

EXTRACTION OF PLATINUM AND GOLD FROM COPPER ANODE SLIMES BY A PROCESS OF CHLORINATING ROASTING FOLLOWED BY CHLORINATING LEACHING

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Abstract

A novel process of chlorinating roasting followed by chlorinating leaching to extract platinum and gold from copper anode slimes was proposed in this research. Results of thermodynamic analysis and experimental research showed that the platinum was chlorinated into $PtCl_2$, while the gold existed in the form of metallic Au during the roasting process. With the copper anode slime being directly leached using a traditional process, the Pt recovery rate was low and came to 80.72%. After the roasting process with sodium chloride and concentrated sulfuric acid in oxygen atmosphere, the recovery rate of Pt increased to a value around 95%. Moreover, with excessive addition of concentrated sulfuric acid, more H_2O (g) was generated and the formation of Cl_2 (g) decreased due to the transition from HCl (g) and Cl_2 (g), as a result of which the Pt recovery rate decreased. In addition, this chlorinating roasting had little effect on the Au recovery due to its difficulty to be chlorinated.

Keywords: Copper anode slime; Recovery of platinum and gold; Chlorinating roasting; Chlorinating leaching

1. Introduction

Platinum (Pt) and gold (Au) are costly precious metals with wide applications in the manufacturing of catalysts, electronic devices, space materials, biomedical devices, jewelry and so on, due to their distinct properties, such as chemical inertness, corrosion resistance, catalytic activity, thermoelectric stability and stable electrical properties [1-4]. However, their limited availability in nature and the abundance in the earth's crust are below $10\mu\text{g/g}$ in addition to the fact that the mineral ores are becoming depleted [5-6]. To meet the future demand, it is necessary to process the secondary platinum and gold containing materials, such as spent catalysts, electronic scraps, anode slimes, and so on [7-8].

The copper anode slime, a byproduct of the copper electro-refining process, is a valuable secondary resource containing valuable elements such as Cu, Se, Sb, Ag, Au, and platinum group metals [9-11]. Based on the different composition and mineralogy in anode slimes, numerous approaches have been made to

utilize it following the processes of pyro-, pyrohydro-, hydro-pyro-, and hydrometallurgical methods [12-16]. These processes can be divided into two steps of the separation and recovery of non-ferrous metals first and then the recovery of target precious metals [17]. The leaching process is mainly used to recover precious metals from the secondary slime using different lixiviants, such as sulfuric acid, nitric acid, hydrochloric acid, chloride, or iodide solutions, etc. J. Hait found that the recovery rate of Au was low with sulfuric acid leaching process without any additive [18]. Also in the thiosulfate leaching process, the Au extraction rate was only 88% [19]. High leaching rates of Au and Pt could be reached using aqua regia solution; however, it could cause environmental pollution [20]. With the anode slime dissolved with the help of chlorine gas in aqueous medium, a high recovery of gold of 94.4% could be obtained, but the chlorine gas causes serious corrosion of equipment [21]. Compared to these methods, a chlorinating leaching process using $\text{NaCl-H}_2\text{SO}_4\text{-NaClO}_3$ mixed solution has a few advantages, including a higher

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dissolution rate of gold, lower price of leaching reagents, as well as the simplicity of the process itself [22]. However, the leaching rate of platinum is low at the optimal condition favoring Au leaching, which causes a considerable amount of platinum to be distributed in the leaching residue and consequently the recovery rate of platinum can be difficult to increase.

In this paper, a novel process of chlorinating roasting followed by a chlorinating leaching to recovery platinum and gold from copper anode slime was investigated. Thermodynamic simulation of the chlorinating roasting procedure was discussed, and the effects of roasting temperature, oxygen flow rate, dosage of sodium chloride and concentrated sulfuric acid on recovery rates of platinum and gold were researched using XRD, SEM, and chemical analysis.

2. Experimental

2.1 Materials

The copper anode slime used in this work was obtained from a copper smelter located in Yunnan province of China. As presented in Table 1, the major elements contained in this slime were Cu, Sn, Sb, Ag, As, S, O, and Te, and the contents of Pt and Au were 93.76 g/t and 1812.15 g/t respectively. Fig.1 shows that this copper anode slime was mainly composed of SnO_2 , $\text{Sn}_{.918}\text{Sb}_{.109}\text{O}_2$, PbSO_4 , Cu_2S , SbAsO_5 and Ag_2SO_4 phases, while Pt and Au phases could not be detected due to their little contents. Generally, they mainly occur in metallic state [23].

The reagents used in the experiments were all analytical grade, including sodium chloride, sodium chlorate, sodium hydroxide, concentrated sulfuric acid, gold powder, and platinum powder. Besides, deionized water was used to prepare solutions in all leaching experiments.

