circuit resulted in a 3.7% higher PGE recovery and the Cr_2O_3 entrainment was reduced from 2.2% to

1.5%, which was a 32% Cr_2O_3 reduction overall. The variances in the masspull, 4E, Cr_2O_3, and water recoveries for the repeated standard flotation tests on the low grade ore were 0.1%, 4.5%, 0.1% and 0.2% respectively. The variances in the masspull, and 4E recoveries for the repeated flotation tests with an 88% energy re-distribution to the lights mill were 0.2% and 5.5%, with a 0% for both the Cr_2O_3, and water recoveries.

These results compare quite well with the findings with the high grade platinum sample and suggest that this process route has significant benefit in reducing Cr_2O_3 entrainment whilst providing a marginal increase in platinum recovery as noted from these results. These results have been plotted to compare the trend with the initial findings of the high grade sample. Figures 4.11 and 4.12 illustrate the recovery profiles for the repeat runs on the low grade 4E sample with the initial runs on the high grade 4E samples.

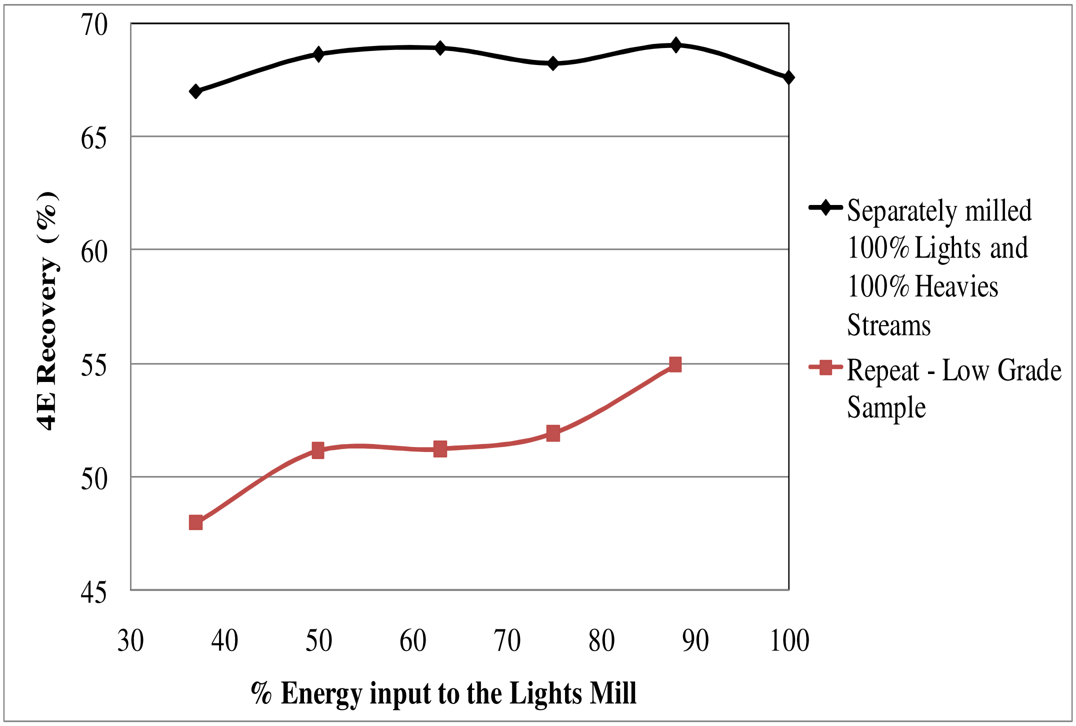


Figure 4.11: 4E recoveries as function of the energy distribution to the lights mill for the initial and repeat runs

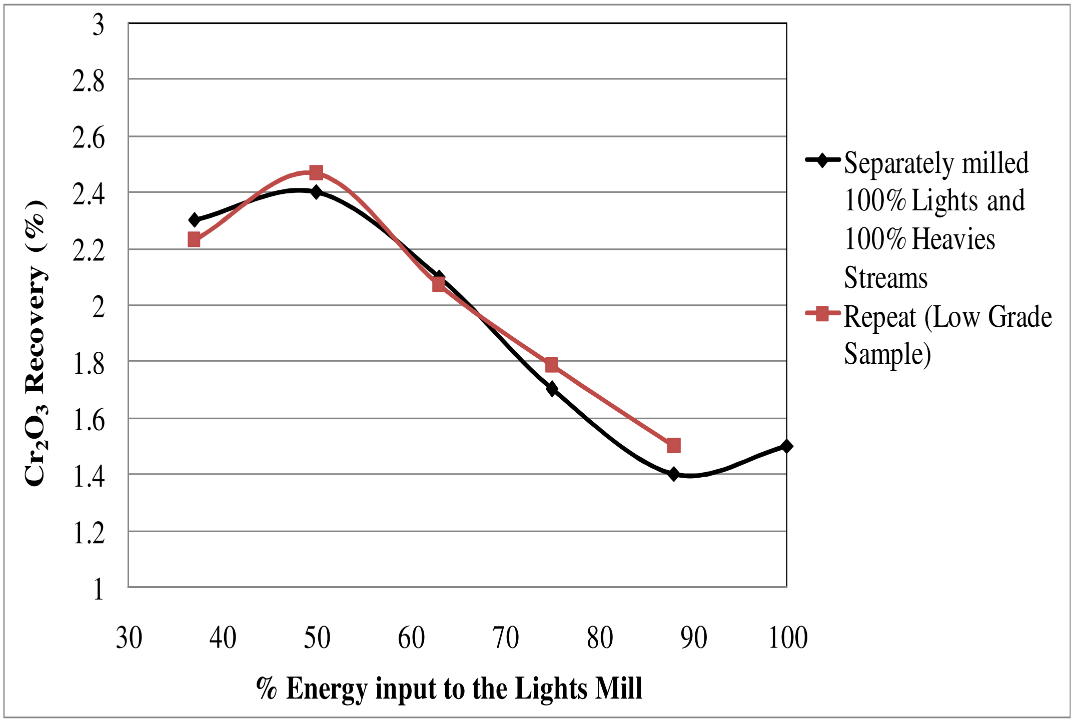


Figure 4.12: Cr_2O_3 recoveries as function of the energy distribution to the lights mill for the initial and repeat runs

The graphs in Figure 4.11 show that the 4E recoveries are of the order of 20% lower in platinum mineral content for the low grade repeat sample. These results are encouraging as they result in a fairly similar PGE recovery profile as in the case of the high grade material. However, the low grade curve shows an increasing 4E recovery trend with an increasing energy to the lights mill. This is not similar for the high grade ore and suggests that the PGMs present in the higher grade ore may have a different mode of occurrence. Figure 4.17 shows the closeness of the Cr_2O_3 graphs for the Cr_2O_3 entrainment curves for the initial and repeat results. The curves in Figure 4.12 emphasise the importance of energy re-distribution in the secondary ball milling circuit for UG-2 ores and provides evidence that energy re-distribution through separate milling of the silicates and chromite particles is a vital method of reducing Cr_2O_3 recoveries to the rougher concentrate. The percentage water recovered in the flotation froth with the solids was measured for each test to investigate its effect on the chromite entrainment to the rougher concentrate. Figure 4.13 shows the resulting relationship.

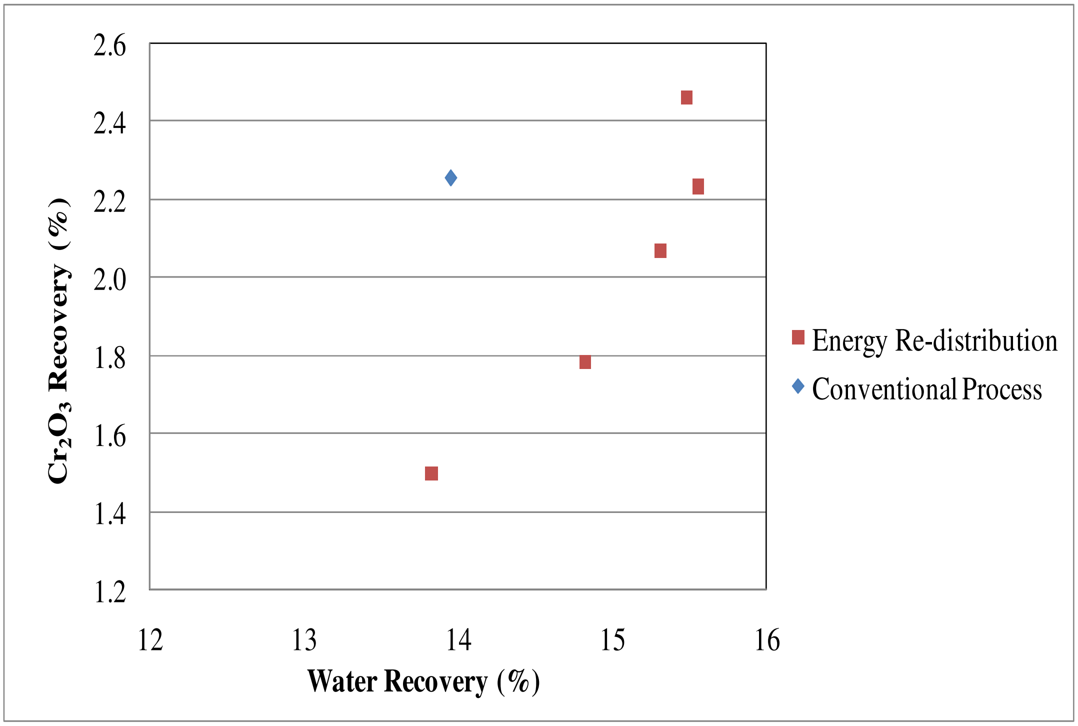


Figure 4.13: Cr_2O_3 recovery versus water recovery for the low grade repeat tests

The graph showed that the data would fit a linear relationship for four out of the five measured data points for the energy re-distribution tests. The lowest water recovery of about 14% was noted for the 88% energy distribution to the lights mills. This data was in agreement with entrainment theory (Section 2.3) which indicated that an increase in water recovery to the froth corresponds to an increase in chromite entrainment. A comparison of the water recovery of the conventional process route shows that the Cr_2O_3 entrainment is much higher for a similar water recovery for the conventional process (2.3% Cr_2O_3) as compared to the 88% energy distribution run (1.5% Cr_2O_3). This indicated that the recovery of the higher amount of Cr_2O_3 in the conventional process may be due to the secondary flotation feed containing a higher proportion of liberated Cr_2O_3 as compared to separate milling tests. Since only 12% of the total milling energy was distributed to the chromite-rich ball mill, the reduced Cr_2O_3 recovery may be possibly due to a lower amount of liberated chromite. The standard method may contain more liberated Cr_2O_3 and thus the recovery may be due to pure flotation rather than entrainment. This was considered when comparing the size by assay analyses for the milled products and is discussed below.

Size-by-assay analyses of the milled products were compared for the standard process and for two separate milling options, i.e. a 50% energy distribution (8 minutes milling each) to the lights and heavies mill, and for the 88% energy distribution to the lights mill (14 minutes milling) and 12% energy distribution to heavies mill (4 minutes milling) to determine the cumulative product size distributions and the 4E and Cr_2O_3 mineral deportment per size fraction. This information may provide more scientific evidence for the choice of the secondary milling circuit design as it may link the mineral liberation to the flotation results for the platinum and chromite minerals. Table 4.19 shows the cumulative product size distributions for the milling products with the associated flotation recoveries.

Table 4.19: Cumulative % passing each screen size for assayed laboratory milled products

4E Recovery (%)	51.3	53.6		51.1	
Cr_2O_3 Recovery (%)	2.3	1.5		2.5	
Screen Size (µm)	Standard (16 min)	Lights (14 min)	Heavies (2 min)	Lights (8 min)	Heavies (8 min)
+106	95.9	98.6	83.6	96.0	98.6
-106+75	80.1	91.3	46.4	77.0	83.3
-75+53	55.9	71.6	26.5	58.7	58.8
-53+38	43.8	62.2	22.7	48.8	50.9

Table 4.19 indicates that the standard test results in 80% -75µm after 16 minutes of batch laboratory ball milling. This milled sample resulted in a 51.3% 4E recovery with a 2.3% Cr_2O_3 entrainment to the rougher concentrate for the low grade runs. When implementing the new circuit design which comprised separation with a spiral concentrator followed by separate ball milling of the lights and heavies, it can be seen that an equal energy distribution to both streams results in 77% -75µm for the lights (silicates) material and 83.3% -75µm for the heavies (chromite) material. This indicates that for an equal mass loading of 250g to the mill and with an equal energy input to each mill, the chromite particles were found to have a marginally lower work index than the silicates. This is possibly due to the composite nature of the silicate particles that may result in their being slightly harder. The flotation results for the 50:50 energy distributions showed a 51.1% flotation recovery with a 2.5% chromite recovery. The sizing data above indicates that the high Cr_2O_3 content is due to over-grinding of the chromite, as 83.3% of the material was found -75µm, with more than 50% of this material -38µm. This possibly liberated more chromite minerals as the 12% energy distribution showed a sizing of only 46.4% of the material +75µm with only 22.7% of the material -38µm. The resulting flotation recovery was 53.6% 4E and most significantly a 35% drop in the Cr_2O_3 content to 1.5% was noted in the rougher concentrate.

This reduction in Cr_2O_3 is strong evidence that supports the implementation of this design in the secondary milling circuit as it clearly shows the effect on chromite liberation whilst also maintaining similar platinum mineral recoveries. The 88% energy distribution to the lights mill resulted in 91.3% of the silicates being -75μm as compared with 80% for the standard case, and 77% for the equal energy distribution with 62.2% of the material also -38μm. A fine grind in excess of 90% -75μm is typically achieved with alternative milling technologies such as SMD's and IsaMills, which are added as tertiary milling devices. These units may also contribute to higher energy costs whereas the implementation of a fairly low cost gravity separation device, such as the spiral concentrator, a good split in silicates and chromite can be achieved. This can be incorporated with two separate ball mills to achieve much finer grinds on silicates and coarser grinds on the chromitite-rich fraction. Tables 4.20 and 4.21 show the 4E and Cr_2O_3 distributions in each size fraction.

Table 4.20: 4E distribution for size-by-assay results of laboratory mill products

	Standard Product (16min)	Rougher Lights Product (8min)	Rougher Heavies Product (8min)	Rougher Lights Product (14min)	Rougher Heavies Product (2min)
Screen Size (μm)	4E Distribution (%)				
+106	2.5	1.1	11.7	3.5	6.0
-106+75	15.5	16.9			12.0
-75+53	25.7	11.0	22.0	10.3	31.5
-53+38	13.9	5.3	5.9	4.6	5.2
-38	42.4	65.7	60.4	81.6	45.3
Total	100.0	100.0	100.0	100.0	100.0

Table 4.21: Cr_2O_3 distribution for size-by-assay results of laboratory mill products

	Standard Product (16min)	Rougher Lights Product (8min)	Rougher Heavies Product (8min)	Rougher Lights Product (14min)	Rougher Heavies Product (2min)
Screen Size (μm)	Cr_2O_3 Distribution (%)				
+106	2.5	1.6	17.7	10.8	21.0
-106+75	13.6	12.0			25.2
-75+53	25.6	17.6	24.0	16.9	22.7
-53+38	13.4	12.8	7.6	9.8	4.5
-38	44.8	56.0	50.7	62.5	26.7
Total	100.0	100.0	100.0	100.0	100.0

The 4E distributions showed that the standard test still contained about 18% of the PGEs in the coarse (+75μm) fractions whereas when separated as lights and heavies, with 88% of the energy re-distributed to the lights, this sample contained only 3.5% of the PGEs in the -106+75µm fraction. The 12 % energy input to the rougher heavies product (2 min milling) does contain 18% of PGEs in the +75µm fraction; however, since about 62% of the total platinum minerals present in the densifier underflow are recovered to the rougher lights when using a spiral separator, the actual quantity of coarse contained platinum in the heavies is less than 7% of the total platinum content of the densifier underflow which feeds this process.

It is interesting to note that with the equal energy distribution to both the lights and heavies (8 minutes milling each), the heavies product still contained almost 12% of the 4E compared with 18% for the 2 minutes milled heavies fraction. This extra energy input (~33%) resulted in almost double the amount of Cr_2O_3 distributed to the fines (-38µm) in the heavies product of the 8 minute sample, thus it may not be worthwhile to utilise extra energy on the heavies to try and reduce the PGE content in the coarse chromitite, as it may result in an excess of fine chromite being liberated without much benefit to the platinum mineral liberation.

When comparing the rougher lights and heavies it can be noted from both Tables 4.20 and 4.21 that the 4E and Cr_2O_3 minerals are generally found in similar proportions in most size fractions. The Cr_2O_3 distributions for the 8 minute (50% energy input) and 2 minute milling (12% energy input) of the heavies in Table 4.21 showed 17.7% and 46.2% Cr_2O_3 in the coarse fractions +75µm respectively. The energy re-distribution significantly lowers the amount of chromite in the heavies product although the proportion of PGM in that fraction is slightly higher as discussed above. Table 4.22 shows the product size distributions for the non-assayed samples from the energy re-distribution tests during the laboratory batch milling tests with the associated flotation recoveries.

Table 4.22: Cumulative % passing each screen size for non-assayed laboratory milled products

4E Recovery (%)	51.9%		48.0		51.2	
Cr_2O_3 Recovery (%)	1.8		2.2		2.1	
Screen Size (µm)	12 min Lights (75%)	4 min Heavies (25%)	6 min Lights (37%)	10 min Heavies (63%)	10 min Lights (63%)	6 min Heavies (37%)
+106	98.3	89.9	92.8	98.8	97.6	95.8
-106+75	90.6	61.2	73.6	88.7	86.3	76.4
-75+53	72.8	38.3	53.2	67.2	65.6	50.5
-53+38	62.1	31.5	45.1	57.1	55.8	42.0

This data also provided further evidence of how the over-grinding of the chromite results in higher levels of chromite entrainment to the flotation concentrate. The 75% energy distribution to the lights (12 minutes milling) results in over 90% - 75µm and the 25% remaining energy that was re-distributed to the heavies ball mill shows only 61% -75µm with only 31.5% - 38µm. The resulting recovery of platinum is higher than the standard option with the Cr_2O_3 content about 22% lower than the standard test. The data also showed that the corresponding increase in the energy input to the heavies resulted in higher Cr_2O_3 recoveries and lower 4E recoveries. This also corresponds to the further fine chromite particles with the heavies material increasing to 88.7% -75µm, with 57.1% -38µm.

These results confirmed that separate milling and energy re-distribution were major contributors to the improved PGE recoveries noted (~2% 4E for the low grade ore) and the significant reduction in Cr_2O_3 entrainment of over 30% to the secondary rougher flotation concentrate.

Further test work was conducted on the effect of attritioning the chromite heavies fraction based on the outcome of these results which suggested that only a small amount of energy may be required for this material.

4.6. Attrition tests on the chromite-rich heavies fraction

Representative samples of the rougher spiral heavies product were subjected to attritioning tests with a Denver attritioning unit at 30 and 40% solids respectively. The unit was run at an impeller speed of 450 rpm for five minutes. A time of five minutes was arbitrarily selected since only 12% of the re-distributed energy (2min milling) was required for the heavies ball mill for the 250g milled sample. It was decided to select an attritioning time of a little more than double that of the ball milling time as the impact of ball milling would be more intensive. A size by assay analysis was conducted for each attritioned sample and compared with the spiral heavies product. The results are shown in Table 4.23.

Table 4.23: Size by assay analyses for attritioning at 30 and 40% solids

	Spiral Heavies Product			Attritioning with 40% solids			Attritioning with 30% solids		
Screen Size (µm)	Mass (%)	4E (%)	Cr_2O_3 (%)	Mass (%)	4E (%)	Cr_2O_3 (%)	Mass (%)	4E (%)	Cr_2O_3 (%)
+106	50.2	30.6	53.4	47.2	16.4	52.1	51.6	29.8	54.3
-106+75	34.4	34.1	32.8	31.6	45.1	28.5	29.2	35.7	27.5
-75+53	11.8	23.1	10.5	16.9	26.1	15.5	15.6	21.9	14.9
-53	3.5	12.2	3.2	4.2	12.4	3.9	3.6	11.7	3.3

The size-by-assay results showed no major difference in the metal deportment with attritioning for both 4E and Cr_2O_3 as compared to the standard spiral heavies product sample, except for the +106µm fraction of the 40% solids attritioned sample. This size fraction showed an almost 50% drop in 4E metal distribution. This may be attributed to the removal of platinum minerals which were attached to the surface of the coarse chromite particles. Since these results were encouraging, it was decided to attrition a spiral heavies sample at 40% solids and combine a lights sample which was ball milled for 14 minutes (88% energy distribution as per Section 4.5). Duplicate flotation tests were performed and the results are shown in the Table 4.24.

Table 4.24: Flotation results for silicate ball milled and attritioned chromite samples

		Recovery		
Concentrates	Mass (%)	4E (%)	Cr_2O_3 (%)	Water (%)
Test 1	5.3	53.9	1	13.5
Test 2 (repeat)	5.2	52.4	0.9	12.6
Average	5.3	53.2	1.0	13.1

The flotation results were also quite encouraging with an average 4E recovery of 53.2 % achieved for this modified circuit design. The variances in masspull, 4E, Cr_2O_3, and water recoveries for the repeated attritioned flotation tests were 0%, 1.3%, 0% and 0.4% respectively The 4E recoveries were about 2% lower than ball milling with an 88% energy distribution to the lights mill and a 12% energy distribution to the chromite ball mill (54.9 % 4E). The most significant results were noted for the Cr_2O_3 recovery which was reduced from 1.5% in the separate ball milling test to about 1% in these tests. This was a reduction in Cr_2O_3 entrainment of 52% when compared to the standard test. The further Cr_2O_3 reduction was also confirmed when comparing the water recoveries. Attritioning resulted in a 13% water recovery to the flotation concentrate as compared with about 14% for ball milling of both heavies and lights streams. This translated to the 1% Cr_2O_3 recovery in the flotation concentrate as compared with 1.5% Cr_2O_3 for the separate ball milling circuit.

Attritioning of the rougher spiral heavies (chromite) stream followed by combined flotation with the ball milled lights stream showed great potential to achieve a Cr_2O_3 entrainment of less than 1% in the secondary rougher concentrate; however, since the chromite stream constitutes about 55% of the total mass of the UG-2 densifier underflow stream, it would have to be determined if the energy costs of attritioning at an impeller speed of 450rpm would prove to be viable on an industrial scale.

4.7. Alternate milling circuit design incorporating a coarse silicates recycle from the spiral concentrator

An alternate milling circuit design was investigated for its potential to improve the platinum recovery and reduce chromite entrainment as shown by Figure 3.9 in Chapter 3.8. This configuration was investigated by conducting a locked cycle test to simulate the effectiveness of recycling the coarse silicates (+38µm). The wet mass of the +38µm recycle stream was recorded at each cycle and was found to reach steady state after the third cycle of the process. The wet mass proportions for each cycle are shown in Table 4.25.

Table 4.25: Wet Flotation results for performance of the recycle spiral concentrator configuration with a single ball mill

Locked Cycle	Streams	Wet Mass (g)	Wet Mass (%)
Cycle 1	-38µm Lights	1412	37.8
	Heavies	1645	44.0
	+38µm Lights Recycle	681	18.2
	Total	3738	100.0
Cycle 2	-38µm Lights	1386	37.5
	Heavies	1238	33.5
	+38µm Lights Recycle	1069	28.9
	Total	3693	100.0
Cycle 3	-38µm Lights	1544	35.7
	Heavies	1632	37.7
	+38µm Lights Recycle	1148	26.5
	Total	4324	100.0

The +38µm lights recycle fraction wet mass for cycle two and three were within a 10% error and therefore it was decided to conclude the tests at cycle three. The recycle stream generally constituted about 20 to 20 percent of the total feed to each cycle. The total energy to the mill was rationed according to the proportion of fresh feed for each cycle, i.e. 100% energy input (16 minutes milling for 500g fresh feed) in cycle 1, with a decreased energy input to cycle 2 and 3 to account for the proportion of recycled material. Locked cycle tests give a good approximation of a continuous operation with a recycle stream as several stages of a process can be simulated by combining batch streams and repeated until steady state is achieved as in a continuous closed circuit process.

The -38µm lights fraction was combined in mass proportion to the heavies stream and floated under standard conditions. The mass proportion of the -38µm lights fraction and the heavies were quite close to a 50:50 mass relationship in the flotation feed. The flotation tests for each cycle were conducted in duplicate to verify the repeatability of the data and the viability of the process, as these results were used to assess the potential benefit of the circuit in terms of platinum recovery and the potential reduction in Cr_2O_3 entrainment. Detailed sizing was not conducted as this circuit would only be considered if the results were encouraging. The 4E and Cr_2O_3 analyses were conducted at Mintek with repeat metal analyses performed on random samples. The average recoveries for each test are shown in Table 4.26. The individual flotation results can be viewed in Appendix B11. The variances in the masspull, 4E and Cr_2O_3 recoveries for the repeated flotation tests were 0%, 8.5%, and 0.1% respectively for cycle 1, 0.1%, 0.8%, and 0% respectively for cycle 2 and 0.2%, 1.6%, and 0.5% respectively for cycle 3.

Table 4.26: Average flotation results for performance of the recycle spiral concentrator configuration with a single ball mill

Locked Cycle Test	Avg. Mass %	Avg. 4E Recovery (%)	Avg. Cr_2O_3 Recovery (%)
Cycle 1	5.4	68.9	2.1
Cycle 2	5.4	64.2	2.5
Cycle 3	6.0	65.8	2.6
Average	5.6	66.3	2.4

The results from cycle 1 represents an open circuit process, i.e. no recycle, and this effectively quantifies the effect on the process if it was decided to discard the coarse +38µm lights recycle fraction, which made up about 20% of the feed mass. The average 4E recovery was 2.1% below the base case recovery of 70% 4E as discussed in Section 4.5. The Cr_2O_3 recovery of 2.1% was achieved and this was about 9% lower than the base case milling configuration (2.3% Cr_2O_3). The base case recovery had an 8.5% mass recovered to the concentrate as compared with 5.4% for cycle 1; however, it must be noted that this sample had a lower head grade than the earlier results as discussed in Section 4.5 and thus mass percentage recovered may be significantly lower.

The effect of recycling the coarse +38µm silicates to the secondary ball by screening the rougher spiral concentrator lights product with a 38µm screen, showed about a 4 to 7% decrease in the overall 4E recovery and a 20% increase in the Cr_2O_3 recovery as compared to cycle 1. The average results of the locked cycle test showed a 66.3% 4E recovery and a 2.4% Cr_2O_3 entrainment to the rougher flotation concentrate. The drop in 4E recovery may be attributed to decreased separation efficiency on the spiral concentrator as the recycled silicates (+38µm lights fraction) may have become finer and thus affected the operation of the spiral.

When this process is compared to the base case or conventional UG-2 secondary milling circuit there is about a 5% drop in 4E recovery and a 5% increase in the Cr_2O_3 entrainment. The increase in Cr_2O_3 entrainment would be around 11% if only cycle 2 and 3 flotation results are averaged and compared to the base case, as the average Cr_2O_3 recovery is about 2.6%.

This suggests that although this process in principle seems to be an intuitive milling circuit design for UG-2 ore (as it allows for the closed circuit milling of the platinum-rich silicates, and for open circuit milling of the high density low platinum-bearing chromite fraction from the rougher spiral concentrator) the overall flotation recoveries do not justify its implementation. On this basis, this design option was rejected and therefore detailed size-by-assay analyses were not considered.

4.8. A laboratory cleaner circuit spiral configuration

Rule and Anyimadu (2007) had indicated that Anglo Platinum had two existing concentrators (Ivan and Mortimer) which had spiral concentrators between the primary and secondary circuits with the purpose of chromite removal. The remaining material was processed in a conventional ball milling circuit. The objective of this run was to determine whether a clean chromite stream with a low platinum content could be rejected from the process and sent for chrome production, whilst only treating the platinum-rich minerals in a single ball mill.

It was therefore decided to conduct a cleaner spiral run as discussed in Chapter 3.9. Three rougher spiral runs were conducted to produce a bulk rougher heavies sample for the cleaner spiral feed. Since it was anticipated that the entire rougher lights or silicates stream would need regrinding, it was not practical to attempt to clean the lights stream which accounted for about 45% of the total feed mass. The remaining 55% of the mass which consisted predominantly of chromite and minor amounts of silicates were cleaned on a heavies cleaner spiral concentrator. A cleaner lights (silicate-rich), middling and tailings (chromite-rich) stream were collected.

Bulk streams of all samples were assayed by size. The results of the overall metal distributions to each stream are shown in Table 4.27.

Table 4.27: The lab cleaner spiral mineral distribution

Distribution (%)			
Streams	Mass (%)	4E (%)	Cr_2O_3 (%)
Lights Product	18.5	40.3	12.4
Middlings Product	17.4	14.2	16.0
Heavies Product	64.1	45.5	71.7
Total	100.0	100.0	100.0

The metal distributions for the cleaner spiral run indicated that 64.1% of the cleaner feed mass reported to the cleaner tailings product. This stream; however, consists of 45.5% of the PGEs and 71.7% of the Cr_2O_3. This data suggested that although the majority of the chromite reports to the tailings (35% of densifier underflow), the overall proportion of platinum minerals deported to this product is excessive (17% of the 4E in densifier underflow stream) therefore the rejection of the entire stream cannot be considered. The remaining 40% of the PGEs in the cleaner reported to the cleaner lights product with only 12.4% Cr_2O_3. This stream constitutes about 19% of the mass. A detailed assay by size analysis of the cleaner heavies product was therefore required to determine whether there existed a specific size interval with a high Cr_2O_3 content and a low 4E content which can be rejected from the process without drastically reducing the overall platinum mineral recovery.

Tables 4.28 to 4.30 show the mass and the 4E and Cr_2O_3 metal distributions for the cleaner spiral runs. The raw data is shown in Appendix B12.

Table 4.28: Mass distribution for the size-by-assay results of the cleaner spiral runs

	Densifier Underflow	Rougher Lights	Rougher Heavies (Cleaner Feed)	Cleaner Lights	Cleaner Middlings	Cleaner Heavies
Screen Size (μm)	Mass Distribution (%)					
+106	49.4	67.9	50.2	21.4	61.0	63.5
-106+75	29.8	18.7	34.4	49.0	29.2	26.3
-75+53	16.7	8.0	11.8	22.5	7.7	7.9
-53+38	3.0	3.3	2.3	4.9	1.5	1.7
-38	1.1	2.0	1.2	2.3	0.6	0.6
Total	100.0	100.0	100.0	100.0	100.0	100.0

Table 4.29: 4E distribution for the size-by-assay results of the cleaner spiral runs

	Densifier Underflow	Rougher Lights	Rougher Heavies (Cleaner Feed)	Cleaner Lights	Cleaner Middlings	Cleaner Heavies
Screen Size (μm)	4E Distribution (%)					
+106	28.7	48.8	30.6	10.2	36.3	32.1
-106+75	39.4	32.3	34.1	52.5	44.9	38.0
-75+53	23.5	11.6	23.1	22.8	14.4	19.1
-53+38	6.1	3.0	7.1	5.3	3.9	7.7
-38	2.3	4.3	5.1	9.3	0.5	3.1
Total	100.0	100.0	100.0	100.0	100.0	100.0

Table 4.30: Cr_2O_3 distribution for the size-by-assay results of the cleaner spiral runs

	Densifier Underflow	Rougher Lights	Rougher Heavies (Cleaner Feed)	Cleaner Lights	Cleaner Middlings	Cleaner Heavies
Screen Size (μm)	Cr_2O_3 Distribution (%)					
+106	48.5	70.7	53.4	9.4	62.3	64.5
-106+75	29.1	9.8	32.8	51.9	27.2	25.2
-75+53	17.6	8.7	10.5	28.4	8.3	8.0
-53+38	3.5	6.6	2.2	7.0	1.4	1.7
-38	1.3	4.2	1.1	3.3	0.7	0.6
Total	100.0	100.0	100.0	100.0	100.0	100.0

The results indicated that over 63% of the material in the spiral cleaner heavies product were found in the +106μm fraction. This fraction contained about 32% of the PGE in that stream together with 64.5% of the Cr_2O_3 minerals. If this stream were to be rejected from the process, it would have resulted in a 23% mass rejection which consists of about 5% of the overall PGE and 36% of the overall Cr_2O_3 minerals in the densifier underflow stream. The remaining 4E and Cr_2O_3 were distributed in the +75 and +53μm fractions, with the -53μm material containing low quantities of both minerals. The size by assay for all the other streams further provided evidence that platinum-bearing chromite minerals are generally concentrated in the same size fraction, and thus it is difficult to reject a size fraction without a significant loss in platinum recovery. Perhaps these streams could be processed using alternative strategies, but additional options may result in more complex secondary circuit designs which may not have a potential cost benefit.

An overall loss of about 5% of the PGEs through the rejection of the +106μm fraction of the cleaner heavies fraction may not be a viable process route. It would be better to try and keep those minerals in the circuit and develop improved techniques to scavenge the chromite heavies, as although discarding this fraction may reduce the overall mass of treatable secondary ore, the 5% loss of PGE revenue may be of great significance and thus this laboratory option will not be considered as an improved secondary milling circuit design.

