

University of Nevada

Reno

"Studies on Thiourea Leaching of Finely-Divided
Gold Ores"

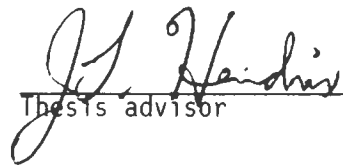
A thesis submitted in partial fulfillment of the
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in Metallurgical Engineering

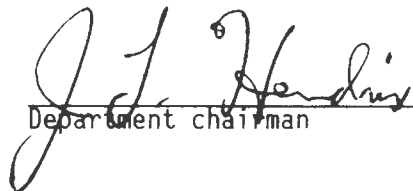
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ABSTRACT

Laboratory leaching experiments were conducted to ascertain the effectiveness of acidic solutions of thiourea, $CS(NH_2)_2$, for dissolving gold from finely disseminated gold ores. The ores tested were from the Chipmunk orebody of the Northumberland mining district in Nye County, Nevada, USA, and carbonaceous ore from the Carlin orebody in Eureka County, Nevada, USA.

Extractions of up to 90 percent were attained from the Chipmunk ore. Extraction was shown to be dependent on thiourea concentration, ferric sulfate oxidant concentration, acid concentration, and leaching time. Optimum conditions were an ambient temperature leach near pH 1.0 utilizing 100-200 kg/t thiourea and 5 kg/t ferric sulfate for 8 to 24 hours.

The Carlin carbonaceous ore proved to be more amenable to leaching with thiourea solutions than standard cyanidation. Recoveries of up to 60 percent were attained compared to 3 percent with cyanide treatment. Behavior was similar to that of Chipmunk ore.

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INTRODUCTION

As research into hydrometallurgical extraction and purification continues, many novel processes are introduced to the mining/metals industry. For various reasons, the vast majority of these processes remain on the laboratory shelf. One process which shows promise for implementation is the use of thiourea, $CS(NH_2)_2$ as a complexing/extracting agent for gold and other precious metals [1]. Laboratory testing of this reagent has indicated that a thiourea process for gold extraction has several advantages over conventional processing techniques.

The use of cyanide for the extraction of gold from ores and concentrates is the predominant treatment method for primary recovery of this metal throughout the world. Indeed, cyanidation has proven to be very effective and economical for processing many types of gold-bearing materials. However, certain characteristics and disadvantages of the cyanidation process warrant the investigation and development of alternative gold extractant systems.

The major concerns involving the cyanidation process are the health and environmental problems associated with it. These include the possible formation of deadly hydrogen cyanide gas, the ingestion or absorption through the skin of cyanide salts [2], and the formation of the free forms of cyanide (HCN and CN^-) in effluent waters which are toxic to aquatic life at concentrations higher than 0.04 mg/L [3]. The United States government recommended standards at present are 11 mg HCN/m^3 of air in the workplace [2] and 0.001 mg/L total cyanide in effluent waters for protection of aquatic life and wild life [4]. The

latter value was under revision as of June 1979 [4].

In light of increasing environmental awareness and responsibility, it is possible that continued stiffening of environmental and health standards will occur. Certainly, increased monitoring of air and water quality can be expected; while some states, such as California, may possibly ban new cyanide processing altogether. This does not even begin to address the future problems (and potential liabilities) related to cyanide heap and dump leaches, which may release significant amounts of cyanide species into the environment long after metal production at the site has ceased.

Thiourea, on the other hand, is not restricted by the same toxicity factors as cyanide. Thiourea toxicity data indicate high threshold limit values for mammals [5] and a lethal dose of 10g/kg for humans [6]. On this basis thiourea is to be considered much safer than cyanide. Indeed, the low pH required in a thiourea process would probably be of more concern. However, it must be mentioned that thiourea has been shown to be a carcinogen to rats and possibly to trout [7]. Even so, thiourea has been used for years in the treatment of thyroid diseases in humans and at this time is considered to be non-carcinogenic to humans [7,8]. Recent literature discussing thiourea processing as a potential replacement for cyanidation fails to mention the possibility of thiourea being a carcinogen [9,10,11,12]. It is not known what bearing these facts will have on the future establishment of a plant processing gold with thiourea; however, thiourea has long been used in the United States in the photography and photo-copying paper industries as well as in various organic syntheses [7].

Another aspect of cyanidation which can be a problem is the somewhat slow kinetics of dissolution of gold by the cyanide species [9,13,14]. This can especially be a problem in certain vat and heap leaching operations which process ores containing fairly large-grained gold particles. Extended leaching periods required to obtain acceptable tailings translate directly into higher overall operating costs, thus reducing profitability.

The kinetics of thiourea dissolution of gold have been studied extensively. Initial interest in thiourea as an extractant/complexing agent for gold dates back to a 1941 paper by Plaskin and Kozhukhova [15]. Interest was renewed by these researchers in 1960 [16], and since then several investigators have studied the gold-thiourea system [1,9,10,11,13,17,18,19,20,21,22,23,24,25]. Several of these investigators studied the rate of dissolution of pure gold in acidic thiourea solutions. Under proper conditions of acid and added oxidant, a thiourea medium has been shown to dissolve the gold as much as twelve times faster than a cyanide medium [1,9,10,13,17,26]. The acids studied included sulfuric, hydrochloric, and nitric. The various oxidants examined were ferric ion, hydrogen peroxide, sodium peroxide, formamidine disulfide, dissolved oxygen [1], ozone [18,19], and potassium permanganate [20,21]. The fastest leaching system was generally concluded to be the ferric ion-sulfate media system, although choride media results were very similar. Rates are dependent on thiourea and oxidant concentrations, and appear to be controlled partially by chemical phenomena and partially by transport of oxidant species to the gold surface. A transition from diffusion control to kinetic control takes place at higher stirring speeds. These factors are similar to

characteristics of cyanide dissolution of gold [17].

With the exception of one study [11], investigations on actual ground gold ores comparing cyanidation leaching rates to those of acidic thiourea solutions supported the results of the work on pure gold [1,9,13]. However, the increase in leaching rate was not as large as for pure gold samples. Other studies have been conducted on actual ores and/or concentrates [19,23,27] with recoveries of greater than 90% claimed using leaching times of eight hours or less [19,23].

Leaching rates tended to decrease somewhat with time, possibly due to the deposition of elemental sulfur on the gold surfaces. The sulfur originates from the breakdown of the thiourea with time [1,9,10,17,22].

A third well-known disadvantage of cyanidation is the inhibition of the dissolution of gold caused by certain metal ions and other species [14,28,29]. These agents form from dissolution of constituents present in the ore. Such elements as copper and zinc tend to form ionic complexes with cyanide species, thus reducing the availability of cyanide ion for gold dissolution, as well as increasing cyanide consumption. These metal-cyanide species may also indirectly inhibit the gold dissolution reactions. Other elements, such as arsenic and antimony, react with cyanide solutions in such a way as to reduce the availability of dissolved oxygen necessary for the dissolution of gold. Still other species, such as pyrite and pyrrhotite, inhibit gold dissolution in both ways. The presence of certain carbonaceous materials can greatly reduce gold extraction by adsorbing the gold-cyanide species, with subsequent losses of gold to the tailings.

Thiourea leaching has been reported to be affected to a much lesser degree by the presence of some of these cyanidation inhibitors

[9,13,16,17,23,24]. In fact, one study suggests that copper, antimony, and arsenic minerals, and activated carbon actually increase the effectiveness of thiourea processing of gold [23]. However, other investigators studying thiourea in nitric and hydrochloric acid media report a decrease in solubility in the presence of zinc, copper, and lead salts due to complex formation of these metals with thiourea [16,22]. Iron minerals present may serve as a partial supply of ferric ion oxidant for the thiourea-gold interaction [1,11]. All in all, it has been concluded that thiourea has an advantage over cyanidation in the presence of these species.

The major disadvantage of the thiourea process is reported to be the higher cost due to high thiourea and acid consumptions [1,9,11]. Excessive thiourea consumption is partially due to the oxidation-degradation of thiourea by ferric iron. Thiourea degradation will be discussed further later in this thesis. Other factors which contribute to thiourea degradation are temperature [11] and complex formation with other metals [16,22]. It has been shown that thiourea consumption can be reduced by an acid pre-leach to remove impurities. Also, in tests where leach liquor was recycled through three separate batches of ore, the thiourea was shown to be consumed effectively while gold extraction dropped only slowly [4,30].

Acid consumption is obviously dependent upon the mineralogy of the ore or concentrate treated: ores high in oxide minerals will be especially reactive with acid. Fortunately, concentrated sulfuric acid is a relatively inexpensive pH modifier with excellent availability.

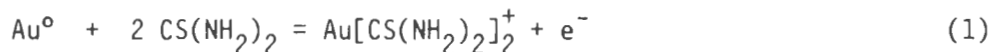
Reviewing the above-mentioned aspects of thiourea leaching as compared to cyanidation, it does appear that certain types of gold-

bearing materials may be more amenable to a thiourea extraction process assuming overall costs can be controlled at competitive levels. However, environmental considerations may reduce the importance of a cost disadvantage.

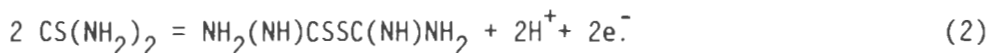
At the present time Nevada has several gold ore processing facilities under construction and many more potential orebodies being evaluated. Many of these are finely-disseminated Carlin-type deposits (see METHODS AND MATERIALS Section) which contain certain cyanidation inhibitors, notably carbonaceous material and/or arsenic minerals. Given the reported advantages of a thiourea process, it was decided to undertake groundwork experimentation on leaching of Carlin-type ores. The parameters studied include thiourea concentration, oxidant requirements, pH, leaching time, and leaching temperature. Optimizations of overall gold extraction and thiourea usage were the goals of the experiments. The results are presented in this thesis.

THEORETICAL CONSIDERATIONS

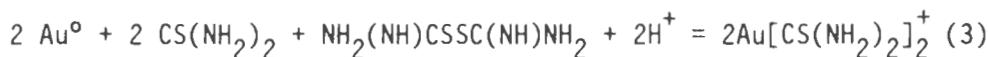
Thiourea in an acid medium forms a single cationic specie with gold according to the following simple dissolution/complexation reaction:



This reaction is considered reversible [10] with an as written electrode potential (E°) at 25°C of -0.38V [10,17]. In the presence of an oxidizing agent such as hydrogen peroxide or ferric ion, thiourea will oxidize in successive stages to a number of products. The first step is the formation of formamidine disulfide:



This is a rapid reaction with an electrode potential of +0.42V. It has been shown that formamidine disulfide itself is a very active oxidant and is considered to be required for actual gold dissolution [1]. A combination of (1) and (2) yields the suggested overall equation [17]:

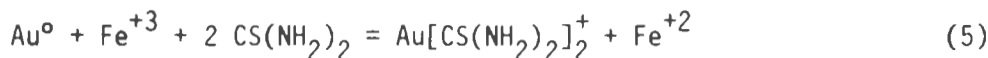


with an $E^\circ = +0.04\text{V}$ and $G^\circ = -1845 \text{ cal/mole}$ (for two electrons transferred).

In a sulfate medium containing ferric ion, oxidation is provided by the $\text{Fe}^{+3}/\text{Fe}^{+2}$ couple:



with $E^\circ = +0.77\text{V}$. Combining (4) with (1) yields the following:



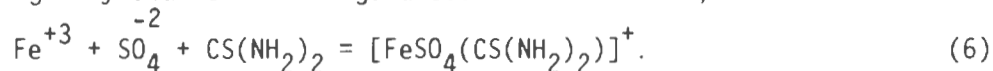
with $E^\circ = +0.39\text{V}$ and $G^\circ = -8994 \text{ cal/mole}$ [17]. A comparison of the free energies of (3) and (5) indicates a greater tendency for (5) to proceed; thus the presence of an added oxidant as well as formamidine

disulfide is advantageous. It has also been suggested that a ferric ion-thiourea complex is formed which also aids in dissolution of gold [9].