2.2 Experimental procedures

The experimental apparatus and process flow sheet investigated for extractions of platinum and gold from copper anode slimes are given in Fig.2. For the experimental procedure, the copper anode slime was first dried at 100 °C for 24 h, ground to below 0.15 mm, and then mixed with a given amount of sodium chloride in a sample boat. After the tube

furnace was heated to a proper temperature, a given amount of concentrated sulfuric acid was added to the sample boat, and then it was immediately put in the tube furnace and roasted in a certain flow rate of O_2 . The exhaust gas was absorbed with the saturated sodium hydroxide solution. After a proper roasting time, the O_2 was cut off and the roasted samples were cooled down to room temperature before they were taken out of the tube furnace, and then weighed and stored for analysis and subsequent leaching experiments. In order to study the effects of roasting variables on the leachability of gold and platinum, each roasting product was leached under the identical conditions (80 °C; 4h; 3mL/g liquid-solid ratio; 120 g/L sulfuric acid, which was taken as the leaching agent; 60 g/L sodium chloride solution, which was used to provide Cl⁻; and 18 g/L sodium chlorate solution, which was taken as the chlorinating agent). For the leaching procedure, the roasted copper anode slime was first ground to below 0.074mm, put into a 400 mL glass conical flask containing the leaching solution, and then the glass conical flask was placed in an oscillation box equipped with a thermostatic water bath. It is noteworthy that the sodium chlorate solution was added per half an hour apart to ensure the high partial pressure of chlorine. After the leaching process finished, the solution was filtrated, and then the filter residue was collected, dried and weighed for analysis.

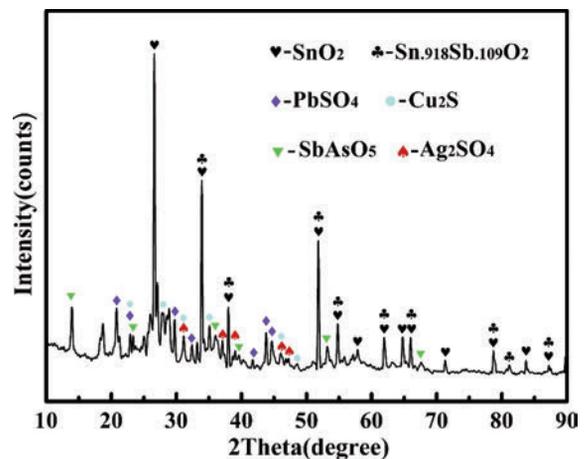


Figure 1. X-ray diffraction pattern of the copper anode slime

Table 1. Elemental composition of the copper anode slime (mass fraction, %)

Element	Pt(g/t)	Au(g/t)	Cu	Sn	Sb	Ag	As	S	Pb	Ni	Zn
Content	93.76	1812.15	12.77	13.72	9.83	4.79	6.21	8.32	3.18	3.2	0.99
Element	Bi	Se	Te	Si	O	others					
Content	3.15	3.23	4.37	0.5	23.78	1.96					



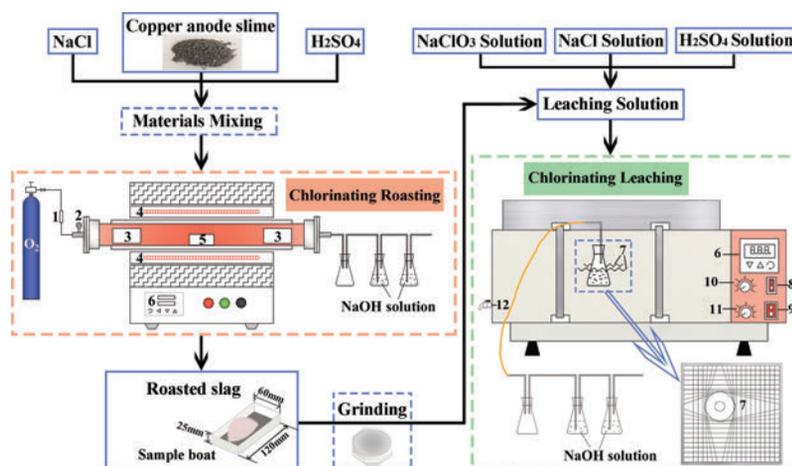


Figure 2. Experimental apparatus: 1—Flowmeter; 2—Pressure gauge; 3—Oven tap; 4—SiC heating component; 5—Sample boat; 6—Intelligent temperature controller; 7—Spring supported mounting; 8—Oscillation switch; 9—Power switch; 10—Oscillation rate regulator; 11—Timing device; 12—Water faucet

2.3 Analytical methods

The chemical composition of the copper anode slime was characterized by chemical analysis. The contents of Pt and Au in samples were determined by flame assaying method. Mineralogy of the samples was characterized by scanning electron microscope analysis (SEM-Tecnai G2 TF30, FEI, Netherlands). Phase composition of the sample was detected by the X-ray diffraction analysis (XRD, Rigaku, TTR- III), which was carried out using Cu K α radiation with the scanning rate of 8°/min and the 2 θ varying from 10 to 90°. The leaching rate of platinum and gold was calculated according to:

$$\varepsilon_p = \left(1 - \frac{m_r w_p}{m_i w_{op}} \right) \times 100\% \quad (1)$$

$$\varepsilon_a = \left(1 - \frac{m_r w_a}{m_i w_{oa}} \right) \times 100\% \quad (2)$$

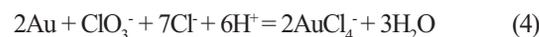
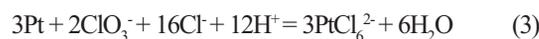
Here, ε_p and ε_a represent the leaching rate of platinum and gold, respectively; m_i and m_r are the masses of the original slime and leaching residue in g, respectively; w_{op} and w_{oa} are the contents of platinum and gold in the original slime in mass%, respectively; and w_p and w_a are the contents of platinum and gold in the leaching residue in mass%, respectively. The thermodynamic data of species were given by FactSage 7.2 thermochemical software.

