SEYED AHMAD HASHEMI*, BAHRAM REZAI*,
MOHAMMAD REZA TAVAKOLI MOHAMMADI**, SEPIDEH JAVANSHIR***

CHARACTERIZATION AND CONCENTRATION STUDIES OF JALAL ABAD IRON MINE

CHARAKTERYSTYKA I BADANIA KONCENTRACJI W KOPALNI RUD ŻELAZA JALAL ABAD

Characterization and determination of liberation degree are the first stages of ore dressing. Block 4 of Jalal Abad mine, Kerman province, Iran, has three kinds of iron ores; D1, D2 and D3, with different grades. In this research, chemical analysis, mineralogy, liberation degree and magnetic enrichment studies were done by XRF, XRD, microscopic sections and Davis tube, respectively. The results indicated that D1, D2 and D3 had average iron grades of 58, 52 and 38%, respectively. The minerals of Magnetite, Hematite, Dolomite, Calcite and Quartz were distinguished. Average liberation degree was estimated about 500 μm by Microscopic studies. The results of magnetic tests showed that iron grade of D1, D2 and D3 concentrates increased to 70.46, 63.98 and 45.37%, respectively. The optimization of blending was investigated for production of accumulated concentrate with desirable iron grade (68%) using MATLAB software.

Keywords: Jalal Abad iron mine, Characterization, Davis tube, Magnetic separation

Charakterystyka i określenie stopnia uwalniania to pierwsze etapy w procesie oczyszczania rud. W bloku 4 kopalni rud żelaza Jalal Abad w prowincji Kerman w Iranie, znajdują się trzy rodzaje złóż rud żelaza: D1, D2, D3, różnej klasy. W pracy tej przedstawiono wyniki analizy chemicznej, mineralogicznej, stopnia uwalniania oraz separacji magnetycznej wykonanych przy zastosowaniu metod XRF, dyfrakcji (XRD) oraz metod mikroskopowych i rurki Davisa. Wyniki analiz wskazują, że złoża D1, D2, D3 to złoża w klasie o zawartości odpowiednio 58, 52 i 38%. W próbkach określono zawartość magnetytu, hematytu, dolomitu, kalcytu oraz kwarcu. Średni poziom uwalniania określony przy pomocy metod mikroskopowych oszacowano na 500 μm. Wyniki badań magnetycznych wskazują, że zawartości żelaza w koncentracie D1,D2 i D3 wzrosły odpowiednio do 70.46, 63.98 i 45.37%. Przeprowadzono optymalizację mieszania koncentratów w celu produkcji końcowego koncentratu o pożądanym poziomie zawartości żelaza (68%) przy zastosowaniu oprogramowania MATLAB.

Słowa kluczowe: kopalnia rud żelaza Jalal Abad, charakterystyka, rurka Davisa, separacja magnetyczna

*   METALLURGY AND PETROLEUM FACULTY OF AMIR KABIR UNIVERSITY OF TECHNOLOGY, TEHRAN, IRAN
**   ENGINEERING FACULTY OF TARBIAT MODARES UNIVERSITY, TEHRAN, IRAN
*** FACULTY OF ENGINEERING, UNIVERSITYOF BIRJAND, BIRJAND, IRAN

1. Introduction

Since long ago, man has experienced the natural magnetic properties of some objects, but in reality, it was Faraday's discoveries and investigations in 1845 that developed magnetic separation. The concept of magnetic separation is based on the ability to magnetize a particular mineral and then physically collect it. There has been significant advancement in the design and application of magnetic separators in the recent decades (Lewis & Horst, 1985; Alanyah et al., 2006).

Magnetic separation is usually used to separate magnetic compounds such as magnetite and hematite from attendant tailings in Rougher, Cleaner and Scavenger stages. Drum separators are the most common type of magnetic separators to separate ferromagnetic minerals in both dry and wet magnetic separation methods (Norrgan & Mankaso, 2002; Rayner & Napier-Munn, 2003).

In a magnetic separator, several competing forces such as gravity force, inertial force, hydrodynamic drag and surface and inter-particle forces are acting on the particles. As shown schematically in Fig. 1, these forces attempt to remove particles from the surface of drum.

Feed

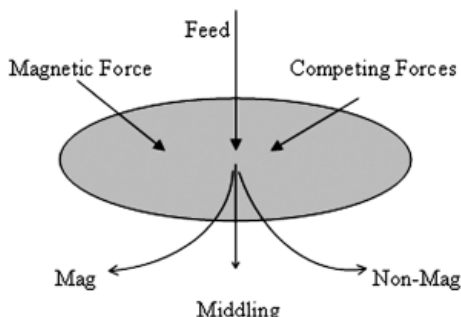Magnetic Force   Competing Forces

Mag   Non-Mag

Middling

Fig. 1. Schematic diagram of the process of magnetic separation

When a magnetable particle is placed in a non-homogeneous magnetic field, it is acted upon by the magnetic force [N] given by Eq. 1:

$$\vec{F}_m = \frac{k}{\mu_0} V B \vec{\nabla} B \tag{1}$$

where, k is the volumetric magnetic susceptibility of the particle, μ_0 is the magnetic permeability of vacuum [$4\pi * 10^{-7}$H/m], V is the volume of the particle [m^3], B is the external magnetic induction [T] and $\vec{\nabla} B$ is the gradient of magnetic induction [T/m]. Thus magnetic force is proportional to the product of both the external magnetic field and the field gradient, and has the direction of the gradient. In a homogeneous magnetic field, in which $\vec{\nabla} B = 0$, the force on a particle is zero.

