Evaluation of the flotation process for producing a DRI pellet feed from a Canadian low-grade iron ore

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1. Introduction

The Mont Reed iron deposit is situated in the southern section of the Labrador Trough (Greenville tectonic province), Québec, Canada (Fig 1). The mining property was initially acquired by Québec Cartier Mining (QCM - now ArcelorMittal) during the 1960s and remains unexploited since (Québec Cartier Mining Company, 1977).

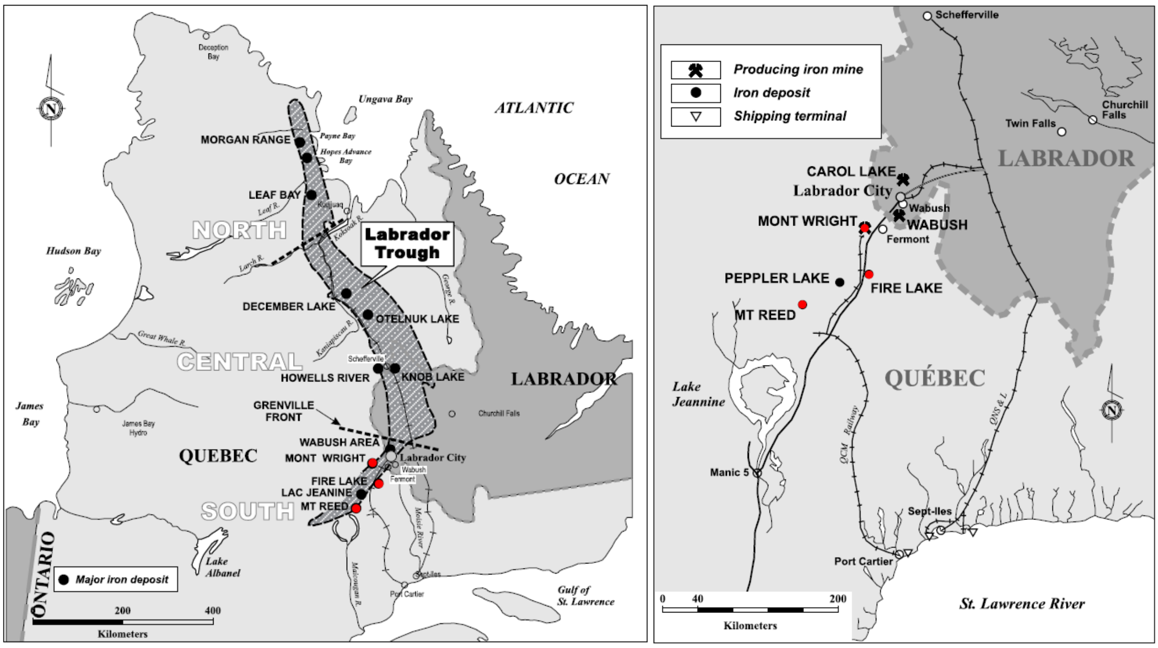


Fig 1. Mont Reed location (ArcelorMittal assets in red).

The Labrador Trough contains world-class Palaeoproterozoic, Lake Superior-type, banded iron formations (BIFs, 1.88 Ga), that were weakly to intensely metamorphosed during the late Precambrian orogeny (ca. 1.0 Ga). In this context, the iron formation at Mont Reed is composed of fine to medium grained magnetite-hematite-quartz units, associated with quartzite, marble, gneiss and silicate-carbonate facies. The orebody has an average grade of 30% Fe and extends over an area of about 2 × 3 km, with a thickness ranging from 60 to 100 m. Geological interpretations describe Mont Reed as primarily folded in an antiform shape that was later refolded orthogonally (Ibrango, 2013). In addition to iron oxides, most of mineral assemblages of the iron formation consist of quartz, Mg-Fe orthopyroxenes (enstatite–ferrosilite series), Ca-clinopyroxenes to a lesser extent (eg diopside–hedenbergite series) and carbonates (Klein, 1978).

One particularity of the Mont Reed deposit when compared to the other ArcelorMittal Mines Canada (AMMC) operations lies in the fact that the ore contains both magnetite and hematite-rich ores, with an overall magnetite to hematite ratio of about 2:1, together with the occurrence of a Mn-rich layer that contaminates the orebody (Belissont, 2021). Furthermore, the liberation degree of the particles in the Mont Reed deposit is lower compared to other deposits in the region. Additionally, the mineralogy of the gangue is much more complex, characterised by the presence of pyroxenes, carbonates, silicates and amphiboles (Mesquita *et al*, 2023).

Hence, in addition to geological and resource modelling, one key issue to make the mine exploitable is to find an optimal process route to concentrate the ore (recovering both hematite and magnetite) and avoiding the presence of gangue minerals in the concentrate. To address this issue ArcelorMittal Mining and Mineral Processing Team (MMP) have been studying Mont Reed ore from 2020 in order to develop a feasible process route to concentrate the material and produce a high-quality pellet feed (SiO_2 <1.8 per cent) which can be fed into a direct reduction furnace. Throughout the research process, a comprehensive characterisation study was conducted using representative samples from the Mont Reed deposit (Belissont, 2021). Additionally, a thorough series of bench-scale tests was carried out to evaluate various methodologies for the concentration of the ore (Mesquita, 2021). Building upon the findings from these investigations, the initial flowsheet presented in Fig 2 was developed.

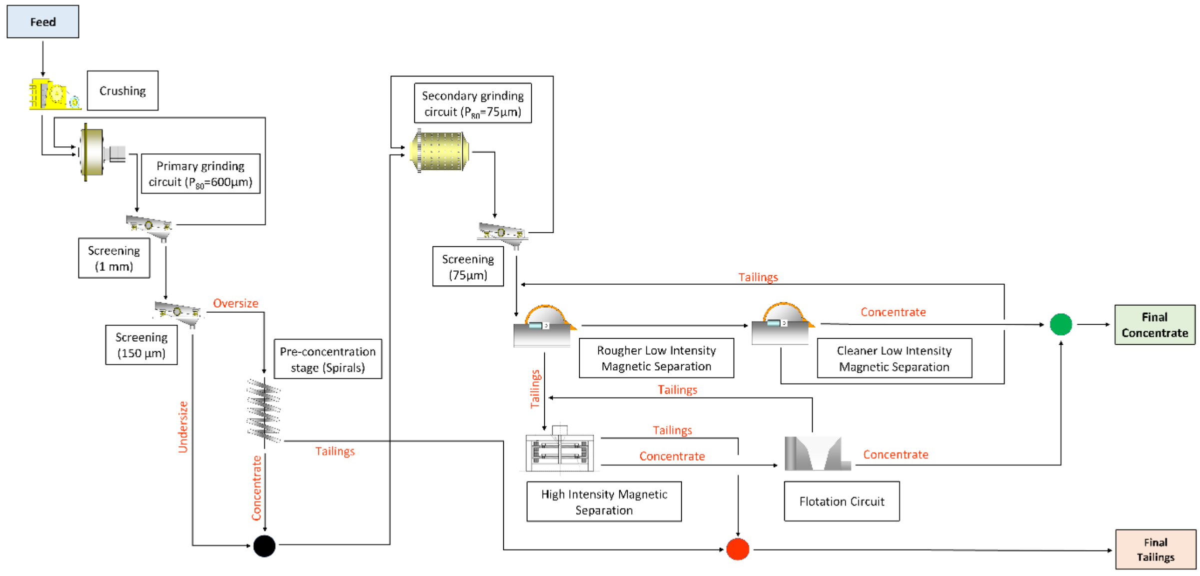


Fig 2. Mont Reed preliminary flowsheet

The flowsheet comprises several stages, beginning with a primary crushing step to adjust the run-of-mine (ROM) material before proceeding to the SAG mill circuit. The main objective of the SAG circuit is to liberate a portion of the gangue, producing a product with a P_{80} of 600 µm, which is then subjected to classification at 150 µm. The oversize material from the screening operation is directed to a pre-concentration stage, utilising spirals. Importantly, the classification is performed before the pre-concentration study to prevent any loss of fine iron oxide particles within the spirals. The concentrate obtained from the spirals and the undersize material from the 150 µm classification are then fed into the secondary grinding circuit, ensuring further liberation of the particles. The magnetite present in the ground product is recovered through a low-intensity magnetic separation circuit (LIMS) composed by two stages: rougher and cleaner. On the other hand, to recover the hematite from the tailings of LIMS stage, a different circuit is used, consisting of wet high-intensity magnetic separation (WHIMS), followed by reverse cationic flotation.