3. Results and discussion

3.1 Thermodynamic analysis

Gold and platinum, as precious metals, have a low chemical activity and show a high ability of resistance to dissolution. Traditionally, a chlorination leaching

method is directly used to extract Pt and Au from copper anode slimes, in which the Pt and Au are chlorinated and transferred to acidsoluble PtCl_6^{2-} and AuCl_4^- through reactions (3)-(4) in a solution of $\text{NaCl-H}_2\text{SO}_4\text{-NaClO}_3$. The stability Eh-pH diagrams for systems of Au-Cl-H $_2$ O and Pt-Cl-H $_2$ O at 80 °C are presented in Figs.3 (a) and (b) respectively. The potential for Au leachable ranges from 0.96 V to 1.32 V with the pH from -2 to 5.45 seen from Fig.3 (a), and that for Pt ranges from 0.75 V to 1.58 V with the pH from -2 to 9.69 as shown in Fig.3 (b). To achieve these high potentials for obtaining high leaching rates of Au and Pt, much NaClO_3 should be employed in the leaching process. However, it causes the utilization efficiency of chlorine to be decreased due to the higher emission of Cl_2 during the leaching process.



An innovative process of copper anode slimes roasted with NaCl and H_2SO_4 in O_2 atmosphere was proposed in this research to convert Pt and Au to acidsoluble chlorides [24]. The equilibrium compositions of systems of Pt-NaCl-H $_2$ SO $_4$ -O $_2$ and Au-NaCl-H $_2$ SO $_4$ -O $_2$ were first calculated with Equilib Module of Factsage 7.2, in which some certain species originated from 40g copper anode slimes (Table 2), 4.00E-02mol H_2SO_4 , 1.61mol O_2 , and 6.84E-3mol (or 3.42E-02 mol) NaCl were used.

The results in Table 3 indicate that the chlorination of Pt proceeded through a progressive phase transformation process as the temperature rose, as described in Eq. (3) through reactions (4) to (6). Compared to that with 1% NaCl addition, the partial pressure of Cl_2 was higher when 5% NaCl was added, which can be seen from Table 3, which further

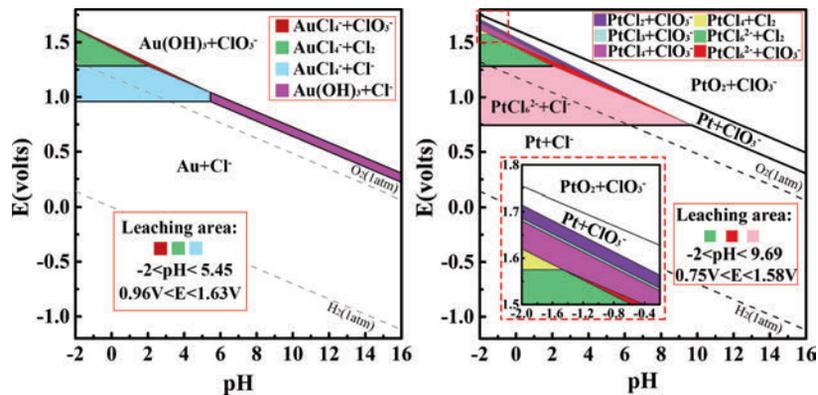


Figure 3. Eh-pH diagrams for (a): Au-Cl-H₂O system at 80 °C (b): Pt-Cl-H₂O system at 80 °C

Table 2. Species amounts used in the calculation for equilibrium composition of 40 g copper anode slimes roasted with H₂SO₄ and NaCl in O₂ (mass fraction, %)

Reagent	Pt	Au	SnO ₂	Cu ₂ S	PbSO ₄
mol	1.92E-07	3.68E-06	4.61E-02	4.02E-02	6.14E-03
Reagent	SbAsO ₃	Ag ₂ SO ₄	NiO	SiO ₂	ZnO
mol	3.31E-02	8.88E-03	2.17E-02	7.14E-03	6.15E-03

increased the phase transition temperature of both PtCl₄ to PtCl₃ and PtCl₃ to PtCl₂, as shown in Table 3 from the viewpoint of chemical equilibrium. In addition, with the temperature increase, the partial pressure of Cl₂ corresponding to the PtCl_x (x=2, 3, 4) stable region increased, which can be seen from the predominance diagram of Pt-Cl-O system at 50 °C, 250 °C, and 450 °C respectively as shown in Fig.4.

