

Production of magnetite concentrate from copper concentrator plant tailings

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Abstract: Waste management is one of the most critical issues in ore enrichment plants. Enriching valuable minerals found in plant waste to make them saleable contributes to waste management operations and increases the plant's economic input. Following the detection of magnetite in the waste of the ETİ Bakır Siirt Copper Concentrator Plant, the objective was to recover and concentrate magnetite to enhance resource efficiency and economic sustainability. This study includes low-field high-intensity wet magnetic separation and flotation tests performed on waste from the Siirt concentrator plant. A three-stage magnetic separation process was applied to samples taken from the plant waste. Flotation tests were conducted under different conditions on the concentrate obtained from magnetic separation, ensuring the removal of sulfide minerals. As a result of the studies, a concentrate containing 77.50% magnetite was obtained from the waste sample, which contained 3.44% magnetite, with a recovery of 52.06%. Final iron concentrate contains 64.68% Fe, 0.02% Cu, and 0.25% S. To determine the industrial viability of the magnetite concentrate, a reduction process was carried out. Following the reduction process, pig iron was produced, and ingots with a purity of 98.76% were obtained.

Keywords: magnetite, magnetic separation, flotation, pyrometallurgy, pig iron

1. Introduction

The sustainable management of industrial by-products is an important component of modern economic and resource policy in the global metallurgical industry. The rapid depletion of high-quality primary iron ore deposits, combined with an unprecedented increase in global industrial demand, has driven major changes in the iron and steel industries, shifting focus to the systematic valorization of secondary resources (Kinnunen and Kaksonen, 2018; Dubiński, 2013). Copper concentration operations worldwide generate massive quantities of mine tailings annually (Silva et al., 2021). Large residual concentrations of important ferrous elements remain in the flotation tailings of advanced copper concentrator operations that were previously classified as processing waste; these are now widely considered as “anthropogenic ores” or artificial deposits (Lottermoser, 2010). The presence of a large magnetite (Fe₃O₄) fraction in the mineral processing tailings of massive sulfide deposits has revealed a promising technical route to recover these important phases. The valorization of these tailings is consistent with the basic principles of the circular economy and, furthermore, provides great economic advantages by transforming a conventional waste stream into a highly marketable industrial commodity.

The economic and technical viability of iron recovery from these secondary resources is closely related to the high-quality requirements of pyrometallurgical reduction processes, especially in the production of pig iron in blast furnaces or direct reduced iron (DRI). Pig iron is the basic intermediate feedstock for modern steelmaking processes and typically contains 3.5–4.7% carbon and very low limits on deleterious elements such as silicon, manganese, phosphorus, and sulfur (NLMK, 2021; Wills and Finch, 2015). Sulfur is usually perceived as the most problematic contaminant in the blast furnace charge (Soltanmohammadi et al., 2011). To avoid “hot shortness”, a serious structural defect that makes steel brittle at high temperatures, the steel industry has set strict chemical specifications on the iron ore

inputs, with the total sulfur content of the final iron concentrate or pellet feed required to be well below the limit of 0.1% to 0.2% (Rao and Leja, 2004; Soltanmohammadi et al., 2011).

Introducing high-sulfur concentrates directly into a blast furnace severely disturbs burden chemistry. Under reducing conditions, sulfur partitioning between molten pig iron and the slag phase is mainly controlled by thermodynamic temperature and slag basicity (CaO/SiO_2) (Bada et al., 2013; Nakhaei and Irannajad, 2018). High sulfur input requires operators to increase slag basicity or significantly change MgO and Na_2O concentrations to increase the sulfur-carrying capacity (sulfide capacity) of the slag phase. Any increase in slag basicity or slag volume significantly increases specific coke consumption, reduces fluid operating temperatures, and increases melt viscosity, all of which reduce the overall thermal efficiency and productivity of the blast furnace system. Therefore, the efficient desulfurization of iron concentrates prior to agglomeration (sintering or pelletizing) is important to produce high-purity pig iron and for stable and economical furnace operation.

The primary technological limitation in recovering iron from complex massive sulfide copper tailings, such as chalcopyrite-pyrite deposits in the Siirt-Madenköy region, is the mineralogical variability of the feed material. LIMS enables efficient separation of diamagnetic silicate gangue minerals at low magnetic field strengths (800–1000 Gauss) (Silva et al., 2021). Magnetite, owing to its strongly ferromagnetic properties, is highly suitable for LIMS. However, processing complex sulfide tailings solely via magnetic separation presents selectivity constraints. A major technical challenge lies in the complex mineralogical intergrowths and mineralogical entrapment of magnetite and iron sulfides, particularly FeS_2 (pyrite). During high-throughput wet magnetic separation, fine pyrite particles are easily mechanically entrained or physically trapped in rapidly developing magnetic flocs of magnetite, resulting in less selectivity and increased sulfur content of the magnetic fraction (Yao et al., 2020).

