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Developing an optimum beneficiation route for a low-grade chromite ore

İlkay Bengü Can [1], Büşra Özsoy [2], Ş. Levent Ergün [1]

[1] Department of Mining Engineering, Hacettepe University, Beytepe, Ankara, Turkey

[2] Gördes Meta Nickel Cobalt Facility, Manisa, Turkey

Corresponding author: ilkay@hacettepe.edu.tr (İlkay B. Can)

Abstract: In this study, an optimum beneficiation route was developed for an existing concentrator that processes low-grade chromite deposits. This objective is challenging as the grade of the ore under consideration is approximately 5%, which is the lowest grade processed in Turkey. Detailed characterization, process mineralogy and liberation studies were performed. The optimum fineness of grind was found to be 100% finer than 0.4 mm after the initial beneficiation tests. Laboratory test work was performed using a combination of a teetered bed separator (TBS), spiral concentrator and shaking table. Two different circuit alternatives including a TBS followed by only shaking tables and a combination of a spiral concentrator, TBS, and shaking tables were compared. It was found that an alternative comprising a spiral concentrator, TBS, and shaking table provided better results, required less equipment, etc. Considering these results, a provisional flowsheet was developed, and a total final concentrate of 6.8% by weight with 49.5% Cr_2O_3 grade and 71.51% Cr_2O_3 recovery could be obtained. The detailed laboratory test work was followed by plant-scale trials for the verification of the experimental findings with different circuit configurations. From roughing and scavenging spiral groups, ~85% of chromite could be obtained prior to the shaking table concentration, and this led to a minimum increase of 20% in the total recovery with less footprint of the plant, reduced water usage, and lower operating costs.

***Keywords*:** chromite, beneficiation, spiral concentrator, teetered bed separator

1. Introduction

The beneficiation of chromite (Burt 1984; 1999) and processing of its low-grade ores (Ağaçayak, 2004; Amer and İbrahim, 1996; Sönmez and Turgut, 1998) are traditionally performed using gravity concentration methods. It has been observed in many studies that to provide a consistent concentrate production, different equipment combinations of the same enrichment method and/or combinations of different enrichment methods can be used in the beneficiation of chromite ores and plant design (Tripathy et al., 2013; Banerjee et al., 2006; Gence, 1999; Murthy et al., 2011; Ozdag et al., 1993). Tripathy et al. (2012) and Aslan and Kaya (2009) reported examples of the use of different enrichment methods to improve the Cr/Fe ratio and produce a saleable concentrate using gravity-concentration high-intensity magnetic-separation combinations. It is apparent that the liberation degree of ore minerals, type of host rock minerals, grades of Al_2O_3, SiO_2, FeO, and Cr_2O_3 in the concentrate, Cr/Fe ratio, and some physical and chemical properties of all these minerals in the feed designate the selection of the beneficiation methods and routes (Bergmann et al., 2016; Tripathy et al., 2013). Even if the mineral to be enriched is the same, to beneficiate the ores obtained from different regions, the corresponding distinctive process flowsheets should be studied. In other words, the design of the beneficiation routes should be ore based.

Shaking tables may be regarded as the most efficient gravity concentration equipment within a 1–0.1-mm range. However, they cause high water consumption and suffer from a low capacity per unit area. The conventional hydraulic classifier, known as a hydrosizer, is commonly used prior to using

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shaking tables to sort material in narrow size/density fractions in order to improve the metallurgy (Burt 1984). Besides conventional systems, a new gravity concentration equipment, the vibrating table, has recently been developed and tested for chromite ores (Gülsoy and Gülcan, 2019).

The hydraulic-classification shaking-table-based flowsheet is predominantly used in Turkey. There exist over 100 plants using similar flowsheets. These are characterized by poor instrumentation and a lack of water control. The recovery has been low in these plants, and several studies have been published to recover chromite from tailings (Cicek and Cocen, 2002; Mohan Rao et al., 2006) and ultrafine particles (Guney et al., 2001; Tripathy et al., 2010; Tripathy and Murthy, 2012; Atalay and Ozbayoglu, 1992). Multi-gravity separators were preferred in many researches as the major fine gravity separator for treating chromite fines and plant tailings (Özkan and İpekoğlu, 2001; Ozdag et al., 1994; Ucbas and Ozdag, 1994; Aslan and Kaya, 2009; Gence, 1999; Ozdag et al., 1993). The column flotation method was also investigated for the recovery of chromite from slime fraction and wastewater streams (Feng and Aldrich, 2004; Guney et al., 1991). In contrast to the conventional beneficiation methods, the hydro cyclone and water-only cyclone seems to have potential for the recovery of chromite fines (Tripathy et al., 2017).

It was reported that 83% of the known chromite deposits of Turkey comprise less than 10% Cr_2O_3 grade (Karahan and Özkan, 2011). Although it is difficult to process these deposits because of their low grade, Turkish chromites are considered to be good for metallurgical use owing to their higher Cr/Fe ratio. While the silicate ratio is important for the energy requirement in ferrochrome production, the ratio of Cr/Fe increases the economic value of the ore. Therefore, in addition to providing a cost advantage, chromite ores with such properties are widely preferred by ferrochrome producers (Erol and İnce, 2012). In contrast, the price of chromite concentrate has been fluctuating. This fluctuation has forced mining companies to shut down their operations in low-price periods, which has further increased the operation costs. Decreasing feed grades, fluctuating chromite prices, and poor concentrator operations result in an undesirable environment for Turkish chromite producers. A paradigm change is necessary for sustainable chromite mining and processing in Turkey.

