INVESTIGATING IMPACT OF PULP DENSITY ON FLOTATION PERFORMANCE

By

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ABSTRACT

The Red Dog Mine, located in northwest Alaska, is one of the world's largest zinc/lead mines. The processing mill feed consists of a blend of ores from two different pits, namely, the Aqqaluk pit and the Qanaiyaq pit respectively. The mill circuit consists of grinding and multiple flotation circuits which separate zinc and lead minerals from their gangue contents depending on the interfacial tension between hydrophilic/hydrophobic mineral surfaces and their environment. The flotation circuit feed is characterized by high percent solids (~ 50%). Percent solids can potentially have a significant effect on the grade/recovery curve. Thus, it is very common that low-density slurries give better flotation response (high grades), particularly in flotation systems containing a significant amount of liberated hydrophilic unwanted mineral particles. Moreover, the blended feed is metallurgically complex and weathered, thus adversely affecting the performance of the mill.

This project investigated the effects of pulp density on Red Dog flotation circuit performance and develop strategies to maximize recovery at 50% solids. Higher solids content increases the rheology of the slurry thereby causing turbulence and froth instability. To study the impacts of slurry density on flotation kinetics, a series of experiments were conducted by varying various operating and process parameters and assessing circuit optimization strategies.

Initial batch tests performed on cyclone overflow samples showed that residence time, rotor revolution per minutes (RPM), and slurry density are important factors affecting flotation performance. Lower slurry densities usually lead to better kinetics. However, in the case of the initial tests, results indicated that slurry density has a minimal effect if residence time is increased. It was shown that yields as high as 73% with Lead (Pb) recovery values of 86.20% is possible even at 60% solids concentration by increasing the residence time. If the slurry is sufficiently diluted then higher rotor speeds combined with higher residence time would provide higher yields and recoveries.

Initial results indicate that at lower RPM ranges, adequate residence time and higher slurry densities lead to better bubble loading and froth stability. Lead (Pb) and Zinc (Zn) recovery values of 89.42% and 80.33% were achieved at 20% solids and 1800 RPM rotor speed. Future test work includes investigation of froth stability and pulp phase kinetics, statistically, and designed programs to optimize flotation performance in high-density slurries.

Several parameters including dosage, and type of collector, pH, the dosage of frother, dosage of depressant, the dosage of activator, type of grinding media, particle size, and bubble size were controlled in the optimization tests. The optimized condition was obtained for both galena, and sphalerite at different solid%. The locked cycle tests were designed based on the Red Dog flotation circuit. At the optimized condition, the grade, and recovery for solid 30% improved by around 0.5%. The optimized condition had a further impact on the flotation performance at a higher solid%. By increasing the solid%, the grade was improved by 1.84%, and 2.24% at galena concentrate for 40%, and 50%, respectively, compared to the normal condition. Recovery was improved for both solid% by less than 1%. The optimized condition increased Zn grade at the flotation circuit by 1%, and recovery by 4% for 40% of solid. In addition, the optimized condition increased grade at the flotation circuit by 5%, and recovery by 4% for 50% of solid.

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DEDICATION

To my beloved family and teachers

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CHAPTER 1 INTRODUCTION

1.1. Introduction

Lead and zinc are playing a significant role in the global market. They are crucial in a variety of technological fields. Zinc is a widely utilized industrial metal, mainly for the production of brass, bronze, chemicals, and zinc-based alloys, as well as in galvanizing steel for the automotive industries , and construction (Harper et al., 2015; Tabayashi et al., 2009). There is substantial demand for Lead in automotive industries in the lead-acid batteries as there is no replacement for lead in battery production. Thus, it is a crucial metal in the battery industries. Additionally, it is employed in the manufacture of weapons, building supplies, power and communication cables, brass, bronze, construction-use solders, and electrical devices (Harper et al., 2015). Between 1990 and 2014, the amount of zinc used to galvanize steel increased from about 50% to 80%. With more than 80% of the lead produced going toward battery manufacturing, this industry continues to be the leading consumer of lead. Globally, in comparison to lead production, which climbed by 70% between 1990 and 2014, zinc production increased by nearly 90% (Mohr et al., 2018).

Lead and zinc are primarily obtained from sedimentary lead-zinc deposits. Sedimentary deposits provide over 50% of the lead and zinc produced worldwide. Black shales contain sedimentary lead-zinc deposits, including those at Red-Dog (USA), Cirque (Canada), Mount Isa, Elura, and Kooshk (Iran). Galena and sphalerite (Lead and Zinc minerals) commonly co-exist in the same ore deposit. Froth flotation is a conventional method of mineral processing that is used to separate these minerals, with sphalerite being selectively depressed while galena is first recovered by flotation. Then, flotation is used to recover sphalerite from the lead tailings. While the recovery of both lead and zinc and enhancing plant profitability depend on lead-zinc flotation selectivity, the presence of organic matter, carbonaceous pyrite, and host gangue minerals in these deposits affect the process' selectivity between galena, sphalerite, and pyrite species (Basilio et al., 1996; Heydari et al., 2018). The selectivity and effectiveness of flotation can be impacted by a wide range of parameters. The particle size distribution of the feed, the solid percent (pulp density), the dosage and type of the collector, frother, depressant, activator, the pH of the pulp, regrinding, multi-stage cleaning of the rougher concentrates, and the water quality are among these factors (EPA, 1994).

The solid percent of pulp is one of the most important factors. However, it has a significant effect on the flotation, but it has been investigated from limited aspects like its effect on the particle residence time for a narrow range of minerals. Moreover, it is necessary to examine the role of pulp density in both the slurry and froth phases of the froth flotation process. The premise of the current project is to further investigate the role of high pulp density in the Red Dog circuit and develop relationships between effective parameters on the flotation performance. These relationships will provide metallurgists at Red Dog with tools to determine the effects of pulp density and optimize the recovery performance of the flotation circuit.

1.2. Purpose

Mineral processing can be considered one of the most intensive water-consuming processes so water is a significant requirement for this process. In areas where water may be difficult or expensive to gain due to an unfavorable climate like the Red Dog mine, assessment of the availability of an adequate and reliable supply of water for a processing plant is essential. For most low-grade mines which usually have a lifespan that exceeds two decades, the availability of water is a long-term concern. In addition, the water needed for a flotation plant may outweigh all the other uses of water at a mine site, and the need to maintain a water balance is critical for the plant to work efficiently. Unfortunately, it is possible to lose water during processing operation or not immediately accessible to reuse in the processing plant. Then replacing the lost water will be an important issue for mines because of lack of fresh sources (Bleiwas, 2012).

There are a few solutions to solve water problems in processing plant 1) discover fresh water resources 2) treatment of wastewater 3) consume less amount of fresh water. Due to unfavorable climate condition in Red Dog mine replacing fresh water resources with the old one is not affordable and easy. Moreover, consuming fresh water generates more wastewater and pollutes the environment. To prevent consequent problems, recycling and reusing of water would be suggested. However, water treatment seems to be a reliable solution to recycle a greater percentage of the used process water within the flotation cycle because of different reasons which includes shortage of fresh water resources as well as to minimize the discharge of the contaminated water to the environment, but the chemical properties of the recycled water are dissimilar to that of fresh water due to remnant chemicals and other components in recycled water after treatment. These substances include rest flotation reagents, cations of metals, anions of organic, suspended solid/colloid particle, and inorganic pollutants. The chemistry of water is very critical to the effectiveness of the flotation as there is a great concern about possible influence of the recycled water on the efficiency of flotation (Liu & Zhang, 2000; Rao & Finch, 1989). In order to decrease related concerns of recycled water, treatment costs and environmental pollution, this research focuses on decreasing the amount of required water in flotation circuits by increasing solid percent in slurry. Increasing solid percent in flotation circuits affects the grade and recovery of flotation process. In the ensuing sections, these effects would be investigated, and any negative effect was modified.

1.3. Objective

The main objective of this research is to study the effects of pulp density on Red Dog flotation circuit performance indices and develop strategies to maximize recovery at 50% solids. High percentage of solids in the flotation circuit intensifies slurry viscosity and increases sanding issues. In an effort to better understand the impact of slurry density on flotation kinetics, a series of controlled experiments were conducted by varying the feed density and various process parameters, to determine the effect of these variations on flotation rate as well as develop circuit optimization strategies.

The specific objectives are summarized as follows:

- Decrease consumption of water in flotation circuits by increasing solid % in the circuits
- Conduct controlled experiments with various pulp densities
- Control effective parameters on the flotation performance
- Determine chemical treatments to maximize mineral grade and recovery at different solid percentages
- Optimize dosage of chemicals
- Increase the grad and recovery of valuable minerals

CHAPTER 2 BACKGROUND

This chapter presents a general review of flotation process, discussing flotation variables and sub-processes that have an impact on grade and recovery. It further discusses the progress that has been made to date on pulp phase studies. It also reviews chemicals and reagents which are used to improve froth formation, froth stability, flotation selectivity and separation efficiency.

2.1. Introduction to Flotation

Froth flotation is a complicated physicochemical separation procedure that is frequently used to separate desirable minerals and undesired gangue minerals. It is based on differences in the surface characteristics of minerals. The different surface properties of the minerals in the slurry can be seen after using the suitable reagents, such as depressants, activators, promoters, collectors, etc. In the froth flotation, the most crucial mechanism, which accounts for the majority of the particles collected as concentrate, involves the attachment of the precious minerals to the air bubbles. (Zavadskas et al., 2016).

The most commonly used method to concentrate lead-zinc minerals is froth flotation. Several categories of factors affect the process outputs, including chemistry, operational, or equipment. The oxidation degree of the ore, the presence of iron sulfides, the abundance of copper minerals, and the basic or acidic nature of the gangue are the main factors related to the nature of the ore body which influences the effectiveness of the technique (Nagaraj, 2005). In addition to those factors related to the mineralogy of the ore, other factors are also important in the success of flotation activities. These factors include particle size distribution of the feed, solid percent, dosage and type of collector, frother, depressant, and activator, pH of the pulp, regrinding, multi-stage cleaning of the rougher concentrates, etc., and water conditions (EPA, 1994). For optimizing the process flow sheet for a given ore sample, it is necessary to examine the effects of the main variables on the process performance.

2.2. Effective Factors in Froth Flotation

More than 100 variables affect the flotation process which make the process very difficult to understand. Efficient flotation depends on the characteristics of both the pulp and froth phases. For performing efficient flotation, it is necessary to know effective factors and their roles in flotation performance. These factors include (Arbiter, 1962):

1. Ore
    - Mineralogical properties
    - Methods of grinding
    - Particle sizes
2. Reagents
    - Character
    - Quantity
3. Water
    - Quantity with respect to solid percent (pulp density)
    - Quality
    - pH
    - Flotation Agents

  - Dissolved Salts

4. Apparatus
    - Method of aeration
    - Method of froth removal
5. Agitation
6. Time of treatment
7. Temperature of pulp
8. Rheology of pulp

2.2.1. Pulp Density

The solid percent of pulp has a significant effect on flotation which is currently not well understood. It has been investigated for limitations like its effect on the particle residence time for a narrow range of minerals. It may also have a further impact on the slurry and froth phases of the froth flotation process. Lynch et al. studied the influence of high pulp density and found that it causes bubble overload, and can even make flotation impossible (Lynch, 1981). A high density of slurry may also create very fine bubbles which are too heavy to carry particles to the froth zone (Schubert, 1979). Luo et al. tested the effect of various parameters on flotation, including type and dosage of collector, grinding time, flotation pH, and solid-in-pulp concentration. They found increasing solid percentages of the slurry has the most important effect on the recovery and selective separation of lead minerals. Increasing solid percentage from 27% to 55% causes an increase in recovery from 60% to 80% and grade from 27.5% to 29.1% for galena (Luo et al., 2016). Mohamed, and Amina floated aluminum scrap particles with sizes between 0.5 and 1 mm.

The results showed that recovery increased to 85% with increasing solid concentration of the range below 10%. While with increasing solid concentration by more than 10%, recovery will decrease (Mohamed & Amina, 1991).

Alsafasfeh found the grade and recovery of the P_2O_5 will increase from 22% to 23.3% and 28% to 77%, respectively, when solid percent in a slurry increases from 20% to 60% (Alsafasfeh, 2017). An increase of the recovery with increasing the slurry density was observed in other studies as well (Althyabat et al., 2011). This increase could be due to several possible reasons that one of them was explained by the low solid concentration of the slurry, the individual mineral particles are at a larger distance from each other as compared to high pulp density.

Consequently, bubbles will be loaded with less particles and bubble-particle aggregates will be less stable. Lu et al. reported flotation of phosphate minerals at high solid concentration by fatty acid collectors. These collectors have a strong tendency to spread at the surface of apatite and fluorapatite surface and a weak tendency to spread at the quartz surface and making flotation more selective (Lu et al., 1999). Frew showed the percentage of solid in slurry strongly affected the flotation rate at the zinc cleaning circuit. By changing the rate of dilution of water, the solid percent decreased from 55 to 39. Results indicated solid percent has a significant effect on flotation rate as lower solid percent results in faster flotation rates. An exponential function between the flotation rate coefficient and the pulp density could explain this behavior in each cell. The dependency on pulp density was largely independent of mineral type but was dependent upon particle size. Coarse particles (+208 microns) were more considerably affected (Frew, 1982). Runge et al. (2012) indicated changes in slurry solid percent in an industrial-scale machine has a significant impact on flotation rate. Faster flotation kinetics occurred at a lower slurry solid percent in slurry phases presumably because the probability of bubble particle collision and attachment will increase. However, a lower slurry solid percent has a negative impact on the froth phase probably due to lesser froth stability as a result of less hydrophobic solids in the froth. Slurry solid percent was demonstrated to affect the overall flotation rate constant considerably in flotation cells in a fine particle cleaning circuit (P80 ~ 20 microns) with rates at 35% solids being five times slower than the rates at 10% solids. Other tests were performed for fibrous ore at a site processing. Tests showed at the different solid percent of slurry, there are no changes in the kinetics of flotation but in the low-solid percent there is a decrease in froth recovery (Runge et al., 2012). Other researchers have also shown a positive effect on flotation rates by decreasing in solid percent of slurry (Lins & Adamian, 1993; Mehrotra & Kapur, 1974). Some other researchers, including (Gaudin, 1979) have also demonstrated the opposite.

2.2.2. Particles Properties

The floatability of minerals is a function of their particle size, mineralogy and liberation characteristics as well as the chemical environment in the flotation process. Hence, before designing a suitable reagent scheme it is necessary to do a quantitative mineralogical analysis (Celik, 2015; Lotter & Bradshaw, 2010). The mineralogical studies of the ore will be useful for the next steps including; designing flowsheet and reagent schemes, the performance evaluation and optimization of the process (Celik, 2015; Pyecha et al., 2006). In general, a metallurgical assessment is performed, based on particles sizes distribution, the distributions of minerals, and metals among the streams and their liberation (Celik, 2015).

However, even for the same particle size and liberation class, a composite particle with a complex locking texture has a better floatability because of the size and distribution of the hydrophobic phases in the particle (Wang & Fornasiero, 2010). By categorizing the particles into liberated and locked binaries, Sutherland studied how liberation affected the flotation rate of products in terms of particle compositions. He claimed that the flotation rate of the ideal size was higher than that of a fully liberated coarse particle so the coarse particles floated more slowly than intermediate sizes at the same liberation class. The floatability of each liberation class will fluctuate due to the amount of the relevant mineral in a locked particle, so classifying the liberation classes as only liberated and locked would not be appropriate in multi-mineral systems(Sutherland, 1989).

Froth flotation only responds well to mineral particles in a narrow size range, out of which flotation response falls substantially (Collins & Jameson, 1976). This size range varies with minerals and reagent conditions, for instance, 6-70 microns for galena (Figure 1), 8-90 microns for sphalerite, 15-60 microns for chalcopyrite and 20-150 microns for pyrite (Trahar, 1981). It is generally accepted that the problem of mineral fines flotation is mainly due to the low mass and high surface area, leading to a low probability of collision and adhesion of mineral particles to air bubbles. However, other factors, such as surface composition, oxidation, mineralogical alterations, and dissolved ions concentration, etc., are also attributed to the problem (Somasundara, 1980).

There are two ways to improve the poor flotation response of mineral fines, namely increasing the size of floated particles and reducing the size of air bubbles. The former is realized through selective aggregation of desired mineral fines, followed by conventional flotation of the fines in the form of floes, which is termed floc flotation (Xu & Yoon, 1989). Song and et al. used flocs to flocculate fine particles (-20 microns) of galena and sphalerite. At good operating conditions, the floc flotation of galena and sphalerite fines can reach floatability of 100%, in comparison to conventional flotation obtaining floatability of about 40%. Furthermore, addition of kerosene greatly improved the floc flotation, saving a large amount of potassium amyl xanthate (PAX) (Song et al., 2001).

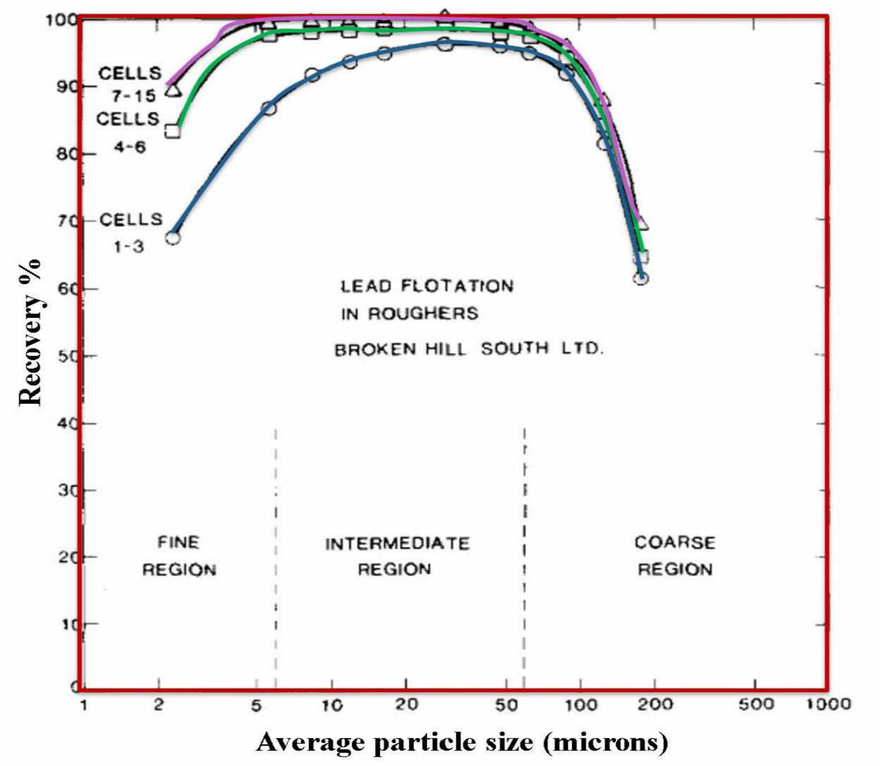


Figure 1. Cumulative lead recoveries in lead roughers at Broken Hill South Ltd.

2.2.3. Reagents

The most important factor to ensure that optimal flotation performance is achieved is the chemical state of the pulp. In this process, various reagents including collectors, frothers, and regulators are added to the pulp. Some reagents are added to modify the mineral surfaces for selective attachment of the valuable mineral on the bubble surfaces; whilst other reagents are used to adjust conditions within the pulp phase and air bubble properties. The reagents that were used in this study were collectors, frothers, pH modifiers, activators, and depressants.

2.2.3.1. Collector

To increase the hydrophobicity of the valuable minerals, collectors are added to the pulp. Each molecule of collectors consists of 2 main parts including a hydrophobic tail that is non-polar and a hydrophilic head which is polar. The polar part of collectors adsorb onto the solid particles because of the physical, chemical, and electrical attraction between the polar part and the mineral, on the other hand, their non-polar part suspends in the pulp (Galbadrakh, 2006). However, the addition of collector might improve the hydrophobicity of particles, but large amount of collectors could potentially have a negative effect on flotation performance. To avoid recovery of undesired minerals in flotation, it is important for collectors to be selective. In order to separate valuable sulfide minerals from gangue, thiol collectors are usually used. Sometimes, thiol collector mixtures have been used because of evidence of enhanced performance over single collectors.

2.2.3.1.1. Flotation of sulfide minerals with thiol collectors

Thiol collectors include the dithiophosphates and the dithiocarbonates (Xanthates). Xanthates are soluble in water and float sulfide minerals but in an unselective manner. The length of carbon chain of non-polar group impact on the recovery power because selectivity declines as carbon chain length increases. Dithiophosphates are frequently used with xanthates for the flotation of complex sulfide minerals.

A xanthate collector is often attached to the surface of a sulfide mineral to control the redox potential through an electrochemical procedure. Three processes are involved in the adsorption of xanthate collectors on surfaces of sulfide minerals: charge transfer during chemisorption of xanthate ions (Eq. 1), oxidation of xanthate to dixanthogen (Eq. 2), and/or formation of metal xanthate (Eq. 3). With the exception of pyrite, practically all electrochemical reactions result in the formation of metal xanthate on the anode and oxygen reduction on the cathode. In the anodic reaction of pyrite, xanthate is primarily oxidized to dixanthogen. Dixanthogen is only formed if the mineral surface reaches a mixed potential higher than the $X^-/X2$ equilibrium value. However, the first molecular layer of xanthate on galena is in the form of chemisorbed xanthate radicals, rather than lead xanthate molecules (Allison et al., 1972; Buckley & Woods, 1997; Kartio et al., 1999; Woods et al., 1997). Based on the specific collector and the mineral, the anodic reaction produces could be chemisorption, metal xanthate or dixanthogen:

$$X^- = X^{ads} + e \qquad (1)$$

$$MS + 2X^- = MX_2 + S + 2e \qquad (2)$$

$$2X^- = X_2 + 2e \qquad (3)$$

$$E = E° - (0.059/2) \log ([X^-]_2 / [X_2]) \qquad (4)$$

In these equations, MS, MX_2, X^-, X_{ads}, X_2 and S represent the sulfide mineral, the metal xanthate, the xanthate ion, the adsorbed xanthate, the dixanthogen and elemental sulfur or polysulphide, respectively. E is redox potential for the oxidation of xanthate to dixanthogen(Arbiter, 1985). Eo is the standard redox potential for the dixanthogen-xanthate couple. Chandra and Gerson have reported the standard redox potential is more positive with decreasing the number of carbon atoms in the hydrocarbon chain of collector (Chandra & Gerson, 2009). Some standard reduction potentials for dithiolates are given in Table 1.

For example, on galena the reduction involves only two electrons, while, in pyrite it is four electrons on a pyrite surface. Minerals surface acted as an electrocatalyst for both the anodic and cathodic reactions so pyrite is more effective catalyst for the reduction of oxygen (Ahlberg &

Elfström Broo, 1996). To be effective the electrochemical interaction between collector and the sulfide mineral: 1) the mineral must be conducting or semi-conducting because electrons can move from the anodic site to the cathodic site, and 2) dissolved oxygen or another oxidizer must be present in solution to act as the electron accepting element.

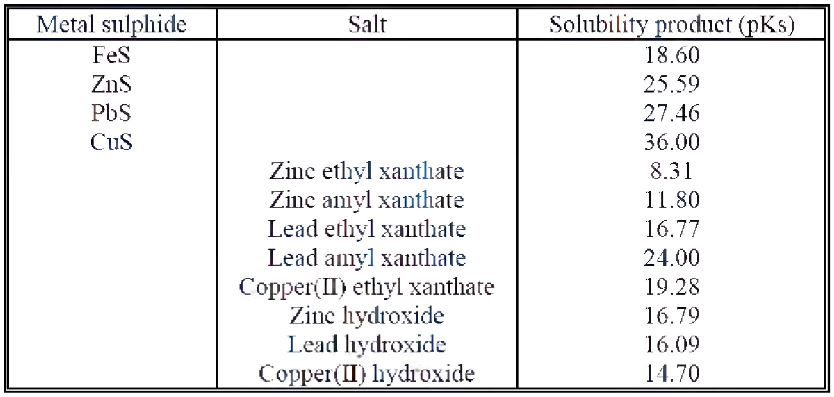
Table 1. Standard reduction potentials for dithiolate/thiol couples of varying chain lengths (Crozier, 1991).

Xanthate	Potential (V SHE)
Methyl	-0.004
Ethyl	-0.060
Propyl	-0.091
Isopropyl	-0.096
Butyl	-0.127
Isobutyl	-0.127
Amyl	-0.159

The solubility products of some metal sulfides, xanthates, and Hydroxides are given in Table 2. The flotation of copper and lead minerals with xanthate will be easier than zinc mineral flotation because their solubility is less than zinc minerals. Moreover, the higher stability of the metal xanthate promotes stronger adsorption of xanthate on the surface of the sulfide mineral.

Guarnaschelli investigated the adsorption of xanthate on galena and discovered that it involved the transfer of an electron from the galena to the xanthate, which was three times more common on the surface of p-type galena semiconductors than n-type. Based on the discovery that p-type galena absorbs three times more xanthate than n-type galena. Galena delivered electrons to xanthate, which, due to its characteristics as a semiconductor, would react with negatively charged thiol collectors(Guarnaschelli, 1968). Galena's surface is quickly oxidized, making it potentially hydrophobic enough for collectorless flotation (Ralston, 1994).

Table 2. Solubility products of some metal sulfides, xanthates, and hydroxides at 25° C(Kakovskii, 1957)



Metal sulphide	Salt	Solubility product (pKs)
FeS		18.60
ZnS		25.59
PbS		27.46
CuS		36.00
	Zinc ethyl xanthate	8.31
	Zinc amyl xanthate	11.80
	Lead ethyl xanthate	16.77
	Lead amyl xanthate	24.00
	Copper(II) ethyl xanthate	19.28
	Zinc hydroxide	16.79
	Lead hydroxide	16.09
	Copper(II) hydroxide	14.70

However, rather than being active in the present of specific mineral, xanthates are active in the present of the entire class of sulfide minerals. Thus, modifiers are employed to make the action of the collector more specific and increase separation efficiency in order to float a specific mineral from a mixture of minerals belonging to the same sulfide class (Finkelstein, 1997). It is important to search for collectors which are capable of selecting one specific mineral rather than the entire class because the desired result is not always obtained during the flotation process of a complex composition like sulfide ores using xanthate.

There are many studies on chelating agents as collectors more selective than xanthates (Barbaro et al., 1997; Ma et al., 2017; Marabini & Barbaro, 1990; Marabini et al., 1991). The selectivity in the flotation separation of complex sulfide minerals increases by using chelate-type reagents (Marabini & Barbaro, 1990; Marabini et al., 1991). Chelating agents, which are actually special complexing agents made up of large organic molecules capable of joining to the metal ion via two or more functional groups and forming one or more rings, create a very stable bond with the metal ion. Numerous parameters that control the selectivity and specificity of the chelation reaction as collector affect the stability of metal chelates. Studies on conventional thiol collectors provided examples of the employment of chelating reagents in flotation. In actuality, thiocarbamate and dixanthogen both function as chelating agents (Ackerman et al., 1987).

However, xanthate is a powerful collector, it is unable to provide the desired selectivity in a complex ore that contains pyrite. Since other collectors draw attention to their usage in sulfide mineral flotation. They consist of collectors of dithiophosphate (DTP), dithiophosphinate (DTPI) or the thionocarbamate type, which are used either alone or in combination with xanthates. They have a significant collectivity power and satisfy the selectivity of other sulfide minerals flotation against iron sulfides (Shen et al., 1998). It is generally advised to use dithiophosphinate and thionocarbamate to separate lead, zinc, copper, and other valuable metals from minerals rich in pyrite (Fairthorne et al., 1997; Pecina-Treviño et al., 2003).

2.2.3.1.2. Mixture of Collectors

According to various reports (Adkins & Pearse, 1992; Bradshaw & O'Connor, 1994; Makanza et al., 2008; Valdiviezo & Oliveira, 1993), utilizing mixes of collectors can improve flotation performance. Plaskin and Zaitseva discovered that a mixture of ethyl and butyl xanthate improves galena recovery by covering the surface more densely and uniformly because these collectors have varied chain lengths(Plaskin & Zaitseva, 1960). Combining cyclohexyl dithiocarbamate with another dithiocarbamate improves recovery, grade, and flotation rate of coarser pyrite ore.

According to Hanson et al.'s research, the glycine promoted more xanthate adsorption because metal glycinate complexes formed in a binary xanthate-glycine system (Hanson et al., 1988). Additionally, xanthate will adsorb in several layers onto a mineral surface in a flotation system and will require minimum assistance from a co-collector (Goh et al., 2008). A mechanism was proposed that either demands for the selective adsorption of collectors onto specified spots or for the molecules' unique orientation to provide good surface coverage (Bradshaw & O'Connor, 1994). In other theory, it was proposed when mixture of xanthate and dithiophosphinate was used as a collector, weaker collectors adsorbed by more oxidized sites and stronger one was adsorbed by less oxidized sites which would result to denser and even more coverage of sulfide minerals surfaces (Bagci et al., 2007). When the dosages of the stronger collector increases, the beneficial effects of collector mixtures become less noticeable (Bradshaw & O'Connor, 1994).

2.2.3.2. Activator

Activators are generally used in the flotation of certain sulfide minerals such as Sphalerite which do not respond well to flotation with short chain thiol collectors. Zinc–xanthate complex is unstable in the pulp hence activators are necessary to improve the adsorption between collector molecules and the sphalerite surface (Chandra & Gerson, 2009). Activators react with the minerals surfaces and modify the chemical nature of them toward more floatable after their interactions with collector.

The activators usually ionize in the solution and then react with the mineral surface (Finkelstein, 1997). The cupric ion (Cu^{2+}) is widely used as activator in the form of sulfate or nitrate. Other heavy metal ions such as Pb^{2+}, Ag^{2+}, Cd^{2+}, mercury and Fe^{2+}/Fe^{3+} are also known to activate zinc sulfide minerals and promote their flotation with xanthate, but they are not used commercially or cause impurities in flotation process (Popov & Vučinić, 1990b; Ralston & Healy, 1980a; Ralston & Healy, 1980b) .The activation mechanisms of sphalerite have been investigated using different methods including micro flotation, electrochemistry, atomic force microscopy, X-ray photoelectron spectroscopy (XPS), scanning electron microscopy with energy dispersive X-ray spectroscopy, and density functional theory computation (Chandra & Gerson, 2009; Popov & Vučinić, 1990a; Tong et al., 2007).

When Cu $^{2+}$ is used to activate sphalerite surfaces, after adsorption, it is reduced to Cu^{1+} with oxidation of sulfide. In this condition, Cu(I)-ethyl xanthate was the dominant surface species (Gerson et al., 1999; Pattrick et al., 1999). Various processes that may occur simultaneously during copper activation of sphalerite were shown in Figure 2. These processes cause the production of hydrophobic and hydrophilic species. However, selective adsorption of Cu($^{2+}$) is the favorite process, but, depending on activation conditions, precipitation of hydrophilic copper containing hydroxide species onto sphalerite surface takes place. In addition, the reaction between aqueous Cu^{2+} and/or Cu (OH)2 with collector molecules leading to nonselective adsorption of hydrophobic CuX or dixanthogen (X2) on sphalerite surfaces. On the other hand, the presence of other sulfide minerals such as pyrite in the pulp has a negative effect on sphalerite flotation activated with copper, their surface can be activated by copper ions and they would be floated simultaneously with sphalerite. There are several factors which control sphalerite flotation activated with copper including impurities, such as iron, surface oxidation, copper and xanthate concentration, activation time, solution O_2 concentration and pH (Chandra & Gerson, 2009).

The mechanism of copper activation of sphalerite is pH-dependent due to the differences in the dominating copper species at acidic and alkaline pH values (Finkelstein, 1997; Girczys et al., 1972; Prestidge et al., 1997) . As illustrated in Figure 3, the copper species can be formed at different pH(Huang & Grano, 2005). The most predominant species at acidic and alkaline pH values are Cu2+ and Cu (OH)2, respectively. In addition, it shows at pH=5.6 for a total copper concentration of 10^{-4}M, Cu (OH)2 precipitation will take place. The initial stage of sphalerite activation at acidic pH is controlled by a chemical reaction in which Cu^{2+} ions replace Zn^{2+} ions in the sphalerite lattice as shown by the following reaction (Finkelstein, 1997; Wang & Forssberg, 1996) :

$$ZnS(s) + Cu^{2+} = CuS(s) + Zn^{2+} \quad (5)$$

While at alkaline conditions, the initial stage of sphalerite activation will be controlled by the precipitation of copper hydroxide on sphalerite surface as shown by the reaction below:

$$ZnS(s) + Cu\ (OH)_2 = ZnSCu(OH)_{2(s)} \quad (6)$$

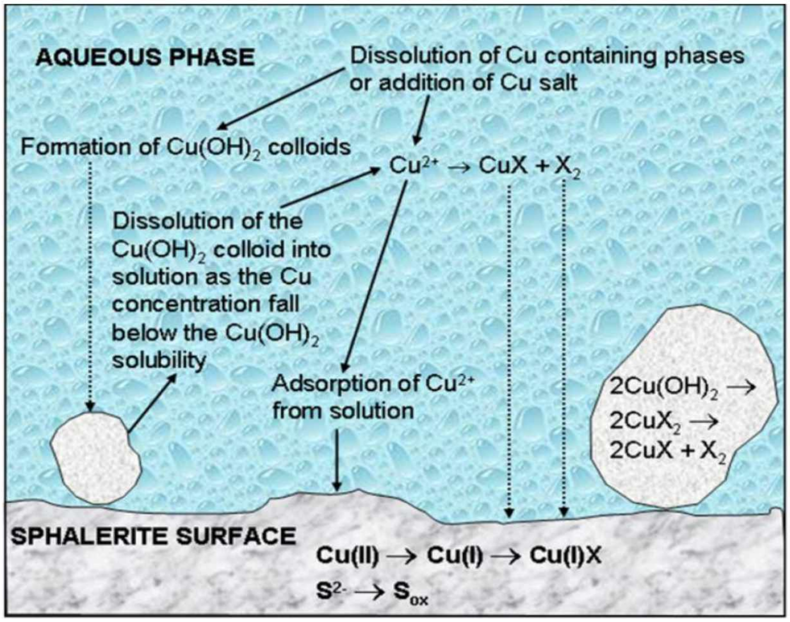


Figure 2. Schematics of sphalerite copper activation showing the various simultaneous processes likely to occur under different activation conditions.

If the conditioning copper concentration is relatively high, then a thick layer of $Cu(OH)_2$ may form. Formation of $Cu(OH)_2$ on the surface can lead to flotation of undesired mineral or gangue, through reaction with xanthate. The $Cu(OH)_2$ layer also obscures the Cu(I)–sulfide layer so lead to surface hydrophobicity. Moreover, $Cu(OH)_2$ precipitated on sphalerite surface may undergo ion exchange with surface Zn^{2+} followed by reduction to Cu(I). During activation time, this $Cu(OH)_2$ will continue replacing with Zn and will slowly move into the solution. This exchange between iron on the pyrite surface and adsorbed $Cu(OH)_2$ does not occur. If copper concentration falls below $Cu(OH)_2$ saturation, then some $Cu(OH)_2$ precipitates will also be lost back into the solution. The effect of $Cu(OH)_2$ on flotation depends on conditioning copper and xanthate concentrations, conditioning time and pH. In higher pH, pyrite tends to promote the formation of iron hydroxide and copper hydroxide species which not only reduce collector adsorption sites but also makes the surface more hydrophilic. Higher pH is ideal for depressing pyrite while lower pH can be used for floating auriferous pyrite. Overall, separation of sphalerite and pyrite can be achieved by the use of high pH, the right oxidation potential and reagent concentration/ conditioning, a short time delay between sequential reagent addition, suitable particle size fraction and minimizing pulp contaminants (Chandra & Gerson, 2009).

Even very low concentrations of $Cu(^{2+})$ ions(10^{-10}M Cu(II) for 1M of Zn(Kakovskii), under equilibrium conditions, it will be able to activate sphalerite. However, at alkaline pH values, such low copper concentrations are not enough to activate sphalerite efficiently. The activation of sphalerite using $Cu(^{2+})$ in alkaline medium will take place in the three steps including: Surface precipitation of the activator metal hydroxide onto the sulfide mineral surface, Cu-Zn replacement into the lattice of the sulfide mineral due to the thermodynamic instability of the Cu(OH)2, and The desorption of the Zn(OH)2 from the surface of sphalerite to the solution would then be important to expose the copper sulfide formed on the surface of sphalerite to the collector (Prestidge et al., 1997; Wang & Forssberg, 1996):

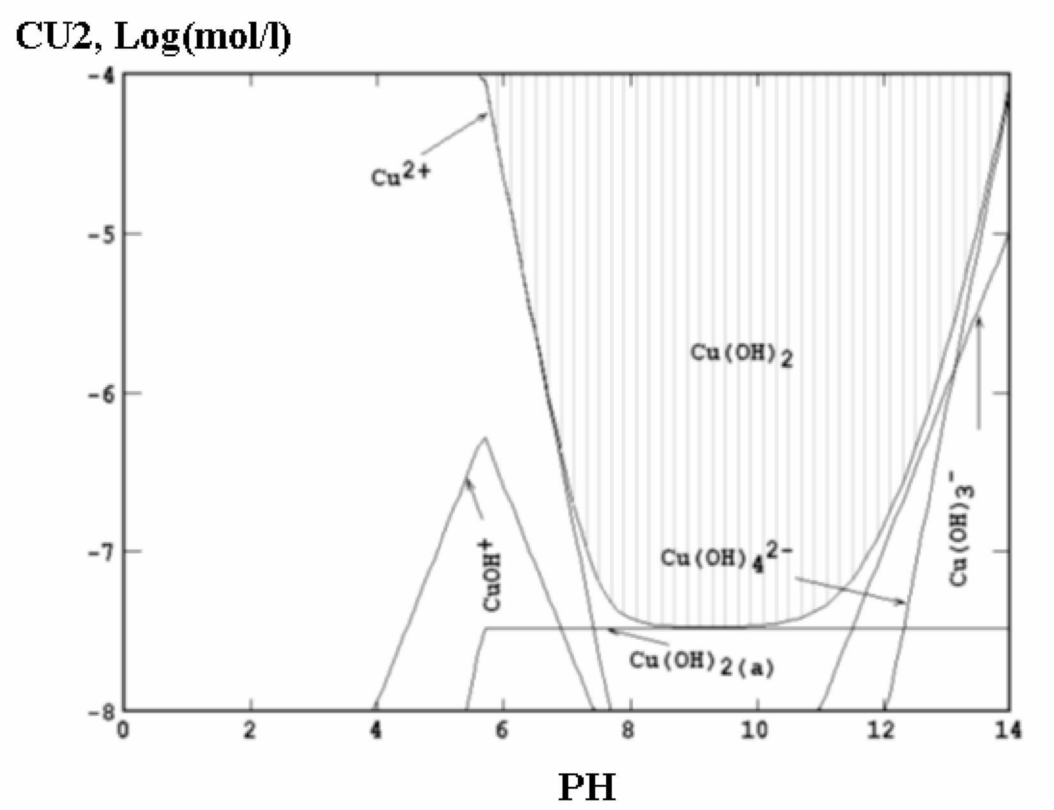


Figure 3. Species distribution diagrams for Cu ($^{2+}$) as function of pH at 25°C. Total copper concentration of 10^{-4}M. Stabcal software, NBS database (Shaded area shows the solid species).

2.2.3.3. Dispersion & Aggregation

There is no clear correlation between the amount of dispersion utilized and flotation performance, but dispersion is occasionally required for the flotation process to avoid the requirement for desliming (Pereira & Peres, 2005; Peres et al., 1994). Non-sulfide flotation techniques employ dispersion to improve selectivity. However, sulfide minerals flotation systems do not typically use it (Silvestre et al., 2009).

Churaev and Derjaguin presented the DLVO theory which explains the theoretical aspects of ultrafine particles aggregation and dispersion by(Churaev & Derjaguin, 1985), and this theory was extended by Adamian and Israelachvili (Adamian, 2000; Israelachvili, 1991). There is an interrelation between dispersion and depressant action in flotation systems. polyphosphates play a special role among dispersing/depressant inorganic agents, and low molecular weight acrylate polymers deserve special attention among organic species.

Rashchi and Finch examined the chemistry and applications of polyphosphates. They found the metal ions-polyphosphate complexes formation may control the action of interfering ions in the flotation process as well as it prevents uncontrolled activation or depression effects (Rashchi & Finch, 2000). Bulatovic used organic polymers in the polymetallic ore flotation. He discussed aspects of the chemistry of starches, dextrins, modified lignosulfonates, and quebracho (Bulatovic, 1999). Bandini et al. found the iron oxides slimes coated galena surface in the flotation. Since, slimes may be removed by a shear conditioning and pH control combination, as well as the use of anionic reagents (carboxymethylcellulose and polyphosphates) (Bandini et al., 2001).

According to Lange et al., aggregating fine sphalerite (up to 100 microns) during xanthate conditioning at pH 5 increased the flotation of them (<20 microns) (Lange et al., 1997). Wightman et al. (2000) demonstrated that the use of a polymer as an aggregation agent can enhance the floatability of ultrafine galena (<5 microns) (Wightman et al., 2000).

2.2.3.4. Depression

2.2.3.4.1. Depression of Pyrite

In the zinc-lead flotation circuits, at the first step, galena is usually floated, and other minerals would be depressed. Pyrite is one of the presented minerals in the ore which must be depressed in lead flotation circuit. Due to natural hydrophobicity of pyrite or the collector-induced hydrophobicity, it is readily floated. Pyrite floatability can also be enhanced by either the activation of copper or lead ions dissolved from complex sulfide minerals or exited ions in recycled water (Dichmann & Finch, 2001; Mu et al., 2016; Zhang et al., 1997).

pH modifier: Increasing the pulp pH with NaOH, Na_2CO_3, and lime is one of the most popular ways to depress pyrite. Several theories can be used to explain why pyrite is depressed, including: 1) Hydrophilic iron hydroxyl on the pyrite surface prevents the adsorption of xanthate as a result of the interaction between hydroxyl ions and surface species. 2) The competitive adsorption of hydroxyl and xanthate anions is the depressing mechanism. Exchange between OH- and X- may cause X- to desorb from the surface of the pyrite. 3) Because pyrite oxidation is favoured by a rise in pH, oxidation of pyrite has supplanted oxidation of xanthate as the predominant anodic process. Thus, at high pH, dixanthogen production is prevented (Gaudin et al., 1942; Janetski et al., 1977; Wark et al., 1932)

Oxidation: The surface composition of sulfide minerals was impacted by Eh because of superficial oxidation. Oxidation of the surface may enhance the flotation of sulfide minerals by forming elemental sulfur, metal-deficient or sulfuric species which facilitating collectorless flotation (He et al., 2005; Luttrell & Yoon, 1984; Tolley et al., 1996). However, extensive oxidation generally reduces mineral floatability by forming hydrophilic and stable metallic oxide/hydroxide coatings (Martin et al., 1991; Senior & Trahar, 1991; Smart, 1991). The outcomes of flotation can be dramatically changed by oxygenating the pulp because it changes the electrochemical atmosphere. Due to preferential reduction of oxygen on pyrite surface, it causes to depress pyrite and decrease its recovery. Additionally, it causes the development of hydrophilic species like sulfate and copper/iron hydroxide. It may be possible to improve the separation of pyrite from sphalerite, galena, and gold by using hydrophilic species to stop or diminish the adsorption of collector and copper ions on pyrite (Monte et al., 1997; Owusu et al., 2013; Peng et al., 2003; Shen et al., 1998). The activity of several sulfide minerals including arsenopyrite, bornite, chalcocite, chalcopyrite, covellite, galena, pentlandite, pyrite and pyrrhotite was investigated for the oxygen reduction in acidic and alkaline solutions and the results are shown in Figure 4. The oxidation rate of sulfide minerals is in the order of pyrite, chalcopyrite, sphalerite, and galena. It can be seen that pyrite is the most active among all the sulfide minerals and galena has the lowest activity (Kelsall et al., 1999; Rand, 1977; Yin et al., 1995).

Boulton et al. studied the flotation of iron sulfide minerals. They found depression of iron sulfide minerals will occur after oxygen conditioning time greater than 20 min, which is time-consuming. It is also found that a combination of oxygen conditioning and additional usage of other depressants can significantly reduce the conditioning time and the dosage of depressants(Boulton et al., 2001a).

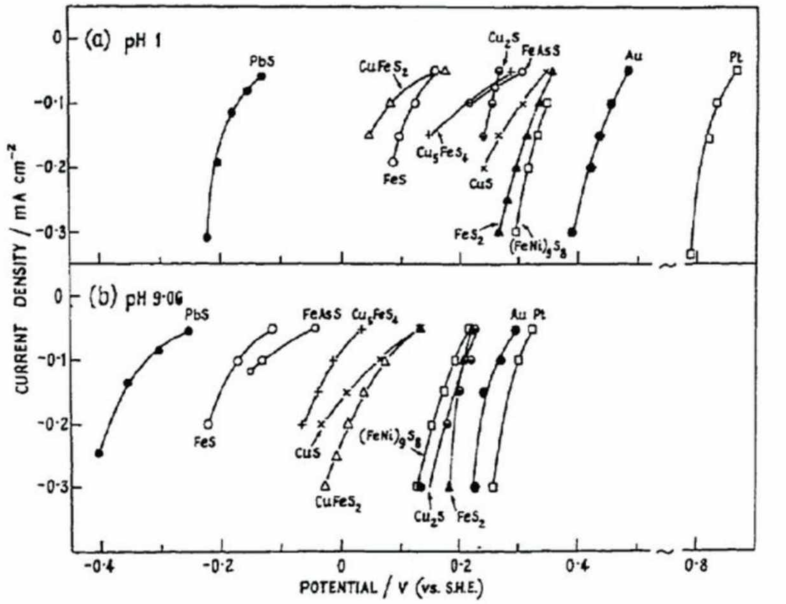


Figure 4. Activation controlled current for oxygen reduction on sulfides (Rand, 1977)

Inorganic depressant: Sodium cyanide is commonly used as a selective pyrite depressant (Bonnissel-Gissinger et al., 1998; Boulton et al., 2001a). The following are some of the potential processes for pyrite flotation depression: 1) Cyanide rather than xanthate ions and hydroxyl ions may adsorb on the surface of pyrite. 2) Insoluble iron cyanide compounds are created when cyanide adsorbs on the surface of pyrite. 3) Thiocyanate may be formed when cyanide reacts with sulfur or polysulfides created by surface oxidation (hydrophilic compound) 4) The addition of cyanide might lower the mixed potential and stop electrochemical action on the pyrite surface, which would prevent xanthate from being chemisorbed and oxidized. Concerns about the environment arise when sodium cyanide is consumed. Consequently, the appropriate selective depressant is needed for the selective separation of lead-zinc sulfide minerals from pyrite (Guo et al., 2014; Guo et al., 2015; Prestidge et al., 1997; Wark et al., 1932). A variety of sulfur-oxy species, including sodium sulfide and zinc sulfate, can be utilized as pyrite depressants to minimize environmental problems. They must be used in pH ranges that are neutral or slightly alkaline. Zinc sulfate causes depression of pyrite by inducing sulfoxyl and hydroxide species on the mineral surfaces. The efficiency of zinc sulfate as pyrite depressant depends on Eh as it cannot function well at lower Eh values due to the less formation of hydroxide groups on the pyrite surface. For depressing pyrite, high dosages of sulfoxyl reagents are usually required so it increases operation costs which limits their application(Bulut et al., 2012; Chandra & Gerson, 2009; He et al., 2006; Khmeleva et al., 2002).

