FLOTATION BEHAVIOR OF CASSITERITE IN A PURE SYSTEM AND IN A NATURAL ORE

by

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ABSTRACT

Micro-flotation studies were performed with cassiterite (SnO_2) and pyrite (FeS_2) to determine their flotation responses to oleate, alkyl aryl sulfonate, dodecyl sulfate, and octyl hydroxamate as collectors. These responses were compared with electrokinetic data to establish the adsorption mechanisms with each of these collectors.

The experimental data for pyrite indicate that both physical and chemical mechanisms of adsorption are occurring.

In the cassiterite system, several flotation mechanisms are also involved, i.e., coulombic attraction of the collector to the mineral surface, metal-ion activation, and chemisorption of oleic acid and hydroxamic acid after the hydroxylation of the mineral surface.

Increased temperature and conditioning time produced an increase in the flotation recovery of cassiterite.

On the basis of these results obtained with the pure minerals, the flotation response of a natural Bolivian ore with these collectors was also investigated. Because the experimental results with the ore indicated that the tin is in solid solution or is contained as a complex sulfide compound in the gangue, the amount of recoverable tin from the ore is thus limited.

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INTRODUCTION

Tin, an element of strategic importance, ranks fifth in the world output among the non-ferrous metals, behind aluminum, copper, lead, and zinc.

The main sources of tin are placer and alluvial deposits, derived from weathering and erosion of vein and disseminated replacement tin deposits. In placer deposits cassiterite is relatively free of impurities because it has had greater resistance to weathering than minerals originally associated with it. Primary tin deposits are usually associated with granites and pegmatites. In contrast to placer deposits, primary tin deposits, usually have other minerals, mostly sulfides containing antimony, iron, lead, zinc, arsenic, bismuth, copper, and silver in the veins with cassiterite.

At present, around 60 percent of the world's tin is mined in southeast Asia in countries such as Malaysia, Indonesia, and Thailand. The other main tin-producing countries are Bolivia, Nigeria, and the Republic of Congo. These nations together account for most of the western world tin production. In most of these countries the deposits are mined by dredging operations. In Bolivia, which accounts for about 16 to 17 percent of the world's tin production, the deposits are predominantly in primary lodes requiring underground mining methods and a more complex processing of the ore. The main source of tin is cassiterite (SnO_2). Minor quantities of the metal occur as complex sulfides in some regions, as in Bolivia.

Because placer cassiterite is so easily recovered by gravity, little attention has been paid to its flotation properties. The main effort of flotation research has been directed toward the lode tin deposits.

Theoretically, cassiterite should contain 78.6 percent tin, however, the mineral is impure due to solid solutions with other elements. The cassiterite ores present great difficulties to mineral processing because of the extraordinary mineral complexity.[1/] In addition to their complex

[1/]Because of this high degree of complexity in the Bolivian ores, selective flotation processes become quite complicated. Similarity in the flotation response of the materials associated with tin using conventional collectors make the separation process very difficult.

character, the surfaces of cassiterite are often contaminated by adherent coatings such as iron and inclusions of other minerals. Table 1 shows some of the Bolivian complex minerals of tin.[1]

Recently, extensive work has been done by many organizations to float cassiterite with varying degrees of success. However, up to the present none of these techniques have been applied on an industrial scale. Most of the studies have been directed toward a practical separation of SnO_2 from the ore; some of these can be found in the following references[2,3].

Table 1 - Bolivian Tin Compounds

Mineral	Composition
Canfieldite	$4Ag_2S.SnS_2$
Frankeite	$5PbS.2SnS_2.Sb_2S_3$
Herzembergite	SnS
Plumbostannite	$SnO_2.PbO$
Stannite	$SnS_2.Cu_2S.FeS$
Souxite	$SnO_2.XH_2O$
Teallite	$SnS.PbS$

Theoretical considerations of collector adsorption mechanisms involved in the flotation of cassiterite have received little attention in the past.

Recently Lapointe and Pommier[4] showed that maximum adsorption of oleic acid on cassiterite takes place in the pH range 4.0 to 6.5 with resultant optimum flotation in this range. Subsequent work showed that the pH of the pulp is not critical so long as it does not exceed a value of 8.5.

Polkin[5] concluded that uncontaminated cassiterite is more responsive to oleate flotation than many other oxide minerals, including hematite, magnetite, zircon, and tantalite. The presence of impurities in the crystal lattice increased the density of the collector layer but reduced recovery.

The purpose of this thesis is to study the possibilities and problems encountered in the flotation of cassiterite from a complex sulfide ore. For this purpose the work has been divided into two parts.

In the first part the flotation responses of cassiterite and pyrite have been studied independently. That is to say, in each case cassiterite and pyrite were run with the same collector concentration and the same conditions, except for the hydroxamate system in which the interaction of this collector with cassiterite was studied in more detail.

Pyrite response to these collectors was studied in detail since it is one of the most common minerals associated with cassiterite.

The second part of this investigation considers the flotation separation of cassiterite from the ore. This study is based upon the experimental results obtained from the pure system.

PART I - EXPERIMENTS IN THE PURE SYSTEMS

In this part of the investigation, flotation experiments were run with pyrite and cassiterite independently. The flotation responses of these minerals with each collector are analyzed and compared in order to establish the possibility of a selective separation in the natural ore.

EXPERIMENTAL MATERIALS AND METHODS

The following materials and methods were used in this investigation:

Minerals

Pure crystals of cassiterite and pyrite were used in this investigation. The purity of each mineral was confirmed by x-ray diffraction. Some of the distinguishing characteristics of these minerals are given below.

Mineral	Hardness (Mohs Scale)	Chemical Composition	Crystallography	Specific Gravity
Cassiterite	6 - 7	SnO_2	Tetragonal	6.80
Pyrite	6 - 6½	FeS_2	Cubic	5.02

Sample Preparation

Grinding the crystals of these minerals by hand with a porcelain mortar and pestle avoided sample contamination. The ground minerals were sized to 48 x 100 mesh by dry screening.

For the surface-charge-determination experiments, the minerals were ground to approximately 5 microns.

Water

Conductivity water, prepared by passing distilled water through a column of anionic-cationic exchange resin (Amberlite MB-3) was used for the experimental work.

Reagents

The following collectors were used:

Name	Formula	Abbreviated Formula
Oleic acid	$C_{17}H_{33}$-COOH	HOl
Sodium dodecyl sulfate	$C_{12}H_{25}OSO_3Na$	$NaRSO_4$
Potassium octyl hydroxamate	HO-C-R ‖ KO-N	KHA

The exact formula for the type of alkyl aryl sulfonate used in this experiment is not precisely known, but this reagent has the following properties: Sulfonate content 95 to 97 percent by weight; molecular weight 450 to 470; total carbon atom content 25 to 27 atoms [6].

Reagent grade of both hydrochloric acid and potassium hydroxide were each used interchangeably for pH adjustment. Stannous ion was added to the systems as the chloride salt.

Flotation Apparatus

Micro-flotation studies were conducted with a modified Hallimond Tube [7,8,9].

The modified Hallimond tube has a fritted glass bottom (with a 60-micron average pore size) for gas entrance. The Hallimond cell used during this investigation was constructed to hold 150 ml of pulp (Figure 1). Nitrogen was purified by being passed through Ascarite to remove carbon dioxide before the nitrogen entered the bottom of the tube through the glass frit. Admittance of the gas to the system was controlled with a regulating valve, a water manometer, a constant head reservoir, and a gas measuring burette (by water displacement).

These are the main advantages of this type of equipment:

(a) Operating variables such as flotation time, gas flow rate, and agitation rate can be controlled accurately;

(b) Samples of 1 to 3 grams can be floated;

(c) Reagent concentration and amount of solution are kept constant during each test; and

(d) Contamination is minimal for no metallic parts are involved in the flotation system. A schematic diagram of the micro-flotation cell appears in Figure 2.

Zeta Meter

Zero-points-of-charge were determined for each of the minerals used in the investigation. The surface charge of the samples in conductivity water were measured with a Zeta Meter, supplied by Zeta-Meter, Inc. The apparatus consists of a microscope-cell combination and a control panel for applying a DC voltage.

The electrophoresis cell, with the mineral suspension placed into it, is set on the stage of the microscope. The suspension is used as the electrolyte, and a known

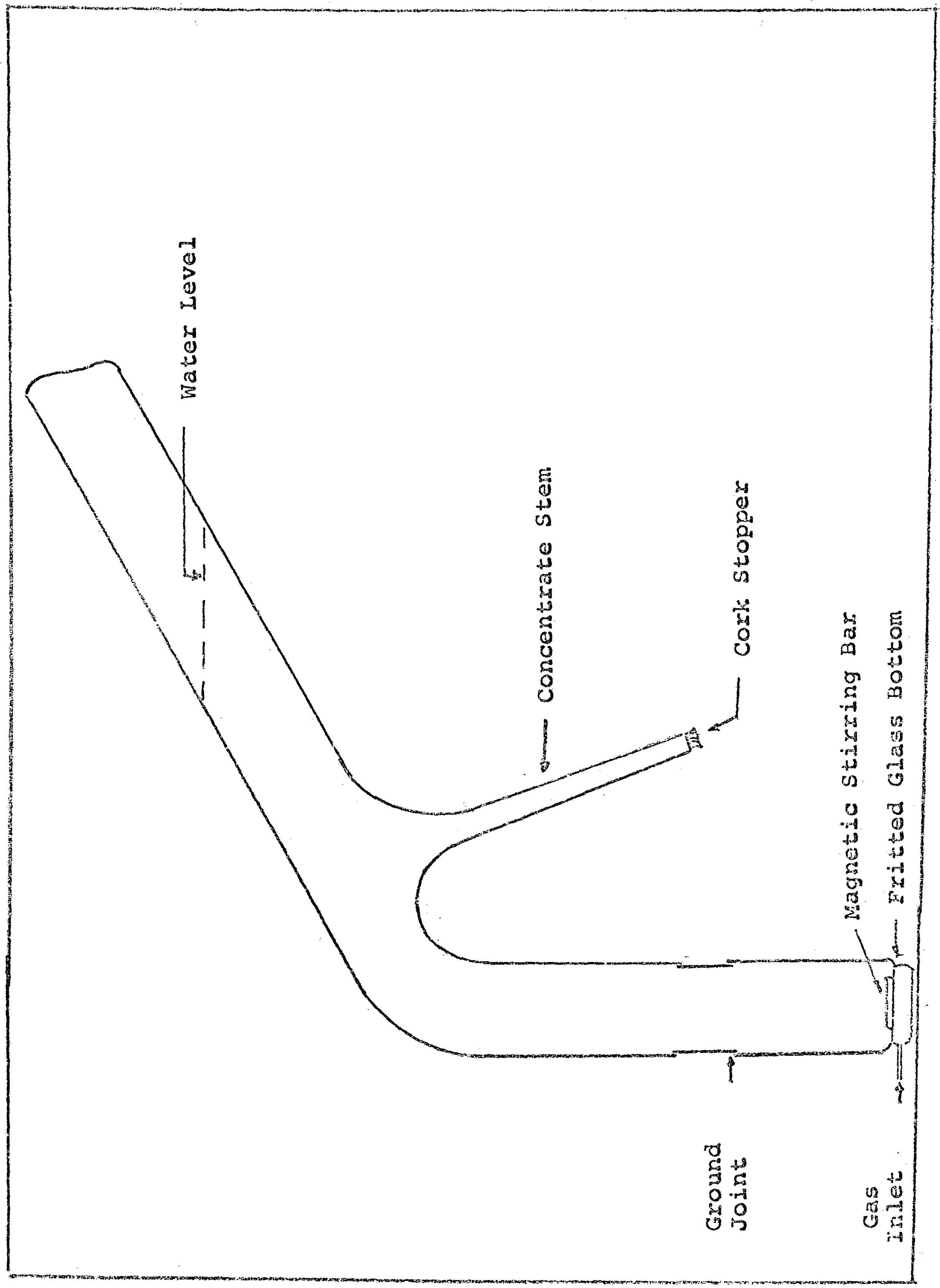

Figure 1. Schematic diagram of the modified Hallimond tube.

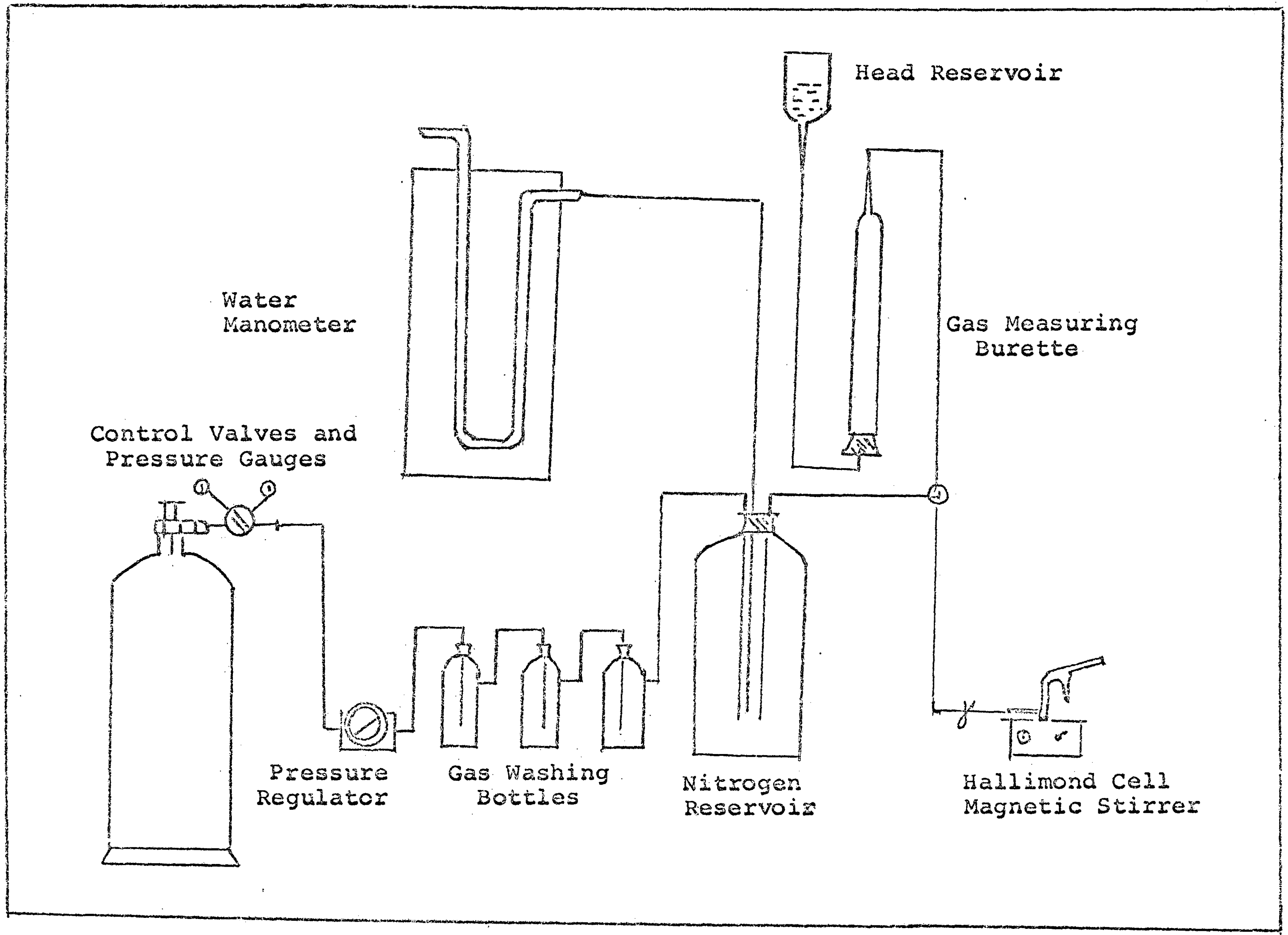

Figure 2. Schematic diagram of the micro-flotation apparatus.

voltage is applied to the electrodes of the cell. The movement of the particles is observed by means of a scale in the ocular of the microscope.

