Recovery of iron from cyanide tailings with reduction roasting–water leaching followed by magnetic separation☆

Yali Zhang, Huaimei Li, Xianjin Yu*

College of Chemical Engineering, Shandong University of Technology, 255081, Zibo, Shandong, China

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ABSTRACT

Cyanide tailing is a kind of solid waste produced in the process of gold extraction from gold ore. In this paper, recovery of iron from cyanide tailings was studied with reduction roasting–water leaching process followed by magnetic separation. After analysis of chemical composition and crystalline phase, the effects of different parameters on recovery of iron were chiefly introduced. Systematic studies indicate that the high recovery rate and grade of magnetic concentrate of iron can be achieved under the following conditions: weight ratios of cyanide tailings/activated carbon/sodium carbonate/sodium sulfate, 100:10:3:10; temperature, 50 °C; time, 60 min at the reduction roasting stage; the liquid to solid ratio is 15:1 (ml/g), leaching at 60 °C for 5 min and stirring speed at 20 r/min at water-leaching; exciting current is 2 A at magnetic separation. The iron grade of magnetic concentrate was 59.11% and the recovery ratio was 75.12%. The mineralography of cyanide tailings, roasted product, water-leached sample, magnetic concentrate and magnetic tailings were studied by X-ray powder diffraction (XRD) technique. The microstructures of above products except magnetic tailings were also analyzed by scanning electron microscope (SEM) and energy disperse spectroscopy (EDS) to help understand the mechanism.

1. Introduction

With the increasing demand both in China and abroad, Chinese gold industry has developed rapidly. The total production of gold in China added up to 340.88 tons in 2010 which had increased 8.57% compared with that of 2009. China ranks the first place among gold production countries in the world. Meanwhile, a construction increase of hazardous industrial wastes like cyanide tailings generated by gold industry is becoming a serious problem. In China, according to incomplete statistics, about more than 20 million tons of cyanide tailings are produced each year [1–3]. These residues occupy precious land, and the construction and maintenance of cyanide tailings disposal sites also increase the production cost of gold-making plants. In addition, cyanide tailings usually have high content of hazardous materials, which could cause serious environmental pollution. At the same time, there are a lot of valuable metals like gold and iron, and in our research, we took a reasonable and effective method to recover these iron and gold existing in the cyanide tailings for getting more benefit. But in this paper, we only introduced the relative contents of iron recovery. Iron content of cyanide tailings is high, and the processing of cyanide tailings as a

secondary resource rather than waste has become a more attractive route for sustainable production on account of the depletion of iron ores and rapid rise of metal prices [4–9]. The concerns over cyanide tailings are not exclusively environmental, but economic as well.

Although total iron content in cyanide tailings is relatively high, its recovery has not been practiced due to high content of impurities oxides such as SiO_2 and Al_2O_3 and the complexity of the chemical and mineralogical compositions of the cyanide tailings. To date, many researchers have conducted numerous studies on iron recovery. One such example is the use of magnetizing roasting–magnetic separation technique to recover iron from red mud [10–12], and iron ore tailings [3]. In other studies, sponge iron or reduced iron is produced from pyrite cinder [13], red mud [12,14,15], oily hot rolling mill sludge [16] and copper matte smelting slag [17] with a direct reduction-magnetic separation technique. It has also been reported that traditional mineral processing techniques being used for recovering iron from ore tailings and smelting slag [18], nickel metallurgical slag [19], blast furnace gas ash [20] and copper converter slag [21]. Magnetizing roasting or direct reduction followed by magnetic separation has been demonstrated to be effective for iron recovery from solid wastes [3,11–17].

In our group, the comprehensive utilization of cyanide tailings is the focus of our research. Based on the above descriptions of different ways of utilization of cyanide tailings, the grade of magnetic concentrate and the recovery rate of iron are lower, because of the massive existence of impurities minerals, so our innovative

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Table 1
Chemical composition of cyanide tailings.

Au[a]	Ag[a]	T Fe	Fe(Fe$_2$O$_3$)[b]	SiO$_2$	Al$_2$O$_3$	B$_2$O$_3$	SO$_3$
5.625	49.94	27.69	27.1	23.9	6.35	3.96	5.37
CO$_2$	CaO	Na$_2$O	K$_2$O	MgO	PbO	TiO$_2$	ZnO
23.5	2.61	1.79	1.41	0.848	0.513	0.531	0.510

[a] The content unit was g/ton.
[b] The iron content of Fe$_2$O$_3$.

idea is combining the improving of the grade of magnetic concentrate with the reducing of the content of Si and Al together. In this study, recovery of iron from cyanide tailings with reduction roasting–water leaching followed by magnetic separation is investigated. The combination points are roasting, by which it can reduce hematite to magnetite, and water leaching, by which soluble substances can solve in the solution, then insoluble and nonmagnetic substances can enter into the magnetic tailings after magnetic separation. Process parameters that affect the recovery of iron are tested and optimized. The way of reduction roasting–water leaching followed by magnetic separation is demonstrated to be effective to recover iron from cyanide tailings for the first time. After the water leaching process, the nonmagnetic substances are used to recover another valuable metal of gold, and then the nonmagnetic residues can also be used to make building materials for creating more economic benefit, such as baking-free brick and cement which will be introduced in the later work.

2. Experimental

2.1. Raw materials

Cyanide tailings used in the study were the residual slags after extracting gold from gold-bearing ores using conventional cyanidation or roasting–cyanidation process. The main chemical composition is given in Table 1.

The total iron content was approximately 27.69%, but its content of SiO$_2$ and Al$_2$O$_3$ which will be detrimental to the iron recovery were relatively high in the tailings. X-ray power diffraction (XRD) pattern of cyanide tailings is shown in Fig. 1. According to the phase analysis, most of iron exists as hematite (Fe$_2$O$_3$), SiO$_2$ exists as quartz and Al$_2$O$_3$ exists as muscovite.

