

AN INVESTIGATION INTO THE EFFECTS OF UG2 ORE VARIABILITY ON FROTH FLOTATION

N.V. Ramlall, BSc (Eng), MSc (Eng), UKZN

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Doctor of Philosophy in the
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PREFACE

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ABSTRACT

South Africa is the world's largest producer of platinum group elements (PGEs). Mining takes place in the Bushveld Complex, and recent statistics, (Mudd, 2010), showed that the UG2 reef is the main source of production, accounting for approximately 60% of world mining production. However, recovery by flotation is complicated by variations in the mineral composition, the need to grind fine and entrainment of chromite, which has an adverse effect on the subsequent smelting of the concentrate. The recovery of PGEs is variable, and it is influenced by PGE feed properties such as degree of liberation, mineral type and grain size. Conventional rougher batch flotation tests on drill core samples do not provide sufficient information for predicting plant performance.

The aim of this research was to develop a rigorous method for the testing of UG2 drill core samples. A rougher-cleaner flotation test procedure was developed, and statistical tests were applied to select an appropriate model, which included entrainment of hydrophilic minerals. Fifty UG2 samples from across the Bushveld were milled at a fixed energy input, and the new test procedure was applied to derive model parameters for all samples. There was a significant variability in the PGE recovery, and typical feed characteristics such as PGE feed grade and grind did not show a clear link to the PGE recovery. This was due to the complex mineralogy of the PGE minerals and variations in ore hardness. Hence, a statistical modelling algorithm was used to determine the factors affecting PGE recovery, and an empirical model was developed, which relates the PGE recovery to feed properties. The model can be used to estimate PGE recovery based on feed properties. Samples which had a high base metal content (e.g. high nickel to iron ratio) had a high PGE recovery, and samples which were altered (e.g. high Rb/Sr ratio and loss on ignition) had a low PGE recovery.

Depressant addition is used in PGE flotation to control the recovery of gangue, but it also affects the flotation of composite PGE/gangue particles. Seven of the fifty UG2 samples were selected for a more detailed investigation, using a more advanced batch flotation test and a mineralogical liberation analysis. The advanced batch flotation test was a new development, in which flotation model parameters were derived simultaneously for flotation after two stages of grinding and a combined cleaning stage. The effect of a range of depressant additions was also modelled. The floatable PGE fraction, determined from batch modelling,

was linked to the mineralogical liberation analysis of the feed. The model is the first of its kind, and it makes it possible to predict the mineralogical characteristics of the feed from flotation data.

A spread-sheet simulator was developed, to demonstrate how batch data (from the advanced flotation test) could be used to predict plant performance. Scale-up parameters were derived by using pilot-plant data for one of the ores. The spread-sheet was then used to optimise the plant design and depressant addition for an ore, while constraining, the mass of concentrate and the chromite content. The example showed that there was an optimum depressant addition and rougher-cleaner volume capacity for an ore.

The gambit of this study was the linking of feed chemical assay and mineralogical properties to PGE recovery. The application of mineralogical tests and modelling of data from the advanced flotation test has demonstrated that the link is relatively complex.

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NOMENCLATURE

SYMBOL	DESCRIPTION	UNITS
A_{cell}	Cross sectional area of cell	m^2
β	Froth stability parameter	min^{-1}
d_i	Depressant dosage for cleaner test i	g/t
d_p	Data points	–
ENT	Degree of entrainment	–
FH	Froth height	m
FRT	Froth residence time	min
η	Non-draining fraction in froth	–
k	Flotation rate constant	min^{-1}
k_{fast}	Species fast floating rate constant	min^{-1}
k_{max}	Species maximum rate constant	min^{-1}
k_{slow}	Species slow floating rate constant	min^{-1}
m	Model parameters	m
m_{ent}	Mass fraction of entrainable species in feed to flotation bank	–
m_{fast}	Mass fraction of fast floating PGMs in feed to flotation bank	–
m_{float}	Mass fraction of floatable species in feed to flotation bank	–
m_{slow}	Mass fraction of slow floating PGMs in feed to flotation bank	–
n	Number of flotation cells in a flotation bank	–
P_1	Flotation-mineralogical model parameter for recovery of liberated PGMs	–
$P_2(d_i)$	Flotation-mineralogical model parameter for recovery of composite PGMs. Parameter is a function of depressant dosage, d_i	–
Q_{air}	Volumetric flow rate of air	$\text{m}^3 \cdot \text{hr}^{-1}$
R	Species recovery	–
R_{bank}	Species recovery from a flotation bank by flotation or entrainment	–
R_{cell}	Species recovery from a flotation cell in a bank	–
$R_{\text{c, cell}}$	Species collection zone recovery for a flotation cell in a bank	–
R_e	Species recovery by entrainment	–
R_{ent}	Species entrainable fraction	–
$R_{\text{ent, cell}}$	Species entrainment recovery for a flotation cell in a bank	–
R_{exp}	Experiment species recovery	–
$R_{\text{f, cell}}$	Species froth recovery for a flotation cell in a bank	–
R_{fast}	Species fast floating fraction	–
R_{float}	Species floatable recovery	–
$R_{\text{float, cell}}$	Species floatable recovery for a flotation cell in a bank	–
R_{max}	Species maximum recovery	–
$R_{\text{o, bank}}$	Species overall recovery from a bank	–

R_{slow}	Species slow floating fraction	–
R_{sim}	Simulated species recovery	–
R_w	Water recovery	–
$R_{w,\text{cell}}$	Water recovery for a flotation cell in a bank	–
$\dot{S}_{\text{feed}}$	Feed slurry volumetric flow rate	$\text{m}^3 \cdot \text{hr}^{-1}$
$\dot{S}_{\text{tail}}$	Tail slurry volumetric flow rate	$\text{m}^3 \cdot \text{hr}^{-1}$
t	Batch flotation time	min
V_{bank}	Effective volume of a bank	m^3
τ	Average pulp residence time for a continuous flotation bank	min
θ	Fitting parameter that describes effect of depressant dosage on floatable fraction	t/g

SUBSCRIPT	DESCRIPTION
prim	Primary rougher
sec	Secondary rougher
fast	Fast floating species
slow	Slow floating species

ABBREVIATION	DESCRIPTION
A/sCo	Acid soluble cobalt
A/sCu	Acid soluble copper
A/sFe	Acid soluble iron
AG	PGMs attached to gangue
A/sNi	Acid soluble nickel
BMS	Base metal sulphides
BSD	Backscattered electron detector
CLC	Cleaner concentrate
CLT	Cleaner tail
CSCV	Cleaner scavenger
ECD	Equivalent circular diameter
EDS	Energy dispersive X-ray spectrometer
G	PGMs locked in gangue
GOI	Gain on ignition
ICP-OES	Inductive coupled plasma spectroscopy
IRUP	Iron-rich ultramafic pegmatite
L	Liberated PGMs
LOI	Loss on ignition
MF1	Single stage mill float circuit
MF2	Staged mill–float–mill–float circuit

MoO	Mode of occurrence
MSC	Model selection criterion
MSE	Mean square error of model
P_{80}	The sieve aperture size that 80% of material passes
PGE(s)	Platinum group element(s)
2PGE+Au	Pt, Pd and associated Au assay
3PGE+Au	Pt, Pd, Rh and associated Au assay
5PGE+Au	Pt, Pd, Rh, Ru, Ir and associated Au assay
PGM(s)	Platinum group mineral(s)
PR	Primary rougher
PRC	Primary recleaner
PR-C	Primary rougher concentrate
PRF	Fast floating primary rougher
PRS	Slow floating primary rougher
r_{adj}^2	Adjusted coefficient of determination
RC	Rougher concentrate
RT	Rougher tail
RTA	Rougher tail A for duplicate batch test
RTB	Rougher tail B for duplicate batch test
SAG	PGMs attached to BMS which is associated with gangue
SC	Secondary cleaner
SEM	Scanning electron microscope
SG	PGMs associated with BMS which is enclosed in gangue
SL	PGMs attached to liberated BMS
S_p	Statistical measure for model selection
SRC	Secondary recleaner
SR-C	Secondary rougher concentrate
SR	Secondary rougher
SRF	Fast floating secondary rougher
SRS	Slow floating secondary rougher
SRT	Secondary rougher tail
SS_{res}	Sum of squared residuals
SS_{total}	Squared error of model
Total PGE+Au	Total platinum group elements and Au assay
UG2	Upper Group two chromitite reef
XRF	X-Ray florescence

CHAPTER 1. INTRODUCTION

1.1. Background to study

The rate of mining of UG2 ore has increased dramatically in the last decade, due to dwindling reserves of Merensky ore and an increase in the demand for platinum group metals. According to Mudd (2010), the UG2 reef accounts for nearly 60% of production. The new platinum mines on the Eastern Limb of the Bushveld Complex are all mining only one reef, namely the UG2 reef. However, there is a significant scatter in feed grades and the mineralogy of PGMs in the UG2 reef, and this results in a variable recovery of PGMs. Mintek has carried out extensive test work for various platinum miners across the Bushveld Complex over the years, and these studies have shown that many operations are not aware of the variability inherent in their deposit. According to Gaylard (2012), metallurgical inefficiencies can have a significant influence on platinum operations. Therefore, an additional 1% improvement in PGM recovery makes a significant difference to the valuation of the mining operation, and their ability to remain in production during difficult economic periods. The inefficiencies in PGM recovery can be related to variable feed properties.

Historical records have shown that the laboratory test methods, which were used in the past to assess drill core samples, were not adequate for making meaningful predictions of plant performance. The batch tests and modelling thereof were limited to rougher tests, with occasional cleaner tests. Some geological and mineralogical tests were done but these were not linked to flotation data. Furthermore, no attempt was made to ascertain the geological and mineralogical factors affecting PGM recovery (from flotation) for a problematic UG2 ore.

Hence, this study was initiated in order to develop a simple bench-scale test methodology, which links geology (through feed chemical assay), mineralogy and batch flotation data, so that more value is created in the feasibility phase of a project. Therefore, the objective is to develop a more structured approach to understanding UG2 ore variability, and this will be elaborated in the next section.

1.2. Thesis objectives

Figure 1 illustrates the scope of the study. The main purpose of this study is to link the variation in the recovery of PGMs to measurable characteristics of the feed, such as feed

chemical assay and mineralogy. Models will be regressed to batch flotation data as a means of characterising ore samples. The batch flotation model parameters will be used to predict and quantify the effect of ore variability on plant performance.

The key outcome of addressing the objectives is an improvement in the current knowledge base on UG2 processing, since value can be added to the feasibility phase of a mining project. Furthermore, by increasing the value at the feasibility phase, the risks involved in the important early part of the life of the mine can be reduced, by identifying regions in the deposit which are problematic with respect to metallurgy.

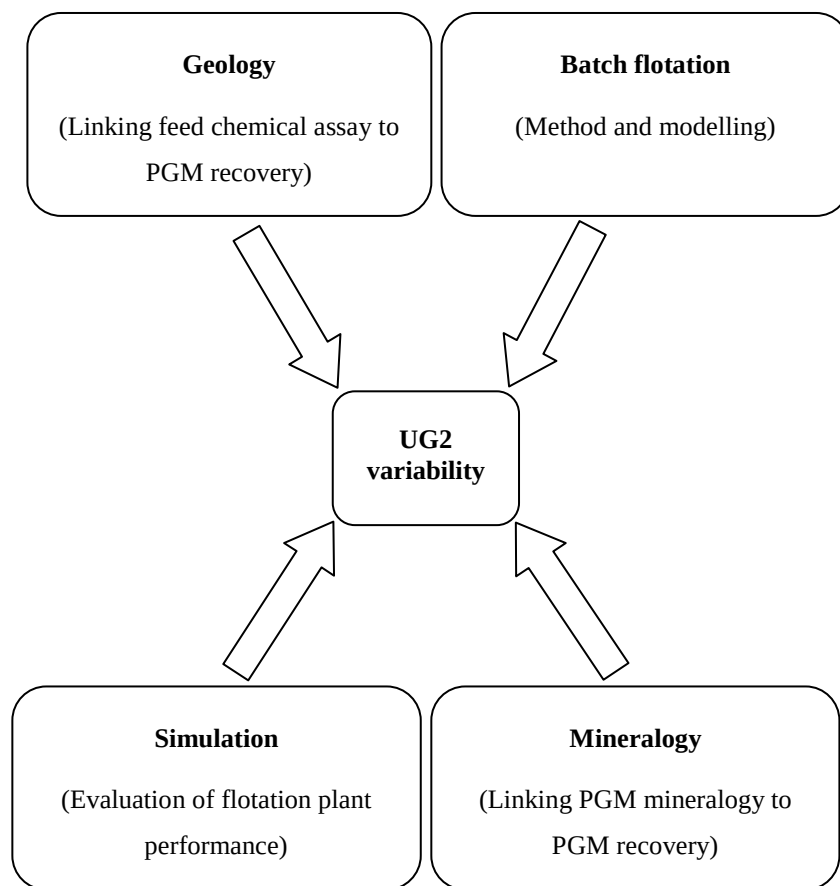


Figure 1: Illustration of scope of study

1.3. Structure of thesis

A brief discussion of each chapter will be given in the subsections to follow.

1.3.1. Chapter 2

The main purpose of this chapter is to, review literature on the influence of ore variability on project feasibility during exploration, the processing of PGMs in South Africa (mainly with

respect to the UG2 reef), and the flotation process used to recover the economic metals. The effect of ore variability on project feasibility and mine profitability is also discussed. South Africa is the foremost producer of PGEs in the world, as a result of the PGM reefs in the Bushveld Complex. The review is focused mainly on the UG2 reef, which is the subject of this study. Geological, mineralogical and metallurgical aspects are discussed. The review also covers certain aspects of flotation, particularly batch flotation, modelling of flotation and simulation of flotation plant performance.

1.3.2. Chapter 3

A generic procedure for highlighting the extent of PGM recovery variability is developed in this chapter. The procedure was designed to be simple and economical so that a large number of samples from the Bushveld Complex could be examined. Based on literature the key factors considered in the procedure were sample selection, feed chemical assay, batch flotation test method, reproducibility of the batch test and the chemical assays, and lastly, selection of the flotation models for species recovery. The term species in this study refers to PGMs, gangue and chromite. Gangue refers to all minerals which are not PGMs and chromite, and is calculated using the PGM and chromite assays, together with the mass flows. Various batch entrainment models are reviewed, and a new approach to entrainment modelling is proposed. The entrainment model, together with several batch flotation recovery models, is evaluated using statistical methods.

1.3.3. Chapter 4

The procedure formulated in Chapter 3 is applied to fifty UG2 samples selected from across the Bushveld Complex, and the results are discussed. Aside from illustrating the extent of ore variability and its effect on recovery, attempts were made to link the PGM recovery to feed chemical assays. The feed chemical assays were used as proxies for geological events, therefore, links to PGM recovery can be interpreted with respect to these events. A statistical approach called predictive modelling was used to determine the relationship between PGM recovery and feed chemical assay.

1.3.4. Chapter 5

Results from Chapter 4 indicated that feed chemical assays can be used to estimate PGM recovery. However, the mineralogical properties of the PGMs, such as liberation and mineral associations, provide a more quantitative measure of the potential recovery of PGMs.

Therefore, a smaller set of samples, which covered a spread of grade and mineralogy, were examined using additional batch flotation tests and mineralogy. The objective was to establish a link between mineralogy and flotation. A more sophisticated batch flotation test procedure is discussed in this chapter, involving two stages of grinding/rougher flotation and cleaning. Batch flotation recovery models from Chapter 3 are extended in this chapter, to include re-grinding and depressant addition (in the cleaner). The modelling approach is a new development, which considers rougher and cleaner data simultaneously. A depressant model which relates the effect of depressant addition to mineral floatability is also discussed.

1.3.5. Chapter 6

The floatable fraction of PGMs, determined from batch flotation modelling in Chapter 5, is linked to PGM liberation data, which was obtained by scanning electron microscope tests on ground ore samples. The effect of depressant on composite particle flotation is also included in the mineralogy-flotation model. The model was validated and application of the model in assessing factors affecting PGM recovery were illustrated and discussed.

1.3.6. Chapter 7

The flotation and entrainment parameters obtained in Chapter 5, for a sample called A, are used in this chapter for simulating plant performance. A simple spread-sheet flotation simulator, which uses batch data together with plant data (for calibration) was developed. Applications of the simulator are illustrated. It must be noted that the simulator is not as advanced as other simulators that are commercially available, but it does illustrate the application of a well-designed batch test in obtaining parameters for evaluating plant performance. This closes the loop in the study, by providing a means of assessing the variability observed at batch scale with respect to simulation exercises on a continuous scale.

1.3.7. Chapter 8

The key outcomes of the study is summarised in this chapter. Furthermore, all aspects shown in Figure 1 are discussed with respect to developing a tool for assessing UG2 ore variability.

1.3.8. Chapter 9

Recommendations are made in this chapter based on the outcomes presented in Chapter 8.

CHAPTER 2. LITERATURE REVIEW

2.1. Introduction

Variation in the geological process across the deposit, results in varying degrees of alteration, lithology and metal grades. A repercussion of geological variation is variability in minerals recovery and ore hardness, which affects the stability of the minerals processing circuit. The field of geometallurgy addresses the problem of ore variability. Geometallurgy attempts to link metallurgical variability such as variation in ore hardness, recovery and upgrading to intrinsic measures of the ore such as feed grade, mineralogy and lithology.

The main purpose of this chapter is to present literature, which can be used to explain ore variability. The literature is presented in three sections; the concept of ore variability, platinum processing and batch flotation testing of an ore. A discussion on ore variability is necessary to illustrate the influence on mineral processing operations and project feasibility. A review of platinum processing with respect to Merensky, UG2 and Platreef ore is essential for understanding geological factors that formed these reefs and the influence it has on beneficiation. Lastly, batch flotation tests are used to recover PGMs from a UG2 ore at laboratory scale. Therefore, a review of batch flotation tests and modelling was carried out, since this is important for developing a test method for investigating PGM recovery variability.

2.2. Significance of ore variability on project feasibility

The main purpose of exploration is to identify areas, which contain material of intrinsic economic interest. However, the form, quantity and quality of the material will determine if there exist reasonable prospects for eventual economic extraction. In addition, investors require attestation that the economic metals occur in sufficient quantity and can be economically extracted. Hence three internationally recognised reporting codes were developed:

- South African code for reporting exploration results, mineral resources and mineral reserves (SAMREC, 2009)
- Australasian Joint Ore Reserves Committee (JORC, 2004)

- National Instrument 43–101 (NI 43–101, 2010) which was developed by the Canadian Institute of Mining, Metallurgy and Petroleum

The three codes have similar requirements. Therefore, a unified code called the Committee for Mineral International Reporting Standards (CRIRSCO) was introduced. The codes are regarded as good practice and are used in the listing rules for many stock exchanges. A discussion on the common points from the codes will be presented.

Figure 2 shows a graphical illustration of the details given in the codes and presents a general relationship between exploration results and mineral resources and reserves. As mentioned earlier, exploration identifies an area, which may contain material of intrinsic interest. The area is referred to as a mineral resource. The mineral resource provides an estimation of the amount of economic metals in a deposit, however; this is subdivided into three categories based on the level of geological confidence:

- The “inferred mineral resource” refers to that part of a resource which is estimated with a low level of confidence. The reason for the low level of confidence is; inadequate geological knowledge, limited sampling data, data is uncertain or of poor quality and uncertain geological and/or grade continuity. Technical and economic studies cannot be carried out because of the low level of confidence.
- An “indicated mineral resource” provides a more reasonable estimation of the resource than an inferred mineral resource. The reason is that more geological data is available. Hence, technical and economic studies can be carried out to enable an evaluation of the economic feasibility of the resource.
- A “measured mineral resource” refers to the part of the resource which is measured with a high level of confidence. Technical and economic studies can be carried out to evaluate the economic feasibility because of the greater level of confidence for this resource.

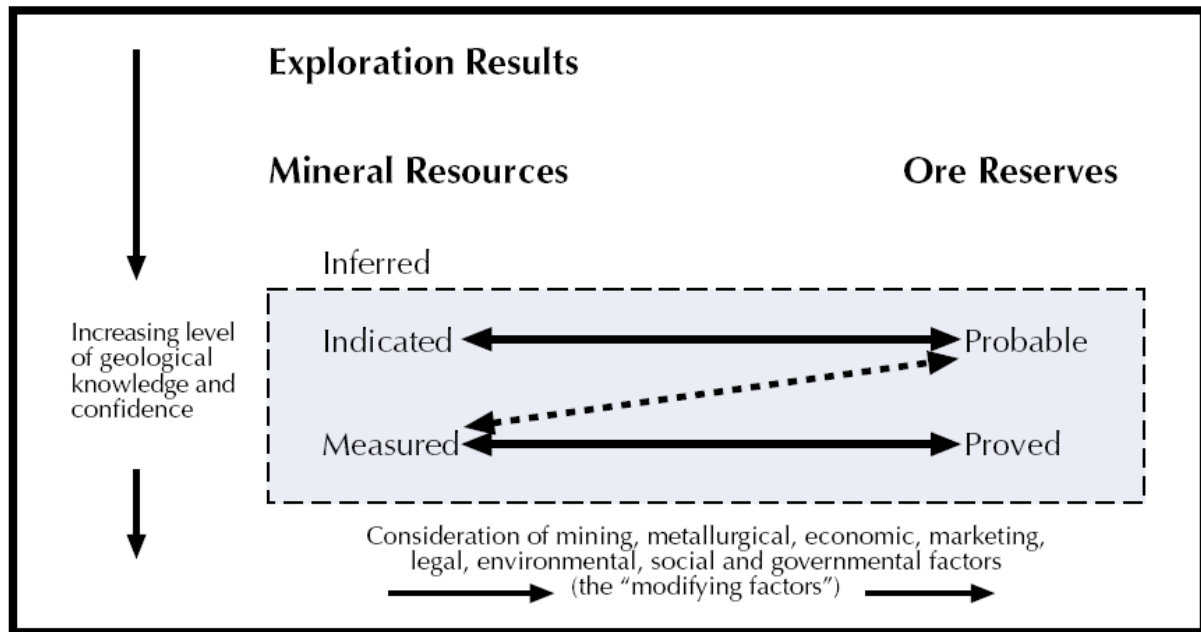


Figure 2: General relationship between exploration results, mineral resources and mineral reserves, after McKenzie (2009)

Once an area is elevated to resource status, modifying factors need to be considered in order to classify the resource into a mineral reserve. The term mineral reserve refers to the mineable part of a mineral resource, which can be indicated or measured. A mineral reserve is derived from detailed technical and economic studies, which consider realistically assumed; mining, metallurgical, economic, marketing, legal, environmental, social and governmental factors. The mineral reserve can be classified into two categories based on confidence in the estimation of the resource and modifying factors:

- A “probable mineral reserve” refers to the mineable part of an indicated and in some cases a measured resource; it considers the addition and loss of material that may occur when it is mined. Determination of a probable mineral reserve is based on appropriate technical and economic studies, which consider realistically assumed modifying factors.
- A “proved mineral reserve” refers to the mineable part of a measured mineral resource. This reserve has a higher level of confidence than the probable mineral reserve, since it is based on a resource which has a higher level of geological confidence. Furthermore, the applied modifying factors have a higher degree of confidence.

Once a deposit is elevated to reserve status, it becomes an economic entity and an asset, upon which loans and equity can be drawn for the extraction of the material. However, there are problems, which may be encountered once mining takes place. One such problem is the metallurgical factor, which may not be fully understood, when generating the bankable document for the deposit. Mineral deposits are heterogeneous and variability in the ore mineralogy and metal grades can occur across the deposit. Hence, adequate metallurgical testing of samples needs to be carried out, for determination of the expected recovery of the economic metals, with respect to feed properties such as mineralogy and metal grades.

Samples processed early in the life of the mine, are often used as a projection of future production estimates when, in fact, the material mined later in the life of the operation is significantly different. It follows that the metallurgical variability must be investigated with a high degree of confidence during the determination of the mineral reserve. If the metallurgical variability is understood sufficiently, appropriate actions can be taken so that production estimates can be met. Furthermore, understanding ore variability earlier on in a project life cycle assists in designing metallurgical circuits, which are flexible in processing the variable feed.

Many authors have noted the importance of variability on project feasibility and the need to quantify the influence on metallurgy:

- Dobby *et al.* (2002) stated that “Variability is a significant problem to address during the design phase of a flotation circuit. For existing operating plants variability needs to be clearly understood and accounted for if plant performance is to be optimised.”
- Williams and Richardson (2004) stated that “Geometallurgical mapping is a new team-based approach that documents variability within an ore body and quantifies the impact of geology (host rock, alteration and structure) and mineralogy on grinding, metallurgical response and metal recovery processes. Thus it is an important tool to reduce the technical risk associated with new mine developments or expansions.”
- Bulled and McInnes (2005) stated that “The key to successful flotation plant design, production planning and mine/mill optimisation is a solid understanding of the resource to be processed.”

- Dunham and Vann (2007) stated that “Incorrect characterisation of the metallurgical recovery/throughput can (and has) led to misspecification of the scale of the projects, and thus can be seriously value destructive.”
- Coward *et al.* (2009) stated that “The value proposition of geometallurgy is simple and compelling. By improving the understanding of the spatial nature of relevant rock properties the mining and ore treatment operations can be improved, both at the design phase and operation of mineral projects.”

2.3. Platinum processing

The review will begin with a discussion on the global platinum ore resources and reserves; this will be followed by a review on the geological setting of platinum ore in South Africa, i.e. major ore types, geology, metal grades and mineralogy. Lastly, the beneficiation process will be discussed.

2.3.1. Global resources and reserves of platinum group elements

Generally, ores that have platinum (Pt) has a major economic metal have five other metals that occur in addition to platinum, which are palladium (Pd), rhodium (Rh), ruthenium (Ru), iridium (Ir) and osmium (Os). The six metals are collectively referred to as platinum group elements (PGEs). Hoffman and Dunn (2002) discussed the geological settings of primary PGE deposits, which varies from low-grade large tonnage (Lac des Isles Complex in Canada) to high-grade, narrow reef type deposits (Merensky reef and UG2 in South Africa, Stillwater Complex in the United States and Great Dyke in Zimbabwe); and nickel-copper ores (Norilsk-Talnakh in Russia, Sudbury in Canada, Farrow, Lightfoot and Kambalda in Australia). The major reserves are in South Africa, Russia and Canada. South Africa is the world's foremost supplier of platinum, producing almost 80% of the world's platinum, as indicated by Table 1. On the other hand, Russia's Norilsk-Talnakh deposit is the largest producer of palladium.

Table 1: Pt, Pd and Rh production in 2006, after Pincock *et al.* (2008a)

PGE	South Africa/ [tonnes]	Russia/ [tonnes]	North America/ [tonnes]	Total Supply/ [tonnes]
Pt	170.0	29.0	13.3	221.0
Pd	85.0	98.4	28.4	224.0
Rh	21.8	2.8	0.5	25.6

South Africa dominates platinum and rhodium production because of a massive reserve base called the Bushveld Complex. Table 2 shows South Africa's dominance over other PGE producing countries as a result of the Bushveld Complex. The PGEs are extracted from three mineralised horizons, which are the repository for in the order of 75 to 80% of the worlds' reserves of PGEs (Merkle and McKenzie, 2002). The three mineralised horizons are the Merensky reef, Platreef and UG2 reef.

Table 2: PGE resources and reserves for primary PGE suppliers, after Pincock *et al.* (2008b)

Country	Reserves/ [tonnes]	Resources/ [tonnes]
South Africa	63 000	70 000
Russia	6200	6600
United States	900	2000
Canada	310	390
Other Countries	800	850
World Total	71 000	80 000

2.3.2. Geological setting of PGEs in South Africa

The PGEs in South Africa are mined and beneficiated from the Bushveld Complex, which is more than 2 billion years old. The Bushveld Igneous Complex was formed by a series of geological events taking place over an extensive period. Lava or magma was injected repeatedly into a sub-volcanic shallow level chamber. The cooling and crystallisation of the magma was extremely slow because of the large volume of magma involved. The slow cooling resulted in different minerals being precipitated when the temperature decreased; the minerals precipitated in a sequence determined by phase relations and the instantaneous composition of the magma. Minerals accumulated into mineralised horizons building from the base of the shallow chamber. Intermittent replenishment by hotter magma, due to repeated volcanic activity, led to repetition of the crystallisation sequence, which repeated the mineral layering (Cawthorn, 1999).

A series of geological events together with aging resulted in an irregular saucer-shaped feature, which covers 65 000 km² with thickness reaching 7 km (Cawthorn and Webb, 2000 and Schouwstra *et al.*, 2000). It is the world's largest layered intrusion and at its current levels of erosion, there are three Limbs that exist, the Eastern, Western and Northern Limbs, as shown in Figure 3. The Merensky reef and UG2 are exploited primarily in the Western and Eastern Limbs. Economically exploited PGEs in the Potgietersrus (Northern Limb) are limited to the Platreef.

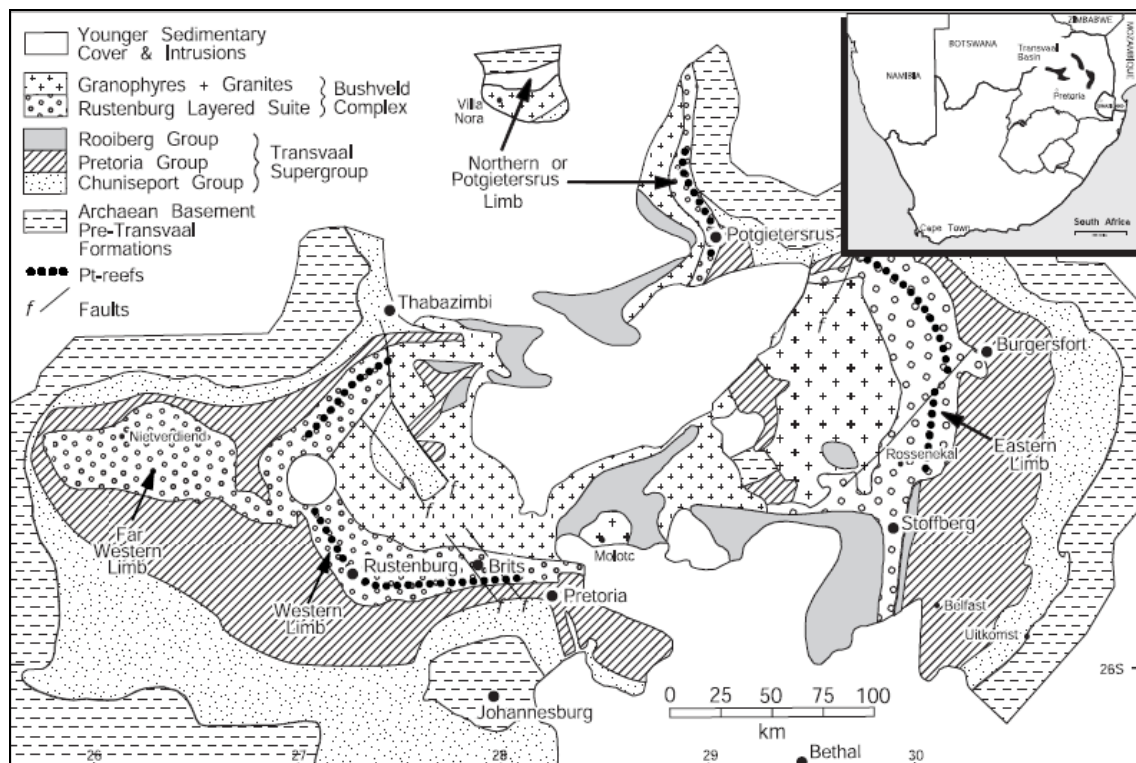


Figure 3: Geology of the Bushveld Complex, after Barnes and Maier (2002)

The Bushveld Complex is characterised by large-scale layering, which formed the basis for the subdivisions of the complex into upper, main, critical and lower zones. Figure 4 illustrates the subdivisions which maintain their individuality (i.e. laterally continuous) except for minor magmatic erosional discontinuities known as potholes. The critical zone hosts the Merensky reef and UG2 (Schouwstra *et al.*, 2000).

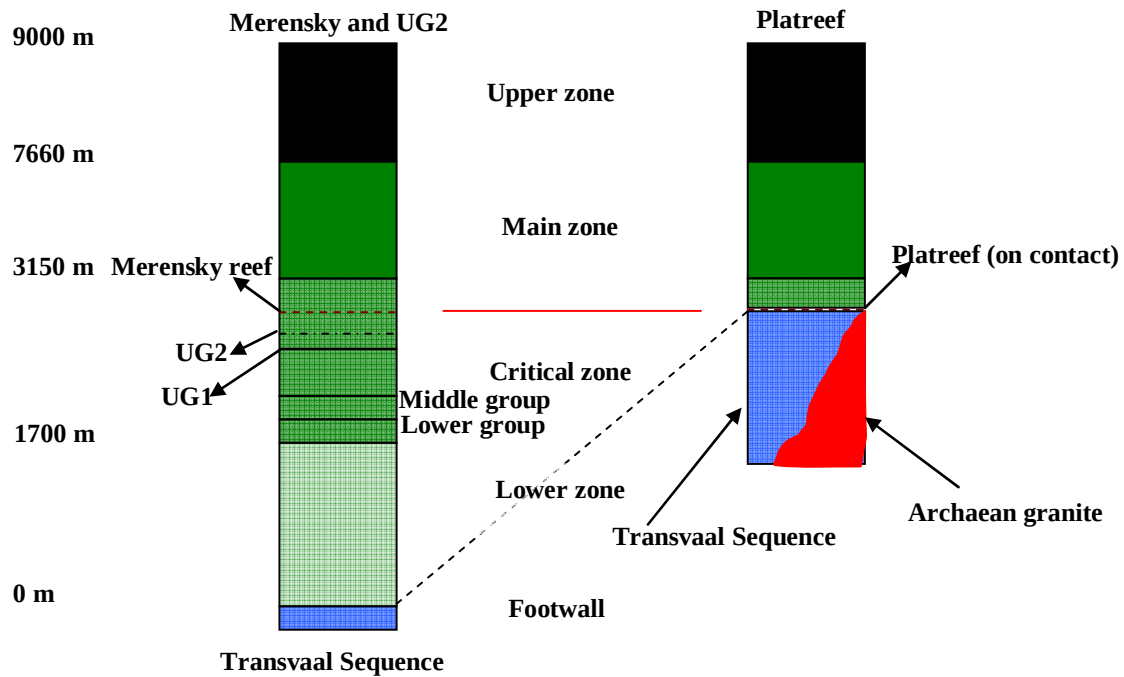


Figure 4: Stratigraphic column of the reefs, reproduced after Schouwstra *et al.* (2000). The figure illustrates the position of the UG2 reef relative to the Merensky reef; the Platreef is interpreted as the Merensky reef equivalent.

The middle and lower group are also chromitite layers, which have a lower PGE content relative to the Merensky and UG2 reef. The Platreef lies directly on the rocks of the Archaean sediments and granites, and the Merensky reef lies on the Bushveld rocks of the critical zone. In the subsections that follow the three mineralised layers will be discussed. A more detailed discussion will be given for the UG2 reef, since it is the subject of this study.

2.3.2.1. The Merensky reef

This reef was exploited since 1925 and for several decades, it was the principal source of PGEs. In 1999, the reef accounted for just over 50% of all PGE bearing ores processed in South Africa (Matthey, 2008). At any locality in the Bushveld Complex the Merensky reef is developed above the UG2 reef; the vertical distance between the two can vary from 20 m to about 350 to 400 m (Cawthorn, 1999; Schouwstra *et al.*, 2000 and Merkle and McKenzie, 2002). Although the Merensky reef is continuous, large variations in thickness, reef composition and mineralisation are present. The reef is typically less than 30 cm and can be traced for 300 km across Bushveld Complex and to depths of 5 km. The reef has a number of favourable characteristics that justified its mining and Table 3 shows some of the characteristics.

Table 3: Characteristics of Merensky reef, modified after Merkle and McKenzie (2002)

Parameter	Merensky reef
Rock type	pyroxenite
PGE content	4–10 ppm
Ni content	1300 ppm
Cu content	800 ppm
BMS content	1–10%
BMS grain size	Up to 10 mm
PGM grain size	Up to 350 μm
Density	3.2 $\text{g}\cdot\text{cm}^{-3}$
PGE Prill distribution	
Pt	59.4
Pd	25.2
Rh	3.0
Ru	8.1
Ir	1.0
Os	0.8
Au (associated element)	2.5

The main rock type for the reef is pyroxenite, and it consists predominately of orthopyroxene (60%), plagioclase feldspar (20%), pyroxene (15%), phlogopite (5%) and occasionally olivine. The reef is characterised by high concentrations of PGEs, nickel (Ni) and copper (Cu) relative to the other reefs. The Ni and Cu make up the base metal sulphides (BMS). Schouwstra et al. (2000) reported that the BMS consisted of pyrrhotite (40%), pentlandite (30%), chalcopyrite (15%) and trace amounts of millerite (NiS), troilite (FeS), pyrite (FeS₂) and cubanite (Cu₅FeS₄). The PGMs are mostly, cooperite (PtS), braggite [(Pt, Pd) NiS], sperrylite (PtAs₂), PGE alloys and in some areas laurite (RuS₂) may be abundant. The PGMs in the reef are typically associated with BMS therefore, the higher BMS content and PGM grain size, relative to the UG2 and the Platreef, results in higher PGM recovery by flotation. The Merensky reef has a lower density than the UG2, but it has a higher comminution work index. The income from base metals in the Merensky reef constitutes approximately 10% of the revenue, compared to 5% from the UG2 and 35% from the Platreef (Merkle and McKenzie, 2002).

2.3.2.2. The UG2 reef

The relatively high grade of PGEs in the UG2 reef was known for some time. However, the processing of this reef had to wait, until a suitable processing technique was available for the separation of the PGEs from the refractory host, called chromite (FeCr_2O_4). The higher PGE content and lower comminution work index, compared to the Merensky reef, made this reef an attractive resource for mining. However, this is partially offsetted by the abrasive nature of chromite and the high chromite content in the final products, which affects smelting*. The processing of the reef commenced in 1970 and by 1999, it accounted for just over 42% of all PGE bearing ores processed in South Africa (Matthey, 2008).

The reef is typically 1 m thick although, it may vary from about 0.4 to 2.5 m. It dips at an angle ranging from 5 to 70 degrees towards the centre of the Bushveld Complex (Cabri, 1981). Chromite seams may be found in the hanging wall and footwall of the main chromitite seam and are generally less than 20 cm; in some instances, these chromitite stringers may be assimilated into the mining program and fed to the flotation plant. Figure 5 shows an idealistic view of the reef typically, there is a higher concentration of PGEs and base metal sulphides at the top and bottom of the main seam, which is referred to as bottom and top loading.

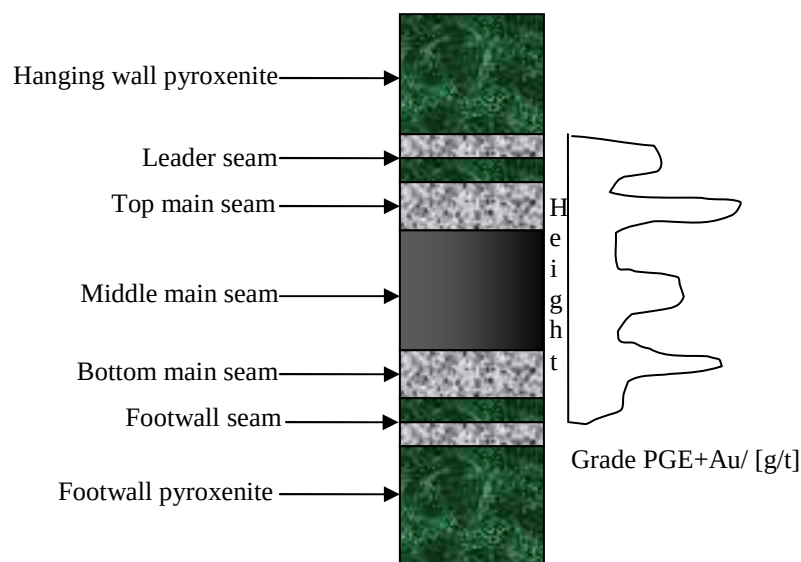


Figure 5: Idealistic view of the UG2 reef, reproduced after Schouwstra *et al.* (2000)

* Chromite is a spinel mineral which accumulates on the walls of the smelter; over time this reduces the furnace capacity which consequently results in a lower production of PGEs and other economic metals.

Table 4 shows the characteristics of the reef. The main rock type is chromitite (60–90%v/v) with minor amounts of silicate minerals, typically pyroxene (5–30%v/v) and plagioclase (1–10%v/v) (McLaren and De Villiers, 1982; Penberthy *et al.*, 2000; Merkle and McKenzie, 2002; Schouwstra *et al.*, 2000 and Cole and Ferron, 2002). Other minerals present in subordinate concentrations include phlogopite, biotite, clinopyroxene, talc, chlorite, quartz, serpentine, ilmenite, magnetite, rutile and calcite (Penberthy *et al.*, 2000). The reef has the lowest Ni and Cu content from all the reefs mined for PGEs. The PGE content is marginally higher than the Merensky reef but variable across the Bushveld Complex. The BMS consists primarily of pentlandite, chalcopyrite, pyrrhotite and to a lesser extent pyrite (Miller *et al.*, 2005). The PGMs are primarily associated with the BMS (Cabri, 1981; Merkle and McKenzie, 2002 and Bryson, 2004a). Furthermore, Mathez and Mey (2005) indicated that 90% of the PGMs occur in an interstitial assemblage consisting of base metal and PGE sulphides, sulfosalts, tellurides and bismuthides. Some of the PGEs, usually laurite and Ir–Os alloys, are found as microscopic inclusions in chromite (Von Gruenewaldt *et al.*, 1986). The PGMs, which remain enclosed in chromite after grinding are virtually non-recoverable, except by sophisticated pyrometallurgical techniques. Figure 6 shows a microscope view of the chromite, silicate and base metal sulphide assemblage in a UG2 ore.

Table 4: Characteristics of UG2, modified after Merkle and McKenzie (2002)

Parameter	UG2 chromitite
Rock type	chromite
PGE content	4–10 ppm
Ni content	700 ppm
Cu content	180 ppm
BMS content	< 1%
BMS grain size	30 μm
PGM grain size	up to 10 μm
Density	4 $\text{g}\cdot\text{cm}^{-3}$
PGE Prill distribution	
Pt	41.0
Pd	34.0
Rh	9.0
Ru	12.0
Ir	1.9
Os	1.7
Au (associated element)	0.4

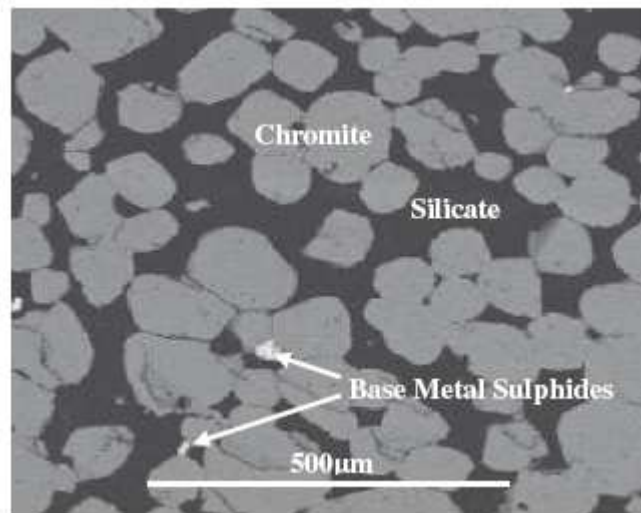


Figure 6: General view of UG2, after Hay and Roy (2010)

Penberthy (2001) discussed a number of different UG2 ore types, which formed as a result of different geological events. The different ore types are discussed briefly in the subsections below.

a. Normal or unaltered UG2

Normal UG2 ore has few signs of replacement, cataclasis⁺, recrystallisation and talcification. Furthermore, approximately 60–90%v/v of chromite occurs with interstitial orthopyroxene and plagioclase. The major base metal sulphides, which occur in minor amounts, are chalcopyrite, pyrrhotite, pyrite and pentlandite. These minerals occur at the chromite–silicate grain boundaries. The PGMs occur preferentially at the grain boundaries of the base metal sulphides with chromite and/or silicates.

b. Pegmatoid footwall UG2 chromitite

This type of UG2 is underlain by a coarse-grained felspathic pegmatoid. The pegmatoid has the same texture and mineralogy as the underlying pyroxenite, but the grain size is coarser. The chromitite layer is also affected by sintering, which is the annealing or densification of the chromite. This occurs due to injection of molten material. Sintering results in enlarging of

⁺ Cataclasis is the process where broken mineral grains are cemented together by secondary silicates like pumpellyite, epidote, albite, talc, chlorite and quartz.

the chrome grains that boarder each other and which, consequently, eliminates the interstitial silicates and PGMs.

c. UG2 affect by iron–rich ultramafic replacement pegmatoid (IRUP)

IRUP is caused when fluid containing Fe, Ti, V and Ca is introduced into the chromitite layer. This results in an increase in the chromite, sulphides and possibly PGE minerals' grain size. There is an alteration of orthopyroxene and plagioclase, which results in the formation of low temperature hydrous silicates such as amphibole, chlorite, talc, serpentine and clay. Experiments carried out by Mintek (Makhanya, 2007b) on an ore affected by IRUP showed that the naturally floatable gangue minerals produced a large mass recovery to the final concentrate (~30%). However, the PGM recovery was low (~45%). The low PGM recovery combined with a high mass recovery to concentrate was attributed to the PGM assemblage being dominated by alteration minerals such as alloys, arsenides and Pt and Pd compounds associated with Te, Bi, Sb, Hg and Pb.

d. UG2 associated with pothole structures

Potholes are circular structures, which disturb the UG2 and surrounding lithologies. An area affected by potholes has the UG2 reef displaced below the normal stratigraphic elevation. Some potholes have been found having diameters several hundred meters wide and a few meters deep (Penberthy, 2001). Mining is not carried out near potholes because of the earth being unstable near these structures.

e. UG2 associated with faulting and fracturing

The UG2 reef here is characterised by cataclasis texture and an increase in the concentration of hydrous phyllosilicate phases like talc.

f. UG2 associated with surface weathering

UG2 from areas exposed to surface weathering have supergene alteration. Samples tested from near surface at Mintek (Marais and Nisbett, 1989; Fletcher and Bryson, 2004 and McLaren and Duarte, 2004) showed that the ore is softer than normal UG2 ore and required significantly less energy to achieve the conventional grind of 80% passing 75 µm. This is an indication of surface alteration. Furthermore, the PGM recovery was low; due to the PGMs occurring as alloys and non-sulphides. Flotation experiments carried out at Mintek (Marais

and Nisbett, 1989 and Fletcher and Bryson, 2004) showed that the mass of concentrate was large, due to the presence of floatable gangue. The concentrate had a low PGM grade and when attempts were made to reduce the mass recovery, by adding more gangue depressant, the PGM recovery decreased significantly. The reason given by the authors was that the PGMs occurred mainly as composites associated with silicates therefore, when depressant was added the PGE recovery decreased.

2.3.2.3. The Platreef

The Platreef was mined since 1920, but it was not until 1993 that the reef was mined on a large scale (Matthey, 2008). The Platreef is mined in the Northern Limb of the Bushveld Complex using open cast mining because it is much thicker than other reefs (generally 5 to 90 m). The reef consists of a complex assemblage of pyroxenites, serpentinites and calc-silicates, and it differs from the other reefs in that the magma reacted with the lime rich floor rock, which resulted in the formation of abundant lime rich minerals. The base metal and PGE concentrations are variable and there is a significant amount of PGE telluride minerals, that contribute between 20 and 45% of the PGMs present in the Platreef. Shackleton *et al.* (2007) showed that the PGE tellurides are difficult to recover by flotation. Table 5 shows the characteristics of the Platreef. Since this reef has variable characteristics no specific value or range can be assigned to the BMS content, BMS grain size, PGM grain size and the ore density.

Table 5: Characteristics of Platreef, modified after Merkle and McKenzie (2002)

Parameter	Platreef
Rock type	Pyroxenite
PGE content	3 ppm
Ni content	3600 ppm
Cu content	1800 ppm
BMS content	n/a
BMS grain size	n/a
PGM grain size	n/a
Density	n/a
PGE Prill distribution	
Pt	42.1
Pd	46.1
Rh	3.0
Ru	4.0
Ir	0.8
Os	0.6
Au (associated element)	3.4

2.3.3. Beneficiation of PGE ores

As mentioned earlier, the Merensky reef was the principal source of PGEs since 1925. The beneficiation of this ore was carried out by a single-stage mill operated at a grind between 55 and 65% passing 75 μm . The PGMs in the milled product were concentrated using a simple flotation circuit, having a rougher stage followed by two stages of concentrate cleaning. This was called the mill–float circuit or MF1 circuit, refer to Figure 7.

It was realised in the 70s that there was going to be an increase in the demand for platinum to meet the expected growth in the autocatalysis industry (Deeplaul and Bryson, 2004). Furthermore, it was apparent that the dwindling Merensky reef reserves would not be able to support the demand for platinum hence; other reserves had to be mined to meet the demand. At the time, processing of the UG2 ore was not favoured because of the lower BMS content and high chromite content in the final concentrate. Mintek, in collaboration with Lonhro, carried out extensive experiments at pilot–scale to investigate a processing route for the PGMs from the UG2 ore.

The same metallurgical flow sheet used to concentrate the PGMs from the Merensky reef was used as a starting point, but it became apparent that a finer grind was required to achieve adequate liberation of the PGMs. A grind of 80% passing 75 μm was used, together with a number of cleaning stages, to produce a concentrate, which had a suitable PGM grade and chromite content (typically less than 2.5%). However, this processing route showed a significant loss in PGMs, with recovery varying between 70 and 80 furthermore; the final concentrate had high chromite content.

The MF1 circuit may also overgrind actively floating gangue minerals such as talc. Talc has the disproportionate effect of enhancing froth stability and increasing entrainment of other gangue minerals (Martinovic *et al.*, 2005). To circumvent this problem of high PGM loss, high chromite grade and low final concentrate grade, Mintek introduced the mill–float–mill–float circuit or the MF2 circuit, shown in Figure 8. The purpose of this circuit was to address the shortcomings of the mill–float circuit, particularly the overgrinding of chromite.

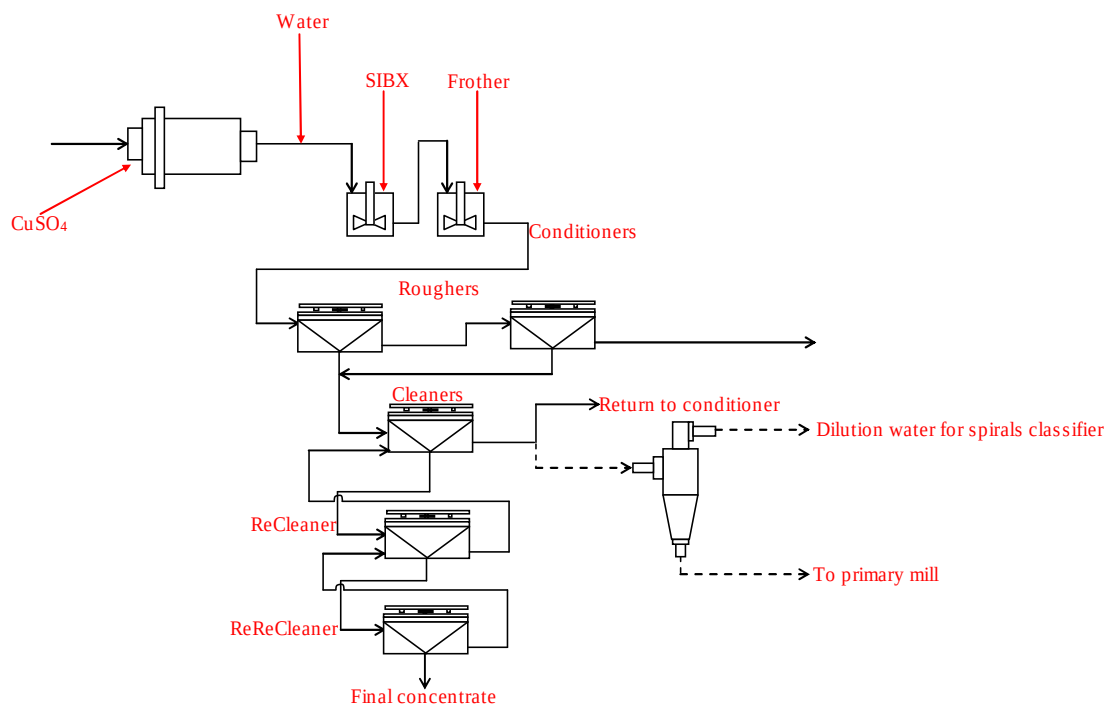


Figure 7: Typical mill–float circuit used initially to concentrate the PGMs from a UG2 ore, reproduced after Deeplaul and Bryson (2004)

separately. This results in two concentrates being produced, a high-grade concentrate and a low-grade concentrate. The PGM recovery from this circuit is typically between 80 to 85%; having a concentrate mass recovery of 1.3% and containing 300 to 600 g/t of PGMs (Woodcock *et al.*, 2007).

Dunne (2007) indicated that the reagents used in the flotation of UG2 ore are:

- Copper sulphate (CuSO_4) which is added to the milling stages as an activator;
- Xanthate (sodium isobutyl xanthate, SIBX) which is added to the two rougher stages as a sulphide and metallic collector;
- Depressant (carboxyl methyl cellulose, CMC) which is added to the rougher and cleaner stages for control of the floatable gangue;
- Frother (Dow200) which is added to the rougher and cleaner stages;
- And flocculants (anionic) which are added to the concentrate thickeners to achieve the desired density.

The reagents are added using peristaltic pumps, and flotation is carried out at a pH of 8.5, which is the natural pH of the ore.

2.4. The flotation process

Since the purpose of this study is to investigate the effect of UG2 ore variability on PGM recovery by flotation; a review of literature on batch flotation testing and modelling is necessary. The review will commence with a description of flotation and the factors which affect the process. The models which describe the various mechanisms by which particles are recovered will also be discussed. Batch flotation techniques used for obtaining parameters for plant design will be reviewed; since attempts will be made in this study to investigate the performance of an ore processed by a flotation pilot-plant.

2.4.1. Overview of flotation

Flotation is a process used to separate a finely divided solid suspension into a product containing economic metals and a reject containing gangue. The approach is based on contacting the suspended economic metal minerals with the rising air bubbles. A reagent suite consisting typically of a collector, depressant and frother is used to create a favourable chemical environment for the recovery of minerals. The flotation process can be summarised

by the interaction between two zones, the pulp (also called collection zone) and the froth zone. The pulp and froth zone collectively influences the recovery and grade of the product. Air is charged into the pulp zone and the action of the impeller together with the frother, which reduces surface tension, disperses the air into small bubbles, which rise to the surface. A collector is added to create minerals with a hydrophobic layer. Particles collide with the air bubbles, and the hydrophobic particles form bubble–particle aggregates. As the bubble–particle aggregate travels in the pulp the particle is swept from the front to the rear of the bubble, due to the relative motion of the bubble. This results in more capacity being available at the front for further particle adhesion (Jameson *et al.*, 1977).

When bubbles enter the froth layer, some liquid (and suspended solids) follows the water, resulting in unselective recovery of gangue particles. The recovery mechanism is called entrainment and as the froth ages the liquid and entrained material drain into the preceding froth layers. The drainage causes the films between the bubbles to be thin, leading to bubble coalescence, which results in the releasing of minerals with the liquid. If the particles are sufficiently hydrophobic, they can reattach, otherwise it drains into the pulp. The process of particle attachment and recovery is affected by many variables.

2.4.2. Factors affecting flotation

A number of factors affect the flotation process and many of these interact with one another making it extremely difficult, if not impossible to model the flotation process. Sutherland and Wark (1955) discussed some of the variables that affect the flotation process and concluded that modelling of the flotation process is complicated by a number of variables that occur simultaneously. Klimpel (1984) summarised these variables into three components, the chemical, equipment and operational, refer to Figure 9. Depending on the settings of the various components in Figure 9 it is possible to achieve a wide range of flotation performances.

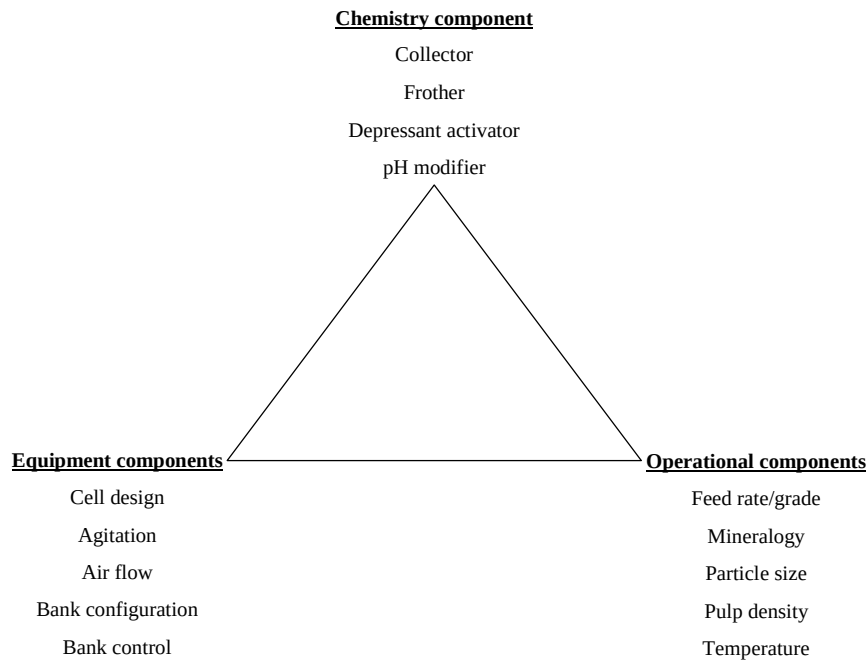


Figure 9: Interaction among the three components that drive the flotation process, reproduced after Klimpel (1984)

Schubert (1999) showed that smaller particles require more turbulence so that a higher frequency of collision can occur between the particles and the bubbles. Deglon *et al.* (2000) carried out an investigation on the hydrodynamics across various platinum concentrators and concluded that the concentrators are operated at higher turbulence so that more collisions can be created between bubbles and particles.

Laplante (1983) cited in Ek (1991) showed that, the rate of flotation increased with increasing air flow rate to a maximum and then decreased as air flow rate was increased further. Jameson *et al.* (1977) found that the rate of flotation increased when bubble size decreased.

Crawford and Ralston (1988) suggested that particle size, together with contact angle, leads to the concept of flotation domain a region which determines whether a particle can be recovered by flotation. Morris (1952) found that the maximum flotation rate was obtained in intermediate particle size fractions; the rate decreased in the coarse and fine sizes. Trahar and Warren (1976) confirmed this when they found that the recovery is greatest for particles of intermediate size, i.e. in the 10-100 μm size range.

Klimpel (1984) stated that chemistry in flotation is important, and its addition needs to be regulated to prevent overdosing. The author found that overdosing with collector can have the disproportionate effect of reducing the flotation rate, which means that additional flotation volume is required to recover the economic metals.

2.4.3. Discussion on batch flotation models

In the platinum industry many, mineral deposits currently being developed are deep level and bankable feasibility studies have to rely entirely on batch experiments, carried out on drill core samples (Hay and Roy, 2010). Barbary *et al.* (1986) discussed the application of batch flotation tests for flotation circuit design and concluded that, there are significant differences between batch and plant data. However, despite the differences, the authors accepted that the simplicity of the batch test makes it favourable for ore characterisation.

The various models used to characterise minerals recovery from a batch test will be discussed. This review is instrumental in determining the most appropriate model for batch data, since it influences the interpretation of the minerals recovery variability.

2.4.4. Pulp phase models

The review on pulp flotation models is based upon an article presented by Fichera and Chudacek (1992). The review is expanded by including publications after 1992.

a. The first order model and description of the rate coefficient

Zuniga (1935) cited in Fichera and Chudacek (1992) made the first attempt to model the mineral recovery with time from a batch test. The author assumed that the flotation rate constant was governed by first-order kinetics. This assumption is analogous to a first-order chemical reaction which is given by:

$$\frac{dC}{dt} = -kC^n \quad (2.1)$$

For a mineral concentration C in the feed with first-order kinetics (i.e. $n = 1$) the recovery with time is:

$$R=1-e^{-kt} \quad (2.2)$$

Higher orders were considered, but Fichera and Chudacek (1992) stated that the deviation from the first-order relationship was most likely due to the inherent heterogeneous characteristics of the flotation feed. Morris (1952) stated that it was not unusual for all particles to be capable of floating due to surface oxidation, liberation, etc. Hence a limiting recovery parameter called maximum recovery ($R_{\max}$) was introduced to account for non-recoverable material:

$$R=R_{\max} \left(1-e^{-kt}\right) \quad (2.3)$$

Schuhmann (1942) cited in Fichera and Chudacek (1992) considered the rate to be determined by particle–bubble collision. The rate was given by an analytical equation, which considered the probability of particle collision (P_c) and adhesion (P_a) to a bubble. A fruitless factor (ϕ) was also included to account for the detaching of particles during flotation:

$$k=P_c P_a \phi \quad (2.4)$$

Bloom and Heindel (1999, 2002); Sarrot *et al.* (2005); Newell and Grano (2006, 2007); Ralston *et al.* (2007); Koh and Schwarz (2008) and Shahbazi *et al.* (2008) have all carried out investigations which used models for the collision, attachment and detachment probability.

Sutherland (1948) cited in Fichera and Chudacek (1992) assumed direct encounter between a particle and bubble and arrived at the following analytical expression for the rate:

$$k = \phi 3\pi R_b r_p V_b N' \operatorname{sech}^2 \left(\frac{3V_b \xi}{4R_b} \right) \quad (2.5)$$

ϕ – Fruitless factor similar to that defined in Equation 2.4

R_b – Bubble radius

r_p – Particle radius

V_b – Bubble velocity relative to particles

N' – Number of bubbles

ξ – Induction time

Nguyen and Nguyen (2008) studied the generalised Sutherland equation, which is an extension of Equation 2.5.

Experiments carried out by Gorain *et al.* (1997) and Kracht *et al.* (2005) showed that the rate can be related to gas dispersion parameters and a floatability parameter, which describes the surface characteristics of a mineral.

b. Distributed models

Gaudin and Plato Malozemoff (1933) cited in Fichera and Chudacek (1992) was the first to carry out tests, which considered particle size. He suggested that the rate was distributed across the particle size range for a mineral. A first-order model was used to determine the rate for each particle size class, and the total recovery was given by:

$$R = \sum_{i=1}^s \omega_i (1 - e^{-k_i t}) \quad (2.6)$$

Where i refers to the different particle size classes, which varies from 1 to s , and ω_i refers to the mass fraction of the mineral having particle size class i in the feed.

c. Distributed floatability models

The distributed floatability model considers the combined properties of a mineral such as size, shape, surface properties, etc. The basic assumption is that the more floatable particles are recovered first, and the least floatable particles are recovered towards the end of the test.

Kelsall (1961) acknowledged a floatability distribution and suggested that for modelling purposes particles may be classified into two groups, i.e. fast and slow floating particles:

$$R_o = R_{fast} (1 - e^{-k_{fast} t}) + R_{slow} (1 - e^{-k_{slow} t}) \quad (2.7)$$

$$R_{fast} + R_{slow} = 100$$

R_o - Overall recovery

R_{fast} - fast floating fraction

R_{slow} - slow floating fraction

k_{fast} - fast floating rate constant

k_{slow} - slow floating rate constant

At an infinite time for flotation the exponential term approaches unity and the overall recovery is equal to the sum of the fast and slow fractions, i.e. 100%.

Imaizumi and Inoue (1963) applied a graphical technique to determine the mineral floatabilities. The total recovery is the sum of the recovery from each floatability class:

$$R = \sum_{j=1}^p \sigma_j (1 - e^{-k_j t}) \quad (2.8)$$

Where j indicates the different floatabilities, which varies from 1 to p for the feed, and σ_j refers to the mass fraction of the mineral having a floatability j in the feed.

Loveday (1966) recognised that the feed has a distribution of floatabilities and proposed a parametric solution, which assumed that the rate constant distribution can be approximated by a gamma function. The recovery equation with fitting parameters a and b was:

$$R = 1 - \left[1 - \left(\frac{b}{b+t} \right)^{a+1} \right] \quad (2.9)$$

Klimpel (1980) assumed a rectangular distribution for the floatability which varies from zero to a maximum flotation rate called k_{max} . A maximum recovery term was introduced:

$$R = R_{max} \left[1 - \frac{1}{k_{max} t} (1 - e^{-k_{max} t}) \right] \quad (2.10)$$

d. Double distributed models

Double distributed models consider the distribution of the rate constant has a function of particle size and particle floatability. Hence, for every particle size class there is a floatability distribution:

$$R = \sum_{i=1}^s \sigma_{i,j} \sum_{j=1}^p \omega_{i,j} (1 - e^{-k_{i,j}t}) \quad (2.11)$$

2.4.5. Froth phase models

An extensive review on the various froth models was presented by Mathe *et al.* (2000) and Vera *et al.* (2002). Therefore, the review presented here is based on the literature presented by these authors.

The froth phase classifies particles based on hydrophobicity. Particles, which are strongly hydrophobic, will most likely be retained in the froth and eventually recovered but particles, which are weakly hydrophobic, will most likely drop back into the pulp phase. However, particle size and reagents influence the classification process. Finch and Dobby (1990) modelled the interaction between the pulp/collection phase and the froth phase. Figure 10 illustrates this interaction, which was mathematically expressed as:

$$R_o = \frac{R_f R_c}{1 - R_c + R_c R_f} \quad (2.12)$$

R_o - Overall recovery

R_c - Collection/pulp phase recovery

R_f - Froth phase recovery

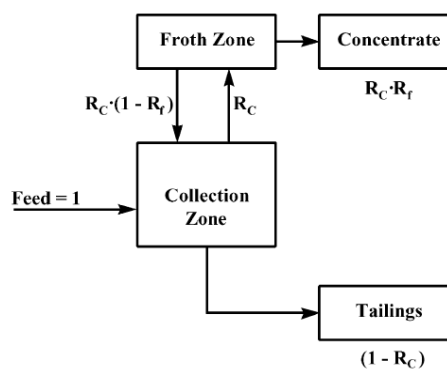


Figure 10: Interaction between pulp/collection phase and the froth phase for a single cell, after Vera *et al.* (2002)

Vera *et al.* (2002) illustrated the influence of the pulp and froth phase recovery on the overall recovery, refer to Figure 11. It was shown that for a pulp phase having recovery between 60 and 99%, and froth phase having recovery between 10 and 90%; the overall recovery can be as low as 50%.

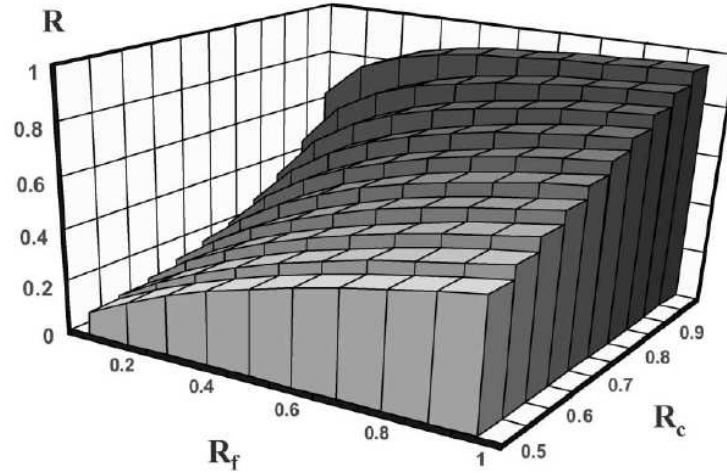


Figure 11: Graphical illustration of Equation 2.12, after Vera *et al.* (2002)

Various methods are available for determining the froth recovery from a batch flotation test. The most common method is that proposed by Vera *et al.* (1999), which simultaneously predicts the pulp phase rate constant and the froth recovery by varying the froth height. The froth recovery was given by:

$$R_f = \frac{k}{k_c} \quad (2.13)$$

Where k is the apparent rate constant and k_c is the pulp phase rate constant. The rate constants can be determined by varying the froth height and fitting the recovery data to a conventional model. The apparent rate constant is plotted against froth height, which was shown by various authors to be a linear relationship. Extrapolation to a zero froth height gives the pulp phase rate constant. However, Yianatos (2007) stated that the froth recovery factor given by Equation 2.13 assumes that the collection of particles in the pulp phase is a first-order process, with 100% maximum recovery, which is not strictly true.

Mathe *et al.* (2000) suggested an exponential decay function for froth recovery with increasing froth retention time (FRT), which can be used for relating batch to continuous froth performance. The model proposed has a non-draining fraction (η) and a froth stability parameter (β):

$$R_f = (1 - \eta)e^{(-\beta \times \text{FRT})} + \eta \quad (2.14)$$

Gorain *et al.* (1998a) presented a model for estimating the froth retention time, which is a function of the froth height (FH) and the superficial gas velocity (J_g):

$$\text{FRT} = \frac{\text{FH}}{J_g} \quad (2.15)$$

The superficial gas velocity can be estimated using the volumetric gas flow rate (Q_g) and the cross sectional area of the flotation cell (A_c):

$$J_g = \frac{Q_g}{A_c} \quad (2.16)$$

2.4.6. Entrainment models

Entrainment is an unselective recovery process in flotation that can result in an undesirable dilution of the concentrate grade and/or desirable increase in the recovery of values. Several authors (Warren, 1985; Çilek and Umucu, 2001; Zheng *et al.*, 2006b, Gülsoy, 2005; and Yianatos and Contreras, 2009) have used the relationship that gangue recovery by entrainment (R_e) is proportional to the recovery of water (R_w), which can be expressed mathematically as:

$$R_e = CF \times R_w \quad (2.17)$$

CF refers to the classification function, which describes the degree of entrainment. Since entrainment is strongly related to particle size, the above equation can be expanded for particle size classes i :

$$R_{e,i} = CF_i \times R_w \quad (2.18)$$

George *et al.* (2004) reviewed three conventional techniques for assessing the recovery by true flotation and entrainment. These techniques will be briefly discussed.

2.4.6.1. The method of Trahar and Warren (1976)

This method compares the water and solids recovery using two flotation tests. In the first test, both collector and frother are used, and in the second test only frother is used. It is hypothesised that the recovery in the second test was due to entrainment only therefore, the minerals that were recovered by true flotation are estimated by the difference in recovery between test one and two. The method assumes that entrainment occurs only when frother is added, which is not strictly true, since the froth is influenced by the presence of floatable particles.

2.4.6.2. The method of Warren (1985)

This method fits a linear relationship to data obtained from tests at various froth depths and froth removal rates. The linear water–solids recovery relationship is extrapolated to zero water recovery to estimate the recovery by true flotation. The technique can be mathematically expressed as the total recovery (R) which is the sum of the recovery by true flotation (R_{true}) and the entrainment recovery (R_e):

$$R = R_{\text{true}} + R_e \quad (2.19)$$

The entrainment recovery was obtained by plotting the solids recovery against water recovery. The relationship is assumed to be linear therefore; the slope of the line gives the degree of entrainment. The degree of entrainment is multiplied by the water recovery to obtain the recovery by entrainment, similar to Equation 2.17.

2.4.6.3. The method of Ross and Van Deventer (1988)

In this method, the concentration of particles in the water contained in the froth is assumed to be identical to the concentration of particles in the pulp. The recovery by entrainment is estimated by multiplying the water recovery to the concentrate by the concentration of solids

in the pulp at that time. The overall recovery is similar to that given by Equation 2.19 but the entrainment recovery is estimated by the following equation:

$$R_e(t) = W(t)(C - bt) \quad (2.20)$$

$R_e(t)$ – Entrained recovery at time t

$W(t)$ – Mass recovery of water at time t

C – Initial concentration of mineral in pulp i.e. at $t = 0$

b – Is a model derived parameter which is determined from the slope of the graph of the mineral concentration remaining in the pulp versus time

George *et al.* (2004) compared the above techniques in addition to a technique developed by the authors. They concluded that all these techniques have drawbacks, which include changing froth structure, changing particle morphology and changing surface chemistry.

The entrainment models discussed above are all expressed in terms of water recovery. Gülsoy (2005) and Shivakumar and Nikkam (2005) proposed a kinetic model for estimating the water recovery in a batch test. The model proposed by Gülsoy was:

$$R_w = \gamma(1 - e^{-\kappa t}) \quad (2.21)$$

Where γ and κ are the model fitting parameters. The model proposed by Shivakumar and Nikkam was:

$$R_w = R_{w,max} \left(1 - e^{-\zeta(t+\varphi)} \right) \quad (2.22)$$

Where $R_{w,max}$ is the model-fit derived maximum water recovery, ζ is the model-fit derived water rate constant and φ is the experimental time lag.

