An investigation into the impact of residual reagents on flotation in response to process water recirculation

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Plagiarism Declaration

I declare that the contents of this thesis, submitted to the University of Cape Town for the degree of Master of Science in Engineering, are original and have not been submitted prior to this for an academic examination towards any qualification at this university or any other institution.

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Synopsis

In the mineral pulp phase of the flotation process, water makes up about 80-85% of the constituents, thus making water an important component. The increasing scarcity of water has resulted in greater environmental restrictions which have led to the necessity of mine operations recycling water from tailing dams, thickener overflow, dewatering and filter products. However, water quality is found to influence the process of ore flotation, the results of various studies investigating the effects of water and recirculating water have seen an adverse effect on flotation selectivity.

The particular focus of the study was to consider the impact of dewatering reagents which may have been present in water recycled from the tailings back into the flotation process. The ore was from the Merensky reef which is found in the Bushveld Igneous Complex (BIC). The complex contains the world largest deposits of platinum group elements. About 75% of the world's platinum reserves and 50% of the world's palladium reserves are in the bushveld igneous complex, making it one of the most economically significant mineral deposit complexes in the world.

The primary dewatering reagents considered were flocculants and coagulants. There are two mechanisms by which flocculation occurs: the polymer bridging theory and the patch model. Polyacrylamides (PAM) are the most common flocculants. Coagulants use a charge neutralisation mechanism to take effect with the most common type being used are salts containing highly charged cations such as Al^{3+} and Fe^{3+}.

The experimental design chosen was a factorial with midpoint analysis. Three factors that were considered at varying dosages; the depressant, flocculant and coagulant. The two levels were high and low (where low was no addition of the respective reagent and high was equivalent to the concentrate seen on site recirculated process water). The key performance parameters used to determine effects were solid recovery, water recovery, copper and nickel mineral recovery, copper and nickel mineral grade, residual concentrations of collector and depressant pre and post flotation as well as particle settling of the tails.

For all test conditions relative to the base condition where none of the varied reagents were added, the solid and water recovery increased, the nickel recovery increased slightly while its grade decreased, the copper recovery was unaffected, but the grade decreased, there was no impact on the residual concentration of the collector and depressant pre and post flotation. The settling of the tailings also remained unaffected.

The solids and water recovery increased due to the agglomeration of finer particles in the system which resulted in the particles having better momentum for particle bubble attachment and faster recovery rates. There may have been a slight effect of the increase in ionic concentration of the water caused by the presence of additional reagents which resulted in a more stable froth and may have increased the recoveries of the solids. The recoveries of the value bearing minerals did not change, the grade however decreased because the increase in solids recoveries is attributed to increased gangue recovery owing to the fine particle size, which, in the presence of dewatering agents would have a faster recovery rate. The residual reagents concentration did not change as the primary impact on their adsorption would have been a change in the surface area due to an increase in particle size, which was not varied by grinding in this study. However, owing to the addition of dewatering agents, the particles may have agglomerated and hence become larger. The depressant adsorption may have been unimpacted owing to the inability of the depressant to fully cover the particle surfaces under standard flotation conditions. The collector adsorption was unimpacted and this is assumed to be owing to the concentration of the dewatering agents used within this study not being high enough to compete with the collector for mineral surface area. The settling of the tails remained unaffected as the finer particles which, in the presence of dewatering agents reported to the concentrate and had little impact on the composition of the tails.

It can be deduced from the results that the possible introduction of the dewatering reagents into the flotation system due to recycling of tailings water may be detrimental to the system. The primary impact of flocculants and coagulants in the flotation process was the introduction of more gangue in the concentrate which decreases the grade of the concentrate and will need to be further refined downstream. Furthermore, there is no positive impact on the settling of the tails, implying that the flocculant and coagulant action is suppressed during flotation and further flocculant and coagulant would need to be added during dewatering but may need careful screening prior to the recirculated water entering the milling and flotation stages for a mineral processing circuit.

Abbreviations

BIC	Bushveld Igneous Complex
PGEs	Platinum Group Elements
PGM	Platinum Group Minerals
CMC	Carboxymethyl Cellulose
UG2	Upper Group 2
ANOVA	Analysis of variance
SIBX	Sodium Isobutyl Xanthate
XRF	X-ray Fluorescence
μL	Microliter
g/ton	Grams per ton
mg/kg	Milligrams per kilogram
mg/L	Milligrams per litre
m/m	Mass per mass
L/min	Litres per minute
NTU/min	Nephelometric Turbidity Units per minute
Ca^{2+}	Calcium ion
Cl^{-}	Chloride ion
Mg^{2+}	Magnesium ion
K^{+}	Potassium ion
Na^{+}	Sodium ion
H^{+}	Hydrogen ion
µm	Micrometre
Fe^{3+}	Iron (III) ion
Al^{3+}	Aluminium (III) ion
TDS	Total Dissolved Solids
mm	Millimetre
kg	Kilogram
CO_3^{2-}	Carbonate ion
SO_4^{2-}	Sulfate ion
mL	Millilitre
mol/L	Mol per litre
km^2	Kilometres Square

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1 Introduction

1.1 Background to the Study

Flotation is an important and versatile mining technique that was originally patented in 1906 (Napier-Munn and Wills 2005). The process is used to separate value-bearing minerals in crushed ore from unwanted material by using or inducing hydrophobic behaviour in the value-bearing minerals.

In the mineral pulp phase of the flotation process water makes up about 80-85% of the constituents making it an important component (Muzenda 2010). The increasing scarcity of water has resulted in greater environmental restrictions which have led to the necessity of mine operations recycling water from their operations as seen in Figure 1. The water is recycled from tailing dams, thickener overflow and dewatering and filter products (Muzenda 2010). The advantage to recycling water within a flotation plant is that it lowers the water demand and discharge from the process while also retaining reagents that can be used again thus also lowering reagent consumption (Muzenda 2010). However, water quality is found to have an effect on the process of ore flotation, the results of various studies investigating the effects of water and recirculating water show that this can have an adverse effect of flotation selectivity (Muzenda 2010).

The ore that will be the focus of the study is the Merensky reef which is found in the Bushveld Igneous Complex (BIC). The BIC is in South Africa, primarily in the North Western province. It covers a massive area of about 66,000 km and contains the world largest deposits of Platinum Group Elements (PGEs). About 75% of the world’s platinum reserves and 50% of the world’s palladium reserves are in the BIC making it one of the most economically significant mineral deposit complexes in the world (Hunt and Lever 1969).

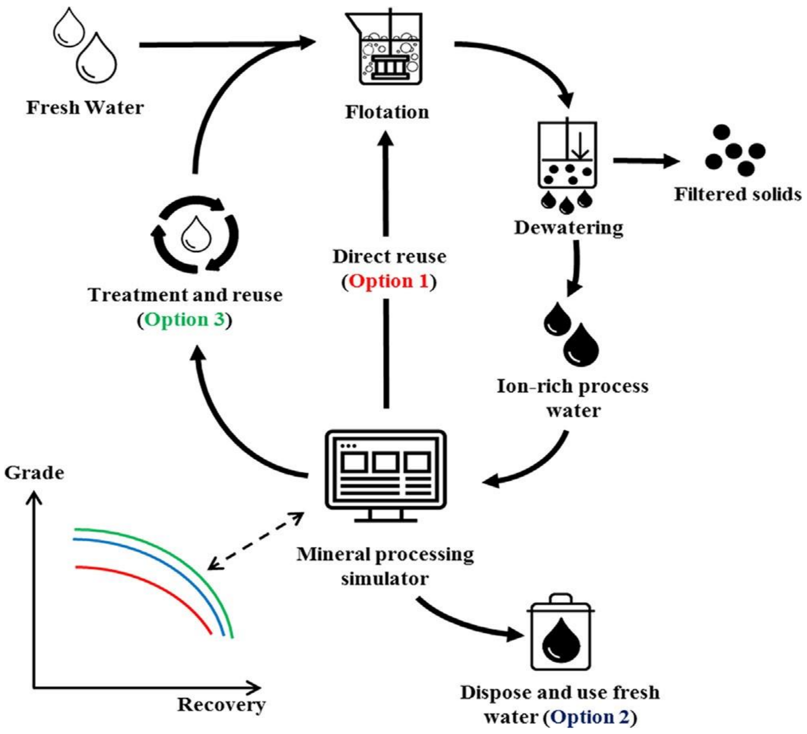

Figure 1 Uses of Recycled Water in the Flotation Process (Michaux et al. 2018).

The particular focus of this study is the impact of dewatering reagents on the flotation process. The primary reagents that are used in the dewatering process are flocculants and coagulants. Flocculants and coagulants are primarily used for the agglomeration of fine particles leading to their settling and easy removal from mixture (Sharma et al. 2006).

With water being increasingly recycled in the mining industry, the effects of these reagents on the flotation needs to be studied. The primary effect expected from these reagents being recycled is the effect on particle size of the flotation feed. The size of particles has long been studied and found to be a crucial factor that impacts the outcome of the flotation process (Feng and Aldrich 1999). Thus, making this study on the impacts of these reagents necessary to determine if there exists any detrimental or beneficial effect from dewatering reagents as very little has been done to examine their effects.

1.2 Overall Aim of the Study

The aim of this study is to investigate the effects of dewatering reagents on the flotation process and possible impact on the settling of the tails from the flotation process.

1.3 Scope, Limitations, and Key Issues

There are two key types of reagents that are used for the thickening process, namely, flocculants and coagulants. The dosage of these along with the depressant dosage will act as the manipulated variable. The study will aim to observe the effects of each factor and if they have interactive effects as well. The key outputs monitored to determine the flotation process are grade and recovery.

The study considers the impact of flocculants and coagulants on the flotation process and the impact on the settling rate of flotation tailings. The parameters used to determine the effect on the flotation process are:

- Final solids recoveries
- Final water recoveries
- Final copper grades and recoveries
- Final nickel grades and recoveries
- Impact on collector and depressant adsorption
- The settling rate of the tailings

The key issues to this study are the concentrations of the dewatering reagents used, these are not an accurate or actual representation of the residual concentrations at an actual site but were synthesized from an actual site process water quality.

2 Literature review

2.1 Bushveld complex minerology

The bushveld igneous complex (BIC) seen in Figure 2 covers a land area of almost 60,000 km^2 in size and is the world's largest reserve of platinum group elements (PGEs) (Dzvinamurungu et al. 2013).

The PGE are six noble, precious metallic elements that are found together on the periodic table. The six elements are: ruthenium, rhodium, palladium, osmium, iridium, and platinum. These metallic elements have similar properties and tend to occur together in similar mineral deposits (Harris, 1991). They have useful properties such as: catalytic properties, resistance to wear and tear, resistance to chemical attack, high temperature characteristics, high mechanical strength, good ductility, and stable electric properties. These properties make it a valuable asset in various fields such as jewellery, industries, anticancer drugs, dentistry, electronics and vehicle exhaust catalyst (Hunt and Lever 1969). The various properties and uses that these metals offer make the BIC an economic asset.

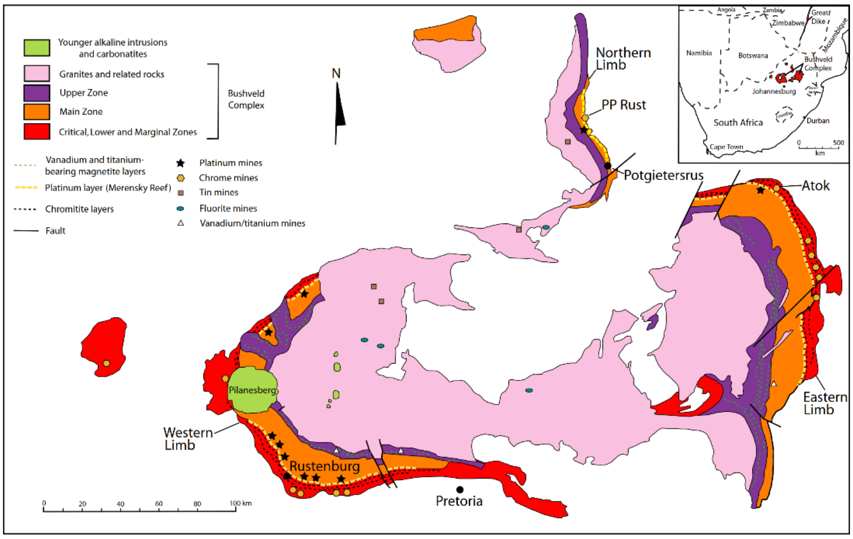

Figure 2 Bushveld Igneous Complex (Bushveld Igneous Complex - Wikimedia Commons n.d.).

One of the largest ore horizons that are mined in the southern sector are the Upper Group 2 (UG2) and Merensky reefs. The Merensky is a pyroxenitic layer that lies between a footwall and a hanging wall of igneous rock called anorthosite. The layer thickness ranges from 0.6 m to 15 m. The layer consists of chromite stringers, with which the sulfide mineralization tends to be associated (Liddel et al. 1985). There are significant amounts of gangue minerals that are contained in the Merensky reef, these can be seen in Table 1 which gives the composition of a typical Merensky ore. The major gangue minerals are talc, feldspar, and pyroxene. They are generally widespread and can be found in between the sulfides and the pyroxenes.

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Table 1 Mineralogy of a Merensky ore (Liddel et al. 1985).

Mineral	Abundance %
Talc	0.5-2
Feldspar	20-40
Pyroxene	50-70
Sulfides	<1
Other	6-7

The PGEs found in the BIC Merensky are strongly associated with base metal sulfide minerals. The major base metal sulfide is pyrrhotite, the other sulfides are pentlandite, chalcopyrite, pyrite and others which are present in smaller quantities. These sulfide minerals collectively are known as base metal sulfides (BMS) (Dzvinamurungu et al. 2013). Table 2 gives the composition of the sulfides found in a typical Merensky ore.

Table 2 Base metal sulfide ore composition (Liddel et al. 1985).

Sulfide	Abundance %
Pyrrhotite	45
Pentlandite	32
Chalcopyrite	16
Pyrite	2-4
Others	3-5

2.2 Fundamentals of Flotation

Flotation is a physico-chemical separation process that utilises the difference in surface properties of valuable minerals and unwanted gangue mineral (Napier-Munn and Wills 2005). The separation process is based on contacting solids with upward moving air bubbles in an agitated air vessel. The solid mineral particles present are interacted with chemical reagents (collectors) that create favourable conditions for valuable minerals to attach to air bubbles. The valuable minerals are then extracted from a froth stabilized by chemicals (frothers) and removed from the flotation cell (Klimpel 1984).

There are three mechanisms by which this process takes place: selective attachment to air bubbles (true flotation), entrainment of particles in the water that reports to the froth and physical entrapment between particles in the froth attached to air bubbles (Napier-Munn and Wills 2005).

The true flotation mechanism is the dominant mechanism and is responsible for the majority of particles that are recovered to the concentrate (Napier-Munn and Wills 2005). The true flotation mechanism relies on the difference in the wettability surface property of particles, the particle that need to be floated must selectively attach to the bubble surface thus requiring them to be hydrophobic (Shackleton 2020). Figure 3 depicts a typical flotation cell and how the mineral makes its way up to the froth. There are only a few minerals that carry this surface property of hydrophobicity, an example would be sulfide minerals, most minerals require the aid of surface-active reagents called collectors (Shackleton 2020).

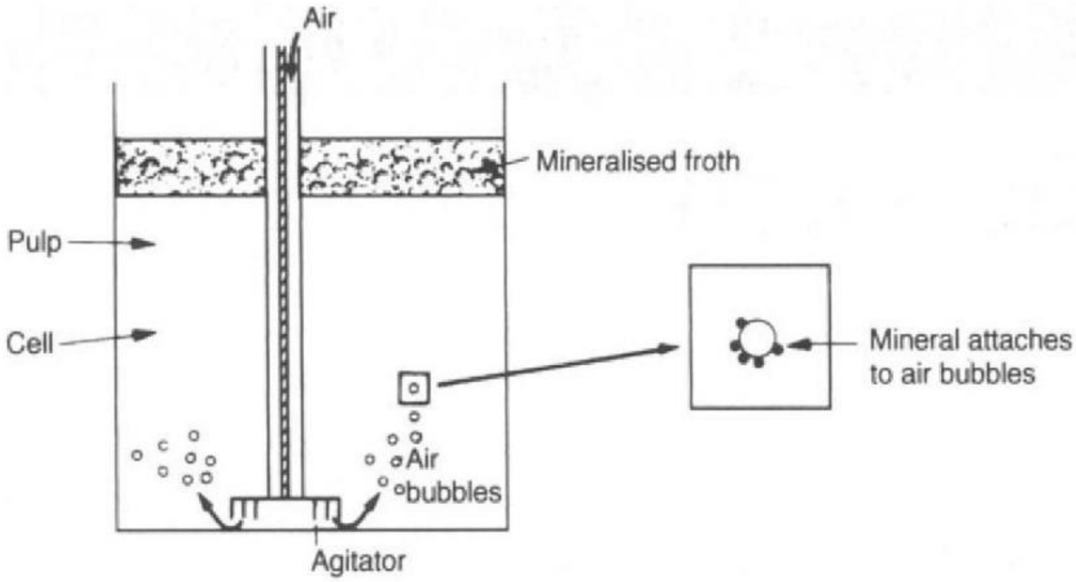

Figure 3 Flotation cell (Napier-Munn and Wills 2005).

The other two mechanisms; entrainment and entrapment do not contribute as many particles to the froth as true flotation but play an important role towards the efficiency of the separation. The entrainment and entrapment mechanisms are non-selective in nature which means they tend to take both the valuable mineral and the gangue to the froth thus impacting the separation efficiency in terms of the grade of the concentrates.

In the process of entrainment, the particles are conveyed by the froth through mechanical means, but they do not adhere to the air bubbles. This means that they do not accurately reflect the true flotation process. The reason the particles do not stick to the air bubbles is because they are hydrophilic, and instead they occupy the spaces between the bubbles in the water. Unfortunately, it is often the presence of slimes, consisting of unwanted minerals, that are conveyed by entrainment and can reduce the concentrate's grade(Rao 2004).

Entrapment like entrainment involves hydrophilic gangue minerals reporting to the froth, but they differ in mechanism. In entrainment the hydrophilic gangue mineral being squeezed within hydrophobic particles thus reporting to the froth, decreasing the separation efficiency of the process(Qiu et al. 2022).

2.3 Factors that Impact Flotation

There are three factors that affect the flotation system, chemistry, equipment, and operation (Figure 4). However, the highly interactive nature of these factors can present difficulty for operators to perform economic optimization of the process (Klimpel 1984).

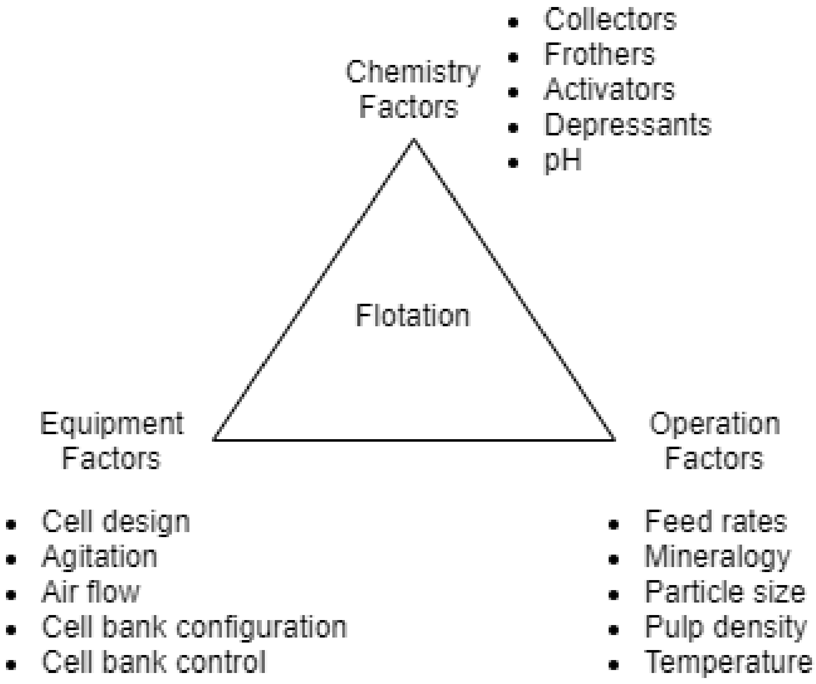

Figure 4 Factors that affect flotation (Klimpel 1984)

2.4 Reagents used in Flotation.

2.4.1 Collectors

Hydrophobicity can be manipulated in minerals in order to determine their floatability. This can be achieved with the aid of surfactants known as collectors (Napier-Munn and Wills 2005; Fuerstenau and Han 2009). Collectors are added to the pulp and act by interacting at the mineral surface by adsorption or by chemical reaction (Rao 2004). The collector molecule interacts with the surface of the mineral, reducing the stability of the hydrated layer separating the mineral surface from the air bubble to such an extent that on contact the particle attaches itself to the bubble (Napier-Munn and Wills 2005). The collector molecules can be classified into two groups of ionising (soluble) and non-ionising compounds (insoluble compounds). The insoluble compounds act by creating a thin film around the surface of the mineral making it hydrophobic (Rao 2004).

Soluble collectors are the group that are more widely used in flotation, they are asymmetric in structure and are heteropolar. They consist of a non-polar hydrocarbon group and polar group. The non-polar hydrocarbon radicle exhibits hydrophobic properties while the polar groups reacts with the mineral surface, this can be seen in Figure 5 (Napier-Munn and Wills 2005; Fuerstenau and Han 2009).

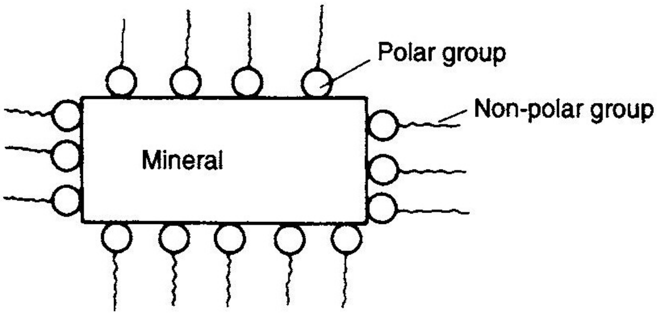

Figure 5 Collector mineral interaction (Napier-Munn and Wills 2005).

The ionising collectors can be classified by the type of ion, anion or cation, present that is responsible for producing the water-repellent effect. The classification of the collectors can be represented as shown in Figure 6.

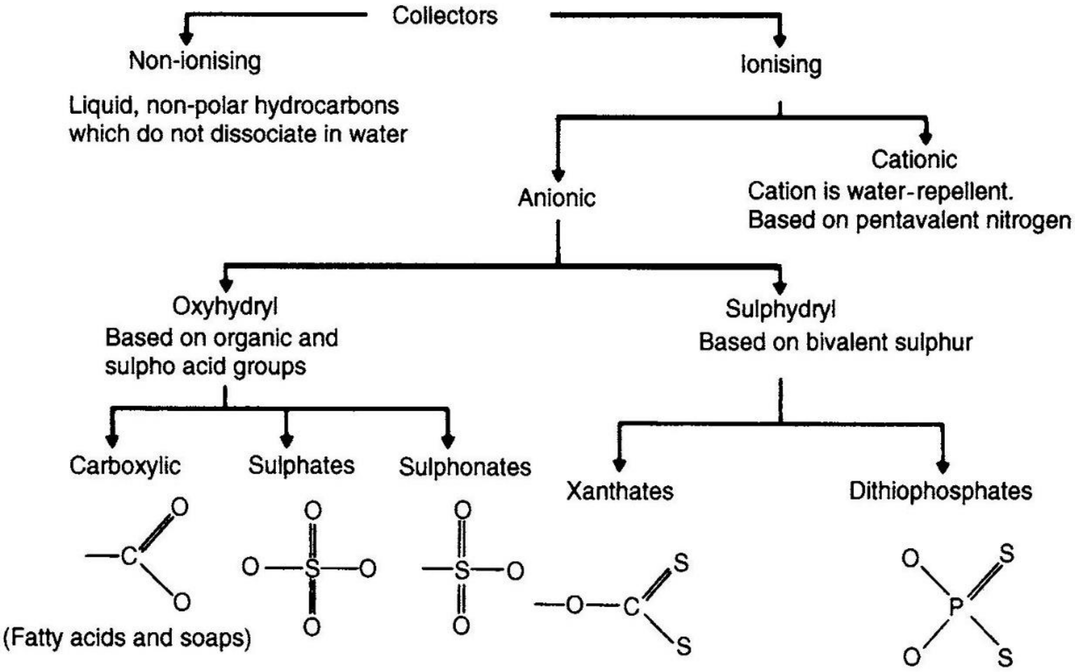

Figure 6 Collector reagent classification (Napier-Munn and Wills 2005).

Sulfide mineral collectors are all based on divalent sulfur and the majority have short hydrocarbon chains that do not exceed C5. The sulfide minerals group is the largest in terms of tonnage that is processed by froth flotation and in terms of volume reagent used and the alkyl xanthates are by far the most used sulfide collectors (Pearse 2005). The chemical structure of alkyl xanthates and a few other commonly used collectors can be seen in Figure 7.

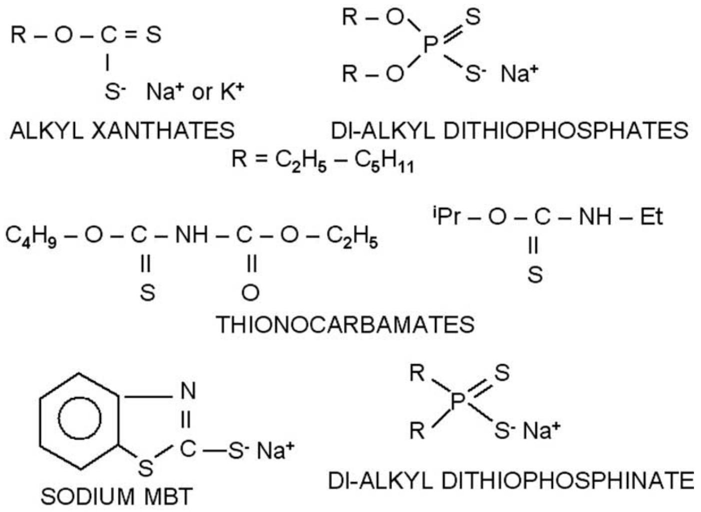

Figure 7 Sulfide collectors (Pearse 2005).

2.4.2 Frothers

Frothers are added to stabilise bubble formation in the pulp phase, the creation of a stable froth allows selective drainage of entrained gangue and increases flotation grade and kinetics. A good frother has negligible collecting power and the ability to produce a stable enough froth that does not result in froth build up (Napier-Munn and Wills 2005; Rao 2004).

Frothers are chemically similar to ionic collectors, they are generally heteropolar surface-active organic reagents capable of being adsorbed at the air-water interface. When frothers react with water the dipoles easily combine with the polar groups and hydrate them, but there is no reaction with the non-polar hydrocarbon group. This results in the non-polar hydrocarbon being associated with the air phase (Napier-Munn and Wills 2005; Fuerstenau and Han 2009).

The frother action can be simplified to the frother molecule’s ability to adsorb on the air-water interface because of its surface activity and to reduce the surface tension, thus stabilising the air bubble. The action of the frother can be seen in *Figure* 8.

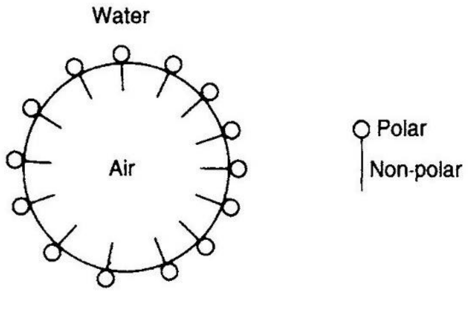

Figure 8 Action of the frother (Napier-Munn and Wills 2005).

Frothers are neutral molecules made up of medium chain length hydrocarbon and a polar group. The polar groups of frother are usually hydroxyls in the form of alcohols or glycols. A few common frothers can be seen in Figure 9.

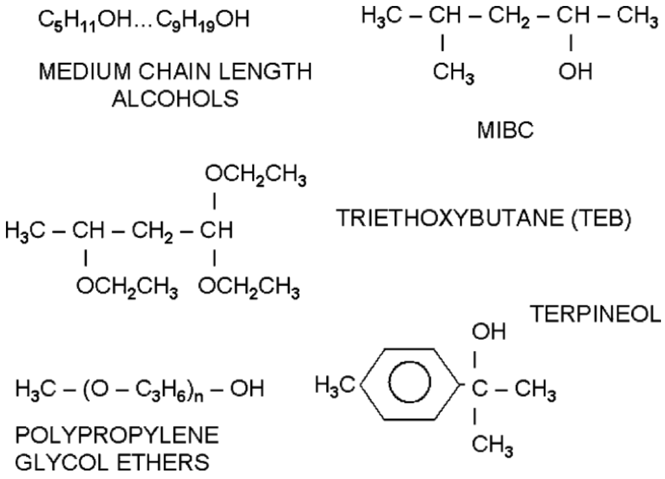

Figure 9 A few common frothers (Pearse 2005).

2.4.3 Depressants

Depressants are used to increase the selectivity of the flotation process. They act by rendering certain minerals hydrophilic, thus preventing them from interacting with air bubbles and reporting to the froth. There are many types of depressants; their role in flotation can be complex and varied, and in most cases not fully understood. This makes it difficult to control compared to other reagents (Napier-Munn and Wills 2005; Fuerstenau and Han 2009).

In the South African platinum group mineral (PGM) industry, polymeric depressants such as carboxymethyl cellulose (CMC) seen in Figure 10 are widely used to depress talcaceous gangue minerals. An advantage of polymeric depressants is that they are less environmentally hazardous when compared with the more widely used inorganic depressants such as sodium silicate and sodium sulfite (Napier-Munn and Wills 2005; Bratby 2016).

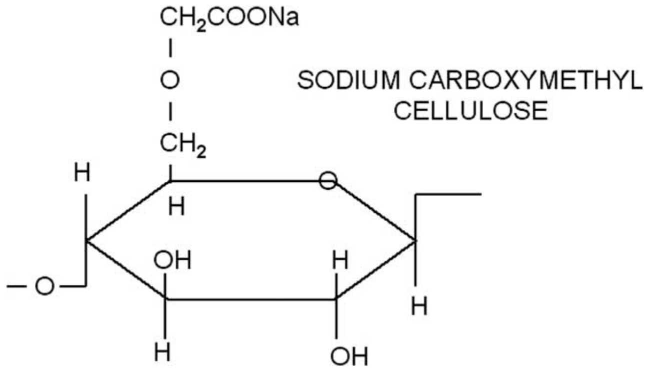

Figure 10 CMC molecule structure (Pearse 2005).

The mechanisms through which depressants act on the surface of gangue includes blocking and binding of the gangue surface which prevents collector adsorption and renders the particle hydrophilic (Pearse 2005).

2.5 Water Recovery

Most mineral processes involve a substantial amount of water, and the final concentrate must be separated from a pulp in which the water-solids ratio may be high. There are multiple dewatering methods that can be used but the bulk of the water about 80% can be recovered by sedimentation (Napier-Munn and Wills 2005; Bratby 2016).

The sedimentation process is the rapid settling of solid particles in liquid which produces a clarified liquid that is to be decanted, leaving behind a thickened slurry. It is most efficient when there is a large density difference between the liquid and the solid particles. Particles that are only a few microns in diameter can take extremely long to settle by gravity alone, to tackle this problem the coagulation and flocculation processes can be used sometimes with the aid of dewatering chemical reagents (Napier-Munn and Wills 2005; Fuerstenau and Han 2009).

2.5.1 Coagulation

The coagulation process causes extremely fine colloidal particles to adhere directly to each other. All particles exert mutual attraction forces known as Van der Waals forces. In normal circumstances adhesion due to these forces is prevented by the electrically charged region around each particle which generates repulsive forces between particles approaching each other. This results in a balance in any given system between the attractive and repulsive forces present at the solid-liquid interface which can be represented in Figure 11 (Napier-Munn and Wills 2005; Bratby 2016).

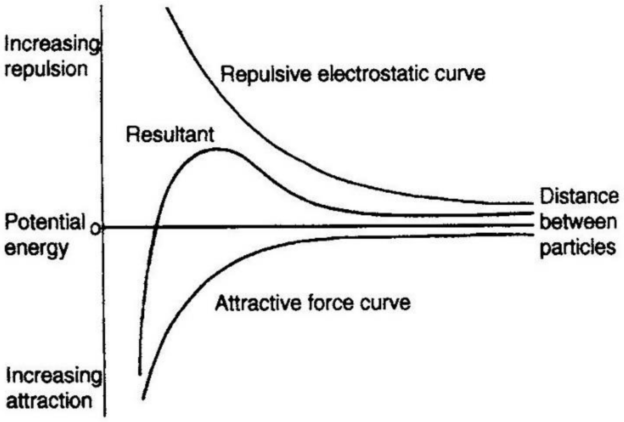

Figure 11 Potential energy curves for two particles approaching each other (Napier-Munn and Wills 2005).

Reagents known as coagulants are used to overcome these repulsive forces which would then cause the fine particles to adhere to each other. Coagulants are electrolytes having an opposite charge to the particles, this causes the neutralization of the system when dispersed. Inorganic salts have long been used for this purpose and as counter ions in aqueous systems are commonly positively charged. Salts containing highly charged cations such as Al^{3+} and Fe^{3+} are mainly used (Napier-Munn and Wills 2005; Bratby 2016).

2.5.2 Flocculation

Flocculation is a process whereby finely divide or dispersed particles are aggregated together to form large particles which can then settle and cause the clarification of the system. Flocculants are the reagents which are used in this process to decrease the repulsive forces and increase the attractive forces (Sharma et al. 2006).

