Int. J. Pure Appl. Sci. 1: 1-8 (2015)

IJPAS
ijpas@tunceli.edu.tr
ISSN: 2149-0910

Research/Araştırma

Copper Recovery from Roasted Chalcopyrite Concentrate by Using Hydrochloric Acid or Water Leaching Method

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Abstract

In this study, the sulphation roasting in atmospheric medium of chalcopyrite concentrate and hydrochloric acid or water leaching of roasted chalcopyrite concentrate were investigated. The effects of roasting temperature (200-700 ^{o}C) and roasting time (15-120 min.) on recovery of copper were also researched. The optimum roasting temperature and roasting time were determined to be 450 ^{o}C and 60 min., respectively. The roasted chalcopyrite concentrate is leached with hydrochloric acid solution or water for recovery of copper. It was shown that the copper extractions from roasted chalcopyrite concentrate are high as 85-90%, while recoveries of iron were are only 6-20%. The leaching conditions with hydrochloric acid solution or water for roasted chalcopyrite concentrate were selected as follows: liquid/solid (v/w) ratio, 10 mL/g; leaching temperature, ambiance temperature; leaching time, 15 min; concentration of hydrochloric acid solution, 0.1 N. Under these conditions, for roasting at 450 ^{o}C, recoveries of copper and iron with hydrochloric acid leaching were 89.7% and 20.1%, respectively. On the other hand, copper and iron extraction yields with water leaching were 86.8% and 13.5%, respectively. For all conditions, it was determined that the amounts of copper obtained from hydrochloric acid leaching are higher than values obtained from water leaching.

Keywords: Chalcopyrite concentrate, Hydrochloric acid leaching, Roasting, Sulphation, Water leaching.

Hidroklorik Asit veya Su Liçi Yöntemi Kullanarak Kavrulmuş Kalkopirit Konsantresinden Bakır Kazanılması

Özet

Bu çalışmada, kalkopirit konsantresinin atmosferik ortamda sülfatlama kavurması ve kavrulmuş kalkopirit konsantresinin hidroklorik asit veya su liçi incelendi.

Research/Araştırma

Bakır kazanılması üzerine kavurma sıcaklığı (200-700 °C) ve kavurma zamanının (15-120 dk) etkileri de araştırıldı. En uygun kavurma sıcaklığı ve kavurma süresi, sırasıyla, 450 °C ve 60 dk olarak belirlendi. Kavrulmuş kalkopirit konsantresi, bakırın kazanılması için, hidroklorik asit çözeltisi veya su ile liç edildi. Kavrulmuş kalkopirit konsantresinden sadece % 6-20 demir kazanımı olurken % 85-90 gibi yüksek bakır ekstraksiyonu olduğu görüldü. Kavrulmuş kalkopirit konsantresi için hidroklorik asit çözeltisi ve su ile liç şartları, müteakip şekilde seçildi: sıvı/katı (v/w) oranı, 10 mL/g; liç sıcaklığı: ortam sıcaklığı; liç süresi: 15 dakika; hidroklorik asit derişimi: 0.1 N. Bu şartlar altında, 450 °C'deki kavurma için, hidroklorik asit liçi ile bakır ve demir kazanımları, sırasıyla, % 89.7 ve % 20.1'dir. Diğer taraftan, su liçi ile bakır ve demir ekstraksiyon miktarları, sırasıyla, % 86.8 ve % 13.5'tir. Tüm şartlar için, hidroklorik asit liçinden elde edilen bakır miktarlarının su liçinden elde edilen değerlerden daha yüksek olduğu belirlendi.

Anahtar Kelimeler: Hidroklorik asit liçi, Kalkopirit konsantresi, Kavurma, Sülfatlama, Su liçi.

INTRODUCTION

Most of the copper in the world is produced from low-grade sulphidic ores. Chalcopyrite ($CuFeS_2$), the most common ore of copper, is usually processed by the pyrometallurgical routes after a beneficiation by flotation of ore to a concentrate. An alternative to conventional pyrometallurgical methods to solve the problem of high SO_2 emission is to use radically different process chemistry to permit the production of copper metal without or minimum SO_2 generation (Riveros et al., 2004). Thus, in search of alternative processes, several hydrometallurgical and pyro-hydrometallurgical routes have been attempted in the recent past (Prasad and Pandey, 1998).

