ABSTRACT: Herein, a new clean extraction technology for the decomposition of bastnasite concentrate by utilizing the microwave radiation is proposed, which prevented Ce(III) from being oxidized to its tetravalent form. The process includes microwave radiation roasting to nonoxidatively decompose the bastnasite concentrate, mechanism analysis of Ce(III) not being oxidized to Ce(IV), hydrochloric acid leaching of the nonoxidative roasted ore, and kinetics analysis of the leaching process. The experiments were carried out concentrating on the effect of roasting temperature and holding time on the decomposition rate of the bastnasite concentrate and the oxidation rate of cerium and the effect of acidity, liquid–solid ratio, leaching temperature, and stirring rate on the leaching kinetics of the nonoxidative roasted ore. When the roasting temperature is 1100 °C, the holding time is 20 min, and the \(m(C)/m(REFCO_3)\) ratio is 0.2, the results show that the leaching efficiency of rare earths can reach 85.45% under the conditions 3 mol/L HCl, 90 °C, 60 min, 9 mL/g liquid–solid ratio, and 300 rpm stirring rate. The X-ray diffraction and scanning electron microscopy analyses of the samples before and after acid leaching show that the rare earth oxides were completely leached and Ce(III) was not oxidized to its tetravalent form. The apparent activation energies of leaching rare earths were calculated as 14.326 kJ/mol, and the HCl leaching process can be described by a new variant of the shrinking-core model, in which both the interfacial transfer and the diffusion through the product layer influenced the reaction rate. Furthermore, a semiempirical rate equation was created to describe the leaching process of the nonoxidative roasted ore.

INTRODUCTION

Bastnasite \((REFCO_3)\) is one of the most critical light rare earth resources and provides approximately 70% rare earth production in China and also in the world.\(^1\)\(^,\)\(^2\) The oxidative roasting hydrochloric acid leaching process is a mature industrial process of decomposing bastnasite concentrate.\(^3\) Bastnasite can be easily decomposed to rare earth oxides during the roasting process.\(^4\) However, Ce(III) in bastnasite is unavoidably oxidized to Ce(IV) during the oxidation roasting process, and the oxidation rate of cerium is higher than 96%.\(^5\)\(^,\)\(^6\) Many studies suggest that pure CeO\(_2\) can hardly be leached by hydrochloric acid. However, Ce\(^{4+}\) and F\(^-\) can easily generate a stable complex \([\text{CeF}_x]\)\(^{4-\cdot x}\)\(^5\)\(^,\)\(^6\) and this complexation can be leached under the conditions of high temperature and highly concentrated hydrochloric acid.\(^7\) However, a large volume of Cl\(_2\) is discharged during the leaching process because the Ce\(^{4+}\) is reduced to Ce\(^{3+}\), while Cl\(^-\) is oxidized to Cl\(_2\), and this process also introduced harmful F\(^-\) to the RECl\(_3\) solution. The chemical reaction is listed as follows

\[
2[\text{CeF}_x]^{4-\cdot x} + 2\text{Cl}^- \rightarrow 2\text{Ce}^{3+} + 2x\text{F}^- + \text{Cl}_2
\]

To inhibit the generation of Cl\(_2\) during the leaching process, a reductant must be added,\(^8\) which increases expenditure on reductant costs significantly. If Ce(III) is not oxidized to Ce(IV), all rare earths in the roasted ore can be kept in their trivalent form.\(^9\) In this case, the reductant does not have to be added, and fewer impurities such as F\(^-\) go into the RECl\(_3\) solution. Thus, a new nonoxidative roasting hydrochloric acid leaching process is urgently required to alleviate the burdens of the ecosystems and economy. Unfortunately, few studies have investigated the nonoxidative roasting hydrochloric acid leaching process of bastnasite concentrate.