The cleaning circuit option was also investigated on an industrial scale at SGS South Africa with a new densifier underflow sample supplied by Lonmin Platinum to determine whether the loss of platinum minerals could be substantially reduced in the chromite-rich cleaner stream based on the latest and most efficient spiral separation technology.

Figure 4.14 was plotted based on the laboratory size-by assay 4E and Cr_2O_3 grades achieved in each spiral cleaner product stream. The PGE and Cr_2O_3 grades were normalized relative to the head grade of the process, i.e. the densifier underflow.

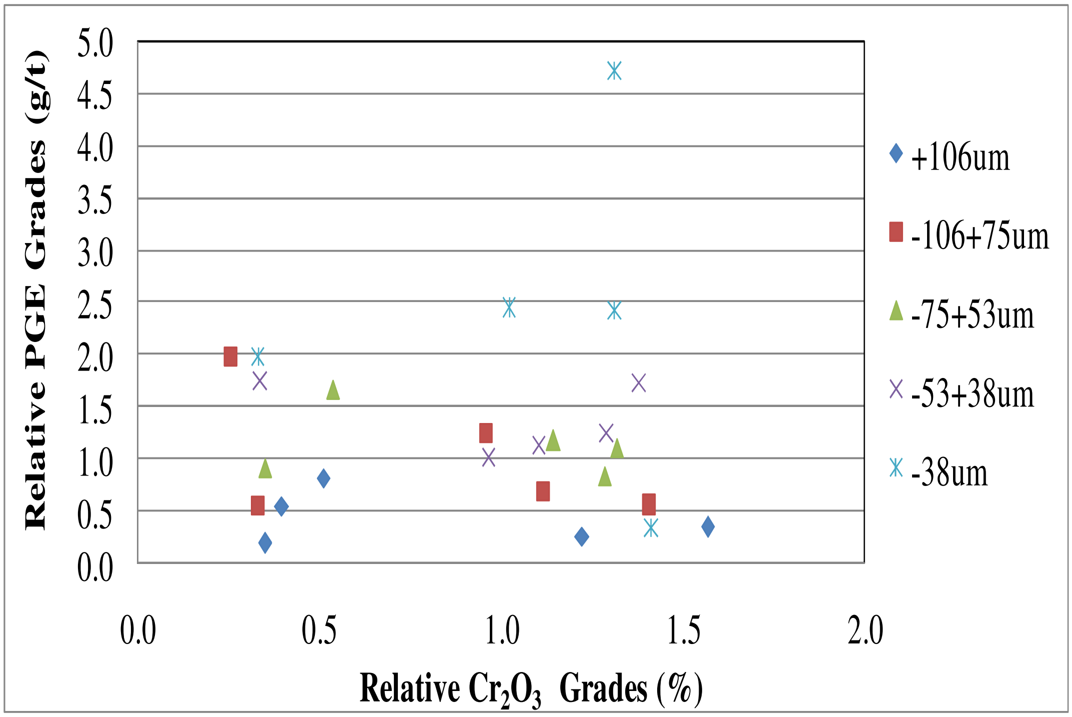


Figure 4.14: Laboratory separation with spirals - relative PGE grades versus relative Cr_2O_3 grades in each size fraction

Figure 4.14 indicated that the coarse fractions contained significant proportions of PGEs and Cr_2O_3 minerals, and although the +106µm and the -106+75µm points suggest a decreasing profile, the relative PGE grades are quite high in the high Cr_2O_3 zone (> 1% Cr_2O_3). The -38µm fractions were also found to contain significant proportions of both minerals.

Since a chromite-rich fraction could not be discarded, it was evident that separating milling of the lights and heavies with an 88% energy distribution to the lights ball mill and a 12% energy distribution to the heavies ball mill, was the most significantly improved laboratory design for the secondary milling circuit for processing the UG-2 platinum ore. This was considered as the optimal laboratory circuit design based on the results of the current testwork campaign.
Section 4.9 and 4.10 compare the flotation rate test results and the size-by-assay analyses of the flotation tailings, for the optimal laboratory circuit design and the standard milling circuit.

4.9. Comparison of flotation tailings samples

The flotation tailings for the standard process and the optimal laboratory milling circuit design were analysed for the 4E and Cr_2O_3 losses in each size fraction after conducting standard rougher flotation tests. The results are shown in Table 4.31.

Table 4.31: The size-by- assay of the flotation tailings

Avg. 4E Rougher Recovery (%)	52.8			54.9		
Avg. Cr_2O_3 Rougher Recovery (%)	2.1			1.5		
Avg. Concentrate Mass %	6.0			6.7		
Size-by-Assay	Standard rougher tailings			Optimal design rougher tailings		
Screen Size (mm)	Mass (%)	4E (%)	Cr_2O_3 (%)	Mass (%)	4E (%)	Cr_2O_3 (%)
+106	2.7	0.8	1.4	11.2	2.1	15.8
-106+75	15.5	18.2	21.5	19.6	5.8	25.3
-75+53	22.2	5.7	22.0	18.0	5.6	20.2
-53+38	8.5	1.9	8.8	5.8	4.1	5.8
-38	45.2	20.5	44.2	38.7	27.5	31.5
Total in Rougher Tailings (%)	94.0	47.2	97.9	93.3	45.1	98.5

A comparison of the rougher tailings of the standard process showed that it contained 18% of the PGEs from densifier underflow that were lost to the -106+75µm fraction. These fractions contain about 16% of the total mass. The optimal milling circuit design resulted in about an 8% total PGE loss in the +106 and -106+75µm fractions (~11% coarse 4E reduction) in a larger mass of about 31%. About 2% of the PGE were lost to the +106µm in the optimal rougher tailings as compared to 0.8% 4E for the standard rougher tailings. The corresponding Cr_2O_3 minerals present in the +106 and -106+75µm fractions of the optimal design was about 18%

higher as compared with the standard tailings. This showed the benefit of separate milling as the over-grinding of chromite can be significantly reduced. This was confirmed by comparing the -53+38μm and -38μm fractions, where the optimal process resulted in about a 16% lower Cr_2O_3 mineral content. The marginally higher PGE (2.1%) in the +106μm of the optimal design may also indicate that a proportion of the PGM are associated with the chromite minerals.

A recap of the plant audit data is shown in Table 4.32.

Table 4.32: Plant audit SRT sample size-by-assay data

Secondary Rougher Tails (SRT)		Distribution (%)	
Screen Size (μm)	Mass (%)	4E	Cr_2O_3
+106	8.9	12.2	8.7
-106+75	23.8	24.9	23.5
-75+53	11.3	11.6	11.1
-53+38	5.5	5.7	5.4
-38	50.5	45.6	51.2
Total	100.0	100.0	100.0

When comparing the results of the secondary rougher tailings from the concentrator with that of the optimal laboratory milling circuit tailings, it was evident that this new milling circuit configuration definitely reduces the coarse platinum mineral losses (37.1% PGEs in SRT) as compared to 17% PGEs for optimal design. The SRT also compared quite well with the laboratory tailings of the standard method which indicates that the milling and flotation characteristics in the laboratory are a good indication of the plant performance. An additional 10% of the chromite minerals are not over-ground when comparing the Cr_2O_3 distribution in the coarse fractions for the SRT and optimal laboratory design.

The results show that the implementation of the separate milling process for the silicates and chromite particles definitely reduces the losses of coarse PGEs in the secondary tailings which are normally a major concern to platinum producers. This optimal laboratory secondary milling circuit design has demonstrated the ability to deal with this problem by using the spiral concentrator to achieve a good separation of low density and high density material. The following section looks at the degree of fast floating material produced into the secondary circuit by comparing the cumulative flotation concentrates of the standard circuit and the optimal circuit design.

4.10. Flotation rate tests for optimal circuit design

Once the optimal laboratory circuit design was established it was decided to collect more timed concentrates during the rougher flotation tests to determine the grade versus recovery relationship. It was also important to investigate whether the optimal process design could liberate more fast floating platinum minerals into the pulp through the modified secondary milling circuit configuration. Three timed concentrates of two, five and ten minutes were collected for each test and the repeatability was also confirmed by conducting a bulk rougher test to compare the final combined concentrate grades, as well as the tailing grade for both the standard process and optimal separate milling process. The water recovery for each concentrate was also noted. The grades for these tests were normalised to illustrate the effect of the grade recovery relationship of the two processes as thus far only recoveries were mentioned. Figures 4.15 to 4.19 show the comparison of the two processes.

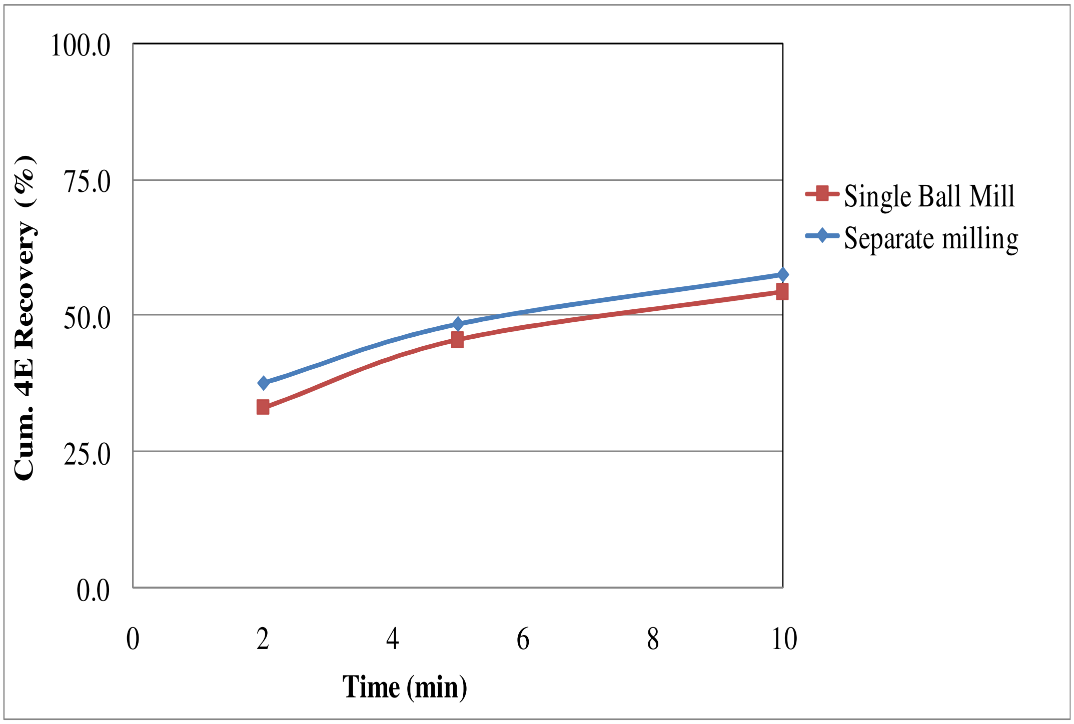


Figure 4.15: Cumulative 4E recovery versus time

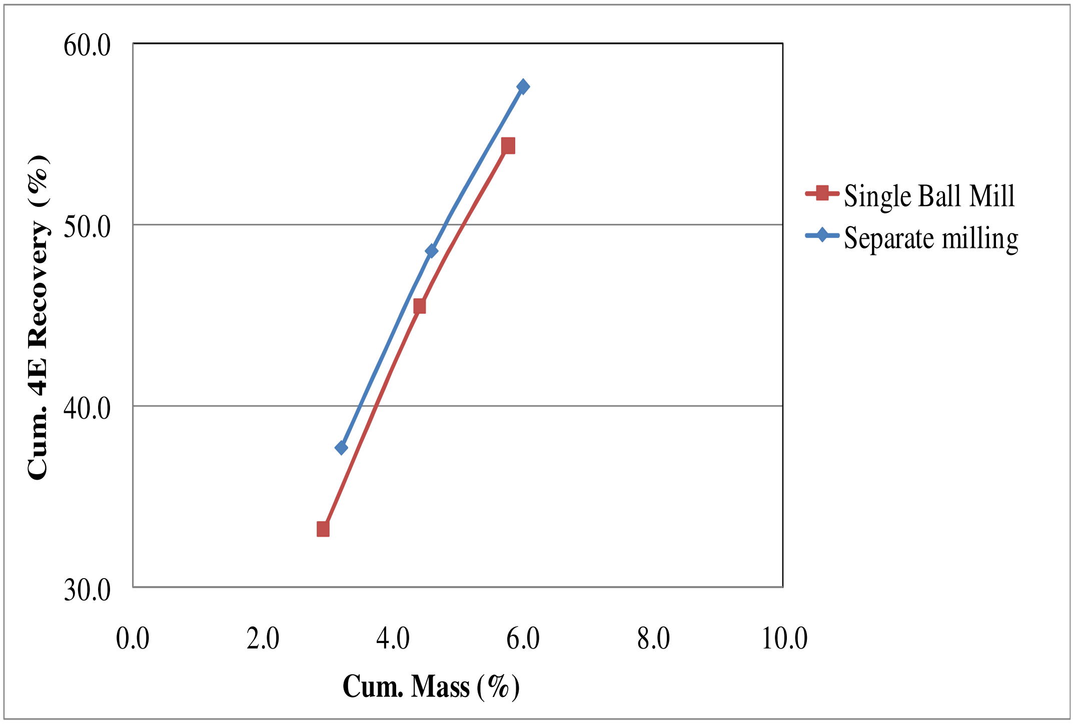


Figure 4.16: Cumulative 4E recovery versus cumulative mass %

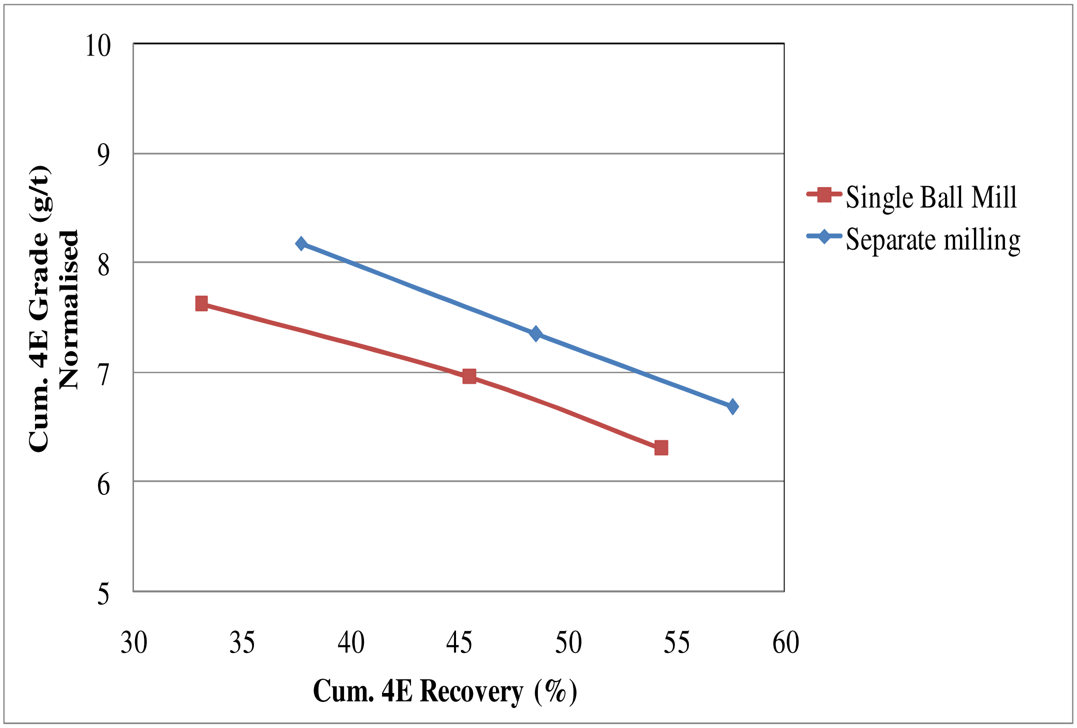


Figure 4.17: Normalised 4E grade versus cumulative 4E recovery

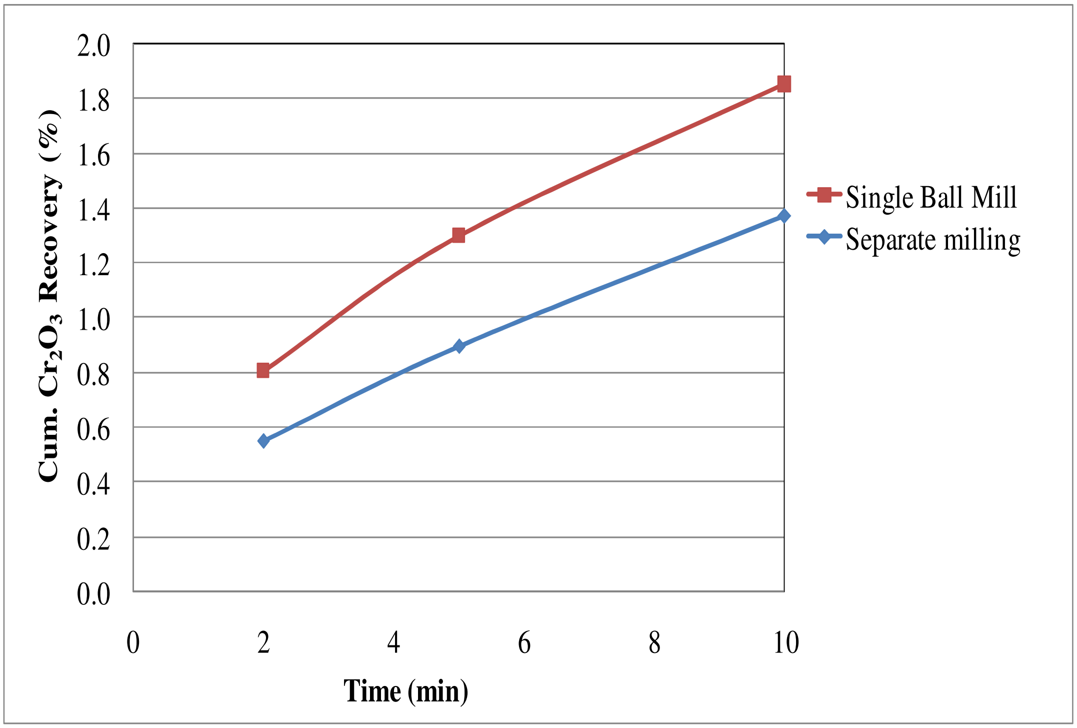


Figure 4.18: Cumulative Cr_2O_3 recovery versus time

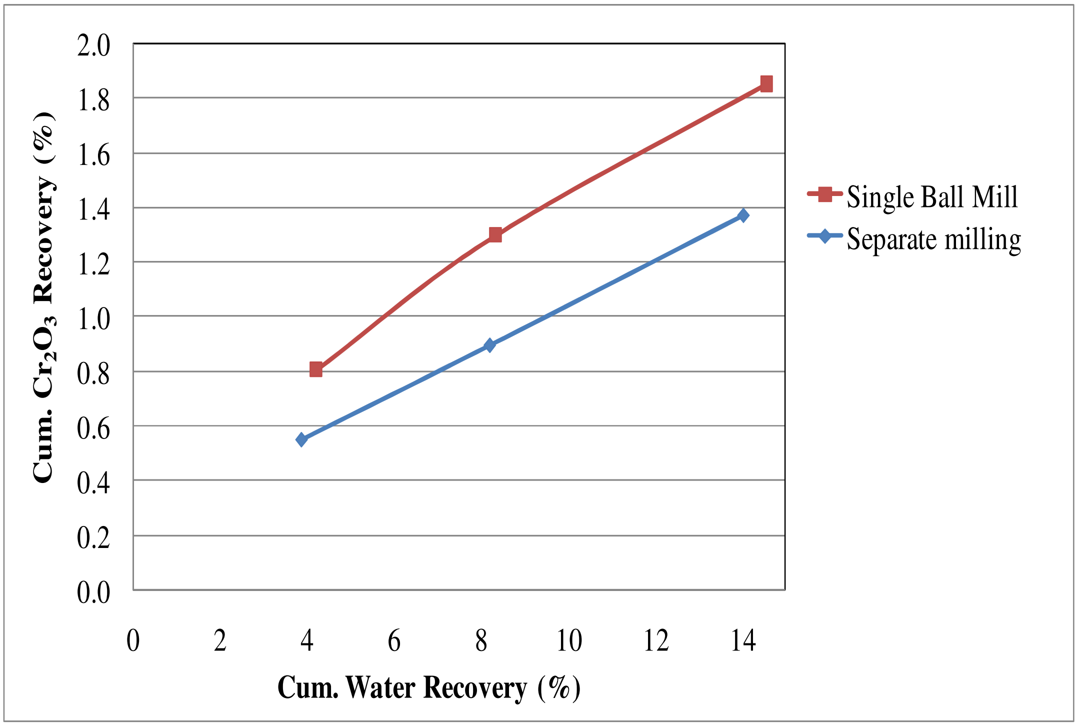


Figure 4.19: Cumulative Cr_2O_3 recovery versus cumulative water recovery

The cumulative flotation results show the benefit of separate milling on the flotation performance. Figures 4.15 and 4.16 showed that the cumulative 4E recovery for the separate milling process was marginally higher than the conventional process. Separate milling resulted in an overall recovery of 57.6% PGE as compared with 54.1% in the standard (single ball mill) process. The recalculated recoveries based on the cumulative concentrates were within a 5% variance of the low grade duplicate tests for both processes as discussed in Section 4.5. The overall mass recovery to the concentrate was marginally higher (<1%) which may have contributed to the slightly higher mineral recoveries that were noted; however, the overall results have shown good repeatability and consistency.

The normalised grade versus recovery curves were plotted for the 4E rate data and this graph (Figure 4.17) showed a positive shift in the grade versus recovery relationship for the platinum minerals. This indicated that the optimal laboratory circuit with separate milling has an advantage of increasing the amount of high grade floatable platinum minerals as compared to the conventional process. This would improve the overall platinum concentrate grade, which will be of high revenue especially after further upgrade in the cleaners and re-cleaners.

The reduction in the Cr_2O_3 recovery in the separate milling process route was the main benefit especially when considering the large smelter penalties that platinum producers face for high Cr_2O_3 content (> 3% after re-cleaning). Figures 4.18 and 4.19 showed evidence of this benefit as each cumulative concentrate differs by about a factor of two as the flotation time was increased, with the first high grade concentrate having a 0.2% lower recovery for the separate milling option. The comparisons of the cumulative Cr_2O_3 recoveries versus the cumulative water recoveries shown in Figure 4.18 provided further evidence of the severity of Cr_2O_3 entrainment for the standard process as compared to the separate milling design. A large drop in the water recoveries were noted as compared to over-grinding of the chromite particles in the standard process (Fig. 4.19).

A scatter graph shown in Figure 4.20 provided a summary of all the 4E flotation recovery data versus the mass percent recovered to the rougher concentrate during the laboratory test work. This scatter of results was based on the bulk concentrates for the various tests for the low grade 4E sample to assess the overall consistency of the flotation process.

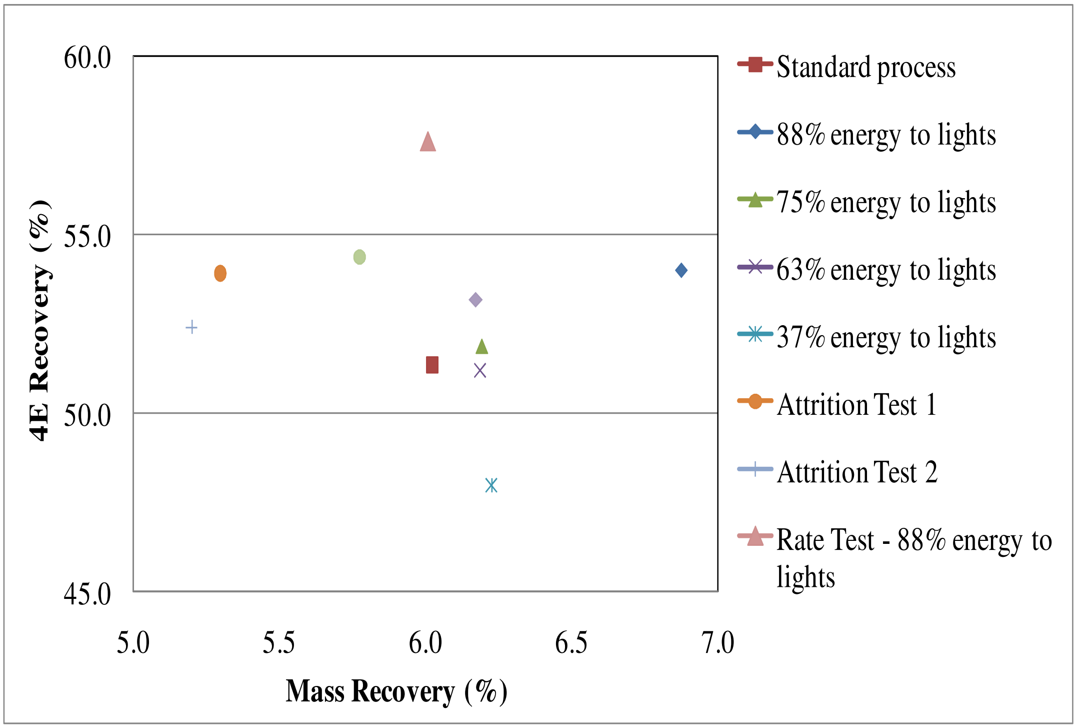


Figure 4.20: The bulk 4E platinum mineral recoveries for the low grade sample

Figure 4.20 indicated that the 4E recoveries are fairly consistent within about a 5% scatter for most tests. The mass percentage was predominantly between 5 to 6.5%; however, the change in PGE recoveries were marginal. This data clearly indicated that the 88% energy input to the lights mill was the optimal laboratory result with a mass pull of 6% to the concentrate and an overall recovery of 57.6% PGE, based on the flotation rate test. The overall performance of this improved secondary milling circuit design based on a triplicate set of flotation results for the low grade ore reveals an average PGE recovery of 55% with a 6.7% mass pull.

These results were quite promising and were thereafter compared with the results of the pilot plant spiral test work at SGS South Africa which is discussed in the following section. This test work served as a confirmation of the laboratory results achieved based on silicate and chromite separations conducted on the MG-type spiral concentrator. It was expected that the latest spiral concentrator supplied by Downer EDI Mining (Roche) from Richards Bay may produce a more efficient gravity separation and more favourable flotation results. It would also be investigated whether current spiral concentrators could be operated to produce a relatively clean (low 4E) discardable chromite fraction.

CHAPTER 5: PILOT PLANT RESULTS AND DISCUSSION

5.1. Comparison of new densifier underflow sample supplied by Lonmin Platinum to SGS for pilot plant campaign

A new sample of the densifier underflow was collected from the same UG-2 concentrator at Lonmin Platinum in September 2010. This was supplied to SGS South Africa for the pilot plant campaign. It was therefore necessary to analyse the feed stream for comparative purposes with the 2008 plant audit sample, which was used for the entire laboratory campaign. The pilot plant feed sample (densifier underflow) was found to have a platinum 4E mineral grade of about 30% lower than the low grade laboratory sample with quite comparable Cr_2O_3 grades.

The size-by-assay profiles for the 2008 and 2010 densifier underflow (rougher spiral feed) are shown in Table 5.1.

Table 5.1: Comparison of size-by-assay results for the densifier underflow samples

	2010 Densifier underflow sample for SGS run			2008 Densifier underflow sample for laboratory work at UKZN		
Screen Size (µm)	Mass (%)	4E (%)	Cr_2O_3 (%)	Mass (%)	4E (%)	Cr_2O_3 (%)
+106	52.7	30.4	46.1	49.4	30.8	46.0
-106+75	26.5	17.8	28.4	29.8	37.2	31.0
-75+53	12.3	14.8	15.1	16.7	23.6	18.3
-53+38	2.3	6.0	3.2	3.0	6.2	3.4
-38	6.2	31.0	7.3	1.1	2.2	1.4
Total	100.0	100.0	100.0	100.0	100.0	100.0

The densifier underflow samples were found to be quite similar in mass and Cr_2O_3 distribution throughout all size fractions. The 4E distribution for the +106µm fraction were also comparable; however, the main difference can be noted in the +75 µm fraction which has decreased by about 20% in PGE distribution since the sampling campaign in 2008. This increase in PGEs has been re-distributed mainly in the -38µm fraction which increased by almost 30% 4E as compared to the 2008 sample. This may be attributed to a few possibilities such as the variation in the PGE content due to ore mineralogy or an improved liberation in the primary crushing and milling circuit. The head grade of this new sample was found to be about 30% lower than the 2008 low grade 4E sample; however, the overall properties of the sample were fairly similar.

5.2. The cleaner spiral confirmation test work at SGS

The SGS pilot plant test work campaign was conducted on two VHG spiral concentrators which were run in a similar configuration as done in the laboratory at UKZN (Figure 3.10 in Section 3.9). The SGS tests served two main purposes, i.e. firstly to compare the optimal process flowsheet design achieved in the laboratory on the MG-type spiral with the results of the VHG spiral concentrator run, and secondly to determine whether a clean chromite fraction could be discarded prior to secondary milling, as this may potentially reduce the overall tonnage to the secondary ball milling circuit. The engineers at SGS commissioned the pilot plant circuit with the objective of achieving a clean chromite heavies stream for rejection from the concentrator circuit. This stream would be chromite-rich and could be potentially used for ferrochrome production.

Each spiral product stream received from SGS was analysed in duplicate for 4E and Cr_2O_3 grades at Mintek. The densifier underflow was analysed in triplicate as this was the feed to the spiral circuit. The percentage variances in these repeat grade analyses were generally below 4% for the majority of the streams, with only two streams, i.e. densifier underflow and rougher lights having a variance of about 8% and 12% respectively. Majority of the data was within the industry norm of a 10% variance. These comparisons shown in Appendix B14 verify the reproducibility of the sample preparation technique used in the laboratory at UKZN and the consistency of the grades. The mass and mineral balance results for the pilot plant spiral circuit are shown in Tables 5.2, 5.3 and 5.4. The comparative laboratory results achieved at UKZN on the full-scale MG-type spiral concentrator have been included with these results in each case.

Table 5.2: Comparison of SGS and UKZN rougher spiral circuit results

Rougher Spiral Run	SGS Distributions (%)			UKZN Distribution (%)		
Streams	Mass (%)	4E (%)	Cr_2O_3 (%)	Mass (%)	4E (%)	Cr_2O_3 (%)
Lights Product	6.1	7.5	2.5	45.0	62.4	22.2
Heavies Product	93.9	92.5	97.5	55.0	37.6	77.8
Total	100	100	100.0	100	100	100.0

The mass distribution of the SGS spiral circuit as compared to its 4E and Cr_2O_3 distributions showed a 6.1 % mass split to the rougher lights stream as compared to 45% mass distribution to the lights noted for the UKZN run. The SGS run targeted a clean chromite discard stream and therefore the emphasis of the operation focused on ensuring that most of the chromite particles reported to the cleaner spiral feed. The spiral circuit was commissioned such that a relatively clean rougher lights product was achieved in 1.8% of the mass with a 2.5% 4E and 0.7% Cr_2O_3 distribution. It was expected to recover the majority of the remaining PGEs (92.5%) in the cleaner lights and middlings streams. The heavies product stream (cleaner spiral feed stream) contained 93.9% of the overall feed mass distribution with the 4E and Cr_2O_3 mineral distributions found to be proportional to the mass distribution. The UKZN results showed an almost equal split in mass distribution with 22.2% Cr_2O_3 and 62.4% PGEs reporting to the rougher lights product.

Table 5.3: Comparisons of the SGS and UKZN cleaner-only spiral run results

Cleaner-only Spiral	SGS Distributions (%)			UKZN Distribution (%)		
Streams	Mass (%)	4E (%)	Cr_2O_3 (%)	Mass (%)	4E (%)	Cr_2O_3 (%)
Lights Product	1.8	2.5	0.7	18.5	40.3	12.4
Midds Product	39.4	28.1	11.3	17.4	14.2	16.0
Heavies Product	58.7	69.4	88.0	64.1	45.5	71.7
Total	100.0	100.0	100.0	100.0	100.0	100.0

The cleaner spiral results showed that the SGS run achieved an efficient gravity separation of chromite as about 88% of the Cr_2O_3 was concentrated to the heavies stream in about 60% of the mass; however, almost 70% of the PGEs in the cleaner feed also followed the Cr_2O_3 minerals to the heavies product.

