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Pilot-Scale Demonstration of Increasing Iron Recovery from Minnesota Oxidized Iron Resources

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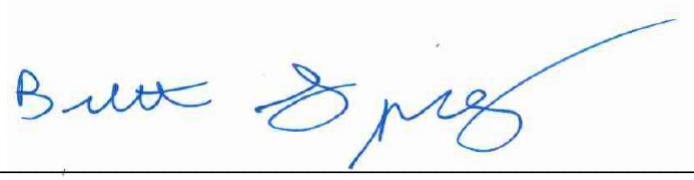
By: 

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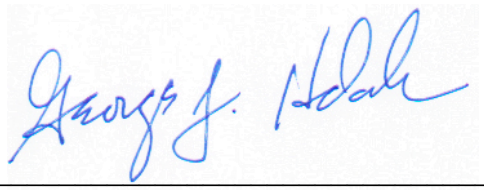
Approved By: 

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Executive Summary

Magnetite is one of several iron oxide minerals that occur within the iron ores on the Mesabi Range. Current Minnesota taconite mining and processing operations extract and concentrate magnetite from the ores to produce taconite pellets. At present, recovery of other iron oxide minerals in the ores (e.g., hematite, goethite) negatively impacts the current process to make taconite pellets. As a result, iron oxide minerals other than magnetite report to the waste materials during taconite processing. This means that a significant amount of iron within the ores is currently not being recovered and reports to waste streams. As well, the abundance of hematite and goethite limit the extent to which taconite ores are currently mined, and limits the extent of ores and ore reserves on the Mesabi Range. This results from the current inability to effectively and efficiently process these iron oxide minerals using currently employed processing schemes. As well, it has been generally observed that iron formation with an abundance of oxidized minerals contains increased concentrations of phosphorus (which commonly occurs in the minerals apatite and goethite), an element that is deleterious to the production of high quality steel products.

In an effort to reduce waste and utilize undeveloped Minnesota iron resources, the University of Minnesota Duluth Natural Resources Research Institute (UMD NRRI) undertook a research project to investigate technologies that may allow for the recovery of oxidized iron minerals (specifically hematite and goethite). This project was one of five research projects conducted by UMD NRRI as part $2.6 million Mining and Water Innovation Grant provided by the Minnesota Legislature.

The UMD NRRI has developed a process that has been shown to remove quartz, apatite, and some gangue (waste) minerals while recovering magnetite, hematite, and goethite in a limited number of samples. This flotation process, known as NatFlot[1], is conducted at the natural pH of the ore slurries in contrast to the industry proven high pH selective flocculation-deslime-flotation (SFDF) process. Implementing a process that occurs at natural pH may result in the elimination of pH modifier chemicals and dispersants that have associated costs and potential environmental and operational safety impacts. The *Pilot-Scale Demonstration of Increasing Iron Recovery from Minnesota Oxidized Iron Resources* report documents the bench- and pilot-scale research results, potential applications, and limitations of the NatFlot process. This research project focused on two of the UMD NRRI research goals: reduction of waste and resource stewardship (e.g., reducing environmental impact). The project objectives were to: 1) demonstrate technology for increasing iron-recovery from Minnesota oxidized iron ore resources at bench- and pilot-scale; 2) provide baseline induration data; and 3) characterize the process water and tailings from the pilot circuit testing.

The definitions for the iron formation used in this report are summarized in Table 6. Oxidized iron resources can be classified into two broad groups as either waste or in-situ (or in-the-ground) resources. In-situ oxidized iron resources consist of oxidized and partially oxidized iron formation that does not respond well to current fine grinding and low-intensity magnetic

[1] Patent application title: Cationic Flotation of Silica and Apatite from Oxidized Iron Ores at Natural pH. International Patent Application No. PCT/US2017/014372, Publication Number WO 2017/127704 A1, published July 27, 2017.

separation processes. Waste oxidized iron resources from early mining consist of lean "ore" waste piles, jig rock tailings piles, and tailings within tailings ponds. Taconite waste oxidized iron resources include lean "ore" waste piles and tailings in tailings ponds. Iron formation from taconite operations is wasted because it may be low-grade, have a low weight recovery with fine grinding and low intensity magnetic separation, or produce a concentrate that is sufficiently oxidized to negatively impact induration and thus, pellet production and quality.

Table 1: Definitions for Iron Formation Types Used in this Report

Iron Formation Type	Iron Oxides	Gangue Minerals	Response to Low-Intensity Magnetic Separation
Oxidized	Hematite, goethite	Quartz (chert)	Low weight recovery
Partially-Oxidized	Magnetite, hematite, goethite	Quartz, carbonate(s), silicate(s)	Moderate weight recovery, low quality concentrate
Taconite	Magnetite	Quartz, carbonate(s), silicate(s)	High weight recovery, high quality concentrate

Current fine particle separation technologies for processing of iron minerals mainly rely on differences in the magnetic susceptibility, specific gravity, and surface properties of the minerals. The typical taconite process relies on the separation of magnetite (a ferrimagnetic mineral) from gangue minerals by fine grinding and low-intensity magnetic separation (LIMS). However, para-magnetic iron minerals (those that are weakly attracted to a magnet), namely hematite and goethite, do not report to the concentrate stream with the conventional LIMS process. Historically, reduction roasting and magnetic separation was found to effectively recover iron minerals on a variety of oxidized iron samples, but the combination is an energy intensive process and is more costly than other beneficiation technologies. Two other technologies used for beneficiation of fine-liberating oxidized iron minerals are wet high-intensity magnetic separation (WHIMS) and elevated-pH flotation.

The intent of this project was to evaluate the NatFlot process on various oxidized iron resources on the west end of the Mesabi Iron Range to determine applicability of this technology on Minnesota oxidized iron resources. The technical team engaged with industry geologists and metallurgists for the selection of bench- and pilot-scale materials for this research project. The locations for test samples were limited to the western portion of the Mesabi Iron Range, specifically the iron resources between Grand Rapids, Minnesota, and Hibbing, Minnesota, as this area contains a large amount of oxidized iron resource (Zanko et al., 2003). The test material was to contain oxidized iron minerals (e.g., hematite, goethite) and not satisfy the typical taconite ore cut off criteria, including Davis Tube concentrate silica grade, concentrate weight recovery, and/or amount of oxidized iron remaining in concentrate. Bench and bulk sample selection was limited to samples available in stockpiles or pit faces. A total of seven bench-scale (drill core or "grab") samples and three pilot-scale bulk (>10 ton each) samples were acquired for this test program. The bench-scale samples are summarized in Table 2.

Table 2: Bench-Scale Sample Summary

Bench Sample ID	Sampling Origin	Sample Type	Sample Selection Rationale
17-1 through 17-4	Drill Core	Taconite with fine-grained primary hematite	Low Davis Tube Test weight recovery
18-1	"Grab" from Pit Wall	Oxidized Iron Formation	Low magnetic iron content
18-2	"Grab" from Pit Wall	Partially Oxidized Iron Formation	Low magnetic iron content
18-3	"Grab" from Stockpile	Partially Oxidized Iron Formation	Davis Tube concentrate contained elevated hematite

Baseline NatFlot tests and conventional beneficiation process comparative testing was conducted on the Bench 17-1 through 17-4 and Bench 18-1 through 18-3 samples. The purpose of these tests was: 1) to determine how the samples reacted to the different processes; and 2) to determine which sample(s) would be selected for bulk sampling and pilot-scale testing. Through this test program, it was determined that the Bench 17-1 through 17-4 samples could not be successfully upgraded to the silica target 4.0-4.5% SiO_2 with either the NatFlot or SFDF technology. The high-silica and low-iron grade was due to concentrating the ankerite and stilpnomelane in the final concentrate. The amount of ankerite and stilpnomelane reporting to concentrate due to selectivity or particles locked with silica is not known at this time. Davis Tube separation testing found that target silica grades could be achieved with the Bench 17-1 through 17-4 samples, but at a low weight recovery. Therefore, these samples were not pursued for further research at this time.

In reviewing the results with the Bench 18-1 through 18-3 samples, it was found that the Bench 18-1 sample could be beneficiated near the concentrate silica target with both NatFlot and SFDF flotation technologies. However, the phosphorus grade for the both concentrates was elevated compared to typical Minnesota magnetite concentrate, possibly due to the high amount of goethite in the final concentrate. Typically, increasing the goethite concentration also increases the phosphorus content in the concentrate. Therefore, this sample was excluded from future research at this time. Both NatFlot and SFDF produced low-silica concentrates with the Bench 18-2 sample; the SFDF baseline test achieving those results at a lower flotation time and weight and increased iron over the NatFlot baseline of approximately 7% and 10%, respectively. The initial results from this baseline test series justified the acquisition of a bulk sample of this material for further research. Lastly, both NatFlot and SFDF technologies resulted in concentrate silica grades above the target with the Bench 18-3 sample due to quartz being recovered to the concentrate. However, the concentrates produced were considered to have reasonable concentrate chemical analysis and recovery. Davis Tube results indicated it was possible to produce a magnetic concentrate at the silica target with relatively good recovery. Scavenging of Bench 18-3 LIMS tailings using WHIMS suggested some potential, but the concentrates produced with WHIMS had large amounts of quartz and relatively low weight recoveries. Regardless, this sample showed enough interesting results to pursue the acquisition of a bulk sample for further research.

Based upon the results of the bench-scale testing program and sample availability, a total of three bulk pilot-scale samples were acquired for basic characterization, baseline testing, and decision

making for selection of sample(s) to use for pilot-scale beneficiation and metallurgical testing. The bulk (pilot-scale) samples are summarized in Table 3.

Table 3: Bulk (Pilot-Scale) Sample Summary

Bulk Sample ID	Sampling Origin	Sample Type	Sample Selection Rationale
16-1	Bulk from Stockpile	Partially Oxidized Iron Formation	Davis Tube concentrate contained elevated hematite
19-1	Bulk from Stockpile	Partially Oxidized Iron Formation	Davis Tube concentrate contained elevated hematite
19-2	Bulk from Pit Wall	Oxidized Iron Formation	Low magnetic iron content

Baseline NatFlot tests and conventional beneficiation process comparative testing was conducted on the Bulk 16-1, 19-1, and 19-2 samples. In summary, both the Bulk 16-1 and 19-1 samples achieved better metallurgical results with the typical low-intensity magnetic separation process than with flotation, but liberated oxidized iron minerals were lost with these processes. Wet high-intensity magnetic separation recovered the oxidized iron minerals, but contained a large amount of quartz and other gangue minerals (e.g., talc). More optimization testing could possibly improve on the grade-recovery relationship with the partially oxidized samples such as Bench 18-3 and Bulk 16-1, but the baseline testing concluded that implementing WHIMS as a scavenger system for this type of sample may be impractical due to a low-iron concentrate produced at a low weight recovery. Contrary to the Bulk 16-1 and 19-1 samples, the Bulk 19-2 found better results with flotation than with magnetic separation.

Although the SFDF process had slightly higher recovery at the same silica grade, it was determined that the results from bench testing merited further investigation of the NatFlot process with the Bulk 19-2 material on pilot-scale. This decision was based on the potential to recover oxidized iron minerals (reduction of waste) with a natural pH process (reduction of chemicals to reduce environmental impact). The goals of the pilot program were to investigate the metallurgical performance of the NatFlot process with the Bulk 19-2 sample at pilot-scale and to produce at least 272 kg (600 lbs.) of concentrate at a grade of 4.0-4.5% SiO_2 with no more than 80% of the particles finer than 30 µm for pyrometallurgical testing.

Approximately 8.3 tons of the Bulk 19-2 material was processed with high-pressure grinding rolls processing from 19.1 mm (0.75 inch) to 100% passing 3.4 mm (6 Mesh). At a specific press force of 3.4 N/mm^2 and at a rolls speed of 15 RPM, the grinding energy consumption was calculated at 1.0 kWh/metric ton. Ball mill Bond Work Index testing resulted in an estimated energy consumption of 18.7 kWh/metric ton. Following sample preparation, a total of six continuous pilot-scale tests were executed to investigate the NatFlot process with the Bulk 19-2 material. The total run time was approximately 40 hours and throughput was approximately 0.20-0.24 metric tons per hour. One of the most critical aspects discovered in the early runs was the formation of a large froth volume that was generated, especially in the rougher flotation bank, which caused difficult control of the circuit. Froth control spray bars were added to combat this issue, however, resulted in a lower flotation residence time in the circuit.

After six tests, it was discovered that the NatFlot process was able to produce concentrate at the required silica grade of 4.0-4.5% SiO_2 at pilot-scale, but at a lower weight and iron recovery than projected by the bench testing. While the first five runs allowed for problem solving and optimization of the circuit, Run 6 was the most successful run demonstrating the potential of this technology. The estimated depressant (starch) and collector (amine) dosages were 0.65 kg/metric ton (1.46 lb./LT) and 0.16 kg/metric ton (0.36 lb./LT), respectively. The Run 6 cleaner flotation concentrate sample was composed of 65.8% goethite, 30.2% hematite, 2.9% quartz, and 1.1% calcite. The Run 6 cleaner flotation tailings sample was composed of 45.9% goethite, 17.3% hematite, and 36.8% quartz. A scavenger NatFlot test was conducted with the Run 6 total tails sample. The effect of scavenging was an increase in weight and iron recovery of approximately 70%, while making a finer composite concentrate. Furthermore, scavenging may also be a method for final concentrate silica grade control. The "typical" Minnesota taconite concentrate versus the concentrate produced with NatFlot during the pilot-scale testing can be seen in Table 4.

Table 4: "Typical" Minnesota Taconite Concentrate vs. Concentrate Produced with NatFlot Pilot-Scale Testing

Sample Type	Total Fe (%)	SiO_2 (%)	Phos (%)	LOI (%)		Size (P80, µm)	Blaine (cm^2/g)		Weight Recovery (%)
"Typical Target" for Minnesota Taconite Concentrate	> 67.0	< 5.5	< 0.02	~1-2		~42-46	~1,800-2,200		~28-32
NatFlot Cleaner Concentrate Produced from Pilot Testing	61.4	4.2	0.03	~7.1		42	1,715		27.9
NatFlot Cleaner+Scavenger Concentrate Produced from Pilot and Scavenger Bench Testing (Interpolated to 4.0% SiO_2)	61.5	4.0	0.03	N/A		~35	N/A		47.5

During pilot Run 2 through Run 6, Pace Analytical took a composite sample of the tailings thickener overflow to analyze for various water parameters. The objective of the baseline water quality characterization of tailings thickener overflow was to understand what components could be a potential concern for process water recycle and future water treatment. Of particular interest in this study was the concentration of mercury, amine, major ions (e.g., alkalinity, sulfate, and chloride), and heavy metals during Run 6. It is important to note that the pilot-scale testing was not conducted to optimize the performance of the tailings thickener or control of the tailings thickener. Run 6 found a low-level mercury measurement of 1.8 ng/L and no detected dissolved mercury. Run 6 had an estimated amine concentration of 33.1 mg/L into the process and a measured 1.8 mg/L residual amine remaining in the tailings thickener overflow water, which equated to an approximated removal rate of 94.6%. The amine removal rate was approximately 1.5% lower in Run 6 than in the bench test, which may be at least partially due to the use of the froth control spray bars that reduced the residence time in the flotation circuit. Run 6 measured 75.5 mg/L chloride and 122 mg/L sulfate. Based upon these average measurements, the pilot-scale NatFlot testing with the Bulk 19-2 sample resulted in an increase in chlorides in the tailings thickener overflow, with a very slight increase in sulfate. For Run 6, the testing identified arsenic, chromium, and lead levels of 1.7 µg/L, 4.5 µg/L, and 0.65 µg/L, respectively. No dissolved heavy metals were detected throughout the testing. For Run 6, the chemical oxygen

demand (COD) and biochemical oxygen demand (BOD) levels were 70.6 mg/L and 14.0 mg/L, respectively.

When conducting a basic comparison of the tailings thickener overflow water to the local wastewater treatment plant or Minnesota's Freshwater Quality standards for Class 2A waters, the water quality met regulatory limits with the exceptions of total dissolved solids and diesel range organics (DRO). The total suspended solids metric, however, would likely improve based upon proper operation of the tailings thickener and basic water treatment. COD and BOD were detected in the process effluent, likely due to the amine and starch. The sulfate concentrations were below the Secondary Drinking Water standard of 250 mg/L, but were an order of magnitude higher than the sulfate standard of 10 mg/L applied to Wild Rice waters.

Approximately 296 kg (652 lbs.) of Run 5 cleaner flotation concentrate and 373 kg (822 lbs.) of Run 6 cleaner flotation concentrate were blended together to form a composite sample for bench- and pilot-scale pelletizing and induration testing. Batch balling and mini-pot studies were used to develop a firing cycle that could produce a pellet which meets the physical requirements of a blast furnace pellet. The developed mini-pot firing cycle was then translated to a pot grate cycle to simulate a possible commercial furnace cycle. The pot grate firing cycle was designed to mimic a grate-kiln furnace design, with an extended pre-heat time and increased temperature to compensate for the relatively large amount of goethite compared to typical Minnesota magnetite concentrate. Pellets produced from two pot grate experiments were then analyzed to determine if they meet physical and metallurgical pellet standards.

Mini-pot testing indicated it was possible to produce pellets to meet the physical requirements of blast furnace feed using a mixture of the concentrate produced at pilot-scale with the Bulk 19-2 sample, 6,696 g/metric ton (15 lb./LT) bentonite, and 1.5% coke breeze addition. Two pot grate tests were conducted; one with 1.5% and one with 1.2% coke breeze addition. The green ball quality of the pellets for the pot grate tests met the industry standards of a dry compressive strength above 17.8 N (4 lbf) and drop number above four. The green ball moisture for both trials is higher than the traditional 9-10% seen in Minnesota taconite processing. The porosity of the fired pellets was relatively high at 33.6% and 33.0% for the two tests, respectively. The addition of the coke breeze and the dehydroxilation of the goethite to hematite are the likely causes of the higher porosity compared to typical Minnesota blast furnace pellets. The average of two pot grate trials produced fired pellets that meet the chemical, physical and metallurgical specifications of blast furnace pellets, with the exception of the phosphorus content, as shown in Table 5.

Table 5: "Typical" Minnesota Taconite Pellet vs. Av. Pellets Produced with NatFlot Pilot-Scale Testing

	Chemical Assay					Physical Characteristics			Metallurgical Characteristics		
Sample Type	**Total Fe (%)**	**Fe^{++} (%)**	**SiO_2 (%)**	**Phos (%)**		**Compressive Strength (lbf)**	**Abrasion Index +6.3 mm (%)**		**Reducibility (DR/DT)**	**Swelling (%)**	**LTD Tumble Index +6.3 mm (%)**
"Typical Target" for Minnesota Blast Furnace-Grade Pellet (Acid)	> 65	< 1.0	< 5.5	< 0.02		> 450	> 95		> 0.6	< 20	> 80
Av. NatFlot Pellet Produced from Pilot Testing Cleaner Concentrate (Acid)	65.5	0.44	5.1	0.03		485	95.4		0.66	16.6	96.8

As a consequence of recovering oxidized iron minerals such as hematite and goethite rather than magnetite, the resulting concentrate is different than typical Minnesota magnetite concentrate. Pure magnetite is 72.3% Fe by weight, whereas hematite and goethite are 69.9% and 62.9% by weight, respectively. The iron grade for oxidized iron concentrate produced in this test was approximately 61-62% total Fe while typical taconite or magnetite concentrate is typically above 67% total Fe. The phosphorus content for concentrates produced from oxidized iron formation is higher (~0.03-0.04% phosphorus) than for magnetite concentrate (<0.02% phosphorus). The reason for this decrease in iron and increase in phosphorus is the abundance of goethite in the concentrate.

Besides the changes in chemical assay, oxidized iron minerals like goethite also change the pyrometallurgical characteristics of the material. The presence of goethite and hematite instead of magnetite in the concentrate negatively impacts pellet production rates, production costs, and pellet quality. Goethite is a hydrated iron-oxide mineral, meaning additional energy is required to produce an oxide pellet from goethite-rich concentrate as compared to a typical pellet produced with magnetite-rich concentrate. During induration goethite loses hydroxide and is converted to hematite. The loss-on-ignition (LOI) value for oxidized iron concentrate is higher than for taconite or partially oxidized taconite concentrate. For example, the LOI of the Bulk 19-2 concentrate was approximately seven percent, resulting in seven percent weight loss during the induration process compared to approximately one percent for taconite. This means that for a similar amount of pellet plant feed, oxidized ores would yield fewer tons of pellets than for typical taconite ore. However, the results from this study found an increased concentrate weight recovery from crude ore compared to a typical taconite operation, meaning more tons of pellets could be produced from each ton of crude ore. This could significantly reduce mining- and tailings disposal costs to offset the higher energy consumption during induration. After

indurating, the pellets produced with taconite and oxidized iron concentrates would have similar total iron content but different phosphorus values, based on the feed phosphorus content.

The most significant operational challenge from the pilot-scale NatFlot testing was managing the froth level while controlling the metallurgical performance of the process. The use of the spray nozzles in the feed and discharge ends of the flotation banks effectively controlled the froth volume; however, this also added water to the circuit and decreased the residence time. In future tests, high pressure/low volume spray nozzles would be recommended for this application. Furthermore, it would be advised to use tank cells versus banks of mechanical cells. Utilization of tank cells could reduce bypass and allow better control of the process by tight control on the amount of material floated per stage. The performance of the pilot flotation cells was poor compared to the performance of the bench flotation cells. The rougher froths were higher in iron which made the scavenger circuit a necessity. The scope of the project did not allow for optimization of the circuit which should be considered in any follow-up testing. Furthermore, continued testing of alternative flotation reagent testing should be explored in the future.

In the course of this research, it became apparent that there is a poor understanding of the iron ore resource that lies between historic direct shipping pits and the current taconite pits. Geometallurgical characterization is critical to understanding the remaining iron resource in Minnesota. The iron resource is likely composed of a mixture of taconite, partially oxidized iron formation, and oxidized iron formation. It is essential to understand the mineralogy and liberation characteristics of iron formation in order to determine the direction research should take in order to meet the changing needs of the iron industry in Minnesota. The physical and chemical properties of talc, stilpnomelane, and the common carbonate minerals should assessed for opportunities to make changes to the NatFlot process to improve its ability to process a wider variety of ore types. A combination of physical separation and pyrometallurgical techniques could also be explored for processing partially oxidized iron formation. Furthermore, a separate test program targeting the production of low-silica concentrate could be explored to support the emerging direct-reduced iron (DRI) market. Lastly, pelletizing and induration research should be continued. Assuming that the beneficiation technology shows potential for economic production of iron concentrate at acceptable quality, some areas of research in the near future should include binder evaluation, ore blends, firing cycles, and alternative pellet applications.

In summary, NatFlot performed well with hard water and oxidized ores consisting of goethite, hematite, magnetite, and quartz. NatFlot performed less well when carbonate minerals and certain silicate minerals (talc and stilpnomelane, for example) were present. Based upon the current understanding of the NatFlot process and the oxidized iron resources in Minnesota, it is unlikely that the current taconite waste (tailings) would be amenable to beneficiation with the NatFlot process due to the mineralogy of the waste stream; specifically carbonates (e.g., ankerite) and silicates (e.g., talc and stilpnomelane). It is more likely that areas that have a relatively simple and more fully oxidized mineralogy could be processed with this technology. For example, if zones of highly oxidized materials above or between taconite ore zones are found, NatFlot may be able to process these areas and reduce the costs associated with stripping and/or tailings management. However, these zones may be relatively small and difficult to selectively mine. Furthermore, the recovered concentrate would either need to be blended with the magnetite concentrate, which would likely require modifications to the existing induration

system, or report to a separate agglomeration and pellet plant, which would require significant capital. Regardless, characterization will be key to developing and maintaining a blending strategy capable of producing a consistent concentrate and pellet product at current taconite facilities.

Alternatively, in-situ oxidized iron formation and the remaining legacy direct shipping ore pit walls, legacy stockpiles, and tailings in tailings ponds may be amenable to beneficiation by the NatFlot process, but additional characterization and metallurgical testing is needed to more fully understand these opportunities. To produce an oxide pellet with these resources would also likely require modifications to an existing pellet facility or an entirely new agglomeration and pellet plant. It is important to obtain a greater understanding of the mineralogy of the remaining resource in order to identify the direction of beneficiation research to ensure effective and efficient processing of remaining iron resources in the State. Minnesota has taconite reserves and resources to meet the demands of domestic iron ore production for many decades, but it will likely take many years to develop the technologies needed to assure continued mining, reduction of waste, and a reduction of the environmental impact of iron mining in Minnesota.

Table of Contents

Table of Figures

Table of Tables

Glossary of Terms

- Blaine: measurement of a total surface area of material (in cm^2/g).
- BOD: biochemical oxygen demand.
- BOD5: 5-day BOD test. In this case of this research study, BOD and BOD5 are the same.
- BWI: Bond Work Index (in kWh/ton).
- Ca^{++}: calcium ions.
- COD: chemical oxygen demand.
- Deslime: removal of fine particles, or slimes, of particles less than 5 µm in size.
- DI: de-ionized.
- Dist.: distribution.
- DT: Davis Tube.
- DTT: Davis Tube test.
- Fe^{++}: ferrous iron (FeO).
- Ferrimagnetic: a material is one that has populations of atoms with opposing magnetic moments, as in antiferromagnetism; however, in ferrimagnetic materials, the opposing moments are unequal and a spontaneous magnetization remains (Wikipedia, Ferrimagnetism, 2018).
- Flocculation: process by which fine particulates are caused to clump together into a floc (Wikipedia, Flocculation, 2018).
- Head: raw material; starting feedstock.
- HPGR: high pressure grinding rolls.
- Hydrocycloning: a vessel used for separating two liquids with different densities, by the circular movement of fluid (Collins Dictionary).
- Induration: heat hardening of the concentrate to shippable pellets.
- LIMS: low-intensity magnetic separation.
- LOI: loss on ignition, or the amount of mass lost in air at a specific temperature.
- LTD: ISO 13930 – dynamic test for low temperature disintegration.
- Mg^{++}: magnesium ions.
- NatFlot: natural pH flotation developed by the UMD NRRI.
- NRRI: Natural Resources Research Institute.
- O/F: overflow.
- Oxidized iron formation: iron formation that is composed dominantly of hematite, goethite, and quartz (chert).
- P80: 80% particle size (in microns).
- Paramagnetic: weakly attracted by an externally applied magnetic field, and form internal, induced magnetic fields in the direction of the applied magnetic field (Wikipedia, Paramagnetism, 2018).
- Partially oxidized iron formation: iron formation that is composed of variable amounts of quartz (chert), magnetite, hematite, goethite, carbonate, and silicate minerals.
- R40: ISO 4695 – determination of reducibility.
- Rec.: recovery.
- Sat Mag Fe: Satmagan magnetic iron.

- Scavenger: recovering valuable material from tailings (waste) streams.
- SEM: scanning electron microscope.
- SFDF: selective flocculation-deslime-flotation.
- Taconite: iron formation in which the major iron oxide is magnetite and it responds well to fine-grinding with reasonable concentrate weight recovery (>25%) with low-intensity magnetic separation.
- Tailings: waste or less-valuable products from a process.
- TKN: total Kjeldahl nitrogen.
- Total Fe Recovery: the percentage of total iron that was recovered in a process as a concentrate product.
- TSS: total suspended solids.
- U/F: underflow.
- UMD NRRI: University of Minnesota Duluth Natural Resources Research Institute.
- Weight Recovery: the percentage of mass that was recovered in a process as a concentrate product.
- WHIMS: wet high-intensity magnetic separation.
- WIDRO: diesel range organics.
- WRA: whole rock analysis.
- XRD: X-ray diffraction.
- XRF: X-ray fluorescence.

Glossary of Units

- cm: unit of length in centimeters (1 cm = 0.39 in = 10 mm).
- cm^2: unit of area in centimeters-squared (1 cm^2 = 0.16 in^2 = 100 mm^2).
- g: unit of mass in grams (1 g = 0.022 lbs. = 0.001 kg).
- gal: unit of volume in gallons (1 gal = 3.79 L).
- h: unit of time in hours (1 h = 60 min). Also abbreviated as hr.
- HP: unit of power in horsepower (1 HP = 0.746 kW = 745.7 W).
- in: unit of length in inches (1 in = 2.54 cm = 25.4 mm).
- kg: unit of weight in kilograms (1 kg = 2.2 lbs. = 1,000 g).
- kW: unit of power in kilowatts (1 kW = 1.34 HP = 1,000 W).
- kWh: unit of energy in kilowatt-hours (1 kWh = 1.34 HPh).
- mg/L: unit of concentration in milligrams per liter (1 mg/L = 1 ppm).
- mm: unit of length in millimeters (1 mm = 0.04 in = 0.1 cm).
- mm^2: unit of area in millimeters-squared (1 mm^2 = 0.0016 in^2 = 0.01 cm^2).
- L: unit of volume in liters (1 L = 0.26 gal).
- lbf: unit of force in pounds-force (1 lbf = 4.45 N = 0.45 kgf).
- lbs.: unit of mass in pounds (1 lbs. = 454 g = 0.45 kg).
- LT: unit of mass in long tons (1 LT = 2,240 lbs = 1.02 metric tons).
- m: unit of length in meters (1 m = 3.28 ft = 100 cm).
- min: unit of time in minutes (1 min = 0.02 h).
- mt: unit of mass in metric tons (1 mt = 0.98 LT = 1.10 st).
- N: unit of force in Newtons (1 N = 0.22 lbf = 0.102 kgf).
- ppm: unit of concentration in parts-per-million (1 ppm = 1 mg/L).
- RPM: unit of rotation in rotations per minute.
- st: unit of mass in short ton (1 st = 2,000 lbs. = 0.91 metric tons).
- ts/hm^3: unit of specific throughout for HPGR, defined as the throughput (in tons/hour) divided by the diameter (in meters), width (in meters), and speed of the rolls (in meters/second).
- µg/L: unit of concentration in micrograms per liter (1 µg/L = 0.001 ppm).
- µm: unit of length in microns (1 µm = 0.001 mm).
- W: unit of power in watts (1 W = 0.0013 HP = 0.001 kW).

Introduction and Background

There are a significant amount of magnetite reserves remaining on the Mesabi Iron Range of Minnesota to satisfy demand in the next few decades (US Geological Survey, 2018); however, most taconite operations are challenged by the presence of partially oxidized iron formation that does not respond well to the efficient low intensity magnetic separation and induration methods employed on the iron range. As a result, significant iron units are not being extracted from Minnesota iron ores. Oxidation affects the processing of iron ore. Partially oxidized iron ore is often treated as waste or, if mined, causes reduced weight recovery of concentrate due to non-magnetic iron oxides (hematite and goethite) not being recovered and reporting to tailings. Other iron resources in Minnesota also include in-situ oxidized iron formation and legacy direct shipping pit walls, stockpiles, and tailings. The *Pilot-Scale Demonstration of Increasing Iron Recovery from Minnesota Oxidized Iron Resources* project investigated methods for increasing iron recovery from oxidized iron resources located at the western end of the Mesabi Range. This report documents the bench- and pilot-scale research into one possible method, natural pH flotation ("NatFlot"), for increasing iron recovery from oxidized iron ore. The report also discusses the potential applications and limitations of the NatFlot process.

Global Iron Ore Market

Iron ore is sold as lump, fines, or pellets. Global reserves of iron ore are 170 billion metric tons. Within the United States, there are 2.9 billion tons of iron ore reserves. Australia has reserves of 50 billion metric tons and Brazil has reserves of 23 billion metric tons. Australia and Brazil provide iron ore to China and India. China has a big influence on the iron ore and steel market. China produced 704 million metric tons of pig iron and 808 million tons of raw steel in 2016, while the United States produced 22 million metric tons of pig iron and 78 million metric tons of raw steel. The iron ore mining in Minnesota and Michigan supplies iron ore pellets for the blast furnaces on the Great Lakes, Illinois, and Pennsylvania. In 2016, the US produced 278.5 million metric tons of raw steel. From this production, the US exported 8.4 million metric tons of steel mill products while 36.1 million metric tons of steel mill and semi-finished products (U.S. Geological Survey, 2018).

Minnesota Iron Range Geology

The Mesabi Iron Range is one of the largest iron ranges in the world. It is 193 km (120 mi) long, averages 1.6-3.2 km (1-2 mi) wide, and comprises rocks that belong to the Paleoproterozoic Animikie Group. The Animikie Group consists of three conformable geological formations: the Pokegama Formation at the base; the Biwabik Iron Formation in the middle; and the overlying Virginia Formation (Severson et al., 2003). Figure 1 shows the location of the Biwabik Iron Formation in Minnesota.

Figure 1: Location of Biwabik Iron Formation in Minnesota (modified after Peterson, 2018)

Since the early 20th century, the Biwabik Iron Formation has been subdivided into four informal members (Figure 2) referred to as (from bottom to top): Lower Cherty, Lower Slaty, Upper Cherty, and Upper Slaty (Wolff, 1917). The cherty members are typically a granular (sand-sized) and thick-bedded (beds ≥ several cm thick); whereas, the slaty members are typically fine-grained (mud-sized) and thin-bedded (≤1 cm thick beds). Note that "slaty" is a miner's term to denote thin-bedded rocks that part parallel to bedding. In this usage, slaty is not indicative of metamorphism or slaty cleavage. The cherty members are largely composed of chert, iron oxides (dominantly magnetite and hematite), and iron silicates, while the slaty members are composed of iron silicates, magnetite, and iron carbonate with local chert beds. Both cherty and slaty iron-formation types are interlayered at all scales, but one rock type or the other predominates in each of the four informal members, and they are so-named for this dominance (Severson et al., 2009). Taconite mining is concentrated on the cherty members of the iron-formation but also occur in the Upper Slaty in the eastern part of the Biwabik Iron Formation. Figure 2 displays the Biwabik Iron Formation geological column.

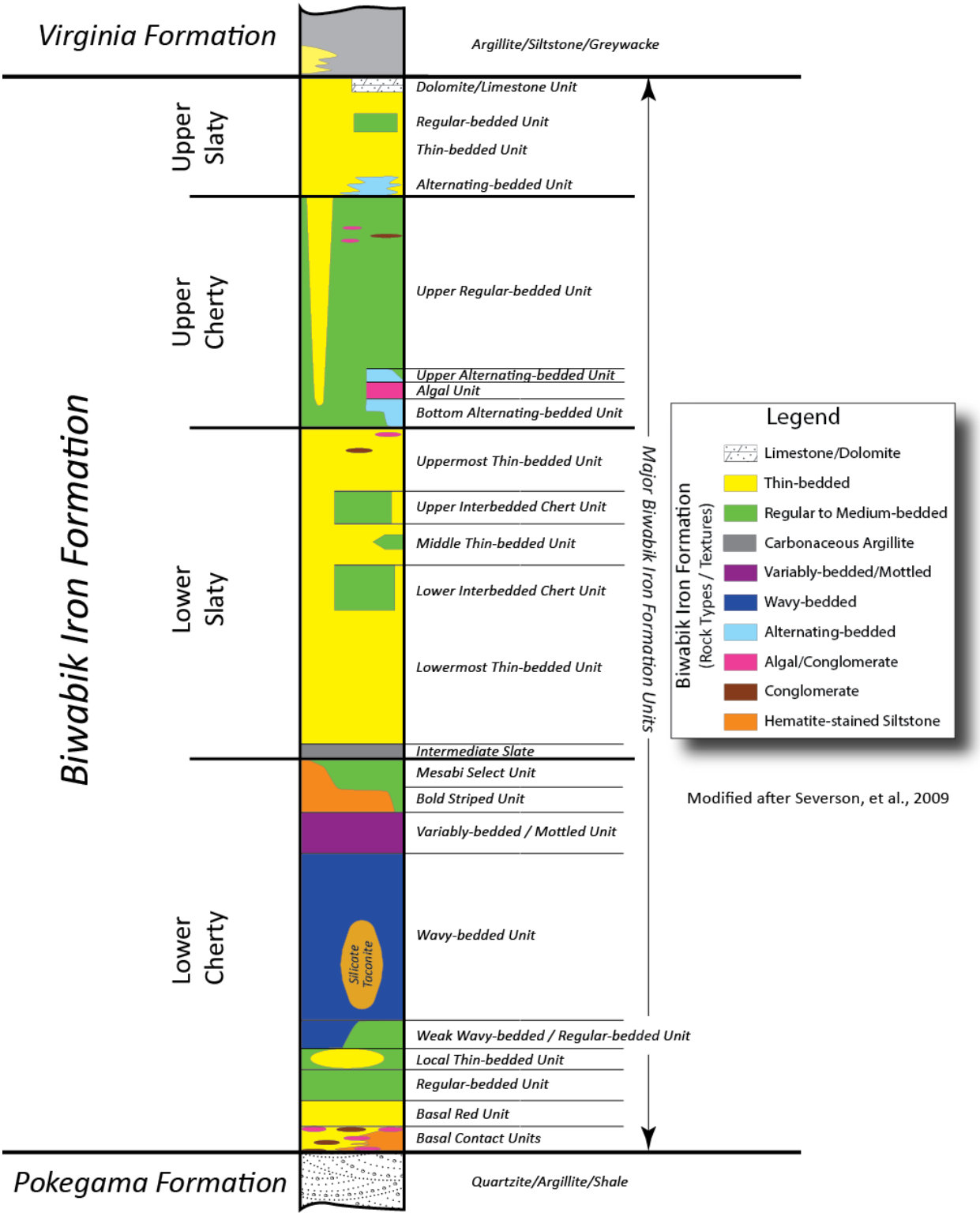


Figure 2: Biwabik Iron Formation Geological Column (after Severson et al., 2009)

The Biwabik Iron Formation has been variably oxidized along most of its length. Oxidation occurs adjacent to faults and at the surface of the iron formation. Oxidation results in the conversion of magnetite into hematite and/or goethite while carbonate and silicate minerals are

converted to goethite. The depth of the oxidation varies along the length of the Biwabik Iron Formation, but tends to be deeper along the western portions of the Mesabi Iron Range. Oxidation affects the processing characteristics of the iron formation and pellet quality.

For this report, the term *oxidized iron formation* is used for iron formation that is composed dominantly of hematite, goethite, and quartz (chert). The term *partially oxidized iron formation* is used for iron formation that is composed of variable amounts of quartz (chert), magnetite, hematite, goethite, carbonate, and silicate minerals. The term *taconite* refers to iron formation in which the major iron oxide is magnetite and it responds well to fine-grinding with reasonable concentrate weight recovery (>25%) with low-intensity magnetic separation. The definitions for the iron formation used in this report are summarized in Table 6.

Table 6: Definitions for Iron Formation Types Used in this Report

Iron Formation Type	Iron Oxides	Gangue Minerals	Response to Low-Intensity Magnetic Separation
Oxidized	Hematite, goethite	Quartz (chert)	Low weight recovery
Partially-Oxidized	Magnetite, hematite, goethite	Quartz, carbonate(s), silicate(s)	Moderate weight recovery, low quality concentrate
Taconite	Magnetite	Quartz, carbonate(s), silicate(s)	High weight recovery, high quality concentrate

A list of common “ore” (e.g., valuable) and “gangue” (e.g., waste) minerals associated with iron formation relevant to this study can be seen in Table 7. Of these minerals, magnetite is highly magnetic while hematite and goethite are weakly magnetic, also known as para-magnetic.

Table 7: Common "Ore" and "Gangue" Minerals Associated with Iron Formation

Ore Mineral	Type	Chemical Analysis	Magnetic Properties
Magnetite	Iron Oxide	Fe_3O_4	Highly-Magnetic
Hematite	Iron Oxide	Fe_2O_3	Para-Magnetic
Goethite	Iron Hydroxide	FeO(OH)	Para-Magnetic

Gangue Mineral	Type	Chemical Analysis	Magnetic Properties
Quartz	Silicate	SiO_2	Non-Magnetic
Talc	Silicate	$Mg_3Si_4O_{10}(OH)_2$	Non-Magnetic
Minnesotaite	Silicate	$(Fe^{2+},Mg)_3Si_4O_{10}(OH)_2$	Non-Magnetic
Siderite	Carbonate	$FeCO_3$	Non-Magnetic
Ankerite	Carbonate	$Ca(Fe,Mg,Mn)(CO_3)_2$	Non-Magnetic
Stilpnomelane	Phyllosilicate	$K(Fe^{2+},Mg,Fe^{3+})_8(Si,Al)_{12}(O,OH)_{27}\cdot n(H_2O)$	Non-Magnetic
Greenalite	Phyllosilicate	$(Fe^{2+},Fe^{3+})_{2-3}Si_2O_5OH_4$	Non-Magnetic
Apatite	Phosphate	$Ca_5(PO_4)_3(F,Cl,OH)$	Non-Magnetic

Direct shipping ores were highly oxidized ores in which most of the magnetite had been converted to hematite and goethite and the carbonates and iron silicates converted to goethite. During the oxidation process silica became mobile and was removed from the iron formation further increasing the iron grade of the ore. The high-grade oxidized ores (direct shipping ores) were largely depleted and the iron mining industry turned its attention to taconite ores. The direct shipping ore pits were concentrated in the western end of the Mesabi Range. The location

of the historic natural (direct) ore pits and the current taconite pits in the western Mesabi Range are shown in Figure 3.

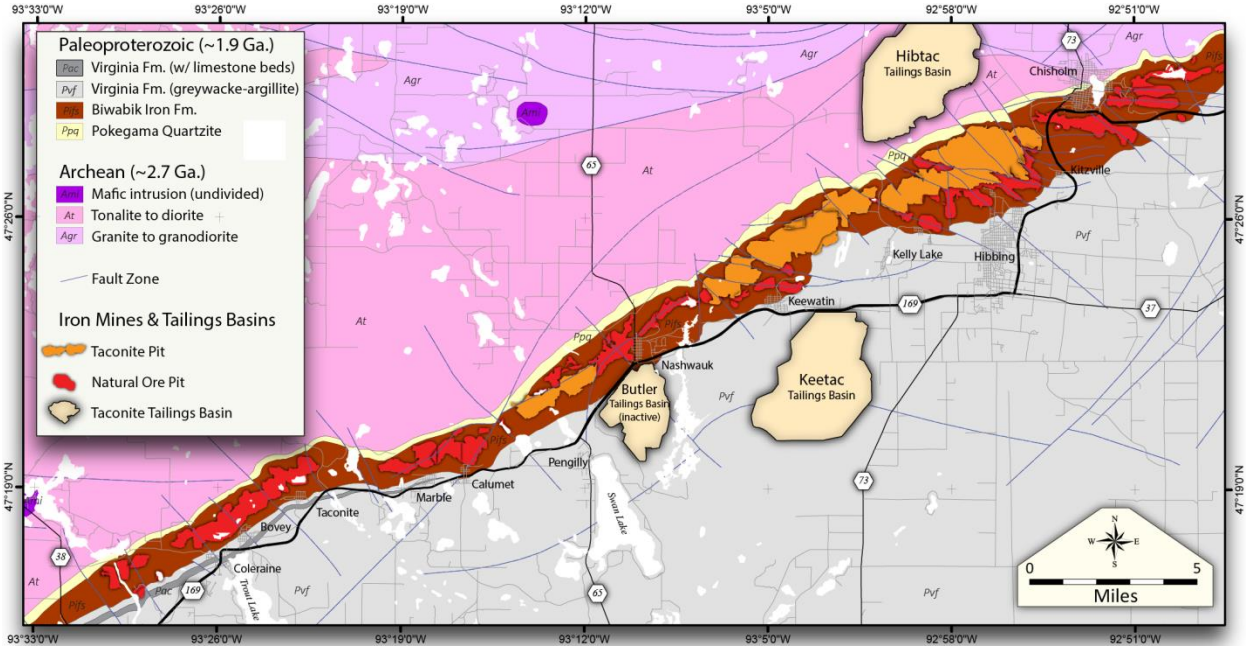


Figure 3: Historic Natural (Direct) Shipping Ore Pits and Current Taconite Pits in the Western Mesabi Range (after Peterson, 2018)

Magnetic taconite ores are not oxidized and responds well to fine grinding and recovery of magnetite by low intensity magnetic separation. A large part of the remaining iron ore resource is composed of partially oxidized iron formation. Low intensity magnetic separation and induration (pelletizing) processes are very sensitive to low levels of oxidation because of the replacement of magnetite by hematite. This is because the oxidation of magnetite to hematite is an exothermic process and provides much of the heat needed during the induration process. Therefore, partially oxidized iron formation is categorized by current taconite mining operations as waste. The location of the taconite ore pits in the Mesabi Range is shown in Figure 4.

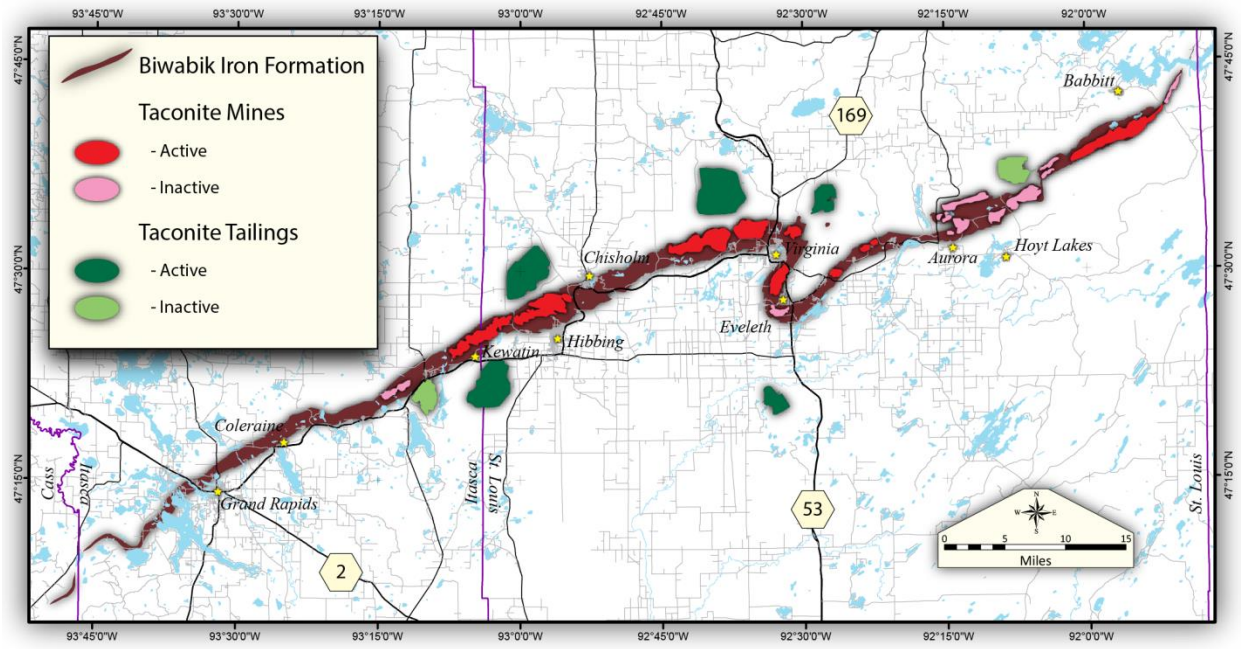


Figure 4: Active and Inactive Taconite Mines and Tailings Pits (after Peterson, 2018)

Sources of Oxidized Iron

Oxidized iron resources can be classified into two broad groups as either waste or in-situ (or in-the-ground) resources. Furthermore, waste iron resources can be subdivided into waste products associated with early mining (direct shipping and wash ores) or more recent taconite mining.

In-Situ Oxidized Iron Resources

In-situ oxidized iron resources consist of oxidized and partially oxidized iron formation that does not respond well to current fine grinding and low-intensity magnetic separation processes. These resources are poorly understood since direct shipping miners were not interested in ores that were not almost completely oxidized and taconite miners are not interested in ores that are not almost completely un-oxidized. Some of the in-situ resources may respond well to alternative beneficiation methods.

Early Mining – Waste Oxidized Iron Resources

Waste oxidized iron resources from early mining consist of lean "ore" waste piles, jig rock tailings piles, and tailings within tailings ponds. The iron oxides that are contained in the jig rock tailings piles and the tailings ponds are generally coarser-grained than the iron oxides found in tailings from taconite operations. However, the oxides contained within these tailings are commonly mixtures of hematite and goethite with minor amounts of magnetite. Hence, the concentrate produced from these tailings would be of lower quality than the concentrate produced from taconite operations. The mineralogy and processing characteristics of the lean "ore" waste piles from this period are not well understood and the response to any beneficiation method is unknown.

Taconite Waste Oxidized Iron Resources

Taconite waste oxidized iron resources include lean "ore" waste piles and tailings in tailings ponds. Iron formation from taconite operations is wasted because it may be low-grade, have a low weight recovery of concentrate with fine grinding and low intensity magnetic separation, or produce a concentrate that is sufficiently oxidized to negatively impact induration and thus, pellet production and quality. Although already ground to a fine size, taconite tailings are lower grade than tailings from early mining operations and provide less opportunity for economic recovery of iron units because it contains lesser iron oxide units. When hematite is present in taconite ores it is in many cases liberated, and could potentially be recovered from tailings by alternative beneficiation methods.

Processing Technologies Overview

Current fine particle separation technologies for processing of iron minerals mainly rely on differences in the magnetic susceptibility, specific gravity, and surface properties of the minerals. As described above, the typical taconite process relies on the separation of magnetite (a ferrimagnetic mineral) from gangue minerals by fine grinding and low-intensity magnetic separation (LIMS). However, para-magnetic iron minerals, namely hematite and goethite, do not report to the concentrate stream with the conventional LIMS process. Generally speaking, Minnesota taconite operations produce a taconite concentrate of greater than 67% total iron, less than 5.5% SiO_2, and less than 0.02% phosphorus at a weight recovery of 28-32%. The required liberation size for a typical Minnesota taconite is approximately 80% passing 44 µm (325 Mesh), with the taconite concentrate having a surface area (Blaine) of approximately 1,800-2,200 cm^2/g.

Historically, reduction roasting and magnetic separation was found to effectively recover iron minerals on a variety of oxidized iron samples, but the combination is an energy intensive process and is more costly than other beneficiation technologies. Two other technologies used for beneficiation of fine-liberating[2] oxidized iron minerals are wet high-intensity magnetic separation (WHIMS) and flotation. High-intensity magnetic separation technology has been successfully employed around the world for the processing of para-magnetic material, but has been known to have plugging issues in operation and often does not recover fine magnetic particles, reducing the effectiveness of the technology with fine liberating minerals (particles less than 30 µm). Furthermore, there have been previous experiences that WHIMS alone cannot produce a consistent silica concentrate grade for the blast furnace pellet market (Hopstock, 1979).

Flotation systems are also used around the world for the recovery of fine-grained oxidized iron minerals. Mining companies in Brazil, for example, use desliming via hydrocycloning before reverse flotation of silica at a pH of 9.5-10.5. The hydrocycloning process is unselective and therefore, this type of processing flowsheet has lower iron recovery. The selective flocculation-deslime-flotation (SFDF) process currently employed at the Tilden Mine in Michigan uses selective flocculation to reduce the iron losses in desliming. This process requires an elevated pH between 10-11 and low amount of calcium ions (5-10 ppm Ca^{++}). It is selective and achieves total iron recovery typically above 70%, but has high reagent costs and difficulties due to the

[2] Generally defined as liberating at a particle size less than 105 µm (150 Mesh).

tight control of the reuse water system. In the 1970's, the Hanna Mining company discovered improved selectivity in cation flotation by using well water with a pH range of about 7 to 8 (Dicks and Morrow, 1978). However, the selective flocculation-deslime portion of the process still required softened water at an elevated pH. Maintaining a dual-water system would be very difficult, if not impractical, at a production facility.

Natural pH (NatFlot) Technology Overview

The University of Minnesota Duluth Natural Resources Research Institute (UMD NRRI) has previously developed a process that has been shown to remove quartz, apatite, and some gangue minerals while recovering magnetite, hematite, and goethite in a limited number of samples (Mlinar and Petersen, International Patent Application No. PCT/US2017/014372). This flotation process, known as NatFlot, is conducted at the natural slurry pH[3] in contrast to the industry proven elevated-pH selective flocculation-deslime-flotation (SFDF) process.

The standard NatFlot system evaluated with this research is a reverse cationic silica flotation technology typically conducted in four-to-six individual flotation stages. Conditions of the NatFlot system maintained during this research include:

- No independent conventional deslime.
- No wet high-intensity magnetic separation (WHIMS).
- No frother reagent.
- No pH control (e.g., flotation conducted at a natural pH between 6-9).
- No recirculation of streams.

The generic NatFlot process block diagram can be seen in Figure 5, below.

[3] pH is not modified or controlled.

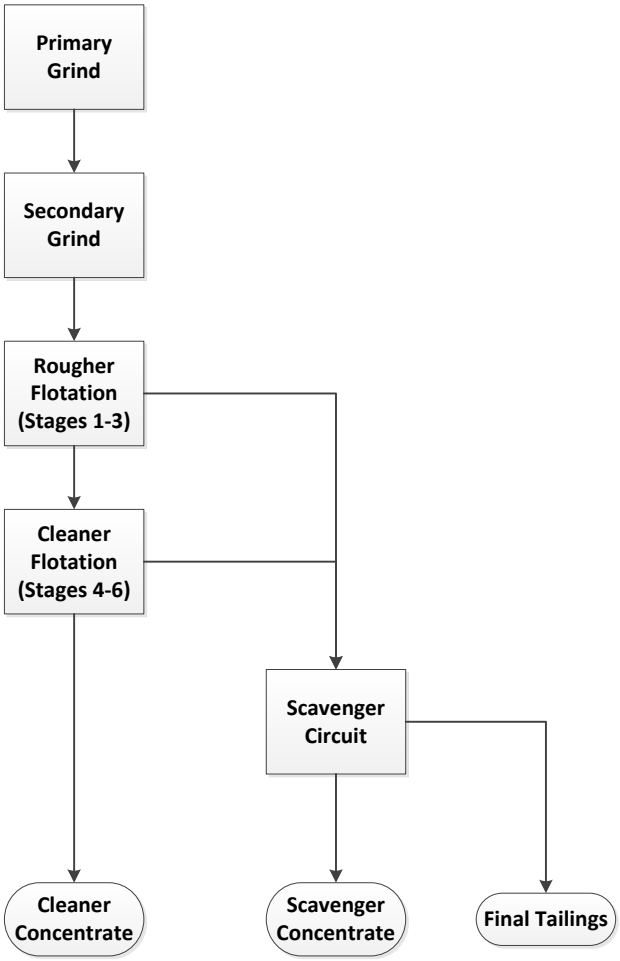


Figure 5: Generic NatFlot Process Block Diagram

The standard NatFlot system uses a four reagent system:

- Water: elevated amount of divalent cations (e.g., Ca^{++}, Mg^{++}) at a natural pH.
- Dispersant (apatite activator)[4]: polyphosphate (e.g., Glass H).
- Iron depressant: polysaccharide (e.g., causticized modified corn starch).
- Silica collector: amine (e.g., 1-Propanamine, 3-(octyloxy); Isodecylether propylamine, 3-(Isodecyloxy) propylammonium acetate).

The NatFlot and SFDF processes are similar in the fact that both technologies rely on removing the gangue from the iron oxide minerals in a reverse flotation process. The SFDF process uses a desliming step at pH of 10.5 where starch selectively flocculates the iron mineral and dispersants reject the fine silica into the deslime thickener overflow. The deslimed material then proceeds to a high-pH reverse flotation circuit for final beneficiation. In place of selective flocculation-deslime, the NatFlot system relies on divalent cations to achieve selectivity in the reverse silica flotation process because selective flocculation-deslime is not effective at a natural pH.

The NatFlot and SFDF flotation processes were both developed for processing of oxidized iron formation. NatFlot was designed specifically for rejecting quartz while recovering hematite and

[4] Polyphopshate is only used only when apatite is present in the sample.

goethite. Previous testing during the development of NatFlot found that the process typically requires fine grinding for liberation of the gangue minerals, flotation times of greater than 15 minutes on bench-scale, and is not selective with some silicate and carbonate minerals (Petersen and Mlinar, 2016). Over thirty different gangue minerals (besides quartz) have been identified in taconite and partially oxidized iron ore in the Minnesota iron range. Neither the NatFlot nor SFDF processes were developed to remove all of the associated gangue minerals. That would be difficult, as the flotation of each gangue mineral may require a different set of conditions. Therefore, this research project was focused on finding various oxidized iron formation samples to use for comparative bench-scale research and pilot-scale demonstration.

Research Motivation

The UMD NRRI's mission is to deliver research solutions to balance economy, resources and environment for resilient communities. UMD NRRI focuses on one or more of the three primary research goals in each of its research projects:

- Reduction of waste: investigate, develop, and demonstrate technologies that may allow for higher yield to reduce production cost, cost avoidance, and repurpose of products and by-products.
- Resource stewardship: investigate, develop, and demonstrate technologies that may reduce environmental impact as differentiator for competitive advantage.
- Broaden portfolio and create higher value: investigate, develop, and demonstrate technologies that may allow moving up the value chain to drive economic resilience and retain value within the state of Minnesota.

Current Minnesota iron mining companies avoid mining or waste the iron formation containing hematite and goethite (non-magnetic minerals). Some of this waste is avoided in the mine plan and left in place while some is stripped and stockpiled in lean iron "ore" stockpiles. If mined, the non-magnetic iron minerals report to the tailings (waste) ponds. Furthermore, magnetite reserves will continue to be depleted over the course of the next few decades (Minnesota Department of Revenue, 2018). In an effort to reduce waste and utilize existing Minnesota iron resources, the UMD NRRI undertook a research project to investigate technologies that may allow for the recovery of oxidized iron minerals from Minnesota iron resources. This research project would focus on two of the UMD NRRI research goals: reduction of waste and resource stewardship. Some of the potential benefits of recovering oxidized iron minerals include:

- Increased utilization of existing Minnesota iron resources;
- Expanding mine life;
- Reduction of iron mining tailings volume; and
- Reduction in production cost due to increasing total iron recovery.

The NatFlot system was of particular interest in this study, as this technology has been shown to recover oxidized iron minerals at a reduced pH. Implementing a process that occurs at natural pH may result in the elimination of pH modifier chemicals (e.g., sodium hydroxide, carbon dioxide) and dispersants that have associated costs and potential environmental and operational safety impacts. Lastly, controlling the water system in a selective flocculation-deslime system requires great care and operational cost. Utilizing a process at a natural pH that is applicable to hard water may result in a more robust total iron recovery process at lower economic costs based

on previous bench-scale research conducted by UMD NRRI (Mlinar and Petersen, International Patent Application No. PCT/US2017/014372).

Project Goals, Objectives, and Deliverables

The intent of this project was to evaluate the NatFlot process on various oxidized iron resources on the west end of the Mesabi Iron Range to determine applicability of this technology on Minnesota oxidized iron resources. This part of the Mesabi Iron Range was chosen because it contains a large amount of variably oxidized iron formation (Zanko et al., 2003). NatFlot was evaluated for metallurgical performance as compared to traditional processing technologies. UMD NRRI engaged with industry geologists to acquire test samples. Testing was conducted at the bench-scale first, then by a continuous pilot-scale operation on a scale that is relevant to industry. Pilot-scale testing required samples of at least 10 tons each.

Project objectives included:

- Demonstrate technology for increasing iron-recovery from Minnesota oxidized iron ore resources at bench- and pilot-scale;
- Provide baseline induration data; and
- Characterize the process water and tailings from the pilot circuit testing.

Project deliverables included:

- Acquire and characterize samples;
- Optimize process on bench-scale;
- Complete pilot-scale testing; and
- Provide final report.

Materials and Methods

Testing Samples Introduction

The technical team engaged with industry geologists and metallurgists for the selection of bench- and pilot-scale materials for this research project. The locations for test samples were limited to the western portion of the Mesabi Iron Range, specifically the iron resources between Grand Rapids, Minnesota, and Hibbing, Minnesota, as this area contains a large amount of oxidized iron resource (Zanko et al., 2003). The test material was to contain oxidized iron minerals (e.g., hematite, goethite) and not satisfy the typical taconite ore cut off criteria, including Davis Tube concentrate silica grade, concentrate weight recovery, and/or amount of oxidized iron remaining in concentrate. For this study, taconite ore was defined as iron formation that produced a Davis Tube concentrate with less than 5.5% SiO_2 and a weight recovery above 25%, at a particle size no more than 80% passing 44 µm (325 Mesh). Each mine has criteria for the amount of oxidation they can accept in the iron concentrate; this information was supplied by the operation's geologists or metallurgists. Bench and bulk sample selection was limited to samples available in stockpiles or pit faces. A total of seven bench-scale (drill core or "grab") samples and three pilot-scale bulk (>10 ton each) samples were acquired for this test program; the sampling details and sample characterization are discussed in the *Results* section, below.

Solids Characterization Methods

The following methods were utilized to characterize materials in this study:

- Sample preparation for chemical analysis: >95% passing 74 µm with ring mill processing.
- NRRI chemical analysis:
    - Total iron: internal NRRI titration procedure with certified calibration standards; results are estimated 0.0+0.3%.
    - Ferrous iron: internal NRRI titration procedure with certified calibration standards. Estimated level of detection is 0.2%.
    - Magnetic iron: Rapiscan Systems/Satmagan 135; internal NRRI procedure with NRRI calibration standards. Estimated level of detection is 0.1%.
    - Metal oxides: JY Horiba ICP-OES; internal NRRI procedure with certified calibration and check standards.
        - Al_2O_3: estimated level of detection is 0.04%.
        - CaO: estimated level of detection is 0.2%.
        - MgO: estimated level of detection is 0.5%.
        - Phosphorus (phos): estimated level of detection is 0.006%.
        - SiO_2: estimated level of detection is 0.5%.
    - Metals in solution: JY Horiba ICP-OES; internal NRRI procedure with certified calibration and check standards.
    - Loss-on-ignition (LOI): internal NRRI titration procedure with certified calibration standards. Estimated level of detection is 0.05%.
- Screen analyses: 20.3 cm (8 inch) Gilson-style metallurgical screens with rotap.
- Blaine analyses: Precision Scientific Company Category 24360.
- Whole Rock Analysis (WRA): Activation Laboratories (Ancaster, Ontario); packages WRA+ICP 4Litho, 4B1, 4B-INAA, 4F FeO. (Activation Laboratories Ltd., 2018).
- X-ray diffraction (XRD): samples for XRD analyses were ground to -74 µm (-200 Mesh). A split of -74 µm (-200 Mesh) material was ground further to approximately -37 µm (-400 Mesh) using an agate mortar and pestle. Scans were collected from 5° to 50° 2θ, 0.02° step size, 0.5 second dwell time, at 40 mA, and 45 kV. XRD scans were collected of each sample using a Phillips X'pert MPD theta-theta diffractometer, at the Research Instrumentation Laboratory, University of Minnesota, Duluth. Background was removed and phases matched using Sieve+ software (ICDD) and the PDF4+ powder diffraction database (ICDD). The abundance of minerals identified by XRD was determined by Rietveld analysis using Siroquant V4 (Sietronics).
- Scanning Electron Microscope (SEM): Stratified polished mounts were prepared for SEM analysis. Stratified polished mounts help to eliminate bias in sample preparation due to size and density variation. Images and EDS spectra were collected on a JEOL JSM-6490LV SEM with Oxford Instruments Inca EDS System. The SEM was selected to identify the distribution of phosphorus in the samples. The elemental detection limits of this configuration of SEM are approximately 0.1 to 0.2 weight percent with a spatial resolution of approximately 5 microns.
- Handheld XRF: Thermo Scientific Niton XL3t GOLDD+,Ag anode (6-50 kV, 0-200 µA max), stationary (bench-top) test stand

- Low temperature disintegration (LTD): ISO 13930 – dynamic test for low temperature disintegration (LTD) (ISO 13930, 2015).
- Reducibility: ISO 4695 – determination of reducibility of pellets (R40) (ISO 4695, 2015).
- Swelling test: ISO 4698 – determination of the free-swelling index (ISO 4698, 2007).
- Pellet porosity: internal NRRI titration water displacement procedure.

Water Characterization Methods

Water characterization was conducted by NRRI at the Coleraine Research Laboratory and by Pace Analytical (Virginia, Minnesota). At NRRI, the JY Horiba ICP was used for the analysis for total calcium and magnesium ions.

