INVESTIGATION OF THE EFFECT OF HYDROCARBON SPILLAGES AND THEIR INTERACTION WITH ALTERATION MINERALS ON THE FLOTATION OF UG2 PGE ORE

by

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Synopsis

With growing safety, low productivity and operational cost concerns in the local PGE mining industry, mechanized mining is becoming more frequent to address these challenges and offer easy access to ore bodies compared to conventional mining methods. The heavy-duty machinery used may contaminate the ore because of oil inadvertently leaking from their hydraulic components during mining. Such contaminants have been anecdotally linked to the reduction of downstream flotation performance. Anecdotal evidence suggests that such problems are further exacerbated in the presence of phyllosilicate alteration minerals (serpentine, chlorite and talc) from weathered or altered ores. The near-surface Two Rivers Platinum (TRP) UG2 ore is routinely exposed to hydrocarbon contamination when the underground ore is excavated and transported to surface, resulting in reduced PGE flotation recoveries. Therefore, this study aimed to decouple the effects of oil contaminants on the flotation performance of PGE bearing ores and further establish the interaction mechanism(s) between oil and phyllosilicate alteration minerals.

Two ores of varying alteration degrees (normal and altered UG2) were utilized to assess the effects of oil. Batch flotation tests were conducted to decouple the effect of oil on the flotation performance of the two UG2 ores by varying the oil dosage from 0 to 500 g/t, followed by conducting supplementary experiments in which the key interests were on column froth stability, rheology, and oil adsorption tests. These were performed to understand the mechanism(s) leading to the reduced flotation performance. Quantitative evaluation of minerals by scanning electron microscopy (QEMSCAN) was utilized to characterize the bulk mineralogy as well as PGM. The bulk mineralogy was validated with quantitative X-ray diffraction (QXRD). The last phase on the study involved using sodium metasilicate (1500 g/t) and a degreaser (500 g/t) as a potential mitigation measure to the deleterious effect of oil on flotation performance.

As was expected, the QEMSCAN bulk mineralogy recorded a higher percentage of the phyllosilicate alteration minerals (serpentine, chlorite, and talc) in the altered UG2 (9.1 wt.%) relative to the normal UG2 (6.2 wt.%), consistent with the observations of alteration during sampling. Talc contributed around half of the total phyllosilicate alteration minerals in both ores; however, it was slightly higher in the altered UG2 compared to the normal UG2 (5.8 wt.% versus 3.8 wt.%). Therefore, the altered ore was expected to experience more severe detrimental flotation effects. The results of batch flotation tests indicated a decrease in Pt (~3 g.t) and Pd (~3 g/t) concentrate grades in the normal ore with increasing oil dosage with no significant negative effects on recovery (Pt increased by 4% and Pd was constant). In contrast, the oil resulted in detrimental effects to both Pt and Pd recovery for the altered ore, where a recovery loss of 6% Pt and 12 % Pd were observed. The concentrate grades remained unaffected.

The addition of oil increased the froth stability for both ores (34% in the normal ore and 47% in the altered) as well as the pulp viscosity (including pure talc). The increase in viscosity was greater in the altered ore. The oil adsorption study on the two feeds revealed that most of the oil coats the particles with no preferential adsorption between the two ores (nearly the same concentration). The oil adsorption study on the concentrate and the tails showed no preferential recovery to either the concentrate or tails in the normal ore whereas more selective recovery of the oil-coated particles to the tails was observed for the altered ore.

The observed decrease in grade in the normal ore due to oil contamination was attributed to oil improving the froth stability which caused excess entrainment of gangue materials and thereby diluting the concentrate grade. The decrease in recovery in the altered ore was ascribed to oil forming particle agglomerates (particularly complex talc agglomerates) which increased the pulp viscosity and subsequently resulting in poor gas dispersion and reduced particle-bubble collision. After understanding the mechanisms in each ore, batch flotation with sodium metasilicate and a degreaser were conducted in the presence of oil, as a potential mitigation measure.

Sodium metasilicate (SS) improved the grades in the normal ore floats and had no detrimental effect toward the recovery (constant recoveries, no loss in Pt and Pd). For the altered ore, both the grade and recovery (Pt-8% and Pd-5%) were improved by the adding of SS. The effects on the grade were most noticeable at low oil concentration (0 and 300 compared to 500 g/t). For the degreaser, only the flotation experiments with the degreaser were performed due to laboratory technical problems-new flotation cell was used. Thus, there was no comparison with the no degreaser condition. The trends (% difference between 0 g/t and 500 g/t) were compared to the no degreaser (oil only) conditions conducted in the first cell. The degreaser was shown to improve 2E grade and recovery in both ores. However, the improvement in flotation performance was unquantified.

The deleterious effects were successfully mitigated when using sodium metasilicate due to its beneficial effects in reducing pulp viscosity as well as improving the froth drainage. With the degreaser, the flotation performance was not quantified. Therefore, this study demonstrated that, SS can be implemented to mitigate the viscous effects caused by oil spillages and alteration minerals (particle agglomeration) in mechanized mines. However, SS should not be implemented blindly as other downstream processes should be considered (settling in the thickener). To implement the degreaser, more tests need to be conducted.

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List of abbreviations and definitions

ARM	African Rainbow Minerals
BMS	Base metal sulphides
CMR	Centre for Mineral Research
CeBER	Centre for Bioprocess Engineering Research
2E	Platinum and Palladium
GSD	Grain size distribution
ICP-OES	Inductively Coupled Plasma Optical Emission Spectroscopy
g/t	gram per ton
Liberated	PGM, particle area is over 80%
km	Kilometre
PGE	Platinum group elements
PGM	Platinum group minerals
Pyroxenite	An igneous rock consisting primarily of pyroxene minerals. clinopyroxene and orthopyroxene
Norite	An igneous rock that primarily consist of plagioclase, olivine, and orthopyroxene
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy
SS	Sodium metasilicate
TOC	Total Organic Carbon
UG2	Upper Group 2
Vol.%	Volume percentage
Wt.%	Weight percentage

1. Introduction

1.1. Background

Platinum Group Elements (PGE) have unique characteristics that render them useful in a wide range of industrial applications and chemical processes; this includes their higher densities, thermal conductivities, melting points, catalytic reaction abilities, and corrosion resistance (Thormann et al., 2017). These elements are widely used in the automotive industry in catalytic converters, glass manufacturing industry, jewellery, medical and the petroleum industry, among others. The six PGE are platinum, palladium, rhodium, osmium, iridium and ruthenium. The primary deposits where these elements are extracted are the magmatic PGE-Ni-Cu deposits of the Bushveld Igneous Complex (BIC) in South Africa, the Great Dyke in Zimbabwe, the Stillwater Complex in the United States of America, and the Ni-Cu dominant PGE magmatic sulphide deposits in the Norilsk-Talnakh Complex in Russia, and the Sudbury Basin in Canada (Godel, Maier & Barnes, 2008; Thormann et al., 2017). It is estimated that the BIC holds over 80% of all known PGE reserves in the world (Mudd, Jowitt & Werner, 2018).

In the BIC, mechanized mining has emerged as the preferred method of ore extraction, especially in the new mines (Rupprecht, 2018; Willis et al., 2004). 2004). This is due to the constant orebody thickness and other benefits it offers compared to conventional mining methods, which include increased productivity, safety, profit, small mining crew, and quick orebody access (Pickering, 1995; Willis et al., 2004). During ore extraction, various trackless mobile machinery (e.g load-haul-dump, drill rigs and front loaders) are employed, mostly in the narrow reefs (Willis et al., 2004). These mining machines employ hydraulic systems to accomplish a variety of tasks such as mucking, drilling, and ore sorting. Oil may inadvertently leak from the hydraulic system as well as the engine during mining activities, likely due to piping failures, oil ageing (making it less viscous), and inadequate maintenance, further contaminating the ore (Mavhungu, Nheta & Rose, 2022). This may subsequently impact the downstream flotation performance in the concentrator plant. According to Bos and Quast (2000), hydrocarbons such as hydraulic oil, degreaser, and engine oil may impact the flotation of sulphide ores negatively.

Many near-surface PGE ores are exposed to weathering in the form of hydration reactions (Guilbert & Park, 2007). This may result in the formation of alteration minerals such as talc, serpentine, chlorite, and magnetite, which are amongst the most common secondary minerals in the BIC rocks (Schouwstra, Kinloch & Lee, 2000). Moreover, some of these minerals may be formed at deeper levels through hydrothermal alteration. Alteration minerals are widely known to harmfully affect the PGE flotation recovery and concentrate grade (Dzingai et al., 2021). Anecdotal observations from the Two Rivers Platinum Mine in South Africa, have also linked the presence of altered phyllosilicates to the exacerbation of poor flotation performance from oil contamination. Due to the design of some mechanized mines, oil contamination may unavoidably perpetuate throughout the mining, crushing and grinding circuits, leading to downstream flotation problems in the concentrator plant.

It is also of importance to note that hydrocarbons (oils) are widely used as collectors in the flotation of minerals such as coal, graphite, and molybdenite (Lin et al., 2018; Polat, Polat & Chander, 2003; Vasumathi et al., 2015; Xia et al., 2019); however, minimal attention has been directed towards studying their effects on the flotation of PGE ores. Thus, a comprehensive understanding of the mechanism (s) leading to poor flotation performance is required to identify an effective mitigation strategy or solution.

1.2. Problem statement

There is a gap in the literature to uncover the effects of oil on the flotation performance of PGE ores, since no attention has been given to this. Oil contamination affects the flotation performance of PGE ores, and the presence of phyllosilicate alteration minerals, mainly in the near-surface ores, may exacerbate this. Thus, there is a need to understand the interaction mechanism between hydrocarbons from undesired spillages and phyllosilicate alteration minerals and how this affects the flotation of the valuable species in the PGE ores, as this will likely occur in mechanized mines.

1.3. Objectives

The overarching objective of this study is to investigate the effect of hydrocarbon contamination on the flotation of Upper Group 2 chromitite (UG2) ore and to establish the mechanism(s) by which the flotation performance is compromised. Ultimately, understanding the mechanism(s) will enable the development of possible mitigation strategies.

More detailed objectives are as follows:

- Determine the critical concentration where the flotation performance of UG2 ore is compromised through hydrocarbon contamination.
- Determine why and how hydrocarbon contamination compromises the flotation performance of PGE ores. This includes investigating the effects of hydrocarbon contamination on the ore rheology and mineral hydrophobicity, and the interaction between hydrocarbon, phyllosilicate alteration minerals and valuable minerals.
- Identify any potential mitigation method or solution to these effects.

1.4. Project Scope and limitations

This project employs a case study approach using ore from the Two Rivers Platinum (TRP) mine. The study intends to understand the interaction mechanism between hydrocarbon contaminants and phyllosilicate alteration minerals, and how this leads to poor flotation recovery and grade of the UG2 ore. Laboratory equipment and various analysis methods will be used to better understand the problem at TRP. The flotation experimental work will be set to mimic conditions at the TRP concentrator plant (grinds, reagent suite, and dosages) using a laboratory-modified 8L Leeds batch flotation cell. Two UG2 ores (normal and altered) of varying alteration degrees will be intentionally contaminated at varying hydrocarbon concentrations, followed by batch flotation test work. Mineralogical characterisation of the normal and altered ore will be performed with Quantitative Evaluation of Minerals by Scanning

Electron Microscopy (QEMSCAN) to determine the bulk mineralogy, PGE distribution, liberation, and association.

Quantitative X-ray diffraction (XRD) and X-ray fluorescence spectrometry (XRF) will be utilized to quantify the bulk mineralogy and validate the back-calculated elemental compositions with the measured compositions. By comparing the flotation response to varying oil concentrations and the mineralogy of the two ores, the effects of oil will be decoupled. Supporting experiments will be used to develop an understanding of the effect of oil on the ore rheology as well as the mineral hydrophobicity. Froth stability and slurry rheology (shear stress, yield stress, and apparent viscosity) studies employing two ores and at different oil dosages are among the ore rheology investigations. The hydrophobicity oil studies on the feeds and concentrates will determine if the oil is coating the particle surfaces and if all oils are recovered in the froth phase along with the concentrates or reporting to the tails. Total organic carbon analysis and oil extraction with petroleum ether will be used to study the oil adsorption on the particle surfaces. Both methods will be used in the feeds and on the concentrates, whereas the solvent extraction method will only be performed on the selected oil concentrates (with oil).

This study solely focuses on understanding the mechanism(s) leading to a decrease in UG2 flotation recovery and grade, and from which possible mitigation strategies may be explored with no assurance that they will be successful. Optimisation of any of these strategies also falls beyond the scope of this project. Although mine sludge contaminated with hydrocarbons is known to be a challenge at some sites, this is beyond the scope of this study. As part of this study, the source of hydrocarbon contamination will be identified based only on the information obtained from a mine visit and will not be thoroughly examined.

1.5. Thesis outline

The first chapter is the introduction, which focuses on the study background, problem statement, objectives, scope and limitations. This is followed by the second chapter, which critically reviews the relevant literature. The study hypotheses and research questions are covered at the end of the second chapter after critically reviewing the relevant literature. The methodology is outlined in the third chapter, which addresses the objectives and key questions. The results and discussion are covered in chapters four and five, respectively. Chapter six presents the study's conclusions and recommendations for future work. The Appendix comprises further details of selected measurements as well as the raw datasets.

2. Literature review

2.1. The Bushveld Igneous Complex

The Bushveld Igneous Complex located in South Africa is the world's largest layered intrusion covering an area of approximately 90 000 km^2 with an average thickness of about 5 km (Barnes & Maier, 2002; Finn et al., 2015). Over 80% of the worlds PGM reserves are located within the Bushveld Igneous Complex (Mudd, Jowitt & Werner, 2018). Due to its enormous area and uniqueness, most layered igneous intrusions are compared to it. The BIC comprises four major rock units, the Rustenburg Layered Suite (RLS), Lebowa Granite Suite (LGS), Rashoop Granopyhre Suite (RGS), and Rooiberg Groups (Hatton & Schweitzer, 1995). The Rustenburg Layered Suite is the most economically important suite, and it hosts the main primary sources of PGE: i) the Upper Group 2 chromitite reef (UG2), ii) the Merensky reef, and iii) the Platreef (Schouwstra, Kinloch & Lee, 2000). These three ore bodies are spread across different limbs of the BIC. The UG2 and Merensky reefs occur in the western and eastern limbs, whereas the Platreef is only found in the northern limb. Unlike the UG2 and Merensky reefs, where the PGE mineralization is primarily the sulphides, the Platreef has a complex mineralization consisting mainly of tellurides, arsenides, and sulphides (Schouwstra, Kinloch & Lee, 2000). The ore of interest to this project is the eastern limb UG2.

2.1.1. UG2 chromitite reef

The UG2 is a platiniferous chromitite layer that consists mainly of chromite (60-90 vol%), pyroxene (5-30 vol%) and plagioclase (1-10 vol%), with the percentages varying depending on the locality within the Bushveld Igneous Complex (Barnes & Maier, 2002; Vermaak, 1995). It is found 40 to 400 m beneath the Merensky reef (Cawthorn, 1999). The chromitite layers are found within the critical zone which further subdivide into three stratigraphic groups: Lower Group (LG), Middle Group (MG) and Upper Group (UG) (Eales & Cawthorn, 1996). The UG2 is the second layer within the Upper Group zone.

The PGE mineralisation in the UG2 predominately comprises PGE sulphides, namely: laurite, braggite, cooperite and an unnamed PtRhCuS (Schouwstra, Kinloch & Lee, 2000). These platinum group minerals (PGM) normally occur in association with the base metal sulphides (BMS) such as chalcopyrite, pentlandite, pyrrhotite and pyrite. Moreover, they also occur in association with chromite and silicate minerals (Schouwstra, Kinloch & Lee, 2000). Other minerals which are generally present in small amount in the UG2 ores apart from the BMS are the secondary minerals such as talc, serpentine and chlorite; silicates such as phlogopite and biotite, and oxides (ilmenite and rutile) (Penberthy, Oosthuyzen & Merkle, 2000; Schouwstra, Kinloch & Lee, 2000). A typical UG2 composition is given in Table 2-1.

The 6E head grade in the UG2 reef varies depending on the location and mining cut; however, the average is between 4 and 7 g/t (Schouwstra, Kinloch & Lee, 2000). The two main precious metal elements in the BIC reefs are platinum (Pt) and palladium (Pd), and the ratio Pt:Pd is used to determine their distribution within the reef. The western limb Pt:Pd ratio is double the eastern limb UG2, 2.2 compared to 1.1 (Schouwstra, Kinloch & Lee, 2000). PGE that do not occur in discrete PGM may be hosted via solid solution substitution, with pentlandite known to potentially host up to 55% of the Pd budget in the UG2 (due to the substitution of Pd for Ni) (Osbahr et al., 2014).

Table 2-1: The general composition of UG2 ores. Adapted from Corin et al. (2021).

Mineral	Volume %
Chromite	50-75
Pyroxene	15-30
Feldspar	3-9
Talc	<1
Serpentine	1
Chlorite	<1
Amphibole	<1
Mica	<1
BMS	<1
Other*	<1

**Other refers to other minor silicates, iron-oxides, and carbonates.*

2.1.2. Mineral processing of PGM from UG2 ore

PGM grain sizes are generally fine to very fine, with most falling below 10 µm (Penberthy, Oosthuyzen & Merkle, 2000). According to Schouwstra, Kinloch and Lee (2000), the average PGM grain size in UG2 ores is about 12 µm, with sizes larger than 30 µm being rare. The recovery of PGM is through the process of froth flotation, where particles are separated based on the difference in their surface properties. Due to their very fine grain size, PGM require very fine grinding to be liberated prior to flotation. However, fine grinding of UG2 ore comes with its challenges. UG2 reef contains a substantial amount of chromite which may be recovered via entrainment to the concentrate as a result of very fine grinding. Chromite ($FeO \cdot Cr_2O_3$) is problematic in the downstream smelting process. The chromite consists of around 40-50% chromium oxide (Cr_2O_3), and the chromium oxide in the concentrate needs to be limited to 3% to avoid or minimise build-up of chromite spinel in the furnace during smelting (Hay & Roy, 2010). The chromite build-up reduces the operational volume of the furnace by about one-third, which may result in slag losses. The slag may still have an appreciable amount of PGMs. To overcome the entrainment problem, mines utilize flow sheets such as MF2 (two stages of grinding and flotation) and MF3 (three stages of grinding and flotation) circuits. A typical circuit used in the processing of UG2 ore is the MF2 shown in Figure 2-1, usually with the addition of another cleaning stage. Additionally, fine grinding of PGM may also result in the liberation of fine-grained alteration mineral, which if not controlled may adversely affect the flotation grade and recovery (Becker et al., 2009; Dzingai et al., 2021; Ndlovu, Farrokhpay & Bradshaw, 2013).

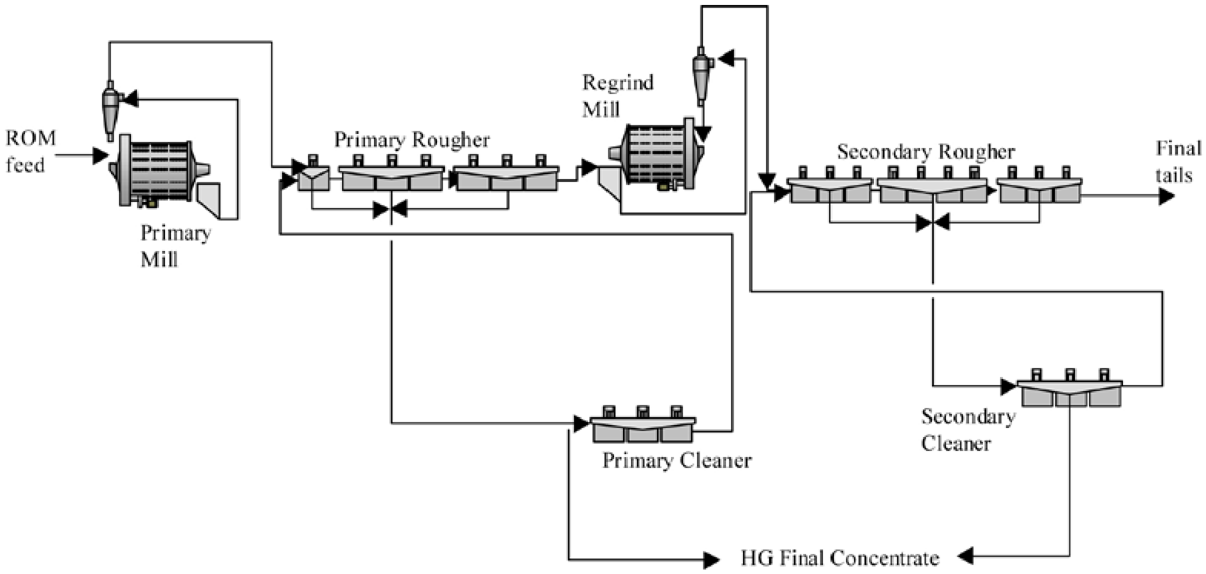

Figure 2-1: Standard MF2 flotation circuit (Nel, Martin & Raabe, 2004).

2.2. Case study: Two Rivers Platinum mine (TRP)

TRP is located on the Dwarsrivier farm, which is about 65 km northwest of Lydenburg in Mpumalanga, South Africa. Implats owns 46% of the operation whereas African Rainbow Minerals (ARM) owns the remaining stake (54%) (ARM MRR report, 2022). The mine is situated on the southern lobe of the eastern limb of the BIC, shown in Figure 2-2. The mine contains both the UG2 chromitite reef and the Merensky reef. At the time of writing, only the UG2 reef (both normal and multiple split reefs) was being exploited for recovery of PGM and chromite. Mining or extraction of the UG2 reef is through two shaft systems, the north and main decline, located about three kilometres apart. The mine has an on-site crusher plant, PGM concentrator plant and a small chromite plant. The PGMs are recovered through froth flotation, whereas the chromite is recovered as a by-product in the chromite plant with gravity concentration. The flowsheet in the PGM concentrator plant has three stages of grinding and is divided into high-grade and low-grade sections. After the initial processing phase, the concentrates are transported to Impala Refining services for further smelting and refining of PGE.

TRP is a fully mechanized mine that employs machines such as the Atlas Copco S1L Rocket Boomer for drilling, load-haul-dump (LHD) for mucking, and front-loaders (Mabuza, 2007). Additionally, conveyor belts are used to assist with moving the mined ore to the surface. The mining activities or extraction of UG2 reef at TRP is through a bord and pillar layout. Each mining site is made up of eight 12 m rooms with the support pillar (6 x 6 m) increasing in size (area) as the depth below the ground increases (Mabuza, 2007).

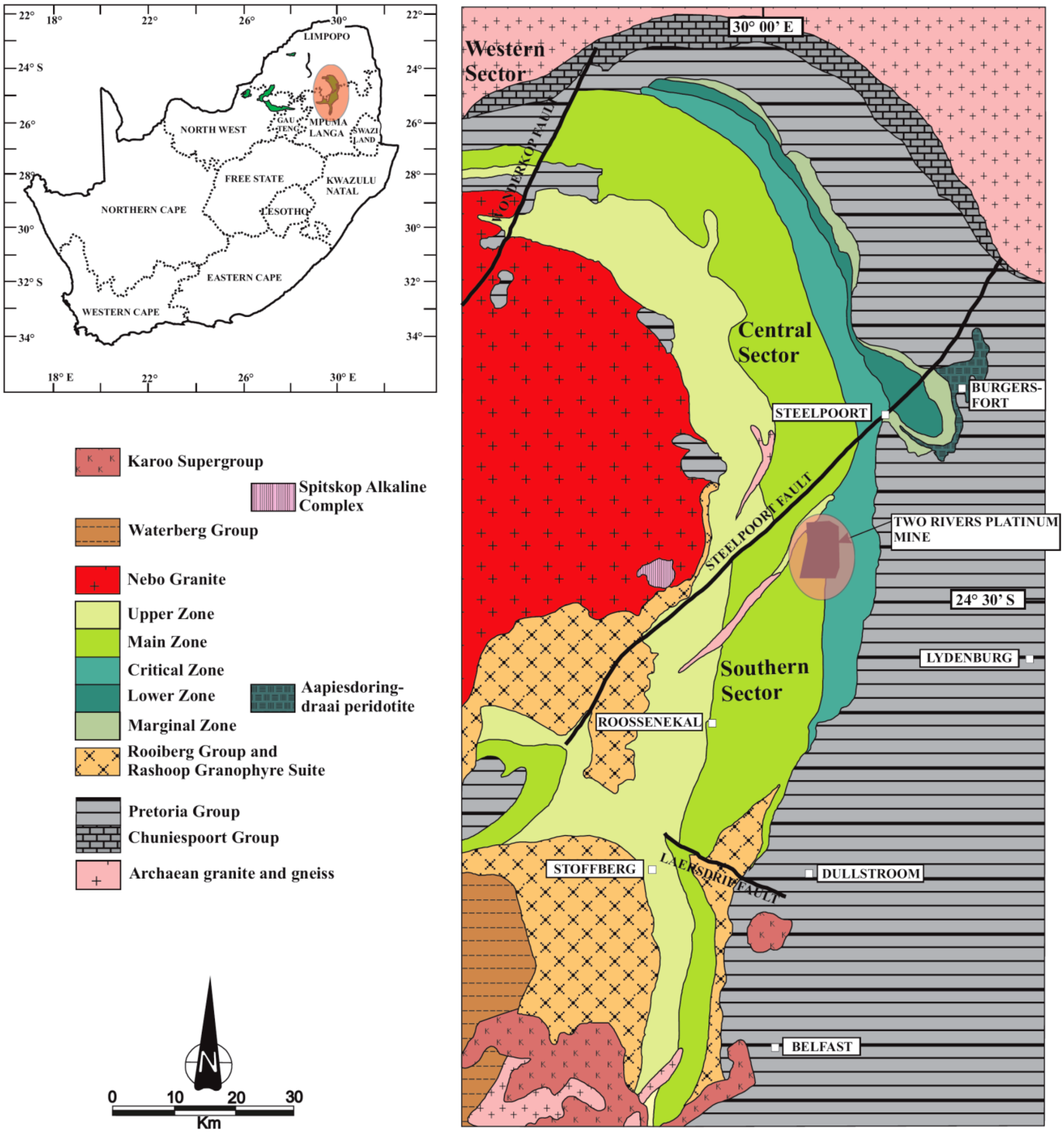

Figure 2-2: Location of TRP in the Eastern Limb (Rose, 2016).

2.2.1. UG2 chromitite reef at TRP

The UG2 at TRP primarily consists of medium-sized chromite grains along with orthopyroxene and plagioclase (Mabuza, 2007). It is overlain by the Merensky by a vertical distance of around 140 m. The thickness of the UG2 varies between 1.2 and 1.4 m, with an average thickness of around 1.2 m. Two different UG2 reef types have been identified at TRP: normal reef and split reef. The former is a typical chromitite reef, whereas the latter is a typical reef that has been subdivided into multiple reefs by the norite (internal pyroxenite/dilutions) as illustrated in Figure 2-3. High levels of alteration minerals associated with the pyroxenite are common in the split reef regions. The internal dilution thickness varies between 40 and 192 cm, depending on the locality (Rose, 2016). The footwall of the normal UG2 reef is the pegmatoidal pyroxenite, with a thickness of about 25 cm and the hanging wall (melanorite) has an average thickness of 20 cm. The PGE mineralisation occurs predominantly as the PGE sulphides in association with chromite as well as trace amount of BMS: chalcopyrite, pentlandite and pyrrhotite. The average historical UG2 PGE head grade (4E) at TRP is 4.22 g/t, with a peak at the reef's base (Mabuza, 2007). At the time of writing, the UG2 4E head grade at TRP was around 3 g/t (ARM MRR report, 2022)

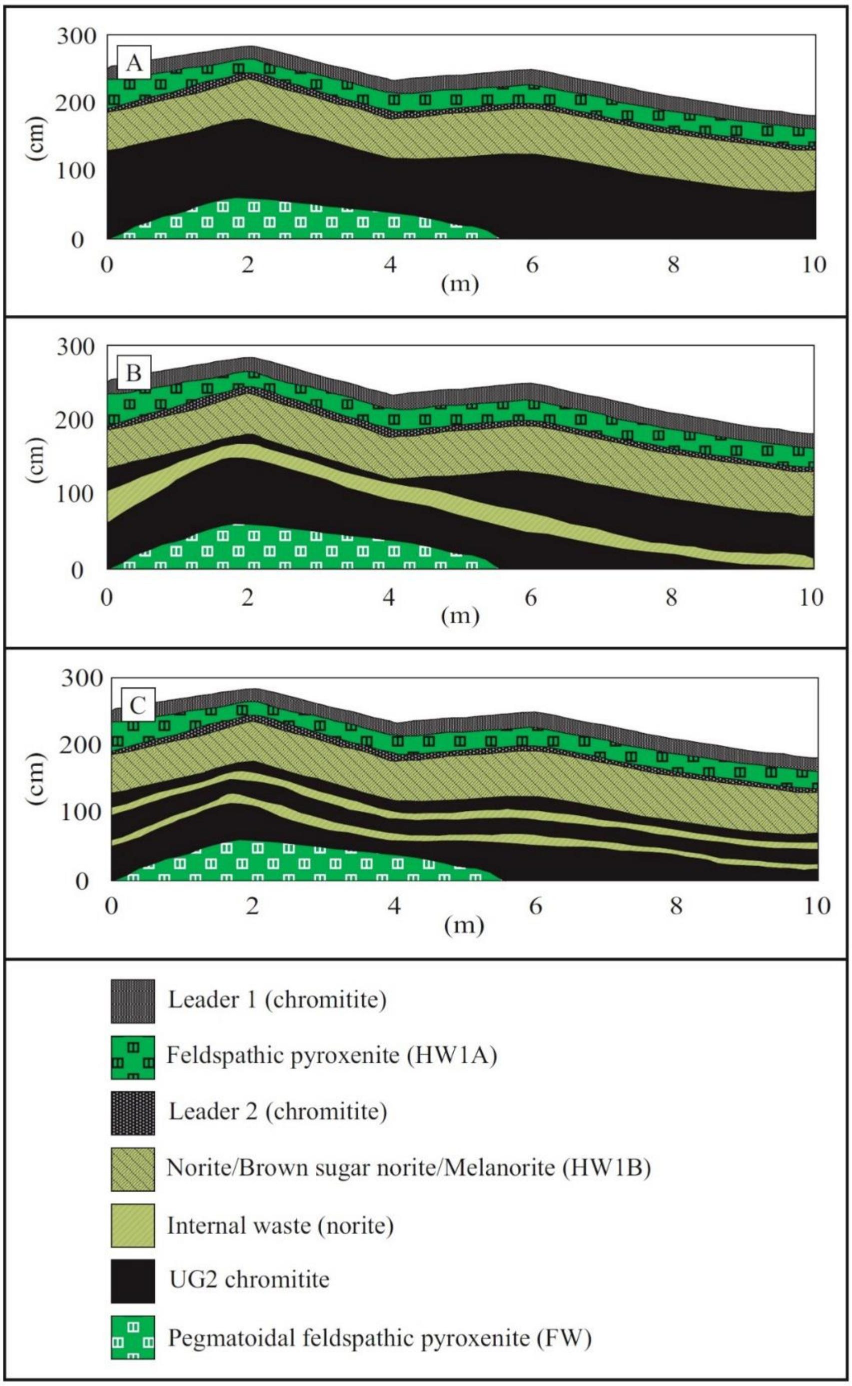

Figure 2-3: Different UG2 reef types identified at TRP. The underground mapping of various mining cuts was conducted by Rose (2016). A is the normal UG2 reef, B is the split reef and C is the multiple split reef type.

2.3. Phyllosilicate minerals in PGM ores

With the increased need to mine and process low-grade ores, it is of great importance to understand the gangue mineralogy, as a way to predict problems and help improve the recovery of valuable minerals (Ndlovu, Farrokhpay & Bradshaw, 2013). Among these gangue minerals are the phyllosilicates, these minerals are generally formed due to weathering/alteration reactions of PGM ores, especially those near-surface ores. They include minerals such as talc, chlorite, serpentine, and clays (kaolinite, montmorillonite, smectite and others). Phyllosilicates are a group of silicate minerals that are composed of different ratios of tetrahedral (T) and octahedral (O) sheets. The varying ratio/proportion of these layers result in minerals that have similar structures, but different chemical and physical properties. The tetrahedral layer consists of silica $(SiO_4)^{4-}$ units, where four oxygen atoms are aligned symmetrically around the silica cation. cation. The octahedral layer consists of a core metal cation that is surrounded by six hydroxyl groups. The octahedral layer can take a brucite structure ($Mg(OH)_2$) if the cation is Mg^{2+} or gibbsite ($Al(OH)_3$) in Al^{3+} cation. The name "phyllosilicate" is derived from the Greek word "phyllo" which means leaf; hence, phyllosilicate minerals generally have a platy sheet-like morphology consisting of faces and edges (Nadziakiewicza, Kehoe & Micek, 2019). Chemical structures of various phyllosilicate minerals are shown in Figure 2-4.

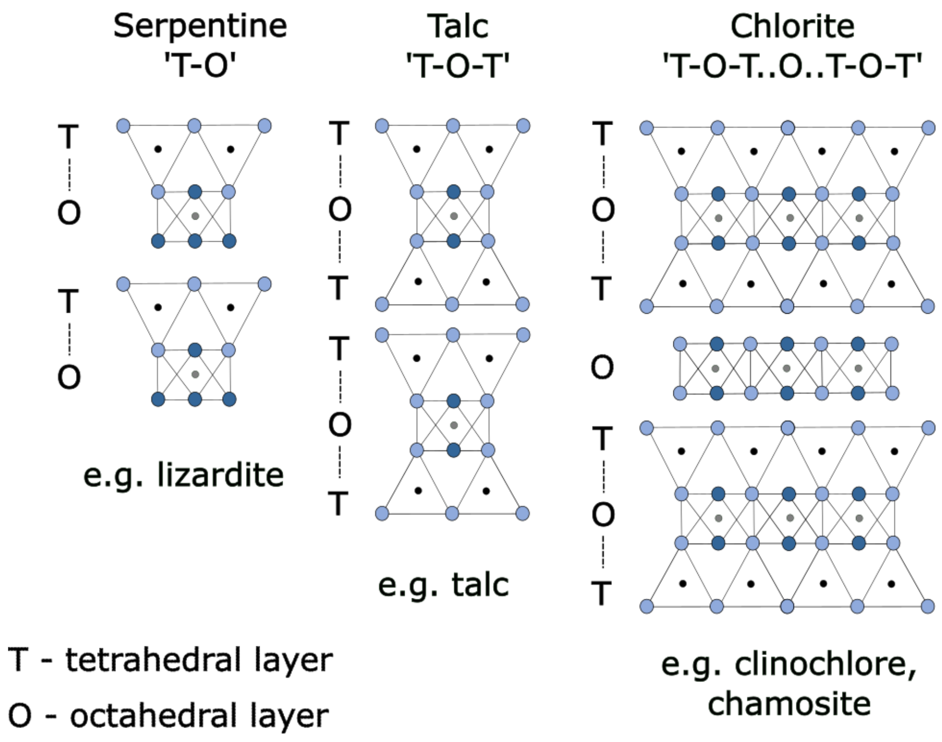

Figure 2-4: Structure of various phyllosilicate minerals. Adapted from Ndlovu et al. (2014).

