

**Associated Sulfide Minerals in Thiosulfate Leaching of  
Gold: Problems and Solutions**

**by**

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## Abstract

The effects of some associated minerals on thiosulfate gold leaching were studied through thermodynamic analysis and leaching experiments on composite ore samples containing various minerals and a reference silicate gold ore.

In the leaching test on the reference gold ore, about 93% of gold was extracted within 3.0 hours. The presence of various amount of pyrite, pyrrhotite, chalcopyrite, arsenopyrite, chalcocite, bornite, and some lead species, has significant detrimental effects.

Under reduced oxygen conditions, the thiosulfate consumptions could be significantly reduced. High gold extractions (i.e.  $\geq 90\%$ ) were observed in the leaching tests with reduced dissolved oxygen (i.e., 0.7% oxygen in the supplied gas) in the absence or in the presence of sulfide minerals such as pyrite, pyrrhotite, arsenopyrite and chalcopyrite. High copper concentration and a pre-aeration step was also found to largely increase the gold extractions under such conditions.

Thiosulfate-copper-ethylenediamine system was found effective in the leaching of gold. The leaching kinetics was significantly slower than that of the conventional thiosulfate-copper-ammonia leaching. The consumption of thiosulfate, however, was largely reduced. This leaching system worked effectively on the reference gold ore within a wider pH range (e.g., 6-11), with or

without ammonia. The presence of ammonia in a low concentration improved the leaching rate but also increase the consumption of thiosulfate. Comparable gold extractions were observed in the leaching of the composite ores containing various sulfide minerals, such as pyrite, pyrrhotite, chalcocite, galena and chalcopyrite.

The leaching of gold in the presence of iron sulfides was also improved by applying chemical additives, such as, carbonate, calcium, galena, phosphate, and additional hydroxide anion. It is proposed that these additives either passivated the harmful surface of sulfide minerals or masked some detrimental aqueous species.

Finally, some improved leaching methods concluded in this study were applied on a few industrial ore samples in order to demonstrate the effectiveness of these methods. It was found that by comprehensively applying these improved thiosulfate leaching strategies, satisfactory gold extractions and thiosulfate consumption results were archived on these ores.

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# Tables of Contents

<b>ABSTRACT</b> .....	<b>I</b>
<b>ACKNOWLEDGEMENT</b> .....	<b>III</b>
<b>TABLE OF CONTENTS</b> .....	<b>IV</b>
<b>LIST OF FIGURES</b> .....	<b>VIII</b>
<b>LIST OF TABLES</b> .....	<b>XV</b>
<b>LIST OF ABBREVIATIONS</b> .....	<b>XVII</b>
<b>CHAPTER 1. INTRODUCTIONS</b> .....	<b>1</b>
1.1. CYANIDATION AND ITS LIMITATIONS .....	1
1.2. ALTERNATIVE LIXIVIANTS .....	2
1.3. THIOSULFATE LEACHING OF GOLD .....	3
1.4. OBJECTIVES OF THIS INVESTIGATION.....	5
<b>CHAPTER 2. LITERATURE REVIEW</b> .....	<b>7</b>
2.1. DEVELOPMENT AND HISTORY OF THIOSULFATE-GOLD LEACHING .....	7
2.2. CHEMISTRY OF GOLD DISSOLUTION IN THIOSULFATE SOLUTIONS.....	11
2.3. CHEMISTRY OF THIOSULFATE DECOMPOSITION .....	15
2.4. LEACHING OF NATURAL GOLD ORES WITH THIOSULFATE.....	19
2.5. ASSOCIATED MINERALS IN THIOSULFATE LEACHING OF GOLD.....	21
<b>CHAPTER 3. THEORETICAL CONSIDERATIONS</b> .....	<b>25</b>
3.1. EFFECTS OF ASSOCIATED MINERALS .....	25
3.1.1. <i>Fundamental principles required for plotting Eh-pH diagrams</i> .....	25
3.1.2. <i>Eh-pH diagram of the thiosulfate-gold leaching system</i> .....	28
3.1.3. <i>Impacts of associated minerals on thiosulfate leaching of gold</i> .....	39

3.2. STRATEGIES FOR IMPROVING THIOSULFATE LEACHING OF GOLD .....	68
3.2.1. <i>Reagent compositions</i> .....	69
3.2.2. <i>Oxygen control and alternative oxidation catalyst</i> .....	74
3.2.3. <i>Oxidative pre-leach (pre-aeration)</i> .....	79
3.2.4. <i>Masking of detrimental surface or ions with chemical additives</i> .....	80
3.2.5. <i>Summary</i> .....	88
<b>CHAPTER 4. EXPERIMENTAL .....</b>	<b>89</b>
4.1. MATERIALS .....	89
4.1.1. <i>Ore samples</i> .....	89
4.1.2. <i>Minerals</i> .....	90
4.1.3. <i>Reagents</i> .....	92
4.2. EXPERIMENTAL PROCEDURES .....	92
<b>CHAPTER 5. RESULTS AND DISCUSSION .....</b>	<b>95</b>
5.1. EFFECTS OF ASSOCIATED MINERALS: LEACHING RESULTS.....	95
5.1.1. <i>Effects of various minerals: a preliminary comparison</i> .....	96
5.1.2. <i>Effects of pyrite, pyrrhotite and arsenopyrite</i> .....	101
5.1.3. <i>Effects of chalcopyrite, chalcocite and bornite</i> .....	108
5.1.4. <i>Effects of lead minerals</i> .....	114
5.1.5. <i>Summary</i> .....	121
5.2. OPTIMIZATION OF LEACHING CONDITIONS .....	124
5.2.1. <i>Ammonia concentration</i> .....	124
5.2.2. <i>Copper concentration</i> .....	136
5.2.3. <i>Thiosulfate concentration</i> .....	146
5.3. EXAMPLES OF USING CHEMICAL ADDITIVES TO ENHANCE THE LEACHING PERFORMANCE IN THE PRESENCE OF DETRIMENTAL MINERALS .....	152
5.3.1. <i>Additional hydroxide anion</i> .....	153

5.3.2. Calcium ion .....	161
5.3.3. Phosphate anion .....	162
5.3.4. Lead ion (Galena).....	169
5.3.5. Carbonate anion.....	171
5.3.6. Summary .....	173
5.4. THIOSULFATE LEACHING WITH LIMITED OXYGEN SUPPLY .....	173
5.4.1. The reference ore.....	174
5.4.2. Pyrite, pyrrhotite and arsenopyrite and chalcopyrite ores .....	177
5.4.3. Leaching of gold with limited oxygen after pre-aeration .....	182
5.4.4. Summary .....	186
5.5. THIOSULFATE LEACHING WITH THIOSULFATE-EDA-COPPER SOLUTION .....	186
5.5.1. Properties of ethylenediamine .....	187
5.5.2. Thermodynamic consideration of thiosulfate-EDA-copper leaching system .....	191
5.5.3. Thiosulfate-ethylenediamine-copper leaching of gold .....	197
5.5.4. Summary .....	206
5.6. VALIDATION OF OPTIMAL LEACHING CONDITIONS ON NATURAL GOLD ORES .....	207
5.6.1. Ore #1.....	208
5.6.2. Ore #2.....	209
5.6.3. Ore #3.....	210
5.6.4. Ore #4.....	212
<b>CHAPTER 6. CONCLUSIONS AND FUTURE WORK.....</b>	<b>214</b>
6.1. CONCLUSIONS.....	214
6.2. FUTURE WORK .....	218
<b>REFERENCES .....</b>	<b>220</b>
<b>APPENDIX 1 .....</b>	<b>235</b>
<b>APPENDIX 2 .....</b>	<b>237</b>



## List of Figures

FIGURE 2-1: GOLD DISSOLUTION SKETCH (AMMONIACAL COPPER THIOSULFATE SYSTEM) .....	12
FIGURE 2-2: EH-PH DIAGRAMS OF S-H <sub>2</sub> O SYSTEM .....	15
FIGURE 2-3: OXIDATION REACTIONS OF THIOSULFATE.....	18
FIGURE 3-1: EH-PH DIAGRAM OF AU-H <sub>2</sub> O-S <sub>2</sub> O <sub>3</sub> <sup>2-</sup> NH <sub>3</sub> /NH <sub>4</sub> SYSTEM (I): EFFECT OF AMMONIA CONCENTRATION.....	31
FIGURE 3-2: EH-PH DIAGRAM OF AU-H <sub>2</sub> O-S <sub>2</sub> O <sub>3</sub> <sup>2-</sup> NH <sub>3</sub> /NH <sub>4</sub> SYSTEM (II): EFFECT OF THIOSULFATE CONCENTRATION.....	31
FIGURE 3-3: EH-PH DIAGRAM OF CU-S-N-H <sub>2</sub> O SYSTEM (I) .....	33
FIGURE 3-4: EH-PH DIAGRAM OF CU-S-N-H <sub>2</sub> O SYSTEM (II) .....	35
FIGURE 3-5: EH-PH DIAGRAM OF CU-S-N-H <sub>2</sub> O SYSTEM (III) .....	36
FIGURE 3-6: EH-PH DIAGRAM OF S-H <sub>2</sub> O SYSTEM (I).....	37
FIGURE 3-7: EH-PH DIAGRAM OF S-H <sub>2</sub> O SYSTEM (II).....	38
FIGURE 3-8: EH-PH DIAGRAM OF S-H <sub>2</sub> O SYSTEM (II).....	40
FIGURE 3-9: EH-PH DIAGRAM OF FE-H <sub>2</sub> O SYSTEM: OXIDATION AND PRECIPITATION OF FERROUS ION....	43
FIGURE 3-10: EH-PH DIAGRAM OF FE-S-H <sub>2</sub> O SYSTEM: IRON SULFIDE MINERALS.....	45
FIGURE 3-11: EH-PH DIAGRAM OF FE-CU-N-S-H <sub>2</sub> O SYSTEM: CO-PRECIPITATION OF COPPER WITH IRON HYDROXIDES.....	47
FIGURE 3-12: EH-PH DIAGRAM OF CU-FE-N-S-H <sub>2</sub> O SYSTEM --- CHALCOCITE (I).....	50
FIGURE 3-13: EH-PH DIAGRAM OF CU-FE-N-S-H <sub>2</sub> O SYSTEM --- CHALCOCITE (II).....	51
FIGURE 3-14: EH-PH DIAGRAM OF CU-FE-N-S-H <sub>2</sub> O SYSTEM --- BORNITE (I) .....	52
FIGURE 3-15: EH-PH DIAGRAM OF CU-FE-N-S-H <sub>2</sub> O SYSTEM --- BORNITE (II) .....	53
FIGURE 3-16: EH-PH DIAGRAM OF CHALCOPYRITE (I) .....	54
FIGURE 3-17: EH-PH DIAGRAM OF CHALCOPYRITE (II) .....	55
FIGURE 3-18: EH-PH DIAGRAM OF ARSENOPYRITE (I).....	58

FIGURE 3-19: EH-PH DIAGRAM OF ARSENOPYRITE (II).....	59
FIGURE 3-20: EH-PH DIAGRAM ARSENOPYRITE (III).....	60
FIGURE 3-21: EH-PH DIAGRAM OF Pb-S-H <sub>2</sub> O SYSTEM IN THE PRESENCE OF SULFATE ANION (I).....	62
FIGURE 3-22: EH-PH DIAGRAM OF Pb-S-H <sub>2</sub> O SYSTEM IN THE PRESENCE OF SULFATE ANION (II).....	63
FIGURE 3-23: EH-PH DIAGRAM OF Pb-S-H <sub>2</sub> O SYSTEM IN THE ABSENCE OF SULFATE ANION (I).....	64
FIGURE 3-24: EH-PH DIAGRAM OF Pb-S-H <sub>2</sub> O SYSTEM: DISSOLUTION OF LEAD .....	66
FIGURE 3-25: CATALYZED OXIDATION OF THIOSULFATE ON PYRITE SURFACE (SKETCH) .....	68
FIGURE 3-26: SPECIFICATION OF AMMONIA/AMMONIUM AND SOLUTION PH .....	71
FIGURE 3-27: EH-PH DIAGRAM OF THIOSULFATE LEACHING SYSTEM: A COMPARISON OF IMPORTANT REDOX COUPLES .....	74
FIGURE 3-28: OXYGEN SUPPLY STRENGTH VS. DISSOLVED OXYGEN CONCENTRATION .....	76
FIGURE 3-29: A COMPARISON OF STANDARD OXIDATION POTENTIALS OF SOME CUPRIC COMPLEXES (ESTIMATED FROM THE COMPLEX STABILITY CONSTANTS).....	77
FIGURE 3-30: PHOSPHATE SPECIFICATION DIAGRAM.....	82
FIGURE 3-31: EFFECTS OF PHOSPHATE ANION THE SURFACE OF IRON SULFIDE MINERALS.....	83
FIGURE 3-32: EH-PH DIAGRAM OF As-Ca-Cu-Fe-H <sub>2</sub> O SYSTEM.....	85
FIGURE 3-33: EH-PH DIAGRAM OF Pb-S-N-Cu-C-H <sub>2</sub> O SYSTEM.....	87
FIGURE 5-1: LEACHING KINETICS OF THE REFERENCE GOLD ORE .....	96
FIGURE 5-2: EFFECT OF ASSOCIATED MINERALS ON GOLD EXTRACTIONS - A PRELIMINARY COMPARISON STUDY (I) .....	97
FIGURE 5-3: EFFECT OF ASSOCIATED MINERALS ON THIOSULFATE CONSUMPTIONS - A PRELIMINARY COMPARISON STUDY (II).....	98
FIGURE 5-4: EFFECTS OF PYRITE, PYRRHOTITE AND ARSENOPYRITE (I) ON GOLD EXTRACTIONS .....	101
FIGURE 5-5: EFFECTS OF PYRITE, PYRRHOTITE AND ARSENOPYRITE (II) ON THIOSULFATE CONSUMPTIONS .....	102
FIGURE 5-6: EFFECT OF PYRITE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION.....	103
FIGURE 5-7: EFFECT OF PYRRHOTITE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION .....	105
FIGURE 5-8: EFFECT OF ARSENOPYRITE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION .....	107

FIGURE 5-9: EFFECTS OF CHALCOCITE, CHALCOPYRITE AND BORNITE ON GOLD EXTRACITONS .....	108
FIGURE 5-10: EFFECTS OF CHALCOCITE, CHALCOPYRITE AND BORNITE ON THIOSULFATE CONSUMPTION .....	109
FIGURE 5-11: EFFECT OF CHALCOPYRITE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION ....	110
FIGURE 5-12: EFFECT OF CHALCOCITE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION .....	111
FIGURE 5-13: EFFECT OF BORNITE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION.....	113
FIGURE 5-14: EFFECTS OF LITHARGE, GALENA AND LEAD CARBONATE ON GOLD EXTRACTIONS.....	114
FIGURE 5-15: EFFECTS OF LITHARGE, GALENA AND LEAD CARBONATE ON THIOSULFATE CONSUMPTIONS .....	115
FIGURE 5-16: EFFECT OF LEAD NITRATE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION .....	116
FIGURE 5-17: EFFECT OF GALENA ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION.....	117
FIGURE 5-18: EFFECT OF LITHARGE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION .....	118
FIGURE 5-19: EFFECT OF LEAD CARBONATE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION.	119
FIGURE 5-20: EFFECTS OF THE ADDITIONAL AMMONIUM HYDROXIDE ON SLURRY PH.....	125
FIGURE 5-21: EFFECT OF AMMONIA ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRITE.....	126
FIGURE 5-22: EFFECT OF AMMONIA ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRRHOTITE.....	126
FIGURE 5-23: EFFECT OF AMMONIA ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 4.0% ARSENOPYRITE .....	127
FIGURE 5-24: EFFECT OF AMMONIA ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 8.0% CHALCOPYRITE .....	128
FIGURE 5-25: EFFECT OF AMMONIA ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 4.0% CHALCOCITE.....	129
FIGURE 5-26: EFFECT OF AMMONIA ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 2.0% BORNITE .....	130
FIGURE 5-27: EFFECT OF AMMONIA CONCENTRATION ON THIOSULFATE CONSUMPTIONS (I).....	131
FIGURE 5-28: EFFECT OF AMMONIA ON GOLD EXTRACTION AT 0.5 HOUR (I).....	132

FIGURE 5-29: EFFECT OF AMMONIA ON GOLD EXTRACTIONS AT 3.0 HOURS (I) .....	133
FIGURE 5-30: EFFECT OF AMMONIA CONCENTRATION ON THIOSULFATE CONSUMPTIONS (II).....	134
FIGURE 5-31: EFFECT OF AMMONIA ON GOLD EXTRACTION AT 0.5 HOUR (II).....	135
FIGURE 5-32: EFFECT OF AMMONIA ON GOLD EXTRACTION AT 3.0 HOURS (II).....	135
FIGURE 5-33: EFFECT OF COPPER ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRITE.....	137
FIGURE 5-34: EFFECT OF COPPER ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRRHOTITE.....	138
FIGURE 5-35: EFFECT OF COPPER ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 4.0% ARSENOPYRITE.....	139
FIGURE 5-36: EFFECT OF ADDITIONAL COPPER IN PRESENCE OF 8.0% CHALCOPYRITE .....	140
FIGURE 5-37: EFFECT OF COPPER CONCENTRATION ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 4.0% CHALCOCITE .....	141
FIGURE 5-38: EFFECT OF COPPER ADDITION ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 2.0% BORNITE .....	142
FIGURE 5-39: EFFECTS OF COPPER ON ATS CONSUMPTION.....	143
FIGURE 5-40: EFFECTS OF COPPER ON GOLD EXTRACTION AT 0.5 HOUR LEACH .....	144
FIGURE 5-41: EFFECTS OF COPPER CONCENTRATION ON GOLD EXTRACTION AT 3.0 HOURS LEACH.....	144
FIGURE 5-42: EFFECT OF ATS ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRITE.....	146
FIGURE 5-43: EFFECT OF ATS ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRRHOTITE.....	146
FIGURE 5-44: EFFECT OF ATS ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 4.0% ARSENOPYRITE.....	147
FIGURE 5-45: EFFECT OF ATS ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 8.0% CHALCOPYRITE .....	148
FIGURE 5-46: EFFECT OF ATS ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 4.0% CHALCOCITE.....	149

FIGURE 5-47: EFFECT OF ATS ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN PRESENCE OF 2.0% BORNITE .....	150
FIGURE 5-48: EFFECTS OF NaOH ON THE CONCENTRATION OF $\text{NH}_3/\text{NH}_4^+$ AND SLURRY PH.....	154
FIGURE 5-49: EFFECT OF NaOH IN THE PRESENCE OF 16.0% PYRITE.....	155
FIGURE 5-50: EFFECT OF NaOH ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRRHOTITE.....	156
FIGURE 5-51: EFFECT OF NaOH ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 4.0% ARSENOPYRITE .....	157
FIGURE 5-52: EFFECT OF NaOH ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 8.0% CHALCOPYRITE .....	158
FIGURE 5-53: EFFECTS OF NaOH ON GOLD EXTRACTIONS.....	159
FIGURE 5-54: EFFECTS OF NaOH ON THIOSULFATE CONSUMPTIONS.....	160
FIGURE 5-55: EFFECT OF $\text{Ca}(\text{OH})_2$ ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 4.0% ARSENOPYRITE .....	161
FIGURE 5-56: EFFECT OF TRISODIUM PHOSPHATE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRITE .....	163
FIGURE 5-57: EFFECT OF TRISODIUM PHOSPHATE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRRHOTITE .....	163
FIGURE 5-58: EFFECT OF SODIUM PHOSPHATE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN PRESENCE OF 8.0% CHALCOCITE .....	164
FIGURE 5-59: EFFECT OF TRISODIUM PHOSPHATE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 8.0% CHALCOPYRITE.....	165
FIGURE 5-60: EFFECT OF TRISODIUM PHOSPHATE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 2.4% GALENA .....	166
FIGURE 5-61: EFFECT OF TRISODIUM PHOSPHATE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN PRESENCE OF 4.0% ARSENOPYRITE.....	167
FIGURE 5-62: EFFECTS OF PHOSPHATE ON GOLD EXTRACTIONS .....	168
FIGURE 5-63: EFFECTS OF PHOSPHATE ON ATS CONSUMPTIONS .....	168

FIGURE 5-64: EFFECT OF GALENA ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16% PYRITE.....	170
FIGURE 5-65: EFFECT OF GALENA ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16% PYRRHOTITE.....	170
FIGURE 5-66: EFFECT OF $\text{Na}_2\text{CO}_3$ ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 2.4% GALENA .....	172
FIGURE 5-67: EFFECT OF OXYGEN PERCENTAGE IN THE LEACHING OF THE REFERENCE ORE .....	174
FIGURE 5-68: EFFECT OF AMMONIA CONCENTRATION ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION UNDER LIMITED OXYGEN CONTENT. ....	175
FIGURE 5-69: EFFECT OF COPPER CONCENTRATION ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION UNDER LIMITED OXYGEN CONTENT. ....	175
FIGURE 5-70: EFFECT OF ATS CONCENTRATION ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION UNDER LIMITED OXYGEN CONTENT.....	176
FIGURE 5-71: EFFECT OF OXYGEN PERCENTAGE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRITE.....	177
FIGURE 5-72: EFFECT OF OXYGEN PERCENTAGE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN PRESENCE OF 16.0% PYRRHOTITE .....	178
FIGURE 5-73: EFFECT OF COPPER ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 16.0% PYRITE.....	179
FIGURE 5-74: EFFECT OF CU ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN PRESENCE OF 16.0% PYRRHOTITE .....	180
FIGURE 5-75: EFFECT OF CU ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 4.0% ARSENOPYRITE.....	181
FIGURE 5-76: THIOSULFATE LEACHING OF GOLD IN THE PRESENCE OF PYRITE WITH REDUCED OXYGEN SUPPLY AFTER PRE-AERATION .....	183
FIGURE 5-77: THIOSULFATE GOLD LEACHING IN THE PRESENCE OF PYRRHOTITE WITH REDUCED OXYGEN SUPPLY AFTER PRE-AERATION .....	184

FIGURE 5-78: THIOSULFATE GOLD LEACHING IN THE PRESENCE OF CHALCOPYRITE WITH REDUCED OXYGEN SUPPLY AFTER PRE-AERATION.....	185
FIGURE 5-79: SPECIFICATION OF EN, HEN <sup>+</sup> OR H <sub>2</sub> EN <sup>2+</sup> .....	188
FIGURE 5-80: MOLECULAR STRUCTURE OF Au(EN) <sub>2</sub> <sup>3+</sup> .....	189
FIGURE 5-81: A PROPOSED HALF REACTION MODEL FOR GOLD DISSOLUTION IN THIOSULFATE-EDA-COPPER SOLUTION (AMMONIA OR AMMONIUM ANION IS OPTIONAL).....	191
FIGURE 5-82: EH-PH DIAGRAM OF THIOSULFATE-EDA-AMMONIA-COPPER SYSTEM (I).....	194
FIGURE 5-83: EH-PH DIAGRAM OF THIOSULFATE-EDA-COPPER SYSTEM (II).....	195
FIGURE 5-84: EH-PH DIAGRAM OF THIOSULFATE-EDA-AMMONIA-COPPER SYSTEM (III).....	196
FIGURE 5-85: EFFECT OF CUEN <sub>3</sub> CONCENTRATION ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION.....	199
FIGURE 5-86: EFFECT OF STS CONCENTRATION ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION.....	200
FIGURE 5-87: EFFECT OF EDA:CU MOL RATIO ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION.....	201
FIGURE 5-88: EFFECT OF OXYGEN PERCENTAGE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION (STS 0.1M, CU 1.2 MM).....	202
FIGURE 5-89: EFFECT OF PH ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION (STS 0.1M, CU 1.2 MM, OXYGEN 21.1%).....	203