At Pace Analytical, the following methods were used for water characterization:

- Low-level mercury (raw sample and dissolved in water): EPA 1631E (2001) (EPA: Method 1631, 2002).
- Amine: ASTM D2327 (ASTM: D2327-80).
- Anions: EPA 300.0 (EPA: Method 300.0, 1993).
- Alkalinity: SM 2320B (NEMI: 2320 B, 2005).
- Chemical oxygen demand (COD): SM 5220D (NEMI: 5220D).
- Biochemical oxygen demand (BOD): SM 5210B (Aggregate Organic Constituents).
- Total suspended solids (TSS): SM 2540D (1997) (Standard Methods for the Examination of Water and Wastewater, 1997).
- Total Kjeldahl nitrogen (TKN): EPA 351.2 (EPA: Method 351.2, 1993).
- Nitrate: EPA 353.2 (EPA: Method 353.2, 1993).
- Total phosphorus: EPA 365.1 (EPA: Method 365.1, 1993).
- Diesel range organics (DRO): WI Mod DRO (WDRO C10-C28) (Wisconsin DNR: Modified D R O Method for Determining Diesel Range Organics, 1995).
- pH: pH at 25 degrees C.
- Total nitrogen: TKN+NO_3+NO_2 Calculation.
- Specific conductance: SM 2510B (NEMI: 2510B).
- Metals (raw sample and dissolved in water): EPA 200.7 (EPA: Method 200.7, 1994).
- Heavy metals (raw sample and dissolved in water): EPA 200.8 (EPA: Method 200.8, 1994).
- Naphthalene: EPA 8260B (EPA: Method 8260B, 1996).

Flotation Chemical Reagents

The most commonly used flotation chemical reagents used in this testing program were the following:

- Israel Chemicals Limited (ICL) sodium hexametaphosphate (aka Glass H) prepared to 5% solution strength with deionized (DI) water (bench testing).
- Grain Processing Corporation (GPC) Supercore S12F modified corn starch prepared via causticization method to 1% solution strength with deionized (DI) water (bench testing) or to 2% solution strength with tap water (pilot testing).

- Air Products Tomamine PA-12 mono amine prepared to 1% solution strength with DI water (bench testing).
- Clariant Flotigam EDA mono amine prepared to 1% solution strength with DI water (bench testing) or to 1% solution strength with tap water (pilot testing).
- Aluminum sulfate flocculant prepared to 1% solution strength with DI water.
- Neo Solutions NS6850 polymer flocculant prepared to 0.1% solution strength with DI (bench testing).
- Neo Solutions NS6850 polymer flocculant prepared to 0.1% solution strength with tap water (pilot testing).

The specific reagent type and reagent preparation procedure for each chemical was chosen based upon the chemical producer's suggestions.

Bench-Scale Flotation Methods

Unless otherwise noted, all baseline, optimization, and comparative bench-scale flotation testing was conducted by staged-amine addition with the end-point determined by the flotation operator. All of the flotation tests throughout this research program were conducted by the same flotation operator to maintain consistency in bench-test results. The important elements tracked via chemical analysis were total iron, silica, and phosphorus, as these are three of the most critical elements of productivity and quality-control in iron- and steel-making. The methods used for comparing the performance of flotation tests were a combination of grade vs. recovery plots and test data interpolated to a specified concentrate silica target.

Pyrometallurgical Materials

Materials used during the pelletization and induration studies included:

- Iron ore concentrate produced at pilot-scale (discussed below);
- Coke breeze: Added to the concentrate to compensate for energy that would have been generated during oxidation of magnetite to hematite; if magnetite were present. The coke breeze was ground to 100% passing 150 µm (100 Mesh); and
- Bentonite: added at 6.7 kg/metric ton (15 lb./LT) of dry concentrate. Coleraine "house" bentonite was used for the study.

Water System

It was found during the development of the NatFlot process that the divalent cations (e.g., Ca^{++}, Mg^{++}) that improved process recovery could either be naturally occurring or added as a salt to elevate the ion levels (Mlinar and Petersen, International Patent Application No. PCT/US2017/014372). During this testing it was also discovered that Coleraine, Minnesota, tap water had elevated divalent cation levels in the range of approximately 335-416 ppm as $CaCO_3$ range. Optimizing the water system and water sourcing was not part of this testing scope, therefore, Coleraine tap water was chosen for this testing program except when otherwise noted (e.g., SFDF testing or blended water NatFlot testing). Coleraine tap water was also used for magnetic separation testing and the pelletizing portion of this research project.

Testing Equipment

A brief description of the bench- and pilot-scale equipment used for this research program can be found in Appendix 1.

Results

Sampling and Preparation of Bench-Scale Samples

A total of seven samples were acquired for basic characterization, baseline testing, and bench-scale testing. The data collected from baseline and bench-scale testing were utilized to decide which bulk samples to collect. The identification for each sample was based upon both the arrival date and the sampling location of the sample. The first four samples were supplied by a western Mesabi Iron Range mining operation. Approximately 50 crushed drill core samples were acquired and split into four composite samples based upon increasing levels of magnetic iron in the sample. These samples were denoted as 17-1 through 17-4, with the higher the number denoting the higher relative amount of magnetic iron in the sample. These samples were chosen as they represent significant resource but do not meet Davis Tube weight recovery specifications.

The next three samples were taken as "grab" samples at a western Mesabi Iron Range mining operation by the NRRI with the assistance of the mining company's geologists. Two 19 L (5-gallon) buckets were taken at three different locations. The first sample, denoted as Bench 18-1, was a sample of highly oxidized upper cherty material, taken from a pit wall. The second sample, denoted as Bench 18-2, was a sample of highly oxidized lower cherty material, also taken from a pit wall. The third sample, denoted as Bench 18-3, was a sample of taconite ore that was taken from a stockpile. The first two samples taken were considered strip material as they did not meet ore criteria at the taconite mine; the third sample was taconite iron formation with oxidized iron levels higher than acceptable with current induration methods. The bench-scale sample summary can be seen in Table 8.

Table 8: Bench-Scale Sample Summary

Sample ID	Sampling Origin	Sample Type	Sample Selection Rationale
17-1	Drill Core	Taconite with fine-grained primary hematite	Low Davis Tube Test weight recovery
17-2	Drill Core	Taconite with fine-grained primary hematite	Low Davis Tube Test weight recovery
17-3	Drill Core	Taconite with fine-grained primary hematite	Low Davis Tube Test weight recovery
17-4	Drill Core	Taconite with fine-grained primary hematite	Low Davis Tube Test weight recovery
18-1	"Grab" from Pit Wall	Oxidized Iron Formation	Low magnetic iron content
18-2	"Grab" from Pit Wall	Partially Oxidized Iron Formation	Low magnetic iron content
18-3	"Grab" from Stockpile	Partially Oxidized Iron Formation	Davis Tube concentrate contained elevated hematite

Each of the samples was crushed to approximately 3.4 mm (6 Mesh) via open-circuit laboratory-scale jaw and gyratory crushing at UMD NRRI. The samples were then blended and stage-crushed via laboratory-scale rolls crushing with a 0.85 mm (20 Mesh) closing screen. The samples were then split into representative batches of 600 grams for further testing via rotary and riffle splitting.

Characterization of Bench-Scale Samples

The chemical and mineralogical characterization for each of the bench sample "head" (e.g., starting material) is briefly discussed, below. The magnetic iron content was measured via Satmagan (e.g., Sat Mag Fe) and the amount of weight loss during high-temperature processing estimated via loss-on-ignition (LOI) testing at two different temperatures (650°C and 950°C). The chemical assays of the Type 17-1 through 17-4 bench samples can be seen in Table 9. Note the increase of magnetic iron from approximately 6% to 14% from samples Bench 17-1 to 17-4, respectively.

Table 9: Bench 17-1 through 17-4 Head Chemical Assay

Sample ID	Total Fe (%)	Sat Mag Fe (%)	SiO_2 (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	Phos (%)	LOI 650°C	LOI 950°C
Bench 17-1	28.38	6.09	48.13	0.55	2.60	1.82	0.017	5.88	5.76
Bench 17-2	27.56	8.13	49.41	0.49	2.54	2.04	0.016	5.40	5.49
Bench 17-3	28.29	10.81	51.84	0.52	1.40	2.10	0.015	4.23	4.43
Bench 17-4	29.37	14.44	47.46	0.70	1.57	1.94	0.017	5.37	5.85

The chemical assays of the Type 18-1 through 18-3 bench samples can be seen in Table 10. Note the magnetic iron ranging from under 1% with the Bench 18-1 sample to over 17% with the Bench 18-3 sample.

Table 10: Bench 18-1 through 18-3 Head Chemical Assay

Sample ID	Total Fe (%)	Sat Mag Fe (%)	SiO_2 (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	Phos (%)	LOI 650°C	LOI 950°C
Bench 18-1	42.61	0.51	33.26	0.64	0.08	0.07	0.090	5.16	5.22
Bench 18-2	44.13	6.87	31.39	0.29	0.18	0.15	0.025	1.05	0.84
Bench 18-3	34.43	17.23	49.58	0.12	BDL[5]	0.74	0.020	5.65	5.71

The mineralogy of the Bench 17-1 through 17-4 bench samples via X-ray diffraction (XRD) analysis can be seen in Figure 6, below. Note that while hematite is more abundant than magnetite, goethite was not identified. It is generally recognized that the hematite in these samples is primary and not associated with the secondary oxidation process. Primary hematite occurs as discrete plates and aggregates of plates rather than as secondary reaction rims on magnetite grains. The Type 17 samples were considered partially oxidized based upon the definitions in Table 6, above.

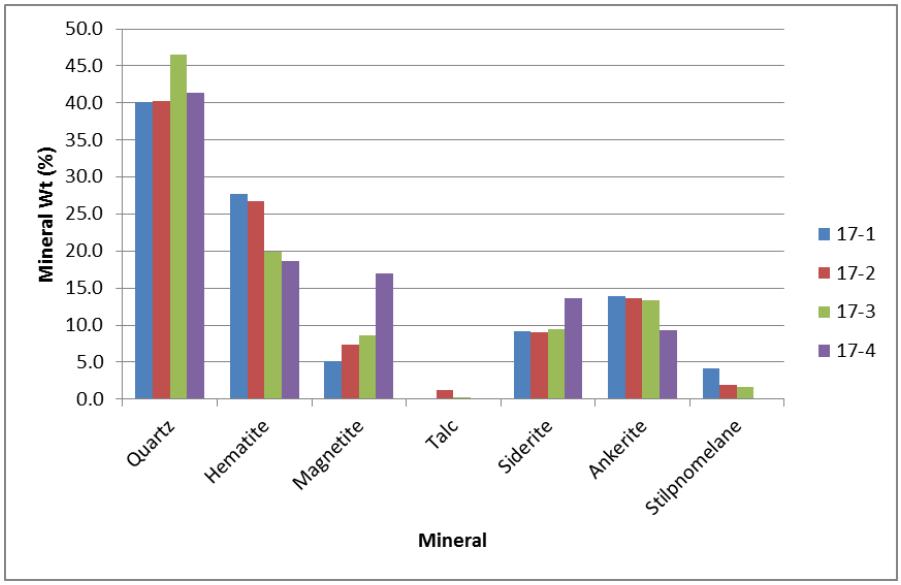


Figure 6: Bench 17-1 through 17-4 Head Mineralogy

The hematite-to-magnetite-to-goethite ratio for the 17-1 through 17-4 bench samples can be seen in Table 11.

Table 11: Bench 17-1 through 17-4 Sample Head Hematite : Magnetite : Goethite Ratio

Sample ID	Hematite	Magnetite	Goethite
17-1	1.0	0.2	0.0
17-2	1.0	0.3	0.0
17-3	1.0	0.4	0.0
17-4	1.0	0.9	0.0

The mineralogy of the Bench 18-1 through 18-3 bench samples can be seen in Figure 7.

[5] BDL = Below Detection Limit.

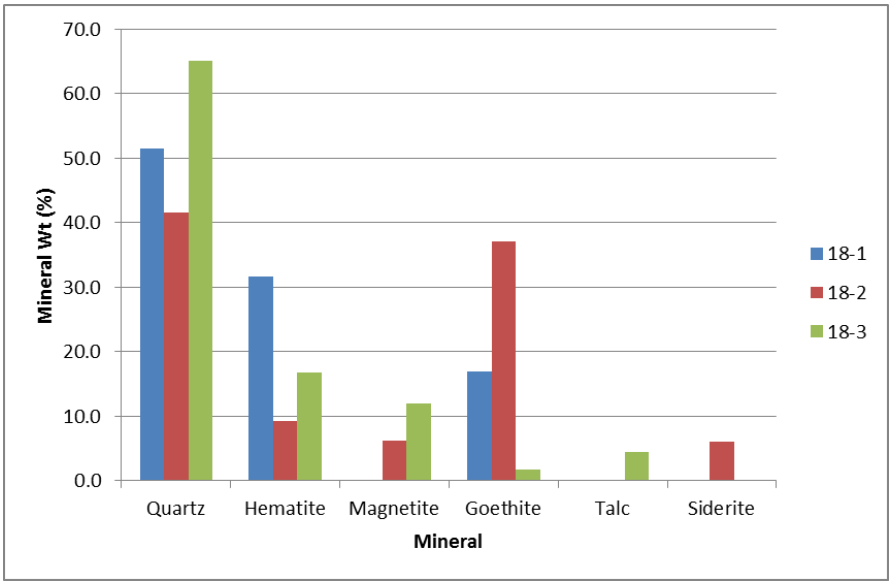


Figure 7: Type 18-1 through 18-3 Head Mineralogy

The hematite-to-magnetite-to-goethite ratio for the 18-1 through 18-3 bench samples can be seen in Table 12. The presence of magnetite, hematite, and goethite indicates that Bench 18-2 and 18-3 samples are partially oxidized iron formation, whereas the Bench 18-1 sample would be considered an oxidized iron formation sample based upon the definitions in Table 6, above.

Table 12: Bench 18-1 through 18-3 Sample Head Hematite : Magnetite : Goethite Ratio

Sample ID	Hematite	Magnetite	Goethite
18-1	1.0	0.0	0.5
18-2	1.0	0.7	4.0
18-3	1.0	0.7	0.1

A summary of the seven bench-scale testing samples can be seen in Table 13.

Table 13: Bench-Scale Sample Characterization Summary

Sample ID	Total Fe (%)	Sat Mag Fe (%)	SiO_2 (%)	Phos (%)		Quartz (%)	Hematite (%)	Magnetite (%)	Goethite (%)	Other Associated Minerals
17-1	28.38	6.09	48.13	0.017		40.1	27.7	5.1	N.D.	Siderite, Ankerite, Stilpnomelane
17-2	27.56	8.13	49.41	0.016		40.2	26.8	7.3	N.D.	Talc, Siderite, Ankerite, Stilpnomelane
17-3	28.29	10.81	51.84	0.015		46.5	19.9	8.6	N.D.	Talc, Siderite, Ankerite, Stilpnomelane
17-4	29.37	14.44	47.46	0.017		41.4	18.7	17.0	N.D.	Siderite, Ankerite, Stilpnomelane
18-1	42.61	0.51	33.26	0.090		51.4	31.7	N.D.	16.9	N.D.
18-2	44.13	6.87	31.39	0.025		41.5	9.3	6.2	37.0	Siderite
18-3	34.43	17.23	49.58	0.020		65.0	16.8	12.0	1.7	Talc

Bench-Scale Beneficiation Testing Summary

Baseline NatFlot tests and conventional beneficiation process comparative testing was conducted on the Bench 17-1 through 17-4 and Bench 18-1 through 18-3 samples. The purpose of these tests were to determine how the samples reacted to the different processes, as well as to determine which sample area(s) would be pursued for acquiring bulk samples for pilot-scale testing. The concentrate silica target was 4.0-4.5% SiO_2, as is a typical silica range for Minnesota taconite concentrates. Note that the *flotation underflow is considered the concentrate* and the *froth is considered the tailings* in the future results and discussion sections.

Baseline NatFlot Testing Conditions

A series of baseline tests were conducted on the samples to provide a basic understanding of how the samples performed in the NatFlot process. Baseline NatFlot test parameters consisted of:

- Process water and temperature: As-is Coleraine, Minnesota, tap water at ambient temperature ~18°C (65°F).
- Dispersant type and dosage: Israel Chemicals Limited (ICL) sodium hexametaphosphate (aka Glass H) prepared to 5% solution strength with deionized water. Typical dosage was 0.11 kg/metric ton (0.25 lb./LT).
- Fine Grind: bench-scale rod mill grinding for 10-15 minutes (depending on material type) at approximately 65% solids (w/w).
- Flotation: bench-scale Denver D1 with 2 L flotation bowl, naturally aspirated, at approximately 25% solids flotation feed density (600 g charges).
- Depressant type and dosage: Grain Processing Corporation (GPC) Supercore S12F modified corn starch to 1% solution strength with deionized water. Typical dosage was 0.67 kg/metric ton (1.50 lb./LT).

- Collector type: Air Products Tomamine PA-12 prepared to 1% solution strength with deionized water.

The typical number of flotation stages and collector (e.g., amine) schedule for the NatFlot process can be seen in Table 14, below. Test samples that required more than six stages to fully complete the flotation process were added at an incremental rate of 44.6 g/metric ton (0.10 lb./LT) collector dosage per stage.

Table 14: Typical NatFlot Stages and Collector Schedule

Flotation Stage	Collector Dosage (g/met ton)	Collector Dosage (lb./LT)
1	22.3	0.05
2	22.3	0.05
3	22.3	0.05
4	44.6	0.10
5	44.6	0.10
6	44.6	0.10
Total	**200.9**	**0.45**

Scavenging testing was not considered during the bench-scale testing phase, but is discussed in the *Pilot Testing Results* section on page 67.

Baseline NatFlot Testing Results

The Bench samples 17-1 through 17-4 were subjected to the baseline NatFlot tests with a 15-minute rod mill grind. The results are summarized in Table 15. As can be seen in the table, the 15-minute rod mill grind resulted in a very fine grind (~98% passing 25 µm) and concentrate silica grades ranging from approximately 8-10% SiO_2 and phosphorus levels below 0.02%. However, the concentrate total iron was low compared to typical Minnesota iron concentrates at the same silica levels. Therefore, XRD analysis was performed on each of the concentrate samples to determine the mineralogy.

Table 15: Baseline NatFlot Metallurgical Data Summary for Bench 17-1 through 17-4 (15-min Grinds)

Test	Sample	Grind Size (% -25 µm)	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
17-23	17-1	98.1	36.0	52.94	7.60	0.015	66.3	0.35	N/A
17-24	17-2	97.7	35.4	53.05	8.72	0.016	68.1	0.45	21.0
17-25	17-3	97.6	36.8	55.29	10.42	0.015	71.2	0.50	27.5
17-26	17-4	97.9	37.3	54.35	10.16	0.014	69.9	0.45	24.5
Average		**97.8**	**36.4**	**53.91**	**9.23**	**0.015**	**68.9**	**0.44**	**24.3**

The mineralogy of the head and NatFlot concentrate for each baseline test can be seen in Figure 8. The results of the XRD analysis indicate that the NatFlot process generally reduced the quartz content and concentrated hematite, magnetite, siderite, ankerite, and stilpnomelane. Concentrations of ankerite and stilpnomelane above a few percent decrease the total iron content

in the concentrate and are detrimental to pellet quality. The chemical formulas for these minerals can be seen in Table 7.

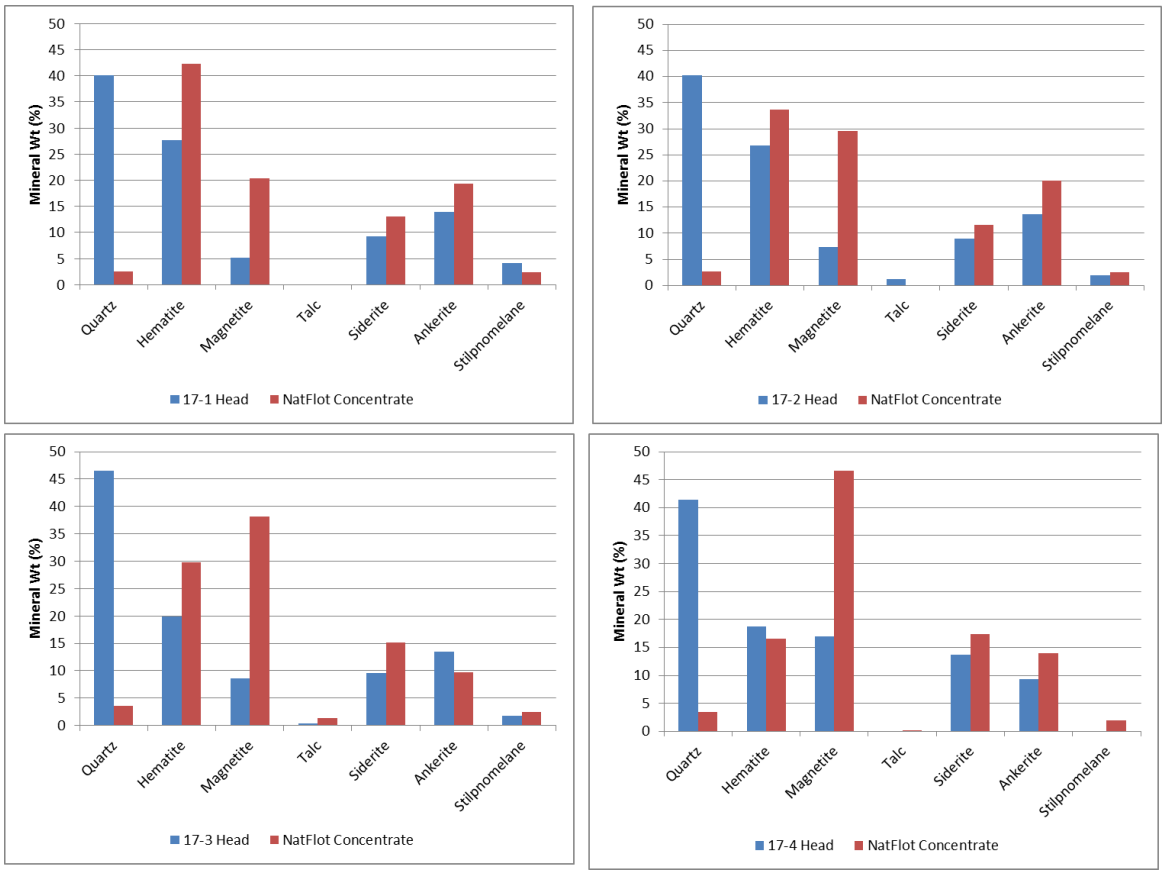


Figure 8: Bench 17-1 through 17-4 Head and NatFlot Concentrate Mineralogy

A single test was conducted with the Bench 17-2 sample to determine the effect of varying the particle size of flotation feed. This test was pursued due to the very fine flotation feed grinds found during the 15-minute rod mill grind baseline NatFlot tests. The result of the grind sensitivity test for Bench 17-2 is summarized in Table 16. The 8-minute rod mill grind produced a flotation feed of approximately 70% passing 25 µm. The concentrate total iron was approximately 50% Total Fe and 13% SiO_2 at the same total collector dosage as the 15-minute rod mill grind test. Decreasing the grind time resulted in no improvement over the baseline NatFlot tests in terms of concentrate quality or total flotation time. This testing confirms that NatFlot is not capable of selectively floating all of these different gangue minerals in one flotation step. The metallurgical test data for this test series can be found in Appendix 2.

Table 16: Grind Sensitivity NatFlot Metallurgical Data Summary for Bench 17-2 (15- vs. 8-min Grind)

Test	Sample	Grind Size (% -25 µm)	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
17-24	17-2	97.7	35.4	53.05	8.72	0.016	68.1	0.45	21.0
17-31	17-2	70.4	37.5	49.58	13.36	0.020	67.6	0.45	22.0

Similar to the Bench 17-1 through 17-4 samples, the Bench 18-1, 18-2, and 18-3 samples were subjected to baseline NatFlot tests and the results summarized in Table 17. The 15-minute rod mill grinds produced flotation feed approximately 90-95% passing 25 µm and concentrate silica grades ranging from 2-10% SiO_2. The NatFlot concentrate produced with the Bench 18-1 sample had a concentrate slightly above the target silica range with nearly 43% weight recovery. However, the phosphorus content was approximately seven times that of a typical Minnesota magnetite concentrate. The NatFlot concentrate produced with the 18-2 sample had a silica grade well below the 4.0-4.5% SiO_2 target by consuming approximately 0.36 kg/metric ton (0.55 lb./LT) of amine collector and floating for a total of 33 minutes. Weight and iron recovery was found to be 44% and 63%, respectively, at a silica grade of 1.7% SiO_2 and a phosphorus grade of nearly 0.04% P. Lastly, the NatFlot concentrate produced with the 18-3 sample had an elevated silica grade of nearly 10% SiO_2 at a weight recovery of 44% and total iron recovery of 79%. The phosphorus grade in this test was the lowest of the three tests at approximately 0.03%.

Table 17: Baseline NatFlot Metallurgical Data Summary for Bench 18-1 through 18-3

Test	Sample	Grind Size (% -25 µm)	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
17-28	18-1	92.6	42.7	58.98	6.04	0.119	58.7	0.55	27.0
17-29	18-2	95.0	44.1	63.28	1.66	0.037	63.0	0.80	32.5
17-30	18-3	89.2	43.7	61.84	9.79	0.033	78.7	0.60	22.5

These NatFlot concentrates were also subjected to XRD analysis to determine the mineralogical content in the baseline concentrate. The head and NatFlot concentrate mineralogy for Bench 18-1, 18-2, and 18-3 can be seen in Figure 9, Figure 10, and Figure 11, respectively. The NatFlot process with the Bench 18-1 sample rejected a high amount of quartz and some hematite, while highly concentrating the goethite. Phosphorus tied up in goethite is the likely the cause of the high phosphorus content in the final concentrate. Similar to the Bench 18-1 sample, the NatFlot process with the Bench 18-2 sample rejected nearly all of the quartz and some hematite and siderite, while slightly concentrating the magnetite and highly concentrating the goethite. This concentrate sample also contained nearly 80% goethite, but contained significantly lower phosphorus than the Bench 18-1 concentrate. Lastly, the NatFlot process with the Bench 18-3 sample rejected a significant amount of quartz and some talc, while slightly concentrating the hematite and highly concentrating the magnetite and goethite. The quartz was significantly reduced but not enough to produce the target silica grade.

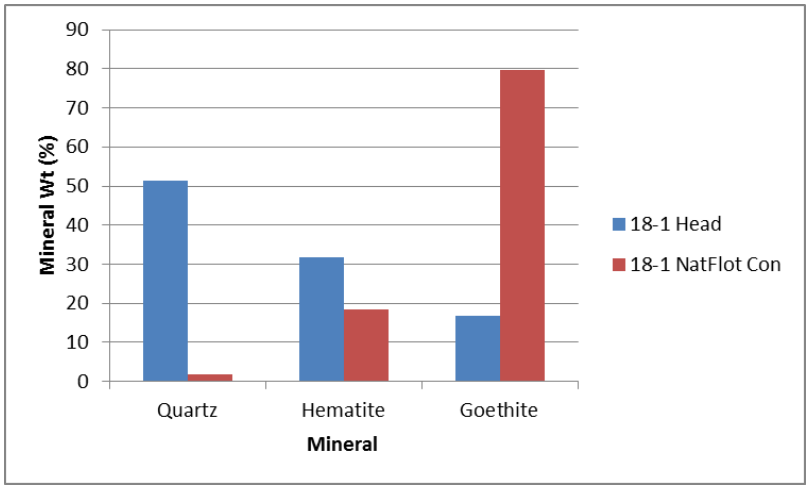


Figure 9: Bench 18-1 Baseline NatFlot Head vs. Concentrate Mineralogy

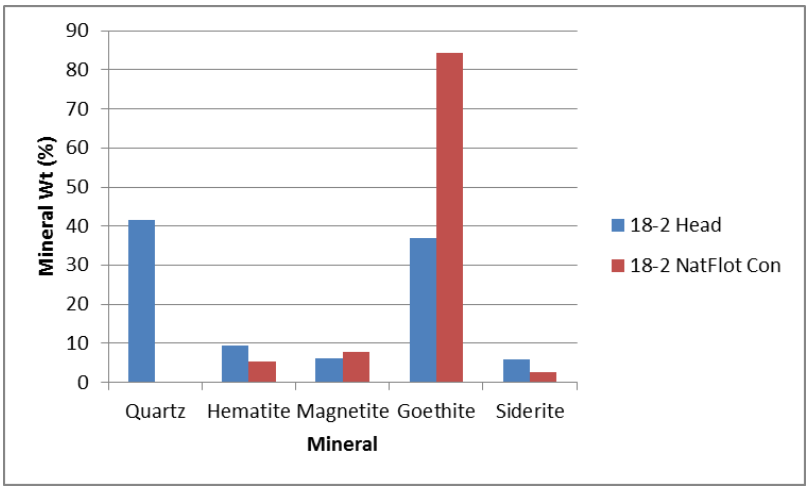


Figure 10: Bench 18-2 Baseline NatFlot Head vs. Concentrate Mineralogy

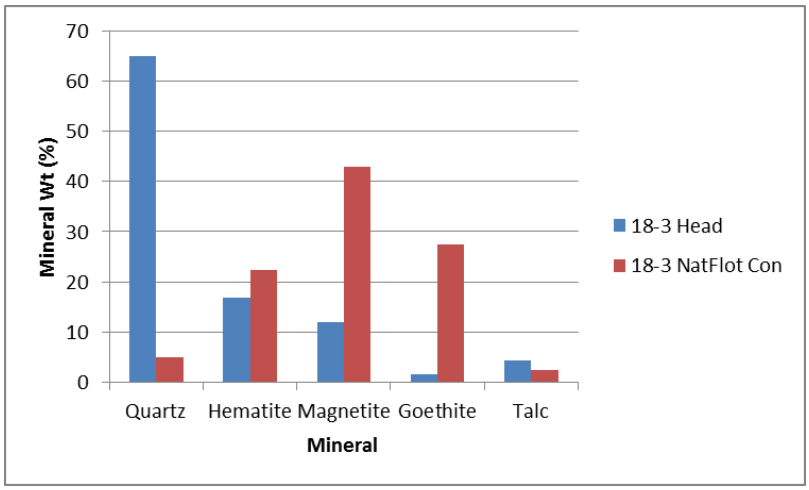


Figure 11: Bench 18-3 Baseline NatFlot Head vs. Concentrate Mineralogy

Testing on the Bench 18-2 and 18-3 samples were repeated to determine the effect of flotation feed particle sizing[6]. The rod mill grind times for the Bench 18-2 and 18-3 samples were reduced from 15 to 8-minutes and 10-minutes, respectively. The results of the grind sensitivity test for Bench 18-2 and 18-3 are summarized in Table 18 and indicate that the NatFlot concentrate produced with the reduced grind of the Bench 18-2 sample had a silica content slightly above target but at a much higher weight and iron recovery. The collector dosage and total flotation times were also reduced by approximately one-third. In terms of the Bench 18-3 sample, the reduced grind test found a slightly higher weight recovery and iron recovery, but at a concentrate silica grade significantly higher than the target. The metallurgical test data for this series can be found in Appendix 3.

Table 18: Grind Sensitivity NatFlot Metallurgical Data Summary for Bench 18-2 and 18-3

Test	Sample	Grind Size (% -25 µm)	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
17-29	18-2	95.0	44.1	63.28	1.66	0.037	63.0	0.80	32.5
17-47	18-2	62.2	57.6	60.56	6.39	0.038	78.1	0.55	20.5
17-30	18-3	89.2	43.7	61.84	9.79	0.033	78.7	0.60	22.5
17-48	18-3	71.6	48.5	58.32	16.14	0.031	82.1	0.60	19.0

Bench Samples Conventional Beneficiation Process Comparative Testing

A series of bench-scale tests with conventional beneficiation technologies were conducted as a comparison to the metallurgical performance of the NatFlot process. These included conventional beneficiation technologies for iron processing with fine-particles, namely, wet low-intensity magnetic separation (LIMS), wet high-intensity magnetic separation (WHIMS), and selective flocculation-deslime-flotation (SFDF).

Bench Samples Davis Tube Testing Summary

A test series was conducted using liberation grind and Davis Tube testing as a proxy for low-intensity magnetic separation performance. Each of the Bench 17-1 through 17-4 samples were subjected to a 6-, 12-, and 15-minute liberation grind and a Davis Tube test. The grind size vs. Davis Tube concentrate silica grade and Davis Tube concentrate silica grade vs. weight recovery for each sample can be seen in Figure 12. The Davis Tube tests found that it is possible to produce a sub-four percent concentrate, with the 17-3 and 17-4 samples having slightly coarser liberation than the first two samples. The estimated liberation size for these samples was approximately 80% passing ("P80") 35-45 µm to produce a 4.0-4.5% SiO_2 concentrate. However, the Davis Tube weight recoveries were lower than traditional taconite specifications require. Therefore, these samples would not be considered viable ore at most taconite operations. The metallurgical test data for this series can be found in Appendix 4.

[6] The Bench 18-1 sample was not chosen for this test series due to the high phosphorus content in the Test 17-28 baseline NatFlot concentrate.

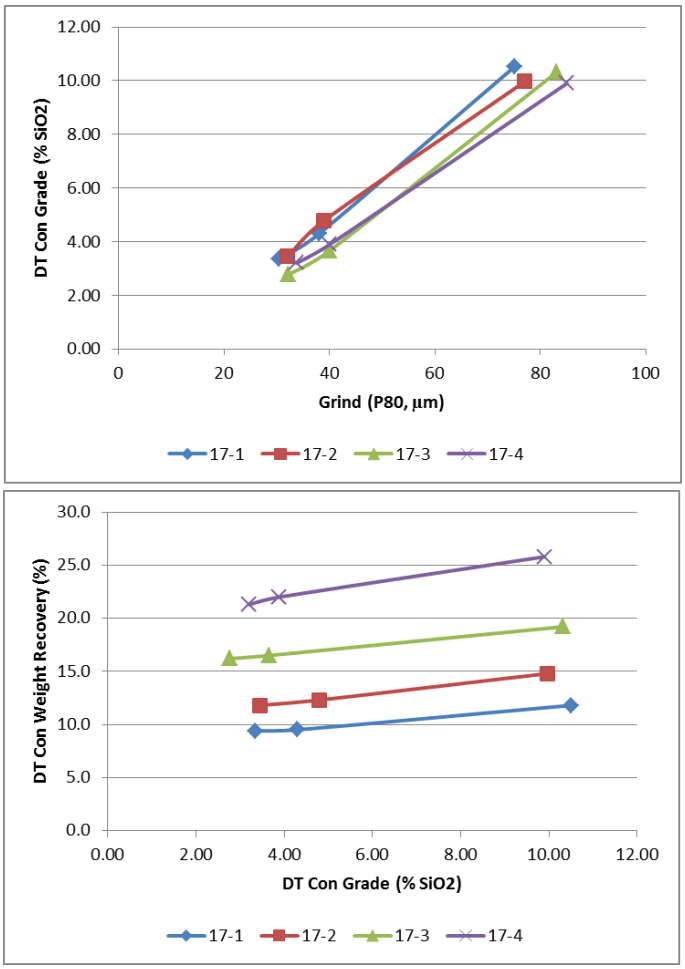


Figure 12: Davis Tube Metallurgical Data Summary for Bench 17-1 through 17-4

The Bench 18-1 and 18-2 samples did not have appreciable magnetite content in the feed, and therefore a liberation grind and Davis Tube test series was not conducted with these samples. However, a liberation grind and Davis Tube test series was performed with the Bench 18-3 sample, similar to the testing with the Bench 17-1 through 17-4 samples, with the exception that the liberation grind times used were 3-, 6-, and 12-minutes instead of 6-, 12-, and 15-minutes based upon a coarser liberation size discovered during testing. The grind size vs. Davis Tube concentrate silica grade and Davis Tube concentrate silica grade vs. weight recovery for 18-3 can be seen in Figure 13. The Davis Tube tests found that it is possible to produce a sub-four percent concentrate, with an estimated liberation size of approximately P80 60-65 µm to produce a 4.0-4.5% SiO_2 concentrate. As well, the Davis Tube weight recovery was similar to traditional taconite operations. The metallurgical test data for this series can be found in Appendix 5.

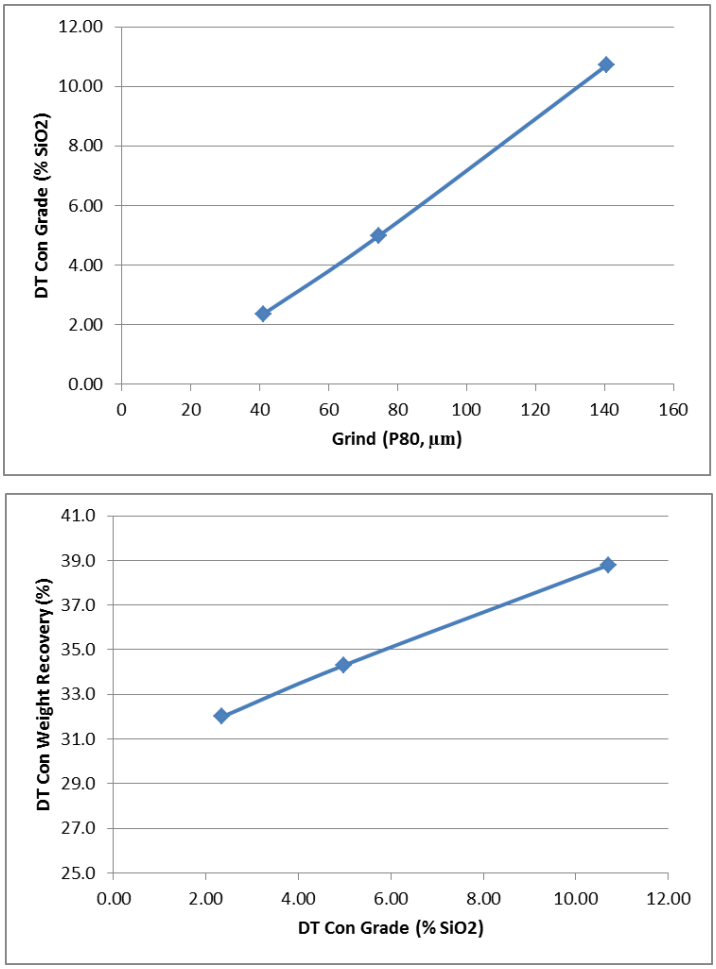


Figure 13: Davis Tube Metallurgical Data Summary for Bench 18-3

Bench Samples Wet Low-Intensity Magnetic Separation (LIMS) and Wet High-Intensity Magnetic Separation (WHIMS) Testing Summary

Based on the Davis Tube results with the Bench 18-3 sample, a small wet high-intensity magnetic separation (WHIMS) test series was conducted. This test series was conducted to determine the potential for recovering, or "scavenging", the paramagnetic iron minerals that were lost to the tailings with conventional low-intensity magnetic separator (LIMS) technology to increase the total iron recovery of the system. A Stearns Magnetics concurrent electro-magnetic drum separator was used for LIMS testing and an Eriez cyclic pulsating wet high gradient separator was used for the WHIMS testing. Both the LIMS and the WHIMS used a rougher-cleaner configuration (e.g., two-pass), where the non-magnetics from both passes were combined and considered the final tailings. The LIMS field intensity was held constant at 0.135 Tesla while the WHIMS field intensity was varied between 0.1-1.2 Tesla. The other test conditions for the WHIMS unit can be found in Appendix 3. In this test series, the magnetic portion was considered the concentrate whereas the non-magnetics were considered the tailings.

In summary, the Bench 18-3 sample was ground with a rod mill for 3-minutes to target a LIMS feed size of approximately P80 67 µm. The cleaner LIMS concentrate was retained while the total tailings were split into batches for WHIMS testing. The cleaner LIMS concentrate produced was 66.1% total iron and 6.4% SiO_2, with a weight and iron recovery of 35.7% and

68.6%, respectively. The total LIMS tails (e.g., WHIMS feed) was 16.8% total iron and 71.3% SiO_2, with a weight and iron recovery of 64.3% and 31.4%, respectively. The total LIMS tails was subjected to WHIMS testing by varying the magnetic field intensity between 0.1-1.2 Tesla. The WHIMS concentrate grade vs. recovery plot can be seen in Figure 14, which indicates the highest concentrate grade the WHIMS technology produced during this test series was approximately 35% total iron and 50% SiO_2 at a weight and iron recovery of about 17% and 40%, respectively.

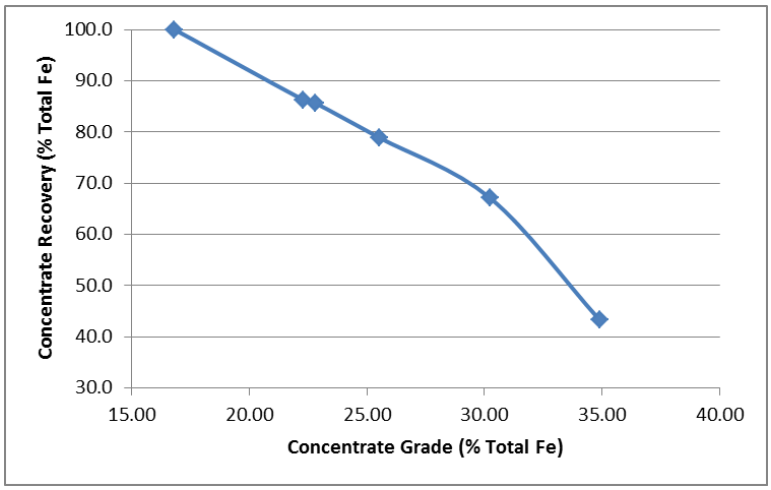


Figure 14: WHIMS Concentrate Total Iron Grade vs. Recovery Plot for Bench 18-3

The Bench 18-3 head, cleaner LIMS concentrate, 0.1 Tesla WHIMS cleaner concentrate, and 0.8 Tesla WHIMS cleaner concentrate were sampled and submitted for XRD analysis to determine the mineralogical content of each product. The XRD data is summarized in Figure 15, and indicates that the LIMS rejected the majority of the quartz and talc while greatly concentrating the magnetite and goethite and slightly concentrating the hematite. The total LIMS tail was estimated to be approximately 77% quartz, 11% hematite, 7% goethite, and 5% talc. The 0.1 Tesla WHIMS data showed that the concentrate was composed mainly of quartz, hematite, and goethite, with minor amounts of talc and siderite. Likewise, the 0.8 Tesla WHIMS concentrate found that the product contained a large amount of quartz, with lesser amounts of hematite, goethite, and talc. These results suggested that the sample would need a re-grind to liberate more of the quartz and talc, but at a potential penalty in concentrate weight recovery.

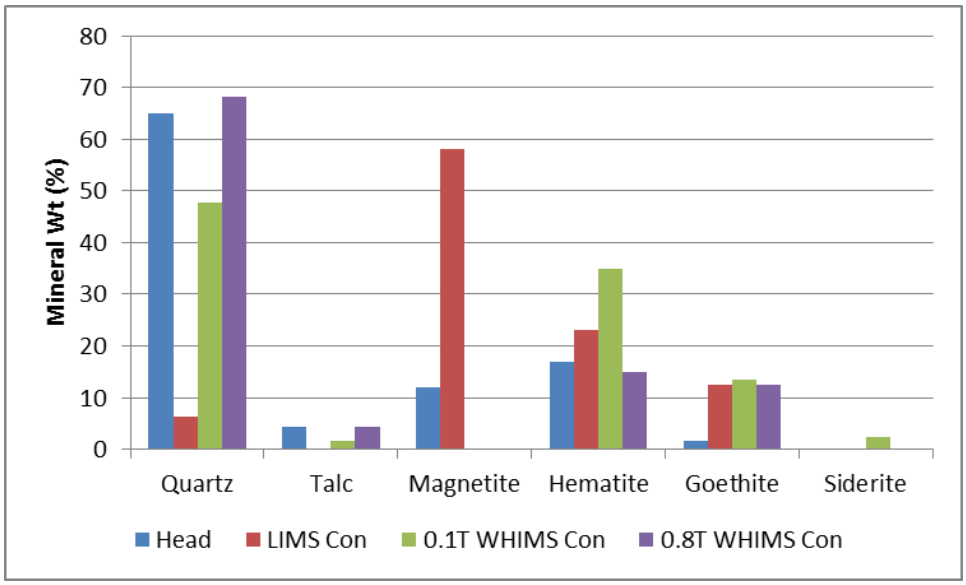


Figure 15: LIMS and WHIMS Product Mineralogy for Bench 18-3

The data for LIMS and WHIMS cleaner concentrates were combined to produce a composite grade vs. recovery plot for the magnetic separation test series. This data is summarized in Figure 16. As can be seen in the figure, it may be possible to increase the iron recovery of the LIMS circuit with the use of WHIMS technology with the Bench 18-3 sample, but the WHIMS concentrate would decrease the chemical purity of the concentrate at a relatively small incremental weight recovery. Re-grinding of the LIMS tails may be necessary to decrease the amount of quartz and talc in the sample before feeding into a WHIMS scavenger system, but may cause a reduction in scavenger concentrate weight recovery. As a result of these findings, it was concluded that implementing WHIMS as a scavenger system for this type of sample may be impractical due to a low-iron concentrate produced at a low weight recovery. The metallurgical test data for this series can be found in Appendix 6.

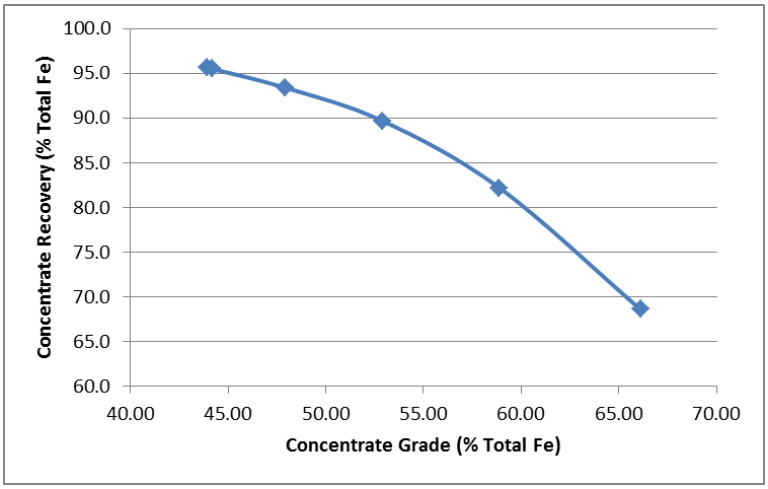


Figure 16: Composite LIMS+WHIMS Concentrate Total Iron Grade vs. Recovery Plot for Bench 18-3

Bench Samples Selective Flocculation-Deslime-Flotation (SFDF) Testing Summary

A single selective-flocculation-deslime-flotation (SFDF) test was performed with the Bench 17-2 sample as a direct comparison to the NatFlot bench test. The same 8-minute rod mill grind was

used as in the NatFlot grind sensitivity test with the Bench 17-2 sample, with a flotation feed size of approximately 70% passing 25 µm. Baseline SFDF conditions were used in this testing, including blended tap and deionized water and chemical reagents at typical dosages for SFDF testing. The water used for the SFDF tests was a deionized and tap water blend to target 5-10 ppm Ca^{++} at a pH of approximately 10.5-11.0. Sodium silicate and sodium hydroxide were added to the rod mill at a dosage of 0.33 kg/metric ton (0.75 lb./LT) and 0.67 kg/metric ton (1.5 lb./LT), respectively. GPC Supercore S12F causticized modified starch was added to the deslime and flotation steps at a dosage of 0.11 kg/metric ton (0.25 lb./LT) and 0.45 kg/metric ton (1.00 lb./LT), respectively. Air Products Tomamine PA-12 was used as the amine collector with incremental dosages of 0.45 kg/metric ton (0.10 lb./LT) per flotation stage.

The metallurgical data is summarized in Table 19, and indicates that the concentrate chemical assays and recoveries for both the NatFlot and SFDF tests are similar, but the total flotation time for the SFDF test was approximately half of the NatFlot tests. However, the concentrate silica grade for both the NatFlot and the SFDF tests was still significantly higher (13.4-15.4%) than the 4.0-4.5% SiO_2 target. Neither the NatFlot nor the SFDF process was capable of producing an acceptable concentrate with the Type 17 samples. The metallurgical test data for this series can be found in Appendix 7.

Table 19: NatFlot vs. SFDF Metallurgical Data Summary for Bench 17-2

Test	Sample	Flowsheet	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
17-31	17-2	NatFlot	37.5	49.58	13.36	0.020	67.6	0.45	22.0
17-46	17-2	SFDF	41.7	47.52	15.36	0.024	71.9	0.50	10.5

A series of selective-flocculation-deslime-flotation (SFDF) tests were also performed with the Bench 18-1, 18-2, and 18-3 samples as a direct comparison to the NatFlot bench tests. The same 15-minute rod mill grind was used as in the NatFlot grind sensitivity tests. The flotation feed sizes for the Bench 18-1, 18-2, and 18-3 samples were approximately 93%, 95%, and 89% passing 25 µm, respectively. The same conditions were used as described above in the Bench 17-2 SFDF test.

The metallurgical data from this test series is summarized in Table 20, which indicates that the SFDF test with the Bench 18-1 sample achieved a slightly higher weight and iron recovery at a lower silica content than the NatFlot baseline test. However, the phosphorus grade was similar to the NatFlot test at approximately 0.12%. The SFDF test with the Bench 18-2 sample produced a slightly higher concentrate silica grade, but at higher weight and iron recovery. The collector dosage and total flotation time for the SFDF were also significantly lower than the NatFlot baseline test. Both the NatFlot and SFDF tests show that a concentrate could be produced below the silica grade target with a phosphorus grade of 0.034-0.037%. Lastly, the SFDF test with the Bench 18-3 sample indicated similar weight and iron recoveries at a lower silica grade than the NatFlot test, but both tests achieved a silica grade above the target.

Table 20: NatFlot vs. SFDF Metallurgical Data Summary for Bench 18-1, 18-2, and 18-3

Test	Sample	Flowsheet	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
17-28	18-1	NatFlot	42.7	58.98	6.04	0.119	58.7	0.55	27.0
17-33	18-1	SFDF	46.0	60.65	4.25	0.121	65.0	0.50	8.5
17-29	18-2	NatFlot	44.1	63.28	1.66	0.037	63.0	0.80	32.5
17-34	18-2	SFDF	56.1	62.62	2.21	0.034	79.7	0.60	11.5
17-30	18-3	NatFlot	43.7	61.84	9.79	0.033	78.7	0.60	22.5
17-35	18-3	SFDF	42.7	65.03	5.72	0.027	80.7	0.50	12.0

The low-silica concentrate grade with the Bench 18-2 sample gave justification to conduct a more detailed grade vs. recovery analysis. The concentrate grade vs. recovery plots for the NatFlot and SFDF tests with the Bench 18-2 sample can be seen in Figure 17[7]. As can be seen in the figure, the SFDF had higher recovery in the flotation process compared to the baseline NatFlot test. Interpolated the data to the same final silica grade (2.21% SiO_2), the SFDF test achieved a weight and iron recovery increase over the NatFlot baseline of approximately 7% and 10%, respectively. The data suggests that the selective flocculation-deslime process was more selective than the first flotation stage in the NatFlot process. The metallurgical test data for this series can be found in Appendix 8.

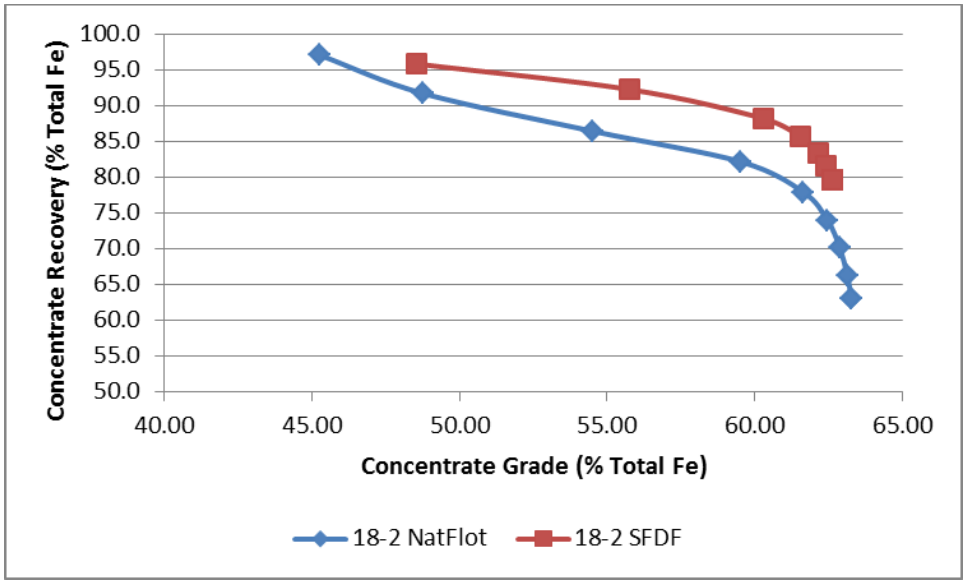


Figure 17: NatFlot and SFDF Concentrate Total Iron Grade vs. Recovery Plot for Bench 18-2

[7] It should be noted that the first SFDF data point is after desliming and the second NatFlot data point is after the first flotation stage.

Summary of Bench-Scale Beneficiation Testing and Bulk Sampling Selection

Baseline NatFlot tests and conventional beneficiation process comparative testing were conducted on the Bench 17-1 through 17-4 and Bench 18-1 through 18-3 samples. The purpose of these tests were to determine how the samples reacted to the different processes, as well as to determine which sample(s) would be selected for bulk sampling and pilot-scale testing. Through this test program it was determined that the Bench 17-1 through 17-4 samples could not be successfully upgraded to the silica target (4.0-4.5% SiO_2) with either the NatFlot or SFDF technologies. The high-silica and low-iron grade resulted from concentrating the ankerite and stilpnomelane in the final concentrate. The amount of ankerite and stilpnomelane reporting to concentrate due to selectivity or particles locked with silica is not known at this time. Davis Tube separation testing found that target silica grades could be achieved with the Bench 17-1 through 17-4 samples, but at a low weight recovery. Therefore, these samples were not pursued for further research in this program.

In reviewing the results with the Bench 18-1 through 18-3 samples, it was found that the Bench 18-1 sample could be beneficiated near the concentrate silica target with both NatFlot and SFDF flotation technologies. However, the phosphorus grade for both concentrates was elevated compared to typical Minnesota magnetite concentrate. This may be due to the high amount of goethite in the final concentrate (80% by weight). There was no appreciable amount of magnetite, so magnetic separation was not considered. Therefore, this sample was excluded from future research in the current program.

Both NatFlot and SFDF produced low-silica concentrates with the Bench 18-2 sample, with the SFDF baseline test achieving those results at a lower flotation time and higher weight and iron recovery over the NatFlot baseline of approximately 7% and 10%, respectively. However, the initial results from this baseline test series justified the acquisition of a bulk sample of this material for further research. Lastly, both NatFlot and SFDF technologies found concentrate silica grades above the target with Bench 18-3 sample due to quartz reporting to the concentrate. However, the concentrates produced were considered to have reasonable concentrate chemical composition and recovery. Davis Tube results indicate it was possible to produce a magnetic concentrate at the silica target with relatively good recovery. Scavenging of Bench 18-3 LIMS tailings using WHIMS showed some potential, but the concentrates produced with WHIMS had large amounts of quartz and relatively low weight recoveries. Regardless, this sample showed enough interesting results to pursue the acquisition of a bulk sample for further research.

Sampling and Preparation of Bulk Samples

Based upon the results of the bench-scale testing program and sample availability, a total of three bulk pilot-scale samples were acquired for basic characterization, baseline testing, and decision making for selection of sample(s) to use for pilot-scale beneficiation and pyrometallurgical testing. The first bulk sample, denoted as 16-1, was a partially oxidized taconite sample recommended and supplied by a western Mesabi Iron Range mining operation. This sample was selected because it does not satisfy taconite ore criteria due to partial oxidation. The partial oxidation results in rims of hematite around the magnetite grains. This rim reduces heat provided by the conversion of magnetite to hematite in the induration process and results in reduced pellet production and quality. Hence, it is treated as waste.

The next two bulk samples were sampled from a local mine site by a contractor[8]. The second bulk sample, denoted as 19-1, was stockpiled material from a previous blast at the mine site. This sample was taken from the same location as the Bench 18-3 sample. This material was sampled with a front-end loader and loaded directly on to side-dump trucks for transportation. Approximately 20-tons of sample was taken and transported to the contractor's location for crushing.

The third bulk sample, denoted as 19-2, was taken with an excavator on the side of an exposed pit face. This sample was taken from the same location as the Bench 18-2 sample. The excavator dug approximately one meter into the bank to remove the top layer, and sampled a few scoops of material. Unlike the method used to collect the grab samples, this sampling method included fine-grained sandy material in addition to rock. The excavator also scraped coarser material from the side of the bank above the original dig location. Approximately 20-tons of sample was taken and transported to the contractor's site for crushing. The pilot-scale sample summary can be seen in Table 21.

Table 21: Pilot-Scale Sample Summary

Sample ID	Sampling Origin	Sample Type	Sample Selection Rationale
16-1	Bulk from Stockpile	Partially Oxidized Iron Formation	Davis Tube concentrate contained elevated hematite
19-1	Bulk from Stockpile	Partially Oxidized Iron Formation	Davis Tube concentrate contained elevated hematite
19-2	Bulk from Pit Wall	Oxidized Iron Formation	Low magnetic iron content

The same contractor was also hired to crush the Bulk 19-1 and 19-2 material from top size down to 100% passing 19 mm (0.75 inch). This particle sizing was chosen because this is a typical feed size to high pressure grinding roll (HPGR) technology. The HPGR product for each sample would then be split out for characterization and metallurgical bench testing. The crushing occurred in late November, 2017. The mobile crushing plant consisted of a feeder, vibratory screen, gyratory crusher, recycle conveyors, and load out conveyors. After crushing, the samples were then transported to NRRI Coleraine facility to be prepared for grinding, characterization, and bench-scale metallurgical testing.

Upon preliminary investigation of the as-received Bulk 19-1 sample, it was discovered that the sample had lower total iron and magnetic iron than the "grab" sample, as seen in Table 22. However, the chemical assays were considered reasonably similar, and therefore the sample was retained as-is for further HPGR processing, characterization, and bench-scale testing.

[8] Hammerlund Construction of Grand Rapids, Minnesota.

Table 22: Comparison of Bench 18-3 and Bulk 19-1 Chemical Assay

Sample ID	Total Fe (%)	Sat Mag Fe (%)	SiO_2 (%)
Bench 18-3	34.43	17.23	49.58
Bulk 19-1 "As-Received"	28.72	13.87	55.64

Upon preliminary investigation of the as-received Bulk 19-2 sample, it was discovered that the sample was contaminated with sandy "dirt". This was confirmed by comparison of the bulk and bench chemical assays, as can be seen in Table 23. The contamination was confirmed as glacial till by the presence of feldspars and granitic fragments observed by optical microscopy and XRD analyses. This glacial till was likely from the overburden at the sampling site, from contaminated trucks, and/or from the lay-down area at the contractor site. Granitic rocks are found to the north of the Biwabik Iron Formation and granitic sand is a common component of sandy tills in the area. The till contamination diluted the iron formation sample.

Table 23: Comparison of Bench 18-2 and Bulk 19-2 Chemical Assay

Sample ID	Total Fe (%)	Sat Mag Fe (%)	SiO_2 (%)
Bench 18-2	44.13	6.87	31.39
Bulk 19-2 "As-Received"	18.08	0.07	74.85
Bulk 19-2 "3.4 mm (6 Mesh) Screen U/S"	40.96	0.07	36.13

This contamination was determined to be primarily in the fine fractions by visual and chemical analysis; the assay by size data found in Appendix 9. As a result, the Bulk 19-2 sample was screened at 3.4 mm (6 Mesh) and the undersize discarded. This resulted in approximately 48% of the Bulk 19-2 sample to be lost in the process. The screen oversize was retained and subjected to HPGR processing to 100% passing 3.4 mm (6 Mesh). The chemical assay of the screened material can be seen in Table 23.

High-Pressure Grinding Rolls (HPGR) Processing of Bulk 19-1 and 19-2

Approximately six barrels of the 19.1 mm (0.75 inch) Bulk 19-1 and six barrels of the Bulk 19-2 samples were processed with the KHD/Weir HPGR in lock-cycle with a 3.4 mm (6 Mesh) closing screen to determine metallurgical performance of the HPGR technology, as well as to prepare samples for characterization and bench testing. The HPGR was operated at a specific press force of 3.4 N/mm^2 and at a rolls speed of 15 RPM.

Characterization of Bulk Samples

The chemical and mineralogical characterization for each of the bulk samples is briefly discussed, below. The chemical assays of the bulk samples can be seen in Table 24.

Table 24: Bulk Sample Head Chemical Assay

Sample ID	Total Fe (%)	Sat Mag Fe (%)	SiO_2 (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	Phos (%)	LOI 650°C	LOI 950°C
Bulk 16-1	33.73	18.24	50.64	0.23	0.01	1.37	0.027	1.09	1.09
Bulk 19-1	28.72	13.87	55.64	1.60	0.61	0.82	0.026	1.26	0.96
Bulk 19-2	40.96	0.07	36.13	1.23	0.49	0.27	0.025	4.53	4.59

The Bulk 19-1 and 19-2 samples were also submitted to Activation Laboratories for a whole rock analysis (WRA) analysis. The WRA data analysis for major oxides can be seen in Table 25 and the full WRA data for Bulk 19-1 and 19-2 can be found in Appendix 10.

Table 25: Whole Rock Analysis for Major Oxides for Bulk 19-1 Head and Bulk 19-2 Head

Analyte Symbol	SiO_2	Al_2O_3	Fe_2O_3	MnO	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	LOI	Total
Unit Symbol	%	%	%	%	%	%	%	%	%	%	%	%
Detection Limit	0.01	0.01	0.01	0.001	0.01	0.01	0.01	0.01	0.001	0.01		0.01
Bulk 19-1 Head	55.01	1.59	34.1	0.13	0.79	0.65	0.35	0.29	0.071	0.05	1.17	100.6
Bulk 19-2 Head	34.37	1.29	57.22	0.068	0.27	0.49	0.29	0.16	0.067	0.05	4.66	99.83

The chemical assay of the Bulk 19-1 and 19-2 heads conducted at NRRI Coleraine and Activation Laboratories can be seen in Table 26.

Table 26: Bulk 19-1 and 19-2 Sample Head Chemical Assay (NRRI vs. WRA)

Sample ID	Total Fe (%)	Sat Mag Fe (%)	SiO_2 (%)	Phos (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	LOI (%)
Bulk 19-1 (NRRI)	28.72	13.87	55.64	0.026	1.60	0.61	0.82	0.96
Bulk 19-1 (WRA)	28.36	N/A	55.01	0.022	1.59	0.65	0.79	1.17
Bulk 19-2 (NRRI)	40.96	0.07	36.13	0.025	1.23	0.49	0.27	4.59
Bulk 19-2 (WRA)	40.64	N/A	34.37	0.022	1.29	0.49	0.27	4.66

The mineralogy of the bulk samples can be seen in Figure 18.

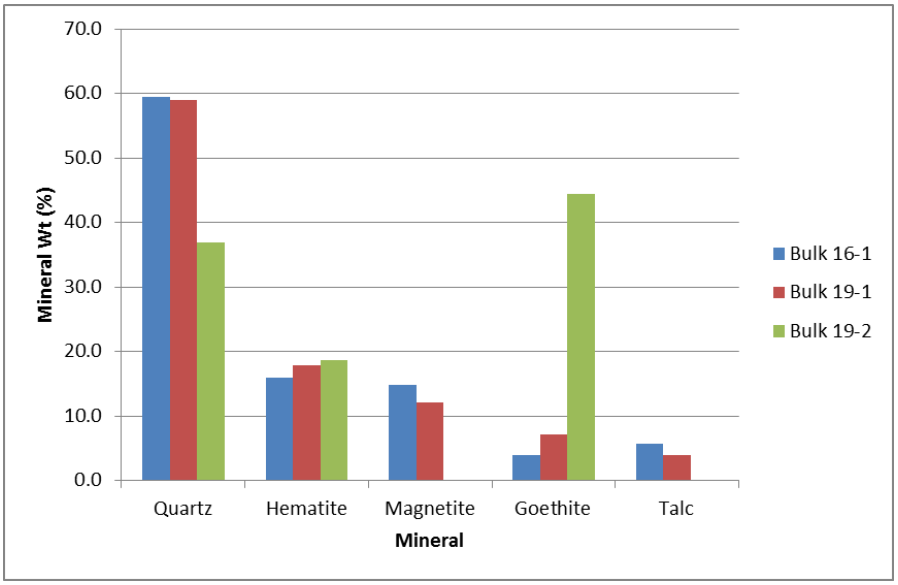


Figure 18: Bulk Sample Head Mineralogy

According to the definitions used in this report, Bulk 16-1 and 19-1 are partially oxidized iron formation, while the Bulk 19-2 sample is an oxidized iron formation. The hematite-to-magnetite-to-goethite ratio for each of the bulk samples can be seen in Table 27.

Table 27: Bulk Sample Head Hematite : Magnetite : Goethite Ratio

Sample ID	Hematite	Magnetite	Goethite
Bulk 16-1	1.0	0.9	0.2
Bulk 19-1	1.0	0.7	0.4
Bulk 19-2	1.0	0.0	2.4

A summary of the three bulk testing samples can be seen in Table 28.