2.4.7. Discussion on scale-up in flotation

Scale-up of batch results are important for estimating the residence time for the various stages of the plant. Yianatos *et al.* (2010) stated that from a metallurgical perspective, the

problem of scale-up has not been completely solved. Several authors have investigated flotation scale-up and some of these investigations will be discussed.

Kalapudas (1985) investigated the scale-up factor, which was defined as the ratio between the plant and batch flotation retention time. A similar rate constant was used for both the plant and batch system. He stated that the scale-up factor was between 1.5 and 3.5, and longer residence times are required for the plant, due to residence time distribution, i.e. short-circuiting in continuous operations.

Gorain *et al.* (1998b) suggested the use of bubble flux (S_b) as the scale-up criterion for ores having similar floatability. The authors found that the flotation rate was linearly related to S_b , which is:

$$S_b = \frac{6Jg}{d_b} \quad (2.23)$$

The bubble flux is a function of the gas velocity (which was defined in § 2.4.5) and the Sauter mean bubble diameter (d_b).

Yianatos *et al.* (2003) fitted a conventional batch model (Equation 2.10 from § 2.4.4c), and continuous model to data obtained from batch and plant tests respectively. The authors obtained the scale-up factors at the optimum mineral separation point according to:

$$\frac{\tau}{t} = \frac{[k_{\max}]_{\text{batch}}}{[k_{\max}]_{\text{plant}}} \quad (2.24)$$

$$\frac{R(\tau)}{R(t)} = \frac{[R_{\max}]_{\text{plant}}}{[R_{\max}]_{\text{batch}}} \quad (2.25)$$

Equation 2.24 indicates that the residence time ratio between the plant (τ) and batch (t) is equal to the ratio between the maximum rate from the batch and the plant at optimal separation. Optimal separation refers to the point when the concentrate grade is equal to the feed grade to the bank. The plant and batch time is used to determine the maximum recovery for the plant, provided that the maximum recovery for the batch test is known.

Dobby and Savassi (2005) and Amelunxen and Amelunxen (2009b, c) proposed similar techniques for the scale-up of batch rates of flotation. The batch data was fitted to conventional pulp models, and the pulp rate obtained was applied directly (i.e. no scale-up) in evaluating plant performance. Variation in plant and batch performance was related to the froth recovery.

Recently, Yianatos *et al.* (2010) presented an empirical scale-up model:

$$\frac{k_{app}}{k_b} = \varepsilon \Phi \chi \psi \quad (2.26)$$

k_{app} is the apparent flotation rate for an industrial cell which is also called the overall rate constant; k_b is the batch rate constant; ε is a scale-up factor which is the ratio between the actual flotation pulp rate constant in a plant and the batch rate constant, Φ is the froth factor, χ is the cell mixing factor and ψ is the particle segregation factor. The authors proposed empirical equations for estimating each factor.

2.4.8. Application of batch flotation tests in evaluating flotation plant performance

The ability to understand ore variability is improved significantly if batch data can be used to determine plant performance. However, there are many difficulties encountered when attempting to extrapolate batch data for understanding plant performance; these were discussed in § 2.4.2. Four flotation simulators will be discussed in this section. The simulators, in the author's opinion, are the most well-known and have significant publications that illustrate its application. The simulators are called MinnovEx Flotation Test (MFT) also called FLEET, Aminpro's Full Kinetic Test (FKT), SUPASIM and JKSimFloat. A brief review of these simulators will be presented.

2.4.8.1. MinnovEx (FLEET) Flotation Test

This simulator was proposed by Dobby and Savassi (2005) and uses a simple batch flotation test to derive parameters for evaluating plant performance. The methodology is divided into three steps:

1. Direct measurement of the rate of true flotation in the pulp which is independent of hydraulic entrainment and other froth effects in the batch cell;
2. Benchmarking of the industrial plant over a broad range of ore types and operating conditions;
3. And model calibration to describe the froth effects in the plant with a minimal number of parameters.

The procedure for the batch test can be summarised as follows:

1. A size-by-size assay is carried out on the feed, combined concentrate and tail. The combined concentrate is composed of four individual concentrates, taken at different times during the test.
2. The test is carried out with a shallow froth, which is scraped at a rapid rate so that the effect of the froth phase on recovery is minimised.
3. The results are then interpreted as minerals rather than metal recovery. A phenomenological approach is used to decouple the recovery by true flotation from the recovery by entrainment.
4. The rates for true flotation are modelled using a frequency distribution of the rate constants, which is mathematically similar to the Rosin–Rammner equation used in particle size distribution analysis.
5. Since mineral recovery information is available, as a function of particle size, the rate frequency distribution can be used to predict the recovery for changes in the feed grind. The assumption used here, is that mineral recovery per size interval remains constant, with changes in grind affecting the distribution in mass between the intervals.
6. The pulp rates from the batch test are used directly with calibrated froth parameters, which are determined from plant survey data. The parameters are used to optimise the mineral recovery from the plant. Historical froth parameters are used in conjunction with the batch generated pulp rate frequency distribution for designing of a new plant.

2.4.8.2. Aminpro's Full Kinetic test

This method was proposed by Amelunxen and Amelunxen (2009b, c) and is similar to the MFT (FLEET) method described above, the procedure for this test is:

1. Only the feed and tail are sized from a batch flotation test therefore, the recovery of minerals to the concentrate is determined by mass balance.
2. The minerals recovery in the pulp phase is modelled according to size, with each size class having a maximum recovery and rate (Equation 2.6 from § 2.4.4b). The sum of the recovery from each size class gives the total minerals recovery in the pulp; entrainment and froth recovery parameters are also included in the batch model.
3. No formal froth recovery model is used when fitting the model to batch data.
4. The degree of entrainment for each size class is modelled using a Swebrec function.
5. The pulp rate constants and entrainment parameter are used directly in plant simulation. Historical froth recovery factors are used to evaluate the recovery and to determine the water recovery through a calibration function.

2.4.8.3. SUPASIM

This method was proposed by Hay (2005) and uses batch data to diagnose and understand the flotation behaviour for plant design. The Kelsall model given by Equation 2.7 in § 2.4.4c is fitted to batch flotation data for each mineral. Each mineral has a fast and slow fraction and corresponding rate constant, which describes the recovery. The author mentioned that many factors affected the scale-up between plant and batch data. The parameters obtained from batch scale tests are used to predict larger-scale behaviour by applying suitable froth factors from a database.

2.4.8.4. JKSimFloat

JKSimFloat is the most widely used of the simulators discussed thus far. It is the product of several years of research (Runge *et al.*, 1998; Alexander *et al.*, 2000; Harris *et al.*, 2002 and Schwarz and Kilgariff, 2005). The simulation of plant performance is accomplished by separating ore and machine factors. Batch flotation test together with plant survey data is used to model the plant performance and to optimise minerals recovery. Batch tests are used to determine the floatable mass fractions for streams in the plant. Plant data is used to calibrate froth, entrainment and pulp models. Recently, Varadi *et al.* (2010) proposed a method that determines the floatable mass fraction for a new plant feed using batch tests; the parameters were superimposed onto a historical plant model to determine plant performance. The historical plant model was calibrated in a previous plant survey, which determined froth, entrainment and pulp parameters.

2.5. Conclusions drawn from chapter

A review of literature pertaining to ore variability and its influence on minerals processing, processing of PGMs in South Africa and batch flotation testing and modelling was presented. The concept of ore variability with respect to minerals processing was discussed. Literature shows that ores which are sufficiently characterised at the bench-scale (with respect to mineralogy, geology and metallurgy) reduce the risks involved in plant design and operation. A review on PGM processing in South Africa showed that the UG2 ore has a diverse mineralogy, due to various geological events, and this influences the PGM recovery. Lastly, a review of literature on batch flotation testing and modelling showed that batch flotation tests are favoured in characterising an ore, because a small sample mass can be used and various test conditions can be examined.

However, the large number of batch flotation models available, presents a problem when attempting to model batch tests; since no particular model has been identified as the most appropriate for modelling batch tests. The parameters from the batch models are important for creating a link to mineralogy and geology hence, the selection of a suitable model is crucial in understanding PGM recovery variability. The information obtained from this review will be used in subsequent chapters to develop a test program for illustrating the PGM recovery variability from UG2 samples. Furthermore, links to feed properties will be examined, and attempts will be made to relate batch flotation parameters to plant performance.

CHAPTER 3. FORMULATION OF A PROCEDURE FOR ILLUSTRATING THE EXTENT OF PGM RECOVERY VARIABILITY

3.1. Introduction

The main purpose of this chapter is to formulate a procedure for illustrating the effect of UG2 ore variability on the PGM recovery by flotation. The procedure will consider various factors such as batch flotation test method, feed chemical assays, reproducibility of batch flotation test setup and assay techniques and appropriate modelling of the minerals recovery from the batch flotation test. These factors were considered because it facilitates an examination between the variable PGM recovery observed (from flotation) and the variable feed properties (through feed chemical assay). However, this relationship will be discussed in Chapter 4.

3.2. General theory for designing a variability test

Literature pertaining to the subject of ore variability testing will be briefly discussed in this section.

Dunne *et al.* (2002) discussed the interpretation of flotation data for the design of the process plant. The authors stated that the first step in designing the flotation plant is sample selection, which gives an indication of the geology and mineralogy of the deposit. The samples are evaluated using a simple and reproducible laboratory batch flotation test, because a process with unnecessary steps or reagents is difficult to analyse.

Williams and Richardson (2004) noted that geology, chemistry, geotechnical, mineralogy, metallurgical response and physical properties of the ore, are the various disciplines which are important in understanding ore variability. Table 6 presents a summary of the various tests proposed by the authors. Application of all the tests proposed is not practical, since there are a number of samples in a variability study hence; this will be uneconomical and time consuming. Furthermore, these tests require a large sample mass, which is often difficult to source during the feasibility phase of a project.

Lane *et al.* (2005) discussed the design and engineering of flotation circuits; the authors noted that mineralogy, liberation and chemical properties of the ore are key parameters to consider when designing flotation circuits.

Dunham and Vann (2007) gave an economic discussion on the importance of understanding ore variability. The authors stated that in addition to the traditional measures such as feed grade, other aspects must also be measured. Some of the measures suggested were concentration of deleterious elements, hardness, grindability, mineralogy and metallurgical recovery.

Table 6: Summary of various tests that can be used to quantify ore variability, after Williams and Richardson (2004)

Discipline	Parameter	Testing possible
Geology	Field relationships	Mapping, drilling, decline
Chemistry	Grade	Assay
Mineralogy	Zonation	Mineral identification, association, size, textural and liberation data by quantitative electron microscopy scan (QEMSCAN)
Physical properties	Hardness–grinding	Bond Work Indices, JK drop weight test, SPI Index, MacPherson 18" mill test
Metallurgical response	Recovery	Flotation kinetics, locked cycle tests
Geotechnical measures	Site preparation and environmental review	Solid density, ground water flow, slope stability

The literature presented indicates that there are a number of factors, which must be considered when carrying out a variability study. However, it is not practical to have a complex study which has a number of samples that requires mineralogy, feed chemical assay, rock breakage testing and extensive flotation testing. Therefore, it was necessary to develop a simple test procedure, which could be applied to a large number of UG2 samples collected from across the Bushveld Complex.

Since the objective of this study is to illustrate PGM recovery variability and to relate this to feed properties; the test procedure will consider sample selection, batch flotation test method,

feed chemical assay and batch flotation recovery modelling. Mineralogy is an essential test but due to the high cost of carrying out this test; it was decided that a limited number of samples would be evaluated, to determine the mineralogical factors influencing PGM recovery. This will be discussed further in Chapter 5 and 6.

3.3. Formulation of a procedure for investigating variability

Figure 12 shows the factors that were considered in formulating a procedure for determining the PGM recovery variability. The reproducibility of the batch flotation test and feed chemical assays will also be examined. Each of the factors will be discussed in the sections to follow.

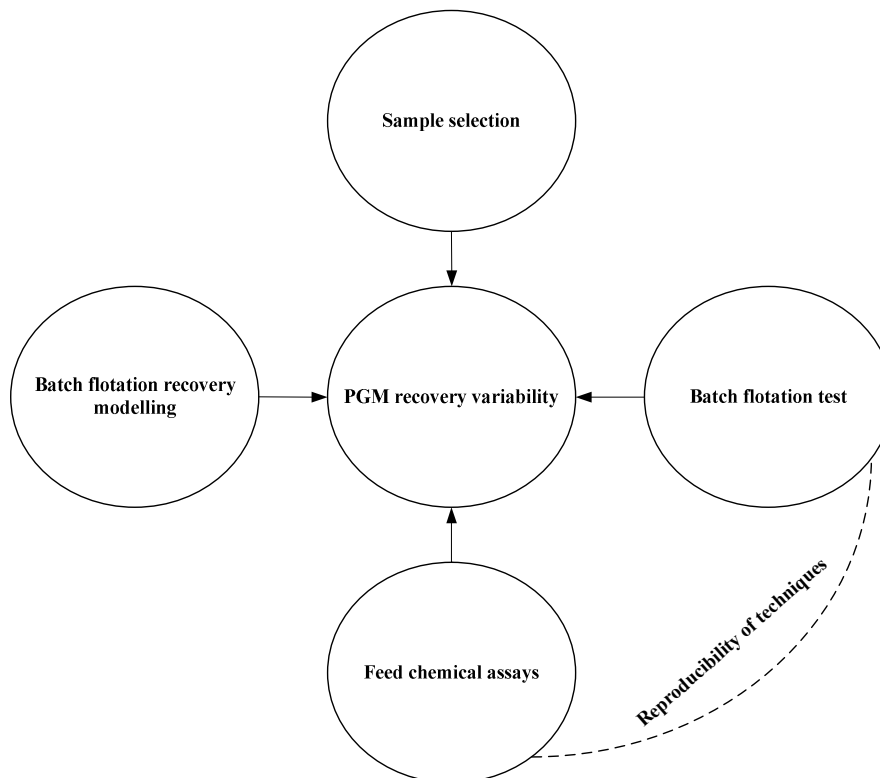


Figure 12: Factors considered in variability study

3.3.1. Sample selection

Williams *et al.* (2002) stated that the selection of samples must be considered before metallurgical test work, and a corollary of this is, poor sample selection can lead to poor or misleading metallurgical results. The selection of samples for characterisation is composed of two requirements, sampling methodology and sample extraction.

With regards to sampling methodology, operations must select samples that reflect the characteristics in the deposit, which are lithology, grade and geology. The metallurgical test work is compromised if the samples selected are not representative of the characteristics in a deposit.

Sample extraction is also an important aspect to consider when selecting samples. There three common sampling techniques employed are core drilling, reverse circulation drilling and channel sampling.

Core drilling is the most expensive of the three techniques, but this sampling is usually preferred, because the sample is more resistant to oxidation. For UG2 samples this is not a major concern, since the sample has low mineral sulphide content and can be stored as crushed material in closed containers, which is practised at Mintek.

Reverse circulation is the least-expensive method of extracting samples, but the sample is contacted with fluid that circulates down the interior of the drill. This may result in contamination of the sample.

Channel samples are usually taken from two parallel cuts in the reef, using a pick or rock hammer. The benefit of this technique is that large samples can be taken economically unlike core drilling.

For this study, the following constraints were applied when selecting samples:

- Correct mining depth- This avoided the selection of near surface material, which is generally oxidised material that has poor PGM recovery. Material from near the surface will not be considered, since it is known that the poor recovery is due to surface alteration.
- Typical mining width- Mining widths are realistic samples, which are similar to the plant feed. This constraint prevents the situation where only the reef material is provided, which generally has good metallurgical characteristics, since there is no dilution waste. This constraint is also important if the waste contains floatable gangue and PGMs.
- Sample supervision- The selection and extraction of samples were supervised by a senior geologist. This gave more confidence in the samples selected, since the geologists are aware of locations in the deposit which have varying properties.

- Sampling technique- Samples were taken as core drillings or rock chippings. Core drilling enabled the inclusion of Greenfield operations that could not take large samples, but could source material from cores used in their basic/definitive feasibility study. For established operations, rock chippings were used to obtain large samples.

3.3.2. Feed chemical assay

3.3.2.1. General

Feed chemical assay is the most preferred factor in variability studies, since it is inexpensive and can be carried out promptly. Table 7 shows the chemical assays which were selected for characterising each feed sample. These are typical assays used in geochemical studies[#] (Von Gruenewaldt *et al.*, 1986; Barnes, 1990; Wilson *et al.* 1999; Barnes and Maier, 2002; McSween *et al.*, 2003; McDonald *et al.*, 2005; Hanley, 2005 and Mungall, 2005). It is believed that having detailed assays, as those used in geochemical studies, may assist in interpreting the variability in PGM recovery.

Table 7: Summary of feed chemical assays

Assays
Pt, Pd, Rh, Ru, Ir and Au
Total sulphur and sulphide sulphur
Cr ₂ O ₃ , A/sNi, A/sCu, A/sFe
MgO, Al ₂ O ₃ , SiO ₂ , TiO ₂ , V ₂ O ₅ , MnO, Fe, P, Cl, Ca,
Co, Ni, Cu, Zn, Ga, As, Se, Rb, Sr, Y, Zr
LOI: 1050, 900, 600, 300°C
and SG

Reproducibility is important when designing a test procedure because; tests that are not reproducible can give results which lead to incorrect conclusions. The reproducibility for each assay was determined by assaying one UG2 sample ten times. The results were used to determine the mean and confidence interval for each assay. The confidence interval was

[#] Geochemistry is the study of the chemical changes on the earth (Krauskopf, 1967). It involves studying the absolute and relative abundances of chemical elements in the; minerals, soils, ores, rocks, water and atmosphere of the earth. This study focuses on understanding the movement of elements from one place to another in an ore deposit, as a result of their chemical and physical properties. This has important implications in prospecting since it determines the age of rocks.

determined at a 95% level using a student's t-distribution. A brief description of the assay techniques will be presented here; a more detailed description can be found in Appendix A1.

3.3.2.2. PGE assay

There are four PGE assay techniques available at Mintek these are total PGE+Au, 2PGE+Au, 3PGE+Au and 5PGE+Au.

Total PGE+Au refers to the total platinum group metal content of a sample, i.e. Pt, Pd, Rh, Ru, Ir and Au. The drawback of using this method is the high metal detection limit (which is 0.28 g/t), and individual assays for each metal cannot be determined.

2PGE+Au refers to the Pt, Pd and Au content of a sample. Only the major metals Pt, Pd and associated Au are measured. The lower detection limit for each metal is 0.1 g/t.

3PGE+Au refers to the Pt, Pd, Rh and Au content of a sample. Individual assays are obtained for each of the metals mentioned, and the lower detection limit for each metal is 0.1 g/t.

5PGE+Au refers to the Pt, Pd, Rh, Ru, Ir and Au content of a sample. Individual assays are obtained for each of the metals mentioned, and the lower detection limit for each metal is 0.1 g/t.

Table 8 shows the reproducibility statistics for the four PGE assay techniques discussed. The 2PGE+Au assay has the lowest mean assay value, and this can be attributed to Rh not being assayed. The 5PGE+Au assay has the highest mean assay value, and this is due to the assaying of more PGEs. The assay techniques have reasonable reproducibility, since the confidence interval for each assay is narrow. The 5PGE+Au assay will be used to determine the PGE content of the feed samples.

Table 8: Statistics for PGE assay techniques

	Total PGE+Au [g/t]	2PGE+Au [g/t]	3PGE+Au [g/t]	5PGE+Au [g/t]
Mean	3.10	2.76	2.89	3.22
Confidence Level (95.0%)	±0.10	±0.05	±0.06	±0.11

3.3.2.3. Total sulphur and sulphide sulphur

The total sulphur content was determined using the conventional LECO technique, and the sulphide sulphur was determined by difference between total sulphur and sulphate sulphur

content. Table 9 shows total sulphur and sulphide sulphur reproducibility statistics, both techniques have similar confidence levels, which is acceptable for this study.

Table 9: Statistics for total sulphur and sulphide sulphur assay

	Total sulphur [%]	Sulphide sulphur [%]
Mean	0.13	0.09
Confidence Level (95.0%)	±0.01	±0.01

3.3.2.4. Cr₂O₃ and acid soluble base metals

The feed samples were assayed for chrome oxide (Cr₂O₃) and acid soluble base metals- A/sNi, A/sCu and A/sFe. The Cr₂O₃ assay is normally carried out for UG2 samples, since it is related to the chromite (FeCr₂O₄) content. Table 10 shows the statistics for these assays which are acceptable for this study.

Table 10: Statistics for Cr₂O₃ and acid soluble base metal assays

	Cr ₂ O ₃ [%]	A/sNi [g/t]	A/sCu [g/t]	A/sFe [g/t]
Mean	16.54	303.95	85.85	8131.00
Confidence Level (95.0%)	±0.16	±3.40	±1.83	±98.00

3.3.2.5. X-Ray fluorescence scan

The feed samples were scanned using X-Ray fluorescence (XRF) to determine the major and minor elements. The XRF scan gave assays for MgO, Al₂O₃, SiO₂, TiO₂, V₂O₅, MnO, Fe, P, Cl, Ca, Co, Ni, Cu, Zn, Ga, As, Se, Rb, Sr, Y and Zr. Table 11 shows the statistics for the XRF scan.

Table 11: Statistics for assays from the XRF scan

	MgO	Al ₂ O ₃	SiO ₂	TiO ₂	V ₂ O ₅	
	[%]	[%]	[%]	[%]	[%]	
Mean	17.53	12.89	33.10	0.72	0.17	
Confidence Level (95.0%)	±1.63×10 ⁻¹	±2.36×10 ⁻¹	±6.50×10 ⁻¹	±1.37×10 ⁻²	±7.75×10 ⁻³	
	MnO	Fe	P	Cl	Ca	
	[%]	[%]	[%]	[%]	[%]	
Mean	0.18	12.88	0.31	0.04	2.57	
Confidence Level (95.0%)	±3.43×10 ⁻³	±3.71×10 ⁻¹	±1.55×10 ⁻²	±1.80×10 ⁻³	±1.03×10 ⁻¹	
	Co	Ni	Cu	Zn	Ga	
	[%]	[%]	[%]	[%]	[%]	
Mean	2.59×10 ⁻²	8.99×10 ⁻²	1.06×10 ⁻²	2.74×10 ⁻²	3.08×10 ⁻³	
Confidence Level (95.0%)	±1.61×10 ⁻³	±7.07×10 ⁻³	±7.30×10 ⁻⁴	±1.48×10 ⁻³	±1.72×10 ⁻⁴	
	As	Se	Rb	Sr	Y	Zr
	[%]	[%]	[%]	[%]	[%]	[%]
Mean	3.62×10 ⁻⁴	9.04×10 ⁻⁵	3.63×10 ⁻⁴	6.94×10 ⁻³	3.85×10 ⁻⁴	1.11×10 ⁻³
Confidence Level (95.0%)	±6.04×10 ⁻⁵	±2.97×10 ⁻⁵	±2.34×10 ⁻⁵	±3.13×10 ⁻⁴	±3.64×10 ⁻⁵	±1.37×10 ⁻⁴

3.3.2.6. LOI and SG

Loss on ignition (LOI) was carried out on each feed sample. According to literature (Lewis, 1996; Christidis, 1998; Chen and Xia, 2008 and Ngom *et al.*, 2010) LOI can be used to determine alteration in a sample. A high LOI value indicates the presence of large amounts of secondary silicates and other volatiles such as H₂O, F, Cl, etc. Alteration occurs as a result of late injection of hydrothermal fluid. The LOI was determined at four temperatures- 300, 600, 900 and 1050 °C. In some instances mass can be gained instead of lost, this is called gain on ignition (GOI), which is reported as a negative loss on ignition. GOI is not uncommon and occurs when the volatile species form oxidation products with air in the oven; this increases the mass of the sample. Reproducibility for this assay was determined at the two end temperatures (300 and 1050 °C). Table 12 shows the statistics for LOI and the specific gravity (SG), which was determined using a pycnometer.

Table 12: Statistics on LOI and SG analysis

	LOI		SG
	300 °C	1050 °C	
Mean	0.14	0.07	3.56
Confidence Level (95.0%)	2.32×10^{-3}	2.36×10^{-3}	3.69×10^{-3}

3.3.3. Batch flotation test procedure

3.3.3.1. Flow sheet selection

Batch flotation tests can vary from a simple rougher test to a more complicated multistage cleaning test. The test can have a single recovery point or progressive collection of concentrate with time, which is called a rate test. There can be various batch flotation test permutations therefore; the selection of a test method is based on simplicity, and the need to get maximum information from a limited mass of sample.

The simple rougher test does not provide sufficient information on the upgrading of minerals in an ore and the effect of gangue depressant on minerals recovery. In UG2 flotation, depressant is important for constraining the floatable gangue therefore, a test method with rougher concentrate cleaning will be used. Furthermore, a prolonged rate test will be carried out because it provides more information than a single recovery point test. Figure 13 shows the flow sheet that was selected for investigating PGM recovery variability. The milled sample is concentrated in a rougher stage, and the concentrate is re-floated in a cleaning stage. The experimental conditions will be discussed in the next section.

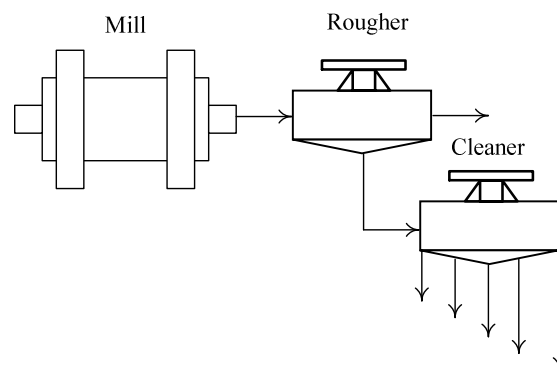


Figure 13: Flow sheet selected for investigating metallurgical variability

3.3.3.2. Experimental

A conventional Denver D12 laboratory flotation machine was used to carry out flotation tests, refer to Figure 14. The machine has a variable-speed drive and the air flow rate to the machine was controlled and measured using a rotameter, which has a linear scale with 100% referring to 21 Nℓ/min at a line pressure of 4.85 bar absolute. The line pressure was maintained using a pressure regulator. The air supplied was filtered using an in-line air cleaner, which prevents dust and oil from entering the flotation cell from the air compressor.



Figure 14: Laboratory batch experiment for UG2 flotation

Typical UG2 reagents which are, collector (SIBX), gangue depressant (KU5) and frother (Dow200) were used. The reagents were prepared to 1%w/w standards using distilled water. The samples were milled in a rubber lined rod mill with a defined stainless steel rod distribution and mass charge, refer to Appendix A3. Rod mills are preferred for laboratory flotation tests because it is easier to discharge the mill product, by comparison to a ball or stirred media mill. Furthermore, a ball and stirred media mill requires more water to wash the pulp off the media, and as a result the milled sample requires filtering, to obtain the correct solids concentration for flotation.

A milling curve could not be generated for each sample due to the limited mass available. Since the objective of this study is to find relationships between feed properties and PGM recovery, it is more logical to have a large number of samples evaluated, irrespective of the limited sample mass available. Hence, a fixed milling energy was used. The energy was determined by generating a milling curve on one sample, which had a sufficient mass, this sample was also used in investigating the feed chemical assay reproducibility discussed earlier. The sample required 16 kWh/t of energy to achieve a grind of 80% passing 75 μm . The test was also partially optimised for each ore by varying the depressant dosage in the cleaner stage.

The experimental procedure is as follows:

1. A 1 kg sample was milled to a fixed energy of 16 kWh/t (i.e. 43 minutes) using a rod mill and water from the local municipality (Rand Water Board). The milled sample was transferred into a 2.5 ℓ flotation cell and more water was added to achieve the correct pulp level; Figure 15 shows the flotation cell used for the tests. The cell has a perspex window for observing and controlling the pulp level. The pulp level was maintained at a height of 108 mm from the base of the flotation cell.
2. The pulp was agitated at an impeller speed of 1200 rpm. The first reagent added was sodium isobutyl xanthate (SIBX) this reagent was added at a dosage of 150 g/t, and the pulp was conditioned for 2 minutes. KU5 was added at a dosage of 30 g/t, and the pulp was conditioned for 3 minutes. The last reagent added was Dow200 at a dosage of 20 g/t; the pulp was conditioned for a minute.
3. After reagent conditioning, air was charged into the cell at 6.3 Nℓ/min. The rougher was floated for 20 minutes to produce a bulk concentrate; the rougher test was repeated to generate sufficient mass for the cleaner stage and for chemical assay. The bulk concentrate was obtained by scraping the froth every 15 seconds with two paddles. The paddles were used to scrape the froth from the back of the flotation cell and into a froth collector tray.



Figure 15: Description of the 2.5 ℓ float cell which has a window for observing (and thus allowing controlling) of the pulp level

4. The duplicate bulk rougher concentrates were transferred into another 2.5 ℓ flotation cell for cleaning. The depressant additions were varied in the cleaner stage. Preliminary development of the froth, with the introduction of air, gave an indication of the froth stability and if too stable, the air was switched off and more depressant was added. The test was started once the froth structure was satisfactory. The premise of adding an incremental amount of depressant, and observing the influence on the froth structure, was based on maximising the selectivity between PGMs and gangue. This is an empirical exercise, and Figure 16 illustrates the method. The froth was stable at 0 g/t KU5 due to floatable gangue, but as the depressant dosage increased, the floatable gangue was depressed. At the final dosage of 80 g/t KU5 it was believed that the selectivity was maximised, further depressant addition would have created an unstable froth.
5. Five sequential concentrates were collected from the cleaner stage at flotation times; 1, 3, 7, 20 and 30 minutes. When collecting the concentrates separate wash bottles (which were weighed prior to the test) were used to clean the scrapers. The additional water used for cleaning the scrapers was subtracted from the wet mass of the concentrate to determine the water recovered with each concentrate.
6. All samples were filtered, dried and weighed prior to sample preparation for assaying.
7. The samples were pulverised in a steel bowl and assayed for Cr₂O₃ and major PGEs, Pt, Pd and associated Au (2PGE+Au).
8. The grind of the samples was estimated by sizing the rougher tail at 75 µm.

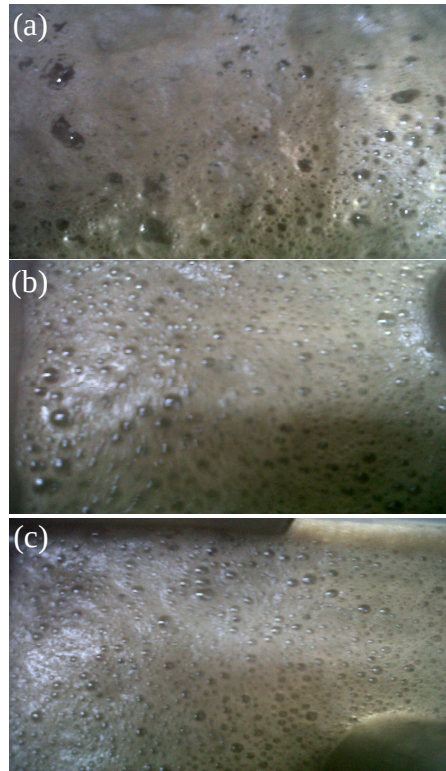


Figure 16: Variation of froth structure in the cleaner stage with depressant addition, (a) 0 g/t KU5, (b) 50 g/t KU5 and (c) 80 g/t KU5

3.3.3.3. Batch flotation reproducibility

The reproducibility of the batch flotation method (i.e. scraping and pulp level control technique) was evaluated by carrying out ten rougher rate tests. The sample used for determining the assay reproducibility and the milling curve was used for batch flotation test work. The reproducibility of the rate of recovery of PGMs, gangue, chromite and water is shown in Figure 17, Figure 18, Figure 19 and Figure 20 respectively. The average recovery from the ten tests is given as a solid line; the dashed lines give the lower and upper 95% confidence interval, which was determined from a student's t-distribution. The experimental data for each test is also shown in the figures.

In general, the batch flotation test method gives reproducible data because the deviation in the recovery points from each test is minimal. Most of the data occurs within the confidence range. However, the chromite data shows some variation, particularly towards the end of the flotation test. The variation may be due to the unstable froth, which occurred towards the end of the test. The unstable froth results in an increase in water recovery, which carries a variable amount of chromite by entrainment. The rate of recovery of water was fairly consistent however; marginal variations in the water recovery and froth conditions can affect

the recovery by entrainment. Nevertheless, the test method gave reasonably consistent data, which indicates that the scraping and level control techniques are reproducible.

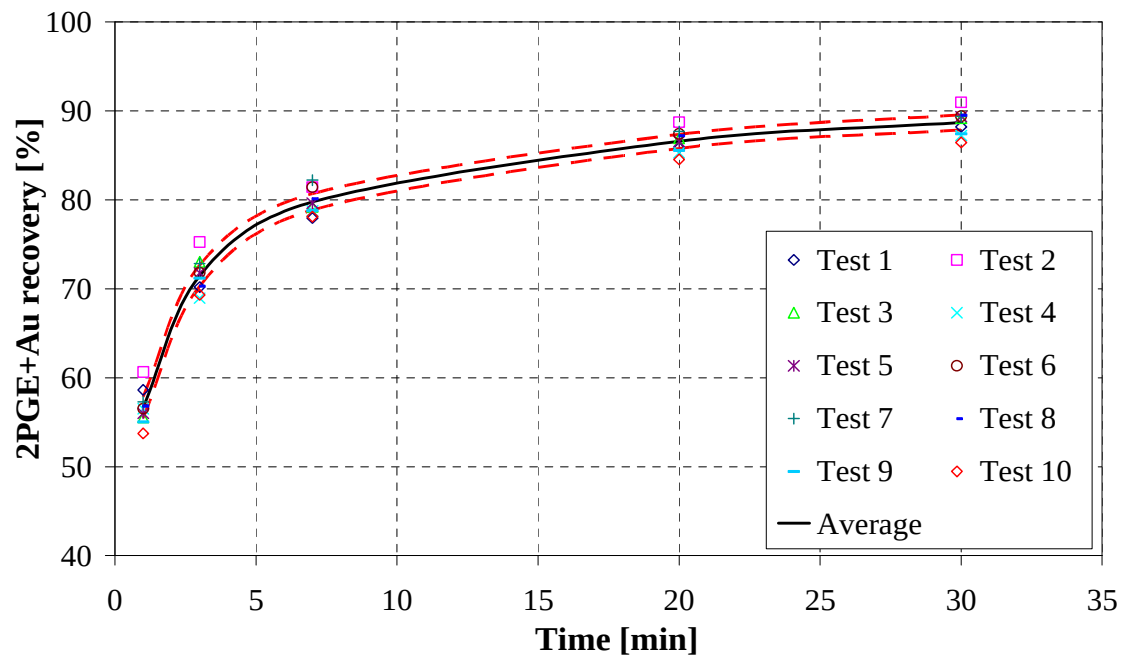


Figure 17: 2PGE+Au recovery reproducibility for batch rougher flotation test

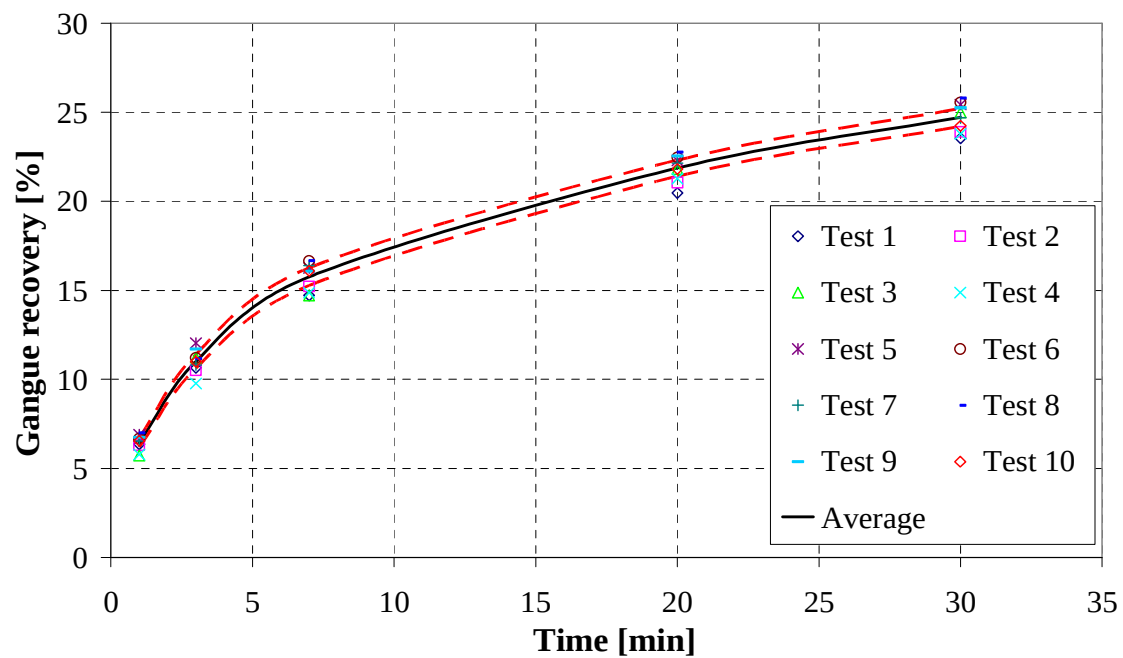


Figure 18: Gangue recovery reproducibility for batch rougher flotation test

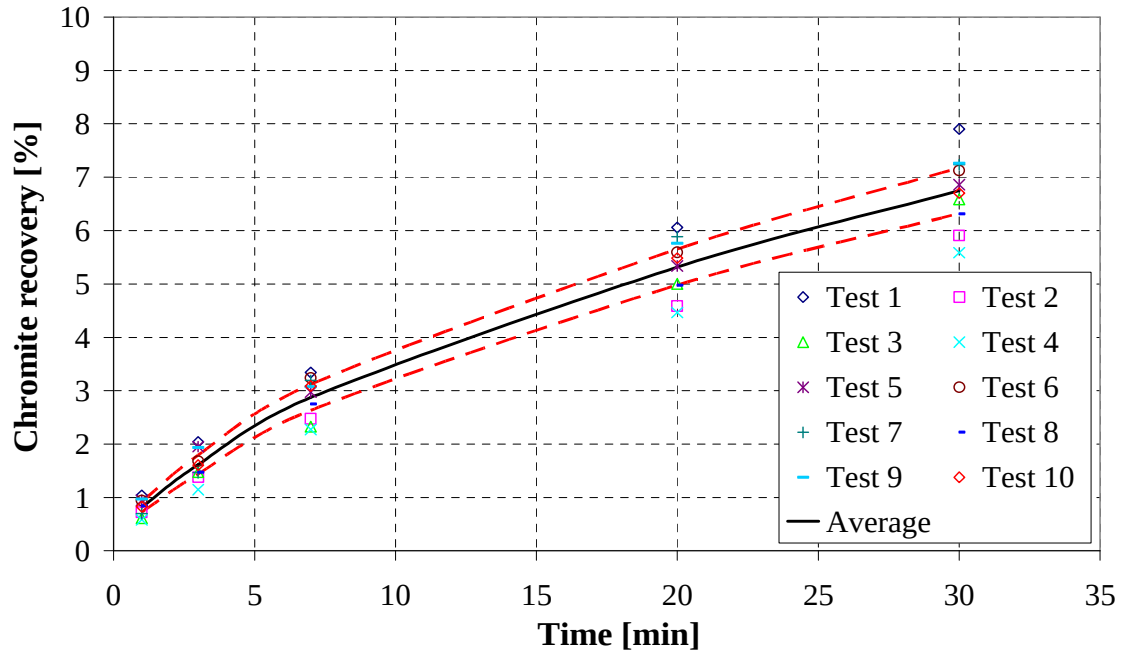


Figure 19: Chromite recovery reproducibility for batch rougher flotation test

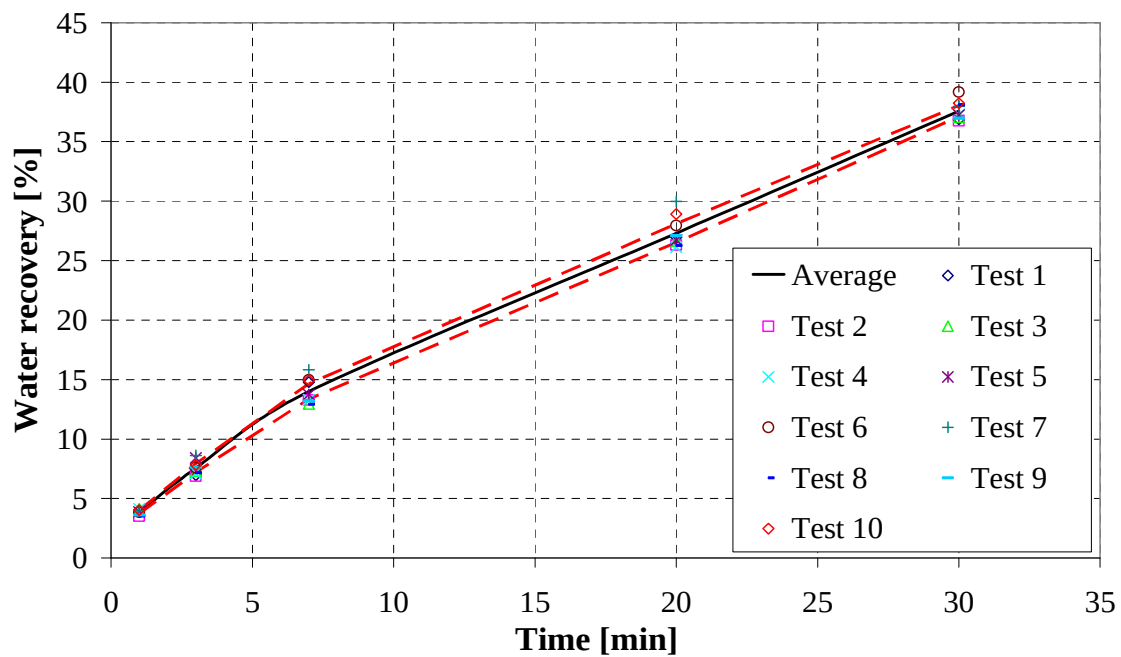


Figure 20: Water recovery reproducibility for batch rougher flotation test

3.3.4. Selection of flotation model for mineral recovery

3.3.4.1. Models evaluated

Fitting of batch flotation models to recovery data is essential for determining flotation parameters such as floatable fractions and rate constants. The parameters can be related to feed properties such as feed chemical assay and mineralogy to explain the flotation recovery

variability observed. However, there are many batch flotation models presented in literature and some of these models were discussed in Chapter 2. A suitable model, which provides a meaningful description of the recovery, for the different species (PGMs, gangue and chromite) in UG2 flotation is required.

Various recovery models will be evaluated statistically to determine the most appropriate model for the rate of recovery of the different species in UG2 flotation. Table 13 shows the flotation models which will be evaluated. Mathe *et al.* (2000) and Vera *et al.* (2002) showed that for batch flotation tests the shallow froth levels give a 100% froth recovery. Hence, in this study modelling of the batch flotation froth will not be considered. The recovery by entrainment will be considered in the evaluation of the different models, and this will be discussed in the next section.

Table 13: Flotation recovery models evaluated

Model	Equations
Classic	$R = R_{\max} (1 - e^{-kt})$
Klimpel	$R = R_{\max} \left[1 - \frac{1}{k_{\max} t} (1 - e^{-k_{\max} t}) \right]$
Second order Klimpel	$R = R_{\max} \left[1 - \frac{1}{k_{\max} t} \ln(1 + kt) \right]$
Second order model	$R = \frac{R_{\max}^2 kt}{1 + R_{\max} kt}$
Kelsall	$R = R_{\text{fast}} (1 - e^{-k_{\text{fast}} t}) + R_{\text{slow}} (1 - e^{-k_{\text{slow}} t})$ $R_{\text{fast}} + R_{\text{slow}} = 100\%$
Modified Kelsall	$R = R_{\text{fast}} (1 - e^{-k_{\text{fast}} t}) + R_{\text{slow}} (1 - e^{-k_{\text{slow}} t})$ $R_{\text{fast}} + R_{\text{slow}} = R_{\max}$

3.3.4.2. Modelling of entrainment in batch flotation

Çilek and Umucu (2001) presented the conventional equation used in modelling batch entrainment recovery:

$$R_e = \text{ENT} \times R_w \quad (3.1)$$

R_e is the recovery by entrainment, ENT is the entrainment factor and R_w is the water recovery. Equation 3.1 is similar to Equation 2.17 presented in Chapter 2 however; ENT is an average entrainment factor for the batch test.

Gülsoy (1999) proposed a modified form of the classic flotation model, which considered the recovery by flotation and entrainment:

$$R = R_{\max} (1 - e^{-kt}) + (1 - R_{\max}) R_e \quad (3.2)$$

Another model commonly used for recovery by flotation and entrainment is:

$$R = R_{\text{float}} + (1 - R_{\text{float}}) R_e \quad (3.3)$$

R_{float} is the recovery by flotation, which can be modelled using any one of the equations given in Table 13.

The major concern in using Equation 3.2 is material which is not recovered by flotation ($1 - R_{\max}$) is assumed to be recovered by entrainment, which is not strictly true. The model assumes that the recovery (R) for a mineral can be 100%, which is unlikely since some non-floatable particles can have a large particle size therefore, it may not be recovered by entrainment. Furthermore, particles which are floatable can be recovered by entrainment (Runge *et al.*, 1998) and this is not considered in the model structure.

Equation 3.3 assumes that floatable material, which is not recovered by flotation, is recovered by entrainment, which is also not strictly true. The model does not consider particles, which have no floatable properties (i.e. hydrophilic) but can be recovered by entrainment. Furthermore, in using Equation 3.3 the floatable fraction is usually determined first, and the entrainment fraction is calculated as an add-on for the overall recovery. Recovery by flotation and entrainment occurs simultaneously therefore, using this approach is also incorrect.

The assumptions made in modelling entrainment using the above equations can result in flotation parameters and rates, which have wrong estimates. A new approach for modelling entrainment in batch flotation is proposed here. The method will be developed for a batch test

that has mineral and water rate recovery data, which is the typical data obtained from batch tests.

Figure 21 illustrates the logic used in developing the batch flotation model. The UG2 feed containing PGMs, gangue and chromite are apportioned into three different classes, called floatable, entrainable and non-recoverable material. The floatable class can be expanded to describe the heterogeneous characteristics of the floatable mineral fraction, due to varying mineral surface properties, particle size and density. However, expanding the floatable fraction depends on the number of data points available from the batch flotation test. A floatable particle can be recovered by entrainment, if the particle detaches from the bubble and goes into the interstitial water between bubbles; or if the particle is trapped in the interstitial water between bubbles, due to the upward movement of the floatable particles. Entrainable particles are assumed to be recovered by the mechanism of entrainment only, which is logical since these particles do not have any floatable properties, according to the model definition of classes. The non-recoverable material refers to material which is not recoverable by flotation and entrainment, and this is generally minerals locked in large unfloatable gangue particles or large gangue particles, which cannot be entrained. The classes described, will be used to develop an approach for modelling entrainment and flotation for a batch flotation test.

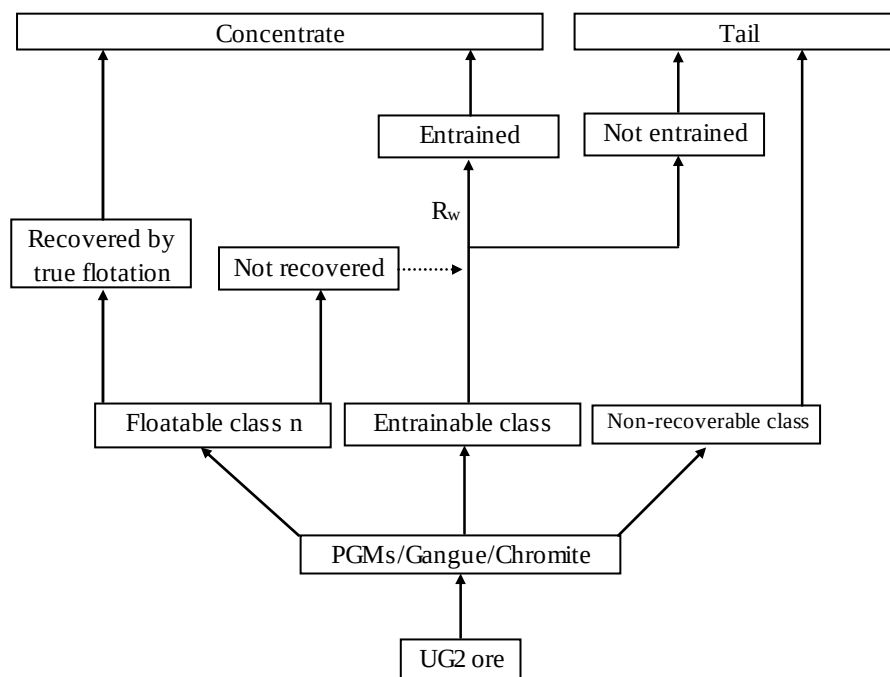


Figure 21: Illustration of mineral classes adopted in modelling batch tests for UG2 ore

Using the definition of floatable, entrainable and non-recoverable material given earlier, the recovery of a mineral (R) can be expressed mathematically as:

$$R = R_{\text{float}} + \int_0^t [(1 - R_{\text{float}}) + R_{\text{ent}}] \times \text{ENT} \times R_w(t) dt \quad (3.4)$$

The mineral recovery is made up of material recovered by flotation (R_{float}), which can be modelled using any of the equations given in Table 13. The floatable material fraction not recovered by flotation at time t ($1 - R_{\text{float}}$) and the entrainable material fraction at time t (R_{ent}) presents the fraction of material which can be entrained. ENT is the average degree of entrainment, and R_w is the water recovery. An integral is used to determine the recovery by entrainment because of the time-varying nature of the batch test. This approach is different from other methods, which use a direct determination of recovery by entrainment, without considering the time-varying characteristic of the batch test.

A suitable batch flotation water recovery model is required for application of Equation 3.4. Two batch models were discussed in Chapter two, and both assume an exponential model structure for the water rate of recovery. For minimalism only one of the models will be discussed. The model proposed by Gülsoy (Equation 2.21 from Chapter 2) has two fitting parameters γ and κ :

$$R_w = \gamma(1 - e^{-\kappa t})$$

The model was applied to the water rate of recovery data, and the model fit and residual plot is shown in Figure 22 and Figure 23 respectively. Examination of both the fit and residual plot indicates that the model is not appropriate for modelling water recovery data. The recovery residuals were calculated by taking the difference between the experimental and model simulated recovery. According to the residual plot, the model under predicts most of the water recovery data and this gives a systematic structure in the residuals. Tsai *et al.* (1998) examined various residual plots and concluded that systematic structures in a residual plot indicate that the model is incorrect for the system and that an alternate model is required.

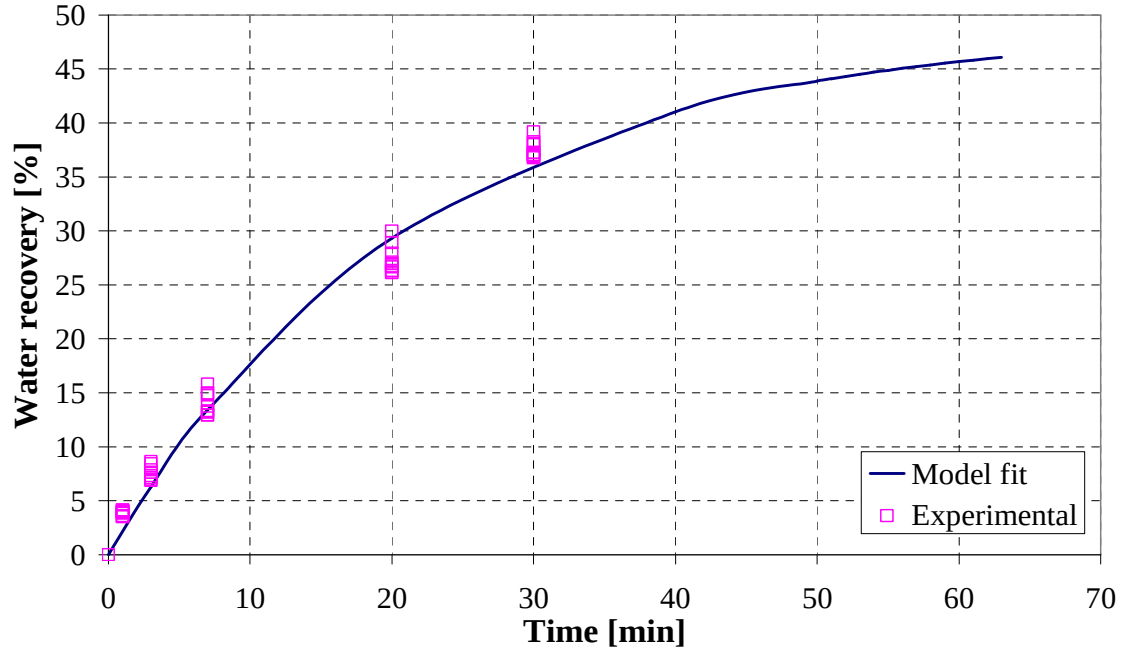


Figure 22: Gülsoy water model fit to experimental data

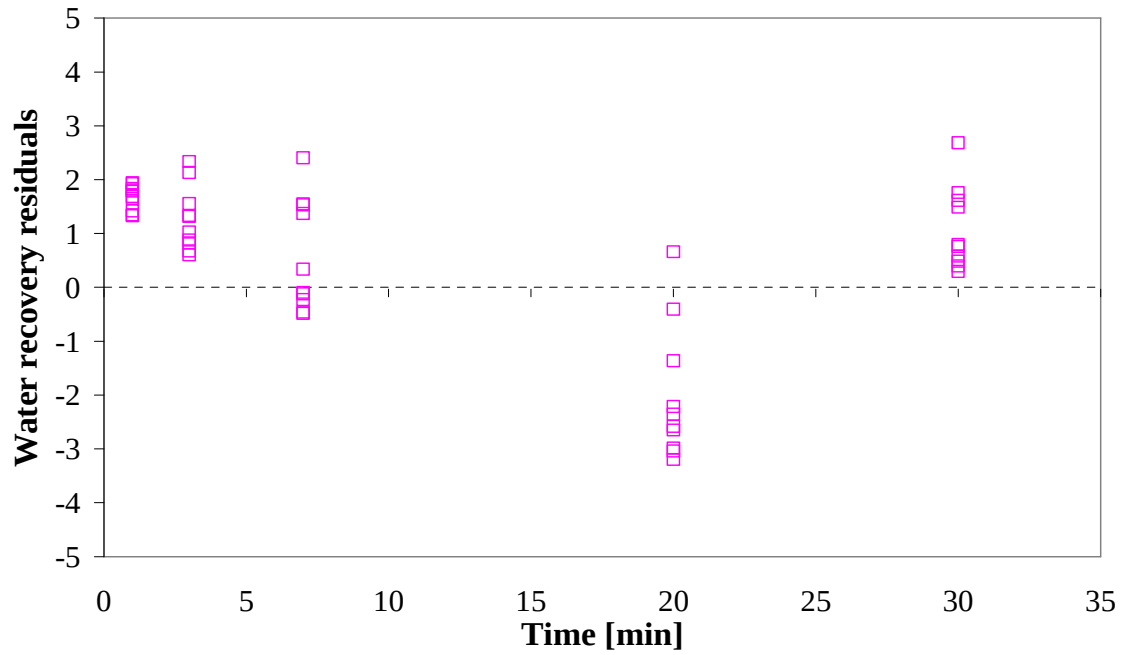


Figure 23: Gülsoy water model residual plot for experimental data

It was decided that a discrete approach will be used to solve Equation 3.4, since there was no suitable model for batch flotation water recovery found in literature. The discrete approach considers the incremental recovery between the timed concentrates collected. The incremental recovery at each time interval can then be summed to obtain the cumulative recovery. The change (Δ) in the mineral recovery for Equation 3.4 between time t_1 and t_2 is:

$$\Delta R = \Delta R_{\text{float}} + \left[\Delta (1 - R_{\text{float}}) + \Delta R_{\text{ent}} \right] \text{ENT} \times \Delta R_w$$

$$\therefore R(t_1, t_2) = R_{\text{float}}(t_1, t_2) + \left[(1 - R_{\text{float}}(t_1, t_2)) + R_{\text{ent}}(t_1, t_2) \right] \text{ENT} \times R_w(t_1, t_2) \quad (3.5)$$

However, the average mineral content in the pulp between t_1 and t_2 must be used to determine the incremental recovery by flotation and entrainment. A log-mean average (Equation 3.6) will be used for the floatable mineral recovery. A linear average (Equation 3.7) will be used for the entrainable mineral recovery since the model structure is linear.

$$(1 - R_{\text{float}}(t_1, t_2)) = \frac{\left[(1 - R_{\text{float}}(t_2)) - (1 - R_{\text{float}}(t_1)) \right]}{\ln \left[\frac{(1 - R_{\text{float}}(t_2))}{(1 - R_{\text{float}}(t_1))} \right]} \quad (3.6)$$

$$R_{\text{ent}}(t_1, t_2) = \frac{R_{\text{ent}}(t_2) + R_{\text{ent}}(t_1)}{2} \quad (3.7)$$

Figure 24 illustrates the calculation of the average mineral content not floated and not entrained in a time interval. The average was used to determine the amount of material available for incremental recovery by entrainment and flotation in a time interval. The maximum recovery is the sum of the floatable and entrainable fraction.

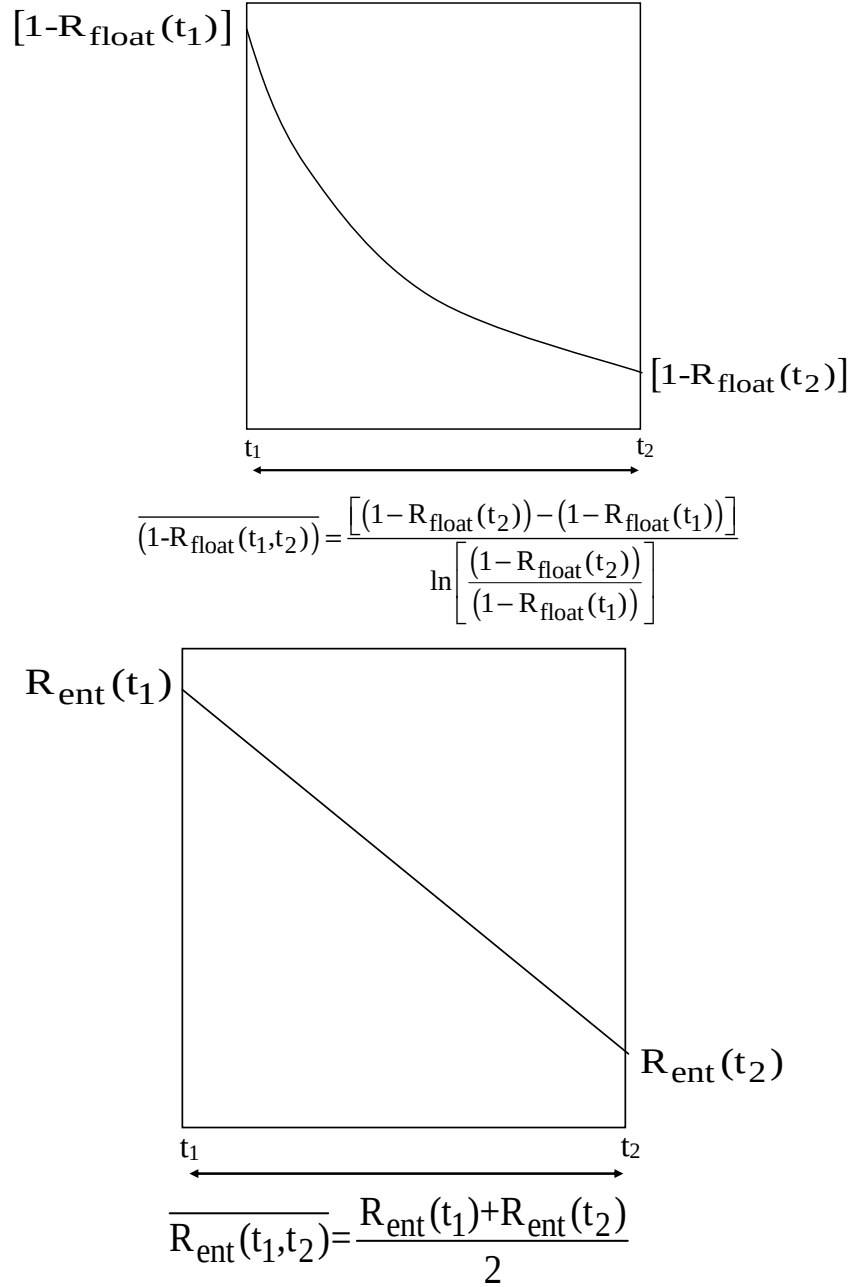


Figure 24: Illustration of average floatable and entrainable material used to determine the incremental recovery

3.3.4.3. Statistical methods used in evaluating recovery models

Various batch flotation models (with and without entrainment) will be examined. Statistical measures will be used to determine the most appropriate model for each species in UG2 flotation. The measures are adjusted coefficient of determination (r_{adj}^2), mean square error of model (MSE), model selection criterion (MSC) and S_p criterion.

The adjusted coefficient of determination describes the fit of a model to the experimental data. Equation 3.8 gives the mathematical form of the statistic, which considers the number

of experimental data points (d_p) and the number of model parameters (m). SS_{res} refers to the sum of the squared residuals between the experimental recovery (R_{exp}) and the simulated recovery (R_{sim}). SS_{total} is the squared error of the model with average experimental recovery $\bar{R}_{exp}$. The statistic weighs the fit with respect to the degrees of freedom for the model. Hence, the benefit of additional model parameters can be assessed from the fit. The statistic varies between 0 and 1. At 0, the model does not fit the experimental data and at 1, the model fits the experimental data completely. The objective is to get a model that fits the data completely i.e. to get close to 1.

$$r_{adj}^2 = 1 - \left[\frac{\frac{SS_{res}}{d_p - m - 1}}{\frac{SS_{total}}{d_p - 1}} \right] \quad (3.8)$$

$$SS_{res} = \sum_{i=1}^{d_p} (R_{exp,i} - R_{sim,i})^2$$

$$SS_{total} = \sum_{i=1}^{d_p} (R_{exp,i} - \bar{R}_{exp})^2$$

The mean square error describes the standard deviation between the simulated data from a model and the experimental data therefore, a smaller value for MSE is preferred. According to Brown (2001), the MSE value can be used to construct the confidence interval for the model parameters using a student's t-distribution. Equation 3.9 gives the mathematical form of the statistic. All parameters have the same meaning as above.

$$MSE = \sqrt{\frac{\sum_{i=1}^{d_p} (R_{exp,i} - R_{sim,i})^2}{df}} \quad (3.9)$$

$df = d_p - m$

There are many criteria available from information theory that can be used when discriminating between models. Some commonly used model selection equations are the Akaike information criterion (AIC), the Schwartz criterion (SC), the model selection criterion

(MSC) and the S_p criterion (Leeb, 2008). Due to commonality between the equations, two model selection equations will be considered. This is the model selection criterion (Equation 3.10) and the S_p criterion (Equation 3.11). The parameters have the same meaning as those presented above. Both equations consider the degrees of freedom in the model being investigated; however, when comparing models a higher value for MSC and a lower value for S_p implies a better model.

$$MSC = \text{Ln} \left(\frac{SS_{\text{total}}}{SS_{\text{res}}} \right) - \frac{2m}{d_p} \quad (3.10)$$

$$S_p = \left(\frac{SS_{\text{res}}}{d_p - m} \right) \left(\frac{d_p - 1}{d_p - 1 - m} \right) \quad (3.11)$$

3.3.4.4. Model selection for the PGM recovery data

a. Model fit

The PGM recovery data from the ten batch rougher tests present in § 3.3.3.3 was used for evaluating the various flotation models (with and without entrainment recovery). The models were fitted simultaneously to the data. This gave an average parameter estimate for the data. The adjusted coefficient of determination was maximised for each constrained regression. The constraints applied were logical bounds such as positive rate constants. The maximum recovery (floatable + entrainment fraction) was fixed to be less than 100% but greater than the recovery at 30 minutes. The recovery at 30 minutes is the last data point for the rate test.

Figure 25 shows the fit of the various models to the experimental data. For models which have entrainment, the recovery estimate by flotation and entrainment is also given in the figure. Figure 26 shows the recovery residual plot for the various models. The fit and residual plots will be used to discuss the models.

In general, models which do not consider entrainment have a poor fit to the experimental data than models which consider entrainment. This may be due to the use of additional parameters for entrainment modelling however; the adjusted coefficient of determination (refer to Figure 27) was larger for the models with entrainment. This implies that the use of the additional parameters, for entrainment modelling, is beneficial for modelling the PGM recovery from

the batch system. Also the incremental water recovery justifies the use of the additional entrainment parameters.

For models without entrainment (a to f from Figure 25 and Figure 26) the fit and residual plot indicates that the models have difficulty in fitting some of the data points. This can be summarised as follows:

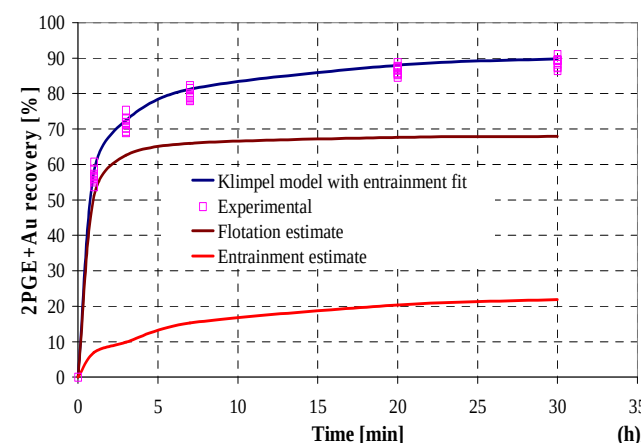
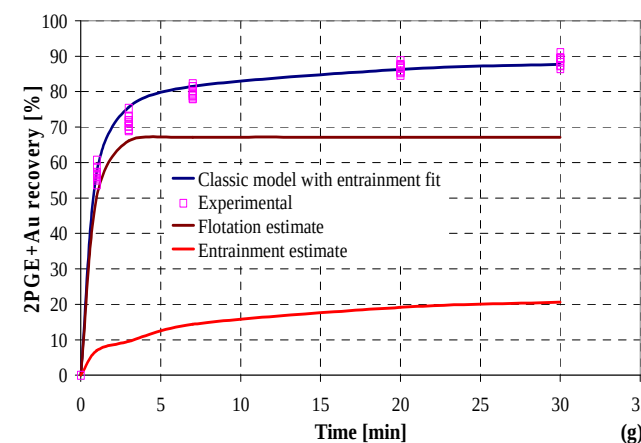
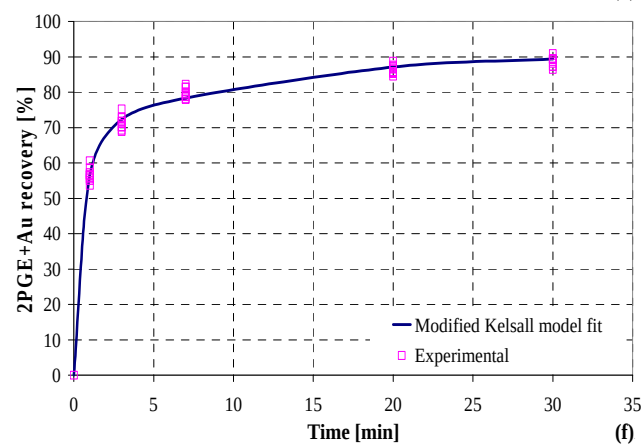
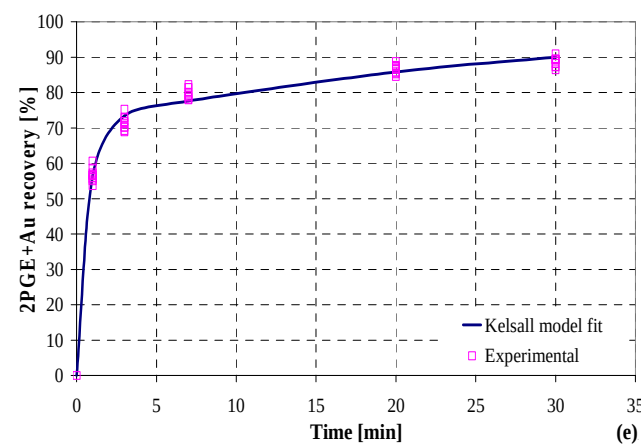
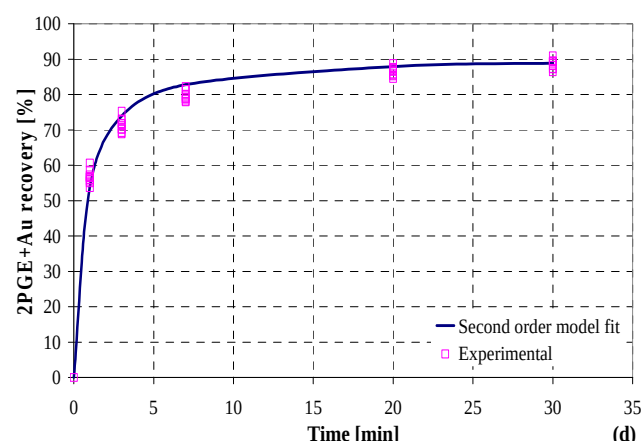
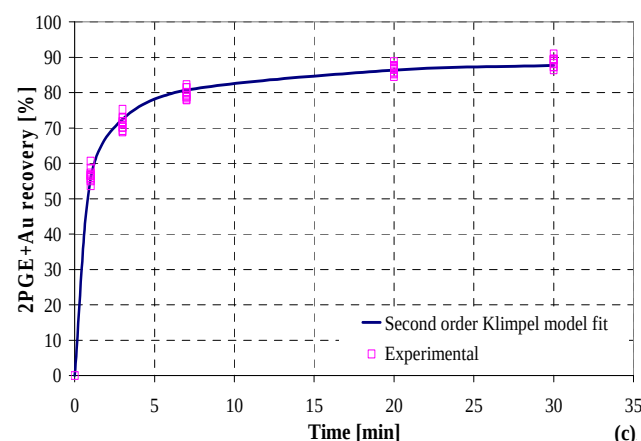
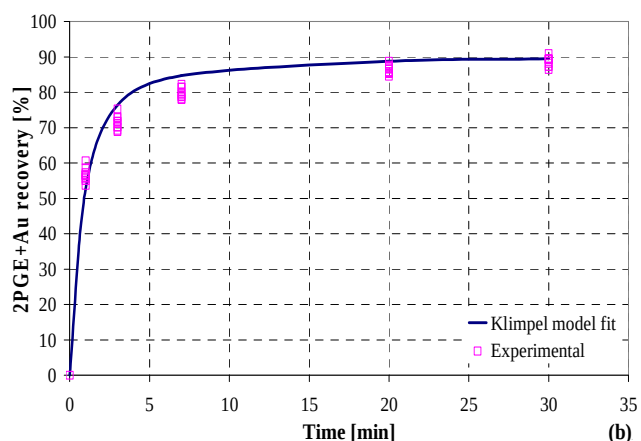
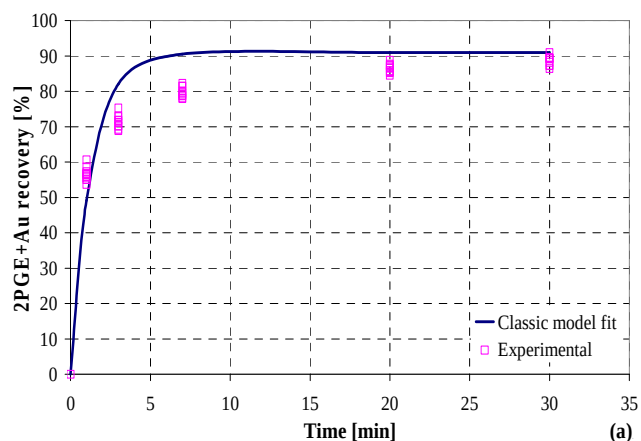
- The classical model has the worse fit; the initial recovery point is underestimated, and all recovery points thereafter are overestimated. The poor fit may be attributed to the existence of another class of recovery (i.e. a second floatable class or an entrainment class), which cannot be approximated by the single curve slope proposed by the model.
- The Klimpel model also has a poor fit to the data, even though the adjusted coefficient of determination indicates that the model accounts for more than 88% of the experimental recovery variation. The model shows similar fitting problems displayed by the classical model hence; the argument of another recovery class may apply here.
- The second-order Klimpel and second order model fit the experimental data better than the previous models, as indicated by the smaller model residuals. However, the models underestimate the recovery of the first concentrate and overestimate the recovery of the second and third concentrates. It is believed that the assumption of the rate of recovery being second-order results in higher recovery estimate for second and third concentrates.
- The Kelsall model (which assumes 100% mineral recovery) and the modified Kelsall model (which assumes a maximum mineral recovery) consider the varying characteristics of the feed material in a flotation test. The feed material contains minerals which have a distribution of floatabilities. This is attributed to variation in surface properties, absorption of reagents and particle size. Both models gave a satisfactory fit to the data.
- However, the problem with applying the Kelsall model is that the assumption of a 100% minerals recovery. This is not entirely correct since minerals can be completely enclosed in large non-floatable gangue thereby, rendering it non-recoverable by flotation and entrainment. The model fit and residual plot indicates that the model provides a good fit to the first two concentrates, which intuitively are fast floating minerals, but the fit worsens for latter recovery points, which are intuitively slow

floating minerals. The poor fit to the latter points is due to the assumption of a 100% mineral recovery; this assumption results in the curve slope being flatter, as a result of the model assuming a larger amount of slow floating minerals. Therefore, the latter data points are underestimated.

