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Development a new flotation refining technology for magnetite concentrates at the Northern Mining and Processing Plant Private Joint-Stock Company

Abstract. The purpose of this study was to develop a new flotation refining technology for magnetite concentrates at the Northern Mining and Processing Plant Private Joint-Stock Company to produce a final commercial product with a total iron content of at least 69.0% and silica content not exceeding 4%. Mineralogical analysis was used to study mineral liberation, chemical analysis to determine the qualitative composition of the raw material, and sieve analysis to determine the granulometric composition of the material. The study also employed methods of processing, analysing, and synthesising research results to establish optimal conditions for separating mineral grains by size. The results showed that achieving the maximum total iron content in the concentrate and minimising the silica content, while ensuring maximum iron recovery, depends on grinding conditions, which are determined by the degree of mineral grain liberation. It was found that the flotation rate constant for quartz depends on the mass fraction of quartz in the feed and ranges from 0.05 to 0.10 min^{-1}. Mineralogical studies of the raw concentrate revealed that, in addition to monomineral magnetite particles, it includes aggregates of magnetite-quartz (up to 2.52%), magnetite-quartz-silicate, magnetite-quartz, silicate-carbonate (up to 7.5%). Carbonates are mainly represented by sideroplesite and calcite; silicates by cummingtonite, biotite, and chlorite. Gangue particles consist of monomineral quartz or silicate-quartz, carbonate-silicate-quartz aggregates. According to quantitative mineralogical calculations, monomineral magnetite particles are concentrated in the fine-grained fraction, with 92.3% in the -0.045 + 0.02 mm class and 98.1% in the -0.02 mm class. Laboratory tests of the magnetic-flotation refining of the concentrate demonstrated the feasibility of obtaining high-quality concentrate from the magnetite concentrates of the Northern Mining and Processing Plant Private Joint-Stock Company with a total iron content of 69.0% and silica content of 3%, achieving an 88.9% yield and 92.9% iron recovery in the concentrate from the operation. The practical significance lies in the development of a technological scheme for refining raw magnetite concentrate under the conditions of the Northern Mining and Processing Plant Private Joint-Stock Company to produce high-quality concentrate with

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maximum yield, a total iron content above 69.0%, and silica below 4.0%, while ensuring iron recovery in the concentrate of more than 90%

Keywords: iron ore concentrate; reverse cationic flotation; magnetic separation; mass fraction; granulometric composition

Introduction

As of 2024, Ukraine holds a prominent position in the global market for the extraction and processing of iron ore quartzites. Mining and processing enterprises in the Kryvyi Rih Iron Ore Basin play a significant role in this sector. The Kryvyi Rih plants supply raw materials to ferrous metallurgy enterprises both within Ukraine and abroad. However, the quality of iron ore concentrates from Kryvyi Rih beneficiation plants, with an iron content of 65-68% and SiO_2 content of 6.0-9.0%, is considerably lower than that of similar plants in Canada, Sweden, and the USA. Such factors reduce the marketability of Kryvyi Rih enterprises and diminish the competitiveness of Ukrainian products.

The quality of iron ore concentrate is crucial for further processing and is primarily determined by two key factors: the content of the useful component and the level of harmful impurities. Hence, there is a growing demand for high-quality concentrates with a high content of the useful component and low silica content, suitable for direct iron reduction. However, Ukrainian plants face challenges due to beneficiation technologies developed in the 20th century, which do not ensure the production of competitive quality concentrate for the global market. Thus, the development of new enrichment methods and techniques has become a pressing issue. This challenge is being addressed by both mining and processing enterprises and researchers.

The study by M.M. Bulayani *et al.* (2024) highlights that the beneficiation of low-grade ores is a critical area of research in the mineral processing sector. Despite their lower iron quality and high impurity levels, low-grade iron ores constitute a significant portion of global iron ore reserves. The beneficiation of low-grade iron ore is an essential process for utilising ore deposits, especially as demand for iron and its alloys continues to grow due to rapid industrialisation and the depletion of high-quality reserves. The mineralogy of iron ore and impurity levels dictate the beneficiation pathways for obtaining high-quality iron ore concentrate. Effective beneficiation processes not only enhance the economic viability of low-grade ore deposits but also promote sustainable resource management and environmental conservation. The study discusses various methods of low-grade iron ore beneficiation, including grinding, gravity separation, flotation, and magnetic separation.