Table 5.4: Comparisons of the overall SGS and UKZN spiral circuit results

Overall Spiral Circuit	SGS Distributions (%)			UKZN Distribution (%)		
Streams	Mass (%)	4E (%)	Cr_2O_3 (%)	Mass (%)	4E (%)	Cr_2O_3 (%)
Rougher Lights	6.1	6.8	2.4	45.0	64.1	23.3
Cleaner Lights	1.7	2.3	0.7	10.2	14.4	9.5
Cleaner Midds	37.0	26.2	11.0	9.5	5.1	12.2
Cleaner Heavies	55.1	64.7	85.9	35.3	16.3	55.0
Total	100.0	100	100.0	100.0	100	100.0

The overall spiral circuit results for the SGS and UKZN runs confirm that the rejection of the cleaner heavies product stream is not viable as majority of the PGEs are contained in this stream (64.7% 4E for the SGS run). Although the UZKN cleaner heavies stream contained a fairly small amount of PGEs (16.3% overall) as compared to the SGS run, it would not be feasible to discard such a large proportion of potential platinum revenue rather than exploring downstream reprocessing options. It is evident from the SGS pilot plant run that the platinum minerals are associated with the chromite to some degree as a large proportion of platinum (64.7% 4E) followed the 85.9% Cr_2O_3 minerals to the cleaner heavies stream.

Figure 5.1 compares the relative PGE and Cr_2O_3 grades for the spiral product streams for both the UKZN and SGS run normalized relative to the process head grade, i.e. densifier underflow feed.

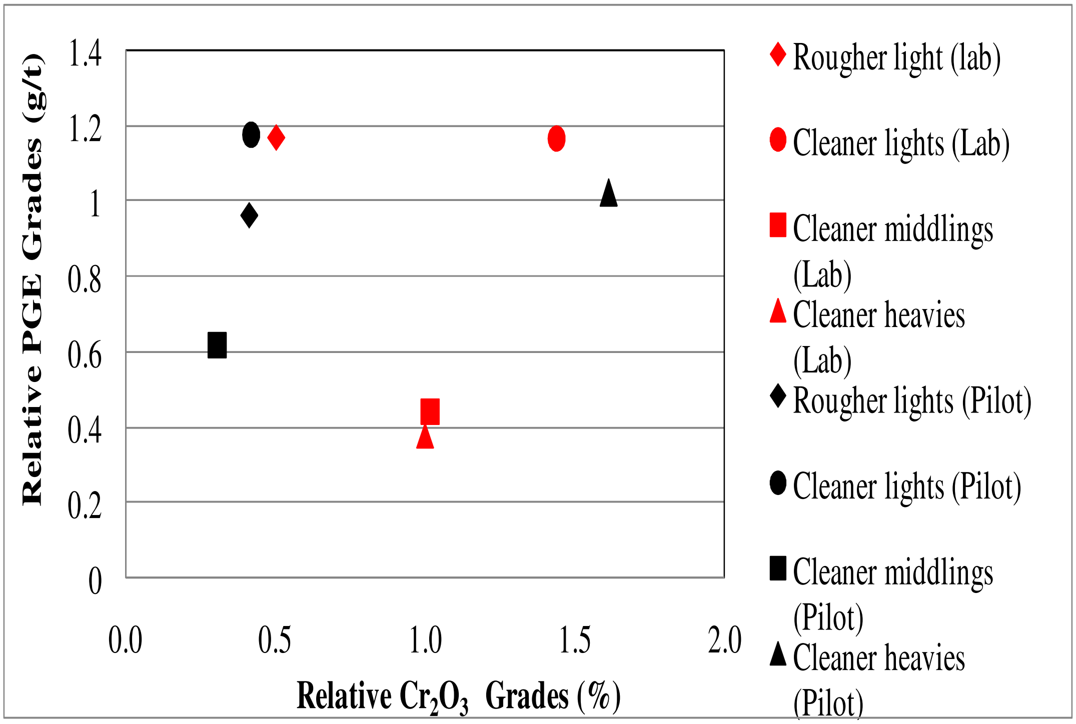


Figure 5.1: Separation with UKZN and SGS spirals -relative PGE grades versus relqtive Cr_2O_3 grades

It is evident from Figure 5.1 that the SGS pilot run resulted in a high grade Cr_2O_3 product in the cleaner heavies stream as compared to the UKZN laboratory run; however, the PGEs were also concentrated in this stream. This also provided further evidence that the PGMs are associated with the chromite minerals, and therefore it may be difficult to achieve a relatively pure chromite stream for discard.

Size-by-assay analyses were thereafter conducted to determine whether a clean chromite stream existed in a particular size fraction from the middlings and heavies spiral product streams for the pilot plant circuit.

5.3. Size-by-assay analyses

The cleaner middlings and heavies streams were analysed to determine the distribution of 4E and Cr_2O_3 minerals in each size fraction. This was conducted to determine whether a chromite-rich size fraction with trace PGEs was discardable. Laboratory size-by-assay of the UKZN cleaner spiral run in Section 4.8 had already indicated that the 4E losses were quite high (>5% 4E loss overall) and therefore did not warrant the consideration of such an option, so these tests were conducted to confirm the laboratory findings. The cleaner middlings and cleaner heavies mineral deportment results based on the SGS pilot plant circuit run are shown in Table 5.5.

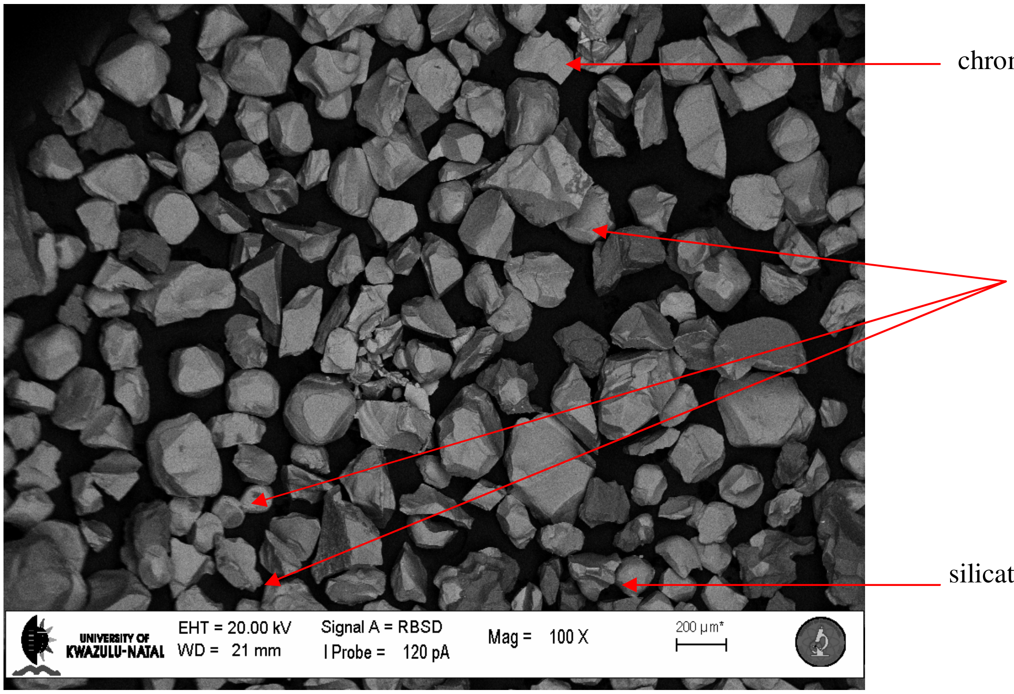
Table 5.5: Size-by-assay of the SGS cleaner middlings and heavies product streams

	Cleaner Spiral - Middlings Product			Cleaner Spiral - Heavies Product		
Screen Size (µm)	Mass (%)	4E (%)	Cr_2O_3 (%)	Mass (%)	4E (%)	Cr_2O_3 (%)
+106	69.2	65.4	66.3	45.8	21.0	47.7
-106+75	21.9	22.5	18.8	32.7	22.5	31.9
-75+53	5.7	4.7	5.0	13.9	19.0	13.0
-53+38	1.1	0.7	2.7	2.7	8.8	2.6
-38	2.2	6.5	7.2	5.0	28.8	4.8
Total	100.0	100.0	100.0	100.0	100.0	100.0

These results also confirmed the laboratory findings. It was evident that the PGE and chromite minerals cannot be exclusively separated using this approach without employing further separation devices downstream. The chromite-rich fractions, i.e. +106 and +75µm size intervals for both the middlings and heavies streams contained large proportions of the PGEs as seen in Table 5.5. The cleaner heavies stream contained 55% of the overall mass distribution with 64.7% 4E and 85.9% Cr_2O_3. Table 5.5 showed that the +106µm fraction of the cleaner spiral heavies product contained 21% PGEs and 47.7% Cr_2O_3 in 45.8% of the mass therefore it would result in the removal of about 25% of the total mass from the process, about 40% of the overall Cr_2O_3 content and 14% of the overall PGEs. Majority of the PGEs are found in the coarse fractions, i.e. +106. 75, and +53µm which indicated that these fractions would need further regrinding to liberate more PGEs to the secondary flotation feed.

The overall mass and Cr_2O_3 distribution was similar to the laboratory cleaner sizing results discussed in Section 4.8 (rejection of 23% total mass, 36% Cr_2O_3 and 5% 4E overall); however, the PGE loss was excessive (14% overall) and therefore it would not be recommended to reject a chromite-rich fraction from the process. Separate milling of this stream is required as it contained a large proportion of PGEs; however, the milling energy would have to be much lower. Separate reprocessing of this +106µm fraction may be considered; however, this may result in a more complex circuit which is normally difficult to operate.

Samples of the sized SGS and UKZN cleaner heavies spiral products were examined under a Scanning Electron Microscope (SEM) unit at the UKZN EM unit. A small sample of each size fraction was prepared, coated with carbon and analysed to observe whether any evidence of locked silicates were visible in these chromite fractions under the high magnification settings. Since the sample masses were low (<1g) no statistical relevance can be inferred to the entire size fraction based on these images, and the results can be treated as random observations of these particles. Images which incorporated the largest possible amount of particles were included to have a general view of some major occurrences. It must be noted that that the light particles represented the high density chromite particles and the darker particles denoted the silicate particles. Figures 5.2 and 5.3 are the images of the 106µm fraction of the SGS and UKZN cleaner spiral heavies' streams respectively which have been included for comparison purposes. The remaining images are available in Appendix B17.



chromite particle

Some random occurrences of possible attached silicates on chromite particles

silicate particle

Figure 5.2: SEM image of the +106µm fraction from SGS cleaner heavies stream (silicates appear darker)

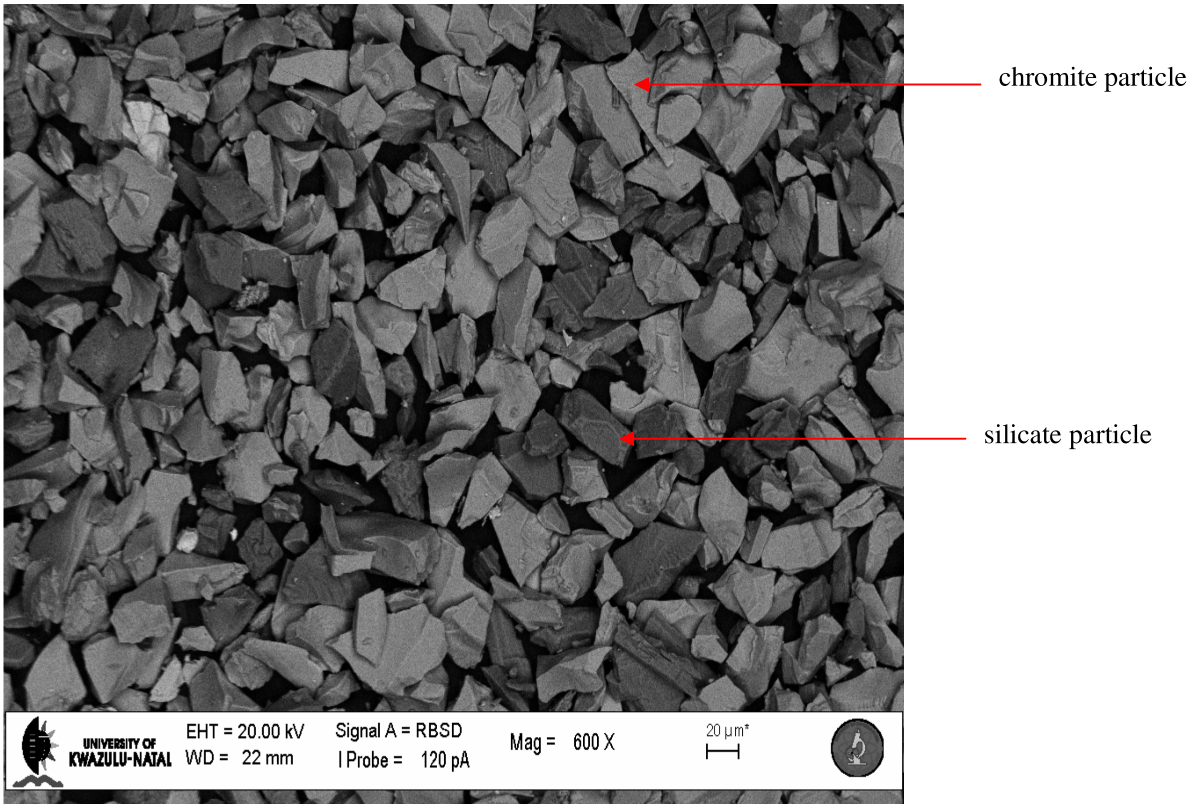


Figure 5.3: SEM image of the +106µm fraction from UKZN cleaner heavies stream (silicates appear darker)

The SEM images in Figures 5.2 and 5.3 indicate that the +106µm fractions from both the SGS and UKZN cleaner spiral heavies streams contain a substantial amount of coarse liberated silicate particles based on the analysis of these grab samples. There are some noticeable chromite particles as indicated in Figure 5.2 that seem to have silicate particles attached on the surfaces; however, since these are not polished cross-sections, no evidence of locked silicates can be noted. These images do; however, indicate that the chromite and silicate may be attached to some degree and thus may require some milling or attrition scrubbing. Since the sample quantities analysed by SEM were quite low, these observations do not provide conclusive evidence.

A few scoping flotation tests investigated the option of implementing a chromite-rich discard of the entire cleaner spiral heavies stream and the results are briefly covered in the following section.

5.4. Spiral lights-only flotation results

It was initially decided to quantify the impact of discarding the cleaner spiral heavies product stream on the 4E and Cr_2O_3 mineral secondary flotation rougher recoveries. These flotation tests were completed prior to having received the 4E and Cr_2O_3 mineral distributions of the cleaner spiral product streams. Two lights-only flotation tests were conducted in duplicate with two different energy inputs to the ball mill. The first test was milled with 88% of the total standard energy input and with the second test milled with 63% of the total standard energy distribution. The overall 4E recovery to the rougher concentrate based on an averaged duplicate tests was less than 20% with a 0.7% Cr_2O_3 recovery. This provided further evidence that discarding the heavies fraction severely reduced the overall platinum recovery, as the SGS run contained significant proportions of PGEs in the chromite-rich stream. The remaining flotation tests therefore focused on comparing the standard process of conventional ball milling to the optimal laboratory milling circuit configuration.

5.5. Comparative flotation results based on the optimal laboratory secondary milling circuit design

The SGS spiral circuit run confirmed that chromite rejection prior to secondary ball milling was not viable as the platinum minerals followed the chromite minerals to the cleaner heavies stream resulting in high PGE losses. The SGS run also achieved an extremely efficient gravity separation of chromite as 85.9% of the Cr_2O_3 minerals were distributed to the cleaner heavies product stream in 55% of the total mass. This was very encouraging and based on the overall mass distribution shown in Table 5.4, suggested the option of exploring the viability of combining the rougher lights, cleaner lights and cleaner middlings streams into a bulk lights (silicates-rich) stream to be milled in the lights ball mill. This stream contained 45% of the total mass, 35.3% of the PGEs and most importantly only 14.1% Cr_2O_3 minerals.

Although the combined stream had a lower 4E distribution as compared to the UKZN rougher spiral run (62.4% PGEs in 45% of the total mass), the lower Cr_2O_3 mineral content may prove advantageous to the overall process when considering separate ball milling with an 88% energy re-distribution to the lights mill. This may have the potential to result in further reductions in the chromite recoveries to the secondary rougher concentrate; however, the effect on the PGE recovery would be an obvious concern as the majority of the PGEs (64.7%) would now only receive 12% of the regrinding energy input as they are concentrated in the heavies ball mill.

This milling option was investigated as it was important to determine the effect on the flotation recoveries of the PGEs and Cr_2O_3 minerals as compared to the standard process.

The flotation tests for the standard milling option were performed in duplicate to benchmark the conventional single ball milling approach based on this new sample. The separate milling option (88% energy input to lights mill, 12% energy input to heavies mill) was conducted in triplicate to verify all results obtained. It was also to be determined whether these results would confirm the laboratory test work conclusions which indicated that this option was the optimal milling circuit design. Three concentrates were collected to compare the amount of fast floating material, the normalised grade-recovery relationship and the effect of water recovery on chromite entrainment. The rougher flotation results are tabulated in Tables 5.6 and 5.7.

Table 5.6: Rougher flotation results for the standard method for the SGS spiral circuit run

Standard Tests	Mass %	4E Recovery (%)	Cr_2O_3 Recovery (%)
Test 1	5.6	64.8	2.0
Test 2 (Repeat)	6.0	64.6	2.4
Average	5.8	64.7	2.2

Table 5.7: Rougher flotation results for the separate milling option for the SGS spiral circuit run

Separate Milling	Mass %	4E Recovery (%)	Cr_2O_3 Recovery (%)
Test 1	5.9	68.4	1.4
Test 2 (Repeat)	6.3	68.8	1.7
Test 3 (Repeat)	5.6	68.1	1.3
Average	5.9	68.4	1.5

The repeated secondary rougher flotation results in Tables 5.6 and 5.7 have confirmed that the separate milling secondary circuit design with an 88% energy distribution to the lights mill and 12% energy to the heavies mill was significantly improved. An average 4E recovery benefit of 3.7% was noted with a 32% reduction in the Cr_2O_3 recovery to the secondary rougher concentrate. The variances in the masspull, 4E, Cr_2O_3, and water recoveries for the repeated standard flotation tests were 0.1%, 0%, 0.1% and 1.1% respectively. The variances in the masspull, 4E, Cr_2O_3, and water recoveries for the repeated separate milling flotation tests were 0.1%, 0.1%, 0% and 0.2% respectively. Both the UKZN and SGS spiral circuit runs followed by separate batch ball milling and combined flotation resulted in a similar overall benefits to the process (~2% 4E recovery benefit and about 30% Cr_2O_3 reduction for UKZN low grade 4E sample) thus these results validate the laboratory test work campaign on the MG-type spiral. Combining the rougher lights, cleaner lights, and cleaner middlings streams for separate milling in the lights ball mill proved to be beneficial to the overall process. This data provides conclusive evidence that the incorporation of gravity separation followed by secondary milling with energy re-distribution is justified in the secondary milling circuit for a UG-2 concentrator.

Sizing of the mill products were also conducted during the laboratory milling and flotation tests to determine the variation in ore hardness as compared to the UKZN low grade 4E sample. The comparisons are tabulated in Figure 5.8 with the 4E and Cr_2O_3 rougher flotation recoveries.

Table 5.8: Comparison of sized mill products for laboratory and pilot plant samples

Avg. 4E Recovery (%)	52.8	64.7	54.9	68.4	54.9	68.4
Avg. Cr_2O_3 Recovery (%)	2.1	2.2	1.5	1.5	1.5	1.5
Mass % to concentrate	6.0	5.8	6.7	5.9	6.7	5.9
Sizing	Standard milling		88% Ball milling of Lights		12% Ball milling of Heavies	
	UKZN	SGS	UKZN	SGS	UKZN	SGS
Screen Size (µm)	Mass (%)					
+106	4.1	2.8	1.4	1.7	16.4	18.1
-106+75	15.9	28.2	7.3	9.8	37.2	38.4
-75+53	24.2	17.8	19.7	19.0	19.9	16.9
-53+38	12.0	4.5	9.4	6.1	3.8	5.4
-38	43.8	46.7	62.2	63.5	22.7	21.2

The ore utilized for the pilot plant spiral test work was found to be harder than the plant audit ore from the same UG-2 concentrator, as a grind of 69% -75µm was achieved for the pilot plant sample as compared to 80% -75µm for the laboratory sample. The lights and heavies mill product results do not vary significantly with the SGS separately milled products containing about 1 to 2% more material in each size fraction. Although the new ore was found to be slightly harder and of a lower 4E grade, the overall PGE recovery was over 10% higher for a similar mass pull to the concentrate.

Figures 5.4, 5.5 and 5.6 show the 4E and Cr_2O_3 comparisons for the standard and separate milling circuit flotation results based on the recovery of stage-wise concentrates. The variances in masspull, 4E, Cr_2O_3, and water recoveries for the repeated standard flotation rate tests were 0.1%, 1.9%, 0.2% and 0.7% respectively. The variances in the masspull, 4E, Cr_2O_3, and water recoveries for the repeated separate milling flotation rate tests were 0%, 0.1%, 0.1% and 0.1% respectively.

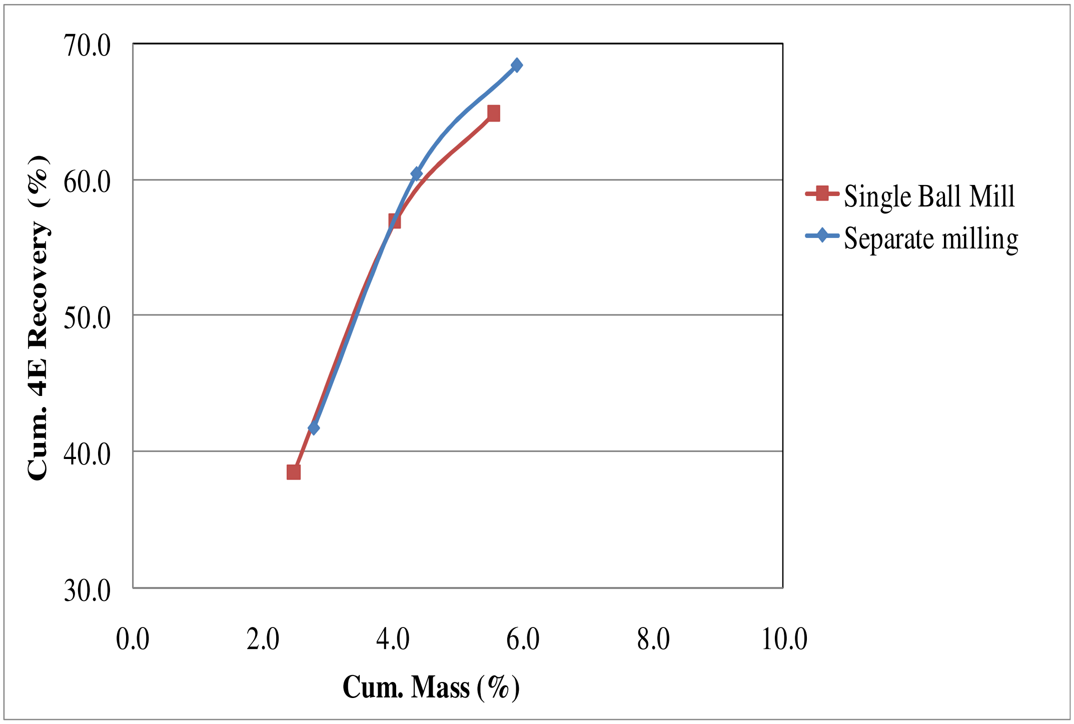


Figure 5.4: Cumulative 4E recovery versus cumulative mass recovery for flotation tests

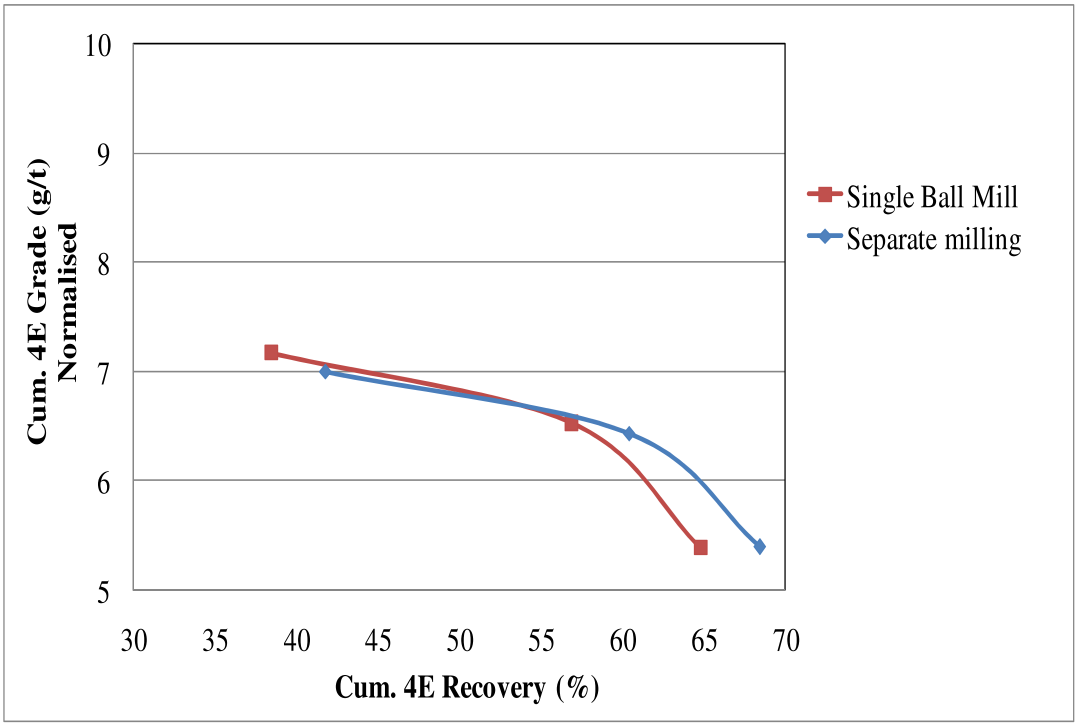


Figure 5.5: Normalised 4E Grade versus cummulative 4E recovery for flotation tests

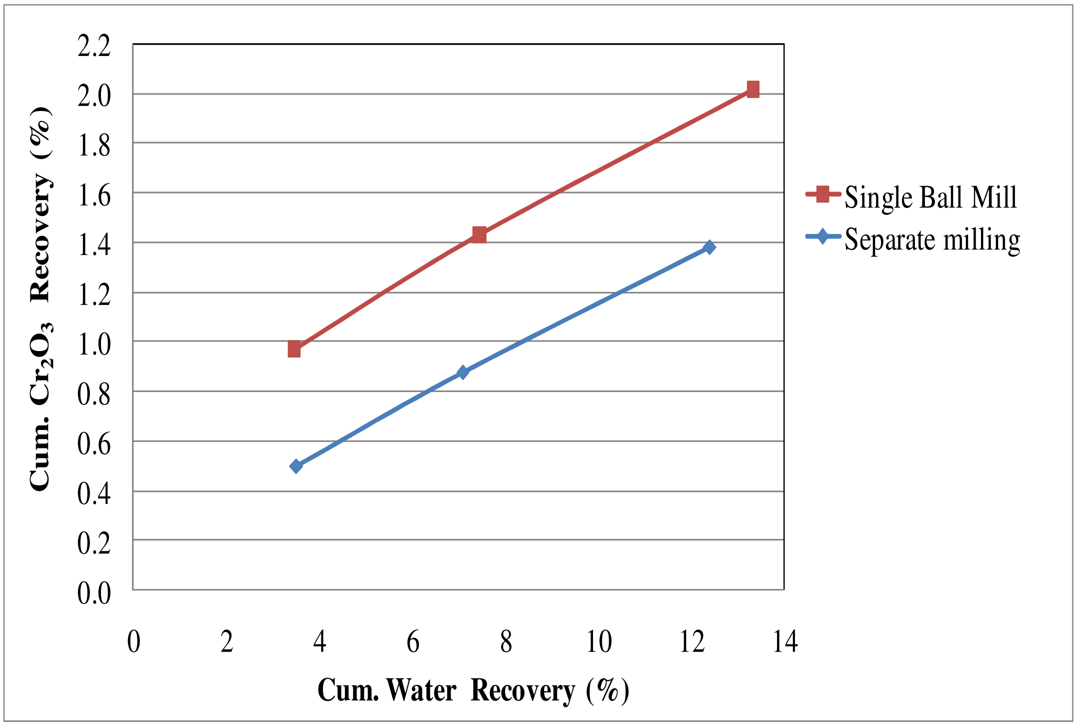


Figure 5.6: Cumulative Cr_2O_3 recovery versus cumulative water recovery for flotation tests

Figures 5.4 and 5.5 show that the cumulative 4E recoveries relationships are quite similar for both the single ball mill and the separate milling option, with minor benefits in the 4E recoveries noted in the last concentrate for both profiles. The major advantage of the separate milling process is noted in Figure 5.6 as although a marginal gain in PGE recovery is achieved (about 4% in secondary rougher of densifier underflow), the reduction in chromite entrainment is substantial (over 30% Cr_2O_3). The separate milling process showed a significant decrease in cumulative water recovery with a large shift in the chromite curve noted for the separate milling option. This data was comparable with the laboratory results in Section 4.10 and further validates the benefit of a gravity separation circuit followed separate milling of the chromite and silicate streams.

A comparison of the secondary rougher flotation tailings for both milling circuit options was also conducted and is discussed in Section 5.6.

5.6. Comparison of flotation tailings samples

The comparison of the rougher flotation tailings data serves as a final confirmation of the laboratory and pilot plant results to justify the benefits for adopting the separate milling option for the secondary milling circuit design for processing UG-2 ore. The rougher tailings were screened into five size fractions using standard laboratory techniques and submitted to Mintek for 4E and Cr_2O_3 analyses. The results are tabulated in Table 5.9.

Table 5.9: Comparison of the Rougher flotation tailings samples for the standard and separate milling option for the SGS pilot plant run

Avg. 4E Rougher Recovery (%)	64.7			68.4		
Avg. Cr_2O_3 Rougher Recovery (%)	2.2			1.5		
Avg. Concentrate Mass %	5.8			5.9		
Size-by-Assay	Standard rougher tailings			Optimal design rougher tailings		
Screen Size (mm)	Mass (%)	4E (%)	Cr_2O_3 (%)	Mass (%)	4E (%)	Cr_2O_3 (%)
+106	3.0	1.0	2.4	9.9	2.4	16.2
-106+75	18.5	6.1	12.5	20.0	5.0	26.9
-75+53	23.7	6.2	24.6	20.8	5.2	7.8
-53+38	4.9	1.2	6.2	4.8	1.2	5.5
-38	44.2	20.8	52.1	38.7	17.7	42.0
Total in Rougher Tailings (%)	94.2	35.3	97.8	94.1	31.6	98.5

A comparison of the rougher tailings for both processes showed that about 83% of the Cr_2O_3 minerals were distributed -75µm in the standard tailings; however, with the separate milling of the chromitite-rich and silicates-rich particles, only 55% of the chromite was -75µm. This translates to a 28% reduction in fine chromite. This reduction in fines into the secondary rougher flotation also correlates quite well with the reduction in Cr_2O_3 present in the rougher flotation concentrate (32% reduction). The coarse fractions, i.e. +106µm and -106+75µm fractions contained similar proportions of PGEs (~7%). The separate milling tailings also showed a 10% reduction in the liberation of ultra-fine (-38µm) chromite minerals as compared to the standard tailings.

This data justified the benefit of considering separate milling with energy re-distribution. The spiral concentrator provided an efficient gravity separation of the chromite and silicate particles. These results also confirmed the size-by-assay analyses of the flotation tailings in Section 4.9 as similar conclusions were noted for the milling and flotation results for the MG-type spiral concentrator products. The under-grinding of the chromite minerals is accompanied by a pronounced decrease in the chromite entrainment which contributes to the positive flotation results as discussed in Section 5.6.

The pilot plant test work has therefore provided sufficient scientific evidence to validate the laboratory results and indicated that the use of a cleaner spiral circuit for separation followed by separate milling of the chromite and silicates is a viable option for the secondary milling circuit design. The following section gives an overview of all laboratory and pilot plant results and confirms the choice of the optimal design for this process.

CHAPTER 6: OVERALL DISCUSSION OF OPTIMAL DESIGN

This section summarises the results and discussion on the laboratory and pilot plant test work by showing the final results of all the milling circuit configurations that were investigated. The basis for the design choice was not only the circuit that showed a significant improvement in the PGE recovery and a major reduction Cr_2O_3 entrainment to the secondary rougher concentrate, but also one that would be the most practical application and that could be easily be implemented on current UG-2 concentrators.

The final flowsheets for the various design options are summarised in Figures 6.1 to 6.7. The final mass, 4E and Cr_2O_3 recoveries are also indicated for each secondary milling circuit configuration and whether it was achieved with the low grade or high 4E laboratory sample, from the sampling campaign in 2008 or with the 2010 pilot plant sample.