In this study, a novel flowsheet for a low-grade, large chromite deposit was developed. The existing concentrator, which had an installed capacity of 200 tph and included a hydraulic classifier and shaking tables, had to be suspended owing to the decreasing metal prices. The metallurgical performance of the final concentrate was recorded as 45–50% recovery with 46–48% Cr_2O_3 content when the feed size was 100% finer than 0.85 mm (Ergun et al., 2017). However, it was found that the ore deposit had a measured + indicated reserve of 150 million tons with a further inferred reserve, but with much finer grains. Therefore, it was decided to double the capacity of the plant to improve the economics of the project further. Installing such a plant with that capacity into an existing building requires a much smaller footprint and proper selection of equipment. However, as the ore grade is relatively low, more careful planning of the enrichment steps is required to obtain a concentrate of the desired metallurgical quality. In this study, spiral concentrators were selected as a pre-concentration stage since it was possible to remove nearly 50% of the feed material as tail and thereby the amount of material which would be the feed of shaking tables were significantly reduced. Considering the feed particle size and capacity of the plant, it is obvious that the concentration with only shaking tables will result in an inefficient and hard to control process as the number equipment will be drammatically increased. Concentration with only spirals were also considered but it was not possible to attain a product with target grade at desired recovery, therefore shaking tables were used after spirals for upgradation purpose. As teetered bed separators are more efficient at high capacities rather than conventional hydraulic classifiers, they were used for classification pupose prior to shaking tables. Thus, the objective of this study is to develop a more metallurgically and costwise efficient flowsheet for the existing concentrator. Therefore, different circuit alternatives including various combinations of spirals, teetered bed separator (TBS), and shaking tables were studied. A detailed characterization study of minerals and lockings was also conducted on both a run-of-mine (ROM) ore and some specific streams to facilitate the selection of the appropriate equipment. In the case of both the beneficiation alternatives, the objective was to obtain a saleable concentrate having a target grade of 48% Cr_2O_3 with an increased metal recovery, and as a result, a suitable flowsheet was proposed. At the second stage of the study, plant-scale trials were performed based on the proposed flowsheet, and these trials confirmed the findings obtained in the laboratory.

2. Materials and methods

The ore samples were supplied from a new low-grade chromium ore deposit located in Eastern Anatolia in Turkey, and they were characterized in detail in terms of their physical, chemical, and mineralogical properties. The grindability of the ore was determined using the standard Bond Work Index method with a test sieve having a size of 106 µm. The size-by-size Cr_2O_3 content was analyzed using the wet titration chemical analysis method. The modal mineralogy of the ROM ore and mineral liberation/lockings on the size basis was determined using a Mineral Liberation Analyzer (MLA). The mineralogical analysis was performed using an FEI MLA 650F, with XMOD for modal mineralogy and the XBSE (X-ray back-scattered electron) method for particle data. Using a combination of image analysis with atomic number contrast imaging (from the back-scattered electron (BSE) signal intensity) and Energy Dispersive Spectrometry (EDS) using Bruker 5010 SDD detectors, minerals and other attributes were directly measured using automated scanning electron microscope systems. The BSE signal intensity is proportional to the mean atomic number of the minerals. The product streams were also examined at each beneficiation step using Clemex Image Analysis. Size-by-size fractions of the ore were also subjected to heavy liquid tests to determine the appropriate size and gravity concentration method to be implemented.

Different beneficiation routes were tested based on the data gathered from the characterization tests. In these routes, a TBS, spiral concentrator, and shaking table were used for classification, pre-concentration, and final concentration purposes, respectively. In the majority of the chromite processing plants, typical pocket type hydrosizers are used prior to the shaking tables. In contrast, TBS was preferred as a classifier owing to its potential for use in coal processing (Drummond et al., 2002), chromite mineral upgradation (Kari et al., 2006; Kapure et al., 2007; Kumar et al., 2009), and enrichment of iron ore tailings (Ozcan and Celik, 2016) and mineral sands (Galvin et al., 2016).

In one of the circuit configurations, a spiral concentrator was tested within a combination of TBS and shaking tables as it is widely used to beneficiate a vast number of ores such as chromite, iron ore, coal, and beach sands primarily owing to its operational simplicity and cost effectiveness (Mishra and Tripathy, 2010; Tripathy and Murthy, 2012). Some literatures report on the optimization of the process parameters such as feed rate, pulp density, and splitter position and agree that among all these variables, the splitter position has significant influence on both the grade and recovery (Tripathy et al., 2010). The spiral concentrator has been used primarily for gangue mineral rejection and for reducing the number of equipment in downstream processes (Özsoy, 2016).

The conformity of the data obtained in the case of the best beneficiation alternative was verified with plant application onsite.

3. Results and discussion

3.1. Characterization

Size-by-assay analyses were conducted on ROM ore, and the obtained results showed that the head assay of the ore was 5.13% of Cr_2O_3 and indicated an accumulation in −0.425 mm size (Fig. 1). The elemental analysis of the feed sample is presented in Table 1.

MLA was used on crushed and sieved ore of −200+20 µm size fraction in order to characterize the modal mineralogy. The −20-µm fraction was omitted because it contained very fine particles that could affect the accuracy of the liberation analysis by means of agglomeration. Mineralogical studies have shown that the ore is mainly associated with magnesium silicates (forsterite) and oxide minerals (brucite) with ~5% of chromite content. The dominance of forsterite as compared to the other minerals-including the Cr content-except for chromite, simplifies the identification of the ore (Table 2). As forsterite and brucite have lower density values than the chromite mineral, it is expected that they will be advantageous in the gravity enrichment. The percentages of Cr and Fe in chromite were 46.46% and 24.94%, respectively.

The degree of liberation is a crucial parameter that designates the separability and efficiency of all the processes. Fig. 2 shows the liberation and mineral locking graph of chromite and forsterite as it is the major gangue mineral in the ore. It can be observed that both minerals have a liberation degree of more than 80% in the particle size range of −200+20 µm. The grindability of the ore was measured as 16.03 kWh/ton.