Reverse flotation is the dominant finishing process for low-grade iron concentrates today, overcoming structural constraints in meeting stringent desulfurization and silica-rejection requirements for high-quality pellet feed production (Silva et al., 2021). In reverse sulfide flotation, the iron oxide matrix is selectively depressed while specialized cationic collectors are used to float the sulfur-bearing or silicate gangue minerals into the foam fraction (Silva et al., 2021). Current mineral processing research mainly focuses on developing improved amine-based cationic collectors that are effective under highly alkaline conditions (pH 11.0 or above). In such situations, increased alkalinity naturally and highly selectively depresses magnetite surfaces (Deng et al., 2021; Li et al., 2023), altering the surface charge (zeta potential), and hydrolyzed amine molecules are strongly adsorbed on the negatively charged sulfide surfaces through ion–molecular complex formation ($\text{RNH}_2 - \text{RNH}_3^+$) (Filippov et al., 2014). This configuration allows effective sulfur removal without corrosive acidic conditions, avoiding equipment corrosion and reducing reagent costs.

Demagnetization, or the elimination of magnetization, is a crucial stage in wet magnetic separation circuits that directly improves separation selectivity. Because of residual magnetization, magnetite particles moving through the LIMS stage aggregate into magnetic flocs that physically trap undesirable fine gangue particles like silicates and pyrite, causing mechanical entrainment (Yao et al., 2020; Wills and Finch, 2015). The efficacy of subsequent inter-stage classification and reverse flotation procedures is greatly increased when this residual magnetism is reset using an alternating magnetic field, which disperses the flocs and releases the trapped gangue particles (Svoboda, 2004; Wills and Finch, 2015).

This study is part of a fully optimized two-stage beneficiation circuit to upgrade low-grade copper concentrator tailings from the ETİ Bakır Siirt plant into a high-grade, low-sulfur magnetite concentrate for industrial pig iron production. The proposed flowsheet is a three-stage LIMS setup running at a constant field intensity of 800 Gauss, systematically combined with inter-stage demagnetization to break magnetic flocs and minimize mechanical gangue entrainment. The pre-concentration stage is fully integrated with a reverse sulfide flotation circuit at a high alkaline pH of 11.0 using an amine-based cationic collector (Green Chemicals Mine-Treat 3452-1) to ensure full compatibility with the existing operating chemistry of the plant. The following sections provide a comprehensive assessment of the synergistic relationship between magnetic field kinetics, particle size distribution (d_{80}), and enhanced flotation kinetics, thus presenting a validated framework for efficient and selective recovery of iron from complex sulfide tailings.

2. Materials and methods

The tailings samples included in this investigation were extracted from the ETİ Bakır CO Siirt Madenköy copper concentrator facility, which processes massive sulfide copper ore. Chalcopyrite (CuFeS_2) is the deposit's primary copper mineral and is extracted by froth flotation. This study utilizes the discharge from the copper flotation circuit, generally considered a waste product, as feed material.

2.1. Chemical analysis

Table 1 presents the analysis results graphically.

Table 1. Chemical analysis results of Siirt sample (mass fraction, %)

Element	Fe	Cu	Zn	S	Magnetite
Grade	33.70	0.13	0.34	27.79	3.43
Method	*T	*ARD, ICP-OES		*LECO	*Satmagan

*ARD: Aqua regia digestion, *T: Titration method, *LECO: CS-844 elemental carbon-sulfur analyzer, *Satmagan: Saturation Magnetization magnetite analyzer by Outokumpu

2.2. Mineralogical analysis

The retrieved magnetite concentrate was analyzed with a mineral microscope utilizing reflected light on a polished section. A Leica DM2700 microscope with top illumination was utilized in the studies. The results indicated that the sample predominantly comprised the magnetite phase, with minor quantities of hematite and pyrite minerals present. Most magnetite was found in free form, whereas hematite was predominantly locked with pyrite. Fig. 1 shows the microscope images.