Continued oxidation of thiourea results in the slow formation of products in which sulfur has higher oxidation states. At alkaline pH values oxidation products are urea and sulfate ion [1]. Formamidine disulfide has been shown to break down with time to cyanamide, hydrogen sulfide, and elemental sulfur [1,11].

It has been noted that these secondary redox-reaction products tend to cause a passivation of the gold surface, thus partially reducing the rate of dissolution of gold [1,9,10,15,17,22]. However, the presence of ferric ion as an oxidant reduces the rate of these secondary reactions; thus, the most rapid rate of gold dissolution is obtained with ferric ion as oxidant [1].

Thiourea tends to react slowly with ferric iron in a sulfate media forming very stable metal-ligand coordination bonds, such as:



This process continues with the reduction of the iron to the complex $[\text{Fe}(\text{CS}(\text{NH}_2)_2)]\text{SO}_4$ [1]. At increased thiourea concentrations these reactions become more pronounced [9].

With regard to pH, it was noted that the formamidine disulfide formation reaction (Equation (2)) is a reversible reaction, and the reduction potential value of +0.42 V is independent of pH from 0 to 4.3. However, it was also noted that the rate of dissolution of gold is strongly dependent on pH; i.e., increasing with decreasing pH.

METHODS AND MATERIALS

Ore MaterialsChipmunk Gold Ore

The majority of the experimentation utilized material from the Chipmunk orebody of the Northumberland mining area in Nye County, south-central Nevada, USA. The orebody is located in the Toquima Range at the head of Northumberland Pass, about 175 kilometers from Tonopah. The Chipmunk orebody is currently owned by the Cyprus Mines Corporation which at present is conducting a cyanide heap leaching operation at the site. The ore itself is a finely-disseminated gold-bearing ore of the "Carlin" type as described by Wargo [31]:

"The classical Carlin-type gold deposit, as exemplified at the Carlin mine, contains micron-sized, high-fineness gold particles in a dolomitic siltstone. The mineralization is disseminated in the sense that it occurs more or less randomly in a block whose diameter might be a centimeter, a meter, or up to a kilometer. Randomly, however, does not mean accidentally. The gold occupies specific sites based on the chemistry of deposition and the nature of the structures present at the time of mineralization."

These types of deposits occur at various locations in Nevada and Utah [31,32,33] and "appear to be associated with hydrothermal processes related to late Tertiary tectonism and volcanism." [32,p.71]. The Chipmunk ore is a quartz matrix with minor amounts of dolomite and trace muscovite. It contains 2.4 grams of gold per ton of ore (0.07 tr. oz. Au/avoir. ton rock) as determined by fire assay techniques. A

semi-quantitative mass spectrographic analysis of the ore was conducted by the United States Bureau of Mines Reno Metallurgy Research Center and is presented in Table I. The somewhat charcoal-colored appearance of the ore indicates the presence of some type of carbon material; however, this carbonaceous material is not activated and does not interfere with cyanide or thiourea extraction.

The sample as received was crushed to minus 12.7 mm (minus 1/2 inch). It was mixed thoroughly and quartered with a riffle-type splitter. Initially, one quarter was further pulverized to a fine powder using a Braun Type-UA pulverizer. A screen analysis of the powder is presented in Table II. This batch of ore was consumed in testing; therefore, a second quarter of the original sample was pulverized for use in the remainder of the testing program. A screen analysis of this second batch is also presented in Table II.

Carlin Carbonaceous Gold Ore

Upon development of acceptable thiourea leaching parameters, the extraction testwork was extended to a carbonaceous gold ore from the well-known Carlin Gold Mine in northern Eureka County, northeastern Nevada, USA. The sample of ore was representative of the carbonaceous gold ore found in several zones of the orebody. The presence of activated carbon and other long-chained hydrocarbon-type compounds results in adverse effects on cyanide processing by adsorbing the gold-cyanide complex ions, with an ultimate loss of the gold to the solid tailings [34,35].

The Carlin ore sample contains 10 grams of gold per ton of ore (0.30 tr.oz. Au/avoir. ton ore) as determined by fire assay techniques,

and the carbon content was about one weight per cent as determined by weight loss during roasting. The sample was pulverized as received and a screen analysis is presented in Table III. A typical elemental analysis of Carlin's carbonaceous ore is presented in Table IV [36]. The level of arsenic is especially notable.

Analytical Methods

Gold

1. Solid Samples

Standard fire assay techniques [37] were employed to determine the gold content of head and tailing samples. A modified silica flux was employed and is presented in Table V.

2. Liquid Samples

Filtrate samples from thiourea extraction tests were analyzed by fire assaying by evaporating 50 ml of solution in an assay-grade lead foil boat, combining with a fluxing mixture, and proceeding with a standard crucible assay. This flux mixture is also presented in Table V. Liquid samples from cyanidation tests were also run in this fashion, but analytical checks were conducted using a Perkin-Elmer 303 atomic absorption spectrophotometer and following the standard practices recommended in Perkin-Elmer's manual [38].

Thiourea

The concentration of thiourea in solution was determined by titration with 0.10 N mercuric nitrate solution using diphenylcarbazide as indicator [39].

Free Cyanide

The concentration of free cyanide in solution was determined by titration with 0.025 N silver nitrate solution using para-dimethylamine-benzylidene rhodanine as indicator [2,p.99].

pH/mV Measurements

The pH values of filtrate solutions were measured using a Corning Model 130 pH meter in conjunction with a Radiometer GK2321C combination electrode. PHydrion pH indicator paper was used occasionally as a pH check. The oxidation-reduction potentials of the leach liquors were monitored using the above-mentioned pH meter with a Markson 1202 combination platinum/reference electrode.

Testing Procedures

Cyanidation Bottle Roll Tests

All of the cyanidation tests were conducted at room temperature using the standard bottle roll technique. The tests were designed to provide a standard gold recovery with which to compare the thiourea test results.

In this method 100 grams of ore were placed in a 500 ml polyethylene bottle. To the bottle were added 200 ml of distilled water, 0.5 grams sodium cyanide, and sufficient calcium hydroxide to produce a solution pH between 10 and 11 (usually 0.05g Ca(OH)₂). The bottle was then capped and placed in a rolling mill for agitation at approximately 30 rpm for a predetermined length of time. At the end of the test period, the contents of the jug were placed in a 9.0 cm diameter plexiglass pressure filter at 206.8 kPa (30 psig) for liquid-solid separation.

After recording the volume of leach liquor, the liquor pH was recorded. The liquor was then analyzed for residual free cyanide and assayed for gold concentration. The solid tailings were washed with three volumes of distilled water (total wash water volume was approximately 500 ml) after which they were removed from the filter, weighed, and placed in a drying oven for 24 hours. The dry weight was recorded, after which the tailings were pulverized and analyzed by fire assay. The weights, volumes, and analyses were then used to ascertain gold extraction and check the material balance.

Thiourea Bottle Roll Tests

The majority of the thiourea testwork was conducted at room temperature using the bottle roll technique to investigate the effects of pH, leaching time, thiourea concentration, and oxidant concentration on overall gold extraction and thiourea consumption. This method was basically the same as that for the cyanidation testing. For the thiourea tests, 100 grams of ore were placed in a 500 ml polyethylene bottle. A predetermined volume of 5 N sulfuric acid was then added to the bottle, and the mixture was allowed to condition for approximately 20 minutes. An acid strength of 5 N was determined to be a convenient concentration because reasonable variations in acid volume used resulted in effective control of the final pH of the slurry. At the same time this allowed for reaction of the acid-consuming constituents in the ore during the conditioning period. At the end of the conditioning period a known weight of reagent-grade thiourea was added along with a known weight of reagent-grade ferric sulfate (optional). Enough distilled water was added to obtain a total of 200 ml of liquid in each

test. The result was a slurry of approximately 30 weight percent solids. The ferric sulfate was supplied as an oxidant, but a large number of tests omitted the oxidant.

The bottle containing the slurry was stoppered with a one-hole stopper (to allow for pressure release) and placed in the rolling mill at about 30 rpm for the predetermined leaching period. Liquid-solid separation was then performed as in the cyanide tests. After recording the filtrate volume, 50 ml of solution were evaporated in a lead foil boat prior to fire assay. The remainder of the filtrate was analyzed for pH, oxidation potential (on later tests), and thiourea consumption.

From the above experimentation, a set of "optimum" parameters was established for use in further testing involving temperature variation, acid pre-washing, recycling of filtrate, and two-stage leach tests.

Agitated Open Beaker Tests

In order to investigate the effects of temperature on the efficiency of gold extraction with thiourea, several 24 hour tests were conducted in 600 ml open beakers. An impeller-type agitator with a variable speed electric motor was used to provide mixing at 180-200 rpm. This arrangement also allowed for continuous monitoring of oxidation potential if desired. Slurry preparation, liquid-solid separation, etc. were identical to that of the bottle roll tests.

For higher-temperature tests the beaker containing the slurry was placed in a Precision Scientific Model 25 water shaker bath. Tests were conducted at 40 and 60°C.

A low-temperature test was conducted at 10°C by placing the beaker on a Thermoelectrics Unlimited Stir-Kool Model SK-12 cooling plate.

Acid Pre-Wash of Ore

In an attempt to ascertain the effects of pretreatment of the ore with a sulfuric acid solution, a test was conducted in which 100 grams of ore was contacted with 500 ml of 3.57 N (16.3 wt.%) sulfuric acid for three hours. The ore was then filtered in the pressure filter, and the cake was weighed to obtain the amount of liquid hold-up. The cake was repulped in a plastic bottle to 200 ml of liquid with distilled water. Ten grams of thiourea were added and the bottle was run through the bottle roll test sequence with a 24 hour leach time. A 1 to 5% sulfuric acid wash has been used successfully by several investigators to lower overall acid and thiourea consumptions [9,13,16,23].

Recycle of Leachant

In order to obtain information pertaining to re-use of leach liquor, as would be the case in a larger-scale continuous-type leaching operation, a test was run in which 50 ml of filtrate from a 20 hour bottle roll test were mixed in a bottle with 50 grams of fresh ore and 50 ml of 5 N sulfuric acid. The bottle was then agitated for 20.5 hours followed by the standard bottle-roll treatment. However, only 25 ml of the second filtrate were evaporated for analysis. The original solution and both tailings were also analyzed.

Two-Stage Leach

Two-stage leaches were conducted as a further step toward simulation of continuous operation, as well as to provide insight into possible establishment of chemical equilibria in a one-stage test. This was accomplished by repulping the filter cake from a room-temperature

open-beaker test (after a pre-set leaching time) with additional amounts of acid, water and thiourea. This second slurry was leached at room temperature for a pre-determined time in an open beaker; both filtrates and the tailing were analyzed.

RESULTS

Chipmunk Gold OreCyanidation

Six 24-hour bottle roll cyanidation tests were conducted to obtain a "standard" extraction with which to compare thiourea leaching test results. Using a 10 kg/t sodium cyanide concentration yielded an average extraction of 83%. Sodium cyanide and calcium hydroxide consumptions were 1 and 0.6 kg/t, respectively. A one-hour test was also conducted at the same cyanide treatment level yielding an 81% extraction.

Thiourea

Numerous bottle roll thiourea leach tests as well as several open-beaker tests were conducted to determine the effects of the various parameters on gold extraction and reagent consumption. Initially, two to three tests were conducted at each set of conditions in order to obtain a reliable average recovery value. However, as more tests were conducted and confidence in the testing and analytical methods increased, the number of tests at a given set of conditions was reduced to one or two. This was especially true when verification of a trend was the major goal of a given test.

Table VI presents the results of 24-hour tests conducted to ascertain the effect of pH on the extraction of gold for three different thiourea concentrations. These results are also presented in graphical form in Figure 1. The maximum extraction is 86% at pH 0.6 using the 35 kg/t thiourea concentration. At each concentration level the behavior

is similar: increasing extraction with decreasing pH. Plateaus in the extraction are reached at approximately 62% near pH 2 for the 10 kg/t concentration, 82% near pH 3 for the 200 kg/t concentration, and 85% at pH 1 for the 35 kg/t concentration level. Sulfuric acid consumption was about 250 kg/t for the leach tests near pH 1.0.