The force of gravity [N] can be written as:

$$\vec{F}_g = \rho V \vec{g} \tag{2}$$

where, ρ is the particle density [kg/m^3] and g is the gravity acceleration [m/s^2]. The hydrodynamic drag [N] is given by:

$$\vec{F}_d = 6\pi b \vec{v}_p \tag{3}$$

where, η is the dynamic viscosity of the fluid [kg/m·s], b is the particle radius [m] and v_p is the relative velocity of the particle with respect to the fluid [m/s]. Magnetic particles will be separated from "nonmagnetic" (or more magnetic particles will be separated from less magnetic particles), if the following conditions are met:

$$\vec{F}_m^{\,mag} > \Sigma \vec{F}_c^{\,mag} \text{ and } \vec{F}_m^{\,non\text{-}mag} < \Sigma \vec{F}_c^{\,non\text{-}mag} \tag{4}$$

where, F_c is a competing force [N] while F^{mag} and $F^{non\text{-}mag}$ are forces acting on the magnetic and "non magnetic" particles [N], respectively. To achieve high recovery of magnetic particles, the magnetic separating force must be greater than the sum of the competing forces, as shown in Eq. 4.
    If:

$$\vec{F}_{mag} > \vec{F}_{comp} \tag{5}$$

selectivity of separation will be poor, as no distinction will be made between various magnetizable particles (Svoboda, 2004). The selectivity is critically determined by the relative values of the magnetic and competing forces, which are affected by a correct choice of a separator and operating conditions (Svoboda & Fujita, 2003; Brozek, 2002; Ohara et al., 2001; Rayner & Napier-Munn, 2000).

Drum separator is a simple form of magnetite separators. It consists of a rotary nonmagnetic drum, containing 3-6 stationary magnets of alternating polarity. These drums are in the form of electromagnet and permanent magnet (Vijayendra, 1995). The material is fed at the top of the rotating drum surface; the nonmagnetic fraction leaves the drum while the magnetic fraction is retained by the magnetic force at the drum surface until arrives outside (Svoboda, 1994, 1998, 2000). Dry low intensity magnetic separation is confined mainly to the concentration of coarse sands, which are strongly magnetic. This process is known as cobbing (Wills, 1985).

The desired separation can be achieved by controlling such parameters as particle size, magnetic field intensity, non-magnetic surface distance from the magnet, the radius of drum, the rotation rate of drum, initial load discharge, spatial angle and the distance between the poles (Rezaei, 1989).

Jalal Abad iron mine, located 45 km far from the Zarand city in Kerman Province is among the most important iron ore mines in Iran. Its iron ore deposit is nearly 2×10^{11} Kg and it More than 50% of valuable reserve is in blocks 1 and 2, and other blocks have no economic importance. Block 4 has three iron ore bulks, namely D1, D2 and D3, with the average iron grade of 58, 52 and 38%, respectively (Hatch Company, 2004).

In this study, characterization, mineralogical and liberation degree studies along with Davis tube tests of block 4 iron ore were performed. Furthermore, concentration and blending optimization for a variety of low and high-grade iron ores of this block was evaluated using magnetic separation method in order to achieve an accumulated concentrate with suitable iron grade (68%) for the production of steel by a direct reduction process.

2. Experimental method

2.1. Materials and equipment

The mineral was comminuted using laboratory jaw and drum crushers. For magnetic tests, Davis tube and a low intensity dry laboratory separator (Field strength: 0.12 T, Drum diameter: 0.250 m) were used. The samples were analyzed by chemical method. An initial 125 kg iron ore sample was prepared from the block 4 of Jalal Abad mine.

2.2. Characterization studies

2.2.1. Sample preparation

The aim of sampling is preparing a small amount of mineral bulk that represents the mass (Nematallahy, 1986). For concentration and characterization studies, 104 Kg of the extracted ore types (D1, D2 and D3) were prepared. The samples were approximately divided into +0.6, 0.25-0.6, 0.1-0.25 and −0.1 m fractions and 1000 Kg sample was selected from different parts by the following method (Fig. 2).

2.2.2. Mineralogical studies and determining of the ore liberation degree

a) Chemical analysis

XRF analysis was conducted for determination of the elements in the ore sample (Mor & Modberi, 1990). For this purpose, 0.2 Kg of -10^{-3} m fraction (Fig. 2) was pulverized to -150 μm (100 meshes) and prepared for XRF analysis. Ideal iron ore concentrate has 0.7% sulfur and 0.05% phosphorus. As shown in Table 1, the percentage of sulfur and phosphorus in the samples is desirable, so there was no need for desulfurization and dephosphorization.

TABLE 1

Chemical composition of the D1, D2 and D3 samples (%)

Sample	Fe	FeO	P	S	SiO_2	MgO	Al_2O_3	CaO	L.O.I
D1	58.54	7.38	0.046	0.074	4.65	0.82	0.21	1.86	26.42
D2	52.08	5.07	0.035	0.071	6.54	0.73	0.26	3.56	31.65
D3	38.29	3.08	0.039	0.057	8.61	3.46	0.52	7.33	38.61