Organic depressant: Various reagents are applied in the selective separation and depression of iron sulfide minerals. They include inorganic depressants such as cyanide, lime, oxygen conditioning, sulfite, etc., as well as organic reagents such starch, dextrin, chitosan, carboxymethyl cellulose, polyacrylamides, wood extracted biopolymers, a combination of these reagents (Bogusz et al., 1997; Bulatovic, 1999; Huang et al., 2013; Shen et al., 1998). However, inorganic depressants are effective, but they are not environmentally friendly, furthermore, they are costly. Organic depressants have biodegradable nature and they are relatively inexpensive so they have the potential to function as alternative depressants (Bicak et al., 2007; Bulut et al., 2012; López Valdivieso et al., 2004; Mu et al., 2018). To improve the selectivity of polymers towards the target mineral, It is suggested to combine the use of polysaccharides with other metal selective chelating agents, organic or inorganic (Bogusz et al., 1997). Boulton et al. studied the iron sulfides depression in sphalerite flotation. Low molecular weight polyacrylamides, thiourea, were efficient pyrite depressants. The depressant action was stronger on coarse particles and the polyacrylamides must be added prior to the collector addition (Boulton et al., 2001b). Guar gum was used as a pyrite depressant. The result showed the addition of as low as 0.1 mg L^{-1} guar gum caused a dramatic decrease in pyrite recovery. Dosage of 10 mg L-1 of it was recommended. Pyrite depression with guar gum was more efficient at high pH because of the formation of hydrophilic iron oxyhydroxy coating. Molecular weight (MW) of guar gum plays an insignificant role in the depression of pyrite flotation (Bicak et al., 2007). Polymer chemistry such as molecular weight, content of functional groups, charge density, and configuration are important factors to consider their effects on pyrite depression. Moreover, the adsorption, conformation and degree of hydration of the polymers on pyrite surface are affected by solution pH, ionic strength, etc. One of disadvantages of organic polymers is at high dosages may result in the depression of all sulfide minerals. However, a major advantage of organic depressants over inorganic reagents is that the polymers have greater flexibility and potential to be modified through various processes, which can effectively increase their selectivity as well as the depression effect (Mu et al., 2016).

2.2.3.4.2. Depression of Sphalerite

The presence of copper and lead ions in the pulp can adsorb to the surface of sphalerite during the lead flotation process, accidentally activating it. These ions boost the sphalerite's recovery by encouraging collector adhesion to the surfaces (Finkelstein, 1997; Seke & Pistorius, 2005). Inorganic depressants are typically used to treat sphalerite, either singly or in combination. In most situations, sphalerite is deactivated and depressed using a mixture of NaCN and $ZnSO_4$, and galena is floated using xanthate (Wang & Forssberg, 1991). $ZnSO_4$ has been used in a number of operations to depress sphalerite and has been shown to have best-countered sphalerite activation in test work and plant operation. ZnSO4 depresses sphalerite by replacing zinc in the form of zinc hydroxyl species with the attached activating copper ions on sphalerite surfaces. The best results of depression were gained at pH 8-10 because of the formation of zinc hydroxyl and zinc hydroxide species (Cao & Liu, 2006). Zinc hydroxyl adsorbs and zinc hydroxide precipitates on the sphalerite surface and create hydrophilic surfaces (Laskowski et al., 1997).

Iron sulfate has been shown and used at plant scale as an alternative to zinc sulfate. Other reagents, such as sodium sulfide and organic depressants, have also been used as depressants in laboratory test work and possibly at plant scale. Huang and et al. tested chitosan as a potential depressant in the differential Pb\Zn sulfide flotation separation. The results showed that the amine groups and the hydroxyl groups in chitosan were involved in the strong interaction between chitosan and sphalerite whether the latter was coated by copper or not (Huang et al., 2013).

2.2.4. pH

The potential difference of a mineral-solution interface, which is indicated by pulp potential, has been shown to be closely related to the floatability of sulfide minerals (Rand & Woods, 1984). Minerals can be floated or sink by oxidation or reduction of them in the pulp so the measured potentials determine whether or not the mineral will float. The measured potential in flotation systems is a mixed potential and it is somewhere between the potential of the minerals and potential of the solution and different electrode materials can yield different Eh values in the same solution (Finch & Labonté, 1989). The collection of Eh-pH data for an operating process may decrease reagent additions, and provide useful additional and helpful information for solving an operational problem for flotation plants (Johnson & Munro, 1988). Eh/pH measurements were examined in relation to the flotation for pyrrhotite by Natarajan and Iwasaki (1973) and Eh was found to be a good indicator of the floatability of minerals.

As it can be seen in Figure *5*, Figure *6*, and Figure 7 the flotation of sulfide minerals with xanthate depends strongly on the pH of the pulp. Figure 5 shows almost 100% recovery of galena obtained at pH between 3 and 10 with $10^{-4}M$ *ethyl-xanthate*. Another research by Ü. Yenial and A. Gül showed galena could float in every pH with high recovery. 100% recovery was obtained at pH between 7-9 using100 g/t of 3418A as collector, H_2SO_4 and lime as pH modifier and 10% of solid (Yenial et al., 2015). However, sphalerite does not float above pH 5 (Figure 7).

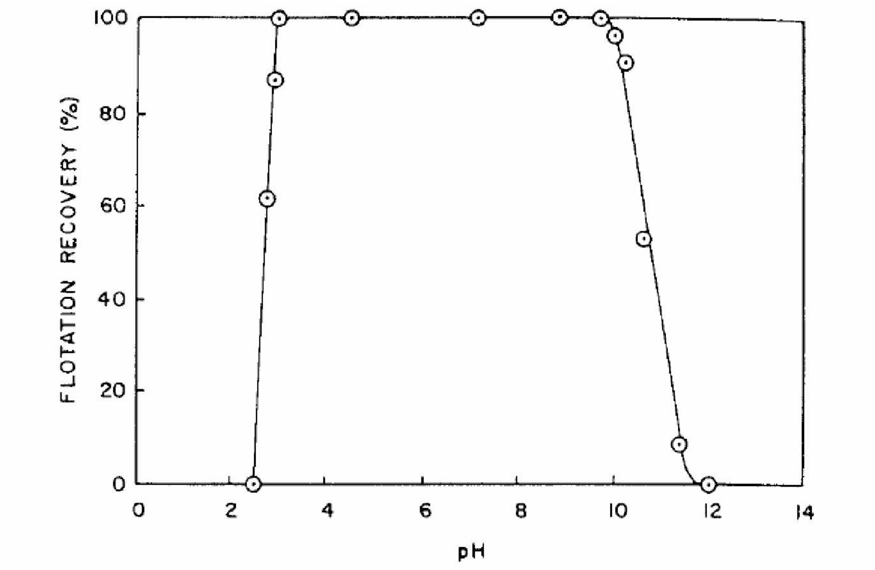


Figure 5. Flotation of galena as a function of pH in the presence of 10^{-4}M ethyl-xanthate (Fuerstenau, 1982).

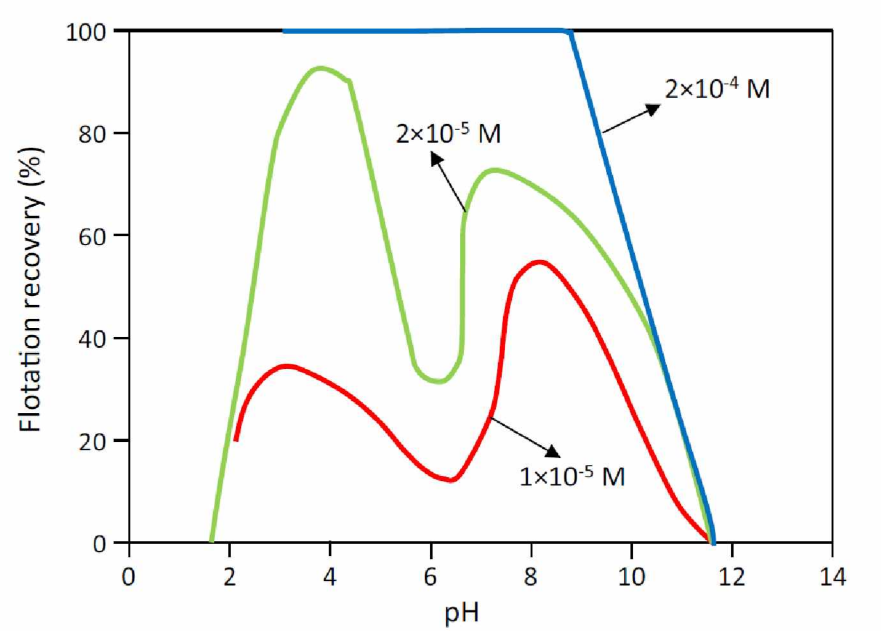


Figure 6. Flotation recovery of pyrite as a function of pH with various dosages of potassium ethyl xanthate (Fuerstenau et al., 1968).

As it can be seen in Figure 6, flotation recovery of pyrite with PEX =< $2*10^{-5}$ M presents a maximum at around pH 4 because of the formation of sulfur-rich products and dixanthogen, and depression around pH 7 due to the formation of iron hydroxide-xanthate complexes on pyrite surface (Fornasiero & Ralston, 1992; Leppinen, 1990; López Valdivieso et al., 2005; Wang & Forssberg, 1991). The pyrite depression With a rise in xanthate concentration, the high ferric hydroxide density on pyrite at pH 7 decreases because ferric hydroxides are converted to ferrous species as a result of xanthate being oxidized to dixanthogen (López Valdivieso et al., 2005). Pyrite is depressed at pH levels below 2 due to the breakdown of xanthate, whereas at pH levels over 11, dixanthogen is thermodynamically unstable and ferric hydroxide islands form on the surface of the pyrite (Leppinen, 1990).

Pyrite flotation or depression is dependent on ionic species in pulp adsorbing or precipitating onto the pyrite surface as well as the quantity of hydrophobic and hydrophilic species present on the surface (He et al., 2005; López Valdivieso et al., 2005). Numerous ionic species can alter the surface characteristics of pyrite and alter its flotation behavior in the flotation system of the sulfide ore complex. Pyrite flotation is improved when copper or lead species interact with xanthate collectors as they adsorb or precipitate on the surface of the mineral (Chandra et al., 2012; Leppinen, 1990) .

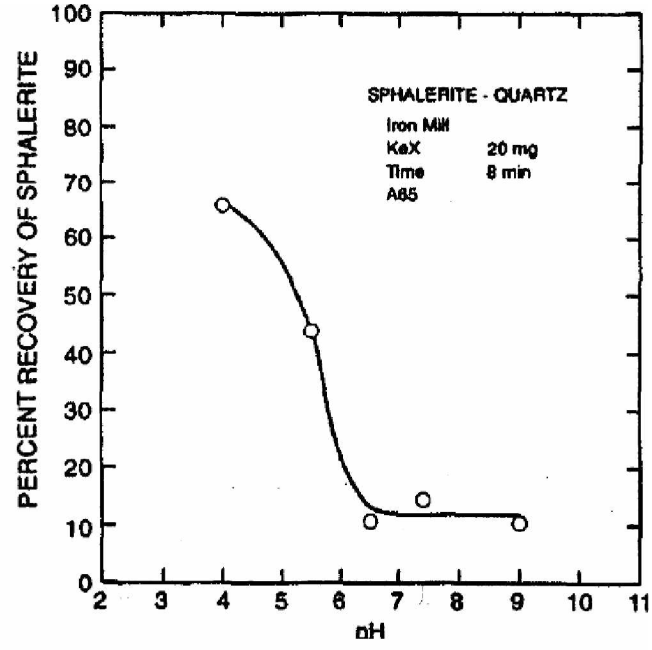


Figure 7. Flotation of sphalerite as a function of pH with 1.3x10^{-4}M ethyl-xanthate (Trahar et al., 1997).

2.2.5. Rheology

The study of rheology examines the connection between a fluid's flow behavior under stress and the subsequent deformation. Particle-particle interactions are also reflected in the rheological behavior of disperse systems, in addition to the properties of the liquid medium in which the solid particles are dispersed and the suspended solid particles. Rheological measurements are frequently used to investigate these interactions. Particle size, size distribution, shape, solids content, pulp chemistry, and surface characteristics of the particles are all important factors that affect the rheology of pulp. A little change in any of these slurry qualities could have a major impact on a process performance. In general, suspensions can either exhibit Newtonian or non-Newtonian behavior. Newton assumed that all materials have a viscosity that is independent of the shear rate at a given temperature. Then, viscosity can be defined by:

$$\eta = \frac{\tau}{\gamma} = \frac{shearstres\,s}{shearrate} \tag{7}$$

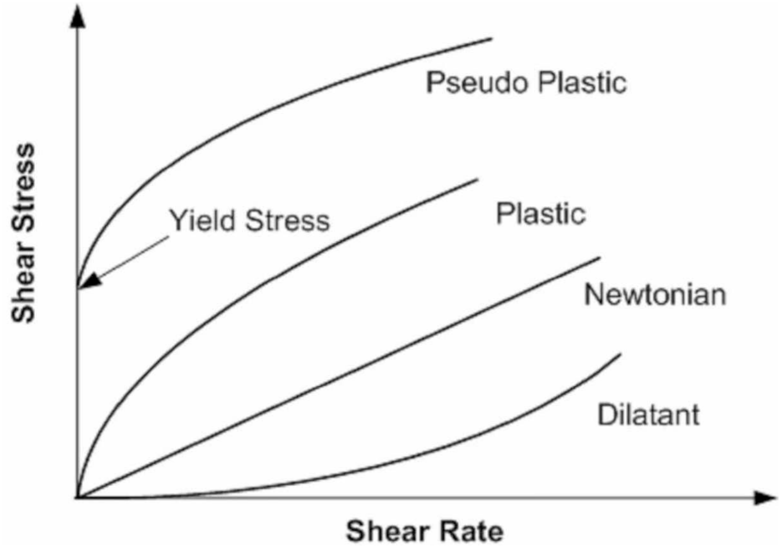


Figure 8. Rheological curves for different rheological systems.

Ideal liquid-like behavior is explained as Newtonian behavior as the applied shearing stress is directly proportioned to the rate of shear, consequently, the viscosity of fluid remains constant as the shear rate is varied. Non-Newtonian behavior is including dilatant, plastic, pseudo-plastic and Bingham behaviors (Figure 8). In non-Newtonian systems, shear stress is not linearly proportional to shear rate anymore, the viscosity for this system is named “apparent viscosity”. In most cases, the viscosity decreases with increasing shear rate and this is called “shear thinning” behavior(Merve Genc et al., 2012; Shabalala et al., 2011).

Studies have demonstrated that the rheological behavior of pulps can significantly influence various processing phases. Changes in rheology can affect the flotation process, particularly the hydrodynamics inside flotation cells, and the transportation of pulps. The rheological behavior demonstrates the degree of inter-particle interaction or aggregation, including gas dispersion and particle-bubble attachment, as well as the mobility of particles and mineralized bubbles in the pulp phase. This behavior can suggest a direct approach to further understanding of inter-particle interactions. Particle-particle interactions in pulp for oxide minerals, clays, clay minerals, coal, sulfide minerals, and even gangue minerals like talc have also been taken into account (Bakker et al., 2009; Dinçer et al., 2003; Farrokhpay, 2012; Farrokhpay et al., 2010; McFarlane et al., 2005; Morris et al., 2002; Mpofu et al., 2003; Muster & Prestidge, 1995; Nosrati et al., 2011; Rubio-Hernández et al., 2006; Shabalala et al., 2011; Taylor et al., 2003; Zhang & Peng, 2015; Zhou et al., 2012)

According to Mueller et al., pulp rheology is an intricate relationship between the processes and physical characteristics of suspended particles. Because the shear field causes a preferred particle orientation, elongated particles or particles with high aspect ratios exhibit anisotropic behavior (Mueller et al., 2009). A non-spherical particle has a different flow than a spherical one, and it also contributes differently to the viscosity of the suspension. For example, the degree of interaction between non-spherical particles will be larger than that between spherical particles at the same volume percentage (Zhang & Peng, 2015).

According to certain studies, particles with a limited size distribution in suspension exhibit higher yield stress and constant shear viscosity values compared to particles with a larger size range at the same volume percentage (He et al., 2004; Turian et al., 1997). Furthermore, it has been noted that the insertion of fine particles improves the viscosity of silica-based solutions (Olhero & Ferreira, 2004). More stable bubble-particle aggregates led to an increase in the recovery of coarse particles with an increased medium viscosity that was regulated by adding a specific amount of glycerol (Xu et al., 2012). Increasing pulp viscosity in flotation causes a decrease in turbulent energy dissipation, a critical parameter that directly controls the stability of a particle-bubble aggregate and quantifies the mean shear forces acting on the bubble-particle aggregate (Farrokhpay, 2012; O'Connor et al., 1990; Pyke et al., 2003). Because of the high bubble-particle detachment efficiency, flotation of coarse particles is particularly challenging (Pyke et al., 2003).

2.2.5.1. Effects of Solid Percentage on Rheology

Increasing the solid content in the pulp has a drastic influence on its rheological properties. The average distance between neighboring particles will be substantially greater than their size if the solids concentration is minimal. Inter-particle forces are minimal in this scenario. As the volume proportion of solids in the system increases, the particles are packed closer together, it is harder for them to move freely, there are more particle-particle interactions, and the viscosity increases (Johnson et al., 2000).

Low solid content pulp exhibits Newtonian rheological behavior, whereas high solid content pulp exhibits non-Newtonian rheological behavior. As seen in cases of extremely concentrated fine particle suspensions, non-Newtonian behavior is substantially influenced by particle size. When there are coarse particles in the suspension, the apparent viscosity does not dramatically rise as the solids content increases, and Newtonian behavior may be observed up to a relatively high solids concentration (Krieger, 1972).

For spherical particles, the effects of solids concentration on pulp viscosity have been taken into consideration. The maximum packing fraction and hence the relationship between concentration and viscosity are strongly influenced by particle shape. Clark looked into how irregularly shaped particles affected viscosity and found that rod-shaped particles had a bigger impact than disc-shaped ones (Figure 9) (Howard et al., 1989). The apparent viscosities and yield stresses of such suspensions are enhanced when the surface area to volume ratio of such particles is large(Leong et al., 1996; Megías-Alguacil et al., 2000). As shown in Figure 10, Horie and Pinder discovered that the yield stress increases with particle length-to-diameter (L/D) ratio at any concentration. It was discovered that the yield stress increases with particle concentration (Pinder, 1979). With a decrease in particle size, it was found that both plastic viscosity and yield stress dramatically rose(Xu et al., 2011). Clearly, the slurry's rheological behavior is substantially influenced by particle size and distribution.

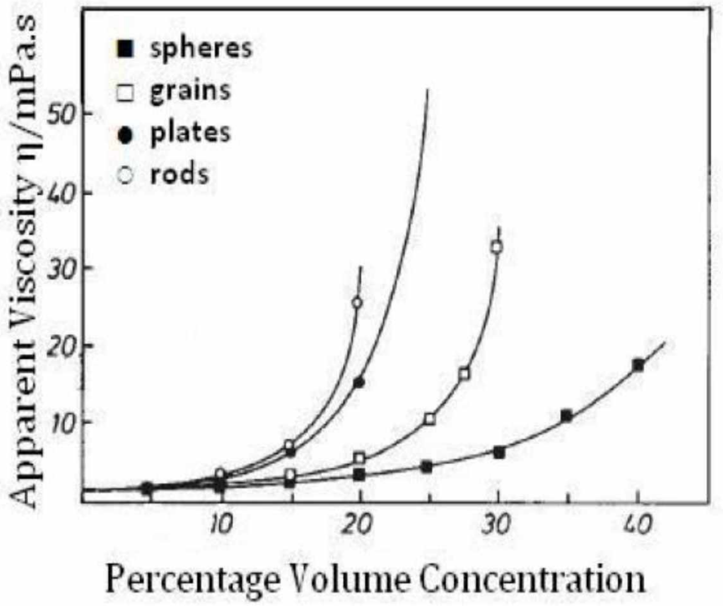


Figure 9. The effects of differently shaped particles on apparent viscosity at various volume concentrations (at a shear rate of 300 s^{1-})" (Howard et al., 1989).

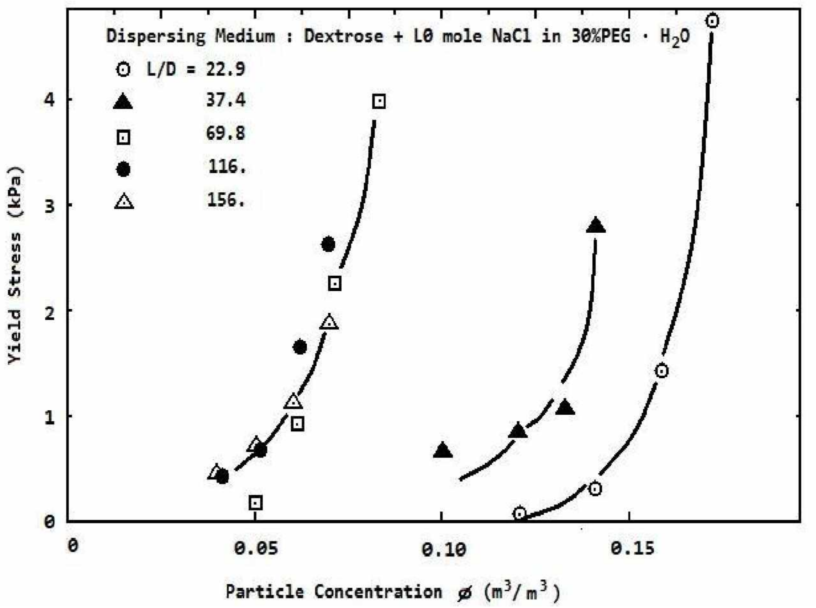


Figure 10. Effects of particle concentration and particle L/D ratio on yield stress(Pinder, 1979).

Any form of particle aggregation in a highly diluted dispersing system causes a decrease in stability, which is followed by particle sedimentation. In contrast, the aggregation in a suspension with a high load may stop the particles from settling. The network of interacting particles formed by the aggregation dramatically raises the viscosity of the fluid (Xu et al., 2011).

2.2.5.2. Effect of Froth Rheology on Flotation

It is well recognized that froth qualities play a significant role in determining flotation performance. Since it can affect both froth mobility and froth stability, two crucial froth qualities in flotation, it is thought that rheology is a significant froth attribute. Froth mobility encompasses both the horizontal motion toward the concentrate overflow weir and the vertical motion of the froth from the pulp and froth interface toward the top surface. Bubble bursting, particle separation, and the drainage of water and solids from the froth are all activities that take place within the froth as it is flowing and are all explained by the concept of froth stability.

The stability of bubble–particle aggregates has been described to be higher when the viscosity of the suspending medium is higher (Li et al., 2018; Vergouw et al., 1998b). The relationship between the pulp viscosity and froth stability (maximum froth height attained in a column) of zinc ore pulp at different pH values is summarized in Table 3 (Farrokhpay & Zanin, 2012). As it can be observed from Table 3 froth stability increases by increasing viscosity at lower pH.

Table 3. Effect of pH on the viscosity and froth stability of a zinc ore slurry(Farrokhpay & Zanin, 2012).

pH	**Viscosity (cP)**	**Max froth height (McFarlane et al.)**
11	1.0	400
9	1.2	450
7	1.5	550
6	2.0	750
4	2.5	900
3	3.0	N/A

It is obvious that changes in froth rheology are caused by changes in air flow rate and froth height. But it is still unclear why the froth rheology changed (Farrokhpay & Zanin, 2012). It is still unknown, nevertheless, whether froth rheology is a separate parameter or only a by-product of pulp rheology (Patra et al., 2012).

The froth height increases with increasing metal ion concentrations in the pulp, particularly for aluminum or calcium ions. In the presence of metal ions, the ore particles aggregates, with higher viscosity is observed at higher metal ion concentrations. This suggests that the presence of metal ions favors particle aggregation probably due to the neutralization of negatively charged species (of the ore surface) and reducing electrostatic stabilization. Vergouw et al. described that particle interaction depends on the charge developed on a mineral surface in aqueous media (Vergouw et al., 1998a). However, the presence of metal ions may cause particle aggregation by reducing electrostatic repulsion between those particles through ionic exchange or precipitation (Kurniawan et al., 2011).

CHAPTER 3 METHODOLOGY

3.1. Sample Collecting and Preparation

The samples were collected from the classifying cyclone overflow stream in mineral processing plant of the Red Dog Mine. 20 of wet 5-gallon sample buckets are needed to perform tests. They were de-canned and dried at Mineral Industry Research Lab, UAF. In the next step, they were mixed to create a homogeneous test feed. The samples were split, using a riffle and were stored in a cold room to minimize oxidation.

3.2. Mineralogical Studies

Characterization of samples were perform using X-ray powder diffraction (XRD) and mineral liberation analyzer (MLA) for original samples. Representative samples were sieved and divided in 7 fraction sizes including -38, +38-45, +45-53, +53-90, +90-150, +150-212 and +212. Then, from each fraction sizes, representative samples were prepared and the degree of freedom and grade of valuable minerals were studied. Grade and particle sizes distribution (PSD) were determined. After plotting the PSD diagram, P80 (hole of sieve that 80% of materials passes it) were extracted. Based on documented PSD and P80 of Red Dog mineral processing plant and previous studies, it weas decided martials require more grinding or not before froth flotation tests. In the next step, it was considered which size ranges are appropriate for flotation of galena and sphalerite as size ranges vary with different minerals and reagent conditions (Trahar, 1981).

3.3. Froth Flotation Studies

Flotation tests was conducted utilizing both conventional lab/bench scale mechanical cells and a customized column flotation unit (Figure 11) at the Mineral Industry Research Lab, UAF. *Figure 12* shows the various stages of the test program in a simplified flow-sheet.

Figure 11. Eriez column flotation unit and Denver cell setup (UAF)

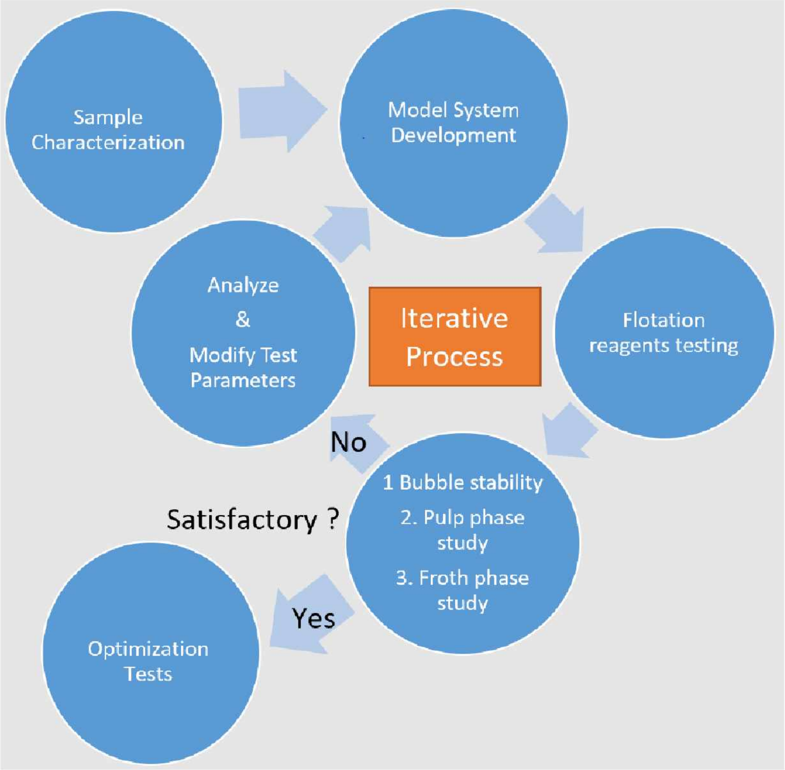


Figure 12. Flowsheet of methodology and test program

3.3.1. Rate and RPM Test

Rate tests were performed in a mechanical Denver cell to investigate floatability of the samples. The dosage of chemicals was purposefully kept low to determine the floatability characteristics flotation kinetics of lead and zinc at different solid% of pulp (30, 40 and 50%). A rate test is where a number of concentrates are collected over various time periods in order to generate recovery-time, grade-time, and mass-time curves. The collected concentrates for each interval time were weighted and analyzed by a commercial lab. The purpose of this procedure is to generate information to determine the metallurgical variability of samples, ore types and ore mixes from the deposit. It also helps to ascertain what metallurgical changes result in an improvement of recovery/grade such as changes in the grinding, reagent type and/or addition, pH, pulp density.

Furthermore, it generates data to determine flotation kinetics as a means of optimizing flotation plant performance.

Agitation was kept constant at 900 RPM for all tests. The pulp level was kept 10-15mm below the froth overflow lip. The froth paddles were sized to extend 10mm below the overflow lip. When air was turned on, pulp level rised and it should rise to 10-15mm below the froth overflow lip. Froth height was kept level with the overflow lip. Froth was collected from top of the cell with interval 30, 60, 90, 120, 210, 330, 510 and 750 seconds. Floated minerals for each interval time were weighted and analyzed by a commercial lab.

The next set of tests examined the effect of varying the impeller speed (RPM) of the mechanical Denver cell in order to investigate the effect of turbulence in the pulp with various solid contents. The chemical dosages were unchanged. Tests were done for different solid% for 800, 900, 1200, 1500 and 1800 rpm to select an appropriate rpm for flotation with high solid %.

3.3.2. Locked Cycle Tests

Locked cycle tests were designed based on a simplified schematic of the Red Dog flotation circuit as you can see in *Figure 13*. First of all, the perforation was done to remove organic material. Then sinking materials was reported to rougher, scavenger and cleaner cells to float galena and followed by sphalerite at separate circuits. After each step of flotation, 5 grams of samples from concentrate and tailing were taken and analyzed to take the best decision to improve the grade and recovery of flotation. Moreover, it was decided products or tailing need more grinding or not as a feed for the next steps. In addition, type and amount of used chemicals was be discussed and selected.

For pre flotation, rougher and scavenger steps, mechanical cell and for cleaner steps, column cells were used (*Figure 11*). Aeration rate, cavitation pressure residence time and volume measurements were conducted for the column unit and impeller speed (RPM) for mechanical flotation cell. Operating parameters to be investigated for both flotation cells that are including feed flowrate, froth depth, bubble stability, etc., and several process parameters such as solid percent, the dosage of reagents, particle size, and residence time, etc. Samples were weighed wet and dry to determine the percentage of solid and analyzed for ore content. Furthermore, statistically designed tests based on the number of variables were conducted.

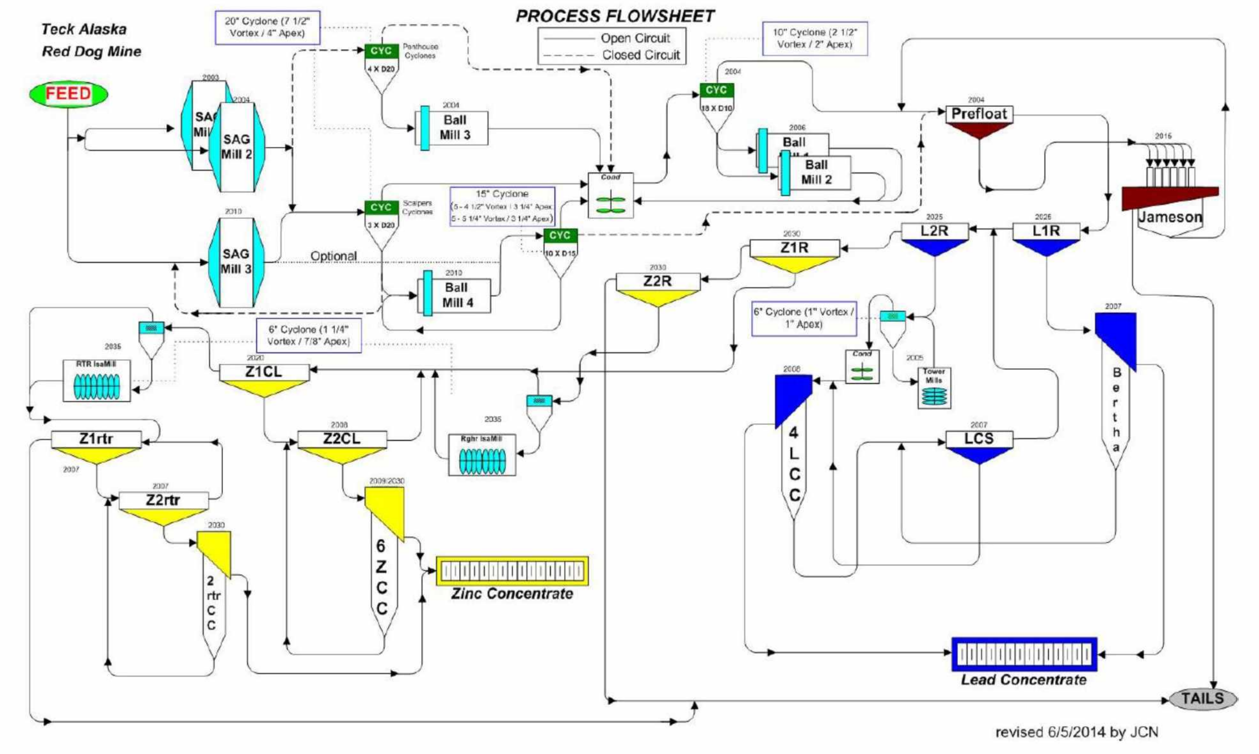


Figure 13. simplified schematic of Red Dog flotation circuit

Additional parameters to be measured on a selective basis are listed as follows (not all proposed measurements can be undertaken during all tests):

1. Bubble size measurements using a standard UAF bubble size analyzer (UAFBSA)

The efficiency of particle-bubble contact and attachment is significantly influenced by the bubble size within the pulp phase of a flotation cell. In general, smaller particles have more tendency to attach to smaller and larger bubbles are more suited for larger particles. In addition, smaller bubbles improve the flotation kinetics so the characterization of bubble size and performance in the pulp would be an important factor. The most common technique developed to measure bubble size is the McGill bubble size analyzer (Figure 14), due to its simplicity and robustness (Biswal et al., 1994; Diaz-Penafiel & Dobby, 1994) .

The McGill bubble size analyzer has a sampling tube that is attached to a viewing chamber with a window inclined at 15° from vertical. The bottom of the sampling tube should be put just below the pulp/froth interface in the quiescent zone. The closed tube is filled with water of similar nature to that in the flotation cell. Bubbles rise up through the sampling tube and slide up the inclined window of the viewing chamber. Their images are taken by a digital video camera as they slide up the inclined window illuminated from behind. The light source illuminates the image area, resulting in each bubble casting a shadow, which can be detected by the software associated with the bubble viewer (Chen et al., 2001; Hernandez-Aguilar et al., 2004). Froth vision would be recorded, and where possible, used to estimate froth stability by counting the number of bubbles bursting on the surface as a function of time.

The McGill bubble viewer produce a bubble size distribution, the average bubble diameter and the Sauter mean bubble diameter for each measurement. The average bubble diameter is calculated from Equation 8 and the Sauter mean bubble diameter is determined from Equation 9:

$$d_b = \frac{\sum_{i=1}^{N} d_i}{N} \qquad (8)$$

$$d_{32} = \frac{\sum_{i=1} d_i^3}{\sum_{i=1} d_i^2} \tag{9}$$

In these equations *di* is the measured bubble diameter and *N* is the number of bubbles measured(Harbort & Schwarz, 2010). Sauter mean bubble sizes range for mechanical cell is between 1.0 mm to 2.0 mm.

UAF bubble size analyzer was designed based on the McGill Bubble size analyzer. Instead of using McGill bubble viewer. All pictures were processed using image processing technique, Thus, bubble sizes as well as their mean were gained.

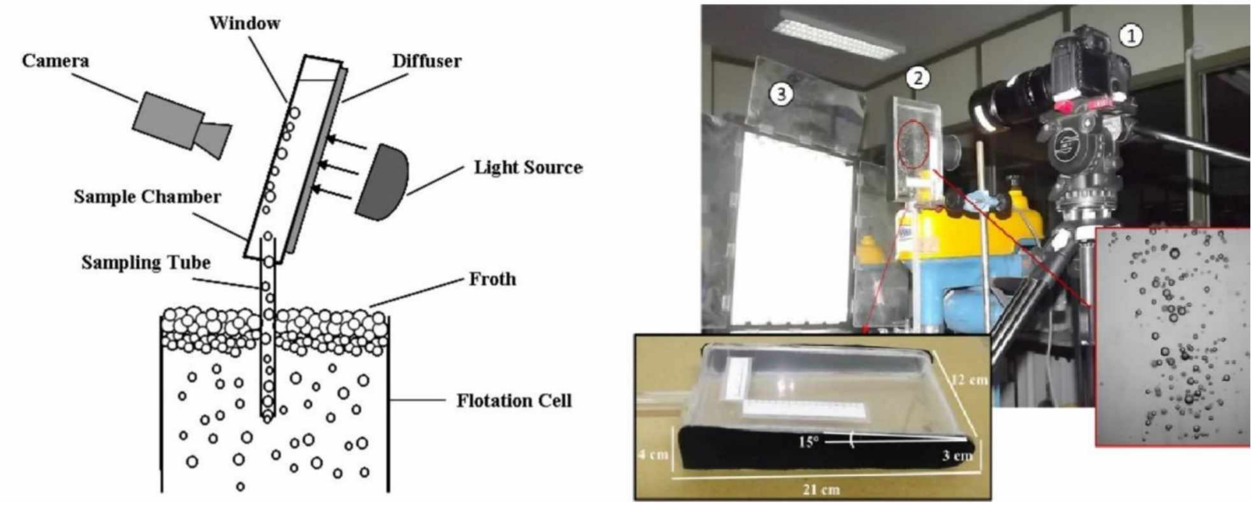


Figure 14. Schematic of McGill bubble size analyzer and typical set-up. 1) digital camera, 2) bubble viewer, 3) light source (Girgin et al., 2006; Mazahernasab & Ahmadi, 2016)

2. Characterization of suspension using in-pulp sampler

Suspension can be characterized by collecting samples of the pulp at different heights using an in-pulp sampler. This step will be done to investigate the changes of pulp density in different heights of pulp (Zheng et al., 2005).

3. Evaluation of metallurgical performance

Metallurgical performance can be evaluated by conducting a recovery vs froth depth study. Typically, overall recovery decreases with froth depth, but a few studies have indicated occasional higher ore recovery at high densities and at very deep froth depths (Runge et al., 2012). The behavior of Red Dog ore under such conditions needs to be investigated.

4. Investigation of the pulp phase performance

Pulp phase performance can be calculated using a changing froth depth analysis technique. This technique involves calculating the overall flotation rate of the ore at each froth depth using the tailing flowrate, cell volume and concentrate recovery values (Equation 10). Investigating the effect of % solids on pulp zone kinetics and the flotation rate would help in understanding potential flotation rate decrease with increase in pulp density(Vera et al., 1999).

$$R = \frac{\mathrm{k}\tau}{1+\mathrm{k}\tau} \qquad \tau = \frac{\mathrm{Vcell}}{\mathrm{Qpulp}} \qquad (10)$$

Where, k is the flotation rate, R is flotation recovery, τ is cell residence time, V_{cell} is the cell volume and Q_{pulp} represents the volumetric flowrate.

5. Optimization of chemical reagents

Four main reagents were used in flotation circuits including promoter (collector), frother, activator, and depressant. In the lead circuit of the Red Dog mineral processing plant, xanthates are used as a collector, methyl isobutyl carbinol as a frother, sodium cyanide as a pyrite depressant, and zinc sulfate as a sphalerite depressant. In zinc circuits, xanthates are used as a collector, methyl isobutyl carbinol as a frother, copper sulfate as a sphalerite activator. In the Red Dog mineral processing plant, they are used in a certain dosage. A series of batch experiments were designed by Minitab

(3-level Box Behnken method, and Taguchi method) in 3 different levels of chemical dosages and different types of chemicals as reagents. Typical reagents used in lead and zinc flotation are presented in Table 4. Some of the common reagents in Red Dog was replaced by new reagents and their effects was considered. After performing all the designed tests, products were analyzed. In the next step, using the results of experiments and Minitab, the best condition to get the highest grade and recovery were estimated.

6. Control pH

PH should be controlled in all tests. It will be measured by pH meter. The potential difference of a mineral-solution interface, which is indicated by pulp potential, has been shown to be closely related to the floatability of sulfide minerals (Rand & Woods, 1984).

Table 4. Commonly Used Reagents in lead and zinc Flotation(EPA, 1994).

Type of Reagent	Lead Flotation	Zinc Flotation
Collector	Aeroflotas Aerophine MinreC A Xanthates Aero88	Xanthates Aeroflotas Sodium Aeroflotas
Frother	Dow Froth Methyl Isobutyl Carbinol Pine Oil	Dow Froth Methyl Isobutyl Carbinol Pine Oil Flotal B
Activator	Alkaline Sulfides Phosphate	Copper Sulfate
Depressant	Cyanide (Sodium or Calcium) Sodium Bisulfate Dichromate Zinc Hydrosulfate Zinc Sulfate	Sulfur Dioxide Sodium Hydrogen Sulfide Zinc Sulfate Sodium Cyanide

3.4. Data Analysis

All obtained data from experiments was analyzed. Relationships between data was discovered. Optimum condition was achieved using Minitab software.

CHAPTER 4 RESULTS AND DISCUSSION: INITIAL STUDIES

4.1. Sample Preparation

The MIRL (UAF) research team visited the Red Dog Mine Plant during July 30 to August 2, 2018. The first batch of samples, collected from the classifying cyclone overflow stream, arrived at UAF facility in October, 2018. The wet 5-gallon sample buckets, upon receipt, were sorted, de-canned and dried for 12 days, then they were mixed to create a homogeneous test feed. The samples were then split, using a riffle, into 160 storage bags and stored in a cold room to minimize oxidation (Figure *15*).

Figure 15. Preparation of representative samples and storage in cold room

4.2. Sample Characterization

Preliminary characterization included particle size analysis and mineralogical tests performed by a commercial lab. Majority of the sample (almost 90%) were found to be passing 100 mesh (150 micron) size with a d50 of approximately 90 microns, as shown in Figure 16. The average grade of Zinc (Zn) in the sample was 15.35%, that of Lead (Pb) was 4.35% and Iron (Fe) was 4.7%. The barite content was low for the current sample (Table 5). No grinding of the as-received sample has been performed during the initial phase of characterization tests. However, grinding of the rougher feed was necessary to match Red Dog's operational PSD in future experiments.

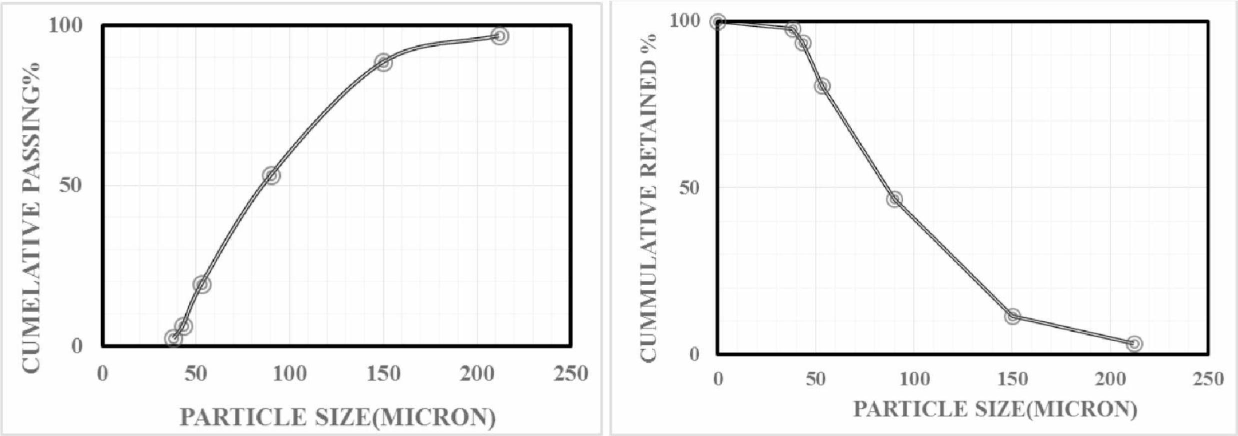


Figure 16. Particle size distribution of the Red Dog samples as received

Table 5. The grade of flotation tests feed

Pb %	**Zn %**	**Fe %**	**Al %**	**Ca %**	**K %**	**Mg %**	**Na %**	**S %**
4.35	15.35	4.76	0.04	0.25	0.01	0.01	<0.01	>10.0

P80 of received samples (Figure 16) is around 130 microns, while the P80 of Rougher cell feed is around 70-80 microns. To obtain similar P80, a series of grinding tests were done using steel grinding media, and stainless-steel (ceramic) grinding media at ball mill. Grinding tests were conducted by ceramic ball mill for 3 , 5, 10, 12 minutes. Grinding in 5, and 12 minutes were shown in Figure 17. As can see, P80 after 5 minutes grinding is around 110 microns and after 12 minutes grinding is around 70 microns.