PROCEDURE

The investigation was conducted in two separate phases: 1) flotation experiments, and 2) electrophoretic experiments.

Flotation Experiments

The flotation experiments were conducted in the following manner:

1. A predetermined volume of conductivity water was combined with the necessary volumes of collector solution and modifying reagents in a 250-ml beaker.
2. The pH of the system was adjusted to the desired value with hydrochloric acid or potassium hydroxide.
3. One gram of the sized mineral was deslimed and then added to the solution.
4. The conditioning times and the sequence of additions of modifying reagents and collector were as follows:
   (a) A cassiterite flotation curve was run as a function of conditioning time, at constant pH and collector concentration.
   (b) For the stannous chloride-activated cassiterite experiments, the $SnCl_2$ solution (pH 2.6) was added to the beaker containing the collector-mineral solution which had previously been adjusted to the desired pH value, and then conditioned for five minutes.

(c) Also a flotation curve was run at constant 50°C bath temperature, and at a constant pH value of 8.0. The pulp for each test was also conditioned for five minutes.

(d) The rest of the experiments of cassiterite and pyrite were run as a function of pH and conditioning time.

5. The pH of the system was measured at the end of the conditioning time. This was considered the flotation pH.
6. The pulp was then transferred to the Hallimond cell, and 30 ml of purified nitrogen was passed in 60 seconds.
7. The pH of the pulp was re-measured and the value termed final pH.
8. The flotation products were dried and weighed, and the recoveries were calculated.

Electrophoresis Experiments

The required particle size for electrophoretic measurements is about 5 microns. This particle size was obtained by the following procedure:

1. A small amount of the sample was ground for two hours with a porcelain mortar and pestle.
2. A suspension of 300 ml was made with two grams of this finely-ground sample and conductivity water.
3. The suspension was allowed to settle for 30 minutes (see appendix).

4. 100 ml of the upper part of the solution which contained the required particle size was removed.
5. 6 ml taken from the solution of part 4 was used for each electrophoretic measurement.
6. In each case the sample taken for each electrophoretic measurement was adjusted to the desired pH value.
7. The Zeta-Meter cell was filled with an aliquot of the suspension.
8. Ten individual particles were timed in their traverse of one micrometer division (160 microns) under a known applied EMF.
9. The polarity switch was reversed and ten more particles were timed during movement in the opposite direction.
10. The pH of the suspension was measured again, and an average of the initial and final readings was taken as the experimental value.
11. From the electrophoretic movement of the particles, zeta potentials were determined by means of tables based upon the Helmholtz-Smoluchovsky equation. This equation is as follows:

$$\zeta = \frac{4\pi\mu E.M.}{D_t}$$, where

ζ = zeta potential in volts

μ = viscosity of the fluid in dyne-sec/cm^2

D_t = dielectric constant of the fluid in coul/volt-cm

$= 80 \times 1.112 \times 10^{-12}$ for water;

E.M. = electrophoretic mobility in cm^2/sec-volt

E.M. = v/V

v = velocity of the suspended particle, in cm/sec

V = voltage per cm length of fluid

RESULTS

The following flotation and electrophoretic results were obtained in the course of this investigation:

Flotation Experiments

The experimental results obtained in this part of the study show the flotation response of cassiterite and pyrite as a function of pH and collector concentration. The data also show the flotation response of cassiterite as a function of temperature, conditioning time, and stannous chloride additions.

An explanation of the results follows:

Cassiterite-Oleate System. The flotation recovery of cassiterite as a function of pH for a collector addition of 1×10^{-4} mole per liter oleate is depicted in Figure 3. Complete flotation occurred over the pH range from 6.4 to 9.1. Two regions of depression were noted at pH 5.8 and 9.7.

Pyrite-Oleate System. The effect of oleate addition and pH on pyrite flotation were also examined. Figure 4 [2/] shows the flotation recovery of pyrite at 2×10^{-5} mole per liter potassium oleate.

Complete flotation occurred over the pH range from 2.2 to 8.0.

[2/] The flotation curve depicted in Figure 4, was taken from: Elgillani, D. A., Mechanisms Involved in Cyanide Depression of Pyrite: Golden, Colorado, Colorado School of Mines, Doctoral Thesis 1181, p. 14 (1968).

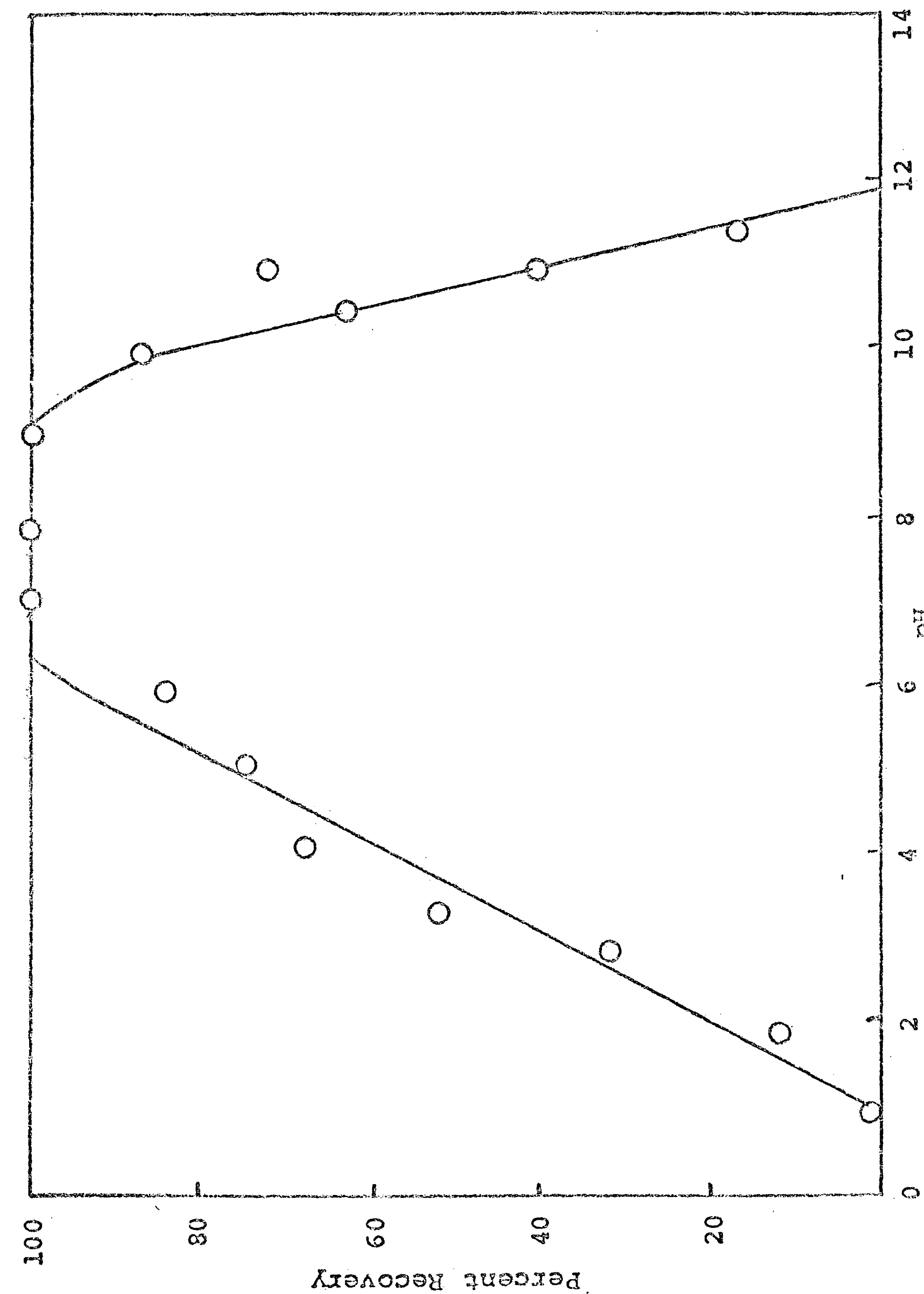

Figure 3. Flotation recovery of cassiterite as a function of pH. Kol 1×10^{-4} mole per liter.

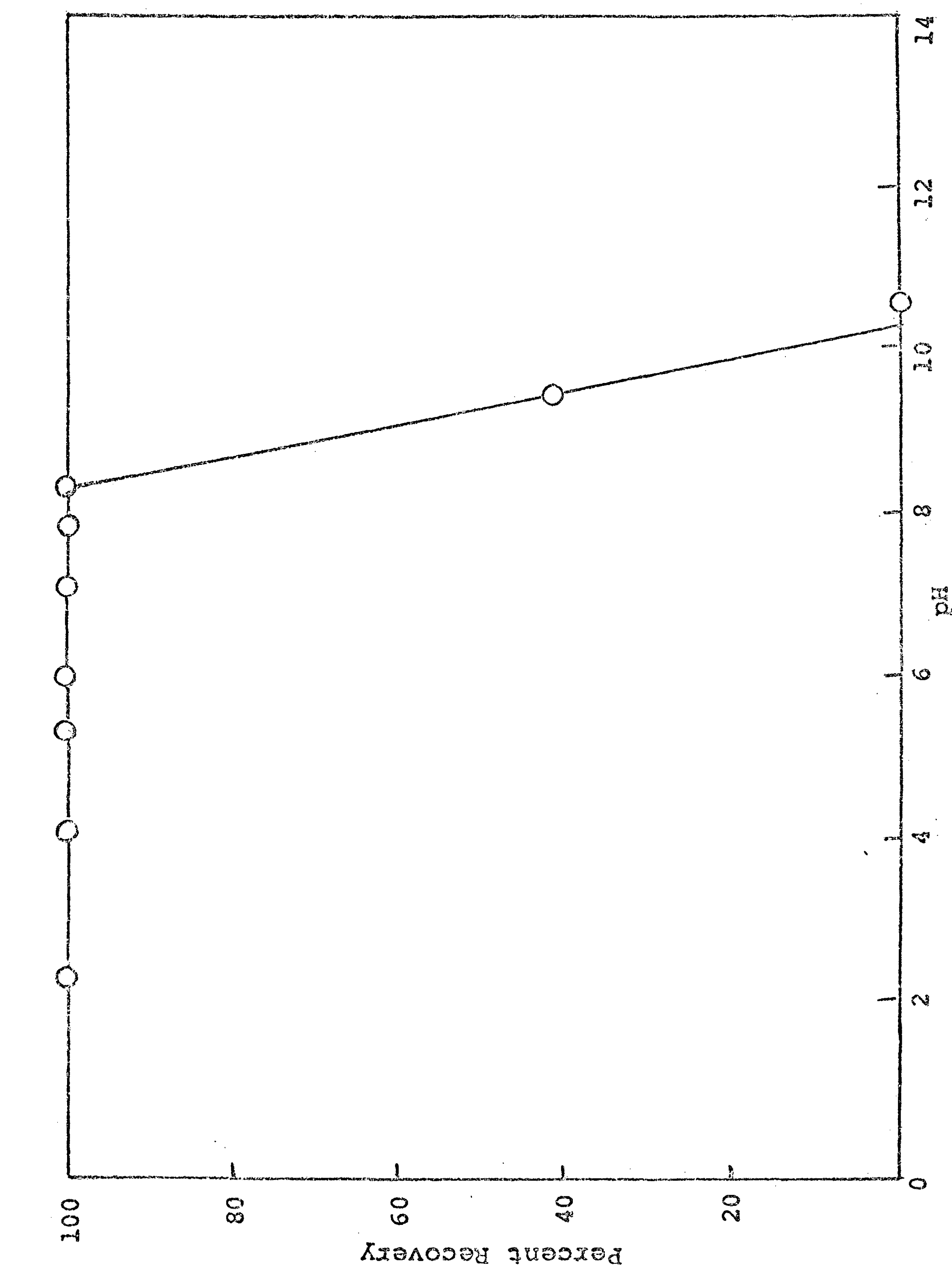

Figure 4. Flotation recovery of pyrite as a function of pH. Kol $2x10^{-5}$ mole per liter.

Cassiterite-Sulfonate System. The sulfonate concentration and pH were varied in this system. The results are shown in Figure 5.

With a concentration of 1×10^{-4} mole per liter sulfonate, complete flotation was obtained in the pH range 1.8 to 7.6. Two regions of depression were noted at pH 1.5 and pH 8.6.

With a concentration of 1×10^{-5} mole per liter sulfonate, recovery and flotation pH range decreased considerably. Maximal recovery of only about 38 percent was obtained. The two regions of depression were noted at pH 1.5 and 6.0.

Pyrite-Sulfonate System. Figure 6 shows the effect of sulfonate additions as a function of pH. For concentrations of 1×10^{-5} mole per liter sulfonate the flotation of pyrite occurred only below its zpc (pH 6.9). It was also noted that as the concentration of the collector was increased the pH range of flotation was broadened. With additions of 5×10^{-5} and 1×10^{-4} mole per liter sulfonate, complete flotation was achieved in the pH region from 1.8 to 8.4.