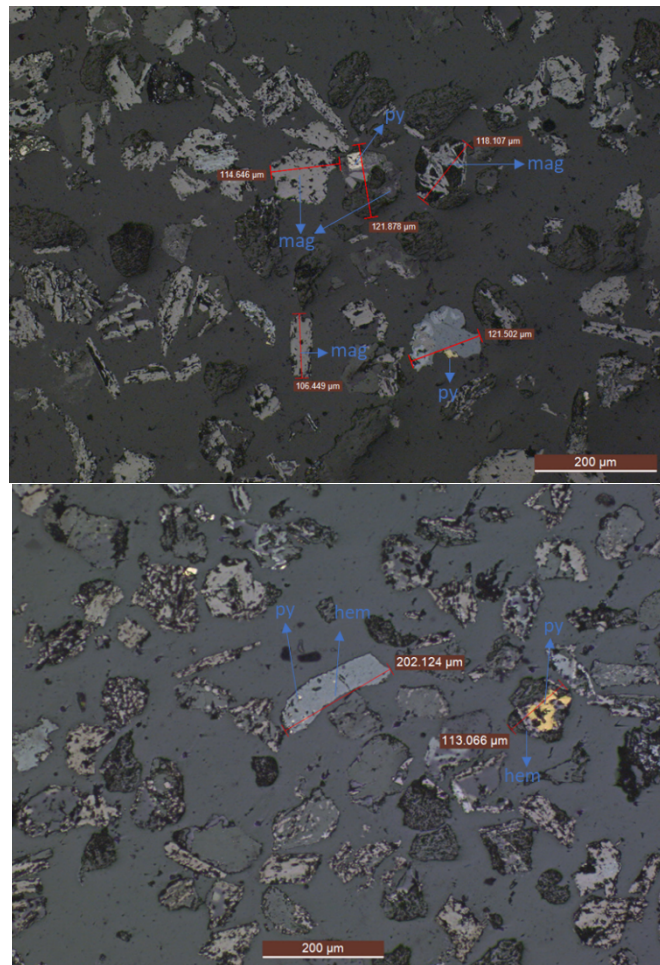


Fig. 1. The images from the ore microscope (Hem: hematite, py: pyrite, mag: magnetite)

An EDS elemental analysis of the magnetite concentrate sample was also performed using a scanning electron microscope. The SEM analyses were conducted using a Thermo Fisher Scientific Apreo 2C. The results of the analysis revealed that the iron content increased in the fine fractions. In the coarse fractions, however, an increase in gangue minerals was observed. Fig. 2 shows the EDS images captured using the microscope.