- The modified Kelsall model overcomes the model fitting issues displayed by the Kelsall model and fits the experimental data equally well. There is, however, a marginal underestimation of the third recovery point. This may be due to the transitioning from fast to slow floating minerals, which the model has difficulty in differentiating.

The recovery by entrainment was modelled for each flotation model discussed above (g to n for Figure 25 and Figure 26). These models gave a better fit to the experimental data, with smaller recovery residuals than the models which did not consider entrainment. The flotation estimate is greater than the entrainment estimate which is logical, since the PGMs are mainly floatable. The entrainment estimate shows a slight peak in the recovery during the initial phase of the test (i.e. within 5 minutes) the recovery then flattens out approaching a linear approximation. Moys (1984) presented a mass transfer model for the froth and indicated that minerals are carried to an interface between the pulp and froth. For floatable minerals, the transfer of material to this interface results in an enriched mineral zone, which provides more material for entrainment. Therefore, during the initial period of the test the zone is highly enriched with PGMs, which if not attached to bubbles can be carried with the water recovered. Conversely, for latter periods of the test the zone contains fewer minerals, due to most being recovered at the beginning of the test. As a result, the recovery by entrainment is lower.

For the models which assume two floatabilities (Kelsall and modified Kelsall), entrainment was modelled on both floatabilities and on the slow floatability only. Considering entrainment on the slow floating minerals assumes that floatable mineral drops off from the slow floating minerals and is available for entrainment. As a result, there is a lower entrainment estimate, according to Figure 25l and Figure 25n, by comparison to the Figure 25k and Figure 25m. It is difficult to resolve in a batch system if entrainment is mainly from slow floating minerals or from both the fast and slow floating mineral fractions. For this reason, the assumption of entrainment occurring from slow floating minerals will not be considered for PGM recovery modelling.



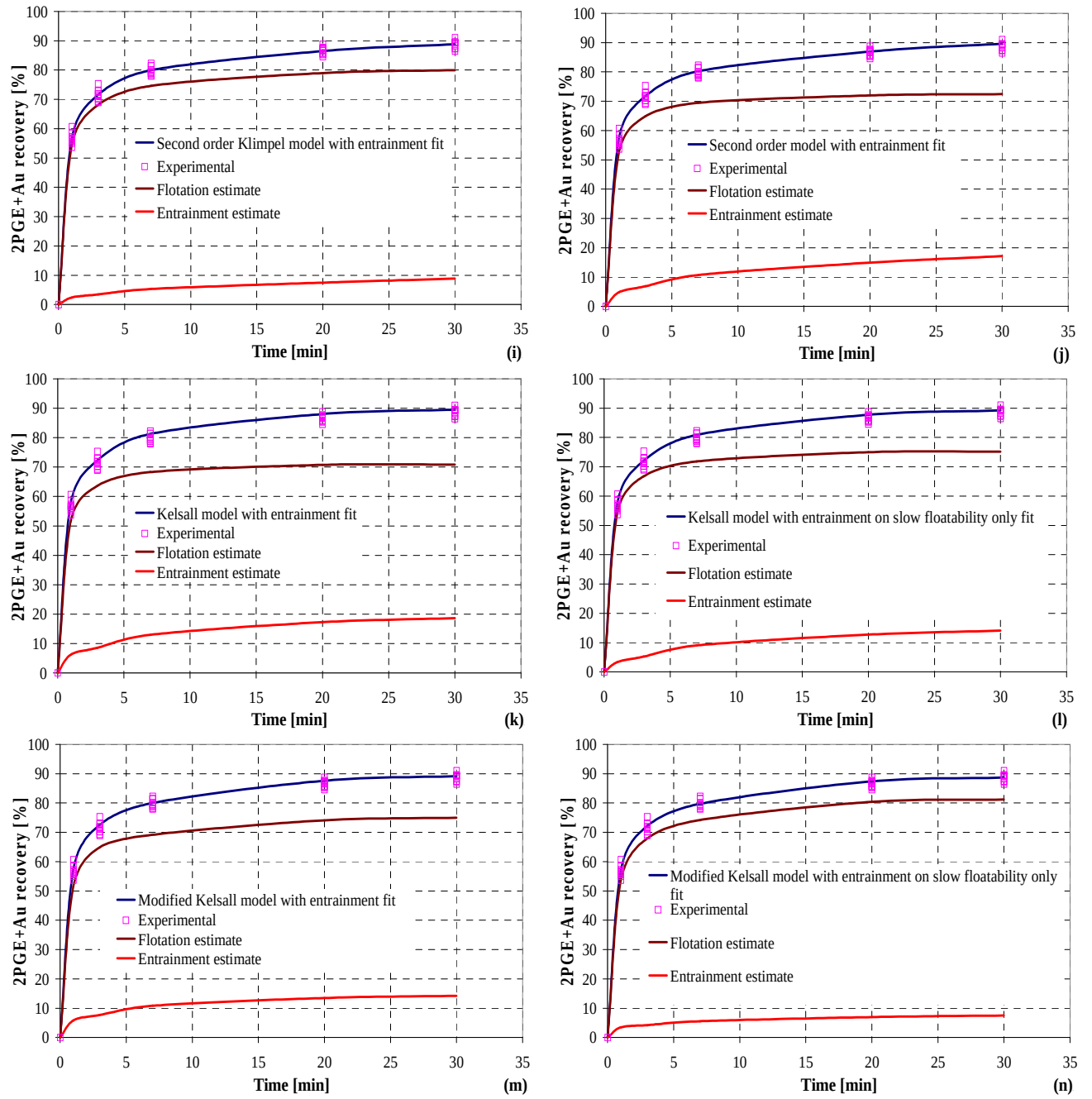
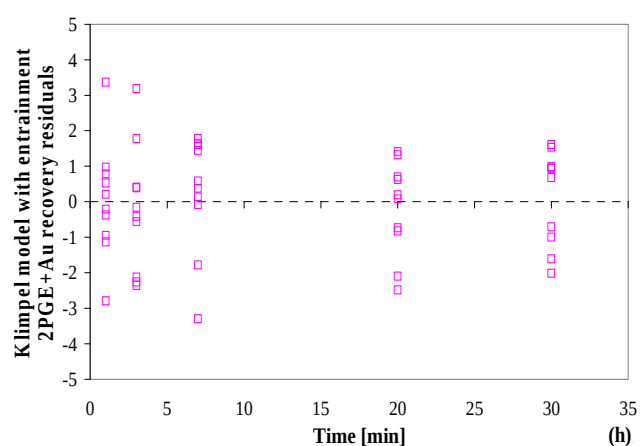
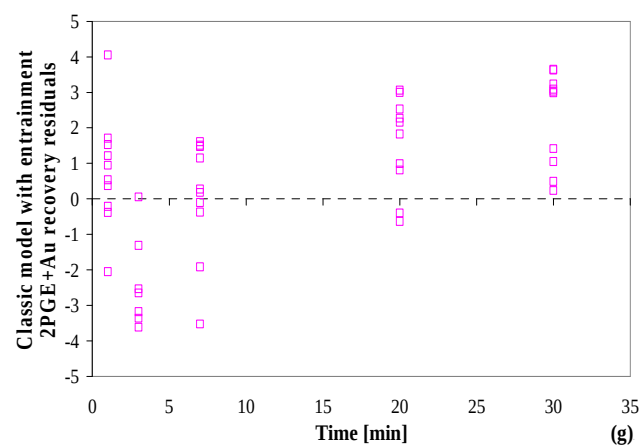
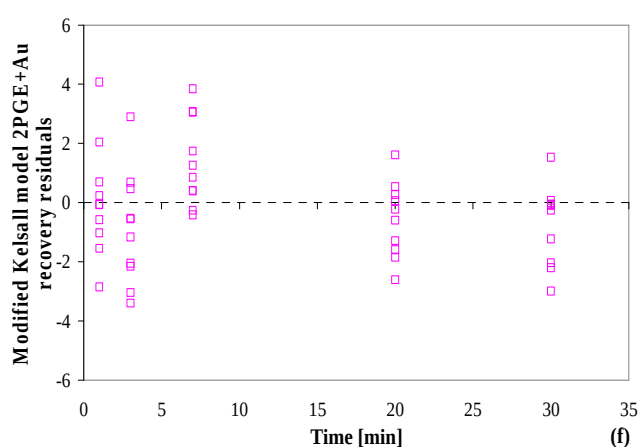
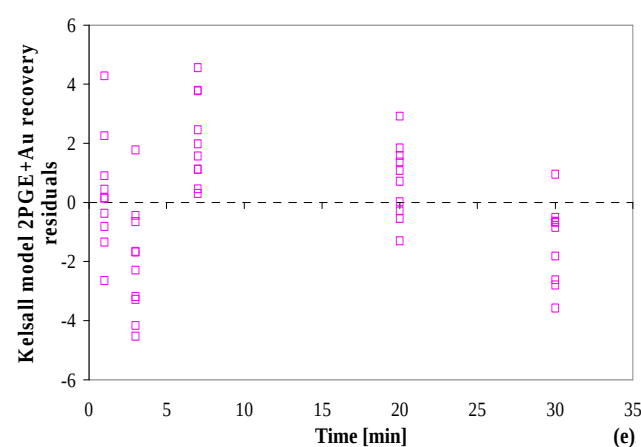
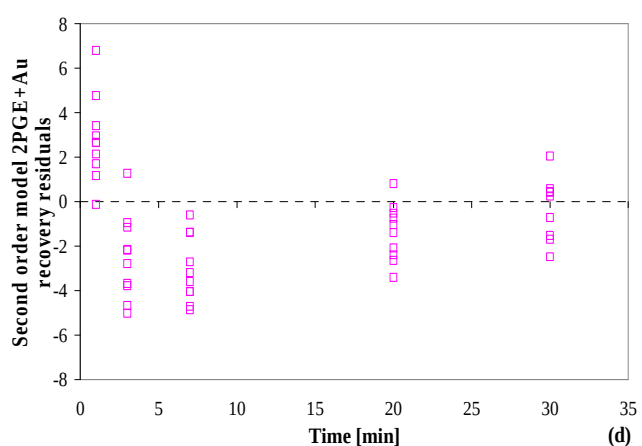
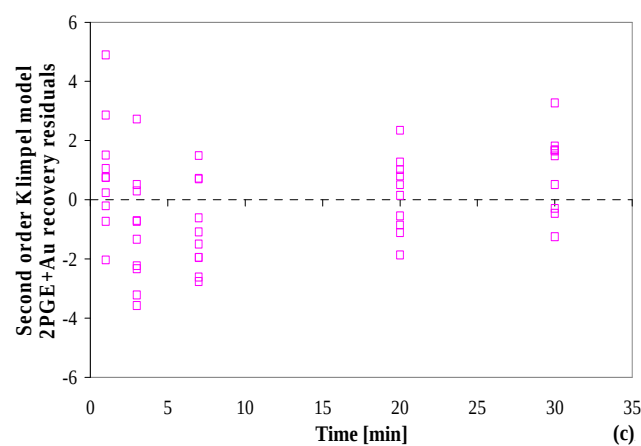
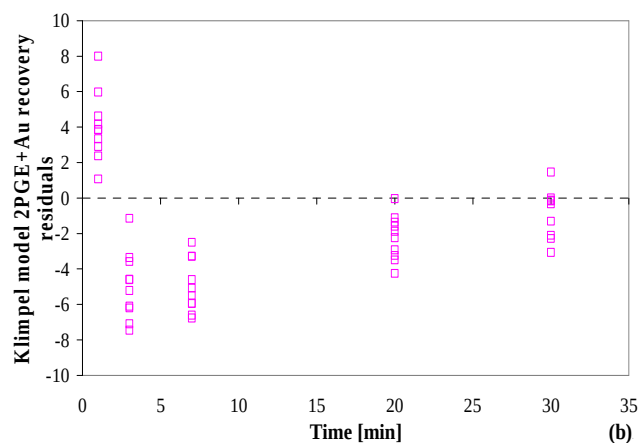
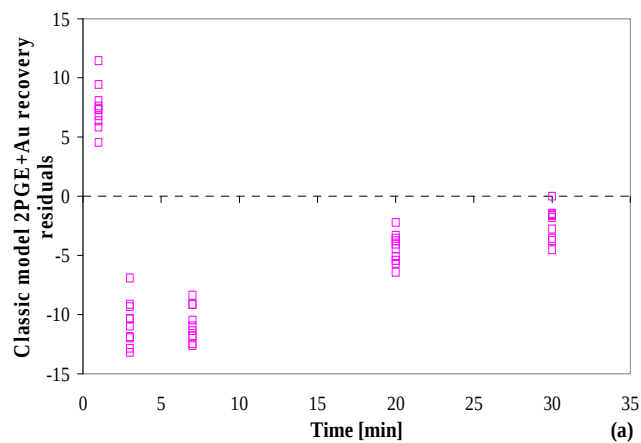


Figure 25: Fit of various flotation models (with and without entrainment) to batch rougher PGM data



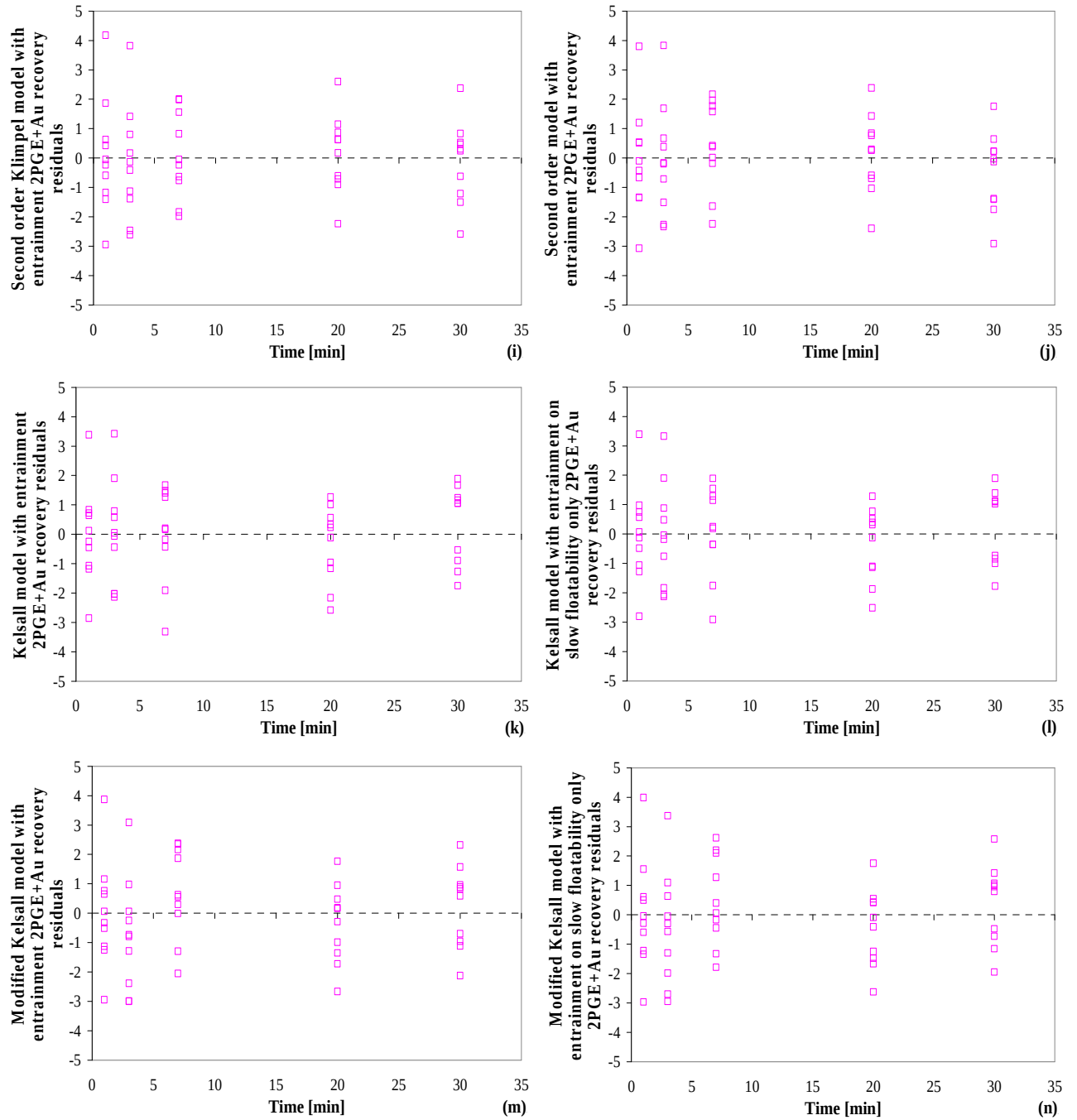


Figure 26: Residual plot of various flotation models (with and without entrainment) to batch rougher PGM data

b. Model statistics

Figure 27 to Figure 30 provides a summary of the statistics for the models evaluated. The adjusted coefficient of determination summarizes the model fit, to each experimental data, with a single index. The larger the index the better the fit, however; this index does not provide information about the fit to each data point. Information on the fit to each experimental data point can be obtained from the residual plot which was discussed earlier. It

follows from Figure 27 that the models with entrainment fit the experimental data better than the models without entrainment. The models, which considered entrainment, accounted for more than 90% of the variation in the experimental data. Furthermore, aside from the classic model with entrainment (which has a poor fit); no particular model stands out from the entrainment models, since all have similar fitting coefficients. Hence, other statistics will be examined.

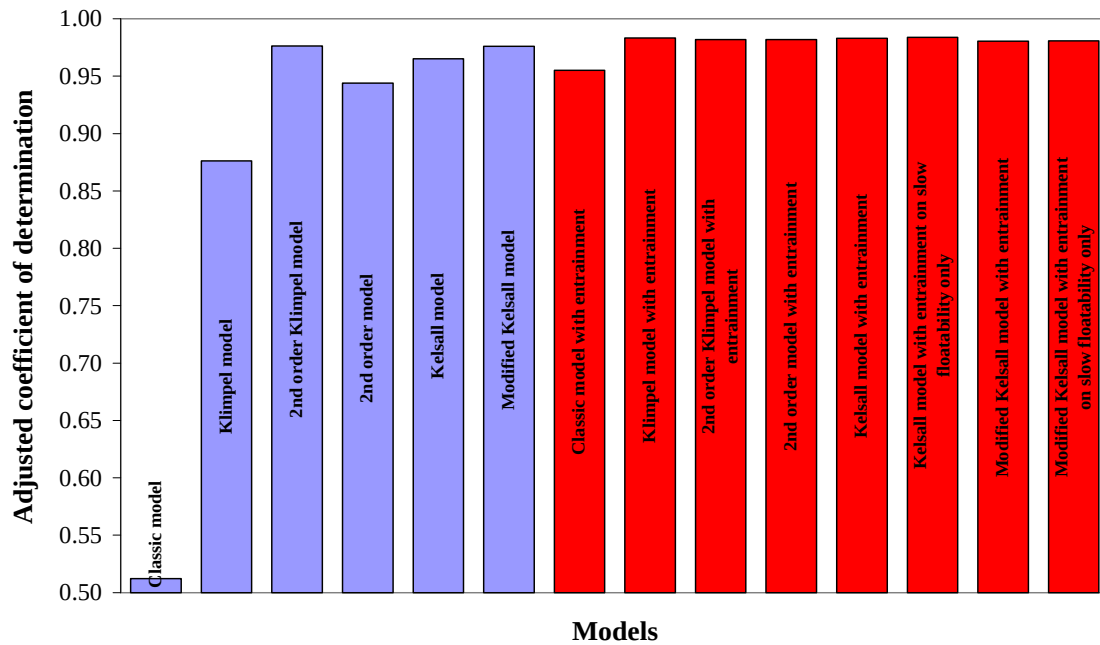


Figure 27: Comparison of adjusted coefficient of determination for models fitted to PGM recovery data

Figure 28 shows the model squared error. The models without entrainment have large model errors, while the models with entrainment have small model errors. Aside from the classic model with entrainment the other entrainment models have similar model errors. The Klimpel model with entrainment and modified Kelsall with entrainment have marginally lower model errors than the other models.

Figure 29 and Figure 30 compares the model selection criterion (MSC) and S_p criterion for the various models respectively. According to both statistics, the entrainment models are better suited for representing a batch system. However, the models with entrainment have similar values for both statistics, and no model stands out completely from the rest.

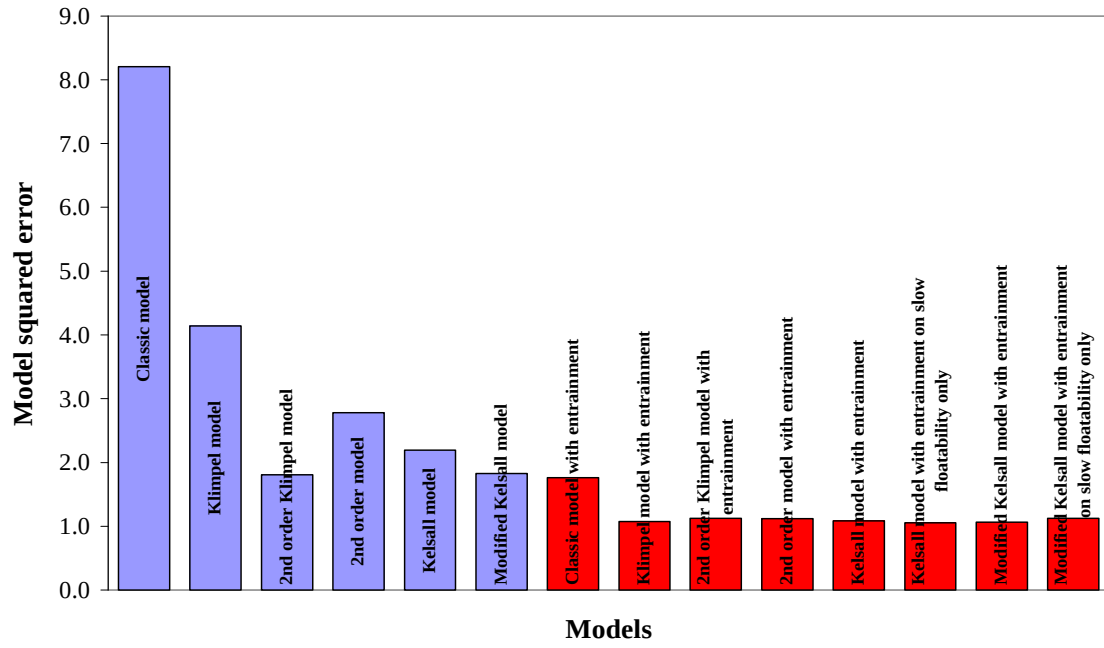


Figure 28: Comparison of MSE for models fitted to PGM recovery data

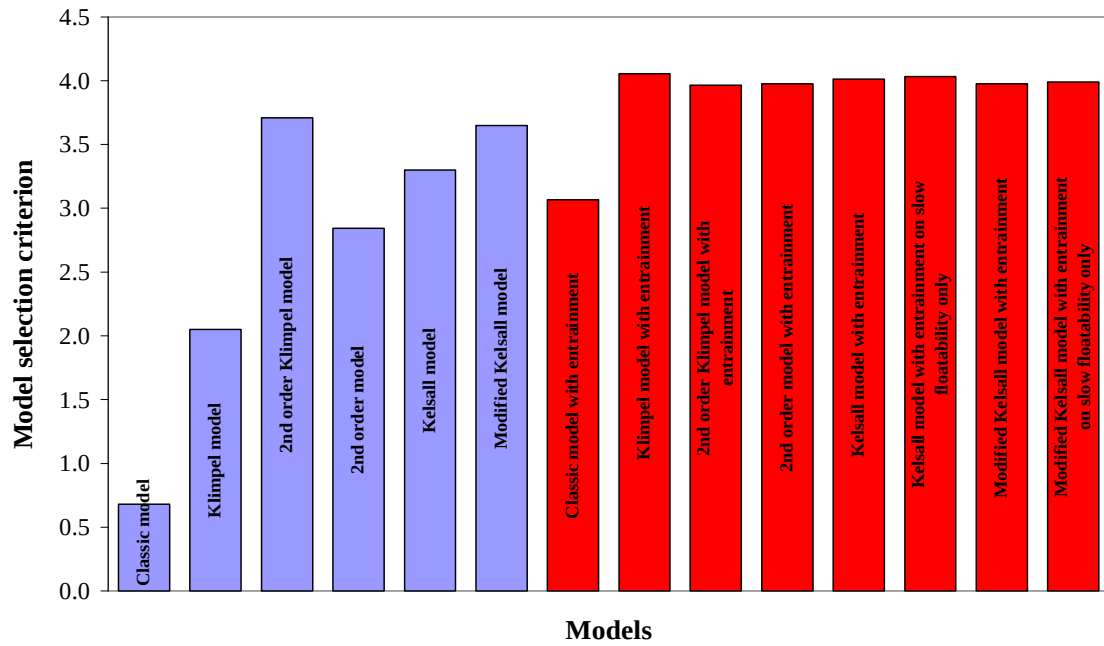


Figure 29: Comparison of MSC for models fitted to PGM recovery data

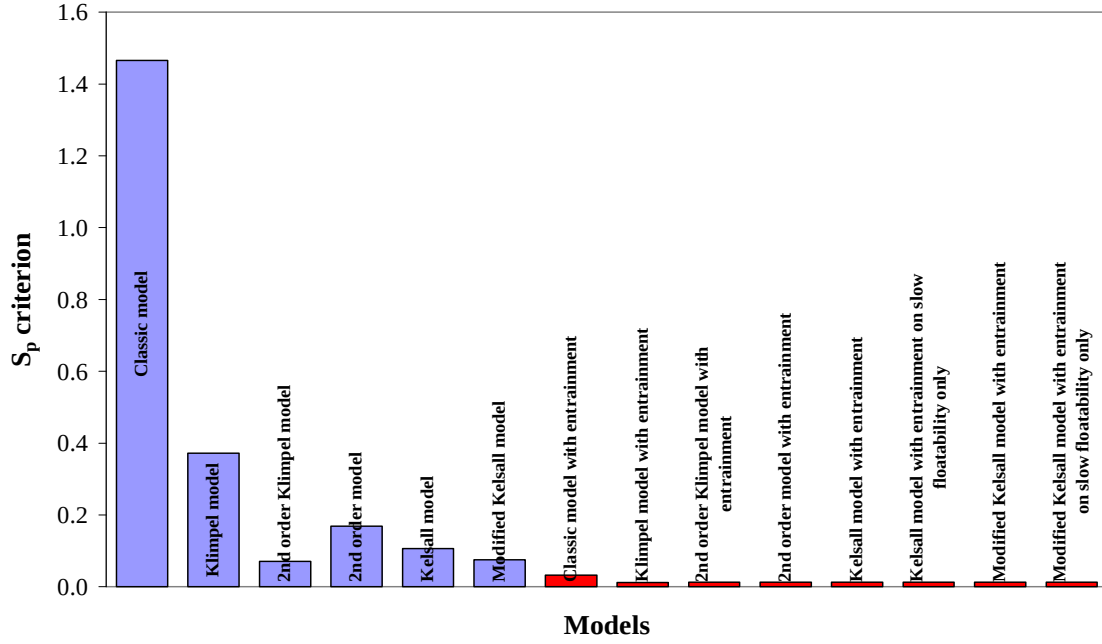


Figure 30: Comparison of S_p criterion for models fitted to PGM recovery data

In summary, the models which consider entrainment provide a better description of the batch PGM recovery. The statistical measures used to differentiate between models gave similar results for most of the entrainment models. Therefore, an intuitive argument can be used to select the most appropriate batch PGM recovery model. The modified Kelsall model considers the heterogeneous characteristics of the feed and has a maximum recovery term which implies that some PGMs are non-recoverable. The other models do not consider the heterogeneous characteristics of the feed. The Kelsall model considers a heterogeneous feed but assumes that all PGMs are recoverable, which is unlikely. Hence, the modified Kelsall model will be used for modelling PGM recovery. The parameter estimates for all models with 95% confidence intervals are given in Appendix A5.

3.3.4.5. Model selection for the gangue recovery data

a. Model fit

Figure 31 and Figure 32 shows the fit and residual plot for the various models. All models fit the experimental data well, since the adjusted coefficient of determination indicates that the models account for more than 90% of the variation in the experimental data.

The models with a single floatable fraction and no entrainment show a good fit to the experimental data. But, examination of the residual plot indicates that these models are not

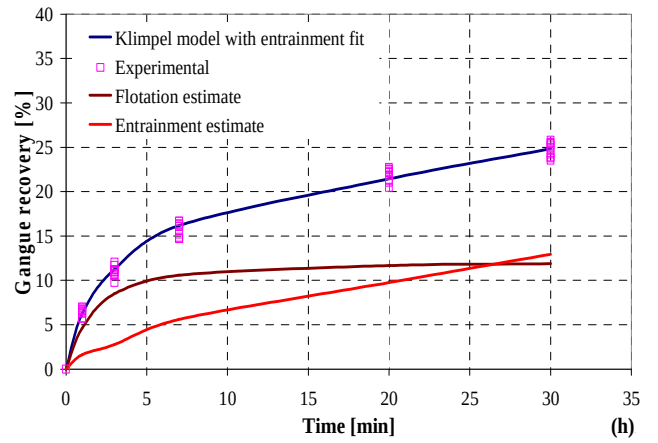
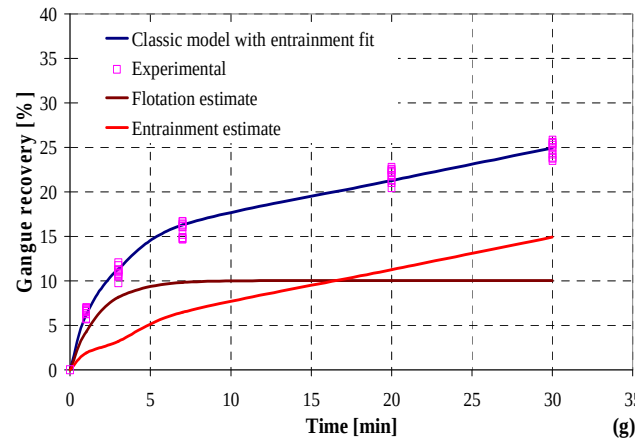
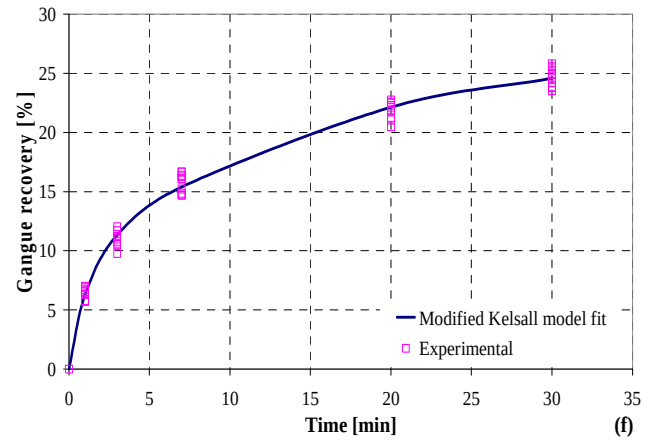
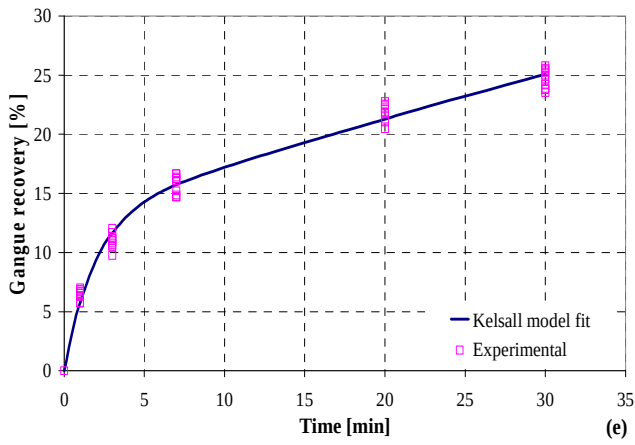
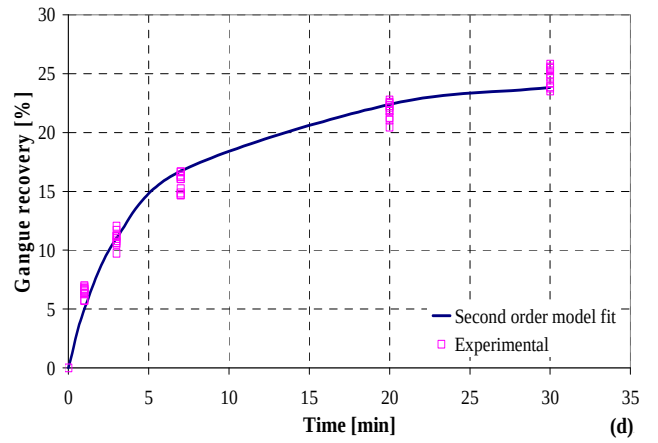
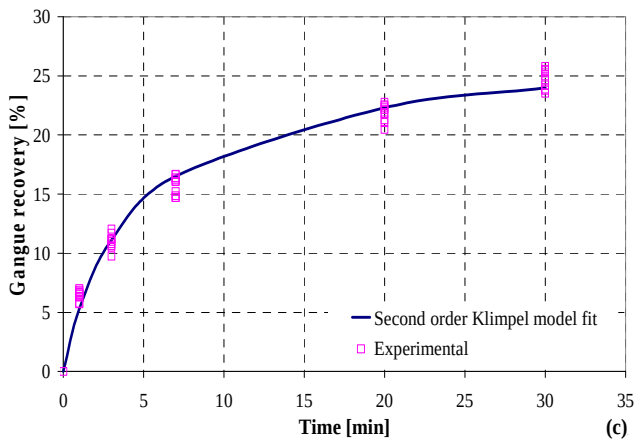
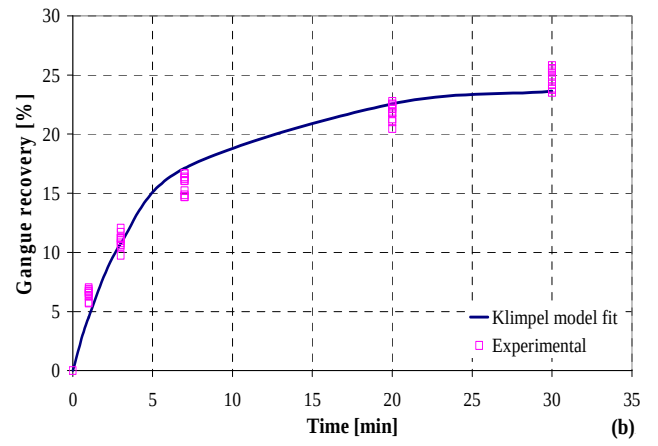
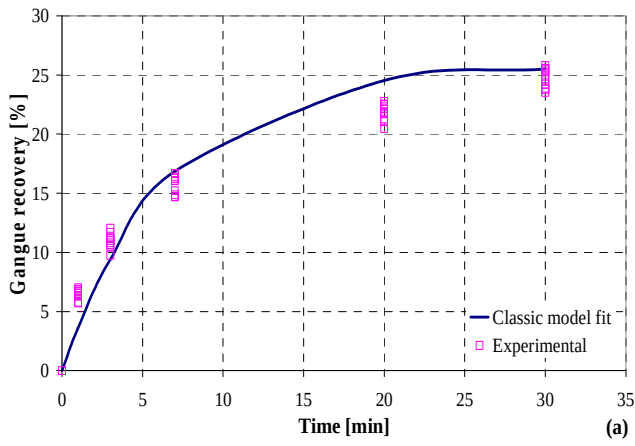
appropriate for modelling gangue recovery. The models have a systematic pattern in the residuals, i.e. the model underestimates the recovery in the initial parts of the test and then compensates by overestimating other recovery points. As mentioned earlier, systematic patterns in the residual plot indicate that the model is incorrect for the system being investigated.

The distributed models (Kelsall and modified Kelsall model) overcome the systematic residual pattern observed for the single floatability class, by assuming the latter recovery can be modelled as slow floating material. However, this raises the question of whether the material is slow floating or recovered by entrainment. The recovery by entrainment may be a more correct recovery mechanism for this material, since the plot is approximately linear between the 7 and 30 minute recovery. This will be examined further by using the entrainment model with each flotation model.

The models which considered entrainment and flotation gave a better fit to the experimental data. Furthermore, these models have smaller residuals, which are randomly distributed across the zero residual line. This implies that entrainment is important in modelling the gangue recovery because, a random distribution of the residuals means that there is no systematic error in the model hypothesis. Therefore, recovery by entrainment is necessary. Likewise for the PGM recovery, the entrainment estimate for gangue shows an increase in the estimate for the initial period. Hence, the same argument can be applied here. During the initial period of the flotation test, large amounts of floatable material (together with non-floatable gangue trapped in water interstitial to bubbles) are carried to the interface between pulp and froth. This results in the interface being enriched with gangue minerals, which provides more material for entrainment. The slope of the entrainment recovery decreases towards the end of the batch test, owing to interface having fewer gangue minerals. The material in the pulp is mainly non-floatable therefore; the depletion of frother and floatable material reduces the chance of this material reaching the interface. As a result, entrainment is lower. Entrainment recovery can increase if the pulp level is increased (which generally occurs for secondary rougher stages in a UG2 plant) but for consistency in the batch test, the level was maintained at a fixed height and with no additional frother added.

The classic model with entrainment gave a good resolution of the overall recovery. The flotation estimate approximates the sharp slope observed for the initial data points, which is

mainly due to floatable gangue like talc. At the latter points, the recovery is mainly due to entrainment. The other models with entrainment show a greater recovery estimate by flotation than by entrainment. This may be possible in flotation. However, in UG2 ores secondary minerals such as talc and amphiboles have floatable characteristics and are recovered quickly (regardless of particle size). But these constitute a small proportion of the feed, i.e. $< 6\%w/w$ (Penberthy, 2001). Hence, most of the gangue is entrainable. The occurrence of a slow floating gangue fraction (according to the Kelsall and modified Kelsall models) is believed to be due to model interaction between the floatable and entrainable parameters. A suitable gangue model will be determined from the model statistics, which will be discussed next.



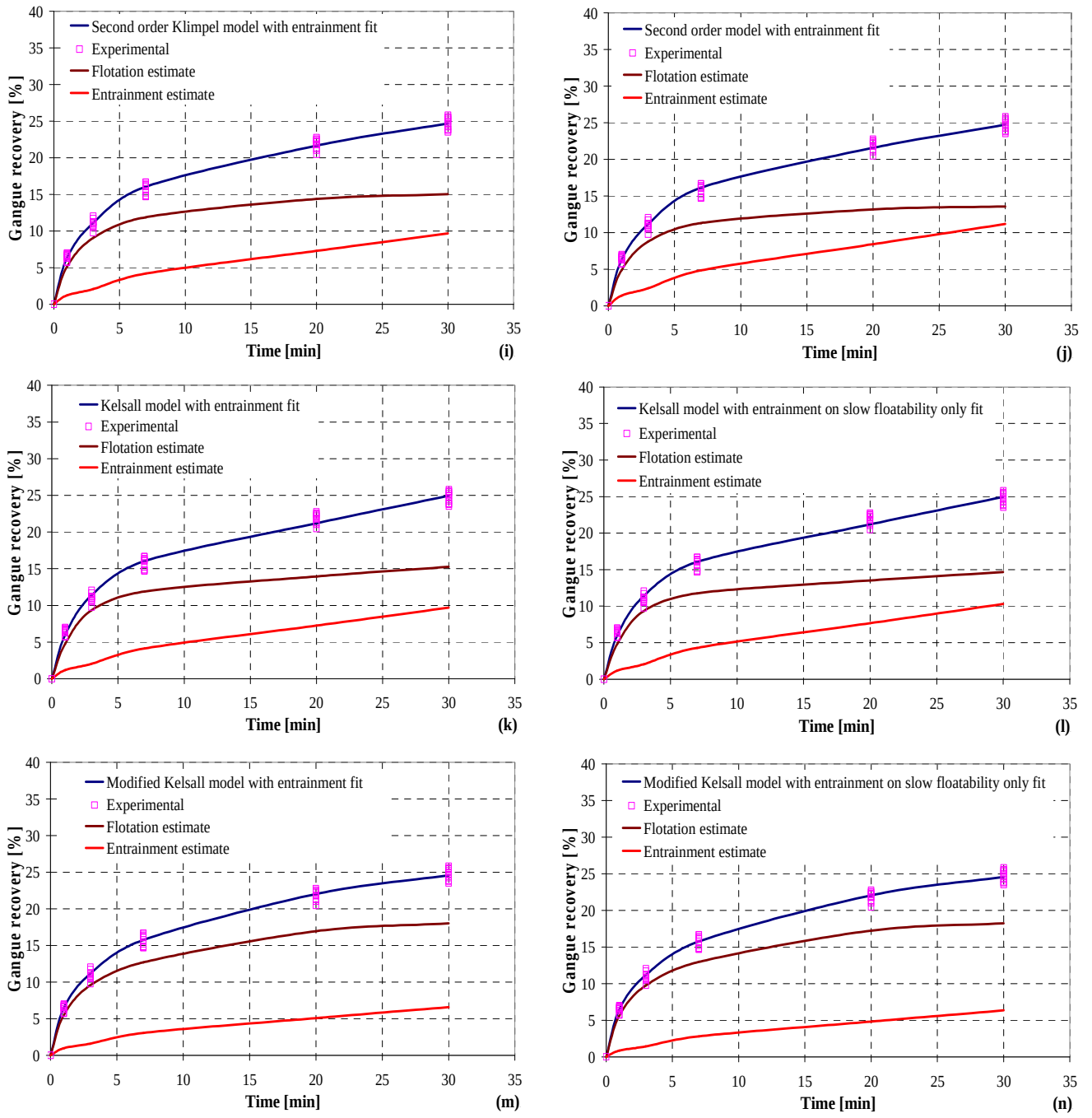
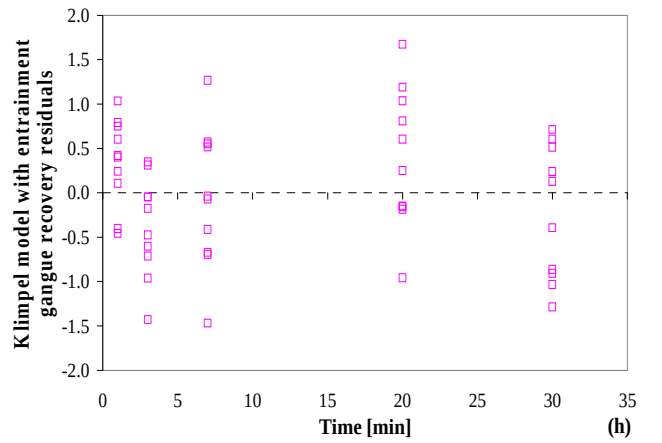
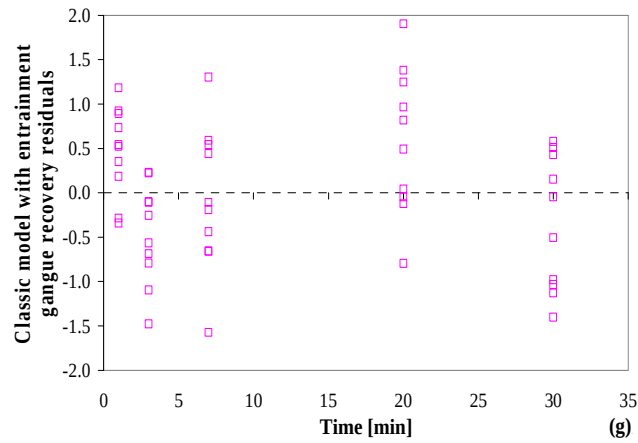
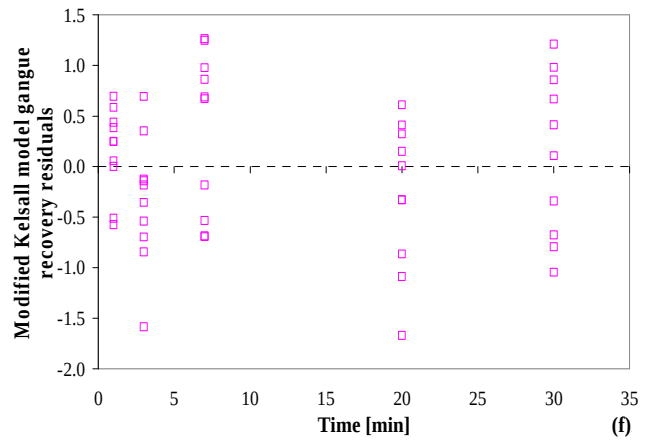
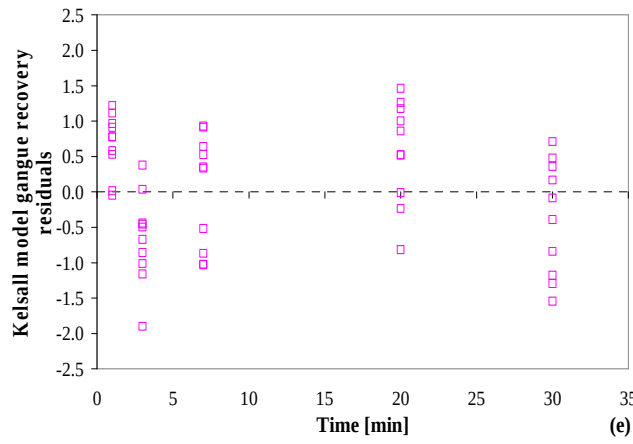
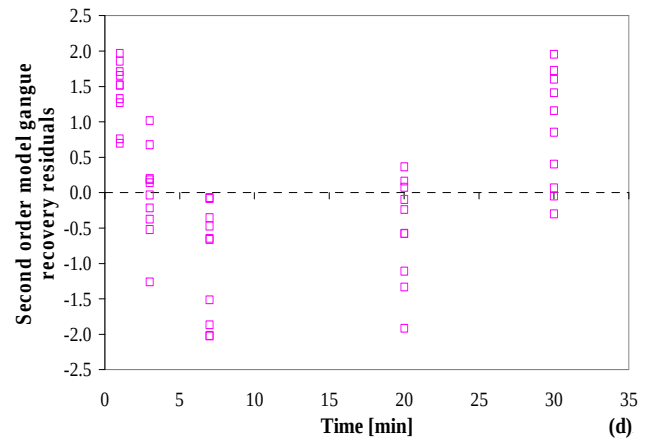
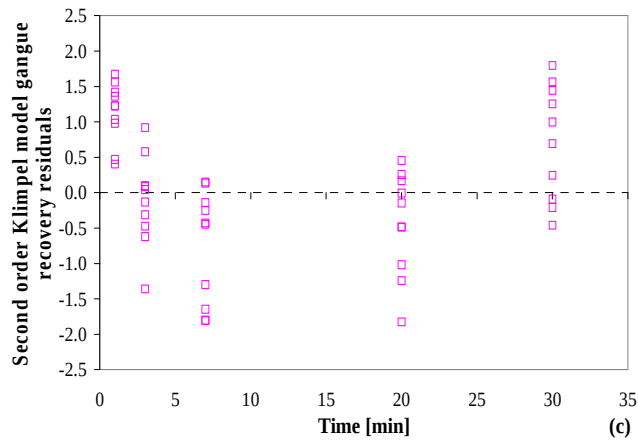
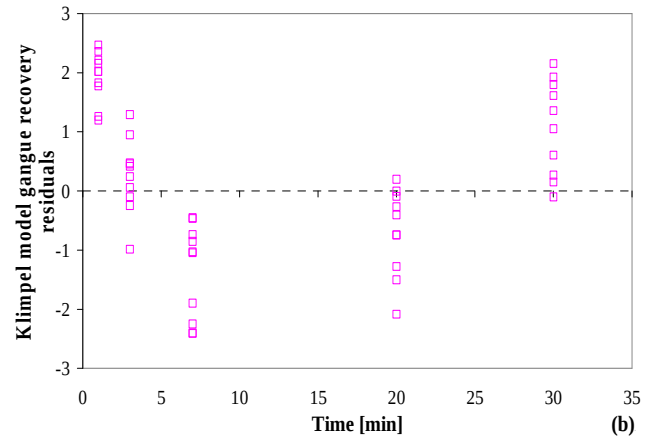
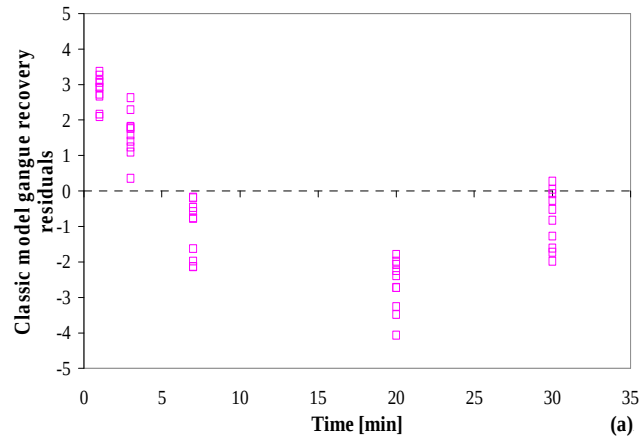


Figure 31: Fit of various flotation models (with and without entrainment) to batch rougher gangue data



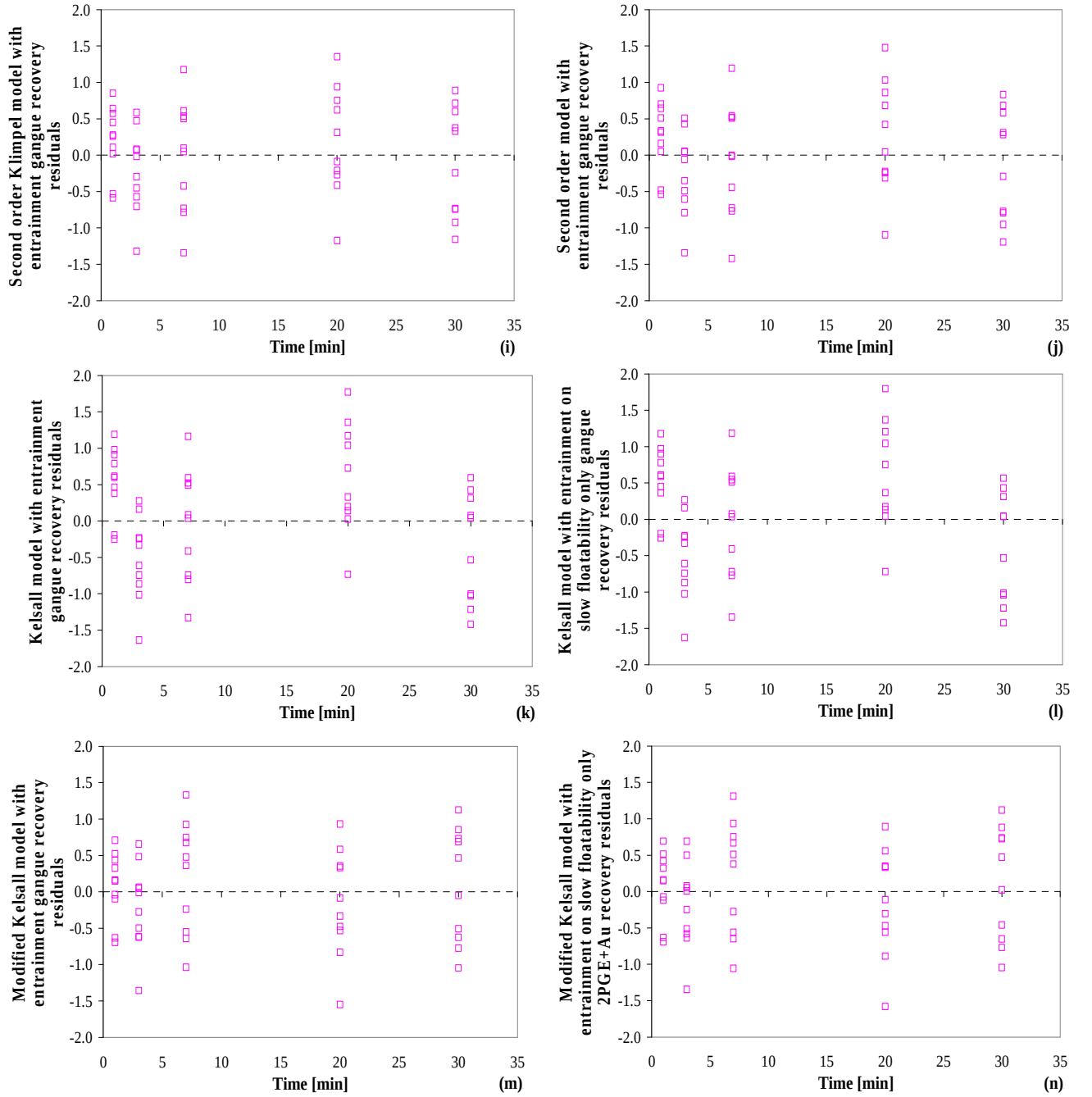


Figure 32: Residual plot of various flotation models (with and without entrainment) to batch rougher gangue data

b. Model statistics

The discussion on the model fit and residual plot, together with the statistics (Figure 33 to Figure 36), will be used to determine the most appropriate model for gangue recovery. The models with entrainment gave a better description and fit for gangue recovery. There are marginal variations in the statistics for the entrainment models. Based on the statistics, the model fit and residual plot, the classic model is better suited for modelling gangue recovery from a batch test, because:

- The model provides a good fit to the experimental data, and the residual plot has small residuals, which are randomly distributed across the zero residual line.
- The model has fewer parameters than the Kelsall model with entrainment and the modified Kelsall model with entrainment and provides a similar fit to the experimental data.
- The model gives a logical description of the gangue recovery by resolving the recovery into a floatable fraction, which is dominant in the initial part of the test, and an entrainment fraction which is dominant towards the end of the test.

The parameter estimates for each model together with the confidence interval is presented in Appendix A5.

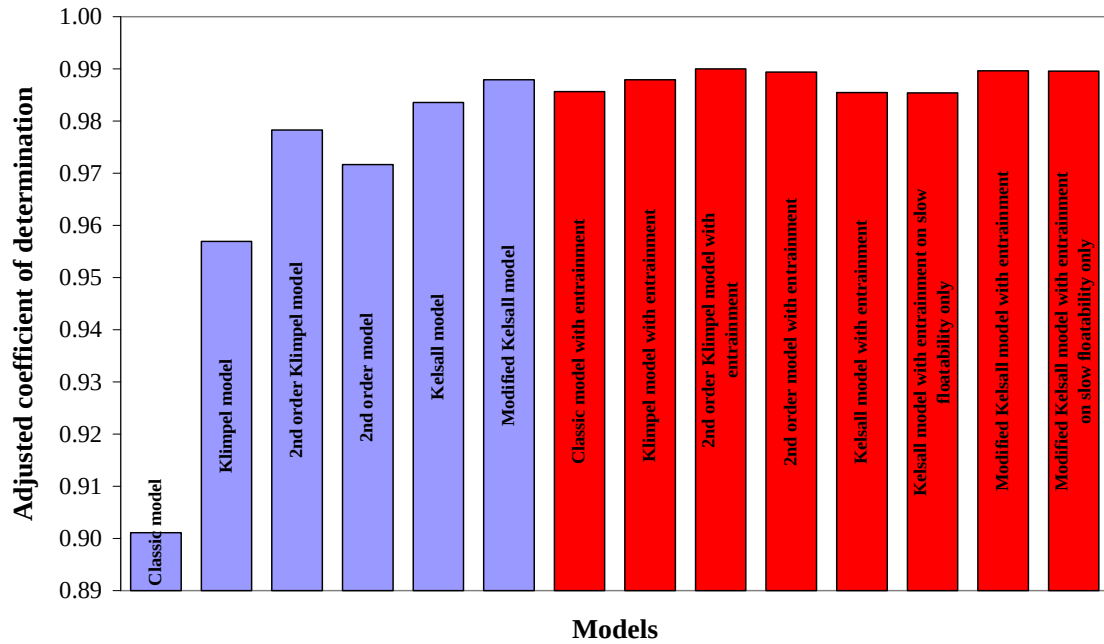


Figure 33: Comparison of adjusted coefficient of determination for models fitted to gangue recovery data

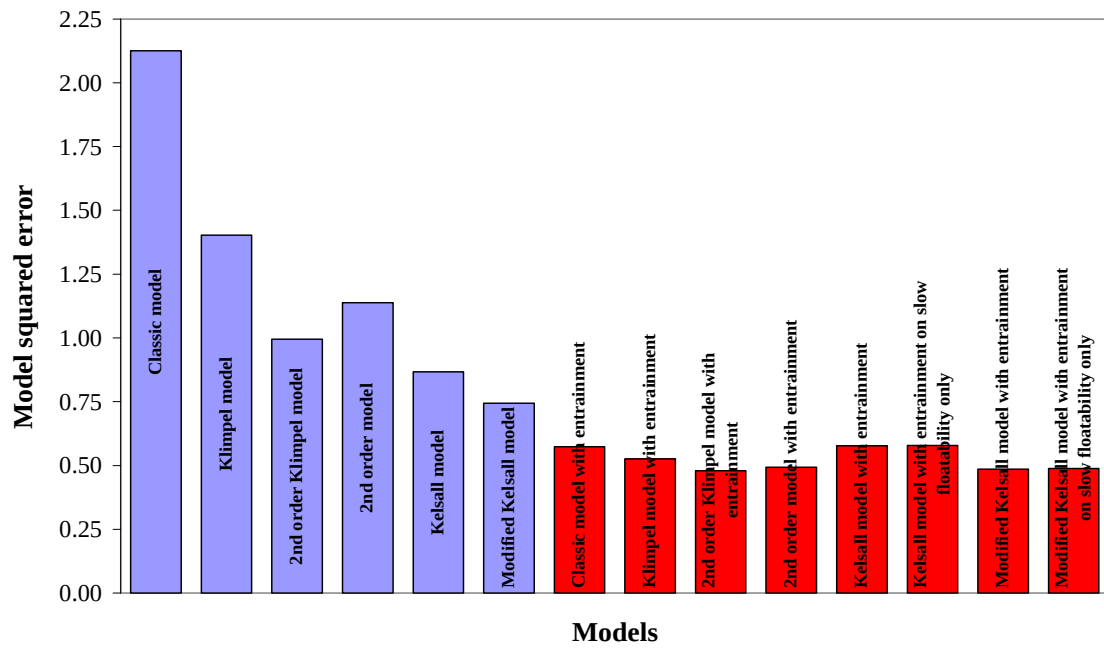


Figure 34: Comparison of model squared error for models fitted to gangue recovery data

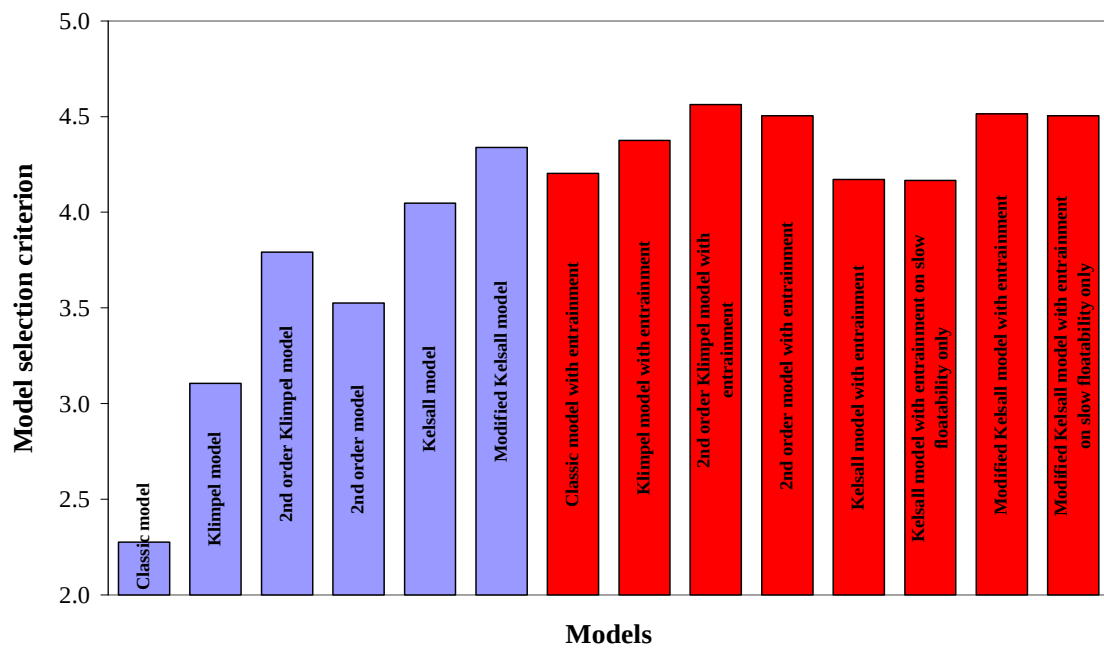


Figure 35: Comparison of model selection criterion for models fitted to gangue recovery data

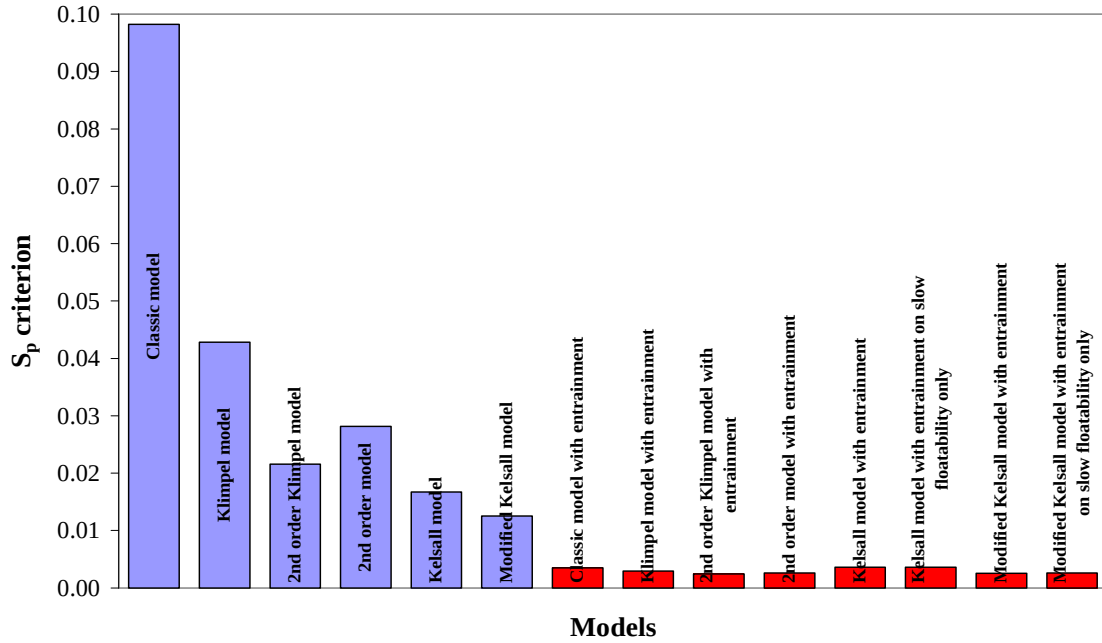


Figure 36: Comparison of S_p criterion for models fitted to gangue recovery data

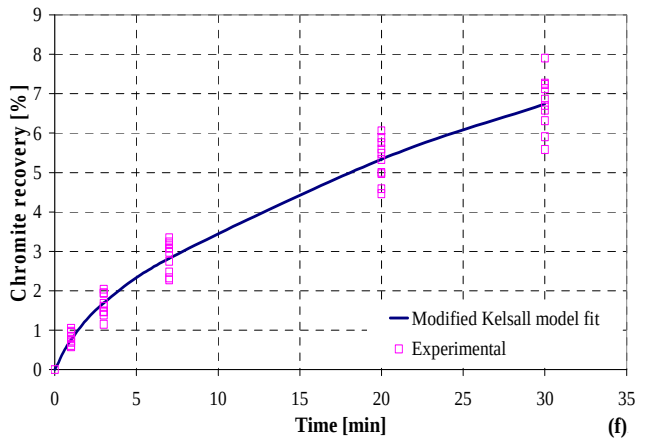
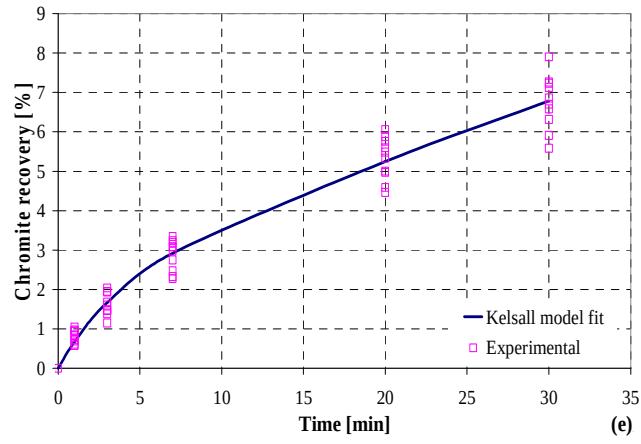
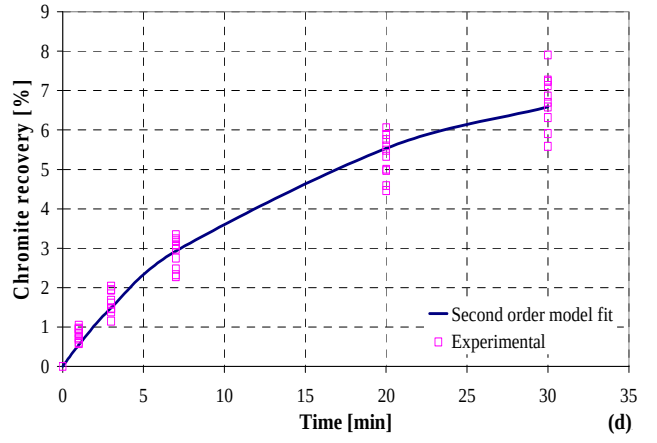
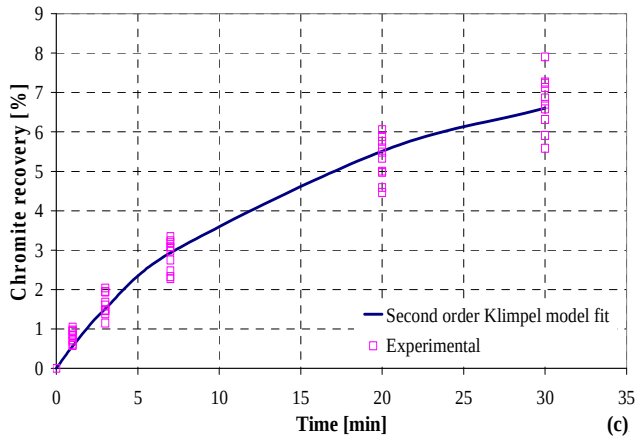
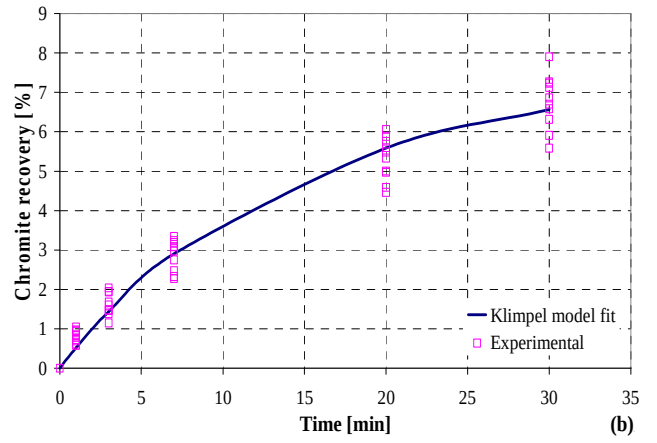
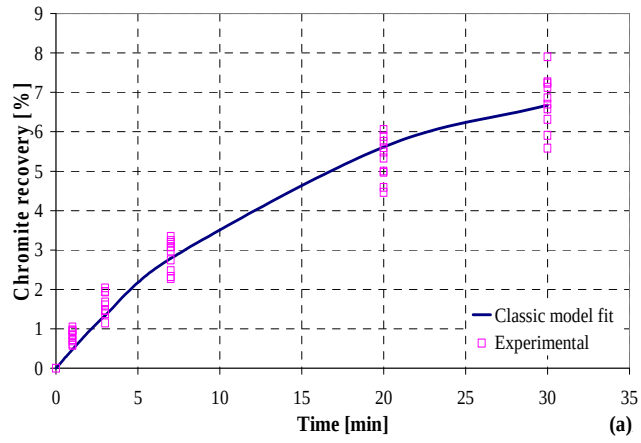
3.3.4.6. Model selection for the chromite recovery data

a. Model fit

Batch flotation models (with and without entrainment modelling) were fitted to rougher chromite recovery data. An additional model which assumed that all chromite is recovered by entrainment was also examined.

Figure 37 and Figure 38 shows the model fit to the experimental data and the residual plot. The models with entrainment have a better fit to all the experimental recovery points. The model, which considers recovery by entrainment only, fits the experimental data well, however; the model residual plot indicates a poor fit for the initial data points. There is a variation in the initial period of the test, which results in a peak in the recovery. This may be due to rapid changes in the froth structure, during the initial period, which influences the water recovery and consequently, the chromite recovery. During this period floatable material transports water (in spaces between bubbles) to the froth which is recovered by scraping. The high floatable content transports more chromite and water (in bubble spaces) to the pulp-froth interface. The scraping of the froth results in a sharp increase in the chromite recovery during this period.

Models with more than one floatable fraction and with entrainment were also fitted to the data. The regression did not converge to a solution since either the rate or mass fraction of one of the floatabilities was forced to zero. Mathematically, this indicates that a single floatable fraction is sufficient for representing the initial recovery period and entrainment modelling is sufficient for the latter recovery. This is a logical result since chromite does not have a distribution of floatabilities like PGMs because it is recovered mainly by entrainment. The strong linear approximation obtained for the entrainment estimate for all models justifies the chromite recovery being largely the result of entrainment.