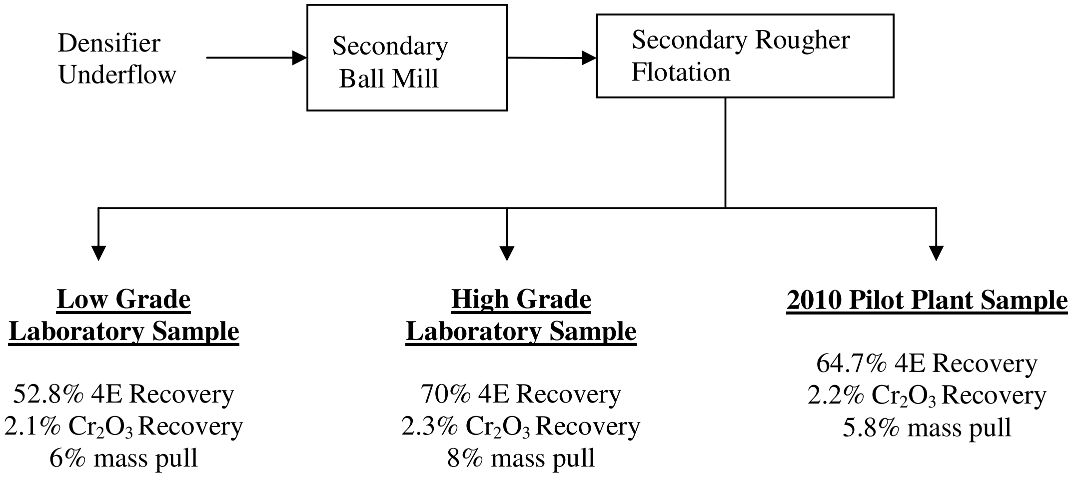


Figure 6.1: Standard MF2 secondary milling and rougher flotation circuit results for a UG-2 concentrator (BASE CASE)

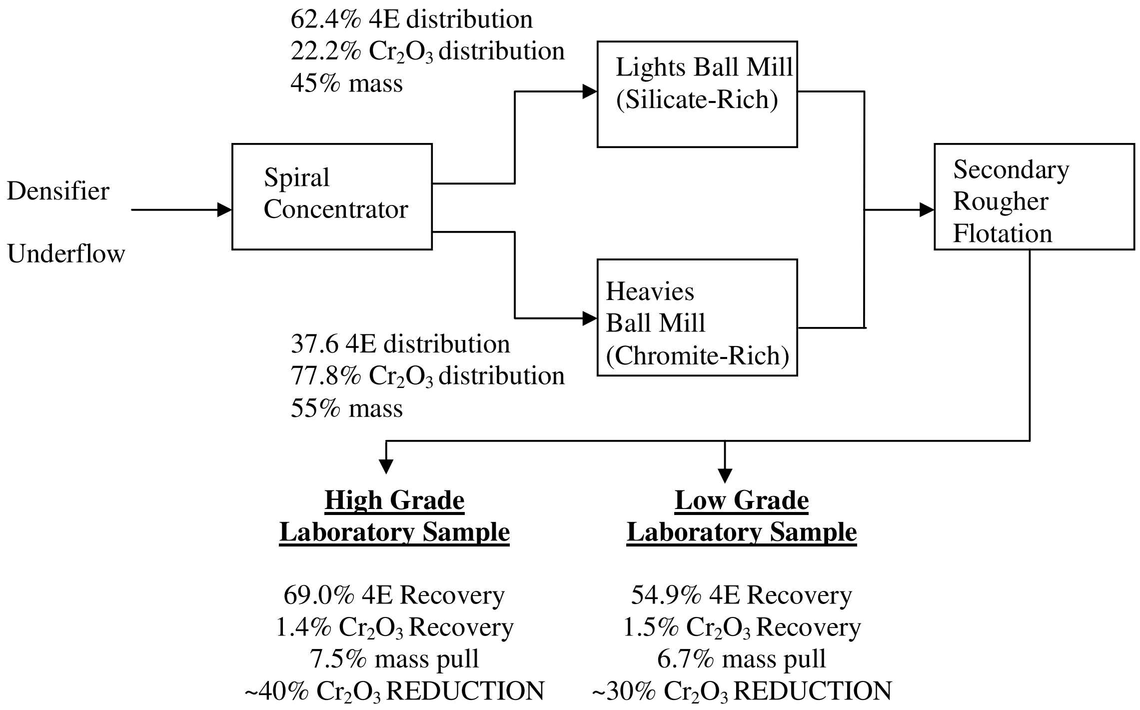


Figure 6.2: Separate ball milling flowsheet of the lights and heavies with 88% of the energy redistributed to the lights mill

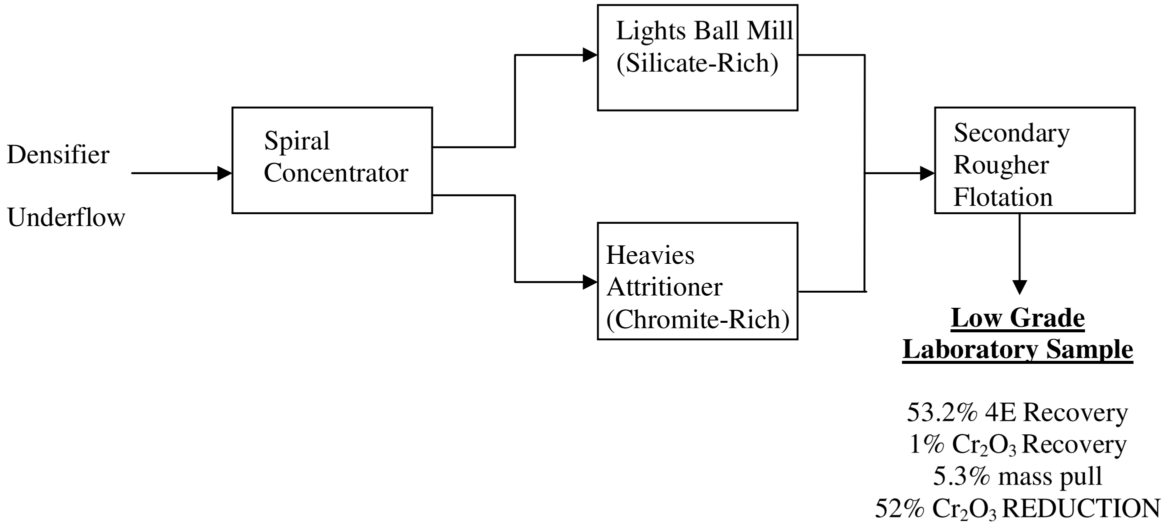


Figure 6.3: Separate ball milling flowsheet for the spiral lights and attritioning of the spiral heavies

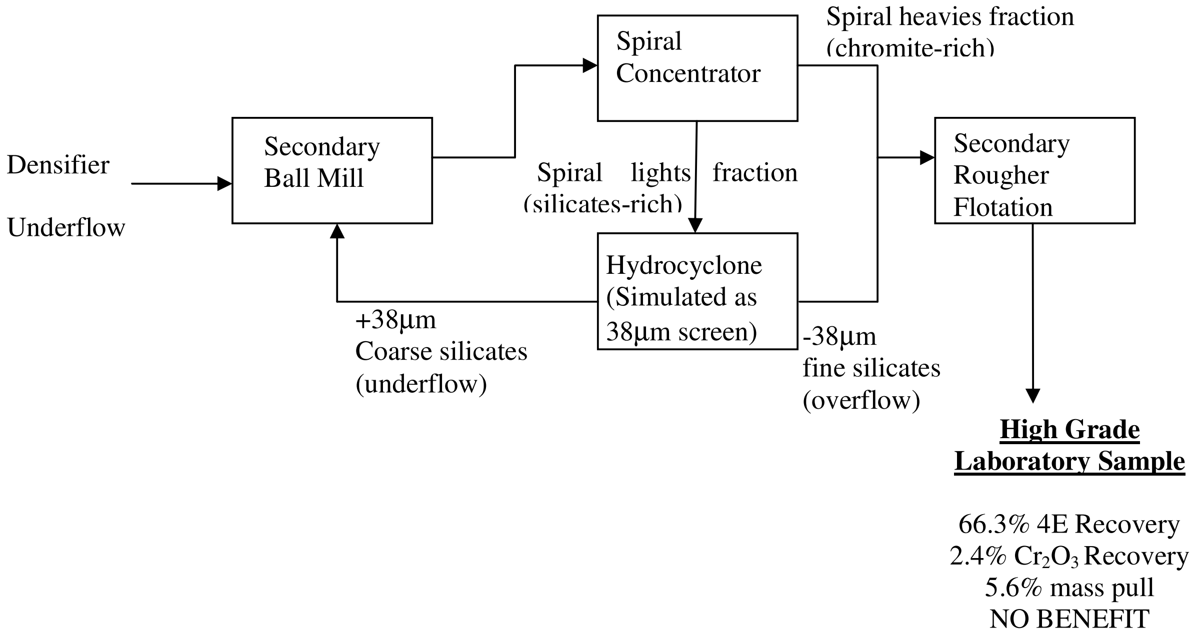


Figure 6.4: Flowsheet incorporating closed circuit ball milling of the spiral lights and open-circuit ball milling of the spiral heavies stream

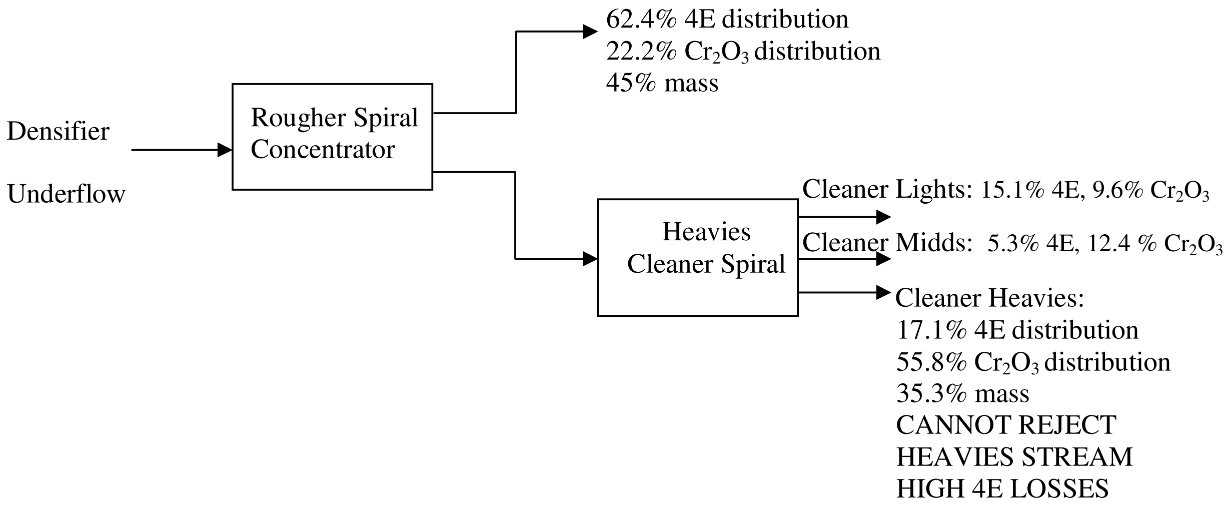


Figure 6.5: The laboratory cleaner MG-type spiral circuit for the potential to reject a "clean" chromite stream

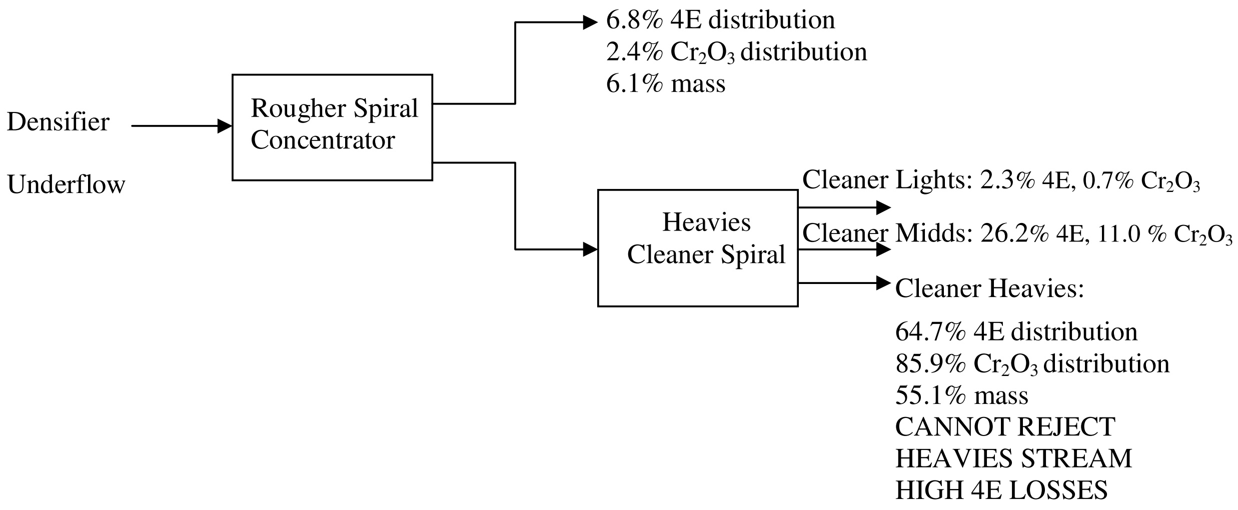


Figure 6.6: SGS pilot plant cleaner spiral circuit for the potential to reject a "clean" chromite stream

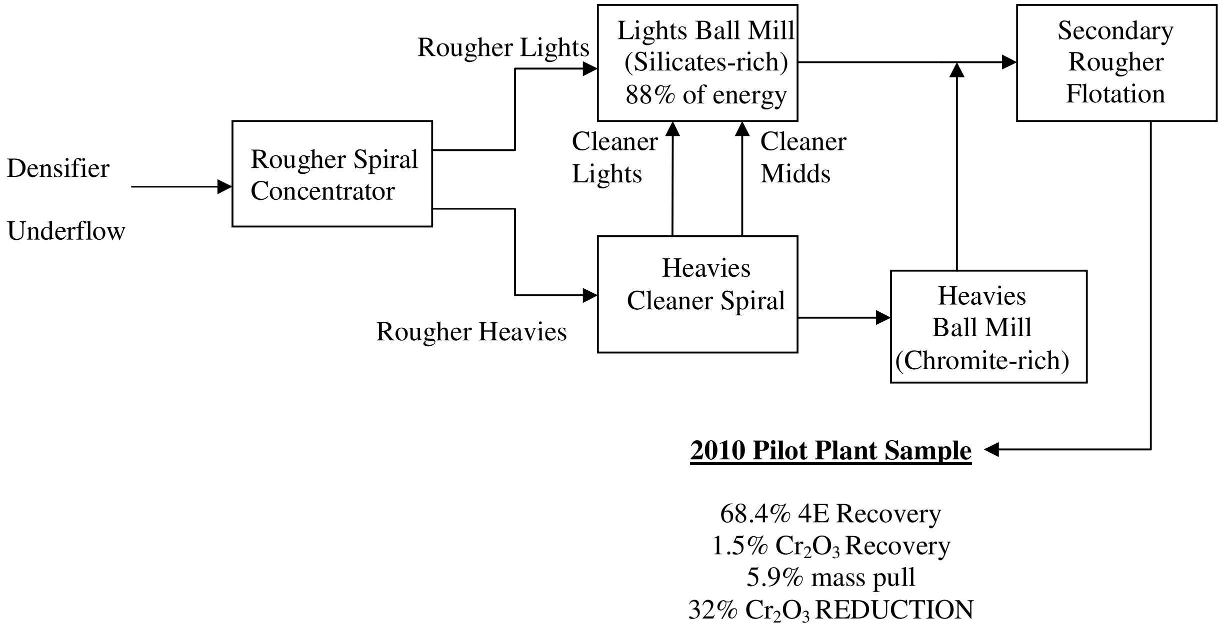


Figure 6.7: SGS pilot plant cleaner spiral flowsheet with separate ball milling and combined flotation

The standard process was benchmarked for the high grade, low grade and pilot plant 4E samples. These recoveries were vital when comparing the viability of each option. The Cr_2O_3 recoveries for the standard process were generally about 2.3% Cr_2O_3 as the Cr_2O_3 head grade was quite constant whereas the 4E recoveries varied considerably from between 50 to 70% overall depending on the 4E grade.

Various milling circuit investigations were considered and two milling circuits were found to produce encouraging results, i.e. the separate milling option with the 88% energy distribution to the lights mill (Figure 6.2) and a similar circuit as in Figure 6.2; however, with attritioning of the spiral heavies fraction as opposed to ball milling (Figure 6.3).

The cleaner spiral circuit (Figures 6.5 and 6.6) did not produce a clean chromite fraction that could be discarded for both the laboratory and pilot plant results. The pilot plant run was found to provide a much more efficient gravity separation of chromite to the heavies stream; however the majority of the PGM also followed the chromite minerals and since significant proportions of 4E were also found to be in the coarse fractions (+75µm), it is clear that this stream does require further regrinding.

The +106µm fraction of the laboratory and pilot plant cleaner spiral heavies stream was found to contain a high amount of PGEs, i.e. 5% and 14% respectively, which confirmed that a clean chromite fraction could not be removed and discarded. Tertiary milling options may be considered for processing the coarse fractions in these circuits (Figures 6.5 and 6.6); however, since the platinum minerals were found in Section 5.2 to follow the chromite minerals, over-grinding of the chromite would always be a concern when adding further stages downstream and the Cr_2O_3 entrainment may increase to above the base case levels. Further tertiary milling stages may not be required when grinds in excess of 90% -75µm can be achieved in the lights ball mill with re-distribution of the milling energy.

Attritioning of the rougher heavies in Figure 6.3 resulted in a 52% reduction in Cr_2O_3 recovery with a marginal increase in the 4E recovery (0.4% 4E). The circuit configuration produced quite favourable overall results; however, since the actual energy utilisation and operating costs for attritioning are undefined, and since it has not been implemented on many concentrators, it was not considered. A detailed study would be required to identify whether any economic or operational benefits would actually accrue from this option.

The separate ball milling design with 88% of the energy re-distributed to the lights (silicate-rich) ball mill and the remaining energy to the heavies ball mill (12%) was found to be the most significantly improved secondary circuit design choice, as shown in Figure 6.2. This circuit would require a capital investment of a second ball mill and the installation of a spirals circuit after the hydrocyclone (densifier); however, since high operating costs or major operating difficulties are not envisaged, it may prove to be an ideal choice for implementation.

Since the % Cr_2O_3 to the final flotation concentrate should not exceed 3%, separate milling with energy re-distribution was an intuitive option to avoid over-grinding. The 12% energy distribution to the chromite ball mill was confirmed by the geology of the BIC (Section 2.1), which indicated that majority of the PGMs were not locked within the chromite but rather interstitial with the chromite grains. A low energy grind or attrition may provide sufficient energy to achieve liberation. Majority of the energy re-distributed to the silicates mill allowed for ultra-fine grinding of the silicates to liberate the platinum minerals locked below 3µm (Section 4.1.3).

The pilot plant test work at SGS achieved an effective gravity separation of the chromite minerals with 86% Cr_2O_3 reporting to the cleaner heavies stream; however, 64.7% PGEs followed the chromite minerals. The silicates were found to be distributed amongst the rougher lights, cleaner lights and cleaner middlings in about 45% of the mass as shown in Figure 6.6. Combining these streams to be treated in the lights ball mill with 88% of the total milling energy with the remaining 12% of milling energy utilised for ball milling of the heavies stream resulted in a 3.7% improvement in the 4E recovery with a 32% reduction in the Cr_2O_3 mineral entrainment as compared to the standard process. The pilot plant results were in agreement with the laboratory results as both indicated that the current milling circuit design may be significantly improved by incorporating spiral concentrator technology.

The statistical reliability of the laboratory and pilot plant data were quantified at various stages of the testwork due to the heterogeneous nature of the feed material and representative sampling. This may have introduced uncertainties on the final flotation recoveries; however, reproducibility was verified for all major findings. A significant number of flotation results were repeated in duplicate or triplicate in certain instances (e.g. Tables B8.14, B9.11, B9.12, B15.5, & B15.6). The repeat analyses of the flotation testwork revealed that the 4E recoveries for the high grade, low grade, and pilot plant sample had variances of below 4%, 5.5% and 1% respectively. The variances for the 4E and Cr_2O_3 head grades for the pilot plant samples were generally below 4%, except for the densifier underflow and rougher lights with an 8.4% and 12.5% variance in the 4E grades respectively. Since this was a scoping study to identify potential milling circuit improvements, a detailed statistical evaluation was not conducted; however, similar quantitative data from the same UG-2 ore, i.e. 2008 high grade sample, 2008 low grade sample and the 2010 pilot plant sample, support the veracity of the data.

A preliminary costing exercise was undertaken for the pilot plant results to determine what value the proposed circuit could add. The capital investment for the spiral concentrator circuit was sourced from an external spiral vendor and the cost of an additional secondary ball mill for the chromite-rich stream was sourced from an external mill vendor based on an estimation of the power draw. These quotations are shown in Appendix B17. The total capital investment for these improvements to the UG-2 concentrator amounted to approximately R72 million. These estimates were based on a dry solids feed tonnage of 180 t/h. The indirect costs were estimated from the data given in Peters and Timmerhaus (1991). The smelting and refining costs were assumed to be 10% of the platinum commodity price. Analyses of the 4E grades revealed that platinum minerals accounted for approximately 75% of the PGEs. For an additional 3.7% 4E recovery, an additional revenue of approximately R50 000 per day could result based upon the Pt mineral recovery only. The other precious metals, i.e. Pd and Rh were neglected in the calculation and would therefore further increase the overall revenue. The preliminary costing revealed that the minimum payback period based on the current prices would be approximately 4 years. This was quite encouraging and suggested that the improved secondary milling circuit design could be considered as a viable option.

The spiral concentrator provides an effective gravity separation of the high density chromite and the low density silicates. The laboratory ball mill simulates closed circuit milling and therefore inefficiencies such as slurry pooling and density effects do not affect the process as in open-circuit grinding on the plant. Separate milling provides the option of closed- circuiting of the silicates which would improve the platinum mineral liberation. Separate milling has been successfully implemented for over five years at Anglo Platinum; however, most other major platinum producers still favour the Mintek MF2 circuit design. The variations in UG-2 ore have resulted in the processing of chromite-rich feed material in concentrators thus final target recoveries (>85% 4E and <3% Cr_2O_3) can be difficult to achieve without optimising the current process flowsheet.

The implementation of a secondary milling circuit design which incorporates a gravity separation device, i.e. a spiral concentrator circuit would be an inexpensive and effective way to achieve good density separation, and in addition this device can be easily incorporated into most plants. The results of the pilot plant spiral circuit run at SGS South Africa have shown that these units are effective gravity separation devices for treating chromite-rich UG-2 platinum ores. A marginally higher platinum recovery (~4% PGE) was achieved with over a 30% reduction in the secondary rougher Cr_2O_3 recovery.

CHAPTER 7: CONCLUSIONS

The efficiency of secondary grinding of UG-2 platinum ore is currently limited by the use of open-circuit grinding in many plants, which is inherently inefficient, due to short-circuiting of coarse particles and possible segregation by gravity in the mill.

The main research question investigated in this project was to determine whether it was possible to improve the design of the secondary milling circuit for treating UG-2 platinum ores to improve the platinum recovery and to reduce the chromite entrainment to the secondary flotation circuit.

A plant audit was conducted at a UG-2 concentrator at Lonmin Platinum to benchmark the process for one sample. Size-by-assay analyses of all major streams of the secondary milling circuit indicated that over 90 percent of the PGE in the feed (densifier underflow) were in the coarse fractions (+53µm) and would therefore require further re-grinding to liberate these minerals. The secondary ball mill was also found to be operating inefficiently, as the ball mill discharge contained over 28% of PGEs in the +75µm size range. Mineralogical evaluations of the SRT plant sample showed losses of about 37% of the PGEs in that stream to the coarse fractions (+75µm).

Attempts to measure segregation within the mill, using batch laboratory tests, (which do not simulate the short-circuiting effect), were inconclusive.

The use of gravity separation of the chromite and silicate particles, prior to secondary grinding, was investigated in various forms. The initial laboratory spiral tests indicated over 60% separation of PGEs in a 45% mass fraction of lights. However, this also contained over 20% of chromite minerals. Separate grinding after gravity separation yielded promising results, particularly a 40% reduction in the chromite content of the rougher concentrate with a similar platinum (4E) recovery of approximately 70% 4E. This was achieved with a re-distribution of 88% of the total milling energy input to the silicates ball mill, with the remaining 12% of the energy distributed to the chromite ball mill. Repeat tests on a low grade 4E laboratory sample confirmed these results with a 2% increase in 4E recovery and approximately 30% reduction in chromite entrainment to the secondary rougher concentrate.

Laboratory attritioning of the spiral heavies stream followed by combined flotation with a ball milled rougher spiral lights stream resulted a 52% reduction in the Cr_2O_3 recovery with a marginal increase in the PGE recovery (0.4%) as compared to the standard process.

Laboratory cleaner spiral tests revealed that a chromite-rich stream could not be discarded as a large proportion of the PGEs (16% overall) would be discarded with about 55% of the chromite minerals. Pilot plant cleaner spiral test work with two VHG spiral concentrators' confirmed that chromite rejection was not viable, despite the improved separation of chromite and silicates. The cleaner heavies stream contained over 86% Cr_2O_3; however, about 65% of the PGEs followed the chromite particles. The +106μm cleaner heavies fraction contained about 40% of the overall Cr_2O_3 from the process; however, 14% of the overall PGEs followed the Cr_2O_3 which revealed that PGMs are associated to some degree with the chromite minerals in the UG-2 ore.

The pilot plant spiral circuit run at SGS confirmed the results of the laboratory spiral circuit test work campaign at UKZN. The use of gravity separation, i.e. a spiral concentrator circuit followed by separate milling of the silicates and chromite particles with energy re-distribution and combined flotation was found to be the most significantly improved secondary milling circuit design. The resulting 4E recoveries were found to be about 4% higher than the standard process and the Cr_2O_3 entrainment was reduced by 32% in the secondary rougher concentrate.

Repeat analyses of the head grades for the high grade sample revealed that the variance was below 0.5%. Selected flotation results were repeated for the high grade sample and the 4E and Cr_2O_3 recoveries reported variances of below 4% and 7% respectively. The flotation results for the standard and improved milling circuit for the low grade sample reported variances in the 4E recoveries of below 5.5%. The variances for the repeated 4E grades of the pilot plant samples were below 4% for the majority of the streams, with only two streams, i.e. densifier underflow and rougher lights having a variance of about 8% and 12% respectively. The pilot plant results showed good reproducibility since the variances for the 4E and Cr_2O_3 recoveries were quite low (<1%).

The costs and financial benefits of installing spirals and a new mill for separate secondary grinding of the chromite-rich fraction have been estimated, for a concentrator processing 180 tonnes per hour. The capital costs for the gravity separation circuit and chromite-rich ball were estimated by external vendors. The total capital investment for the implementation of the improved milling circuit design was approximately R72 million. An additional revenue of approximately R50 000 per day could result based upon the Pt mineral recovery only, with a minimum payback period of 4 years. This was encouraging and indicated the benefit of this improved secondary milling circuit design.

The batch silicates laboratory ball mill was able to achieve much finer grinds in excess of 90% -75µm which indicated that the move towards finer grinding technology may be avoided by considering this improved secondary ball milling circuit design.

The conventional ball mill was able to achieve an improvement in the liberation of platinum bearing minerals through redistribution of grinding energy to the lights fraction.

This improved secondary milling circuit design based on one UG-2 sample may provide a significant contribution to improving the overall recovery of PGEs on an industrial scale.

CHAPTER 8: RECOMMENDATIONS

After concluding this study on the design of a secondary UG-2 concentrator milling circuit for a particular UG-2 sample, a few notable recommendations are proposed for future work to confirm the implementation of this circuit on an industrial scale.

A plant audit campaign at various UG-2 concentrators is recommended to benchmark the process for several UG-2 samples. The purpose of this study would be to collect a representative 200kg densifier underflow sample from the secondary milling circuits of existing concentrators in different areas of the UG-2 reserve for laboratory spiral and flotation testwork. These ores should be of varied Cr_2O_3 and PGE content to obtain a detailed analysis against which any claimed circuit improvements could be compared.

Cleaner flotation testwork is recommended based on a typical reagent suite for the cleaner circuit with depressant addition. The densifier overflow sample should be combined in proportion with the milled spiral products to reflect a typical secondary flotation feed sample. This will aid in quantifying the 4E and Cr_2O_3 recovery benefit for the entire secondary circuit based on the improved milling circuit design. Laboratory locked cycle tests are recommended for this testwork campaign. The effect of separate flotation of the chromite and silicate streams can also be investigated as part of this laboratory flotation optimisation testwork.

It would be worthwhile to commission different spiral concentrators on the existing ore to confirm the appropriateness of the VHG spiral concentrator prior to pilot plant testwork. It is envisaged that a continuous five day pilot plant test work campaign incorporate gravity separation with a spiral concentrator circuit, with separate milling and flotation to determine the benefit of the improved design configuration. This campaign can be conducted at an external spiral vendor or at a UG-2 concentrator. This circuit can be run with the silicates ball mill in closed-circuit using a hydrocyclone and with open-circuit ball milling for the chromite minerals. Closed-circuit grinding is significantly more efficient for reducing the distribution of sizes, as coarse particles which short-circuit due to mixing, are re-cycled.

The placement of the cleaner middlings stream with the silicates-rich or chromite-rich streams can be investigated for both closed and open-circuit milling configurations as batch spiral runs do not quantify the effect of proper placement of middlings in the process.

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APPENDICES

APPENDIX A: LABORATORY AND PILOT PLANT EQUIPMENT AND EXPERIMENTAL PROCEDURES

APPENDIX A1: EXPERIMENTAL PROCEDURE FOR PLANT AUDIT AND SAMPLING CAMPAIGN

The bulk densifier underflow sample was collected in 15 buckets using a large sample cutter to produce approximately 250kg of sample for laboratory milling and flotation test work. The sample cutter was immersed into the flowing stream at one go to ensure that a representative cut of the sample stream was obtained. All samples were stored wet to prevent oxidation of the mineral surfaces which inhibits flotation.

A1.1. Sample preparation for the plant audit circuit samples

The sample preparation technique for the plant audit samples is shown below:

- Each bucket was weighed without the lid to confirm the mass recorded at the plant
- The top water was decanted into a separate bucket and any fines noted in the water were filtered and the solids were returned to the main sample while the remaining water was discarded.
- The main sample remaining after decanting was then air dried for three days and the dry mass was recorded. This was used to determine the % solids of the stream.
- The samples were cone and quartered and riffled to produce representative sub-samples of 500g for screening
- The sub-samples were then wet screened with the smallest screen size, i.e. 38µm to remove the fines and the +38µm and -38µm fractions were dried in the oven at 105^{o}C.
- The dried +38µm samples were then screened using 106, 75, 53, 38µm sieves and the -38µm fraction was adjusted to account for the additional -38µm fraction that was removed during wet screening.
- The mass fraction in each size interval was recorded and a sub-sample of each size fraction was bagged for analysis at Mintek.

The samples were then sent to Mintek to undergo a 4E (Pt, Pd, Rh, Au) analysis for PGEs' and ICP analysis for Cr_2O_3.

A1.2. Sample preparation for the densifier underflow bulk sample

The bulk sample was stored in 15 buckets and since each bucket contained three sample cuts taken periodically from the densifier underflow, it was decided that each bucket would be representative of the process stream and therefore could be used independently for each set of laboratory tests, instead of blending the entire sample together which was in excess of 250kg.

Each bucket was therefore prepared as mentioned for the plant audit sample; however, since the water content was very minimal drying was not required. Each bucket contained approximately 30kg of material. This material was shovelled with a spade several times on a canvas to ensure that it was well mixed. Thereafter a cone and quartering technique was used to prepare smaller batches of 2kg which were then riffled further to produce the required sample mass for milling and flotation tests. The remaining samples were stored in buckets and sealed. Since milling was conducted for each sample oxidation was not a concern as the milling activated the mineral surfaces for flotation.

APPENDIX A2. Experimental Procedure for the Mineralogy at MINTEK

The SRT sample was split into two representative sub-samples. One sub-sample was pulverised for analysis by X-ray diffraction (XRD), for the identification of the minerals present (crystalline mineral detection limit of > 3% by mass). Polished sections were prepared from the second sub-sample for analysis by automated SEM. A target of about 50 grains on the SRT was sought for the PGM analysis.

The SEM analysis used for the MINTEK mineralogical evaluations was based on a bright phase search, which seeks out PGM based on their higher backscatter intensity under the electron beam, compared with gangue minerals. The bright PGMs were identified and described in terms of the species present, their grain size distribution, liberation characteristics and mineral associations (i.e. what other minerals are associated with the identified species in particles).

The detection limit for the bulk modal analysis was approximately 0.5 mass%. Minerals present in amounts less than this were grouped together, owing to the high uncertainties associated with their proportions in the sample.

The determination of platinum liberation is a particularly challenging task; in view of the very low concentration of the platinum minerals, only 61 grains of the PGM were analysed.