There are two mechanisms by which flocculation occurs: the polymer bridging theory and the patch model. In the bridging theory, the destabilization of the particle occurs when the polymer chain adsorbs on more than one particle, this results in the linking of multiple particles together. At the contact point between a colloidal particle and polymer, the interaction between the two is limited to a site leaving the rest of the polymer molecule available for further adsorptions. The remaining segments of the polymer molecule are extended into the solution as seen in Figure 12 a, the polymer would then adsorb onto the surface in a series of loops (segments extending in the solution and trains (segments adsorbed on the surface). These loops and trains can further attach themselves to colloidal particles that have free adsorption sites which result in the formation of a particle-polymer-particle aggregate which can be seen in Figure 12 b, the polymer in this aggregate serve as the bridge. In the event that excess polymer is added in the system, then the particles would restabilize by surface saturation and can be sterically stabilized as in Figure 12 c (Sharma et al. 2006).

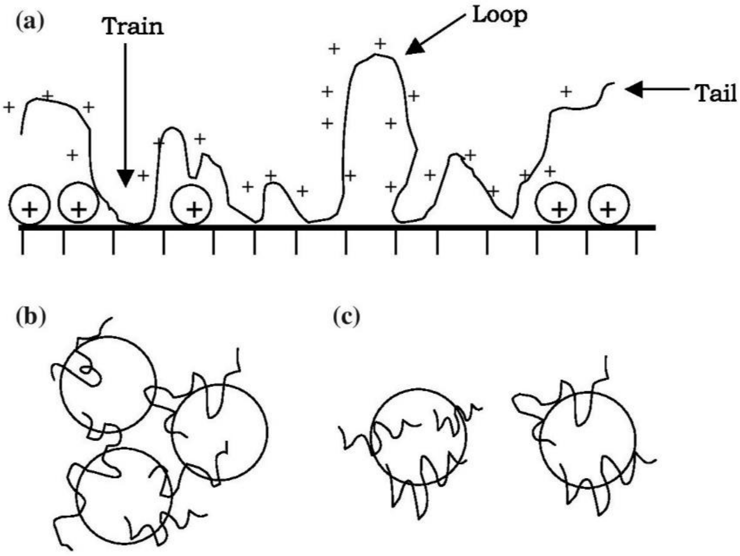

Figure 12 Polymer bridging mecahnism (Sharma et al. 2006).

The patch model or the electrostatic mechanism, involves the use of uneven distribution of charges resulting from adsorption of discrete patches of polymers on the surface. Highly cationic polymers are adsorbed onto a negative particle surface as seen in Figure 13, this promotes flocculation by reducing the overall negative charge which reduces the inter repulsive forces, this is referred to as charge neutralization. In addition to charge neutralization, positive regions of the polymer are attracted to negative regions on other particles, this is referred to as heterocoagulation (Sharma et al. 2006).

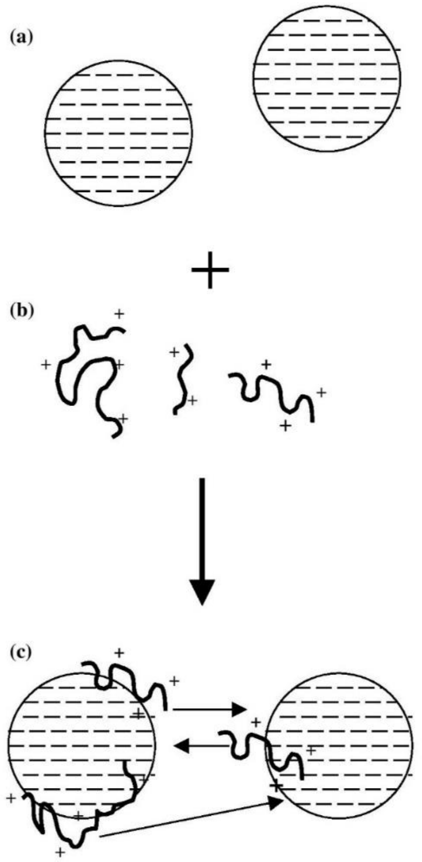

Figure 13 The patch model (Sharma et al. 2006).

There are two major categories of flocculants; these are organic and inorganic flocculants. Inorganic flocculants have been used for an extremely long time and consist of salts of multivalent metals such as aluminium and iron. However, there are several disadvantages in using them; they are consumed in large quantities and produce large amount of sludge, are highly sensitive to pH, work on a limited number of systems, and do not work on very fine particles. These reasons have led them to be almost abandoned in favour of organic flocculants (Sharma et al. 2006).

The organic flocculant category can be further divided into natural bio flocculant and grafted flocculants. It is the polymeric flocculants under these categories that have become vastly popular due to their natural inertness to pH changes, high efficiency with low dosage and easy handling (Lee et al. 2014). However, they have their cons in their degradability and environmental friendliness (Lee et al. 2014). The polyacrylamides (PAM) are the most common flocculants, these can be manufactured to have non-ionic, anionic, and cationic character. The structure of PAM can be seen in Figure 14 (Bulatovic 2016)

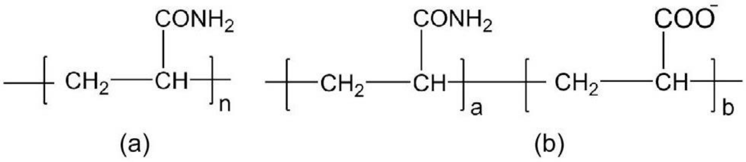

Figure 14 Chemical structure of PAM (Napier-Munn and Wills 2005).

2.6 Effects of Particle Size on Flotation

The effect of particle size on flotation is an area that has been significantly explored and having a feed that is too fine or coarse is known to have a detrimental effect on the performance of the process. Thus, making it important to operate at the optimum grind size (Feng and Aldrich 1999).The size at which a particle is classified as fine is less than 10 µm, while intermediate is classified between 10 to 70 µm and coarse being larger than 70 µm (Trahar 1981).

There are various factors that are impacted by the size of the flotation feed but in this study the particle-bubble interaction is a key factor. Particle size impacts the probability of the bubbles colliding, attaching to the bubble and staying attached to the bubble in the pulp phase (Feng and Aldrich 1999; Trahar 1981).

When the particle size is too fine, it results in slower recovery rates, due to reduced bubble-particle collision and results in more entrainment. These fine particles are also known to have much larger surface area, that can lead to excessive adsorption of reagents and other effects related to chemically active particles. This can lead to poor results in grades and recoveries (Feng and Aldrich 1999).

Conversely, when the particles are too large, it causes a disruption in bubble-particle aggregation in turbulent zones impacting the recovery rate of the particles. In addition, the buoyancy of the particle-bubble aggregate decreases relative to the pulp (Feng and Aldrich 1999).

This behaviour can be attributed to two properties seen in coarse particles. Coarse particles require a higher degree of hydrophobicity to achieve better floatability and when finer particles are present in the system, they tend to consume more of the collector reagent which leads to less surface coverage of the coarse particle leading to less floatability (Trahar 1981). However, Trahar (1981) noted that there is insufficient evidence proving that finer particles consume more reagent Trahar 1981).

Intermediate particles provide the best flotation results as they are well liberated and float at a rate that provides optimum recovery. Furthermore, the principal mechanism by which particles are recovered is true flotation, providing the best selectivity (Trahar 1981).

2.7 Effects of the Ionic Concentration of Process Water on Flotation

There have been several studies that have been conducted on the effects of ions in flotation systems due to increased recycling of water in the mining industry. The recycled water contains several inorganic and organic dissolved compounds that alter the chemistry of the system (Rao and Finch 1989).

Corin et al. (2011) and Manono et al. (2012) considered the effects of increasing the ionic strength of the water used in the flotation of sulfide ores. From these studies the authors were able to determine that as the ionic concentration of the plant water increased, the mass and water recoveries increased while the recoveries of the valuable minerals remained constant, but the grade however had decreased. These results obtained were attributed to the stabilisation and mobilisation of the froth seen with increased water recovery (Manono et al. 2012) and the decrease in grade was attributed to the fast-floating nature of the valuable minerals which resulted in most of the minerals being recovered and any additional recoveries being primarily gangue, reducing the grade of the concentrate.

Muzenda (2010), Levay et al. (2001), Farrokhpay (2011,) and Farrokhpay and Zanin (2012) studied how water quality plays a crucial role in the performance of flotation. One of the factors in water quality that impacts flotation was found to be the impact of ions and their concentration. The ions are known to impact the froth stability of the flotation process and cause the formation of smaller and more stable bubbles that result in an increase in water recoveries.

2.8 Summary and the Gap in Knowledge

The mining industry is increasingly recycling water, the effects of a change in water quality need to be studied as water quality is known to have an impact on the flotation process. The particular focus of this study is the potential impact that dewatering reagents used in thickeners could have if they were recycled back into flotation systems.

The key properties that dewatering reagents have are their ability to agglomerate smaller particles into larger ones and the ionic properties the reagents contain. It has been noted that the particle size plays a crucial role in the performance of the flotation process and works best on optimum size that is neither too coarse nor too fine. As having particles that are too fine results in poor particle-bubble attachment due to a lack of momentum in smaller particles. However, having particles that are too large is also detrimental to the system as they have less buoyancy, and the mineral particle may be less liberated (Feng and Aldrich 1999)

The other property is the ionic property of the dewatering reagent, there have been several studies that have been done on the effects of increasing ionic concentration of flotation water. These studies have seen an increase in froth stability, which has increased water and solid recoveries. The valuable mineral recoveries of copper and nickel are not impacted while a decrease in grade is noted (Corin et al. 2011; Manono et al. 2012).

There are some gaps in knowledge due to the novelty of the focus of this study, one being the impact of the dewatering reagents on flotation performance when these are contained in the water that is recycled into flotation. Hence the following gaps have been identified:

- The effect of dewatering reagents on flotation performance.
- How the dewatering reagents interact with the flotation reagents.
- Understanding the impact on the settling of the tailings due to a possible increase in the concentration of the dewatering reagents in recirculated water.

3 Research Aim, Key Questions, and Hypothesis Objectives

3.1 Overall Aim

The aim of the study is:

- To investigate the effects of residual flotation reagents and dewatering agents from water recirculation on flotation, with a particular focus on the coagulant, flocculant, and depressant.

3.2 Key Questions

The key questions of the study are:

- What is the effect of dewatering agents on key flotation outcomes?
- How do the selected dewatering reagents affect key flotation reagents?
- Do the dewatering reagents impact the settling of tailings?

3.3 Hypothesis

The hypothesis of the study is:

- With increasing residual dewatering agent concentration, the performance of the flotation system will deteriorate due to the ionic structure of the residual dewatering agents which may compete with flotation reagents for the mineral surface.

3.4 Sustainable Development Goals

Figure 15 is an image displaying the 17 sustainable development goals to be a blueprint to achieve a better and more sustainable future for all.

Figure 15 UN sustainable development goals (#Envision2030: 17 goals to transform the world for persons with disabilities | United Nations Enable n.d.)

The project addresses the following sustainability goals:

- Goal 6 - Clean water and sanitation: This goal is addressed with an attempt at making more efficient use of the wastewater produced to maximise recycling and a more efficient flotation system.
- Goal 12 - Responsible consumption and production: This goal is addressed with an attempt at achieving higher grades and recoveries for the minerals that will be extracted as this would allow maximum utilisation of a precious resource.

4 Experimental Method and Materials

4.1 Research Methodology

The research methodology that was implemented in this study is the factorial design. This is a method that is used when the experiment contains two or more factors, each with discrete levels and whose experimental units take on all possible combinations of these levels across all such factors.

The factorial design for this experiment considered three factors at three levels and therefore can be denoted as 3^3 factorial design. The three factors that were varied were the dosages of the depressant, flocculant, and coagulant. The three dosage levels that were varied can be seen in Table 3. The dosage concentrations were adapted from a South African PGM concentrator.

Table 3 Dosages for each level for the factors that were varied.

Levels	Reagent concentration variation (mg/ kg ore)		
	Depressant	Coagulant	Flocculant
Low (L)	0	0	0
Medium (M)	30	7.5	2
High (H)	60	15	4

The number of experiments required for this study for a full factorial method would have been twenty-seven excluding duplicates, this was reduced by cutting down to a three-factor two-level factorial design with 2 midpoint runs, the remaining midpoints were predicted with the aid of statistical analysis software Design-Expert®. Thus, the total number of runs required was twenty including duplicates, that ensured reproducibility. The experimental plan can be seen in Table 4 where D, C and F denote depressant, coagulant and flocculant respectively.

Table 4 2^3 factorial design experimental plan.

Number of experiments	Run name	Factors and levels for each factor		
		Coagulant dosage	Flocculant dosage	Depressant dosage
1	None	L	L	L
2	D	L	L	H
3	C	H	L	L
4	F	L	H	L
5	DC	H	L	H
6	DF	L	H	H
7	CF	H	H	L
8	DCF	H	H	H
9	Mid-Point	M	M	M
10	Mid-Point	M	M	M

4.2 Synthetic Plant Water Preparation

In order to best mimic the process water on site, a unique synthetic plant water recipe was developed using the methodology from Smith et al, (2002). This method considers each ion in a specific sequence to ensure the correct concentrations for each component are achieved.

The resulting plant water recipe are shown in Table 5 and Table 6, the salts were all supplied in powder form by Merck and were of analytical grade.

Table 5 Plant water recipe

Chemical Name	Chemical Formula	Concentration mg/L
Calcium Chloride hydrated	$CaCl.2H_2O$	63.64
Magnesium Chloride	$MgCl_2$	47.13
Potassium Chloride	KCl	24.23
Sodium Chloride	$NaCl$	176.43
Sodium Hydrogen Carbonate	$NaHCO_3$	200.17
Sodium Sulfate	Na_2SO_4	286.07

Table 6 Concentration of ions.

Ions	Concentrations mg/L
Ca^{2+}	17.32
Cl^-	184.60
Mg^{2+}	12.03
K^+	12.68
Na^+	216.89
H^+	2.38
CO_3^{2-}	142.98
SO_4^{2-}	193.34
TDS	782.22

4.3 Reagents

Table 7 shows the reagents that were used for the flotation and related tests which were dosed as supplied. The sulfuric acid and phenol were supplied by Merck, and the remaining reagents were provided by the mine site. The reagents were dosed as supplied and not corrected for active content.

Table 7 Reagents used in experimentation.

Reagent name	Main component	Quantity	Purpose
Gemcol	SIBX (Sodium Isobutyl Xanthate)	143 mg/kg at 11.5%	Collector
FinnFix300	CMC	see Table 3 and 4	Depressant
Flomin F200	Alcohol and ether	40 µL	Frother
WM 7530	Acrylamide-acrylate co-polymer	see Table 3 and 4	Coagulant
WM 8305	Poly Hydroxyalkylene dimethyl ammonium chloride	see Table 3 and 4	Flocculant
-	Sulfuric acid	5 mL at min 95%	Depressant concentration analysis
-	Phenol	1 mL at 5% m/m	Depressant concentration analysis

4.4 Batch Flotation Procedure

4.4.1 Milling Curve

A typical sample of a Merensky ore was received from the operation, it was crushed to a size of -1 mm, before being blended, riffled, and split using a rotary splitter into 1 kg samples which were stored in airtight bags in a sealed drum at room temperature. The required particle size was 80% passing 75 μm. The samples were milled in a 1 kg stainless-steel rod mill, the rods used were six of 25 mm x 285 mm, eight of 20 mm x 285 mm and six of 16 mm x 285 mm dimensions. The slurry solid percentage was 66% which was achieved with the addition 500 mL of synthetic water. To determine the milling time required to achieve this grind size the samples were milled at 15, 25, 39, and 50 min respectively. Thereafter, the milled samples were wet screened through a 75 μm screen. The procedure resulted in the milling curve that can be seen in Figure 16. The milling curve has a significant R^2 value that confirms the validity linear relationship. Using this equation, it was determined that the time required to achieve the target grind size of 80% passing 75 μm was 46 min and 46 sec, which was confirmed with further milling and screening at the time.

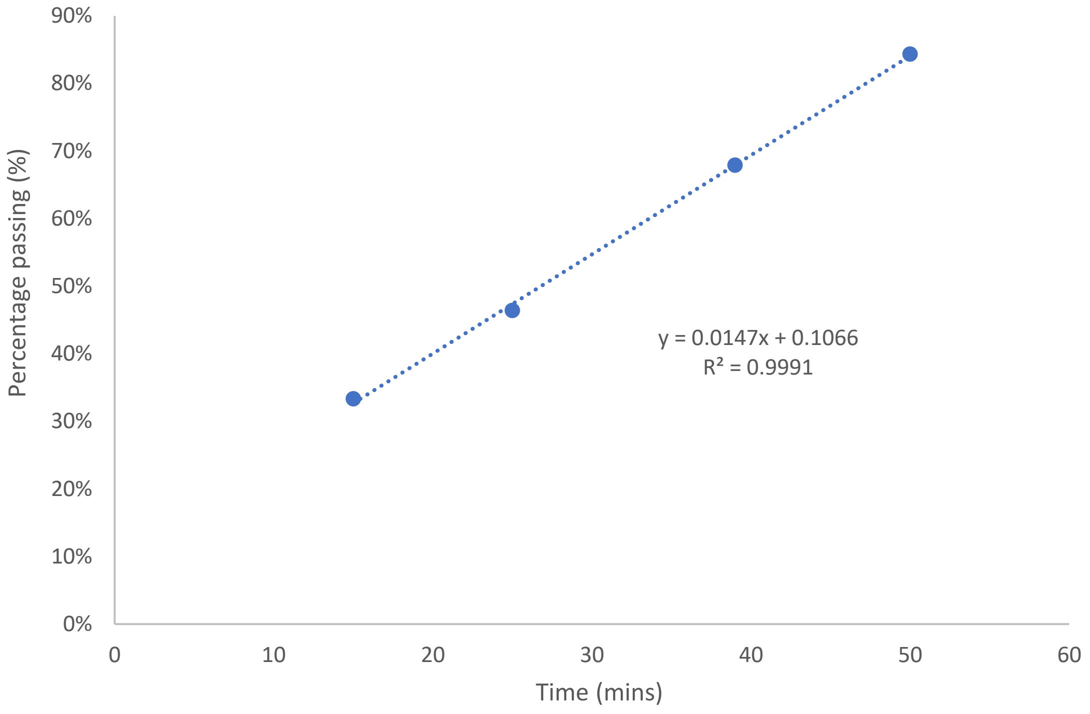

Figure 16 Milling curve for the selected Merensky ore.

4.4.2 Apparatus used in this Study

The key apparatus used was a 3 L Barker flotation cell equipped with an impeller and an airflow regulator which can be seen in Figure 17. Accompanying this float cell were 4 collection pans each with its respective wash bottle, to collect four concentrates, and two top-up water bottles all of which consisted of synthetic water.

Figure 17 Barker Flotation cell used during this study.

4.4.3 Milling and Procedure

The milling and flotation process began by weighing the ore sample for mass balance. The sample was then placed in the mill, where the necessary residual reagents, collector, flocculant, and coagulant were added according to the dosages listed in Table 7. To maintain a slurry concentration of 66%, 500 mL of synthetic plant water was added to the mill. The reagents were measured by mass rather than volume, due to their viscous nature. To ensure accurate delivery, the measuring cylinder was washed three times with synthetic water to ensure all the reagent was delivered.

During the milling, the collection trays, wash bottles, top-up water bottles, and filter papers were weighed. At the end of the milling process, the slurry was removed from the mill and transferred to the float cell, rinsed carefully with a limited amount of additional synthetic water to prevent loss of solids. The flotation process was then conducted using the standard UCT batch flotation procedure as outlined in Table 8 (Wiese 2005).

Table 8 UCT batch flotation procedure adapted.

Time (minutes)	Step
Prestart	Impeller speed is at 1200 rpm
00:00	Start the stopwatch and dose the required amount of CMC using a 10 mL syringe
01:00	Take 2 feed samples using a 10 mL syringes for X-ray Fluorescence (XRF - Olympus Vanta from InnoX Africa) analysis of solids and residual reagent analysis of water
02:00	Dose Frother 40 μl using a micro pipette
03:00	Open feed gas, ensure air flowrate is maintained at 7 L/min
	From this point, the scraping/ collecting process is carried out every 15 seconds
05:00	The first sample tray is removed (2 min flotation)
09:00	The second sample tray is removed (4 min flotation)
15:00	The third sample tray is removed (6 min flotation)
23:00	The fourth sample tray is removed (8 min flotation). The feed gas is turned off

Notes or pointers to aid in reproducibility:

- While cleaning with the wash bottle after scraping ensure that water is not accidentally put back into the cell as top up water (this disrupts water balance). The scraper should be cleaned in the tray rather than the edge of the float cell.
- The top-up water needs to be checked after each scrape and wash. While adding the top-up water, particularly the first tray, ensure that the addition rate is consistent across floats.
- An additional consideration must be made in marking the quantity of top water used for each tray.

After completing the flotation process, three tailings' samples were taken using 10 mL syringes for XRF analysis of the solids (using the Olympus Vanta from InnoX Africa) and residual reagent analysis of the water, as well as a settling test. The process then continued by weighing all the trays and water bottles used in the flotation. The final step involved filtering the concentrate samples through a Buchner flask. The solid concentrate was collected on a filter paper and stored in an oven at 80°C overnight to dry, while the liquid concentrate was stored in sample jars at room temperature for further testing.

4.5 Residual Reagent Test

Four water samples in total were tested for residual reagents, two were feed samples and two were tailings samples. The samples were taken from the float cell using syringes, as noted in Section 4.5.2, these were filtered using a Buchner flask. The water collected in the flask was then transferred into sample jars.

4.5.1 Collector Reagent Residual Test

The collector used was SIBX, it has an absorbance peak at 301 nm determined using UV/Vis spectrophotometer. The samples were prepared by passing them through a 0.45 µm sieve micro filter and then placing them in quartz cuvettes for the spectrometer. The samples were compared to distilled water and synthetic water as references.

The spectrometer provides output in the form of absorbance, the absorbance is based on the beer lambert law which is represented by the Equation 1 (The Beer-Lambert Law - Chemistry LibreTexts n.d.)

$$A = \epsilon l c \tag{1}$$

The absorbance can be transformed into concentrations with the development of a standard calibration graph, that conforms with previous projects (adopted from Manenzhe (2018)). The linear equation shown in Equation 2 was obtained.

$$Y = 16360X \; with \;\; an \; R^2 \; value \; of \; 0.988 \tag{2}$$

Where the Y term represents the absorbance while the slope represents the constants ϵl, leaving the X term to represent concentration.

4.5.2 Depressant Reagent Residual Test

The depressant, unlike the collector, is not readily visible on the spectrometer but due to its chemical composition, it can undergo the colorimetric method to give a visible peak at 480 nm.

The method below was adapted from Dubois et al., (1956). The water sample analysis process begins by filtering 2 mL of the sample through a micro filter into a test tube. Next, 1 mL of 5% phenol (m/m) is added to the test tube using a micro pipette. Then, 5 mL of 95.5% sulfuric acid is added rapidly, with the stream of acid directed towards the liquid surface. The tubes are left to stand for ten minutes, then shaken and left to stand for an additional 10 to 20 minutes. Finally, the solutions are transferred to cuvettes and run through a spectrometer for analysis.

The spectrometer provides absorbances at 480 nm that needs to be converted to concentrations using a calibration curve shown in Appendix 10.1.

The equation for the depressant calibration curve is:

.

$$\mathrm{Y} = 0.0053\mathrm{X} \text{ with an } R^2 \text{ value } 0.97 \tag{3}$$

Where the Y term represents the absorbance while the slope represents the constants ϵl, leaving the X term to represent concentration.

4.6 Settling Rate Measurements

The particle settling rate measurements were done by taking a 10 mL slurry sample using a syringe from the float cell at the end of the flotation process, as noted in Section 4.5.2. The sample was transferred to a sample vial and shaken for 30 seconds before being placed in a Hannah turbidity meter. A stopwatch was immediately started, and turbidity readings were taken at 90 second intervals for one hour. This method was developed and adapted from studies considering the effects of flocculants and coagulants on particle settling (Arjmand et al. 2019; Ma et al. 2019).

The methodology specified above resulted in graphs for each run shown in Figure 18. It can be seen in the figure that the graph followed a logarithmic decay which can be confirmed by the significant R^2 values for the lines. The slope of these lines can be interpreted as the settling rate as it signifies the change in turbidity readings over the unit of time giving the relationship below.

$$Settling\ rate = \frac{Turbidity\ (NTU)}{Time} \tag{4}$$

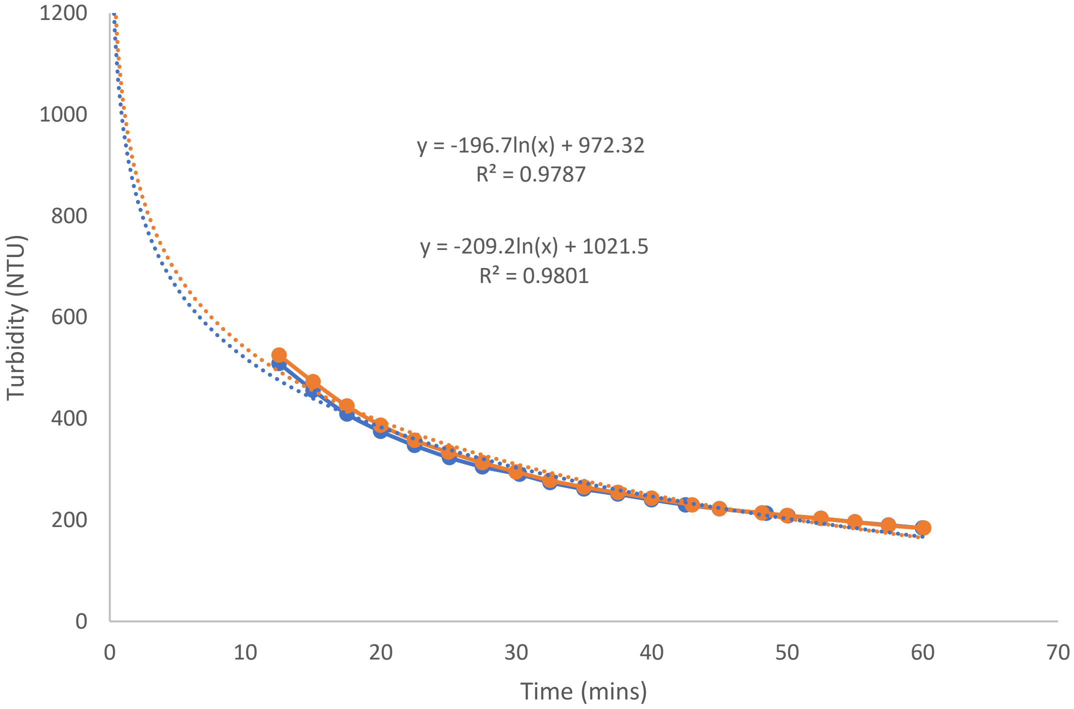

Figure 18 Settling rate graph for run 2 with duplicate.

5 Results

This Chapter is divided into two sections, the first section displays results that were obtained from experimental data and then underwent statistical analysis. In this section the responses to the varied factors have been mapped out on graphs with the error bars representing the standard deviation of each experiment. The second section displays the statistical analysis results that were run in the software Design-Expert® to develop models and predict mid-point tests.

Refer to Table 4 under Section 4.1 to interpret test conditions for the legend provided in the graphs.

5.1 Water and Solid Recovery

5.1.1 Water Recovery vs Time

Figure 19 shows cumulative water recoveries and time for different reagent conditions.

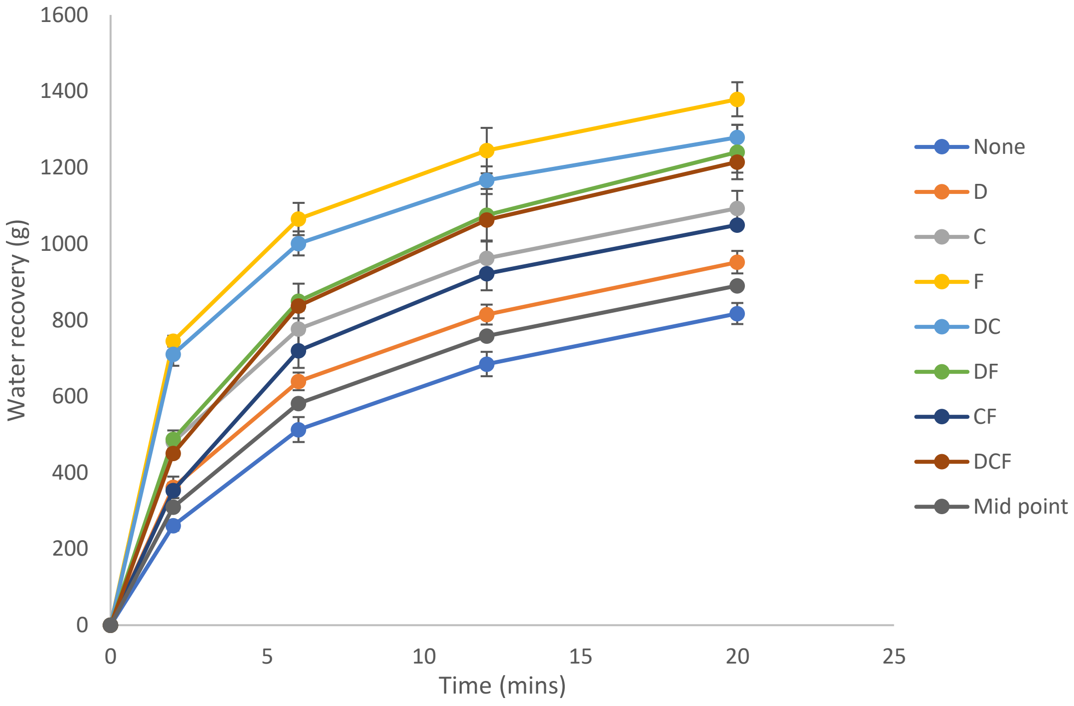

Figure 19 Water recovery vs time.

Figure 19 shows that the rate of recovery is highest in the first 8 min of the process after which the rate slows down significantly. The addition of any of the variable reagents or combination thereof increases water recovery. A higher flocculant dosage (F), in the absence of a depressant and coagulant, has the highest impact on water recovery while the absence of all three polymers (None) results in the lowest water recovery.

5.1.2 Solids Recovery vs Time

Figure 20 shows the relation between cumulative solids recovery and time.

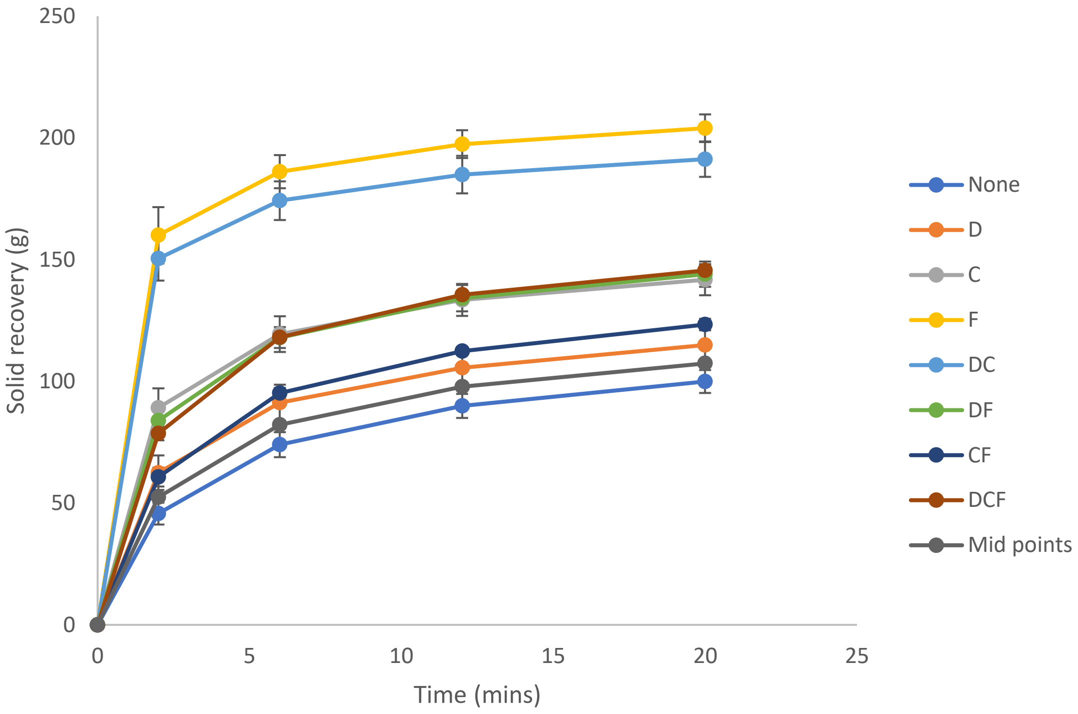

Figure 20 Solids recovery vs time.

Figure 20 shows that the majority of the solids is recovered in the first 2 min of the process. The highest solids recoveries were achieved under the conditions of high flocculant (F) and high depressant and high coagulant (DC) in the system. While the lowest recoveries were obtained in the absence of all three polymers (None).

5.1.3 Solids Recovery vs Water Recovery

Figure 21 shows the relationship between cumulative solids recovery and cumulative water recovery for the different reagent conditions.