Cassiterite-Dodecyl Sulfate System. As shown in Figure 7, the flotation response of cassiterite to pH is rather sensitive. With an addition of 1×10^{-4} mole per liter dodecyl sulfate, optimum flotation recovery was obtained only between pH values of 2.3 and 4.4. The flotation response of cassiterite to dodecyl sulfate decreases below pH 2.0 and above pH 5.0.

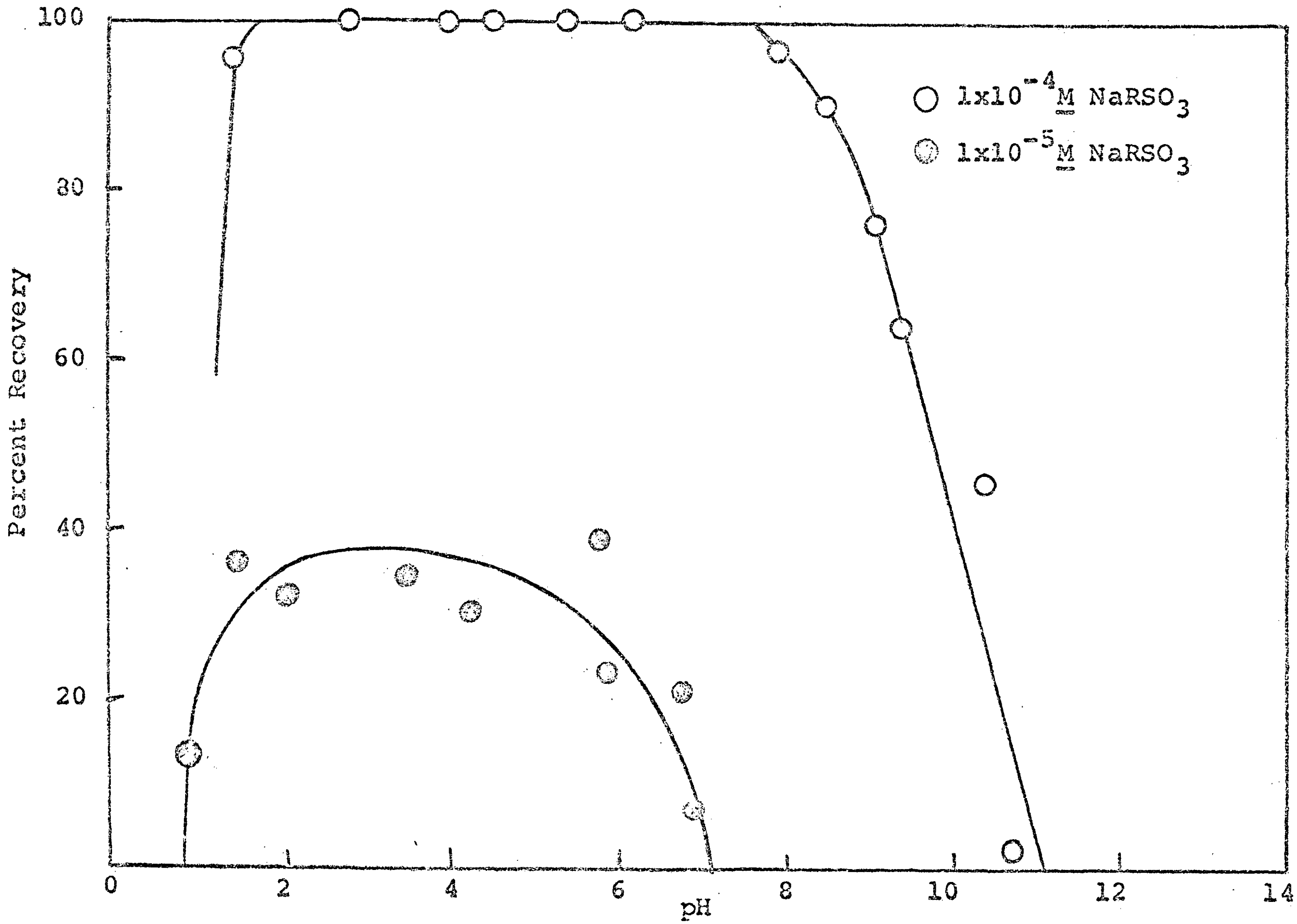

Figure 5. Flotation recovery of cassiterite as a function of pH, in the presence of 1×10^{-4} and 1×10^{-5} mole per liter $NaRSO_3$.

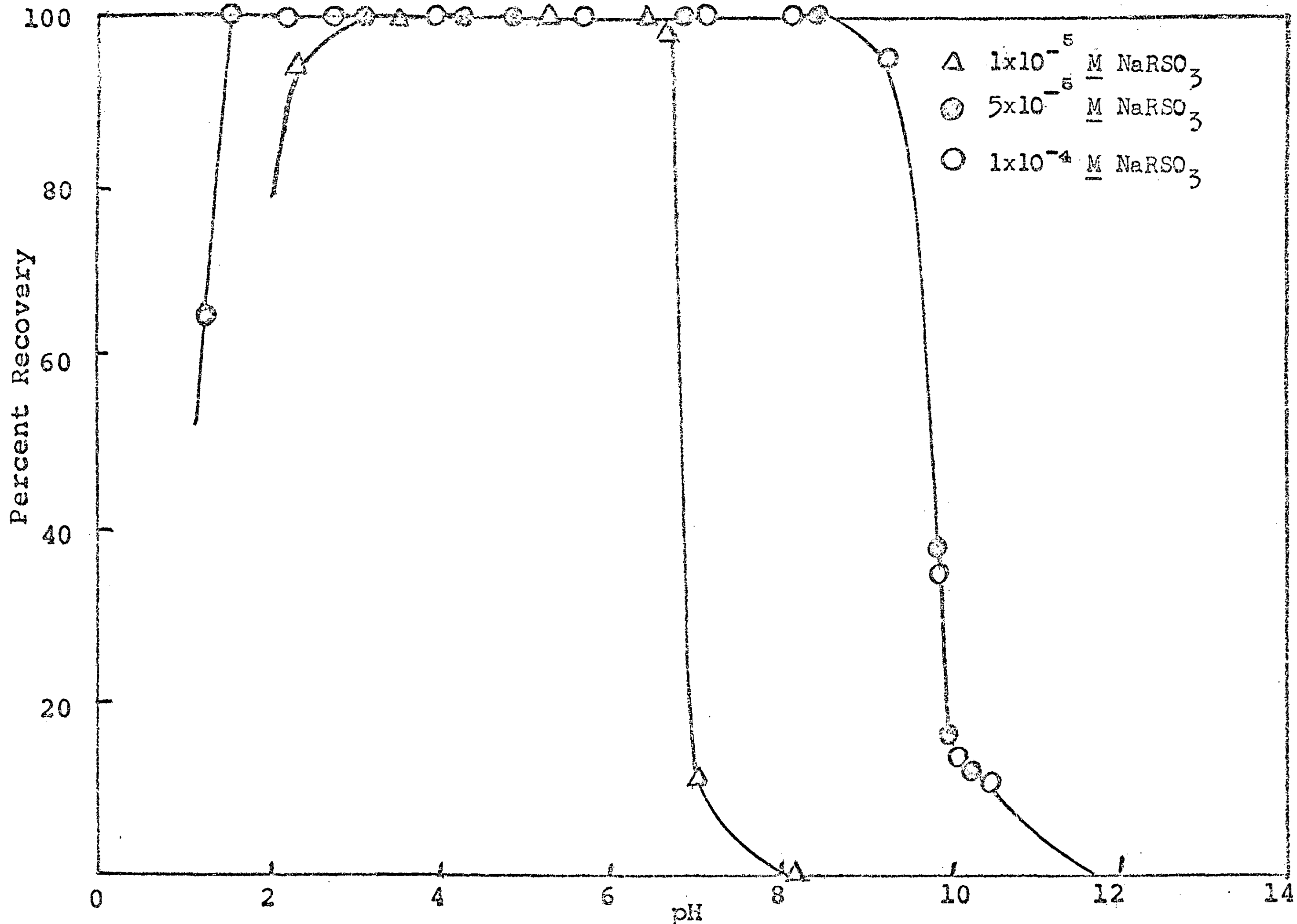

Figure 6. Flotation recovery of pyrite as a function of pH in the presence of $1x10^{-5}$, $5x10^{-5}$, and $1x10^{-4}$ mole per liter $NaRSO_3$.

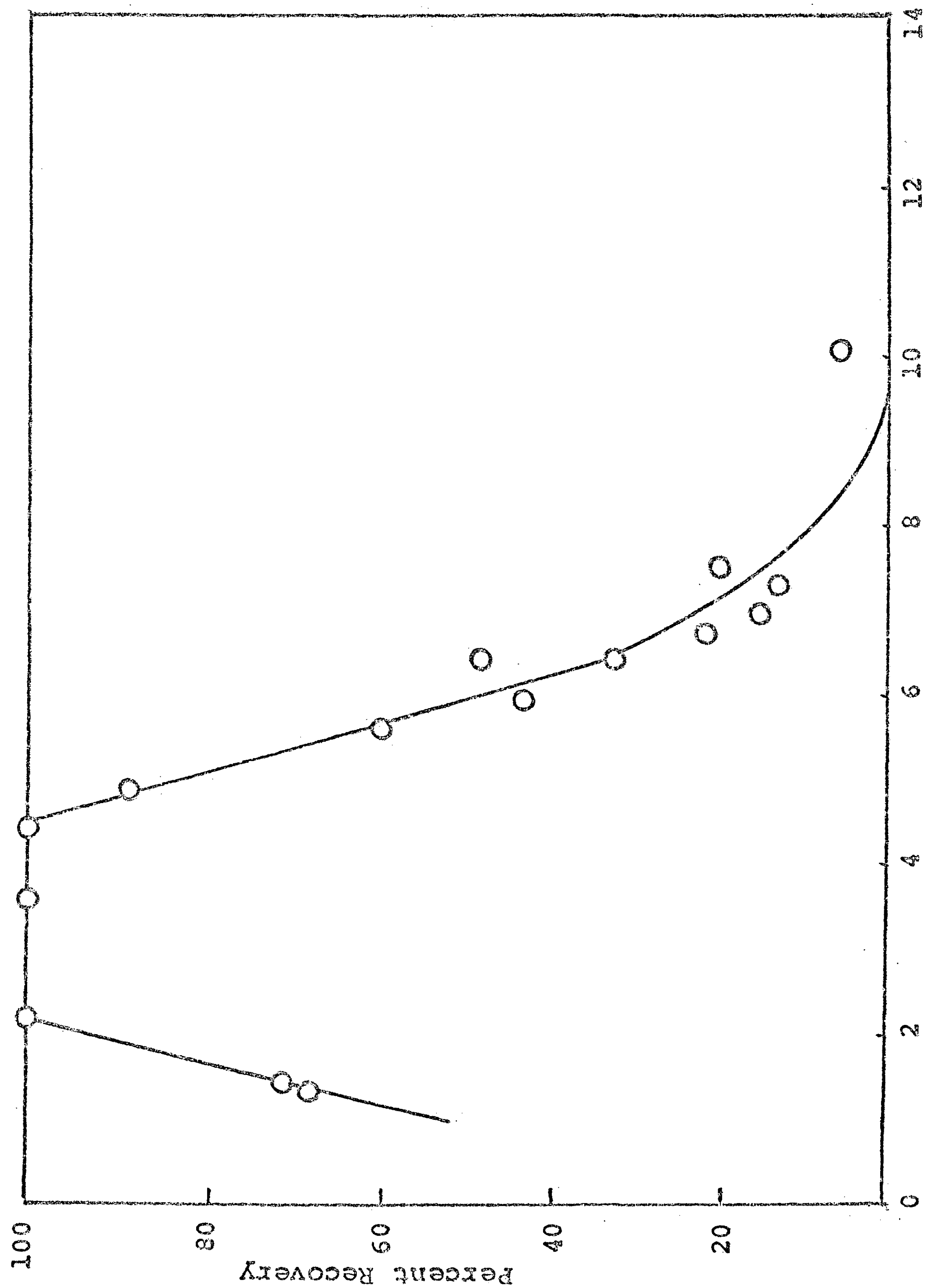

Figure 7. Flotation recovery of cassiterite as a function of pH. $NaRSO_4$ $1x10^{-4}$ mole per liter.

Pyrite-Dodecyl Sulfate System. The effect of dodecyl sulfate with pyrite is shown in Figure 8. The comparison of the flotation response of pyrite to that of cassiterite in Figure 7 shows that pyrite floated in a wider pH range. At pH 7.3 with the pyrite system, optimum flotation still was obtained, whereas in the cassiterite system at the same pH only 18 percent recovery occurred.

Cassiterite-Octyl Hydroxamate System. The flotation response of cassiterite as a function of pH and collector additions are shown in Figures 9, 10, and 11, for collector additions of 1×10^{-4}, 3×10^{-4}, and 5×10^{-4} mole per liter octyl hydroxamate, respectively. It is seen in Figure 9, a maximal recovery of 45 percent was obtained between pH 6 and 7. Complete depression was noted below pH 4.0 and 8.2. As plotted in Figure 10, the flotation recovery was increased to 70 percent between pH 7.5 and 8.5. Complete depression was observed below pH 4.0 and 10.5. Finally, Figure 11 shows 100 percent flotation at pH 8.0.

In another series of experiments, the flotation behavior of stannous-activated cassiterite was considered as a function of pH for a constant collector addition of 3×10^{-4} mole per liter octyl hydroxamate (same as in Figure 10), and 1×10^{-5} mole per liter stannous chloride. The flotation response of cassiterite was increased from 70- to 100-percent recovery between pH values 7.3 to 8.2.