APPENDIX A3. Experimental Procedure for the MINTEK Laboratory Heavy Liquid Separation (HLS) Tests

Two representative 1.5kg sub-samples of the densifier underflow UG-2 sample were sent to Mintek for HLS testwork. The samples were classified into two size classes, i.e. -212+150µm and -150+106µm. A representative sub-sample of each fraction was taken for head grade analysis.

A solvent extraction flask was used for the testwork. A solution of clean Tetra-Bromo-Ethane (TBE) with a density of 2960 kg/m^3 was poured into the flask. The sample was then introduced into the media for separation and approximately 500g of material was treated at a time.

The material was given time to split (separate) into the respective fractions i.e. floats and sinks. Once there was no noticeable particle movement taking place, a collection beaker was used to capture the resulting sinks separately from the floats products.

APPENDIX A4. Equipment utilised for laboratory test work

Figure A4.1: 200mm diameter steel cylindrical mill

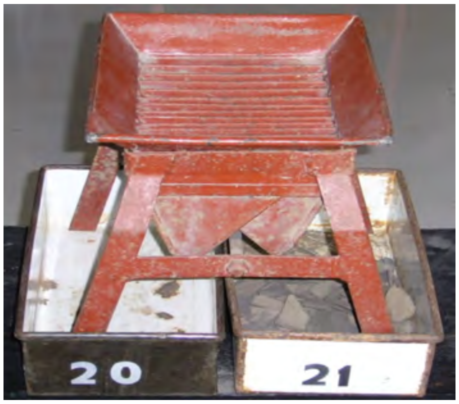


Figure A4.2: Riffle Box used in experiments

Figure A4.3: 300mm diameter variable speed mill with large sieve used to separate the slurry from the steel balls

Figure A4.4: Pressure Filter Apparatus

A 38 µm sieve is shown in Figure A4.5 on a vibrating shaker.

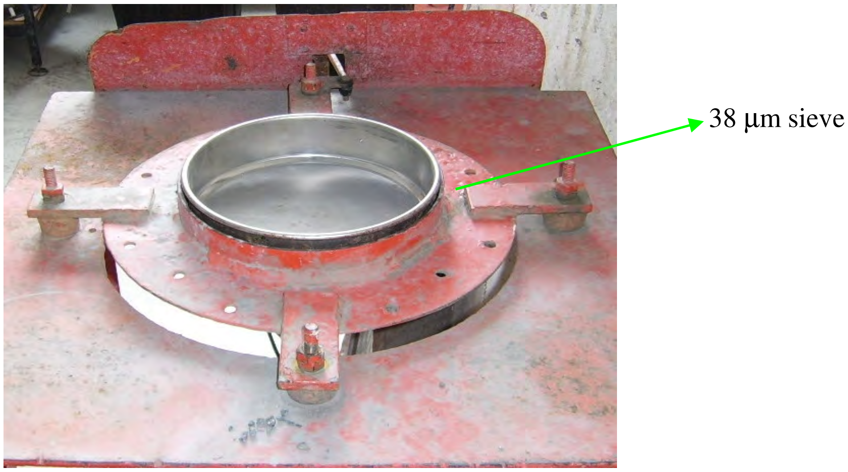


Figure A4.5: Vibrating Shaker with sieve

Figure A4.6: Sieve shaker

Flexible plastic piping was used to transfer the feed to the top of the spiral as is shown in Figure A4.7 below. The flowrate of the feed was determined at this point. The feed tank was at the bottom of the spiral.

Figure A4.7: Picture showing the spiral feed point

As the slurry flowed down the spiral it split via the splitter opening (Figure A4.8). The heavy density fraction flowed through the splitter opening and into the inner spiral. The light density fraction flowed along the outer rim of the spiral. After the splitter there was a cover from the outer rim to the inner rim to prevent the overflow of slurry.

Figure A4.8: Picture showing the splitter and splitter opening (6cm)

Figure A4.9 shows where the heavy, middle and light density fractions exit the spiral. The exit streams flow onto the sample point cover and then back into the feed tank. A distribution plate on the sample points prevented the slurry from splashing out of the tank when the different fractions hit the sample point cover. When the sample point cover is removed, the three density fractions flow into three separate ports for each of the splits.

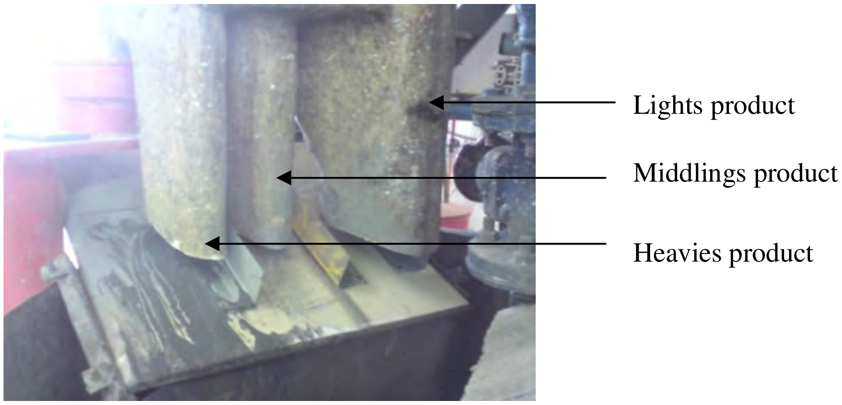


Figure A4.9: Picture showing the heavy, middle and light density fractions exit onto sample point cover.

Figure A4.10: Picture showing the sample collection points and the slurry pump.

The density fractions then flowed down the plastic pipes and into the sample buckets for collection as shown in Figure A4.10. The slurry pump can also be seen in this figure.

APPENDIX A5. Experimental Procedure for the effect of varying proportions of dense minerals

The experimental procedure is detailed below:

- Feed size distributions for the un-milled ilmenite and limestone were conducted.
- The masses of the mill charge were determined based on the volumetric loadings i.e. grinding media (27mm steel balls), limestone, and ilmenite.
- The mill was loaded with the total charge and the mill speed was set at 70 rpm and operated for a standard time of 20 minutes
- After milling the product and grinding media were removed by using wash water.
- The product was then pressure filtered to remove the excess water.
- The filter cake sampled by a cross-cut method i.e. to ensure that a representative slice on either side of the filter cake was removed to form the sub-sample for analysis.
- The sub-sample was wet screened at 38µm.
- The +38µm and filtered -38µm samples were dried overnight in an oven.
- After drying, the samples were screened using sieves on a vibrating sieve shaker.
- The mill products were screened using root two series sieves for each run.
- A representative sub-sample (10g) from each size fraction from the mixed tests were dissolved in excess HCl (20ml) to determine the proportion of limestone and ilmenite in each size fraction.
- About 85% of the limestone was dissolved by this acid digestion technique due to the impurities present in the limestone.
- A particle size distribution for each mineral was determined by plotting the log of cumulative mass percent passing versus the log of the aperture sizes.

These profiles were compared to determine the most effective ratio of limestone to ilmenite for the mill feed mixture.

APPENDIX A6. Experimental Procedure for the MG-type rougher spiral concentrator

The detailed laboratory procedure is shown below:

- The spiral concentrator was configured such that it operated in batch mode with the product streams (lights, middlings, heavies) emptying back into the base tank with a slurry pump to feed the unit at the top.
- The spiral base tank was first filled with 80 litres of water and then the slurry pump was switched on.
- The flowrate was set by adjusting main flow valve on the pump with a fixed recycle rate maintained on the base tank recycle valve. The flowrate was measured using a beaker and stopwatch.
- Thereafter solids were filled gradually into the base tank and allowed to mix with the water until steady state was achieved. The amount of solids was initially 17kg (70% by mass of ilmenite and 30% limestone); however, this was varied depending on the variation in parameters when optimising the unit.
- The system was allowed to run for ten minutes before a density can test was performed to determine the percent solids of the slurry.
    - In the density can test a 1 litre beaker was filled with slurry at the feed point. The beaker had a hole at the 1 litre mark to ensure an exact volume.
    - The beaker was wiped and weighed to determine the percentage solids content of the feed knowing the density of the ore.
- The percent solids was repeatedly checked and either the water or solids adjusted until it was at setpoint and reached a steady state operation.
- The sample board covers were removed to expose the three sample hoses to the sample buckets which were used to collect sufficient sample.
- The slurry was captured in buckets at the three sample points and the time taken to was recorded.
- Repeat samples were taken in another three buckets to verify the results.
- The sample board covers were then replaced and the water valve was turned on while bleeding the unit of excess sample which was collected, dewatered and discarded as a fresh sample was prepared for each run.
- The six samples were pressure filtered, dried and riffled to produce a sub-sample for acid digestion tests with HCL to determine the proportion of limestone and ilmenite in each fraction, i.e. lights, midds, and heavies products of spiral

- A solution of 10ml of 32% HCL and 10ml of water was used for each test. Each sub-sample obtained was then added to this solution. Once the reaction was complete the solution was filtered and dried to calculate the mass of limestone that dissolved, accounting for the 85% solubility for these tests.

APPENDIX A7. Acid digestion technique developed for ICP analyses

The sample preparation and acid digestion technique is shown below:

- A 5 gram sub-sample was prepared from the UG-2 spiral run product samples
- 10ml of hydrochloric acid (HCl) and 10ml of nitric acid (HNO_3) were diluted with 10ml of distilled water and added to sample which was placed on a heater and a magnetic stirrer heated to 50°C for 5 minutes.
- The solution was allowed to cool before filtering and then diluted to 500ml
- A 10 ml vial of this solution was filtered with a micropore filter and then sent for ICP analysis
- 5ppm, 10ppm, 20ppm, 50ppm, 100ppm, and 200ppm Cu, Ni and Cr standards were also prepared and sent with the samples to calibrate the unit before analysis. The required apparatus are shown in Figure A7.1

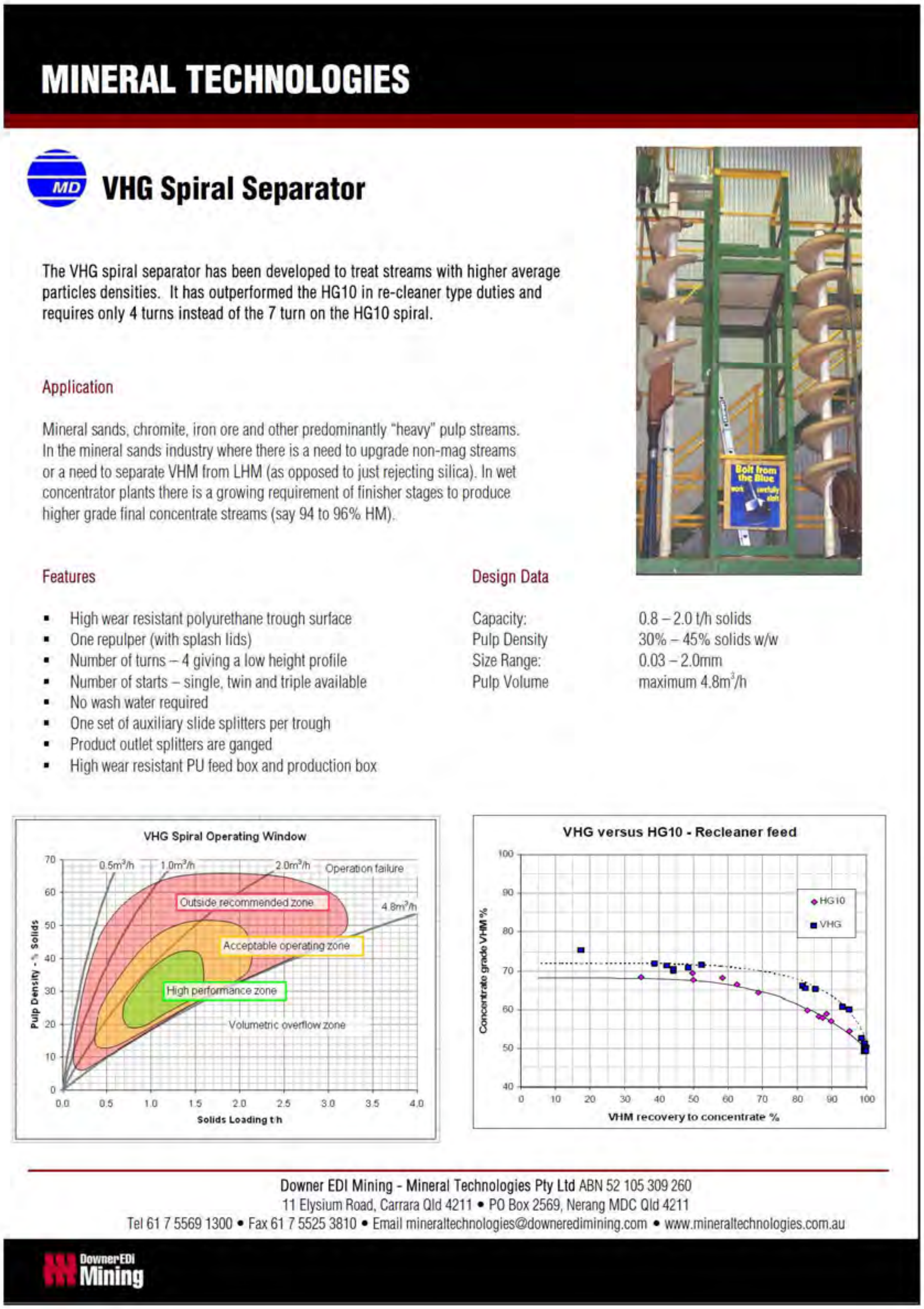
Figure A7.1: Picture of acid digestion apparatus for ICP analyses to determine the metal concentrations

APPENDIX A8. Experimental procedure for UG-2 rougher spiral runs for separate milling tests

- The spiral was initially started up with fresh water (96 litres) and the 17kg of UG-2 ore was gradually added to the feed tank.
- The flowrate of the spiral was then set to 0.7 litres/s by adjusting slurry pump valve.
- The system was allowed to run for ten minutes before a density can test was performed. Density can test-
    - A one litre beaker was filled with slurry at the feed point to the spiral.
    - The beaker was weighed to determine the density of the slurry. This was used to set the feed % solids to 15%.
- The density can test was repeated to ensure that steady state was achieved.
- The sample board covers were removed to expose the sample points to the exit of the spiral.
- The middlings cut was closed so as to collect a lights and heavies fraction only.
- The slurry was captured in buckets at the two sample points and time taken was recorded.
- The sample board covers were then replaced and the unit was then shut down but bleeding out any excess sample using fresh water.
- The samples were then filtered and dried and sub-sampled for milling test work.

A similar method of operation was used for the cleaner spiral run at a flowrate of 0.5 litres/s and at 15% solids. A cleaner lights, middlings and heavies product were collected.

APPENDIX A9. VHG Spiral Concentrator Specifications



MINERAL TECHNOLOGIES

VHG Spiral Separator

The VHG spiral separator has been developed to treat streams with higher average particles densities. It has outperformed the HG10 in re-cleaner type duties and requires only 4 turns instead of the 7 turn on the HG10 spiral.

Application

Mineral sands, chromite, iron ore and other predominantly "heavy" pulp streams. In the mineral sands industry where there is a need to upgrade non-mag streams or a need to separate VHM from LHM (as opposed to just rejecting silica). In wet concentrator plants there is a growing requirement of finisher stages to produce higher grade final concentrate streams (say 94 to 96% HM).

Features

- High wear resistant polyurethane trough surface
- One repulper (with splash lids)
- Number of turns – 4 giving a low height profile
- Number of starts – single, twin and triple available
- No wash water required
- One set of auxiliary slide splitters per trough
- Product outlet splitters are ganged
- High wear resistant PU feed box and production box

Design Data

Capacity:	0.8 – 2.0 t/h solids
Pulp Density	30% – 45% solids w/w
Size Range:	0.03 – 2.0mm
Pulp Volume	maximum 4.8m^3/h

Downer EDI Mining - Mineral Technologies Pty Ltd ABN 52 105 309 260
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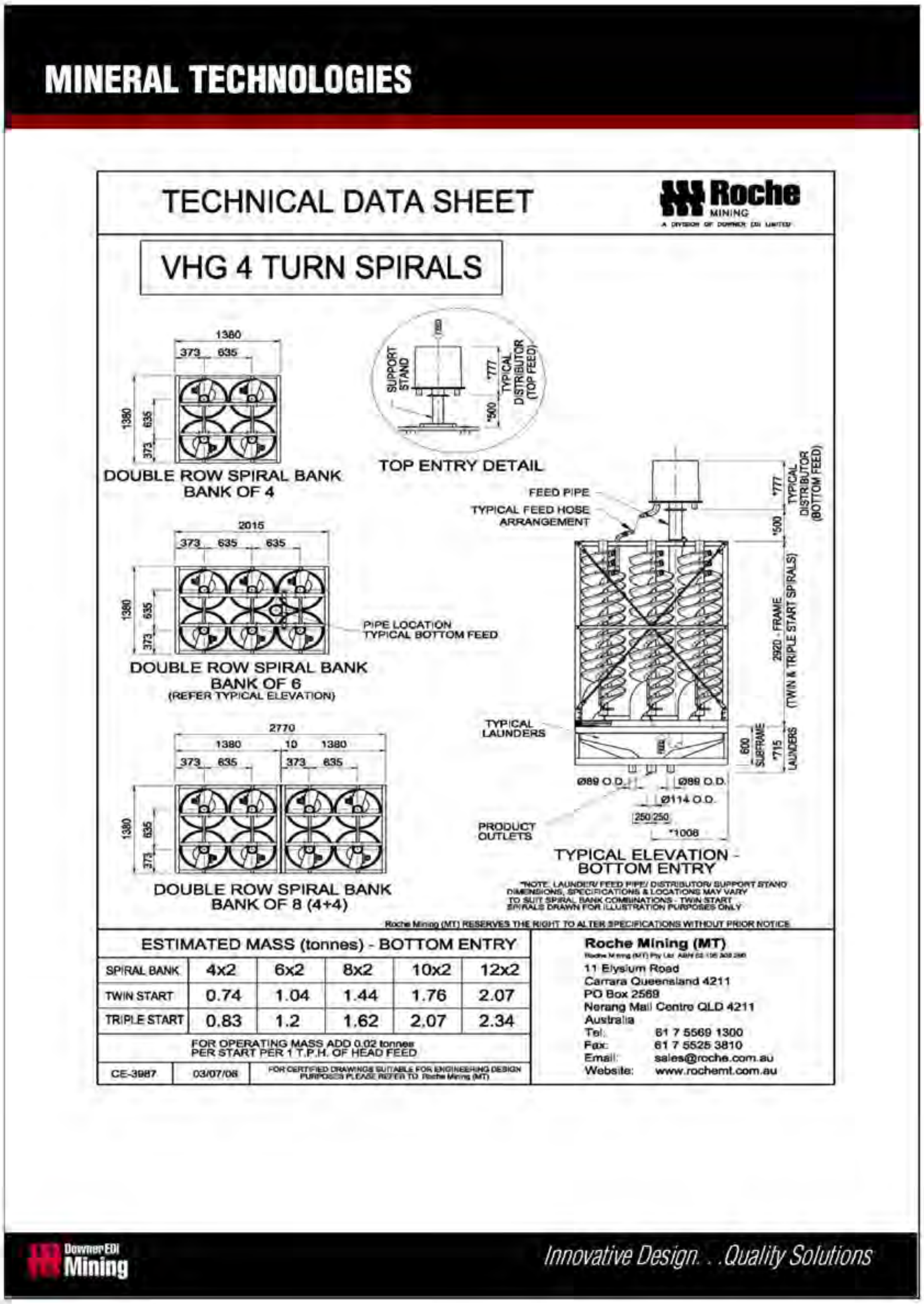
Downer EDi Mining



MINERAL TECHNOLOGIES

TECHNICAL DATA SHEET

Roche MINING
A DIVISION OF DOWNER EDI LIMITED

VHG 4 TURN SPIRALS

*NOTE: LAUNDER/ FEED PIPE/ DISTRIBUTOR/ SUPPORT STAND DIMENSIONS, SPECIFICATIONS & LOCATIONS MAY VARY TO SUIT SPIRAL BANK COMBINATIONS - TWIN START SPIRALS DRAWN FOR ILLUSTRATION PURPOSES ONLY

Roche Mining (MT) RESERVES THE RIGHT TO ALTER SPECIFICATIONS WITHOUT PRIOR NOTICE

ESTIMATED MASS (tonnes) - BOTTOM ENTRY

SPIRAL BANK	4x2	6x2	8x2	10x2	12x2
TWIN START	0.74	1.04	1.44	1.76	2.07
TRIPLE START	0.83	1.2	1.62	2.07	2.34

FOR OPERATING MASS ADD 0.02 tonnes PER START PER 1 T.P.H. OF HEAD FEED

CE-3987 | 03/07/08 | FOR CERTIFIED DRAWINGS SUITABLE FOR ENGINEERING DESIGN PURPOSES PLEASE REFER TO Roche Mining (MT)

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APPENDIX B: ADDITIONAL TABLES, FIGURES & RAW DATA

Appendix B1: Milling curve for densifier underflow sample

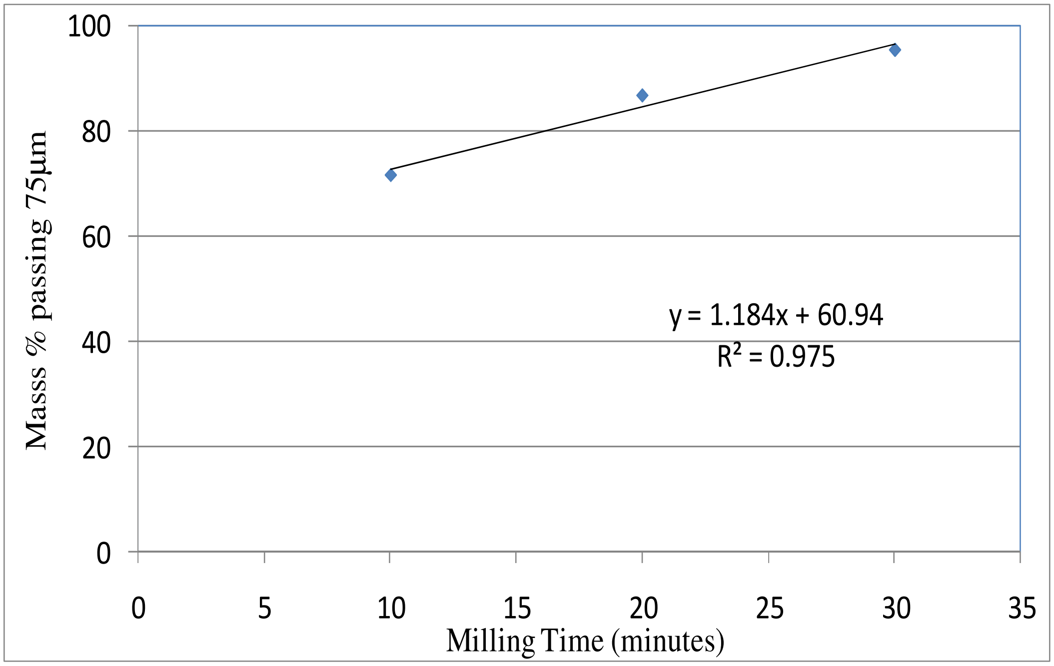


Figure B1.1: Milling Curve for UG-2 densifier underflow laboratory sample

Appendix B2: Tables of plant audit results

Table B2.1: Laboratory size by assay analyses of PRT

Primary Rougher Tails (PRT)		Distribution (%)	
Screen Size (µm)	Mass (%)	4E	Cr_2O_3
+106	34.3	31.6	36.1
-106+75	21.1	16.0	22.9
-75+53	11.1	7.8	11.2
-53+38	3.9	2.8	3.5
-38	29.6	41.8	26.2
Total	100.0	100.0	100.0

Table B2.2: Laboratory size by assay analyses of densifier overflow

Densifier Overflow		Distribution (%)	
Screen Size (µm)	Mass (%)	4E	Cr_2O_3
+106	4.4	4.1	1.0
-106+75	14.9	8.6	6.8
-75+53	11.1	4.9	7.9
-53+38	5.2	1.8	5.2
-38	64.4	80.5	79.0
Total	100.0	100.0	100.0

Table B2.3: Laboratory size by assay analyses of mill discharge

Mill Discharge		Distribution (%)	
Screen Size (µm)	Mass (%)	4E	Cr_2O_3
+106	11.4	10.0	11.2
-106+75	26.7	17.7	28.5
-75+53	11.5	7.1	12.2
-53+38	4.9	3.4	5.0
-38	45.5	61.7	43.1
Total	100.0	100.0	100.0

Table B2.4: Laboratory size by assay analyses of SFF

Secondary Float Feed (SFF)		Distribution (%)	
Screen Size (µm)	Mass (%)	4E	Cr_2O_3
+106	9.6	5.3	9.1
-106+75	16.0	8.8	15.5
-75+53	9.4	5.1	9.1
-53+38	4.1	2.1	4.0
-38	60.9	78.6	62.3
Total	100.0	100.0	100.0

Table B2.5: Laboratory size by assay analyses of SRT

Secondary Rougher Tails (SRT)		Distribution (%)	
Screen Size (µm)	Mass (%)	4E	Cr
+106	8.9	12.2	Cr_2O_3
-106+75	23.8	24.9	23.5
-75+53	11.3	11.6	11.1
-53+38	5.5	5.7	5.4
-38	50.5	45.6	51.2
Total	100.0	100.0	91.3

Table B2.6: Laboratory size by assay analyses of densifier underflow

Densifier Underflow (Mill Feed)		Distribution (%)	
Screen Size (µm)	Mass (%)	4E	Cr_2O_3
+106	49.4	28.7	48.5
-106+75	29.8	39.4	29.1
-75+53	16.7	23.5	17.6
-53+38	3.0	6.1	3.5
-38	1.1	2.3	1.3
Total	100.0	100.0	100.0

Table B2.7: Laboratory size by assay analyses of densifier underflow (repeat)

Densifier Underflow (Mill Feed) - Repeat		Distribution (%)	
Screen Size (µm)	Mass (%)	4E	Cr_2O_3
+106	49.4	30.8	46.0
-106+75	29.8	37.2	31.0
-75+53	16.7	23.6	18.3
-53+38	3.0	6.2	3.4
-38	1.1	2.2	1.4
Total	100.0	100.0	100.0

Appendix B3: Mintek sizing results on plant audit "A" and "B" samples

Table B3.1: Cumulative percentage passing versus screen size for densifier underflow (U/F) and densifier overflow (O/F)

Size (mm)	Densifier U/F (A)	Densifier U/F (B)	Densifier O/F (A)	Densifier O/F (B)
1.700	100.00	99.89	100.00	100.00
1.180	100.00	99.79	100.00	100.00
0.850	100.00	99.64	100.00	100.00
0.600	100.00	99.04	100.00	100.00
0.425	99.00	97.85	100.00	100.00
0.300	95.31	94.31	100.00	100.00
0.212	85.77	84.71	100.00	100.00
0.150	60.54	58.78	99.86	99.66
0.106	37.94	36.22	98.07	96.29
0.075	20.13	17.68	90.77	83.97
0.053	12.68	10.04	81.95	76.00
0.038	8.78	5.98	71.02	68.30

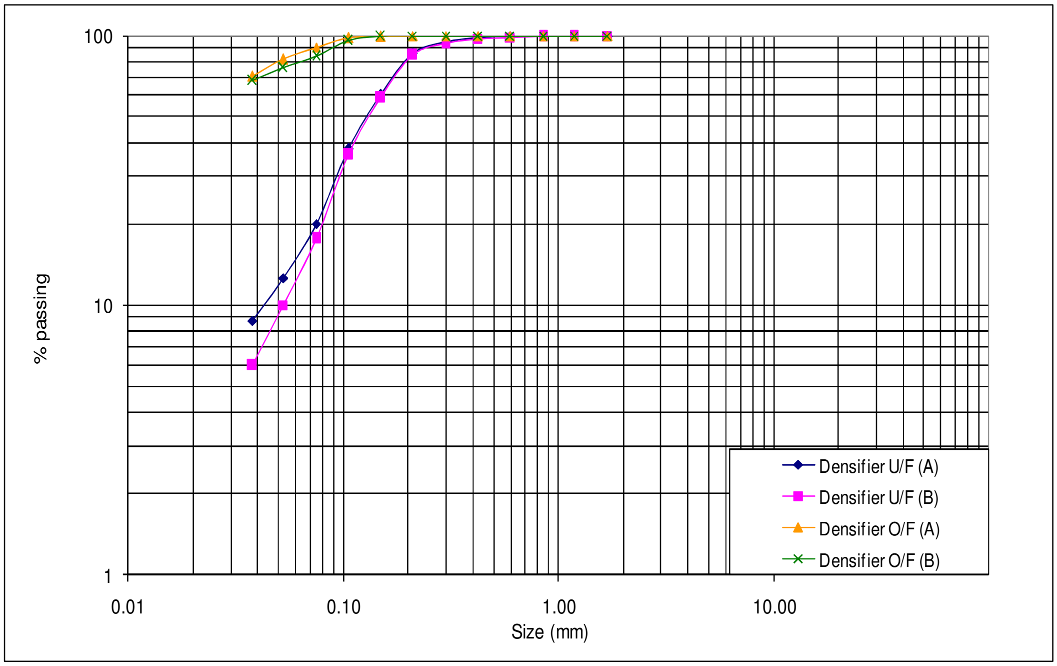


Figure B3.1: Cumulative % passing versus screen size for densifier underflow and overflow samples

Appendix B4: Mintek HLS results

Table B4.1: 4E results for Sample 1

-212+150µm				Recovery [%]			
Density	RD to sinks	Mass [g]	Mass [%]	Au	Pd	Pt	Tot PGM+Au
+2.96*	3.80	1094.4	82.42	56.52	74.21	70.47	71.90
-2.96	2.50	233.4	17.58	43.48	25.79	29.53	28.10
Total		**1327.80**	**100**	100.00	100.00	100.00	100.00

Table B4.2: 4E results for Sample 2

-150+106µm				Recovery [%]			
Density	RD to sinks	Mass [g]	Mass [%]	Au	Pd	Pt	Tot PGM+Au
+2.96*	3.80	1154.0	87.51	78.67	85.38	82.45	83.70
-2.96	2.50	164.7	12.49	21.33	14.62	17.55	16.30
Total		**1318.70**	**100**	100.00	100.00	100.00	100.00

Table B4.3: Cr_2O_3, SiO_2, and Fe_2O_3 results for Sample 1

-212+150µm				Recovery [%]		
Density	RD to sinks	Mass [g]	Mass [%]	Cr_2O_3	SiO_2	Fe_2O_3
+2.96*	3.80	1094.4	82.42	99.69	55.98	98.21
-2.96	2.50	233.4	17.58	0.31	44.02	1.79
Total		**1327.80**	**100**	100.00	100.00	100.00

Table B4.4: Cr_2O_3, SiO_2, and Fe_2O_3 results for Sample 2

-150+106µm				Recovery [%]		
Density	RD to sinks	Mass [g]	Mass [%]	Cr_2O_3	SiO_2	Fe_2O_3
+2.96*	3.80	1154.0	87.51	99.77	71.22	99.07
-2.96	2.50	164.7	12.49	0.23	28.78	0.93
Total		**1318.70**	**100**	100.00	100.00	100.00

Appendix B5: Mintek Mineralogy results for SRT plant audit sample

Table B5.1: Bulk modal analysis results for SRT

Modal Analysis	
UKZN - SRT Sample	
Mineral	**Mass %**
Pentlandite	<1
Chalcopyrite	<1
Pyrrhotite	<1
Pyrite	<1
Other Sulphides	<1
Pyroxene	16
Feldspar	14
Amphibole	<1
Olivine	1
Talc	1
Serpentine	1
Mica	1
Quartz	<1
Chromite	65
Fe/Ti Oxides	1
Other Oxides	1
Calcite/Apatite	<1
Other Silicates	<1
Other	<1
Total	**100**

Table B5.2: PGM grain floatability index

PGM grain Floatability Index			
	Particle Characteristics	**PGM Vol %**	**No of PGMs**
Fast Floating	Liberated PGM's >3µm ECD	9.3	5
	Liberated BMS >10µm ECD	18.3	6
Slow Floating 1	Liberated PGM's <3µm ECD	3.7	4
	Liberated BMS <10µm ECD	0	0
	PGM's >3µm ECD attached to gangue	17.7	7
	BMS >10µm ECD attached to gangue	11.4	10
Slow Floating 2	PGM's <3µm ECD attached to gangue	6.3	8
	BMS <10µm ECD attached to gangue	1.4	1
No Floating	PGM's and/or BMS locked in gangue	31.8	20
		100.0	**61**

Table B5.3: PGM grain liberation index

PGM grain Liberation Index			
Index	**No of PGMs**	**PGM Vol %**	**Num %**
<0.2	41	61.8	67.2
0.2 - 0.4	3	5.4	4.9
0.4 - 0.6	1	0.5	1.6
0.6 - 0.8	1	1.0	1.6
0.8 - 1.0	15	31.4	24.6
	61	**100.0**	**100.0**