In this context, the pilot-scale testing of the LIMS circuit returned different results compared to the bench-scale one conducted using a Davis Tube tester. In the first, the LIMS circuit failed to generate a final product meeting the desired quality criteria (SiO_2 < 1.8% Fe). Instead, the operation produced a final concentrate with 66.28% Fe and 6.90% SiO_2 (28.3% yield / 58.4% Fe recovery), which is a much higher silica level than the initial target. Considering these findings, the MMP Team has made the decision to assess the inclusion of a flotation stage after the LIMS circuit to reduce the silica content to the targeted levels (Filippov, 2014). Therefore, this study presents the results obtained with the objective of producing a high-quality concentrate through the flotation process.

2. Materials & Methods

The work was divided in five different phases, as presented in Fig 3. The initial phase of the study revolved around characterizing the sample to be tested. This entailed a comprehensive chemical and mineralogical analysis of the material, accompanied by a study on its liberation characteristics.

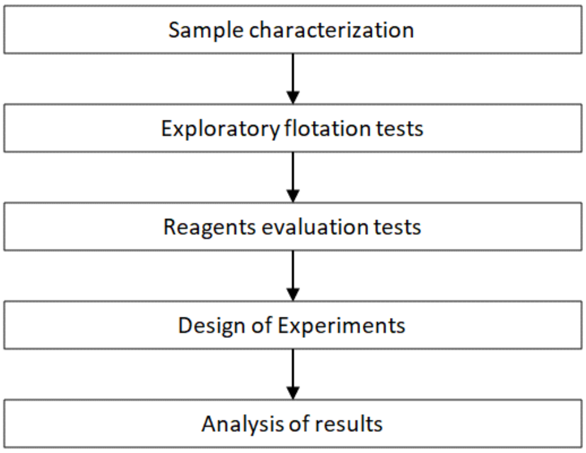


Fig 3. Research methodology

The following phase of the study involved the execution of exploratory bench-scale flotation tests. The primary objective was to assess the feasibility of employing reverse cationic flotation techniques to concentrate the magnetite contained in LIMS circuit product (Veloso, 2018). This phase scrutinized various critical factors, including reagent dosages, desliming procedures, and the use of frother. Building upon the insights gained from the initial two phases, the subsequent stage of the research focused on evaluating different collectors and depressants to concentrate the magnetite found in Mont Reed ore (Araujo, 2005). Based on the outcomes of the third phase, the most promising collector and depressant were selected, and a design of experiments (DoE) plan was developed to examine the influence of reagent dosages, pH levels, and solids percentage in the flotation process, while targeting to maximize the Fe recovery in the process and minimize the SiO_2 grade in the final concentrate. With all the information generated in the preceding stages, a comprehensive analysis was conducted to address the initial objectives of this research.

2.1. Materials

2.1.1. Sample

The sample selected for the study corresponds to a representative aliquot of the LIMS cleaner concentrate collected during the pilot campaign testwork executed with Mont Reed ore.

2.1.2. Desliming apparatus

Desliming operation was executed using a desliming loop supplied by Brastorno coupled with a 1” hydrocyclone. The operation was performed with a pulp of 20% of solids and using a pressure of 2.5kpa.

2.1.3. Flotation cell

Flotation experiments were conducted employing a Denver Metso Flotation Cell model D12, featuring a conventional 1-liter cell (Fig 4).

Fig 4. Flotation apparatus

2.1.4. Reagents

FLOTIGAM 5944 (etheramine), FLOTIGAM 2835-2 (etherdiamine), FLOTINOR 5530 (amidoamine) and FLOTINOR 16551 (phosphoric ester) supplied by Clariant were employed as collectors, while corn starch, dextrin from maize starch, carboxymethyl cellulose (CMC) and humic acid supplied by Sigma-Aldrich were utilized as depressants (Veloso, 2018 and Turrer, 2010). Montanol 800 (provided by Clariant) was evaluated as frother due to its exceptional efficiency with ultrafine particles.

1.1.1. Reagent's preparation

For bench flotation tests, 4 $g \cdot L^{-1}$ (wt/v) reagent solutions were prepared as described below.

Collectors

1. FLOTIGAM 7100 (etheramine) was prepared by diluting 2 g of reagent with demineralized water to reach a total volume of 500 mL under continuous stirring.
2. FLOTIGAM 2835-2 (etherdiamine) was prepared by diluting 2 g of reagent with demineralized water to reach a total volume of 500 mL under continuous stirring.
3. FLOTINOR 5530 (amidoamine) was prepared by diluting 2 g of reagent with demineralized water to reach a total volume of 500 mL under continuous stirring.
4. FLOTINOR 16551 (phosphoric ester) was prepared by diluting 2 g of reagent with demineralized water to reach a total volume of 500 mL under continuous stirring.

Depressants

1. To prepare the starch solution, 2 g of the reagent was diluted in 20 mL of water. Simultaneously, 0.5 g of NaOH was dissolved in the same quantity of water using a separate container. Once both reagents were fully dissolved, they were combined in a beaker and mixed thoroughly using a magnetic agitator until the starch was completely gelatinized. The resulting gelatinized solution was then further diluted with

demineralized water to a total volume of 500 mL and was used for conducting the flotation tests.

2. Dextrin was prepared by diluting 2 g of the reagent salt with demineralized water to reach a total volume of 500 mL under continuous stirring.
3. CMC was prepared by diluting 2 g of carboxymethyl cellulose sodium salt with demineralized water to reach a total volume of 500 mL under continuous stirring.
4. Humic acid was prepared by diluting 2 g of humic acid sodium salt with demineralized water to reach a total volume of 500 mL under continuous stirring.

Frother

1. Montanol 800 was prepared by diluting 2 g of the reagent solution with demineralized water to reach a total volume of 500 mL under continuous stirring.

1.1.2. Flotation tests

Bench-scale flotation tests were carried out conditioning the reagents in a pulp containing 50 wt.% solids and agitation at 1200 rpm. The conditioning time of reagents was settled in 5 minutes. Subsequently, the flotation operation was executed at a reduced speed of 900 rpm with an airflow rate of 4 L·min^{-1}. Froth collection continued until exhaustion.

1.1.2.1. Exploratory tests

Exploratory tests were performed aiming at evaluating in a global way the influence of desliming, frother and reagents dosage in flotation process. The parameters adopted in the essays are presented in Table 1.