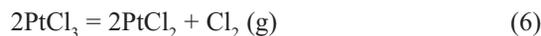
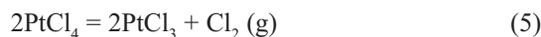
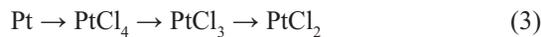


Table 4 shows that the chlorination of Au can only be carried out at a low temperature, and then the AuCl₃ will be transformed to Au completely at temperature higher than 106 °C. Also, as shown in Fig.5, the Au existed stably in a high partial pressure of Cl₂, and the temperature rise increased the upper limit of Cl₂ partial pressure for Au stable region.

The thermodynamic analysis above supports a conclusion that the platinum is more easily to be chlorinated than gold. Furthermore, an experiment was designed to verify this, in which 10 g copper anode slimes mixed with 4 g concentrated sulfuric acid, 0.5 g NaCl, powders of 0.5 g Pt and 0.5 g Au, were roasted at 450 °C for 180 min in O₂ atmosphere

Table 3. Equilibrium composition of Pt-containing phases in Pt-Cl₂ system calculated by Factsage 7.2

T (°C)	P(Cl ₂) (atm)	Pt phase equilibrium composition (mol)		
		PtCl ₄	PtCl ₃	PtCl ₂
NaCl amount: 6.84E-3 mol (NaCl addition: Copper anode slimes = 1%, mass %)				
50	2.06E-03	1.92E-07	0	0
250*	1.79E-03	0	1.92E-07	0
305*	1.61E-03	0	0	1.92E-07
450	9.54E-04	0	0	1.92E-07
500	7.42E-04	0	0	1.92E-07
NaCl amount: 3.42E-02 mol (NaCl addition: Copper anode slimes = 5%, mass %)				
50	1.05E-02	1.92E-07	0	0
283*	9.48E-03	0	1.92E-07	0
352*	8.78E-03	0	0	1.92E-07
450	7.43E-03	0	0	1.92E-07
500	6.62E-03	0	0	1.92E-07

*The temperature data corresponding to the Pt phase transition temperature.

and the X-ray diffraction pattern of the corresponding roasted residue is presented in Fig. 6. The presence of PtCl₂ peak and Au peak in Fig.6 validates the thermodynamic analysis.

During the roasting process, the partial pressures of Cl₂ and O₂ determine the chlorination behavior of Pt and Au [25]. The equilibrium composition of NaCl-H₂SO₄-O₂ system was calculated by Equilib Module in Factsage 7.2 and the results are shown in Fig.7.

Fig.7 (a) indicates that with the increase in NaCl dosage, Cl₂(g), HCl(g), and Na₂SO₄ amounts increase along with the obvious reduction of SO₃(g) at 450°C. Thus, the release of chlorine from NaCl might be carried out through reactions (7) - (9). Fig.7 (b) shows that with increasing H₂SO₄ amount, the Cl₂(g) amount first increased and then decreased slightly while

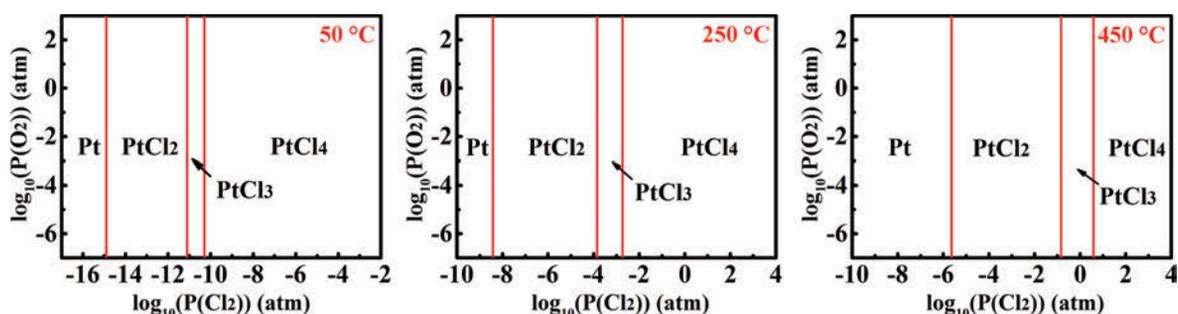


Figure 4. Predominance area diagrams for the Pt-Cl-O system at 50, 250 and 450 °C