b) XRD analysis

XRD analysis was used to identify the minerals and present phases in the samples. According to the XRD analysis, Magnetite (Fe_3O_4), Hematite (Fe_2O3), Goethite (FeOOH), Dolomite ($CaMg(CO_3)_2$), Calcite ($CaCO_3$) and Quartz (SiO_2) were identified. Some other minerals (sulfur and phosphorous containing minerals) were not detected by the XRD analysis due to very low content.
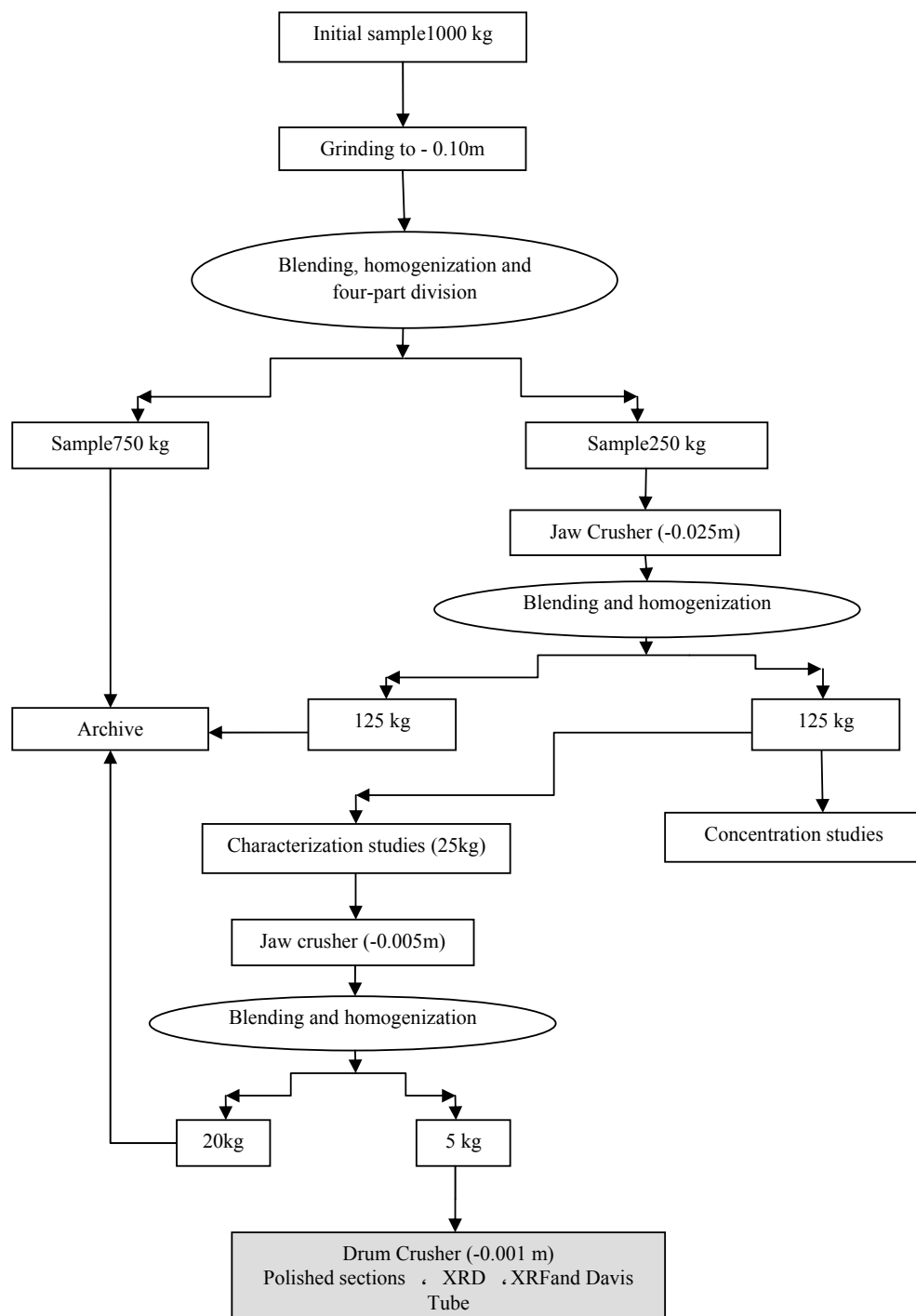
Fig. 2. Representative sample preparation method

c) Microscopic studies

Comprehensive mineralogical studies were performed for determination of the ore's liberation degree. For this purpose, 0.01Kg representative sample from each fraction (850-1000, 600-850 and 425-600 µm) was used to make 12 polished sections. The results indicated that the major iron containing minerals were Magnetite, Hematite, Martite and Goethite; and sulfur containing minerals were Pyrite and Chalcopyrite but in small amounts. The magnetic minerals, which in ordinary light under the microscope included:

- *Magnetite:* The reflected light intensity is moderate to low and creamy white to gray in color, isotropic and non polychromic. It is usually condensed granularand is converted into secondary minerals such as Hematite and Martite.
- *Hematite and Martite:* They are formed by weathering and alteration of Magnetite. First, Magnetite is converted into Hematite, and the latter is converted into Martite. In these crystals, the intensity of reflected light is medium to high and the color is white to gray. Thay are low anisotropic and too weak polychromic in some crystals. If the Hematite particles are small and fine, internal reflection in the crystals is very sharp and red to brown in color. These crystals are usually seen on the magnetite particles along the surface of fractures and cleavages.
- *Goethite:* In these crystals, the intensity of reflected light is moderate to low and white to gray in color. There is usually reflective polychroism.