Table 28: Bulk Sample Characterization Summary

Sample ID	Total Fe (%)	Sat Mag Fe (%)	SiO_2 (%)	Phos (%)		Quartz (%)	Hematite (%)	Magnetite (%)	Goethite (%)	Other Associated Minerals
16-1	33.73	18.24	50.64	0.027		59.5	16.0	14.9	3.9	Talc
19-1	28.72	13.87	55.64	0.026		59.0	17.9	12.1	7.1	Talc
19-2	40.96	0.07	36.13	0.025		36.9	18.7	N.D.	44.4	N.D.

Bulk Samples Bench-Scale Beneficiation Testing Summary

Baseline NatFlot tests and conventional beneficiation process comparative testing were conducted on the Bulk 16-1, 19-1, and 19-2 samples. The purpose of these tests was to determine how the samples reacted to the different processes, as well as to determine which sample(s) would be pursued for pilot-scale beneficiation testing.

Bulk Sample Baseline NatFlot Testing

A baseline NatFlot test series was conducted with the Bulk 16-1, 19-1, and 19-2 samples using the same baseline conditions described in the *Baseline NatFlot Testing* section found on page 19. The results are displayed in Table 29. In summary, the baseline NatFlot testing indicated that the concentrate produced by the NatFlot process with the Bulk 16-1 sample had elevated silica compared to the 4.0-4.5% SiO_2 concentrate target with fine flotation feed grind sizes of 75-87% passing 25 µm. Little material was rejected via the NatFlot process (18-30% weight), even at high collector dosage rates of 0.51 kg/metric ton (1.15 lb./LT) and long flotation times (22-27 minutes).

Similar to the Bulk 19-1 sample, the baseline NatFlot testing found that the concentrate produced by the NatFlot process had relatively high silica compared to the 4.0-4.5% SiO_2 concentrate target with fine flotation feed grind sizes of 63-83% passing 25 µm. A significant amount of material was rejected by the process (40-44%), but the concentrate silica grade was in the 12-17% SiO_2 range. The total collector dosage was 0.29 kg/metric ton (0.65 lb./LT) and flotation time was nearly 30-minutes. Based on previous flotation experience, it is likely the presence of talc in these two samples contributed to the difficulty in selectively floating the silica minerals.

In contrast to the first two bulk samples, the single baseline NatFlot testing of the Bulk 19-2 sample found that the concentrate produced by the NatFlot process was below the 4.0-4.5% SiO_2 target with a weight recovery above 34% and an iron recovery above 54%. The total collector dosage was 0.20 kg/metric ton (0.45 lb./LT) and flotation time was nearly 20-minutes. The metallurgical test data for this series can be found in Appendix 11.

Table 29: Baseline NatFlot Test Summary with Bulk 16-1, 19-1, and 19-2 Samples

Test	Sample	Grind Time (min)	Grind Size (% -25 µm)	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
17-12	16-1	15	87.0	70.8	41.76	36.76	84.7	1.15	27.0
17-14	16-1	10	74.5	82.0	35.53	45.13	87.6	1.15	22.0
17-77	19-1	15	83.3	36.6	59.21	12.45	74.6	0.65	30.0
17-78	19-1	10	62.9	39.8	56.55	16.54	78.0	0.65	29.5
18-7	19-2	13	87.2	35.4	62.99	2.03	54.5	0.45	20.0

Bulk 19-1/19-2 Blend Testing

As a comparison to the individual baseline testing, a series of three NatFlot tests were conducted to investigate the effect of different Bulk 19-1 and 19-2 sample flotation feed ratios on the NatFlot concentrate silica grade. The Bulk 19-1 and 19-2 materials were varied from 100% 19-2 to a 50:50 blend of Bulk 19-1 and 19-2. A 13-minute rod mill grind and the baseline NatFlot conditions were used for this test series.

The results from this test series are summarized in Table 30. To achieve a 4.0% SiO_2 concentrate with the NatFlot process at this flotation feed grind, a maximum of 16% Bulk 19-1 could be used. At a 16% Bulk 19-1 to 84% Bulk 19-2 flotation feed ratio and a 4.0% SiO_2 concentrate grade target, the estimated NatFlot concentrate weight and iron recovery would be 40.8% and 65.2%, respectively. Although it was determined that a blend of the Bulk 19-1 and 19-2 samples could produce a satisfactory concentrate silica grade, the blend ratio relied heavily on the 19-2 sample. Therefore, it was determined that future testing would treat each of the bulk samples separately. The metallurgical test data for this series can be found in Appendix 12.

Table 30: Baseline NatFlot Blend Testing Summary with Bulk 19-1 and 19-2 Samples

Test	Sample	Grind Size (% -25 µm)	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
18-7	100% 19-2	87.2	35.4	62.99	2.03	0.032	54.5	0.45	20.0
18-16	10:90 19-1:19-2	75.3	40.1	62.89	3.00	0.031	63.5	0.45	20.5
18-15	25:75 19-1:19-2	72.8	41.8	61.43	5.41	0.032	67.5	0.45	19.0
18-10	50:50 19-1:19-2	84.6	39.5	60.13	8.43	0.031	68.5	0.55	20.0

Bulk Sample Conventional Beneficiation Process Comparative Testing

A series of bench-scale tests with conventional beneficiation technologies were conducted as a comparison to the metallurgical performance of the NatFlot process. The technologies studied included conventional beneficiation technologies for iron processing with fine-particles, namely, wet low-intensity magnetic separation (LIMS), wet high-intensity magnetic separation (WHIMS), and selective flocculation-deslime-flotation (SFDF).

Bulk Sample Davis Tube Testing Summary

A test series was conducted using liberation grind and Davis Tube testing as a proxy for low-intensity magnetic separation performance. The Bulk 16-1 sample was subjected to a 6-, 9-, and 12-minute liberation grind and a Davis Tube test. The grind size vs. Davis Tube concentrate silica grade and Davis Tube concentrate silica grade vs. weight recovery for each sample can be seen in Figure 19. As can be seen in the figure, the Davis Tube tests found that it is possible to produce a sub-four percent concentrate with over 30% weight recovery. This would generally be considered ore at this taconite operation, however, the partial oxidation of the concentrate precluded it from being "ore" at the specific taconite mine that supplied it. The estimated liberation size for this sample was approximately 80% passing ("P80") 55-60 µm to produce a 4.0-4.5% SiO_2 concentrate. The metallurgical test data for this series can be found in Appendix 13.

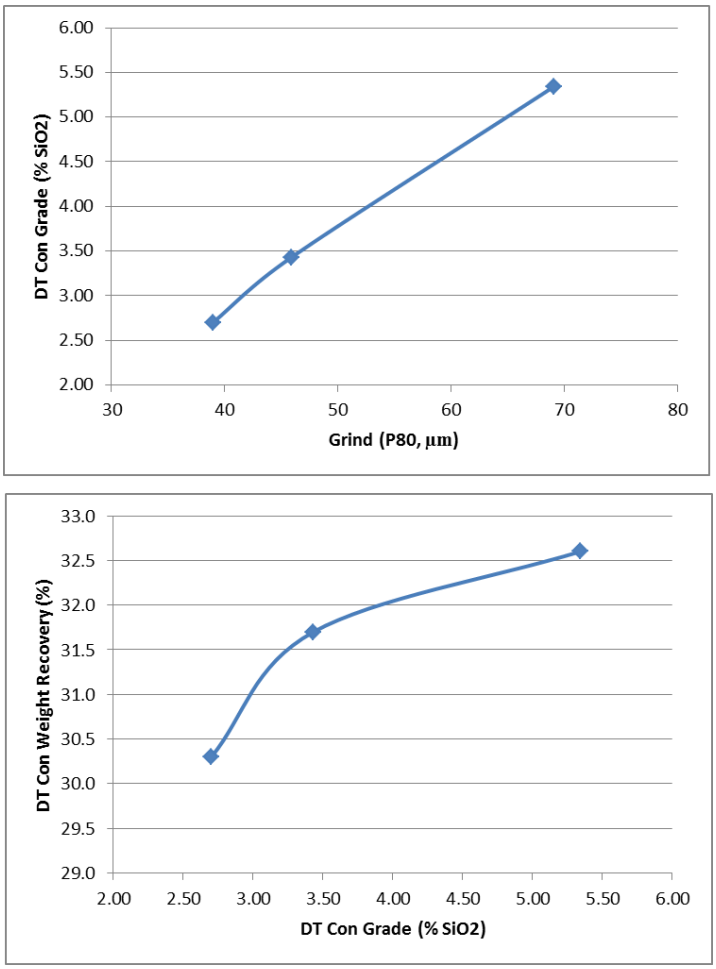


Figure 19: Davis Tube Metallurgical Data Summary for Bulk 16-1

A liberation grind and Davis Tube series was also performed with the Bulk 19-1 sample. This testing was similar to the testing with the Bulk 16-1 sample, with the exception that the liberation grind times used were 5-, 10-, and 20-minutes instead of 6-, 9-, and 12-minutes. The grind size vs. Davis Tube concentrate silica grade and Davis Tube concentrate silica grade vs. weight recovery for Bulk 19-1 can be seen in Figure 20. As can be seen in the figure, the Davis Tube tests indicated that it is possible to produce a sub-four percent concentrate with over 28% weight recovery. The estimated liberation size of the sample was approximately P80 65-70 µm to produce a 4.0-4.5% SiO_2 concentrate. The Bulk 19-2 sample did not have appreciable magnetite content in the feed, and therefore a liberation grind and Davis Tube test series was not conducted with this sample. The metallurgical test data for this series can be found in Appendix 14.

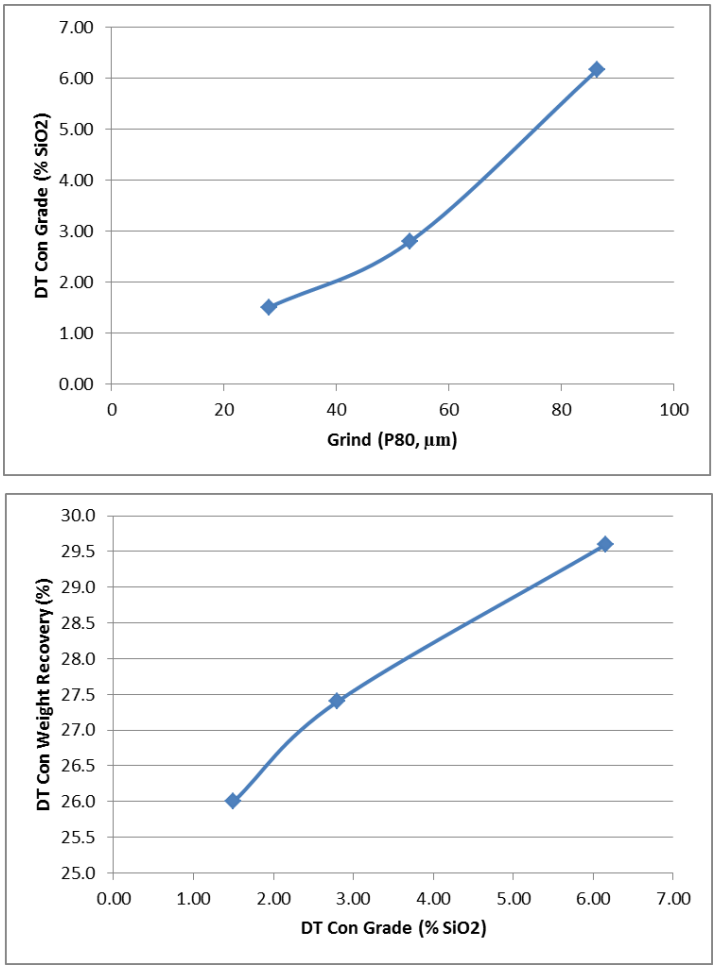


Figure 20: Davis Tube Metallurgical Data Summary for Bulk 19-1

Bulk Sample Wet Low-Intensity Magnetic Separation (LIMS) and Wet High-Intensity Magnetic Separation (WHIMS) Testing Summary

Similar to the Bench 18-3 sample testing, a small wet high-intensity magnetic separation (WHIMS) test series was conducted with the Bulk 16-1 sample. This test series was to determine the potential for recovering, or "scavenging", the paramagnetic iron minerals that were lost to the tailings with conventional low-intensity magnetic separator (LIMS) technology to increase the total iron recovery of the system. The same LIMS and WHIMS conditions were used as in the *Bench Samples Wet Low-Intensity Magnetic Separation (LIMS) and Wet High-Intensity Magnetic Separation (WHIMS) Testing Summary* section on page 26.

In summary, the Bulk 16-1 sample was ground with a rod mill for 3-minutes to target a LIMS feed size of approximately P80 60 µm. The cleaner two-pass LIMS concentrate was retained while the total tailings were split into batches for WHIMS testing. The cleaner LIMS concentrate produced was 66.3% total iron and 6.6% SiO_2, with a weight recovery and iron recovery of 35.1% and 69.0%, respectively. The total LIMS tails (e.g., WHIMS feed) was 16.1% total iron and 76.3% SiO_2, with a weight recovery and iron recovery of 64.9% and 31.0%, respectively. The total LIMS tails was subjected to WHIMS testing by varying the magnetic field intensity between 0.1-1.2 Tesla. The WHIMS concentrate grade vs. recovery plot can be seen in Table 26. The highest concentrate grade produced by the WHIMS technology during this

test series was approximately 42% total iron at an iron recovery of about 26%. Furthermore, the silica grade was approximately 34% SiO_2 with a fairly low weight recovery of 10%.

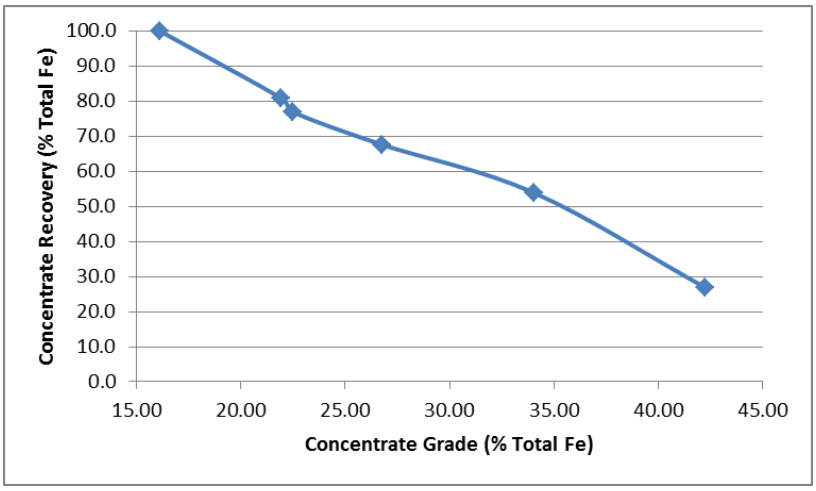


Figure 21: WHIMS Concentrate Total Iron Grade vs. Recovery Plot for Bulk 16-1

The Bulk 16-1 head, cleaner LIMS concentrate, 0.1 Tesla WHIMS cleaner concentrate, and 0.8 Tesla WHIMS cleaner concentrate were sampled and submitted for XRD analysis to determine the mineralogical content of each product. The XRD data is summarized in Figure 22. Similar to the Bench 18-3 sample, the analysis indicated that the LIMS rejected the majority of the quartz and some talc while greatly concentrating the magnetite and slightly concentrating the hematite and goethite. The total LIMS tail was estimated to be approximately 65% quartz, 13% hematite, 11% goethite, and 9% talc. The 0.1 Tesla WHIMS data showed that the concentrate was composed mainly of quartz, hematite, and goethite, with some minor amounts of talc and siderite. Likewise, the 0.8 Tesla WHIMS concentrate found that the product contained a large amount of quartz, with lesser amounts of hematite, goethite, talc, and siderite. These results suggested that the sample would need to be re-ground to liberate more of the quartz and talc, but at a potential penalty in concentrate weight recovery.

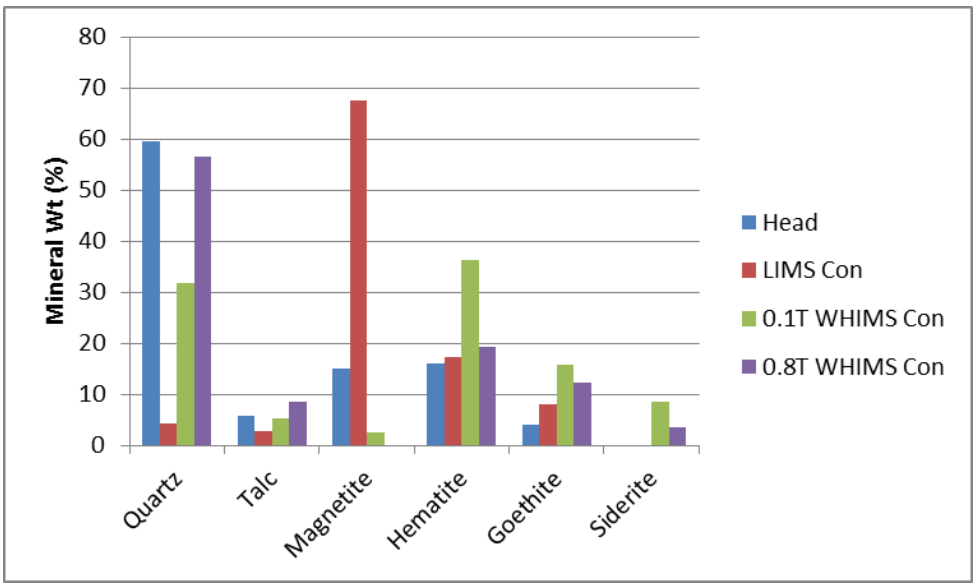


Figure 22: LIMS and WHIMS Product Mineralogy for Bulk 16-1

The LIMS and WHIMS cleaner concentrate data was combined to produce a composite grade vs. recovery plot for the magnetic separation test series. As can be seen in Figure 23, it may be possible to increase the iron recovery of the LIMS circuit with the use of WHIMS technology with the Bulk 16-1 sample, but the WHIMS concentrate would decrease the chemical purity of the concentrate at a relatively small incremental weight recovery. Re-grinding of the LIMS tails may be necessary to decrease the amount of quartz and talc in the sample before feeding into a WHIMS scavenger system, but may cause a reduction in scavenger concentrate weight recovery. Sending the two-pass WHIMS concentrate through additional WHIMS passes may also improve the grade, but likely not at a significant level. The metallurgical test data for this series can be found in Appendix 15.

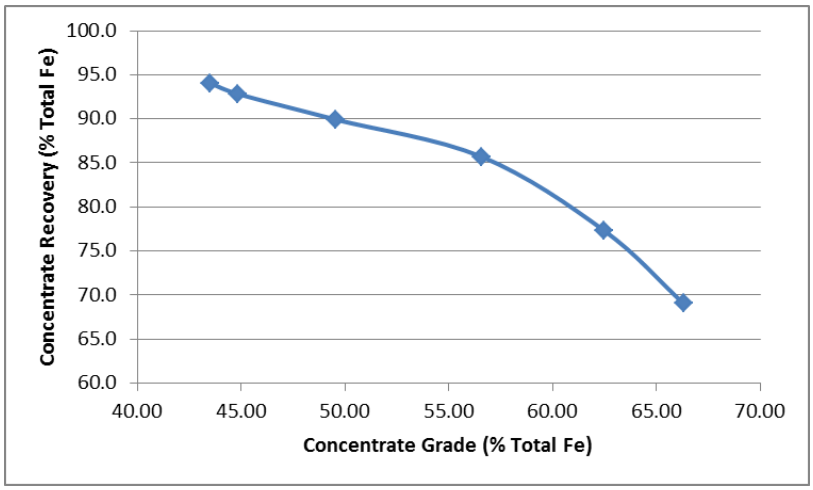


Figure 23: Composite LIMS+WHIMS Concentrate Total Iron Grade vs. Recovery Plot for Bench 16-1

A WHIMS study was conducted with the Bulk 19-2 sample to determine the concentrate grade and recovery relationship with this technology and sample. The same Eriez WHIMS unit was used in a rougher-cleaner configuration (e.g., two-pass), where the non-magnetics from both passes were combined and considered the final tailings. The WHIMS field intensity was varied between 0.1-0.8 Tesla. The other test conditions for the WHIMS unit can be found in the Appendix 3. In this test series, the magnetic portion was considered the concentrate whereas the non-magnetics were considered the tailings. The Bulk 19-2 sample was ground to two different size specifications with a rod mill to target a WHIMS feed size of approximately P80 30 µm and P80 40 µm. The samples were split out and tested with the WHIMS unit at three different magnetic field intensities (0.1 Tesla, 0.4 Tesla, and 0.8 Tesla).

The WHIMS results with the Bulk 19-2 sample are summarized in Figure 24. In summary, the WHIMS technology was unable to produce a concentrate at the target silica grade in this test series. The WHIMS technology was able to reduce the silica content in the sample from approximately 36% SiO_2 to as low as 8-10% SiO_2 when using a 0.1 Tesla magnetic field intensity, however, found low weight recoveries when producing a silica product below 15% SiO_2. More optimization testing could possibly improve on the grade-recovery relationship, but similar to the Bench 18-3 WHIMS results, the baseline testing with the Bulk 16-1 sample concluded that implementing WHIMS as a scavenger system for this type of sample may be

impractical due to a low-iron concentrate produced at a low weight recovery. The metallurgical test data for this series can be found in Appendix 16.[9]

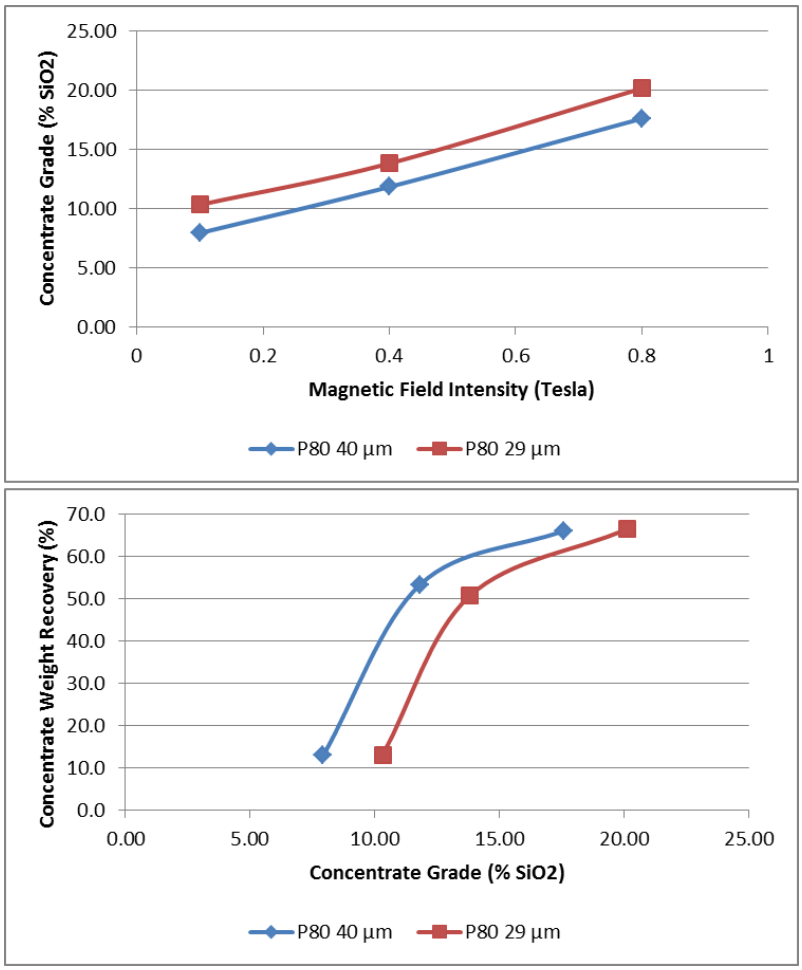


Figure 24: WHIMS Metallurgical Data Summary for Bulk 19-2

Bulk 16-1 and 19-1 Sample Selective Flocculation-Deslime-Flotation Testing

A selective-flocculation-deslime-flotation (SFDF) test was performed with the Bulk 16-1 sample as a direct comparison to the NatFlot bench test. The same 15-minute rod mill grind was used as in the NatFlot baseline test with the Bench 16-1 sample, with a flotation feed size of approximately 87% passing 25 µm. The same baseline SFDF conditions were used in this testing as described in the *Bench Samples Selective Flocculation-Deslime-Flotation (SFDF) Testing Summary* section on page 28.

The metallurgical results are summarized in Table 31. It was discovered that the SFDF process was able to further reduce the silica content in the concentrate compared to the NatFlot baseline test at a similar collector dosage and flotation time, but was above the silica target. The baseline SFDF test achieved a 6.8% SiO_2 concentrate at a weight and iron recovery of 37.8% and 72.3%,

[9] The Bulk 19-1 sample was not included in this test series because the sample was acquired from the same location as the Bench 18-3 sample, and characterization was determined to be similar so it was assumed the metallurgical performance would also be similar.

respectively. Both tests found high consumption of collector (amine) of over 0.45 kg/metric ton (1.0 lb./LT) and 27 minutes of total flotation time. The metallurgical test data for this series can be found in Appendix 17.

Table 31: NatFlot vs. SFDF Metallurgical Data Summary for Bench 16-1

Test	Sample	Flowsheet	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
17-12	16-1	NatFlot	70.8	41.76	36.76	0.024	84.7	1.15	27.0
17-17	16-1	SFDF	37.8	64.01	6.77	0.028	72.3	1.00	27.0

The stark differences in flotation results between the NatFlot and SFDF tests prompted a mineralogical study. The Bulk 16-1 sample head, NatFlot concentrate, and SFDF concentrate were submitted for XRD analysis. As seen in Figure 25, the NatFlot process moderately reduced the quartz, goethite, and talc content, with moderate concentration in hematite and magnetite. The SFDF process, however, greatly reduced the quartz and talc, while greatly concentrating the hematite, magnetite, and goethite minerals. It was also found that the talc content in the concentrates for both the NatFlot and SFDF processes were approximately the same. This was of interest, as the deslime weight rejection in the SFDF baseline test was nearly 34%.

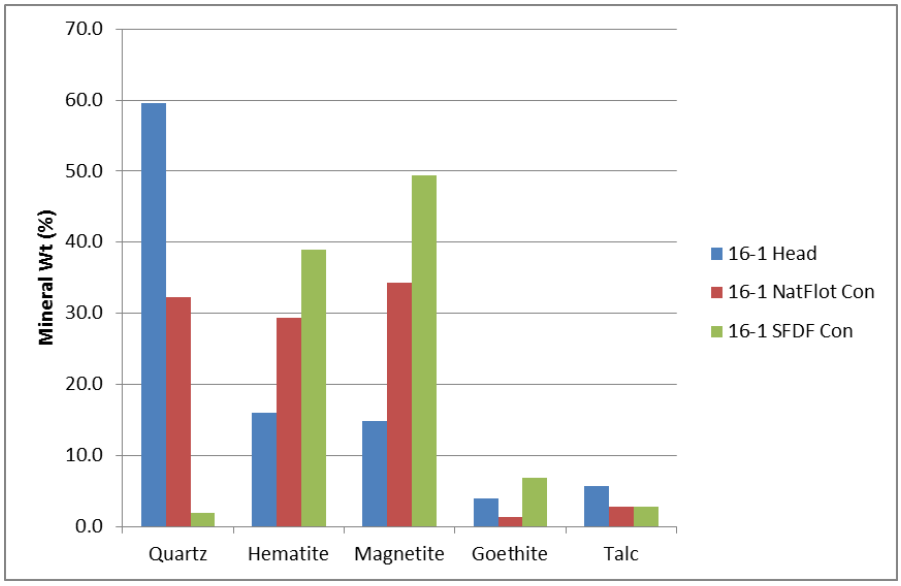


Figure 25: NatFlot vs. SFDF Product Mineralogy for Bulk 16-1

A single SFDF test was also performed with the Bulk 19-1 sample as a direct comparison to the NatFlot bench test. The same 15-minute rod mill grind was used as in the NatFlot baseline test with the Bench 19-1 sample, with a flotation feed size of approximately 83% passing 25 µm. The same baseline SFDF conditions were used in this testing as described in the *Bench Samples Selective Flocculation-Deslime-Flotation (SFDF) Testing Summary* section on page 28, with the exception that the total starch dosage was increased from 0.56 kg/metric ton (1.25 lb./LT) to 0.67 kg/metric ton (1.50 lb./LT) to match the NatFlot baseline starch dosage.

The metallurgical results are summarized in Table 32. It was discovered that the SFDF process was able to further reduce the silica content in the concentrate compared to the NatFlot baseline

test at a similar collector dosage and half the total flotation time, but was above the silica target. The baseline SFDF test found a 8.1% SiO_2 concentrate at a weight and iron recovery of 35.2% and 76.2%, respectively, whereas the NatFlot baseline test found a 12.5% SiO_2 concentrate at similar weight and iron recoveries. The metallurgical test data for this series can be found in Appendix 18.

Table 32: NatFlot vs. SFDF Metallurgical Data Summary for Bench 19-1

Test	Sample	Flowsheet	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
17-77	19-1	NatFlot	36.6	59.21	12.45	0.033	74.6	0.65	30.0
18-69	19-1	SFDF	35.2	62.55	8.05	0.033	76.2	0.60	14.5

Neither the NatFlot nor SFDF process could produce a satisfactory silica grade compared to the target, suggesting that either the liberation or mineralogy of the sample is hindering the processes. As a diagnostic tool, an XRD analysis was conducted with the NatFlot products to determine what minerals were present at each stage of the process. The results are summarized in Figure 26. In summary, the first three froths produced little weight rejection while consuming 67.0 g/metric ton (0.15 lb./LT) collector and 10.5 minutes of flotation time at essentially the same mineralogy as the head sample. This is a common response in flotation with talc; the talc will consume the amine collector and inhibit flotation. Froths four to six found a higher weight rejection of 11-19% and high quartz content ranging from 70-90%, with lesser amounts of hematite, magnetite, goethite, and talc. The last two froths were characterized by lesser weight recovery (5-8%) composed of similar amounts of the same minerals as found in froths four to six. The final flotation underflow (concentrate) was found to be composed of about 38-40% hematite and magnetite, 9% goethite, 8% quartz, 5% talc, a small amount of calcite.

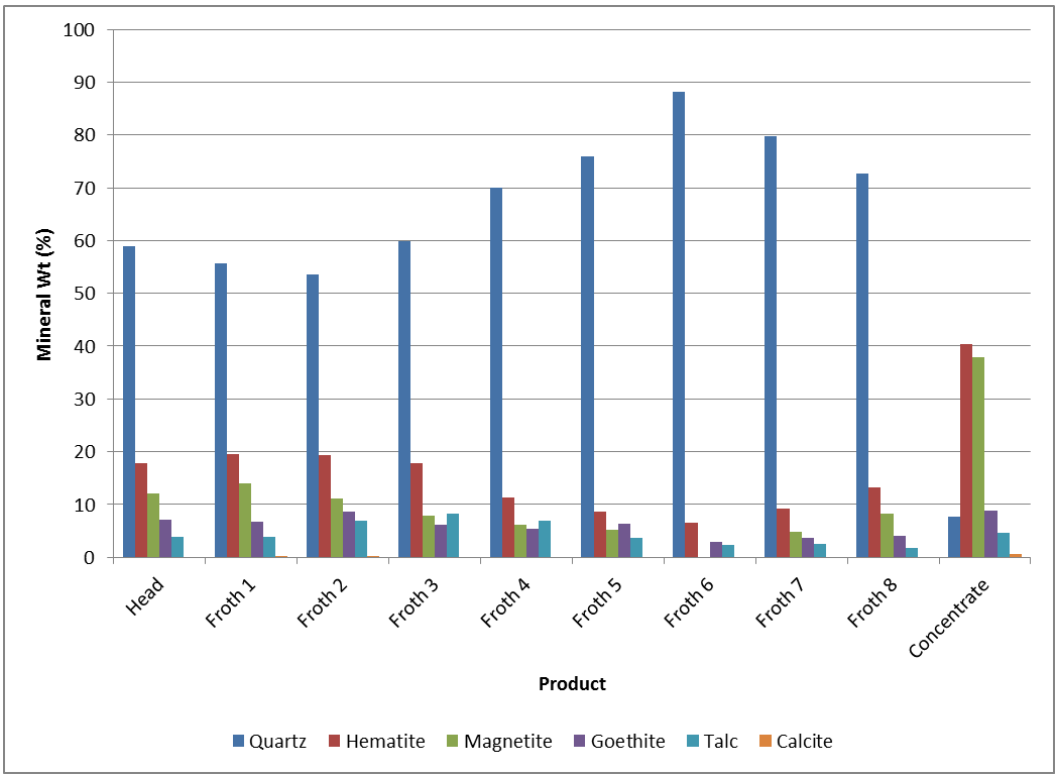


Figure 26: NatFlot Baseline Mineralogy per Product for Bulk 19-1

Contrary to the Bulk 16-1 and 19-1 samples, the baseline NatFlot test with the Bulk 19-2 found an acceptable silica concentrate grade could be produced. As a result, and before conducting a baseline NatFlot vs. SFDF study, a series of optimization and water system tests were conducted with the NatFlot process and Bulk 19-2 sample. The SFDF vs. NatFlot test summary can be found in the *Bulk 19-2 Sample Selective Flocculation-Deslime-Flotation Testing* section on page 63.

Bulk 19-2 Optimization Testing

The Bulk 19-2 sample was chosen to conduct optimization testing to determine the effect of grind, starch dosage, flotation feed pH, polyphosphate dosage, collector (amine) type, and process water. The goal of this testing was to optimize certain test parameters in the NatFlot bench-test to determine which conditions should be targeted for pilot-scale testing. The concentrate silica target of interest in this test series was 4.0-4.5% SiO_2. For the following tests the total iron grade vs. total iron recovery plots were used as a method to determine relative differences between individual tests at each flotation stage, whereas the data tables were used for a direct comparison of each test at target silica grade (typical interpolation target was 4.0-4.5% SiO_2).

Effect of Flotation Feed Grind Size

A series of three individual NatFlot tests were performed to determine the effect of grind size in the system. The grind size was varied by varying the rod mill grinding time from 13-minutes, 10-minutes, and 7-minutes. The tests were performed with the Bulk 19-2 material using a total of 0.67 kg/metric ton (1.50 lb./LT) GPC S12F causticized modified starch, 0.11 kg/metric ton

ICL Glass H (0.25 lb./LT), and six-stage flotation with a total of 0.20 kg/metric ton (0.45 lb./LT) Air Products PA-12 amine.

The concentrate grade vs. concentrate recovery plot and concentrate grade/recovery data interpolated to 4.5% SiO_2 table are presented in Figure 27 and Table 33, respectively. As illustrated, the flotation feed grinds of 80% passing ("P80") 34 µm and 46 µm tests had relatively similar metallurgical results. The finer grind test (P80 23 µm) resulted in a reduction in amine dosage and total flotation time and could produce a slightly higher iron grade concentrate, but had additional iron losses compared to the coarser grind tests. Therefore, the coarsest grind time (7-minutes) was selected for future testing. The metallurgical test data can be found in Appendix 19.

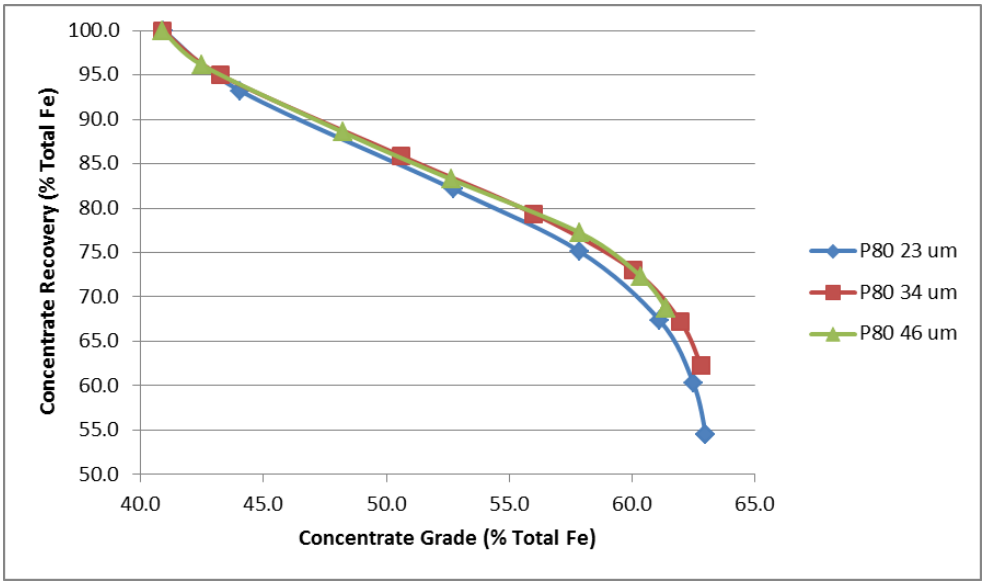


Figure 27: Effect of Grind Con. Grade vs. Recovery Plot (Test 18-7, Test 18-9, Test 18-11)

Table 33: Effect of Grind Con. Grade/Recovery Data Interpolated to 4.5% SiO_2 (Test 18-7, Test 18-9, Test 18-11)

Concentrate Grade/Recovery @ 4.5% SiO_2								
Test	**Mill Grind Time (min)**	**Grind (P80, µm)**	**Grind (% -25 µm)**	**Weight Recovery (%)**	**Total Fe (%)**	**Total Fe Recovery (%)**	**Amine (lb./LT)**	**Time (min)**
18-7	13	23	87.2	45.6	60.91	67.9	0.24	12.7
18-9	10	34	65.2	46.3	61.27	69.3	0.28	16.1
18-11	7	46	49.0	46.9	61.04	69.9	0.42	15.5

Effect of Starch Dosage

A series of three individual NatFlot tests were performed to determine the effect of starch dosage in the system. GPC causticized modified starch was varied from 0.33 kg/metric ton (0.75 lb./LT), 0.67 kg/metric ton (1.50 lb./LT), and 1.34 kg/metric ton (3.00 lb./LT). The tests were performed with the Bulk 19-2 material, 0.11 kg/metric ton ICL Glass H (0.25 lb./LT), and six-stage flotation with a total of 0.20 kg/metric ton (0.45 lb./LT) Air Products PA-12 amine. A 7-minute rod mill grind produced a flotation feed at approximately P80 46 µm (49% -25 µm).

The concentrate grade vs. concentrate recovery plot and concentrate grade/recovery data interpolated to 4.5% SiO_2 table can be seen in Figure 28 and Table 34, respectively. As can be seen in the figure and table, the recovery improved with an increase in starch dosage. However, the recovery was relatively similar when using 0.67 kg/metric ton (1.50 lb./LT) and 1.34 kg/metric ton (3.00 lb./LT) causticized modified starch, but resulted in significant weight and iron losses when decreasing the dosage to 0.33 kg/metric ton (0.75 lb./LT). The amine consumption with the starch dosage of 0.67 kg/metric ton (1.50 lb./LT) was also slightly lower than with the starch dosage of 1.34 kg/metric ton (3.00 lb./LT). Therefore, the future bench testing maintained a starch dosage of 0.67 kg/metric ton (1.50 lb./LT). Another result of this testing series was that the final flotation concentrate was marginally above 4.0% SiO_2 at a causticized modified starch dosage of 0.67 kg/metric ton (1.50 lb./LT), therefore, the rod mill grind time was increased from 7-minutes to 9-minutes in the future bench tests. The metallurgical test data can be found in Appendix 20.

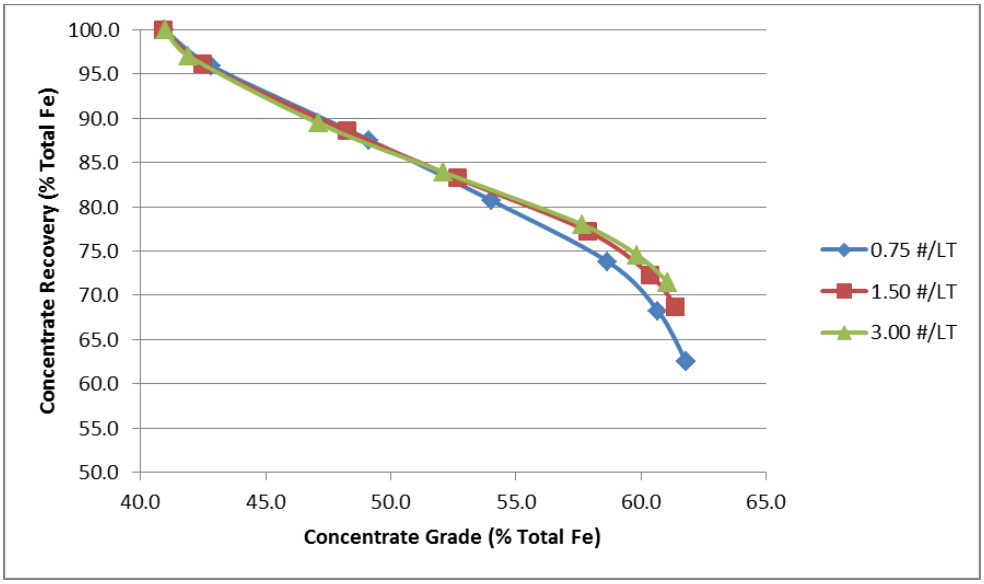


Figure 28: Effect of Starch Dosage Con. Grade vs. Recovery Plot (Test 18-11, Test 18-20, Test 18-21)

Table 34: Effect of Starch Dosage Con. Grade/Recovery Data Interpolated to 4.5% SiO_2 (Test 18-11, Test 18-20, Test 18-21)

Concentrate Grade/Recovery @ 4.5% SiO_2						
Test	**Starch (lb./LT)**	**Weight Recovery (%)**	**Total Fe (%)**	**Total Fe Recovery (%)**	**Amine (lb./LT)**	**Time (min)**
18-20	0.75	43.9	61.19	65.5	0.40	16.7
18-11	1.50	46.9	61.03	69.9	0.42	15.5
18-21	3.00	47.9	61.07	71.5	0.45	16.0

Effect of Polyphosphate (Glass H) Dosage

A series of four individual NatFlot tests were performed to determine the effect of polyphosphate (Glass H) in the system. The ICL Glass H dosage was varied between 0.00 kg/metric ton (0.00 lb./LT), 0.11 kg/metric ton (0.25 lb./LT), 0.22 kg/metric ton (0.50 lb./LT), and 0.45 kg/metric

ton (1.00 lb./LT). The NatFlot tests were performed with the Bulk 19-2 sample using a total of 0.67 kg/metric ton (1.50 lb./LT) GPC S12F causticized modified starch and six-stage flotation with a total of 0.20 kg/metric ton (0.45 lb./LT) Air Products PA-12 amine. The same tap water was used in each test. A 9-minute rod mill grind produced a flotation feed at approximately P80 38 µm (59% -25 µm).

The concentrate grade vs. concentrate recovery plot and concentrate grade/recovery data interpolated to 4.0% SiO_2 can be seen in Figure 29 and Table 35, respectively. As can be seen in the figure and table, the increase in Glass H dosage found a decrease in amine consumption but a decrease in weight recovery and iron recovery as well. The higher Glass H dosage improved the final concentrate grade slightly but also increased the phosphorus content in the final concentrate. The metallurgical test data can be found in Appendix 21

This finding promoted the request for a preliminary scanning electron microscope study. In this study, it was discovered that all of the phosphorus present in the Bulk 19-2 sample was associated with the goethite; no apatite was detected in the sample. Therefore, the Glass H dispersant was not used in the future optimization test series.

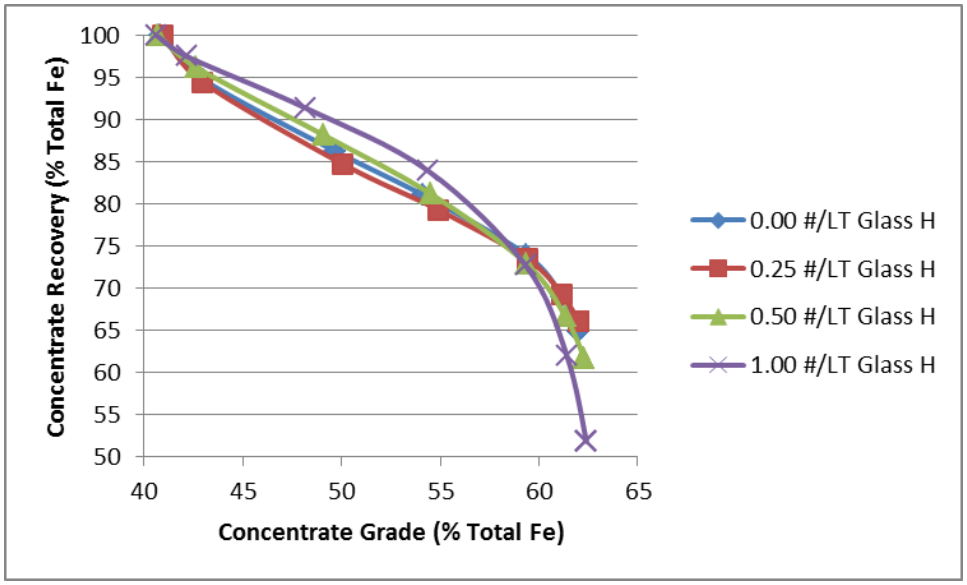


Figure 29: Effect of Polyphosphate Dosage Con. Grade vs. Recovery Plot (Test 18-26 to Test 18-29)

Table 35: Effect of Polyphosphate Dosage Con. Grade/Recovery Data Interpolated to 4.0% SiO_2 (Test 18-26 to Test 18-29)

Concentrate Grade/Recovery @ 4.0% SiO_2							
Test	**Glass H (lb./LT)**	**Weight Recovery (%)**	**Total Fe (%)**	**Phos (%)**	**Total Fe Recovery (%)**	**Amine (lb./LT)**	**Time (min)**
18-27	0.00	45.4	61.46	0.032	68.1	0.39	16.1
18-26	0.25	45.1	61.41	0.034	67.8	0.38	16.9
18-28	0.50	42.9	61.65	0.034	65.0	0.38	20.0
18-29	1.00	40.7	61.48	0.040	61.4	0.36	18.6

Effect of Process Water Type

A series of seven individual NatFlot tests were performed to determine the effect of process water in the system. The process water was varied between different ratios of tap water and deionized (DI) water, starting from no DI water to 100% DI in increments of 25%. A 50:50 Coleraine tap:distilled water test was also conducted to compare to the 50:50 Coleraine tap:DI water test. The NatFlot tests were performed with the Bulk 19-2 sample using no Glass H, a total of 0.67 kg/metric ton (1.50 lb./LT) GPC S12F causticized modified starch, and six-stage flotation with a total of 0.20 kg/metric ton (0.45 lb./LT) Air Products PA-12 amine. A 9-minute rod mill grind produced a flotation feed at approximately P80 38 µm (59% -25 µm).

The concentrate grade vs. concentrate recovery plot and concentrate grade/recovery data interpolated to 4.0% SiO_2 table can be seen in Figure 30 and Table 36, respectively. The figure and table indicate the tests with the higher ratio of Coleraine tap water, specifically the tests with 100% and 75% Coleraine tap water, had the highest weight and iron recovery. The increase in the amount of DI water produced a lower process water pH but a higher flotation feed pH. The tests with 50:50 Coleraine tap:DI and 50:50 Coleraine tap:distilled water had similar total hardness levels and similar metallurgical results. Therefore, as-is Coleraine tap water was used in the future optimization test series. The metallurgical test data can be found in Appendix 22.

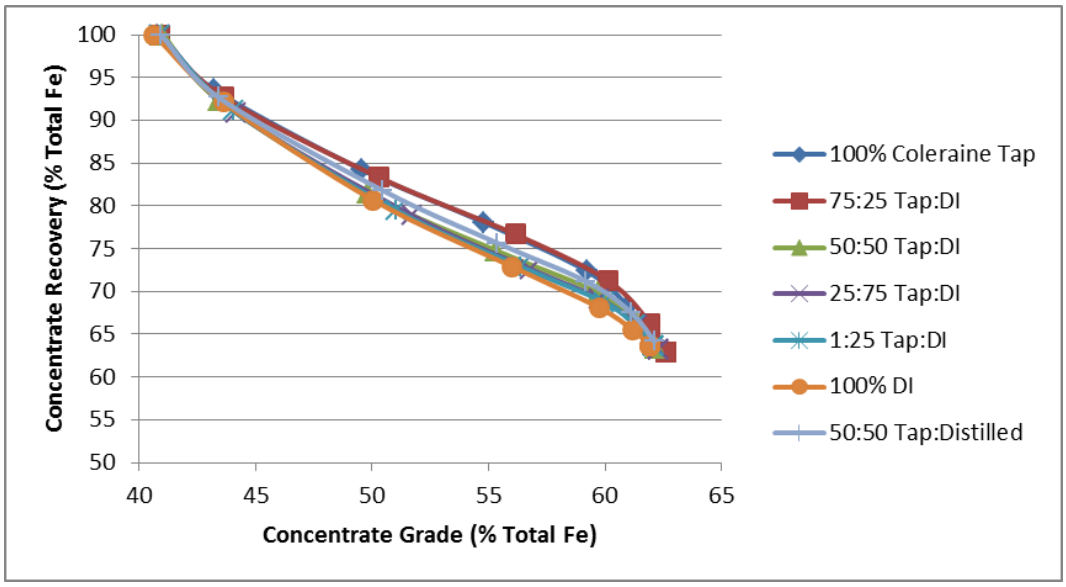


Figure 30: Effect of Process Water Type Con. Grade vs. Recovery Plot (Test 18-37 to Test 18-43)

Table 36: Effect of Process Water Type Con. Grade/Recovery Data Interpolated to 4.0% SiO_2 (Test 18-37 to Test 18-43)

Concentrate Grade/Recovery @ 4.0% SiO_2											
Test	**Water**	**Ca^{++} (ppm)**	**Mg^{++} (ppm)**	**Total Hardness as $CaCO_3$ (ppm)**	**Process Water pH**	**Flot Feed pH**	**Weight Recovery (%)**	**Total Fe (%)**	**Total Fe Recovery (%)**	**Amine (lb./LT)**	**Time (min)**
18-37	100% Coleraine Tap	108.4	35.4	416	7.09	8.06	44.5	61.30	66.9	0.38	17.1
18-38	75:25 Tap:DI	88.2	28.7	338	7.65	8.17	44.7	61.58	67.2	0.33	16.9
18-39	50:50 Tap:DI	53.8	17.5	206	7.60	8.25	43.5	61.64	65.4	0.38	16.9
18-40	25:75 Tap:DI	30.8	10.5	120	7.69	8.47	43.6	61.52	65.6	0.37	14.7
18-41	1:25 Tap:DI	3.9	2.0	18	7.00	8.79	43.5	61.49	65.5	0.37	N/A
18-42	100% DI	BDL[10]	0.3	1	6.46	9.05	42.4	61.67	64.3	0.41	15.5
18-43	50:50 Tap:Distilled	62.2	20.2	238	8.16	8.34	43.7	61.62	65.8	0.40	17.8

Effect of Flotation Feed pH

A series of six individual NatFlot tests were performed to determine the effect of flotation feed pH in the system. The flotation feed pH was varied between the pH target range of 6-10 using hydrochloric acid to decrease pH and sodium hydroxide to increase pH. The NatFlot tests were performed with the Bulk 19-2 sample using no Glass H, a total of 0.67 kg/metric ton (1.50 lb./LT) GPC S12F causticized modified starch, and six-stage flotation with a total of 0.20 kg/metric ton (0.45 lb./LT) Air Products PA-12 amine. The as-is Coleraine tap water measured 98.3 ppm Ca^{++} and 34.8 ppm Mg^{++}, for a calculated hardness of 389 ppm as $CaCO_3$. A 9-minute rod mill grind produced a flotation feed at approximately P80 38 µm (59% -25 µm).

The concentrate grade vs. concentrate recovery plot and concentrate grade/recovery data interpolated to 4.0% SiO_2 table can be seen in Figure 31 and Table 37, respectively. These indicate that the grade/recovery relationship was fairly similar between the test series. However, at a flotation feed pH of 10, the final concentrate produced a lower final concentrate grade than the other tests; the average silica for the pH range 6-9 was 2.9% SiO_2 whereas the pH 10 test final concentrate silica was 3.8% SiO_2. Furthermore, the amine dosage to achieve a 4.0% SiO_2 concentrate was about 44.6 g/metric ton (0.10 lb./LT) more than the other tests in the series. Therefore, the NatFlot process was found to be fairly robust for this particular sample in the pH range 6-9. The flotation feed pH was not adjusted in the future optimization test series. The metallurgical test data can be found in Appendix 23.

[10] Below detection limit.

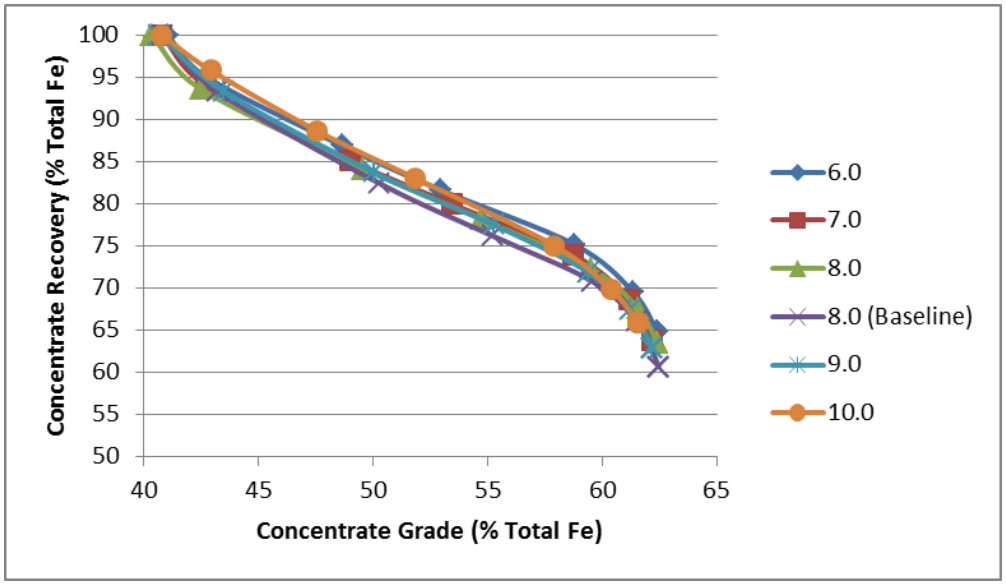


Figure 31: Effect of Flotation Feed pH Con. Grade vs. Recovery Plot (Test 18-45 to Test 18-49)

Table 37: Effect of Flotation Feed pH Con. Grade/Recovery Data Interpolated to 4.0% SiO_2 (Test 18-44 to Test 18-49)

Concentrate Grade/Recovery @ 4.0% SiO_2							
Test	**Flotation Feed pH Target**	**Measured Flotation Feed pH**	**Weight Recovery (%)**	**Total Fe (%)**	**Total Fe Recovery (%)**	**Amine (lb./LT)**	**Time (min)**
18-45	6	6.0	45.6	61.57	68.5	0.37	18.9
18-46	7	7.0	45.6	61.19	68.5	0.35	18.1
18-47	8	8.0	44.8	61.41	68.2	0.33	18.2
18-44	8.0 (Baseline)	8.0	44.5	61.34	66.9	0.33	17.7
18-48	9	9.1	45.1	61.14	67.8	0.34	17.8
18-49	10	10.1	44.2	61.38	66.5	0.44	N/A

Effect of Amine Type

The Air Products Tomamine PA-12 amine was used in all of the baseline and optimization studies; however, it was discovered during the research program that this product is no longer available commercially. Therefore, a small test series was conducted to test comparable monoamines that are commercially available with the goal of choosing one for future bench- and pilot-scale testing. The NatFlot tests were performed with the baseline conditions, namely the Bulk 19-2 sample using 0.11 kg/metric ton ICL Glass H (0.25 lb./LT), a total of 0.67 kg/metric ton (1.50 lb./LT) GPC S12F causticized modified starch, and six-stage flotation with a total of 0.20 kg/metric ton (0.45 lb./LT) total amine. The as-is Coleraine tap water measured 106.4 ppm Ca^{++} and 35.1 ppm Mg^{++}, for a calculated hardness of 410 ppm as $CaCO_3$. A 9-minute rod mill grind produced a flotation feed at approximately P80 38 µm (59% -25 µm).

The concentrate grade vs. concentrate recovery plot and concentrate grade/recovery data interpolated to 4.0% SiO_2 table can be seen in Figure 32 and Table 38, respectively. As can be seen in the figure and table, it was found that all of the monoamines that were tested had slightly higher total iron recovery and/or lower total flotation time than the baseline PA-12 amine. The

two tests that were the most promising were tests Test 18-32 and Test 18-33. Of these two tests, the Test 18-33 indicated slightly lower recovery but shorter total flotation time. The chemical used for this test was Clariant Flotigam EDA; this was the amine chosen for pilot-scale testing based on the bench results and chemical availability. This test series should be repeated and confirmed with various other mono- and diamines in future optimization testing programs. The metallurgical test data can be found in Appendix 24.

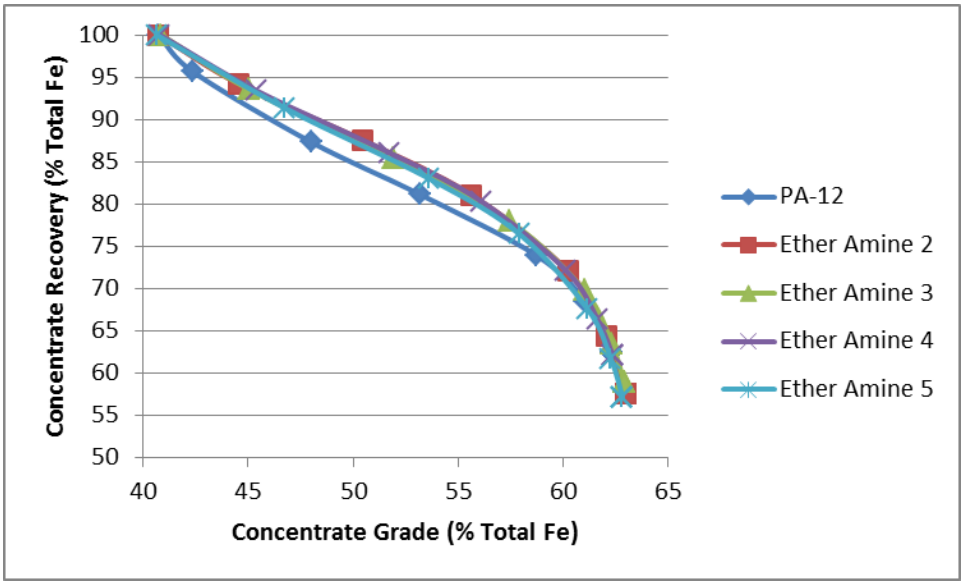


Figure 32: Effect of Amine Type Con. Grade vs. Recovery Plot (Test 18-30 to Test 18-34)

Table 38: Effect of Process Water Type Con. Grade/Recovery Data Interpolated to 4.0% SiO_2 (Test 18-30 to Test 18-34)

Concentrate Grade/Recovery @ 4.0% SiO_2						
Test	**Amine**	**Weight Recovery (%)**	**Total Fe (%)**	**Total Fe Recovery (%)**	**Amine (lb./LT)**	**Time (min)**
18-30	PA-12	43.8	61.52	66.3	0.39	17.4
18-31	Ether Amine 2	43.9	61.62	66.3	0.33	14.9
18-32	Ether Amine 3	45.0	61.47	67.6	0.28	14.8
18-33	Ether Amine 4	44.4	61.50	67.0	0.34	13.8
18-34	Ether Amine 5	43.8	61.45	66.1	0.28	N/A

Reproducibility Testing

At the conclusion of this optimization test series, it was determined that the following conditions be used for further testing. These optimized baseline conditions included a flotation feed grind target of approximately P80 38 µm (59% -25 µm), no polyphosphate (e.g., Glass H), 0.67 kg/metric ton (1.50 lb./LT) GPC S12F causticized modified starch, and staged Clariant Flotigam EDA amine. The Coleraine, Minnesota, tap water was used at a natural pH for process water. The optimized baseline and 25:75 Coleraine tap-to-distilled water blends were investigated for repeatability and resolution on the data generated.

Optimized Baseline Triplicate Testing

Three individual NatFlot tests were performed to determine repeatability and resolution for the bench flotation data with the optimized parameters discussed in the *Bulk 19-2 Optimization Testing* section, above. The optimized NatFlot test was performed in triplicate with the Bulk 19-2 sample using a total of 0.67 kg/metric ton (1.50 lb./LT) GPC S12F causticized modified starch and six-stage flotation with a total of 0.20 kg/metric ton (0.45 lb./LT) Clariant Flotigam EDA amine. The process tap water, collector dosage, and flotation time per stage were held constant in each test; the collector and flotation time per stage can be seen in Table 39. The froth (tailings) from these tests were combined and retained for tailings thickening testing[11].

Table 39: Collector Dosage and Flotation Time per Stage (Test 18-51 to Test 18-53)

Flotation Stage	Collector Dosage (g/met ton)	Collector Dosage (lb./LT)	Flotation Time (min)
1	22.3	0.05	3.5
2	22.3	0.05	4.0
3	22.3	0.05	3.5
4	44.6	0.10	3.0
5	44.6	0.10	2.0
6	44.6	0.10	1.0
Total	**200.9**	**0.45**	**17.0**

The concentrate grade vs. concentrate recovery plot and concentrate grade/recovery data interpolated to 4.0% SiO_2 table can be seen in Figure 33 and Table 40, respectively. The three-test series resulted in an average weight recovery and total iron recovery of 44.4% and 67.2%, respectively. The standard deviation for the weight recovery and total iron recovery were 0.23% and 0.35%, respectively. The average total amine and flotation time interpolated to 4.0% SiO_2 found very similar dosages at 0.17 kg/metric ton (0.38 lb./LT) and 16.4 minutes, respectively. The metallurgical test data can be found in Appendix 25.

[11] See the *Flotation Froth (Tailings) Settling Testing* section on Page 56 for more information.

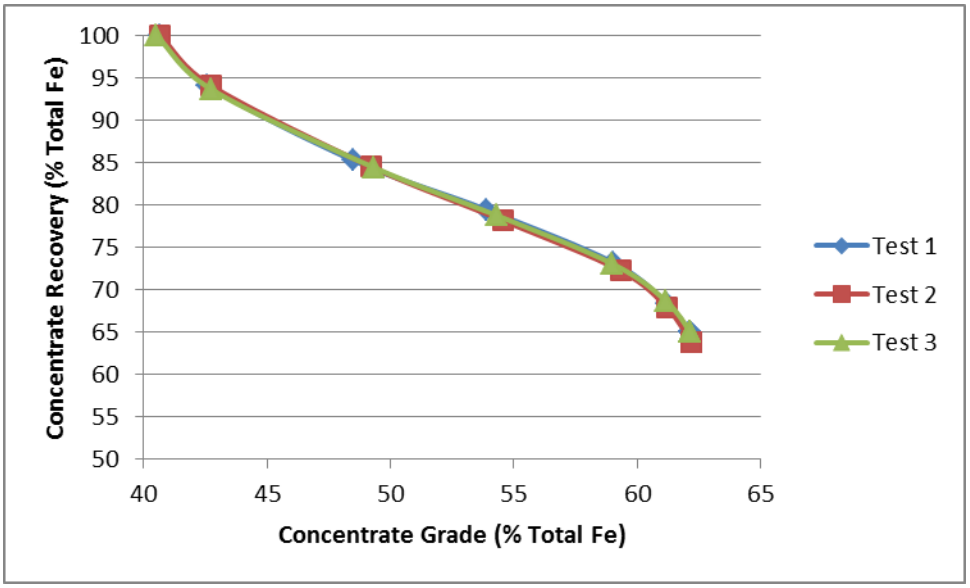


Figure 33: Concentrate Grade vs. Recovery Plot (Test 18-51 to Test 18-53)

Table 40: Concentrate Grade/Recovery Data Interpolated to 4.0% SiO_2 (Test 18-51 to Test 18-53)

Concentrate Grade/Recovery @ 4.0% SiO_2						
Test	**Test**	**Weight Recovery (%)**	**Total Fe (%)**	**Total Fe Recovery (%)**	**Amine (lb./LT)**	**Time (min)**
18-51	Test 1	44.7	61.44	67.5	0.37	16.4
18-52	Test 2	44.1	61.49	66.7	0.38	16.4
18-53	Test 3	44.4	61.51	67.4	0.39	16.5
Average		**44.4**	**61.48**	**67.2**	**0.38**	**16.4**
Std. Dev.		**0.23**	**0.03**	**0.35**	**0.01**	**0.08**

It should be noted that a new Bulk 19-2 sample was split out for bench testing following this series. Therefore, it was expected that the following tests may show slightly different results when comparing to this and previous test series.