Table 1. Exploratory flotation tests parameters

Test	**Desliming**	**Operational Parameters**							
		wt.% of solids	**pH**	**Collector**		**Depressant**		**Frother**	
				Name	**Dosage (g/t)**	**Name**	**Dosage (g/t)**	**Name**	**Dosage (g/t)**
1	No	30	10.5	FLOTIGAM 7100	150	Corn Starch	300	Montanol 800	50
2	Yes	30	10.5	FLOTIGAM 7100	150	Corn Starch	300	Montanol 800	50
3	Yes	30	10.5	FLOTIGAM 7100	150	Corn Starch	300	Montanol 800	0
4	Yes	30	10.5	FLOTIGAM 7100	300	Corn Starch	800	Montanol 800	50

1.1.2.2. Reagent's evaluation tests

Aiming at optimising the flotation operation, different collectors and depressants were evaluated. The methodology adopted during the reagent's evaluation campaign is presented in Fig 5. All tests were performed in pH 10.5 with 30% of solids in pulp. Due to its specific characteristics (collector/depressant dual action), amidoamine was evaluated with and without addition of depressant (Silva et *al*, 2022).

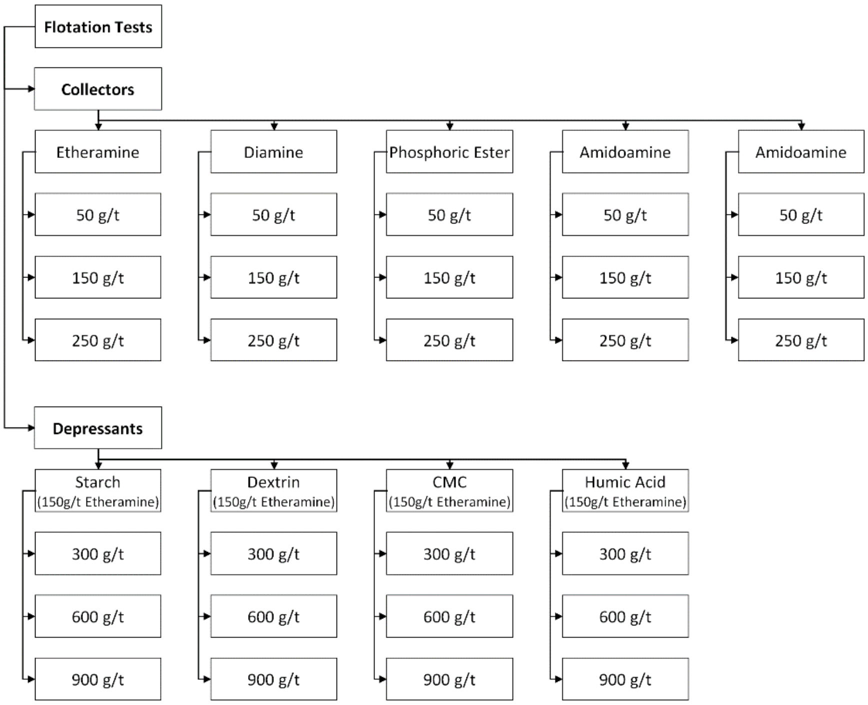


Fig 5. Reagent's evaluation tests methodology

1.1.2.3. Design of Experiments tests

Based on the results achieved in the previous phases, a DoE was developed aiming at optimising the flotation operation. The parameters adopted in the tests are presented in Table 2.

Table 2. DoE tests parameters

Test	Type	Desliming	Operational Parameters					
			% of solids	pH	Collector		Depressant	
					Name	Dosage (g/t)	Name	Dosage (g/t)
1	Reverse Cationic	Yes	30	9	FLOTIGAM 7100	150	Corn Starch	800
2	Reverse Cationic	Yes	40	11	FLOTIGAM 7100	350	Corn Starch	400
3	Reverse Cationic	Yes	40	11	FLOTIGAM 7100	50	Corn Starch	200
4	Reverse Cationic	Yes	40	10	FLOTIGAM 7100	350	Corn Starch	200
5	Reverse Cationic	Yes	30	11	FLOTIGAM 7100	50	Corn Starch	600
6	Reverse Cationic	Yes	40	9	FLOTIGAM 7100	250	Corn Starch	200
7	Reverse Cationic	Yes	30	10	FLOTIGAM 7100	250	Corn Starch	600
8	Reverse Cationic	Yes	40	9	FLOTIGAM 7100	350	Corn Starch	800
9	Reverse Cationic	Yes	40	10	FLOTIGAM 7100	150	Corn Starch	400
10	Reverse Cationic	Yes	30	9	FLOTIGAM 7100	350	Corn Starch	400
11	Reverse Cationic	Yes	30	9	FLOTIGAM 7100	50	Corn Starch	200
12	Reverse Cationic	Yes	40	9	FLOTIGAM 7100	50	Corn Starch	600
13	Reverse Cationic	Yes	30	11	FLOTIGAM 7100	250	Corn Starch	200
14	Reverse Cationic	Yes	40	10	FLOTIGAM 7100	250	Corn Starch	600
15	Reverse Cationic	Yes	40	11	FLOTIGAM 7100	150	Corn Starch	800
16	Reverse Cationic	Yes	30	10	FLOTIGAM 7100	150	Corn Starch	400
17	Reverse Cationic	Yes	30	11	FLOTIGAM 7100	350	Corn Starch	800

3. Results & Discussion

3.1. Sample characterisation

The size-by-size results of LIMS cleaner concentrate sample are presented in Table 3 and its mineralogical composition is presented in Table 4.

Table 3. Head sample size-by-size (sieving)

Size (µm)	Mass (%)	Cum OS (%)	Cum US (%)	Grade (%)												
				Fet	FeO	SiO_2	Al_2O_3	CaO	MgO	TiO_2	P	Mn	S	Na_2O	K_2O	LOI
150	3.0	3.0	97.0	40.35	18.65	41.60	0.05	0.80	1.25	0.01	0.003	0.26	0.005	0.02	<0.001	-1.19
106	6.4	9.4	90.6	57.60	23.70	18.80	0.09	0.37	0.66	0.01	0.002	0.26	0.004	0.02	<0.001	-2.35
75	22.0	31.4	68.6	65.95	26.80	8.14	0.09	0.24	0.48	0.01	0.002	0.27	0.003	0.02	<0.001	-2.76
53	20.8	52.2	47.8	68.03	28.00	5.14	0.10	0.21	0.44	0.01	0.002	0.28	0.003	0.02	<0.001	-2.80
45	13.7	65.9	34.1	68.83	29.40	3.19	0.10	0.21	0.40	0.01	0.001	0.29	0.004	0.02	<0.001	-2.85
38	34.1	100.0	0.0	69.47	28.60	1.78	0.10	0.58	0.37	0.01	0.001	0.32	0.047	0.07	0.01	-2.40
<38	3.0	3.0	97.0	40.35	18.65	41.60	0.05	0.80	1.25	0.01	0.003	0.26	0.005	0.02	<0.001	-1.19
Feed	**100.0**	**-**	**-**	**66.28**	**27.58**	**6.90**	**0.10**	**0.37**	**0.46**	**0.01**	**0.002**	**0.29**	**0.018**	**0.04**	**0.00**	**-2.58**

Table 4. Head sample mineralogy

Mineral	Mineral Class	Chemical Formula	Proportion (%)
Magnetite	Oxides	Fe_3O_4	92.07
Quartz	Tectosilicate	SiO_2	6.14
Hematite	Oxides	Fe_2O_3	1.38
Dolomite	Carbonates	$CaMg(CO_3)_2$	0.33
Calcite	Carbonates	$CaCO_3$	0.05
Talc	Phyllosilicate	$Mg_3Si_4O_{10}(OH)_2$	0.03

The ore is mainly composed of magnetite (92.07%) with minor quantities of quartz (6.14%) and hematite (1.38%). There are also trace amounts of carbonates such as dolomite and calcite, which explains the CaO and MgO observed in the chemical analysis. The complete particle size distribution (PSD) of the material is presented in Fig 6.