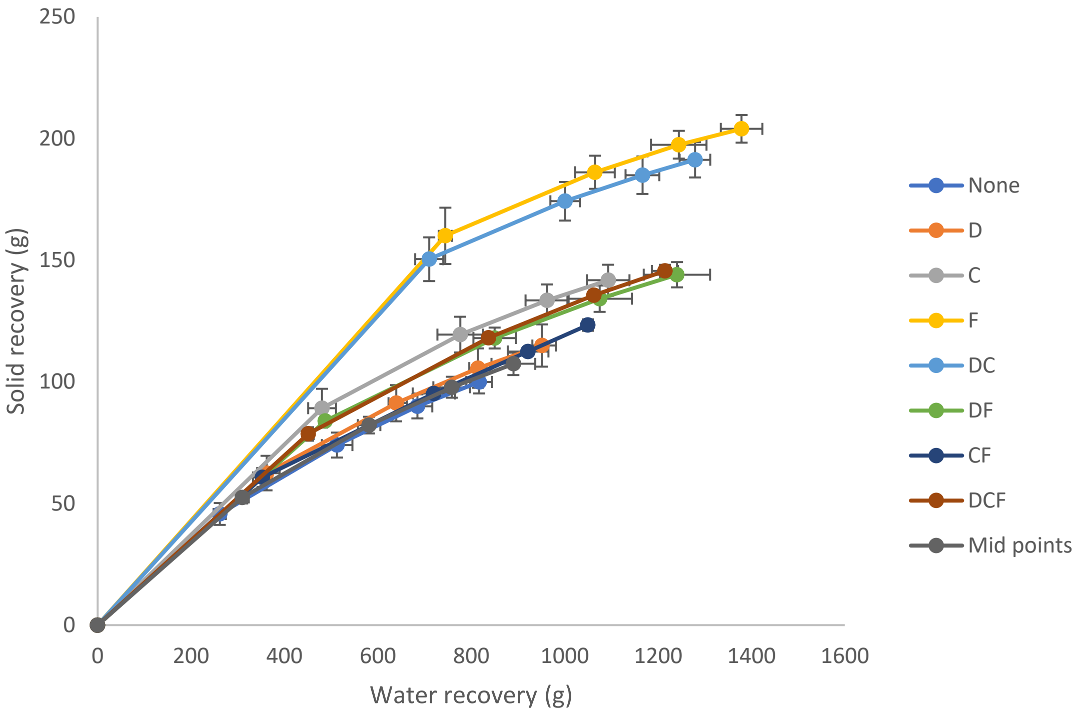

Figure 21 Final solid vs Final water recovery.

As seen in Figures 19 and 20, the high flocculant (F) condition has the greatest effect on solids per unit water recovery while the absence of all three polymers (None) has the lowest solids per unit water recovery. The impact of reagent dosage can also be seen when comparing the high depressant, high coagulant and high flocculant (DCF) and mid-point conditions, the water and solids recoveries both increased with increasing dosage.

5.1.4 Final Solids and Water Recoveries

Figure 22 shows the final solids and water recoveries for the different reagent conditions.

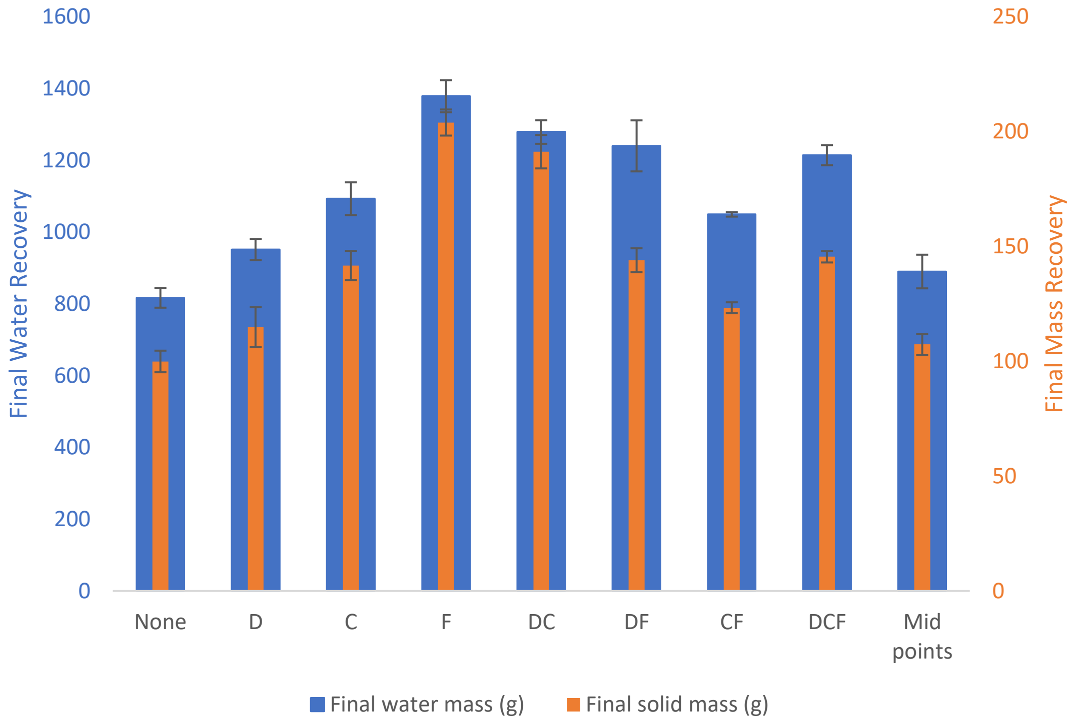

Figure 22 Final mass and water recovery.

Figure 22 shows that the addition of any of the reagents resulted in an increase in water and solids recovered. A high flocculant (F) had the highest recovery of all reagents, however when a flocculant (F) was paired with any other reagent both the solids and water recoveries decreased despite the other reagents individually having an increasing effect on water and solid recoveries.

A high coagulant (C) and high depressant (D), when added individually, resulted in an increase in water and solids recoveries. When interacting with other reagents they showed increased water and solid recoveries over and above that of their individual recoveries.

Combining all three variables, D, C, and F, led to increased water and solids recoveries. Surprisingly, this combination did not yield the highest recoveries, despite each individual reagent resulting in improved water and solids recovery when used alone. Furthermore, the impact of dosage is evident when comparing the mid-point dosage with DCF. There exists a proportional relationship between dosage and recoveries, indicating that higher dosages correspond to increased recoveries.

5.2 Copper Recovery and Grade

5.2.1 Copper Recovery vs Time

Figure 23 shows the relationship between copper recovery and time for the different reagent conditions.

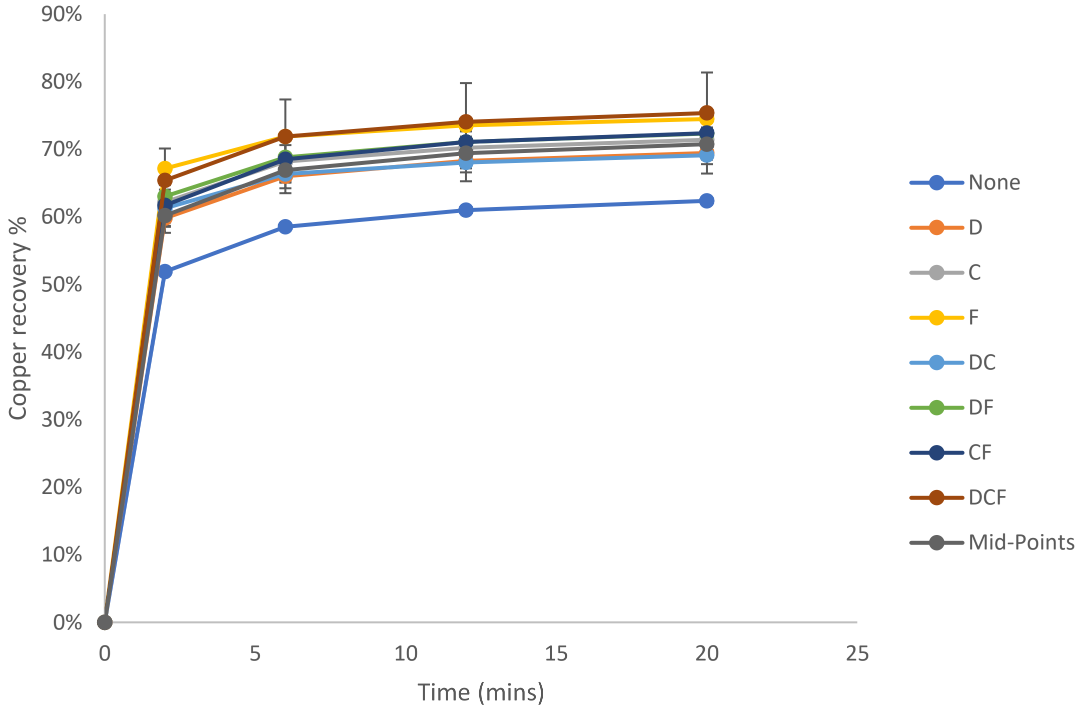

Figure 23 Copper recovery vs time.

Figure 23 shows that most of the copper was recovered in the first 2 min of the process. The addition of reagents had a notable impact on copper recovery, leading to higher copper recoveries in the process.

5.2.2 Copper Recovery vs Water Recovery

Figure 24 shows the cumulative copper recovery versus the cumulative water recovery.

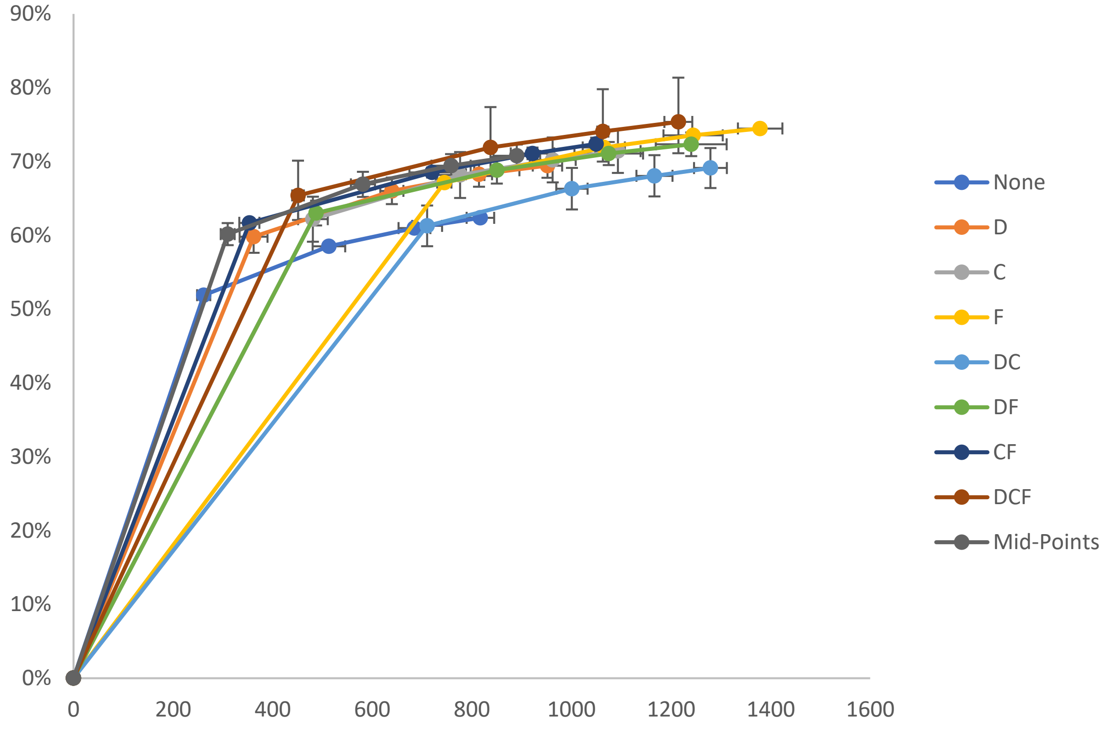

Figure 24 Copper recovery vs water recovery.

From Figure 24, it can be seen that as water recovery increases so does the copper recovery. The addition of reagents to the system can be seen to have an increasing impact on the recoveries of solids per unit water recovery, this can be seen when comparing None to the other conditions. The impact of reagent dosage can also be seen when comparing DCF and mid-point conditions with both water and solid recoveries increasing with increasing dosage.

5.2.3 Copper Grade vs Copper Recovery

Figure 25 shows the cumulative copper grade vs cumulative copper recovery for the different reagent conditions.

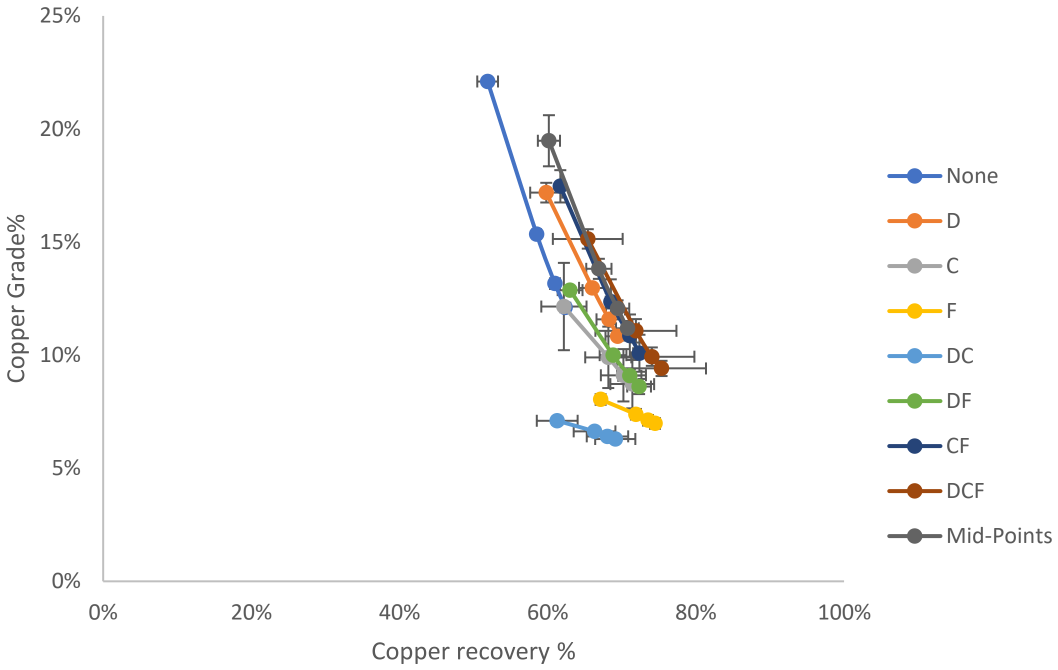

Figure 25 Copper grade vs Copper recovery.

Figure 25 presents a general trend that shows an inverse relationship between grade and recovery that applies to all reagent conditions. Taking a closer look at the results one can notice that despite having similar recoveries some of the interactions have different grades. For example, looking at conditions DCF and F, both these conditions have a similar recovery but achieve significantly different grades with F having a much lower grade than DCF. A particular outlier is DC which shows the lowest grades of all conditions, with recoveries comparable to C and DF.

5.2.4 Final Copper Recovery and Grade

Figure 26 shows the final grades and recoveries for the copper.

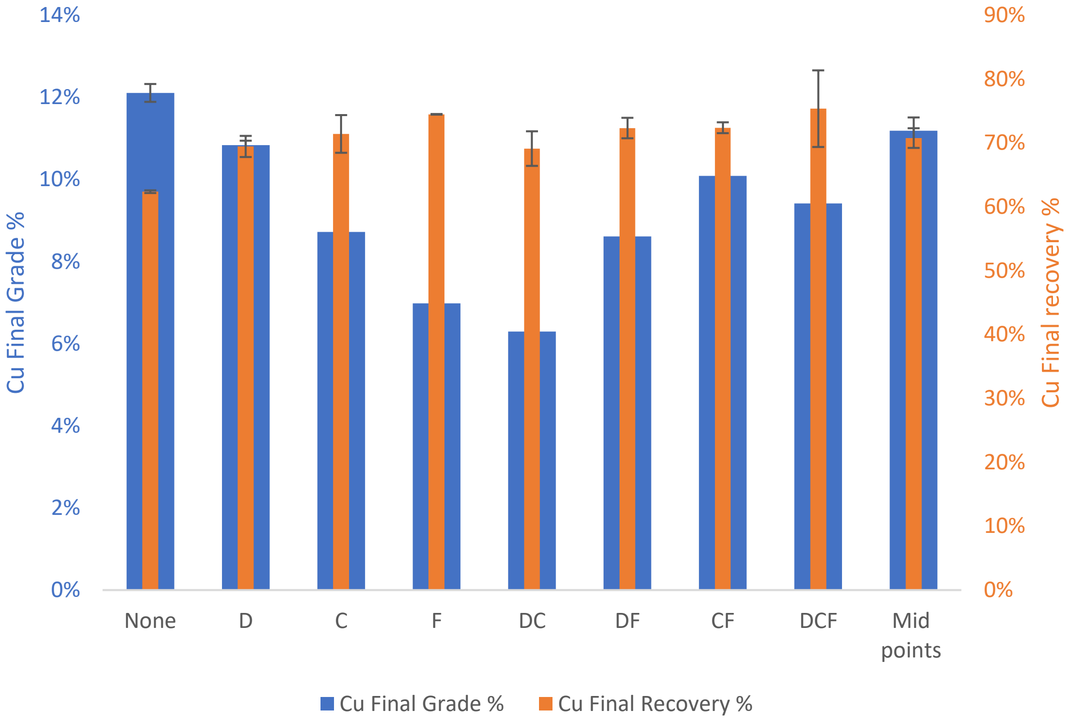

Figure 26 Final copper grades and recoveries.

It can be seen, from Figure 26, that adding reagents to the system resulted in a slight increase in the recoveries of the system. The recoveries were quite similar to one another, except for None where the recovery was the lowest.

The grade was impacted by the reagent conditions more than the recovery. DC and F had the most negative impact on the grade of the system but achieved the highest solid recoveries. The highest grade was achieved by the None condition.

5.3 Nickel Recovery and Grade

5.3.1 Nickel Recovery vs Time

Figure 27 shows the cumulative nickel recovery versus time for all the tested conditions.

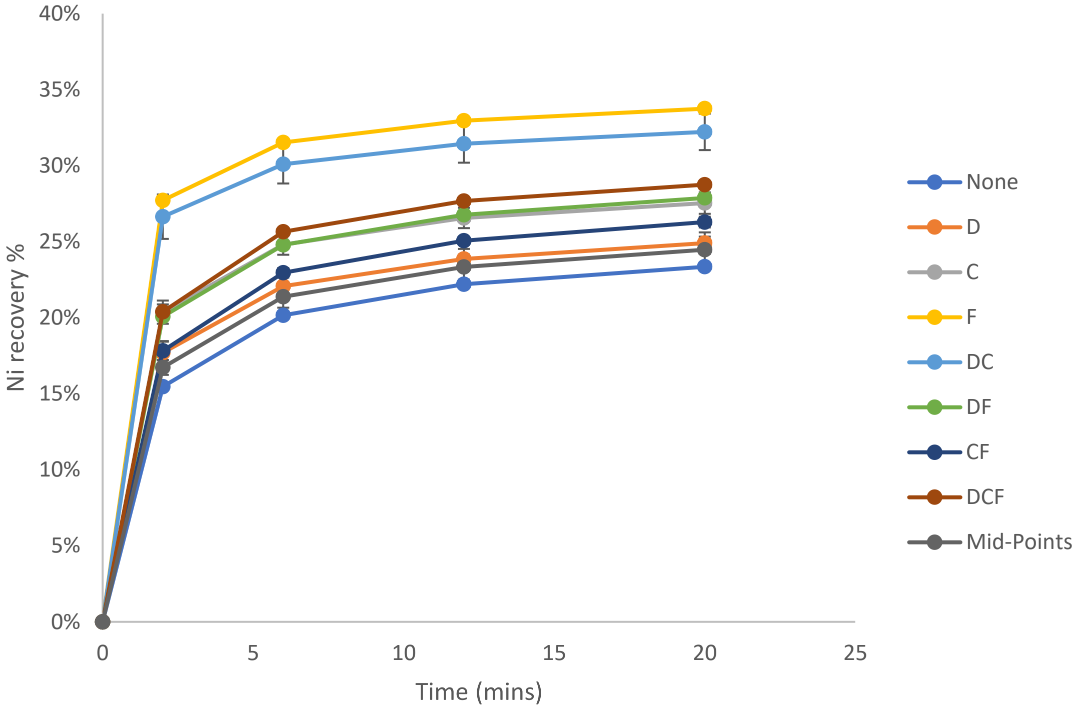

Figure 27 Nickel recovery vs time.

Figure 27 shows that most of the recovery occurs in the first 2 min. The F and DC conditions had the highest recovery values while having no reagents in the system resulted in the lowest recovery values. Generally, it could be seen that the addition of reagents resulted in higher nickel recoveries.

5.3.2 Nickel Recovery vs Water Recovery

Figure 28 shows the cumulative nickel recovery versus the cumulative water recovery for all tested conditions.

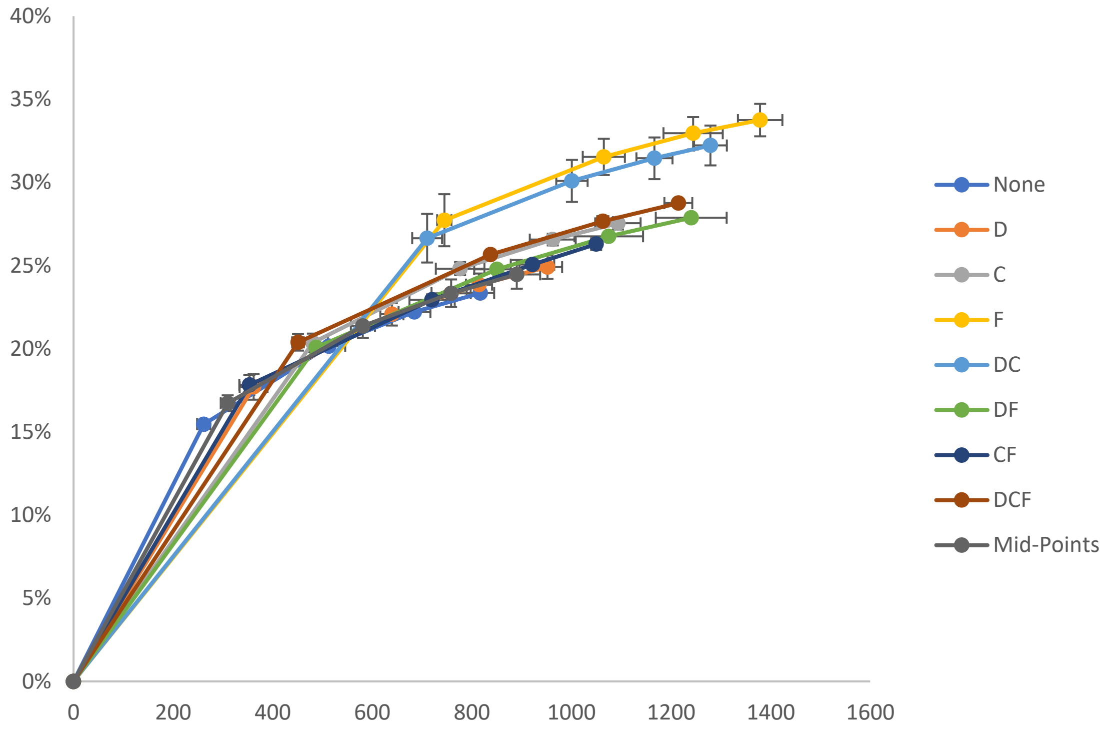

Figure 28 Nickel recovery vs water recovery.

Figure 28 shows that the condition None resulted in the lowest recovery per unit water while adding reagents in the system resulted in a higher nickel recoveries per unit water, with conditions F and DC resulting in the highest recoveries of nickel and water.

5.3.3 Nickel Grade vs Recovery

Figure 29 shows the cumulative nickel grade versus the cumulative nickel recovery for all tested conditions.

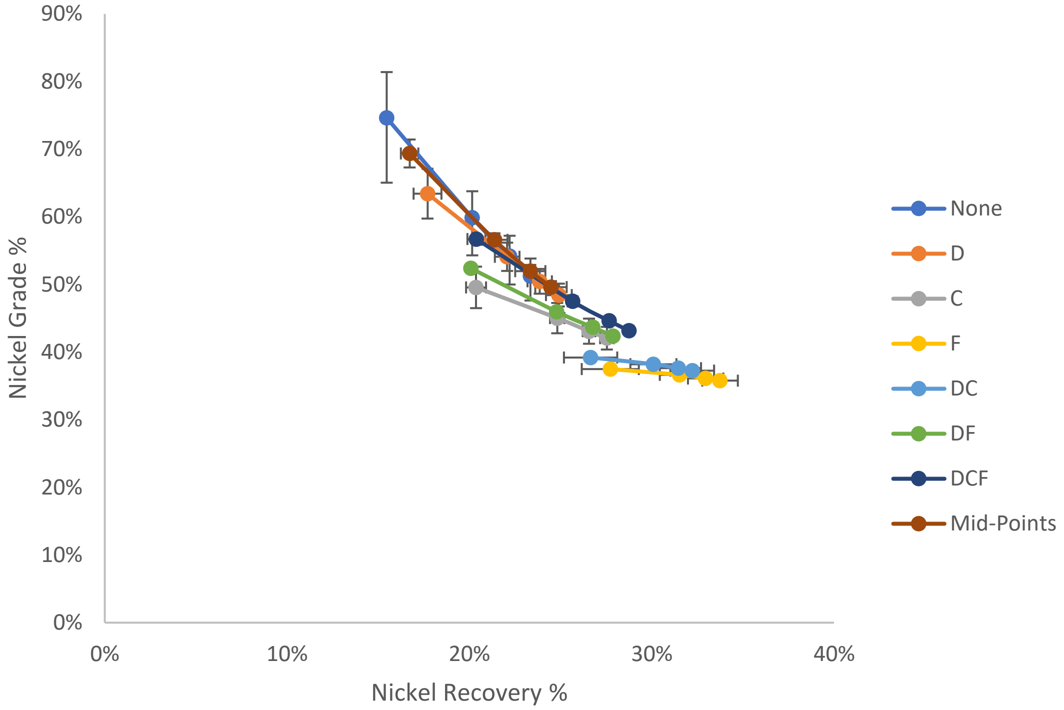

Figure 29 Cumulative Nickel Grade vs Cumulative Nickel Recovery.

From Figure 29, it can be observed that the grade and recovery of nickel have an inverse relationship. The reagent conditions DC and F reported the highest nickel recoveries and can be seen to have the lowest nickel grade while None is seen to have the best nickel grade but the worst nickel recovery.

5.3.4 Final Nickel Grade and Recovery

Figure 30 shows the final nickel grade and recoveries for the different test conditions.

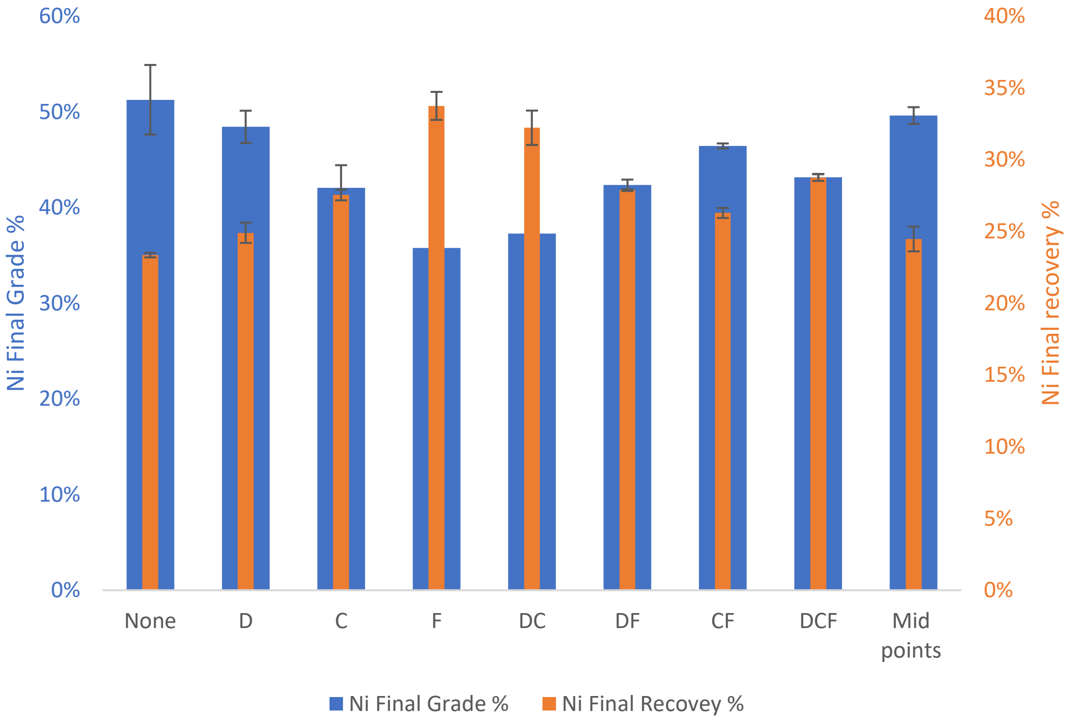

Figure 30 Final nickel recovery and grade.

Figure 30 yet agian shows an inverse relationship between the nickel grade and recovery. The condition with the highest nickel grade is None which also has the lowest nickel recovery. The condition with the lowest nickel grade is F which also has the highest nickel recovery.

5.4 Settling of Tailings

5.4.1 Turbidity vs Time

Figure 31 is of turbidity versus time for all tested conditions.

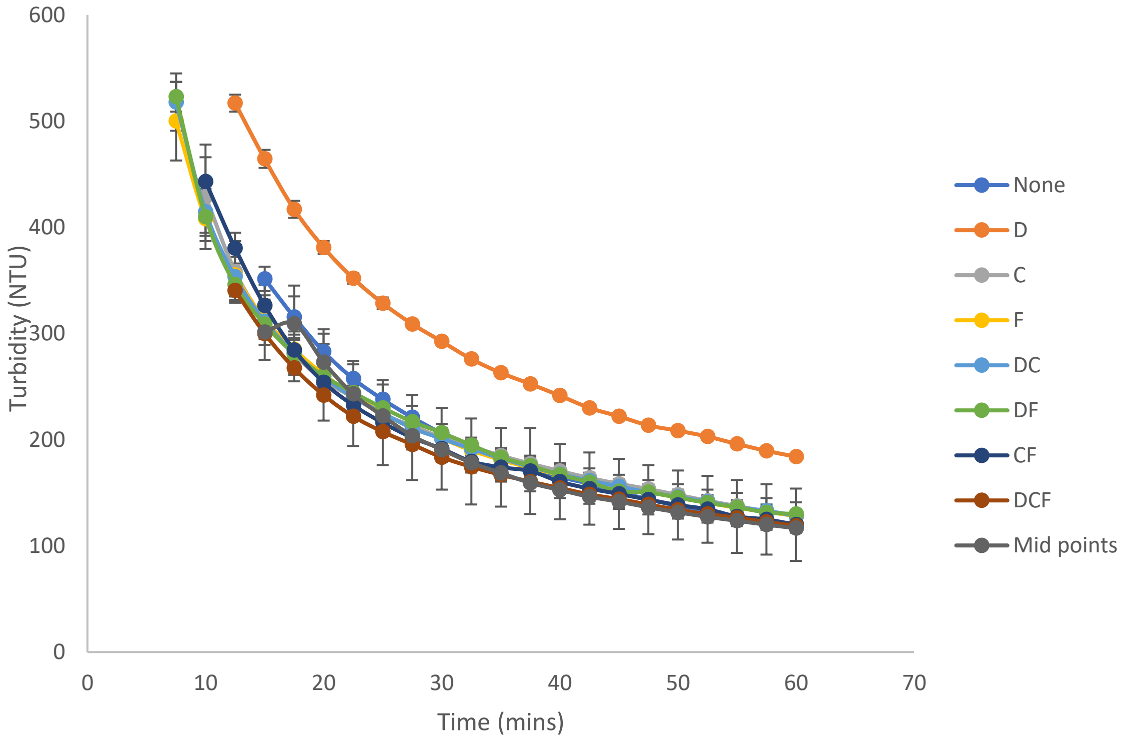

Figure 31 Turbidity vs time.

Figure 31 shows that there is one condition clearly distinct from the others and that is D. The system with depressant takes longer to settle than the other conditions that all settle in a similar pattern.

5.4.2 Settling Rate

Figure 32 shows a comparison of the settling rates for the different conditions tested in this study.

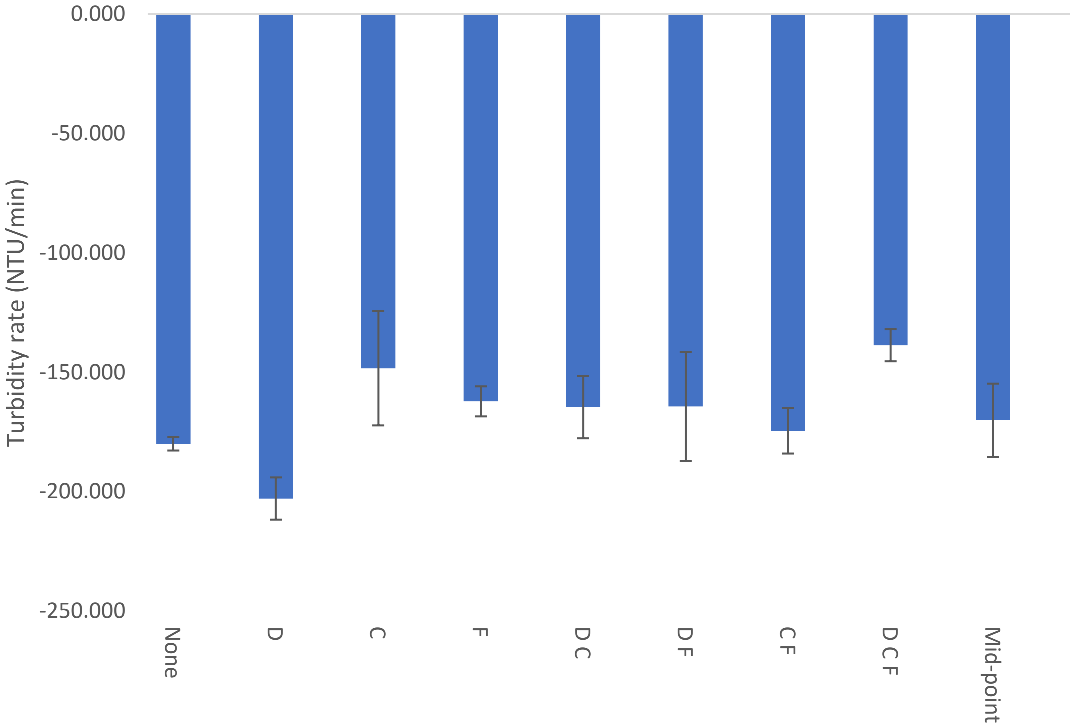

Figure 32 Settling rate for the different conditions.