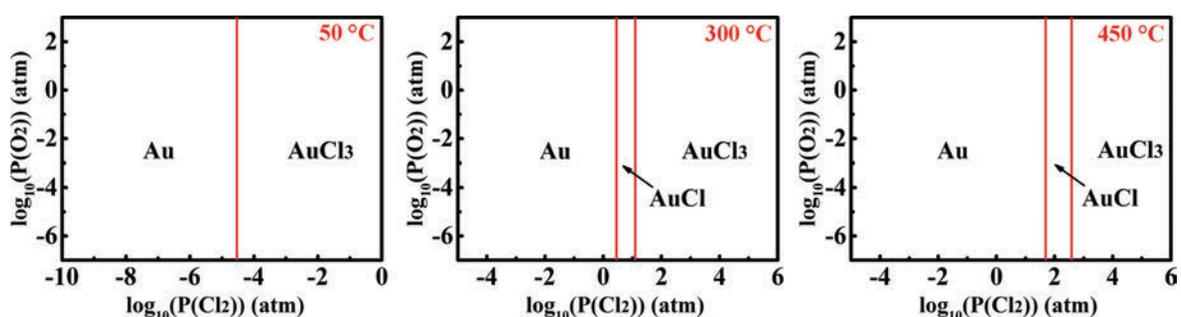


Figure 5. Predominance area diagrams for the Au-Cl-O system at 50, 300 and 450 °C

Table 4. Equilibrium composition of Au-containing phases in Au-Cl₂ system calculated by Factsage 7.2

T (°C)	P(Cl ₂) (atm)	Au phase equilibrium composition (mol)	
		AuCl ₃	Au
50	2.06E-03	3.68E-06	0
106*	2.05E-03	0	3.68E-06
300	1.61E-03	0	3.68E-06
450	9.54E-04	0	3.68E-06
500	7.42E-04	0	3.68E-06

*The temperature data corresponding to the Au phase transition temperature.

coupled with an always increase of HCl (g) amount, which might be due to the translation between HCl (g) and Cl₂ (g) according to Eq. (10). More H₂O (g) could be generated through reaction (7) with more H₂SO₄ added, and consequently the generation of HCl (g) was promoted in reaction (10). Meanwhile, the reaction (10) was a reversible exothermic reaction, due to which more Cl₂ (g) could also be transformed to HCl (g) with the temperature increase as shown in Fig.7 (c). The increase in O₂ (g) amount enhanced Cl₂ (g) formation obviously seen from Fig.7 (d), and the NaCl was almost transferred to Cl₂ (g) and HCl (g) at O₂ (g) amount of 0.2 mol. Increasing O₂ (g) amount further, the amounts of Cl₂ (g) and HCl (g) changed slightly while the partial pressures of them decreased. An appropriate NaCl amount, roasting temperature, and O₂ (g) amount is important for the Cl₂ (g)

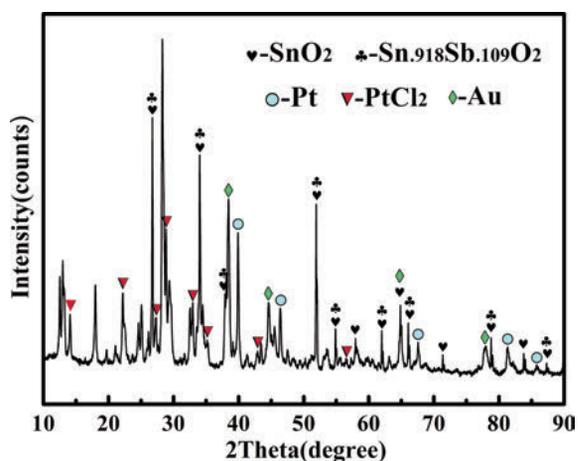
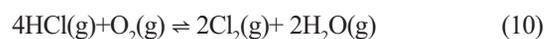
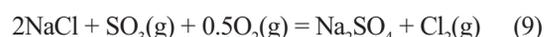
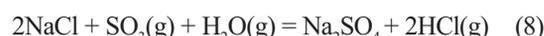


Figure 6. X-ray diffraction pattern of the roasted residue of copper anode slimes mixed with powders of Pt and Au

generation and Pt chlorination.