In the work of A.F.D.V. Rodrigues *et al.* (2023), it was proven that reverse cationic flotation technology is the most widely used method for fine iron ore beneficiation. However, this technology faces numerous inefficiencies in the technological cycle, including fine grinding, desliming, flotation, and product thickening. The study explores new advances in crushing and grinding technologies, as well as magnetic separation and gravity methods, as alternatives or complete replacements for reverse flotation technology. Researchers have demonstrated that innovative approaches to iron ore beneficiation will play a decisive role in the transition of the global industry to so-called "green steel".

In the work of X. Zhang *et al.* (2019), it was established that flotation is considered the most promising method for obtaining high-quality iron from low-grade, finely disseminated iron ores. Flotation can be used as a standalone method or as a complement to other separation techniques, such as gravity separation and magnetic separation. Flotation is highly dependent on the chemical composition of the mineral surfaces to be separated. Reverse flotation is the most promising flotation pathway for further development. The choice between cationic or anionic reverse flotation depends on the mineralogy and available reagents. Cationic reverse flotation offers advantages over anionic flotation due to its higher flotation speed, simpler reagent systems, operational simplicity and reliability, and suitability for operation at low temperatures. However, cationic collectors have relatively poor selectivity, foaming properties, and high toxicity, which can lead to product losses and environmental pollution. Anionic reverse flotation is less sensitive to slimes, has a lower collector cost, and higher selectivity due to calcium activators, which have an activating effect on quartz and a depressing effect on iron ore. However, anionic reverse flotation requires significant quantities of activators, as well as high temperatures and alkalinity of the process.

In the study by B. Luo *et al.* (2021), the new amphoteric surfactant LDEA was investigated. The research showed that during froth flotation, the properties of the collector, especially its solubility in the pulp, adsorption on the mineral surface, and hydrophobicity, play a crucial role in achieving optimal iron content and recovery rates. It was determined that pH stabilisation affects not only the chemical composition of the surface but also the ionisation state of the collector, which determines its adsorption mechanism on mineral surfaces and the efficiency of particle separation.

In the work of J.T.G. Junior *et al.* (2023), it was proven that flotation is the most widespread method for processing iron oxides in the typical size range of -150+40 μm. The study noted that ultrafine particles below 40 μm exhibit low collision rates with air bubbles, which risks the loss of fine and ultrafine magnetite and haematite particles alongside quartz. Researchers demonstrated that the use of cationic surfactants, such as diamine LILAFLOT-811M and monoamine ether LILAFLOT-919, showed high efficiency in processing ultrafine particles. Both biodegradable collectors effectively separate magnetite and quartz. At pH-9, both collectors facilitated quartz recovery of 95.9% (with diamine LILAFLOT-811M) and 97.7% (with monoamine ether LILAFLOT-919) and exhibited relatively similar flotation kinetics.

In the study by T. Oliinyk *et al.* (2023), researchers examined the challenges of developing efficient iron ore beneficiation technologies and applying various enrichment methods, including gravity, magnetic, and flotation techniques. The ore under investigation contained particles below 74 μm. The flotation enrichment reagent regime consisted of Lilaflot-811M, Lilaflot-D817M, and Lilaflot-D819M as collectors, dextrin as a depressor for iron ore minerals, and caustic soda as a medium regulator. It was proven that the most efficient beneficiation schemes included two stages of ore grinding, wet magnetic separation, and reverse flotation in two steps with regrinding of the froth product. Another effective scheme involved pre-enrichment of the ore using dry magnetic separators, two-stage grinding to a fraction below 74 μm, wet magnetic enrichment, and reverse cationic flotation in two steps with regrinding of the froth product.

The technological potential for enriching magnetite quartzites of the Northern Mining and Processing Plant Private Joint-Stock Company (PJSC "Northern MPP") has been studied between 2021 and 2024 but remains inconclusive. This allows for a more detailed investigation of the magnetic, flotation, and magnetic-flotation refining technologies to produce a competitive product. Therefore, the aim of this work was to determine the optimal technological scheme for flotation refining of magnetite concentrates at PJSC "Northern MPP" to obtain a final commercial product with a total Fe content of at least 69.0% and SiO_2 content not exceeding 4%.