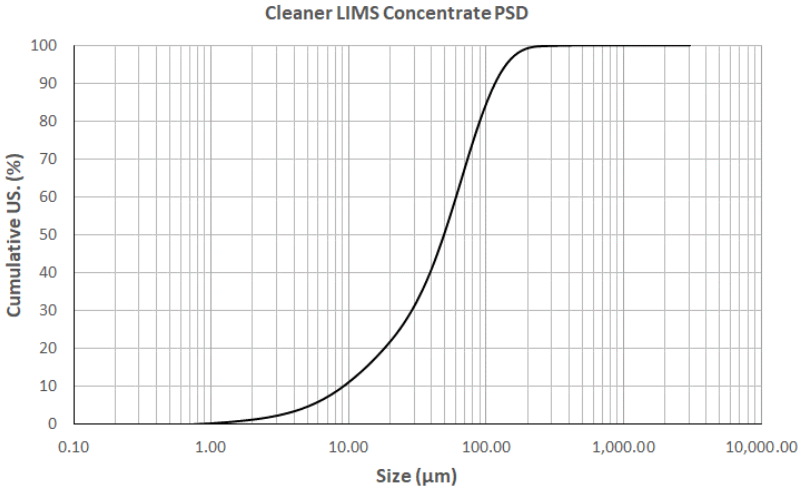


Fig 6. Head sample PSD (laser analyser)

The sample exhibits a P_{80} value of 90 µm, a P_{50} value of 50 µm, and 10% of particles passing at 10 µm. In terms of liberation, iron oxides are well liberated while a higher quantity of gangue mixed particles is observed (Table 5).

Table 5. Liberation analysis

Mineral	Proportion (%)		
	Liberated (>90% in area)	**Middling (<90 >30% in area)**	**Locked (<30% in area)**
Iron oxide	97.79	1.51	0.70
Quartz	82.42	16.45	1.13
Dolomite	31.29	18.94	49.77
Calcite	4.43	62.26	33.32

As expected, most part of mixed gangue particles are contained in the coarser size fractions (e.g. >53µm) as observed in the size-by-size results and illustrated in Fig 7.

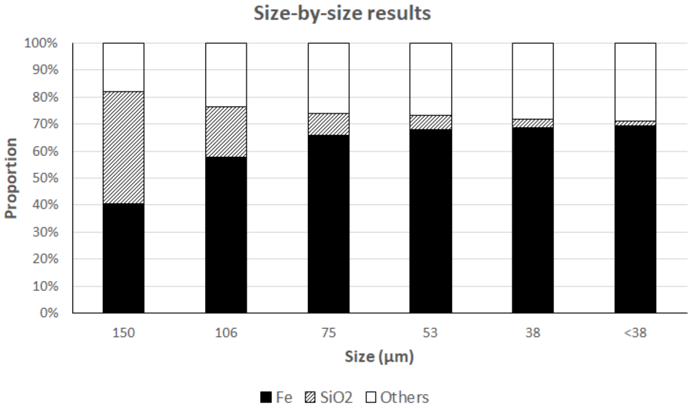


Fig 7. Main elements distribution per size fraction

This fact is due to the locked characteristics of coarse gangue particles (Fig 8). The disseminated iron oxides within quartz particles makes this type of particle be attracted by the magnetic field of LIMS operations, contaminating thus the concentrate. This is also an important point for flotation tests, since these particles may be adsorbed by both depressant and collectors, which may jeopardize the selectivity of the process. In this way it is extremely important to find the optimum dosages of reagents to perform an efficient separation of magnetite particles contained in the sample.

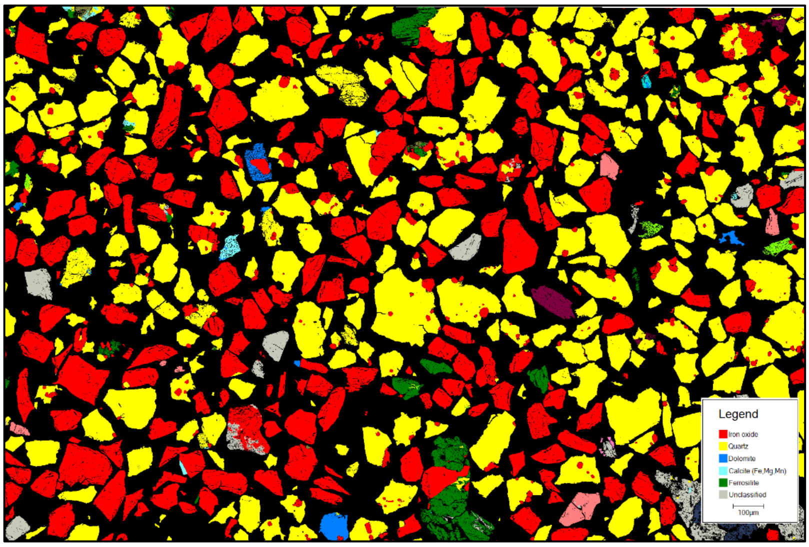


Fig 8. Cleaner LIMS concentrate >150µm size fraction SEM image

3.2. Exploratory tests

During the exploratory tests campaign, the first parameter evaluated was the addition of a desliming stage previously to the flotation operation. The results obtained are presented in Table 6.

Table 6. Desliming test results

Product	Stage Distribution (%)			Grade (%)													
	Mass	Fe	SiO_2	Fe	FeO	SiO_2	Al_2O_3	CaO	MgO	TiO_2	P	Mn	S	Na_2O	K_2O	LOI
Feed	100.0	100.0	100.0	66.28	28.13	6.90	0.43	0.24	0.48	0.00	0.00	0.28	0.00	0.01	0.00	-2.92
Deslimed	98.1	98.2	98.7	66.31	28.20	6.94	0.44	0.21	0.47	<0.01	0.001	0.28	<0.001	0.01	<0.001	-2.96
Slimes	1.9	1.8	1.3	64.46	24.70	4.86	0.16	1.65	0.88	0.01	0.005	0.47	0.045	0.09	0.02	-0.94

Despite the loss of mass (1.9%) and iron (1.8%) in the process, desliming proved to be beneficial for the flotation process, mainly when considering the removal of SiO_2 from the concentrate, as presented in Fig 9.

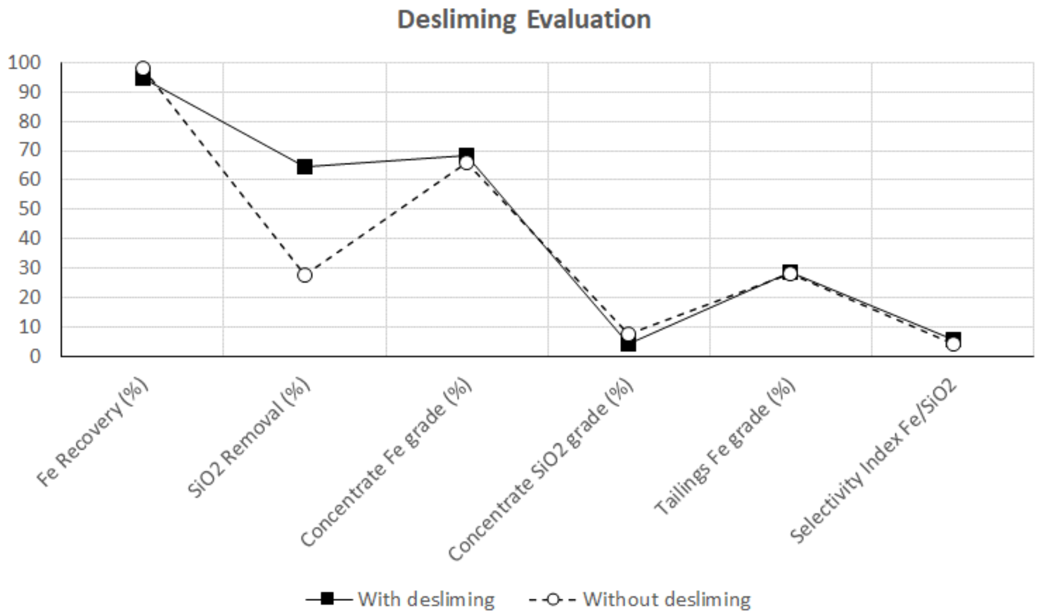
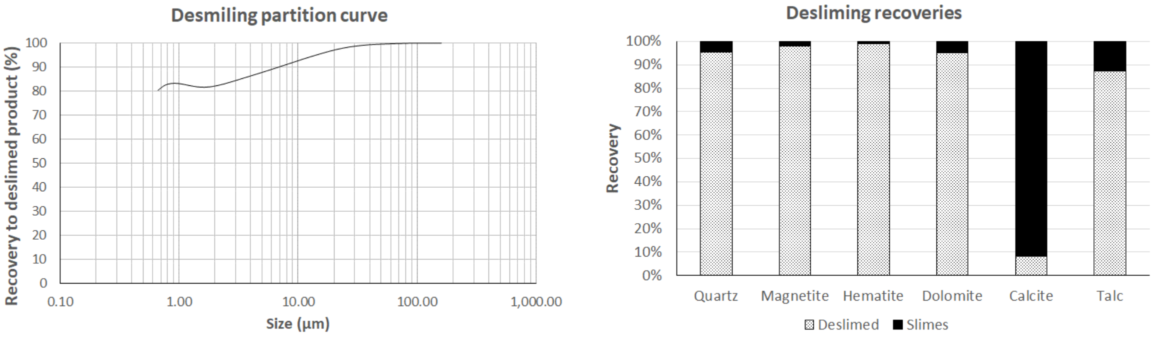


