

# Reducing the Environmental Impact of the Flotation of Sphalerite: An Alternative Reagent Scheme

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Daniel Hamilton

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Supervisor: Dr. I. Nirdosh

Department of Chemical Engineering

Lakehead University  
955 Oliver Road, Thunder Bay, ON

P7B 5E1

# Abstract

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*N*-Hydrocinnamoyl-*N*-phenylhydroxylamine (HCNPHA) was found to float sphalerite without activation by copper sulphate. However, concomitant flotation of sulphidic gangue minerals, such as pyrite, significantly reduced the grade. Adding 200 g/t of copper sulphate, which is approximately 80% reduction in the amount consumed in the xanthate reagent scheme currently used in the industry, improved the grade and the recovery. Batch flotation tests were conducted using 350 g and 1 kg ore samples to optimize the flotation variables (2.5 Lpm air flow rate; 200 g/t collector dosage; 80% passing through 63  $\mu\text{m}$  particle size). Carboxymethylcellulose (CMC) and sodium silicate were added to suppress sulphidic and non-sulphidic gangue minerals. This improved the grade of sphalerite from 45.5% to 60.1%. CMC also helped regulate the frothing characteristics of HCNPHA, minimizing entrainment of gangue in the float concentrate (froth). Flotation kinetics studies confirmed the improved selectivity using the auxiliary chemicals CMC and sodium silicate. The selectivity index (SI) for sphalerite with respect to pyrite increased from 0.88 to 2.35 upon the addition of copper sulphate, and for non-sulphidic gangue it improved from 1.93 to 6.29 on adding CMC. With the combination of all auxiliary chemicals studied, the SI for sphalerite with respect to pyrite and non-sulphidic gangue further improved to 2.58 and 8.01, respectively. By using a drastically reduced amount of copper sulphate and adding two environmentally friendly auxiliary chemicals (CMC and sodium silicate), the mineral processing industry has the potential for savings of over \$15 million/year.

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# Nomenclature

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CMC: carboxymethylcellulose

Ga: galena

GR: grade ratio

k: first order rate constant

$K_m$ : modified rate constant

NSG: non-sulphidic gangue

PAX: potassium amyl xanthate

Pb-R: lead rougher

PS: polysaccharide

Py: pyrite

$R_\infty$ : maximum recovery

$R_t$ : cumulative recovery at time  $t$

SI: selectivity index

Sp: sphalerite

Zn-R: zinc rougher

$\Phi$ : time correction factor

# Section 1.0 Introduction

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## **1.0 Introduction**

This research explores an alternative reagent scheme to the xanthate reagent scheme currently used by the industry for the flotation of sphalerite. To better understand the concepts discussed in the following sections an introduction to the terms, concepts and principles of froth flotation need to be provided.

### **1.1 Froth Flotation**

Froth flotation is a separation process for the beneficiation of low grade ores. Flotation separates minerals that are suspended in slurry by facilitating attachment to air bubbles to provide selective levitation of the solid particles. Minerals can be classified into non-polar and polar types. Non-polar minerals are difficult to hydrate, have weak molecular bonds and are to a degree hydrophobic (water hating) whereas polar minerals are easily hydrated, strong covalent or ionic bonding and are hydrophilic (water loving) (Crozier, 1992). Flotation can be classified into *bulk flotation* and *differential flotation*. Bulk flotation is a general term describing all rougher or scavenger flotation's where a singular valuable mineral is separated from the gangue (non-valuable) minerals in a one step process. Differential flotation involves the separation of complex ores. It refers to removing valuable minerals from a poly-mineral ore using a combination of collectors, suppressants, modifiers, frothers and activators. These reagents play specific roles but all are generally interfacial surface tension modifiers, surface chemistry modifiers.

Froth flotation is a complex system that is not fully understood, however, it can be viewed as made up of parts; a bubble generation zone where minerals are selectively rendered

hydrophobic to aid in adhesion to air bubbles and a frothing zone where the bubbles coagulate into a loose froth. These water rich zones are known as the pulp. The final zone is where the air bubbles dominate and water returns to the pulp as the froth condenses.

## **1.2 Collectors**

Collectors are the most important group of speciality mineral processing chemicals. According to Urbina (2003), collectors are amphiphilic compounds that affect the wettability of solid surfaces through adsorption at the solid-liquid interface. These organic surfactant chemicals selectively adsorb onto the valuable mineral(s) surface and render it hydrophobic helping to facilitate bubble attachment. Flotation of sulphide minerals usually employ thiol collectors whereas non-sulphide minerals generally use fatty acids, amine, quaternary ammonium compounds, etc. (Wills, 1997, Crozier, 1992). Collectors contain both a non-polar hydrocarbon group and a polar group which adsorbs on the mineral surface leaving the non-polar group in contact with the bulk solution thus providing a hydrophobic film. Selectivity of a collector is dependant on functional groups and chain length (Bradshaw, 2005). Wills (1997) classified them into two types; ionising collectors (dissociate into ions in water) and non-ionising collectors (insoluble in water). It should be noted that the amount of collector added to the system should be carefully monitored, and ideally only enough to form a monomolecular layer on the mineral surface should be added because excess amounts of collector will cause a multilayer to form on the mineral surface interfering with air bubble adhesion. However, instead of increasing the concentration (dosage) of a collector it is possible to increase the chain length to impart a greater hydrophobicity. This has a drawback of rapidly becoming insoluble as chain

length increases and in order for chemisorption to occur the collector needs to ionize (Wills, 1997).

### **1.3 Frothers**

Frothers are surface active reagents that facilitate air induced froths (Crozier, 1992, Urbina, 2003). They enhance flotation performance through (Wills, 1997):

- reducing surface tension stabilizing bubble formation
- creating a stable froth which allows selective drainage of entrained gangue
- increasing the kinetics of flotation

However, it is important that a frother does not act as a collector and only create a froth stable enough to facilitate collection and transfer of the concentrate (froth). Bubble formation is very important in froth flotation and frothers allow for smaller uniform bubbles to be generated thus increasing the surface area of each bubble. Higher surface area is equivalent to greater carrying capacity. The mechanism of frothers is to adsorb on the air-water interface, and to this extent, frothers should be sparingly soluble in water to allow for even distribution in the system. The most effective frothers contain the following functional groups: hydroxyl, carboxyl, carbonyl, amino and sulpho groups. Alcohols tend to be the most widely used due to their negligible collector property and form a brittle froth that aids in regulating the frothing characteristics (Wills, 1997).

## **1.4 Modifiers**

Also known as regulators, modifiers are used in conjunction with collectors to either increase or decrease the hydrophobic effect on the mineral surface, thus making the collector more selective to the desired valuable minerals. Modifiers are best summarized by Urbina (2003); “chemical compounds that regulate the solution chemistry and the pH or by controlling the dispersion or aggregation of solid particles.” There are three main types of modifiers; pH modifiers, activators and suppressants. While there are distinctions between each type of modifier, some modifiers that are added have multiple functions. For example, lime can be used to adjust the pH of the system (pH modifier); however, the calcium cation is a known depressant for pyrite in copper flotation (Crozier, 1992). Therefore, it is important to understand the effect that each reagent added to the system will have in order to maximize mineral flotation.

### **1.4.1 Activators**

Activators alter the chemical nature of the surface of minerals thus; facilitate collectors, to render them hydrophobic. They are generally soluble salts that ionize upon introduction in water, and the ions formed react with the mineral surface. A classic example is the activation of sphalerite by copper sulphate and this is discussed in detail in Section 2.2.2: Mechanism of Action of Activators and Depressants.

### **1.4.2 Suppressants**

Suppressants, also known as depressants, intensify the selectivity of collectors by rendering undesirable minerals (gangue) hydrophilic. They are inorganic or organic compounds



which increase minerals interactions with water molecules. The process for how suppressants work is varied and complex. This makes suppressants the most difficult reagents to control. In general, a suppressant works by coating the mineral surface and retarding collector adsorption. A naturally occurring example of suppression is slime coating, where slimes in an ore coat the valuable minerals surface retarding collector adsorption. This generally becomes an issue with particle sizes less than 20  $\mu\text{m}$  (Wills, 1997). Addition of sodium silicate increases the charge on the slime particles causing them to disperse. Depressants work either by (Bradshaw, 2005):

- a) preventing a collector from adsorbing onto an existing hydrophilic mineral
- b) adsorbing onto a hydrophobic mineral surface and rendering it hydrophilic

Bradshaw (2005) goes on to state that inorganic depressants belong to group (a) above whereas polymer based depressants tend to belong to group (b). While the addition of suppressants is focused on the gangue minerals, there are possible side effects that can impact sulphide minerals favourably and unfavourably. One is the aforementioned slime coating which can reduce the natural hydrophobicity of a sulphide mineral. Removing slimes through the use of depressants will increase the hydrophobicity of the affected minerals as well as decrease froth stability. A decrease in froth stability will reduce the amount of entrained gangue minerals. Also, polymer based suppressants at higher dosages were observed to depress sulphide minerals. This was theorized by Bradshaw (2005) to be due to the suppressants adsorbing more strongly onto sulphides or due to the decrease in charge which facilitates agglomeration. This agglomeration traps the sulphide minerals within the matrix of gangue minerals.

## ***1.5 Grinding***

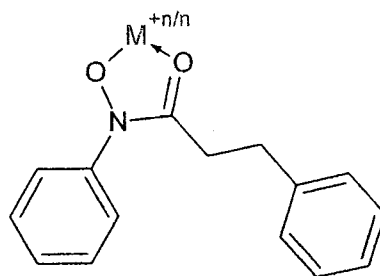
Grinding is a key element to successful flotation. It is imperative that care be taken when grinding samples to avoid contamination by grease, oil or extraneous iron (as extra iron in the system will distort the overall mass balance). Wet grinding should be done prior to flotation to prevent oxidation of mineral surfaces. Oxidation inhibits collector adsorption and also enhances dissolution of heavy metal ions which can inadvertently activate other mineral surfaces (Wills, 1997). Finer grinds can increase the possibility of the presence of slimes. Minerals which are strongly hydrophobic naturally may work well with a coarser grind and, according to Wills, other minerals such as sphalerite and pyrite are more easily depressed at larger particle sizes. However, it is important to grind finely enough that the minerals are of a size small enough to be levitated by the air bubbles passed through the pulp.

## ***1.6 Objectives of the Research Project***

In the mineral processing industry, xanthates are universally used to float sulphide minerals. With this collector, sphalerite is commonly activated with copper sulphate to enhance xanthate adsorption by forming a less soluble and more stable chemical compound on the mineral surface. According to Natural Resource Canada (<http://mmsd.mms.nrcan.gc.ca/stat-stat/prod-prod/2007-eng.aspx>, accessed Feb. 20/09), in 2007, 594,113 tonnes of zinc were produced. Depending on the grade of the ore the amount of copper sulphate needed for activation varies. Copper sulphate is used in consumed in quantities, 200 g/t to 1.2 kg/t, that far exceed all other reagents used in processing zinc ores, making it the most expensive auxiliary chemical even though it is not a specialty chemical such as collectors and frothers. Natarajan et

al. (Natarajan, 1995, Natarajan et al., 1997, Natarajan and Nirdosh, 2001, Natarajan and Nirdosh, 2006, Nirdosh and Natarajan, 2002) studied several alternative collectors to eliminate copper sulphate in the flotation of sphalerite. Derivatives of cupferron did not require activation of sphalerite by copper sulphate for its flotation (Natarajan et al., 1997, Nirdosh and Natarajan, 2002). Unfortunately, cupferrons had toxic properties and the concomitant flotation of pyrite caused major limitations on the modified cupferron reagents. Hydroxamic acids,  $R^1N(OH)C(=O)R^2$ , were chosen as a structurally similar alternative.

Hydroxamic acids belong to the O-O type chelating agents. Their chemistry and ability with several metal ions are well documented in literature (Agrawal, 1979, Agrawal and Tandon, 1974, Chatterjee, 1978, Natarajan and Nirdosh, 2001, Yale, 1943) and is shown in Figure 1. The stability constants of the chelates formed by hydroxamates follow the order: alkaline earth metals < transition metals < rare earths < cations with high charges. Of all the hydroxamic acids tested by Natarajan et al., *N*-hydrocinnamoyl-*N*-phenylhydroxylamine (HCNPHA) gave the best results. However, HCNPHA was observed to have a major limitation in concomitant flotation of pyrite with sphalerite. In order to make HCNPHA a viable alternative, gangue minerals such as pyrite and silica should be suppressed.



**Figure 1: Chelation of HCNPHA with metal ion of charge +n divided by number of chelates (n)**

Entrainment of gangue into the froth (concentrate) was due to the formation of a thick froth by HCNPHA and the affinity of HCNPHA for pyrite. Moreover, it was thought that sphalerite suppressed in the lead rougher stage needs reactivation. The present research aimed at eliminating these limitations to explore the viability of using HCNPHA as a collector for the flotation of sphalerite from a lead-zinc ore. Several polysaccharides were considered to suppress pyrite, improve the sphalerite grade of the concentrate, and regulate the frothing characteristics. Effect of copper sulphate on reactivating sphalerite suppressed in the lead-rougher stage was studied to fix the minimum amount of copper sulphate required to attain maximum sphalerite grade and recovery. Sodium silicate was tested as a suppressant for silica (non-sulphidic gangue). As flotation is a complex process involving control of several variables, attempts were made to optimize the collector concentration, air flow rate and particle size. In addition to the batch flotation tests, a large number of tests were conducted on flotation kinetics to study the differential flotation achieved for the combination of new reagents. The research project aimed to come up with an alternative reagent scheme for the flotation of sphalerite from a lead-zinc ore.

## Section 2.0 Literature Review

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## ***2.1 Xanthate Reagent Scheme***

Xanthates ( $\text{RO}(\text{CS})\text{S}^-\text{M}^+$ ), where R is a hydrocarbon group containing one to six carbon atoms, are known as universal collectors for base metal sulphide ores. Xanthates are prepared by reacting an alcohol, alkali hydroxide and carbon disulphide, and the most used versions are ethyl, isopropyl, isobutyl, amyl or hexyl hydrocarbon chains. They are commonly sold commercially as solutions, but also as powders or pellets, with pellets being the more common as it avoids dusting issues (xanthate dust is toxic and an irritant) as well as improves storage stability (Crozier, 1992). Classified as *thiol* collectors belonging to the anionic type of collectors, these are the most widely used collectors in the mineral processing industry. Xanthates oxidize easily and have a short lifespan in storage as after several weeks they develop a strong odour (hydrogen sulphide gas) and a deeper colour due to the formation of dixanthogen. This decomposition can be inhibited to an extent by using xanthates in an alkaline medium (Wills, 1997).

The xanthate reagent scheme for lead-zinc sulphide ores is a two stage flotation process. Galena is floated first with inorganic depressants (zinc sulphate, cyanide, metabisulphite) added to suppress the sphalerite. Xanthate is the collector used for both the galena and sphalerite however; they differ in structure (ethyl xanthate for galena, amyl xanthate for sphalerite). For the flotation of sphalerite, the sphalerite surface has to be reactivated in order for the collector to properly adsorb on the surface. This is achieved through the addition of an activator, in this case copper sulphate. Copper ions adsorb preferentially on zinc sites which facilitates the formation of copper-xanthate complexes. This in turn allows the sphalerite to be rendered hydrophobic and floated. The specific mechanisms of action for the collectors, activator and depressants are discussed in the following sections.

### ***2.1.1 Mechanism of Action of Xanthates***

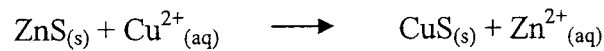
Nagaraj (1987) classified xanthates as a bidentate chelating agent of the S-S type. It is assumed by Wills (1997) that xanthates adsorb on sulphide mineral surfaces due to the chemical forces, forming insoluble metal xanthates which are highly hydrophobic. However, the xanthate complexes formed with zinc are more soluble and less stable than those formed with copper, lead, silver and mercury. The mechanism of action of xanthates has been generally agreed on to occur in one of three theories (Leja, 1982):

1. adsorption (multilayer chemisorption)
2. ion exchange
3. neutral molecule adsorption

Unstable complexes formed with xanthates, for example zinc-xanthate, require activators to enhance collector adsorption onto the mineral surface. The cupric ion ( $\text{Cu}^{2+}$ ), in the form of a sulphate or nitrate, is the most widely used activator (Chandra, 2009).

### ***2.1.2 Mechanism of Action of Activators and Depressants***

As mentioned earlier, in the flotation of sphalerite, copper sulphate is used to activate the sphalerite surface facilitating xanthate adsorption. While the activation of sphalerite via copper ions is generally agreed on, the actual mechanism is still debated. Copper ions activate the sphalerite surface through an ion exchange mechanism where each  $\text{Cu}^{2+}$  ion releases one  $\text{Zn}^{2+}$  ion into solution (Wills, 1997, Crozier, 1992, Chandra, 2009) as is shown by the following reaction:



Thus, allowing the xanthate molecule to react and the copper surface complex formed renders sphalerite hydrophobic. However, sphalerite can be inadvertently activated by other heavy metal ions present in the pulp which can be problematic as this accidental activation by other metal ions reduces the selectivity of separation of sphalerite from complex ores.

Suppression of various minerals such as sphalerite and pyrite during the lead-rougher stage in lead-zinc ores is achieved using cyanides either alone or in combination with chemicals such as metabisulphite or zinc sulphate. Cyanides are commonly used in mineral processing for separation of polymetallic sulphide ores, specifically when the depression of pyrite and other sulphide based metals are involved (Wills, 1998). Zinc sulphate is also used for suppressing sphalerite during the flotation of galena. Research done by Wang and Forssberg (1995) on the electrochemistry of sulphide-xanthate-cyanide systems theorized that cyanides depress the flotation of sulphides, with the exception of galena, due to one or more of the following process:

- Cyanides form metal complexes with surface metal ions preventing formation of metal xanthates.
- Cyanide reduces the redox potential of the pulp while consuming oxygen preventing the oxidation and chemisorption of xanthates.
- Cyanides react with sulphur to form thiocyanate thereby removing the hydrophobicity of the mineral.
- Cyanide is preferentially adsorbed on sulphide mineral surfaces thus preventing xanthate adsorption.



Zinc sulphate suppresses sphalerite in a different manner than cyanides. When used in conjunction with cyanides, zinc sulphates helps reduce the amount of cyanide needed to suppress sphalerite (Wang and Forssberg, 1995). Zinc sulphate readily dissociates in water and, at alkaline pH values, hydroxylation of zinc occurs. Zinc hydroxide then adsorbs onto the mineral surface preventing collector adsorption or undergoes ion-exchange with the heavy metal sulphides present on the sphalerite surface, thereby releasing the activating metal ions.

## **2.2 Hydroxamic Acids**

Hydroxamic acids ( $R^1N(OH)C(=O)R^2$ ) are chelating agents that form five-membered bidentate chelates with several metal ions (Somasundaran and Moudgil, 1987, May et al., 1998, Agrawal, 1973, Agrawal and Kaur, 2001). This notoriety is due to their ability to form stable complexes with transition metals such as iron. Figure 2 shows an example of a hydroxamic acid bonding to a metal ion. As such, much research has been done into using hydroxamic acids for medicinal purposes as a cure for iron poisoning (Agrawal and Kaur, 2001). Hydroxamic acids have also found use in analytical laboratories as colorimetric reagents and gravimetric reagents in the past (Chatterjee, 1978). While hydroxamic acids achieve efficient metal selectivity in analytical practice, this transfer to actual practice in mineral processing is far from straightforward.

There exist many different ways to synthesize hydroxamic acids (depending on the substituents desired) with most methods deriving from 'Blatt's method', (Chatterjee, 1978), where an alkyl or aryl ester reacts with a hydroxylamine in the presence of an alkali. Generally speaking, hydroxamic acids are weak organic acids, confirmed by high  $pK_a$  values.

*N*-aryhydroxamic acids being even weaker acids due to intramolecular hydrogen bonding (Chatterjee, 1978).

### ***2.2.1 Hydroxamic Acids as Chelating Collectors***

According to Nagaraj (1987), there are two basic requirements that are necessary but not sufficient for the chelating agent to form metal chelates:

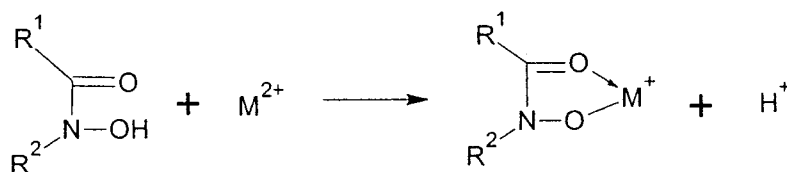
- a) Suitable functional groups should be contained within the molecule.
- b) The placement of the functional groups should be such as to allow for the formation of a ring with the metal as the central atom.

For mineral processing applications, three more requirements exist:

- c) In order for the chelating agent to function as a collector the complex it formed with a metal ion should be neutral. Complexes that are neutral tend to be insoluble in water, which in turn encourages hydrophobicity.
- d) Also, for the chelating agent to function as a collector, the bond formed must be sufficiently strong to resist scaling off.
- e) In order to function as a depressant, the chelates should be charged and highly hydrophilic. It should be noted that for a polymeric depressant, this is not a requirement as long as it contains enough hydrophilic groups on its backbone.

Hydroxamic acid chemistry has been extensively researched and derivatives of hydroxamates are continuously synthesised and tested in order to better understand the mechanism of action and the selectivity with respect to valuable minerals.

As mentioned earlier, pK values of hydroxamic acids differ significantly based on functional groups; the stability constants for hydroxamic acids with metal ion, however, do not vary significantly (Agrawal, 1977). The structure of hydroxamic acids can be varied greatly. Different derivatives vary significantly in pK<sub>a</sub> values which are influenced by either: groups having positive or negative inductive effect, the substituents present on the *N*-phenyl ring, and the conjugated groups attached to the carbonyl group (Nagaraj, 1987, Chatterjee, 1978). Substituents such as fluorine or chlorine induce a negative effect thereby lowering the pK<sub>a</sub> whereas groups such as methyl induce a positive effect and consequently increase the pK<sub>a</sub>. The stability constants of the chelates formed by hydroxamates follow the order: alkaline earth metals < transition metals < rare earths < cations with charges greater than two. It has been stated that hydroxamates are chelating agents of the O-O type, and while they do contain nitrogen, the nitrogen does not participate directly in chelation (Figure 2). However, it does affect the electron density of the oxygen attached to it.



**Figure 2: Example of hydroxamic acid bonding to a metal ion**

Hydroxamic acids can exist in either an acid medium (behave as monobasic acid due to one replaceable hydrogen) or an alkaline medium (contain two replaceable hydrogen atoms). Although hydroxamic acids have these two tautomeric forms, metal complexes are formed in an acidic medium (Chatterjee, 1978, Nagaraj, 1987, Agrawal, 1977). The pK<sub>a</sub> of hydroxamic acids is approximately 9, which is also the pH value of maximum flotation (Nagaraj, 1987). Studies

have theorized that this is due to one of two factors; either co-adsorption of ions at the interface or reactions of metal hydroxy complexes at the mineral surface (only if the system involves hydrolyzing metal ions). This was further confirmed by Hu et al. (1997) who stated that chelating collectors adsorb through chemical interactions between the metal ions on the mineral surface and the chelating group at the mineral-water interface. It has been indicated that after chemisorption of the hydroxamate, the metal hydroxamate precipitates on the mineral surface thus inducing its hydrophobicity and allowing the mineral to be floated (Hu et al., 1997). Similar to xanthates, hydroxamic acids are susceptible to multilayer formation. This is possibly due to excessive hydrogen bonding or hydrophobic bonding, which is all dependant on the nature of the system in question.

### ***2.2.2 Applications in Mineral Processing***

Chelating agents of the O-O type are the most extensively used reagents in mineral processing, with alkyl hydroxamic acids being the most investigated chelating agents for use as collectors (Nagaraj, 1987). Wide application for hydroxamic acids exists due to the fact that chelating agents of the O-O type (which hydroxamic acids belong to), form chelates with most metals and have increased selectivity compared to other types of collectors such as carboxylic acids. Polyhydroxamic acids are becoming more and more useful in the field of metallurgy, such as for recovery of alkali metal carbonates (Agrawal and Kaur, 2001). Flotation of oxide ores, such as chrysacolla by xanthates, traditionally uses sulphidization in order to float them. Sulphide/oxide blends, it is commonly practiced to float each component separately; however, according to Lee et al. (2008), and with hydroxamates it is possible to float both components simultaneously without sacrificing grade or recovery

While not commercially used in North America, hydroxamic acids have been used for over 50 years in Russia, under the name IM-50, and China (Nagaraj, 1987). However, specific details of the structure and use of this collector are not known. Alkyl hydroxamates have found use in the flotation of copper oxide ores when used in combination with xanthates. Alkyl hydroxamates have been in use to float copper-cobalt ores in Africa since the 1970's (Lee et al., 2008). The lack of success of hydroxamic acid collectors in the mineral processing industry may be based on choice of complexing agent. Most of the complexing agents tested as collectors have been based on successful chelation and not on commercial viability leading many new collectors as laboratory curiosities. Hydroxamic acids have many advantages over xanthates as collectors (stable in crystalline form, environmentally friendly, etc.), however, in order to be a viable alternative in the mineral processing industry, structures should be chosen for use on the basis of commercial viability and the introduction of such reagents should pose no change to current industrial operations.

### ***2.3 Polysaccharides in Mineral Processing***

Polysaccharides have found roles in mineral processing as depressants, dispersants or flocculants. They are mostly applied to the selective flotation of oxides, silicates and iron ores. However, their use with sulphide ores is limited due to the lack of adsorption selectivity (Bulatovic, 1999). Polymer depressants need to be selective, either in depression of gangue or flocculation, so that the entrapment of valuable minerals in flocs is reduced. However, if the polymer can selectively suppress and flocculate gangue minerals, it would be an effective depressant in flotation (Liu et al., 2006). Another limitation with polysaccharides is ensuring sufficient dissolution and hydration, as well as being careful not to add too much as viscosity

becomes an issue with pumping (Wiese et al., 2008). The polymers (such as starch and dextrin) that are used in flotation are generally of unknown chemical composition. That is to say that there exist approximately 75 different types of starches and over 120 dextrans, almost all of which have chemical structures that are not clearly defined or are highly suspect (Bulatovic, 1999). Not knowing the exact chemical structure, specifically the impurities present in natural polymers, can make predicting how the polysaccharide will interact with the minerals when introduced to the flotation system difficult.

Polysaccharides (starch, guar gum, dextrin) consist of basic sugar monomers with the different types just having the monomers structured together uniquely. The first patent to use starch and lime in the clarification of colliery effluents was issued in 1928 (Laskowski et al., 2007). Since then, starch and lime have been tested and employed in flotation of sulphide minerals. Since pyrite is the most abundant and widespread metal sulphide which lowers base metal concentrate grades, it is desirable to find alternatives to process ores using the inorganic (cyanides, ferrocyanides, etc.) sulphidic gangue suppressants because of these environmental concerns (Bicak et al., 2007). Polysaccharides fill this role effectively as they are naturally occurring, environmentally friendly molecules that are biodegradable.

Polymers can be categorized based on their functional groups into one of the following: non-ionic (starch, dextrin), anionic (CMC, guar gum), cationic, and amphoteric (polymers which contain both anionic and cationic groups). Anionic polymers are of interest in flotation due to modifications that can be made to the polymer which improves the depressing properties for easily floatable magnesium-bearing minerals (Bulatovic, 1999). The specific mechanisms for actions are mentioned in the following section.

### ***2.3.1 Mechanism of Action for Polysaccharides***

Mechanism of action for polysaccharides is not well understood. However, it has been suggested that it could be due to one of the following reasons:

- Hydrogen bonding.
- Hydrophobic interaction (dispersing effect of polymers).
- Chemical interaction.
- Electrostatic attraction.

It is agreed upon that interaction of polymer based depressants is chemical in nature and employs a blocking action; that is to say they form a hydrophilic film on the mineral surface preventing the collector (Rashchi et al., 2004). Research has indicated that this interaction tends to occur via the metal hydroxyl species on the mineral surface.

Polysaccharides do not have clearly defined mechanisms; however, through extensive research, several properties of polysaccharides that seem consistent when used in flotation have been discovered. Polymer depressants adsorption has been observed to be independent of pH (Wang et al., 2005); however, the diffusion rate significantly increases with an increase in temperature (Liu et al., 2006). Adsorption was theorized to decrease with lower molecular weights. This has been also explained by adsorption being favoured by the polymer containing more carboxyl functional groups which interact with the mineral surface (Liu et al., 2006). In terms of specific polymers, Wang et al. (2005) observed that guar gums adsorption density did not significantly change with an increase of pH, suggesting that electrostatic forces are not the driving force. The same applies to hydrophobic forces, as ionic strength had negligible effect on adsorption. This was further confirmed by Bicak et al. (2007), who suggested guar gum worked

through hydrogen bonding and Bronsted acid/base interactions depending on the pH. The adsorption of carboxymethylcellulose (CMC) can be greatly enhanced by the addition of calcium ions and it was found to be detrimental in suppressing pyrite when the CMC contained a higher degree of substitution. This was stated to be the result of higher electrostatic forces between the CMC molecules and the pyrite surface (Bicak et al., 2007).

Most researchers conclude that polysaccharides adsorb at the mineral-aqueous solution interface through reactions with the hydroxylated metal species on the mineral surface (Laskowski et al., 2007). These interactions can be categorized as acid/base reactions with the polysaccharides acting as acids. Therefore, the stronger the basic properties of the mineral surface the stronger the reaction with the polysaccharides will be, and the reaction will manifest a chemical interaction (Laskowski et al., 2007, Liu et al., 2000). Strongly acidic mineral surfaces will display a weak acid/base interaction resulting in light hydrogen bonding. Liu et al. (2000) observed that non-hydrolysable metal cations can enhance polysaccharide adsorption onto highly acidic mineral surfaces. There still needs to be more research into clearly defining how polysaccharides interact in complex ore systems. This research should be conducted on both pure minerals, to clearly see trends in regards to specific minerals, and in polymetallic ore systems to determine if selectivity for gangue minerals can be achieved.

### ***2.3.2 Polysaccharide Applications in Mineral Processing***

Polysaccharides have been used in the mineral process industry for over 70 years (Liu et al., 2000). Polysaccharides such as dextrin, guar gum, starch and CMC have found use as depressants in flotation of polymetallic ores for suppression of hydrophobic non-sulphidic gangue minerals. However, as of late, the focus of research has been the selective separation of



sulphide minerals. Liu et al. (2006) states that polysaccharides are effective suppressants for talc and magnesia-bearing minerals as impurities in sulphide ores. To that extent, high molecular weight polymers are effective for suppressing slimes and reducing entrainment of gangue in the concentrate froth.

According to Bulatovic (1999), CMC, cellulose gums and guar gum are widely used for suppression of activated silicates in the flotation of polymetallic ores such as copper-zinc, copper-lead-zinc and platinum group metals (PGM). Low molecular weight starches are also used to suppress naturally hydrophobic gangue minerals (Wiese et al., 2008). Modified starches have also been tested; however, unlike their CMC and guar gum counterparts, a significant portion (approximately 30%) of the starches remained in solution, (Weise et al., 2008). Starches, dextrans and CMCs are all suitable for depression of sulphides, whereas, anionic polymers function well in depressing non-sulphidic gangue minerals (such as silicates) during differential flotation of sulphides. CMCs are widely used in mineral processing plants to suppress talc, activated silicates and magnesium-bearing minerals (Bulatovic, 1999). Guar gum is also used as a suppressant for talc in PGM processing plants (Want et al., 2005). Bicak et al. (2007) observed that guar gum selectively suppressed pyrite at an alkaline pH (pH 9) using low dosages, whereas higher dosages and a higher pH were required for CMC to achieve similar results.

As polysaccharides become better understood and optimized, their role in mineral processing might flourish. The advantages are many: environmental, economically cost effective, biodegradable, long shelf life, etc. With more research in optimizing each polymer, the goal of achieving a more selective depressant comes closer.