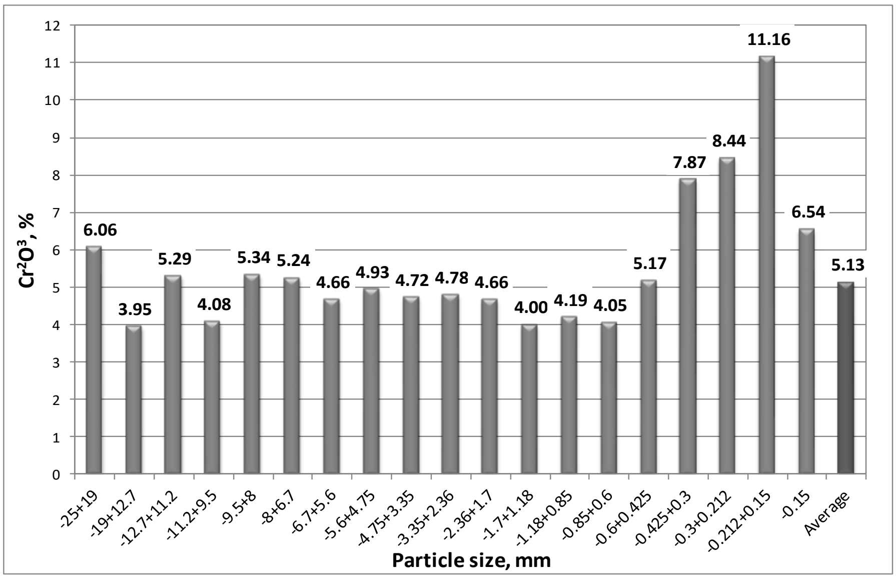

Fig. 1. Size-by-assay analysis of ROM chromium ore

Table 1. Complete chemical analysis of feed sample

Element	Weight, %	Element	Weight, %
Al	0.07	H	0.20
C	0.17	Li	0.00
Ca	0.61	Mg	32.28
Cr	2.44	Ni	0.01
Cu	0.01	O	45.09
F	0.01	S	0.03
Fe	1.56	Si	17.51
Total	100.00		

Table 2. Modal mineralogy of ROM ore

Mineral	Density, g/cm³	Chemical Formula	Distribution, %
Forsterite	3.27	$Mg_2(SiO_4)$	86.46
Chromite	4.8	$Fe^{2+}Cr_2O_4$	5.25
Brucite	2.4	$Mg(OH)_2$	5.48
Calcite	2.71	$Ca(CO_3)$	1.24
Dolomite	2.85	$CaMg(CO_3)_2$	0.18
Quartz	2.63	(SiO_2)	0.21
Hematite	5.3	$Fe^{3+}2O_3$	0.21
Clinochlore	2.65	$(Mg,Fe^{2+})5AlSi_3Al_2O_{10}(OH)_8$	0.39
Diopside	3.4	$CaMg(Si_2O_6)$	0.20
Goethite	3.8	$Fe^{3+}O(OH)$	0.07
Actinolite	3.04	$Ca_2Mg_3Fe^{2+}2Si_8O_{22}(OH)_2$	0.03
Plagioclase	2.69	$Na_{0.5}Ca_{0.5}Si_3AlO_8$	0.05
Pyrite	5.01	$Fe^{2+}S_2$	0.04
Kaersutite	3.24	$NaCa_2Mg_4Ti(Si_6Al_2O_{23})(OH)_2$	0.08
Others	(Fluorite, chalcopyrite, muscovite, barite, etc.)		1.57
Total			100

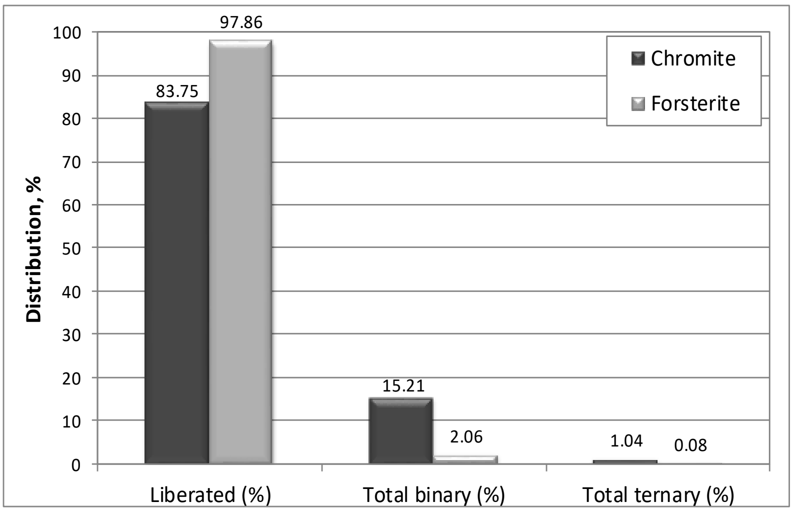

Fig. 2. Liberation and locking states of chromite and forsterite minerals

It can be observed that 15% of the chromite is locked with other minerals in the binary form. The MLA listed all the locking minerals and 13.46% of this amount was resulted from forsterite mineral (Table 3). Forsterite was more liberated than chromite; approximately 2% of it showed locking with chromite and other minerals.

Table 3. Mineral locking forms of chromite and forsterite

Chromite Locked with	Binary Form (%)	Ternary Form (%)
Forsterite	13.46	0.59
Brucite	0.15	0.06
Kaersutite	0.00	0.01
Calcite	0.13	0.00
Hematite	0.60	0.16
Clinochlore	0.85	0.09
Plagioclase	0.00	0.11
Diopside	0.01	0.00
Clinozoisite	0.00	0.02
Forsterite Locked with	**Binary Form (%)**	**Ternary Form (%)**
Pentlandite	0.01	0.00
Chromite	0.94	0.02
Brucite	0.33	0.00
Kaersutite	0.00	0.01
Calcite	0.20	0.01
Dolomite	0.01	0.00
Quartz	0.01	0.00
Hematite	0.26	0.01
Clinochlore	0.01	0.01
Actinolite	0.01	0.00
Diopside	0.07	0.01
Galena	0.00	0.00
Goethite	0.14	0.00
Awaruite	0.04	0.00
Clinozoisite	0.02	0.00

A stereomicroscope equipped with Clemex Vision P.E 5.0 image analysis software was also used to determine the liberation of the chromite mineral at coarser particle sizes. Considering the gravity enrichment methods for application, the ore was crushed down to −1 mm and sieved to narrow particle size fractions to facilitate the liberation analysis. According to results presented in Fig. 3, the degree of liberation of chromite is very low, particularly in the case of coarse sizes; however, it is clear that significant liberation can be achieved at sizes finer than 0.300 mm, which was also supported by the MLA.