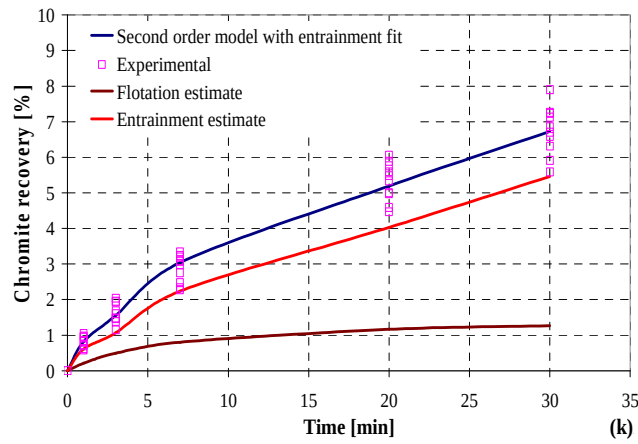
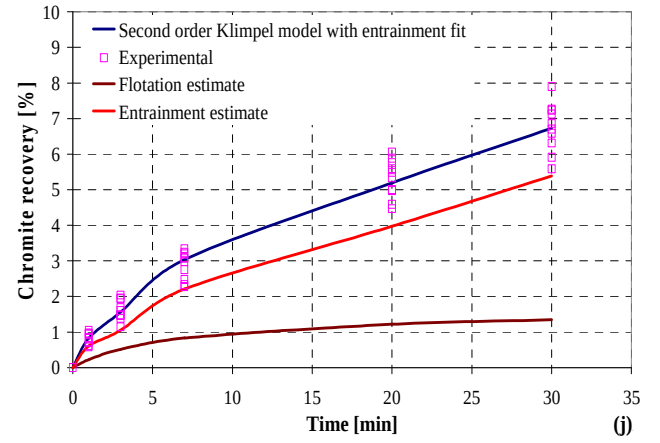
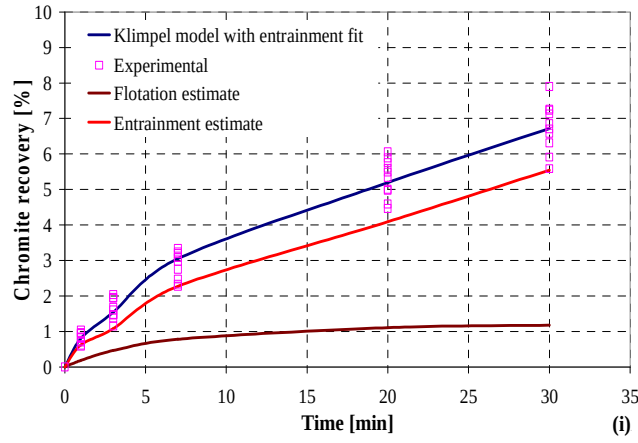
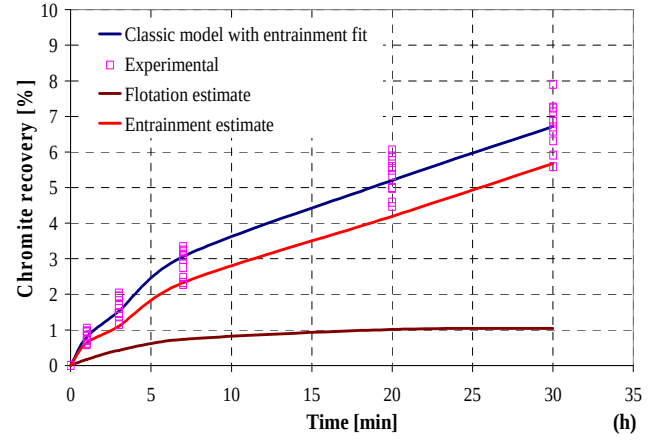
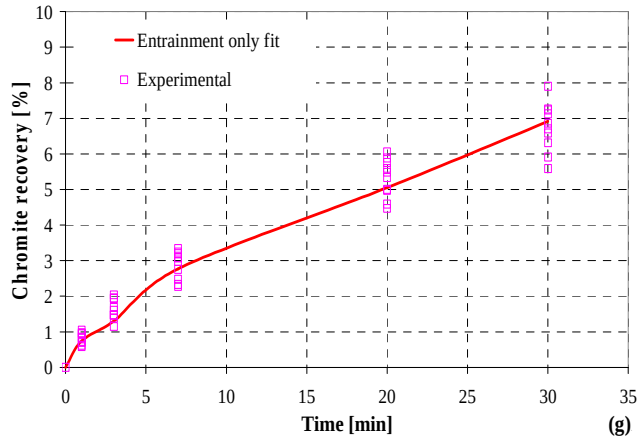
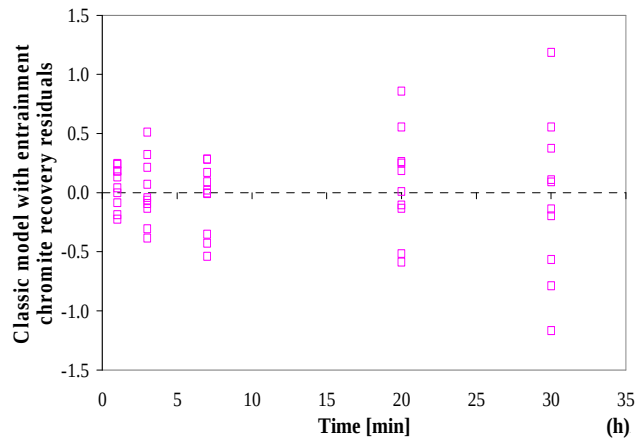
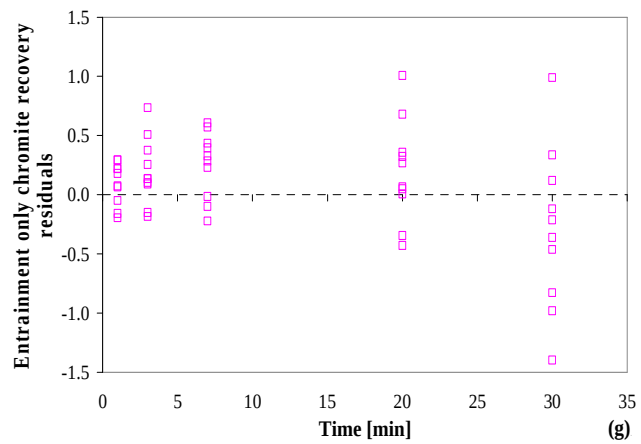
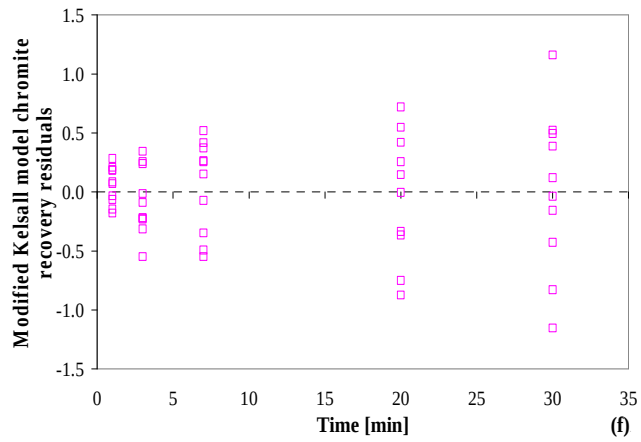
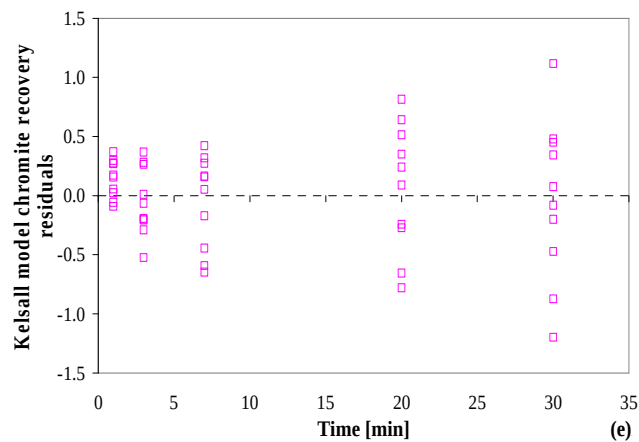
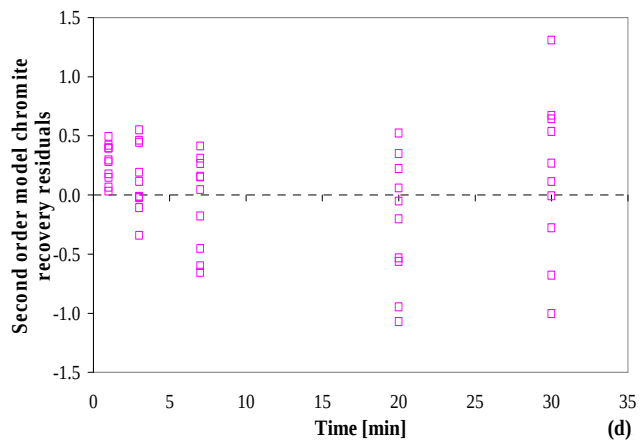
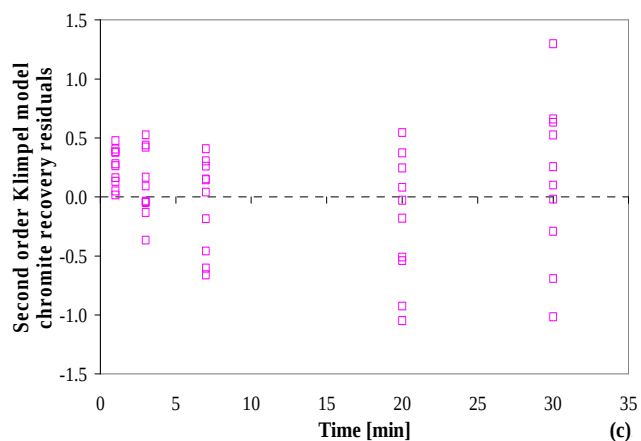
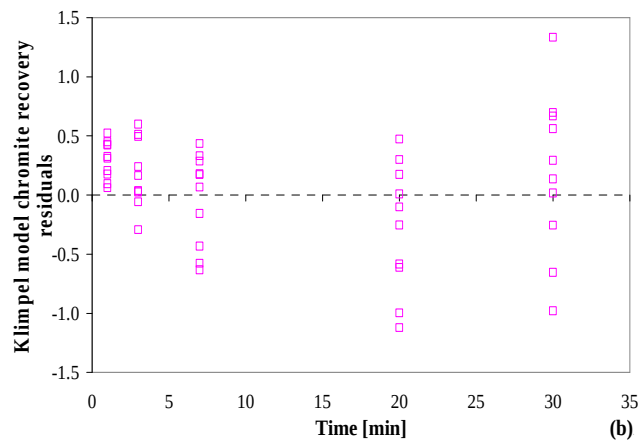
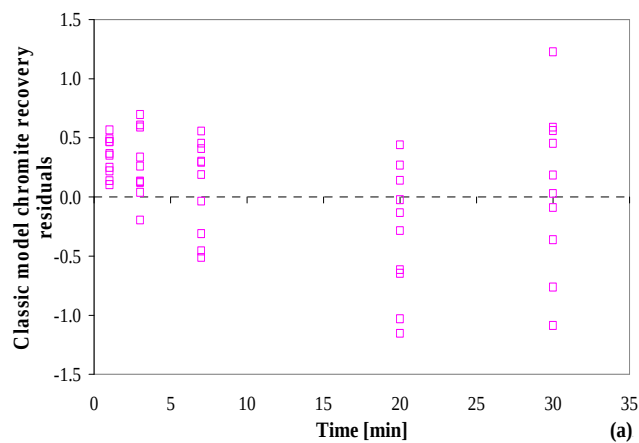


Figure 37: Fit of various flotation models (with and without entrainment) to batch rougher chromite data



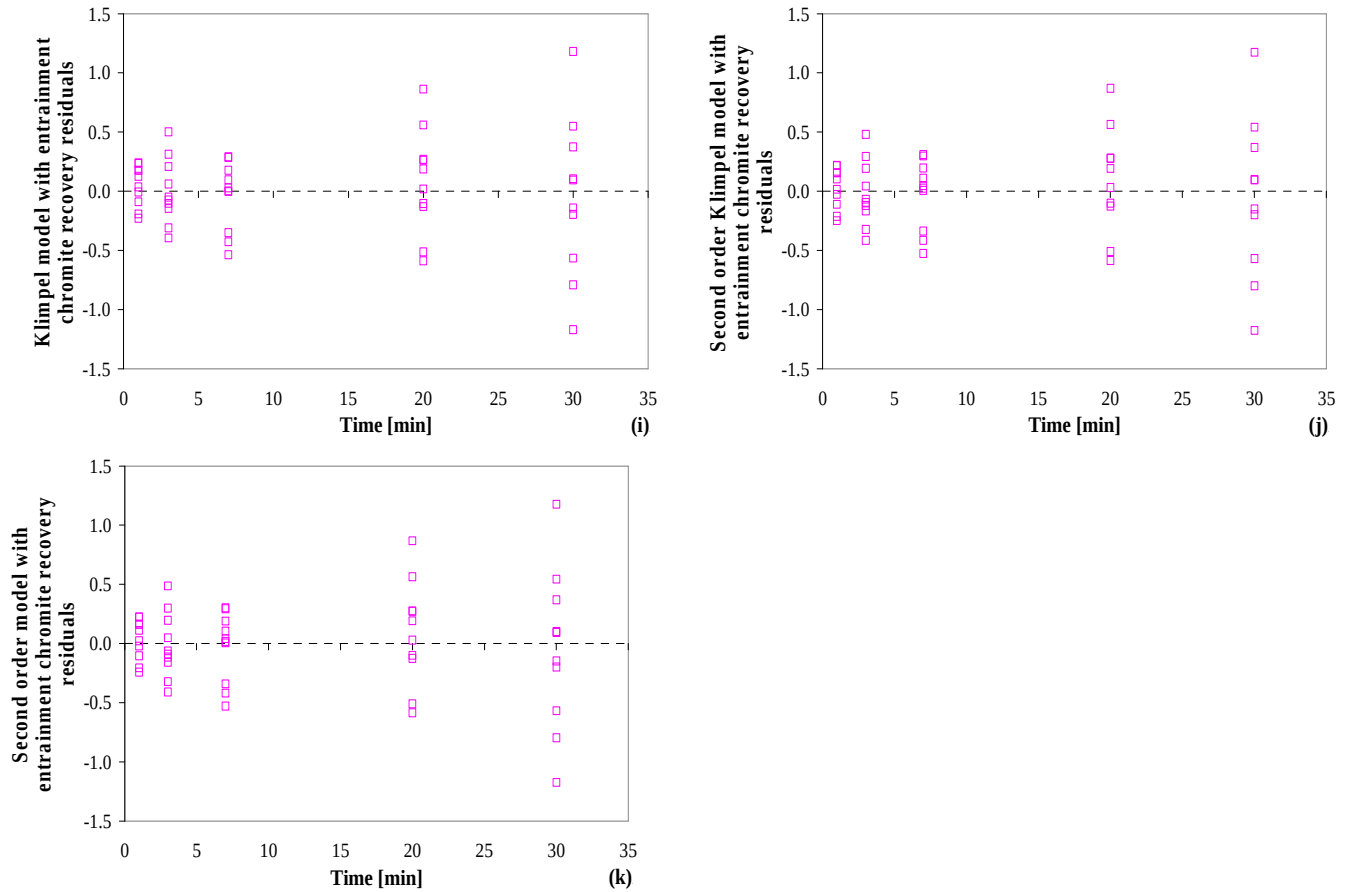


Figure 38: Residual plot of various flotation models (with and without entrainment) to batch rougher chromite data

b. Model statistics

Based on the statistical measures (Figure 39 to Figure 42) and the model fitting discussion given earlier; the classic model with entrainment was selected for modelling the chromite recovery, because:

- The model provides a good fit to the chromite recovery data, and it estimates the recovery at each period in the batch test fairly well.
- It describes the rapid changes in the initial period (mainly recovery by flotation) and the entrainment recovery towards the end of the test.
- Both model selection statistics (MSC and S_p) indicate that this is a suitable model for modelling the chromite recovery.

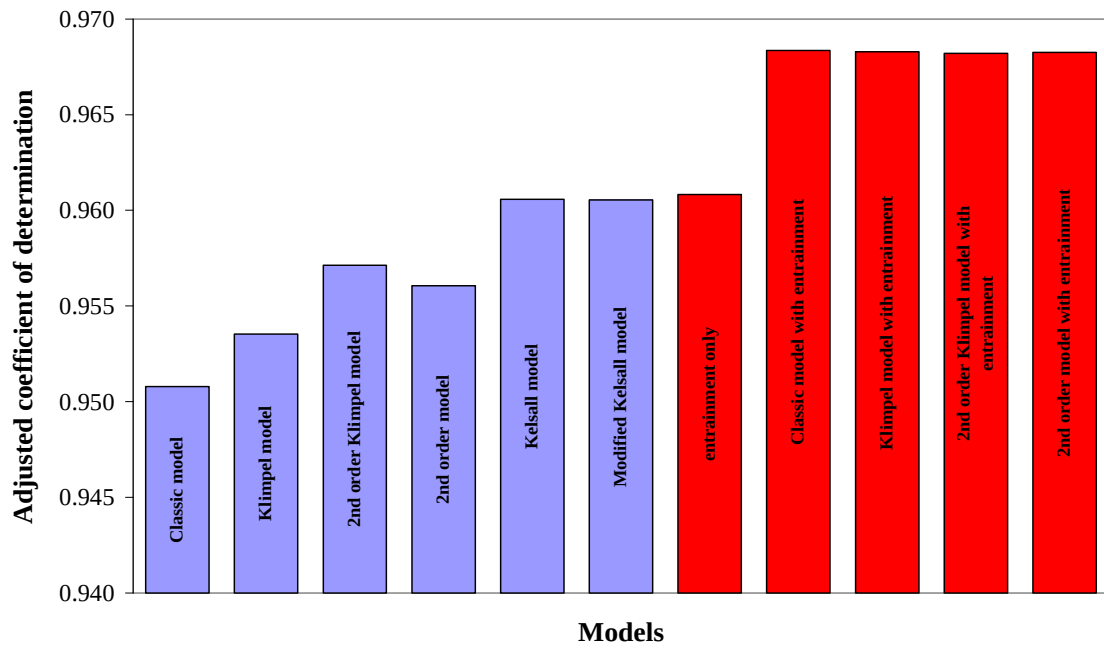


Figure 39: Comparison of adjusted coefficient of determination for models fitted to chromite recovery data

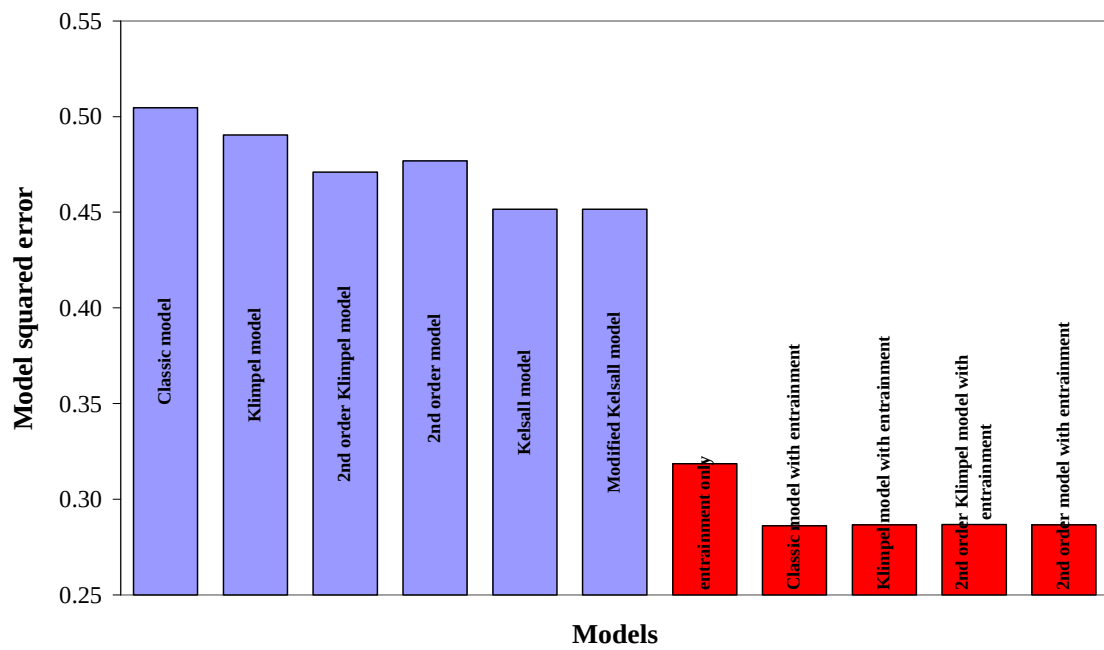


Figure 40: Comparison of model squared error for models fitted to chromite recovery data

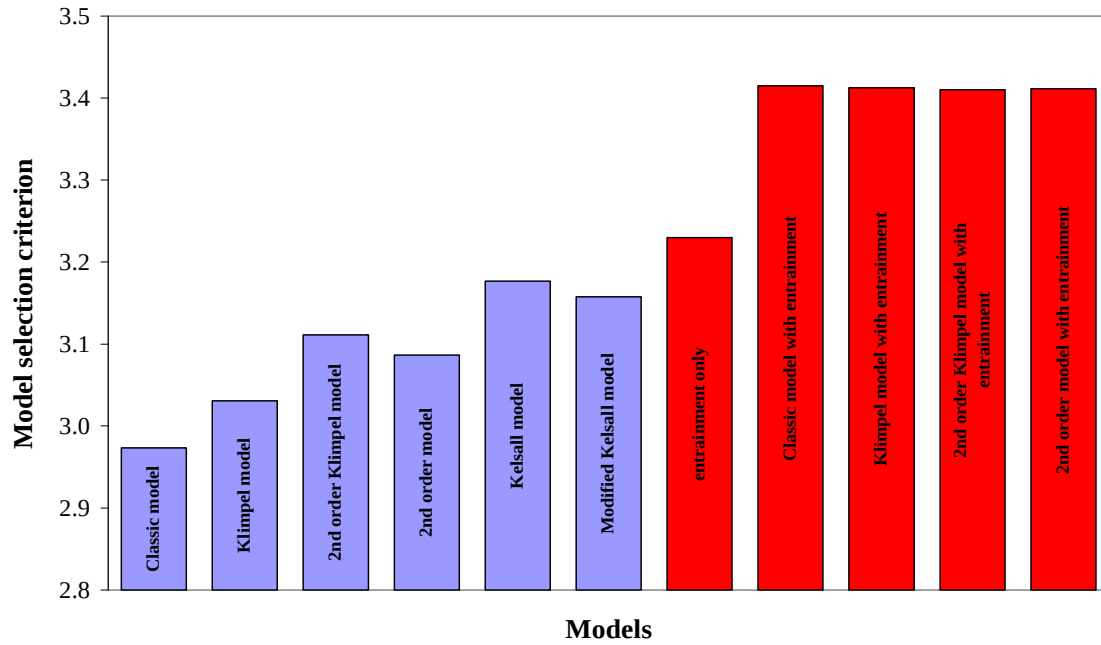


Figure 41: Comparison of model selection criterion for models fitted to chromite recovery data

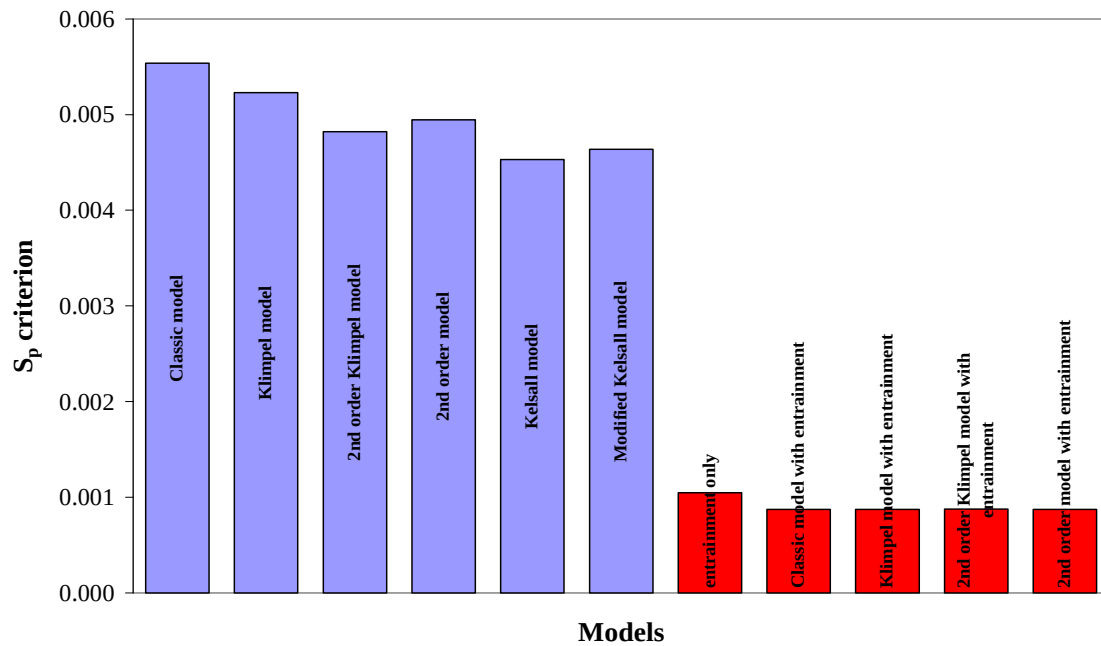


Figure 42: Comparison of S_p criterion for models fitted to chromite recovery data

3.4. Chapter summary

A procedure for determining the PGM recovery variability from UG2 ore was presented. In developing the procedure economics and sample mass were considered. Therefore, the procedure can be applied to more samples from the Bushveld Complex. This made the study more generic. The procedure considered sample selection, feed chemical assays, batch

flotation test, reproducibility of chemical assays and batch flotation method and lastly, models for the rate of mineral recovery. The reproducibility of the chemical assay and batch flotation test was satisfactory for the study. Various batch flotation models were applied to the reproducibility data for PGMs, gangue and chromite to determine suitable models for the rate of recovery. Gangue and chromite recovery was included to make the study complete.

Models which considered recovery by flotation and entrainment were found to be better for modelling the rate data. A new approach was proposed for entrainment modelling in a batch system. The approach was logical, and this was justified by the fitting and residual statistics. From the statistical evaluation of models, the PGM recovery will be modelled using the modified Kelsall model with entrainment. The gangue and chromite recovery will be modelled using a single floatable fraction with entrainment. The models will be applied in the next chapter to illustrate the recovery variability.

CHAPTER 4. RESULTS AND DISCUSSION ON UG2 VARIABILITY

4.1. Introduction

A test procedure was formulated in Chapter 3 for measuring flotation performance. The procedure was applied to fifty UG2 samples collected from across the Bushveld Complex. The variability with respect to PGM feed grade, grind and recovery of PGMs will be shown and discussed here. Since the primary focus of the study is to determine factors affecting PGM recovery; the variability of gangue and chromite recovery will be shown in Appendix B- to reduce the size of the main body of the thesis. Depression of gangue is an important feature of platinum flotation, and the recovery of gangue and chromite will be considered in the simulation example presented in Chapter 7.

Attempts will be made to link the PGM floatable recovery to feed chemical assay. A simple statistical methodology, called predictive modelling, will be used to develop an empirical relationship between PGM recovery and feed chemical assay.

4.2. Sample selection

Figure 43 illustrates the location of the fifty UG2 samples that were received from the various operations situated in the Bushveld Complex. More samples were received from the Western Limb than the Eastern Limb because there are more platinum operations in the Western Limb. Points on the map which have the same colour indicate operations that belong to the same company, e.g. Anglo Platinum, Lonmin, Impala, etc. Some operations submitted more samples, and this is shown as clusters on the map.

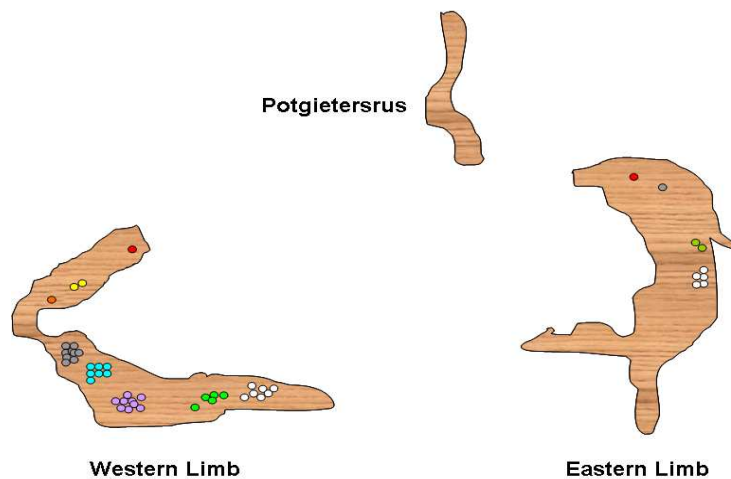


Figure 43: Illustration of the locations from which the samples were collected for the study

4.3. 2PGE+Au feed variability

Figure 44 shows the feed 2PGE+Au assay variability for samples from the Western and Eastern Limb of the Bushveld Complex. The 2PGE+Au grade is shown since this assay was used for flotation test work presented later in this chapter. The Western Limb has a higher PGE grade than Eastern Limb, which corroborates literature (Cawthorn, 1999). The PGE grade depends strongly on mining cut. The Eastern Limb has larger mining cuts than the Western Limb, due to a greater mineralisation in the footwall. Therefore, these samples have a lower PGE grade than the Western Limb samples. Figure 45 shows the influence of silicate content on the PGE grade. Samples with higher silicate content (i.e. a greater dilution of the reef, due to larger mining cuts) have a lower PGE feed grade.

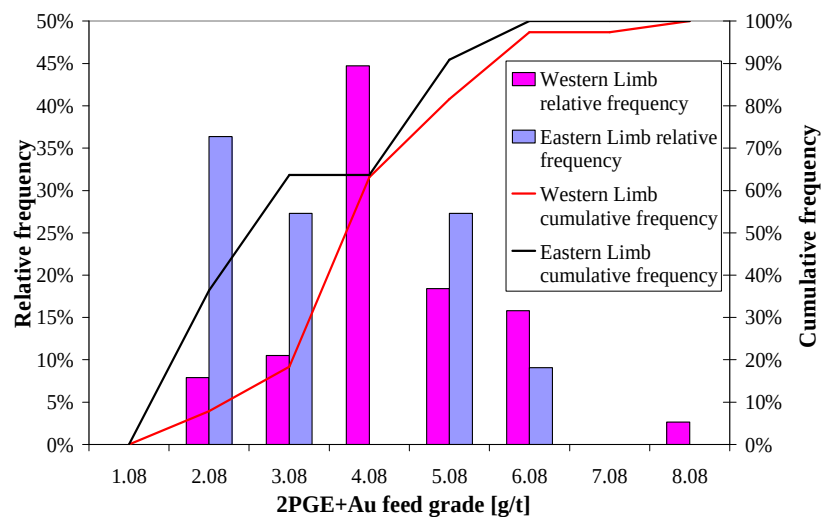


Figure 44: Variation in 2PGE+Au feed grade for samples from the Western and Eastern Limb

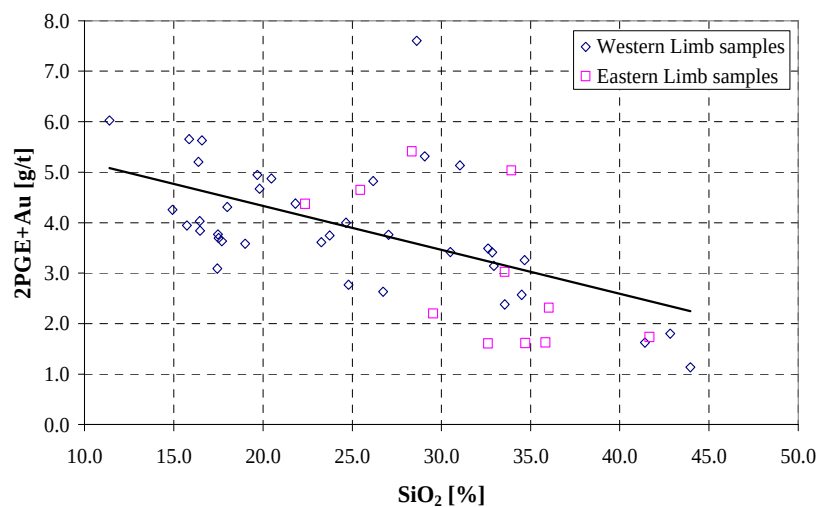


Figure 45: Relationship between 2PGE+Au and SiO₂ feed grade

4.4. Grind variability

A fixed energy was used to mill all samples for flotation test work; therefore, the grind (per cent passing 75 μm) varied. The grind for each sample was estimated by sizing the rougher tail from the flotation test at 75 μm . Figure 46 shows the variation in the grind for the samples. The Eastern Limb samples are softer than the Western Limb samples. According to Fletcher and Bryson (2004) softer ores have more secondary silicates, which are clay-like minerals like talc, chlorites, amphiboles, serpentines, etc. The clay-like minerals are derived from chemical alteration of primary silicates.

Geochemical proxies make use of changes in bulk rock geochemical composition caused by alteration. According to Bahlburg and Dobrzinski (2009), a simple proxy that describes alteration is the Ruxton Ratio R ($\text{SiO}_2/\text{Al}_2\text{O}_3$). The ratio assumes that Al_2O_3 remains immobile during alteration; therefore, changes in R reflect silica loss due to alteration. A lower value for R implies that there are more secondary silicate minerals. Figure 47 shows the relationship between grind and the R ratio, which although moderate, suggests that samples with more secondary silicates (i.e. lower R values) have finer grind.

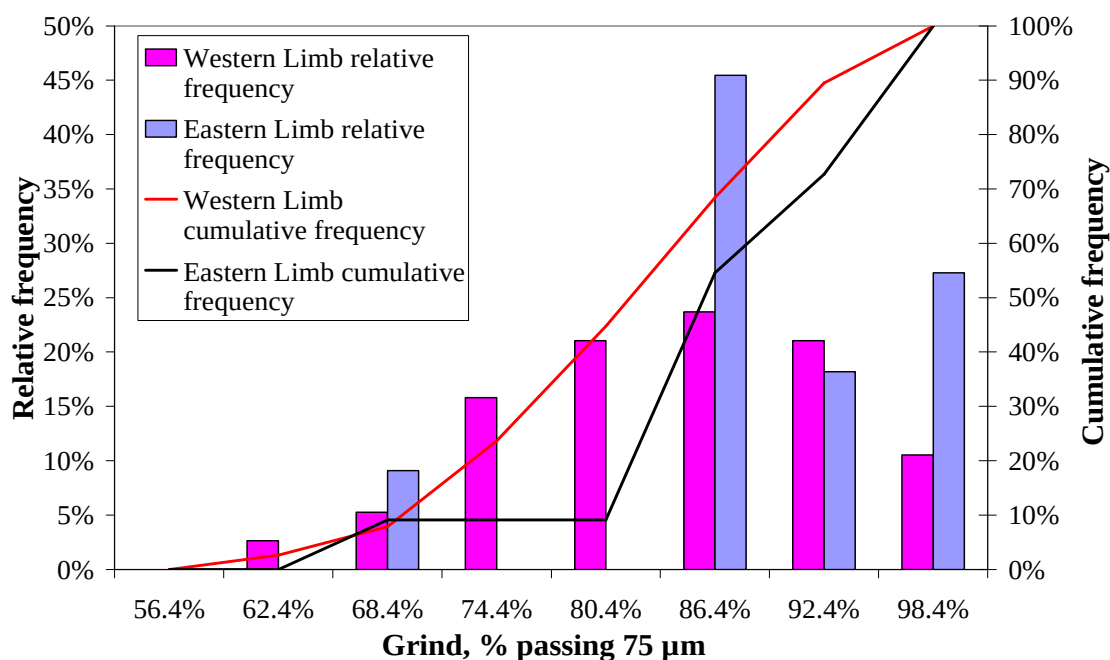


Figure 46: Variation in grind for samples from the Western and Eastern Limb

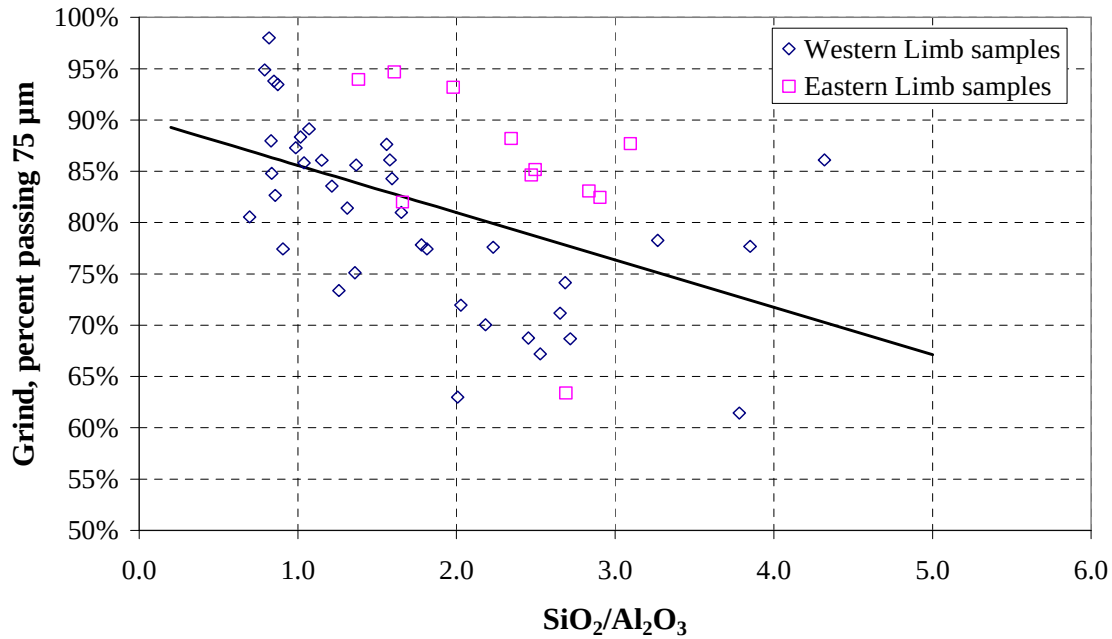


Figure 47: Relationship between grind and Ruxton ratio

4.5. PGM recovery variability

4.5.1. Illustration of PGM recovery variability

The modified Kelsall model with entrainment was found statistically to be the most suitable model for representing the batch PGM recovery data in Chapter 3. The model was fitted to overall recovery to cleaner concentrate for each sample, and the model curves are shown in Figure 48. The quality of the model fit for each sample is shown in the parity chart of Figure 49. The figure compares the recovery obtained from the model with the experimental recovery; the comparative line (in red) provides a measure of the simulated data variation from the experimental data. The model gave a good fit to all experimental data points and for all samples.

The premise of fitting the model to the recovery data was to determine the flotation parameters that illustrate the extent of PGM recovery variability. Furthermore, it enables links between flotation recovery and feed chemical assay to be examined. The parameters obtained were the fast floating fraction (R_{fast}), slow floating fraction (R_{slow}), entrainable fraction (R_{ent}), fast floating rate constant (k_{fast}), slow floating rate constant (k_{slow}) and the average degree of entrainment factor (ENT). These six parameters were fitted to a total of thirteen independent data inputs for each ore type. The independent data points were calculated from five cleaner concentrate samples, two rougher tailings samples, one cleaner

tailing sample, and five measurements of water mass in concentrate. In some cases, it was necessary to combine dried concentrate samples, to provide sufficient mass for PGM analysis. Figure 50 illustrates the statistical significance of the regressions using a histogram of the adjusted r^2 values.

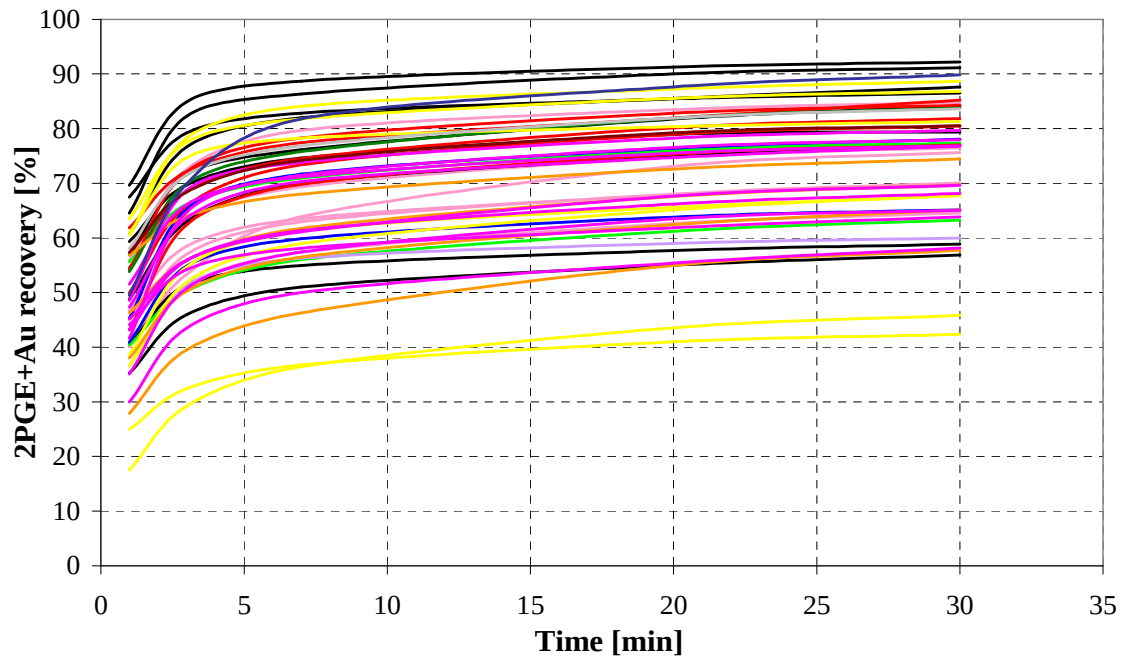


Figure 48: Overall recovery of PGMs to cleaner concentrate

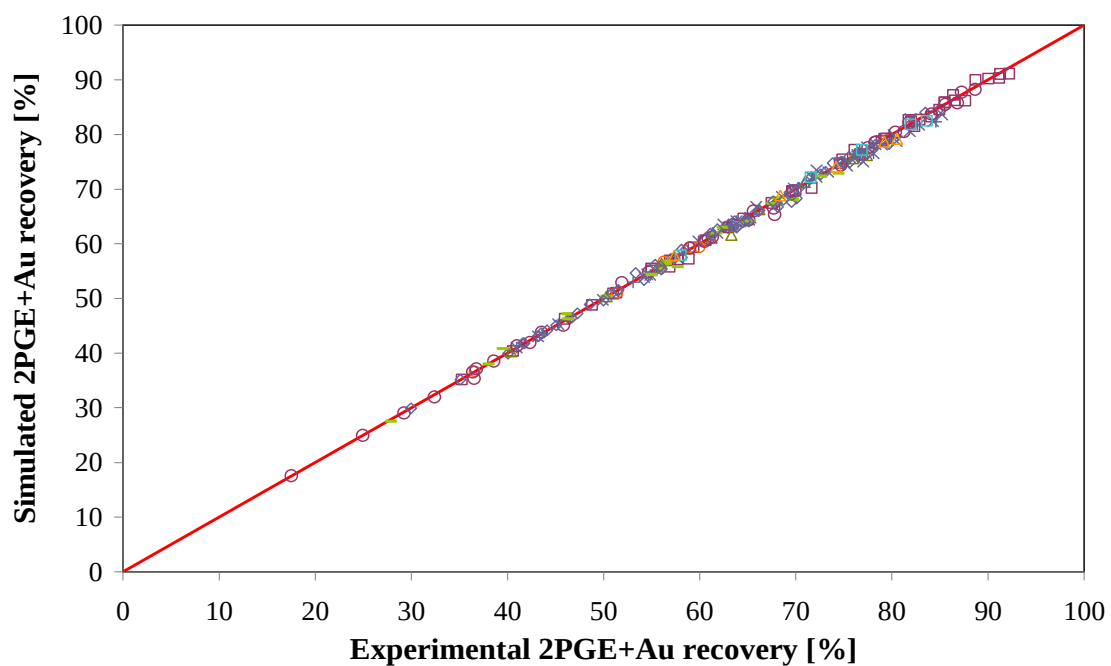


Figure 49: Model fit to cleaner PGM rate of recovery for each sample

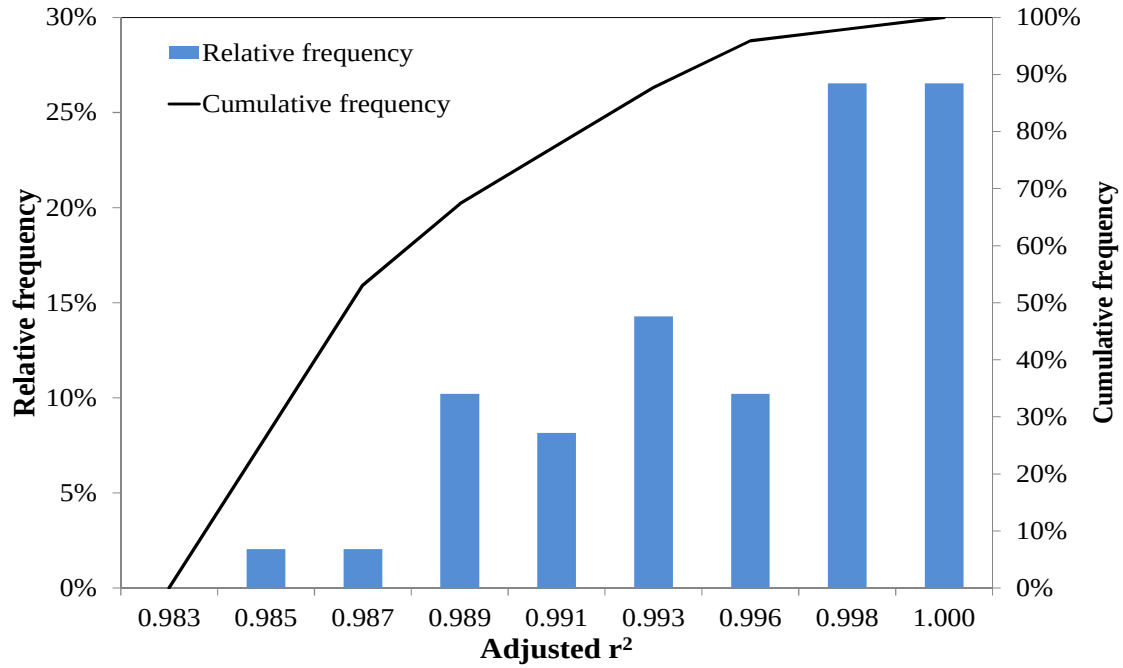


Figure 50: Histogram of adjusted r^2 for model fits to PGM recovery data

Figure 51 shows the extent of the PGM variability with respect to the model parameters for samples from the Eastern and Western Limb. The Eastern Limb samples have larger fractions of the PGMs that are fast floating and entrainable than the Western Limb. However, both Limbs have a similar distribution of slow floating PGMs. The distribution of the rate constants and the degree of entrainment for samples from each Limb is also similar. The factors influencing the variation observed will be discussed in the next section.

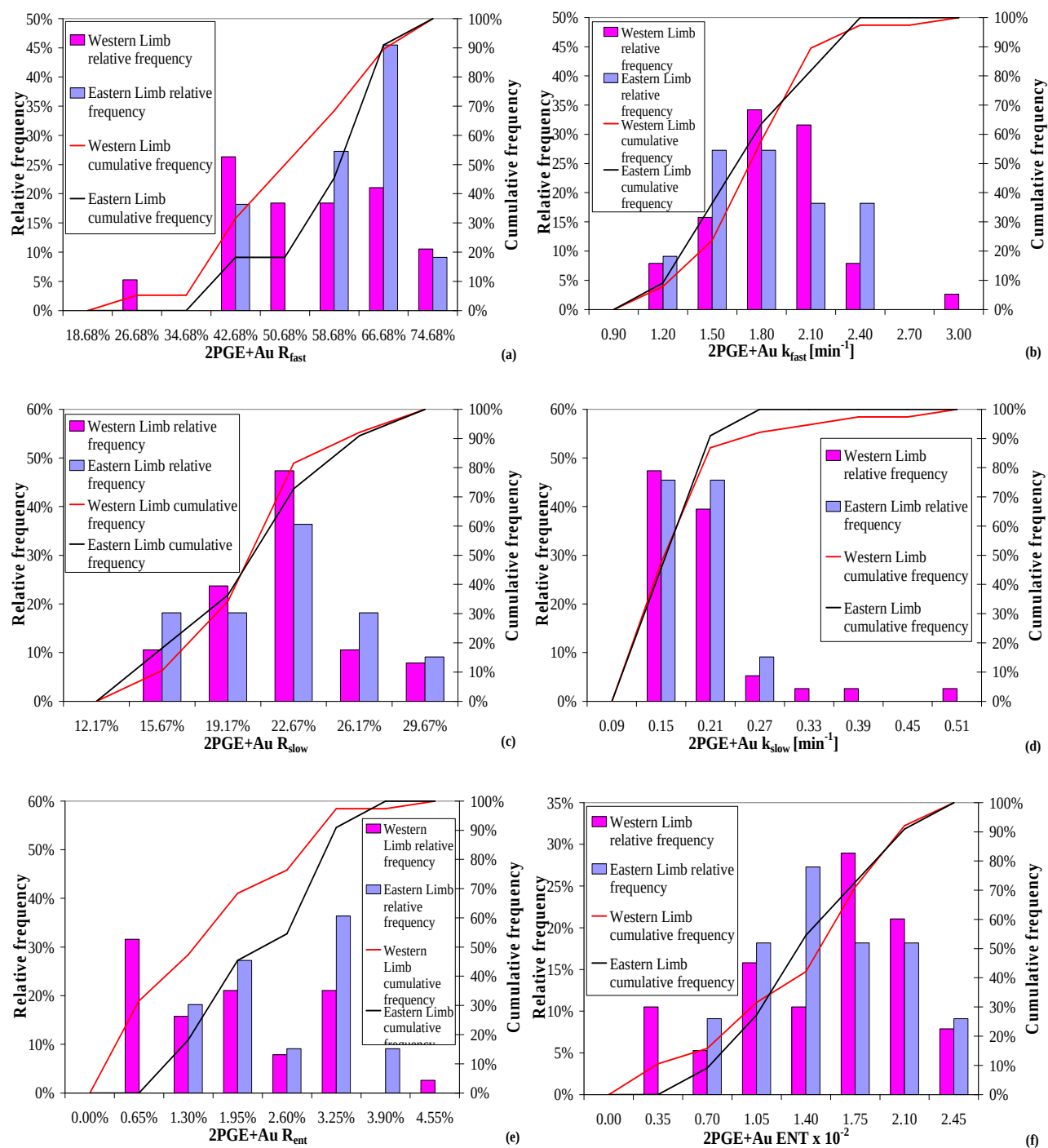


Figure 51: Variability of batch flotation parameters for PGM recovery

4.5.2. Linking PGM recovery variability to feed chemical assay

4.5.2.1. General

There is a significant variation in the flotation response of the PGMs. Samples have different floatable and entrainable fractions, rate constants and degree of entrainment. The variation may be attributed to various factors such as grind and feed chemical assay. It was initially thought that samples with finer grind would have a higher recovery due to better liberation of

PGMs, and that higher PGE feed grades give better recovery of PGMs. However, the study showed some surprising results of these two variables.

The recovery profile for two samples (B8 and L1), with similar grinds, are shown in Figure 52. Sample L1 had a lower recovery despite having a much higher PGM content. This can be attributed to two factors; insufficient liberation of PGMs and minerals may have been altered. Both factors are shown and discussed in Chapter 6. With regards to liberation, a UG2 ore may have a fine grind (e.g. 80% passing 75 μm), but the actual platinum group mineral can have poor liberation, i.e. the minerals can be locked in gangue such as chromite and silicates. According to Penberthy (2001), the poor liberation of PGMs at fine grinding is due to the minerals having undergone alteration, i.e. silicate overprint of the minerals. Hence, ultra-fine grinding of the ore is required for liberation of PGMs. With regards to feed grade, higher PGE grade does not necessary imply better mineral recovery by flotation. Voordouw *et al.* (2010) examined the platinum group minerals in UG2 ore and found that more than a 100 minerals occur. This is significant by comparison to any ore body. Furthermore, he found that some of the sulphide mineral had been altered due to injection of hydrothermal fluid. The result was an alteration of the sulphide minerals to arsenides, tellurides and alloys- also known as secondary platinum group minerals. Shackleton *et al.* (2007) found these minerals to have poor floatability and are lost to the flotation tail.

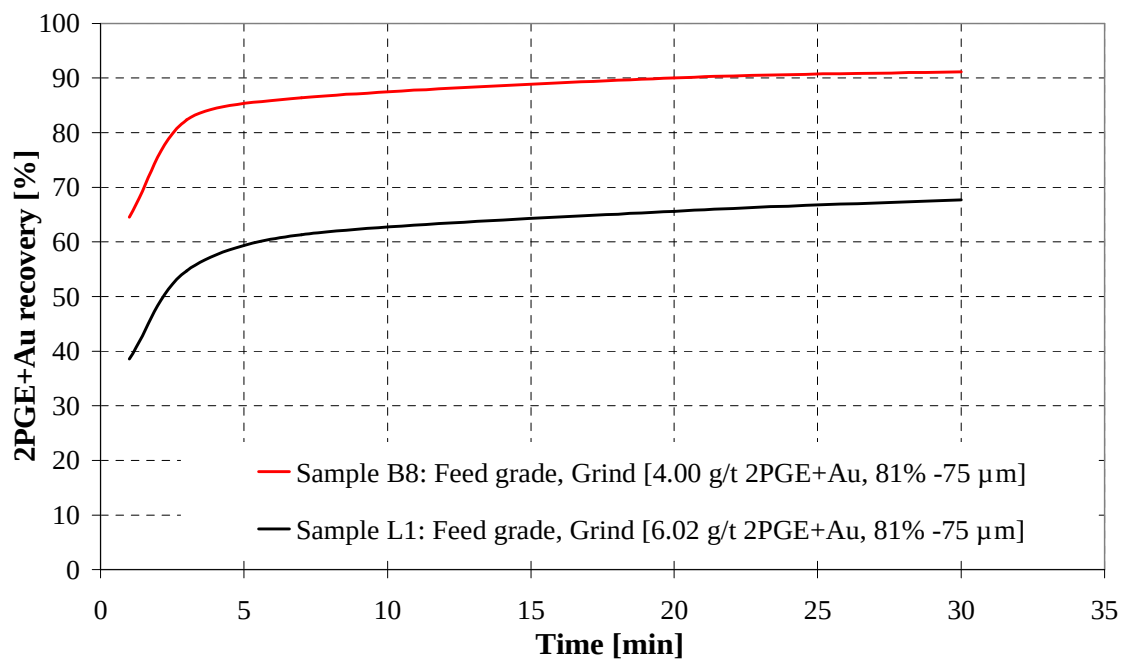


Figure 52: Significance of feed PGE grade on PGM recovery at similar feed grinds

Figure 53 illustrates that samples which have similar PGE feed grade do not necessarily have better performance at fine grinds. This justifies the explanation given earlier regarding the key mineralogical characteristics affecting recovery, which is poor liberation of PGMs (even at a fine grind) and alteration of PGMs to secondary minerals with poor floatability.

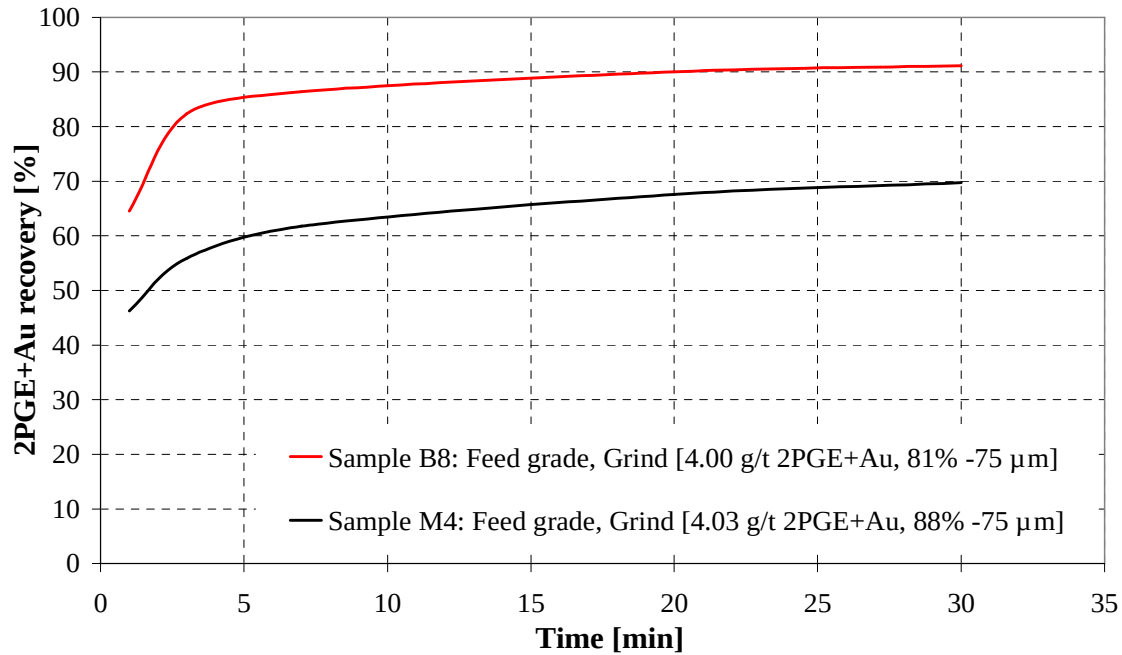


Figure 53: Significance of feed grind on PGM recovery at similar feed PGE grade

The premise of understanding PGM recovery variability is the ability to relate the variability to a discernible characteristic of the sample. Whole rock assays indicate changes in the geology therefore; changes in recovery may be explained with respect to geological variation. However, there is no structured method for determining such a relationship. For a metallurgist the problem is determining what whole rock assays and ratios should be used. On the other hand, for a geologist the problem is determining the necessary metallurgical parameters.

Furthermore, the functional form of the relationship between whole rock assay and recovery by flotation may be linear, non-linear or a combination of several variables. A statistical method will be presented in the next section for determining the link between PGM recovery and geochemical assays. This is a new development and can be applied for other types of mineralisations.

4.5.2.2. Predictive modelling- linking PGM recovery to feed properties

a. General theory

A brief discussion on predictive modelling will be presented followed by application of the method. NIST (2012) gives a complete discussion on predicting modelling; an illustration of the procedure is given by Figure 54. The first step is the selection of an appropriate response variable and predictor variable(s) for modelling. In this study, the response variable will be the floatable mass fractions, i.e. fast and slow floating. The predictor variables will be assays and ratios of assays. Only the floatable fraction (the sum of the fast and slow floating fractions) was chosen as the response variable. This fraction is recovered by attachment to bubbles, unlike the unfloatable, but entrainable fraction which does not have any mineral properties that enable attachment to a bubble. The entrainment fraction is relatively small, and it tends to get lost in multiple-stage cleaning. The link to various measurable features of the ore will be investigated using statistics.

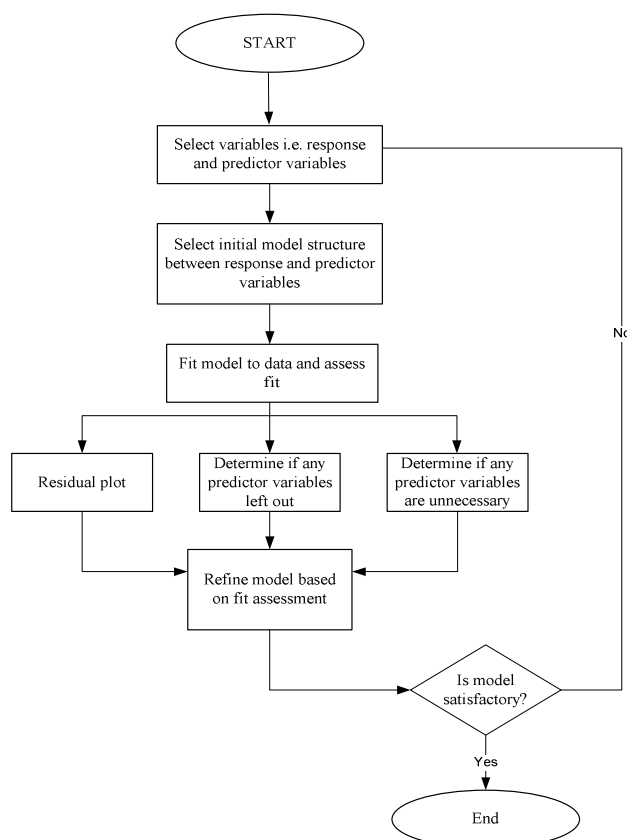


Figure 54: Algorithm used for linking PGM recovery to feed chemical assay

The second step involves determining an appropriate model structure between the recovery and chemical assays. According to NIST (2012), the model structure can be determined from

theory. However, the literature reviewed in this study on UG2 beneficiation did not show any relationships between PGM recovery and chemical assays. Therefore, a simple multi-linear model structure was assumed between recovery and chemical assay. This assumption may be incorrect, since some chemical assays may have a non-linear relationship with the recovery. Nevertheless, the model fit and assessment step enables an evaluation of the functional form of the chemical assays in the proposed model. The adjusted coefficient of determination was used as a fitting statistic for the multi-linear regression analysis.

Residual plots serve two purposes in predictive modelling; it enables an assessment of the functional form of the predictor variables, and it enables an assessment of predictor variables, which may have been left from the initial model structure. Residual plots for each predictor variable can indicate if the variable is incorrect in the model. Generally, a random and scattered distribution of the residuals across the zero residual line indicates a correctly modelled variable. On the other hand, residual plots which have a systematic pattern, like large residuals for small values of the variable and small residuals for large values of the variable, indicate an incorrect functional form of the variable in the model. NIST (2012) indicates that transformation of the variable using logs, inverse or square roots can rectify most incorrect functional forms. The residual plot can also be used to determine if predictor variables were left out of the model structure. If a plot of the residuals with potential predictor variables (i.e. other chemical assays) show a systematic pattern, then the variable should be included in the model proposed. In other words, the systematic pattern indicates that the variable should be included but its functional form needs to be corrected in the model structure.

Lastly, a T-statistic can be used to determine if the model has unnecessary variables in the structure. This is important since a model with unnecessary variables can result in smoothing of the ‘noise’ in the experimental data. In other words, unnecessary variables compensate for noise in a causal manner rather than actually fitting the data meaningfully. This is typically seen for polynomials of higher order having five or more inflection points. The absolute T-statistic is calculated using:

$$T - \text{statistic} = \left| \frac{\text{Coefficient of variable}}{\text{Standard error of variable}} \right| \quad (4.1)$$

The coefficient and standard error of the variable are obtained from the model fit to the experimental data. Since the t-distribution is symmetrical the absolute value of the T-statistic from Equation 4.1 can be used. The T-statistic is compared to the value from the t-distribution having a model degree of freedom (df) and a confidence level of 95%. If the T-statistic is less than the value from the t-distribution, the variable is unnecessary and can be removed from the model structure (i.e. the null hypothesis). However, if the T-statistic is greater than the value from the t-distribution, then the null hypothesis cannot be rejected at a 95% confidence level.

b. Application of predictive modelling

In UG2 flotation, several chemical assays and ratios may be considered for describing the recovery of PGMs. Based on literature and bivariate plots of PGM recovery by flotation with chemical assays, the following variables were selected for the initial model:

- 2PGE+Au– Refers to the PGE feed grade of the sample. The flotation products were assayed for Pt, Pd and associated Au hence, the feed 2PGE+Au content needs to be considered only.
- Grind– This variable influences the liberation of the PGMs and consequently the recovery.
- Pt/Pd ratio – Oberthür and Melcher (2005) stated that Pd is more mobile than Pt and is dispersed in the supergene environment. In other words, deposits which have undergone some form of geological alteration or weathering, have less Pd in the reef since it is removed during alteration. A higher Pt/Pd ratio indicates a supergene environment. Farrow *et al.* (2005) stated that “The genetic implications of the higher Pt/Pd trend with decreasing sulphur content is due to Pt solubility being lower than that of Pd in most aqueous halogen-bearing fluids, it is possible that Pt-bearing sulphides precipitated before the higher sulphide, i.e. lower Pt mineralisation.” Cameron and Hattori (2005) stated that in most cases, there is an increase in the Pt/Pd ratio in altered material.
- Ni/[Ni+Fe]– Guo *et al.* (1999) used the Ni/[Ni+Fe] ratio to describe the base metal content in sulphide ores. This ratio is considered to be important since the PGMs are generally associated with BMS therefore, a high BMS content will inherently imply better PGM recovery. Furthermore, higher Fe contents relative to Ni (i.e. smaller Ni/[Ni+Fe] ratio) implies that there is more pyrrhotite present in the sample.

Pyrrhotite has a lower floatability than pentlandite and occurs mainly in altered environments, i.e. Fe-rich hydrothermal fluid altered environments. In this study, the acid soluble Ni and Fe assay will be used because this assay is directly related to sulphide Ni and Fe minerals, unlike Ni and Fe, which includes oxide minerals.

- Rb/Sr– Xu *et al.* (2010) used this ratio for determining the extent of chemical weathering in deposits. During fractional crystallization Sr tends to become concentrated in the solid phase while Rb remains in the liquid phase. The Rb/Sr ratio in residual magma may increase over time resulting in rocks with increasing Rb/Sr ratios with increasing differentiation. With respect to chemical weathering, Sr will be higher in the solid phase for weathered samples than for samples with minimal weathering, i.e. a lower ratio.
- LOI– Loss on ignition refers to the decrease in the mass of the sample when heated in a controlled environment. The more volatile species, which occur due to alteration, are removed when heated. Therefore, samples with a high LOI value have more volatiles, which indicate a higher degree of alteration.

Figure 55 shows bivariate plots of the variables discussed above with the floatable PGM recovery, which is the sum of the fast and slow floating fraction. In some plots, trend lines are included to highlight relationships. The plots indicate:

- The feed grade, grind and Pt/Pd ratio do not have a significant influence on the PGM recovery.
- The base metal ratio ($A/sNi/[A/sNi+A/sFe]$), Rb/Sr ratio and LOI have weak to moderate relationships with the recovery.
- Samples with a high base metal ratio (Figure 55d) have better recovery of PGMs than samples with a lower ratio.
- Samples which have a small Rb/Sr ratio have a lower recovery, which implies that these samples are more weathered hence, the recovery is lower.
- Samples with a high LOI value have a lower recovery, which implies that these samples are more altered hence, the recovery is lower.

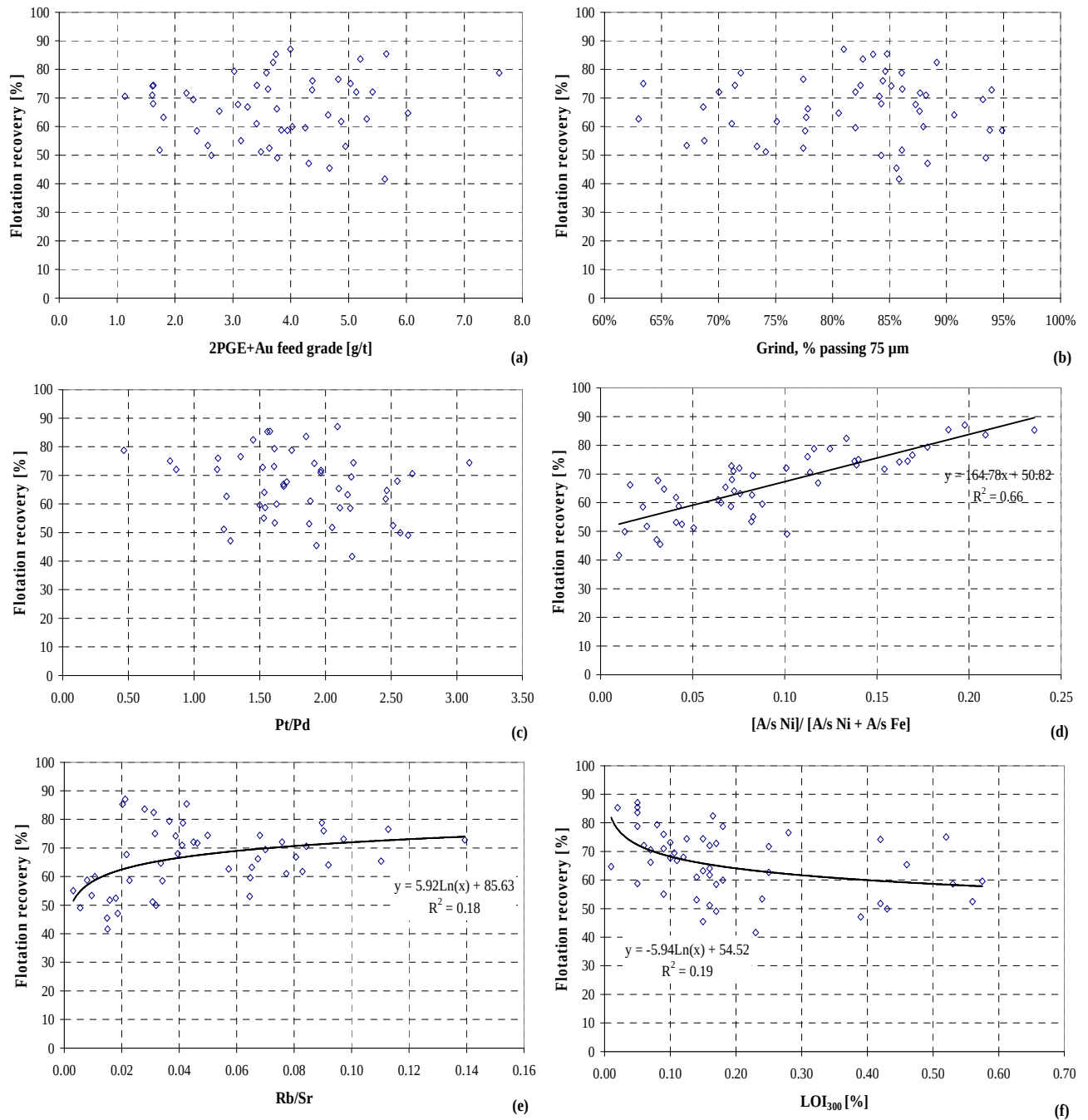


Figure 55: Plots of potential predictor variables which may influence PGM recovery

Multi-linear regression was used to link the recovery and assays described above. Figure 56 presents the model fit to the experimental data, Equation 4.2 gives the multi-linear model and, Figure 57 shows the residual plot for each predictor variable. Using these three sets of information the following summary can be made for the initial model:

1. The model has an adjusted coefficient of determination of 0.75, which is a moderate fit to experimental data. The model fit plot shows difficulty in modelling the lower

PGM recovery data. This may be due to mineralogical factors of the samples, which are not sufficiently represented by the assays used in modelling, such as mineral association and degree of liberation.

2. The residual plots indicate that the 2PGE+Au, fractional grind, Pt/Pd and [A/sNi]/[A/sNi + A/sFe] ratio have random and scattered residuals hence; these variables have a correct functional form in the model.
3. On the other hand, Rb/Sr and LOI have scattered residuals but larger residuals occur at lower values of the variables, and smaller residuals occur at the larger values of the variables. This indicates that the functional form of the variables is incorrect. The variables may need to be transformed using $\ln(\text{variable})$, $1/\text{variable}$ or $\text{variable}^{-0.5}$ transformations, to correct the functional form of the variable such that the residuals are not structured.
4. In Equation 4.2 the 2PGE+Au grade is in g/t; the grind is fractional and LOI is in per cent. The coefficient for each variable was used to determine the absolute T-statistic (according to Equation 4.1); this statistic was compared to the value from a t-distribution having 43 degrees of freedom at a 95% confidence level- this was 2.02. The 43 degrees of freedom is the difference between the data points (50) and the model parameters (7). The analysis (refer to Table 14) indicates that the 2PGE+Au, fractional grind and Pt/Pd ratio are unnecessary variables in the model. Since their absolute T-statistic was less than the value from the t-distribution. This was expected since the bivariate plots (Figure 55) showed that these variables had no significant influence on the PGM recovery.