## Section 3.0 Materials and Methods

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### **3.1 Ore**

Ore was obtained from *Teck Cominco Ltd.*, British Columbia, Canada. The ore has an average assay of 25-35% sphalerite (zinc sulphide), 2-8% galena (lead sulphide), 5-10% pyrite (iron sulphide), 4-5% barite (barium sulphate) and 42-64% silica. The buckets of ore received were combined together and homogenized before partitioning into 350 g and 1 kg samples. Three percent of iron is always associated with sphalerite (Ref. private communication from *Teck Cominco*), hence, the pyritic iron mentioned in the following sections refers to the total iron calculated as follows:

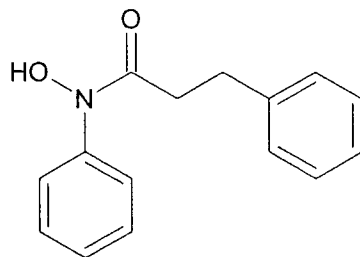
$$\text{Py (pyritic iron)} = \text{Total iron} - \text{iron in sphalerite}$$

### **3.2 Grinding**

Grinding was carried out in a polyurethane lined rod mill, to prevent extraneous iron from leaching into the system, using 24 stainless steel rods varying in diameter (1", 0.5" and 0.25"). The rod mill and rods are cleaned prior to use, adding distilled water and rolling for 10 minutes to remove any lingering traces of ore from previous tests. Distilled water was added to maintain 67% solids during grinding and the rod mill discharge was transferred to the flotation cell where the bulk density was reduced to 35%. Grinding time was sufficient to achieve a particle size of 80%  $\leq 63 \mu\text{m}$  based on sieve tests and particle size analysis (refer to Section 4.2.1 for results).

### 3.3 Collector

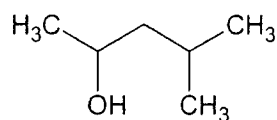
*N*-hydrocinnamoyl-*N*-phenylhydroxylamine (HCNPHA) is an *N*-arylhydroxamic acid collector of the type *N*-aryl-*C*-aralkyl. A sample previously synthesized in the research group was used for the present research. HCNPHA's structure is shown in Figure 3 and HCNPHA was crystallized from a water-ethanol mixture (75% alcohol) and produces a colorless crystalline structure. Although, HCNPHA in the solid form has a long shelf life, its solutions in alkalis decomposed during storage. Solutions were prepared by weighing out the appropriate amount to create a 1% by weight solution, transferring to a volumetric flask using distilled water, and adding 1 N sodium hydroxide drop-wise until dissolution occurs. The stock solution thus prepared was stored in a refrigerator.



**Figure 3: Structure of HCNPHA**

### 3.4 Frother

Methyl isobutyl carbinol (MIBC) was the frother used for all flotation tests. It is an alcohol-based frother currently being used by *Teck Cominco* (private communication from *Teck Cominco*) and its structure is shown in Figure 4. A stock solution of 0.1% by weight was prepared in distilled water and used for all experiments.



**Figure 4: Structure of MIBC**

### **3.5 pH Modifiers**

Sodium hydroxide (1 N), sulphuric acid (1 N) and quick lime (saturated solution) were used to adjust the pH during the zinc rougher stage as outlined in Section 3.9: Flotation Tests. All pH modifiers were reagent grade, and distilled water from Lakehead university supply system was used to make up all solutions.

### **3.6 Other Reagents**

All reagents were reagent grade and made into solutions using distilled water from the Lakehead university supply system.

Sodium metabisulphite (MBS)	-	1% solution by weight
Zinc sulphate	-	1% solution by weight
Copper sulphate	-	1% and 10% solutions by weight
Lime (quick lime)	-	Saturated solution of lime
Sodium cyanide	-	1% solution by weight
Sulphuric acid	-	1 N solution
Sodium hydroxide	-	1 N solution

Potassium ethyl xanthate (PEX)	-	1% solution by weight
Potassium amyl xanthate (PAX)	-	1% solution by weight
Suppressants (excluding CMC's)	-	1% solution by weight

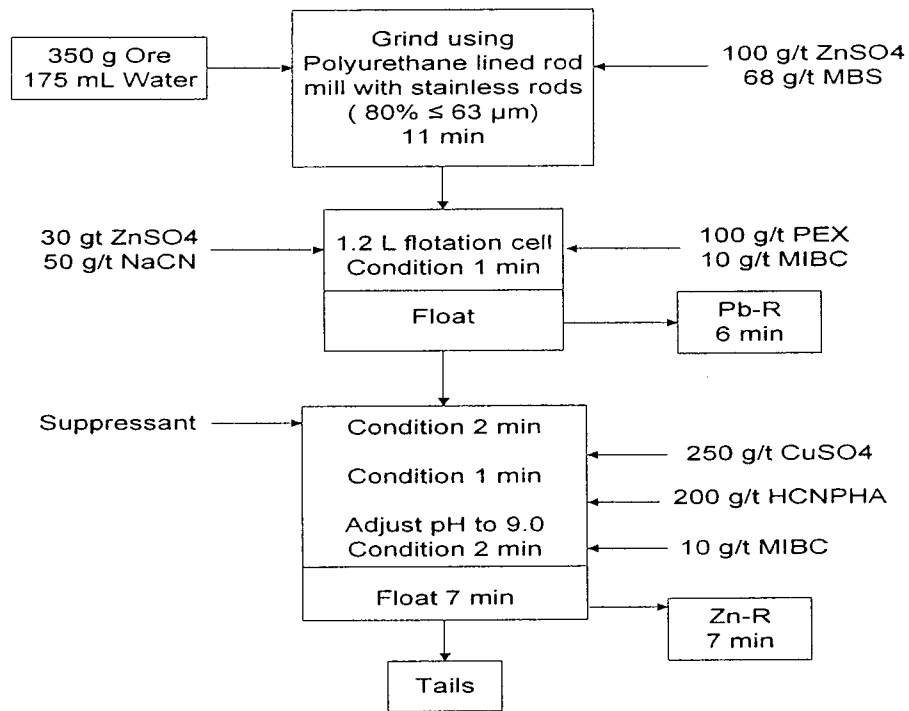
### ***3.7 Preparation of Carboxymethylcellulose (CMC) Solution***

Solutions of CMCs were prepared according to a procedure suggested by the suppliers of the samples, Cambrian Chemicals Inc., Ontario, Canada (private communication). Using a magnetic stirrer, a vortex was created in the beaker, such that the vortex reached the bottom of the beaker. Water at pH 10 to 11 was used to dissolve the CMCs. CMC was slowly added down the center of the vortex, allowing sufficient time to dissolve, to create a 1% by weight solution. It should be noted that the polysaccharide suppressants were also prepared following this procedure.

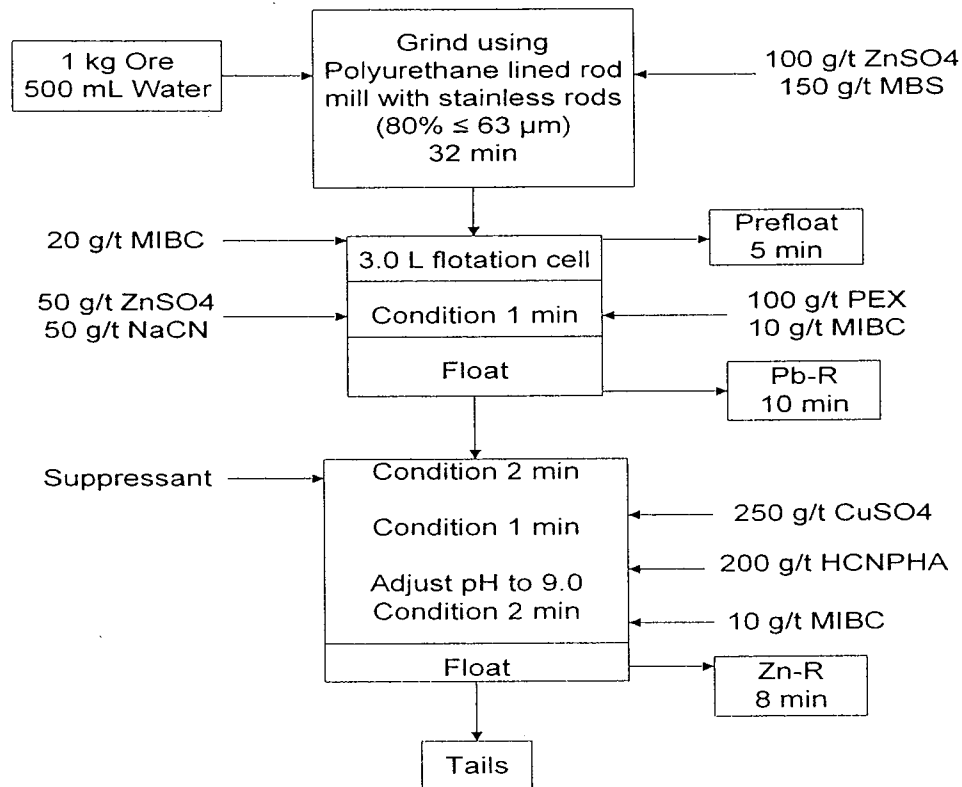
### ***3.8 Flotation Tests***

#### ***3.8.1 Batch Flotation***

Once the ground ore was transferred to the flotation cell, it was moved to the Denver laboratory flotation machine. Distilled water was added to bring water level 1.5 cm below the lip which also lowered the bulk density to approximately 35%. Experiments were carried out using 350 g of ore or 1 kg of ore; the general procedures are given in Figure 5 and Figure 6, respectively.



**Figure 5: General procedure for 350 g ore samples**



**Figure 6: General procedure for 1 kg ore samples**

For all flotation tests, makeup water at pH 9.0 was used. The specific changes made before each batch of experiments are given in the results section of this report. All floats and tails were oven dried at 60°C overnight, weighed, homogenized thoroughly before digesting a representative sample.

### ***3.8.2 Flotation Kinetics***

Flotation kinetics was studied only for the zinc rougher stage, and therefore, the general procedure remained the same. However, timed samples were collected (1, 2, 3, 5, 8, 14, and 22 minutes) and the frother was added in three stages (0, 5, and 10 minutes) to maintain a constant froth during the collection time. Also, the froth density was measured by weighing each tray before use then weighing after collection. The difference of the two weights divided by the dry weight of the cut gives the froth density.

### ***3.9 Acid Digestion***

Digested samples were analyzed using inductively coupled argon plasma atomic emissions spectroscopy (ICAP). An exactly weighed representative sample of a float or tails was digested using the following five day procedure:

Day 1: 7 mL distilled water and 3 mL HNO<sub>3</sub> were added, and the contents were heated to 90°C.

Day 2: 10 mL HNO<sub>3</sub> were added and the contents heated to 90°C.

Day 3: 3 mL HF and 7 mL HNO<sub>3</sub> were added and the contents heated to 90°C.



Day 4: 3 mL HF and 7 mL HNO<sub>3</sub> were added and the contents heated to 150°C.

Day 5: 10 mL HNO<sub>3</sub> were added and the contents heated to 90°C then cooled to room temperature.

The crucibles were cooled and each solution was made into 500 mL using volumetric flasks. Thus, the final solution was in 2% HNO<sub>3</sub>. Each of these solutions was analyzed by ICAP for lead, zinc and iron. The final solution was prepared in nitric acid instead of hydrochloric acid to avoid precipitation of lead as lead chloride.

### ***3.10 Particle Size Analysis***

Both ground and unground ore samples were analyzed using a Malvern MasterSizer 2000 with a Malvern Hydro 2000MU dispersion unit and Mastersizer 2000E v 5.20 analytical software. Results were obtained using the following procedure:

- Equipment was warmed-up for 30 minutes in order for the baseline to stabilize.
- Bubbles were removed from the system by increasing and decreasing the speed of the pump several times to provide a cleaner background signal which gives more accurate results.
- Program variables were set by selecting the solvent and its respective refractive index (water = 1.33); sample measuring time (45 seconds); desired obscuration level (5-10%) and measuring cycles (three measurements per sample).
- Background was measured before each sample.

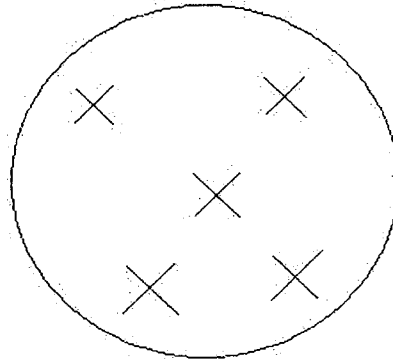
- Sample was introduced into the dispersion unit until the obscuration level reached 5-10%, after which the sample measurements were taken with results automatically graphed by the analytical software.
- Equipment was adequately cleaned between each sample with de-ionized water to ensure no particles remained in the tubing or sample cell.

The particle size analyzer (PSA) is a specialized instrument to measure the distribution of different sized particles. It uses a two laser system (two single wavelength lasers; red – 633 nm for large particles; blue – 466 nm for fine particles) where the lasers are passed through the cell. The laser beams are scattered at different angles due to different sizes, shapes and opacities of the particles present. Detectors measure the scattered light and generate a plot based on volume.

### ***3.11 Scanning Electron Microscope (SEM) Imaging***

A JEOL JSM-5900 LV SEM was used for all images and the software program to run the SEM was the SEM Control User Interface Version 5.27. X-ray analysis was carried out on samples to confirm and identify all minerals present as well as to show which minerals were liberated and which minerals were trapped inside other matrices. The X-ray analysis used a back scatter electron (BSE) detector which detects the primary electrons deflected off the nuclei. These electrons produce a spectrum containing an atomic number contrast. Elements with high atomic numbers show up as bright white spots on the SEM images, whereas elements with low atomic numbers, such as carbon, appear as black. Sphalerite contains holes in its crystal lattice which have been assumed to be occupied by iron. It was assumed that sphalerite contained approximately 3% iron. X-ray analysis was used to determine whether any iron contained in the

sphalerite matrix was naturally associated with it or due to the presence of pyrite. Samples were prepared by using double sided carbon tape attached to metal stubs. The homogenized ore samples were lightly tapped onto the tape. Using light air pressure, any extra ore was then removed so that only what was attached to the tape was left on the stub. In order to ensure that a complete image of each sample was obtained, several spots across the sample were imaged as shown in Figure 7. Sphalerite was the main interest so areas that showed a high amount of sphalerite were chosen for all images. Images were then color mapped using Imagepro software to help clearly display the liberation, or lack thereof, of the minerals.



**Figure 7: Schematic diagram outlining spots on sample where images were taken**

## Section 4.0 Results and Discussion

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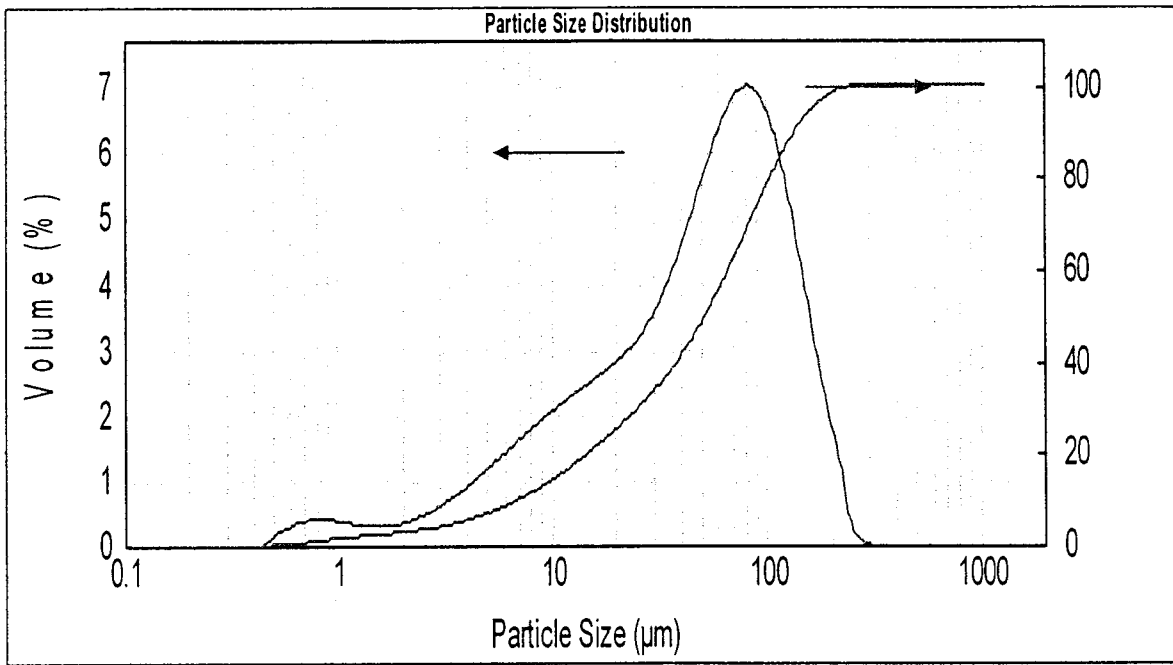
## ***4.1 Fixing Flotation Tests Variables***

In order to determine the effect of the suppressants tested, it is first necessary to optimize flotation test variables such as air flow rate, collector dosage, particle size and copper sulphate dosage. In each of these tests 350 g ore sample were used. All three variables were tested separately with the specific changes to the general procedure are stated in their respective sections. All modifications were applied to the zinc rougher stage, as the focus of the research was flotation of sphalerite. Therefore, in all the following sections, any changes to dosages or experimental parameters are only applied to the zinc rougher stage, with the exception being grinding time, unless otherwise stated, i.e., if changed from the general procedures given in Section 3.9.1: Batch Flotation.

### ***4.1.1 Particle Size Analysis***

Particle size is very important to the flotation process as achieving the proper particle size for an ore encourages adhesion to air bubbles and improves the floatability of the ore as the particles are not too large so as to prevent levitation by air bubbles. It was necessary to confirm that the previously stated condition of 80% passing through 63  $\mu\text{m}$  particle size, was achieved using an 11 minute grinding time. While 100  $\mu\text{m}$  particle size is sufficient to float, it was desirable to grind to a specific particle size in order to liberate the maximum amount of valuable mineral from the host mineral (silica). Therefore, sieve analysis on the ore sample was conducted. Samples were ground for 11 minutes and the slurry was filtered. A piece of the filter-cake (ground ore) was removed as a representative sample. This was washed with acetone and dried. The dried solids were sieved to obtain the particle size distribution. If the desired

particle size was not achieved, the solids were mixed together and grinded for an additional three minutes and the procedure was repeated. To confirm the results of the sieve analysis, a Malvern particle size analyzer was used to determine the particle size distribution. Figure 8 shows the particle size distribution of the ore supplied by *Teck Cominco*.



**Figure 8: Particle size distribution for the ore supplied by *Teck Cominco***

The black arrow denotes the curve that is read using the left axis whereas the red arrow indicates the curve that uses the right axis (this set-up is consistent for all graphs presented in this section). Here it can be seen that the ore has 62% passing through 63 µm. In comparison, Figures 9 and 10 show the results for the two different grinding times. It is clearly shown that 11 minutes grinding time was not producing the desired particle size (only 65% passing through 63 µm), this is confirmed by the sieve analysis in Table 1. A 14 minute grinding time was found to achieve 79% passing through 63 µm.

**Table 1: Sieve analysis for 11 minutes grinding time**

Mesh ( $\mu\text{m}$ )	11 minutes		
	Mass (g)	Wt. %	Grade (wt. %)
+75	341.9	36.4	29.7
53	172.6	54.7	31.2
-53	425.8	100.0	33.2

Based on the above observations, a grinding time of 14 minutes was used for all batch tests with 350 g ore samples in order that the particle size conforms to that of *Teck Cominco*. Agglomerating the slimes is another option because the finer particles from the system prevent detrimental effects later in the process such as slime coating. The ore samples of different particle size ranges were analyzed by ICAP for mineral distribution. It was found that -38  $\mu\text{m}$  particle had 33.2% sphalerite. However, agglomeration of the slimes was discarded as an option following ICAP analysis of samples collected from a sieve analysis. The sieve analysis showed that the fine particles (slimes) still contained a high amount of sphalerite (sphalerite grade for -38  $\mu\text{m}$  was 33.3%) which would be lost.

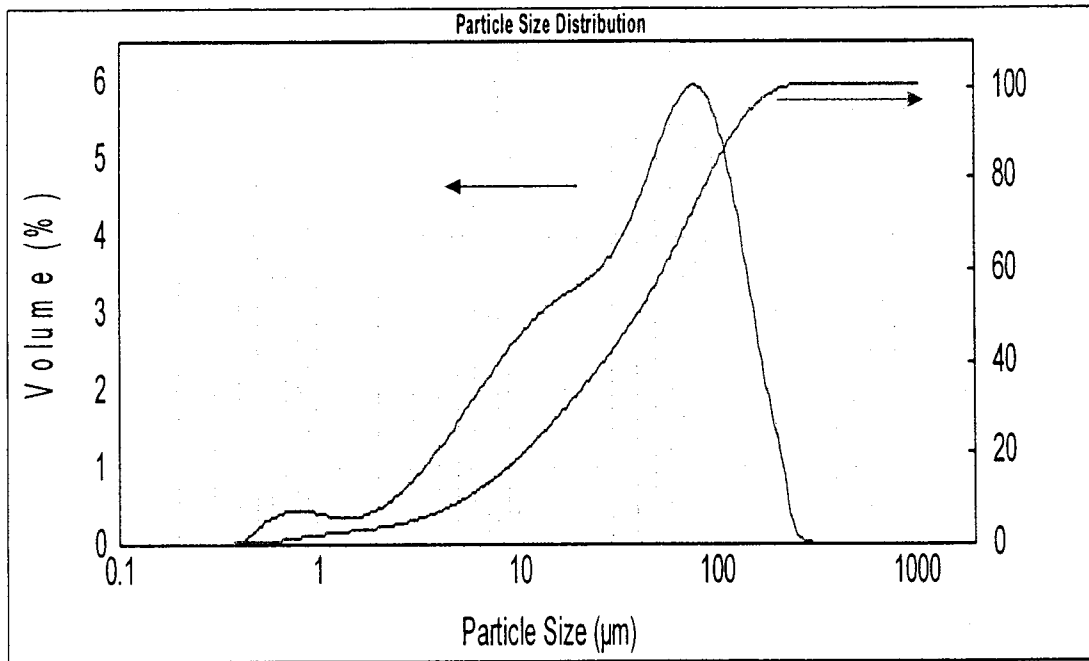


Figure 9: Particle size distribution for 11 minutes grinding time

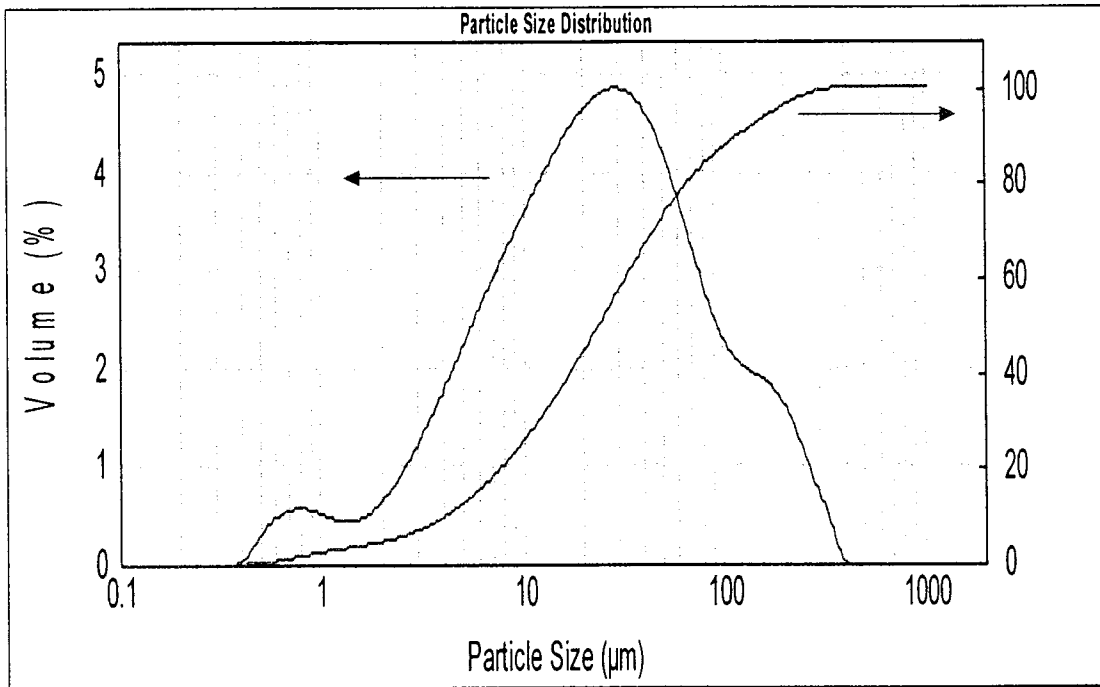


Figure 10: Particle size distribution for 14 minutes grinding time



#### 4.2.2 Scanning Electron Microscopy (SEM)

To confirm the results from Section 4.2.1: Particle Size Analysis, a scanning electron microscope was used in conjunction with semi-quantitative X-ray analysis to confirm the liberation of valuable minerals from the host matrices. Both the zinc rougher and tails for samples ground for 11 minutes and 14 minutes were analyzed. Figure 11 and Figure 12 show the results of the zinc rougher stage.

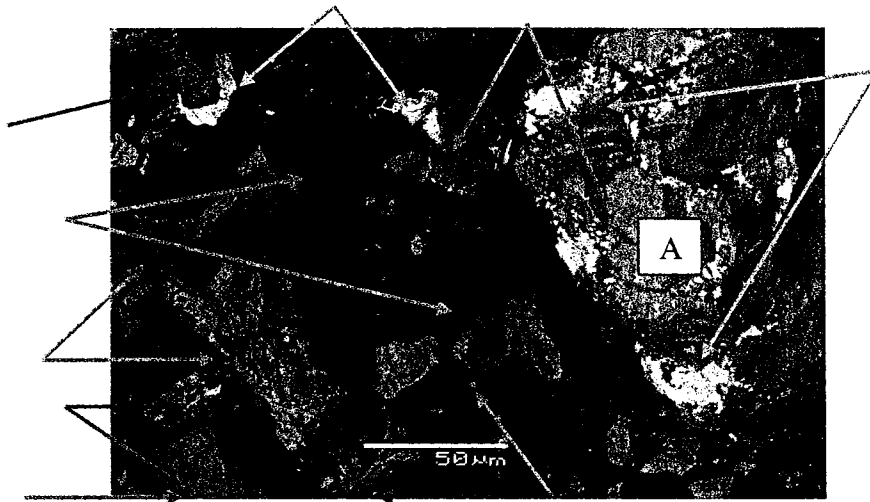


Figure 11: SEM image of zinc rougher stage after 11 minutes grinding (SP-53 Zn 4)

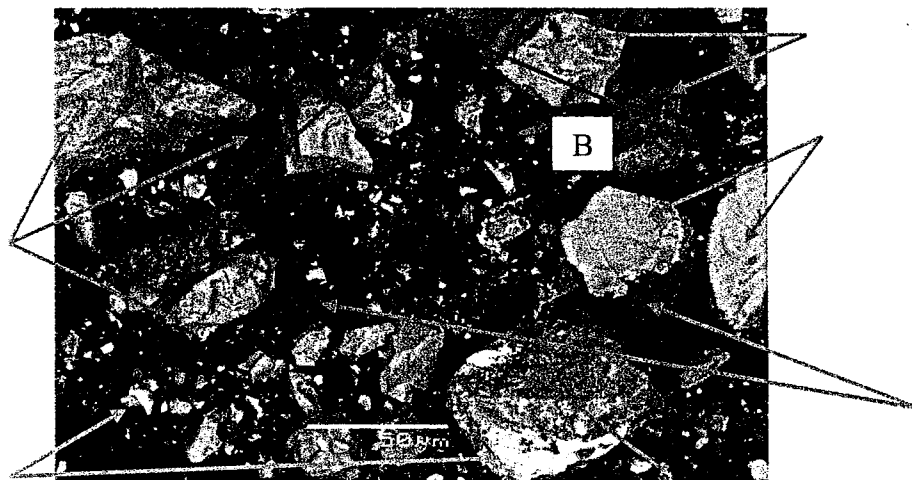


Figure 12: SEM image of zinc rougher stage after 14 minutes grinding (SP-55 Zn 3)

(Note: the blue arrows denote sphalerite, red arrows for pyrite, green for galena and brown for silica and this convention is uniformly followed in all SEM images in this section.)

A back scatter electron detector was used which works based on atomic number. The higher the atomic number the brighter (whiter) the element will appear in the image. Therefore, the white spots are galena, the light grey is sphalerite, dark grey is pyrite and the almost black particles are silica. It is clearly observed that in general the particle sizes are smaller when ground for 14 minutes than with 11 minutes. Also, since there is approximately 3 % iron associated with sphalerite semi-quantitative X-ray analysis was conducted on sphalerite particles to determine if the iron present was from pyrite or the naturally occurring iron within sphalerite. Table 2 shows the X-ray analysis results for the outlined particles (refer to Appendix 2.0 for a complete listing of results). X-ray analysis carries with it some inherent error:

- The spot size of the beam can be made only so small therefore, may encompass the particles around the area that is being analyzed
- While leaving the beam in one spot for an extended period of time will give a cleaner signal, the beam will spread out through the sample striking secondary targets which will show up on analysis

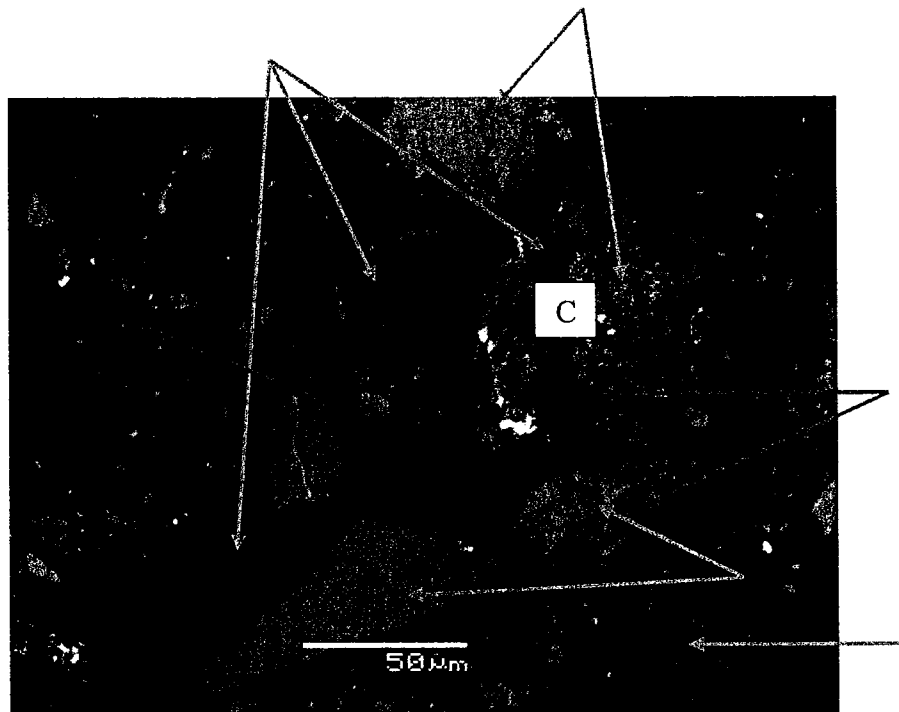
However, using a consistent method of measuring each sample helps minimize these errors.