In order to eliminate the negative effect of carbon on iron recovery experiment, commercial analysis grade activated carbon was used as a reducing agent.
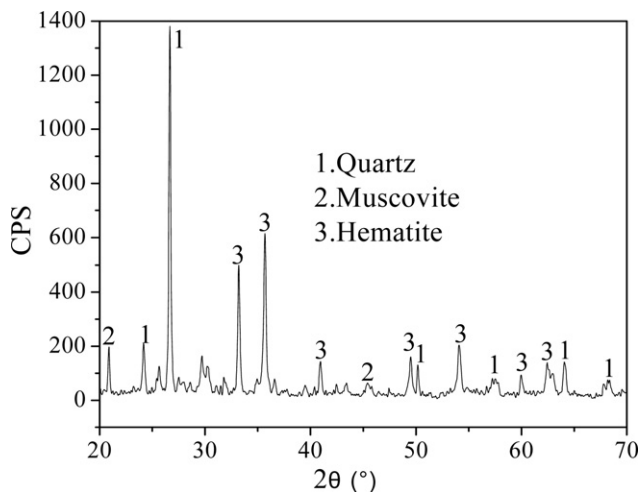

Fig. 1. XRD pattern of cyanide tailings.

2.2. Experimental method
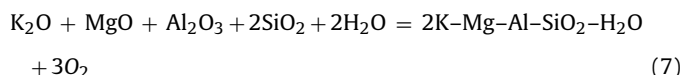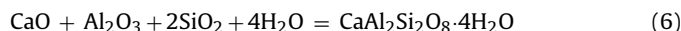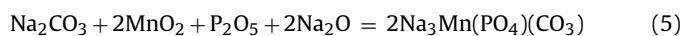
The raw material was screened to the required particle size ($\leq$74 μm), and then heated in dry oven. Original samples were mixed thoroughly with activated carbon and compound additives according to specific proportions, and after that, the mixture was put in a porcelain crucible with a sealed cover, next placed into the muffle furnace preheated to the required temperature. The charge was roasted under experimental conditions. After a given time, roasted samples were taken out rapidly and quenched with water quickly, then vaccum filtered, kiln dried. The dried samples were ground and put into glass beakers with water to form slurry with a certain liquid-solid ratio. The beaker was placed in an electrical heating oil-bath vessel equipped with a continuous agitating device, and leached with warm water at the required temperature and duration, and then the slurry was separated by buchcer funnel. The filter cake was dried and matched into about the concentrate of 30% slurry, and the slurry was separated by the low intensity magnetic separator (model: XCGS-φ50) with different intensity of working electrical current. The grade of magnetic concentrate was analyzed by chemical method, and the recovery rate of iron was deduced according to mass balance in magnetic separation process.

In our experiments, all experimental data were the averages of duplicate determinations. The relative errors of the data were about 3%.

2.3. Theoretical fundamental

#### 2.3.1. Reduction roasting process

Activated carbon can be used as the reductant, the reduction reaction of activated carbon was regarded as the combination of indirect reduction reaction (carbon monoxide was used as reductant) and Boudol reaction. The chemical reactions are as follows:

$$Fe_2O_3 + C = 2Fe_3O_4 + CO \qquad (1)$$

$$C + CO_2 = 2CO \qquad (2)$$

$$3Fe_2O_3 + CO = 2Fe_3O_4 + CO_2 \qquad (3)$$

$$MgO + Fe_2O_3 = MgFe_2O_4 \qquad (4)$$

$$Na_2CO_3 + 2MnO_2 + P_2O_5 + 2Na_2O = 2Na_3Mn(PO_4)(CO_3) \qquad (5)$$

$$CaO + Al_2O_3 + 2SiO_2 + 4H_2O = CaAl_2Si_2O_8 \cdot 4H_2O \qquad (6)$$

$$K_2O + MgO + Al_2O_3 + 2SiO_2 + 2H_2O = 2K–Mg–Al–SiO_2–H_2O$$
$$+ 3O_2 \qquad (7)$$

When the equilibrium partial pressure of carbon dioxide (CO$_2$) in Boudol reaction exceeds the partial pressure of CO$_2$ produced in reduction reaction of Fe$_2$O$_3$, carbon monoxide (CO) is produced through Boudol reaction with carbon and CO$_2$ in the system, and then CO reacts with Fe$_2$O$_3$. CO$_2$ is consumed and reductant CO is produced constantly by Boudol reaction, so the reaction may continue until activated carbon runs out or all of the Fe$_2$O$_3$ is reduced.

2.3.2. Water leaching process

The main chemical reactions occurred during the leaching process:

$$K–Mg–Al–SiO_2–H_2O(S) = K–Mg–Al–SiO_2–H_2O(L) \qquad (8)$$
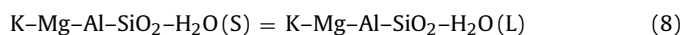
In the water leaching process, K–Mg–Al–SiO$_2$–H$_2$O is easy to dissolve into the water solution, causing the decrease of impurities content, and so the magnetite was beneficiation.
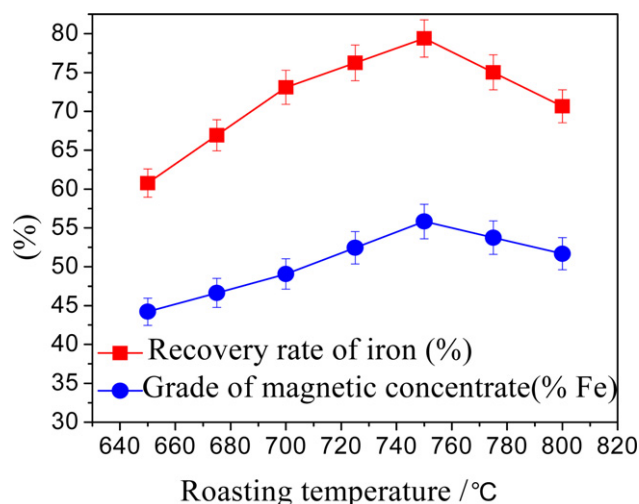
Fig. 2. Effects of roasting temperature on iron recovery.