## List of Tables

TABLE 2-1: Au(I) COMPLEX STABILITY CONSTANTS (K) [NAVARRO ET AL., 2002].....	7
TABLE 2-2: LEACHING CONDITIONS OF SOME NATURAL GOLD ORES .....	20
TABLE 3-1: ESTIMATED REDOX POTENTIALS (VOLTS) OF $CuL_x^y/Cu(S_2O_3)_3^{5-}$ .....	78
TABLE 3-2: PKSP OF METAL PHOSPHATE PRECIPITATES [DEAN, 1985] .....	82
TABLE 4-1: MINERAL COMPOSITIONS OF THE NATURAL MINERAL SAMPLES USED IN THIS RESEARCH ...	4-91
TABLE 4-2: ELEMENTAL COMPOSITIONS OF THE NATURAL MINERALS SAMPLES USED IN THIS RESEARCH	92
TABLE 5-1: INITIAL SLURRY POTENTIALS IN THE PRESENCE OF VARIOUS MINERALS (STANDART HYDROGEN ELECTRODE) .....	99
TABLE 5-2: PYRITE OXIDATION % (AT 3.0 HOURS) .....	104
TABLE 5-3: EFFECT OF AMMONIA ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 2.4% GALENA .....	131
TABLE 5-4: EFFECT OF COPPER ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN PRESENCE OF 2.4% GALENA .....	143
TABLE 5-5: EFFECT OF ATS CONCENTRATION IN THE PRESENCE OF 2.4% GALENA .....	151
TABLE 5-6: EFFECTS OF NaOH ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 2.4% GALENA .....	159
TABLE 5-7: EFFECT OF SODIUM PHOSPHATE ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION (THE REFERENCE ORE) .....	162
TABLE 5-8: EFFECT OF Cu ON GOLD EXTRACTION AND THIOSULFATE CONSUMPTION IN THE PRESENCE OF 8% CHALCOPYRITE.....	182
TABLE 5-9: THIOSULFATE-EDA-COPPER EXPLORATIVE LEACHING TESTS .....	198
TABLE 5-10: TS-EDA LEACH OF CHALCOCITE-BEARING GOLD ORE .....	204
TABLE 5-11: THIOSULFATE-EDA-COPPER LEACHING OF CHALCOPYRITE-BEARING GOLD ORE .....	204
TABLE 5-12: THIOSULFATE-EDA-COPPER LEACHING OF PYRITE AND PYRRHOTITE BEARING GOLD ORE	205
TABLE 5-13: THIOSULFATE-EDA-COPPER LEACHING OF GALENA-BEARING GOLD ORE .....	206

TABLE 5-14: THIOSULFATE-EDA-COPPER TESTS ON ORE #3 .....	211
TABLE 5-15: THIOSULFATE-EDA-COPPER TESTS ON ORE #4 .....	212

## List of Abbreviations

TS	= Thiosulfate, $S_2O_3^{2-}$
ATS	= Ammonium thiosulfate, $(NH_4)_2S_2O_3$
STS	= Sodium thiosulfate, $Na_2S_2O_3 \cdot 5H_2O$
AH	= Ammonia, Ammonium hydroxide, $NH_4OH$
CS	= Copper sulfate, $CuSO_4 \cdot 5H_2O$
EDTA	= Ethylene diamine tetracetic acid, $(HO_2CCH_2)_2NCH_2CH_2N(CH_2CO_2H)_2$
EDA	= Ethylene diamine, $H_2NCH_2CH_2NH_2$
en	= Ethylene diamine, $H_2NCH_2CH_2NH_2$
mM	= 0.001 mol/L
M	= 1 mol/L
kg/t	= kg per tonne of ore
CDTA	= Cyclohexanediamine tetraacetic acid
DETA	= Diethylenetriamine
TETA	= Triethylenetetramine
NTA	= Nitrilotriacetic acid

# Chapter 1. Introductions

## 1.1. Cyanidation and its limitations

Gold dissolution in an alkaline cyanide solution was first reported by Elsner in 1846. MacArther patented the cyanidation process in 1887. Nowadays, about 92% of all the gold metal is produced using the cyanidation process in the world [McNulty, 2001].

One of the major problems of the cyanidation process is its reduced efficiency, i.e. high reagent consumption and/or low gold extraction, during the treatment of refractory and complex ores containing various detrimental minerals. While the easiest ore being consumed preferentially, the future gold industry may have to treat ores with increasing refractoriness [La Brooy et al, 1994].

Another significant problem of cyanidation is the use of cyanide, an extreme toxic chemical that has been banned in some countries and areas, e.g., Czech Republic, Turkey, Greece, Colorado, Wisconsin, Montana State, Esquel (Argentina) and Costa Rica [Rainforest Information Centre, 2007]. In the rest regions of North America and Australia, the legal banning of cyanidation was unsuccessful. However, under growing pressures due to environmental concerns, mining companies are now applying the stringent regulations and codes on the transportation, handling, usage and disposal of cyanide containing materials. Unfortunately, these measures may not be sufficient to completely avoid cyanide

related incidents. A list of recent cyanide spillage disasters would suggested that the only ultimate solution for this problem is to replace cyanide with a lixiviant that is significant environmently friendly [Rosenhek, 2007]. Therefore, it is believed that the use of cyanide in mining will become increasingly restricted.

## **1.2. Alternative lixiviants**

Alternative lixiviants were investigated to provide a solution for the treatment of refractory ores and an environmentally friendly gold leaching process. Among these non-cyanide gold lixiviants, thiourea, halogen, thiocyanate and thiosulfate have received most attention in recent years [Wan et al., 1993].

Each of them has significant problems, and therefore has not yet been accepted by the industry. Most of these lixiviants have a higher overall production cost and/or a process that is more difficult to control. For chemistry reasons, some of these new technologies were found to be applicable only to a few specific types of ores. For instance, thiourea is a gold lixiviant requiring an acidic leaching condition. Intensive academic researches and several industrial pilot studies were conducted in the 1980's and 1990's [Tremblay et al, 1996]. However, high reagent consumption, gold passivation problems, and high capital costs associated with the acidic leaching conditions have hindered further developments of this lixiviant. The acidic leaching conditions also produced a high TDS problem (total dissolved solid). Further more, thiourea is considered to be a potential carcinogen. Compared to thiourea, thiosulfate has demonstrated

greater potentials in both leaching performance and environmental considerations. Therefore, thiosulfate has become the most promising non-cyanide lixiviant and is receiving extensive attentions from both academic and industrial areas.

### **1.3. Thiosulfate leaching of gold**

Thiosulfate is capable of forming Au(I) complexes and can, therefore, be used as a gold lixiviant. A typical thiosulfate leaching process uses 0.05 to 2 mol/L ammonium thiosulfate (ATS) and 0.05 to 6 mol/L ammonium hydroxide (AH) as well as 0.01 mM to 500 mM copper ion.

One significant advantage of thiosulfate leaching is its ability to treat preg-robbing ores containing carbonaceous materials [Wan, 1997; West-Sell and Hackl., 2005]. The adsorption of thiosulfate gold complexes on activated carbon or organic carbon was found to be insignificant. Compared to cyanide, thiosulfate (ammonium) is a much safer chemical. The toxicity data rating of sodium cyanide is: TLV/TWA 5 mg m<sup>-3</sup>, ORL-RAT LD50 6.4 mg kg<sup>-1</sup>, IPR-RAT LD50 4.3 mg kg<sup>-1</sup>. While the toxicity rating of ammonium thiosulfate is: ORL-RAT LD50 2890 mg kg<sup>-1</sup> [MSDS]. Thiosulfate salts, including ATS, are regarded as nonhazardous by Worksafe Australia [NOHSC, 1999]. They are also classified as “General Recognized as Safe” in the US and not considered as dangerous substances by European standards [Bean, 1997; EEC/FDA, 2001; Langhans et al., 1992]. The major environmental issue with thiosulfate leaching is the use of ammonia

(toxicity rating is: TLV/TWA 35 mg m<sup>-3</sup>, ORL-RAT LD50 350 mg kg<sup>-1</sup>). It is also thought that thiosulfate leaching might provide an alternative option for the leach of copper-bearing ores, which are not economical in cyanidation. Another fact that has to be considered is that ammonium thiosulfate (US\$0.1-0.15/kg) is about 10 times cheaper than sodium cyanide (US\$1.8-2.0/kg) in bulk chemical market.

Research has demonstrated that thiosulfate leaching has similar kinetics than cyanidation. However, there are numerous of obstacles that hinder the application of thiosulfate process. First of all, the reaction mechanism of the ATS-AH-Cu leaching system is a lot more complicated than cyanidation and most of other leaching processes, which results in more difficulties in the research and application of this technology [Wan, 1997]. The instability of thiosulfate was also found to be a severe problem of thiosulfate leaching, which is responsible for elevated reagent costs, reduced gold extractions, difficulties in solution recycling, and resin poisoning in the ion exchange recovery of gold. In a recent publication, Aylmore [2001] reported a thiosulfate consumption at 36 to 165 kg/t ton of ore after 96 hours leach of a copper bearing sulfidic ore. Another problem of thiosulfate leaching is the use of ammonia that could lead to health hazards and may become another source of pollution when not handled correctly.

More work is required in the area of gold recovery from the thiosulfate leaching slurry or pregnant solution. The most popular recovery technology in cyanidation, CIP/CIL (carbon-in-pulp or carbon-in-leach), was proved to be inefficient in thiosulfate leaching. Zinc and copper cementation were found to be effective but

problematic, either because of the high reagent and operating costs or the contamination of barren solution and the precipitate [Navarro et al., 2004]. Solvent extraction is another effective but immature gold recovery technology under development [Zhao et al., 1997; Liu et al., 2002]. Ion exchange technology was believed to be the most promising one because of the reduced operating costs by recovering gold directly from the slurry without a solid-liquid separation step. However, more effort needs to be placed on overcoming the resin poisoning problems that is caused by the preferential adsorption of polythionates on resin [Zhang, et al., 2004].

In the treatment of sulfidic ores, thiosulfate technology also displays lower efficiency, i.e., high reagent consumption, low gold extraction or both,. The interferences of associated minerals may increase the complexity of the reaction chemistry and the instability of thiosulfate. Therefore, it is necessary to conduct systematic investigations on the effects of minerals on thiosulfate leaching of free milling gold.

#### **1.4. Objectives of this investigation**

- To study the effects of some associated minerals, including iron sulfides, copper sulfides, lead minerals/salts and arsenopyrite, on thiosulfate gold leaching;
- To analyze the effects of minerals using thermodynamic approaches, such as Eh-pH diagrams;

- To study the effects of reagent concentrations, including thiosulfate, copper and ammonia concentrations, on thiosulfate gold leaching in the presence of various detrimental minerals and;
- To suggest and examine various methods for increasing gold extractions and decreasing thiosulfate consumptions in the presence of detrimental minerals.

## Chapter 2. Literature review

### 2.1. Development and history of thiosulfate-gold leaching

Thiosulfate forms complexes with many metal ions. These properties of thiosulfate are related to the sulfide-like sulfur atom. As shown in Table 2-1, the Au(I) ion forms strong complex with the thiosulfate, which is more stable than the thiourea and the thiocyanate complexes.

**Table 2-1: Au(I) complex stability constants (K) [Navarro et al., 2002]**

Ligand	Log K
Cyanide	38.3
Thiocyanate	17.0
Thiourea	23.3
Ammonia	26.0
Thiosulfate	28.0

Thiosulfate was first reported as a silver lixiviant in the 19<sup>th</sup> century. Following a chlorination roast, the silver ore was leached with STS. The dissolved silver was then recovered by Na<sub>2</sub>S as silver sulfide [Liddell, 1945]. Berezowsky and Gromely [1978; 1979] described an ATS leaching process under atmospheric condition on an ammonia oxidative pressure leach residue. Up to 95% and 98% gold and silver were extracted within 2-4 hours at 0.4-0.8 mol/L ATS, 40-60°C. Kerley [1981; 1983] patented a thiosulfate process using sulfite as a stabilizer. The laboratory tests were successful using a leaching condition of 1.2 mol/L ATS, 1-4 g/L Cu(II), >0.05% sulfite and pH>7.5 (by ammonia) but the process failed in production scale. Perez and Galaviz [1987] tried to improve the thiosulfate

processes by increasing the pH to 9.5 and above. Zipperian and Raghavan [1988] revealed the importance of using a copper catalyst under atmospheric conditions. A copper bearing ore was leached at 2 mol/L ATS, 4.1 mol/L NH<sub>3</sub> and 6 g/L Cu(II), and 50 °C within 2 hours. To avoid the precipitation of Cu ion as Cu<sub>2</sub>S, they suggested to maintain a pH>10 and Eh above 200 mV. The total consumption of thiosulfate, in their work, was about 50%. A low concentration formula was used by Langhans et al [1988] in the heap leaching of an oxidized ore. The pH was controlled between 9 and 11 with 0.05-0.2 mol/L ATS. Consequently, the ATS consumption was reduced to 0.4 kg/t. Cao et al [1992] confirmed the possibility of leaching gold with low concentration of thiosulfate. 95% gold was extracted within 2 hours at 0.2 mol/L TS without addition of sulfite. An industrial pilot study was conducted by Newmont Gold in the 1990's. An ATS technology was applied in their heap leaching of a bio-oxidized double-refractory gold ore [Wan, 1997]. Acceptable gold extractions were achieved at 0.1 mol/L ATS, 0.1 mol/L NH<sub>3</sub> and 30 ppm Cu(II). The ATS consumption was 12-15 kg/t. Marchbank et al [1996] reported an ATS leaching process on a pressure oxidized ore. Following the neutral or alkaline pressure oxidation, the ore was leached at 0.025 – 0.1 mol/L ATS, 50-100 ppm Cu(II), 40-55 °C, pH 7.0-8.7. 1 mM sulfite was used to stabilize thiosulfate. In 2000, Barrick Gold carried out an extensive trial of ATS leaching of their pressure oxidized double refractory gold ore [Fleming, 2003]. In a recent study of ATS leaching of a copper bearing sulfide concentrate [Aylmore, 2001], a high ATS concentration was used. At 0.8 mol/L ATS, 4 M NH<sub>3</sub>, pH 10.2, 0.05 M Cu(II), 96% gold was leached. However, the

consumption of ATS reached 36-165 kg/t. Therefore, he investigated an ATS leaching process with thiosulfate *in situ* generated by adding elemental sulfur and sulfur dioxide in the leaching process. Up to 89% gold was leached. Ji, et al., [2003] published an STS leaching process without any addition of Cu(II) or NH<sub>3</sub>. Close to 80% gold was extracted within 6-8 hours at 60-80 °C, 10-100 psi oxygen pressure. Gold was recovered with an ion exchange method with success.

Generally, in the early stage of thiosulfate leaching, high temperatures and pressures were used to ensure an acceptable gold extraction. Since the 1980's, most works have focused on atmospheric thiosulfate leaching under ambient or slightly elevated temperatures. More recently, efforts have been made to leach oxidized ores using diluted ATS-Cu(II)-NH<sub>3</sub> solutions. However, it was also noticed that diluted leaching solutions were not suitable for complex sulfide ore and therefore, high thiosulfate concentration and elevated temperature-pressure appeared again in some recent investigations (See Table 2-2).

Academic research on the leaching of gold with thiosulfate has significantly improved the understanding of this technology. A research group at Queen's University led by Dr. Wan-Tai Yen studied thiosulfate leaching of gold comprehensively since 1990's. It has been found that:

- 1) Gold leaching requires only small amounts of copper as the oxidation catalysts. For stabilizing the thiosulfate, the copper concentration must be minimized and carefully controlled [Xia, 2002, 2003].

2) Gold leaching requires only a small amount of dissolved oxygen (DO 0.3 ppm). The gold extraction was not affected significantly by reducing the Do level from 8.6 to 0.3 ppm. However, by limiting the dissolved oxygen concentration, the thiosulfate consumption can be minimized [Xia, 2002, 2003].

3) For the control of the copper species in the leaching solution and to minimize the thiosulfate oxidation by Cu(II) ammine, copper ligands, other than ammonia, can be used as chemical additives so that the consumption of thiosulfate may be minimized. Two examples of these copper ligands tested are EDTA and NTA [Xia, 2002, 2003].

4) Nickel ammine, instead of Cu(II) ammine was suggested by Arima et al. [2004] as a possible oxidation catalyst for thiosulfate-gold leaching. The thiosulfate consumption was reduced and the copper co-adsorption on the resin was avoided.

Works conducted by a research group at Monash University focused on the interpretation and understanding for thiosulfate leaching kinetics and chemical reaction using electrochemistry methods. Their results confirmed the importance of minimizing oxygen and the copper supply and the addition of copper ligands in thiosulfate leaching of gold and suggested that anions, such as phosphate may significantly improve the stability of thiosulfate [Breuer et al, 2002, 2003, 2004]. REQCM (Rotating Electrochemical Quartz Crystal Microbalance) was widely used as a new tool in the investigations of the leaching kinetics. Inner sphere reaction theory was repeatedly applied to explain the observations in their

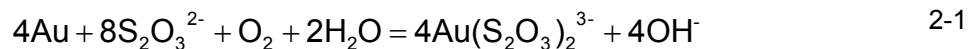
research and was the fundamental mechanism for the inhibition of thiosulfate consumption with phosphate anion.

New advances were also made in the research to find alternative leaching catalysts other than copper and ammonia. One example is the use of Fe-EDTA-TU (thiourea) in the thiosulfate leaching at pH 6-7 [Zhang, et al., 2005]. Another example is the use of Fe-Oxalate-TU (thiourea) in the thiosulfate leaching at pH 6-7 [Chandra et al., 2005].

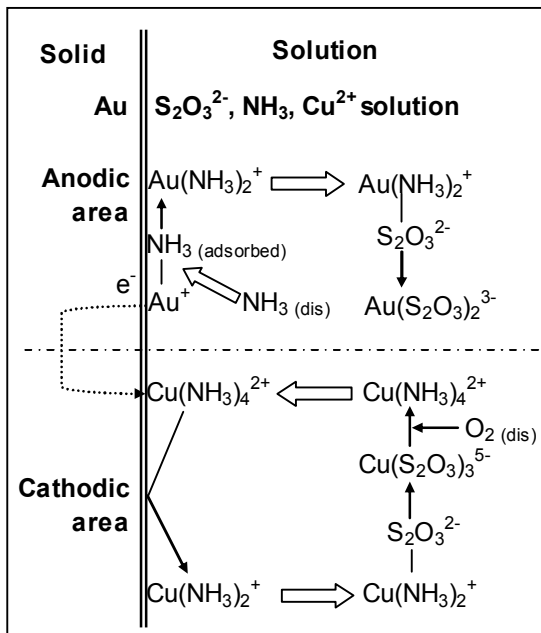
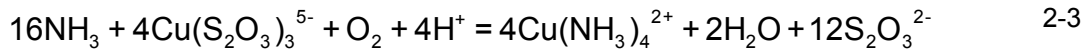
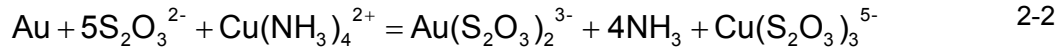
Attentions were also paid to the *in situ* generation of thiosulfate in the treatment of sulfidic ores [Fang, et al., 2002]. It is expected that through these technologies, thiosulfate reagents will be produced during or prior to the thiosulfate leaching of sulfidic ores so that the reagent costs will be minimized.

## **2.2. Chemistry of gold dissolution in thiosulfate solutions**

The leaching of gold in thiosulfate solutions is a metal oxidation process as described in an overall equation 2-1.



In ammoniacal thiosulfate gold leaching using cupric ammine as an oxidant, or more precisely, oxidation catalyst, the reaction chemistry can be explained by the intermediate reactions (Equation 2-2 and 2-3).



**Figure 2-1: Gold dissolution sketch (ammoniacal copper thiosulfate system)**

The actual reaction system is much more complicated; But the most important question is how ammonia participates in this reaction mechanism. Other than forming copper ammonia complex, it was believed that ammonia is preferentially adsorbed on gold surface and forms a Au(I) complex,  $\text{Au}(\text{NH}_3)_2^+$ . This complex

diffuses to the bulk solution, reacts with the thiosulfate anion so that the dissolved Au(I) is ultimately stabilized as  $\text{Au}(\text{S}_2\text{O}_3)_2^{3-}$ . Both thermodynamic studies and electrochemistry studies have supported this theory. However, no direct evidence has been found for the existence of  $\text{Au}(\text{NH}_3)_2^+$ . A reaction model based on this theory is shown in Figure 2-1.

According to this theory, the dissolution of gold is a “corrosion” process. The ultimate oxidant is oxygen or any other oxidative chemicals that are capable of oxidizing Cu(I) ammine to Cu(II) ammine. Unlike cyanidation, thiosulfate leaching of gold does not require a diffusion of oxygen onto the solid-liquid interface of gold, which can be very slow under atmospheric conditions. The direct oxidant of gold is Cu(II) ammine after it is adsorbed on to the surface of gold in the cathodic area. After accepting one electron, Cu(II) ammine is reduced to Cu(I) ammine, which is desorbed and diffused into the bulk solution phase. In the anodic area of gold surface, a gold atom loses one electron and produces an aurous ion. It is then complexed with ammonia. The aurous ammonia complex diffuses to the bulk solution phase where ammonia is substituted and aurous ion is stabilized as a more stable complex,  $\text{Au}(\text{S}_2\text{O}_3)_2^{3-}$ . There are four diffusion steps: the diffusions of ammonia and Cu(II) ammine to the gold solid-liquid interface and the diffusions of Cu(I) ammine and Au(I) ammine complex to the bulk solution phase. In all these diffusion steps, ammonia plays important roles. This partially explains the slow dissolution rates in the absence of sufficient ammonia ligand. First of all, ammonia works as ligand of copper, which stabilizes copper and forms the oxidation catalyst in the anodic area. Other than the ammonia complexed with

copper, free ammonia must present so that they can adsorb on the surface of gold and form an aurous complex.

