



Ammoniacal thiosulfate leaching of pressure oxidized sulfide gold concentrate with low reagent consumption



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ABSTRACT

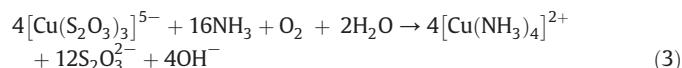
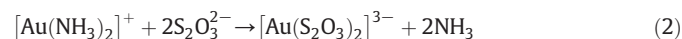
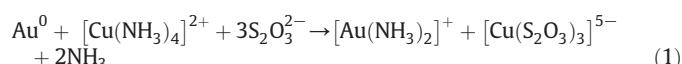
Ammoniacal thiosulfate leaching of pressure oxidized sulfide gold concentrate received from a gold mine is studied. The main chemical composition of the concentrate was Au 32 g/t, Ag 12 g/t, Fe 59%, S 21% and As 19%. The experimental variables studied were ammonia concentration, thiosulfate concentration, copper concentration, and temperature. The highest gold extraction (89%) was achieved with the following leaching solutions: 0.2 mol S₂O₃²⁻/L, 0.2 mol NH₃/L, and 0.1 g Cu/L; and with the following conditions: solids, 30 m-%; pressure, 1 atm; continuous aeration (0.2 L/min); temperature, 30 °C; and leaching time, 6 h. Measured thiosulfate and copper concentrations after 6 h of leaching were 0.19 M and 0.08 g/L, respectively. The results from the leaching experiments showed that the pressure oxidized gold concentrate was effectively leached in an ammoniacal thiosulfate leaching solution with minimal reagent consumption. The study shows that the widely reported problem of high reagent consumption can be circumvented in leaching of pressure oxidized sulfide gold concentrate by the use of low reagent concentrations. The results indicate good premises for gold recovery by ion-exchange resins, since minimal thiosulfate consumption also ensures low polythionate concentration, which is an important issue in gold recovery with ion-exchange resins. The minimal reagent consumption also indicates that there are good possibilities to circulate the leaching solution for re-use in the leaching stage after gold recovery and solid–liquid separation.

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1. Introduction

The undesirable environmental and safety aspects of the prevailing cyanidation techniques used in gold production have driven study in the area towards a search for alternative lixivants. The use of thiosulfate in the presence of ammonia and copper(II) has been considered to be a suitable alternative to cyanidation techniques, and extensive research work has been carried out in this area and comprehensive reviews were presented (Aylmore and Muir, 2001; Grosse et al., 2003; Muir and Aylmore, 2004). However, widespread commercialization of thiosulfate processes has not yet been achieved. The major problems facing thiosulfate processes are the high consumption of thiosulfate and the challenges posed by the recovery of the gold after the leaching stage (Muir and Aylmore, 2004; Hilson and Monhemius, 2006). One of the challenges to evaluation and adoption of the thiosulfate process is the complicated chemistry of the leaching system, resulting from the simultaneous presence of complexing ligands (ammonia and thiosulfate), the stability of thiosulfate in the solution, and the Cu(II)–Cu(I) redox couple. The reactions that describe the leaching behavior of gold in an

ammoniacal thiosulfate system are shown below (Eqs. (1)–(4)) (Grosse et al., 2003).



The net reaction is:



Muir and Aylmore (2004) summarized published studies on thiosulfate leaching of various ores and noted that a wide range of conditions and concentrations of reagents have been used. Emphasis has been on carbonaceous ores that give poor gold recoveries using cyanide and copper–gold ores. Gold dissolution rates and recovery percentages have varied considerably, depending on the ore type and conditions

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used. Navarro et al. (2002) found that under mild conditions, thiosulfate leaching compared very favorably with cyanidation when leaching gold from flotation concentrate was studied. Muir and Aylmore (2004) studied some relatively simple Australian oxide ores and presented that while most of the gold and silver were readily leached in 2–4 h, a fraction of the cyanide-soluble gold always remained in the thiosulfate leach residue. Langhans et al. (1992) studied low grade oxidized gold ore and the results from their study indicate that copper-catalyzed thiosulfate leaching may be competitive with conventional cyanidation methods for application to heap, dump, or in-situ leaching techniques. Barrick Gold Corporation (Barrick) patented a process for recovering gold from an ore having a combination of refractory sulfide and carbonaceous material using pressure oxidation followed by thiosulfate leaching (Marchbank, 1996). Barrick subsequently patented a combined pressure oxidation, thiosulfate and resin-in-pulp process for treatment of refractory gold ores (Thomas, 1998). The patent deposited by Barrick (Thomas, 1998) included a study in which pressure-leached arsenopyrite ores from Barrick were examined and a large number of stirred tank tests carried out at pH 8–9. Gold recovery in their study varied from 50% to 95%. The work by Feng and van Deventer (2010) showed that oxidative pre-treatment of sulfide ore followed by thiosulfate leaching provided an efficient approach for the treatment of sulfide containing ore. Feng and van Deventer (2010) showed that consumption of thiosulfate in leaching was negligible after an over 1 h oxidative ammonia pre-treatment.