Fig. 3. Effects of ratio of activated carbon to cyanide tailings on iron recovery.

3. Results and discussion

3.1. Iron recovery by reduction roasting–water leaching followed by magnetic separation

3.1.1. Effect of roasting temperature

The effect of roasting temperature on extraction of iron was investigated under the following conditions: cyanide tailings, activated carbon, sodium sulfate and sodium carbonate at a mixing ratio of 100:10:10:3, 60 min for roasting time, 60 °C for leaching temperature, 5 min for leaching time, 5:1 for the liquid-to-solid ratio, 20 r/min for stirring speed and 2 A for the exciting current. The results are shown in Fig. 2.

It can be seen from Fig. 2, both the grade of magnetic concentrate and the recovery rate of iron increase rapidly with the rising of roasting temperature from 650 °C to 750 °C, however, when the roasting temperature is over 750 °C, both the grade of magnetic concentrate and the recovery rate of iron tend to decrease. This is probably because that the speed of reaction would be much slow when the roasting temperature is too low, so the hematite cannot be transformed to magnetite completely; when the roasting temperature is too high, some hematite may be over-reduced to ferrous oxide which is weakly magnetic and difficult to be recycled. Moreover, it may cause the form of some agglomeration and the decrease of materials porosity at high temperature, so the activated carbon can hardly contact with the materials. When the temperature is 750 °C, the grade and recovery rate are both high. Hence, the optimum roasting temperature is determined to be 750 °C.

3.1.2. Effect of mass fraction of activated carbon

The effect of mass fraction of activated carbon on extraction of iron was investigated under the following conditions: cyanide tailings, sodium sulfate and sodium carbonate at a mixing ratio of 100:10:3, 750 °C for roasting temperature, 60 min for roasting time, 60 °C for leaching temperature, 5 min for leaching time, 5:1 for the liquid-to-solid ratio, 20 r/min for stirring speed and 2 A for the exciting current. The results are shown in Fig. 3.

It can be seen from Fig. 3 that while the mass fraction of activated carbon is below 10%, both the grade of magnetic concentrate and the recovery rate of iron increase with the increase of mass fraction of activated carbon. When the mass fraction of activated carbon keeps increasing, the two indexes tend to decrease rapidly. It may be due to that hematite cannot be reduced completely when the activated carbon is insufficient, but when the activated carbon is excessive, some hematite could be over-reduced to ferrous oxide
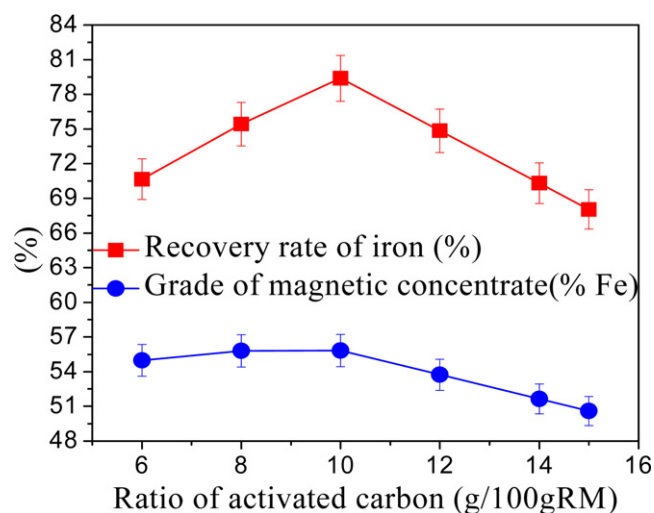
which is weakly magnetic. Hence, the optimal ratio of activated carbon was determined to be 10%.

3.1.3. Effect of roasting time

The effect of roasting time on extraction of iron was investigated under the following conditions: cyanide tailings, activated carbon, sodium sulfate and sodium carbonate at a mixing ratio of 100:10:10:3, 750 °C for roasting temperature, 60 °C for leaching temperature, 5 min for leaching time, 5:1 for the liquid-to-solid ratio, 20 r/min for stirring speed and 2 A for the exciting current. The results are shown in Fig. 4.

The results in Fig. 4 reveal that while the roasting time is in the range of 40–60 min, both the grade of magnetic concentrate and recovery rate of iron increased with the prolonging of duration. But when the roasting time is over 60 min, both the two indexes tend to decrease, especially the recovery rate of iron decreased quickly. This is perhaps because when the roasting time is too long, the excessive agglomerations were formed, and then easily prevented the carbon monoxide from spreading, so the hematite cannot be reduced effectively. Therefore, the best roasting time is 60 min.
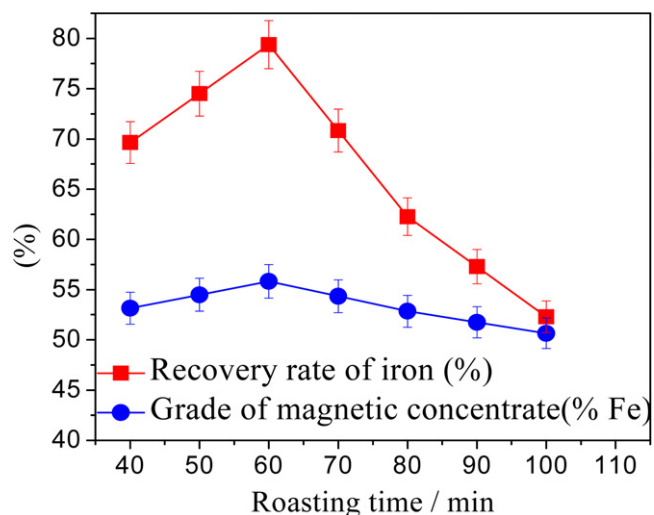