More recently, a research has claimed that the Cu(I) complex is actually a mixed ammonia-thiosulfate complex [Choo et al., 2004] (Equation 2-3). Based on inner sphere reaction mechanism, coordination of other anions, such as,  $\text{OH}^-$ ,  $\text{PO}_4^{3-}$ ,  $\text{SO}_4^{2-}$ ,  $\text{NO}_3^-$  in the structure of Cu(II) ammine was also suggested to have some influence on the reaction chemistry. It was claimed that in presence of these anion, the coordination of thiosulfate in Cu(II) ammine will be retarded and therefore the oxidation of thiosulfate by this Cu(II) complex will be significantly slower. The significance of this claim on the actual leaching performance is not clear yet. Other than Cu(II) ammine, it was also claimed that Ni(II) ammine may also work as the oxidation catalyst of gold [Arima, et al, 2004, 2003]. Instead of  $\text{Cu}(\text{NH}_3)_4^{2+}/\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$ , another redox couple,  $\text{Ni}(\text{NH}_3)_6^{2+}/\text{Ni}_3\text{O}_4(\text{hydr})$ , works as the oxidation catalyst of gold. As  $\text{Ni}_3\text{O}_4$  is solid, therefore, the success of this mechanism must be kinetically dependant on the rate of dissolved oxygen diffusion to the solid surface of  $\text{Ni}_3\text{O}_4$ . More investigation is required to further understand Ni catalyzed gold dissolution. Iron was also reported as a potential catalyst in thiosulfate leaching of gold. Iron (III) EDTA or the iron (III) oxalate complex was reported to be able to oxidize gold in presence of thiourea at neutral or weak acidic condition (pH 6-7) [Zhang, et al., 2005; Chandra et al., 2005].

## 2.3. Chemistry of thiosulfate decomposition

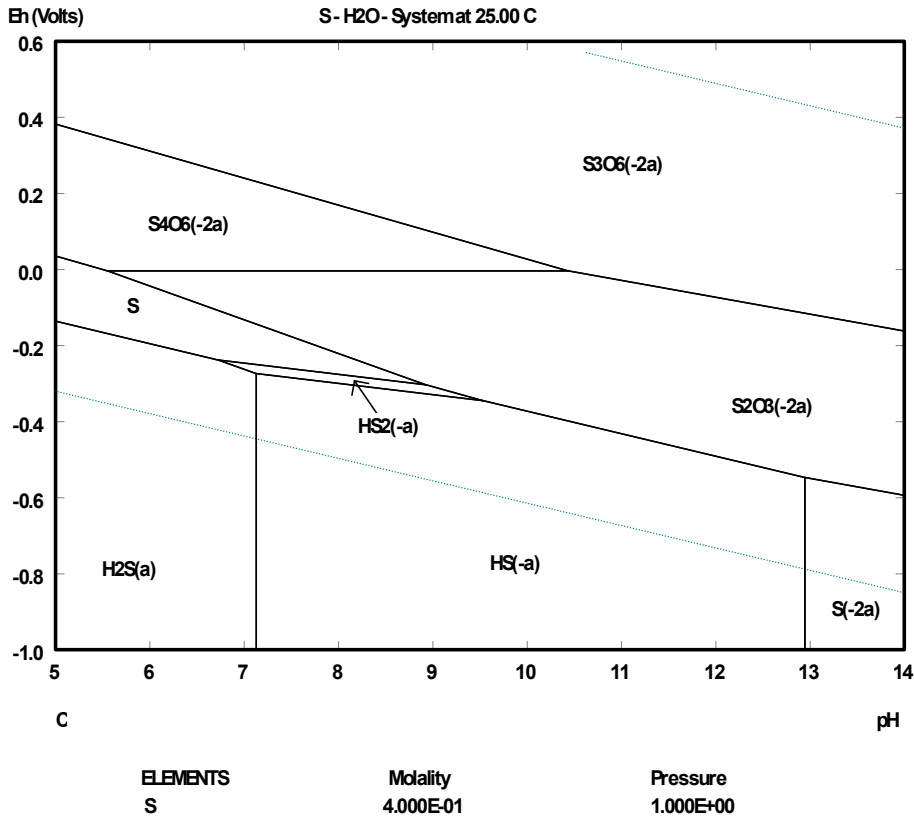


Figure 2-2: Eh-pH diagrams of S-H<sub>2</sub>O system

Thiosulfate is a meta-stable ion. As shown in Figure 2-2, thiosulfate may be decomposed to the sulfide ion ( $S^{2-}$  or  $HS^-$ ) under a strong reductive condition. The major immediate products of thiosulfate under oxidative conditions are tetrathionate and trithionate. At  $pH < 9.5$ , thiosulfate is possibly decomposed to  $HS_2^-$  and elemental sulfur.

Due to the reaction kinetics, the actual chemical reactions are much more complicated. The potential reactions that are expected in a thiosulfate solution can be summarised as follows:.

1) Decomposition of  $S_2O_3^{2-}$  to "S-" and "SO<sub>3</sub>"



Equations 2-4 to 2-9 shows some reactions when the sulfide-like sulfur is detached from thiosulfate anion. Depending on the redox potential, the products of such a chemical decomposition can be either elemental sulfur or sulfide ions (including hydrogen sulfides). Poly-sulfides are possible intermediate products in this process. The formation of elemental sulfur precipitate may occur on the surfaces of both the gold and the associated minerals [Hamilton et al., 1983]. Such a precipitate, if found on the gold surface may substantially passivate the leaching reactions. Under a neutral or acidic condition, the formation of hydrogen

sulfide gas becomes possible, which will result in a significant loss of thiosulfate reagent.

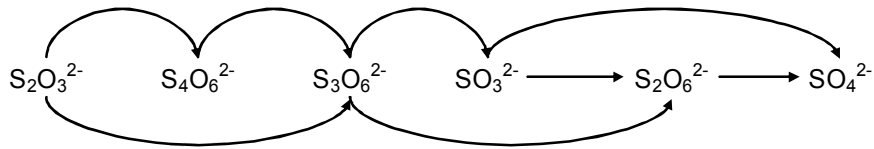
## 2) Reactions with sulfide ion

Sulfide is a reductive species in aqueous solution. Meanwhile, sulfide is a precipitation agent of many metal ions. Therefore, even at low concentrations, sulfide ions may significantly affect the stabilities of dissolved metals.



As shown in Equations 2-10 to 2-13, both gold and copper ions can be precipitated by the sulfide ion. These reactions will jeopardize the gold leaching reactions by either reducing the dissolved gold back to native gold or by passivating the gold surface and slowing down the dissolutions of gold. On the other hand, the precipitation of gold by the sulfide ion can also be used for recovering gold from a clear thiosulfate pregnant solution [West-Sells et al., 2005].

### 3) Oxidation of thiosulfate



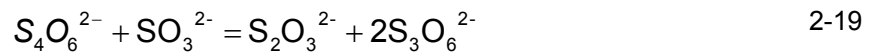
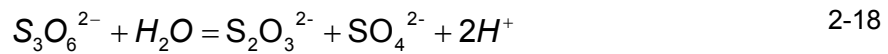
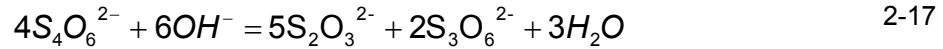
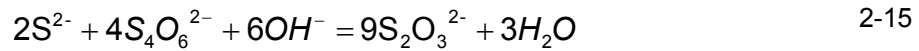
**Figure 2-3: Oxidation reactions of thiosulfate**

As shown in Figure 2-3, the oxidations of thiosulfate and subsequent meta-stable S-O species are quite complicated. Thiosulfate can be oxidized to polythionates (tetrathionate or trithionate). These polythionate can be further oxidized to  $SO_3^{2-}$ ,  $S_2O_6^{2-}$  and  $SO_4^{2-}$ . More importantly, many of these oxidation reactions can be accelerated by the presence of multi-valent metal ions, i.e.,  $Cu^{2+}/Cu^+$ , and  $Fe^{3+}/Fe^{2+}$ . In thiosulfate-copper-ammine leaching system,  $Cu^{2+}$  and  $Cu(NH_3)_4^{2+}$  are capable of catalyzing the oxidations of many S-O species and result in an increased oxidation rate of thiosulfate. These catalysis reactions largely increase the complexity of thiosulfate leaching reaction system.

### 4) Interactions between various S-O species

Further to the copper catalyzed oxidation reactions, there are a few interactions between various S-O species taking place in this system (see Equations 2-14 to 2-19). These reactions further complicate the chemistry of thiosulfate in the gold leaching system.





## 2.4. Leaching of natural gold ores with thiosulfate

Many reports about thiosulfate leaching of natural gold ores were published in the past few decades. As shown in Table 2-2, the leaching conditions and type of ores were listed. In most cases, ambient temperature or slightly elevated temperature (30-70°C) was used.

The most studied ore types include:

- 1) Oxidative pre-treatment residue of carbonaceous sulfide ore

This type of ore is typically very easy to leach with thiosulfate. Therefore, diluted ATS-AH-Cu solutions were used, i.e. ATS <0.1 M, Cu <1 mM, pH 7-9.

- 2) Complex metal sulfide gold ore

**Table 2-2: Leaching conditions of some natural gold ores**

Authors	Type of ore	TS (M)	Cu (mM)	Temp. (oC)	pH/AH (M)	Au Extraction %	ATS consumption
Berezowsky et al., 1978	Cu-S	0.5-1	30-60	25-50	9-10		
Kerley, 1983	Cu-Mn-S	12-25%	15-60	40-60	7-9	95%	4 kg/t
Block-Bolten et al., 1985	Zn-Pb-S	0.1-0.5		21-50	0.75 M	90%	45 lb/t
Perez et al., 1987	Cu-Mn-S	5-15%	15-60	25-40	10-11		
Zipperian et al., 1988	Rhyolite, Mn	0.1	200	50	10	90%	50%
Hemmati et al., 1989	Carbonaceous	0.7	150	35	10.5	73%	15-19%
Ji et al., 1991	Oxidized Cu	0.1-2	8-300	30-65	0.5-5.2 M		
Hu et al., 1991	Sulfide	1	16	40	2 M	95.6%	
Murthy, 1991	Pb-Zn-S	0.1-0.5		21-70	6.9-8.5	95%	
Cao et al., 1992	Sulfide concentrate	0.2-0.3	47	60	10-10.5	95%	4.8 kg/t
Langhans et al., 1992	Oxidized Cu	0.2	1	Ambient	11	90%	2 kg/t%
Gong et al., 1993	Sulfide concentrate	0.7	60	50	1 M		
Wan et al., 1994	Carbonaceous, Sulfide	0.1-0.2	0.5	Ambient	9-10		
Abbruzzese et al., 1995		2	100	25	8.5-10.5	80%	
Groudev et al., 1995	Oxide	0.1	8	Ambient	8.5-9		
Groubev et al., 1996	Fe-Cu-Zn-Pb-S	0.14	8	Ambient	9-10	80%	
Marchbank et al., 1996	Carbonaceous, Sulfide	0.025-0.1	0.8-8	25-80	7-8.7	85%	
Yen et al., 1996	Cu-S	0.4	30	Ambient	11		
Wan, 1997	Carbonaceous, Sulfide	0.1	0.5	Ambient	9		8 kg/t
Yen et al., 1998	Cu-S	0.5	100	Ambient	10	90%	30 kg/t
Thomas et al., 1998	Oxidized sulfide	0.03-0.05	0.1-2	80-85	7-9	85%	
Felming, 2000	Carbonaceous, Sulfide	0.05	0.5	40-60	7.5-8	90%	
Balaz et al., 2000	Sulfide concentrate	0.5	62	70	6-7		
Aylmore, 2001	Cu-S concentrate	0.1-0.8	Dec-62	Ambient	10.2	80%	
Schmitz et al., 2001	Carbonaceous, Sulfide	0.4	16	Ambient	1.9 M		
Molleman et al., 2002	Cu	0.2	15.7	35	10		
Ficeriova et al., 2002	Cu-Pb-Zn-S	0.5	157	70			
Aylmore et al., 2004	Oxide	0.075	1.5	Ambient	0.4 M	90%	
Bhakta, 2003	Carbonaceous, Sulfide	0.01-0.09	0.5-1		8.8-9.2		
West-sells et al., 2005	Carbonaceous, Sulfide, POX	0.1	0.8	Ambient	9	80%	

These types of ore are much more difficult to treat due to the presence of various metal sulfides. Therefore, in most examples, high concentrations of ATS, Cu and ammonia were necessary. As a result of using high reagent concentration, the consumption of reagents was typically on the high side. Under such strong leaching conditions, the gold leaching rate was fair and the final gold extraction was acceptable.

## **2.5. Associated minerals in thiosulfate leaching of gold**

Oxide minerals were found to be capable of adsorbing copper, nickel and cobalt from ammonia solutions [Osseo-Asare et al., 1979]. If this phenomena occurs in thiosulfate leaching, the copper catalyst will be affected and loss it effectiveness.

The results indicated that above 60% of the Ni and Cu could be adsorbed by hematite (160 m<sup>2</sup>/L) in a 0.1 M ammonia/ammonium solution. The adsorption percentage increased with the increase of pulp density. The total adsorbed metal was found to be proportional to the total amount of added metal. The adsorption of metals on the hematite was also found to be affected by the pH. At pH between 9 and 10, the adsorption of the Cu was significantly inhibited. At a higher pH, e.g., 11, the adsorption appeared to more significant. The order of adsorbability appeared to be: hematite>alumina>quartz.

In the leaching of silver with ammonia, it was also observed that silver was lost to the surface of complex sulphide and chalcopyrite concentrates [Acharya, et al., 1986]. It was found that the ferrous and ferric oxides that were freshly generated on the mineral surfaces were responsible for the loss of silver.

Feng et al. [2001] studied the preg-robbing phenomena of some minerals and claimed that chalcopyrite, pyrite, arsenopyrite, sphalerite and kaolinite were preg-robbing in thiosulfate leaching. However, this study was conducted mostly at 0.0 M free thiosulfate and the preg-robbing was not a significant problem under reasonable thiosulfate concentrations, e.g., 0.1 M. For both silica and kaolin

minerals, the adsorption of gold was found to be physical. Whereas, on sulfide minerals, the adsorption was more typically chemical. The problems with sulphide minerals were not further discussed. It was also stated that lead and zinc ions were able to compete with gold in the adsorption on surfaces of the minerals.

Other than adsorption, dissolution of the associated minerals is another type of interference during the leaching performance. The dissolution of minerals in thiosulfate-ammonia leaching is attributed to either thiosulfate or ammonia ligands.

The leaching and oxidation behaviours were studied for some sulfide minerals in ammonia solution [Rao et al., 1998]. An investigation on the effects of minerals in the thiosulfate solution was conducted by Feng et al., [2001]. The leaching behaviours of pyrite, pyrrhotite, arsenopyrite and chalcopyrite were studied. It was found that these minerals could be dissolved partially and consume significant amount of thiosulfate. Chalcopyrite was found to be the most dissolvable mineral among the studied sulfides. Pyrite, pyrrhotite and arsenopyrite were less reactive due to surface passivation. Further more the leaching of these minerals increased at higher concentrations of ammonia or copper. The addition of 0.25 M sulphate ion was found to slightly reduce the leaching rate of these sulfides. Meanwhile, these minerals were found to accelerate the consumption of thiosulfate. In this work, the interaction of leaching

minerals with gold extraction was not discussed and no improving method was provided.

In a study on the oxidative ammonia leaching of sphalerite, copper was found to be an oxidation catalyst [Ghosh et al., 2002].

The effects of leached metal ions on thiosulfate leaching were also investigated and indicated that a low concentration of lead ion could slightly enhance the dissolution of gold. At higher concentration, the lead ion reduced the extraction of gold. The authors claimed that  $\text{AuPb}_2$  formed in the presence of the lead ion. Zinc ion was also detrimental in thiosulfate leaching by reducing the gold extraction. Cobalt and chromium ions were found to be less efficient than copper in catalyzing gold extraction in thiosulfate leaching [Feng et al., 2002]. Most of the results of this investigation indicated that foreign ions were detrimental to gold extraction in thiosulfate leaching. They also consumed free thiosulfate. As in presence of these ions, better gold extractions required higher thiosulfate, ammonia and copper concentration.

Xu et al. [1995] studied the effect of pyrite on thiosulfate decomposition and suggested that the pyrite surface catalyzed thiosulfate oxidation to tetrathionate. In another investigation by Feng et al. [2005], thiosulfate decomposition was studied in the presence of pyrite and pyrrhotite. It was found that both sulfides accelerated the oxidation of thiosulfate, and the concentrations of copper and oxygen were found to significantly affect the consumption of thiosulfate.

In a publication about solution chemistry factors in heap leaching, the behaviour of some sulfides, i.e., pyrite, was discussed [Wan et al, 2003]. Eh-pH diagram for pyrite was discussed and the rest potentials of some minerals were compared. It was found that the rest potentials have the following order: chalcocite < gold < chalcopyrite ~ arsenopyrite < covellite < pyrite.

Due to the lack of a systematic research in the effects of associated minerals during the thiosulfate leaching of gold, the practical impacts of these minerals has yet to be confirmed. No improving approaches have been suggested for overcoming the observed problems.

## **Chapter 3. Theoretical considerations**

### **3.1. Effects of associated minerals**

In this section, a thermodynamic analysis of the thiosulfate leaching system in the presence of some associated minerals will be discussed. The Eh-pH diagram will be used as an important tool for thermodynamic analysis. The fundamental principles required for plotting Eh-pH diagrams will be briefly introduced. Eh-pH diagrams of the thiosulfate leaching system and species distribution diagrams in the absence or presence of mineral impurities will be shown.

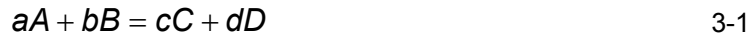
Through this work, some general minerals problems will be analyzed and then the mechanisms discussed in this section will be used in later sections to explain the observations in the leaching tests.

#### **3.1.1. Fundamental principles required for plotting Eh-pH diagrams**

In order to plot an Eh-pH diagram, it is necessary to calculate the equilibrium states based on some specific physical-chemistry principles, which will be briefly introduced as follows.

The direction of any chemical reaction can be determined by the change of the standard Gibbs free energy. The standard Gibbs free energy of formation ( $\Delta G_f^0$ ) of a compound is the change of Gibbs free energy that accompanies the formation of 1 mole of that substance from its component elements, at their

standard states (the most stable form of the element at 298 K and 100 kPa). The change of Gibbs free energy ( $\Delta G^0$ ) of a chemical reaction (e.g. Equation 3-1) can be calculated with Equation 3-2. Only when the  $\Delta G^0$  value is less than zero, does the reaction proceed forwards.



$$\Delta G^0 = c\Delta G_{fC}^0 + d\Delta G_{fD}^0 - a\Delta G_{fA}^0 - b\Delta G_{fB}^0 \quad 3-2$$

where, A and B are the reactants; C and D are the products; a, b, c, d are the molar numbers for balancing of this reaction. The relation between  $\Delta G^0$  and the equilibrium constant of a reaction (K) can be defined by Equation 3-3.

$$\Delta G^0 = -RT \ln K \quad 3-3$$

where, R is the ideal gas law constant (8.314 J/mol K) and T is the temperature in degree Kelvin.

In a complex reaction that has no exchange of electrons, e.g., Equation 3-4,  $\Delta G^0$  can be expressed as Equation 3-5.



$$\Delta G^0 = -RT \ln K = -RT \ln \frac{[A]^a [B]^b}{[C]^c} = c \Delta G_{fC}^0 - a \Delta G_{fA}^0 - b \Delta G_{fB}^0 \quad 3-5$$

For a redox reaction, the Nernst Equation can be used as a bridge between  $\Delta G^0$  and  $E^0$  (standard redox potential). The redox potential of a reaction (E) under a constant temperature can be calculated from  $\Delta G^0$  using Equation 3-6 and 3-7

$$E^0 = \frac{\Delta G^0}{-nF} \quad 3-6$$

$$E = E^0 - \frac{RT}{nF} \ln Q = E^0 - \frac{RT}{nF} \ln \frac{[C]^c [D]^d}{[A]^a [B]^b}$$

where, n is the molar number of electrons transferred in this redox reaction; F is the Faraday constant (96485 c/mol); R is the ideal gas law constant (8.314 J/mol K); and Q is the reaction quotient. In a closed system, Q tends to ultimately approach a constant value K (the equilibrium constant). Then Equation 3-7 becomes Equation 3-8.

$$E = E^0 - \frac{RT}{nF} \ln K \quad 3-8$$

The equilibrium states of most chemical reactions in aqueous solutions can be expressed in an Eh-pH diagram. The above described thermodynamic principles

can be used to calculate the Nernst equations of those reactions of interest in this system. Using these Nernst equations, an Eh-pH diagram can be built [Pourbaix,1966]. Through the use of Eh-pH diagrams, the stability of a mineral or a chemical species in terms of Eh and pH will be shown.

### **3.1.2. Eh-pH diagram of the thiosulfate-gold leaching system**

In the first two Eh-pH diagrams (Figures 3-1 and 3-2), the Nernst equations were listed together with the diagrams. For the remaining diagrams, a thermodynamic computing program was used to assist with the calculations. The detailed calculation steps will not be shown for those Eh-pH diagrams.

When using an Eh-pH diagram to analyze the thiosulfate leaching system, it should be noted that,

- A. In this study, the thiosulfate leaching systems were studied only under ambient temperature (298 K) and atmospheric pressure (100 kPa).
- B. In most of the Eh-pH diagrams, some final reaction products were intentionally eliminated from the calculation for the following purposes: (i) The equilibrium state of the leaching system is not reached during the leaching period. To study the reaction chemistry and the effects of some species of interest, some final products that are not important to the leaching system, i.e. sulfate anion, must be eliminated; (ii) In a thiosulfate leaching system under ambient and atmospheric conditions, the Eh-pH area

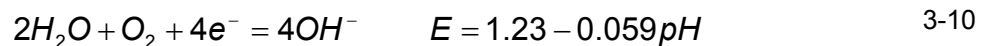
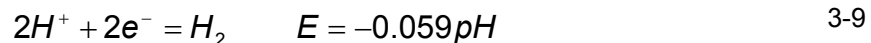
of interest was mainly between +0.5 to -0.5 V and pH 6 to pH 14. Products appearing outside of this area might be ignored in the calculations.

- C. For a complicated reaction system such as the thiosulfate leaching of gold, multiple Eh-pH diagrams must be plotted to reflect the reaction chemistry at each different stages, in order to analyze different species of interest and to serve different purposes of study.
- D. An Eh-pH diagram is plotted based on known thermodynamic data collected from various publications or databases. Inaccurate thermodynamic data may result in deviations in the Eh-pH diagrams.

#### **3.1.2.1. Gold dissolution by thiosulfate and ammonia**

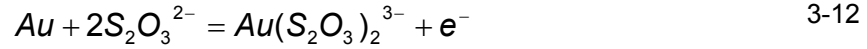
The most important reactions in a thiosulfate leaching system are the oxidations of gold and the formation of the aurous complex. Therefore, two Eh-pH diagrams were discussed as follows, focusing on the dissolution of gold.

As discussed in section 3.1.1, an Eh-pH diagram requires Nernst equations for defining the boundaries of each window of species. The equations required for Figures 3-1 and 3-2 are listed as follows (see Equations 3-9 to 3-15).