Microwave irradiation extraction of minerals in metallurgy has shifted into the focus of attention in recent years.\(^10\)\(^-\)\(^12\) Compared with traditional heating, the major advantages of microwave irradiation are the rapid internal heating, selective heating, higher thermal efficiency, pollution-free environment,
convenient automatization, and several nonthermal effects of microwaves. Because of the selective heating of microwave irradiation, it only affects polar substances. Activated carbon is an effective microwave absorber by which the heating rate by microwave irradiation can reach 21 K/s, and it can be heated to 1053–1556 K in a short time. The purpose of using activated carbon to improve the microwave radiation performance of minerals to achieve a rapid temperature rise is a commonly used method in metallurgy. Activated carbon-assisted microwave irradiation roasting of bastnasite concentrate not only significantly increases the heating rate but also prevents Ce(III) from being oxidized to its tetravalent form.

The aim of this research is to study the mechanism of Ce(III) not being oxidized to Ce(IV) during the microwave radiation roasting process, hydrochloric acid leaching process of the nonoxidative roasted ore, and hydrochloric acid leaching kinetics. The kinetic model concluded as a new shrinking core model controlled by interfacial transfer and diffusion controls across the product layer, and the activation energy of the leaching process was calculated as 14.326 kJ/mol. Furthermore, we created a semiempirical rate equation to describe the leaching process of the nonoxidative roasted ore. It can provide theoretical support for the mechanisms of leaching rare earths from nonoxidative roasted ores and help us optimize the leaching conditions.

**RESULTS AND DISCUSSION**

**Roasting Process. Effect of Roasting Temperature.** When the activated carbon/bastnasite concentrate ratio \[m(C)/m(REFCO_3)\] was 0.2, the roasting temperature could rapidly rise to 1100 °C in less than 10 min and could also remain relatively stable at 1100 °C. The heating curve shown in Figure 1 is consistent with the results reported by other research groups. We chose such a high roasting temperature because we can take full advantage of the rapid internal heating of microwave radiation, and the highest microwave roasting temperature that the ore can reach is 1100 °C. The decomposition reaction of the bastnasite concentrate is an endothermic reaction, which is more favorable for promoting the decomposition of the bastnasite concentrate at a high temperature. The thermogravimetric (TG)−differential scanning calorimetry (DSC) curves of the bastnasite concentrate are shown in Figure 2. It is known that the bastnasite concentrate begins to decompose at 427 °C and peaks at 482.1 °C. We believe that higher roasting temperature can significantly shorten the roasting time to achieve the purpose of saving energy. Therefore, the microwave roasting temperature was selected to be 1100 °C as the optimum condition.

The decomposition products were mainly rare earth oxides, as shown in reaction eqs 2 and 3

\[
2\text{REF}_2\text{RE}(\text{CO}_3)_3 = \text{RE}_2\text{O}_3 + 3\text{REF}_2 + 6\text{CO}_2
\]  
\[
3\text{REF}_2 = \text{RE}_2\text{O}_3 + \text{REF}_3
\]

**Effect of Holding Time.** To study the oxidation degree of trivalent cerium and decomposed behavior of the bastnasite concentrate in the microwave heating process, bastnasite concentrate mixed with activated carbon was treated by microwave radiation for different holding times. As shown in Figure 3, when the holding time was gradually extended, the decomposition rate of the bastnasite concentrate was increased in the first 20 min and reached a maximum value of 99.4% at a holding time of 20 min. When the roasting time was extended

![Figure 1. Temperature rise curve of bastnasite concentrate mixed with activated carbon (1200 W).](image1)

![Figure 2. TG–DSC curve of bastnasite concentrate.](image2)

![Figure 3. Decomposition rate of bastnasite concentrate and oxidation rate of cerium with samples roasted for different times by microwave radiation (roasting temperature 1100 °C, \[m(C)/m(REFCO_3)\] 0.2, 1200 W).](image3)
beyond 20 min, the decomposition rate of the bastnasite concentrate decreased obviously, and it could be deduced that the ore was sintered after 20 min. As the holding time was extended, the sintering degree of the ore also increased, which agreed with the experimental phenomenon of the decrease in the decomposition rate. Moreover, the oxidation rate of cerium was lower than 0.3% in the first 30 min with increasing holding time; it could be deduced that Ce(III) was not oxidized to Ce(IV) in the first 30 min. However, at a holding time of 40 min, the oxidation rate of cerium was 1.3%. This phenomenon indicates that Ce(III) started to oxidize at the holding time of 40 min and activated carbon had been exhausted. To obtain a high decomposition rate of the bastnasite concentrate under the premise of ensuring that Ce(III) was not oxidized, a roasting temperature of 1100 °C and a holding time of 20 min were the best microwave roasting conditions.