As-Is vs. Reduced-Hardness Triplicate Water Testing

A test series was conducted to obtain additional baseline data of the metallurgical performance of the NatFlot process with the Bulk 19-2 sample using as-is Coleraine tap water versus reduced-hardness water. A triplicate series of the optimized NatFlot test conditions with as-is Coleraine tap water was compared to a triplicate series of Coleraine tap water diluted with distilled water. The reduced-water was diluted at a ratio of 25:75 as-is Coleraine tap water to distilled water. The as-is Coleraine tap water measured 92.2 ppm Ca^{++} and 36.2 ppm Mg^{++}, for a calculated hardness of 379 ppm as $CaCO_3$. The 25:75 Tap:Distilled water measured 21.5 ppm Ca^{++} and 9.4 ppm Mg^{++}, for a calculated hardness of 92 ppm as $CaCO_3$.

Each test used the optimized NatFlot test conditions; specifically, a total of 0.67 kg/metric ton (1.50 lb./LT) GPC S12F causticized modified starch and six-stage flotation with a total of 0.20

kg/metric ton (0.45 lb./LT) Clariant Flotigam EDA amine. The end point for each flotation stage was determined by the flotation operator and the time per flotation stage recorded.

The concentrate grade vs. concentrate recovery plot and concentrate grade/recovery data interpolated to 4.0% SiO_2 table are illustrated in Figure 34 and Table 41, respectively. Figure 34 shows a higher apparent difference between the 100% tap water and diluted water blend in the 45-60% total iron concentrate grade/recovery range, but the differential becomes less when nearing final concentrate grade (>60% Total Fe). The average of the 100% tap water triplicate interpolated to 4.0% SiO_2 concentrate silica grade found an apparent increase of 1.6% weight recovery and 2.1% iron recovery compared to the diluted water triplicate tests at a slightly lower amine dosage level (0.35 lb./LT versus 0.38 lb./LT). However, the total flotation time for the 100% tap water test was approximately three minutes longer. This suggests that water with a lower concentration of cations may reduce the floatability of certain minerals at a natural pH, as was also found in the *Effect of Process Water Type* test series. The impact of the water system on the performance of the NatFlot system should be investigated further in future research programs. The metallurgical test data can be found in Appendix 26.

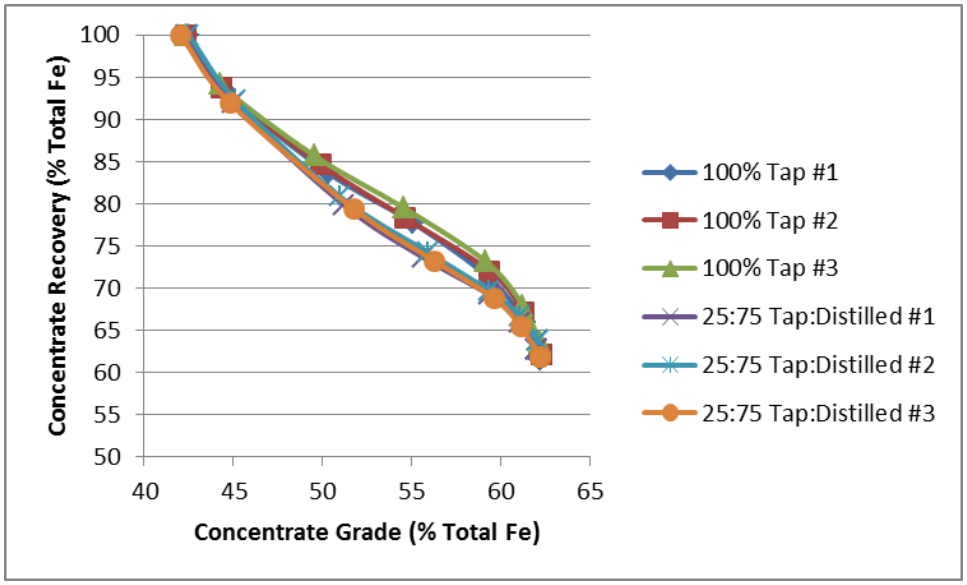


Figure 34: Concentrate Grade vs. Recovery Plot (Test 18-62 to Test 18-67)

Table 41: Concentrate Grade/Recovery Data Interpolated to 4.0% SiO_2 (Test 18-62 to Test 18-67)

Concentrate Grade/Recovery @ 4.0% SiO_2						
Test	**Test**	**Weight Recovery (%)**	**Total Fe (%)**	**Total Fe Recovery (%)**	**Amine (lb./LT)**	**Time (min)**
18-62	100% Tap #1	45.8	61.27	66.1	0.35	18.4
18-63	100% Tap #2	46.7	61.17	67.4	0.34	18.4
18-64	100% Tap #3	46.8	61.27	67.8	0.35	18.6
Average		**46.4**	**61.24**	**67.1**	**0.35**	**18.5**
Std. Dev.		**0.44**	**0.05**	**0.72**	**0.00**	**0.08**
18-65	25:75 Tap:Distilled #1	44.5	61.47	64.6	0.39	15.6
18-66	25:75 Tap:Distilled #2	45.3	61.50	65.5	0.39	15.6
18-67	25:75 Tap:Distilled #3	44.6	61.27	65.0	0.36	15.2
Average		**44.8**	**61.42**	**65.0**	**0.38**	**15.5**
Std. Dev.		**0.33**	**0.10**	**0.38**	**0.01**	**0.21**

Recovery by Size Testing

A single NatFlot test with the Bulk 19-2 sample was performed with the optimized test parameters mentioned above to investigate the recovery by size. The Bulk 19-2 flotation feed and flotation underflow (concentrate) were screened from 53 µm down to 25 µm, and the screen products submitted for total iron and silica assay. The assay by size on the tailings was then balanced from the feed and concentrate size distribution and assays.

The concentrate assay and recovery by size data can be seen in Table 42. The table indicates that the total iron recovery is highest in the -37 µm + 25 µm range, and decreases above and below this range. The highest recovery was found at the 25 µm (500 Mesh) particle size. The concentrate silica grade decreases as the particle size decreases, as was expected. The metallurgical data can be found in Appendix 27.

Table 42: Concentrate Assay and Recovery by Size

Tyler Mesh	**Micron**	**Concentrate Grade**		**Concentrate Recovery**		
		Total Fe (%)	**SiO_2 (%)**	**Weight Rec (%)**	**Total Fe Rec (%)**	**SiO_2 Rec (%)**
+270	53	61.54	4.58	43.9	58.7	8.2
325	44	61.54	3.39	67.5	85.1	10.9
400	37	62.09	2.93	57.8	75.6	6.9
500	25	62.79	2.36	64.7	94.7	4.6
-500	-25	63.05	2.17	29.3	44.5	1.8

Flotation Froth (Tailings) Settling Testing

The combined froth (tailings) from the baseline triplicate testing (Test 18-51 to Test 18-53) was used for a small tailings settling testing program. The tailings had a specific gravity of 3.11 g/cc. The tailings material was tested in a 2 L cylinder and mixing conducted manually with a mixing paddle. The baseline test was conducted without any added chemical reagents, and this was compared against four dosage levels of aluminum sulfate (alum) and three dosage levels of Neo Solutions NS6850 polymer flocculant.

In summary, the aluminum sulfate did not increase the settling rates compared to the baseline (see Figure 35). In fact, the tests with alum may have settled slower than the baseline test. Conversely, the tests with the NS6850 polymer flocculant indicated much faster settling rates than the baseline test (see Figure 36). Using 10.0 cm (or approximately 45% solids w/w to the underflow) as a reference point, the polymer flocculant produced a settling rate of 1.57 cm/min, 2.53 cm/min, and 3.67 cm/min for the NS6850 dosages rates of 0.4 g/metric ton (0.001 lb./LT), 4.5 g/metric ton (0.01 lb./LT), and 45 g/metric ton (0.1 lb./LT), respectively, compared to the baseline settling rate of 1.35 cm/min. The settling rate increased 16.0%, 87.4%, and 172.0% for the NS6850 dosages rates of 0.4 g/metric ton (0.001 lb./LT), 4.5 g/metric ton (0.01 lb./LT), and 45 g/metric ton (0.1 lb./LT), respectively, compared to the baseline test. The metallurgical test data can be found in Appendix 28.

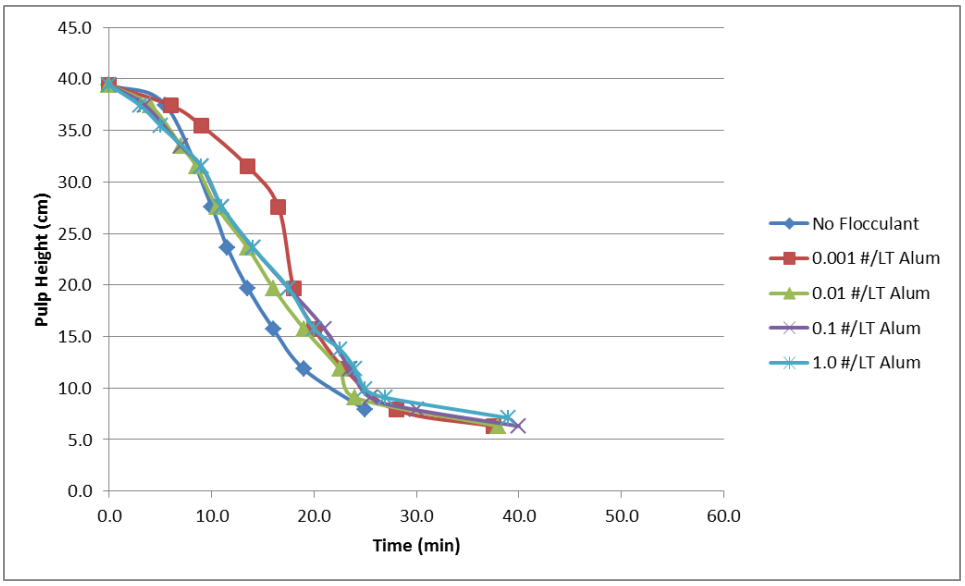


Figure 35: Baseline and Aluminum Sulfate Settling Tests

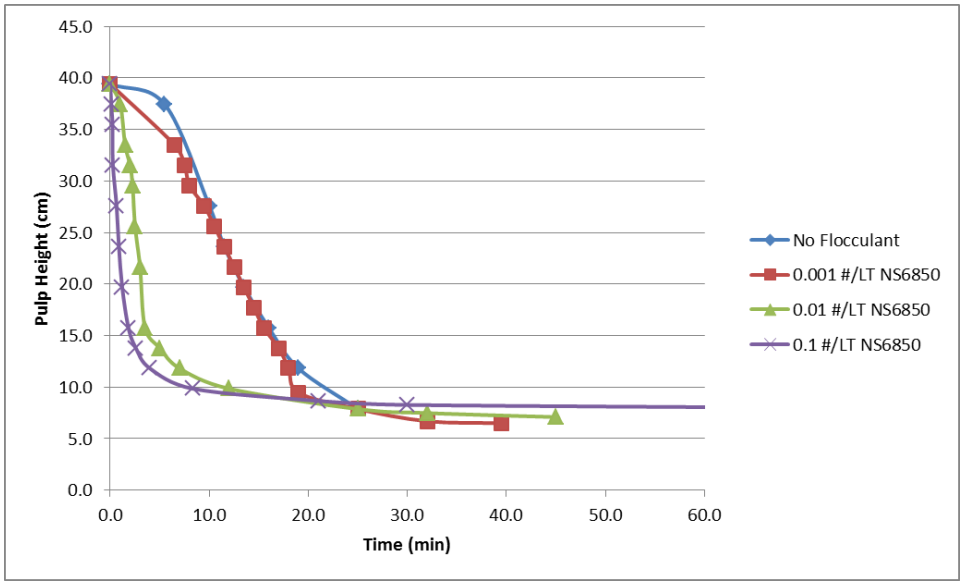


Figure 36: Baseline and Neo Solutions NS6850 Polymer Flocculant Settling Tests

Water System Testing

Calcium and Magnesium Leaching Testing

A series of three tests were conducted to determine the effect of calcium and magnesium ions leaching into the process water during grinding and mixing. Six hundred grams of each of the three bulk samples, 16-1, 19-1, and 19-2, were ground in the bench-scale rod mill for 10-minutes with 325 mL of Coleraine tap water. After 10-minutes, the material was discharged and the mill was rinsed into the bench-scale Denver D1 flotation bowl for 10-minutes. The amount of water required to rinse out the mill and into the flotation bowl was recorded. After 10-minutes, the mixed slurry was poured into a pressure filter (without additional water) and the filtrate captured for water characterization.

The objective of this testing was to determine if the hardness of the process water would be influenced by the presence of calcium and magnesium minerals in the ore samples. Figure 37 shows the concentration of calcium and magnesium ions before and after the addition of individual ore samples to the process water, and indicates that the hardness of the water changes marginally with the addition of ore samples. The results indicate that the calcium and magnesium minerals, if present, do not easily leach divalent cations into the process water.

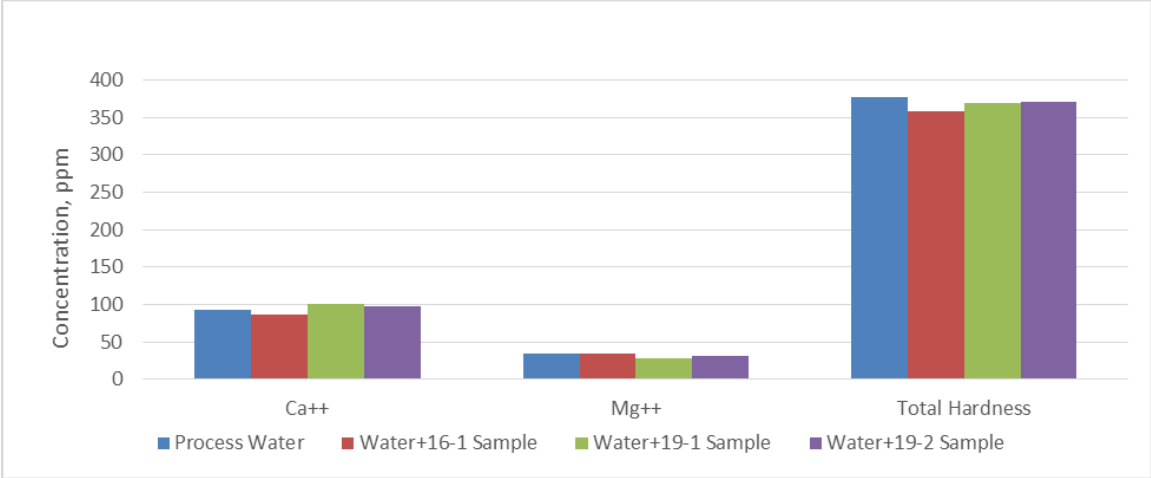


Figure 37: Concentration of Calcium and Magnesium Ions in Process Water and Ground Slurry Samples for Bulk 16-1, 19-1, and 19-2

Lock Cycle Water Testing

A lock-cycle test series was conducted to determine the effect of recycling the process water and to characterize concentrate quality, concentrate recovery, and basic water characterization. The objective was to find how reagent quantities must be adjusted to compensate for the circulating load of reagents and the effect of cations in a re-circulating water system; it was not to find an increase in recovery, which can be expected from recirculating the cleaner tailings, or to find whether slimes or other objectionable solid or soluble material will build up and interfere with flotation.

The optimized NatFlot test was performed with the 19-2 sample using a total of 0.67 kg/metric ton (1.50 lb./LT) GPC S12F causticized modified starch and six-stage flotation with a total of 0.20 kg/metric ton (0.45 lb./LT) Clariant Flotigam EDA amine. After completion of each cycle, the concentrate was filtered as-is and the filtrate water was retained. The tailings were also filtered and the filtrate retained, but were mixed and filtered with 0.45 kg/metric ton (0.10 lb./LT) NS6850 polymer flocculant. The filtrate from the concentrate and tailings were combined and used for the mill dilution water and flotation makeup water on the next cycle. The concentrate and tailings were submitted for chemical assay (e.g., total iron, silica, and phosphorus) and the water was submitted for water analysis (e.g., Ca^{++}, Mg^{++}, pH, and conductivity). The lock-cycle process flow diagram can be found in Appendix 29.

A few initial lock-cycle passes were conducted, which indicated that adding amine in the first stage of the next cycle created a large amount of froth volume. Therefore, the test was repeated without the use of the additional amine in the first stage in Cycles 2-6. The lock-cycle concentrate total iron, silica grade, weight recovery, and total iron recovery through six cycles produced a flotation concentrate containing an average of 62.6% total iron and 2.4% SiO_2 at an iron recovery of 58.1% (see Figure 38). Total collector (amine) consumption was constant at 0.18 kg/metric ton (0.40 lb./LT) per cycle. The calcium and magnesium measurements for each cycle can be seen in Table 43, and show that the calcium and magnesium levels decreased from approximately 356 ppm total hardness as $CaCO_3$ and stabilized at approximately 300 ppm total hardness as $CaCO_3$. Through six locked-cycle repetitions, it was concluded that the recycling of

water did not negatively impact the NatFlot concentrate grade or recovery. The metallurgical test data can be found in Appendix 30.

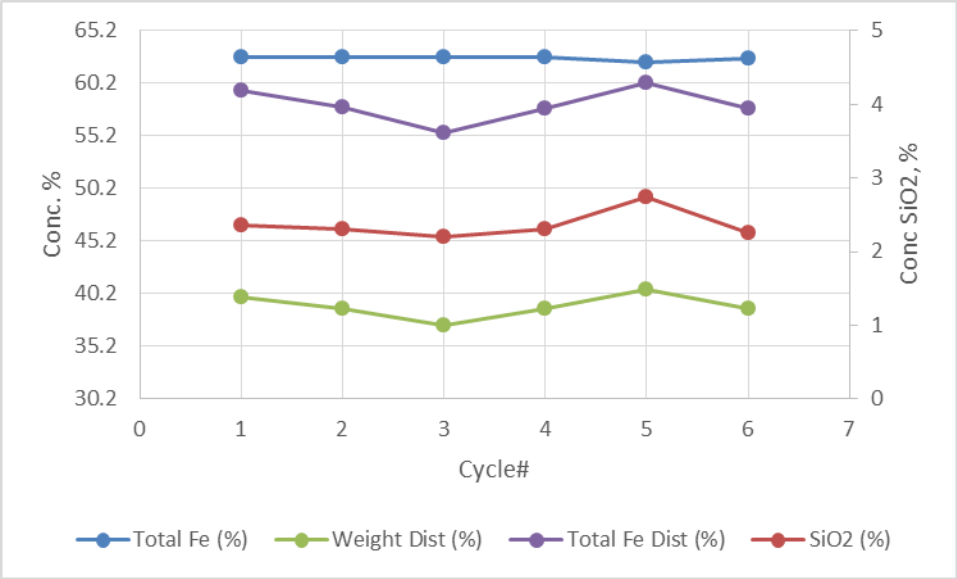


Figure 38: Concentrate Iron Grade, Silica Grade, Weight Distribution, and Total Iron Distribution per Cycle

Table 43: Calcium and Magnesium Ions per Cycle

Cycle	Ca^{++} (ppm)	Mg^{++} (ppm)	Total Hardness as $CaCO_3$ (ppm)
0 (Coleraine Tap)	91.3	31.0	356
1	84.2	30.0	334
2	83.4	27.8	323
3	82.4	25.1	309
4	82.9	22.3	299
5[12]	N/A	N/A	N/A
6	83.6	23.4	305
Average	**83.3**	**25.7**	**314**
Std. Dev.	**0.6**	**2.8**	**12.7**

Residual Amine Testing

Based upon the findings from the lock-cycle testing, a test series was conducted to obtain baseline information on the amount of residual amine that may be present in the process water after the NatFlot process. The optimized NatFlot test was performed with the Bulk 19-2 sample using a total of 0.67 kg/metric ton (1.50 lb./LT) GPC S12F causticized modified starch and six-stage flotation with a total of 0.20 kg/metric ton (0.45 lb./LT) Clariant Flotigam EDA amine.

[12] The water sample for Cycle 5 was mishandled and therefore could not be analyzed.

The amount of water added during each flotation stage was recorded, as well as the volume of water added to rinse the products into the pressure filter to determine total water volume. The concentrate and tailings were filtered, filtrate blended together, and a sample of the combined filtrate submitted to Pace Analytical for amine analysis.

The Test 18-82 test indicated that approximately 4,200 mL of water was used in total, with 1838 mL for flotation feed, 1175 mL of water for flotation makeup water, 1175 mL of water for product rinsing, and 11.6 mL of water from the dilute amine solution. Approximately 16 mL water remained in the concentrate and tailings filter cake. The amine concentration in the test, based upon total amine and total water usage, was estimated at 27.8 mg/L. Pace Analytical determined that the filtrate water had an amine concentration of 1.1 mg/L. Based upon the test measurements, the estimated amine removal rate was approximately 96.0%. The metallurgical test data and removal rate calculations can be found in Appendix 31. It should also be noted that the amine concentration in the effluent is likely lower than would be expected at pilot- or production-scale because of the relatively large amount of product rinse water used during the bench-scale test.

Bulk 19-2 Mineral Floatability Testing

During the residual amine testing test (Test 18-82), the products from each flotation were retained and submitted for XRD analysis to determine the floatability of each mineral in the NatFlot process with the Bulk 19-2 sample. The results are summarized in Figure 39. In summary, the amount of quartz decreased and the amount of hematite and goethite increased through each stage of the NatFlot process with the Bulk 19-2 material. The first two froth products are characterized as having high weight recovery (13-22%) and are composed primarily of quartz (>60%). The hematite-to-goethite ratio was about the same at approximately 1.0:1.6. The next two froth products were found to have moderate weight recovery of about 8% and composed of a mixture of quartz, hematite, and goethite, but was primarily quartz. The final two froth products found low weight recovery (3-6%), primarily composed of goethite (~65%), hematite (~20-30%), and a small amount of quartz (3-8%) and siderite (2-3%). The flotation underflow (concentrate) had very little quartz (1.5%), with the remaining being composed of approximately one-third hematite and two-thirds goethite. The data indicates that quartz is the most floatable while goethite and hematite are the least floatable. The carbonate minerals siderite and ankerite were less floatable than quartz.

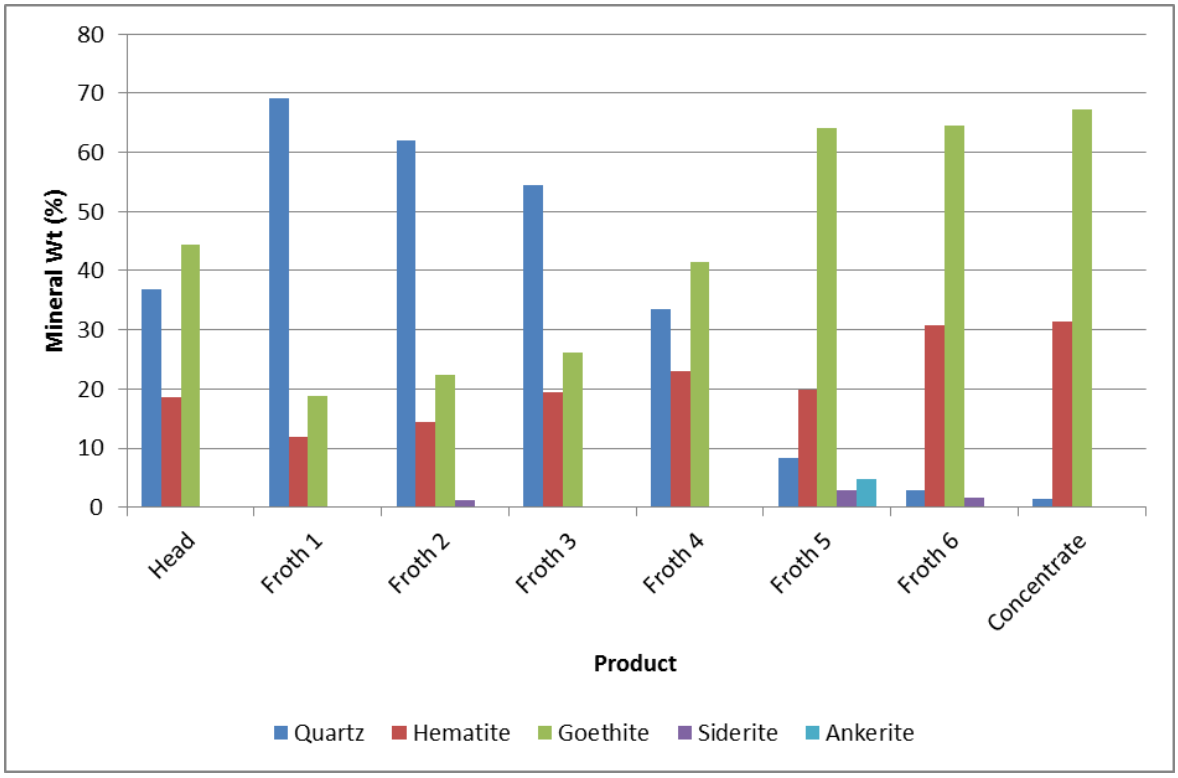


Figure 39: Optimized NatFlot Mineralogy per Product for Bulk 19-2

The floatability of each mineral, by total collector dosage and total collector time, can be seen in Figure 40 and Figure 41, respectively. The data indicated that 90% of the quartz was rejected with a total of 79.1 g/metric ton (0.18 lb./LT) collector and a total flotation time of 12.3 minutes. At 95% rejection of quartz, these values changes to 104.2 g/metric ton (0.23 lb./LT) and 14 minutes, respectively.

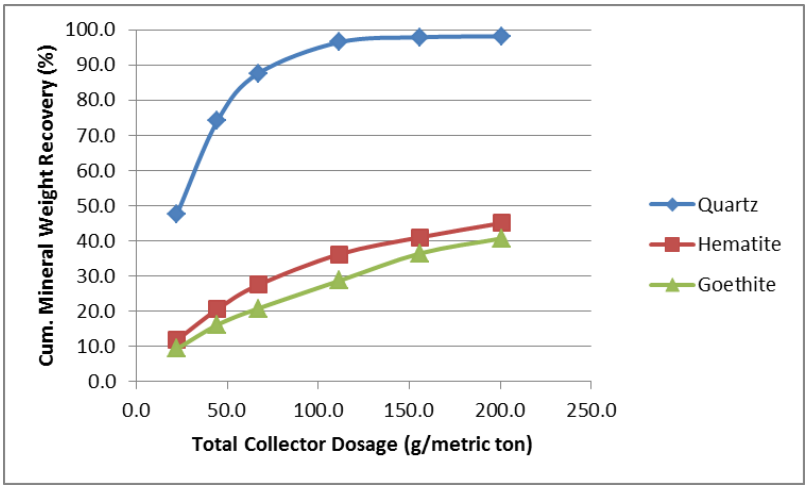


Figure 40: Cumulative Mineral Weight Recovery vs. Total Collector Dosage for NatFlot Test with Bulk 19-2

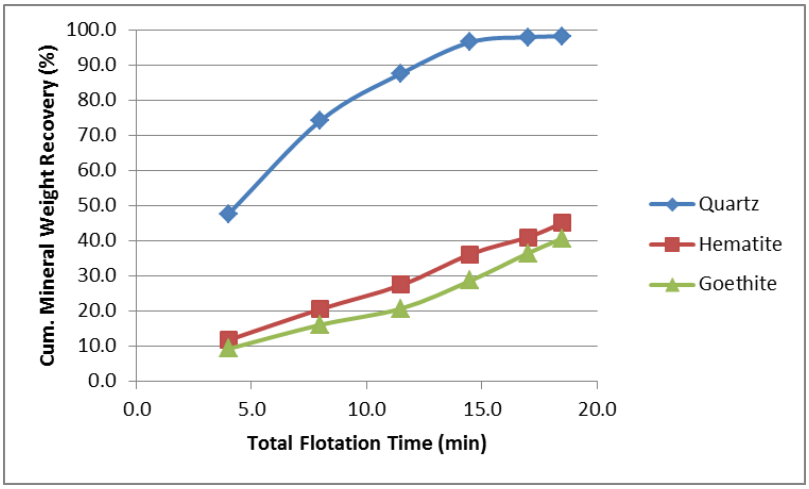


Figure 41: Cumulative Mineral Weight Recovery vs. Total Flotation Time for NatFlot Test with Bulk 19-2

Bulk 19-2 Sample Selective Flocculation-Deslime-Flotation Testing

After the conclusion of the optimization and water study, a single SFDF test was performed with the Bulk 19-2 sample as a direct comparison to a NatFlot bench test with the optimized conditions described in the *Residual Amine Testing* (Test 18-82) on page 60. The same 9-minute rod mill grind was used as in the NatFlot optimized test with the Bench 19-2 sample. The same baseline SFDF conditions were used in this testing as described in the *Bench Samples Selective Flocculation-Deslime-Flotation (SFDF) Testing Summary* section on page 28, with the exception that the total starch dosage was increased from 0.56 kg/metric ton (1.25 lb./LT) to 0.67 kg/metric ton (1.50 lb./LT) to match the optimized NatFlot starch dosage.

The metallurgical results are summarized in Table 44. In summary, it was discovered that the NatFlot process was able to further reduce the silica content in the concentrate compared to the SFDF baseline test at a reduced collector dosage, but took approximately nine minutes longer; an increase of 85%. It should be noted that the NatFlot used optimized conditions while the SFDF test used baseline conditions for this comparison.

Table 44: NatFlot vs. SFDF Metallurgical Data Summary for Bench 19-2

Test	Sample	Flowsheet	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
18-82	19-2	NatFlot	39.4	62.67	2.34	0.033	58.5	0.35	18.5
18-68	19-2	SFDF	43.5	62.24	2.97	0.035	63.7	0.50	10.0

To directly compare the results of the metallurgical performance of the two tests, a grade vs. recovery plot was produced. The data is summarized in Figure 42 and Table 45, and shows that the SFDF process had higher recovery but could produce a lower final concentrate silica product. Interpolating the test data to a concentrate grade of 4.0% SiO_2, the SFDF had a weight and iron increase over the NatFlot process by 2.0% and 2.5%, respectively. Similar to other baseline comparison tests, the SFDF baseline test found a decrease in flotation time of about half. However, the baseline SFDF test used approximately 31.2 g/metric ton (0.07 lb./LT) more amine

than the baseline NatFlot tests to achieve the silica target. The metallurgical test data for this series can be found in Appendix 32.

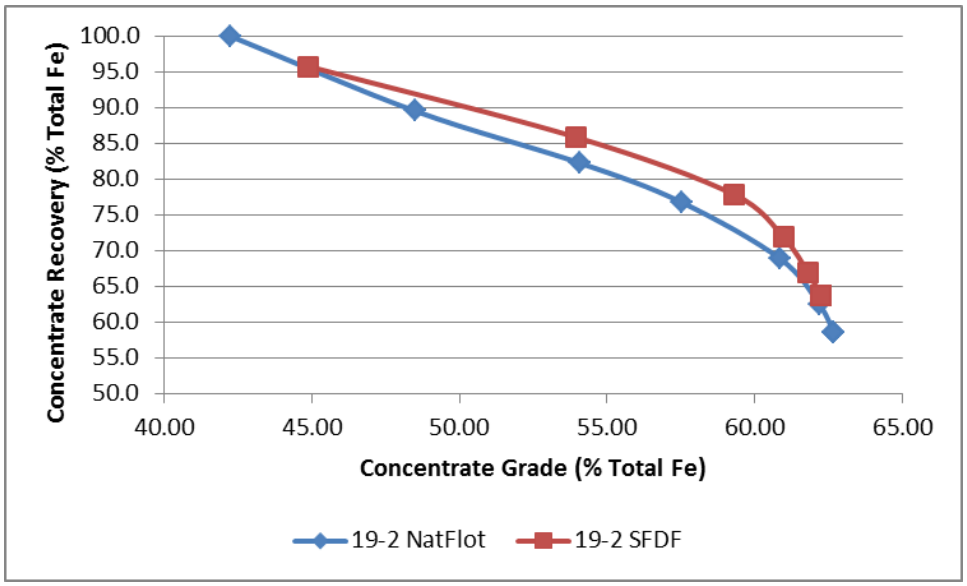


Figure 42: NatFlot and SFDF Concentrate Total Iron Grade vs. Recovery Plot for Bench 19-2

Table 45: Concentrate Grade/Recovery Data Interpolated to 4.0% SiO_2 (Test 18-82 to Test 18-68)

Concentrate Grade/Recovery @ 4.0% SiO_2							
Test	**Sample**	**Flowsheet**	**Weight Recovery (%)**	**Total Fe (%)**	**Total Fe Recovery (%)**	**Amine (lb./LT)**	**Time (min)**
18-82	19-2	NatFlot	46.0	61.29	66.8	0.28	15.3
18-68	19-2	SFDF	48.0	61.40	69.3	0.35	7.5

Summary of Bench-Scale Beneficiation Testing and Bulk Sampling Selection

Baseline NatFlot tests and conventional beneficiation process comparative testing was conducted on the Bulk 16-1 and Bulk 19-1 and 19-2 samples. The purpose of these tests was to determine how the samples reacted to the different processes, as well as to determine which sample(s) would be pursued for pilot-scale testing. In summary, both the Bulk 16-1 and 19-1 samples achieved better metallurgical results with the typical low-intensity magnetic separation process than with flotation, but liberated oxidized iron minerals were lost with these processes. Wet high-intensity magnetic separation helped to increase recovery of the oxidized iron minerals, but contained a large amount of quartz and other gangue minerals (e.g., talc). More optimization testing may improve on the grade-recovery relationship with partially oxidized samples such as Bench 18-3 and Bulk 16-1, but the baseline testing concluded that implementing WHIMS as a scavenger system for this type of sample may be impractical due to a low-iron concentrate produced at a low weight recovery.

Contrary to the Bulk 16-1 and 19-1 samples, the Bulk 19-2 found better results with flotation than with magnetic separation. After conducting a small optimization study, it was discovered that the NatFlot process was able to further reduce the silica content in the concentrate compared

to the SFDF baseline test at a reduced collector dosage, but had a flotation residence time increase of approximately 85%. Interpolating to a 4.0% SiO_2 concentrate grade, the SFDF had a slight weight and iron increase over the NatFlot process by 2.0% and 2.5%, respectively. Similar to other baseline comparison tests, the SFDF baseline test indicated a decrease in flotation time of about half. However, the baseline SFDF test used more amine than the baseline NatFlot tests to achieve the silica target.

Although the SFDF process had slightly higher recovery at the same silica grade, it was determined that the results from bench testing had enough merit to investigate the NatFlot process with the Bulk 19-2 material on pilot-scale. This decision was based on the potential to recover oxidized iron minerals (reduction of waste) with a natural pH process (with reduction of chemicals and lower environmental impact). The scope, goals, objectives, and deliverables for this pilot-study are discussed in the following sections.

Pilot-Scale Beneficiation Testing Summary

The original pilot-scale testing scope called for producing 1-2 tons of concentrate for agglomeration and pyrometallurgical testing. However, the scope was modified to reduce the amount of pilot-scale concentrate required and limit the bench- and pilot-scale pyrometallurgical to up to four mini-pot and two pilot pot-grate tests. The scope was altered due to project timing constraints and the limited amount of Bulk 19-2 material available for pilot testing because of the sampling contamination.

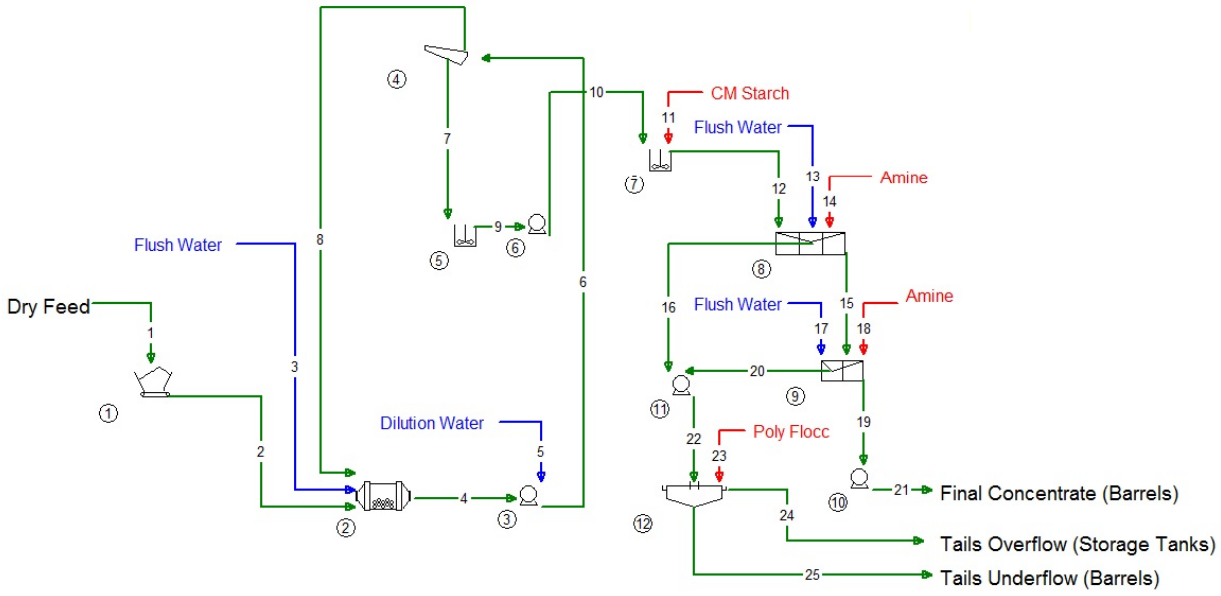
To investigate the NatFlot process on pilot-scale, a program was conducted with the Bulk 19-2 material from late-June to mid-July, 2018. The goals of this testing program were to 1) investigate the metallurgical performance of the NatFlot process at relevant scale, 2) characterize the solid products (e.g., concentrate and tailings) and process water, and 3) produce at least 272 kg (600 lbs.) of concentrate for agglomeration and pyrometallurgical testing. The target concentrate specification was a concentrate with 4.0-4.5% SiO_2 with no more than 80% of the particles finer than 30 µm.

Pilot Testing Sample, Flowsheet, Material Handling, and Test Conditions

The remaining Bulk 19-2 material was processed via HPGR with the same conditions referenced in the *High-Pressure Grinding Rolls (HPGR) Processing of Bulk 19-1 and 19-2* section, and blended to be used for feed in the continuous pilot-scale testing program. The NatFlot process flowsheet used for continuous pilot-scale testing was the following:

1) Closed-circuit ball milling closed with a 44 µm (325 Mesh) screen,
2) Screen undersize conditioned for approximately three minutes in a conditioning tank with causticized modified starch for iron depression,
3) A total of five flotation stages in a rougher-cleaner configuration with mono-amine for silica collection, and
4) Tails thickening in thickener tank with polymer flocculant as a flocculation aid.

The process flowsheet can be seen in Figure 43, below.



Key for Unit Operations

1) Dry Feeder
2) Ball Mill
3) Ball Mill Discharge Pump
4) Derrick Stacksizer Screen
5) Screen Undersize Sump
6) Screen undersize Pump
7) Conditioning Tank
8) Rougher Flotation Bank
9) Cleaner Flotation Bank
10) Concentrate Pump
11) Total Tails Pump
12) Tailings Thickener

Key for Process Streams

1) Dry Feed In
2) Ball Mill Fresh Feed
3) Ball Mill Feed Flush Water
4) Ball Mill Discharge
5) Screen Feed Dilution Water
6) Screen Feed
7) Screen Undersize
8) Screen Oversize
9) Screen Undersize Sump Discharge
10) Conditioning Tank Feed
11) Causticized Modified Starch In
12) Rougher Flotation Feed
13) Rougher Flotation Flush Water
14) Rougher Amine In
15) Rougher Flotation Underflow (Con)
16) Rougher Flotation Froth (Tailings)
17) Cleaner Flotation Flush Water
18) Cleaner Amine In
19) Cleaner Flotation Underflow (Con)
20) Cleaner Flotation Froth (Tailings)
21) Concentrate Barrel Feed
22) Tailings Thickener Feed
23) Polymer Flocculant In
24) Tailings Thickener Overflow (Water)
25) Tailings Thickener Underflow (Solids)

Figure 43: Continuous Pilot-Scale Process Flowsheet

The crushing and HPGR processing were conducted in batches, with the products from each of these being blended before progressing to the next stage of the testing. The ball milling, conditioning, and flotation process were tested in a continuous process, as the adhesion of starch to iron requires fresh-ground surfaces.

The thickener was operated as a sump, with the underflow (e.g., high-solids slurry) pumped out of the thickener into barrels every 2-3 hours during operation. During this time, the thickener overflow was continuously gravity-fed to a series of septic tanks for holding, characterization, and ultimately, for disposal at a local water treatment facility. Dewatering of the final cleaner underflow (e.g., cleaner concentrate) and tails thickener underflow (e.g., total tailings) were completed via settling and decanting from barrels; the water was also stored in the septic tanks for disposal.

The process water for this testing series was the city of Coleraine, Minnesota, tap water heated with a single 14-kW on-demand electric hot water heater. The process water was not recycled during this testing phase.

A total of five flotation stages were used in this pilot-testing; three 101 L (26.7 gal) cells in the rougher bank and two 100 L cells in the cleaner bank, for a total volume of 505 L (133.4 gal). The divider weirs between each flotation cell were completely blanked off with a divider except

for a small 5.1 cm (2.0 in) hole in the center. The flotation cell divider design can be found in Appendix 33.

The purpose of this flotation cell divider design was to simulate discrete flotation stages and more closely mimic the bench results. This design would also allow for more control on each flotation stage by controlling the amount of collector (amine) added at a specific time with the goal of improving selectivity. Maintaining discrete cells within the flotation banks would also allow for a longer flotation residence time minimizing material losses to the froth product stream.

Pilot Sampling Plan

During each run the pilot operators would measure specific areas of interest providing data to evaluate the overall process performance. In the grinding circuit, the operators would manually measure dry feedrate, ball mill discharge density, and screen undersize. The mill power draw would be automatically recorded via a Fluke power datalogger. In the flotation circuit, the operators would measure flotation feed density, temperature, and pH. The reagent dosages would also be measured and recorded in each flotation cell. The engineers would record water usage, on-demand hot water target, screen frequency, and screen panel size.

The basic sampling plan for each pilot run called for taking cuts every 20-30 minutes throughout the duration of the run to form a composite sample for size, metallurgical balance, and concentrate quality analyses. There were a total of seven sampling points for the composite samples. Sampling began a minimum of one hour after the start of the pilot test and continued throughout the run with the exception of periods of process interruption. The sampling plan for the pilot runs can be found in Appendix 34.

In addition to the composite sampling points, there were typically four cleaner flotation concentrate (underflow) samples taken throughout the run to determine the concentrate grade. In the final few runs, samples were also taken of the flotation slurry to measure calcium and magnesium ion levels and to obtain slurry for bench-scale flotation testing.

Pace Analytical acquired a composite sample over the course of one hour of the tailings thickener overflow and analyzed for: low-level mercury (raw sample and dissolved in water), amine, anions, alkalinity, chemical oxygen demand (COD), biochemical oxygen demand (BOD), total suspended solids (TSS), total Kjeldahl nitrogen (TKN), nitrate, total phosphorus, diesel range organics (WIDRO), pH, total nitrogen, specific conductance, metals (raw sample and dissolved in water), heavy metals (raw sample and dissolved in water), and naphthalene.

Pilot Testing Results

A total of approximately 8.3 metric tons of the Bulk 19-2 sample was processed with a KHD/Weir HPGR in lock-cycle from 100% passing 19.1 mm (0.75 inch) to 100% passing 3.4 mm (6 Mesh) using the same conditions as referenced in the *High-Pressure Grinding Rolls (HPGR) Processing of Bulk 19-1 and 19-2* section, above. The HPGR was operated at a specific press force of 3.4 N/mm^2 and at a rolls speed of 15 RPM. The HPGR operating parameters and test data for the first five cycles can be seen in Table 46. The grinding energy consumption was

calculated at 1.0 kWh/metric ton and the average specific throughput[13] was estimated at 407 ts/hm³.

Table 46: HPGR Operating Parameters and Test Result Summary

Cycle		1	2	3	4	5
Operating Parameter Set Point						
Nitrogen Pressure	bar	28	28	28	28	28
Hydraulic pressure	bar	15	15	15	15	15
Rolls Speed	RPM	15	15	15	15	15
Rolls Speed	m/s	0.71	0.71	0.71	0.71	0.71
Test Data						
Throughput	tph	63.8	66.1	65.3	65.3	63.1
Specific Throughput	ts/hm³	401.0	415.6	410.7	410.7	396.7
Specific power, Net	kWh/t	1.1	0.9	1.0	1.0	0.9

A sample of the 3.4 mm (6 Mesh) screen undersize was taken for a ball mill Bond Work Index (BWI) test to estimate the energy consumption required for the Bulk 19-2 sample. This test resulted in an estimated energy consumption of 18.7 kWh/metric ton (19.0 kWh/LT[14]) for the Bulk 19-2 sample[15].

After completion of three commissioning runs, a total of six tests were executed to test the NatFlot process with the Bulk 19-2 material at pilot-scale. The goals of the pilot program were to investigate the metallurgical performance of the NatFlot process with the Bulk 19-2 sample at pilot-scale and to produce at least 272 kg (600 lbs.) of concentrate at a grade of 4.0-4.5% SiO_2 with no more than 80% of the particles finer than 30 µm for pyrometallurgical testing. Total run time was approximately 40 hours at approximately 0.20-0.24 metric tons per hour.

The optimized conditions from the bench-scale testing program were used as the baseline conditions for the six tests. Specifically, the following parameters were targeted for the pilot-scale testing program:

- Pilot Test Material: Bulk 19-2.
- Fine Grind: P80 <40 µm with >450% recirculating load.

[13] Note that specific throughput is defined as the throughput (in tons/hour) divided by the diameter (in meters), width (in meters), and speed of the rolls (in meters/second). The units are given in ts/hm³, and it signifies the capacity of a machine with a roll diameter of 1 m, a width of 1 m and a peripheral speed of 1 m/s.
[14] 19.0 kWh/LT = 17.0 kWh/st.
[15] The BWI test used a 44 µm sieve for screening the mill product during each cycle.

- Process water and temperature: Coleraine, Minnesota, tap water heated to 18- 20°C (65-68°F)[16]. The water system was open-circuit (e.g., no water recycling was conducted in the pilot testing phase).
- Depressant type and dosage: Grain Processing Corporation Supercore S12F modified corn starch, 0.67 kg/metric ton (1.5 lb./LT).
- Collector type and dosage: Clariant Flotigam EDA amine, 0.16 kg/metric ton (0.35 lb./LT) to 0.20 kg/metric ton (0.45 lb./LT).
- Ball mill discharge density: 60-65% solids (depending on test).
- Screen size and frequency: Derrick Stacksizer 45 µm polyurethane panel at 40 Hz.
- Screen undersize (aka flotation feed) density: 25-35% solids (depending on test).
- Tailings flocculant and dosage: Neo Solutions NS6850 polymer flocculant, up to 44.6 g/metric ton (0.1 lb./LT), and adjusted manually by visual of thickener overflow clarity.
- Maximum flotation residence time: approximately 65 minutes[17].

The designed ore and water balance for this process flowsheet can be found in Appendix 35.

The parameters that were varied throughout the six pilot-scale runs were dry feed rate, mill critical speed, ball mill discharge density, screen undersize density, and collector (amine) dosage schedule in the five flotation cells. All other parameters were held constant, as referenced above. The summary table of key process parameter targets for Run 2-6 can be seen in Table 47, below[18]. The feedrate in Run 2 was increased from 0.20 tons/hour to 0.24 tons/hour mid-run to increase screen undersize particle size based upon size distribution data generated during the test. The total amine dosage in Run 3 was increased from 0.16 kg/metric ton (0.35 lbs./LT) to 0.20 kg/metric ton (0.45 lbs./LT) mid-run to improve concentrate silica grade based upon XRF chemical assay generated during the test. The other process parameter targets were changed between runs based upon the previous run results and operator/engineer observations. The ball mill critical speed and ball mill density targets were chosen to reduce the power input into the ore as the mill was slightly oversized.

[16] All runs targeted 18°C (65°F) process water temperature with the exception of Run 3. During Run 3, the feed and discharge box control spray bars were piped from an unheated water source and the process water was therefore increased to 20°C (68°F) to compensate. The feed and discharge box control spray bars are discovered below.
[17] Actual flotation residence time is discussed in further detail, below.
[18] Pilot Run 1 was a short commissioning run with the Bulk 19-2 material, therefore, was not included in the summary table.

Table 47: Run 2-6 Pilot-Scale Beneficiation Process Targets

Run	Dry Feedrate (mt/h)	Mill Critical Speed (%)	Ball Mill Disch. Density (%)	Screen U/S Density (%)	Ro. Flot 1 (g/t)	Ro. Flot 2 (g/t)	Ro. Flot 3 (g/t)	Cl. Flot 1 (g/t)	Cl. Flot 2 (g/t)	Total Amine (g/t)[19]
2 (early-morning)	0.20	75	60-65	25-35	22.3	22.3	22.3	44.6	44.6	156.2
2 (mid-morning)	0.24	75	60-65	25-35	22.3	22.3	22.3	44.6	44.6	156.2
3 (morning)	0.24	70	60-62	25-35	22.3	22.3	22.3	44.6	44.6	156.2
3 (afternoon)	0.24	70	60-62	25-35	22.3	22.3	22.3	67.0	67.0	200.9
4	0.24	70	60-62	25-35	22.3	22.3	22.3	67.0	67.0	200.9
5	0.24	70	60-62	25-35	13.4	13.4	22.3	44.6	62.5	156.2
6	0.22	66	60-65	35	13.4	13.4	22.3	44.6	62.5	156.2

The cleaner flotation concentrate weight recovery and silica grade per pilot run can be seen in Figure 44, below. The cleaner flotation silica grade vs. total iron recovery plot for Runs 2-6 can be seen in Figure 45[20]. The metallurgical balance summary for the composite sample for each run can be found in Appendix 36. The pilot-scale operator sheet summary can be found in Appendix 37, and the individual run summary sheets can be found in Appendix 38 through Appendix 42.

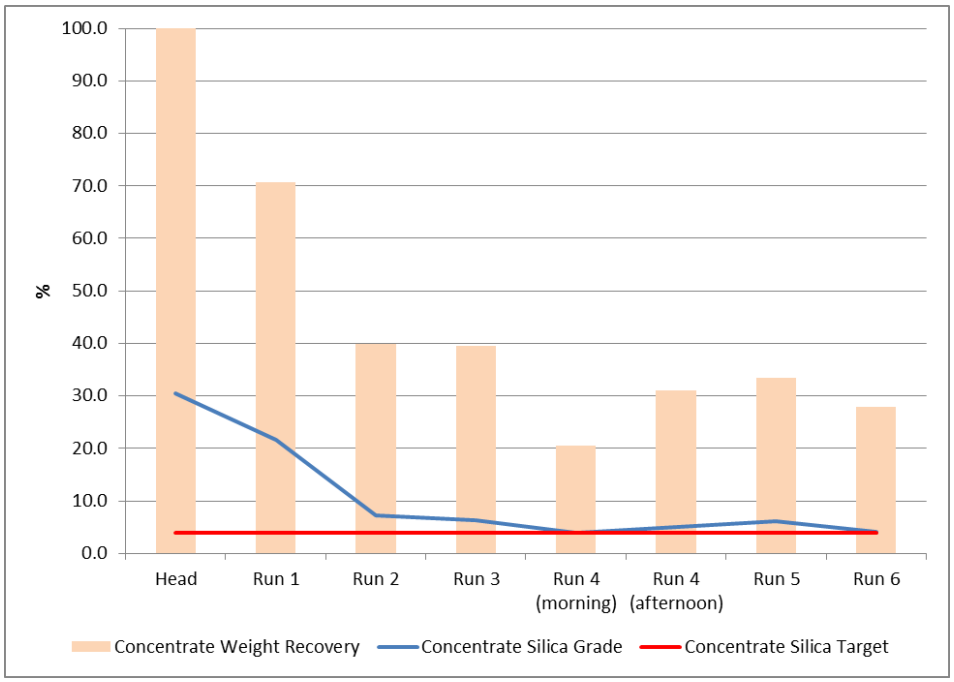


Figure 44: Cleaner Flotation Concentrate Weight Recovery and Silica Grade per Pilot Run

[19] Reference: 0.16 kg/metric ton = 0.35 lb./LT; 0.20 kg/metric ton = 0.45 lb./LT.
[20] Pilot Run 1 was a short commissioning run with the Bulk 19-2 material, therefore, was not included in the summary table.

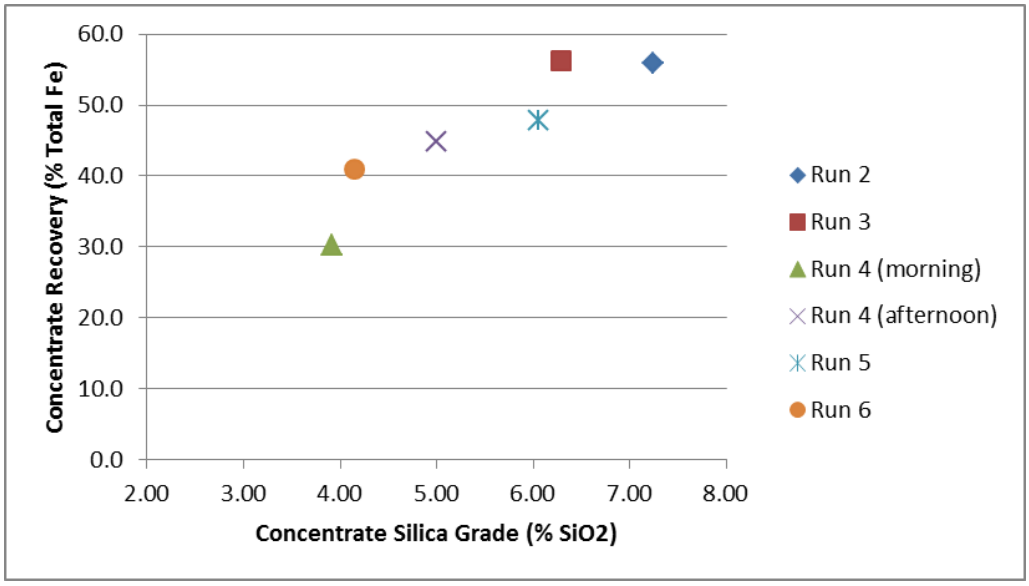


Figure 45: Cleaner Flotation Concentrate Silica Grade vs. Iron Recovery per Pilot Run

As seen in Figure 44 and Figure 45, the NatFlot rougher-cleaner flotation process was able to produce concentrate at the required silica grade of 4.0-4.5% SiO_2 at pilot-scale, but at a lower weight and iron recovery than projected by the bench testing. As a comparison to the pilot-scale results, the optimized NatFlot bench test[21] resulted in a weight and iron recovery of 46% and 66.8%, respectively, at a 4.0% SiO_2 concentrate silica grade.

One of the most critical aspects discovered in the early runs was the formation of a large froth volume, especially in the rougher flotation bank. Within the flotation cells, the froth volume was high, but could be managed by the use of conventional froth conveying paddles. However, in the feed and discharge boxes of the flotation banks, the froth would at times overload the boxes causing material loss.

After the second run, water spray bars were added to the feed and discharge ends on both the rougher and cleaner flotation banks to control the froth build-up. The addition of the spray bars in the feed and discharge worked well to control the froth, but added a significant amount of water to the circuit. This additional water caused dilution in the flotation circuit and therefore shorter residence time. This was confirmed by a re-cleaner bench flotation test on the cleaner flotation underflow (concentrate) during Run 4.

During Run 4, a cut was taken of the cleaner flotation underflow (e.g., final concentrate) and a re-cleaner NatFlot bench test was performed on the sample. The slurry sample was added to the Denver D1 2L flotation cell with no additional reagents. The slurry was mixed for one minute and air added to the cell. The slurry sample was floated until completion, approximately three minutes, and the products submitted for chemical analysis. The metallurgical test data is summarized in Table 48, below.

[21] Test 18-82.

Table 48: Run 4 Re-Cleaner Bench Flotation Test

Product	Weight (%)	Total Fe (%)	SiO_2 (%)		Total Fe Distribution (%)	SiO_2 Distribution (%)
Run 4 Cl. U/F Bench Froth	12.1	51.97	14.44		10.2	45.2
Run 4 Cl. U/F Bench U/F	87.9	62.76	2.41		89.8	54.8
Head (calc)	100.0	61.45	3.87		100.0	100.0

As can be seen in the table, the cleaner underflow sample continued to float without additional amine and reduced the silica grade from approximately 3.9% SiO_2 to 2.4% SiO_2 at a weight recovery of 88% and an iron recovery of 90%. The concentrate silica grade found from this re-cleaner bench test was similar to the previous bench testing; the Run 4 re-cleaner NatFlot bench test resulted in a concentrate silica grade of 2.4% SiO_2 vs. 2.3% SiO_2 for bench Test 18-82 at an amine dosage of 0.20 kg/metric ton (0.45 lb./LT). This finding suggests that using the froth control spray bars in the feed and discharge boxes of the rougher and cleaner flotation banks decreased the flotation residence time and therefore led to a higher concentrate silica grade. Following Run 4, the water flow to the froth control spray bars were reduced as much as possible but still maintained sufficient flow to control the froth buildup in the flotation bank feed and discharge boxes.

A set of samples were also taken from the froth products on each cell and the final underflow (concentrate) during Run 4 to compare the product chemical assay compared to the optimized NatFlot baseline Test 18-82[22]. The silica assay per product is plotted in Figure 46, and indicates that the average product for the first three (rougher) stages assayed about 10% higher in silica in the bench test compared to the results found in Run 4. The difference between the bench and pilot test are closer in the final two (cleaner) stages, but the bench test still found slightly higher selectivity as compared to the pilot test. The data for this comparison can be found in Appendix 43.

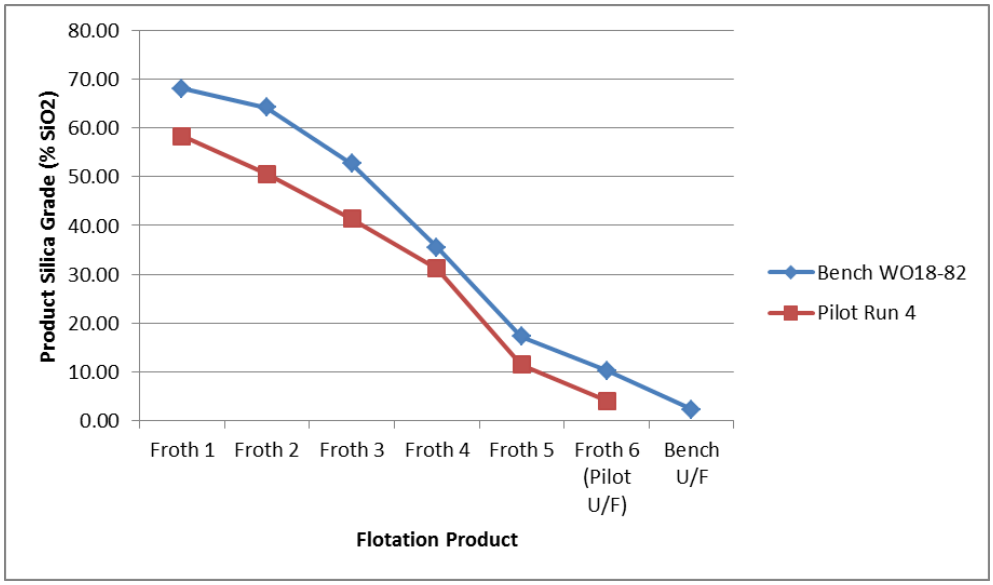


Figure 46: Pilot Run 4 Silica Grade by Product

[22] Recoveries were not calculated due to sample locations presenting difficulty in acquiring time-weighted samples.

One explanation for the difference between the bench- and pilot-scale testing was the difference between open-circuit rod mill grinding for the bench tests and the closed-circuit ball mill circuit for the pilot scale. The average 9-minute rod mill grind and Run 4 flotation feed is plotted in Figure 47. The figure shows that the bench test products were slightly finer than the Run 4 flotation feed sizing. Another explanation was that the residence time was reduced too much to achieve similar selectivity as the bench test; the residence time for the pilot flotation circuit is explored in greater detail for Run 6, below.

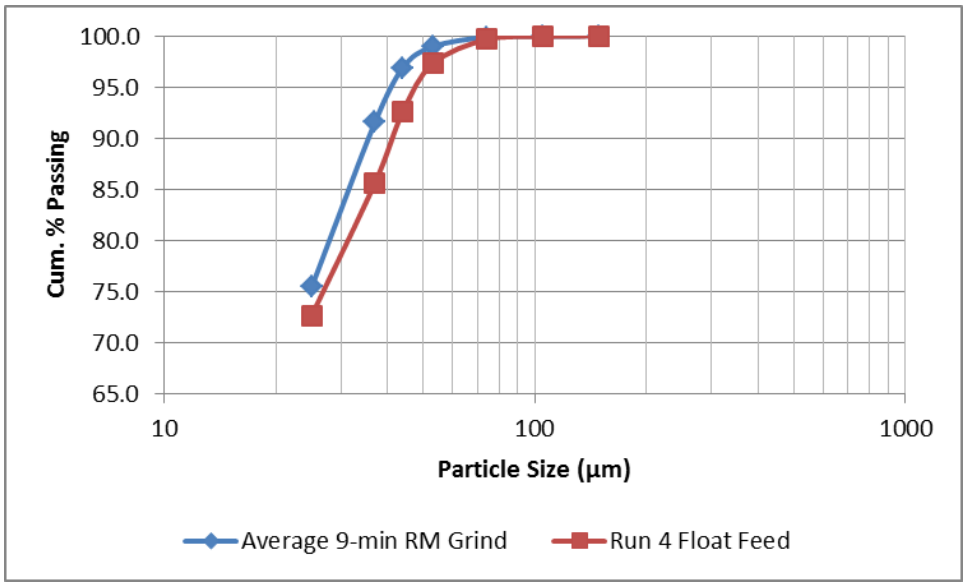


Figure 47: 9-Minute Rod Mill Grind vs. Run 4 Pilot Flotation Feed

Following Run 4, the amine dosage in the first three cells was decreased in an attempt to increase selectivity in the rougher bank; total collector dosage also decreased from 0.20 kg/metric ton (0.45 lb./LT) to 0.16 kg/metric ton (0.35 lb./LT). The composite cleaner flotation underflow (concentrate) produced from Run 5 was a 6.1% SiO_2 at a weight and iron recovery of 33.4% and 47.8%, respectively. During Run 5, a sample of the cleaner underflow (concentrate) was taken for a re-cleaner NatFlot bench test and a sample of the rougher and cleaner flotation bank (total tails) was taken for a scavenger NatFlot bench test.[23] It was found during the re-cleaner bench test that an additional 2.8% weight could be floated without additional collector or starch. The re-cleaner NatFlot bench test resulted in the pilot cleaner concentrate silica grade being reduced from 4.0% to 3.3% SiO_2 by floating for an additional two minutes while recovering 98% of the total iron. It was also shown that the concentrate silica could be reduced as low as 2.0% SiO_2 by adding an additional 0.14 kg/metric ton (0.30 lb./LT) amine and floating for an additional eight minutes, while recovering 74.4% of the total iron. This bench test found that reducing the amine and froth control spray water volume resulted in concentrate silica values that more closely matched the bench test results.

A scavenger NatFlot test was also conducted with the Run 5 total tails sample. A total of 0.22 kg/metric ton (0.50 lb./LT) GPC causticized modified starch and multiple stages of 44.6 g/metric ton (0.1 lb./LT) of Clariant Flotigam EDA were used for the test. The scavenger bench test data is summarized in Figure 48. The scavenger NatFlot bench test found the ability to produce a

[23] Bench testing was conducted the same day as the pilot run.

concentrate product with a silica content as low as 2.6% SiO_2 by floating with an additional 0.22 kg/metric ton (0.50 lb./LT) of collector for 18 minutes. When interpolating the test data to a concentrate grade of 4.0% SiO_2, the scavenger bench test found that an additional weight and iron recovery of 23.0% and 43.3%[24], respectively, could be achieved by floating for 15-minutes and 0.16 kg/metric ton (0.36 lb./LT) of collector. Run 5 bench test data can be found in Appendix 44. The finding that iron scavenging could successfully recover a significant amount of weight at target silica grade is an important consideration for modifying the process with the objective of increasing the weight recovery in future process development.