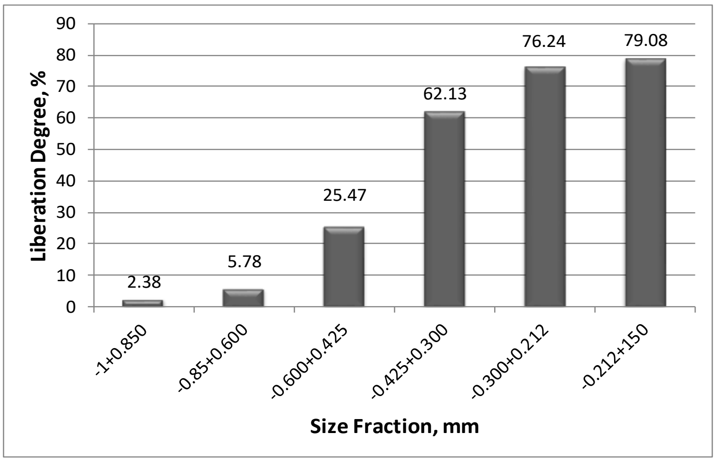

Fig. 3. Liberation degree of chromite at sizes less than 1 mm

It is well known that there is a critical balance between sufficient grinding for liberation and fine losses. Therefore, a study was conducted with a material of −1 mm size, and the preliminary works performed on this size showed that it was possible to obtain a salable grade (48% Cr_2O_3) concentrate. However, the weight of the products and metal recovery values were significantly low. The images captured using the Clemex Vision stereomicroscope indicated that the major factor affecting the separation performance was insufficient liberated chromite particles, particularly at sizes just below 1 mm (Fig. 4). Therefore, there arose the necessity for enrichment at finer particle sizes.

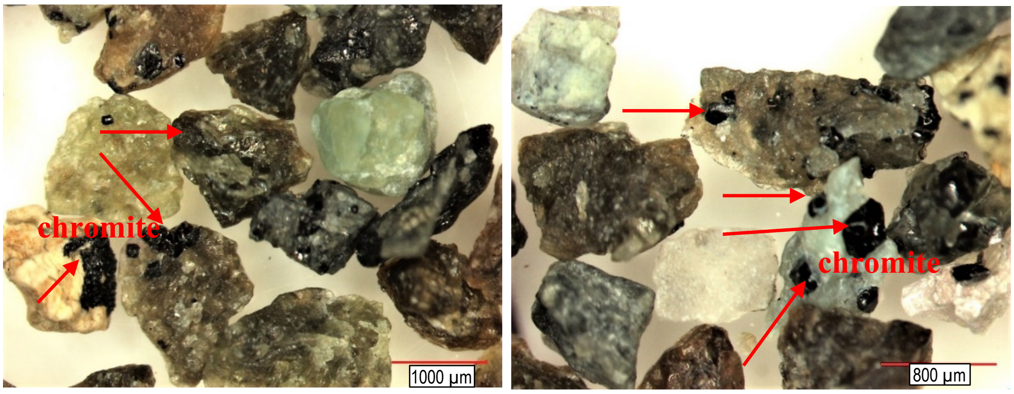

Fig. 4. Stereomicroscope images of chromite particles at −1 mm size

However, to decide the appropriate enrichment size and method, the ore was prepared in narrow size fractions such as −600+75 µm, −500+75 µm, −425+75 µm, and −300+75 µm, and then each was subjected to a heavy liquid test at a density of 2.6 g/cm^3. Prior to the heavy liquid test, the particles finer than 75 µm in each fraction were discarded through decantation as it was anticipated that they would be lost (particularly the light particles) in the slime fraction. The amount of slime fraction was recorded in coarser (+300 µm) size fractions as approximately 35% of the material and nearly 20% metal was lost; however, in the case of the −300 µm size, the amount of slime was increased to 41% with 24% Cr_2O_3 grade. It was observed from the heavy liquid test results that the values obtained from the −425+75 µm and −300+75 µm fractions were similar (Table 4). However, the decantation results indicated that as the size became finer, the metal lost was expected to be greater. Therefore, the following beneficiation tests were performed at size of −425 µm, which was selected as the optimum fineness of grind.

Table 4. Heavy liquid test results

Ore Fractions		Weight (%)	Cr_2O_3 (%)	Recovery (%)	According to Feed (discarded) Total Recovery (%)
−600+75 µm	2.6 Sink	16.26	33.31	83.84	67.12
	2.6 Float	83.74	1.25	16.16	
	Total	100.00	6.46	100.00	
−500+75 µm	2.6 Sink	14.68	34.90	74.55	59.68
	2.6 Float	85.32	2.05	25.45	
	Total	100.00	6.87	100.00	
−425+75 µm	2.6 Sink	20.39	31.85	94.98	76.73
	2.6 Float	79.61	0.43	5.02	
	Total	100.00	6.84	100.00	
−300+75 µm	2.6 Sink	20.40	31.39	98.56	75.22
	2.6 Float	79.60	0.12	1.44	
	Total	100.00	6.50	100.00	

3.2. Beneficiation tests

Based on the characterization study, the feed material used in all the tests was prepared as 100% −425 µm (P_{80}: 288 µm, P_{50}: 160 µm) through crushing and grinding. It was necessary to remove the very fine particles, which may have negatively affected the gravity concentration. Therefore, the slime rejection from the 75 µm particles was carefully performed based on the light particles. The amount of −75 µm size particles in the feed was measured as approximately 29%. As for the +75 µm size particles, a TBS was used to prepare the appropriate feed streams to the shaking tables. While the underflow of the TBS was processed at coarse-size shaking tables, the overflow was processed at fine-size shaking tables. Two beneficiation alternatives including the use of a TBS followed by only shaking tables and a combination of a spiral concentrator, TBS, and shaking tables were studied. Based on the metallurgical results, the middling of the shaking tables was re-processed at tables, particularly in the case of coarse particle sizes.