From Figure 32, it can be seen that the settling rates for all conditions are similar, except for D. Most of the tested conditions have a settling rate range of between -138 and -180 Ntu/min. However, D settles the slowest with -200 Ntu/min.

5.5 Residual Reagent Concentrations

For this section the feed concentration readings are samples that were taken 1 min after the depressant was added to the flotation process and the tailings samples are taken at the end of the process while the tailings were still in the cell. This has been highlighted under Section 4.5.2.

5.5.1 SIBX Reagent Feed and Tails Residual Concentrations

Figure 33 shows the concentration of the collector SIBX pre and post flotation.

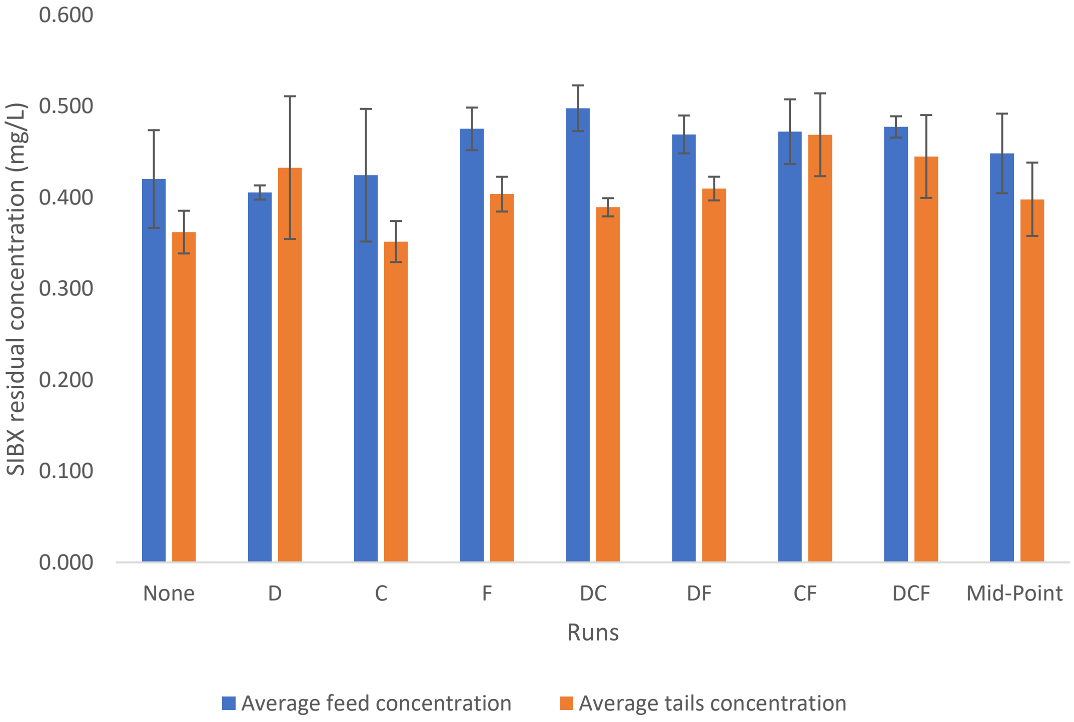

Figure 33 SIBX residual feed and tails concentration.

It can be seen from Figure 33 that there is no impact on the concentration readings under the various different conditions as they all produce a similar level of concentration. The initial concertation of the collector at dosage was 143 mg/kg ore. In Figure 33 it can clearly be seen that most of the collector is being consumed under all conditions.

5.5.2 CMC Reagent Feed and Tails Concentration

Figure 34 depicts the concentration of CMC pre flotation and post flotation for all the conditions tested.

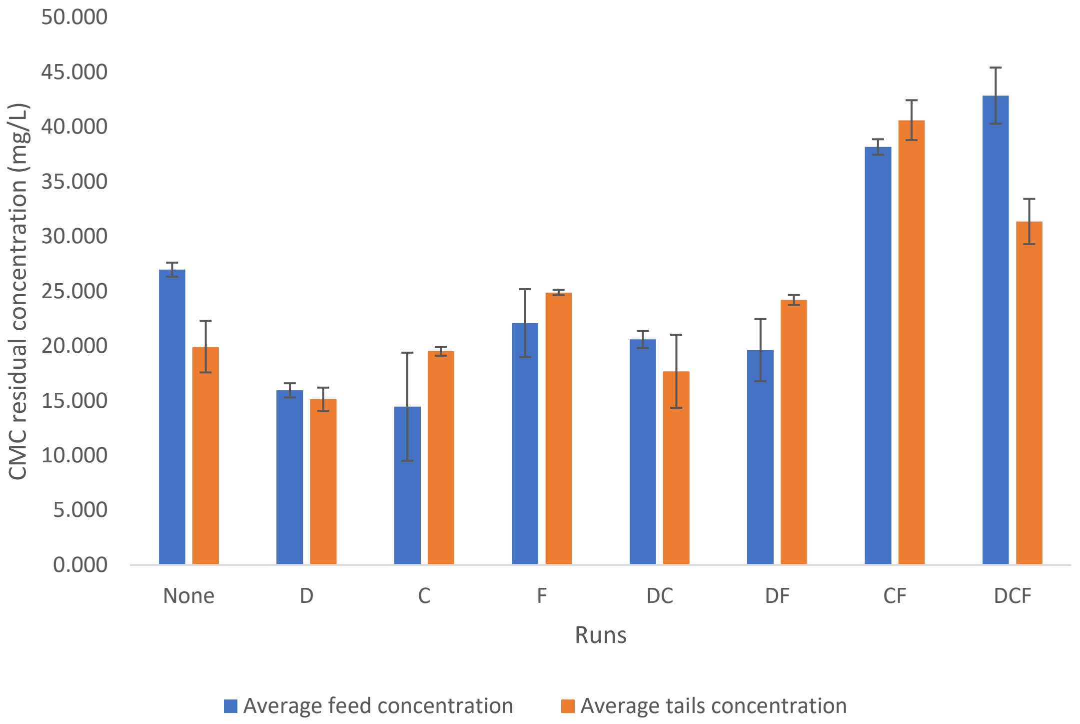

Figure 34 CMC residual feed and tail concentration

The initial dosage of CMC added to the system is 60 mg/kg ore. Figure 34 shows that under the None condition when no CMC was added to the system, a reading was still obtained indicating a substance similar to CMC is present, therefore providing a higher reading. As CMC is added and under different conditions, it can be noted that the CMC concentration does not increase or decrease significantly, indicating no impact on CMC adsorption upon the addition of dewatering reagents.

5.6 Statistical Modelling for Interactive Effects

A statistical analysis was used to better understand how the three conditions interact and impact the flotation process. The software used is Design-Expert®.

The software runs an analysis of variance (ANOVA) that models the conditions in a linear equation and provides a p-value that indicates how likely it is to obtain an extreme result given the null hypothesis is true. For a condition to be truly significant a 95% confidence interval is considered.

All response results have been tabulated to present the significant condition p-values for the particular response. Non-effective conditions are removed from the model equation.

A Pareto chart shows the absolute values of the standardized effects from the largest to the smallest. Conditions below the significance lines were omitted from the development of the model equation.

When having only a single significant condition a 2D graph is sufficient to portray its effect, when having two significant conditions a 3D graph is more appropriate to show the interactions. For responses with three significant conditions, 3 graphs are used to show the effects of all three variables.

5.6.1 Water Recovery

The following conditions and interactions were significant for water recovery and are highlighted in Table 9.

Table 9 Table indicating conditions and interactions that impact the water recovery of the flotation process statistically.

Conditions	Symbol	P-test
Depressant	A	0.0172
Coagulant	B	0.023
Flocculant	C	< 0.0001
Depressant and Coagulant	AB	0.0041
Depressant and Flocculant	AC	0.0109
Coagulant and Flocculant	BC	< 0.0001
Depressant, Coagulant, and Flocculant	ABC	0.0216
Model		< 0.0001

The model equation developed by the software is:

$$\begin{aligned} &Final\ water\ recovery \\ &\quad = 817.18 + 2.24(A) + 18.41(B) + 140.46(C) + 0.057(AB) - 1.31(AC) \\ &\quad - 10.09(BC) + 0.081(ABC) \end{aligned}$$

It can be seen from Table 9 and Figure 35 that all the conditions are significant toward the response seen in the Final water recovery. The conditions with the most significant effect are the flocculant and the conditions BC and AB.

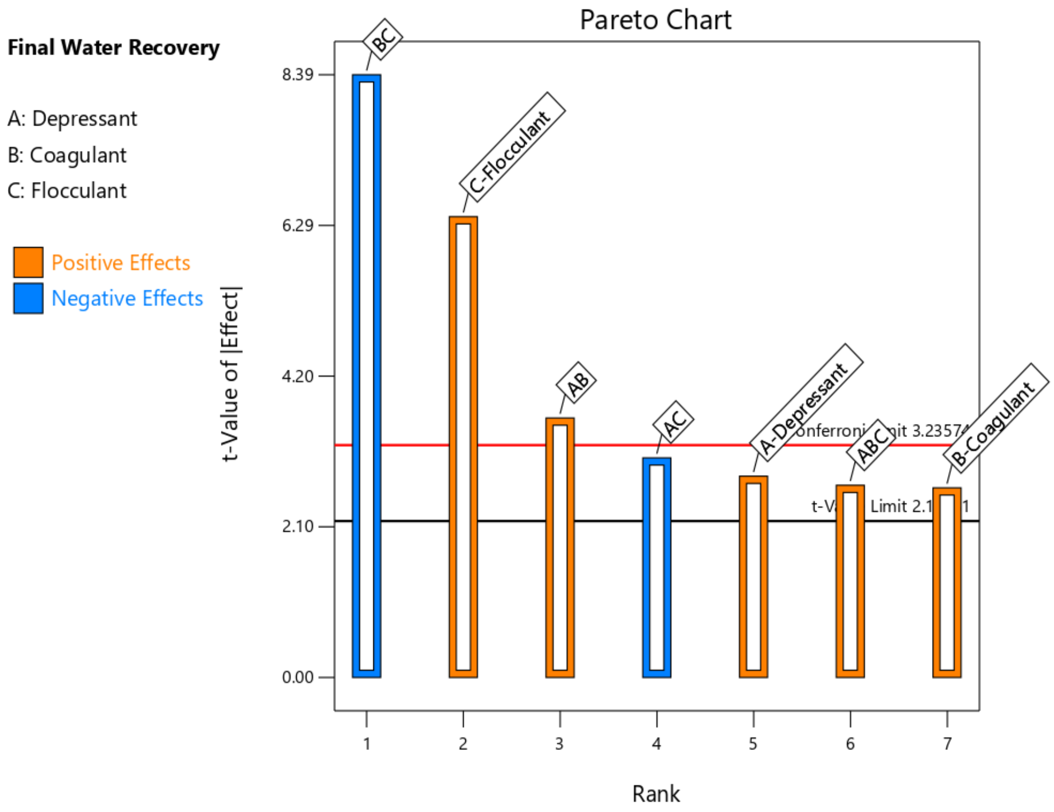

Figure 35 Pareto displaying conditions affect.

The effect of the flocculant is visible when focusing on the point where the depressant and coagulant dosages are zero. As the flocculant quantity is increased from Figure 36 to Figure 38 the water recovery also increases.

The interactive effect between the depressant and coagulant can be seen in Figure 36 where there is no depressant in the system. The contour lines on the X-Y plane depict increasing water recovery slopes with increasing depressant and coagulant dosage.

The effect of the coagulant and flocculant can be seen by comparing Figure 36 and Figure 38. When focusing on the line where the depressant dosage is zero and the coagulant dosage is increasing, it can be seen that in Figure 36 the water recovery increases as the coagulant dosage increases. However, in Figure 38 when the flocculant is introduced into the system the water recovery is not impacted by the increasing coagulant dosage.

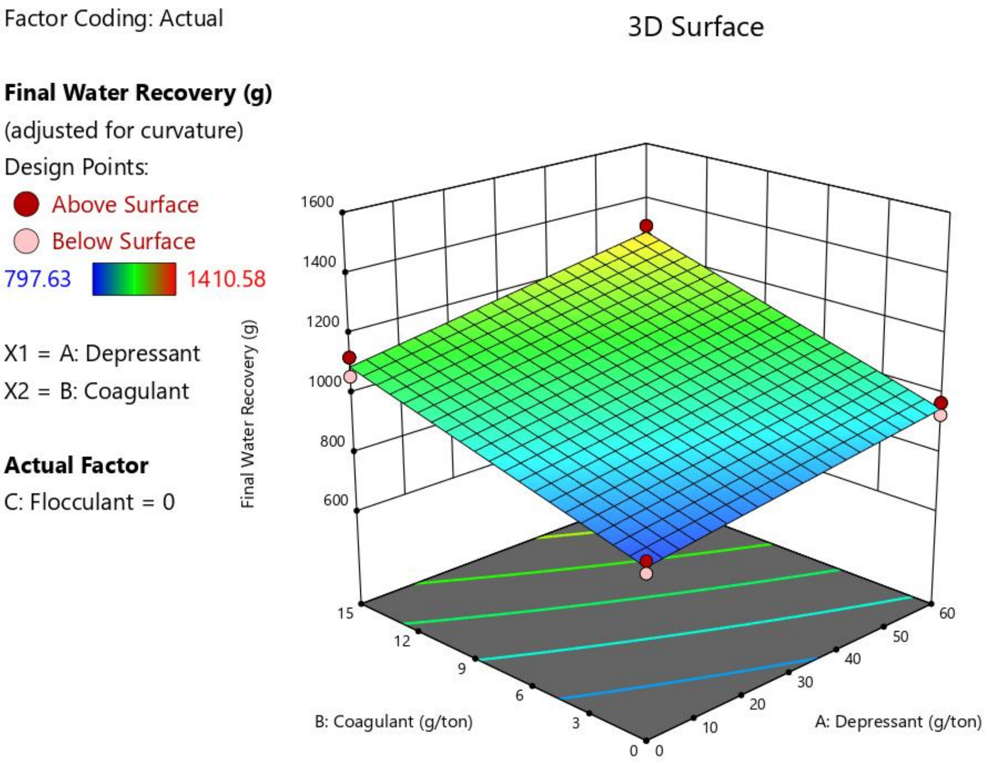

Figure 36 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final water recovery (g) at 0 g/ton Flocculant dosage.

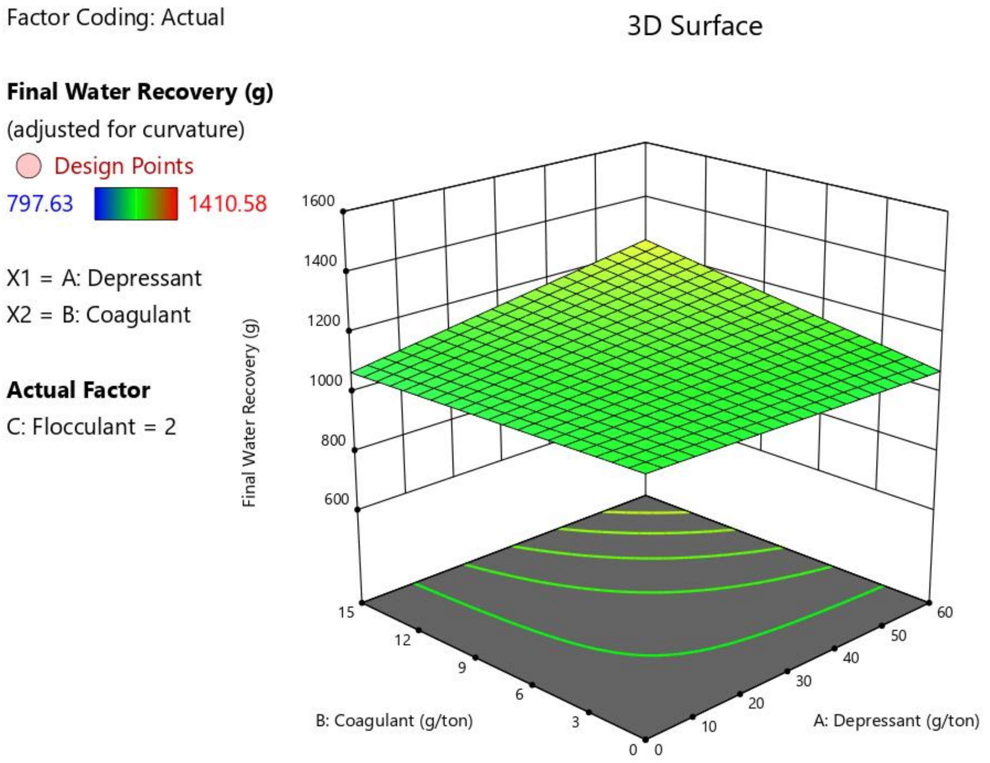

Figure 37 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final water recovery (g) at 2 g/ton Flocculant dosage.

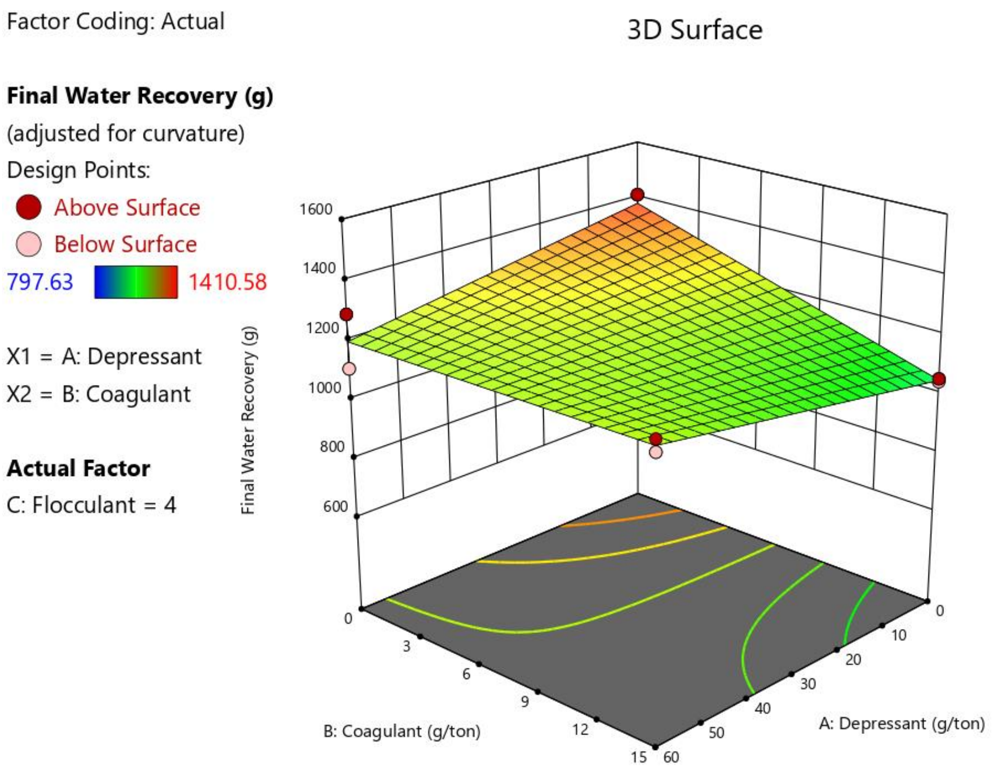

Figure 38 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final water recovery (g) at 4 g/ton Flocculant dosage.

5.6.2 Solids Recovery

The following conditions and interactions were significant for recovery and are highlighted in Table 10.

Table 10 Table indicating conditions and interactions that impact the solid recovery of the flotation process statistically.

Conditions	Symbol	P-test
Depressant	A	0.0362
Coagulant	B	0.0052
Flocculant	C	< 0.0001
Depressant and Coagulant	AB	< 0.0001
Depressant and Flocculant	AC	< 0.0001
Coagulant and Flocculant	BC	< 0.0001
Depressant, Coagulant, and Flocculant	ABC	0.0014
Model		< 0.0001

The model equation developed by the software is:

$$\begin{aligned} Final\ solids\ recovery &= 99.95 + 0.25(A) + 2.79(B) + 26.01(C) + 0.038(AB) - 0.31(AC) \\ &- 2.04(BC) + 0.013(ABC) \end{aligned}$$

It can be seen from Table 10 and Figure 39 that all the conditions are significant towards the response seen in the Final solid recovery. The interactions between variables BC, AB and AC have the most significant impact on solid recovery.

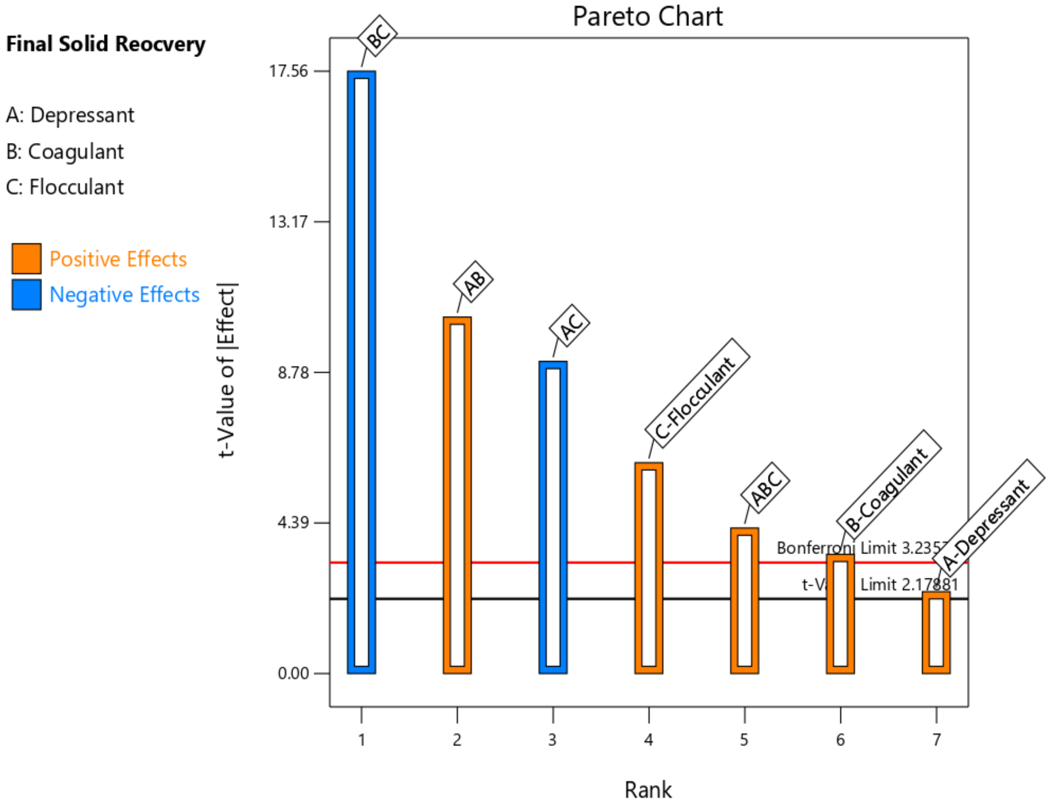

Figure 39 Pareto chart for conditions affect.

The AB condition impact can be seen in the X-Y plane of Figure 40, the contour lines on the graph indicate as the dosage of the reagents increases so does the solid recovery.

The BC condition impact can be seen when focusing on the point where the depressant dosage is zero, it can be seen that in Figure 40 the water recovery increases as the coagulant dosage increases however in Figure 42 when flocculant is added to the system the water recovery is not impacted by the increasing coagulant dosage.

A similar observation can be made for AC condition, with the focus shifting towards the zero-coagulant line and it can be observed that as the depressant dosage increases the water recovery decreases.

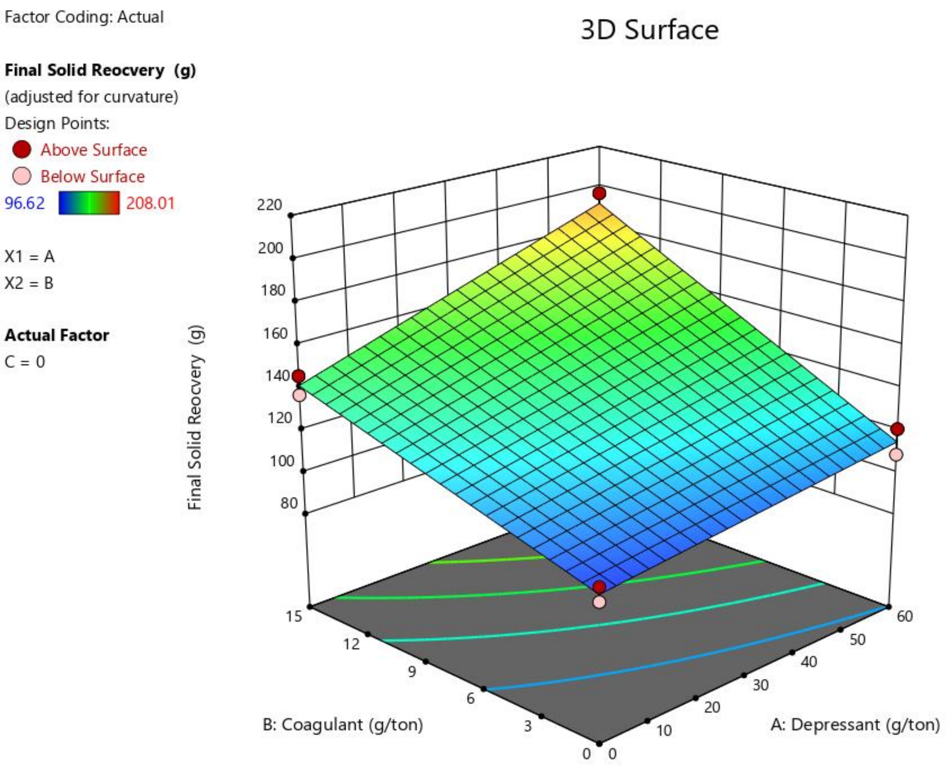

Figure 40 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final solid recovery (g) at 0 g/ton Flocculant dosage.

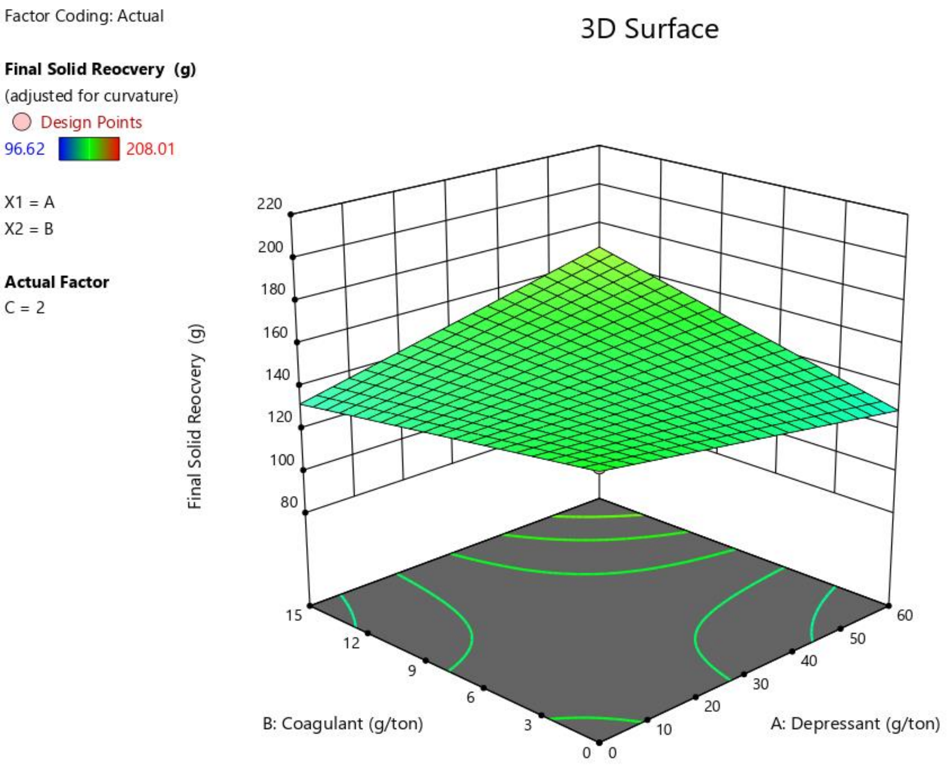

Figure 41 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final solid recovery (g) at 2 g/ton Flocculant dosage.

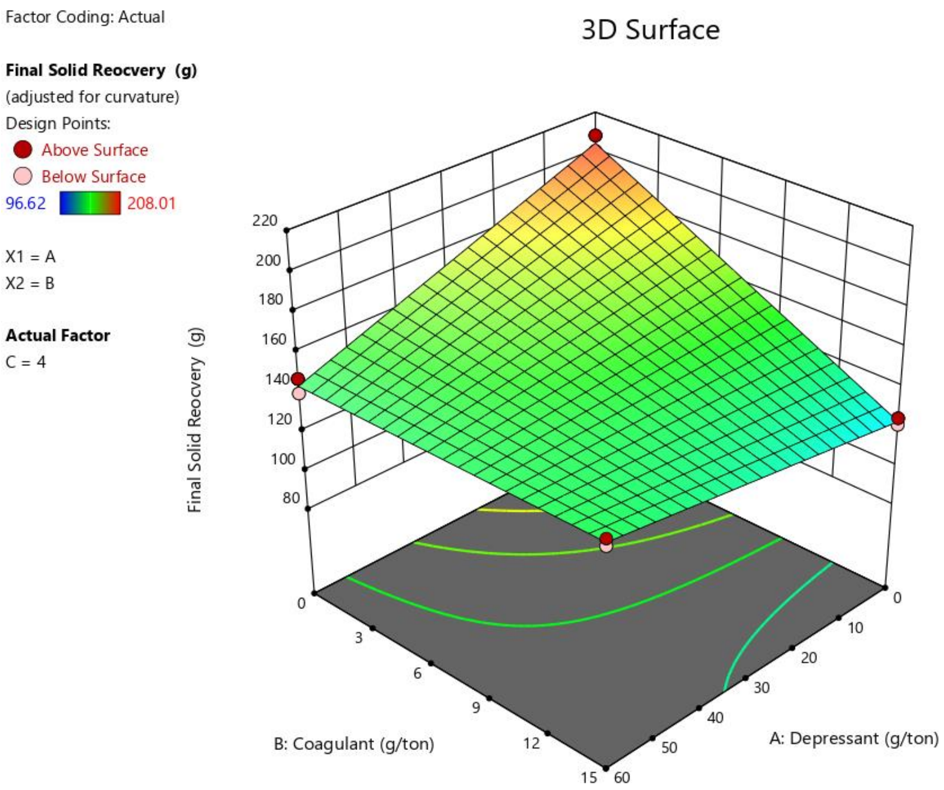

Figure 42 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final solid recovery (g) at 4 g/ton Flocculant dosage

5.6.3 Final Copper Grade

The following conditions and interactions were significant toward the final copper grade and are highlighted in Table 11.

Table 11 Table indicating conditions and interactions that impact the copper grade of the flotation process statistically.

Conditions	Symbol	P-test
Depressant	A	0.0129
Coagulant	B	0.0129
Flocculant	C	0.0591
Depressant and Coagulant	AB	0.0028
Depressant and Flocculant	AC	0.0028
Coagulant and Flocculant	BC	< 0.0001
Model		< 0.0001

The model equation developed by the software is:

$$\begin{aligned} Final\ copper\ grade \\ &= 11.94 - 0.015(A) - 0.19(B) - 1.22(C) - 0.003(AB) + 0.0094(AC) \\ &+ 1.024(BC) \end{aligned}$$

Table 11 and Figure 43 shows the conditions that impact copper grade, the most significant conditions were BC, AB and AC.

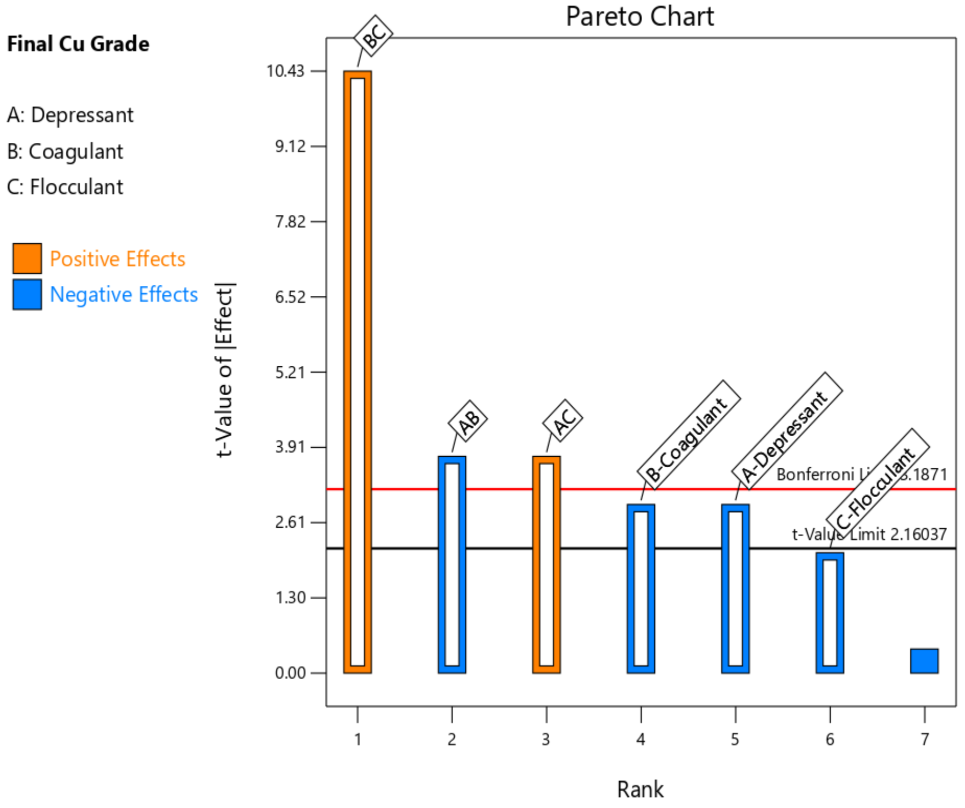

Figure 43 Pareto chart for conditions affect.