Hydrometallurgical extraction of copper from chalcopyrite is considered to be both more economical and environmentally friendly than pyrometallurgical extraction, especially when the copper-bearing sulfide minerals are present at low grade in the ore (Pradhan et al., 2008). But, due to compact structure of chalcopyrite, leaching of chalcopyrite is very difficult. For that reason, the leaching behaviors of chalcopyrite were investigated in numberless leaching processes (Hirato et al., 1987; Hackl et al., 1995; Lu et al., 2000; Adebayo et al., 2003; Ruiz et al., 2011; Cai et al., 2012; Solis-Marcial and Lapidus, 2013; Lu and Dreisinger, 2013; Qian et al., 2014; Solis-Marcial and Lapidus, 2014) and bioleaching processes

(Sandström et al., 2005; Watling, 2006; Gericke et al., 2010). On the other hand, some commercial hydrometallurgical processes such as Sherrit-Cominco, Cymet, Cyprus, Clear, Arbiter (Prasad and Pandey, 1998) and HydroCopper (Hyvärinen and Hämäläinen, 2005) processes have been developed for copper production from chalcopyrite concentrate.

Some researchers have been investigated chalcopyrite oxidative roasting parameters and mechanism (Bayer and Wiedemann, 1992; Zivkovic et al., 1996; Chakravarthy and Srikanth, 2000). In this study, the sulphation roasting in atmospheric medium of chalcopyrite concentrate and hydrochloric acid or water leaching of roasted chalcopyrite concentrate were investigated.

MATERIAL AND METHOD

Material

The chalcopyrite concentrate used in this study was from Cengiz Holding of Turkey. The chemical composition of chalcopyrite concentrate for - 100 mesh is given in Table 1.

Method

5±0.05 grams of concentrated chalcopyrite were placed in porcelain capsules (12 cm in diameter and 3 cm in height) and concentrated chalcopyrite samples were spread into capsules. Then, these capsules were placed in muffle furnace. Each sample was roasted at different times and temperature conditions. Roasted chalcopyrite concentrates were cooled in a desiccator until room temperature. These roasted samples were weighed.

Table 1. The chemical composition of concentrated chalcopyrite.

Constituents	Composition, w/w
Co	1250 mg/kg
Cu	18.24%
Fe	34.71%
Ni	670 mg/kg
S	31.82%
Zn	1690 mg/kg
LOI* (80-900°C)	13.53%

*Lost of Ignition

The mass differences were calculated using Eq (1):

$$m = \frac{(m_b - m_a)}{m_b} * 100 \qquad (1)$$

Where, m is mass difference (%), m_b is weigh of concentrated chalcopyrite before roasting (g) and m_a is weigh of concentrated chalcopyrite after roasting (g).

Distilled water and 0.1 N HCl solution were used to be leaching solution for recovery of copper from roasted chalcopyrite concentrates. All leaching experiments were carried out in 150 mL glass conical flasks on a stirrer at ambience temperature (about 20 ^{o}C). Roasted chalcopyrite concentrates in room temperature were leached under the condition of liquid/solid ratio of 10 mL/g and leaching time of 15 min at 300 rpm. After a predetermined period, reaction mixtures were separated by filter papers. Then, volume and pH of the supernatants were measured. The supernatants were analyzed for Cu and Fe by an atomic absorption spectrometer (Perkin Elmer, Analyst 800).

Analytical Methods

The composition of chalcopyrite concentrate was determined. For this purpose, chalcopyrite concentrate sample were dissolved in the microwave unit and then, obtained solution were analyzed by an atomic absorption spectrometer.

RESULTS AND DISCUSSIONS

The effect of Roasting Temperature

The chalcopyrite concentrate samples were roasted at 200-700 ^{o}C and then, roasted chalcopyrite concentrate samples were leached with 0.1. N hydrochloric acid solution or distilled water. The effects of roasting temperature on amounts of mass difference and metal recoveries with hydrochloric acid or water leaching for 200-700 ^{o}C are shown in Figure 1.