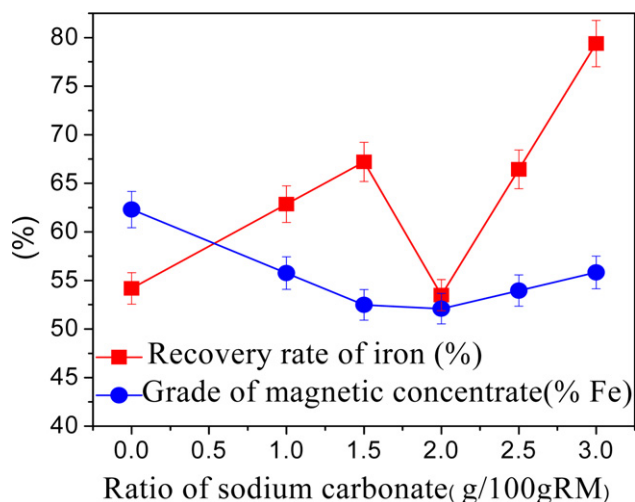**Fig. 4.** Effects of roasting time on iron recovery.

Fig. 5. Effects of ratio of sodium carbonate to cyanide tailings on iron recovery.

**Fig. 6.** Effects of leaching time on iron recovery.

3.1.4. Effect of ratio of sodium carbonate to cyanide tailings

The effect of ratio of sodium carbonate to cyanide tailings on extraction of iron was investigated under the following conditions: cyanide tailings, activated carbon, sodium sulfate at a mixing ratio of 100:10:10, 750 °C for roasting temperature, 60 min for roasting time, 60 °C for leaching temperature, 5 min for leaching time, 5:1 for the liquid-to-solid ratio, 20 r/min for stirring speed and 2 A for the exciting current. The results are shown in Fig. 5.

As shown in Fig. 5, there is non-linear relationship between each parameter of iron recovery and sodium carbonate ratio, especially at ratio of sodium carbonate (2.0 g/100 g RM), iron recovery appeared a obvious "wave" curve, it is because that the added sodium carbonate can change the mineral structure, and make it become more loose which is of great benefit for the reduction of hematite, with increase of the content of sodium carbonate, causing the increase of yield of magnetic concentrate. But when the ratio of sodium carbonate was about 2%, it can be occurred the following reaction:

$$Fe_2O_3 + Na_2CO_3 = Na_2O \cdot Fe_2O_3 + CO_2\uparrow \qquad (9)$$
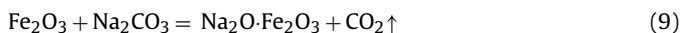
When the amount of sodium carbonate continues to increase, another reaction can occur as follows:

$$Na_2O \cdot Fe_2O_3 + CO_2 + Al_2O_3 = Na_2O \cdot Al_2O_3 + Fe_2O_3 \qquad (10)$$
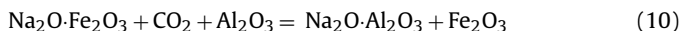
It means that when the amount of sodium carbonate continues to increase, more and more $Na_2O \cdot Fe_2O_3$ will be produced, and it will react with Al_2O_3, taking Fe_2O_3 out, so there will be more hematite can be reduced, that is to say, the yield of magnetic concentrate increases again, making the recovery rate of iron increases finally. With the recovery rate of iron achieving maximum when sodium carbonate ratio was about 3%, meanwhile, the grade of magnetic concentrate was about 55.82%, basically meeting the requirements of industrial production. In a word, it was inferred that the optimum sodium carbonate ratio was about 3%.

3.1.5. Effect of leaching time

The effect of leaching time on extraction of iron was investigated under the following conditions: cyanide tailings, activated carbon, sodium sulfate and sodium carbonate at a mixing ratio of 100:10:10:3, 750 °C for roasted temperature, 60 min for roasting time, 60 °C for leaching temperature, 5:1 for the liquid-to-solid ratio, 20 r/min for stirring speed and 2 A for the exciting current. The results are shown in Fig. 6.

It can be seen from Fig. 6 that the recovery rate of iron decreased remarkably at the initial leaching stages and then decreased slowly.
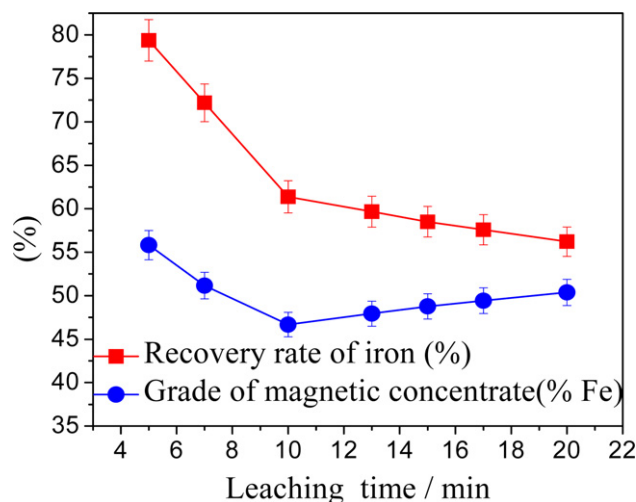
However, the grade of magnetic concentrate decreased considerably at the initial leaching time of 5 min and then increased. It is maybe because the content of iron gesso increase gradually with the extension of leaching time, and soluble compounds produced in the roasting process entered into the solution. Therefore, it is inferred that the optimum leaching time is 5 min, during which both the grade of magnetic concentrate and recovery rate of iron acquired are the highest.

3.1.6. Effect of leaching temperature

The effect of leaching temperature on extraction of iron was investigated under the following conditions: cyanide tailings, activated carbon, sodium sulfate and sodium carbonate at a mixing ratio of 100:10:10:3, 750 °C for roasted temperature, 60 min for roasting time, 5 min for leaching time, 5:1 for the liquid-to-solid ratio, 20 r/min for stirring speed and 2 A for the exciting current. The results are shown in Fig. 7.