The impact of BC can be seen when focusing on the line where the depressant dosage is zero, comparing Figures 44 and 46. In Figure 44 as the coagulant dosage increases the grade decreases, however when flocculant is added to the system as seen in Figure 46 the opposite occurs with the grade which increases as the coagulant dosage increases.

The impact of the AB condition can be observed on the X-Y plane depicted in Figure 44. As the dosages increase, the contour lines indicate a decrease in grade.

The AC condition can be observed by focusing on the line where the coagulant dosage is zero. In Figure 44 when there is no flocculant in the system, it can be seen that as the depressant dosage increases the grade decreases, however in Figure 46 when flocculant is introduced in the system, as the depressant dosage increases so does the grade.

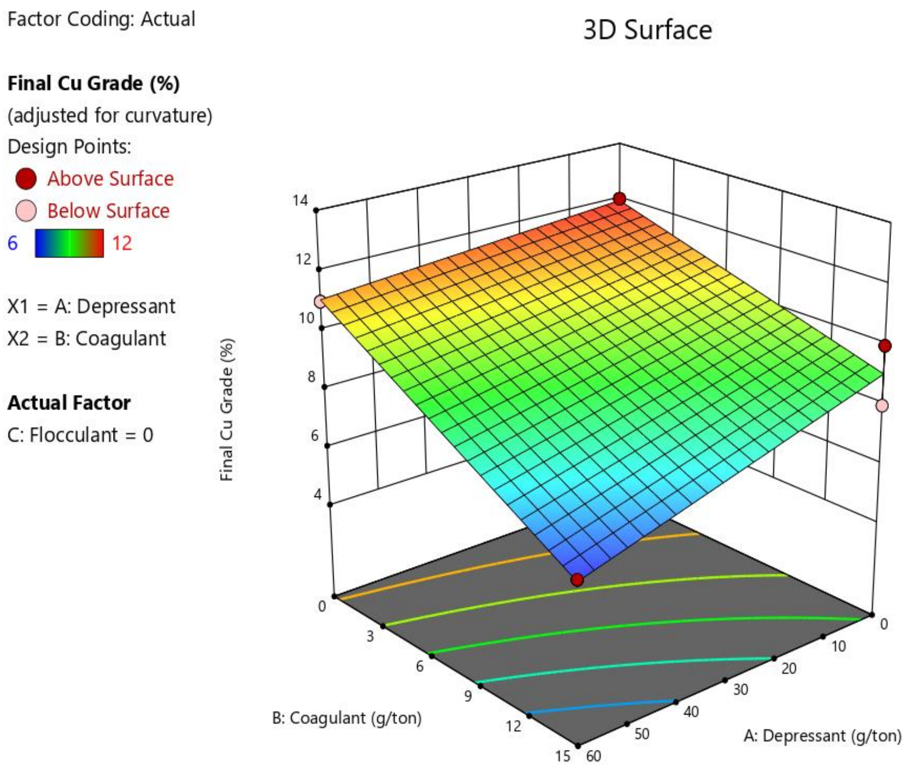

Figure 44 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Copper grade (%) at 0 g/ton Flocculant dosage

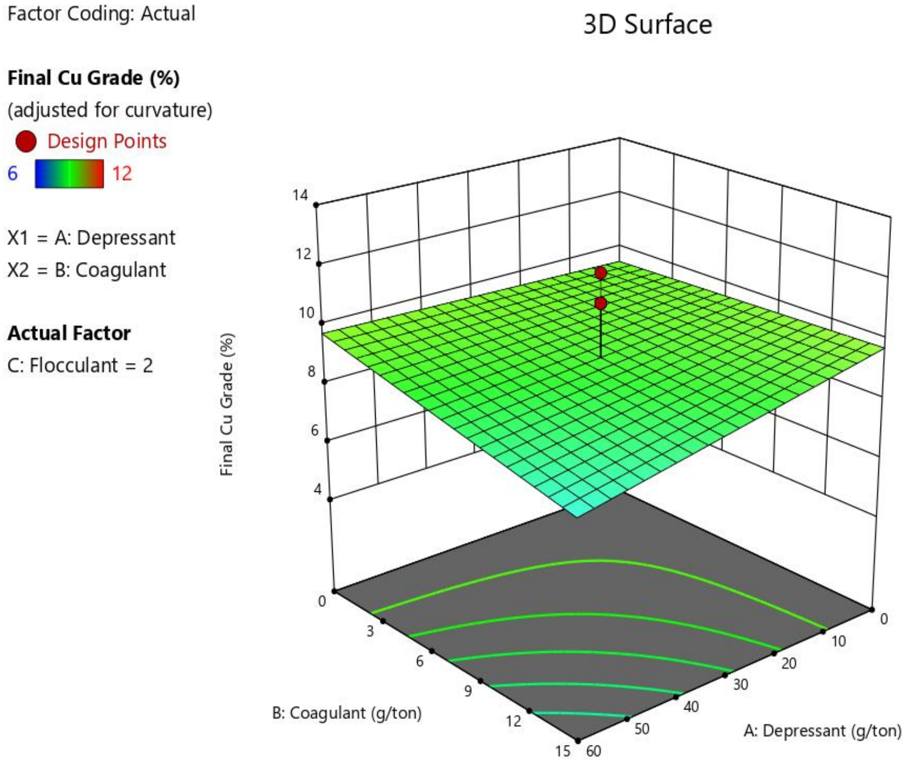

Figure 45 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Copper grade (%) at 2 g/ton Flocculant dosage.

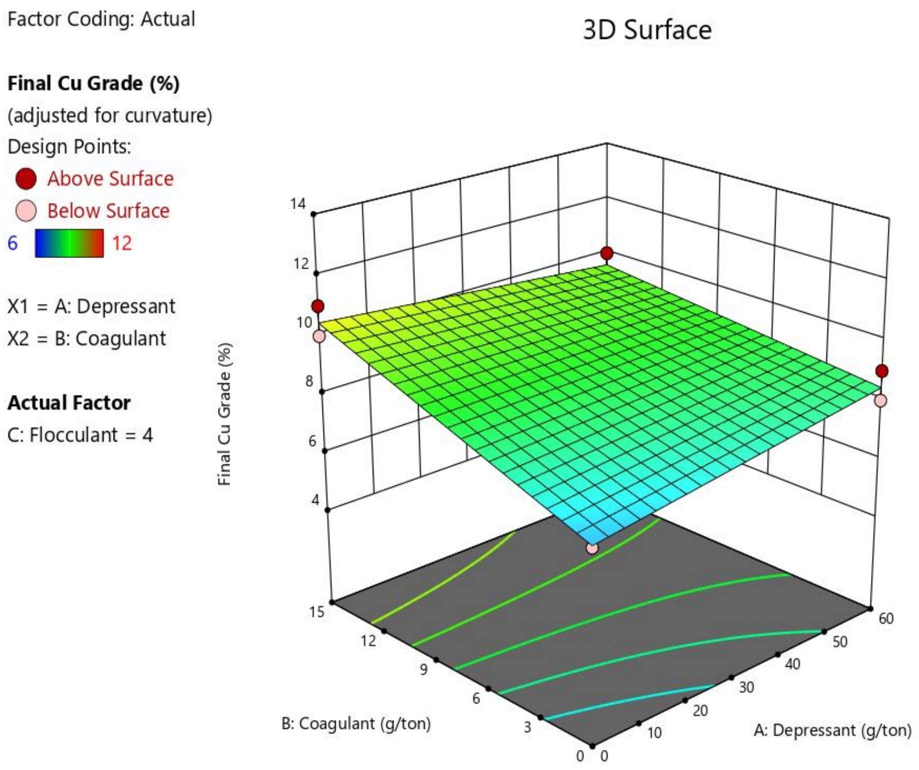

Figure 46 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Copper grade (%) at 4 g/ton Flocculant dosage.

5.6.4 Copper Recovery

The following conditions and interactions were significant toward recovery and are highlighted in Table 12

Table 12 Table indicating conditions and interactions that impact the copper recovery of the flotation process statistically.

Conditions	Symbol	P-test
Depressant	A	0.2681
Coagulant	B	0.108
Flocculant	C	0.0013
Depressant and Coagulant	AB	0.5715
Depressant and Flocculant	AC	0.4532
Coagulant and Flocculant	BC	0.2008
Depressant, Coagulant, and Flocculant	ABC	0.014
Model		0.0102

The model equation developed by the software is:

$$Final\ copper\ recovery = 62.5 + 0.12(A) + 0.57(B) + 3(C) - 0.01(AB) - 0.04(AC) - 0.18(BC) + 0.0042(ABC)$$

It can be seen in Table 12 and Figure 47 that there are only two conditions that impact copper recovery, these are C and ABC. The other conditions are included in the model equation as the condition ABC cannot be modelled without incorporating the other conditions.

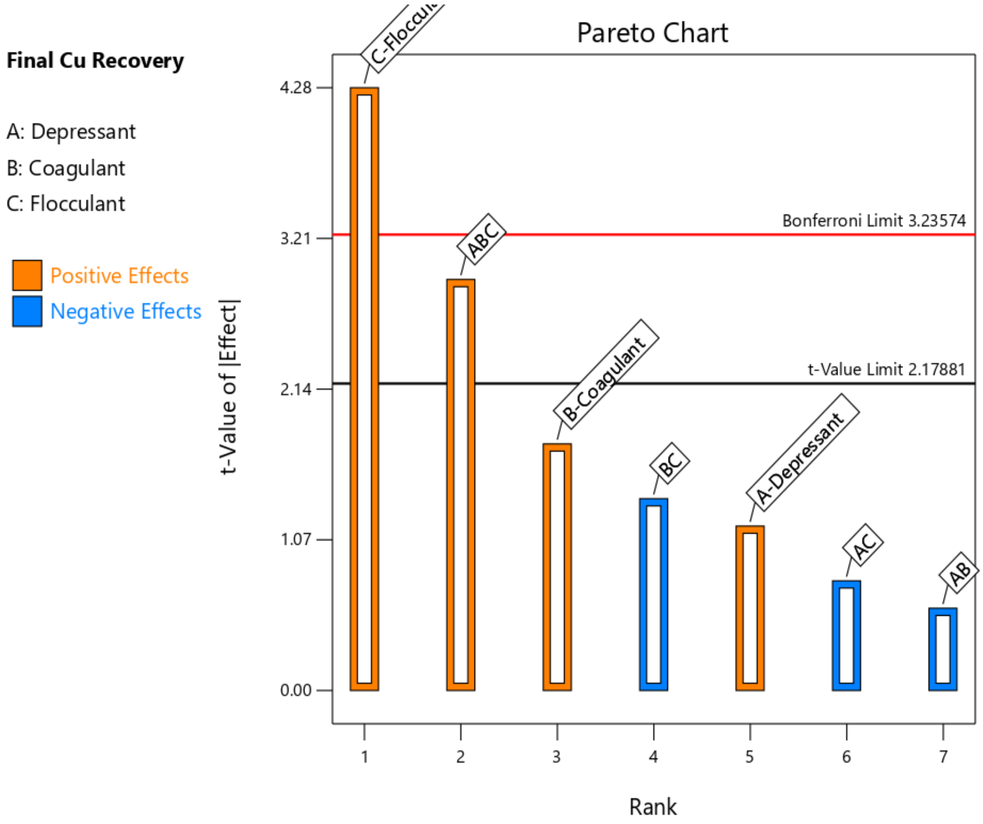

Figure 47 Pareto displaying conditions affect.

The impact of condition C can be seen by focusing on the point where the depressant and coagulant dosage is zero in Figures 48,49 and 50. It can be seen as the flocculant dosage rises, the final copper recovery also increases with it.

The impact of condition ABC can be seen by focusing on the point where the depressant and coagulant dosages are maximum in Figures 48,49 and 50. It can be seen as the flocculant dosage rises, the final copper recovery also increases with it.

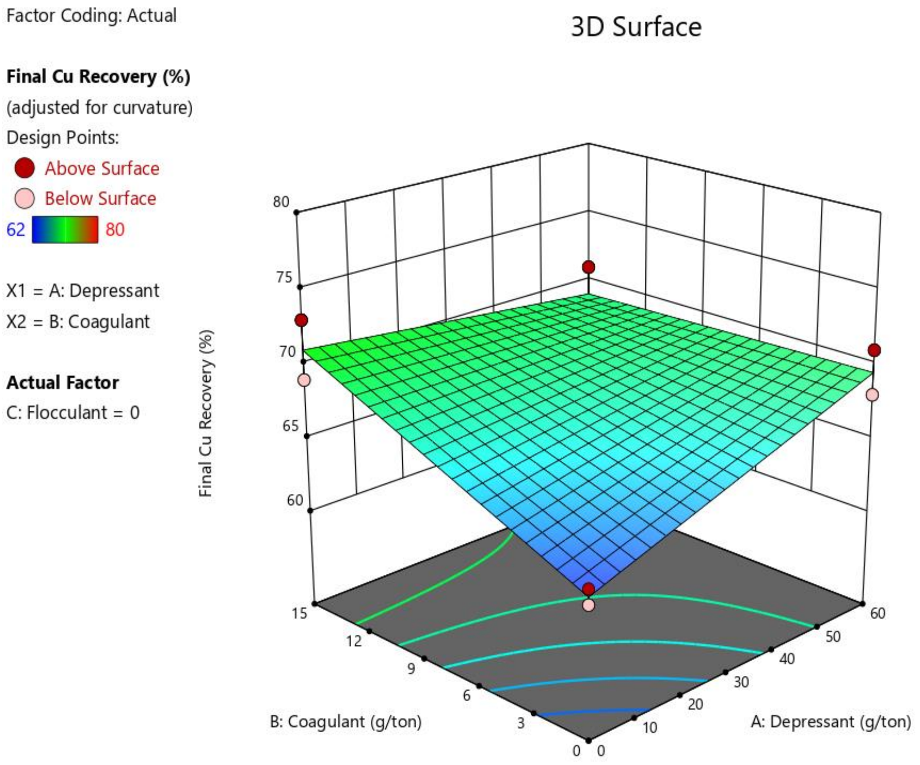

Figure 48 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Copper recovery (%) at 0 g/ton Flocculant dosage.

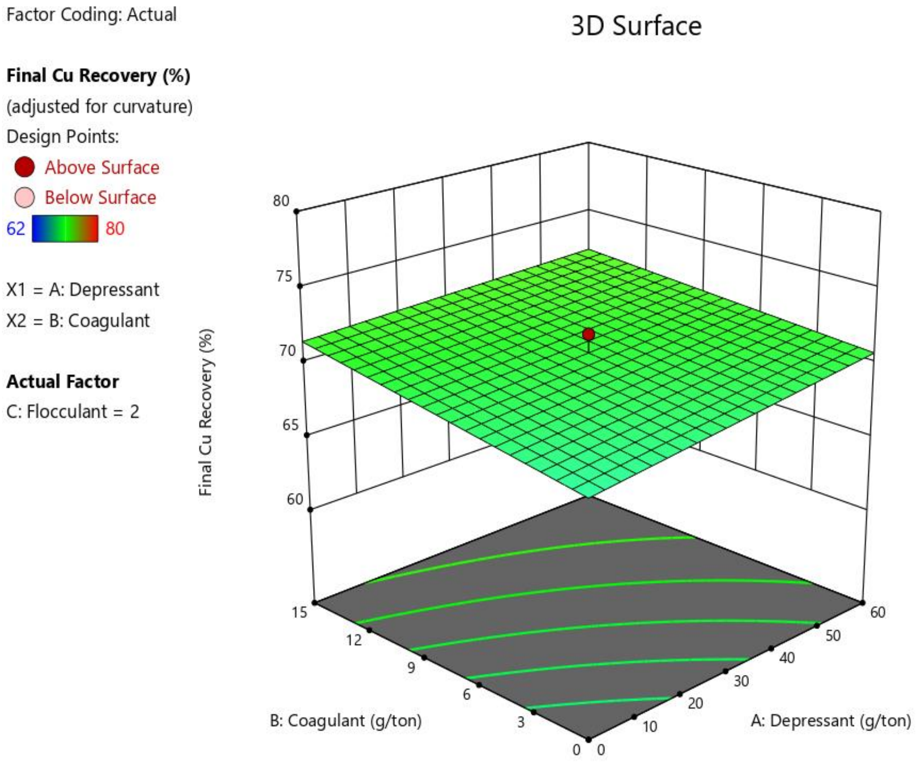

Figure 49 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Copper recovery (%) at 2 g/ton Flocculant dosage.

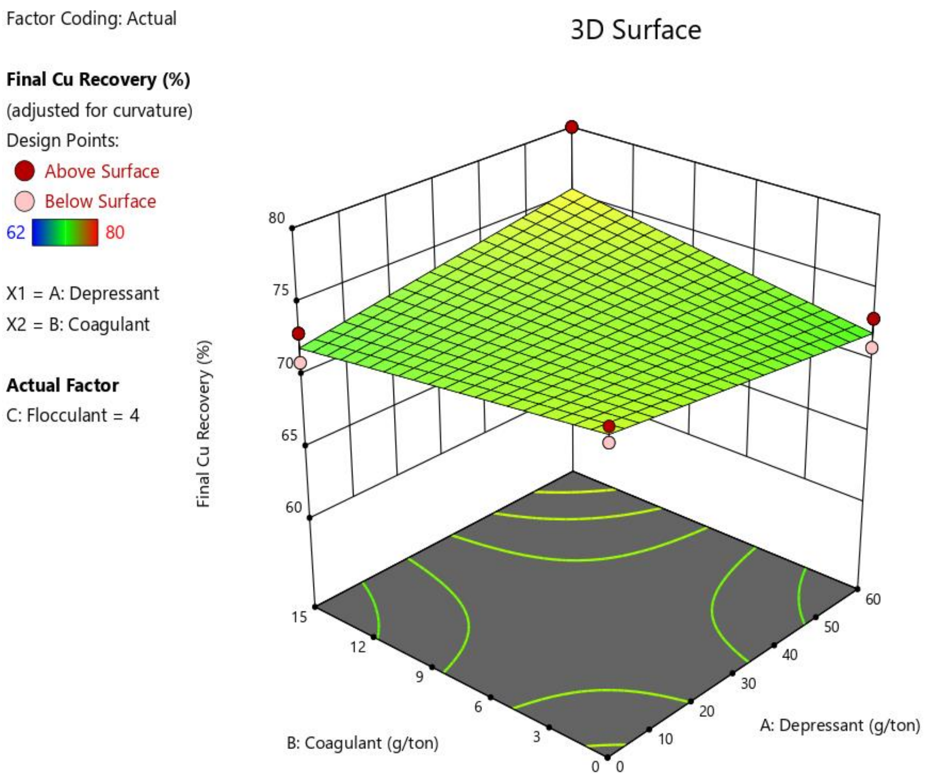

Figure 50 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Copper recovery (%) at 4 g/ton Flocculant dosage.

5.6.5 Nickel Grade

The following conditions and interactions were significant toward recovery and are highlighted in Table 13.

Table 13 Table indicating conditions and interactions that impact the nickel grade of the flotation process statistically.

Conditions	Symbol	P-test
Depressant	A	0.1519
Coagulant	B	0.0105
Flocculant	C	0.0061
Depressant and Coagulant	AB	0.0035
Depressant and Flocculant	AC	0.0061
Coagulant and Flocculant	BC	< 0.0001
Depressant, Coagulant, and Flocculant	ABC	0.0315
Model		< 0.0001

The model equation developed by the software is:

$$\begin{aligned} &Final\ nickel\ grade \\ &\quad = 51.5 - 0.05(A) - 0.63(B) - 3.86(C) - 0.002(AB) + 0.04(AC) \\ &\quad + 0.33(BC) - 0.002(ABC) \end{aligned}$$

It can be seen from Table 13 and Figure 51 that nearly all the conditions had an impact on the grade of nickel except for A. The condition that stood out and had the most significant condition was BC.

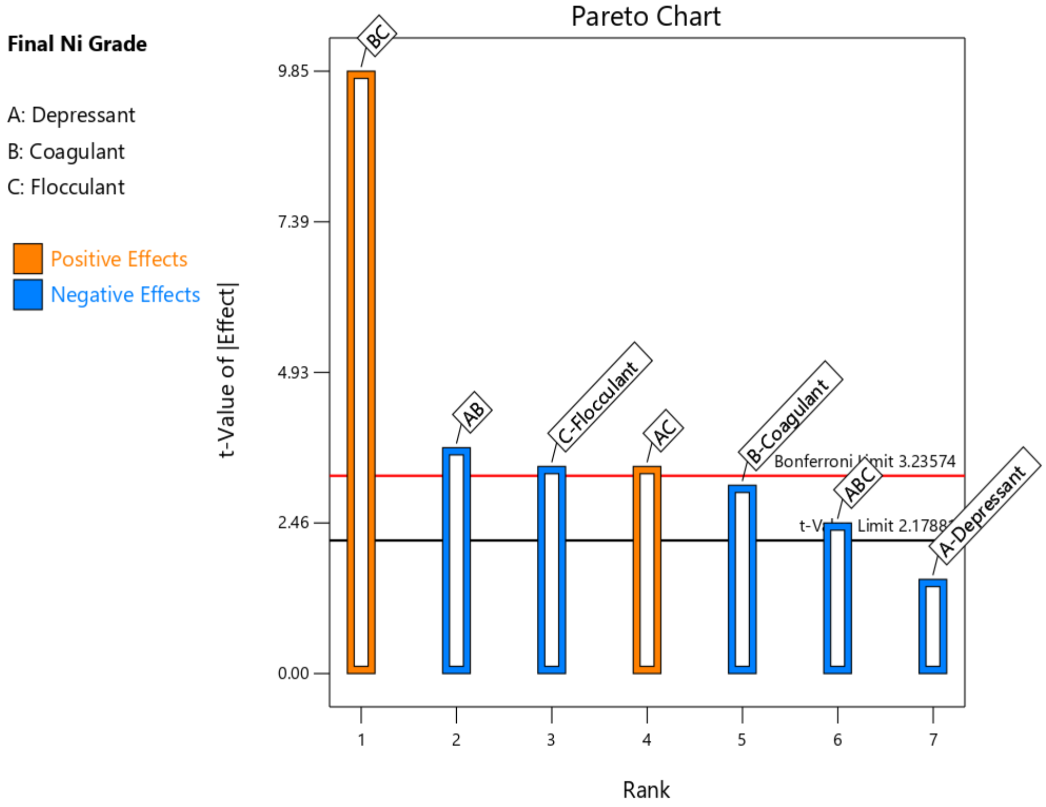

Figure 51 Pareto displaying conditions affect.

The impact of BC can be seen when focusing on the line where the depressant dosage is zero. It can be seen in Figure 52 that as the coagulant dosage increases the grade decreases. However, when flocculant is introduced in Figure 54 the opposite effect is seen, as the coagulant dosage increases so does the grade of nickel.

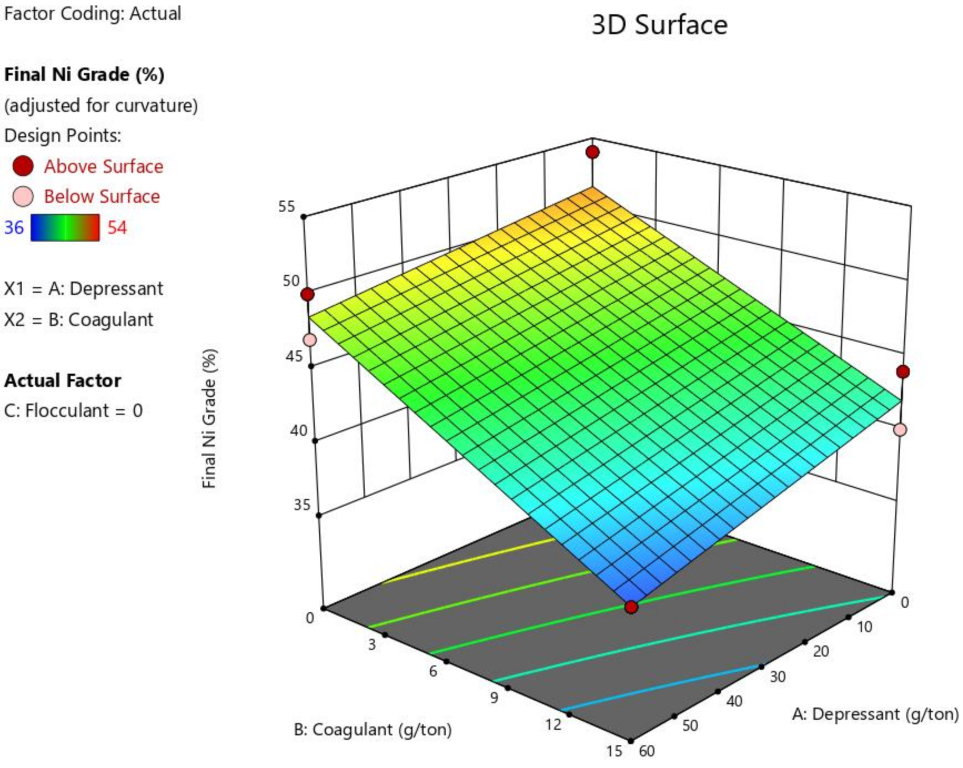

Figure 52 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Nickel grade (%) at 0 g/ton Flocculant dosage.

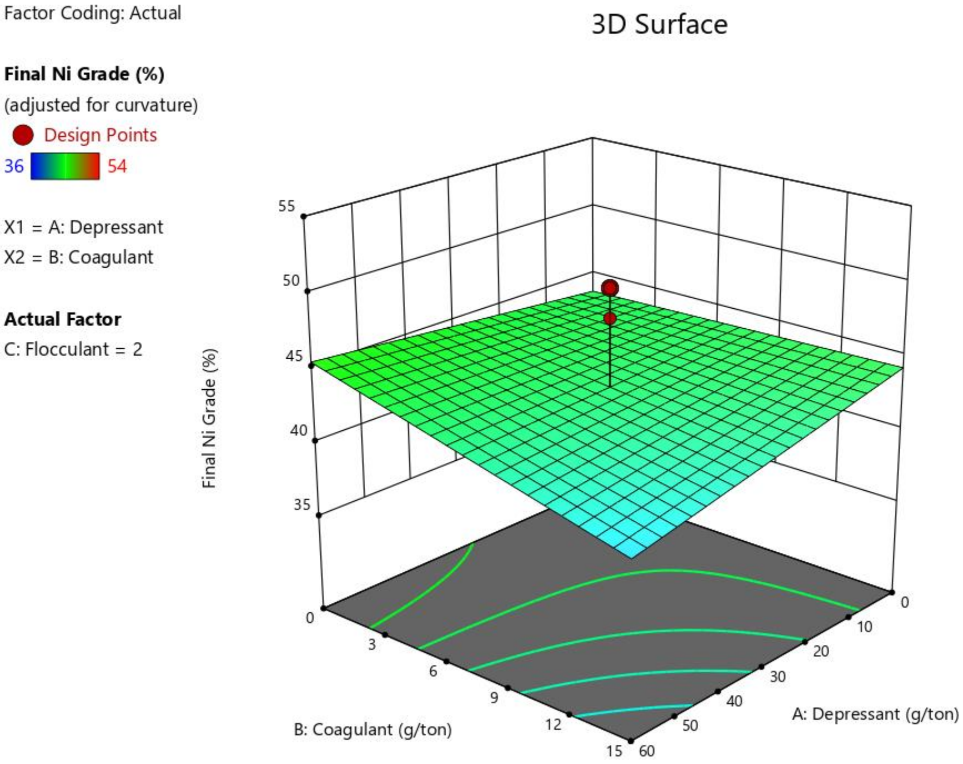

Figure 53 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Nickel grade (%) at 2 g/ton Flocculant dosage.

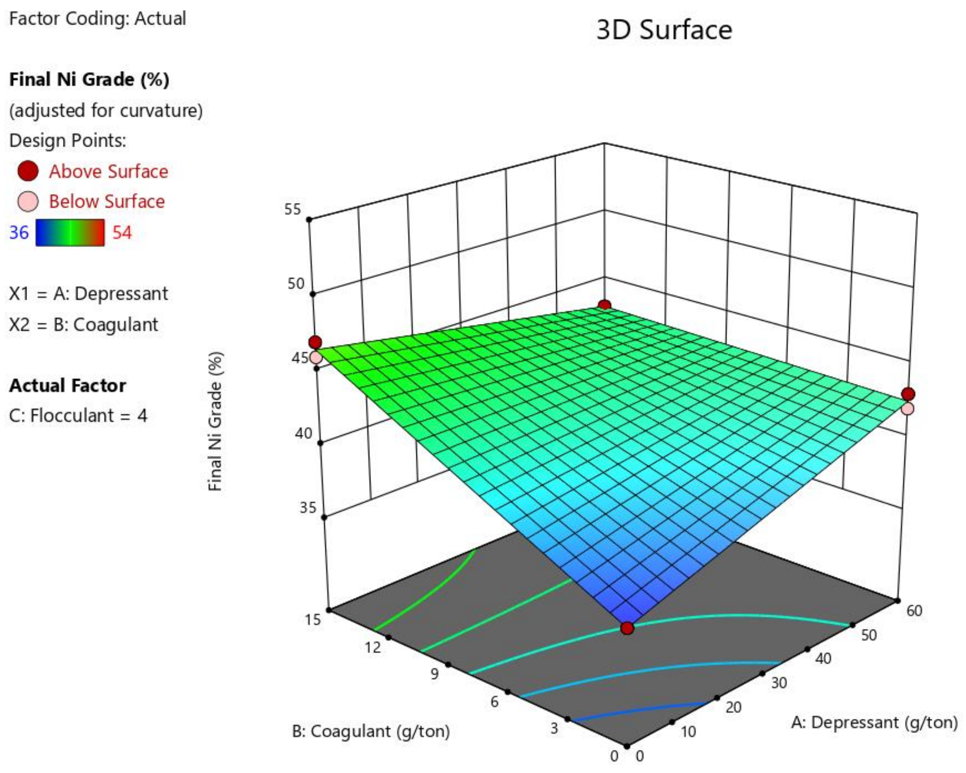

Figure 54 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Nickel grade (%) at 4 g/ton Flocculant dosage.

5.6.6 Final Nickel Recovery

The following conditions and interactions were significant toward recovery and are highlighted in Table 14.

Table 14 Table indicating conditions and interactions that impact the nickel recovery of the flotation process statistically.

Conditions	Symbol	P-test
Depressant	A	0.048
Coagulant	B	0.001
Flocculant	C	< 0.0001
Depressant and Coagulant	AB	< 0.0001
Depressant and Flocculant	AC	< 0.0001
Coagulant and Flocculant	BC	< 0.0001
Depressant, Coagulant, and Flocculant	ABC	0.0035
Model		< 0.0001

The model equation developed by the software is

$$\begin{aligned}Final\ nickel\ recovery &= 23 + 0.025(A) + 0.3(B) + 2.625(C) + 0.0033(AB) - 0.029(AC) \\ &- 0.19(BC) + 0.0014(ABC)\end{aligned}$$

It can be seen from Table 14 and Figure 55 that all the conditions impact the final nickel recovery. The most significant conditions that impacted nickel recovery are BC, AB, and C

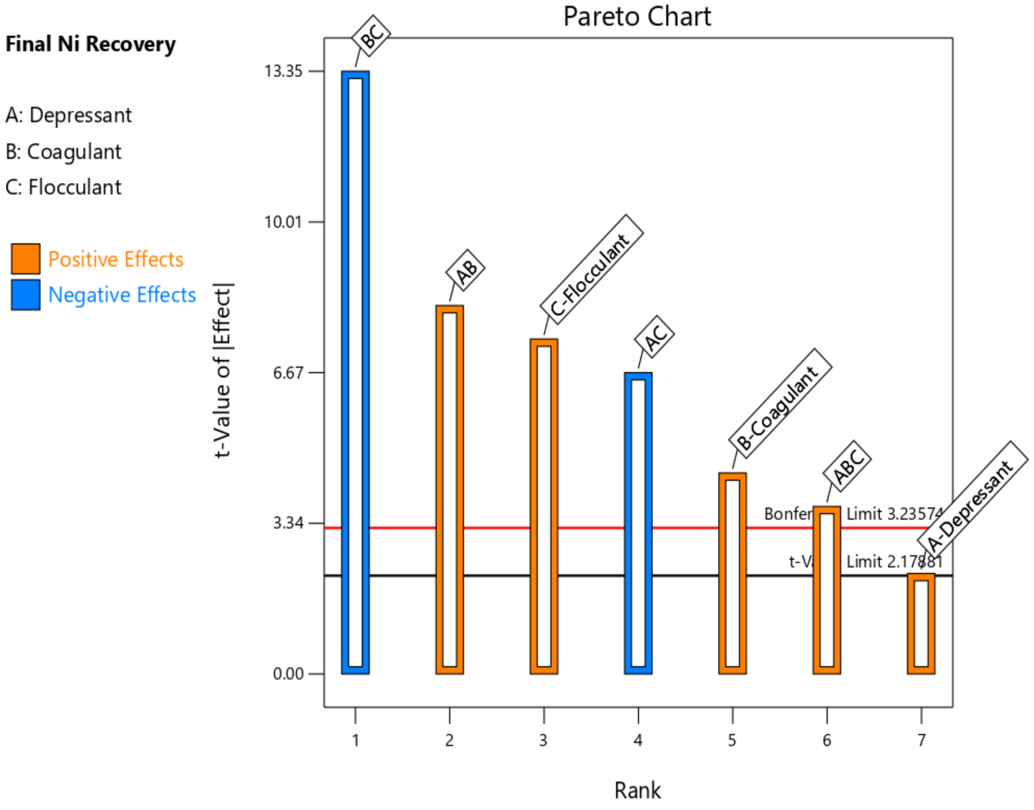

Figure 55 Pareto displaying conditions affect.

The BC condition can be seen to have the most significant impact, it can be observed by focusing on the line at the point where the depressant dosage is zero in the system. In Figure 56 As the coagulant dosage increases so does the nickel recovery. However, when flocculant is added to the system as in Figure 58 nickel recovery seems to be unimpacted by the increasing coagulant dosages.