The results of increasing thiourea concentration on gold extraction are presented in Table VII and Figure 2. These 24-hour tests were run at three different sets of pH and oxidant leaching conditions. All three curves appear to level off above about 25 kg/t thiourea loading. The highest extraction (88%) was attained using 200 kg/t thiourea and 5 kg/t $\text{Fe}_2(\text{SO}_4)_3$. The other curves attain the 80-85% extraction level and remain there throughout the range of concentrations.

Figure 3 shows the change in the extraction of gold as related to the amount of $\text{Fe}_2(\text{SO}_4)_3$ oxidant used. Four different thiourea concentration and pH values are compared. These data are also presented in Table VIII along with test results at 200 kg/t thiourea loading and 0 to 5 kg/t $\text{Fe}_2(\text{SO}_4)_3$. The results at the different thiourea levels do not appear to form a definite pattern: recoveries at the 25 and 40 kg/t loadings tend to drop with increasing $\text{Fe}_2(\text{SO}_4)_3$ present, while the recovery at the 35 kg/t loading increases at the 5 kg/t $\text{Fe}_2(\text{SO}_4)_3$ level and then falls off. The 100 kg/t level remains fairly constant at about 86%. The data at 200 kg/t thiourea also suggest an increase in recovery upon addition of 5 kg/t $\text{Fe}_2(\text{SO}_4)_3$, but no data are available at higher oxidant levels.

The variation in gold extraction with respect to leaching time is observed in Figure 4 and the data are presented in Table IX. Six sets of leaching conditions were examined. At the lowest thiourea concen-

tration (10 kg/t), the change in extraction is somewhat erratic as leaching time is increased. It is observed that at 168 hours (7 days) of leaching the extraction appears to drop off from the 60-65% level of the shorter leach times. At 35 kg/t thiourea and 5 kg/t $\text{Fe}_2(\text{SO}_4)_3$, the extraction increases steadily from 75% at 2 hours to 85% at 24 hours. Data at this thiourea level but without oxidant indicate a similar trend with a leveling off of extraction at 86% up to and beyond 24 hours of leaching. At 100 kg/t thiourea it is observed that 77% of the gold can be extracted in one hour and extraction continues to increase to 90-plus % at 48 hours. At the highest thiourea level of 200 kg/t, the extraction is constant at about 81% over the range studied (1 to 48 hours). No data are available at leaching times of less than one hour.

Much data were obtained on thiourea consumption as each of the previously mentioned parameters were varied. However, some difficulty arose in obtaining a highly accurate consumption number for any particular test. The main problem encountered was the obscuring of the titration end point due to the presence of large quantities of iron species, sulfates, and other salts. Also, the nature of the calculation results in large increases in consumption values for very small decreases in volume of titrant used. Coupling these two facts results in the necessity to downplay the specific consumption number. Instead, one must seek out any trends which may be established by comparison of consumption values which vary considerably for a small change in a given parameter.

The consumption data for several series of tests are presented in Table X. Initially, it is noted that a strong pattern relating thiourea consumption to gold extraction is not clearly established.

Thiourea consumption is plotted as a function of leaching time for three sets of initial thiourea concentrations in Figure 5. In all cases the consumption jumps considerably from the short leaching period values (0.5 to 2.5 kg/t) to the 24 hour values (9 to 11 kg/t). However, beyond 24 hours the results become erratic; i.e., at the 35 and 100 kg/t loadings the consumption appears to drop somewhat, while at the 200 kg/t level the consumption continues to increase to 17.5 kg/t as leach time continues to 48 hours.

The variation of thiourea consumption with amount of $\text{Fe}_2(\text{SO}_4)_3$ added is shown in Figure 6. It is observed that for these sets of conditions thiourea consumption increases with increasing oxidant concentration. The curve at the 40 kg/t treatment level appears to be leveling off above the $\text{Fe}_2(\text{SO}_4)_3$ loading of 50 kg/t, but there is insufficient data at the 200 kg/t thiourea level to check this effect.

The effect of acid level on thiourea consumption is shown in Figure 7 for two sets of leaching conditions. Since the pH effect on gold recovery had been established, further testing at higher pH values was not conducted. However, it does appear that the acid levels near pH 1.0 (100-120Kg/t H_2SO_4) have little effect on thiourea consumption.

Figure 8 presents plots of thiourea consumption and gold extraction as functions of leaching temperature for a single set of leaching conditions. Thiourea consumption is fairly constant up to 40°C, after which it rises considerably. The temperature effects on gold recovery are somewhat dramatic in that leaching above room temperature results in a considerable drop in recovery (90% at 21°C to 54% at 60°C).

The change in thiourea consumption with changing initial thiourea treatment level is presented in Figure 9 for one set of leaching para-

meters. It is observed that the consumption value is fairly constant (9 to 11 kg/t) over the thiourea concentration range studied.

Oxidation potential measurements were taken on the bottle roll test filtrates of the final portion of the testwork, after some of the most effective leaching conditions were established. Also, oxidation potential was monitored continuously on several open beaker tests. Table XI presents the final readings along with leaching conditions, and Table XII presents the oxidation potential change with leaching time at constant leaching conditions.

It is noted that insufficient data have been collected to establish clear-cut patterns regarding the effects of the various parameters on oxidation potential and gold extraction. It does appear, however, that a potential above about +150 mV is required to obtain an extraction of 80-85%; but potentials higher than +150 mV do not result in a pronounced increase in extraction above 85%. Temperature data are also somewhat incomplete; however, it is noted that lower oxidation potentials for the 10 and 60°C tests coincide with lower recoveries as compared to room temperature tests.

The oxidation potential-leach time data of Table XII are presented graphically in Figure 10. Also plotted is gold extraction vs. leach time data for the same experiments. The potential rises rapidly from +124 mV at 0.16 hours to +157 mV at 5 hours where the curve flattens between +158 and +170 mV; the extraction curve seems to follow the same pattern, although there is an acute lack of data between 1 and 20 hours of leaching.

In Table XIII the results of the acid pre-wash test are compared to identical tests without pre-wash. It is observed that the extrac-

tion and oxidation potential are fairly constant at about 85% and +170 mV, respectively. However, thiourea consumption for the pre-wash appears to be somewhat higher at 9 kg/t versus 6 kg/t, although this could be within the error margin of the analytical procedure.

Results of the leachant recycle are presented in Table XIV in comparison with a 24-hour bottle roll test using similar leaching conditions. The first stage of the leach produced results nearly identical to those of the standard test. However, the second-stage recycle extraction is somewhat lower at 78% (vs. 85%) while the oxidation potential jumps from +164 mV to +193 mV. Thiourea consumption is reduced to 2 kg/t versus 5-6 kg/t for the standard and first stage tests.

The two-stage leach test results are compared to a standard test at similar conditions in Table XV. Total extraction by the two stages was similar to that of the standard test at approximately 85%. Thiourea consumption increased from 0.5 kg/t in the first stage to 6kg/t in the second. Second stage consumption was the same as that of the standard test.

Carlin Carbonaceous Gold Ore

Cyanidation

Several cyanidation bottle roll tests were conducted on Carlin carbonaceous ore. Ore samples were leached for 24 hours at 21°C using 5-10 kg/t sodium cyanide at a pH of 10-11. The average extraction was 3%. Cyanidation of a sample of Carlin ore which was roasted at 600°C for one hour resulted in a gold extraction of 87%. This behavior is typical of ores containing activated carbon materials.

Thiourea

A limited number of bottle roll and open beaker tests were conducted using thiourea as extractant. The results are presented in Table XVI. While there is not enough data to construct meaningful plots, numerous trends can be observed from the table.

The effect of pH on gold extraction is notable, as extraction increased from 3% to about 55% when the pH was lowered from 5.7 to about 1.0.

At the lower pH values the effect of leaching time is apparent by comparing the first stage of the 6-hour two-stage leach (42% extraction) to the 24 hour leach at similar conditions (52% extraction). The long-term 120-hour test yielded an extraction of 60%.

The presence of ferric sulfate oxidant appears to have little noticeable effect on either gold extraction or thiourea consumption.

Thiourea consumption is extremely erratic, varying from 0.5 kg/t to 49 kg/t. No hard-and-fast relationships can be gleaned from this data.

Oxidation potential data are almost non-existent; therefore no observations can be made regarding this parameter.

The effect of varying initial thiourea concentration is similar to that for Chipmunk ore. Under favorable pH conditions, extraction increases from 29% at 20 kg/t to about 55% at the 200-300 kg/t level.

A comparison of temperature tests indicates that leaching at 40°C is possibly detrimental to gold recovery since extraction dropped from 56% at 21°C to 48% at the higher temperature.

Two tests were run using two leaching stages in an attempt to increase extraction above the 55-60% level. The 6-hour two-stage leach

resulted in an overall extraction of 55%: 42% for the first stage with an additional 13% picked up in the second stage. The 24-hour two-stage test yielded 48% in the first stage and 11% in the second for an overall extraction of 59%.

DISCUSSION

Chipmunk Gold Ore

Cyanidation

The bottle roll cyanidation test results on Chipmunk ore indicate that cyanide leaching will yield an 80-plus % extraction even at short leaching times. This is probably due to the finely-disseminated nature of the gold particles in the ore. Although extensive testwork was not conducted on the cyanidation aspect, it is noted that extractions approaching 90% were not attained.

Thiourea

The results given in Table VI and Figure 1 reveal that thiourea extraction is a function of pH; i.e., increasing extraction with decreasing pH until extraction levels off near pH 2. This is consistent with the gold complexation reaction (3), as well as results published by other investigators [9,19,30]. Although the highest extraction given in the table occurs at the 35 kg/t thiourea level, it is noted that higher thiourea concentrations allow a larger range of effective leaching pH's before extraction begins to drop off.

Gold extraction is also a function of thiourea concentration as evidenced by Table VII and Figure 2. All three sets of data indicate that thiourea concentrations of greater than 50 kg/t are required to obtain an 80-plus % extraction. This effect is also consistent with the literature [1,9,15,16,17,19,30] as well as the gold complexing reactions (1) and (3). The drop in recovery at the higher concentrations for the pH 1.0 series is possibly due to the breakdown of forma-

midine disulfide resulting in a sulfur coating of the gold particles as discussed in the section on theory.

Regarding the presence of ferric sulfate as oxidant (Table VIII and Figure 3), it is observed that in every case the highest extraction was obtained using 5 kg/t $\text{Fe}_2(\text{SO}_4)_3$. However, extractions of 80-plus % were obtained when no oxidant was added. It is therefore assumed that the majority of oxidant required is supplied by the ore itself. Indeed, the spectrographic analysis of Table I indicates that the ore is 7% iron. However, a small additional amount is required; but higher $\text{Fe}_2(\text{SO}_4)_3$ additions result in a slight drop in extraction. This could be due to the breakdown of formamidine disulfide in the presence of higher amounts of ferric ion, forming sulfur and other secondary redox reaction products which are known to passivate the gold surface [1,9, 10,15,17,22]. Another explanation could be a depletion in the supply of thiourea resulting from complexing with ferric ion (Reaction (6)). The latter explanation could be especially applicable to the lower initial thiourea testwork (25,35,40 kg/t).

The data on leaching time versus gold extraction as presented in Table IX and Figure 4 are interesting in that the reported kinetic factor is observed only at the higher thiourea concentrations. Indeed, comparison with the one-hour cyanidation extraction of 81% suggests that for this particular ore, thiourea may have no kinetic advantage at all. It is possible that at shorter leaching times (less than one hour) thiourea would extract gold noticeably faster than cyanide, but no data are available to demonstrate this. Lower extractions at the 10 kg/t thiourea level are probably due to a shortage of available thiourea. At the 35 and 100 kg/t levels the extraction seems to be a

direct function of leaching time, while at the 200 kg/t level the rapid leveling of the extraction at 80-82% is again possibly due to passivation of the gold surfaces.