**Mechanism Analysis of Ce(III) Not being Oxidized to Ce(IV).** The reason why Ce(III) was not oxidized to Ce(IV) is the following three mechanisms: [1] cerium oxide underwent carbothermal reduction reactions to reduce Ce(IV) to Ce(III). [2] Activated carbon oxidized to form a reducing gas CO, and the CO reduced Ce(IV) to Ce(III). [3] Activated carbon reacted with O₂ to form CO₂ and CO, so it consumed oxygen on the surface of mineral particles. The surface of the mineral particles was a CO₂ atmosphere because \( \rho(CO_2) > \rho(O_2) > \rho(CO) \). Therefore, Ce(III) was not oxidized to Ce(IV) in the CO₂ atmosphere during the roasting process. The reactions are summarized as follows

\[
\begin{align*}
4CeO_2 + C &= 2Ce_2O_3 + CO_2 \\
2CeO_2 + C &= Ce_2O_3 + CO \\
2CeO_2 + CO &= Ce_2O_3 + CO_2 \\
C + O_2 &= CO_2 \\
2C + O_2 &= 2CO
\end{align*}
\]

The Gibbs free energy of the above reactions needs to be calculated to understand the reason why Ce(III) was not oxidized to Ce(IV), and the results are shown in Figure 5. According to thermodynamics calculation and analysis, the carbothermal reduction reactions 4 and 5 can be carried out at 1100 °C, but reaction 6 cannot be carried out at 1200 °C, and therefore, mechanism [2] is excluded; reactions 7 and 8 are thermodynamically easy. Therefore, mechanisms [1] and [3] may coexist.

Activated carbon could be rapidly heated by microwave radiation and then transferred heat to the bastnasite concentrate. That is, the temperature of activated carbon particles was higher than that of bastnasite concentrate particles. Before the bastnasite concentrate reached decomposition temperature, activated carbon had consumed the oxygen on the surface of mineral particles. Because the gas density is \( \rho(CO_2) > \rho(O_2) > \rho(CO) \), CO₂ is denser than O₂, so the generated CO₂ was deposited around the bastnasite concentrate particles. At the roasting temperature of 1100 °C, the Ce(IV) can also be deoxygenized to Ce(III) by carbothermal reduction reactions 4 and 5 during the roasting process. Because of bastnasite particles being placed in a CO₂ atmosphere and undergoing the carbothermal reduction reactions, Ce(III) could not be oxidized to Ce(IV) during the roasting process. Based on the above mechanisms, Ce(III) would inevitably be oxidized to Ce(IV) if activated carbon had been exhausted.

**Leaching Process.** Effect of the Initial Concentration of HCl. All the nonoxidized roasted ores used in the leaching experiments were roasted under the best microwave roasting conditions. The effect of the initial concentration of HCl on the leaching efficiency of rare earths was investigated, as shown in Figure 6. The initial concentration of HCl is selected from 1.0 to 5.0 mol/L. It was observed that higher acidity of HCl could accelerate the leaching out of rare earths, and the leaching efficiency of rare earths reached the equilibrium in approximately 60 min, and then, the increase rate gradually slows down. Moreover, with the increase in the initial concentration of HCl from 3.0 to 5.0 mol/L, the leaching efficiency of rare earths could not significantly improve (85.5–87.3%). Therefore, 3.0 mol/L HCl was selected as the optimum initial concentration for further investigations.

**Effect of Liquid–Solid Ratio.** The effect of the liquid–solid ratio (L/S) on the leaching efficiency of rare earths was studied, and the results are shown in Figure 7. With the L/S increased from 5 to 10 mL/g, when the leaching efficiency of rare earths was increased from 66.9 to 86.4% after 60 min. However, the leaching efficiency of rare earths reached 85.5 and 86.4% after 60 min at 9 and 10 mL/g, respectively. No significant difference was observed for the leaching efficiency of rare earths within the liquid–solid ratio ranging from 9 to 10.
mL/g. Therefore, the liquid–solid ratio (L/S) of 9 mL/g was selected as the optimum liquid–solid ratio for further investigations.