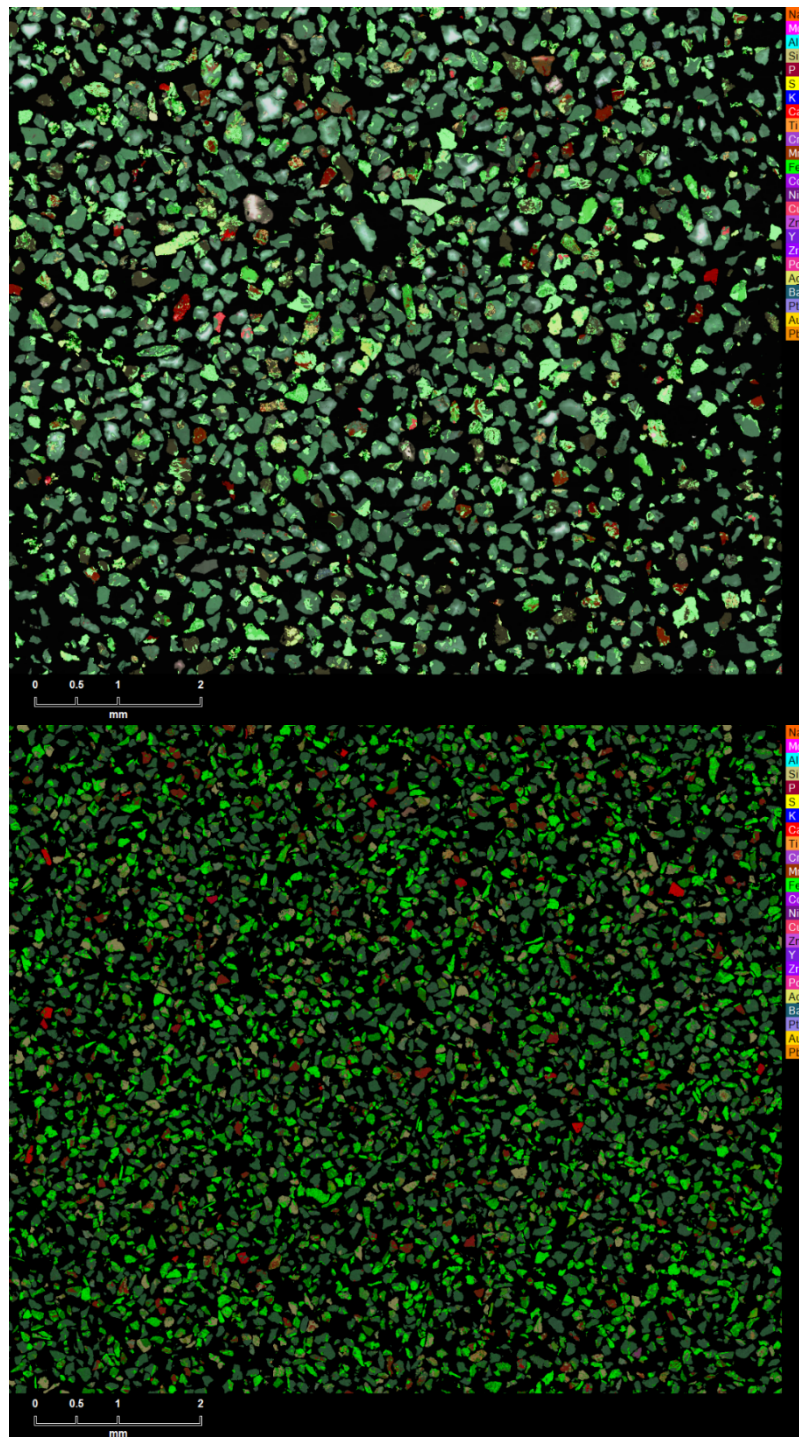


Fig. 2. The EDS images of magnetite concentrate captured using the scanning electron microscope

Table 2 presents the fractional modal mineralogy assay of the magnetite concentrate, based on MLA results. Using SEM-MLA, the fractional release rates of the minerals were determined. The release rate of magnetite increased as the analysis proceeded to smaller fractions. Table 3 presents the mineral release contents.

Table 2. Modal mineralogy table of magnetite concentrate by SEM-MLA

Mineral	-212+150	-150+106	-106+75	-75+53	-53+38	-38+20	-20
<i>Magnetite</i>	18.34	34.44	60.02	75.52	86.27	92.70	91.58
<i>Pyrite</i>	1.70	0.74	0.40	0.36	0.24	0.11	0.09
<i>Hematite</i>	0.22	0.39	0.66	0.86	1.40	1.10	2.12
<i>Almandine</i>	55.49	39.16	15.13	5.35	1.72	0.46	0.06
<i>Quartz</i>	4.64	4.75	3.98	3.07	1.56	0.77	0.24
<i>Ferroan Dolomite</i>	1.64	2.92	2.64	1.42	0.59	0.14	0.04
<i>Goethite</i>	0.65	1.39	2.55	2.57	1.82	1.14	3.05
<i>Chamosite</i>	2.33	3.05	1.77	1.04	0.44	0.14	0.02
<i>Other</i>	14.98	13.15	12.85	9.82	5.96	3.44	2.79

Table 3. Fractional liberation rates of magnetite concentrate sample

Mineral	-212+150	-150+106	-106+75	-75+53	-53+38	-38+20	-20
<i>Magnetite</i>	1.96	10.32	14.32	27.34	45.59	67.32	80.06
<i>Pyrite</i>	0.25	0.28	0.30	9.97	13.89	34.18	45.18
<i>Hematite</i>	0.00	0.00	0.07	0.12	0.19	0.46	9.36
<i>Almandine</i>	8.34	6.11	6.26	8.87	17.23	22.78	24.49
<i>Quartz</i>	8.09	9.86	9.61	11.36	11.63	20.90	27.96
<i>Ferroan Dolomite</i>	0.16	0.59	1.45	3.91	11.27	11.57	30.67
<i>Goethite</i>	0.13	0.11	0.10	0.07	0.13	0.46	20.22
<i>Chamosite</i>	0.04	0.01	0.04	0.03	0.22	0.37	12.23
<i>Olivine</i>	0.18	0.20	0.20	0.12	0.25	0.16	16.65

2.3. Magnetic separation tests

The pre-concentration of magnetite was conducted using a Low-Intensity Magnetic Separator (LIMS). The operation was performed at a constant magnetic field intensity of 800 Gauss. This stage aimed to separate the ferromagnetic magnetite fraction from the non-magnetic silicate matrix and some sulfide minerals. The resulting magnetic concentrate was collected as the feed for the subsequent cleaning stage. Table 4 presents the parameters for the magnetic separation experiments.