The results of microscopic study of the D1, D2 and D3 samples are as follows:

- *D1 sample:* It mainly contains Magnetite crystals and very small amount of Hematite and Martite in the margin of Magnetite crystals. Amorphic structure of Hematite crystals confirms its secondary nature. Sulfide minerals and Goethite crystals were scarcely seen in some sections. The results also indicated that in 850-1000 and 600-850 µm fractions, iron containing ores are found in intergrowth with Gong, but in 425-600 µm fractions, more than 80% of these minerals are released.
- *D2 sample:* It was found that the ore consists of Magnetite, which is altered into Martite slightly. The martitization process has largely formed on the sidelines of the grains, in fractures and on the surface of cleavages. Goethite wasalso found in the D2 sample to some extent. Alteration in D2 is more than in D1.The results also indicated that in 850-1000 and 600-850 µm fractions, iron ores are found in intergrowth with gangue. Also more than 80% of these minerals are released in 425-600 µm fractions.
- *D3 sample:* The study of microscopic sections of D3 sample showed that it also has Magnetite minerals, which have almost changed into Martite. Thus, D3 sample is altered more than the D1 and D2 samples. Goethite is also abundant in this sample. More than 80% of iron minerals are released in 425-600 µm fraction. Microscopic images of minerals are illustrated in Figs. 3-14.
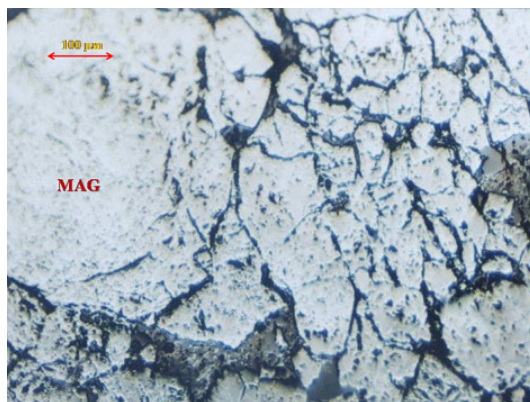
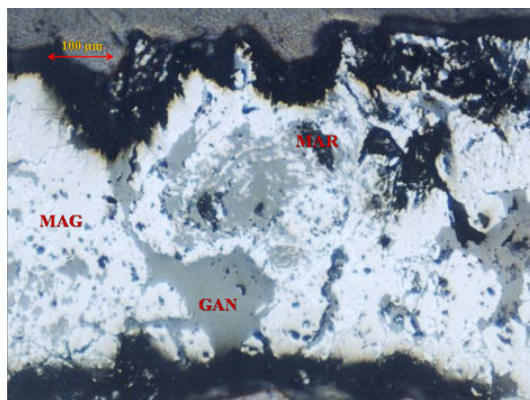Fig. 3. The surface of completely Magnetic sample – Section of the D1 sample (LN * 100)

Fig. 4. Magnetite coexistent with Martite and gangue – 850-1000 μm fraction of the D1 sample (LN * 100)
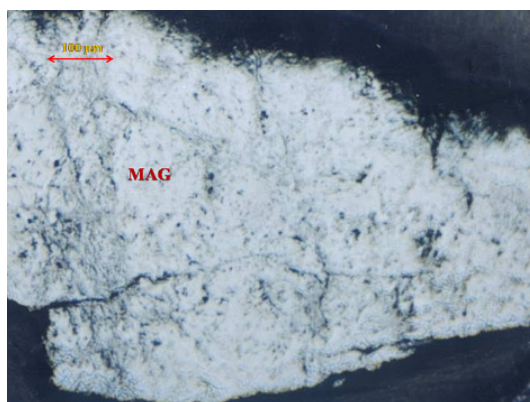

Fig. 5. Completely released Magnetite grain – 425-600 μm fraction of the D1 sample (LN * 100)
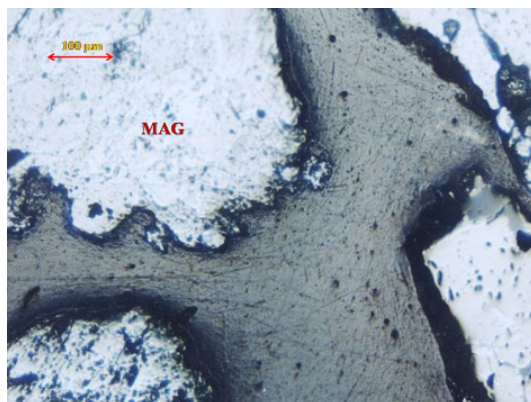
736

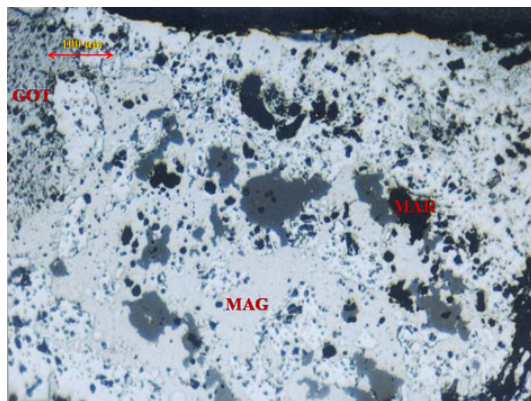Fig. 6. Partially released Magnetite grain – 425-600 µm fraction of the D1 sample (LN * 100)

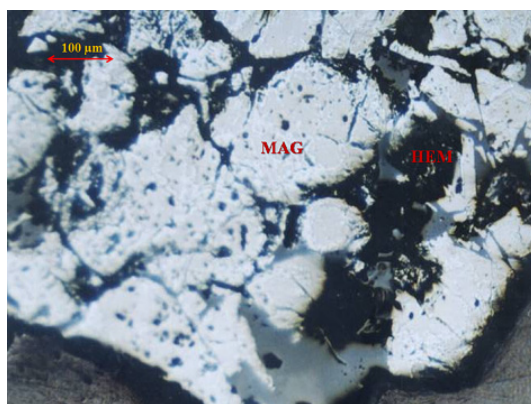Fig. 7. Magnetite associated with Goethite and Martite – Section of the D2 sample (LN * 100)