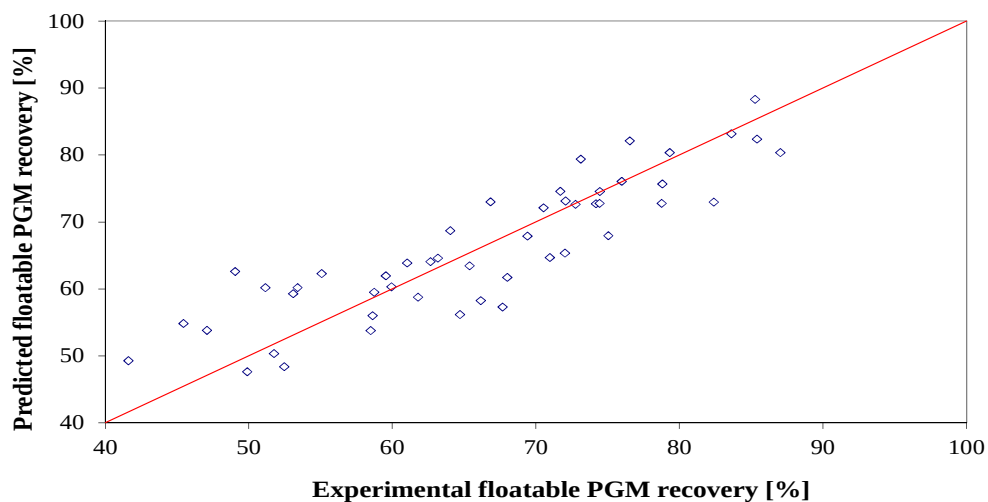


Figure 56: Parity chart for initial model fit

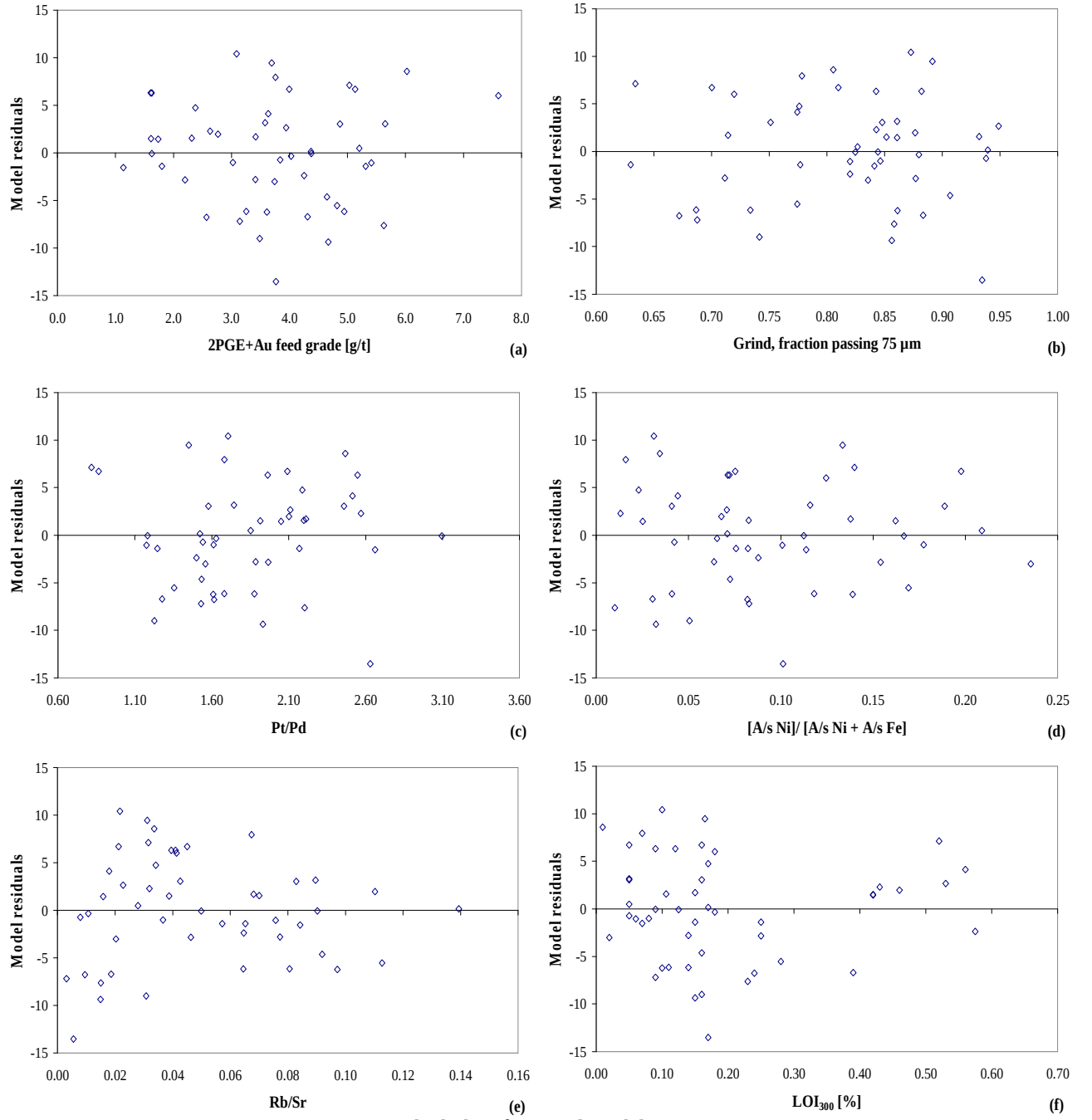


Figure 57: Residual plots for initial model structure

$$R_{\text{float}} = -0.27[2\text{PGE} + \text{Au}] + 9.47[\text{Grind}] - 3.25\left[\frac{\text{Pt}}{\text{Pd}}\right] + 146.90\left[\frac{\text{A/sNi}}{\text{A/sNi} + \text{A/sFe}}\right] + 82.02\left[\frac{\text{Rb}}{\text{Sr}}\right] - 14.79[\text{LOI}] + 50.50 \quad (4.2)$$

Table 14: Summary of model parameters for determination of T-statistic

Variable	Coefficients	Standard Error	Absolute T-statistic
Intercept	50.50	10.58	4.77
2PGE+Au	-0.27	0.75	0.36
Fractional grind	9.46	10.91	0.87
Pt/Pd	-3.25	2.02	1.61
AsNi/[A/sNi+A/sFe]	146.93	15.70	9.36
Rb/Sr	82.02	26.47	3.10
LOI	-14.79	5.78	2.56

The predictive modelling approach was repeated, but the unnecessary variables were removed, and the incorrect functional form of the variables was corrected using transformations of the variable. The transformations which gave the best model fit and scatter in the residuals was used. Equation 4.3 presents the improved model and Figure 58 shows the improved model fit.

$$R_{\text{float}} = 140.68 \left[\frac{\text{A/sNi}}{\text{A/sNi} + \text{A/sFe}} \right] - 1.32 \left[\frac{\text{Rb}}{\text{Sr}} \right]^{-0.5} - 3.33 \ln \left[\text{LOI}_{300^{\circ}\text{C}} \right] + 54.08 \quad (4.3)$$

The model has an adjusted coefficient of 0.80, which according to NIST (2012) is not large enough to conclude that the model may be used for prediction. Generally, an adjusted coefficient >0.92 indicates that the model can be used for prediction. Nevertheless, it is believed that the model provides a reasonable qualitative correlation between PGM floatable recovery and chemical assay. The model indicates that samples with a high base metal sulphide ratio have high PGM recovery. This implies greater association of the PGMs with pentlandite. On the other hand, samples with high Rb/Sr ratio and LOI value, which are proxies for weathering and alteration respectively, have a poor PGM recovery.

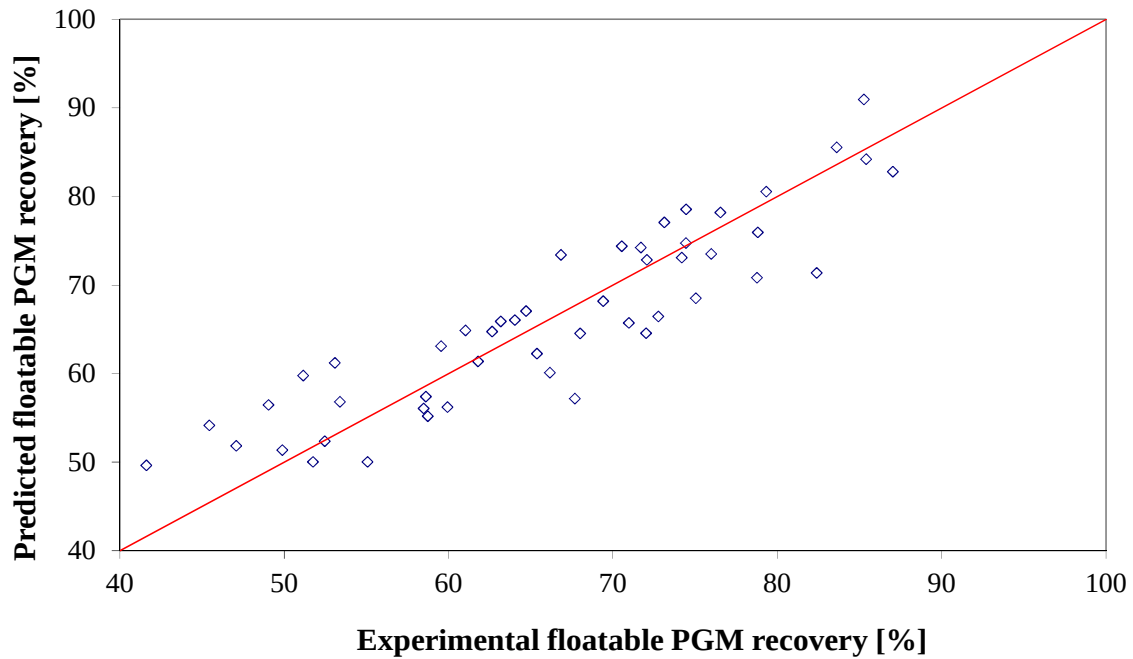


Figure 58: Parity chart for improved model fit

4.6. Chapter summary

The variability of the PGM recovery from cleaner batch flotation tests was illustrated for fifty UG2 samples. A suitable model was fitted to the cleaner recovery data, and the parameters obtained were used to illustrate the extent of the PGM variability, with respect to floatable and entrainable fractions. Attempts were made to determine the feed chemical assays which influence the PGM floatable recovery, i.e. the fast and slow floating PGM fractions. However, no significant relationship could be found. Predictive modelling was used to determine an empirical link between PGM recovery and feed chemical assay. The empirical model obtained can be used to determine qualitatively if a UG2 sample has recovery issues. It is believed that the poor recovery predictability of the model is due to bulk assays not providing sufficient information on the mineralogy of the PGMs, i.e. liberation, associations and PGM speciation. Therefore, in the next chapters the influence of mineralogy on PGM recovery will be investigated, with the primary purpose of determining a more rigorous model for determining the factors influencing PGM recovery.

CHAPTER 5. MODELLING OF BATCH FLOTATION TESTS WITH GRINDING AND DEPRESSANT EFFECTS

5.1. Introduction

The variability in the flotation properties of samples of UG2 ore was illustrated in the previous chapter. An empirical link between feed chemical assay and PGM recovery was developed, but there was significant scatter. The reason is that feed chemical assay does not account for the spectrum of platinum group mineral associations with gangue and base metal sulphides.

Mineralogy can be used to identify the mode of occurrence of PGMs and its contribution to recovery by flotation. The objective of this chapter is to develop and model a batch flotation test method, which quantifies the effects of stage grinding and depressant addition, i.e. an advanced batch flotation test. Both grinding and depressant addition are important in UG2 flotation. The link between model parameters and mineralogy will then be investigated. The experimental method, which includes sample selection and batch flotation tests, will be discussed. The methods for model selection used in Chapter 3 will be extended to include regrinding and gangue depression.

5.2. Experimental

5.2.1. Sample selection

Seven samples were selected for further test work; Figure 59 shows the location of the samples in the Bushveld Complex. The selection of the samples was based on the amount of drill core material available for additional tests. Nevertheless, they do represent a reasonable spread of mineralogy and feed grade.

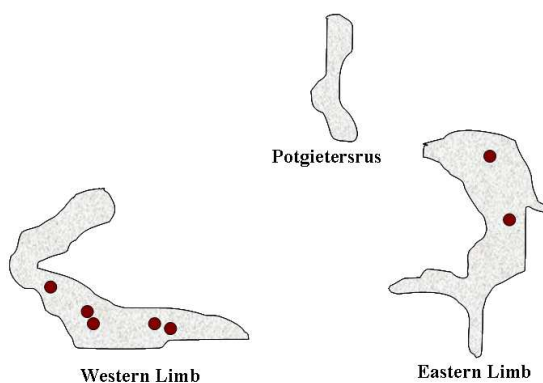


Figure 59: Location of samples selected for further test work

5.2.2. Batch flotation tests

The importance of characterising a sample using batch flotation tests has been recognised by many authors, (Kaya and Laplante, 1986; Barberly *et al.*, 1986; Gochin and Smith, 1987; Villas Bôas, 1991; Agar *et al.*, 1998; Brown and Hall, 1999; Dunne *et al.*, 2002; Yianatos *et al.*, 2003; Hay and Rule, 2003; Williams and Richardson, 2004; Dobby and Savassi, 2005; Alves and Gonçalves, 2006; Mpapwa *et al.*, 2008; Amelunxen and Amelunxen, 2009b & c; Torres *et al.*, 2009; Varadi *et al.*, 2010 and Suazo *et al.*, 2010). However, there are limitations to the batch flotation test, including the use of empirical scale-up parameters, periodic removal of the froth by scraping and the relatively low loading of the bubbles in a typical laboratory flotation machine. Nevertheless, the batch flotation test is still considered the principal method for characterising a sample because simple standard procedures can be used, and the effects of reagents can be tested on small sample masses.

A simple rougher test is normally carried out to determine the recovery attributes of the sample. However, rougher test work does not provide a full understanding of the process, particularly when the gangue must be depressed in the cleaning stages. This is demonstrated for a simple rougher and cleaner test in Figure 60. A sample was milled to a fine grind of 80% passing 75 μm and material was collected at different times. Another test was carried out, but the rougher concentrate was cleaned with a small amount of depressant (50 g/t KU5 based on feed to rougher). The overall recovery for PGMs, gangue and chromite is shown. All species in the cleaner stage were recovered at a slower rate than that observed in the rougher stage. The addition of depressant impeded the flotation of floatable gangue and composite PGM particles, resulting in a lower recovery. Therefore, roughing together with cleaning tests (at different depressant additions) is necessary for obtaining better estimates of mineral floatable mass fractions, rates and entrainment parameters. Clearly, rates of flotation should be measured in both roughing and cleaning.

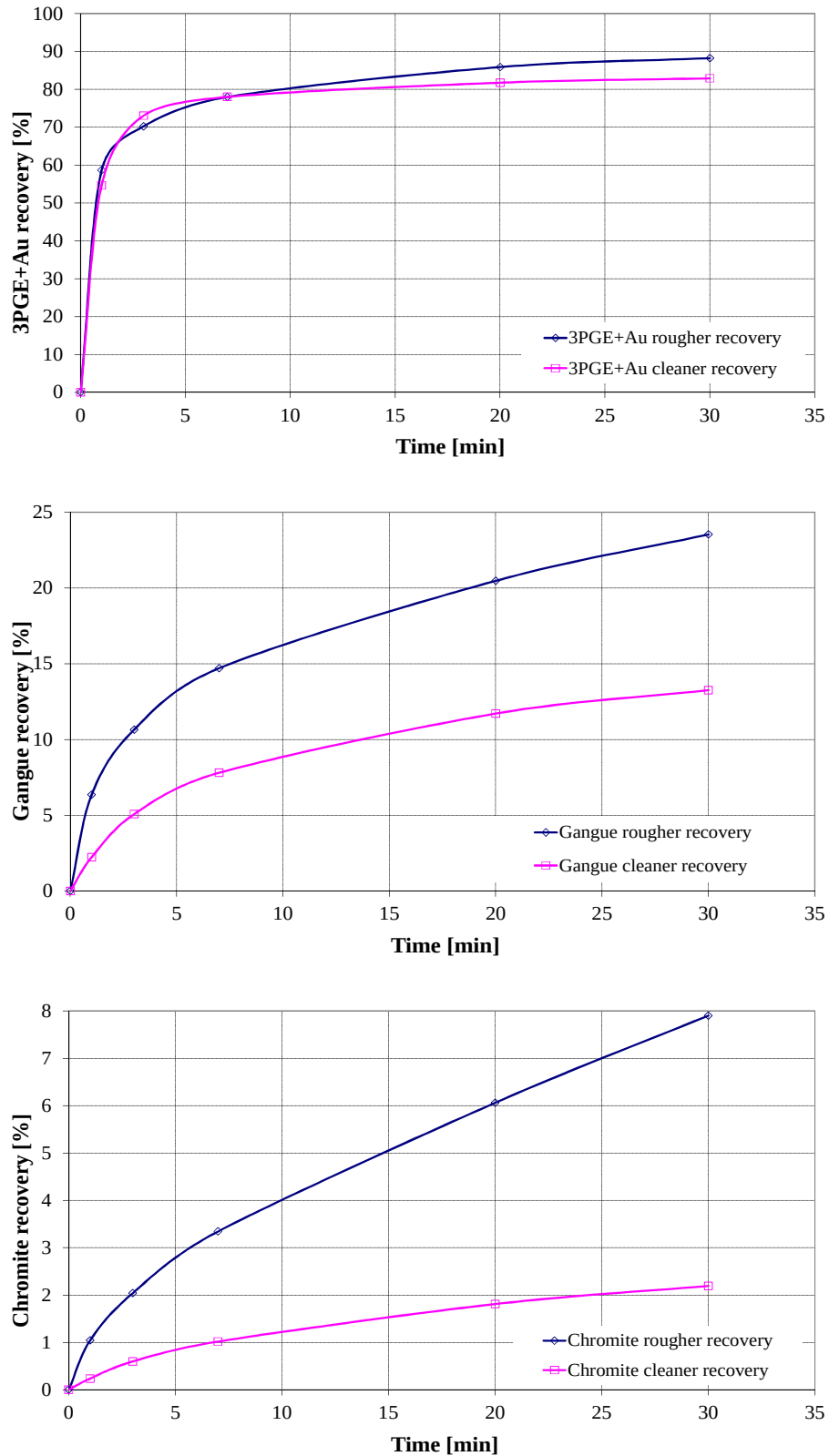


Figure 60: Comparison of the overall recovery of PGMs, gangue and chromite from rougher and cleaner tests

Figure 61 shows the batch flotation experiments used to characterise the samples. A primary rougher, secondary rougher and four cleaner tests (at different depressant additions) were

carried out for all samples. In carrying out the cleaner tests, sufficient mass from each cleaner concentrate is required to create a prill for assaying. Therefore, roughing was carried out twice with the masses combined and used for subsequent cleaning tests. This gave an additional rougher tail for each cleaner test, which was used in determining the mineral recovery.

Primary and secondary roughing was carried out to simulate the process used in most UG2 plants. A coarse grind was used to liberate large grain PGMs attached to BMS thereafter, a finer grind was used to liberate smaller grain PGMs locked in gangue. The application of stage grinding prevents the unnecessary detachment of PGMs from BMS, which results in a smaller particle that has a lower probability of being recovered. Stage grinding also limits the overgrinding of chromite, and prevents the generation of fine chromite particles, which have a higher probability of being recovered by entrainment. Depressant addition to the cleaner stage was also examined, since it was shown earlier that depressant impedes the flotation of minerals and affects the recovery.

The experimental procedure for the primary rougher stage was as follows:

1. A 1 kg sample was milled to a target grind of 40% passing 75 μm , which is a typical primary grind used in UG2 flotation, (Hay and Schroeder, 2005 and Dunne, 2007).
2. Thereafter, the milled sample was transferred into a 2.5 ℓ flotation cell, and the pulp was agitated at an impeller speed of 1200 rpm. The pulp was conditioned with typical reagents using dosages that were found from the extensive historical test work carried out by Mintek on UG2 samples (Overbeek *et al.*, 1980; Liddell and Dunne, 1984; Liddell and Burger, 1988; Hinde and Joosub, 1997; Harris *et al.*, 2000; Naik, 2000; Darley, 2000; Hlahane, 2002; Fickling and Maharaj, 2002; Dass, 2003; Morgan, 2003; Maharaj *et al.*, 2004; Powell, 2004; Bowers and Maharaj, 2004; Roy, 2005; Roy and Dhilwayo, 2006; Powell and Stallknecht, 2006; Mbonambi and Mogosetsi, 2006; Makhanya, 2007a Sibanyoni, 2007; Maharaj *et al.*, 2007 and Mbonambi *et al.*, 2009).
3. The first reagent added was the collector SIBX at a dosage of 150 g/t, the pulp was conditioned for 2 minutes. It should be noted that a typical amount of depressant was added to the primary rougher stage (KU5 at 20 g/t) and the pulp was conditioned for a

further 3 minutes. Lastly, Dow200 frother was added to the pulp at a dosage of 40 g/t and the pulp was conditioned for a minute.

4. After conditioning, air was introduced into the agitated pulp at a rate of 8.4 Nℓ/min. The froth was scraped every 15 seconds, and five concentrates were collected at times 1, 3, 7, 20 and 30 minutes. The description of the experimental apparatus, i.e. flotation cell and machine were presented in § 3.3.3.2.

Once rate information in the primary rougher had been obtained the next test was carried out on the secondary rougher:

1. A 1 kg sample was milled and concentrated as per the test procedure described for the primary rougher, but in the primary rougher a concentrate was collected for 5 minutes only. This flotation time was chosen based on the PGM rate of recovery observed for the various UG2 samples from the variability study in Chapter 4. The rate response (Figure 48 in § 4.5.1) indicated that at 5 minutes, the curve ‘knead’ suggesting a transitioned from fast floating PGMs to slow floating PGMs.
2. Grinding tests on the primary rougher tail sample were performed to determine the time required to obtain the typical grind of 80% passing 75 µm (Hay and Schroeder, 2005 and Dunne, 2007).
3. The milled sample was then transferred into a 2.5 ℓ flotation cell, and it was conditioned at an impeller speed of 1200 rpm with 75 g/t SIBX, for 2 minutes, followed by a typical depressant dosage (KU5 at 20 g/t) for 3 minutes. Dow200 frother at a dosage of 40 g/t was added and the pulp was conditioned for a minute.
4. After conditioning, air was introduced into the flotation cell at a rate of 8.4 Nℓ/min; the froth was scraped every 15 seconds, and five concentrates were collected at times 1, 3, 7, 20 and 30 minutes.

Lastly, cleaner tests at different depressant additions were carried out according to the following procedure:

1. The rougher stages, both primary and secondary, were carried out according to the procedure outlined above. But, it was carried out in duplicate to generate sufficient mass for cleaner test work and for assaying. Sufficient mass is required to generate a

prill for assaying and the mass requirement increases with a lower grade concentrate and tail hence, the rougher was carried out twice.

2. In the cleaner stage, four depressant dosages were evaluated. The first dosage was 0 g/t, this can be considered as a blank test, which enables the linking of the rate information from the rougher to the cleaner stage. Three other depressant dosages were evaluated, the dosages selected depended on the sample being tested. For instance, a sample with significant floatable gangue requires a higher amount of depressant than a sample with a smaller amount of floatable gangue. The initial test at 0 g/t depressant was used to identify samples, which had abnormal amounts of floatable gangue. This was determined by examining the mass recovery with time and adjusting the depressant incrementally for each test, i.e. more depressant was added for tests, which had a higher cleaner mass recovery.

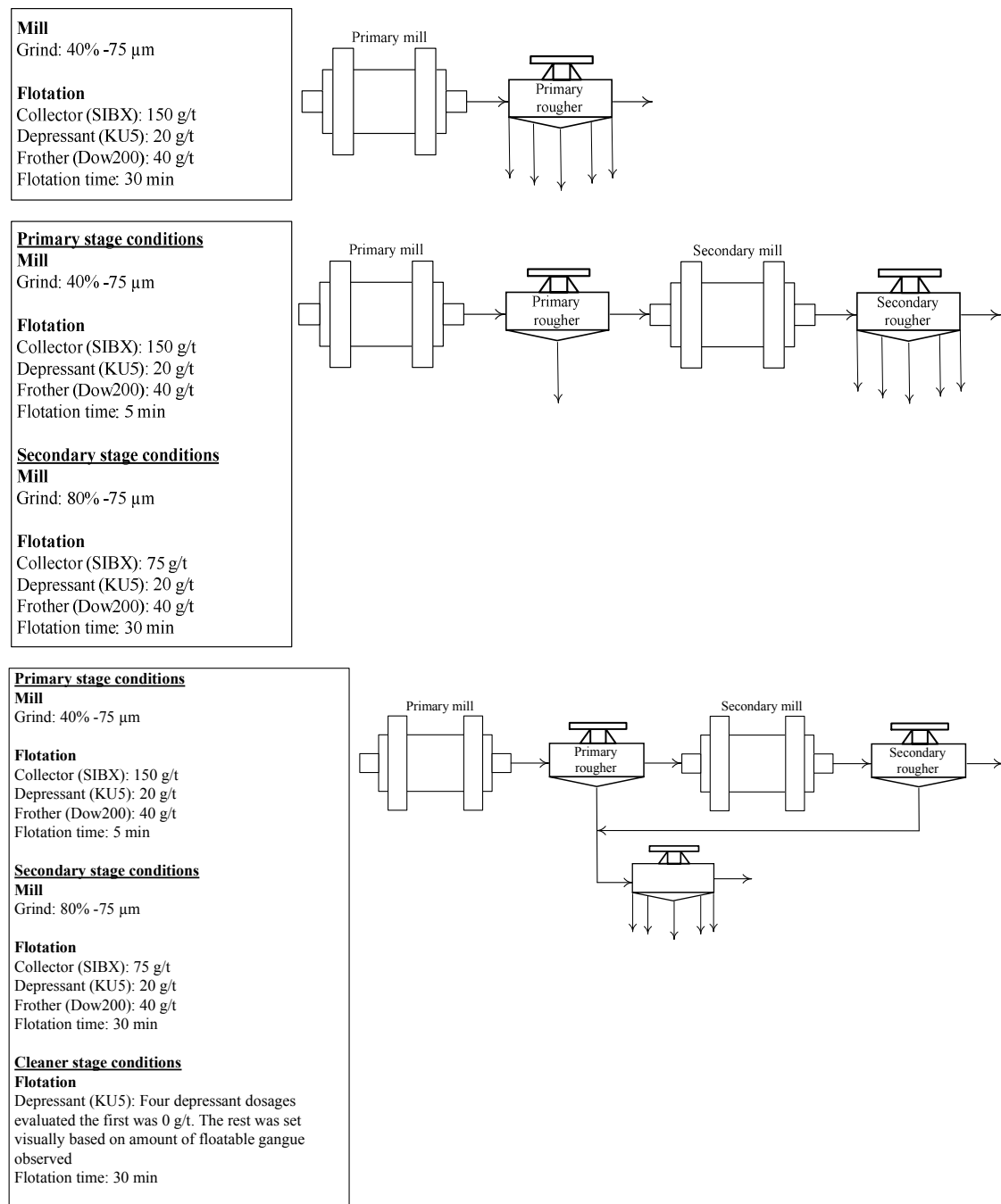


Figure 61: Batch flotation tests used to characterise the effects of grind and depressant addition

5.3. Modelling of batch flotation tests with grind and depressant effects

Suitable models for PGM, gangue and chromite recovery were chosen in Chapter 3 based on statistical discrimination. It was found that a model which includes entrainment of particles provides a better description of the minerals recovery. The inclusion of additional parameters for entrainment was justified by additional data in the form of incremental water recovery. However, the flotation test work proposed in this chapter has stage grinding and depressant

addition effects that extend across both roughing and cleaning. Therefore, a modelling methodology which encapsulates these effects is required.

Models for simulation of flotation such as JKSimFloat (Runge *et al.*, 1998 and Harris *et al.*, 2002), SUPASIM (Hay and Rule, 2003) and FLEET (Dobby and Savassi, 2005) assume that the flotation characteristics are maintained at all stages. The JKSimFloat method uses a global regression of data from batch tests on various plant products. The data is linked via a mass balance for the plant. An alternative approach is a direct regression of a model to plant data. The latter was used by Loveday and Hemphill (2006) for modelling a platinum flotation circuit. They accounted for the effects of regrinding by assuming a new set of parameters (mass fractions and rate constants) for material entering the secondary rougher.

The various batch tests in this work will be linked in a global regression, which is similar to the method developed at the Julius Kruttschnitt Mineral Research Centre (JKMRC) (Runge *et al.*, 1998; Alexander *et al.*, 2000 and Harris *et al.*, 2002) However, development of a new model was required to incorporate the effects of regrinding and addition of depressant.

In view of significant parameter inter-action between mass fractions and rate constants observed in Chapter 3, it was decided that the global regression should use the same rate constants for primary and secondary rougher flotation. Hence, the effect of re-grinding would be shown in an understandable way, by changes in the mass fractions. The finer particles produced by regrinding are recovered more readily by entrainment. For modelling purposes, it is expected that regrinding will increase the amount of entrainable material (R_{ent}) and that the value of ENT may also be affected. The parameters R_{ent} and ENT interact in regression and some simplifying assumptions are required for application of a global regression. The entrainment factor (ENT) for the cleaner stage was approximated by using the average of the values obtained in the two roughing stages, weighted in terms of the mass fractions of entrainable material:

$$ENT_{Cleaner} = \frac{(R_{ent} \times ENT)_{Primary\ rougher} + (R_{ent} \times ENT)_{Secondary\ rougher}}{R_{ent, Primary\ rougher} + R_{ent, Secondary\ rougher}} \quad (5.1)$$

The modelling exercise will be illustrated on one of the seven samples, and the best method will be applied to the rest of the samples. The sample selected is referred to as sample A.

5.3.1. PGM modelling

5.3.1.1. Model hypothesis

A basic requirement for modelling of a flotation circuit is that the particles retain their flotation or entrainment characteristics when they move from one stage to the next. Regrinding changes all the particles and a new set of parameters are required. However, where significant parameter interaction occurs, it is possible to assume some common parameters (Loveday and Marchant, 1972). Figure 62 illustrates the total number of parameters ($6 \times \text{primary rougher} + 6 \times \text{secondary rougher} + [6 \times \text{cleaner stage}] \times 4 = 36$). The total number of data points, for each mineral type, including the water recovery is 62. The behaviour in the cleaning stage must be linked to the model parameters in the two roughing stages. Various approaches are discussed below.

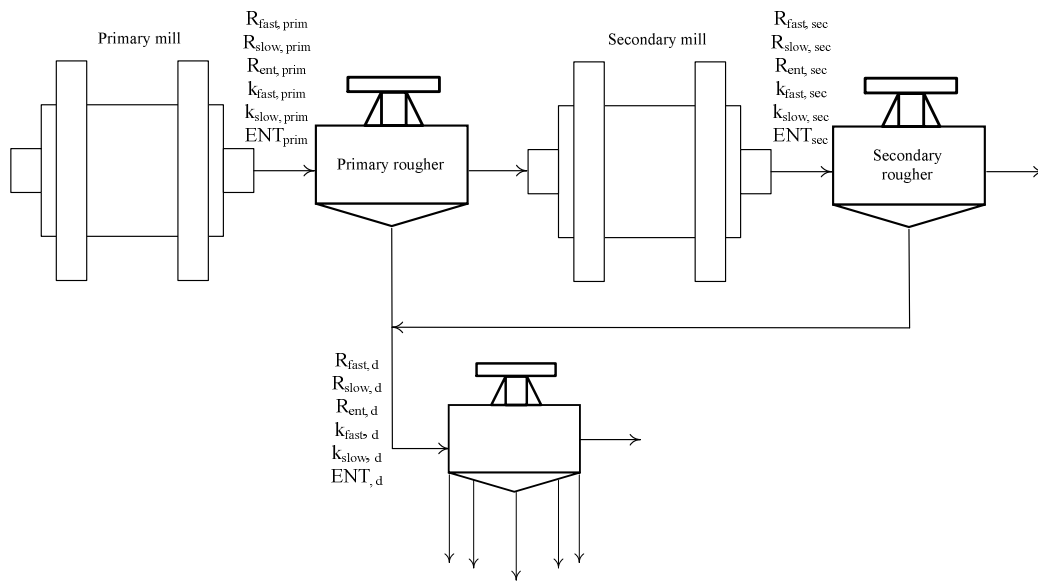


Figure 62: Parameters used in modelling PGMs for rougher–cleaner test work

Approach One– This approach has 13 parameters (i.e. $R_{fast, prim}$, $R_{slow, prim}$, $R_{ent, prim}$, ENT_{prim} , $R_{fast, sec}$, $R_{slow, sec}$, $R_{ent, sec}$, ENT_{sec} , $R_{slow, d2}$, $R_{slow, d3}$, $R_{slow, d4}$, k_{slow} and k_{fast}), which are determined by a global regression to the timed flotation tests, linked by a mass balance. The mass fractions (fast, slow and entrainable) are estimated for each rougher stage. The entrainment factor was also determined for primary and secondary roughing, and this was used to determine the entrainment factor in the cleaner stage using Equation 5.1. The mass

fractions sum up to a maximum recovery for each stage also; the mass fractions together with the rate constants and entrainment factors can be used to model the feed flotation properties to the cleaner stage. Note, the primary rougher was carried out for a prolonged time. However, for secondary roughing and cleaning tests a 5 minute primary rougher concentrate was taken. The prolonged primary rougher test provides more data points for modelling this stage.

Since additional depressant (d_2 , d_3 and d_4) was added to the cleaner, it was hypothesized that depressant influences the slow floating material only, which is mainly composite material. The fast floating material is mainly liberated large grain PGMs and PGMs attached to BMS therefore; these particles should intuitively not be influence by depressant addition. Depressant acts on gangue. Therefore, the assumption of the slow floating material being affected by depressant addition is logical. The mass fraction of the slow floating PGMs in the cleaner at each depressant addition ($R_{slow, d2}$, $R_{slow, d3}$, $R_{slow, d4}$) was estimated from the regression. A global fast and slow rate constant was also determined.

Approach Two– This approach also has 13 parameters (i.e. $R_{fast, prim}$, $R_{slow, prim}$, $R_{ent, prim}$, ENT_{prim} , $R_{fast, sec}$, $R_{slow, sec}$, $R_{ent, sec}$, ENT_{sec} , $k_{slow, d2}$, $k_{slow, d3}$, $k_{slow, d4}$, k_{slow} and k_{fast}). In this approach, the slow floating mass fractions in the cleaner stage are assumed to be constant, but the rate of flotation of the slow floating material was allowed to vary for each depressant addition. The other parameters are calculated in the same way as mentioned for Approach One.

Approach Three – This approach has 16 parameters (i.e. $R_{fast, prim}$, $R_{slow, prim}$, $R_{ent, prim}$, ENT_{prim} , $R_{fast, sec}$, $R_{slow, sec}$, $R_{ent, sec}$, ENT_{sec} , $k_{slow, d2}$, $k_{slow, d3}$, $k_{slow, d4}$, $R_{slow, d2}$, $R_{slow, d3}$, $R_{slow, d4}$, k_{slow} and k_{fast}). In this approach, both the rate of flotation and mass fraction of the slow floating PGMs in the cleaner are assumed to be variable with each depressant addition. The rest of the parameters were calculated as above.

5.3.1.2. Model evaluation

The fit of the modelling approaches with respect to the experimental data is shown by the parity charts in Figure 63. All the approaches gave a good fit to the experimental data. The adjusted coefficient of determination, which considers the parameters used in modelling, indicated that more than 90% of the data variation was accounted by the approaches.

There are significant differences between model and data in the recovery range 40 to 60%, this corresponds to the first concentrate from the primary rougher and cleaner tests. These data points occurred when there were rapid changes in the froth structure. The two points tending to 60% are for the primary rougher and cleaner at 0 g/t; these points are underestimated by the approaches, and this may be due to a large amount of floatable gangue occurring in this period. The floatable gangue causes the material to be recovered at a faster rate; this was also observed during the experiments. The froth was stable and the bubbles on the surface of the froth were very large and heavily mineralised. On the other hand, with increasing depressant dosage the initial concentrates from the cleaner tests were overestimated. The froth structure was less stable and the bubbles on the surface of the froth were smaller when the depressant dosage was increased, as shown by Figure 16 in Chapter 3. When the depressant dosage was increased some of the floatable gangue was suppressed, and this caused the froth to be less stable. Hence, the PGM recovery was lower.

Modelling of the froth using techniques discussed by Vera *et al.* (2002) may resolve this issue, however, the froth structure changes rapidly during a batch flotation test. This is due to depletion of the frother and the floatable gangue. The depletion is greatest at the beginning but decreases with time over the duration of the test. This is a challenge in the modelling of the batch froth, since it is time varying and dependent on frother and floatable gangue content in the pulp. Nevertheless, the marginal variation in the initial concentrates does not affect the overall model fits significantly, as indicated by the parity charts.

Approach Two shows a poor fit at the lower PGM recoveries (<20%); this corresponds to the secondary rougher data. Examination of this data showed that the approach, by comparison to the others, over predicts the entrainable material from the secondary rougher. This can be attributed to a constant slow floating mass fraction used in the cleaners. The approach apportions more material to the entrainable fraction. As more material is apportioned, to the entrainable fraction, less material was classified as slow floating and this provided a better cleaner fit, but poor secondary rougher fit.

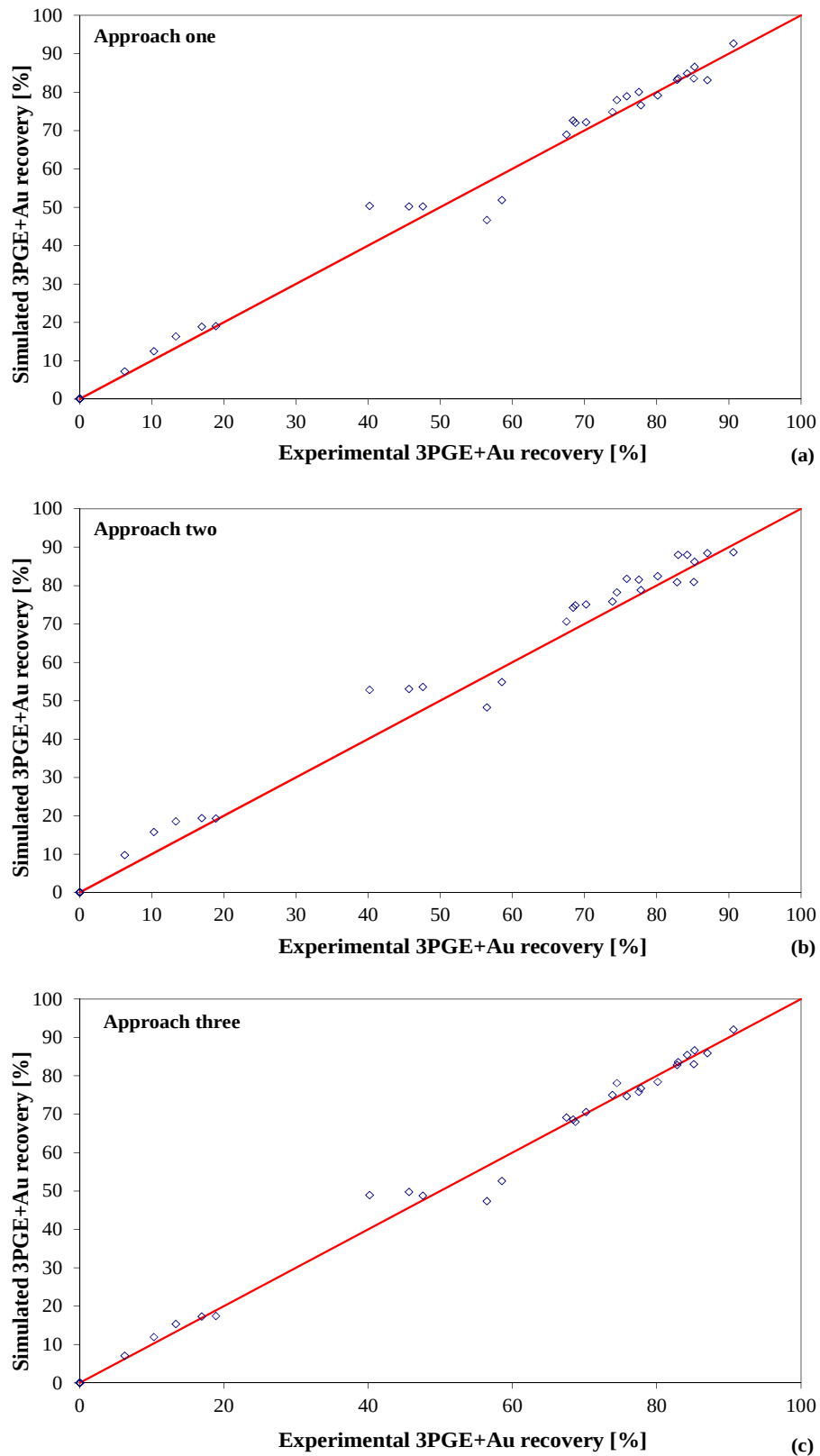


Figure 63: Parity charts for modelling approaches applied to rougher-cleaner PGM data

Figure 64 shows the flotation estimate obtained from the modelling approaches. The primary rougher flotation estimate is for a test carried out over a prolonged time, and the secondary

rougher flotation estimate is based on a 5 minute primary rougher concentrate. Therefore, the material sent to the cleaner stage is composed of the 5 minute primary rougher and the 30 minute secondary rougher flotation and entrainment estimate. Material that detaches during flotation can be entrained.

In general, the approaches indicate that depressant affects the rate of recovery. Both Approach Two and Three suggest that if the time was extended the recovery for a high depressant test would approach the recovery for a low depressant test. This is unlikely since the recovery should intuitively be lower for higher depressant additions. Therefore, application of Approach Two and Three is not logical. On the other hand, Approach One presents a better description of the PGM rate of recovery for depressant addition.

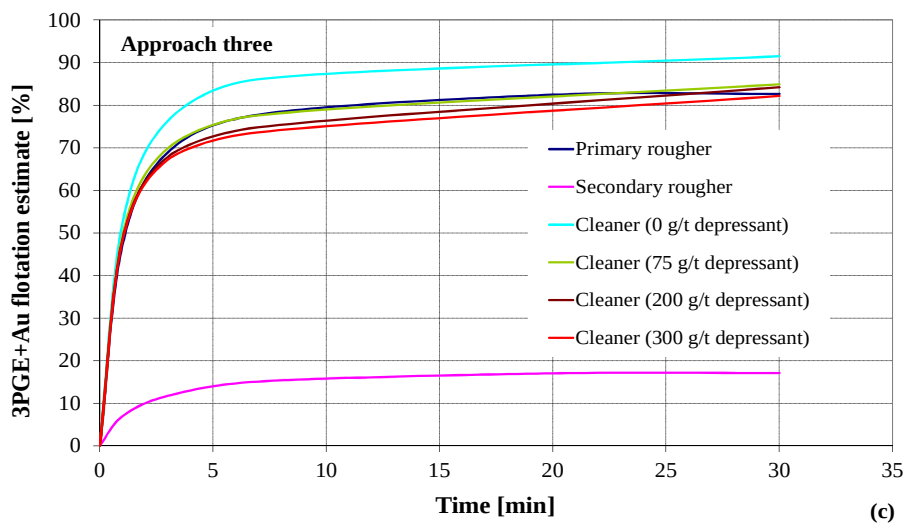
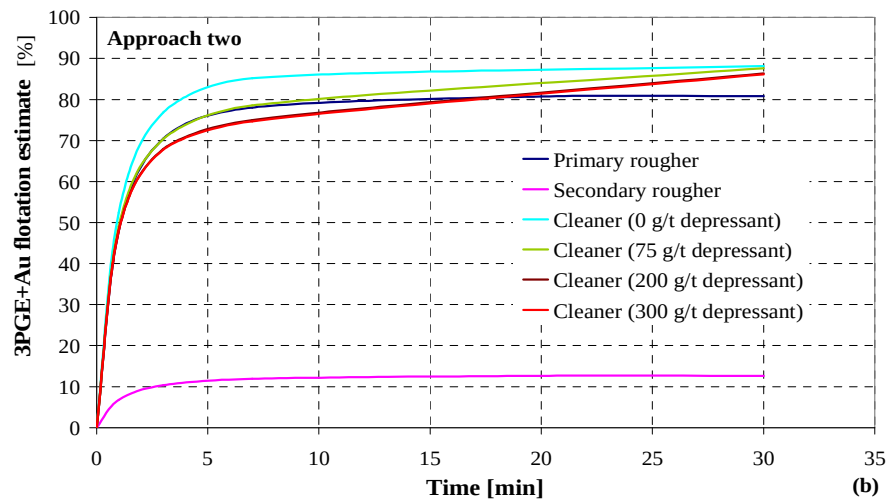
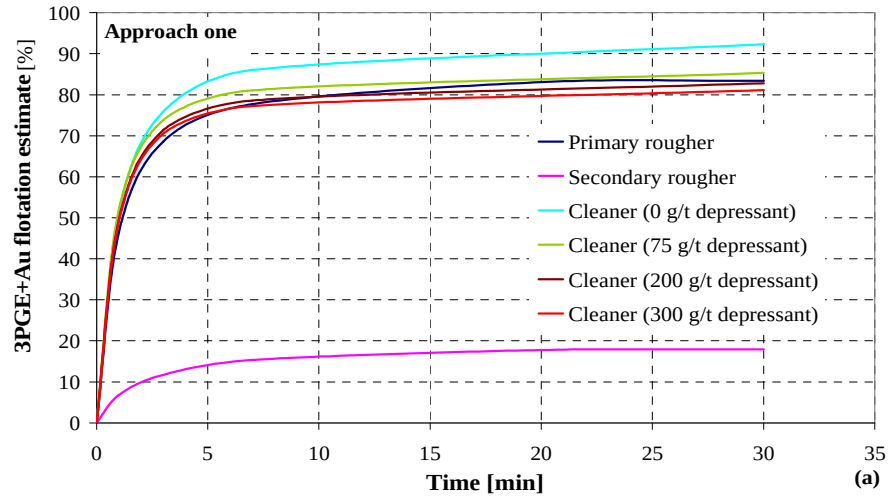


Figure 64: Flotation estimates from modelling approaches applied to rougher-cleaner PGM data

Figure 65 shows the entrainment model estimate for the approaches with time and with water recovery. The recovery by entrainment is small by comparison with the recovery by flotation.

Konopacka and Drzymala (2010) investigated different types of water recovery-entrainment plots and indicated that curves such as that seen in Figure 65 is due to less frother in the system and the feed containing coarse particles. The PGMs have a small grain size therefore; the bending of the curves cannot be attributed to the presence of coarse particles. Less frother may be a more rational explanation, since the curve bends around 5 minutes into the flotation test. Frother concentrations may be low at this time, due to most of the material being collected by flotation, which contains most of the frother with the water collected to the concentrates.

A further reason for the bending of the curve may be the concentration of PGMs in the pulp. During the test, most of the PGMs are recovered relatively quickly by flotation, and this reduces the PGM concentration in the pulp. The lower concentration limits the material available for entrainment. This is shown by the cleaner test at 0 g/t, for this test the curve asymptotes due to most of the PGMs being recovered by flotation. On the other hand, for the cleaners at higher depressant addition the curve has a lower degree of asymptoting, because depressant addition impedes the composite particle flotation. Hence, more PGMs are present in the pulp for entrainment.

The same behaviour is exhibited by the primary rougher, which has a lower degree of asymptoting by comparison to the secondary rougher. It is believed that the primary rougher has more PGMs in the pulp, and this material contributes to the entrainment estimate with time. On the other hand, the secondary rougher feed contains less PGMs due to the recovery of most of the PGMs from the primary rougher. Therefore, the limited amounts of PGMs in the pulp results in an asymptoting of the entrainment estimate with time and water recovery.

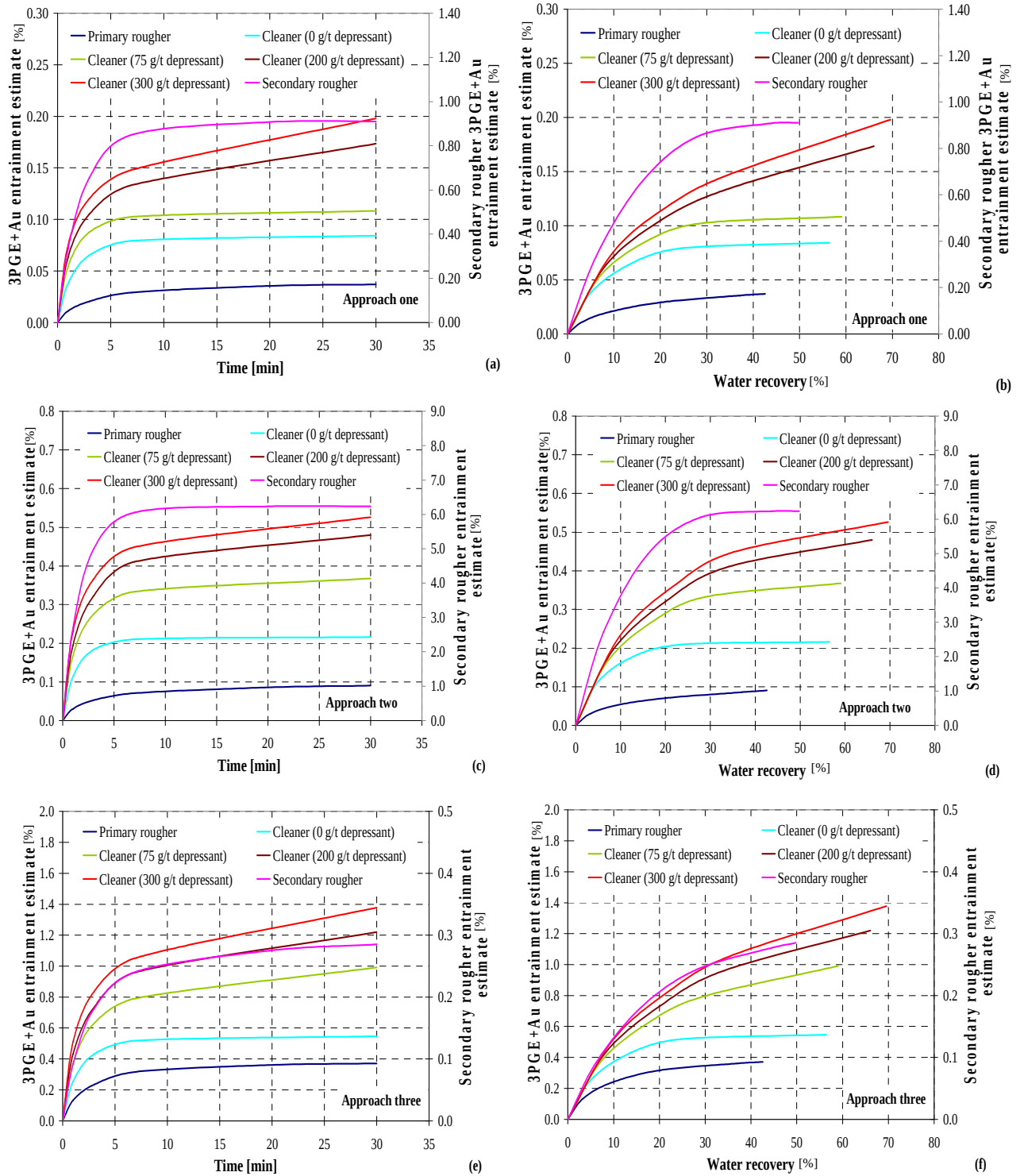


Figure 65: Model entrainment estimates for approaches applied to rougher-cleaner PGM data

Figure 66 presents the variation in the cleaner parameters for the three approaches. For Approach One an exponential function correlates well with the slow floating fraction. No notable trend exists for the parameters obtained by the other approaches. However, the model fitting indicates that depressant reduces the floatability of the slow floating material; since the

rate of flotation and the slow floating fraction decreases with increasing depressant dosage. Approach One offers insight into the tightening of the modelling of the rougher-cleaner data; a single parameter model which fits the mass fractions can be used to reduce the number of parameters. The result of this is a more compact modelling approach. This will be examined in the next section, when the selection of the modelling approach for PGMs will be evaluated using statistical methods presented in Chapter 3 for model selection.

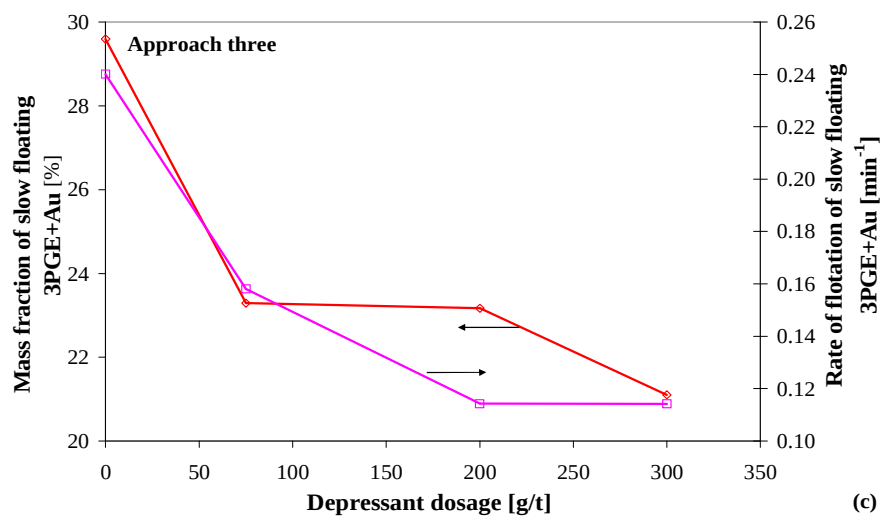
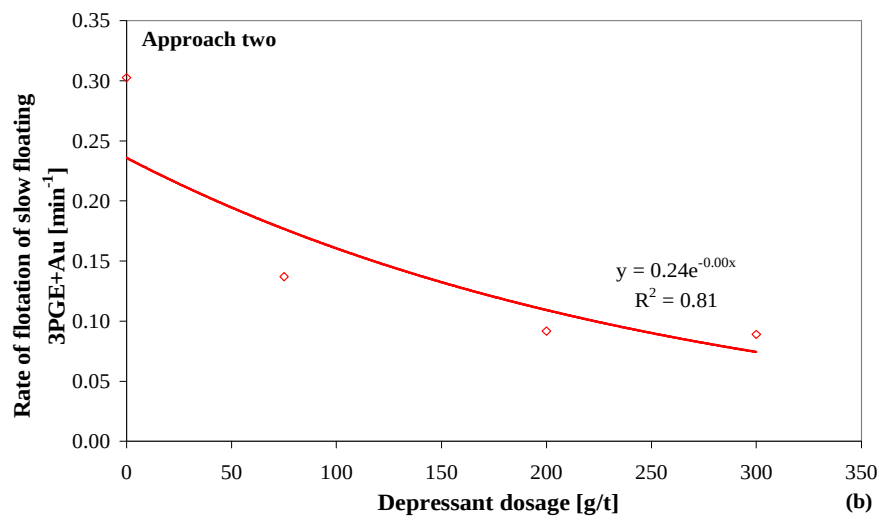
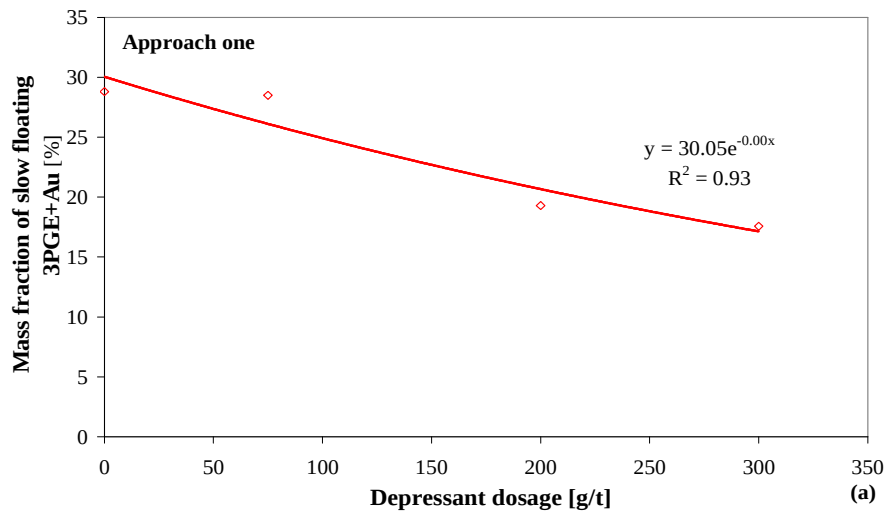


Figure 66: Variation in cleaner parameters with depressant for modelling approaches applied to rougher-cleaner PGM data

5.3.1.3. Model selection

Three approaches for modelling the rougher-cleaner data were evaluated in the previous section. The approaches, which considered variation in the slow floating rate constant, do not provide a logical interpretation of the action of depressant in flotation. Hence, the approach which assumes a variation in the slow floating mass fraction was selected, since it gave a better description of the test. Examination of the variation in the mass fraction indicates it can be correlated with depressant addition, using an exponential function. This correlation was examined as another modelling approach, called Approach Four.

Figure 67 presents a summary of the model fit, and the flotation and entrainment model estimates. The parity chart indicates that Approach Four offers a good fit to the experimental data. This is expected since the application of an exponential function should not result in a significant loss in the fit, because it is modelling the observation for an unconstrained regression, i.e. Approach One. The flotation and entrainment estimates do not deviate significantly from those observed for Approach One.

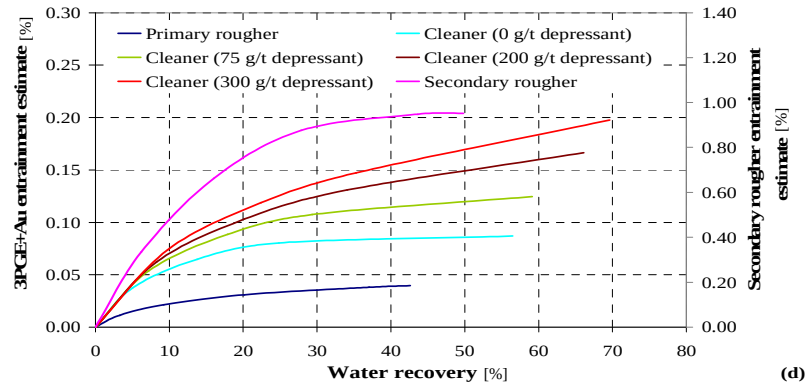
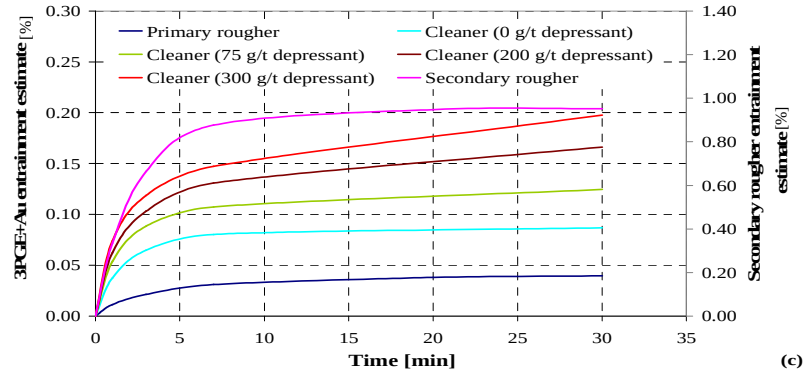
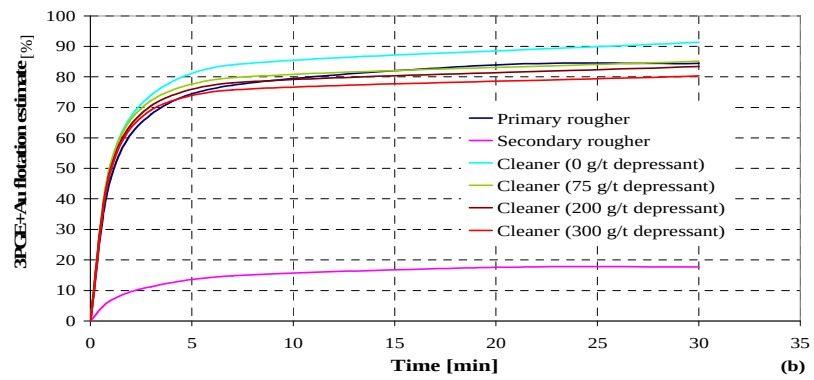
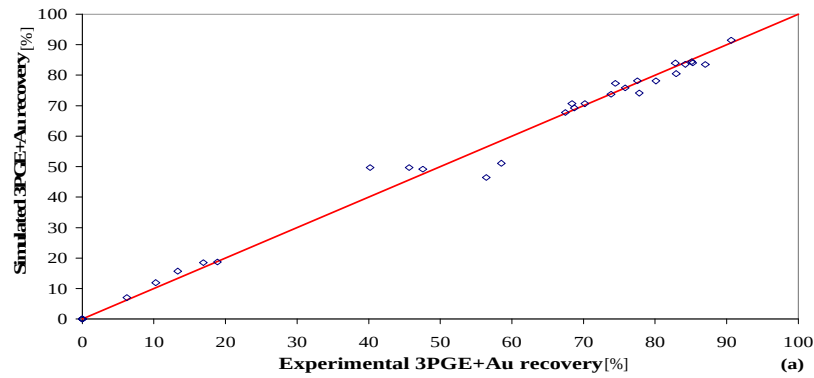


Figure 67: Summary of modelling approach assuming an exponential decay for variation in slow PGM mass fraction

Table 15 presents a summary of the statistics for the modelling approaches. Approach Two and Three are included in this summary, even though it did not provide a logical description of the rate of flotation of the PGMs. Approach One or Four can be used to model the PGM data. Approach Four is a compact model, which gives a good fit to the data by comparison to Approach One. Furthermore, the model selection and the S_p criteria, indicate that Approach Four is marginally better than Approach One. The lower model squared error for Approach Four implies that the parameters obtained from regression have a narrow range of variation, which is due to the fewer parameters used in modelling. Hence, Approach Four is the best approach because it is mathematically compact and is statistically better than Approach One.

Table 15: Summary of statistics for model approaches applied to PGM rougher-cleaner data

Statistic	Approach One	Approach Two	Approach Three	Approach Four
r_{adj}^2	0.986	0.977	0.990	0.988
MSC	4.093	3.586	4.378	4.237
S_p	0.152	0.250	0.118	0.129
MSE	2.725	3.568	2.301	2.542

Table 16 shows the parameter estimate from the different modelling approaches. The parameters for the primary rougher are for an extended flotation test, and the parameters for the secondary rougher are for a primary rougher having a 5 minute concentrate collected.

Approach Four, which is the approach selected for modelling PGM data, indicates that most of the PGMs are recovered from the primary rougher, because there is a larger floating component at this stage. The secondary rougher has slower floating material, which is reasonable, since this stage has composite PGMs and PGMs with a smaller grain size, by comparison to the primary rougher.

The parameter θ is an exponential parameter, which links depressant dosage to the slow floating PGM mass fraction. Equation 5.2 expresses the relation between mass fraction and depressant dosage. This indicates that for a depressant dosage d_i , the fraction of slow floating material available for recovery by flotation, is the product of the slow floating fraction at a dosage of 0 g/t in the cleaner ($R_{slow, d0}$), and the value from the exponential function for depressant addition d_i . The relationship has many advantages such as, providing a

mathematical relationship for modelling depressant addition on flotation mineral recovery. This is attractive for understanding the effect of depressant addition with respect to composite particles (mineralogy) and for plant simulation. Chapter 6 and 7 will discuss these applications in greater detail.

$$\frac{R_{\text{slow},d_i}}{R_{\text{slow},d_0}} = e^{(-\theta \times d_i)} \quad (5.2)$$

Table 16: Summary of parameter estimates obtained from global regression for PGM rougher–cleaner data

Parameter	Approach One	Approach Two	Approach Three	Approach Four
$R_{\text{fast, prim}}$	57.62	58.12	56.46	56.41
$R_{\text{slow, prim}}$	25.82	22.72	26.47	28.13
$R_{\text{ent, prim}}$	0.74	0.87	2.25	0.74
ENT_{prim}	2.64×10^{-3}	1.61×10^{-3}	7.10×10^{-4}	2.54×10^{-3}
$R_{\text{fast, sec}}$	6.24	6.29	6.23	6.23
$R_{\text{slow, sec}}$	12.59	12.61	11.06	12.50
$R_{\text{ent, sec}}$	1.09	1.04	1.61	1.09
ENT_{sec}	3.63×10^{-3}	3.03×10^{-2}	1.00×10^{-3}	3.62×10^{-3}
$R_{\text{slow, d2}}$	28.48	-	23.29	-
$R_{\text{slow, d3}}$	19.29	-	23.17	-
$R_{\text{slow, d4}}$	17.57	-	21.10	-
$k_{\text{slow, d2}}$	-	0.14	0.16	-
$k_{\text{slow, d3}}$	-	0.09	0.11	-
$k_{\text{slow, d4}}$	-	0.09	0.11	-
$k_{\text{fast}} [\text{min}^{-1}]$	1.26	1.29	1.34	1.33
$k_{\text{slow}} [\text{min}^{-1}]$	0.22	0.30	0.24	0.19
θ [t/g]	-	-	-	-1.60×10^{-3}

5.3.2. Gangue modelling

5.3.2.1. Model hypothesis

The gangue species, which are minerals that are not chromite or PGMs, was modelled in Chapter 3 using a single floatable fraction and recovery by entrainment. The methods developed earlier for evaluation of the PGM model were repeated for gangue recovery. However, fewer parameters were used, since there is a single floatable fraction.

5.3.2.2. Model evaluation

The fit for the different approaches with respect to the experimental data is shown in Figure 68. The fit to the data is significantly worse for Approach Two.

There is more deviation for the modelling of the gangue data than that observed for the PGM modelling. Gangue is recovered mainly by entrainment, as indicated by Figure 69 and Figure 70. It is believed that the use of a single entrainment factor does not account for the various silicate minerals, which have different densities. The gangue can be resolved into several dominant silicate phases to improve the modelling, but this would require mineralogical examination, which will be expensive to implement on each flotation product. Modelling, according to particle size, may improve the definition of the entrainment factor. Harris (2000) modelled the UG2 gangue recovery for pilot plant data according to particle size, but found that the fit was under or overestimated in certain parts of the plant. The resolution of the gangue into several dominant silicate species, together with modelling according to particle size, will offer a more improved model for gangue recovery. However, this approach will result in an expensive batch flotation method, which from an industrial perspective is not favourable, due to cost and time constraints. Nevertheless, Approach One and Three offer a reasonable fit to the data despite the gangue having a spectrum of densities and particle sizes.

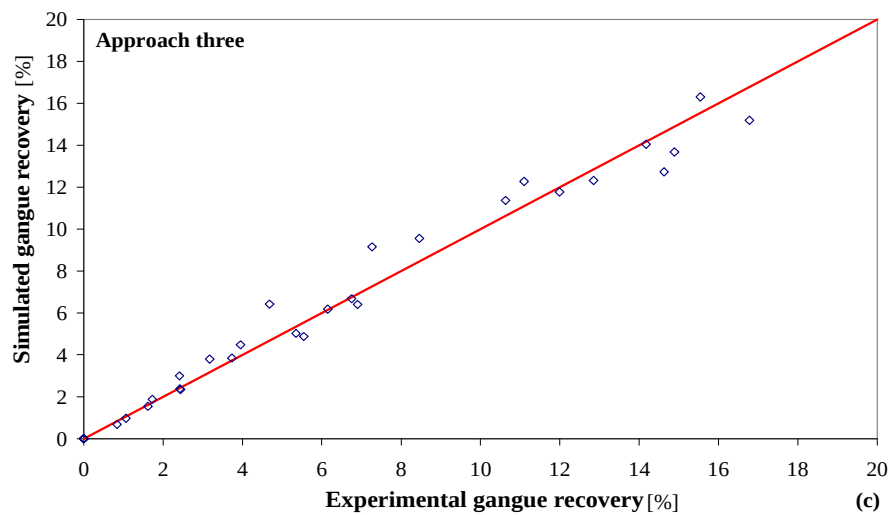
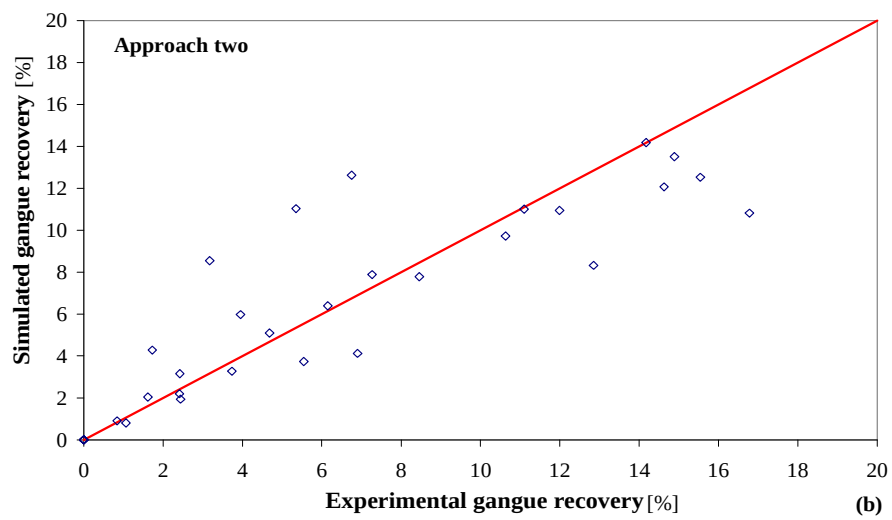
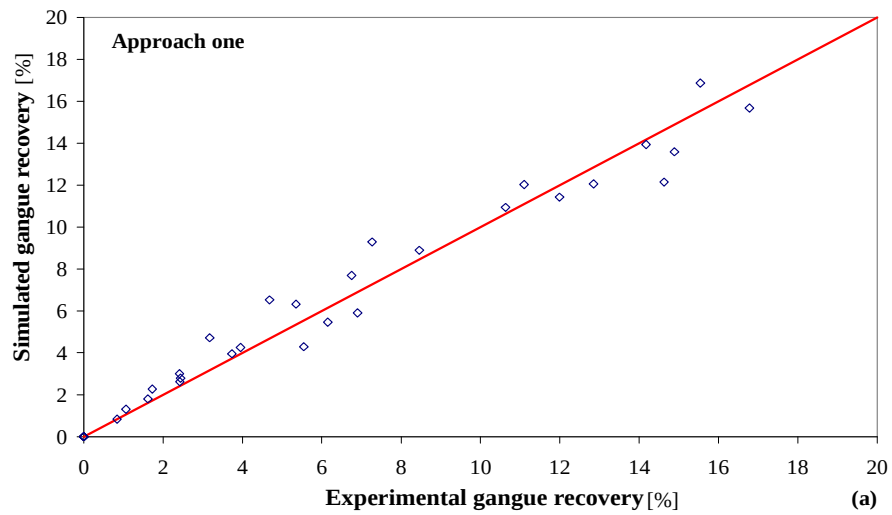


Figure 68: Parity charts for approaches used to model gangue for rougher-cleaner test work

Figure 69 and Figure 70 shows the flotation and entrainment estimates for the different approaches. Approach One provides a logical description of the flotation and entrainment estimate than Approach Three. Approach Three forces the rate of flotation at the highest cleaner depressant dosage to zero; this means that there is no gangue recovery by flotation. This is unlikely since depressant was added incrementally such that the froth was not completely depressed and as a result there was floatable gangue present. The low floatable estimate is likely due to interaction between the entrainment and flotation terms in the modelling approaches.

Approach One is better suited for describing the gangue recovery. The approach indicates that the flotation estimate in the cleaner stage decreases when depressant dosage increases. The gangue recovery reaches an asymptote with each depressant dosage, which is typical for depressant addition. The higher gangue recovery at 0 g/t depressant addition is due to the gangue recovered from both the primary and secondary rougher.

The slope of the entrainment estimate with water recovery increases with increasing depressant in the cleaner. This can be attributed to depressant suppressing the floatable gangue, which results in a loose froth structure that has more water in the spaces between the bubbles. The water contains entrainable material, and this is recovered with each scrape to the concentrate collector tray. The secondary rougher has a higher entrainment estimate than the primary rougher. The higher secondary rougher entrainment is due to finer grinding of the primary rougher tails, which generates more material in the finer particle range. This material has a higher probability of being entrained.

Figure 71 shows the variation in the cleaner parameters with depressant for the different approaches. Approach One indicates that the variation in the floatable gangue fraction can be modelled using an exponential function, similar to that observed for the PGM modelling. The variation in the rate constant could also be modelled using an exponential function but the function does not model the rate at the lower depressant dosages sufficiently. Approach Three shows that the rate and mass fraction decrease with increasing depressant addition. However, it does not follow any notable trend. A polynomial of order n may be used to model the variation in rate and mass fraction, as applied by Amelunxen and Amelunxen (2009b) in the modelling of reagent effects for a molybdenum ore. This does not simplify the model if the order is greater than two.