The effects of condition AB can be seen on the X-Y plane of Figure 56, the contour lines indicate that increase in depressant and coagulant dosage results in an increase in the recovery of nickel.

The effects of condition C can be seen focusing on the point where the dosage of the depressant and coagulant is zero. Moving from Figure 56 where there is no flocculant to Figure 58 where the flocculant dosage is maximum, it can be seen that the nickel recovery increases as the flocculant dosage increases.

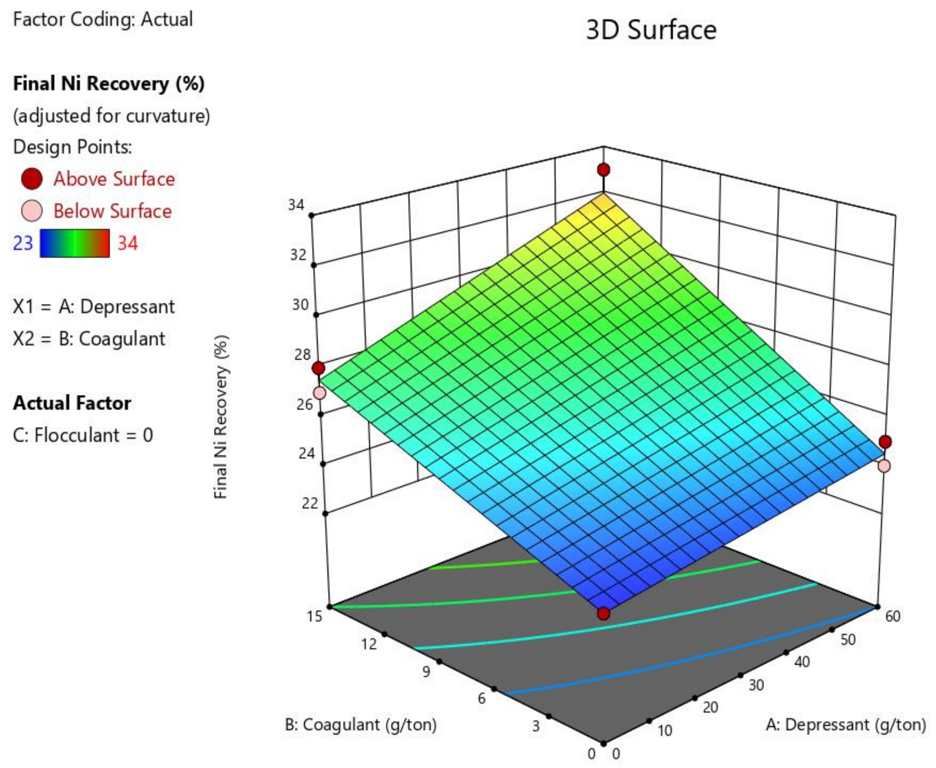

Figure 56 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Nickel recovery (%) at 0 g/ton Flocculant dosage.

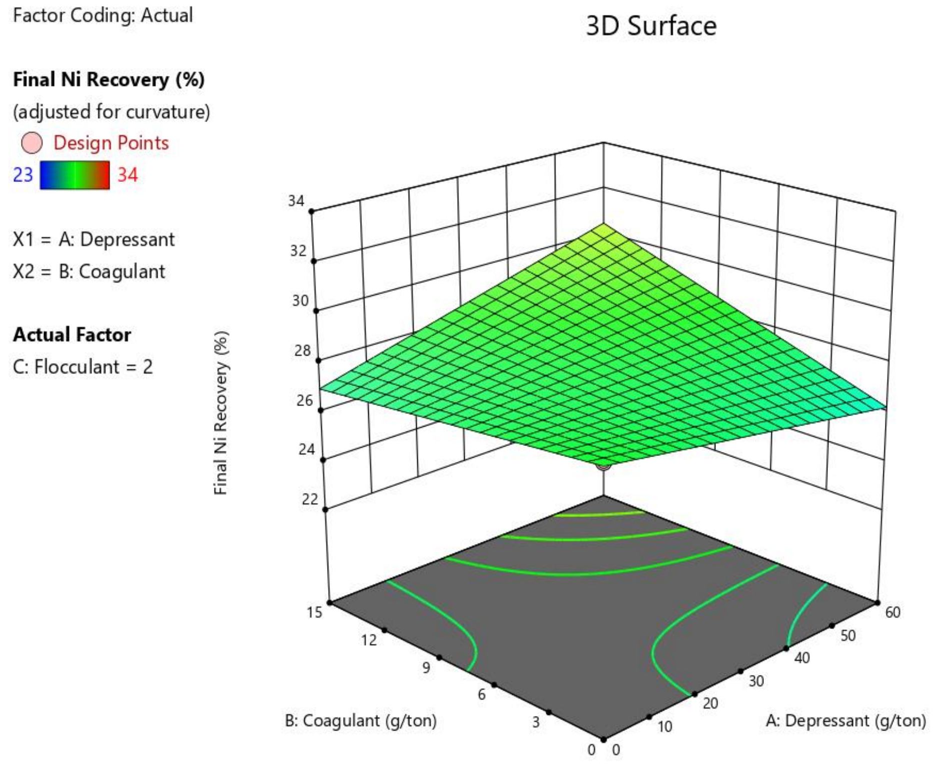

Figure 57 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Nickel recovery (%) at 2 g/ton Flocculant dosage.

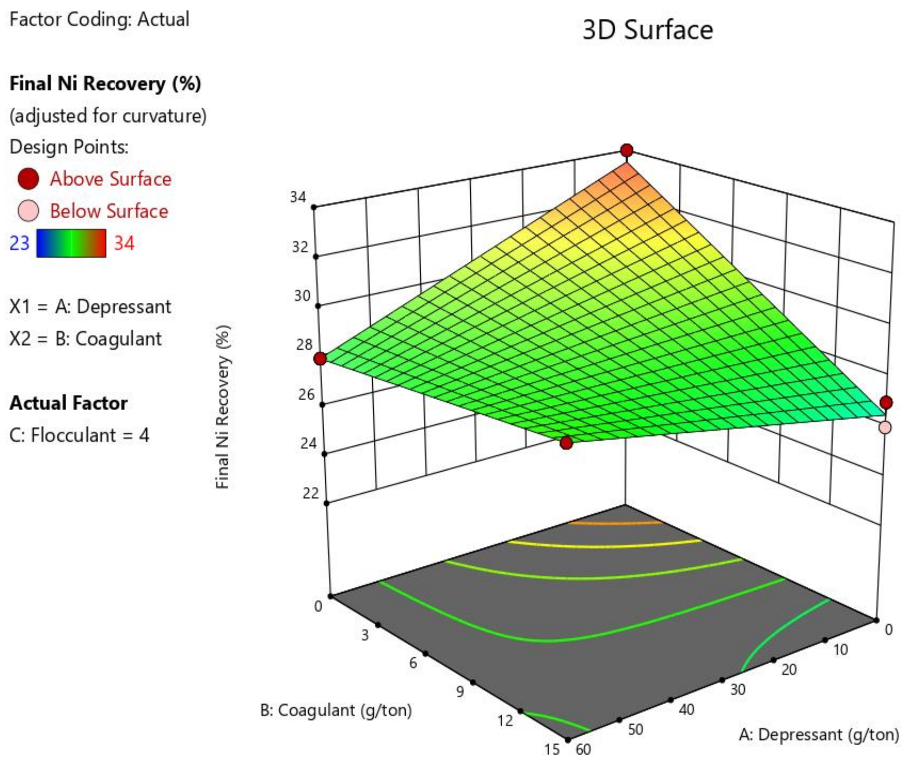

Figure 58 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Final Nickel recovery (%) at 4 g/ton Flocculant dosage.

5.6.7 Settling Rate

The following conditions and interactions were significant towards recovery and are highlighted in Table 15.

Table 15 Table indicating conditions and interactions that impact the settling rate of the flotation tailings statistically.

Conditions	Symbol	P-test
Depressant	A	0.904
Coagulant	B	0.0934
Flocculant	C	0.2426
Depressant and Coagulant	AB	0.3465
Depressant and Flocculant	AC	0.1362
Coagulant and Flocculant	BC	0.2379
Depressant, Coagulant, and Flocculant	ABC	0.5044
Model		0.2629

The model equation developed by the software is:

$$\begin{aligned} Settling\ rate = {} & -180 - 0.38(A) + 2.11(B) + 4.43(C) + 0.0074(AB) + 0.087(AC) \\ & - 0.73(BC) + 0.0087(ABC) \end{aligned}$$

It can be seen from Table 15, Figure 59, and Figure 60 that the settling rate of the tailings is unaffected by any of the conditions and while a model is generated, the model itself is insignificant and of little value in interpreting the data.

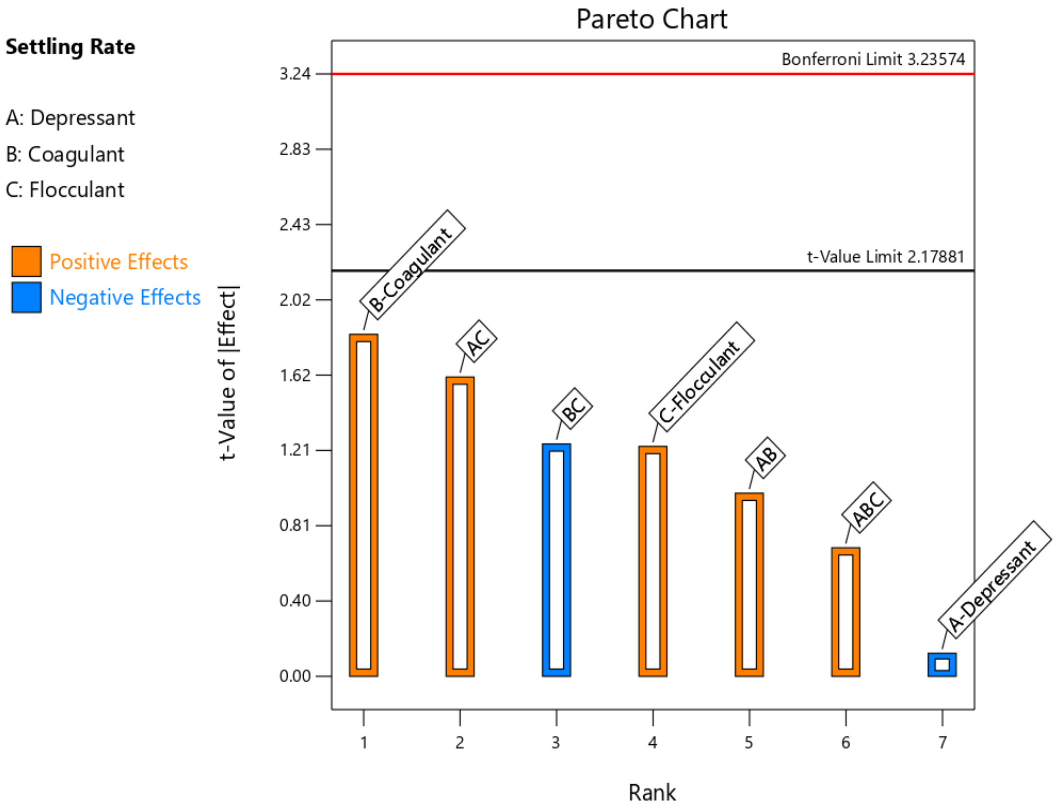

Figure 59 Pareto displaying conditions affect.

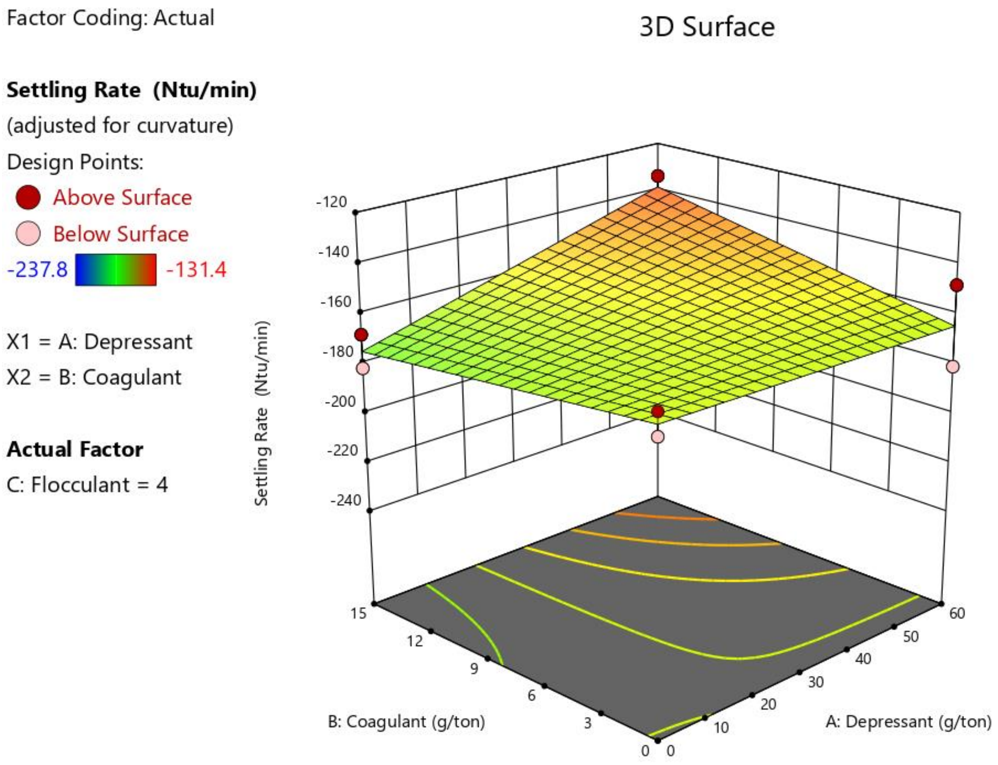

Figure 60 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs Settling rate (Ntu/min) at 4 g/ton Flocculant dosage.

5.6.8 Depressant feed concentrations

The following conditions and interactions were significant toward recovery and are highlighted in Table 16.

Table 16 Table indicating conditions and interactions that impact the depressant feed concentration of the flotation process statistically.

Conditions	Symbol	P-test
Depressant	A	0.0095
Coagulant	B	0.0835
Flocculant	C	0.2087
Coagulant and Flocculant	BC	0.0384
Model		0.0114

The model equation developed by the software is:

$$Depressant\ feed\ concentration = 16.05 + 0.14(A) - 0.08(B) - 0.66(C) + 0.21(BC)$$

It can be seen from Table 16 and Figure 61 that there are two conditions that affect the feed concertation of the depressant these are BC and A. The A condition is the depressant and it’s obvious that dosing a higher depressant dosage would result in a higher concentration reading. The condition BC effects can be seen on the X-Y plane contours on Figure 62, 63 and 64, it can be seen that with increasing concentrations of BC the CMC feed reagent concentration also increases. However, it must be taken into consideration that the BC interaction is not a significant effect as seen in the pareto chart in Figure 61.

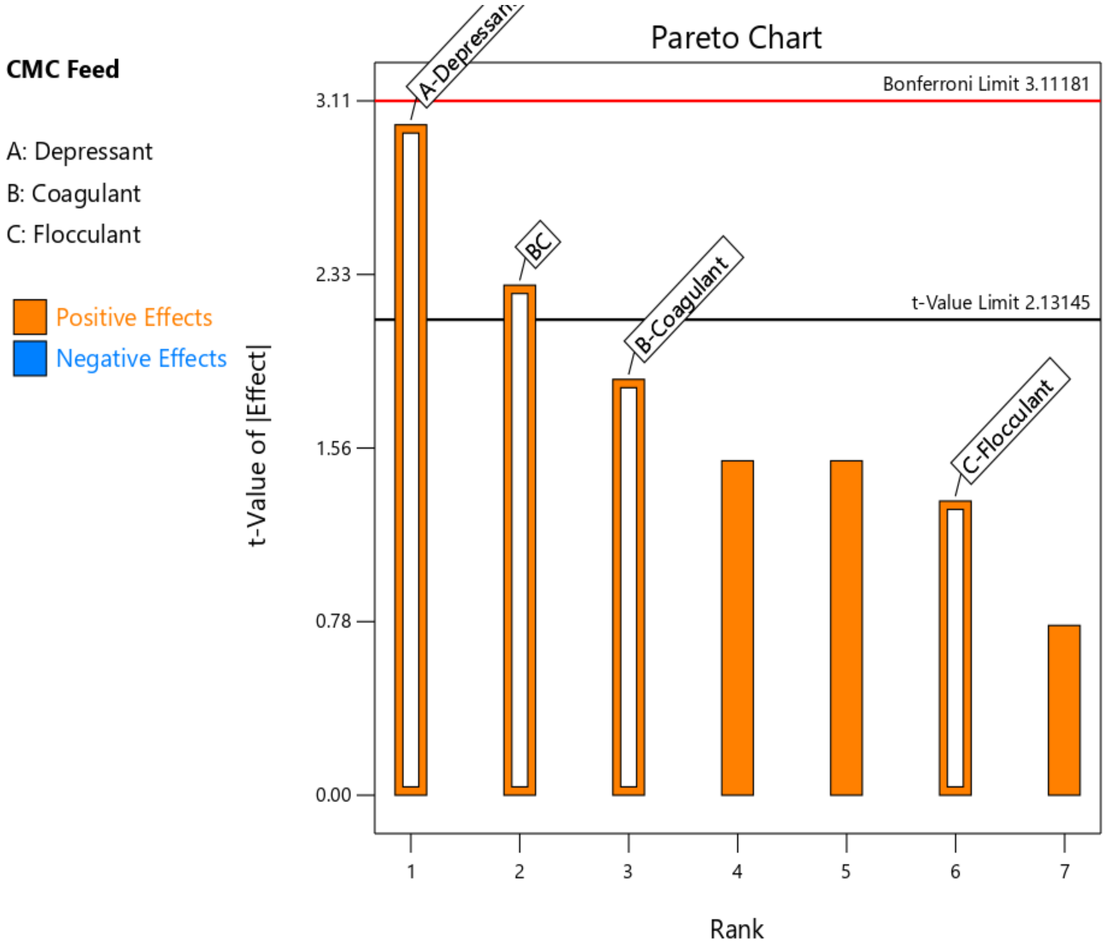

Figure 61 Pareto displaying conditions affect.

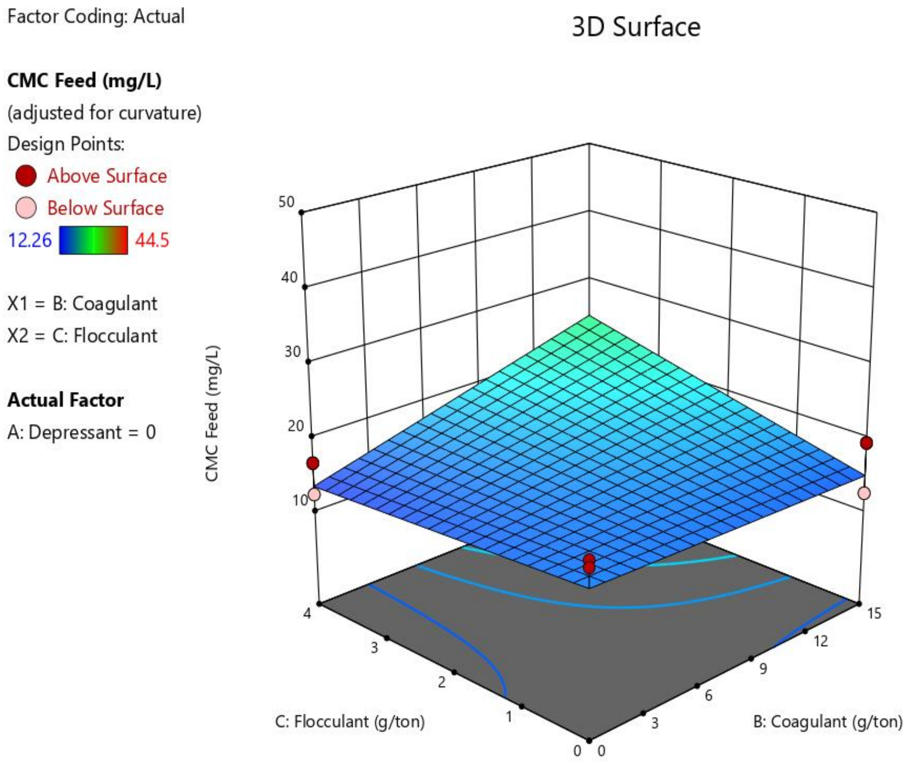

Figure 62 Flocculant dosage (g/ton) vs Coagulant dosage (g/ton) vs CMC feed concentration (mg/L) at 0 g/ton depressant dosage.

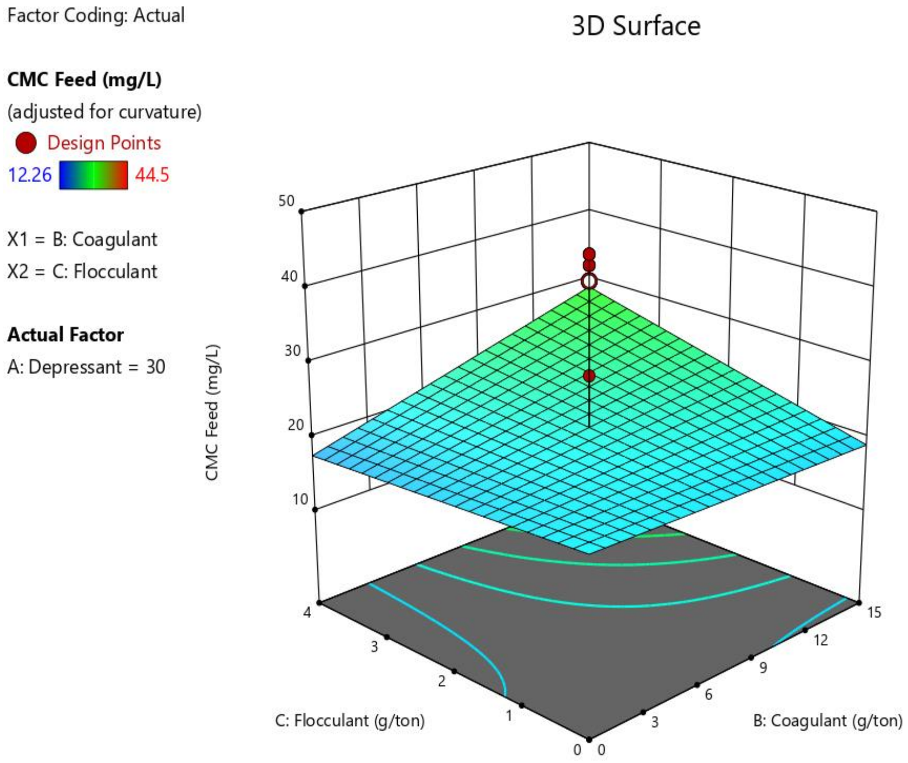

Figure 63 Flocculant dosage (g/ton) vs Coagulant dosage (g/ton) vs CMC feed concentration (mg/L) at 30 g/ton depressant dosage.

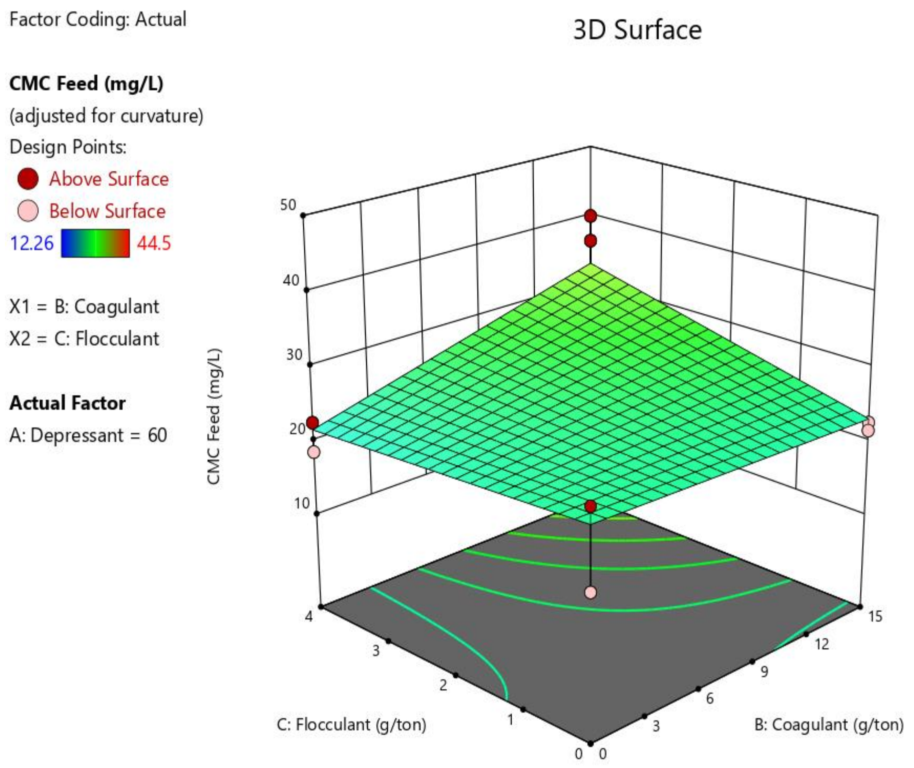

Figure 64 Flocculant dosage (g/ton) vs Coagulant dosage (g/ton) vs CMC feed concentration (mg/L) at 60 g/ton depressant dosage.

5.6.9 Depressant tailings concentrations

The following conditions and interactions were significant toward recovery and are highlighted in Table 17.

Table 17 Table indicating conditions and interactions that impact the depressant tailings concentration of the flotation process statistically.

Conditions	Symbol	P-test
Depressant	A	0.0391
Coagulant	B	0.6583
Flocculant	C	0.7791
Depressant and Coagulant	AB	0.0030
Coagulant and Flocculant	BC	0.0001
Model		0.0005

The model equation developed by the software is:

$$1/Depressant\ tailings\ concentrations = 0.03 + 0.00008(A) + 0.002(B) + 0.0045(C) - 0.000029(AB) - 0.00064(BC)$$

The model developed by the system determined that an inverse model would be more suitable for the system for the tailings depressant concentration.

It can be seen from Table 17 and Figure 65 that the two conditions that impact the residual depressant concentration are BC and AB.

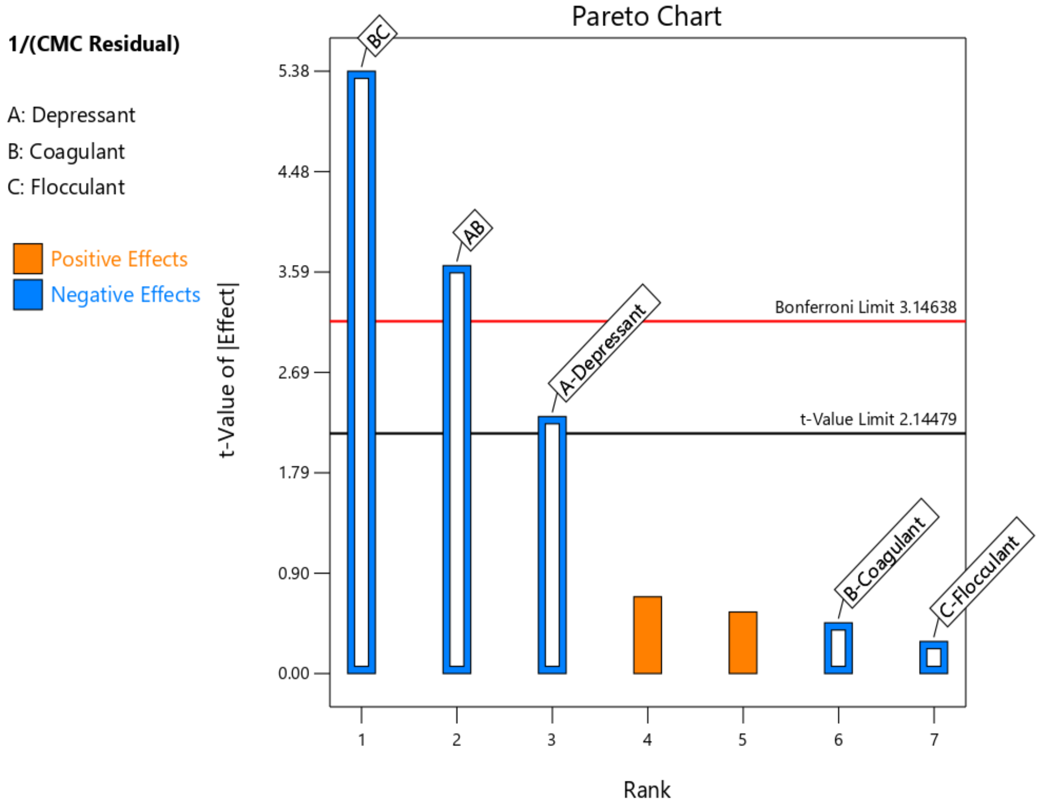

Figure 65 Pareto displaying conditions effect.

The condition AB can be seen with the Aid of the contour line on the X-Y plane of Figure 66, 67, and 68. As the reagent concentrations increase so does the amount of depressant in the tails.

The BC condition can be identified by examining the line representing zero depressant dosage on Figures 66 and 68. In Figure 66 as the coagulant dosage increases the depressant concentration decreases, however when flocculant is added to the system as seen in Figure 67 there is a minimum effect on concentration as the coagulant dosage increases.

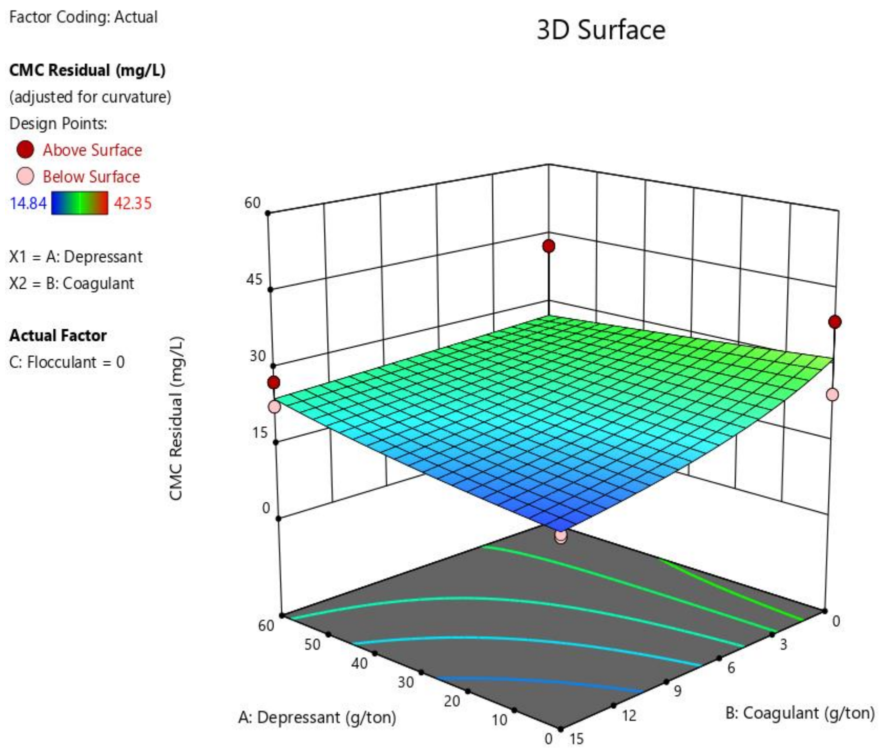

Figure 66 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs CMC residual concertation (mg/L) at 0 g/ton Flocculant dosage.

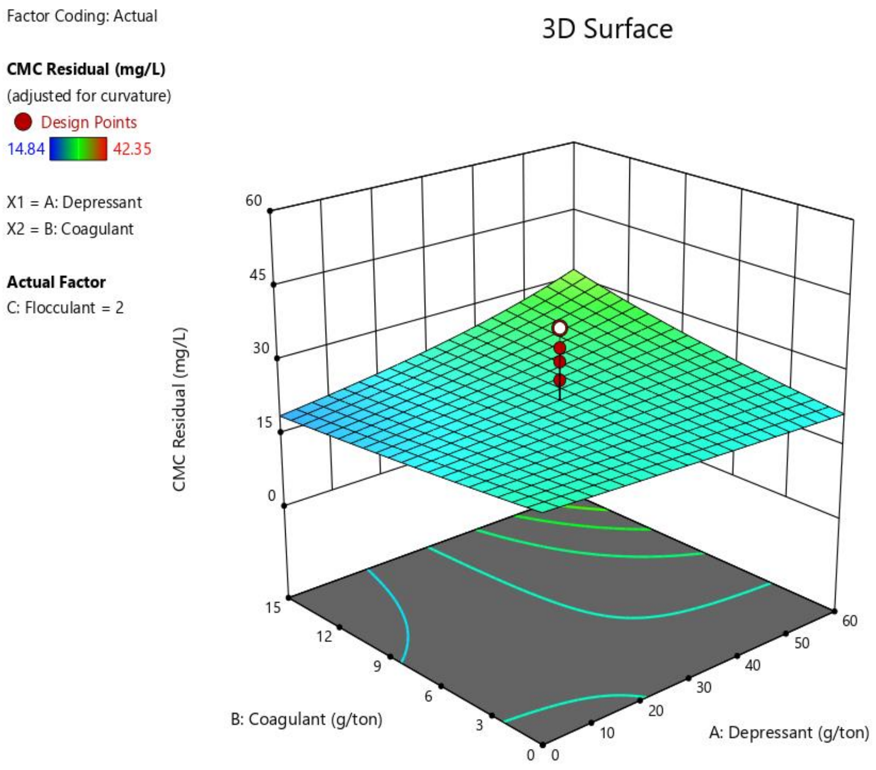

Figure 67 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs CMC residual concertation (mg/L) at 2 g/ton Flocculant dosage.