Fig. 8. Magnetite associated with Hematite and gangue – 850-1000 µm fraction of the D2 sample (LN * 100)
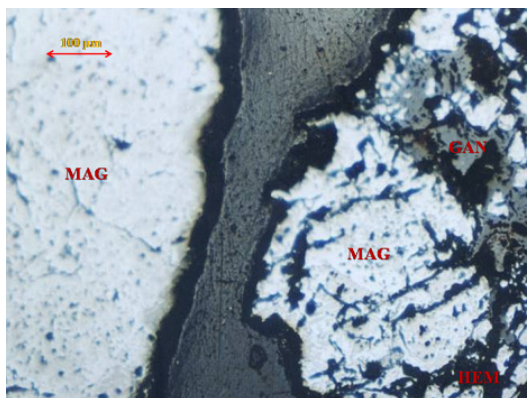
Fig. 9. Magnetite involved with Hematite and gangue (right) and free Magnetite (left) – 600-850 µm fraction of the D2 sample (LN * 100)
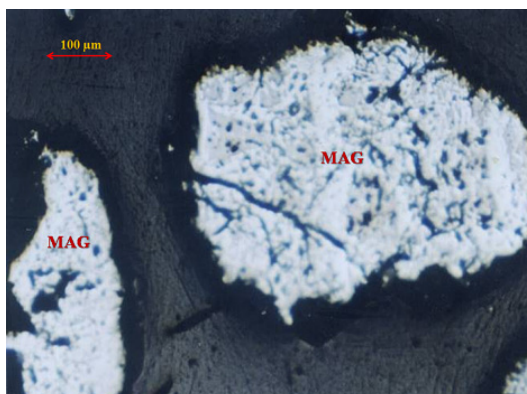
Fig. 10. Mainly released Magnetite grains – 425-600 µm fraction of the D2 sample (LN * 100)
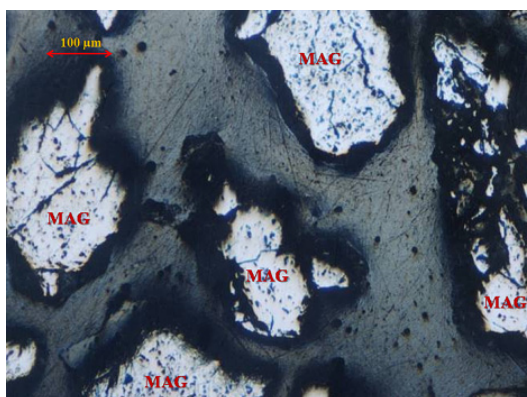
Fig. 11. Mainly released Magnetite grains – 425-600 µm fraction of the D3 sample (LN * 100)

Fig. 12. Coarse Magnetite grains surrounded by Goethite – 600-850 µm fraction of the D3 sample (LN * 100)

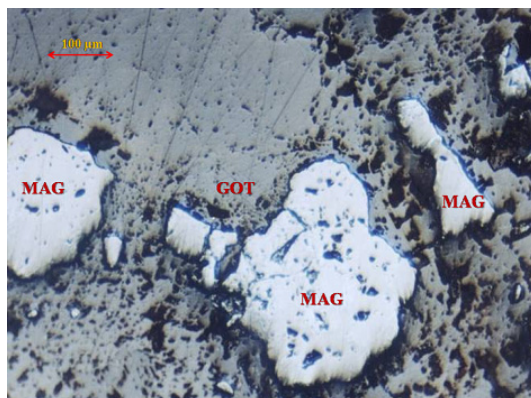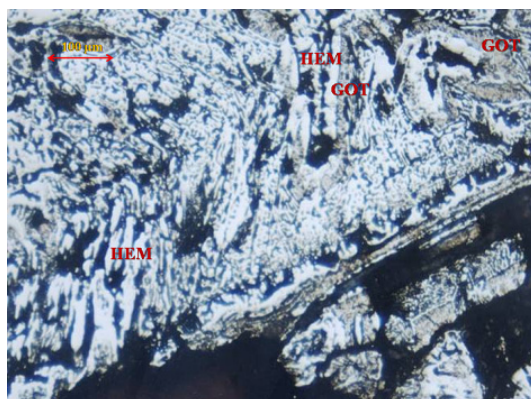Fig. 13. Hematite blades inside Goethite – 600-850 µm fraction of the D3 sample (LN * 100)
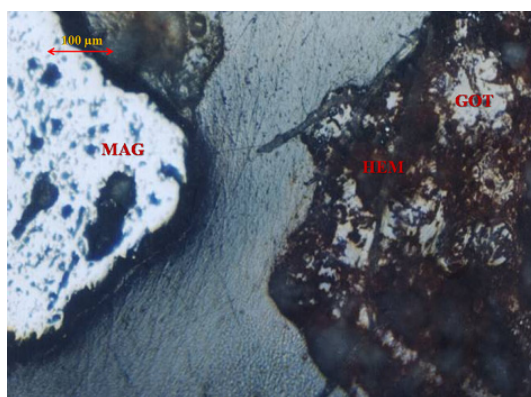

Fig. 14. Blending of Goethite and fine grain Hematite with internal reflection
in brown color (right) – 425-600 µm fraction of the D3 sample (LN * 100)

2.3. Davis tube tests

Davis tube studies were carried out to confirm the degree of liberation results. For this purpose, some samples of –10-3 m fraction (Fig. 2) were ground up to –1000, –850, –600 and –425 μm and Davis tube tests were performed on the typical samples of each fraction. Feed, concentrate and tailings were chemically analyzed. The results, given in Table 2-4, showed that the recovery increases in smaller particles. Therefore, the liberation degree of particles increases in the finer fractions. Also grinding efficiency increased as particle size decreased from –1000 to –600 μm, but not in the finer fraction (425-600 μm). Therefore, 80% of particles were released in the 425-600 μm fraction.