Materials and Methods

To determine the feasibility and necessity of refining magnetite quartzites under laboratory conditions, experiments were conducted on the beneficiation of raw concentrate from PJSC "Northern MPP". The chemical composition of the concentrate revealed the following: total iron content – 66.0%, magnetite iron content – 62.3%, iron oxide – 27.6%, harmful impurities: silica (SiO_2) – 6.79%, titanium dioxide (TiO_2) – 0,041%, aluminium dioxide (Al_2O_3) – 0.18%, manganese oxide (MnO) – 0,038%, calcium oxide (CaO) – 0.47%, magnesium oxide (MgO) – 0.53%, phosphorus oxide (P_2O_5) – 0,018%, sulphur (S) – 0.07%, carbon dioxide (CO_2) – 0.33%, potassium oxide (K_2O) – 0,053%, sodium oxide (Na_2O) – 0.15%, loss on ignition (LOI) – 0,062%. The granulometric composition of the raw concentrate is presented in Table 1.

Table 1. *Granulometric characteristics of ordinary concentrate*

Size class, mm					
+0.071	-0.071 +0.056	-0.056 +0.045	-0.045 +0.02	-0.02	Total
0.7	1.1	2.4	36.5	59.3	100

Source: *authors' development*

The tests were conducted at the laboratory of PJSC "Northern MPP" and the Department of Mineral Processing at Kryvyi Rih National University. The pre-grinding of the concentrate was performed using a laboratory mill of the "Rollgang" type, designed to simulate the grinding processes of industrial mills. Magnetic beneficiation was carried out using a laboratory magnetic analyser AM-2A (NTC MAGNIS LTD, Luhansk, Ukraine) with a magnetic field induction of 1,000 G. Flotation tests on the concentrate were conducted using a laboratory flotation machine "237 FL" (SCMA Lab, Kryvyi Rih, Ukraine), equipped with interchangeable cells of 0.5 and 1.0 litres capacity. Laboratory studies were performed based on three technological schemes for enriching the raw concentrate: 1. Magnetic refining scheme. This scheme included preliminary grinding of the entire raw concentrate, classification in hydrocyclones, and magnetic beneficiation. The scheme is illustrated in Figure 1. 2. The flotation refining scheme. This scheme involved a reverse cationic flotation cycle. The froth product of the main flotation cycle was classified in a closed circuit with the mill, enriched using a magnetic separator with low magnetic induction, processed in a control flotation cell, and returned to the main flotation process. The flotation process was carried out with the addition of the amine reagent Lilaflot 811M. The scheme is illustrated in Figure 2. 3. The magnetic-flotation refining scheme. This scheme involved the classification and grinding of raw concentrate in a closed circuit. The classifier overflow was directed to magnetic separation and subsequently to the main flotation cells, followed by the enrichment of the cell product using magnetic separators. The froth product was classified and ground in a closed circuit. The hydrocyclone overflow, after magnetic beneficiation and control flotation, was returned to the main flotation cycle. The main flotation process was conducted with the addition of the amine reagent Lilaflot 811M. The scheme is illustrated in Figure 3.

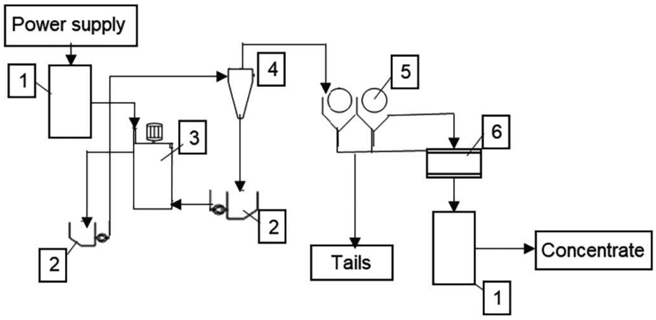

Figure 1. *Technological scheme No. 1 for magnetic refining of raw concentrate*
Notes: *1 – mixer (homogeniser) for feed product; 2 – technological sumps with pumps; 3 – vertical mill; 4 – hydrocyclone battery; 5 – wet magnetic separation; 6 – magnetic desliming unit*
Source: *authors' development*