The surface charge may differ across the edges and the faces in which case the mineral may be known as anisotropic. It is broadly accepted that the face of phyllosilicate minerals carries a permanent negative charge, as a result of the isomorphous substitution between higher valence ions of Si^{+4} and lower valence ions valence ions of Al^{+3} (Chen, Xumeng & Peng, 2018; Ndlovu et al., 2014; Xu et al., 2018). The charge on the edges of the sheets can be positive or negative depending on the pH, due to the interaction of exposed aluminol (Al–OH) and silanol (Si–OH) amphoteric group with hydrogen or hydroxyl ions in the aqueous phase (Ndlovu et al., 2014).

These minerals have widespread occurrence in a number of different rock types. In the Bushveld complex, the most common phyllosilicate minerals are talc, serpentine, chlorite and mica (Schouwstra, Kinloch & Lee, 2000). Talc is known to be the most problematic and common form of natural floating gangue mineral (NFG), which may adversely dilute the concentrate grade due to its naturally hydrophobic nature (Becker et al., 2009; Dzingai et al., 2021). In addition, the strong association between talc and orthopyroxene renders composite particles of talc and orthopyroxene hydrophobic which may further reduce the concentrate grade if not controlled (Becker et al., 2009).

Fine-grained alteration phyllosilicate minerals are also widely known to cause slime coating (Chen, Xumeng & Peng, 2018; Molifie et al., 2023; Yu et al., 2017). This occurs when the valuable minerals and the phyllosilicates carry opposite charges resulting in electrostatic attraction. Slime coating inhibits the collector adsorption on the surface of valuable minerals which ultimately affects the recovery. For example, the low recovery of valuable species because of serpentine slime coating (Bremmell, Fornasiero & Ralston, 2005; Molifie, 2021). In addition, fine liberated phyllosilicates such as talc and serpentine are widely known to have complex rheology, which may result in poor recovery (Burdukova et al., 2008; Shabalala et al., 2011).

2.3.1. Alteration reactions in PGM-bearing ores

There are various types of reactions that alter the ore mineral assemblage. These reactions include and are not limited to hydration, oxidation, hydrolysis, silicification, decarbonation, and sulfidation (Guilbert & Park, 2007). For the interest of this study, only hydration will be addressed. This is because the near-surface ores at TRP are more prone to it; this may lead to the formation of secondary minerals (alteration phyllosilicates) which are known to be detrimental to flotation. Hydration reactions occur when water is added to certain silicate minerals, resulting in phyllosilicate alteration minerals. Some examples of these reactions include the formation of serpentine from olivine in equation 2-1, chlorite from feldspar in equation 2-2 (Eggleton & Boland, 1982; Guilbert & Park, 2007), serpentine from pyroxene in equation 2-3 (Rouméjon & Cannat, 2014:726), and talc from orthopyroxene in equation 2-4 (Iyer et al., 2008:80). Other examples include the formation of montmorillonite and chlorite from amphiboles and biotites. The hydration of minerals also occurs in deep mining sites through hydrothermal alteration as is the case of the Great Dyke PGM ores (Li et al., 2008). In this reaction, the hydrothermal fluids react with minerals. The extent of this alteration increases with increasing ore depth as opposed to weathering.

$$2Mg_2SiO_4 + 2H_2O + 2H^+ \leftrightarrow Mg_3Si_2O_5(OH)_4 + Mg^{+2} \tag{2-1}$$

Olivine Serpentine

$$KAlSi_3O_8 + 10H_2O + 6.5Mg^{+2} \leftrightarrow Mg_{6.5}(Si_3Al)O_{10}(OH)_8 + 12H^+ + K^+ \tag{2-2}$$

K-feldspar Chlorite

$$3(Mg,Fe)SiO_3 + 2H_2O \rightarrow Mg_3Si_2O_5(OH)_4 + SiO_2\,(aq) \tag{2-3}$$

Pyroxene Serpentine

$$5Mg_2Si_2O_6 + 2H_2O \rightarrow 2Mg_3Si_4O_{10}(OH)_2 + 2Mg_2SiO_4 \tag{2-4}$$

Orthopyroxene Talc Olivine

It is also worth noting that the ore alteration as a result of hydration reaction may occur simultaneously with other reactions. One of the examples is outlined in equation 2-1, where it occurs concurrently with hydrolysis (addition of H^+ to the ore). Moreover, oxidation may occur in the near-surface ore due to oxygen exposure. Secondary sulphide minerals such as violarite, chalcocite, covellite, oxides, and sulphate are formed during the oxidation of BMS in Ni-Cu sulphide ores (Guilbert & Park, 2007). Oxidation leads to the formation of secondary minerals such as kaolinite. Serpentine can also be formed from the oxidation of olivine in the presence of hydrogen ions.

2.4. Froth flotation

Froth flotation (Figure 2-5) is a widely used physicochemical process that separates minerals based on the surface property differences between valuable minerals and gangue minerals (Wills & Finch, 2015). It enables the recovery of valuable minerals from the complex ores and low-grade ores. Moreover, it is widely used to beneficiate the sulphide ores. The recovery of minerals in froth flotation is through three different mechanisms: i) true flotation, ii) entrainment and iii) entrapment. Of these mechanisms, true flotation is the means by which hydrophobic minerals selectively attach to air bubbles with the aid of reagents and are transported from the pulp phase to the froth phase for recovery. Entrainment is a non-selective mechanism where mineral particles are recovered along with water. The amount of entrained gangue recovered in flotation is directly proportional to the water recovery; therefore, the lower the recovery of water, the less the entrained gangue minerals and the higher the grade (Engelbrecht & Woodburn, 1975; Neethling & Cilliers, 2009; Zheng, Johnson & Franzidis, 2006). Some of the factors that affect particle entrainment include slurry and froth viscosity, particle size, froth characteristics (bubble size, froth height, stability and others) and density differences between solids and liquids (Yianatos & Contreras, 2010). In entrapment, gangue minerals are sandwiched between air bubbles.

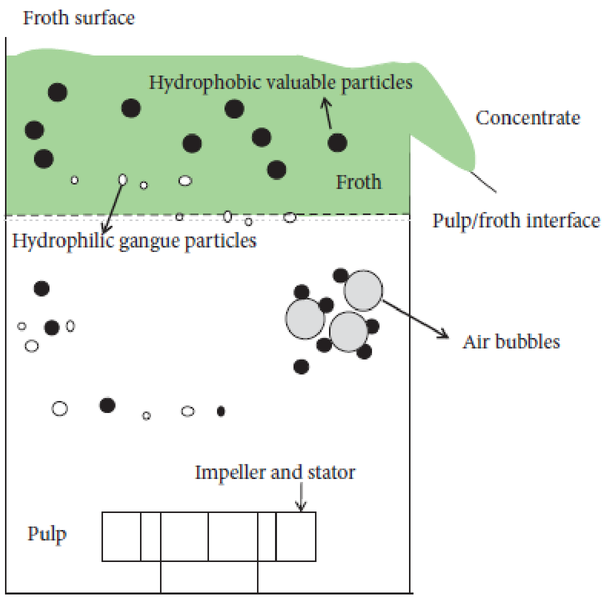

Figure 2-5: Froth flotation process (Wang, Lei & Li, 2020).

Froth flotation can either be direct or reverse (Wills & Finch, 2015). In direct flotation, valuable minerals report to the concentrate, as is the case with PGM processing. In contrast, reverse flotation is where valuable minerals are recovered in the pulp product and gangue minerals in the froth product.

2.4.1. Pulp Phase

The pulp phase is a slurry-dominated zone where bubble-particle collisions occur to form aggregates that are later recovered in the froth zone. The air introduced in the flotation cell acts as carrier of hydrophobic particles. The interactions are enhanced by the introduction of reagents such as collectors and activators. Amongst other factors, rheology is one of the pulp phase essential elements, along with pH, pulp potential (Eh), and ionic strength (Sheni, Corin & Wiese, 2018). The effects of rheology on the pulp phase will be covered in Section 2.4.4.

2.4.2. Froth Phase

During flotation, the hydrophobic minerals in the pulp phase are attached to the air bubbles, which transport them to the surface (froth zone). It is here where the valuable minerals are recovered as concentrates. The role of the froth phase in flotation is to limit or reduce the recovery of unwanted minerals to the concentrate. The froth phase is the final stage of flotation; therefore, it determines the overall flotation performance (grade and recovery) (Ekmekci et al., 2006). The drainage of entrained gangue minerals from the froth phase occurs as a result of differences in densities. According to Ata (2012), the entrained particles are mainly composed of gangue minerals.

Froth stability refers to the bubble's ability to withstand coalescence and rupture. An ideal froth in flotation is characterised by the least amount of coalescence and rupture (Farrokhpay, Saeed & Bradshaw, 2012). Froth stability has a significant influence on the overall flotation recovery and grade of valuable minerals. When the froth is too stable, gangue minerals may be entrained, consequently lowering the concentrate grade and in the case of an unstable froth, valuable minerals (hydrophobic) are lost back to the pulp zone, which further reduces the flotation recovery (Bradshaw, Oostendorp & Harris, 2005). Additionally, it is widely known that a slurry with high amounts of talc produces a highly stable froth, which further promotes the recovery of gangue minerals (Becker et al., 2009).

One of the most widely used methods to determine froth stability is the froth column (Bikerman, 1973). Froth stability can be determined using various tests such as the dynamic and static tests. In the dynamic test, the equilibrium height of an aerated slurry is determined, whereas in the static test, the froth formed is allowed to decay without restoration of the froth by the gas and the decay rate determined. The dynamic test resembles the dynamics of the true froth flotation system (Farrokhpay, Saeed & Bradshaw, 2012). The dynamic stability equation given by Bikerman (1973) is used to determine the froth stability and the equation is given below.

$$\Sigma = \frac{H_{max}A}{Q} \tag{2-5}$$

where Σ is the dynamic stability factor, H_{max} is the maximum froth height, A is the column cross-section area and Q is the air flowrate.

In the study conducted by Bos and Quast (2000) on the effect of oils and lubricants on froth flotation of a Cu ore, the authors found that the addition of oils such as hydraulic oil, degreaser, and transmission oil resulted in the formation of a heavy froth, which further led to the reduction of chalcopyrite grade. The above phenomenon was observed at low concentrations for some oils. This was attributed to the coating of the oil on the mineral’s surface, causing some hydrophilic gangue to become hydrophobic; thereby, promoting the flotation of liberated gangue minerals.

2.4.3. Flotation reagents

Reagents are used in flotation to facilitate the interaction between minerals and bubbles, thus achieving mineral separation. These include collectors, frothers, depressants, and regulators/modifiers. Some of the widely used reagents in the local PGM includes, xanthate collectors, alcohol-based frothers, carboxymethyl cellulose (CMC) depressant, and activators such as copper sulphate (Wiese, Harris & Bradshaw, 2008; Wills & Finch, 2015). The reagents of particular interest to this study are sodium silicate, which is a flotation modifier, and an oil degreaser. These are covered in the paragraphs below.

Sodium silicate

Sodium silicate is a chemical compound that is primarily used as a flotation dispersant and has mild depressant properties (De Castro & De Hoces, 1993; Wang, Weiqing et al., 2018; Zhang, Wencai & Honaker, 2018). It is also known as "water glass" and has a formula of $(Na_2O)_X.(SiO_2)_Y$, where the ratio x:y is referred to as the modulus. The main forms are the sodium metasilicate where x:y is 1, sodium orthosilicate (x:y is 2) and sodium pyrosilicate (x:y is 3). Moduli ranging between 1 and 3.75 are commercially available (Lagaly et al., 2000). The compounds with a ratio of less than 2.85 are alkaline, and those with a greater ratio are neutral.

The aqueous dissolution of sodium silicate produces three main species: $Si(OH)_4$ at pH < 9.4, $SiO(OH)_3^-$ at pH > 9.4, and $SiO_2(OH)_2^-$ at pH > 12.6 (Feng et al., 2012). The two negative species are responsible for gangue rejection of silica as well as dispersion. Molifie et al. (2021) discovered that sodium metasilicate could effectively act as a dispersant, rheology modifier, and depressant in highly altered PGM ore. According to the same study, high concentrations of sodium silicate (>1000 g/t) reversed the charge on serpentine minerals, resulting in electrostatic repulsion between valuable minerals and serpentine. This dispersion prevented slime coating while also lowering pulp viscosity, which if too high can cause low recoveries due to poor gas dispersion. Additionally, sodium metasilicate was found to improve froth drainage by lowering the froth stability which subsequently increased the concentrate grade.

Degreaser (ethylene glycol butyl ether)

Ethylene glycol butyl ether (EGBE) is a non-volatile organic surfactant that is widely used as a cleaning agent in oil-related spills (Adamy, 1994; Desnoyers et al., 1983; Kuchierskaya et al., 2021). Apart from being a surface cleaning agent, it is used in paints, surface coatings, inks, and dyes (ETHER, 2007). Its chemical structure is shown in Figure 2-6. The molecule is composed of a hydrophilic end (polar) and a hydrophobic end (nonpolar), thus making it an amphiphilic molecule. These amphiphilic properties aid in oil removal in an aqueous solution as the hydrophobic end attracts the oil and the hydrophilic end attracts the water molecules. This further lowers the interfacial surface tension between oil and water, thus ensuring the dislodgement of oil from the water-oil interface. ETBE is also known as 2-Butoxyethanol in the literature.

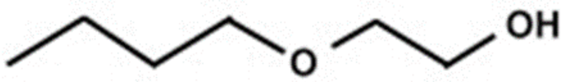

Figure 2-6: Ethylene glycol butyl ether molecular structure.

2.4.4. Rheology background and its effects on flotation

Rheology is the study of deformation and flow of matter under applied stress. In froth flotation, the slurry flow behaviour is described graphically by a flow curve or rheogram. The rheogram is obtained by plotting the shear stress against the shear rate. The rheological complexity of the slurry is well known to cause mineral processing challenges. Therefore, understanding the slurry's rheological behaviour is vital, as it is indicative of the level of inter-particle interaction or aggregation.(Farrokhpay & Saeed, 2012). Figure 2-7 depicts the rheological behaviour of different types of fluid. The slurry behaviour can either be Newtonian or non-Newtonian. The Newtonian fluids are characterized by a linear relationship between shear stress and shear rate with no yield stress, as depicted in Figure 2-7, whereas the non-Newtonian fluids are represented by plastic, Bingham, pseudoplastic, and dilatant behaviours.

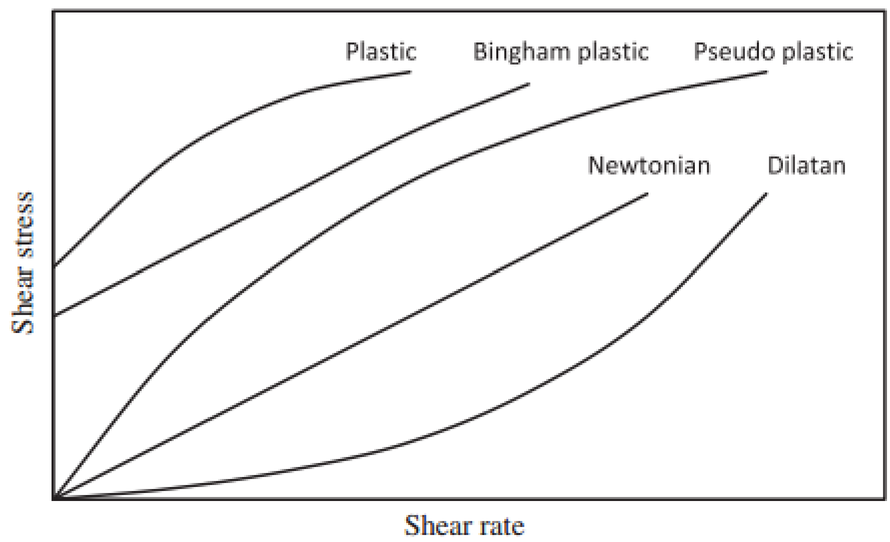

Figure 2-7: Flow behaviour of different fluids (Farrokhpay, 2012).

The two main parameters that are used to describe the rheological behaviour of mineral slurries are apparent viscosity and yield stress. These aid in determining the extent of particle aggregation and dispersion. Apparent viscosity refers to the viscosity of a non-Newtonian fluid at a particular shear rate. Mineral slurries generally exhibit a non-Newtonian, pseudoplastic or Bingham behaviour. Bingham and Casson's models are widely used to estimate the rheological properties of the mineral suspensions (Farrokhpay & Saeed, 2012). The Bingham model is given by the equation below.

$$\tau = \tau_B + \eta_{pl} D \tag{2-6}$$

where τ is the shear stress, τ_B is the yield stress, η_{pl} is plastic viscosity, and D is the shear rate.

Effects of rheology on froth flotation

The primary requirement in froth flotation is gas dispersion, and this plays a vital role in particle suspension and bubble-particle attachment. From a study conducted by Bakker, Meyer and Deglon (2010), the authors found that the rheological complexity of the slurry as caused by high yield stress and viscosity that may result in the formation of a 'cavern' around the impeller. Moreover, this may lead to poor gas dispersion throughout the flotation cell and reduced interaction between bubbles and particles (Shabalala et al., 2011). The above phenomenon is associated with fine-grained particles and ores that contain appreciable mineralogical content of phyllosilicate minerals, especially the minerals such as talc (Becker et al., 2013).

Wang, Lei and Li (2020) highlighted the importance of understanding the flotation rheology as this influences the separation of minerals. The authors identified solids concentration, particle size and shape, surface charge and particle interaction, as the key slurry rheological factors. The increase in slurry solids concentration is associated with an increase in pulp viscosity. This may result in the formation of cross-linked network structures in the presence of fine-grained phyllosilicate minerals (Chen, Xumeng & Peng, 2018; Ndlovu, Farrokhpay &

Bradshaw, 2013). To prevent this from occurring, flotation cells are usually operated at concentrations in the region of 40 wt.% (~20 vol%) with the upper limit of 50 wt.% (Bakker, Meyer & Deglon, 2009). Becker et al. (2013) also demonstrated that, at high solids percent the slurry viscosity increases exponentially.

It is widely known that the pulp viscosity generally increases with a decrease in particle size. From a study conducted by Farrokhpay, Bradshaw & Dunne (2013), an increase in apparent viscosity was observed with the decrease in particle sizes. Fine particles also display electroviscous effects due to the high volume fraction and formation of double layers as a result of reduced distance between particles (Zhou, Scales & Boger, 2001). These electroviscous effects are more pronounced in high solids concentrations. Particle shape affects pulp rheology as well; however, its effect on pulp rheology has not been thoroughly researched (Wang, Lei & Li, 2020). A study by Collins and Sciarone (1974) found that, for a particular density and particle size in a ferrosilicon suspension, irregular (rough and angular) particles display higher viscosity than spherical particles.

In an aqueous suspension, fine-grained phyllosilicate minerals may form a network structure due to different modes of interaction: edge-edge (E-E), edge-face (E-F), and face-face (F-F), caused by charge heterogeneity. The E-E and E-F interactions give rise to greater viscosity in a suspension (Chen, Xumeng & Peng, 2018). An example of these modes of interaction is illustrated in Figure 2-8. The kind of particle aggregates generated is determined by the surface charges (edges and faces), as well as the suspension pH. The F-F interaction is characterized by the presence of laminar structures and low yield stress; alternatively, the E-E and E-F are rheological complex due to the presence of lengthy 3D structures which result in high slurry viscosities (Gupta et al., 2011; Johnson, Russell & Scales, 1998).

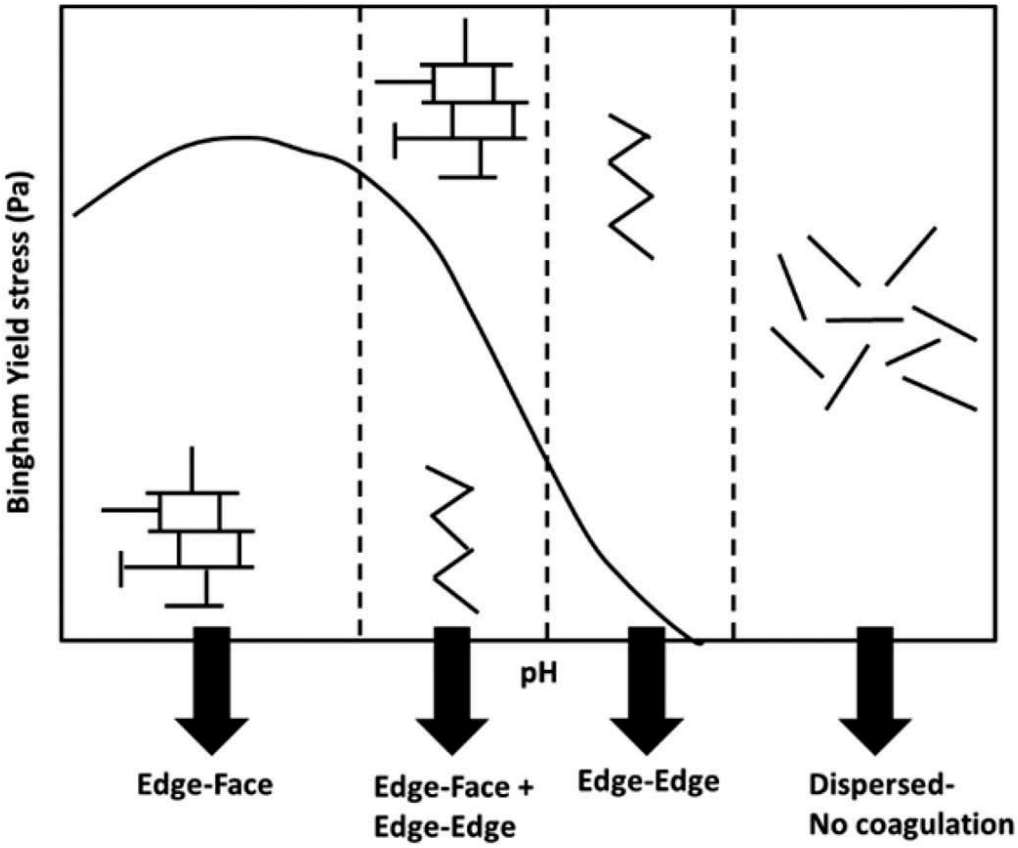

Figure 2-8:The effect of pH on kaolinite's Bingham yield stress with regard to different mode of interaction (Chen, Xumeng & Peng, 2018).

2.5. Effects of oil on flotation and its interaction with talc

This section focuses on the interaction mechanism between oil and phyllosilicate alteration minerals. Hydraulic oils are mainly derived from the processing and purification of crude oil. The main fluid is predominantly a base stock (mineral oil or synthetic) with small percentages of additives depending on the oil's intended function or properties.

2.5.1. Adsorption mechanism between talc and oil

The adsorption of oil on the talc surface occurs via three steps: i) the diffusion of oil molecules onto the surface, ii) entrapment in the mineral surface through capillary action, and iii) the agglomeration of oil droplets in the mineral structure (Polowczyk & Kozlecki, 2017; Sabir, 2015). Hydrophobicity is the main factor which governs the interaction (Zadaka-Amir, Bleiman & Mishael, 2013). The oil accumulates on the mineral surface due to the hydrophobic attraction which in turn results in formation of an oil bridge between particles. This further pulls individual particles together to form agglomerates. As a result of the oil adsorption on the mineral surface, hydrophobicity becomes enhanced. Additionally, due to particle agglomeration, the slurry viscosity increases (Luo & Tomac, 2018; Yang et al., 2012). Talc is used in environmental oil clean ups due to this interaction mechanism with oil (Zadaka-Amir, Bleiman & Mishael, 2013).

2.5.2. Effect of oil on froth flotation

Hydrocarbons are widely used as collectors in the flotation of coal (Jia, Harris & Fuerstenau, 2002) and molybdenite (Lin et al., 2018). They are known to enhance the flotation recovery by increasing the hydrophobicity of these particles; therefore, increasing the attachment between valuable minerals and bubbles. It is also worth noting that molybdenite, a naturally hydrophobic mineral, exhibits similar anisotropic surface properties to talc (Castro, Lopez-Valdivieso & Laskowski, 2016). Molybdenite is naturally hydrophobic due to the vast difference in the basal plane to edge ratio (Yuan et al., 2019).

Bos and Quast (2000) conducted a study on the effect of hydrocarbons (oil and lubricant) on the flotation of chalcopyrite (P_{80} of 75 µm). The authors found that the addition of oil at 500 g/t produced a very sticky froth, which led to the increase in the recovery of chalcopyrite. The increase in recovery was attributed to the ability of the distillate oil to form chalcopyrite aggregates. However, when other hydrocarbons: engine oil, hydraulic oil, degreaser, and transmission oil, were used, the authors observed a decrease in selectivity. This was ascribed to the coating of hydrocarbon across the surface of gangue minerals, causing them to float and diluting the grade. Hornn et al. (2020) also demonstrated that chalcopyrite recovery improves in the presence of oil due to agglomeration. From a study conducted by He et al. (2011) on the effect of nonpolar compositions on flotation of molybdenite, the authors found that six different diesel oils each had different effects on the recovery of molybdenite. Although the molybdenite fines were liberated, the variation in recoveries was linked to the carbon chain length in oils that tends to affect the oil dispersibility. The effects of various types of oil on different ore types are summarized in Table 2-2.

From the studies conducted by both He et al. (2011) and Bos and Quast (2000), it is clear that different oil types used in flotation may induce different effects on the same mineral. Although studies have been conducted on the effect of oil on the flotation of various minerals, there is still a dearth of understanding on the effect of oil on the flotation of PGM bearing ores because so little has been reported, and mainly the reported cases are based on anecdotal observations. It is of utmost importance in this study to understand how oil used in the mechanised mining activities of UG2 ore affects the downstream recovery of PGMs.

Table 2-2: Summary table of the effects of oils on different ore types.

Oil type	**Ore type/mineral**	**Effect (s)**	**Reference (s)**
TFC 430 (hydraulic)	Copper sulphide	Reduced the copper grade	Bos and Quast (2000)
Distillate	Copper sulphide	Increased chalcopyrite recovery	Bos and Quast (2000)
Transformer oil	Molybdenite	Promote aggregation of particles	Lin et al. (2018)
Kerosene	Molybdenite	Used as a collector to enhance the recovery	Li et al. (2021) Lin et al. (2018)
Kerosene	Coal	Used as collector to increase the mineral hydrophobicity	Chen, Yumeng et al. (2022) Zhang, Qingshan et al. (2021)
Diesel oil	Graphite	Used as a collector	Vasumathi et al. (2015)

2.6. Summary and critical synthesis

PGMs are naturally fine-grained, with grain sizes mostly below 10 µm within the UG2 ore. To recover them, they require liberation from the ore at very fine grinds. The liberation of PGM at fine grinds result in the creation of very fine particles, including alteration minerals. This presents challenges due to the rheological complexities associated with altered phyllosilicate minerals, especially the naturally floatable gangue. This will affect the rheology and froth stability. The presence of oil in such systems may further cause flotation problems. A study by Bos and Quast (2000) demonstrated that oils might result in detrimental effects on the flotation performance. From the same study, it was deduced that oil coating the surface of the gangue minerals promotes the flotation of gangue. Various studies have shown the collection abilities of oil (kerosene, diesel oil, transformer) in the flotation of molybdenite (He et al., 2011; Lin et al., 2018; Song et al., 2012) and coal (Jia, Harris & Fuerstenau, 2002). These hydrocarbons are used to enhance the surface hydrophobicity of the valuable mineral by adsorbing on the minerals, further promoting the interaction between the minerals and bubbles. Oil may also coat on the hydrophilic minerals, making them hydrophobic and resulting in their recovery as well as grade dilution (Bos & Quast, 2000).

However, there is still a gap in understanding the fundamental mechanism of hydrocarbon interaction with valuable minerals and gangue in PGM ores. Up to this point, minimal attention has been given to study the effect of hydrocarbons on the flotation of PGE bearing ores, which may contain a high amount of phyllosilicate alteration minerals that are known to compromise the recovery of valuable minerals. Various studies have linked the drop in flotation recovery and complex slurry behaviour to the presence of phyllosilicate alteration minerals in the slurry (Ndlovu, Farrokhpay & Bradshaw, 2013; Shabalala et al., 2011). The current study seeks to investigate the interactive effect of hydrocarbons and phyllosilicate minerals on the flotation performance of a UG2 PGM-bearing ore.

2.7. Hypotheses

Based on the literature, the following were hypothesized:

- The presence of phyllosilicate alteration minerals and a high concentration of hydrocarbons in the slurry will increase pulp viscosity, due to the agglomeration of phyllosilicate minerals with increased hydrophobicity. High pulp viscosity leads to reduced bubble-particle interaction and poor gas dispersion. This, in turn, will lead to a reduction in flotation recovery.
- High oil concentrations will reduce the concentrate grade due to oil coating both hydrophobic and hydrophilic gangue minerals, rendering them hydrophobic while also decreasing depressant adsorption and enhancing gangue mineral recovery.
- Sodium metasilicate will enhance recovery and grade due to electrostatic repulsion between particles upon adsorption, increasing the dispersion of particles, reducing the viscosity and increasing froth drainage.
- The amphiphilic properties of the degreaser will aid in dislodging oil, thus improving both the recovery and grade.

2.8. Key Questions

1. What are the effects of hydrocarbons on the flotation performance of UG2 ore?
2. At what oil concentration is the flotation recovery of UG2 ore compromised through hydrocarbon contamination?
3. What are the effects of hydrocarbon on slurry rheology and froth stability, and how is this affected by the presence of high phyllosilicate alteration minerals content?
4. Is the oil adsorbing on the particle surface and also targeting the more hydrophobic particles?
5. What are the effects of using sodium silicate and the degreaser on flotation performance?

3. Experimental methodology

This chapter covers the methodology developed to assess the proposed hypotheses and the key questions presented in Chapter 2. Chapter 3 starts with a description of the underground sampling and ore preparation, followed by milling curves and baseline flotation test work and lastly the supplementary tests: column stability, rheology, and oil adsorption studies. Unless otherwise indicated, all sample preparation and experimental test work took place in the Centre for Minerals Research (CMR) facilities in the Department of Chemical Engineering at the University of Cape Town (UCT). The study plan is shown in Figure 3-1.

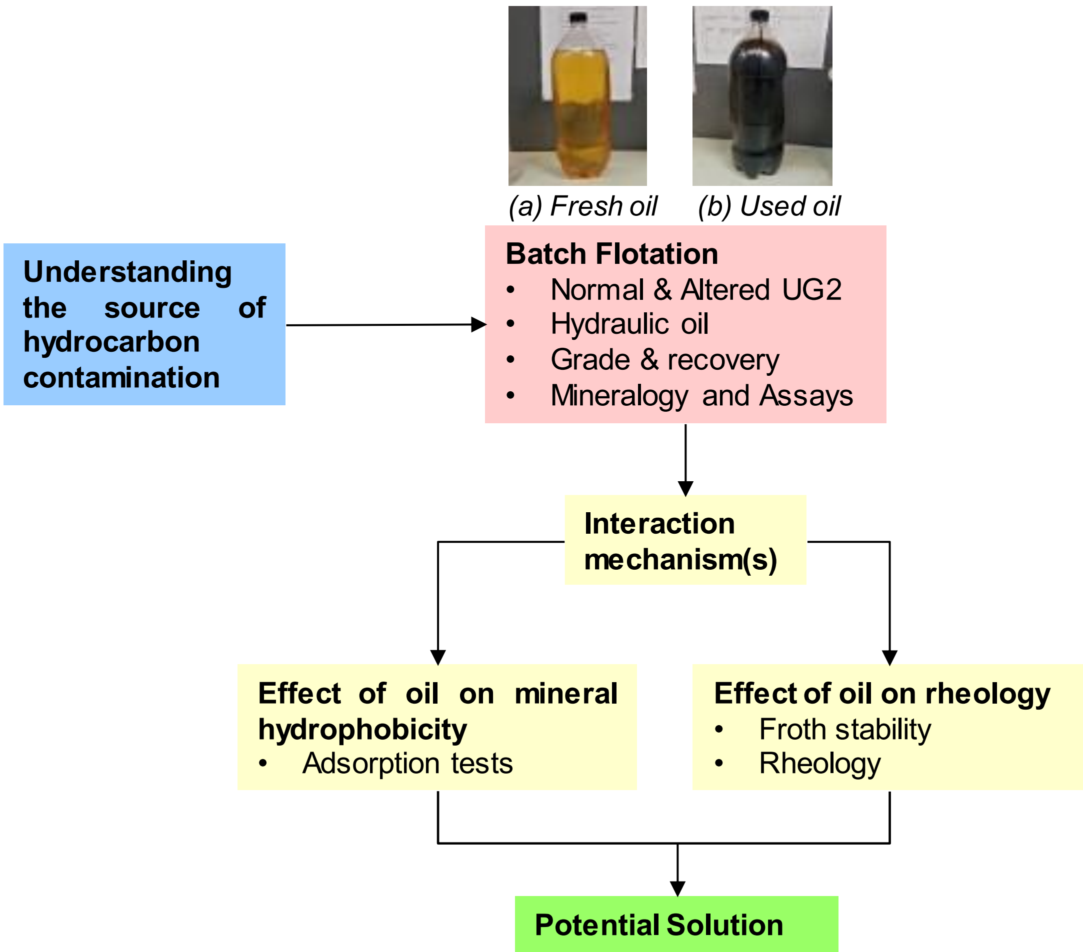

Figure 3-1: Study research plan.

3.1. Ore sampling and preparation

Following a site visit to TRP in April 2021, the UG2 ore was sampled underground from various regions. This comprised sampling at two locations with varying degrees of alteration: a less altered site (normal reef) and an altered reef. The sampling was in line with the underground mining cut employed at TRP. The altered ore was sampled from the south shaft level 5 F-drive (S5F) UG2 reef, whereas the less altered (normal UG2 reef) ore was obtained from the northern section (N8). The former was pale black in colour, whereas the latter was dark black. The region where the altered ore was sampled is characterised by the presence of a single split reef with internal waste dilution as shown in Figure 3-1. This is similar to the single split reef identified at TRP (Figure 2-3 b) and has strong presence of alteration mineral associated with pyroxene.