**Table 2: Element percent results for zinc rougher stage for various grinding times**

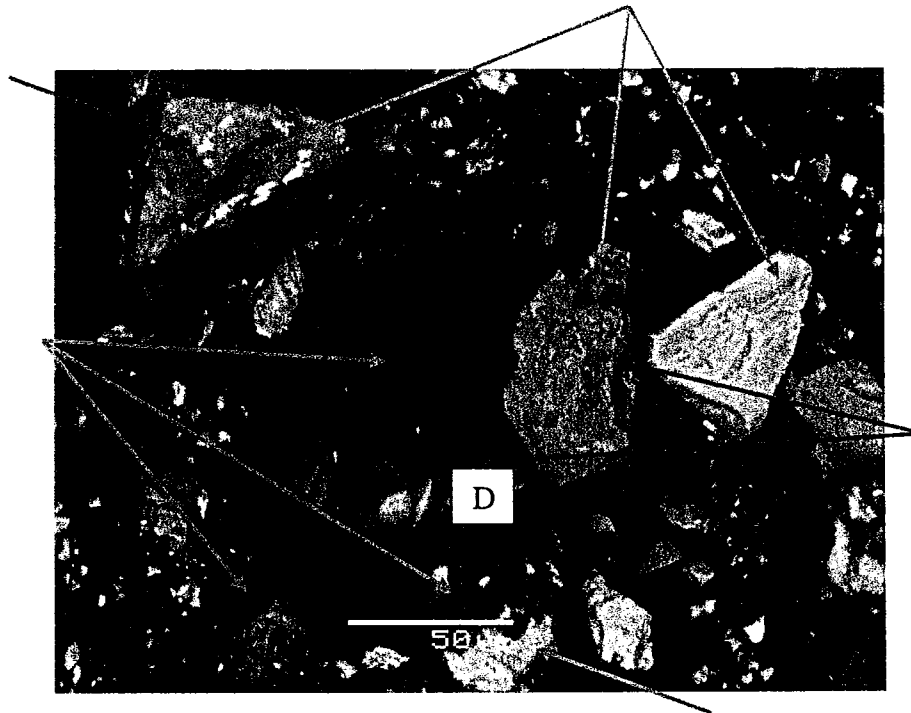
Sample Site	Element %			
	Pb	Zn	Fe	Si
A	8.51	60.96	6.89	2.76
B	6.91	28.97	38.98	2.33
C	59.27	14.91	6.60	4.63
D	5.53	43.42	1.35	18.30

Generally speaking, it was found that there was a quantity of galena (on average 1% to 5%) present wherever sphalerite was detected. High pyrite values were rarely detected inside

sphalerite particles except in case (B) where the color has darkened in comparison to (A) and the pyrite content increases to 38.98%. Increasing the grinding time also increases the amount of galena liberated from other mineral matrices. Analysis of the tails of the two grinding times; it was observed that fewer particles are seen with sphalerite trapped within a silica matrix. In the zinc rougher float, the darkening of particles was due to the presence of pyrite; however, in the tails the darkening of the particles was caused by silica. In both the zinc rougher and the tails which were ground for 14 minutes, there are more sphalerite particles liberated from other matrices which confirm the results from the sieve and particle size analysis.



**Figure 13: SEM image of tails after 11 minutes grinding (SP-53 T2)**



**Figure 14: SEM image of tails after 14 minutes grinding (SP-55 T3)**

#### ***4.2 Preliminary Reagents Tests***

Based on an extensive literature survey (refer to Section 2.0: Literature Review), it was decided to test the effect of polysaccharides and carboxymethylcelluloses (CMC's) on pyrite suppression. These tests were carried out for a preliminary evaluation into the effectiveness as a suppressant for pyrite. This was done to determine if new suppressants were needed or if one of the types tested showed promise for use in future tests, meaning if one of the suppressants depresses sulphidic gangue minerals such as pyrite. All tests followed the general procedure for 350 g ore samples outlined in Section 3.9.1: Batch Flotation.

##### ***4.2.1 Effectiveness of Polysaccharides as Pyrite Suppressants***

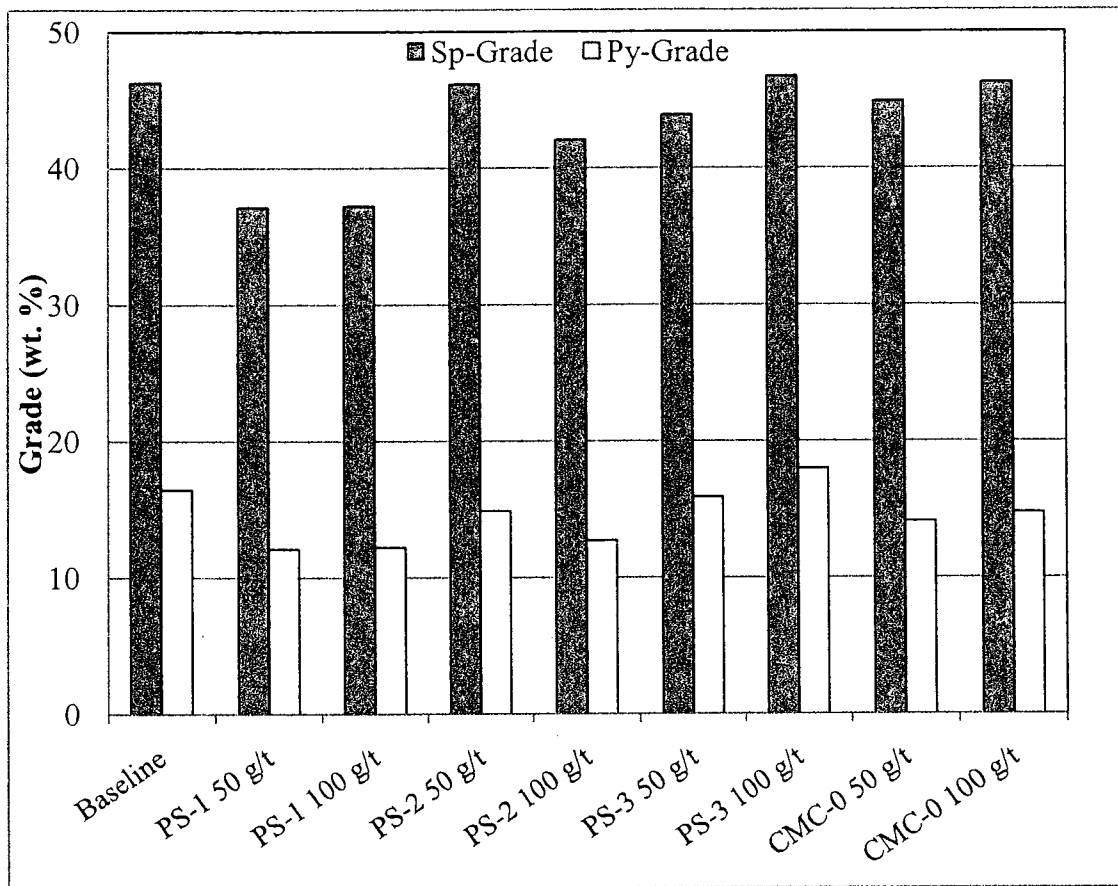
A baseline test, where no other auxiliary chemicals were added, using only HCNPHA, was conducted in order to have a basis of comparison. Each suppressant was tested at two

dosages, 50 g/t and 100 g/t, with all other variables kept constant. Table 3 gives the experimental conditions for these batch flotation tests. For these tests, an optimum collector dosage (200 g/t), comparable to the amount of potassium amyl xanthate (PAX) used by *Teck Cominco*, was used. Results of these tests (P1 to P9) are given in Table 15 of the Appendix 1.0.

**Table 3: Procedure specifics for types of suppressants experiments**

Test	Stage	Time (min.)	MBS (g/t)	ZnSO <sub>4</sub> (g/t)	NaCN (g/t)	PEX (g/t)	HCNPHA (g/t)	MIBC (g/t)	Air (Lpm)	pH
P-1 to P-9	Grind	11	68	100						
	Pb-R Cond.-1	1		30	50					
	Pb-R Cond.-2	1				100				6.33-6.48
	Pb-R Float	6						10	2.0	
	Zn-R Cond.-1	2					200			8.97-9.01
	Zn-R Float	7						10	2.0	

Figure 15 gives the comparison of sphalerite grade and pyrite grade in the zinc rougher concentrates obtained in tests P1 to P9.



**Figure 15: Sphalerite and pyrite grades for suppressant tests**

Both show suppression in pyrite grade, from 16.4 (baseline) to 12.2 (PS-1) and 16.4 to 14.1 (CMC-0) respectively, where as, the other two polysaccharide suppressants neither change the pyrite grade nor float more pyrite further increasing the grade (PS-3). It was concluded from these experiments to continue testing with PS-1 and CMC-0. It was considered to fix the flotation variables (air flow rate, collector dosage, particle size) need to be optimized before launching extensive testing on suppressants or any other auxiliary chemical.

#### 4.2.2 Effect of Air Flow Rate

To determine the optimum air flow rate, air flow was varied from 0.5 Lpm to 3.0 Lpm, increasing in 0.5 Lpm increments (tests AF-1 to AF-6). The effect of the air flow rate was studied on the recoveries and grades of sphalerite (Sp), pyrite (Py), and non-sulphidic gangue (NSG). The experimental conditions and the dosages of the chemicals used were the same as in Table 3. Since sphalerite was the mineral of interest, effect of air flow rates for the flotation of galena in the lead rougher stage were not studied.

Recoveries of sphalerite, pyrite and non-sulphidic gangue in the zinc rougher stage for the various air flow rates tested are given in Figure 16 (please refer to Table 16 in Appendix 1.0). No significant improvement beyond an air flow rate of 1.5 Lpm was observed. Therefore, it was decided to use an optimum air flow rate of 2.5 Lpm for all subsequent tests.

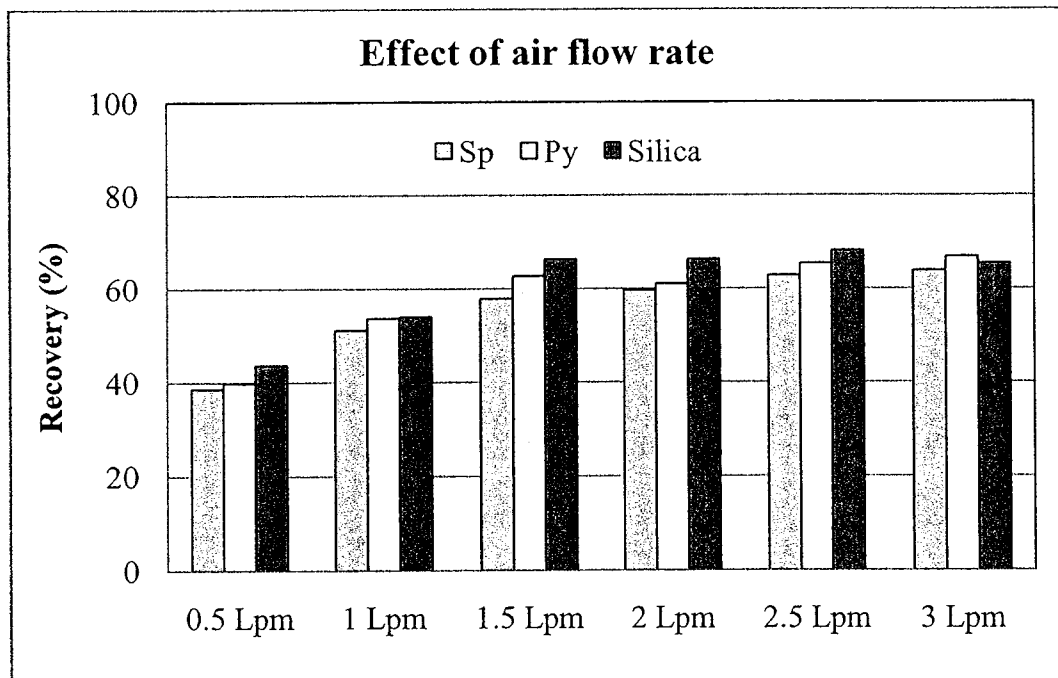


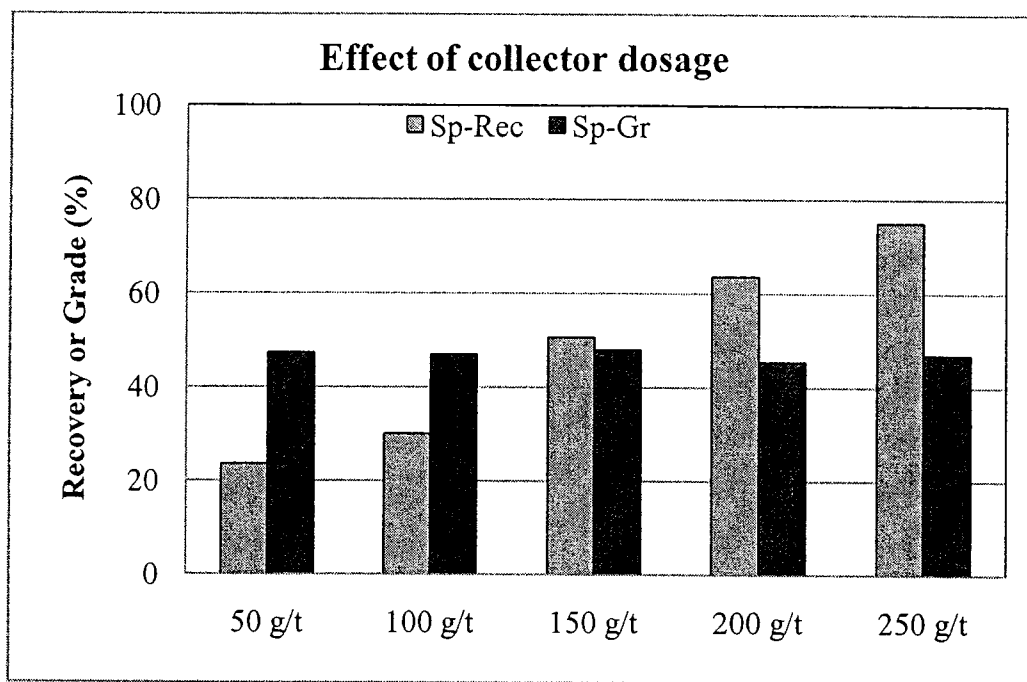
Figure 16: Effect of air flow rate on sphalerite recovery

### ***4.2.3 Effect of Collector (HCNPHA) Dosage***

After fixing the air flow rate at 2.5 Lpm, the effect of collector dosage was studied by adding different amounts of HCNPHA from 50 to 250 g/t, in 50 g/t increments (tests CC-1 to CC-6). In tests CC-1 to CC-6, experimental conditions and dosages of chemicals used were the same as in Table 3. The sphalerite grade and recovery for each of the tests is given in Figure 17 and detailed results are presented in Table 17 in Appendix 1.0.

The sphalerite grade stayed relatively constant regardless of collector dosage. However, the sphalerite recovery continually increased with higher amounts of collector. This indicated that sphalerite was not being enriched in the float, and this was due to the direct result of concomitant flotation of gangue minerals. These gangue minerals were most likely entrained by the thick froth formed by the collector itself. This especially thick froth was consistent with every test and is a major concern in that if the frothing characteristics of the system using HCNPHA cannot be changed, then it will be problematic to achieve higher grades of sphalerite even with the suppression of gangue minerals. Therefore, it was decided to use 200 g/t HCNPHA and improve the grade of the concentrate by adding reagents that would suppress gangue materials by changing the frothing characteristics and by changing the surface characteristics of sphalerite.





**Figure 17: Effect of collector dosage on sphalerite recovery and grade**

#### **4.2.4 Effect of Copper Sulphate Dosage**

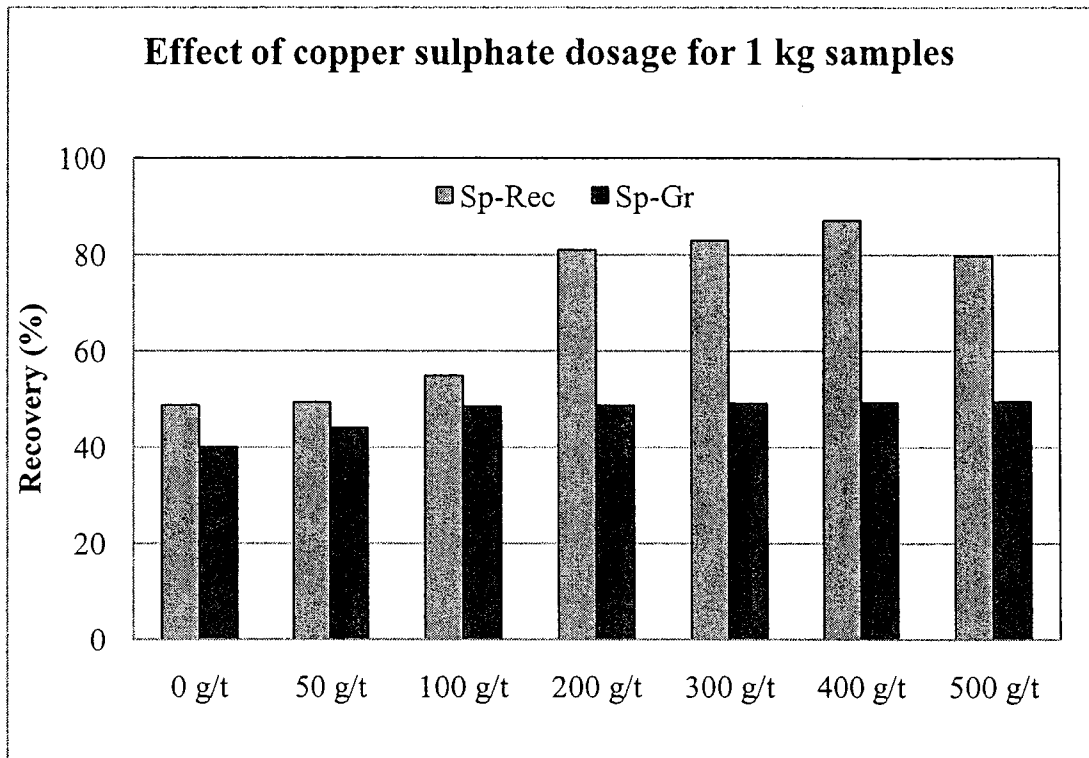
Having set the air flow rate (2.5 Lpm) and the collector dosage (200 g/t), it was decided to determine the optimum dosage of copper sulphate. Currently, *Teck Cominco* uses 1.2 kg/t of copper sulphate in the xanthate reagent scheme. However, the alternative collector HCNPHA, was expected to reduce the amount of copper sulphate needed. Tests were conducted using both 350 g (CS-1 to CS-13) and 1 kg (CS-14 to CS-20) of ore samples; the procedures for which are outlined Section 3.9.1: Batch Flotation. The experimental conditions for tests CS-1 to CS-13 are given in Table 3 and the experimental conditions for CS-14 to CS-20 are in Table 4. The dosages of copper sulphate tested are 0, 50, 100, 200, 300, 400 and 500 g/t, with detailed results in Table 18 and Table 19 in Appendix 1.0 for 350 g tests and 1 kg tests, respectively. Extensive tests were done using 350 g ore samples (CS-1 to CS-13) before working with the 1 kg ore samples. Tests using 350 g ore samples indicated that in order to make the copper sulphate act

effectively, pH must be increased incrementally. When copper sulphate was conditioned at pH 9, the optimum pH for conditioning the collector, sphalerite grades and recoveries of the zinc rougher concentrates were poor. This appeared to be due to precipitation of copper sulphate at pH 9. A procedure was then standardized in which the pH was increased to 7.5 for the copper sulphate conditioning and then further increased to pH 9 for conditioning with the collector. This modification in the procedure gave consistent and reproducible results and the effect of adding copper sulphate was clearly observed.

**Table 4: Experimental conditions for effect of copper sulphate tests using 1 kg ore samples**

Test	Stage	Time (min.)	MBS (g/t)	ZnSO <sub>4</sub> (g/t)	NaCN (g/t)	PEX (g/t)	CuSO <sub>4</sub> (g/t)	HCNPHA (g/t)	MIBC (g/t)	Air (Lpm)	pH
CS-14 to CS-20	Grind	40	150	100					20	2.5	
	Pre-float	5									
	Pb-R Cond.-1	1		50		100			10	2.5	6.06-6.31
	Pb-R Cond.-2	1									
	Pb-R Float	10									
	Zn-R Cond.-1	1					50-500				
	Zn-R Cond.-2	2						200			
	Zn-R Float	8							10	2.5	9.02-9.16

The sphalerite grades and recoveries of the zinc rougher concentrates for tests CS-14 to CS-20 (using 1 kg ore sample) are shown in Figure 18.



**Figure 18: Effect of copper sulphate dosage on sphalerite recovery and grade**

From these tests it was concluded that there was no significant change in either the recovery or the grade of sphalerite past a dosage of 200 g/t of copper sulphate. The copper sulphate dosage of all subsequent tests (unless otherwise stated) was set at 200 g/t. This dosage was set solely on the sphalerite recovery/grade and not on the amount of the gangue floated because the grade and recovery of gangue remained relatively constant throughout all the tests, with a consistent grade/recovery of pyrite and NSG being approximately 16% / 45% and 30% / 40%, respectively, for each test.

### ***4.3 Improvement of Sphalerite Grade using Suppressants***

Extensive testing was conducted to improve the sphalerite grade in the zinc rougher stage by regulating the frothing characteristics of HCNPHA, and by suppressing gangue minerals. Based on previous results (Section 4.1.1) wherein CMC and polysaccharides showed initial signs of pyrite suppression, two polysaccharides and seven CMC's were chosen for further testing. The CMC's varied in their molecular masses with their exact chemical composition and the details are not discussed in order to guard the intellectual property. DETA (Joly et al., 2004) and sodium silicate were also tested as they have been used as suppressants for sulphidic and non-sulphidic gangue (NSG), respectively, in the mineral processing industry. This batch of tests on suppressants was conducted over three sphaes, namely;

1. Preparing a list of the suppressants and select the best suppressant from each type (polysaccharide, CMC and amine).
2. Determining the optimum dosage for each suppressant.
3. Combining the sulphidic gangue suppressants with the non-sulphidic gangue suppressant (sodium silicate) and optimizing the conditions.

#### ***4.3.1 Short-listing Suppressants***

DETA, two polysaccharides and seven CMC's were all tested at a dosage of 100 g/t (tests SP-1 to SP -15). This dosage was arbitrarily chosen and each suppressant was tested individually. Each test was conducted using 350 g ore samples (refer to Section 3.9.1: Batch Flotation for procedure) with the specific details of the experimental parameters given in Table

3. Since the objective of these tests was to improve the grade of sphalerite without overly affecting the recovery, grade ratio (GR) was used to compare the results (Eqn. 1).

$$\text{Grade ratio} = \frac{\text{Valuable mineral grade}}{\text{Gangue mineral grade}} \quad (1)$$

The valuable mineral in this study was of course sphalerite collected in the float concentrate.

**Table 5: Batch flotation results for sulphidic gangue suppressants**

Sample	Stage	Grade (wt %)				Recovery (%)				
		Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Wt. %
Baseline	Pb-R	23.9	28.6	13.1	34.3	54.4	9.6	8.9	6.7	9.5
	Zn-R	2.9	46.6	16.8	33.6	37.1	86.7	63.0	41.2	52.6
	Tail	0.9	2.8	10.4	85.9	8.5	3.7	28.1	52.1	37.9
	<b>Feed</b>	<b>4.2</b>	<b>28.3</b>	<b>14.1</b>	<b>53.5</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CMC-1	Pb-R	28.7	26.6	13.2	31.5	58.5	8.8	8.8	6.5	9.3
	Zn-R	3.0	47.4	16.7	32.9	32.6	84.4	59.9	41.3	52.4
	Tail	1.1	5.2	11.9	81.7	9.0	6.8	31.4	52.2	38.4
	<b>Feed</b>	<b>4.6</b>	<b>29.3</b>	<b>14.5</b>	<b>51.5</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CMC-2	Pb-R	25.6	27.5	13.7	33.3	52.7	8.4	8.5	6.8	10.5
	Zn-R	3.2	45.9	17.7	33.3	40.0	86.7	67.5	47.0	57.0
	Tail	1.0	4.5	11.1	83.4	7.3	4.8	24.0	46.2	32.4
	<b>Feed</b>	<b>4.8</b>	<b>30.5</b>	<b>15.1</b>	<b>49.5</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CMC-3	Pb-R	25.5	27.6	14.3	32.6	51.8	8.4	8.7	6.9	11.2
	Zn-R	3.2	46.8	17.4	32.6	39.8	86.9	64.3	47.4	57.0
	Tail	1.2	4.6	13.1	81.1	8.4	4.7	27.0	45.6	31.8
	<b>Feed</b>	<b>5.1</b>	<b>31.2</b>	<b>15.7</b>	<b>48.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CMC-4	Pb-R	29.8	26.0	14.6	29.5	52.1	7.4	8.2	7.1	12.7
	Zn-R	3.7	47.8	18.7	29.9	41.4	87.5	67.7	53.3	61.1
	Tail	1.3	6.5	15.5	76.7	6.5	5.1	24.1	39.6	26.2
	<b>Feed</b>	<b>6.4</b>	<b>34.2</b>	<b>17.3</b>	<b>42.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CMC-5	Pb-R	26.7	26.5	14.1	32.7	53.2	8.3	8.8	6.9	11.1
	Zn-R	3.3	46.3	17.7	32.7	39.5	86.1	65.2	46.6	56.3
	Tail	1.1	5.2	12.1	81.6	7.3	5.6	26.0	46.5	32.6
	<b>Feed</b>	<b>5.2</b>	<b>30.7</b>	<b>15.5</b>	<b>48.6</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

Sample	Stage	Grade (wt %)				Recovery (%)				Wt. %
		Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	
CMC-6	Pb-R	26.9	27.7	13.6	31.7	46.9	8.3	7.9	7.1	10.7
	Zn-R	4.5	49.8	20.0	25.6	42.9	81.4	63.0	40.2	52.1
	Tail	1.5	8.9	13.0	76.6	10.2	10.3	29.1	52.7	37.2
	<b>Feed</b>	<b>5.8</b>	<b>32.2</b>	<b>16.7</b>	<b>45.2</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CMC-7	Pb-R	19.1	29.1	13.6	38.2	42.2	7.9	8.1	8.5	11.2
	Zn-R	3.2	47.0	17.7	32.1	49.9	89.2	74.4	59.6	67.7
	Tail	1.6	5.0	13.4	79.9	7.9	3.0	17.5	31.9	21.1
	<b>Feed</b>	<b>4.7</b>	<b>36.1</b>	<b>16.4</b>	<b>42.9</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
PS-1	Pb-R	24.5	27.2	12.4	35.9	47.0	9.4	7.9	7.0	7.7
	Zn-R	4.0	46.3	16.9	32.8	39.7	83.4	56.5	39.1	51.2
	Tail	1.6	5.0	13.2	80.2	13.3	7.2	35.5	53.9	41.2
	<b>Feed</b>	<b>4.6</b>	<b>27.8</b>	<b>15.0</b>	<b>52.6</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
PS-2	Pb-R	23.1	33.2	15.5	28.2	48.2	9.8	9.1	7.0	12.3
	Zn-R	3.5	48.7	19.8	28.0	42.2	83.5	67.8	45.9	55.9
	Tail	1.4	6.9	11.9	79.8	9.7	6.7	23.2	47.1	31.8
	<b>Feed</b>	<b>5.2</b>	<b>33.5</b>	<b>16.8</b>	<b>44.5</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
DETA	Pb-R	17.7	37.0	15.0	30.3	49.1	11.8	10.0	7.2	12.5
	Zn-R	2.6	46.1	16.1	35.2	40.5	83.1	60.3	47.5	55.5
	Tail	1.1	4.9	13.8	80.2	10.4	5.1	29.7	45.3	32.0
	<b>Feed</b>	<b>4.0</b>	<b>31.8</b>	<b>15.2</b>	<b>49.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

Three percent of iron is always associated with sphalerite (Ref. private communication from *Teck Cominco*); hence, the pyritic iron mentioned in this section and the following sections was calculated as follows:

$$\text{Py (pyritic iron)} = \text{Total iron} - \text{iron in sphalerite}$$

Table 6 displays the grade ratio for each test. In the baseline test (SP-1) no suppressant was added, it gave a grade ratio of 2.77 for pyrite and 1.38 for NSG. The grade ratios of all other tests were compared with these values. CMC-1, DETA and PS-1 all achieved grade ratios equal to or higher than that of the baseline for pyrite showing that for these three reagents, suppression

of pyrite and non-sulphidic gangue was observed. This is an indication that pyrite suppression is taking place.

**Table 6: Grade ratios with respect to pyrite and NSG in the float concentrate**

Suppressant	Grade Ratio	
	Pyrite	NSG
<i>Baseline</i>	2.77	1.38
<b>CMC-1</b>	<b>2.84</b>	<b>1.44</b>
CMC-2	2.60	1.38
CMC-3	2.69	1.44
CMC-4	2.56	1.60
CMC-5	2.62	1.42
CMC-6	2.49	1.95
CMC-7	2.65	1.46
<b>PS-1</b>	<b>2.74</b>	<b>1.41</b>
PS-2	2.45	1.74
<b>DETA</b>	<b>2.87</b>	<b>1.31</b>

(Note: data in bold represents significant results)

Many of the suppressants tested showed promise in suppressing non-sulphidic gangue minerals with CMC-6 having the highest grade ratio (1.95). However, it was decided to continue further tests with CMC-1 instead of CMC-6 due to CMC-1 rejecting less sphalerite to the tails (CMC-1 left 6.3% sphalerite in the tails while CMC-6 left 10.3% in the tails). PS-1 and DETA both had minimal rejection of sphalerite to the tails (5.0% and 4.9% respectively) with all three suppressants achieving a sphalerite recovery of greater than 83%. It was decided to conduct further testing only with these three suppressants to determine their optimum dosage.

#### **4.3.2a Effect of Varying the Pyrite Suppressants Dosage**

It was decided to test CMC-1, PS-1 and DETA as pyrite suppressants. The optimum dosage for each suppressant was determined by using the experimental parameters outlined in Table 3. Each suppressant was tested at the following dosages; 100, 150, 200 and 250 g/t (SP-16



to SP-31). The grades and recoveries of the minerals for these batch flotation tests are given in Table 6 with detailed results in Appendix 1.0 - Table 21.

It was important to maximize the sphalerite grade, without affecting the recovery significantly. Also, any improvement in the sphalerite grade was a direct result of reductions in both sulphidic and non-sulphidic gangue minerals.