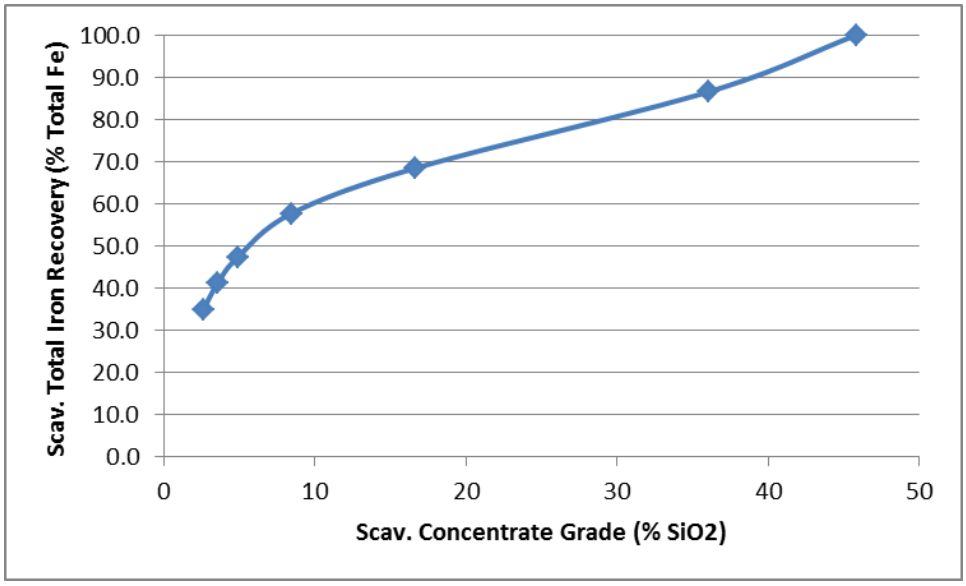


Figure 48: Run 5 Scavenger Silica Grade vs. Total Iron Recovery

While the first five runs allowed for problem solving and optimization of the circuit, Run 6 was the most successful run demonstrating the potential of this technology. The composite sample achieved a cleaner flotation underflow grade of 4.2% SiO_2 at a relatively good weight recovery compared to the previous runs. This pilot run will be discussed in detail in the following sections and the products submitted for detailed characterization.

Pilot Run 6 Rougher and Cleaner Flotation Pilot Testing Summary

The sixth pilot test was conducted using the refinements learned from the previous five tests. After confirming that the froth control spray bars reduced residence time in the flotation circuit, a slightly lower feed rate was targeted (0.22 mt/h instead of 0.24 mt/h) and the volume of water to the froth control spray bars were minimized as much as possible. The froth control spray nozzle in the rougher feed box was mostly off and the froth control spray nozzle in the rougher discharge box was manually pulsed when the froth level reached the top of the box. The froth control spray bar in the feed and discharge box in the cleaner flotation bank was continuously operated but at a low flow. Another process change in Run 6 was the screen undersize density target was raised from 30% solids to 35% to increase the flotation residence time.

[24] Recovery values based on scavenger feed.

The process operated consistently for over four hours before the rougher flotation dart valve discharge port plugged causing a short process disruption. Total run time during this test was approximately seven hours. The composite cleaner flotation underflow (concentrate) for the run was a 61.4% total iron and 4.2% SiO_2, with a weight and iron recovery of 27.9% and 40.9%, respectively. The 80% passing size ("P80") for flotation feed, underflow (concentrate), and total froth (tailings) were measured to be 31 µm, 45 µm, and 26 µm, respectively. The particle size distributions of the seven products streams can be seen in Figure 49.

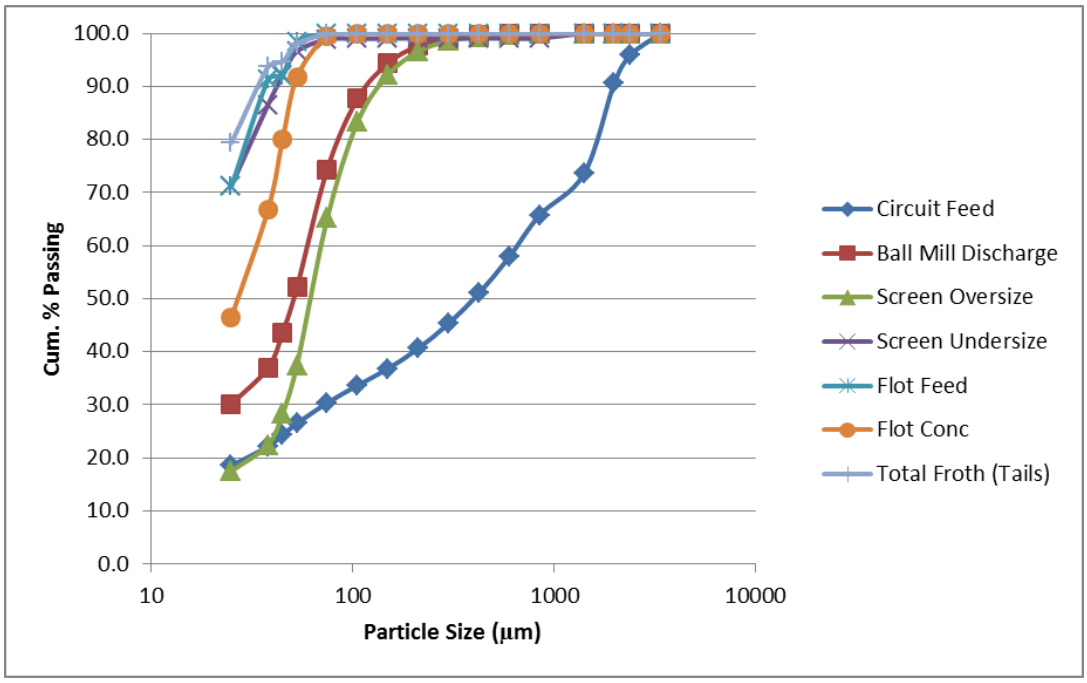


Figure 49: Run 6 Particle Size Distribution of Pilot Products

The measurements taken during Run 6 and estimates were used to produce an ore and water balance for Run 6. It was estimated that the total energy and water consumption for Run 6 was 25.6 kWh/metric ton (26.0 kWh/LT) and 3,726 L/metric ton (1,000 gallon/LT), respectively. The circulating load ratio and Bond Work Index efficiency were 360% and 101%, respectively. Flotation feed density, temperature, and pH were measured to be 35.5%, 22°C (72°F), and 8.37, respectively. The estimated starch and collector dosages were 0.65 kg/metric ton (1.46 lb./LT) and 0.16 kg/metric ton (0.36 lb./LT), respectively. Using the measurements and assumptions found in Appendix 45, the estimated flotation residence time for the rougher-cleaner flotation process was 50 minutes. Appendix 37 contains the Run 6 process parameter targets and field measurements, and Appendix 45 contains the estimated ore and water balance for Run 6.

Four slurry samples were taken in the afternoon for analysis of the calcium and magnesium ions in the water. The data is summarized in Table 49. The highest total hardness ions (e.g., Ca^{++}, Mg^{++}) were found in the process water and total tailings streams, with lesser ions in the flotation feed and concentrate streams. Similar results were found in Run 4 and Run 5, as well. The total froth stream having the total hardness levels most similar to the process water seems logical as this stream contained the most tap water due to the froth spray bars. The flotation feed had the lowest total hardness of the three samples.

Table 49: Run 6 Flotation Circuit Calcium and Magnesium Ion Levels

Stream	Ca^{++} (ppm)	Mg^{++} (ppm)	Total Hardness (ppm, as $CaCO_3$)
Process Water	115.5	36.8	440
Flot Feed	79.7	16.7	268
Cl. Flot U/F (Con)	85.6	28.3	330
Total Froth (Tails)	110.4	31.1	404

Pilot Run 6 Re-Cleaner and Scavenger Flotation Bench Testing Summary

During Run 6, a sample of the cleaner underflow (concentrate) was taken for a re-cleaner NatFlot bench test and a sample of the rougher and cleaner flotation bank (total tails) was also taken for a scavenger NatFlot bench test.[25] It was found during the re-cleaner bench test that an additional 1.2% weight could be floated without additional collector. The re-cleaner NatFlot bench test found that the pilot cleaner concentrate silica grade was reduced from 4.0% to 3.8% SiO_2 by floating for an additional 1.5 minutes while recovering 99% of the total iron. It was also found that the concentrate silica could be reduced as low as 1.6% SiO_2 by adding an additional 0.13 kg/metric ton (0.30 lb./LT) amine and floating for an additional eight minutes, while recovering 52% of the total iron.

A scavenger NatFlot test was conducted with the Run 6 total tails sample. A total of 0.22 kg/metric ton (0.50 lb./LT) GPC causticized modified starch and multiple stages of 44.6 g/metric ton (0.1 lb./LT) of Clariant Flotigam EDA were used for the test. The scavenger bench test data is summarized in Figure 50. The scavenger NatFlot bench test resulted in a concentrate product with a silica content as low as 2.1% SiO_2 by floating with an additional 0.22 kg/metric ton (0.50 lb./LT) of collector for 22 minutes. The final bench scavenger concentrate was found to have a particle sizing of P80 25 µm. The discovery of a much finer scavenger concentrate compared to the cleaner concentrate suggests fine iron losses to the froth fraction in the pilot circuit. Interpolating the test data to a 4.0% SiO_2 concentrate grade, the scavenger bench test resulted in an additional weight and iron recovery of 28.0% and 49.5%, respectively, could be achieved by floating for 17-minutes and 0.13 kg/metric ton (0.28 lb./LT) of collector. Run 6 bench test data can be found in Appendix 46. The finding that iron scavenging could successfully recover a significant amount of weight at target silica grade was important for increasing the weight recovery and for future process development.

[25] The samples taken for bench testing were kept as a slurry and the bench tests ran five days after the day of the pilot run.

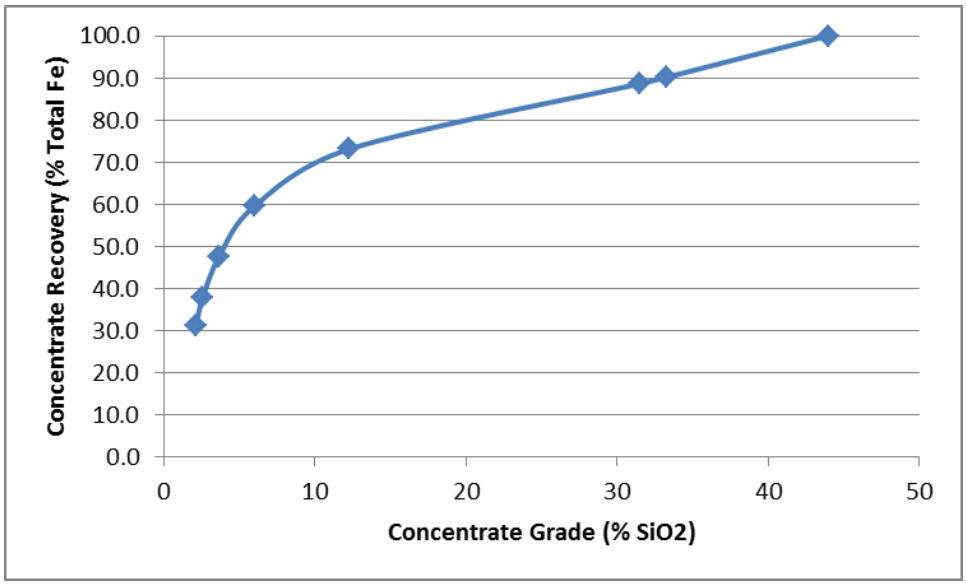


Figure 50: Run 6 Scavenger Silica Grade vs. Total Iron Recovery

The pilot cleaner underflow (concentrate) and bench scavenger data was combined to determine the estimated grade and recovery of the cleaner and scavenger concentrate, as shown in Figure 51. Run 6 combined pilot and bench test data can be found in Appendix 47. Assuming the cleaner flotation concentrate produced in the pilot circuit was combined with the bench scavenger concentrate to produce a composite 4.0% SiO_2 concentrate, the estimated weight and iron recovery would be 47.5% and 69.4%, respectively (see Table 50). If a 4.5% SiO_2 concentrate product was targeted, the estimated weight and iron recovery would be 50.1% and 72.7%, respectively. The composite phosphorus grade and 80% passing particle size ("P80") was estimated at 0.033% P and 35 µm, respectively. The effect of scavenging was an increase in weight and iron recovery of approximately 70%, while making a finer composite concentrate (estimated P80 42 µm cleaner flotation concentrate vs. a P80 35 µm for composite concentrate). Furthermore, scavenging may also be a method for final concentrate silica grade control.

Figure 51 also shows that the pilot cleaner concentrate with bench scavenger concentrate had slightly higher recovery than the bench baseline test. Interpolating the test data to a 4.0% SiO_2 concentrate grade, the pilot NatFlot cleaner and scavenger concentrate had approximately 1.4% and 2.6% higher weight and iron recovery than the optimized NatFlot bench test, respectively. However, the piloting and bench test scavenger NatFlot tests also resulted in a total collector dosage approximately double the baseline and over four times the total flotation time as found in the optimized bench test.

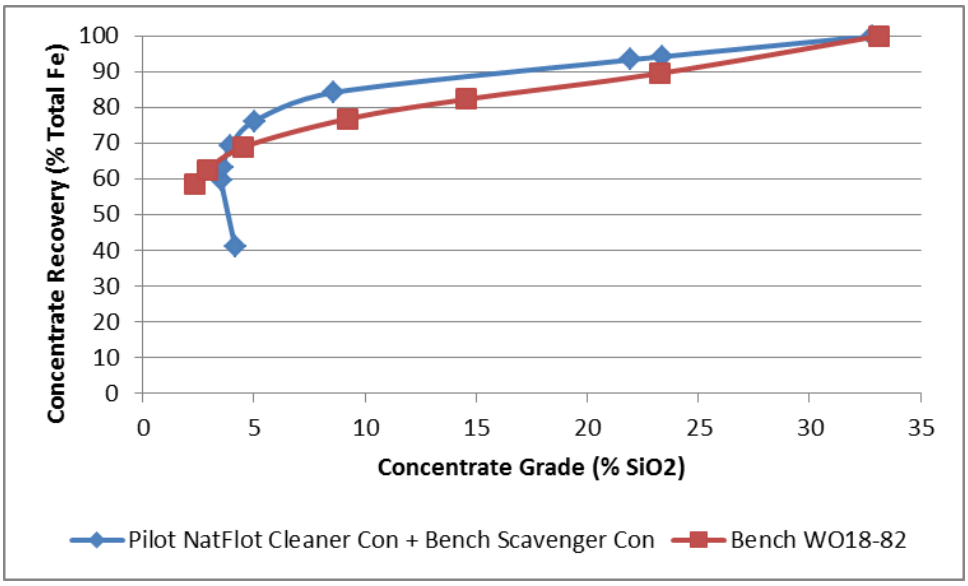


Figure 51: Run 6 NatFlot Pilot Cleaner Concentrate + NatFlot Scavenger Product and Bench NatFlot Test 18-82 Bench Silica Grade vs. Total Iron Recovery

Table 50: Summary of Pilot NatFlot Cleaner Concentrate, Bench NatFlot Scavenger Concentrate, and Optimized NatFlot Baseline Concentrate

Product	Weight Recovery (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Recovery (%)	Amine (lb./LT)	Time (min)
Pilot NatFlot Cl. Con	27.9	61.40	4.15	0.032	40.9	0.35	50.3
Bench NatFlot Scav. Con	17.2	63.32	2.11	0.030	31.3	0.50	21.5
Pilot NatFlot Cl. Con + Bench NatFlot Scav. Con Interpolated to 4.0% SiO_2	47.5	61.54	4.00	0.033	69.4	0.64	67.6[26]
Bench Optimized NatFlot Test 18-82 Interpolated to 4.0% SiO_2	46.0	61.29	4.00	0.034	66.8	0.28	15.3

Characterization of Pilot-Scale Beneficiation Process Products and Effluent

The products from Run 6 were further characterized via XRD with Rietveld refinement and chemical assay for a more detailed understanding of the product specifications and to help guide the design of the pyrometallurgical testwork. The tailings thickener overflow water was analyzed for basic characterization to be used for baseline water quality data.

[26] Total flotation time was calculated as the estimated Run 6 flotation residence time plus the bench test time required to produce a composite 4.0% SiO_2 concentrate product.

Pilot Run 6 Concentrate and Tailings Characterization

The Run 6 cleaner flotation underflow (concentrate) sample was composed of 65.8% goethite, 30.2% hematite, 2.9% quartz, and 1.1% calcite. The Run 6 cleaner flotation underflow (concentrate) chemical assay had an estimated total iron, silica, and phosphorus grade of 61.4% Total Fe, 4.2% SiO_2, and 0.032% phosphorus, respectively. The hematite-to-goethite ratio was 1.0:2.2, compared to 1.0:2.4 for the head analysis. This shows a slightly higher concentration of goethite compared to hematite in the NatFlot rougher-cleaner pilot-scale flotation process.

The Run 6 cleaner flotation froth (tailings) sample was composed of 45.9% goethite, 17.3% hematite, and 36.8% quartz. The Run 6 cleaner flotation froth (tailings) chemical assay had an estimated total iron, silica, and phosphorus grade of 34.3% Total Fe, 45.2% SiO_2, and 0.024% phosphorus, respectively. The pilot-scale Run 6 cleaner flotation underflow and froth samples were also submitted to Activation Laboratories for whole rock analysis (WRA). The WRA analysis for major oxides can be seen in Table 51 and the full WRA data for Run 6 cleaner flotation underflow and froth can be found in Appendix 48. The Al_2O_3 and Na_2O are likely due to feldspars (microcline and albite) that were in the sandy till that contaminated the iron formation samples. Most of the feldspars reported to the tails.

Table 51: Whole Rock Analysis for Major Oxides for Run 6 Flotation Cleaner Concentrate and Tailings

Analyte	SiO_2	Al_2O_3	Fe_2O_3	MnO	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	LOI	Total
Unit Symbol	%	%	%	%	%	%	%	%	%	%	%	%
Detection Limit	0.01	0.01	0.01	0.001	0.01	0.01	0.01	0.01	0.001	0.01		0.01
RUN 6 7-20-18 FLOT U/F (CON)	4.28	0.46	85.63	0.082	0.16	0.53	0.08	0.04	0.05	0.09	7.01	99.41
RUN 6 7-20-18 FLOT FROTH (TAILS)	46.57	1.71	43.84	0.064	0.38	0.64	0.38	0.22	0.081	0.06	4.17	99.22

The chemical assay of the Run 6 flotation concentrate and tailings conducted at NRRI Coleraine and Activation Laboratories can be seen in Table 52, below.

Table 52: Pilot-Scale Run 6 Cleaner Flotation Underflow and Froth Chemical Assay

Sample ID	Total Fe (%)	Fe^{++} (%)	Sat Mag Fe (%)	SiO_2 (%)	Phos (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	LOI (%)
Run 6 Cl. U/F (NRRI)	61.40	N/A	N/A	4.15	0.032	N/A	N/A	N/A	N/A
Run 6 Cl. U/F (WRA)	60.60	0.9	N/A	4.28	0.04	0.46	0.53	0.16	7.01
Run 6 Froth (Tails) (NRRI)	34.27	N/A	N/A	45.15	0.024	N/A	N/A	N/A	N/A
Run 6 Froth (Tails) (WRA)	31.44	1.0	N/A	46.57	0.03	1.71	0.64	0.38	4.17

Pilot Water Characterization

During pilot Run 2 through Run 6, Pace Analytical obtained a composite sample of the tailings thickener overflow to analyze for: low-level mercury (raw sample and dissolved in water),

amine, anions, alkalinity, chemical oxygen demand (COD), biochemical oxygen demand (BOD), total suspended solids (TSS), total Kjeldahl nitrogen (TKN), nitrate, total nitrogen, total phosphorus, pH, specific conductance, metals (raw sample and dissolved in water), heavy metals (raw sample and dissolved in water), diesel range organics (WIDRO), and naphthalene. The tailings thickener overflow sample was taken at Stream #24 in Figure 52, below. Pace Analytical technicians obtained four cuts over the course of one hour to form the composite sample from each pilot run for analysis. Pace Analytical also obtained one process water sample, which is Coleraine tap water, before Run 2 on June 20, 2018, for a baseline water characterization. The process water sample was taken at Stream #3 in Figure 52, below.

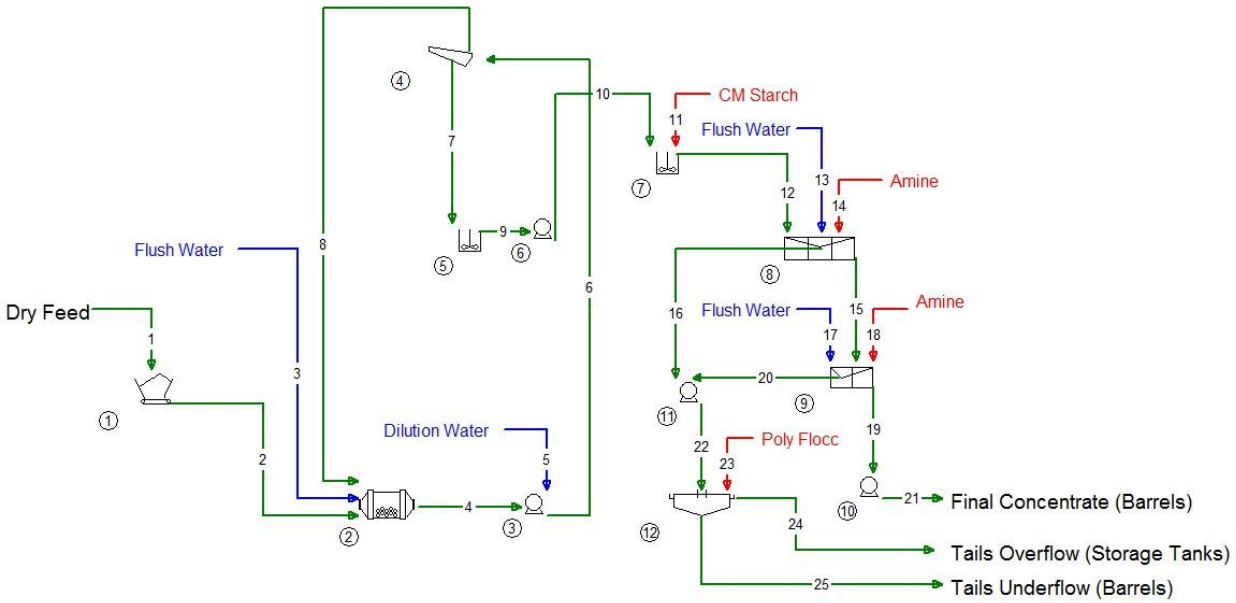


Figure 52: Continuous Pilot-Scale Process Flowsheet

The objective of the baseline water quality characterization of tailings thickener overflow was to understand what components could be a potential concern for process water recycle and future water treatment. The data was also used as effluent characterization for disposal at the local wastewater treatment plant. Of particular interest in this study was the concentration of mercury, amine, major ions (e.g., alkalinity, sulfate, and chloride), and heavy metals. The full process water and thickener tailings overflow data set can be found in Appendix 49. It is important to note that the pilot-scale testing was not conducted to optimize the performance of the tailings thickener or control of the tailings thickener. Note also that polymer flocculant addition was based upon visual clarity observation of the thickener overflow. Furthermore, the feedwell in the tailings thickener was not present during this testing program so contact between the polymer flocculant and slurry may have been impacted during pilot operation.

The average of the pilot runs and the individual Run 6 results were explored further, especially in relation to the mercury, amine, anions, and heavy metal analyses. In summary, pilot Run 2 through Run 6 had an average of 2.8 ng/L mercury in the tailings thickener overflow cut samples, compared to 0.6 ng/L in the process water. Run 6 had a low-level mercury of 1.8 ng/L. In terms of dissolved mercury, the process water and Run 2, Run 5, and Run 6 had no detectable low-level mercury readings with a 0.50 ng/L reporting limit. Run 3 and Run 4 had a dissolved low-level mercury value of 0.59 ng/L and 0.56 ng/L, respectively.

Pilot Run 2 through Run 6 had an average of 1.8 mg/L amine in the tailings thickener overflow, with Run 6 also measuring 1.8 mg/L residual amine. Run 6 had an estimated amine concentration of 33.1 mg/L into the process and a measured 1.8 mg/L residual amine remaining in the tailings thickener overflow water, which equated to an approximated removal rate of 94.6%. This estimated amine removal rate assumes the water in the tailings thickener underflow (tailings solids) and cleaner flotation underflow (concentrate) filtrate had similar residual amine concentration. As a comparison, the single bench test discussed in the *Residual Amine Testing* section had an amine concentration of 27.8 mg/L into the test and 1.1 mg/L residual amine remaining in the recycle water[27], which equated to an estimated amine removal rate of approximately 96.0%. The amine removal rate was approximately 1.5% lower in Run 6 than in the bench test, which may be at least partially due to the use of the froth control spray bars that reduced the residence time in the flotation circuit. Appendix 50 contains the amine removal rate calculations and ore and water balance for Run 6.

The raw process water had 22.7 mg/L and 118 mg/L chloride and sulfate, respectively. Pilot Run 2 through Run 6 had an average of 63.7 mg/L chloride and 122 mg/L sulfate in the tailings thickener overflow, with Run 6 measuring 75.5 mg/L chloride and 122 mg/L sulfate. Based upon these average measurements, the pilot-scale NatFlot testing with the Bulk 19-2 sample resulted in an increase in chlorides in the tailings thickener overflow, with a very slight increase in sulfate. Total sulfur measured in the Bulk 19-2 sample head solids was 0.009%. Total sulfur in the Run 6 underflow (concentrate) and froth (tailings) solids were 0.009% and 0.011%, respectively.

In terms of heavy metals, the testing of the tailings thickener overflow water detected the presence of arsenic, chromium, and lead. On average for Run 2 through Run 6, the testing found arsenic, chromium, and lead levels of 1.4 µg/L[28], 2.7 µg/L, and 0.70 µg/L[29], respectively. For Run 6, the testing found arsenic, chromium, and lead levels of 1.7 µg/L, 4.5 µg/L, and 0.65 µg/L, respectively. No dissolved heavy metals were detected throughout the testing. Arsenic, chromium, and lead in the Bulk 19-2 sample head solids were measured to be 10.9 ppm, 40 ppm, and 14 ppm, respectively. Arsenic, chromium, and lead in the Run 6 underflow (concentrate) solids were 13.4 ppm, 25 ppm, and 6 ppm, respectively. Arsenic, chromium, and lead in the Run 6 froth (tailings) solids were 11.2 ppm, 25 ppm, and < 5 ppm, respectively.

Of the other metals measured[30], the largest concentration changes compared to the baseline process water were found with iron, manganese, and sodium. The baseline process water had a concentration of 674 µg/L iron, 11.4 µg/L manganese, and 15.0 mg/L sodium. The average of Run 2 through Run 6 was 4,170 µg/L iron, 108.6 µg/L manganese, and 7.4 mg/L sodium. The Run 6 measurements were 6,050 µg/L iron, 126 µg/L manganese, and 51.8 mg/L sodium.

Another area of interest was in the chemical oxygen demand (COD) and biochemical oxygen demand (BOD5) analyses. On average for Run 2 through Run 6, the testing had a COD level of

[27] Combined froth (tailings) and underflow (concentrate) filtrate water.
[28] Arsenic was not detected in the Run 2 tailings thickener overflow sample with 1.0 µg/L detection limit.
[29] Lead was not detected in the Run 3 tailings thickener overflow sample with 0.5 µg/L detection limit.
[30] Calcium, copper, iron, magnesium, manganese, nickel, potassium, sodium, and zinc.

61.9 mg/L. In terms of BOD, there was no detection during Run 2 and Run 3, and an average of 58.6 mg/L for Run 4 through Run 6. For Run 6, the COD and BOD levels were 70.6 mg/L and 14.0 mg/L, respectively. The levels for both COD and BOD were low compared to typical treated wastewater. Generally, the COD analyses were higher than the BOD analyses, suggesting some chemicals are less, or not, biodegradable, but can be oxidized with strong chemical oxidants. It is hypothesized that the majority of the chemicals were adsorbed to the mineral surfaces and removed via the tailings thickener, with the COD and BOD detection a result of the residual amine and starch in the water.

In discussion with a local wastewater treatment plant[31], it was explained that their permit requires monitoring total and dissolved mercury, dissolved oxygen, total phosphorus, total Kjeldahl nitrogen (TKN), nitrate and nitrite (NO_2+NO_3), total dissolved solids (TDS), and specific conductance in their effluent discharge for reporting purposes. Their effluent discharge limits include: 8.0 mg/L ammonia nitrogen, pH of 6-9, carbonaceous biochemical oxygen demand (CBOD or BOD5) of 25 mg/L (monthly average), total suspended solids (TSS) of 30 mg/L (monthly average), total zinc of 418 µg/L (maximum do not exceed), and fecal coliform of 200 colony-forming unit (CFU) per 100 mL (J. Frost, personal communication, 9 August, 2018).

In practice, the tailings thickener overflow water would report back to the concentrator as makeup water in the process. Any water released to the environment from a processing operation would go through an additional water treatment plant before release to the environment. However, as a simple baseline comparison, the water characterization and local wastewater permit requirements were compared. When comparing these treated water limits to the tailings thickener overflow water as a baseline, the pilot-scale testing values for ammonia nitrogen, pH, BOD, and total zinc were found to be within the local wastewater treatment plant permit limits, but the TSS values were above the limit. The average TSS in Run 2 through Run 6 was 124.5 mg/L, with Run 6 having a TSS of 63.2 mg/L. The elevated TSS value was likely due to the operation of the thickener instead of material properties.

A comparison between the process water influent (e.g. Coleraine tap water), process effluent (e.g., tailings thickener overflow water), and Minnesota's Freshwater Quality standards for Class 2A (most sensitive) waters can be found in Appendix 51. Appendix 51 also contains Minnesota Pollution Control Agency (MPCA) water quality standard summary tables. The data was broken into four categories, including: 1) dissolved heavy metals, 2) water quality parameters, 3) dissolved major ions, iron, and manganese, and 4) nutrients. To calculate the means and standard values for comparison, any values below the reporting limits were assigned values of one-half of the reporting limit based upon "Method 4" of EPA guidelines on handling chemical concentration data near the detection limit (EPA, 2017). Some preliminary observations by NRRI water scientists was that for the dissolved heavy metals, all heavy metal concentrations in the NatFlot process effluent were below detection at levels below chronic toxicity standards, except for mercury. Both total and dissolved mercury concentrations, though measureable, were below chronic toxicity standards.

[31] The Grand Rapids Public Utilities Commission (GRPUC) wastewater treatment Plant treats a total of approximately 7.5 mgpd water, 6.0 mgpd of which is industrial wastewater.

For the water quality parameters, the nitrogen amine values were above detection in the NatFlot process effluent and were a result of dosing the system with flotation chemicals. At the time of this report, no appropriate water quality standards for comparison and guidance were found for this parameter. Next, COD was below detection at 25 mg/L in the process influent and increased to 62 ± 18 mg/L in the effluent. Similarly, BOD5[32] increased from below detection (2.4 mg/L) to 12.2 ± 9.6 mg/L in the process effluent. This value is below the traditional wastewater treatment plant (WWTP) value of 25 mg/L which is typically used for WWTP NPDES permits. The total suspended sediment (TSS) levels increased from approximately 11 mg/L in the influent to >100 mg/L in the process effluent, well above the water quality standard of 10 mg/L (to be exceeded <10% of the time) for northern Minnesota rivers. WWTPs typically must generate an effluent with < 30 mg TSS/L. The process effluent pH compared to influent water pH was approximately the same value. Specific electrical conductivity was relatively high (762 µS/cm) in influent water and increased approximately 11% in the effluent. Diesel Range Organic levels (WDRO) were well above detection in the process effluent and non-detectable in process influent. The measured concentrations of approximately 2 mg/L exceeded the MPCA Class 2a standard for "oil" of 0.5 mg/L. No naphthalene, an indicator of PAH, was detected, at a detection limit (1.0 µg/L) well below the standard for human health (65 µg/L).

For the dissolved major ions, iron, and manganese values, the NatFlot process increased chloride concentrations in the influent from approximately 23 to 64 ± 10 mg/L, well below levels chronically toxic to trout (230 mg/L), but much higher than the Northern Lakes & Forests Ecoregion range of 0.6–1.2 mg/L[33]. The sulfate concentrations were relatively unaffected by the process and were an order of magnitude higher than the SO_4 standard of 10 mg/L applied to Wild Rice waters, but below the Secondary Drinking Water standard of 250 mg/L based on taste and potential laxative effects[34]. Alkalinity values were indicative of moderately hard influent water (262 mg $CaCO_3$/L); the process actually reduced alkalinity by approximately 17%. Base metal cations calcium and magnesium were relatively unchanged by the process, but potassium and sodium increased substantially, although there are no water quality standards for these constituents. Dissolved iron was below detection in the influent and effluent; dissolved manganese increased from <10 µg/L to 53 ± 34 µg/L in the effluent. At these levels, there are no toxicity concerns. However, there are Secondary Drinking Water standards for each based on potential staining.

For the nutrient values, no significant "red flags" are associated with the nutrient content of the process effluent in terms of potential for the effluent to affect the primary productivity of a receiving water body. The influent total phosphorus was unusually high (0.64 mg P/L), presumably due to the City of Coleraine adding polyphosphate in the filter bed in the water treatment process (H. Bertram, personal communication, 27 September, 2018). Most, if not all, of this phosphorus was removed by the NatFlot process since effluent concentrations were below detection (i.e. < 0.14 mg P/L).

[32] BOD 5-day test.
[33] See MPCA summary tables in Appendix 51 for more information.
[34] See MN Dept. of Health website for more information:
http://www.health.state.mn.us/divs/eh/risk/guidance/waterguidance.html

Characterization of Pelletizing Feed Composite Sample

Approximately 296 kg (652 lbs.) of Run 5 cleaner flotation concentrate and 373 kg (822 lbs.) of Run 6 cleaner flotation concentrate were blended together to form a composite sample for bench- and pilot-scale pelletizing and induration testing. The composite concentrate sample was composed of 67.3% goethite, 29.9% hematite, and 2.8% quartz. The composite concentrate sample had an estimated total iron, silica, and phosphorus grade of 61.3% Total Fe, 4.4% SiO_2, and 0.03% phosphorus, respectively. The WRA analysis for major oxides can be seen in Table 53 and the full WRA data for the composite pelletizing feed sample can be found in Appendix 52.

Table 53: Whole Rock Analysis for Major Oxides for Pelletizing and Induration Testing

Analyte	SiO_2	Al_2O_3	Fe_2O_3	MnO	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	LOI	Total
Unit Symbol	%	%	%	%	%	%	%	%	%	%	%	%
Detection Limit	0.01	0.01	0.01	0.001	0.01	0.01	0.01	0.01	0.001	0.01		0.01
Composite Filter Cake for Pelletizing	4.49	0.52	86.12	0.081	0.17	0.55	0.09	0.05	0.056	0.07	7.09	100.2

The chemical assay of the composite pelletizing feed sample conducted at NRRI Coleraine and Activation Laboratories can be seen in Table 54, below.

Table 54: Pelletizing Feed Composite Sample Chemical Assay

Sample ID	Total Fe (%)	Fe^{++} (%)	Sat Mag Fe (%)	SiO_2 (%)	Phos (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	LOI (%)
Pellet Feed (NRRI)	61.30	0.64	0.15	4.36	0.033	0.54	0.40	0.13	7.07
Pellet Feed (WRA)	60.86	0.8	N/A	4.49	0.03	0.52	0.55	0.17	7.09

The composite sample particle size was a P80 of 42 µm and a Blaine of 1,715 cm^2/g. The particle size distribution can be seen in Figure 53, below.

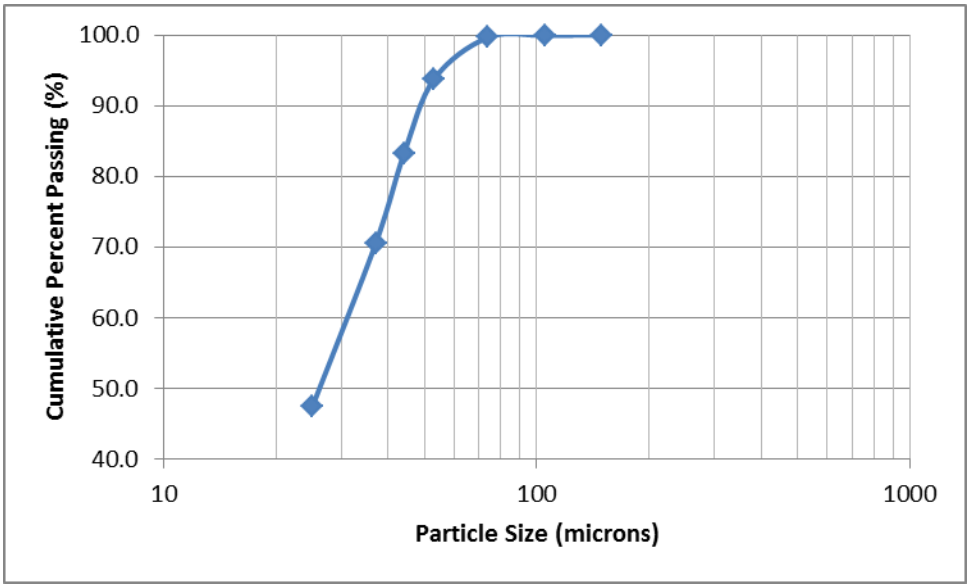


Figure 53: Pelletizing Feed Composite Sample Particle Size Distribution

Pyrometallurgical Testing Introduction

A pelletization and induration study was carried out using the iron concentrate produced from the pilot-scale NatFlot study using the Bulk 19-2 material. Characterization of this feed material can be seen in the *Characterization of Pelletizing Feed Composite Sample* section, above. The objective of the pelletization and induration studies was to assess if a blast furnace-grade pellet[35] could be produced using the concentrate produced during the pilot-scale testing. The scope included conducting up to four mini-pot and two pot grate tests.

A series of bench-scale batch balling/mini-pot and pot grate studies were conducted to determine if the iron concentrate produced from the pilot-scale NatFlot testing with the Bulk 19-2 material could be used to produce a blast furnace-grade pellet. Batch balling and mini-pot studies were used to develop a firing cycle that could produce a pellet which meets the physical requirements of a blast furnace pellet. The developed mini-pot firing cycle was then translated to a pot grate cycle to simulate a possible commercial furnace cycle. Pellets produced from pot grate experiments were then analyzed to determine if they meet physical and metallurgical pellet standards. Standard industry metrics were used to assess the pellet quality at each stage of testing.

Batch Balling and Mini-Pot Test Plan

Batch balling and mini-pot experiments were run with the objective of developing a firing cycle that could produce pellets that meet physical specifications required of blast furnace pellets. Mini-pot experiments are not representative of the conditions a pellet would see in a commercial furnace. Because of this, mini-pot experiments are used exclusively to determine if a proposed firing cycle and pellet blend can produce fired pellets that meet the physical constraints of blast

[35] Generally defined as an oxide pellet with 4.0-5.5% SiO_2 and industry-acceptable metallurgical properties. Acceptable metallurgical properties are described in future sections.

furnace pellets. Therefore, a larger test volume is required to be fired to conduct the metallurgical tests; this is simulated using a pot grate test.

For the batch balling and mini-pot work, a blend of 6.7 kg/metric ton (15 lb./LT) bentonite and 1.5% coke breeze was mixed with the concentrate. Bentonite was added as a binding agent and coke breeze was added to compensate for the lower heat of reaction for the hematite and goethite during induration compared to a typical Minnesota taconite concentrate. Green ball quality (unfired pellet) was evaluated using:

- dry compressive strength,
- green ball moisture,
- 45.7 cm (18-inch) wet drop number, and
- size analysis.

Mini-pot experiments used 1,600 grams of green balls that were pelletized using a balling tire. Post mini-pot fired pellet quality was measured using the compressive strength and the tumble test to measure abrasion.

Pot Grate Test Plan

Pot grate experiments simulate what a green ball would see if it were traveling through a commercial scale furnace. This results in pellets that are representative of commercial pellets, with respect to physical and metallurgical properties. Thus, the purpose of the pot grate studies was to examine if a pellet could be produced that meets the physical and metallurgical specifications for blast furnace grade pellets. A total of two pot grate tests were scoped and completed for this current research project.

For the pot grate simulations, 32.7 kg (72 lbs.) of green balls were prepared in a pelletizing disc. A 15.2 cm (6 inch) pellet bed depth and 5.1 cm (2 inch) hearth layer was used. The feed was composed of equal weights -12.7 mm to +11.1 mm (-1/2 inch to +7/16 inch) and -11.1 mm to +9.5 mm (-7/16 inch to +3/8 inch) green balls. The blend consisted of 6.7 kg/metric ton (15 lb./LT) bentonite and 1.2-1.5% coke breeze.

The physical quality of the pellet was measured in the same manner as with the batch balling and mini-pot studies. Metallurgical quality was examined by running an ISO 13930 – dynamic test for low temperature disintegration (LTD), ISO 4695 – determination of reducibility (R40), and a swelling test. Porosity of the pellet was also measured using a water displacement method. Chemically, the pellet was analyzed for total iron (Fe_{tot}), ferrous iron (Fe^{++}), silica (SiO_2), and phosphorus (Phos).

Bench-Scale Pyrometallurgical Testing Summary

The mini-pot study had two objectives:

1. Produce pellets that meet the physical requirements, green and fired, of blast furnace grade pellets, and
2. Develop a firing cycle that could be used to produce blast furnace grade pellets using the iron concentrate described above.

Green ball quality for the pellets produced for the mini-pot trials can be seen in Table 55. Two trials, M18108 and M18109, were conducted for repeatability.

Table 55: Green Ball Quality of Mini-Pot Pellets

				Size Analysis			
Test	**Dry Compressive Strength (lbf)**	**Green Ball Moisture (%)**	**Wet Drop No.**	**% +1/2"**	**% +7/16"**	**% +3/8"**	**% -3/8"**
M18108	14.7	12.4	4.8	33.92	40.85	21.20	4.03
M18109	13.8	12.5	4.8	16.52	43.06	35.56	4.86

Note: Concentrate Moisture = 11.8%; 1.5% Coke Breeze of Dry Concentrate Weight; 15 lb./LT Bentonite of Dry Concentrate Weight; Coleraine, MN Water; 18 inch drop height.

For green ball quality, it is traditionally preferred that the dry compressive strength and wet drop number be above 17.8 N (4 lbf) and four drops. Both mini-pot results shown in Table 55 exceed the physical requirements for dry compressive strength and drop number. The green ball moisture for both trials is higher than the typical 9-10% found with Minnesota magnetite processing. It is likely that the increased green ball moisture is due to the large fraction the goethite present. Goethite tends to be in the form of fine particles which can trap moisture. The only noticeable difference between the green ball quality of the two trials was the green ball growth rate, or size analysis. This was a notable difference, however, does not impact mini-pot trials because the mini-pot feed was composed of equal weights -12.7 mm to +11.1 mm (-1/2 inch to +7/16 inch) and -11.1 mm to +9.5 mm (-7/16 inch to +3/8 inch) green balls.

Once green balls of the appropriate physical quality could be made, the next step was to develop a mini-pot firing cycle to produce fired pellets that meet the physical requirements for blast furnace feed. The baseline mini-pot firing cycle that was developed can be seen in Appendix 53. This was the second attempt at developing the mini-pot firing cycle. The first attempt resulted in fired pellets with unacceptable fired compressive strengths of approximately 1,624 N (365 lbf). The physical quality of the pellets fired using the cycle found in Appendix 53 can be seen in Table 56. A comparison of the two mini-pot temperature profiles can be found in Appendix 54.

Table 56: Mini-Pot Fired Pellet Quality

					Abrasion Index	
Test	**Compressive Strength (lb_f)**	**Standard Deviation (lb_f)**	**% Below 300 lb_f**	**% Below 200 lb_f**	**% +6.3 mm**	**%-32M**
M18108	526	99	2	0	97.7	2.2
M18109	525	86	2	0	97.5	2.5

Note: Compression strengths are the average of 100 pellets

For fired pellets, it is preferred that the average compression strength be above 2,002 N (450 lbf), with a minimal amount below the 1,334 N (300 lbf) and 890 N (200 lbf) threshold and that the abrasion index show 95% + 6.3 mm. Results in Table 56 show that both trials exceed an average compression strength 2,002 N (450 lbf) with a total of only 2% below 1,334 N (300 lbf). The abrasion index is also well above the 95% threshold for both trials.

Table 56 shows that, using the mini-pot cycle found in Appendix 53, it was possible to produce pellets to meet the physical requirements of blast furnace feed using the concentrate produced at pilot-scale with the Bulk 19-2 sample. The next step was to translate the mini-pot firing cycle to a pot grate cycle to produce pellets for metallurgical testing.

Pilot-Scale Pyrometallurgical Testing Summary

The objective of the pot grate study was to produce pellets that could meet the physical and metallurgical standards for blast furnace grade pellets using the concentrate produced during the pilot-scale NatFlot testing with the Bulk 19-2 material. The pot grate firing cycle was designed to mimic a grate-kiln furnace design, with an extended pre-heat time and increased temperature to compensate for the relatively large amount of goethite compared to typical Minnesota magnetite concentrate. Thus, a 15.2 cm (6 inch) bed depth was used with a low differential pressure during firing. The pot grate firing cycle, adapted from the mini-pot study, can be seen in Appendix 55.

Two pot grate tests were conducted varying the amount of coke breeze in the pellet. The purpose of the two tests was to determine the sensitivity of final product quality to reduction in coke breeze addition rate. The green ball quality of the pellets produced for the pot grate study can be seen in Table 57.

Table 57: Green Ball Quality for Pot Grate Study

Test	% Coke Breeze	Dry Compressive Strength (lbf)	Greenball Moisture (%)	Wet Drop No.
P18305	1.5	15.4	12.5	5.7
P18306	1.2	14.5	12.0	4.6

The green ball quality of the pellets for the pot grate tests seen in Table 57 meets the industry standards of a dry compressive strength above 17.8 N (4 lbf) and drop number above four. The green ball moisture for both trials is higher than the traditional 9-10% seen in Minnesota taconite processing. As was the case with the mini-pot green balls, it is likely that the increased moisture content is due to the large fraction the goethite present.

After it was confirmed that the green balls met the required physical specifications, they were fired in the pot grate using the cycle seen in Appendix 55. The physical quality of the fired pellets can be seen in Table 58.

Table 58: Pot Grate Fired Pellet Quality

							Abrasion Index	
Test	% Coke Breeze	Compressive Strength (lb_f)	Standard Deviation (lb_f)	% Below 300 lb_f	% Below 200 lb_f	Porosity (%)	% +6.3mm	% -32M
P18305	1.5	475.0	123.0	9.0	2.0	33.63	95.3	4.2
P18306	1.2	495.0	132.0	9.0	3.0	32.99	95.5	4.2

Both pot grate trials produced fired pellets that meet the physical specifications of blast furnace pellets. Results in Table 58 show that both trials exceed an average compression strength 2,002 N (450 lbf) with 9% below 1,334 N (300 lbf). Both trials resulted in greater than 95% plus 6.3

mm for the abrasion index. The porosity of the fired pellets was relatively high at 33.6% for P18305 and 33.0% for P18306. The Bulk 19-2 concentrate contained roughly 66% goethite. The addition of the coke breeze and the dehydroxilation of the goethite to hematite, seen in Equation 1, are the likely causes of the higher porosity compared to typical Minnesota blast furnace pellets.

$$\alpha\text{-}FeOOH_{(s)} \rightarrow \alpha\text{-}Fe_2O_{3\,(s)} + H_2O_{(g)} \qquad \textbf{Equation 1}$$

The metallurgical test results for both trials can be seen in Table 59 and the two pot grate temperature profiles can be found in Appendix 56.

Table 59: Metallurgical Test Result Summary from Pot Grate Tests

							LTD Tumble Index (%)	
Test	**Fe_{tot} (%)**	**Fe^{++} (%)**	**SiO_2 (%)**	**Phos (%)**	**Reducibility (DR/DT)**	**Swelling (%)**	**+6.3mm**	**-32M**
P18305	65.48	0.44	4.99	0.034	0.65	17.42	96.9	2.9
P18306	65.46	0.44	5.11	0.033	0.66	15.84	96.7	2.7

For pellet metallurgical results, it is preferred that the swelling be less than 20%, the LTD tumble index be 80% greater than 6.3 mm, and that the ferrous iron content be less than 1%. Test P18305 resulted in a swelling of 17.42%, a LTD tumble index of 96.9% greater than 6.3 mm, and a ferrous iron of 0.44%. Test P18306 resulted in a swelling of 15.84%, a LTD tumble index of 96.7% greater than 6.3 mm, and a ferrous iron of 0.44%. The reducibility for P18305 and P18306 was 0.65 and 0.66. Generally, reducibility values over 0.6 are considered acceptable. The total iron and silica values are in line with typical Minnesota blast furnace quality pellets. The phosphorus value of 0.03% for both tests is not uncommon for pellets made with concentrates containing a large amount of hematite and goethite, but is higher than the standard Minnesota taconite pellet. In comparison, typical iron ore oxide pellets produced with Minnesota taconite are typically below 0.02%.

In summary, the objective of the pelletization and induration studies was to assess if a blast furnace-grade pellet could be produced using the iron concentrate from the pilot-scale NatFlot testing. From the mini-pot study, it was concluded that concentrate could be used to create a fired pellet that meets the physical specifications of blast furnace pellets. The pot grate studies proved that not only could a pellet be made to meet physical specifications, but also metallurgical specifications of blast furnace pellets. Overall, it was concluded that the concentrate produced during the pilot-scale NatFlot testing using the Bulk 19-2 material could potentially be used to produce blast furnace-grade pellets.

Discussion

The motivation for this research project was to investigate methods and technologies for reducing waste with the potential for reducing costs and environmental impacts by increasing recovery of iron. This research project included acquisition of samples for bench testing, evaluation of several beneficiation techniques on the bench, evaluation of tested techniques, acquisition of bulk samples for pilot testing, and pilot testing. Due to time and cost constraints, bulk sample selection was limited to samples available in stockpiles or pit faces. The research test samples were selected and obtained with the assistance of industry geologists. The criteria for the test materials were to acquire samples that would represent some of the oxidized iron resources in Minnesota. However, it was not possible to collect a sample or assemble a bulk sample that was representative of *all* of the collective iron resource of the Western Mesabi Range without extensive research and drilling.

In the course of this research, it became apparent that there is a poor understanding of the iron ore resource that lies between historic direct shipping pits and the current taconite pits. Iron ore miners have been and are pragmatic. In the course of assessing iron ore resources, iron ore miners drill and analyze core until they encounter iron formation that does not meet their criteria for ore and then stop or greatly reduce the frequency of drilling. Hence, the historic direct shipping ore iron miners drilled until the oxidation of the iron formation decreased and the head iron fell and the silica rose. Similarly, the taconite ore iron miners drill until the oxidation increased and the weight recovery by low intensity magnetic separation dropped and the hematite in the concentrate increased. Much of the remaining iron resource consists of partially oxidized iron formation, oxidized iron formation, taconite, and mixtures of the aforementioned iron formation types. Therefore, it is currently difficult to identify the characteristics of the remaining resource and to determine a method or combination of methods that will be necessary for economic beneficiation.

The first phase of research targeted investigating the NatFlot technology with some baseline comparisons to conventional processing technologies on bench-scale. It was found during the course of the bench test program that the NatFlot process was able to reject the quartz but, as expected, also concentrated certain gangue minerals like ankerite and stilpnomelane. There was also some evidence that talc minerals interfered with the NatFlot process, as found in the testing with the Bulk 16-1 sample. The selective flocculation-deslime-flotation (SFDF) process was able to produce a significantly lower silica grade than the NatFlot process with the Bulk 16-1 sample. With the low weight rejection in the first few froths and the high amount of talc remaining in the final concentrate, one hypothesis of why the SFDF was able to reduce the silica content in the final concentrate more than the NatFlot process is that the talc is difficult to remove from the process and consuming a large amount of the collector at a natural pH versus a high-pH process. The SFDF process utilizes desliming to remove a significant amount of silica before flotation and desliming is not negatively impacted by talc. In either case, neither NatFlot nor SFDF was able to successfully treat the Bulk 16-1 sample. The testing using wet high-intensity magnetic separation (WHIMS) as a scavenger found little potential, as the concentrates contained high amount of quartz and talc. The weight recovery also dropped significantly when attempting to increase the concentrate grade. Further testing is required to determine the factors limiting the effectiveness of WHIMS.

The data from this study suggest that the NatFlot process is most effective when processing oxidized iron formation that contains varying amounts of hematite, goethite, and quartz. The processing of more mineralogically complicated partially oxidized iron formation will likely require consideration of other beneficiation technologies or a combination of technologies, possibly including NatFlot, LIMS, WHIMS, SFDF, or new technologies. The taconite and partially oxidized ores in the Minnesota iron ranges contain over thirty different gangue minerals besides quartz. Determining the zeta potential of these gangue minerals at various pH could help guide future flotation research. If the floatability of certain gangue minerals at a natural pH is not adequate, the NatFlot process would then need to be adapted by adding additional beneficiation steps and/or varying the process to reject these minerals.

Goethite has a unique morphology that results in the generation of a disproportionate amount of ultra-fine particles during grinding. The ultra-fine particles cause slurries to become "slimy". If a highly goethitic feed material, similar to the Bulk 19-2 sample, were to be processed at a production concentrator, the grinding circuit would need to be carefully designed to reduce the amount of over-grinding and slimes generated by this mineral. Some concepts that could be implemented to reduce slimes would be to use high recirculating loads in the circuit, multi-stage grinding, and screens in place of hydrocyclones. Grinding media type and sizing would also need to be carefully considered. Even with a carefully designed grinding and classification circuit, certain minerals may still pose a problem for flotation. However, the bench-scale NatFlot scavenger test with the Run 6 tailings showed potential for recovering more fine-grained goethite and hematite. The flotation process developed by Hanna Mining Company in the 1970's also found an advantage of using higher-hardness water when conducting cationic flotation at a natural pH, but relied on a low-hardness, high pH, water for desliming (Dicks and Morrow, 1978). If "slimy" ores are to be processed, this technology may be beneficial to investigate; however, the cost and operational complexity of maintaining a dual-water system would also have to be carefully considered.

As a consequence of recovering oxidized iron minerals such as hematite and goethite rather than magnetite, the resulting concentrate is different than typical Minnesota magnetite concentrate. The iron grade for oxidized iron concentrate is approximately 61-62% total Fe while typical taconite or magnetite concentrate is typically above 67% total Fe. The phosphorus content for concentrates produced from oxidized iron formation is higher (~0.03-0.04% phosphorus) than for magnetite concentrate (<0.02% phosphorus). The reason for this decrease in iron and increase in phosphorus is the abundance of goethite in the concentrate. The NatFlot process produced a high-goethite concentrate with the Bulk 19-2 sample at a phosphorus grade of approximately 0.03%, whereas the concentrate produced with the Bench 18-1 process was also high-goethite but at a phosphorus grade nearly four times higher. Characterization will be a key factor for understanding proper feed blends to produce concentrate for pellet production and desired quality.

Besides the changes in chemical assay, oxidized iron minerals like goethite also change the pyrometallurgical characteristics of the material. The presence of goethite and hematite instead of magnetite in the concentrate negatively impacts pellet production rates, production costs, and pellet quality. Goethite is a hydrated iron-oxide mineral, meaning additional energy is required

to produce an oxide pellet from goethite-rich concentrate as compared to a typical pellet produced with magnetite-rich concentrate. During induration goethite loses hydroxide and is converted to hematite. The loss-on-ignition (LOI) value for oxidized iron concentrate is higher than for taconite or partially oxidized taconite concentrate. For example, the LOI of the Bulk 19-2 concentrate was approximately seven percent, resulting in seven percent weight loss during the induration process compared to approximately one percent for taconite. This means that for a similar amount of pellet plant feed, oxidized ores would yield fewer tons of pellets than for typical taconite ore. However, the results from this study found an increased concentrate weight recovery from crude ore compared to a typical taconite operation, meaning more tons of pellets could be produced from each ton of crude ore. This could significantly reduce mining- and tailings disposal costs to offset the higher energy consumption during induration. After indurating, the pellets produced with taconite and oxidized iron concentrates would have similar total iron content but different phosphorus values, based on the feed phosphorus content.

The second phase of this study was to investigate the selected technology on pilot-scale. The most significant operational challenge from the pilot-scale NatFlot testing was managing the froth level while controlling the metallurgical performance of the process. The use of the spray nozzles in the feed and discharge ends of the flotation banks effectively controlled the froth volume; however, this also added water to the circuit and decreased the residence time. In future tests, high pressure-low volume spray nozzles would be recommended for this application. Furthermore, it would be advised to use tank cells versus banks of mechanical cells. Utilization of tank cells could reduce bypass and allow better control of the process by tight control on the amount of material floated per stage. The performance of the pilot flotation cells was poor compared to the performance of the bench flotation cells. The rougher froths were higher in iron which made the scavenger circuit a necessity. The scope of the project did not allow for optimization of the circuit which should be considered in any follow-up testing.

Although the NatFlot process could produce a concentrate that met the silica grade constraint in a rougher-cleaner pilot-scale process, the recovery was lower than expected. A secondary key finding was the need for a scavenging circuit. The NatFlot process produced a pilot-scale cleaner concentrate of 61.4% total iron and 4.2% SiO_2 at a weight and iron recovery of 27.9% and 40.9%, respectively. Mathematically combining the bench-scale scavenger concentrate at a 4.0% SiO_2 concentrate silica grade, the combined cleaner-scavenger concentrate had an estimated weight and iron recovery of 47.5% and 69.4%, respectively, at a total amine consumption of approximately 0.29 g/metric ton (0.64 lb./LT). This suggests that the scavenger process may be recovering the fine goethite and hematite being lost in the rougher and cleaner flotation cells during the pilot-scale testing. One result of adding a scavenging circuit to the process was a final concentrate with a finer size (estimated P80 42 µm cleaner flotation concentrate vs. a P80 35 µm for composite concentrate). Furthermore, scavenging may also be a method for composite concentrate silica grade control.

The third phase of this study was to investigate the production of a blast furnace-grade pellet. The pyrometallurgical studies discussed in this report were conducted using the cleaner flotation concentrate. However, the concentrate produced at a commercial operation would likely be composed of the cleaner and scavenger streams to achieve a specific concentrate grade and recovery. It is possible that the slight decrease in the particle size would impact

pyrometallurgical results. The physical and metallurgical properties of pellets depend on several variables, and particle size is just one of the various parameters that need to be considered.

The oxidized iron concentrate produced from the Bulk 19-2 could be considered a unique iron ore concentrate due to the mineralogy of the material, specifically the high amount of goethite present. This unique mineralogy introduces another variable often not seen in commercial pelletization and induration processes. To fully understand how the particle size and mineralogy of each oxidized iron concentrate may influence the physical and metallurgical properties of pellets, a detailed experimental design must be conducted for each type of material produced.

The brief pelletization and induration studies conducted during this project have shown that it is possible to produce a pellet that meets the physical and metallurgical specifications of blast furnace grade pellets using the concentrate produced during the pilot-scale NatFlot testing with the Bulk 19-2 material. However, this was not a comprehensive study aimed at optimizing the pelletization or induration components for this project. As a consequence, there are several areas of pelletization and induration that need to be researched to optimize using this highly-goethitic oxidized iron concentrate for blast furnace grade pellet production. Areas of interest include binder evaluation, various ore blends, furnace design, firing cycle, and alternate pellet applications. In addition, the economics should also be considered, including the likely increased cost to produce concentrate from goethite-rich concentrate, and the likely lower value of a pellet with an elevated phosphorus content. Included in the study should be the potential benefits for higher concentrate weight recovery and the reduction in tailing disposal cost.

Basic water characterization was also conducted as part of the pilot-scale testing; it should be noted that the tailings thickener overflow water would report back to the concentrator as makeup water in the process. Any water released to the environment from a processing operation would go through an additional water treatment plant before release to the environment. Furthermore, the results of the water characterization were based on an open-circuit water system. In production, this water would be continually recycled. It was found during the pilot-scale testing that the highest total hardness ions were found in the process water and total tailings streams, with lesser ions in the flotation feed and concentrate streams. The total froth stream having the total hardness levels most similar to the process water seems logical as this stream contained the most tap water due to the froth spray bars. The flotation feed had the lowest total hardness of the three samples. This may be a result of NatFlot process, where the newly created mineral surfaces during grinding would adsorb divalent cations. This should be explored further in future testing programs.

It was also found that the amine removal rate in the pilot-scale testing was slightly lower than in the bench-scale testing (94.5% vs. 96.0%), at measurements of approximately 1.8 mg/L and 1.1 mg/L for the pilot- and bench-test, respectively. The finding that there was detectible levels of residual amine in the both the pilot- and bench-scale water could explain why the lock-cycle testing did not require the first stage of amine after the first cycle. This finding will also important for future research, as the water recycle system may need to be capable of managing the residual amine in the process at pilot- and commercial-scale.

When conducting a basic comparison of the tailings thickener overflow water to the local wastewater treatment plant or Minnesota's Freshwater Quality standards for Class 2A waters, the water was generally found to be within regulatory limits with the exception of total dissolved solids and diesel range organics (DRO). The total suspended solids metric, however, would likely improve based upon proper operation of the tailings thickener and basic water treatment. COD and BOD were detected in the process effluent, likely due to the amine and starch. The sulfate concentrations were below the Secondary Drinking Water standard of 250 mg/L, but were an order of magnitude higher than the SO_4 standard of 10 mg/L applied to Wild Rice waters.

In summary, NatFlot performed well with hard water and oxidized ores consisting of goethite, hematite, magnetite, and quartz. NatFlot performed less well when carbonate minerals and certain silicate minerals (talc and stilpnomelane, for example) were present. Based upon the current understanding of the NatFlot process and the oxidized iron resources in Minnesota, it is unlikely that the current taconite waste (tailings) would be amenable to beneficiation with the NatFlot process due to the mineralogy of the waste stream; specifically carbonates (e.g., ankerite) and silicates (e.g., talc and stilpnomelane). It is more likely that areas that have a relatively simple and more fully oxidized mineralogy could be processed with this technology. For example, if zones of highly oxidized materials above or between taconite ore zones are found, NatFlot may be able to process these areas and reduce the costs associated with stripping and/or tailings management. However, these zones may be relatively small and difficult to selectively mine. Furthermore, the recovered concentrate would either need to be blended with the magnetite concentrate, which would likely require modifications to the existing induration system, or report to a separate agglomeration and pellet plant, which would require significant capital. Regardless, characterization will be key to developing and maintaining a blending strategy capable of producing a consistent concentrate and pellet product at current taconite facilities.

Alternatively, in-situ oxidized iron formation and the remaining legacy direct shipping ore pit walls, legacy stockpiles, and tailings in tailings ponds may be amenable to beneficiation by the NatFlot process, but additional characterization and metallurgical testing is needed to more fully understand these opportunities. To produce an oxide pellet with these resources would also likely require modifications to an existing pellet facility or an entirely new agglomeration and pellet plant. It is important to obtain a greater understanding of the mineralogy of the remaining resource in order to identify the direction of beneficiation research to ensure effective and efficient processing of remaining iron resources in the State. Minnesota has taconite reserves and resources to meet the demands of domestic iron ore production for many decades, but it will likely take many years to develop the technologies needed to assure continued mining, reduction of waste, and a reduction of the environmental impact of iron mining in Minnesota.

Future Research Recommendations

Geometallurgical characterization is critical to understanding the remaining iron resource in Minnesota. The iron resource is likely composed of a mixture of taconite, partially oxidized iron formation, and oxidized iron formation. It is essential to understand the mineralogy and liberation characteristics of iron formation in order to determine the direction research should take in order to meet the changing needs of the iron industry in Minnesota. The first phase of a comprehensive geometallurgical characterization program will include careful logging of existing core, identification of mineralogy and variability of mineralogy in potential ore and waste beds, and liberation characteristics. The data collected during the first phase will be used to focus research efforts in the second phase. The second phase would include research into existing and alternative beneficiation methods and development of alternative increased value iron products. The characterization will include floatability and zeta potential studies to identify opportunities for improving flotation with the reduction of chemicals or use of more environmentally friendly chemicals. Understanding the variability of the mineralogy, liberation characteristics, and metallurgical response to various beneficiation methods will be critical for developing a sustainable mine plan for processing mixed partially oxidized, oxidized, and taconite iron ore. Furthermore, continued testing of alternative flotation reagent testing should be explored in the future.

A thorough review of the characteristics of the problematic minerals encountered during the development of the NatFlot process should be conducted. The physical and chemical properties of talc, stilpnomelane, and the common carbonate minerals should be assessed for opportunities to make changes to the NatFlot process to improve its ability to process a wider variety of ore types. Selective flocculation-deslime technology could also be evaluated as a process to handle mixed ore types likely to be encountered at the western end of the Mesabi Iron Range. While WHIMS did not prove to be a viable option using specific operating conditions, a study of the ore characteristics that prevented the method being selective from should be conducted. For example, the influence of fineness of grind, particle size distribution, and flocculation of fine particles on selectivity should be studied.