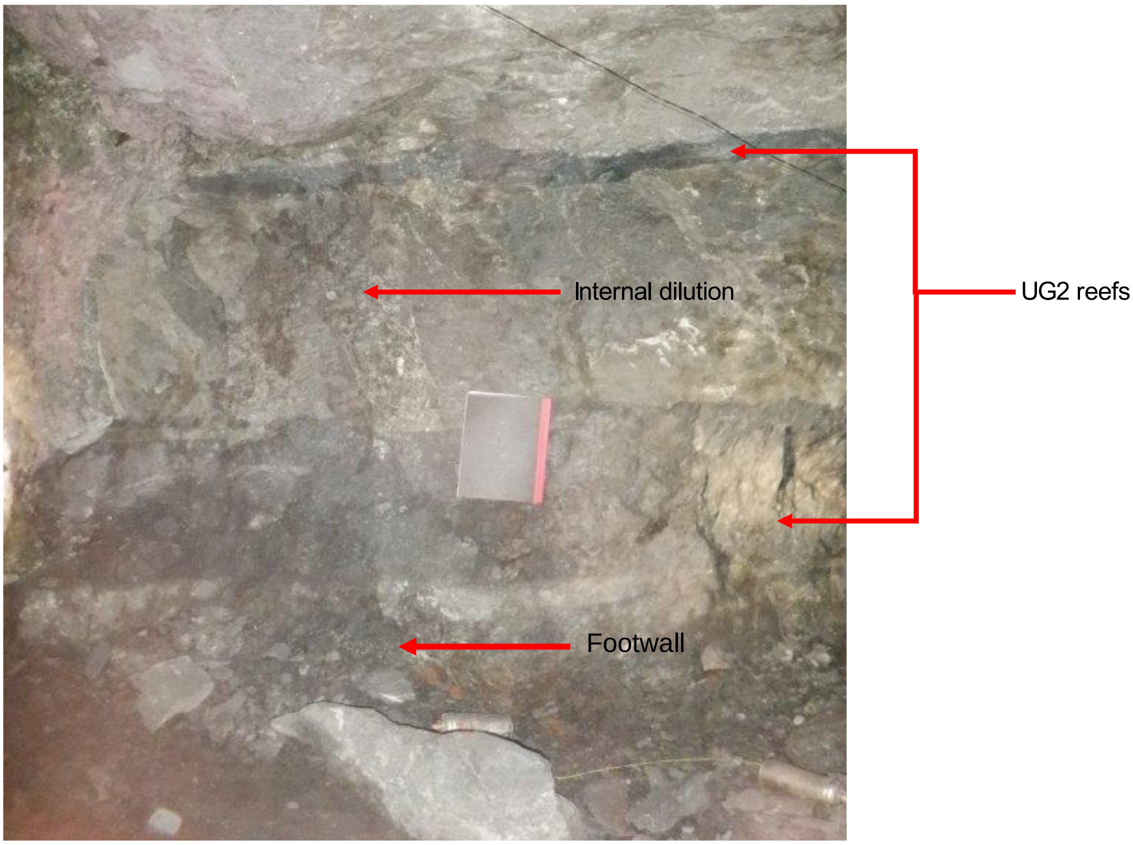

Figure 3-2: Split reef identified at TRP in S5F. An A4 textbook (29.5 cm × 20.5 cm) was used to scale the reef. Figure scale – 1 cm: 22.4 cm.

Following sampling, the two underground UG2 ore samples (normal and altered, 300 kg each) were couriered to UCT. Upon arrival, the two samples were dried in a pan overnight, followed by jaw crushing. After crushing, the samples were dry screened to ensure that a particle top size of 100% passing 3 mm was achieved. Each sample was blended, riffled, and split using a 10-way Dickie and Stockler rotary splitter into ±4.1 kg representative aliquots. This mass corresponds to the solids concentration of 37 wt.% in an 8L flotation cell. The samples were labelled accordingly and stored.

3.2. Milling curves

The approximate primary and secondary rougher grinds at TRP are 40% and 70% passing 75 µm, respectively (TRP Internal Mine Report, 2021). The test work was performed at the secondary rougher grind of 70% passing 75 µm because this is where potential problems arising from phyllosilicate alteration minerals and hydrocarbon contamination are likely to be the most noticeable compared to the primary grind. To determine the time required to achieve the secondary grind, the ±4.1 kg ore samples of both normal and altered ore, were ground in a 3 kg stainless steel rod mill at various time intervals to generate a milling curve. The mill was charged at 66 % solids with 22 stainless steel rods (equal diameter of 20.5 mm), and the speed was fixed at 75 % critical speed - equivalent to 56.4 revolutions per minute (rpm). The mill discharge was filtered, followed by an overnight oven-drying at 80°C. The dried samples were deagglomerated and split with a 10-way Dickie and Stockler rotary splitter to get a representative 250 g aliquot for each ore type. Each aliquot was wet screened at 75 μm, and both the oversize and undersize were dried and weighed to determine the percentage passing through the 75 μm screen aperture. The milling curves developed are displayed in Figure 3-3.

The required milling time (at the grind of 70% passing 75 μm) for the altered ore of 26.8 min was considerably less than the good ore (48.3 min), providing some of the first indications that the altered ore has higher proportions of softer phyllosilicate alteration minerals present.

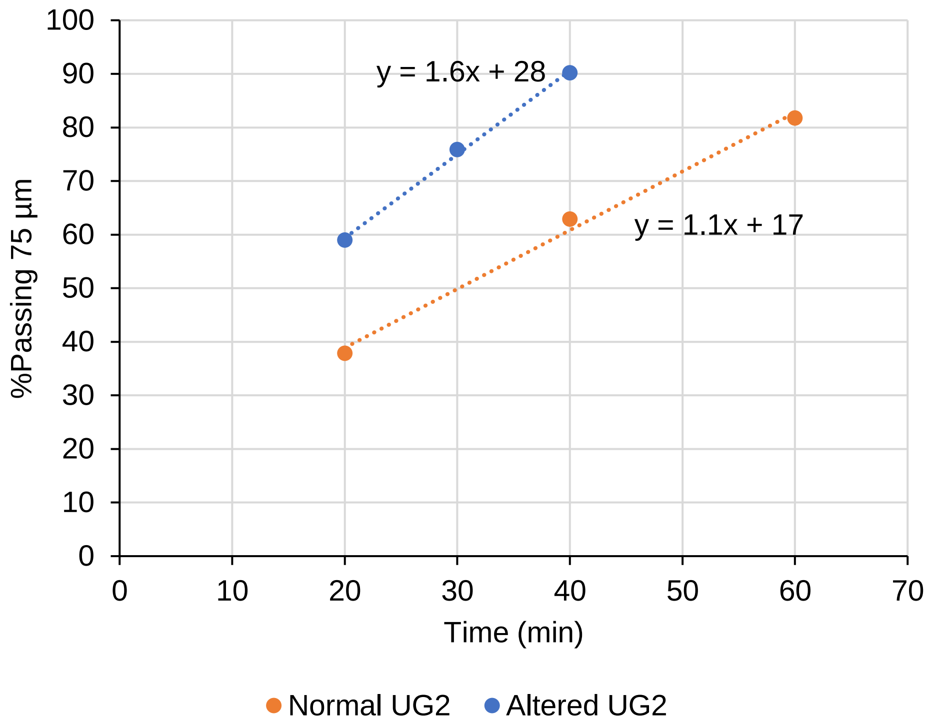

Figure 3-3: The normal and altered UG2 milling curves.

3.3. Feed characterization

Quantitative Evaluation of Minerals by Scanning Electron Microscopy (QEMSCAN) was used to mineralogically characterize the unsized feeds of the normal and altered ore. Representative subsamples of the flotation feed from each ore were prepared for mineralogical analysis on an FEI QEMSCAN 650F instrument. Samples were mixed with nanocarbon and prepared into polished 30 mm diameter ore mounts using a series of grinding and polishing steps followed by carbon coating to ensure charge dispersal in the scanning electron microscope. All measurements were run at 25kV and 10nA. Mineral grades for each ore were measured using the bulk mineral analysis method on QEMSCAN with a pixel size of 3 μm and field size of 732 μm. The PGM mineralogy was characterised using the trace mineral search analysis method on QEMSCAN with a 0.75 μm pixel size and field size of 450 μm. A total of 28 epoxy resin mount surfaces per ore were analysed in the PGM search.

X-ray Fluorescence (XRF) spectrometry and quantitative X-ray diffraction (XRD) were used to validate the bulk mineralogical composition. Equivalent aliquots were submitted for XRF analysis on a Panalytical XRF instrument in the Department of Geological Sciences, UCT to determine the bulk chemical composition of the ores. In combination with those from X-ray powder diffraction (XRD), these results were used to validate the QEMSCAN measured mineral grades (Appendix A, Figure A-1). For XRD, a 3.5 g representative aliquot of each ore was micronized in the presence of ethanol to produce fine powder for analysis. A Bruker D8 X-ray diffractometer with a LynxEye detector and CoKα radiation hosted in the Centre for Catalysis Research in the Department of Chemical Engineering, UCT was used for the analysis. The mineral phase identification was conducted with the Bruker EVA software and phase quantification with the Rietveld refinement method using TOPAS software.

3.4. Flotation reagents

The reagents employed in the batch flotation and froth stability tests are those currently in use at the TRP concentrator plant (TRP Internal Mine Report, 2021). No changes were made except for the frother dosage, which was increased from the initially provided dosage to obtain a suitable mass recovery in the batch flotation cell. Dosage scoping trials were carried out for this purpose. The reagents and dosages used in this project are covered below.

3.4.1. Collector

The collector used was Senkol 821, provided by AECI Mining Chemicals. This is a blend of 70 % dithiophosphate (DTP), 15 % sodium isobutyl xanthate (SIBX), and 15% sodium ethyl xanthate (SEX). The collector arrived in two parts: a dry component of xanthate and a solution component of the liquid collector with water to make up 40% active ingredient. The collector had a shelf life of 14 days upon mixing the two components and was therefore only mixed when required. Prior to milling, a dose of 110 g/t of Senkol 821 was added to each sample in the mill.

3.4.2. Depressant

An AECI Mining Chemicals' depressant (Sendep 30D) was used throughout the flotation experiments. This is a carboxymethyl cellulose (CMC) depressant. A 1 wt. % stock solution was prepared by dissolving 1 g of the dry product in 100 mL of distilled water for an hour every five days. The same dosage (20 g/t) was used for both the normal and altered ores (irrespective of the amount of naturally floating gangue).

3.4.3. Frother

Senfroth 200 frother was supplied by AECI Mining Chemicals. This is a polypropylene glycol frother. For all flotation experiments, the frother dosage was kept constant at 30 g/t (130 µL).

3.4.4. Hydraulic oil

TRP provided two 2 L of fresh and used hydraulic circulation oil (AWS68) that was used for this project. Along with the collector, the hydraulic oil was added to the mill.

Before commencing with the flotation experiments, the two hydraulic oil samples supplied by TRP were sent for analysis to Oilwatch in Cape Town. The aim was to determine if there was any significant difference in terms of the viscosity, impurities, and additives between the samples (see Table 3-1).

Table 3-1: Summarized analysis of oil properties and impurities.

	V40 (cSt)	Oxidation index	%H_2O	TAN	TBN	%Fuel	%Soot	Nitration index	Ca (ppm)
Fresh oil	68.5	2	ND	0.59	0	0	0	0	42
Used oil	60.5	4	ND	0.59	2.73	10.9	0.5	4	524

**V40-represents the viscosity at 40°C and cSt is centistoke, TAN is the acidity KOH/g, TBN is the antiacid KOH/g, Fuel represents other fuel contaminants, and ND is not detected. The oxidation index is normally used to predict the oil's shelf lives and the nitration index is used to indicate the presence of nitrogen oxide compounds which may help stimulate oxidation.*

The major differences were in the oil viscosity, which is lower for the used oil compared to the fresh oil, and the calcium levels, which were slightly higher in the used oil (524 ppm compared to 42 ppm in the fresh oil).

3.4.5. Sodium metasilicate and the degreaser

For the potential mitigation of the effects of the oil addition, sodium metasilicate (SS) and the degreaser (Solox 180) were used separately. The sodium metasilicate was supplied by Sigma Aldrich whereas the degreaser (ethylene glycol butyl ether) was supplied by AECI Mining Chemicals. A modulus of 1 was used for all the mitigation tests with SS and a dosage of 1500 g/t. This was administered into the flotation cell after transferring the slurry from the mill. For the degreaser, a dosage of 500 g/t was used.

3.4.6. Synthetic plant water

Synthetic plant water was used in all experimental work, including milling. This was done to emulate the typical salt ion concentration of South African PGM concentrator plants. This was prepared by dissolving a weighed amount of different salts, given in Table 3-2, in distilled water (Wiese, Harris & Bradshaw, 2005).

Table 3-2:Various salt quantities used to prepare 40 L of synthetic plant water.

Inorganic salt name	Chemical formula	Salt mass in 40 L (g)
Magnesium Sulphate heptahydrate	$MgSO_4.7H_2O$	24.60
Magnesium Chloride hexahydrate	$Mg(NO_3)_2.6H_2O$	4.28
Calcium Nitrate tetrahydrate	$Ca(NO_3)_2.4H_2O$	9.44
Calcium Chloride dihydrate	$CaCl_2.2H_2O$	5.88
Sodium Chloride	NaCl	14.24
Sodium Carbonate	Na_2CO_3	1.20

3.5. Batch flotation experiments

The flotation experiments were conducted in an 8 L modified Leeds batch flotation cell shown in Figure 3-4. The oil concentration was varied from 0 to 500 g/t in increments of 100 g/t. The collector and hydraulic oil were all added to the mill prior to milling. After milling the 4.1 kg sample, the slurry was transferred into the 8 L Leeds flotation cell, and the cell capacity maintained at 8 L by topping up with synthetic plant water bringing the solids concentration to 37 wt.%. The cell impeller speed was kept constant at 1200 rpm and the air flowrate at 15 L/m - equivalent to a Jg of 0.69 cm/s. The froth height was maintained at 2 cm.

After transferring the slurry, the depressant was added to the cell and allowed to condition for 2 minutes. Following the depressant, the frother was added and allowed to condition for 1 minute. The air valve was then opened after conditioning with the frother to initiate flotation. Four concentrates (C1 to C4) were collected into a pan by scraping the froth every 15 seconds at the interval of 0-2 (C1), 2-6 (C2), 6-12 (C3), and 12-20 minutes (C4). The concentrates were vacuum filtered, and the tailings were filtered with a filter press. Prior to floating, a feed sample was collected with a 60 mL syringe and three tailing samples (each 60 mL) at the end. The feed, concentrates, and tails were oven-dried and weighed. Furthermore, the water recovery in each experiment was determined by weighing the filter paper, four wash bottles, and concentrate pans before and after floating. The experiments were conducted in duplicate to produce sufficient mass for assays and to enable the calculation of the experimental error.

Figure 3-4: 8 L Leeds batch flotation cell.

For possible mitigation of the deleterious effects of oil addition, two reagents were used separately: SS and a degreaser. The use of SS to mitigate the viscous effects was evaluated first, where a dosage of 1500 g/t was used. This was conditioned for 5 min after transferring the slurry into the cell. Thereafter, it was followed by administering the depressant and frother. For the degreaser experiments, the same procedures were followed only replacing SS with 500 g/t of the degreaser. Both these two reagents were evaluated at the extremes of the investigated oil dosage (500 g/t).

The concentrates, tailings and two milled feeds were split with a rotary splitter to get representative samples (30 g for each concentrate, 300 g for the tail and feed). These samples were sent externally to SGS South Africa (Pty) Ltd Geochemistry laboratory in Rustenburg (North West) for Platinum (Pt), Palladium (Pd), Copper (Cu) and Nickel (Ni) assays. The samples were fire assayed for Pt and Pt using lead (Pb) as a collector and finished with Inductively coupled plasma - optical emission spectrometry (ICP-OES). The detection limit for Pt and Pd was 0.02 g/t and Cu detection limit was 0.5 g/t and 1 g/t for Ni. The Cu and Ni analyses were conducted using ICP-OES, in which the samples were prepared with a sodium peroxide fusion. The duplicate standard error for the grade and recovery were determined and plotted.

3.6. Froth stability

A perspex frothing column (Figure 3-5) with a diameter of 20 cm and height of 100 cm was utilised to investigate the effect of varying oil concentration on the froth stability of the UG2 ores. The normal and altered UG2 ores were utilised in the 3-phase froth stability experiments, and the oil concentration was varied from 0 to 500 g/t in increments of 100. The same reagents and dosages were utilised in the same order as the batch flotation tests. The milled slurry was transferred to the column feed bucket maintaining the same solids percent used in flotation, and the volume was brought to the 8 L mark by topping up with synthetic plant water. This was followed by the addition of the depressant and frother. The slurry content in the bucket was constantly agitated with an overhead stirrer to prevent the settling. Before transferring the slurry to the column, the overhead column stirrer was turned on to prevent settling within the column. The slurry was pumped from the feed bucket to the froth column with the aid of a peristaltic pump until a pulp height of 15 cm was achieved. The column was fitted with four baffles to prevent the formation of the vortex and four pore-2 size ceramic frits for controlling the bubble size. The ceramic frits were cleaned in an ultrasonic bath prior to each experiment. A Jg of 0.69 cm/s, equivalent to an airflow rate of 13 L/m, was used for all the experiments. This was the same Jg used in flotation. The air to the column vessel was turned on, and the change in froth height was tracked every 2 seconds until the maximum steady height was reached, after which the air supply was cut off. A video was recorded to track the height since the growth was very rapid in the first few minutes. All the experiments were performed in triplicate and the standard error determined.

Figure 3-5: Column used to investigate the effect of oil on the froth stability of the normal and altered ore.

3.7. Rheology

AR1500EX-TA rheometer shown in Figure 3-6 was used to conduct all rheological measurements. Pure talc derived from a mineral specimen supplier (<75 µm) and two types of ore (normal and altered UG2; each with a grind of 70% <75 µm) were used. The purity of the talc sample was determined using quantitative XRD (with the same procedure as given in section 3.3), which was 90.5 wt.% talc, 1.3 wt.% calcite, and 8.2 wt.% amphibole. Rheological studies were undertaken to investigate the effect of oil on talc pulp viscosity in the absence of other minerals, followed by studies on the two ores' pulp viscosity. This was done to test how the talc would react to oil and what to expect if the mineralogy of the particular ore of interest has a high proportion of talc content. The talc experiments were carried out at solid concentrations of 20 and 30 vol.%, whereas the ores were tested at 40 vol.%. The feed aliquots were prepared using a rotary riffler splitter. Distilled water was used to prepare the slurry suspensions. Prior to the experiments, the two UG2 ore samples were sent to UIS Analytical Services in Centurion to determine the densities, using the Micromeritics AccuPyc II 1340 Gas Pycnometery instrument. The densities were used to determine the solid volume concentration (vol.%) in each respective sample. The densities for normal and altered ore were 3.612 g/cm^3 and 3.419 g/cm^3, respectively. For talc experiments, a density of 2.7 g/cm^3 was used for all solid volume concentration (vol.%) calculations (Patnaik, 2003).

The suspensions were prepared in the rheometer's sample holder cup before mounting and lowering the standard vaned rotor geometry to the desired depth. Thereafter, the TA software package was used to set up the conditions of various samples: temperature, pre-shear, pre-shear duration and equilibration time. The temperature was kept constant for all the experiments at 22°C. Other parameters were changed depending on the sample's solids concentration. For the talc tests, the pre-shear used was between 300/s and 400/s for a duration of 60 seconds and an equilibration time of 5 seconds. A pre-shear of 400/s was used for the two ores for a duration of 60 seconds and an equilibration time of 5 seconds. Where the rheogram was curved and did not follow a Bingham-type behaviour, which is defined by a linear connection between shear stress and shear rate as well as the existence of a yield stress, the conditioning parameters were adjusted accordingly. All measurements were conducted in triplicate and the errors determined (shear stress, shear strain and viscosity).

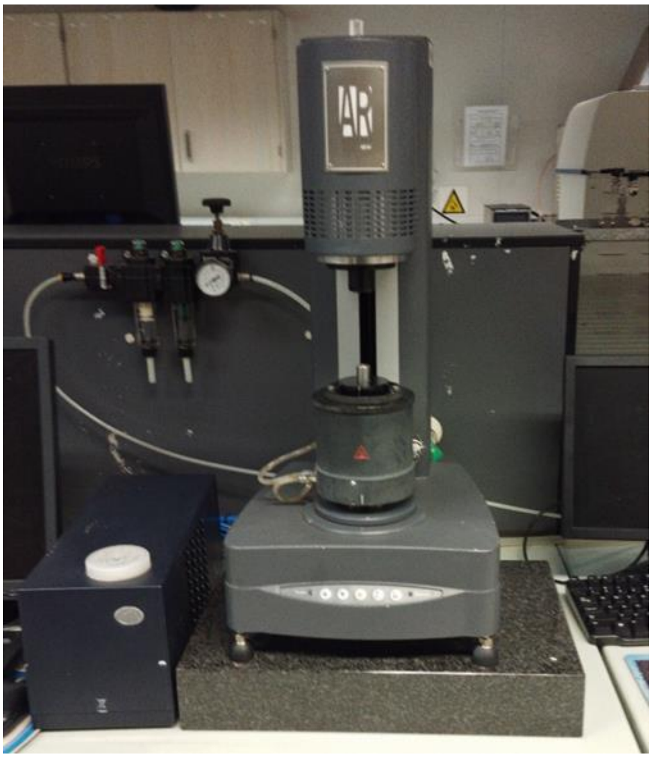

Figure 3-6: AR1500EX-TA rheometer fitted with a standard vaned rotor geometry.

3.8. Oil adsorption studies

The objective of the oil adsorption tests was to decouple if the hydraulic oil was adsorbing on the particle surfaces or remaining in the slurry suspension. Two separate experiments were conducted: (i) total organic carbon analysis and (ii) oil extraction with petroleum ether.

3.8.1. Total organic carbon (TOC)

A 4.1 kg feed sample of each ore type was milled for a predetermined duration to achieve the target grind of 70 % passing 75 µm. Only the oil (200 g/t and 500 g/t) was added in the rod mill prior to grinding, and no other reagents were added thereafter. This was to ensure that the only carbon detected in the analysis was from the oil or from the ore (naturally occurring). Distilled water was used for milling. The slurry was filtered post-milling, and the residual liquid was retained. The residual liquid was transferred to a tank fitted with baffles and an overhead stirrer for proper mixing. Thereafter, a 40 mL representative sample was collected with a syringe from the tank. A multi-N/C 3100 analyser from the centre for bioprocess engineering research (CEBER) lab, UCT was used to analyse the TOC in each representative sample. Before analysing, the vortex shaker was used to ensure mixing. The analysis was conducted in triplicate. A blank test without the oil was conducted for each ore. Equation 3-1 was used to determine the adsorption density (adsorbed oil/area) of each ore.

$$\Gamma = \frac{(C_0 - C_f) \times V}{m\ A} \tag{3-1}$$

where Γ is the adsorption density in mg/m^2, C_0 is the initial oil concentration in mg/L,C_f is the final oil concentration in mg/L, V is the liquid volume in L, m is the ore mass in g and A is the surface area of the ores in m^2/g. The Brunauer, Emmett and Teller (BET) surface areas of the two ores at the same grinds (70 %<75 µm) were conducted in the analytical lab at UCT. The normal UG2 milled feed had a BET surface area of 0.71 m^2/g, in contrast to 1.43 m^2/g of the altered UG2. Prior to analysing, a calibration curve (Figure 3-7) was developed using a known dosage of oil prepared in distilled water (200 and 500 ppm) and was vigorously mixed with a vortex shaker. Immediately after, the samples were analysed for TOC. The curve correlates the known oil dosage to the TOC in mg/L.

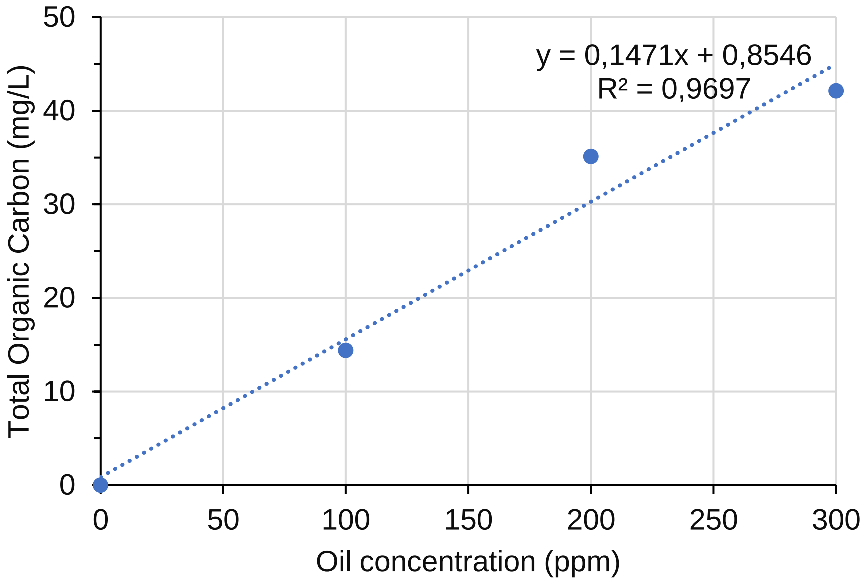

Figure 3-7: TOC calibration curve from known oil concentrations.

3.8.2. Solvent extraction

The oil extraction was conducted on the feed and concentrates of each ore using the method obtained from TRP (TRP Internal Mine Report, 2021) and Sui et al. (2014). Prior to this study, oil extraction trial tests were conducted by the metallurgical team at the mine to confirm that the oil dosage range mimics the contaminated ore. The feed experiments aimed to validate the TOC results, whereas the tests on the concentrates were used to determine if the adsorbed oil remained in the pulp phase or was recovered in the froth phase. The slurry was milled for the predetermined time and filtered post-milling. For each ore, a 500 g feed sample was prepared using a rotary splitter and 500 g/t of oil was added to the feed slurry prepared using distilled water. The filtered feed was transferred to a beaker, and 400 mL of pre-measured petroleum ether was administered, ensuring that the solids were completely submerged. The petroleum ether-solids mixture was stirred for at least 5 minutes to ensure that the oil adequately dissolved in petroleum ether. This was vacuum filtered, and the filtrate retained. The petroleum ether used was sourced locally from Kimix chemical and laboratory supplies in Cape Town. The boiling point of petroleum ether was between 40°C - 60°C. The filtrate was then transferred to a pre-weighed 400 mL glass beaker (mass 1). The filtrate beaker was placed in a 1 L pre-heated water bath beaker which was placed on the hot plate. This is a safety precaution to prevent direct heating of petroleum ether which is highly flammable and has a low flash point. The hot water bath was maintained at 100°C. The 400 mL beaker was allowed to cool at room temperature once all the petroleum ether had evaporated. This was followed by wiping the outside of the beaker to ensure there was no water before weighing. The beaker was weighed (mass 2) after evaporating all the petroleum ether and the mass of adsorbed oil was determined gravimetrically (mass 2 – mass 1). The oil dosage (g/t) was determined by dividing the mass of oil by the sample mass. A reference sample of petroleum ether was evaporated to determine that there was no residual mass present in this solvent.

For the concentrates, a 4.1 kg feed sample of each ore was milled, adhering to the batch flotation procedures. The collector was added to the mill along with 500 g/t of oil. The depressant was added and conditioned in the cell, followed by the frother. Concentrates C1 to C4 were collected and combined after flotation, after which the combined concentrate was filtered. The same extraction procedures as described in the above paragraph were carried out, and the amount of adsorbed oil in the concentrates was determined. The oil in the bulk tails was determined as the difference between the oil in the feed and the one extracted from the concentrates. Protective PPE and a respiratory mask were always worn during the experiments and the petroleum ether evaporation was carried out in a fume hood.

4. Results

This chapter provides the experimental results that address the hypotheses and key questions outlined in Chapter 2. Section 4.1 addresses the geochemical and mineralogical characterization of the two feeds, followed by the baseline flotation tests (Section 4.2). The individual behaviour of each ore in the presence of oil and mitigation reagents is covered in Sections 4.3 and 4.4. The results summary is given in section 4.5.

4.1. Feed characterization

The QEMSCAN technique was used to quantify the bulk mineralogy of the two ores and the results were validated using XRD and XRF (Refer to Figure A-1 in Appendix A). Additional information such as the PGE distribution, false colour images, grain size distribution, and the PGM liberation and association results, were also obtained with QEMSCAN.

4.1.1. Feed grades

The feed grades of the normal and altered ores are shown in Table 4-1. Comparing the 2E grade of the two UG2 feeds, the normal UG2 had slightly higher Pt and Pd grades than the altered UG2. However, the Pt:Pd ratio in the two ores was similar. Both Cu and Ni appeared in small quantities (Cu<0.01% and Ni-0.1%) in these two ores, which is typical of the UG2 ore (McLaren & De Villiers, 1982).

Table 4-1: Normal and altered UG2 ore head grades.

Ore	Pt (g/t)	Pd (g/t)	Pt:Pd	Cu (%)	Ni (%)
Normal	1.9	1.2	1.6	<0.01	0.1
Altered	1.4	1.0	1.4	<0.01	0.1

4.1.2. Bulk mineral grades

The bulk mineralogy provided in this section aided in quantification of phyllosilicate alteration minerals that are widely known to adversely affect the PGE flotation performance, as well as other gangue minerals.

The QEMSCAN bulk mineralogy of the two ores is provided in Table 4-2. As expected, chromite is the predominant mineral in both ores, 42.4 wt.% in the normal ore compared to 33.9 wt.% in the altered ore. The slightly lower percentage of chromite in the altered ore is due to the internal dilution in the split reef as shown in Figure 3-2. The second most common mineral in both ores was orthopyroxene, with a similar proportion of ~31 wt.%. The plagioclase content in the altered ore was almost double the quantity in the normal ore (20.9 wt.% compared to 11 wt.%). Both ores contained phyllosilicate alteration minerals (serpentine, talc, and chlorite; Figure 4-1), with the proportion higher in the altered UG2 than in the normal UG2 (9.1 wt% versus 6.2 wt.%). The serpentine content was relatively low in both ores relative to talc and chlorite. The chlorite content of the altered UG2 was approximately double that of the normal (3.2 wt.% versus 1.9 wt.%). Talc contributed around half of the total phyllosilicate alteration minerals in both ores; however, slightly higher in the altered UG2 compared to the normal UG2 (5.8 wt.% versus 3.8 wt.%). The XRD results (Appendix B) however, indicate the differences in talc content may be up to 4 times greater between the ores (1.8 wt.% in the normal and 8.1 wt.% in the altered. The real value most likely lies in-between that analysed by the two techniques (Appendix B). Talc is of interest to this study due to its froth stabilising effect, being naturally floatable and due to the potential formation of agglomerates in the presence of oil. The altered ore is expected to experience more severe effects compared to the normal ore, due to a slightly higher phyllosilicate alteration content. Other minerals detected include olivine, quartz, carbonate, amphibole, mica, and minor base-metal sulphides (BMS). The main BMS were chalcopyrite, pentlandite and pyrrhotite.

Table 4-2: QEMSCAN bulk mineralogical composition (wt.%) of the two different PGM bearing ores.

Mineral	**Normal ore (wt.%)**	**Altered ore (wt.%)**
Chalcopyrite	0.1	< 0.1
Pentlandite	< 0.1	< 0.1
Pyrrhotite	0.1	< 0.1
Other BMS	0.1	< 0.1
Chromite	42.4	33.9
Orthopyroxene	30.3	31.5
Plagioclase	11	20.9
Clinopyroxene	5.3	2.4
Talc	3.6	5.8
Chlorite	1.9	3.2
Amphibole	1.8	1.1
Olivine	1.0	0.1
Mica	0.9	0.4
Serpentine	0.7	< 0.1
Quartz	0.4	0.1
Carbonate	0.2	0.4
Other	0.2	0.2

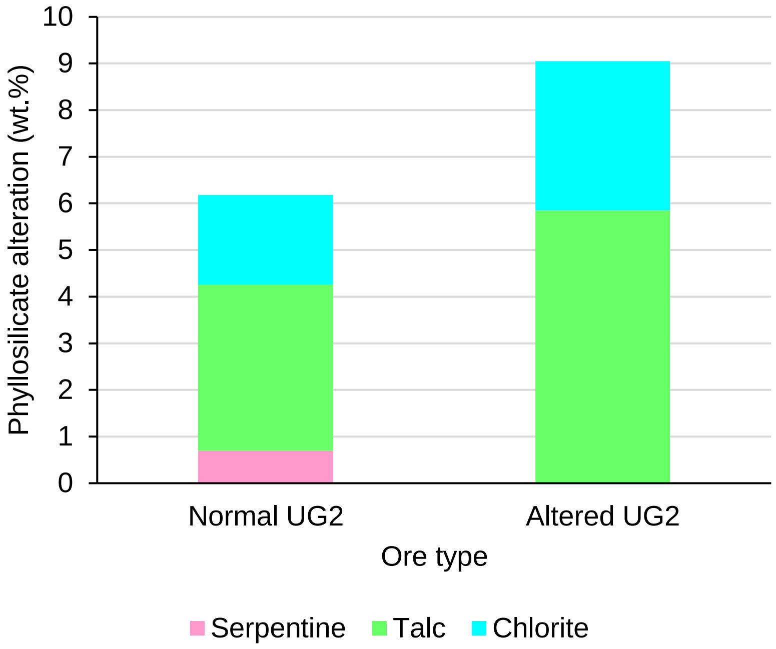

Figure 4-1: Comparison of the phyllosilicate alteration mineral contents within the normal and altered UG2 ores as determined with QEMSCAN.

4.1.3. PGM characterization and distribution

The distribution of various PGE species in the two feeds is illustrated in Figure 4-2. The results include gold which is normally part of the analysis, but not a PGE categorically. PGE sulphides are the dominant species with the PGM area of around 66 PGM area % in both ores. The dominant constituents comprising the PGE sulphides are cooperite, braggite and laurite. The second most common PGE in the normal ore was the arsenides group, predominantly sperrylite, with a PGE area of 19.2%. On the other hand, the PGE arsenides group in the altered ore was relatively low, at 3 PGM area %. The PGE alloys comprising of PdPb, PdSb, PtFe, PdAu and OsIrRu, was second most common group in the altered ore with 22 PGM area %. In the normal ore, the PGM alloys content was low, at around 10 PGM area %, with the same constituents except the PdAu alloy. The gold contents in the two ores were minor—0.5 and 3.9 in the normal and altered ores, respectively. Other PGE groups which appeared in minor quantities in both ores include the bismuthotellurides, bismuthides and sulpharsenides.