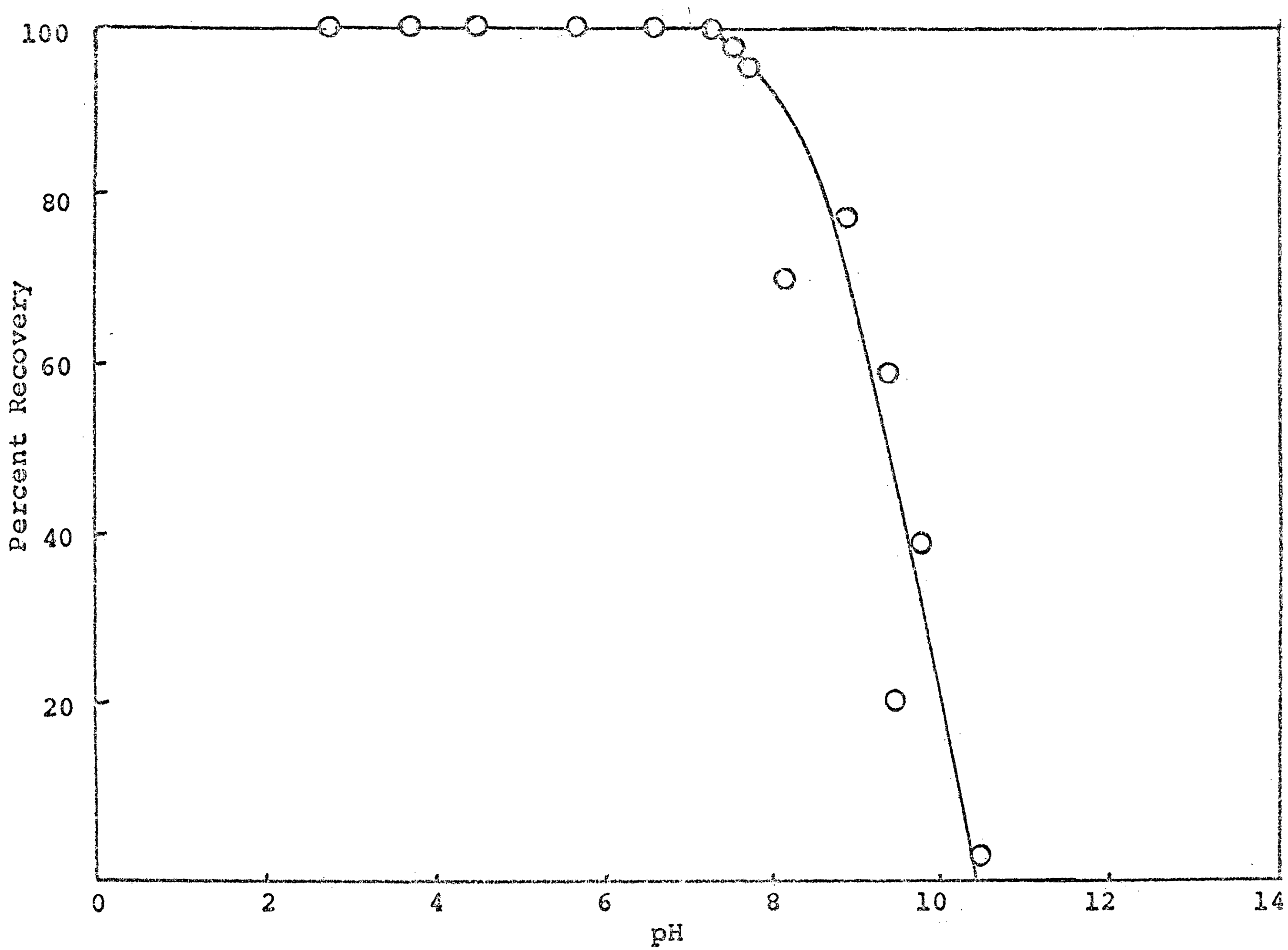

Figure 8. Flotation recovery of pyrite as a function of pH.
$NaRSO_4$ 1×10^{-4} mole per liter.

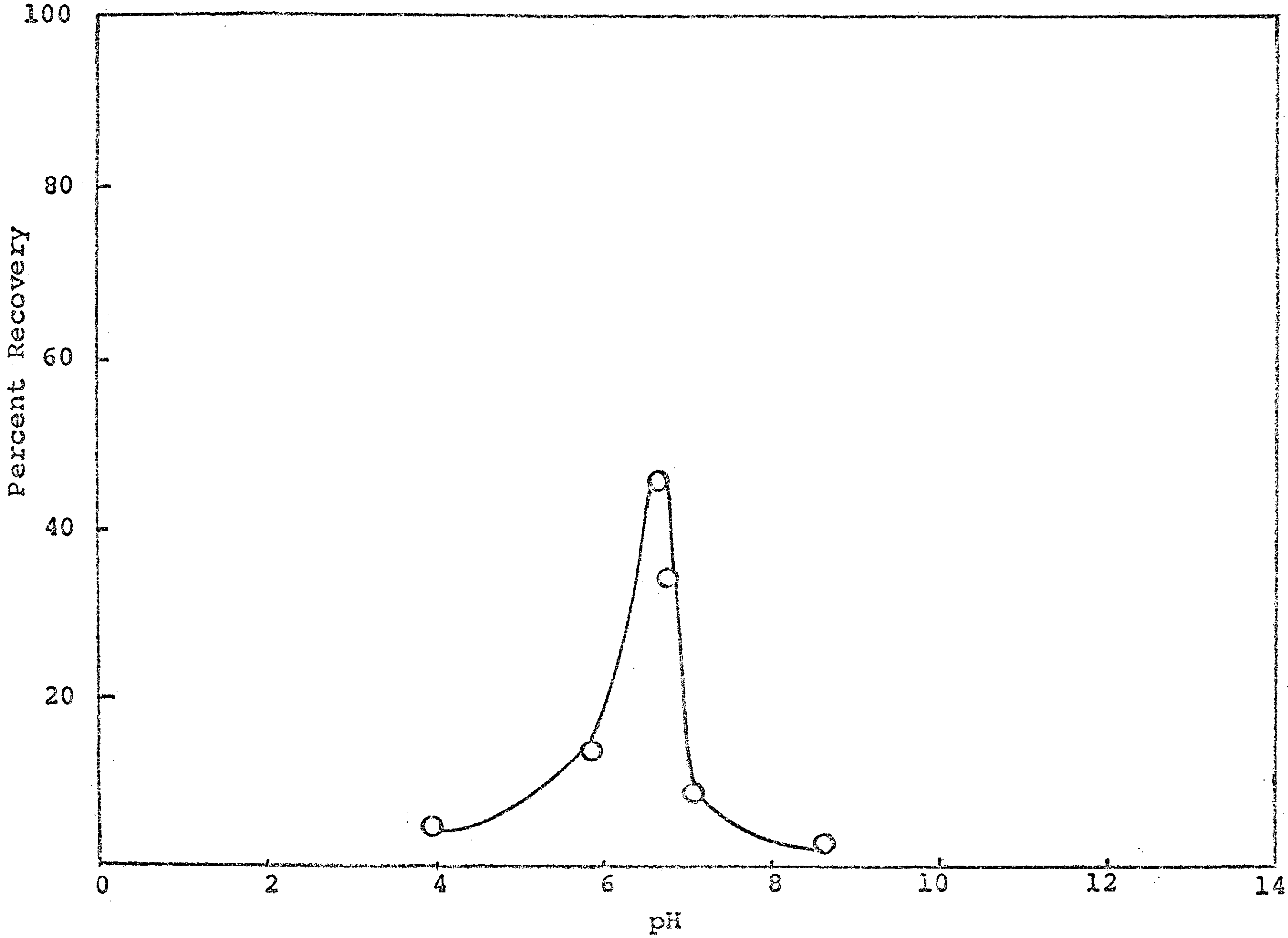

Figure 9. Flotation recovery of cassiterite as a function of pH. KHA 1×10^{-4} mole per liter.

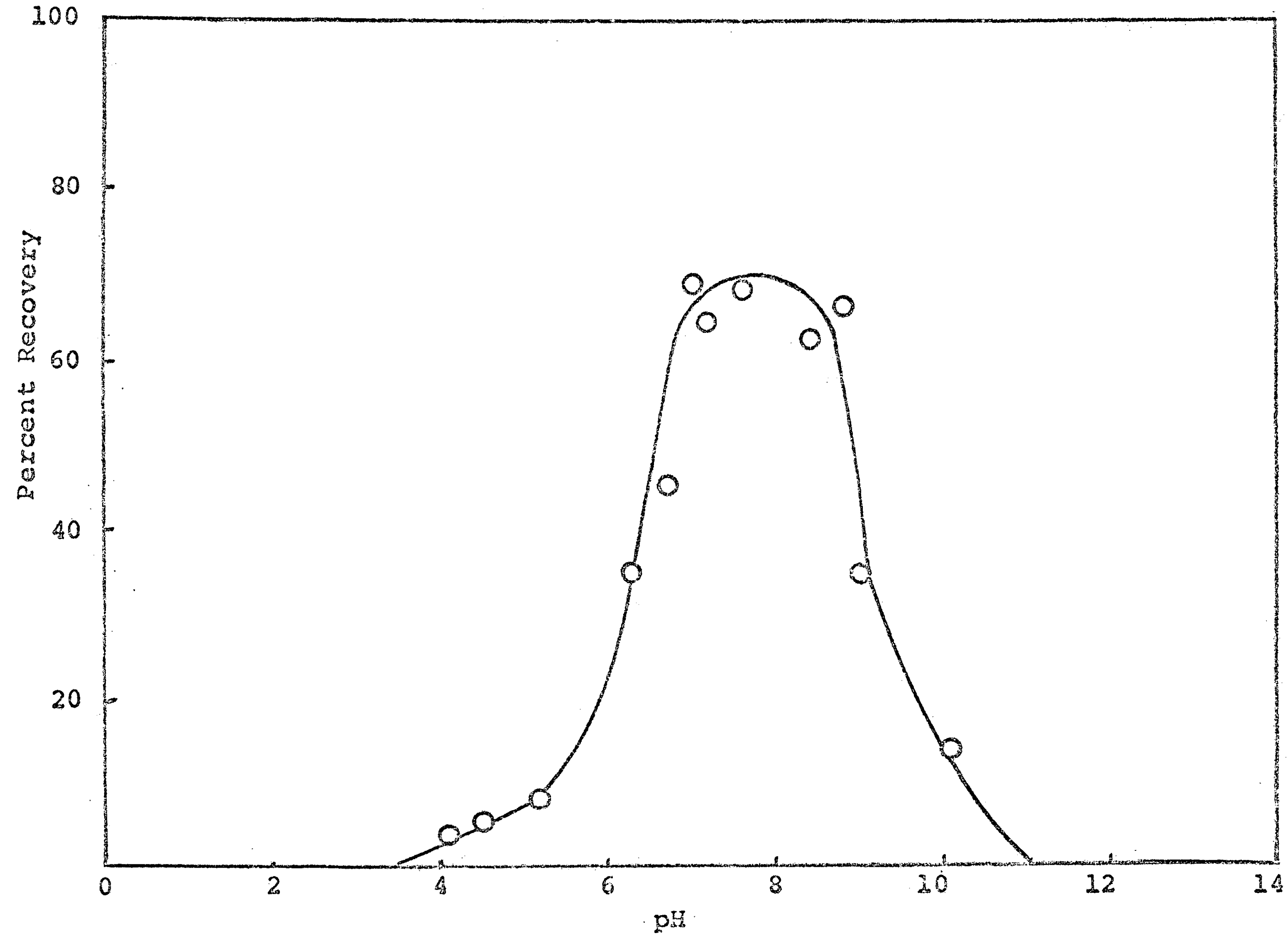

Figure 10. Flotation recovery of cassiterite as a function of pH. KHA $3x10^{-4}$ mole per liter.

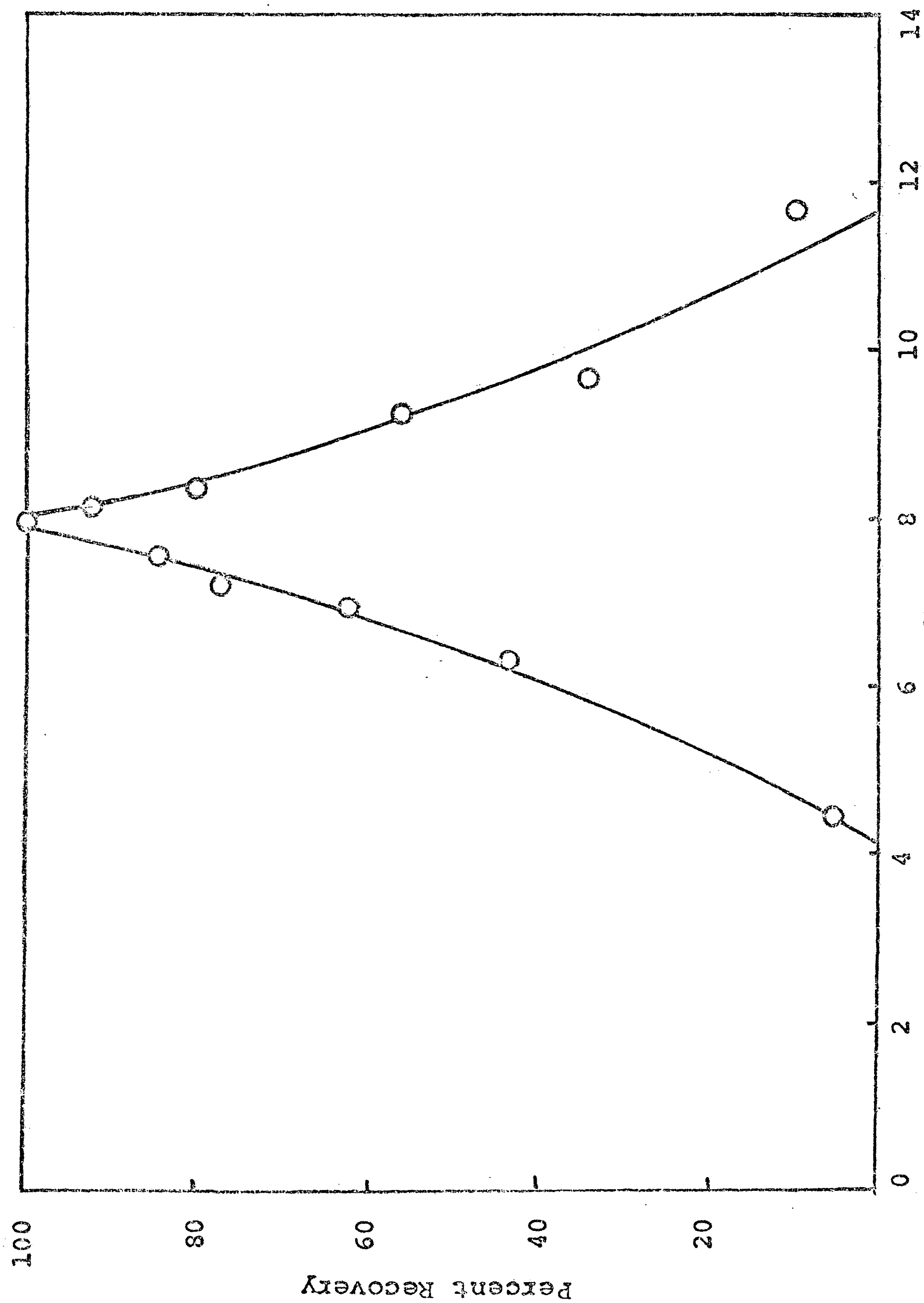

Figure 11. Flotation recovery of cassiterite as a function of pH. KHA 5×10^{-4} mole per liter.