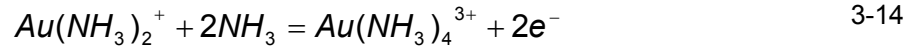
$$[NH_4OH] = ([NH_4^+] + [NH_4OH]) / (1 + 10^{9.25-pH})$$



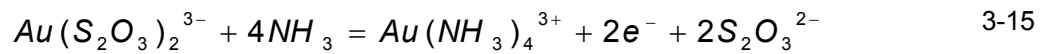
$$E = 0.15 + 0.059 \log[Au(S_2O_3)_2^{3-}] - 0.118 \log[S_2O_3^{2-}]$$



$$E = 0.12 + 0.059 \log[Au(NH_3)_2^+] - 0.118 \log[NH_3^+]$$



$$E = 0.89 + 0.059 \log\{[Au(NH_3)_4^{3+}] / [Au(NH_3)_2^+]\} - 0.059 \log[NH_3^+]$$



$$E = 0.81 + 0.059 \log\{[Au(NH_3)_4^{3+}] / [Au(S_2O_3)_2^{3-}]\} \\ - 0.118 \log[NH_3^+] + 0.059 \log[S_2O_3^{2-}]$$

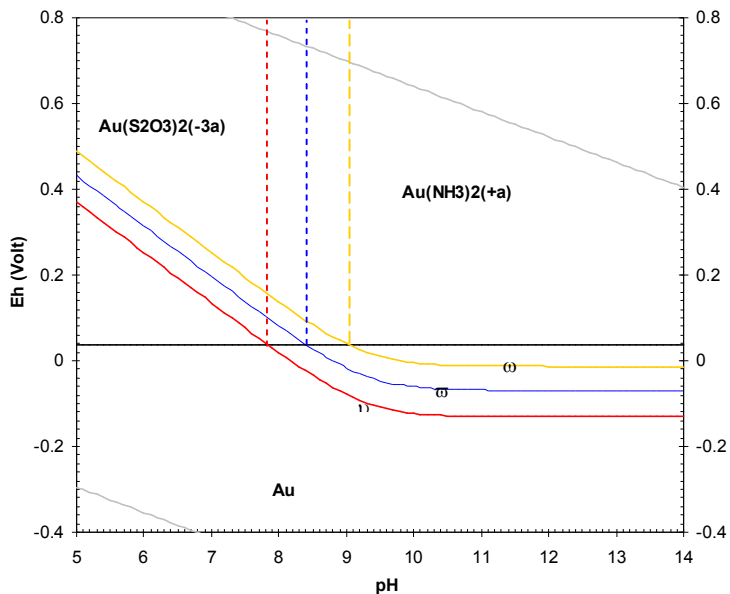


Figure 3-1: Eh-pH diagram of  $\text{Au-H}_2\text{O-S}_2\text{O}_3^{2-}\text{-NH}_3/\text{NH}_4$  system (I): effect of ammonia concentration

(Au: 0.5 mM, S: 0.40 M,  $\text{NH}_3$ : 3.4 M,  $\text{NH}_4^+$ : 1.3 M,  $\text{NH}_2^-$ : 0.4 M)

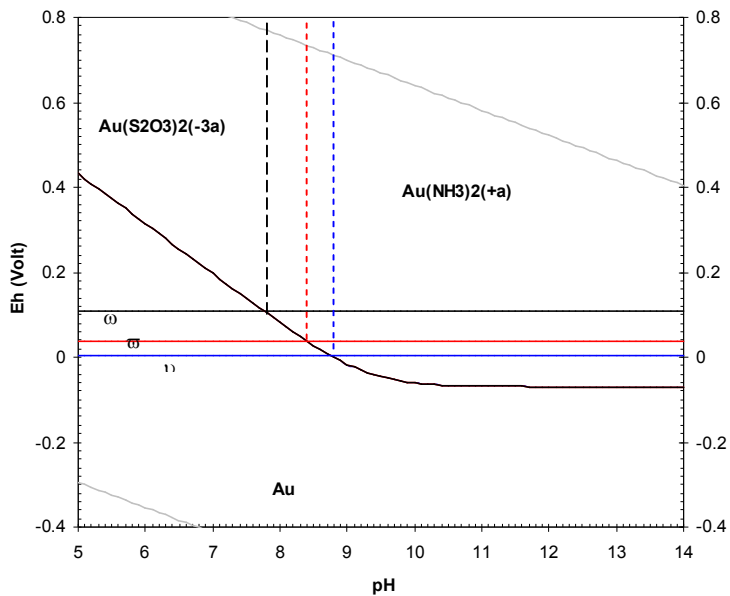


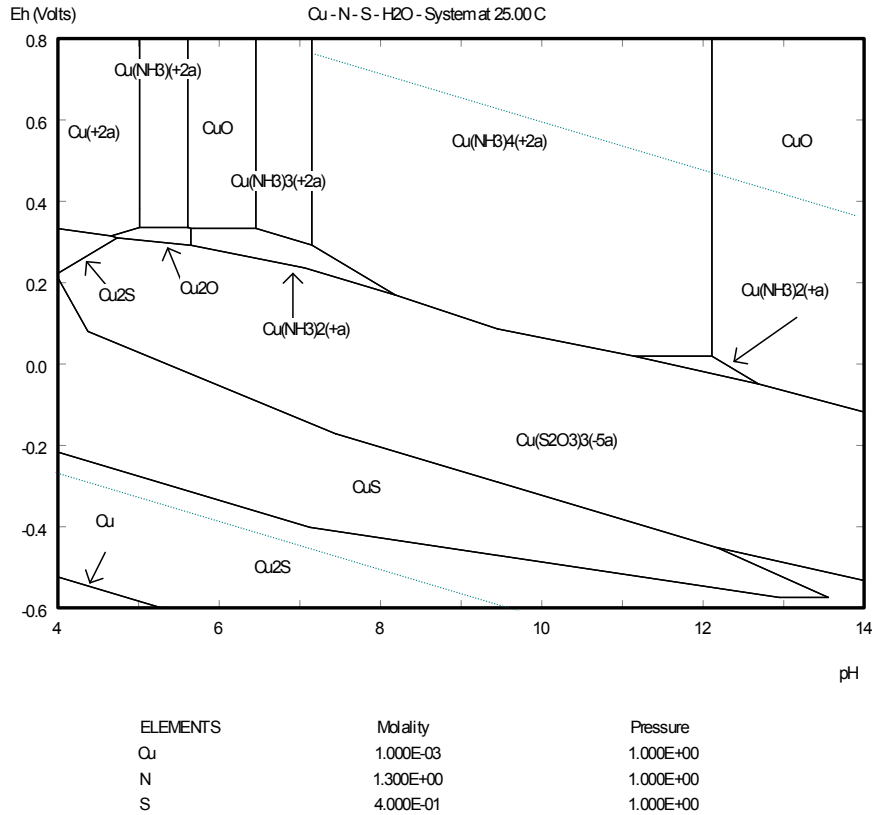
Figure 3-2: Eh-pH diagram of  $\text{Au-H}_2\text{O-S}_2\text{O}_3^{2-}\text{-NH}_3/\text{NH}_4$  system (II): effect of thiosulfate concentration

(Au: 0.5 mM, N: 0.9 M,  $\text{S}_2\text{O}_3^{2-}$ : 0.80 M,  $\text{S}_4\text{O}_6^{2-}$ : 0.40 M,  $\text{S}_2\text{O}_3^{2-}$ : 0.10 M)

Figures 3-1 and 3-2 are the Eh-pH diagrams of the Au-H<sub>2</sub>O-S<sub>2</sub>O<sub>3</sub><sup>2-</sup>-NH<sub>3</sub>/NH<sub>4</sub> system. The effects of ammonia and thiosulfate concentrations are demonstrated with these diagrams respectively. To oxidize gold, the oxidation potential should be higher than 0 to -0.2 V, depending on the concentration of ammonia. The potential shifts downwards with increasing the ammonia concentration (Figure 3-1). To stabilize the dissolved gold as the thiosulfate complex, the oxidation potential should be higher than 0 to 0.1 V, depending on the concentration of thiosulfate. The potential shifts downwards when the thiosulfate concentration is higher (Figure 3-2). In general, increasing the ammonia and thiosulfate concentrations are beneficial for the dissolution of gold because a less oxidative leaching condition is required.

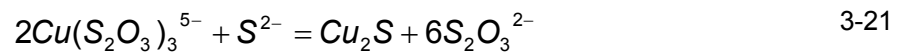
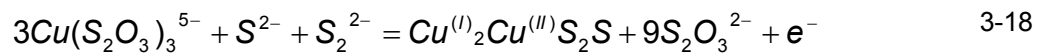
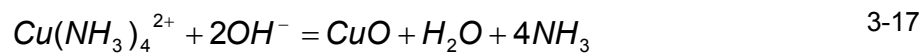
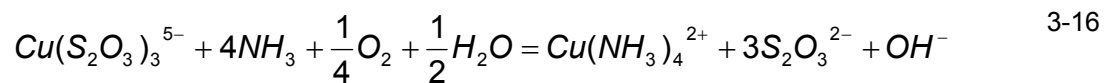
#### **3.1.2.2. The stability of copper ammine in thiosulfate leaching**

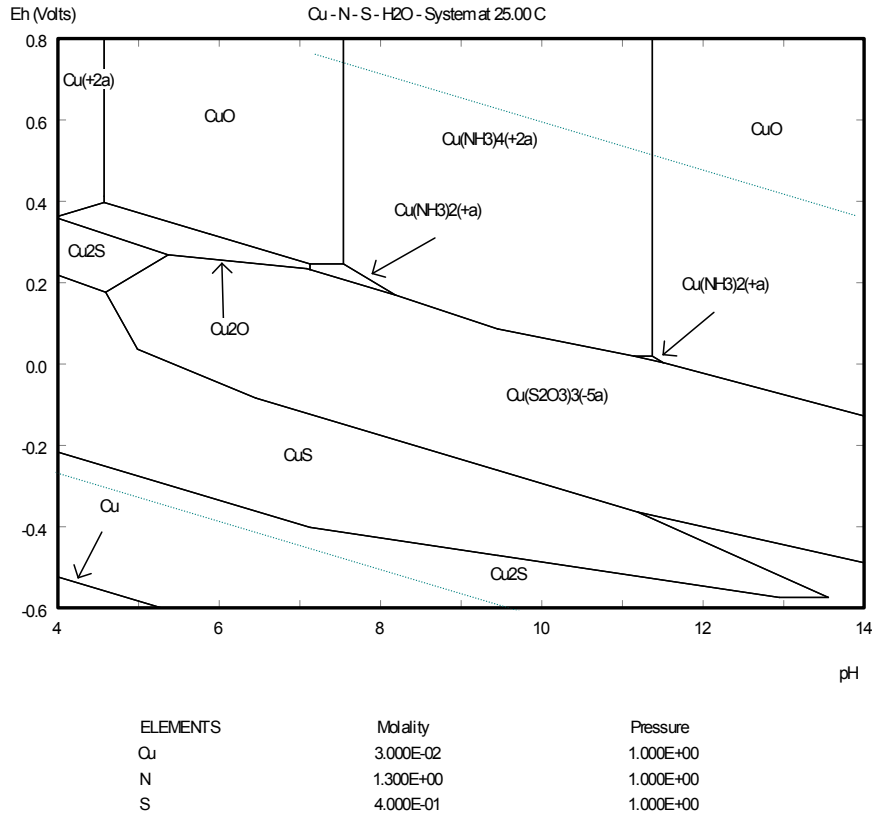
Due to reasons of kinetics, gold dissolution using dissolved oxygen as an oxidant is slow in thiosulfate solutions. Copper forms a tetra ammine complex with ammonia and this complex is a better oxidant of gold.



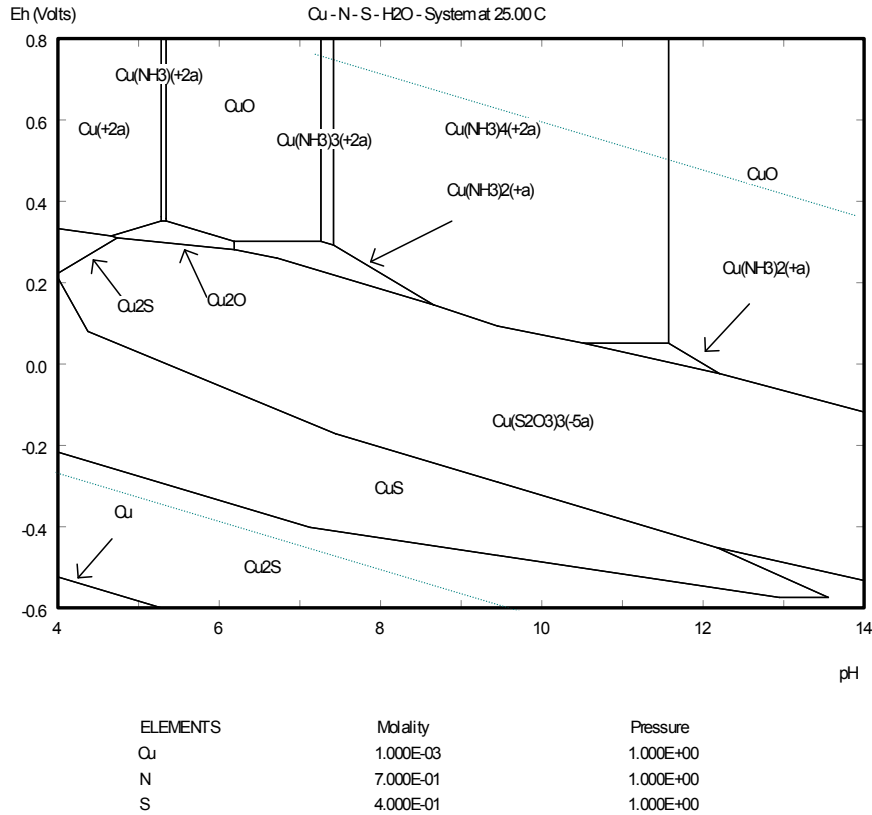
**Figure 3-3: Eh-pH diagram of Cu-S-N-H<sub>2</sub>O system (I)**

Figure 3-3 is an Eh-pH diagram of the Cu-S-N-H<sub>2</sub>O system at 1 mM copper, 0.9 M ammonia and 0.20 M ATS.  $\text{Cu}(\text{NH}_3)_4^{2-}$  remains stable between pH 7.3 and 11.3. Outside of this range copper can be precipitated as CuO. The reduction potential of  $\text{Cu}(\text{NH}_3)_4^{2-}$  is from about 0.3 to 0.1 V depending on the pH value. The reduction product of cupric ammonia is cuprous thiosulfate complex. A cuprous ammonia complex may appear as an intermediate product. Cuprous sulfides appear at -0.4 V and below (see Equations 3-16 to 3-21).





**Figure 3-4: Eh-pH diagram of Cu-S-N-H<sub>2</sub>O system (II)**



**Figure 3-5: Eh-pH diagram of Cu-S-N-H<sub>2</sub>O system (III)**

To demonstrate the effects of copper and ammonia concentrations on the stability of copper ammine, the same Cu-S-N-H<sub>2</sub>O system was plotted under altered conditions. Figure 3-4 is the Cu-S-N-H<sub>2</sub>O system with higher copper concentration (i.e., 30 mM instead of 1 mM). Figure 3-5 is the Cu-S-N-H<sub>2</sub>O system with lower ammonia concentration (0.3 M instead of 0.9 M). Both changes result in a narrower window of  $\text{Cu}(\text{NH}_3)_4^{2-}$ , which diminishes the efficiency of the thiosulfate leaching system.

### 3.1.2.3. Sulfur species in thiosulfate leaching system

The instability of S-O anions, such as thiosulfate, is another reason that makes the thiosulfate leaching system very complicated. It is necessary to study the sulphur species to reach a better understanding of the leaching results.

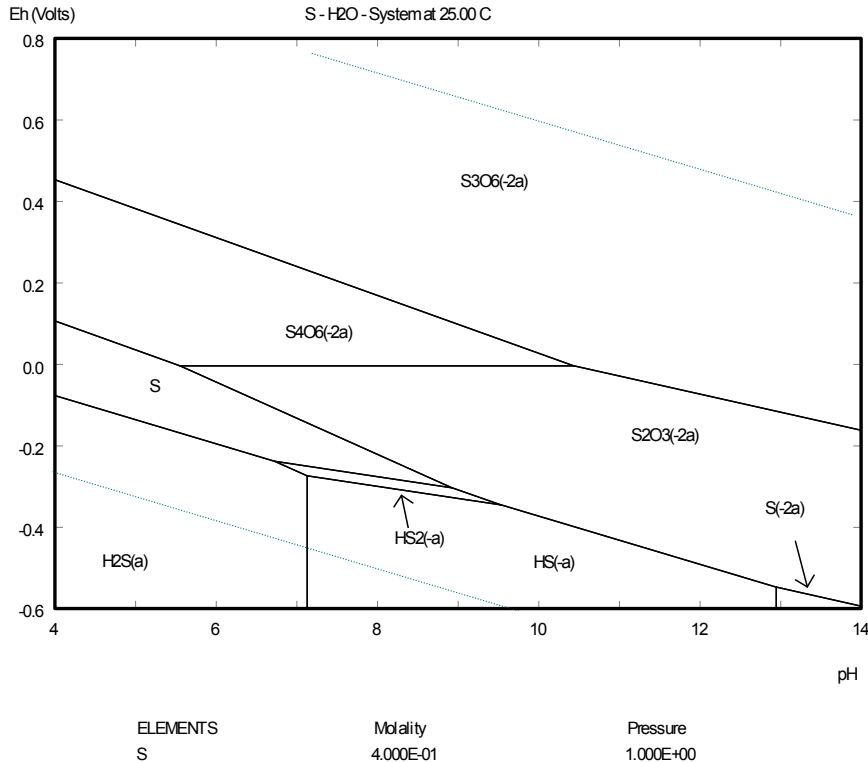


Figure 3-6: Eh-pH diagram of S-H<sub>2</sub>O system (I)

Figure 3-6 is an Eh-pH diagram of S-H<sub>2</sub>O system containing 0.2 M thiosulfate. Under typical leaching conditions, the thiosulfate may be oxidized to either tetrathionate or trithionate depending on the pH conditions (see Equations 3-22 to 3-24). The oxidation potential of thiosulfate to tetrathionate is about 0.0 V. At a pH higher than 10.5, the thiosulfate is oxidized to trithionate and the oxidation potentials required become lower.

When the pH is lower than 9, the formation of elemental sulphur is possible. Note that the sulphur may cover the surface of the gold and passivate the leaching reactions. Such a low pH should be avoided.

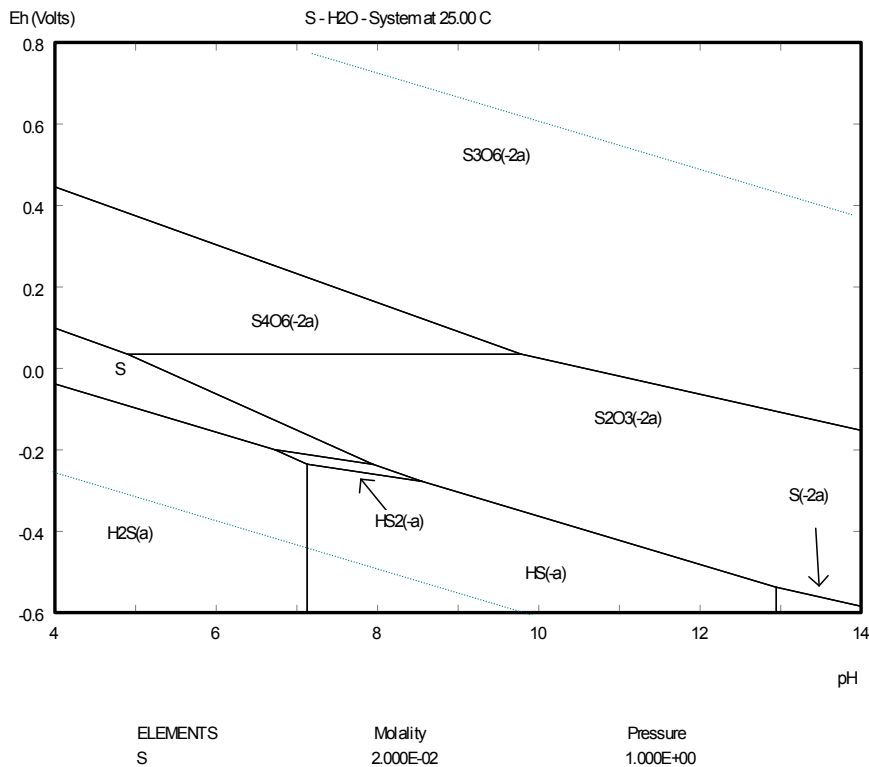
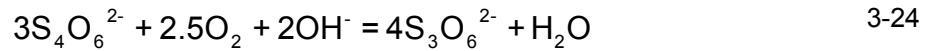
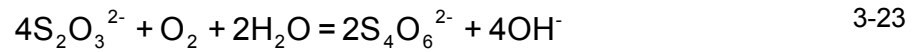
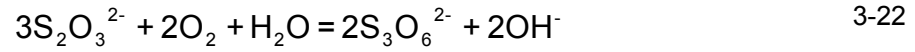


Figure 3-7: Eh-pH diagram of S-H<sub>2</sub>O system (II)

Figure 3-7 is an Eh-pH diagram of S-H<sub>2</sub>O system containing 0.01 M thiosulfate. Due to the low thiosulfate concentration, the oxidation potential of thiosulfate shifts upwards to about 0.4 V. Higher oxidation potential of thiosulfate indicates that the decomposition of this reagent is more difficult if the slurry potential remains the same. With the decrease of thiosulfate concentration, the critical pH for the generation of elemental sulphur shifts from about 9 to about 8, leaving a wide range of pH that is suitable for the leaching of gold.

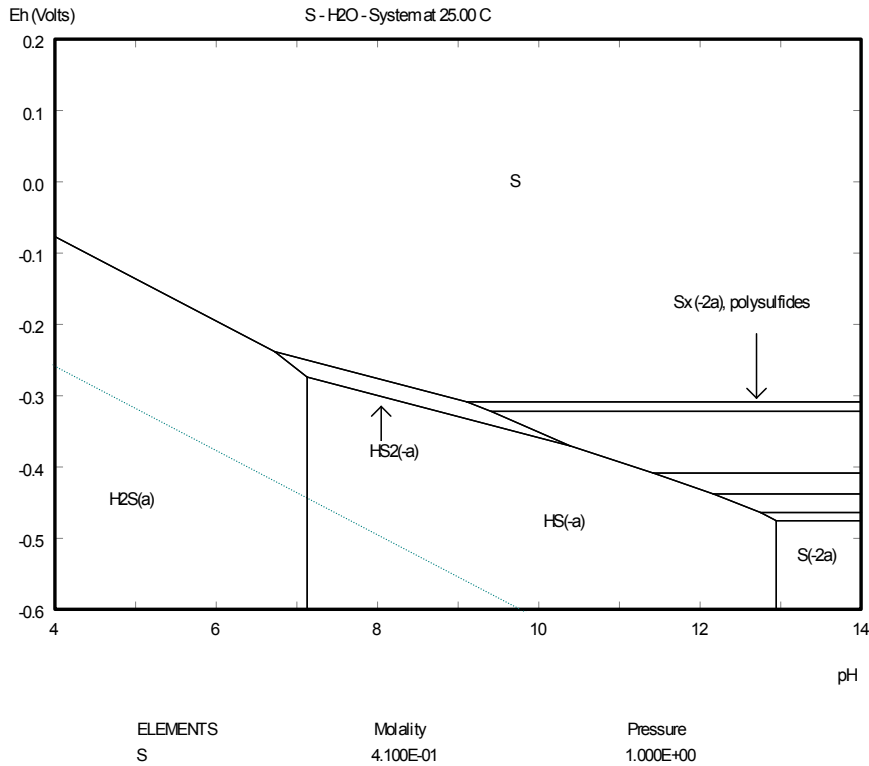
### **3.1.3. Impacts of associated minerals on thiosulfate leaching of gold**

In this section, possible interference of associated minerals on gold leaching performance will be analyzed with thermodynamic methods. The detrimental effects of the sulfide anion, the ferrous ion, dissolved copper, arsenic species and dissolvable lead species will be discussed respectively.