The Davis Tube test was conducted to recover magnetite from the sulfur concentrate obtained through sulfur flotation. An ERIEZ Model EDT Davis Tube tester was used in the experiments. Table 5 lists the experimental parameters.

Table 4. Parameters of magnetic separation (LIMS) tests

Drum Speed (rpm)	Feed Size (d80) (μm)	Solid Ratio (%)	Magnetic Field (Gauss)	Drum Dimensions (mm)	Flowrate (dm ³ /min)
30	68.41	28-30	800 (Natural Magnet)	Ø 200x300	3.5

Table 5. Parameters of Davis Tube test on sulfur concentrate

Magnetic Field (Gauss)	Amount of Sample (g)	Tube Speed (rpm)	Tube Angle (degree)
800	60	1200	45

2.4. Flotation tests

Process parameters were selected for the reverse sulfide flotation stage to reject residual pyrite from the LIMS concentrate selectively. The pulp pH was adjusted to 11.0 to meet the operating conditions of the copper concentrator facility. This allowed for a seamless integration into the industrial process and eliminated the need for additional pH adjustment chemicals. Under this very alkaline regime, the amine-based cationic collector Green Chemicals Mine-Treat 3452-1 was applied for selective flotation of

sulfide minerals from magnetite surfaces. According to the reverse flotation principle, pyrite and other sulfur-containing minerals were floated in the froth fraction. The experiment produced a magnetite concentrate with lower sulfur content and higher iron grade. Table 6 presents the sulfur flotation details.

Table 6. Sulfur flotation test parameters

Collector (3452-1) (g/Mg)	Frother (MIBC) (g/Mg)	Solid (g)	Cell Volume (cm ³)	pH
50	50	500	1200	11
Rotation Speed (rpm)	Air Feed (dm ³ /min)	Conditioning Time (min)	Flotation Time (min)	Solid Ratio (%)
850	2	5*	5	31.88

*After the addition of the collector, the conditioning was allowed for 5 min. The frother was added after the 4th min.

Note: The flotation parameters presented in Table 6 were applied consistently across all experimental runs. To verify process reproducibility, reverse sulfide flotation tests were initially carried out in triplicate (n = 3) under these identical conditions. Following the consistent and highly repeatable results obtained from these initial runs, all subsequent flotation experiments were maintained under the exact same parameters. The experimental results demonstrated high statistical repeatability, yielding a final magnetite concentrate with mean values of 64.26 ± 0.51% Fe, 0.31 ± 0.09% S, and 78.24 ± 0.66% Magnetite, thereby confirming the reliability of the flotation circuit.

2.5. Reduction of magnetite and pig iron production

The resultant magnetite concentrate underwent a sequence of pyrometallurgical procedures for refinement. Initially, the magnetite concentrate was combined with coal and petcoke before undergoing pelletization. Table 7 lists the ratios of the products used in the pelletizing process.

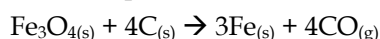
Table 7. Quantities and compositions of the products used in the pelletization

Product	Mass (g)	Mass (%)	Fe (%)	*C (%)	*S (%)
Magnetite Concentrate	9,000	76.53	64.68	0.00	0.25
Coal	1,290	10.97	0.00	99.50	0.23
Petcoke	1,290	10.97	0.00	95.00	5.00
Bentonite	90	0.77	0.00	0.00	0.00
Lime	90	0.77	0.00	0.00	0.00
Total	11,760	100.00	49.50	21.34	0.77

*C, S elements measured with LECO C-S analyzer

The initial moisture content was established at 5%, and a final moisture content of 13% was attained post-pelletizing. The pellet dimensions were established at -8+3 mm.

The pelletized samples were subjected to a furnace previously heated temperature of 1000°C for a duration of 30 min. During the reduction tests, a Protherm PLF 130/15 chamber-type ash furnace was used. The reduction process was carried out by directly reducing magnetite ore with carbon (Fruehan, 1977). The reaction involved in the reduction process is as follows:



This procedure culminated in reduced “sponge iron”. Fig. 3 shows images of the pellets during the roasting process.