TABLE 2

Results of the Davis tube test of D1 sample

Fraction (μm)	Feed			Concentrate			Tailings			Recovery of iron	
	Weight (×10⁻³ Kg)	Fe (%)	FeO (%)	Weight (×10⁻³ Kg)	Fe (%)	FeO (%)	Weight (×10⁻³ Kg)	Fe (%)	FeO (%)	Weight (%)	The total (%)
-1000	10	58.39	7.18	6.07	66.88	9.98	3.93	45.22	2.91	60.70	69.53
-850	10	58.51	7.21	6.16	68.02	10.11	3.84	43.20	2.49	61.60	71.61
-600	10	58.32	7.12	6.30	70.29	10.12	3.70	38.14	1.95	63.00	75.93
-425	10	58.22	7.01	6.34	70.70	10.2	3.66	36.52	1.69	63.40	76.99

TABLE 3

Results of the Davis tube test of D2 sample

Fraction (μm)	Feed			Concentrate			Tailings			Recovery of iron	
	Weight (×10⁻³ Kg)	Fe (%)	FeO (%)	Weight (×10⁻³ Kg)	Fe (%)	FeO (%)	Weight (×10⁻³ Kg)	Fe (%)	FeO (%)	Weight (%)	The total (%)
-1000	10	52.46	5.02	5.09	59.34	7.97	4.91	45.21	1.99	50.90	57.58
-850	10	52.36	5.13	5.16	60.73	8.11	4.84	43.31	1.97	51.60	59.85
-600	10	52.85	4.89	5.38	62.25	8.08	4.62	41.80	1.07	53.80	63.37
-425	10	53.04	5.17	5.41	63.10	8.38	4.59	41.09	1.30	54.10	64.36

TABLE 4

Results of the Davis tube test of D3 sample

Fraction (μm)	Feed			Concentrate			Tailings			Recovery of iron	
	Weight (×10⁻³ Kg)	Fe (%)	FeO (%)	Weight (×10⁻³ Kg)	Fe (%)	FeO (%)	Weight (×10⁻³ Kg)	Fe (%)	FeO (%)	Weight (%)	The total (%)
-1000	10	38.27	3.24	4.12	43.07	5.87	5.88	34.86	1.44	41.20	46.37
-850	10	38.41	3.39	4.23	44.25	5.94	5.77	34.11	1.42	42.30	48.73
-600	10	38.23	3.08	4.14	45.90	5.66	5.59	32.26	1.11	44.10	52.95
-425	10	37.98	2.99	4.42	46.00	5.40	5.58	31.68	1.07	44.20	53.53

2.4. Dry magnetic tests

The typical sample ($\approx$ 30 kg) was prepared for concentration studies. Before magnetic separation experiments, the sample was crushed up to 1000 μm and divided into three parts. To study the effect of grinding on recovery, one part of the sample was ground till –850 μm and the other part was ground up to –600 μm using ball mill. Therefore, the dry magnetic tests were performed on three fractions of –1000, –850 and –600 μm using a low intensity drum separator. The main parameters affecting these parathion efficiency are magnetic field intensity, drum diameter, drum rotation speed, particle size and feed rate (Rezaei, 1989).

In this research, feeding rate and drum diameter were considered as constant and drum rotation speed and particle size range were considered as variable data. The separating field intensity, drum diameter and feeding rate were adjusted at 0.12T, 0.25 m and 8.3×10^{-3} Kg/s, respectively.

Magnetic experiments were done in two stages; rougher and cleaner (Fig. 15). First, the samples were fed into the rougher separator and divided into two parts; concentrate and tailing.
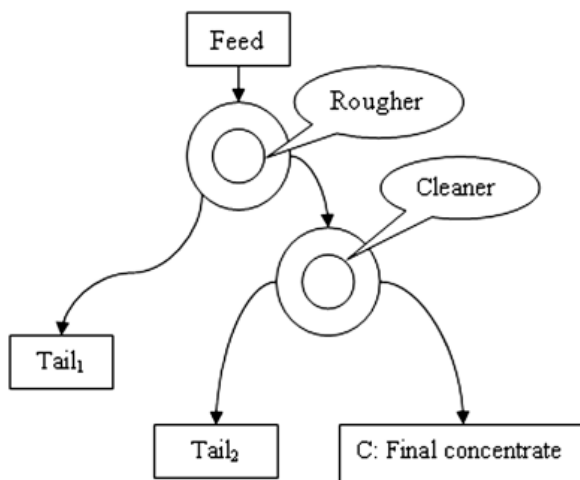

Fig. 15. The flowsheet of process

Rougher concentrate was fed into the cleaner (with a higher speed). Finally, cleaner concentrate was considered as final concentrate, and final tailing contain both rougher and cleaner tailing. The drum speed was set at 1 Hz and 1.33 Hz in the rougher and cleaner separators, respectively. Some samples from concentrate and tailing of the rougher and cleaner were chemically analyzed. The results of D1, D2 and D3 samples are given in Tables 5-7.

The results are summarized as follows:

In D1 sample, iron concentration at cleaner stage increased to 68.75% (>68%) in the –800 μm fraction. By approaching the degree of liberation at smaller particle sizes (–600 μm fractions), iron concentration was also raised to 70.46%.

In D2 sample, the iron grade at cleaner stage increased to 63.98% in the –600 μm fraction. However, it just reached to 49.87% in D3 sample, which is far from the required grade (68%).

Therefore, the iron grade of D1 and D2 samples should be blended into special ratio for steel producing by direct reduction method.