#### **3.1.3.1. Sulfide anion**

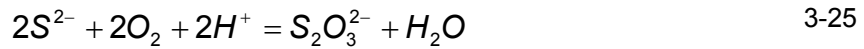
The sulfide anion is the most important component of all sulfide minerals. It may react with various aqueous species. These reactions occur either on the surface of sulfide minerals or in the bulk solution. The major forms of sulfide under alkaline conditions are HS<sup>-</sup> and S<sup>2-</sup>. For convenience, both species will be referred to as sulfide (anion) hereafter. The chemistry of the sulfide anion can be very complicated in aqueous solution. However, there are mainly two types of reactions of interest in the thiosulfate leaching system: the oxidation of sulfide, and the formation of metal sulfide precipitates.

## 1) Oxidation of the sulfide anion

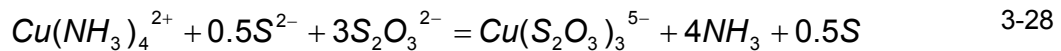
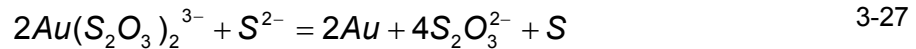


**Figure 3-8: Eh-pH diagram of S-H<sub>2</sub>O system (II)**

Under typical leaching conditions, the sulfide anion in the bulk solution phase tends to form thiosulfate by oxidation (Figure 3-6 and Figure 3-7). The extraction of metal ions from sulfide minerals leaves more reactive sulfide anion on the leached surface. As a result, the sulfur ratio increases on the leached surface. Polysulfides and elemental sulfur are produced on the surface of some sulfide minerals following Equation 3-26. Under these circumstances, the Eh-pH diagram of S-H<sub>2</sub>O system is shown in Figure 3-8.



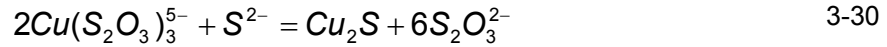
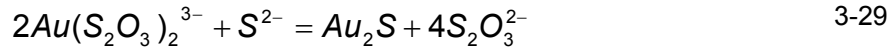
The oxidation potential of the sulfide ion is very negative, i.e. -0.2 to -0.6 V. The oxidants that react with the sulfide include at least dissolved oxygen, the cupric ion and its ammonia complexes (Equation 3-28), the aurous ion and its thiosulfate complexes (Equation 3-27), and meta-stable polythionates (Equation 2-15).



## 2) Formation of sulfide precipitate

Under a reductive condition, the sulfide ion is also capable of complexing many different metal ions, e.g.,  $Au^+$  and  $Cu^+$ , and producing metal sulfide precipitates (Equation 3-29 and 3-30). Copper species can be reduced and precipitated as various copper sulfides, i.e.,  $Cu_5FeS_4$ ,  $CuFeS_2$ , or  $Cu_2S$ , depending on the abundance of iron on the leached surface. Most of these sulfide precipitates will be extracted with ammonia back into the solution phase, especially when the

solution condition becomes oxidative. Therefore, to avoid the loss of gold and copper, it is necessary to have the surface of the mineral oxidized.



### 3) Summary

The sulfide anion either in the bulk solution or on the surface of minerals plays important roles in the thiosulfate leaching. It is detrimental to gold dissolution due to its highly reductive properties and its capability of forming metal sulfide precipitates. On the other hand, the sulfide anion may suppress the oxidation of thiosulfate due to consumption of the oxidant by the sulfide. The production of thiosulfate from the sulfide anion may also reduce the overall consumption of thiosulfate. However, it was found that this is not necessarily true for all sulfide minerals. The reasons will be discussed in a latter section (3.1.3.6).

#### **3.1.3.2. Iron species**

Iron-containing sulfides are the most common sulfidic minerals associated with gold ores. Therefore, the behaviour of iron becomes important. The reactive surface of many sulfides contains both the ferrous ion and the ferric ion, with the ferrous ion being the majority. The ferric ion on the surface of sulfide minerals,

such as chalcopyrite, can be converted to the ferrous ion under a reductive condition.

The ferrous ion is slightly dissolvable on the surface of most sulfides and the presence of this ion may significantly affect the reaction chemistry. The ferrous ion is both a reducing agent and a co-precipitating agent that may reduce and precipitate metal ions. Meanwhile, the final product, FeOOH, is known for its capacity of passivating the surface of sulfide minerals.

### 1) Oxidation and precipitation of ferrous ion

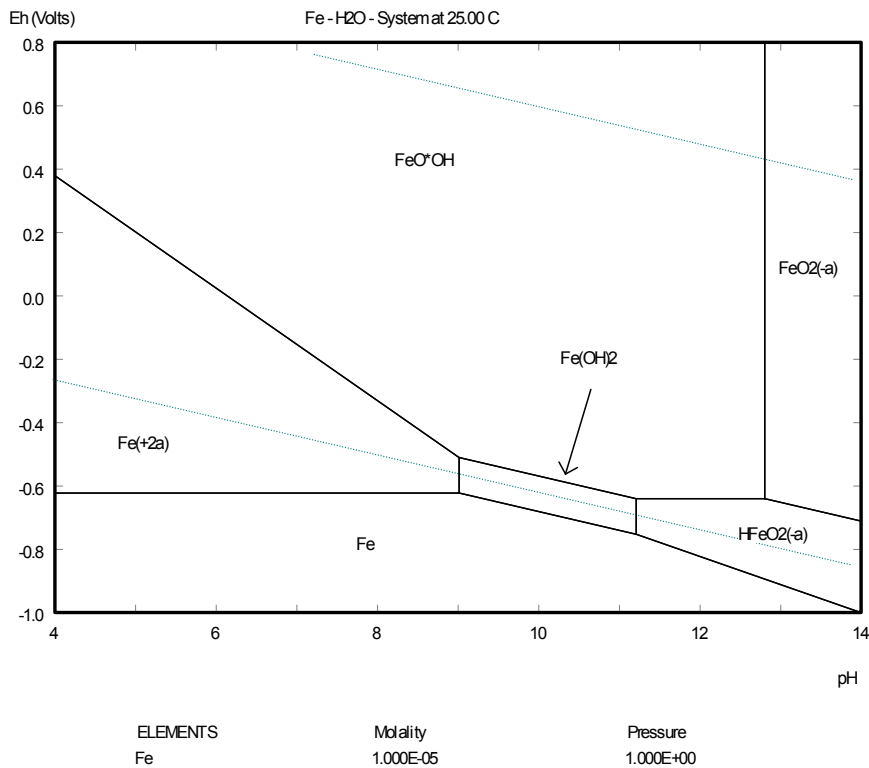
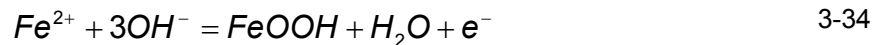
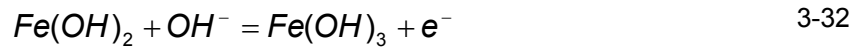
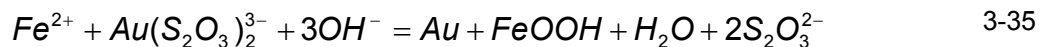


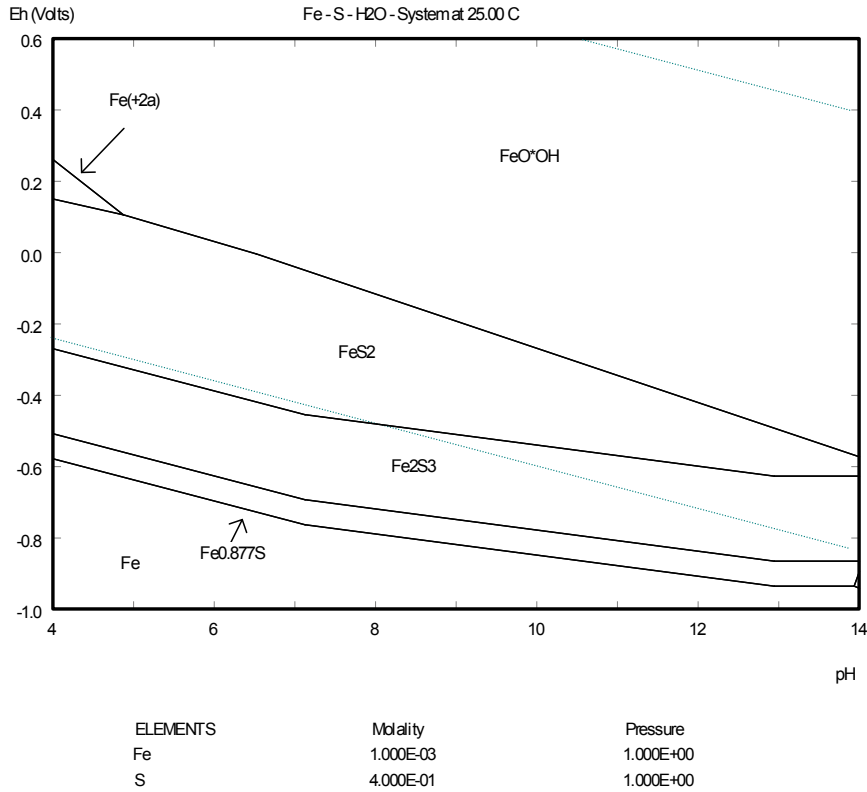
Figure 3-9: Eh-pH diagram of Fe-H<sub>2</sub>O system: oxidation and precipitation of ferrous ion

The ferrous ion, either in bulk solution or on the surface of sulfides can play important roles in several chemical and physical reactions. As shown in Figure 3-9, the ferrous ion is not stable at a pH above 9.5. It tends to be oxidized and precipitated directly or indirectly as FeOOH (Equation 3-31 to 3-34).



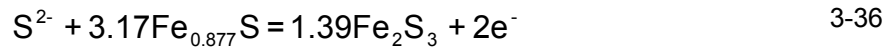
The oxidation potential of the ferrous ion is found to be more negative than -0.5 V. Considering that the oxidation of gold and stabilization of aurous thiosulfate requires a redox potential that is higher than -0.1 V, the ferrous ion is obviously capable of precipitating gold or preventing gold from dissolution (Equation 3-35).

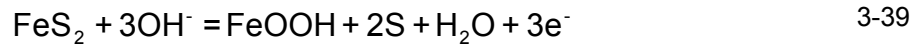
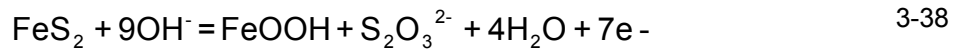




**Figure 3-10: Eh-pH diagram of Fe-S-H<sub>2</sub>O system: iron sulfide minerals**

Figure 3-10 is an Eh-pH diagram of two iron sulfides minerals (pyrite and pyrrhotite) during the thiosulfate leaching. On the surface of the pyrite or the pyrrhotite, the oxidation follows the order of pyrrhotite → Fe<sub>2</sub>S<sub>3</sub> → pyrite → FeOOH (see Equation 3-36 to 3-39). Pyrrhotite is more reductive than pyrite.

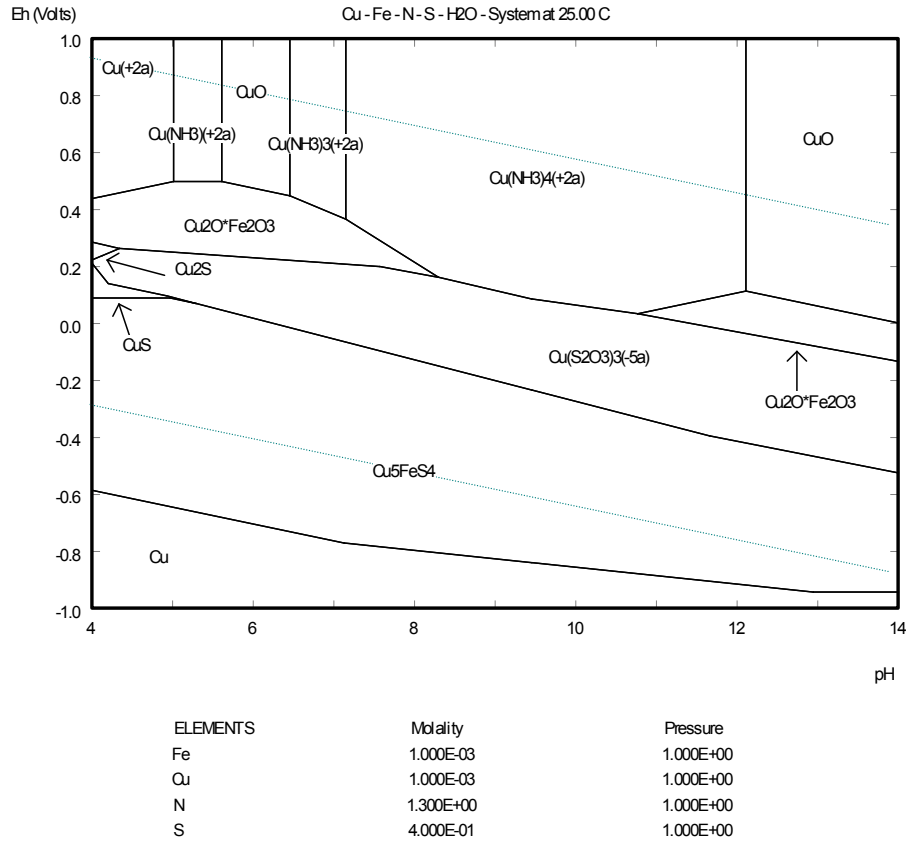




## 2) Co-precipitation of gold with iron hydroxides

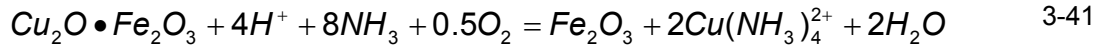
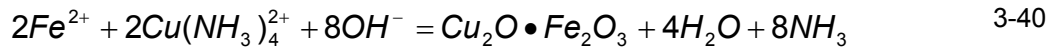
Reduction and precipitation of gold can be accelerated due to the presence of  $\text{Fe}(\text{OH})_2$  and  $\text{Fe}(\text{OH})_3$  as the intermediate products of ferrous oxidation and precipitation. These hydroxides tend to co-precipitate or adsorb other metals, and therefore, may result in an instant loss of dissolved gold.

## 3) Co-precipitation of copper with iron hydroxides



**Figure 3-11: Eh-pH diagram of Fe-Cu-N-S-H<sub>2</sub>O system: co-precipitation of copper with iron hydroxides**

Other than dissolved oxygen and aurous complexes, the cupric ion and its complexes can also be reduced by ferrous ion. As shown in Figure 3-11, cupric ammine can be reduced to  $\text{Cu}_2\text{O}\cdot\text{Fe}_2\text{O}_3$  in the presence of ferrous ion on the surface of sulfides. (Equation 3-40). The precipitation of copper on the reductive surface of the sulfide is either a co-precipitation or an adsorption procedure. The precipitation is instant and the copper species in the precipitate can be oxidized and leached back to solution as cupric ammine under an oxidative condition (Equation 3-41).



The co-precipitation or chemical adsorption of copper on the sulfide surface is beneficial to gold extraction due to the competition of copper with gold in the iron hydroxide co-precipitation or chemical adsorption procedures. On the other hand, the presence of the cupric ion catalyzes the oxidization of sulfide surface, which is also beneficial to the gold leaching performance.

#### 4) Surface passivation by iron hydroxides (oxides)

The formation of FeOOH due to the oxidation and precipitation of the ferrous ion may produce an inert film on the surface of some iron-containing sulfides. However, to enable the surface passivation, there must be sufficient iron in the lattice of the sulfide minerals. For pyrite (FeS<sub>2</sub>), pyrrhotite (FeS), chalcopyrite (CuFeS<sub>2</sub>) and arsenopyrite (FeAsS), the iron ratios are on the high side (1:2 or 1:3). Therefore, passivation is expected when FeOOH covers the surface of these sulfides. However, for some other iron-containing sulfide minerals, e.g., bornite (Cu<sub>5</sub>FeS<sub>4</sub>), the iron ratio is much lower (1:10). Passivation by FeOOH is, therefore, not favourable.

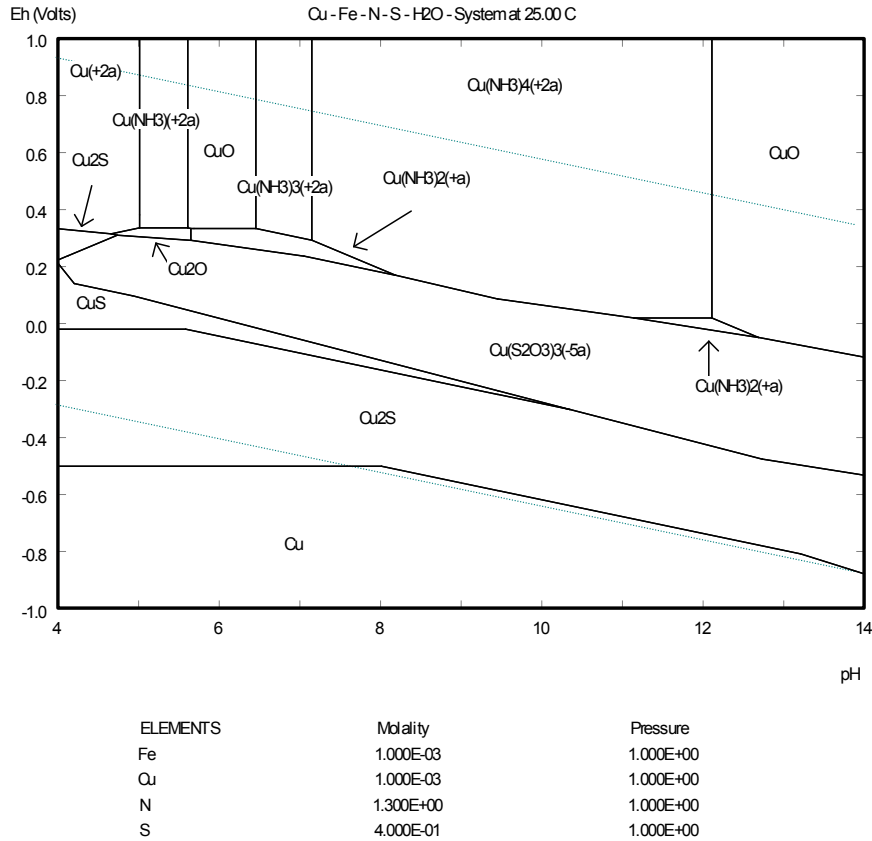
#### 5) Summary

The above analysis of the ferrous ion would indicate that the presence of the ferrous ion on the leached surface of sulfides would reduce the gold extraction due to its reductive property and its capacity of co-precipitating or adsorbing other metal ions out of solution. The cupric ion could enhance the oxidation of ferrous anion and suppress gold co-precipitation with ferrous ion. The oxidation and precipitation of ferrous ion ultimately produce FeOOH, which may passivate the reductive surface of iron-containing sulfides that have high iron ratio in the formula.

### **3.1.3.3. Copper species**

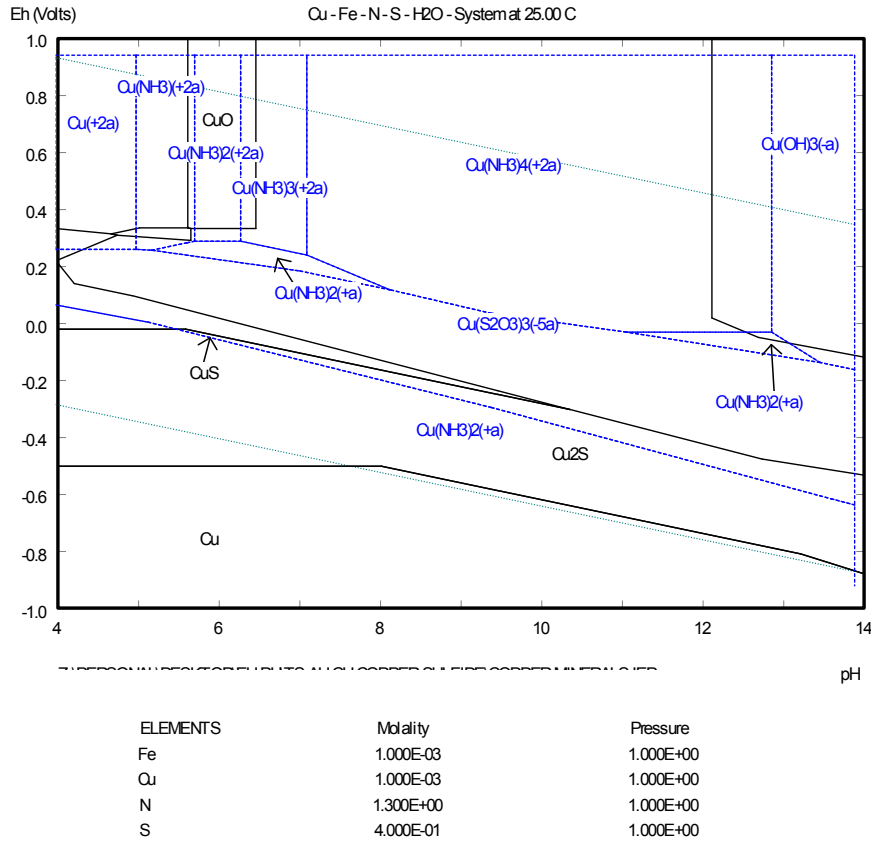
On the surface of copper-bearing sulfides, there are at least two reductive species: the cuprous ion and sulfide anion. The oxidation of the sulfide anion has been discussed in a previous section (3.1.3.1). The behaviour of the cuprous ion, therefore, is another interesting topic that requires attention. Due to the presence of copper ligands, copper minerals tend to dissolve in ammonia-thiosulfate solution. The dissolved copper may have both positive and negative effects on the leaching of gold. The leached surfaces become more reductive and will react with many aqueous species.

#### 1) Dissolution of copper



**Figure 3-12: Eh-pH diagram of Cu-Fe-N-S-H<sub>2</sub>O system --- chalcocite (I)**

The major copper sulfide mineral that has no iron in the lattice is chalcocite (Cu<sub>2</sub>S). Figure 3-12 is an Eh-pH diagram for demonstrating the dissolution of copper sulphide in ammonia-thiosulfate solution. The dissolution and oxidation of copper species follows the order of Cu<sub>2</sub>S → CuS → Cu(S<sub>2</sub>O<sub>3</sub>)<sub>3</sub><sup>5-</sup> → Cu(NH<sub>3</sub>)<sub>4</sub><sup>2+</sup>. The oxidation potentials of Cu<sub>2</sub>S and CuS are about -0.4 and -0.2 V respectively and are sufficiently negative to reduce dissolved oxygen, cupric ammine and aurous thiosulfate.