The results in Fig. 7 indicate that while the leaching temperature is less than 60 °C, the grade of magnetic concentrate increases remarkably with the increase of leaching temperature. This indicates that the higher temperature increases the molecular motion and reduces solution viscosity, which is more conductive to leaching. However, it can be seen that the grade of magnetic concentrate decreases slowly when the leaching temperature is higher than
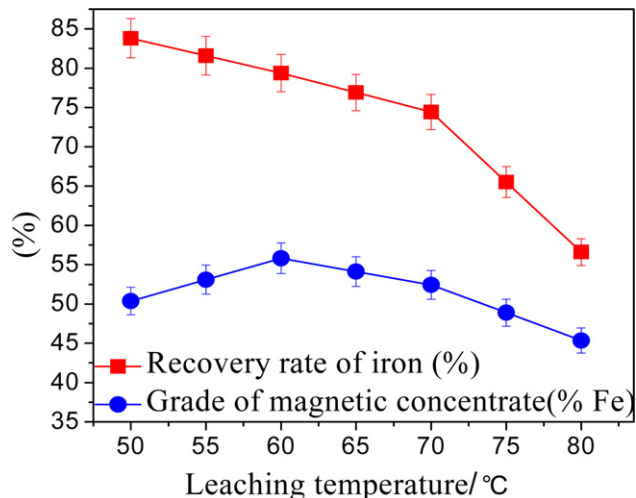

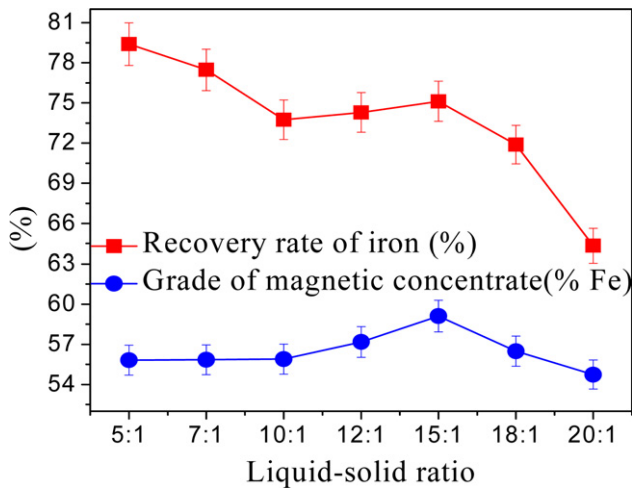**Fig. 7.** Effects of leaching temperature on iron recovery.

Fig. 8. Effects of liquid–solid ratio on iron recovery.

Fig. 9. Effects of exciting current on iron recovery.

60 °C. It is maybe because magnetite produced in roasting process is easy to form complex iron and compounds with some impurities, which causes the loss of iron concentrate, and the decrease of both the grade of magnetic concentrate and recovery rate of iron. This may be because the burning loss of water-leached ore keeps increasing with the increase of leaching temperature, so the recovery rate of iron tends to decrease gradually. The optimum leaching temperature, therefore, is 60 °C.

3.1.7. Effect of liquid-to-solid ratio

The effect of liquid-to-solid ratio on extraction of iron was investigated under the following conditions: cyanide tailings, activated carbon, sodium sulfate and sodium carbonate at a mixing ratio of 100:10:10:3, 750 °C for roasted temperature, 60 min for roasting time, 5 min for leaching time, 60 °C for the leaching temperature, 20 r/min for stirring speed and 2 A for the exciting current. The results are shown in Fig. 8.

It indicates that when the liquid-to-solid ratio is smaller, pulp viscosity is high, and solution is easy to reach saturation in leaching interface, preventing the leaching reactions of bounding materials. When the liquid-to-solid ratio keeps increasing, mass transfer speed is improved correspondingly and the solution materials in the liquid-to-solid interface keep speeding outside, the reactions go with the best speed. But when the liquid-to-solid ratio goes on increasing, the concentration of untreated and residual sodium carbonate in solution drops which is detrimental to the dissolution of the impurities. So, considering the grade of magnetic concentrate and recovery rate of iron, simultaneously, the optimum liquid-to-solid ratio is chosen to be 15:1.

3.1.8. Effect of exciting current

The effect of exciting current on extraction of iron was investigated under the following conditions: cyanide tailings, activated carbon, sodium carbonate and sodium sulfate at a mixing ratio of 100:10:10:3, 750 °C for roasted temperature, 60 min for roasting time, 5 min for leaching time, 60 °C for the leaching temperature, 15:1 for liquid-to-solid ratio and 20 r/min for rotate speed. The results are shown in Fig. 9.

It can be seen from Fig. 9 that no magnetic concentrate can be separated by the magnetic separator when the exciting current is 1 A. While the exciting current is over 1 A, the recovery rate of iron increases remarkably from 1 A to 1.5 A and almost shows linear relationship, and then increases slowly from 1.5 A to 3 A. There was non-linear relationship between grade of magnetic concentrate and exciting current, when exciting current was about
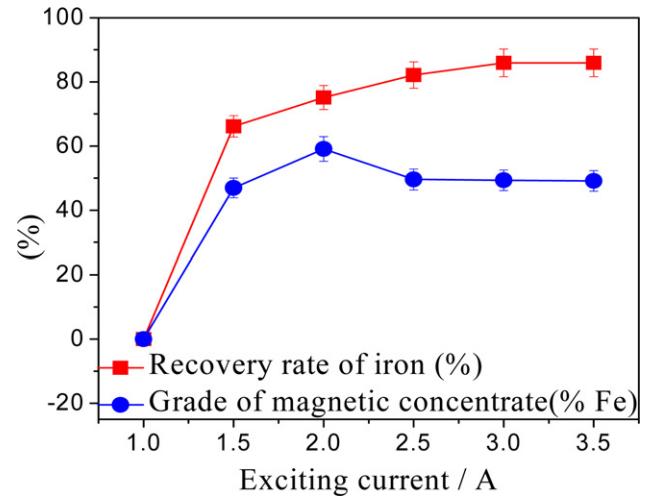
2 A, the grade of magnetic concentrate was achieving a peak. It is maybe because magnetic concentrate is hard to be chosen completely when the exciting current is smaller (namely magnetic field intensity is weaker). When the exciting current is bigger, impurity mineral mingles with the magnetic concentrate, producing magnetic reunion phenomenon and causing the yield of magnetic concentrate to increase, but the grade decreases due to the existence of nonmagnetic impurities mineral. Based on the results of the experiment, the best exciting current is 2 A.