Fig 9. Flotation tests desliming evaluation

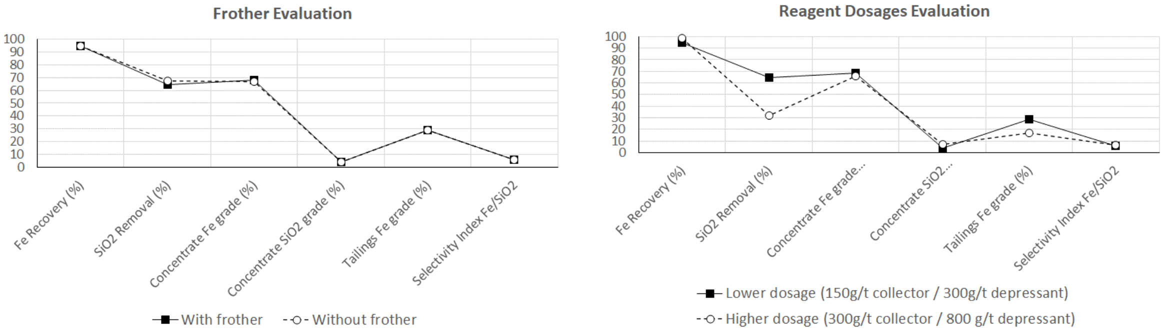
The improvement in the silica removal is due to two factors acting together: (i) the reduction of ultrafine particles in the system (mainly below 5μm); (ii) the removal of carbonates from the ore, which increases the presence of ions in the pulp, jeopardizing the separation process (Fig 10). These two factors acting together allowed a considerable increasing in silica removal, indicating thus the necessity of a desliming stage previously to flotation operations.



a) b)

Fig 10. a) Desliming operation partition curve. b) Mineral's recovery in desliming operation

The second parameter evaluated was the use of a frother to help in the stabilization of the foam. The results indicated that the reagent had no big effect on the process and due to that the use of the modifier was discarded. Lastly, the influence of the reagent's dosages into the flotation results were evaluated as presented. The results showed that the increasing in reagents dosage had a bad effect on silica removal. This is most probably due to the high quantity of starch added, which seems to have depressed part of the quartz particles, keeping them in the concentrate. By consequence, the quantity of iron in the tailings also decreased. These results highlight the necessity of avoiding the overdosage of reagents in the flotation operation. Both facts are illustrated in Fig 11.



a) b)

Fig 11. a) Forther evaluation. b) Reagent's dosage evaluation

The exploratory tests were very important to understand better the material and define some premisses for the next stages of the study. The key findings are summarized below:

- A desliming stage is mandatory to produce a high-quality from through Mont Reed LIMS circuit concentrate.
- The use of a frother did not presented any improvement in the process.
- The dosage of reagents has a strong impact on process efficiency and high dosages must be avoided, mainly when considering depressants.

3.3. Reagent's evaluation

As mentioned before, the reagent's evaluation study was divided in two phases, being the first one focused on evaluating different collectors and the second one focused on the depressants. The results obtained in the first part of the study are presented in Fig 12.

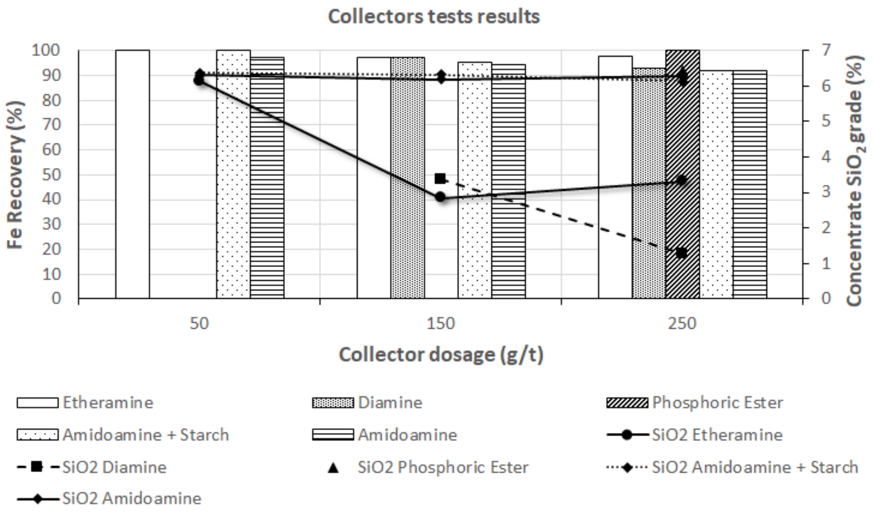


Fig 12. Collectors' tests results

In general, low dosages of collectors (50g/t) did not led to an effective separation with tests delivering concentrates with high silica grade. No froth was observed when using diamine and phosphoric ester in this dosage. When working with higher dosages, both etheramine and diamine returned promising balances between Fe recovery and SiO_2 grade in the concentrate (Fig 13). The higher dosage diamine test returned a concentrate with less than 1.8% SiO_2 (250g/t) but with a slighter decrease in Fe recovery. By the other side, both phosphoric ester and amidoamine did not led to a good selectivity in the process. In addition, no big differences were observed between the tests using amidoamine with and without depressant. In a general way, the etheramine was the collector with the best performance among the four tested.