**Figure 3-13: Eh-pH diagram of Cu-Fe-N-S-H<sub>2</sub>O system --- chalcocite (II)**

To understand the reaction sequence of copper dissolution from chalcocite, the Eh-pH diagrams of the predominant ions were plotted (Figure 4-13). The thin lines in this diagram are the boundaries of the predominant ions. Under a negative potential, e.g., -0.4 V, the predominant ionic species is  $\text{Cu}(\text{NH}_3)_2^+$ . This indicates that in the first step of the copper dissolution procedure, i.e.,  $\text{Cu}_2\text{S} \rightarrow \text{CuS}$ , it is ammonia that works as the major lixiviant of copper. Therefore, the actual steps of copper dissolution from chalcocite shall be:  $\text{Cu}_2\text{S} \rightarrow \text{CuS} + \text{Cu}(\text{NH}_3)_2^+ \rightarrow \text{Cu}(\text{S}_2\text{O}_3)_3^{5-} \rightarrow \text{Cu}(\text{NH}_3)_4^{2+}$  (see Equations 3-42 to 3-44).

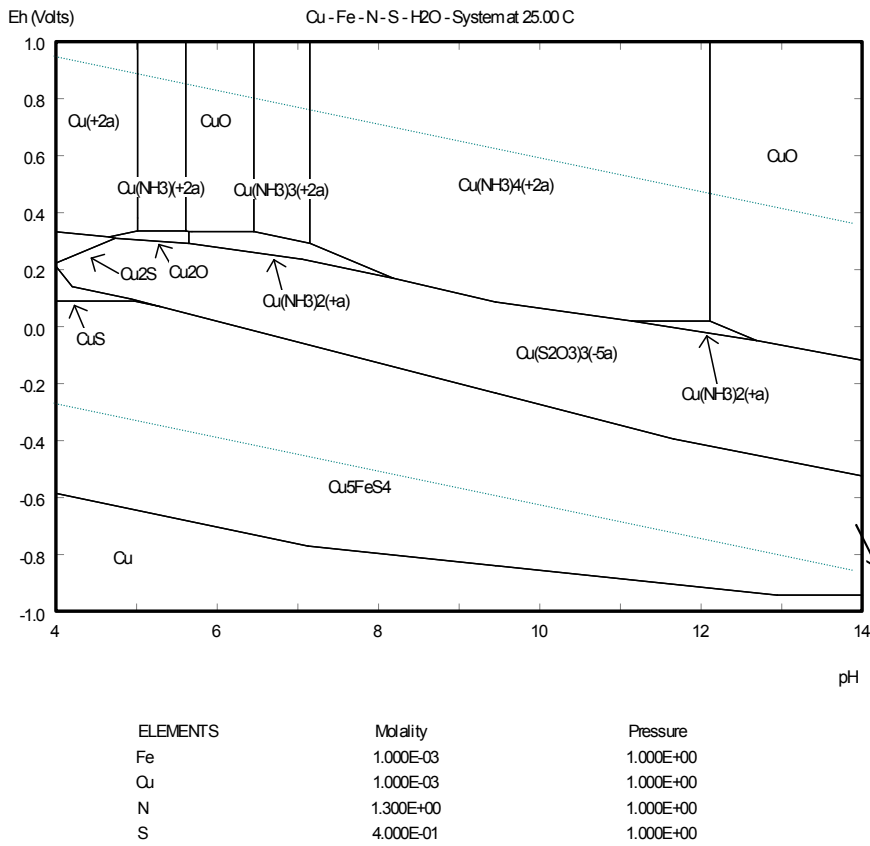
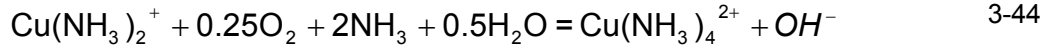
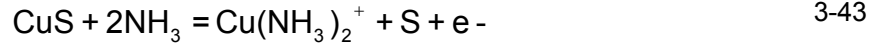
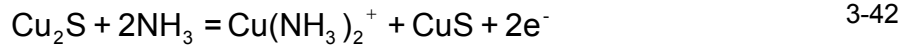
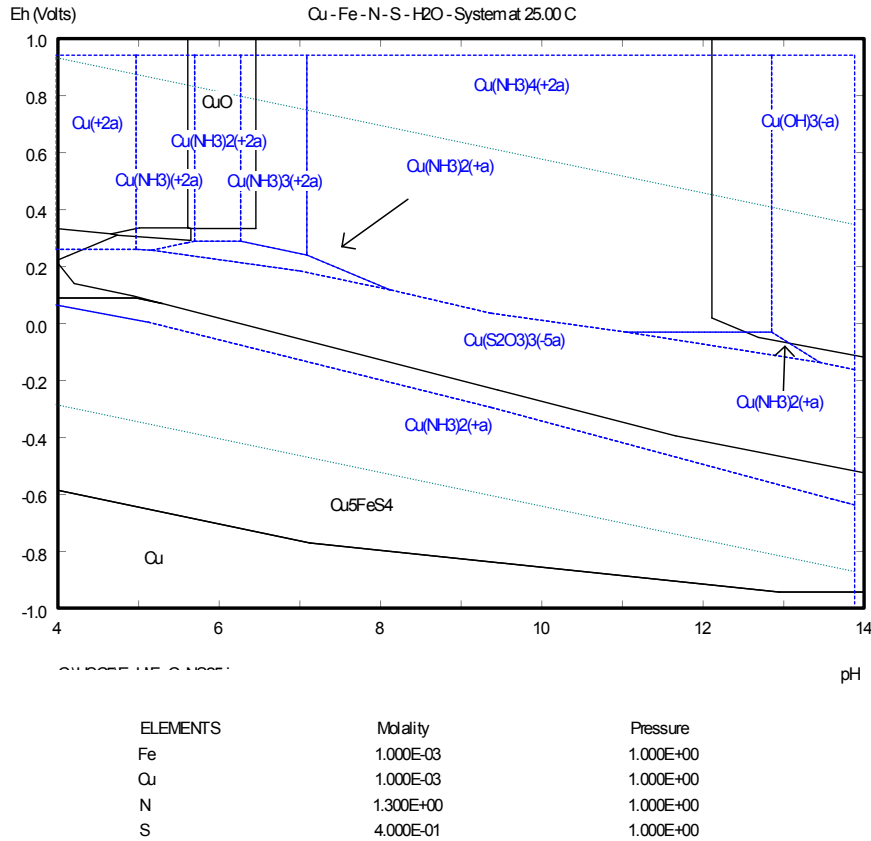
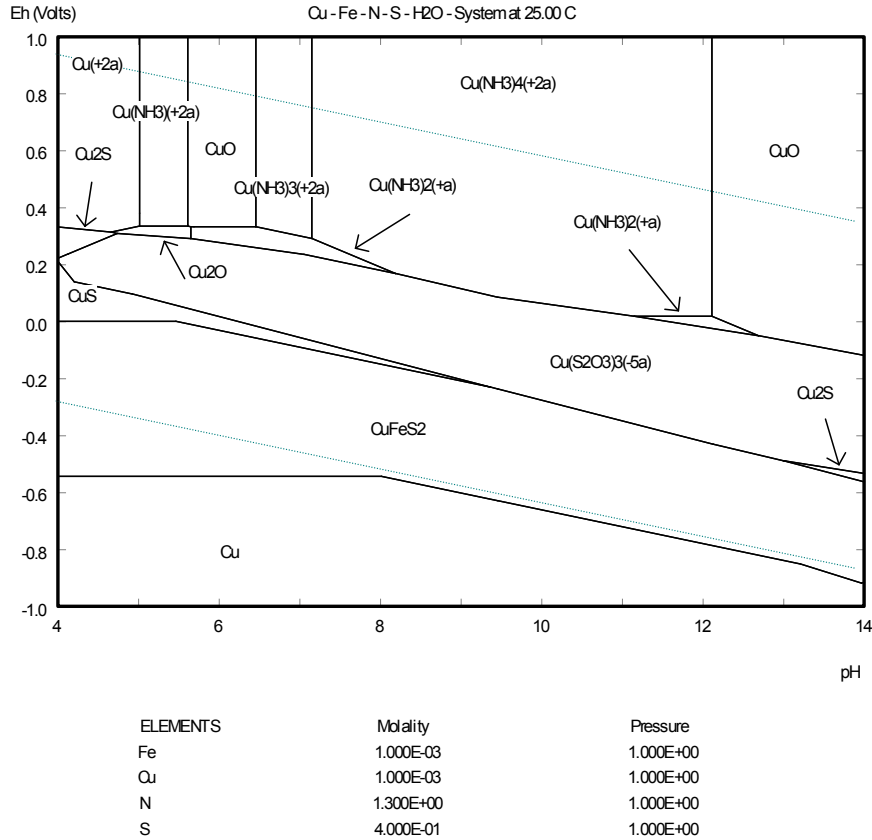
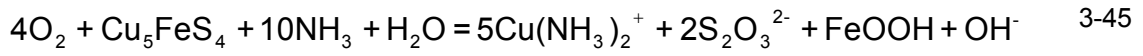


Figure 3-14: Eh-pH diagram of Cu-Fe-N-S-H<sub>2</sub>O system --- bornite (I)

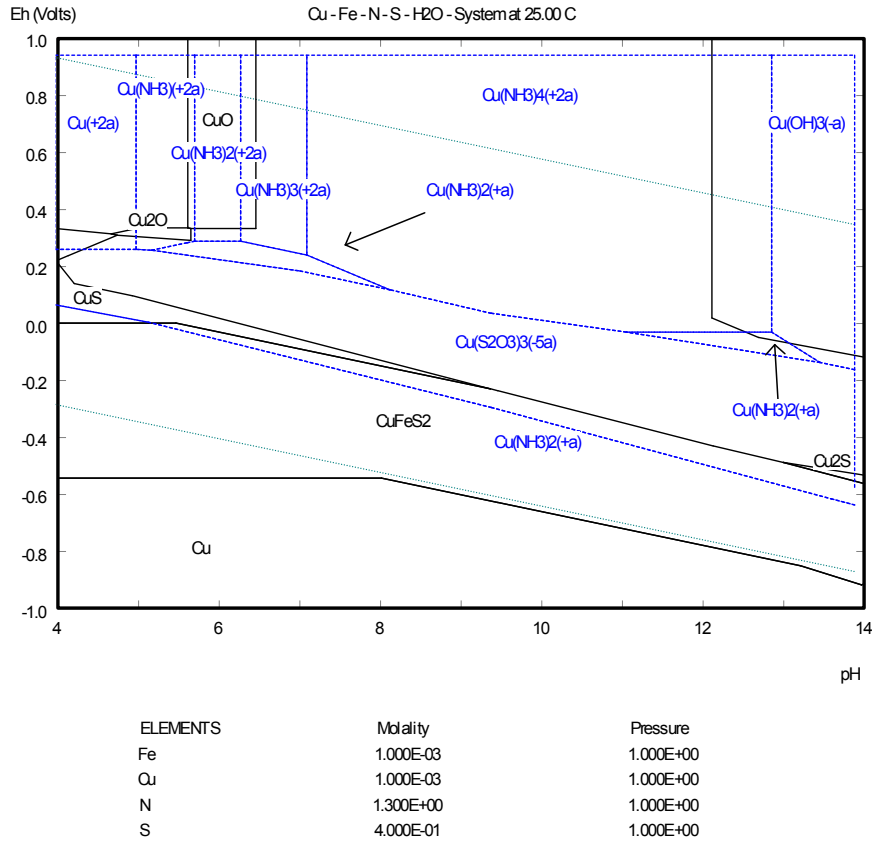


**Figure 3-15: Eh-pH diagram of Cu-Fe-N-S-H<sub>2</sub>O system --- bornite (II)**

Chalcopyrite and bornite are common copper sulfide minerals that have iron in their lattices. Figure 3-14 is an Eh-pH diagram for bornite in the thiosulfate leaching system. Figure 3-15 displays the predominant ions in the same system.  $\text{Cu}(\text{NH}_3)_2^{2+}$  appears to be the leaching product within a negative potential range (see Equation 3-45), i.e., -0.4 to -0.3 V at pH 10.0-11.0. Therefore, the actual steps of copper extraction and oxidation shall have the order of  $\text{Cu}_5\text{FeS}_4 \rightarrow \text{Cu}(\text{NH}_3)_4^{2+} \rightarrow \text{Cu}(\text{S}_2\text{O}_3)_3^{5-} \rightarrow \text{Cu}(\text{NH}_3)_2^{2+}$ . The formation of CuS and  $\text{Cu}_2\text{S}$  during the leaching of bornite is also possible due to the higher thermodynamic stability of these two species over bornite.

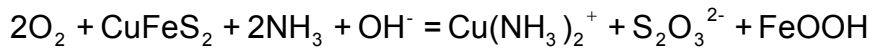


**Figure 3-16: Eh-pH diagram of chalcopyrite (I)**



**Figure 3-17: Eh-pH diagram of chalcopyrite (II)**  
**(Cu: 1 mM, S: 0.40 M, N: 0.9 M, Fe: 1 mM)**

Figure 3-16 is an Eh-pH diagram of chalcopyrite in thiosulfate leaching solution. Due to the high iron ratio in the lattice, the iron interference must be considered. As a result, in this diagram,  $\text{Cu}_2\text{O}\cdot\text{Fe}_2\text{O}_3$  cannot be ignored as compared to the Eh-pH diagram of bornite. Figure 3-17 displays the predominant ions of the same system. The leaching and oxidation follow the order of  $\text{CuFeS}_2 \rightarrow \text{Cu}(\text{NH}_3)_2^+ \rightarrow \text{Cu}(\text{S}_2\text{O}_3)_3^{5-} \rightarrow \text{Cu}_2\text{O}\cdot\text{Fe}_2\text{O}_3 \rightarrow \text{Cu}(\text{NH}_3)_4^{2+}$ . Ammonia appears to be the major lixiviant of copper in the leaching of chalcopyrite (see Equation 3-46).



3-46

## 2) Oxidation of the sulfide anion in copper sulfides

The behaviour of the sulfide anion has been discussed in section 3.1.3.1. In the leaching of copper sulfide minerals, due to the highly dissolvable nature of copper, the leached surface of the sulfide contains more reactive sulfide anion. Consequently, more thiosulfate and/or elemental sulfur will be produced on the surface of these minerals. In addition, the minerals' surface will be more reductive than iron sulfide minerals.

## 3) Passivation of leached surface

In the presence of iron, the formation of FeOOH may result in passivation of the reactive surfaces. On the chalcopyrite surface, this passivation shall be very effective due to the high ratio of iron in the lattice. On the bornite surface, however, the passivation may be less effective due to the low ratio of iron.

For the iron free copper sulfide, i.e., chalcocite, the surface of the leached mineral will not be passivated by FeOOH. However, the formation of elemental sulfur may slow down the leaching rate of copper.

## 4) Excessive copper in solution

The copper ion leached from copper minerals, on one hand, can provide more necessary oxidant for the dissolution of gold. On the other hand, the

concentration of copper will ultimately become excessive. As discussed in previous sections, copper ammine is an effective oxidant and an oxidation catalyst for gold; thiosulfate, sulfide anion, ferrous ion and many other reductive species exist with this system. Therefore, excessive copper may accelerate the oxidations of both the thiosulfate and the sulfide surfaces.

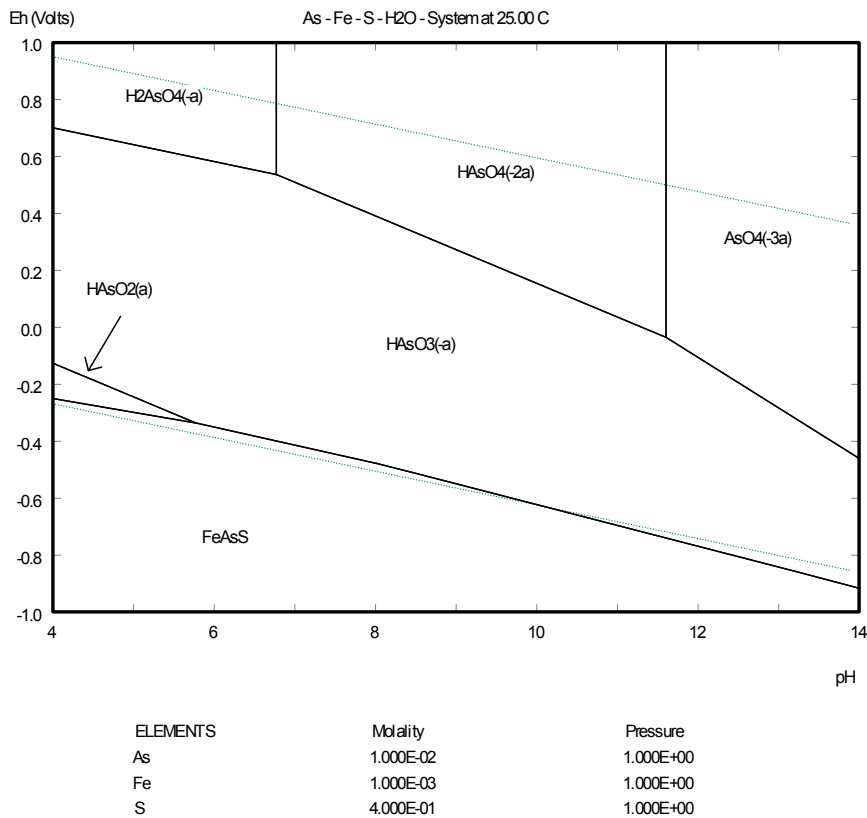
## 5) Summary

The presence of copper sulfide minerals may significantly change the reaction chemistry of the thiosulfate leaching system due to their reductive properties and more importantly their highly dissolvable natures. Among the studied minerals, chalcopyrite is much less dissolvable than chalcocite and bornite. Ammonia is the major ligand that dissolves copper from these sulfide minerals. The dissolved copper may accelerate the oxidation of thiosulfate and increase the consumption of thiosulfate reagent. Its impacts on gold extractions, however, are very complicated and will be further analyzed. On one hand, the dissolved copper accelerates the oxidation of gold and sulfide surfaces. But it also reduced the degree of gold co-precipitation with iron. On the other hand, higher copper content is responsible for increased decomposition rate of thiosulfate, which will also consume more oxidant. With the increase of copper concentration in solution, the re-precipitation of copper sulfide on gold surface becomes more significant, which may ultimately affect the extraction of gold.

### 3.1.3.4. Arsenic species

The presence of the sulfide anion, and the ferrous ion on the surface of arsenopyrite may result in a reduction of dissolved gold and consume oxidants such as cupric ammine and dissolved oxygen. The oxidation and precipitation of arsenic species are also important and will require more attention.

#### 1) Oxidation of arsenopyrite



**Figure 3-18: Eh-ph diagram of arsenopyrite (I)**

As shown in Figure 3-18, FeAsS can be leached and oxidized to form  $\text{As}^0$ , which is further oxidized to  $\text{HAsO}_3^-$  and then  $\text{HAsO}_4^{2-}$  (see Equations 3-47 to 3-49). The oxidation potential of FeAsS was sufficiently negative to result in a reduction of

aurous thiosulfate. Dissolved oxygen and cupric ammine can also be consumed by these two arsenic species.

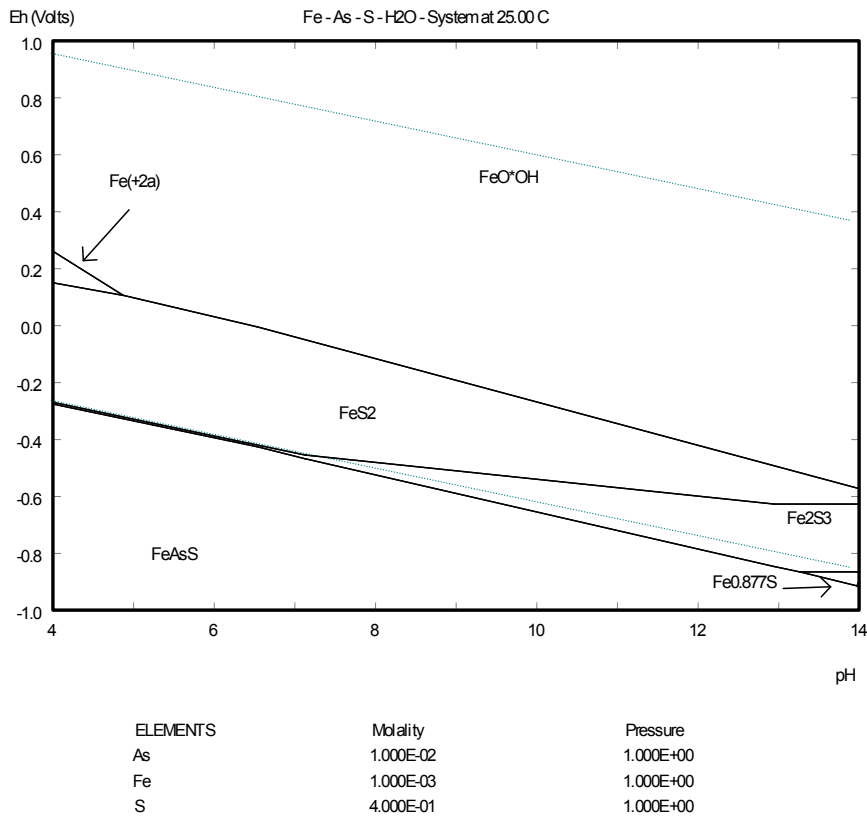
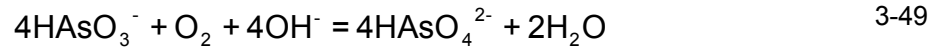
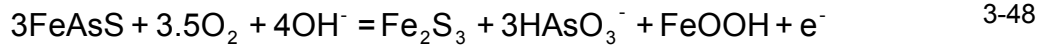
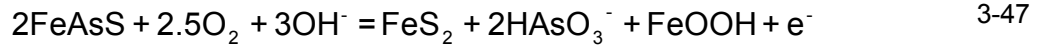
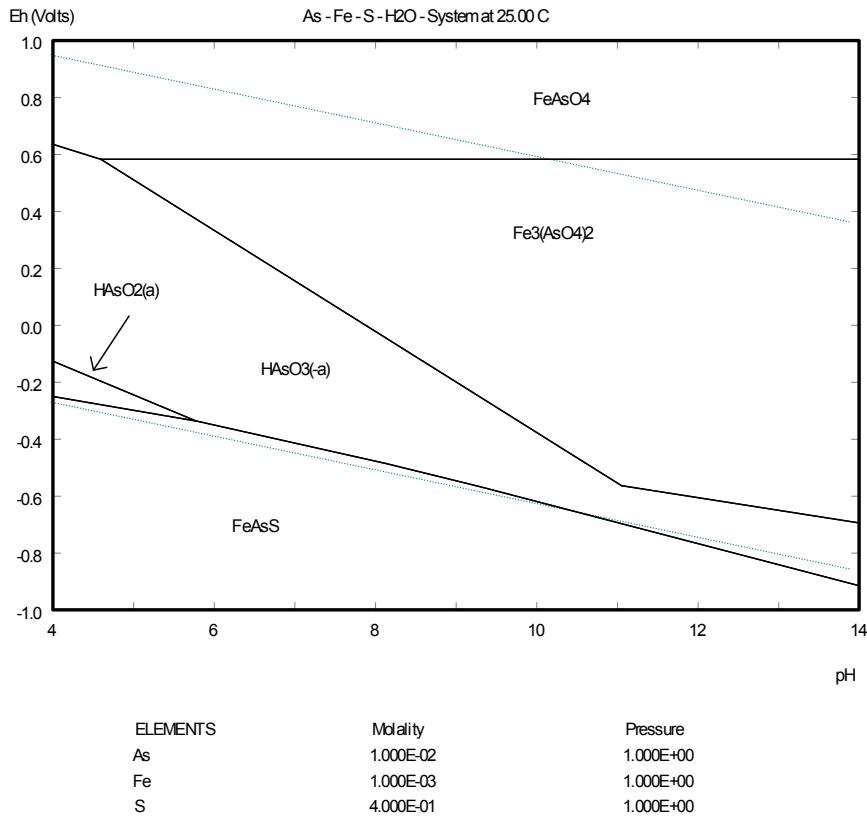


Figure 3-19: Eh-ph diagram of arsenopyrite (II)

Figure 3-19 is an Eh-pH diagram of the same system with the focus on the chemistry of the iron in arsenopyrite. The leached surface of arsenopyrite produces pyrrhotite and pyrite as intermediate products and the iron portion of arsenopyrite will be ultimately oxidized to FeOOH.