A combination of physical separation and pyrometallurgical techniques could also be explored for processing partially oxidized iron formation. This study should begin with a thorough study of historic reduction methods. While historic reduction methods have proved to be technically successful, they have been cost prohibitive. Furthermore, a separate test program targeting the production of low-silica concentrate could be explored to support the emerging direct-reduced iron market.

As mentioned in the *Pilot-Scale Pyrometallurgical Testing Summary* section, there are several areas of pelletization and induration that could be researched to optimize the use of this lower quality concentrate for blast furnace grade pellet production. Assuming that the beneficiation technology shows potential for economic production of iron concentrate at acceptable quality, some areas of research in the near future should include the following:

- Binder evaluation – Studies to optimize bentonite dosage and/or an organic/bentonite binder blend should be conducted to optimize pellet chemistry and green ball properties.

- Ore blends – Mixtures of concentrates from partially oxidized, oxidized and taconite would be assessed to reduce, or eliminate, the use of an additional carbon source (e.g., coke breeze).
- Firing cycle – Due to the large amount of goethite present in the concentrate, optimizing the drying and preheat section of the firing cycle should be assessed.
- Alternate pellet applications – There is no doubt that blast furnaces are currently the largest consumer of iron ore pellets. However, the use of lower quality concentrate produced from processing partially oxidized and oxidized iron formation should be considered in alternate iron making technologies.

Conclusions

The findings in this research study suggest that the NatFlot process is applicable for processing oxidized iron formation that contains hematite, goethite, and quartz. Alternative methods or combinations of beneficiation technologies will be required to process iron formation that contains carbonate and platy silicate minerals. A combination of physical separation and pyrometallurgical techniques could also be explored when processing partially oxidized iron formation. The bench-scale testing using wet high-intensity magnetic separation (WHIMS) as a scavenger of LIMS tails under a fixed set of conditions was not successful, as the concentrates contained high amount of quartz and talc. The weight recovery also dropped significantly when attempting to increase the concentrate grade, suggesting that re-grinding and flotation may be required to make final concentrate silica target.

The pilot-scale testing found that the NatFlot process could produce a concentrate that met the silica grade constraint in a rougher-cleaner pilot-scale process, but the recovery was lower than expected. A bench-scale NatFlot test found potential for a scavenging circuit to recover 70% more weight and iron, with the concentrate being finer than the cleaner concentrate. The brief pelletization and induration studies conducted during this project have shown that it is possible to produce a pellet that meets the physical and metallurgical specifications of blast furnace grade pellets using the concentrate produced during the pilot-scale NatFlot testing with the Bulk 19-2 material.

Basic water characterization was also conducted as part of the pilot-scale testing; it should be noted that the tailings thickener overflow water would report back to the concentrator as makeup water in the process. Any water released to the environment from a mining operation would go through an additional water treatment plant before release to the environment. Furthermore, the results of the water characterization were based on an open-circuit water system. In production, this water would be continually recycled. When conducting a basic comparison of the tailings thickener overflow water to the local wastewater treatment plant or Minnesota's Freshwater Quality standards for Class 2A waters, the thickener overflow water complied with Minnesota water quality regulations with the exception of total dissolved solids and diesel range organics (DRO).

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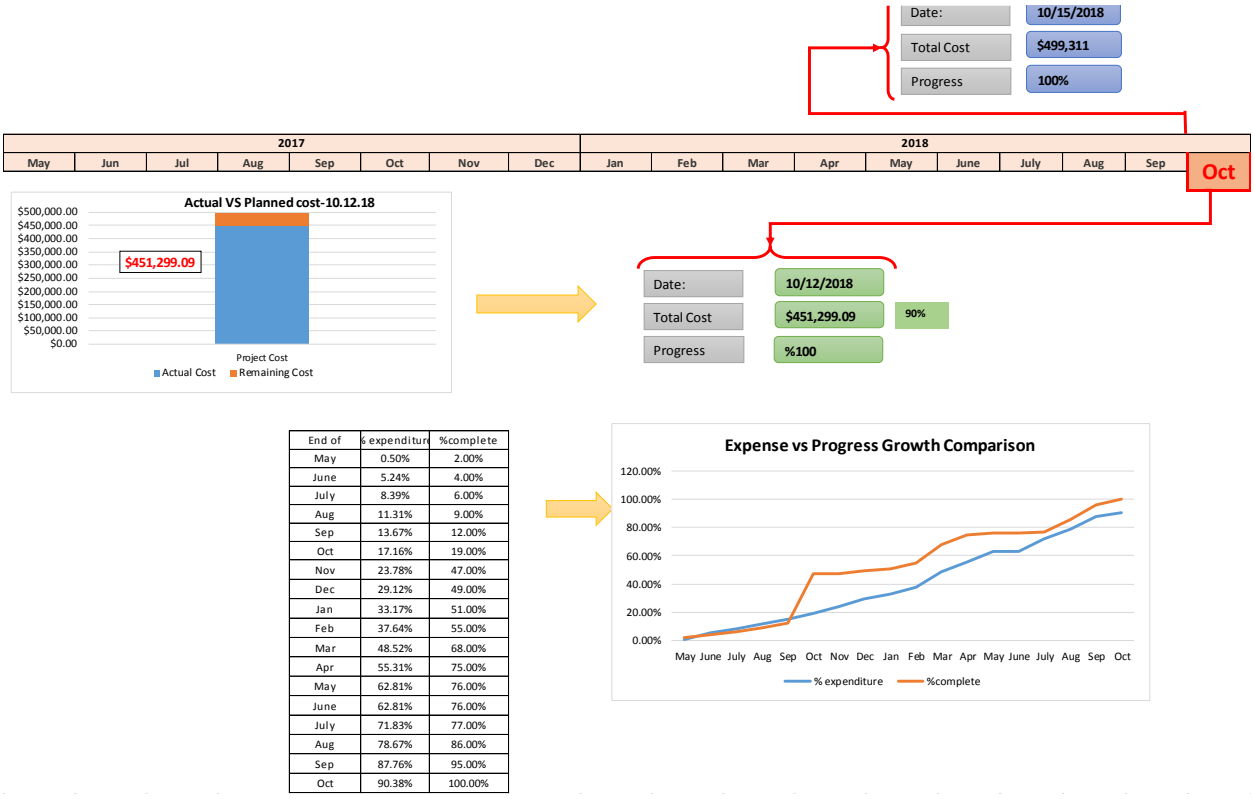
Project Management Summary

Total project timeline was approximately 17 months and total cost was $451,299.



End of	% expenditure	%complete
May	0.50%	2.00%
June	5.24%	4.00%
July	8.39%	6.00%
Aug	11.31%	9.00%
Sep	13.67%	12.00%
Oct	17.16%	19.00%
Nov	23.78%	47.00%
Dec	29.12%	49.00%
Jan	33.17%	51.00%
Feb	37.64%	55.00%
Mar	48.52%	68.00%
Apr	55.31%	75.00%
May	62.81%	76.00%
June	62.81%	76.00%
July	71.83%	77.00%
Aug	78.67%	86.00%
Sep	87.76%	95.00%
Oct	90.38%	100.00%

Appendices

Appendix 1: Bench- and Pilot-Scale Equipment Description

Bench Testing Equipment

- Rod mill: 21.3 cm (8.4 inch) diameter by 24.8 cm (9.8 inch) length with no lifters. The mill was loaded to approximately 41.8% volumetric loading with the ninety five (95) 1.3 cm (0.5 inch) diameter steel rods. The total rod charge weight was 21.1 kg (46.3 lbs). The feed to the mill was targeted to be 65% (w/w).
- Low-intensity magnetic separator: Stearns Magnetics concurrent electro-magnetic 30.5 cm width by 38.1 cm diameter wet drum low-intensity magnetic separator,
- Wet high-intensity magnetic separator: Eriez SSS-I-145 cyclic pulsating wet high gradient separator. The magnetic field intensity was varied from 0.1 T – 1.2 T with a 2.0 mm rod matrix. The operational parameters included frequency of 45 Hz, bottom orifice of 9 mm, and stroke length of 20 mm. The feed density was approximately 25%, 200 gram sample charges, and 10 second flush.
- Liberation grind mill: 15.2 cm (6.0 inch) diameter by 10.3 cm (4.1 inch) length mill with no lifters. The mill charge was broken into approximately seventy-seven 1.9 cm grinding media (68% by weight) and one hundred-seventeen 1.3 cm grinding media (32% by weight).
- Davis Tubes: Dings model MDTT with magnetic field intensity of 5,000 Gauss. Other operational parameters included sample feed weight of 10.00 grams per test, 1.5 liter per minute (0.4 gallon per minute), 90-100 strokes per minute, and 8-minute residence time.
- Thickener/settling testing: glass 2 L cylinder 8.9 cm (3.5 inch) diameter by 48.3 cm (19 inch) length.
- Ball mill Bond Work Index mill: Bico-Braun Bond work index (BWI) ball mill.
- Flotation: Denver D1 naturally aspirated mechanical cell with a 2 L bowl. Float feed charges were typically 600 grams at approximately 25% solids.
- Batch balling tire: 39.4 cm (15.5 inches) inside diameter with a 12.7 cm (5 inches) width.
- Mini-pot furnace: A cylindrical, refractory line vessel with a 11.43 cm (4.5 inches) inside diameter and a height of 12.7 cm (5 inches).

Pilot Testing Equipment

- Jaw crusher: Telsmith 10x16 pilot jaw crusher.
- Gyratory crusher: Allis Chalmers 9640 pilot gyratory (cone) crusher.
- Rolls crusher: Sturtevant Mill Co. Balanced Crushing Roll (No. 911).
- High pressure grinding rolls: KHD/Weir model RP90/25 with studded lining on the rolls. Roll sizing was 90 cm diameter by 25 cm width (35.4 in x 9.8 in). Nitrogen pressure was pre-set externally before conducting the grinding test. Installed motors were two (2) 75 kW (101 HP). Operational data was recorded with a datalogger.
- Screen: Sweco VIBRO-Energy Separator, model 1S30S6666 76 cm (30 in) system equipped with a 3.4 mm (6 Mesh) screen panel.
- Ball mill: 0.76 m (30 inch) diameter by 1.32 m (52 inch) length pilot ball mill with 11.2 kW (15 HP) motor installed. The grinding media profile can be seen in the table, below.

Ball Size (in)	Ball Size (cm)	Weight (lbs)	Weight (kg)	Weight (%)
+2.00	+5.1	241	109	11.1
-2.00 + 1.50	-5.1 + 3.8	1016	461	46.8
-1.50 + 1.25	-3.8 + 3.2	657	298	30.2
-1.25 + 1.00	-3.2 + 2.5	199	90	9.2
-1.00 + 0.75	-2.5 + 1.9	58	26	2.7
Total		**2171**	**985**	**100.0**

- Screen: Single-deck Derrick Stacksizer with 45 µm screen panels installed.
- Conditioning tank: 75 L conditioning tank.
- Rougher flotation bank: bank of three mechanical flotation cells at a total volume of 303 L.
- Cleaner flotation bank: bank of two mechanical flotation cells at a total volume of 202 L.
- Tailings thickener tank: 3,140 L (830 gallon) thickener tank with rake.
- Balling disc: 91.44 cm (36 inches) inside diameter with a 17.78 cm (7 inches) width.
- Pot grate furnace: A cylindrical, refractory lined vessel with a 34.29 cm (13.5 inches) inside diameter and a height of 60.96 cm (24 inches).

Appendix 2: Metallurgical Test Data for Baseline NatFlot Tests with Bench 17-1 through 17-4 Samples (Test 17-23 to Test 17-26 and Test 17-31)

Test Information						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
17-23	17-41	NatFlot Baseline with Type 17-1 (15-min RM #3 grind)	Type 17-1	28.38	28.76			36.0	52.94	7.60	66.3
17-24	17-42	NatFlot Baseline with Type 17-2 (15-min RM #3 grind)	Type 17-2	27.56	27.62			35.4	53.05	8.72	68.1
17-25	17-43	NatFlot Baseline with Type 17-3 (15-min RM #3 grind)	Type 17-3	28.29	28.57			36.8	55.29	10.42	71.2
17-26	17-44	NatFlot Baseline with Type 17-4 (15-min RM #3 grind)	Type 17-4	29.37	29.00			37.3	54.35	10.16	69.9
17-31	17-49	NatFlot Baseline with Type 17-2 (8-min RM #3 grind)	Type 17-2	27.56	27.47			37.5	49.58	13.36	67.6

	Test Conditions															
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
17-23	NatFlot Rougher	98.1		7.8		Tap									1.5	0.35
17-24	NatFlot Rougher	97.7		7.75		Tap									1.5	0.45
17-25	NatFlot Rougher	97.6		N/A		Tap									1.5	0.50
17-26	NatFlot Rougher	97.9		N/A		Tap									1.5	0.45
17-31	NatFlot Rougher	70.4		7.99		Tap									1.5	0.45

	Product Weights										Product Chemical Analyses								
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
17-23		79	119.3	76.5	61.5	46.2				215.2	597.7		9.58	9.64	12.41	21.20	35.50				52.94
17-24		19.7	113.8	98	87.1	39.7	26.2			211	595.5		9.64	8.33	8.42	14.52	26.72	36.86			53.05
17-25		56.6	101.9	81.3	80.3	35.6	22.3			220.1	598.1		11.28	9.51	9.83	13.20	22.97	28.52			55.29
17-26		18.4	87.2	96.6	106.3	38.4	26.5			222.2	595.6		14.37	11.66	10.31	12.33	21.25	29.94			54.35
17-31		22.8	90.7	83.7	94.3	44.9	36.4			223.4	596.2		10.06	8.78	9.69	13.57	23.34	31.30			49.58

	%SiO2									
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2
17-23										7.6
17-24										8.72
17-25										10.42
17-26										10.16
17-31										13.36

Other Assays of Interest
Con phos: 0.015%
Con phos: 0.016%
Con phos: 0.015%
Con phos: 0.014%
Con Phos: 0.020%

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
17-23	PA12	0.05	0.05	0.05	0.1	0.1				0.35	4.0	4.5	4.0	3.0	Not Recorded				15.5
17-24	PA12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	2.5	5.0	5.0	4.0	3.0	1.5			21.0
17-25	PA12	0.1	0.05	0.05	0.1	0.1	0.1			0.50	5.5	6.5	6.5	4.5	3.0	1.5			27.5
17-26	PA12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.0	6.0	6.0	5.0	2.5	2.0			24.5
17-31	PA12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	6.0	5.5	3.5	2.0	1.5			22.0

Appendix 3: Metallurgical Test Data for Baseline NatFlot Tests with Bench 18-1 through 18-3 Samples (Test 17-28 to Test 17-30, Test 17-47, and Test 17-48)

Test Information						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
17-28	17-47	NatFlot Baseline Rougher with 15 min Grind	Type 18-1	42.61	42.91			42.7	58.98	6.04	58.7
17-29	17-46	NatFlot Baseline Rougher with 15 min Grind (Note: 9 stages of flotation so deslime = F1)	Type 18-2	44.13	44.28	5.1	25.8	44.1	63.28	1.66	63.0
17-30	17-50	NatFlot Baseline Rougher with 15 min Grind	Type 18-3	34.43	34.31			43.7	61.84	9.79	78.7
17-47	17-60	NatFlot Baseline Rougher with 8 min Grind	Type 18-2	44.13	44.62			57.6	60.56	6.39	78.1
17-48	17-59	NatFlot Baseline Rougher with 10 min Grind	Type 18-3	34.43	34.44			48.5	58.32	16.14	82.1

Test Conditions																
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
17-28	NatFlot Rougher	92.6		7.19		Tap									1.5	0.55
17-29	NatFlot Rougher	95.0		7.37		Tap									1.5	0.80
17-30	NatFlot Rougher	89.2		7.6		Tap									1.5	0.60
17-47	NatFlot Rougher	62.2		7.45		Tap									1.5	0.55
17-48	NatFlot Rougher	71.6		7.47		Tap									1.5	0.60

Product Weights											
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g
17-28		22.7	56.1	66	125.1	42.1	16.7	11.6		253.6	593.9
17-29	29.9	68.7	76.8	53.8	30.4	20.9	17.7	17	14.1	260	589.3
17-30		6.4	19.2	34.6	124.4	102	31.1	18.9		261.1	597.7
17-47		4.8	22.5	47.2	81.8	50.4	28.3	19		344.8	598.8
17-48		4.9	13.9	30.1	89.2	91.3	49.8	28.5		289.3	597.0

Product Chemical Analyses									
Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
	43.92	41.02	34.26	23.11	25.89	31.93	39.09		58.98
25.82	20.31	17.88	20.81	36.50	49.53	55.70	58.86	60.39	63.28
	25.46	22.60	18.19	10.94	9.02	14.00	22.02		61.84
	35.03	30.59	24.57	17.19	21.24	26.12	31.92		60.56
	25.91	23.55	19.84	10.94	7.86	10.21	15.22		58.32

%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2
17-28										6.04
17-29	58.56	69.33	73.97	69.59	42.6	21.91	11.5	6.49	7.38	1.66
17-30										9.79
17-47										6.39
17-48										16.14

Other Assays of Interest
Con Phos: 0.119%; Con 650 LOI: 6.65; Con 950 LOI: 7.20
Con Phos: 0.037%; Con 650 LOI: 7.02; Con 950 LOI: 7.33
Con Phos: 0.033%; Con 650 LOI: 0.31; Con 950 LOI: 0.27
Con Phos: 0.038%
Con Phos: 0.031%

Amine Additions										
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)
17-28	PA12	0.05	0.05	0.05	0.1	0.1	0.1	0.1		0.55
17-29	PA12	0.15	0.05	0.1	0.1	0.1	0.1	0.1	0.1	0.80
17-30	PA12	0.1	0.05	0.05	0.1	0.1	0.1	0.1		0.60
17-47	PA12	0.05	0.05	0.05	0.1	0.1	0.1	0.1		0.55
17-48	PA12	0.1	0.05	0.05	0.1	0.1	0.1	0.1		0.60

Flot Times								
1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
3.0	5.0	5.5	5.5	4.5	2.0	1.5		27.0
10.0	5.5	4.0	4.0	3.0	2.5	2.0	1.5	32.5
2.0	3.5	4.0	5.5	4.0	2.0	1.5		22.5
2.0	3.0	5.0	4.0	3.0	2.0	1.5		20.5
2.0	2.5	4.0	4.0	3.0	2.0	1.5		19.0

Appendix 4: Metallurgical Test Data for Davis Tube Tests with Bench 17-1 through 17-4 Samples (Test 17-32 and Test 17-40)

WO17-32 and WO17-40

17-1

Lib Grind Time (min)	Grind (P80, µm)	Product	Wt (%)	Total Fe (%)	SiO_2 (%)	Phos (%)		TFe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
6	75	Feed	100.0	28.38	48.13	0.017		100.0	100.0	100.0
		DTC	11.8	62.12	10.51	0.014		25.8	2.6	9.7
		DTT	88.2	23.87	53.16	0.017		74.2	97.4	90.3
12	38	Feed	100.0	28.38	48.13	0.017		100.0	100.0	100.0
		DTC	9.5	68.08	4.29	0.011		22.8	0.8	6.1
		DTT	90.5	24.21	52.73	0.018		77.2	99.2	93.9
15	31	Feed	100.0	28.38	48.13	0.017		100.0	100.0	100.0
		DTC	9.4	69.46	3.35	0.007		23.0	0.7	3.9
		DTT	90.6	24.12	52.78	0.018		77.0	99.3	96.1

17-2

Lib Grind Time (min)	Grind (P80, µm)	Product	Wt (%)	Total Fe (%)	SiO_2 (%)	Phos (%)		TFe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
6	77	Feed	100.0	27.56	49.41	0.016		100.0	100.0	100.0
		DTC	14.8	62.76	9.97	0.013		33.7	3.0	12.0
		DTT	85.2	21.45	56.26	0.017		66.3	97.0	88.0
12	39	Feed	100.0	27.56	49.41	0.016		100.0	100.0	100.0
		DTC	12.3	68.29	4.79	0.011		30.5	1.2	8.5
		DTT	87.7	21.85	55.67	0.017		69.5	98.8	91.5
15	32	Feed	100.0	27.56	49.41	0.016		100.0	100.0	100.0
		DTC	11.8	69.56	3.44	0.007		29.8	0.8	5.2
		DTT	88.2	21.94	55.56	0.017		70.2	99.2	94.8

17-3

Lib Grind Time (min)	Grind (P80, µm)	Product	Wt (%)	Total Fe (%)	SiO_2 (%)	Phos (%)		TFe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
6	83	Feed	100.0	28.29	51.84	0.015		100.0	100.0	100.0
		DTC	19.2	62.92	10.31	0.014		42.7	3.8	17.9
		DTT	80.8	20.06	61.71	0.015		57.3	96.2	82.1
12	40	Feed	100.0	28.29	51.84	0.015		100.0	100.0	100.0
		DTC	16.5	68.89	3.66	0.012		40.2	1.2	13.2
		DTT	83.5	20.27	61.36	0.016		59.8	98.8	86.8
15	32	Feed	100.0	28.29	51.84	0.015		100.0	100.0	100.0
		DTC	16.2	69.97	2.77	0.010		40.1	0.9	10.8
		DTT	83.8	20.23	61.33	0.016		59.9	99.1	89.2

17-4

Lib Grind Time (min)	Grind (P80, µm)	Product	Wt (%)	Total Fe (%)	SiO_2 (%)	Phos (%)		TFe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
6	85	Feed	100.0	29.37	47.46	0.017		100.0	100.0	100.0
		DTC	25.8	62.77	9.90	0.015		55.1	5.4	22.8
		DTT	74.2	17.76	60.52	0.018		44.9	94.6	77.2
12	40	Feed	100.0	29.37	47.46	0.017		100.0	100.0	100.0
		DTC	22.0	67.99	3.89	0.013		50.9	1.8	16.8
		DTT	78.0	18.48	59.75	0.018		49.1	98.2	83.2
15	34	Feed	100.0	29.37	47.46	0.017		100.0	100.0	100.0
		DTC	21.3	69.13	3.20	0.010		50.1	1.4	12.5
		DTT	78.7	18.61	59.44	0.019		49.9	98.6	87.5

Appendix 5: Metallurgical Test Data for Davis Tube Tests with Bench 18-3 Sample (Test 17-43 and Test 17-45)

WO17-43 and WO17-45

18-3

Lib Grind Time (min)	Grind (P80, µm)	Product		Wt (%)	Total Fe (%)	SiO_2 (%)	Phos (%)		TFe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
3	141	Feed		100.0	34.43	49.58	0.020		100.0	100.0	100.0
		DTC		38.8	62.92	10.71	0.021		70.9	8.4	40.7
		DTT		61.2	16.37	74.22	0.019		29.1	91.6	59.3
6	75	Feed		100.0	34.43	49.58	0.020		100.0	100.0	100.0
		DTC		34.3	67.27	4.98	0.016		67.0	3.4	27.4
		DTT		65.7	17.29	72.86	0.022		33.0	96.6	72.6
12	41	Feed		100.0	34.43	49.58	0.020		100.0	100.0	100.0
		DTC		32.0	69.45	2.35	0.013		64.5	1.5	20.8
		DTT		68.0	17.95	71.81	0.023		35.5	98.5	79.2

Appendix 6: Metallurgical Test Data for Low-Intensity Magnetic Separation and Wet High-Intensity Magnetic Separation with Bench 18-3 Sample (Test 17-49 and Test 17-52)

18-3 LIMS+WHIMS

WO17-49 and WO17-52

Rod Size: 2 mm

Pulse Freq: 45 Hz

Orifice Size: 8 mm

Product	Grind (P80, µm)	Product	Wt (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	TFe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
3-min	Target was ~67 um	Head	100.0	34.43	49.58	0.020	100.0	100.0	100.0
		LIMS Con (1350 G)	35.7	66.10	6.35	0.016	68.6	4.3	50.9
		LIMS Tail (1350 G)	64.3	16.83	71.31	0.027	31.4	95.7	49.1
	As-Is	12000 Gauss							
		Tails 1+2	43.0	6.62	90.06	N/A	13.7	28.7	N/A
		Con	57.0	22.31	65.80	N/A	86.3	71.3	N/A
		Head (calc)	100.0	15.56	76.24	N/A	100.0	100.0	N/A
	As-Is	8000 Gauss							
		Tails 1+2	43.0	6.56	91.45	0.020	14.3	8.1	37.0
		Con	57.0	22.80	69.95	0.034	85.7	91.9	63.0
		Head (calc)	100.0	15.81	79.20	0.028	100.0	100.0	100.0
	As-Is	4000 Gauss							
		Tails 1+2	55.0	7.40	88.76	N/A	21.1	47.1	N/A
		Con	45.0	25.53	60.67	N/A	78.9	52.9	N/A
		Head (calc)	100.0	15.55	76.13	N/A	100.0	100.0	N/A
	As-Is	2000 Gauss							
		Tails 1+2	67.7	8.84	85.88	N/A	32.9	64.7	N/A
		Con	32.3	30.25	54.4	N/A	67.1	35.3	N/A
		Head (calc)	100.0	15.76	75.70	N/A	100.0	100.0	N/A
	As-Is	1000 Gauss							
		Tails 1+2	83.3	12.06	82.72	0.026	56.7	77.8	88.3
		Con	16.7	34.89	48.07	0.038	43.3	22.2	11.7
		Head (calc)	100.0	15.88	76.92	0.028	100.0	100.0	100.0

Appendix 7: Metallurgical Test Data for Selective Flocculation Tests with Bench 17-2 Sample (Test 17-46)

Test Information						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
17-46	17-61	SFDF Baseline with Type 17-2 (8-min RM #3 grind)	Type 17-2	27.56	27.56	3.2	5.2	41.7	47.52	15.36	71.9

	Test Conditions															
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
17-46	SFDF Rougher			10.76; 10.65		25:1 DI:Tap						Sodium Silicate; NaOH	Mill	Sodium Silicate (0.75 #/LT); NaOH (1.5 #/LT)	1.25	0.50

	Product Weights										
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g
17-46	18.8	82.1	106.1	69.5	45.2	25.2				248.2	595.1

Product Chemical Analyses									
Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
5.22	11.49	11.32	13.32	18.28	24.36				47.52

	%SiO2									
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2
17-46										15.36

Other Assays of Interest
Con Phos: 0.024%

	Amine Additions									
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)
17-46	PA12	0.1	0.1	0.1	0.1	0.1				0.50

Flot Times								
1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
2.5	2.5	2.0	2.0	1.5				10.5

Appendix 8: Metallurgical Test Data for Selective Flocculation Tests with Bench 18-1, 18-2, and 18-3 Samples (Test 17-33, Test 17-34, and Test 17-35)

Test Information					Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
17-33	17-53	SFDF Baseline Rougher with 15 min Grind	Type 18-1	42.61	42.93	31.8	28.9	46.0	60.65	4.25	65.0
17-34	17-54	SFDF Baseline Rougher with 15 min Grind	Type 18-2	44.13	44.12	13.0	14.3	56.1	62.62	2.21	79.7
17-35	17-57	SFDF Baseline Rougher with 15 min Grind	Type 18-3	34.43	34.39	28.3	6.1	42.7	65.03	5.72	80.7

	Test Conditions															
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
17-33	SFDF Rougher			10.46; 10.60		25:1 DI:Tap						Sodium Silicate; NaOH	Mill	Sodium Silicate (0.75 #/LT); NaOH (1.5 #/LT)	1.25	0.50
17-34	SFDF Rougher			10.64; 10.61		25:1 DI:Tap						Sodium Silicate; NaOH	Mill	Sodium Silicate (0.75 #/LT); NaOH (1.5 #/LT)	1.25	0.60
17-35	SFDF Rougher			10.94; 10.6		25:1 DI:Tap						Sodium Silicate; NaOH	Mill	Sodium Silicate (0.75 #/LT); NaOH (1.5 #/LT)	1.25	0.50

	Product Weights										
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g
17-33	189.5	52.1	31.1	13.9	10.7	11.5	12.6			273.9	595.3
17-34	77.4	84	50.4	18.7	13.2	8.9	9.2			335.1	596.9
17-35	169	32.7	78.9	32.7	18.5	10.8				254.9	597.5

	Product Chemical Analyses									
Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
28.94	12.25	20.74	32.15	42.43	50.66	54.98			60.65
14.26	11.21	21.19	35.74	45.85	51.63	54.67			62.62
6.12	8.05	11.42	21.67	33.52	41.28				65.03

	%SiO2									
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2
17-33										4.25
17-34										2.21
17-35										5.72

Other Assays of Interest
Con Phos: 0.121%
Con Phos: 0.034%
Con Phos: 0.027%

	Amine Additions									
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)
17-33	PA12	0.1	0.1	0.1	0.1	0.1				0.50
17-34	PA12	0.1	0.1	0.1	0.1	0.1	0.1			0.60
17-35	PA12	0.1	0.1	0.1	0.1	0.1				0.50

	Flot Times								
1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
2.0	2.0	1.5	1.5	1.5				8.5
2.0	2.5	2.0	2.0	1.5	1.5			11.5
2.0	3.5	2.5	2.5	1.5				12.0

Appendix 9: As-Received Bulk 19-2 3.4 mm (6 Mesh) Screen Undersize Assay by Size

Tyler Mesh	Micron	Weight %	Total Fe (%)	Total Fe Dist (%)
6	3327	0.0	N/A	0.0
8	2362	0.4	N/A	0.0
10	1651	0.9	N/A	0.0
14	1168	5.6	37.56	11.7
20	841	3.6	35.51	7.1
28	595	4.4	35.24	8.5
35	420	4.5	34.16	8.4
48	297	4.2	33.29	7.7
65	210	3.9	32.54	7.0
100	149	3.7	30.60	6.3
150	105	4.8	23.02	6.1
200	74	9.0	14.17	7.0
270	53	13.9	9.72	7.5
325	44	4.4	6.70	1.6
400	37	5.3	6.47	1.9
500	25	7.9	3.28	1.4
-500	-25	23.7	13.54	17.8
Head (calc)		**100.0**	**18.07**	**100.0**

Appendix 10: Whole Rock Analysis of Bulk 19-1 and 19-2 Samples from Activation Laboratories

Report Number: A18-00644
Report Date: 6/2/2018

Analyte Symbol	SiO2	Al2O3	Fe2O3	MnO	MgO	CaO	Na2O	K2O	TiO2	P2O5	LOI	LOI2	Total	Total 2	Fe2O3(T)	FeO
Unit Symbol	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%
Detection Limit	0.01	0.01	0.01	0.001	0.01	0.01	0.01	0.01	0.001	0.01			0.01	0.01	0.01	0.1
Analysis Method	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	TITR
Bulk 19-1 Head	55.01	1.59	34.1	0.13	0.79	0.65	0.35	0.29	0.071	0.05	1.17	0.52	100.6	99.99	40.55	5.8
Bulk 19-2 Head	34.37	1.29	57.22	0.068	0.27	0.49	0.29	0.16	0.067	0.05	4.66	4.57	99.83	99.74	58.11	0.8

Analyte Symbol	Au	As	Br	Cr	Ir	Sb	Sc	Se	Mass	Cd	Cu	Ni	Zn	S	Ag	Pb
Unit Symbol	ppb	ppm	ppm	ppm	ppb	ppm	ppm	ppm	g	ppm	ppm	ppm	ppm	%	ppm	ppm
Detection Limit	2	0.5	0.5	5	5	0.2	0.1	3		0.5	1	1	1	0.001	0.3	5
Analysis Method	INAA	INAA	INAA	INAA	INAA	INAA	INAA	INAA	INAA	TD-ICP	TD-ICP	TD-ICP	TD-ICP	TD-ICP	TD-ICP	TD-ICP
Bulk 19-1 Head	< 2	10.3	< 0.5	55	< 5	< 0.2	1.1	< 3	38.4	0.8	9	10	9	0.011	< 0.3	< 5
Bulk 19-2 Head	< 2	10.9	< 0.5	40	< 5	0.8	1.6	< 3	40.5	1.2	7	12	14	0.009	0.3	< 5

Analyte Symbol	Sc	Be	V	Cr	Co	Ni	Cu	Zn	Ga	Ge	As	Rb	Sr	Y	Zr	Nb
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit	1	1	5	20	1	20	10	30	1	1	5	2	2	1	2	1
Analysis Method	FUS-ICP	FUS-ICP	FUS-ICP	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-ICP	FUS-ICP	FUS-ICP	FUS-MS
Bulk 19-1 Head	2	1	25	70	6	< 20	< 10	< 30	3	9	11	9	39	< 1	27	1
Bulk 19-2 Head	2	1	25	40	6	< 20	10	< 30	2	9	12	5	25	< 1	21	< 1

Analyte Symbol	Mo	Ag	In	Sn	Sb	Cs	Ba	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit	2	0.5	0.2	1	0.5	0.5	2	0.1	0.1	0.05	0.1	0.1	0.05	0.1	0.1	0.1
Analysis Method	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-ICP	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS
Bulk 19-1 Head	< 2	< 0.5	< 0.2	< 1	< 0.5	< 0.5	79	4.4	8.9	0.97	3.7	0.7	0.23	0.7	0.1	0.7
Bulk 19-2 Head	3	< 0.5	< 0.2	< 1	0.9	< 0.5	45	3.4	7.9	0.81	3.3	0.7	0.23	0.8	0.1	0.9

Analyte Symbol	Ho	Er	Tm	Yb	Lu	Hf	Ta	W	Tl	Pb	Bi	Th	U
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit	0.1	0.1	0.05	0.1	0.01	0.2	0.1	1	0.1	5	0.4	0.1	0.1
Analysis Method	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS
Bulk 19-1 Head	0.1	0.4	0.06	0.4	0.05	0.6	< 0.1	5	< 0.1	< 5	< 0.4	0.8	0.4
Bulk 19-2 Head	0.2	0.6	0.08	0.5	0.08	0.3	< 0.1	6	< 0.1	< 5	< 0.4	0.9	0.9

Appendix 11: Metallurgical Test Data for Baseline NatFlot Tests (Test 17-12, Test 17-14, Test 17-77, Test 17-78, and Test 18-7)

						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
17-12	17-31	NatFlot Baseline Rougher with 15 min Grind on 16-1	Type 16-1	33.73	34.88			70.8	41.76	36.76	84.7
17-14	17-33	NatFlot Baseline Rougher with 10 min Grind on 16-1	Type 16-1	33.73	33.26			82.0	35.53	45.13	87.6
17-77	18-1	NatFlot baseline with 15-min RM grind on 19-1 HPGR 6M U/S	Type 19-1 HPGR 6M U/S	28.72	29.03			36.6	59.21	12.45	74.6
17-78	18-2	NatFlot baseline with 10-min RM grind on 19-1 HPGR 6M U/S	Type 19-1 HPGR 6M U/S	28.72	28.81			39.8	56.55	16.54	78.0
18-7	18-5	NatFlot with 13-min RM grind on 19-2 "HPGR" 6M U/S	19-2 HPGR 6M U/S	40.96	40.92			35.4	62.99	2.03	54.5

	Test Conditions														
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
17-12	NatFlot Rougher	87.0	7.34			Tap									1.5	1.15
17-14	NatFlot Rougher	74.5	7.34			Tap									1.5	1.15
17-77	NatFlot Rougher	83.3		7.88		Coleraine Tap									1.5	0.65
17-78	NatFlot Rougher	62.9		7.9		Coleraine Tap									1.5	0.65
18-7	NatFlot Rougher	87.2		8.10		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45

	Product Weights											Product Chemical Analyses									
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
17-12		4.6	7.6	11.3	22.2	27.1	37.6	46.2		379.4	536.0		28.77	27.50	26.28	24.23	21.05	15.90	11.03		41.76
17-14		8.6		12.7		26.1		32.8	24.7	479.5	584.4		27.97		27.32		25.24		21.84	17.76	35.53
17-77		7.1	9.1	17.8	63.1	98.2	111.5	44.4	27.6	218.6	597.4		24.85	22.41	19.27	14.94	10.02	7.68	9.94	16.36	59.21
17-78		3.9	7	12.7	50.1	77	106.3	66.8	35.5	237.1	596.4		23.93	21.68	18.71	14.73	9.78	6.92	8.56	13.91	56.55
18-7		77	129.9	60.6	46.2	31.9	23.3			202.3	571.2		20.69	19.83	27.02	39.55	51.51	58.10			62.99

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
17-12										36.76	Conc Phos: 0.024%
17-14										45.13	
17-77										12.45	Con Phos: 0.033%
17-78										16.54	Con Phos: 0.031%
18-7							35.61	15.85	6.96	2.03	Con Phos: 0.032%; Con LOI: 6.94%, 67.56% Tfe, 2.30% SiO2, 0.035% phos

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
17-12	PA-12	0.45	0.1	0.1	0.1	0.1	0.1	0.1	0.1	1.15	1.5	2.0	2.5	4.5	4.0	4.5	4.0	4.0	27.0
17-14	PA-12	0.45	0.1	0.1	0.1	0.1	0.1	0.1	0.1	1.15	1.5	2.0	2.5	3.0	3.0	4.0	3.5	2.5	22.0
17-77	PA-12	0.05	0.05	0.05	0.1	0.1	0.1	0.1	0.1	0.65	2.5	3.0	5.0	7.0	4.5	4.0	2.0	2.0	30.0
17-78	PA-12	0.05	0.05	0.05	0.1	0.1	0.1	0.1	0.1	0.65	2.0	3.0	4.0	7.0	5.0	4.0	2.5	2.0	29.5
18-7	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	5.0	4.0	3.0	2.5	1.5			20.0

Appendix 12: Metallurgical Test Data for Baseline NatFlot Blend Tests (Test 18-7, Test 18-10, Test 18-15, Test 18-16)

						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-7	18-5	NatFlot with 13-min RM grind on 19-2 HPGR 6M U/S	19-2 HPGR 6M U/S	40.96	40.92			35.4	62.99	2.03	54.5
18-10	18-7	NatFlot with 13-min RM grind on 50:50 19-2 HPGR 6M U/S: 19-1 HPGR 6M U/S	50:50 19-2 HPGR 6M U/S : 19-1 HPGR 6M U/S	34.84	34.67			39.5	60.13	8.43	68.5
18-15	18-11	NatFlot with 13-min RM grind on 75:25 19-2 HPGR 6M U/S: 19-1 HPGR 6M U/S	75:25 19-2 HPGR 6M U/S : 19-1 HPGR 6M U/S	37.90	38.04			41.8	61.43	5.41	67.5
18-16	18-12	NatFlot with 13-min RM grind on 90:10 19-2 HPGR 6M U/S: 19-1 HPGR 6M U/S	90:10 19-2 HPGR 6M U/S : 19-1 HPGR 6M U/S	39.74	39.70			40.1	62.89	3.00	63.5

	Test Conditions														
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-7	NatFlot Rougher	87.2		8.10		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45
18-10	NatFlot Rougher	84.6		8.20		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.55
18-15	NatFlot Rougher	72.8		8.16		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45
18-16	NatFlot Rougher	75.3		8.12		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45

	Product Weights										
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g
18-7		77	129.9	60.6	46.2	31.9	23.3			202.3	571.2
18-10		6.2	18.7	34.2	132.2	98.2	41	23.5		231	585.0
18-15		15.5	28.3	84.9	136.2	48.2	20.3			239.7	573.1
18-16		26.6	84.2	108.8	73.4	27.3	23			229.9	573.2

	Product Chemical Analyses									
Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
	20.69	19.83	27.02	39.55	51.51	58.10			62.99
	33.59	28.05	24.22	16.76	13.03	17.67	26.08		60.13
	31.74	24.72	19.29	16.46	25.51	38.09			61.43
	25.26	19.27	18.21	24.49	39.46	49.86			62.89

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
18-7							35.61	15.85	6.96	2.03	Con Phos: 0.032%; Con LOI: 6.94%, 67.56% Tfe, 2.30% SiO2, 0.035% phos
18-10										8.43	Con Phos: 0.031%
18-15										5.41	Con Phos: 0.032%
18-16										3.00	Con Phos: 0.031%

	Amine Additions									
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)
18-7	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45
18-10	PA-12	0.05	0.05	0.05	0.1	0.1	0.1	0.1		0.55
18-15	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45
18-16	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45

	Flot Times								
1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
4.0	5.0	4.0	3.0	2.5	1.5			20.0
2.0	2.5	3.0	5.5	3.5	2.0	1.5		20.0
2.5	3.0	5.0	4.0	3.0	1.5			19.0
3.0	4.0	5.0	4.0	2.5	2.0			20.5

Appendix 13: Metallurgical Test Data for Davis Tube Tests with Bench 16-1 Sample (Test 17-37)

T16-1 WO17-37

Lib Grind Time (min)	Grind (P80, µm)	Product	Wt (%)	Total Fe (%)	SiO_2 (%)	Phos (%)		TFe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
6	69	Feed	100.0	33.73	50.64	0.027		100.0	100.0	100.0
		DTC	32.6	67.13	5.34	0.022		64.9	3.4	26.6
		DTT	67.4	17.58	72.55	0.029		35.1	96.6	73.4
9	46	Feed	100.0	33.73	50.64	0.027		100.0	100.0	100.0
		DTC	31.7	68.92	3.43	0.014		64.8	2.1	16.4
		DTT	68.3	17.40	72.55	0.033		35.2	97.9	83.6
12	39	Feed	100.0	33.73	50.64	0.027		100.0	100.0	100.0
		DTC	30.3	69.60	2.70	0.013		62.5	1.6	14.6
		DTT	69.7	18.14	71.48	0.033		37.5	98.4	85.4

Appendix 14: Metallurgical Test Data for Davis Tube Tests with Bench 19-1 Sample (Test 17-76)

WO17-76

Grind Time (min)	Grind (P80, µm)	Wt (%)	Total Fe (%)	SiO_2 (%)	Phos (%)		Total Fe Rec (%)	SiO_2 Rec (%)	Phos Rec (%)
0 (head)	N/A	100.0	28.72	55.64	0.026		100.0	100.0	100.0
5	86	29.6	66.37	6.16	0.017		68.4	3.3	19.4
10	53	27.4	69.19	2.80	0.015		66.0	1.4	15.8
20	28	26.0	70.19	1.50	0.013		63.5	0.7	13.0

Appendix 15: Metallurgical Test Data for Low-Intensity Magnetic Separation and Wet High-Intensity Magnetic Separation with Bench 16-1 Sample (Test 17-51 and Test 17-53)

16-1 LIMS+WHIMS

WO17-51 and WO17-53

Rod Size: 2 mm

Pulse Freq: 45 Hz

Orifice Size: 8 mm

Product	Grind (P80, µm)	Product	Wt (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	TFe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
3-min	Target was ~59 um	Head	100.0	33.73	50.64	0.027	100.0	100.0	100.0
		LIMS Con (1350 G)	35.1	66.31	6.63	0.016	69.0	4.8	9.1
		LIMS Tail (1350 G)	64.9	16.10	76.29	0.029	31.0	95.2	90.9
	As-Is	12000 Gauss							
		Tails 1+2	42.8	7.60	87.61		19.2	61.8	#DIV/0!
		Con	57.2	21.93	63.11		80.8	38.2	#DIV/0!
		Head (calc)	100.0	15.06	74.86		100.0	100.0	#DIV/0!
	As-Is	8000 Gauss							
		Tails 1+2	47.9	8.28	86.19	0.026	23.1	66.8	62.8
		Con	52.1	22.48	61.97	0.036	76.9	33.2	37.2
		Head (calc)	100.0	15.67	73.58	0.031	100.0	100.0	100.0
	As-Is	4000 Gauss							
		Tails 1+2	60.1	8.80	84.71		32.5	80.1	#DIV/0!
		Con	39.9	26.77	54.47		67.5	19.9	#DIV/0!
		Head (calc)	100.0	15.98	72.63		100.0	100.0	#DIV/0!
	As-Is	2000 Gauss							
		Tails 1+2	76.6	9.97	82.79		46.2	90.5	#DIV/0!
		Con	23.4	34.05	43.55		53.8	9.5	#DIV/0!
		Head (calc)	100.0	15.60	73.62		100.0	100.0	#DIV/0!
	As-Is	1000 Gauss							
		Tails 1+2	89.7	13.12	77.08	0.029	73.2	99.2	100.0
		Con	10.3	42.25	33.58	0.059	26.8	0.8	0.0
		Head (calc)	100.0	16.12	72.60	0.032	100.0	100.0	100.0

Appendix 16: Metallurgical Test Data for Wet High-Intensity Magnetic Separation with Bench 19-2 Sample (Test 18-89 to Test 18-91)

19-2
WHIMS
WO18-
89/90/91

2 mm Rods
Rougher-
Cleaner

Product	Grind	Grind (P80, µm)	Magnetic Field Intensity (Tesla)	Wt (%)	Total Fe (%)	SiO_2 (%)	Total Fe Dist (%)	SiO_2 Dist (%)
Con	7-min	40	0.1	12.9	60.99	7.93	18.9	3.1
Tail			0.1	87.1	38.85	37.31	81.1	96.9
Con	7-min	40	0.4	53.3	56.02	11.82	71.2	18.9
Tail			0.4	46.7	25.80	57.83	28.8	81.1
Con	7-min	40	0.8	66.0	52.23	17.58	82.2	34.8
Tail			0.8	34.0	21.92	63.88	17.8	65.2

Product	Grind	Grind (P80)	Magnetic Field Intensity (Tesla)	Wt (%)	Total Fe (%)	SiO2 (%)	Total Fe Dist (%)	SiO2 Dist (%)
Con	9-min	29	0.1	13.0	59.11	10.31	18.4	4.1
Tail			0.1	87.0	39.19	36.54	81.6	95.9
Con	9-min	29	0.4	50.6	54.62	13.80	66.2	21.3
Tail			0.4	49.4	28.56	52.19	33.8	78.7
Con	9-min	29	0.8	66.4	50.51	20.13	80.1	39.9
Tail			0.8	33.6	24.71	59.85	19.9	60.1

Appendix 17: Metallurgical Test Data for Selective Flocculation Tests with Bulk 16-1 Samples (Test 17-12 and Test 17-17)

						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
17-12	17-31	NatFlot Baseline Rougher with 15 min Grind on 16-1	Type 16-1	33.73	34.88			70.8	41.76	36.76	84.7
17-17	17-37	SFDF Baseline Rougher with 15 min Grind on 16-1	Type 16-1	33.73	33.45	33.7	10.2	37.8	64.01	6.77	72.3

	Test Conditions														
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
17-12	NatFlot Rougher	87.0	7.34			Tap									1.5	1.15
17-17	SFDF Rougher	87 (?)	10.3; 10.35	25:1 DI:Tap								Sodium Silicate; NaOH	Mill	Sodium Silicate (0.75 #/LT); NaOH (1.5 #/LT)	1.25	1.00

	Product Weights										Product Chemical Analyses									
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
17-12		4.6	7.6	11.3	22.2	27.1	37.6	46.2		379.4	536.0		28.77	27.50	26.28	24.23	21.05	15.90	11.03		41.76
17-17	201.6	22.3	37.7	41	31.8	19.2	11.2	7.2		225.8	597.8	10.18	13.57	14.98	18.31	21.72	27.35	33.57	39.12		64.01

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
17-12										36.76	Conc Phos: 0.024%
17-17										6.77	Conc Phos: 0.028%

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
17-12	PA-12	0.45	0.1	0.1	0.1	0.1	0.1	0.1	0.1	1.15	1.5	2.0	2.5	4.5	4.0	4.5	4.0	4.0	27.0
17-17	PA-12	0.4	0.1	0.1	0.1	0.1	0.1	0.1		1.00	3.0	5.0	6.0	5.5	3.5	2.5	1.5		27.0

Appendix 18: Metallurgical Test Data for Selective Flocculation Tests with Bulk 19-1 Samples (Test 17-77 and Test 18-69)

						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-7	18-5	NatFlot with 13-min RM grind on 19-2 HPGR 6M U/S	19-2 HPGR 6M U/S	40.96	40.92			35.4	62.99	2.03	54.5
18-9	18-6	NatFlot with 10-min RM grind on 19-2 HPGR 6M U/S	19-2 HPGR 6M U/S	40.96	40.90			40.6	62.79	2.69	62.3
18-11	18-8	NatFlot with 7-min RM grind on 19-2 HPGR 6M U/S	19-2 HPGR 6M U/S	40.96	40.89			45.8	61.38	4.07	68.7

	Test Conditions														
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-7	NatFlot Rougher	87.2		8.10		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45
18-9	NatFlot Rougher	65.2		8.11		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45
18-11	NatFlot Rougher	49.0		8.08		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45

	Product Weights										Product Chemical Analyses									
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
18-7		77	129.9	60.6	46.2	31.9	23.3			202.3	571.2		20.69	19.83	27.02	39.55	51.51	58.10			62.99
18-9		58.2	116.3	65.2	46.8	30.6	21.6			231.5	570.2		20.14	18.25	23.29	31.55	44.13	53.08			62.79
18-11		42.9	99	59.4	57.2	31.6	18.3			260.6	569.0		21.11	17.76	20.89	24.39	36.03	45.60			61.38

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
18-7							35.61	15.85	6.96	2.03	Con Phos: 0.032%; Con LOI: 6.94%, 67.56% Tfe, 2.30% SiO2, 0.035% phos
18-9		64.47	70.19	61.95	48.51	25.88	13.79			2.69	Con Phos: 0.032%; Con LOI: 7.00%, 67.13% Tfe 2.93% SiO2, 0.034% phos
18-11							24.03			4.07	Con Phos: 0.033%; Con LOI: 6.85%, 65.84% Tfe 4.42% SiO2, 0.034% phos

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
18-7	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	5.0	4.0	3.0	2.5	1.5			20.0
18-9	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.5	4.0	3.0	2.5	1.5			19.0
18-11	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.0	4.0	3.0	2.5	2.0	1.5			16.0

Appendix 19: Metallurgical Test Data for Effect of Grind (Test 18-7, Test 18-9, and Test 18-11)

						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-7	18-5	NatFlot with 13-min RM grind on 19-2 HPGR 6M U/S	19-2 HPGR 6M U/S	40.96	40.92			35.4	62.99	2.03	54.5
18-9	18-6	NatFlot with 10-min RM grind on 19-2 HPGR 6M U/S	19-2 HPGR 6M U/S	40.96	40.90			40.6	62.79	2.69	62.3
18-11	18-8	NatFlot with 7-min RM grind on 19-2 HPGR 6M U/S	19-2 HPGR 6M U/S	40.96	40.89			45.8	61.38	4.07	68.7

	Test Conditions														
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-7	NatFlot Rougher	87.2		8.10		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45
18-9	NatFlot Rougher	65.2		8.11		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45
18-11	NatFlot Rougher	49.0		8.08		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45

	Product Weights										Product Chemical Analyses									
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
18-7		77	129.9	60.6	46.2	31.9	23.3			202.3	571.2		20.69	19.83	27.02	39.55	51.51	58.10			62.99
18-9		58.2	116.3	65.2	46.8	30.6	21.6			231.5	570.2		20.14	18.25	23.29	31.55	44.13	53.08			62.79
18-11		42.9	99	59.4	57.2	31.6	18.3			260.6	569.0		21.11	17.76	20.89	24.39	36.03	45.60			61.38

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
18-7							35.61	15.85	6.96	2.03	Con Phos: 0.032%; Con LOI: 6.94%, 67.56% Tfe, 2.30% SiO2, 0.035% phos
18-9		64.47	70.19	61.95	48.51	25.88	13.79			2.69	Con Phos: 0.032%; Con LOI: 7.00%, 67.13% Tfe 2.93% SiO2, 0.034% phos
18-11							24.03			4.07	Con Phos: 0.033%; Con LOI: 6.85%, 65.84% Tfe 4.42% SiO2, 0.034% phos

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
18-7	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	5.0	4.0	3.0	2.5	1.5			20.0
18-9	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.5	4.0	3.0	2.5	1.5			19.0
18-11	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.0	4.0	3.0	2.5	2.0	1.5			16.0

Appendix 20: Metallurgical Test Data for Effect of Starch Dosage Rate (Test 18-11, Test 18-20, and Test 18-21)

						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-11	18-8	NatFlot with 7-min RM grind on 19-2 HPGR 6M U/S	19-2 HPGR 6M U/S	40.96	40.89			45.8	61.38	4.07	68.7
18-20	18-15	NatFlot with 7-min RM grind on 19-2 HPGR 6M U/S with Lower Starch (0.75 #/LT)	19-2 HPGR 6M U/S	40.96	40.94			41.5	61.78	3.80	62.5
18-21	18-16	NatFlot with 7-min RM grind on 19-2 HPGR 6M U/S with Higher Starch (3.0 #/LT)	19-2 HPGR 6M U/S	40.96	40.98			47.9	61.07	4.51	71.5

	Test Conditions															
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-11	NatFlot Rougher	49.0		8.08		Coleraine Tap						Glass H	Mill	0.25 #/LT	1.5	0.45
18-20	NatFlot Rougher			7.94		Coleraine Tap						Glass H	Mill	0.25 #/LT	0.75	0.45
18-21	NatFlot Rougher	48.0		7.84		Coleraine Tap						Glass H	Mill	0.25 #/LT	3.0	0.45

	Product Weights										Product Chemical Analyses									
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
18-11		42.9	99	59.4	57.2	31.6	18.3			260.6	569.0		21.11	17.76	20.89	24.39	36.03	45.60			61.38
18-20		47.8	107.1	67.3	55.2	31.2	26.3			237.1	572.0		20.13	18.29	23.76	29.28	41.74	50.67			61.78
18-21		29.4	97.7	67.6	60.2	25.4	17.7			274.5	572.5		23.72	18.25	19.19	22.95	32.08	40.91			61.07

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
18-11							24.03			4.07	Con Phos: 0.033%; Con LOI: 6.85%, 65.84% Tfe 4.42% SiO2, 0.034% phos
18-20							17.28			3.80	Con Phos: 0.036%
18-21							30.32			4.51	Con Phos: 0.037%

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
18-11	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.0	4.0	3.0	2.5	2.0	1.5			16.0
18-20	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.0	4.0	4.0	3.0	2.0	1.5			17.5
18-21	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	2.0	4.0	3.5	3.0	2.0	1.5			16.0

Appendix 21: Metallurgical Test Data for Effect of Polyphosphate Dosage Rate (Test 18-26 to Test 18-29)

						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-26	18-27	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Glass H series (0.25 #/LT)	19-2 HPGR 6M U/S	40.96	40.85			42.7	62.02	3.26	64.8
18-27	18-28	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Glass H series (0.00 #/LT)	19-2 HPGR 6M U/S	40.96	40.95			43.6	61.99	3.35	66.1
18-28	18-29	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Glass H series (0.50 #/LT)	19-2 HPGR 6M U/S	40.96	40.71			40.3	62.29	3.28	61.7
18-29	18-30	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Glass H series (1.00 #/LT)	19-2 HPGR 6M U/S	40.96	40.61			33.7	62.40	2.96	51.8

	Test Conditions															
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-26	NatFlot Rougher	60.4	7.46	7.98	8.16	Coleraine Tap	107.4	35.6	415	7.72	0.675	Glass H	Mill	0.25 #/LT	1.5	0.45
18-27	NatFlot Rougher	58.0	"	7.91	8.03	Coleraine Tap	"	"	#VALUE!	"	"				1.5	0.45
18-28	NatFlot Rougher		"	8.09	8.24	Coleraine Tap	"	"	#VALUE!	"	"	Glass H	Mill	0.50 #/LT	1.5	0.45
18-29	NatFlot Rougher		"	8.05	8.18	Coleraine Tap	"	"	#VALUE!	"	"	Glass H	Mill	1.00 #/LT	1.5	0.45

	Product Weights										
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g
18-26		57.6	108.5	55.8	58.4	27.8	19.2			243.9	571.2
18-27		57.7	118.1	58.5	49	24.1	15.8			250.3	573.5
18-28		46.9	106.8	71.2	61.1	33	22.1			230.5	571.6
18-29		35.3	96.6	82.4	74.6	50.2	41.7			193.8	574.6

	Product Chemical Analyses									
Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
	21.50	18.17	21.81	28.16	41.98	50.37			62.02
	22.81	19.20	21.90	28.09	40.30	47.77			61.99
	19.07	17.44	22.74	31.69	44.32	51.43			62.29
	16.78	14.74	21.03	35.32	49.61	56.80			62.40

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
18-26							17.29			3.26	Con Phos: 0.033%; F6 Phos: 0.045%
18-27							20.88			3.35	Con Phos: 0.032%; F6 Phos: 0.041%
18-28							15.49			3.28	Con Phos: 0.034%; F6 Phos: 0.041%
18-29							9.27			2.96	Con Phos: 0.039%; F6 Phos: 0.043%

	Amine Additions									
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)
18-26	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45
18-27	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45
18-28	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45
18-29	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45

	Flot Times								
1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
3.5	5.0	3.0	3.0	2.0	1.5			18.0
3.5	4.0	3.0	3.0	2.0	1.5			17.0
4.0	5.0	5.0	3.0	2.5	1.5			21.0
3.0	4.0	4.5	4.0	3.0	2.0			20.5

Appendix 22: Metallurgical Test Data for Effect of Process Water Type (Test 18-37 to Test 18-43)

					Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-37	18-38	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S 100% Tap	19-2 HPGR 6M U/S	40.96	40.78			42.2	62.01	3.16	64.1
18-38	18-39	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S 75:25 Tap:DI	19-2 HPGR 6M U/S	40.96	40.89			41.1	62.60	2.77	62.9
18-39	18-40	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S 50:50 Tap:DI	19-2 HPGR 6M U/S	40.96	41.01			41.8	62.19	3.33	63.4
18-40	18-41	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S 25:75 Tap:DI	19-2 HPGR 6M U/S	40.96	40.90			41.5	62.28	3.10	63.3
18-41	18-42	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S 1:25 Tap:DI	19-2 HPGR 6M U/S	40.96	40.89			41.9	62.09	3.31	63.7
18-42	18-43	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S 100% DI	19-2 HPGR 6M U/S	40.96	40.64			41.8	61.94	3.67	63.7
18-43	18-44	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S 50:50 Tap:Distilled	19-2 HPGR 6M U/S	40.96	40.93			42.3	62.11	3.31	64.2

	Test Conditions														
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-37	NatFlot Rougher		7.09	7.90	8.06	100% Coleraine Tap	108.4	35.4	416	7.71	0.688				1.5	0.45
18-38	NatFlot Rougher		7.65	8.01	8.17	75:25 Tap:DI	88.2	28.7	338	7.97	0.521				1.5	0.45
18-39	NatFlot Rougher		7.60	8.11	8.25	50:50 Tap:DI	53.8	17.5	206	7.98	0.373				1.5	0.45
18-40	NatFlot Rougher		7.69	8.35	8.47	25:75 Tap:DI	30.8	10.5	120	7.97	0.205				1.5	0.45
18-41	NatFlot Rougher		7.00	8.74	8.79	1:25 Tap:DI	3.9	2.0	18	7.66	0.045				1.5	0.45
18-42	NatFlot Rougher		6.46	8.89	9.05	100% DI	BDL	0.3	#VALUE!	7.50	0.000				1.5	0.45
18-43	NatFlot Rougher		8.16	8.20	8.34	50:50 Tap:Distilled	62.2	20.2	238	8.30	0.382				1.5	0.45

	Product Weights										Product Chemical Analyses									
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
18-37		66.7	109.3	64.2	47.4	24.1	19.8			241.8	573.3		22.37	20.09	22.38	28.16	40.19	48.38			62.01
18-38		74.7	110.2	68	42	27.4	15			235.3	572.6		22.72	20.03	22.87	30.00	43.39	51.52			62.60
18-39		74.2	116.1	66.8	42.8	22	13.7			241.2	576.8		24.55	22.14	23.58	27.15	39.61	47.88			62.19
18-40		91	125.6	58.1	28.4	19.2	14.2			239.1	575.6		23.52	22.54	25.43	28.53	38.20	46.27			62.28
18-41		88.2	121	62.2	34.8	16	12.4			241.5	576.1		23.50	22.80	24.89	28.39	39.20	45.78			62.09
18-42		81.5	116.8	72.5	38	16	9.9			240.3	575.0		22.63	22.87	25.05	29.68	37.58	42.89			61.94
18-43		72.8	121	60.3	38.6	22.7	17.2			243.6	576.2		23.75	21.17	24.10	27.39	37.30	46.91			62.11

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
18-37						31.91	19.35			3.16	Con Phos: 0.032%
18-38						27.55	15.52			2.77	Con Phos: 0.032%
18-39						32.75	20.74			3.33	Con Phos: 0.033%
18-40						36.18	22.08			3.1	Con Phos: 0.032%
18-41						31.75	22.05			3.31	Con Phos: 0.032%
18-42						34.85	26.79			3.67	Con Phos: 0.032%
18-43						36.56	21.64			3.31	Con Phos: 0.031%

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
18-37	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.0	4.0	3.0	2.0	2.0			18.5
18-38	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.0	4.0	3.0	3.0	2.0			19.5
18-39	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	4.0	3.5	3.0	2.0	1.5			18.0
18-40	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	3.5	3.0	2.0	2.0	1.5			16.0
18-41	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.0	3.0	2.5	2.0	Not Recorded			15.0
18-42	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	3.5	3.0	2.5	2.0	1.5			16.0
18-43	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	5.0	3.5	2.5	2.0	1.5			18.5

Appendix 23: Metallurgical Test Data for Effect of Flotation Feed pH (Test 18-44 to Test 18-49)

						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-44	18-45	NatFlot with 9-min RM grind on19-2 HPGR 6M U/S pH Series (Natural)	19-2 HPGR 6M U/S	40.96	40.83			39.7	62.46	2.69	60.7
18-45	18-46	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S pH Series (pH 6)	19-2 HPGR 6M U/S	40.96	41.02			42.6	62.38	3.04	64.8
18-46	18-50	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S pH Series (pH 7)	19-2 HPGR 6M U/S	40.96	40.78			41.8	62.19	2.91	63.8
18-47	18-51	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S pH Series (pH 8)	19-2 HPGR 6M U/S	40.96	40.32			41.0	62.40	2.87	63.5
18-48	18-52	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S pH Series (pH 9)	19-2 HPGR 6M U/S	40.96	40.68			41.1	62.19	2.81	62.8
18-49	18-53	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S pH Series (pH 10)	19-2 HPGR 6M U/S	40.96	40.77			43.7	61.54	3.80	65.9

	Test Conditions														
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-44	NatFlot Rougher		6.82	7.86	8.00	Coleraine Tap	98.3	34.8	389	8.71	0.621				1.5	0.45
18-45	NatFlot Rougher		"	7.98	6.04	Coleraine Tap	"	"	#VALUE!	"	"				1.5	0.45
18-46	NatFlot Rougher		"	8.00	6.98	Coleraine Tap	"	"	#VALUE!	"	"				1.5	0.45
18-47	NatFlot Rougher		"	7.98	8.00	Coleraine Tap	"	"	#VALUE!	"	"				1.5	0.45
18-48	NatFlot Rougher		"	8.09	9.10	Coleraine Tap	"	"	#VALUE!	"	"				1.5	0.45
18-49	NatFlot Rougher		"	8.01	10.07	Coleraine Tap	"	"	#VALUE!	"	"				1.5	0.45

	Product Weights										Product Chemical Analyses									
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
18-44		68	122.8	61.1	44.6	27	24.1			228.5	576.1		23.11	20.94	24.01	28.22	41.48	52.28			62.46
18-45		50.6	103.6	57.3	62.5	34.3	22.3			245.7	576.3		23.35	18.58	21.47	24.59	38.80	49.80			62.38
18-46		58.8	109.4	57	55.9	32.1	22.8			241.7	577.7		23.92	19.13	21.43	25.35	38.81	50.10			62.19
18-47		64.3	118.1	62.3	49.9	27.5	18.7			236.9	577.7		23.15	18.70	22.14	27.49	40.46	50.80			62.40
18-48		74	112	63.9	45.5	25.8	21.5			239	581.7		21.42	20.29	22.96	29.21	40.99	50.52			62.19
18-49		51.6	86.8	61.1	71.7	32.9	19.8			251.3	575.2		19.03	19.47	21.42	26.09	37.38	45.71			61.54

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
18-44							14.59			2.69	Con Phos: 0.032%
18-45							17.86			3.04	Con Phos: 0.031%
18-46							16.09			2.91	Con Phos: 0.031%
18-47							16.05			2.87	Con Phos: 0.030%
18-48							16.09			2.81	Con Phos: 0.030%
18-49							23.11			3.8	Con Phos: 0.030%

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
18-44	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	5.0	4.0	3.0	2.5	2.0			20.0
18-45	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	5.0	3.5	3.5	3.0	2.0			20.5
18-46	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	5.0	3.5	3.0	2.5	2.0			20.0
18-47	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	5.0	4.0	3.5	2.5	1.5			20.0
18-48	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	4.5	4.0	3.0	2.5	2.0			20.0
18-49	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	2.5	3.5	3.0	3.5	2.5	Not Recorded			15.0

Appendix 24: Metallurgical Test Data for Effect of Collector Type (Test 18-30 to Test 18-34)

					Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-30	18-31	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S PA12	19-2 HPGR 6M U/S	40.96	40.72			40.8	62.34	3.05	62.5
18-31	18-32	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Amine 2	19-2 HPGR 6M U/S	40.96	40.74			37.2	62.97	2.46	57.6
18-32	18-33	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Amine 3	19-2 HPGR 6M U/S	40.96	40.86			38.2	62.91	2.37	58.9
18-33	18-34	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Amine 4 (Clariant EDA)	19-2 HPGR 6M U/S	40.96	40.80			40.6	62.38	2.98	62.1
18-34	18-35	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Amine 5	19-2 HPGR 6M U/S	40.96	40.65			37.0	62.84	2.40	57.2

	Test Conditions														
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-30	NatFlot Rougher		7.36	7.98	8.12	Coleraine Tap	106.4	35.1	410	7.86	0.705	Glass H	Mill	0.25 #/LT	1.5	0.45
18-31	NatFlot Rougher		"	7.90	8.08	Coleraine Tap	"	"	#VALUE!	"	"	Glass H	Mill	0.25 #/LT	1.5	0.45
18-32	NatFlot Rougher		"	8.04	8.23	Coleraine Tap	"	"	#VALUE!	"	"	Glass H	Mill	0.25 #/LT	1.5	0.45
18-33	NatFlot Rougher		"	8.06	8.21	Coleraine Tap	"	"	#VALUE!	"	"	Glass H	Mill	0.25 #/LT	1.5	0.45
18-34	NatFlot Rougher		"	8.02	8.29	Coleraine Tap	"	"	#VALUE!	"	"	Glass H	Mill	0.25 #/LT	1.5	0.45

	Product Weights										Product Chemical Analyses									
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
18-30		45.6	102.1	68.1	62.1	32	27.7			232.9	570.5		22.01	18.96	21.07	26.97	39.76	50.23			62.34
18-31		79.1	87.4	64.3	60.5	36.5	28.7			211.5	568.0		17.02	17.53	23.30	34.57	48.29	55.43			62.97
18-32		84.8	99.4	65.6	49.2	28.6	19			214.6	561.2		17.36	18.94	25.72	38.37	50.27	55.84			62.91
18-33		92.5	91.8	54.1	54.2	28.9	18.9			232.7	573.1		16.74	18.65	24.75	35.25	47.02	52.73			62.38
18-34		117.6	94.5	53	49.9	26.7	19			211.6	572.3		17.09	20.53	28.74	41.21	51.43	56.11			62.84

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
18-30						33.01	17.11			3.05	Con Phos: 0.034%
18-31						20.5	10.69			2.46	Con Phos: 0.034%
18-32						16.86	10.13			2.37	Con Phos: 0.034%
18-33						22.28	13.88			2.98	Con Phos: 0.035%
18-34						15.96	9.72			2.40	Con Phos: 0.031%

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
18-30	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	4.0	3.0	3.0	2.5	2.5			19.0
18-31	Amine 2	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	3.5	3.0	3.0	2.5	2.0			17.5
18-32	Amine 3	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	3.5	3.5	3.0	2.5	1.5			18.0
18-33	Amine 4 (Clariant Flotigam EDA)	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	3.5	2.5	2.5	2.0	1.5			15.5
18-34	Amine 5	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	3.5	3.0	3.0	2.0	Not Recorded			15.0

Appendix 25: Metallurgical Test Data for Triplicate Testing (Test 18-51 to Test 18-53)

						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-51	18-47	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Repeatability #1	19-2 HPGR 6M U/S	40.96	40.65			42.6	62.12	3.25	65.0
18-52	18-48	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Repeatability #2	19-2 HPGR 6M U/S	40.96	40.65			41.7	62.19	3.16	63.7
18-53	18-49	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S Repeatability #3	19-2 HPGR 6M U/S	40.96	40.50			42.4	62.14	3.24	65.0

	Test Conditions														
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-51	NatFlot Rougher		N/A	8.08	8.13	Coleraine Tap			0						1.5	0.45
18-52	NatFlot Rougher		"	8.03	8.13	Coleraine Tap			0						1.5	0.45
18-53	NatFlot Rougher		"	8.04	8.14	Coleraine Tap			0						1.5	0.45

	Product Weights										Product Chemical Analyses									
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
18-51		58.6	106.9	66.2	55.2	29.1	16.1			246	578.1		23.61	19.60	20.48	26.66	39.40	47.41			62.12
18-52		60.7	114.1	66.5	50.6	25.7	19.5			240.8	577.9		23.01	19.74	22.28	27.56	40.13	49.19			62.19
18-53		65.3	111.5	61.2	49.6	26.8	18			244.5	576.9		22.92	19.23	21.64	26.90	37.90	47.68			62.14

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
18-51							19.36			3.25	Con Phos: 0.032%
18-52							18.67			3.16	Con Phos: 0.031%
18-53							20.67			3.24	Con Phos: 0.032%

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
18-51	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.0	3.5	3.0	2.0	1.5			17.5
18-52	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.0	3.5	3.0	2.0	1.5			17.5
18-53	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.0	3.5	3.0	2.0	1.5			17.5

Final Concentrate Grade/Recovery

WO	Test	Weight Recovery (%)	Total Fe (%)	SiO2 (%)	Total Fe Recovery (%)	Amine (#/LT)	Time (min)
18-51	Test 1	42.6	62.12	3.25	65.0	0.45	17.5
18-52	Test 2	41.7	62.19	3.16	63.7	0.45	17.5
18-53	Test 3	42.4	62.14	3.24	65.0	0.45	17.5
Average		42.2	62.15	3.22	64.6	0.45	17.5
Std. Dev.		0.38	0.03	0.04	0.60	0.00	0.00

Concentrate Grade/Recovery @ 4.0% SiO2

WO	Test	Weight Recovery (%)	Total Fe (%)	Total Fe Recovery (%)	Amine (#/LT)	Time (min)
18-51	Test 1	44.7	61.44	67.5	0.37	16.4
18-52	Test 2	44.1	61.49	66.7	0.38	16.4
18-53	Test 3	44.4	61.51	67.4	0.39	16.5
Average		44.4	61.48	67.2	0.38	16.4
Std. Dev.		0.23	0.03	0.35	0.01	0.08

Appendix 26: Metallurgical Test Data for As-Is vs. Reduced-Hardness Triplicate Water Testing (Test 18-62 to Test 18-67)

					Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-62	18-58	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S triplicate series (Hard Water #1)	WO18-57 19-2 HPGR 6M U/S	42.01	42.46			41.9	62.26	2.88	61.5
18-63	18-59	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S triplicate series (Hard Water #2)	WO18-57 19-2 HPGR 6M U/S	42.01	42.33			42.2	62.32	2.67	62.1
18-64	18-60	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S triplicate series (Hard Water #3)	WO18-57 19-2 HPGR 6M U/S	42.01	42.33			43.2	62.20	2.96	63.5
18-65	18-61	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S triplicate series (Soft Water #1)	WO18-57 19-2 HPGR 6M U/S	42.01	42.39			42.9	62.01	3.33	62.8
18-66	18-62	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S triplicate series (Soft Water #2)	WO18-57 19-2 HPGR 6M U/S	42.01	42.50			43.7	62.06	3.33	63.8
18-67	18-63	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S triplicate series (Soft Water #3)	WO18-57 19-2 HPGR 6M U/S	42.01	42.06			41.8	62.21	2.93	61.8

	Test Conditions															
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-62	NatFlot Rougher		6.72	7.43	7.92	Coleraine Tap	92.2	36.2	379	8.78	0.623				1.5	0.45
18-63	NatFlot Rougher		"	7.57	7.96	Coleraine Tap	"	"	#VALUE!	"	"				1.5	0.45
18-64	NatFlot Rougher		"	7.53	7.95	Coleraine Tap	"	"	#VALUE!	"	"				1.5	0.45
18-65	NatFlot Rougher	67.0	7.61	8.30	8.48	25:75 Tap:Distilled	21.5	9.4	92	8.19	0.195				1.5	0.45
18-66	NatFlot Rougher		"	8.31	8.49	25:75 Tap:Distilled	"	"	#VALUE!	"	"				1.5	0.45
18-67	NatFlot Rougher		"	8.39	8.52	25:75 Tap:Distilled	"	"	#VALUE!	"	"				1.5	0.45

	Product Weights										Product Chemical Analyses									
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g	Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
18-62		64	106.5	61.6	53	29.7	21.5			242.8	579.1		25.40	22.47	23.58	31.00	42.50	50.84			62.26
18-63		61.5	102.2	64.4	54.3	28.4	24.4			244.8	580.0		25.09	21.85	24.02	28.62	41.64	50.73			62.32
18-64		57.5	97.2	66.5	54.4	30.9	21.7			249.5	577.7		24.81	21.30	22.69	28.86	41.08	50.23			62.20
18-65		77.3	118.6	58	38.8	21.2	16.2			248.2	578.3		25.38	25.08	26.34	28.21	38.08	46.79			62.01
18-66		76.2	112.1	63.7	38.1	20.1	15.3			252.9	578.4		25.06	24.90	25.61	29.67	36.32	45.34			62.06
18-67		79.6	126.6	56.1	35.9	20.2	18.4			241.8	578.6		24.71	24.25	26.51	30.19	40.35	47.22			62.21

	%SiO2										
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	Other Assays of Interest
18-62						30.4	15.51			2.88	Con Phos: 0.031%
18-63						30.69	15.88			2.67	Con Phos: 0.033%
18-64						32.24	16.41			2.96	Con Phos: 0.034%
18-65							22.28			3.33	Con Phos: 0.033%
18-66							23.49			3.33	Con Phos: 0.032%
18-67							20.06			2.93	Con Phos: 0.034%

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
18-62	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	4.5	4.0	3.5	2.5	2.0			20.5
18-63	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	4.5	4.0	3.5	2.5	2.0			20.5
18-64	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	4.0	4.5	4.0	3.5	2.5	2.0			20.5
18-65	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.0	3.0	2.5	2.0	1.5			16.5
18-66	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.0	3.0	2.5	2.0	1.5			16.5
18-67	PA-12	0.05	0.05	0.05	0.1	0.1	0.1			0.45	3.5	4.0	3.0	2.5	2.0	1.5			16.5

Concentrate Grade/Recovery @ 4.0% SiO2						
Water		**Weight Recovery (%)**	**Total Fe (%)**	**Total Fe Recovery (%)**	**Amine (#/LT)**	**Time (min)**
100% Tap	Average	46.4	61.24	67.1	0.35	18.5
	Std. Dev.	0.44	0.05	0.72	0.00	0.08
25:75 Tap:Distilled	Average	44.8	61.42	65.0	0.38	15.5
	Std. Dev.	0.33	0.10	0.38	0.01	0.21

Appendix 27: Bulk 19-2 Assay by Size Data

Product	Weight (g)	Weight (%)	Total Fe (%)	SiO_2 (%)		Total Fe Dist (%)	SiO_2 Dist (%)
Flotation Con	353.4	60.7	29.38	53.23		42.1	97.2
Flotation Tail	228.5	39.3	62.55	2.33		57.9	2.8
Head (calc)	581.9	100.0	42.41	33.24		100.0	100.0

WO18-79											
BF-71											
Bulk 19-2											
9-min RM#3 Grind											
9-min Head											
Wt %	100.0										
Tyler Mesh	Micron	Weight %	Passing Cum. %	Total Fe (%)	SiO2 (%)		Total Fe Dist (%)	SiO2 Dist (%)		Total Fe Units	SiO2 Units
+270	+53	2.1	97.9	45.96	24.43		2.3	1.6		0.99	0.53
325	44	2.9	95.0	48.75	20.99		3.3	1.8		1.41	0.61
400	37	6.6	88.4	47.42	24.68		7.3	4.8		3.11	1.62
500	25	19.0	69.4	42.87	32.99		19.2	18.7		8.15	6.27
-500	-25	69.4	0.0	41.44	35.27		67.8	73.1		28.76	24.48
Calculated Head		100.0	31.7	42.41	33.50		100.0	100.0		42.41	33.50
9-min NatFlot Con											
Wt %	39.3										
Tyler Mesh	Micron	Weight %	Passing Cum. %	Total Fe (%)	SiO2 (%)		Total Fe Dist (%)	SiO2 Dist (%)		Total Fe Units	SiO2 Units
+270	+53	2.4	97.6	61.54	4.58		2.4	4.5		1.48	0.11
325	44	5.0	92.6	61.54	3.39		4.9	6.9		3.05	0.17
400	37	9.6	83.0	62.09	2.93		9.5	11.7		5.99	0.28
500	25	31.3	51.7	62.79	2.36		31.3	30.5		19.65	0.74
-500	-25	51.7	0.0	63.05	2.17		51.9	46.3		32.60	1.12
Calculated Head		100.0	35.8	62.77	2.42		100.0	100.0		62.77	2.42
Direct Head				62.55	2.33						
9-min NatFlot Tail											
Wt %	60.7										
Tyler Mesh	Micron	Weight %	Passing Cum. %	Total Fe (%)	SiO2 (%)		Total Fe Dist (%)	SiO2 Dist (%)		Total Fe Units	SiO2 Units
+270	+53	2.0	98.0	33.80	39.92		2.3	1.5		0.67	0.79
325	44	1.5	96.5	22.30	57.38		1.2	1.7		0.35	0.89
400	37	4.6	91.9	27.37	54.40		4.3	4.6		1.25	2.48
500	25	11.1	80.8	6.43	89.02		2.4	18.4		0.71	9.85
-500	-25	80.8	0.0	32.50	48.96		89.8	73.9		26.27	39.58
Calculated Head		100.0	N/A	29.25	53.59		100.0	100.0		29.25	53.59
Recovery by Size											
Tyler Mesh	Micron	Total Fe Re	SiO2 Rec (%)								
+270	+53	58.7	8.2								
325	44	85.1	10.9								
400	37	75.6	6.9								
500	25	94.7	4.6								
-500	-25	44.5	1.8								

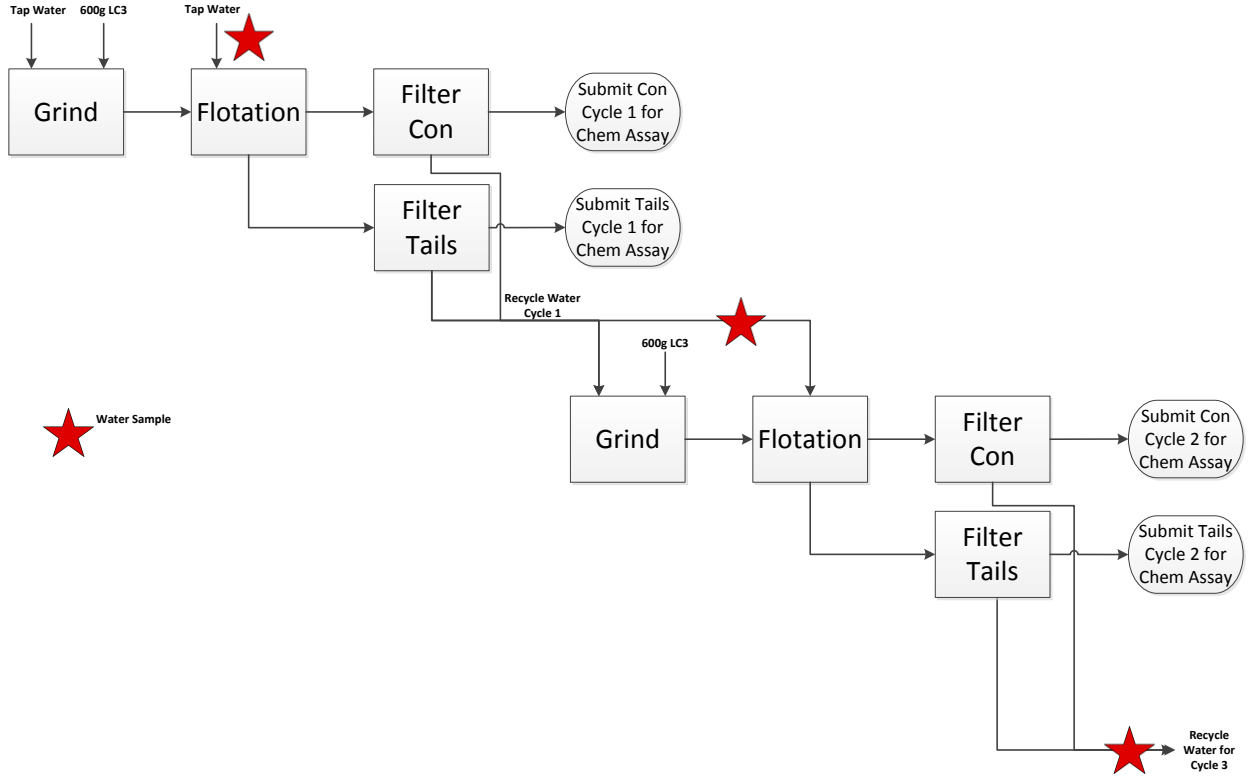
Appendix 28: Metallurgical Test Data for Froth (Tailings) Settling Tests (Test 18-55 to Test 18-56)

No Flocculant	
Pulp Level (cm)	Time (min)
39.4	0.0
37.4	5.5
27.6	10.0
23.6	11.5
19.7	13.5
15.8	16.0
11.8	19.0
7.9	25.0

Aluminum Sulfate							
0.001 #/LT Alum		0.01 #/LT Alum		0.1 #/LT Alum		1.0 #/LT Alum	
Pulp Level (cm)	Time (min)	Pulp Level (cm)	Time (min)	Pulp Level (cm)	Time (min)	Pulp Level (cm)	Time (min)
39.4	0.0	39.4	0.0	39.4	0.0	39.4	0.0
37.4	6.0	37.4	4.0	37.4	3.5	37.4	3.0
35.5	9.0	33.5	7.0	33.5	7.0	35.5	5.0
31.5	13.5	31.5	8.5	31.5	9.0	31.5	9.0
27.6	16.5	27.6	10.5	27.6	11.0	27.6	11.0
19.7	18.0	23.6	13.5	23.6	14.0	23.6	14.0
15.8	20.0	19.7	16.0	19.7	17.5	19.7	17.5
11.8	23.0	15.8	19.0	15.8	21.0	15.8	20.0
7.9	28.0	11.8	22.5	11.8	23.5	13.8	22.5
6.3	37.5	9.1	24.0	9.1	25.5	11.8	24.0
		6.3	38.0	7.9	30.0	9.9	25.0
				6.3	40.0	9.1	27.0
						7.1	39.0

Polymer Flocculant (NS6850)					
0.001 #/LT NS6850		0.01 #/LT NS6850		0.1 #/LT NS6850	
Pulp Level (cm)	Time (min)	Pulp Level (cm)	Time (min)	Pulp Level (cm)	Time (min)
39.4	0.0	39.4	0.0	39.4	0.0
33.5	6.5	37.4	1.0	37.4	0.1
31.5	7.5	33.5	1.5	35.5	0.2
29.6	8.0	31.5	2.0	31.5	0.3
27.6	9.5	29.6	2.3	27.6	0.6
25.6	10.5	25.6	2.5	23.6	0.9
23.6	11.5	21.7	3.0	19.7	1.2
21.7	12.5	15.8	3.5	15.8	1.8
19.7	13.5	13.8	5.0	13.8	2.6
17.7	14.5	11.8	7.0	11.8	4.0
15.8	15.5	9.9	12.0	9.9	8.3
13.8	17.0	7.9	25.0	8.7	21.0
11.8	18.0	7.5	32.0	8.3	30.0
9.5	19.0	7.1	45.0	7.9	91.0
7.9	25.0				
6.7	32.0				
6.5	39.5				

Appendix 29: Lock-Cycle Process Flow Diagram



Appendix 30: Metallurgical Test Data for Bench-Scale Lock-Cycle Water Testing (Test 18-80)

	Cycle 1 (4/17)		**Cycle 2 (4/17)**		**Cycle 3 (4/19)**		**Cycle 4 (4/19)**		**Cycle 5 (4/23)**		**Cycle 6 (4/23)**	
Stage	Amine (#/LT)	Time (Min)	Amine (#/LT)	Time (Min)	Amine (#/LT)	Time (Min)	Amine (#/LT)	Time (Min)	Amine (#/LT)	Time (Min)	Amine (#/LT)	Time (Min)
1	0.05	4.0	0.00	4.5	0.00	4.5	0.00	5.0	0.00	1.0	0.00	3.0
2	0.05	4.0	0.05	4.0	0.05	3.5	0.05	5.0	0.05	2.5	0.05	4.0
3	0.05	3.5	0.05	2.5	0.05	2.5	0.05	3.5	0.05	3.5	0.05	3.0
4	0.10	3.0	0.10	2.5	0.10	2.5	0.10	2.5	0.10	4.0	0.10	2.5
5	0.10	2.5	0.10	2.0	0.10	2.0	0.10	2.0	0.10	2.5	0.10	2.0
6	0.10	2.0	0.10	1.5	0.10	1.0	0.00	1.5	0.10	2.0	0.10	1.5
Total	**0.45**	**19.0**	**0.40**	**17.0**	**0.40**	**16.0**	**0.30**	**19.5**	**0.40**	**15.5**	**0.40**	**16.0**

Cycle	**Water Source**	**Ca^{++} (ppm)**	**Mg^{++} (ppm)**	**Total Hardness as $CaCO_3$ (ppm)**	**Chem Lab pH**	**K25**
0	Tap	91.3	31.0	356	8.04	0.718
1	Tap	84.2	30.0	334	8.31	0.742
2	Recycle Water from Cycle 1	83.4	27.8	323	8.19	0.828
3	Recycle Water from Cycle 2	82.4	25.1	309	7.40	0.950
4	Recycle Water from Cycle 3	82.9	22.3	299	7.64	1.030
5	Recycle Water from Cycle 4	N/A	N/A	N/A	N/A	N/A
6	Recycle Water from Cycle 5	83.6	23.4	305	8.09	1.048

Cycle	Total Fe (%)	SiO_2 (%)	Phos (%)	Weight Dist (%)	Total Fe Dist (%)
0	N/A	N/A	N/A	N/A	N/A
1	62.64	2.36	0.033	39.9	59.5
2	62.68	2.31	0.034	38.8	57.9
3	62.75	2.21	0.034	37.2	55.5
4	62.67	2.31	0.034	38.7	57.8
5	62.25	2.75	0.034	40.6	60.2
6	62.59	2.26	0.035	38.8	57.8
Average	**62.60**	**2.37**	**0.034**	**39.0**	**58.1**
Std. Dev.	**0.16**	**0.18**	**0.001**	**1.1**	**1.5**

Appendix 31: Bench-Scale Residual Amine Testing: Metallurgical Balance and Removal Rate Calculations

Work Order	BF Number	Description											
18-82	18-74	NatFlot with 9-min RM grind on bulk 19-2 for makeup water measurements and XRD											
Product	**Wt (g)**	**Wt (%)**	**Total Fe (%)**	**SiO2 (%)**		**Total Fe Dist (%)**	**SiO2 Dist (%)**		**Total Fe Units**	**SiO2 Units**			**Amine (#/LT)**	**Time (min)**
F1	127.4	22.1	20.07	68.02		10.5	45.3		4.43	15.01			0.05	4.0
F2	79.1	13.7	22.37	64.14		7.3	26.5		3.07	8.79			0.05	4.0
F3	45.5	7.9	29.42	52.67		5.5	12.5		2.32	4.15			0.05	3.5
F4	49.2	8.5	38.88	35.5		7.8	9.1		3.31	3.03			0.10	3.0
F5	31.5	5.5	50.34	17.26		6.5	2.8		2.75	0.94			0.10	2.5
F6	16.9	2.9	55.93	10.33		3.9	0.9		1.64	0.30			0.10	1.5
Con	227.6	39.4	62.67	2.34		58.5	2.8		24.71	0.92				
Head	577.2	100.0	42.23	33.15		100.0	100.0		42.23	33.15				
Combined Products		**Wt (%)**	**Total Fe (%)**	**SiO2 (%)**		**Total Fe Dist (%)**	**SiO2 Dist (%)**		**Total Fe Units**	**SiO2 Units**			**Amine (#/LT)**	**Time (min)**
Con		39.4	62.67	2.34		58.5	2.8		24.71	0.92			0.45	18.5
Con+F6		42.4	62.20	2.89		62.4	3.7		26.35	1.23			0.35	17.0
Con+F5-6		47.8	60.85	4.53		68.9	6.5		29.10	2.17			0.25	14.5
Con+F4-6		56.3	57.53	9.22		76.8	15.7		32.41	5.19			0.15	11.5
Con+F3-6		64.2	54.08	14.55		82.2	28.2		34.73	9.35			0.10	8.0
Con+F2-6		77.9	48.50	23.27		89.5	54.7		37.80	18.13			0.05	4.0
Con+F1-6		100.0	42.23	33.15		100.0	100.0		42.23	33.15			0.00	0.0
Interp:		**Wt (%)**	**Total Fe (%)**	**SiO2 (%)**		**Total Fe Dist (%)**	**SiO2 Dist (%)**						**Amine (#/LT)**	**Time (min)**
4.00	SiO2	46.0	61.29	4.00		66.8	5.6						0.28	15.3

WO18-82											
				Inputs						
Stage	Amount of Float Makeup Water Added (mL)	Amount of Product Rinse Water Added (mL)		Total Solids	579.0	g				
Stage 1	375			Solids SpG	3.5839	g/cc				
Stage 2	300			Amine Dosage	0.45	#/LT	-->	0.000201083	g amine/g solids	
Stage 3	125			Amine Solution	1.00	%				
Stage 4	125			Filter Cake Moisture Content	10.00	%				
Stage 5	150									
Stage 6	100			***Calculations***						
Total	1175	1175		Total Amine Used	0.1164	g				
				Solids	161.6	mL				
				Float Feed Water	1838.4	mL				
				Froth Makeup Water	1175.0	mL				
				Amine Solution	11.6	mL				
				Product Rinse Water	1175.0	mL				
				Total Water In	**4200.1**	**mL**				
				Water Remaining in Filter Cakes	16.2	mL				
				Total Water Out	**16.2**	**mL**				
				Outputs						
				Chem in	116.4	mg				
				Water in - water out	4.2	L				
				Amine Concentration In	27.8	mg/L				
				Amine Concentration Out	1.1	mg/L				
				Removal Rate	96.0	%				

Appendix 32: Metallurgical Test Data for Selective Flocculation Tests with Bulk 19-2 Samples (Test 17-68 and Test 18-82)

						Desliming		Rougher Flotation			
Work Order	BF Number	Description	Feed Matr'l	Head Fe (Direct)	Head Fe (Calc)	Slime %Wt	Slime %Fe	Conc %Wt	Conc %Fe	Conc %SiO2	Conc %FeRec
18-68	18-64	SFDF with 9-min RM grind on 19-2 HPGR 6M U/S	WO18-57 19-2 HPGR 6M U/S	42.01	42.53	9.3	19.5	43.5	62.24	2.97	63.7
18-82	18-74	NatFlot with 9-min RM grind on 19-2 HPGR 6M U/S for makeup water measurements and XRD	WO18-57 19-2 HPGR 6M U/S	42.01	42.23			39.4	62.67	2.34	58.5

	Test Conditions															
Work Order	Flowsheet	% -25um	Process Water pH (Batch Lab)	Mill Discharge pH	Flot Feed pH	Water Used	Ca++ in water ppm	Mg++ in water ppm	Hardness as CaCO3 (ppm)	pH (Chem Lab)	K25	Special Reagent Type	Reagent Injection Location	Special Reagent Amount (#/LT)	Starch #/LT	Total Amine #/LT
18-68	SFDF Rougher			10.75	10.51	1:25 Tap:DI			0			Sodium Silicate; NaOH	Mill	Sodium Silicate (0.75 #/LT); NaOH (1.5 #/LT)	1.5	0.50
18-82	NatFlot Rougher			8.07	8.18	Coleraine Tap			0						1.5	0.35

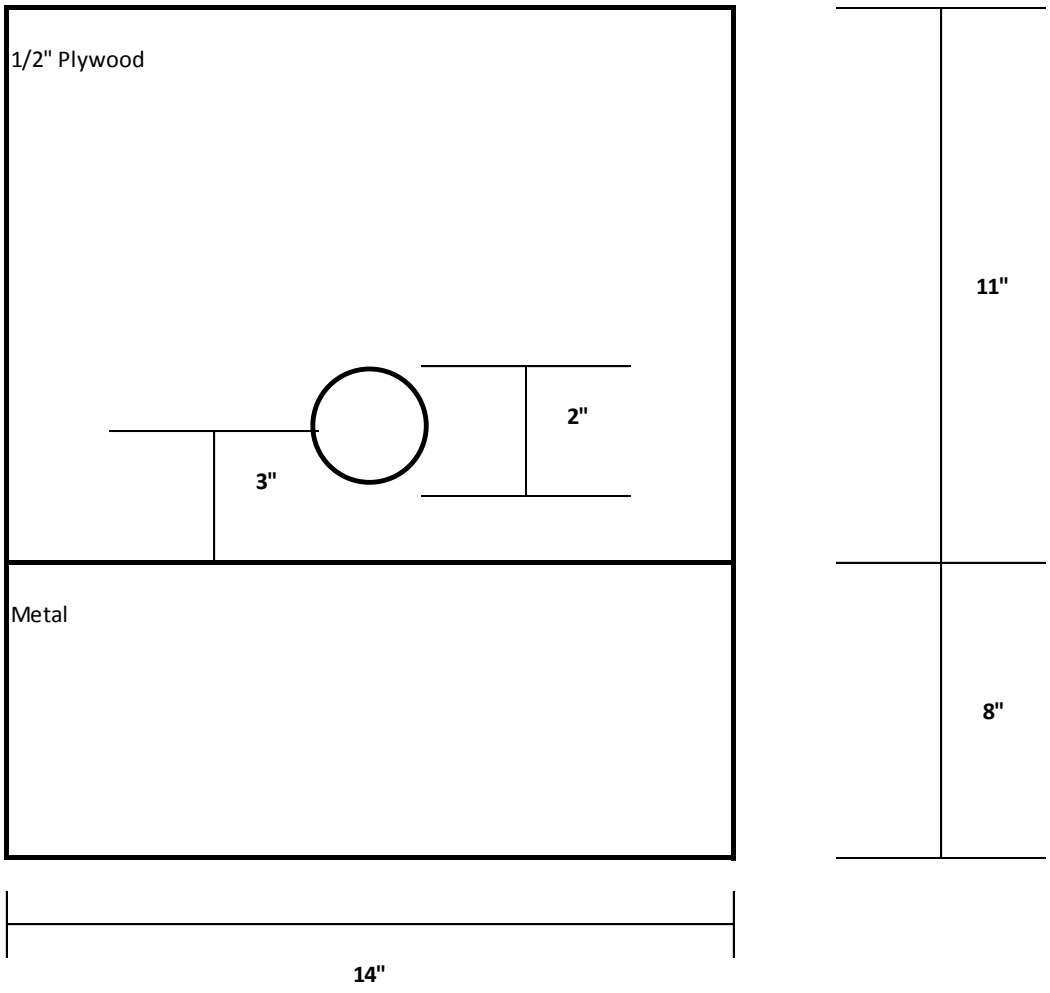
	Product Weights										
Work Order	Slime g	1st Froth g	2nd Froth g	3rd Froth g	4th Froth g	5th Froth g	6th Froth g	7th Froth g	8th Froth g	Conc g	Total Feed g
18-68	53.8	132.9	68.8	32.7	23.8	14.2				251.5	577.7
18-82		127.4	79.1	45.5	49.2	31.5	16.9			227.6	577.2

	Product Chemical Analyses									
Slime %Fe	1st Froth %Fe	2nd Froth %Fe	3rd Froth %Fe	4th Froth %Fe	5th Froth %Fe	6th Froth %Fe	7th Froth %Fe	8th Froth %Fe	Conc %Fe
19.47	18.27	28.76	44.70	51.59	54.35				62.24
	20.07	22.37	29.42	38.88	50.34	55.93			62.67

	%SiO2										Other Assays of Interest
Work Order	Deslime %SiO2	1 St Froth %SiO2	2 St Froth %SiO2	3 St Froth %SiO2	4 St Froth %SiO2	5 St Froth %SiO2	6 St Froth %SiO2	7 St Froth %SiO2	8 St Froth %SiO2	Conc %SiO2	
18-68					16.02	12.67				2.97	Con Phos: 0.035%
18-82		68.02	64.14	52.67	35.5	17.26	10.33			2.34	Con Phos: 0.033%; F1 phos: 0.018%, F2 phos: 0.021%, F3 phos: 0.024%, F4 phos: 0.033%, F5 phos: 0.042%, F6 phos: 0.042%

	Amine Additions										Flot Times								
Work Order	Type	1st Stage Amine (#/LT)	2nd Stage Amine (#/LT)	3rd Stage Amine (#/LT)	4th Stage Amine (#/LT)	5th Stage Amine (#/LT)	6th Stage Amine (#/LT)	7th Stage Amine (#/LT)	8th Stage Amine (#/LT)	Total Amine (#/LT)	1st Stage (min)	2nd Stage (min)	3rd Stage (min)	4th Stage (min)	5th Stage (min)	6th Stage (min)	7th Stage (min)	8th Stage (min)	Total Flot Time (min)
18-68	PA-12	0.1	0.1	0.1	0.1	0.1				0.50	2.0	2.5	2.0	2.0	1.5				10.0
18-82	Clariant Flotigam EDA	0.05	0.05	0.05	0.1	0.1				0.35	4.0	4.0	3.5	3.0	2.5	1.5			18.5

Appendix 33: Pilot-Scale Flotation Cell Divider Design



Appendix 34: Metallurgical Sampling Plan for Pilot Testing

NRRI Increasing Iron Recovery Pilot Plant
Sampling Plan

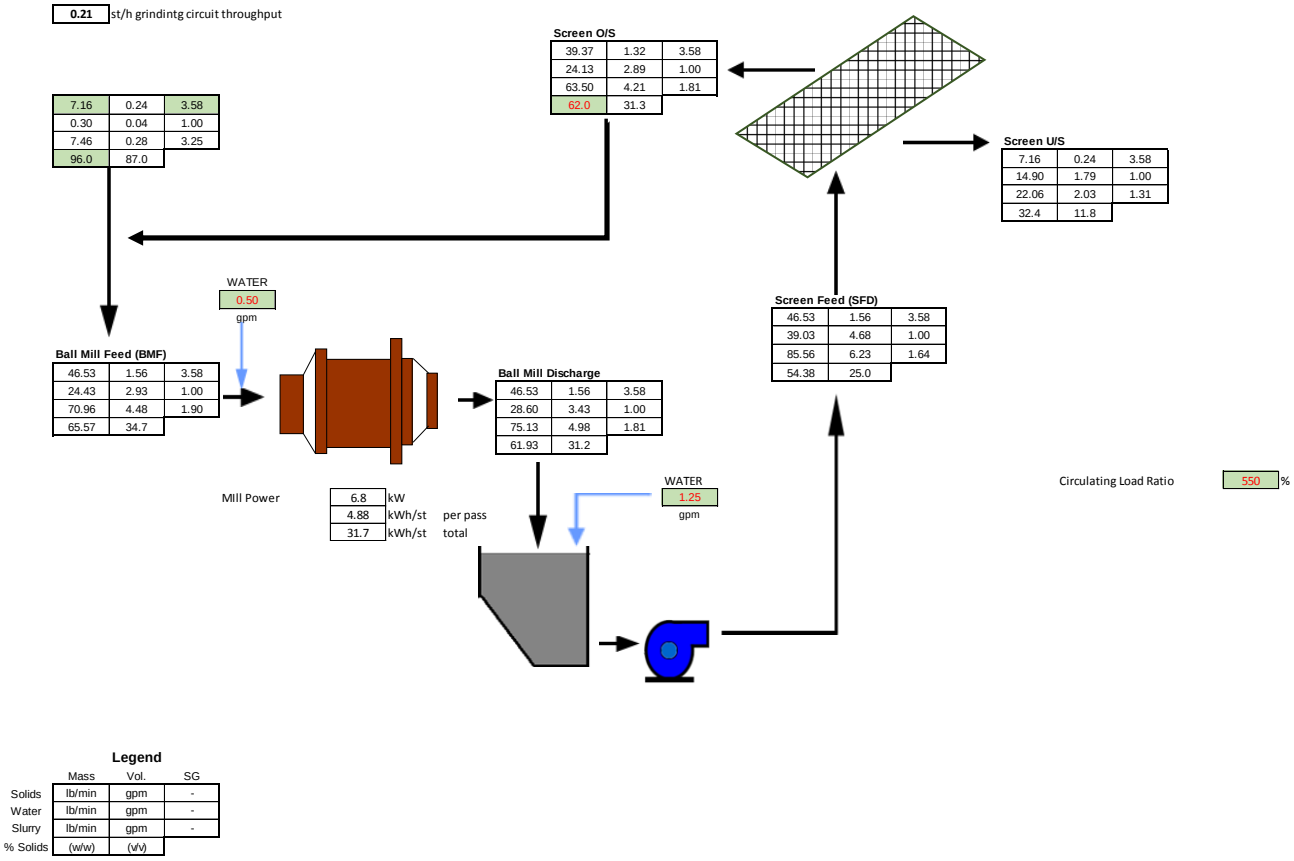
6/26/2018

Sample Points

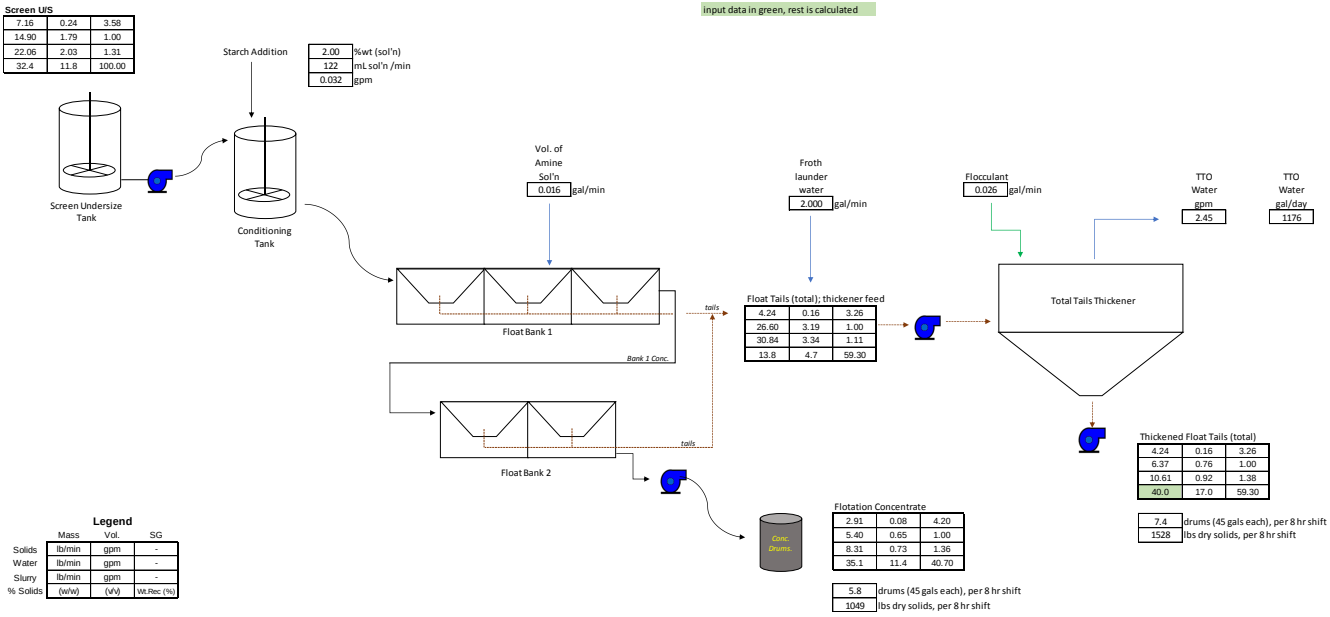
Circuit Feed	Screen Oversize	Ball Mill Discharge	Screen Undersize	Float Feed	Float Conc (U/F)	Float Tails (O/F)
% Solids	% Solids	% Solids	% Solids	% Solids	% Solids	% Solids
SG	SG	SG	SG	SG	SG	SG
Size Dist.	Size Dist.	Size Dist.	Size Dist.	Size Dist.	Size Dist.	Size Dist.
% Total Fe				% Total Fe	% Total Fe	% Total Fe
% SiO2				% SiO2	% SiO2	% SiO2
				% Phos	% Phos	% Phos

Appendix 35: Ore and Water Balance for Pilot-Scale Testing

Grinding circuit:



Flotation circuit:



Appendix 36: Metallurgical Balances for Pilot Runs 2-6

Run	Product	% Solids	Specific Gravity	Size (P80, µm)	Weight Dist (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
2	Circuit Feed	100.0	3.6175	1688	100.0	42.19	30.66	N/A	100.0	100.0	100.0
	Ball Mill Discharge	62.9	3.6770	91	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Screen Oversize	61.1	3.6537	103	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Screen Undersize	32.2	3.6389	33	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Flot Feed	30.9	3.6368	32	100.0	41.88	31.75	0.025	100.0	100.0	100.0
	Flot Conc (U/F)	18.9	4.2880	30	39.9	58.64	7.25	0.033	55.9	9.1	52.7
	Flot Tails (O/F)	8.9	3.3214	23	60.1	30.73	47.37	0.022	44.1	90.9	47.3

Run	Product	% Solids	Specific Gravity	Size (P80, µm)	Weight Dist (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
3	Circuit Feed	98.2	3.6380	1647	100.0	42.40	30.26	N/A	100.0	100.0	100.0
	Ball Mill Discharge	60.5	3.6953	97	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Screen Oversize	65.0	3.7199	110	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Screen Undersize	33.0	3.6529	33	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Flot Feed	32.2	3.6514	33	100.0	41.87	30.84	0.027	100.0	100.0	100.0
	Flot Conc (U/F)	17.3	4.3238	37	39.6	59.60	6.29	0.032	56.3	8.1	46.9
	Flot Tails (O/F)	11.1	3.3082	27	60.4	30.26	47.43	0.022	43.7	91.9	53.1

Run	Product	% Solids	Specific Gravity	Size (P80, µm)	Weight Dist (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
4	Circuit Feed	N/A	N/A	N/A	100.0	43.19	30.79	N/A	100.0	100.0	100.0
	Ball Mill Discharge	61.7	3.6489	84	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Screen Oversize	62.8	3.7010	102	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Screen Undersize	N/A	3.6313	34	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Flot Feed	31.7	3.6386	32	100.0	41.78	32.85	0.027	100.0	100.0	100.0
	Flot Conc (U/F)	10.7	4.3841	43	20.5	61.62	3.91	0.033	30.3	2.4	25.1
	Flot Tails (O/F)	13.3	3.4759	27	79.5	36.65	40.42	0.027	69.7	97.6	74.9

Run	Product	% Solids	Specific Gravity	Size (P80, µm)	Weight Dist (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
5	Circuit Feed	81.9	3.6312	1657	100.0	42.52	31.48	N/A	100.0	100.0	100.0
	Ball Mill Discharge	49.1	3.6925	94	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Screen Oversize	79.9	3.7208	106	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Screen Undersize	34.0	3.6564	38	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Flot Feed	34.0	3.6758	37	100.0	41.76	32.10	0.026	100.0	100.0	100.0
	Flot Conc (U/F)	N/A	4.3184	43	33.4	59.73	6.06	0.031	47.8	6.3	39.8
	Flot Tails (O/F)	18.9	3.3840	31	66.6	32.74	45.41	0.023	52.2	93.7	60.2

Run	Product	% Solids	Specific Gravity	Size (P80, µm)	Weight Dist (%)	Total Fe (%)	SiO_2 (%)	Phos (%)	Total Fe Dist (%)	SiO_2 Dist (%)	Phos Dist (%)
6	Circuit Feed	97.5	3.6248	1877	100.0	43.09	31.84	N/A	100.0	100.0	100.0
	Ball Mill Discharge	65.4	3.6822	88	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Screen Oversize	68.3	3.6858	100	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Screen Undersize	38.0	3.6305	32	N/A	N/A	N/A	N/A	N/A	N/A	N/A
	Flot Feed	34.6	3.6330	31	100.0	41.83	33.40	0.027	100.0	100.0	100.0
	Flot Conc (U/F)	17.9	4.3697	45	27.9	61.40	4.15	0.032	40.9	3.5	33.0
	Flot Tails (O/F)	17.3	3.4177	26	72.1	34.27	45.15	0.024	59.1	96.5	67.0

Appendix 37: Operator Sheet Run Summary

Grinding Circuit and Water Consumption Summary

Pilot Run #	**1**	**2a**	**2b**	**2 composite**	**3a**	**3b**	**3 composite**	**4**	**5**	**6**
Date (2018)	*20-Jun*	*22-Jun*	*22-Jun*	*22-Jun*	*27-Jun*	*27-Jun*	*27-Jun*	*9-Jul*	*17-Jul*	*20-Jul*
Run Start	10:52 AM	8:20 AM	9:45 AM	8:20 AM	8:50 AM	11:45 AM	8:50 AM		8:14 AM	8:18 AM
Run End	2:55 PM	9:45 AM	12:15 PM	12:15 PM	11:45 AM	1:49 PM	1:49 PM		2:15 PM	3:15 PM
Run Time (hrs)	4.05	1.42	2.50	3.92	2.92	2.07	4.98	0.00	6.02	6.95
Target Feed Rate (wet), lb/min	7.50	7.50	9.00	N/A	9.00	9.00	9.00	9.00	9.00	8.10
Measured Feed Rate (wet), lb/min	7.54	7.45	9.24	8.59	8.22	10.47	9.15	8.68	8.81	8.42
Dry Feed Rate (4% moist), LTPH	0.202	0.200	0.248	0.230	0.220	0.280	0.245	0.233	0.236	0.226
Mill Speed (%Cs)	111	75	75	75	70	70	70	70	70	66
Mill Motor Input Power (kW)	9.22	7.38	7.38	7.38	6.99	6.99	6.99	7.02	7.01	6.49
Estimated Pinion Power (kW)	8.32	6.66	6.66	6.66	6.31	6.31	6.31	6.34	6.33	5.86
Estimated Mill Specific Energy (kWh/LT)	41.2	33.4	26.9	28.9	28.7	22.5	25.7	27.2	26.8	26.0
Circulating Load Ratio (by mass balance)	N/A	-	-	300	-	-	315	N/A	275	360
Target Mill Discharge % Solids (w/w)	65	65	65	65	62	62	62	62	62	65
Measured Mill Discharge % Solids (w/w), Marcy	N/A	63.3	64	-	60.9	62.8	-	61.3	63	66
Measured Mill Discharge % Solids (w/w), Lab	N/A	-	-	62.9			60.5	61.7	49.0	65.4
F80, circuit (µm)	646	-	-	1688	-	-	1647	N/A	1657	1877
P80, circuit (µm)	39	-	-	33	-	-	33	34	38	32
WIo, kWh/LT	34	-	-	19	-	-	17	-	19	17
Approx Wi_{EFF}, (%), using Bwi = 17 kWh/LT	50	-	-	88	-	-	99	-	87	101
Total Water Consumtion (gallons)	587	-	-	1568	-	-	2077		1586	1568
Water Consumption Rate (gal/min)	2.42	-	-	6.67	-	-	6.95		4.39	3.76
Water Consumption Rate (gal/LT_{ore})	718	-	-	1739	-	-	1700		1117	1000

Flotation Circuit Summary

	Field Measurements			**Flotation Reagent Targets**							**Field Measurements**						
Run	**Av. Float Feed Density (Marcy Cup)**	**Av. Float Feed Temperature (°C)**	**Av. Float Feed pH**	**Causticized Modified Starch (#/LT)**	**Ro. Flot 1 (#/LT)**	**Ro. Flot 2 (#/LT)**	**Ro. Flot 3 (#/LT)**	**Cl. Flot 1 (#/LT)**	**Cl. Flot 2 (#/LT)**	**Total Amine (#/LT)**	**Causticized Modified Starch (#/LT)**	**Ro. Flot 1 (#/LT)**	**Ro. Flot 2 (#/LT)**	**Ro. Flot 3 (#/LT)**	**Cl. Flot 1 (#/LT)**	**Cl. Flot 2 (#/LT)**	**Total Amine (#/LT)**
1	N/A	N/A	N/A	1.50	0.05	0.05	0.05	0.10	0.10	0.35	N/A	N/A	N/A	N/A	N/A	N/A	0.00
2 (early-morning)	28.5	N/A	N/A	1.50	0.05	0.05	0.05	0.10	0.10	0.35	N/A	N/A	N/A	N/A	N/A	N/A	0.00
2 (mid-morning)	32.0	N/A	8.29	1.50	0.05	0.05	0.05	0.10	0.10	0.35	1.34	0.06	0.05	0.04	0.08	0.09	0.32
3 (morning)	29.2	22.0	7.94	1.50	0.05	0.05	0.05	0.10	0.10	0.35	1.60	0.05	0.05	0.05	0.11	0.10	0.36
3 (afternoon)	29.7	22.0	8.03	1.50	0.05	0.05	0.05	0.15	0.15	0.45	1.28	0.04	0.04	0.05	0.12	0.13	0.37
4	29.9	22.0	8.26	1.50	0.05	0.05	0.05	0.15	0.15	0.45	1.56	0.05	0.05	0.05	0.16	0.16	0.47
5	31.5	22.0	8.38	1.50	0.03	0.03	0.05	0.10	0.14	0.35	1.54	0.03	0.04	0.05	0.11	0.16	0.39
6	35.5	22.0	8.37	1.50	0.03	0.03	0.05	0.10	0.14	0.35	1.46	0.03	0.03	0.05	0.09	0.15	0.36

Appendix 38: Pilot-Scale Run 2 Summary

Date	22-Jun-18
Technician	Joe Cannella
Engineer	Shashi Rao

Stream	Total Fe, %	Silica, %	Phos, %	Tare Wt	Wet Wt	Dry Wt	% Solids	SG	Wt Rec	Fe Rec	SiO2 Rec	Phos Rec
Circuit Feed	42.19	30.66	N/A	426.8	1307.0	880.2	100.0	3.6175	100.0	100.0	100.0	100.0
Ball Mill Discharge	N/A	N/A	N/A	426.6	4683.5	2677.6	62.9	3.6770	N/A	N/A	N/A	N/A
Screen Oversize	N/A	N/A	N/A	374.7	5316.0	3019.1	61.1	3.6537	N/A	N/A	N/A	N/A
Screen Undersize	N/A	N/A	N/A	427.6	4752.0	1394.2	32.2	3.6389	N/A	N/A	N/A	N/A
Flot Feed	41.88	31.75	0.025	427.6	3226.2	865.9	30.9	3.6368	100.0	100.0	100.0	100.0
Flot Conc	58.64	7.25	0.033	427.8	2210.3	337.5	18.9	4.2880	39.9	55.9	9.1	52.7
Flot Tails	30.73	47.37	0.022	427.0	4057.0	323.9	8.9	3.3214	60.1	44.1	90.9	47.3

Data Not Available

Cleaner U/F 11:08am (grab)	59.64	6.65
Cleaner U/F 1:05pm (grab)	58.15	8.38

			Circuit Feed			Ball Mill Discharge			Screen Oversize			Screen Undersize			Flot Feed			Flot Conc			Tails Thickener Feed		
Inch/Tyler Mesh	(mm)	(µm)	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing
6	3.400	3400		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
8	2.400	2400	13.2	4.4	95.6		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
10	2.000	2000	17.6	5.9	89.6		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
14	1.420	1420	53.4	17.9	71.7		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
20	0.850	850	21.3	7.2	64.5	0.3	0.1	99.9		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
28	0.600	600	23	7.7	56.8	0.2	0.1	99.8	0.2	0.1	99.9		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
35	0.425	425	19.6	6.6	50.2	0.4	0.2	99.6	0.4	0.2	99.7		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
48	0.300	300	16.2	5.4	44.8	1.4	0.6	99.0	1.9	0.9	98.9		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
65	0.212	212	13.6	4.6	40.2	3.7	1.6	97.3	4.8	2.2	96.7		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
100	0.150	150	10.9	3.7	36.5	8.4	3.7	93.6	11.2	5.1	91.7		0.0	100.0	0.2	0.1	99.9		0.0	100.0		0.0	100.0
150	0.106	106	9.4	3.2	33.4	16.1	7.1	86.5	21.8	9.8	81.8	0.2	0.1	99.9	0.2	0.1	99.9	0.1	0.1	99.9	0.1	0.1	99.9
200	0.075	75	9.9	3.3	30.1	30.8	13.7	72.8	41.2	18.6	63.2	0.6	0.3	99.6	1	0.3	99.6	0.3	0.2	99.7	0.1	0.1	99.9
270	0.053	53	10.3	3.5	26.6	46.6	20.7	52.1	54.8	24.7	38.5	5.4	2.4	97.2	8.6	2.3	97.3	2.3	1.7	98.0	0.2	0.1	99.7
325	0.045	45	7.5	2.5	24.1	17.2	7.6	44.4	21.6	9.7	28.7	10	4.5	92.7	16.3	4.4	92.9	4.3	3.2	94.8	2.4	1.6	98.1
400	0.038	38	6.6	2.2	21.8	14.1	6.3	38.2	11.6	5.2	23.5	17.6	7.9	84.8	20.5	5.5	87.4	8.9	6.6	88.1	0.2	0.1	97.9
500	0.025	25	11.1	3.7	18.1	15.5	6.9	31.3	11.5	5.2	18.3	28.7	12.9	71.8	59	15.9	71.5	18.8	14.0	74.1	17.1	11.7	86.2
-500	-0.025	-25	53.9	18.1	0.0	70.5	31.3	0.0	40.6	18.3	0.0	159.4	71.8	0.0	265.8	71.5	0.0	99.4	74.1	0.0	126	86.2	0.0
			297.5			225.2			221.6			221.9			371.6			134.1			146.1		
				P80, µm	1688		P80, µm	91		P80, µm	103		P80, µm	33		P80, µm	32		P80, µm	30		P80, µm	23

Appendix 39: Pilot-Scale Run 3 Summary

Date	27-Jun-18
Technician	Joe Cannella
Engineer	Shashi Rao

Data Not Available

Stream	Total Fe, %	Silica, %	Phos, %	Tare Wt	Wet Wt	Dry Wt	% Solids	SG	Wt Rec	Fe Rec	SiO2 Rec	Phos Rec
Circuit Feed	42.40	30.26	N/A	428.5	1590.0	1140.6	98.2	3.6380	100.0	100.0	100.0	100.0
Ball Mill Discharge	N/A	N/A	N/A	430.9	3841.6	2063.4	60.5	3.6953	N/A	N/A	N/A	N/A
Screen Oversize	N/A	N/A	N/A	430.5	3530.8	2015.9	65.0	3.7199	N/A	N/A	N/A	N/A
Screen Undersize	N/A	N/A	N/A	444.1	2986.4	838.9	33.0	3.6529	N/A	N/A	N/A	N/A
Flot Feed	41.87	30.84	0.027	447.1	2986.4	818.4	32.2	3.6514	100.0	100.0	100.0	100.0
Flot Conc	59.60	6.29	0.032	450.9	2452.7	346.4	17.3	4.3238	39.6	56.3	8.1	46.9
Flot Tails	30.26	47.43	0.022	444.4	6981.7	726.8	11.1	3.3082	60.4	43.7	91.9	53.1

Cleaner U/F 10:10am (grab)	59.84	6.37	Total of 0.35 #/LT Amine (based on 9 #/min feed)
Cleaner U/F 11:56am (grab)	58.41	8.00	Total of 0.35 #/LT Amine (based on 9 #/min feed)
Cleaner U/F 1:10pm (grab)	60.53	5.57	Total of 0.45 #/LT Amine (based on 9 #/min feed)
Cleaner U/F 1:45pm (grab)	59.63	6.34	Total of 0.45 #/LT Amine (based on 9 #/min feed)

			Circuit Feed			Ball Mill Discharge			Screen Oversize			Screen Undersize			Flot Feed			Flot Conc			Tails Thickener Feed		
Inch/Tyler Mesh	(mm)	(µm)	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing
6	3.400	3400		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
8	2.400	2400	8.4	4.1	95.9		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
10	2.000	2000	12.4	6.0	89.9		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
14	1.420	1420	33.6	16.3	73.6		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
20	0.850	850	15.9	7.7	65.9	0.7	0.4	99.6	1	0.5	99.5		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
28	0.600	600	15.8	7.7	58.3	0.3	0.2	99.5	0.5	0.2	99.3		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
35	0.425	425	13.8	6.7	51.6	0.5	0.3	99.2	1	0.5	98.8		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
48	0.300	300	11.5	5.6	46.0	1.5	0.8	98.4	2.4	1.1	97.7		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
65	0.212	212	9.6	4.7	41.3	3.7	1.9	96.5	5.5	2.5	95.2		0.0	100.0		0.0	100.0		0.0	100.0	0.2	0.1	99.9
100	0.150	150	7.8	3.8	37.6	8	4.2	92.3	11.8	5.5	89.7		0.0	100.0		0.0	100.0		0.0	100.0	0.2	0.1	99.8
150	0.106	106	6.6	3.2	34.4	15.5	8.1	84.3	23	10.7	79.0		0.0	100.0		0.0	100.0		0.0	100.0	0.2	0.1	99.6
200	0.075	75	7.4	3.6	30.8	29.8	15.5	68.8	45.4	21.0	58.0	0.4	0.2	99.8	0.4	0.2	99.8	0.5	0.3	99.7	0.5	0.3	99.3
270	0.053	53	7.4	3.6	27.2	38.1	19.8	48.9	60.3	28.0	30.0	4.5	2.6	97.2	4.5	2.6	97.1	4.7	2.8	96.9	2.8	1.7	97.7
325	0.045	45	5.3	2.6	24.6	21.1	11.0	37.9	16.8	7.8	22.3	8.8	5.1	92.1	8.5	5.0	92.2	13.8	8.2	88.6	5.6	3.3	94.3
400	0.038	38	3.2	1.6	23.1	9.4	4.9	33.0	9.4	4.4	17.9	10.9	6.3	85.8	13.5	7.9	84.3	12.7	7.6	81.1	9.8	5.9	88.5
500	0.025	25	9.4	4.6	18.5	13.1	6.8	26.2	8.1	3.8	14.1	25.4	14.7	71.1	20.8	12.1	72.2	38.3	22.9	58.2	16	9.6	78.9
-500	-0.025	-25	38.2	18.5	0.0	50.3	26.2	0.0	30.5	14.1	0.0	122.8	71.1	0.0	123.8	72.2	0.0	97.4	58.2	0.0	131.9	78.9	0.0
			206.3			192			215.7			172.8			171.5			167.4			167.2		
				P80, µm	1647		P80, µm	97		P80, µm	110		P80, µm	33		P80, µm	33		P80, µm	37		P80, µm	27

Appendix 40: Pilot-Scale Run 4 Summary

Date	2-Jul-18
Technician	Joe Cannella
Engineer	Shashi Rao

Data Not Available

Stream	Total Fe, %	Silica, %	Phos, %	Tare Wt	Wet Wt	Dry Wt	% Solids	SG
Circuit Feed	43.19	30.79	N/A	N/A	N/A	N/A	#VALUE!	N/A
Ball Mill Discharge	N/A	N/A	N/A	833.2	5580.1	2930.7	61.7	3.6489
Screen Oversize	N/A	N/A	N/A	834.1	4015.0	1997.5	62.8	3.7010
Screen Undersize	N/A	N/A	N/A	N/A	N/A	1253.4	#VALUE!	3.6313
Flot Feed	41.78	32.85	0.027	869.9	2754.2	596.4	31.7	3.6386
Flot Conc	61.62	3.91	0.033	864.4	3286.9	259.1	10.7	4.3841
Flot Tails	36.65	40.42	0.027	812.5	4215.6	453.8	13.3	3.4759

Wt Rec	Fe Rec	SiO2 Rec	Phos Rec
100.0	100.0	100.0	100.0
N/A	N/A	N/A	N/A
N/A	N/A	N/A	N/A
N/A	N/A	N/A	N/A
100.0	100.0	100.0	100.0
20.5	30.3	2.4	25.1
79.5	69.7	97.6	74.9

Cleaner U/F 10:00am (grab)	61.65	3.54
Cleaner U/F 11:00am (grab)	59.40	5.99
Cleaner U/F 12:40pm (grab)	60.66	5.16
Cleaner U/F 1:55pm (grab)	61.05	4.34

Stream	Total Fe, %	Silica, %	Phos, %
Flot Feed (afternoon)	41.98	32.54	0.027
Flot Conc (afternoon)	60.65	5.00	0.033
Flot Tails (afternoon)	33.57	45.67	0.025

Wt Rec	Fe Rec	SiO2 Rec	Phos Rec
100.0	100.0	100.0	100.0
31.1	44.9	4.8	38.0
68.9	55.1	95.2	62.0

Stream	Ca++	Mg++	Total Hardness (as CaCO3)
Process Water	96.0	34.6	382.2
Flot Feed	69.5	24.9	276.1
CL Flot U/F (Con)	73.9	19.8	266.1
Total Froth (Tails)	71.6	27.8	293.3
(11:00 am)			

			Circuit Feed			Ball Mill Discharge			Screen Oversize			Screen Undersize			Flot Feed			Flot Conc			Tails Thickener Feed		
Inch/Tyler Mesh	(mm)	(µm)	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing
6	3.400	3400	N/A	#VALUE!	#VALUE!		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
8	2.400	2400	N/A	#VALUE!	#VALUE!		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
10	2.000	2000	N/A	#VALUE!	#VALUE!		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
14	1.420	1420	N/A	#VALUE!	#VALUE!		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
20	0.850	850	N/A	#VALUE!	#VALUE!	0.4	0.2	99.8		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
28	0.600	600	N/A	#VALUE!	#VALUE!	0.2	0.1	99.6		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
35	0.425	425	N/A	#VALUE!	#VALUE!	0.2	0.1	99.5	0.6	0.4	99.6		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
48	0.300	300	N/A	#VALUE!	#VALUE!	1.5	0.9	98.6	1.4	0.8	98.8		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
65	0.212	212	N/A	#VALUE!	#VALUE!	1.6	1.0	97.6	3.6	2.2	96.6		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
100	0.150	150	N/A	#VALUE!	#VALUE!	4.4	2.7	95.0	7.7	4.6	92.0	0.3	0.2	99.8		0.0	100.0		0.0	100.0		0.0	100.0
150	0.106	106	N/A	#VALUE!	#VALUE!	9.6	5.8	89.2	16	9.6	82.4	0.2	0.1	99.7	0.1	0.1	99.9	0.1	0.1	99.9	0.2	0.1	99.9
200	0.075	75	N/A	#VALUE!	#VALUE!	21.6	13.1	76.1	32.8	19.7	62.6	0.7	0.4	99.3	0.5	0.3	99.6	0.7	0.5	99.5	0.4	0.2	99.7
270	0.053	53	N/A	#VALUE!	#VALUE!	35.7	21.6	54.5	39.5	23.8	38.8	4.7	2.7	96.6	4	2.4	97.3	8.6	5.9	93.6	3.2	1.8	97.8
325	0.045	45	N/A	#VALUE!	#VALUE!	12.6	7.6	46.8	20.5	12.3	26.5	8.9	5.2	91.4	8	4.7	92.6	14.6	10.0	83.6	5.1	2.9	94.9
400	0.038	38	N/A	#VALUE!	#VALUE!	11.8	7.1	39.7	8.3	5.0	21.5	12	7.0	84.4	11.8	7.0	85.6	17.1	11.7	71.9	9.3	5.4	89.5
500	0.025	25	N/A	#VALUE!	#VALUE!	11.7	7.1	32.6	8.4	5.1	16.4	23.6	13.7	70.6	21.9	12.9	72.7	30.1	20.6	51.3	19.9	11.4	78.1
-500	-0.025	-25	N/A	#VALUE!	#VALUE!	53.8	32.6	0.0	27.3	16.4	0.0	121.3	70.6	0.0	123.1	72.7	0.0	75.1	51.3	0.0	135.7	78.1	0.0
			0			165.1			166.1			171.7			169.4			146.3			173.8		
				P80, µm	#VALUE!		P80, µm	84		P80, µm	102		P80, µm	34		P80, µm	32		P80, µm	43		P80, µm	27

Appendix 41: Pilot-Scale Run 5 Summary

Date	17-Jul-18
Technician	Joe Cannella
Engineer	MAM

Data Not Available

Stream	Total Fe, %	Silica, %	Phos, %	Tare Wt	Wet Wt	Dry Wt	% Solids	SG
Circuit Feed	42.52	31.48	N/A	837.1	1861.2	838.5	81.9	3.6312
Ball Mill Discharge	N/A	N/A	N/A	859.6	5619.0	2334.8	49.1	3.6925
Screen Oversize	N/A	N/A	N/A	809.5	4564.2	2999.5	79.9	3.7208
Screen Undersize	N/A	N/A	N/A	692.9	3032.3	795.2	34.0	3.6564
Flot Feed	41.76	32.10	0.026	1034.8	3413.5	809.9	34.0	3.6758
Flot Conc	59.73	6.06	0.031	823.4	2415.1	N/A	#VALUE!	4.3184
Flot Tails	32.74	45.41	0.023	812.7	9723.4	1683.0	18.9	3.3840

Wt Rec	Fe Rec	SiO2 Rec	Phos Rec
100.0	100.0	100.0	100.0
N/A	N/A	N/A	N/A
N/A	N/A	N/A	N/A
N/A	N/A	N/A	N/A
100.0	100.0	100.0	100.0
33.4	47.8	6.3	39.8
66.6	52.2	93.7	60.2

	ICP		XRF	
	Total Fe, %	Silica, %	Total Fe, %	Silica, %
Cleaner U/F 9:45am (grab)	61.18	4.63	66.97	4.49
Cleaner U/F 11:20am (grab)	57.53	8.91	62.74	8.41
Cleaner U/F 12:30pm (grab)	60.05	5.86	65.72	5.28
Cleaner U/F 2:00pm (grab)	60.84	4.78	66.88	4.66

Stream	Ca++	Mg++	Total Hardness (as CaCO3)
Process Water	110.2	35.5	421.4
Flot Feed	78.9	19.6	277.7
Cl. Flot U/F (Con)	76.3	25.1	293.9
Total Froth (Tails)	90.6	28.0	341.5
(time?)			