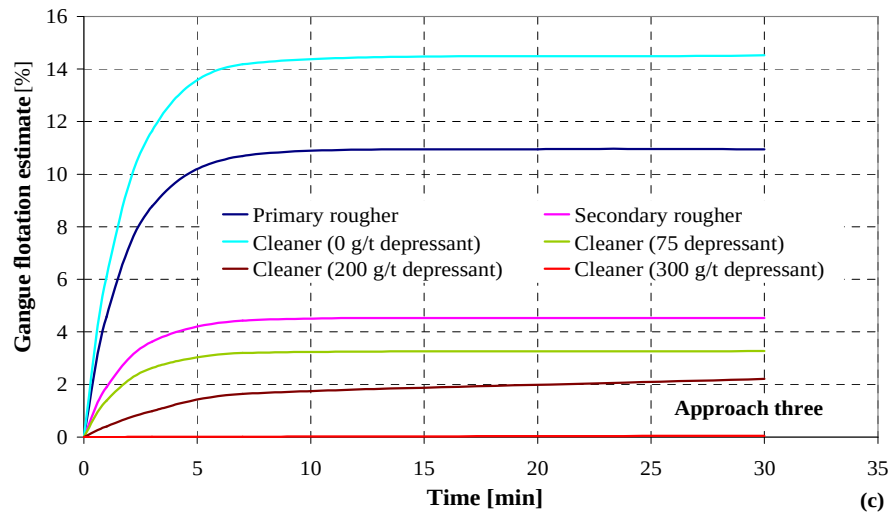
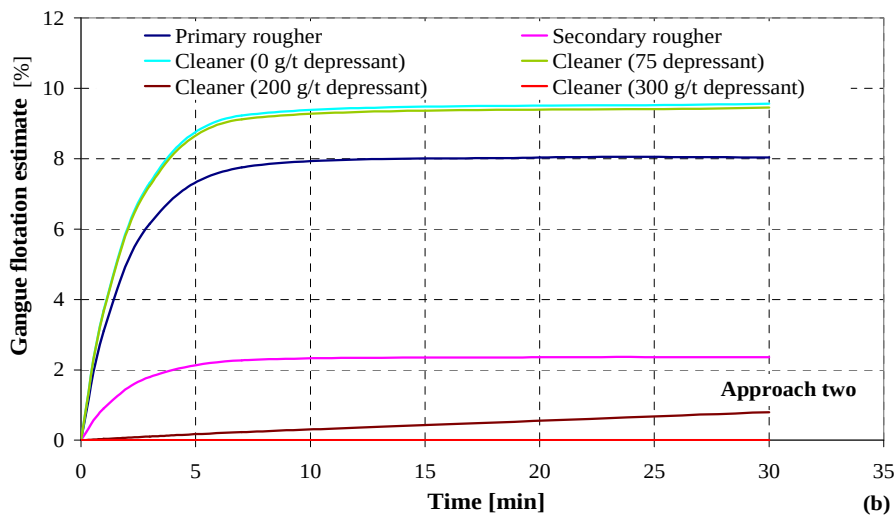
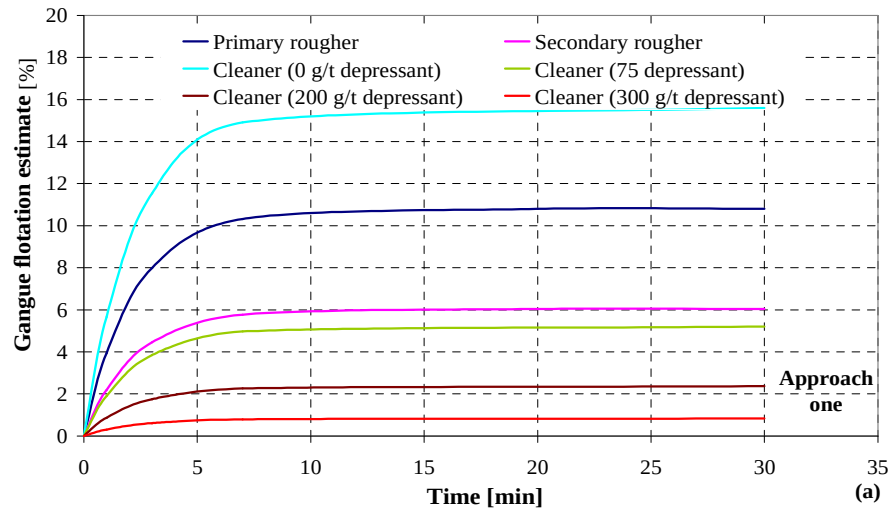


Figure 69: Gangue model flotation estimate for rougher-cleaner test work

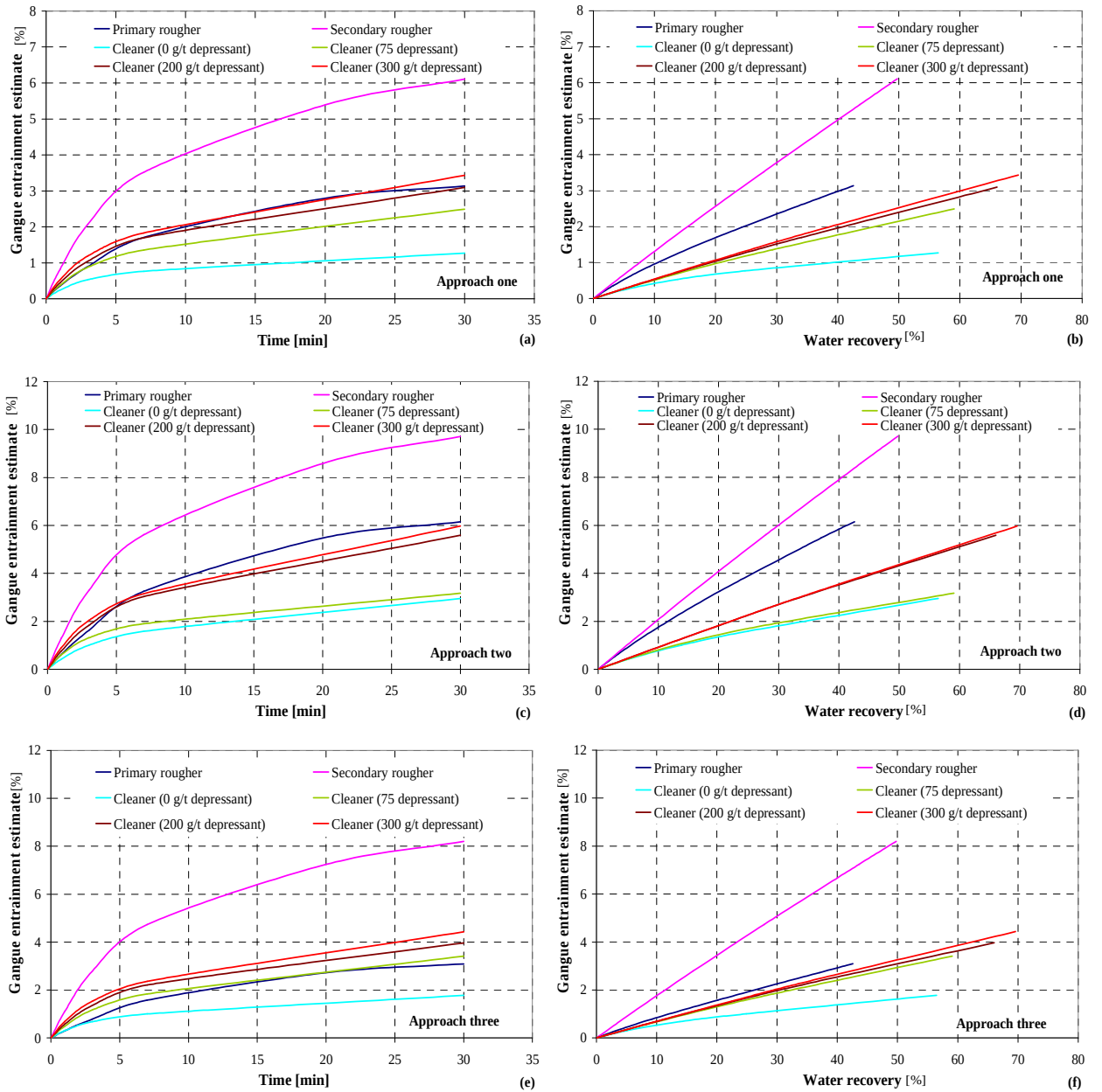


Figure 70: Gangue model entrainment estimate for rougher-cleaner test work

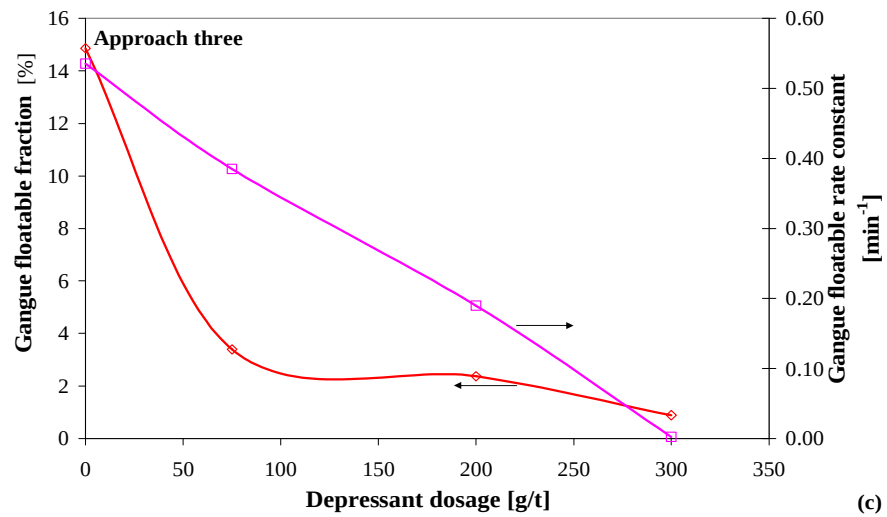
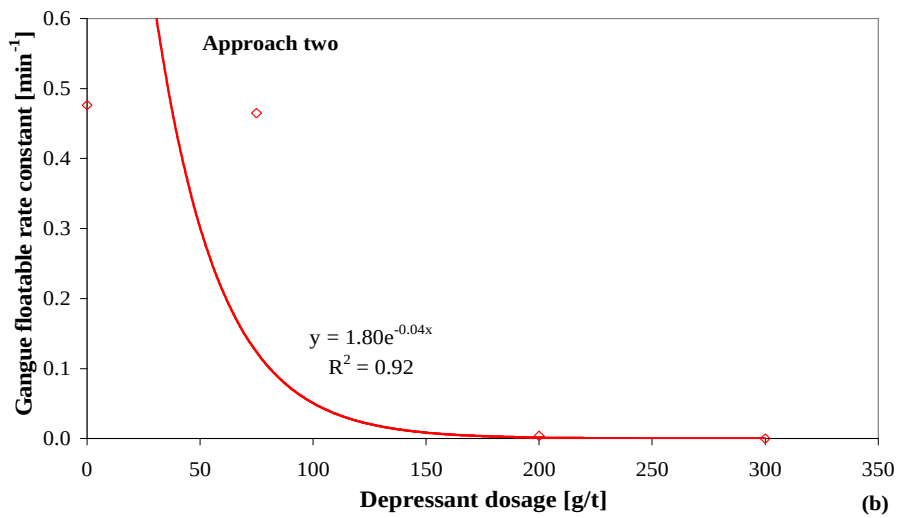
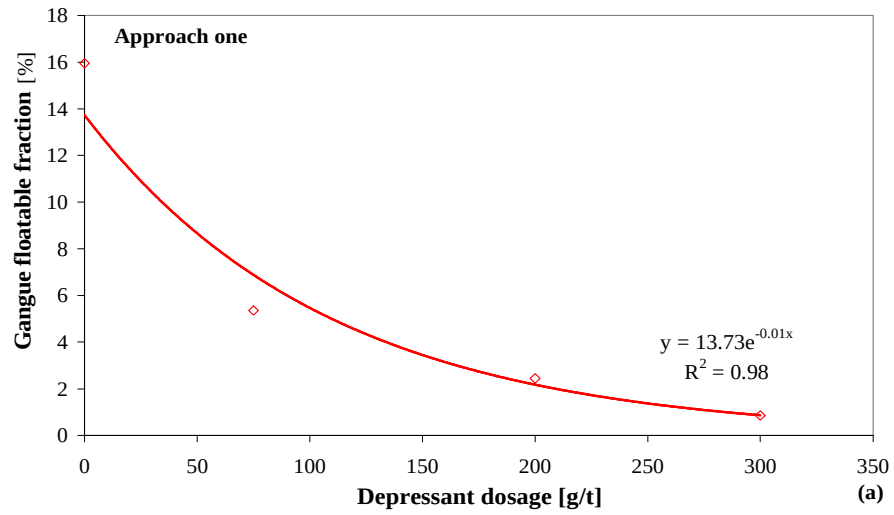


Figure 71: Variation in cleaner gangue model parameters for rougher-cleaner test work

5.3.2.3. Model Selection

The approach which considered a variation in the mass fraction with depressant addition gave a good fit and description of the rougher-cleaner data. In addition, it was found that an exponential function describes the variation in the mass fraction with depressant addition. A fourth modelling approach with an exponential variation in the mass fractions was applied to the data; Figure 72 presents a summary of the fit and flotation and entrainment estimate for this approach. This approach has a good fit to the experimental data in addition, the description of the flotation and entrainment estimates was logical.

Table 17 presents the statistical measures for the approaches. The second and third approach did not provide a good fit and/or description of the rougher-cleaner data therefore; these approaches cannot be used to model the data. The statistical measures suggest that Approach One is better than Approach Four, but the difference in the measures is marginal. Furthermore, as discussed in the PGM data, a compact modelling approach is preferred. Since fewer parameters infer that fewer data points are required in regression, and there is less parameter interaction. Hence, Approach Four will be used to model the gangue data.

Table 18 gives a summary of the parameters obtained from the approaches. The primary rougher parameters are for a prolonged test, and the parameters for the secondary rougher and cleaner tests are based on a primary rougher carried out for 5 minutes. The sum of the floatable and entrainable fraction gives the maximum recovery for the rougher stage; material not recovered may be too coarse for entrainment or have no floatable properties. In both rougher stages, there is more entrainable material than floatable material, which is expected since the modal proportion of floatable gangue (talc, serpentine, amphiboles and most secondary silicates) is smaller by comparison to the non-floatable gangue (pyroxene, feldspar, quartz and most primary silicates). Most of the floatable gangue is obtained from the primary rougher, and a smaller fraction is obtained after finer grinding. This concurs with industrial and pilot plant operations, which operate these stages with a shallow froth height and additional frother, so that a stable froth can be obtained, due to less floatable gangue. θ is a fitting parameter for the exponential modelling of the floatable mass fractions with depressant addition.

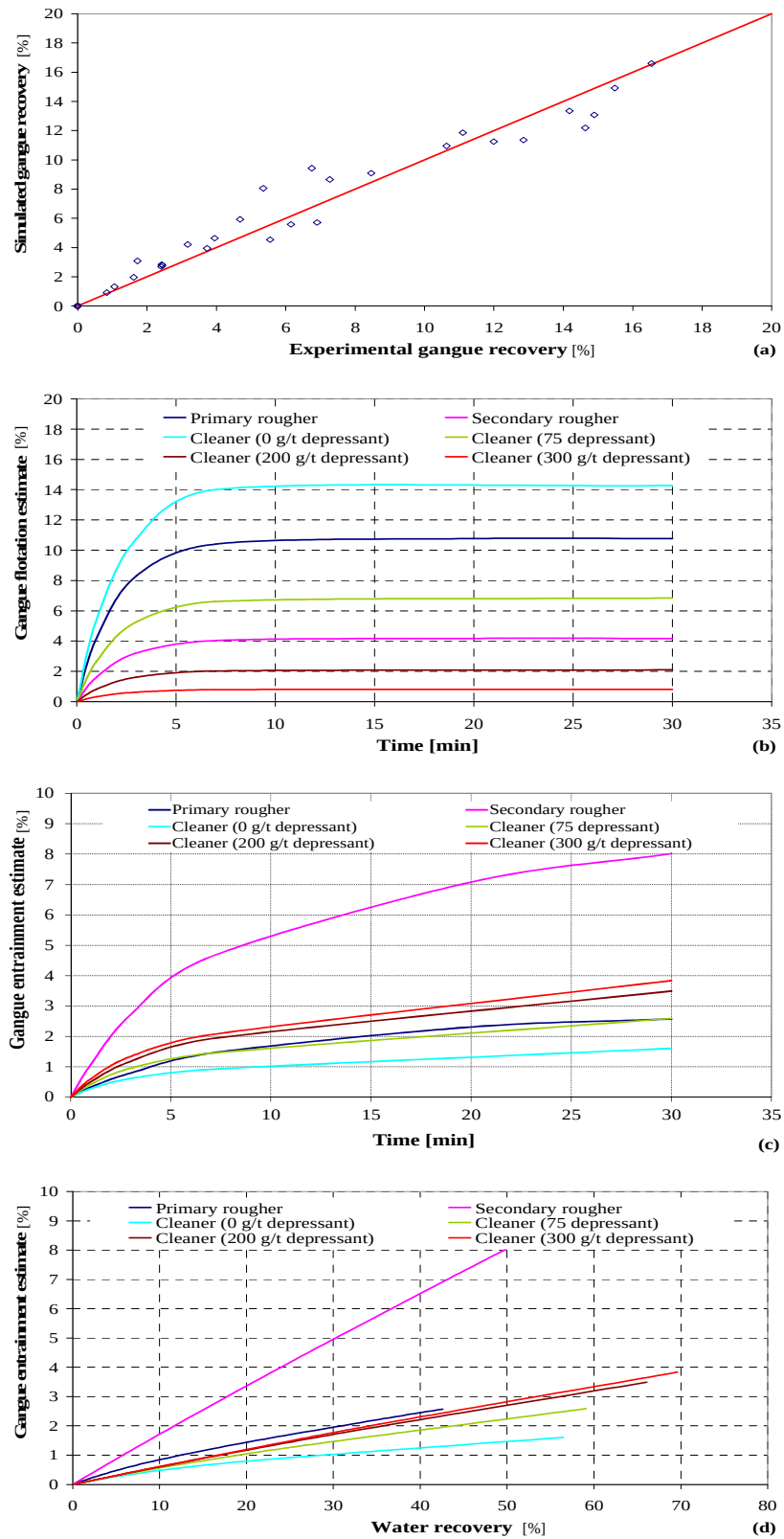


Figure 72: Summary of modelling approach assuming an exponential function for variation in gangue floatable mass fraction

Table 17: Summary of statistics for model approaches applied to gangue rougher–cleaner data

Statistic	Approach One	Approach Two	Approach Three	Approach Four
r_{adj}^2	0.962	0.740	0.972	0.939
MSC	3.122	1.207	3.386	2.678
S_p	0.010	0.070	0.008	0.016
MSE	0.730	1.902	0.628	0.937

Table 18: Summary of parameter estimates obtained from global regression for gangue rougher–cleaner data

Parameter	Approach One	Approach Two	Approach Three	Approach Four
$R_{float, prim}$	11.10	8.32	11.09	11.10
$R_{ent, prim}$	17.83	28.49	31.68	12.30
ENT_{prim}	4.08×10^{-3}	5.47×10^{-3}	2.30×10^{-3}	4.77×10^{-3}
$R_{float, sec}$	6.14	2.42	4.61	4.26
$R_{ent, sec}$	83.23	86.95	84.70	85.01
ENT_{sec}	1.51×10^{-3}	2.37×10^{-3}	2.03×10^{-3}	1.97×10^{-3}
$R_{float, d2}$	5.35	-	3.39	-
$R_{float, d3}$	2.44	-	2.37	-
$R_{float, d4}$	0.86	-	0.88	-
$k_{float, d2} [min^{-1}]$	-	0.47	0.38	-
$k_{float, d3} [min^{-1}]$	-	3.87×10^{-3}	0.19	-
$k_{float, d4} [min^{-1}]$	-	1.47×10^{-5}	2.19×10^{-3}	-
$k_{float} [min^{-1}]$	0.44	0.48	0.54	0.48
$\theta [t/g]$	-	-	-	-9.44×10^{-3}

5.3.3. Chromite modelling

5.3.3.1. Model hypothesis

The modelling approach used for the gangue species was applied in the modelling of the chromite data. Since chromite and gangue have the same batch models, i.e. a single floatable fraction and an entrainment component.

5.3.3.2. Model evaluation

The fit of the approaches to the experimental data is shown in Figure 73. Approach One and Three have a better fit than Approach Two. As was found for the PGM and gangue data, assumption of a variable rate constant only does not fully describe cleaner depressant addition. Therefore, modelling of the chromite data can be accomplished using Approach One or Three, since Approach Two does not provide a reasonable description of the chromite recovery. Figure 74 describes the flotation estimate for the approaches. In Approach Three, the interaction between the floatable rate constant and mass fraction indicates that the cleaner at 200 g/t depressant approaches the same recovery as the 75 g/t test, which is not possible. Approach One, on the other hand, gives a better description of the flotation estimate, which shows a decrease in the chromite rate of recovery with increasing depressant addition.

Figure 75 shows the entrainment estimates for the modelling approaches. Approach One will be considered because of the good model fit and logical flotation and entrainment estimate. The higher recovery with depressant was also observed for the PGMs and gangue, and is due to a brittle froth structure. A brittle froth contains more water in the bubble voids, and this water carries entrainable material, which is recovered with scraping. Furthermore, at higher depressant dosages, there is less floatable gangue therefore; the froth is less stable at prolonged flotation times, which was observed for times greater than 20 minutes. No additional frother was used to stabilise the froth because this affects the recovery and result in an additional variable in the batch modelling. The unstable froth promotes water recovery, which consequently, encourages recovery by entrainment.

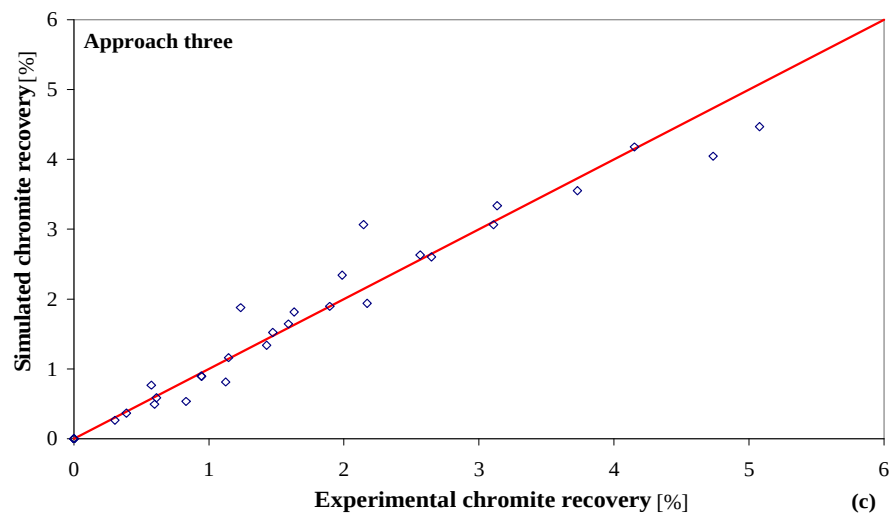
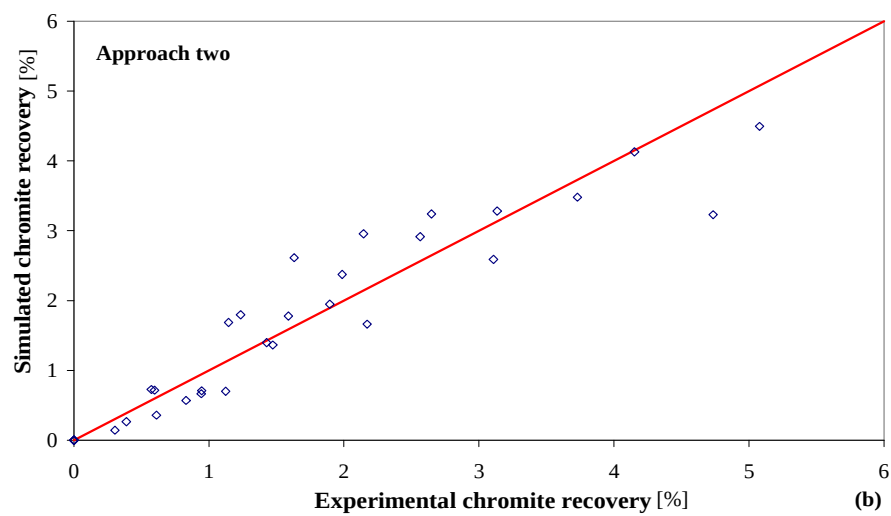
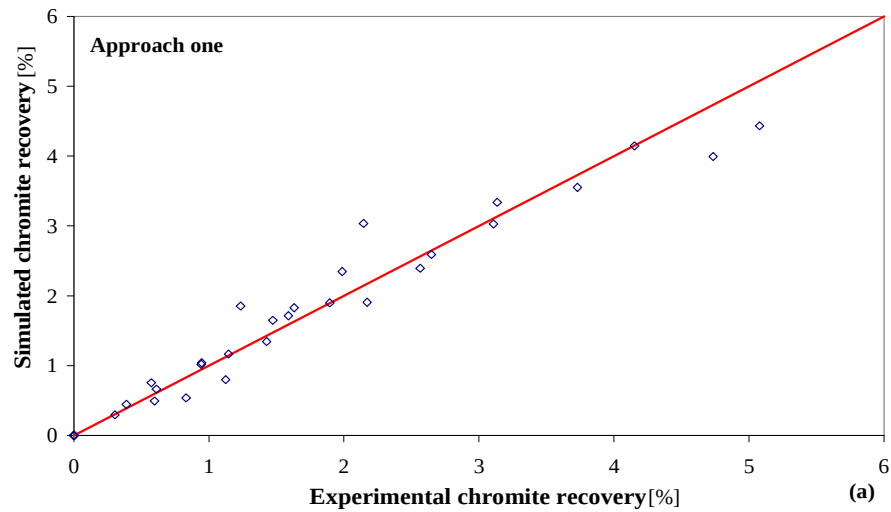


Figure 73: Parity charts for approaches used to model chromite for rougher-cleaner test work

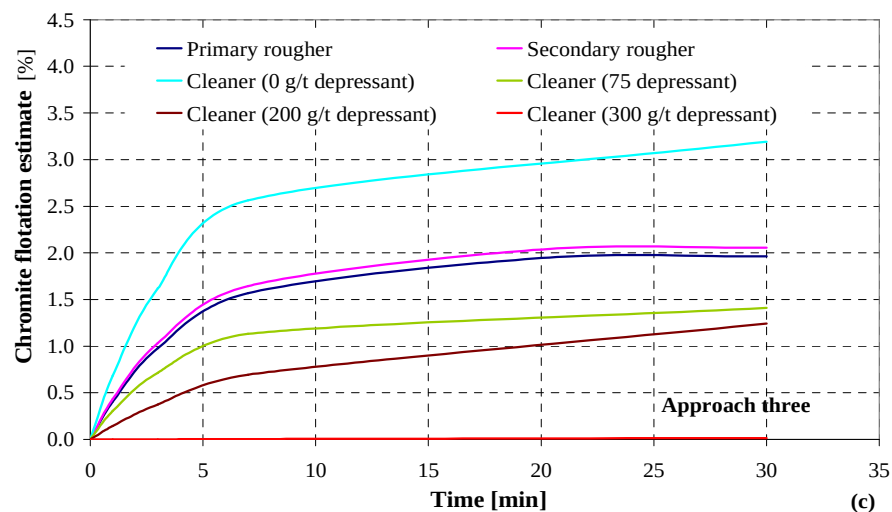
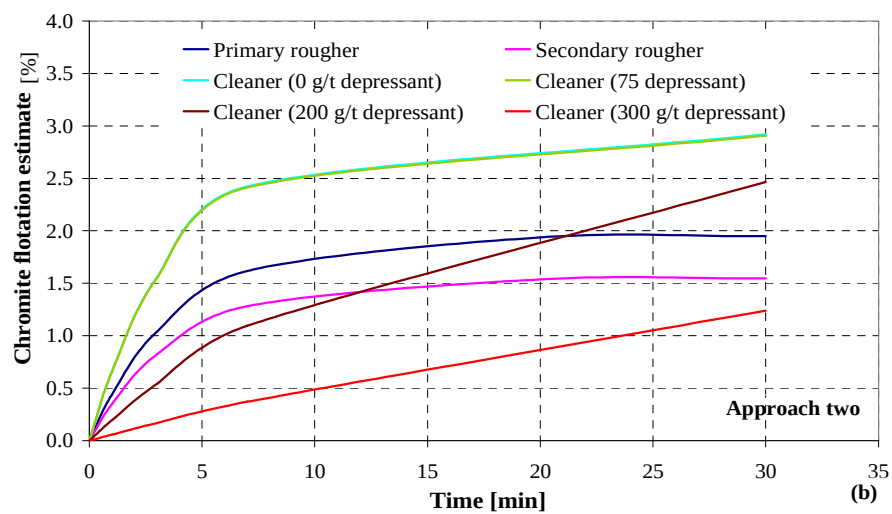
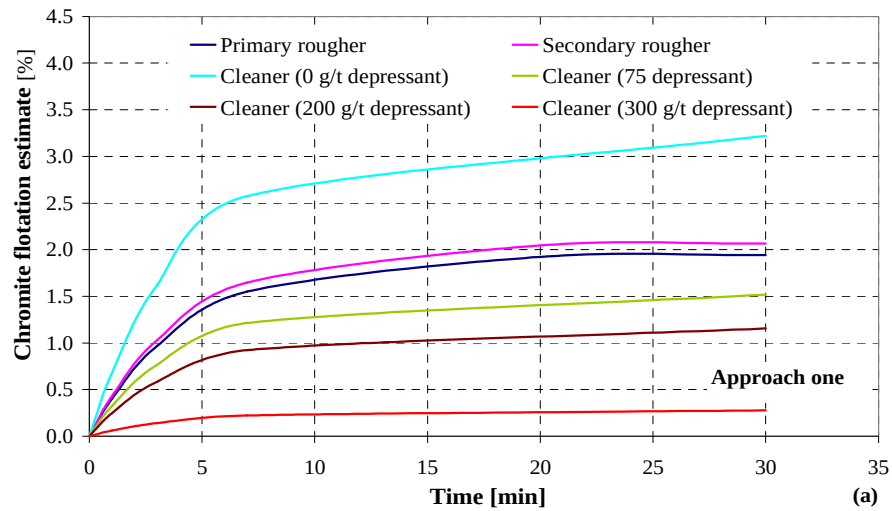


Figure 74: Chromite model flotation estimate for rougher-cleaner test work

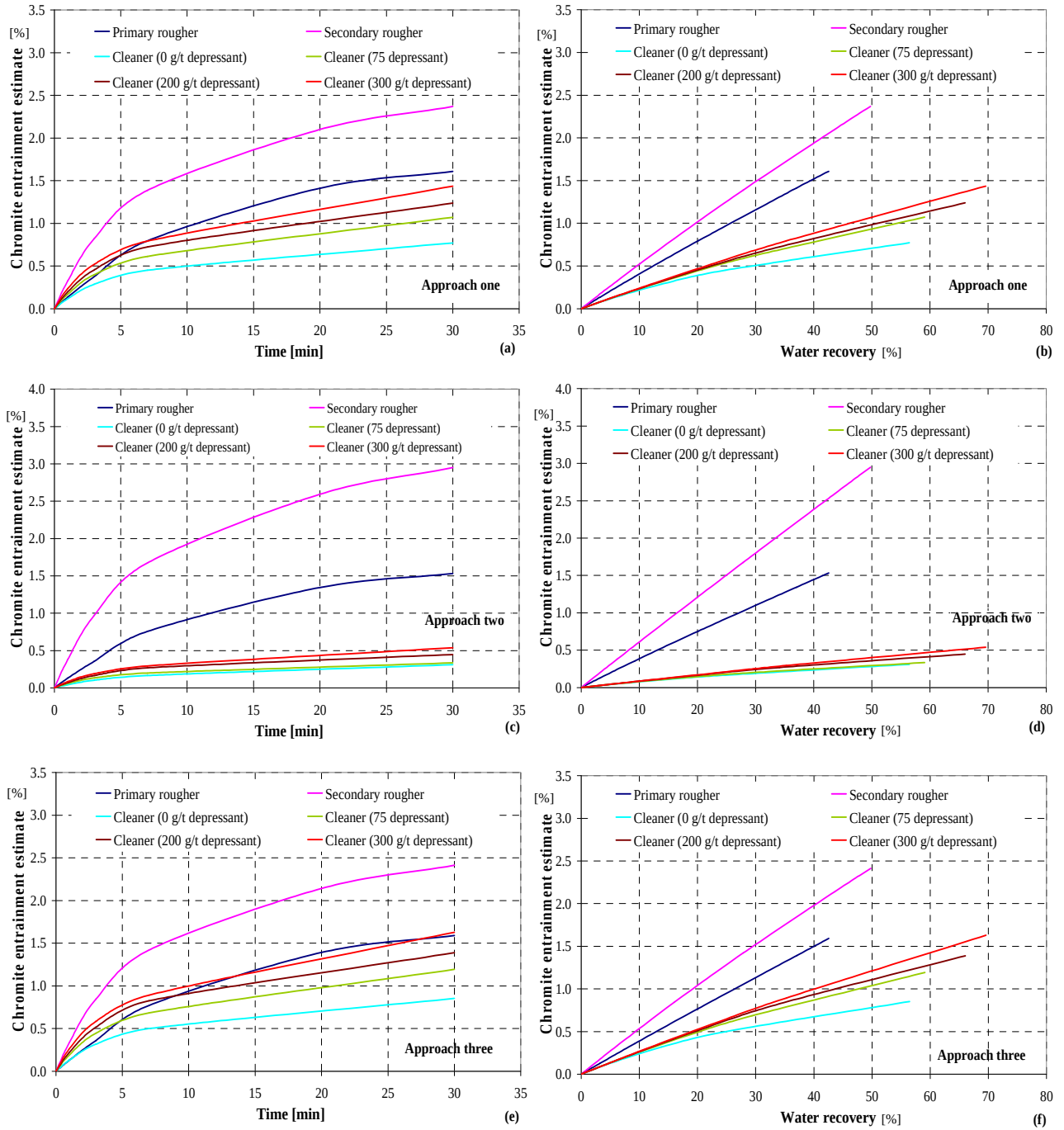


Figure 75: Chromite model entrainment estimate for rougher-cleaner test work

Figure 76 shows the variation in the cleaner parameters for the approaches. The variation in the chromite floatable fraction can be correlated with an exponential function. The rate constant can also be correlated with an exponential function, but it overestimates the rate at zero depressant addition. No trend exists for the variable rate and mass fraction from Approach Three but both parameters decrease with depressant, indicating a decrease in floatability.

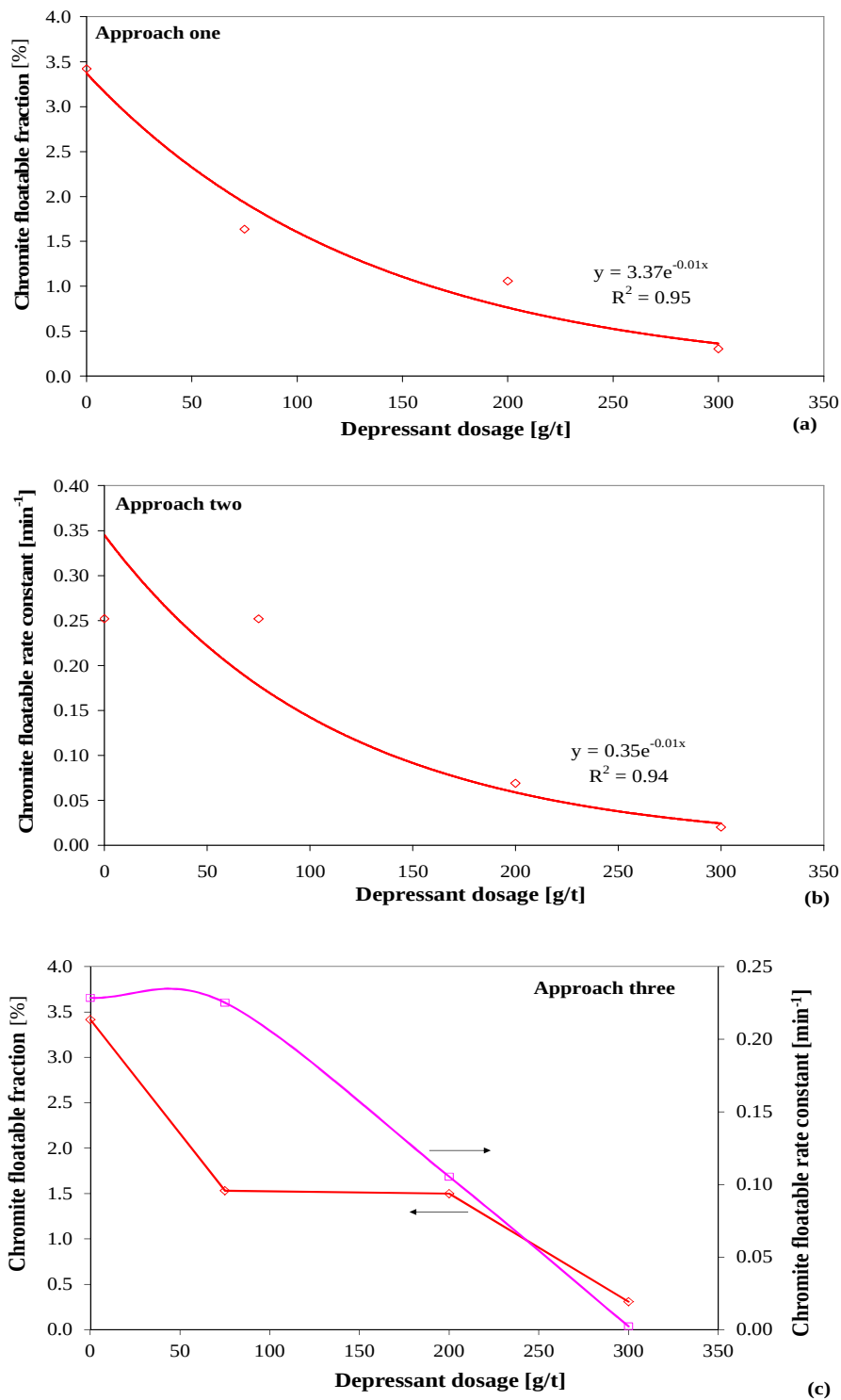


Figure 76: Variation in cleaner chromite model parameters for rougher-cleaner test work

5.3.3.3. Model Selection

The rougher-cleaner species modelling indicates that the variation in floatable mass fraction gives a good representation of the data. In addition, an exponential model correlates the

variable mass fractions with depressant. This provides a significant simplification in the modelling and enables the evaluation of the effect of depressant on mineral recovery at dosages not tested.

Figure 77 shows the results for modelling of the chromite data using an exponential function for the mass fractions in the cleaner stage. There is no significant loss in the model fit by comparison to Approach One, and the flotation and entrainment estimates from the model are reasonable. The approaches are compared statistically in Table 19. Approach Four by comparison to Approach One does not show a significant difference in the statistical measures, since the fitting statistic (R_{adj}^2) and the model selection statistics (MSC and S_p) vary marginally. Hence, model Approach Four will be used to model chromite recovery.

Table 20 presents a summary of the parameter estimates for the different approaches. There is a small amount of floatable chromite present in the flotation tests, and this may be chromite that is associated with floatable gangue or PGMs such as Laurite (Von Gruenewaldt *et al.*, 1986). The maximum recovery of chromite in both stages is less than that predicted for PGMs and gangue, and is due to chromite particles being coarse and having a high mineral density.

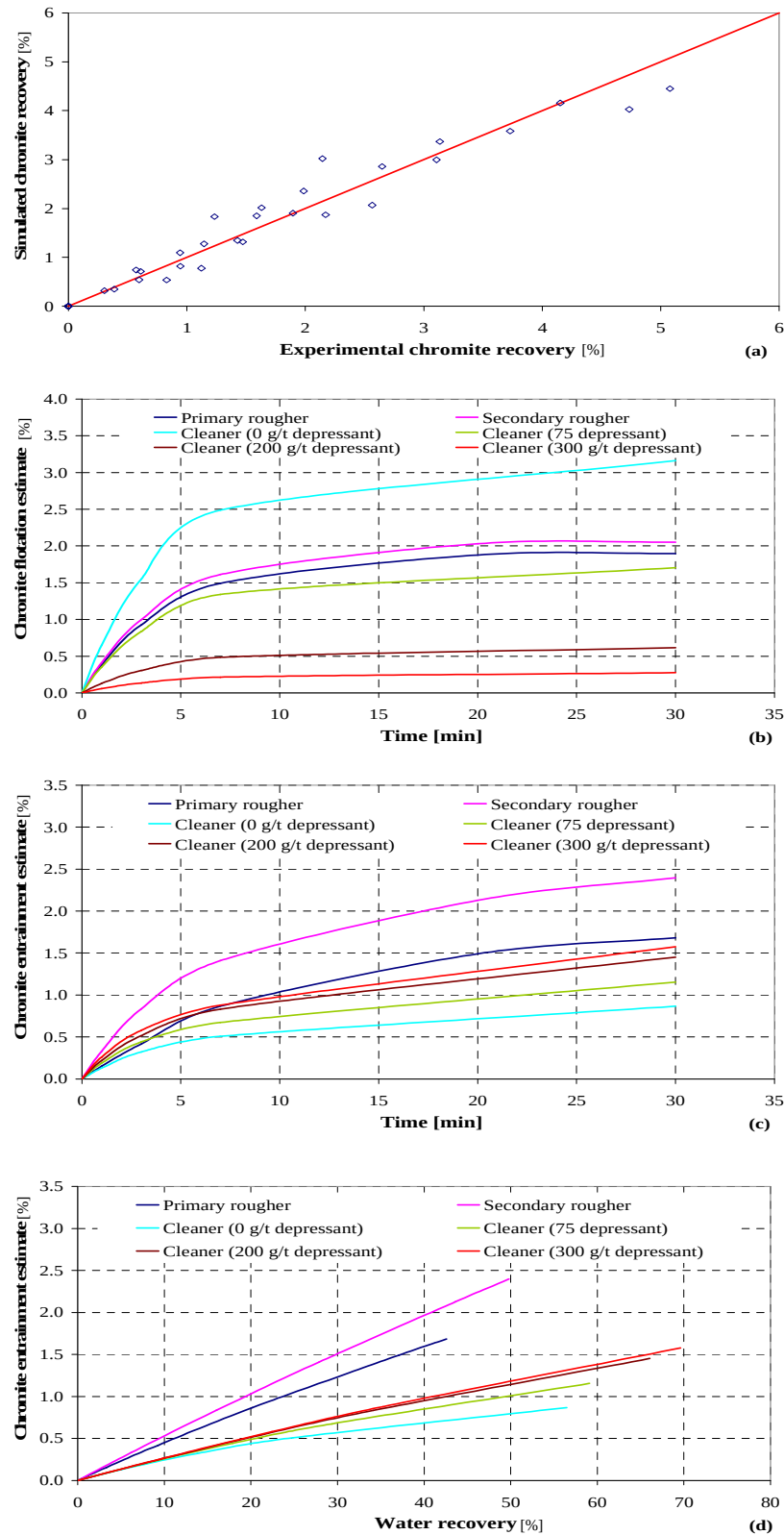


Figure 77: Summary of modelling approach assuming an exponential decay for variation in chromite floatable mass fraction

Table 19: Summary of statistics for model approaches applied to chromite rougher–cleaner data

Statistic	Approach One	Approach Two	Approach Three	Approach Four
r_{adj}^2	0.946	0.872	0.946	0.939
MSC	2.778	1.913	2.736	2.651
S_p	1.09×10^{-3}	2.37×10^{-3}	1.06×10^{-3}	1.13×10^{-3}
MSE	0.228	0.351	0.208	0.243

Table 20: Summary of parameter estimates obtained from global regression for chromite rougher–cleaner data

Parameter	Approach One	Approach Two	Approach Three	Approach Four
$R_{float, prim}$	1.99	1.90	1.89	1.92
$R_{ent, prim}$	20.70	21.24	37.78	11.33
ENT_{prim}	1.85×10^{-3}	1.72×10^{-3}	9.96×10^{-4}	3.59×10^{-3}
$R_{float, sec}$	2.14	1.57	2.12	2.10
$R_{ent, sec}$	22.64	72.34	20.59	21.10
ENT_{sec}	2.15×10^{-3}	8.30×10^{-4}	2.42×10^{-3}	2.34×10^{-3}
$R_{float, d2}$	1.64	-	1.53	-
$R_{float, d3}$	1.06	-	1.50	-
$R_{float, d4}$	0.30	-	0.31	-
$k_{float, d2} [min^{-1}]$	-	0.25	0.23	-
$k_{float, d3} [min^{-1}]$	-	0.07	0.11	-
$k_{float, d4} [min^{-1}]$	-	0.02	2.21×10^{-3}	-
$k_{float} [min^{-1}]$	0.23	0.25	0.23	0.22
$\theta [t/g]$	-	-	-	-8.04×10^{-3}

5.4. Application of modelling to UG2 samples

The PGMs, gangue and chromite were modelled using approaches that considered a variation in the floatable mass fraction, rate constant and both mass fraction and rate constant in the cleaner stage for depressant addition. The best approach was a variation in the floatable mass fraction, since it gave a good model fit and logical description of the rougher-cleaner flotation

data. Furthermore, the approach was simplified by modelling the floatable mass fraction as a function of depressant using an exponential model. The simplified approach, called Approach Four, will be applied to the other samples to determine the flotation parameters. The model fits and parameter estimates will be discussed in the next sections. Reasons for the variation in the PGM parameters will also be discussed in the next chapter.

5.4.1. PGM model fit

Figure 78 shows the model fit to the PGM rougher-cleaner data for all the samples evaluated. The model provides a good fit to the data and it illustrates the ability of the model to model samples with different feed characteristics. Table 21 presents a summary of the PGM parameters. The slow floating fraction in the cleaner stage at 0 g/t is also shown, since this together with the exponential depressant model (having a parameter θ) was used to model the variation in the slow floating recovery with depressant addition. The recovery in the primary rougher consists mainly of fast floating PGMs, which have a large grain size or is associated with larger BMS. The secondary rougher has slower floating PGMs and entrainable PGMs, which is due to finer grinding.

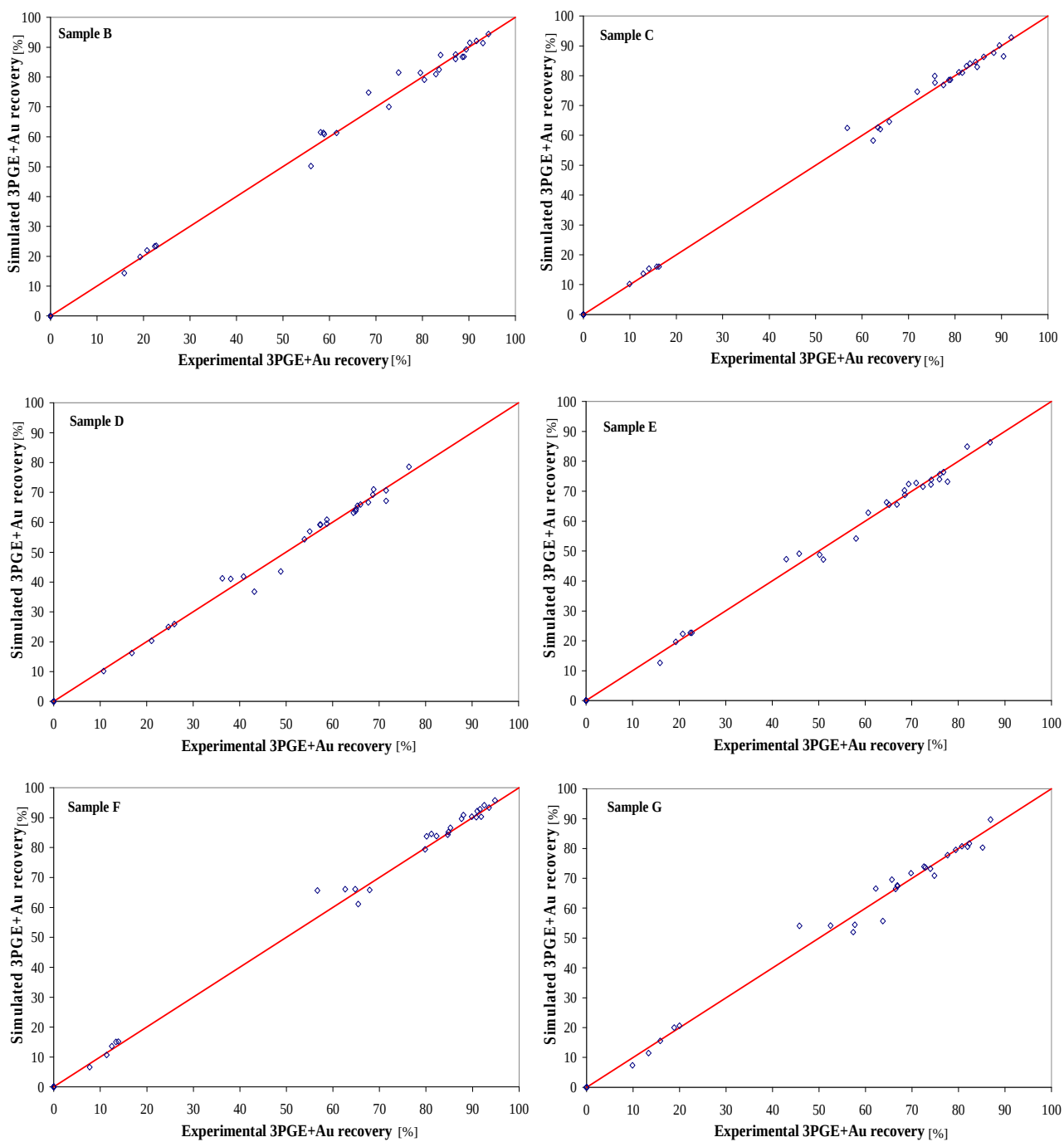


Figure 78: PGM parity charts for other samples evaluated

Table 21: PGM parameter estimates for other samples evaluated

Parameter	Sample B	Sample C	Sample D	Sample E	Sample F	Sample G
$R_{fast, prim}$	57.11	55.51	44.07	38.05	66.23	53.49
$R_{slow, prim}$	29.46	25.67	22.25	34.88	23.94	28.13
$R_{ent, prim}$	0.96	0.39	1.37	0.99	1.00	0.74
ENT_{prim}	8.54×10^{-3}	5.55×10^{-2}	2.43×10^{-2}	5.31×10^{-3}	3.45×10^{-3}	1.19×10^{-2}
$R_{fast, sec}$	15.87	9.96	10.75	4.28	4.04	6.35
$R_{slow, sec}$	7.20	6.04	14.07	18.32	10.02	14.41
$R_{ent, sec}$	1.01	2.79	4.96	1.15	2.30	4.90
ENT_{sec}	1.70×10^{-2}	2.68×10^{-4}	7.15×10^{-3}	2.98×10^{-2}	1.80×10^{-2}	1.00×10^{-4}
$k_{fast} [min^{-1}]$	1.48	1.94	1.21	2.00	1.78	1.92
$k_{slow} [min^{-1}]$	0.17	0.32	0.12	0.49	0.26	0.15
$\theta [t/g]$	-3.57×10^{-3}	-1.62×10^{-3}	-2.57×10^{-3}	-2.05×10^{-3}	-2.64×10^{-3}	-2.88×10^{-3}
$R_{slow \rightarrow cl} [0 g/t]$	22.42	27.31	24.03	44.32	25.20	30.19

5.4.2. Gangue model fit

The model fit, and the parameter estimates from the model regression are shown in Figure 79 and Table 22 respectively. The model fit charts show more variation than that observed for the PGMs, and this may be due to the gangue being made up of minerals with varying densities. The variation in mineral density influences the calculation of the entrainable material, which is the major contributor to gangue recovery. The gangue entrainment factors (for primary and secondary roughing) are less than that calculated for the chromite discussed later, refer to Table 23. This is due to the gangue minerals having a lower density (SG = 2.8-3.4) by comparison to the chromite (SG = 4.6). The lower density results in a lower entrainment throughout the size classes.

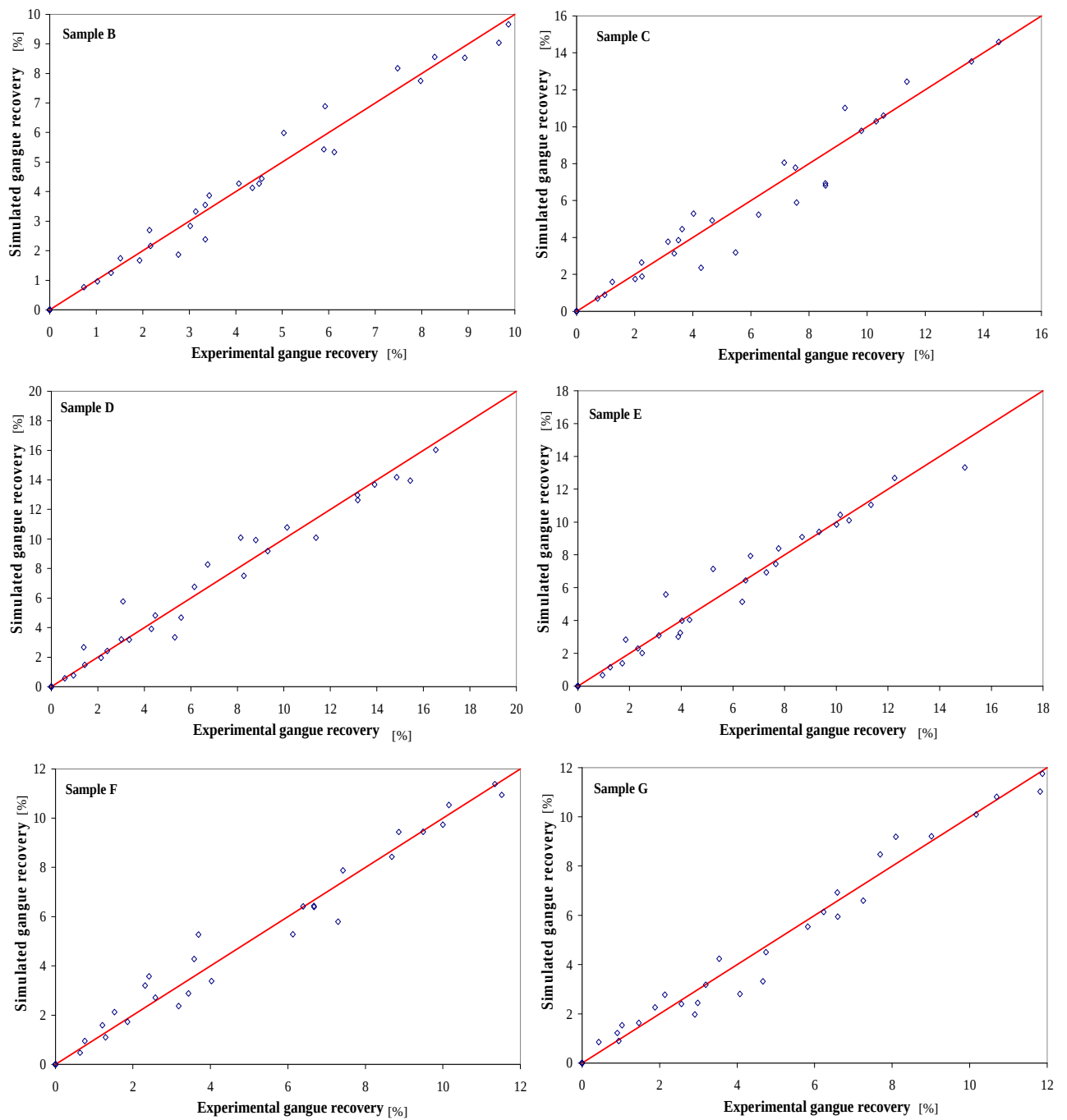


Figure 79: Gangue parity charts for other samples evaluated

Table 22: Gangue parameter estimates for other samples evaluated

Parameter	Sample B	Sample C	Sample D	Sample E	Sample F	Sample G
$R_{\text{float, prim}}$	4.56	6.21	10.14	7.45	7.42	7.69
$R_{\text{ent, prim}}$	17.43	69.86	45.35	46.45	37.05	50.09
ENT_{prim}	1.54×10^{-3}	1.41×10^{-3}	1.60×10^{-3}	1.11×10^{-3}	1.36×10^{-3}	1.53×10^{-3}
$R_{\text{float, sec}}$	5.91	5.48	6.74	6.09	4.59	4.44
$R_{\text{ent, sec}}$	65.78	86.95	83.74	70.65	88.73	53.54
ENT_{sec}	9.95×10^{-4}	2.29×10^{-3}	1.78×10^{-3}	1.28×10^{-3}	1.40×10^{-3}	2.04×10^{-3}
$k_{\text{float}} [\text{min}^{-1}]$	0.27	0.40	0.34	0.50	0.30	0.26
θ [t/g]	-1.89×10^{-2}	-9.75×10^{-3}	-6.48×10^{-3}	-8.64×10^{-3}	-2.79×10^{-2}	-1.29×10^{-2}
$R_{\text{float} \rightarrow \text{cl}}$ [0 g/t]	9.25	10.68	14.89	12.79	10.27	9.92

5.4.3. Chromite model fit

The model fit to the chromite data is shown in Figure 80, and the parameters obtained from the model are given in Table 23. There is a small amount of floatable chromite occurring in the roughers. Most of the chromite is recovered to the concentrate by entrainment, but there is a proportion of chromite which is non-recoverable owing to large particle size and its inability to attach to a bubble. There is more entrainable chromite occurring in the secondary rougher due to finer grinding, which increases the probability of entrainment of this mineral. The flotation rate for chromite is lower than that observed for the gangue and for the slow floating PGMs.

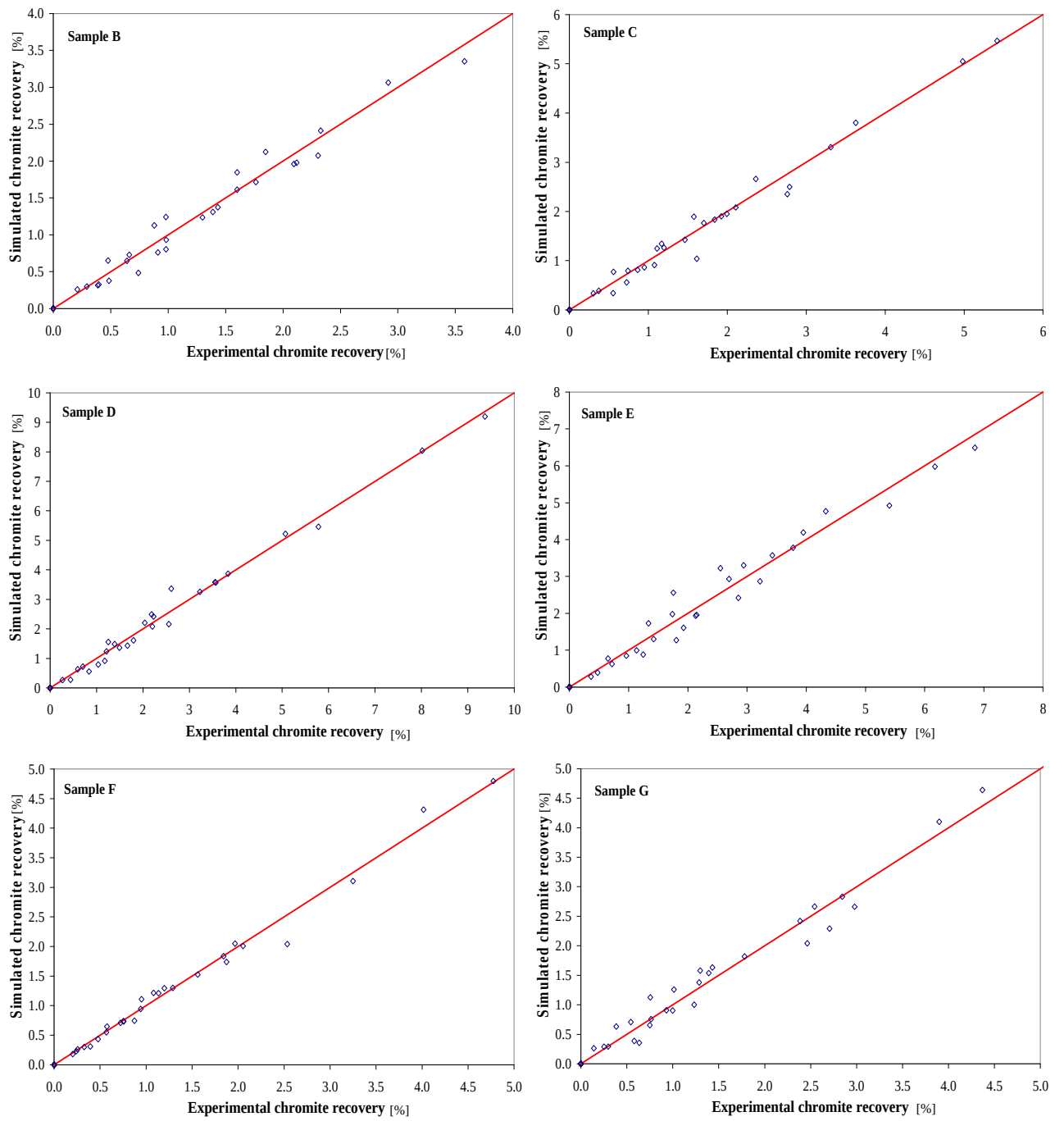


Figure 80: Chromite parity charts for other samples evaluated

Table 23: Chromite parameter estimates for other samples evaluated

Parameter	Sample B	Sample C	Sample D	Sample E	Sample F	Sample G
$R_{\text{float, prim}}$	0.86	1.01	1.92	2.70	0.91	1.39
$R_{\text{ent, prim}}$	11.23	10.48	5.82	14.16	17.66	14.19
ENT_{prim}	2.24×10^{-3}	2.21×10^{-3}	6.85×10^{-3}	1.66×10^{-3}	1.38×10^{-3}	1.74×10^{-3}
$R_{\text{float, sec}}$	1.85	1.82	3.18	2.34	1.29	2.01
$R_{\text{ent, sec}}$	21.24	27.01	53.91	40.22	40.94	35.86
ENT_{sec}	1.52×10^{-3}	3.01×10^{-3}	2.34×10^{-3}	1.92×10^{-3}	1.58×10^{-3}	3.15×10^{-3}
$k_{\text{float}} [\text{min}^{-1}]$	0.15	0.30	0.17	0.31	0.14	0.13
θ [t/g]	-5.06×10^{-3}	-4.71×10^{-3}	-5.50×10^{-3}	-8.29×10^{-3}	-1.13×10^{-2}	-2.19×10^{-3}
$R_{\text{float} \rightarrow \text{cl}} [0 \text{ g/t}]$	2.23	2.53	4.17	4.38	1.68	2.60

5.5. Conclusions

Batch rougher and cleaner data were modelled simultaneously for PGMs, gangue and chromite. The model considered the effect of grind variation in the rougher stage and depressant addition in the cleaner stage. The model provides a logical description of the flotation and entrainment of minerals in the batch tests. The following observations were made from the modelling of PGMs, gangue and chromite:

- The rate of flotation in increasing order is, chromite<gangue<PGMs (fast and slow)
- The effect of depressant on the species floatability was linked using the exponential parameter, θ . The larger the parameter, the greater the effect of depressant on the floatable mass fraction, i.e. less material is floatable with depressant addition. The parameter increased in the order, PGMs<chromite<gangue. This indicates gangue is more sensitive to the depressant than chromite and slow floating PGMs.

The flotation parameters estimated from this chapter will be used in the next chapter to determine if there is a link between the feed mineralogy and the batch flotation recovery of PGMs. Furthermore, the PGM, gangue and chromite parameters will be used in simulation exercises (Chapter 7).

CHAPTER 6. LINKING MINERALOGY TO PGM FLOTATION

6.1. Introduction

Batch flotation tests are used to determine the recovery attributes of the minerals in a sample. However, it does not provide details for the recovery variability. Attempts were made to relate the PGM recovery variability to feed chemical assay in Chapter 4 but the relationship was scattered. The feed chemical assay does not explain the mineral properties such as, associations with other minerals and the degree of liberation of minerals. Mineralogy, on the other hand, provides these key mineral properties. Mineralogy provides information on the PGMs in the samples such as, the mode of occurrence, which refers to the association of the PGMs with other minerals, like gangue and base metal sulphides.

The objective of this chapter is to develop a link between the floatable PGM fraction ($R_{\text{fast}} + R_{\text{slow}}$), obtained in the previous chapter, and the mineralogy of the PGMs in the feed. The model hypothesis for linking recovery with mineralogy will be discussed. The model will be validated and the application of the model in understanding PGM recovery/loss will be illustrated.

6.2. Model development

6.2.1. Model hypothesis

The mineralogy of PGMs in UG2 ore is more complex than the mineralogy of minerals in a massive sulphide ore. Massive sulphide ores have economic metals occurring in a limited range of minerals. For example, copper samples from Congo have copper minerals that occur mainly as chalcopyrite, chalcocite and malachite. Nickel samples from Botswana have nickel minerals that occur mainly as pentlandite or millerite. The UG2 ore on the other hand, has PGMs which occur in a broad spectrum of minerals, and which also have associations with other minerals such as, gangue and base metal sulphides. Hence, in Chapter 4 it was difficult to derive a qualitative correlation. Furthermore, typical factors such as, PGE grade and grind were not significant in the correlation, since the platinum group minerals have a complex mineralogy, which chemical assay cannot describe completely. In PGM mineralogy, four factors influence the floatability of a PGM and these are particle size, degree of liberation, speciation and mode of occurrence.

Particle size influences the probability of collision of a particle with an air bubble. Heindel and Bloom (1999) examined the exact and approximate relations for particle-bubble collision, and showed that smaller particles have a lower momentum, by comparison to larger particles. The lower momentum results in a lower probability of collision with an air bubble, which results in a lower recovery.

The degree of liberation refers to the proportion of a particle which is floatability and it is calculated using Equation 6.1. The floatable area is determined using the equivalent circular diameter of the PGM and BMS. The non-floatable area, which is generally gangue, is determined using the gangue equivalent circular diameter. The ratio of the floatable area to the entire particle area provides an index, called the degree of liberation, which varies between zero and unity. An index closer to zero indicates that the particle is mostly gangue and an index equal to zero indicates that the PGM is locked in gangue. For a liberation index approaching unity the particle is primarily floatable and an index equal to one indicates that the PGM (or host BMS) is fully liberated. The area data is obtained by viewing polished sections of particles mounted in epoxy. The calculation of the floatable and non-floatable area for three-dimensional particles requires mathematical transformations. The equivalent circular diameter maps the areas onto a Cartesian plane and ignores the depth occupied by the particle occurring outside the plane. Stereology is then used to infer 3-dimensional geometric features from a particle with a 2-dimensional measurement (Napier-Munn *et al.*, 2005). More advanced methods, which map the particle into a three-dimensional plane, are being examined by mineralogists, but to the author's knowledge, these methods are still being developed.

$$\text{Degree of liberation} = \frac{\text{Floatable Area}}{\text{Floatable Area} + \text{Non-floatable Area}} \quad (6.1)$$

The quantification of mineral associations for PGMs is particularly challenging, due to the low concentrations and small particle size of the PGMs. Figure 86 illustrates the PGM concentration relative to BMS and gangue content. A large number of samples mounted in epoxy are scanned automatically, using a scanning electron microscope (SEM). The PGM mineralogical analyses was carried out using a QEMSCAN that consisted of a Zeiss scanning electron microscope and Bruker energy dispersive X-ray spectrometer, which was controlled by custom hardware and software system developed by FEI Australia.

The mineralogy of the seven samples, which were tested extensively by flotation (Chapter 5), was determined at the final grind of 80% passing 75 μm , as this would determine ultimate recovery.

The procedure was as follows:

1. The UG2 sample was milled in a laboratory batch rod mill to 80% passing 75 μm . The mill product, which was dried overnight, was de-agglomerated using an 850 μm screen to break the lumps. The sample was homogenised so that a 200 g subsample could be taken for the analysis. The subsample was prepared into several polished sections for the analysis.
2. The QEMSCAN was used to automatically search for the PGM particles in the sample. However, due to the low grade and submicron PGM grain sizes; PGM particles are very difficult to locate without such an instrument.
3. The backscattered electron detector (BSD) on the SEM provided an input image to the system; in which a mineral's brightness was determined by its average atomic number. PGMs are distinguishable from gangue minerals in such an image, due to their high average atomic number. Once a possible PGM is detected, the EDS system is used to determine its chemical composition by comparing the analysis obtained to a previously established database of PGM compositions.
4. Several carefully prepared polished sections for each sample were automatically scanned overnight. By gathering information on a number of PGM particles in a sample, it was possible to report on the composition, size, and relationships of the PGM particles to other minerals.
5. The analysis was carried out by the mineralogical division at Mintek.

Speciation refers to the different platinum group minerals which occur in a sample. In a UG2 sample, a plethora of platinum group minerals is present and in many cases a formal identification of the PGM types is difficult. Voordouw *et al.* (2010) acknowledged the scarcity of many PGM types and proposed that replicable and statically significant differences can be greatly reduced by grouping the PGMs as indicated by Table 24. Furthermore, the authors discussed the geological impact on the generation of the minerals and indicated that some minerals originate under alteration conditions such as PGE tellurides, arsenides, ferroplatinum and alloys. These are referred to as secondary PGMs. Conversely, primary PGMs originate under normal conditions and have a good floatability. The alteration

minerals have poor floatability and Penberthy *et al.* (2000) found that the floatability decreases in the order, Braggite [Pt–Pd]S > Cooperite [PtS] > Malanite [Pt, Rh, Ir]₂CuS₄ > Ferroplatinum [Fe–Pt] and non-sulphide platinum species > Laurite. However, it stands to reason that if the PGMs with a poor floatability are associated with a larger base metal sulphide particle (such as chalcopyrite, pentlandite or pyrite), then the floatability of that particle is improved significantly. Hence, the mode of occurrence also needs to be considered.

Table 24: PGM groups, after Voordouw et al. (2010)

Group	Typical composition
Pt sulphide	Pt(±Ni)S
	PtPbS
	PtSnS
Pt–Pd sulphide	Pd(±Ni)S
Pt–Rh–Cu sulphide	PtRhCuS
Laurite	Ru(±Fe)S
Fe–Pt	FePt
	FeCuPt
PGE telluride	PdTe
	PdTeBi
	PtTeBi
	PtTe
	PdPtBiTe
PGE (S-)As	PtAs
	PtRhRuSAs
	PtSAs
	RhSAs
	Pd(±Ni)As
PGE alloys	PdPb
	PdAsSb
	PdSb
	PtPdSb
	PdHg
	PdSn
	PtSb
	PdTeHg
	PtSbAs
	PdTePb

The mode of occurrence refers to the associations of the PGMs with other minerals. There are six modes of occurrence for PGMs, and Figure 81 illustrates these modes:

- L refers to PGMs that are liberated.
- SL refers to PGMs that are associated with liberated base metal sulphides.
- AG refers to PGMs that are attached to silicate or oxide gangue particles.
- SAG refers to PGMs that are associated with base metal sulphides that are attached to silicate or oxide gangue particles.
- SG refers to PGMs that are associated with base metal sulphides that are enclosed in silicate or oxide gangue particles.
- G refers to PGMs that are enclosed in gangue particles.

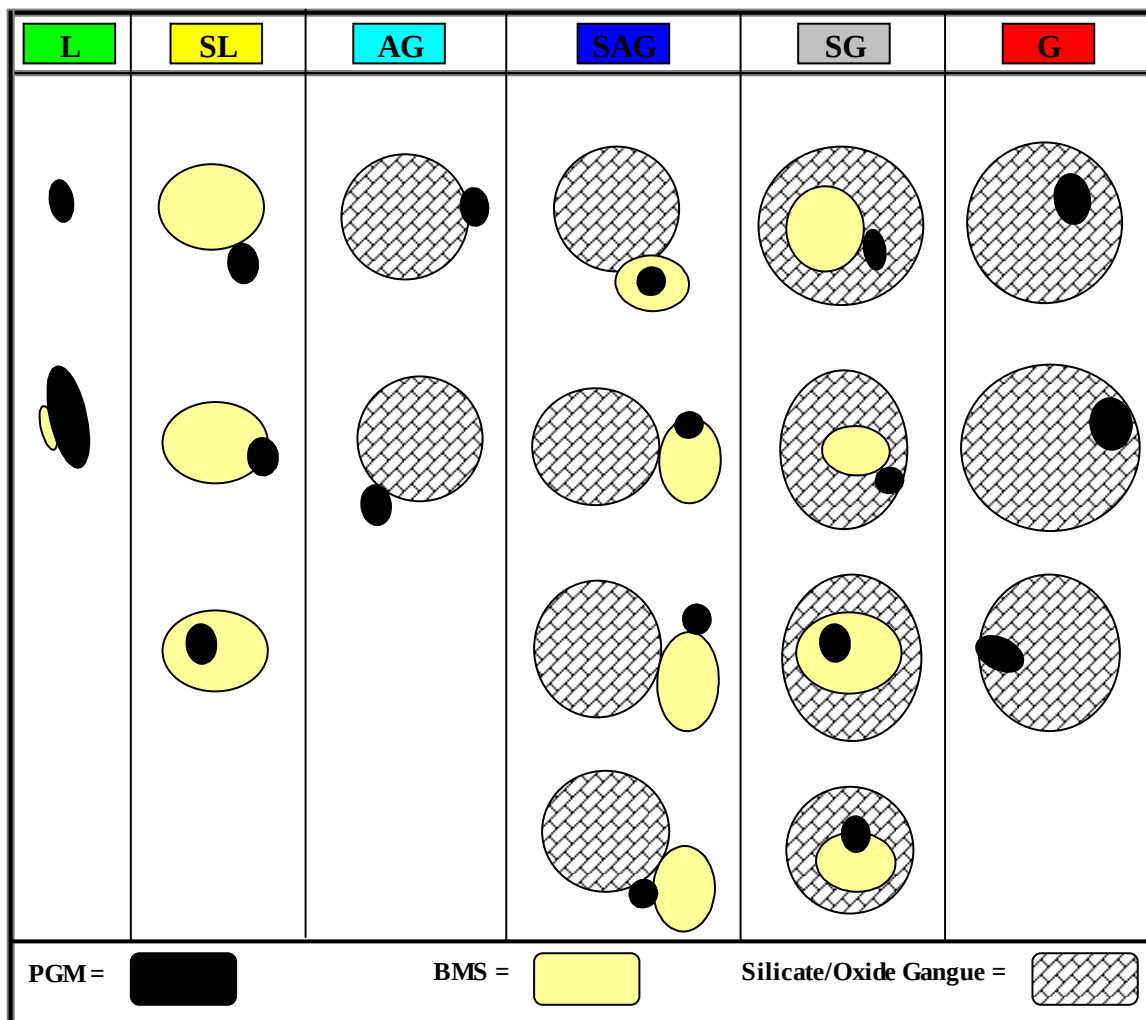


Figure 81: Mode of occurrence of PGMs

The development of a link between mineralogy and flotation of PGMs requires mineralogical characteristics such as, particle size, the degree of liberation, speciation and mode of occurrence. This approach requires a significant amount of mineralogy and flotation data, i.e. an analysis of the mineralogy of the feed, tails and concentrate would be required. For example, the feed would need to be apportioned into the different modes of occurrence, then for each mode of occurrence the PGM speciation needs to be considered. A simplification can be made by dividing the speciation into primary and secondary PGMs for each mode of occurrence- due to the large number of PGMs. The speciation for each mode would need to be further divided into classes for particle size and liberation. This type of modelling is called a nested model and requires knowledge of the various flotation products, to determine the proportion of each PGM in that class, which can be recovered. The economics of such a modelling approach is not practical.