The effect of conditioning time on the flotation recovery of cassiterite was also studied with the use of the same collector concentration as in Figure 12, and at constant pH value of 8.0. Figure 13 shows the flotation response of cassiterite as a function of conditioning time. Below 20 minutes of conditioning time, the recovery remained constant at about 70 percent. At 25 and above 28 minutes 81- and 100-percent recoveries were obtained, respectively.

Finally a separate experiment was undertaken to ascertain the possibility of increasing the flotation recovery at elevated temperatures of 50°C. This experiment was run as a function of pH and also, as in the last two cases, a constant collector concentration of 3×10^{-4} mole per liter octyl hydroxamate was added. As depicted in Figure 14, the flotation recovery of cassiterite at 50°C also shows 100% recovery in the pH range from 6.8 to 8.7. The two regions of depression were observed below pH 4 and 9.8.

Pyrite-Octyl Hydroxamate System. The flotation recovery of pyrite as a function of pH for collector additions of 1×10^{-4} and 5×10^{-4} mole per liter octyl hydroxamate is shown in Figure 15. For both collector additions good flotation of pyrite occurred. The flotation results of this mineral compared to that of cassiterite (Figures 9, 10, and 11) were markedly different when the same collector was used. In the pyrite system at pH values below 4, and about 10.5,

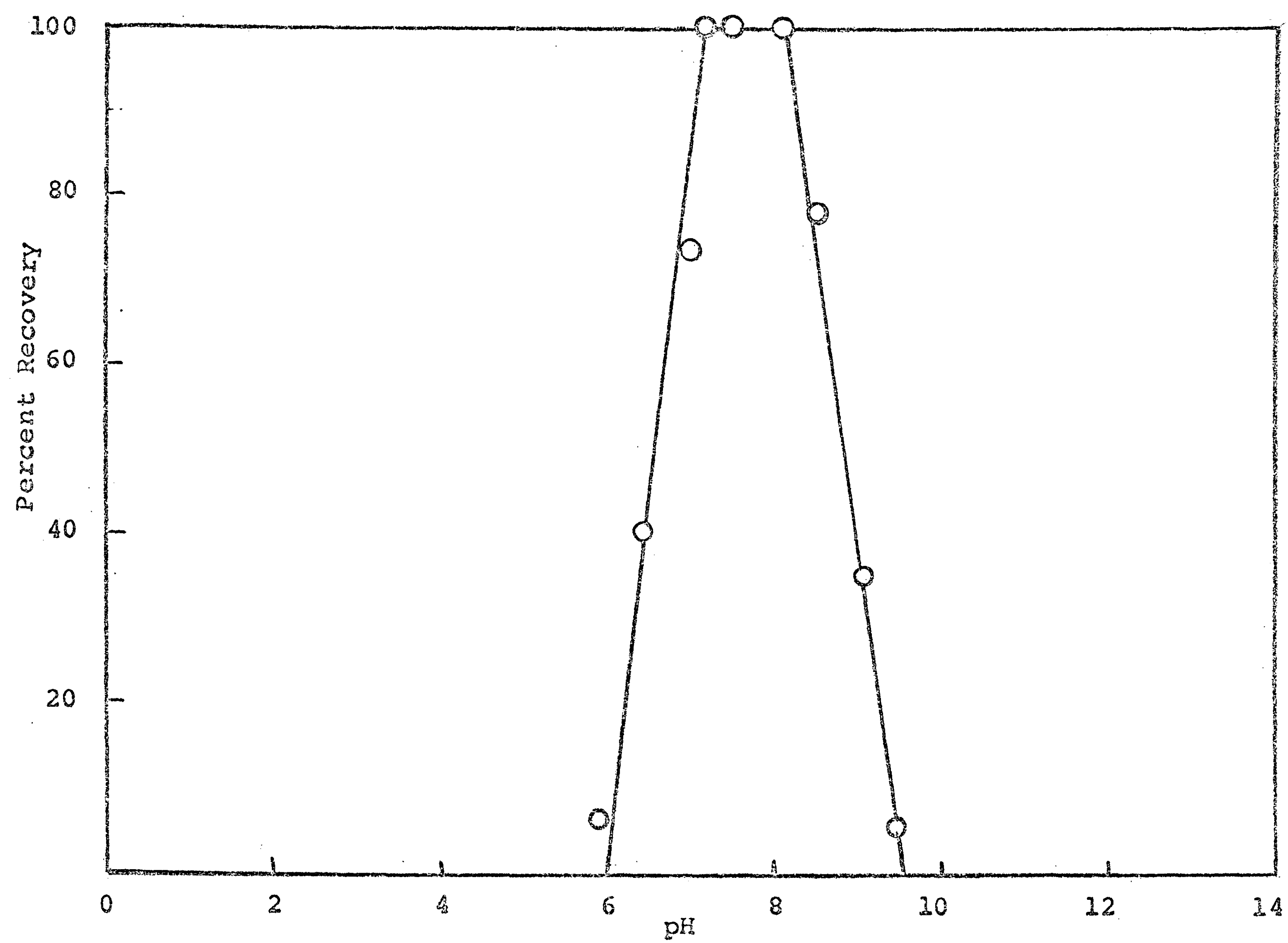

Figure 12. Flotation recovery of cassiterite as a function of pH for additions of 1×10^{-5} mole per liter $SNCl_2$ and 3×10^{-4} mole per liter KHA.

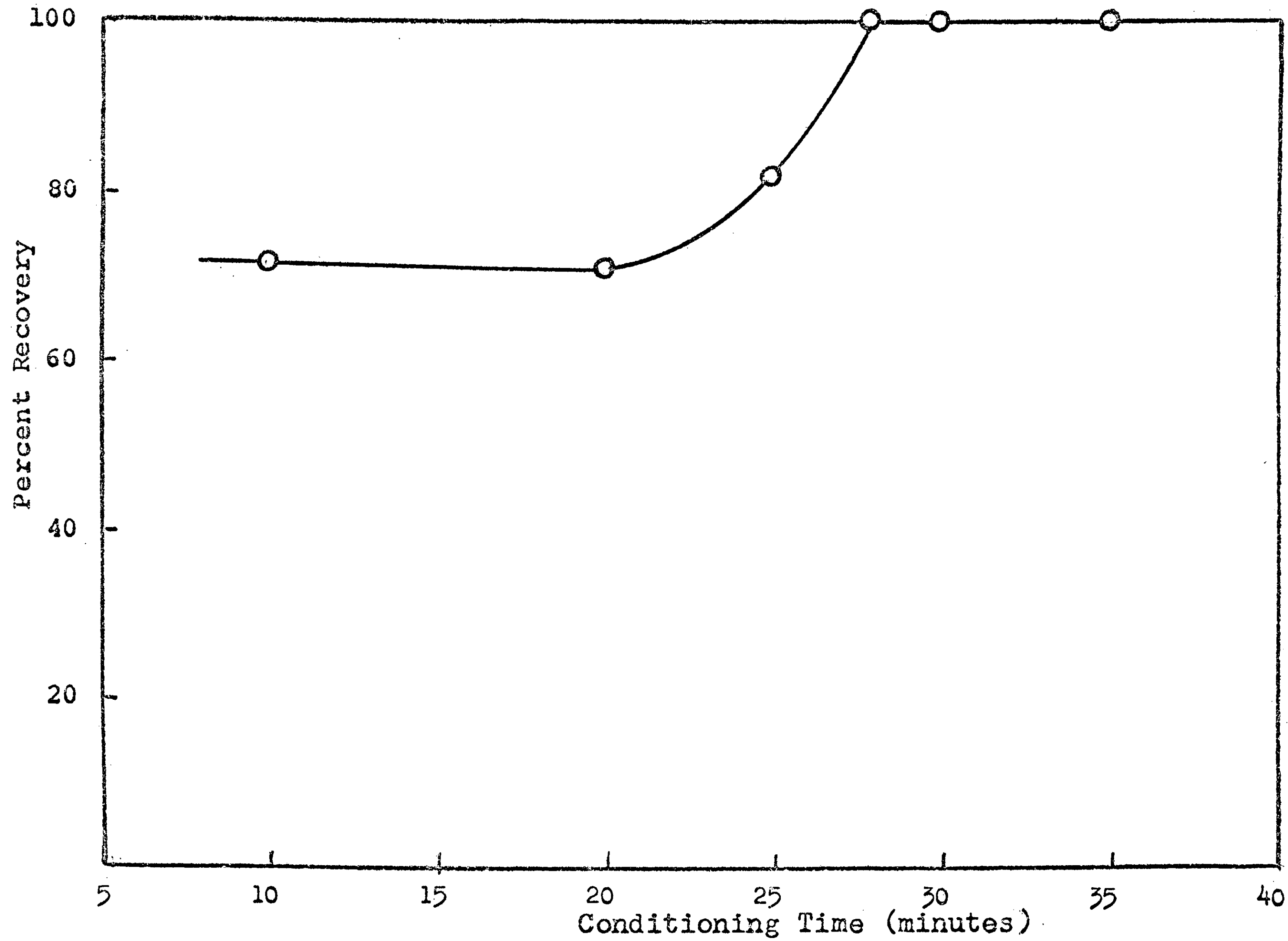

Figure 13. Flotation recovery of cassiterite as a function of conditioning time at constant pH 8.0 KHA $3x10^{-4}$ mole per liter.

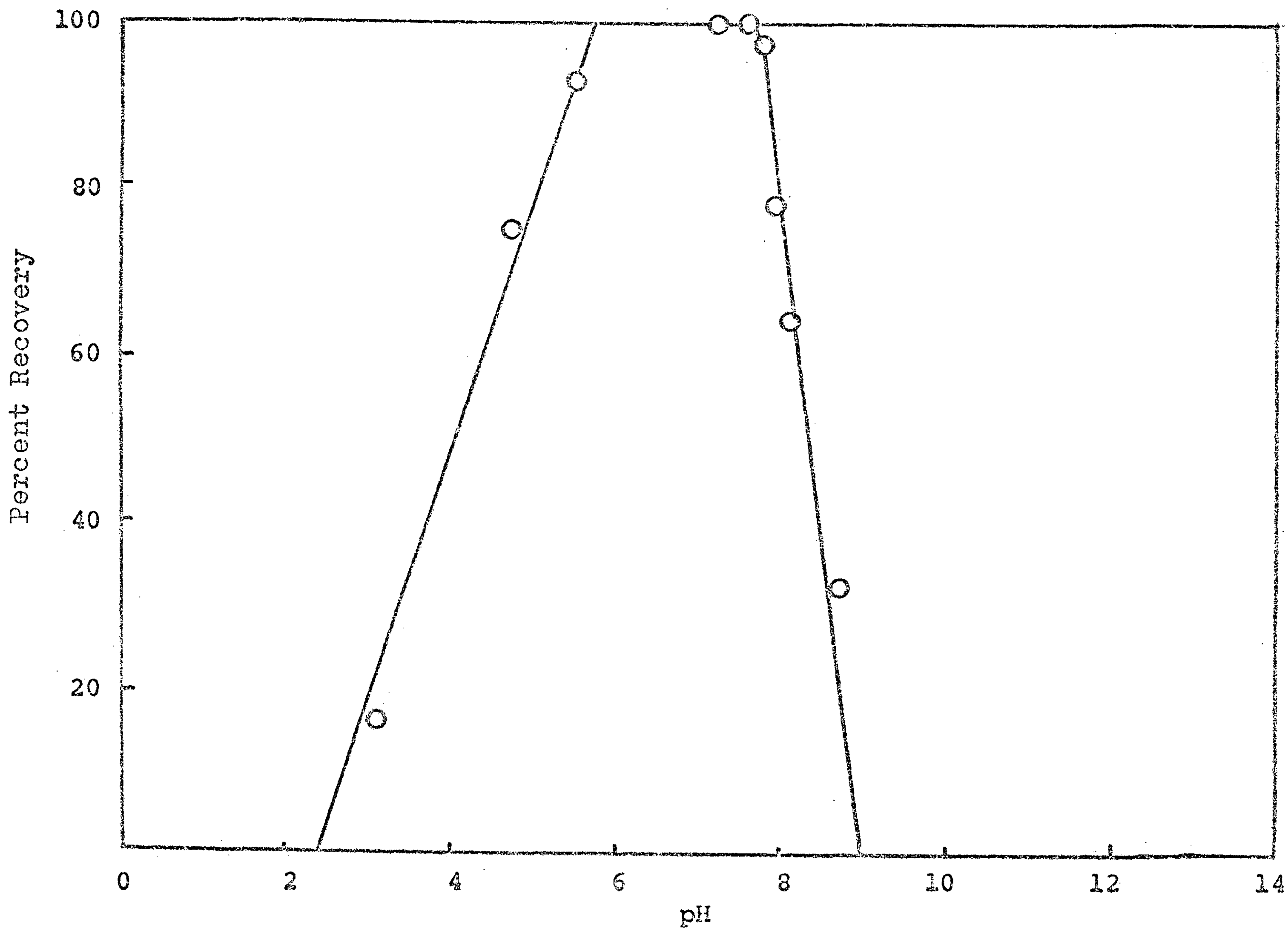

Figure 14. Flotation recovery of cassiterite as a function of pH in the presence of 3×10^{-4} mole per liter KHA and at constant $50^{\circ}C$.