According to the literature, thiourea consumption should vary directly with ferric ion concentration, leaching time, temperature, and the presence of certain impurities [1,9,11,16,22]. Except for the latter factor which was not studied, the data in Table X and Figures 5 through 9 support these conclusions. The remaining three factors are closely related in that increased consumption is due to the ultimate breakdown of the formamidine disulfide to sulfur and other species which do not aid in gold complexation. Ferric ion will complex thiourea as well as aid in its degradation, thereby compounding its affect on consumption. Regarding leaching time, it would be expected that consumption would steadily increase as leaching proceeded. However, only at the 200 kg/t thiourea level is this effect noted past 48 hours. The absence of added oxidant in the 35 and 100 kg/t series could indicate that by 24 hours of leaching the ferric ion present in the ore has all been consumed (reduced to ferrous) and further thiourea degradation is slowed. At the 200 kg/t level, the presence of such a high excess of thiourea could contribute to a continuation of thiourea degradation. This may provide a clue to the relatively low extractions observed at the 200 kg/t level in the leaching time/extraction data of Table IX.

A lack of data on acid level versus thiourea consumption prohibits speculation as to whether increased acid level would result in increased consumption. However, the equations for thiourea breakdown suggest that this would not be the case.

The effect of leaching temperature on thiourea consumption and gold extraction as shown in Figure 8 indicates that thiourea is fairly stable in solution until temperatures rise above 40°C. The other consumption numbers are within experimental error. The drop in recovery at 60°C could possibly be due to thermal degradation of the gold-thiourea complex itself, as well as a loss of available thiourea/formamidine disulfide due to thermal effects.

Thiourea consumption does not vary with increasing initial thiourea concentration, maintaining a constant consumption of approximately 10 kg/t for 24 hours tests. Literature values vary between 1.4 and 5 kg/t although leaching conditions are not fully specified [9]. It is observed from Tables IX and X that to obtain a reasonable gold recovery (greater than 80%) from this ore requires either very high thiourea concentrations or leaching times of 8 hours or greater. For the former case, thiourea consumption will be relatively high; a lack of thiourea consumption data for intermediate leaching times at lower thiourea concentrations prevents further discussion of these variables. It should be noted that high thiourea consumption coupled with the very high acid consumption of 250 kg/t could result in a rather expensive gold extraction process.

The relative scarcity of oxidation potential data for leaching of Chipmunk ore (Tables XI and XII) precludes serious discussion of this parameter. As mentioned previously, however, a potential of +150-160 mV is required for 80-plus % extraction of the gold. Potentials beyond this level, however, appear to be related to factors other than gold-thiourea-formamidine disulfide complexation.

The addition of an acid pre-wash to a 24 hour leach exhibited no advantage whatsoever (Table XIII). Actually, a slight drop in recovery and an increase in thiourea consumption are observed. But it is suspected that these differences are within experimental error. Several literature sources mention the advantage of the use of a pre-wash to reduce thiourea consumption [9,13,16,23], and it is curious that such an effect was not observed in this case. However, it is possible that a shorter leaching time would help to reduce thiourea consumption while still maintaining an acceptable extraction.

The leachant recycle experiment indicates that a thiourea solution can be used for successive leaches while maintaining respectable extraction. The drop in consumption from 5 to 2 kg/t is consistent with work done by others and is encouraging regarding continuous ore processing. This drop could possibly be due to the initial conversion of a portion of the thiourea to formamidine disulfide, which would register as a loss of thiourea but would still be effective in gold dissolution.

Results of the two-stage leach (Table XV) indicate that an increase in gold extraction above 85-90% may not be possible on this ore. It can be concluded that this extraction ceiling is not due to a solution equilibrium being established in the single stage batch tests. Rather, this ceiling effect could be due to gold surface passivation by redox reaction products. A further explanation could be that the fraction of gold that is not extracted is locked inside insoluble particles and thereby shielded from the thiourea species. This would explain the fact that cyanidation is also limited to about 80% extraction. The slight extraction advantage of thiourea would then be due to the dissolving effect of the acid treatment.

The increase in thiourea consumption in the second stage confirms the results of the acid pre-wash in that relatively high consumptions are still encountered even after pre-treatment with an acid liquor. However, in this case the leaching time was shorter (12 hours versus 24 hours) and it is known that very little of the thiourea was tied up in gold complexation.

Carlin Carbonaceous Gold Ore

Cyanidation

Cyanidation of gold ores containing activated carbon species and hydrocarbon chains is notoriously ineffective unless a prior oxidation treatment is employed. The preferred method at the present time is the use of chlorine to effect oxidation of the carbonaceous materials via the hypochlorite ion, thus eliminating the deleterious effects of these species. Extraction values of 90-plus % have been reported using this technique [35]. Another very effective (but expensive) pre-treatment method is roasting of the ore to provide thermal oxidation of the carbonaceous matter. Gold extraction from Carlin carbonaceous ore increased from 3% without roasting to 87% when roasting was employed, thus establishing a reasonable recovery value to use in comparison with thiourea leaching results.

Thiourea

The results of the limited thiourea testing of the Carlin ore as presented in Table XVI indicate that the relationships observed in leaching the Chipmunk ore are also evident in this case. A pH of near 1.0 appears to be required although higher acid consumptions are en-

countered to attain this pH level.

Oxidant levels had no effect on extraction, indicating that ample oxidant is supplied by the ore itself. Initial thiourea concentrations required to obtain the higher extractions of 50-60% were on the order of 200-300 kg/t which is somewhat higher than for Chipmunk ore. It should be noted here that the Carlin ore contains about 4 times as much gold as the Chipmunk ore.

Leaching time appears to be important for obtaining higher extractions, since recovery increases from 42% at 6 hours of leaching to 60% at the 120 hour leach time. Comments on temperature are limited by a lack of data, although a slight decrease in recovery was noted at the 40°C leaching temperature. This is similar to results on Chipmunk ore.

The erratic nature of the thiourea consumption values suggests the presence of a higher level of interference in the titration samples. Virtually none of the thiourea consumption patterns found in leaching Chipmunk ore are consistently observed in the Carlin data.

The increase in gold extraction using thiourea versus cyanide is significant as recovery increases to near 60% with thiourea as compared to a 3% extraction for straight cyanidation: a 20-fold increase. As reported in the literature, a thiourea process is affected to a much lesser degree by carbon species [23].

However, 60% extraction is still well below the 87% extraction obtained by cyanidation of the roasted sample, as well as below a minimum extraction acceptable for an industrial application. The two-stage extraction tests did nothing to break through this extraction ceiling. Possible explanations are passivation of the gold surfaces by thiourea breakdown products or the presence of a type of carbonaceous

material which is capable of adsorbing gold-thiourea species. This subject is extremely interesting and deserves more attention. Perhaps a modification of leaching procedures would result in an 80-plus % extraction, thus solving a problem which has long plagued the gold industry. However, this must wait for further research.

CONCLUSIONS

Chipmunk Ore

1. Thiourea leaching of Chipmunk ore is technically possible, with gold extractions equal to or exceeding extraction obtained by cyanidation. However, relatively high consumptions of thiourea and sulfuric acid may make thiourea processing of this ore expensive in comparison to cyanidation.
2. Due to the finely-disseminated nature of the gold in this ore, thiourea leaching does not exhibit a practical kinetic advantage over cyanidation.
3. Optimum leaching conditions for an 80-plus % gold extraction using thiourea ore:

Thiourea	-	50-100 kg/t
Ferric Sulfate	-	5 kg/t
Temperature	-	Ambient
Time	-	8-24 hours
pH	-	1.0
Oxidation Potential	-	+160 mV
4. Pre-washing Chipmunk ore with an acid solution does not yield advantages in thiourea consumption or gold extraction.
5. Re-use of thiourea leach liquors results in effective use of

the thiourea, thus reducing thiourea consumption while maintaining adequate extraction levels.

Carlin Carbonaceous Ore

1. Use of thiourea for extracting gold from Carlin ore results in gold recovery values of near 60% as compared to only 3% using straight cyanidation. This indicates that thiourea processing is affected to a much lesser degree by the presence of carbonaceous materials. However, other unknown factors have prohibited attaining recoveries in the 80-90% range. More work needs to be done in determining the nature of these factors.
2. The thiourea leaching behavior of this ore is similar to that of the Chipmunk ore. However, higher acid and thiourea treatment levels are required to obtain the maximum extraction.
3. Oxidant necessary for the gold complexation reactions is supplied totally by the ore itself.

Table I. Semi-quantitative spectrographic analysis of Chipmunk
ore.

<u>Element</u>	<u>Weight Percent</u>	<u>Element</u>	<u>Weight Percent</u>
Ag	ND*	Na	ND
Al	3.0	Nb	ND
As	ND	Ni	0.004
Au	ND	P	ND
B	0.008	Pb	ND
Ba	ND	Pt	ND
Be	ND	Re	ND
Bi	ND	Sb	ND
Ca	2.0	Si	M**
Cd	ND	Sn	ND
Co	ND	Sr	ND
Cr	0.001	Ta	ND
Cu	0.001	Te	ND
Fe	7.0	Ti	0.1
Ga	ND	Tl	ND
Hf	ND	V	0.01
In	ND	W	ND
Mg	1.0	Zn	ND
Mn	0.06	Zr	0.007
Mo	ND		

* Not Detected.

** Major Constituent.

Table II. Screen Analyses of Chipmunk ore samples.

<u>Screen</u>		<u>Sample No. 1</u>		<u>Sample No. 2</u>	
<u>Mesh*</u>	<u>Microns</u>	<u>Weight Percent</u>	<u>Weight Percent Passing</u>	<u>Weight Percent</u>	<u>Weight Percent Passing</u>
+48	297	27.0	73.0	7.3	92.6
65	210	10.2	62.8	10.6	82.0
100	149	7.4	55.4	11.9	70.1
200	74	12.6	42.8	14.0	56.1
-200	-	42.8	0	56.1	0
		<u>100.0</u>		<u>99.9</u>	

*Tyler Screen Mesh Designation

Table III. Screen analysis of Carlin carbonaceous ore sample.

<u>Screen</u>			
<u>Mesh*</u>	<u>Microns</u>	<u>Weight Percent</u>	<u>Weight Percent Passing</u>
+48	297	12.0	88.0
65	210	7.9	80.1
100	149	5.8	74.3
200	74	10.8	63.5
325	45	13.1	50.4
-325	--	50.4	-0-
		100.0	

*Tyler Screen Mesh Designation

Table IV. Typical elemental analysis of Carlin carbonaceous ore.*

<u>Element</u>	<u>PPM</u>	<u>Element</u>	<u>PPM</u>
Ag	ND**	Ni	15
As	1500	Pb	15
Au	30	Sb	ND
B	30	Sc	7
Ba	150	Sr	70
Co	5	Tl	300
Cr	30	V	50
Cu	20	W	ND
Ga	15	Y	15
La	30	Yb	1.5
Mn	150	Zn	ND
Mo	15	Zr	70
Nb	L***		

* After Wells and Mullens [36, p. 188]

** Not Detected at limit of detection

*** Detected, but below limit of determination

Table V. Fire assay flux mixtures.

<u>Flux Ingredient</u>	<u>Solution Assay*</u>	<u>Solids Assay**</u>
SiO ₂	10 grams	12 grams
Na ₂ CO ₃	40	40
PbO	35	70
Na ₂ B ₄ O ₇	10	15
Flour	0.5	3

* For 50 ml evaporated solution.

** For one assay ton solids (one assay ton = 29.166g).

Table VI. Effect of pH on thiourea extraction of gold from Chipmunk ore.

Leach Conditions: Temperature - 21°C
Time - 24 hr.

Thiourea (kg/t)	Fe ₂ (SO ₄) ₃ (kg/t)	Acid Level (ml 5N H ₂ SO ₄)	Final pH	% Extraction
10	5	120	0.6	62
10	5	100	1.0	61
10	5	80	1.8	61
10	5	70	4.0	45
10	5	60	4.9	17
35	5	150	0.1	84
35	5	120	0.6	86
35	5	90	1.2	81
35	5	80	1.9	81
35	5	75	2.7	74
35	5	60	4.8	54
200	0	120	0.6	82
200	0	100	1.0	80
200	0	90	1.6	82
200	0	75	3.3	79
200	0	54	5.6	33

Table VII. Effect of initial thiourea concentration on gold extraction from Chipmunk ore.