Fig. 3. Images from reduction process

The sponge iron was introduced into an induction furnace for conversion into pig iron. Pig iron was produced through the smelting process. An Inductotherm VIP-1 induction furnace was used in the production of pig iron. Images of the melting process are shown in Fig. 4.



Fig. 4. Images from pig iron production

1.4 kg of ST-37 steel was utilized to establish the bath environment within the induction furnace. The yield of the pig iron was determined by summing the weight of the steel component with the total calculation. Table 8 shows the total iron content of the products fed into the induction furnace.

Table 8. Composition of the material fed into the induction furnace

Product	Mass (kg)	Mass (%)	Fe (%)
ST-37 Steel	1.40	13.46	98.50
Magnetite Concentrate	9.00	86.54	64.68
Total*	10.40	100.00	69.23

*Calculated

3. Results and discussion

3.3. Magnetic separation tests

Since the total iron (Fe) content in massive sulfide ore structures encompasses iron derived from both magnetite and pyrite, the enrichment performance in this study was evaluated based specifically on the magnetite content to ensure metallurgical accuracy. According to Fig. 1 and Table 1, the three-stage LIMS circuit operating at 800 Gauss showed high selectivity toward magnetite minerals. The magnetite content, which was initially only 3.43% in the feed, increased systematically to 64.18% in the final magnetic concentrate, aided by the disruption of magnetic flocs via inter-stage demagnetization. These results indicate that the magnetic separation circuit achieved a significant enrichment ratio of approximately 18.7 for the targeted magnetite mineral.

Magnetite recovery reached 79.38% after the three stages of separation. The concurrent reduction in impurities, such as sulfur (from 27.79% to 7.43%) and copper (from 0.14% to 0.06%), confirms the successful rejection of the pyrite-associated iron fraction and the enrichment of the product in terms of magnetite. However, the residual sulfur content of 7.43% in the final magnetic concentrate suggests that certain pyrite particles remain either mineralogically interlocked with magnetite or were subject to residual magnetic entrainment. Notably, the overall iron recovery remained as low as 7.90%, which serves as a concrete technical indicator that the non-magnetic iron within the pyrite structure was successfully diverted to the tailings, thereby validating the process's focus on selective magnetite recovery. Fig. 5 shows the current diagram and metallurgical balance tables for the three-stage magnetic separation test. After each magnetic separation stage, the product was demagnetized to prevent particle aggregation and mineral entrapment. The demagnetizer operates by exposing samples to a reverse current via its internal coil.