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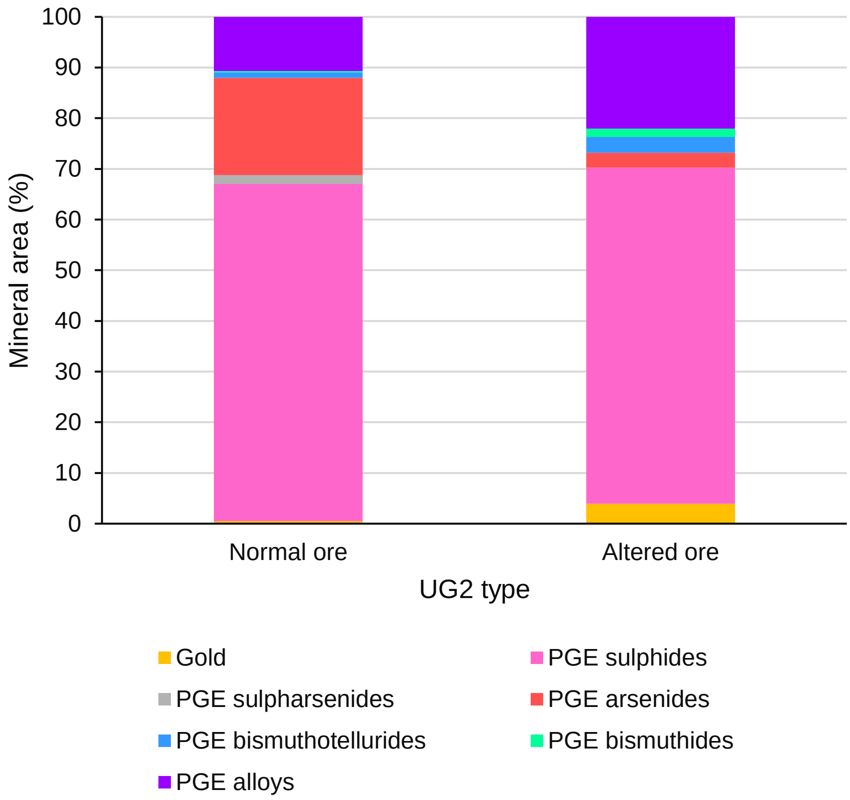

Figure 4-2: Distribution of the PGM in the two UG2 ores (Normal and altered). A total of 167 particles were analysed in the normal ore and 161 in the Altered ore. Secondary grinds were used (70%<75 µm).

It was also essential to understand the PGM grain size distribution within the two ores. This is critical in understanding the degree of milling required for liberation of PGMs. It can be deduced from Figure 4-3, that most PGM are fine to ultra-fine with grain sizes below 10 µm. This accounts for about 74% of the total particles analysed in the normal ore and 86% in the altered ores. Overall, the normal ore had a slightly coarser PGM grain size distribution than the altered ore; the coarsest PGM grain size examined in the normal ore had an equivalent circular diameter of 20 µm and 13 µm in the altered ore.

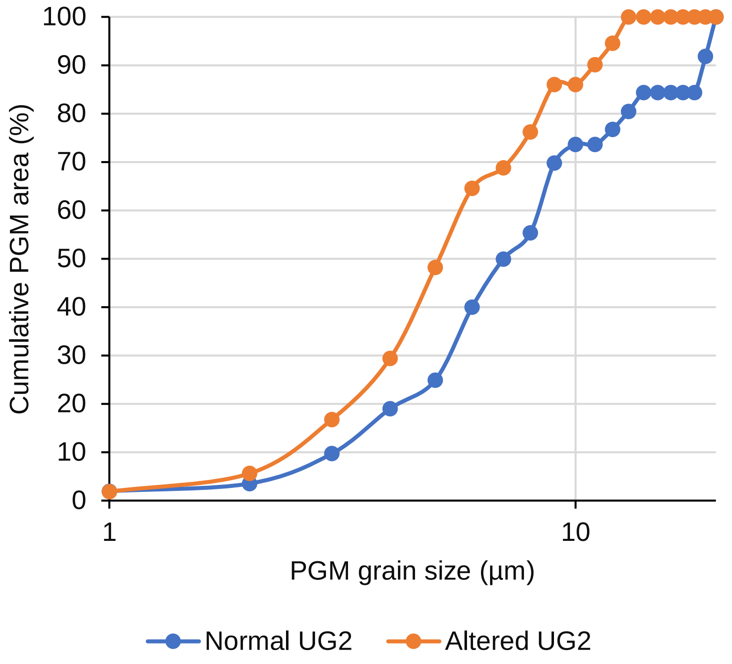

Figure 4-3: PGM grain size distribution (GSD) in the normal and altered ores. The number of particles analysed using QEMSCAN was 167 and 161 in the normal and altered ore. The PGM grain size represents the equivalent circle diameter.

Figure 4-4 depicts the mineral liberation and association results of the normal and altered ore, which is another quantitative analysis essential for PGM characterization. Comparing the two ores, major proportion of the PGMs were liberated in the normal ore (46 PGM area %) relative to the altered ore (25 PGM area %). A higher effective liberation % (Liberated PGM and PGM locked in liberated BMS(PGM/BMS)) was reported in the normal ore as compared to the altered ore- 67 PGM area % in the normal ore and 43 PGM area % in the altered ore. This inferred that a significant proportion of PGMs remained associated with gangue minerals and unliberated, about 33 and 57 % of the total PGMs in the normal and altered ore, respectively. This may be difficult to recover during flotation because a significant amount of PGMs may be rejected alongside the gangue minerals. The results show a strong association of PGM/BMS with silicates in the altered ore- approximately 17 PGM area %. Approximately 24 and 10 PGM area % were associated with oxides (enclosed and attached). About 11% of the PGM in the altered ore were in closed in the alteration silicates compared to 9% in the normal ore. In terms of association with oxide, 10% PGM were enclosed with the oxide ore in the altered ore compared to 7% in the normal ore. Overall, PGM and BMS had a stronger association with primary silicates and alteration silicate minerals than with oxide (chromite) minerals.

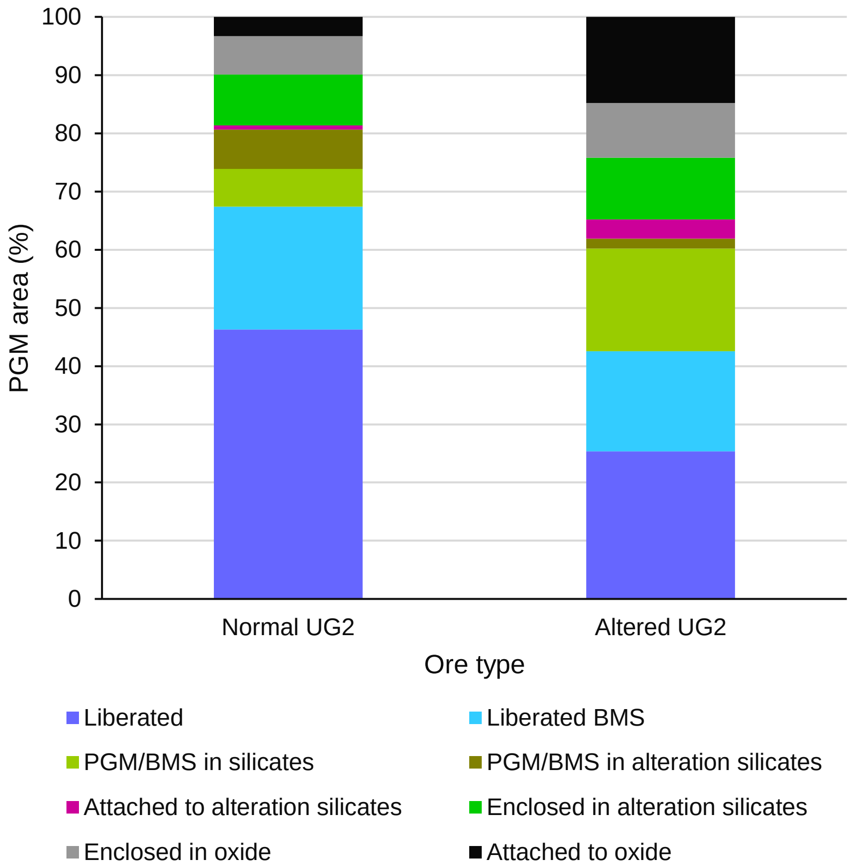

Figure 4-4: The proportion of liberated PGM and various minerals associated with them. The 'PGM/BMS' category refers to the PGM associated with BMS which are in turn associated with other minerals. The number of particles analysed using QEMSCAN was 167 and 161 in the normal and altered ores, respectively. A liberated PGM is defined has >80% of the PGM particle by area.

4.2. Baseline batch flotation

The intention of this section is to present the baseline flotation performance prior to any intentional 'contamination' with oil. The cumulative solids and water recovery data obtained from the normal and altered ore batch flotation experiments are presented in Figure 4-5. Greater mass of solids were recovered in the altered ore in comparison to the normal ore. On the contrary, a greater water recovery was observed in the normal ore float.

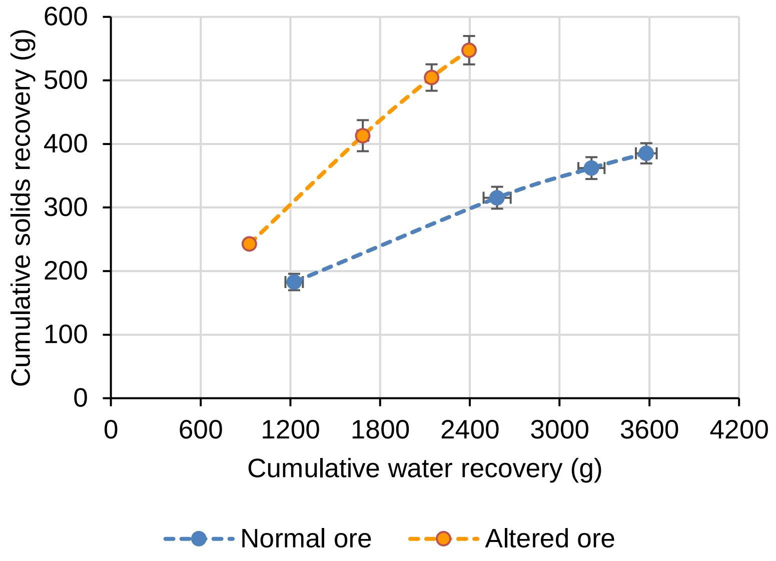

Figure 4-5: The baseline cumulative solids recovered as a function cumulative recovered for the normal and altered ore. Error bars represent standard error of duplicate floats.

The baseline flotation performances of Pt and Pd as function of percentage solids recovered are shown in Figure 4-6. The recovery of Pt and Pd (Figure 4-6 a-b) in both ores follows a similar trend, which corresponds to the conventional flotation behaviour in which recovery increases with the percentage solids recovered. The highest recovery for both metals was observed in normal ore and at lower solids recovery in comparison to altered ore (9.45 versus 13.45 wt.%). It is clear that Pt and Pd recovery versus solids recovery curve is shifted up and that the normal ore gives higher Pt and Pd recoveries for any given solids recovery. In addition to this, when the Pt and Pd grades (Figure 4-6 c-d) in the normal and altered ore floats were compared, the normal ore gives higher grades than the altered ore for any given solids recovery.

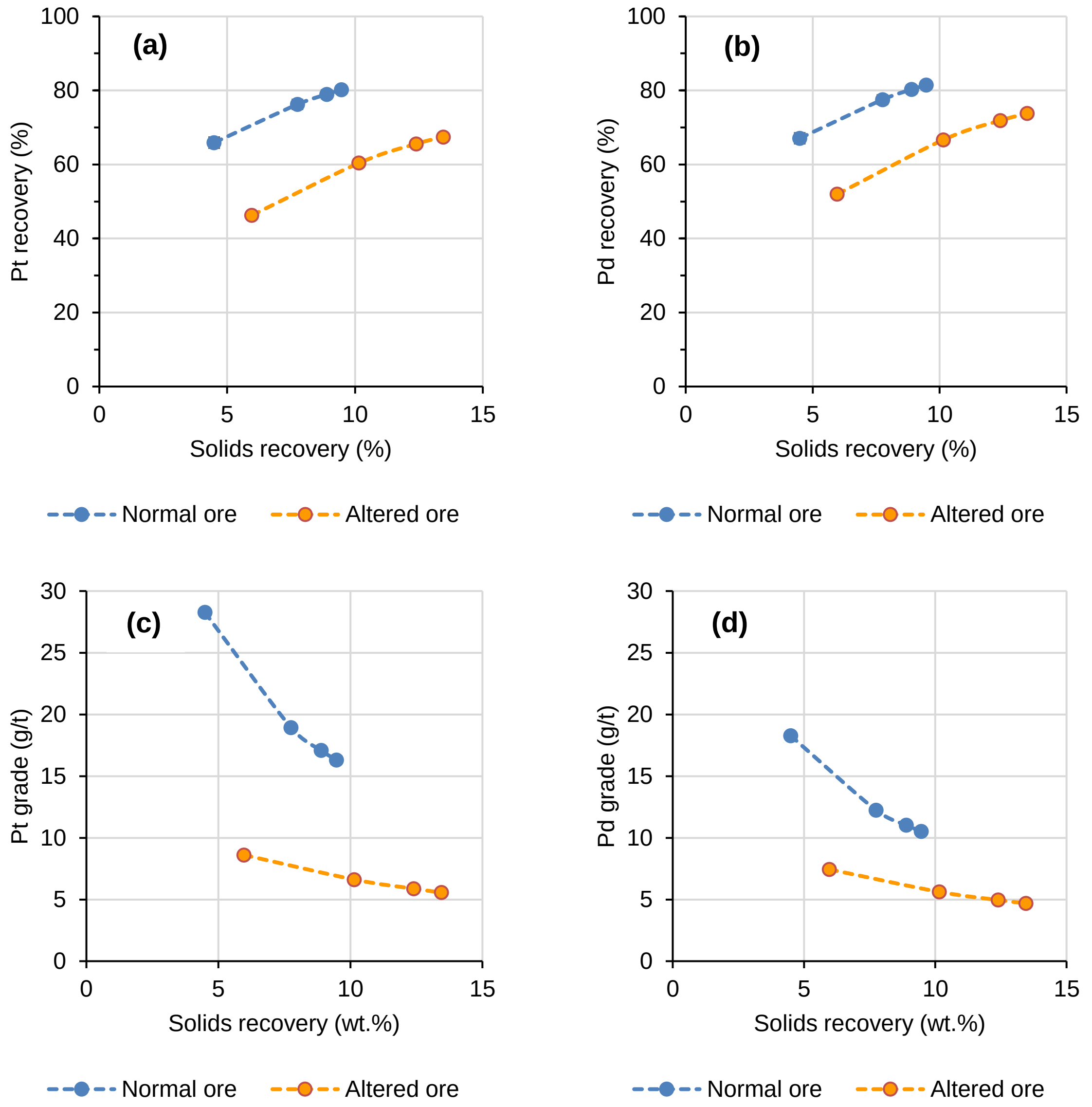

Figure 4-6: Pt (a) and Pd (b) cumulative recovery at various percent solids recovered. Pt (a) and Pd (d) cumulative grade at various percent solids recovered. Error bars represent standard error of duplicate runs.

4.3. Effect of oil addition on the normal ore

This section addresses the flotation behaviour of the normal ore in the presence of oil. This includes the effect of oil on flotation grade and recovery as well as the key supplementary measurements such as rheology, oil adsorption, froth stability tests needed to decouple the multiple effects.

4.3.1. Batch flotation

4.3.1.1 Baseline tests

Figure 4-7 depicts the cumulative solids and water recovered, as well as the total solids and water recovered at various oil concentrations. Increasing oil concentration resulted in an increasing trend of both solids and water recovered with the highest solids and water recovered at higher oil concentrations. The solids recovered increased by about 45% from 0 g/t to 500 g/t oil addition, whereas the water recovered increased by 26%.

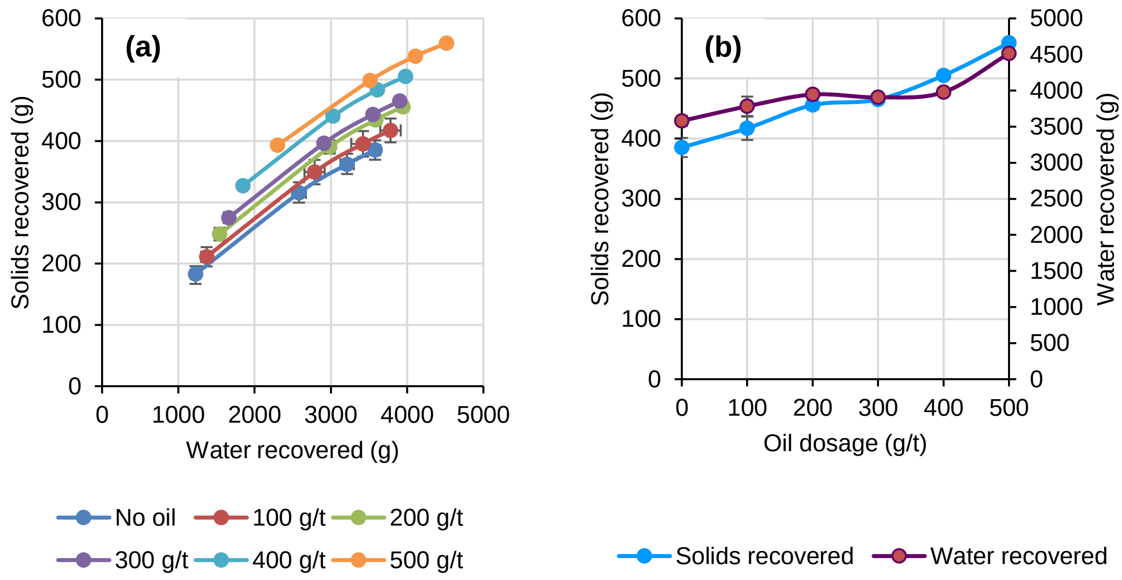

Figure 4-7: (a) Cumulative mass of solids and water recovered at different oil concentrations. (b) Total solids and water recovered at various oil concentration. Error bars represent the standard error of duplicate floats.

4.3.1.2 Response to oil addition

The normal ore flotation response to oil with respect to recovery and grade is illustrated in Figure 4-8. The results yielded multiple key observations. Varying the oil concentration (0-500g/t) resulted in an increase in Pt recovery of 4% and constant Pd recovery of roughly 80%, with the exception of 400 g/t which could be an outlier or an assay error. In contrast to the recovery, the oil had a significant impact on the grade with both Pt and Pd grades decreasing by roughly 3 g/t from the “zero oil” to the 500 g/t oil addition. A gradual, almost linear, decrease in grades was observed with increasing oil concentration.

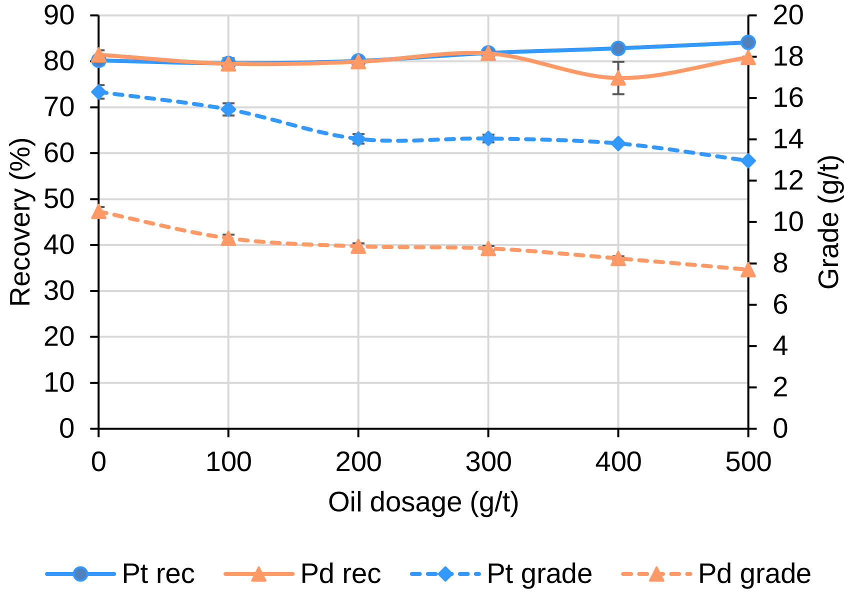

Figure 4-8: Final Pt , Pd grade and recovery at various oil concentrations. Error bars represent the standard error of duplicate floats.

To further assess the effect of oil on flotation performance of the normal ore, Pt and Pd recovery as a function of solids recovered as well as Pt and Pd grade versus recovery were plotted in Figure 4-9. The Figure 4-9 a-b shows that the recovery increases with solids recovery, as expected. At a constant solids recovery, the Pt and Pd recovery decreases as the oil concentration increases. This is particularly clear for Pd, where at 400 g of solids recovery, Pd recovery drops from approximately 80% percent in the absence of any oil to approximately 70% when 400 g/t of oil is present. The effects of this can also be seen in the grade versus recovery curves where for a constant grade, especially Pd, the recovery decreases with increasing oil concentration.

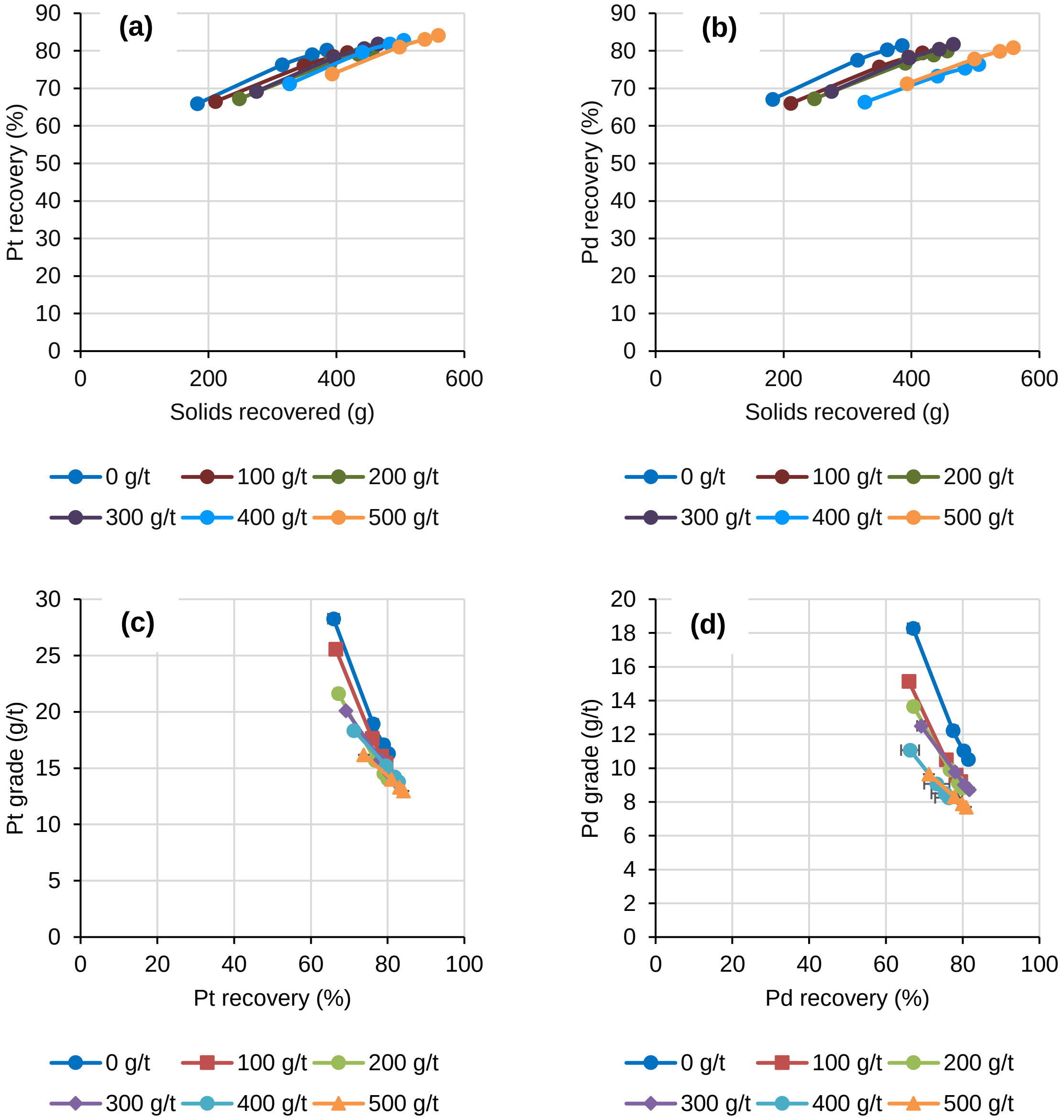

Figure 4-9: Cumulative Pt (a) and Pd (b) recovery as a function of mass pull for different oil concentration. Pt (c) and Pd (d) grade versus Pt recovery curve at various oil concentration. Error bars represent standard error of duplicate runs.

The two metal grades as a function of mass pull and water recovery were plotted in Figure 4-10. The grade was of particular importance because it was adversely affected by the presence of oil. As the amount of oil added increases, there is a small downward shift in the grade versus solid recovered curves, while the grade versus water recovered curves experience a greater downward shift. This suggests that the addition of oil has a bigger impact on water recovery, and that the decrease in grade is mostly caused by entrainment, with only a secondary effect of true flotation.

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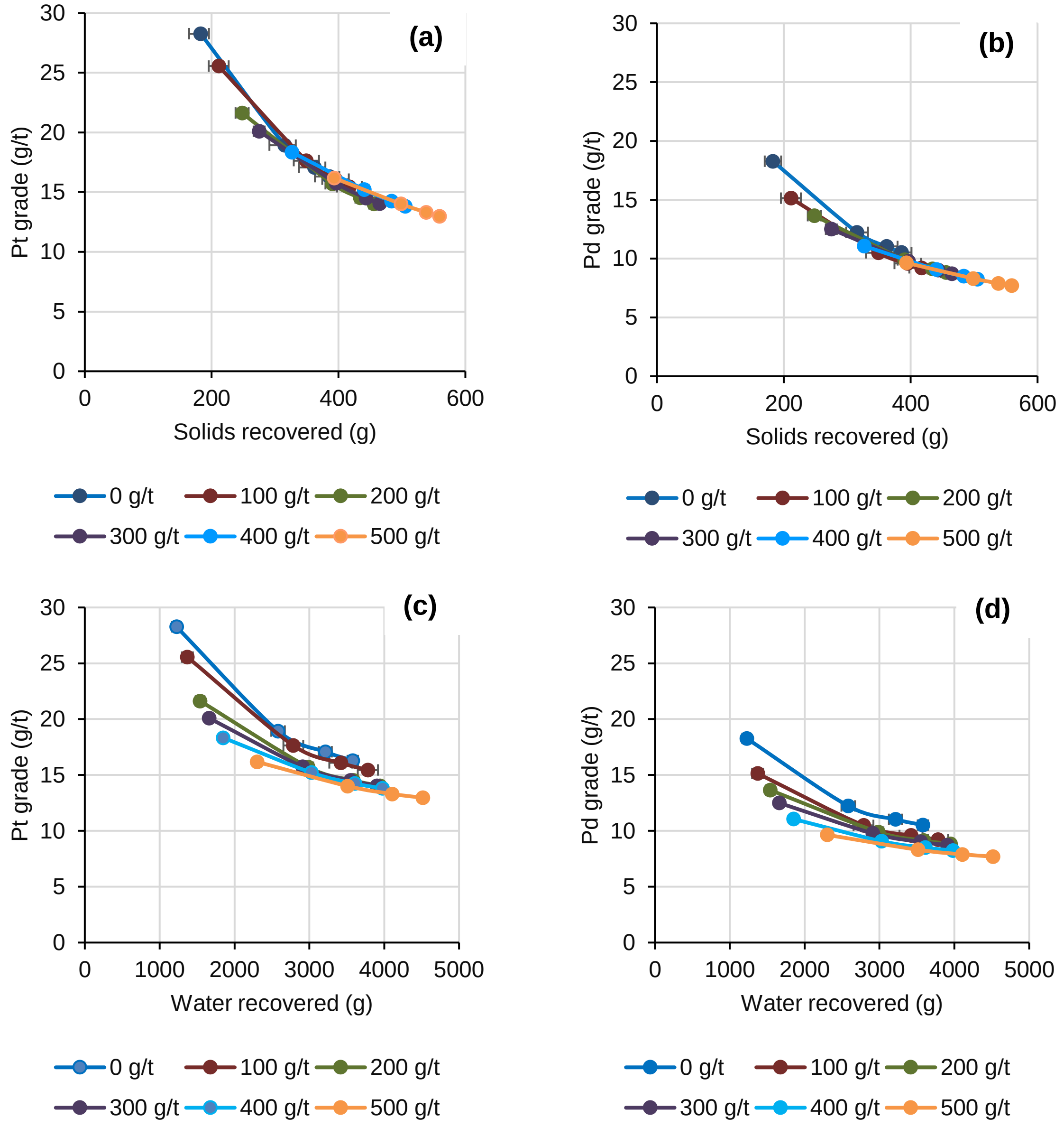


Figure 4-10: Pt(a) and Pd (b) grade as a function of mass recovered. Pt (c) and Pd (d) grade as a function of water recovered. Error bars represent standard error of duplicate runs.

4.3.1.3 Concentrate mineralogy

The bulk mineralogy was performed on selected combined flotation concentrates using the QEMSCAN approach to study the effects of oil from a mineralogical standpoint. The primary goal was to compare and identify the mineral (s) responsible for grade dilution resulting from oil addition. Also, investigate the effect of oil on chromite recovery since this might affect the downstream smelter problem as high chromite levels might lower the operating volume of the furnace, as covered in section 2.1.2. Figure 4-11 depicts the bulk composition of the two concentrates analysed. Mineralogical composition of the two concentrates (0 and 500 g/t oil) was similar, with the largest components being orthopyroxene, talc, plagioclase, and chromite. More of these minerals were recovered in the presence of oil (500 g/t), a difference of 50 g in orthopyroxene content between the 100 and 500 g/t concentrates, 19 g for talc, 21 g for plagioclase, and 50 g for chromite. These showed that there was unselective recovery of gangue minerals as a result of oil addition, including both hydrophobic (talc) and hydrophilic (orthopyroxene, plagioclase, and chromite) minerals. It is also worth mentioning that the very high talc recovery in the two concentrates suggests that the depressant dosage could be further optimized. Additionally, the Cr_2O_3 grade in the two concentrates (around 9 wt.% for both ores, Appendix B-XRF) was relatively higher in comparison to the acceptable level in the smelter (~3 wt.%), as previously stated in Chapter 2. This infers that further cleaning of the concentrate would be required to reduce the Cr_2O_3 grade.

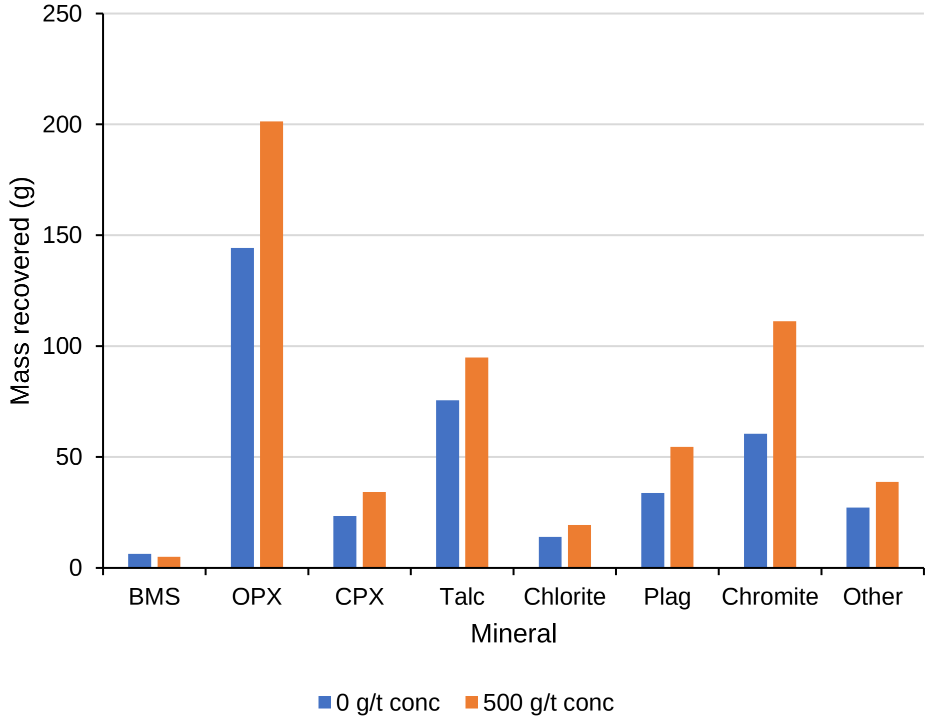


Figure 4-11: Bulk mineralogical composition of the concentrate at 0 and 500 g/t oil. OPX is orthopyroxene, CPX is clinopyroxene, Plag is plagioclase and other represents the remaining minerals which were recovered in small quantities.

4.3.1.4 Used versus fresh oil

Up to this point, the fresh oil was used for all batch tests. Used hydraulic oil of the same type as the fresh oil was utilized to assess if there would be changes in the flotation performance owing to the nature or condition of the oil. Figure 4-12 depicts the comparison of Pt, Pd recovery and grade at two selected fresh and used oil dosages (200 g/t and 300 g/t). The grade and recovery for both Pt and Pd were similar, with no significant difference between used and fresh oil at the dosages.

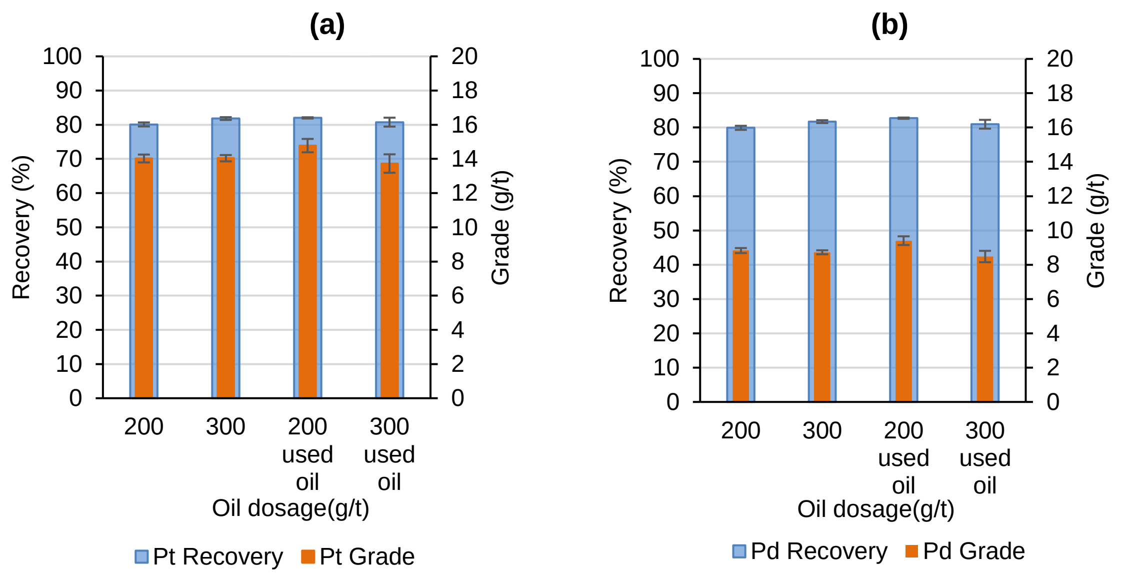


Figure 4-12: (a) A comparison of Pt recovery and grade as function of fresh and used oil dosage. (b) A comparison of Pd recovery and grade as function of fresh and used oil dosage. Error bars represents standard error of duplicate runs.