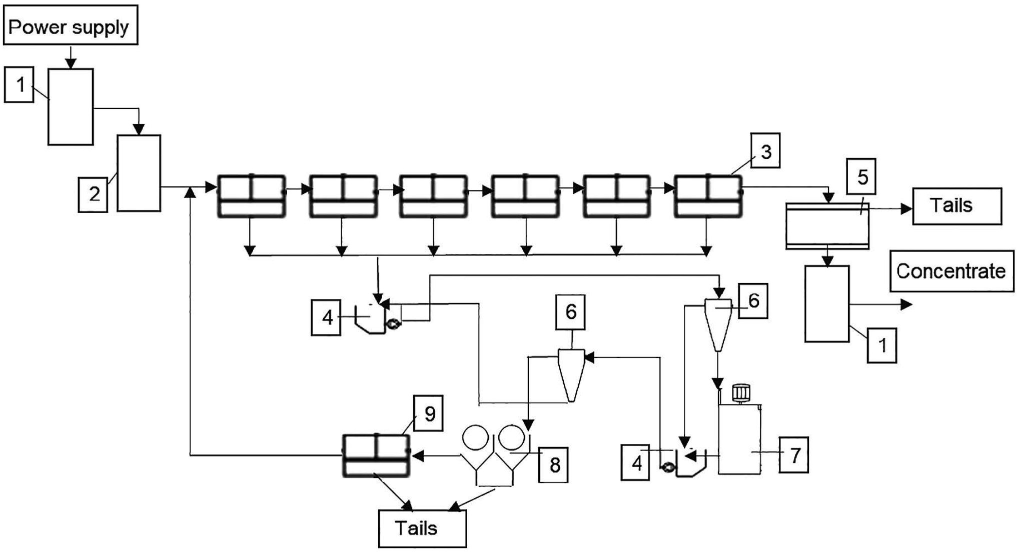

Figure 2. *Technological scheme No. 2 for flotation refining of raw concentrate*
Notes: *1 – mixer (homogeniser) for feed product; 2 – contact tank with reagent; 3 – main flotation cells; 4 – technological sumps with pumps; 5 – magnetic desliming unit; 6 – hydrocyclone batteries; 7 – vertical mill; 8 – wet magnetic separation; 9 – control flotation cell*
Source: *authors' development*

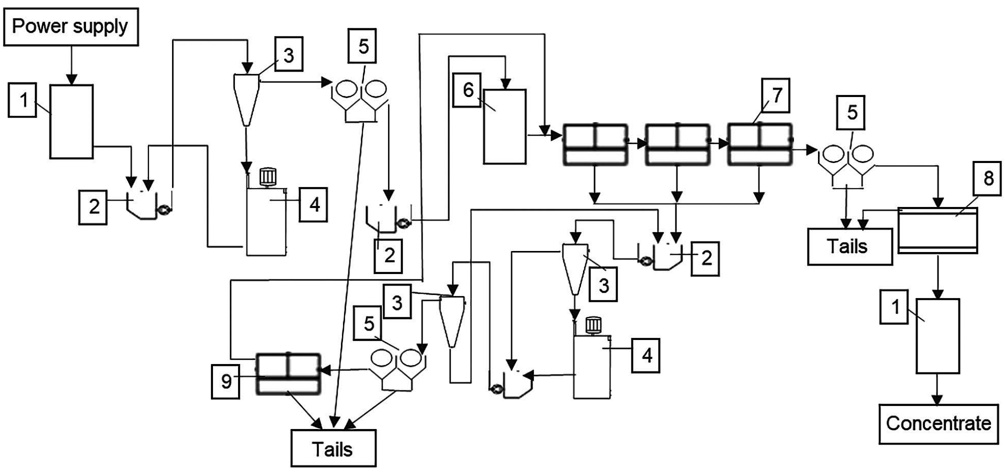

Figure 3. *Technological scheme No. 3 for magnetic-flotation refining of raw concentrate*
Notes: *1 – mixer (homogeniser) for intermediate product; 2 – main flotation cells; 3 – hydrocyclone batteries; 4 – vertical mill; 5 – wet magnetic separation; 6 – contact tank with reagent; 7 – main flotation cells; 8 – magnetic desliming unit; 9 – control flotation cell*
Source: *authors' development*

The research was conducted using the reverse cationic flotation method, employing the cationic reagent amine Lilaflot 811M, produced by AKZO NOBEL. The process did not include the addition of hydrolysed starch as an iron depressant due to the presence of poor, non-liberated aggregates in the flotation feed, which are susceptible to the depressing action of hydrolysed starch, negatively affecting concentrate quality. Additionally, no pH stabiliser was used.

Flotation and reagent conditions were identical across all experiments and included:

1. Amine dosage: 200 g/t;
2. Contact time: 1 minute;
3. Solid content in the flotation feed: 30-35%;
4. Flotation duration: 10 minutes.