3.2.1. Alternative 1: TBS followed by shaking tables

A total concentrate of 4.17% by weight was obtained from the coarse, middling, and fine shaking table stages with 49.55% Cr_2O_3 grade and 41.89% recovery (Fig. 5). This calculation represents the experimental condition in which there was no circulating stream in the entire circuit. It was found that the use of a single concentration stage was impossible, and thus, the middling fractions were required to be re-concentrated accordingly. Consequently, the number of shaking tables would be greatly increased. Considering the proposed size distribution and treatable capacity of the plant (Bogdanov, 1983), the number of shaking tables for the coarse and fine size processing was calculated to be 242 and 272, respectively. This is equal to a total of 514 shaking tables, which requires a large amount of equipment and results in inefficient and difficult process control.

3.2.2. Alternative 2: Combination of spiral concentrator, TBS, and shaking tables

As the ore is regarded as a low-grade deposit, another alternative including stages comprising a spiral concentrator is considered in which obtaining a pre-concentrate is possible. For this purpose, an Australia-made Mineral Deposit A87D type of spiral concentrator (∅: 600 mm, L: 5 m, 7 turns) was run with an average capacity of 1 tph with 35% solid in feed and operated in three stages. The first stage was used as the rougher stage, while the following two stages were used as scavenger stages. The concentrates of the rougher spiral and first stage scavenger spiral were mixed based on their Cr_2O_3 content and classified using a TBS to prepare the feed streams of the coarse- and fine-size shaking tables (Fig. 6).

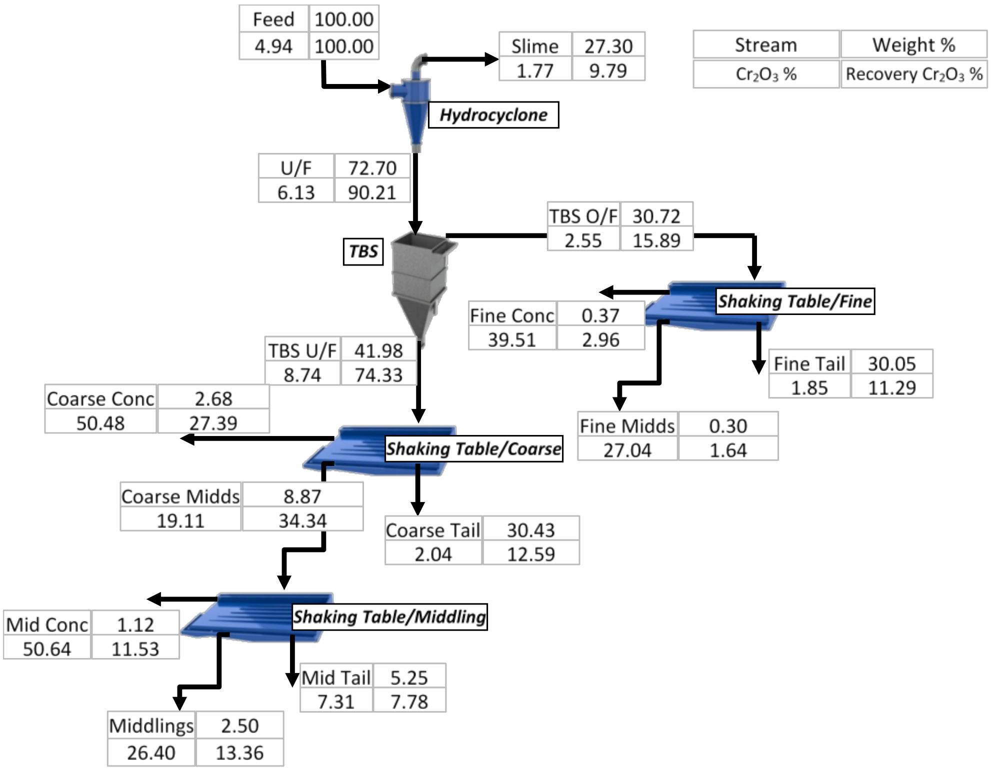

Fig. 5. Beneficiation test results of alternative 1 (TBS followed by shaking tables)