In this study, the sulphation roasting in atmospheric medium of chalcopyrite concentrate and hydrochloric acid or water leaching of roasted chalcopyrite concentrate were investigated. Sokic at al. (2008) studied the TG-DTA of chalcopyrite in a stream of air at 300-800 ^{o}C and reported heating reaction of chalcopyrite from 300 ^{o}C to 800 ^{o}C. These reactions are as follows:

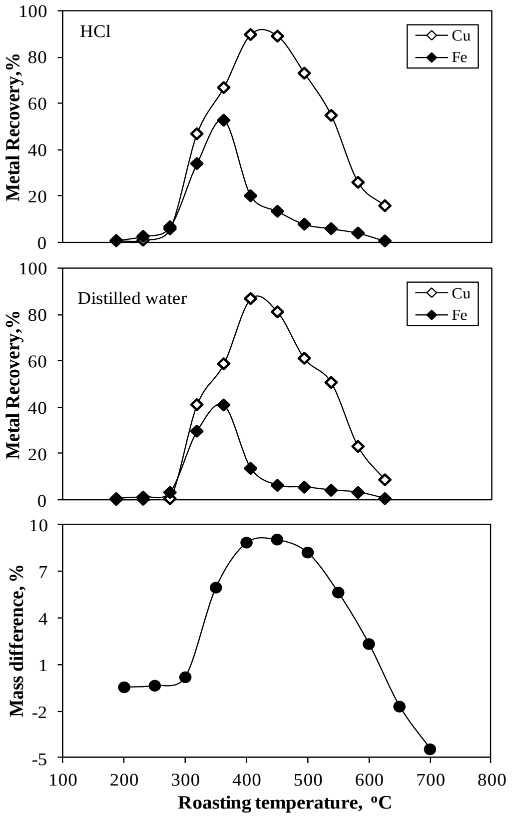

Figure 1. The effects of roasting temperature on amounts of mass difference and metal recoveries (amount of chalcopyrite sample: 5 g, roasting time: 60 min., concentration of hydrochloric acid solution: 0.1 N, leaching time: 15 min, agitation speed of leaching: 300 rpm,

liquid/solid ration: 10 mL/g).

$2CuFeS_2(s) + O_2(g) \rightarrow Cu_2S(s) + 2FeS(s) + SO_2(g)$ (330-357 °C) (3)

$2CuFeS_2(s) + O_2(g) \rightarrow Cu_2S(s) + 2FeS(s) + SO_2(g)$ (404-440 °C) (4)

$FeS(s) + 2O_2(g) \rightarrow FeSO_4(s)$ (502-555°C) (5)

$Cu_2S(s) + SO_2(g) + 3O_2(g) \rightarrow 2CuSO_4(s)$ (6)

$2Cu_2S(s) + 5O_2(g) \rightarrow 2CuO{\cdot}CuSO_4(s)$ (7)

$2FeSO_4(s) \rightarrow Fe_2O_3(s) + SO_2(g) + SO_3(g)$ (555-613 °C) (8)

$CuSO_4(s) \rightarrow CuO(s) + SO_3(g)$ (Above 750 °C) (9)

$CuO{\cdot}CuSO_4(s) \rightarrow 2CuO(s) + SO_3(g)$ (10)

According to Eq. 5-7, The formation of $CuSO_4$ and $FeSO_4$ leads to mass increase. For this purpose, in this study, the formations of $CuSO_4$ were followed mass differences.

It was seen in Fig. 1 that mass gain is maximum value at 450 °C. The mass differences are negative values below 300 °C and above 600 °C. Further, the maximum copper recovery was obtained from leaching with hydrochloric acid solution of chalcopyrite concentrate which roasted at 450 °C. The recovery of Cu and Fe with hydrochloric acid leaching were 89.7 % and 20.1 %, respectively. The Cu and Fe extraction yields with water leaching were 86.8 % and 13.5 %, respectively. The recoveries of copper for water and hydrochloric acid solution leaching were near. Iron is undesirable compound in pregnant solutions due to it causes several problems while copper is obtained to be metallic copper by cementation. The iron recovery with water leaching was smaller than iron recovery obtained from acid leaching. Therefore, water leaching of roasted chalcopyrite concentrate is preferable. On the other hand, the copper extraction yields were also very high at 500 °C.

The Effect of Roasting Time

The effects of roasting time on amounts of mass difference and metal recoveries at 450 °C are seen in Fig. 2. According to Fig. 2, the mass rapidly increased until 45 min. of roasting time, after 45 min., the increasing of mass difference slowed down. The copper extraction yields do not significantly change after 60 min of roasting time.

The effects of roasting time on amounts of mass difference and metal recoveries at 500 °C are seen in Fig. 3. According to Fig. 3, until 45 min. of roasting time, the mass differences increased, but after 60 min. of roasting time, the mass differences do not change. The iron extraction yields obtained with water leaching of roasted chalcopyrite concentrate at 500 °C was very small as 6.2 %. Under these conditions, the Cu extraction yield was 86.8 %.

Conclusions

1. The application of oxidative roasting of chalcopyrite concentrate in

atmospheric medium is high efficiency method.