3.2. Analysis of roasted sample

3.2.1. XRD analysis

The roasted product prepared in optimized conditions as following: roasting temperature of 750 °C, roasting time of 60 min, weight ratios of cyanide tailings/sodium carbonate/sodium sulfate/activated carbon of 100:3:10:10, and particle size 85% less than 74 μm in the roasted products was analyzed by XRD. The XRD pattern is shown in Fig. 10.

From Fig. 10, it can be seen that the hematite had been reduced to magnetite, which is easy to be separated by low intensity magnetic separation. Compared with those in Fig. 1, the peaks of silica were getting lower, which indicates that silica can react with some other components in the roasting process. Moreover, the peaks
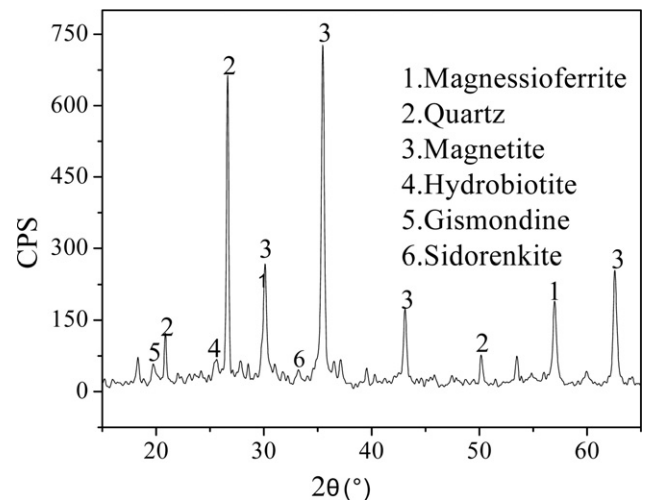

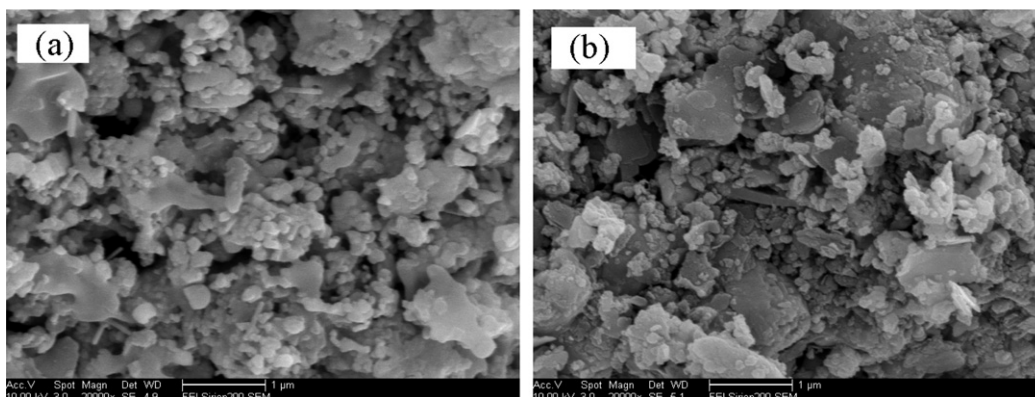**Fig. 10.** XRD pattern of roasted sample.

Fig. 11. The SEM images of roasted sample and raw material (a: roasted sample; b: raw material).
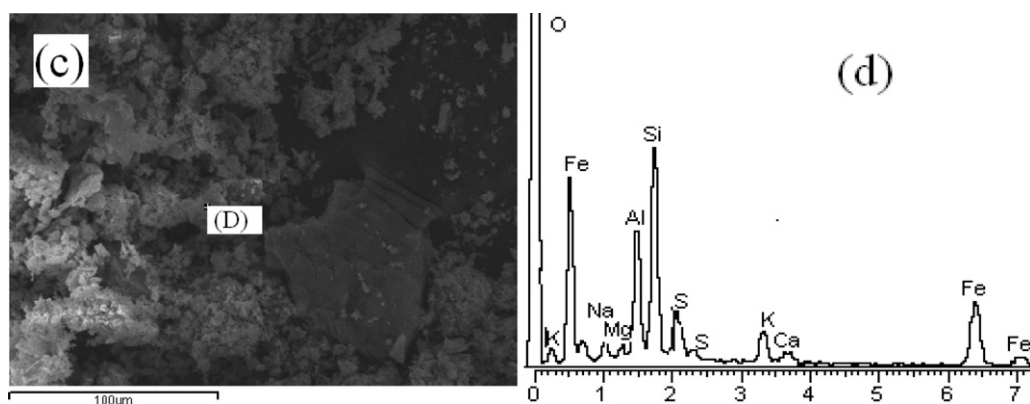

Fig. 12. The SEM image of roasted sample (a) and its corresponding EDS at area D (b).

of muscovite and hematite disappear completely. But the peaks of intermediate products such as magnesioferrite, hydrobiotite, gismodine and sodorenkite appear simultaneously in roasted process. Therefore, it can be concluded that hematite was completely reduced to magnetite, some main impurities and the additives reacted with each other, and formed these complex compounds. The reactions (Eqs. (5)–(7)) also show the formation of these complex compounds.

3.2.2. SEM–EDS analysis

Fig. 11 shows SEM images of the roasted product under optimized conditions and the EDS results are also presented in Fig. 12.

As shown in Fig. 11(a) and (b), at the temperature of 750 °C for 60 min, the roasted samples become loose and porous by reduction roasting, which will be beneficial to the recovery of iron concentrate.