4.3.1.5 Oil deportment

An investigation of the oil adsorption was thereafter carried out to determine if the oil was adsorbing on the particle surfaces or remained unabsorbed in the bulk liquid. In the instance of adsorption, this would eventually confirm whether the target was the more hydrophobic minerals. The absorbed carbon content measured using the total organic carbon (TOC) analyser at different initial oil dosages is shown in Figure 4-13 a. Figure 4-13 b illustrates the oil adsorbed obtained using the petroleum ether extraction method. Only the 500 g/t condition was conducted in the solvent extraction method. The TOC experiment showed that nearly all of the oil was adsorbed onto the feed solids. The oil-solvent extraction method, on the other hand, extracted 342 g/t of oil from the ore. This accounted for about 71 % of the initial oil (500 g/t). The rest of the oil likely remained in the slurry suspension unabsorbed or adsorbed to the mill shell. Even though the absolute values of the two methods varied, both methods showed that the majority of the oil adsorbs on the particle surfaces.

Additionally, the oil extraction method was performed on the combined concentrate from tests at 500g/t oil to further investigate if the adsorbed oil was recovered in the froth zone along with the particles or remained in the pulp zone. The flotation cell mass balance results are shown in Figure 4-14. The mass balance shows non-selective recovery of oil coated particles to either the concentrate or the bulk tails. The oil per solids recovered was nearly the same in the concentrate and tails, at around 490 g/t. Around 13% of the initially administered oil was recovered with the concentrates, and around 88% in the tails.

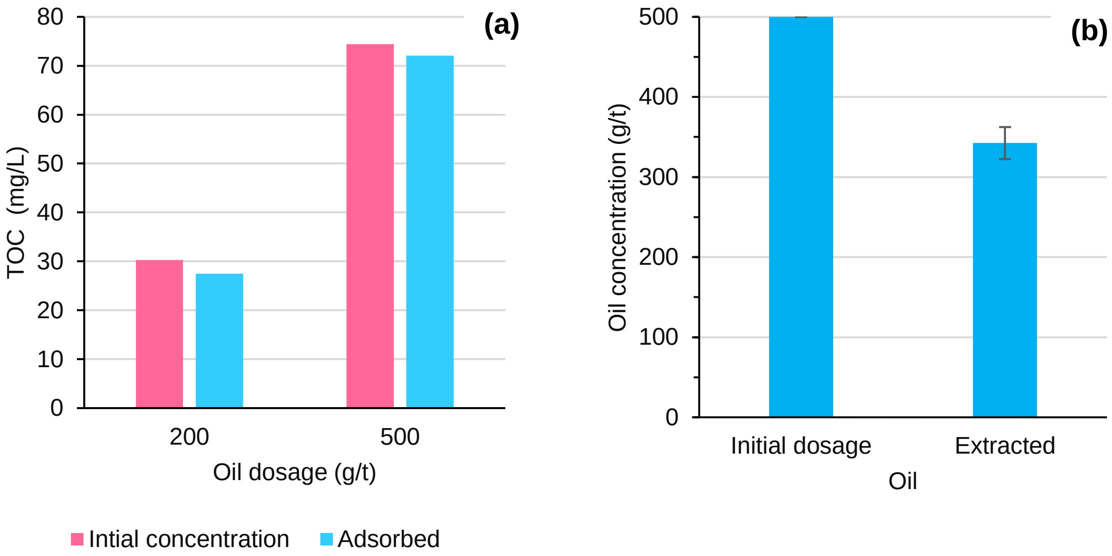


Figure 4-13: (a) The oil adsorption studies using the total organic carbon analyser. The carbon concentration is represented in mg/L. The initial concentration represents the TOC concentration in the initial administered oil. The adsorbed bars represent the amount of oil adsorbed on the mineral particles. (b) The extraction of oil from the milled feed with petroleum ether. The error bars represent the standard error of duplicate tests.

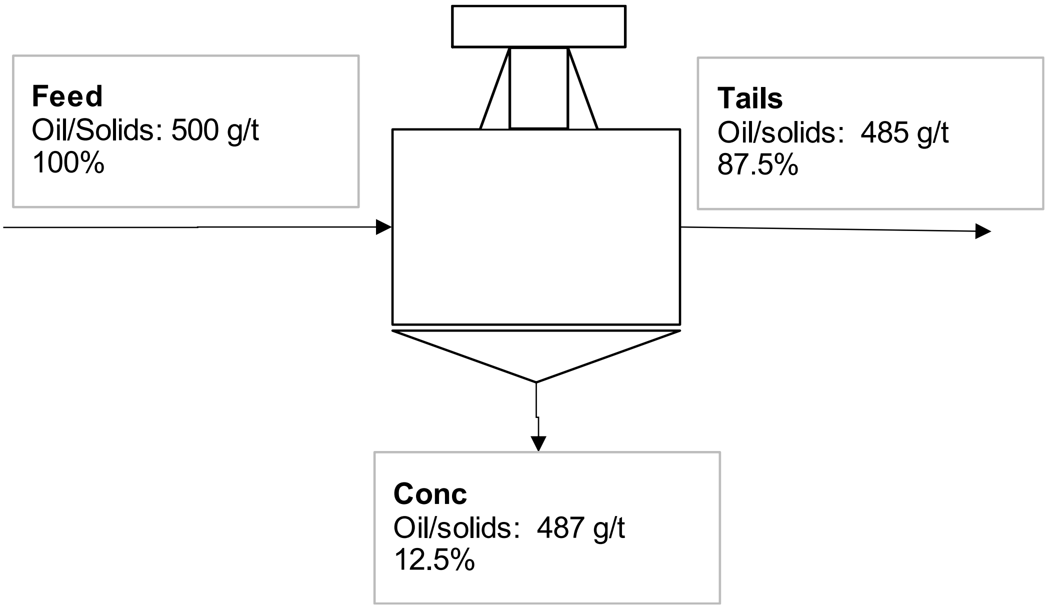


Figure 4-14: Oil mass balance conducted around the flotation cell. This was based on the oil added on the feed prior to milling, extracted oil from combined concentrates (c1 to c4) using petroleum ether and oil in the bulk tailings.

4.3.2. Rheology

Rheological measurements were conducted to assess the possibilities of rheological complexity arising in a flotation cell due to the presence of oil; their results are presented in Figure 4-15 and Figure 4-16. Figure 4-15 illustrates a rheogram of the normal ore, performed at a constant solids concentration (40 vol.%) and different oil concentrations. The Bingham model was applied to obtain the linear rheogram, confirming plastic behaviour. The regression fit (r) for all investigated concentrations was over 0.90. This figure clearly shows that the shear stress increases as oil concentration increases. This was also true for yield stress (y-intercept from the equations), which increased as oil concentration increased.

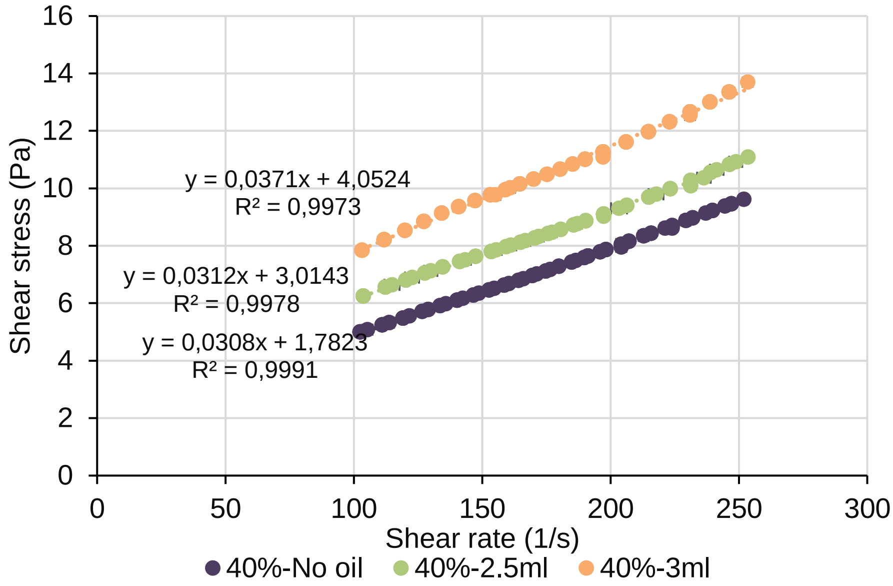


Figure 4-15: Bingham modelled rheogram of the normal ore (secondary grinds: 70 %<75 µm) at various oil concentrations. Error bars (which are too small to be seen) represents a standard error of triplicate runs.

Figure 4-16 illustrates the effect of oil on the Bingham apparent viscosity of the normal ore. The operating shear rate of a mechanical flotation cell is around 160/s (Shabalala et al., 2011); therefore, all apparent viscosities were reported at this shear rate. The results indicate an increase in the slurry's apparent viscosity at elevated oil concentrations. This indicates that rheological complexity is more likely to occur at higher oil concentrations.

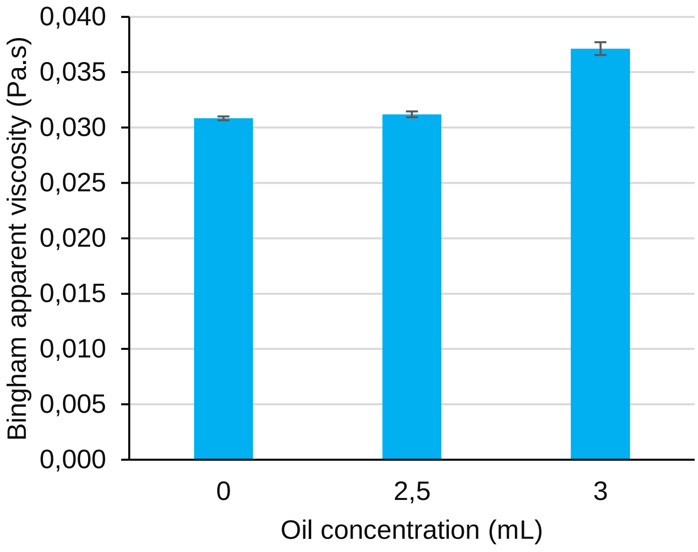


Figure 4-16: Bingham apparent viscosity of the milled normal ore feed (Grinds: 70 %<75 µm) as a function of oil concentrations. Samples with solids concentration of 40 vol% were used for all the oil concentrations. The error bars represent the standard error of triplicate runs.

4.3.3. Froth stability

This section will investigate the effect of oil on froth stability of the normal ore. Figure 4-17 portrays the 3-phase dynamic froth stability at various oil concentrations. The key observation from Figure 4-17 was the increase in froth stability as oil concentration increased. The overall increase in froth stability over the investigated range was 34%. The highest froth stability factor was 82 s at 400 g/t oil. From 0-100 g/t, the froth stability was nearly constant after which an increase was observed.

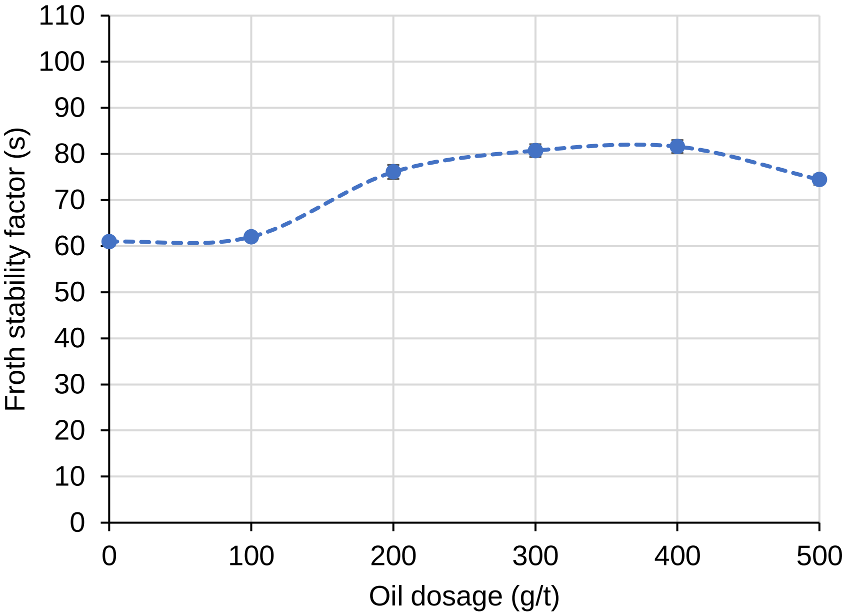


Figure 4-17: The effect of varying oil concentration on 3-phase froth stability of the normal ore. The error bars represent the standard errors of triplicate measurements.

4.4. Effect of oil addition on the altered ore

This section presents the altered ore behaviour in the presence of oil. Key experimental results will be presented, including batch flotation, rheology, oil adsorption, and froth stability.

4.4.1. Batch flotation

4.4.1.1 Baseline tests

The effect of oil on solids and water recovered during flotation is summarized in Figure 4-18: (a) demonstrates the cumulative solids recovered as function of water recovered and (b) illustrates the total solids and water recovered for each individual oil concentration. The two figures show that the solids and water recovered increased marginally from 0 g/t to 100 g/t before plateauing. This was an interesting observation because the solids and water recovered were expected to increase with increasing oil concentration, as with normal ore (Figure 4-7), and due to the probable agglomeration of minerals such as talc in the presence of oil. When comparing the solids recovered for each investigated oil concentration between the normal (Figure 4-7) and altered ore, the altered ore had higher recoveries. Another key observation was that the total water recovered in the altered ore was relatively low compared to the normal ore.

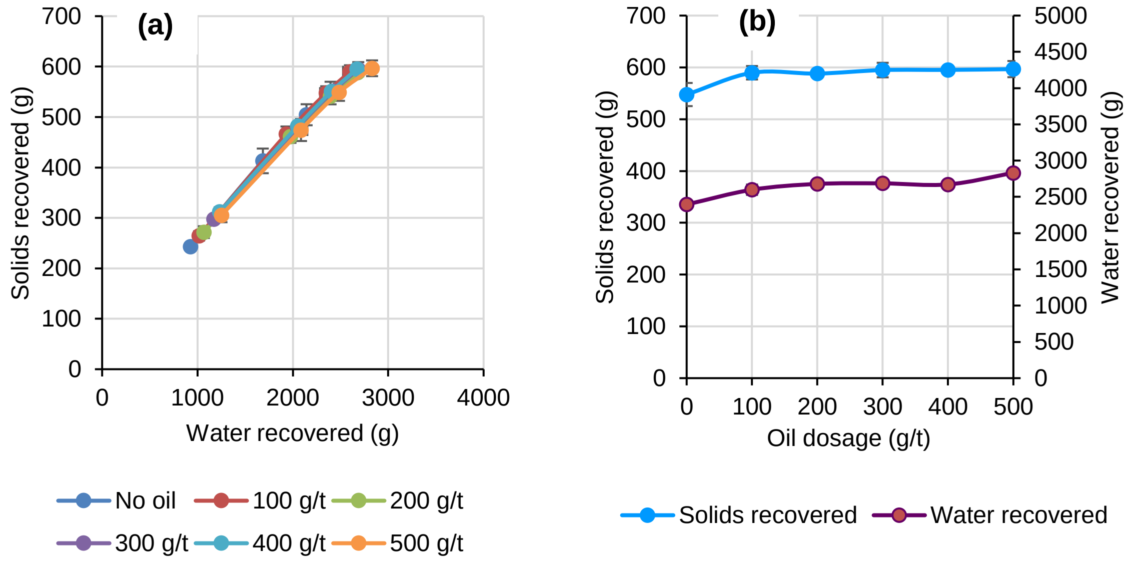


Figure 4-18: Cumulative mass of solids recovered, and water recovered at various oil concentrations. (b) Total solids and water recovered at various concentrations. Error bars represent the standard error of duplicate floats.

4.4.1.2 Response to oil addition

Figure 4-19 depicts the final Pt, Pd recovery and grade for various oil dosages. The first point to note is that the baseline Pt and Pd recoveries in the absence of oil are lower for the altered ore relative to the normal ore (Figure 4-8). Pt and Pd recoveries were 67.4% and 73.8%, respectively, for the altered ore and 81.4% and 79.6%, respectively, for the normal ore.

The oil had a significant impact on the recovery of both Pt and Pd but had no effect on their grades. A gradual decrease in Pt and Pd recovery was observed from 0 to 200 g/t oil, after which it became constant until 500 g/t oil condition. The Pt recovery decreased by 6% from the baseline recovery of 67% to 61% at 500 g/t oil. Pd experienced a recovery drop of 12% over the investigated oil range. Therefore, the oil had a more severe effect on Pd relative to Pt. In terms of the Pt and Pd grade, the addition of different oil dosages had no detrimental effect. The grades for both metals were nearly constant, Pt at around 5 g/t and Pd at around 4 g/t over the entire oil range.

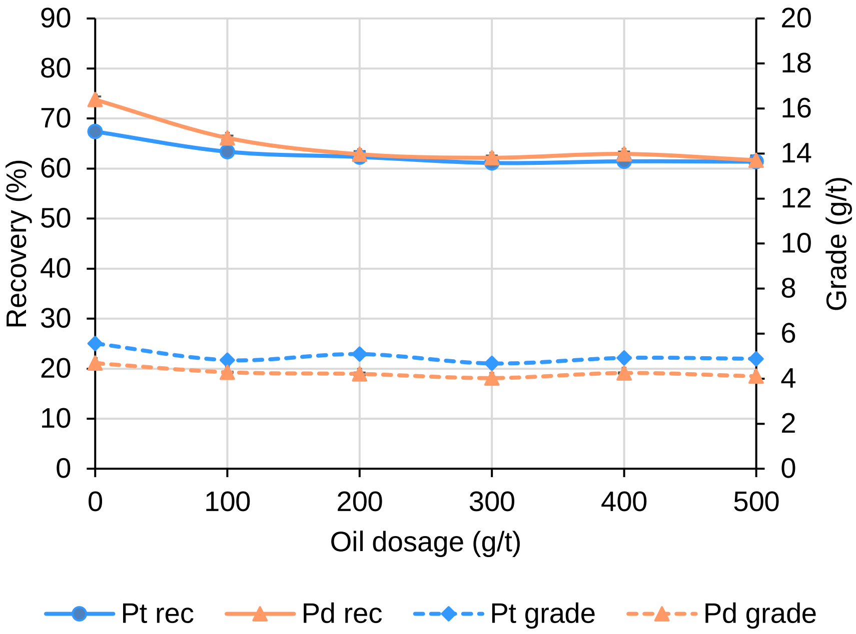


Figure 4-19: Final Pt, Pd grade and recovery at various oil concentrations. Error bars (which are too small to be seen) represent the standard error of duplicate floats.

Figure 4-20 illustrates the Pt and Pd recovery of the altered ore as a function of solids recovered (a-b) and their grade-recovery curves (c-d). Figure 4-20 a-b indicate that there is a downward shift in the Pt and Pd recoveries as a function of solids recovered when going from zero oil addition to 100 g/t oil addition, indicating that there is a reduction in recovery at the same mass pull. Thereafter, the different oil dosages lie on a similar curve. This corresponded to the grade-recovery curves (Figure 4-20 c-d), which demonstrated that all oil concentrations lie on a similar profile.

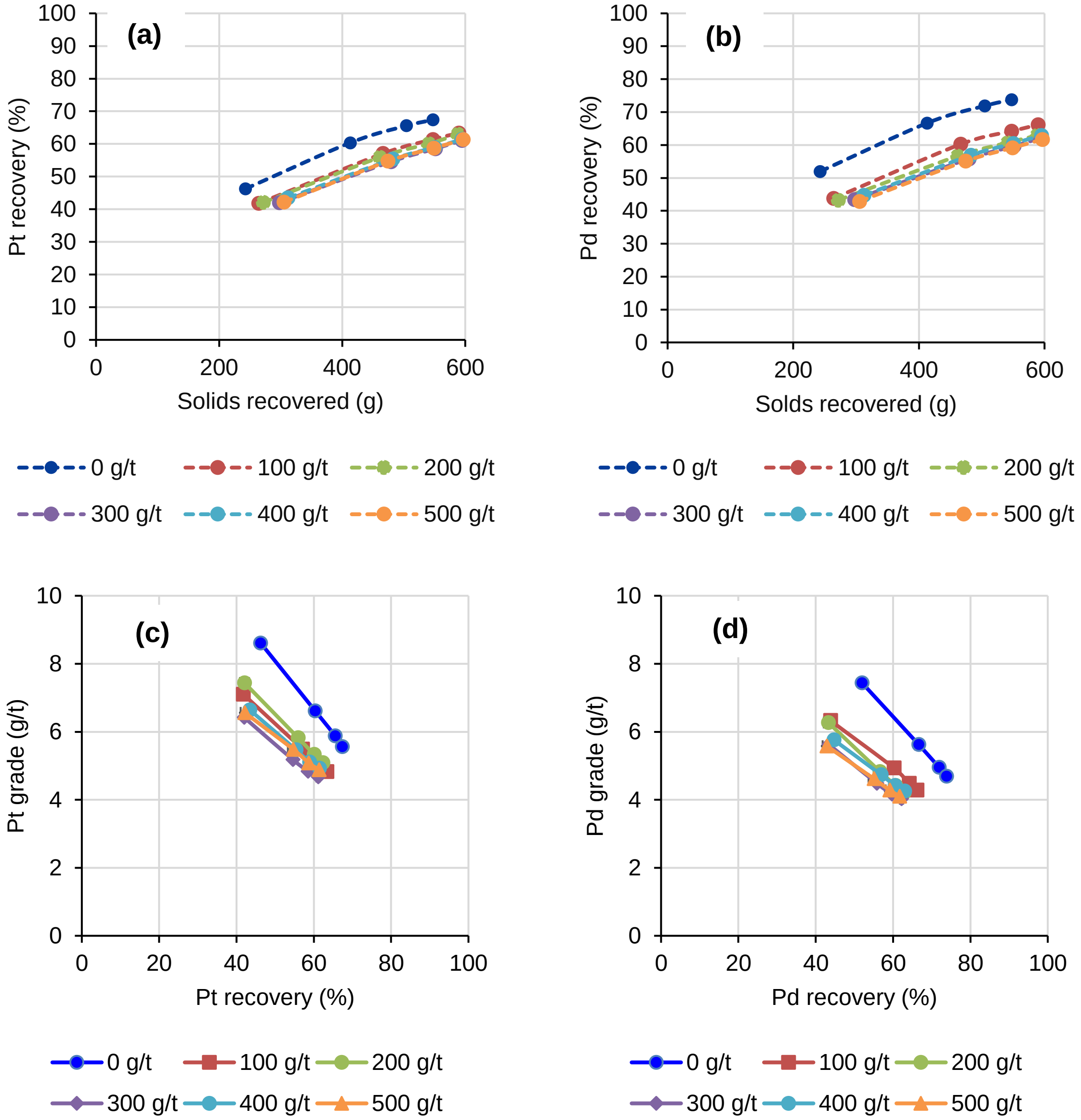


Figure 4-20: Cumulative Pt (a) and Pd (b) recovery as a function of mass pull for different oil concentration. Pt (c) and Pd (d) grade versus Pt recovery curve at various oil concentration. Error bars represent standard error of duplicate runs.

To further understand the effect of oil on the altered ore performance, Pt and Pd recovery at various oil dosages were plotted as function of water recovered and their grades as function of solids recovered as shown in Figure 4-21. Figure 4-21 a and b demonstrate that at a constant water recovery, the recovery (Pt and Pd) decreases with increasing oil concentration. It was more apparent with Pt recoveries at 2000 g water recovery. Similar trends were observed in the grades as well as shown in Figure 4-21 c-d.

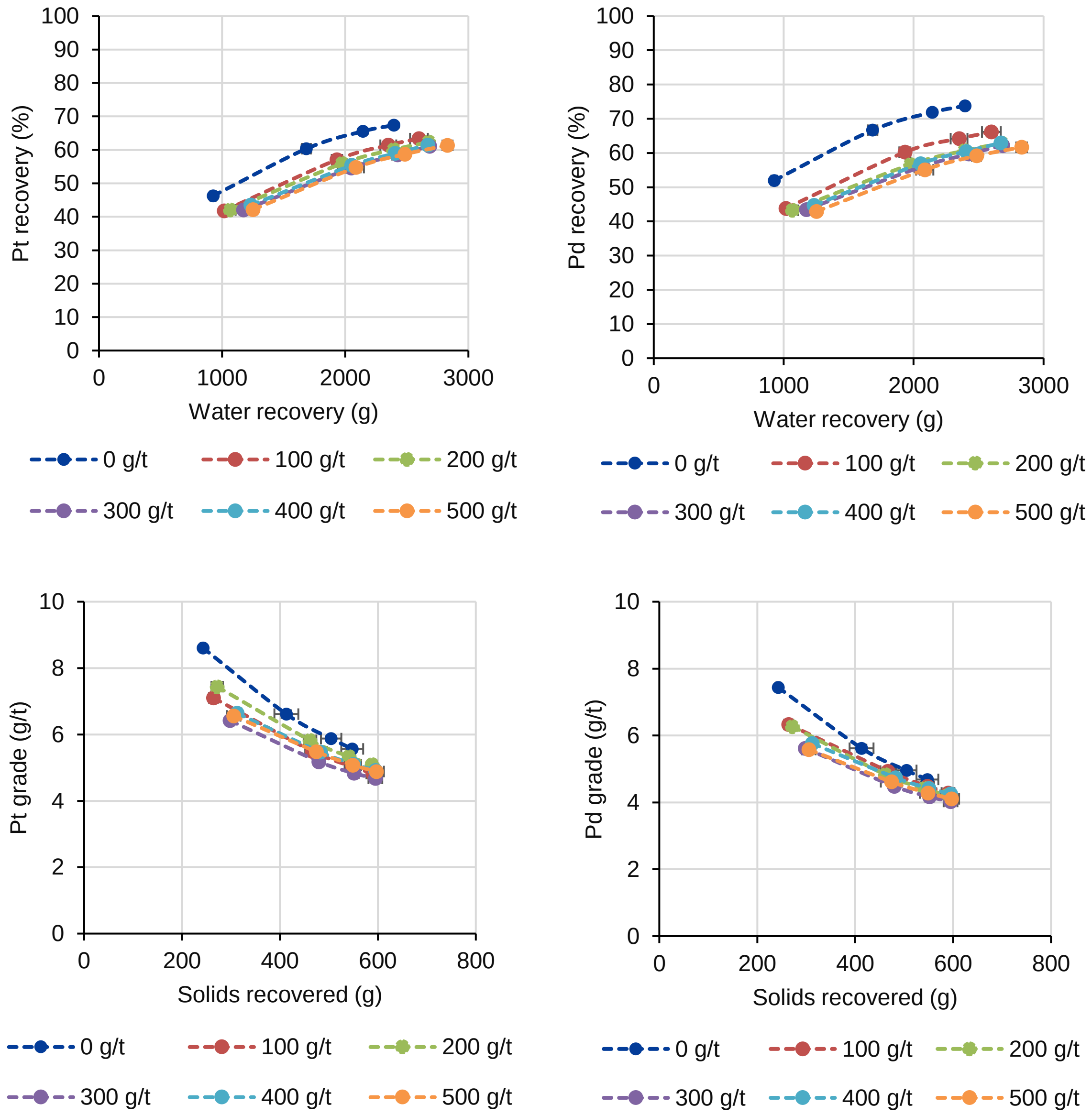


Figure 4-21: Cumulative Pt (a) and Pd (b) recovery as a function of water recovered at different oil concentration. Pt (c) and Pd (d) grade as a function of mass pull at various oil concentration. Error bars represent standard error of duplicate runs.

4.4.1.3 Concentrate mineralogy

QEMSCAN analysis was performed on the altered ore flotation concentrate, as with the normal ore, to gain an understanding of the minerals recovered as a result of oil addition; the results are shown in Figure 4-22. The quantities of minerals recovered in the two flotation concentrates were comparable. This was consistent with the observed grade trends (Figure 4-19) where no significant reduction was observed with varying oil dosage. Similar to the normal ore, it appears that the addition caused a slight increase in recovery of the major minerals present, orthopyroxene, talc, plagioclase, and chromite. Another key observation from Figure 4-22 was the lower BMS content in the 500 g/t concentrate compared to the baseline concentrates. The BMS content in the 0 g/t oil concentrates was 0.74 wt.% and 0.34 wt.% in the 500 g/t oil concentrates. This was consistent with the observed drop in Pt and Pd recovery due to oil addition (Figure 4-19). This was also consistent with the Cu and Ni trends (Refer to Appendix B).

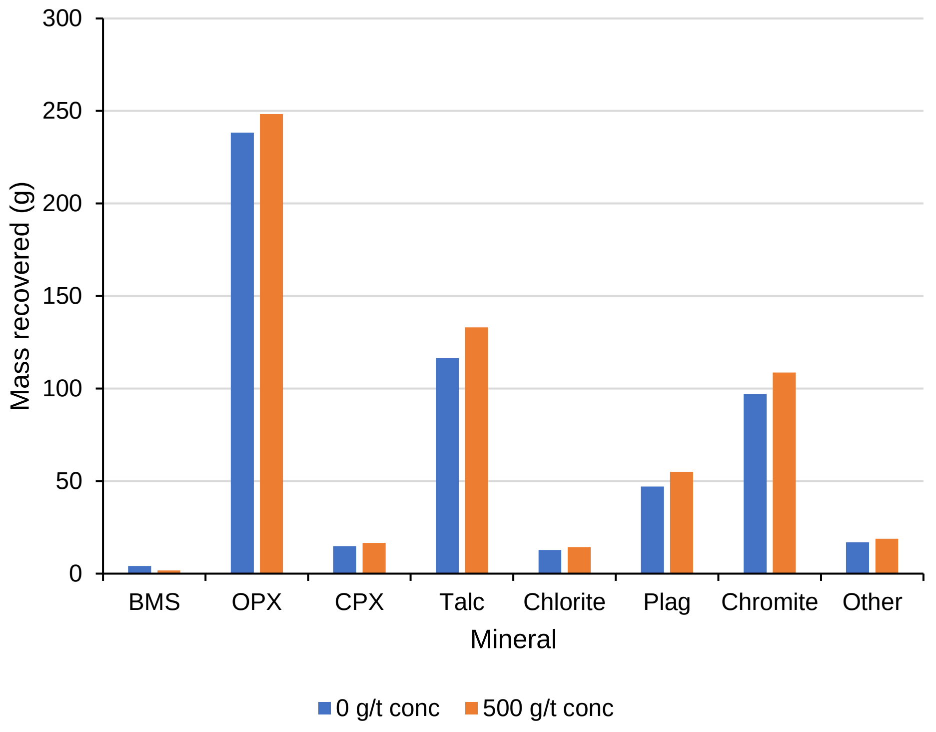
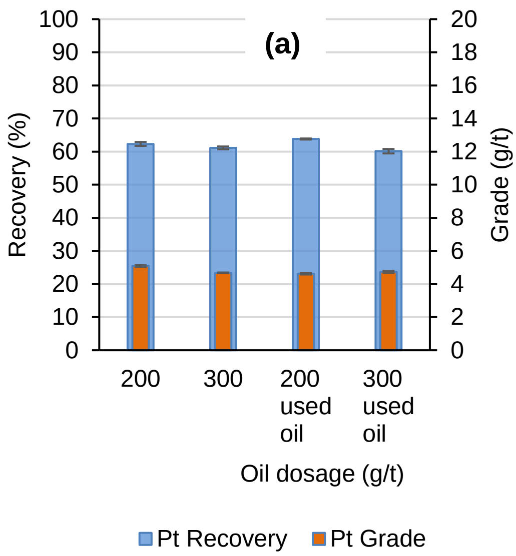


Figure 4-22: Bulk mineralogical composition of the concentrate at 0 and 500 g/t oil. OPX is orthopyroxene, CPX is clinopyroxene, Plag is plagioclase and other represents the remaining minerals which were recovered in small quantities.

4.4.1.4 Used versus fresh oil

Flotation experiments were also conducted with the used hydraulic oil to assess the flotation performance with a different oil. The results of used and fresh oils are shown in Figure 4-23. No substantial differences of Pt and Pd were observed for both the grade and recovery when using either fresh or used oil.



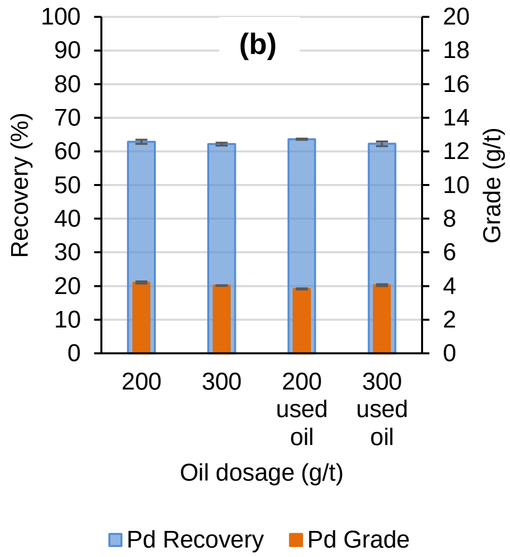


Figure 4-23: (a) A comparison of Pt recovery and grade as function of fresh and used oil dosage. (b) A comparison of Pd recovery and grade as function of fresh and used oil dosage. Error bars represent standard error of duplicate runs.

4.4.1.5 Oil deportment

The oil adsorption studies performed on the altered ore feed samples are displayed in Figure 4-24 (a), which shows the TOC adsorption at different oil dosages; and (b) with shows oil extraction with petroleum ether. Figure 4-24(a) demonstrates that most of the oil adsorbed on the particle surfaces for both investigated oil concentrations. This result was validated with the oil extraction method conducted at an initial oil dosage of 500 g/t, where 354 g/t of adsorbed oil was extracted from the feed post milling. The oil adsorbed determined using the oil extraction method was similar to the normal ore (342 g/t oil adsorbed). This infers that there was no preferential oil adsorption onto the greater amount of phyllosilicate minerals that were present in the altered ore.

As shown in Figure 4-24 and earlier in Section 4.3.1.5, the TOC method demonstrated that almost all the oil adsorbed compared to the extraction method, which was about 70% adsorbed. The difference could be in the analysis method, as the residual liquid was analysed in the TOC and the adsorbed oil was determined by the extraction of the oil coating the particles using petroleum ether as a solvent.