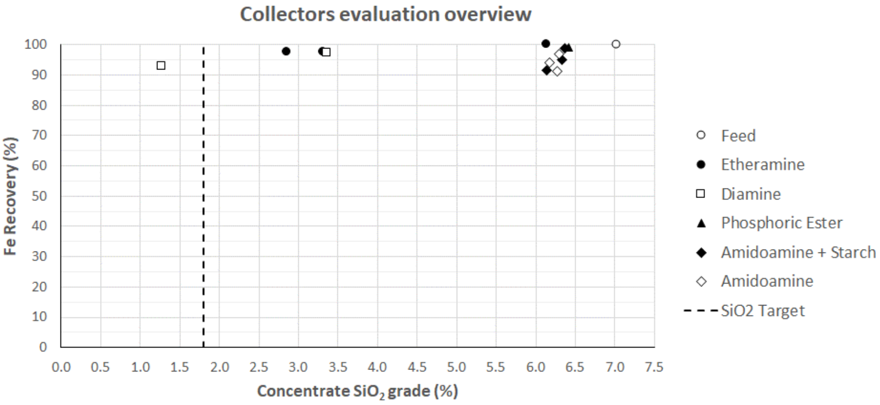


Fig 13. Collectors' evaluation overview

Regarding the depressants, both starch and dextrin presented interesting balances between Fe recovery and SiO_2 grade in the concentrate as presented in Fig 14. In both cases, overdosage of depressant jeopardized the process selectivity (as previously observed during exploratory tests), returning poorer concentrates. For CMC, stabilization was observed when using dosages higher than 300 g/t, with no big impacts related to overdosages as observed for the previous cases. The SiO_2 grade in this case stayed considerably stable among the tests but the Fe recovery using the reagent was low. Lastly, the humic acid was able to return the lowest SiO_2 grades among all tests, but the iron recovery was strongly affected.

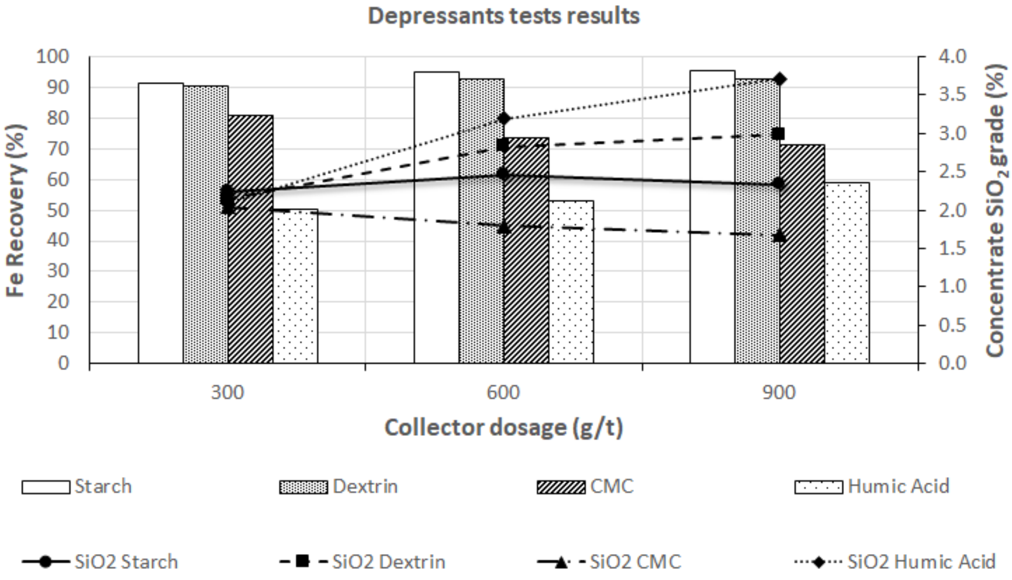


Fig 14. Depressant's tests results

In general, despite not attending the initial target, starch was the depressant with the best compromise between quality and recovery (Fig 15).

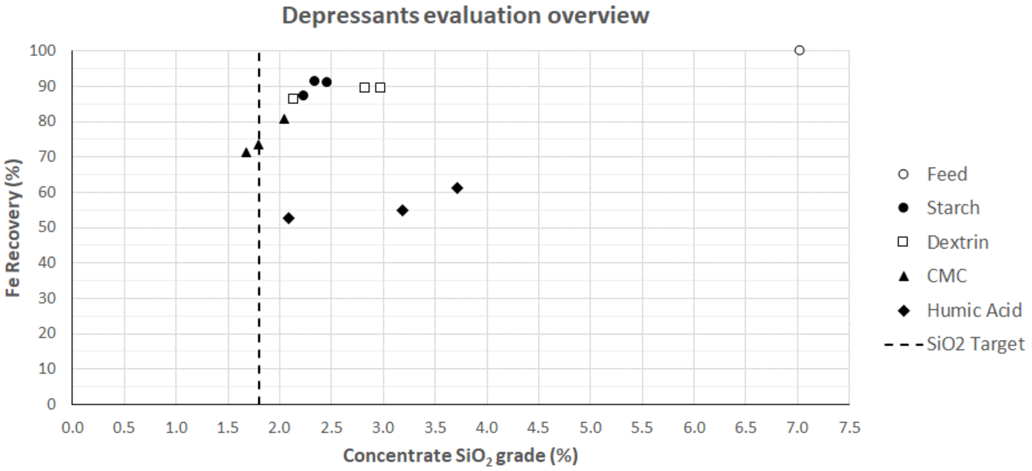


Fig 15. Depressant's tests results overview

Given the results obtained in this phase of the study, etheramine and starch were selected to be tested in the DoE phase with the objective of optimising the flotation process and arriving in the desired SiO_2 levels in the concentrate.

3.3.1. Design of experiments

The results obtained in DoE flotation tests are presented in Table 7.

Table 7. Design of experiment flotation tests results

Test	% of solids	pH	Dosage (g/t)		Concentrate				Tailings			
					Distribution (%)		Grade (%)		Distribution (%)		Grade (%)	
			Collector (g/t)	Depressant (g/t)	Mass	Fe	Fe	SiO_2	Mass	Fe	Fe	SiO_2
1	30	9.0	150	800	87.9	92.6	70.14	1.86	12.1	7.4	40.72	41.80
2	40	11.0	350	400	78.6	82.8	69.95	2.06	21.4	17.2	53.29	24.50
3	40	11.0	50	200	94.7	95.4	67.29	6.17	5.3	4.6	56.86	16.30
4	40	10.0	350	200	56.8	60.6	70.90	0.92	43.2	39.4	60.52	14.70
5	30	11.0	50	600	96.9	97.3	66.61	6.23	3.1	2.7	57.24	16.50
6	40	9.0	250	200	45.9	49.0	71.54	1.18	54.1	51.0	63.20	11.65
7	30	10.0	250	600	84.1	89.2	70.60	1.40	15.9	10.8	45.32	34.70
8	40	9.0	350	800	90.8	95.5	69.86	2.01	9.2	4.5	32.76	53.10
9	40	10.0	150	400	84.9	89.9	70.63	1.68	15.1	10.1	44.32	36.20
10	30	9.0	350	400	94.3	97.1	69.14	3.96	5.7	2.9	34.73	49.70
11	30	9.0	50	200	88.4	91.0	68.67	4.29	11.6	9.0	51.72	26.10
12	40	9.0	50	600	93.1	96.0	68.49	3.81	6.9	4.0	38.08	45.50
13	30	11.0	250	200	78.9	83.4	70.26	1.82	21.1	16.6	52.27	25.30
14	40	10.0	250	600	81.1	86.0	70.54	1.28	18.9	14.0	49.25	30.00
15	40	11.0	150	800	90.4	93.4	69.07	3.91	9.6	6.6	46.29	33.00
16	30	10.0	150	400	85.8	90.6	70.04	1.68	14.2	9.4	43.97	37.60
17	30	11.0	350	800	87.4	91.4	69.55	2.44	12.6	8.6	45.16	35.70

The experiments proved the feasibility of producing a concentrate with less than 1,8% of silica from Mont Reed LIMS circuit concentrate using flotation (Fig 16).