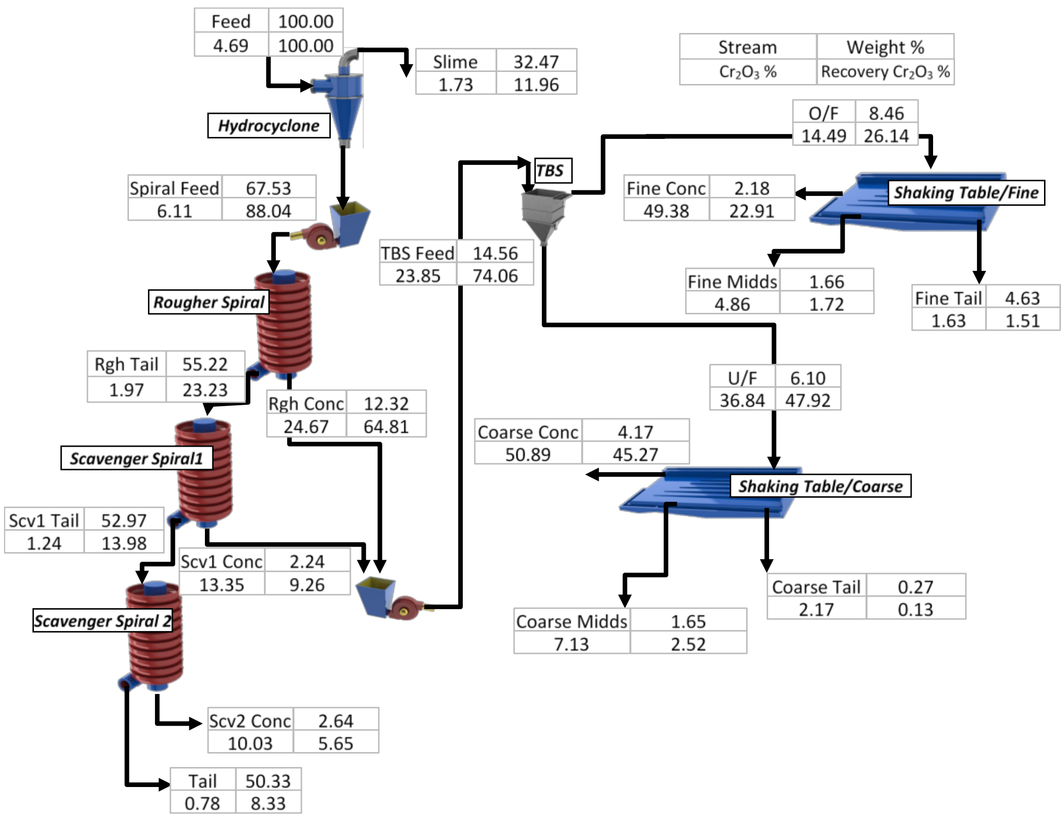

Fig. 6. Beneficiation test results of alternative 2 (spiral concentrator, TBS, and shaking tables) open circuit

In alternative 2, the total concentrate was the combined product of the coarse- and fine-fractions processing of the shaking tables. Using the spiral concentrators, the Cr_2O_3 grade of the final tail could be reduced to 0.78%, and approximately 50% of the feed was removed by tail; thereby, only approximately 15% of the entire material was processed using shaking tables. This means that this process will be cost efficient in terms of the number of equipment as a significant portion of the feed leaves the system in the pre-concentration step before the shaking tables. It is expected that checking the chromite liberation and its locking type should contribute to a better understanding of the beneficiation mechanisms, performance of the separator, and prediction of a chromite particle behavior in downstream processing, as has also been emphasized by Leroy et al. (2011) and Pascoe et al. (2007). Therefore, the locking statements of the second-stage scavenging spiral concentrate and the middlings of the shaking tables were examined under a stereomicroscope, and the degree of chromite liberation was calculated based on size in order to determine whether they were required to be recycled or milled. It was found that an acceptable liberation of chromite of 87% occurred at −106 µm for the middling of the fine-size shaking table and of 89% and 95% occurred at −150 µm for the middlings of the coarse-size shaking table and scavenging spiral concentrate, respectively. Considering that the optimum fineness of grind was −150 µm, two stages of middling beneficiation circuit including a grinding section was designed based on a simulation study using the JKSimMet software program (Fig. 7).

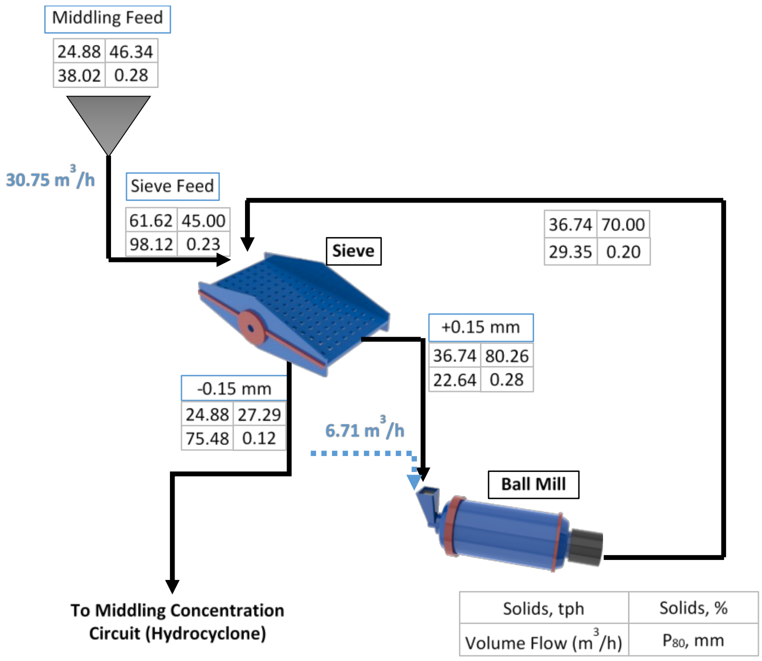

Fig. 7. Grinding circuit of middling products

As the recovery was quite low for a single-stage concentration of the middlings, two stages of concentration were designed (Fig. 8). Even though the design of the circuit treating the middlings was performed in terms of the integrity of the work, the amount of chromite in the middlings was approximately 10%, and an increase in metal recovery of only 1% could be achieved. Therefore, it is recommended that a space be allocated for such a circuit in the plant design, which may be installed if required.

As this concentration alternative provided better results than only the shaking table alternative, a mass balance with a planned capacity was performed by taking into consideration the circulation of the middling products back to their feeds using an Excel-based program. The final beneficiation flowsheet was proposed as shown in Fig. 9, as no significant achievement was attained by re-concentrating the middlings after the milling. According to this, it can be observed that the coarse and fine middlings were recirculated back to the TBS while the second-stage scavenging spiral was omitted. In this case, a total final concentrate of 6.8% by weight with 49.5% Cr_2O_3 grade and 71.51% Cr_2O_3 recovery could be achieved. Owing to the operating conditions, the use of a dewatering equipment was envisaged, which may be required to remove the excess water from the overflow of the TBS.