## 2) Formation of metal arsenate



**Figure 3-20: Eh-pH diagram arsenopyrite (III)**

It was claimed that iron arsenate was found on the leached surface of arsenopyrite [Feng et al., 2001]. Figure 3-20 depicts the formation of iron arsenate on the surface of arsenopyrite. FeOOH was removed in the calculation of this diagram in order to show the formation of iron arsenate. This is possibly

due to the preferential reaction of dissolved arsenates with ferrous ion under an oxidative condition. In this case, iron is precipitated as  $\text{Fe}_3(\text{AsO}_4)_2$  instead of  $\text{FeOOH}$  (Equation 3-50). Under a more oxidative condition, ferric arsenate,  $\text{FeAsO}_4$ , is predominant (Equation 3-51).

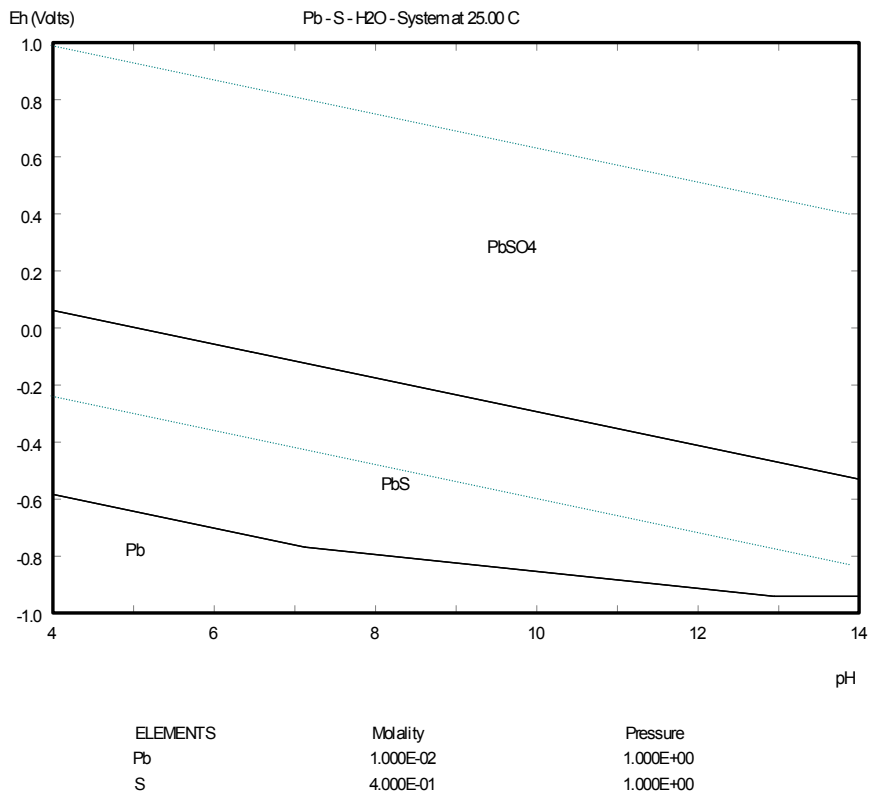


### 3) Passivation of arsenopyrite surface

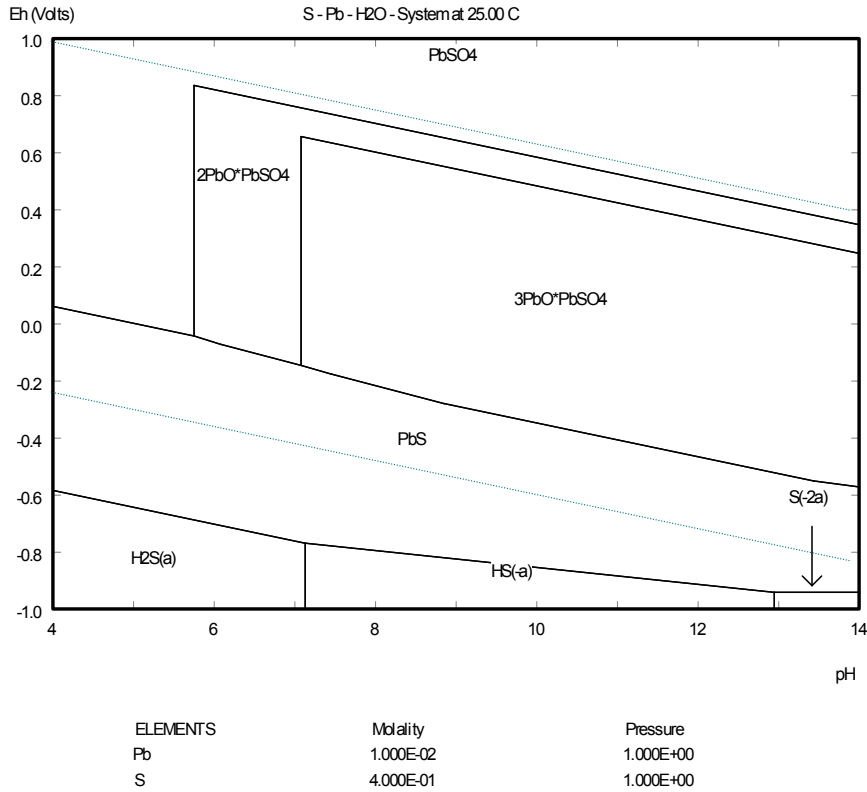
As stated in the above analysis, the iron on the surface of arsenopyrite may produce either  $\text{FeOOH}$  or iron arsenates. Both species may form an inert layer on the leached surface of arsenopyrite. Passivation may occur due to this layer of precipitate.

#### **3.1.3.5. Lead species**

##### 1) Lead precipitate in the presence of sulfate anion



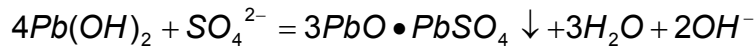
**Figure 3-21: Eh-pH diagram of Pb-S-H<sub>2</sub>O system in the presence of sulfate anion (I)**



**Figure 3-22: Eh-pH diagram of Pb-S-H<sub>2</sub>O system in the presence of sulfate anion (II)**

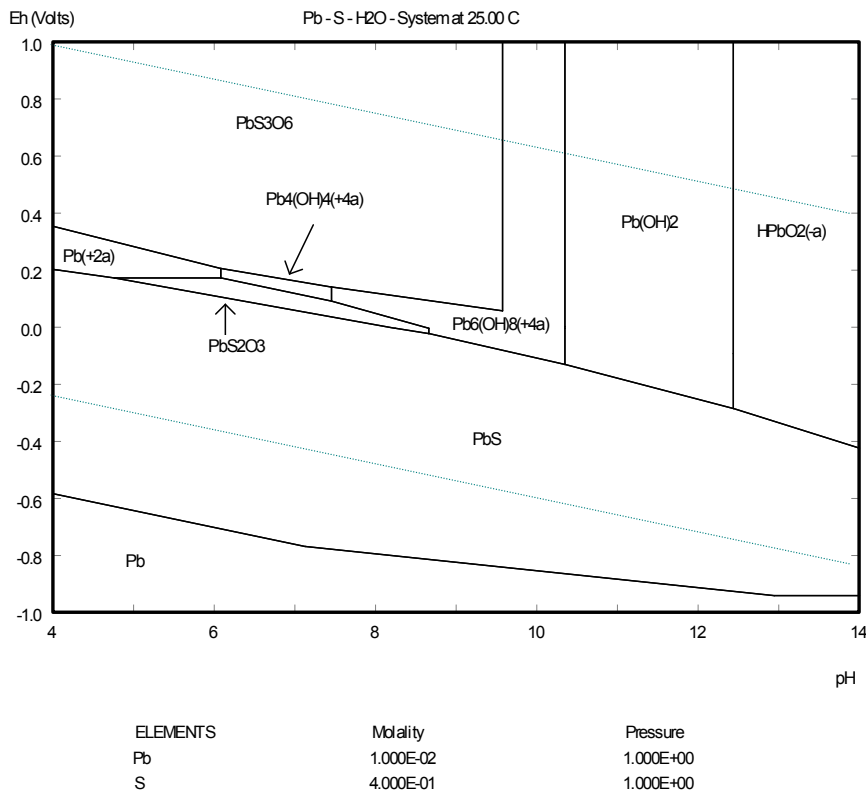
As shown in Figure 3-21, the Pb-S-N-H<sub>2</sub>O system has PbSO<sub>4</sub> as the only final oxidation product (Equation 3-52). However, the actual reaction system is more complicated. Figure 3-22 is an Eh-pH diagram for the same system. However, this diagram focuses on the sulfur species. It may be seen that before lead sulfate is produced, a combined solid product, 3PbO•PbSO<sub>4</sub>, could be formed (Equation 3-53). This product may be the major intermediate precipitation product.





3-53

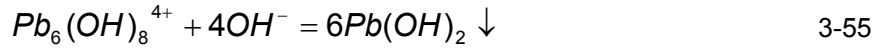
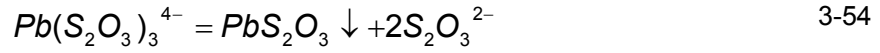
## 2) Lead precipitate in the absence of sulfate



**Figure 3-23: Eh-pH diagram of Pb-S-H<sub>2</sub>O system in the absence of sulfate anion (I)**

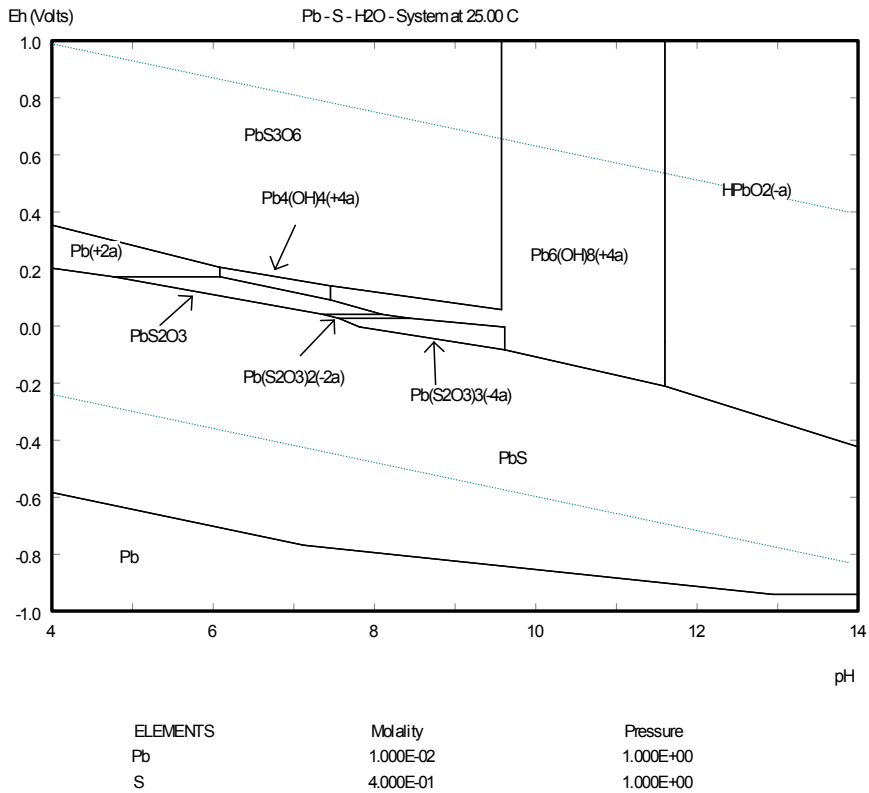
Figure 3-23 is an Eh-pH diagram of the Pb-S-H<sub>2</sub>O system with all sulfate species removed in the calculation. This is because, within the early stage of thiosulfate leaching, the sulfate concentration is minimal and the hydroxide as well as the thiosulfate concentrations are much higher. Lead hydroxide, Pb(OH)<sub>2</sub>, is one of the major precipitates of lead (Equation 3-55). Lead thiosulfate precipitate only occurs as a lead product during the early stage of leaching (Equation 3-54).

During a later stage of the leach, when more sulfate anion is available,  $3\text{PbO}\cdot\text{PbSO}_4$  and ultimately  $\text{PbSO}_4$  will appear.



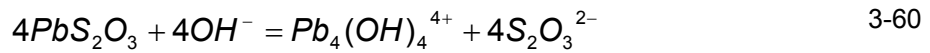
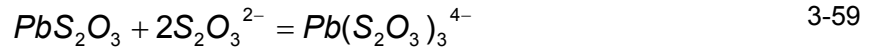
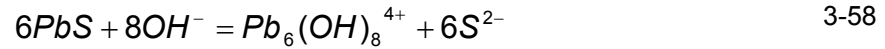
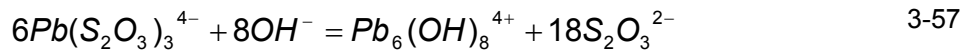
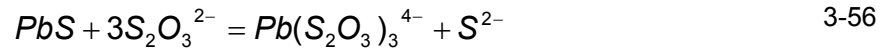
### 3) Dissolution of lead species

To study the dissolution of lead minerals, both lead hydroxide and lead sulfate solid products were removed from the calculation so that the dissolution chemistry can be revealed.



**Figure 3-24: Eh-pH diagram of Pb-S-H<sub>2</sub>O system: dissolution of lead**

Figure 3-24 is an Eh-pH diagram with lead hydroxide and the sulfate precipitates eliminated. Figure 3-24 indicates that galena can be dissolved by the hydroxide anion and form a complex,  $Pb_6(OH)_8^{4+}$ . At lower pH (< 9.3), the dissolved lead can be precipitated as  $PbS_2O_3$ . This precipitate can be re-dissolved by excessive thiosulfate anion as a thiosulfate complex,  $Pb(S_2O_3)_3^{4-}$ . These dissolution reactions can be described by Equations 3-56 to 3-61.



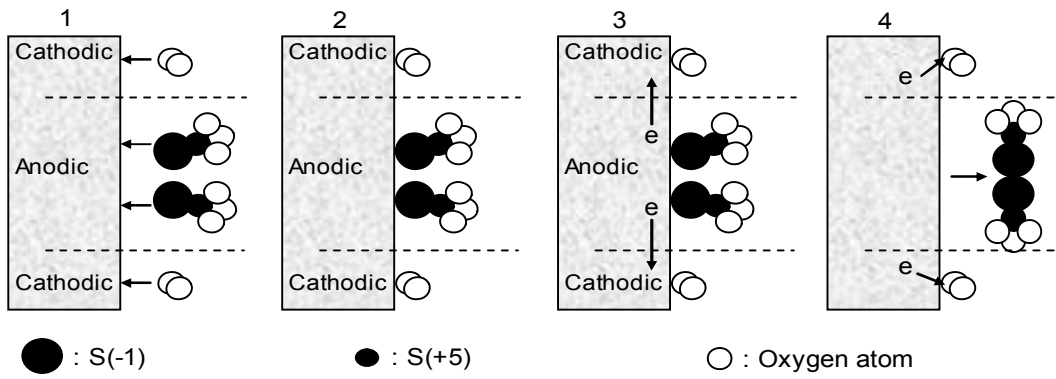
4) Passivation of minerals or gold surfaces by lead precipitates

The above analysis of the Pb-S-H<sub>2</sub>O system indicates that in a thiosulfate leaching system, galena and other lead minerals may be dissolved as hydroxide complexes. Then, a precipitate of lead thiosulfate, PbS<sub>2</sub>O<sub>3</sub>, may formed; this precipitate can be re-dissolved by excessive thiosulfate or hydroxide anion as Pb(S<sub>2</sub>O<sub>3</sub>)<sub>3</sub><sup>4-</sup> and Pb<sub>6</sub>(OH)<sub>8</sub><sup>4+</sup>. Once dissolved, the lead ion may migrate to the surfaces of particles of gold and other minerals. If the lead ion is precipitated as hydroxide or sulphate on the surface of particles, surface passivation may occur on both the gold and other mineral particles.

### 5) Summary

Lead minerals are dissolvable in alkaline solution as hydroxide complexes and produce a temporary precipitate, lead thiosulfate. This precipitate is re-dissolvable by excessive thiosulfate or hydroxide anions. The dissolved lead may migrate to and precipitate on the surface of gold, to passivate the gold surface with a layer of lead hydroxide film.

#### 3.1.3.6. Oxidation of thiosulfate catalyzed by sulfide surface



**Figure 3-25: Catalyzed oxidation of thiosulfate on pyrite surface (sketch)**

As previously discussed (section 3.1.3.1), the sulfide anion may compete with the thiosulfate for the consumption of oxidants. The oxidation products include both thiosulfate and elemental sulfur. Therefore, it is expected that thiosulfate consumption will be reduced in the presence of sulfide minerals. However, thiosulfate leaching tests on sulfide minerals have indicated that the thiosulfate tends to decompose faster in the presence of most sulfides. Such a result can be explained by the following catalysis theory.

A reaction model (Figure 3-25) [Moses et al., 1987] claimed that pyrite is semi-conductive. In the presence of thiosulfate and dissolved oxygen (or other oxidants, such as cupric ammine), the thiosulfate anion is adsorbed onto the surface of iron sulfides. By losing an electron from the anodic area to cathodic area, thiosulfate was oxidized to tetrathionate. The electron is accepted by the oxidants in the cathodic area. Cupric species, such as cupric ammine may replace the dissolved oxygen in this model and further speed up the oxidation of the thiosulfate. This reaction model further complicates the chemistry on the surface of pyrite. Other than pyrite, other iron-containing sulfide minerals, e.g., pyrrhotite, and chalcopyrite may also have similar problems to pyrite.

## **3.2. Strategies for improving thiosulfate leaching of gold**

General strategies that may improve the leaching performance in the presence of detrimental minerals will be discussed in this section. In this section, various

methods will be proposed. In a later section, some examples of using these proposed strategies to improve the thiosulfate-gold leaching will be introduced.

### **3.2.1. Reagent compositions**

The essential reagents in thiosulfate leaching include: ATS (ammonium thiosulfate), AH (ammonia hydroxide) and copper (typically as copper sulfate). In the presence of detrimental minerals, it is necessary to adjust the reagent compositions accordingly.

#### **3.2.1.1. Thiosulfate**

Thiosulfate anion is the lixiviant of gold. A reduced thiosulfate concentration will lead to an unfavourable thermodynamic condition for the dissolution of gold.

- A. By reducing thiosulfate concentration, the reduction potential of  $\text{Au}(\text{S}_2\text{O}_3)_2^{3-}$  is significantly shifted upwards (Figures 3-6 and 3-7). This means that the stability of this complex will become weaker despite the oxidative condition of the slurry remaining the same.
- B. The reduction potential of  $\text{Cu}(\text{NH}_3)_4^{2+}$  shifted slightly downwards and resulting in a weaker oxidative power of this oxidation catalyst (Figure 3-3).

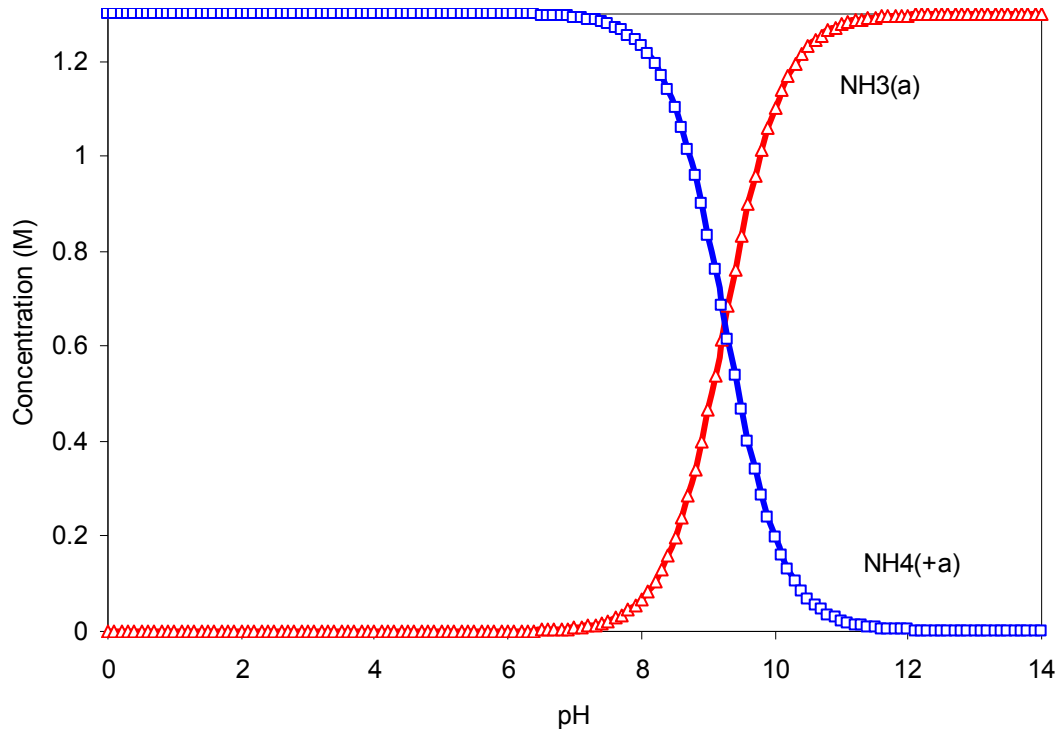
On the other hand, thiosulfate is a reducing agent; excessive additions of thiosulfate will increasingly compete with the gold in the reactions with oxidants, such as cupric ammine or dissolved oxygen. The oxidation products of thiosulfate, polythionates, also have detrimental effects on the leaching of gold.

In general, there is a range of thiosulfate concentrations for the optimum gold leaching performance. The optimum range of thiosulfate concentrations may change with the compositions of the associated minerals. The major factors to consider here are:

- A. Thiosulfate may compete with sulfides in the consumption of oxidants;
- B. Free thiosulfate anion may be consumed by certain metal ions, e.g. Pb, Zn, and Cu;
- C. Thiosulfate may enhance the leaching of certain minerals, e.g. galena.

Therefore, the optimum thiosulfate concentration might vary for different ore samples.

### 3.2.1.2. Ammonia



**Figure 3-26: Specification of ammonia/ammonium and solution pH**  
( $N_{\text{total}}$  1.3 M)

Ammonia is generally a weak alkaline that stabilizes the solution pH between 9 and 11. The co-relation of pH and specifications of  $\text{NH}_3$  and  $\text{NH}_4^+$  was calculated and is shown in Figure 3-26.