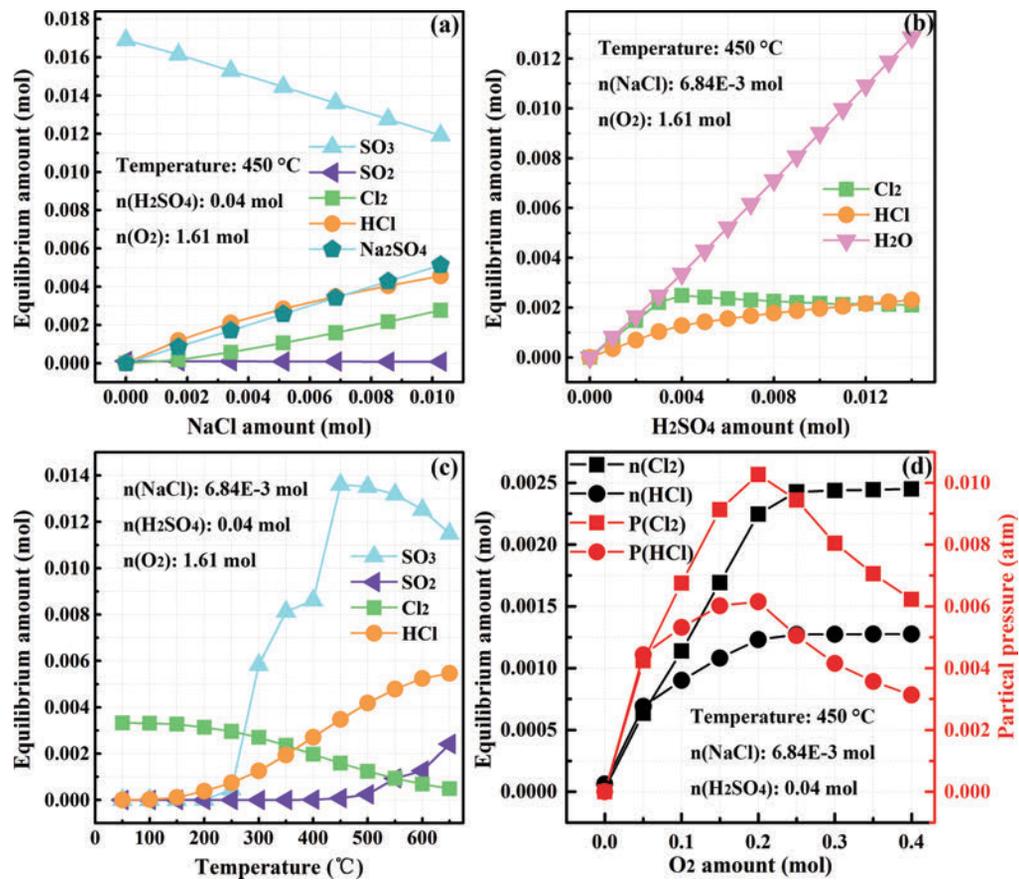


Figure 7. Effects of temperature, NaCl, H₂SO₄ and O₂ amounts on the equilibrium composition of NaCl-H₂SO₄-O₂ system

3.2 Results

This process consists of two main stages, the first one being chlorination roasting and chlorination leaching the second, and the leaching procedure was carried out under the same conditions as described in Section “2.2 Experimental procedures”. The parameters of roasting temperature, oxygen flow rate, addition amounts of concentrated sulfuric acid and sodium chloride were focused to study the effect of the roasting variables on leachability of gold and platinum, and the results are shown in Table 5.

3.3 Discussion

3.3.1 Effect of roasting temperature

First in the blank test, with the copper anode slime being directly treated through a traditional leaching process using the parameters described in Section “2.2 Experimental procedures”, the Pt recovery rate was low and went to 80.72% causing the massive waste of Pt resources.

In Table 5, a significant improvement is observed in the recovery of platinum from 95.09% to 98.37% with the temperature increasing from 400 °C to 450

°C. With the roasting temperature over 450 °C, the recovery rate of platinum decreased. It might be attributed to the decrease of Cl₂ (g) formation and further the Pt chlorination deduced from Fig.7 (c). In addition, the massive volatilization of SO₃ (g) caused by the quick decomposition of concentrated sulfuric acid at higher temperatures also resulted in a small portion of it available to participate in Cl₂ (g) formation. Meanwhile, it is noteworthy that the sintering degree increased at higher temperatures, which can be seen from Fig.8, which slowed down the diffusion of SO₃ (g) and O₂ (g) to the surface of NaCl as well as diffusion of the generated Cl₂ (g) to the surface of platinum, thereby inhibiting the Pt chlorination. Obviously, roasting at 450 °C resulted in a good leaching performance for Pt. In addition, though the Pt can be chlorinated to PtCl₂ at 450 °C based on the result in Fig.6, it cannot be detected in Fig.9 due to the small concentration of Pt in the roasted residue.

3.3.2 Effect of sodium chloride and concentrated sulfuric acid amounts

The data in Table 5 show that the Pt recovery rate



Table 5. Effects of the roasting variables on recovery of gold and platinum

Blank test			Pt recovery (%)		Au recovery (%)	
Direct leaching without roasting			80.72		97.03	
T (°C)	NaCl addition (wt%)	H ₂ SO ₄ addition (wt%)	O ₂ flow rate (mL/min)	Time (min)	Pt recovery (%)	Au recovery (%)
400	1	40	200	120	95.09	98.68
425	1	40	200	120	97.52	98.84
450	1	40	200	120	98.37	98.76
475	1	40	200	120	98.25	98.46
500	1	40	200	120	97.17	98.36
525	1	40	200	120	94.61	98.08
T (°C)	NaCl addition (wt%)	H ₂ SO ₄ addition (wt%)	O ₂ flow rate (mL/min)	Time (min)	Pt recovery (%)	Au recovery (%)
450	0	40	200	120	93.85	98.63
450	0.5	40	200	120	96.56	98.67
450	0.75	40	200	120	97.63	98.27
450	1	40	200	120	98.37	98.76
450	1.25	40	200	120	97.7	98.44
450	1.5	40	200	120	96.52	98.32
T (°C)	NaCl addition (wt%)	H ₂ SO ₄ addition (wt%)	O ₂ flow rate (mL/min)	Time (min)	Pt recovery (%)	Au recovery (%)
450	1	30	200	120	97.1	98.69
450	1	35	200	120	97.8	98.78
450	1	40	200	120	98.37	98.76
450	1	45	200	120	96.87	98.58
450	1	50	200	120	96.26	98.65
450	1	55	200	120	96.1	98.76
T (°C)	NaCl addition (wt%)	H ₂ SO ₄ addition (wt%)	O ₂ flow rate (mL/min)	Time (min)	Pt recovery (%)	Au recovery (%)
450	1	40	50	120	94.73	98.99
450	1	40	100	120	94.88	99.22
450	1	40	150	120	95.76	98.66
450	1	40	200	120	98.37	98.76
450	1	40	250	120	97.93	99.1
450	1	40	300	120	96.57	98.71