A low L/S inhibited the leaching efficiency of rare earths; on the other hand, a higher L/S can reduce the viscosity of slurry to improve the mass transfer rate between the solid and liquid but will significantly reduce the concentration of rare earths which will improve the wastewater and decrease the operational efficiency of the solvent extraction.

**Effect of Leaching Temperature.** Figure 8 presents the effects of leaching temperature on the leaching efficiency of rare earths as a function of time. When the temperature rose from 50 to 90 °C, the leaching efficiency of rare earths increased from 72.3 to 85.4% after 60 min. It can be observed from Figure 8 that the leaching temperature has a significant influence on the leaching efficiency of rare earths. The leaching efficiency of rare earths reached equilibrium in approximately 60 min, and then, the increase rate slows down. To maximize the leaching efficiency of rare earths, the leaching temperature should be 90 °C as the best experimental condition.

**Effect of Stirring Rate.** The effect of the stirring rate on the leaching efficiency of rare earths was investigated at 100, 200, 300, and 400 rpm. The results in Figure 9 show that the leaching efficiency of rare earths increased from 74.7 to 85.4% with the increase in stirring rate from 100 to 300 rpm and reached equilibrium after 60 min. However, the leaching efficiency of rare earths under 400 rpm was less than that under 300 rpm, which indicated that a too large stirring rate affects the adsorption and diffusion on the mineral surface and the leaching efficiency was weakened under 400 rpm.

Through comprehensive consideration, an initial HCl concentration of 3 mol/L, a liquid–solid ratio of 9 mL/g, a leaching temperature of 90 °C, a leaching time of 60 min, and a stirring rate of 300 rpm are the best leaching conditions.
respectively. Figure 10 shows the morphology changes in the nonoxidative roasting ore before and after hydrochloric acid leaching. Compared with the morphology of the bastnasite concentrate as shown in Figure 20A, the particles of the roasting ore presented a rough and broken surface (see Figure 10A). Moreover, after hydrochloric acid leached, the particle surface became porous and cracked (see Figure 10B). Scanning electron microscopy (SEM) and XRD analysis indicated that the RE2O3 had completely leached into the hydrochloric acid (see Figure 11). However, REF3 (represented by (Ce, Nd, La)-F3) was almost not leaching into hydrochloric acid because REF3 is not easily soluble in dilute hydrochloric acid. From the SEM mapping of the leaching residue in Figure 12, the same results can also be obtained. That is the reason why the leaching efficiency of rare earths can only reach about 85%. However, the leaching process can effectively prevent the introduction of harmful F− into the RECl3 solution.

**Kinetic Analysis.** The leaching process of nonoxidized roasted ores was analyzed using the shrinking core model to understanding the leaching mechanism. According to the SEM image of nonoxidative roasted ore particles, the particles can be assumed to be perfect spherical particles. So, the hydrochloric
The leaching process was analyzed using the shrinking core model, which is used in many studies on the leaching of the rare earth concentrate. Different shrinking core models were tested to find the most suitable model equation, and the different kinetic equations of the leaching process were as follows:

- Outer diffusion or chemical reaction controls
  \[ 1 - (1 - X)^{1/3} = K_d t \]  
  \[ \text{Inner diffusion controls} \]
  \[ 1 - 2/3X - (1 - X)^{2/3} = K_i t \]  
  \[ \text{Interfacial transfer and diffusion controls} \]
  \[ 1/3\ln(1 - X) + [(1 - X)^{-1/3} - 1] = K_d t \]  

where, \( X \) is the leaching efficiency of rare earths (%); \( K_d, K_i \) and \( t \) are the different rate constants; and \( t \) is the leaching time (min).