Leach Conditions: Temperature - 21°C
Time - 24 hours

<u>Thiourea</u> (kg/t)	<u>Fe₂(SO₄)₃</u> (kg/t)	<u>pH</u>	<u>% Extraction</u>
5	5	0.7	36
10	5	0.7-1.0	61
20	5	0.8-1.0	71
25	5	0.7-1.0	78
35	5	0.7	86
40	5	0.7-1.0	81
50	5	0.7	83
100	5	0.7	87
200	5	0.7-0.9	88
10	0	0.6	57
35	0	0.6	82
100	0	0.6	83
200	0	0.6	82
20	0	1.0	84
35	0	1.0	86
75	0	1.0	82
100	0	1.0	87
125	0	1.0	79
200	0	1.0	80

Table VIII. Effect of ferric sulfate concentration on thiourea extraction of gold from Chipmunk ore.

Leach Conditions: Temperature - 21°C
Time - 24 hours

<u>Thiourea</u> <u>(kg/t)</u>	<u>Fe₂(SO₄)₃</u> <u>(kg/t)</u>	<u>pH</u>	<u>% Extraction</u>
25	5	0.6	78
25	10	0.6	76
25	20	0.6	74
35	0	0.6	82
35	5	0.6	86
35	10	0.6	84
35	30	0.5	85
40	5	1.0	86
40	10	1.0	80
40	50	1.0	71
40	100	1.0	75
100	0	0.6-1.0	86
100	5	0.6	87
100	10	1.0	80
100	20	1.0	86
200	0	0.6-1.0	81
200	5	0.6-1.0	88

Table IX. Effect of leaching time on thiourea extraction of gold from Chipmunk ore.

Leach Conditions: Temperature - 21°C

Thiourea (kg/t)	Fe ₂ (SO ₄) ₃ (kg/t)	Acid Level (ml 5N H ₂ SO ₄)	pH	Time (hr)	% Extraction
10	5	120	0.6	4	65
10	5	120	0.6	8	67
10	5	120	0.6	12	57
10	5	120	0.7	24	62
10	5	120	0.7	48	61
10	5	100	1.0	24	61
10	5	100	1.3	168	51
35	5	120	0.4	2	75
35	5	100-120	0.7-0.4	4	76
35	5	120	0.5	9	80
35	5	100-120	0.7-0.5	12	81
35	5	120	0.6	24	85
35	0	100	0.7	8	77
35	0	100	1.0	24	86
35	0	100	1.0	48	86
100	0	100-120	0.8-0.6	1	77
100	0	100	0.9	20	84
100	0	100	1.0	24	87
100	0	100	1.0	48	92
200	0	100	0.6	1	82
200	0	100	0.7	4	81
200	0	100	0.8	8	81
200	0	100	1.0	24	80
200	0	100	1.4	48	82

Table X. Effects of various parameters on thiourea consumption: Chipmunk ore.

Series	Time (hr)	Fe ₂ (SO ₄) ₃ (kg/t)	Acid (ml 5N H ₂ SO ₄)	Temp. (°C)	Thiourea (kg/t)	Consumption (kg/t)	% Extraction
Time	8	0	100	21	35	1.5	77
	24	0	100	21	35	9	86
	48	0	100	21	35	7	86
	1	0	100	21	100	0.5	73
	24	0	100	21	100	9.5	89
	48	0	100	21	100	6	92
	4	0	100	21	200	2.5	77
	24	0	100	21	200	11	80
	48	0	100	21	200	17.5	82
Fe ₂ (SO ₄) ₃	24	5	100	21	40	6.5	82
	24	50	100	21	40	14.5	71
	24	100	100	21	40	17	75
	24	0	100	21	200	2.5	75
	24	5	100	21	200	10	77

Continued.....

Table X. Effects of various parameters on thiourea consumption: Chipmunk ore.
(continued)

Series	Time (hr)	$\text{Fe}_2(\text{SO}_4)_3$ (kg/t)	Acid (ml 5N H_2SO_4)	Temp. °C	Thiourea (kg/t)	Consumption (kg/t)	% Extraction
Acid	24	0	100	21	35	9	86
	24	0	120	21	35	10.5	82
	1	0	100	21	100	0.5	73
	1	0	120	21	100	0.5	81
Temp.	24	0	100	10	100	12.5	68
	24	0	100	21	100	9.5	90
	24	0	100	40	100	12.5	81
	24	0	100	60	100	22	54
Thiourea	24	0	100	21	35	9	86
	24	0	100	21	100	9.5	89
	24	0	100	21	200	11	80

Table XI. Final oxidation potential values for various thiourea leaching conditions: Chipmunk ore.

Thiourea (kg/t)	Fe ₂ (SO ₄) ₃ (kg/t)	Acid (ml 5N)	Time (hr)	Temp (°C)	+mV	%Extraction	Comments
35	50	100	24	21	224	80	
35	0	100	48	21	191	86	
100	0	100	48	21	158	92	
100	0	100	24	21	160	90	
150	0	100	24	21	158	86	
100	0	100	24	21	174	83	Acid pre-wash
100	0	100/50	20/20	21	164/193	84/78	Two Stage
100	0	100	24	21	168	84	Continuous Monitor
100	0	100	24	10	144	68	
100	0	100	24	21	160	90	
100	0	100	24	40	NA*	81	
100	0	100	24	60	109	54	

*Not Available

Table XII. Effect of leaching time on oxidation potential and gold extraction using thiourea: Chipmunk ore.

Leach Conditions: Temperature - 21°C
 Thiourea - 100 kg/t
 $\text{Fe}_2(\text{SO}_4)_3$ - 0

<u>Leach Time (hr)</u>	<u>mV (+)</u>	<u>% Extraction</u>
0.16	124	NA**
0.33	133	NA
0.75	138	NA
1	140	73
2.5	148	NA
4	153	NA
5	157	NA
20	164	84
23	169	NA
24	164*	87
48	158	92

* Average of Two Tests

** Not Available

Table XIII. Acid pre-wash comparison: Chipmunk ore.

Pre-wash conditions: Acid Solution - 3.57 N
 Acid Volume - 500 ml
 Wash Time - 3 hours
 Weight of Ore - 100g

Leach Conditions: Temperature - 21°C
 Time - 24 hours
 Thiourea - 100kg/t
 Fe₂(SO₄)₃ - 0
 pH - 0.8

<u>Test Type</u>	<u>No. of Tests</u>	<u>mV</u>	<u>Consumption (kg/t)</u>	<u>% Extraction</u>
Standard	5	+164	6	87
Pre-Wash	1	+174	9	83

Table XIV. Results of leachant recycle: Chipmunk ore.

Leach conditions: Temperature - 21°C
 Thiourea - 100 kg/t/stage
 $\text{Fe}_2(\text{SO}_4)_3$ - 0
 pH - 0.9 -1.0

Test Type	Leach Time (hr)	Acid Level (ml 5N H_2SO_4)	mV	Consumption (kg/t)	% Extraction
Standard	24	100	+164	6	87
Recycle:					
Stage 1	20	100	+164	5	84
Stage 2	20.5	50	+193	2	78

Table XV. Results of two-stage leach: Chipmunk ore.

Leach Conditions: Temperature - 21°C
 Thiourea - 100 kg/t/stage
 Fe₂(SO₄)₃ - 0
 pH - 0.8

<u>Test Type</u>	<u>Leach Time (hr)</u>	<u>Acid Level (ml 5N H₂SO₄)</u>	<u>Consumption (kg/t)</u>	<u>% Extraction</u>
Standard	24	100	5	86
Two-Stage:				
Stage 1	11	100	0.5	81
Stage 2	<u>12</u>	10	<u>6</u>	<u>3</u>
Total	23		Total 6.5	Total 84

Table XVI. Results of thiourea leaching of Carlin carbonaceous ore.

Thiourea (kg/t)	Fe ₂ (SO ₄) ₃ (kg/t)	Acid (ml 5N)	pH	Time (hr)	Temp (°C)	mV	Consumption (kg/t)	% Extraction	Comments
200	0	100	5.5	4	21	NA*	2.5	2	
200	0	100	5.7	24	21	NA	2.5	3	
200	50	150	1.8	24	21	NA	18	54	
200	0	150	2.0	24	21	NA	0.5	52	
300	0	160	0.7	24	21	NA	49	57	
300	100	160	0.7	24	21	NA	49	56	
**200	0	150	0.8	6	21	+132	44	42	Two Stage
**150	0	20	0.7	6	21	+148	0.5	13	
								55	Total
**300	0	150	0.8	24	21	NA	42	48	Two Stage
**300	0	130	1.0	24	21	NA	40	11	
								59	Total
20	0	150	1.0	24	21	NA	1	29	
300	0	160	0.8	24	40	NA	35	48	
300	0	150	1.0	120	21	NA	34	60	