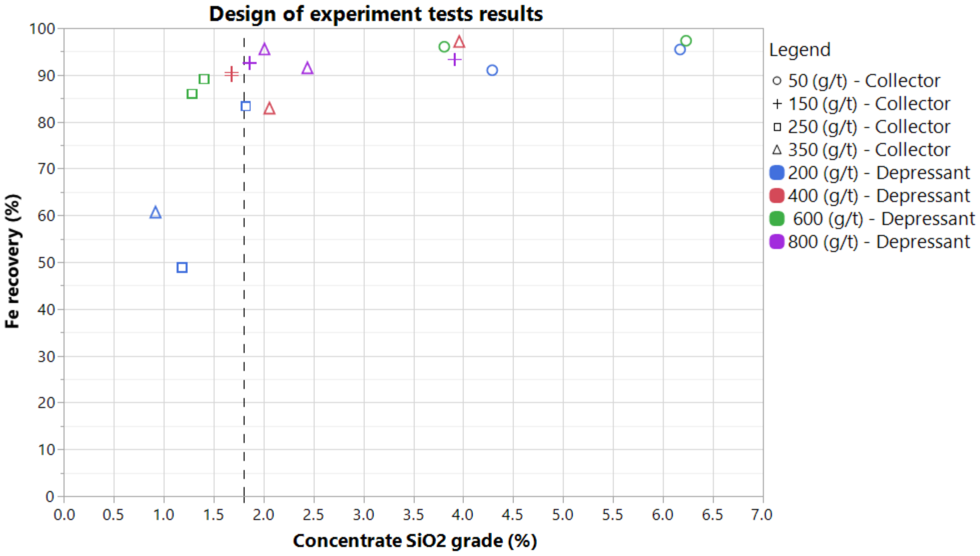
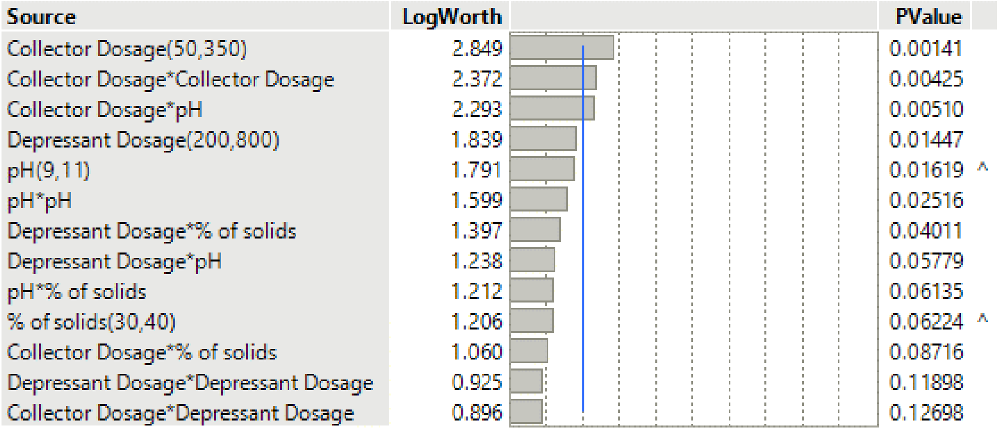


Fig 16. DoE tests results evaluation

From the graph it is possible to observe that tests with collector and depressant dosages of 150g/t and 400g/t, respectively, presented the best compromise between Fe recovery and SiO_2 grade. Despite presenting higher qualities, tests with bigger collector dosages led to a considerable decrease in recovery. In this context, the DoE analysis showed that the main factor affecting the flotation process corresponds to the dosage of collector (Fig 17). This parameter also presents an important synergy with the pH of the pulp.



Source	LogWorth	PValue
Collector Dosage(50,350)	2.849	0.00141
Collector Dosage*Collector Dosage	2.372	0.00425
Collector Dosage*pH	2.293	0.00510
Depressant Dosage(200,800)	1.839	0.01447
pH(9,11)	1.791	0.01619 ^
pH*pH	1.599	0.02516
Depressant Dosage*% of solids	1.397	0.04011
Depressant Dosage*pH	1.238	0.05779
pH*% of solids	1.212	0.06135
% of solids(30,40)	1.206	0.06224 ^
Collector Dosage*% of solids	1.060	0.08716
Depressant Dosage*Depressant Dosage	0.925	0.11898
Collector Dosage*Depressant Dosage	0.896	0.12698

Fig 17. Parameters effect summary

When looking for the surfaces generated during DoE analysis, it is possible to observe an interesting compromise between quality and recovery when using collector dosages between 150 and 250 g/t near to pH 10 (Fig 18).

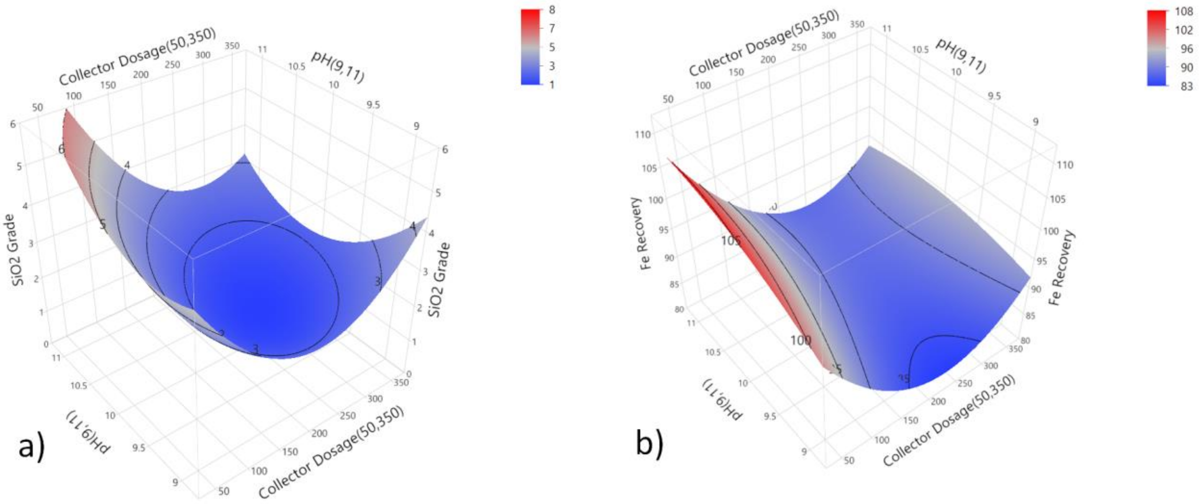
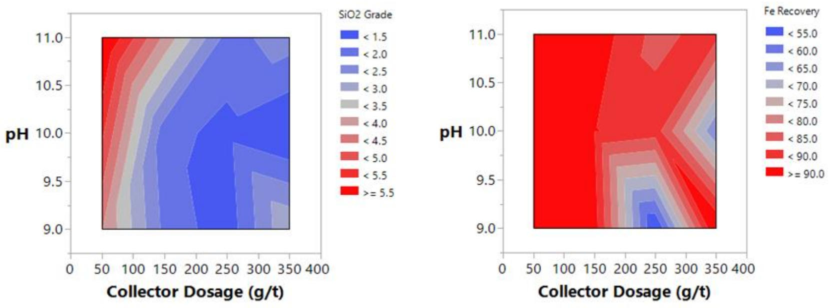


Fig 18. Collector dosage vs pH surface

The statement above is well illustrated in the contour plot showed in Fig 19. It is interesting to observe that bad qualities are obtained when adopting low dosage of collector in high pHs, while a considerable drop in Fe recovery is observed in low pHs and high dosages of reagent.



a) b)

Fig 19. Collector dosage vs pH contour plot

Despite presenting a lower impact in the results, the dosage of depressant is still an important factor to consider. In this way, the surface presented in Fig 20 indicates a good performance in flotation process for collector dosages between 150 and 250 g/t and depressant dosages between 400 and 600 g/t.