As shown in Figure 13, the rate constants \( R^2 \) for temperature increase from 50°C to 90°C were calculated for the leaching efficiency of rare earths. Moreover, the correlation coefficients \( R^2 \) of the different kinetic equations are listed in Table 1. We evaluated that the correlation coefficient \( R^2 \) values of these models, eq 11, were the most suitable to fit the kinetic data. As shown in Table 2, the correlation coefficient \( R^2 \) of eq 9 and 10 is not all upon 0.95, indicating that the leaching reaction control does not conform to the outer diffusion controls or chemical reaction and inner diffusion control models. However, the correlation coefficient \( R^2 \) of eq 11 is all above 0.98, indicating that the leaching process fitted the new shrinking core model well.

Arrhenius plots for extractions of rare earths are shown in Figure 14. The apparent activation energies \( E_a \) of 14.326 kJ/mol was obtained from the slope of the straight line \( (-E_a/R) \). The activation energy of a diffusion-controlled process was characterized to be less than 13 kJ/mol, while for a chemically controlled process, it is usually more than 42 kJ/mol. The apparent activation energies of 14.326 kJ/mol indicate that the reaction rate was not controlled by the outer diffusion controls or chemical reaction and inner diffusion. It also can be inferred that the HCl leaching process of nonoxidative roasted ore fitted the new shrinking core model controlled by interfacial transfer and diffusion controls across the product layer.

As a result, the kinetic expression of the hydrochloric acid leaching of the nonoxidative roasted ore can be used to describe the leaching process:

\[ 1/3\ln(1 - X) + [(1 - X)^{-1/3} - 1] = K_d t \]  

where \( k_d \) is the Arrhenius constant, which is the intercept of the straight line \( (-E_a/R) \) in Figure 13. \( E_a \) is the apparent activation energy, \( R \) is the universal gas constant (J/mol/K), \( T \) is the leaching temperature (K), and \( t \) is the leaching time (min).

As shown in Figure 15, the fitting plot of 1/3ln(1 \( - X \)) + [(1 \( - X \))^{-1/3} - 1] versus time was obtained for different initial concentrations of HCl. The slope value of the straight line was the apparent rate constant \( k_a \) value and \( \ln(k_a) \) versus ln[HCl] plot shown in Figure 16 was used to determine the empirical reaction order of dependency concerning the initial concentration of HCl. In the same way, from the slopes of the straight line shown in Figure 17, the apparent rate constant \( k_a \) value was determined and a plot of \( \ln(k_a) \) versus ln[L/S] shown in Figure 18 was used to determine the empirical reaction order of dependency concerning liquid–solid ratio. The reaction order of dependency concerning the initial concentration of HCl and the liquid–solid ratio were 1.4361 and 1.51133, respectively. The relationship of the apparent rate constant and control factors including temperature, initial concentration of HCl, and the liquid–solid ratio was provided as follows:

\[ K_a = 4 \prod DV_m C_0 = K_0(C_{HCl})^{a}(L/S)^b e^{-E_a/RT} \]  

where \( D \) is the diffusion coefficient, \( V_m \) is the volume of product, and \( C_0 \) is the concentration of leaching agent (HCl).

Equation 13 is the combination of eqs 11 and 12:

\[ 1/3\ln(1 - X) + [(1 - X)^{-1/3} - 1] = K_0(C_{HCl})^{a}(L/S)^b e^{-E_a/RT} t \]  

According to eq 14, initial concentration of HCl, leaching temperature, and liquid–solid ratio have a significant effect on the leaching efficiency of rare earths, and the key control steps...
of the leaching process are both interfacial transfer and diffusion controls across the product layer. To improve the leaching efficiency of rare earths, the leaching process should be to raise the leaching temperature to increase the diffusion coefficient ($D$), to raise the initial concentration of HCl to increase the concentration of leaching agent ($C_0$), to raise the liquid–solid ratio to increase the volume of product ($V_m$), and to choose the adequate stirring rate to improve leaching efficiency of rare earths.