* Not Available

** Two Stage

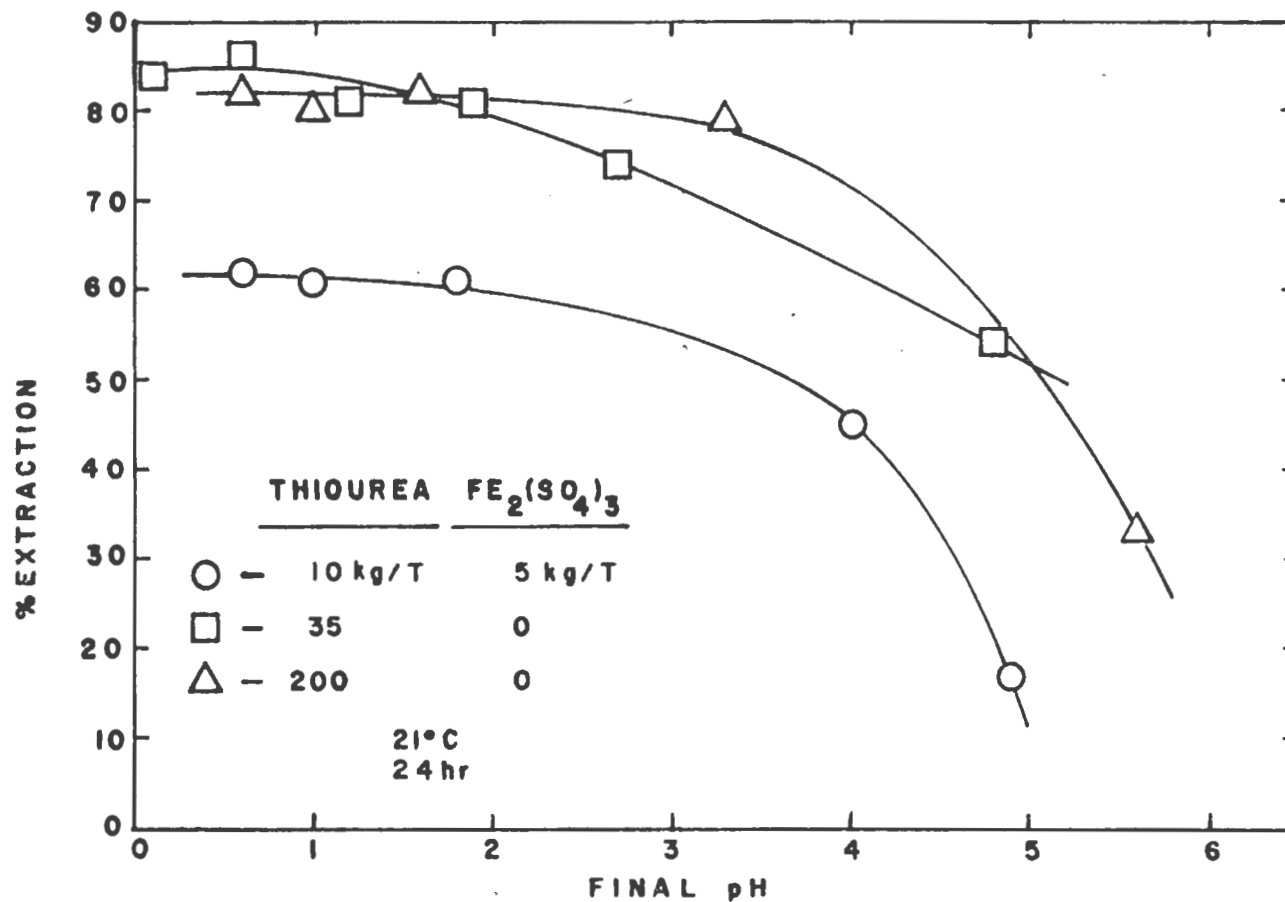


FIG. 1. PERCENT GOLD EXTRACTION VS. pH FOR CHIPMUNK ORE

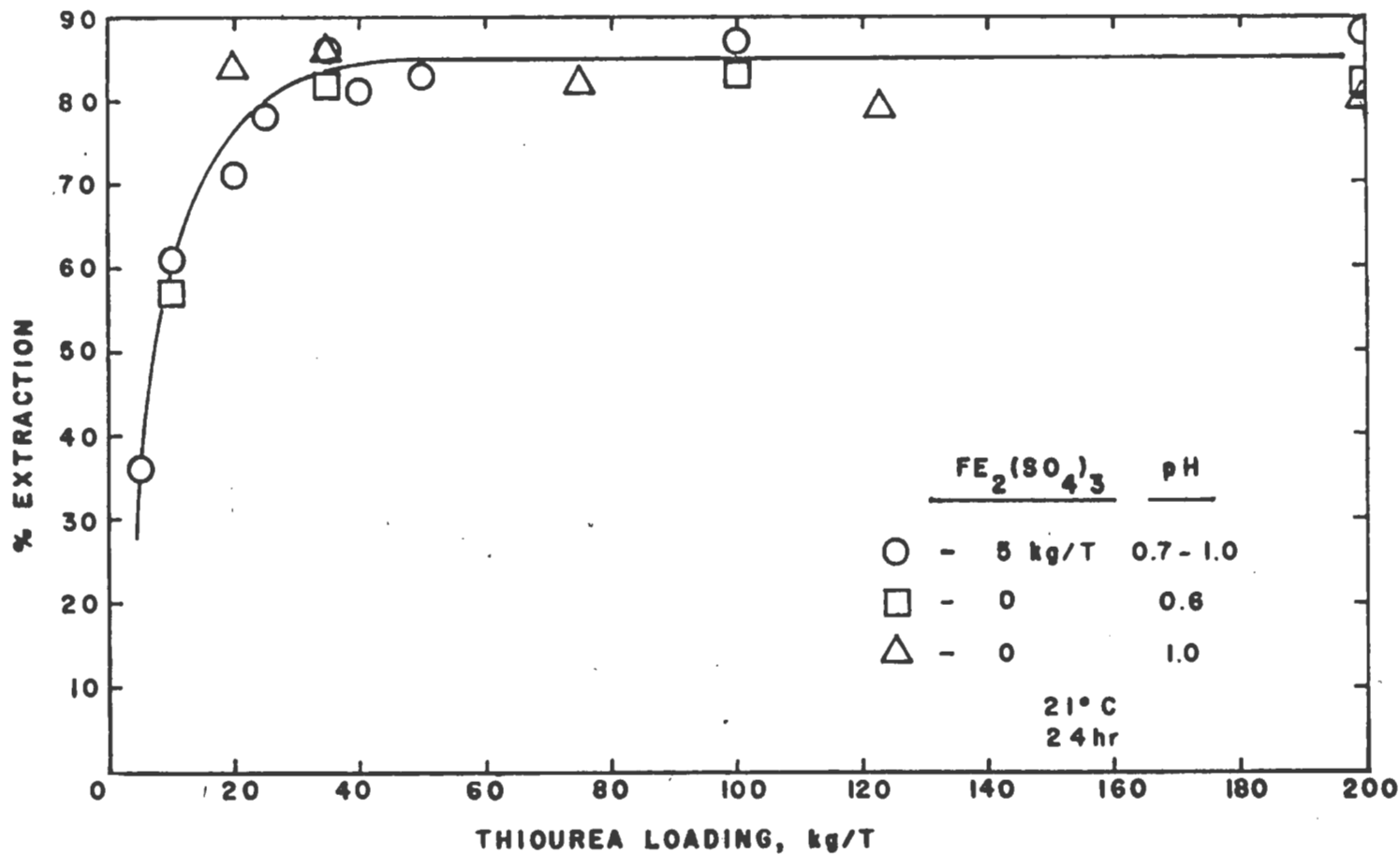


FIG. 2. PERCENT GOLD EXTRACTION VS. THIOUREA LOADING FOR CHIPMUNK ORE

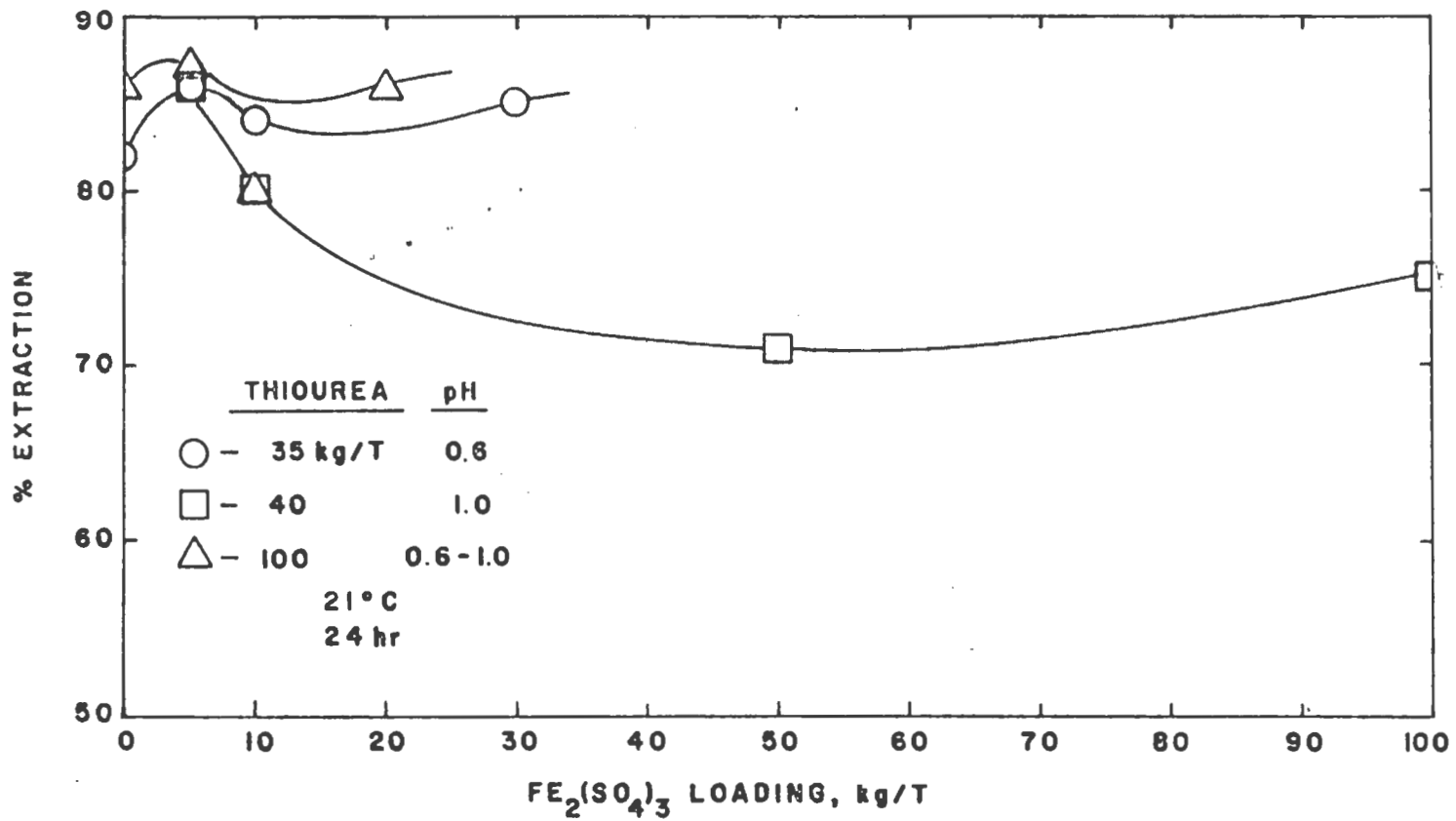


FIG. 3. PERCENT GOLD EXTRACTION VS. $Fe_2(SO_4)_3$ LOADING FOR CHIPMUNK ORE

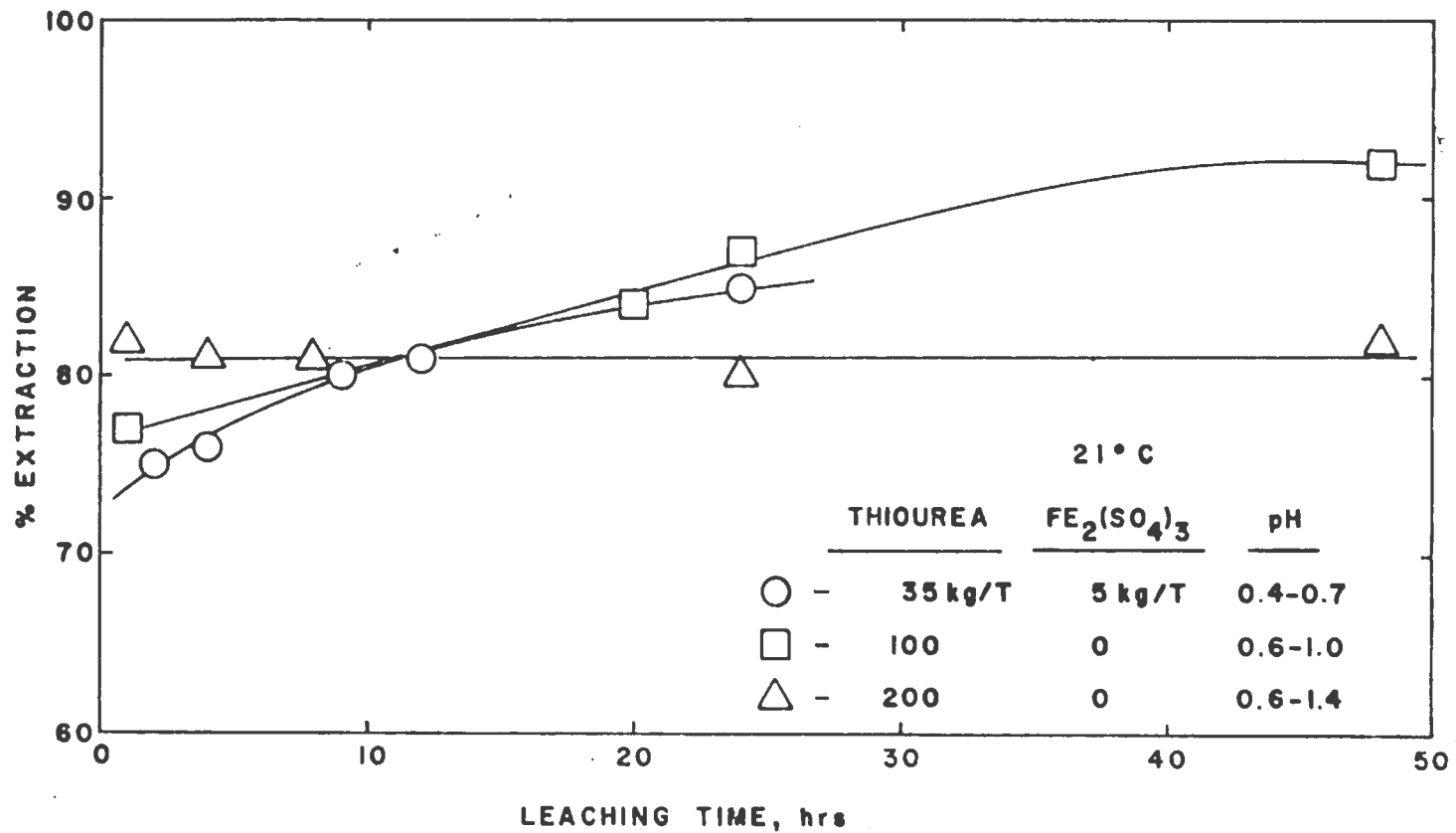