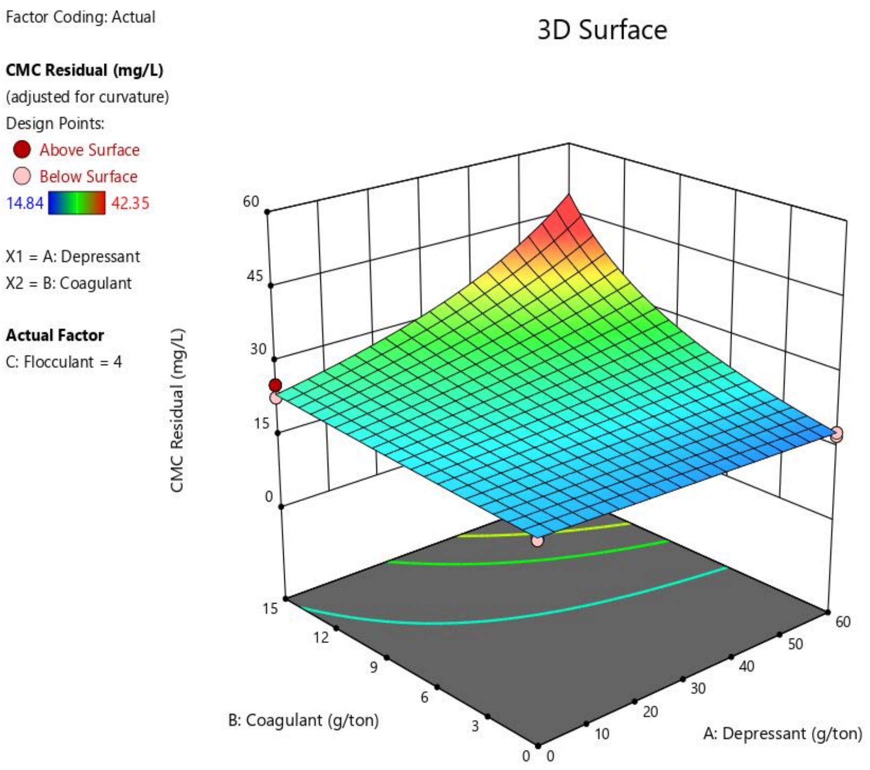

Figure 68 Depressant dosage(g/ton) vs Coagulant dosage(g/ton) vs CMC residual concertation (mg/L) at 4 g/ton Flocculant dosage

5.6.10 Collector feed concentrations

The following conditions and interactions were significant for recovery and are highlighted in Table 18.

Table 18 Table indicating conditions and interactions that impact the collector feed concentration of the flotation process statistically.

Conditions	Symbol	P-test
Depressant	A	0.1770
Coagulant	B	0.9090
Flocculant	C	0.8490
Depressant and Coagulant	AB	0.9090
Depressant and Flocculant	AC	0.1566
Coagulant and Flocculant	BC	0.7323
Depressant, Coagulant, and Flocculant	ABC	0.7323
Model		0.6928

The model equation developed by the software is:

$$SIBX\ feed\ concentration = 0.42 + 0.0018(A) + 0.014(C) - 0.000033(AB) - 0.0005(AC) + 0.000012(ABC)$$

It can be seen from Table 18 and Figure 69 that there were no significant conditions that affected the collector feed concentration and while a model is generated, the model itself is insignificant and of little value in interpreting the data.

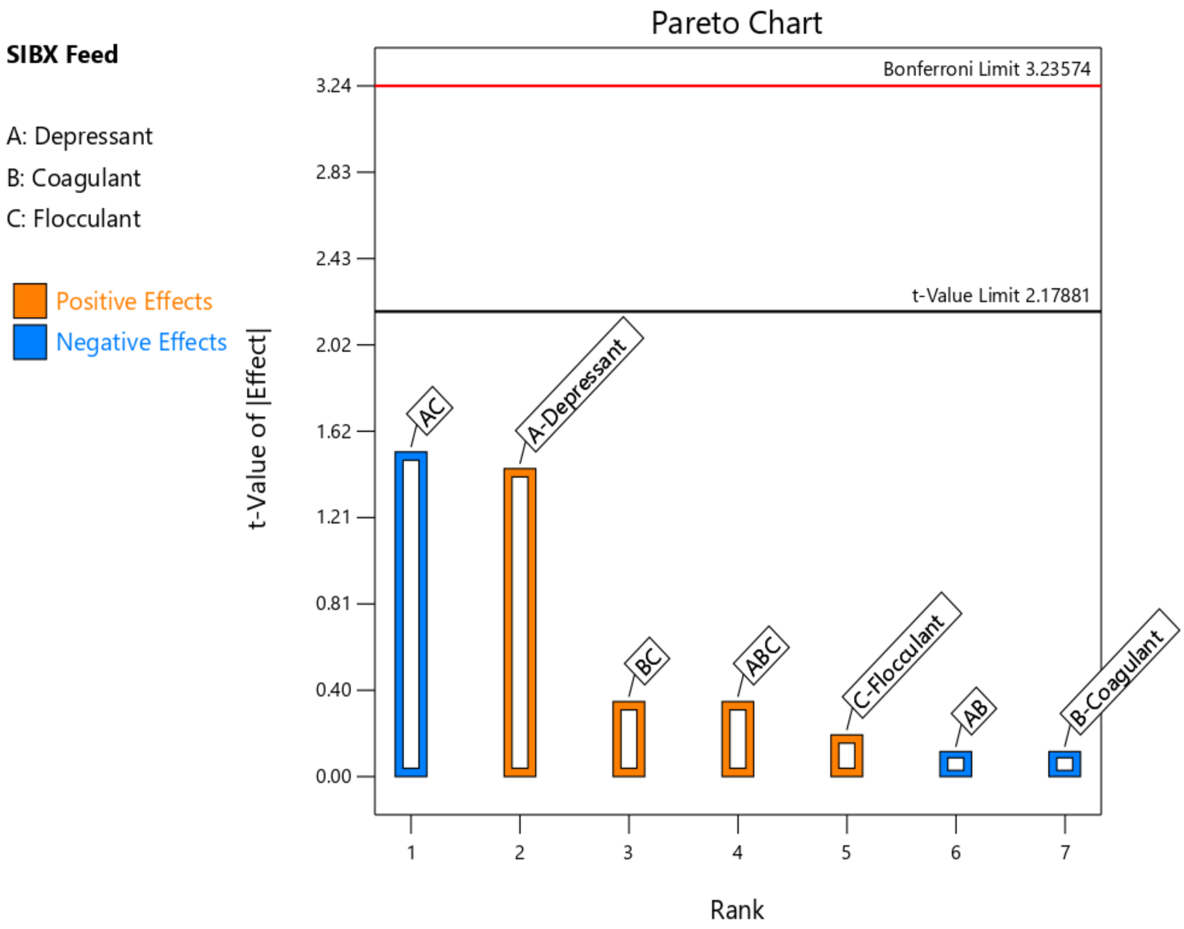

Figure 69 Pareto displaying conditions affect.

5.6.11 Collector tailings concentrations

The following conditions and interactions were significant for recovery and are highlighted in Table 19.

Table 19 Table indicating conditions and interactions that impact the collector tailings concentration of the flotation process statistically.

Conditions	Symbol	P-test
Flocculant	C	0.0430
Model		0.043

The model equation developed by the software is:

$$SIBX\ tails\ concentration = 0.39 + 0.012(C)$$

It can be seen from Table 19 and Figure 70 that there was only one condition that impacted the collectors' tailings concentration. However, the condition itself can barely be considered significant as seen in Figure 70. Figure 71 shows the impact of increasing flocculant dosage on the tailing's concentration of the collector, as the flocculant concentration increases so does the concentration of the collector in the tailings.

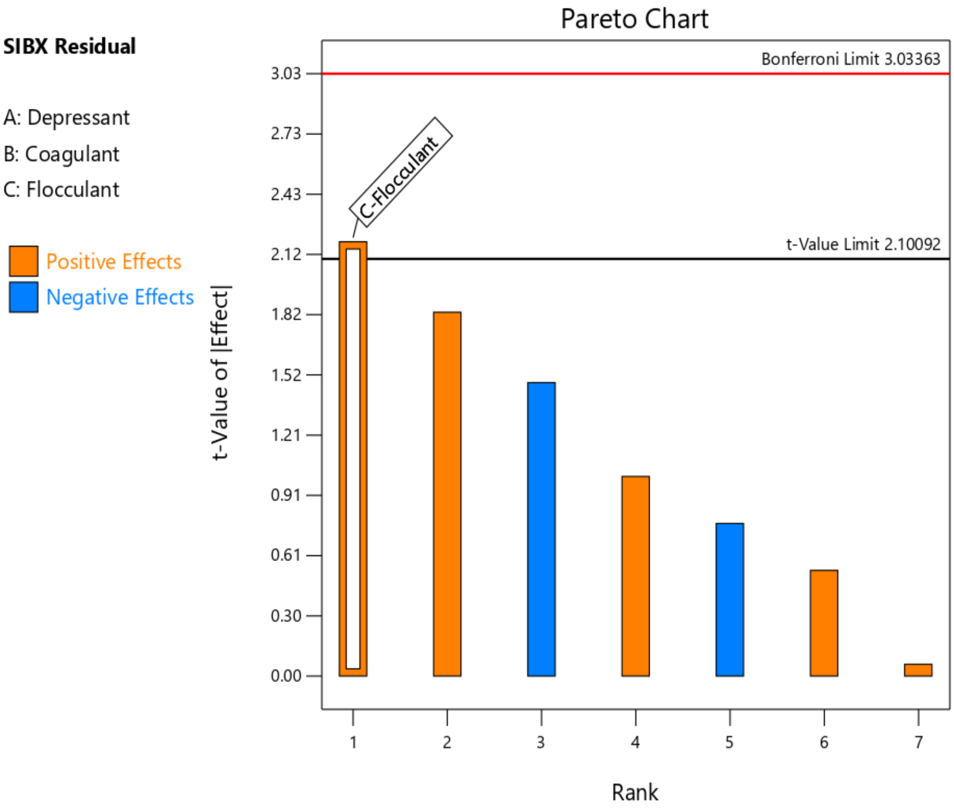

Figure 70 Pareto displaying conditions affect.

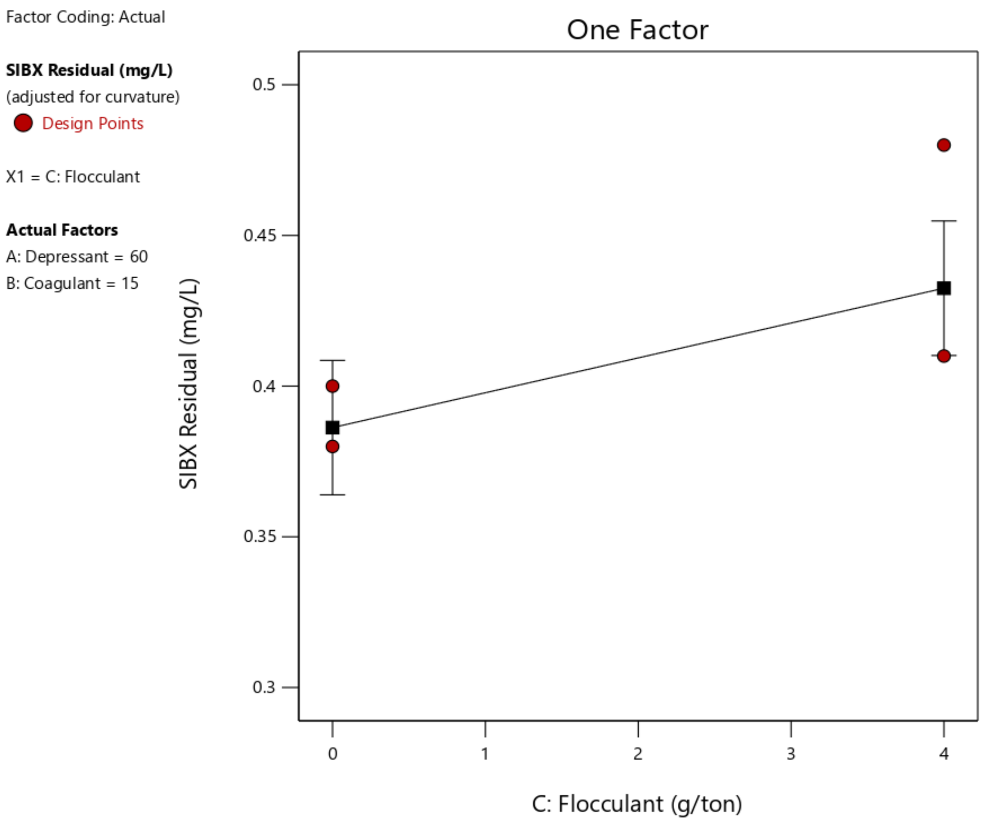

Figure 71 SIBX residual (g/ton) vs Flocculant dosage(g/ton) at depressant dosage 60 g/ton and coagulant dosage 15 g/ton.

6 Discussion

6.1 What is the Effect of Dewatering Reagents on Key Flotation Outcomes?

All variations resulted in an increase in the final water and solids recoveries relative to the base where none of the reagents were added (as seen in Figure 19 and 20). This observation can be further supported with the aid of the statistical analysis that was reported under Sections 5.6.1 and 5.6.2 that clearly showed that all reagent variables and combinations thereof resulted in an increase in water and solids recoveries.

The increase in solids and water recovery may be owing to several reasons. It could be attributed to the ionic properties of the reagents that were added to the system. An increase in ionic concentration of the system is known to increase the stability of the froth, that results in an increase of solids and water recoveries (Corin et al. 2011). The presence of ions in the froth impacts the surface tension and inhibits bubble coalescence, resulting in smaller and more stable bubbles and thus creating a more stable froth (Quinn et al. 2007). Another factor that could contribute to a rise in solid recoveries is the compression of the electrical double layer caused by ionization, that enhances the adherence of bubbles to particles (Farrokhpay and Zanin 2012), or more particles being trapped in the plateau borders due to ions slowing inter-bubble drainage (Manono et al. 2013).

A further possible reason for the increase in solids and water recoveries could be that coagulants cause particles to adhere to each other while flocculants adhere fine particles to form larger particles (Napier-Munn and Wills 2005; Sharma et al. 2006). When these settling reagents are added to the system, it can be expected that the size of the agglomerated particles would increase, and the quantity of smaller sized agglomerates and individual particles would be less. Particle size is a key factor in the performance of a flotation system, thus making it essential to have the optimal size of particles that is not too fine or coarse (Feng and Aldrich 1999; Norori-McCormac et al. 2017). The increase in agglomerate particle size that was to be expected should have had a negative impact on the solid recoveries due to poor bubble-particle aggregation in turbulent zones and decrease in buoyancy of the particle-bubble aggregate relative to the pulp (Feng and Aldrich 1999).

The decreased presence of solids in the froth would also mean a decrease in the froth stability as the presence of solids is a well-known factor that increases froth stability which would result in a decrease in water recoveries (Feng and Aldrich 1999). However, the results observed from this study are contrary as both water and solids recoveries increased with the introduction of dewatering reagents possibly indicating that the impact was observed because off the ionic properties of the reagents.

The contradiction stated above could be owing to the grind size of 80% passing 75 µm which contains a significant portion of fine particles. Fine particles have slow recovery rates due to decreased particle-bubble collisions and have large specific areas, that can lead to excessive adsorption of reagents and higher oxidation of mineral surfaces (Dunn 1997; Feng and Aldrich 1999; Lange et al. 1997). When dewatering reagents were added to the system, they could have interacted with the fine particles primarily due to their large surface area resulting in aggregates. These newly formed aggregates have better momentum resulting in more particle-bubble collisions and a faster recovery rate due to their larger mass (Lange et al. 1997).

The copper recovery results can be seen in Figure 23, when reagents or a combination thereof are added to the system, there is a very slight increase in the recoveries of the copper bearing minerals that was deemed to be an insignificant increase in the statistical analysis Section 5.6.5. The final copper grade was impacted significantly by the addition of reagents, the grade had an inverse relationship with the final solid recoveries; conditions which had the highest solid recoveries had the lowest copper grades, which is expected and similar to the results obtained by others (Corin et al. 2011; Manono et al. 2012).

The final nickel recovery can be seen in Figure 27, when reagents or a combination thereof are added to the system, it can be seen that there is an increase in the recoveries of the nickel bearing mineral that is proportionate to the solid recoveries achieved under the different conditions. The final nickel grade seems to have an inverse relation with the recovery of the nickel bearing mineral, with increasing solid and nickel recoveries, the nickel grades decreased similar to the copper results these were also observed in previous studies done by (Corin et al. 2011; Manono et al. 2012; Farrokhpay and Zanin 2012).

The trends seen in the copper and nickel recoveries and grades, indicate that almost all the copper has been recovered while most of the nickel has been recovered. This could be attributed to the rates at which the minerals associated with these metals, float. It is widely acknowledged that chalcopyrite, which is a copper-bearing mineral, has a fast rate of floatation, while pentlandite, which is the primary carrier of nickel in sulfide Cu-Ni-PGM ores, floats relatively slowly (Corin et al. 2011). The decrease in grade can be associated with a significant portion of the increased solids recovered being gangue. Gangue recovery is increased through entrainment, which results from the increase in ionic strength. This increase in ionic strength stabilizes the froth, which can be observed by the corresponding increase in water recoveries. (Manono et al. 2012; Farrokhpay and Zanin 2012).

The results obtained could be attributed to two prevalent properties that are found in dewatering reagents. The ionic property that would have resulted in an increase of the ionic concentration of the system and could have caused an increase in the froth stability resulting in increased recoveries of gangue. The second property of the dewatering reagents being the adhesion of fine particles resulting in faster recovery rates due to better bubble-particle attachment.

6.2 How Do the Dewatering Reagents Affect Key Flotation Reagents?

Pre-flotation and post flotation concentrations of the collector and depressant were used to study the possible effect of the dewatering reagents. It was observed that the addition of the variable reagents and a combination thereof had no effect on the concentration of the flotation reagents pre and post flotations.

The collector concentrations pre-flotation and post flotation can be seen in Figure 33, the trend, seen clearly, indicates that the concentration of residual collector remained consistent under all conditions. The statistical analysis done for the pre-flotation concentration under Section 5.6.10 and post flotation concentration under Section 5.6.11, also confirmed no effect on both factors tested. These results indicate the adsorption of the collector at the mineral surface was not impacted by the addition of the dewatering reagents.

The depressant concentrations pre-flotation and post flotation under the different tested conditions can be seen in Figure 34. Similar to the collectors, there was no visible impact of the dewatering reagents to be seen in the concentrations pre and post flotation. The statistical analysis done on both factors tested are shown under Sections 5.6.8 and 5.6.9. These results indicate that the adsorption of the depressant on the gangue material was not impacted by the addition of dewatering reagents to the flotation system.

The primary impact of dewatering agents is to cause the agglomeration of fine particles into larger particles. There is an inverse relationship that exists between particle size and surface area. It is expected that with a decrease in surface area of the particle there would be an impact on the adsorption of the flotation reagents on the mineral surface, however the results discussed above indicates that the possible reduction in surface area has not had an impact on the adsorption of the flotation reagents.

Several studies have been done to study the impact of depressant molecular weight on the adsorption of the reagent. From these studies it was determined that the molecular weight of a CMC depressant does not impact the adsorption of the reagent on the mineral surface (Parolis et al. 2004; Wiese et al. 2008). This could be attributed to depressants being unable to achieve complete surface area coverage under flotation conditions (McFadzean et al. 2011). This indicates that as the molecular weight of the depressant increased (while its surface area decreased) it had little or no impact on the adsorption of the reagent. This can similarly be applied to the results seen in this study in that the probable change (decrease) in the surface area of the minerals was not significant enough to impact the adsorption of the reagent.

Finer particles with larger surface area are expected to consume most of the collector that is present in the system as they have a larger surface area and higher specific free energy (Trahar 1981). When dewatering reagents are added to the system the quantity of fine particles reduces thus impacting the adsorption and causing more interaction between the collector and more coarse particles. Coarser particles would require longer to achieve their critical hydrophobicity thus possibly impacting the recovery of the solids (Mcfadzean et al. 2012). Furthermore, there would have been a change in flotation feed collector concentration as collector is added to the milling process as highlighted in Section 4.5.2.

Since the recovery of the minerals copper and nickel remain unaffected, it can be assumed that the collector adsorption is unimpacted. This can be due to the dosage levels of the dewatering reagents used in this study not being high enough to reduce the surface area to such an extent that collector adsorption would be impacted. If adsorption was impacted, then recoveries would have decreased.

Additionally (Trahar 1981) indicated that tests done to prove the relationship between the surface area and adsorption were not necessarily proven as the methodology of testing surface area techniques using gas adsorption was subject to differences in specific free energy. This would lead to vastly different results.

6.3 Do the Dewatering Reagents Affect the Settling of Tailings?

There was no effect seen on settling rate of the tailings with the addition of the variable reagents and combinations thereof. This can be confirmed with the results in Figure 31 and Figure 32 and the statistical analysis done on these results seen in Section 5.6.7.

While Ma et al. (2019) showed that the addition of flocculants and coagulants tends to result in an increase in the rate of settling with increasing dosage to a certain extent after which the excess dosage results in slower rates. In the system considered in this study, the addition of the flocculants and coagulants did not impact the settling of the tails.

According to Lange et al. (1997) finer particles have a larger surface area and therefore are more likely to interact with the flocculant and coagulant resulting in larger aggregates that are better suited to particle bubble attachment that primarily is the main driver for particles that report to the concentrate (Lange et al. 1997). The lack of impact on settling tests noted in this study can be attributed to the flocculants and coagulants primarily interacting with the significant number of finer particles present in the system due to a grind size of 80% passing 75 µm.

The settling test was done on the tails of the flotation procedure, the study has been heavily focused on the change in the recovery of the concentrates of the experiments. If a closer look is taken at the tail’s recovery, it can be seen that the highest solid recovery of the tails can be seen under condition 1 in Table 4 of section 4.1 where no variable reagents were added, the value being 856.85 while the lowest tails recovery was under condition 4 with a value of 749.93. The relative change of solids reporting to the tails was 11.62 %. This indicated that the majority of the tailing’s composition remained as they were and probably the reason why the settling was not impacted. There is a possibility that a higher dosage of dewatering reagents would be required to see an impact on the settling on tailings.

7 Conclusions

The objective of this study was to investigate the effects of residual reagents from the dewatering thickening process on the flotation process. This was achieved by observing the impact on water recovery, solid recovery, copper recovery and grade, nickel recovery and grade, settling rate of the tails, and the impact of these reagents on the adsorption of flotation reagents on mineral surfaces.

The reagents were varied using the factorial design which produced the combinations to be tested. Table 4 under Section 4.1 shows all the conditions tested with condition 1 (None) being the base condition for comparison.

7.1 What is the Effect of Dewatering Agents on Key Flotation Outcomes?

When dewatering reagents and a combination thereof were added to the flotation system. It could be seen that all conditions relative to the base conditions resulted in the following outcomes:

- The water and solid recoveries increased.
- The copper recovery increased, and grade decreased.
- The nickel recovery increased slightly, and the grade decreased.

These outcomes were achieved due to the primary role that flocculants and coagulants play as reagents; they are used to agglomerate small particles into larger ones. In most cases the expected result of these would be larger particles that would be too heavy to report to the concentrate. However, in this case due to the experimental conditions having fine particle size of 80% passing 75 microns, these extremely fine particles tend to have less momentum resulting in slow recovery rates and poor particle-bubble attachment. When dewatering reagents are added to the system, they result in the agglomeration of the particle into a more optimum size that has better momentum resulting in faster recovery rates.

The copper and nickel recoveries remained unaffected despite the increase in solid recoveries. The final grade of both these metals decreased because the total solid recoveries increased but the individual metal recoveries did not increase indicating that there was more gangue reporting to the concentrate.

These results indicate that the introduction of dewatering reagents to the system is detrimental to the flotation system as the recoveries of the valuable metals did not increase but more gangue reported to the concentrate. This results in a greater need for separation downstream, which could result in an increase in the cost of operations. There would also be a need to evaluate the cost to treat the recycled water vs the decrease in grade.

7.2 How Do the Dewatering Reagents Affect Key Flotation Reagents?

The effect of the dewatering reagents on the flotation reagents was determined by measuring the concentration of the collector and depressant pre flotation and post flotation. It could be seen under all conditions compared to the base condition (which can be seen in Table 4 Section 4.1) that there was no impact on concentration readings of the pre flotation and post flotation. The concentration levels remained consistent, and the small differences were determined to be noise by the statistical analysis done under Section 5.

The key effect of the dewatering reagent in the system is the increase in particle size due to agglomeration of the smaller particles. This would result in the decrease of the surface area of the particles for the reagent to adsorb onto. From the results of the study, it could be seen that the reduction in the surface did not impact the adsorption of the reagents, for the depressant this could be attributed to depressants inability to achieve complete surface area coverage under flotation conditions. While for the collector the results obtained in terms of solid recovery in the concentrate and the concentration of the collectors remained constant, indicating that the decrease in surface area coverage was not significant enough to impact the adsorption of the collector. Additionally, there is no concrete proof that a relationship exists between collector surface area and adsorption.

This result can be seen as a neutral outcome, there is no negative impact on the adsorption of the reagent but also does not mean that the quantity of the current dosage can be decreased. This however applies to only the dosage levels that have been currently being tested.

7.3 Do the Dewatering Reagents Affect the Settling of Tailings?

The settling tests were done on the tailings from the float cell. The results obtained indicated that there was no impact seen on the settling rate when dewatering reagents and a combination thereof were added to the system compared to the base conditions.

This could be attributed to the dewatering reagents primarily interacting with the fine particles in the mill which report to the concentrate. While the settling tests were conducted on the tails of the flotation process. Furthermore, the solid content of the tail’s composition did not change significantly between the base condition and the most impacted testing condition.

8 Recommendation

From the results, discussion, and conclusions, the following recommendations can be put forward.

- Froth column studies to determine whether a greater impact was obtained from the flocculant and coagulant ionic properties or the agglomeration properties of the reagents, particularly on the stability of the froth.
- A different experimental path where recycled water is implemented in the flotation process should be explored.
- Adsorption studies on particles of different sizes to determine, how much of an impact particles size has on the adsorption of flotation reagents.
- A different methodology to test the settling of the particles over longer possible period, methods such as jar test.

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10 Appendices

10.1 Calibration Curve Graphs

CMC calibration curve adopted from Manenzhe (2018)

CHAPTER 4

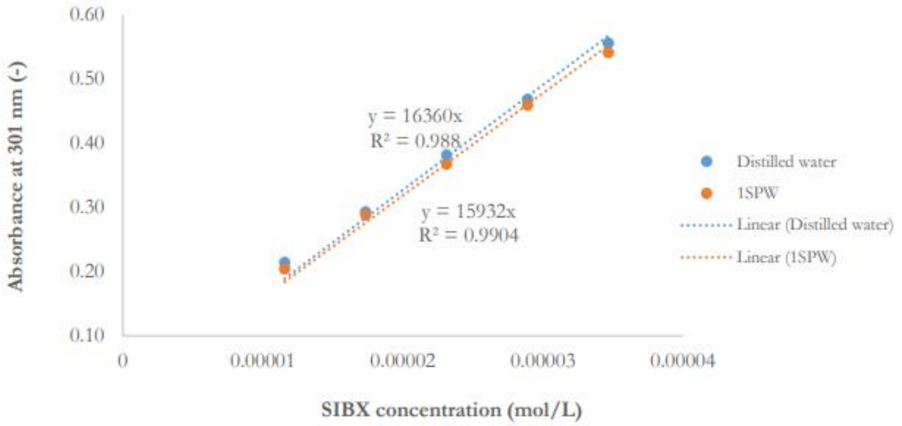
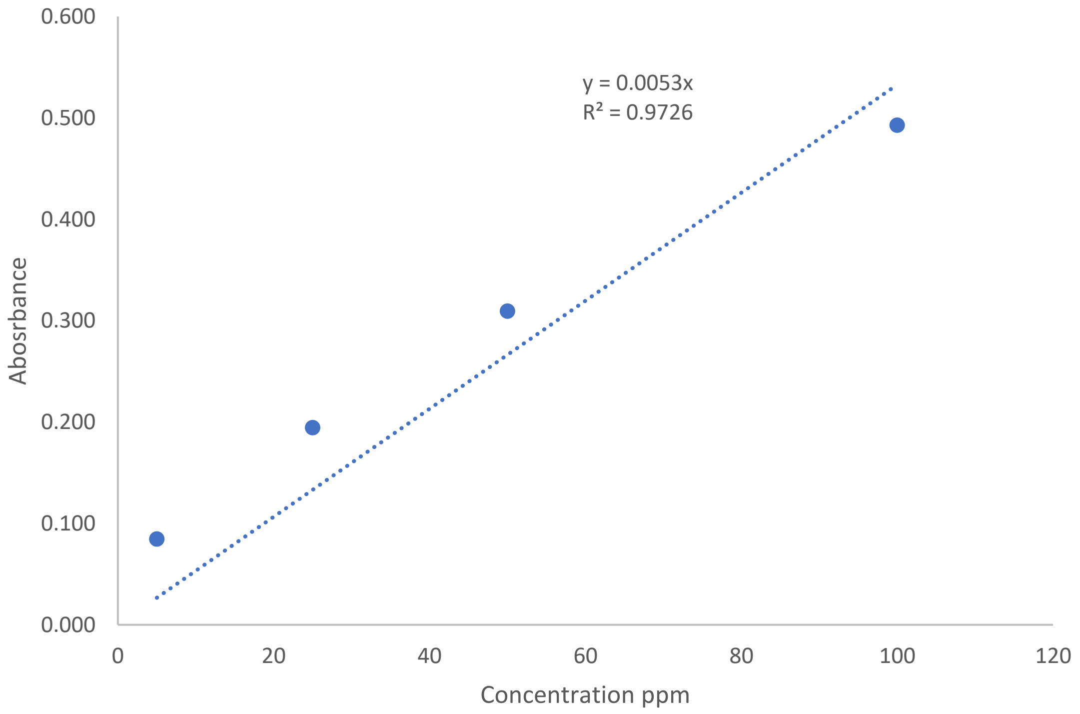


Figure 4.13: Calibration curves for distilled water and 1SPW for the SIBX concentration range of 2 to 6 ppm

SIBX calibration curve from experimental analysis



10.2 Flotation Mass Balance Results

RUN 1			WATER									SOLIDS								
Depressant	L	TIME		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR
		5	C1	270.720	270.720	252.020	252.020	6.908	261.370	13.223	9.350	C1	48.860	48.860	42.510	42.510	12.996	45.685	4.490	3.175
		9	C2	265.310	536.030	237.780	489.800	8.625	512.915	32.690	23.115	C2	28.780	77.640	27.870	70.380	9.351	74.010	5.134	3.630
Flocculant	L	15	C3	171.310	707.340	172.330	662.130	6.392	684.735	31.968	22.605	C3	15.770	93.410	16.050	86.430	7.472	89.920	4.936	3.490
		23	C4	129.380	836.720	135.500	797.630	4.672	817.175	27.641	19.545	C4	9.870	103.280	10.190	96.620	6.448	99.950	4.709	3.330
			F1									F1	16.700		16.610					
Coagulant	L		F2									F2	16.780		16.860					
			T1									T1	14.260		14.130					
			T2									T2	13.830		14.080					
												Tailings	848.560		865.140					
													856.850	99.950	0.896					
run 2			WATER									SOLIDS								
Depressant	H	TIME		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR
		5	C1	381.390	381.390	341.820	341.820	10.375	361.605	27.980	19.785	C1	67.520	67.520	57.490	57.490	14.855	62.505	7.092	5.015
		9	C2	274.320	655.710	281.080	622.900	5.004	639.305	23.200	16.405	C2	28.960	96.480	28.470	85.960	10.904	91.220	7.439	5.260
Flocculant	L	15	C3	177.180	832.890	173.000	795.900	4.441	814.395	26.156	18.495	C3	14.900	111.380	13.970	99.930	10.280	105.655	8.096	5.725
		23	C4	139.770	972.660	135.080	930.980	4.285	951.820	29.472	20.840	C4	9.690	121.070	8.860	108.790	10.143	114.930	8.683	6.140
			F1									F1	16.420		16.870					
Coagulant	L		F2									F2	17.230		17.070					
			T1									T1	13.810		13.490					
			T2									T2	14.140		13.040					
												Tailings	867.340		848.660					
run 3			WATER									SOLIDS								
Depressant	L	TIME		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR
		5	C1	459.710	459.710	501.860	501.860	-9.169	480.785	29.805	21.075	C1	83.530	83.530	94.810	94.810	-13.504	89.170	7.976	5.640
		9	C2	282.200	741.910	309.060	810.920	-9.302	776.415	48.797	34.505	C2	30.700	114.230	29.790	124.600	-9.078	119.415	7.333	5.185
Flocculant	L	15	C3	187.930	929.840	184.170	995.090	-7.017	962.465	46.139	32.625	C3	14.590	128.820	13.510	138.110	-7.212	133.465	6.569	4.645
		23	C4	131.200	1061.040	130.440	1125.530	-6.078	1093.285	45.601	32.245	C4	8.430	137.250	8.150	146.260	-6.565	141.755	6.371	4.505
			F1									F1	16.980		16.650					
Coagulant	H		F2									F2	16.900		16.790					
			T1									T1	13.550		13.130					
			T2									T2	13.680		13.190					
												Tailings	807.940		805.900					

run 4			WATER									SOLIDS								
Depressant	L	TIME		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR
		5	C1	754.760	754.760	734.530	734.530	2.680	744.645	14.305	10.115	C1	168.190	168.190	151.820	151.820	9.733	160.005	11.575	8.185
		9	C2	280.310	1035.070	360.380	1094.910	-5.781	1064.990	42.313	29.920	C2	22.760	190.950	29.510	181.330	5.038	186.140	6.802	4.810
Flocculant	H	15	C3	167.200	1202.270	191.550	1286.460	-7.003	1244.365	59.531	42.095	C3	10.550	201.500	12.090	193.420	4.010	197.460	5.713	4.040
		23	C4	145.160	1347.430	124.120	1410.580	-4.687	1379.005	44.654	31.575	C4	6.510	208.010	6.550	199.970	3.865	203.990	5.685	4.020
			F1									F1	16.630		16.720					
Coagulant	L		F2									F2	16.460		17.120					
			T1									T1	12.010		12.150					
			T2									T2	11.680		12.550					
												Tailings	740.710		759.140					
													749.925	203.990	0.786					