TABLE 5

The results of dry magnetic separation of the rougher and cleaner stages of D1 sample

Fraction (µm)	Feed			Cleaner concentrate			Rougher Tailings			Cleaner Tailings			Recovery of iron	
	Weight (Kg)	Fe (%)	FeO (%)	Weight (Kg)	Fe (%)	FeO (%)	Weight (Kg)	Fe (%)	FeO (%)	Weight (Kg)	Fe (%)	FeO (%)	Weight (%)	The total (%)
−1000	2754	58.34	7.26	1343	67.30	10.84	1170	48.11	4.01	241	57.90	2.82	48.77	56.25
−850	2526	57.95	7.53	1284	68.75	11.30	1024	44.39	3.85	218	57.59	2.89	50.83	60.30
−600	2985	58.14	7.51	1610	70.46	11.10	1163	45.11	3.66	212	58.86	2.51	53.94	65.37

TABLE 6

Results of dry magnetic separation of the rougher and cleaner stages of D2 sample

Fraction (µm)	Feed			Cleaner concentrate			Rougher Tailings			Cleaner Tailings			Recovery of iron	
	Weight (Kg)	Fe (%)	FeO (%)	Weight (Kg)	Fe (%)	FeO (%)	Weight (Kg)	Fe (%)	FeO (%)	Weight (Kg)	Fe (%)	FeO (%)	Weight (%)	The total (%)
−1000	2.628	51.86	5.12	1.055	59.97	8.66	1.293	44.65	2.69	0.280	53.82	2.75	40.14	46.42
−850	2.874	52.13	5.44	1.226	61.76	8.82	1.378	42.71	2.91	0.270	56.39	2.73	42.66	50.54
−600	2.814	52.30	5.20	1.281	63.98	8.38	1.297	40.88	2.74	0.236	52.02	2.08	45.52	55.69

TABLE 7

Results of dry magnetic separation of the rougher and cleaner stages of D3 sample

Fraction (µm)	Feed			Cleaner concentrate			Rougher Tailings			Cleaner Tailings			Recovery of iron	
	Weight (Kg)	Fe (%)	FeO (%)	Weight (Kg)	Fe (%)	FeO (%)	Weight (Kg)	Fe (%)	FeO (%)	Weight (Kg)	Fe (%)	FeO (%)	Weight (%)	The total (%)
−1000	2.549	38.25	3.22	0.766	41.60	6.75	1.476	36.33	1.48	0.307	39.71	2.91	30.05	32.68
−850	2.768	38.32	3.18	0.908	43.73	6.60	1.554	35.18	1.35	0.306	39.05	2.51	32.80	37.43
−600	2.894	38.54	3.16	1.025	45.37	6.15	1.574	34.01	1.62	0.295	38.93	1.76	35.42	41.69

2.5. Blending of samples

Magnetic separation results showed that the iron concentration of D1, D2 and D3 samples in cleaner stage (−600 µm fraction) increased to 70.46%, 63.91% and 45.37%, respectively. Therefore, the iron grade of sample D2 is less than desirable and it should be blended with D1 concentrate by a certain weight ratio. The composition of D1 and D2 samples (to achieve the optimal iron grade of 68%) is calculated by Eq. 6:

$$aA + bB = (A + B)68 \leftrightarrow \frac{B}{A} = \frac{a - 68}{68 - b} \tag{6}$$

where, a is iron grade [%], A is the concentrate weight of D1 sample [kg], b is iron grade [%], B is the concentrate weight of D2 sample [kg] and B/A is the weight ratio of concentrate D2 to D1. For example, this value is about 0.61 in −600 µm fraction for different samples.

2.6. Optimization of samples blending

Regarding the variable concentrate grade and favorable grade, it is necessary to calculate the weight ratio of concentrate blending in different conditions. For this purpose, MATLAB software was used to optimize the blending of samples with maximum concentration recovery. The main equation (Eq. 7) and the objective function (Eq. 8) are defined as follows:

Equation:
$$aA + bB + cC = (A + B + C)g \qquad (7)$$

Objective Function:
$$Max(A + B + C) \qquad (8)$$

where, a, b and c are the iron grade of D1, D2 and D3 concentrates [%], respectively, A, B and C represent the corresponding weight [kg] for these samples and G is desirable iron grade for the accumulated concentrate [%].

The program gives weight ratio of the concentrate blending of D1, D2 and D3 samples by asking some preliminary questions. The program's code in MATLAB software is given in Appendix. Some of these typical questions are given below:

1. Which sample (D1, D2 or D3) is going to be combined?
2. What is the grade of D1, D2 and D3 concentrates?
3. What is the optimum iron grade for the accumulated concentrate?

3. Results

In this research, first a representative sample from the block 4 of Jalal Abad iron mine in Kerman province, Iran was prepared, grinded, homogenized and divided using conventional methods. Therefore, a representative sample was prepared for characterization and concentration studies. The XRF analysis indicated the iron grade of D1, D2 and D3 samples as 58%, 52% and 32%, respectively. Also the analytical results showed that sulfur and phosphorus grades in the samples were less than optimum in the iron concentrates, and so desulfurization and dephosphorylation of the concentrates would not be necessary. The XRD results confirmed that the dominant minerals are Magnetite, Hematite, Goethite, Dolomite, Calcite and Quartz. Microscopic studies showed that the majority of iron containing ores (about 80%) is released in 425-600 µm fraction. Therefore the average degree of liberation for the ore can be regarded as 500 µm.

Davis tube tests carried out on the different fractions (−1000, −850, −600 and −425 µm). The results confirmed the liberation degrees obtains in −600 to +425 µm fraction .