**Table 7: Batch flotation results for varying suppressant dosage**

Sample	Stage	Grade (wt %)				Recovery (%)				
		Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Wt. %
CMC-1 100 g/t	Pb-R	28.7	26.6	13.2	31.5	58.5	8.8	8.8	6.5	9.3
	Zn-R	3.0	47.4	16.7	32.9	32.6	84.4	59.9	41.3	52.4
	Tail	1.1	5.2	11.9	81.7	9.0	6.8	31.4	52.2	38.4
	<b>Feed</b>	<b>4.6</b>	<b>29.3</b>	<b>14.5</b>	<b>51.5</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CMC-1 150 g/t	Pb-R	30.2	26.3	12.6	30.9	50.9	7.5	7.4	6.7	9.8
	Zn-R	4.2	53.2	18.0	24.6	36.8	79.2	55.5	37.2	49.5
	Tail	1.7	10.9	14.6	72.9	12.3	13.3	37.1	56.0	40.7
	<b>Feed</b>	<b>5.7</b>	<b>33.3</b>	<b>16.1</b>	<b>44.9</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CMC-1 200 g/t	Pb-R	28.1	27.1	12.7	32.1	50.6	7.8	7.6	6.8	9.9
	Zn-R	4.2	53.8	18.4	23.6	36.7	75.0	53.5	32.7	45.0
	Tail	1.5	12.4	13.4	72.8	12.7	17.2	38.9	60.5	45.0
	<b>Feed</b>	<b>5.3</b>	<b>32.5</b>	<b>15.6</b>	<b>46.6</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CMC-1 250 g/t	Pb-R	28.9	27.6	12.7	30.9	52.9	8.6	8.0	6.5	9.7
	Zn-R	4.3	54.6	17.7	23.4	33.5	73.2	48.4	28.7	41.0
	Tail	1.4	11.3	13.3	74.0	13.6	18.2	43.6	64.8	49.3
	<b>Feed</b>	<b>5.3</b>	<b>30.7</b>	<b>15.1</b>	<b>49.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
PS-1 100 g/t	Pb-R	24.5	27.2	12.4	35.9	47.0	9.4	7.9	7.0	7.7
	Zn-R	4.0	46.3	16.9	32.8	39.7	83.4	56.5	39.1	51.2
	Tail	1.6	5.0	13.2	80.2	13.3	7.2	35.5	53.9	41.2
	<b>Feed</b>	<b>4.6</b>	<b>27.8</b>	<b>15.0</b>	<b>52.6</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

Sample	Stage	Grade (wt %)				Recovery (%)				
		Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Wt. %
PS-1 150 g/t	Pb-R	29.6	27.7	14.2	28.5	49.1	7.6	7.5	6.9	11.5
	Zn-R	4.0	48.8	18.2	29.0	43.0	86.2	62.3	52.2	60.6
	Tail	1.6	7.6	19.0	71.7	7.9	6.2	30.1	40.9	28.0
	<b>Feed</b>	<b>6.3</b>	<b>34.9</b>	<b>18.0</b>	<b>40.9</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
PS-1 200 g/t	Pb-R	30.0	26.6	13.7	29.7	54.5	7.6	7.9	6.9	11.7
	Zn-R	3.3	48.1	16.8	31.7	37.9	86.1	60.3	51.7	59.4
	Tail	1.4	7.2	18.3	73.1	7.6	6.3	31.8	41.4	28.8
	<b>Feed</b>	<b>5.9</b>	<b>33.8</b>	<b>16.9</b>	<b>43.4</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
PS-1 250 g/t	Pb-R	31.1	28.7	13.6	26.6	50.6	8.2	7.9	6.4	10.8
	Zn-R	4.1	50.7	17.1	28.1	38.6	84.3	57.7	44.8	55.1
	Tail	1.9	7.3	16.5	74.3	10.8	7.5	34.4	48.8	34.0
	<b>Feed</b>	<b>6.2</b>	<b>33.6</b>	<b>16.5</b>	<b>43.7</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
DETA 100 g/t	Pb-R	17.7	37.0	15.0	30.3	49.1	11.8	10.0	7.2	12.5
	Zn-R	2.6	46.1	16.1	35.2	40.5	83.1	60.3	47.5	55.5
	Tail	1.1	4.9	13.8	80.2	10.4	5.1	29.7	45.3	32.0
	<b>Feed</b>	<b>4.0</b>	<b>31.8</b>	<b>15.2</b>	<b>49.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
DETA 150 g/t	Pb-R	23.0	41.0	13.1	23.0	52.4	13.7	8.8	6.1	12.0
	Zn-R	2.8	45.0	15.0	37.2	30.8	71.9	48.4	41.3	47.6
	Tail	1.8	10.6	15.6	72.0	16.8	14.4	42.7	52.7	40.4
	<b>Feed</b>	<b>4.8</b>	<b>30.6</b>	<b>15.0</b>	<b>49.5</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
DETA 200 g/t	Pb-R	26.5	28.9	13.2	31.4	53.1	8.8	8.1	7.0	10.7
	Zn-R	3.1	46.3	15.4	35.1	33.7	75.9	50.5	45.0	52.1
	Tail	1.7	13.1	17.7	67.4	13.2	15.3	41.4	48.0	37.2
	<b>Feed</b>	<b>5.1</b>	<b>32.1</b>	<b>16.1</b>	<b>46.7</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
DETA 250 g/t	Pb-R	23.7	28.7	13.5	34.1	45.9	8.1	7.8	7.5	10.8
	Zn-R	3.8	49.0	16.1	31.1	41.4	77.1	51.7	44.3	52.7
	Tail	1.7	13.6	18.3	66.4	12.7	14.8	40.6	48.2	36.5
	<b>Feed</b>	<b>5.2</b>	<b>33.9</b>	<b>16.6</b>	<b>44.3</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

During these tests, it was interesting to observe that increasing the dosage of CMC-1 had a marked affect on the frothing characteristics of the pulp. With the ability to regulate the froth, entrainment of gangue minerals in 'foamy' froth was reduced and this resulted in higher sphalerite grades (baseline sphalerite grade 46.6%; CMC-1 at 250 g/t sphalerite grade 54.6%). This phenomenon was also exhibited to a lesser extent by PS-1. A clearer picture of the dosage

that maximized sphalerite grade (by depressing gangue minerals) is given by the grade ratios as seen in Table 8.

**Table 8: Grade ratios with respect to pyrite grade and NSG grade in the float concentrate**

Suppressant	Grade Ratio	
	Pyrite	NSG
<i>Baseline</i>	2.77	1.38
CMC-1 100 g/t	2.84	1.44
CMC-1 150 g/t	2.96	2.17
CMC-1 200 g/t	2.92	2.28
<b>CMC-1 250 g/t</b>	<b>3.08</b>	<b>2.33</b>
PS-1 100 g/t	2.74	1.41
PS-1 150 g/t	2.69	1.68
PS-1 200 g/t	2.86	1.52
<b>PS-1 250 g/t</b>	<b>2.97</b>	<b>1.81</b>
DETA 100 g/t	2.87	1.31
DETA 150 g/t	3.00	1.21
DETA 200 g/t	3.01	1.32
<b>DETA 250 g/t</b>	<b>3.04</b>	<b>1.58</b>

(Note: data in bold represent the best results obtained for each suppressant.)

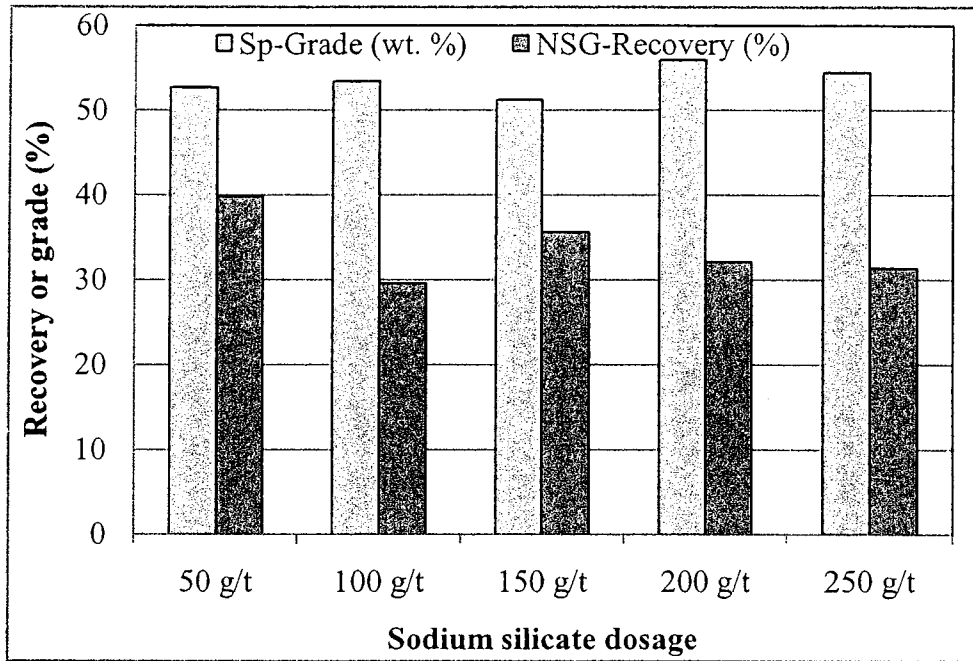
From the grade ratios given above, noticeable improvement with respect to sulphidic gangue is seen in all suppressants with CMC-1 at 250 g/t having the best result of 3.08 and DETA achieving a similar result of 3.04. CMC-1 had the highest grade ratio with respect to non-sulphidic gangue as well due to its ability to regulate the frothing characteristics preventing gangue minerals from being entrained in the float concentrate. PS-1 showed the second highest improvement in terms of non-sulphidic gangue with an increase from 1.41 to 1.81, and this can be attributed to the slight effect it had on the frothing properties. Higher dosages were not tested for CMC-1 as CMC's are known to increase slurry viscosity at higher concentrations and also because a decrease in sphalerite recovery was observed with increasing the dosage, 100 g/t CMC had 84.4 % sphalerite recovery; 250 g/t CMC had 73.2% sphalerite recovery (~12% reduction).

This drop-off in recovery was not observed for DETA or PS-1. DETA shows only minor improvements in grade ratio in terms of sulphidic gangue beyond 150 g/t, as well as displaying no ability to regulate the frothing characteristics. Therefore, it was decided to use 250 g/t CMC-1, 250 g/t PS-1 and 150 g/t DETA in further tests.

#### ***4.3.2b Effect of NSG Suppressant Dosage***

The dosage of the three selected sulphidic gangue suppressants (CMC-1, PS-1 and DETA) was optimized. The frothing characteristics of HCNPHA were regulated by the addition of CMC. It was then decided to add sodium silicate to suppress silica (~55 %). Currently in the mineral processing industry sodium silicate ( $\text{Na}_2\text{SiO}_3$ ) is used as a depressant for NSG such as silica. Experimental conditions for these batch flotation tests (SP-32 to SP-39) are given in Table 3. Figure 19 shows the plot of sphalerite grade and NSG recovery for tests Sp-32 to Sp-39 and the details of each test is shown in Table 22 in Appendix 1.0.

Comparing the grades of sphalerite to the baseline test from section 3.3.1, there has been an increase from 46.6 to 55.9%. This is a significant increase and supports the observations that entrainment of gangue minerals, specifically NSG such as silica, into the froth was responsible for the low sphalerite grades. The recovery of NSG was decreased from 41.2% observed in the baseline test to 32.1%, using 200 g/t sodium silicate. Increasing the sodium silicate dosage beyond 200 g/t did not significantly improve the grade of sphalerite. Hence, it was decided to use 200 g/t sodium silicate in conjunction with CMC-1 and PS-1 in order to achieve maximum sphalerite grade by suppressing both sulphidic and non-sulphidic gangue minerals.

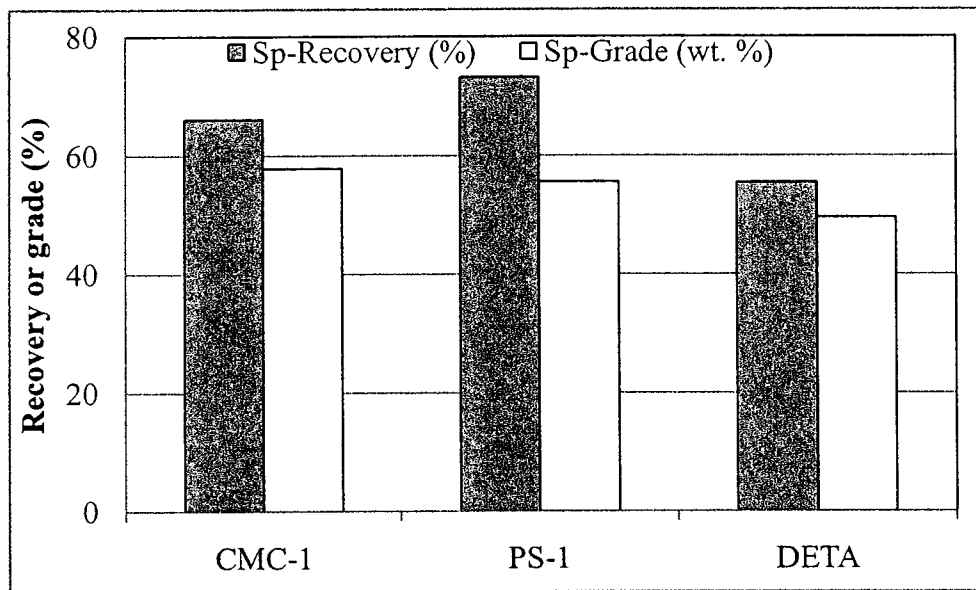


**Figure 19: Effect of varying the dosage of sodium silicate on sphalerite grade and NSG recovery**

#### ***4.3.3 Effect of Combing both Sulphidic and Non-sulphidic Gangues Suppressants***

In the previous section (Section 4.3.2a), it was decided to continue testing with the following sulphidic gangue suppressants at the indicated dosages: CMC-1 (250 g/t), PS-1 (250 g/t) and DETA (150 g/t). Sodium silicate was also chosen as a non-sulphidic gangue suppressant and was used at a dosage of 200 g/t. The next set of batch flotation experiments combined each sulphidic gangue suppressant with sodium silicate at their stated dosages. Experimental parameters are shown in Table 3; this procedure is modified in that both types of suppressant (for sulphidic and non-sulphidic gangue) are added at the same time and conditioned together for two minutes before the copper sulphate is introduced to the system. Figure 20 shows the plot of sphalerite recovery and grade for the zinc rougher stage for these tests (SP-40 to SP-52).

The result of combining both types of suppressants had a noticeable effect on the recovery of sphalerite, dropping from ~80% from individual suppressant tests to ~60%, which indicated that dosage for each suppressant needed to be adjusted when used in conjunction with sodium silicate. While the highest sphalerite grade (57.8%) was achieved with this batch of



**Figure 20: Combining sulphidic and non-sulphidic gangue suppressants and their effect on sphalerite grade and recovery**

tests, further optimization was needed because the results indicated that at these dosages the suppressants indiscriminately masked all sites and not just gangue minerals. However, from these initial tests, it was decided to drop DETA because it did not show as much promise as the other suppressants, having the lowest sphalerite grade and recovery as well as displaying no frothing regulating properties. PS-1 also was not used in further testing even though it produced results comparable to that of CMC-1, because its ability to regulate froth was less than that of the CMC. Therefore, it was decided to use CMC-1 in order to improve the sphalerite grade.

Extensive testing was done changing several variables independently, such as: CMC dosage, sodium silicate dosage, pH modifier, amount of frother added, etc (refer to tests SP-49 to

SP-54 in Table 23 in Appendix 1.0 Flotation Data). Changing the dosages of the suppressants slightly improved the recovery without compromising the grade; however, the NSG recovery values increased significantly. It was therefore decided to keep the dosages at 250 g/t CMC-1 and 200 g/t sodium silicate. Reducing and/or eliminating the frother from some scouting tests resulted in a noticeable decrease in sphalerite recovery (from 66.1% down to 54.4%). Replacing sodium hydroxide with quick lime as the pH modifier brought no significant change to the sphalerite grade (57.8 to 60.1%) or recovery (66.1 to 70.9%). Due to economic reasons, quick lime was preferred as the pH modifier instead of sodium hydroxide.

#### ***4.4 Flotation Kinetics***

For the tests on flotation kinetics, the same procedure for 1 kg ore samples as outlined in Section 3.9.1: Batch flotation was used. However, instead of collecting for a set time as in batch flotation, the zinc rougher was collected at specific time intervals (1, 2, 3, 5, 8, 14, and 22 minutes). The reagents were added in the same dosages with the same conditioning time as outlined in Table 4 with the exception of the frother which was added in three stages (0; 5 and 10 minutes) in order to maintain constant frothing characteristics of the pulp. Froth density is a measure of the amount of solids contained in the froth. A denser froth contains more solids and, since the valuable mineral (sphalerite) is denser than the gangue minerals (both sulphidic and non-sulphidic gangues), an increase in froth density was attributed to an increase in sphalerite grade. Froth density was also measured for each cut to give an indication of where the grade of sphalerite starts to decrease and gangue minerals are starting to float in relative abundance. Effects of several variables on sphalerite recovery and grade were studied by following the kinetics of flotation. These include: adding each reagent individually; then the order in which

the reagents were to be added to the system, air flow rate, and collector concentration. Since sphalerite is the valuable mineral of interest, all experimental results were focused on the zinc rougher stage in order to understand the surface modification and differential flotation of the minerals present.

#### ***4.4.1 Effect of Reagents Order on Flotation Kinetics***

Flotation kinetics was studied using 200 g/t HCNPHA as collector: (a) baseline using only HCNPHA with no auxiliary reagents added (KT-1), (b) HCNPHA with 200 g/t copper sulphate (KT-2), (c) HCNPHA with 200 g/t copper sulphate and 250 g/t CMC-1 (KT-3) and (d) HCNPHA with 200 g/t copper sulphate, 250 g/t CMC-1 and 250 g/t sodium silicate (KT-4).



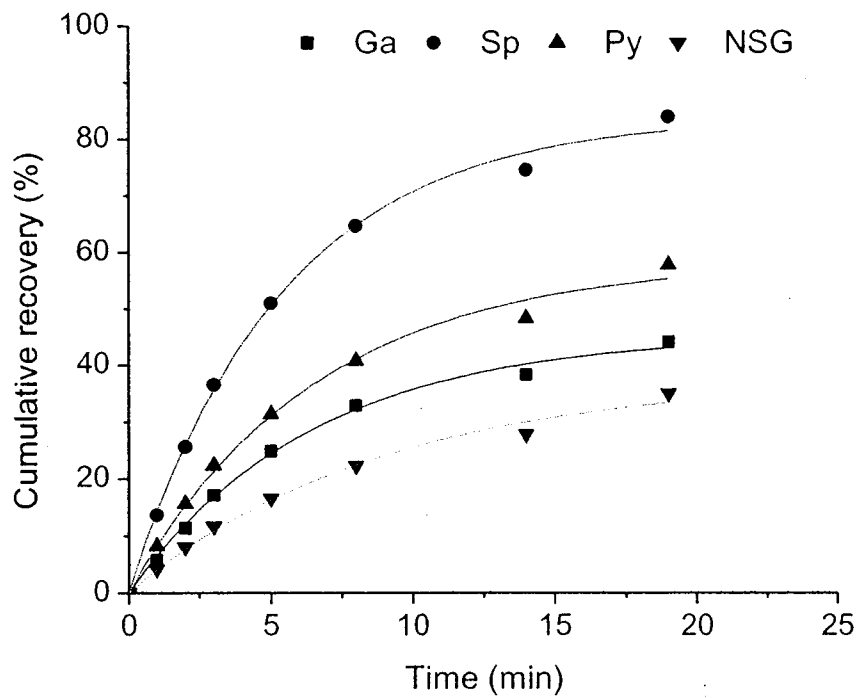
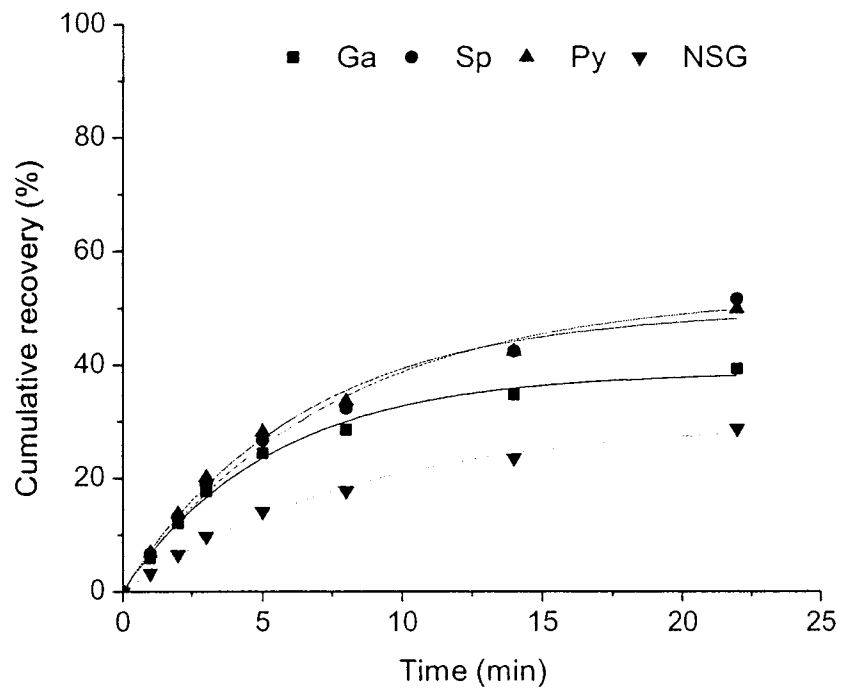


Figure 21a) 200 g/t HCNPHA no auxiliary reagents added (KT-1); b) 200 g/t copper sulphate and 200 g/t HCNPHA (KT-2)

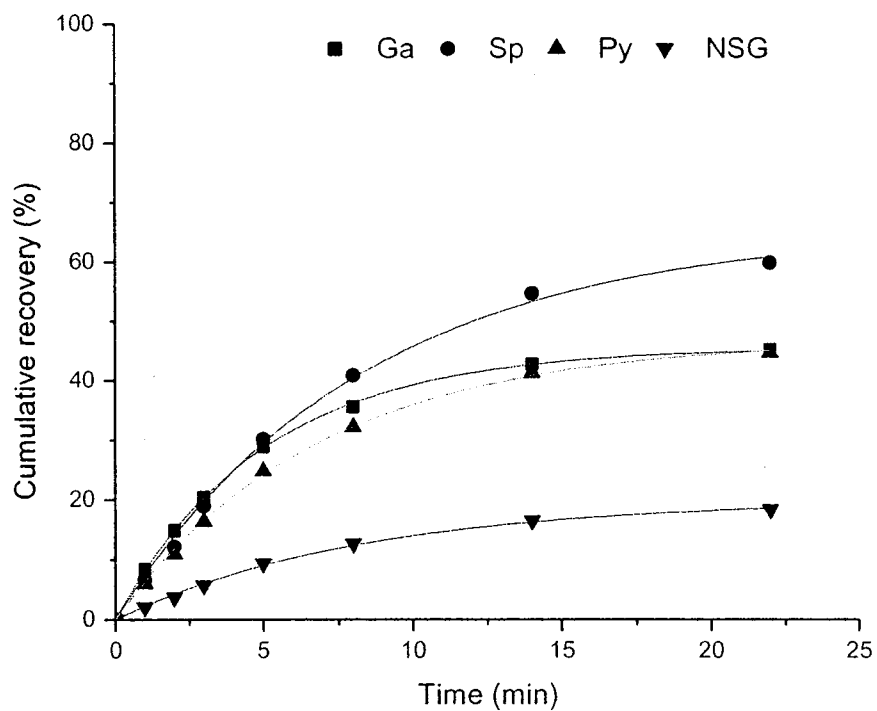
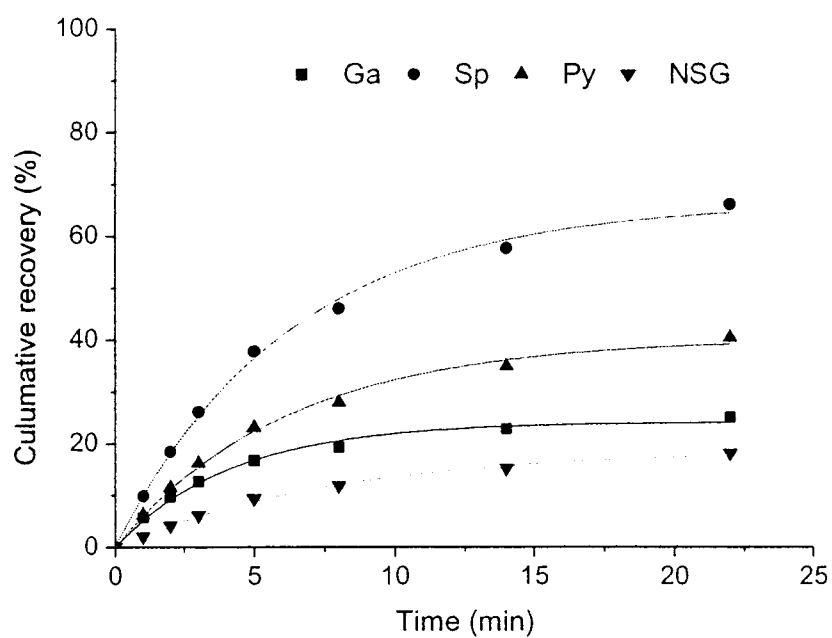


Figure 22a) 250 g/t CMC-1, 200 g/t copper sulphate and 200 g/t HCNPHA (KT-3); b) 250 g/t sodium silicate, 250 g/t CMC-1, 200 g/t copper sulphate and 200 g/t HCNPHA (KT-4)

It was observed that upon the addition of copper sulphate, the recovery of sphalerite improved significantly (Figure 21a and 21b) without overly affecting the recovery of the other minerals. This produced a better sphalerite grade in the float concentrates. The recovery and grade of sphalerite are shown in Table 9. This matched the results obtained from the “Effect of copper sulphate”, Section 4.2.4. Using copper sulphate improved the maximum sphalerite recovery ( $R_{\infty}$ ) from 53% to 83% with only an increase of 8% for pyrite and non-sulphidic gangue which supports preferential adsorption of copper cation species on zinc sites and the facilitation of chelation by HCNPHA.

**Table 9: Sphalerite grade and recovery for kinetics tests**

Test	Mass Floated wt. %	Sp. Recovery %	Sp. Grade wt. %
KT-1	39.3	51.6	39.4
KT-2	53.1	83.9	46.3
KT-3	36.9	66.1	54.9
KT-4	37.1	59.8	50.2

While testing the effect of CMC-1 on the kinetics of flotation, differential flotation was still maintained however there was a noticeable decrease in the recovery of sphalerite which could suggest that CMC-1 was indiscriminately masking the zinc sites. This could be contingent from the drop in recoveries of all the minerals. Time recovery plots and the grade of the concentrates, do allow for some qualitative observations to compare the effect of auxiliary chemicals, however, quantitative estimation of the increase or decrease in selectivity for a given mineral was not possible.

The selectivity for a collector, HCNPHA in this specific case, for a certain mineral over other minerals under given experimental conditions depends on the first-order rate constant ( $k$ )

and the value of maximum recovery ( $R_\infty$ ). Therefore, it is challenging to interpret any change in selectivity between a valuable mineral and a gangue mineral from their recoveries alone. Xu (1998), suggested modified rate constant  $K_m$  to overcome this and  $K_m$  whose value is determined by the following equation:

$$K_m = R_\infty \times k \quad (2)$$

This modified rate constant then can be used to define the selectivity index ( $SI$ ) or the relative rate constant of one mineral ( $M_1$ ) to that of the other mineral ( $M_2$ ). The following equation defines the selectivity index:

$$SI_{(M_1/M_2)} = \frac{K_m \text{ of } M_1}{K_m \text{ of } M_2} \quad (3)$$

Hence, taking the case of defining the selectivity index with respect to a valuable mineral  $M$  and a gangue mineral  $G$  the equation can be transformed into:

$$SI_{(M/G)} = \frac{K_{mM}}{K_{mG}} \quad (4)$$

The selectivity index allows for the quantification of the selectivity of a collector (HCNPHA) for a particular mineral (sphalerite) over another mineral (gangue) under a given set of system variables. The factors that have a beneficial effect on SI in order to maximize the separation between the selected minerals should be discussed. The modified first-order rate equation for flotation kinetics is:

$$R_t = R_\infty(1 - e^{-k(t+\phi)}) \quad (5)$$

where  $R_t$  is the cumulative recovery at time  $t$ ,  $R_\infty$  is the maximum recovery or the cumulative recovery at time  $\infty$ ,  $k$  is the first-order rate constant and  $\phi$  is the time correction factor. Using Eqn. 5 was used to fit the time recovery plots in Figure 21a,b and Figure 22a,b, the first-order rate constant and the maximum recovery of the minerals were obtained by curve fitting. The free-ware CurveExpert 3.1 was used for curve fitting. The time-recovery curves obtained have a  $r^2 \geq 0.97$ . Equations (2) and (4) were used to calculate the modified rate constant and the selectivity index. The calculated parameters for the flotation kinetic tests are given in Table 10.

**Table 10: Flotation kinetic parameters for effect of reagent addition tests**

Test	Mineral	k	$R_\infty$	Km	SI wrt NSG	SI wrt Py
KT-1	Sp	0.125	53.4	6.67	1.93	0.88
	NSG	0.113	30.6	3.46		
	Py	0.151	50.1	7.56		
KT-2	Sp	0.186	83.4	15.52	3.73	1.78
	NSG	0.112	37.2	4.17		
	Py	0.150	58.1	8.72		
KT-3	Sp	0.126	64.8	8.16	3.29	1.18
	NSG	0.126	19.7	2.48		
	Py	0.148	46.8	6.93		
KT-4	Sp	0.151	67.8	10.24	4.37	1.63
	NSG	0.119	19.7	2.34		
	Py	0.152	41.3	6.28		

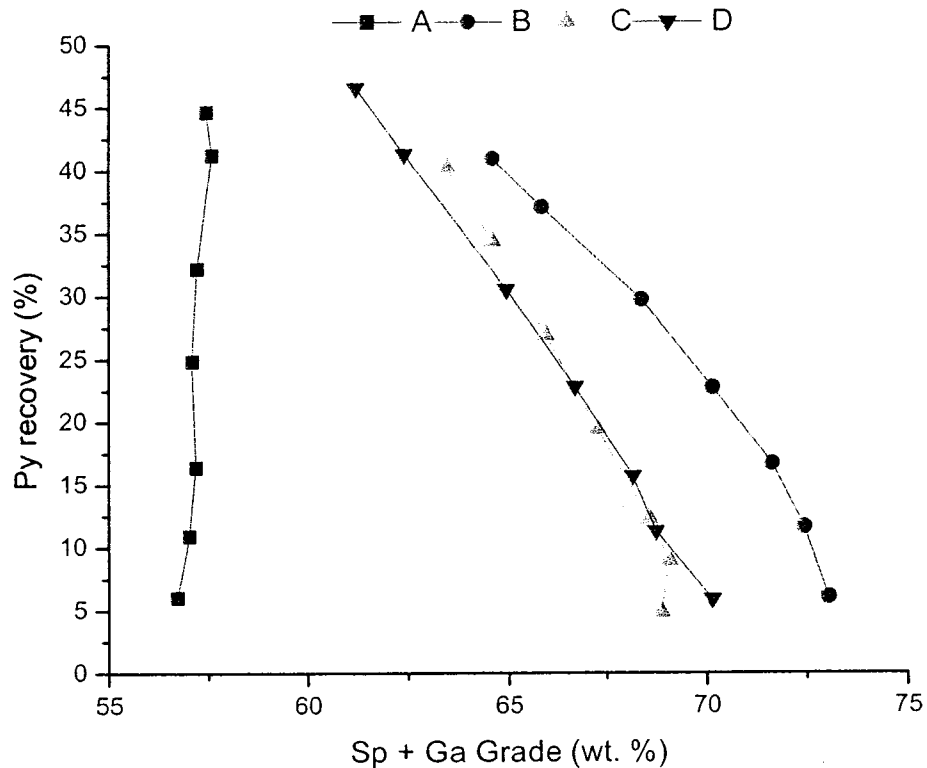
Addition of 200 g/t copper sulphate increased the selectivity of HCNPHA with respect to pyrite from 0.88 to 1.78 and for non-sulphidic gangue from 1.93 to 3.73. It appears that suppression of gangue minerals is taking place; however, a closer inspection of the first-order rate constant ( $k$ ) reveals that  $k$  remains practically unaltered for both types of gangue minerals but is significantly increased for sphalerite (from  $0.125 \text{ min}^{-1}$  to  $0.186 \text{ min}^{-1}$ ). Therefore it can be surmised that copper sulphate facilitates differential flotation and preferentially modifies the

sphalerite surface in comparison to the other minerals. The modified rate constant ( $K_m$ ) for pyrite decreased from 8.72 to 6.93 (~20% drop) while the  $K_m$  for non-sulphidic gangue decreased from 4.17 to 2.48 (~40% drop) for experiments KT-2 and KT-3. This indicates that CMC-1 helped suppress NSG more than pyrite. The addition of sodium silicate, test KT-4, helped further reduce the modified rate constant for both pyrite and non-sulphidic gangue.

#### ***4.4.2 Impact of Changing Reagents Order***

Upon the addition of CMC-1, a drop was observed in sphalerite recoveries as well as selectivity with respect to both sulphidic and non-sulphidic gangue. Having previously optimized the other variables (air flow rate, particle size, etc.), it was decided to look into the order in which the reagents should be added to determine the cause of the drop in values. Four different variations were tested:

- A. CMC-1 and sodium silicate added prior to copper sulphate and collector.
- B. Copper sulphate added and conditioned before CMC-1 and sodium silicate.
- C. CMC-1 and sodium silicate added after conditioning with copper sulphate and collector.
- D. Sodium silicate added first, and then copper sulphate and collector followed by CMC-1.



**Figure 23: Pyrite recovery versus concentrate (Sp+Ga) grade for reagent order tests**

By plotting the pyrite recovery *versus* the sphalerite plus galena grade in the concentrate, it can be determined that scheme (B) is the most advantageous way to add the reagents as seen in Figure 23. Test B gives the highest concentrate grade for any amount of pyrite recovered. This is more clearly defined when looking at the flotation kinetics parameters and selectivity indices (*SI*) in Table 11. For tests A and B, there is really no change in the rate constants for the gangue minerals whereas the sphalerite rate constant is noticeably improved (0.151 to 0.229).