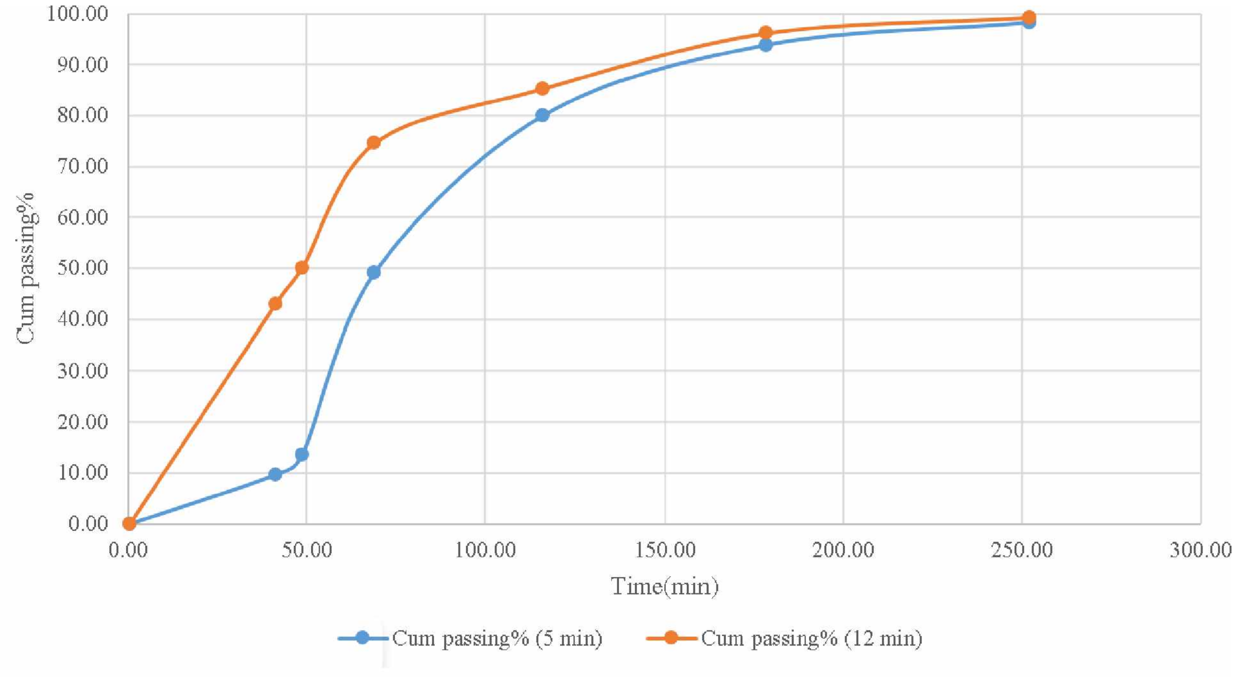


Figure 17. The particle size distribution of the Red Dog samples after 5, and 12-minutes grinding

4.3. Flotation: Rate and RPM Test

A simple exploratory test was performed in a 3-liter Denver cell to gauge floatability of the received sample in the MIRL lab. The dosage of chemicals (Table 6) was purposefully kept low to determine the floatability characteristics of lead at 20% solids. As it can be seen from Table 6, upgradation was below expected thresholds and selectivity was non-optimal. Lead recovery was found to be low with significant amounts of sphalerite reporting to the concentrate. Thus, it became evident that selectivity is a function of the reagent dosage and low amounts of depressant were insufficient in depressing sphalerite and pyrite. The chemical dosages were adjusted for subsequent tests. The cell agitator was set at 900 RPM.

Table 6. Chemical dosages (exploratory test 1)

	Chemicals	**Dosage (g/t)**
Galena Flotation	NaCN	130
	ZnSO4	320
	Aero 88	15
	MIBC	5
Sphalerite Flotation	Aero 88	25
	MIBC	5
	CuSO4	292
	Lime	110

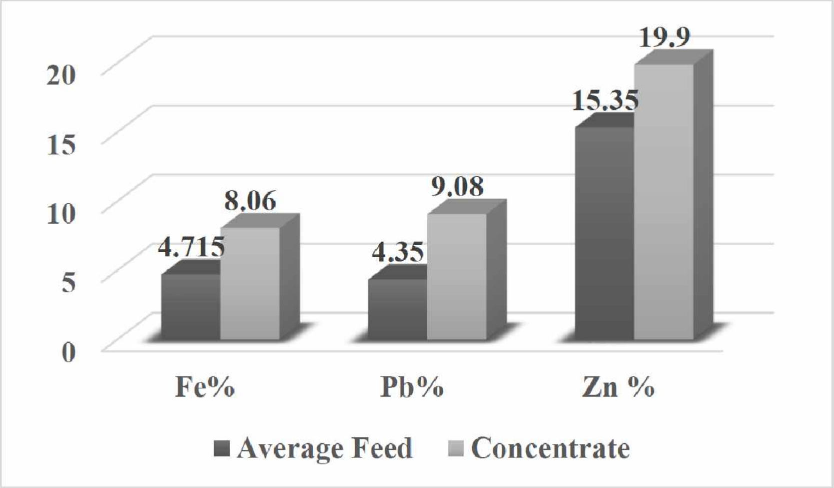


Figure 18. The grades of feed and concentrate of test 1

The next set of experiments were batch rate tests conducted for 3 different slurry densities (20%, 40% and 60%) to estimate ore characteristics and flotation kinetics. A rate test is where a number of concentrates are collected over various time periods in order to generate recovery-time, grade-time and mass-time curves. A batch test is when a single concentrate is collected for the full duration of the test. The purpose of this procedure is to generate information to determine metallurgical variability of samples, ore types and ore mixes from the deposit. It also helps to ascertain what metallurgical changes result in an improvement of recovery/grade such as changes in grind, reagent type and/or addition, pH, pulp density. Furthermore, it generates data to determine flotation kinetics as a means of optimizing flotation plant performance. Cell volume was 3-liters and agitation were kept constant at 900 RPM for all tests. The pulp level was kept 10-15mm below the froth overflow lip as shown in Figure 19.The froth paddles are sized to extend 10mm below the overflow lip. When air is turned on, pulp level rises and it should rise to 10-15mm below the froth overflow lip as represented in Figure 19. The cell was marked and calibrated for pulp level with agitation and no air so that with airflow, pulp level is 10-15mm below the froth overflow lip. Froth height was kept level with the overflow lip. During the test-run the default air setpoint was used and pulp height was about 15mm below the overflow lip. Batch rate tests were conducted using two different collectors (Xanthate and Aero 88) to consider impact of them on flotation rate, grade and recovery. Table 7 details the chemical dosages used in the tests. Froth was collected from top of the cell with interval 30, 60, 90, 120, 210, 330, 510 and 750 seconds (Figure 20). Floated minerals for each interval time were weighted and analyzed by a commercial lab.

The cumulative recovery curve for Lead (Pb) using Xanthate is shown in Figure 21. The comparison between Xanthate and Aero 88 at 40% solids is provided in Figure 22. Interestingly, the maximum recovery and yield values of lead were achieved at 60% solids contrary to expectations (Table 8). Xanthate performed slightly better than Aero 88 with a cumulative recovery of 83.77% as compared to 79.64% at 40% solids respectively.

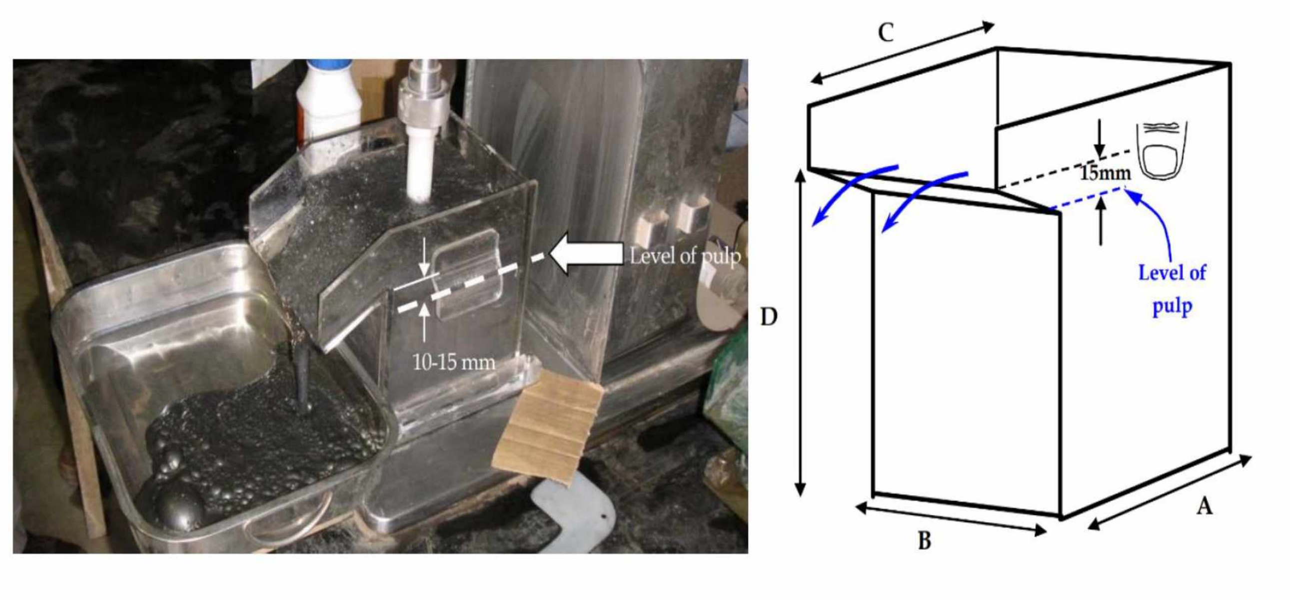


Figure 19. Rule of thumb for pulp and froth level for batch rate test

Table 7. Chemical dosages used in batch rate tests (20%, 40% and 60% solids)

	Chemicals	**Dosage (g/t)**	**Dosage (g) for 20% of solid**	**Dosage (g) for 40% of solid**	**Dosage (g) for 60 % of solid**
Galena flotation	NaCN	130	0.065	0.173	0.390
	ZnSO4	600	0.300	0.800	1.800
	Xanthate	60	0.030	0.080	0.180
	MIBC	21	0.011	0.007	0.015
	Aero 88	60	0.030	0.080	0.180
Sphalerite flotation	Xanthate	160	0.080	0.213	0.480
	MIBC	21	0.011	0.007	0.015
	CuSO4	400	0.200	0.533	1.200
	Aero 88	160	0.080	0.213	0.480
	Lime	110	0.055	0.147	0.330

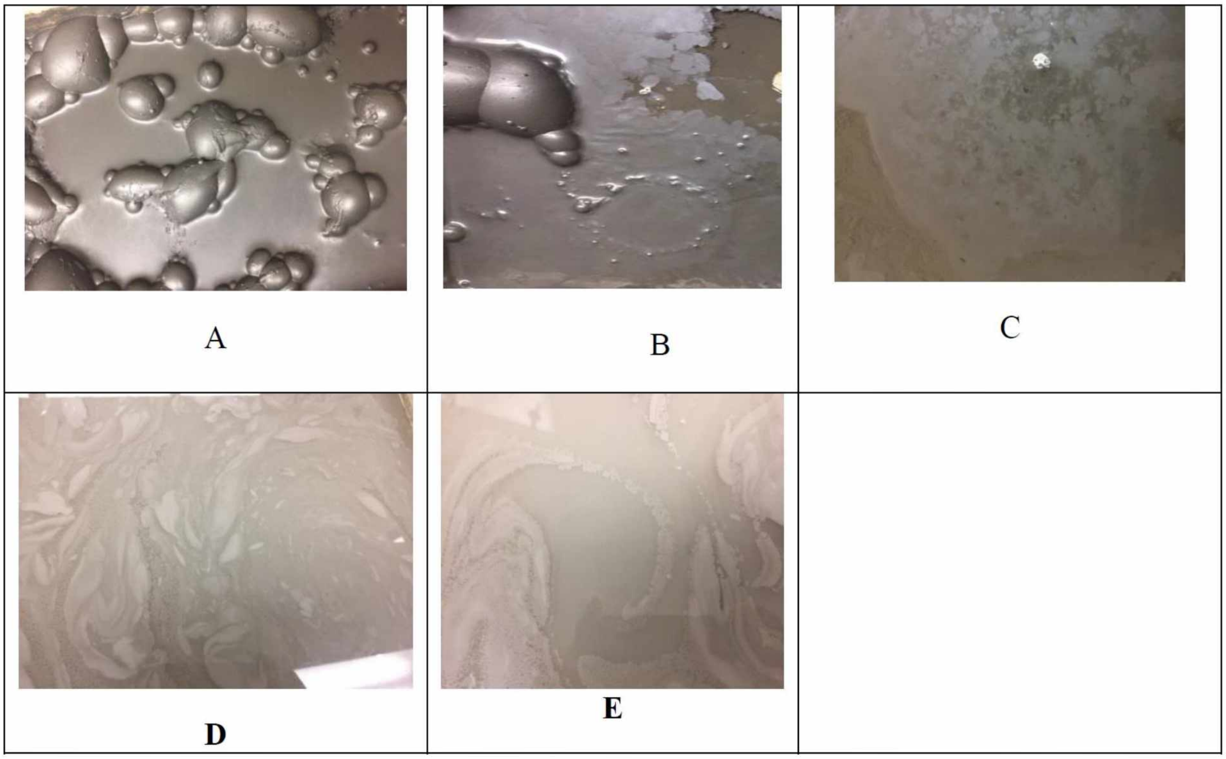


Figure 20. Rate test after removal of froth at A) 30 seconds B) 60 seconds C) 120 seconds D) 210 seconds E) 330 seconds

Table 8. Cumulative recovery of Lead (Pb) at 20%, 40% and 60% solids [Xanthate]

Time (s)	**Cumulative Recovery Lead (Pb) [Xanthate]**		
	20% Solids	40% Solids	60% Solids
30	12.04	17.27	6.09
60	17.48	26.14	10.18
120	28.31	43.59	27.23
210	40.22	55.29	50.78
330	49.53	64.77	65.36
510	63.29	74.36	76.69
750	78.35	83.77	86.20

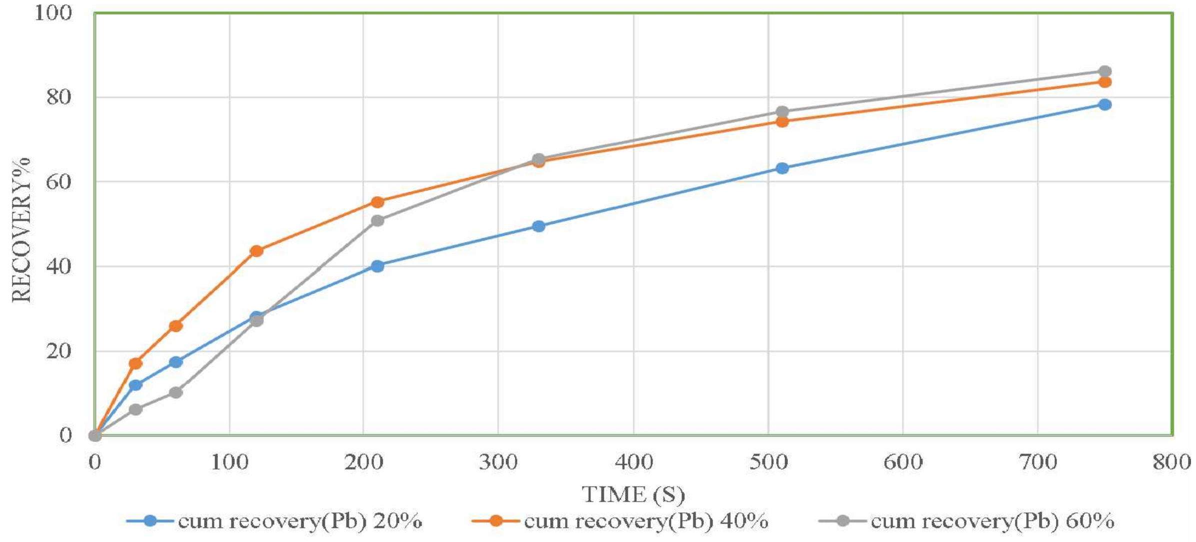


Figure 21. Cumulative recovery Lead (Pb) for 20%, 40% and 60% solids (Xanthate)

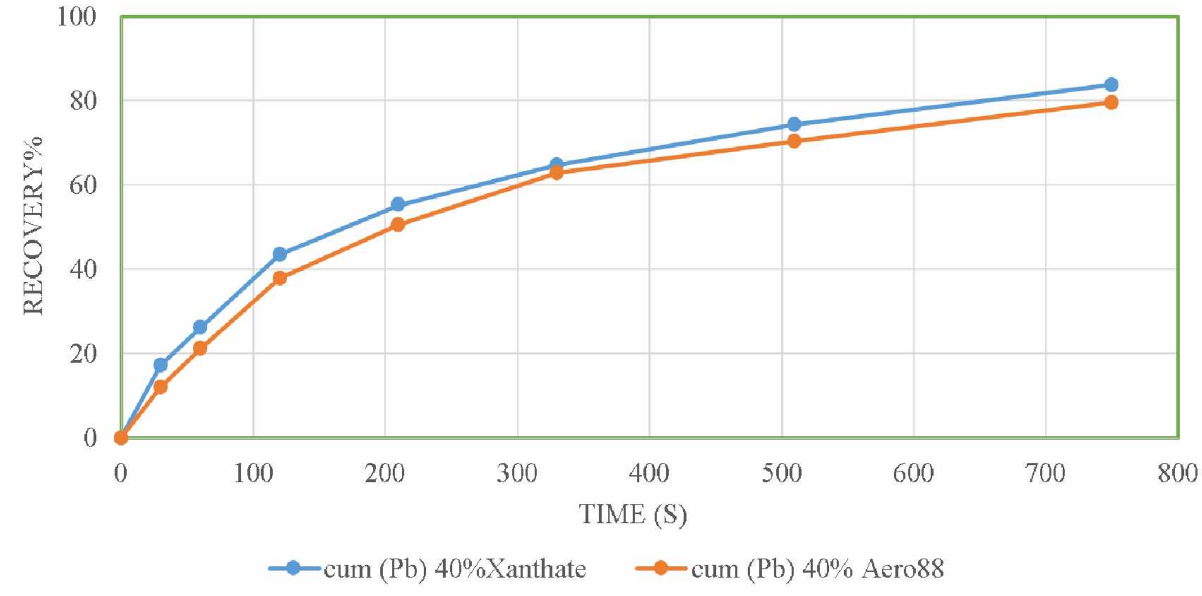


Figure 22. Cumulative recovery Lead (Pb) at 40% solids (Xanthate vs Aero 88)

The preliminary analysis of the batch rate test results demonstrates the significance of residence time as an important factor in improving flotation performance. The overall yield values at 20% 40% and 60% solids for Xanthate are 68%, 69.6% and 73% respectively. Better flotation performance at higher slurry densities indicates optimal bubble loading and froth stability. It is expected that lower slurry densities would lead to better kinetics, however, in case of the initial tests, results indicated that slurry density has minimal effect if residence time is increased. The next set of tests examined the effect of varying the rotor RPM of the 3-liter Denver cell from 800 ~ 1800. The chemical dosages were unchanged as detailed in Table 7, however, in this iteration only Xanthate was used as the primary collector. Slurry density was kept at 20%, 40% and 60% solids respectively. Yield was significantly higher at low densities with 76.1% feed material reporting to the concentrate at 20% solids at 800 RPM. The yield values at 40% and 60% solids were substantially lower at 74.25% and 69.88% respectively at the same RPM. Figure 23 and Figure 24 show the cumulative recovery of Lead (Pb) and Zinc (Zn) for different slurry densities respectively.

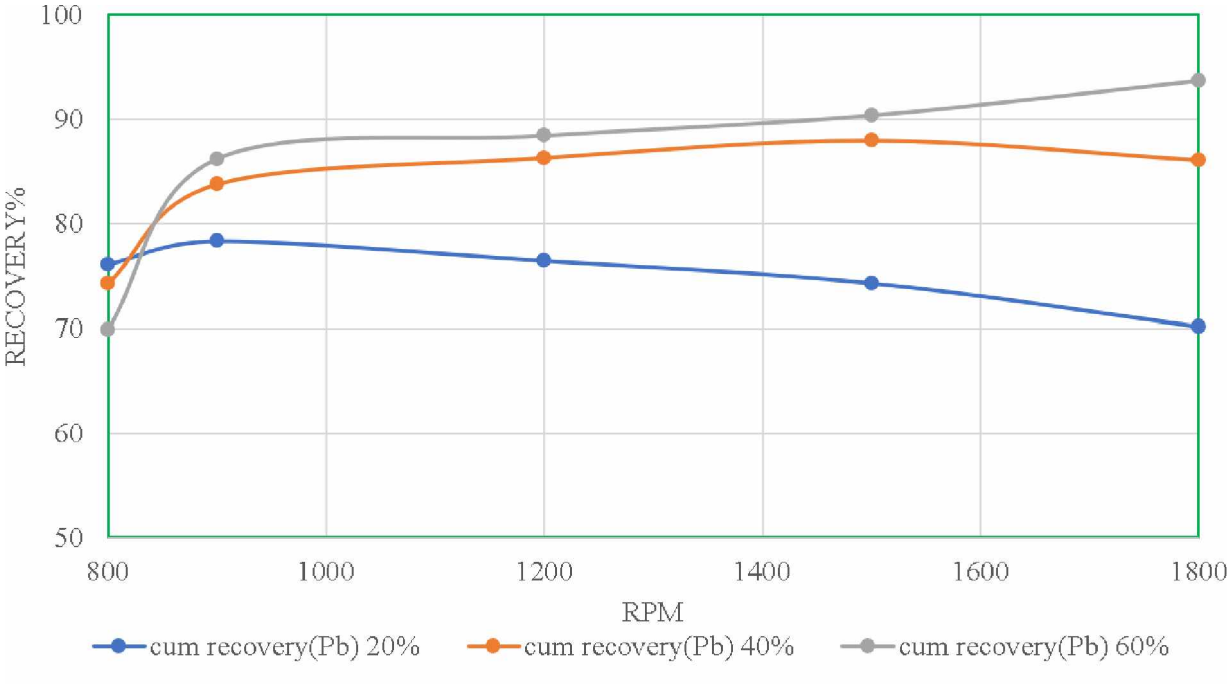


Figure 23. Cumulative recovery Lead (Pb) for 20%, 40% and 60% solids (Xanthate)

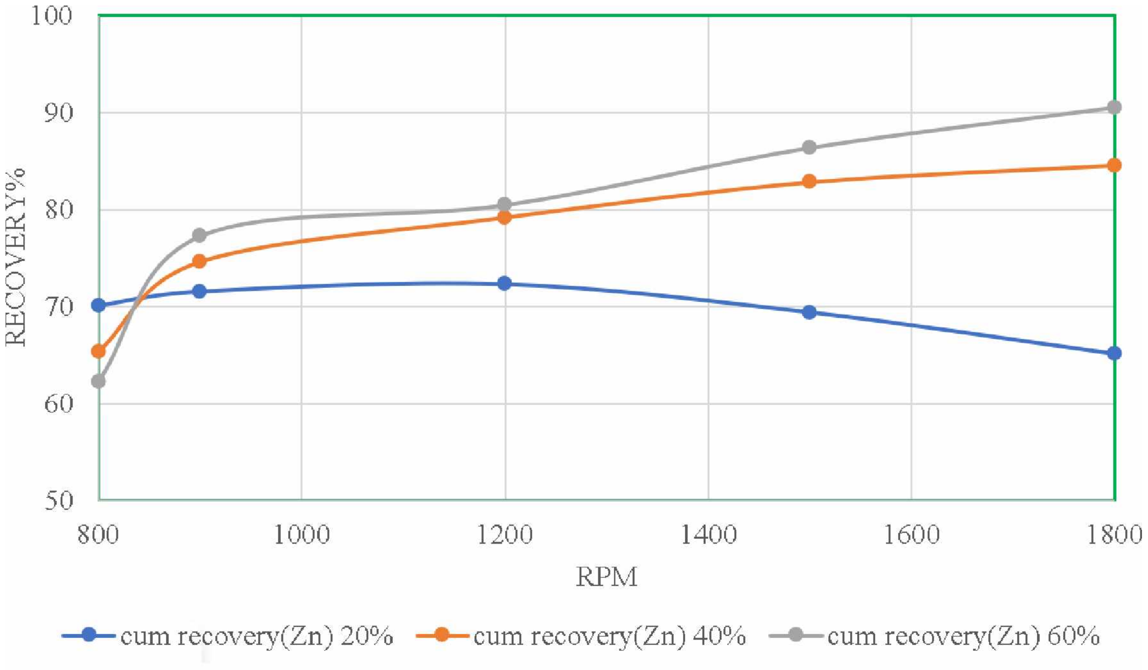


Figure 24. Cumulative recovery Zinc (Zn) for 20%, 40% and 60% solids (Xanthate)

The results conclusively prove that at high slurry densities higher RPM rate leads to better recovery. Lead (Pb) and Zinc (Zn) recovery values at 20% solids and 1800 RPM rotor speed are 70.12% and 65.1% respectively. However, there is significant difference in performance between 40% solids and 60% solids, with 60% solids displaying better floatability throughout the rotor RPM envelope. This phenomenon can be attributed to better bubble loading at higher slurry densities, which leads to better bubble-particle collision and adhesion rate. Froth phase is stable at higher percent solids and material loss from froth phase to pulp phase is reduced. Moreover, there is significant turbulence effect at higher RPM ranges (1500 ~ 1800), whereby, it was observed that the froth phase was not stable and bubble burst was occurring at an alarmingly higher rate. The initial tests have demonstrated the significance of residence time and slurry density, which leads to higher bubble loading rate, as major factors affecting flotation performance. If solids content is significantly reduced then the agitation rate becomes a deciding factor along with residence time.

Lower slurry densities with high agitation rates have shown lower recoveries. The optimum RPM for low slurry densities is 900. The optimum RPM for higher slurry is 1200.

4.4. Effects of Particle Sizes on Flotation

As all above experiments were done for flotation feed with PSD 150 microns. A series of experiments were performed for flotation feed with PSD 70-80 microns. The results show decreasing the particle sizes in all 3 different pulp density causes the improvement of flotation performance. More grinding means more liberation of valuable minerals from gangues. As it can be seen in Figure 25 at lower pulp solid%, by decreasing the particles sizes, recovery increased due to more liberation of galena particles. But by increasing the pulp solid% as well as liberation, the bubbles were overloaded so some of them burst before reach to top of the cell. Thus, it has a negative effect on the recovery at high solid percentage. On the other hand, by increasing the liberation, the concentrate grade improved in all 3 different pulp densities from 9.08%, 10.12%, and 6.85% to 14.32%, 12.96%, and 9.27%, respectively. The pulp with PSD 80 microns has the highest grade and recovery at the lowest pulp density. However, pulp with PSD 150 microns has the highest recovery at pulp with 60% solid, and the highest grade at pulp with 40% solid (Figure 25, and 26).

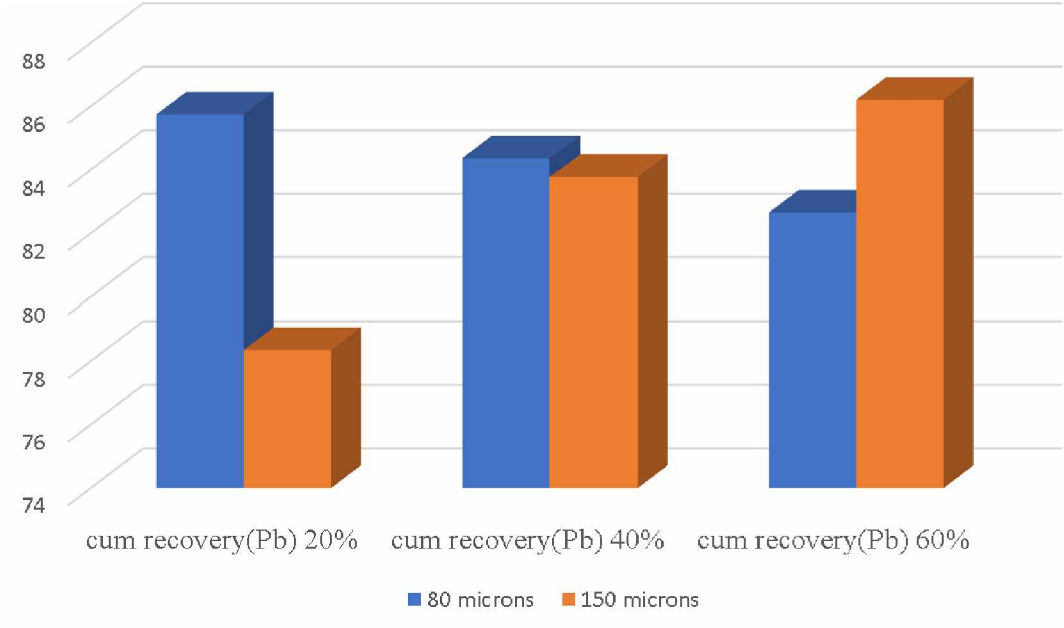


Figure 25. Effect of particles sizes on the flotation recovery

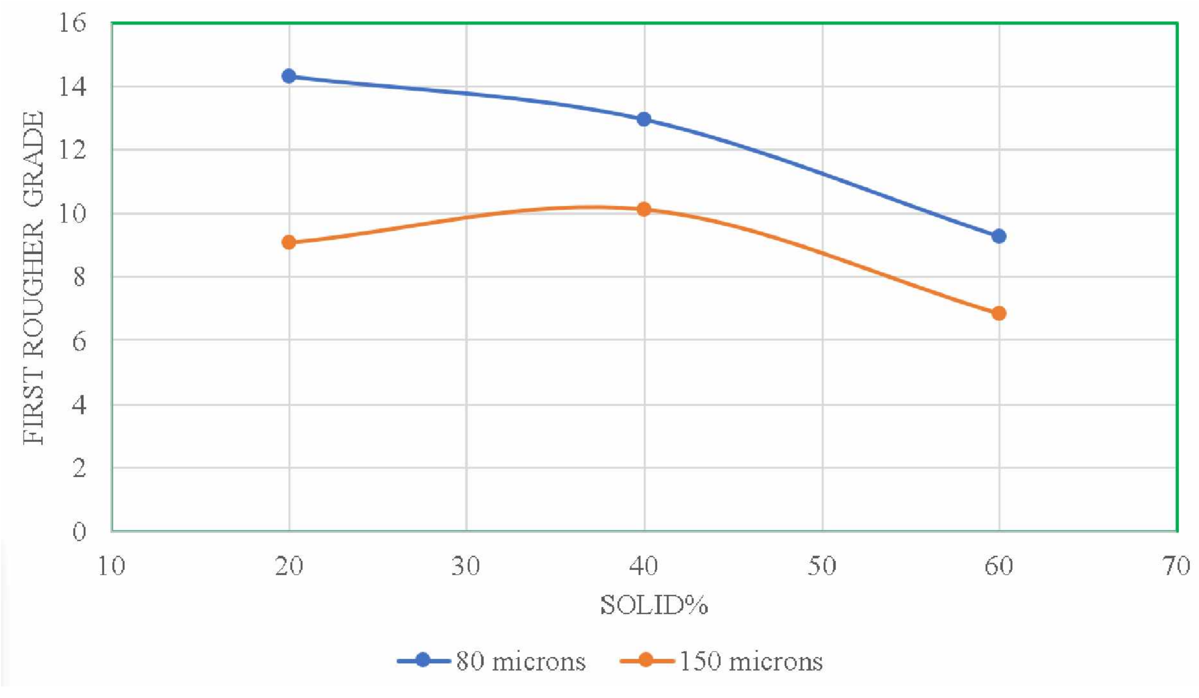


Figure 26: Effect of particles sizes on the flotation grade

CHAPTER 5 RESULTS AND DISCUSSION: EFFECT OF CHEMICALS & OPTIMIZATION

5.1. Effectiveness of Grinding Media on Flotation Performance

The PSD of received samples were 150 microns so it was needed extra grinding to reach the similar PSD (70-80 microns) of pre-flotation feed. To achieve the appropriate PSD, the ball mill was used. Two different type of grinding media was used to reduce particles sizes including steel grinding media, and stainless-steel (ceramic) grinding media. Grinding environment is a place that you can control some parameters that causes the improvement of flotation performance. To get the similar PSD, the grinding time was 3 minutes in ball mill with steel grinding media, and 12 minutes in ball mill with stainless-steel (ceramic) grinding media. The PSD can be seen in the Figure 17 .

Flotation results shows feed was grinded by stainless-steel (ceramic) grinding media has higher grade, and recovery compare to steel grinding media. Using stainless-steel grinding media instead of the steel grinding media increased the rougher concentrate grade by 0.53, 0.27, and 1.23 for pulp with 20, 40, and 60 solid percentage, respectively. In addition, the recovery was improved by 0.48, 0.99, and 2.48 for pulp with 20, 40, and 60 solid percentage, respectively (Figure 27).

The steel grinding media dissolve iron ions. Some of iron species such as hydrolyzed one adsorb on galena particles. In addition, Fe^{2+} ions react with dissolved O_2, consequently the product of this reaction will react with Fe^{2+}. Finally, H_2O_2 will be produced as below(Cohn et al., 2006):

$$Fe^{2+} + O_2 \rightarrow Fe^{3+} + (O_2^0)^- \qquad (11)$$

$$Fe^{2+} + (O_2^0)^- \rightarrow Fe^{3+} + H_2O_2 \qquad (12)$$

The grinding media causes the formation of H_2O_2 at wet grinding environment. Steel grinding media because of dissolving higher amount of Fe^{2+} ions lead to produce higher amount of H_2O_2.

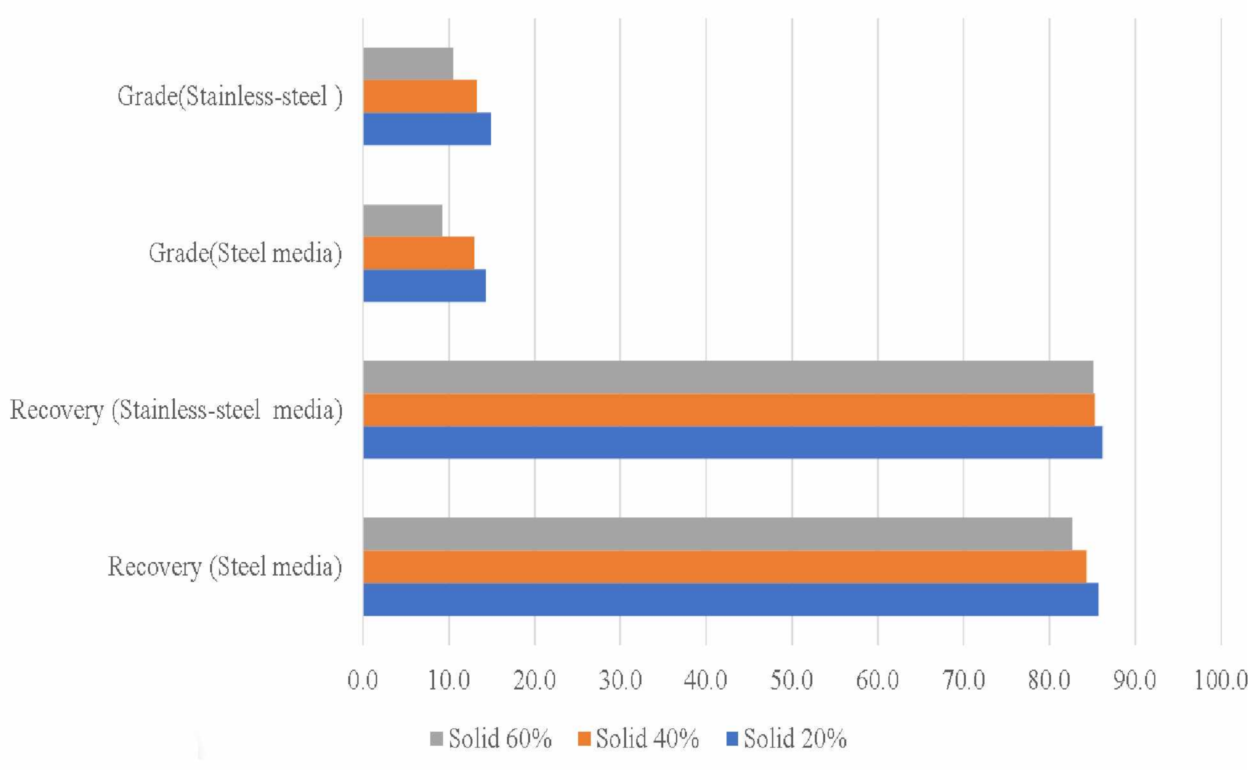


Figure 27. Effect of grinding media on the grade and recovery at different pulp density

H_2O_2 oxidizes the galena surface. Some studies demonstrated some oxidized species such as sulfoxy and hydroxyl on galena surface. oxidation H_2O_2 leads to depression of galena partially, but if its amount exceeds 10^{-3} M it will completely depress galena(Ikumapayi et al., 2012; Javadi, 2013). In addition, the present of H_2O_2 increase the oxidation and decomposition of lead xanthate(Hu et al., 2020).

$$[Pb(EX)_2]_{ads} + H_2O_2 \rightarrow Pb(OH)_2 + (EX)_2 \qquad (12)$$

By increasing solid%, amount of Fe^{2+} will increase, consequently, more H_2O_2 will be produced. Thus, at higher pulp solid%, using steel grinding media has higher negative effects on galena flotation. It will decrease both grade and recovery. In order to improve flotation performance with higher solid percentage in the mining and mineral processing, the replacement of stainless-steel grinding media could be a suggestion after considering all economic aspects.

5.2. Collectorless Flotation

The purpose of this step is to control prefoliation step. In the preflotation step, organic material, and sulfurs would be removed from the flotation feed. It would be performed only by adding frother. It is vital to know what is the optimum time to do preflotation with lowest probability of losing valuable minerals, and with the high probability of removing organic materials. In addition, this step can help to better understanding of sulfide ores floatability in the collectorless flotation.

Flotation tests were performed for 10 minutes. They were conducted using MIBC (5g/t) as frother at pH=7. Recovery was around 51%, and 43% for Pb, and Zn. According to Figure 28 .the results show preflotation in these experiments should be performed under 1 minute to minimize losing valuable, and maximize organic material removal. The losing of galena and sphalerite also is related to the amount of total organic carbon (TOC) in the feed. By increasing TOC, losing galena and sphalerite will increase. In the experiments, we almost lose 2-5% of the sphalerite during preflotation.

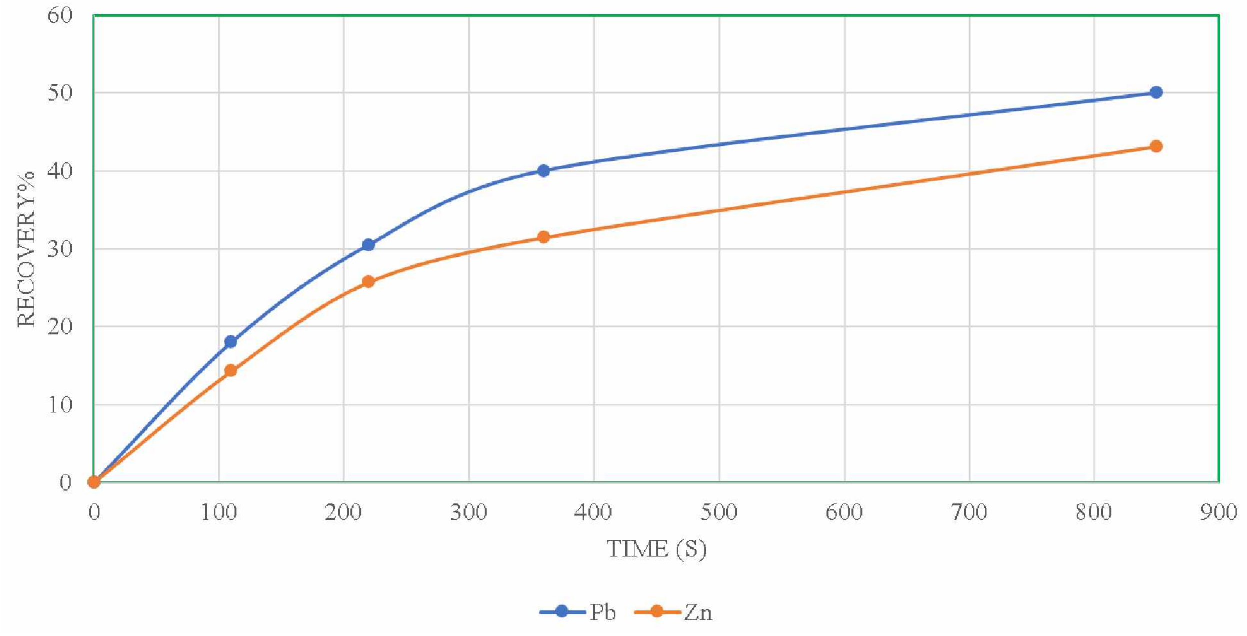


Figure 28. Collectorless flotation of Galena and Sphalerite

The galena collectorless flotation is depend on the relative amount of hydrophobic species (such as elemental sulfur) and hydrophilic species (such as lead hydroxide). The amount of $[S^0]/[OH^-]$ can describe the collectorless flotation behavior. Greater $[S^0]$ means greater hydrophobicity as well as better floatability at a certain pH. The amount of $[S^0]/[OH^-]$ is determined by pH, and Eh. At acidic pH due to the formation of elemental sulfur, the collectorless flotation recovery is greater than Alkaline conditions (Figure 29)(Hu et al., 2020; Luttrell & Yoon, 1984). Thus, the grade and recovery of products can be controlled by controlling pH-Eh during the flotation. To loss less galena and sphalerite, it is better flotation to be done at higher pH and Eh. In addition, a cleaning step after preflotation step is needed for floated material to return part of the galena and sphalerite to the circuit.

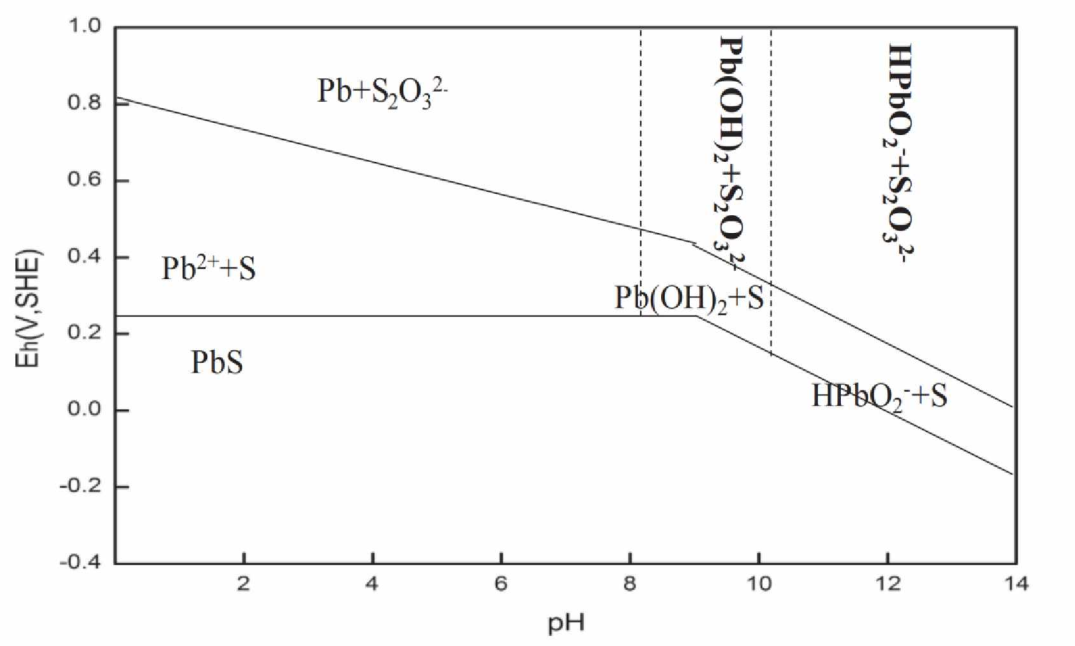


Figure 29. Eh–pH diagram for the PbS–H2O system in aqueous solution for dissolved species at 10^{-4}M

5.3. Optimization Tests for Galena Flotation

In order to optimatze the galena flotation, a series of experiment were designed according to the Table 9 using Minitab (3-level Box Behnken method) in 3 different levels of solid%, chemical dosages and different types of chemicals as reagents. Experiments were done in 3 different pH including 7, 8, and 9. 3 different dosage of collector (50, 140, 500 g/t), frother (15, 25, 50g/t), depressants ZnSO4 (0, 300, 600g/t), NaCN (50, 100 and 150 g/t) were selected. Tests were conducted in 3 different solid % including 30, 40, and 50. Flotation tests were done in 3L Denver cell. After adjusting pH, preflotation was conducted for one minute using MIBC (5g/t). After preflotation, again pH was controlled. Then NaCN, ZnSO4 and collector were added with one minute interval, respectively. In the next step, 5 minutes preparation and mixing were done and MIBC was added. Aeration and froth removal were performed for 6 minutes. The purpose of this step is to obtain the best flotation condition to get highest amount of grade and recovery, and consequently highest profit. The designed experiments for galena are given in Table 9.