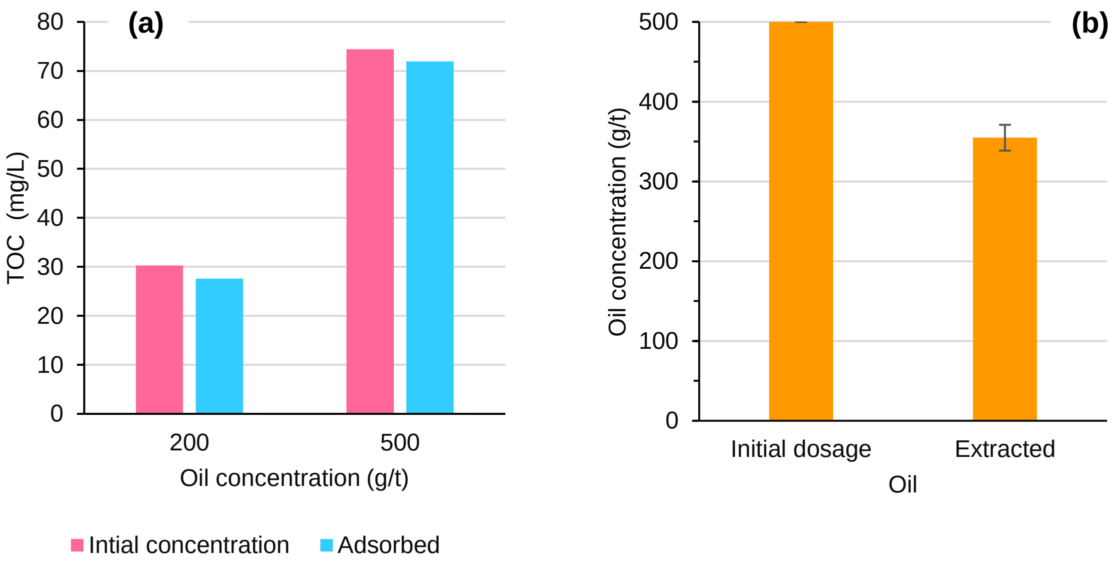


Figure 4-24: (a) The oil adsorption studies using the total organic carbon analyser. The carbon concentration is represented in mg/L. The initial concentration represents the TOC concentration in the initial administered oil. The adsorbed bars represent the amount of oil adsorbed on the mineral particles. (b) The extraction of oil from the milled feed with petroleum ether. The error bars represent the standard error of duplicate tests.

Oil extraction with petroleum ether was also performed on the 500 g/t batch concentrate. The flotation cell mass balance results are displayed in Figure 4-25. The results indicate that most of the oil coated solids reported to the bulk tailings as compared to the concentrate. The oil per solids mass concentration was 555 g/t in the tails and 232 g/t in the concentrate. This is a significant difference which infers that during flotation most of coated oil particles are in the pulp phase.

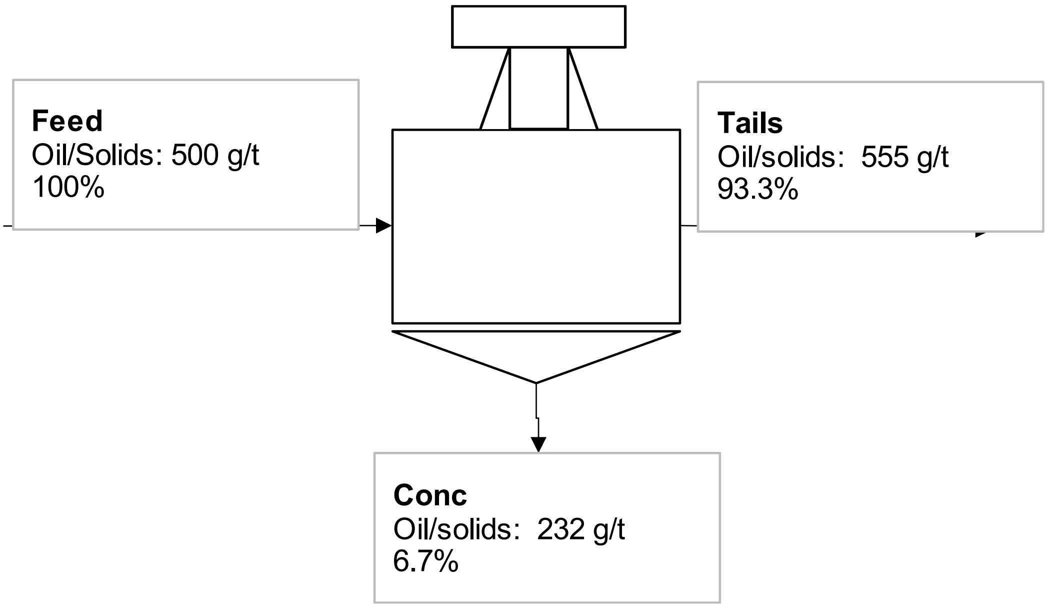


Figure 4-25: Oil mass balance conducted around the flotation cell. This was based on the oil added on the feed prior to milling, extracted oil from combined concentrates (c1 to c4) using petroleum ether and oil in the bulk tailings.

4.4.2. Rheology

Figure 4-26 shows the rheogram of the altered ore at various oil concentrations. Similar to the normal ore rheogram (Figure 4-15), the shear stress of the fitted Bingham model increased as a result of increasing oil concentration, and this became apparent at higher oil dosages. A good correlation coefficient of over 0.99 was achieved for all concentrations.

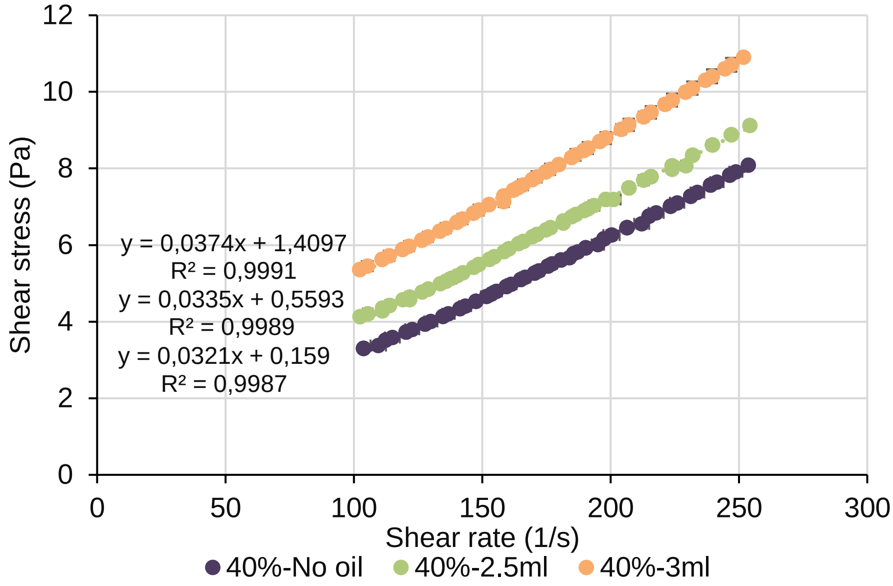


Figure 4-26: Bingham modelled rheogram of the altered ore (secondary grinds: 70 %<75 µm) at various oil concentrations. Samples with solids concentration of 40 vol% were used. Error bars (which are too small to be seen) represents a standard error of triplicate runs.

Figure 4-27 illustrates the effect of oil on the Bingham apparent viscosity of the altered UG2 ore. As expected, the apparent viscosity increased as the oil concentration increased. The difference with respect to the baseline apparent viscosity (no oil) was more noticeable at higher oil concentrations. A similar trend was observed in the apparent viscosity of the normal oil, where the viscosity increases with increasing oil concentration (Figure 4-16). Additionally, the baseline apparent viscosity of the altered ore is higher than that of the normal ore (altered ore-0.0322 Pa.s and normal ore-0.0308 Pa.s, refer to Figure 4-16 for comparison) which can be attributed to the greater abundance of talc in the altered ore.

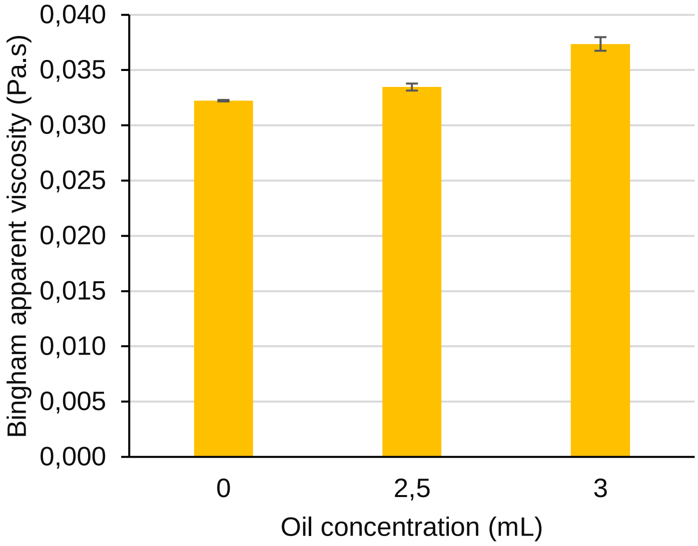


Figure 4-27: Bingham apparent viscosity of the milled altered ore feed (Grinds: 70 %<75 µm) as a function of oil concentrations. Samples with solids concentration of 40 vol% were used for all the oil concentrations. The error bars represent the standard error of triplicate runs.

4.4.3. Froth stability

Figure 4-28 depicts the 3-phase dynamic froth stability factor as a function of oil dosage. The froth stability significantly increased with increasing oil concentration, from a baseline (0 g/t) froth stability factor 63 s to a maximum of 93 s at 400 g/t. This accounted for an overall increase of approximately 47%, which was greater than in normal ore (34%, Figure 4-17). Thus, the oil had a more considerable effect on the froth stability of the altered ore over the normal ore. For the first two oil concentrations as shown in Figure 4-28, the effects of oil were negligible, peaking thereafter at 400 g/t.

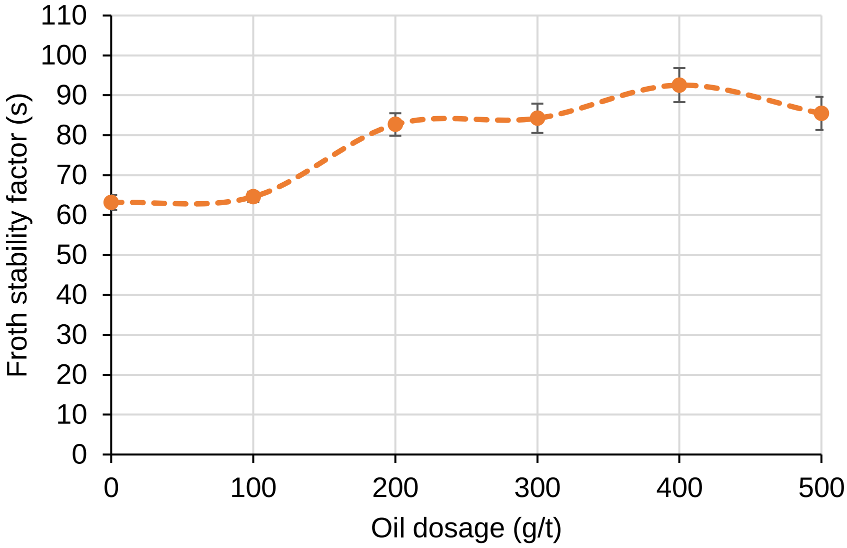


Figure 4-28: 3-phase froth stability at various oil concentration. The error bars represent the standard errors of triplicate trials.

4.5. Single mineral study

This section presents the rheological studies performed on pure talc at various solids concentrations. Here, the goal was to determine how the rheological characteristics of an ore containing a significant amount of talc would change in the presence of oil.

4.5.1. Rheology

The rheogram of water and talc are shown in Figure 4-29. As highlighted in the figure, the increase in oil concentration consequently shifts the curve to higher shear stresses. This was more noticeable at the higher solids volume percent for talc sample (20 vs 30 vol%).

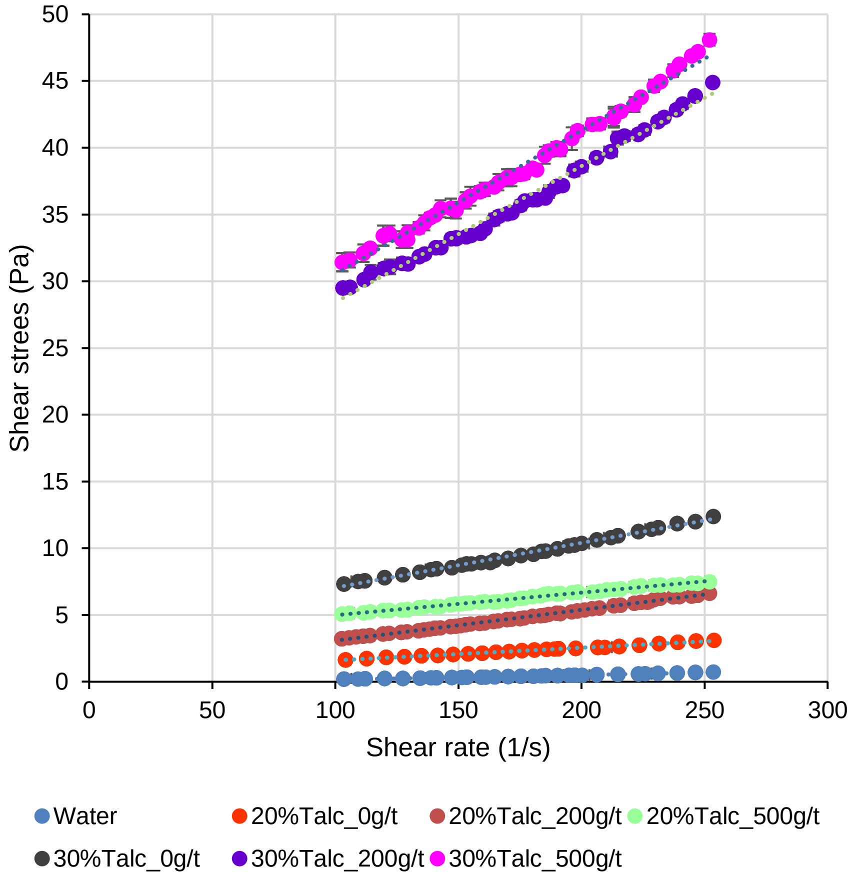


Figure 4-29: Bingham modelled rheogram of talc (grinds: 100 %<75 µm) at various solids and oil concentrations. Samples with solids concentration of 20 and 30 vol% were used. Error bars (which are too small to be seen) represents a standard error of triplicate runs.

The Bingham apparent viscosity of pure talc at 20% and 30% solids concentration and two oil concentrations is shown in Figure 4-30. An increase in the apparent viscosity of the 20 vol% and 30 vol% sample was observed with increasing oil concentration. However, for the 20 vol% sample, there was a small decline at 500 g/t oil which could simply be an outlier. The effect of increasing oil concentration on pulp viscosity was more severe at the higher solid concentrations. The baseline experiments (0 g/t oil) demonstrated how increasing solids concentration influences pulp viscosity, which is, the higher the pulp density the higher the apparent viscosity. The apparent viscosity of the 30 vol% sample increased significantly from

200 g/t. The difference in apparent viscosity between pure talc and the ores was an order of magnitude greater for talc. The talc experimental result is consistent the batch flotation result of both ores where a gradual decrease in grade (normal ore, Figure 4-8) and recovery (altered ore, Figure 4-19) was reported starting at 100 g/t oil. With talc, the oil effect was significant from 200 g/t.

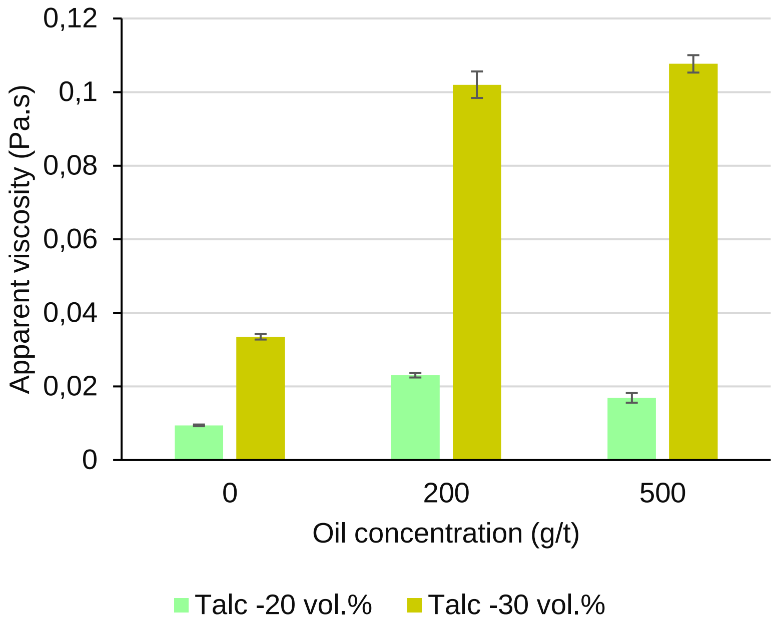


Figure 4-30: The Bingham apparent viscosities of pure talc at varying solids and oil concentrations. The grind used was 100 %<75 µm. The error bars represent the standard error of triplicate runs.

4.6. Mitigation of the deleterious effects of oil with sodium metasilicate and a degreaser

The mitigation of the detrimental effect of oil on the flotation performance of the normal and altered ores with sodium metasilicate and a degreaser will be addressed in this section.

4.6.1. Mitigation in the normal ore

Sodium metasilicate (SS)

Figure 4-31 highlights the effect of SS in the presence of oil with respect to the cumulative solids and water recovered. A high concentration of SS was adopted after a study conducted by Molifie (2021) which showed the dispersing abilities at high concentrations (1000 to 2000 g/t). When comparing the flotation results in the absence and presence of SS, the addition of SS successfully lowered both the mass of solids and water recovered for each individual oil concentration.

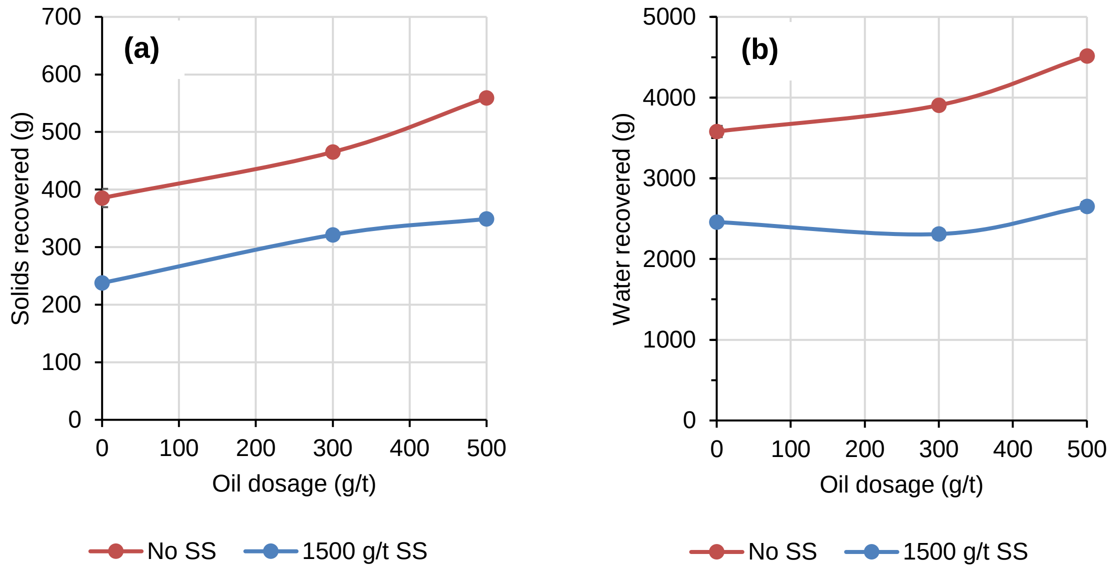


Figure 4-31: Cumulative solids (a) and cumulative water (b) recovered at 0 g/t and1500 g/t sodium metasilicate and different oil dosages. The error bars represent the standard error of duplicate floats.

The effect of adding 1500 g/t SS on Pt and Pd grade and recovery is illustrated in Figure 4-32. A small drop in recovery (Figure 4-32 a-b) was observed upon adding SS compared to no SS. This was most noticeable for Pt in which a drop of around 2% was observed. This became greater at high oil concentration. For Pd, a slight drop of around 1.5 % was observed at 0 and 300 g/t oil. This was likely due to the reduced solids recovered in the presence of SS however, at 500 g/t, the same recoveries were recorded in the absence and presence of SS. With regard to the grade, the adding of SS improved both Pt and Pd grade. The grade improvements were more significant at low oil dosage (0 and 300 g/t oil compared to 500 g/t oil).

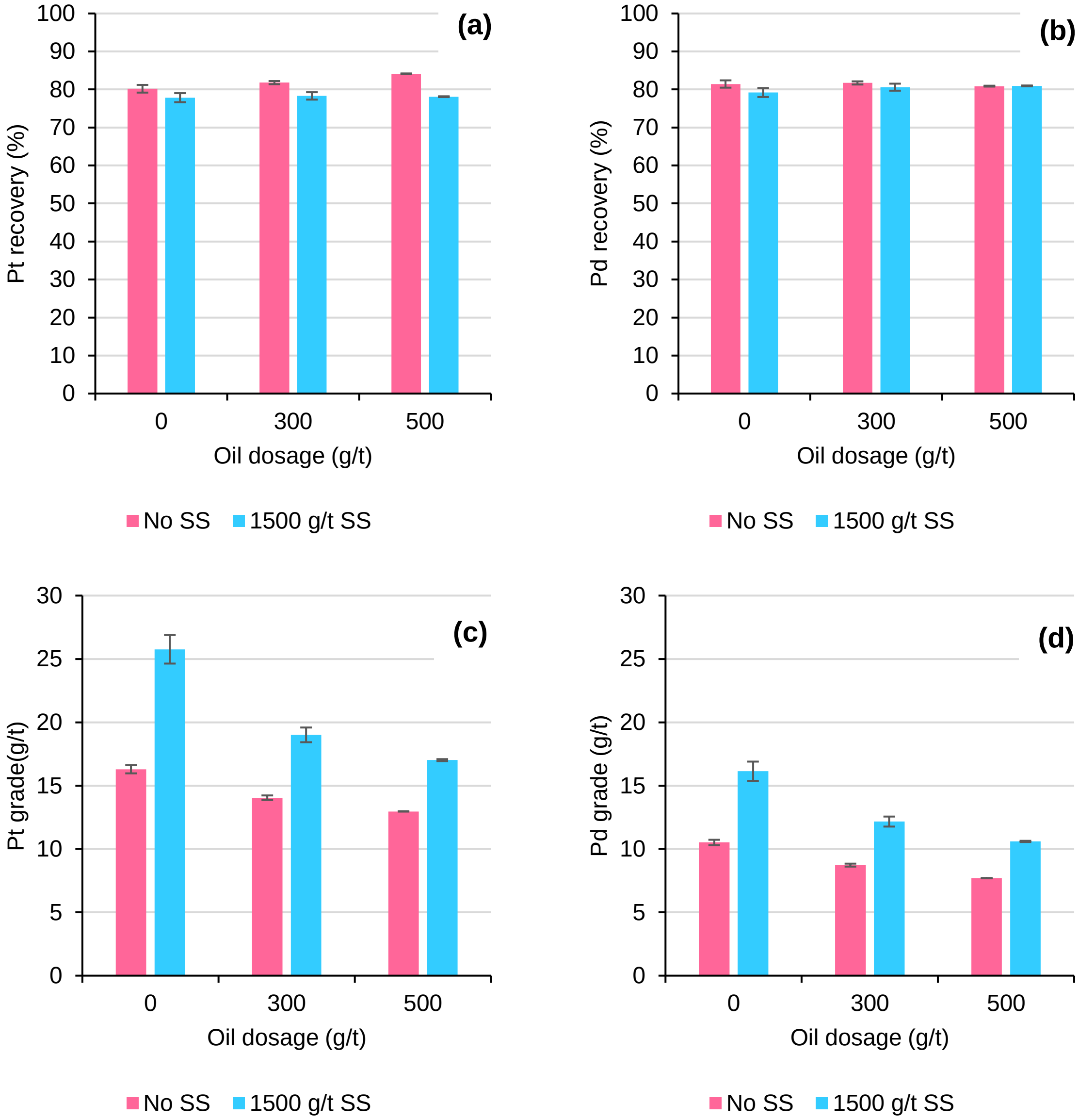


Figure 4-32: Effect SS on the final Pt (a) and Pd (b) recovery at different oil concentrations. Effect SS on the final Pt (c) and Pd (d) grade at different oil concentrations. The error bars represent the standard error of duplicate floats.

The relationship between recovery and solids recovered is depicted in Figure 4-33, as well as the grade and solids recovered. The use of SS enhanced the recovery of both Pt and Pd at a certain constant solids recovery with increasing oil concentration. This is particularly noticeable for Pd at 200 g, where recovery increased from 70% in the absence of SS to roughly 80% in the presence of SS. Similar trends were observed for the grades.

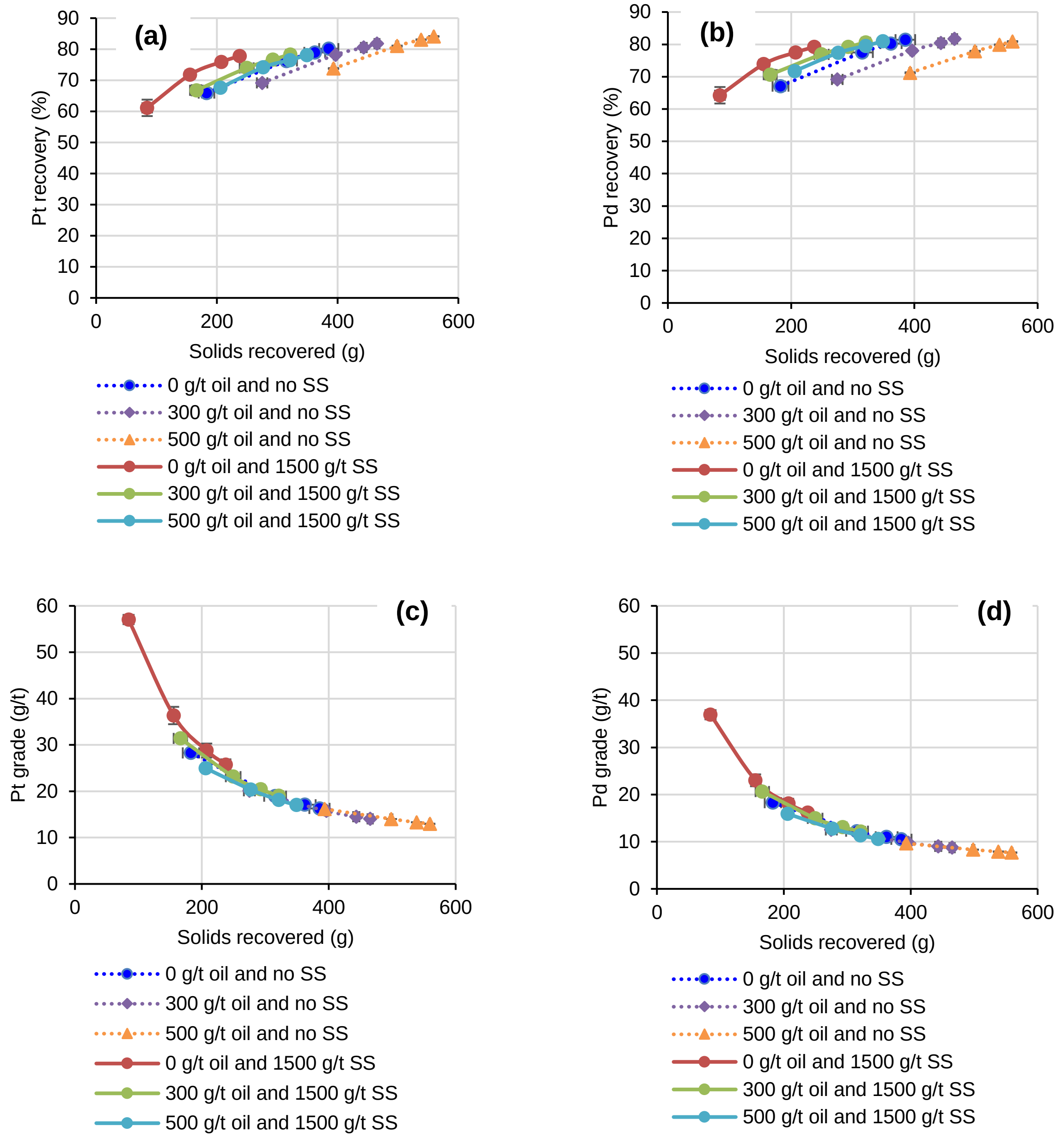


Figure 4-33: Pt (a) and Pd (b) recovery as a function of solids recovered at 1500 g/t SS and various oil concentrations. Pt (b) and Pd (b) grade as a function of solids recovered at 1500 g/t SS and various oil concentrations. Error bars represent the standard error of duplicate floats.

Degreaser

AECI Mining Chemicals supplied the degreaser with the recommendation of dosing it in an equal ratio with oil, thus 1:1. A dosage of 500 g/t of the degreaser was added to the batch flotation cell. Following a mechanical failure of the first flotation cell, these studies were conducted in a new cell, therefore, will be compared amongst each other.

The Pt and Pd final grades and recoveries at 500 g/t degreaser and at zero and 500 g/t oil dosage were plotted to determine the effect of the degreaser on flotation performance as shown in Figure 4-34. The degreaser had a positive effect on recovery but had a slight negative effect on the grade. The Pt recovery increased by around 3% and Pd by 5%. The addition of the degreaser resulted in a decrease in grade of about 1 to 1.5 g/t for Pt and Pd, respectively.

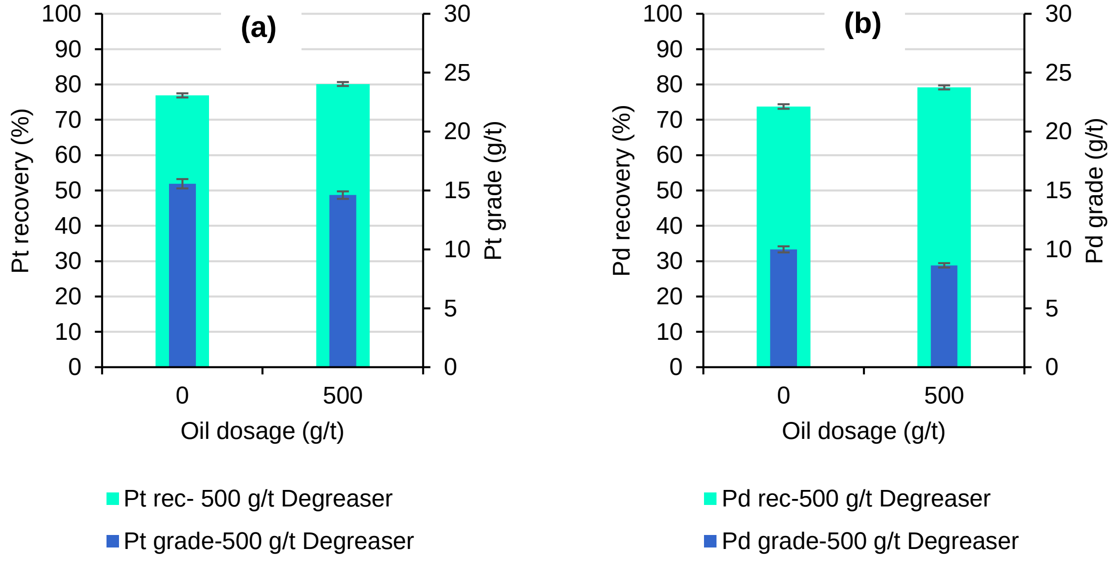


Figure 4-34: Final Pt (a) and Pd (b) grade and recovery at 500 g/t degreaser and varying oil dosages. Error bars represent the standard error of duplicate floats.

4.6.2. Mitigation in the altered ore

Sodium metasilicate

Figure 4-35 illustrates the cumulative solids and water recovered at 1500 g/t SS. It is evident that the introduction of SS reduced both water and solids recovery compared to when no SS was added, and most notably when no oil was present. At 500 g/t oil, the solids recovered was slightly higher than in the absence of SS which might be an indication that the dosage should be increased or that oil is coating everything, hindering the adsorption of SS on particles.

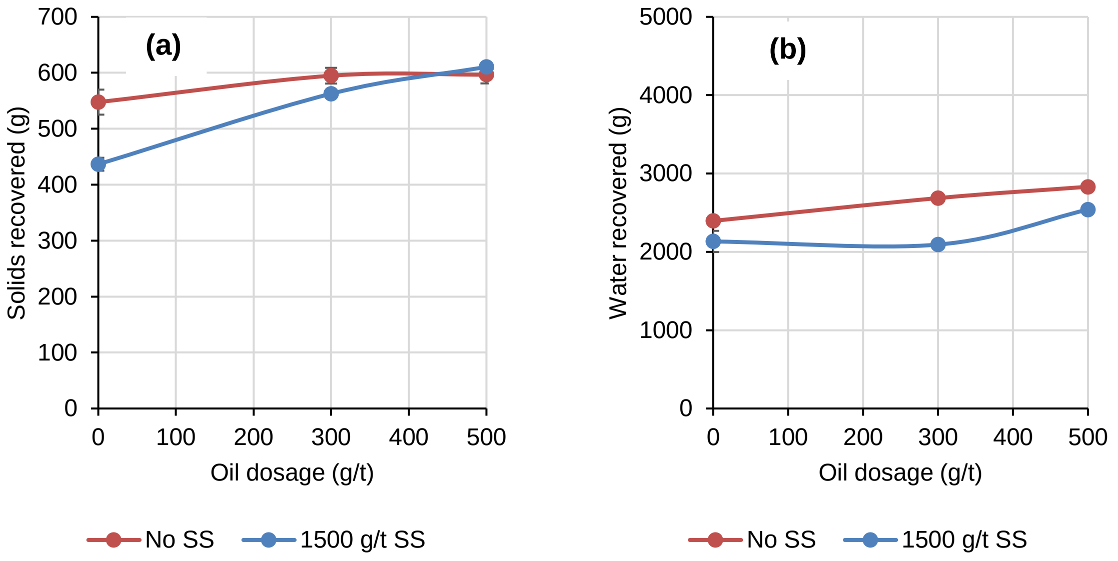


Figure 4-35: Cumulative solids (a) and cumulative water (b) recovered at 0 g/t and1500 g/t sodium metasilicate and different oil dosages. The error bars represent the standard error of duplicate floats.

The effects of SS on the altered ore flotation performance with respect to the grade and recovery is illustrated in Figure 4-36. In the presence of oil, SS had a significant effect on both the grade and recovery. The Pt and Pd recovery (Figure 4-36 a-b) increased overall by approximately 8% from a recovery of 67% at 0 g/t as a result of adding SS and 6% for Pd. When comparing the no SS and 1500 g/t SS floats, the effect of SS was significant at 500 g/t oil condition – where 13.4 % increase in Pt and 13.6% in Pd recovery was observed. SS also enhanced the grade (Figure 4-36 c-d) which was most significant at lower oil concentrations for both Pt and Pd.