The model can be simplified by considering combinations of the six modes of occurrence into three groups, as shown in Figure 82. The PGMs are grouped according to their degree of liberation as follows, liberated PGMs (L and SL), composite PGMs (AG and SAG) and non-recoverable PGMs (SG and G). The liberated PGMs (and host BMS) are defined as not being attached to any gangue mineral. Hence, composite particles have a reduced probability of attachment, and their recovery may depend upon the hydrophobicity of the adjacent gangue minerals. The recovery of PGMs in composite particles is expected to be affected by the addition of depressant. PGMs locked in gangue (SG and G) are non-recoverable. Therefore, these particles will not contribute to the recovery by flotation, but possibly to the recovery by entrainment.

Furthermore, only the PGM recovery by flotation (R_{fast} and R_{slow} from Chapter 5) will be considered because the recovery by entrainment cannot be linked to mineralogy in a logical way. Entrainment recovery is complicated by physical properties in the batch flotation test such as, particle size and froth stability, which in turn are affected by the concentration of frother and depressant, as shown in Chapter 5. These properties, together with the water rate of recovery, determine recovery by entrainment. On the other hand, the mode of occurrence will determine the probability of particle attachment to air bubbles and ultimately the recovery to the concentrate.

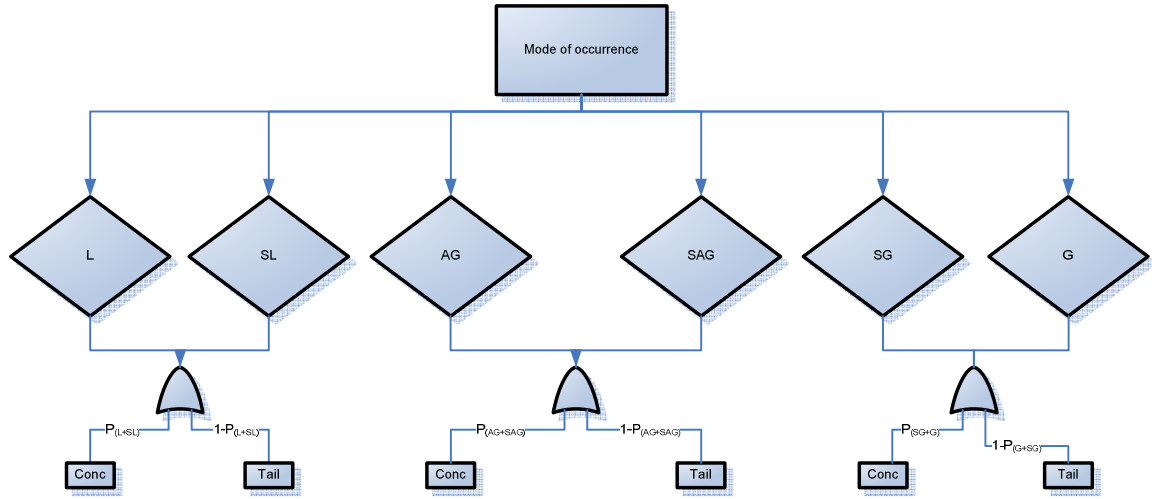


Figure 82: Simplified model for linking mineralogy to flotation

A simple linear model is proposed as a link between mineralogy and floatability; Equation 6.2 shows the model. The PGM feed mineralogy is determined as volume per cent apportioned to the various modes, this is standard industry practice. Therefore, the parameters in the model should vary between zero and one. It is assumed that the liberated particles are not influenced by depressant and should be fully recovered. Hence, parameter P_1 should be close to unity. This is logical since depressant is designed to act on silicate gangue minerals. Therefore, for liberated PGMs, which do not have a gangue component, there should not be any effect on the floatability in the pulp when depressant is added. However, depressant does change the froth structure, and smaller liberated PGMs may have difficulty in being recovered, due to brittle froth structures. Nevertheless, tests (both laboratory and plant) are not operated at brittle froth structures, i.e. high depressant dosages, which depressant floatable gangue that stabilises the froth. Hence, the assumption of having no depressant addition effects on the liberated PGMs is logical in the model hypothesis.

The composite particles, on the other hand, also have a fraction (P_2) which is recovered but this fraction is a function of depressant added. Intuitively at higher depressant dosages the recovery of the composite particles should be less, as shown in Chapter 5 for the slow floating PGM fraction. The floatable PGM recovery was determined in Chapter 5, and it is the sum of the fast (R_{fast}) and slow (R_{slow}) floating PGMs at different cleaner depressant dosages.

$$\begin{aligned}
R_{\text{float}} &= P_1 \times [L + SL] + P_2 (d_i) \times [AG + SAG] \\
R_{\text{float}} &= R_{\text{fast}} + R_{\text{slow}} (d_i)
\end{aligned}
\tag{6.2}$$

The parameters from the model P_1 and P_2 will be determined by regression to the floatable PGM recovery (obtained from batch flotation tests of Chapter 5) and the feed mineralogical data. This will be shown in the next section.

6.2.2. Model fit

Mineralogy was carried out on each feed sample milled to 80% passing 75 μm . The single-stage milling of the sample may result in some of the PGMs detaching from the BMS, but this approach was necessary in order to limit the cost of analysis. One of the samples, called sample G, was not included in the regression. This sample will be used to determine if the model hypothesis is valid.

Table 25 shows the data used for the regression. The parameters for flotation were extracted from data in Chapter 5. The mode of occurrence in ground feed samples is also shown. The mode of occurrence is generally determined in terms of the estimated per cent by volume (Napier-Munn *et al.*, 2005), instead of number or mass per cent. Mode of occurrence based on number per cent is clearly inappropriate, as it gives a disproportionate weight to small particles. Mode of occurrence based on mass per cent cannot be used for the PGMs because of the many species and the difficulty in identifying minerals using an electron microprobe. The proportions by volume can be estimated directly from the mineralogical data, irrespective of mineral density and is therefore, a better estimate of mode of occurrence.

It should be noted that the flotation data, which was extracted from Chapter 5, is based on the model for floatable PGMs, which was regressed to data from two stages of grinding and flotation. Recovery refers to the potential overall recovery by flotation (and not entrainment) at infinite time, and it is the sum of the fast and slow floating fractions. Hence, the ‘experimental’ recovery of the slow floating fraction is 100% when the depressant dosage is zero. It should also be noted that the depressant dosage shown in Table 25 refers to the depressant added to the cleaner, and it does not include the standard 20 g/t of depressant, which was added to both the rougher stages. This is consistent with the modelling of depressant addition in Chapter 5.

Table 25: Summary of data used for linking mineralogy to flotation

	Flotation		Mineralogy					
	Depressant [g/t]	Per cent recovery by flotation	L % v/v	SL % v/v	AG % v/v	SAG % v/v	SG % v/v	G % v/v
Sample A	0	92						
	75	88	48.04	17.84	10.40	15.81	0.00	7.91
	200	84						
	300	81						
Sample B	0	94						
	40	91	25.60	65.41	2.63	1.91	0.50	3.95
	60	90						
	80	89						
Sample C	0	93						
	100	89	38.01	13.79	21.20	21.33	0.00	5.67
	200	85						
	275	83						
Sample D	0	79						
	100	73	23.75	3.26	36.00	17.17	0.00	19.82
	350	65						
	500	61						
Sample E	0	86						
	75	80	35.20	3.15	31.00	15.81	0.38	14.46
	200	71						
	300	66						
Sample F	0	95						
	25	94	32.20	26.00	15.10	23.60	0.00	3.10
	50	92						
	100	89						

The mineralogical and flotation data were fitted to Equation 6.2 to determine the model parameters P_1 and $P_2(d_i)$. There were 24 data points and two model parameters, P_1 and P_2 . The initial regression was done by making no assumptions about the model for P_2 as a function of depressant addition. Values of P_2 at the selected depressant additions were used as parameters in the regression, giving a total of 14 parameters. Figure 83 shows the parity chart. The adjusted coefficient of determination indicates that 92% of the variation in recovery is accounted for by the mineralogy model (in blue). The prediction based on a larger number of chemical assays, developed in Chapter 4, is also shown for comparison (in green). It follows that the mineralogy-flotation model is better for predicting PGM recovery, because

it considers the liberation and mineral associations, which is the key to PGM recovery from a UG2 ore.

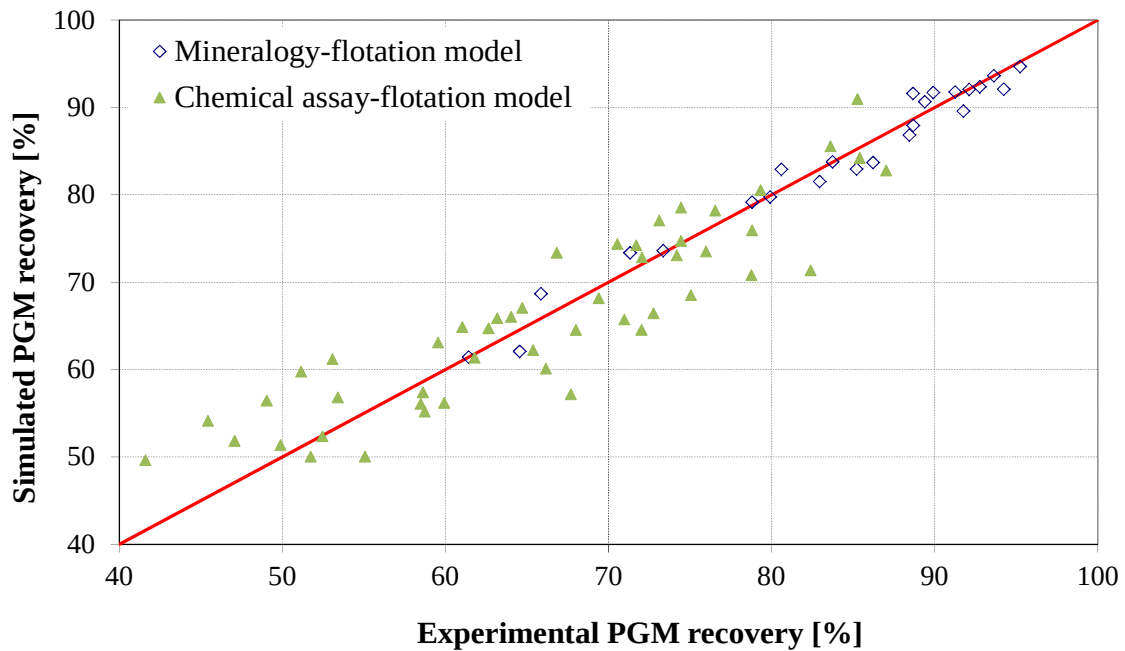


Figure 83: Parity chart for mineralogy-flotation model fit. The chemical assay-flotation model is shown for comparison

The parameter P_1 was found to be 0.96, which implies that 96% of the liberated PGMs are recovered by flotation. The remaining liberated particles may be too small to be recovered by flotation but may be recoverable by entrainment. The variation in parameter P_2 with depressant addition (d_i) is shown in Figure 84. Initial effect of depressant is strong, but the effect becomes less, with increasing depressant addition. The variation was correlated empirically with three equations. Other models such as polynomials of n -order can also be examined however; these models have various local maxima and minima, which is not a logical description of the data. Therefore, addition of depressant should reduce the composite PGM floatability to a point that further depressant addition does not affect floatability. The additional depressant cannot attach to a particle which already contains depressant. Hence, higher-order polynomials do not consider this, and will indicate improved floatability at higher depressant additions, which is intuitively impossible.

The objective was to have a simple equation with few parameters, and which describes the variation in a logical way. The linear and exponential models do not give a good fit to the data, as indicated by Figure 84 and Table 26.

A two fraction model, with a fraction affect by depressant and another fraction not affected by depressant, gave a better fit and description of the variation of P_2 with depressant addition. The model indicates that depressant does not affect a proportion of the composite particles. This is logical since this proportion may have a large liberation index (i.e. larger BMS and PGM area relative to the adjacent gangue). Hence, the recovery is not significantly affected by depressant addition. The model will be used to model the variation in P_2 with depressant addition. Therefore, the mathematically compact mineralogical-flotation model is:

$$R_{\text{float}} = P_1 [L + SL] + P_2(d_i) [AG + SAG] \quad (6.3)$$

$$P_1 = 0.96$$

$$P_2 = 0.57 + 0.43e^{\left(-3.43 \times 10^{-3} d_i\right)}$$

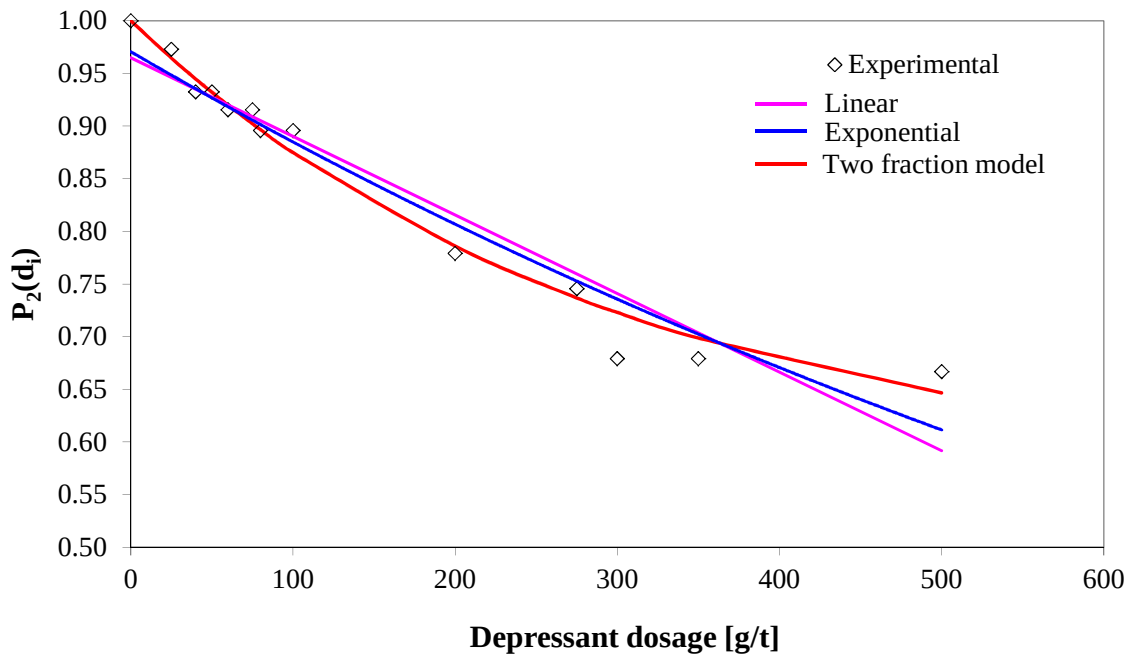


Figure 84: Variation in composite particle linear parameter (P_2) with depressant addition (d_i)

It is worth noting that grinding of the ore influences the liberation of PGMs. Therefore, finer grinding will liberate locked and composite PGM particles, but various factors will influence

the recovery of the liberated PGM. These factors include particle size and PGM type, i.e. secondary PGMs (altered minerals with poor floatability) and primary PGMs (normal unaltered minerals with good floatability).

Table 26: Summary of equations used to model effect of depressant addition on composite particle recovery

Equation type	Equation	Adjusted coefficient of determination
Linear	$P(d_i) = -7.46 \times 10^{-4} (d_i) + 9.65 \times 10^{-1}$	0.92
Exponential	$P(d_i) = 0.97e^{(-9.24 \times 10^{-4} d_i)}$	0.93
Two fraction model	$P(d_i) = 0.57 + 0.43e^{(-3.43 \times 10^{-3} d_i)}$	0.96

Figure 85 shows the model fit for Equation 6.3, the model fit with no depressant equation, and the chemical assay-flotation model. The application of the two fraction model (with one exponential effect), for variation in parameters with depressant, does not result in a significant loss in fit. The approach gives a compact mineralogical-flotation model, which can be used to predict flotation recovery given feed mineralogical properties. The two fraction model was derived in the depressant addition range 0 to 500 g/t. To the author's knowledge the upper dosage of 500 g/t is the highest dosage applied to a UG2 ore. Due to the stability in the two fraction model, PGM recovery for dosages above 500 g/t can be predicted.

Given the flotation recovery at two depressant additions the liberated and composite PGM content can be estimated. The PGMs locked in gangue can be determined by difference, i.e. 100-liberated PGMs-composite PGMs. This relationship is essential for understanding the factors constraining PGM recovery. However, the model will be evaluated in the next section using sample G, which was omitted from the regression. The objective will be to determine if the model proposed for linking mineralogy to flotation is valid.

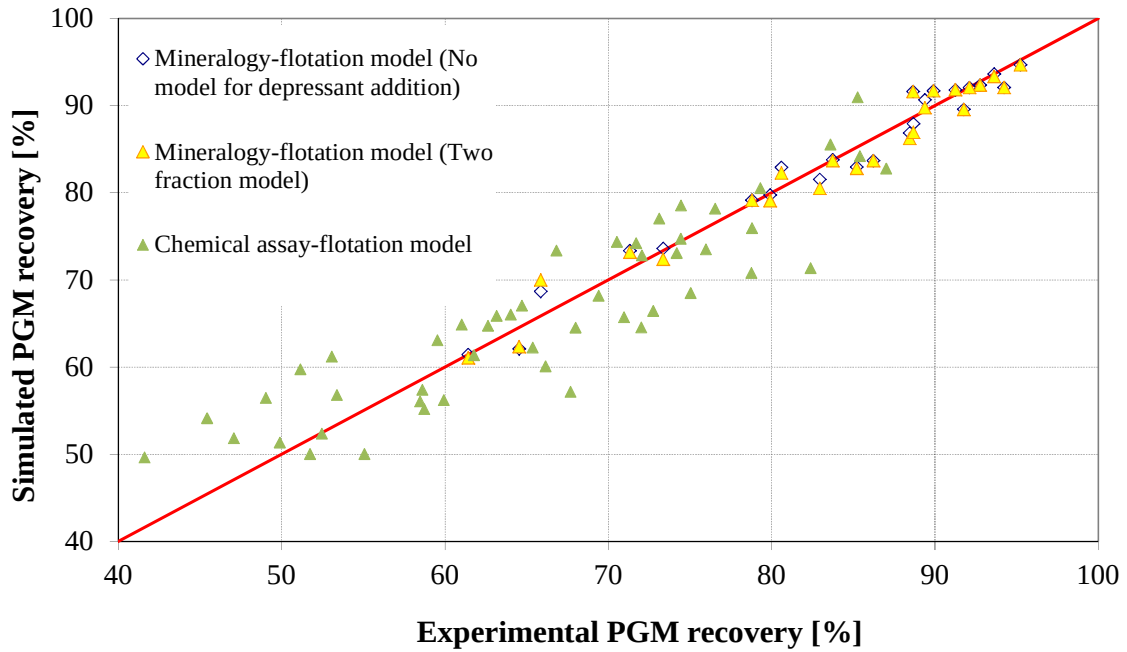


Figure 85: Parity chart for mineralogy-flotation model fit using an empirical equation for depressant addition

6.2.3. Model validation

Table 27 shows the mineralogical data for sample G; this data was used to predict the floatable PGM recovery using the model developed earlier i.e. Equation 6.3. Table 28 presents a comparison between floatable PGM recovery, obtained from batch cleaner flotation tests, carried out in Chapter 5, and the floatable recovery predicted from feed mineralogy. The results compare well although, the predicted recovery is marginally but consistently lower than the recovery from the batch flotation tests. This was also observed in the regression results. It is believed that the single-stage milling of the sample for mineralogy resulted in overgrinding of some of the PGMs hence, a smaller grain size, which is more difficult to recover by flotation. Therefore, the recovery is marginally lower. Nevertheless, the model proposed for linking mineralogy to flotation is simple and gives a reasonable prediction of the floatable PGM recovery, despite the complex mineralogy of the UG2 ore. Applications of this model will be discussed in the next section.

Table 27: Summary of feed mineralogy for sample G

Mode of occurrence	Volume Per cent
L	21.3
SL	3.0
AG	40.7
SAG	24.0
SG	0.4
G	10.6

Table 28: Comparison of flotation recovery from batch flotation tests on sample G and the recovery predicted using mineralogy

Depressant	Flotation recovery	Predicted recovery
0	89.95	89.00
80	83.75	82.32
140	79.96	78.39
200	76.76	75.19

6.3. Applications of model

The mineralogy-flotation model can be used to examine factors influencing the loss in recovery. This is illustrated by way of an example using sample D from Table 25. Figure 86 shows the mineralogy false colour map of the PGM particles in general, this sample has most of the PGMs locked in gangue or associated with gangue (composite particles).

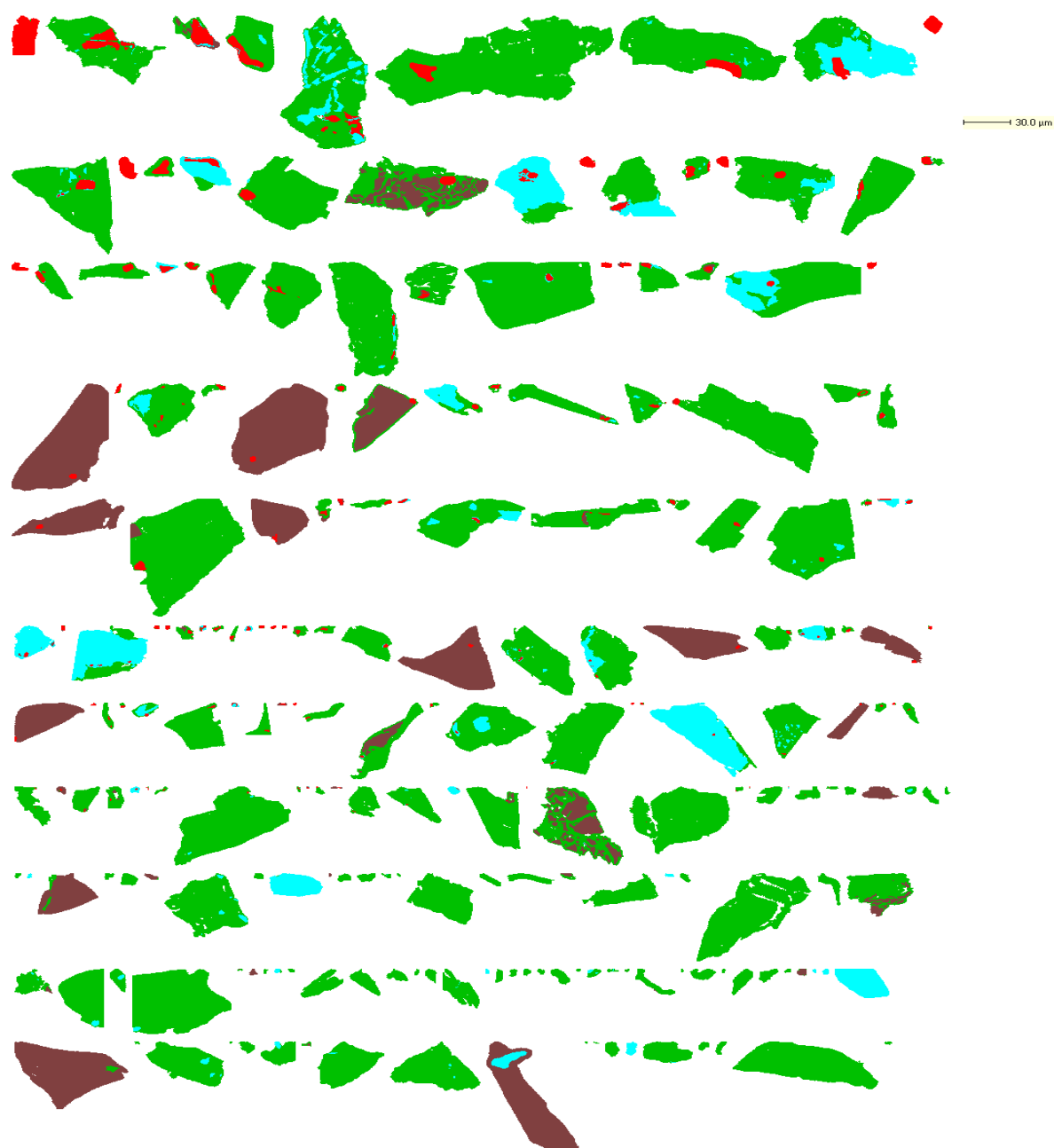
Table 29 presents a summary of the analysis. Using Equation 6.3 at a depressant dosage of 100 g/t in the cleaner enables determination of the floatable recovery. The loss in recovery (to both the rougher and cleaner tail) is determined by difference between the feed mode of occurrence mineralogy, and the recovery predicted for each mode using Equation 6.3. The normalised loss in recovery was determined, as this provides a better assessment of the loss, which is mainly due to locked PGMs and PGMs associated with gangue. Mineralogy was carried out on the combined rougher and cleaner tail, for the batch test carried out in Chapter 5, to determine the nature of the losses and to check against the model prediction. The results are presented in Table 29. The predicted values do not match the experimental values exactly, but illustrates the same logic, that the loss is mainly due to locked PGMs and composites.

It must be noted that the tailings mineralogy is extremely difficult to analyse because there are fewer PGMs particles in tailing samples. Several mineralogical sections of the tails were examined to improve the reliability of the measurements. Nevertheless, the mineralogical-flotation model proposed gives a meaningful description of the recovery variability and the nature of the PGM losses.

Table 29: Loss of PGMs to tail for Sample D

Mode of occurrence	Feed/ [%]	Predicted recovery at 100 g/t/ [%]	Predicted loss ^r / [%]	Normalised predicted loss to tail/ [%]	Tails mineralogy/ [%]
L	23.75	22.80	0.95	3.45	6.3
SL	3.26	3.13	0.13	0.47	0.0
AG	36.00	31.51	4.49	16.32	21.2
SAG	17.17	15.03	2.14	7.78	8.2
SG	0.00	0.00	0.00	0.00	1.5
G	19.82	0.00	19.82	71.97	62.8
	100.00	72.46	27.54	100	100

^rPredicted loss determined by difference between Feed (column 2) and predicted recovery (column 3)



False colour map key	
	Background
	PGM
	BMS
	Oxide gangue
	Silicate gangue

Figure 86: False colour map for Sample D feed

6.4. Conclusions

A model was developed, which links mineralogy to recovery by flotation. The model enables predictions of floatable PGM recovery when using a liberation analysis of the feed at the final grind of 80% passing 75 μm , which is a typical grind for UG2 ore. The model was validated using independent data, and it was concluded that the accuracy of the prediction was adequate. Application of the model was also examined, and it was shown that the mineralogical factors influencing PGM loss in flotation can be determined by the model.

It was noted that finer grinding of a UG2 ore will result in improved liberation, but the recovery of the liberated particles is influenced by other factors. These factors include particle size and mineral type, i.e. primary PGMs (which have good floatability) and secondary PGMs, which are altered and have poor floatability (Penberthy, 2001). Nevertheless, the complexities in UG2 ore mineralogy were related to PGM recovery by flotation, using a simple mathematical expression, which considered PGM mode of occurrence. The effect of depressant on mineralogical characteristics of the feed and its relation to PGM floatability was included in the model. This is a significant finding, with respect to plant operation.

CHAPTER 7. EVALUATION OF PLANT PERFORMANCE USING BATCH DERIVED FLOTATION PARAMETERS

7.1. Introduction

The simulation of flotation circuit performance using batch flotation data has been examined by many authors. The key flotation simulators in the author's opinion are, JKSimFloat (Runge *et al.*, 1998; Alexander *et al.*, 2000; Harris *et al.*, 2002 and Schwarz and Kilgariff, 2005), SUPASIM (Hay and Rule, 2003; Hay and Schroeder, 2005; Hay, 2005 and Hay and Roy, 2010), Aminproc (Amelunxen and Amelunxen, 2009a, b, c) and MinnovEx (Dobby and Savassi, 2005). These flotation simulators were discussed in Chapter 2. The simulators make use of batch flotation tests together with plant measurements to determine plant performance. In the event that plant data is not available, suitable historical parameters are used from a database. The models used in these simulators use a simple correction factor (i.e. froth factor) to link plant performance to batch tests. These factors are empirical and may differ for each mineral. If this is the case, the batch data is effectively force-fitted to the plant data, but the parameters derived from batch tests, nevertheless, provide a reasonable basis for extrapolation of plant performance.

Simulation provides an opportunity for case studies and automatic optimisation of plant operating conditions, particularly the flow of froth from cleaners and re-cleaners. The automatic search for optimum conditions can be constrained to fix conditions such as, mass of final concentrate and/or concentration of impurities. Practical limits on operating conditions, such as froth flow per launder length can also be imposed. A simple spread-sheet flotation simulator is proposed here for evaluating plant performance. The simulator is not as advanced as the approaches mentioned above, but it provides individuals who do not have access to these simulators with a means of evaluating plant performance, given batch tests and plant data. The simulator methodology and application will be illustrated by way of an example.

The evaluation of plant performance will be shown in two steps in this chapter. The first step is to configure the spread-sheet, so that it matches plant configuration. The batch rougher tests described in Chapter 5 were used to calculate the rates of flotation and entrainment, and the associated mass fractions. The batch parameters are inserted into the simulator, and the simulator is regressed to plant stream data to determine the froth parameters. The froth

parameters link the batch data to the plant and are essentially ‘scale-up’ factors. The next step in evaluating plant performance, involved carrying out optimisation studies using the calibrated plant model. Various plant design and operating conditions can be simulated. It was decided that the distribution of cell volume in the plant and the use of depressant in the cleaners would be of interest for UG2 flotation. For volume distribution, incremental changes were made for one flotation bank at a time and the final PGM recovery was optimised in the simulator, by varying the froth residence time (and hence the ‘pull’ rate) for each flotation bank. The froth residence time can be varied for optimisation studies, since the froth model parameters were obtained during the calibration exercise. The final concentrate was constrained to typical industrial values, i.e. 1.0% for mass recovery and 2.5% for chromite grade. Plots for the effect of volume distribution on final PGM recovery was generated for each flotation bank.

Depressant addition to the cleaner stage was also investigated. Incremental changes were made for depressant addition to the cleaner stage, and the PGM recovery was maximised, by varying the froth residence time for each bank. The plant volume distribution was fixed for this exercise. The final concentrate was constrained as discussed above for the volume distribution simulations.

The simulations produced relative optimums and not global. The EXCEL® search procedure uses gradients and a constrained search will depend upon the starting point, particularly when there are a large number of variables. Interaction between variables is a common phenomenon. Nevertheless, the simulations for each operating variable provided a means for evaluating the flotation plant sensitivity with respect to volume and depressant for a UG2 feed.

7.2. Simulator algorithm

7.2.1. Modelling of bank recovery

Finch and Dobby (1990) showed that the recovery from a flotation bank having n mechanical cells with the same pulp residence time per cell can be modelled according to Figure 87. In practice, the pulp residence time per cell increases down the flotation bank and other factors come into play, such as changes in froth stability. A good model of froth flow as a function of froth depth is vital for the successful application of parameters obtained from hand-scraped

batch flotation tests. A simple model was selected, in which the froth flow is specified per flotation bank, (not per cell), to reduce the number of variables in the simulator.

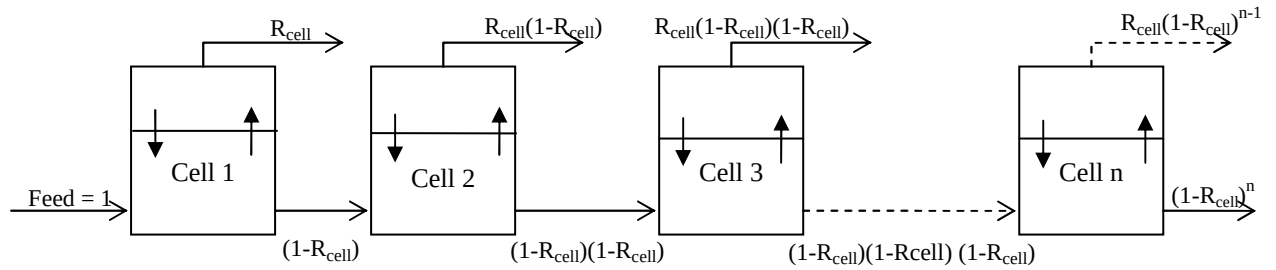


Figure 87: Conceptual configuration of the pulp and froth phase for a bank having n cells

The recovery of floatable or entrainable material from a bank (R_{bank}) having n cells can be modelled using the equation:

$$R_{bank} = 1 - (1 - R_{cell})^n \quad (7.1)$$

R_{cell} refers to the recovery by flotation or entrainment from each cell. The floatable recovery per cell ($R_{float,cell}$) was modelled using the model proposed by Finch and Dobby (1990):

$$R_{float,cell} = \frac{R_{f,cell} R_{c,cell}}{1 - R_{c,cell} + R_{c,cell} R_{f,cell}} \quad (7.2)$$

$R_{f,cell}$ and $R_{c,cell}$ is the froth and collection/pulp zone recovery per cell respectively. The recovery by entrainment per cell ($R_{ent,cell}$) was modelled using the conventional model, but froth recovery was included. A high froth recovery implies a shallow froth therefore, the recovery by entrainment should be greater and the opposite applies for low froth recovery.

$$R_{ent,cell} = ENT \times R_{w,cell} \times R_{f,cell} \quad (7.3)$$

ENT is the entrainment factor and $R_{w,cell}$ is the water recovery per cell, $R_{f,cell}$ is the froth recovery per cell, which is analogous to Equation 7.2.

The model for the froth proposed by Mathe *et al* (2000) was used, because it links froth recovery to froth residence time and hence to froth depth. The froth levels in the pilot-plant test, which will be used to calibrate the simulator, were automatically controlled. Froth recovery is expressed in terms of froth residence time and two empirical parameters η and β . The models for each species (PGMs, gangue and chromite) will be discussed next.

7.2.1.1. PGMs

A recovery model, which assumes two floatable fractions (fast and slow), together with entrainment, was found in Chapter 3 to be sufficient for modelling PGM recovery. The model was also extended in Chapter 5 for modelling stage grinding and depressant addition. Hence, it will be used in plant simulations as well.

a. Floatable PGMs

The recovery of fast and slow floating PGMs for each cell was modelled using Equation 7.4 and 7.6. The recovery from the collection/pulp zone was modelled using the equation for a continuous stirred tank reactor. The rates of flotation, k_{fast} and k_{slow} are used to characterise PGM flotation. The average residence time was used to determine the recovery from a cell in the flotation bank hence, the overall flotation bank residence time (τ) is divided by the number of cells (n) in the flotation bank. The overall flotation bank residence time was calculated using the flotation bank volume (V_{bank}) and the linear average of the feed and tail slurry volumetric flow rate ($\dot{s}_{\text{feed}}$ and $\dot{s}_{\text{tail}}$). The froth was modelled using Equation 7.9, which was proposed by Mathe *et al.* (2000) and used by Harris (2000) in UG2 plant simulations. The parameters η and β are the average froth non-drainage fraction and froth stability term for the entire plant respectively. Harris (2000) used different froth parameters for cleaning and roughing stages because of the difference in froth loading. However, in this study, a single set of froth parameters is proposed for each species in the circuit, and these are called scale-up parameters. A new set can be calculated for cleaning and roughing and this will most probably improve the fit, but due to parameter interaction, a reduced set of parameters is recommended. Since this will have a better statistical significance.

The average froth residence time (FRT) for a flotation bank was calculated according to Equation 7.10; A_{bank} is the cross-sectional area of the cells in the flotation bank; FH is the average froth height, and Q_{air} is the average volumetric air flow rate.

$$R_{\text{fast,cell}} = \frac{R_{\text{f,cell}} R_{\text{c,fast,cell}}}{1 - R_{\text{c,fast,cell}} + R_{\text{c,fast,cell}} R_{\text{f,cell}}} \quad (7.4)$$

$$R_{\text{c,fast,cell}} = \left[1 - \frac{1}{1 + k_{\text{fast}} \frac{\tau}{n}} \right] \quad (7.5)$$

$$R_{\text{slow,cell}} = \frac{R_{\text{f,cell}} R_{\text{c,slow,cell}}}{1 - R_{\text{c,slow,cell}} + R_{\text{c,slow,cell}} R_{\text{f,cell}}} \quad (7.6)$$

$$R_{\text{c,slow,cell}} = \left[1 - \frac{1}{1 + k_{\text{slow}} \frac{\tau}{n}} \right] \quad (7.7)$$

$$\tau = \frac{V_{\text{bank}}}{\left[\frac{\dot{S}_{\text{feed}} + \dot{S}_{\text{tail}}}{2} \right]} \quad (7.8)$$

$$R_{\text{f,cell}} = (1 - \eta) e^{-\beta \times \text{FRT} + \eta} \quad (7.9)$$

$$\text{FRT} = \frac{A_{\text{cell}} \text{FH}}{Q_{\text{air}}} \quad (7.10)$$

It is worth noting that if the froth recovery was assumed to be 100%, the recovery from a cell is just the recovery from the collection zone (Equation 7.5 and 7.7). Substitution of the collection zone model into the flotation bank recovery model, given by Equation 7.1, gives:

$$R_{\text{bank}} = 1 - \left[\frac{1}{1 + k \frac{\tau}{n}} \right]^n \quad (7.11)$$

Equation 7.11 is the traditional flotation bank recovery model used by Loveday and Marchant (1972) in simulating flotation circuits.

b. Entrainable PGMs

The recovery by entrainment for a cell was determined using Equation 7.12. The average flotation bank water recovery (R_w/n) is used to determine the entrainment recovery for each cell. The froth recovery is the same as that determined for the fast and slow floating PGMs.

$$R_{ent,cell} = \left(ENT \times \frac{R_w}{n} \times R_{f,cell} \right) \quad (7.12)$$

The overall PGM recovery from a flotation bank ($R_{o,bank}$) is given by:

$$R_{o,bank} = m_{fast} \left[1 - (1 - R_{fast,cell})^n \right] + m_{slow} \left[1 - (1 - R_{slow,cell})^n \right] + \left\{ m_{fast} \left[(1 - R_{fast,cell})^n \right] + m_{slow} \left[(1 - R_{slow,cell})^n \right] + m_{ent} \right\} \left[1 - (1 - R_{ent,cell})^n \right] \quad (7.13)$$

The fast and slow floating PGM recovery is multiplied by the fast and slow mass fractions (m_{fast} and m_{slow}) in the feed to each flotation bank. This gives the floatable recovery to the concentrate of the flotation bank. The recovery of PGMs by entrainment was determined using the entrainment recovery for a flotation bank, which is multiplied by the entrainable material, which consists of PGMs not recovered by flotation and locked PGMs which are entrainable (m_{ent}). The recovery of PGMs by flotation and entrainment is a simultaneous process therefore; the assumption of entrainment occurring for PGMs, which are not floated, is not strictly true. However, this assumption is necessary in order to facilitate simulation, and is used by JKSimFloat in carrying out simulations (Runge *et al.*, 1998).

7.2.1.2. Gangue and chromite

Gangue and chromite are modelled in the same way as the PGMs, but with a single floatable fraction and entrainment. This modelling approach was found to be suitable for both species in Chapter 3. The overall flotation bank recovery for each species is:

$$R_{o,bank} = m_{float} \left[1 - (1 - R_{float,cell})^n \right] + \left\{ m_{float} (1 - R_{float,cell})^n + m_{ent} \right\} \left[1 - (1 - R_{ent,cell})^n \right] \quad (7.14)$$

m_{float} is the floatable mass fraction in the feed to the flotation bank and $R_{float,cell}$ and $R_{ent,cell}$ is the floatable and entrainable recovery per cell respectively. The floatable and entrainment recovery was modelled in the same way as the PGMs.

7.2.2. Executing simulation

The simulation of a flotation circuit was carried out by writing the equations in mass balance form for each flotation bank in the circuit. However, the recovery models had to be

populated, and this was done by using batch derived parameters determined in Chapter 5 and regression to smoothed plant data for the same ore. The batch parameters from Chapter 5 are the rate constants (k), entrainment factor ENT, the primary and secondary rougher floatable and entrainable mass fractions. The smoothed plant data was used to determine the froth recovery model parameters, i.e. η and β for each species. Depressant was also modelled in the cleaner stage using Equation 5.2 presented in Chapter 5, which relates the floatable mass fraction for each species (only slow for PGMs) to depressant addition.

However, the simulation cannot proceed until suitable concentrate solids concentration for each flotation bank is assumed. This assumption is also used in JKSimFloat (Runge *et al.*, 1998). A review of plant data makes it possible to select a suitable concentration of solids in the concentrate, for estimating the flow of water in the concentrate. Hence, a water balance can be generated for the simulation. Table 30 presents average pilot plant solids concentration for UG2 ores processed (Harris, 2001a; Harris, 2001b; Thiele and Sibanyoni, 2003; Bowers and Maharaj, 2004; Maharaj *et al.*, 2004, Maharaj *et al.*, 2007 and Mabaso and Bryson, 2009).

The determination of the froth parameters (η and β) is iterative. The parameters are varied in the calibration, and this gives a froth recovery which calculates the mass flow rate (sum of all species mass flows). The mass flow rate is used to calculate the water flow for each concentrate, which is then used to determine the water flow rate across a flotation bank. The calibration continues until suitable froth parameters are determined, and which minimises the difference between the smoothed experimental data and the simulation output.

The froth parameters for each species can be used to carry out simulation exercises, which vary the froth residence time for each bank subject to some constraint. Variation of volume in each bank and effect of depressant addition to cleaner was investigated for a fixed final concentrate mass recovery and chromite grade, which are 1.0% and 2.5% respectively. These values are industrial constraints and both influence smelter operations, which require a low mass of concentrate and a low chromite grade.

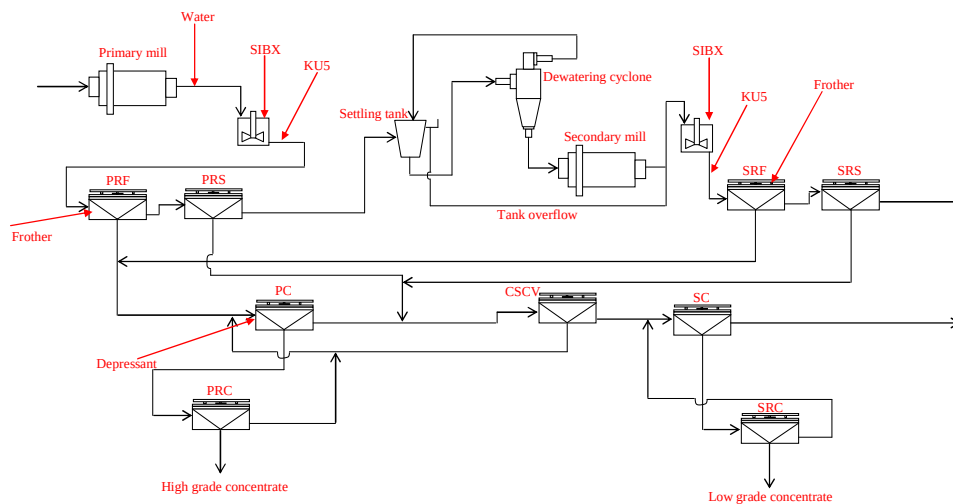
Table 30: Average solids concentration in concentrate, obtained from Mintek reports for UG2 pilot plant campaigns

Stage	Percentage solids concentration
Roughers	10
Scavenger	10
Cleaners	15
Recleaner	25

7.3. Model calibration

Figure 89 shows the circuit which will be used to illustrate the calibration of the flotation model discussed above and for investigation of volume changes on PGM recovery. The UG2 ore processed by this circuit was referred to as sample A in Chapter 5. The sample was taken during a pilot plant campaign carried out at Mintek (Mabaso and Bryson, 2009) hence, the pilot plant data is from that study. The circuit is a typical UG2 flotation circuit having stage grinding with depressant added to the primary cleaner stage. The concentrates from the rougher stage are matched such that fast and slow floating PGMs are treated in separate cleaning circuits. Collector, frother and depressant are also added into the rougher stage, and this was the same as the dosages used in the batch test work presented in Chapter 5.

The simulator was calibrated according to procedure illustrated by Figure 89. The species flow rates from the plant were smoothed to provide balanced data; this was carried out using a spread-sheet, and the mass smoothing technique called the matrix method, which was applied by Harris (2000) in UG2 plant optimisation studies. The smoothed data together with the batch parameters obtained from Chapter 5 for sample A were used to calibrate the model. The calibration was achieved by fitting the plant model with batch parameters to the smoothed plant data, such that the froth parameters (η and β) for each species was varied until the difference in the smoothed plant and simulated data was minimised. The froth residence time for each flotation bank provided additional data for the calibration. The froth model essentially ‘scales-up’ the batch parameters and provides a link between flotation batch tests and the plant. This is analogous to the approach used in SUPASIM (Hay and Rule, 2003). The SUPASIM simulator is more mature and has a database of froth scale-up factors between batch flotation and plant data.



Key	
PRF	Primary rougher fast
PRS	Primary rougher slow
SRF	Secondary rougher fast
SRS	Secondary rougher slow
PC	Primary cleaner
PRC	Primary recleaner
CSCV	Cleaner scavenger
SC	Secondary cleaner
SRC	Secondary recleaner

Figure 88: Flow sheet used in simulation exercise

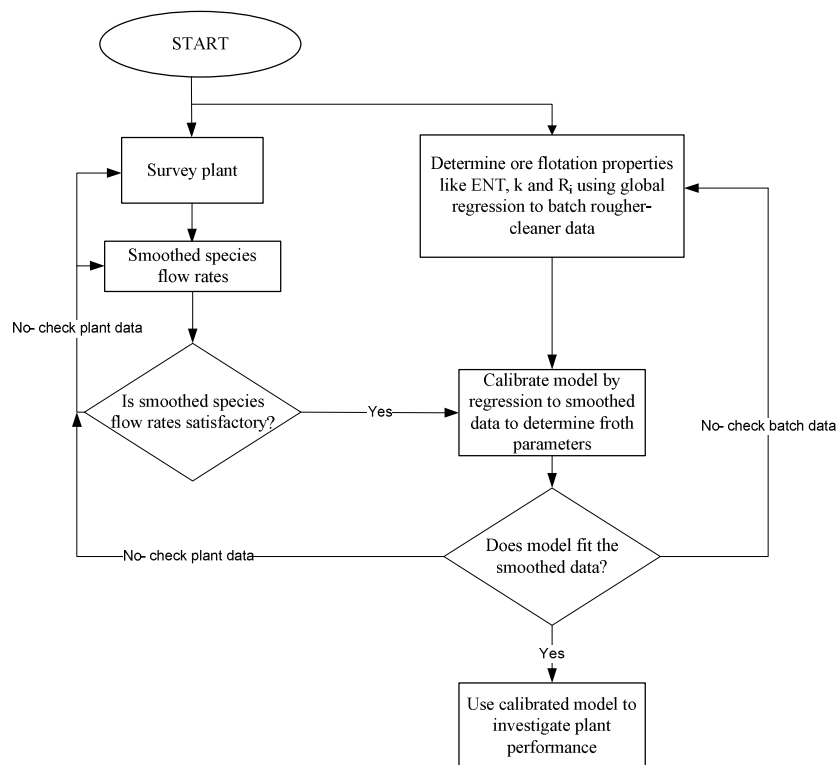


Figure 89: Approach for calibration of flotation simulator

Figure 90 shows the parity chart which compares the smoothed species distribution with the experimental (plant) species distribution in the mass smoothing exercise. The species distribution was calculated relative to the plant feed, i.e. to the primary mill in Figure 88. Hence, all the species can be observed on the same figure. Significant deviation between the data is circled in the figure; additional stream data for this plant configuration was not available; therefore, the deviation could not be examined further to determine if it was a sampling or assay error. Nevertheless, the remaining 63 data points compared well with the plant data and showed a small deviation from the parity line; therefore, the plant was stable and can be used for calibration to determine the species froth parameters.

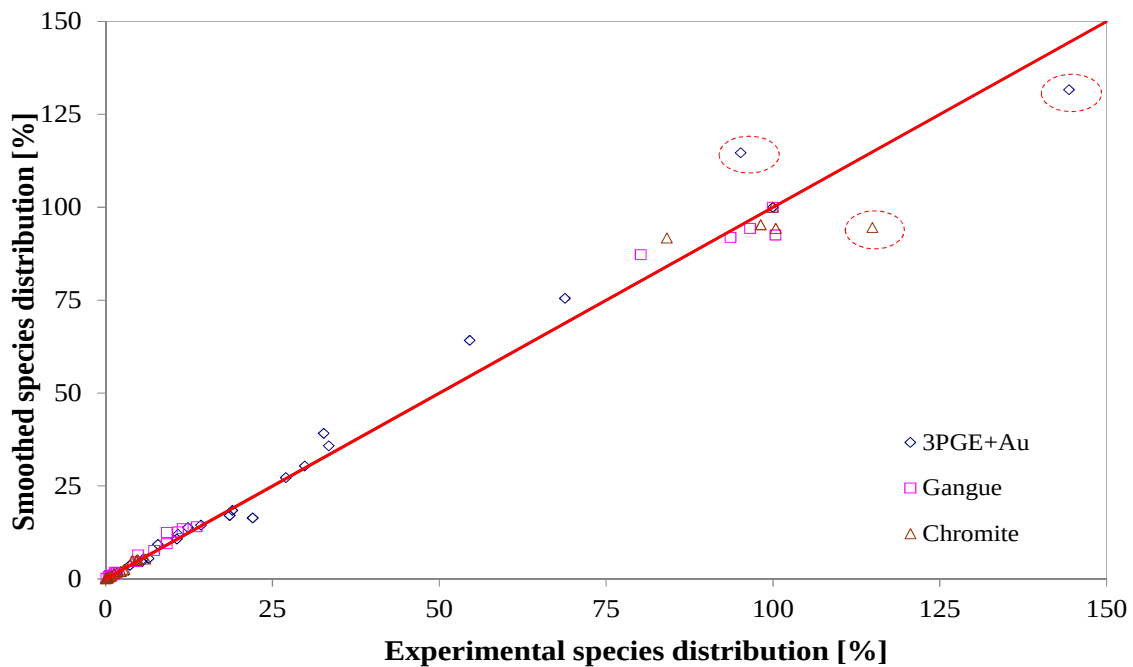


Figure 90: Parity chart for experimental and smoothed species distribution, which is relative to plant feed flow rate

Figure 91 shows the parity chart which compares the simulated species distribution (from model calibration) for each stream with the smoothed species distribution. The simulated data compares well with the smoothed data, however; there is significant deviation for the PGM data which is circled in the figure. This data corresponds to the primary cleaner (PC) flotation bank feed and concentrate. The two points are the PGM data which had a poor fit in the mass smoothing exercise given by Figure 90. Therefore, it may be a sampling problem that resulted in the data being anomalous. The other data points were modelled sufficiently by the simulator since their deviation from the smooth species distribution was small.

Table 31 shows the froth parameters obtained from the calibration of the simulator. The parameters have a physical meaning; β refers to froth stability and η refers to the non-draining material fraction in the froth. The froth model is an exponential function therefore; higher values for β (chromite and gangue) imply that the froth is more selective towards these species than the PGMs. The non-draining parameter is lower for gangue and chromite, which is logical since these species detach in the froth more readily than the PGMs resulting in better PGM upgrading.

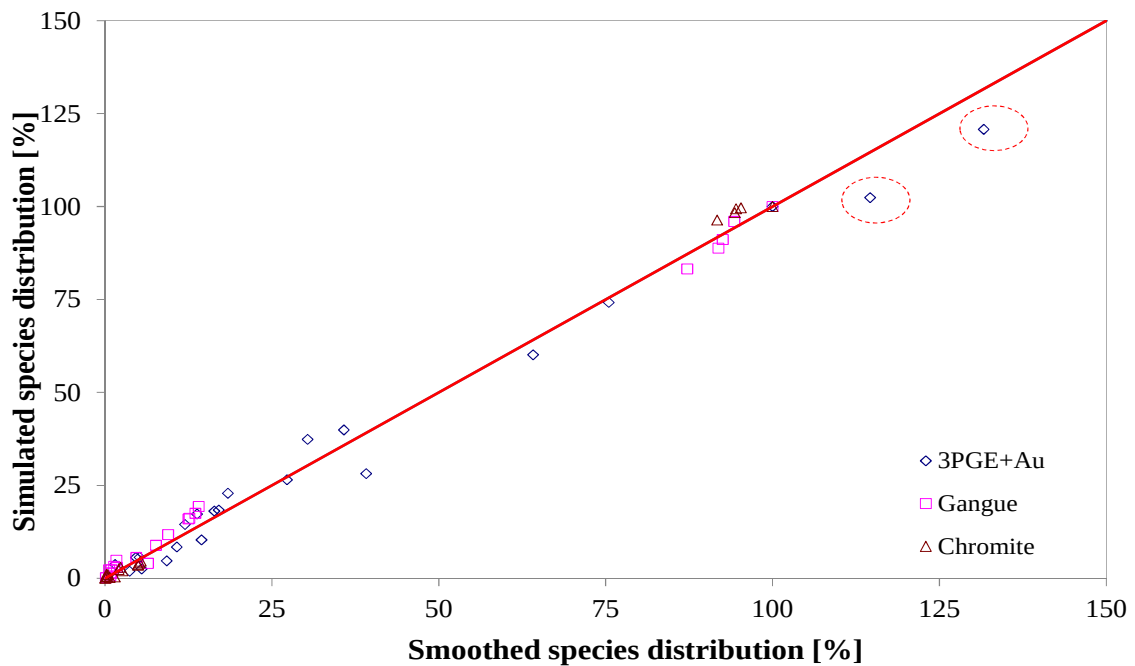


Figure 91: Parity chart for smoothed and simulated species distribution after model calibration

Table 31: Froth parameters obtained from simulator calibration

	β / [min ⁻¹]	η
PGMs	1.61	1.81×10^{-1}
Gangue	4.98	6.18×10^{-2}
Chromite	2.32	2.74×10^{-4}

7.4. Evaluation of effect of volume on PGM recovery

Figure 92 shows the approach used to evaluate the sensitivity of final PGM recovery for variations in flotation bank volume distribution. The calibrated simulator discussed earlier was used to carry out simulations. The volume of each flotation bank was changed incrementally, and the simulator was used to optimise final PGM recovery subject to final

product constraints. The constraints were a fixed mass recovery of 1.0% and a chromite grade less than 2.5%; the depressant dosage to the primary cleaner was fixed at 70 g/t, which was the dosage used for the actual pilot-plant campaign. The mass recovery and chromite grade are typical industrial constraints, which are used for proper operation of the smelter. Optimisation of the final PGM recovery was determined using SOLVER®, which varies the froth residence time per flotation bank. The PGM recovery obtained is a relative optimum. A profile for the final PGM recovery with variation in flotation bank volume was generated to determine the sensitivity of PGM recovery with flotation bank volume.

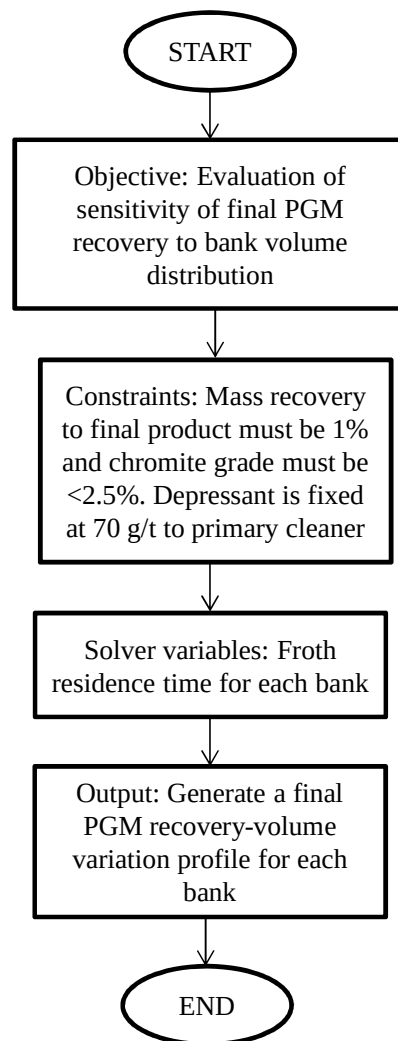


Figure 92: Approach for evaluating effect of volume distribution on final PGM recovery

The graphs in Figure 93 show the effect of changing the volume of selected flotation banks on the final PGM recovery. The volume of each bank is expressed as a percentage of the total cell volume in the plant. The dotted red line is the simulated optimum recovery for the existing distribution of cell volume. Therefore, the intersection of the lines with the dotted red

line gives the existing flotation bank volume relative to the total plant volume. Plant conditions are listed in Appendix E.

The rougher stage makes up a significant amount of the total plant capacity. Final PGM recovery is relatively insensitive to volume changes in primary roughers (PRF and PRS). The most significant change to the final PGM recovery was observed for the cleaner stages, i.e. PC, PRC, SC and SRC. These flotation banks make up a small amount of the total plant capacity but have the most influence on the final PGM recovery. For example, if the primary cleaner (PC) volume is increased from 5.9% to 6.8%, final PGM recovery is increased by 2.5%. Furthermore, decreasing the original capacity by almost 3% gives a PGM loss of more than 15%. Changes in the volume of other cleaner stages show similar trends, but the effects on the final PGM recovery are less pronounced. It is believed that sensitivity of PGM recovery to cleaner volume is due to the importance of the cleaner stage in improving selectivity between gangue and PGMs, and for reducing chromite entrainment, by making it possible to increase circulation, without loss in recovery. Hence, additional capacity is crucial in these stages since it provides more residence time, which improves selectivity subject to final concentrate constraints.

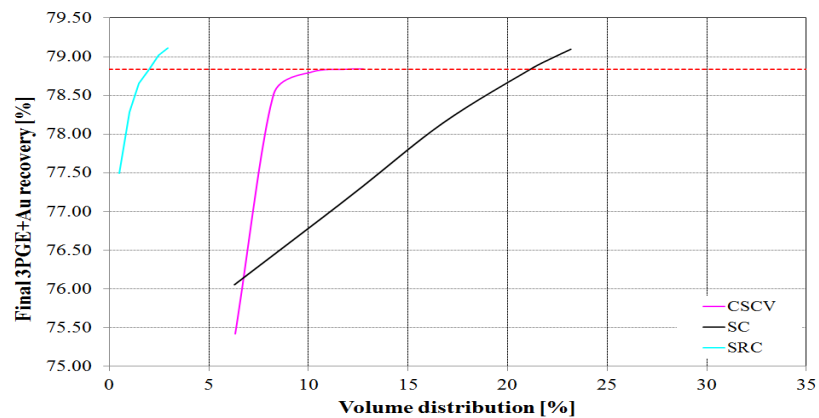
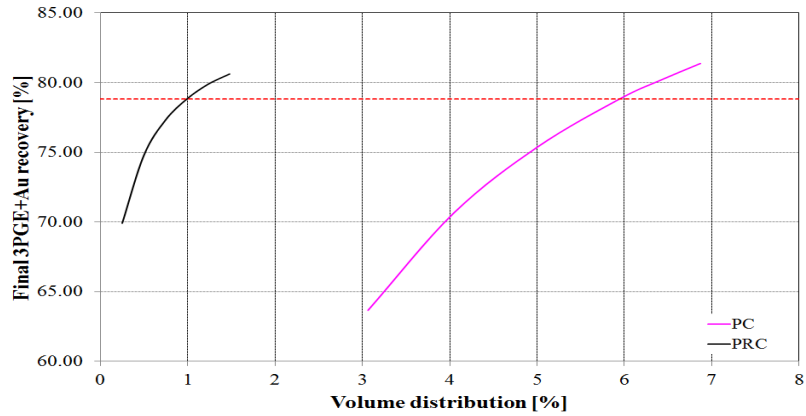
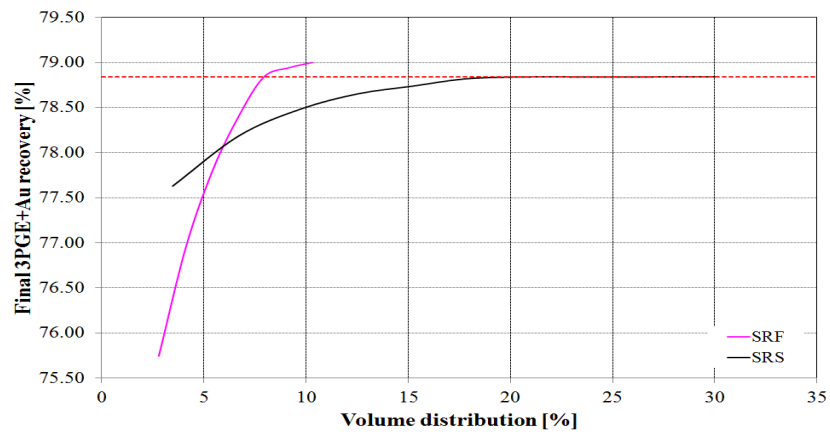
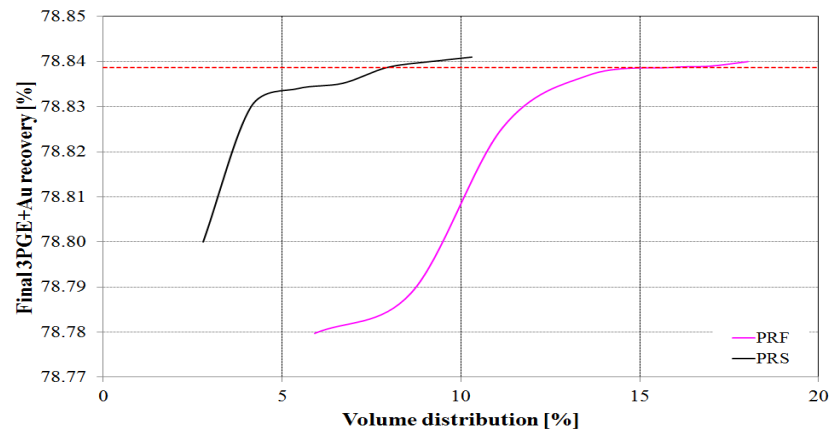


Figure 93: Evaluation of flotation bank volume reduction on final PGM recovery

7.5. Effect of depressant addition on PGM recovery

Figure 94 shows the approach used to evaluate the effect of cleaner depressant addition on the final PGM recovery. In this simulation exercise the plant volume distribution was fixed at the original plant distribution. The depressant to the cleaner stage was changed incrementally, and the simulator was used to find an optimum PGM recovery, using changes in froth residence time. The final product was constrained to a fixed mass recovery of 1.0% and a chromite grade less than 2.5%. The effect of cleaner depressant addition on final PGM recovery is shown in Figure 95. The results give a relative optimum for the cleaner depressant addition. At low dosages the PGM recovery is low and this may be attributed to poor selectivity between gangue and PGMs. The selectivity improves with depressant addition since more gangue is depressed; therefore, the recovery of PGMs improves to a relative optimum. At higher depressant additions, the recovery decreases and this may be attributed to depression of composite PGM particles.

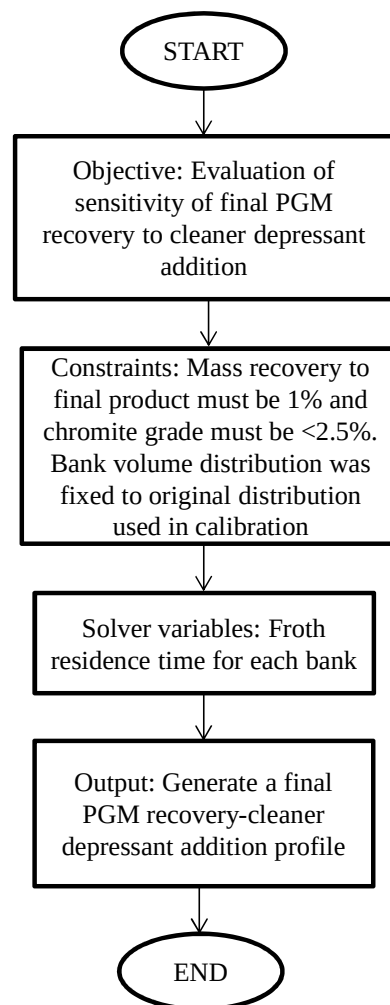


Figure 94: Approach for evaluating effect of cleaner depressant addition on final PGM recovery

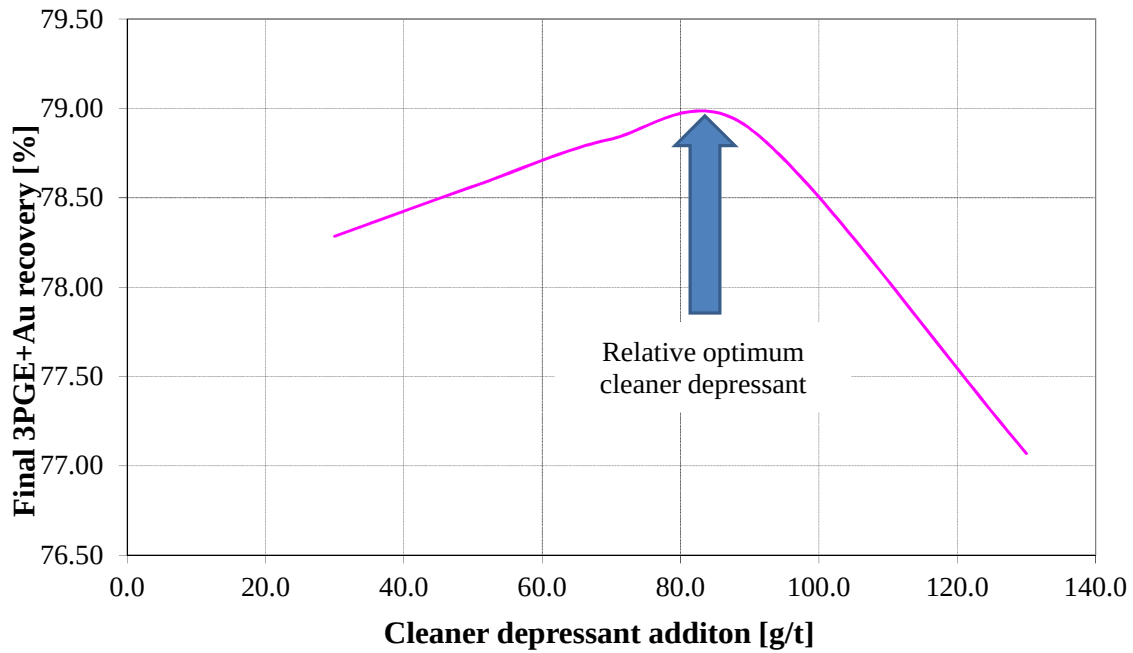


Figure 95: Effect of cleaner depressant addition on PGM recovery to final concentrate

7.6. Conclusions

A method of using batch and plant data to simulate and optimise plant performance has been presented. The simulator uses batch flotation data with calibration to plant flotation data to determine froth parameters (η and β) for each species. The parameters link batch flotation tests to plant performance by accounting for the difference in the froth behaviour. The froth height and hence the average froth residence time was measured in the flotation pilot-plant and incorporated in the model. This is considered to be an improvement on the simple froth factors used in other simulators such as JKSimFloat, SUPASIM, Aminproc and FLEET (also known as MinnoVex). It offers individuals who do not have access to these simulators an opportunity to develop their own simulators using a spread-sheet. Furthermore, the simulator relates well with the batch flotation test work presented earlier and enables an extrapolation of the parameters from these tests, together with suitable froth parameters, for evaluating plant performance.

Applications for this simulator include simulation of plant performance based on drill core samples of future mining of UG2 ore, and identification of problematic ores in the deposit. This is important for developing strategies for treating different plant feeds, i.e. change in plant volume, change in circuit configuration, optimisation of depressant addition and identifying liberation issues. Hence, PGM recovery variability is investigated on plant-scale.

CHAPTER 8. CONCLUSIONS

This study was focused upon the development of better test procedures for UG2 ore. The following conclusions were reached:

Preliminary batch flotation model development

A set of ten batch rougher flotation tests, on a single sample, was used to evaluate batch flotation models. Statistical tests were used to investigate different batch flotation models, and it was concluded that a two rate constant model was adequate to represent floatable PGM particles. A single rate constant model was adequate to represent floatable gangue and chromite particles. However, it was found that the recovery of unfloatable and floatable particles by entrainment was significant for PGMs, gangue and chromite therefore; parameters for entrainment of these minerals were included in the batch flotation model.

Application of the preliminary batch flotation test to illustrate UG2 variability

A standard rougher-cleaner batch flotation test was applied to fifty UG2 samples obtained from across the Bushveld Complex. A fixed energy was applied in grinding therefore, grind (per cent passing 75 μm) varied due to changes in ore hardness. Depressant was added to the cleaner stage to achieve a suitable froth structure. The batch flotation models discussed earlier (for PGMs, gangue and chromite) were regressed to the rougher-cleaner data. There was significant scatter in the regressed parameters, and this can be attributed to the feed being variable.

Linking PGM recovery variability to feed chemical characteristics

The floatable PGM mass fractions, obtained from model regression to rougher-cleaner data, were related to feed properties. Initially, it was found that there was no simple relationship between the mass fraction of floatable PGM particles and the grind and PGE feed grade. It was concluded that this was due to the complex mineralogy of the UG2 feeds, which have varying degrees of liberation, mineral association and mineral texture. A statistical algorithm was used to develop a relationship between PGM floatable recovery and feed characteristics. It was found that the recovery of floatable PGM particles was affected positively by base metal content and negatively by alteration indicators such as high Rb/Sr ratio and loss on ignition.

Development of an advanced flotation test procedure

A more advanced batch flotation test procedure was developed, with two stages of grinding and flotation. The primary grind was 40% -75 μm and secondary grind was 80% -75 μm . The concentrates were combined and floated in a cleaner stage, where the effect of a progressive increase in depressant addition was measured. A two rate constant model (and entrainment) was found to fit the data well, with the same rate parameters used in all three stages. The effect of regrind was to increase the mass fraction of the floatable minerals. A new model was used to link depressant addition to the mass fraction of slow floating PGMs and the floatable mass fractions for gangue and chromite. This was a new development. The advanced batch flotation test and modelling were used for developing a link to mineralogy and for simulation of flotation circuit performance.

Linking the PGM floatable recovery from the advanced batch flotation test to feed mineralogy

Seven samples of UG2 ore were selected for more detailed tests. The advanced flotation test was used on these samples, and mineralogical tests were carried out on feed samples milled to the final grind of 80% -75 μm . Data from automated SEM analyses of size fractions were collected on PGM degree of liberation, mineral association, mineral type and grain size. A simple link between floatable PGM recovery and PGM liberation was found. The model also included the effect of depressant addition on composite PGM particle flotation. The model can be used to estimate PGM liberation data (typically obtained from mineralogy) using the less expensive advanced batch flotation test.

Simulation of flotation circuit performance

Parameters for one of the ores, obtained from the advanced batch flotation test, were used to simulate flotation circuit performance. Two constrained simulation exercises were carried out, the first examined the effect of flotation bank volume on the final PGM recovery, and the second examined the effect of depressant addition on the final PGM recovery. In both simulations, the recovery of mass and chromite content were constrained to typical operating values. The simulator is a new development and illustrates how batch testing can be used for making a preliminary assessment of plant performance. This is important when developing a mining schedule for a deposit.