Table 9. Optimization Tests for Galena

Run	Solid%	Collector (g/t)	pH	Frother(g/t)	ZnSO4 (g/t)	NaCN(g/t)	Pb Grade%	Pb Recovery%	Separation Efficiency (SE)	Yield%
1.00	50.00	140.00	8.00	50.00	600.00	100.00	27.35	55.18	51.50	8.78
2.00	40.00	140.00	8.00	25.00	300.00	100.00	28.64	59.55	55.53	9.04
3.00	40.00	140.00	9.00	15.00	300.00	150.00	24.93	57.64	53.09	10.06
4.00	40.00	140.00	9.00	15.00	300.00	50.00	23.21	59.23	53.95	11.10
5.00	40.00	140.00	8.00	25.00	300.00	100.00	28.15	60.21	56.00	9.30
6.00	40.00	140.00	8.00	25.00	300.00	100.00	29.07	59.98	55.99	8.98
7.00	40.00	140.00	7.00	15.00	300.00	50.00	22.49	58.50	53.14	11.32
8.00	40.00	50.00	8.00	25.00	600.00	150.00	25.26	41.93	39.56	7.22
9.00	30.00	140.00	9.00	25.00	300.00	150.00	28.28	61.06	56.76	9.39
10.00	50.00	140.00	9.00	25.00	300.00	50.00	25.20	56.34	52.06	9.73
11.00	50.00	50.00	8.00	15.00	300.00	100.00	26.76	36.46	34.81	5.93
12.00	40.00	50.00	9.00	25.00	600.00	100.00	24.86	43.13	40.57	7.55
13.00	30.00	140.00	7.00	25.00	300.00	150.00	28.94	60.54	56.45	9.10
14.00	40.00	500.00	7.00	25.00	600.00	100.00	23.09	64.40	58.12	12.13
15.00	30.00	500.00	8.00	50.00	300.00	100.00	25.48	65.24	59.58	11.14
16.00	40.00	500.00	8.00	25.00	600.00	50.00	22.25	64.99	58.28	12.71
17.00	50.00	140.00	8.00	15.00	600.00	100.00	26.61	54.73	50.97	8.95
18.00	30.00	140.00	8.00	50.00	-	100.00	27.25	62.30	57.59	9.95
19.00	40.00	140.00	8.00	25.00	300.00	100.00	28.83	59.54	55.56	8.98
20.00	30.00	50.00	8.00	50.00	300.00	100.00	24.38	51.72	47.95	9.23
21.00	40.00	140.00	8.00	25.00	300.00	100.00	28.65	60.12	56.03	9.13
22.00	50.00	50.00	8.00	50.00	300.00	100.00	23.10	40.39	37.92	7.61
23.00	40.00	50.00	7.00	25.00	600.00	100.00	25.04	43.21	40.67	7.51
24.00	40.00	500.00	9.00	25.00	-	100.00	21.74	65.29	58.31	13.06
25.00	30.00	140.00	8.00	50.00	600.00	100.00	28.23	61.25	56.91	9.44
26.00	40.00	50.00	8.00	25.00	-	50.00	20.28	45.44	41.75	9.75

Table 9 continued

27.00	40.00	140.00	7.00	50.00	300.00	150.00	27.10	63.27	58.37	10.16
28.00	40.00	50.00	9.00	25.00	-	100.00	24.73	45.22	42.39	7.95
29.00	30.00	50.00	8.00	15.00	300.00	100.00	22.06	54.86	50.02	10.82
30.00	40.00	140.00	9.00	50.00	300.00	50.00	26.21	65.17	59.73	10.82
31.00	50.00	140.00	8.00	50.00	-	100.00	26.53	57.08	52.98	9.36
32.00	30.00	140.00	9.00	25.00	300.00	50.00	27.74	63.16	58.43	9.90
33.00	50.00	500.00	8.00	15.00	300.00	100.00	19.45	64.23	56.46	14.37
34.00	40.00	500.00	8.00	25.00	-	50.00	20.27	67.23	59.14	14.43
35.00	40.00	500.00	8.00	25.00	-	150.00	21.97	66.75	59.55	13.22
36.00	50.00	140.00	7.00	25.00	300.00	50.00	25.59	56.88	52.60	9.67
37.00	40.00	50.00	8.00	25.00	600.00	50.00	24.42	44.39	41.62	7.91
38.00	50.00	500.00	8.00	50.00	300.00	100.00	17.09	69.74	59.01	17.75
39.00	30.00	140.00	8.00	15.00	-	100.00	26.23	59.17	54.69	9.81
40.00	40.00	50.00	7.00	25.00	-	100.00	23.86	45.67	42.64	8.33
41.00	40.00	500.00	9.00	25.00	600.00	100.00	23.55	63.18	57.29	11.67
42.00	40.00	140.00	7.00	15.00	300.00	150.00	25.08	57.32	52.86	9.94
43.00	40.00	140.00	9.00	50.00	300.00	150.00	26.58	64.10	58.94	10.49
44.00	30.00	500.00	8.00	15.00	300.00	100.00	24.26	63.74	57.97	11.43
45.00	40.00	500.00	8.00	25.00	600.00	150.00	23.89	62.05	56.47	11.30
46.00	50.00	140.00	7.00	25.00	300.00	150.00	26.20	55.47	51.53	9.21
47.00	40.00	140.00	8.00	25.00	300.00	100.00	28.38	59.80	55.70	9.17
48.00	40.00	50.00	8.00	25.00	-	150.00	25.20	44.15	41.52	7.62
49.00	30.00	140.00	8.00	15.00	600.00	100.00	26.96	57.23	53.19	9.23
50.00	50.00	140.00	8.00	15.00	-	100.00	23.22	56.27	51.51	10.54
51.00	30.00	140.00	7.00	15.00	300.00	50.00	25.97	59.94	55.28	10.04
52.00	40.00	140.00	7.00	50.00	300.00	50.00	25.74	64.28	58.86	10.86
53.00	50.00	140.00	9.00	25.00	300.00	150.00	23.67	62.35	56.65	11.46
54.00	40.00	500.00	7.00	25.00	-	100.00	21.39	63.50	56.76	12.91

Separation efficiency is calculated as below:

$$SE = R\ c_m(c - f)/c(c_m - f) \qquad 13$$

Where c is %assayed element in the concentrate

c_m is assayed element in mineral being concentrated

f is assayed element in feed

R is recovery of c in the concentrate

In this section, the effects of different factors on these outcome would be considered.

5.3.1. Effect of Solid% on Grade, Recovery, and Separation Efficiency

After conducting the experiments for rougher steps, the data was analyzed using one-way ANOVA. Three different outcomes were considered here that include grade, recovery and Separation Efficiency. In this section, the effects of different factors on these outcome would be considered.

a) Grade:

Grade is one of the main parameters to consider eather a concentrate or process performance is economical or not. If grade be lower than a certain amount, the probability of getting fine increases so it is necessary to track it in the flotation circuts. In addition, the grade of materials shows the quality of them as well as the performance of the process that are important parameters for the mineral processing plants. The results indicate at different condition the mean of grade for 30%, 40%, and 50% of solid are 26.3%, 24.8%, and 24.3%, respectively. By increasing solid%, galena grade in rougher concentrate decreases. However by changing solid% from 30 to 40 as well as 40 to 50 there is not significant differences based on information in Table , and Figure . But increasing solid% by 20 from 30 to 50 has a significant effect on the galena concentrate grade. As P-Value obtained by Fisher Individual Tests for level 30-50% solid is 0.049 that is less than 0.05 (Table 10, and Figure 30).

b) Recovery:

Another main parameter to consider the flotation performance is recovery of the products. However, sometimes recovery cannot represent a clear point of view about the product quality. Thus, recovery should be considered with grade together for taking final decision of changing flotation condition in order to improve the flotation performance. As it can be seen in Table changing solid% by either from 30 to 40 or 40 to 50 does not have a significant effect on the recovery because their P-Values are 0.32, and 0.488 that both of them are greater than 0.05. On the other hand, changing solid percent from 30 to 50 has a significant negative effect on the recovery (Table 11, and Figure 31).

Table 10. Mean & Fisher Individual Tests for Differences of Means for Effect of Solid% on Galena Grade

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
40 - 30	-1.486	0.889	(-3.271, 0.298)	-1.67	0.101
50 - 30	-2.08	1.06	(-4.22, 0.05)	-1.96	0.049
50 - 40	-0.598	0.889	(-2.382, 1.187)	-0.67	0.504
%Solid	**N**	**Mean**	**StDev**	**95% CI**	
30	12	26.315	2.014	(24.807, 27.823)	
40	30	24.829	2.564	(23.875, 25.783)	
50	12	23.231	3.165	(22.723, 25.739)	

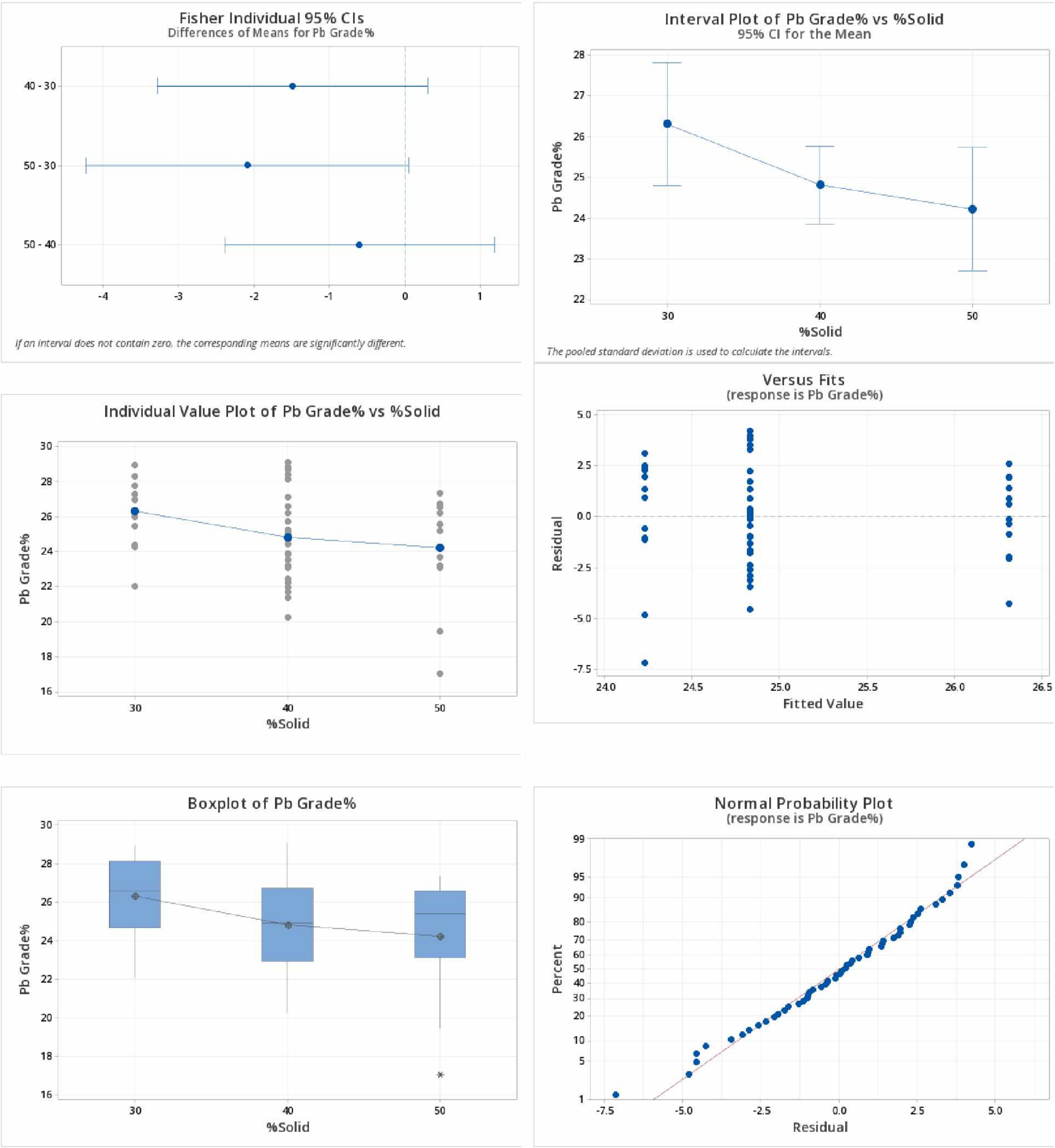


Figure 30. Statistics Graph of the Impact of Solid% on Galena Grade

Table 11. Mean & Fisher Individual Tests for Differences of Means for Effect of Solid% on Galena Recovery

%Solid	N	Mean	StDev	95% CI
30	12	60.02	3.85	(55.44, 64.59)
40	30	57.31	8.49	(54.41, 60.20)
50	12	55.43	9.16	(50.85, 60.00)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
40 - 30	-2.71	2.70	(-8.12, 2.70)	-1.00	0.320
50 - 30	-4.59	3.22	(-11.06, 1.88)	-1.42	0.0106
50 - 40	-1.88	2.70	(-7.29, 3.53)	-0.70	0.488

c) Separation Efficiency:

Separation Efficiency is defined as the ratio of concentration that has been eliminated from the process feed to the initial concentration in the feed. The results show by increasing solid%, the separation efficiency is decreasing As by increasing solid% by 20%, the SE decreases almost 5%. Increasing solid percentage by 10% from 30 to 40 or 40 to 50% does not show significant differences. Because their P values are 0.218, and 0.088 that both of them are greater than 0.05. On the other hand increasing solid% by 20% from 30 to 50 has a significant effects on SE and decrease it by 5%. As it can bee seen in Table 12 its P value is 0.023. It generally means the increasing of solid % has a negative effects of flotation performance when changes is more than 10% (Table 12, and Figure 32) .

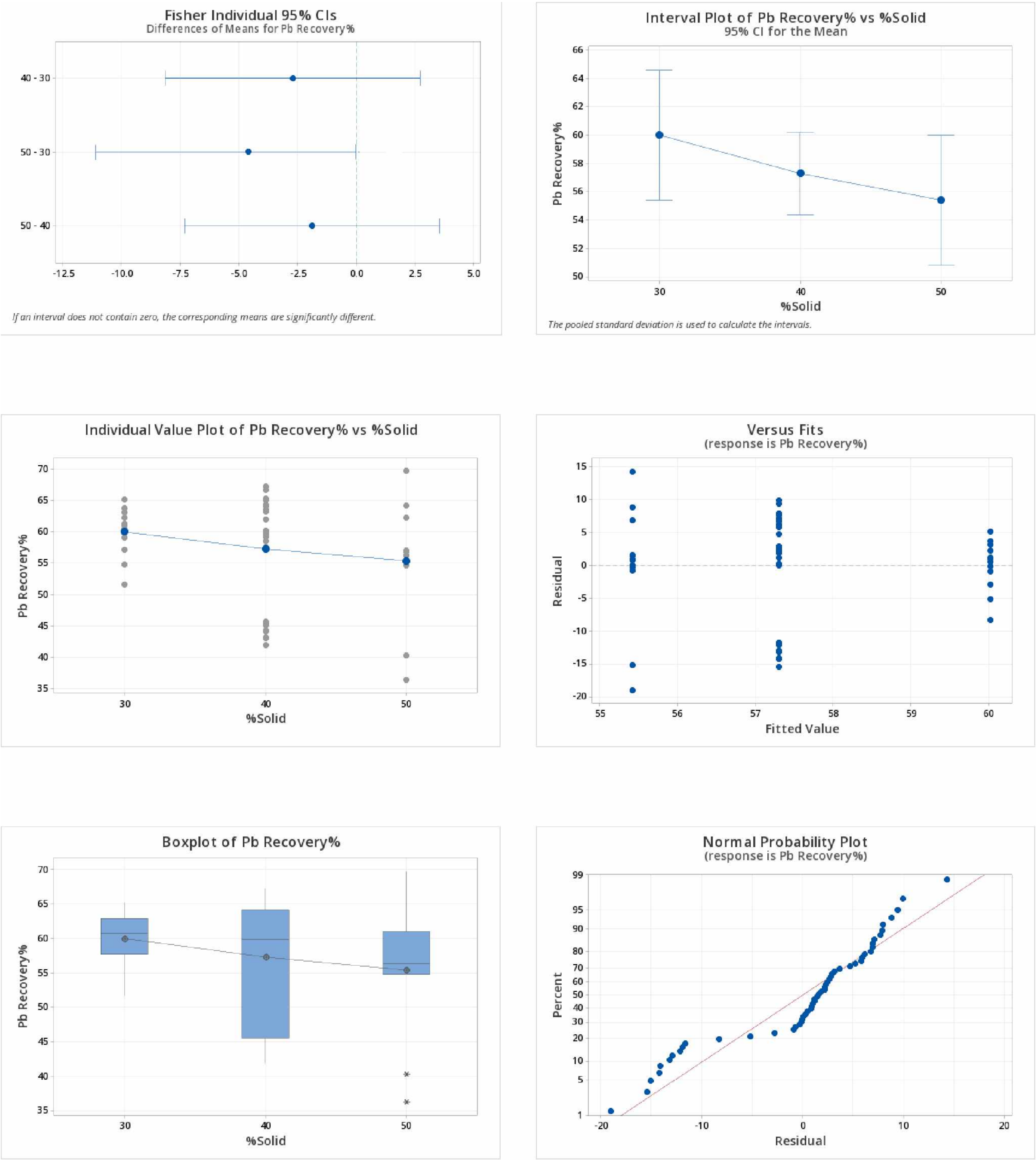


Figure 31 Statistics Graph of the Impact of Solid% on Galena Recovery

Table 12. Mean & Fisher Individual Tests for Differences of Means for Effect of Solid% on Galena Separation Efficiency

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
40 - 30	-2.79	2.24	(-7.28, 1.70)	-1.25	0.218
50 - 30	-4.74	2.67	(-10.11, 0.63)	-1.77	0.023
50 - 40	-1.95	2.24	(-6.44, 2.55)	-0.87	0.088
Solid%	Number	Mean	Simultaneous confidence level = 91.93% Significance level $\alpha = 0.05$		
30	12	55.40			
40	30	52.61			
50	12	50.67			

5.3.2. Effect of Collector(g/t) on Grade, Recovery, and Separation Efficiency

a) Grade

Collector is added to the flotation to increase the hydrophobicity of target minerals by binding to them. Collector has a significant effect on the grade and recovery of the flotation. Selecting an appropriate dosage of collector is so critical for flotation performance. Adding too low or too high dosage of collectors to the circuit can has a negative effect on the flotation performance. Low dosage of collector means there is not enough collector to attach to the surface of valuable minerals and float them. High dosage of collector also can create other problems in the flotation circuit and decreases both grade and recovery for example by increasing the collector dosage the probability of floating other sulfide minerals such as Pyrite and Sphalerite, and binary particle of sulfide minerals will increase that causes to decrease the grade of products. The data at table shows by increasing collector dosage from 50 to 140 the grade of galena product increase by 2.4% while by increasing collector dosage from 140 to 500 g/t, grade decreases by 4.5%. The highest grade was gained at 140 g/l of collector dosage. All P-Values are below 0.05 that show changing collector dosage has a significant impact on flotation performance in all the levels (Table 13, and Figure 33)

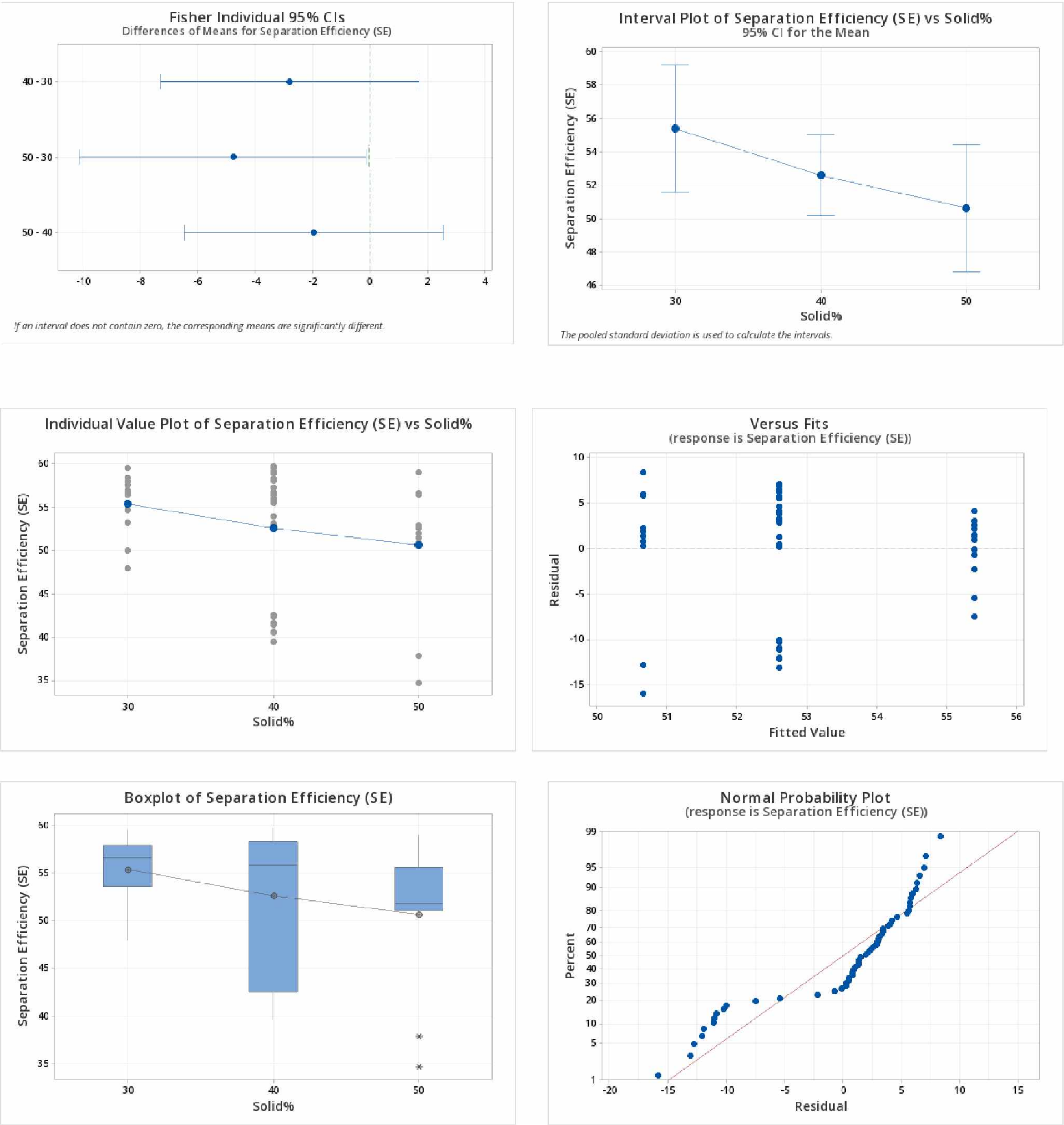


Figure 32. Statistics Graph of the Impact of Solid% on Galena Separation Efficiency

Table 13. Mean & Fisher Individual Tests for Differences of Means for Effect of Collector (g/t) on Galena Grade

Collector (g/t)	**N**	**Mean**	**StDev**	**95% CI**	
50	12	24.162	1.690	(23.054, 25.271)	
140	30	26.568	1.826	(25.867, 27.269)	
500	12	22.036	2.306	(20.927, 23.144)	
Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
140 - 50	2.405	0.653	(1.094, 3.717)	3.68	0.001
500 - 50	-2.127	0.781	(-3.694, -0.559)	-2.72	0.009
500 - 140	-4.532	0.653	(-5.843, -3.220)	-6.94	0.000

Simultaneous confidence level = 87.93%

b) Recovery

According to Table by increasing the collector dosage from 50 to 500 g/t. Mean Recovery increase from 44.71 to 65.028. The P-Value for all 3 collector levels is less than 0.05 that indicate a significant effect of collector on the recovery. However, by increasing collector dosage from 140 to 500 g/t, galena grade decreases, but recovery increase for these levels too. The highest amount of recovery was achieved with collector dosage of 500 g/t. Because by increasing of the collector dosage the probability of flotation of binary sulfide minerals as well as binary sulfide minerals that contain galena will increase (Table 14, and Figure 34).

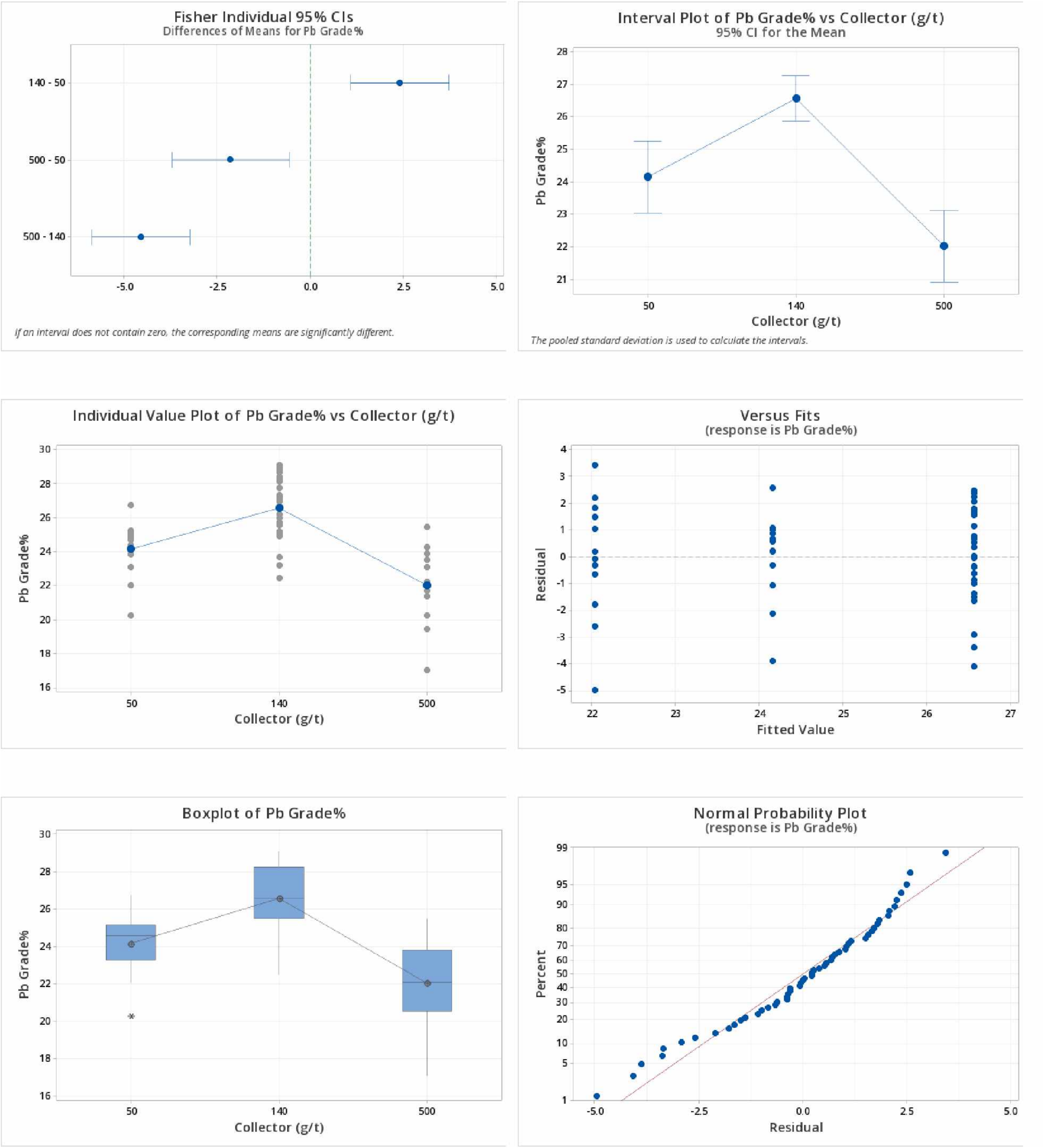


Figure 33. Statistics Graph of the Impact of Collector on Galena Grade

Table 14. Mean & Fisher Individual Tests for Differences of Means for Effect of Collector (g/t) on Galena Recovery

Collector (g/t)	N	Mean	StDev	95% CI
50	12	44.71	4.80	(42.84, 46.59)
140	30	59.589	2.833	(58.403, 60.774)
500	12	65.028	2.077	(63.153, 66.903)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
140 - 50	14.87	1.10	(12.66, 17.09)	13.46	0.00001
500 - 50	20.31	1.32	(17.66, 22.97)	15.38	0.00002
500 - 140	5.44	1.10	(3.22, 7.66)	4.92	0.00016

Simultaneous confidence level = 87.93%

c) Separation Efficiency

By increasing collector dosage from 40 to 500 g/t the separation efficiency will increase from 41.78% to 58.08%. The P-Value shows increasing the collector dosage has a significant effect on separation efficiency (Table 15, and Figure 35).

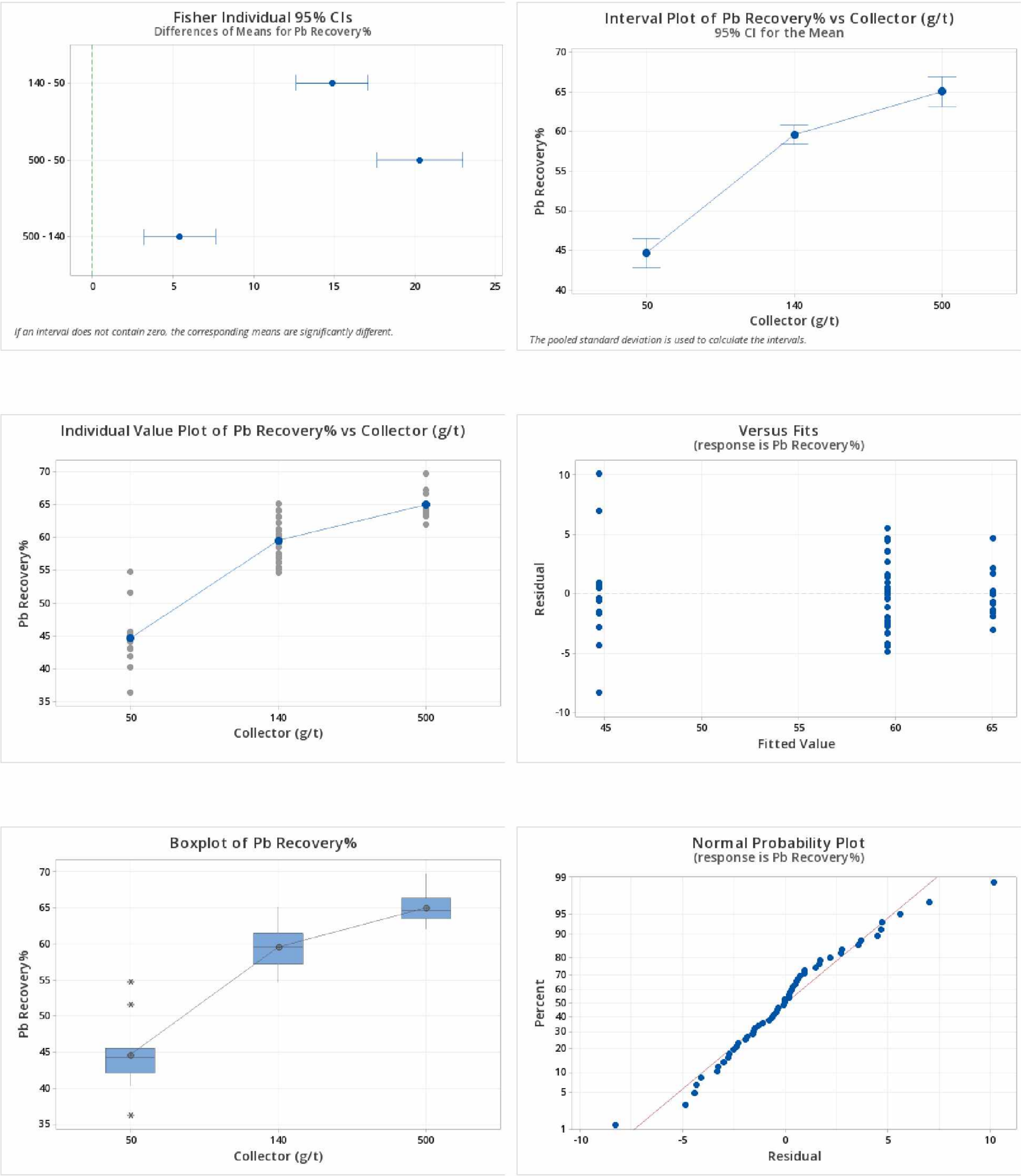


Figure 34. Statistics Graph of the Impact of Collector on Galena Recovery

Table 15. Mean & Fisher Individual Tests for Differences of Means for Effect of Collector (g/t) on Galena Separation Efficiency

Collector (g/t)	**N**	**Mean**	**StDev**	**95% CI**
50	12	41.78	4.03	(40.21, 43.36)
140	30	55.095	2.529	(54.097, 56.093)
500	12	58.079	1.132	(56.501, 59.656)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
140 - 50	13.310	0.930	(11.443, 15.177)	14.31	0.000
500 - 50	16.29	1.11	(14.06, 18.53)	14.66	0.000
500 - 140	2.984	0.930	(1.117, 4.851)	3.21	0.002

Simultaneous confidence level = 87.93%

5.3.3. Effect of pH on Grade, Recovery, and Separation Efficiency

Controling pH in the flotation is a vital method to control selective mineral seperation. In this section, the range of pHs including7, 8, and 9 that usually is applied at galena flotation was considered. Then effect of pH on grade, recovery, and separation efficiency was examined.

a) Grade

However pH is one of the main parameters that any significant changes in it can has an adverse effect on the flotation grade, but it has not a singnificat effect on the flotation grade. Because in this series of experiments , a limited range of pH was used that the result shows all of them has almost same mean grade at galena flotation. In addition P-values for all of them are greater than

0.05 that prove it is not an effective parameter in these series of experiment (Table 16, and Figure 36).

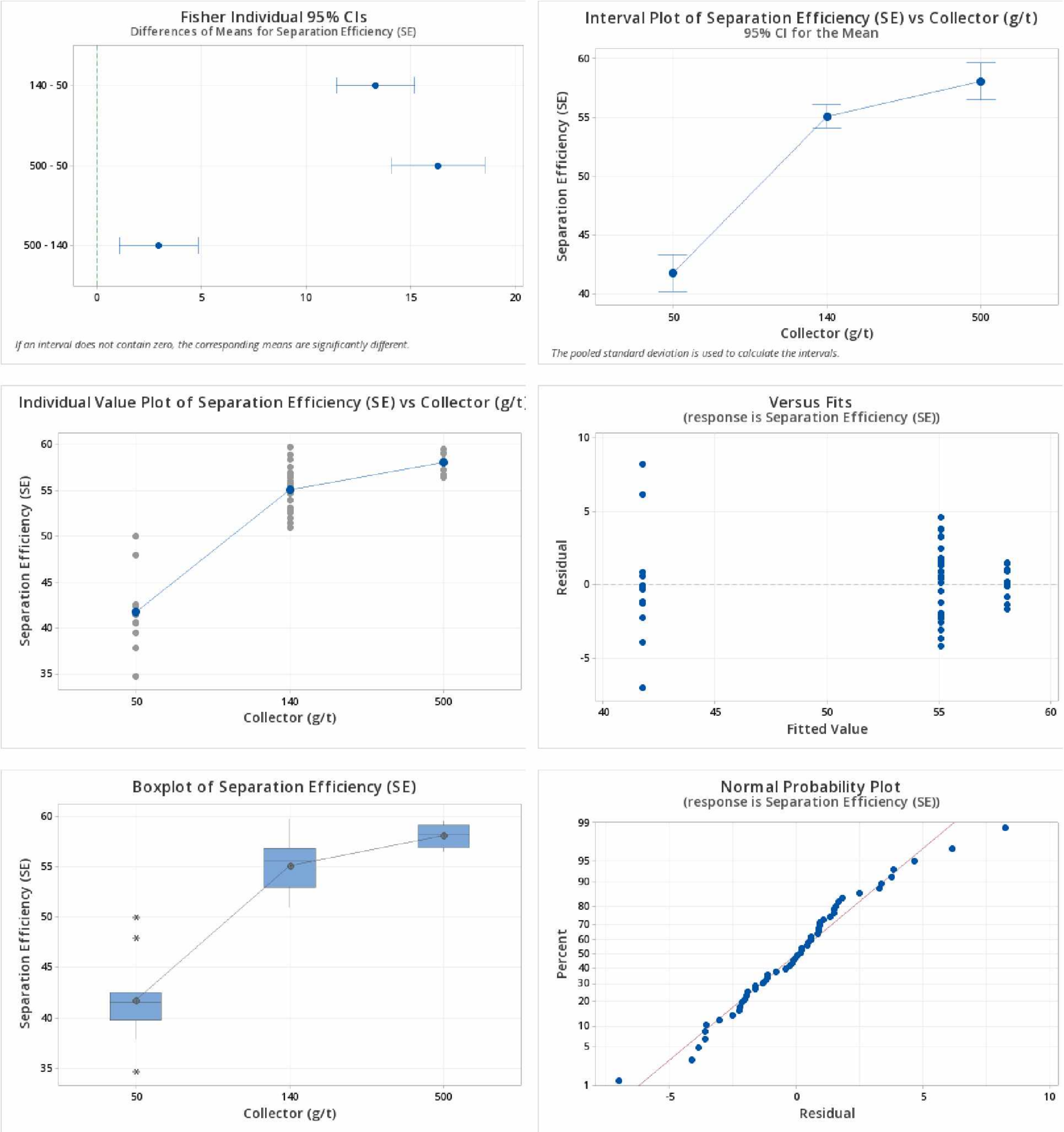


Figure 35. Statistics Graph of the Impact of Collector on Galena Separation Efficiency

Table 16. Mean & Fisher Individual Tests for Differences of Means for Effect of pH on Galena Grade

pH	**N**	**Mean**	**StDev**	**95% CI**
7	12	25.041	2.078	(23.471, 26.610)
8	30	25.007	3.144	(24.015, 26.000)
9	12	25.058	1.908	(23.489, 26.628)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
8 - 7	-0.033	0.925	(-1.891, 1.824)	-0.04	0.971
9 - 7	0.02	1.11	(-2.20, 2.24)	0.02	0.987
9 - 8	0.051	0.925	(-1.806, 1.908)	0.06	0.956

Simultaneous confidence level = 87.93%

b) Recovery

As mentioned above , because of the selection of the limited range of pH, even it does not have a signifficant affect on the flotation recovery. The average recoveryfor different level changes between 57.7% to 58.8%. The P-Values are 0.75, 0.74, and o.48 for pH 7, 8, and 9, respectively. All P-values are greater than 0.05 that shows pH does not have effective impact on the experiments (Table 17, and Figure 37).

c) Separation efficiency

According to table , pH also does not have any significant effect on separation efficiancy. All P-Values are greater than 0.05. By increasing pH, almost separation efficiency remains the same amount (Table 18, and Figure 38).

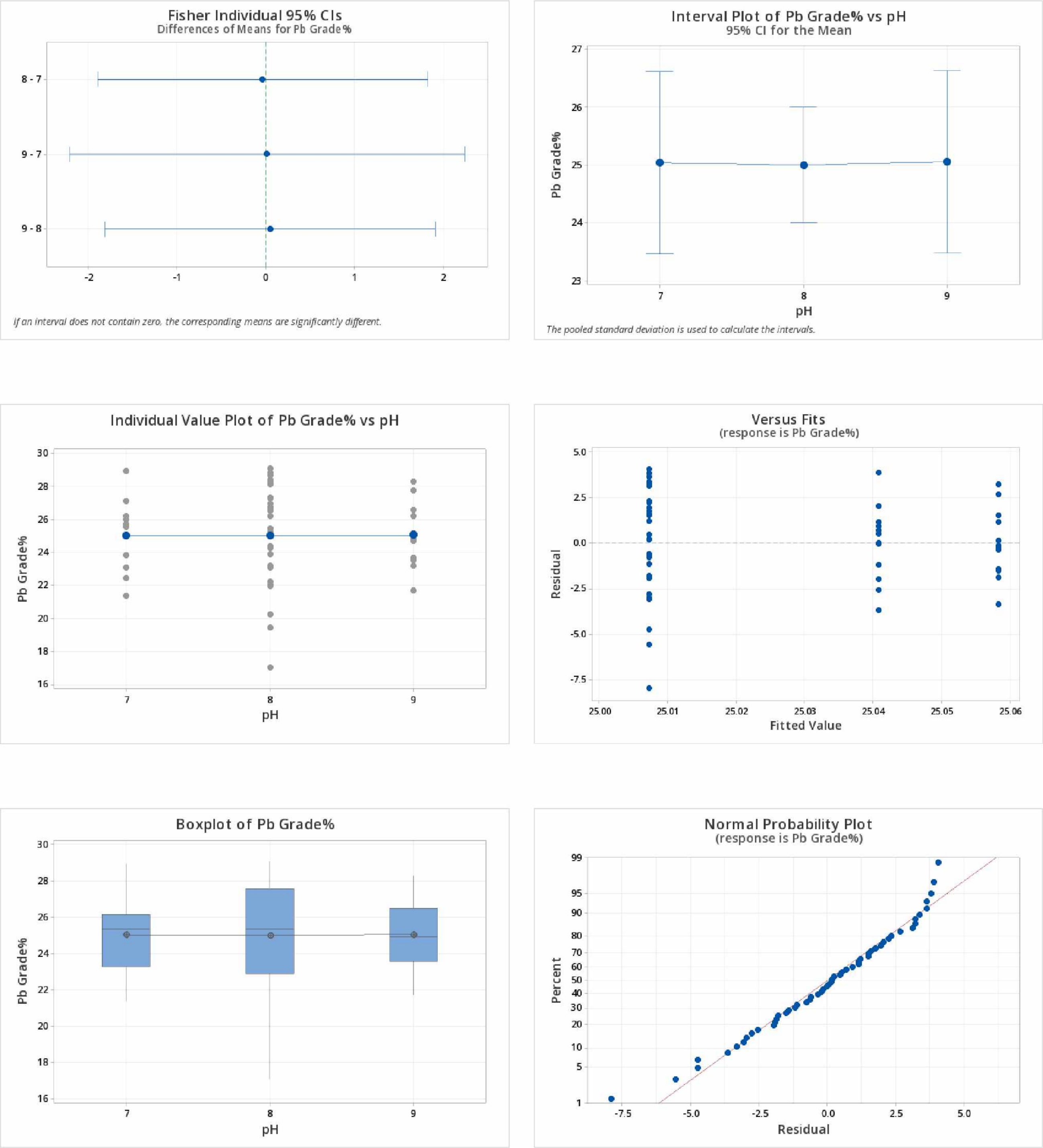


Figure 36. Statistics Graph of the Impact of pH on Galena Grade

Table 17. Mean & Fisher Individual Tests for Differences of Means for Effect of pH on Galena Recovery

pH	**N**	**Mean**	**StDev**	**95% CI**
7	12	57.75	6.92	(53.11, 62.39)
8	30	56.86	8.59	(53.92, 59.79)
9	12	58.82	7.41	(54.18, 63.46)

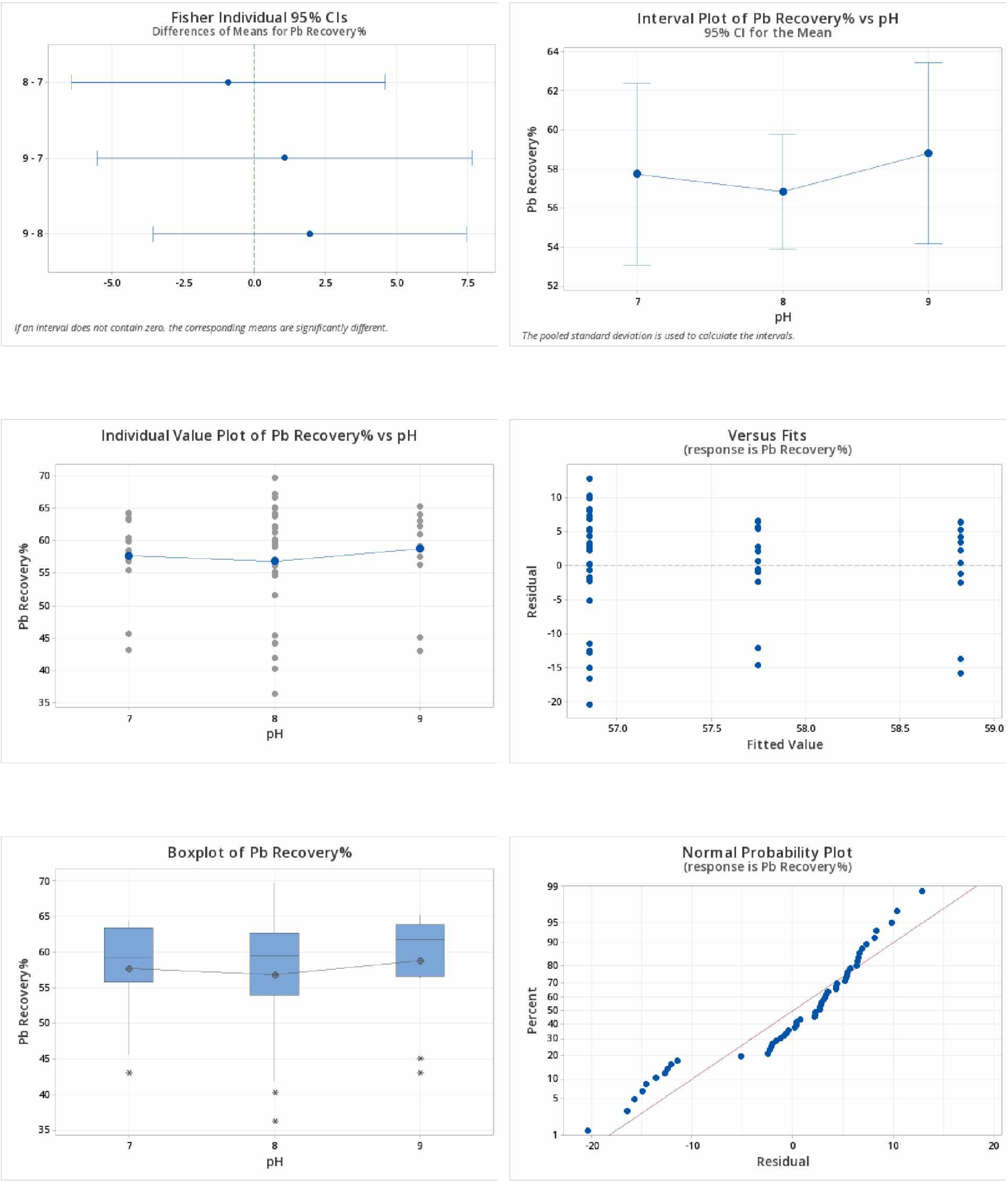
Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
8 - 7	-0.89	2.74	(-6.38, 4.60)	-0.33	0.746
9 - 7	1.07	3.27	(-5.49, 7.64)	0.33	0.744
9 - 8	1.97	2.74	(-3.53, 7.46)	0.72	0.476

Simultaneous confidence level = 87.93%

Table 18. Mean & Fisher Individual Tests for Differences of Means for Effect of pH on Galena

pH	**N**	**Mean**	**StDev**	**95% CI**
7	12	53.11	5.89	(49.22, 56.99)
8	30	52.19	7.13	(49.73, 54.65)
9	12	54.01	6.33	(50.13, 57.90)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
8 - 7	-0.91	2.29	(-5.52, 3.69)	-0.40	0.692
9 - 7	0.91	2.74	(-4.59, 6.41)	0.33	0.742
9 - 8	1.82	2.29	(-2.78, 6.42)	0.80	0.430


Fisher Individual 95% CIs
Differences of Means for Pb Recovery%

If an interval does not contain zero, the corresponding means are significantly different.