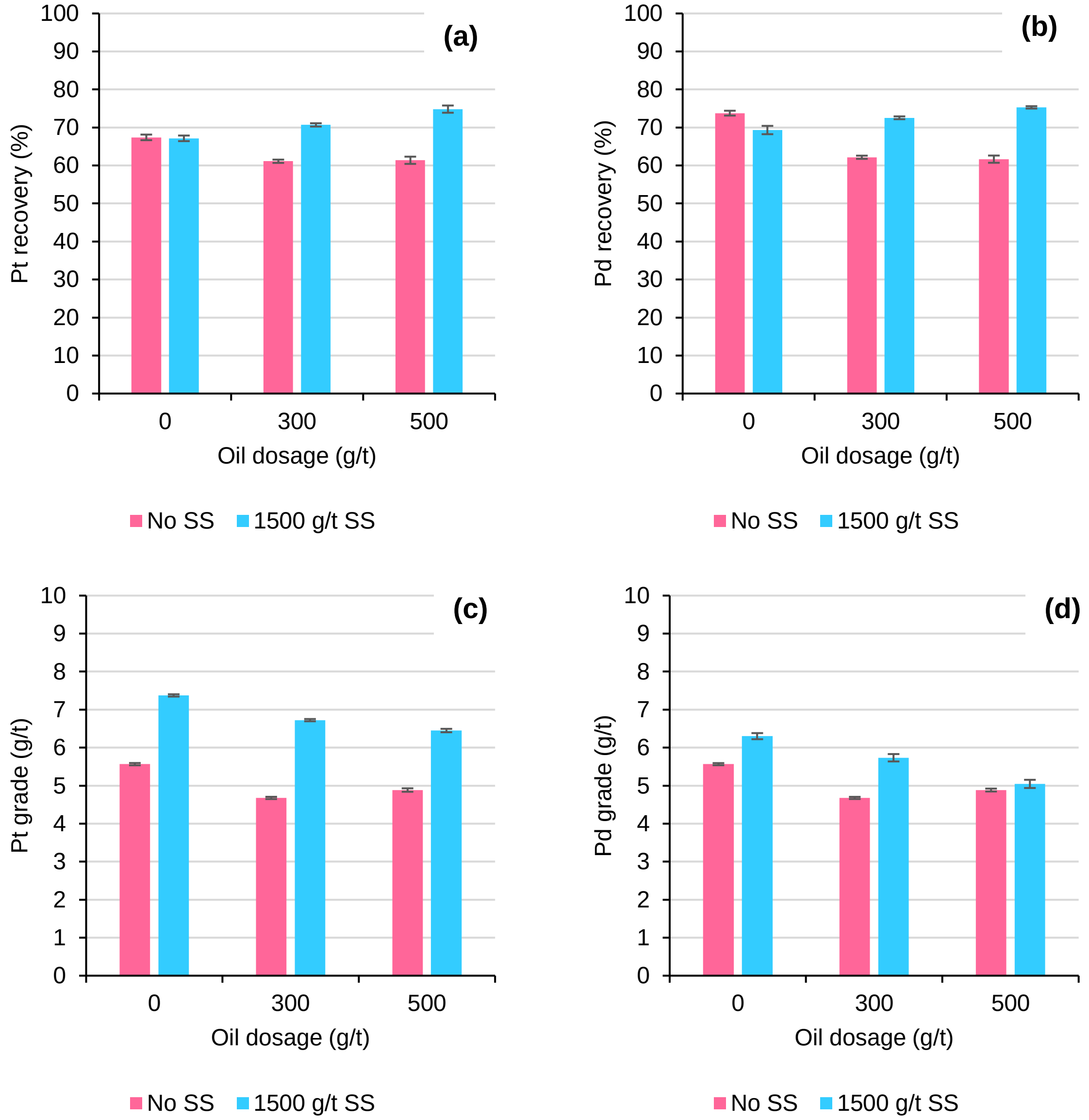


Figure 4-36: Effect SS on the final Pt (a) and Pd (b) recovery at different oil concentrations. Effect SS on the final Pt (c) and Pd (d) grade at different oil concentrations. The error bars represent the standard error of duplicate floats.

Figure 4-37 plots the flotation performance of Pt and Pd with respect to the solids recovered. At a constant solids recovery (e.g., 400 g in Figure 4-37 a-b), the addition of SS increased both the Pt and Pd recovery. The SS effect were pronounced for at higher oil concentration especially for Pd. With respect to Pt and Pd grade, similar trends were observed.

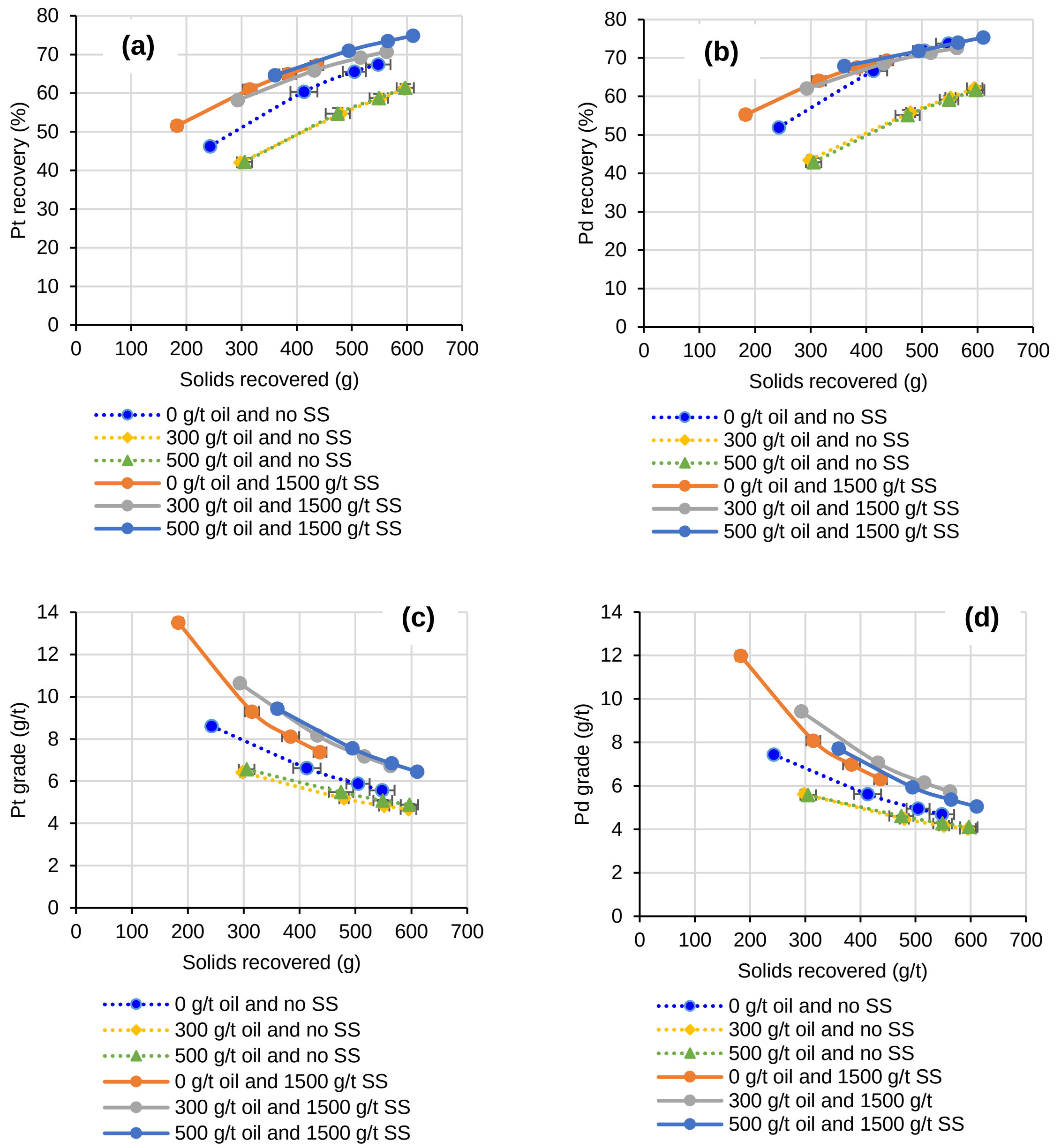


Figure 4-37: Pt (a) and Pd (b) recovery as a function of solids recovered at 1500 g/t SS and various oil concentrations. Pt (b) and Pd (b) grade as a function of solids recovered at 1500 g/t SS and various oil concentrations. Error bars represent the standard error of duplicate floats.

Degreaser

To assess the effect of the degreaser on the flotation performance of the altered ore, the metal recovery and grade curves with no oil and at 500 g/t oil dosage were plotted as shown in Figure 4-38. Same Pt recovery (~70%) was observed at 0 g/t and 500 g/t oil, as result of adding the degreaser. The degreaser sightly increased the Pd recovery by around 1.5% and was constant for Pd. For both metals, same grades were observed at 0 g/t and 500 g/t oil upon adding the degreaser.

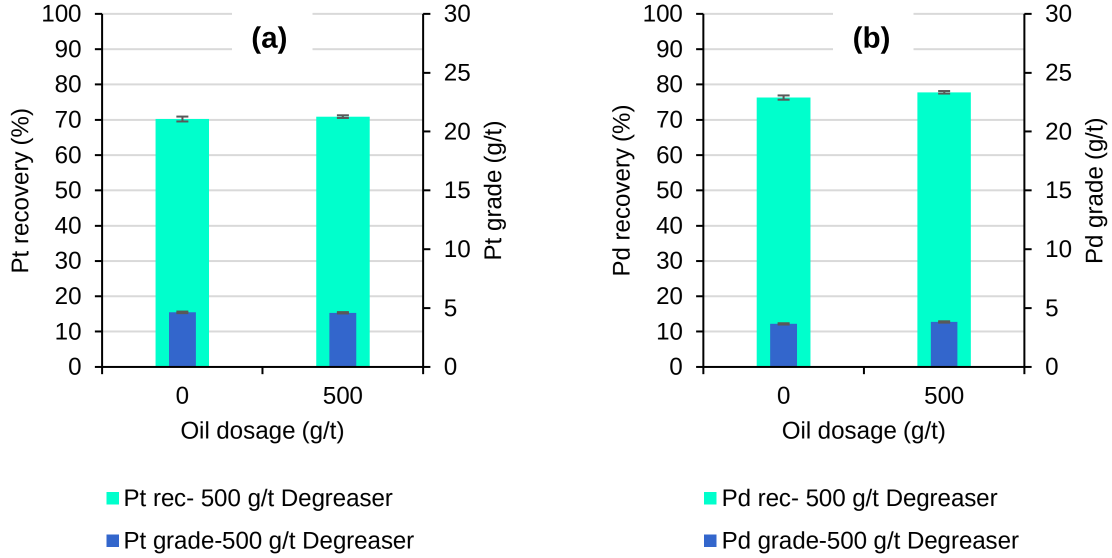


Figure 4-38: Final Pt (a) and Pd (b) grade and recovery at 500 g/t degreaser and varying oil dosages. Error bars represent the standard error of duplicate floats.

4.7. Summary

The key observations from this chapter are summarized below and also tabulated in Table 4-3:

- The altered ore had a greater concentration of phyllosilicate alteration minerals in the ore, with between 2 and 4 times greater talc concentration.
- Higher Pt and Pd recovery were obtained in the baseline floats of the normal ore compared to the altered ore.
- The addition of oil in the normal ore resulted in a gradual decrease in grade and slight increase in recovery. In contrast to the normal ore, the oil in the altered ore resulted in gradual decrease in recovery and negligible effect of the grade. Refer to Table 4-3.
- Nearly all of the oil adsorbed on the particle surfaces in both ores, rather than remaining in the liquid.
- No selective recovery of the oil coated solids to either the concentrate or the tails occurred in the normal ore in contrast to the selective recovery of oil coated particles to the bulk tails in the altered ore.

Table 4-3: A summary of key findings from batch flotation tests, froth stability and rheology (viscosity).

	Ore type	**Oil (baseline tests)**	**SS (in presence of oil)**	**Degreaser (in presence of oil)**
Solids recovery	Normal UG2	Increased	Decreased	Increased
	Altered UG2	Nearly constant	Decreased	Increased
Water recovery	Normal UG2	Increased	Decreased	Increased
	Altered UG2	Nearly constant	Decreased	Increased
Recovery	Normal UG2	Pt: Increased (4%) Pd: Constant	Pt: Decreased (2%) Pd: Decreased (1.5%)	Pt: Increased (3%) Pd: Increased (5%)
	Altered UG2	Pt: Decreased (6%) Pd: Decreased (12%)	Pt: Increased (8%) Pd: Increased (6%)	Pt: Increased (1.5%) Pd: Constant
Grade	Normal UG2	Pt: Decreased (3 g/t) Pd: Decreased (3 g/t)	Pt: Increased Pd: Increased	Pt: Decreased (1 g/t) Pd: Decreased(1.5 g/t)
	Altered UG2	Pt: Constant Pd: Constant	Pt: Increased Pd: increased	Pt: Constant Pd: Constant
Froth stability	Normal UG2	Increased (34%)	-	-
	Altered UG2	Increased (47%)		
Viscosity	Normal UG2	Increased	-	-
	Altered UG2	Increased		
	Talc	Increased		

Legend

Increased
Decreased
Constant

5. Discussion

This chapter discusses the experimental results presented in Chapter 4. The chapter is structured to cover the mineralogical implications on flotation in Section 5.1, the effects of oil on the flotation performance of normal and altered ores (including mitigation) in Section 5.2 and Section 5.3 respectively, and lasty the summary in Section. The summary section highlights the proposed mechanism(s) leading to the detrimental effects on flotation performance observed in each ore.

5.1. Implications of mineralogy on the UG2 flotation performance

5.1.1. Bulk mineralogy

The normal and altered ores are primarily composed of chromite, orthopyroxene, and plagioclase, as indicated in Table 4-2. Due to their abundance, these minerals may pose flotation challenges related to grade dilution as they may likely be recovered via entrainment. Moreover, high chromium oxide (Cr_2O_3) grade in the concentrate reduces the furnace volume during smelting due to chromite spinel build-up, as mentioned earlier in chapter 2. This may lead to PGM losses along with the slag material. The high chromium oxide grade will likely cause more problems in the normal ore, as the normal ore contains higher quantities (42 wt.%) compared to the altered ore (34 wt.%). QEMSCAN results indicated that the alteration mineral contents in the altered ore were twice the amount of the normal ore (Figure 4-1). Phyllosilicate alteration minerals are known to be detrimental to flotation performance (Becker et al., 2009; Dzingai et al., 2021; Ndlovu, Farrokhpay & Bradshaw, 2013; Ndlovu et al., 2011; Shabalala et al., 2011); consequently, the altered ore was expected to experience more severe effects compared to the normal ore.

The alteration mineral of particular interest to this study was talc due to the potential formation of agglomerates in the presence of oil (Sabir, 2015). The results from the QEMSCAN analysis (Figure 4-1) indicate that the talc content in the altered ore is twice that of the normal ore; however, validation of these results with quantitative XRD (Figure A-1 in Appendix A) indicates that the difference could be as much as four times – hence it is most likely that the true value is somewhere in between. Talc is a naturally occurring floatable gangue mineral that is widely known for diluting concentrate grade and stabilizing the froth phase. As a result of the froth stabilizing effects, the solids and water recoveries increase which may consequently lead to the recovery of other gangue minerals via entrainment, and further diluting the grade (Becker et al., 2009; Becker, Wiese & Ramonotsi, 2014; Dzingai et al., 2021). Additionally, talc has a strong association with orthopyroxene, which may induce hydrophobicity to composite particles, making it more likely to be recovered (Becker et al., 2009). The agglomeration of finely grained talc particles because of oil results in increases in the slurry viscosity, which may in turn lead to reduced recovery due to poor gas dispersion and reduced bubble-particle collision (Sabir, 2015; Shabalala et al., 2011). Therefore, the altered ore was expected to indicate low flotation grade and recovery, as well as higher water and solids recovery, relative to the normal ore.

5.1.2. PGM characterization

The PGM characterization was performed using the QEMSCAN technique. The results (Figure 4-2 showed that the PGE sulphides predominate the UG2 mineralogy, with approximately 66 PGM area % in both ores. Cooperite was the main constituent of the sulphides group in both ores. A strong presence of the arsenides group (19.2 PGM are %) comprising predominately of sperrylite was reported in the normal ore, and a minor proportion in the altered ore (3 PGM area %). The PGE asernides, particularly sperrylite, are well known for their slow floatability, which causes them to report to the tailings (Shackleton, Malysiak & O'Connor, 2007). As result, this might contribute towards recovery loss and was expected to be greater in the normal ore. Another key observation was the high PGE alloys percentages, which were relatively high in the altered ore (22 PGM area %) in contrast to the normal ore (10 PGM area %). The high PGE alloys proportions were consistent with the alloys abundances from the eastern Bushveld Igneous Complex UG2 (McLaren & De Villiers, 1982).

Figure 4-3 demonstrated that the bulk of the PGM grains were less than 10 µm. This was consistent with the observed UG2 grain sizes by Penberthy, Oosthuyzen and Merkle (2000) and Schouwstra, Kinloch and Lee (2000). It is apparent these PGMs requires fine to very fine liberation, which may in turn create problems related to fine grinding as previously mentioned in Chapter 2. A moderate effective liberation of 67 PGM area % was reported in normal ore in contrast to 43 PGM area % in the altered ore (Figure 4-4). The degree of liberation in the normal ore was in line with the secondary grinds liberation observed in various UG2 samples (Penberthy, 2006). However, the liberation observed in the altered ore was relatively low. Some of the PGMs were reported to be enclosed in oxide minerals and alteration silicates, in both ores. The enclosed PGMs may presents flotation recovery challenges as they are unrecoverable due to unexposed PGM area.

5.2. Baseline batch flotation

The baseline flotation recoveries (Figure 4-6) of Pt and Pd in the absence of oil in the normal ore (both at 80%) were higher relative to the altered ore (Pt -67 % and Pd -74%). This was due to the greater liberation of PGM in the normal ore (67 PGM area %) compared to the altered ore (43 PGM area %), as shown in Figure 4-4. The flotation grades were also higher for the normal ore, which was expected due to the lesser degree of alteration in the normal ore as shown by the bulk mineralogy in Table 4-2. Looking at the bulk mineralogy (Table 4-2) of the two ores, the slime coating and the rheological effects mentioned in Chapter 2 would be a problem in the altered ore.

5.3. The effects of oil on the flotation performance of the normal ore

5.3.1. Grade

This section aims to address and discuss the grade hypothesis from Chapter 2, which is as follows: *"High oil concentrations will reduce the concentrate grade due to oil coating both hydrophobic and hydrophilic gangue minerals, rendering them hydrophobic while also decreasing depressant adsorption and enhancing gangue mineral recovery."*

The addition of oil (0 -500 g/t) resulted in a gradual decrease in grades of both Pt and Pd, with the effect being more detrimental at high oil dosages. An overall grade reduction of approximately 3 g/t was observed for both Pt and Pd (Figure 4-8). This was consistent with the grade-recovery curves in Figure 4-9 c-d, which demonstrated that at a constant recovery the grade decreases with increasing oil concentration. Similar trends were observed for the grade-solids recovery (Figure 4-10 a-b) and grade-water recovery curves (Figure 4-10 c-d). Additionally, the solids and water recovery as a function of oil concentration (Figure 4-7) showed a significant increase in both water and solids recovery with increasing oil concentration. The presence of oil also increased the froth stability by 34% as shown in Figure 4-17. This was consistent with the rheological measurements (Figure 4-15 and Figure 4-16) which revealed an increase in yield stress, and apparent viscosity with increasing oil concentration.

According to Neethling and Cilliers (2009) and Zheng, Johnson and Franzidis (2006) the flotation grade decreases with increasing solids and water recoveries. This conventional flotation trend was consistent with the Pt and Pd grade reduction trends observed with increasing solids and water recovery in Figure 4-10. The lowest grades were attained at the highest water and solids recovered as well as the oil dosage (500 g/t) as shown in Figure 4-9 c-d and Figure 4-10. The 3-phase froth stability result corresponded with the solids recovery and water recovery which all increased with increasing oil concentration. Conventionally, increasing froth stability leads to an increase in solids recovered (Achaye, Wiese & McFadzean, 2021). According to Bradshaw, Oostendorp and Harris (2005), a significant increase in froth stability may result in additional gangue recovery particularly if the froth is too stable. The observed froth stability increase along with increasing solids and water recovery contributed to the drop in grade for Pt and Pd.

The bulk mineralogical evaluation of the baseline batch flotation concentrate (no oil) and the 500 g/t oil concentrate (Figure 4-11) revealed that orthopyroxene, talc, plagioclase, and chromite were the major minerals responsible for Pt and Pd grade dilution. Orthopyroxene, plagioclase and chromite are hydrophilic gangue minerals that are normally recovered through entrainment. Hay and Roy (2010) investigated the entrainment of chromite in the flotation of UG2 ore and observed a significant increase in chromite entrainment with increasing water and solids recovery, which is consistent with the current study. Talc is a naturally floatable gangue mineral which also co-exists with orthopyroxene (Becker et al., 2009). The grade dilution due to talc and orthopyroxene may be due to the recovery of liberated talc via flotation as well as unliberated talc-orthopyroxene composites and orthopyroxene through entrainment. Therefore, the concentrate mineralogy along with the grade versus water recovery and grade versus solids recovery in Figure 4-10 confirm the grade dilution to be from the recovery of gangue minerals, likely due to both entrainment and true flotation. These findings were also consistent with Bos and Quast (2000) where the addition of hydraulic oil reduced the flotation grade in copper ore.

Therefore, the decrease in Pt and Pd grade (Figure 4-8) was attributed to the increase in froth stability with increasing oil dosage that resulted in an overall increase in solids and water recovery consequently leading to greater recovery of gangue minerals.

5.3.2. Recovery

The hypothesis that will be addressed in this section is stated below:

"The presence of phyllosilicate alteration minerals and a high concentration of hydrocarbons in the slurry will increase pulp viscosity, due to the agglomeration of phyllosilicate minerals with increased hydrophobicity. High pulp viscosity leads to reduced bubble-particle interaction and poor gas dispersion. This, in turn, will lead to a reduction in flotation recovery."

The Pt recovery (Figure 4-8) increased by 4% from a baseline recovery of 80% as the oil dosage was gradually increased from 0 to 500 g/t, whereas the Pd recovery remained nearly constant at 80%. Figure 4-7 b depicts the increase in solids and water recovery as the oil concentration increases. This corresponded to the Pt recovery trend which increased with increasing solids and water recovery as oil concentration was increased (Figure 4-8 and Figure 4-9 b). Also, the solids recovery trend observed in Figure 4-7 b was also consistent with an overall froth improvement of 34% in the normal ore (Figure 4-17). Higher froth stability is known to improve the recovery of valuable minerals at the expense of grade (Barbian et al., 2005; Morar et al., 2006). This was consistent with the Pt grade versus recovery curve where a trade-off between the grade and recovery is visible at higher oil dosages (Figure 4-9c). Conventionally, high flotation recovery is linked to high solids recovery (Murhula, Hashan & Otsuki, 2022). This was consistent with the solids recovery (Figure 4-7 b) and Pt recovery (Figure 4-8) observed in this study. The Pd recovery contradicted this observation as it remained constant with increasing solids recovery at higher oil concentrations. Thus, the increase in Pt recovery was attributed to the froth stability improvement which resulted in increased solids recovery with increasing oil concentration.

Although the final recoveries increased with oil increasing concentration, Figure 4-9 a-b illustrated that for a constant mass of solids recovered, the Pt and Pd recovery decreased with increasing oil concentration. This was worth noting as it can result in recovery losses at full-scale if the flotation cells are operated at constant mass pull.

5.4. The effects of oil on the flotation performance of the altered ore

5.4.1. Grade

The same hypothesis as stated in section 5.3.1 will be addressed here. The addition of oil had little effect on the final Pt and Pd grade in the altered ore (Figure 4-19). Pt and Pd grades were nearly constant with increasing oil dosage, at around 5 g/t and 4 g/t respectively. The grades trends were consistent with the water and solids recovery trends (Figure 4-18), which were nearly constant from 100 g/t to 500 g/t oil condition. This was unexpected, since the water and solids recovery normally increase with improved froth stability leading to grade dilution due to entrainment (Bradshaw, Oostendorp & Harris, 2005; Hay & Roy, 2010; Morar et al., 2006).

However, when the grades are viewed with respect to recoveries or solids recovered, it is apparent that there is a slight negative effect of increasing oil addition. When operating at constant solids recovery (especially low solids recovery; Figure 4-21), the grade decreases with increasing oil concentration. Although the drop was minor, for plant-scale operations, this could result in final concentrate grade reduction if the plant is operating at constant solid mass pull.

5.4.2. Recovery

The hypothesis mentioned in section 5.3.2 will also be addressed below. The batch flotation results of the altered ore revealed (Figure 4-19) a significant decrease in the recovery of Pt (6%) and Pd (12%) with increasing oil dosage. The recovery drop plateaued after the 100 g/t oil condition. This is also shown by the Pt, Pd recovery versus solids recovery (Figure 4-20 a-b) and Pt, Pd recovery versus water recovery curves (Figure 4-21 a-b), that at a constant mass of solids and water recovered, the recoveries decrease upon addition of 100 g/t oil and, thereafter, becomes nearly constant with increasing oil concentration. It is also worth mentioning that the solids and water recovery were almost constant as the oil concentration was increased.

The bulk mineralogy of the altered ore contained considerable amounts of phyllosilicate alteration minerals relative to the normal ore (Table 4-2), particularly talc. XRD results revealed the talc content of around 8 wt.% (4 times greater than in the normal ore, Appendix B). As indicated earlier, talc in the presence of oil, subsequently increased the viscosity (Polowczyk & Kozlecki, 2017; Sabir, 2015). The slurry rheological experiments (Figure 4-27) displayed an increase in yield stress (y intercepts from the equations in Figure 4-26) and apparent viscosity with increasing oil concentration. Furthermore, the rheological experiments of pure talc (Figure 4-30) demonstrated an exponential increase in the viscosity of talc especially at higher oil concentration. The observed rheological increases are thought to be caused by talc-oil agglomeration, coupled with talc's platy shape (Ndlovu, Farrokhpay & Bradshaw, 2013; Ndlovu et al., 2011; Polowczyk & Kozlecki, 2017; Sabir, 2015).

Various studies have linked high slurry yield stress and viscosity with poor gas dispersion and reduced particle-bubble collisions within a flotation cell (Bakker, Meyer & Deglon, 2009; Bakker, Meyer & Deglon, 2010; Shabalala et al., 2011). This limits bubble-particle interaction, thus lowering the transfer of valuable minerals (solids) from the pulp to the froth phase. The constant solids recoveries (Figure 4-18 a-b) observed with increasing oil concentration (from 100 g/t to 500 g/t oil), suggested a limited transfer of solids to the froth zone (solids recovered). Figure 4-19 a-b illustrated that the recoveries of both Pt and Pd decreased and became nearly constant after 100 g/t oil concentration. Therefore, the recovery loss is believed to be caused by poor gas dispersion because of increased slurry viscosity, as well as slimes coatings of ultrafine alteration minerals on the valuable minerals. Additionally, the oil-solvent extraction data (Figure 4-25) shows a selective recovery of oil-coated particles to the tailings relative to the concentrate, inferring that most of the oil was in the pulp zone during flotation likely creating a solids 'trap zone' which limited the recovery and also gave rise to the observed viscosity (Figure 4-27).

The flotation results showed that the oil had a more negative impact on the recovery of Pd in comparison to Pt. This was worth noting as the altered ore feed had a higher Pt grade than Pd (Table 4-1). Osbahr et al. (2014) conducted a study on the distribution of PGE in the western and eastern limb UG2 ores and discovered that the pentlandite hosts a higher concentration of PGE compared to other BMS, with Pd predominating (~55 wt.%) in pentlandite. The bulk mineralogical composition in Table 4-2 shows that the pentlandite content was similar to the other BMS (<0.1 wt.%). Therefore, the flotation results suggest that the oil had a negative impact on the pentlandite (host mineral) which contains elevated amount of Pd, thus significantly reducing the recovery. This also suggests that pentlandite might be difficult to recover compared to the other two BMS, and the presence oil and fine phyllosilicate alteration minerals worsened it.

5.5. Mitigating the effects of oil with sodium metasilicate and a degreaser

This section discusses the mitigation strategies that were evaluated, namely: sodium metasilicate and degreaser. The hypotheses considered in this section are listed below.

Sodium metasilicate: *"Sodium metasilicate will enhance recovery and grade due to electrostatic repulsion between particles upon adsorption, increasing the dispersion of particles, reducing the viscosity and increasing froth drainage."*

Degreaser: *"The amphiphilic properties of the degreaser will aid in dislodging oil, thus improving both the recovery and grade."*

5.5.1. Sodium metasilicate

Normal ore

Grade

The addition of 1500 g/t SS in both the absence and presence of oil improved both Pt and Pd grades (Figure 4-32 c-d). The effect of SS on the grade was most noticeable at low oil dosages (0 and 300 g/t compared to 500 g/t). When comparing the quantities of solids and water recovered in the absence and presence of SS (Figure 4-31 a-b), the addition successfully lowered the solids and water recovered in the presence of oil. Additionally, as illustrated in Figure 4-33, the addition of SS improved the grade as the oil concentration increased for a constant solids recovery value.

Amongst the primary grade diluents (Figure 4-11) as a result of oil addition was talc, which is known to dilute concentrate grade as covered earlier in Chapter2 and also possess a strong froth stabilizing effect (Becker et al., 2009; Becker, Wiese & Ramonotsi, 2014; Dzingai et al., 2021). It is also found as finely disseminated grains within orthopyroxene, which makes the composite particles hydrophobic (Becker et al., 2009). Molifie (2021) conducted a study on the use of sodium metasilicate to enhance flotation performance of a highly altered PGM ore. It was deduced that high dosages of SS successfully increased the concentrate grade by depressing gangue minerals, particularly talc. The study attributed that to the dissolution of Mg^{2+} ions from talc's surface that facilitated the adsorption of SS on talc edge and face, thus depressing it. At SS high dosage (1000-2000 g/t), the slurry pH becomes basic (~11) (Feng et al., 2012; Molifie, 2021). The dissolution of SS produces $SiO(OH)_3^-$ (formed at pH > 9.4), which is on the species known to depress gangue minerals. This postulated to have contributed to the improved flotation grade.

Molifie (2021) also demonstrated that the addition of high SS dosages improves gangue drainage by lowering the froth stability and the quantity of solids recovered. This was consistent with the lower solids recovery observed in the presence of SS as compared to when no SS was added (Figure 4-31). Molifie (2021) attributed the improved froth drainage to the dispersion of aggregated particles which would have otherwise hindered or limited the froth drainage in the absence of SS.

Recovery

Varying the oil concentration had no negative impact on the normal ore flotation recovery (Figure 4-8). However, the addition of 1500 g/t SS slightly reduced the recovery (Figure 4-32 a-b). This was more visible for Pt in which recovery decreased by around 1% at 0 g/t oil, 1.5% at 300 g/t oil and around 5% at 500 g/t oil condition. For Pd the recovery decreased slightly by 1% at these oil conditions. Both solids and water recovery decreased significantly due to the addition of SS as illustrated in Figure 4-31 a-b. This was consistent with the observed Pt and Pt reduction as SS was added. Therefore, the observed Pt and Pd recovery drop was due to the lower mass recovery of solid particles caused by the mild depressing properties and froth drainage effects of SS.

Altered ore

Grade

Varying the oil concentration had a negligible effect on the concentrate grade for the altered ore as shown in Figure 4-19. Pt and Pd grade were nearly constant with increasing oil concentration, Pt at around 5 g/t and Pd at around 4 g/t. However, the addition of 1500 g/t SS improved both Pt and Pd grades (Figure 4-36 c-d). Similar to the normal ore grade behaviour, the SS effect was most effective at lower oil concentrations (300 g/t compared to 500 g/t). Also, lower solids and water recovery was observed upon adding SS as indicated in Figure 4-35. This infers that more gangue minerals were rejected as result of adding SS. Figure 4-37 c-d illustrated the improvement of grade at constant solids recovery because of SS addition. The improved Pt and Pd grade with lower solids and water recovery due to SS addition pointed toward improved drainage, as previously stated in the discussion on the normal ore. The improved grade was ascribed to improved froth drainage caused by the dispersion of aggregated particles (Molifie, 2021).

Recovery

The oil had a detrimental effect of the flotation recovery of the altered ore resulting in an overall Pt drop of 6% and Pd by 12 %. This was attributed to the rheological complexity arising with increased slurry viscosity (Figure 4-27) consequently leading to poor gas dispersion and reduced particle-bubble collision. The oil adsorption study on the feed and concentrate also revealed that the oil-coated particles selectively report to the tailings which inferred that the problem was in the pulp phase (Figure 4-25). The addition of 1500 g/t SS as a mitigation measure increased the recovery for both Pt and Pd (Figure 4-36 a-b). The Pt recovery increased by 8% from a baseline (0 g/t oil) recovery of 67% and Pd by 6% from 70%. The effect of SS was most noticeable at higher oil dosage (500 compared to 300 g/t) in terms of the recovery.

Similar observations were reported by Molifie (2021) in which the addition of high SS dosages (up to 2000 g/t) increased the PGE recovery of a highly altered PGM ore. The recovery increase was ascribed to SS significantly reducing the slurry viscosity by adsorbing on the surfaces of talc and serpentine. The adsorption of SS on serpentine was found to make the mineral more negatively charged, thus resulting in electrostatic repulsion with negatively charged PGM. Upon adsorption on talc face and basal plane, SS resulted in the dispersion of these particles. The altered ore had relatively high alteration minerals content compared to normal ore (Table 4-2), and the effect of SS on recovery was more noticeable in the altered ore. It was also shown that increasing the oil concentration resulted in higher viscosities. Since SS is known to lower the slurry viscosity, this reduces the likelihood of flotation problems such as poor gas dispersion and reduced bubble-particle interaction, which may result in recovery losses (Becker et al., 2013; Ndlovu, Farrokhpay & Bradshaw, 2013; Shabalala et al., 2011). Lower pulp viscosity contributes to better cell agitation, which leads to enhanced particle-bubble collision and, as a result, higher recoveries.

SS also resulted in lower solids recovery (Figure 4-35) with an increase in Pt and Pd recovery, except at 500 g/t oil, where the solids in the absence and presence of SS were relatively similar. The reduced solids recovery infers that the interaction between possible oil-agglomerated solid particles have been reduced, leading to lower viscosities and hence improved recovery. It should be noted that the addition of SS enhanced both the grade and recovery of the altered ore.

5.5.2. Degreaser

Normal ore

Grade

Figure 4-34 demonstrates that adding 500 g/t of the degreaser resulted in about 1 and 1.5 g/t decrease in Pt and Pd grades, respectively compared to using degreaser with no oil present. However, there is no comparison against the condition where no degreaser was used due to laboratory equipment problems. One can only compare trends (% difference between 0 g/t and 500 g/t), and it was shown that, in the absence of degreaser, the grade of the Pt and Pd in the normal ore decreased by 20.4% and 26.7%, respectively, when going from no oil addition to 500 g/t oil addition. On the other hand, when degreaser was added, there was only a 6.2% and 13.6% reduction in grade. Therefore, it appears that the degreaser was able to improve grades relative to when no degreaser was used. This observation was in contrast to the study conducted by Bos and Quast (2000), where the addition of the degreaser in the flotation cell was found to reduce the copper concentrate grade. The study attributed that to the coating of gangue minerals by the degreaser, which caused the hydrophilic gangue minerals to become floatable hydrophobic minerals. It is also worth noting that the use of a degreaser is not extensively covered in froth flotation.