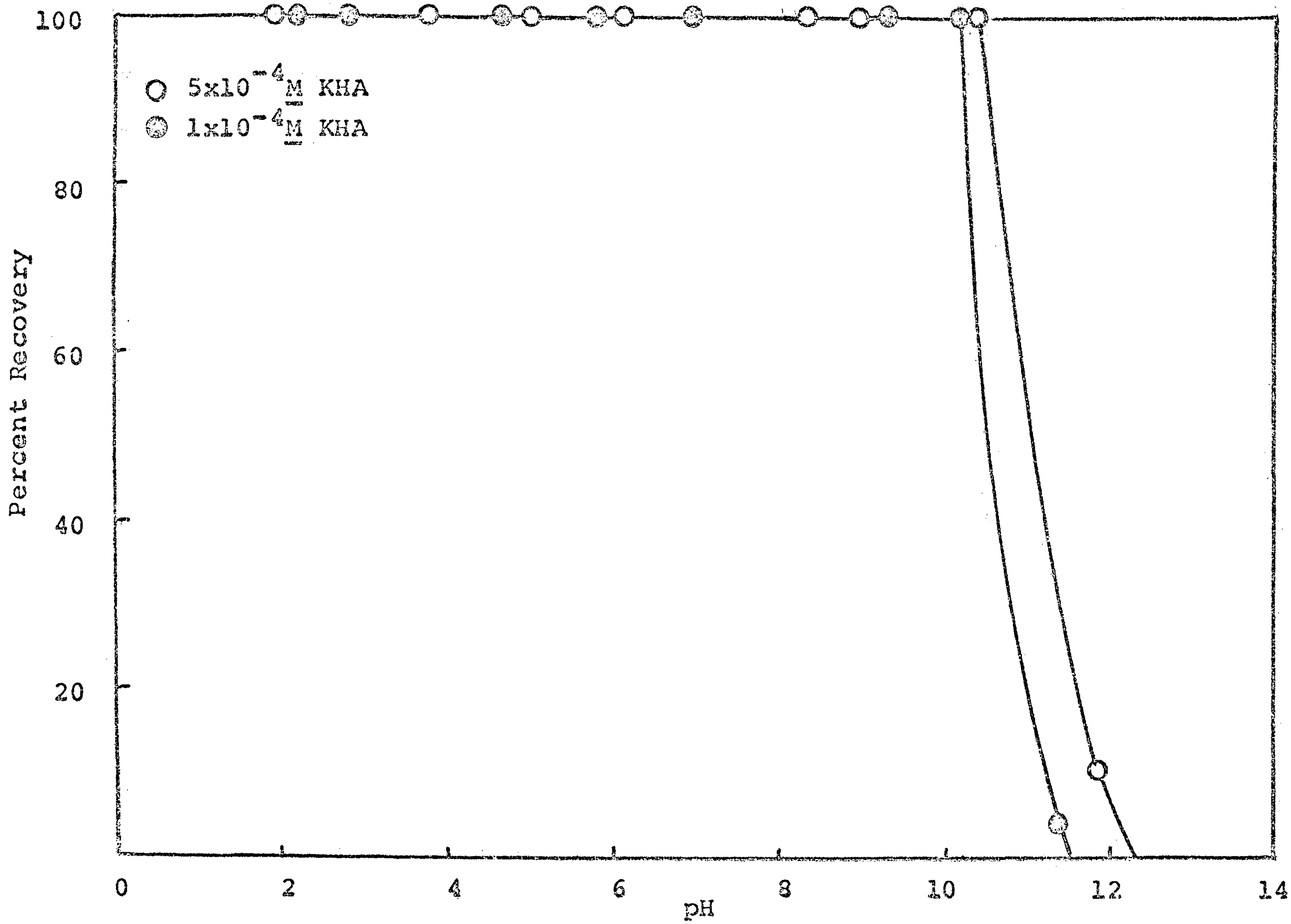

Figure 15. Flotation recovery of pyrite as a function of pH in the presence of 5×10^{-4} and 1×10^{-4} moles per liter KHA.

good flotation recoveries were still obtained, but in the cassiterite system at the same pH values, practically no flotation occurred.

Electrophoretic Experiments

Figures 15 and 16 show the zeta potential as a function of pH for pyrite and cassiterite, respectively.

The pH corresponding to the zero point of charge (zpc) is also depicted in these figures. Below this pH, the particles were positively charged; whereas above this pH, the surface was negative.

The zpc of pyrite, when placed in conductivity water, was found to be at pH 6.9, and the zpc of cassiterite in pure water was determined to be at pH 6.1.

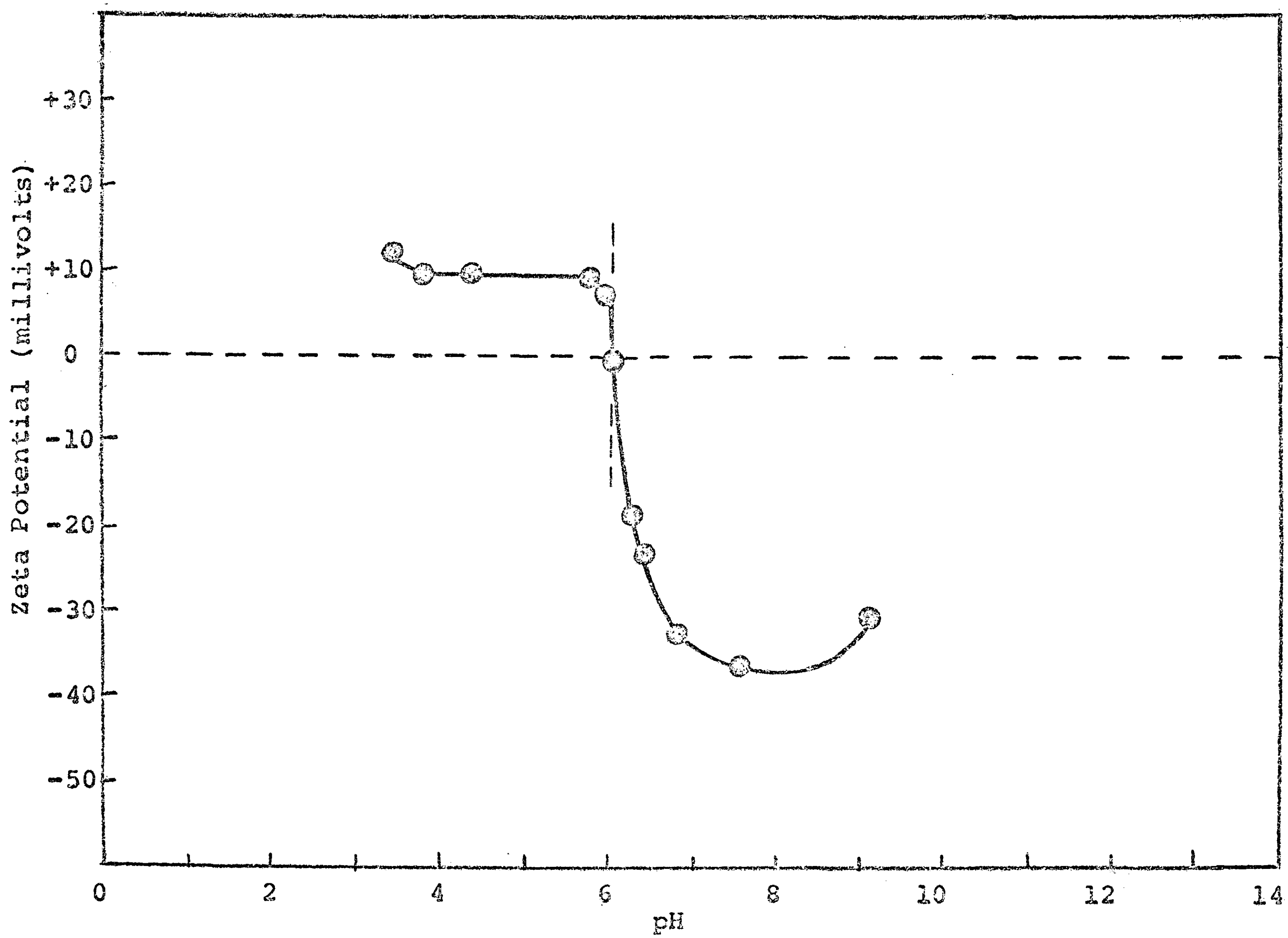

Figure 16. Zeta potential of cassiterite as a function of pH.

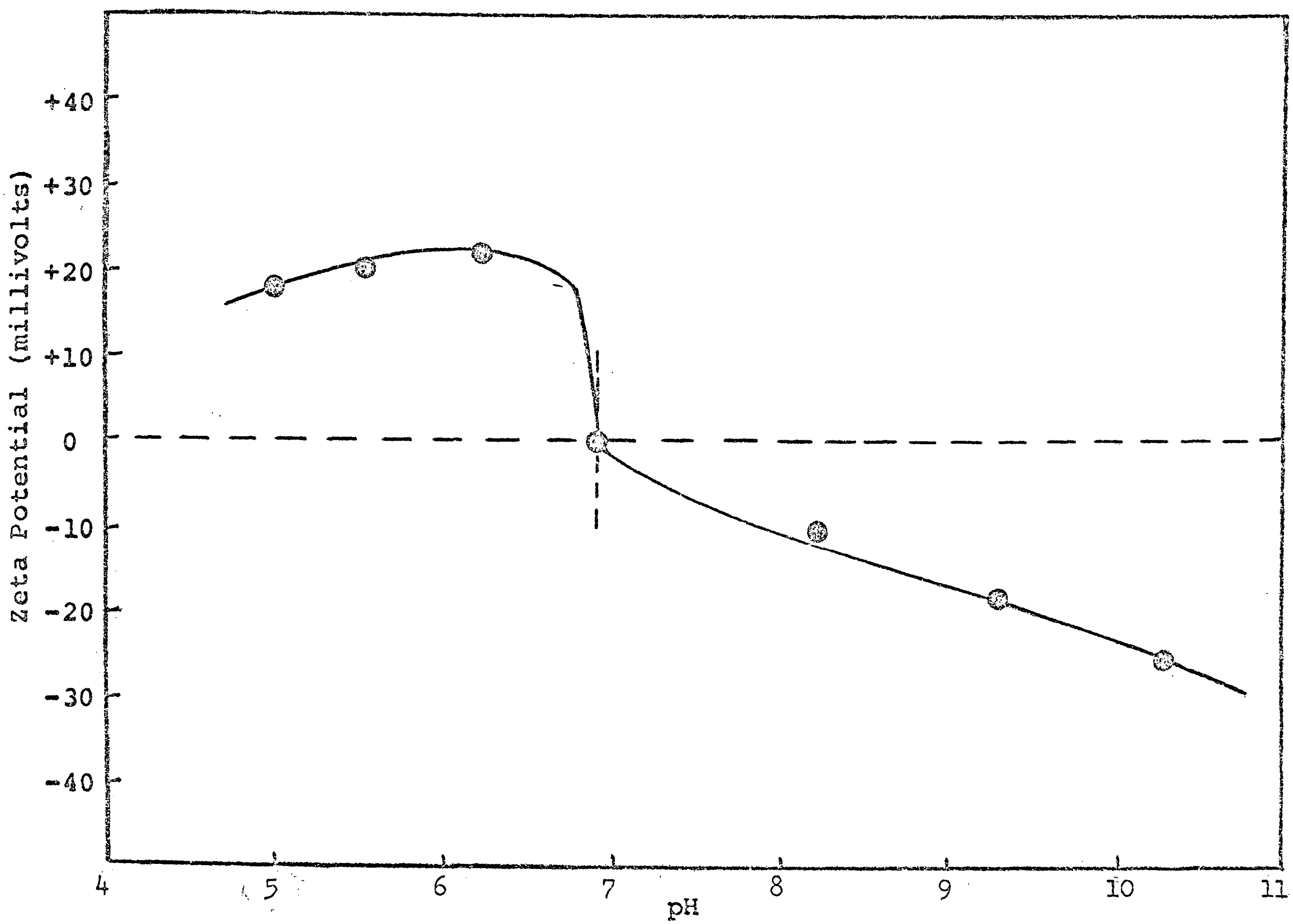

Figure 17. Zeta potential of pyrite as a function of pH.

DISCUSSION OF RESULTS

The experimental results obtained with cassiterite and pyrite will be discussed independently for each collector-mineral system.

Cassiterite-Oleate System. The study of the flotation behavior of cassiterite in this thesis has been oriented toward a practical rather than a theoretical approach. For this reason, little emphasis has been placed on some of the mechanisms of interaction between collector and cassiterite.

Recent studies on hematite [10] and pyrolusite [11] have suggested that oleic acid may adsorb on oxides by a molecular mechanism. These authors also found out that maximum chemisorption of oleic acid occurs near the iso-electric point of the oxide [12] where molecular adsorption is highly warranted [13]. These results are in contrast to physical adsorption based upon surface charge interaction where the amount adsorbed is a minimum at the zpc.

By analogy the cassiterite-oleate system (Figure 3) shows a flotation recovery of 96 percent at pH 6.1, which was found to be the zpc of cassiterite (Figure 16), and 100 percent at pH 6.5. The flotation of cassiterite with oleate in the basic pH range is probably due to the formation of insoluble stannous oleate at the surface. Apparently the species responsible for flotation is $SnOH^{+}$, the concentration of which is a function of pH. This aspect will be discussed more in detail in the hydroxamate section.

Depression below pH 2 can probably be attributed to the formation of oleic acid (aqueous) from oleate ion, and at pH~11 depression may be due to the formation of $Sn(OH)_2(s)$.

The reaction of cassiterite with oleic acid may be expressed by the following equation:

$$>M - O - MOH + HOl \rightleftharpoons >M - O \cdot M - Ol + H_2O \qquad (1)$$

Where HOl represents oleic acid; $>M - O - MOH$, the mineral surface site with chemisorbed water; and $>M - O \cdot M - Ol$, the mineral surface site with chemisorbed collector.

<u>Pyrite-Oleate System</u>. Figure 4 shows the flotation response of pyrite for a collector addition of 2×10^{-5} mole per liter potassium oleate. Good flotation recovery was obtained between pH 2 and 8.0. The flotation response of this mineral to oleate is probably due to the formation of insoluble $Fe(Ol)_3(s)$ at the mineral surface with the emission of surface anions to the solution, and the decrease in flotation at pH above 8.0 is due to the presence of $Fe(OH)_3(s)$ at the surface of pyrite. This work is in agreement with that done by Elgillani [14].

<u>Cassiterite-Sulfonate System</u>. Figure 5 shows that at a concentration of 1×10^{-5} mole per liter sulfonate, a

maximum recovery of about 38 percent is obtained. Depression at about pH 1.0 is due to the competition between chloride and sulfonate ions for the mineral surface. At pH values above 6.0, the flotation recovery is also decreased. The zpc of SnO_2 was found to be at pH 6.1, and the flotation response of cassiterite to sulfonate occurs only in the pH range in which the mineral is positively charged. This behavior suggests that sulfonate is adsorbing on the cassiterite by electrostatic attraction. With an addition of 1×10^{-4} mole per liter sulfonate, flotation recovery is increased considerably in the acid and basic media. The flotation response above the zpc of SnO_2 may be due either to chain association or to chemisorption; however, more research needs to be done on this matter.

Pyrite-Sulfonate System. Figure 6 shows that for a collector addition of 1×10^{-5} mole per liter sulfonate, good flotation recovery is obtained below pH 7. The zpc of pyrite is at pH 6.9 (Figure 17), and the flotation response of sulfonate at 1×10^{-5} mole per liter takes place only below the zpc of the mineral. This observation indicates that flotation occurs by physical attraction. At the higher collector additions of 5×10^{-5} and 1×10^{-4} mole per liter sulfonate, the flotation recovery is enhanced up to about pH 10.0. Since pyrite is negatively charged, above pH 6.9 the flotation response of this mineral above its zpc is probably due to chain association or to the formation of the stable $Fe(RSO_3)(s)$ compound, which is quite insoluble with a ksp on the order of 1×10^{-21} (15).