Interval Plot of Pb Recovery% vs pH
95% CI for the Mean

The pooled standard deviation is used to calculate the intervals.

Individual Value Plot of Pb Recovery% vs pH

Versus Fits
(response is Pb Recovery%)

Boxplot of Pb Recovery%

Normal Probability Plot
(response is Pb Recovery%)

Figure 37. Statistics Graph of the Impact of pH on Galena Recovery

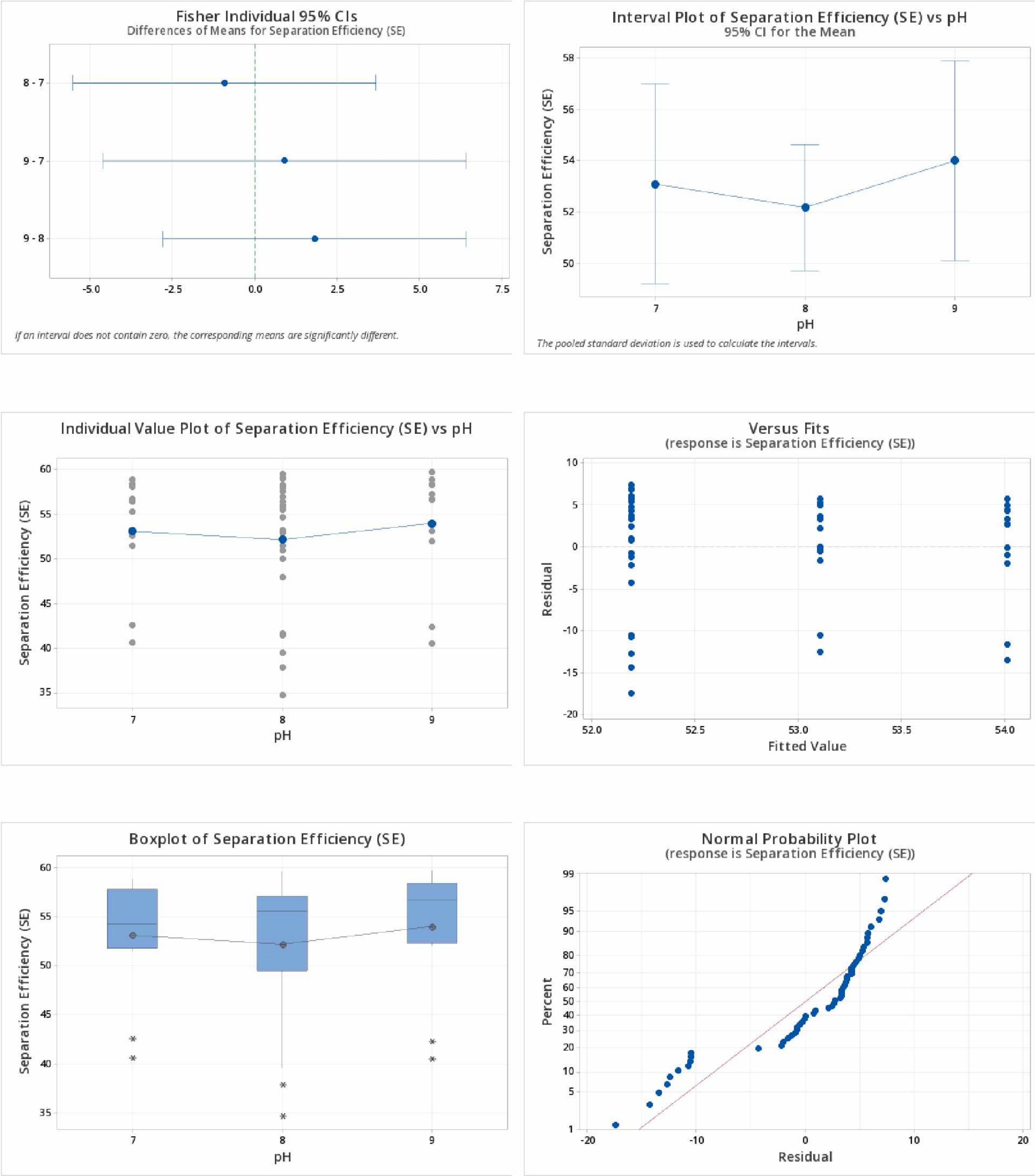


Figure 38. Statistics Graph of the Impact of pH on Galena Separation Efficiency

5.3.4. Effect of Frother on Grade, Recovery, and Separation Efficiency

The frother control the size, and stability of bubbles on the flotation cells. The bubble size, and stability can effects on flotation performance. Therefore, controlling the amount of frother in the flotation cell is necessary.

a) Grade:

The changes of frother from either 15 to 25, and 25 to 50 g/t have not significant impacts on grade. The P-Values for them are 0.41, and 0.77, respectively that are greater than 0.05. But when the amount of frother increase from 15 to 50, it shows it is an important parameter on flotation. Hence, controlling the amount of added frother on flotation cell and bubbles sizes are important. The P-Value for level 50-15 is 0.025 (Table 19, and Figure 39).

Table 19. Mean & Fisher Individual Tests for Differences of Means for Effect of Frother on Galena Grade

Frother	**N**	**Mean**	**StDev**	**95% CI**
15	13	24.402	2.236	(22.909, 25.896)
25	29	25.143	2.737	(24.143, 26.143)
50	12	26.420	3.969	(24.866, 27.974)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
25 - 15	0.740	0.895	(-1.057, 2.537)	0.83	0.412
50 - 15	1.02	1.07	(-1.14, 3.17)	0.95	0.025
50 - 25	0.277	0.921	(-1.571, 2.125)	0.30	0.765

Simultaneous confidence level = 87.93%

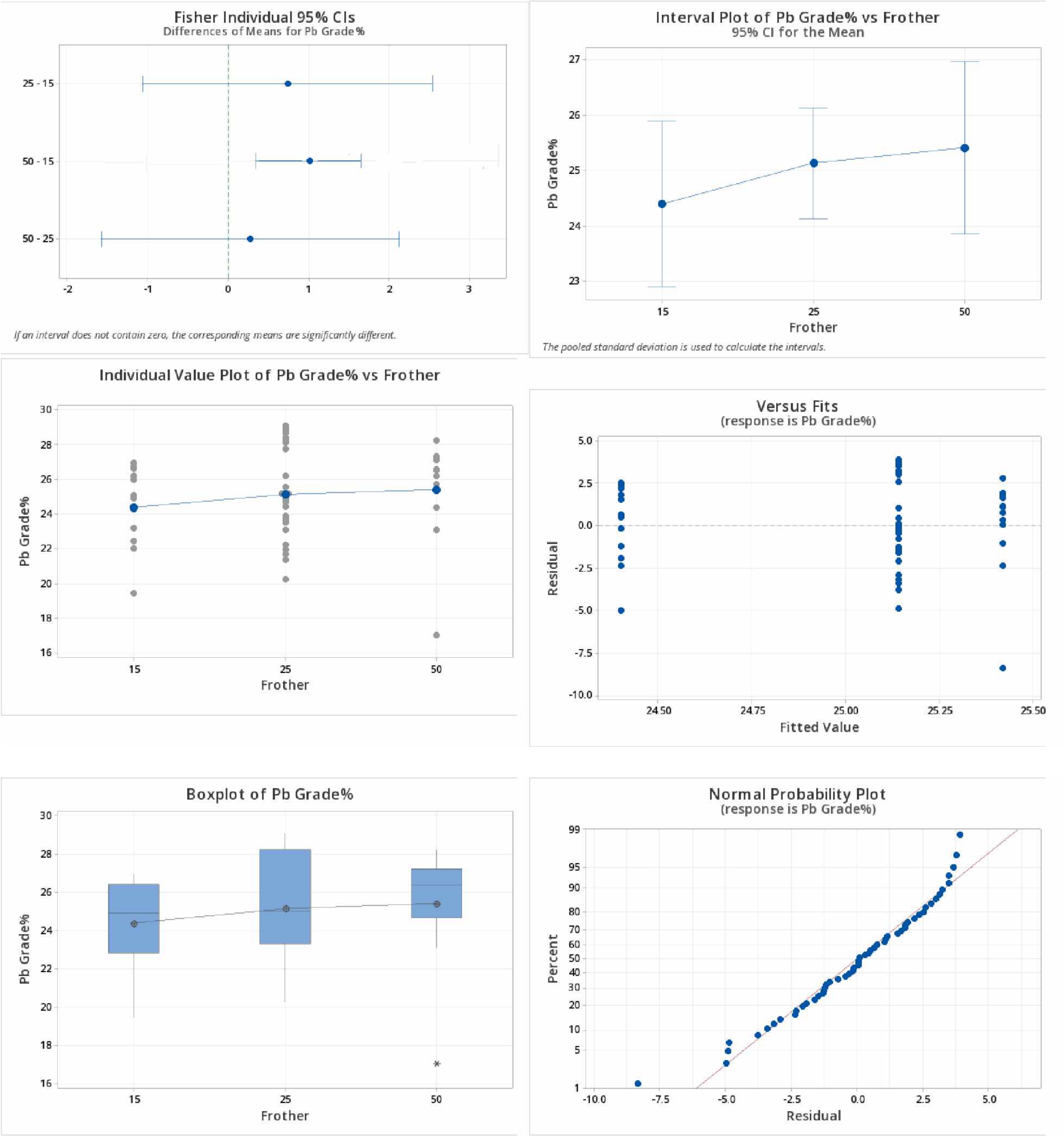


Figure 39. Statistics Graph of the Impact of Frother on Galena Grade

b) Recovery

As the results show, however, by increasing the frother dosage, recovery is increasing slightly, but it does not have significant impact on the flotation performance. The P-Values for all three levels are greater than 0.05 (Table 20, and Figure 40).

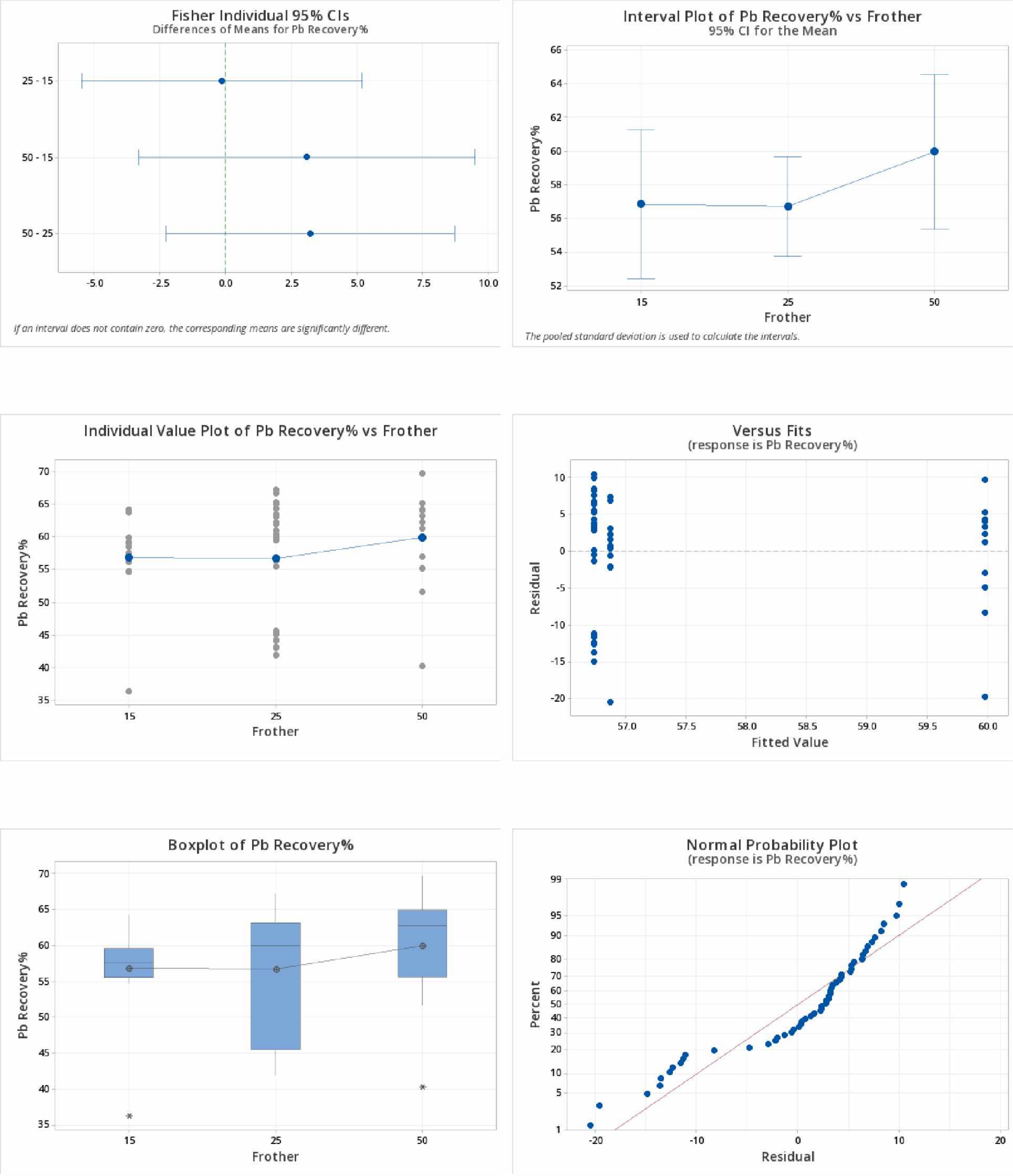


Figure 40. Statistics Graph of the Impact of Frother on Galena Recovery

Table 20. Mean & Fisher Individual Tests for Differences of Means for Effect of Frother on Galena Recovery

Frother	N	Mean	StDev	95% CI
15	13	56.87	6.77	(52.45, 61.29)
25	29	56.74	8.40	(53.78, 59.70)
50	12	59.98	7.90	(55.38, 64.58)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
25 - 15	-0.13	2.65	(-5.45, 5.19)	-0.05	0.962
50 - 15	3.11	3.18	(-3.27, 9.48)	0.98	0.333
50 - 25	3.23	2.72	(-2.23, 8.70)	1.19	0.241

Simultaneous confidence level = 87.93%

Table 21. Mean & Fisher Individual Tests for Differences of Means for Effect of Frother on Galena Separation Efficiency

Frother	N	Mean	StDev	95% CI
15	13	52.15	5.65	(48.45, 55.85)
25	29	52.20	7.08	(49.73, 54.68)
50	12	54.94	6.53	(51.09, 58.80)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
25 - 15	0.05	2.22	(-4.40, 4.51)	0.02	0.980
50 - 15	2.79	2.66	(-2.55, 8.14)	1.05	0.299
50 - 25	2.74	2.28	(-1.84, 7.32)	1.20	0.235

Simultaneous confidence level = 87.93%

c) Separation Efficiency

The mean of separation efficiency for frother dosages of 15, 25, and 50 are 52.12, 52.2, and 54.94%, respectively. The changes are not significant especially by increasing dosage of frother from 15 to 25%. The P-Values for all categories are greater than 0.05. Thus, frother dosage are not a significant factor that impacts on flotation performance (Table 21, and Figure 41).

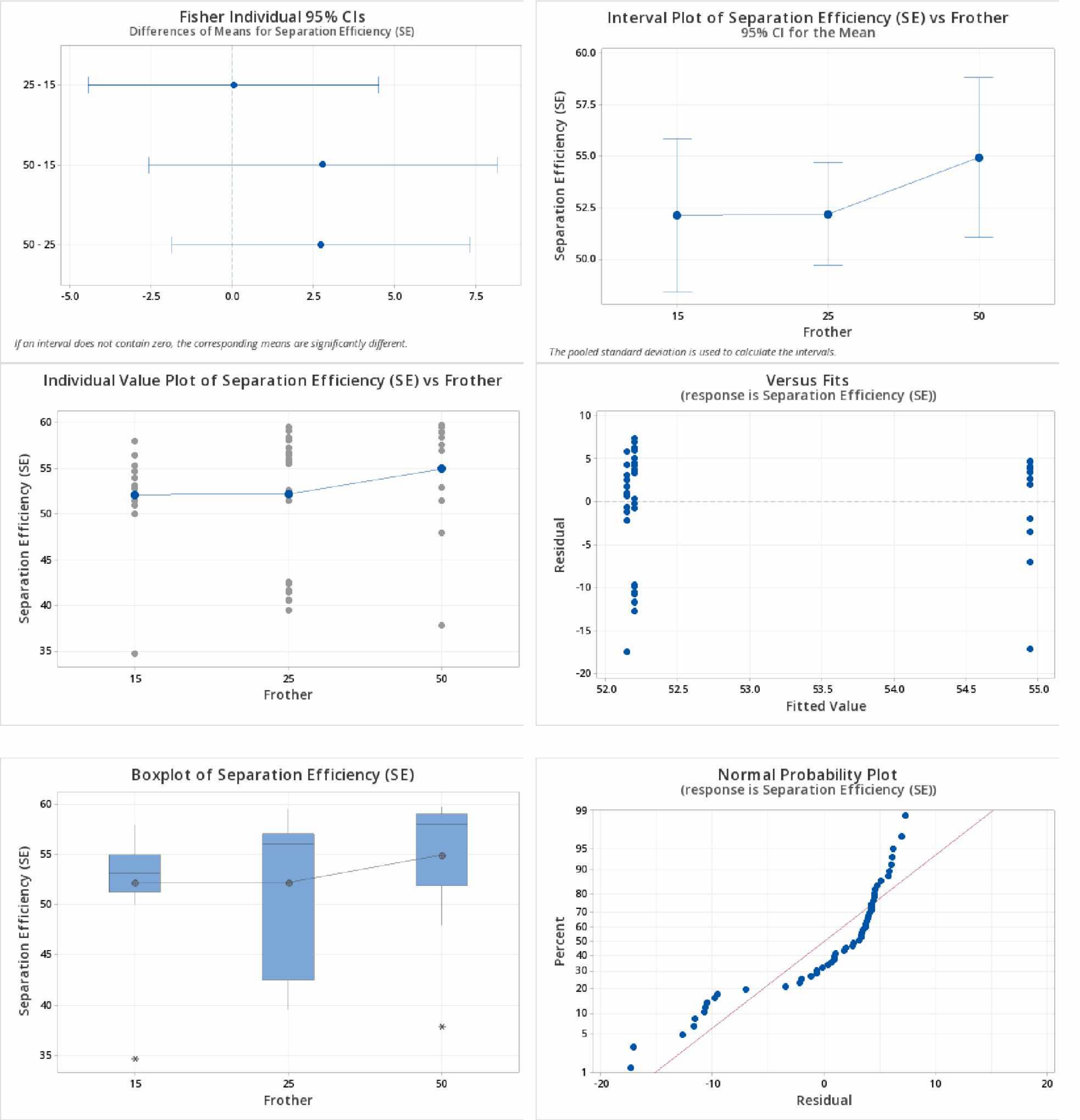


Figure 41. Statistics Graph of the Impact of Frother on Galena Separation Efficiency

5.3.5. Effect of Depressant (ZnSO4) on Grade, Recovery, and Separation Efficiency

Depressants are added to the flotation circuts to control the umwanted materials. In the experiments ZnSO4 wre added to the flotation process to depress Sphalarite. Here I will consider how ZnSO4 can affect grade, recovery , and separation efficiency.

a) Grade:

As the results shows, by increasing ZnSO4 dosage from 0 to 300 g/t at the flotation process, the grade increased significantly as P-Value is 0.026<0.05. Increasing ZnSO4 from 300 to 600 does not show an incridable changes in the grade But adding extra dosage of ZnSO4 decrease the grade slightly, However this changes is ignorable because the P-Value is 0.61>0,05. So it is necessary to keep dosage of ZnSO4 between 300 to 600 to get higher grad (Table 22, and Figure 42).

Table 22. Mean & Fisher Individual Tests for Differences of Means for Effect of ZnSO4 on Galena Grade

ZnSO4 (g/t)	N	Mean	StDev	95% CI
0	12	23.556	2.449	(22.062, 25.050)
300	30	25.574	2.852	(24.629, 26.519)
600	12	25.126	1.838	(23.632, 26.620)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
300 - 0	2.019	0.881	(0.250, 3.787)	2.29	0.026
600 - 0	1.57	1.05	(-0.54, 3.68)	1.49	0.142
600 - 300	-0.449	0.881	(-2.217, 1.320)	-0.51	0.613

Simultaneous confidence level = 87.93%

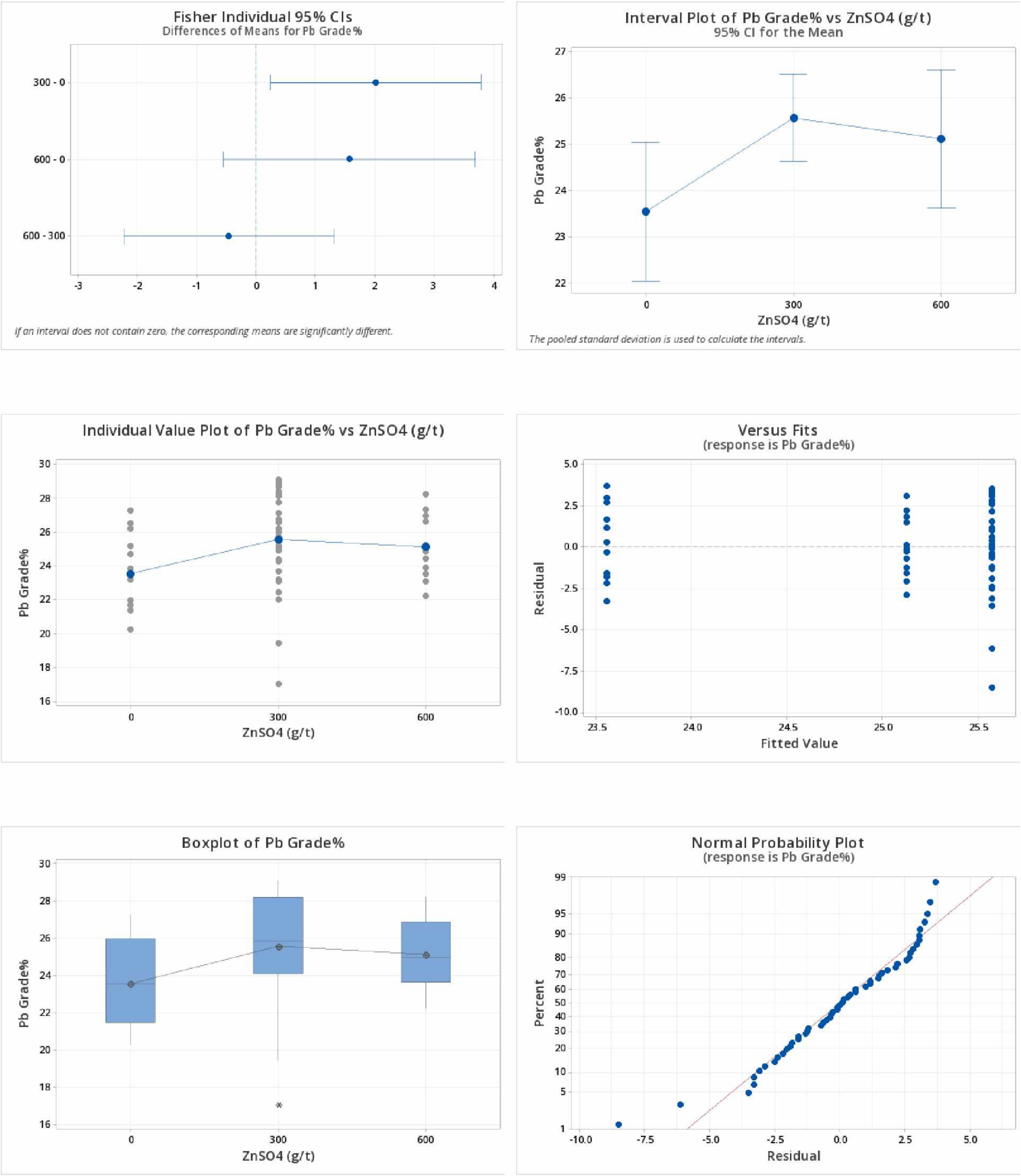


Figure 42. Statistics Graph of the Impact of ZnSO4 on Galena Grade

b) Recovery:

Increasing the amount of ZnSO4 in the flotation process does not have a significant effect on galena recovery. The P-Values for all 3 levels are greater than 0.05. The highest recovery was achieved with 300 g/t of ZnSO4, however the differences between recoveries of the levels based on ZnSO4 dosage are not remarkable (Table 23, and Figure 43).

c) Separation efficiency

The highest separation efficiency was gained when ZnSO4 dosage was around 300 g/ton. The results do not show a considerable change in the SE by changing the dosage of ZnSO4. Because at all levels the P-Values are greater than 0.05 (Table 24, and Figure 44).

Table 23. Mean & Fisher Individual Tests for Differences of Means for Effect of ZnSO4 on Galena Recovery

ZnSO4 (g/t)	**N**	**Mean**	**StDev**	**95% CI**
0	12	56.51	9.08	(51.97, 61.04)
300	30	59.03	6.72	(56.16, 61.90)
600	12	54.64	9.10	(50.10, 59.18)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
300 - 0	2.52	2.67	(-2.85, 7.89)	0.94	0.350
600 - 0	-1.87	3.20	(-8.28, 4.55)	-0.58	0.562
600 - 300	-4.39	2.67	(-9.76, 0.98)	-1.64	0.107

Simultaneous confidence level = 87.93%

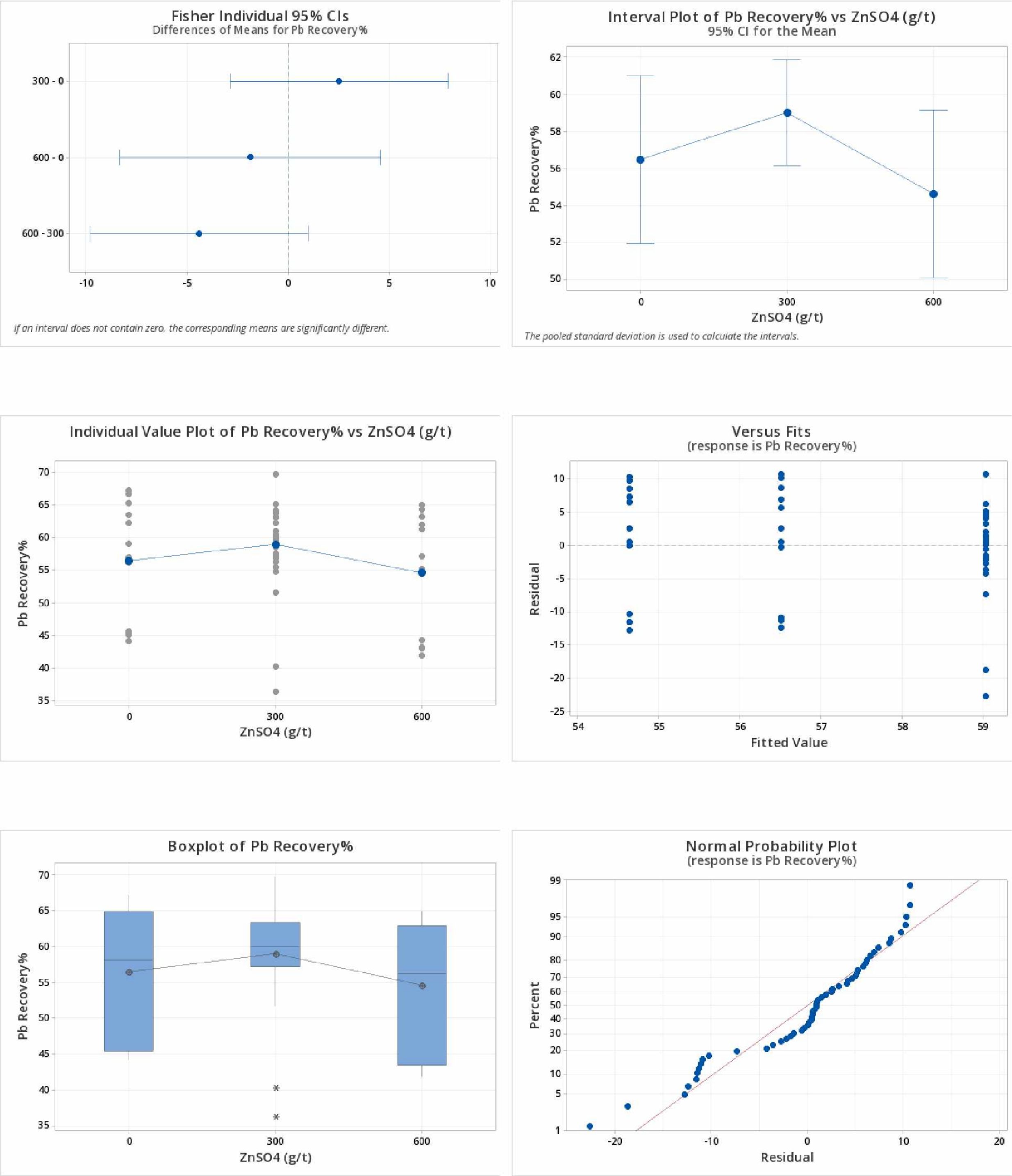


Figure 43. Statistics Graph of the Impact of ZnSO4 on Galena Recovery

Table 24. Mean & Fisher Individual Tests for Differences of Means for Effect of ZnSO4 on Galena Separation Efficiency

ZnSO4 (g/t)	**N**	**Mean**	**StDev**	**95% CI**
0	12	51.57	7.40	(47.78, 55.36)
300	30	54.24	5.67	(51.84, 56.64)
600	12	50.43	7.65	(46.64, 54.22)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
300 - 0	2.67	2.23	(-1.81, 7.15)	1.20	0.237
600 - 0	-1.14	2.67	(-6.50, 4.22)	-0.43	0.671
600 - 300	-3.81	2.23	(-8.29, 0.67)	-1.71	0.094

Simultaneous confidence level = 87.93%

5.3.6. Effect of Depressant (NaCN) on Grade, Recovery, and Separation Efficiency

In Galena flotation NaCN uses as a Pyrite depressant. In this section, effects of NaCN dosage galena grade, recovery, and separation efficiency was considered. Dosage of chemical can be an effective parameters on the flotation performance.

a) Grade

The results show by increasing the dosage of NaCN from 50 to 150 g/t the galena concentrate grade will increase from 24.11 to 25.59%, The increased amount is not a significant amount. Because P-Values for all 3 different level are greater than 0.05 (Table 25, Figure 45).

b) Recovery

The impact of NaCN on the recovery was investigated. It can be seen in Table 26, and Figure 46, adding 100 g/ton NaCN to the flotation cell causes a drop on the recovery however lower or upper dosage cause an increase in the flotation recovery. But the increase of recovery is ignorable because the P-Values for all 3 different levels are greater than 0.05 according to Table 26.

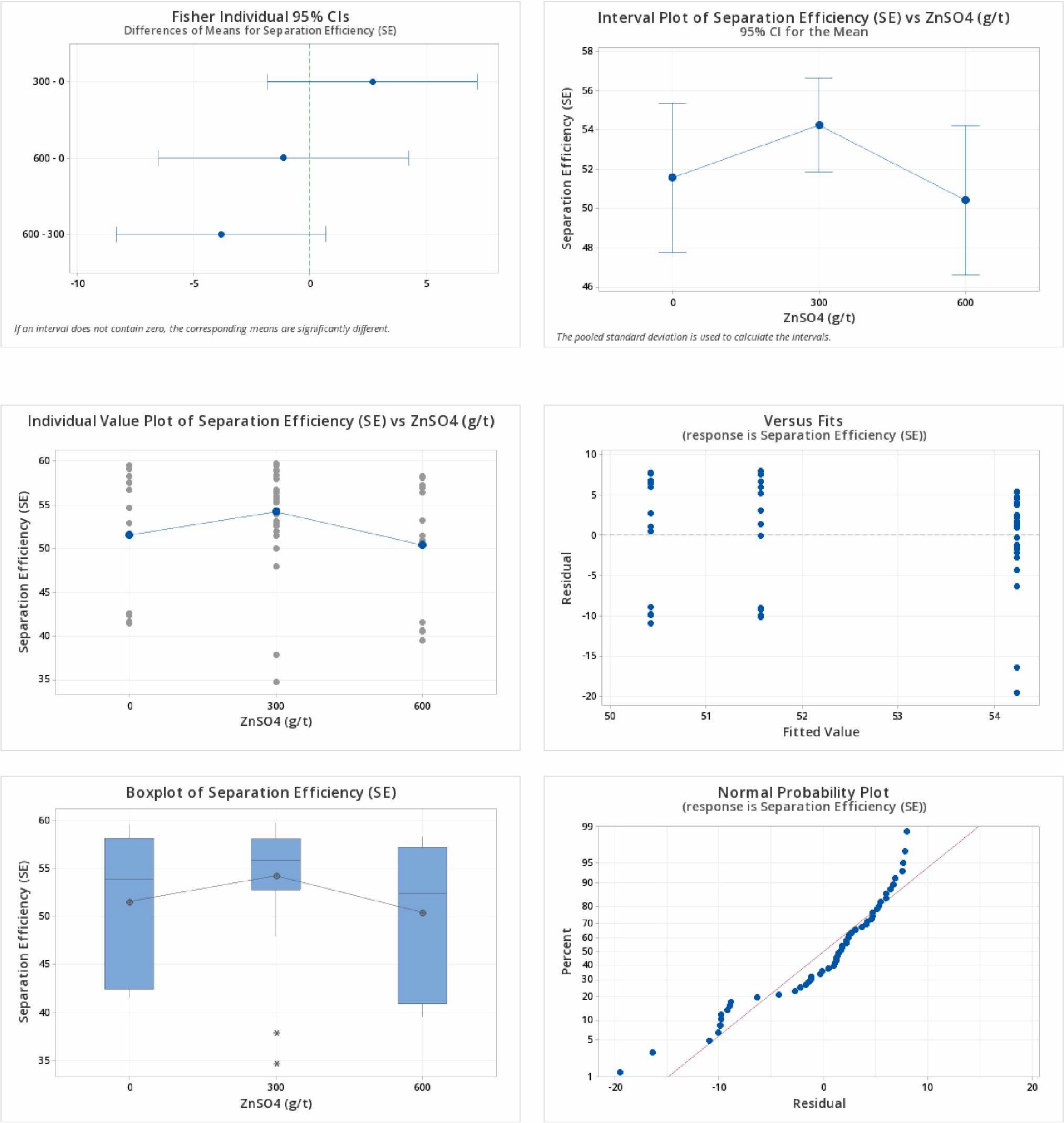


Figure 44. Statistics Graph of the Impact of ZnSO4 on Galena Separation Efficiency

Table 25. Mean & Fisher Individual Tests for Differences of Means for Effect of NaCN on Galena Grade

NaCN(g/t)	**N**	**Mean**	**StDev**	**95% CI**
50	12	24.114	2.402	(22.575, 25.653)
100	30	25.165	2.957	(24.191, 26.138)
150	12	25.592	1.971	(24.053, 27.131)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
100 - 50	1.051	0.907	(-0.771, 2.872)	1.16	0.252
150 - 50	1.48	1.08	(-0.70, 3.65)	1.36	0.179
150 - 100	0.427	0.907	(-1.394, 2.248)	0.47	0.640

Simultaneous confidence level = 87.93%

Table 26. Mean & Fisher Individual Tests for Differences of Means for Effect of NaCN on Galena Recovery

NaCN(g/t)	**N**	**Mean**	**StDev**	**95% CI**
50	12	58.80	7.36	(54.16, 63.43)
100	30	56.75	8.34	(53.81, 59.68)
150	12	58.05	7.69	(53.42, 62.69)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
100 - 50	-2.05	2.73	(-7.54, 3.44)	-0.75	0.457
150 - 50	-0.74	3.27	(-7.30, 5.82)	-0.23	0.821
150 - 100	1.31	2.73	(-4.18, 6.79)	0.48	0.635

Simultaneous confidence level = 87.93%

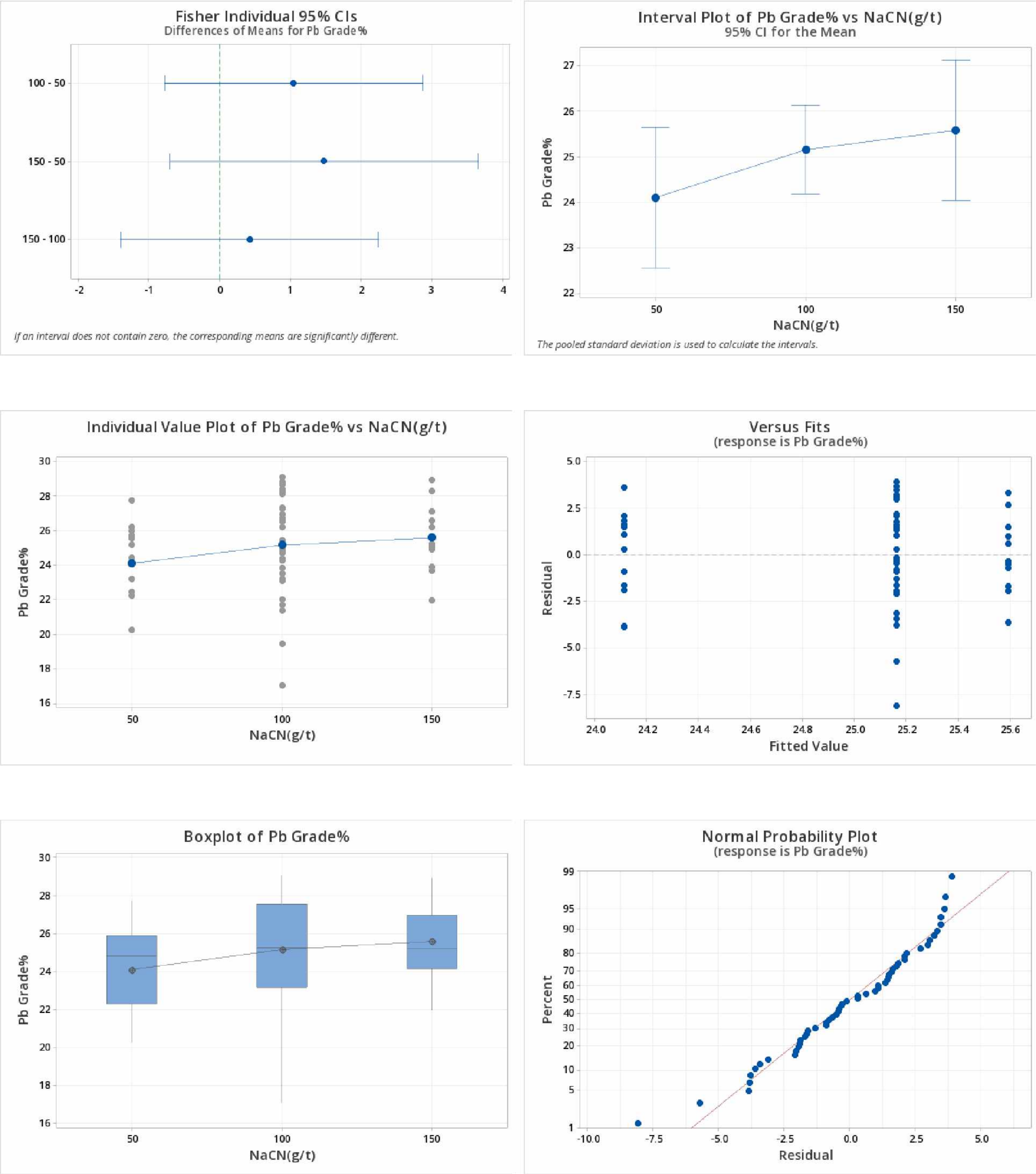


Figure 45. Statistics Graph of the Impact of NaCN on Galena Grade

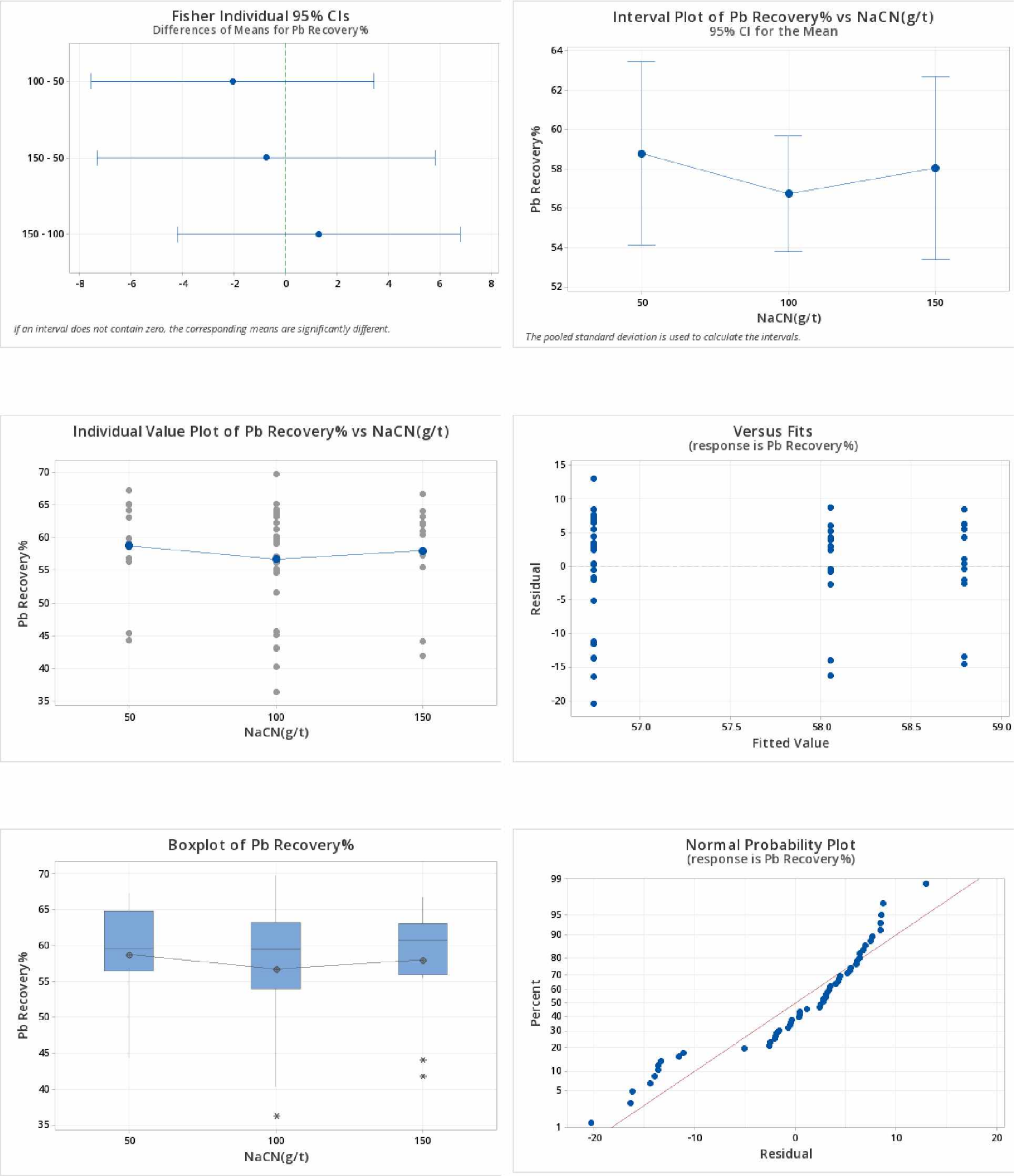


Figure 46. Statistics Graph of the Impact of NaCN on Galena Grade

c) Separation efficiency

Separation efficiency does not show a remarkable change by adding higher dosage of NaCN. All P-Values for different levels are greater than 0.05. So, the dosage of NaCN does not have an importance effect on the flotation performance (Table 27, Figure 47).