As shown in the energy spectrum (Fig. 14), Fe, Si and Al were fine-disseminated together, which made them hard to be separated effectively, so the reasonable and effective method of separating Fe, Si and Al was necessary, such as grinding and water-leaching of the roasted sample.

3.3. Analysis of water-leached sample

3.3.1. XRD analysis

Taking the water-leached samples for analysis, the optimum experimental conditions were as follows: the leaching temperature of 60 °C for 5 min, the stirring speed of 20 r/min and the ratio of liquid to solid of 15:1. Its XRD pattern is shown in Fig. 13.

From Fig. 13, it can be seen that the peaks of magnetite were getting higher obviously, and the peaks of silica were getting lower compared with the XRD pattern of roasted sample. Meanwhile, the peaks of hydrobiotite disappeared completely, but the peaks of gismondine and sidorenkite still existed in the water-leached sample, it indicated that hydrobiotite produced in roasting process are easy to dissolve in the solution. On the contrary, gismondine and sidorenkite were hard to dissolve in the solution, so the magnetite was enriched.
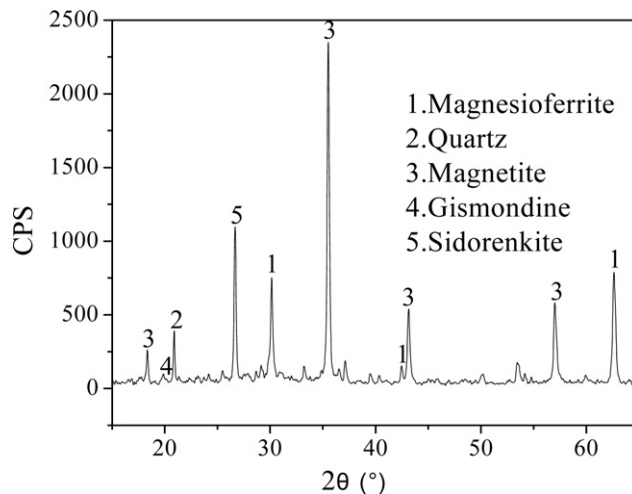

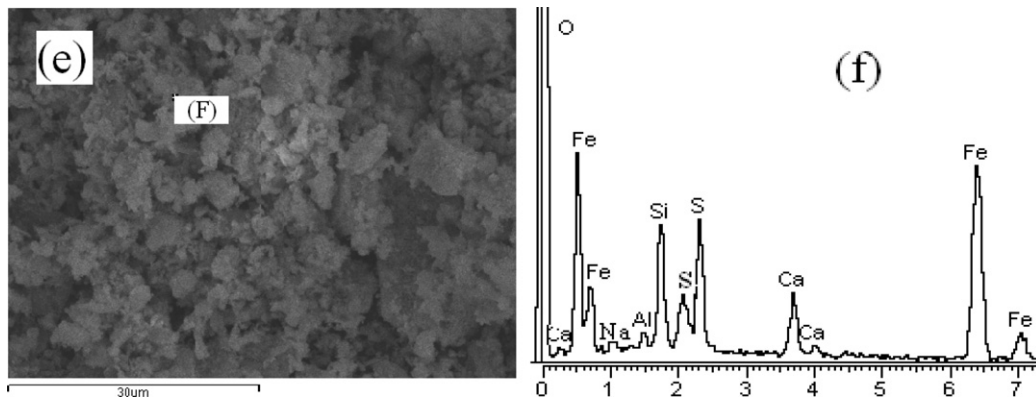**Fig. 13.** XRD pattern of water-leached sample.

Fig. 14. The SEM image of water-leached sample (a) and its corresponding EDS at area F (b).

3.3.2. SEM-EDS analysis

Fig. 14 shows SEM images of water-leached sample under optimized process parameters as above. The EDS results are also presented.

Compared the EDS patterns of roasted samples (Fig. 12(b)) with water-leached samples (Fig. 14(b)), after leached, the peaks of Fe was getting higher, the peaks of Al and Si were getting lower, and the peaks of K and Mg disappeared completely, which was supported by XRD (Fig. 13).

3.4. Analysis of magnetic concentrate

3.4.1. XRD analysis

The magnetic concentrate was separated from water leached samples under optimized process parameters. Its XRD pattern was shown in Fig. 15.

From Fig. 15, magnetite was the main mineral phase of magnetic concentrate, but a small amount of quartz also existed, probably because that the silica which was not disintegrated out of ore in the roasting process was separated together with the magnetic concentrate, or the smaller mineral size produced magnetic reunion phenomenon. In addition, the peaks of gismondine and sidorenkite disappeared completely, which proved that both the gismondine and sidorenkite had no magnetism, and were separated from the magnetic concentrate.

3.4.2. SEM-EDS analysis

The SEM image of magnetic concentrate and its energy spectrum of some areas under the optimum conditions were shown in Fig. 16.

From Fig. 16, iron was the main element in the magnetic concentrate, but elements of Si and Al still existed, which was in accord
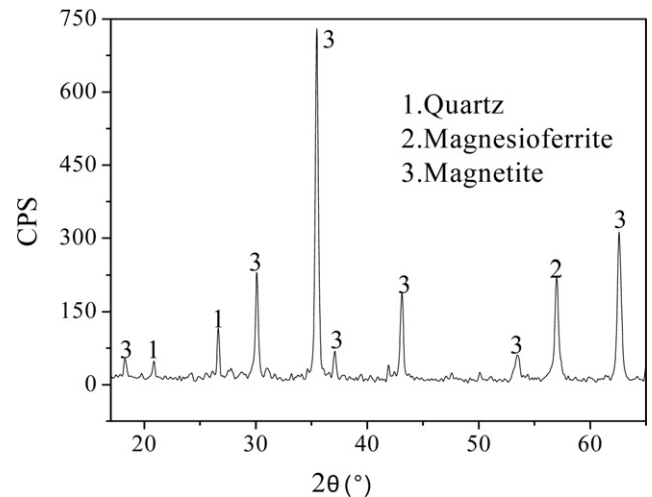

Fig. 15. XRD pattern of magnetic concentrate.

with XRD analysis, which caused the grade of magnetic concentrate not high.