Muir and Aylmore (2004) stated that industry's first task in adoption of the thiosulfate system is to develop a robust overall process flowsheet that can be realistically used and costed for a range of actual ores. The efficiency of the leaching stage has a primary effect on the technical and economic success of a hydrometallurgical business (Crundwell, 2013), so it is evident that managing the leaching stage is a key issue when developing the thiosulfate system. Clearly, there is thus a need for leaching studies with different ore types and pre-treatments that can provide comprehensive data about optimal conditions and that evaluate gold recovery, reagent re-cycling or destruction, and impurity control possibilities. Efficient pre-treatment of ores/concentrates coupled with optimal leaching conditions could offer a way to overcome the problems hitherto identified in the leaching stage, as well as facilitate the following recovery stage. This work studies the leaching of pressure oxidized gold concentrate in thiosulfate solution in the presence of ammonia and copper(II). The aim of the present study is to find optimal leaching conditions for pressure oxidized sulfide gold concentrates by studying the effect of different parameters (reagent concentrations and temperature).

2. Experimental work

2.1. Materials

Pressure oxidized gold concentrate was received from a gold mine. The gold mine processes the ore to concentrate by crushing, grinding and flotation. After flotation, the concentrate consists mainly of pyrite and arsenopyrite. The concentrate from the flotation is subjected to acidic pressure oxidation in an autoclave, where sulfide minerals are oxidized to sulfate. The gold concentrate used in the experiments was taken from the process stream after pressure oxidation. The received gold concentrate was dried at 60 °C for 12–24 h. The dried concentrate was blended and divided into sub-samples, which were divided to representative samples for experimental use by a rotary splitting device. Samples were then collected for chemical analysis and size distribution characterization. Table 1 shows the elemental composition of the concentrate. Gold content was determined by fire-assay followed by ICP-OES and other elements were determined by acid digestion followed by ICP-OES. The particle size distribution of the gold concentrate was analyzed with a LS 13 320 Laser Diffraction Particle Size Analyzer. Mean particle size and particle size distribution are presented in

Table 1
Chemical composition of the pressure oxidized concentrate.

Element	Amount
Fe, mass-%	59.1
S, mass-%	21.3
As, mass-%	19.2
Au, g/t	32
Ag, g/t	12
Cu, g/t	137
Cd, g/t	3
Co, g/t	30
Cr, g/t	69
Mn, g/t	106
Mo, g/t	3
Ni, g/t	63
Pb, g/t	84
Sb, g/t	514
Zn, g/t	104

Table 2. All the chemicals used were of analytical grade and ultrapure water was used in the preparation of the solutions. Thiosulfate was added as sodium thiosulfate, copper as copper sulfate, and ammonia as ammonia water (25%) purchased from Merck.

2.2. Analytical techniques

Metal concentrations (Au, Cu) in the solutions were analyzed by atomic absorption spectroscopy (Thermo Scientific iCE 3000 Series AA spectrophotometer). Samples for gold analyses were first oxidized by hydrogen peroxide to oxidize the sulfur species to stable sulfates. After oxidation, gold sample solutions were acidified by aqua regia and boiled to ensure complete conversion of the metal species to their chloride form. The thiosulfate concentration was determined by ion chromatography (Thermo Scientific DIONEX ICS-1100 Ion Chromatography System). An IonPac AS22 analytical column and IonPac AG22 guard column were used for the analyses. Rosemount Analytical PERph-X high temperature sensors were used to measure mixed solution potential (3300FT-12-30) and pH (3300HT-10-30). All potentials are given herein with respect to the Standard Hydrogen Electrode (SHE). Sensors were hooked up with a Rosemount Analytical Dual Input Analyzer. Dissolved oxygen in the solution was measured with a Marvet Fluids AJ90 analyzer.

2.3. Leaching tests

Leaching tests of the pressure oxidized concentrate were performed in a 1.1 L reactor ($d = 8.4$ cm) as batch experiments in which a total volume of 0.85 L of slurry was used. The reactor was equipped with an overhead mixer (Heidolph RZR 2041) and 4 baffles. A double-pitch-bladed turbine (4 cm diameter) was used for mixing. The stirring speed was maintained at 700 min^{-1} . The temperature in the reactor was controlled by a thermostatic oil bath. The gas was fed from the bottom of the reactor. The flow rate of the gas was controlled with a gas flow control valve (F-201 CV) hooked up with a control unit (Bronkhorst hi-tec, Type E-7100). The solution was first stabilized to

Table 2
Mean particle size and particle size distributions of the concentrate particles.

Diameter	μm
Mean	7.13
<10%	1.29
<25%	2.64
<50%	5.45
<75%	9.76
<90%	16.16

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