Standard research methods applied:

1. Sieve analysis was conducted in accordance with DSTU 3704:2013 (2013).
2. Flotation analysis was conducted in accordance with DSTU 8811.1:2018 (2018).
3. Chemical analysis was conducted in accordance with DSTU 8811.0:2019 (2019).

Results

An analysis of comparable plant operations demonstrated that achieving high beneficiation efficiency depends on improving concentrate quality, optimising raw material utilisation, and reducing specific beneficiation costs. These goals are achieved through various methods, including flotation refining (Bulayani *et al.*, 2024). The feasibility of implementing and selecting a technological refining scheme depends on multiple factors, including the physico-chemical properties and characteristics of the feed material, plant capacity and productivity, available space for refining operations, and the essential testing, especially semi-industrial trials. The beneficiation technologies currently used in Ukraine, particularly at PJSC "Northern MPP" are incapable of producing high-quality concentrates and require improvement. Enhancement efforts focus on incorporating additional refining operations for raw concentrate, such as extra grinding, fine screening, using magnetic separators with varying field gradients and intensities, hydraulic separation, and flotation methods. The Kirkenes plant in Norway refines its raw concentrate through additional grinding and reverse cationic flotation, achieving iron content levels up to 70%. Plants such as Adams (USA), Sherman and Griffith Mines (Canada), the Poltava Mining and Processing Plant, and the Ingulets Mining and Processing Plant (Ukraine) have effectively improved the quality of magnetite concentrate by integrating reverse cationic flotation operations into their technological schemes. Hydraulic fine screening operations are also widely used in mining and processing plants in the USA and Canada, first introduced in 1967 at the Erie Mining plant in the USA. This technology has also been implemented at the Central Mining and Processing Plant (Ukraine), leading to increased quality and productivity (Hubin *et al.*, 2016).

All beneficiation processes, including flotation, are based on the differences in the properties of the minerals being separated. The primary phenomena utilised in flotation are surface tension and wettability. Under specific conditions, these properties ensure the effective progress of flotation. For efficient flotation, selecting the appropriate flotation equipment is critical. Each type of flotation machine has unique characteristics and structural features, and specific machines are used for the flotation of various minerals (Bulayani *et al.*, 2024). Flotation machines are enrichment devices used to separate minerals during the flotation process in a water-mineral pulp. The flotation process is influenced by factors such as flotation time, stirring, and aeration methods. These parameters determine the type, volume, and number of flotation machine cells as well as compliance with energy consumption standards (Smirnov & Biletskyi, 2010).

At PJSC "Northern MPP" raw magnetite concentrate is produced through magnetic beneficiation. The raw material composition for magnetite concentrate exhibits unstable quality indicators (Dovhiy *et al.*, 2017). The composition of the iron ore feed is characterised by mineralogical varieties such as magnetite, haematite-magnetite, silicate-magnetite, and magnetite-silicate. The structure is fine- and microcrystalline, with aggregates of heavy ore grains of magnetite and haematite bound with gangue minerals such as quartz, silicates, and carbonates (Filenko, 2011). The material also contains martite, biotite, iron mica, amphiboles, and cummingtonite. Due to variations in the proportions of easily, moderately, and poorly beneficiated ore minerals, as well as differing levels of harmful gangue impurities, the concentrate displays variable characteristics in its mineralogical, chemical, and granulometric composition (Morkun *et al.*, 2017).

Mineralogical studies revealed that the concentrate predominantly consists of the ore mineral magnetite (82.6%). The samples also contain iron hydroxides, silicates, and carbonates (totalling 4.14%). The gangue component is quartz, with a content of 6.63-7.5%. Harmful impurities include sulphides and apatite, with contents around 0.16%. Mineralogical studies of the raw concentrate also identified aggregates such as magnetite-quartz (up to 2.52%), magnetite-quartz-silicate, magnetite-quartz, and silicate-carbonate (up to 7.5%). The carbonates are primarily represented by sideroplesite and calcite, while silicates include cummingtonite, biotite, and chlorite. Non-ore particles include monomineral quartz or silicate-quartz, carbonate-silicate-quartz aggregates. Quantitative mineralogical calculations revealed that monomineral magnetite particles accumulate in fine fractions class -0.045+0.02 mm: 92.3%; class -0.02 mm: 98.1%. In coarser fractions, the proportion of monomineral particles decreases, while the proportion of poor aggregates and gangue particles increases. For instance, class +0.071 mm: 62.7% poor aggregates, and 6.9% gangue particles; class -0.071+0.056 mm: 41.7% poor aggregates, and 6.2% gangue particles; class -0.056+0.045 mm: 16.8% poor aggregates, and 2.6% gangue particles.