The iron grade of D1, D2 and D3 concentrates in −600 µm fraction was increased at cleaner stage by 70.46%, 63.98% and 45.37%, respectively. Thus, the concentrate grade of D1 sample reached to the desired grade at the steel producing industry, while the D2 and D3 concentrates grade was less than 68%. Therefore, they should be blended with a certain weight ratio. A program in MATLAB software was encoded giving the optimum concentrate blending ratio in different conditions.

4. Acknowledgements

Reference

Alanyali H., Col M., Yilmaz M., Karagoz S., 2006. *Application of magnetic separation to steelmaking slags for reclamation*. Waste Management, vol. 26, no. 10, p. 1133-1139.

Brozek M., 2002. *The distribution of magnetic fields and an analysis of the separation in process in a spiral separater*. Arch. Min. Sci., Vol. 47, No 2, p. .

Hatch Company, 2004. *Jalal-Abad iron oremine prefeasibility stud.report.canada.*

Lewis F.M., Horst W.E., 1985. *SME mineral processing handbook*. vol. 2, Society of Mining Engineering Publishing Co, N.L. Weiss, New York.

Mor F., Modberi S., 1990. *Mineralogy handbook*. vol. 1, ed. 1, Tehran (in Persian).

Nematallahy H., 1986. *Ore dressing*. vol. 2, ed. 2, Tehran University, Tehran (in Persian).

Norrgan D.A., Mankaso M.J., 2002. *Selection and sizing of magnetic concentrating equipment; plant design/ practice Andprocessing*. vol. 1, SME-AIME, New York.

Ohara T.,Kumakura H., Wada H., 2001. *Magnetic separation using superconducting magnets*. Physica C: Superconductivity, vol. 357-360, p. 1272-1280.

Rayner J.G., Napier-Munn T.J., 2000.*The mechanism of magnetics capture in the wet drum magnetic separator*. Minerals Engineering, vol. 13, no. 3, p. 277-285.

Rayner J.G., Napier-Munn T.J., 2003. *A mathematical model of recovery of dense medium magnetic in the wet drum magnetic separator*. International Journal of Mineral Processing, vol. 69, no. 1-4, p. 157-173.

Rezaei B., 1989. *Mineral processing technology*. vol. 1, Amirkabir, Tehran (in Persian).

Svoboda J., 1987. *Magnetic methods for the treatment of minerals*. Elsevier, Amsterdam.

Svoboda J., 1994. *The effect of the magnetic field strength on the efficiency of magnetic separation*. Mineral Engineering, vol. 7, p. 747.

Svoboda J., 2000. *Separation in magnetic fluids: time to meet technological needs*. In: Proc. MINPREX 2000 Congress, Melbourne, Australia, p. 297.

Svoboda J., 2001. *Magnetic methods of material treatment: technology at crossroads*. In: Proc. 4th Int. Conf. Mater. Eng. Res. ICMR 2001, Akita, Japan, p. 131.

Svoboda J., 2004. *Magnetic techniques for the treatment of materials*. Kluwer Academic Publisher, Netherland.

Svoboda J., Fujita T., 2003. *Recent developments in magnetic methods of material separation*. Minerals Engineering, vol. 16, p. 785-792.

Vijayendra G.H., 1995. *A handbook on mineral dressing*. Vikus Publishing House PVT.LTD.

Wills B.A., 1985. *Mineral processing technology*. Pergamon Press.

744

APPENDIX

In order to determine the optimum weight ratio of sample blending, the encoded program in MATLAB software was used as follows:

```
clear
kk=input ('if D1,D2& D3 are blending press 1, elseif D1 & D2 are blending press 2, if D1 &
D3 are blending press 3');
ifkk==1
    g=input('what is the suitable grade of iron for bulk concentrate? ');
    a=input('input iron grade of D1 concentrate? ');
    b=input('input iron grade of D2 concentrate? ');
    c=input('input iron grade of D3 concentrate? ');
    A=1;
    Amax=0;
Bmax=0;
Cmax=0;
    kmax2=0;
for B=0:.0001:1
for C=0:.0001:1
if (g-a)+(g-b)*B+(g-c)*C >-0.001 & (g-a)+(g-b)*B+(g-c)*C < 0.001
        kmax1=A+B+C;
if kmax1 > kmax2
        kmax2=kmax1;
        Amax=A;
Bmax=B;
Cmax=C;
end
end
end
end
    weight_of_D1=Amax
    weight_of_D2=Bmax
    weight_of_D3=Cmax
end
ifkk==2
    g=input('what is the suitable grade of iron for bulk concentrate? ');
    a=input('input iron grade of D1 concentrate? ');
    b=input('input iron grade of D2 concentrate? ');
    A=1;
    C=0;
    Amax=0;
Bmax=0;
Cmax=0;
    kmax2=0;
```

```
for B=0:.0001:2
if (g-a)+(g-b)*B >-0.001 & (g-a)+(g-b)*B < 0.001
          kmax1=A+B+C;
if kmax1 > kmax2
          kmax2=kmax1;
          Amax=A;
Bmax=B;
Cmax=C;
end
end
end
     weight_of_D1=Amax
     weight_of_D2=Bmax
end
ifkk==3
     g=input('what is the suitable grade of iron for bulk concentrate? ');
     a=input('input iron grade of D1 concentrate? ');
     c=input('input iron grade of D3 concentrate? ');
     A=1;
     B=0;
     Amax=0;
Bmax=0;
Cmax=0;
     kmax2=0;
for C=0:.0001:2
if (g-a)+(g-c)*C >-0.001 & (g-a)+(g-c)*C < 0.001
          kmax1=A+B+C;
if kmax1 > kmax2
          kmax2=kmax1;
          Amax=A;
Bmax=B;
Cmax=C;
end
end
end
     weight_of_D1=Amax
     weight_of_D3=Cmax
end
```