3.5. XRD analysis of magnetic tailings

Taking the magnetic tailings for analysis, its XRD pattern was shown in Fig. 17. From the XRD pattern, it can be seen that quartz was the main mineral phase of magnetic tailings, which proved that most of the quartz were separated by the way of reduction roasting–water leaching followed by magnetic separation. Meanwhile, in comparison with the XRD pattern of magnetic
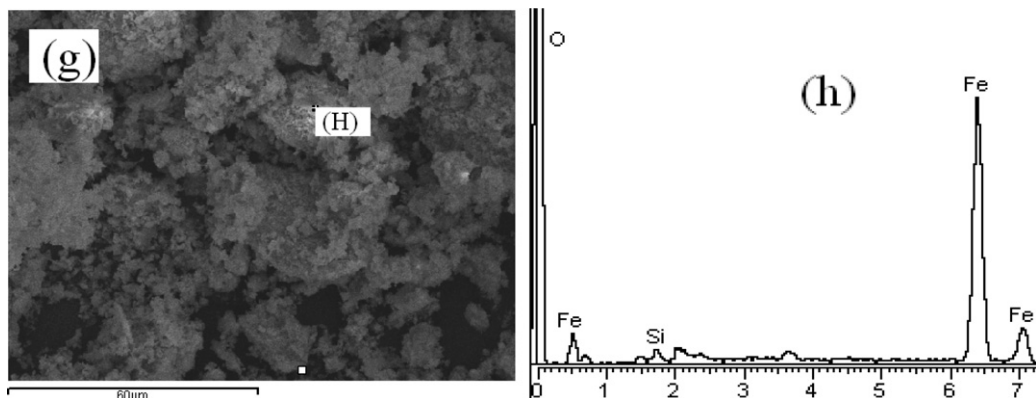

Fig. 16. The SEM image of magnetic concentrate (a) and its corresponding EDS at area H (b).

Table 2
Weight percentage of main elements in original slag, roasted sample and leached sample.

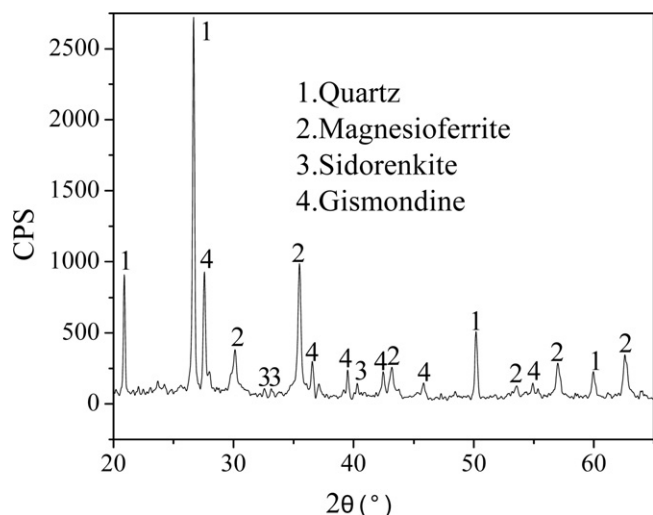|	O/%	Na/%	Mg/%	Si/%	S/%	K/%	Ca/%	Al/%	Fe/%
Original slag	43.60	0.96	0.46	11.50	1.71	1.76	0.98	3.74	27.69
Roasted sample	44.16	1.75	0.66	14.83	1.01	1.96	1.07	4.43	30.09
Leached sample	35.60	1.40	0	7.88	7.11	0	4.53	1.10	42.38

Fig. 17. XRD pattern of magnetic tailings.

concentrate, the peaks of intermediate products, such as magnesioferrite, gismondine and sidorenkite appeared in magnetic tailings, which indicated that they mostly entered into the magnetic tailings due to their nonmagnetic.

3.6. The energy disperse X-ray fluorescence analysis

The weight percentage of main elements in original slag, roasted sample and leached sample from EDX are shown in Table 2. By comparing these data, it can be seen that the content of sulfur decreased after magnetic roasting, but the content of other elements were beneficiation, it may be because that sulfur generated volatile gas in the roasting process. Generally, after the magnetic roasting–water leaching process, the content of Si and Al in water leached sample reduced from 11.50% in original slag to 7.88%, and from 3.74% to 1.10%, respectively, and Fe rose from 27.69% to 42.38%, but the content of Na also increased from 0.96% to 1.40% which were in accord with XRD patterns in Figs. 10, 13, 15 and 17, and the SEM-EDS images in Figs. 12 and 14. Therefore, it can be concluded that effective separation between Fe, Si and Al can be realized by the magnetic roasting–water leaching process, especially the water leaching pretreatment, and both the grade of magnetic concentrate and recovery rate of iron improved.

4. Conclusions

(1) Major chemical compositions of cyanide tailings were Fe_2O_3, SiO_2 and Al_2O_3. Hematite, quartz and muscovite existed in cyanide tailings as main phases, and most of iron and quartz were fine-disseminated together, which were hard to be separated. A new, efficient method of iron extraction is proposed in the present work. The results show that the route is technically viable, and it provides the possibility of comprehensive utilization of high-iron solid waste.

(2) Activated carbon ratio, roasted temperature, roasted time, additive ratio, leaching temperature, leaching time, liquid-to-solid ratio and exciting current are the main factors which affect the iron recovery. The grade of magnetic concentrate of 65.11% Fe and iron recovery rate of 75.12% occur under the following conditions: ratio of activated carbon: cyanide tailing: sodium carbonate: sodium sulfate as 100:10:10:3, roasting at 750 °C for 60 min, ratio of liquid:solid as 15:1, leaching at 60 °C for 5 min with the stirring speed of 20 r/min, and exciting current of 2 A.

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