Laboratory studies demonstrated a direct correlation between silica content and iron content: as the proportion of ore minerals increases, the proportion of gangue minerals decreases, and vice versa. The results are presented in Figure 4.

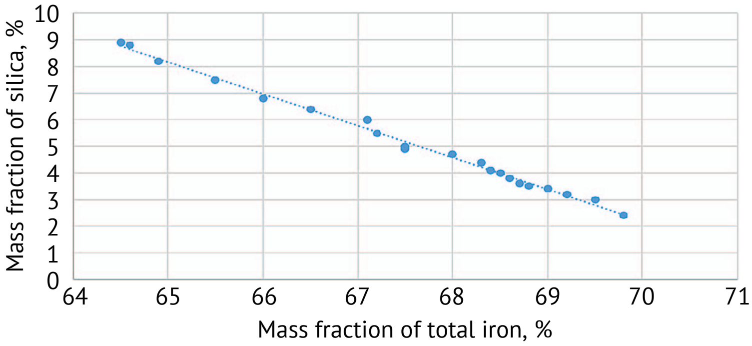

Figure 4. *Relationship between the mass fraction of total iron and silica*

Source: *authors' development*

The analysis of the studies demonstrated that using magnetic, magnetic-flotation, and flotation refining methods for raw concentrate with the reagent collector Lilaflot 811M makes it possible to obtain a final concentrate with a total iron mass fraction ranging from 68.0 to 69.0%, recovery rates between 92.9 and 95.7%, and silica content in the concentrate from 3.4 to 4.7%. The results of the studies are presented in Tables 2 and 3.

Table 2. *Results of raw concentrate refining for technological schemes No. 1-3*

No. of technological scheme	Quality indicators, %								
	Output power supply			Concentrate			Dump tailings		
	Mass share Fe_{total}, %	Removal Fe_{total}, %	Exit Fe_{total}, %	Mass share Fe_{total}, %	Removal Fe_{total}, %	Exit Fe_{total}, %	Mass share Fe_{total}, %	Removal Fe_{total}, %	Exit Fe_{total}, %
1	66.0	100	100	68.3	94.9	91.7	40.6	5.1	8.3
2				68.0	95.7	92.9	40.0	4.3	7.1
3				69.0	92.9	88.9	42.1	7.1	11.1

Source: *authors' development*

Table 3. *Granulometric characteristics of ordinary concentrate finishing according to technological schemes No. 1-3*

No. of technological scheme	Size class, mm						Product
	+0.071	-0.071 +0.056	-0.056 +0.045	-0.045 +0.02	-0.02	Total	
1	-	0.3	1.1	31.7	66.9	100	Concentrate
2	0.5	0.9	3.5	34.2	60.9	100	Concentrate
3	-	0.6	0.9	22.0	76.5	100	Concentrate

Source: *authors' development*

An analysis of the granulometric composition of the concentrates indicates that those obtained in tests 1 and 3 consist of finer material than the sample from test 2. The content of the -0.045 mm fraction in these tests was 98.6 and 98.5%, respectively. Since pure ore minerals are mainly found in the -0.045+0 fraction, the total iron content is higher in samples 1 and 3 than in sample 2, with values of 68.3 and 69.0% compared to 68.0%. This confirms that the overall quality indicators of concentrates obtained using technological schemes that involve vertical mills for feed grinding are superior to schemes where the feed is not ground. This is due to the more effective liberation of grains, which positively impacts the quality parameters of the concentrate.

Discussion

The findings of this study are supported by a substantial body of scientific research. The magnetic-flotation refining method has been identified as the most promising and economically viable approach for raw concentrate beneficiation. The selection of beneficiation equipment and the reagent regime depends on the mineralogy and level of harmful impurities. The current trend in the metallurgical industry, coupled with the rapid growth of "green energy", demands that iron product manufacturers reduce the silica content to 2%. Magnetic or hydraulic separation alone cannot achieve this due to the presence of locked silica minerals. In the work of M. Ma (2012), a comparison of reverse anionic and cationic flotation highlighted that the advantages of anionic flotation include lower reagent costs and better iron recovery. However, the author noted that reverse cationic flotation remains the most widely used method for iron ore beneficiation.