The ultimate recoveries for the gangue minerals were similar for each test. Also, it was noted that the sphalerite recovery was highest in tests B and D: 75.2% and 78.6%, respectively. However, adding copper sulphate first to re-activate the suppressed sphalerite from the lead

rougher stage and then adding the auxiliary chemicals, CMC-1 and sodium silicate, greatly improved the *SI* in regards to both pyrite and NSG, increasing it from 4.37 and 1.64 for pyrite and NSG, respectively, for scheme (A) to 8.01 and 2.58 for scheme (B). Therefore the preferred order of addition of various reagents in the proposed reagent scheme was found to be:

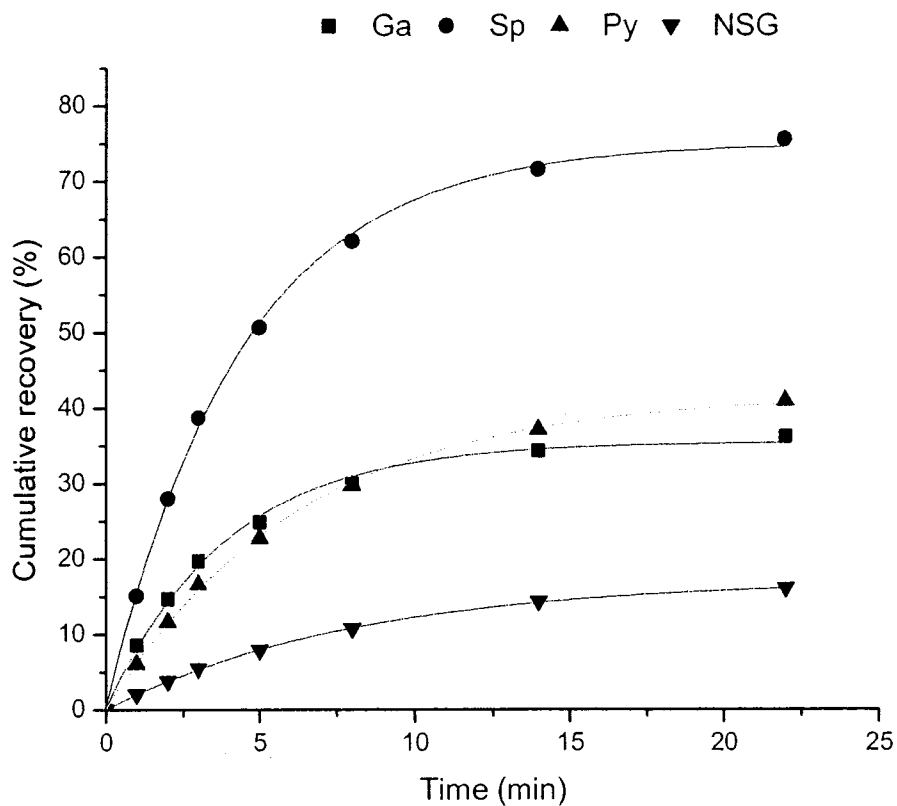
- Condition lead-rougher tails with copper sulphate to activate the sphalerite suppressed in the lead rougher stage. Then add CMC-1 and sodium silicate simultaneously and condition, finally, condition with collector (HCNPHA).

Figure 24 shows the cumulative recovery *versus* time using the preferred reagent addition scheme mentioned above. This was found to give the best result.

**Table 11: Flotation kinetics parameters for effect of reagent order tests**

Test	Mineral	k	$R_{\infty}$	Km	SI wrt NSG	SI wrt Py
A	Sp	0.151	67.8	10.24	4.37	1.63
	NSG	0.119	19.7	2.34		
	Py	0.152	41.3	6.28		
B	Sp	0.229	75.2	17.22	8.01	2.58
	NSG	0.125	17.2	2.15		
	Py	0.159	42.0	6.68		
C	Sp	0.136	69.0	9.38	5.92	1.85
	NSG	0.092	17.2	1.58		
	Py	0.117	43.3	5.07		
D	Sp	0.164	78.6	12.89	6.45	2.12
	NSG	0.098	20.4	2.00		
	Py	0.121	50.2	6.07		

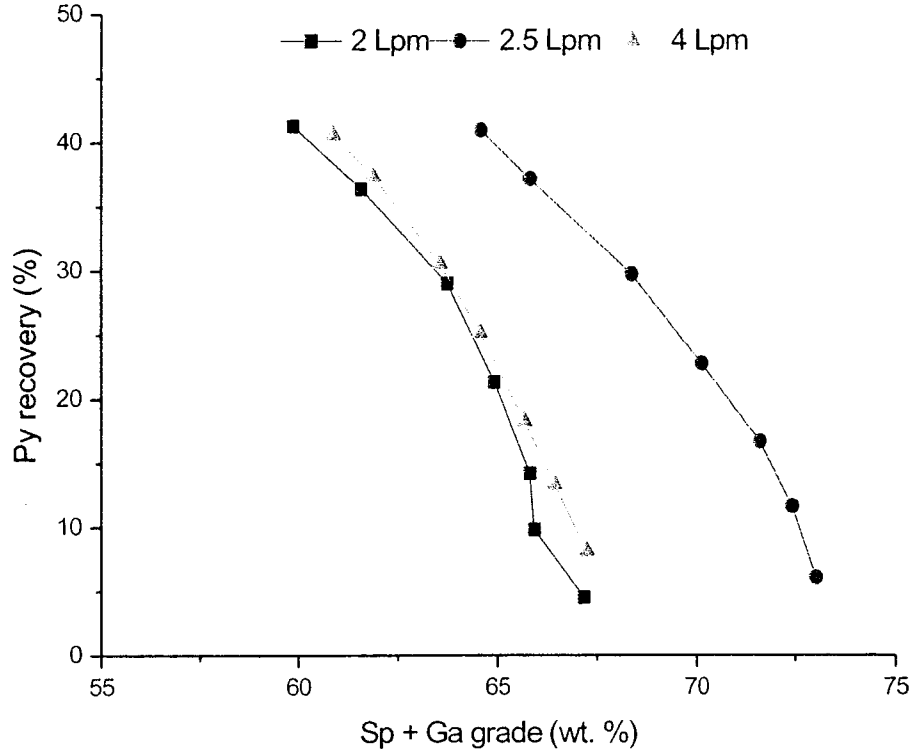




**Figure 24: Reagent scheme B time cumulative recovery plot**

#### ***4.4.3 Effect of Air Flow Rate on Flotation Kinetics***

The introduction of air is a crucial component of flotation, as it plays an effective part in determining the bubble characteristics (size, quantity, etc.) and affect kinetics of flotation. Hence, it was decided to test three air flow rates, 2.0, 2.5 and 4.0 Lpm, using the correct reagent addition scheme B, as discussed above. This would give a clear picture as to whether a lower or higher air flow rate is desirable. Figure 25 shows a plot of the results with the flotation kinetics parameters given in Table 12.



**Figure 25: Pyrite recovery as a function of concentrate grade for different air flow rates**

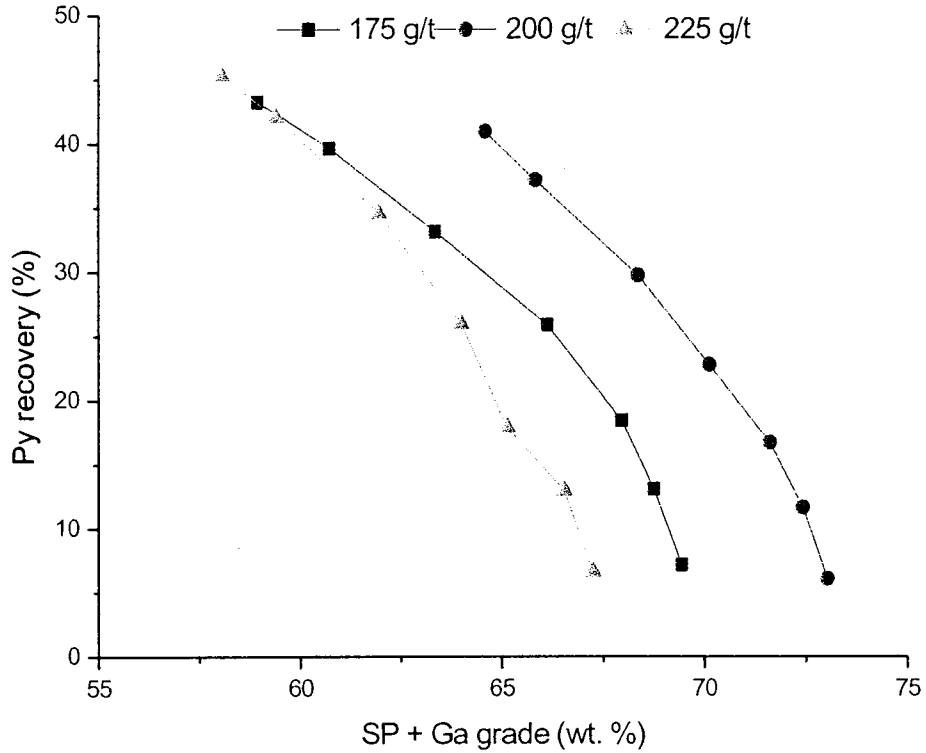
Based on both the plot and the parameters (such as  $SI$ ), an air flow rate of 2.5 Lpm was found to be ideal for this system. Increasing the air flow rate beyond this value caused selectivity to be lost and an increase in the rate constants for the gangue minerals was also observed.

**Table 12: Flotation kinetics parameters for varying air flow rates**

Test	Mineral	k	$R_{\infty}$	Km	SI wrt NSG	SI wrt Py
2.0 Lpm	Sp	0.185	77.2	14.28	6.40	2.43
	NSG	0.120	18.6	2.23		
	Py	0.136	43.3	5.89		
2.5 Lpm	Sp	0.229	75.2	17.22	8.01	2.58
	NSG	0.125	17.2	2.15		
	Py	0.159	42.0	6.68		
4.0 Lpm	Sp	0.225	67.1	15.10	5.55	2.00
	NSG	0.147	18.5	2.72		
	Py	0.184	41.0	7.54		

#### **4.4.4 Effect of Collector Concentration**

Finally, collector dosage was tested using three different concentrations: 175, 200 and 225 g/t. Figure 26 shows the plot of pyrite recovery versus concentrate grade. 200 g/t HCNPHA gave the highest concentrate grade per amount of pyrite recovered, whereas 175 and 225 g/t gave similar results. However, looking at Table 13, 175 g/t achieved the highest selectivity with respect to non-sulphidic gangue.



**Figure 26: Pyrite recovery as a function of concentrate grade for various collector dosages**

It was determined that using 175 or 200 g/t would be ideal for this system as both gave similar results for their respective selective indices. However, increasing the collector dosage should have increased the reaction rate of sphalerite ( $k$ ), yet the reaction rate of sphalerite was observed to decrease with increasing dosages. Therefore it was determined that the optimum dosage for HCNPHA was 200 g/t based on the plot and the  $SI$  values. Concentration of CMC and sodium silicate used for the three different dosages of HCNPHA remained constant. Hence, the frothing characteristics increased with an increase in concentration of HCNPHA. The

increased frothing property of the slurry formed a thick layer of froth on the surface and slowed down the flotation kinetics.

**Table 13: Flotation kinetics parameters for various collector dosages**

Test	Mineral	$k$ ( $\text{min}^{-1}$ )	$R_{\infty}$	$K_m$	SI wrt NSG	SI wrt Py
175 g/t	Sp	0.258	75.6	19.50	8.19	2.50
	NSG	0.124	19.2	2.38		
	Py	0.178	43.8	7.80		
200 g/t	Sp	0.229	75.2	17.22	8.01	2.58
	NSG	0.125	17.2	2.15		
	Py	0.159	42.0	6.68		
225 g/t	Sp	0.204	75.9	15.48	5.09	2.01
	NSG	0.118	25.8	3.04		
	Py	0.165	46.8	7.72		

#### ***Section 4.5 Economic Analysis***

According to Natural Resources Canada, in 2007 594,113 tonnes of zinc were produced in Canada. Assuming an average grade of 10% zinc, that would mean that approximately 6 million tonnes of ore were processed in 2007. Using this assumption, an average copper sulphate dosage of 1 kg/t was used for all calculations. Table 14 shows the economic analysis comparing the alternative reagent scheme to the current xanthate reagent scheme. The costs for the collectors (HCNPHA and xanthate) are not included as the costs for production and use are assumed to be comparable. Frother is also not included as the same frother (MIBC) and dosage (10 g/t) were used in both reagent schemes.

**Table 14: Economic analysis comparing alternative reagent scheme to current xanthate scheme**

Auxiliary Chemical	Cost (\$/kg)	Alternative Reagents		Xanthate Scheme	
		Dosage (g/t)	(\$/yr)*10 <sup>6</sup>	Dosage (g/t)	(\$/yr)*10 <sup>6</sup>
Copper Sulphate	3.5	200	4.2	1000	21
CMC	0.5	250	0.75	-	-
Sodium Silicate	0.5	250	0.75	-	-
Total			5.7		21
<b>Savings</b>			<b>15.3</b>		

From this brief analysis, it can be seen that by adding two new auxiliary chemicals, CMC and sodium silicate, and drastically reducing the amount of copper sulphate, a possible savings of \$15.3 million/year can be achieved in the Canadian mining industry. Also, no changes to the flow sheet given by *Teck Cominco* were made, meaning no additional equipment costs would be incurred by incorporating the new reagent scheme.

## Section 5.0 Conclusions

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## **5.0 Conclusions**

HCNPHA was able to float sphalerite by using 200 g/t of copper sulphate, which was used to reactivate the sphalerite surface. Thus, 80% reduction of the copper sulphate was achieved for processing a similar ore using the xanthate reagent scheme. The addition of copper sulphate increased the sphalerite recovery but also the kinetics of flotation of sphalerite. Copper ions adsorbed preferentially on zinc sites and facilitated surface chelation with HCNPHA. However, the mechanism of collection when using HCNPHA differs from xanthates because complexes formed with copper and zinc are less soluble in water, therefore, activation by copper sulphate is not necessary to form a less soluble complex on the sphalerite surface. However, sphalerite is suppressed during the lead rougher stage, therefore, reactivation of the sphalerite suppressed in the lead rougher stage by copper sulphate was found to be essential. The optimum collector (HCNPHA) concentration and air flow rate for flotation were determined to be 200 g/t and 2.5 Lpm, respectively. The foaming characteristics of HCNPHA at pH 9 with the ore used in the study, and the consequent entrainment of gangue minerals, were alleviated to a considerable extent using a carboxymethylcellulose (CMC). The grade of sphalerite was further improved with the addition of 250 g/t sodium silicate which aided in reducing the flotation of non-sulphidic gangue. The order in which reagents were added had a significant impact on the grade and recovery of sphalerite, and the preferred order of addition of various reagents was found to be: copper sulphate, suppressants (CMC and sodium silicate), collector and finally frother. The auxiliary chemicals added to the flotation process are environmentally friendly and do not require any new equipment or changes to the existing industrial flow sheet. Implementing an 80% reduction in copper sulphate (to the current xanthate reagent scheme) and the addition of two new auxiliary chemicals, the alternative reagent scheme proposed here has the possibility of



generating a savings of \$15.3 million/year for the Canadian industry. This value was reached based on the assumption that the price of the new collector (HCNPHA) and potassium amyl xanthate (PAX) are comparable.

### **5.1 Recommendations**

Testing with pure minerals and HCNPHA is needed to determine if HCNPHA is selective or if minerals floated due to inadvertent activation by free metal ion species in the pulp. While grinding time was adjusted for this ore to achieve the required 80% passing through 63  $\mu\text{m}$  as recommended by *Teck Cominco*, tests should be conducted to determine whether a finer grind or coarser grind would be beneficial for the new alternative reagent scheme. HCNPHA has shown promise in the flotation of sphalerite; however, to improve its commercial application to the industry, tests should be done to see how well it works as a collector for other base metal sulphide ores such as chalcopyrite, galena, etc. Structure of HCNPHA can be easily modified structure and a derivative of it may improve selectivity in the flotation of sphalerite or may reduce the severity of its foaming characteristics. In order to truly determine if the alternative reagent scheme proposed is a viable replacement for the current xanthate reagent scheme, pilot plant testing needs to be done to see how well this combination of reagents works in a continuous process. Also, testing the auxiliary chemicals in conjunction with the xanthate reagent scheme is desirable to determine if it is possible to reduce the amount of copper sulphate consumed during the current flotation process without affecting sphalerite grade or recovery.

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## Section 7.0 Appendices

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## Appendix 1.0 Flotation Data

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**Table 15: Effectiveness of Polysaccharides, CMCs and DETA as Pyrite Suppressants**

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Wt. %
P-1 Baseline	Pb-R	30.2	39.1	26.4	10.5	24.0	58.2	7.1	8.3	8.9	8.8
	Zn-R	166.9	5.1	46.3	16.4	32.2	28.8	47.7	49.9	49.1	48.9
	Tail	144.5	2.6	50.7	15.9	30.8	12.9	45.2	41.8	42.0	42.3
	Feed	<b>341.6</b>	<b>7.0</b>	<b>46.4</b>	<b>15.7</b>	<b>30.9</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
P-2 PS-1 50 g/t	Pb-R	59.3	11.3	30.5	11.5	46.7	45.4	10.6	12.7	12.5	16.9
	Zn-R	170	2.2	37.1	12.1	48.5	35.0	50.4	52.0	51.6	48.5
	Tail	121.3	1.7	40.2	11.5	46.6	19.6	39.0	35.3	35.9	34.6
	Feed	<b>350.6</b>	<b>3.6</b>	<b>37.1</b>	<b>11.8</b>	<b>47.5</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
P-3 PS-1 100 g/t	Pb-R	57.1	34.0	27.7	11.1	27.2	62.9	9.5	11.8	9.0	16.6
	Zn-R	168.5	3.6	37.2	12.2	47.0	25.6	49.4	50.4	55.2	49.1
	Tail	117.9	2.3	44.2	13.1	40.4	11.5	41.1	37.7	35.7	34.3
	Feed	<b>343.5</b>	<b>8.2</b>	<b>38.0</b>	<b>12.3</b>	<b>41.4</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
P-4 PS-2 50 g/t	Pb-R	45.1	35.5	33.7	15.0	15.8	54.7	9.4	12.2	8.6	13.5
	Zn-R	192.9	4.9	46.2	14.9	34.0	33.6	57.1	53.5	64.3	57.9
	Tail	95.4	3.4	54.6	19.3	22.7	11.6	33.5	34.3	27.2	28.6
	Feed	<b>333.4</b>	<b>8.6</b>	<b>46.9</b>	<b>16.2</b>	<b>28.3</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
P-5 PS-2 100 g/t	Pb-R	34.2	48.1	25.3	11.9	14.7	53.5	6.7	9.3	6.7	9.6
	Zn-R	186.5	6.7	42.1	12.7	38.5	31.9	47.6	42.7	59.8	52.5
	Tail	134.7	4.2	55.9	19.8	20.1	14.6	45.7	48.0	33.5	37.9
	Feed	<b>355.4</b>	<b>9.7</b>	<b>45.7</b>	<b>15.3</b>	<b>29.3</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
P-6 PS-3 50 g/t	Pb-R	29.7	47.9	25.2	9.9	17.0	55.5	6.9	7.8	7.0	8.6
	Zn-R	185.5	6.7	43.9	15.9	33.6	32.9	51.1	53.1	55.7	53.6
	Tail	130.8	3.3	51.2	16.6	28.9	11.6	42.0	39.1	37.2	37.8
	Feed	<b>346</b>	<b>8.9</b>	<b>45.0</b>	<b>15.7</b>	<b>30.4</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
P-7 PS-3 100 g/t	Pb-R	28.7	48.5	26.3	10.9	14.3	53.8	7.0	8.2	6.6	8.3
	Zn-R	178.9	8.1	46.7	18.0	27.2	36.9	51.2	55.6	49.7	51.4
	Tail	140.2	2.6	48.6	14.9	33.8	9.3	41.8	36.1	43.7	40.3
	Feed	<b>347.8</b>	<b>9.2</b>	<b>45.8</b>	<b>16.2</b>	<b>28.8</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Wt. %
<b>P-8</b>	Zn-R	184.4	6.2	44.9	14.1	34.8	28.0	51.9	48.2	58.7	52.7
CMC	Tail	126.9	4.7	50.8	18.3	26.2	14.7	40.4	43.0	36.2	36.3
<b>50 g/t</b>	<b>Feed</b>	<b>349.9</b>	<b>10.9</b>	<b>45.2</b>	<b>15.3</b>	<b>28.7</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
	Pb-R	31.0	47.7	26.4	9.4	16.5	48.6	6.9	7.2	7.3	8.9
<b>P-9</b>	Zn-R	176.6	7.6	46.3	14.7	31.4	31.4	49.0	45.8	53.8	50.5
CMC	Tail	142.0	6.0	51.9	18.8	23.2	20.0	44.2	47.0	38.8	40.6
<b>100 g/t</b>	<b>Feed</b>	<b>349.6</b>	<b>10.5</b>	<b>46.8</b>	<b>15.9</b>	<b>26.7</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

Table 16: Air Flow Rate Tests

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Wt. %
AF-1 0.5 Lpm	Pb-R	23.8	31.7	28.4	9.9	30.0	25.5	4.1	4.3	7.2	6.8
	Zn-R1	50.5	9.5	44.6	15.8	30.0	16.3	13.6	14.6	15.3	14.4
	Zn-R2	91.8	9.0	45.4	15.0	30.6	27.8	25.1	25.2	28.4	26.3
	Tail	183.5	4.9	51.9	16.7	26.5	30.4	57.3	55.9	49.1	52.5
	Feed	<b>349.6</b>	<b>8.5</b>	<b>47.5</b>	<b>15.7</b>	<b>28.4</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
AF-2 1 Lpm	Pb-R	48.3	26.7	34.1	12.7	26.5	50.7	10.3	11.5	11.7	13.9
	Zn-R	178	5.0	45.9	16.1	33.0	34.9	51.1	53.7	54.0	51.2
	Tail	121.2	3.0	50.8	15.3	30.8	14.4	38.6	34.8	34.3	34.9
	Feed	<b>347.5</b>	<b>4.3</b>	<b>39.1</b>	<b>13.8</b>	<b>42.8</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
AF-3 1.5 Lpm	Pb-R	45.8	30.7	34.3	12.3	22.7	50.1	9.4	10.4	10.6	13.2
	Zn-R1	112.5	6.4	46.5	17.4	29.7	25.5	31.4	36.1	33.9	32.4
	Zn-R2	94.9	4.8	46.5	15.2	33.5	16.2	26.5	26.5	32.3	27.3
	Tail	94.4	2.5	57.9	15.5	24.2	8.3	32.8	26.9	23.2	27.2
	Feed	<b>347.6</b>	<b>8.1</b>	<b>48.0</b>	<b>15.6</b>	<b>28.3</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
AF-4 2 Lpm	Pb-R	44.3	28.4	29.9	11.8	30.0	40.1	7.0	8.3	11.0	10.9
	Zn-R	246.5	6.0	46.0	15.5	32.5	47.2	59.6	61.0	66.3	60.9
	Tail	114.1	3.5	55.6	16.9	24.0	12.6	33.4	30.7	22.7	28.2
	Feed	<b>404.9</b>	<b>3.7</b>	<b>40.0</b>	<b>14.6</b>	<b>41.7</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
AF-5 2.5 Lpm	Pb-R	47.1	28.2	33.4	12.0	26.4	51.0	9.7	10.6	12.4	13.8
	Zn-R1	145.4	5.3	46.8	16.7	31.2	29.4	41.9	45.6	45.2	42.5
	Zn-R2	70.7	4.2	48.3	14.9	32.7	11.4	21.0	19.8	23.0	20.6
	Tail	79.2	2.7	56.4	16.2	24.7	8.3	27.5	24.0	19.5	23.1
	Feed	<b>342.4</b>	<b>7.6</b>	<b>47.5</b>	<b>15.6</b>	<b>29.3</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
AF-6 3 Lpm	Pb-R	64.6	23.3	36.9	14.6	25.3	57.8	14.2	17.1	15.6	18.2
	Zn-R1	151.6	5.1	47.7	17.8	29.4	30.0	42.9	49.1	42.5	42.8
	Zn-R2	70.9	3.0	49.5	13.8	33.6	8.2	20.9	17.7	22.8	20.0
	Tail	66.9	1.6	55.4	13.3	29.8	4.0	22.0	16.1	19.1	18.9
	Feed	<b>354</b>	<b>6.1</b>	<b>38.7</b>	<b>14.1</b>	<b>41.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

**Table 17: Effect of Collector Dosage**

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Wt. %
CC-1 50 g/t	Pb-R	12.7	29.1	42.5	16.1	12.3	26.2	3.1	3.9	2.2	3.7
	Zn-R	85.7	3.5	47.3	15.2	34.0	21.1	23.6	24.9	26.1	24.8
	Tail	246.6	3.0	51.1	15.1	30.8	52.7	73.3	71.2	71.7	71.5
	<b>Feed</b>	<b>345</b>	<b>4.3</b>	<b>39.1</b>	<b>13.8</b>	<b>42.8</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CC-2 100 g/t	Pb-R	21.4	36.0	34.5	12.9	16.6	27.0	2.7	3.2	2.4	6.3
	Zn-R	104.4	5.0	47.0	16.2	31.8	31.1	30.0	33.2	32.5	30.7
	Tail	213.9	3.3	51.3	15.1	30.2	41.9	67.3	63.6	65.1	63.0
	<b>Feed</b>	<b>339.7</b>	<b>3.7</b>	<b>40.0</b>	<b>14.6</b>	<b>41.7</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CC-3 150 g/t	Pb-R	28.7	42.1	31.4	11.7	14.9	25.7	2.5	2.9	2.2	8.4
	Zn-R	167.2	5.9	48.0	16.4	29.8	47.2	50.7	54.5	52.1	49.0
	Tail	145.1	3.9	51.1	14.8	30.3	27.1	46.8	42.6	45.7	42.6
	<b>Feed</b>	<b>341</b>	<b>6.1</b>	<b>38.7</b>	<b>14.1</b>	<b>41.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CC-4 100 g/t	Pb-R	35.3	37.0	32.1	11.8	19.1	25.4	2.6	3.1	2.6	10.2
	Zn-R	131.6	6.0	47.1	15.5	31.5	42.3	40.0	42.3	41.2	38.2
	Tail	177.5	3.4	50.0	14.8	31.8	32.3	57.3	54.6	56.3	51.5
	<b>Feed</b>	<b>344.4</b>	<b>7.8</b>	<b>47.1</b>	<b>14.8</b>	<b>30.4</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CC-5 200 g/t	Pb-R	36.2	30.1	42.3	16.3	11.3	27.3	3.6	4.3	2.1	10.4
	Zn-R	209.7	3.8	45.5	15.8	34.9	57.3	63.7	69.0	65.4	60.4
	Tail	101.5	2.1	48.3	12.6	37.0	15.4	32.7	26.7	32.4	29.2
	<b>Feed</b>	<b>347.4</b>	<b>6.1</b>	<b>46.0</b>	<b>14.9</b>	<b>33.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CC-6 250 g/t	Pb-R	42.6	38.8	35.6	13.2	12.4	26.9	3.2	3.6	2.2	12.6
	Zn-R	227.3	5.1	46.9	15.2	32.8	62.8	75.1	74.4	74.2	67.3
	Tail	67.8	2.8	45.6	15.1	36.6	10.3	21.7	22.0	23.6	20.1
	<b>Feed</b>	<b>337.7</b>	<b>8.9</b>	<b>45.2</b>	<b>14.9</b>	<b>31.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

**Table 18: Effect of Copper Sulphate Dosage for 350 g Samples**

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				Wt. %
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	
1 CS-1 25 g/t	Pb-R	79.5	29.1	42.5	16.1	12.3	72.2	20.7	24.0	14.7	23.0
	Zn-R	184	3.5	47.3	15.2	34.0	20.0	53.4	52.6	59.8	53.2
	Tail	82.4	3.0	51.1	15.1	30.8	7.8	25.8	23.4	25.5	23.8
	<b>Feed</b>	<b>345.9</b>	<b>4.3</b>	<b>39.1</b>	<b>13.8</b>	<b>42.8</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
2 CS-2 50 g/t	Pb-R	52.2	36.0	34.5	12.9	16.6	58.5	11.2	12.6	9.8	15.0
	Zn-R	204.7	5.0	47.0	16.2	31.8	32.2	59.9	61.9	63.1	59.0
	Tail	90.1	3.3	51.3	15.1	30.2	9.3	28.8	25.5	27.1	26.0
	<b>Feed</b>	<b>347</b>	<b>3.7</b>	<b>40.0</b>	<b>14.6</b>	<b>41.7</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
3 CS-3 100 g/t	Pb-R	41.5	42.1	31.4	11.7	14.9	51.8	7.9	9.0	7.1	11.9
	Zn-R	213.6	5.9	48.0	16.4	29.8	37.2	62.5	65.1	64.2	61.1
	Tail	94.7	3.9	51.1	14.8	30.3	10.9	29.5	25.9	28.7	27.1
	<b>Feed</b>	<b>349.8</b>	<b>6.1</b>	<b>38.7</b>	<b>14.1</b>	<b>41.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
4 CS-4 200 g/t	Pb-R	45.0	37.0	32.1	11.8	19.1	51.2	9.0	10.2	8.6	12.8
	Zn-R	215.9	6.0	47.1	15.5	31.5	39.5	63.1	64.2	64.2	61.5
	Tail	89.9	3.4	50.0	14.8	31.8	9.3	27.9	25.6	27.1	25.6
	<b>Feed</b>	<b>350.8</b>	<b>9.3</b>	<b>45.9</b>	<b>14.8</b>	<b>30.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
5 CS-5 25 g/t	Pb-R	84.8	30.1	42.3	16.3	11.3	75.4	22.7	26.6	14.8	24.4
	Zn-R	161.9	3.8	45.5	15.8	34.9	18.3	46.6	49.1	52.2	46.7
	Tail	100.2	2.1	48.3	12.6	37.0	6.3	30.6	24.3	33.0	28.9
	<b>Feed</b>	<b>346.9</b>	<b>9.8</b>	<b>45.5</b>	<b>15.0</b>	<b>29.8</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
6 CS-6 100 g/t	Pb-R	55.9	38.8	35.6	13.2	12.4	62.1	12.7	14.3	9.0	16.1
	Zn-R	226.8	5.1	46.9	15.2	32.8	32.8	68.3	66.8	69.6	65.2
	Tail	65	2.8	45.6	15.1	36.6	5.2	19.0	19.0	21.3	18.7
	<b>Feed</b>	<b>347.7</b>	<b>10.1</b>	<b>44.9</b>	<b>14.9</b>	<b>30.2</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
7 CS-7 200 g/t	Pb-R	54.9	40.5	36.4	13.4	9.8	63.6	12.7	14.1	8.2	15.8
	Zn-R	195.9	5.3	50.8	15.6	28.3	29.8	63.1	58.8	56.9	56.5
	Tail	95.8	2.4	39.9	14.7	43.0	6.6	24.2	27.1	34.9	27.6
	<b>Feed</b>	<b>346.6</b>	<b>10.1</b>	<b>45.5</b>	<b>15.0</b>	<b>29.4</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				Wt. %
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	
CS-8 200 g/t	Pb-R	22.6	20.6	26.1	9.8	43.5	30.9	6.1	5.0	5.3	6.6
	Zn-R	185.4	4.8	46.2	14.4	34.6	58.6	88.9	61.0	41.8	54.1
	Tail	134.6	1.2	3.6	11.1	84.2	10.5	5.0	34.0	52.9	39.3
	<b>Feed</b>	<b>342.6</b>	<b>4.4</b>	<b>28.1</b>	<b>12.8</b>	<b>54.7</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CS-9 500 g/t	Pb-R	20.4	26.0	24.5	10.3	39.3	33.2	4.8	4.5	4.6	6.0
	Zn-R	209	4.4	45.6	14.9	35.1	57.6	90.8	66.8	49.6	61.0
	Tail	113.1	1.3	4.1	11.8	82.8	9.2	4.4	28.7	45.8	33.0
	<b>Feed</b>	<b>342.5</b>	<b>4.7</b>	<b>30.6</b>	<b>13.6</b>	<b>51.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CS-10 0 g/t	Pb-R	14	23.6	23.1	8.7	44.6	21.1	2.9	2.7	3.5	4.1
	Zn-R	170.2	5.8	43.8	16.6	33.8	63.6	66.8	61.6	40.9	50.0
	Tail	156.1	1.5	21.7	10.5	66.3	15.3	30.3	35.7	55.6	45.9
	<b>Feed</b>	<b>340.3</b>	<b>4.6</b>	<b>32.8</b>	<b>13.5</b>	<b>49.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CS-11 50 g/t	Pb-R	13.3	33.1	21.3	8.5	37.0	19.2	2.4	2.2	3.0	3.9
	Zn-R	201	8.2	47.1	17.6	27.0	72.1	81.0	68.1	46.0	58.9
	Tail	127.1	1.6	15.2	12.2	71.0	8.7	16.5	29.7	50.9	37.2
	<b>Feed</b>	<b>341.4</b>	<b>6.7</b>	<b>34.3</b>	<b>15.2</b>	<b>43.8</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CS-12 100 g/t	Pb-R	12.5	36.6	22.4	8.4	32.5	19.5	2.4	2.0	2.6	3.6
	Zn-R	202.9	8.2	48.9	16.7	26.2	70.8	85.0	64.5	45.0	58.9
	Tail	129.2	1.8	11.4	13.7	73.2	9.7	12.6	33.5	52.4	37.5
	<b>Feed</b>	<b>344.6</b>	<b>6.8</b>	<b>33.9</b>	<b>15.3</b>	<b>44.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
CS-13 500 g/t	Pb-R	10.5	31.1	23.5	9.9	35.5	13.2	2.0	1.9	2.4	3.1
	Zn-R	220	8.8	51.5	15.7	24.0	78.4	93.1	64.4	48.1	64.0
	Tail	113.2	1.8	5.3	15.9	77.0	8.4	4.9	33.7	49.4	32.9
	<b>Feed</b>	<b>343.7</b>	<b>7.2</b>	<b>35.4</b>	<b>15.6</b>	<b>41.8</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