increased obviously from 93.85% to 98.37% with NaCl amount from 0 to 1% when the samples were roasted at 450 °C. Then with NaCl amount beyond 1%, the sample would be sintered (Fig.10) and the gas-solid chlorination of Pt would be hindered, as a result of which the Pt recovery rate decreased slightly. In Fig.11, the foundation of Ag₂SO₄ peak in the X-ray diffraction pattern of the roasted residue at 1.5% NaCl addition indicates the lower level of chlorination of the sample compared to that with 1% NaCl added, which can be seen from Fig.9. In Fig.9, the Ag₂SO₄ in the copper anode slime was transformed to AgCl. Thus, the suitable sodium

chloride amount is 1 %.

The concentrated sulfuric acid had a complicated effect on the Pt chlorination. With its amount ranging from 30 % to 40 %, the Pt recovery increased, as shown in Table 5. However, more H₂O (g) would be generated through reaction (7) when more concentrated sulfuric acid was added, and the formation of Cl₂ (g) might be decreased according to reaction (10) and Fig.7 (b), as a result of which the Pt chlorination and recovery decreased (Table 5). The concentrated sulfuric acid amount should be fixed at 40 %.



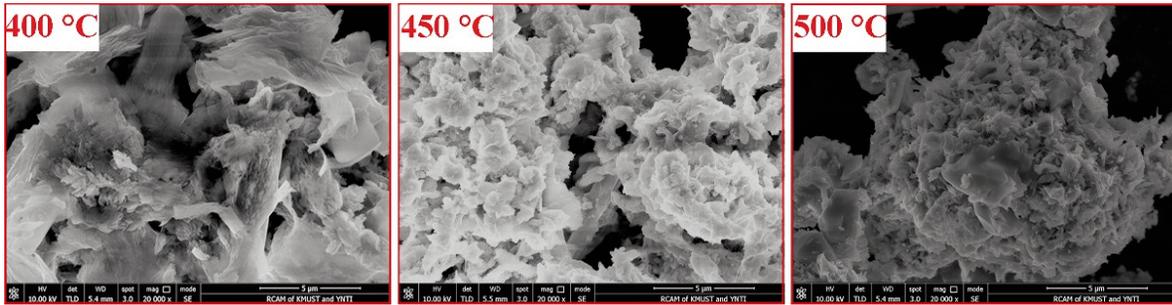


Figure 8. SEM images of roasted residues with additions of 1% NaCl and 40% H₂SO₄ at different temperatures

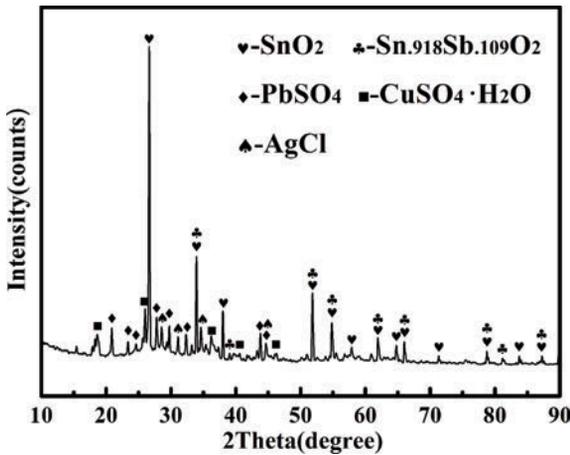


Figure 9. X-ray diffraction pattern of roasted residue with additions of 1% NaCl and 40% H₂SO₄ at 450 °C

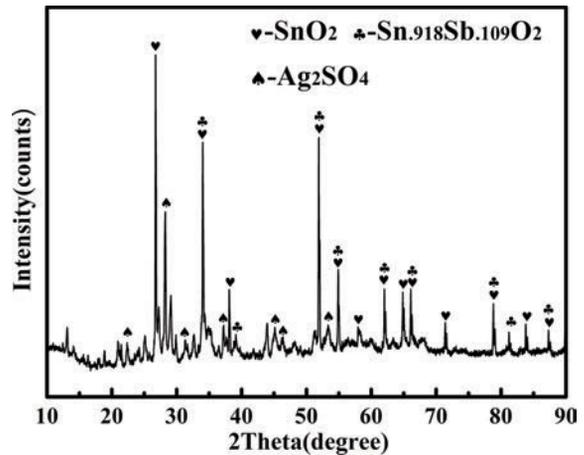


Figure 11. SEM images of the roasted residues with additions of 1% NaCl and 1.5% NaCl respectively at 450 °C and 40% H₂SO₄