Recovery

The degreaser enhanced both Pt and Pd recovery (Figure 4-34) in the normal ore when comparing the 0 g/t and 500 g/t oil condition by 3% and 5% respectively. As previously stated, only the trends will be used to relate between the degreaser and no degreaser conditions; this applies to the altered ore as well. In the absence of the degreaser, the Pt recovery increased by 5.2% (from 0 g/t to 500 g/t) and Pd recovery decreased by 0.7%. In the presence of the degreaser, the Pt recovery increased by 4.2% and Pd by 7.2%, from 0 g/t to 500 g/t. This infers that the degreaser enhanced the overall recoveries relative to the no degreaser condition.

Altered ore

Grade

When comparing the 0 g/t and 500 g/t oil conditions, the addition of 500 g/t degreaser resulted in a constant grade for both metals (Figure 4-38). However, when comparing the trends (%difference between 0 g/t and 500 g/t oil), both Pt and Pd grade decreased by around 12%, respectively, in the absence of the degreaser. Pt grade decreased by around 1% and Pd grade increased by 4.4% in the presence of the degreaser. Thus, the degreaser improved the grade relative to the baseline effect on grade caused by oil. This was attributed to the amphiphilic properties of the degreaser.

Recovery

The addition of the degreaser in the presence of oil was reported to raise the Pd concentration by roughly 1.5% in the altered ore and was constant for Pd (Figure 4-38). By comparing the trends, it was shown that in the absence of the degreaser the Pt recovery decreased by 8.5% from 0 g/t to 500 g/t and Pd by 16%. Adding the degreaser resulted in 1% and 2% increase in Pt and Pd recovery, respectively. Therefore, this suggests that the degreaser increased the recovery compared to when no degreaser was added. The enhanced recovery was attributed to the dislodgement of oil due to the amphiphilic properties of the degreaser, as previously indicated.

5.6. Interaction mechanisms between oil and the UG2 ores

This section summarizes the discussion chapter and also explores the underlying interaction mechanisms between oil and the two ores that resulted in poor flotation performance, as well as the mitigation strategies. The effect of oil on the flotation performance of the normal and altered ore was different. The normal ore mechanism will be covered first, followed by the altered ore. A simplified diagram of the mechanism in each ore is illustrated in Figure 5-1.

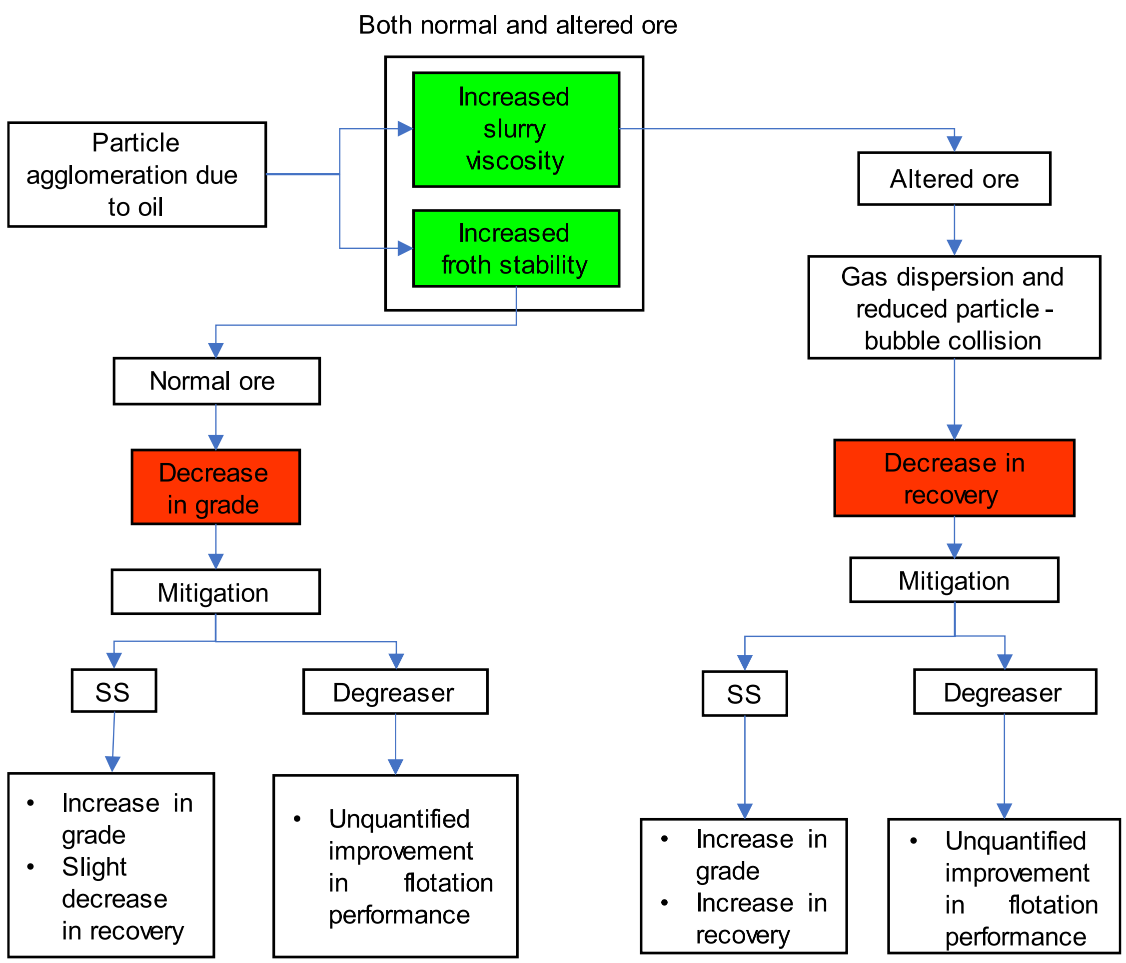


Figure 5-1: A schematic diagram illustrating the mechanisms causing decreased flotation performance in each ore (normal and altered) and the mitigation strategies explored.

The addition of the oil in the normal ore flotation resulted in the adsorption of the majority of the oil on the particle surfaces, which was further postulated to have caused the agglomeration of finely milled particles, especially talc (Polowczyk & Kozlecki, 2017; Sabir, 2015). As a result, the talc-oil agglomerates gave rise to increased froth stability (34% increase, Figure 4-17) as well as increased slurry viscosity (Figure 4-16). The improved froth stability resulted in increased water and solids recovery. This led to the unselective recovery of gangue minerals as evidenced by the concentrate mineralogy results (Figure 4-11), subsequently reducing the Pt and Pd grade. These gangue minerals comprised orthopyroxene, talc, plagioclase, and chromite. The oil had no negative effect on the recoveries. It was further found that no selective recovery of oil coated solids to either the concentrate or the tail was observed in the normal ore (Figure 4-17). To mitigate the effects of the oil addition, SS (1500 g/t) and the degreaser (500 g/t) were used separately. SS improved the Pt and Pd grade, and this was attributed to better gangue drainage in the froth phase. In terms of recovery, a slight drop in recovery was observed and it was due to reduced solid recovery in the presence of SS. Mitigation with degreaser resulted in an unquantifiable improvement in flotation performance.

Similar to the normal ore, most oil adsorbed on the particle surfaces upon addition in the altered ore (Figure 4-25). The altered ore had high phyllosilicate alteration content (particularly talc, Table 4-2) in comparison to normal ore. This is thought to have resulted in greater talc agglomeration compared to the normal ore, which further increased both the froth stability (47% increase) and slurry viscosity. As expected, the viscosity was higher in the altered ore (Figure 4-27) compared to the normal ore (Figure 4-16) due to the higher alteration content and talc agglomerates formed by oil. It was also demonstrated by a viscosity test using pure talc (Figure 4-30) which showed that any ore slurry with a high talc composition would be more viscous in the presence of oil. The increased viscosity was inferred to cause poor gas dispersion and reduced bubble-particle collisions, leading to the recovery losses, 6% drop for Pt and 12% for Pd. The recovery loss was further explained by the oil adsorption study (Figure 4-25), which showed that the majority of the oil coated solids selectively report to the tails. With regard to the grade, the oil had a negligible effect. The addition of 1500 g/t SS increased the baseline recoveries of Pt and Pb by 8 and 6%, respectively. This was attributed to SS reducing the slurry viscosity and thereby preventing poor gas dispersion and reduced bubble-particle collision problems. With respect to the grade, SS improved both Pt and Pd grades because of improved froth drainage. As stated earlier, the addition of the degreaser as a mitigation measure resulted in an unquantifiable improvement in flotation performance.

Up to this point the flotation experiments were conducted using fresh oil. Flotation tests work were also conducted using used oil to compare the difference in flotation behaviour of the fresh and the used hydraulic oil (Figure 4-12 and Figure 4-23). No significant difference in grade (Pt and Pd) was observed when using the used oil compared to the fresh oil. However, the two oil samples had appreciable difference in the viscosity as shown in Table 3-1(68.5 centistoke in the fresh oil compared to 60.5 centistoke in the used oil).

6. Conclusions and Recommendations

6.1. Conclusions

The aim of this study was to investigate the effect of hydraulic oil spillages on the flotation performance of UG2 ores and establish the interaction mechanisms between oil and phyllosilicate alteration minerals leading to detrimental effects on flotation performance. Finally, the use of sodium metasilicate and a degreaser was evaluated as potential mitigation measures to the combined effects of oil and phyllosilicate alteration minerals on flotation performance. The section highlights the key observations and findings from this study, which may support or invalidate the proposed hypotheses from Chapter 2.

6.1.1. Implication of mineralogy

The bulk mineralogical composition of the normal and altered ore revealed a significant presence of chromite, pyroxenes, and plagioclase (in decreasing order). In terms of the alteration minerals, the altered ore had relatively high proportions of phyllosilicate alteration minerals (about 9 wt.%) in contrast to the normal ore (about 6 wt.%). Talc predominated in the composition of the phyllosilicate alteration minerals: 6 wt.% in the altered ore and 3.8 wt.% in the normal ore. As previously stated, the talc will agglomerate in the presence of oil, and this may give rise to rheological challenges within the flotation cell, leading to reduced flotation performance. This includes reduced collisions between particles and bubbles, poor gas dispersion, and slime coating by fine particles. Thus, the altered ore was expected to experience severe rheological effects in the presence of oil in contrast to the normal ore. Serpentine and chlorite contents were relatively low in the two ores; however, along with talc these might create slime coating problems.

The PGM characterization revealed that the majority of the PGMs in both ores are fine to ultrafine with grain sizes below 10 µm. The PGM mineralization in the two ores was predominately the PGE sulphides with approximately 66 PGM area %. A strong presence of the PGE arsenides was reported in the normal ore (~20 PGM area %). Various investigations have shown that the PGE arsenides have poor floatability and may compromise PGE recoveries. In terms of liberation in the two feeds at secondary grinds, an effective liberation of 67% was reported in the normal ore in comparison to 43% in the altered ore. This infers that the majority of PGMs remained unliberated in the altered ore. The QEMSCAN results also illustrated that the majority of the unliberated PGMs in both ores are associated with silicates and alteration silicates over oxide minerals.

6.1.2. Baseline batch flotation tests

High recovery (Pt-80% and Pd-81%) in the baseline flotation test (with no oil) was reported in the normal ore in comparison to the altered oil (Pt-67% and Pd-73%). Similar trends were observed with respect to the grade - normal ore (Pt-16 g/t and Pd-10 g/t) versus altered ore (Pt-5.5 g/t and Pd-5 g/t). The low recovery observed in the altered ore relative to the normal was due to the low liberation of PGMs in the altered ore. Additionally, high phyllosilicate alteration content in altered ore as well as the high slurry viscosity in the absence of oil supported the reported baseline recovery, which suggested rheological and slime coating problems. The low grade in the altered ore was due to being more weathered in comparison to the normal ore, as evidenced by the bulk mineralogy results.

6.1.3. Effect of oil on flotation performance of the normal and altered ore

The addition of oil on the normal ore resulted in a gradual decrease in Pt and Pd grade by approximately 3 g/t, from no oil to 500 g/t oil condition. With respect to the recovery, the oil had no negative impact – a slight in increase in Pt was reported (4%) and constant recovery for Pd. The oil adsorption study revealed that the majority of the oil adsorbed on the particle surfaces, rather than remaining in the liquid, which was postulated to result in particle agglomeration. An increase in the froth stability along with solids and water recovery was reported with increasing oil concentration in the normal ore. The increase in froth stability resulted in unselective recovery of gangue minerals as evidenced by the concentrate mineralogy results, leading to reduced concentrate grade. The concentrate mineralogy demonstrated that the grade dilution was likely due to true flotation (talc recovery) and gangue entrainment (chromite, orthopyroxene and plagioclase). Therefore, the grade dilution was ascribed to the increased froth stability which lead to the increased recovery of gangue minerals. The observed recovery increase was ascribed to an increase to solids recovery with increasing oil concentration.

With respect to the altered oil, the addition of oil (0-500 g/t) had little effect on Pt and Pd grade (nearly constant, Pt~5 g/t and Pd~4 g/t) and an overall gradual decrease their recoveries (Pt -6% and Pd -12%). The grade trends were consistent with constant solids and water recovery observed with increasing oil concentration. Similar to the normal ore, the majority of oil adsorbed on the particle surfaces of the altered ore, and with the altered ore comprising higher talc content relative to the normal ore, this was believed to have resulted in greater particle agglomeration. The addition of oil resulted in a significant increase in froth stability as well as the slurry apparent viscosity, compared to the normal ore. The increased viscosity was postulated to cause poor gas dispersion and reduced bubble-particle collisions as well as slime coating by ultra fine particles, leading to the observed recovery losses. Additionally, the majority of the oil was observed to selectively report to tailings compared to the concentrate. This was consistent with the recovery losses.

With respect to the grade, the finding from the normal ore supported the postulated grade hypothesis which states that *"High oil concentrations will reduce the concentrate grade due to oil coating both hydrophobic and hydrophilic gangue minerals, rendering them hydrophobic while also decreasing depressant adsorption and enhancing gangue mineral recovery".* However, the experimental finding of the altered ore was in contrast with the grade hypothesis, as the grade was nearly constant with increasing oil concentration due to the relatively constant mass recovery.

With respect to the recovery, the altered ore finding supported the hypothesis which is as follows : *"The presence of fine phyllosilicate alteration minerals and a high concentration of hydrocarbons in the slurry will increase pulp viscosity, due to the agglomeration of phyllosilicate minerals with increased hydrophobicity. High pulp viscosity leads to reduced* *bubble-particle interaction and poor gas dispersion. This, in turn, will lead to a reduction in flotation recovery".* The observed recovery trend in the normal ore was in contrast to the above hypothesis, since the recovery slightly increased (Pt) or remained constant (Pd) with increasing oil concentration. This may be due to the lower phyllosilicate content in the normal ore.

6.1.4. Effect of sodium metasilicate on the detrimental effect of oil

For mitigation of the deleterious oil effects, the addition of SS (1500 g/t) in the normal ore increased the grade for both Pt and Pd. This was more apparent at lower oil concentrations (0 and 300 g/t) compared to higher oil concentration (500 g/t). This was ascribed to SS improving the froth drainage by lowering the froth stability as well as the solids and water recovery. With regard to the recovery in the normal ore, SS reduced the recovery slightly, with the decrease being more noticeable at higher oil concentration (500 g/t, Pt-5% and Pd-1%). This was due to the reduced solids recovery in the presence of SS.

In the altered ore, SS increased both the grade and recovery of Pt and Pd. Similar to the normal ore, the grade enhancement was not apparent at lower oil concentration. Pt recovery increased by 8% and Pd by 6%. The increase in the grade was due to the improvement in froth drainage, similar to the normal ore. The increase in recovery was ascribed to dispersion of particles as a result of adding SS, further lowering the slurry apparent viscosity.

The postulated hypothesis regarding sodium silicate is as follows: *"Sodium metasilicate will enhance recovery and grade due to electrostatic repulsion between particles upon adsorption, increasing the dispersion of particles, reducing the viscosity and increasing froth drainage."* The grade improvement in the presence of SS in both ores supported the hypothesis. With respect to the recovery, only the recovery in the altered ore was consistent with the hypothesis, whereas the normal ore recovery trends disproved the hypothesis.

6.1.5. Effect of a degreaser on the detrimental effect of oil

For the degreaser in both ores, there was no comparison against the condition where no degreaser was used due to laboratory equipment problems. Only the trends were compared (percentage difference between 0 g/t and 500 g/t). It was shown that the degreaser improved both the grade and recovery in both ores, as compared to when no degreaser was present. As stated earlier, the addition of the degreaser resulted in an unquantifiable improvement in flotation performance; therefore, further tests are still required.

6.2. Recommendations

The following recommendations are drawn after understanding the interaction mechanisms leading to reduced flotation performance and mitigation of those effects:

1. Both ores contain considerable quantities of phyllosilicate alteration minerals which might result in rheological problems. These are greater in the altered ore compared to the normal ore, and the depressant dosage (20 g/t) was very low in relation to the amount of gangue present. Therefore, additional tests with moderate and high depressant dosage should be performed to obtain an optimum dosage.

2. The presence of oil resulted in particle agglomeration which increased the slurry viscosity. Operating at low solids concentrations would aid in preventing the increased viscosity. Of course, at full-scale this would result in throughput decreases, which may not be acceptable to the operation.
3. The use of sodium metasilicate to mitigate the viscous effects of oil was successful in both ores- with only the recovery in the normal ore slightly negatively affected. Therefore, the optimum SS dosage should be investigated, which would work in both ores without negatively affecting either the grade or the recovery. Mass pull would need to be carefully monitored to ensure no loss in recovery.
4. Additional degreaser experiments should be performed since the increase in flotation performance was unquantified. Also, the effect of the degreaser on froth stability should be assessed.
5. The oil adsorption experimental work showed that majority of the oil adsorbed on particle surfaces. The effect of the oil on mineral hydrophobicity should extensively be studied by measuring the contact angle of oil coated particles.
6. SS successfully worked on the ores; however, it should also be tested on the mine sludge, which contain fine-grained PGM locked in oil.

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Appendix A

Batch flotation calculations

Determining the operating wt.% solids in the 8L modified Leeds batch cell

The ore and slurry density used for this calculation were provided by Two Rivers Platinum mine - emulating the secondary rougher conditions at their concentrator.

ρ_{ore}= 3650 kg/m^3 ρ_{slurry}= 1380 kg/m^3 ρ_{water}= 1000 kg/m^3

$$\%solids = \frac{\rho_{ore}(\rho_{slurry} - \rho_{water})}{\rho_{slurry}(\rho_{ore} - \rho_{water})} \qquad \text{A-1}$$

%solids = 37%

Determining the mass of solids corresponding to %solids

$$m = V.\rho_{slurry}$$

m= 4.1 kg A-2

Determining the water required during the milling phase

Milling at performed at 66 wt.% solids percent.

$$m_{pulp} = \frac{m_{solids}}{\%solids}$$

$$m_{pulp} = 6.21\ kg$$

$$m_{water=}m_{pulp-m_{solids}} \qquad \text{A-3}$$

$$m_{water} = 2.11\ kg$$

$$V_{water} = \frac{m_{water}}{\rho_{water}}$$

$$V_{water} = 2.11\ L$$

Froth stability calculations

Determing the J_g

The column was operated at the air flow rate of 13 L/min

$$J_g = \frac{Air\ volumetric\ flowrate}{Column\ cross\ sectional\ area} \qquad \text{A-4}$$

$$= 0.69\ cm/s$$

Rheology calculations

Determining the UG2 mass required at 40 vol.% solids

The normal and altered ore densities were determined with the pycnometry test.

Table A-0-1: Normal and altred ore pycomentry densisty result.

	P (g/cm^3)
Normal ore	3.612
Altered ore	3.419

The cup volume was 50 ml.

$$solids_{volume} = vol.\%solids\ x\ cup_{volume} \quad \text{A-5}$$

$$mass_{solids} = density\ x\ solids_{volume} \quad \text{A-6}$$

$$water_{volume} = total\ volume - \ solids_{volume} \quad \text{A-7}$$

Equation A-5 to A-7 were used to determine the mass of solids corresponding to 40 vol.% solids and the rest of the calculation are tabulated below.

Table 0-2: Summary of rheological calculations of the normal and altered ore at 1000 g/t oil condition.

	% solids by volume	Sample volume (ml)	Sample mass (g)	Remaining water volume without reagents (ml)	Mass of oil (g)	Volume of oil (ml)	water volume required (water before- reagents volumes)
Normal ore	40%	20	68,38	30	2,634	3,00	27,00
Altered ore	40%	20	68,38	30	2,634	3,00	27,00

Equation A-5 to A-7 were also used for the rheological calculations of talc. Theoretical density of talc was used for these calculations, as cited in chapter 3.

Table 0-3: Summary of rheological calculations of the pure talc (20 and 30 vol.%) at 200 and 500 g/t oil condition

200 g/t Oil

% solids by volume	Sample volume (ml)	Sample mass (g)	Remaining water volume without reagents (ml)	Mass of oil (g)	Volume of oil (ml) @200g/t oil	water volume required (water before- reagents volumes)
20%	10	27,00	40		0,120	39,88
30%	15	40,50	35		0,190	34,81

500 g/t Oil

% solids by volume	Sample volume (ml)	Sample mass (g)	Remaining water volume without reagents (ml)	Mass of oil (g)	Volume of oil (ml) @ 500g/t oil	water volume required (water before- reagents volumes)
20%	10	27,00	40		0,310	39,69
30%	15	40,50	35		0,460	34,54

QEMSCAN validation chart

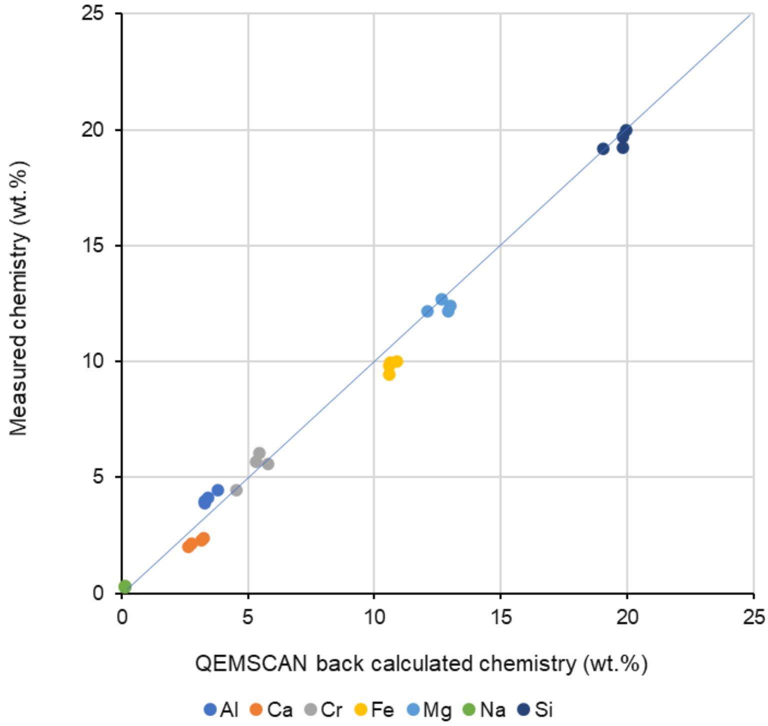

Figure A-1: Comparison of the back calculated elemental chemistry derived from QEMSCAN versus the measured mineral chemistry. The x=y line is also shown.

Froth stability graph

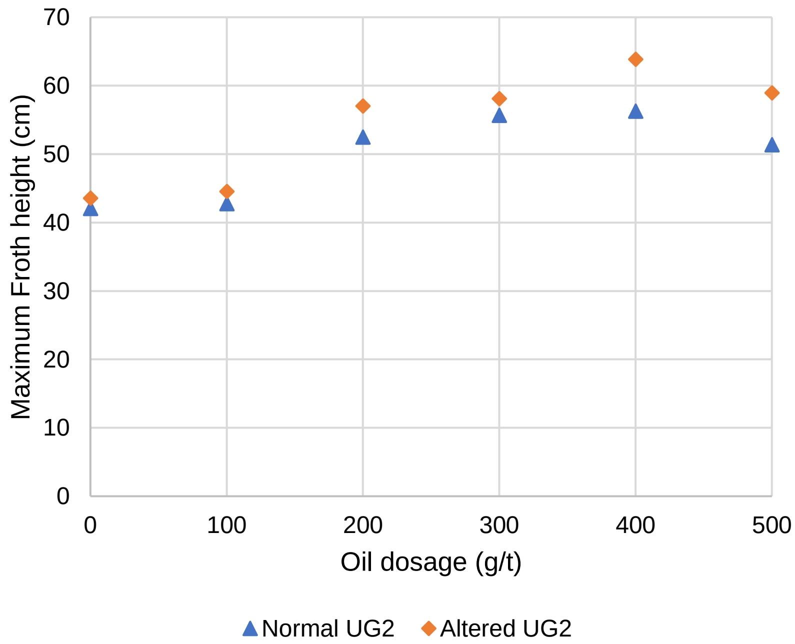

Figure A-2: Average maximum equilibrium height attained in the normal and altered ore.

Oil adsorption tests

The adsorbed oil per area of the particle is illustrated below.

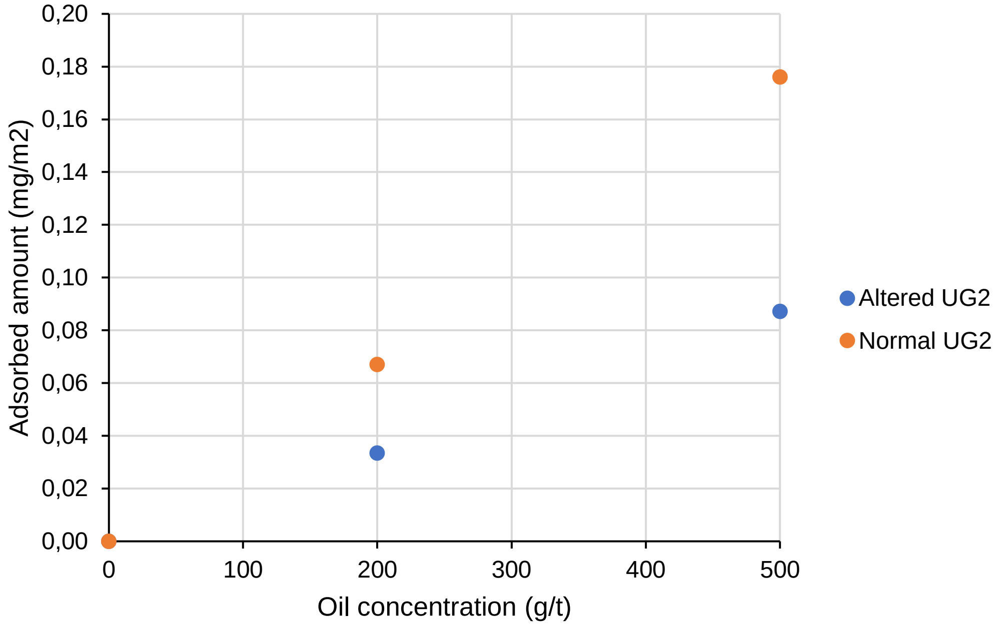

Figure A-3: The normal and altered ore TOC adsorption test.

QEMSCAN False colour image for the two feed samples

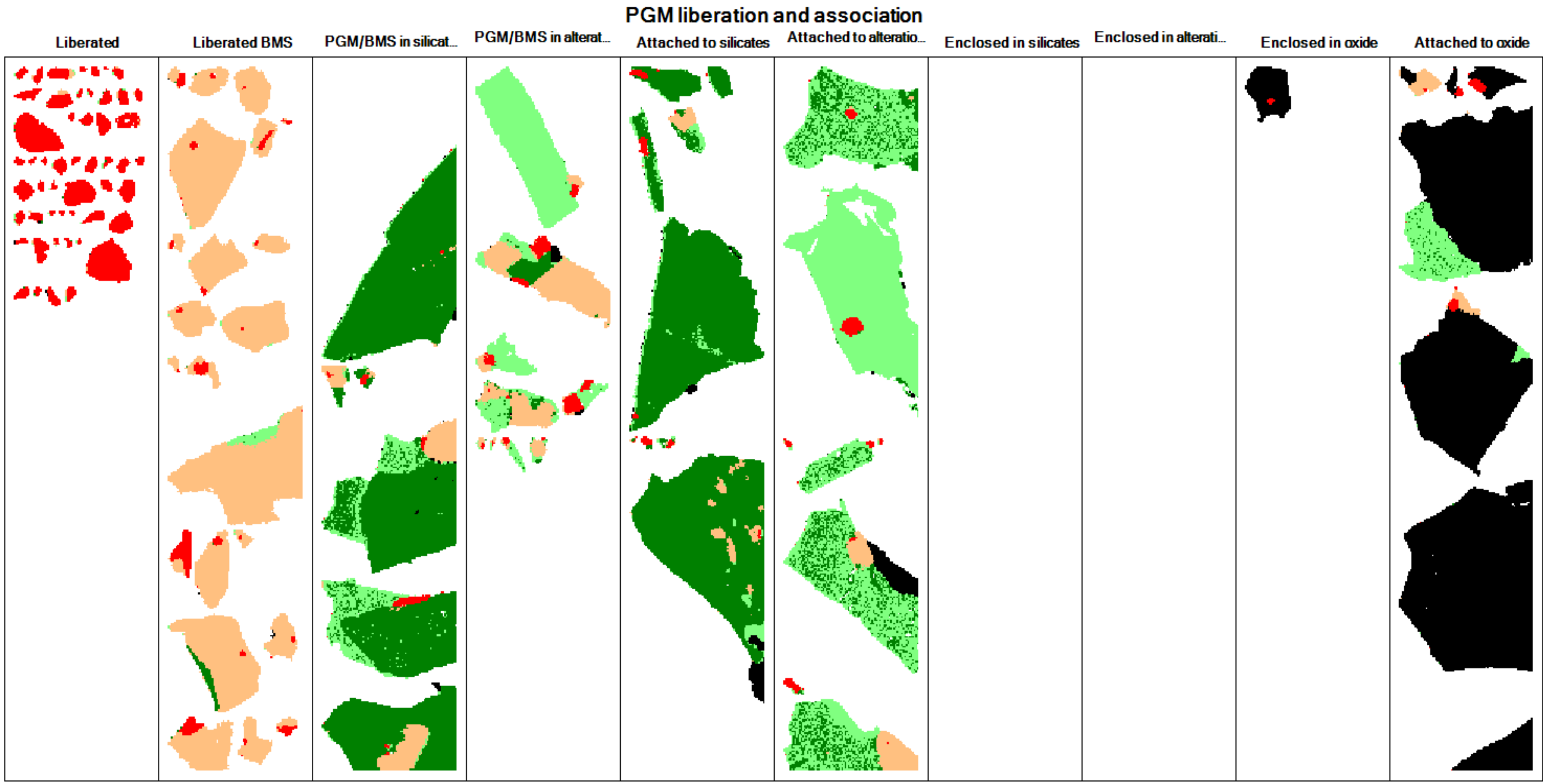

Figure A-4: Normal ore PGM liberation and association.

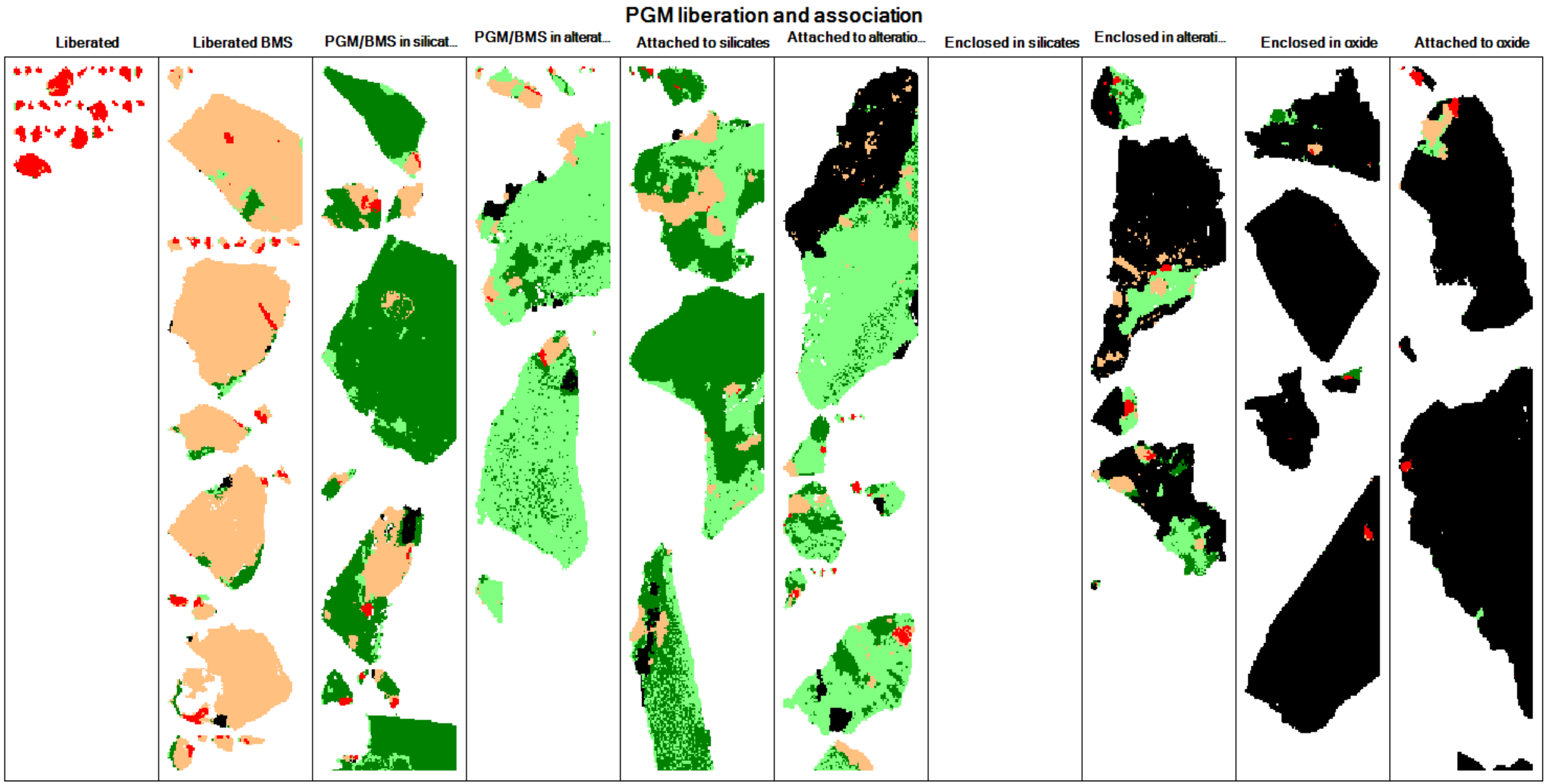

Figure A-5: Altered PGM liberation and association.

Appendix B: online

The raw data of all experiments are available online at the following DOI: https://doi.org/10.25375/uct.24183843.v1