Table 19: Effect of copper sulphate dosage tests for 1 kg samples

Sample	Stage	Mass (g)	Time (min)	Grade (wt %)			Cumulative Grade (wt %)			Cumulative Recovery (%)									
				Ga	Sp	Py	Ga	Sp	Py	Ga	Sp	Py	NSG						
CS-14 0 g/t	PF	11.3		12.5	14.7	17.9	54.9												
	Pb-R	76.5	0	36.1	27.2	12.7	24.0												
	Zn-R1	71.7	1	2.3	43.1	17.9	36.7	2.3	43.1	17.9	36.7	0	0	0	0	0	0	0	0
	Zn-R2	59.3	2	2.4	40.7	17.9	39.1	2.3	42.0	17.9	37.8	3.4	10.8	9.0	9.0	5.1	5.1	5.1	5.1
	Zn-R3	88.6	4	2.7	40.2	19.8	37.2	2.5	41.3	18.6	37.6	6.2	19.2	16.5	16.5	9.6	9.6	9.6	9.6
	Zn-R4	129.6	8	3.3	38.0	18.5	40.2	2.8	40.1	18.6	38.6	11.0	31.6	28.8	28.8	16.1	16.1	16.1	16.1
	Feed		981.3		5.0	28.9	14.3	51.8											
CS-15 50 g/t	PF	12		11.9	15.3	18.5	54.3												
	Pb-R	51.9	0	53.1	18.8	9.0	19.2												
	Zn-R1	57.5	1	5.8	48.4	18.0	27.9	5.8	48.4	18.0	27.9	5.5	9.5	7.0	7.0	3.6	3.6	3.6	3.6
	Zn-R2	61.7	2	6.1	46.1	17.6	30.2	6.0	47.2	17.8	29.1	11.7	19.3	14.4	14.4	7.8	7.8	7.8	7.8
	Zn-R3	88.9	4	6.3	42.6	17.1	34.1	6.1	45.2	17.5	31.2	20.8	32.3	24.7	24.7	14.6	14.6	14.6	14.6
	Zn-R4	118.5	8	5.8	42.1	17.3	34.7	6.0	44.1	17.4	32.5	32.1	49.4	38.6	38.6	23.9	23.9	23.9	23.9
	Feed		944.1		6.4	30.9	15.6	47.1											
CS-16 100 g/t	PF	12.9		12.4	15.1	17.9	54.6												
	Pb-R	50.7	0	53.7	19.0	9.4	18.0												
	Zn-R1	75.7	1	5.3	52.3	14.7	27.7	5.3	52.3	14.7	27.7	6.7	13.3	7.6	7.6	4.3	4.3	4.3	4.3
	Zn-R2	64.5	2	5.6	50.2	14.8	29.4	5.4	51.4	14.7	28.5	12.8	24.2	14.1	14.1	8.2	8.2	8.2	8.2
	Zn-R3	97.4	4	5.8	48.7	15.4	30.1	5.6	50.3	15.0	29.2	22.2	40.1	24.3	24.3	14.3	14.3	14.3	14.3
	Zn-R4	99.9	8	4.9	44.3	15.7	35.1	5.4	48.5	15.2	30.9	30.5	54.9	35.0	35.0	21.6	21.6	21.6	21.6
	Feed		988		6.0	30.2	14.8	49.0											



Sample	Stage	Mass (g)	Time (min)	Grade (wt %)				Cumulative Grade (wt %)				Cumulative Recovery (%)								
				Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Ga	Sp	Py	NSG					
CS-17 200 g/t	PF	14.2		12.2	17.5	18.3	52.0													
	Pb-R	81	0	37.4	23.8	13.0	25.7													
	Zn-R1	99.6	1	2.5	53.1	16.0	28.4	2.5	53.1	16.0	28.4	0	0	0	0	0	0	0	0	
	Zn-R2	100.6	2	2.5	53.5	16.3	27.7	2.5	53.3	16.1	28.1	4.4	17.1	10.4	5.9	17.1	10.4	10.4	5.9	
	Zn-R3	140.8	4	2.9	51.2	16.5	29.5	2.6	52.4	16.3	28.6	9.0	34.5	21.1	11.7	34.5	21.1	21.1	11.7	
	Zn-R4	173	8	3.8	41.4	17.3	37.5	3.0	48.7	16.6	31.6	16.3	57.8	36.2	20.3	57.8	36.2	36.2	20.3	
	Tail	390.2		1.9	9.5	14.0	74.6													
	<b>Feed</b>	<b>999.4</b>		<b>5.5</b>	<b>30.9</b>	<b>15.3</b>	<b>48.2</b>													
CS-18 300 g/t	PF	16.5		13.3	17.4	21.2	48.1													
	Pb-R	93.4	0	40.1	27.7	13.1	19.2													
	Zn-R1	124.2	1	3.4	54.8	17.4	24.4	3.4	54.8	17.4	24.4	6.5	21.0	13.2	6.9	21.0	13.2	13.2	6.9	
	Zn-R2	106.4	2	2.8	54.8	17.5	24.9	3.1	54.8	17.4	24.6	11.2	39.0	24.6	12.9	39.0	24.6	24.6	12.9	
	Zn-R3	158.5	4	3.2	51.0	17.4	28.4	3.2	53.3	17.4	26.2	19.0	63.9	41.4	23.1	63.9	41.4	41.4	23.1	
	Zn-R4	157.9	8	3.9	39.0	18.5	38.6	3.4	49.2	17.7	29.8	28.6	82.9	59.3	36.9	82.9	59.3	59.3	36.9	
	Tail	336.8		1.9	7.9	15.1	75.1													
	<b>Feed</b>	<b>993.7</b>		<b>6.5</b>	<b>32.6</b>	<b>16.4</b>	<b>44.4</b>													
CS-19 400 g/t	PF	13.3		14.3	17.8	20.6	47.3													
	Pb-R	77.4	0	47.0	21.3	12.1	19.7													
	Zn-R1	129.1	1	3.2	57.7	16.7	22.4	3.2	57.7	16.7	22.4	6.1	22.4	12.9	6.6	22.4	12.9	12.9	6.6	
	Zn-R2	126.6	2	3.5	56.4	17.0	23.0	3.3	57.1	16.9	22.7	12.8	43.8	25.7	13.2	43.8	25.7	25.7	13.2	
	Zn-R3	151.3	4	3.6	51.3	17.6	27.4	3.4	54.9	17.2	24.5	21.1	67.1	41.6	22.6	67.1	41.6	41.6	22.6	
	Zn-R4	180.7	8	4.6	36.8	19.0	39.6	3.8	49.4	17.7	29.1	33.5	87.1	62.1	38.8	87.1	62.1	62.1	38.8	
	Tail	329.7		1.8	7.3	15.7	75.3													
	<b>Feed</b>	<b>1008.1</b>		<b>6.6</b>	<b>33.0</b>	<b>16.7</b>	<b>43.7</b>													
CS-20 500 g/t	PF	10.8		10.3	19.4	16.1	54.3													
	Pb-R	56.2	0	40.8	20.2	11.4	27.6													
	Zn-R1	94.7	1	3.1	56.0	15.7	25.2	3.1	56.0	15.7	25.2	6.0	19.4	10.8	4.4	19.4	10.8	10.8	4.4	
	Zn-R2	81.7	2	3.2	55.1	15.8	25.9	3.1	55.6	15.7	25.5	11.3	35.9	20.2	8.3	35.9	20.2	20.2	8.3	
	Zn-R3	115.1	4	3.8	51.6	16.0	28.6	3.4	54.0	15.8	26.8	20.1	57.7	33.6	14.4	57.7	33.6	33.6	14.4	
	Zn-R4	147.9	8	4.3	40.7	16.5	38.5	3.7	49.5	16.1	30.7	33.0	79.8	51.3	24.9	79.8	51.3	51.3	24.9	
	Tail	495.7		1.8	8.4	11.9	77.9													
	<b>Feed</b>	<b>1002.1</b>		<b>4.9</b>	<b>27.2</b>	<b>13.7</b>	<b>54.1</b>													

Table 20: Short listing suppressant tests

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				Wt. %
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	
SP-1 Baseline	Pb-R	32.9	23.9	28.6	13.1	34.3	54.4	9.6	8.9	6.7	9.5
	Zn-R	182.2	2.9	46.6	16.8	33.6	37.1	86.7	63.0	41.2	52.6
	Tail	131.1	0.9	2.8	10.4	85.9	8.5	3.7	28.1	52.1	37.9
	Feed	346.2	4.2	28.3	14.1	53.5	100	100	100	100	100
SP-2 Baseline	Pb-R	33.6	24.4	28.2	13.9	33.6	46.3	8.7	8.5	6.9	9.6
	Zn-R	171.6	4.0	51.4	17.5	27.1	39.4	82.7	56.3	36.5	49.2
	Tail	143.9	1.7	6.4	13.0	78.9	14.3	8.6	35.2	56.5	41.2
	Feed	349.1	5.0	30.6	15.3	49.1	100	100	100	100	100
SP-3 CMC-1	Pb-R	31.5	28.7	26.6	13.2	31.5	58.5	8.8	8.8	6.5	9.3
	Zn-R	177.7	3.0	47.4	16.7	32.9	32.6	84.4	59.9	41.3	52.4
	Tail	130.2	1.1	5.2	11.9	81.7	9.0	6.8	31.4	52.2	38.4
	Feed	339.4	4.6	29.3	14.5	51.5	100	100	100	100	100
SP-4 CMC-2	Pb-R	37.4	25.6	27.5	13.7	33.3	52.7	8.4	8.5	6.8	10.5
	Zn-R	202.2	3.2	45.9	17.7	33.3	40.0	86.7	67.5	47.0	57.0
	Tail	115	1.0	4.5	11.1	83.4	7.3	4.8	24.0	46.2	32.4
	Feed	354.6	4.8	30.5	15.1	49.5	100	100	100	100	100
SP-5 CMC-3	Pb-R	39.5	25.5	27.6	14.3	32.6	51.8	8.4	8.7	6.9	11.2
	Zn-R	200.5	3.2	46.8	17.4	32.6	39.8	86.9	64.3	47.4	57.0
	Tail	111.9	1.2	4.6	13.1	81.1	8.4	4.7	27.0	45.6	31.8
	Feed	351.9	5.1	31.2	15.7	48.0	100	100	100	100	100
SP-6 CMC-4	Pb-R	44.2	29.8	26.0	14.6	29.5	52.1	7.4	8.2	7.1	12.7
	Zn-R	212.4	3.7	47.8	18.7	29.9	41.4	87.5	67.7	53.3	61.1
	Tail	91.1	1.3	6.5	15.5	76.7	6.5	5.1	24.1	39.6	26.2
	Feed	347.7	6.4	34.2	17.3	42.1	100	100	100	100	100
SP-7 CMC-5	Pb-R	38.6	26.7	26.5	14.1	32.7	53.2	8.3	8.8	6.9	11.1
	Zn-R	194.9	3.3	46.3	17.7	32.7	39.5	86.1	65.2	46.6	56.3
	Tail	112.9	1.1	5.2	12.1	81.6	7.3	5.6	26.0	46.5	32.6
	Feed	346.4	5.2	30.7	15.5	48.6	100	100	100	100	100
SP-8 PS-1	Pb-R	25.7	24.5	27.2	12.4	35.9	47.0	9.4	7.9	7.0	7.7
	Zn-R	171.8	4.0	46.3	16.9	32.8	39.7	83.4	56.5	39.1	51.2
	Tail	138.3	1.6	5.0	13.2	80.2	13.3	7.2	35.5	53.9	41.2
	Feed	335.8	4.6	27.8	15.0	52.6	100	100	100	100	100

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Wt. %
SP-9 PS-2	Pb-R	36.6	30.4	25.7	13.7	30.3	48.4	7.0	7.5	7.3	10.7
	Zn-R	228.2	4.2	47.2	19.3	29.2	46.3	89.0	73.3	58.6	67.0
	Tail	75.8	1.4	6.4	15.3	76.9	5.3	4.0	19.3	34.1	22.3
	Feed	340.6	6.4	35.8	17.8	40.0	100	100	100	100	100
SP-10 CMC-6	Pb-R	36.7	26.9	27.7	13.6	31.7	46.9	8.3	7.9	7.1	10.7
	Zn-R	179.1	4.5	49.8	20.0	25.6	42.9	81.4	63.0	40.2	52.1
	Tail	127.8	1.5	8.9	13.0	76.6	10.2	10.3	29.1	52.7	37.2
	Feed	343.6	5.8	32.2	16.7	45.2	100	100	100	100	100
SP-11 CMC-7	Pb-R	40.9	27.3	26.7	14.8	31.2	41.6	7.2	8.3	7.7	11.9
	Zn-R	250.3	4.6	43.8	17.9	33.7	52.9	90.2	76.2	68.8	72.8
	Tail	52.8	2.2	6.0	17.3	74.5	5.5	2.6	15.5	23.5	15.3
	Feed	344	6.9	36.0	17.5	39.6	100	100	100	100	100
SP-12 DETA	Pb-R	41.9	17.7	37.0	15.0	30.3	49.1	11.8	10.0	7.2	12.5
	Zn-R	185.9	2.6	46.1	16.1	35.2	40.5	83.1	60.3	47.5	55.5
	Tail	107.2	1.1	4.9	13.8	80.2	10.4	5.1	29.7	45.3	32.0
	Feed	335	4.0	31.8	15.2	49.0	100	100	100	100	100
SP-13 CMC-4	Pb-R	44.6	17.9	31.1	14.3	36.7	42.5	9.1	8.2	8.4	13.0
	Zn-R	209.1	3.3	46.2	19.3	31.2	49.2	85.8	70.0	53.4	61.2
	Tail	88.1	1.3	6.5	14.3	77.9	8.2	5.1	21.8	38.3	25.8
	Feed	341.8	4.7	34.0	17.3	44.0	100	100	100	100	100
SP-14 PS-2	Pb-R	42.4	23.1	33.2	15.5	28.2	48.2	9.8	9.1	7.0	12.3
	Zn-R	191.9	3.5	48.7	19.8	28.0	42.2	83.5	67.8	45.9	55.9
	Tail	109.2	1.4	6.9	11.9	79.8	9.7	6.7	23.2	47.1	31.8
	Feed	343.5	5.2	33.5	16.8	44.5	100	100	100	100	100
SP-15 CMC-7	Pb-R	38.2	19.1	29.1	13.6	38.2	42.2	7.9	8.1	8.5	11.2
	Zn-R	231.4	3.2	47.0	17.7	32.1	49.9	89.2	74.4	59.6	67.7
	Tail	72.1	1.6	5.0	13.4	79.9	7.9	3.0	17.5	31.9	21.1
	Feed	341.7	4.7	36.1	16.4	42.9	100	100	100	100	100

Table 21: Effect of suppressant dosage tests

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				Wt. %
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	
SP-16 CMC-7 25 g/t	Pb-R	33.9	27.9	32.7	13.6	25.9	50.6	10.2	8.6	6.2	10.1
	Zn-R	218.1	3.7	42.0	17.4	36.9	44.0	87.2	73.5	56.9	64.7
	Tail	85.2	1.2	3.2	10.9	84.7	5.4	2.6	17.9	36.9	25.3
	Feed	337.2	5.5	31.3	15.4	47.9	100	100	100	100	100
SP-17 CMC-7 50 g/t	Pb-R	43.9	30.4	29.5	15.5	24.7	46.1	7.6	8.0	6.8	12.5
	Zn-R	265.6	4.1	43.5	19.1	33.3	49.6	90.7	80.2	74.3	75.6
	Tail	41.7	2.2	5.0	17.8	74.9	4.3	1.7	11.7	18.9	11.9
	Feed	351.2	7.1	37.2	18.5	37.2	100	100	100	100	100
SP-18 CMC-7 75 g/t	Pb-R	39.3	34.9	29.5	14.4	21.2	51.2	7.8	7.7	6.3	11.6
	Zn-R	253.9	3.8	43.9	18.6	33.7	43.0	89.8	76.2	73.8	75.2
	Tail	44.5	2.9	6.5	22.5	68.1	5.7	2.3	16.2	19.8	13.2
	Feed	337.7	7.3	37.3	18.6	36.8	100	100	100	100	100
SP-19 DETA 150 g/t	Pb-R	39.6	23.0	41.0	13.1	23.0	52.4	13.7	8.8	6.1	12.0
	Zn-R	157.4	2.8	45.0	15.0	37.2	30.8	71.9	48.4	41.3	47.6
	Tail	133.6	1.8	10.6	15.6	72.0	16.8	14.4	42.7	52.7	40.4
	Feed	330.6	4.8	30.6	15.0	49.5	100	100	100	100	100
SP-20 DETA 200 g/t	Pb-R	36.1	26.5	28.9	13.2	31.4	53.1	8.8	8.1	7.0	10.7
	Zn-R	176.5	3.1	46.3	15.4	35.1	33.7	75.9	50.5	45.0	52.1
	Tail	126	1.7	13.1	17.7	67.4	13.2	15.3	41.4	48.0	37.2
	Feed	338.6	5.1	32.1	16.1	46.7	100	100	100	100	100
SP-21 DETA 250 g/t	Pb-R	37.7	23.7	28.7	13.5	34.1	45.9	8.1	7.8	7.5	10.8
	Zn-R	183.5	3.8	49.0	16.1	31.1	41.4	77.1	51.7	44.3	52.7
	Tail	127.1	1.7	13.6	18.3	66.4	12.7	14.8	40.6	48.2	36.5
	Feed	348.3	5.2	33.9	16.6	44.3	100	100	100	100	100
SP-22 PS-1 150 g/t	Pb-R	40.1	29.6	27.7	14.2	28.5	49.1	7.6	7.5	6.9	11.5
	Zn-R	212.2	4.0	48.8	18.2	29.0	43.0	86.2	62.3	52.2	60.6
	Tail	97.9	1.6	7.6	19.0	71.7	7.9	6.2	30.1	40.9	28.0
	Feed	350.2	6.3	34.9	18.0	40.9	100	100	100	100	100
SP-23 PS-1 200 g/t	Pb-R	40.6	30.0	26.6	13.7	29.7	54.5	7.6	7.9	6.9	11.7
	Zn-R	205.4	3.3	48.1	16.8	31.7	37.9	86.1	60.3	51.7	59.4
	Tail	99.6	1.4	7.2	18.3	73.1	7.6	6.3	31.8	41.4	28.8
	Feed	345.6	5.9	33.8	16.9	43.4	100	100	100	100	100

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Wt. %
<b>SP-24</b>	Pb-R	37.7	31.1	28.7	13.6	26.6	50.6	8.2	7.9	6.4	10.8
	Zn-R	191.5	4.1	50.7	17.1	28.1	38.6	84.3	57.7	44.8	55.1
	Tail	118.3	1.9	7.3	16.5	74.3	10.8	7.5	34.4	48.8	34.0
	<b>Feed</b>	<b>347.5</b>	<b>6.2</b>	<b>33.6</b>	<b>16.5</b>	<b>43.7</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-25</b>	Pb-R	34.2	30.2	26.3	12.6	30.9	50.9	7.5	7.4	6.7	9.8
	Zn-R	172.6	4.2	53.2	18.0	24.6	36.8	79.2	55.5	37.2	49.5
	Tail	142.2	1.7	10.9	14.6	72.9	12.3	13.3	37.1	56.0	40.7
	<b>Feed</b>	<b>349</b>	<b>5.7</b>	<b>33.3</b>	<b>16.1</b>	<b>44.9</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-26</b>	Pb-R	35	28.1	27.1	12.7	32.1	50.6	7.8	7.6	6.8	9.9
	Zn-R	158.9	4.2	53.8	18.4	23.6	36.7	75.0	53.5	32.7	45.0
	Tail	158.9	1.5	12.4	13.4	72.8	12.7	17.2	38.9	60.5	45.0
	<b>Feed</b>	<b>352.8</b>	<b>5.3</b>	<b>32.5</b>	<b>15.6</b>	<b>46.6</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-27</b>	Pb-R	33.4	28.9	27.6	12.7	30.9	52.9	8.6	8.0	6.5	9.7
	Zn-R	141.2	4.3	54.6	17.7	23.4	33.5	73.2	48.4	28.7	41.0
	Tail	169.6	1.4	11.3	13.3	74.0	13.6	18.2	43.6	64.8	49.3
	<b>Feed</b>	<b>344.2</b>	<b>5.3</b>	<b>30.7</b>	<b>15.1</b>	<b>49.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-28</b>	Pb-R	28.8	34.1	27.8	12.2	25.9	57.7	9.2	7.8	6.0	8.8
	Zn-R	148.6	3.9	45.3	14.8	36.0	29.9	67.6	42.9	38.0	45.4
	Tail	150.1	1.6	15.5	16.8	66.1	12.4	23.3	49.3	56.0	45.8
	<b>Feed</b>	<b>327.5</b>	<b>5.5</b>	<b>30.1</b>	<b>15.5</b>	<b>48.9</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-29</b>	Pb-R	30.4	35.9	23.6	12.2	28.4	56.3	7.1	7.2	6.2	8.9
	Zn-R	196.9	3.7	47.2	15.6	33.5	35.2	84.8	55.3	48.2	57.5
	Tail	115.4	1.6	7.8	18.0	72.7	8.5	8.2	37.5	45.6	33.7
	<b>Feed</b>	<b>342.7</b>	<b>5.9</b>	<b>31.8</b>	<b>16.1</b>	<b>46.2</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-30</b>	Pb-R	28.8	30.8	23.4	12.0	33.8	51.4	6.7	7.1	6.9	8.3
	Zn-R	156.8	4.6	52.8	17.0	25.6	36.5	72.5	47.5	33.4	45.1
	Tail	161.8	1.5	14.7	15.8	68.0	12.1	20.8	45.5	59.7	46.6
	<b>Feed</b>	<b>347.4</b>	<b>5.3</b>	<b>32.6</b>	<b>16.0</b>	<b>46.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-31</b>	Pb-R	30.2	33.8	30.6	12.9	22.7	53.7	9.5	7.9	5.8	9.1
	Zn-R	132.1	4.3	51.1	15.0	29.5	27.7	63.6	37.2	31.4	40.0
	Tail	168.2	2.3	16.9	17.4	63.4	18.7	26.9	54.9	62.8	50.9
	<b>Feed</b>	<b>330.5</b>	<b>6.0</b>	<b>31.8</b>	<b>16.0</b>	<b>46.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

Table 22: Effect of non-sulphidic gangue suppressant dosage tests

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				Wt. %
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	
SP-32 Na <sub>2</sub> SiO <sub>3</sub> 50 g/t	Pb-R	30.3	39.0	29.3	12.4	19.3	55.1	8.9	7.5	5.5	9.4
	Zn-R	165.4	4.8	52.7	16.7	25.9	33.8	80.2	50.6	39.8	51.2
	Tail	127.3	2.0	9.3	18.0	70.7	11.1	10.9	42.0	54.7	39.4
	Feed	323	6.9	33.4	16.8	42.9	100	100	100	100	100
SP-33 Na <sub>2</sub> SiO <sub>3</sub> 100 g/t	Pb-R	29.8	38.1	27.0	12.1	22.8	50.7	7.6	6.9	5.9	8.9
	Zn-R	178.4	5.4	53.4	16.7	24.5	38.6	81.7	52.0	41.1	53.3
	Tail	126.8	2.1	9.9	18.6	69.5	10.7	10.7	41.1	53.0	37.9
	Feed	335	7.0	34.6	17.0	41.4	100	100	100	100	100
SP-34 Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	27.5	42.8	24.5	11.7	21.0	52.9	6.9	6.8	5.1	7.9
	Zn-R	143.7	6.3	54.5	15.9	23.3	34.0	66.9	40.2	30.1	41.4
	Tail	176.2	2.0	17.4	17.0	63.6	13.2	26.2	53.0	64.8	50.7
	Feed	347.4	7.0	33.3	16.1	43.5	100	100	100	100	100
SP-35 Na <sub>2</sub> SiO <sub>3</sub> 150 g/t	Pb-R	29.1	33.3	32.4	11.8	22.4	57.2	11.0	8.0	5.7	9.1
	Zn-R	154.9	4.1	51.2	16.8	27.8	33.2	82.2	53.5	35.6	48.2
	Tail	137.2	1.3	4.7	13.7	80.3	9.6	6.7	38.5	58.7	42.7
	Feed	321.2	5.6	29.7	15.0	49.8	100	100	100	100	100
SP-36 Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	30.4	37.5	28.3	12.2	22.0	54.2	8.4	7.4	5.4	8.9
	Zn-R	155.4	4.9	55.9	16.8	22.4	33.4	78.4	48.2	32.1	45.4
	Tail	156.3	1.8	9.4	15.3	73.5	12.4	13.2	44.3	62.5	45.7
	Feed	342.1	6.4	32.2	15.7	45.7	100	100	100	100	100
SP-37 Na <sub>2</sub> SiO <sub>3</sub> 250 g/t	Pb-R	26.7	33.6	28.4	11.3	26.7	52.3	8.7	7.0	5.8	7.7
	Zn-R	154.3	4.8	54.5	16.0	24.7	35.1	78.0	46.4	31.4	44.7
	Tail	164.1	1.6	8.7	15.1	74.5	12.6	13.3	46.7	62.8	47.6
	Feed	345.1	5.5	30.7	15.2	48.6	100	100	100	100	100
SP-38 Na <sub>2</sub> SiO <sub>3</sub> 100 g/t	Pb-R	25.4	30.5	35.9	11.2	22.3	52.2	11.9	7.6	5.2	7.5
	Zn-R	147.6	4.7	53.4	16.5	25.4	35.8	79.4	49.9	29.6	43.5
	Tail	166.5	1.4	5.2	12.5	80.9	12.0	8.6	42.6	65.2	49.0
	Feed	339.5	5.0	28.4	14.2	52.4	100	100	100	100	100
SP-39 Na <sub>2</sub> SiO <sub>3</sub> 150 g/t	Pb-R	32.2	28.4	42.0	12.4	17.1	49.4	12.7	7.6	5.1	9.5
	Zn-R	153.7	4.1	50.5	16.9	28.5	32.9	71.4	48.3	35.5	45.5
	Tail	152.2	2.2	11.3	15.6	70.9	17.6	15.9	44.1	59.4	45.0
	Feed	338.1	5.5	32.1	15.9	46.5	100	100	100	100	100

Table 23: Effect of Combining Suppressants

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Wt. %
SP-40 PS-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	25.8	36.9	26.6	11.5	25.0	52.6	8.0	7.0	5.8	7.7
	Zn-R	157.1	5.0	54.2	15.7	25.0	34.3	77.6	45.3	34.0	46.8
	Tail	152.9	2.0	10.4	17.1	70.6	13.1	14.4	47.8	60.2	45.5
	Feed	335.8	6.1	32.1	16.0	45.8	100	100	100	100	100
SP-41 CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	25	36.8	24.0	11.4	27.9	49.8	6.8	6.6	5.9	7.1
	Zn-R	149.3	6.1	54.4	18.1	21.5	37.4	70.0	47.6	29.2	42.4
	Tail	177.7	1.7	15.1	14.6	68.5	12.8	23.2	45.9	64.9	50.5
	Feed	352	6.1	32.4	15.9	45.7	100	100	100	100	100
SP-42 PS-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	25	36.2	28.0	11.6	24.2	51.9	8.6	7.2	6.1	7.8
	Zn-R	140.4	5.7	55.5	16.3	22.5	34.7	73.2	43.4	30.9	44.0
	Tail	153.7	2.0	12.6	16.9	68.4	13.4	18.2	49.4	63.0	48.2
	Feed	319.1	6.3	32.7	16.2	44.7	100	100	100	100	100
SP-43 DETA 150 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	22.4	43.8	21.3	10.6	24.3	52.3	6.0	6.3	5.6	6.7
	Zn-R	130.5	7.2	49.6	15.3	28.0	33.9	55.5	36.3	30.4	38.9
	Tail	182.7	2.1	24.6	17.3	56.0	13.8	38.5	57.4	64.0	54.4
	Feed	335.6	6.8	34.1	16.1	43.0	100	100	100	100	100
SP-44 CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	23.2	37.1	29.1	10.9	23.0	43.7	7.8	6.3	5.8	6.8
	Zn-R	140.6	7.9	57.8	19.0	15.3	39.6	66.1	46.8	26.6	41.3
	Tail	176.6	2.6	18.2	15.2	64.0	16.7	26.1	46.9	67.6	51.9
	Feed	340.4	7.1	35.3	16.5	41.1	100	100	100	100	100
SP-45 CMC-1 100 g/t Na <sub>2</sub> SiO <sub>3</sub> 100 g/t	Pb-R	22.9	40.6	28.0	10.9	20.6	57.7	9.1	7.3	5.2	7.1
	Zn-R	147.2	5.2	52.6	17.7	24.6	32.9	76.3	53.2	32.3	45.8
	Tail	151.1	1.4	9.8	12.8	76.0	9.4	14.6	39.4	62.5	47.0
	Feed	321.2	5.9	30.7	14.9	48.5	100	100	100	100	100
SP-46 CMC-1 200 g/t Na <sub>2</sub> SiO <sub>3</sub> 100 g/t	Pb-R	20.5	39.6	26.1	10.7	23.6	58.2	8.4	7.2	4.9	5.9
	Zn-R	136.8	4.9	52.0	16.9	26.2	29.9	69.4	47.0	27.1	39.5
	Tail	189.1	1.4	12.0	11.9	74.7	11.9	22.2	45.9	68.0	54.6
	Feed	346.4	5.0	28.7	13.8	52.5	100	100	100	100	100