2. The mass gain is maximum value at 450 °C for 60 min. of roasting time. The mass differences are negative values below 300 °C and above 600 °C.

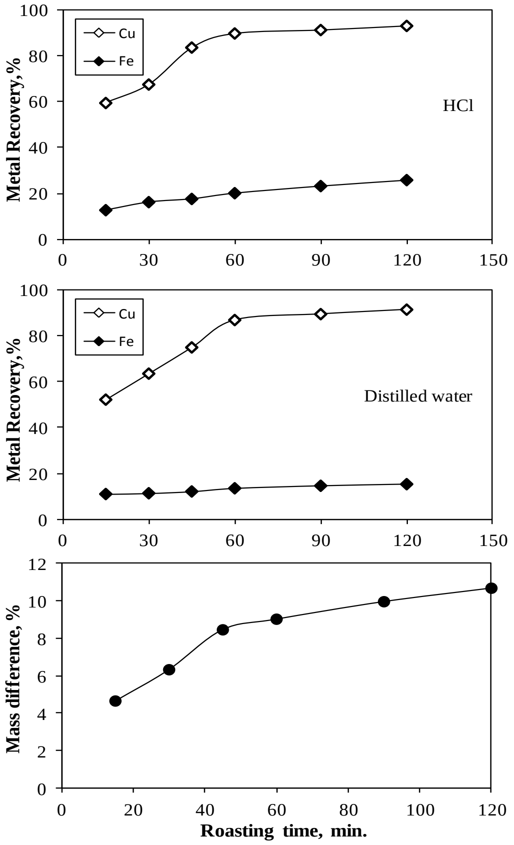

Figure 2. The effects of roasting time on amounts of mass difference and metal recoveries at 450 °C (amount of chalcopyrite sample: 5 g, leaching time: 15 min, concentration of hydrochloric acid solution: 0.1 N agitation speed of leaching: 300 rpm, liquid/solid ration: 10 mL/g).

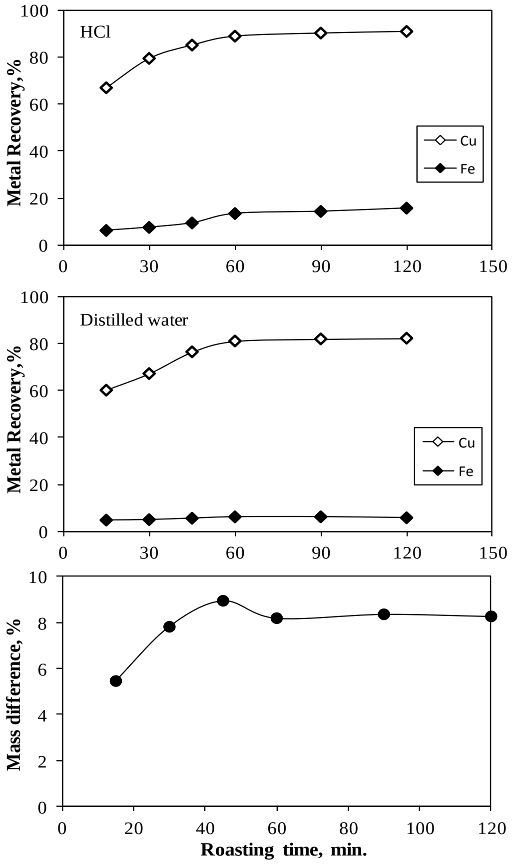

Figure 3. The effects of roasting time on amounts of mass difference and metal recoveries at 500 °C (amount of chalcopyrite sample: 5 g, leaching time: 15 min, concentration of hydrochloric acid solution: 0.1 N agitation speed of leaching: 300 rpm, liquid/solid ration: 10 mL/g).

3. The maximum copper recovery was obtained from leaching with hydrochloric acid of chalcopyrite concentrate which roasted at 450 °C for 60 min. of roasting time. The recoveries of Cu and Fe with hydrochloric acid leaching are 89.7 % and 20.1 %, respectively.

4. At 450 °C, he mass is rapidly increased until 45 min. of roasting time, after 45 min., the increasing of mass difference slows down.

The iron extraction yields obtained with water leaching of roasted chalcopyrite concentrate at 500 °C is very small as 6.2 %. Under these conditions, the Cu extraction yield is 86.8 %.

Acknowledgments

The author is grateful to Tunceli University, Unit of Scientific Research Projects (Project No: MFTUB13-21) for the financial support of this work.

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