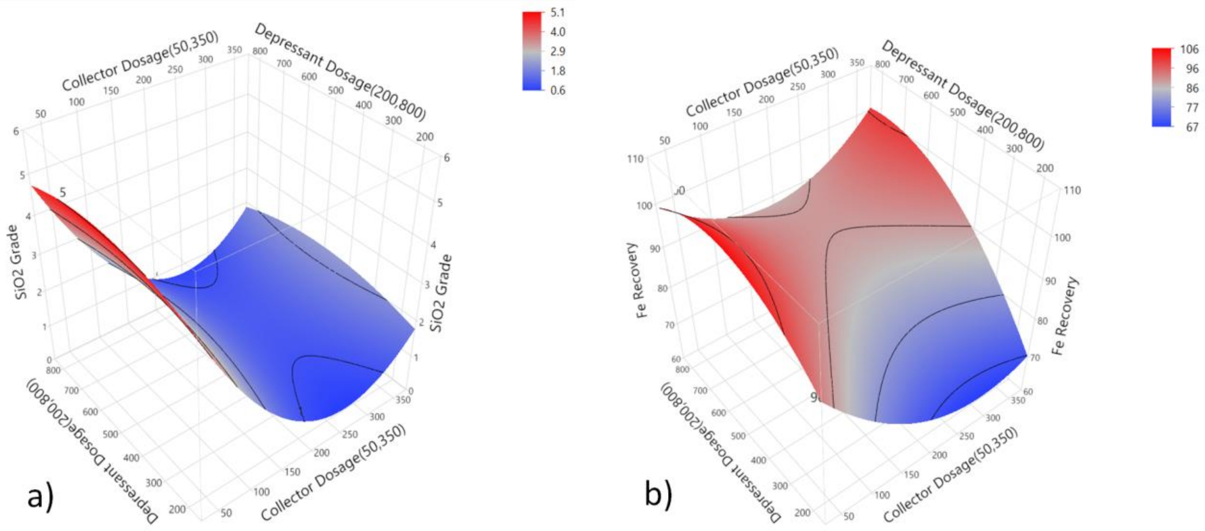
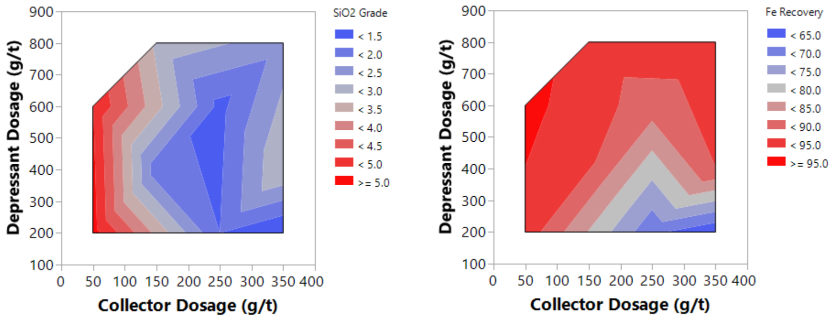


Fig 20. Reagent's dosages surfaces

An important point to highlight corresponds to the fact that it is not possible to achieve the desired target with collector dosages below 150 g/t, as presented in the contour plots of Fig 21.



a) b)

Fig 21. Reagent's dosages contour plot

Finally, the results obtained during the research proved the feasibility of producing a DRI product from Cleaner LIMS concentrate using flotation, thus answering the initial objective of the study. In this context, the original flowsheet proposed by MMP team was updated with the inclusion of the flotation circuit (desliming + flotation) after LIMS operations, as illustrated in Fig 22.

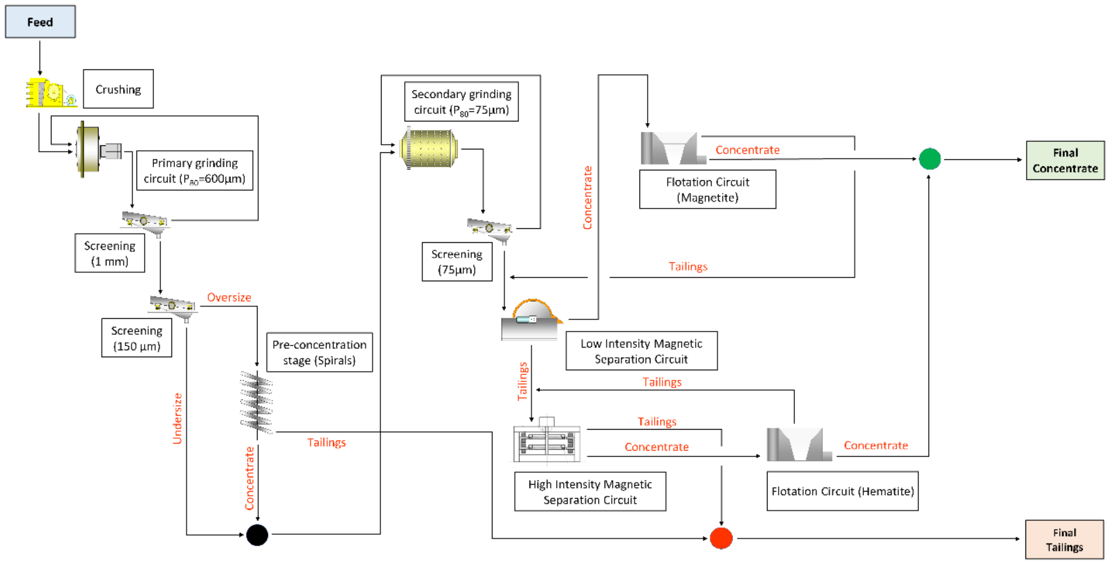


Fig 22. Updated flowsheet

Aiming at showing the improvement achieved, a comparison between the initial scenario and the situation after the addition of the flotation stage is presented in Table 8. In this case, given the results obtained in test 16, it was selected as a good candidate to illustrate flotation process operation.

Table 8. Flotation circuit performance evaluation

Product	**Global Yield (%)**	**Global Fe Rec. (%)**	**Grade (%)**												
			Fe	**FeO**	**SiO_2**	**Al_2O_3**	**CaO**	**MgO**	**TiO_2**	**P**	**Mn**	**S**	**Na_2O**	**K_2O**	**LOI**
Cleaner LIMS concentrate	28.3	58.4	66.28	27.58	6.90	0.10	0.37	0.46	0.01	0.002	0.29	0.018	0.04	0.00	-2.58
Flotation concentrate	23.9	52.0	70.04	29.90	1.68	0.09	0.21	0.44	0.01	0.002	0.30	0.001	0.02	<0.001	-2.99

After the inclusion of flotation operation, the quality of the final magnetite product increased from 66.28% Fe to 70.04% Fe, while the SiO_2 levels decreased from 6.90% to 1.68%. Despite the considerable increase in quality, an impact in the recoveries was notices, as expected. This highlights the importance of recovering the hematite contained in the tailings of LIMS operation, as a solution for increasing the overall efficiency of Mont Reed plant, as it is already considered by MMP team in the proposed flowsheet.

4. Conclusion

In this work the use of flotation was evaluated to increase the quality of Mont Reed magnetite concentrate to DRI levels (<1,8% SiO_2) and proved to be feasible. The methodology was able to decrease the silica amounts of the concentrate in 75% (from 6.90% SiO_2 to 1.68% SiO_2) while returning a high-grade product with 70.04% Fe. A desliming stage showed to be mandatory to arrive in these results, where flotation operation performed using 150g/t of collector, 400g/t of depressant, pH 10 and 30% of solids in pulp presented a good compromise between quality and iron recovery. These results are of uttermost importance for Mont Reed exploitation as they increased the value created on the asset, making it a potential source of DRI pellet feed for the global iron ore market. In addition, similar investigations are suggested for the recovery of hematite within the circuit, assuring then the best use of the mineral resource.

5. References

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