Table B5.4: PGM grain size distribution

PGM grain size distribution			
Class	**No of PGM**	**PGM Vol %**	**Num %**
0 - 3	28	22.6	45.9
3 - 6	30	60.6	49.2
6 - 9	3	16.8	4.9
9 - 12			
12 - 15			
15 - 18			
18 - 21			
21 - 24			
24 - 27			
27 - 30			
>30			

Table B5.5: PGM search data

UKZN SRT Sample										
PGM grains						**BMS**		**Gangue**		**Host Types**
Type	**MoO**	**Floatability Index**	**Lib Index**	**Area**	**µm ECD ***	**Area**	**µm ECD ***	**Area**	**µm ECD ***	/ = Attached, ( ) = Locked
PtFe	L	Fast Floating	1	7.2	3.0					-
PtS	L	Fast Floating	1	7.6	3.1					-
PtS	L	Fast Floating	1	12.2	3.9					-
PtPdSb	L	Fast Floating	1	18.4	4.8					-
PtPdS	L	Fast Floating	1	10.8	3.7					-
PtFe	SL	Fast Floating	1	25.9	5.7	277.2	18.8			Chalcopyrite
PtPdS	SL	Fast Floating	1	4.7	2.4	87.8	10.6			Pyrrhotite
PtS	SL	Fast Floating	1	10.8	3.7	129.2	12.8			Pentlandite
PtRhCuS	SL	Fast Floating	1	37.4	6.9	189.7	15.5			Pentlandite
PtS	SL	Fast Floating	1	2.9	1.9	315.4	20.0			Pentlandite
PtFe	SL	Fast Floating	1	28.4	6.0	91.8	10.8			Pentlandite
PtS	L	Slow Floating 1	1	4.7	2.4					-
PtS	L	Slow Floating 1	1	5.4	2.6					-
PtS	L	Slow Floating 1	1	6.5	2.9					-
PtS	L	Slow Floating 1	1	5.8	2.7					-
PtS	AG	Slow Floating 1	0	10.4	3.6			1255.3	40.0	Orthopyroxene
PtPbCuNiS	AG	Slow Floating 1	0	14.4	4.3			3224.5	64.1	Plagioclase
PtS	AG	Slow Floating 1	0	9.0	3.4			2814.8	59.9	Plagioclase
PtFe	AG	Slow Floating 1	0	20.2	5.1			870.1	33.3	Clinopyroxene
PtPdS	AG	Slow Floating 1	0	9.0	3.4			2831.4	60.0	Plagioclase
PtFe	AG	Slow Floating 1	0	7.9	3.2			1315.1	40.9	Chromite
PtPdBiTe	AG	Slow Floating 1	0	35.3	6.7			4242.2	73.5	Clinopyroxene
PtS	SAG	Slow Floating 1	0	9.0	3.4	979.9	35.3	2881.1	60.6	Pentlandite/Plagioclase
PtAsS	SAG	Slow Floating 1	0	9.0	3.4	337.7	20.7	895.7	33.8	Pentlandite/Clinopyroxene
PtPdS	SAG	Slow Floating 1	0	2.9	1.9	418.0	23.1	564.1	26.8	Chalcopyrite/Plagioclase
PtRhCuS	SAG	Slow Floating 1	0	9.7	3.5	373.3	21.8	2360.9	54.8	Pentlandite/Plagioclase
PtRhCuS	SAG	Slow Floating 1	0	6.5	2.9	601.6	27.7	2435.8	55.7	Chalcopyrite/Plagioclase
PdS	SAG	Slow Floating 1	0	6.1	2.8	126.7	12.7	2657.5	58.2	Pentlandite/Talc
PdS	SAG	Slow Floating 1	0	5.4	2.6	267.1	18.4	2064.6	51.3	Pentlandite/Chlorite
PtS	SAG	Slow Floating 1	1	6.1	2.8	982.1	35.4	630.0	28.3	Pentlandite/Plagioclase
PtFe	SAG	Slow Floating 1	0	5.8	2.7	117.4	12.2	2647.4	58.1	Pentlandite/Plagioclase
PdPb	SAG	Slow Floating 1	0	8.3	3.2	143.3	13.5	2219.8	53.2	Chalcopyrite/Clinopyroxene
PtAsS	AG	Slow Floating 2	0	6.8	3.0			630.4	28.3	Clinopyroxene
PtRhCuS	AG	Slow Floating 2	0	3.6	2.1			354.6	21.2	Plagioclase
PdS	AG	Slow Floating 2	0	4.7	2.4			124.6	12.6	Amphibole
Ru(Os,Ir)S	AG	Slow Floating 2	0	5.8	2.7			190.4	15.6	Orthopyroxene
PtFe	AG	Slow Floating 2	0	4.0	2.2			2594.2	57.5	Orthopyroxene
PdS	AG	Slow Floating 2	0	4.3	2.3			174.2	14.9	Clinopyroxene
PtPdS	AG	Slow Floating 2	0	2.2	1.7			1932.8	49.6	Orthopyroxene
PtS	AG	Slow Floating 2	0	6.8	3.0			904.7	33.9	Plagioclase
PtPdS	SAG	Slow Floating 2	0	8.3	3.2	63.4	9.0	475.6	24.6	Pentlandite/Plagioclase
Ru(Os,Ir)S	SG	No Floating	0	6.8	3.0	27.4	5.9	1403.3	42.3	(Chalcopyrite)Clinopyroxene
PtPdS	SG	No Floating	0	9.4	3.5	28.8	6.1	384.1	22.1	(Pentlandite)Chlorite
PdBiTe	SG	No Floating	0	13.3	4.1	63.7	9.0	3553.9	67.3	(Pentlandite)Plagioclase
PdPb	SG	No Floating	0	10.4	3.6	82.1	10.2	3704.8	68.7	(Pentlandite)Plagioclase
PtFe	G	No Floating	0	2.2	1.7			5989.7	87.3	Clinopyroxene
Ru(Os,Ir)S	G	No Floating	0	6.5	2.9			326.2	20.4	Chromite
Ru(Os,Ir)S	G	No Floating	0	21.2	5.2			1185.5	38.9	Chromite
PtAs	G	No Floating	0	7.2	3.0			1617.8	45.4	Clinopyroxene
PtFe	G	No Floating	0	11.5	3.8			1415.2	42.4	Plagioclase
PtPdS	G	No Floating	0	5.0	2.5			4623.8	76.7	Plagioclase
PtFe	G	No Floating	0	14.4	4.3			27.7	5.9	Clinopyroxene
AuAg	G	No Floating	0	3.6	2.1			839.9	32.7	Plagioclase
PtS	G	No Floating	0	3.2	2.0			1516.3	43.9	Talc
PtS	G	No Floating	0	4.3	2.3			683.3	29.5	Ilmenite
PtFe	G	No Floating	0	19.8	5.0			656.3	28.9	Orthopyroxene
PtPdS	G	No Floating	0	3.2	2.0			1823.4	48.2	Plagioclase
PtS	G	No Floating	0	10.1	3.6			2137.3	52.2	Orthopyroxene
PtAs	G	No Floating	0	9.4	3.5			3041.6	62.2	Plagioclase
PtS	G	No Floating	0	13.0	4.1			2594.2	57.5	Chlorite
PtFe	G	No Floating	0	16.9	4.6			5465.5	83.4	Plagioclase

Appendix B6: Additional sizing results for grinding tests with a synthetic UG-2 ore mixture

Table B6.1: Sizing of pure limestone

Screen size (µm)	Mass (%)	Cum. % Passing - Sample 1	Cum. % Passing (Repeat)
+425	0.1	100.0	100.0
-425+300	0.9	99.9	99.9
-300+212	5.7	99.8	99.0
-212+150	13.0	93.3	93.3
-150+106	16.1	80.6	80.3
-106+75	14.1	62.0	64.2
-75+53	29.5	44.3	50.1
-53+38	9.9	13.7	20.6
- 38	10.8	2.8	10.8

Table B6.2: Sizing of pure ilmenite

Screen size (µm)	Mass %	Cum. % passing	Cum. % passing (Repeat)
-600	2.39	100.00	100.00
-600+425	15.42	97.61	99.12
-425+300	28.18	82.18	83.67
-300+212	28.63	54.00	56.22
-212+150	16.74	25.37	26.13
-150+106	6.56	8.62	8.94
-106+75	1.51	2.06	2.07
-75+53	0.39	0.55	0.52
-53+38	0.16	0.16	0.16
- 38	0.00	0.00	0.00

Table B6.3: Sizing of 20% ilmenite-80% limestone mixture

Screen size (µm)	Mass % - ilmenite	Mass % - limestone	Cum. % passing - ilmenite	Cum. % passing - limestone
+425	0.0	0.0	100.0	100.0
-425+300	0.9	0.8	100.0	100.0
-300+212	20.6	4.0	99.1	99.2
-212+150	37.5	10.0	78.5	95.2
-150+106	17.6	18.2	41.0	85.2
-106+75	10.8	16.8	23.3	66.9
-75+53	6.4	32.6	12.6	50.1
-53+38	4.3	15.3	6.2	17.5
- 38	1.8	2.2	1.8	2.2

Table B6.4: Sizing of 40% ilmenite-60% limestone mixture

Screen size (µm)	Mass % - ilmenite	Mass % - limestone	Cum. % passing - ilmenite	Cum. % passing - limestone
+425	0.0	0.0	100.0	100.0
-425+300	1.0	0.8	100.0	100.0
-300+212	10.8	8.0	99.0	99.2
-212+150	12.0	21.1	88.2	91.2
-150+106	32.7	17.1	76.3	70.1
-106+75	17.4	16.9	43.6	53.0
-75+53	19.1	27.8	26.2	36.1
-53+38	5.9	5.3	7.1	8.4
- 38	1.2	3.1	1.2	3.1

Table B6.5: Sizing of 60% ilmenite-40% limestone mixture

Screen size (µm)	Mass % - ilmenite	Mass % - limestone	Cum. % passing - ilmenite	Cum. % passing - limestone
+425	0.2	0.0	100.0	100.0
-425+300	2.4	1.1	99.8	100.0
-300+212	16.4	6.6	97.4	98.9
-212+150	25.2	9.8	81.0	92.2
-150+106	20.2	13.2	55.8	82.5
-106+75	10.7	14.7	35.6	69.3
-75+53	11.5	22.7	24.9	54.6
-53+38	6.1	11.5	13.5	31.9
- 38	7.3	20.4	7.3	20.4

Table B6.6: Sizing of 80% ilmenite-20% limestone mixture

Screen size (µm)	Mass % - ilmenite	Mass % - limestone	Cum. % passing - ilmenite	Cum. % passing - limestone
+425	0.0	0.0	100.0	100.0
-425+300	1.1	2.0	100.0	100.0
-300+212	14.5	3.3	98.9	98.0
-212+150	24.2	11.4	84.4	94.7
-150+106	22.6	12.2	60.2	83.3
-106+75	12.9	14.7	37.6	71.0
-75+53	14.7	30.6	24.7	56.3
-53+38	4.6	0.4	9.9	25.7
- 38	5.3	25.3	5.3	25.3

Table B6.7: Sizing of 50% ilmenite-50% limestone mixture

Screen size (µm)	Mass % - ilmenite	Mass % - limestone	Cum. % passing - ilmenite	Cum. % passing - limestone
+425	0.00	0.00	100.00	100.00
-425+300	0.00	0.00	100.00	100.00
-300+212	7.52	8.79	100.00	100.00
-212+150	6.58	13.19	92.48	91.21
-150+106	44.20	19.23	85.89	78.02
-106+75	27.59	20.33	41.69	58.79
-75+53	8.78	29.12	14.11	38.46
-53+38	4.08	8.24	5.33	9.34
- 38	1.25	1.10	1.25	1.10

Table B6.8: Repeat test - Sizing of 80% ilmenite-20% limestone mixture

Screen size (µm)	Mass % - ilmenite	Mass % - limestone	Cum. % passing - ilmenite	Cum. % passing - limestone
+150	2.9	6.2	97.1	93.8
-150+106	61.9	62.5	35.2	31.3
-106+75	20.4	16.8	14.8	14.5
-75+53	10.4	8.5	4.4	5.9
-53+38	3.1	3.6	1.3	2.3
- 38	1.3	2.3	0.0	0.0

Table B6.9: Repeat test - Sizing of 50% ilmenite-50% limestone mixture

Screen size (µm)	Mass % - ilmenite	Mass % - limestone	Cum. % passing - ilmenite	Cum. % passing - limestone
+150	4.6	11.3	95.4	88.7
-150+106	74.3	65.9	21.1	22.8
-106+75	12.6	11.6	8.5	11.2
-75+53	6.4	7.8	2.0	3.4
-53+38	1.7	2.3	0.4	1.1
- 38	0.4	1.1	0.0	0.0

Appendix B7: Tables of results of the commissioning runs on the MG-type spiral concentrator

Table B7.1: Percent solids table for running the MG-type spiral concentrator (Density Can Test)

Slurry Density (kg/m^3)	% solids
1000	0
1010	1.4
1020	2.8
1030	4.1
1040	5.4
1050	6.7
1060	8.0
1070	9.3
1080	10.5
1090	11.7
1100	12.9
1110	14.0
1120	15.2
1130	16.3
1140	17.4
1150	18.5
1160	19.5
1170	20.6
1180	21.6
1190	22.6
1200	23.6
1210	24.6
1220	25.5
1230	26.5
1240	27.4
1250	28.3
1260	29.2
1270	30.1

Table B7.2: Effect of flowrate on the UG-2 ore (0.63l/s)

Stream	Mass%	Cu Grade (ppm)	Ni Grade (ppm)	Cr Grade (ppm)	Recovery Cu	Recovery Ni	Recovery Cr
Light Fraction	57.7	139.0	137.2	38.5	65.1	49.0	28.5
Middle Fraction	14.6	167.4	279.1	347.8	19.7	24.8	62.3
Heavy Fraction	27.6	68.0	153.1	26.0	15.2	26.2	9.2
Total	100.0	123.5	162.3	80.2	100.0%	100.0%	100.0%

Table B7.3: Effect of flowrate on the UG-2 ore (0.63l/s repeat)

Stream	Mass%	Cu Grade (ppm)	Ni Grade (ppm)	Cr Grade (ppm)	Recovery Cu	Recovery Ni	Recovery Cr
Light Fraction	43.9	157.0	183.6	41.9	62.3	55.7	57.8
Middle Fraction	21.4	95.3	117.4	26.8	18.4	17.3	18.0
Heavy Fraction	34.7	61.4	112.1	22.2	19.3	26.9	24.2
Total	100.0	110.6	144.6	31.8	100.0	100.0	100.0

Table B7.4: Effect of flowrate on the UG-2 ore (0.51l/s)

Stream	Mass%	Cu Grade (ppm)	Ni Grade (ppm)	Cr Grade (ppm)	Recovery Cu	Recovery Ni	Recovery Cr
Light Fraction	28.5	23.4	23.5	25.8	40.3	27.8	41.4
Middle Fraction	23.9	15.9	17.1	14.5	23.0	17.0	19.4
Heavy Fraction	47.6	12.8	28.0	14.7	36.7	55.3	39.3
Total	100.0	16.5	24.1	17.8	100.0%	100.0%	100.0%

Table B7.5: Effect of flowrate on the UG-2 ore (0.51l/s repeat)

Stream	Mass%	Cu Grade (ppm)	Ni Grade (ppm)	Cr Grade (ppm)	Recovery Cu	Recovery Ni	Recovery Cr
Light Fraction	27.0	32.5	35.4	32.3	41.9	34.1	41.7
Middle Fraction	23.7	27.8	40.6	18.8	31.6	34.4	21.4
Heavy Fraction	49.3	10.5	17.3	15.1	24.8	30.4	35.6
Total	100.0	20.9	28.0	20.8	100.0%	100.0%	100.0%

Table B7.6: Effect of flowrate on the UG-2 ore (0.87l/s)

Stream	Mass%	Cu Grade (ppm)	Ni Grade (ppm)	Cr Grade (ppm)	Recovery Cu	Recovery Ni	Recovery Cr
Light Fraction	42.2	23.1	28.7	30.9	75.1	85.3	67.9
Middle Fraction	22.4	5.7	2.2	12.7	9.8	3.5	14.9
Heavy Fraction	35.4	5.6	4.5	9.4	15.1	11.2	17.3
Total	100.0	13.0	14.2	19.2	100.0%	100.0%	100.0%

Table B7.7: Effect of flowrate on the UG-2 ore (0.87l/s repeat)

Stream	Mass%	Cu Grade (ppm)	Ni Grade (ppm)	Cr Grade (ppm)	Recovery Cu	Recovery Ni	Recovery Cr
Light Fraction	42.8	37.7	54.0	40.3	63.4	61.2	60.4
Middle Fraction	22.8	18.8	28.8	18.5	16.9	17.4	14.7
Heavy Fraction	34.4	15.1	24.1	21.0	20.4	21.9	25.3
Total	100.0	25.5	37.8	28.6	100.0%	100.0%	100.0%

Appendix B8: Tables of flotation results for separate milling tests on high grade 4E laboratory sample

Table B8.1: Feed mixtures for Ratio tests

Feed Mixtures	UG-2 Ore	Mass (%)	250 g Mixture
20L/80H	Chromite	87.4	218.5
	Silicates	12.6	31.5
80L/20H	Chromite	30.3	75.7
	Silicates	69.7	174.3
60L/40H	Chromite	53.6	134.1
	Silicates	46.4	115.9
40L/60H	Chromite	72.3	180.6
	Silicates	27.7	69.4

Table B8.2: Flotation results for (80L/20H) Lights + (20L/80H) Heavies Ratio - 88% Energy input to lights mill

(80L/20H) Lights + (20L/80H) Heavies Ratio - 88% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	7.4	69.6	1.9
Tail	92.6	30.4	98.1
Total	100.0	100.0	100.0

Table B8.3: Flotation results for (80L/20H) Lights + (20L/80H) Heavies Ratio - 75% Energy input to lights mill

(80L/20H) Lights + (20L/80H) Heavies Ratio - 75% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	8.6	58.2	2.7
Tail	91.4	41.8	97.3
Total	100.0	100.0	100.0

Table B8.4: Flotation results for (80L/20H) Lights + (20L/80H) Heavies Ratio - 75% Energy input to lights mill (Repeat)

(80L/20H) Lights + (20L/80H) Heavies Ratio - 75% Energy input to lights mill (Repeat)			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc	42.0	8.4	60.3	2.5
Tail	457.2	91.6	39.7	97.5
Total	499.2	100.0	100.0	100.0

Table B8.5: Flotation results for (80L/20H) Lights + (20L/80H) Heavies Ratio - 63% Energy input to lights mill

(80L/20H) Lights + (20L/80H) Heavies Ratio - 63% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	8.1	63.5	2.4
Tail	91.9	36.5	97.6
Total	100.0	100.0	100.0

Table B8.6: Flotation results for (80L/20H) Lights + (20L/80H) Heavies Ratio - 50% Energy input to lights mill

(80L/20H) Lights + (20L/80H) Heavies Ratio - 50% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	7.0	64.7	1.8
Tail	93.0	35.3	98.2
Total	100.0	100.0	100.0

Table B8.7: Flotation results for (80L/20H) Lights + (20L/80H) Heavies Ratio - 37% Energy input to lights mill

(80L/20H) Lights + (20L/80H) Heavies Ratio - 37% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	7.0	65.6	1.9
Tail	93.0	34.4	98.1
Total	100.0	100.0	100.0

Table B8.8: Flotation results for 100% Lights + 100% Heavies - 100% Energy input to lights mill

100% Lights + 100% Heavies - 100% Energy input to lights mill			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc	39.4	8.0	67.6	1.5
Tail	455.1	92.0	32.4	98.5
Total	494.5	100.0	100.0	100.0

Table B8.9: Flotation results for 100% Lights + 100% Heavies - 88% Energy input to lights mill

100% Lights + 100% Heavies - 88% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	7.5	69.0	1.4
Tail	92.5	31.0	98.6
Total	100.0	100.0	100.0

Table B8.10: Flotation results for 100% Lights + 100% Heavies - 75% Energy input to lights mill

100% Lights + 100% Heavies - 75% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	7.7	68.2	1.7
Tail	92.3	31.8	98.3
Total	100.0	100.0	100.0

Table B8.11: Flotation results for 100% Lights + 100% Heavies - 75% Energy input to lights mill (Repeat)

100% Lights + 100% Heavies - 75% Energy input to lights mill (Repeat)		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	7.6	67.5	1.7
Tail	92.4	32.5	98.3
Total	100.0	100.0	100.0

Table B8.12: Flotation results for 100% Lights + 100% Heavies - 63% Energy input to lights mill

100% Lights + 100% Heavies - 63% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	8.0	68.9	2.1
Tail	92.0	31.1	97.9
Total	100.0	100.0	100.0

Table B8.13: Flotation results for 100% Lights + 100% Heavies - 37% Energy input to lights mill

100% Lights + 100% Heavies - 37% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	7.7	67.0	2.3
Tail	92.3	33.0	97.7
Total	100.0	100.0	100.0

Table B8.14: Comparison of variances for repeat analyses

	Variance (%) for Repeat Analyses		
Repeat Tests	Re-Calculated 4E Head Grades	4E Recovery	Cr_2O_3 Recovery
75% Energy input (100%L)	0.00	0.95	2.26
75% Energy input (80/20 L-H)	0.54	3.65	6.78
Standard Test	0.00	2.89	5.35

Appendix B9: Tables of repeat flotation results for separate milling tests on low grade 4E laboratory sample (100% Lights + 100% Heavies)

Table B9.1: Flotation results for standard process

Standard Test		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	6.0	51.3	2.3
Tail	94.0	48.7	97.7
Total	100.0	100.0	100.0

Table B9.2: Flotation results for 100% Lights + 100% Heavies - 88% Energy input to lights mill

88 % Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	6.9	54.0	1.6
Tail	93.1	46.0	98.4
Total	100.0	100.0	100.0

Table B9.3: Flotation results for 100% Lights + 100% Heavies - 88% Energy input to lights mill (Repeat)

88 % Energy input to lights mill (Repeat)		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	6.2	53.2	1.4
Tail	93.8	46.8	98.6
Total	100.0	100.0	100.0

Table B9.4: Flotation results for 100% Lights + 100% Heavies - 75% Energy input to lights mill

75% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	6.2	51.9	1.8
Tail	93.8	48.1	98.2
Total	100.0	100.0	100.0

Table B9.5: Flotation results for 100% Lights + 100% Heavies - 63% Energy input to lights mill

63% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	6.2	51.2	2.1
Tail	93.8	48.8	97.9
Total	100.0	100.0	100.0

Table B9.6: Flotation results for 100% Lights + 100% Heavies - 50% Energy input to lights mill

50% Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	7.3	51.1	2.5
Tail	92.7	48.9	97.5
Total	100.0	100.0	100.0

Table B9.7: Flotation results for 100% Lights + 100% Heavies - 37% Energy input to lights mill

37 % Energy input to lights mill		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	6.2	48.0	2.2
Tail	93.8	52.0	97.8
Total	100.0	100.0	100.0

Table B9.8: Rate Test – Standard process

Rate test for standard process		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc 1 (2min)	2.9	33.2	0.8
Conc 2 (5min)	1.5	12.3	0.5
Conc 3 (10min)	1.4	8.8	0.6
Tail	94.2	45.7	98.1
Total	100.0	100.0	100.0

Table B9.9: Rate Test – 100% Lights + 100% Heavies - 88% Energy input to lights mill

Rate test for 88 % Energy input to lights		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc 1 (2min)	3.2	37.7	0.6
Conc 2 (5min)	1.4	10.8	0.3
Conc 3 (10min)	1.4	9.0	0.5
Tail	94.0	42.4	98.6
Total	100.0	100.0	100.0

Table B9.10: Summary of water recoveries for separate milling

% Energy Input to Ball Mill		Masspull (%)	Recovery (%)	
Lights	Heavies	Conc	Water	Cr_2O_3
88	2	6.5	13.8	1.5
75	4	6.2	14.8	1.8
63	6	6.2	15.3	2.1
50	8	7.3	15.5	2.5
37	10	6.2	15.6	2.2

Table B9.11: Repeat tests for 100% Lights + 100% Heavies - 88% Energy input to lights mill

		Recovery		
Concentrates	Mass (%)	4E (%)	Cr_2O_3 (%)	Water (%)
Test 1	7.1	57.6	1.4	14.0
Test 2 (repeat)	6.9	54.0	1.6	13.8
Test 3 (repeat)	6.2	53.2	1.4	13.7
Average	6.7	54.9	1.5	13.9

Table B9.12: Repeat tests of standard process

		Recovery		
Concentrates	Mass (%)	4E (%)	Cr_2O_3 (%)	Water (%)
Test 1	6.1	51.3	2.3	14.0
Test 2 (repeat)	5.8	54.3	1.9	14.6
Average	6.0	52.8	2.1	14.3

Appendix B10: Flotation results for the ball milling of the spiral lights and attritioning of spiral heavies fraction

Table B10.1: Test 1 – Ball milling of lights and attritioning of heavies

Ball milling of lights / Attritioning of Heavies		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	5.3	53.9	1.0
Tail	94.7	46.1	99.0
Total	100.0	100.0	100.0

Table B10.2: Test 2 – Ball milling of lights and attritioning of heavies (Repeat)

Ball milling of lights / Attritioning of Heavies (Repeat)		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	5.2	52.4	0.9
Tail	94.8	47.6	99.1
Total	100.0	100.0	100.0

Appendix B11: Flotation results for the alternate milling circuit design incorporating a coarse silicates recycle from the spiral concentrator (Locked Cycle Test)

Table B11.1: Flotation results for cycle 1

Cycle 1		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	5.5	66.9	2.4
Tail	94.5	33.1	97.6
Total	100.0	100.0	100.0

Table B11.2: Flotation results for cycle 1 (repeat)

Cycle 1 - Repeat		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	5.4	71.0	1.9
Tail	94.6	29.0	98.1
Total	100.0	100.0	100.0

Table B11.3: Flotation results for cycle 2

Cycle 2		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	5.7	64.9	2.6
Tail	94.3	35.1	97.4
Total	100.0	100.0	100.0

Table B11.4: Flotation results for cycle 2 (repeat)

Cycle 2 - Repeat		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	5.2	63.6	2.5
Tail	94.8	36.4	97.5
Total	100.0	100.0	100.0

Table B11.5: Flotation results for cycle 3

Cycle 3		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	6.3	66.7	3.1
Tail	93.7	33.3	96.9
Total	100.0	100.0	100.0

Table B11.6: Flotation results for cycle 3 (Repeat)

Cycle 3 - Repeat		Recovery (%)	
Stream	Mass (%)	4E	Cr_2O_3
Conc	5.0	64.4	2.1
Tail	95.0	35.6	97.9
Total	100.0	100.0	100.0

Appendix B12: Tables of size-by-assay results of laboratory spiral products

Table B12.1: Size-by-assay results for the rougher spiral lights product

Rougher Spiral Lights Product		Distribution (%)	
Screen Size (um)	Mass (%)	4E	Cr_2O_3
106	67.9	48.8	70.7
75	18.7	32.3	9.8
53	8.0	11.6	8.7
38	3.3	3.0	6.6
Base	2.0	4.3	4.2
Total	100.0	100.0	100.0

Table B12.2: Size-by-assay results for the rougher spiral heavies product

Rougher Spiral Heavies Product		Distribution (%)	
Screen Size (μm)	Mass (%)	4E	Cr_2O_3
+106	50.2	30.6	53.4
-106+75	34.4	34.1	32.8
-75+53	11.8	23.1	10.5
-53+38	2.3	7.1	2.2
-38	1.2	5.1	1.1
Total	100.0	100.0	100.0

Table B12.3: Size-by-assay results for the cleaner spiral lights product

Heavies Cleaner Spiral - Lights Product		Distribution (%)	
Screen Size (μm)	Mass (%)	4E	Cr_2O_3
+106	21.4	10.2	9.4
-106+75	49.0	52.5	51.9
-75+53	22.5	22.8	28.4
-53+38	4.9	5.3	7.0
-38	2.3	9.3	3.3
Total	100.0	100.0	100.0

Table B12.4: Size-by-assay results for the cleaner spiral middlings product

Heavies Cleaner Spiral - Middlings Product		Distribution (%)	
Screen Size (µm)	Mass (%)	4E	Cr_2O_3
+106	61.0	36.3	62.3
-106+75	29.2	44.9	27.2
-75+53	7.7	14.4	8.3
-53+38	1.5	3.9	1.4
-38	0.6	0.5	0.7
Total	100.0	100.0	100.0

Table B12.5: Size-by-assay results for the cleaner spiral heavies product

Heavies Cleaner Spiral - Heavies Product		Distribution (%)	
Screen Size (µm)	Mass (%)	4E	Cr_2O_3
+106	63.5	32.1	64.5
-106+75	26.3	38.0	25.2
-75+53	7.9	19.1	8.0
-53+38	1.7	7.7	1.7
-38	0.6	3.1	0.6
Total	100.0	100.0	100.0

Appendix B13: Tables of size-by-assay results of milled products

Table B13.1: Size-by-assay results of standard milled product

Standard feed (500g) - 100% Energy input		Distribution (%)	
Screen Size (µm)	Mass (%)	4E	Cr_2O_3
+106	4.1	2.5	2.5
-106+75	15.9	15.5	13.6
-75+53	24.2	25.7	25.6
-53+38	12.0	13.9	13.4
-38	43.8	42.4	44.8
Total	100.0	100.0	100.0

Table B13.2: Size-by-assay results of milled rougher lights with 50% energy input

Rougher Lights - (250g) 50% - Energy input		Distribution (%)	
Screen Size (μm)	Mass (%)	4E	Cr_2O_3
+106	4.0	1.1	1.6
-106+75	19.0	16.9	12.0
-75+53	18.3	11.0	17.6
-53+38	9.9	5.3	12.8
-38	48.8	65.7	56.0
Total	100.0	100.0	100.0

Table B13.3: Size-by-assay results of milled rougher heavies with 50% energy input

Rougher Heavies - (250g) - 50% Energy input		Distribution (%)	
Screen Size (μm)	Mass (%)	4E	Cr_2O_3
+106	16.7	11.7	17.7
-106+75			
-75+53	24.5	22.0	24.0
-53+38	7.8	5.9	7.6
-38	50.9	60.4	50.7
Total	100.0	100.0	100.0

Table B13.4: Size-by-assay results of milled rougher lights with 88% energy input

Rougher Lights - (250g) - 88% energy input		Distribution (%)	
Screen Size (μm)	Mass (%)	4E	Cr_2O_3
+106	8.7	3.5	10.8
-106+75			
-75+53	19.7	10.3	16.9
-53+38	9.4	4.6	9.8
-38	62.2	81.6	62.5
Total	100.0	100.0	100.0

Table B13.5: Size-by-assay results of milled rougher heavies with 12% energy input

Rougher Heavies - (250g) - 12% Energy input		Distribution (%)	
Screen Size (μm)	Mass (%)	4E	Cr_2O_3
+106	16.4	6.0	21.0
-106+75	37.2	12.0	25.2
-75+53	19.9	31.5	22.7
-53+38	3.8	5.2	4.5
-38	22.7	45.3	26.7
Total	100.0	100.0	100.0

Appendix B14: Tables of flotation results with pilot plant spiral products

Table B14.1: % Variances of head grades of each stream of pilot plant sample

Streams	Densifier Underflow		Rougher Lights		Rougher Heavies		Cleaner Lights		Cleaner Middlings		Cleaner Heavies	
Head Grade (ppm/%)	4E	Cr_2O_3	4E	Cr_2O_3	4E	Cr_2O_3	4E	Cr_2O_3	4E	Cr_2O_3	4E	Cr_2O_3
% Variance	8.4	0.9	12.5	4.0	0.5	1.2	2.1	1.6	1.3	1.2	0.4	0.3

Table B14.2: Flotation results of standard rate test from pilot plant spiral products

Standard Rate Test			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc 1 (2min)	12.3	2.5	38.5	1.0
Conc 2 (5min)	7.7	1.6	18.4	0.5
Conc 3 (10min)	7.6	1.5	7.9	0.6
Tail	468.9	94.4	35.2	98.0
Total	496.5	100.0	100.0	100.0

Table B14.3: Flotation results of standard rate test (repeat) from pilot plant spiral products

Standard Rate Test (Repeat)			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc 1 (2min)	13.9	2.8	39.9	1.1
Conc 2 (5min)	7.9	1.6	15.1	0.6
Conc 3 (10min)	8.0	1.6	9.6	0.6
Tail	463.0	94.0	35.4	97.6
Total	492.8	100.0	100.0	100.0

Table B14.4: Flotation results from 100% Lights + 100% Heavies - 88% Energy input to lights mill (Rate Test from pilot plant spiral products)

Rate Test - 88 % Energy input to lights mill			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc 1 (2min)	13.9	2.8	41.7	0.5
Conc 2 (5min)	8.0	1.6	18.7	0.4
Conc 3 (10min)	7.7	1.5	8.0	0.5
Tail	472.1	94.1	31.6	98.6
Total	501.7	100.0	100.0	100.0

Table B14.5: Flotation results from 100% Lights + 100% Heavies - 88% Energy input to lights mill (Repeat Test from pilot plant spiral products)