In the work of F. Dehghani *et al*. (2022) demonstrated that the final ore grinding size depends on the nature of mineralisation, mineral distribution within ore deposits, and associated gangue minerals. The researchers examined two technological schemes for magnetic-flotation concentrate refining: 1) Magnetic separation with a magnetic field intensity of 2,000 G, grinding to a particle size of less than 74 μm, followed by a flotation cycle; 2) Grinding the entire flotation feed, flotation beneficiation, and subsequent wet magnetic separation with a magnetic field intensity of 1,000 G. During flotation, a collector reagent, depressant, and pH stabiliser were applied. The study determined that wet magnetic separation prior to flotation was more effective than using flotation as the primary beneficiation operation. Magnetic separation before flotation reduced grinding costs, the number of flotation cells, and reagent consumption, resulting in a higher-quality product with fewer impurities. In contrast, using flotation as the initial beneficiation step increased the number of flotation cells, raised reagent consumption, and complicated the process.

In the work of S.O. Bada *et al*. (2012) also employed an initial grinding stage to a particle size of less than 75 μm, followed by magnetic separation, a control grinding stage to achieve 95% of particles below 25 μm, wet magnetic separation, and reverse cationic flotation with the collector reagent Lilaflot-817M and starch (iron depressant). The operations increased the iron mass fraction in the concentrate to 65.5%, with reverse cationic flotation further increasing the iron content to 70.0%.

H.D.G. Turrer *et al*. (2007) studied the impact of synthetic flocculants on the beneficiation of fine iron ore fractions. They found that enriching iron ore fractions smaller than 44 μm could increase iron recovery by 7.8% with the addition of non-ionic polyacrylamides combined with a cationic collector, depressant, and pH stabiliser. However, using non-ionic polyacrylamides also increased silica content in the concentrate by 0.12-1.58% due to the presence of polyacrylamide-induced flocs. J.-O. Gustafsson & O. Lima (2013) evaluated the foam formation and stability of cationic collectors, as foam structure and stability are critical factors in mineral flotation. Their work compared the effects of ether diamine Lilaflot-D817M and Lilaflot-628M and found that ore type, water quality, and collector type significantly influence foam properties during magnetite quartzite beneficiation. Water with higher ion content produced more voluminous foam.

In the work of K.P. Babu & M. Aminuddin (2020) studied the impact of low-grade iron ore particle size and changes in reagent regimes on concentrate quality and yield. The study showed that increasing the collector dosage and pulp density reduced concentrate yield while increasing iron content. They also found that pH stabilisation affected concentrate yield and iron content. At lower pH levels, negatively charged silicates electrostatically interact with positively charged cation ions, reducing concentrate yield while increasing iron content. At higher pH levels, silicates become less negatively charged, decreasing interaction with cations, which increases concentrate yield but decreases iron content.

N.P. Lima *et al*. (2013) examined the size-based separation of flotation feed. They divided the feed into two fractions: -150÷+45 μm and below 45 μm and compared the results with an undivided sample of less than 150 μm. The study found that iron recovery was highest for the fraction below 45 μm with lower reagent consumption. However, increased collector dosage reduced iron recovery and silica content. The researchers demonstrated that size-based feed separation in iron ore flotation improved iron recovery by nearly 3%, reduced silica content, and decreased iron content in tailings. G.M. Rocha *et al*. (2022) studied flotation without starch addition and at lower pH levels. They compared amide-amines and etheramines as collectors, finding that increasing amide-amine dosage by 50%, without starch addition, improved iron recovery by up to 9% and concentrate yield by up to 5%. However, this reduced the iron mass fraction in the concentrate by 1%.

To enhance the flotation efficiency for magnetite quartzite beneficiation, H. Hubin *et al*. (2018) found that the initial concentrate contained many technological aggregates formed during grinding, increased pulp viscosity, and the growth of the ground material's total surface area into magnetic flocs, causing mechanical entrapment of natural particles. They demonstrated that ultrasonic treatment improved flotation properties and increased final concentrate yield by 2-5% without compromising product quality by altering pulp viscosity.

To sum it all up, the study, along with an analysis of other scientific works, confirmed that reverse cationic flotation is the most promising method for iron ore beneficiation. However, the technological and economic performance of this process depends on the proper selection of beneficiation equipment and an optimal reagent regime.