CHAPTER 9. RECOMMENDATIONS

The advanced test procedure developed in this study can be used to estimate the mineralogy of PGMs in UG2 samples and to predict plant performance. However, the latter requires more comparative tests, and it is recommended that the following additional work be done:

- The effect of a range of secondary grinds was not investigated. This can be an important operating variable, particularly for ores, which have significant ‘locking’ of PGMs in gangue or ores, which liberate at coarser grinds than that typically used for secondary grinding. One avenue of future test work should be an examination of the effect of changes in the amount of secondary grinding, preferably on one of the selected samples. This effect should be linked to the mineralogical-flotation model developed in this study. The effect of secondary grinding on entrainment of particles is also an important factor, and it should be examined. Excessive grinding can lead to overgrinding of the chromite and its entrainment in the final concentrate. Hence, future test work should examine the influence of secondary grinding on entrainment.
- It is recommended that the advanced flotation test be applied to plant feed samples, and that plant surveys be used to calculate scale-up factors. This is an important phase to validate the test procedure and to make the predictions of plant performance more accurate. Tests on known problematic ores should also be done to validate model predictions.

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APPENDIX A

A1. Description of assay techniques

A1.1. Total PGE+Au analysis

This section describes the method that was used by Mintek's analytical division to determine the total PGE+Au content in a UG2 sample, (Mukwevho, 2005).

1. A subsample was taken from a pulverised bulk sample and mixed with a flux consisting of: fused borax, sodium carbonate, silica, lead oxide and reducing agent (maize meal). The ratio of the lead oxide to the reducing agent was fixed between 10:1 and 11:1.
2. A silver nitrate solution was then added to the mixture before being heated in a furnace. The borax and sodium carbonate fused with the sample to release the PGEs which were collected by the metallic lead. Metallic lead forms from the reduction of lead oxide by maize meal and since it has a high density it falls to the bottom of the crucible.
3. The molten lead and slag was then poured into an iron mould and allowed to solidify. The lead was separated from the slag and placed into a cupel which was then placed into a muffle oven. A cupel is a porous cup that is used in cupellation which is the metallurgical process for separating noble metals by using high temperatures. A muffle oven is a high temperature oven that isolates the subject material, in this case the cupel contents, from fuel and products of combustion like gases and flying ash.
4. In the oven the lead becomes oxidised and is subsequently absorbed by the cupel material, a prill then forms which contains the PGEs.
5. The prill was weighed and the ratio of the prill mass to the total mass taken for analysis was used to determine the total PGE+Au content in g/t.

A1.2. 2PGE+Au analysis

In this section the method used by Mintek's analytical department for determination of the 2PGE+Au content in a UG2 sample will be discussed, (Mukwevho, 2005 and Maluleke, 2009). The method is similar to that described for the total PGE+Au method except, the prill is not analysed by gravimetric methods.

1. The prill obtained from the gravimetric analysis is transferred into a small beaker and 3 ml of concentrated nitric acid is added.
2. The beaker contents are then heated on a hot plate for 5 minutes.
3. The contents are cooled and 9 ml of concentrated hydrochloric acid was added, the contents was reheated until no nitrous oxide fumes were visible.
4. The contents was cooled and transferred into a 25 ml volumetric flask containing 1.25 ml of a 0.2 g/l scandium and 5 ml concentrated hydrochloric acid.
5. The solution was diluted with distilled water so that the PGEs could be read using inductive coupled plasma spectroscopy (ICP–OES).
6. The solution from the 25 ml volumetric flask was transferred into a clean 15 ml plastic auto sampler tube and the ICP–OES instrument was used to read the solution for elements: Pt, Pd and Au.
7. In the event that the elements exceeded the highest standard used to calibrate the instrument the solution was diluted.

A1.3. 3PGE+Au analysis

In this section the method used by Mintek's analytical department for determination of the 3PGE+Au content in a UG2 sample will be discussed, (Mukwevho, 2005 and Maluleke, 2010). The method is similar to that describes for the total PGE+Au method except, the prill is not analysed by a gravimetric method.

1. The prill was transferred into a clean 25 ml volumetric flask containing 1.25 ml of a 0.2 g/l scandium solution.
2. A 1:1 hydrochloric acid solution was used to make up the volume in the flask; the contents of the flask were then agitated by shaking so that the prill dissolved.
3. The solution was then read for elements: Pt, Pd, Rh and Au using the ICP–OES.

A1.4. 5PGE+Au analysis

In this section the method used by Mintek's analytical department for determination of the 5PGE+Au content in a UG2 sample will be discussed, (Maluleke, 2011).

1. A conventional fire assay method, similar to that discussed in § A1.1 was used to produce a NiS button which contains the noble metals: Pt, Pd, Rh, Ru, Ir and Au.
2. The button was crushed into small pieces by a hydraulic press.

3. 10 g of ammonium chloride was transferred into a 600 mL glass beaker.
4. The crushed button was also transferred into the beaker and 300 mL of a 37% analytical grade (i.e. AR grade) hydrochloric acid solution was added.
5. The beaker was covered immediately with a watch glass and placed on a boiling steam bath that was placed in a fume cupboard for approximately 16 hours.
6. The beaker was then removed from the steam bath and checked to ensure complete dissolution of the crushed button; this is usually indicated by a clear green colour with black residue at the bottom.
7. The watch glass and sides of the beaker were washed down with a 40% hydrochloric acid solution.
8. The solution was then filtered using a Buchner funnel under vacuum through a 0.45 µm Durapore membrane filter.
9. The sides of the funnel were washed with a 40% hydrochloric acid solution to remove all traces of Ni. The membrane filter (with precipitate) was transferred into the original 600 mL beaker and 20 mL of a 37% AR grade hydrochloric acid and 10 mL of a 65% AR grade nitric acid solution was added, the filtrate was discarded.
10. The beaker was covered with a watch glass and was gently heated until the entire residue dissolved. Thereafter 2 to 3 drops of hydrogen peroxide was added and the beaker contents were heat for a few minutes to get rid of the excess hydrogen peroxide.
11. The beaker was removed from the hot plate and the filter membrane was removed using a glass rod. The filter was then washed several times with a 40% hydrochloric acid solution into the beaker.
12. The beaker solution was then heated until a volume of approximately 20 mL remained; a 40% hydrochloric acid solution was used to transfer the solution into a 100 mL beaker.
13. The solution was then heated so that a volume of approximately 5 mL remained in the beaker, then 3.5 mL of 37% AR grade hydrochloric acid and 0.5 mL of AR grade nitric acid was added; the solution was boiled for approximately a minute.
14. The solution was cooled and transferred into a 25 mL volumetric flask.
15. 5 mL of 37% AR grade hydrochloric acid and 1.25 mL of a 0.2 g/L scandium solution was added. The volume was then made up to the flask capacity using deionised water and was agitated. The solution was read using an ICP–OES instrument.

A1.5. Total sulphur and sulphide sulphur analysis

The total sulphur and sulphide sulphur content for each sample was determined using a Leco sulphur analyser.

For determination of the total sulphur content a sample with mass between 0.5 and 1.0 g was heated to approximately 1350 °C in an induction furnace while passing a stream of oxygen through the sample. The sulphur dioxide released from the sample was measured by an IR detection system and the total sulphide content was provided.

For determination of the sulphide sulphur content, a sample with mass between 0.5 and 1.0 g was selectively leached in a warm sodium carbonate solution to convert the metal sulphate into insoluble carbonates and soluble sulphate. The sulphate residue was removed by filtration and was washed free of the carbonate solution. The solution was then analysed by Leco sulphur analyser to determine the sulphide sulphur content.

A1.6. Cr₂O₃ analysis

A 1 to 2 g subsample was taken from a pulverised bulk sample, usually 200 g, and was placed into a beaker with 9 ml of concentrated hydrochloric acid and 3 ml of nitric acid. The solid–liquid mixture was then refluxed (using a glass lid) for approximately 15 minutes on a hot plate so that the solids dissolve completely. The lid was washed into the beaker using millipore water and the solution was transferred into a 50 ml plastic beaker having 10 ml hydrochloric acid and 2.5 ml Sc. Millipore water was used to transfer the solution into the flask and to make up the solution to the flask volume. The solution was then read using an ICP–OES instrument which was calibrated for reading Cr. The Cr reading was then converted to Cr₂O₃ using the molecular weight of the elements and the assumption that all the Cr occurs as Cr₂O₃. The assumption that all the Cr occurs as Cr₂O₃ is logical since the UG2 sample is a chromitite ore.

A1.7. Acid soluble base metal sulphides

The acid soluble base metal sulphides: A/sNi, A/sFe, A/sCu and A/sCo were determined for each feed sample using a diagnostic leach. A 2.5 g subsample of the pulverised feed was added to a 250 ml lixiviant. The lixiviant was made from 37.5% ammonia acetate and 0.5% ammonia chloride. The lixiviant dissolves only the: oxides, hydroxides, carbonates and sulphates, but not the sulphides. The reactor was sparged with nitrogen for a period of 1 hour.

On completion of the reaction the residual samples were filtered; the feed assay and the residual samples were analysed for: Ni, Cu, Fe and Co using the ICP–OES instrument. The assay for each stream: feed, residue and filtrate was used to carry out a mass balance to determine the base metal sulphide content.

A1.8. X–ray fluorescence scan

A quantitative X–ray fluorescence (XRF) scan was carried out to determine some of elements present in the feed like: MgO, Al₂O₃, SiO₂, TiO₂, V₂O₅, MnO, Fe, P, Cl, Ca, Co, Ni, Cu, Zn, Ga, As, Se, Rb, Sr, Y and Zr. The instrument was inspected prior to analysing for the elements, the inspection list was used to check on the: the X–ray, spectrometer pressure, cooling water conductivity and temperature. Once these checks were found to be satisfactory an 8 g subsample was transferred into the sample holder for analysis. The sample was then bombarded with high energy X–rays which resulted in each element emitting its own characteristic wavelength. The wavelength was interpreted with a database which allowed for the determination of the element content.

A1.9. Loss on ignition analysis

A 2 g subsample from a pulverised feed sample was dried over night in a hot–air cabinet operating between 105–110 °C. This pre-heating step removes moisture from the feed sample. A muffle oven was pre–heated to the temperature at which the analysis was to be carried out, for this study four temperatures were investigated: 300, 600, 900 and 1050 °C. The mass of the crucible and crucible and subsample was measured before placing into the oven. The sample in the crucible was heated for 1 hour thereafter it was removed from the oven and allowed to cool for 15 minutes before it was reweighed. The difference in the sample mass at the start and end of the test was the mass loss, which can be express has the LOI value which is a percent loss relative to the feed used.

A1.10. Specific gravity analysis

A 15 g subsample for each feed was analysed to determine the specific gravity (SG). The specific density of each sample was relative to the density of water at ambient temperature and pressure. The specific density was determined using a pycnometer.

A2. Raw data for assay reproducibility on base case sample (Sample A)

Assay	Total PGE+Au/ [g/t]	2PGE+Au/ [g/t]	3PGE+Au/ [g/t]	5PGE+Au/ [g/t]
1	3.00	2.72	2.98	3.19
2	3.03	2.73	2.94	3.08
3	3.00	2.70	2.96	3.08
4	3.09	2.71	2.75	3.21
5	3.14	2.75	2.95	3.49
6	3.00	2.81	2.98	3.44
7	2.97	2.89	2.83	3.30
8	3.11	2.68	2.88	3.10
9	3.11	2.85	2.83	3.12
10	3.00	2.73	2.79	3.23

Assay	Sulphide sulphur/ [%]	Total sulphur/ [%]	AsNi/ [g/t]	AsCu/ [g/t]	AsFe/ [g/t]	AsCo [*] / [g/t]	Cr ₂ O ₃ / [%]	SG	LOI/ [%]	
									1050 °C	300 °C
1	0.130	0.197	298.50	83.00	7965.00	10.00	16.45	3.56	0.05	0.14
2	0.092	0.139	299.00	83.00	8000.00	10.00	16.60	3.57	0.06	0.14
3	0.083	0.125	298.00	85.00	7980.00	10.00	16.90	3.56	0.10	0.09
4	0.088	0.134	309.00	88.00	8100.00	10.00	16.40	3.57	0.07	0.14
5	0.079	0.120	311.00	88.00	8090.00	10.00	16.30	3.56	0.08	0.21
6	0.079	0.119	305.00	83.50	8005.00	10.00	16.70	3.57	0.10	0.11
7	0.079	0.119	302.00	83.00	7930.00	10.00	16.80	3.57	0.03	0.15
8	0.083	0.125	306.00	88.00	8240.00	10.00	16.40	3.56	0.04	0.14
9	0.086	0.130	309.00	88.00	8670.00	10.00	16.20	3.56	0.06	0.11
10	0.083	0.125	302.00	89.00	8330.00	10.00	16.60	3.56	0.14	0.12

Assay	MgO/ [%]	Al ₂ O ₃ / [%]	SiO ₂ / [%]	TiO ₂ / [%]	V ₂ O ₅ / [%]	MnO/ [%]	Fe/ [%]	P/ [%]	Cl/ [%]	Ca/ [%]
1	17.92	13.01	31.99	0.72	0.173	0.17	12.87	0.34	0.04	2.61
2	17.23	13.06	32.52	0.73	0.173	0.18	13.06	0.32	0.04	2.52
3	17.87	12.46	33.90	0.71	0.160	0.18	12.97	0.33	0.05	2.72
4	17.44	13.19	32.57	0.72	0.171	0.17	12.92	0.29	0.04	2.49
5	17.54	12.90	33.14	0.71	0.168	0.17	12.81	0.29	0.04	2.57
6	17.44	12.99	33.03	0.73	0.172	0.18	12.57	0.30	0.04	2.42

* Assay below detection limit which is 10 g/t.

7	17.36	12.76	33.03	0.74	0.175	0.17	12.48	0.30	0.04	2.49
8	17.58	13.21	32.93	0.72	0.170	0.17	12.47	0.31	0.04	2.52
9	17.60	12.19	35.20	0.67	0.141	0.18	14.21	0.34	0.05	2.91
10	17.29	13.08	32.38	0.72	0.179	0.17	12.44	0.29	0.04	2.49

Assay	Co/ [%]	Ni/ [%]	Cu/ [%]	Zn/ [%]	Ga/ [%]
1	2.61E-02	7.32E-02	9.29E-03	2.42E-02	3.11E-03
2	2.57E-02	8.93E-02	1.03E-02	2.67E-02	2.92E-03
3	2.91E-02	9.20E-02	1.15E-02	2.99E-02	3.29E-03
4	2.16E-02	9.38E-02	9.85E-03	2.69E-02	2.94E-03
5	2.64E-02	8.35E-02	1.04E-02	2.62E-02	3.10E-03
6	2.55E-02	9.93E-02	1.02E-02	2.68E-02	3.01E-03
7	2.73E-02	9.80E-02	1.10E-02	2.97E-02	3.12E-03
8	2.73E-02	8.24E-02	1.04E-02	2.65E-02	3.00E-03
9	2.72E-02	1.06E-01	1.30E-02	3.08E-02	3.63E-03
10	2.27E-02	8.16E-02	1.05E-02	2.61E-02	2.74E-03

Assay	As/ [%]	Se/ [%]	Rb/ [%]	Sr/ [%]	Y/ [%]	Zr/ [%]
1	3.15E-04	6.48E-05	3.70E-04	6.32E-03	4.63E-04	1.56E-03
2	3.15E-04	6.48E-05	3.33E-04	6.65E-03	3.80E-04	9.26E-04
3	2.25E-04	1.94E-04	4.19E-04	7.50E-03	4.39E-04	1.05E-03
4	4.71E-04	7.38E-05	3.78E-04	6.72E-03	3.32E-04	1.25E-03
5	2.88E-04	9.28E-05	3.25E-04	7.08E-03	4.27E-04	1.15E-03
6	4.73E-04	1.11E-04	3.80E-04	6.72E-03	3.25E-04	1.17E-03
7	3.47E-04	7.51E-05	3.57E-04	6.96E-03	3.47E-04	1.07E-03
8	3.53E-04	1.02E-04	3.63E-04	6.96E-03	3.26E-04	1.01E-03
9	4.66E-04	4.24E-05	3.92E-04	7.81E-03	4.13E-04	9.97E-04
10	3.66E-04	8.23E-05	3.11E-04	6.71E-03	4.02E-04	9.15E-04

A3. Mill setup and milling curve for base case sample

Rod diameter size/ [mm]	Number of rods	Total mass/ [g]
20	10	5405.9
15	10	3269.7
10	15	1847.3
5	9	456.0

Specific energy/ [kWh/t]	-75 μ m	+ 75 μ m
6.19	695.0	311.2
11.99	394.4	590.6
18.38	80.0	907.6

A4. Raw data for flotation reproducibility tests

Stream	Time	Mass recovery for tests/ [g]									
		1	2	3	4	5	6	7	8	9	10
RC-1	1	50.70	48.60	44.20	44.60	53.40	51.70	49.70	53.50	52.00	50.20
RC-2	3	35.00	32.90	43.20	31.10	40.80	35.30	33.20	32.30	39.00	34.70
RC-3	7	33.90	37.80	28.20	40.90	33.90	44.20	45.30	43.70	35.10	41.30
RC-4	20	50.20	48.70	59.70	53.30	50.70	49.00	49.30	50.50	54.60	48.30
RC-5	30	27.60	24.50	27.70	21.50	27.20	26.80	22.10	25.80	23.70	21.00
RT		798.30	803.70	796.00	804.40	790.60	790.50	795.00	790.80	790.30	800.30

RC– Rougher concentrate

RT– Rougher tail

Stream	Time	2PGE+Au assay for tests/ [g/t]									
		1	2	3	4	5	6	7	8	9	10
RC-1	1	30.92	34.00	35.60	36.32	28.52	30.12	32.12	30.91	31.60	31.60
RC-2	3	8.84	12.09	11.46	11.47	10.53	11.94	13.05	12.15	13.26	13.26
RC-3	7	6.12	4.45	6.21	6.88	6.29	6.00	5.78	6.54	6.29	6.29
RC-4	20	4.21	4.10	3.65	3.51	3.71	3.36	3.09	4.09	3.93	3.93
RC-5	30	2.28	2.47	2.32	2.59	2.80	2.02	2.14	2.61	2.69	2.69
RT		0.40	0.31	0.39	0.46	0.37	0.37	0.37	0.39	0.50	0.50

Stream	Time	Cr ₂ O ₃ assay for tests/ [%]									
		1	2	3	4	5	6	7	8	9	10
RC-1	1	3.30	2.57	2.39	2.21	3.08	3.22	2.50	2.82	3.29	2.93
RC-2	3	4.57	3.43	3.42	3.09	4.28	3.69	4.14	3.50	4.37	3.97
RC-3	7	6.16	4.98	5.23	4.67	5.23	6.23	6.82	5.20	5.75	6.37
RC-4	20	8.68	7.47	7.71	6.97	8.08	8.49	9.76	7.86	8.66	8.82
RC-5	30	10.70	9.29	9.82	8.85	9.74	10.10	10.90	9.26	11.20	10.30
RT		18.50	20.15	20.20	19.90	20.45	20.75	20.85	21.15	20.70	20.70

Stream	Time	Water recovery for tests/ [g]									
		1	2	3	4	5	6	7	8	9	10
RC-1	1	159.70	140.70	164.60	165.30	155.10	153.70	159.00	141.50	144.20	161.00
RC-2	3	118.60	134.70	127.80	118.90	181.40	150.20	185.70	144.80	160.30	152.40
RC-3	7	313.00	256.30	225.80	248.60	213.50	294.70	288.00	230.80	223.00	284.40
RC-4	20	487.00	521.40	548.40	512.00	519.50	519.50	566.30	533.90	556.60	558.70
RC-5	30	397.00	418.00	415.90	446.20	420.20	448.60	319.80	473.00	394.70	373.30
RT		1791.40	1947.50	1808.50	1913.30	1918.60	1909.30	1837.00	1916.10	1818.40	1849.50

A5. Parameter estimates and confidence intervals for batch models evaluated

PGMs		
Model	Best fit parameter estimates	95% Confidence interval for best fit parameter estimates
Classic model	$R_{\max} = 90.72$ $k = 0.78 \text{ min}^{-1}$	$R_{\max}$: 74.47-100.00 k : 0.57-1.23 min^{-1}
Klimpel model	$R_{\max} = 90.72$ $K_{\max} = 2.08 \text{ min}^{-1}$	$R_{\max}$: 82.65-99.29 $K_{\max}$: 1.80-2.35 min^{-1}
2nd order Klimpel model	$R_{\max} = 91.14$ $K_{\max} = 4.29 \text{ min}^{-1}$	$R_{\max}$: 87.67-94.63 $K_{\max}$: 3.95-4.64 min^{-1}
2nd order model	$R_{\max} = 90.97$ $k = 1.59 \times 10^{-2} \text{ min}^{-1}$	$R_{\max}$: 85.38-96.57 k : 1.53×10^{-2} - $1.65 \times 10^{-2} \text{ min}^{-1}$
Kelsall model	$R_{\text{fast}} = 71.43$ $R_{\text{slow}} = 28.57$ $k_{\text{fast}} = 1.49 \text{ min}^{-1}$ $k_{\text{slow}} = 3.51 \times 10^{-2} \text{ min}^{-1}$	R_{fast} : 67.75-74.63 R_{slow} : 25.37-32.25 k_{fast} : 1.40-1.61 min^{-1} k_{slow} : 2.69×10^{-2} - $5.01 \times 10^{-2} \text{ min}^{-1}$
Modified Kelsall model	$R_{\text{fast}} = 67.04$ $R_{\text{slow}} = 23.93$ $R_{\max} = 90.97$ $k_{\text{fast}} = 1.68 \text{ min}^{-1}$ $k_{\text{slow}} = 9.16 \times 10^{-2} \text{ min}^{-1}$	R_{fast} : 63.49-70.60 R_{slow} : 23.81-24.04 $R_{\max}$: 87.30-94.64 k_{fast} : 1.61-1.73 min^{-1} k_{slow} : 9.11×10^{-2} - $9.20 \times 10^{-2} \text{ min}^{-1}$
Classic model with entrainment	$R_{\max} = 67.09$ $R_{\text{ent}} = 32.91$ $k = 1.39 \text{ min}^{-1}$ $\text{ENT} = 2.59 \times 10^{-2}$	$R_{\max}$: 62.51-71.65 R_{ent} : 28.35-37.49 k : 1.38-1.40 min^{-1} ENT : 2.57×10^{-2} - 2.61×10^{-2}
Klimpel model with entrainment	$R_{\max} = 68.50$ $R_{\text{ent}} = 31.50$ $K_{\max} = 3.86 \text{ min}^{-1}$ $\text{ENT} = 2.60 \times 10^{-2}$	$R_{\text{float,max}}$: 65.46-71.43 R_{ent} : 28.57-34.54 $K_{\max}$: 3.81-3.93 min^{-1} ENT : 2.50×10^{-2} - 2.63×10^{-2}
2nd order Klimpel model with entrainment	$R_{\max} = 82.54$ $R_{\text{ent}} = 17.46$ $K_{\max} = 5.45 \text{ min}^{-1}$ $\text{ENT} = 9.03 \times 10^{-3}$	$R_{\max}$: 81.06-83.93 R_{ent} : 16.07-18.94 $K_{\max}$: 5.11-5.82 min^{-1} ENT : 7.80×10^{-3} - 1.05×10^{-2}
2nd order model with entrainment	$R_{\max} = 73.38$ $R_{\text{ent}} = 26.62$ $k = 3.44 \times 10^{-2} \text{ min}^{-1}$ $\text{ENT} = 1.81 \times 10^{-2}$	$R_{\max}$: 71.70-75.80 R_{ent} : 24.20-28.30 k : 3.11×10^{-2} - $4.09 \times 10^{-2} \text{ min}^{-1}$ ENT : 1.43×10^{-2} - 2.64×10^{-2}

Kelsall model with entrainment	$R_{fast} = 55.85$ $R_{slow} = 15.00$ $R_{ent} = 29.15$ $k_{fast} = 2.02 \text{ min}^{-1}$ $k_{slow} = 0.25 \text{ min}^{-1}$ $ENT = 2.55 \times 10^{-2}$	$R_{fast}: 53.05-58.40$ $R_{slow}: 14.54-15.73$ $R_{ent}: 25.87-32.41$ $k_{fast}: 1.98-2.03 \text{ min}^{-1}$ $k_{slow}: 0.23-0.29 \text{ min}^{-1}$ $ENT: 2.35 \times 10^{-2}-2.95 \times 10^{-2}$
Kelsall model with entrainment on slow floatability only	$R_{fast} = 59.00$ $R_{slow} = 16.16$ $R_{ent} = 24.84$ $k_{fast} = 2.00 \text{ min}^{-1}$ $k_{slow} = 0.22 \text{ min}^{-1}$ $ENT = 2.45 \times 10^{-2}$	$R_{fast}: 56.34-61.54$ $R_{slow}: 15.54-16.86$ $R_{ent}: 21.60-28.12$ $k_{fast}: 1.99-2.02 \text{ min}^{-1}$ $k_{slow}: 0.20-0.25 \text{ min}^{-1}$ $ENT: 2.42 \times 10^{-2}-2.49 \times 10^{-2}$
Modified Kelsall model with entrainment	$R_{fast} = 60.65$ $R_{slow} = 14.66$ $R_{ent} = 15.66$ $R_{max} = 90.97$ $k_{fast} = 1.73 \text{ min}^{-1}$ $k_{slow} = 1.24 \times 10^{-1} \text{ min}^{-1}$ $ENT = 2.65 \times 10^{-2}$	$R_{fast}: 58.34-62.96$ $R_{slow}: 14.61-14.76$ $R_{ent}: 15.56-15.71$ $R_{max}: 88.66-93.28$ $k_{fast}: 1.69-1.77 \text{ min}^{-1}$ $k_{slow}: 1.23 \times 10^{-1}-1.25 \times 10^{-1} \text{ min}^{-1}$ $ENT: 2.59 \times 10^{-2}-2.69 \times 10^{-2}$
Modified Kelsall model with entrainment on slow floatability only	$R_{fast} = 60.65$ $R_{slow} = 20.74$ $R_{ent} = 9.59$ $R_{max} = 90.97$ $k_{fast} = 1.81 \text{ min}^{-1}$ $k_{slow} = 1.51 \times 10^{-1} \text{ min}^{-1}$ $ENT = 2.34 \times 10^{-2}$	$R_{fast}: 58.36-62.94$ $R_{slow}: 20.63-21.02$ $R_{ent}: 9.31-9.70$ $R_{max}: 88.68-93.26$ $k_{fast}: 1.77-1.85 \text{ min}^{-1}$ $k_{slow}: = 1.50 \times 10^{-1}-1.52 \times 10^{-1} \text{ min}^{-1}$ $ENT: 2.21 \times 10^{-2}-2.42 \times 10^{-2}$

Gangue		
Model	Best fit parameter estimates	95% Confidence interval for best fit parameter estimates
Classic model	$R_{\max} = 25.79$ $k = 0.15 \text{ min}^{-1}$	$R_{\max}$: 21.51-30.06 k : 0.10-0.21 min^{-1}
Klimpel model	$R_{\max} = 25.79$ $K_{\max} = 0.40 \text{ min}^{-1}$	$R_{\max}$: 24.10-28.61 $K_{\max}$: 0.28-0.50 min^{-1}
2nd order Klimpel model	$R_{\max} = 29.77$ $K_{\max} = 0.46 \text{ min}^{-1}$	$R_{\max}$: 29.39-30.57 $K_{\max}$: 0.33-0.63 min^{-1}
2nd order model	$R_{\max} = 27.37$ $k = 8.21 \times 10^{-3} \text{ min}^{-1}$	$R_{\max}$: 26.51-28.60 k : 5.92×10^{-3} - $1.07 \times 10^{-2} \text{ min}^{-1}$
Kelsall model	$R_{\text{fast}} = 13.13$ $R_{\text{slow}} = 86.87$ $k_{\text{fast}} = 0.52 \text{ min}^{-1}$ $k_{\text{slow}} = 4.93 \times 10^{-3} \text{ min}^{-1}$	R_{fast} : 12.46-14.18 R_{slow} : 85.82-87.54 k_{fast} : 0.37-0.68 min^{-1} k_{slow} : 4.35×10^{-3} - $5.36 \times 10^{-3} \text{ min}^{-1}$
Modified Kelsall model	$R_{\text{fast}} = 8.66$ $R_{\text{slow}} = 18.61$ $R_{\max} = 27.27$ $k_{\text{fast}} = 0.90 \text{ min}^{-1}$ $k_{\text{slow}} = 6.44 \times 10^{-2} \text{ min}^{-1}$	R_{fast} : 8.40-9.68 R_{slow} : 17.75-19.99 $R_{\max}$: 26.16-29.67 k_{fast} : 0.53-1.20 min^{-1} k_{slow} : 3.78×10^{-2} - $7.23 \times 10^{-2} \text{ min}^{-1}$
Classic model with entrainment	$R_{\max} = 10.02$ $R_{\text{ent}} = 89.98$ $k = 0.56 \text{ min}^{-1}$ $\text{ENT} = 4.86 \times 10^{-3}$	$R_{\max}$: 9.74-10.60 R_{ent} : 89.40-90.26 k : 0.42-0.70 min^{-1} ENT : 4.48×10^{-3} - 5.14×10^{-3}
Klimpel model with entrainment	$R_{\max} = 12.26$ $R_{\text{ent}} = 87.74$ $K_{\max} = 1.03 \text{ min}^{-1}$ $\text{ENT} = 4.21 \times 10^{-3}$	$R_{\text{float,max}}$: 12.07-12.62 R_{ent} : 87.38-87.93 $K_{\max}$: 0.77-1.31 min^{-1} ENT : 3.84×10^{-3} - 4.48×10^{-3}
2nd order Klimpel model with entrainment	$R_{\max} = 17.01$ $R_{\text{ent}} = 82.99$ $K_{\max} = 0.97 \text{ min}^{-1}$ $\text{ENT} = 3.13 \times 10^{-3}$	$R_{\max}$: 16.81-17.67 R_{ent} : 82.33-83.19 $K_{\max}$: 0.72-1.27 min^{-1} ENT : 2.75×10^{-3} - 3.43×10^{-3}
2nd order model with entrainment	$R_{\max} = 14.44$ $R_{\text{ent}} = 85.56$ $k = 3.56 \times 10^{-2} \text{ min}^{-1}$ $\text{ENT} = 3.62 \times 10^{-3}$	$R_{\max}$: 14.10-14.80 R_{ent} : 85.20-85.90 k : 2.59×10^{-2} - $4.56 \times 10^{-2} \text{ min}^{-1}$ ENT : 3.23×10^{-3} - 3.92×10^{-3}
Kelsall model with entrainment	$R_{\text{fast}} = 11.16$ $R_{\text{slow}} = 35.02$ $R_{\text{ent}} = 53.82$	R_{fast} : 10.95-11.77 R_{slow} : 34.12-42.17 R_{ent} : 46.07-54.93

	$k_{fast} = 0.54 \text{ min}^{-1}$ $k_{slow} = 4.15 \times 10^{-3} \text{ min}^{-1}$ $ENT = 3.10 \times 10^{-3}$	$k_{fast}: 0.42\text{-}0.66 \text{ min}^{-1}$ $k_{slow}: 3.34 \times 10^{-3}\text{-}5.00 \times 10^{-3} \text{ min}^{-1}$ $ENT: 2.51 \times 10^{-3}\text{-}3.34 \times 10^{-3}$
Kelsall model with entrainment on slow floatability only	$R_{fast} = 11.16$ $R_{slow} = 35.02$ $R_{ent} = 53.82$ $k_{fast} = 0.54 \text{ min}^{-1}$ $k_{slow} = 3.51 \times 10^{-3} \text{ min}^{-1}$ $ENT = 3.35 \times 10^{-3}$	$R_{fast}: 10.16\text{-}11.77$ $R_{slow}: 34.71\text{-}35.93$ $R_{ent}: 53.52\text{-}53.92$ $k_{fast}: 0.42\text{-}0.67 \text{ min}^{-1}$ $k_{slow}: 3.37 \times 10^{-3}\text{-}3.57 \times 10^{-3} \text{ min}^{-1}$ $ENT: 2.39 \times 10^{-3}\text{-}3.57 \times 10^{-3}$
Modified Kelsall model with entrainment	$R_{fast} = 6.99$ $R_{slow} = 11.67$ $R_{ent} = 36.50$ $R_{max} = 55.17$ $k_{fast} = 1.00 \text{ min}^{-1}$ $k_{slow} = 9.60 \times 10^{-2} \text{ min}^{-1}$ $ENT = 4.81 \times 10^{-3}$	$R_{fast}: 6.02\text{-}7.95$ $R_{slow}: 10.95\text{-}12.07$ $R_{ent}: 36.04\text{-}37.15$ $R_{max}: 53.01\text{-}57.17$ $k_{fast}: 0.84\text{-}1.14 \text{ min}^{-1}$ $k_{slow}: 9.37 \times 10^{-2}\text{-}9.98 \times 10^{-2} \text{ min}^{-1}$ $ENT: 4.56 \times 10^{-3}\text{-}5.43 \times 10^{-3}$
Modified Kelsall model with entrainment on slow floatability only	$R_{fast} = 6.99$ $R_{slow} = 11.83$ $R_{ent} = 31.67$ $R_{max} = 50.49$ $k_{fast} = 1.03 \text{ min}^{-1}$ $k_{slow} = 9.60 \times 10^{-2} \text{ min}^{-1}$ $ENT = 5.97 \times 10^{-3}$	$R_{fast}: 6.02\text{-}7.96$ $R_{slow}: 11.45\text{-}12.48$ $R_{ent}: 31.42\text{-}41.92$ $R_{max}: 48.89\text{-}62.36$ $k_{fast}: 0.87\text{-}1.15 \text{ min}^{-1}$ $k_{slow}: 9.37 \times 10^{-2}\text{-}9.98 \times 10^{-2} \text{ min}^{-1}$ $ENT: 4.65 \times 10^{-3}\text{-}6.11 \times 10^{-3}$

Chromite		
Model	Best fit parameter estimates	95% Confidence interval for best fit parameter estimates
Classic model	$R_{\max} = 7.90$ $k = 6.21 \times 10^{-2} \text{ min}^{-1}$	$R_{\max}$: 6.92-11.59 k : 2.29×10^{-2} - $7.70 \times 10^{-2} \text{ min}^{-1}$
Klimpel model	$R_{\max} = 9.01$ $K_{\max} = 0.12 \text{ min}^{-1}$	$R_{\max}$: 8.89-14.44 $K_{\max}$: 0.04-0.21 min^{-1}
2nd order Klimpel model	$R_{\max} = 12.47$ $K_{\max} = 0.10 \text{ min}^{-1}$	$R_{\max}$: 10.60-23.63 $K_{\max}$: 0.02-0.22 min^{-1}
2nd order model	$R_{\max} = 10.64$ $k = 5.09 \times 10^{-3} \text{ min}^{-1}$	$R_{\max}$: 9.48-18.89 k : 7.85×10^{-4} - $1.21 \times 10^{-2} \text{ min}^{-1}$
Kelsall model	$R_{\text{fast}} = 2.11$ $R_{\text{slow}} = 97.89$ $k_{\text{fast}} = 0.28 \text{ min}^{-1}$ $k_{\text{slow}} = 1.63 \times 10^{-3} \text{ min}^{-1}$	R_{fast} : 1.43-2.39 R_{slow} : 97.61-98.57 k_{fast} : 0.07-0.90 min^{-1} k_{slow} : 1.30×10^{-3} - $1.90 \times 10^{-3} \text{ min}^{-1}$
Modified Kelsall model	$R_{\text{fast}} = 1.07$ $R_{\text{slow}} = 10.12$ $R_{\max} = 11.19$ $k_{\text{fast}} = 0.60 \text{ min}^{-1}$ $k_{\text{slow}} = 2.74 \times 10^{-2} \text{ min}^{-1}$	R_{fast} : 0.37-1.68 R_{slow} : 9.02-13.02 $R_{\max}$: 9.39-14.70 k_{fast} : 0.53-1.82 min^{-1} k_{slow} : 9.93×10^{-3} - $3.59 \times 10^{-2} \text{ min}^{-1}$
Entrainment only	ENT=0.19	ENT: 0.16-0.21
Classic model with entrainment	$R_{\max} = 1.04$ $R_{\text{ent}} = 95.53$ $k = 1.73 \times 10^{-1} \text{ min}^{-1}$ ENT= 1.66×10^{-3}	$R_{\max}$: 0.47-1.61 R_{ent} : 95.03-97.24 k : 1.63×10^{-1} - $2.22 \times 10^{-1} \text{ min}^{-1}$ ENT: 1.63×10^{-3} - 1.94×10^{-3}
Klimpel model with entrainment	$R_{\max} = 1.32$ $R_{\text{ent}} = 95.53$ $K_{\max} = 0.31 \text{ min}^{-1}$ ENT= 1.61×10^{-3}	$R_{\text{float, max}}$: 1.15-1.67 R_{ent} : 95.51-96.53 $K_{\max}$: 0.07-2.26 min^{-1} ENT: 1.30×10^{-3} - 1.81×10^{-3}
2nd order Klimpel model with entrainment	$R_{\max} = 1.82$ $R_{\text{ent}} = 95.53$ $K_{\max} = 0.29 \text{ min}^{-1}$ ENT= 1.56×10^{-3}	$R_{\max}$: 1.32-2.77 R_{ent} : 95.50-96.13 $K_{\max}$: 0.04-2.99 min^{-1} ENT: 1.46×10^{-3} - 1.78×10^{-3}
2nd order model with entrainment	$R_{\max} = 1.53$ $R_{\text{ent}} = 95.53$ $k = 1.03 \times 10^{-1} \text{ min}^{-1}$ ENT= 1.58×10^{-3}	$R_{\max}$: 1.01-2.07 R_{ent} : 91.59-97.53 k : 9.60×10^{-3} - $4.24 \times 10^{-1} \text{ min}^{-1}$ ENT: 1.46×10^{-3} - 1.67×10^{-3}

APPENDIX B

B1. Raw data for feed chemical assays

Sample	MgO/ [%]	Al ₂ O ₃ / [%]	SiO ₂ / [%]	TiO ₂ / [%]	V ₂ O ₅ / [%]	MnO/ [%]	Fe/ [%]	Total S/ [%]	Cr ₂ O ₃ / [%]	P/ [%]	Cl/ [%]	Ca/ [%]	Co/ [%]	Ni/ [%]	Cu/ [%]	Zn/ [%]	Ga/ [g/t]	As/ [g/t]	Se/ [g/t]	Rb/ [g/t]	Sr/ [g/t]	Y/ [g/t]	Zr/ [g/t]	Pt/ [g/t]	Pd/ [g/t]	Rh/ [g/t]	Au/ [g/t]	Ir/ [g/t]	Ru/ [g/t]
1	12.72	18.22	14.92	0.86	0.33	0.18	17.02	0.10	33.64	0.26	0.02	1.46	0.04	0.14	0.03	0.05	51.17	3.70	3.90	2.20	34.04	2.10	11.61	2.51	1.67	0.46	0.07	0.56	0.82
2	17.20	12.67	41.40	0.60	0.13	0.18	10.40	0.04	13.80	0.29	0.03	3.11	0.03	0.07	0.02	0.02	25.14	1.42	1.42	4.35	110.03	4.92	20.70	1.14	0.45	0.18	0.03	0.43	0.27
3	12.49	19.87	15.74	0.74	0.32	0.17	15.74	0.03	32.45	0.27	0.02	1.97	0.04	0.10	0.01	0.05	54.48	2.16	1.77	1.67	73.56	2.16	3.44	2.64	1.25	0.49	0.06	1.20	0.60
4	19.28	11.13	42.83	0.63	0.13	0.18	11.22	0.04	11.50	0.26	0.02	2.64	0.02	0.07	0.01	0.02	23.36	1.39	0.83	5.19	79.55	5.19	21.79	1.19	0.55	0.17	0.07	0.40	0.25
5	19.80	10.18	43.95	0.63	0.12	0.18	11.20	0.06	10.92	0.29	0.02	2.51	0.02	0.06	0.02	0.02	19.53	0.83	0.65	6.29	74.68	5.64	18.32	0.80	0.30	0.13	0.03	0.44	0.19
6	12.34	19.51	17.70	0.81	0.31	0.17	15.40	0.02	31.29	0.26	0.02	1.98	0.03	0.09	0.01	0.05	50.65	3.51	3.41	1.81	100.70	2.01	6.82	2.57	1.02	0.51	0.04	1.05	0.61
7	15.81	16.63	21.82	0.78	0.29	0.17	15.40	0.04	25.88	0.29	0.02	2.68	0.03	0.09	0.01	0.04	40.25	2.46	1.44	2.77	30.71	3.80	4.83	2.36	2.00	0.40	0.02	0.13	0.68
8	16.07	14.43	26.17	0.67	0.23	0.17	13.47	0.06	26.55	0.26	0.02	1.71	0.03	0.10	0.01	0.04	45.32	3.56	2.21	5.97	52.92	3.66	12.51	2.68	1.98	0.41	0.16	0.17	0.80
9	15.20	16.50	19.00	0.79	0.31	0.17	15.80	0.08	30.00	0.28	0.02	1.61	0.04	0.11	0.02	0.05	45.00	4.50	4.20	4.50	50.21	1.90	8.20	2.25	1.29	0.36	0.04	0.19	0.79
10	16.22	14.72	23.28	0.75	0.25	0.17	14.13	0.07	28.35	0.26	0.02	1.58	0.03	0.11	0.01	0.04	46.16	1.29	0.99	5.17	53.22	3.48	13.43	2.21	1.37	0.38	0.03	0.19	0.80
11	13.60	17.71	18.01	0.95	0.31	0.17	15.21	0.05	31.81	0.30	0.02	1.66	0.04	0.09	0.01	0.04	45.52	2.70	1.20	1.30	69.82	3.60	16.21	2.40	1.88	0.44	0.03	0.16	0.92
12	13.68	15.89	24.79	1.54	0.27	0.18	14.93	0.15	23.73	0.29	0.02	4.33	0.03	0.08	0.01	0.06	37.61	5.83	0.40	2.72	24.63	4.63	32.88	1.87	0.89	0.34	0.01	0.12	0.71
13	13.13	19.55	23.74	0.65	0.27	0.16	13.04	0.05	25.68	0.27	0.02	3.24	0.03	0.10	0.01	0.03	41.15	2.82	3.02	5.25	258.84	4.38	13.23	2.27	1.45	0.33	0.02	0.16	0.70
14	16.66	14.90	24.65	0.82	0.24	0.17	14.22	0.04	25.52	0.26	0.03	2.27	0.03	0.11	0.01	0.06	39.75	1.27	0.88	1.66	78.13	1.66	9.45	2.68	1.28	0.39	0.04	0.16	0.79
15	14.23	16.75	26.72	0.69	0.24	0.15	12.39	0.03	25.66	0.28	0.02	2.66	0.03	0.07	0.00	0.04	40.18	1.84	1.36	2.71	84.92	3.10	12.01	1.88	0.73	0.31	0.02	0.15	0.69
16	16.26	15.64	19.68	0.69	0.26	0.21	16.15	0.01	29.04	0.28	0.02	1.57	0.03	0.10	0.00	0.05	45.70	23.1	2.04	2.25	34.87	3.58	8.28	3.22	1.72	0.66	0.01	0.33	1.10
17	16.85	15.05	20.46	0.63	0.24	0.18	14.85	0.02	29.94	0.27	0.02	1.31	0.03	0.10	0.00	0.04	43.33	2.41	2.21	3.21	38.72	3.11	8.63	3.45	1.40	0.58	0.01	0.22	1.00
18	20.39	12.34	35.84	0.59	0.17	0.17	11.63	0.03	15.35	0.63	0.02	2.69	0.02	0.09	0.01	0.02	29.29	0.94	0.66	2.92	58.49	3.58	8.19	1.22	0.40	0.14	0.01	0.11	0.38
19	19.85	9.54	29.54	0.56	0.23	0.21	13.80	0.13	21.30	0.59	0.03	3.98	0.03	0.13	0.03	0.03	35.83	1.45	0.97	4.12	88.85	4.36	10.77	1.45	0.74	0.21	0.01	0.12	0.57
20	18.31	13.91	32.61	0.64	0.19	0.18	12.73	0.06	17.92	0.50	0.02	2.75	0.03	0.10	0.01	0.03	32.03	1.18	0.29	3.33	81.19	3.62	5.88	1.06	0.54	0.18	0.01	0.15	0.50
21	18.26	13.57	33.55	0.64	0.19	0.17	12.81	0.02	17.40	0.47	0.02	2.76	0.02	0.08	0.01	0.03	31.64	2.49	1.24	2.96	80.96	3.06	11.47	1.86	1.16	0.32	0.00	0.14	0.61
22	17.73	13.90	34.71	0.69	0.18	0.17	12.04	0.02	17.07	0.49	0.03	2.83	0.02	0.07	0.01	0.03	29.11	2.61	0.75	3.27	84.25	3.36	8.68	1.05	0.55	0.20	0.01	0.12	0.44
23	16.43	15.82	25.45	0.74	0.24	0.18	14.32	0.03	24.37	0.24	0.01	1.94	0.03	0.12	0.01	0.04	43.48	1.03	0.93	5.87	63.88	4.12	8.24	2.78	1.81	0.51	0.05	0.07	0.84
24	16.08	16.18	22.36	0.86	0.29	0.18	14.74	0.04	26.69	0.25	0.02	2.08	0.03	0.13	0.01	0.04	41.53	2.89	1.86	2.89	20.70	4.23	29.68	2.61	1.72	0.50	0.04	0.12	0.83
25	13.25	14.70	41.67	0.63	0.16	0.18	12.37	0.03	13.56	0.26	0.02	3.02	0.02	0.08	0.01	0.02	29.19	1.97	0.62	1.66	104.45	3.31	10.04	1.16	0.57	0.21	0.01	0.07	0.79
26	11.75	16.36	17.52	0.79	0.30	0.19	17.28	0.03	34.09	0.24	0.02	1.20	0.04	0.14	0.01	0.05	54.24	1.58	2.22	0.84	27.12	2.85	5.38	2.17	1.50	0.46	0.02	0.00	0.70
27	11.86	16.35	11.40	2.50	0.48	0.17	18.88	0.06	36.32	0.30	0.03	1.43	0.04	0.08	0.02	0.06	65.20	3.40	1.55	0.93	27.65	2.89	3.20	4.28	1.73	0.87	0.01	0.38	1.60
28	13.69	14.48	19.80	0.66	0.23	0.19	14.14	0.03	34.17	0.31	0.02	2.07	0.04	0.11	0.01	0.05	50.91	2.60	1.70	0.91	60.64	3.05	9.84	3.07	1.59	0.59	0.01	0.27	0.72
29	14.52	15.96	16.58	0.99	0.32	0.19	17.30	0.03	32.12	0.23	0.02	1.55	0.04	0.10	0.01	0.05	47.46	4.63	2.37	1.34	88.91	2.88	7.93	3.85	1.75	0.79	0.04	0.40	1.29
30	17.84	14.11	28.62	0.74	0.22	0.18	14.11	0.02	21.66	0.25	0.02	2.06	0.02	0.10	0.01	0.04	38.71	0.98	1.18	2.74	66.45	2.45	9.31	2.42	5.18	0.59	0.01	0.28	0.82
31	12.54	19.11	16.38	0.68	0.28	0.17	15.27	0.02	32.56	0.28	0.02	2.50	0.03	0.09	0.004	0.05	46.01	0.91	2.02	2.33	83.23	2.33	3.84	3.37	1.82	0.71	0.01	0.29	1.06
32	15.34	15.19	27.04	0.74	0.25	0.17	14.40	0.01	23.99	0.24	0.02	2.44	0.03	0.09	0.01	0.04	40.21	1.08	0.88	4.82	71.48	2.95	8.55	2.34	1.39	0.48	0.03	0.26	0.87
33	13.45	18.96	15.86	0.73	0.28	0.17	16.01	0.02	32.31	0.23	0.02	1.77	0.04	0.10	0.01	0.05	49.12	1.30	1.40	2.50	58.72	1.10	4.10	3.44	2.18	0.63	0.03	0.30	1.02
34	11.68	20.01	17.47	0.74	0.31	0.16	13.86	0.02	32.75	0.45	0.02	2.31	0.03	0.13	0.01	0.05	59.72	1.62	0.81	0.71	128.56	2.44	10.77	2.72	1.04	0.49	0.01	0.45	0.77
35	11.95	19.38	16.47	0.78	0.30	0.17	14.96	0.04	33.44	0.45	0.02	1.82	0.03	0.12	0.01	0.05	58.54	2.21	0.90	0.60	76.01	1.91	11.75	2.31	1.50	0.48	0.02	0.34	0.59
36	10.95	17.65	17.44	0.71	0.30	0.17	15.10	0.02	34.13	0.49	0.02	2.81	0.03	0.09	0.01	0.05	57.42	1.91	1.49	1.91	88.43	2.02	6.59	1.94	1.14	0.35	0.01	0.33	0.52
37	11.03	19.80	16.45	0.72	0.30	0.16	14.58	0.02	33.99	0.45	0.02	2.27	0.03	0.12	0.01	0.05	53.98	1.48	0.99	0.99	92.50	1.08	7.19	2.47	1.51	0.49	0.05	0.34	0.61

38	19.18	12.39	32.86	0.68	0.15	0.18	12.63	0.02	18.75	0.47	0.03	2.49	0.03	0.10	0.01	0.02	30.52	0.96	0.77	4.97	64.38	3.64	10.91	2.22	1.18	0.46	0.02	0.10	0.55
39	19.75	8.06	30.49	0.48	0.10	0.23	13.83	0.02	23.58	0.57	0.02	2.64	0.03	0.13	0.01	0.03	36.79	0.74	0.99	5.18	76.17	4.07	10.12	2.35	1.06	0.47	0.01	0.12	0.63
40	18.52	13.64	34.50	0.55	0.17	0.17	11.60	0.03	17.64	0.48	0.01	2.52	0.02	0.09	0.01	0.03	28.36	2.44	0.58	0.68	71.73	2.24	8.58	1.57	0.97	0.27	0.02	0.19	0.54
41	19.14	12.76	34.67	0.62	0.13	0.17	11.71	0.02	18.00	0.44	0.02	2.14	0.02	0.10	0.004	0.03	35.05	0.86	0.57	4.95	61.43	2.86	12.00	2.03	1.21	0.45	0.01	0.20	0.59
42	17.94	15.04	33.56	0.59	0.16	0.17	11.67	0.02	17.46	0.49	0.01	2.73	0.02	0.09	0.004	0.03	32.02	1.16	0.39	2.89	84.77	2.80	3.09	1.62	0.74	0.33	0.01	0.17	0.52
43	14.94	14.48	29.07	0.77	0.20	0.18	14.13	0.02	23.30	0.49	0.02	2.19	0.03	0.12	0.01	0.03	41.73	0.71	0.51	3.55	61.99	3.24	11.95	2.93	2.35	0.63	0.03	0.12	0.73
44	18.47	14.21	31.04	0.66	0.16	0.17	12.47	0.02	20.01	0.43	0.02	2.15	0.02	0.11	0.01	0.03	39.26	1.84	1.74	3.00	66.62	3.67	9.48	2.38	2.75	0.53	0.00	0.22	0.76
45	20.18	12.15	32.62	0.67	0.15	0.18	13.20	0.05	18.08	0.46	0.03	2.01	0.03	0.13	0.02	0.03	32.52	1.53	0.77	2.49	80.84	2.49	8.42	1.90	1.55	0.41	0.03	0.10	0.55
46	19.04	13.42	32.94	0.65	0.16	0.18	12.38	0.02	18.52	0.46	0.02	2.02	0.03	0.10	0.01	0.03	33.99	1.90	1.43	2.76	890.55	3.71	11.90	1.89	1.24	0.40	0.01	0.16	0.60
47	15.86	12.61	33.92	0.65	0.19	0.21	14.98	0.07	19.20	0.33	0.05	1.71	0.03	0.13	0.02	0.03	36.34	5.04	2.47	1.24	39.17	2.16	2.88	2.22	2.72	0.57	0.09	0.25	0.88
48	7.00	18.19	36.03	0.97	0.25	0.17	13.62	0.05	19.26	0.36	0.05	3.91	0.02	0.06	0.01	0.03	4.17	6.00	0.30	11.58	165.44	8.03	36.48	1.58	0.72	0.16	0.02	0.27	0.43
49	12.04	17.07	28.34	0.65	0.22	0.19	15.24	0.23	23.01	0.35	0.05	2.31	0.03	0.16	0.05	0.04	40.12	5.99	2.64	6.09	80.45	3.86	34.33	2.87	2.44	0.58	0.10	0.12	0.55
50	13.99	16.60	24.12	0.74	0.26	0.18	14.40	0.04	26.96	0.320	0.021	2.180	0.031	0.102	0.013	0.041	42.90	2.28	1.61	3.27	77.17	3.46	13.48	2.26	1.21	0.41	0.03	0.41	0.63

Sample	Sulphide S/ [%]	AsCu/ [g/t]	AsNi/ [g/t]	AsFe/ [g/t]	AsCo/ [g/t]	SG	LOI/GOI @ 1050 °C/ [%]	LOI/GOI @900 °C/ [%]	LOI/GOI @ 600 °C/ [%]	LOI/GOI @ 300 °C/ [%]
1	0.070	306.00	671.00	6970.00	10.00	3.99	-0.50	-0.43	3.77	0.58
2	0.031	111.00	459.00	5980.00	10.00	3.41	-0.26	-0.04	0.57	0.12
3	0.019	41.00	380.00	4990.00	15.00	3.95	-1.03	-1.06	0.87	0.53
4	0.034	124.00	419.00	5110.00	10.00	3.41	-0.21	-0.14	0.96	0.15
5	0.043	230.00	820.00	6390.00	20.00	3.40	-0.22	-0.03	0.59	0.07
6	0.013	27.00	440.00	9520.00	10.00	3.90	-0.04	-0.07	0.97	0.56
7	0.011	69.00	485.00	3830.00	10.00	3.70	-0.99	-0.98	-0.29	0.09
8	0.021	36.00	837.00	4110.00	22.00	3.82	-0.84	-0.72	-0.04	0.28
9	0.038	75.00	659.50	5030.00	11.50	3.90	-1.27	-1.18	-0.45	0.05
10	0.020	106.00	1060.00	6570.00	10.00	3.84	-0.78	-0.44	0.50	0.10
11	0.014	110.00	307.00	9750.00	10.00	3.84	0.50	0.42	1.11	0.39
12	0.059	30.50	509.50	7000.00	72.00	3.69	0.77	0.62	0.51	0.46
13	0.016	61.00	930.00	3020.00	10.00	3.70	-1.06	-0.01	-0.46	0.02
14	0.013	77.00	961.00	3900.00	50.00	3.80	-1.22	-0.69	-0.33	0.05
15	0.011	32.50	73.00	5455.00	51.00	3.70	-0.83	-0.56	-0.53	0.43
16	0.005	28.00	233.50	5455.00	10.00	3.92	-0.77	-0.05	0.57	0.14
17	0.012	44.50	224.50	5255.00	10.00	3.93	-0.74	-0.73	0.76	0.16
18	0.019	55.00	880.00	4400.00	50.00	3.64	-0.03	0.37	0.11	0.13
19	0.080	229.00	1093.00	6000.00	62.00	3.61	0.50	1.12	0.56	0.25
20	0.045	105.00	622.00	8000.00	48.00	3.63	-0.24	-0.11	-0.23	0.09
21	0.015	38.50	840.00	3895.00	35.00	3.63	-0.35	-0.62	-2.59	0.08
22	0.017	46.00	755.00	3900.00	36.00	3.64	-0.02	0.45	0.48	0.42
23	0.028	57.00	750.00	9590.00	10.00	3.70	0.55	0.46	0.41	0.16
24	0.032	118.00	830.00	10850.00	10.00	3.83	-0.03	-0.08	0.40	0.17
25	0.018	69.00	481.00	18600.00	47.00	3.42	0.77	0.80	0.62	0.42

26	0.025	53.00	693.00	4500.00	10.00	4.06	-1.09	-0.82	0.64	0.17
27	0.036	136.00	318.00	8900.00	100.00	4.25	-1.77	-1.19	-0.73	0.01
28	0.028	60.00	348.00	10410.00	12.00	3.86	-0.37	0.22	0.36	0.15
29	0.021	80.00	184.50	18150.00	34.00	3.95	0.12	0.19	0.73	0.23
30	0.020	74.00	825.00	5800.00	44.00	3.95	-0.81	-0.36	0.31	0.18
31	0.011	46.00	626.00	2370.00	16.00	3.71	-1.39	-0.45	-0.17	0.05
32	0.014	84.00	323.00	19840.00	9.70	3.76	-0.75	-0.49	0.11	0.07
33	0.014	64.00	885.50	3805.00	10.00	4.04	0.08	0.07	0.58	0.05
34	0.011	48.00	491.00	4360.00	36.00	3.90	-0.62	-0.90	-0.01	0.17
35	0.037	119.00	376.00	8490.00	36.00	3.95	-0.71	-1.08	-0.31	0.05
36	0.018	60.00	114.00	3530.00	32.00	3.91	-0.79	-0.65	-0.26	0.10
37	0.014	73.00	391.00	5580.00	36.00	3.94	-0.78	-0.88	-0.62	0.18
38	0.01	61.50	288.00	4220.00	46.50	3.67	-0.54	-0.26	0.53	0.14
39	0.02	68.00	809.00	5060.00	38.00	3.67	-0.76	-0.43	0.57	0.15
40	0.02	108.00	471.00	5280.00	42.00	3.62	-0.53	-0.37	0.62	0.24
41	0.01	32.00	545.00	4070.00	34.00	3.68	-0.84	-0.47	0.31	0.11
42	0.01	27.00	146.00	6210.00	37.00	3.61	-0.61	-0.33	0.40	0.17
43	0.02	70.00	474.00	5290.00	34.00	3.79	-1.17	-0.97	0.68	0.25
44	0.01	34.00	295.00	3620.00	33.00	3.73	-0.92	-0.78	0.68	0.16
45	0.03	155.00	681.00	12800.00	53.00	3.66	-0.21	-0.15	0.50	0.16
46	0.020	67.00	550.50	6095.00	40.00	3.71	-0.74	-0.55	0.21	0.09
47	0.067	147.50	699.50	4300.00	10.00	3.72	-0.79	-0.81	0.56	0.52
48	0.04	89.00	361.50	4015.00	10.00	3.44	0.20	0.48	0.24	0.11
49	0.155	635.50	1245.00	11100.00	10.00	3.65	0.35	-0.56	-0.44	0.06
50	0.027	95.53125	468.969	6074.063	17.1875	3.79	-0.44	-0.38	0.60	0.33

LOI is a positive percent mass loss while GOI is a negative mass loss

B2. Raw data for 50 UG2 samples

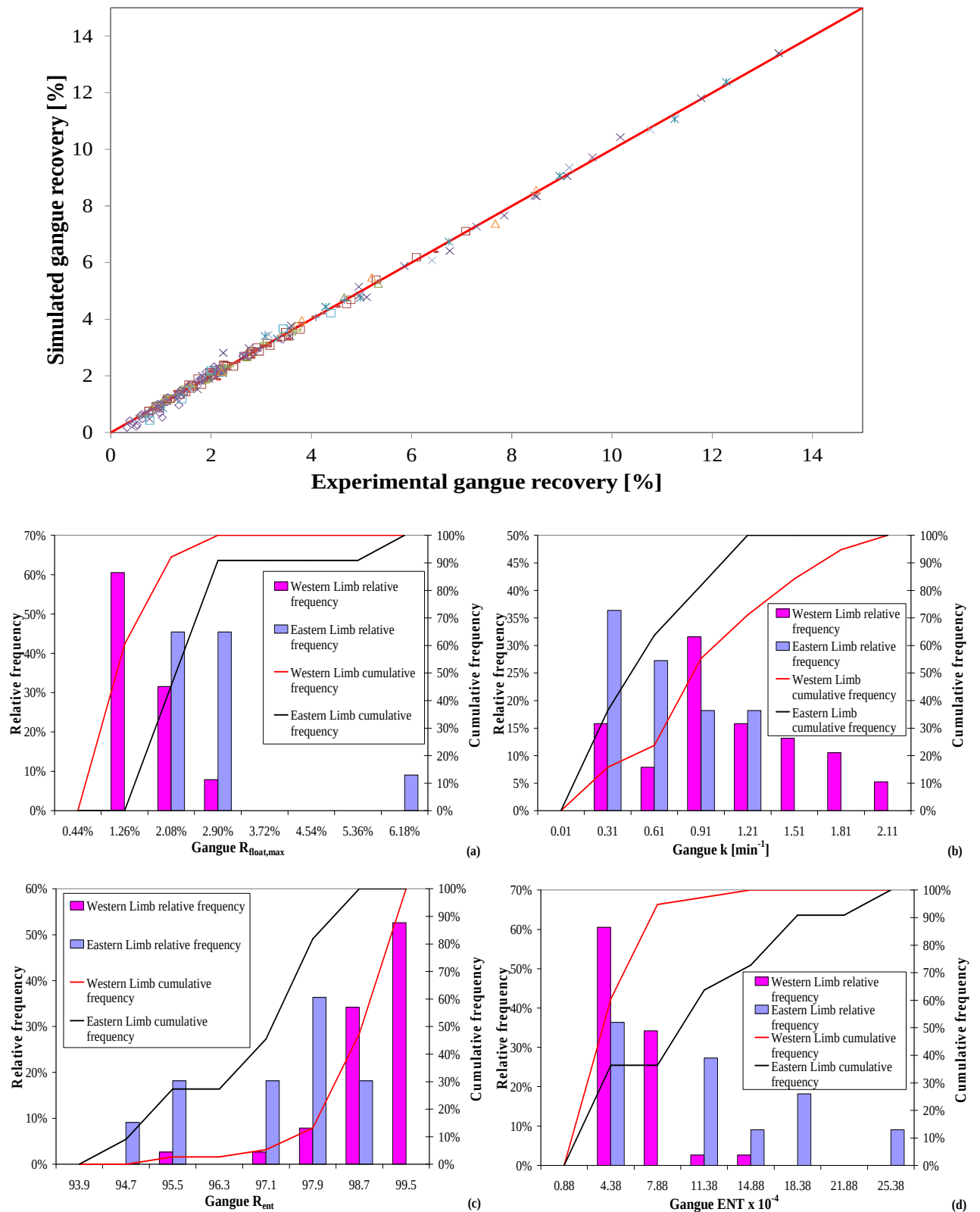
Please refer to Excel file '*50 UG2 samples raw data*' in folder '*Appendix*' on CD. This gives the raw data for the fifty UG2 samples tested in Chapter 4.

Grind estimate based on sizing of rougher tails.

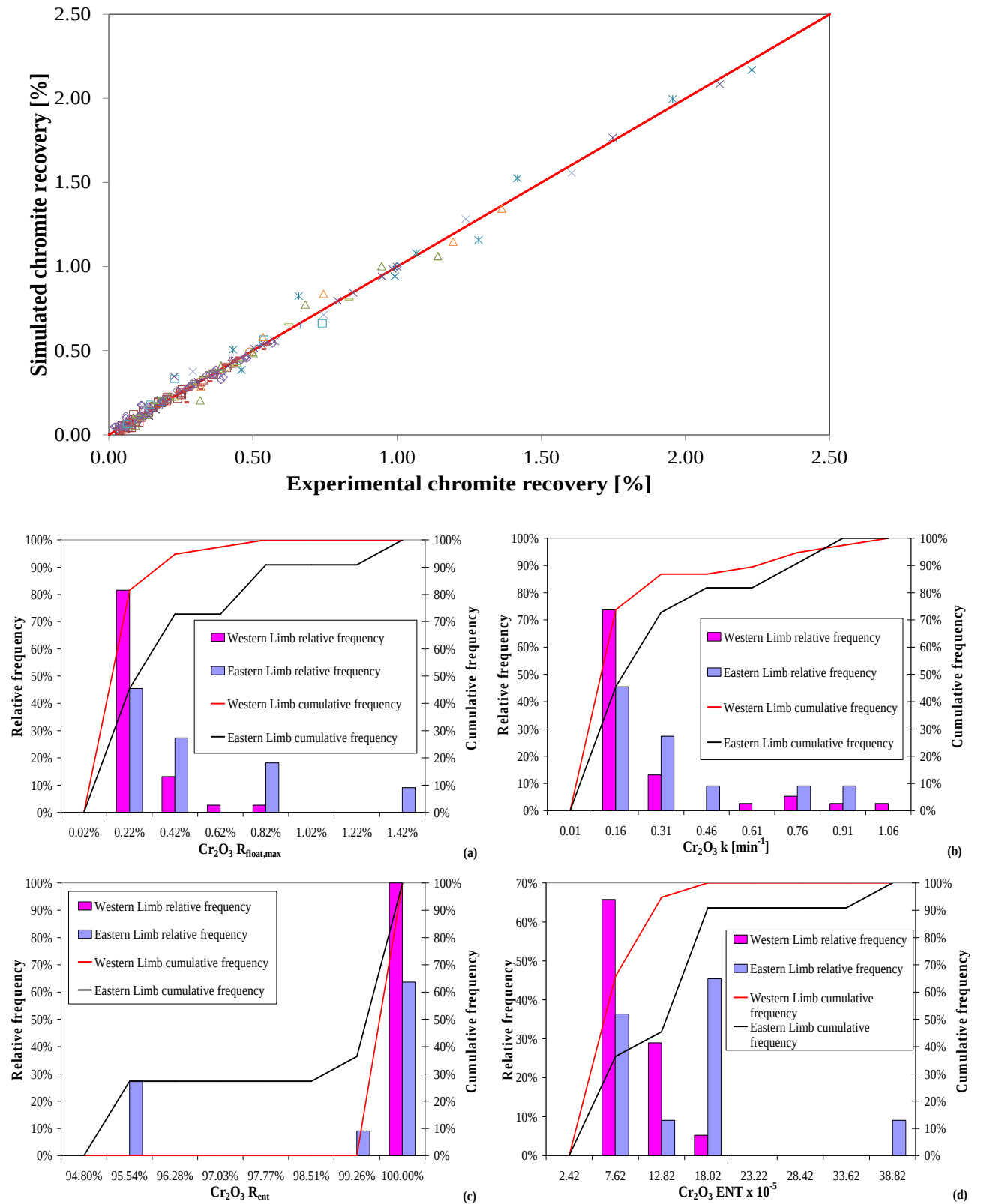
Sample	-75 μ m	Sample	- 75 μ m
1	98%	26	89%
2	78%	27	81%
3	95%	28	86%
4	78%	29	86%
5	86%	30	72%
6	77%	31	83%
7	81%	32	78%
8	77%	33	85%
9	86%	34	93%
10	86%	35	94%
11	88%	36	87%
12	88%	37	88%
13	84%	38	71%
14	81%	39	61%
15	84%	40	67%
16	73%	41	69%
17	75%	42	78%
18	82%	43	63%
19	88%	44	70%
20	88%	45	74%
21	85%	46	69%
22	85%	47	63%
23	95%	48	93%
24	94%	49	82%
25	83%	50	81%

B3. Summary of results for gangue and chromite variability

The quality of the gangue model fit is shown in the parity chart and the histograms show the spread in the model parameters, which describe the gangue flotation.



The quality of the chromite model fit is shown in the parity chart below and the histograms show the spread in the model parameters, which describe chromite flotation.



APPENDIX C

C1. Processed data for rougher and cleaner comparison in Chapter 5

PGM		
Time	Rougher	Cleaner
0	0.00	0.00
1	58.62	54.67
3	70.19	73.02
7	77.95	77.98
20	85.85	81.72
30	88.21	82.88
Gangue		
Time	Rougher	Cleaner
0	0.00	0.00
1	6.35	2.23
3	10.65	5.08
7	14.71	7.81
20	20.47	11.71
30	23.53	13.25
Chromite		
Time	Rougher	Cleaner
0	0.00	0.00
1	1.04	0.23
3	2.04	0.60
7	3.34	1.02
20	6.06	1.81
30	7.90	2.19

C2. Raw data for further batch test work in Chapter 5

Please refer to folder 'Appendix' and Excel spread-sheets '*Sample A-float, Sample B-float, Sample C-float, Sample D-float, Sample E-float, Sample F-float and Sample G-float*' on CD. This gives the raw data for the samples tested in Chapter 5.

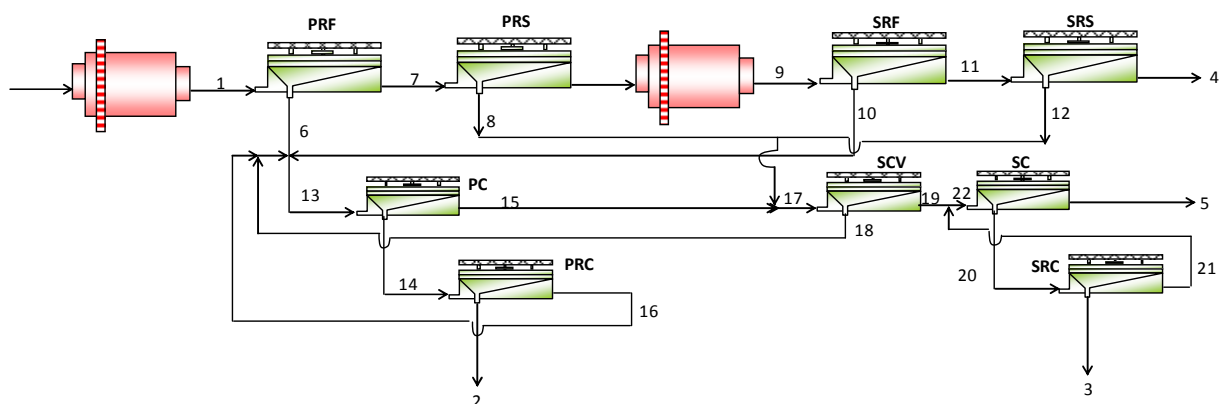
APPENDIX D

D1. Mineralogy raw data

Please refer to folder ‘*Appendix*’ and Excel spread-sheets ‘*Sample A, Sample B, Sample C, Sample D, Sample E, Sample F, Sample G and Sample D-Tail*’ on CD. This gives the mineralogical data used in Chapter 6.

APPENDIX E

E1. Raw data for pilot plant in Chapter 7



Stream	Mass flow/ [kg/hr]		PGE assay [g/t]		Cr2O3 assay [%]		Water/ kg/hr	
	Actual	Smooth	Actual	Smooth	Actual	Smooth	Actual	Smooth
1	699.72	699.72	3.06	3.06	18	18	1199.23	1452.85
2	4.38	4.81	336.35	336.04	2.37	2.4	23.73	23.53
3	0.99	0.89	36.44	36.36	2.94	2.34	7.9	7.92
4	561.1	615.99	0.47	0.48	18.88	18.75	1079	820.63
5	64.24	78.03	2.61	2.54	7.83	7.99	666.01	600.77
6	34.1	38.87	34.25	35.34	4.74	4.8	218.28	204.59
7	675.68	660.85	1.06	1.16	18.3	18.16	1019.19	1248.26
8	11.84	10.73	11.65	10.96	8.41	8.39	113.79	89.62
9	702.39	650.12	0.91	1.00	20.6	18.32	991.12	1158.64
10	3.25	3.96	71.58	64.81	7.66	7.68	112.93	54.75
11	655.19	646.16	0.62	0.61	19.3	18.39	1114.38	1103.89
12	32.53	30.17	3.58	3.29	11	10.93	481.11	283.26
13	64.08	57.1	61.6	49.36	5.31	5.13	702.02	445.04
14	9.69	10.62	210.29	231.12	3.72	3.51	107.03	95.21
15	50.86	46.48	7.83	7.85	5.7	5.51	529.69	349.83
16	4.76	5.81	146.99	144.26	4.27	4.43	65.1	71.68
17	95.6	87.38	6.05	6.68	7.83	7.73	1148.77	722.71
18	9.62	8.46	49.06	41.48	5.86	5.95	98.88	114.02
19	75.98	78.92	3.01	2.92	7.66	7.92	774.08	608.69
20	7.5	6.3	15.81	17.68	4.98	5.14	155.32	146.33
21	4.77	5.41	16.31	14.6	5.79	5.6	131.92	138.41
22	80.75	84.33	3.80	3.67	7.55	7.77	906	747.1

Banks	Bank Vol. [L]
PRF	408
PRS	204
SRF	204
SRS	680
PC	153
PRC	25
CSCV	306
SC	544
SRC	48

bank	Average air flow rate [L/min]	Average froth height [mm]	Length for cell in bank [m]	Width for cell in bank [m]	Cross sectional area of cell in bank [m ²]
PRF	14	95	0.42	0.41	0.17
PRS	27.5	55	0.42	0.41	0.17
SRF	13	65	0.42	0.41	0.17
SRS	37	110	0.42	0.41	0.17
PC	8.5	85	0.30	0.30	0.09
PRC	3	60	0.18	0.15	0.03
CSCV	32	95	0.30	0.30	0.09
SC	10	75	0.42	0.41	0.17
SRC	11	140	0.30	0.30	0.09