Cassiterite-Dodecyl Sulfate System. The flotation behavior of cassiterite as a function of pH for a collector addition of 1×10^{-4} mole per liter dodecyl sulfate (Figure 7) shows that cassiterite floats completely only in the pH region from 2.2 to 4.4. The drop in flotation recovery on the acid side is probably due to the hydrolysis of the collector. At pH 6 and above there is also a definite decrease in flotation recovery. Since cassiterite is positively charged below its zpc (pH 6.1), the interaction between dodecyl sulfate and cassiterite is due to electrostatic attraction.

Pyrite-Dodecyl Sulfate System. Figure 8 shows that good flotation of pyrite is obtained at 1×10^{-4} mole per liter dodecyl sulfate below pH 7.3. The flotation response of pyrite with dodecyl sulfate is probably due to chemical adsorption of collector since good flotation is achieved two units in pH above the zpc.

Cassiterite-Octyl Hydroxamate System. The cassiterite-hydroxamate flotation study has proved to be the most informative and interesting part of the investigation of the pure system. The flotation experiments showed several features that warrant consideration; these features seem to be a function of the following:

(1) Maximum flotation response of cassiterite near the isoelectric point

(2) Degree of surface hydrolyzation and metal ion activation

(3) Effect of temperature and conditioning time

(4) Amount of hydroxamate added

(Items 2, 3, and 4 are parameters.)

(1) As depicted in Figures 9, 10, and 11, the flotation response of cassiterite with octyl hydroxamate occurs near the zpc. Similar effects were also observed in the cassiterite-oleate system (Figure 3). These results are in contrast to physical adsorption where the amount adsorbed is minimum at and above the isoelectric point [13]. From these systems it follows that they would not respond to flotation unless a chemical reaction occurs (on the mineral surface) which can overcome the forces of repulsion by the decrease in free energy resulting from the reaction.

(2) Since tin starts to hydrolyze at pH 2, the concentration of the various stannous ion species were calculated as a function of pH (Table 2) on the assumption that a nominal $SnCl_2$ addition of 1×10^{-5} mole per liter and in accordance with the following equilibrium data from Latimer [16].

$$Sn^{++} + OH^- \rightleftharpoons Sn(OH)^+ \qquad K = 1.87 \times 10^{12} \qquad (2)$$

$$Sn(OH)_2(s) \rightleftharpoons Sn^{++} + 2OH^- \qquad K = 3.24 \times 10^{-27} \qquad (3)$$

As shown in Table 2, maximum concentration of $SnOH^+$ occurs in the pH range of 4.0 to 4.8. This pH range in Figures 9, 10, and 11 corresponds to the beginning of the flotation response of cassiterite to the collector action. The flotation recovery (Figure 10) was enhanced considerably by

Table 2. Concentration of various stannous ion species as a function of pH for nominal additions of 1×10^{-5} molar stannous chloride to water.

PH	Species		
	Sn^{++} mole/liter	$SnOH^{+}_{+}$ mole/liter	$Sn(OH)_{2(s)}$
1.6	5.73×10^{-6}	4.27×10^{-6}	0
1.8	4.59×10^{-6}	5.41×10^{-6}	0
2.0	3.48×10^{-6}	6.51×10^{-6}	0
2.2	2.52×10^{-6}	7.48×10^{-6}	0
2.4	1.75×10^{-6}	8.24×10^{-6}	0
2.6	1.18×10^{-6}	8.82×10^{-6}	0
2.8	7.81×10^{-7}	9.21×10^{-6}	0
3.0	5.08×10^{-7}	9.50×10^{-6}	0
3.2	3.26×10^{-7}	9.67×10^{-6}	0
3.4	2.08×10^{-7}	9.80×10^{-6}	0
3.6	1.32×10^{-7}	9.87×10^{-6}	0
3.8	8.40×10^{-8}	9.91×10^{-6}	0
4.0	5.32×10^{-8}	9.94×10^{-6}	0
4.2	3.36×10^{-8}	9.96×10^{-6}	0
4.4	2.12×10^{-8}	9.97×10^{-6}	0
4.6	1.34×10^{-8}	9.99×10^{-6}	0
4.8	8.14×10^{-9}	9.60×10^{-6}	3.89×10^{-7}
5.0	3.24×10^{-9}	6.06×10^{-6}	3.94×10^{-6}
6.0	3.24×10^{-11}	6.06×10^{-7}	9.39×10^{-6}
7.0	3.24×10^{-13}	6.06×10^{-8}	9.93×10^{-6}
8.0	3.24×10^{-15}	6.06×10^{-9}	9.99×10^{-6}
9.0	3.24×10^{-17}	6.06×10^{-10}	1×10^{-5}

running a flotation experiment with additions of 1×10^{-5} mole per liter stannous chloride in the acid state (pH 2.6) as specified in Figure 12. When the $SnCl_2$ was added in the basic state (pH 8.0), the flotation recovery was not enhanced as in the previous case.

The addition in the acid state showed clearly that there is a definite action between the metal salt added and the collector-mineral system. From Table 2 it can be seen that at pH 8.0 almost all the tin is present as $Sn(OH)_2(s)$. However, in practice, the effect of the metal salt when added in the acid state was quite determinant in the flotation response of cassiterite. A possible explanation for this to occur is that the reaction taking place at the solid-luquid interface is fast enough; that is, the activating $SnOH^+$ species approaching to the surface is favored by the mechanism of reaction before it is precipitated as $Sn(OH)_2(s)$ in solution [3].

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[3] When the $SnCl_2$ was added to the flotation system, it was observed that the cassiterite had a strong tendency to float. In fact, after 5 seconds of flotation time, 100 percent flotation was already attained.

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(3) The increase in temperature (Figure 14) improved the flotation recovery considerably and resulted also in an appreciable reduction in the optimum collector concentration (same as in Figure 10). These temperature effects seem to substantiate the premise that chemical reactions on the mineral surface are responsible for collector action because

the kinetics of most chemical reactions tend to be enhanced by increase in temperature [17]. In addition to increasing the rate of reaction, it seems most likely that an increase in the temperature of the system will considerably increase the mineral solubility. The increase in solubility would provide higher concentration of $SnOH^+$, the specie apparently necessary for activation.

Similar effects were also observed when a flotation recovery curve was run at constant KHA concentration and pH as a function of conditioning time (Figure 13). This effect seems to indicate that the flotation response of cassiterite is not only a function of thermodynamics, but also of kinetics. From this it follows that the longer the conditioning time the more dissolution occurs resulting in the formation of more $SnOH^+$.

(4) Figures 9, 10, and 11 reveal that the higher the collector concentration, the more recovery was obtained. Figure 9 shows 45 percent recovery at pH 6.7; Figure 10, 70 percent between pH 7.5 and 8.0; and Figure 11, 100 percent recovery at pH 8.0. An important aspect is that optimum flotation recovery can be attained with less collector concentration by using the different techniques shown in Figures 12, 13, and 14.

A comparison of the mineral flotation response as a function of pH and collector addition (Figures 9, 10, and 11) indicates that flotation recoveries fall-off below pH

6.5. There are several possible explanations for this behavior.

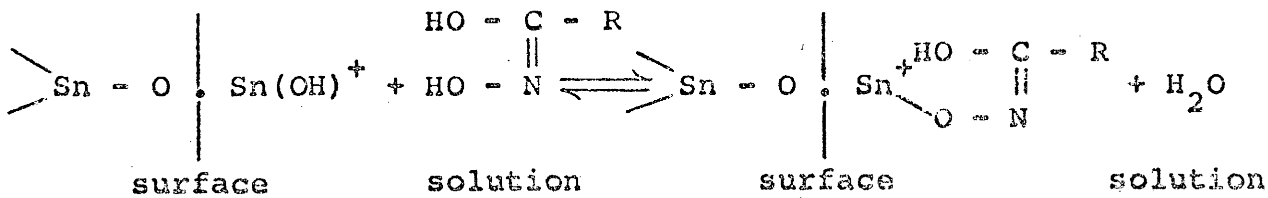
(1) Below pH 6.5, there is probably an insufficient amount of $SnOH^{+}$ adsorbed for flotation to occur.

(2) The pH values at which the curves fall-off on the acid side corresponds closely to the isoelectric point of cassiterite (pH 6.1). Since the monohydroxamic acids are reported to function as weak acids in aqueous solution [18], the neutral molecule, HAX, should predominate in acid media. Although the cassiterite surface is positive below the zpc, there is most probably insufficient AX^{-} present for flotation to occur by electrostatic attraction of the collector ion for the mineral surface.

The fall-off in flotation recovery noted in Figures 12 and 14 is also probably due to the reasons mentioned above. The depression of cassiterite at high pH is probably related to the hydrolysis of adsorbed tin ion to tin hydroxide on the mineral surface. From Table 2, for a nominal addition of 1×10^{-5} mole per liter stannous chloride to water, it may be seen that most of the tin ion is precipitated as tin hydroxide at pH values in excess of 8.5. This reaction was shown in equation 3. The hydrolysis of the tin ion of the mineral surface can be expressed as follows:

$$>\text{Sn}-\text{O}\underset{\text{surface}}{\Big|}\text{Sn(OH)}^{+} + \text{OH}^{-} \rightleftharpoons \text{Sn}-\text{O}\underset{\text{surface}}{\Big|}\text{Sn(OH)}_2 \qquad (4)$$

If the undissociated molecule [18] is functioning as collector, the adsorption mechanism could occur as follows:



(5)

By this mechanism water would be split out, and tin hydroxamate would be formed at the surface.

<u>Pyrite-Octyl Hydroxamate System</u>. Figure 15 shows that good flotation of pyrite is achieved at collector additions of 1×10^{-4} and 5×10^{-4} mole per liter KHA. As depicted in Figure 15, optimum flotation is observed below and above the zpc of pyrite, between pH 2.0 and about pH 10.5. The flotation response of pyrite with KHA seems to be due to chemisorption, and probably the compound formed is $Fe(HA)_3$ at the mineral surface. However, this aspect needs to be studied more in detail.

In the comparison of the flotation curve of pyrite with that of cassiterite (Figures 9, 10, and 11), two possible pH regions of separation are encountered. The first region is below pH 4, and the second above pH 10.0.

CONCLUSIONS

The following are the conclusions regarding the flotation characteristics of cassiterite and pyrite developed from the investigation of the pure system:

(1) The flotation response of cassiterite and pyrite with the various collectors used showed that octyl hydroxamate and dodecyl sulfate appear to be the most promising collectors for a selective separation from a natural ore.

(2) Selective separation of cassiterite from pyrite with dodecyl sulfate is possible using the following technique:

(a) Float first the pyrite at pH 10.0, and

(b) Float the cassiterite at pH 4.0.

(3) A selective separation with octyl hydroxamate should be possible using the following technique:

(a) Float first the pyrite at pH 10.5, and

(b) Float the cassiterite in the pH region 8 ± 0.5.

(4) Reduction of optimum hydroxamate concentration and increase in the pH range over which cassiterite may be floated can be achieved as follows:

(a) Increase the temperature of the system to 50°C. Optimum flotation pH can be controlled in the pH region from 6.80 to 8.60.

(b) Addition of stannous chloride to the flotation system. The flotation pH has to be controlled in the pH region from 7.3 to 8.1 for optimum response.

(c) Increase of conditioning time of the pulp to about 28 minutes. pH has to be fixed as indicated in part 3-b.

(5) For the oleate and sulfonate systems a different technique would be required in order to separate the pyrite from cassiterite. Pyrite would have to be floated first with ethyl or amyl xanthate, and then the cassiterite floated with oleic acid or sulfonate.

PART II - EXPERIMENTS WITH A TYPICAL BOLIVIAN TIN ORE

This second part of the investigation was directed toward studying flotation response of cassiterite from a complex sulfide ore, with the use of dodecyl sulfate and octyl hydroxamate, to determine whether these collectors were sufficiently selective for a practical use.

From the study of the pure system, it was seen that the separation of pyrite from cassiterite appeared to be favorable with the use of the collectors mentioned above. The natural ore, besides containing pyrite, also contained sphalerite; but since the sphalerite in this ore is known to contain iron in solid solution, it was hoped that the flotation responses of the two minerals to hydroxamate and dodecyl sulfate would be similar.

From the course of the investigation in the pure system, it was observed that pyrite floated in alkaline and acid media over a larger pH region compared with that of cassiterite. This showed that the sulfides must be floated prior to cassiterite flotation.

The flotation pH ranges for the two-stage separation of the sulfides and cassiterite were already outlined in conclusions (sections 2 and 3) in the pure system. In this second part of the study dixanthogen [19] was used to compare the efficiency of this collector with dodecyl sulfate and octyl hydroxamate for the sulfide separation.

EXPERIMENTAL MATERIALS

The following materials were used during the second part of this investigation:

Ore

The ore used for this experimental work was a complex sulfide ore of cassiterite from the Ocuri Tin Mine, Departamento de Potosi, Bolivia. The ore which assayed 0.85 percent tin, contained considerable amounts of pyrite and sphalerite. Also there were small amounts of galena, chalcopyrite, and some quartz and clay.

The qualitative and quantitative analysis of the ore by emission spectroscopy and x-ray, respectively, are listed in Table 3. Screen size analysis (Table 4) was conducted with the ore to study the percent distribution of fines. Samples from each size fraction were analyzed microscopically to determine the liberation size of cassiterite. This size was found to be at 100 mesh.

Because the ore was found to have some magnetic properties, the 100 x 200-size fraction was tested in the magnetic separator unit. The analysis of the magnetic fraction showing 0.36 percent tin, indicated that the tin was probably associated with the accompanying material as a complex compound. An investigation of the methods of the treatment of ore containing tin that cannot be liberated by comminution is a research study in itself for another thesis.

Table 3. Qualitative and Quantitative Analysis of the Ore.

Element	Relative Abundance	% Content
Ag	Impurity	---
Al	Minor	---
Cu	Impurity	0.21
Fe	Major	23.0
Li	Impurity	---
Mg	Minor	---
Mn	Minor	---
Mo	Impurity	---
Ni	Impurity	---
Pb	Minor	0.53
Si	Minor	---
Sn	Minor	0.85
Ti	Impurity	---
Zn	Major	15.0
Zr	Impurity	---

Sample Preparation

The standard sample used for each test comprised 500 grams of ore crushed to minus 20 mesh. The ore was ground at 25 percent solids in an 8 x 12 inch rod mill for 7 minutes to obtain a minus 100 mesh product. A wet screen analysis plus the tin content of these products is listed in Table 5.