FIG. 4. PERCENT GOLD EXTRACTION VS. LEACHING TIME FOR CHIPMUNK ORE

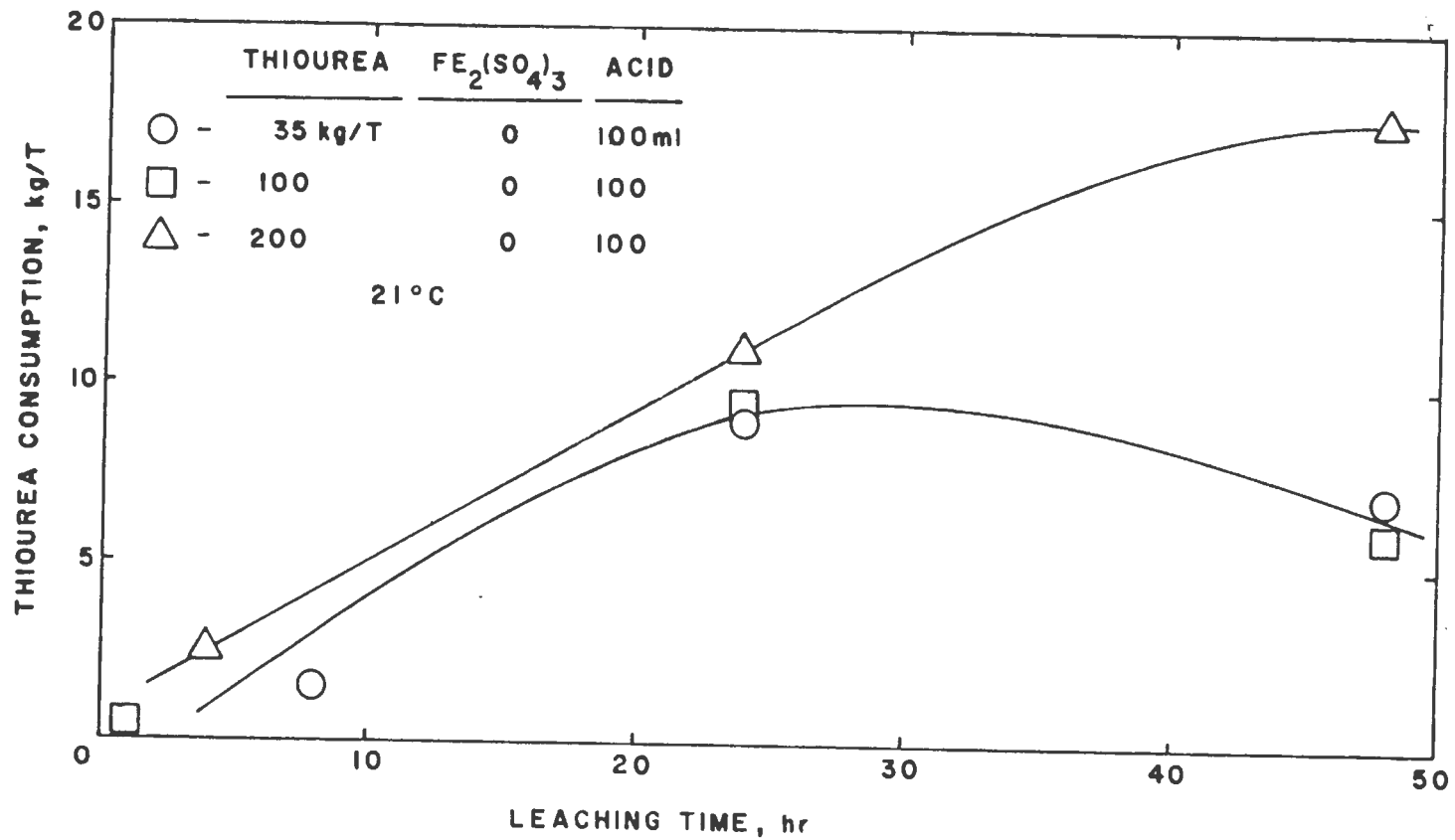


FIG. 5. THIOUREA CONSUMPTION VS. LEACHING TIME FOR CHIPMUNK ORE

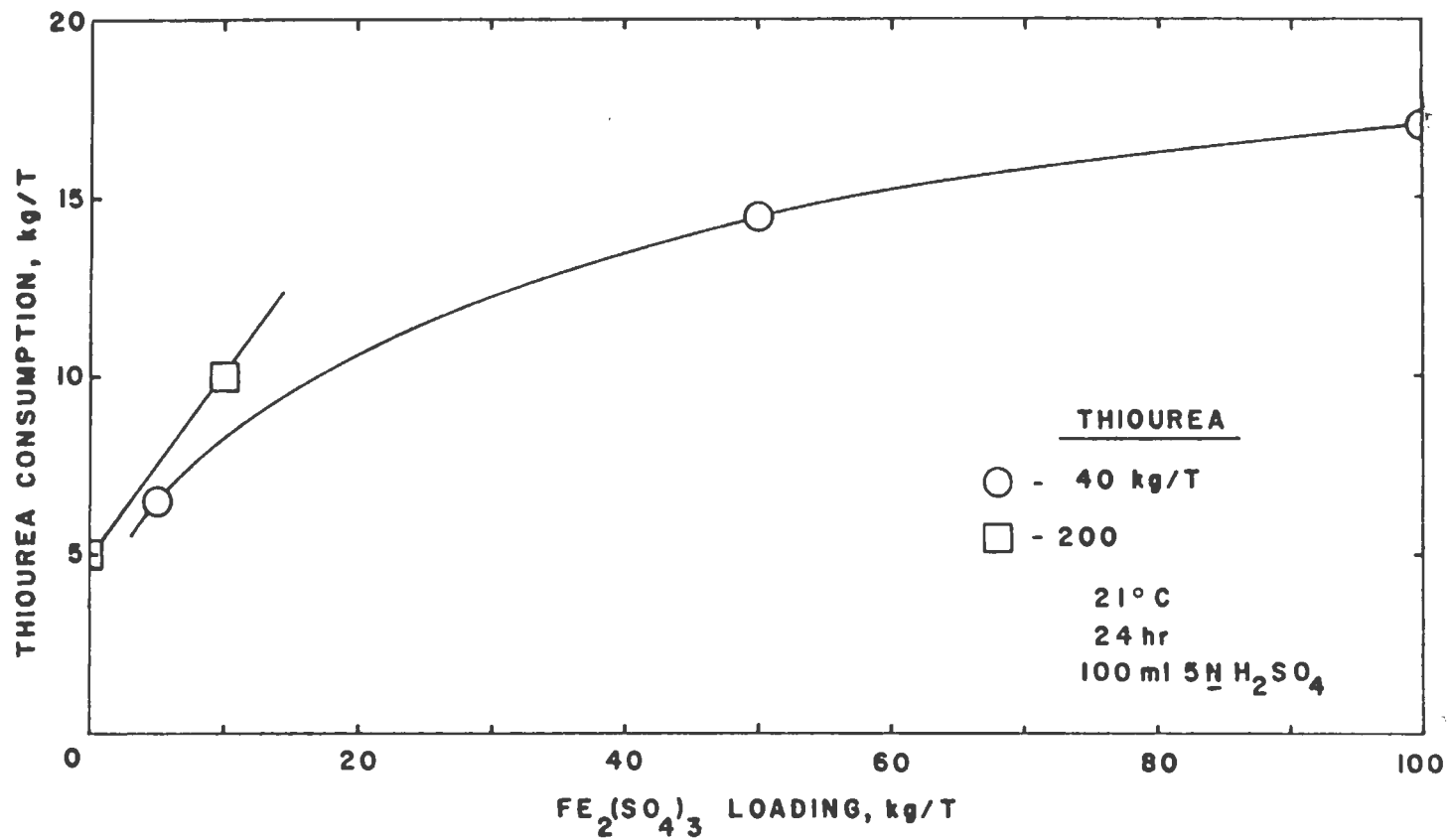


FIG. 6. THIOUREA CONSUMPTION VS. FE₂(SO₄)₃ LOADING FOR CHIPMUNK ORE

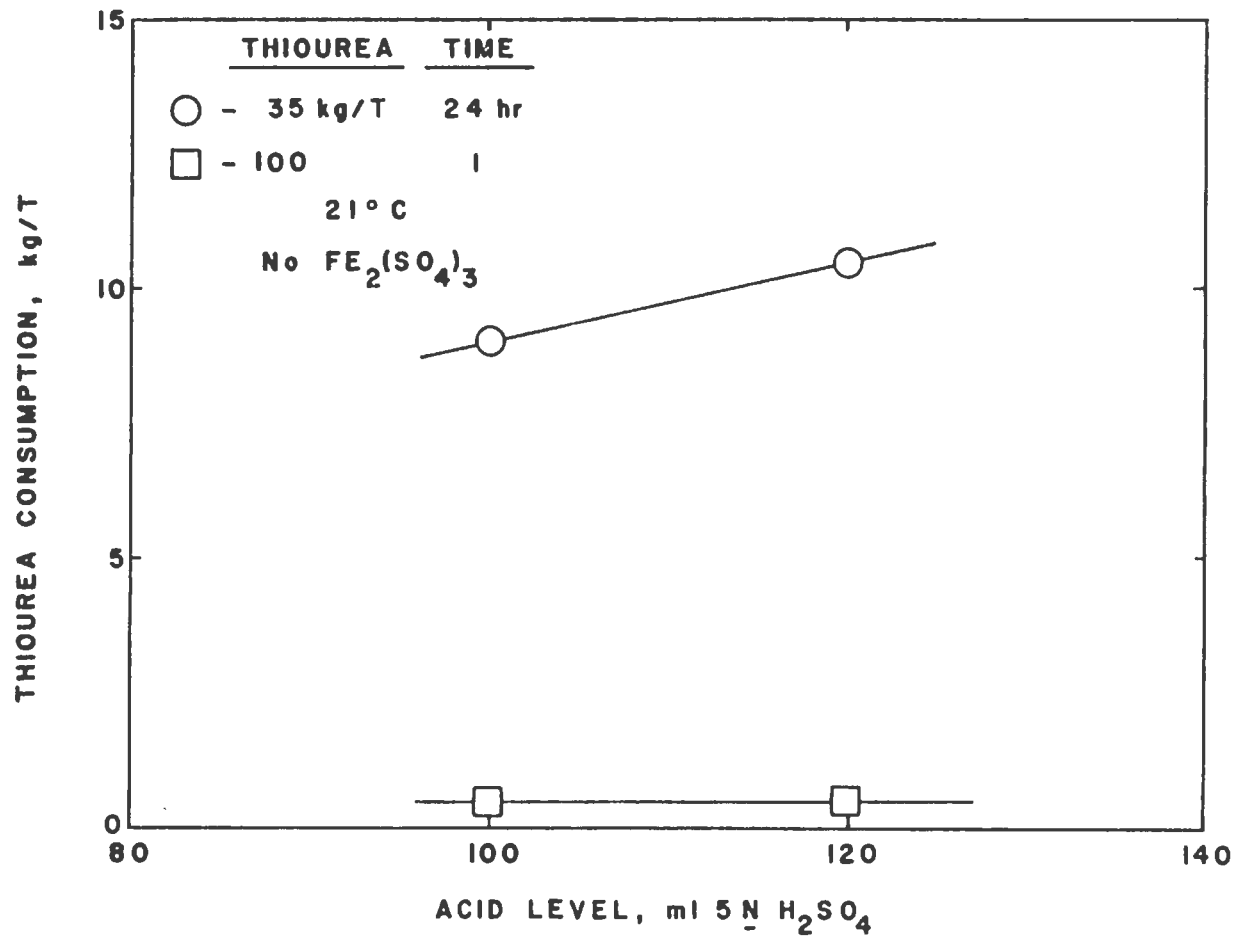


FIG. 7. THIOUREA CONSUMPTION VS. ACID LEVEL FOR CHIPMUNK ORE

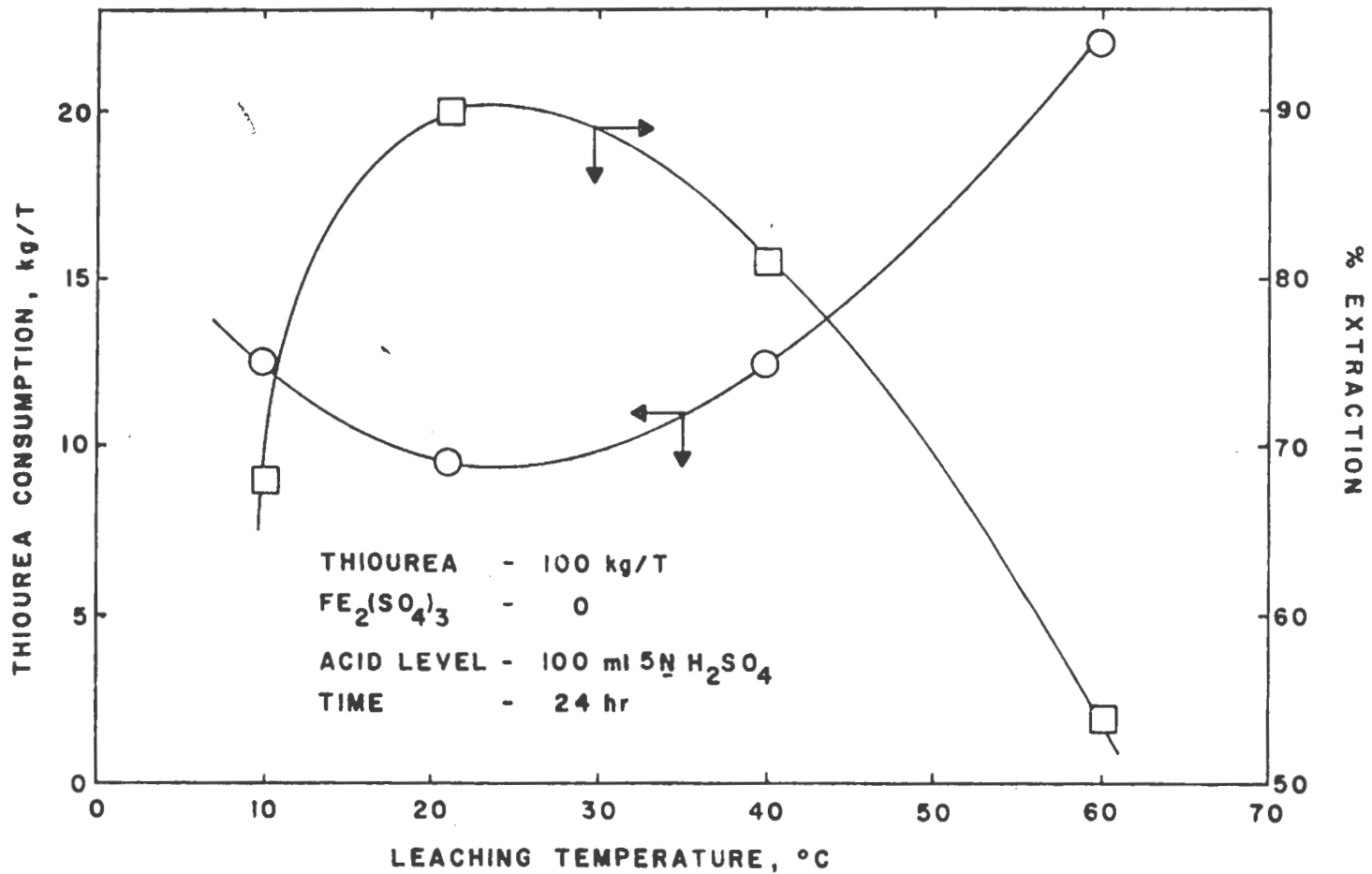