Table 27. Mean & Fisher Individual Tests for Differences of Means for Effect of NaCN on Galena Separation Efficiency

NaCN(g/t)	**N**	**Mean**	**StDev**	**95% CI**
50	12	53.74	6.26	(49.85, 57.63)
100	30	52.15	6.94	(49.69, 54.61)
150	12	53.48	6.54	(49.59, 57.37)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
100 - 50	-1.58	2.29	(-6.19, 3.02)	-0.69	0.493
150 - 50	-0.26	2.74	(-5.76, 5.24)	-0.09	0.926
150 - 100	1.33	2.29	(-3.28, 5.93)	0.58	0.566

Simultaneous confidence level = 87.93%

5.3.7. A Summery of Chemicals Effects on the Galena Grade, Recovery, Separation Efficiency at the Flotation

1) Increasing solid% from 30 to 50% has a negative effect on the grade. As by increasing the solid percentage by 20%, the avarage grade decrease by 3%.
2) Increasing solid% from 30 to 50% has a negative effect on the recovery. The recovery drops from 60.02 to 55.43%.
3) Increasing solid% causes to decrease sepration efficiency from 55.4 to 50.67%

4) Increasing the collector dosage from 50 to 140 g/ton as well as 140 to 500 has a significant impact on the grade. The highest grade was achieved at 140 g/ton of collector dosage.
5) Increasing collector dosage from 50 to 500 g/ton has a significant effect on the galena recovery and separation efficiency.By increasing collector dosage separation efficiency increase from 41.78 to 58.8%
6) Because of the limited range of pH in my experiments, it does not indicate a significant impact on grade, recovery , and separation efficiency of the galena.
7) Increasing the frother dosage from 15 to 50 g/ton shows a remarkable impact on the galena concentrate grade. As by increasing frother dosage from 15 to 50 g/ton, the concentrate grade increased from 24.4 to 26.42%
8) Increasing frother dosage from 15 to 50 g/t does not show a remarkable effect on the recovery and separation efficiency of galena.
9) Increasing ZnSO4 dosage from 0 to 300 g/ton has a significant impact on the galena grade. The grade increased by 2.5%.
10) Increasing ZnSO4 dosage from 0 to 600 does not show significant effects on the galena recovery, and separation efficiency.
11) Increasing NaCN dosage from 50 to 150 g/ton does not show any remarkable affects on galena grade, recovery, and separation efficiency.
12) According to counter plot, performing flotation by collector dosage above 140 g/ton, frother dosage above 25 g/ton, and ZnSO4 dosage between 200 to 400 g/ton keeps separation efficiency above 55% (Figure 48).

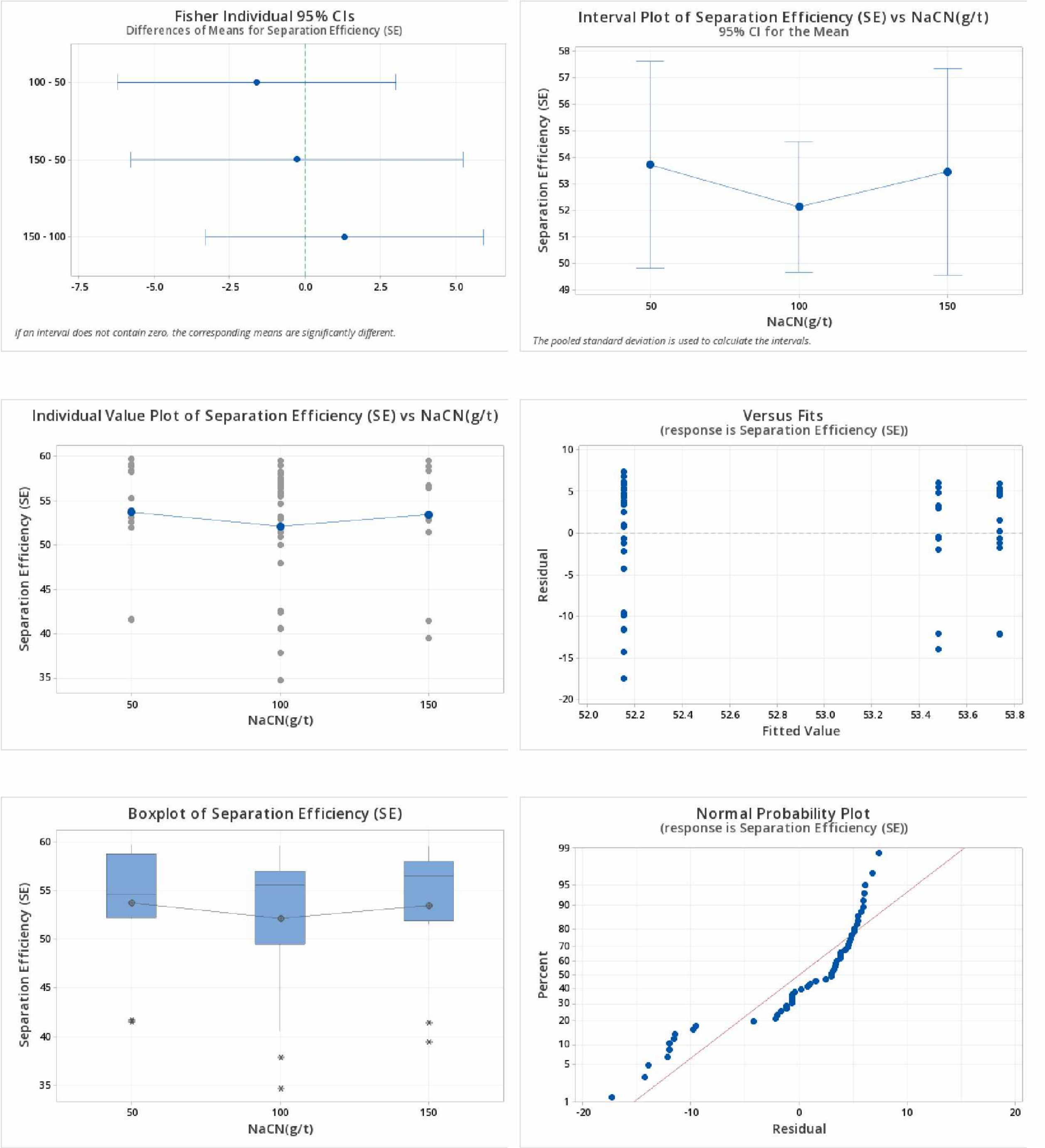


Figure 47. Statistics Graph of the Impact of NaCN on Galena Separation efficiency

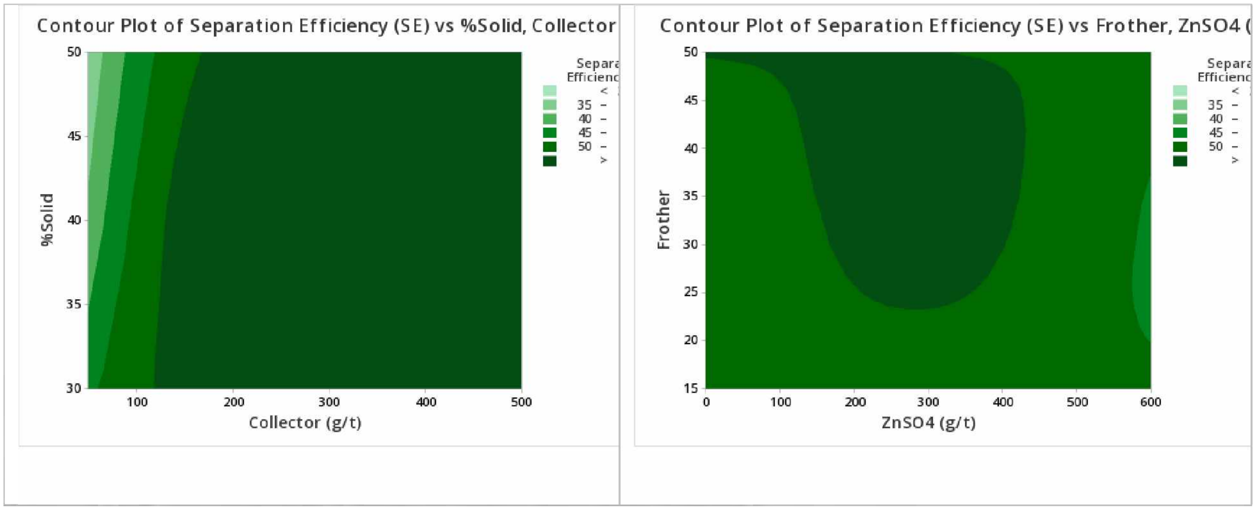


Figure 48. Countor Plot SE vs Solid%, and SE vs Frother

5.4. Optimization Tests for Sphalerite flotation

Another series of optimization tests will perform for sphalerite flotation (Table 28). A series of experiment were desinged using Minitab (3-level Box Behnken method) in 3 different levels of solid%, chemical dosages and different types of chemicals as reagents. Experiments were done in 3 different pH including 10, 11 and 12. 3 different dosage of collector (50, 250, 500 g/t), frother (15, 25, 35g/t), Activator $CuSO_4$ (500, 1000, 1500g/t) was selected. Tests were conducted in 3 different solid % including 30, 40, and 50. Flotation tests were done in 3L Denver cell, similar to the 6.5 section. Then $CuSO_4$ and collector were added with one minute interval, respectively. 20 minutes preparation and mixing were done and MIBC was added. Aeration and froth removal were done for 6 minutes. The purpose of this step is to obtain the best flotation condition to get highest amount of grade and recovery of Sphalerite.

Table 28. Optimization Tests for Sphalerite

Run	%Solid	Collector (g/t)	pH	Frother(g/ton)	CuSO4(g/t)	Zn Grade%	Zn Recovery%	Separation Efficiency(SE)	Yield%
1	30	50	10	15	500	23.17	61.52	55.81	11.55
2	30	50	10	15	1000	25.89	64.36	58.97	10.81
3	30	50	10	15	1500	26.33	65.42	59.97	10.81
4	30	250	11	25	500	25.91	69.30	63.05	11.63
5	30	250	11	25	1000	32.25	74.23	68.97	10.01
6	30	250	11	25	1500	33.17	79.25	73.50	10.39
7	30	500	12	35	500	24.53	62.75	57.24	11.13
8	30	500	12	35	1000	27.55	67.28	61.87	10.62
9	30	500	12	35	1500	28.30	63.78	59.09	9.80
10	40	50	10	15	500	22.17	63.73	57.25	12.50
11	40	50	10	15	1000	23.32	68.89	61.79	12.85
12	40	50	10	15	1500	25.99	67.53	61.62	11.30
13	40	250	11	25	500	24.44	70.12	63.21	12.48
14	40	250	11	25	1000	27.68	68.57	62.98	10.78
15	40	250	11	25	1500	28.42	69.10	63.63	10.58
16	40	500	12	35	500	21.67	64.58	57.72	12.96

Table 28 Continued

17	40	500	12	35	1000	24.70	64.32	58.58	11.33
18	40	500	12	35	1500	25.94	65.19	59.67	10.93
19	50	50	10	15	500	18.32	61.20	53.61	14.53
20	50	50	10	15	1000	21.45	62.12	55.69	12.60
21	50	50	10	15	1500	22.63	62.80	56.67	12.07
22	50	250	11	25	500	20.05	65.12	57.43	14.13
23	50	250	11	25	1000	23.11	67.14	60.32	12.64
24	50	250	11	25	1500	25.18	66.53	60.55	11.49
25	50	500	12	35	500	17.76	59.20	51.82	14.50
26	50	500	12	35	1000	20.64	58.33	52.38	12.29
27	50	500	12	35	1500	21.17	57.74	52.09	11.86

5.4.1. Effect of Solid% on Grade, Recovery, and Separation Efficiency of Sphalerite

a) Grade

In the sphalerite flotation, increasing the solid% has a remarkable negative effect on the concentrate grade. Increasing the solid% from 30 to 50 causes to decrease the grade of concentrate by 6%. That is a significant effect on the product grade and its quality. Increasing grade from 30 to 40% does not cause a significant effect on the product grade because the P-Value for this level is 0.059 that is greater than 0.05. But increasing solid percentage from 40 to 50, and 30 to 50 has a remarkable impact on the product quality. Thus, increasing solid% in the flotation is a critical decision. By increasing the solid% in the flotation process, the mineral particles are at a smaller distance from each other as compared to low pulp density. The probability of bubble-particle collision and attachment will increase. Consequently, the probability of attaching binary particles to the bubble will increase that causes the grade reduction (Table 29, and Figure 49).

Table 29. Mean & Fisher Individual Tests for Differences of Means for Effect of Solid% on Sphalerite Grade

Solid%	N	Mean	StDev	95% CI
30	9	27.46	3.34	(25.59, 29.32)
40	9	24.926	2.319	(23.064, 26.787)
50	9	21.146	2.327	(19.284, 23.007)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
40 - 30	-2.53	1.28	(-5.16, 0.10)	-1.98	0.059
50 - 30	-6.31	1.28	(-8.94, -3.68)	-4.95	0.000
50 - 40	-3.78	1.28	(-6.41, -1.15)	-2.96	0.007

Simultaneous confidence level = 88.11%

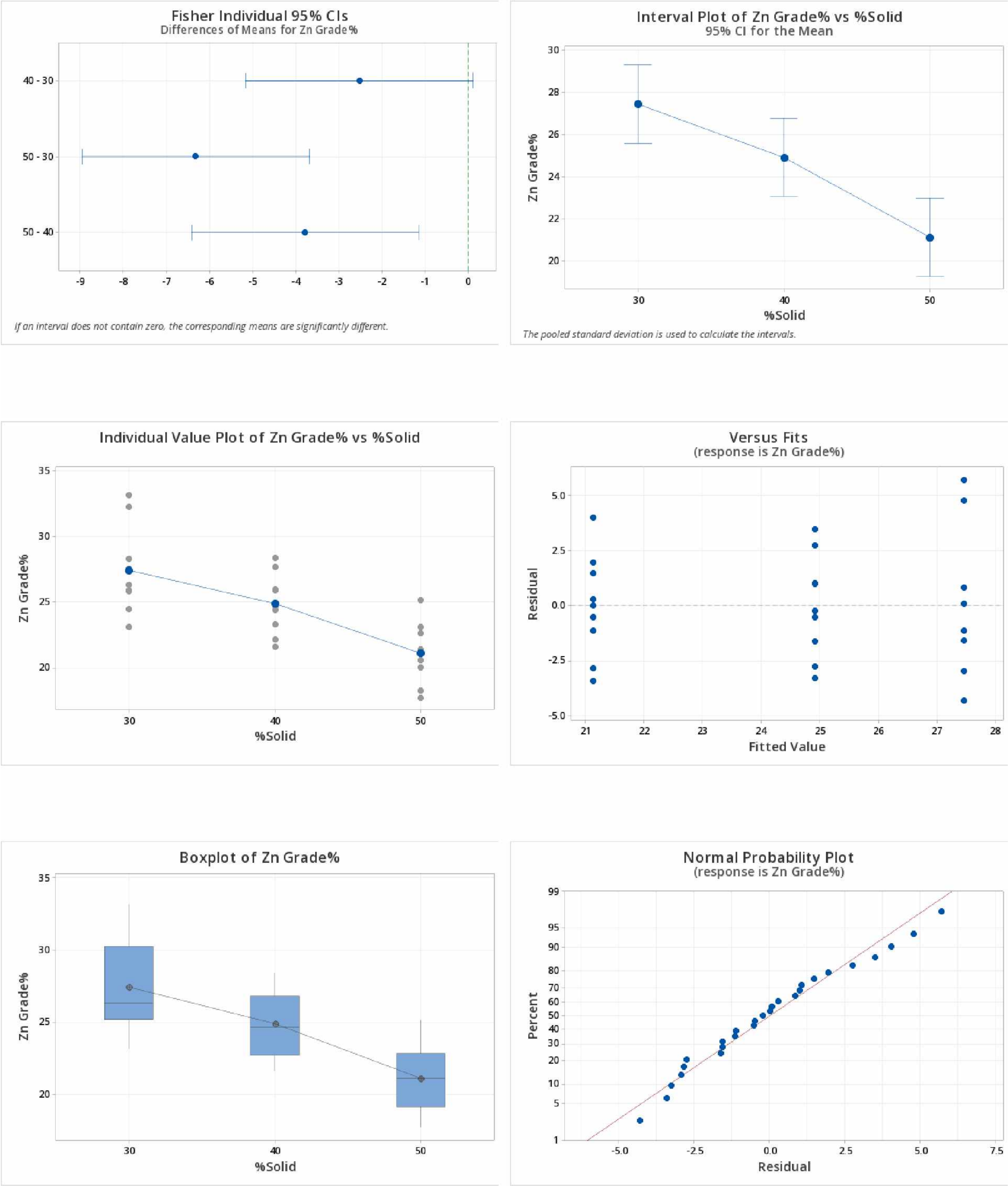


Figure 49. Statistics Graph of the Impact of Solid% on Sphalerite Grade

b) Recovery

Increasing solid percentage has an adverse effect on the flotation recovery. As by increasing the solid% by 20%, the flotation recovery will decrease by 5%. Increasing the solid% from 40 to 50, and 30 to 50 have a remarkable effect on the recovery because the P-Value is less than 0.05. Increasing solid percentage will cause bubble overloading. The bubble overloading will lead to burst the bubble before they arrive to the surface of froth zone in flotation cell. Thus, increasing solid percentage sometimes causes the reduction in the recovery (Table 30, and Figure 50).

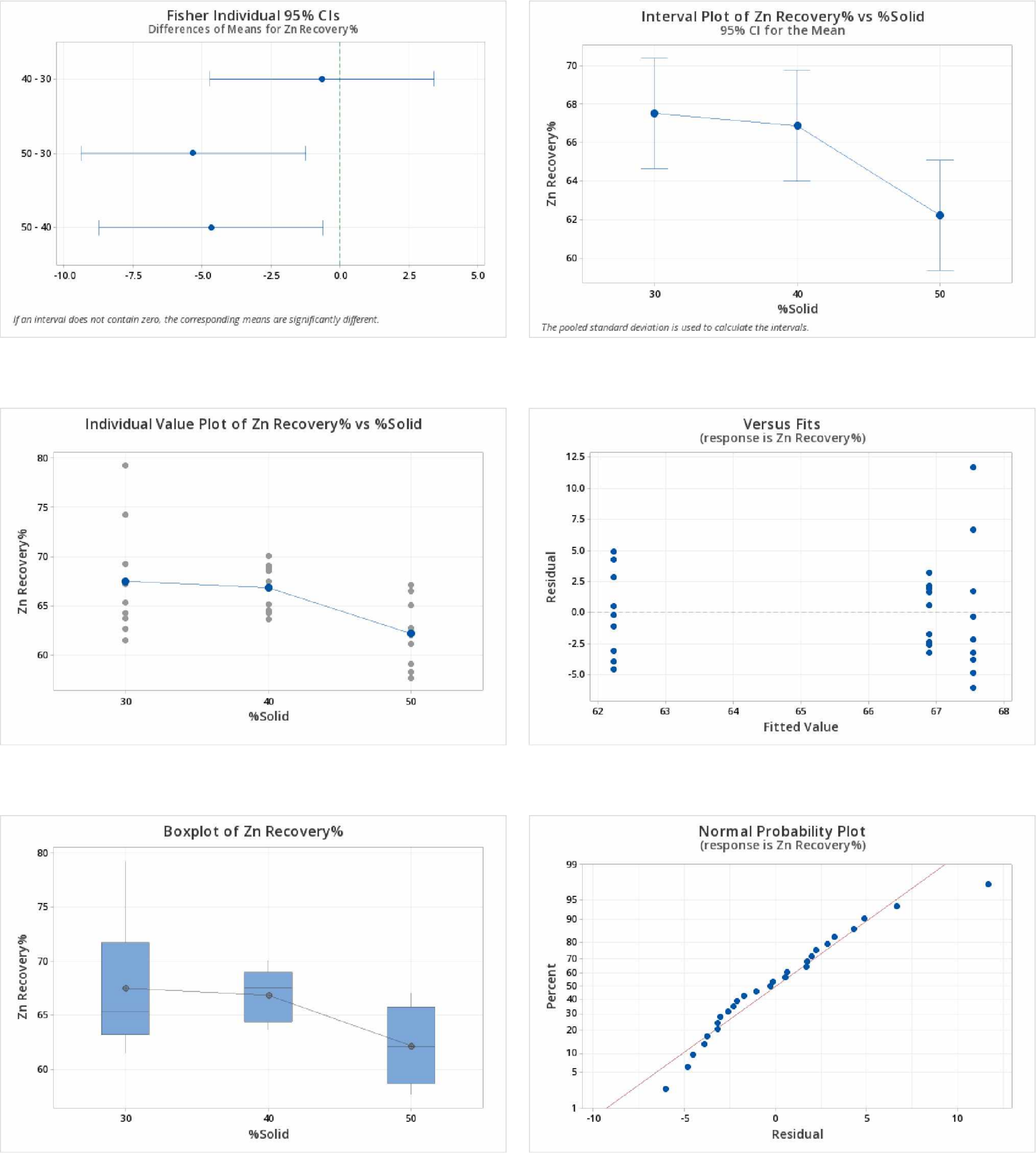
c) Separation efficiency

Increasing solid percentage has an adverse impact on the flotation separation efficiency. By increasing solid percentage from 30 to 50, the mean of separation efficiency decreases from 62.05% to 55.62%. However increasing solid% from 30 to 40 does not show an significant effects on the sphalerite separation efficiency (P-Value is 0.498) , but increasing solid percentage from 40 to 50% , and from 30 to 50 indicate a remarkable effects on the separation efficiency (Table 31).

Table 30. Mean & Fisher Individual Tests for Differences of Means for Effect of Solid% on Sphalerite Recovery (Simultaneous confidence level = 88.11%)

%Solid	**N**	**Mean**	**StDev**	**95% CI**
30	9	67.54	5.84	(64.67, 70.41)
40	9	66.892	2.433	(64.024, 69.761)
50	9	62.24	3.48	(59.37, 65.11)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
40 - 30	-0.65	1.97	(-4.71, 3.41)	-0.33	0.743
50 - 30	-5.30	1.97	(-9.36, -1.24)	-2.70	0.013
50 - 40	-4.65	1.97	(-8.71, -0.59)	-2.37	0.026


Fisher Individual 95% CIs
Differences of Means for Zn Recovery%

If an interval does not contain zero, the corresponding means are significantly different.

Interval Plot of Zn Recovery% vs %Solid
95% CI for the Mean

The pooled standard deviation is used to calculate the intervals.

Individual Value Plot of Zn Recovery% vs %Solid

Versus Fits
(response is Zn Recovery%)

Boxplot of Zn Recovery%

Normal Probability Plot
(response is Zn Recovery%)

Figure 50. Statistics Graph of the Impact of Solid% on Sphalerite Recovery

Table 31. Mean & Fisher Individual Tests for Differences of Means for Effect of Solid% on Sphalerite Separation Efficiency

%Solid	**N**	**Mean**	**StDev**	**95% CI**
30	9	62.05	5.75	(59.23, 64.88)
40	9	60.718	2.461	(57.892, 63.545)
50	9	55.62	3.39	(52.79, 58.44)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
40 - 30	-1.33	1.94	(-5.33, 2.66)	-0.69	0.498
50 - 30	-6.43	1.94	(-10.43, -2.44)	-3.32	0.003
50 - 40	-5.10	1.94	(-9.10, -1.10)	-2.63	0.015

Simultaneous confidence level = 88.11%

5.4.2. Effect of Collector on Grade, Recovery, and Separation Efficiency of Sphalerite

a) Grade

According to Table 32 by increasing collector dosage from 50 to 250 g/t , the concentrate grade of sphalerite increase from 23.25% to 26.69%, but adding more collector than 250g/t causes an adverse effect on the product quality. As by increasing the collector dosage from 250 to 500 g/t the grade of sphalerite decreases by 3%. Because increasing more collector to the flotation cell can help to folate more binary particles. In addition, it reduces the selectivity of sulfide minerals flotation. As other sulfide minerals could be floated as well. Consequently, they will enter to the concentrate and reduce the product quality. Increasing collector dosage from 50 to 250 g/t shows a significant impact on the sphalerite grade because the P-Value is less than 0.05. But increasing the collector dosage from 250 to 500 g/t does not show a remarkable effect on the sphalerite grade, because the P-Value is greater than 0.05 (Table 32, and Figure 52).

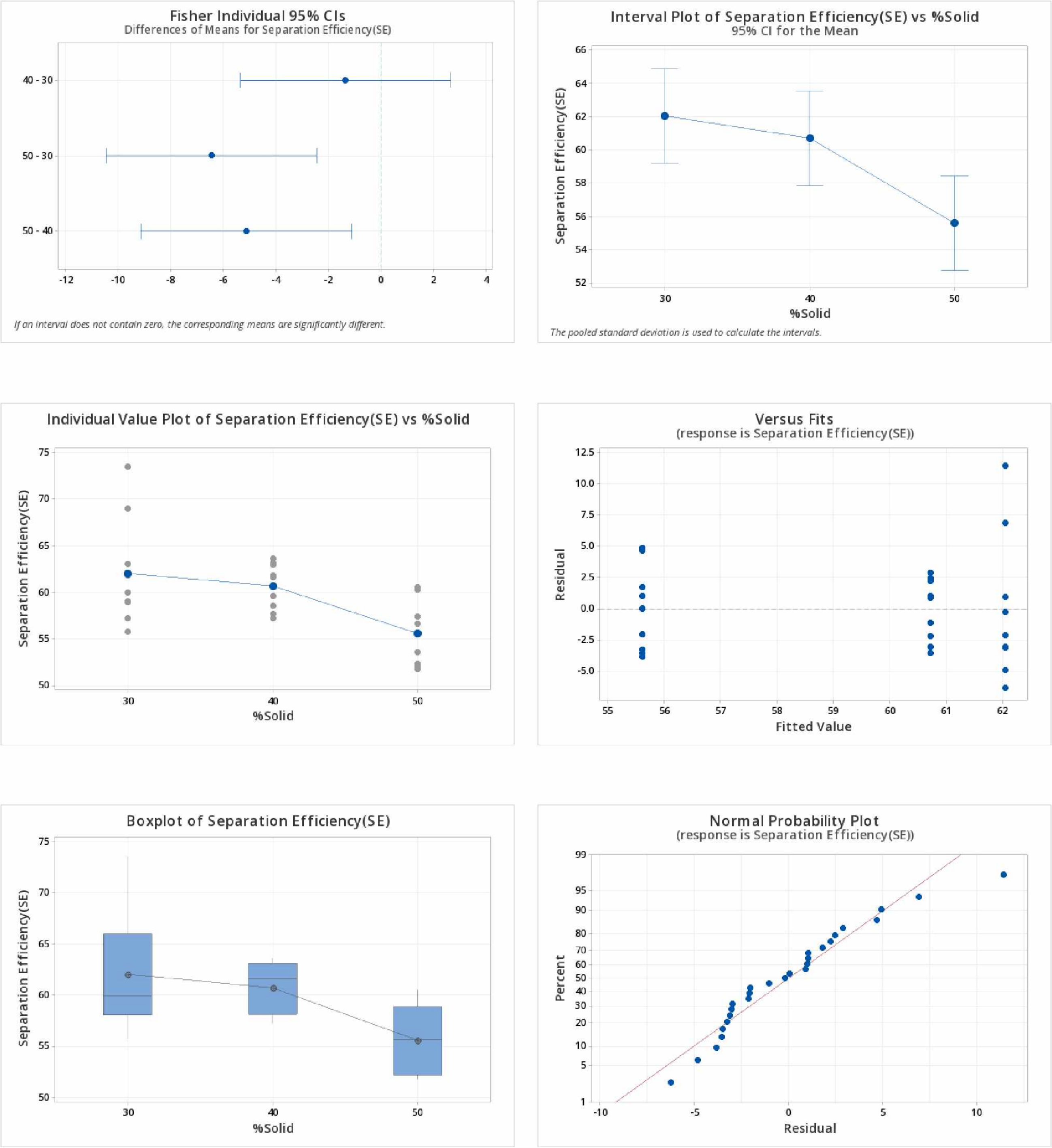


Figure 51. Statistics Graph of the Impact of Solid% on Sphalerite Separation Efficiency

Table 32. Mean & Fisher Individual Tests for Differences of Means for Effect of Collector on Sphalerite Grade

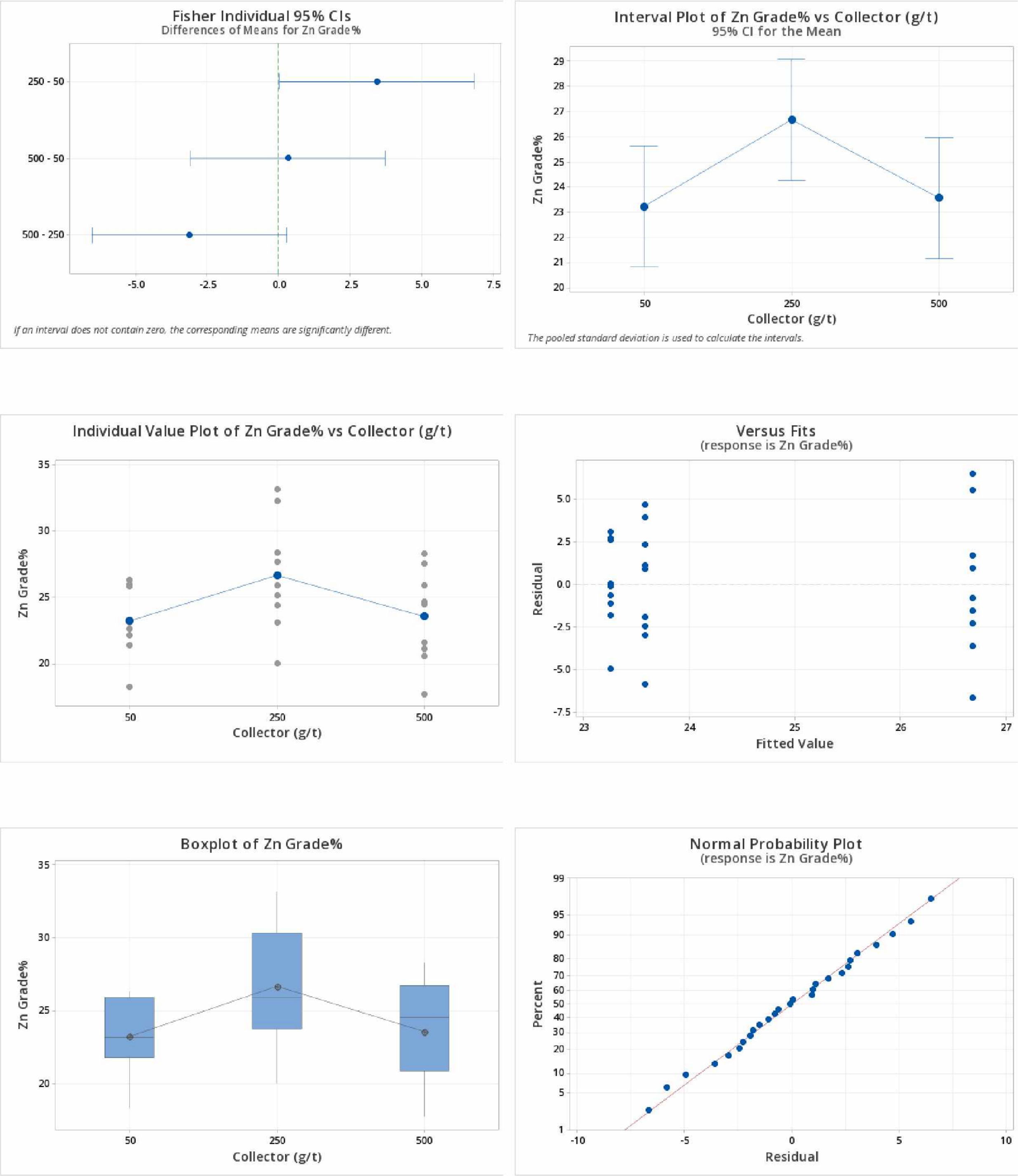
Collector (g/t)	N	Mean	StDev	95% CI
50	9	23.252	2.576	(20.850, 25.654)
250	9	26.69	4.21	(24.29, 29.09)
500	9	23.58	3.50	(21.18, 25.99)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
250 - 50	3.44	1.65	(0.04, 6.83)	2.09	0.048
500 - 50	0.33	1.65	(-3.06, 3.73)	0.20	0.842
500 - 250	-3.11	1.65	(-6.50, 0.29)	-1.89	0.071

Simultaneous confidence level = 88.11%

b) Recovery

Increasing collector dosage from 50 to 250 g/t improves the sphalerite recovery at the concentrate. The recovery increased from 64.17 to 69.93%. Because by increasing the collector dosage the probability of binary particles or sphalerite locked particles will increase so the recovery will increase. But when collector dosage increases from 250 to 500 g/t, it has a negative effect on the recovery. As recovery decrease from 69.93 to 62.57%. By increasing the collector dosage from a certain amount, the probability of the formation of micelle or admicelle will increase. So, collector will lose its own performance, and application, and cannot make the sphalerite surface hydrophobic. The P-Values for both collector level 50 to 250, and 250 to 500 g/t is less than 0.05 that shows the collector dosage is playing an important role on sphalerite recovery. The highest amount of recovery was obtained with collector dosage of 250 g/t (Table 33, and Figure 53).



Fisher Individual 95% CIs

Differences of Means for Zn Grade%

If an interval does not contain zero, the corresponding means are significantly different.

Interval Plot of Zn Grade% vs Collector (g/t)

95% CI for the Mean

The pooled standard deviation is used to calculate the intervals.

Individual Value Plot of Zn Grade% vs Collector (g/t)

Versus Fits

(response is Zn Grade%)

Boxplot of Zn Grade%

Normal Probability Plot

(response is Zn Grade%)

Figure 52. Statistics Graph of the Impact of Collector on Sphalerite Grade

Table 33. Mean & Fisher Individual Tests for Differences of Means for Effect of Collector on Sphalerite Recovery

Collector (g/t)	**N**	**Mean**	**StDev**	**95% CI**
50	9	64.174	2.676	(61.749, 66.600)
250	9	69.93	4.34	(67.50, 72.35)
500	9	62.57	3.36	(60.15, 65.00)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
250 - 50	5.75	1.66	(2.32, 9.19)	3.46	0.002
500 - 50	-1.60	1.66	(-5.03, 1.83)	-0.96	0.345
500 - 250	-7.35	1.66	(-10.79, -3.92)	-4.42	0.000

Simultaneous confidence level = 88.11%

c) Separation efficiency

According to Table 34 by increasing the collector dosage from 50 to 250 g/t the separation efficiency will increase by almost 5%, but increasing the collector dosage from 250 to 500 g/t has an adverse effect on the flotation performance and reduce separation efficiency by 7%. The P-Values for both level50-250, and 250-500 are less than 0.05 that indicate collector dosage has a significant effect on the separation efficiency of sphalerite flotation. The highest separation efficiency was obtained with the collector dosage around 250 g/t (Table 34, and Figure 54).

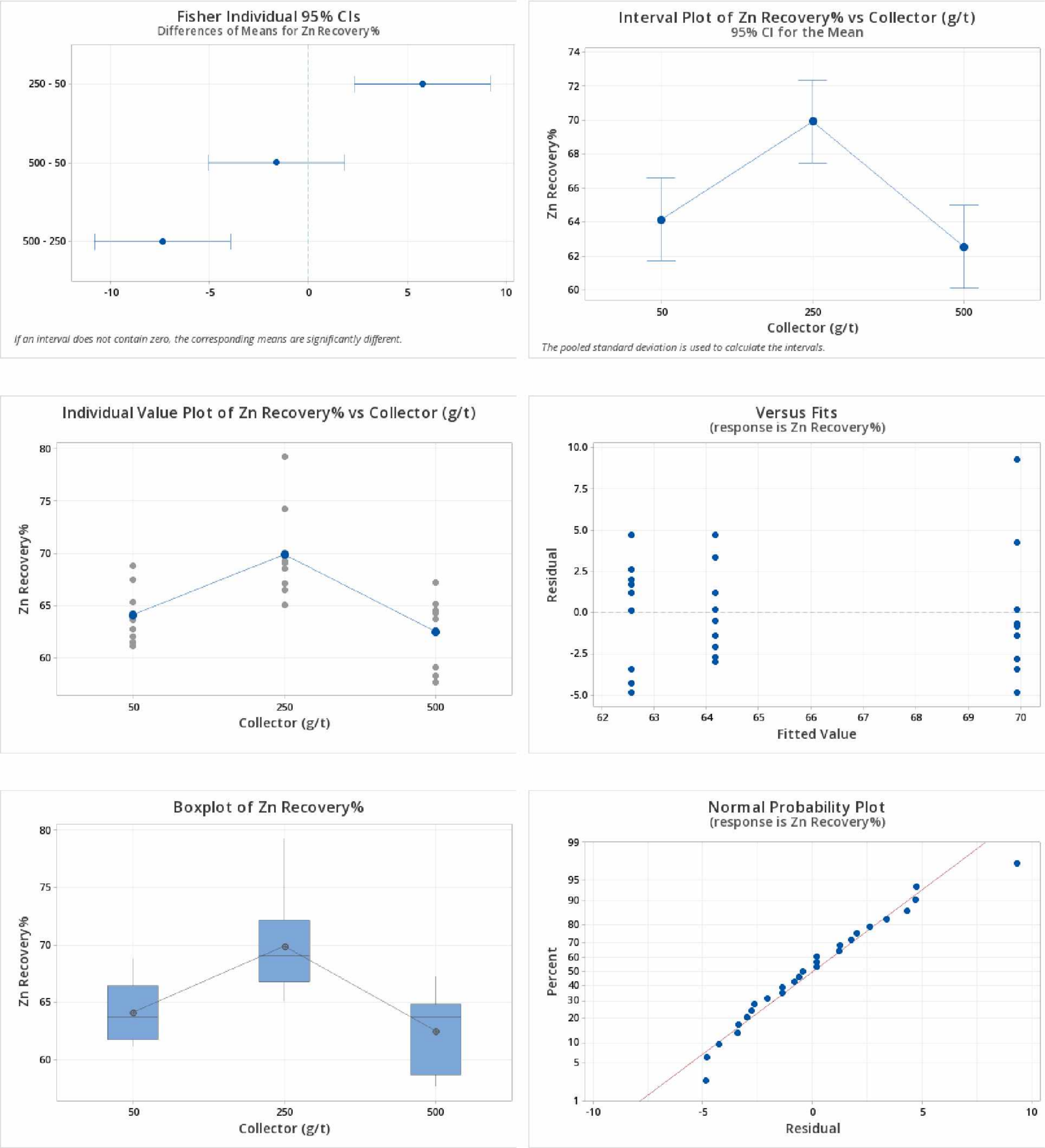


Figure 53. Statistics Graph of the Impact of Collector on Sphalerite Recovery

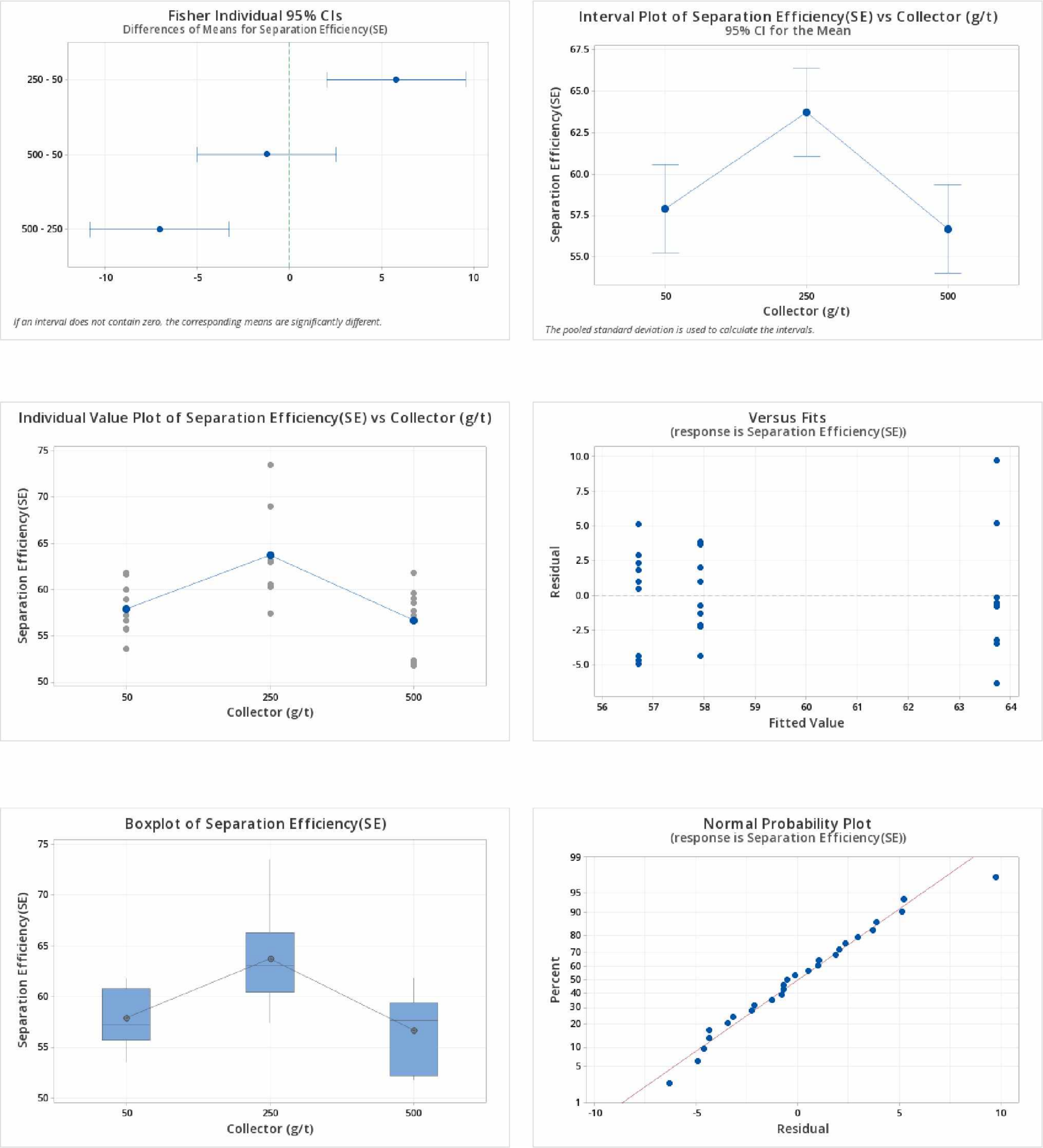


Figure 54. Statistics Graph of the Impact of Collector on Sphalerite Separation Efficiency

Table 34. Mean & Fisher Individual Tests for Differences of Means for Effect of Collector on Sphalerite Separation Efficiency

Collector (g/t)	**N**	**Mean**	**StDev**	**95% CI**
50	9	57.932	2.830	(55.269, 60.594)
250	9	63.74	4.82	(61.07, 66.40)
500	9	56.72	3.71	(54.06, 59.38)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
250 - 50	5.81	1.82	(2.04, 9.57)	3.18	0.004
500 - 50	-1.21	1.82	(-4.98, 2.55)	-0.66	0.513
500 - 250	-7.02	1.82	(-10.78, -3.25)	-3.85	0.001

Simultaneous confidence level = 88.11%

5.4.3. Effect of pH on Grade, Recovery, and Separation Efficiency of Sphalerite

a) Grade

The pH is another important parameter on the sphalerite flotation. The result shows by increasing the pH, from 10 to 11 the grade of sphalerite concentration will increase from 23.25%, but by increasing the pH from 11 to 12 the product grade, and quality will decrease by 3%. The P-Values shows increasing the pH from 10 to 11, and 11 to 12 has a significant effect on the sphalerite grade on the flotation concentrate, because P-Values for both level is less than 0.05 (Table 35, and Figure 55).

b) Recovery

By increasing the pH from 10 to 11, the recovery will increase by 5%. But when the pH increases from 11 to 12, the recovery will decrease by 7%. Changing the pH has a significant impact on the recovery of the sphalerite in both 10-11, and 11-12 levels. The P-value for both levels are 0.002, and 0.00, respectively. Both of them are less than 0.05. The highest recovery was gained at pH around 11 (Table 36, and Figure 56).