Table 1. Correlation Coefficients ($R^2$) of the Different Kinetic Models at Different Leaching Temperatures Rose from 50 to 90 °C

| Leaching Temperature (°C) | Kinetic Expression 1 | Kinetic Expression 2 | Kinetic Expression 3 |
|---------------------------|----------------------|----------------------|----------------------|
| 50                        | $1 - (1 - X)^{1/3}$  | $1 - 2/3X - (1 - X)^{2/3}$ | $1/3\ln(1 - X) + [(1 - X)^{-1/3} - 1]$ |
| 60                        | 0.84941              | 0.94773              | 0.99191              |
| 70                        | 0.84314              | 0.96721              | 0.99398              |
| 80                        | 0.84353              | 0.94526              | 0.99590              |
| 90                        | 0.84905              | 0.94963              | 0.99745              |

Table 2. Chemical Composition of the Bastnasite Concentrate (Mass Fraction, %)

| Composition | REO | CaO | Al₂O₃ | PbO | ZnO | PO₄³⁻ | SrO | F | MgO | ThO₂ |
|-------------|-----|-----|-------|-----|-----|-------|-----|---|-----|------|
| Content (%) | 55.86 | 10.61 | 0.19 | 0.19 | 0.012 | 1.58 | 1.50 | 3.43 | 0.35 | 0.10 |

Figure 14. Arrhenius plots for extractions of rare earths.

Figure 15. Plot of $1/3\ln(1 - X) + [(1 - X)^{-1/3} - 1]$ vs time for different initial concentrations of HCl.

Figure 16. Plot of $\ln K_a$ as a function of the initial concentration of HCl.

Figure 17. Plot of $1/3\ln(1 - X) + [(1 - X)^{-1/3} - 1]$ vs time for different liquid–solid ratios.

increase the concentration of leaching agent ($C_0$), to raise the liquid–solid ratio to increase the volume of product ($V_m$), and to choose the adequate stirring rate to improve leaching efficiency of rare earths.
CONCLUSIONS

This paper proposes a new nonoxidative roasting hydrochloric acid leaching process that inhibits the generation of Cl₂ during the hydrochloric acid leaching process and significantly reduces the cost of production. The optimum roasting conditions were determined as follows: 1100 °C, 0.2 m(C)/m(REFCO₃) ratio, and 20 min holding time. Moreover, the mechanism of Ce(III) not being oxidized to Ce(IV) during the microwave roasting process was studied. In the leaching experiments, the effects of the initial concentration of HCl, liquid–solid ratio, leaching temperature, and stirring rate on the leaching efficiency of rare earths were investigated via the control variate method. In addition, the optimum leaching conditions were determined as follows: 3 mol/L HCl, 90 °C, 60 min, 9 mL/g liquid–solid ratio, and 300 rpm stirring rate. Under these conditions, the leaching efficiency of rare earths can reach 85.45%. The leaching kinetic data fitted a new variant of the shrinking core model, which is controlled by both the interfacial transfer and diffusion through the product layer. The apparent activation energy was 14.326 kJ/mol, and the reaction orders with respect to the initial concentration of HCl and liquid–solid ratio were 1.4361 and 1.51133, respectively. Besides, a semiempirical rate equation of the leaching process was established as

\[
\frac{1}{3}\ln(1 - X) + [(1 - X)^{1/3} - 1] = -0.33228(C_{HCl})^{1.4361}(L/S)^{1.51133} e^{-14326/RT t}
\]

EXPERIMENTAL SECTION

Materials. The bastnasite concentrate used in this experiment was supplied by China Northern Rare Earth (Group) Hi-Tech Co. Ltd., and the ore was dried in an oven at 110 °C for 4 h to remove the free moisture water before use. Activated carbon and hydrochloric used in the experiments were analytical-grade reagents; all the aqueous solutions were prepared with distilled water. The chemical components and mineral components were analyzed by the Baotou Research Institute of Rare Earths and are shown in Tables 2 and 3, respectively. The XRD pattern and microscopic morphology of the bastnasite concentrate were analyzed, and the results were shown respectively in Figures 19 and 20. From the XRD pattern (Figure 19), it can be seen that the bastnasite concentrate was mainly composed of REFCO₃ (represented by CeFCO₃) and CaCe(CO₃)₂F (represented by CeFCO₃). Moreover, it can be seen from Figure 20A, the bastnasite concentrate particle presented a smooth and random surface.