FIG. 8. THIOUREA CONSUMPTION AND PERCENT EXTRACTION VS. LEACHING TEMPERATURE FOR CHIPMUNK ORE

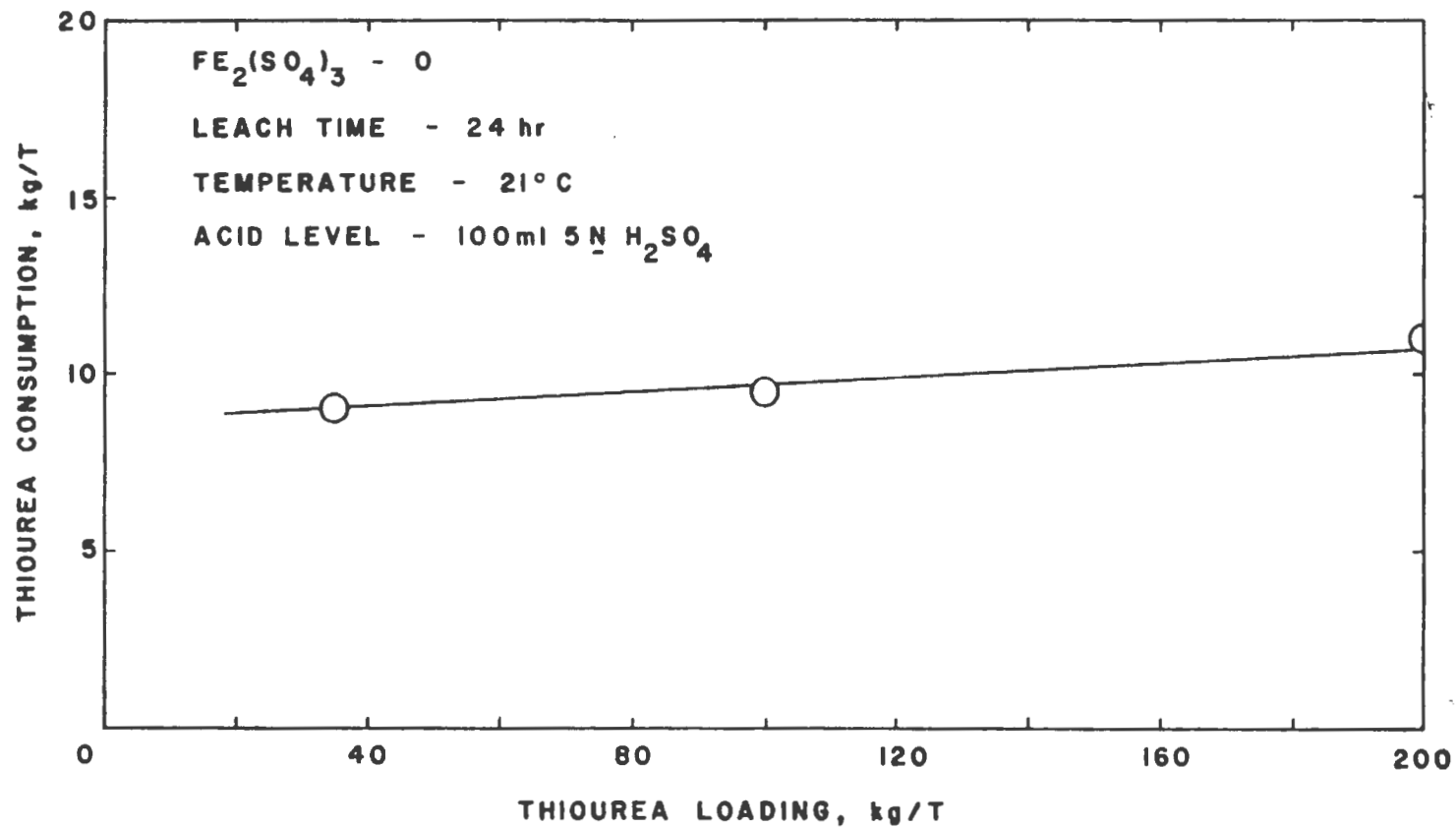


FIG. 9. THIOUREA CONSUMPTION VS. THIOUREA LOADING FOR CHIPMUNK ORE

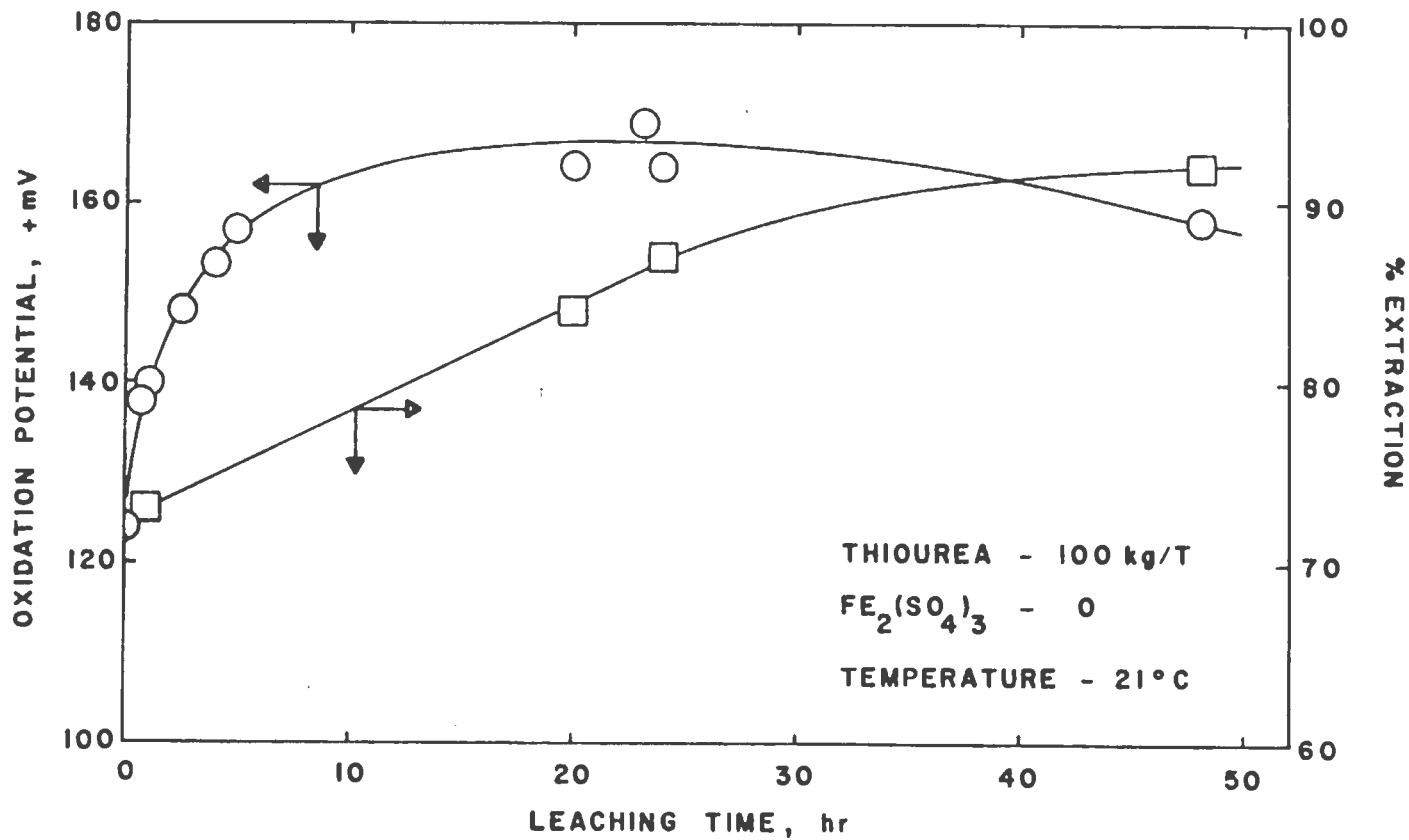


FIG. 10. OXIDATION POTENTIAL AND PERCENT GOLD EXTRACTION VS. LEACHING TIME FOR CHIPMUNK ORE

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