Table 4. Screen Analysis (1000 grams ore)
(Crusher product obtained in crushing rolls)

Mesh Size	Percent Distribution	Cumulative Percentage Coarser
- 20 + 28	18.88	18.88
- 28 + 35	24.67	43.55
- 35 + 48	13.93	57.48
- 48 + 65	10.10	67.58
- 65 + 100	8.76	76.34
-100 + 150	6.39	82.73
-150 + 200	2.49	85.22
-200 + 325	7.50	92.72
-325	7.28	100.00
	100.00	

Table 5. Screen Analysis (500 grams ore)
(Rod mill product)

Mesh size	Cumulative Percentage Coarser	% Sn	Sn Distribution %
-100 + 150	5.90	0.37	2.24
-150 + 200	19.60	0.39	5.47
-200 + 325	47.92	0.63	18.28
-325 + 400	59.60	0.65	7.78
-400	100.00	1.60	66.23
			100.00

The size fraction used in each flotation test was 100 x 400 mesh. The minus 400 material was deslimed by sedimentation. The determination of the settling rates is outlined in Appendix 1.

Reagents

The flotation study of cassiterite from the ore was limited to the use of potassium octyl hydroxamate (KHA) and sodium dodecyl sulfate $(NaRSO)_4$. In some of the experiments for the flotation of the sulfides, dixanthogen was prepared and purified through the following procedure[20].

Amounts in excess of potassium ethyl xanthate and iodine were placed in a 250-ml beaker containing around 80 ml of distilled water, and the solution was stirred for 15 minutes. An oily precipitate was formed at the bottom of the beaker, which was dixanthogen. The following reaction illustrates the process:

$$E\bar{X} + I^{0} \rightleftharpoons 1/2\ (EX)_2 + I^{-}$$

The $(EX)_2$ was extracted from the remaining solution by adding ethyl ether. The ethyl ether phase in which the dixanthogen was present was placed in a separate container and heated to volatilize the ethyl ether. The remaining substance was pure dixanthogen.

RESULTS

Only a limited number of flotation tests were run with the ore. Some of the conditions such as collector concentration and frother were varied according to the flotation effects observed during the experimental work. The $NaRSO_4$ and KHA were added stages. The rest of the flotation conditions for each collector have already been specified in conclusions.

Test No. 1. This test was carried out with $NaRSO_4$ to float first the sulfides (pH 10.0) and then the SnO_2 (pH 4.0). The overall concentration of the collector used was 1.0 lb/ton dodecyl sulfate. For this test a conditioning time of 15 minutes was used, with no addition of frother. The flotation results are listed in Table 6.

Table 6. Flotation results from test No. 1

Product	Weight Grams	% Sn	Sn Weight Grams	Flotation Recovery (% Sn)
Sulfides	147	0.66	0.9702	22.37
Concentrate	227	1.10	2.4970	57.56
Slimes	79	0.96	0.7584	17.48
Tailing	33	0.34	0.1122	2.59
			4.3378	100.00

Test No. 2. In a second test run a total of 0.7 lb/ton KHA was added. When the sulfides were first floated at pH 10.5, it was observed that the pyrite floated first and then the sphalerite. These two products were cleaned once independently and the non-float materials were added together. No collector addition was done during the cleaning stage. The cassiterite was then floated at pH 8.0, from the remaining material. The total conditioning time was 15 minutes, and 4 drops of amyl alcohol were added as a frother. The flotation results are shown in Table 7.

Table 7. Flotation results from test No. 2

Product	Weight Grams	% Sn	Sn Weight Grams	Flotation Recovery (% Sn)
Pyrite	45	0.67	0.3015	6.99
Sphalerite	147	0.48	0.7056	16.37
Concentrate	139	1.50	2.0850	48.36
Sulfide Cleaning	92	0.78	0.7176	16.64
Slimes	45	0.99	0.4455	10.33
Tailing	12	0.47	0.0564	1.31
			4.3116	100.00

Test No. 3. A separate test was run with an initial addition of 0.6 lb/ton $(EX)_2$, which was added to the ore during the grinding stage. The sulfides were floated at pH 5 and 8, to float the pyrite and sphalerite, respectively. An additional amount of 0.2 lb/ton $(EX)_2$ was added to float the rest of the sphalerite from the pulp. The pulp in the cell was conditioned for 10 minutes. No desliming was done and the cassiterite was left with the tails. The results are shown in Table 8.

Table 8. Flotation results from test No. 3

Product	Weight Grams	% Sn	Sn Weight Grams	Flotation Recovery (% Sn)
Pyrite I	60	0.50	0.3000	6.53
Pyrite II	151.7	0.30	0.4551	9.91
Sphalerite	69.3	0.42	0.2910	6.33
Tailing (SnO_2)	197.1	1.80	3.5478	77.23
			4.5939	100.00

Test No. 4. In this test the flotation of the sulfides was done in a similar way as in test No. 3, with the only difference that the products of pyrite and sphalerite were added together. After the sulfides were skimmed, the pH of the pulp was kept constant at 8.0, and the collector at 0.6 lb/ton KHA was added. The pulp was conditioned for 15 minutes and two drops of amyl alcohol were added. The results are shown in Table 9.

Table 9. Flotation results from test No. 4

Product	Weight Grams	% Sn	Sn Weight Grams	Flotation Recovery (% Sn)
Pyrite	358.8	0.60	2.1528	50.40
Concentrate	29.8	4.10	1.2218	28.60
Slimes	90.2	0.99	0.8929	20.90
Tailing	2.1	0.20	0.0042	0.10
			4.2717	100.00

Test No. 5. The separation of the sulfides was done as in test 3 and 4. The pyrite and sphalerite were also skimmed independently. For flotation of cassiterite 0.6 lb/ton $NaRSO_4$ was used, and the pulp was conditioned for 15 minutes. No frother was required. The results are shown in Table 10.

Table 10. Flotation results from test No. 5

Product	Weight Grams	% Sn	Sn Weight Grams	Flotation Recovery (% Sn)
Pyrite	163	0.31	0.5053	10.77
Sphalerite	189	0.39	0.7371	15.71
Concentrate	39	6.30	2.4570	52.36
Slimes	100	0.98	0.9800	20.88
Tailing	2	0.65	0.0130	0.28
			4.6924	100.00

Test No. 6. A final test was conducted with amyl dixanthogen. The pyrite and sphalerite were floated in the same conditions as in the previous tests. Each one of these products was cleaned once as shown in Table 11. The initial addition of dixanthogen during grinding was the same as in test 3. Only an additional amount of 0.1 lb/ton dixanthogen was required during flotation. The cassiterite was left with the tails.

Table 11. Flotation results from test No. 6

Product	Weight Grams	% Sn	Sn Weight Grams	Flotation Recovery (% Sn)
Pyrite	227	0.30	0.6810	16.22
Sphalerite	57	0.40	0.2280	5.43
Cleaner FeS_2	45	0.85	0.3825	9.11
Cleaner ZnS	49	0.77	0.3773	8.99
Tailing (SnO_2)	40	4.70	1.8800	44.78
Slimes	73	0.89	0.6497	15.47
			4.1985	100.00

DISCUSSION OF RESULTS

The experimental conditions for a favorable flotation response that evolved from the work with pure cassiterite were also employed with a natural ore. Table 6 and 10 show the flotation results obtained with dodecyl sulfate ($NaRSO_4$). Flotation recoveries of 57.6 and 52.4 percent were obtained with 1 and 0.6 lb/ton $NaRSO_4$ (1 lb/ton $NaRSO_4$ = 3×10^{-4} mole per liter $NaRSO_4$); in the latter case 0.8 lb/ton dixanthogen $(EX)_2$ was previously added (1 lb/ton EX_2 = 1.6×10^{-3} mole per liter EX_2). The grades of the concentrates were 1.1 and 6.3 percent tin, respectively. The overall recoveries in both cases were about the same. The amount of tin lost during the sulfide flotation, as shown in Tables 6 and 10, is rather high. These losses probably could have been decreased by additional cleaning of the sulfide concentrate, but since much of the tin is probably present in the sulfides not as SnO_2 but as solid solution or chemically bound as tin sulfide complexes, complete recovery cannot be expected. Considerable losses also observed in the slimes were probably due to a high content of fine cassiterite in the ore.

Table 7 and 9 depict the flotation results obtained with octyl hydroxamate (KHA). As we can see from Table 7 an overall recovery of 65 percent (including the sulfide cleaning) with 0.7 lb/ton KHA (1 lb/ton KHA = 4.24×10^{-4} mole per liter KHA). The grade of the concentrate was 1.5 percent tin. The recovery obtained in Table 9 was only 28.6 percent tin with 0.8 and 0.6 lb/ton of EX_2 and KHA, respectively. The grade of the concentrate was 4.1 percent tin. The flotation recovery in Table 7 is much higher than that of Table 9. This difference is due to the amount of tin lost with the sulfides; however, part of the amount of tin lost with the sulfides of Table 9 was due to physical entrapment. This reason is apparent because the tin to sulfide ratio was nearly constant under flotation with all collectors and conditions. The tin losses in the slimes were nearly the same as those of Tables 6 and 10. The tin content in the sulfides as shown in Table 8 was also as high as in the other tests. This experiment was run only with dixanthogen at 0.8 lb/ton. The rougher tails were not subjected to additional cleaning. Because no desliming was done in this test, the recovery of cassiterite was increased to 77.2 percent, but the grade was only 1.8 percent tin. In similar tests also run, the products of the sulfides were cleaned independently (Table 11). A recovery of about 63 percent tin (including cleaners) was obtained. Because of the longer collector carbon chain, only 0.7 lb/ton dixanthogen was required. The amount of material left in the tails was also reduced, and its tin content was raised to 4.7 percent; however, the amount of tin present in the sulfides was the same as in the previous tests. In general, the tin content in the tails was rather high.

A detailed mineralogical examination of each product was not undertaken to determine what constituted the remaining amounts of tin that were floated with the sulfides.

The loss of valuable material encountered in the slimes were quite high. This seems to indicate that because of the large amount of SnO_2 present in the minus 400 fraction (Table 5), the ore would have to be deslimed at a much smaller particle size in a practical separation.

The main problem encountered in this study was the response of some of the valuable material to the collecting action during the sulfide separation. From the tests in Tables 6, 7, 8, and 11, where $NaRSO_4$, KHA, and $(EX)_2$ were used to float the sulfides, about 1 gram of tin (22.5 ± 1.2 percent) was found in the sulfide concentrates. This fact suggests that some of the tin might have been floated due to the following reasons:

(1) Solid solution, that is, the sphalerite or pyrite may contain tin atoms as interstitial or substitutional solid solution.

(2) Another possible reason is that some of the tin might be present as SnS, or some other complex sulfide compound, as listed in Table 1. From this reason it

follows that dixanthogen would adsorb quite well on these compounds, provided that sulfur is the active site [19]. Oxides do not respond to the collecting action of dixanthogen, provided that they are not floated near the zpc of the oxide [19].

(3) From the experimental results obtained from the ore, it was found that the flotation response of the sulfides to $NaRSO_4$ and KHA was good at relatively high collector additions. In these tests the amount of valuable material floated with the sulfides was as high as in the previous tests. This may be due to complex tin sulfide compounds which cannot be liberated from pyrite or sphalerite.

CONCLUSIONS

Because of the limited number of tests run with the ore, it is difficult to draw final conclusions in regard to the flotation response of cassiterite from the ore with either $NaRSO_4$ or KHA. However, the collecting properties with KHA can be improved considerably by combining the flotation effects outlined in conclusions of the pure system (sections 3 and 4). This possibility implies the necessity of doing further research work in regard to the flotation response of SnO_2 from a natural ore, as the intent of this experimental work was simply to study whether the technique devised with the pure system was amenable to natural ores.

From the flotation results obtained with the ore, it can be seen that a detailed mineralogical and quantitative study of the materials present is necessary. This would yield the following information:

(1) Amount of tin content, if present in solid solution, either as interstitial tin or as substituting element

(2) Identification of the tin compounds associated with either pyrite or sphalerite

(3) Possible presence of surface coatings, and

(4) Presence of soluble salts

Provided that the ore would contain some tin in solid solution, as indicated in case 1, the separation of this valuable fraction of material from the ore by flotation techniques would not be amenable at all. If the tin present in this form is comparatively small, it could be left with the rest of the tailings. In case 2, provided that the tin is not associated with one of this sulfides, it would have to be depressed and then floated with the valuable material. This could be done with common depressors and activators such as NaCN, $CuSO_4$, Na_2SO_3, etc., and dixanthogen to float the sulfides, and either $NaRSO_4$ or KHA to float the SnO_2. The products containing the cassiterite would have to be cleaned to up-grade the tin in the final concentrate.

The use of some depressants such as cyanide, sodium carbonate, etc. could also help to neutralize the action of some soluble salts or undesired metal ions. The addition of some dispersants such as sodium silicate would be useful to eliminate possible slime coatings on the surface of fine SnO_2 particles.

APPENDIX

The determination of mineral settling rates from Stoke's Law.

Stoke's Law gives the rate of fall of a small sphere in a viscous fluid.[21]

V = free-falling velocity of a particle in cm/sec

$$= \frac{d^2 \times g(D_p - D_f)}{18\ n} \text{ cm/sec, where}$$

d = Stoke's diameter of the particle in cm

g = gravitational acceleration in cgs units
981 cm/sec^2

D_p = density of the particle in gm/ml

D_f = density of the fluid in gm/ml

n = absolute viscosity of the fluid in dyne sec/cm^2 (poise)

<u>For 5 micron particles</u>

$$V = \frac{(5\times10^{-4})^2\ 981(D_p - 1.0)}{18 \times 0.0113} \text{ cm/sec}$$

$$= (1.21 \times 10^{-3})(D_p - 1.0) \text{ cm/sec}$$

$$= 4.32\ (6.8 - 1.0) \text{ cm/hr}$$

$$V = 25.1 \text{ cm/hr}$$

<u>For 37 micron (400 mesh) particles</u>

$$V = \frac{(37 \times 10^{-4})^2\ 981(D_p - 1.0)}{18 \times 0.0113}\ \text{cm/sec}$$

$$= (6.61 \times 10^{-2})(D_p - 1.0)\ \text{cm/sec}$$

$$= (2.38 \times 10^{2})(6.8 - 1.0)\ \text{cm/hr}$$

$$V = 1379\ \text{cm/hr}$$

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