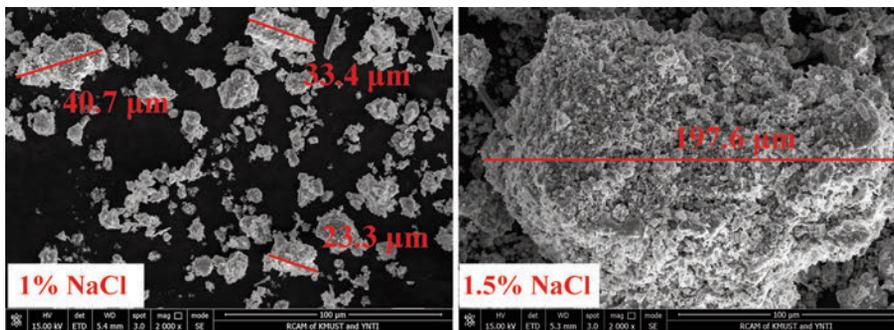


Figure 10. X-ray diffraction pattern of the roasted residue with 1.5% sodium chloride addition at 450 °C and 40% H₂SO₄

3.3.3 Effect of oxygen flow rate

With the copper anode slimes roasted with 1% NaCl and 40% concentrated sulfuric acid at 450 °C, more Cl₂ (g) could be formed with the O₂ flow rate increasing from 50 to 200 mL/min, and correspondingly the recovery rate of Pt increased from 94.73% to 98.37%. However, it then decreased slightly with a further increase of O₂ flow rate, which might be attributed to the decrease in Cl₂ partial pressure at a higher O₂ flow rate obtained from Fig.7 (d) which goes against the Pt chlorination.

Besides, this chlorination roasting process had little effect on the Au recovery based on the results in Table 5, which is related to the fact that it is difficult to chlorinate Au under the roasting conditions and it exists stably in the form of metallic Au after roasted confirmed by Fig.6.

4. Conclusions

The effective recovery of Pt and Au from the copper anode slime through the process of chlorination roasting first and chlorination leaching



followed is feasible. The factors, including roasting temperature, oxygen flow rate, sodium chloride and concentrated sulfuric acid amounts play important roles on the Pt recovery, while they have little effect on the Au recovery due to its difficulty to be chlorinated.

The thermodynamic analysis shows that the chlorination of platinum proceeds through a progressive phase transformation process presented as $\text{Pt} \rightarrow \text{PtCl}_4 \rightarrow \text{PtCl}_3 \rightarrow \text{PtCl}_2$ with the temperature increase, and the transition temperature increases with the increase of Cl_2 partial pressure. With the copper anode slime being directly treated through a traditional leaching process, the Pt recovery rate is low and goes to 80.72%. After roasted with sodium chloride and concentrated sulfuric acid in oxygen atmosphere, the recovery rate of Pt is increased to a value around 95%. However, with excessive concentrated sulfuric acid added, a massive amount of $\text{H}_2\text{O}(\text{g})$ is generated and the formation of $\text{Cl}_2(\text{g})$ is decreased due to the transition from $\text{HCl}(\text{g})$ and $\text{Cl}_2(\text{g})$, as a result of which the Pt recovery rate decreases. Besides, the excessive addition of sodium chloride leads to the sample sintered and the Pt recovery rate also decreases.

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EKSTRAKCIJA PLATINE I ZLATA IZ ANODNOG MULJA POSTUPKOM HLORIZACIJE PRŽENJEM I HLORIZACIJE LUŽENJEM

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Apstrakt

U ovom radu je predstavljen novi postupak hlorizacije prženjem, nakon kojeg sledi hlorizacija luženjem, a koji se koristi za ekstrakciju platine i zlata iz anodnog mulja. Rezultati termodinamičke analize i eksperimentalnog dela istraživanja su pokazali da je platina tokom postupka hlorizacije prešla u $PtCl_2$, a da je zlato nađeno u svom metalnom obliku prilikom postupka luženja. Tokom postupka direktnog luženja anodnog mulja, količina dobijene platine je bila mala i iznosila je 80,72%. Nakon postupka prženja u prisustvu natrijum hlorida i koncentrovane sumporne kiseline u atmosferi sa kiseonikom, stopa dobijene Pt se povećala na oko 95%. Pored toga, prekomernim dodavanjem koncentrovane sumporne kiseline, generisano je više H_2O (g), a formiranje Cl_2 (g) se smanjilo zbog prelaska HCl (g) i Cl_2 (g), što je dovelo do smanjenja stope dobijene Pt. Postupak hlorizacije prženjem je, takođe, imao mali uticaj na stopu dobijenog Au zbog činjenice da se zlato veoma teško podvrgava postupku hlorizacije.

Ključne reči: Anodni mulj; Stopa dobijene platine i zlata; Hlorizacija prženjem; Hlorizacija luženjem