Sample	Stage	Mass (g)	Grade (wt %)				Recovery (%)				Wt. %
			Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	
<b>SP-47</b> CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	22.2	37.4	24.8	10.9	26.9	57.4	8.1	7.4	5.5	6.5
	Zn-R	143.3	4.6	52.5	16.1	26.7	30.8	74.5	47.7	29.3	42.1
	Tail	175	1.5	10.1	12.5	76.0	11.8	17.4	44.9	65.3	51.4
	<b>Feed</b>	<b>340.5</b>	<b>5.1</b>	<b>28.9</b>	<b>13.9</b>	<b>52.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-48</b> CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	26.7	38.8	27.6	12.1	21.5	56.8	8.3	7.5	5.6	8.2
	Zn-R	130	5.4	60.1	18.3	16.2	31.2	70.9	44.6	25.2	39.7
	Tail	170.6	1.6	13.4	15.0	70.0	12.0	20.8	47.9	69.2	52.1
	<b>Feed</b>	<b>327.3</b>	<b>6.1</b>	<b>33.1</b>	<b>16.1</b>	<b>44.7</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-49</b> PS-2 100 g/t CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	22.6	40.9	23.4	10.9	24.8	51.6	6.5	6.2	5.7	6.6
	Zn-R	104.5	7.8	61.5	17.7	13.0	31.2	53.8	32.2	18.0	30.5
	Tail	215.6	2.1	22.0	16.4	59.5	17.2	39.7	61.6	76.3	62.9
	<b>Feed</b>	<b>342.7</b>	<b>6.4</b>	<b>34.1</b>	<b>16.5</b>	<b>43.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-50</b> PS-2 100 g/t CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	22.1	43.7	20.4	10.8	25.1	51.9	5.6	6.0	5.7	6.4
	Zn-R	107.4	7.6	54.7	17.0	20.7	29.7	48.5	30.5	21.9	31.3
	Tail	213.8	2.4	26.0	17.7	53.8	18.4	46.0	63.5	72.4	62.3
	<b>Feed</b>	<b>343.3</b>	<b>6.7</b>	<b>34.6</b>	<b>17.1</b>	<b>41.6</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-51</b> CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	21.2	39.0	23.8	11.1	26.1	53.6	7.3	6.6	5.5	6.2
	Zn-R	117.7	6.1	56.9	17.4	19.6	30.1	62.2	37.1	21.9	34.4
	Tail	203.7	1.9	16.1	15.2	66.8	16.3	30.5	56.2	72.6	59.5
	<b>Feed</b>	<b>342.6</b>	<b>5.7</b>	<b>30.6</b>	<b>15.7</b>	<b>48.0</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-52</b> PS-1 100 g/t CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	28.7	34.6	31.3	12.4	21.7	51.1	9.3	7.2	5.6	8.5
	Zn-R	132.7	5.9	57.1	19.7	17.3	35.1	68.2	46.1	25.9	39.5
	Tail	174.6	1.8	14.3	15.1	68.8	13.7	22.5	46.6	68.5	52.0
	<b>Feed</b>	<b>336</b>	<b>6.2</b>	<b>32.7</b>	<b>16.7</b>	<b>44.4</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-53</b> PS-1 100 g/t CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	20	36.3	27.6	10.5	25.7	51.7	8.5	6.6	5.6	5.9
	Zn-R	127.4	6.1	55.8	17.8	20.3	33.6	67.0	43.7	24.2	37.8
	Tail	189.5	1.8	13.7	13.6	70.9	14.7	24.5	49.7	70.2	56.2
	<b>Feed</b>	<b>336.9</b>	<b>5.5</b>	<b>30.5</b>	<b>15.0</b>	<b>49.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>



Sample	Stage	Mass (g)	Grade (wt %)			Recovery (%)					
			Ga	Sp	Py	Ga	Sp	Py	NSG	Wt. %	
<b>SP-54</b> PS-1 100 g/t CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	20.3	42.3	19.6	11.4	26.8	50.2	5.5	6.1	5.9	5.9
	Zn-R	123.3	7.5	54.9	17.2	20.4	33.6	57.5	34.4	24.9	36.1
	Tail	197.6	2.3	22.0	18.5	57.2	16.2	37.0	59.5	69.2	57.9
	<b>Feed</b>	<b>341.2</b>	<b>6.6</b>	<b>33.8</b>	<b>17.6</b>	<b>42.1</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-55</b> PS-1 100 g/t CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	21.9	43.7	19.0	11.3	26.0	51.3	5.1	6.0	5.6	6.2
	Zn-R	151	6.9	53.9	17.7	21.5	37.3	66.6	43.1	30.0	42.5
	Tail	182.1	1.7	19.0	17.3	62.0	11.4	28.3	50.8	64.4	51.3
	<b>Feed</b>	<b>355</b>	<b>6.5</b>	<b>33.9</b>	<b>17.1</b>	<b>42.5</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>
<b>SP-56</b> DETA 100 g/t CMC-1 250 g/t Na <sub>2</sub> SiO <sub>3</sub> 200 g/t	Pb-R	22.8	34.6	21.2	12.0	32.2	58.9	7.5	8.5	6.6	7.2
	Zn-R	88.3	5.8	57.4	16.5	20.3	26.6	54.4	31.5	17.0	27.7
	Tail	207.5	1.4	17.1	13.4	68.2	14.5	38.1	60.0	76.4	65.1
	<b>Feed</b>	<b>318.6</b>	<b>5.0</b>	<b>28.5</b>	<b>14.1</b>	<b>52.3</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>	<b>100</b>

Table 24: Flotation kinetics data

Sample	Stage	Mass (g)	Time (min)	Grade (wt %)			Cumulative Recovery (%)			Cumulative Grade (wt %)				
				Ga	Sp	Py	Ga	Sp	Py	Ga	Sp	Py	NSG	
KT-1	PF	9.9		9.9	14.6	16.5	59.1	0	0	0	6.4	40.4	20.5	32.7
	Pb-R	52.2	0	44.1	17.7	10.2	28.0	6.0	6.7	6.9	6.5	39.9	20.5	33.1
	Zn-R1	49.1	1	6.4	40.4	20.5	32.7	12.1	13.2	13.8	6.5	39.4	20.5	33.6
	Zn-R2	48.9	2	6.6	39.4	20.6	33.4	17.8	19.2	20.2	9.8	39.0	20.3	34.4
	Zn-R3	46.0	3	6.4	38.4	20.5	34.7	24.4	26.7	28.2	14.1	38.7	19.8	35.4
	Zn-R4	59.0	5	5.9	38.0	19.8	36.3	28.6	32.4	33.6	17.7	39.0	19.2	36.1
	Zn-R5	44.9	8	4.9	37.4	17.6	40.1	34.8	42.4	42.4	23.5	39.4	18.8	36.6
	Zn-R6	74.7	14	4.3	39.9	17.3	38.5	39.3	51.6	49.9	28.7			
	Zn-R7	66.1	22	3.6	41.2	16.5	38.7							
	Tail	539.3		1.4	24.6	12.3	61.7							
<b>Feed</b>	<b>990.1</b>		<b>5.3</b>	<b>29.9</b>	<b>14.8</b>	<b>50.0</b>								
KT-2	PF	8.7		9.2	14.6	15.6	60.6	0	0	0	3.9	52.3	15.9	27.9
	Pb-R	49.8	0	44.9	15.3	9.7	30.2	5.8	13.6	8.3	4.1	51.9	15.9	28.1
	Zn-R1	75.2	1	3.9	52.3	15.9	27.9	11.4	25.6	15.7	4.3	51.5	15.8	28.5
	Zn-R2	67.5	2	4.3	51.5	15.8	28.4	17.2	36.6	22.4	4.4	51.1	15.7	28.7
	Zn-R3	62.5	3	4.7	50.5	15.6	29.2	24.9	51.0	31.4	4.5	49.9	15.7	29.9
	Zn-R4	82.8	5	4.8	50.3	15.6	29.4	32.9	64.6	40.8	4.4	48.5	15.7	31.4
	Zn-R5	86.1	8	4.8	45.8	15.8	33.6	38.4	74.5	48.3	27.8	46.3	15.9	33.5
	Zn-R6	69.8	14	4.0	41.0	15.6	39.3	44.1	83.9	57.8	35.0			
	Zn-R7	80.0	22	3.6	34.0	17.1	45.3							
	Tail	403.3		1.4	9.3	13.6	75.7							
<b>Feed</b>	<b>985.7</b>		<b>5.2</b>	<b>29.3</b>	<b>14.7</b>	<b>50.8</b>								
KT-3	PF	17.2		13.2	15.1	20.9	50.8	0	0	0	7.1	56.7	18.8	17.5
	Pb-R	79.3	0	49.2	20.4	12.3	18.1	5.8	9.9	6.3	6.3	56.1	18.4	19.2
	Zn-R1	52.0	1	7.1	56.7	18.8	17.5	9.7	18.5	11.6	4.1	55.9	18.1	20.1
	Zn-R2	46.1	2	5.5	55.4	18.0	21.1	12.7	26.2	16.2	6.2	55.6	17.9	21.2
	Zn-R3	41.4	3	4.6	55.5	17.5	22.4	16.7	37.8	23.2	9.4	55.5	17.7	21.8
	Zn-R4	62.8	5	4.1	55.0	17.3	23.7	19.3	46.0	28.0	11.8	55.4	17.6	22.3
	Zn-R5	44.7	8	3.7	54.8	16.9	24.6	22.8	57.6	34.9	15.2	54.9	17.6	23.1
	Zn-R6	62.9	14	3.6	55.0	17.2	24.2	25.1	66.1	40.5	18.2			
	Zn-R7	49.0	22	2.9	51.7	17.6	27.7							
	Tail	518.1		1.3	15.8	15.4	67.5							
<b>Feed</b>	<b>973.5</b>		<b>6.6</b>	<b>30.6</b>	<b>16.0</b>	<b>46.8</b>								

Sample	Stage	Mass (g)	Time (min)	Grade (wt %)			Cumulative Recovery (%)			Cumulative Grade (wt %)									
				Ga	Sp	Py	Ga	Sp	Py	Ga	Sp	Py	NSG						
KT-4	PF	4.7		9.6	24.6	15.7	50.1												
	Pb-R	48.5	0	45.0	13.7	10.5	30.7	0	0	0	0	0							
	Zn-R1	43.3	1	11.2	45.6	21.7	21.6	8.4	6.6	6.1	2.1	45.6	21.7	21.6	21.3	21.5	21.3	21.6	
	Zn-R2	35.2	2	10.5	46.9	21.7	20.9	14.9	12.1	11.0	3.7	46.2	21.7	21.3	20.6	22.3	22.7	22.7	22.7
	Zn-R3	41.1	3	7.8	49.7	20.5	22.0	20.5	19.0	16.4	5.8	47.4	21.3	20.6	20.1	22.7	22.7	22.7	23.1
	Zn-R4	67.6	5	7.2	49.7	19.4	23.7	29.0	30.2	24.9	9.3	48.2	20.6	20.1	19.6	22.7	22.7	22.7	23.1
	Zn-R5	61.8	8	6.2	51.5	18.5	23.9	35.6	40.9	32.2	12.6	49.0	20.1	19.6	19.5	22.7	22.7	22.7	23.1
	Zn-R6	76.7	14	5.4	53.5	18.3	22.9	42.8	54.6	41.3	16.5	50.1	19.6	19.5	19.5	22.7	22.7	22.7	23.1
	Zn-R7	30.3	22	4.4	51.4	17.5	26.7	45.2	59.8	44.7	18.3	50.2	19.5	19.5	19.5	22.7	22.7	22.7	23.1
	Tail		549.5		1.7	20.4	14.5	63.4											
	<b>Feed</b>	<b>958.7</b>		<b>6.0</b>	<b>31.2</b>	<b>16.2</b>	<b>46.7</b>												
KT-5	PF	11.8		11.6	13.7	17.8	56.9	0	0	0	0								
	Pb-R	57.4	0	46.0	18.5	10.2	25.3	8.7	15.1	6.1	2.0	66.4	13.4	13.4	13.6	13.7	14.2	15.3	
	Zn-R1	70.2	1	6.7	66.4	13.4	13.6	14.8	28.0	11.7	3.8	66.3	13.8	13.8	14.2	14.2	14.2	14.2	15.3
	Zn-R2	59.8	2	5.5	66.2	14.3	14.0	19.8	38.8	16.7	5.5	65.7	14.2	14.2	14.5	15.3	15.3	15.3	16.5
	Zn-R3	51.4	3	5.3	64.3	15.2	15.3	25.0	50.7	22.8	8.0	64.6	14.5	14.5	15.1	16.5	16.5	16.5	18.4
	Zn-R4	60.0	5	4.7	61.0	15.6	18.8	30.0	62.1	29.8	10.8	63.0	15.1	15.1	15.8	18.4	18.4	18.4	19.2
	Zn-R5	61.9	8	4.4	57.0	17.3	21.2	34.4	71.6	37.2	14.3	60.7	15.8	15.8	16.2	19.2	19.2	19.2	20.8
	Zn-R6	59.3	14	4.0	49.0	19.2	27.8	36.2	75.6	41.0	16.1	59.6	16.2	16.2	17.1	19.2	19.2	19.2	20.8
	Zn-R7	27.5	22	3.7	44.5	21.3	30.5												
	Tail		521.3		1.3	12.1	15.9	70.7											
	<b>Feed</b>	<b>980.6</b>		<b>5.5</b>	<b>31.4</b>	<b>15.7</b>	<b>47.4</b>												
KT-6	PF	16.2		11.2	15.5	19.6	53.7	0	0	0	0								
	Pb-R	54.1	0	56.0	16.0	9.3	18.7	9.8	12.8	5.8	1.4	64.7	15.7	15.7	15.5	12.9	12.9	12.9	
	Zn-R1	63.5	1	10.2	64.7	15.7	9.4	17.2	24.1	11.2	3.7	62.5	15.5	15.5	14.3	16.2	16.2	16.2	17.7
	Zn-R2	60.3	2	8.1	60.1	15.3	16.5	22.9	34.3	16.2	6.0	61.6	15.5	15.5	15.7	16.2	16.2	16.2	17.7
	Zn-R3	54.6	3	6.8	59.8	15.7	17.7	29.4	48.0	23.4	9.6	60.4	15.7	15.7	16.0	17.7	17.7	17.7	20.8
	Zn-R4	76.3	5	5.7	57.5	16.2	20.5	33.6	57.7	29.3	12.9	59.2	16.0	16.0	16.8	19.6	19.6	19.6	20.8
	Zn-R5	58.2	8	4.8	53.8	17.3	24.1	38.2	68.6	37.6	17.6	57.1	16.8	16.8	17.1	19.6	19.6	19.6	20.8
	Zn-R6	72.0	14	4.2	48.1	19.8	27.9	40.0	73.1	42.0	20.4	55.8	17.1	17.1	17.1	19.6	19.6	19.6	20.8
	Zn-R7	35.1	19	3.5	41.4	21.2	33.8												
	Tail		496.3		1.5	15.1	18.4	64.9											
	<b>Feed</b>	<b>986.6</b>		<b>6.7</b>	<b>32.5</b>	<b>17.4</b>	<b>43.4</b>												

Sample	Stage	Mass (g)	Time (min)	Grade (wt %)			Cumulative Recovery (%)			Cumulative Grade (wt %)				
				Ga	Sp	Py	Ga	Sp	Py	Ga	Sp	Py	NSG	
KT-7	PF	9.7		11.0	14.0	17.4	57.6	0	0	0	9.9	58.9	15.7	15.5
	Pb-R	54.6	0	52.1	16.2	9.5	22.2	8.2	9.0	4.9	9.2	59.8	16.0	15.0
	Zn-R1	49.7	1	9.9	58.9	15.7	15.5	13.9	16.6	9.0	8.7	59.8	16.1	15.4
	Zn-R2	40.1	2	8.4	60.9	16.3	14.4	17.8	22.5	12.3	7.8	59.5	16.5	16.2
	Zn-R3	32.3	3	7.3	59.8	16.5	16.4	24.5	34.5	19.5	7.1	58.9	16.8	17.2
	Zn-R4	66.1	5	6.0	58.8	17.3	17.8	30.2	46.4	27.0	6.5	58.1	17.1	18.3
	Zn-R5	67.3	8	5.1	57.2	17.7	20.0	34.8	57.7	34.4	6.1	57.4	17.2	19.3
	Zn-R6	66.1	14	4.1	55.3	18.0	22.6	37.7	66.0	40.3				
	Zn-R7	51.1	22	3.4	52.9	18.4	25.3							
	Tail	585.1		1.3	17.1	15.1	66.5							
	<b>Feed</b>	<b>1022.1</b>		<b>5.9</b>	<b>31.7</b>	<b>15.6</b>	<b>46.8</b>							
KT-8	PF	7.9		7.0	13.6	16.9	62.6	0.0	0.0	0.0	9.3	60.9	14.7	15.1
	Pb-R	46.5	0	47.5	16.2	9.3	27.0	10.8	12.5	5.9	8.3	60.5	14.9	16.3
	Zn-R1	58.3	1	9.3	60.9	14.7	15.1	18.2	23.4	11.4	7.8	60.4	15.2	16.6
	Zn-R2	51.8	2	7.2	60.0	15.2	17.7	23.2	31.7	15.7	7.0	59.7	15.7	17.6
	Zn-R3	39.4	3	6.3	60.2	16.1	17.3	29.5	44.2	22.8	6.3	58.7	16.2	18.9
	Zn-R4	61.2	5	5.1	58.0	16.8	20.0	34.5	56.6	30.6	5.8	56.7	16.9	20.7
	Zn-R5	63.5	8	3.9	55.3	17.7	23.1	40.9	70.7	41.3	5.5	55.7	17.2	21.6
	Zn-R6	80.9	14	4.0	49.8	19.2	27.0	43.3	77.0	46.6				
	Zn-R7	37.9	22	3.1	46.9	20.2	29.8							
	Tail	504.9		1.1	11.3	14.2	73.4							
	<b>Feed</b>	<b>952.3</b>		<b>5.2</b>	<b>29.9</b>	<b>15.2</b>	<b>49.7</b>							
KT-9	PF	13.7		10.9	14.6	18.7	55.8	0	0	0	3.3	80.6	9.8	6.4
	Pb-R	66.5	0	50.4	19.8	10.9	19.0	4.0	20.3	4.6	3.5	80.3	9.8	6.4
	Zn-R1	74.2	1	3.3	80.6	9.8	6.4	8.8	42.1	9.6	3.7	79.6	9.9	6.8
	Zn-R2	80.0	2	3.7	80.1	9.9	6.4	13.1	58.9	13.7	4.1	78.5	10.4	7.0
	Zn-R3	63.1	3	4.1	78.0	10.1	7.8	19.8	78.1	19.5	4.6	76.5	11.3	7.6
	Zn-R4	75.3	5	5.4	75.1	12.1	7.5	28.7	89.8	28.3	4.9	73.9	12.4	8.8
	Zn-R5	38.1	8	8.1	61.3	17.8	12.9	29.9	90.7	30.1	5.0	72.7	12.9	9.4
	Zn-R6	26.9	14	8.9	41.7	26.2	23.2							
	Zn-R7	8.9	19	8.2	27.3	33.2	31.3							
	Tail	537.4		1.5	2.3	18.6	77.7							
	<b>Feed</b>	<b>984.1</b>		<b>6.2</b>	<b>29.9</b>	<b>16.0</b>	<b>48.0</b>							

Sample	Stage	Mass (g)	Time (min)	Grade (wt %)			Cumulative Recovery (%)			Cumulative Grade (wt %)				
				Ga	Sp	Py	Ga	Sp	Py	Ga	Sp	Py	NSG	
<b>KT-10</b>	PF	10.9		11.8	15.2	19.2	53.7	0	0	0	5.8	77.5	10.4	6.3
	Pb-R	61.5	0	56.6	16.3	10.1	17.1	9.8	0	7.3	5.8	77.1	10.6	6.4
	Zn-R1	118.3	1	5.8	77.5	10.4	6.3	17.3	51.3	13.2	5.9	76.5	10.8	6.8
	Zn-R2	90.8	2	5.9	76.6	11.0	6.6	26.2	75.2	20.8	6.0	75.9	11.2	6.9
	Zn-R3	48.6	3	6.0	74.1	11.7	8.3	29.9	83.1	24.2	6.1	75.2	11.7	7.0
	Zn-R4	53.4	5	6.4	73.0	13.1	7.5	33.1	88.4	27.8	6.2	73.8	12.4	7.6
	Zn-R5	36.2	8	7.1	68.8	15.6	8.5	34.1	89.6	28.9	6.3	73.2	12.6	7.9
	Zn-R6	29.2	14	7.9	56.6	20.8	14.7							
	Zn-R7	7.7	19	8.5	48.1	23.6	19.8							
	Tail	547.1		1.9	3.8	20.3	73.9							
	<b>Feed</b>	<b>1003.7</b>		<b>7.0</b>	<b>31.3</b>	<b>16.7</b>	<b>44.9</b>							
<b>KT-11</b>	PF	9.4		8.8	13.7	16.9	60.7	0	0	0	7.8	61.7	14.1	16.5
	Pb-R	53.5	0	47.6	17.2	10.5	24.7	11.0	17.0	7.2	7.0	61.7	14.4	16.8
	Zn-R1	74.8	1	7.8	61.7	14.1	16.5	17.7	30.2	13.1	6.6	61.4	14.9	17.2
	Zn-R2	58.2	2	6.1	61.8	14.9	17.2	22.7	41.2	18.4	6.1	60.1	15.3	18.5
	Zn-R3	49.1	3	5.4	60.5	16.0	18.2	28.7	54.9	25.9	5.7	57.7	15.8	20.8
	Zn-R4	66.1	5	4.7	56.4	16.6	22.4	33.1	65.5	33.2	5.4	55.4	16.3	23.0
	Zn-R5	59.9	8	3.9	48.0	17.8	30.3	37.8	76.0	43.3	5.2	53.8	16.5	24.5
	Zn-R6	48.9	14	3.5	40.6	19.4	36.4							
	Zn-R7	26.8	22	2.9	32.8	19.7	44.6							
	Tail	545.0		1.2	10.0	14.0	74.8							
	<b>Feed</b>	<b>991.7</b>		<b>5.3</b>	<b>27.4</b>	<b>14.8</b>	<b>52.5</b>							
<b>KT-12</b>	PF	9.8		11.1	15.0	17.6	56.3	0	0	0	10.5	56.8	15.5	17.2
	Pb-R	59.3	0	51.9	15.5	9.8	22.7	11.6	12.9	6.7	9.0	57.5	15.8	17.7
	Zn-R1	70.0	1	10.5	56.8	15.5	17.2	19.1	25.0	13.0	8.3	56.9	15.8	19.1
	Zn-R2	63.5	2	7.5	58.3	16.1	18.1	24.3	34.3	17.9	7.3	56.7	16.2	19.8
	Zn-R3	51.7	3	6.3	55.2	15.7	22.8	30.4	48.3	26.0	6.6	55.4	16.5	21.6
	Zn-R4	76.8	5	5.0	56.2	17.1	21.6	39.2	71.3	42.1	6.0	53.4	16.7	23.9
	Zn-R5	80.2	8	4.1	51.2	17.5	27.2	40.4	74.7	45.4	5.8	52.3	16.8	25.1
	Zn-R6	68.3	14	3.3	43.3	17.8	35.6							
	Zn-R7	28.8	22	2.8	36.1	18.6	42.5							
	Tail	491.0		1.2	13.7	16.6	68.6							
	<b>Feed</b>	<b>999.4</b>		<b>6.3</b>	<b>30.8</b>	<b>16.3</b>	<b>46.6</b>							

Sample	Stage	Mass (g)	Time (min)	Grade (wt %)				Cumulative Recovery (%)				Cumulative Grade (wt %)			
				Ga	Sp	Py	NSG	Ga	Sp	Py	NSG	Ga	Sp	Py	NSG
<b>KT-13</b>	PF	8.0		13.0	13.1	16.9	57.0	0	0	0	0	8.3	58.9	13.2	19.6
	Pb-R	58.9	0	44.4	16.7	10.5	28.4	7.7	10.9	4.5	0	7.2	58.8	13.7	20.3
	Zn-R1	47.9	1	8.3	58.9	13.2	19.6	13.9	22.7	9.8	4.1	6.7	59.2	14.0	20.1
	Zn-R2	52.3	2	6.1	58.7	14.1	21.1	18.3	32.2	14.2	5.7	5.9	59.0	14.5	20.5
	Zn-R3	41.1	3	5.5	60.1	14.9	19.5	23.4	46.7	21.4	8.5	5.5	58.2	15.1	21.2
	Zn-R4	64.0	5	4.2	58.8	15.5	21.5	28.8	60.2	29.0	11.4	5.2	56.4	15.6	22.8
	Zn-R5	63.2	8	4.4	55.5	16.9	23.2	32.6	70.7	36.4	14.9	5.0	54.9	16.0	24.1
	Zn-R6	57.1	14	3.5	48.0	18.0	30.5	35.0	76.0	41.3	17.4				
	Zn-R7	34.5	22	3.5	40.2	19.8	36.5								
	Tail	523.1		1.2	9.8	14.2	74.7								
	<b>Feed</b>	<b>950.1</b>		<b>5.4</b>	<b>27.3</b>	<b>14.7</b>	<b>52.5</b>								
<b>KT-14</b>	PF	10.0		14.2	15.0	17.7	53.2	0	0	0	0	9.1	58.1	16.3	16.4
	Pb-R	59.7	0	51.5	14.5	9.6	24.3	11.8	15.2	8.2	2.9	8.1	58.4	16.3	17.2
	Zn-R1	78.8	1	9.1	58.1	16.3	16.4	17.1	25.0	13.4	4.9	7.6	58.1	16.4	17.9
	Zn-R2	50.0	2	6.5	58.7	16.4	18.4	21.8	33.9	18.3	6.9	7.0	57.6	16.6	18.9
	Zn-R3	46.6	3	6.3	57.3	16.6	19.8	27.3	45.9	25.2	10.0	6.6	56.9	16.7	19.7
	Zn-R4	64.1	5	5.2	56.3	17.0	21.5	31.1	54.5	30.6	12.5	6.2	55.7	17.1	21.0
	Zn-R5	48.2	8	4.8	53.7	17.6	23.9	34.9	63.7	37.4	15.9	6.0	54.9	17.3	21.8
	Zn-R6	55.9	14	4.2	49.1	19.2	27.5	36.4	67.4	40.6	17.7				
	Zn-R7	25.2	19	3.5	43.7	20.2	32.6								
	Tail	533.3		1.3	16.5	16.1	66.1								
	<b>Feed</b>	<b>971.8</b>		<b>6.3</b>	<b>30.9</b>	<b>16.2</b>	<b>46.6</b>								

**Table 25: Kinetic parameters for flotation kinetic tests**

Test		k	R <sub>∞</sub>	K <sub>m</sub>	SI wrt NSG	SI wrt Py
KT-1	Sp	0.125	53.4	6.673	1.93	0.88
	NSG	0.113	30.6	3.456		
	Py	0.151	50.1	7.561		
KT-2	Sp	0.186	83.4	15.518	3.73	1.78
	NSG	0.112	37.2	4.165		
	Py	0.150	58.1	8.715		
KT-3	Sp	0.126	64.8	8.165	3.29	1.18
	NSG	0.126	19.7	2.482		
	Py	0.148	46.8	6.926		
KT-4	Sp	0.229	75.2	17.22	8.01	2.58
	NSG	0.125	17.2	2.15		
	Py	0.159	42.0	6.68		
KT-5	Sp	0.136	69.0	9.38	5.92	1.85
	NSG	0.092	17.2	1.58		
	Py	0.117	43.3	5.07		
KT-6	Sp	0.164	78.6	12.89	6.45	2.12
	NSG	0.098	20.4	2.00		
	Py	0.121	50.2	6.07		
KT-7	Sp	0.339	91.8	31.12	25.00	5.17
	NSG	0.166	7.5	1.25		
	Py	0.196	30.7	6.02		
KT-8	Sp	0.410	88.1	36.12	20.58	4.61
	NSG	0.270	6.5	1.76		
	Py	0.278	28.2	7.84		
KT-9	Sp	0.258	75.6	19.50	8.19	2.50
	NSG	0.124	19.2	2.38		
	Py	0.178	43.8	7.80		
KT-10	Sp	0.204	75.9	15.48	5.09	2.01
	NSG	0.118	25.8	3.04		
	Py	0.165	46.8	7.72		
KT-11	Sp	0.185	77.2	14.28	6.40	2.43
	NSG	0.120	18.6	2.23		
	Py	0.136	43.3	5.89		

Test		k	R $\infty$	Km	SI wrt NSG	SI wrt Py
KT-12	Sp	0.225	67.1	15.10	5.55	2.00
	NSG	0.147	18.5	2.72		
	Py	0.184	41.0	7.54		
KT-13	Sp	0.203	73.7	14.96	6.29	2.35
	NSG	0.100	23.9	2.38		
	Py	0.144	44.2	6.36		
KT-14	Sp	0.151	67.8	10.24	4.37	1.63
	NSG	0.119	19.7	2.34		
	Py	0.152	41.3	6.28		

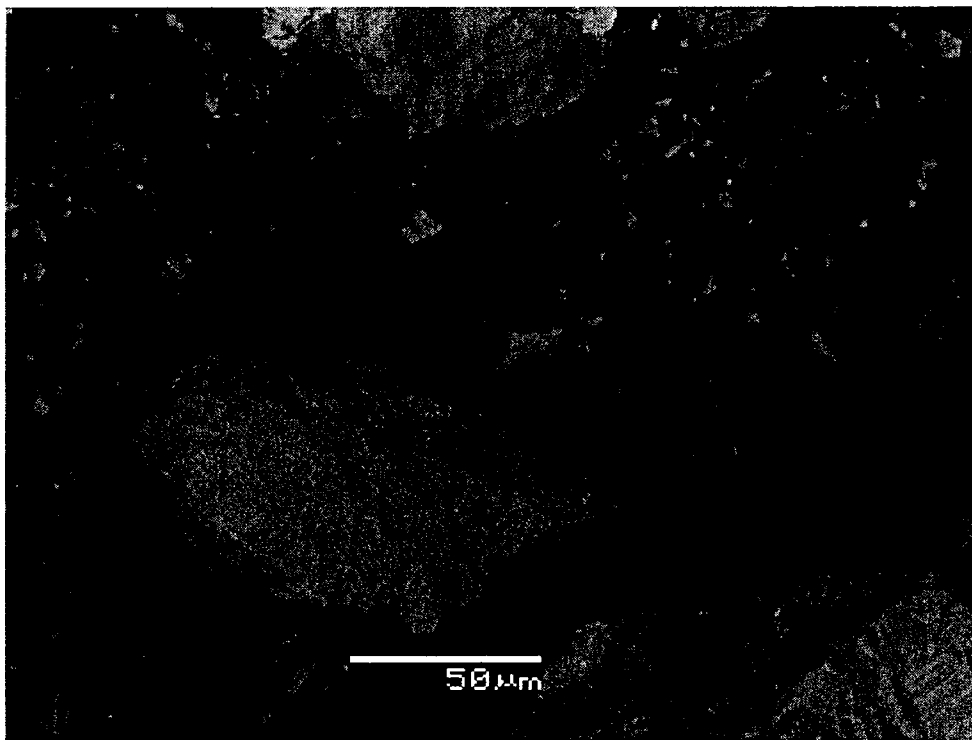
\* wrt: with respect to



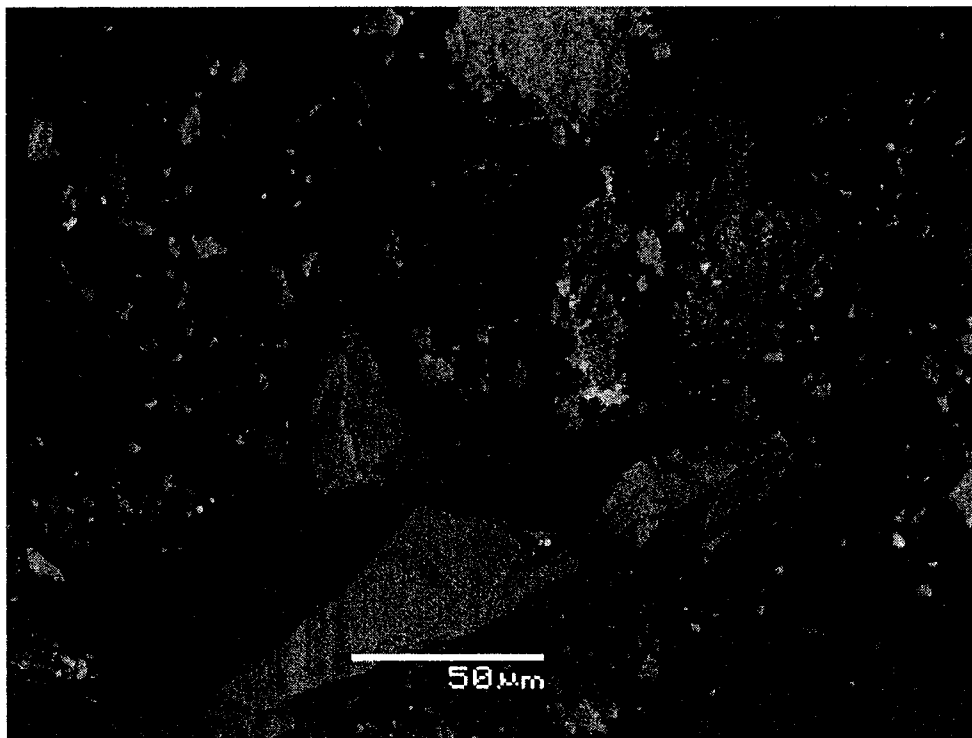
## Appendix 2.0 Scanning Electron Microscope Data