Rate Test - 88 % Energy input to lights mill (repeat)			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc 1 (2min)	15.2	3.1	42.5	0.7
Conc 2 (5min)	8.1	1.6	18.9	0.5
Conc 3 (10min)	7.6	1.5	7.4	0.5
Tail	460.9	93.7	31.2	98.3
Total	491.8	100.0	100.0	100.0

Table B14.6: Flotation results from 100% Lights + 100% Heavies - 88% Energy input to lights mill (Bulk rougher test from pilot plant spiral products)

88 % Energy input to lights mill (bulk conc repeat)			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc	28.0	5.6	68.1	1.3
Tail	475.0	94.4	31.9	98.7
Total	503.0	100.0	100.0	100.0

Appendix B15: Tables of lights-only flotation results with pilot plant spiral products

Table B15.1: Lights-only flotation results from 100% Lights + 100% Heavies - 88% Energy input to lights mill (Discarded cleaner spiral heavies)

Lights -only rate test (88% energy input)			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc 1 (2min)	10.3	2.1	11.0	0.3
Conc 2 (5min)	5.4	1.1	5.7	0.2
Conc 3 (10min)	6.7	1.3	5.8	0.3
Tail	227.6	45.5	20.3	17.5
H Discard	250.0	50.0	57.2	81.7
Total	500.0	100.0	100.0	100.0

Table B15.2: Lights-only flotation results from 100% Lights + 100% Heavies - 88% Energy input to lights mill – Repeat test (Discarded cleaner spiral heavies stream)

Lights -only bulk flotation test (88% energy input)			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc	23.1	4.6	19.4	0.7
Tail	224.3	45.1	20.9	16.5
H Discard	250.0	50.3	59.7	82.9
Total	497.4	100.0	100.0	100.0

Table B15.3: Lights-only flotation results from 100% Lights + 100% Heavies - 63% Energy input to lights mill (Discarded cleaner spiral heavies)

Lights -only rate test (63% energy input)			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc 1 (2min)	11.0	2.2	9.7	0.2
Conc 2 (5min)	6.6	1.3	4.3	0.2
Conc 3 (10min)	7.3	1.5	3.8	0.3
Tail	225.6	45.1	23.0	16.3
H Discard	250.0	50.0	59.3	83.1
Total	500.5	100.0	100.0	100.0

Table B15.4: Lights-only flotation results from 100% Lights + 100% Heavies - 63% Energy input to lights mill – Repeat test (Discarded cleaner spiral heavies stream)

Lights -only bulk flotation test (63% energy input)			Recovery (%)	
Stream	Mass (g)	Mass (%)	4E	Cr_2O_3
Conc	25.1	5.0	20.7	0.7
Tail	225.2	45.0	20.6	16.1
H Discard	250.0	50.0	58.7	83.2
Total	500.3	100.0	100.0	100.0

Table B15.5: Summary of repeat flotation results for standard process

Standard Tests	Mass %	4E Recovery (%)	Cr_2O_3 Recovery (%)	Water Recovery (%)
Test 1	5.6	64.8	2.0	13.4
Test 2 (Repeat)	6.0	64.6	2.4	14.8
Average	5.8	64.7	2.2	14.1

Table B15.6: Summary of repeat separate milling flotation results for the 88% energy input to lights mill

Separate Milling	Mass %	4E Recovery (%)	Cr_2O_3 Recovery (%)	Water Recovery (%)
Test 1	5.9	68.4	1.4	12.4
Test 2 (Repeat)	6.3	68.8	1.7	12.7
Test 3 (Repeat)	5.6	68.1	1.3	13.2
Average	5.9	68.4	1.5	12.8

Appendix B16: Size-by-assay of pilot plant spiral circuit samples

Table B16.1: Size-by-assay results for the densifier underflow (mill feed)

Densifier Underflow (Mill Feed)			Distribution (%)	
Screen Size (mm)	Mass (g)	Mass (%)	4E	Cr_2O_3
+106	316.2	52.7	30.4	46.1
-106+75	159.1	26.5	17.8	28.4
-75+53	73.5	12.3	14.8	15.1
-53+38	13.9	2.3	6.0	3.2
-38	37.3	6.2	31.0	7.3
Total	600.0	100.0	100.0	100.0

Table B16.2: Size-by-assay results for the cleaner heavies spiral run - middlings product

Heavies Cleaner Spiral - Middlings Product			Distribution (%)	
Screen Size (μm)	Mass (g)	Mass (%)	4E	Cr_2O_3
+106	344.0	69.2	65.4	66.3
-106+75	108.8	21.9	22.5	18.8
-75+53	28.1	5.7	4.7	5.0
-53+38	5.5	1.1	0.7	2.7
-38	10.7	2.2	6.5	7.2
Total	497.1	100.0	100.0	100.0

Table B16.3: Size-by-assay results for the cleaner heavies spiral run - heavies product

Heavies Cleaner Spiral - Heavies Product			Distribution (%)	
Screen Size (µm)	Mass (g)	Mass (%)	4E	Cr_2O_3
+106	250.0	45.8	21.0	47.7
-106+75	178.6	32.7	22.5	31.9
-75+53	75.7	13.9	19.0	13.0
-53+38	14.5	2.7	8.8	2.6
-38	27.5	5.0	28.8	4.8
Total	546.3	100.0	100.0	100.0

Appendix B17: Cost estimate for proposed milling circuit design

Table B17.1: Capital investment calculation for improved milling circuit design

Capital Investment		Units		
Spiral Circuit (Feed @ 180t/h)			R 4,500,000	Roche EDI Mining (Personal Communciation - 25/5/2011)
Chromite Ball Mill (~141kW)			R 12,000,000	Metso (Telephonic communication - 27/05/2011)
Installation cost for ball mill (3.2 x capital cost of mill)			R 38,400,000	Metso (Telephonic communication - 27/05/2011)
Subtotal			R 54,900,000	
Add: Indirect costs for engineering, construction, manpower and contingency			R 16,470,000	30% of capital (Peters and Timmerhaus, 1991)
Total			**R 71,370,000**	
Dry feed Tonnage to secondary circuit	180	t/h	4320	t/day (Lonmin - Personal Communication 24/05/2011)
Mass of concentrate (based on 5.6% laboratory rougher mass pull)			241.9	t/day
Pt rougher concentrate grade (ppm) - 75% of 4E	20.3	g/t	4898.9	g/day
Pt commodity price	1800	$/ounce	158	ounce/day
Less: Cost of Smelting and Refining (~10% of Pt price)	180	$/ounce		B.K. Loveday - (Personal Communication 8/06/2011)
Revenue from Pt sales			$256,006	per day
Forex Rate (1$ = R7.00)				
Pt revenue for Standard Circuit (Laboratory Testwork on 2010 Pilot plant sample)			R 1,792,042	per day
Pt revenue for improved milling circuit design (3.7% 4E Rec. x 0.75)	2.8	%	R 1,841,771	per day
Increased revenue (per day)			**R 49,729**	
Minimum payback period			**47**	**months**

Table B17.2: Power draw estimation for chromite ball mill

Estimation of energy requirement for chromite ball mill			
F_{80}	130	µm	Calculated from sizing data of pilot plant VHG spiral heavies stream
P_{80}	106	µm	Calculated from sizing data of milled product of pilot plant
			VHG spiral heavies stream
Work Index of chromite -rich stream (W_i)	15	kWh/t	Hay and Roy, 2010
Work (W)	1.41	kWh/t	Equation 2 and 3 (Section 2.4)
Flowrate (~55% mass flow to VHG spiral heavies stream)	100	tph	
Power	141	kW	

Appendix B18: SEM images from the SGS and UKZN cleaner heavies spiral sizing samples (silicates appear darker)

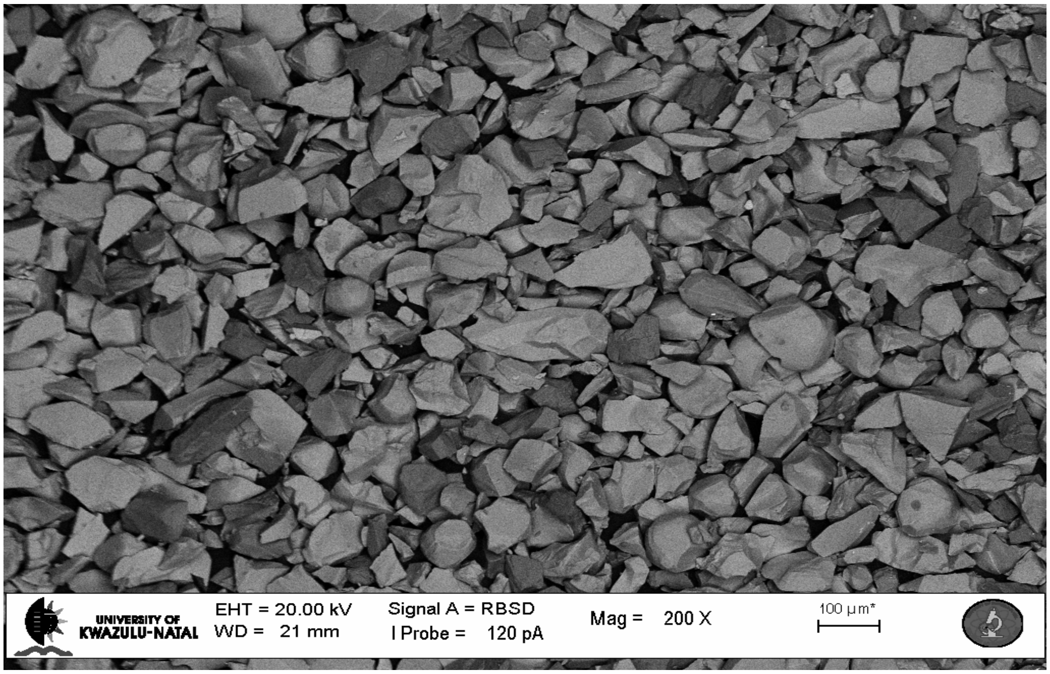


Figure B17.1: SEM image of the +53µm fraction of the pilot plant cleaner heavies spiral stream

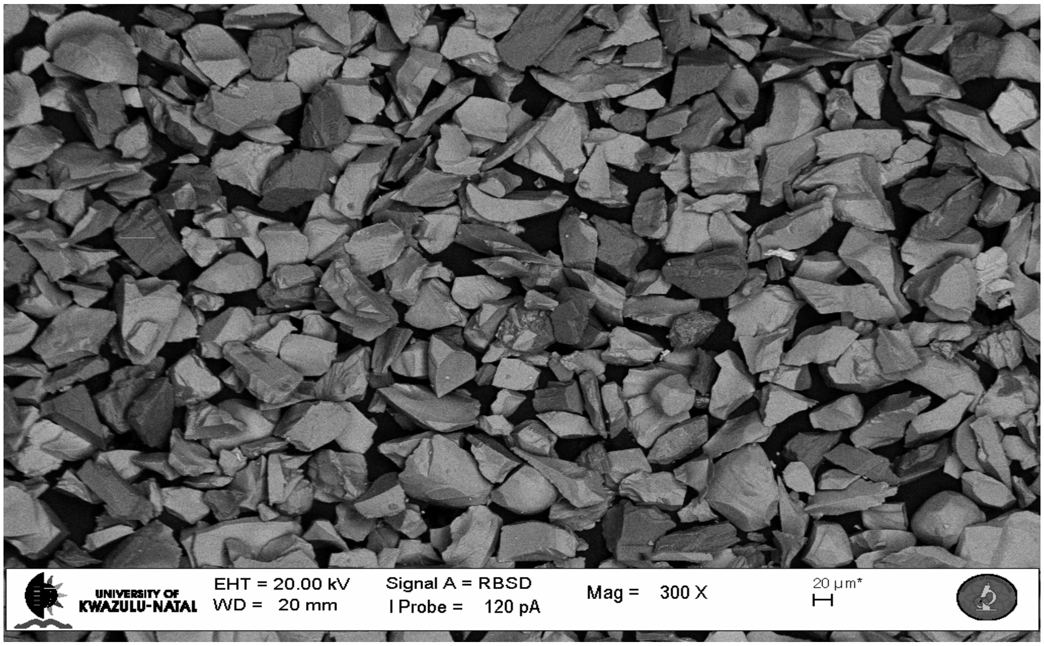


Figure B17.2: SEM image of the +53µm fraction of the pilot plant cleaner heavies spiral stream

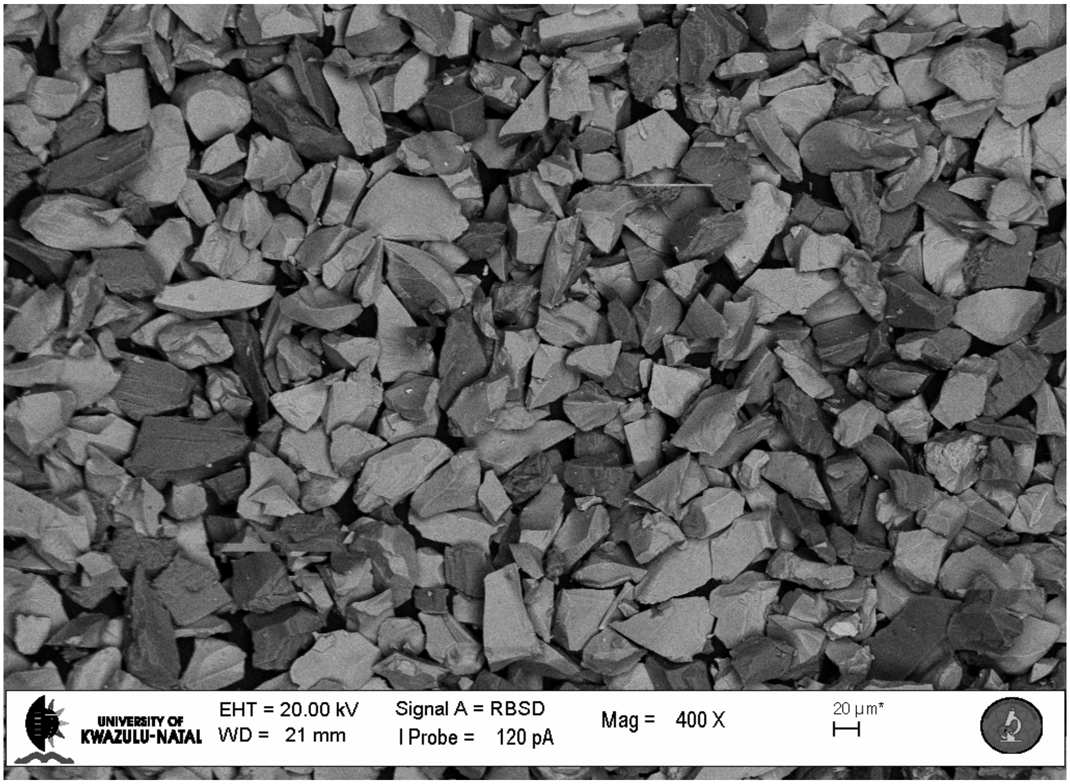


Figure B17.3: SEM image of the +38μm fraction of the pilot plant cleaner heavies spiral stream

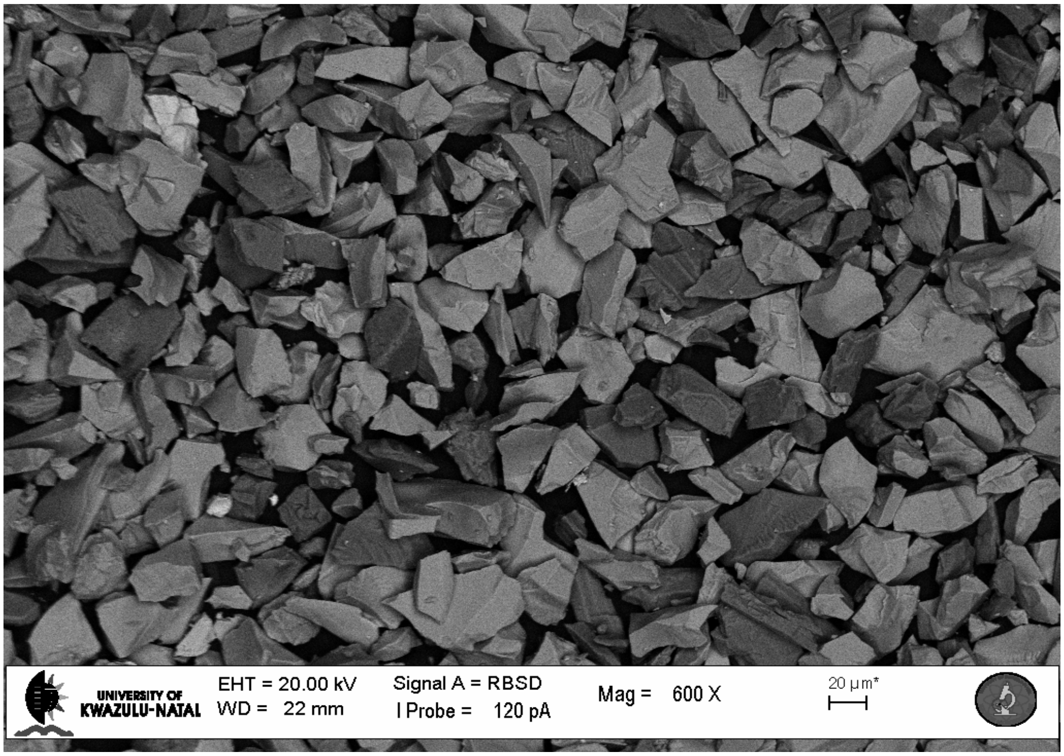


Figure B17.4: SEM image of the -38μm fraction of the pilot plant cleaner heavies spiral stream

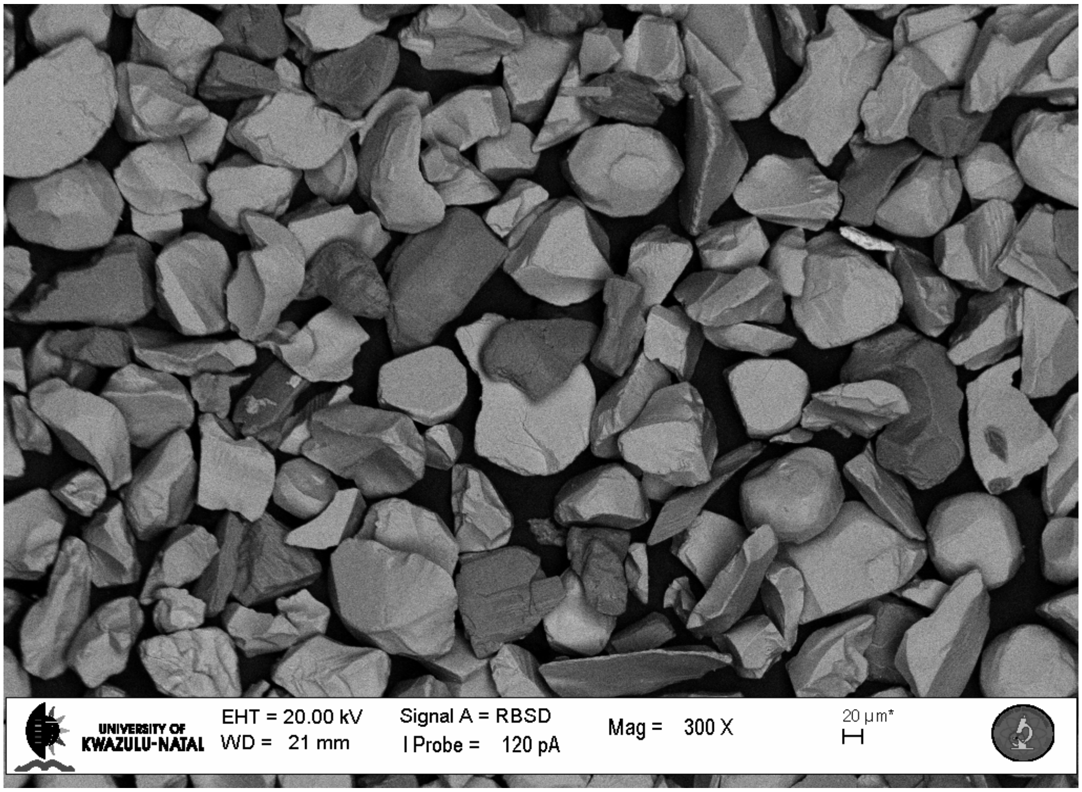


Figure B17.5: SEM image of the +75μm fraction of the UKZN cleaner heavies stream

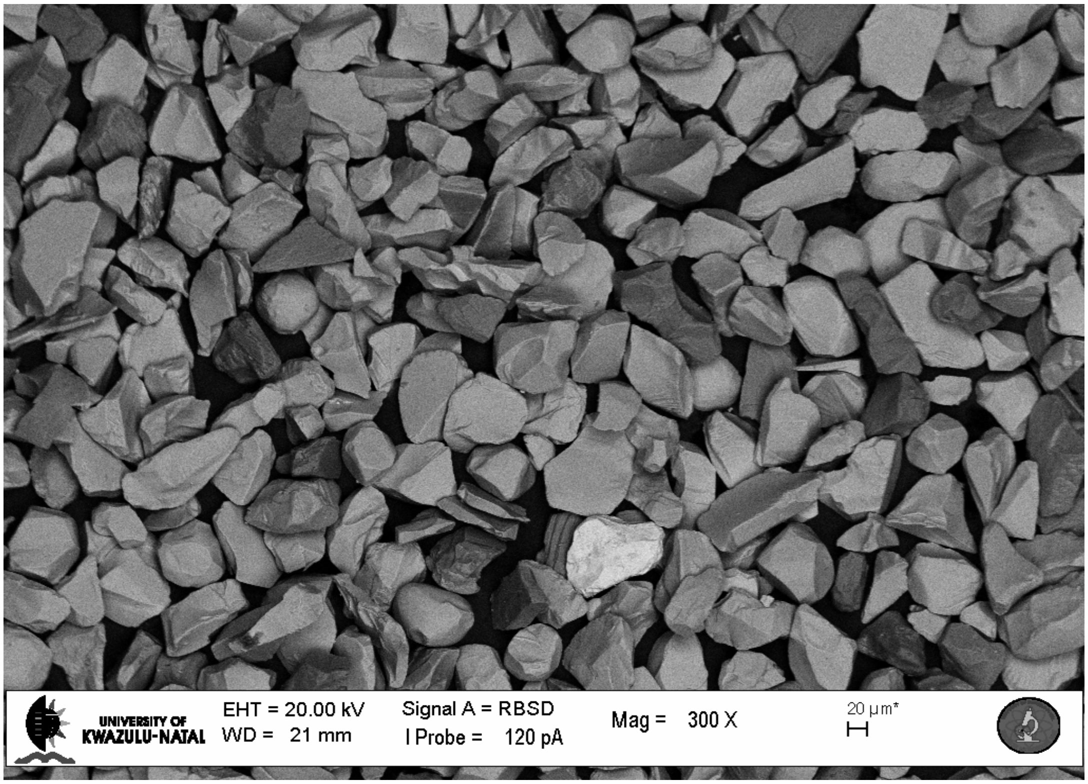


Figure B17.6: SEM image of the +53μm fraction of the UKZN cleaner heavies stream

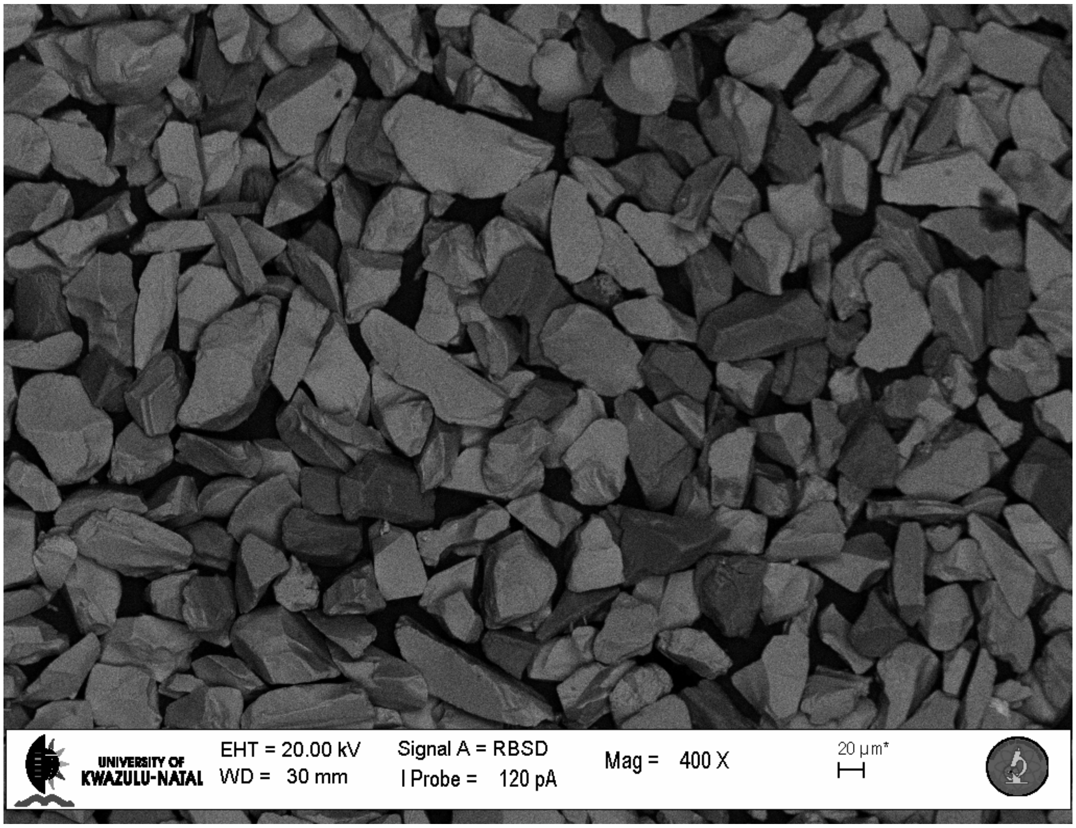


Figure B17.7: SEM image of the +38μm fraction of the UKZN cleaner heavies stream

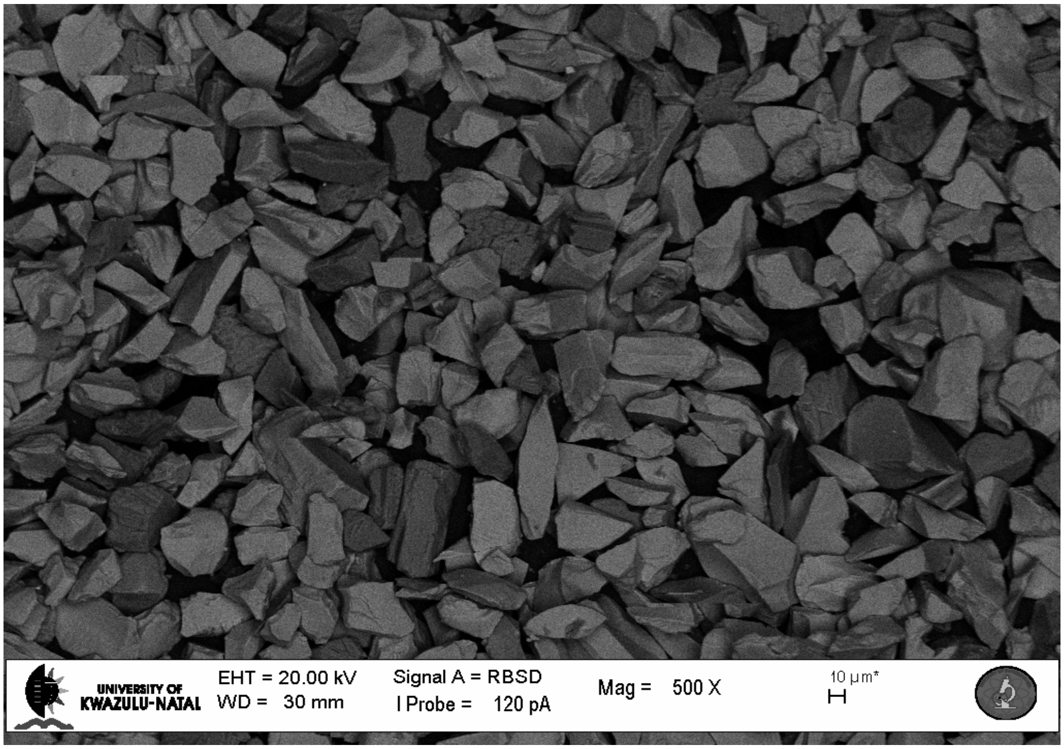
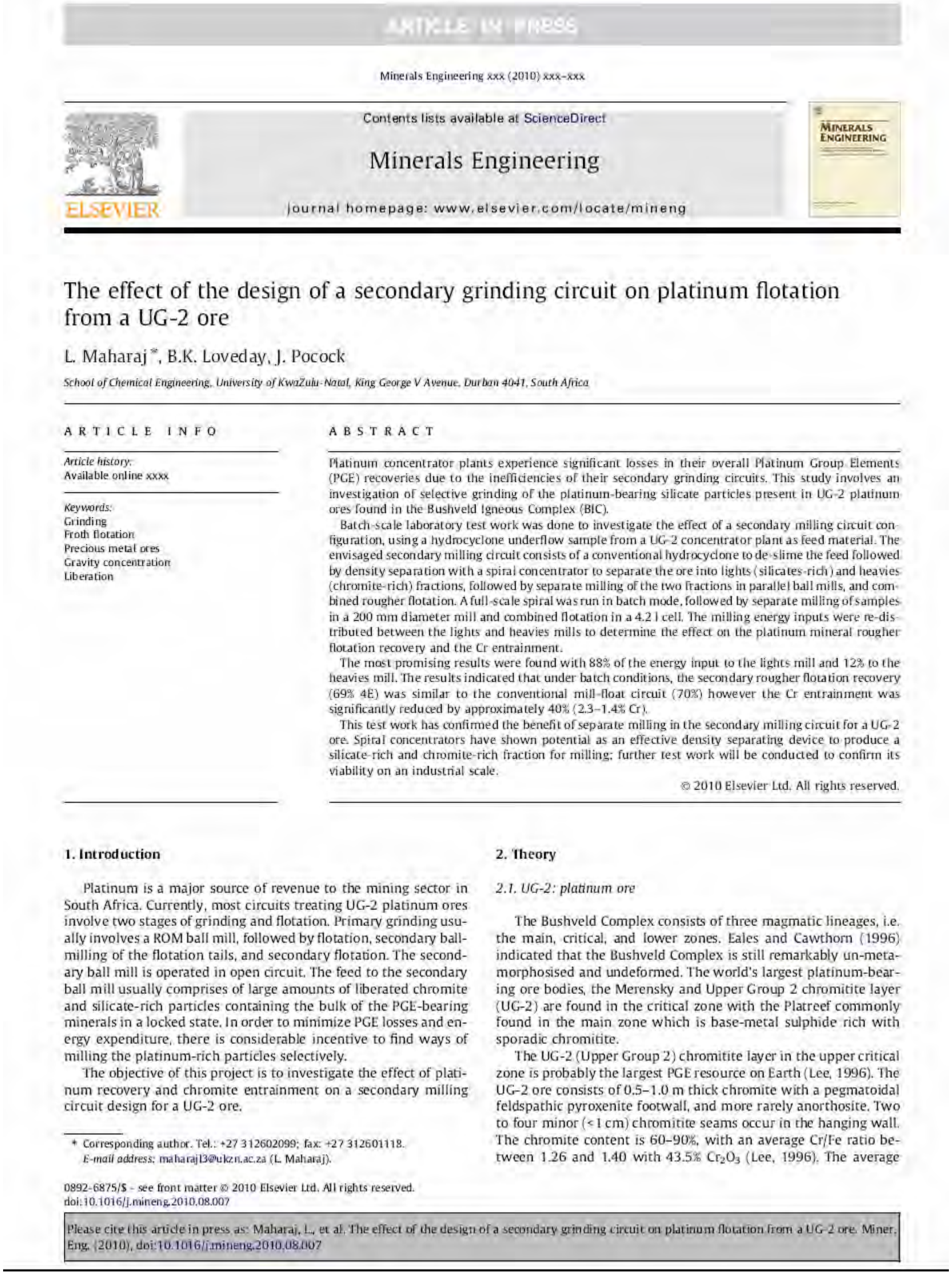


Figure B17.8: SEM image of the -38μm fraction of the UKZN cleaner heavies stream

APPENDIX C: PUBLISHED RESEARCH FROM THIS PROJECT

APPENDIX C1: JOURNAL PUBLICATION



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The effect of the design of a secondary grinding circuit on platinum flotation from a UG-2 ore

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A B S T R A C T

Platinum concentrator plants experience significant losses in their overall Platinum Group Elements (PGE) recoveries due to the inefficiencies of their secondary grinding circuits. This study involves an investigation of selective grinding of the platinum-bearing silicate particles present in UG-2 platinum ores found in the Bushveld Igneous Complex (BIC).