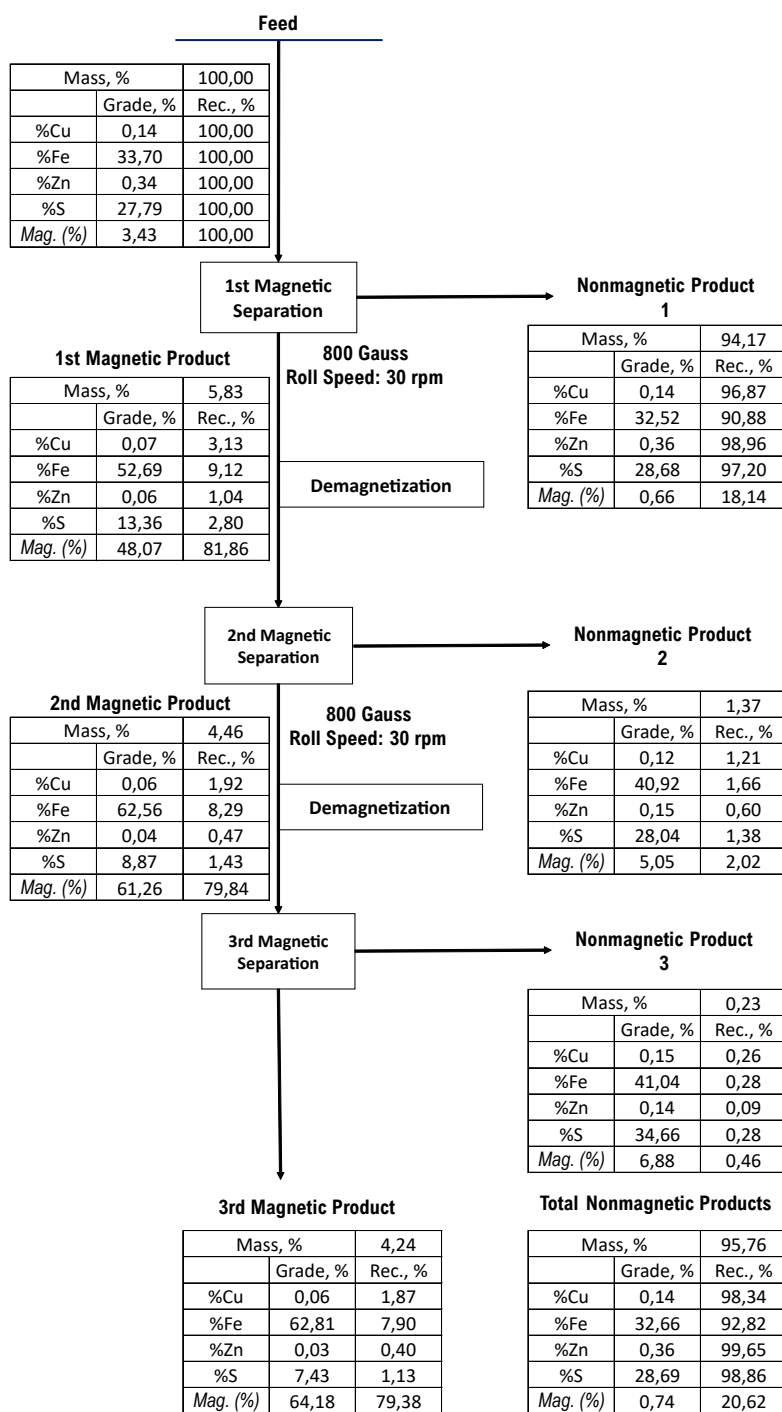


Fig. 5. Flowsheet and metal balance tables for magnetic separation tests

3.4. Flotation tests

The magnetic concentrate from the third stage has a residual sulfur concentration of 7.43% and a magnetite grade of 64.18%, requiring a final purification step via reverse sulfide flotation. The selective separation of magnetite and pyrite was successfully performed using an amine-based collector (Green Chemicals 3452-1) at a dosage of 50 g/Mg in a strongly alkaline medium (pH 11). At this stage, most residual sulfide minerals were effectively removed into the froth product, which contained 15.94% sulfur.

The sink product, the final magnetite concentrate, was of considerably higher quality. The sulfur level was reduced considerably from 7.43 % to 0.25 % S, which is almost the industrial standard for iron ore concentrates. At this stage, the magnetite grade increased from 64.18% to 77.50%, with a partial

magnetite recovery of 65.48%. The reduction of copper content from 0.05% to 0.02% and the constant zinc content of 0.03% further confirm the efficiency of reverse flotation in removing non-ferrous sulfide impurities. The results show that LIMS is effective for initial concentration, but it needs to be combined with reverse flotation to get high-grade magnetite purity from complex massive sulfide tailings. Table 9 presents the sulfur flotation test results.

Table 9. Sulfur flotation of magnetic concentrate's test results

Products	Mass (g)	Mass (%)	Grade (%)					Recovery (%)				
			Cu	Fe	Zn	S	Mag.	Cu	Fe	Zn	S	Mag.
<i>Sulfur Concentrate</i>	228.50	45.77	0.10	63.58	0.04	15.94	48.40	82.51	45.34	54.57	98.18	34.52
<i>Sink Product</i>	270.71	54.23	0.02	64.68	0.03	0.25	77.50	17.49	54.66	45.43	1.82	65.48
<i>Total*</i>	499.21	100.00	0.05	64.18	0.03	7.43	64.18	100.00	100.00	100.00	100.00	100.00

*Calculated values

To provide a better context for the performance of the proposed flow sheet, the metallurgical findings of this study were compared with recent literature studies on magnetite recovery and sulfur removal from copper plant residues. In a similar study, the material was ground to 20 microns (d80) to obtain a magnetite concentrate from copper plant residues. As a result of the grinding, a magnetite concentrate containing over 67% Fe was obtained (Silva et al., 2021). In the present study, a regrinding step was not required as target Fe and S grades were successfully attained. This had a beneficial effect on reducing capital and operating costs. Another benefit is eliminating the regrind step and preserving the coarse particle size distribution, which prevents mechanical entrainment of magnetite into the froth during the reverse flotation stage (Neethling and Cilliers, 2009; Zheng et al., 2020).