Conclusions

Magnetic beneficiation methods do not yield high-quality concentrates due to ore depletion and the inclusion of silicate minerals, which degrade concentrate quality. The concentrate is fine-grained and comprises aggregates that are challenging to separate using magnetic separators. An analysis of flotation beneficiation processes at the Poltava and Ingulets Mining and Processing Plants, as well as enterprises in the USA and Canada, demonstrated that reverse cationic flotation is more efficient. Depending on feed quality, it can increase iron content to 69% or higher while reducing silica content to 3%.

Laboratory studies revealed that additional grinding operations in a closed circuit, combined with wet magnetic separation, can increase the iron mass fraction to 68.0-68.3% and reduce silica content to 4.3-4.6%. The flotation refining scheme that omits grinding the initial feed but includes froth product grinding in a closed circuit showed the poorest qualitative results for the final concentrate. The reduced iron mass fraction in the final concentrate was attributed to the presence of non-liberated grains with a particle size of +0.045÷+0.071 mm, containing 28.2-45.2% total iron. In this technological scheme, the total iron content did not exceed 68.0%.

The most efficient refining scheme for the raw concentrates of PJSC "Northern MPP" is the magnetic-flotation refining scheme with classification and grinding of the raw concentrate in a closed circuit. Laboratory tests confirmed that this approach can produce a concentrate with a total iron content exceeding 69.0% and silica content below 3%. Producing high-quality concentrate will enable its use in the production of DRI-class pellets, which is critical for the global steel industry's transition to "green energy". Future research will focus on improving the technological chain, particularly by determining the optimal feed particle size for flotation cells to enhance the quantitative and qualitative performance of the magnetic-flotation technology and optimise reagent consumption.

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None.

Conflict of Interest

None.

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Розробка технології флотаційного збагачення магнетитових кварцитів приватного акціонерного товариства «Північний гірничо-збагачувальний комбінат»

Анотація. Метою роботи була розробка нової технології флотаційного доведення магнетитових концентратів на основі приватного акціонерного товариства «Північний гірничо-збагачувальний комбінат» для отримання кінцевого товарного продукту з массовою часткою заліза загального не менше 69,0 % і кремнезему не більше 4 %. Використано мінералогічний аналіз для дослідження розкриття мінералів, хімічний аналіз – для визначення якісного складу досліджуваної сировини та ситовий аналіз – для визначення гранулометричного складу матеріалу. В роботі також застосовано методи обробки, аналізу та синтезу результатів досліджень зі створення оптимальних умов розділення мінеральних зерен за крупністю. У результаті визначено, що досягнення максимальної масової частки заліза загального у концентраті та мінімальної масової частки кремнезему, при максимальному вилученні заліза, залежать від умов подрібнення, які визначаються ступенем розкриття мінеральних зерен. Встановлено, що константа швидкості флотації кварцу залежить від масової частки кварцу в живленні і становить 0,05-0,10 $хв^{-1}$. Мінералогічні дослідження рядового концентрату виявили, що до його складу входять, окрім мономінеральних частинок магнетиту, зростки магнетиту – кварцу (до 2,52 %), магнетиту – кварцу – силікату, магнетиту – кварцу, силікату – карбонату (до 7,5 %). Карбонат здебільшого представлений сидероплазитом, кальцитом; силікати – кумінгтонітом, біотитом, хлоритом. Нерудні часточки – мономінеральні кварцові або силікат – кварцові, карбонат – силікат – кварцові. Згідно з кількісним мінералогічним розрахункам, мономінеральні частинки магнетиту накопичуються в тонкозернистій фракції в класі -0,045 + 0,02 мм – 92,3 % та в класі -0,02 мм – 98,1 %. В результаті проведення лабораторних випробувань магнітно-флотаційного доведення концентрату доведено можливість отримання з магнетитових концентратів приватного акціонерного товариства «Північний гірничо-збагачувальний комбінат» високоякісного концентрату з масовою часткою заліза загального 69,0 % і оксиду кремнію 3 %, 88,9 % за виходом і 92,9 % вилученням заліза загального у концентрат від операції. Практична значимість полягає у розробці технологічної схеми доведення рядового магнетитового концентрату в умовах приватного акціонерного товариства «Північний гірничо-збагачувальний комбінат» для отримання високоякісного концентрату з максимальним виходом та масовою часткою заліза загального вище 69,0 % і кремнезему менше 4,0 % при вилученні заліза загального у концентрат більше 90 %

Ключові слова: концентрат залізорудний; флотація катіонна зворотна; сепарація магнітна; частка масова; склад гранулометричний