Batch-scale laboratory test work was done to investigate the effect of a secondary milling circuit configuration, using a hydrocyclone underflow sample from a UG-2 concentrator plant as feed material. The envisaged secondary milling circuit consists of a conventional hydrocyclone to de-slime the feed followed by density separation with a spiral concentrator to separate the ore into lights (silicates-rich) and heavies (chromite-rich) fractions, followed by separate milling of the two fractions in parallel ball mills, and combined rougher flotation. A full-scale spiral was run in batch mode, followed by separate milling of samples in a 200 mm diameter mill and combined flotation in a 4.2 l cell. The milling energy inputs were re-distributed between the lights and heavies mills to determine the effect on the platinum mineral rougher flotation recovery and the Cr entrainment.

The most promising results were found with 88% of the energy input to the lights mill and 12% to the heavies mill. The results indicated that under batch conditions, the secondary rougher flotation recovery (69% 4E) was similar to the conventional mill-float circuit (70%) however the Cr entrainment was significantly reduced by approximately 40% (2.3–1.4% Cr).

This test work has confirmed the benefit of separate milling in the secondary milling circuit for a UG-2 ore. Spiral concentrators have shown potential as an effective density separating device to produce a silicate-rich and chromite-rich fraction for milling; further test work will be conducted to confirm its viability on an industrial scale.

1. Introduction

Platinum is a major source of revenue to the mining sector in South Africa. Currently, most circuits treating UG-2 platinum ores involve two stages of grinding and flotation. Primary grinding usually involves a ROM ball mill, followed by flotation, secondary ball-milling of the flotation tails, and secondary flotation. The secondary ball mill is operated in open circuit. The feed to the secondary ball mill usually comprises of large amounts of liberated chromite and silicate-rich particles containing the bulk of the PGE-bearing minerals in a locked state. In order to minimize PGE losses and energy expenditure, there is considerable incentive to find ways of milling the platinum-rich particles selectively.

The objective of this project is to investigate the effect of platinum recovery and chromite entrainment on a secondary milling circuit design for a UG-2 ore.

2. Theory

2.1. UG-2: platinum ore

The Bushveld Complex consists of three magmatic lineages, i.e. the main, critical, and lower zones. Eales and Cawthorn (1996) indicated that the Bushveld Complex is still remarkably un-metamorphosised and undeformed. The world's largest platinum-bearing ore bodies, the Merensky and Upper Group 2 chromitite layer (UG-2) are found in the critical zone with the Platreef commonly found in the main zone which is base-metal sulphide rich with sporadic chromitite.

The UG-2 (Upper Group 2) chromitite layer in the upper critical zone is probably the largest PGE resource on Earth (Lee, 1996). The UG-2 ore consists of 0.5–1.0 m thick chromite with a pegmatoidal feldspathic pyroxenite footwall, and more rarely anorthosite. Two to four minor (<1 cm) chromitite seams occur in the hanging wall. The chromite content is 60–90%, with an average Cr/Fe ratio between 1.26 and 1.40 with 43.5% Cr_2O_3 (Lee, 1996). The average

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PGE grain sizes are found to be 9.3 µm in diameter. Most of the PGE are attached to the base-metal sulphide grains (pyrite, pentlandite, chalcopyrite, and pyrrhotite) and the PGE may be considered sulphide associated (Lee, 1996). The PGE are invariably interstitial to the chromite grains, and the only PGE commonly enclosed by chromite is laurite throughout the UG-2 ore (Lee, 1996).

2.2. UG-2: ore secondary milling circuit designs

Mintek have assisted Lonmin with the implementation of the MF2 (mill-float-mill-float) circuit and thereafter with the conversion of UG-2 milling circuits to open circuit to reduce the high circulation of chromite to the secondary ball mill and to prevent the silicates from being displaced (Deeplaul and Bryson, 2004). The Mintek secondary milling open-circuit design is shown in Fig. 1. This circuit includes a thickener unit which is fed with primary rougher tailings and hydrocyclone overflow material and thus reduces the amount of coarse silicates that may short circuit through the hydrocyclone overflow and bypass the secondary milling process.

This configuration was shown to make significant inroads to reducing the presence of coarse unliberated silicates which contain majority of the platinum-bearing minerals.

Anglo Platinum have also addressed the chromite entrainment problem and poor liberation of the coarse silicates (Rule and Anyimadu, 2007). They currently employ separate grinding and flotation for their silicate and chromite streams prior to millings at most of their existing concentrators, which are separated using conventional hydrocyclones. They also have a few concentrators (e.g. Ivan and Mortimer) with spiral circuits installed between the primary and secondary milling circuits for the removal and discard of chromite.

Fig. 2 shows a diagram of the Anglo Platinum secondary milling circuit for their UG-2 ore concentrator.

This circuit design indicated that separate milling of the silicates and chromite is necessary to overcome the problem. A 1.67 MW ball mill was used for the silicates re-grind and a combined 1.25 MW power draw for the chromite re-grinds. This was based on mineralogy and assay by size data and showed that a larger proportion of the energy should be distributed to the silicates re-grind to liberate the locked platinum bearing particles. Separate flotation was also conducted as the flotation feed particle size and surface chemistry for silicates and chromite differs.

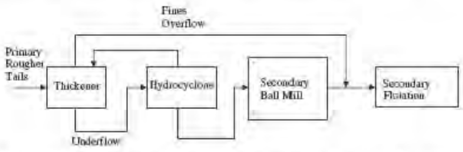

Fig. 1. Mintek secondary milling circuit –open circuit operation for UG2-ore (Deeplaul and Bryson, 2004).

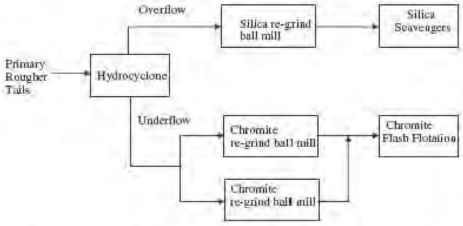

Fig. 2. Overview of a UG-2 operation at Anglo Platinum (Rule and Anyimadu, 2007).

A hydrocyclone was used as separating device however due to the large density difference between the chromite particles (4.5 SG) and silicate particles (~2.6 SG) with associated PGE's, the hydrocylone may provide a poor split with the coarse silicate reporting to the overflow and the fine chromite reporting to the underflow for flotation.

The consequences of such a split are that if the cyclone overflow stream (flotation feed) contains coarse silicates with locked PGEs, a loss in valuable mineral recovery will be noted (Mainza et al., 2004). The fine chromite that is present in the underflow stream is sent back to the grinding circuit resulting in over-grinding of barren chromite (Becker et al., 2008). This can result in a build-up of a high circulating load and loss in fresh feed capacity due to the mill inefficiency and therefore an efficient separating device is required to produce a relatively pure chromite and silicate products for separate milling.

2.3. The use of a three product cyclone as a separating device to improve secondary mill performance

Becker et al. (2008) quantified the influence of classification with a three product cyclone on liberation and recovery of PGEs in UG-2 ore. The three product cyclone has an additional vortex finder concentric in the hydrocyclone resulting in two overflows, i.e. an inner and outer overflow, with one of the overflows comprising of middlings and other fines.

The silica and chromite splits to the product streams show that the three product cyclone can be used to recover coarse silica through screening the middlings product which contains 24% and 19% of the +100 µm silica for normal and reverse classification respectively (Mainza et al., 2004).

The outer overflow is comprised of coarse silicate and the fine chromite fraction which are screened at 100 µm with a Pansep screen. The coarse silicates report to the screen oversize and the fine chromite reports to the screen undersize where it is combined with the fine overflow and sent to the flotation circuit.

Pilot plant trials were performed on a Lonmin Platinum Concentrator UG-2 platinum ore sample by Mainza et al. (2004). The authors showed that the three product cyclone reduced the overall chromite content in the flotation feed stream, (55.1 wt% vs. 42.7%) with less fine chromite and an increased liberation of valuable PGEs (21–34 grains in a 100 grain mineralogical sample size) with the locked PGEs also decreased from 46 grains to 27 grains.

The efficiency of separation of the three product cyclone may be improved through the use of finer screening technology below 75 µm. The investigations carried out by Mainza et al. (2004) and Becker et al. (2008) suggested that alternative gravity separation methods should be investigated to determine if there is potential to achieve a better split of chromite and silicate particles.

3. Experimental

3.1. Circuit configuration

The envisaged milling circuit configuration is shown in Fig. 3.

The circuit incorporates the separate milling technology employed by Anglo Platinum (Rule and Anyimadu, 2007) however a spiral concentrator was used as the separating device and combined rougher flotation was conducted so as to minimize additional changes to the existing secondary flotation circuits on

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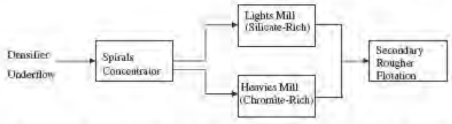

Fig. 3. Diagram of envisaged UG-2 ore secondary milling circuit configuration.

concentrator plants. An existing full scale MG-type rougher spiral concentrator was available in the School of Chemical Engineering at the University of KwaZulu-Natal and it was commissioned with a UG-2 ore to determine the operating conditions. Gulsoy and Kademli (2006) performed tests on the operational parameters of a Reichert spiral (model HG7) using a mica-feldspar ore (avg. SG = 3.4). The authors investigated feed rate, percentage solids content, and splitter opening and specified the following optimised parameters for the Reichert spiral:

- Slurry flow – 0.5–1.5 l/s.
- Solids content – 15%.
- Splitter opening – cut point at 1/3 of splitter opening (~6 cm).

These optimised parameters were found to adapt well to the separation of chromite and silicates in the UG-2 ore as it also has an average SG between 3.2 and 3.4.

3.2. Feed sample preparation

A 20 kg hydrocyclone underflow sample from a UG-2 platinum ore concentrator was run on the spiral concentrator at optimised laboratory settings determined from a series commissioning tests to produce a lights (silicate-rich), and a heavies (chromite-rich) fraction. The hydrocyclone underflow has a 60% passing size of 150 μm with a top size of 425 μm. The spiral had a mass split of 45% to the lights and 55% to the heavies with a platinum 4E analysis revealing that 65% of the platinum group minerals reported to the lights (silicate-rich) fraction. For simplicity it was decided to use a 50:50 split of the spiral products when conducting the batch milling tests.

Previous test work conducted by Maharaj et al. (2009) on a synthetic ore mixture of limestone and ilmenite indicated that a ratio of 80% ilmenite and 20% limestone by volume may influence the preferential grinding of platinum-rich particles during batch ball milling. It was therefore decided to test the proposed ball milling circuit on the 20% lights fraction and a 80% heavies fraction feed ratio as well. In order to ensure that the process remains at steady state, remaining material which constitutes 80% lights and 20% heavies were fed to the second mill for this option. The feed material for these tests was prepared by running the UG-2 hydrocyclone underflow sample on the spiral concentrator to produce a lights and heavies fraction. The spiral products were then combined in the correct mass proportions to achieve the desired volumetric ratios for the separate milling tests.

3.3. Experimental procedure

Separate batch ball milling tests were conducted on the lights and the heavies fractions keeping the overall energy input to the system constant. Sixteen minutes of milling time was required to mill a 500 g hydrocyclone underflow feed to 80% passing 75 μm using a single ball mill under conventional open circuit milling conditions. The separate milling tests were therefore conducted on a fixed mill feed of 250 g to each mill and energy was re-distributed between the lights and heavies mill based on the variation in the milling times shown in Table 1.

Table 1
Milling time.

Milling time (min)		Total time (min)	Energy distribution (%)	
Lights (silicate-rich)	Heavies (chromite-rich)		Lights	Heavies
16	0	16	100	0
14	2	16	88	12
12	4	16	75	25
10	6	16	63	37
8	8	16	50	50
6	10	16	37	63

The milled products were then combined to form a 500 g sample for the flotation tests. Flotation tests were conducted in a Denver flotation machine with an impeller speed of 700 rpm and conditioned for 3 min. A simple reagent suite consisting of a standard sulphide mineral collector (300 g/t SIBX) and a frother (40 g/t XP200) were used for the rougher flotation tests at a natural pH. Analyses were carried out on the flotation concentrate and tailings for 4E (Pt, Pd, Rh, Au) by a fire assay method and Cr by ICP.

4. Results and discussion

The conventional process route of open circuit ball milling of the secondary hydrocyclone underflow stream (feed to secondary mill) followed by rougher flotation resulted in a 70% recovery of 4E platinum minerals and a 2.3% Cr recovery under batch conditions. This base case recovery data must be considered when evaluating the new circuit incorporating a spirals separator to produce a lights fraction (silicate-rich) and a heavies fraction (chromite-rich) followed by separate milling of the spiral lights and heavies using two ball mills at varied milling times. The total energy input to the milling circuit was fixed (16 min) as per the conventional process route.

The flotation results for the effect of separate milling of the spiral product mixtures and for the 100% spiral lights and heavies products are shown in Tables 2 and 3 respectively. A sample of the data has been repeated to test the reproducibility of results and the 4E recoveries have been found to be within a 4% variance with the Cr recoveries having a variance of between 5% and 7%.

The recoveries of the 4E and Cr have been plotted as a function of the energy input to the lights fractions as shown in Figs. 4 and 5.

The 4E platinum mineral recoveries in Fig. 4 show no improvement in the base case recovery (70% 4E) however it is evident that separate milling of the pure lights (100%) and heavies followed by combined flotation is a better option than combining the spiral products into varying feed ratios prior to separate grinding. The 4E recoveries are more than 2–3% higher for majority of the tests with an increase in the lights mill energy input.

In Fig. 5, the Cr recoveries are compared as a function of the energy input to the lights mill. Separate milling shows real benefit as the Cr recovery is significantly reduced from 2.3% to 1.4% as the

Table 2
Flotation performance on the separate milling of the spiral product mixtures.

Energy distribution during milling (%)		Masspull (%)	Recovery (%)	
80% lights, 20% heavies	80% heavies, 20% lights	Concentrate	4E	Cr
88	12	7.4	69.6	1.9
75	25	8.6	59.2	2.6
63	37	8.1	63.5	2.4
50	50	7.0	64.7	1.8
37	63	7.0	65.6	1.9

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Table 3
Flotation performance on the separate milling of the 100% spiral lights and heavies products.

Energy distribution during milling (%)		Masspull (%)	Recovery (%)	
100% lights	100% heavies	Concentrate	4E	Cr
100	0	8.0	67.6	1.5
88	12	7.5	69.0	1.4
75	25	7.7	68.2	1.7
63	37	8.0	68.9	2.1
50	50	8.1	68.6	2.4
37	63	7.7	67.0	2.3

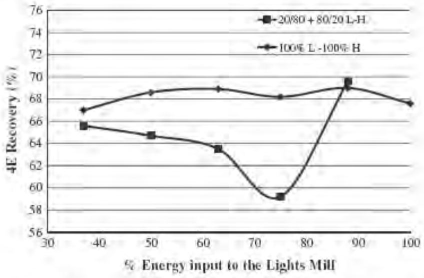

Fig. 4. Graph of 4E recoveries as a function of the energy distribution to the lights mill.

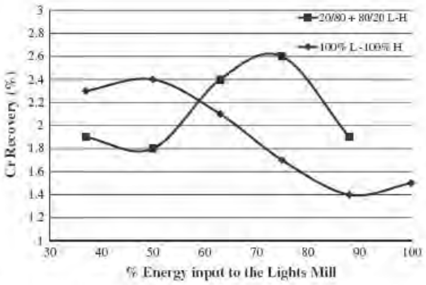

Fig. 5. Graph of Cr recoveries as function of the energy distribution to the lights mill.

lights energy input is increased to 88% of the total energy input. Most significantly the 4E recoveries are quite constant at approximately 69% however a 40% reduction in the Cr entrainment was achieved.

The results have also indicated that the heavies fraction may not need a large proportion of grinding energy as the 4E recovery was 67.6% for the 100% energy input to the lights mill, as compared to 69% for the 88% energy input to the lights mill, i.e. only 12% energy addition to the heavies fraction for a further 1.4% 4E recovery and thus alternative technologies such as attritioning may be explored in this regard.

The results also confirm the benefit of the implementation of separating ball milling of the silicates and chromite fractions by Anglo Platinum in their secondary circuit design (Rule and Anyimadu, 2007). The most important advantage of a separate milling operation is that it allows for the silicates mill to be run in closed circuit to maximize the platinum recovery. It may also reduce the possibility of the hypothesized slurry pooling effect (short-circuiting of silicates) noted for the overflow discharge ball mills.

The significant reduction in the Cr recoveries (~40%) is due to the fact that most of the total milling energy can be re-distributed to the silicates ball mill thus reducing the energy input and thus degree of over-grinding on the chromitite fraction. The lower Cr recoveries may also be attributed to the reduction of any density interactions that may occur between the high-density chromitite and low-density silicate particles in the ball mill.

5. Conclusions

Separate milling of the silicates and chromite fractions of a UG-2 ore have shown significant benefit as a more selective secondary ball milling circuit option. The Cr recovery was significantly reduced by 40% (2.3–1.4% Cr) in the secondary rougher concentrate by incorporating a conventional hydrocyclone as a de-sliming device, followed by spiral concentrators for density separation and then separate milling of the silicates and chromite streams. A large proportion of the total milling energy was distributed to the silicates ball mill (88%) without a drop in the overall platinum mineral rougher recovery (~69% 4E).

Acknowledgements

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APPENDIX C2: CONFERENCE PUBLICATION

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DENSITY SEGREGATION AND SELECTIVE GRINDING OF A SYNTHETIC UG-2 ORE MIXTURE IN A BATCH LABORATORY BALL MILL

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Abstract

Platinum concentrator plants experience significant losses in their overall Platinum Group Elements (PGE) recoveries due to the inefficiencies of their secondary grinding and flotation processes. This project involves an investigation of selective grinding of the platinum-bearing silicate particles present in UG-2 platinum ores found in the Bushveld Igneous Complex (BIC).

The basic hypothesis is that liberated chromite grains and silicate-rich PGE-bearing particles can segregate inside conventional ball mills during secondary grinding with an overflow discharge arrangement, which allows for the formation of a slurry pool. If there is preferential segregation of the silicate-rich particles into the slurry pool, they will bypass the grinding process.

Preliminary test work was conducted in a batch-scale laboratory ball mill with a mild steel ball charge. A synthetic ore mixture consisting of ilmenite (ρ=4600kg/m^3) and limestone (ρ=2650kg/m^3) of similar work indices (hardness), was used as the feed material ("as is" and a -300µm screened feed) to simulate the high density chromite and the low density silicate particles found in UG-2 ore. A mill of 30cm in diameter and 30cm in length was used for the tests. The effects of various volumetric feed ratios of the simulated ore were investigated at a uniform ball size (27mm diameter).

The tests indicated that preferential grinding of the low density limestone material occurred under batch conditions. This was noted at an 80:20 feed ratio of high density particles (ilmenite) to low density particles (limestone).

These preliminary investigations indicated that further test work should focus on a continuous operation. The potential for new milling technology would provide a significant contribution to the ongoing research undertaken by platinum producers and large research institutes to improve the recovery of PGE minerals as an increase in the platinum recovery beyond 90% would of significant benefit.

1. INTRODUCTION

Platinum is a major source of revenue to the mining sector in South Africa. Currently, most circuits treating UG2 platinum ores involve two stages of grinding and flotation. Primary grinding usually involves a ROM ball mill, followed by flotation, secondary ball-milling of the flotation tails, and secondary flotation. The secondary ball mill is operated in open circuit. The feed to the secondary ball mill usually comprises of large amounts of liberated chromite and silicate-rich particles containing the bulk of the PGE-bearing minerals in a locked state. In order to minimise PGE losses and energy expenditure, there is considerable incentive to find ways of milling the silicate-rich particles selectively.

The objective of this project is to conduct batch laboratory ball milling on a synthetic ore mixture consisting of ilmenite and limestone to simulate the high density chromite and the low density silicate particles found in UG-2 ore to investigate the effect of density segregation during milling. The effect on various volumetric ratios of ilmenite and limestone will be considered.

2. THEORY

2.1. UG-2 Platinum Ore

The Bushveld Complex consists of three magmatic lineages, i.e. the main, critical, and lower zones. Eales and Cawthorn (1996) indicated that the Bushveld Complex is still remarkably unmetamorphosised and undeformed.

The world's largest platinum-bearing ore bodies, the Merensky and Upper Group 2 Chromite Layer (UG2) are found in the critical zone with the Platreef commonly found in the main zone which is base-metal sulphide rich with sporadic chromite.

The UG2 (Upper Group 2) chromitite layer in the upper Critical Zone is probably the largest PGE resource on Earth (Vermaak, 1985, cited by Lee, 1996). The UG2 ore consists of 0.5 – 1.0 m thick chromitite with a pegmatoidal feldspathic pyroxenite footwall, and more rarely anorthosite. Two to four minor (<1cm) chromitite seams occur in the hanging wall. The chromite content is 60-90%, with an average Cr/Fe ratio between 1.26-1.4 with 43.5% Cr_2O_3 (Gain, 1985, cited by Lee, 1996). The average PGE grain sizes are found to be 9.3μm in diameter. Although the sulphide content is low, most of the PGE are attached to the base-metal sulphide grains (pyrite, pentlandite, chalcopyrite, and pyrrhortite) so the PGE may still be considered sulphide associated (Lee, 1996).

2.2. Material Transport in Ball Mills

Material transport is one of the major factors that contribute to density segregation patterns in the mill. The different mill configurations do not always result in perfectly mixed pulp and thus result in poor grinding efficiencies. The following section discusses the effect of two typical types of mill discharge mechanisms and their effect on grinding.

Fuerstenau et al (1986) studied the material transport in ball mills by considering material flow through two types of discharge arrangements, namely a grate-or diaphragm discharge (open end) versus an overflow discharge (constricted-end) ball mill as shown in Figure 2 below.

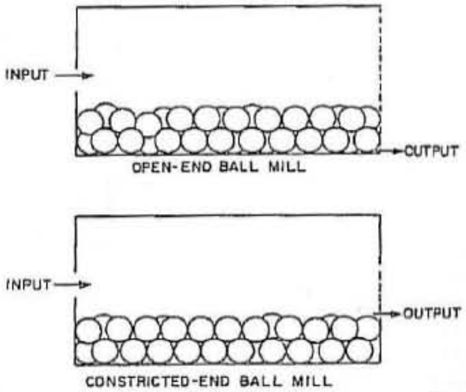

Figure 1. Diagram showing material flow in an open-end (Grate) and constricted-end (overflow) ball mill (Fuerstenau et al, 1986)

Fuerstenau et al (1986) performed investigations on the effect of feed rate, mill rotational speed, and ball filling on both configurations and concluded the following:

- Grate discharge mills are preferred in instances where a relatively coarse product is desired and where it is important to avoid extreme fines for downstream processes
- The overflow discharge arrangements have the advantage of simplicity of design and are employed for finer grinds.
- The difference in performance characteristics of these two types of mill has been attributed to steeper hold-up profiles in grate discharge mills resulting in lower retention in this type of mills.
- Very little other information is known relating to its effects on mill design using breakage kinetics and transport models.

The effect of material transport and slurry pooling (short- circuiting) are major factors which affect the selective grinding process however these effects are not feasible for batch milling.

The current study using the synthetic ore mixture will investigate the effect density segregation under batch milling conditions. The effect of a higher volumetric ratio of the low density material (limestone) or high density material (ilmenite) will be noted. This may give an indication of the degree of settling that occurs at the various ratios of ilmenite and limestone.

3. EQUIPMENT AND EXPERIMENTAL PROCEDURE

3.1. Equipment and feed data

A variable speed batch laboratory mill of 30cm in diameter and 30cm in length as shown in Figure 2 below was used for the experimental work. The length to diameter (L/D) ratio was 1.

Figure 2. Picture of the laboratory scale ball Mill

The mill specifications were fixed for all the runs and are shown in Table 1 below.

Table 1.Mill base case specifications

Mill diameter	0.30m
Mill length	0.30m
L/D ratio	1
Volumetric loading of total charge	40%
Balls to slurry volumetric ratio	50:50
Water to solids volumetric ratio	50:50
Ball size	27mm
Mill rotational speed	70rpm
Milling Time	20min

The table below represents the feed samples used for the milling tests:

Table 2.Data on feed samples

Limestone (Kulu 200C)	Ilmenite (Roaster product)
$W_i = 12.8$ kWh/t	$W_i = 12.3$ kWh/t
$\rho = 2650 kg/m^3$	$\rho = 4600$ kg/m^3

The feed and product size distributions were determined using a vibrating sieve shaker as shown by Figure 3 below.

Figure 3. Picture of a laboratory vibrating sieve shaker used to produce size distribution data

3.2. Experimental Procedure

The effects of seven volumetric feed ratios were investigated on the "as is" (un-sized) synthetic ore mixture as shown below in Table 3.

Table 3.Volumetric ratios of "as is" synthetic ore feed mixtures

Ratio	Volume % of ilmenite	Volume % of limestone
1	100	0
2	80	20
3	60	40
4	50	50
5	40	60
6	20	80
7	0	100

Table 4 shows the tests done on the -300µm ilmenite and limestone feed samples

Table 4.Volumetric ratios of "-300µm" synthetic ore feed mixtures

Ratio	Volume % of ilmenite	Volume % of limestone
1	50	50
2	80	20

The experimental procedure for the "as is" and -300µm feed samples of ilmenite and limestone was similar. In the latter tests on the screened samples, bulk feed samples of "As is" limestone and ilmenite were wet screened at 300µm, and the -300µm fraction was filtered, dried and stored for the tests.

The experimental procedure is detailed below:

- Feed size distributions for the un-milled ilmenite and limestone were conducted.
- The masses of the mill charge based on the volumetric loadings were determined i.e. grinding media (27mm steel balls), limestone, and ilmenite. The volume of water was filled into a measuring cylinder.
- The mill was loaded with the total charge.
- The mill speed was set at 70 rpm and the mill was operated for a standard time of 20 minutes
- After milling the product and grinding media were removed by using wash water.
- The product was then pressure filtered to remove the excess water.
- The filter cake sampled by a cross-cut method i.e. to ensure that a representative slice on either side of the filter cake was removed to form the sub-sample for analysis.
- The sub-sample was wet screened at 38µm.
- The +38µm and filtered -38µm samples were dried overnight in an oven.
- After drying, the samples were screened using sieves on a vibrating sieve shaker.
- The mill products were screened using root two series sieves for each run.
- A representative sub-sample (10g) from each size fraction from the mixed tests were dissolved in excess HCl (20ml) to determine the proportion of limestone and ilmenite in each size fraction.
- About 85% of the limestone was dissolved by this acid digestion technique due to the impurities present in the limestone.
- A particle size distribution for each mineral was determined by plotting the log of cumulative mass percent passing versus the log of the aperture sizes.
- These profiles were compared to determine the most effective ratio of limestone to ilmenite for the mill feed mixture.

4. RESULTS AND DISCUSSION

Sizing of the unmilled "as is" limestone and ilmenite resulted in a F_{80} (Screen size at which 80% of the feed material passes through) of 350µm and 300µm respectively. Table 5 below indicates the F_{80} and P_{80} (Screen size at which 80% of the product material passes through) for the " as is" ilmenite and limestone samples that were milled for 20 minutes under standard milling conditions as defined in Table 1. The feed sample ratio in Table 5 is represented in terms of volume percentage limestone added to the mill.

Table 5.Effect of varying ore ratios on the "as is" feed mixtures

Feed Sample Ratio	Limestone P_{80} (µm)	Ilmenite P_{80} (µm)
0% Limestone	-	210
20% Limestone	141	197
40% Limestone	144	208
50% Limestone	169	169
60% Limestone	177	177
80% Limestone	147	239
100% Limestone	155	-

The P_{80} for pure (100%) ilmenite and limestone was 210 and 155mm respectively. Although the limestone was slightly coarser (F_{80} =350µm), the limestone had a lower P_{80} after grinding although its work index indicated that it is marginally harder. The hardness of the ore however did not vary greatly as noted by the work indices in Table 2, and therefore it's mainly the densities that influence the grinding process. Figure 3 below illustrates a logarithmic plot of the data above.

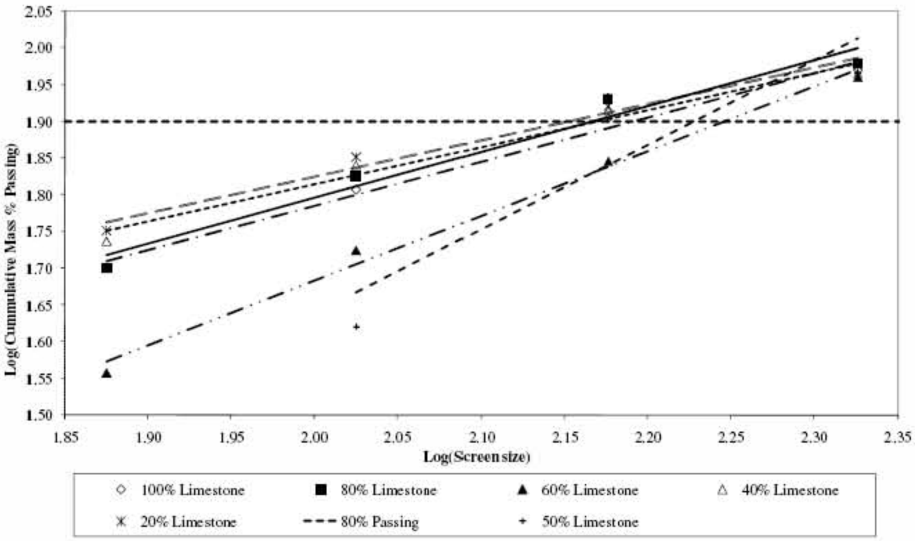

Figure 3. Graph of limestone product distribution for "as is" feed mixture ratios

The effect of varying ratios of limestone to ilmenite indicated that a lower ratio of limestone (20%) and a high ratio of ilmenite (80%) produced the best results. At a ratio of 50% volume of each ore, the grindability is the same indicating that the hardness was similar.

It was expected that the high density material would be more prevalent in the grinding zone as it would slide down the mill and receive additional impact forces. The P_{80} of ilmenite is at its lowest value at this ratio (20:80 limestone to ilmenite) as well which does agree with that hypothesis, so perhaps the lower density limestone mixes well with the ilmenite and also predominates in the grinding zone. This is rather counter intuitive to what would be expected from basic understanding of density effects however after conducting the similar tests on the -300μm ilmenite and limestone samples, a similar trend was noted. Figure 4 below illustrates the similar effect on the limestone grind.

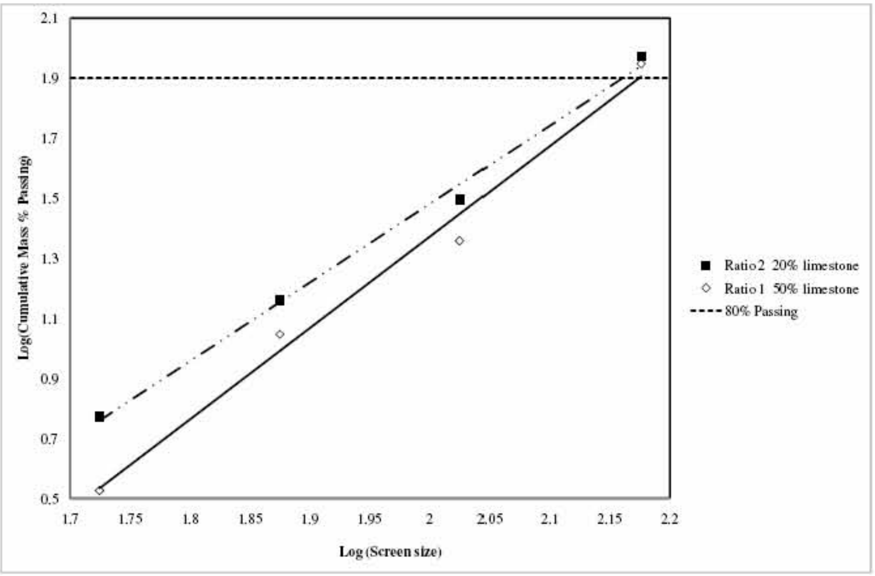

Figure 4. Graph of limestone product distribution for -300μm feed mixture ratios

Although the change in P_{80} is marginal, it does favour the 20% limestone mixture for both the screened and "as is" (unscreened) feed ratio tests. Table 6 below shows the actual P_{80} values for these -300μm tests.

Table 6.Effect of varying ore ratios on the -300μm feed mixtures

Feed sample ratio	Limestone (P_{80})	Ilmenite (P_{80})
20% limestone	138	140
50% limestone	141	144

The 20% limestone and 80% ilmenite does appear to be the best ratio to promote selective grinding of limestone (i.e. silicates) however since the difference is quite small, further test work is necessary.

5. CONCLUSIONS

These preliminary investigations do indicate that selective grinding of the less dense particles is occurring to some extent as the ratio of the low density material is reduced i.e. at a 20% limestone and 80% ilmenite feed ratio. Future test work is required to confirm these findings. These tests will focus on slurry pooling (short-circuiting effect) and other effects of material transport based on a continuous milling operation.

6. ACKNOWLEDGEMENTS

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