Microwave Irradiation Roasting Decomposition. The microwave equipment used in this experiment was made by the key laboratory of unconventional Metallurgy, Ministry of Education, Kunming University of Science and Technology, China. Experiments on the microwave radiation roasting were carried out at the power level of 1200 W with a frequency of 2.45 GHz. Activated carbon was mixed with the bastnasite concentrate and prepared to a slurry of 18 wt% at a mass ratio of 1:1. The mixture was placed in a microwave oven and irradiated at 1200 W for 20 min. After roasting, the mixture was dried in an oven at 110 °C for 4 h before being ground to a size of 40 mesh for testing.

The main mineral components of the bastnasite concentrate are shown in Table 3. The XRD pattern and microscopic morphology of the bastnasite concentrate were analyzed, and the results were shown respectively in Figures 19 and 20. From the XRD pattern (Figure 19), it can be seen that the bastnasite concentrate was mainly composed of REFCO₃ (represented by CeFCO₃) and CaCe(CO₃)₂F (represented by CeFCO₃). Moreover, it can be seen from Figure 20A, the bastnasite concentrate particle presented a smooth and random surface.
concentrate by grinding, and the activated carbon/bastnasite concentrate ratio is 0.2. Then, the ore placed in a corundum crucible was placed in the center of the microwave equipment for irradiation. The roasting temperature was measured continuously by using a stainless-steel sheathed, K-type thermocouple which was inserted into the center of the ore. After the roasting process, the roasted ore was taken out then crushed and ground for further analysis. The particle size of the roasted ore is below 40 μm, and the distribution is narrow.

Leaching Procedures. The leaching experiments were carried out in a conical baker, and a water bath and mechanical stirrer were employed. When the hydrochloric acid solution with a certain concentration in the beaker attained the desired temperature, the required amount of nonoxidative roasting ore was charged into the beaker and then the stirring and timing began. After selected time intervals, 1 mL of the liquid sample was taken from the conical baker and filtered. The liquid samples were used to analyze the content of rare earths.

Experimental Analysis. The mineral structure in the experiment was measured using the PW-1700 X-ray diffractometer (Philips, Netherlands). The morphology of the minerals was analyzed using a Sigma-500 field-emission scanning electron microscope (Zeiss, Germany), and the mineral composition analysis was performed using an XFlash-6160 spectrometer (Brook). The concentration of rare earths in the liquid samples was analyzed by inductively coupled plasma atomic emission spectrometry and presented by oxides. The content of Ce⁴⁺ in the roasted ore was analyzed by the method of ferrous ammonium sulfate titration without the addition of perchloric acid. The decomposition rate of the bastnasite concentrate is determined by the following leaching method and is expressed by the leaching efficiency of rare earths. The leaching process was carried out under the conditions of initial hydrochloric acid concentration of 9.0 mol/L, liquid–solid ratio of 20:1, leaching temperature of 90 °C, leaching time of 60 min, and stirring rate of 300 rpm. The leaching efficiency of rare earths (μ) and the oxidation rate of cerium (η) were calculated with the following equation

\[
\mu = \frac{C_1 \times L}{C_2 \times S} \times 100\% \\
\eta = \frac{\omega_1 \times m_1}{\omega_2 \times m_2} \times 100\%
\]

where μ and η are the leaching efficiency of rare earths and the oxidation rate of cerium, respectively; C₁ and C₂ represent the concentration of rare earths in liquid samples and roasted ore, respectively; L/S is the liquid–solid ratio of leaching; m₁ and m₂ are the mass of the roasted ore and the bastnasite concentrate, respectively; and Ce⁴⁺ is the oxidation state of cerium in the roasted ore, respectively.

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Notes

The authors declare no competing financial interest.

ACKNOWLEDGMENTS

The authors gratefully acknowledge funding support from the Key program of the National Natural Science Foundation of China (grant number 51634005), the Key Program of the National Natural Science Foundation of Inner Mongolia Autonomous Region of China (grant number 2016ZD03), and the Key Program of the National Science Foundation of Inner Mongolia Autonomous Region of China (grant number 2017MS210).

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