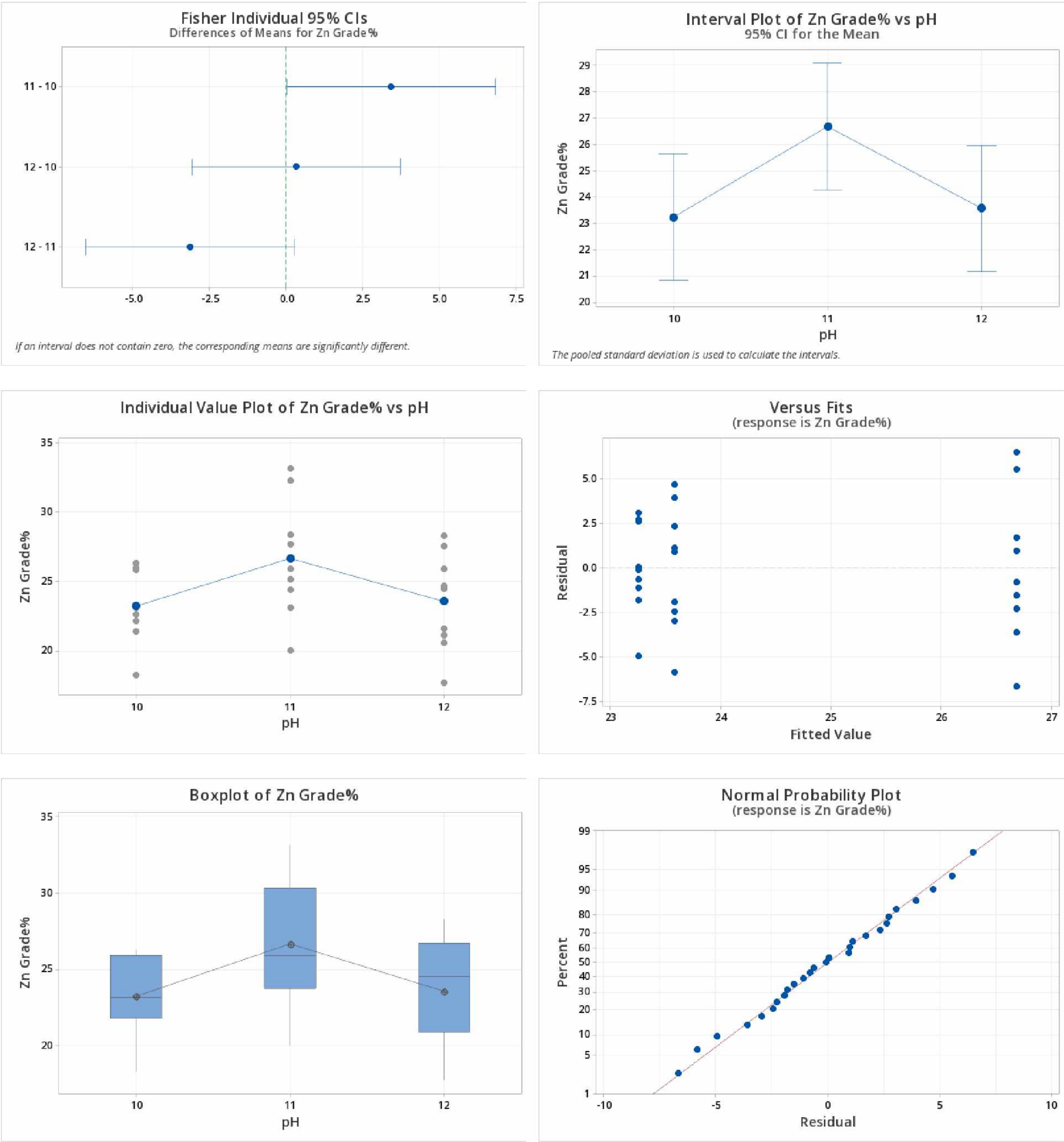


Figure 55. Statistics Graph of the Impact of pH on Sphalerite Grade

Table 35. Mean & Fisher Individual Tests for Differences of Means for Effect of pH on Sphalerite Grade

pH	N	Mean	StDev	95% CI
10	9	23.252	2.576	(20.850, 25.654)
11	9	26.69	4.21	(24.29, 29.09)
12	9	23.58	3.50	(21.18, 25.99)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
11 - 10	3.44	1.65	(0.04, 6.83)	2.09	0.048
12 - 10	0.33	1.65	(-3.06, 3.73)	0.20	0.842
12 - 11	-3.11	1.65	(-6.50, 0.29)	-1.89	0.071

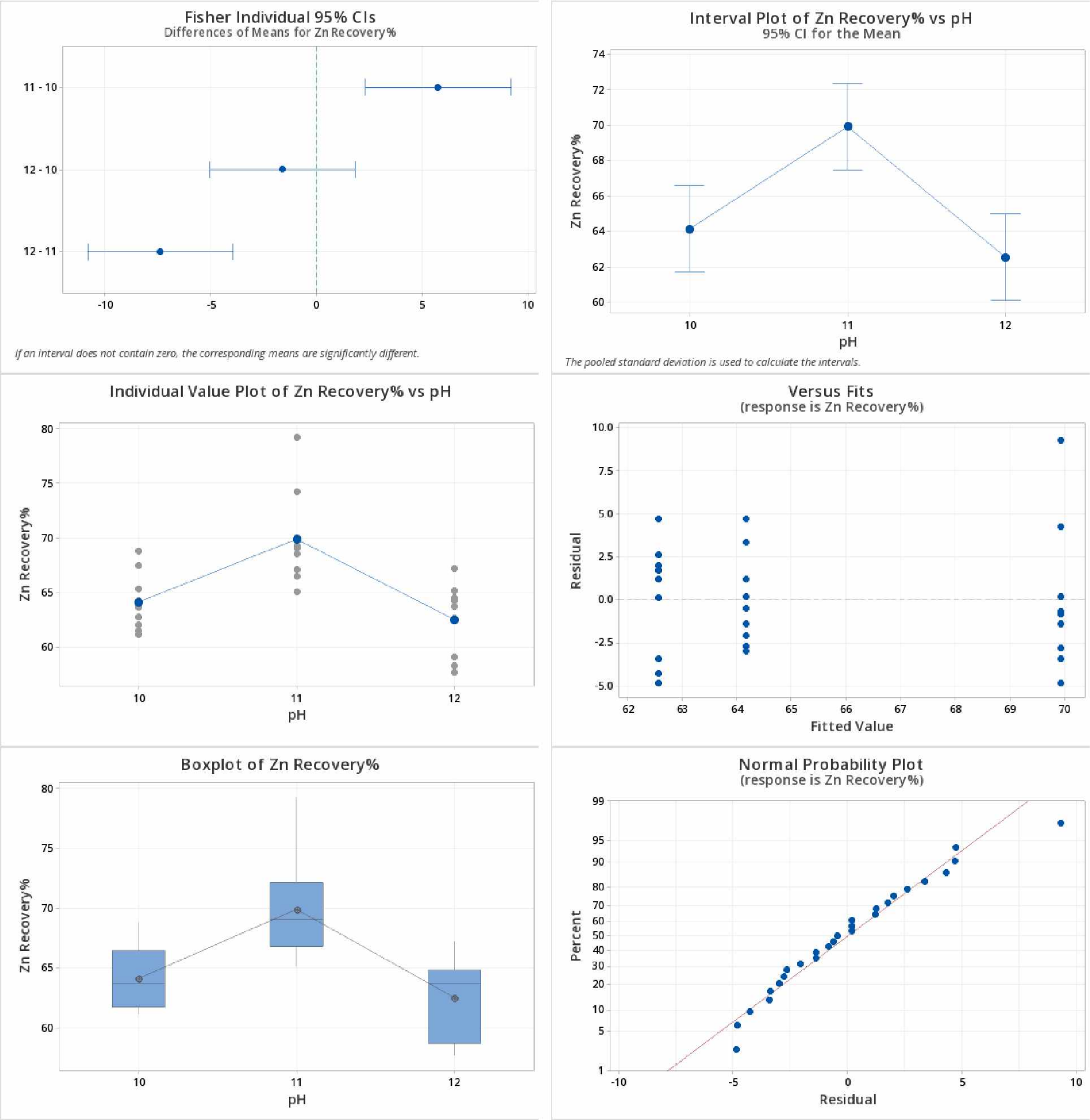
Simultaneous confidence level = 88.11%

Table 36. Mean & Fisher Individual Tests for Differences of Means for Effect of pH on Sphalerite Recovery

pH	N	Mean	StDev	95% CI
10	9	64.174	2.676	(61.749, 66.600)
11	9	69.93	4.34	(67.50, 72.35)
12	9	62.57	3.36	(60.15, 65.00)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
11 - 10	5.75	1.66	(2.32, 9.19)	3.46	0.002
12 - 10	-1.60	1.66	(-5.03, 1.83)	-0.96	0.345
12 - 11	-7.35	1.66	(-10.79, -3.92)	-4.42	0.000

Simultaneous confidence level = 88.11%


Fisher Individual 95% CIs
Differences of Means for Zn Recovery%

If an interval does not contain zero, the corresponding means are significantly different.

Interval Plot of Zn Recovery% vs pH
95% CI for the Mean

The pooled standard deviation is used to calculate the intervals.

Individual Value Plot of Zn Recovery% vs pH

Versus Fits
(response is Zn Recovery%)

Boxplot of Zn Recovery%

Normal Probability Plot
(response is Zn Recovery%)

Figure 56. Statistics Graph of the Impact of pH on Sphalerite Recovery

c) Separation efficiency

Separation efficiency was affected by the pH. By increasing the pH value from 10 to 11, SE increased from 57.93 to 63.74. By increasing the pH from 11 to 12 the SE decreased from 63.74 to 56.72%. The P-Values show pH values has a significant effect on the flotation performance.

The P-value for both levels 10-11, and 11-12 are 0.004, and 0.001 that is less than 0.05. the highest SE was gained at pH 11(Table 37, and Figure 57).

Table 37. Mean & Fisher Individual Tests for Differences of Means for Effect of pH on Sphalerite Separation Efficiency

pH	N	Mean	StDev	95% CI
10	9	57.932	2.830	(55.269, 60.594)
11	9	63.74	4.82	(61.07, 66.40)
12	9	56.72	3.71	(54.06, 59.38)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
11 - 10	5.81	1.82	(2.04, 9.57)	3.18	0.004
12 - 10	-1.21	1.82	(-4.98, 2.55)	-0.66	0.513
12 - 11	-7.02	1.82	(-10.78, -3.25)	-3.85	0.001

Simultaneous confidence level = 88.11%

5.4.4. Effect of Frother on Grade, Recovery, and Separation Efficiency of Sphalerite

a) Grade

The results indicate by increasing frother dosage from 15 to 25 g/t, the sphalerite grade increased in the concentrate from 23.25 to 23.58%. Increasing the frother to the certain amount can increase the bubble stability, and bubble size, in addition the number of bubbles will increase in the flotation cell. Thus, it causes more bubble surface area to carry higher amount of the sphalerite to the froth zone. However, adding extra dosage of frother can have a negative effect on sphalerite grade in the concentrate. Because it can have a huge effect on the bubble size. The bubble size become so small that they lose their capability to carry the sphalerite particles to the froth zone. As increasing the froth dosage from 25 to 35 was reduced sphalerite grade in the concentrate by almost 3%. The

P-Values show increasing the frother dosage from 15 to 25 g/t has a remarkable effect on the grade, while increasing the frother dosage from 25-35 does not have significant effect on the sphalerite grade on the concentrate (Table 38, and Figure 58).

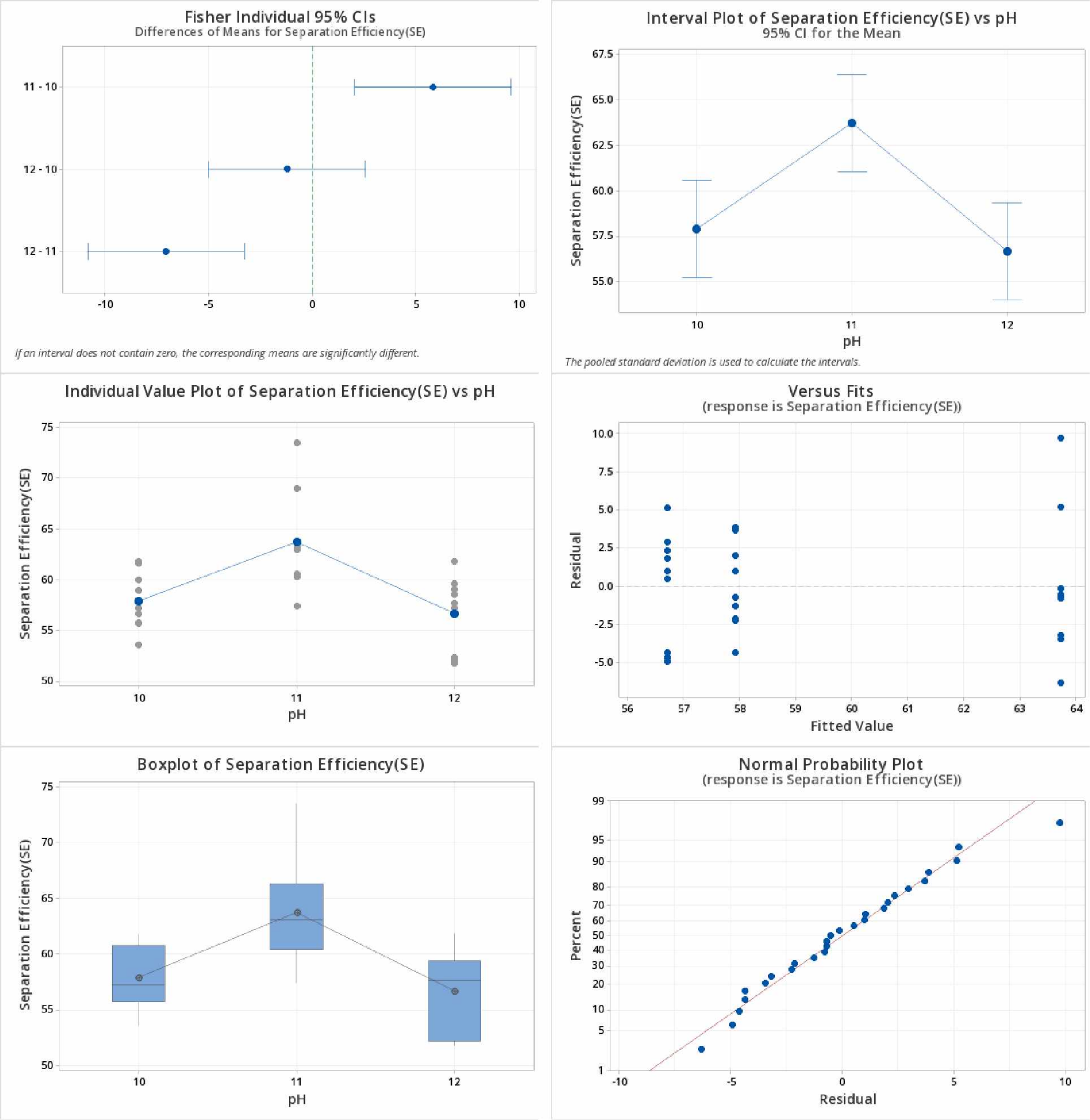


Figure 57. Statistics Graph of the Impact of pH on Sphalerite Separation Efficiency

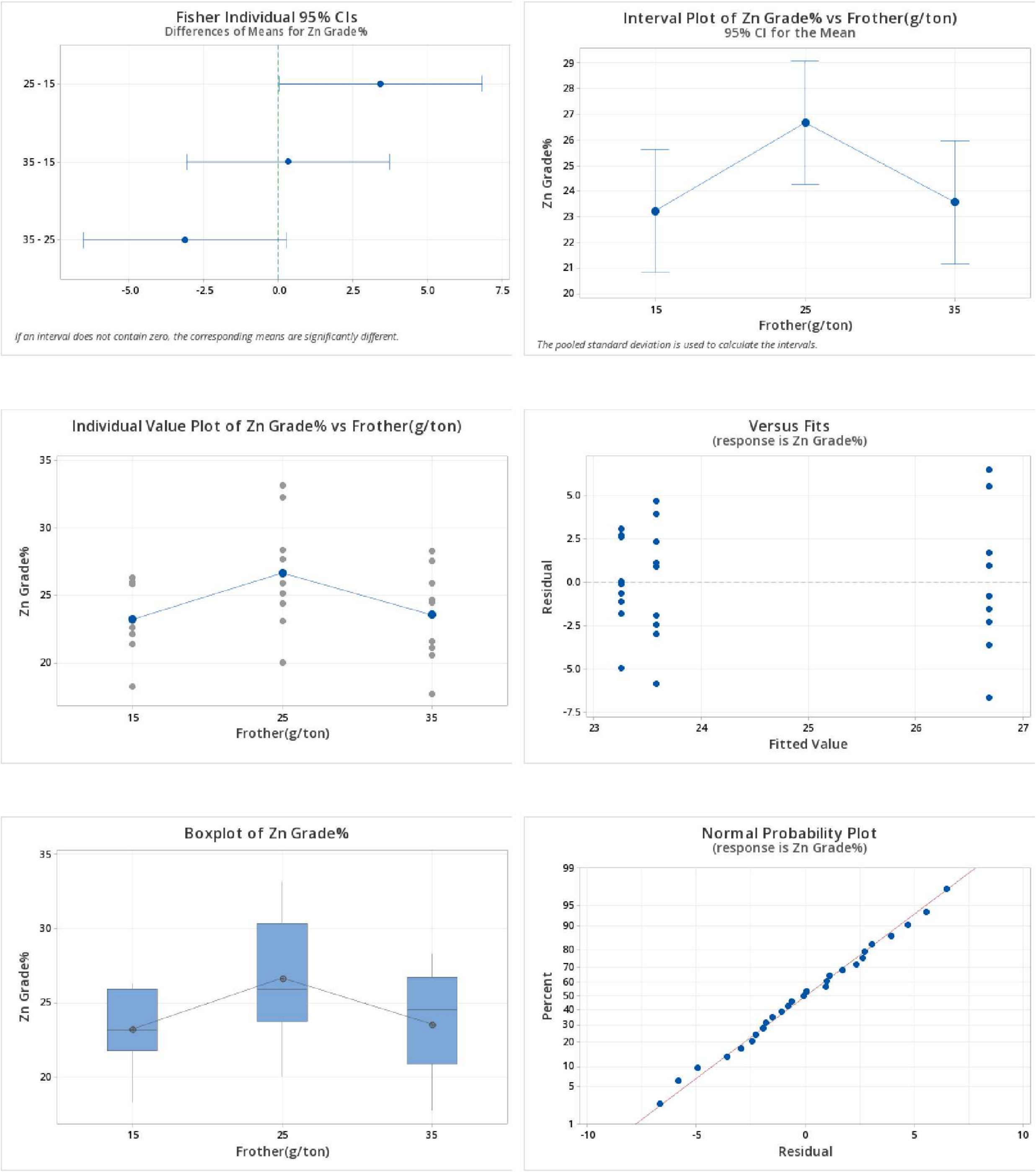


Figure 58. Statistics Graph of the Impact of Frother on Sphalerite Grade

Table 38. Mean & Fisher Individual Tests for Differences of Means for Effect of Frother on Sphalerite Grade

Frother(g/ton)	N	Mean	StDev	95% CI
15	9	23.252	2.576	(20.850, 25.654)
25	9	26.69	4.21	(24.29, 29.09)
35	9	23.58	3.50	(21.18, 25.99)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
25 - 15	3.44	1.65	(0.04, 6.83)	2.09	0.048
35 - 15	0.33	1.65	(-3.06, 3.73)	0.20	0.842
35 - 25	-3.11	1.65	(-6.50, 0.29)	-1.89	0.071

Simultaneous confidence level = 88.11%

b) Recovery

Increasing frother dosage from 15 to 25 g/t has aa positive effect on the sphalerite recovery. It is increase sphalerite recovery by 5.7%. However, when the dosage of frother reach to 35 g/t, the recovery decrease considerably. The recovery decreased by 7.4%. The P-Value for level 15-25, and 25-35 g/t are 0.002 and 0.000. Both P-value is less than 0.05. they show remarkable differences in the recoveries. Thus, attention to dosage of frother in sphalerite flotation can improve the recovery. The highest recoveries were achieved by using 2.5 g/t frother(Table 39, and Figure 59).

c) Separation efficiency

The SE results show by increasing frother dosage from 15 to 25, SE increase from 57.93% to 63.74%. The P-Value is 0.004 that is less than 0.05. So, increasing frother dosage to 25 g/t has a positive remarkable effect on the SE. On the other hand, increasing frother dosage from 25 to 35 g/t has a negative effect on the SE. The P-Value for this level is 0.001that is less than 0.05. It indicates increasing frother dosage from 25 to 35 g/t has a remarkable negative effect on the flotation performance (Table 40, and Figure 60).

Table 39. Mean & Fisher Individual Tests for Differences of Means for Effect of Frother on Sphalerite Recovery

Frother(g/ton)	**N**	**Mean**	**StDev**	**95% CI**
15	9	64.174	2.676	(61.749, 66.600)
25	9	69.93	4.34	(67.50, 72.35)
35	9	62.57	3.36	(60.15, 65.00)

Fisher Individual Tests for Differences of Means

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
25 - 15	5.75	1.66	(2.32, 9.19)	3.46	0.002
35 - 15	-1.60	1.66	(-5.03, 1.83)	-0.96	0.345
35 - 25	-7.35	1.66	(-10.79, -3.92)	-4.42	0.000

Simultaneous confidence level = 88.11%

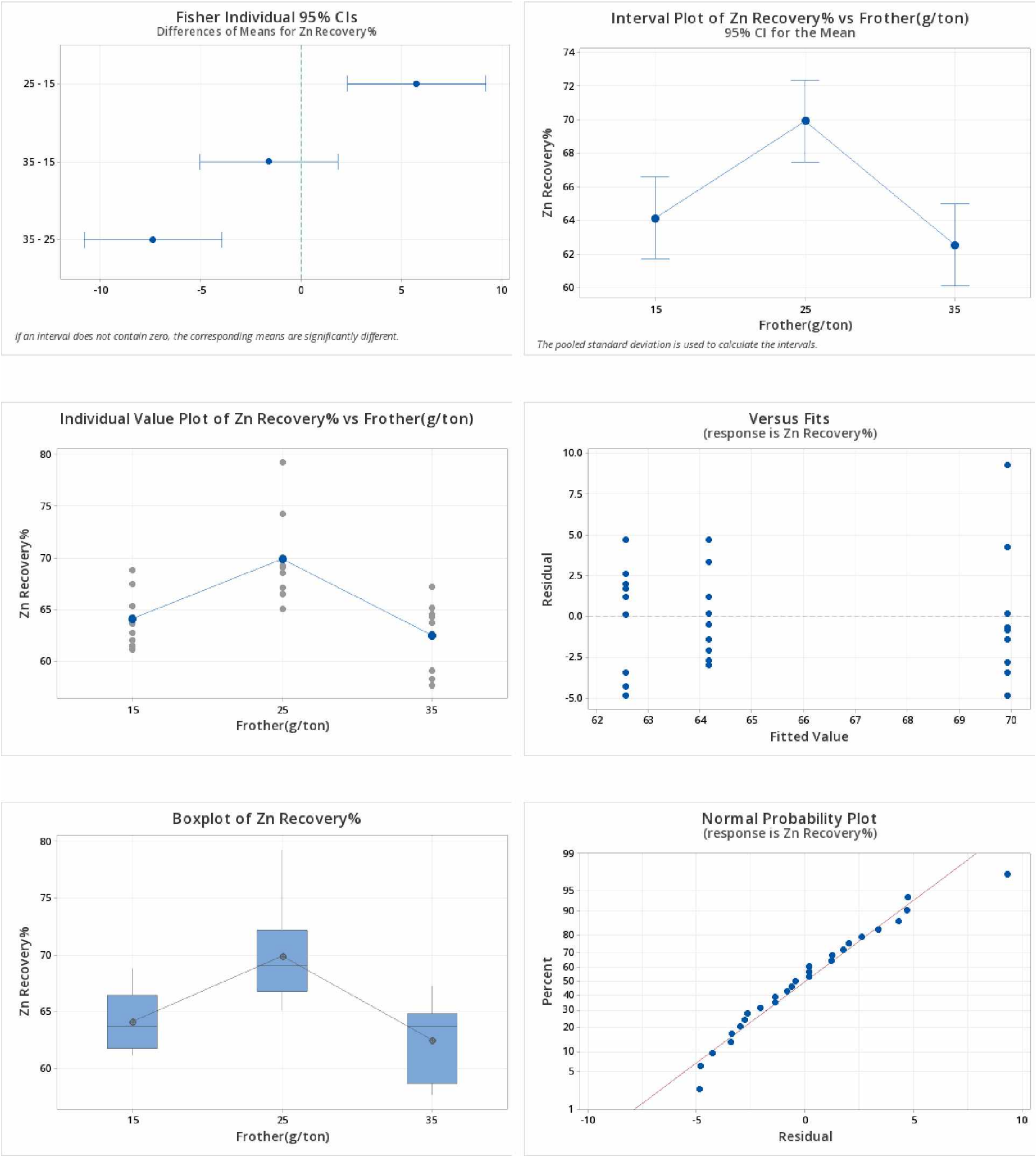


Figure 59. Statistics Graph of the Impact of Frother on Sphalerite Recovery

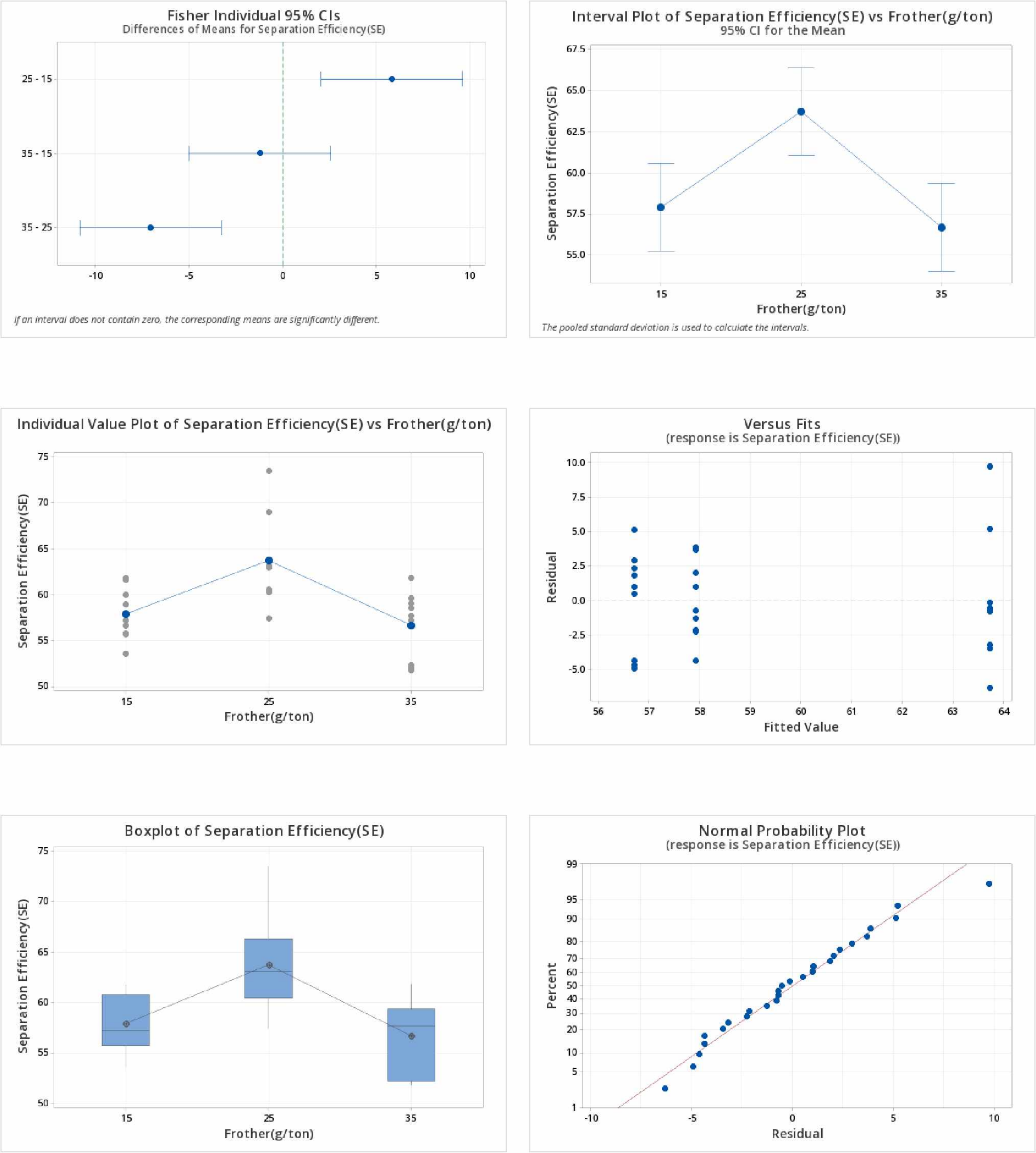


Figure 60. Statistics Graph of the Impact of Frother on Sphalerite Separation Efficiency

Table 40. Mean & Fisher Individual Tests for Differences of Means for Effect of Frother on Sphalerite Separation Efficiency

Frother(g/ton)	**N**	**Mean**	**StDev**	**95% CI**
15	9	57.932	2.830	(55.269, 60.594)
25	9	63.74	4.82	(61.07, 66.40)
35	9	56.72	3.71	(54.06, 59.38)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
25 - 15	5.81	1.82	(2.04, 9.57)	3.18	0.004
35 - 15	-1.21	1.82	(-4.98, 2.55)	-0.66	0.513
35 - 25	-7.02	1.82	(-10.78, -3.25)	-3.85	0.001

Simultaneous confidence level = 88.11%

5.4.5. Effect of CuSO4 on Grade, Recovery, and Separation Efficiency of Sphalerite

a) Grade

CuSO4 was added to the flotation cell during sphalerite flotation. It is an activator that can active sphalerite surface and improve the flotation performance. In the experiments, by increasing the CuSO4 dosage from 500 to 1000 g/t, the product grade was improved from 22% to 25.18%. In addition, increasing CuSO4 dosage from 1000-1500 g/t increased the grade by 1%. The P-Value show the grade differences for level 500-1000 as well as 1000-1500 are not remarkable because the P-Value for both levels are greater than 0.05. However, the P-Value for level 500-1500 is 0.011 that shows, increasing CuSO4 dosage from 500 to1500 has a remarkable impact on the flotation performance. The highest grade was achieved by 1500 g/t of CuSO4 (Table 41, Figure 61).

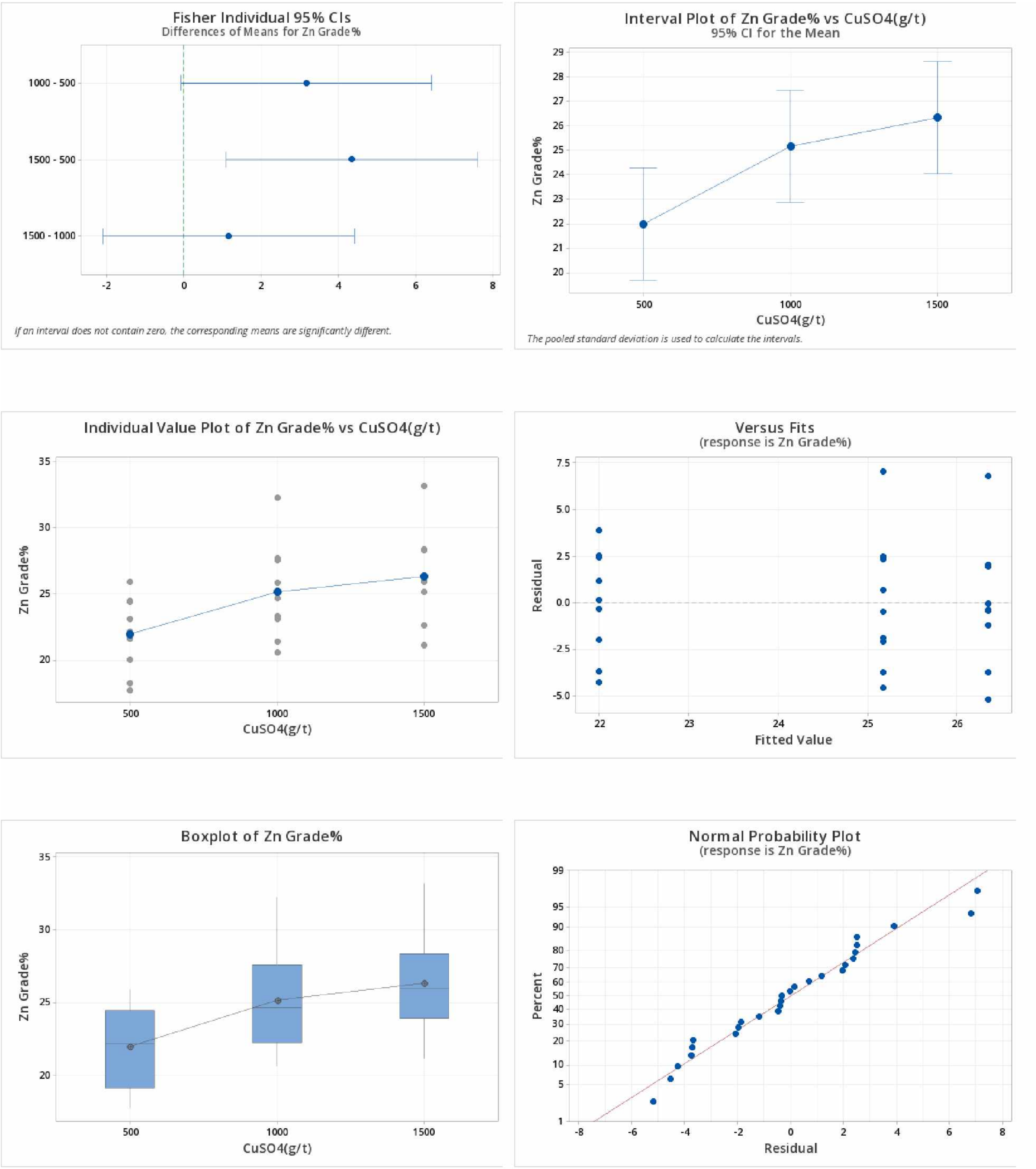


Figure 61. Statistics Graph of the Impact of CuSO4 on Sphalerite Grade

Table 41. Mean & Fisher Individual Tests for Differences of Means for Effect of CuSO4 on Sphalerite Grade

CuSO4(g/t)	**N**	**Mean**	**StDev**	**95% CI**
500	9	22.002	2.839	(19.711, 24.294)
1000	9	25.18	3.62	(22.89, 27.47)
1500	9	26.35	3.48	(24.06, 28.64)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
1000 - 500	3.17	1.57	(-0.07, 6.42)	2.02	0.054
1500 - 500	4.35	1.57	(1.10, 7.59)	2.77	0.011
1500 - 1000	1.17	1.57	(-2.07, 4.41)	0.75	0.463

Simultaneous confidence level = 88.11%

Table 42. Mean & Fisher Individual Tests for Differences of Means for Effect of CuSO4 on Sphalerite Recovery

CuSO4(g/t)	**N**	**Mean**	**StDev**	**95% CI**
500	9	64.17	3.63	(60.90, 67.44)
1000	9	66.14	4.54	(62.87, 69.41)
1500	9	66.37	5.82	(63.10, 69.64)

Difference of Levels	**Difference of Means**	**SE of Difference**	**95% CI**	**T-Value**	**Adjusted P-Value**
1000 - 500	1.97	2.24	(-2.65, 6.59)	0.88	0.388
1500 - 500	2.20	2.24	(-2.42, 6.82)	0.98	0.335
1500 - 1000	0.23	2.24	(-4.39, 4.85)	0.10	0.918

Simultaneous confidence level = 88.11%

b) Recovery

However, the highest recovery was achieved by 1500 g/t of CuSO4, but increasing CuSO4 dosage from 500 to 1500 does not have any remarkable effect on the sphalerite recovery. In all 3 different levels P-Values are greater than 0.05 that shows non-effectiveness of the CuSO4 dosage on the sphalerite recovery (Table 42, and Figure 62).

c) SE

According to table by increasing CuSO4 dosage from 500 to 1500, the SE increased from 57.46 to 60.78%. But the amount of changes is not remarkable because all P-values for all levels are greater than 0.05(Table 43, and Figure 63).

Table 43. Mean & Fisher Individual Tests for Differences of Means for Effect of CuSO4 on Sphalerite Separation Efficiency

CuSO4(g/t)	N	Mean	StDev	95% CI
500	9	57.46	3.77	(54.15, 60.77)
1000	9	60.17	4.68	(56.86, 63.49)
1500	9	60.76	5.78	(57.44, 64.07)

Difference of Levels	Difference of Means	SE of Difference	95% CI	T-Value	Adjusted P-Value
1000 - 500	2.71	2.27	(-1.97, 7.40)	1.19	0.244
1500 - 500	3.30	2.27	(-1.39, 7.98)	1.45	0.160
1500 - 1000	0.58	2.27	(-4.10, 5.27)	0.26	0.800

Simultaneous confidence level = 88.11%

5.5. Taguchi Analysis: Zn Grade%, Recovery%, and Separation Efficiency versus Solid%, Collector (g/t), pH, Frother(g/ton), CuSO4(g/t)

In the next step, the data was analyzed using Taguchi method to examine which parameter is the most effective parameters on the grade, recovery, and separation efficiency.

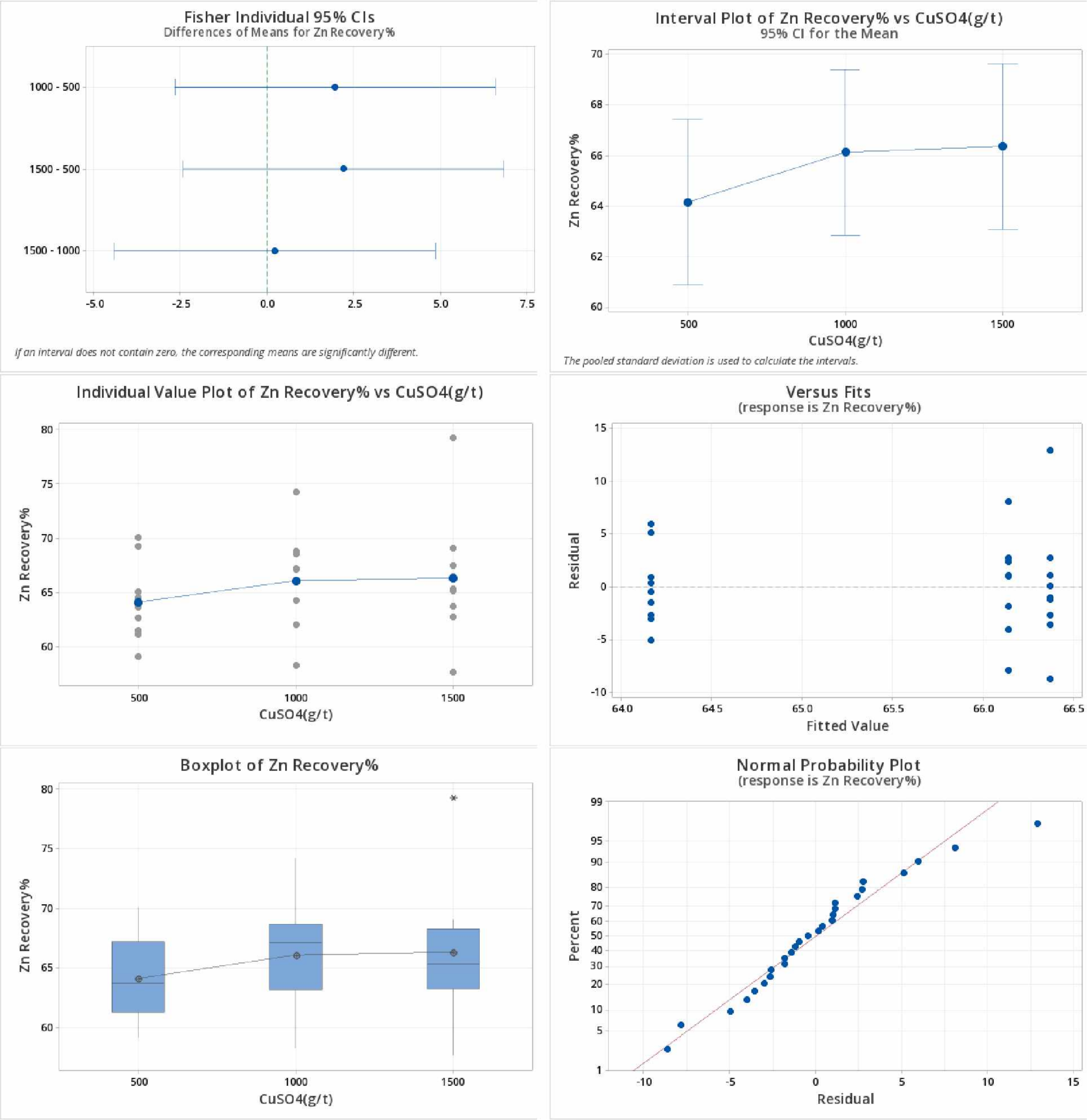
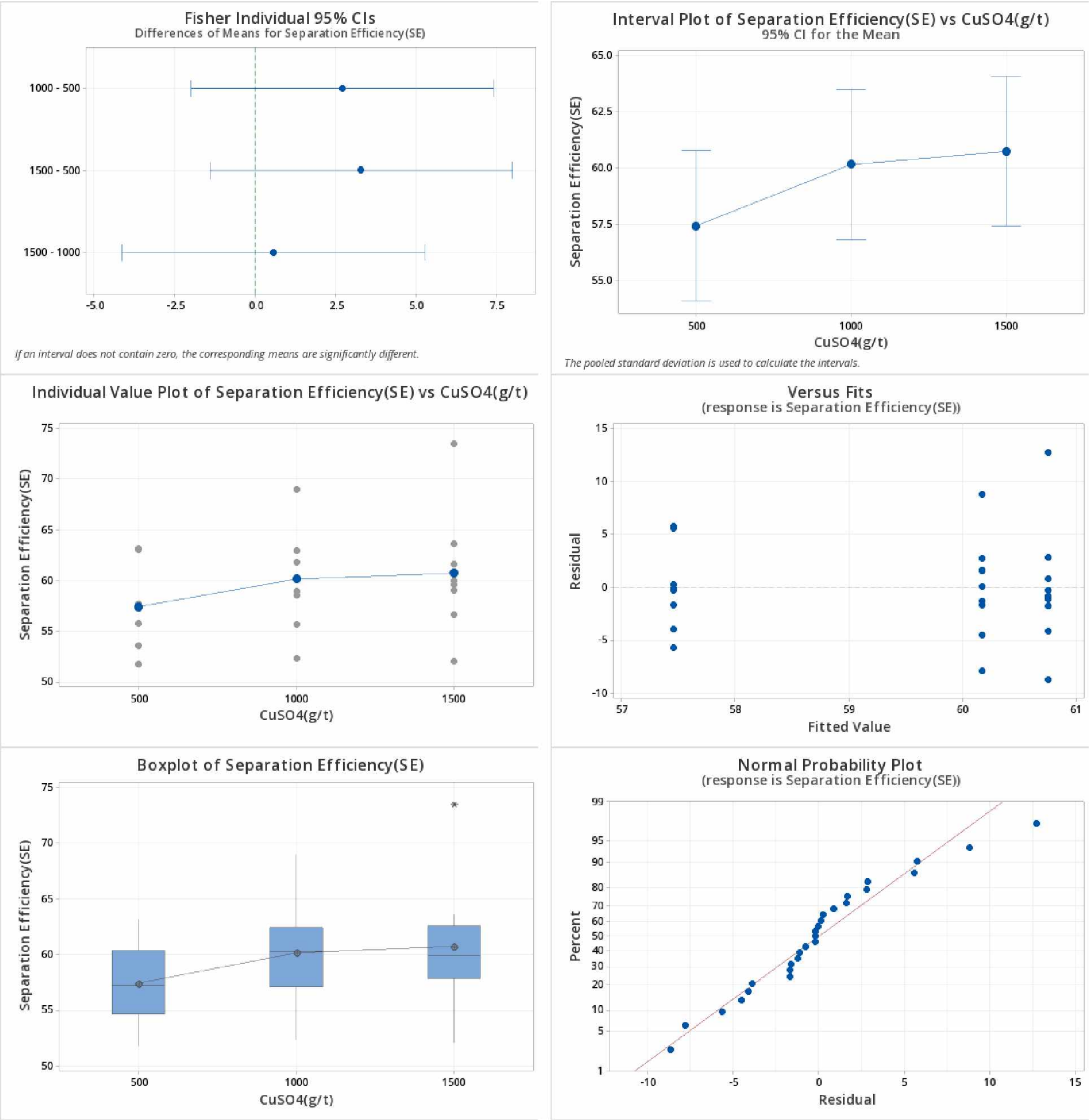


Figure 62. Statistics Graph of the Impact of CuSO4 on Sphalerite Recovery

a) Grade

According Table 44, the solid percentage is the most effective parameters on the sphalerite grade in the concentrate. The highest grade was obtained at 30% of solid. The second effective parameter is dosage of activator (CuSO4). The highest grade was achieved with 1500 g/t of the CuSO4. The third, fourth, and fifth priority are collector dosage (250 g/t), frother dosage (25 g/t), pH(11), respectively (Figure 64).


Fisher Individual 95% CIs
Differences of Means for Separation Efficiency(SE)

If an interval does not contain zero, the corresponding means are significantly different.

Interval Plot of Separation Efficiency(SE) vs CuSO4(g/t)
95% CI for the Mean

The pooled standard deviation is used to calculate the intervals.

Individual Value Plot of Separation Efficiency(SE) vs CuSO4(g/t)

Versus Fits
(response is Separation Efficiency(SE))

Boxplot of Separation Efficiency(SE)

Normal Probability Plot
(response is Separation Efficiency(SE))

Figure 63. Statistics Graph of the Impact of CuSO4 on Sphalerite Separation Efficiency

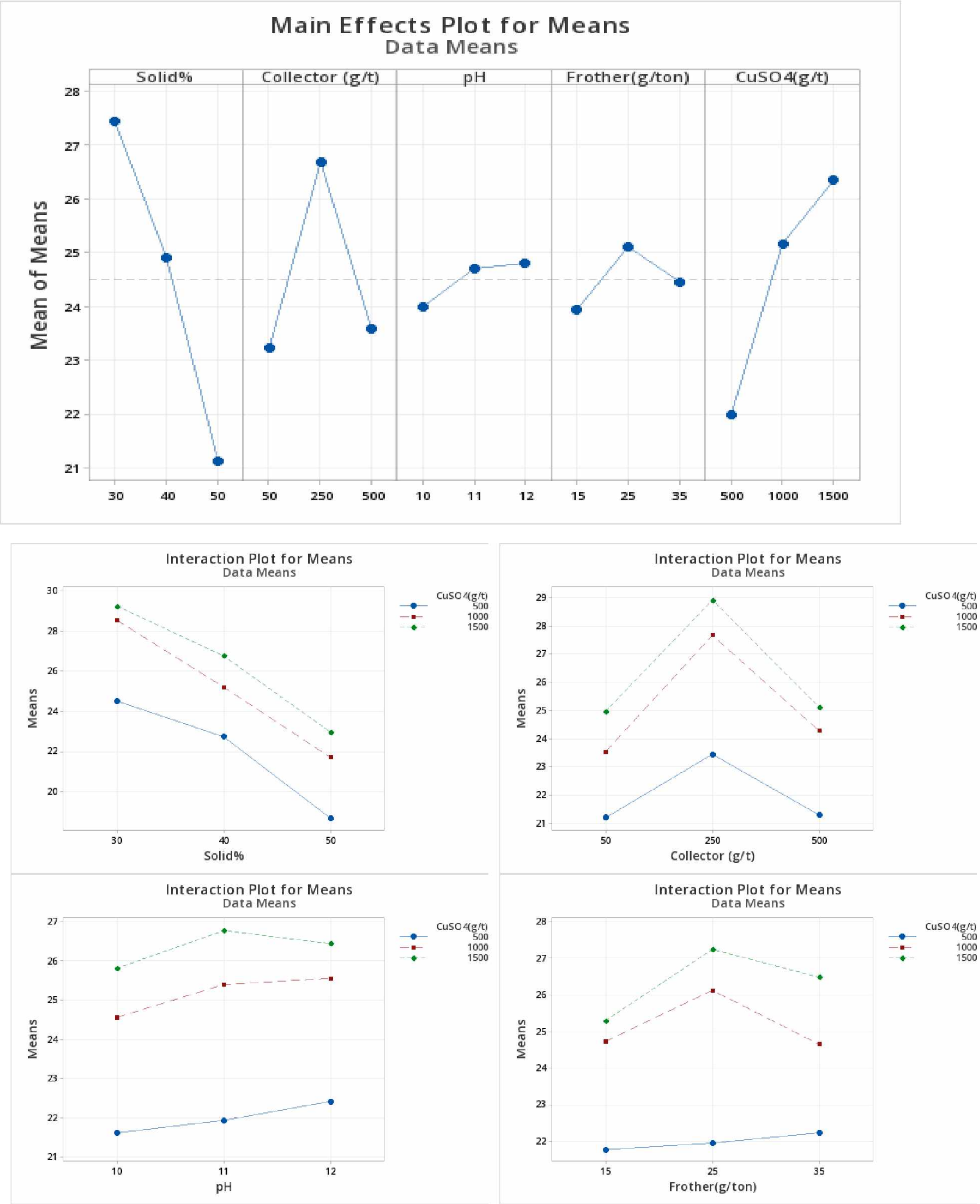


Figure 64. Effects of different parameters on the Sphalerite Grade

b) Recovery

Taguchi analysis show the most important parameter in sphalerite recovery is collector dosage. Flotation with 250 g/t collector had the highest amount of the recovery. The second priority was the solid percentage, the lower solid percentage had the highest recovery. The third, fourth, and fifth effective parameters are froth dosage (25 g/t), CuSO4 dosage (1500 g/t), and pH (11), respectively (Table 45, and Figure 65).