			Circuit Feed			Ball Mill Discharge			Screen Oversize			Screen Undersize			Flot Feed			Flot Conc			Tails Thickener Feed		
Inch/Tyler Mesh	(mm)	(µm)	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing
6	3.400	3400		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
8	2.400	2400	13.8	4.7	95.3		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
10	2.000	2000	14.2	4.9	90.4		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
14	1.420	1420	51.6	17.6	72.8		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
20	0.850	850	23	7.9	64.9	0.6	0.3	99.7	1.4	0.5	99.5		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
28	0.600	600	23.1	7.9	57.0	0.2	0.1	99.6	0.7	0.3	99.2		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
35	0.425	425	19.7	6.7	50.3	0.5	0.2	99.4	1.3	0.5	98.7		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
48	0.300	300	16.4	5.6	44.7	1.5	0.7	98.7	3	1.1	97.6	0.1	0.1	99.9		0.0	100.0		0.0	100.0		0.0	100.0
65	0.212	212	13.5	4.6	40.1	3.9	1.8	97.0	6.7	2.5	95.2	0.6	0.4	99.6	0.7	0.4	99.6		0.0	100.0	1	0.6	99.4
100	0.150	150	10.9	3.7	36.4	8.6	3.9	93.1	13.8	5.1	90.1	1	0.6	99.0	0.2	0.1	99.4		0.0	100.0	1.4	0.8	98.6
150	0.106	106	9.1	3.1	33.3	16.8	7.6	85.6	27.2	10.0	80.0	0.7	0.4	98.6	0.4	0.2	99.2	0.3	0.2	99.8	1.7	1.0	97.6
200	0.075	75	9.4	3.2	30.0	33.1	14.9	70.7	56	20.7	59.4	1.1	0.7	97.9	1	0.6	98.6	1	0.6	99.2	1.4	0.8	96.8
270	0.053	53	11.3	3.9	26.2	51.6	23.2	47.5	75.1	27.7	31.7	7.3	4.4	93.6	5.7	3.5	95.0	12.2	7.7	91.5	3.8	2.2	94.5
325	0.045	45	5.9	2.0	24.2	17.8	8.0	39.5	21.8	8.0	23.6	9.8	5.8	87.7	8.8	5.5	89.6	15	9.5	82.0	5.7	3.3	91.2
400	0.038	38	6.3	2.2	22.0	12	5.4	34.1	12.9	4.8	18.9	12.5	7.4	80.3	13.6	8.5	81.1	14.2	9.0	73.0	9.7	5.7	85.5
500	0.025	25	10	3.4	18.6	14.9	6.7	27.4	11.7	4.3	14.6	22.5	13.4	66.9	19.7	12.2	68.9	33.2	21.0	52.0	16.5	9.7	75.8
-500	-0.025	-25	54.4	18.6	0.0	60.9	27.4	0.0	39.5	14.6	0.0	112.2	66.9	0.0	110.8	68.9	0.0	82.3	52.0	0.0	129.3	75.8	0.0
			292.6			222.4			271.1			167.8			160.9			158.2			170.5		
				P80, µm	1657		P80, µm	94		P80, µm	106		P80, µm	38		P80, µm	37		P80, µm	43		P80, µm	31

Appendix 42: Pilot-Scale Run 6 Summary

Date	20-Jul-18
Technician	Joe Cannella
Engineer	MAM

Data Not Available

Stream	Total Fe, %	Silica, %	Phos, %	Tare Wt	Wet Wt	Dry Wt	% Solids	SG	Wt Rec	Fe Rec	SiO2 Rec	Phos Rec
Circuit Feed	43.09	31.84	N/A	837.1	1861.2	998.5	97.5	3.6248	100.0	100.0	100.0	100.0
Ball Mill Discharge	N/A	N/A	N/A	859.6	5619.0	3112.4	65.4	3.6822	N/A	N/A	N/A	N/A
Screen Oversize	N/A	N/A	N/A	809.5	4564.2	2563.5	68.3	3.6858	N/A	N/A	N/A	N/A
Screen Undersize	N/A	N/A	N/A	692.9	3032.3	889.3	38.0	3.6305	N/A	N/A	N/A	N/A
Flot Feed	41.83	33.40	0.027	1034.8	3413.5	822.4	34.6	3.6330	100.0	100.0	100.0	100.0
Flot Conc	61.40	4.15	0.032	823.4	2415.1	284.4	17.9	4.3697	27.9	40.9	3.5	33.0
Flot Tails	34.27	45.15	0.024	812.7	9723.4	1544.4	17.3	3.4177	72.1	59.1	96.5	67.0

	ICP		XRF	
	Total Fe, %	Silica, %	Total Fe, %	Silica, %
Cleaner U/F 9:30am (grab)	61.61	3.83		
Cleaner U/F 10:10am (grab	59.68	5.71		
Cleaner U/F 12:40pm (grab	61.16	4.32		
Cleaner U/F 2:55pm (grab)	60.20	5.35		

Stream	Ca++	Mg++	Total Hardness (as CaCO3)
Process Water	115.5	36.8	439.9
Flot Feed	79.7	16.7	267.8
CL Flot U/F (Con)	85.6	28.3	330.3
Total Froth (Tails)	110.4	31.1	403.7

Note: cut samples in afternoon (2:45) when circuit was still stabilizing from losing rougher flotation level

			Circuit Feed			Ball Mill Discharge			Screen Oversize			Screen Undersize			Flot Feed			Flot Conc			Tails Thickener Feed		
Inch/Tyler Mesh	(mm)	(µm)	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing	Weight	Wt% Retained	Cum % Passing
6	3.400	3400		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
8	2.400	2400	9.8	4.1	95.9		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
10	2.000	2000	12.6	5.2	90.7		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
14	1.420	1420	40.9	17.0	73.7		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0		0.0	100.0
20	0.850	850	19	7.9	65.8	0.2	0.1	99.9	0.4	0.2	99.8	2	0.9	99.1		0.0	100.0		0.0	100.0		0.0	100.0
28	0.600	600	19	7.9	57.8	0.2	0.1	99.8	0.3	0.1	99.7	0.1	0.0	99.0		0.0	100.0		0.0	100.0		0.0	100.0
35	0.425	425	16.4	6.8	51.0	0.4	0.2	99.6	0.8	0.4	99.3		0.0	99.0		0.0	100.0		0.0	100.0		0.0	100.0
48	0.300	300	13.7	5.7	45.3	1.3	0.6	99.1	1.8	0.8	98.6		0.0	99.0		0.0	100.0		0.0	100.0		0.0	100.0
65	0.212	212	11.3	4.7	40.6	3.2	1.4	97.6	4.5	2.0	96.6		0.0	99.0		0.0	100.0		0.0	100.0		0.0	100.0
100	0.150	150	9.1	3.8	36.8	7.4	3.3	94.4	10.1	4.4	92.2		0.0	99.0		0.0	100.0		0.0	100.0		0.0	100.0
150	0.106	106	7.6	3.2	33.7	15	6.7	87.7	20.2	8.9	83.3	0.1	0.0	99.0		0.0	100.0	0.1	0.0	100.0		0.0	100.0
200	0.075	75	7.9	3.3	30.4	30	13.3	74.4	41.4	18.1	65.1	0.3	0.1	98.9	0.4	0.1	99.9	1.2	0.5	99.5	0.3	0.1	99.9
270	0.053	53	9.3	3.9	26.5	50.2	22.3	52.1	63.1	27.7	37.5	4.9	2.2	96.6	4.7	1.4	98.5	19.1	7.7	91.7	3.8	1.8	98.0
325	0.045	45	5.2	2.2	24.3	19.2	8.5	43.6	20.8	9.1	28.4	9.9	4.5	92.1	21.7	6.5	92.0	29	11.7	80.0	7	3.4	94.7
400	0.038	38	5.3	2.2	22.1	15.1	6.7	36.9	13.5	5.9	22.4	12.2	5.6	86.5	2.7	0.8	91.2	32.9	13.3	66.7	2.1	1.0	93.6
500	0.025	25	8.6	3.6	18.6	15.2	6.7	30.2	11.2	4.9	17.5	33.3	15.2	71.3	67.5	20.1	71.1	50	20.2	46.5	29.7	14.3	79.4
-500	-0.025	-25	44.6	18.6	0.0	68	30.2	0.0	40	17.5	0.0	155.9	71.3	0.0	238.9	71.1	0.0	114.8	46.5	0.0	164.9	79.4	0.0
			240.3			225.4			228.1			218.7			335.9			247.1			207.8		
				P80, µm	1877		P80, µm	88		P80, µm	100		P80, µm	32		P80, µm	31		P80, µm	45		P80, µm	26

Appendix 43: Chemical Assay by Flotation Product Data for Pilot Run 4 and Optimized NatFlot Bench Test (Test 18-82)

Run 4 (afternoon)

Product	Amine (#/LT)	Total Fe (%)	SiO2 (%)
Ro. Froth 1	0.05	25.52	58.25
Ro. Froth 2	0.05	28.62	50.58
Ro. Froth 3	0.05	35.90	41.41
Cl. Froth 1	0.15	42.42	31.16
Cl. Froth 2	0.15	55.98	11.48
Cl. U/F		61.52	4.16
Flot Feed (direct)		42.29	32.91

Bench WO18-82

Product	Amine (#/LT)	Total Fe (%)	SiO2 (%)
Ro. Froth 1	0.05	20.07	68.02
Ro. Froth 2	0.05	22.37	64.14
Ro. Froth 3	0.05	29.42	52.67
Ro. Froth 4	0.10	38.88	35.50
Ro. Froth 5	0.10	50.34	17.26
Ro. Froth 6	0.10	55.93	10.33
Ro. U/F		62.67	2.34
Flot Feed (direct)		42.01	33.37

Appendix 44: Run 5 Bench Test Data

Run 5
Date 7/17/2018
Material Run 5 Total Froth (Tailings Thickener Feed)
Test Scavenger
Starch 0.5 #/LT

Product	Wt (g)	Wt (%)	Total Fe (%)	SiO2 (%)	Total Fe Dist (%)	SiO2 Dist (%)	Amine (#/LT)	Float Time (min)
F1	141.5	27.6	15.76	71.53	13.4	43.0		3.0
F2	150.3	29.3	19.97	64.74	18.1	41.3	0.10	4.5
F3	54.4	10.6	32.93	42.04	10.8	9.7	0.10	3.0
F4	36.8	7.2	46.79	21.05	10.4	3.3	0.10	3.0
F5	19.2	3.7	53.20	12.60	6.2	1.0	0.10	2.0
F6	18.3	3.6	57.02	8.30	6.3	0.6	0.10	2.0
U/F	93.0	18.1	62.19	2.59	34.9	1.0		
Head (calc)	513.5	100.0	32.31	45.86	100.0	100.0	0.50	17.5

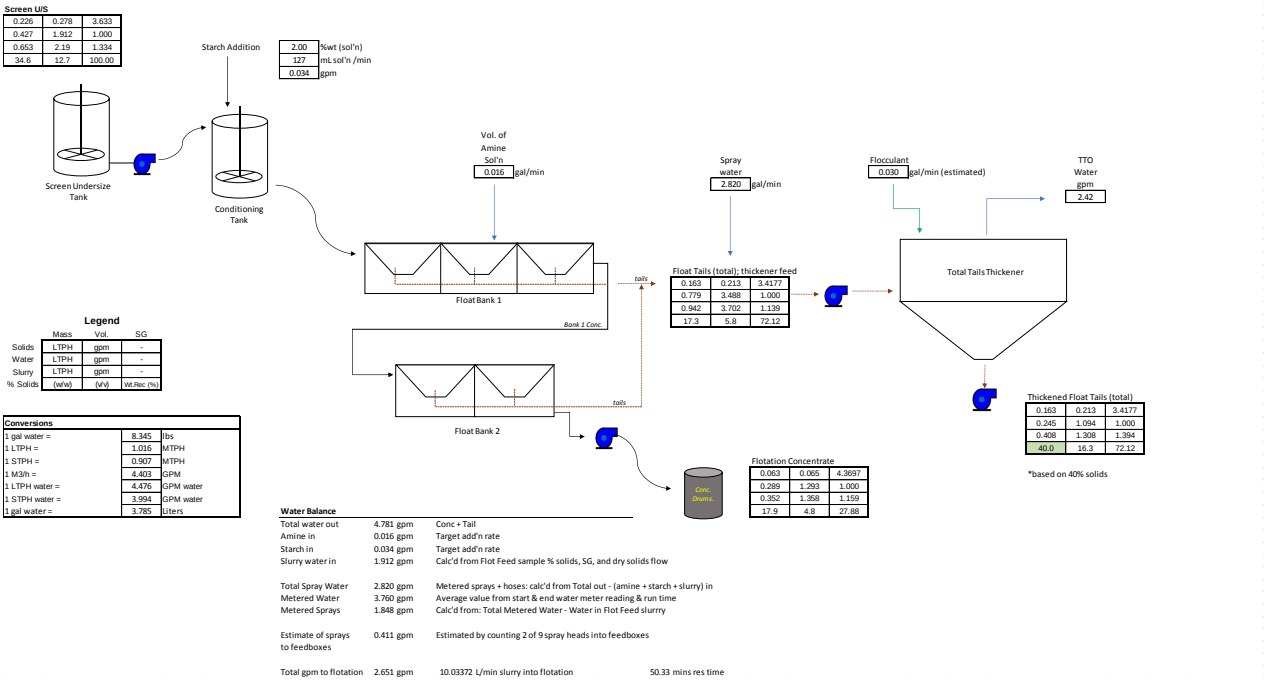
Combined	Wt (%)	Total Fe (%)	SiO2 (%)	Total Fe Dist (%)	SiO2 Dist (%)	Amine (#/LT)	Float Time (min)
U/F	18.1	62.19	2.59	34.9	1.0	0.50	17.50
U/F + F6	21.7	61.34	3.53	41.1	1.7	0.40	15.50
U/F + F5-6	25.4	60.14	4.86	47.3	2.7	0.30	13.50
U/F + F4-6	32.6	57.21	8.42	57.7	6.0	0.20	10.50
U/F + F3-6	43.2	51.25	16.67	68.5	15.7	0.10	7.50
U/F + F2-6	72.4	38.61	36.09	86.6	57.0	0.00	3.00
U/F + F1-6	100.0	32.31	45.86	100.0	100.0	0.00	0.00

Run 5
Date 7/17/2018
Material Run 5 Cleaner Float U/F (Con)
Test Re-Cleaner
Starch None #/LT

Product	Wt (g)	Wt (%)	Total Fe (%)	SiO2 (%)	Total Fe Dist (%)	SiO2 Dist (%)	Amine (#/LT)	Float Time (min)
F1	15.1	2.8	43.79	28.72	2.0	19.8		2.0
F2	49.1	9.0	55.12	11.75	8.1	26.3	0.10	2.5
F3	41.6	7.7	59.90	5.59	7.5	10.6	0.10	2.0
F4	43.5	8.0	61.61	3.69	8.0	7.3	0.10	1.5
U/F	393.4	72.5	62.93	2.01	74.4	36.0		
Head (calc)	542.7	100.0	61.35	4.04	100.0	100.0	0.30	8.0

Combined	Wt (%)	Total Fe (%)	SiO2 (%)	Total Fe Dist (%)	SiO2 Dist (%)	Amine (#/LT)	Float Time (min)
U/F	72.5	62.93	2.01	74.4	36.0	0.30	8.00
U/F + F4	80.5	62.80	2.18	82.4	43.3	0.20	6.50
U/F + F3-4	88.2	62.55	2.47	89.9	53.9	0.10	4.50
U/F + F2-4	97.2	61.86	3.34	98.0	80.2	0.00	2.00
U/F + F1-4	100.0	61.35	4.04	100.0	100.0		

Appendix 45: Run 6 Ore and Water Balance



Water Balance

Total water out	4.781 gpm	Conc + Tail
Amine in	0.016 gpm	Target add'n rate
Starch in	0.034 gpm	Target add'n rate
Slurry water in	1.912 gpm	Calc'd from Flot Feed sample % solids, SG, and dry solids flow
Total Spray Water	2.820 gpm	Metered sprays + hoses: calc'd from Total out - (amine + starch + slurry) in
Metered Water	3.760 gpm	Average value from start & end water meter reading & run time
Metered Sprays	1.848 gpm	Calc'd from: Total Metered Water - Water in Flot Feed slurrry
Estimate of sprays to feedboxes	0.411 gpm	Estimated by counting 2 of 9 spray heads into feedboxes
Total gpm to flotation	2.651 gpm	10.03372 L/min slurry into flotation 50.33 mins res time

Appendix 46: Run 6 Bench Test Data

Run 6
Date 7/20/2018 **Note: Ran test on 7/25/18**
Material Run 6 Total Froth (Tailings Thickener Feed)
Test Scavenger
Starch 0.5 #/LT

Product	Wt (g)	Wt (%)	Total Fe (%)	SiO2 (%)	Phos (%)	Total Fe Dist (%)	SiO2 Dist (%)	Phos Dist (%)	Amine (#/LT)	Float Time (min)
F1	128.4	24.8	13.79	76.44	0.012	9.8	43.1	12.7		4.0
F2	16.9	3.3	16.33	72.60	0.014	1.5	5.4	2.0		1.5
F3	133.7	25.8	20.80	65.71	0.016	15.5	38.6	17.7	0.10	4.5
F4	59.3	11.5	41.14	31.33	0.032	13.6	8.2	15.7	0.10	4.0
F5	40.8	7.9	52.53	13.90	0.040	11.9	2.5	13.5	0.10	3.5
F6	30.3	5.9	58.04	7.57	0.042	9.8	1.0	10.5	0.10	2.5
F7	19.3	3.7	61.14	4.56	0.037	6.6	0.4	5.9	0.10	1.5
U/F	88.9	17.2	63.32	2.11	0.030	31.3	0.8	22.0		
Head (calc)	517.6	100.0	34.73	43.97	0.023	100.0	100.0	100.0	0.50	21.5

Combined	Wt (%)	Total Fe (%)	SiO2 (%)	Phos (%)	Total Fe Dist (%)	SiO2 Dist (%)	Phos Dist (%)	Amine (#/LT)	Float Time (min)
U/F	17.2	63.32	2.11	0.030	31.3	0.8	22.0	0.50	21.5
U/F + F7	20.9	62.93	2.55	0.031	37.9	1.2	27.9	0.40	20.0
U/F + F6-7	26.8	61.86	3.65	0.034	47.7	2.2	38.5	0.30	17.5
U/F + F5-7	34.6	59.74	5.98	0.035	59.6	4.7	51.9	0.20	14.0
U/F + F4-7	46.1	55.12	12.28	0.034	73.1	12.9	67.6	0.10	10.0
U/F + F3-7	71.9	42.79	31.47	0.028	88.6	51.5	85.3	0.00	5.5
U/F + F2-7	75.2	41.64	33.25	0.027	90.2	56.9	87.3	0.00	4.0
U/F + F1-7	100.0	34.73	43.97	0.023	100.0	100.0	100.0		

Run 6
Date 7/20/2018 **Note: Ran test on 7/24/18**
Material Run 6 Cleaner Float U/F (Con)
Test Re-Cleaner
Starch None #/LT

Product	Wt (g)	Wt (%)	Total Fe (%)	SiO2 (%)	Total Fe Dist (%)	SiO2 Dist (%)	Amine (#/LT)	Float Time (min)
F1	3.5	1.2	45.51	24.59	0.9	7.5		1.5
F2	37.3	13.1	53.58	13.90	11.4	45.2	0.10	2.5
F3	47.1	16.6	61.34	4.00	16.5	16.4	0.10	2.0
F4	52.6	18.5	63.03	2.25	19.0	10.3	0.10	1.5
U/F	143.9	50.6	63.46	1.63	52.2	20.5		
Head (calc)	284.4	100.0	61.51	4.03	100.0	100.0	0.30	7.5

Combined	Wt (%)	Total Fe (%)	SiO2 (%)	Total Fe Dist (%)	SiO2 Dist (%)	Amine (#/LT)	Float Time (min)
U/F	50.6	63.46	1.63	52.2	20.5	0.30	7.50
U/F + F4	69.1	63.34	1.80	71.2	30.8	0.20	6.00
U/F + F3-4	85.7	62.96	2.22	87.7	47.2	0.10	4.00
U/F + F2-4	98.8	61.71	3.77	99.1	92.5	0.00	1.50
U/F + F1-4	100.0	61.51	4.03	100.0	100.0		

Appendix 47: Run 6 Combined Pilot and Bench Test Data

Cleaner Products	Wt Rec	Total Fe, %	Silica, %	Phos, %	Fe Rec	Amine (#/LT)	Time (min)
Cl. Con	27.9	61.40	4.15	0.032	40.9	0.35	50.3
Cl. Tails	72.1	34.27	45.15	0.024	59.1		
Head (calc)	100.0	41.83	33.73	0.026	100.0		

Scavenger Products	Wt Rec	Total Fe, %	Silica, %	Phos, %	Fe Rec	Amine (#/LT)	Float Time (min)
U/F	17.2	63.32	2.11	0.030	31.3	0.50	21.5
U/F + F7	20.9	62.93	2.55	0.031	37.9	0.40	20.0
U/F + F6-7	26.8	61.86	3.65	0.034	47.7	0.30	17.5
U/F + F5-7	34.6	59.74	5.98	0.035	59.6	0.20	14.0
U/F + F4-7	46.1	55.12	12.28	0.034	73.1	0.10	10.0
U/F + F3-7	71.9	42.79	31.47	0.028	88.6		5.5
U/F + F2-7	75.2	41.64	33.25	0.027	90.2		4.0
U/F + F1-7	100.0	34.73	43.97	0.023	100.0		

Combined Products	Wt Rec	Total Fe, %	Silica, %	Phos, %	Fe Rec	Amine (#/LT)	Float Time (min)
Cl. Con	27.9	61.40	4.15	0.032	40.9	0.35	50.3
Cl. Con + U/F	40.3	61.99	3.52	0.031	59.4	0.85	71.8
Cl. Con + U/F + F7	42.9	61.94	3.59	0.032	63.3	0.75	70.3
Cl. Con + U/F + F6-7	47.2	61.59	3.94	0.033	69.1	0.65	67.8
Cl. Con + U/F + F5-7	52.9	60.61	5.01	0.033	76.1	0.55	64.3
Cl. Con + U/F + F4-7	61.1	57.98	8.57	0.033	84.1	0.45	60.3
Cl. Con + U/F + F3-7	79.8	49.29	21.92	0.029	93.3	0.35	55.8
Cl. Con + U/F + F2-7	82.1	48.35	23.38	0.029	94.2	0.35	54.3
Cl. Con + U/F + F1-7	100.0	42.16	32.87	0.026	100.0	0.35	50.3

Product	**P80 (µm)**	**Wt Rec**	**Total Fe, %**	**Silica, %**	**Phos, %**	**Fe Rec**	**Amine (#/LT)**	**Time (min)**
Cl. Con	42	27.9	61.40	4.15	0.032	40.9	0.35	50.3
Scav. Con	25	19.3	61.86	3.65	0.034	28.2	0.30	17.5
Final Con	35	47.2	61.59	3.94	0.033	69.1	0.65	67.8

Appendix 48: Whole Rock Analysis of Pilot-Scale Run 6 Cleaner Flotation Underflow (Concentrate) and Froth (Tailings) from Activation Laboratories

Report Number: A18-10052
Report Date: 13/8/2018

Analyte Symbol	SiO2	Al2O3	Fe2O3	MnO	MgO	CaO	Na2O	K2O	TiO2	P2O5	LOI	LOI2	Total	Total 2	Fe2O3(T)	FeO
Unit Symbol	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%
Detection Limit	0.01	0.01	0.01	0.001	0.01	0.01	0.01	0.01	0.001	0.01			0.01	0.01	0.01	0.1
Analysis Method	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	TITR
RUN 6 7-20-18 FLOT U/F (CON)	4.28	0.46	85.63	0.082	0.16	0.53	0.08	0.04	0.05	0.09	7.01	6.91	99.41	99.31	86.64	0.9
RUN 6 7-20-18 FLOT FROTH (TAILS)	46.57	1.71	43.84	0.064	0.38	0.64	0.38	0.22	0.081	0.06	4.17	4.06	99.22	99.11	44.95	1

Analyte Symbol	Au	As	Br	Cr	Ir	Sb	Sc	Se	Mass	Cd	Cu	Ni	Zn	S	Ag	Pb
Unit Symbol	ppb	ppm	ppm	ppm	ppb	ppm	ppm	ppm	g	ppm	ppm	ppm	ppm	%	ppm	ppm
Detection Limit	2	0.5	0.5	5	5	0.2	0.1	3		0.5	1	1	1	0.001	0.3	5
Analysis Method	INAA	INAA	INAA	INAA	INAA	INAA	INAA	INAA	INAA	TD-ICP	TD-ICP	TD-ICP	TD-ICP	TD-ICP	TD-ICP	TD-ICP
RUN 6 7-20-18 FLOT U/F (CON)	< 2	13.4	< 0.5	25	< 5	1	1	< 3	16.9	1.2	75	11	27	0.009	1	6
RUN 6 7-20-18 FLOT FROTH (TAILS)	< 2	11.2	< 0.5	25	< 5	0.6	1.5	< 3	11.3	0.7	93	12	31	0.011	0.7	< 5

Analyte Symbol	Sc	Be	V	Cr	Co	Ni	Cu	Zn	Ga	Ge	As	Rb	Sr	Y	Zr	Nb
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit	1	1	5	20	1	20	10	30	1	1	5	2	2	1	2	1
Analysis Method	FUS-ICP	FUS-ICP	FUS-ICP	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-ICP	FUS-ICP	FUS-ICP	FUS-MS
RUN 6 7-20-18 FLOT U/F (CON)	1	2	27	20	6	< 20	60	< 30	1	12	14	< 2	11	8	34	1
RUN 6 7-20-18 FLOT FROTH (TAILS)	2	1	24	30	5	< 20	80	< 30	2	6	8	7	30	5	30	1

Analyte Symbol	Mo	Ag	In	Sn	Sb	Cs	Ba	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit	2	0.5	0.2	1	0.5	0.5	2	0.1	0.1	0.05	0.1	0.1	0.05	0.1	0.1	0.1
Analysis Method	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-ICP	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS
RUN 6 7-20-18 FLOT U/F (CON)	5	< 0.5	< 0.2	< 1	1.3	< 0.5	57	2.9	6.6	0.66	2.9	0.7	0.23	0.8	0.1	0.9
RUN 6 7-20-18 FLOT FROTH (TAILS)	4	< 0.5	< 0.2	< 1	0.7	< 0.5	93	3.7	8.2	0.89	3.5	0.8	0.23	0.8	0.1	0.9

Analyte Symbol	Ho	Er	Tm	Yb	Lu	Hf	Ta	W	Tl	Pb	Bi	Th	U
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit	0.1	0.1	0.05	0.1	0.01	0.2	0.1	1	0.1	5	0.4	0.1	0.1
Analysis Method	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS
RUN 6 7-20-18 FLOT U/F (CON)	0.2	0.6	0.1	0.6	0.09	< 0.2	< 0.1	4	< 0.1	< 5	< 0.4	0.6	0.9
RUN 6 7-20-18 FLOT FROTH (TAILS)	0.2	0.5	0.08	0.5	0.07	0.4	< 0.1	5	< 0.1	< 5	< 0.4	0.9	0.7

Appendix 49: Process Water and Pilot-Scale Run 2 through Run 6 Tailings Thickener Overflow Water Analysis from Pace Analytical

	Mercury		ASTM D2327 Nitrogen, Amine	300.0 IC Anions 28 Days		2320B Alkalinity	5220D COD	SM5210B, BOD, Low Level	2540D Total Suspended Solids	350.1 Ammonia, Distilled
Parameter	Low Level Mercury	Low Level Mercury (dissolved)	Nitrogen, Amine	Chloride	Sulfate	Alkalinity (Total as CaCO3)	Chemical Oxygen Demand	BOD, 5 Day	Total Suspended Solids	Nitrogen, Ammonia
Method	EPA 1631E (2001)	EPA 1631E (2001)	ASTM D2327	EPA 300.0	EPA 300.0	SM 2320B	SM 5220D	SM 5210B*	SM 2540D (1997)	EPA 350.1 rev. 2 (1993)
Units	ng/L	ng/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
Reporting Limit	0.50	0.50	0.25	1.0	2.0	10.0	25.0	2.4	2.2	0.11
Fresh Process Water (6/20/18)	0.606	ND	ND	22.7	118	262	ND	ND	11.3	ND
Run 2 Thickener Overflow (6/22/18)	2.22	ND	1.6	46.7	121	236	32.4	ND	65.2	ND
Run 3 Thickener Overflow (6/27/18)	3.96	0.59	2.2	61.3	124	224	48.8	ND	120	ND
Run 4 Thickener Overflow (7/2/18)	1.81	0.56	1.2	64.4	122	214	76.3	24.6	263	ND
Run 5 Thickener Overflow (7/17/18)	4.25	ND	2.2	70.7	119	212	81.4	20.0	111	0.15
Run 6 Thickener Overflow (7/20/18)	1.84	ND	1.8	75.5	122	206	70.6	14	63.2	0.13
Average	2.82	N/A	1.8	63.7	122	218	61.9	N/A	124.5	N/A
Max	4.25	0.59	2.2	75.5	124	236	81.4	24.6	263.0	0.15
Std. Dev.	1.07	N/A	0.4	9.8	2	11	18.5	N/A	73.0	N/A
ND = Not Detected								*Run 4 used Hach 10360 BOD		
								Parameter	BOD, 5 day	
								Method	Hach 10360 rev. 1.2 (2011)	
								Units	mg/L	
								Reporting Limit	8.0	

	351.2 Total Kjeldahl Nitrogen	**353.2 Nitrate + Nitrite pres.**	**365.1 Phosphorous, Total**	**WIDRO CGS**	**4500H+ pH, Electrometric**	**Total Nitrogen**	**2510B Specific Conductance**
Parameter	Nitrogen, Kjeldahl, Total	Nitrogen, NO2 plus NO3	Phosphorous	WDRO C10-C28	pH at 25 Degrees C	Total Nitrogen Calculation	Specific Conductance
Method	EPA 351.2	EPA 353.2	EPA 365.1	WI MOD DRO	SM 4500-H+B	TKN+NO3+NO2 Calculation	SM 2510B
Units	mg/L	mg/L	mg/L	mg/L	Std. Units	mg/L	umhos/cm
Reporting Limit	0.50	0.050	0.14	0.095	0.10	0.55	10.0
Fresh Process Water (6/20/18)	ND	ND	0.64	ND	8.0	ND	762
Run 2 Thickener Overflow (6/22/18)	0.79	0.051	ND	0.69	8.1	0.84	812
Run 3 Thickener Overflow (6/27/18)	1.1	ND	ND	1.1	8.2	1.1	856
Run 4 Thickener Overflow (7/2/18)	0.87	0.065	ND	2.4	7.9	0.93	868
Run 5 Thickener Overflow (7/17/18)	1.0	ND	ND	2.2	8.2	1.0	839
Run 6 Thickener Overflow (7/20/18)	1.0	ND	ND	2.4	7.9	1.0	848
Average	0.95	N/A	N/A	1.76	8.1	0.97	845
Max	1.10	0.065	N/A	2.40	8.2	1.10	868
Std. Dev.	0.11	N/A	N/A	0.72	0.1	0.09	19
ND = Not Detected							

	200.7 Met ICP									200.7 Met ICP (dissolved)								
Parameter	Calcium	Copper	Iron	Magnesium	Manganese	Nickel	Potassium	Sodium	Zinc	Calcium (dissolved)	Copper (dissolved)	Iron (dissolved)	Magnesium (dissolved)	Manganese (dissolved)	Nickel (dissolved)	Potassium (dissolved)	Sodium (dissolved)	Zinc (dissolved)
Method	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7	EPA 200.7
Units	mg/L	ug/L	ug/L	mg/L	ug/L	ug/L	mg/L	mg/L	ug/L	mg/L	ug/L	ug/L	mg/L	ug/L	ug/L	mg/L	mg/L	ug/L
Reporting Limit	0.5	10.0	50.0	0.5	10.0	10.0	0.5	0.5	20.0	0.5	10.0	50.0	0.5	10.0	10.0	0.5	0.5	20.0
Fresh Process Water (6/20/18)	98.1	ND	674	36.5	11.4	ND	4.7	15.0	38.0	84.5	ND	ND	31.0	ND	ND	3.9	12.7	37.7
Run 2 Thickener Overflow (6/22/18)	91.0	ND	1120	32.1	67.7	ND	6.5	32.3	ND	89.8	ND	ND	31.4	41.7	ND	6.3	31.7	ND
Run 3 Thickener Overflow (6/27/18)	88.7	ND	3640	29.5	66.4	ND	6.9	40.3	ND	76.1	ND	68.7	25.6	24.8	ND	6.1	36.9	ND
Run 4 Thickener Overflow (7/2/18)	87.6	ND	7010	29.9	210	ND	7.6	44.3	ND	83.1	ND	ND	27.8	118	ND	7.1	42.5	ND
Run 5 Thickener Overflow (7/17/18)	89.1	ND	3030	29.3	73.1	ND	7.9	50.5	ND	75.8	ND	ND	25.0	31.0	ND	6.7	43.4	ND
Run 6 Thickener Overflow (7/20/18)	86.2	ND	6050	28.4	126	ND	8.2	51.8	ND	85.9	ND	ND	27.9	49.5	ND	8.2	51.9	ND
Average	88.5	N/A	4170	29.8	108.6	N/A	7.4	43.8	N/A	82.1	N/A	N/A	27.5	53.0	N/A	6.9	41.3	N/A
Max	91.0	N/A	7010	32.1	210.0	N/A	8.2	51.8	N/A	89.8	N/A	68.7	31.4	118.0	N/A	8.2	51.9	N/A
Std. Dev.	1.6	N/A	2121	1.2	55.3	N/A	0.6	7.1	N/A	5.5	N/A	N/A	2.2	33.6	N/A	0.7	6.8	N/A
ND = Not Detected																		

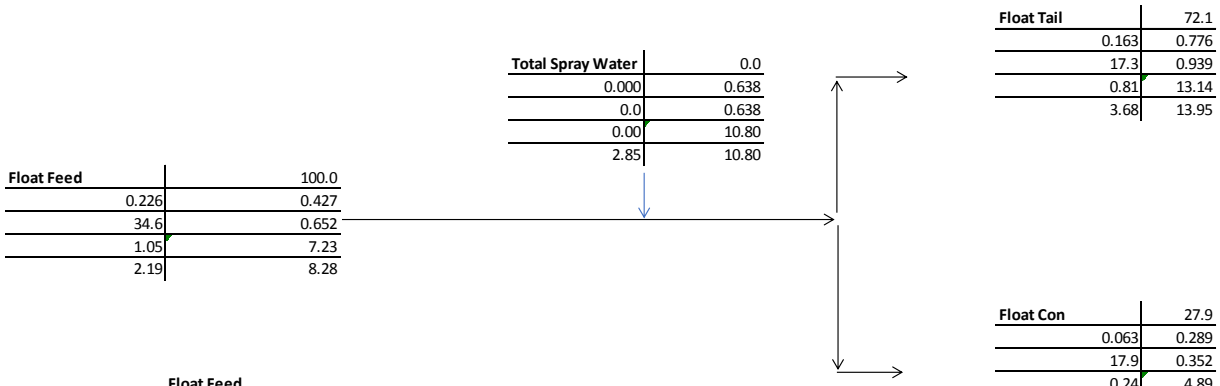
	200.8 MET ICPMS						200.8 MET ICPMS (dissolved)						8260B MSV Low Level
Parameter	Antimony	Arsenic	Beryllium	Cadmium	Chromium	Lead	Antimony (dissolved)	Arsenic (dissolved)	Beryllium (dissolved)	Cadmium (dissolved)	Chromium (dissolved)	Lead (dissolved)	Naphthalene
Method	EPA 200.8	EPA 200.8	EPA 200.8	EPA 200.8	EPA 200.8	EPA 200.8	EPA 200.8	EPA 200.8	EPA 200.8	EPA 200.8	EPA 200.8	EPA 200.8	EPA 8260B
Units	ug/L	ug/L	ug/L	ug/L	ug/L	ug/L	ug/L	ug/L	ug/L	ug/L	ug/L	ug/L	ug/L
Reporting Limit	0.5	1.0	0.2	0.2	1.0	0.5	0.5	1.0	0.2	0.2	1.0	0.5	1.0
Fresh Process Water (6/20/18)	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND
Run 2 Thickener Overflow (6/22/18)	ND	ND	ND	ND	1.4	0.62	ND	ND	ND	ND	ND	ND	ND
Run 3 Thickener Overflow (6/27/18)	ND	1.2	ND	ND	2.1	ND	ND	ND	ND	ND	ND	ND	ND
Run 4 Thickener Overflow (7/2/18)	ND	1.4	ND	ND	3.5	0.67	ND	ND	ND	ND	ND	ND	ND
Run 5 Thickener Overflow (7/17/18)	ND	1.4	ND	ND	2.0	0.85	ND	ND	ND	ND	ND	ND	ND
Run 6 Thickener Overflow (7/20/18)	ND	1.7	ND	ND	4.5	0.65	ND	ND	ND	ND	ND	ND	ND
Average	N/A	N/A	N/A	N/A	2.7	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Max	N/A	1.7	N/A	N/A	4.5	0.85	N/A	N/A	N/A	N/A	N/A	N/A	N/A
Std. Dev.	N/A	N/A	N/A	N/A	1.1	N/A	N/A	N/A	N/A	N/A	N/A	N/A	N/A
ND = Not Detected													

Appendix 50: Pilot-Scale Run 6 Residual Amine Testing: Ore and Water Balance and Removal Rate Calculations

Run	Product	% Solids	Specific Gravity	Weight Dist (%)
6	Circuit Feed	97.5	3.6248	100.0
	Ball Mill Discharge	65.4	3.6822	N/A
	Screen Oversize	68.3	3.6858	N/A
	Screen Undersize	38.0	3.6305	N/A
	Flot Feed	34.6	3.6330	100.0
	Flot Conc	17.9	4.3697	27.9
	Flot Tails	17.3	3.4177	72.1

O&W Balance Key

Product	Wt Dist
Solids (LTPH)	Water (LTPH)
Density	Slurry Flow (LTPH)
Solids (L/min)	Water (L/min)
Slurry Flow (gal/min)	Slurry Flow (L/min)



Float Feed	
Total Amine In	0.35 #/LTPH
	0.1 #/h
	597.3 mg/min
Total Water In	18.03 L/min
Amine in	33.1 mg/L
Amine out	1.8 mg/L
Removal Rate	94.6 %

Appendix 51: NatFlot Process Water Influent, Effluent, and Minnesota's Freshwater Quality Standards for Class 2A Waters

Authors Note: The information below was provided by former NRRI Limnologist and Aquatic Ecologist, Dr. Rich Axler (ret.).

Mean concentrations (± standard deviation) of dissolved heavy metals measured in process influent and effluent. Runs 2-6 as described previously were used for this summary. RL = Report Limits as reported by Pace Labs and are used to estimate the Method Detection Limit (MDL); values below the RL are indicated as ND (non-detectable) and assigned values of 0.5*RL in the calculation of means and standard deviations. *FW criterion (2A)* indicates the current Minnesota freshwater quality standard for its most sensitive Class 2A waters as per MPCA (2018: https://www.pca.state.mn.us/sites/default/files/wq-iw1-04j.pdf). * CS = chronic standard and MS = maximum standard. Since certain heavy metal standards are Hardness dependent, the values listed in this table assume relatively softwater (hardness ~ 100 mgCaCO3/L) for simplicity*. Formulae to calculate these standards for any hardness value are found in MPCA (2018) above; dissolved metals, as opposed to total metals are the preferred concentrations to use for toxicity assessments. Because of the particular importance of mercury in NE Minnesota waters, both dissolved and total Hg are reported although total-Hg is used for the standard. Criteria are based on toxicity to freshwater organisms except for Hg which is based on human health.

	DISSOLVED HEAVY METALS										
	Antimony (ug/L)	Arsenic (ug/L)	Beryllium (ug/L)	Cadmium (ug/L)	Chromium (ug/L)	Copper (ug/L)	Lead (ug/L)	Nickel (ug/L)	Zinc (ug/L)	Mercury (ng/L)	Mercury Total (ng/L)
Reporting Limit	0.5	1.0	0.2	0.2	1.0	10	0.5	10	20	0.5	0.5
FW criterion (2A) (CS/MS)	5.5/90	2.0/360	-/-	1.1/3.9*	-/-	9.8/18 *	3.2/82*	158/1418*	106/117*	-/-	6.9/2400 [human health]
Process Effluent Average	nd	nd	nd	nd	nd	nd	nd	nd	nd	0.4 ± 0.2	2.8 ± 1.1
Process Influent	nd	nd	nd	nd	nd	nd	nd	nd	37.7	nd	0.61

Note 1. FW Criteria taken from Minnesota Administrative Rules 7050.0220 SPECIFIC WATER QUALITY STANDARDS BY ASSOCIATED USE CLASSES (https://www.revisor.mn.gov/rules/7050.0220/) and Minnesota Administrative Rules 7050.0222 SPECIFIC WATER QUALITY STANDARDS FOR CLASS 2 WATERS OF THE STATE; AQUATIC LIFE AND RECREATION (https://www.revisor.mn.gov/rules/7050.0222/).

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Mean concentrations (± standard deviation) of non-metal water quality parameters of potential regulatory concern measured in process influent and effluent. Runs 2-6 as described previously were used for this summary. RL = Report Limits as reported by Pace Labs and are used to estimate the Method Detection Limit (MDL); values below the RL are indicated as ND (non-detectable) and assigned values of 0.5*RL in the calculation of means and standard deviations. *FW criterion (2A)* indicates the current Minnesota freshwater quality standard for its most sensitive Class 2A waters as per MPCA (2018: https://www.pca.state.mn.us/sites/default/files/wq-iw1-04j.pdf).

	WATER QUALITY PARAMETERS								
	Nitrogen, Amine (mg/L)	COD (mg/L)	BOD5 (mg/L)	TSS (mg/L)	pH (std units)	Alkalinity (mg/L)	Spec. Electrical Conductivity (uS/cm)	WDRO C10-C28 (mg/L)	Napthalene (ug/L)
Reporting Limit	0.25	25.0	2.4	2.2	0.1	10	10	0.095	1.0
FW criterion (Class 2A Waters)	?	--	≤ 1.5	Northern. rivers <10	6.5 – 8.5	--	--	0.5 [oil std]	65/409 [human health std]
Process Effluent Average	1.8 ± 0.4	62 ± 18	12.2 ± 9.6	124 ± 73	8.1± 0.1	218 ± 11	845 ± 19	1.76 ± 0.72	nd
Process Influent	nd	nd	nd	11.3	8.0	262	762	nd	nd

Note 1. FW Criteria taken from Minnesota Administrative Rules 7050.0220 SPECIFIC WATER QUALITY STANDARDS BY ASSOCIATED USE CLASSES (https://www.revisor.mn.gov/rules/7050.0220/) and Minnesota Administrative Rules 7050.0222 SPECIFIC WATER QUALITY STANDARDS FOR CLASS 2 WATERS OF THE STATE; AQUATIC LIFE AND RECREATION (https://www.revisor.mn.gov/rules/7050.0222/).

Mean concentrations (± standard deviation) of major cations, anions, iron, and manganese measured in process influent and effluent. Runs 2-6 as described previously were used for this summary. RL = Report Limits as reported by Pace Labs and are used to estimate the Method Detection Limit (MDL); values below the RL are indicated as ND (non-detectable) and assigned values of 0.5*RL in the calculation of means and standard deviations. *FW criterion (2A)* indicates the current Minnesota freshwater quality standard for its most sensitive Class 2A waters as per MPCA (2018: https://www.pca.state.mn.us/sites/default/files/wq-iw1-04j.pdf). Total iron and manganese concentrations are included also.

	DISSOLVED MAJOR IONS. Iron, and Manganese										
	Cl (mg/L)	SO4 (mg/L)	Alk (mg CaCO3/L)	Ca (mg/L)	Mg (mg/L)	K (mg/L)	Na (mg/L)	Fe (ug/L)	Tot-Fe (ug/L)	Mn (ug/L)	Total-Mn (ug/L)
Reporting Limit	1.0	2.0	10	0.5	0.5	10	0.5	50	50	10	10
FW criterion (Class 2A Waters)	230 CS 860 MS	10 (Wild Rice Waters)	--	--	--	--	--	--	300 [Drinking water staining]		50 [Drinking water staining]
Process Effluent Average	64 ± 10	122 ± 1.6	218 ± 11	82 ± 5.5	27 ± 2.2	6.9 ± 0.7	41 ± 6.8	nd	4170 ± 2121	53 ± 34	109 ± 55.3
Process Influent	22.7	118	262	84	31	3.9	13	nd	674	nd	11.4

Note 1. FW Criteria taken from Minnesota Administrative Rules 7050.0220 SPECIFIC WATER QUALITY STANDARDS BY ASSOCIATED USE CLASSES (https://www.revisor.mn.gov/rules/7050.0220/) and Minnesota Administrative Rules 7050.0222 SPECIFIC WATER QUALITY STANDARDS FOR CLASS 2 WATERS OF THE STATE; AQUATIC LIFE AND RECREATION (https://www.revisor.mn.gov/rules/7050.0222/).

Mean concentrations (± standard deviation) of the plant micronutrients nitrogen and phosphorus, and their fractions, measured in process influent and effluent. Runs 2-6 as described previously were used for this summary. RL = Report Limits as reported by Pace Labs and are used to estimate the Method Detection Limit (MDL); values below the RL are indicated as ND (non-detectable) and assigned values of 0.5*RL in the calculation of means and standard deviations. *FW criterion (2A)* indicates the current Minnesota freshwater quality standard for its most sensitive Class 2A waters as per MPCA (2018: https://www.pca.state.mn.us/sites/default/files/wq-iw1-04j.pdf).

	NUTRIENTS				
	Total Phosphorus (mgP/L)	Total Nitrogen (calculated, mgN/L)	Total Kjeldahl Nitrogen (TKN, mgN/L)	[Nitrate + Nitrite]- Nitrogen (mgN/L)	Ammonia- Nitrogen (mgN/L)
Reporting Limit	0.14	0.55	0.5	0.05	0.11
FW criterion (Class 2A Waters)	Lakes < 0.012 Rivers < 0.050	--	--	- No FW std - Drinking water std 10.0	0.016 [as un-ionized NH3-N]
Process Effluent Average	nd	0.97 ± 0.086	0.95 ± 0.11	< RL [0.038 ± 0.017]	< RL [0.089 ± 0.042]
Process Influent	0.64	nd	nd	nd	nd

Note 1. The high TP in the process influent (0.64 mg/L which represents the City of Coleraine's tap water), relative to natural water bodies in the region suggests that the City of Coleraine may be adding a phosphate salt for corrosion control in drinking water infrastructure. Lakes in the Northern Lakes and Forests Ecoregion of MN are expected to have TP < 30 ugP/L (0.030 mgP/L) unless impaired.

Note 2. FW Criteria taken from Minnesota Administrative Rules 7050.0220 SPECIFIC WATER QUALITY STANDARDS BY ASSOCIATED USE CLASSES (https://www.revisor.mn.gov/rules/7050.0220/) and Minnesota Administrative Rules 7050.0222 SPECIFIC WATER QUALITY STANDARDS FOR CLASS 2 WATERS OF THE STATE; AQUATIC LIFE AND RECREATION (https://www.revisor.mn.gov/rules/7050.0222/).

From MPCA (2018) Guidance Manual for Assessing the Quality of Minnesota Surface Waters for Determination of impairment.
https://www.pca.state.mn.us/sites/default/files/wq-iw1-04j.pdf

Table 12. Lake eutrophication water quality standards for aquatic recreation use assessments.

Ecoregion	TP (µg/L)	chl-*a* (µg/L)	Secchi (m)
Northern Lakes and Forest – Lake trout (Class 2A)	< 12	< 3	> 4.8
Northern Lakes and Forest – Stream trout (Class 2A)	< 20	< 6	> 2.5
Northern Lakes and Forest – Aquatic Rec. Use (Class 2B)	< 30	< 9	> 2.0
North Central Hardwood Forest – Stream trout (Class 2A)	< 20	< 6	> 2.5
North Central Hardwood Forest – Aq. Rec. Use (Class 2B)	< 40	< 14	> 1.4
North Central Hardwood Forest – Aq. Rec. Use (Class 2B) Shallow lakes	< 60	< 20	> 1.0
Western Corn Belt Plains & Northern Glaciated Plains – Aq. Rec. Use (Class 2B)	< 65	< 22	> 0.9
Western Corn Belt Plains & Northern Glaciated Plains – Aq. Rec. Use (Class 2B) Shallow lakes	< 90	< 30	> 0.7

Eutrophication standards for Class 2A (coldwater) lakes and reservoirs (MPCA 2018).

Eutrophication standards for class 2A lakes and reservoirs.

Designated lake trout lakes in all ecoregions (lake trout lakes support natural populations of lake trout, *Salvelinus namaycush*):

Phosphorus, total	µg/L	12	NA	--	--	NA
Chlorophyll-a	µg/L	3	NA	--	--	NA
Secchi disk transparency	meters	No less than 4.8	NA	--	--	NA

Designated trout lakes in all ecoregions, except lake trout lakes:

Phosphorus, total	µg/L	20	NA	--	--	NA
Chlorophyll-a	µg/L	6	NA	--	--	NA
Secchi disk transparency	meters	No less than 2.5	NA	--	--	NA

Additional narrative eutrophication standards for class 2A lakes and reservoirs are found under subpart 2a.

Eutrophication standards for class 2A rivers and streams.

North River Nutrient Region:		
Phosphorus, total	µg/L	less than or equal to 50
Chlorophyll-a (seston)	µg/L	less than or equal to 7
Diel dissolved oxygen flux	mg/L	less than or equal to 3.0
Biochemical oxygen demand (BOD_5)	mg/L	less than or equal to 1.5
Central River Nutrient Region:		
Phosphorus, total	µg/L	less than or equal to 100
Chlorophyll-a (seston)	µg/L	less than or equal to 18
Diel dissolved oxygen flux	mg/L	less than or equal to 3.5
Biochemical oxygen demand (BOD_5)	mg/L	less than or equal to 2.0
South River Nutrient Region:		
Phosphorus, total	µg/L	less than or equal to 150
Chlorophyll-a (seston)	µg/L	less than or equal to 35
Diel dissolved oxygen flux	mg/L	less than or equal to 4.5
Biochemical oxygen demand (BOD_5)	mg/L	less than or equal to 3.0

Additional narrative eutrophication standards for class 2A rivers and streams are found under subpart 2b.

From MPCA (Heiskary & Wilson 2005) Lake Water Quality Assessment Report: Developing Nutrient Criteria https://www.pca.state.mn.us/sites/default/files/lwq-a-nutrientcriteria.pdf

Table 2. Reference Lake Data Base Water Quality Summary
(Summer Average Water Quality Characteristics for Lakes by Ecoregion)*

Parameter	NLF	CHF	WCP	NGP
# of lakes	32	43	16	13
Total Phosphorus (ug/l)	14 - 27	23 - 50	65 - 150	122 - 160
Chlorophyll mean (ug/l)	4 - 10	5 - 22	30 - 80	36 - 61
Chlorophyll maximum (ug/l)	< 15	7 - 37	60 - 140	66 - 88
Secchi Disk (feet) (meters)	8 - 15 (2.4 - 4.6)	4.9 - 10.5 (1.5 - 3.2)	1.6 - 3.3 (0.5 - 1.0)	1.3 – 2..6 (0.4 – 0.8)
Total Kjeldahl Nitrogen (mg/l)	0.4 – 0.75	< 0.60 - 1.2	1.3 - 2.7	1.8 - 2.3
Nitrite + Nitrate-N (mg/l)	<0.01	<0.01	0.01 - 0.02	0.01 - 0.1
Alkalinity (mg/l)	40 – 140	75 - 150	125 - 165	160 - 260
Color (Pt-Co Units)	10 – 35	10 - 20	15 - 25	20 - 30
pH (SU)	7.2 - 8.3	8.6 - 8.8	8.2 - 9.0	8.3 - 8.6
Chloride (mg/l)	0.6 – 1.2	4 - 10	13 - 22	11 - 18
Total Suspended Solids (mg/l)	< 1 – 2	2 - 6	7 - 18	10 - 30
Total Suspended Inorganic Solids (mg/l)	< 1 – 2	1 - 2	3 - 9	5 - 15
Turbidity (NTU)	< 2	1 - 2	3 - 8	6 - 17
Conductivity (umhos/cm)	50 – 250	300 - 400	300 - 650	640 - 900
TN:TP ratio	25:1 - 35:1	25:1 - 35:1	17:1 - 27:1	13:1 - 17:1

*Based on Interquartile range (25th - 75th percentile) for ecoregion reference lakes.
Derived in part from Heiskary, S. A. and C. B. Wilson (1990).

Appendix 52: Whole Rock Analysis of the Composite Pelletizing Feed Sample from Activation Laboratories

Report Number: A18-00644
Report Date: 6/2/2018

Analyte Symbol	SiO2	Al2O3	Fe2O3	MnO	MgO	CaO	Na2O	K2O	TiO2	P2O5	LOI	LOI2	Total	Total 2	Fe2O3(T)	FeO
Unit Symbol	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%	%
Detection Limit	0.01	0.01	0.01	0.001	0.01	0.01	0.01	0.01	0.001	0.01			0.01	0.01	0.01	0.1
Analysis Method	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	FUS-ICP	TITR
19-2 Head	34.37	1.29	57.22	0.068	0.27	0.49	0.29	0.16	0.067	0.05	4.66	4.57	99.83	99.74	58.11	0.8
Composite Filter Cake for Pelletizing	4.49	0.52	86.12	0.081	0.17	0.55	0.09	0.05	0.056	0.07	7.09	7	100.2	100.1	87.01	0.8

Analyte Symbol	Au	As	Br	Cr	Ir	Sb	Sc	Se	Mass	Cd	Cu	Ni	Zn	S	Ag	Pb
Unit Symbol	ppb	ppm	ppm	ppm	ppb	ppm	ppm	ppm	g	ppm	ppm	ppm	ppm	%	ppm	ppm
Detection Limit	2	0.5	0.5	5	5	0.2	0.1	3		0.5	1	1	1	0.001	0.3	5
Analysis Method	INAA	INAA	INAA	INAA	INAA	INAA	INAA	INAA	INAA	TD-ICP	TD-ICP	TD-ICP	TD-ICP	TD-ICP	TD-ICP	TD-ICP
19-2 Head	< 2	10.9	< 0.5	40	< 5	0.8	1.6	< 3	40.5	1.2	7	12	14	0.009	0.3	< 5
Composite Filter Cake for Pelletizing	< 2	14	< 0.5	23	< 5	1.1	1.1	< 3	14.4	1	25	11	13	0.009	1	7

Analyte Symbol	Sc	Be	V	Cr	Co	Ni	Cu	Zn	Ga	Ge	As	Rb	Sr	Y	Zr	Nb
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit	1	1	5	20	1	20	10	30	1	1	5	2	2	1	2	1
Analysis Method	FUS-ICP	FUS-ICP	FUS-ICP	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-ICP	FUS-ICP	FUS-ICP	FUS-MS
19-2 Head	2	1	25	40	6	< 20	10	< 30	2	9	12	5	25	< 1	21	< 1
Composite Filter Cake for Pelletizing	1	2	27	< 20	7	< 20	< 10	< 30	1	13	14	< 2	12	8	37	< 1

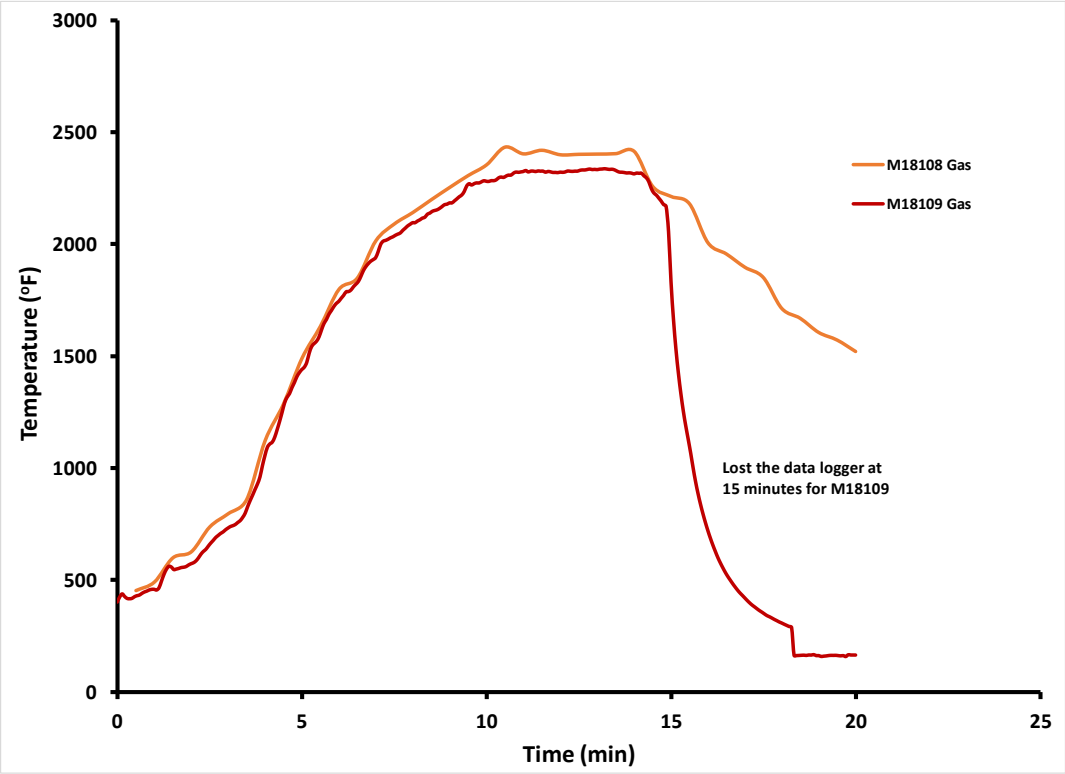
Analyte Symbol	Mo	Ag	In	Sn	Sb	Cs	Ba	La	Ce	Pr	Nd	Sm	Eu	Gd	Tb	Dy
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit	2	0.5	0.2	1	0.5	0.5	2	0.1	0.1	0.05	0.1	0.1	0.05	0.1	0.1	0.1
Analysis Method	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-ICP	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS
19-2 Head	3	< 0.5	< 0.2	< 1	0.9	< 0.5	45	3.4	7.9	0.81	3.3	0.7	0.23	0.8	0.1	0.9
Composite Filter Cake for Pelletizing	5	< 0.5	< 0.2	< 1	1	< 0.5	60	3.2	7.3	0.79	3.4	0.8	0.26	0.9	0.2	1

Analyte Symbol	Ho	Er	Tm	Yb	Lu	Hf	Ta	W	Tl	Pb	Bi	Th	U
Unit Symbol	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
Detection Limit	0.1	0.1	0.05	0.1	0.01	0.2	0.1	1	0.1	5	0.4	0.1	0.1
Analysis Method	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS	FUS-MS
19-2 Head	0.2	0.6	0.08	0.5	0.08	0.3	< 0.1	6	< 0.1	< 5	< 0.4	0.9	0.9
Composite Filter Cake for Pelletizing	0.2	0.8	0.11	0.6	0.09	< 0.2	< 0.1	3	< 0.1	< 5	< 0.4	0.8	0.9

Appendix 53: Mini-Pot Firing Cycle

Time (min)	Temperature (^{o}F)	O_2 (%)	ΔP (in)
0.5	450	20	6
1	500	20	6
2	600	20	6
3	800	20	6
4	1200	20	6
5	1500	20	6
6	1800	20	6
7	2000	20	6
8	2150	17	6
9	2250	17	6
10	2350	17	6
11	2400	17	6
12	2400	17	6
13	2400	17	6
14	2400	17	6
15	2200	20	6
16	2000	20	6
17	1900	20	6
18	1700	20	6
19	1600	20	6
20	1500	20	6

Appendix 54: Comparison of M18108 and M18109 Mini-Pot Temperature Profiles



Note that the data logger was lost for trial M18109 after 15 minutes.

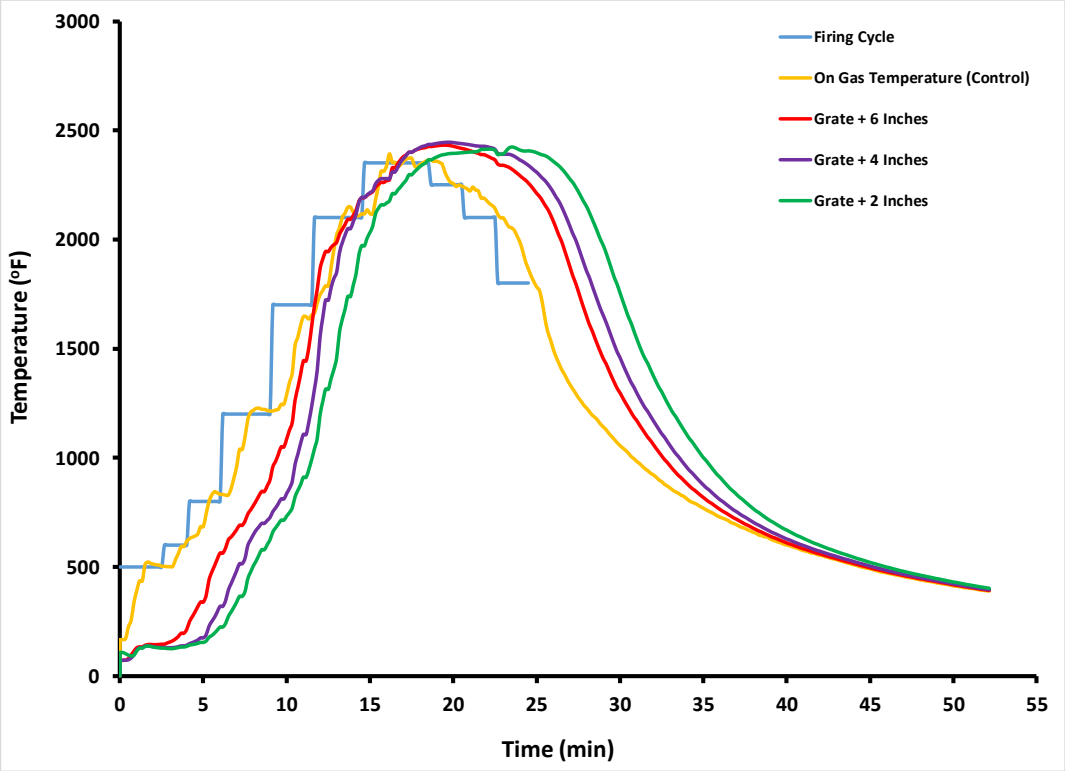
Appendix 55: Pot Grate Firing Cycle

Zone	Time (min)	Temperature (^oF)	ΔP (inches)	O_2 (%)
DD1	02:30	500	15	21
DD2a	01:30	600	12	21
DD2b	02:00	800	12	21
PH1	03:00	1200	12	18
PH2	02:30	1700	12	18
Fire 1	03:00	2100	8	17
Fire 2	04:00	2350	8	17
Fire 3	02:00	2250	6	17
Afterfire 1	02:00	2100	6	16
Afterfire 2	02:00	1800	6	16
Cooling			6	20

Note: DD = Down Draft; PH = Pre-Heat

Appendix 56: P18305 and P18306 Pot Grate Temperature Profiles

Temperature profile for pot grate P18305 (1.5% coke breeze addition).



Temperature profile for pot grate P18306 (1.2% coke breeze addition).