Run 5			WATER									SOLIDS								
Depressant	H	TIME		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR
		5	C1	688.940	688.940	731.390	731.390	-6.162	710.165	30.017	21.225	C1	144.040	144.040	156.760	156.760	-8.831	150.400	8.994	6.360
		9	C2	289.820	978.760	291.870	1023.260	-4.547	1001.010	31.466	22.250	C2	24.570	168.610	23.070	179.830	-6.654	174.220	7.934	5.610
Flocculant	H	15	C3	162.360	1141.120	169.080	1192.340	-4.489	1166.730	36.218	25.610	C3	10.880	179.490	10.560	190.390	-6.073	184.940	7.707	5.450
		23	C4	114.780	1255.900	110.160	1302.500	-3.710	1279.200	32.951	23.300	C4	6.650	186.140	6.040	196.430	-5.528	191.285	7.276	5.145
			F1									F1	16.680		16.860					
Coagulant	L		F2									F2	16.600		17.070					
			T1									T1	12.710		12.310					
			T2									T2	12.730		12.200					
												Tailings	760.800		752.840					

Run 6			WATER									SOLIDS								
Depressant	H	TIME		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR
		5	C1	482.170	482.170	492.260	492.260	-2.093	487.215	7.135	5.045	C1	83.390	83.390	84.420	84.420	-1.235	83.905	0.728	0.515
		9	C2	335.830	818.000	389.920	882.180	-7.846	850.090	45.382	32.090	C2	31.580	114.970	36.620	121.040	-5.280	118.005	4.292	3.035
Flocculant	L	15	C3	208.190	1026.190	241.490	1123.670	-9.499	1074.930	68.929	48.740	C3	15.380	130.350	16.970	138.010	-5.876	134.180	5.416	3.830
		23	C4	164.000	1190.190	167.210	1290.880	-8.460	1240.535	71.199	50.345	C4	10.020	140.370	9.700	147.710	-5.229	144.040	5.190	3.670
			F1									F1	16.800		16.640					
Coagulant	H		F2									F2	16.850		16.390					
			T1									T1	13.360		13.080					
			T2									T2	13.440		13.530					
												Tailings	807.010		812.640					

Run 7				WATER								SOLIDS								
Depressant	L	TIME		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR
		5	C1	367.350	367.350	338.810	338.810	7.769	353.080	20.181	14.270	C1	63.430	63.430	57.950	57.950	8.639	60.690	3.875	2.740
		9	C2	320.550	687.900	413.430	752.240	-9.353	720.070	45.495	32.170	C2	33.280	96.710	35.790	93.740	3.071	95.225	2.100	1.485
Flocculant	H	15	C3	203.020	890.920	200.380	952.620	-6.925	921.770	43.628	30.850	C3	16.960	113.670	17.560	111.300	2.085	112.485	1.676	1.185
		23	C4	163.260	1054.180	92.560	1045.180	0.854	1049.680	6.364	4.500	C4	11.340	125.010	10.370	121.670	2.672	123.340	2.362	1.670
			F1									F1	16.870		16.640					
Coagulant	H		F2									F2	16.840		16.410					
			T1									T1	13.700		13.550					
			T2									T2	13.800		13.600					
												Tailings	838.640		829.100					

Run 8				WATER								SOLIDS								
Depressant	H	TIME		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR
		5	C1	442.530	442.530	459.160	459.160	-3.758	450.845	11.759	8.315	C1	76.610	76.610	80.540	80.540	-5.130	78.575	2.779	1.965
		9	C2	398.790	841.320	374.280	833.440	0.937	837.380	5.572	3.940	C2	41.050	117.660	38.000	118.540	-0.748	118.100	0.622	0.440
Flocculant	H	15	C3	213.740	1055.060	237.440	1070.880	-1.499	1062.970	11.186	7.910	C3	16.760	134.420	18.170	136.710	-1.704	135.565	1.619	1.145
		23	C4	139.820	1194.880	163.600	1234.480	-3.314	1214.680	28.001	19.800	C4	9.370	143.790	10.650	147.360	-2.483	145.575	2.524	1.785
			F1									F1	16.860		16.780					
Coagulant	H		F2									F2	16.700		16.590					
			T1									T1	13.260		13.290					
			T2									T2	13.360		13.550					
												Tailings	819.830		812.490					

Run 9				WATER								SOLIDS								
Depressant	M	TIME		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR
		5	C1	313.040	313.040	302.980	302.980	3.214	308.010	7.113	5.030	C1	51.600	51.600	51.050	51.050	1.066	51.325	0.389	0.275
		9	C2	233.180	546.220	285.050	588.030	-7.654	567.125	29.564	20.905	C2	26.760	78.360	31.340	82.390	-5.143	80.375	2.850	2.015
Flocculant	M	15	C3	158.160	704.380	181.620	769.650	-9.266	737.015	46.153	32.635	C3	14.600	92.960	15.670	98.060	-5.486	95.510	3.606	2.550
		23	C4	118.580	822.960	135.080	904.730	-9.936	863.845	57.820	40.885	C4	9.130	102.090	9.510	107.570	-5.368	104.830	3.875	2.740
			F1									F1	17.100		16.530					
Coagulant	M		F2									F2	17.090		16.590					
			T1									T1	13.790		13.870					
			T2									T2	13.830		13.960					
												Tailings	860.160		821.800					

Run 10			WATER									SOLIDS								
Depressant	M	TIME		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR		0.000	R0 CUM	1.000	R1 CUM	error	AVERAGE	STD DEV	STD ERROR
		5	C1	292.260	292.260	330.240	330.240	-12.995	311.250	26.856	18.990	C1	50.600	50.600	56.760	56.760	-12.174	53.680	4.356	3.080
		9	C2	284.780	577.040	283.420	613.660	-6.346	595.350	25.894	18.310	C2	29.920	80.520	30.850	87.610	-8.805	84.065	5.013	3.545
Flocculant	M	15	C3	168.870	745.910	199.330	812.990	-8.993	779.450	47.433	33.540	C3	15.020	95.540	16.940	104.550	-9.431	100.045	6.371	4.505
		23	C4	134.480	880.390	140.160	953.150	-8.265	916.770	51.449	36.380	C4	9.720	105.260	10.070	114.620	-8.892	109.940	6.619	4.680
			F1									F1	16.710		16.720					
Coagulant	M		F2									F2	16.830		16.520					
			T1									T1	13.850		13.680					
			T2									T2	13.980		14.130					
												Tailings	852.140		840.720					

10.3 Copper Assay Results

Run 1	Cu %		Cu mass (g)		cumulative mass		Copper grade%		Copper recovery %		T Cu feed		T Cu Cc+Tt		average cu grade	std dev	std error	average copper recovery	std dev	std error
	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.211	0.231	10.314	9.816	10.314	9.816	0.211	0.231	0.520	0.518	19.001	21.697	19.841	18.949	0.221	0.014	0.010	0.519	0.001	0.001
C2	0.045	0.046	1.289	1.276	11.604	11.092	0.149	0.158	0.585	0.585	Mass balance				0.154	0.006	0.004	0.585	0.000	0.000
C3	0.029	0.031	0.464	0.496	12.067	11.588	0.129	0.134	0.608	0.612	1.000		1.100		0.132	0.003	0.002	0.610	0.002	0.002
C4	0.028	0.025	0.278	0.259	12.346	11.847	0.120	0.123	0.622	0.625	1.044		0.873		0.121	0.002	0.002	0.624	0.002	0.001
F1	0.020	0.020	0.336	0.336																
F2	0.017	0.022	0.292	0.374																
T1	0.008	0.007	0.116	0.099																
T2	0.009	0.009	0.124	0.125																

Run 2	Cu %		Cu mass (g)		cumulative mass		Copper grade%		Copper recovery %		T Cu feed		T Cu Cc+Tt		average cu grade	std dev	std error	average copper recovery	std dev	std error
	2.000	2.100	2.000	2.100	2.000	2.100	2.000	2.100	2.000	2.100	2.000	2.100	2.000	2.100						
C1	0.168	0.176	11.310	10.130	11.310	10.130	0.168	0.176	0.613	0.583	18.008	17.305	18.437	17.384	0.172	0.006	0.004	0.598	0.022	0.015
C2	0.038	0.040	1.098	1.127	12.407	11.257	0.129	0.131	0.673	0.648	Mass balance				0.130	0.002	0.001	0.660	0.018	0.013
C3	0.027	0.029	0.401	0.404	12.808	11.661	0.115	0.117	0.695	0.671	2.000		2.100		0.116	0.001	0.001	0.683	0.017	0.012
C4	0.022	0.024	0.212	0.209	13.020	11.870	0.108	0.109	0.706	0.683	1.024		1.005		0.108	0.001	0.001	0.695	0.017	0.012
F1	0.017	0.016	0.277	0.273																
F2	0.017	0.018	0.300	0.304																
T1	0.005	0.006	0.073	0.086																
T2	0.007	0.006	0.096	0.081																

Run 3	Cu %		Cu mass (g)		cumulative mass		Copper grade%		Copper recovery %		T Cu feed		T Cu Cc+Tt		average cu grade	std dev	std error	average copper recovery	std dev	std error
	3.000	3.100	3.000	3.100	3.000	3.100	3.000	3.100	3.000	3.100	3.000	3.100	3.000	3.100						
C1	0.141	0.102	11.761	9.690	11.761	9.690	0.141	0.102	0.643	0.600	18.013	18.518	18.278	16.141	0.122	0.027	0.019	0.622	0.031	0.022
C2	0.036	0.032	1.102	0.959	12.863	10.649	0.113	0.085	0.704	0.660	Mass balance				0.099	0.019	0.014	0.682	0.031	0.022
C3	0.025	0.025	0.365	0.338	13.228	10.987	0.103	0.080	0.724	0.681	1.000		1.100		0.091	0.016	0.012	0.702	0.030	0.022
C4	0.025	0.025	0.207	0.203	13.434	11.190	0.098	0.077	0.735	0.693	1.015		0.872		0.087	0.015	0.011	0.714	0.030	0.021
F1	0.018	0.018	0.297	0.300																
F2	0.018	0.019	0.309	0.312																
T1	0.005	0.005	0.072	0.060																
T2	0.006	0.007	0.086	0.096																

Run 4	Cu %		Cu mass (g)		cumulative mass		Copper grade%		Copper recovery %		T Cu feed		T Cu Cc+Tt		average cu grade	std dev	std error	average copper recovery	std dev	std error
	4.000	4.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.076	0.085	12.816	12.844	12.816	12.844	0.076	0.085	0.675	0.668	16.943	18.674	18.993	19.225	0.080	0.006	0.004	0.671	0.005	0.003
C2	0.037	0.033	0.844	0.977	13.660	13.821	0.072	0.076	0.719	0.719	Mass balance				0.074	0.003	0.002	0.719	0.000	0.000
C3	0.028	0.027	0.292	0.331	13.953	14.152	0.069	0.073	0.735	0.736	1.000		1.100		0.071	0.003	0.002	0.735	0.001	0.001
C4	0.029	0.027	0.186	0.174	14.139	14.326	0.068	0.072	0.744	0.745	1.121		1.029		0.070	0.003	0.002	0.745	0.001	0.000
F1	0.017	0.019	0.279	0.321																
F2	0.017	0.018	0.278	0.300																
T1	0.006	0.007	0.074	0.083																
T2	0.007	0.006	0.076	0.072																

Run 5	Cu %		Cu mass (g)		cumulative mass		Copper grade%		Copper recovery %		T Cu feed		T Cu Cc+Tt		average cu grade	std dev	std error	average copper recovery	std dev	std error
	5.000	5.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.071	0.071	10.241	11.114	10.241	11.114	0.071	0.071	0.593	0.632	16.694	17.988	17.260	17.577	0.071	0.000	0.000	0.613	0.028	0.019
C2	0.035	0.039	0.862	0.893	11.104	12.007	0.066	0.067	0.643	0.683	Mass balance				0.066	0.001	0.000	0.663	0.028	0.020
C3	0.028	0.029	0.304	0.304	11.407	12.311	0.064	0.065	0.661	0.700	1.000		1.100		0.064	0.001	0.001	0.681	0.028	0.020
C4	0.029	0.029	0.192	0.174	11.599	12.485	0.062	0.064	0.672	0.710	1.034		0.977		0.063	0.001	0.001	0.691	0.027	0.019
F1	0.018	0.017	0.300	0.293																
F2	0.015	0.018	0.252	0.312																
T1	0.007	0.007	0.084	0.084																
T2	0.008	0.006	0.099	0.077																

Run 6	Cu %		Cu mass (g)		cumulative mass		Copper grade%		Copper recovery %		T Cu feed		T Cu Cc+Tt		average cu grade	std dev	std error	average copper recovery	std dev	std error
	6.000	6.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.129	0.129	10.741	10.865	10.741	10.865	0.129	0.129	0.618	0.642	18.393	20.604	17.370	16.922	0.129	0.000	0.000	0.630	0.017	0.012
C2	0.031	0.027	0.992	0.996	11.732	11.861	0.102	0.098	0.675	0.701	Mass balance				0.100	0.003	0.002	0.688	0.018	0.013
C3	0.028	0.021	0.423	0.355	12.155	12.216	0.093	0.089	0.700	0.722	1.000		1.100		0.091	0.003	0.002	0.711	0.016	0.011
C4	0.021	0.022	0.212	0.216	12.368	12.432	0.088	0.084	0.712	0.735	0.944		0.821		0.086	0.003	0.002	0.723	0.016	0.011
F1	0.019	0.022	0.323	0.371																
F2	0.017	0.018	0.292	0.297																
T1	0.005	0.007	0.072	0.088																
T2	0.007	0.004	0.089	0.054																

Run 7	Cu %		Cu mass (g)		cumulative mass		Copper grade%		Copper recovery %		T Cu feed		T Cu Cc+Tt		average cu grade	std dev	std error	average copper recovery	std dev	std error
	7.000	7.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.180	0.170	11.398	9.828	11.398	9.828	0.180	0.170	0.614	0.620	19.524	16.125	18.570	15.847	0.175	0.007	0.005	0.617	0.005	0.003
C2	0.037	0.031	1.235	1.106	12.633	10.934	0.131	0.117	0.680	0.690	Mass balance				0.124	0.010	0.007	0.685	0.007	0.005
C3	0.028	0.024	0.468	0.421	13.101	11.356	0.115	0.102	0.705	0.717	1.000		1.100		0.109	0.009	0.007	0.711	0.008	0.006
C4	0.020	0.020	0.229	0.211	13.330	11.566	0.107	0.095	0.718	0.730	0.951		0.983		0.101	0.008	0.006	0.724	0.009	0.006
F1	0.022	0.016	0.373	0.268																
F2	0.016	0.016	0.269	0.259																
T1	0.006	0.005	0.079	0.072																
T2	0.006	0.005	0.087	0.064																

Run 8	Cu %		Cu mass (g)		cumulative mass		Copper grade%		Copper recovery %		T Cu feed		T Cu Cc+Tt		average cu grade	std dev	std error	average copper recovery	std dev	std error
	8.000	8.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.147	0.156	11.269	12.540	11.269	12.540	0.147	0.156	0.687	0.621	19.145	19.381	16.395	20.201	0.151	0.006	0.004	0.654	0.047	0.033
C2	0.028	0.032	1.154	1.205	12.423	13.745	0.106	0.116	0.758	0.680	Mass balance				0.111	0.007	0.005	0.719	0.055	0.039
C3	0.023	0.022	0.384	0.394	12.807	14.139	0.095	0.103	0.781	0.700	1.000		1.100		0.099	0.006	0.004	0.741	0.057	0.041
C4	0.026	0.022	0.245	0.229	13.051	14.368	0.091	0.098	0.796	0.711	0.856		1.042		0.094	0.005	0.003	0.754	0.060	0.042
F1	0.020	0.021	0.332	0.356																
F2	0.018	0.017	0.296	0.279																
T1	0.003	0.006	0.041	0.085																
T2	0.005	0.008	0.064	0.102																

Run 9	Cu %		Cu mass (g)		cumulative mass		Copper grade%		Copper recovery %		T Cu feed		T Cu Cc+Tt		average cu grade	std dev	std error	average copper recovery	std dev	std error
	9.000	9.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.185	0.208	9.561	10.598	9.561	10.598	0.185	0.208	0.593	0.619	18.638	18.172	16.122	17.120	0.196	0.016	0.011	0.606	0.018	0.013
C2	0.039	0.038	1.038	1.194	10.600	11.792	0.135	0.143	0.657	0.689	Mass balance				0.139	0.006	0.004	0.673	0.022	0.016
C3	0.031	0.025	0.456	0.398	11.055	12.190	0.119	0.124	0.686	0.712	1.000		1.100		0.122	0.004	0.003	0.699	0.019	0.013
C4	0.025	0.023	0.228	0.215	11.284	12.405	0.111	0.115	0.700	0.725	0.865		0.942		0.113	0.003	0.002	0.712	0.017	0.012
F1	0.019	0.017	0.330	0.279																
F2	0.017	0.020	0.292	0.328																
T1	0.005	0.006	0.063	0.076																
T2	0.006	0.006	0.087	0.078																

Run 10	Cu %		Cu mass (g)		cumulative mass		Copper grade%		Copper recovery %		T Cu feed		T Cu Cc+Tt		average cu grade	std dev	std error	average copper recovery	std dev	std error
	10.000	10.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.201	0.185	10.171	10.518	10.171	10.518	0.201	0.185	0.609	0.586	17.319	18.702	16.711	17.950	0.193	0.011	0.008	0.597	0.016	0.011
C2	0.039	0.039	1.152	1.194	11.323	11.712	0.141	0.134	0.678	0.652	Mass balance				0.137	0.005	0.003	0.665	0.018	0.013
C3	0.028	0.026	0.415	0.447	11.737	12.159	0.123	0.116	0.702	0.677	1.000		1.100		0.120	0.005	0.003	0.690	0.018	0.012
C4	0.023	0.023	0.223	0.233	11.960	12.391	0.114	0.108	0.716	0.690	0.965		0.960		0.111	0.004	0.003	0.703	0.018	0.013
F1	0.015	0.019	0.244	0.321																
F2	0.019	0.018	0.327	0.291																
T1	0.005	0.005	0.075	0.073																
T2	0.005	0.008	0.075	0.106																

10.4 Nickel Assay Results

Run 1	Ni %		Ni mass (g)		cumulative mass		Ni grade%		Ni recovery %		T Ni feed		T Ni Cc+Tt		average Ni grade	std dev	std error	average Ni recovery	std dev	std error
	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.679	0.814	33.152	34.603	33.152	34.603	0.679	0.814	0.155	0.155	230.804	228.074	214.356	223.806	0.746	0.096	0.068	0.155	0.000	0.000
C2	0.357	0.369	10.272	10.287	43.423	44.890	0.559	0.638	0.203	0.201	Mass balance				0.599	0.056	0.039	0.202	0.001	0.001
C3	0.281	0.284	4.431	4.550	47.854	49.440	0.512	0.572	0.223	0.221	1.000		1.100		0.542	0.042	0.030	0.222	0.002	0.001
C4	0.247	0.253	2.437	2.575	50.291	52.015	0.487	0.538	0.235	0.232	0.929		0.981		0.513	0.036	0.026	0.234	0.002	0.001
F1	0.232	0.223	3.868	3.699																
F2	0.224	0.223	3.757	3.760																
T1	0.193	0.189	2.748	2.675																
T2	0.182	0.195	2.512	2.750																

Run 2	Ni %		Ni mass (g)		cumulative mass		Ni grade%		Ni recovery %		T Ni feed		T Ni Cc+Tt		average Ni grade	std dev	std error	average Ni recovery	std dev	std error
	2.000	2.100	2.000	2.100	2.000	2.100	2.000	2.100	2.000	2.100	2.000	2.100	2.000	2.100						
C1	0.608	0.660	41.066	37.961	41.066	37.961	0.608	0.660	0.182	0.172	235.255	228.828	225.108	221.194	0.634	0.037	0.026	0.177	0.008	0.005
C2	0.335	0.344	9.696	9.805	50.761	47.766	0.526	0.556	0.225	0.216	Mass balance				0.541	0.021	0.015	0.221	0.007	0.005
C3	0.269	0.283	4.005	3.958	54.767	51.723	0.492	0.518	0.243	0.234	1.000		1.100		0.505	0.018	0.013	0.239	0.007	0.005
C4	0.251	0.256	2.429	2.266	57.196	53.989	0.472	0.496	0.254	0.244	0.957		0.967		0.484	0.017	0.012	0.249	0.007	0.005
F1	0.231	0.223	3.785	3.755																
F2	0.218	0.227	3.749	3.875																
T1	0.189	0.189	2.606	2.547																
T2	0.186	0.193	2.636	2.521																

Run 3	Ni %		Ni mass (g)		cumulative mass		Ni grade%		Ni recovery %		T Ni feed		T Ni Cc+Tt		average Ni grade	std dev	std error	average Ni recovery	std dev	std error
	3.000	3.100	3.000	3.100	3.000	3.100	3.000	3.100	3.000	3.100	3.000	3.100	3.000	3.100						
C1	0.526	0.465	43.962	44.087	43.962	44.087	0.526	0.465	0.200	0.208	229.990	228.593	220.043	212.441	0.496	0.043	0.031	0.204	0.005	0.004
C2	0.327	0.310	10.033	9.226	53.995	53.313	0.473	0.428	0.245	0.251	Mass balance				0.450	0.032	0.022	0.248	0.004	0.003
C3	0.268	0.269	3.912	3.640	57.906	56.952	0.450	0.412	0.263	0.268	1.000		1.100		0.431	0.026	0.019	0.266	0.003	0.002
C4	0.251	0.259	2.118	2.111	60.025	59.063	0.437	0.404	0.273	0.278	0.957		0.929		0.421	0.024	0.017	0.275	0.004	0.003
F1	0.232	0.227	3.933	3.781																
F2	0.226	0.225	3.811	3.773																
T1	0.193	0.186	2.618	2.436																
T2	0.190	0.183	2.599	2.415																

Run 4	Ni %		Ni mass (g)		cumulative mass		Ni grade%		Ni recovery %		T Ni feed		T Ni Cc+Tt		average Ni grade	std dev	std error	average Ni recovery	std dev	std error
	4.000	4.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.372	0.378	62.583	57.358	62.583	57.358	0.372	0.378	0.288	0.266	223.372	228.717	217.041	215.457	0.375	0.004	0.003	0.277	0.016	0.011
C2	0.331	0.303	7.531	8.927	70.115	66.284	0.367	0.366	0.323	0.308	Mass balance				0.366	0.001	0.001	0.315	0.011	0.008
C3	0.276	0.269	2.910	3.251	73.024	69.535	0.362	0.360	0.336	0.323	1.000		1.100		0.361	0.002	0.001	0.330	0.010	0.007
C4	0.265	0.259	1.723	1.695	74.748	71.231	0.359	0.356	0.344	0.331	0.972		0.942		0.358	0.002	0.002	0.337	0.010	0.007
F1	0.224	0.224	3.722	3.750																
F2	0.221	0.225	3.629	3.855																
T1	0.187	0.184	2.247	2.234																
T2	0.185	0.184	2.163	2.310																

Run 5	Ni %		Ni mass (g)		cumulative mass		Ni grade%		Ni recovery %		T Ni feed		T Ni Cc+Tt		average Ni grade	std dev	std error	average Ni recovery	std dev	std error
	5.000	5.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.393	0.391	56.550	61.356	56.550	61.356	0.393	0.391	0.256	0.277	236.028	225.425	220.781	221.662	0.392	0.001	0.001	0.266	0.015	0.010
C2	0.322	0.318	7.916	7.329	64.467	68.685	0.382	0.382	0.292	0.310	Mass balance				0.382	0.000	0.000	0.301	0.013	0.009
C3	0.277	0.284	3.014	2.996	67.480	71.681	0.376	0.376	0.306	0.323	1.000		1.100		0.376	0.000	0.000	0.315	0.013	0.009
C4	0.270	0.269	1.792	1.624	69.272	73.305	0.372	0.373	0.314	0.331	0.935		0.983		0.373	0.001	0.001	0.322	0.012	0.008
F1	0.233	0.225	3.881	3.800																
F2	0.237	0.222	3.929	3.790																
T1	0.192	0.189	2.444	2.324																
T2	0.193	0.193	2.458	2.353																

Run 6	Ni %		Ni mass (g)		cumulative mass		Ni grade%		Ni recovery %		T Ni feed		T Ni Cc+Tt		average Ni grade	std dev	std error	average Ni recovery	std dev	std error
	6.000	6.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.521	0.527	43.463	44.447	43.463	44.447	0.521	0.527	0.202	0.200	227.417	233.068	215.609	222.145	0.524	0.004	0.003	0.201	0.001	0.001
C2	0.313	0.292	9.885	10.693	53.347	55.140	0.464	0.456	0.247	0.248	Mass balance				0.460	0.006	0.004	0.248	0.001	0.000
C3	0.272	0.265	4.186	4.494	57.534	59.634	0.441	0.432	0.267	0.268	1.000		1.100		0.437	0.007	0.005	0.268	0.001	0.001
C4	0.248	0.246	2.486	2.382	60.020	62.016	0.428	0.420	0.278	0.279	0.948		0.953		0.424	0.005	0.004	0.279	0.001	0.000
F1	0.227	0.229	3.812	3.809																
F2	0.224	0.228	3.781	3.739																
T1	0.187	0.191	2.493	2.494																
T2	0.187	0.191	2.508	2.583																

Run 7	Ni %		Ni mass (g)		cumulative mass		Ni grade%		Ni recovery %		T Ni feed		T Ni Cc+Tt		average Ni grade	std dev	std error	average Ni recovery	std dev	std error
	7.000	7.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.627	0.654	39.752	37.899	39.752	37.899	0.627	0.654	0.182	0.174	227.263	228.783	217.836	217.827	0.640	0.019	0.014	0.178	0.006	0.004
C2	0.323	0.324	10.736	11.607	50.488	49.506	0.522	0.528	0.232	0.227	Mass balance				0.525	0.004	0.003	0.230	0.003	0.002
C3	0.265	0.267	4.489	4.683	54.977	54.189	0.484	0.487	0.252	0.249	1.000		1.100		0.485	0.002	0.002	0.251	0.003	0.002
C4	0.250	0.244	2.840	2.534	57.817	56.724	0.462	0.466	0.265	0.260	0.959		0.952		0.464	0.003	0.002	0.263	0.004	0.003
F1	0.215	0.226	3.624	3.752																
F2	0.229	0.227	3.851	3.727																
T1	0.188	0.190	2.570	2.572																
T2	0.182	0.187	2.510	2.536																

Run 8	Ni %		Ni mass (g)		cumulative mass		Ni grade%		Ni recovery %		T Ni feed		T Ni Cc+Tt		average Ni grade	std dev	std error	average Ni recovery	std dev	std error
	8.000	8.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.565	0.570	43.269	45.868	43.269	45.868	0.565	0.570	0.200	0.207	231.174	231.146	215.968	221.223	0.567	0.003	0.002	0.204	0.005	0.003
C2	0.292	0.291	11.999	11.066	55.268	56.933	0.470	0.480	0.256	0.257	Mass balance				0.475	0.007	0.005	0.257	0.001	0.001
C3	0.254	0.248	4.259	4.508	59.527	61.441	0.443	0.449	0.276	0.278	1.000		1.100		0.446	0.005	0.003	0.277	0.001	0.001
C4	0.237	0.238	2.218	2.534	61.745	63.975	0.429	0.434	0.286	0.289	0.934		0.957		0.432	0.003	0.002	0.288	0.002	0.002
F1	0.219	0.236	3.689	3.958																
F2	0.233	0.217	3.888	3.605																
T1	0.186	0.190	2.468	2.522																
T2	0.178	0.185	2.382	2.505																

Run 9	Ni %		Ni mass (g)		cumulative mass		Ni grade%		Ni recovery %		T Ni feed		T Ni Cc+Tt		average Ni grade	std dev	std error	average Ni recovery	std dev	std error
	9.000	9.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.694	0.715	35.826	36.480	35.826	36.480	0.694	0.715	0.165	0.170	231.489	224.555	216.721	214.566	0.704	0.014	0.010	0.168	0.003	0.002
C2	0.341	0.339	9.128	10.618	44.954	47.098	0.574	0.572	0.207	0.220	Mass balance				0.573	0.001	0.001	0.213	0.009	0.006
C3	0.273	0.280	3.990	4.392	48.944	51.491	0.527	0.525	0.226	0.240	1.000		1.100		0.526	0.001	0.001	0.233	0.010	0.007
C4	0.256	0.267	2.339	2.538	51.283	54.029	0.502	0.502	0.237	0.252	0.936		0.956		0.502	0.000	0.000	0.244	0.011	0.008
F1	0.233	0.229	3.984	3.787																
F2	0.219	0.224	3.744	3.723																
T1	0.187	0.190	2.576	2.630																
T2	0.186	0.188	2.571	2.629																

Run 10	Ni %		Ni mass (g)		cumulative mass		Ni grade%		Ni recovery %		T Ni feed		T Ni Cc+Tt		average Ni grade	std dev	std error	average Ni recovery	std dev	std error
	10.000	10.100			1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100	1.000	1.100						
C1	0.700	0.665	35.435	37.768	35.435	37.768	0.700	0.665	0.161	0.172	226.625	226.655	219.523	219.507	0.683	0.025	0.017	0.167	0.008	0.005
C2	0.340	0.344	10.167	10.606	45.602	48.374	0.566	0.552	0.208	0.220	Mass balance				0.559	0.010	0.007	0.214	0.009	0.006
C3	0.276	0.269	4.150	4.557	49.752	52.931	0.521	0.506	0.227	0.241	1.000		1.100		0.514	0.010	0.007	0.234	0.010	0.007
C4	0.256	0.249	2.488	2.510	52.240	55.442	0.496	0.484	0.238	0.253	0.969		0.968		0.490	0.009	0.006	0.245	0.010	0.007
F1	0.223	0.229	3.730	3.821																
F2	0.222	0.218	3.731	3.593																
T1	0.190	0.190	2.630	2.601																
T2	0.190	0.188	2.660	2.652																

10.5 CMC Concentration Results

Run conditions		Average feed concentration	std dev	std error	% Error
None	Run 1	19.104	0.645	0.456	4.662%
D	Run 2	26.965	0.645	0.456	-3.440%
C	Run 3	15.943	4.936	3.491	35.922%
F	Run 4	14.450	3.091	2.186	-17.821%
DC	Run 5	22.091	0.778	0.550	-5.109%
DF	Run 6	20.597	2.846	2.013	-21.658%
CF	Run 7	19.623	0.712	0.503	-5.263%
DCF	Run 8	38.160	2.557	1.808	9.048%
Mid-Point		42.862	1.433	0.717	
DCFM	Run 9	43.679	1.156	0.818	3.675%
DCFM	Run 10	42.044	1.468	1.038	-5.061%
		Average tails concentration	std dev	std error	% Error
None	Run 1	25.943	2.357	1.667	12.073%
D	Run 2	19.937	1.067	0.755	7.295%
C	Run 3	15.126	0.400	0.283	-3.814%
F	Run 4	19.513	0.245	0.173	1.757%
DC	Run 5	24.874	3.335	2.358	-20.950%
DF	Run 6	17.689	0.467	0.330	-3.804%
CF	Run 7	24.182	1.823	1.289	10.123%
DCF	Run 8	40.613	2.068	1.462	6.951%
Mid-Point		31.360	2.864	1.432	
DCFM	Run 9	30.676	4.603	3.255	19.184%
DCFM	Run 10	32.044	1.245	0.881	5.349%

10.6 SIBX Concentration Results

		Average feed concentration	std dev	std error	% Error
None	Run 1	0.420	0.054	0.038	16.555%
D	Run 2	0.406	0.008	0.006	-2.752%
C	Run 3	0.425	0.073	0.051	21.594%
F	Run 4	0.475	0.023	0.017	-7.200%
DC	Run 5	0.498	0.025	0.018	6.880%
DF	Run 6	0.469	0.021	0.015	6.068%
CF	Run 7	0.472	0.035	0.025	-11.218%
DCF	Run 8	0.477	0.012	0.008	3.401%
Mid-Point		0.448	0.044	0.022	
DCFM	Run 9	0.424	0.039	0.028	12.195%
DCFM	Run 10	0.473	0.042	0.030	-13.536%
		Average tails concentration	std dev	std error	% Error
None	Run 1	0.362	0.023	0.017	8.724%
D	Run 2	0.433	0.078	0.055	22.682%
C	Run 3	0.352	0.022	0.016	8.652%
F	Run 4	0.404	0.019	0.013	-6.897%
DC	Run 5	0.389	0.010	0.007	3.549%
DF	Run 6	0.410	0.013	0.009	4.380%
CF	Run 7	0.469	0.045	0.032	12.821%
DCF	Run 8	0.445	0.045	0.032	13.462%
Mid-Point		0.398	0.040	0.020	
DCFM	Run 9	0.370	0.024	0.017	-9.532%
DCFM	Run 10	0.426	0.033	0.023	-11.533%

10.7 Settling Test Results

run	rate	average	std dev	std error	error	reagents
1.0	-182.000	-180.000	2.828	2.000	2.198%	None
1.1	-178.000					
2.0	-196.700	-202.950	8.839	6.250	-6.355%	D
2.1	-209.200					
3.0	-131.400	-148.350	23.971	16.950	-25.799%	C
3.1	-165.300					
4.0	-166.700	-162.250	6.293	4.450	5.339%	F
4.1	-157.800					
5.0	-173.900	-164.650	13.081	9.250	10.638%	D C
5.1	-155.400					
6.0	-148.200	-164.400	22.910	16.200	-21.862%	D F
6.1	-180.600					
7.0	-181.300	-174.550	9.546	6.750	7.446%	C F
7.1	-167.800					
8.0	-134.000	-138.750	6.718	4.750	-7.090%	D C F
8.1	-143.500					
9.0	-159.300	-170.150	15.344	10.850	0.000%	Mid-point
9.1	-181.000					
10.0	-181.000	-170.150	15.344	10.850	0.000%	M D C F
10.1	-159.300					