Table 44. Rank of parameters effectiveness in the sphalerite grade

Level	**Solid%**	**Collector (g/t)**	**pH**	**Frother(g/ton)**	**CuSO4(g/t)**
1	27.46	23.25	24.00	23.94	22.00
2	24.93	26.69	24.71	25.12	25.18
3	21.15	23.58	24.81	24.47	26.35
Delta	6.31	3.44	0.81	1.17	4.35

Table 45. Ranking of parameters effectiveness in the sphalerite Recovery

Level	**Solid%**	**Collector (g/t)**	**pH**	**Frother(g/ton)**	**CuSO4(g/t)**
1	67.54	64.17	64.91	63.82	64.17
2	66.89	69.93	66.47	67.00	66.14
3	62.24	62.57	65.30	65.86	66.37
Delta	5.30	7.35	1.56	3.18	2.20
Rank	2	1	5	3	4

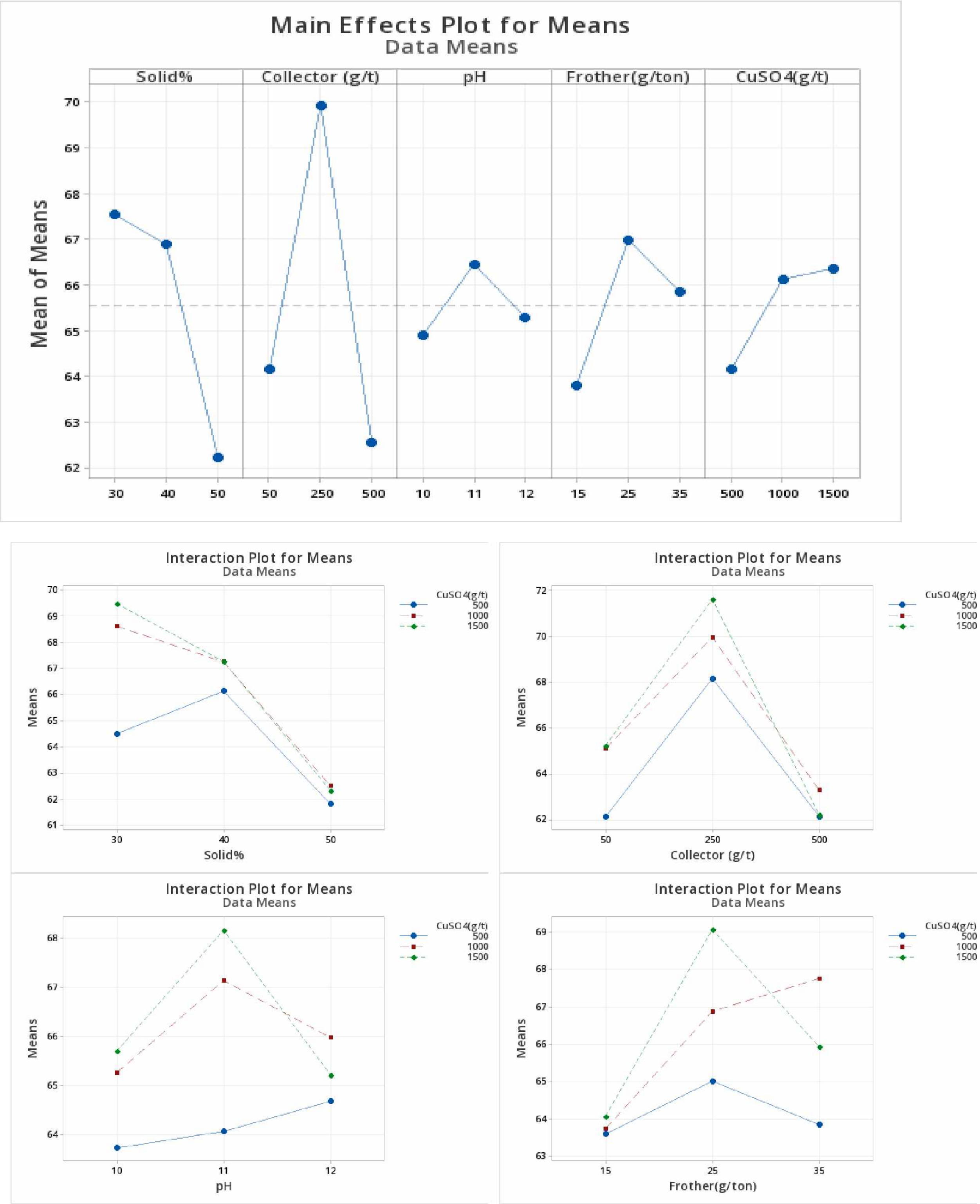


Figure 65. Effects of different parameters on the Sphalerite Recovery

c) Separation Efficiency

In the separation efficiency the dosage of collector (250 g/t) is the most effective parameters. The second priority is solid percentage (30 g/t). The third, fourth and fifth priorities are CuSO4 dosage (1500 g/t), frother dosage(25g/t), and pH(11) (Table 46, and Figure 65).

According to the counter plot the highest amount of separation efficiency can be archive when dosage of collector is between 150 to 420 g/t and solid percentage is less than 35%. Then separation efficiency could be above 70% (Figure 67).

Table 46. Ranking of parameters effectiveness in the sphalerite Recovery Separation Efficiency

Level	**Solid%**	**Collector (g/t)**	**pH**	**Frother(g/ton)**	**CuSO4(g/t)**
1	62.05	57.93	58.78	57.87	57.46
2	60.72	63.74	60.27	60.83	60.17
3	55.62	56.72	59.33	59.68	60.75
Delta	6.43	7.02	1.49	2.96	3.29
Rank	2	1	5	4	3

5.6. Grade, Recovery, and Predict Separation Efficiency Based on the Chemical Dosage

4 Node CART® Regression can be used to predict the product grade, recovery, and separation efficiency. The model was used to predict outcomes of my tests. The input of this model was Solid%, Collector (g/ton), pH, frother(g/ton), and CuSO4(g/t). The output could be grade, recovery or separation efficiency. This model can give you an overall point of view to select the dosage of chemicals in the way that the probability of gaining higher grade, recovery, as well as separation efficiency of the products. For example. Here you can see some information about the model that was designed to predict separation efficiency (Table 47, and Figure 68). More training data means more accurate model.

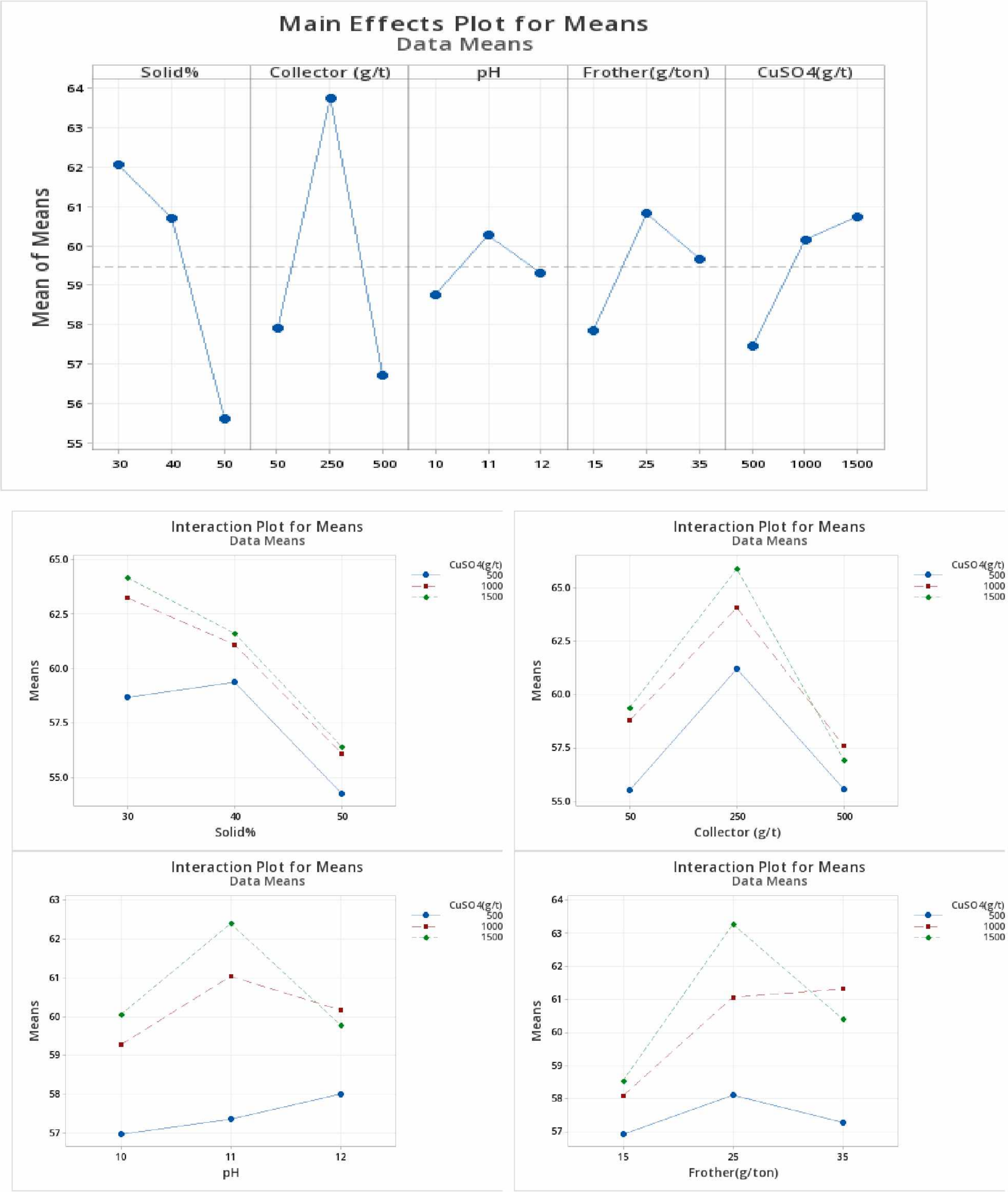


Figure 66. Effects of different parameters on the Sphalerite Separation Efficiency

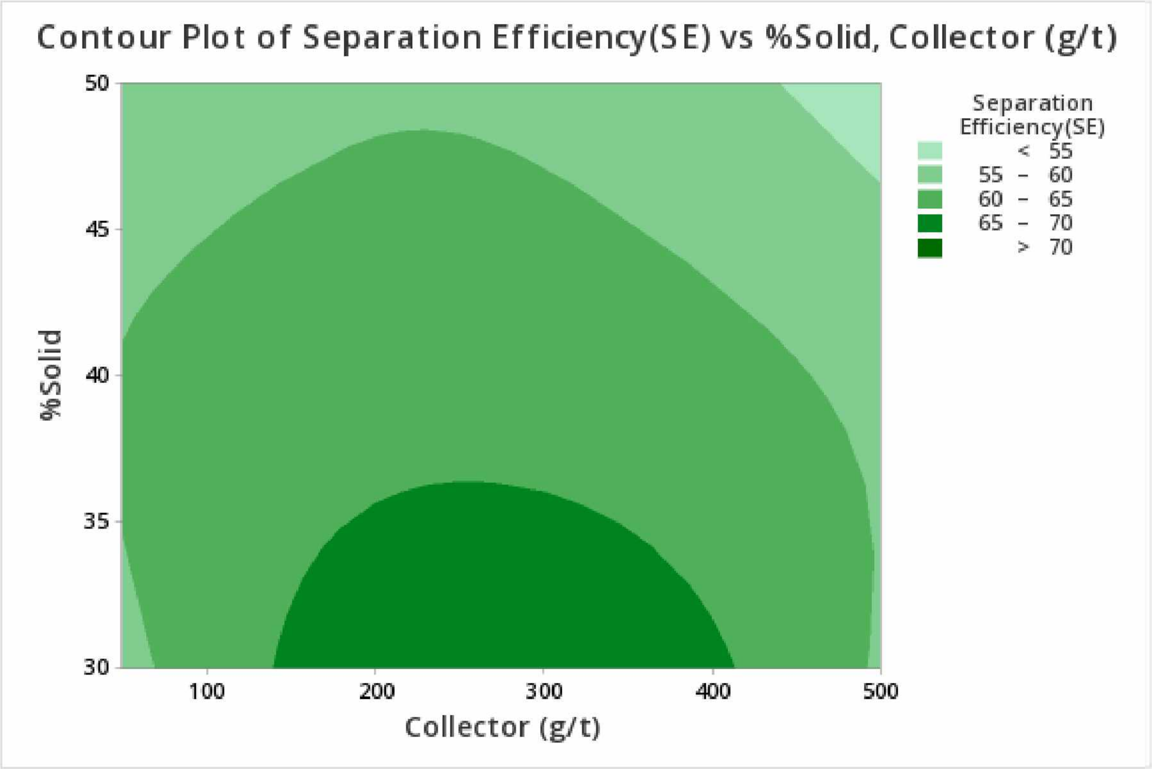


Figure 67. Contour Plot of SE vs Solid%, Collector

Table 47. 4 Node CART® Regression feature to Predict Grade, Recovery, and Predict Separation Efficiency

Node splitting	Least squared error
Optimal tree	Within 1 standard error of maximum R-squared
Model validation	10-fold cross-validation
Rows used	27

Mean	StDev	Minimum	Q1	Median	Q3	Maximum
59.4622	4.85518	51.82	56.67	59.09	61.87	73.5

Statistics	Training	Test
R-squared	63.27%	49.52%
Root mean squared error (RMSE)	2.8877	3.3852
Mean squared error (MSE)	8.3387	11.4599
Mean absolute deviation (MAD)	2.4065	2.8082
Mean absolute percent error (MAPE)	0.0401	0.0467

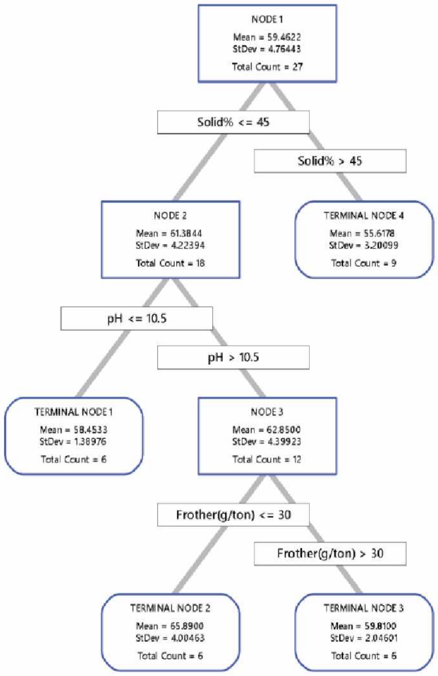


Figure 68. Optimal Tree Diagram to predict outputs

5.7 Optimized Condition for Galena and Sphalerite Flotation

Based on the results achieved from last sections, the optimized conditions for both galena, and sphalerite was estimated at different solid%. Table 48 shows the optimum condition for galena flotation at different solid%. Table 49 shows the optimum condition for Sphalerite flotation at different solid%.

Table 48. Galena flotation estimated optimized condition

Solid%	**Collector(g/t)**	**Frother(g/t)**	**pH**	**ZnSO4**	**NaCN**	**SE**
30	150	17	7	300	50	>55
40	200	40	7	300	100	>55
50	300	65	7	300	150	>55

Table 49. Sphalerite flotation optimized condition

Solid%	Collector(g/t)	Frother(g/t)	pH	CuSO4	SE
30	200	25	11	600	>70
40	250	25	11.8	1000	65-70
50	250	35	12	1200	60-65

The flotation tests were done with the estimated condition. After conducting flotation tests using estimated condition, grade, recovery, and SE were obtained. The results were shown for Galena, and Sphalerite in Table 50, and Table 51.

At optimized condition with 30% of solid, grade, recovery, and SE of galena flotation are 28.28%, 67.31%, and 64.11%, respectively. Grade, recovery, and SE for optimized condition with 40% of solid are 28.61%, 67.31%, and 70.06%, respectively. Grade, recovery, and SE for optimized condition with 50% of solid are 23.21%, 68.45%, and 60.39%, respectively. The lower solid % has a higher grade at galena flotation. The experiment with 40% has the highest galena recovery.

According to Table 51, grade, recovery, and SE for optimized condition with 30% of solid at sphalerite flotation are 34.87%, 80.24%, and 75.32%, respectively. Grade, recovery, and SE for optimized condition with 40% of solid are 29.1%, 71.65%, and 68.38%, respectively. Grade, recovery, and SE for optimized condition with 50% of solid are 27.23%, 67.23%, and 62.24%, respectively. Flotation with low solid percentage shows higher grade and recovery for sphalerite flotation.

Table 50. Sphalerite concentrate grade, recovery, and SE using estimated condition

Solid%	**Collector(g/t)**	**Frother(g/t)**	**pH**	**ZnSO4**	**NaCN**	**Pb Grade%**	**Pb Recovery%**	**SE**
30	150	17	7	300	50	28.28	67.31	64.11
40	200	40	7	300	100	25.61	70.06	61.35
50	300	65	7	300	150	23.12	68.45	60.39

Table 51. Sphalerite concentrate grade, recovery, and SE using estimated condition

Solid%	**Collector(g/t)**	**Frother(g/t)**	**pH**	**CuSO4**	**Zn Grade%**	**Zn Recovery%**	**SE**
30	200	25	11	600	34.87	80.24	74.32
40	250	25	11.8	1000	29.1	71.65	68.38
50	250	35	12	1200	27.23	67.23	62.64

In the next step, extra researches were performed to improve the flotation performance.

5.8. Effects of Mixed Collectors on Galena, and Sphalerite Flotation

In this series of experiments a mixed of 2 collector was used to examine their impacts on galena, and sphalerite flotation (Table 52, and Table 53). These collectors are including xanthate and dithiophosphinate. Equal amount of them used in both galena and sphalerite flotation. The results showed a mixed collectors have a positive effect to improve the recovery in all different solid%. But there are not significant differences in concentrates grades except at 30% that grade decrease by 2%. When mixed collector instead of single collector is used weaker collectors adsorbed by more oxidized sites and stronger one was adsorbed by less oxidized sites which would result to denser and even more coverage of sulfide minerals surfaces (Bagci et al., 2007). Consequently, the probability of galena or sphalerite minerals as well as binary particles will increase. Thus, there is an increase in the flotation recovery.

Table 52. Galena concentrate grade, recovery, and SE using mixed collector

Solid%	Collector(g/t)	Frother(g/t)	pH	ZnSO4	NaCN	Pb Grade%	Pb Recovery%	SE
30	150	17	7	300	50	26.35	68.82	64.73
40	200	40	7	300	100	25.09	72.04	62.05
50	300	65	7	300	150	23.47	69.9	60.94

Table 53. Sphalerite concentrate grade, recovery, and SE using mixed collector

Solid%	Collector(g/t)	Frother(g/t)	pH	CuSO4	Zn Grade%	Zn Recovery%	SE
30	200	25	11	600	34.02	82.37	75.64
40	250	25	11.8	1000	28.68	73.12	69.58
50	250	35	12	1200	27.29	69.23	64.55

5.9. Using Na2S at Sphalerite Flotation to Depress Galena

However, in Lead-Zinc flotation circuit, at the first step galena separated, followed by the separation of sphalerite, but still some galena remains in the sphalerite feed. Therefore, it is necessary to remove the remaining galena in sphalerite feed. One way to remove galena at sphalerite circuit is using Na_2S as a galena depressant. the result showed at higher pH Na2S act so well as it can depress around 80% of galena at pH 11 at high dosage. About dosage of Na2S should be careful. because adding extra Na2S causes to depress binary minerals as well as sphalerite mineral. Adding appropriate amount of Na2S causes to increase the grade of sphalerite concentrate in all 3 levels of solid%. But it will have a small negative effect on the sphalerite recovery.

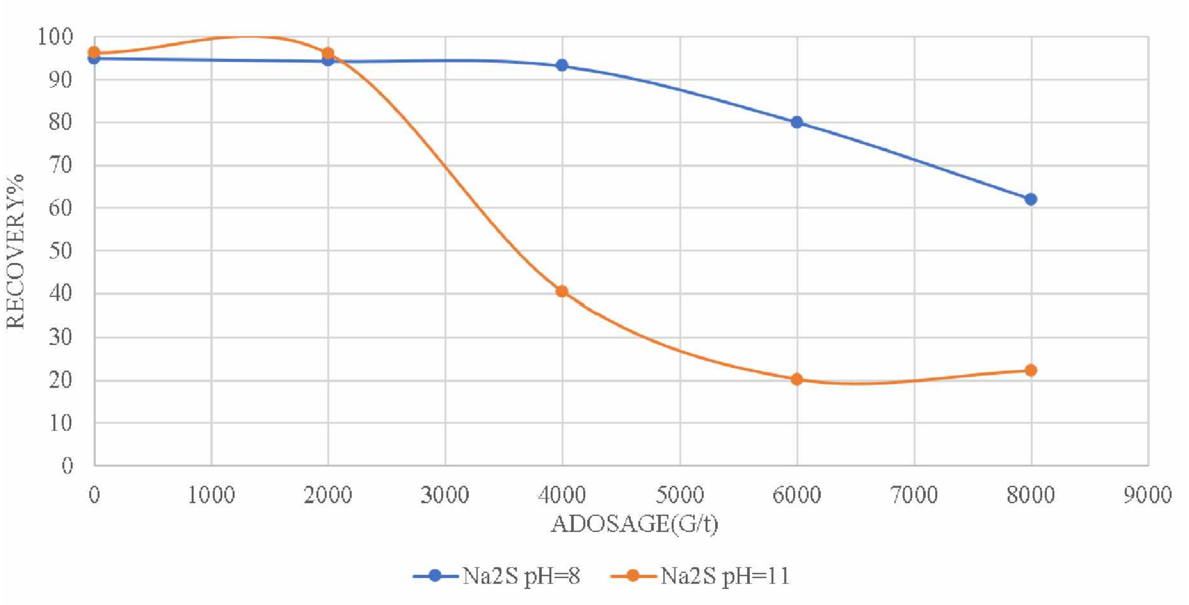


Figure 69.Depression of Galena Using Na2S

Table 54. Effect of Na_2S at Sphalerite flotation on its grade, recovery, and separation efficiency

Solid%	**Collector(g/t)**	**Frother(g/t)**	**pH**	**CuSO4**	**Na2S(g/t)**	**Zn Grade%**	**Zn Recovery%**	**SE**
30	200	25	11	600	3000	36.25	81.24	77.01
40	250	25	11.8	1000	3000	30.15	71.32	70.44
50	250	35	12	1200	3000	29.82	69.17	65.53

5.10. Effect of Bubble Size on the Flotation Performance

Bubble size is one of the main parameters that affect flotation performance. Any decrease or increase on the bubble size can impact on the product grade and recovery. To better understanding about their role on the flotation, some photo was taken from the bubbles during the flotation process. Their average diameter was determined (Figure 70, and Figure 71). The relationship between their sizes and the grade recovery was obtained. The result shows the appropriate bubble size for 30% is around 1000 microns, by increasing or decreasing the bubble size recovery will decrease. For both pulp with 40, and 50% of solid, 1200 microns of bubble size can give the max recovery (Figure 72).

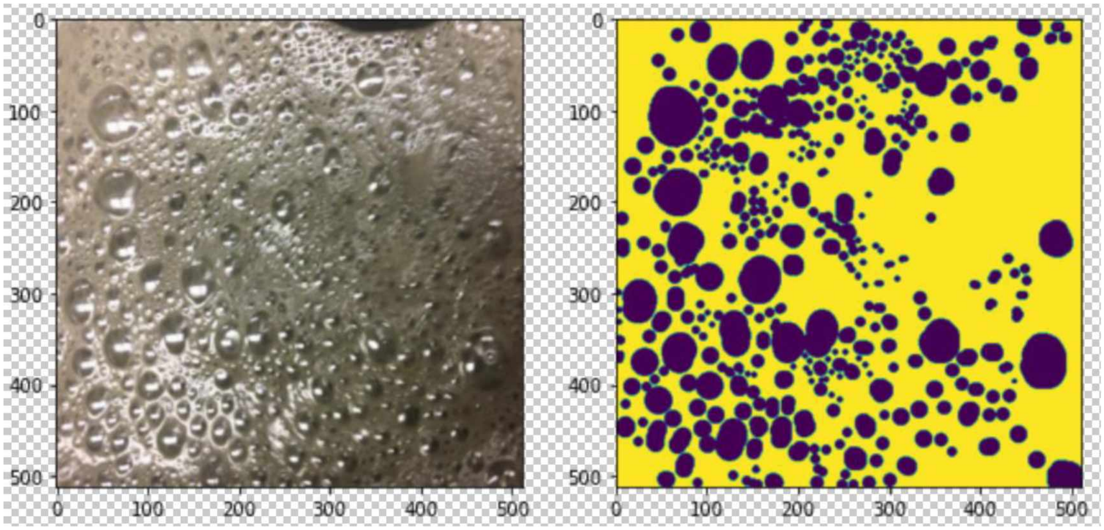


Figure 70. CNN detection of bubbles

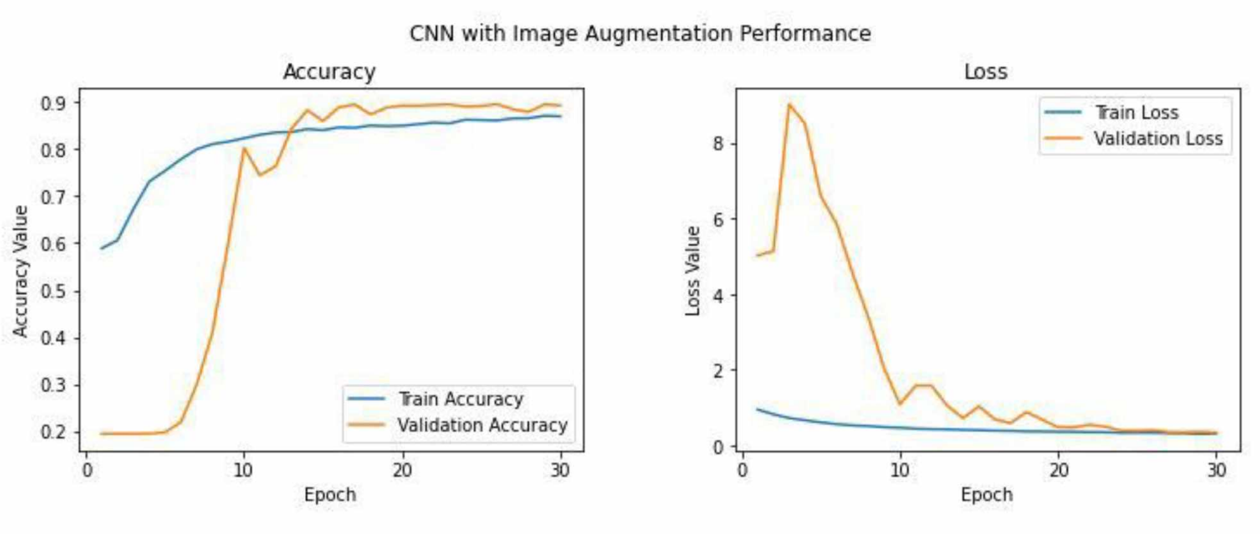


Figure 71. Accuracy of CNN method to recognize bubbles(Malekian, 2017)

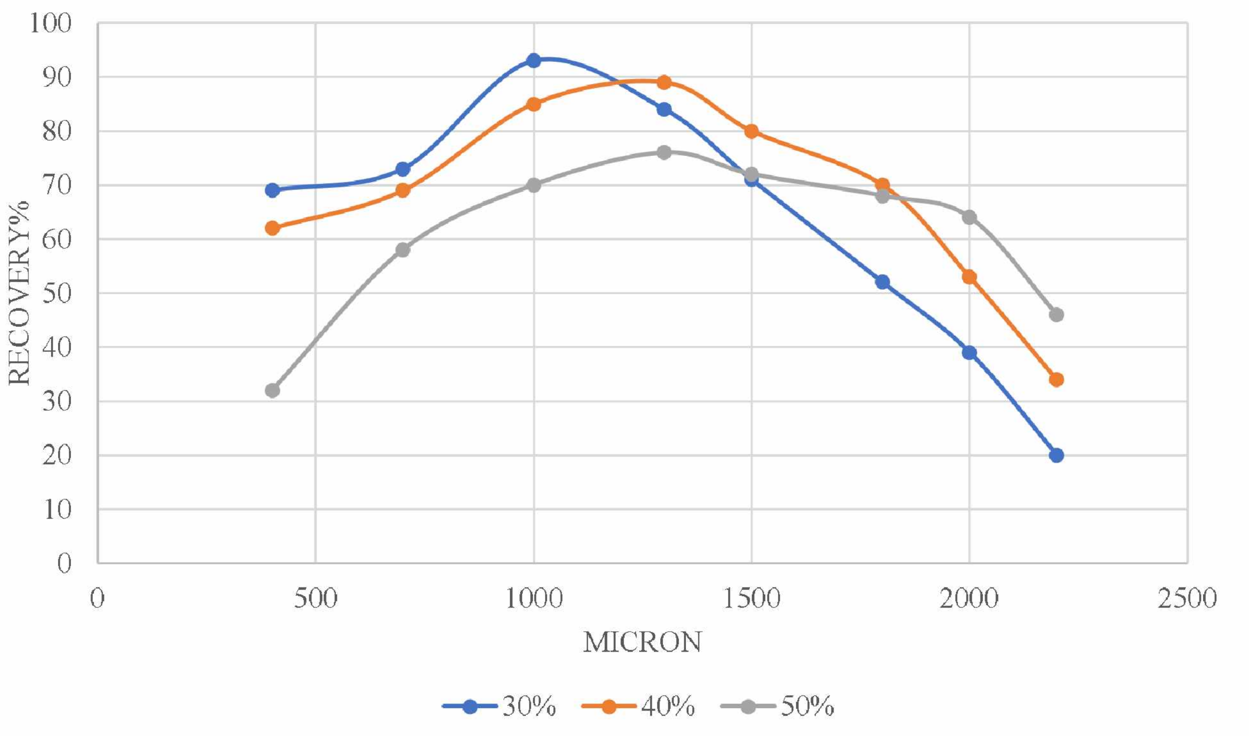


Figure 72. Relationship between Bubble Size and Galena Recovery

5.11. Locked Cycle Tests

Locked cycle tests were done based on the red dog flowsheet. First of all, the locked cycle tests were performed based on the original condition in the red dog for galena. The result showed at 30% of solid, the grade, and recovery of Pb were 57.36%, and 52.73%, respectively. By increasing the soli% to 40, and 50% in the flotation cell, the grade was decreased by almost 4, and 10%. The recovery was decreased by around 1% for 40, and 50 of solid%. Data shows an adverse impact of the increasing solid% in the galena flotation locked cycle tests. In the next step, the optimized condition was used to at galena flotation tests. The grade, and recovery for pulp with % 30 of solid improved by around 0.5%. Optimized condition had a further impact on the flotation performance at higher solid%. As by increasing the solid%, grade was improved by 1.84%, and 2.24% at galena concentrate for 40%, and 50%, respectively. Recovery was improved for both solid% by less than 1%.

In sphalerite circuit, the final grade, and recovery for 30% of solid were 55.52, and 79.11%. in the optimized condition Zn grade was increased by 0.96%, and its recovery decreased by 0.62%. By increasing the solid%, grade and recovery decreased remarkably. Then flotation tests were performed in the optimized condition for both 40%, and 50%. The optimized condition increased Zn grade at flotation circuit by 1%, and recovery by 4% for 40% of solid. In addition, the optimized condition increased grade at flotation circuit by 5%, and recovery by 4% for 50% of solid.

Table 55. Grade and Recovery of Galena Lock Cycle Test

Solid%	**Pb Grade%**	**Pb Recovery%**	**Pb Grade%**(after modified the condition)	**Pb Recovery%**(after modified the condition
30	57.36	52.73	58.09	53.21
40	53.28	51.44	55.12	49.35
50	47.81	50.88	51.17	49.48

Table 56. Grade and Recovery of Sphalerite Locked Tests

Solid%	Zn **Grade%**	Zn **Recovery%**	Zn **Grade%**(after modified the condition)	Zn **Recovery%**(after modified the condition
30	55.52	79.11	56.68	78.49
40	51.13	74.64	53.02	76.05
50	45.38	69.20	49.77	73.16

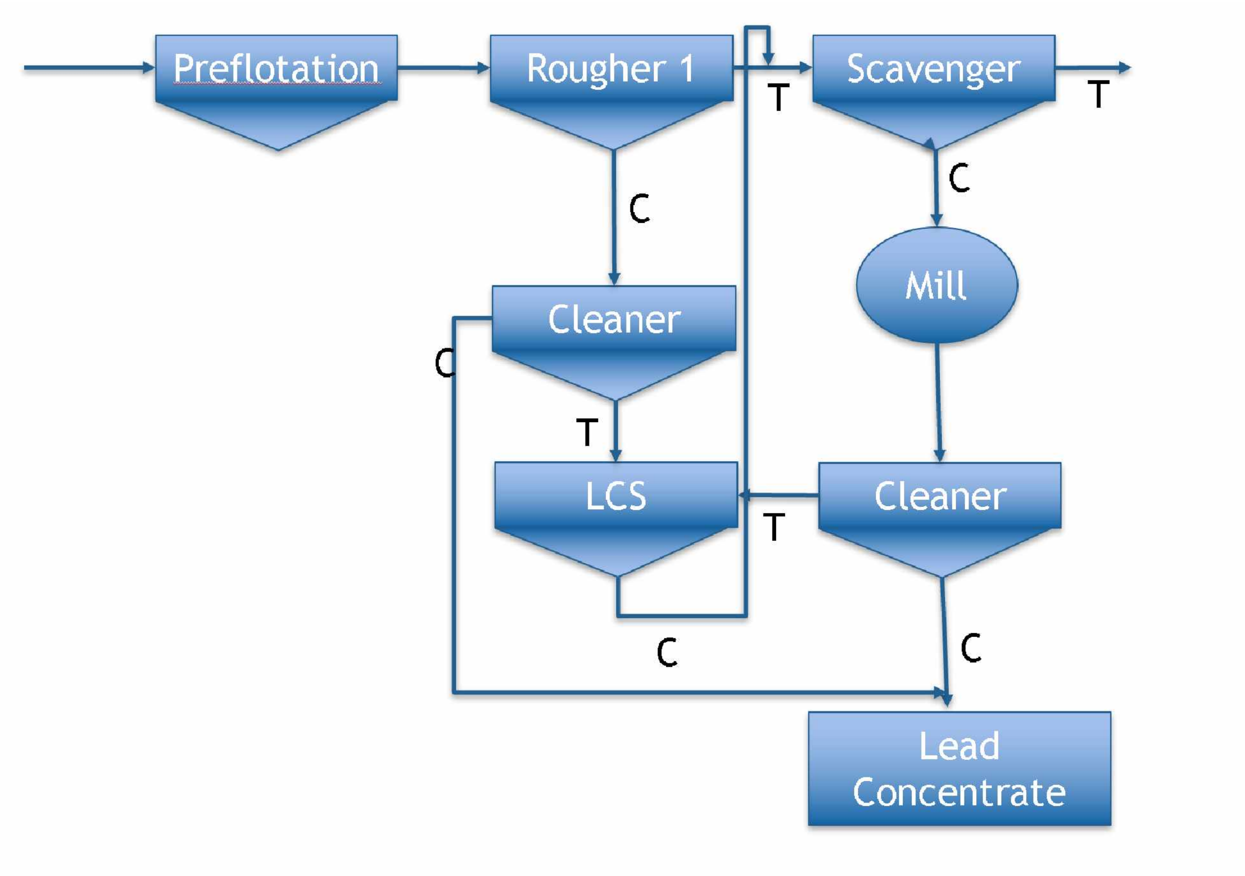


Figure 73. Galena Lock Cycle Test Flowsheet

CHAPER 6 CONCLUSION

1) At flotation test with 20%, 40% and 60% solids, yield was significantly higher at low densities with 76.1% feed material reporting to the concentrate at 20% solids at 800 RPM. The yield values at 40% and 60% solids were substantially lower at 74.25% and 69.88% respectively at the same RPM. Lower RPM means lower turbulence that give enough time to valuable materials to be separated from unwanted materials.
2) The results conclusively prove that at high slurry densities higher RPM rate leads to better recovery. Lead (Pb) and Zinc (Zn) recovery values at 20% solids and 1800 RPM rotor speed are 70.12% and 65.1% respectively. However, there is significant difference in performance between 40% solids and 60% solids, with 60% solids displaying better floatability throughout the rotor RPM envelope. This phenomenon can be attributed to better bubble loading at higher slurry densities, which leads to better bubble-particle collision and adhesion rate. Froth phase is stable at higher percent solids and material loss from froth phase to pulp phase is reduced. Moreover, there is significant turbulence effect at higher RPM ranges (1500 ~ 1800).
3) At lower pulp solid%, by decreasing the particles sizes, recovery increased due to more liberation of galena particles. But by increasing the pulp solid% as well as liberation, the bubbles were overloaded so some of them burst before reach to top of the cell. Thus, it has a negative effect on the recovery at high solid percentage. On the other hand, by increasing the liberation, the concentrate grade improved in all 3 different pulp densities from 9.08%, 10.12%, and 6.85% to 14.32%, 12.96%, and 9.27%, respectively. The pulp with PSD 80 microns has the highest grade and recovery at the lowest pulp density. However, pulp with

PSD 150 microns has the highest recovery at pulp with 60% solid, and the highest grade at pulp with 40% solid

4) Flotation results shows feed was grinded by stainless-steel (ceramic) grinding media has higher grade, and recovery compare to steel grinding media. Using stainless-steel grinding media instead of the steel grinding media increased the rougher concentrate grade by 0.53, 0.27, and 1.23 for pulp with 20, 40, and 60 solid percentage, respectively. In addition, the recovery was improved by 0.48, 0.99, and 2.48 for pulp with 20, 40, and 60 solid percentage, respectively. Steel grinding media because of dissolving higher amount of Fe^{2+} ions lead to produce higher amount of H_2O_2. In addition, the present of H_2O_2 increase the oxidation and decomposition of lead xanthate
5) Increasing solid% from 30 to 50% has a negative effect on the grade. As by increasing the solid percentage by 20%, the avarage grade decrease by 3%. Increasing solid% causes to decrease the selectivety of separation.
6) Increasing soild% from 30 to 50% has a negative effect on the recovery. The recovery drops from 60.02 to 55.43%. Increasing solid% leades bubble be overloaded with particles and brust before reaching to the froth zone. Thus, it has a negative effect on the flotation recovery.
7) Increasing solid% causes to decrease sepration efficiency from 55.4 to 50.67%.
8) Increasing the collector dosage from 50 to 140 g/ton as well as 140 to 500 has a significant impact on the grade. The highest grade was achieved at 140 g/ton of collector dosage. Increasing collector dosage to a certain amount has a positive effect on flotation grade because it causes to improve separation efficiency. But when collector dosage increase

from certain amount can causes micelle or flotation of othe type of sulfide minerals that has a negative effects on the flotation grade.

9) Increasing collector dosage from 50 to 500 g/ton has a significant effect on the galena recovery and separation efficiency.By increasing collector dosage separation efficiency increase from 41.78 to 58.8%
10) Because of the limited range of pH in my experiments, it does not indicate a significant impact on grade, recovery , and separation efficiency of the galena.
11) Increasing the frother dosage from 15 to 50 g/ton shows a remarkable impact on the galena concentrate grade. As by increasing frother dosage from 15 to 50 g/ton, the concentrate grade increased from 24.4 to 26.42%. Adding more frother causes bubble stability as well as producing the smaller bubble to convey more particles to the froth zone.
12) Increasing frother dosage from 15 to 50 g/t does not show a remarkable effect on the recovery and separation efficiency of galena.
13) Increasing ZnSO4 dosage from 0 to 300 g/ton has a significant impact on the galena grade. The grade increased by 2.5%. Increasing ZnSO4 dosage to a certaain dosage causes to depression of more amount of sphalerite so it improves the product grade.
14) Increasing ZnSO4 dosage from 0 to 600 does not show significant effects on the galena recovery, and separation efficiency.
15) Increasing NaCN dosage from 50 to 150 g/ton does not show any remarkable affects on galena grade, recovery, and separation efficiency.
16) According to counter plot, performing flotation by collector dosage above 140 g/ton, frother dosage above 25 g/ton, and ZnSO4 dosage between 200 to 400 g/ton keeps separation efficiency above 55%.

17) For optimized condition with 30% of solid at galena flotation, grade, recovery, and SE are 28.28%, 67.31%, and 64.11%, respectively. Grade, recovery, and SE for optimized condition with 40% of solid are 28.61%, 67.31%, and 70.06%, respectively. Grade, recovery, and SE for optimized condition with 50% of solid are 23.21%, 68.45%, and 60.39%, respectively.

18) The results show grade, recovery, and SE for optimized condition with 30% of solid at sphalerite flotation are 34.87%, 80.24%, and 75.32%, respectively. Grade, recovery, and SE for optimized condition with 40% of solid are 29.1%, 71.65%, and 68.38%, respectively. Grade, recovery, and SE for optimized condition with 50% of solid are 27.23%, 67.23%, and 62.24%, respectively.

19) A mixed collectors have a positive effect to improve the recovery in all different solid%. But there are not significant differences in concentrates grades except at 30% that grade decrease by 2%. Using the mixed collectors in the flotation. When mixed collector is used weaker collectors adsorbed by more oxidized sites and stronger one was adsorbed by less oxidized sites which would result to denser and even more coverage of sulfide minerals surfaces

20) At higher pH Na2S act so well as it can depress around 80% of galena at pH 11 at high dosage. About dosage of Na2S should be careful. because adding extra Na2S causes to depress binary minerals as well as sphalerite mineral. Adding appropriate amount of Na2S causes to increase the grade of sphalerite concentrate in all 3 levels of solid%. But it will have a small negative effect on the sphalerite recovery.

21) Any decrease or increase on the bubble size can impact on the product grade and recovery. To better understanding of their role on the flotation. Some photo was taken from the

bubbles during the flotation process. Their average diameter was determined. The relationship between their sizes and the grade recovery was obtained. The result shows the appropriate bubble size for 30% is around 1000 microns, by increasing or decreasing the bubble size recovery will decrease. For both pulp with 40, and 50% of solid, 1200 microns of bubble size can give the max recovery. When bubble size increase to a certain amount, its capacity to carry the particles improve. But increasing the bubble size from a certain amount, make bubble unstable as well as causes bubble overloading that has a negative effect on flotation performance.

22) At 30% of solid, the grade, and recovery of Pb were 57.36%, and 52.73%, respectively. By increasing the solid% to 40, and 50% in the flotation cell, the grade was decreased by almost 4, and 10%. The recovery was decreased by around 1% for 40, and 50 of solid%. Data shows an adverse impact of the increasing solid% in the galena flotation circuit. In the next step, using the optimized condition, flotation tests for galena were done. The grade, and recovery for solid 30% improved by around 0.5%. Optimized condition had a further impact on the flotation performance at higher solid%. As by increasing the solid%, grade was improved by 1.84%, and 2.24% at galena concentrate for 40%, and 50%, respectively. Recovery was improved for both solid% by less than 1%.

23) In sphalerite circuit, the final grade, and recovery for 30% of solid were 55.52, and 79.11%. in the optimized condition Zn grade was increased by 0.96%, and its recovery decreased by 0.62%. By increasing the solid%, grade and recovery decreased remarkably. Then flotation tests were performed in the optimized condition for both 40%, and 50%. The optimized condition increased Zn grade at flotation circuit by 1%, and recovery by 4% for

40% of solid. In addition, the optimized condition increased grade at flotation circuit by 5%, and recovery by 4% for 50% of solid.

CHAPER 7 SUGGESTIONS FOR FUTURE STUDIES

1) Study the size of the agitator in flotation cell and its effect on flotation performance at high solid%.
2) Study the contact angle between bubbles and particles at different solid% at different flotation condition to find the relationship between contact angle and solid% as well as the bubble stability,
3) Study the degree of freedom of products at different solid% to find the relationship between degree of freedom of particles and the flotation performance to decide grinding time is enough or not to optimize grinding time.
4) Performing the flotation with different solid% at flotation cells with different Hight to consider the relationship between the cell height, solid% and bubble stability.
5) Study the effects of other type of collectors, frother, depressants, and activators at high solid%.

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