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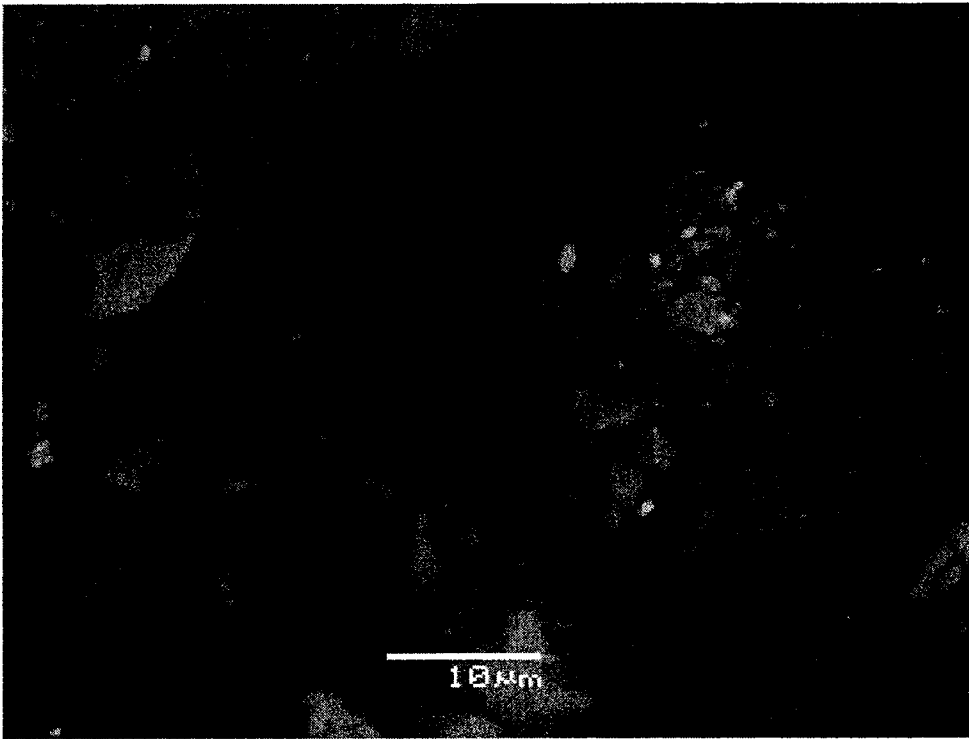
*Appendix 2.1 SEM images*



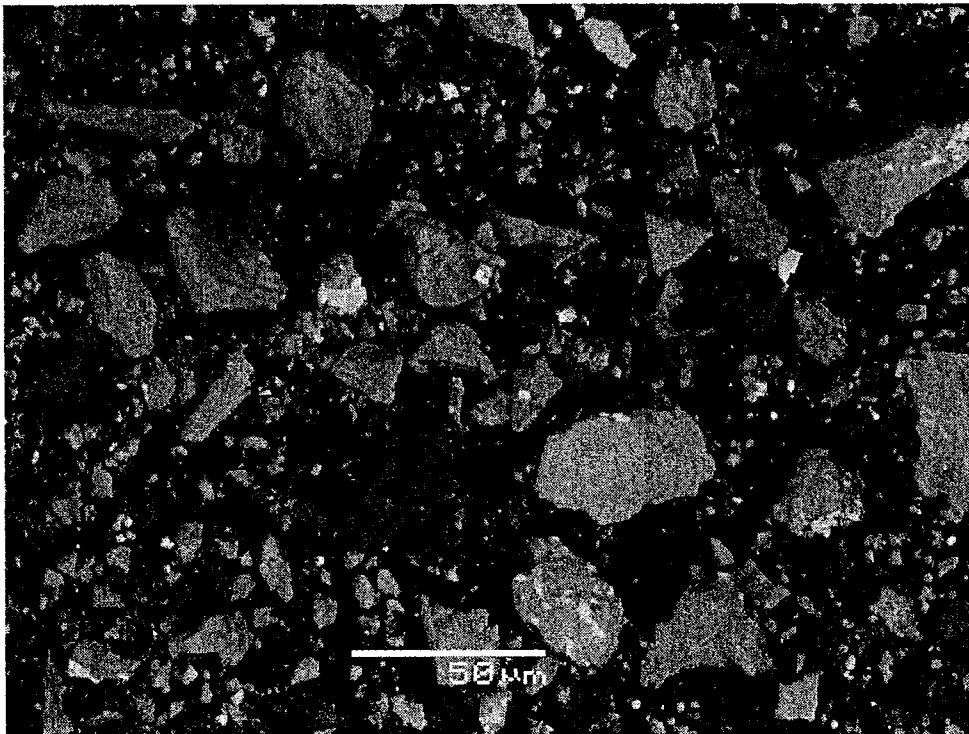
**Figure 27: SEM image of tails for 11 minutes grinding time**



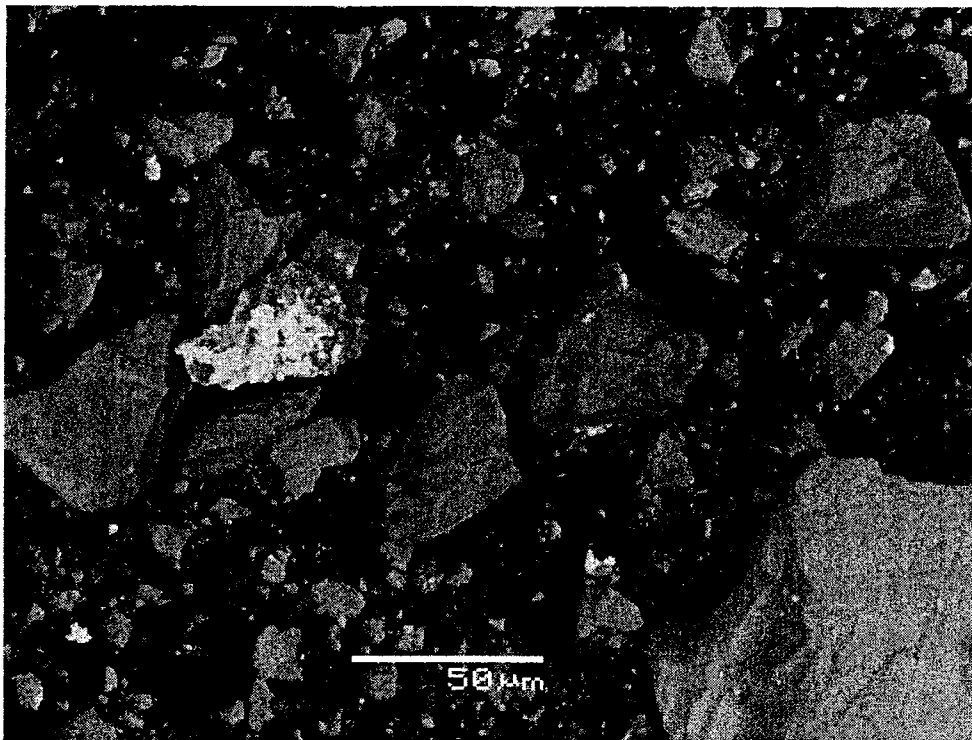
**Figure 28: SEM image of tails for 11 minutes grinding time**



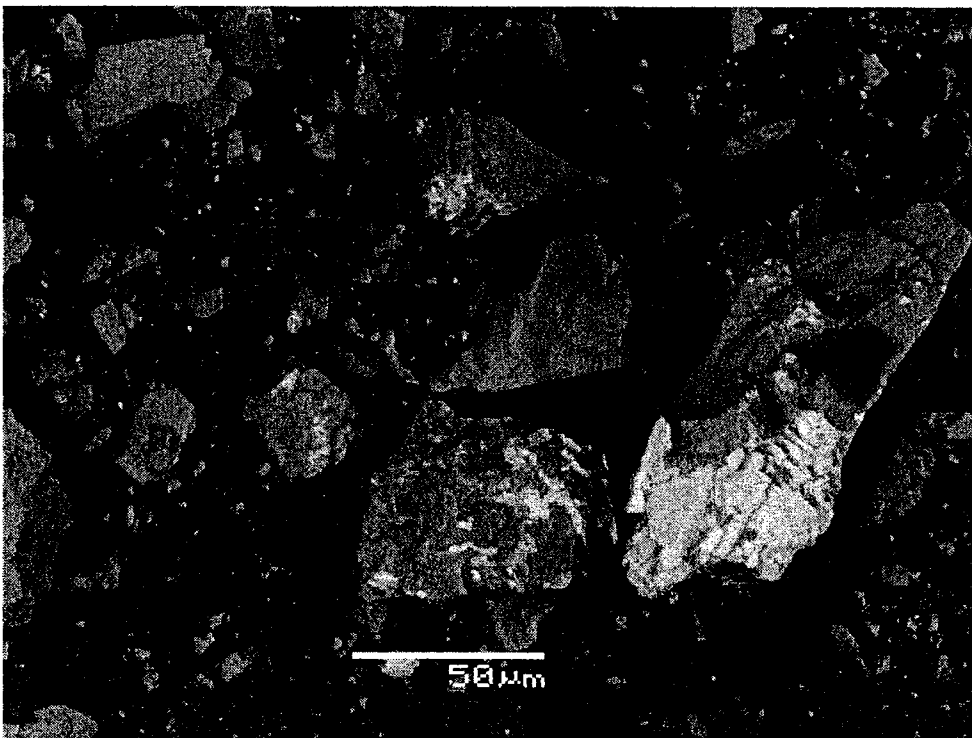
**Figure 29: SEM image of tails for 11 minutes grinding time**



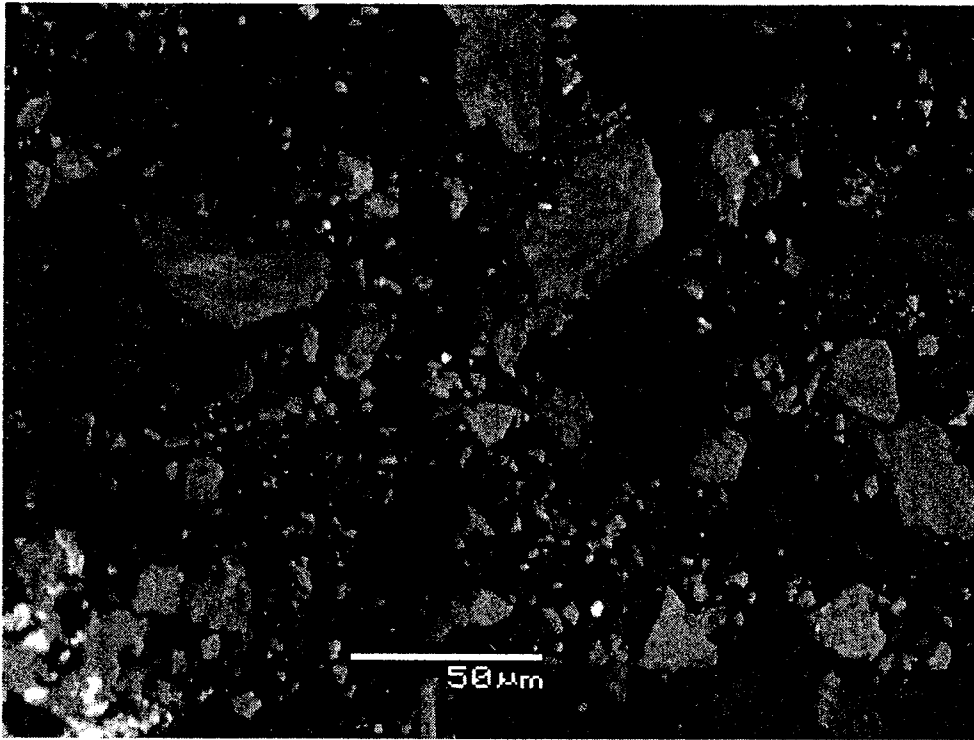
**Figure 30: SEM image of zinc rougher for 11 minutes grinding time**



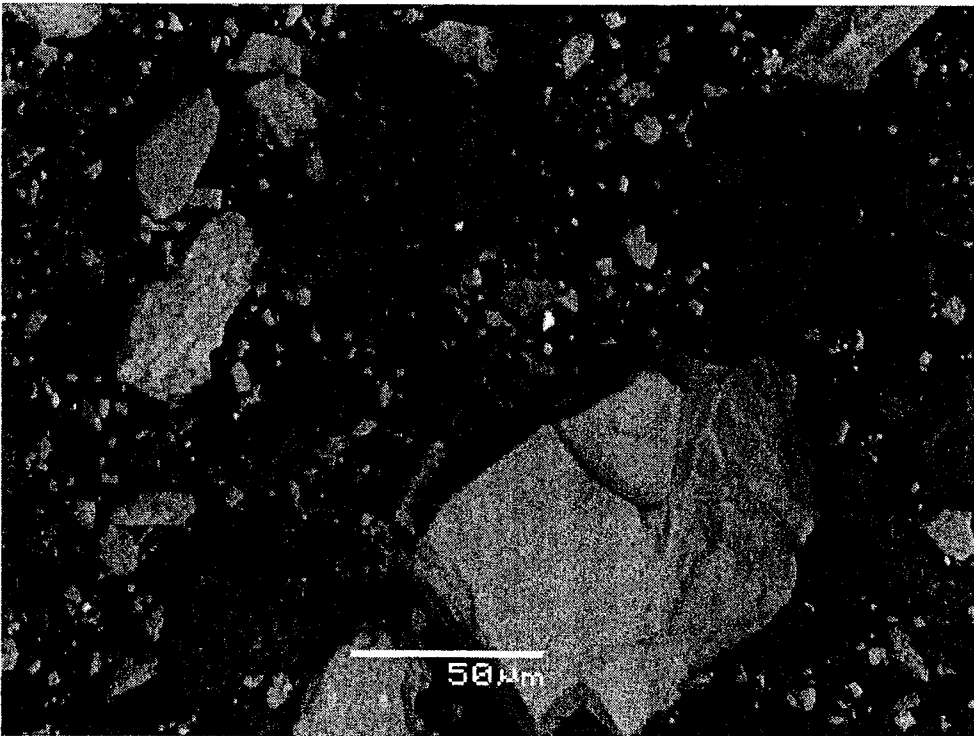
**Figure 31: SEM image of zinc rougher for 11 minutes grinding time**



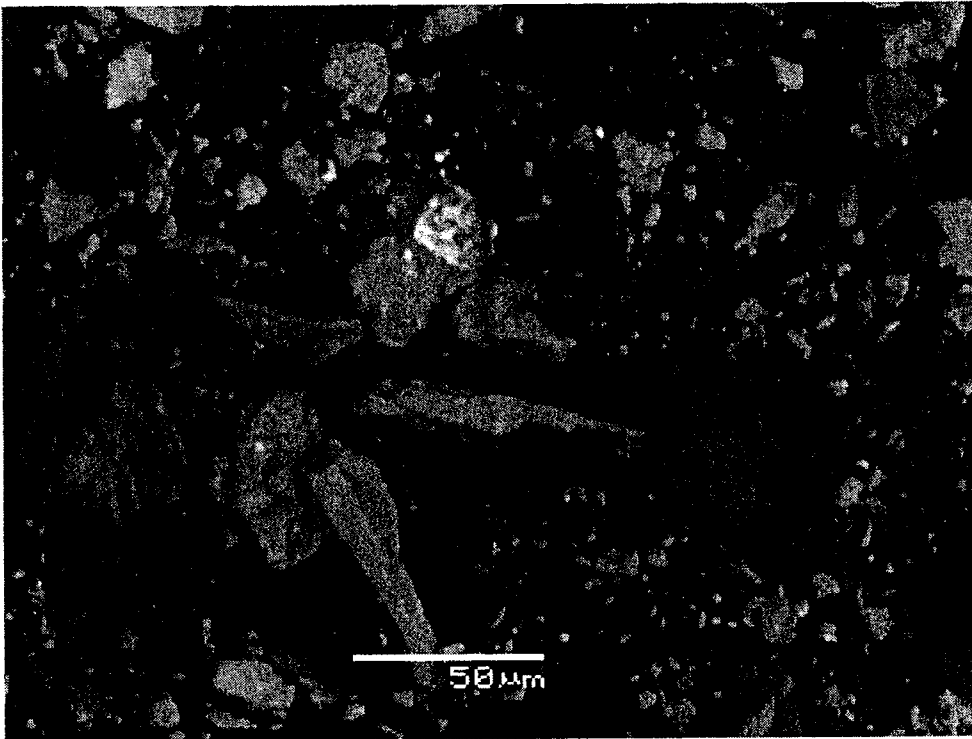
**Figure 32: SEM image of zinc rougher for 11 minutes grinding time**



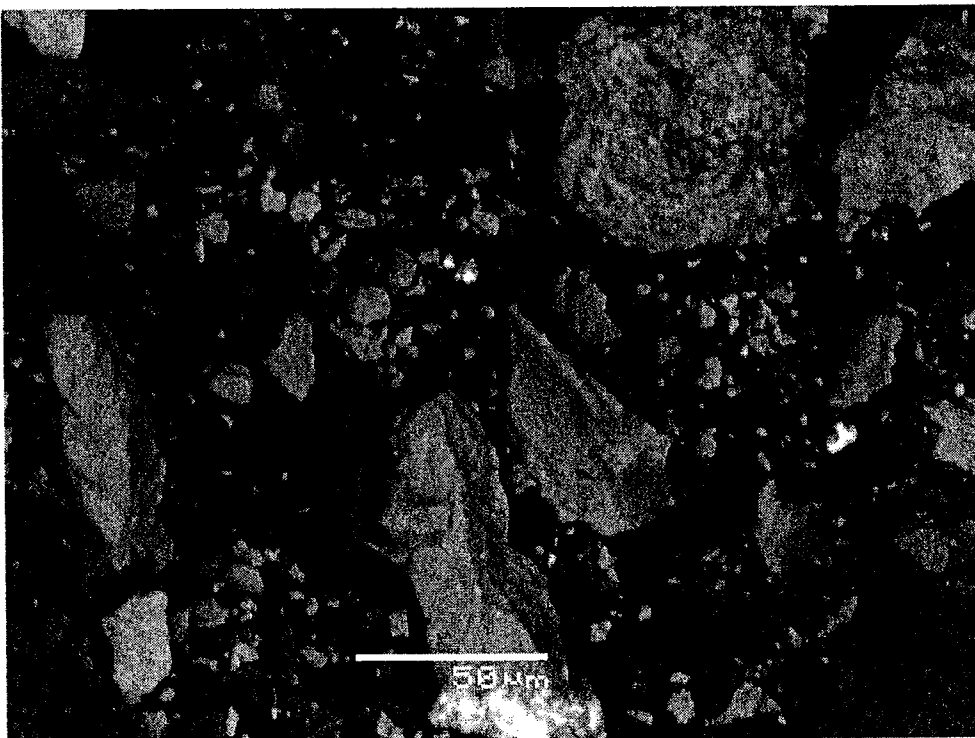
**Figure 33: SEM image of tails for 14 minutes grinding time**



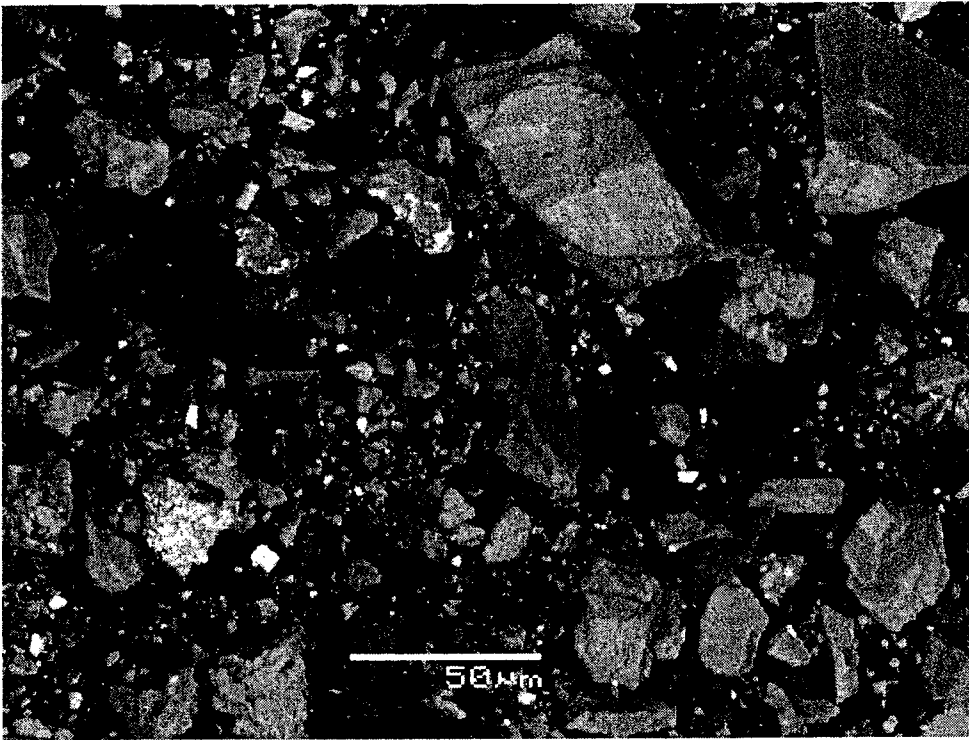
**Figure 34: SEM image of tails for 14 minutes grinding time**



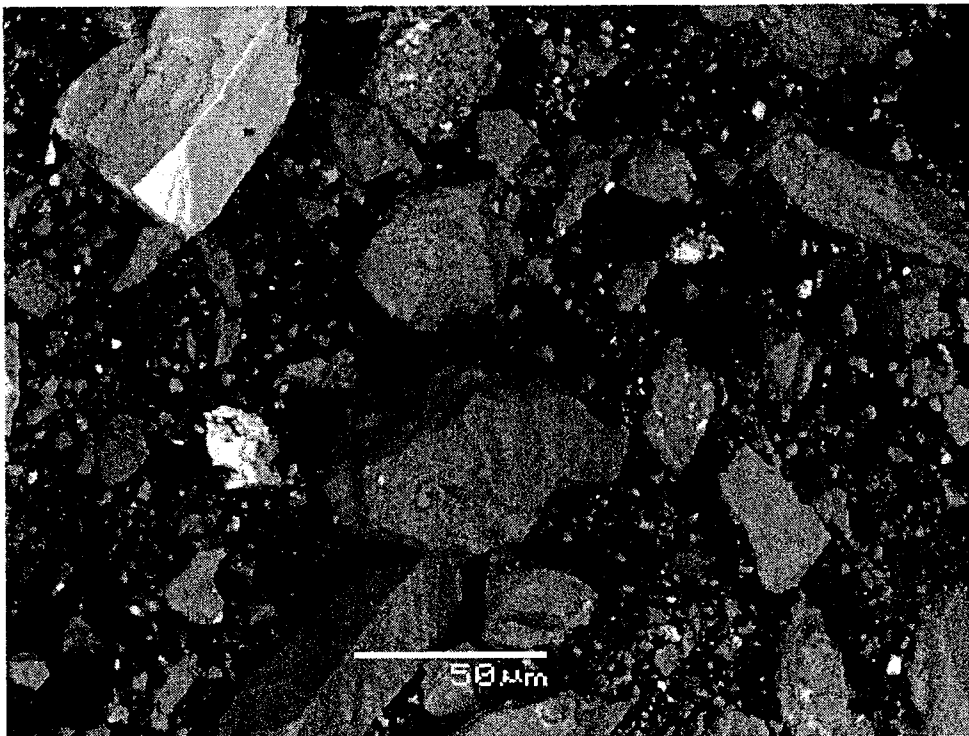
**Figure 35: SEM image of tails for 14 minutes grinding time**



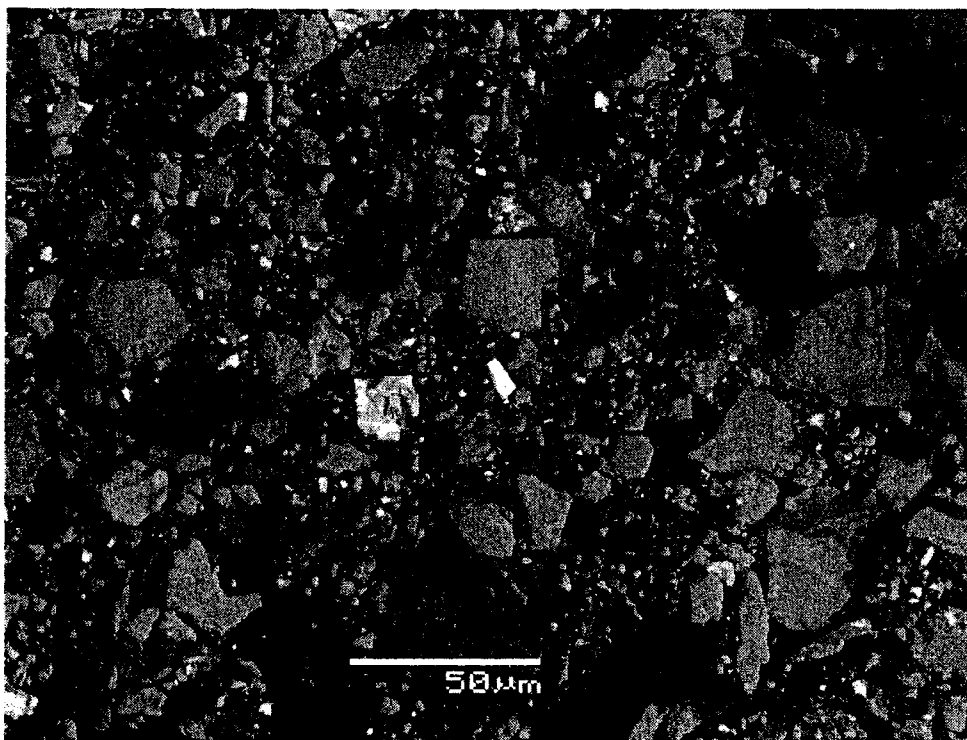
**Figure 36: SEM image of tails for 14 minutes grinding time**



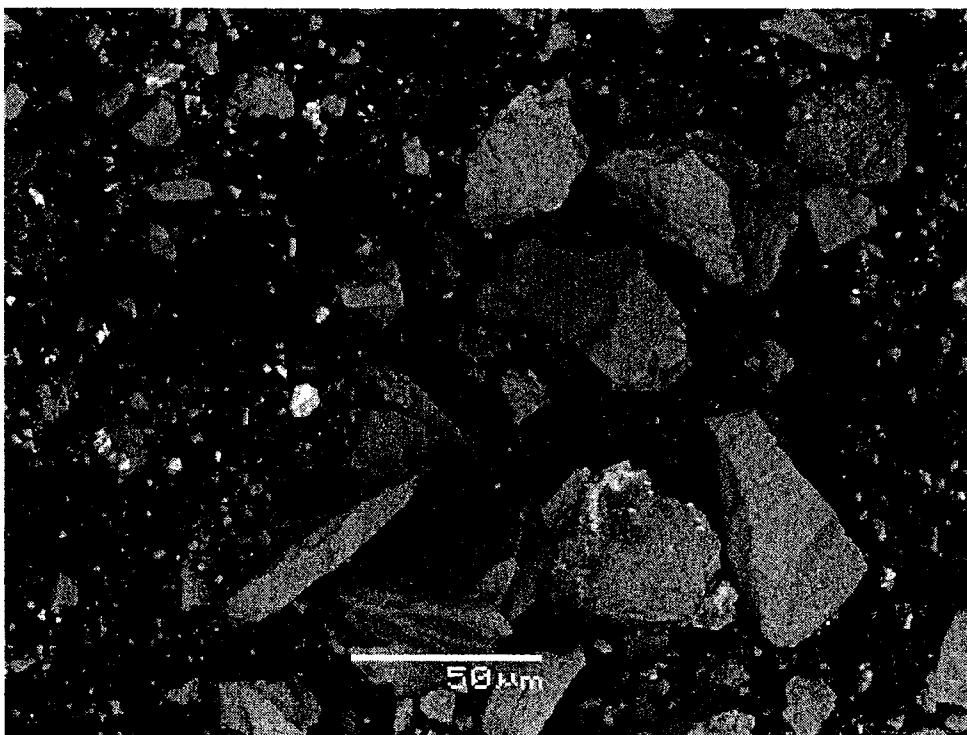
**Figure 37: SEM image of zinc rougher for 14 minutes grinding time**



**Figure 38: SEM image of zinc rougher for 14 minutes grinding time**



**Figure 39: SEM image of zinc rougher for 14 minutes grinding time**



**Figure 40: SEM image of zinc rougher for 14 minutes grinding time**



## Appendix 2.2 X-ray Analysis of Ore

Table 26: X-ray analysis results for ground ore samples

Test	Element %			
	Pb	Zn	Fe	Si
SP-53 Zn-R	2.62	36.93	4.95	1.06
SP-53 Zn-R	46.31	9.72	2.28	1.80
SP-53 Zn-R	0.17	3.51	0.51	42.00
SP-53 Zn-R	2.67	38.45	1.72	1.33
SP-53 Zn-R	11.43	36.09	2.42	3.11
SP-53 Zn-R	2.28	39.11	1.32	1.07
SP-53 Zn-R	45.88	22.64	12.38	2.79
SP-53 Zn-R	46.30	2.18	34.31	1.33
SP-53 Zn-R	9.32	58.41	8.31	3.19
SP-53 Zn-R	15.92	53.44	9.52	1.96
SP-53 Zn-R	3.17	3.68	1.01	42.73
SP-53 Zn-R	25.63	51.19	1.86	2.29
SP-53 Zn-R	76.30	9.68	1.39	1.46
SP-53 Zn-R	15.51	56.00	4.93	3.19
SP-53 Zn-R	40.73	36.29	3.11	2.35
SP-53 Zn-R	7.41	15.66	1.69	32.71
SP-53 Zn-R	43.34	5.48	32.23	1.88
SP-53 Zn-R	54.03	13.21	15.37	2.30
SP-53 Zn-R	75.77	5.66	6.90	1.16
SP-53 Zn-R	71.81	10.17	4.27	2.32
SP-53 Zn-R	9.46	65.61	2.91	2.01
SP-53 Zn-R	6.01	66.01	3.93	2.40
SP-53 Zn-R	8.51	60.96	6.89	2.76
SP-53 Zn-R	79.72	8.03	1.34	1.02
SP-53 Zn-R	57.41	21.50	4.11	2.34
SP-53 Zn-R	14.83	59.06	4.56	2.05
SP-53 Zn-R	76.72	11.04	1.02	0.92
SP-53 T	15.89	1.95	2.45	8.03
SP-53 T	20.69	44.55	4.60	7.20
SP-53 T	13.51	25.68	9.86	18.85
SP-53 T	22.35	50.81	5.01	2.81
SP-53 T	5.36	1.36	55.69	7.63
SP-53 T	70.65	5.85	5.06	4.45
SP-53 T	18.36	49.59	7.79	3.91
SP-53 T	59.27	14.91	6.60	4.63

Test	Element %			
	Pb	Zn	Fe	Si
SP-53 T	9.13	23.29	2.78	6.28
SP-53 T	25.34	47.39	2.37	4.97
SP-53 T	9.77	44.66	2.63	13.92
SP-53 T	10.68	4.84	42.03	13.19
SP-53 T	3.76	47.75	5.62	12.44
SP-53 T	15.45	51.38	1.77	7.12
SP-53 T	28.78	13.19	17.38	13.71
SP-53 T	9.79	26.20	10.00	17.79
SP-55 Zn-R	22.84	57.93	2.67	0.38
SP-55 Zn-R	36.00	22.89	11.13	8.04
SP-55 Zn-R	41.03	1.85	40.10	0.77
SP-55 Zn-R	22.24	52.18	6.15	0.94
SP-55 Zn-R	6.91	28.97	38.98	2.33
SP-55 Zn-R	0.88	76.55	2.64	0.23
SP-55 Zn-R	12.66	14.74	2.37	29.57
SP-55 Zn-R	46.62	0.68	37.56	0.68
SP-55 T	6.38	16.54	0.90	33.24
SP-55 T	38.89	36.36	1.69	5.05
SP-55 T	30.37	47.30	4.04	1.03
SP-55 T	5.53	43.42	1.35	18.30
SP-55 T	25.21	54.66	3.36	0.40
SP-55 T	2.05	20.48	50.52	2.70
SP-55 T	7.05	28.97	38.90	2.57
SP-55 T	0.78	72.81	1.15	3.35
SP-55 T	40.85	28.23	12.98	1.01
SP-55 T	22.91	47.72	4.71	4.41
SP-55 T	76.61	7.86	0.94	2.70