Based on the overall recovery rates, 27.37% of the total magnetite was recovered in the sulfur concentrate. To recover the magnetite from the sulfur concentrate, a magnetic separation test was conducted using a Davis Tube. Based on the high S content of the recovered product (14.51%), it was determined that effective recovery was not attainable due to the presence of unliberated magnetite-pyrite middlings. Table 10 presents the results of the Davis Tube test conducted on the sulfur concentrate.

Table 10. The results of the Davis Tube test conducted on the sulfur concentrate

Products	Mass (g)	Mass (%)	Grade (%)					Recovery (%)				
			Cu	Fe	Zn	S	Mag.	Cu	Fe	Zn	S	Mag.
Magnetic	5.73	9.54	0.09	62.72	0.04	14.51	53.78	8.67	9.69	8.75	8.69	10.60
Nonmagnetic	54.31	90.46	0.10	61.68	0.04	16.09	47.83	91.33	90.31	91.25	91.91	89.40
Total*	60.04	100.00	0.10	61.78	0.04	15.94	48.40	100.00	100.00	100.00	100.00	100.00

3.5. Pig iron production test results

9 kilograms of magnetite concentrate were utilized for pig iron production. Along with 9 kg of concentrate, 2.580 g of coal containing 97.25% carbon, 90 g of bentonite, and 90 g of lime were used. These materials were mixed and pelletized. After the reduction and casting processes, a total of 6.27 kg of pig iron ingots was produced. The ingots contained 98.76% iron. Iron ingots with a purity of 98.76% were produced in an induction furnace with a recovery rate of 86%. Table 11 presents the metallurgical calculations for pig iron production.

Table 11. Metallurgical calculations of pig iron production

Product	Mass (kg)	Mass (%)	Grade Fe (%)	Recovery Fe (%)
Pig Iron	6.27	60.29	98.76	86.00
Slag	4.13	39.71	24.41	14.00
Furnace Feed	10.40	100.00	69.23	100.00

4. Conclusions

This study demonstrates the technological feasibility of recovering high-quality magnetite concentrate from complex massive sulfide tailings through an integrated beneficiation circuit. The multi-stage Low-Intensity Magnetic Separation (LIMS) circuit efficiently enhanced the magnetite concentration, increasing its value from 3.43% in the feed to 64.18% in the final magnetic product. A crucial element of this success was implementing demagnetization between LIMS stages to prevent magnetic flocculation and significantly reduce mechanical entrapment of pyrite and silicate gangue particles. The significant enrichment achieved through magnetic forces resulted in a residual sulfur concentration of 7.43%, necessitating final purification.

The Mine Treat 3452-1 collector was employed in reverse sulfide flotation at pH 11.0 to effectively address this difficulty and selectively eliminate the residual sulfides, resulting in a sulfur content reduction to an astounding 0.25% S. The integrated process yielded a magnetite concentrate with a purity of 77.50% and minimal impurities, achieving an overall magnetite recovery of 52.06%. The results show that the combination of LIMS, inter-stage demagnetization, and reverse flotation constitutes a process that holds great promise for converting low-grade sulfur waste into industrial-grade raw material. Table 12 shows the metal recovery rates of the final magnetite concentrate, based on the feed material.

Table 12. Overall magnetite concentrates recovery rate table

Products	Mass (%)	Grade (%)					Recovery (%)				
		Cu	Fe	Zn	S	*Mag.	Cu	F	Zn	S	*Mag.
<i>Magnetite Concentrate</i>	2.30	0.02	64.68	0.03	0.25	77.50	0.34	4.38	0.20	0.02	52.06
<i>Final Tailings</i>	97.70	0.14	33.24	0.35	28.43	1.68	99.66	95.62	99.80	99.98	47.94
<i>Feeding*</i>	100.00	0.14	33.96	0.34	27.78	3.42	100.00	100.00	100.00	100.00	100.00

*Mag.: Magnetite, *Feeding: Calculated values

Using the magnetite concentrate obtained, pig iron with a purity of 98.76% was produced. In conclusion, recovering the iron content, which constitutes a significant portion of the facility's waste, as a secondary resource directly reduces the total amount of pulp sent to tail dams. The high-quality, low-sulfur iron concentrate obtained is used in the iron and steel industry as pellet feedstock or as an alternative raw material, thereby reducing dependence on primary ore mining and fully supporting the principles of the circular economy. From an economic perspective, reduced environmental liabilities and disposal costs, coupled with the introduction of a new value-added product to the market, support the commercial sustainability of the proposed flow scheme by balancing the energy requirements during the LIMS stage with flotation reagent costs.

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