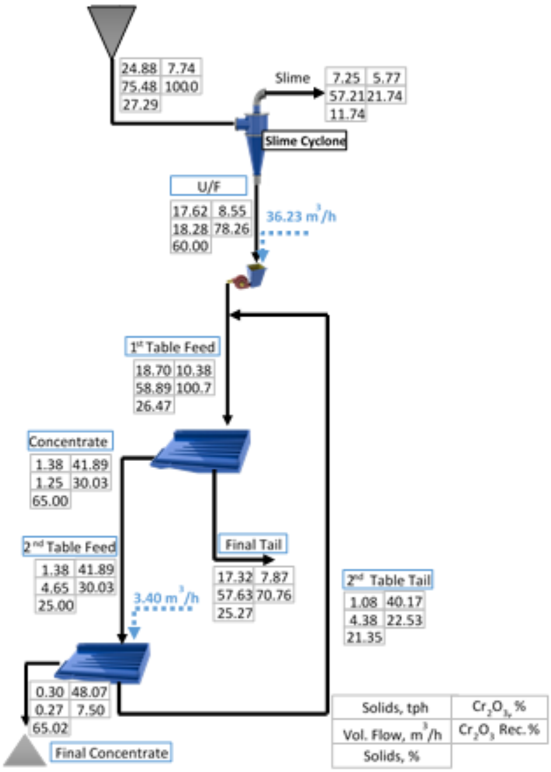

Fig. 8. Concentration circuit of middling products and their mass balance

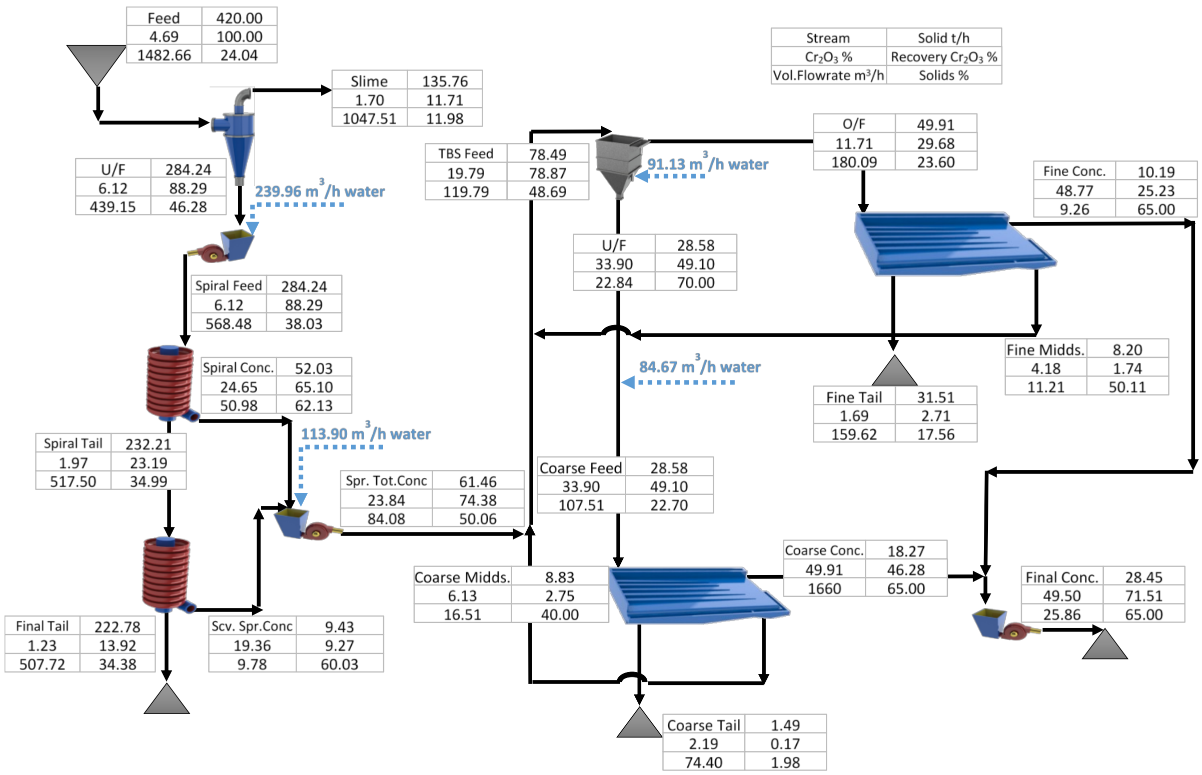

Fig. 9. Proposed beneficiation circuit

In accordance with this proposed flowsheet, the required numbers of equipment were calculated considering a particle size distribution of −425 µm and capacity of 420 tph. For the two stages of spiral concentration, 17 batteries including 36 spirals for each should be used. The bulk concentrate of the spirals should be processed by 107 shaking tables, of which 50 and 57 should be used for the coarse- and fine-size particles, respectively. However, this number was calculated as 514 for the single-stage shaking table processing. It is known that while shaking tables provide more successful separation than spiral concentrators in single-stage separation, they have some disadvantages such as operational difficulties and limited capacities. It is apparent that when shaking tables are used in conjunction with spiral concentrators, which have a much simpler operation, the number of tables required to be used in the plant is significantly reduced, and the process will be much easier to control. In addition, the number of circulating streams, and water and footprint usage in the plant will also be reduced.

The results of the experimental studies indicated that sufficient efficiency cannot be ensured with single-stage shaking tables even if fine grinding is performed. Moreover, middling products must be ground in an additional step. This makes the circuit complex and necessitates the use of a large number of shaking tables. In contrast, spiral tests show that a more efficient pre-concentration is possible, particularly on re-treating the tails.

3.3. Plant trial

The plant trial campaign was conducted to investigate the best possible separation that was achievable using the latest available spiral technology. For this purpose, new spiral concentrators were implemented in the scavenging and cleaning stages. In this scope, a 25-tph trial circuit was thus built in the existing plant, and the effects of the feed rate, spiral settings, and cyclone parameters were evaluated. The roughing (first spiral group) and scavenging (second spiral group) stages of the spiral concentrators were operated with shaking tables in different flowsheet combinations. The average grade and recovery values obtained in these trials are summarized in Table 5. When the performances of the spiral groups are compared, it is observed that 64.34% of chromite is obtained from the first group spiral concentrate while 19.88% of chromite is obtained from the second group spiral concentrate. Subsequent to the processing of the spiral concentrates in the shaking tables, a final concentrate with 48.18% of Cr_2O_3 grade was obtained.

Table 5. Summary of grade and recovery values of streams in plant trials

	Plant Feed	First Spiral Group			Second Spiral Group	
		Feed (U/F of Cyclone)	Concentrate	Second Spiral Feed	Concentrate	Tail
			Grade, %			
Highest	7.53	10.49	28.28	3.65	10.40	3.54
Lowest	4.77	6.40	14.52	2.27	3.07	1.57
Average	6.02	8.22	20.54	3.22	5.34	2.23
			Recovery, %			
Highest	100.00	94.78	72.78	30.95	22.40	8.66
Lowest	100.00	72.36	58.72	17.61	14.69	3.63
Average	100.00	87.12	64.34	24.26	19.88	5.92

In addition to alternative of the spiral concentrator and shaking table combination, the utilization of only spiral options was also tested using various roughing and scavenging combinations. However, it was observed that for equal concentrate grades, the spiral option with shaking table provided better results than the option of only the spiral.

Fig. 10a presents the summary of the plant trials in terms of the concentrate mass pull against the metal recovery and upgrading ratio. As observed, it is possible to obtain a high metal recovery with an increase in the concentrate mass pull; however, considering the upgrading ratio, it would be best to operate the spiral concentrator to obtain a mass pull of approximately 70%. The plant trial results were in good agreement with the results of the laboratory-scale tests.

The separation efficiency is a measure of the metallurgical efficiency, which defines the recovery difference between the valuable and gangue mineral to the concentrate. According to equation (1) given below (Schulz, 1970), the separation efficiency values obtained from the plant trials are also calculated and shown in Fig. 10b.

$$SE = R\frac{c_m(c-f)}{c(c_m-f)} \tag{1}$$

where SE is separation efficiency (%), R is recovery of valued constitute in concentrate (%), c_m is assayed element in mineral being concentrated (%), c is assayed element in the concentrate (%), f is assayed element in feed (%).

In the separation efficiency formula, *f* is considered as the feed just before the spiral concentration stage. From Fig. 10b it can be concluded that a satisfactory separation could be obtained with approximately 70% concentrate mass pull.

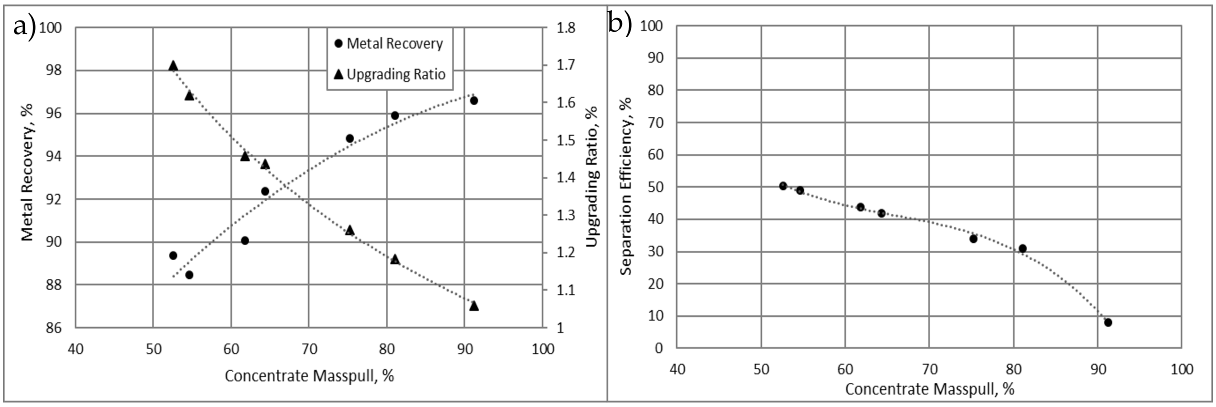

Fig. 10. Metallurgical performance summary (a) and separation efficiency values (b) of plant trials

3.4. Further remarks

As there is an existing concentrator building, shaking tables and thickeners are available, 240 tph of capacity is ready to start-up.

Further studies have been conducted to further improve the performance; however, the details of these studies are not presented in this paper. The design of a larger plant with a capacity of 420 tph will be then be completed. These are:

- The use of high-pressure grinding rolls prior to the grinding mill has resulted in a significantly reduced specific energy consumption. Pilot scale tests performed in Germany proved this without increasing the amount of slime produced. However, no selective liberation was observed after performing a careful mineralogical analysis and heavy liquid tests.
- To further decrease the footprint of the plant, a TBS can also be used for beneficiation purposes. The results of the initial tests were promising.
- The cyclone overflow before the spiral circuit contains 10–12% of the total Cr_2O_3. This fraction may also be beneficiated using a multi-gravity separator (MGS) after a secondary desliming using small-diameter cyclones. Two-stage applications of the TBS and MGS are expected to be tested at the plant next.

4. Conclusion

The following conclusions can be drawn based on the obtained results:

- The different beneficiation routes such as TBS followed by only shaking tables and the combination of a spiral concentrator, TBS, and shaking tables were compared. With the spiral concentrator option, a better metallurgy could be achieved, i.e., a total final concentrate of 6.8% by weight with 49.5% Cr_2O_3 grade and 71.51% Cr_2O_3 recovery was obtained.

- A higher mass pull in the spiral concentrator increased the overall recovery. In the plant trials, up to ~85% of the total chromite recovery was obtained from the roughing and scavenging spiral concentrators prior to the shaking table concentration, and considering the upgradation ratio, the operation of spiral concentrators with nearly 70% of mass pull was proposed.
- The performance predicted in the laboratory and the results of plant trial tests were very similar.
- The performance obtained with the new flowsheet was superior to that obtained with the existing flowsheet. A minimum increase of 20% was obtained in the recovery.
- With this modification, the footprint of the plant, water usage, and operating costs would decrease substantially.
- The metallurgical performance of the chromite beneficiation flowsheets can be improved significantly with the implementation of the aforementioned new technologies and design philosophies.

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