More importantly, ammonia stabilizes the cupric ion and forms a complex that works as the direct oxidant of gold. From the thermodynamic point of view, insufficient addition of free ammonia will result in a reduced window of cupric ammine in the Eh-pH diagram. Therefore the copper species will become less stable and tends to precipitate as oxides. Another problem with the weak

ammonia condition is that the reduction potential of cupric ammine will become more positive (Figure 3-3), so that more copper will present as cuprous thiosulfate rather than cupric ammine. This change reduces the amount of oxidant of gold. The increase of ammonia concentration may result in a reduced oxidation potential of gold, which make gold oxidation easier (Figure 3-1). Ammonia also enhances the gold leaching by accelerating the diffusion of the oxidation product of gold and preventing the formation of a sulfur film on gold. Overall, a higher ammonia concentration makes the gold leaching conditions more favourable.

The increase of ammonia concentration will accelerate the dissolution and oxidation of the surfaces of various sulfide minerals. On one hand, the oxidations of most sulfides become easier at higher pH (see Eh-pH diagrams of various sulphide minerals in previous sections). On the other hand, more cupric ammine is available in the oxidation of sulfide when the ammonia concentration is increased.

In addition, the passivation of some sulfides, e.g., pyrite, pyrrhotite, chalcopyrite and arsenopyrite, will require the formation of FeOOH. With more ammonia, more hydroxide anion can be supplied to enhance the formation of the FeOOH film on the surface of these minerals, and ultimately reduce the interference of these minerals on the leaching of gold.

### **3.2.1.3. Copper**

Copper concentration is a critical factor in thiosulfate leaching. The cupric ion forms a complex with ammonia and this complex works as the direct oxidant of gold. Therefore, it is necessary to remain at an optimum concentration of copper.

On the other hand, the presence of more copper oxidant will also accelerate the oxidation of thiosulfate; the overall reagent consumption will become much higher. Therefore, it is not suggested to add an excessive amount of copper.

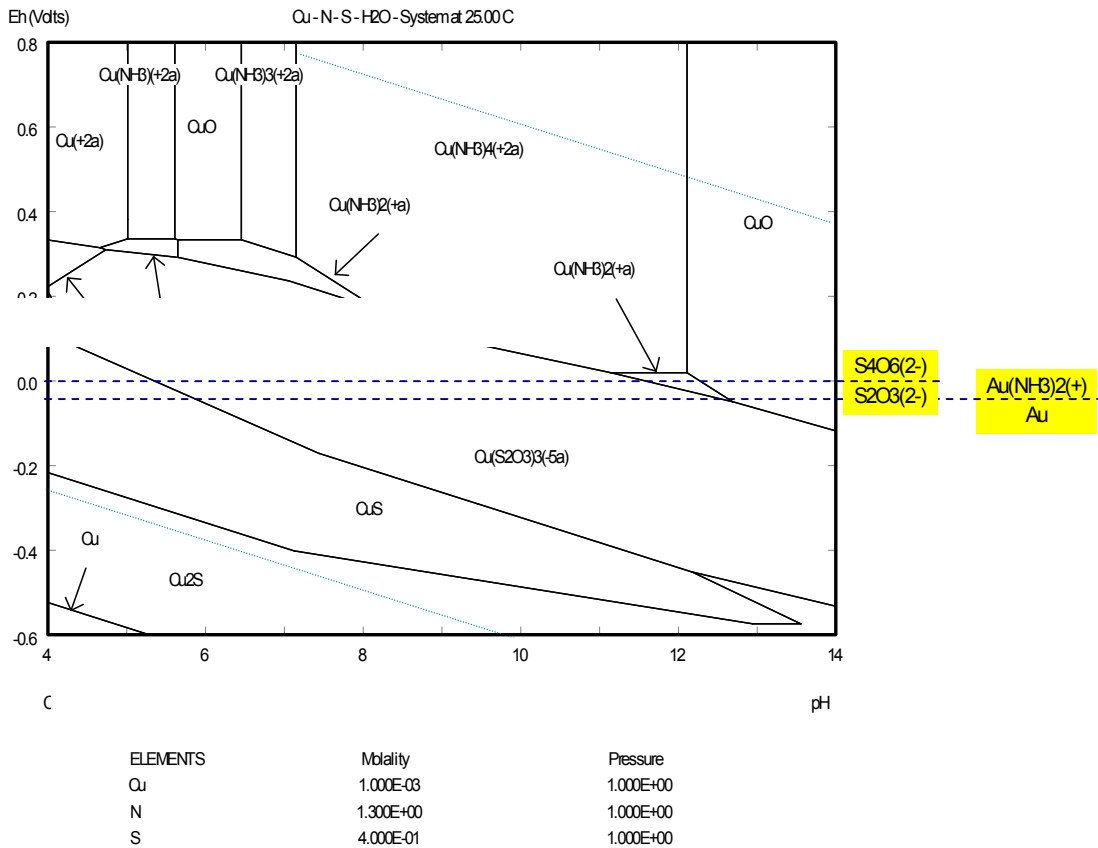
In the presence of some sulfide minerals, e.g., pyrite, pyrrhotite, chalcopyrite and arsenopyrite, copper can be reduced and co-precipitated, or adsorbed on the surface of the sulfides. Meanwhile, cupric ammine will also be consumed by the reductive sulfide surface. Therefore, in the presence of these minerals, additional copper might be required to maintain the acceptable oxidation rate of gold.

On the surface of some sulfides, e.g., pyrite and pyrrhotite, the oxidation of thiosulfate is accelerated due to the surface catalysis phenomena. The increase of copper concentration further accelerates the decomposition of thiosulfate on the surface of these minerals.

In general, by adding more copper in the thiosulfate leaching system, the gold oxidation conditions are enhanced. However, the thiosulfate stability becomes lower.

### 3.2.2. Oxygen control and alternative oxidation catalyst

In the leaching of synthetic gold ores containing various minerals, it was found that the oxidation rate of thiosulfate was much higher. As a result, the consumption of thiosulfate increased. It is challenging to leach complex sulfidic ores without excessive amount of reagents, making the process very expensive.



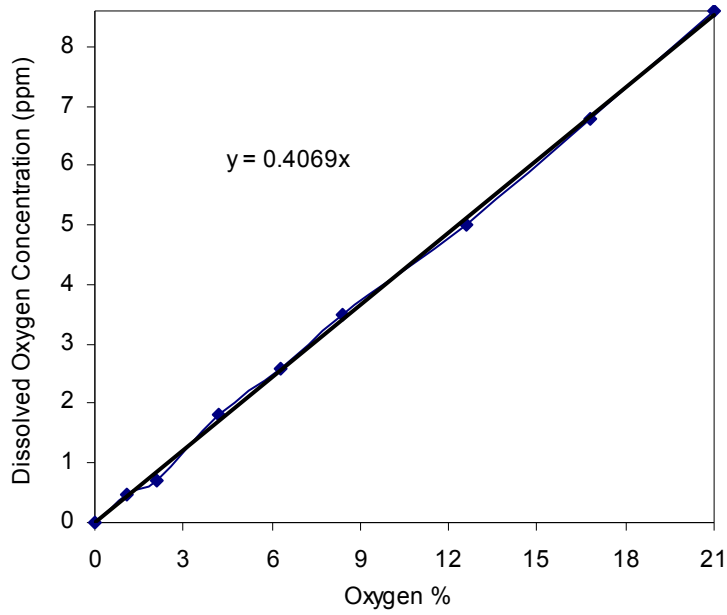
**Figure 3-27: Eh-pH diagram of thiosulfate leaching system: a comparison of important redox couples**

To solve the above-mentioned problem, the electrochemical status of the thiosulfate leaching system should be examined. By merging three the Eh-pH

diagrams (Figure 3-1, Figure 3-7 and Eh-pH diagram of Cu-S-N-H<sub>2</sub>O) into one figure 3-27. The redox potentials have the following relations:  $E(\text{Cu}(\text{NH}_3)_4^{2+}/\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}) > E(\text{S}_4\text{O}_6^{2-}/\text{S}_2\text{O}_3^{2-}) > E(\text{Au}(\text{NH}_3)_2^+/\text{Au}^0)$ . It is obvious that cupric ammine is capable of oxidizing both the thiosulfate and the gold. To reduce the oxidation of thiosulfate, a milder oxidative condition shall be maintained. There are two theoretical options for introducing a milder oxidative condition into the thiosulfate leaching system.

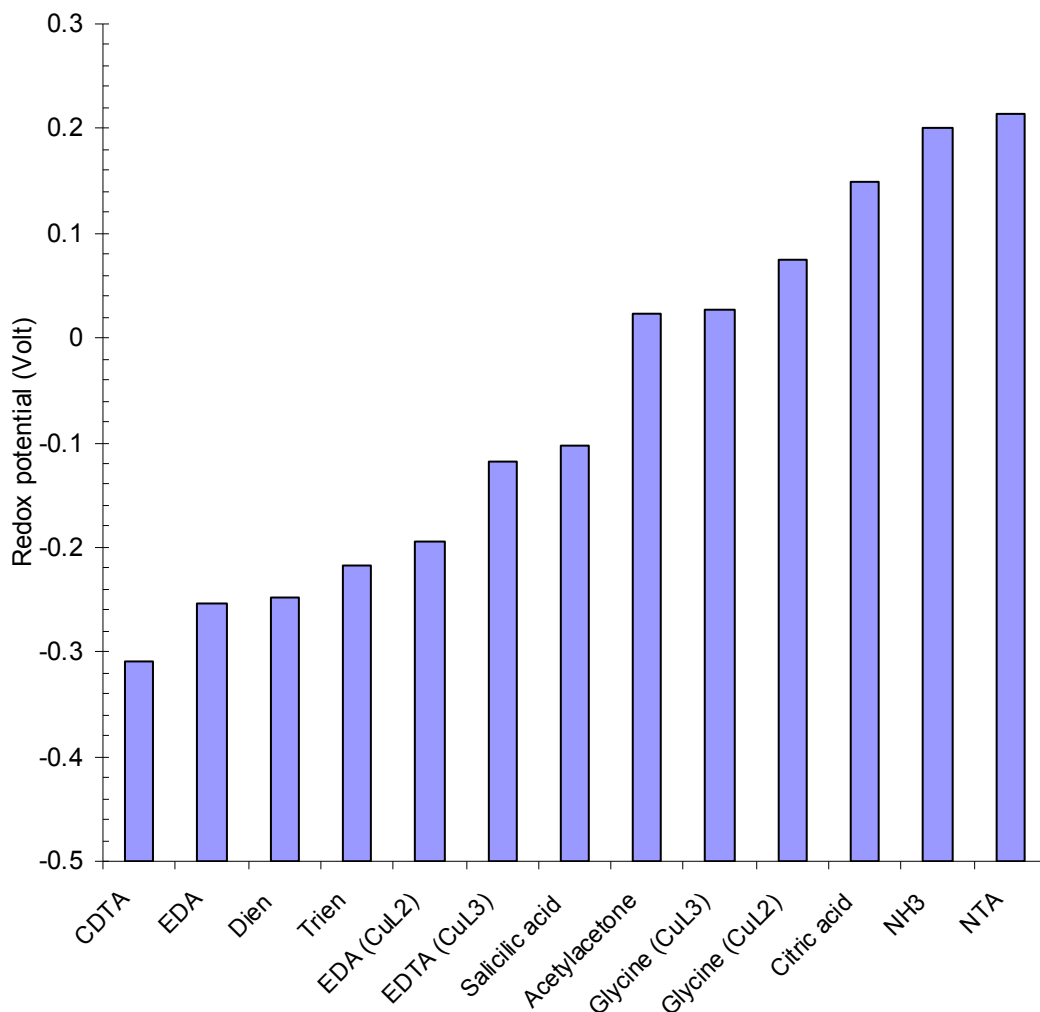
- A. Either to maintain the solution potential at a lower level so that the redox environment is oxidizing to gold but not too oxidizing to thiosulfate
- B. Or to use an alternative redox couple which has a lower oxidation potential to replace  $\text{Cu}(\text{NH}_3)_4^{2+}/\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$ .

Option A can be realized by slurry potential control using electrodes. This method is difficult practically because of the complexity of the slurry compositions. Oxygen control, however, is a mature technique that can be used practically. By limiting the air supply to the agitating tank, injecting inert gases, e.g., nitrogen gas, or elevating the slurry temperature, the concentration of the dissolved oxygen can be readily reduced. The reduction of dissolved oxygen concentration in the leaching solution will ultimately result in a reduced slurry potential.



**Figure 3-28: Oxygen supply strength vs. dissolved oxygen concentration**

As shown in Figure 3-28, by adjusting the percentage of oxygen in the sparging gas, the dissolved oxygen concentration can be maintained at a designated level. The gas mixture was controlled by mixing air and pure nitrogen gas.



**Figure 3-29: A comparison of standard oxidation potentials of some cupric complexes (estimated from the complex stability constants)**

Another way of reducing the oxidative power and stabilizing thiosulfate is to using a milder oxidizer (Option B). Figure 3-29 shows the reduction potentials of various cupric complexes estimated from their complex stability constants. As previously cited, the standard potential of  $\text{CuL}_x^y/\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$  can be estimated based on the standard potential of  $\text{Cu}^{2+}/\text{Cu}^+$  using thermodynamic principles and stability constants of copper complexes. For instance, when the ligand of copper

is ammonia, the reduction potential of  $\text{CuL}_x^y/\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$  can be estimated using Equations 3-62 and 3-63.

$$aA + bB = cC + dD \quad 3-62$$

$$E^0 = -\frac{\Delta G^0}{nF} = \frac{RT \ln K}{nf} = 0.0591 * \log K = 0.0591 * \log \frac{[C]^c [D]^d}{[A]^a [B]^b} \quad 3-63$$

Table 3-1 lists some other ligands of cupric ion, their complex equilibrium constants in log value and their standard reduction potentials. Many of these ligands can be used in thiosulfate leaching to form cupric complexes working as oxidation catalysts that have milder oxidative power.

**Table 3-1: Estimated redox potentials (Volts) of  $\text{CuL}_x^y/\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$**

Ligand	Oxidatant	Log K (Cu-L)	Reductant	Log K (Cu-TS)	Calc $E^0$ (V)
CDTA	CuL	22.0	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	-0.31
EDA	CuL3	21.0	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	-0.25
DETA	CuL2	20.9	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	-0.25
TETA	CuL	20.4	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	-0.22
EDA	CuL2	20.0	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	-0.19
EDTA	CuL	18.7	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	-0.12
Salilic acid	CuL2	18.5	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	-0.10
Acetylacetone	CuL2	16.3	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	0.02
Glycine	CuL3	16.3	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	0.03
Glycine	CuL2	15.5	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	0.07
Citric acid	CuL	14.2	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	0.15
$\text{NH}_3$	CuL4	13.3	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	0.20
NTA	CuL	13.1	$\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$	13.84	0.21

### **3.2.3. Oxidative pre-leach (pre-aeration)**

The reductive surface properties of sulfide are responsible for the decrease of gold extractions under many circumstances. By consuming oxidants and reducing slurry potential, sulfide surfaces make the dissolution of gold more difficult. Even the dissolved gold will be reduced and precipitated by these surfaces. Therefore, in presence of sulfide minerals, an oxidative pre-leach can be an important measure for minimizing the detrimental effects on gold leaching.

In the presence of thiosulfate, the oxidants, such as dissolved oxygen and the cupric species are consumed by both the sulfide surface and the thiosulfate. Therefore, the oxidation of the sulfide surface is less amendable than that in the absence of thiosulfate. Further more, some sulfide surfaces may catalyze the oxidation of thiosulfate and increase the reagent consumption.

Consequently, the oxidation of a sulfide surface should ideally be performed prior to thiosulfate leaching. A pre-aeration step is typically the most economical way of having sulfide partially oxidized. As long as air or oxygen is supplied and efficient agitation is maintained, the oxidative pre-leach can be conducted in water without the addition of any chemicals. However, considering that ammonia and cupric ammine can accelerate the oxidation of sulfides and promote the degree of surface passivation by  $\text{FeOOH}$ , a pre-leach with ammonia and copper may provide more benefits in the pre-leach stage.

As suggested in Section 3.2.2, thiosulfate leaching of gold under a limited oxygen supply can reduce the reagent consumption without affecting gold dissolution. However, with reduced oxygen content, the oxidation of the sulfide surface will become very slow. The gold extraction, therefore, will remain low. However, if the sulfide surface is oxidized in the pre-aeration step, the negative effects of the associated sulfide minerals will be less significant. Therefore, a combination of an oxidative pre-leach process with a thiosulfate leaching under reduced oxygen concentration can be a promising solution to many sulfide gold ores.

#### **3.2.4. Masking of detrimental surface or ions with chemical additives**

To minimize the detrimental effects of either the reactive surfaces or the dissolved ions, additional chemicals might be required to mask or passivate the problematic species. Extra hydroxide, phosphate, lead, carbonate and calcium ions are some examples that can be used to deactivate the harmful species in thiosulfate leaching process.

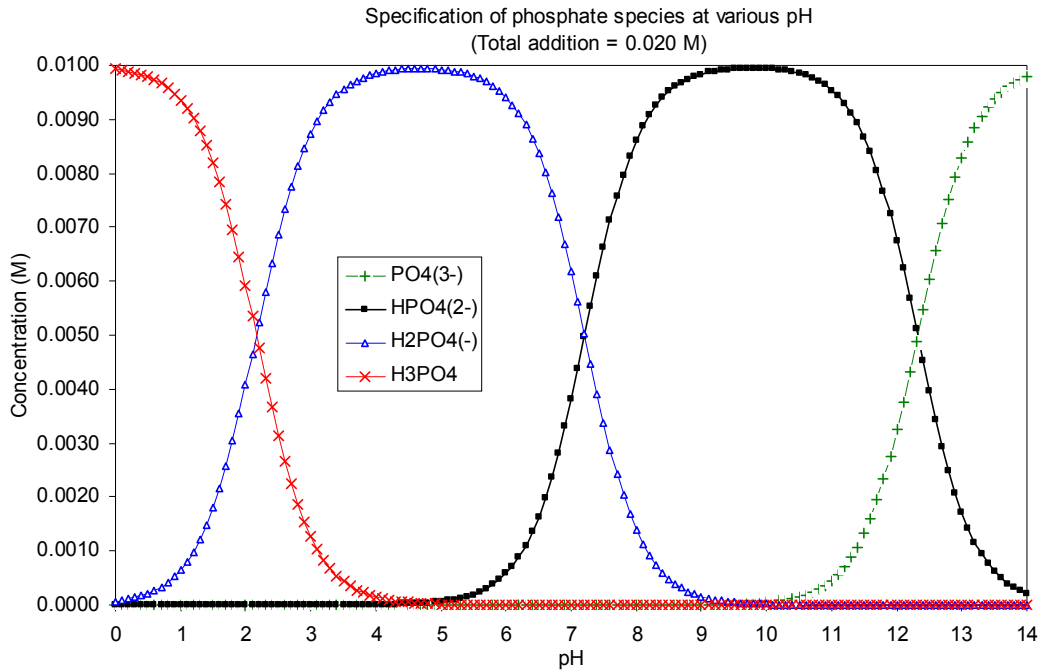
##### **3.2.4.1. Hydroxide anion**

The oxidation of many sulfides is easier at higher pH. This is because the oxidation potentials shift downwards when the pH increases. More importantly, the formation of a passivation layer of FeOOH also requires a stronger alkaline condition. Therefore, additional hydroxide anion shall be beneficial to gold leaching. However, at higher pH, the oxidation catalyst, cupric ammine, becomes

instable, which will ultimately affect the gold extraction. Therefore, the amount of hydroxide anion must be carefully controlled to avoid this problem.

#### **3.2.4.2. Phosphate anion**

As discussed previously, surface passivation of sulfides is a critical measure that diminishes their detrimental effects on thiosulfate leaching of gold. FeOOH is the most important species that helps to passivate the surface of many sulfide minerals. However, with the formation of FeOOH, gold and copper may also be co-precipitated. In addition, some sulfide minerals contain lower level of iron. The passivation effect on these minerals, therefore, is very weak. By the addition of a reagent that could precipitate metal ions on its surface, the reactive surface might be passivated by these precipitates instead of FeOOH. One of the reagents that has been traditionally used as a surface passivation agent is the phosphate anion.



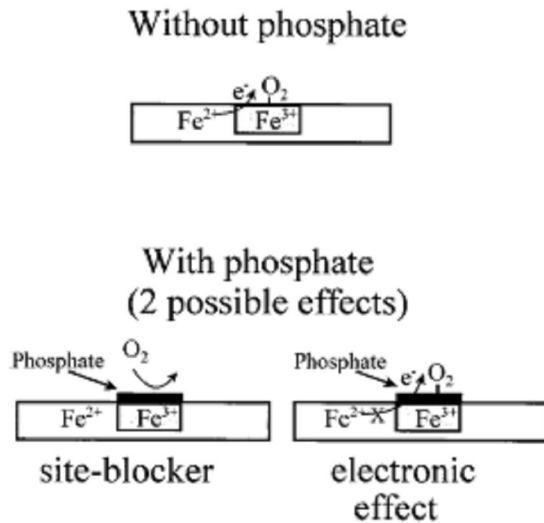
**Figure 3-30: Phosphate specification diagram**

Phosphate acid is a medium-strength acid. By hydrolysis, phosphate acid exists as  $\text{H}_3\text{PO}_4$ ,  $\text{H}_2\text{PO}_4^-$ ,  $\text{HPO}_4^{2-}$ ,  $\text{PO}_4^{3-}$ . The distribution of these species is demonstrated in Figure 3-30. Under typical thiosulfate operating conditions, the leaching pH is between 9 and 11. Thus,  $\text{HPO}_4^{2-}$  will be the major form of phosphate present.

Phosphate forms many stable precipitates with metal ions. As shown in Table 3-2, phosphate may precipitate many metal ions or chemically adsorb on the surface of many metal sulfides. As a result, phosphate was reported in many publications as a surface passivation reagent [Cai et al., 2005].

**Table 3-2: pK<sub>sp</sub> of metal phosphate precipitates [Dean, 1985]**

Phosphate precipitate	pKsp
BiPO <sub>4</sub>	22.9
Ca <sub>3</sub> (PO <sub>4</sub> ) <sub>2</sub>	28.7
Co <sub>3</sub> (PO <sub>4</sub> ) <sub>2</sub>	34.7
Cu <sub>3</sub> (PO <sub>4</sub> ) <sub>2</sub>	36.9
FePO <sub>4</sub>	21.9
Pb <sub>3</sub> (PO <sub>4</sub> ) <sub>2</sub>	42.1
Ni <sub>3</sub> (PO <sub>4</sub> ) <sub>2</sub>	30.3
Zn <sub>3</sub> (PO <sub>4</sub> ) <sub>2</sub>	32.0



**Figure 3-31: Effects of phosphate anion the surface of iron sulfide minerals**  
**[Elsetinow et al., 2001]**

As shown in Figure 3-31, the adsorption of phosphate and its chemical bond with Fe<sup>3+</sup> have two possible functions: blocking the reactive site from external oxidants and limiting the electron transfer. Therefore, phosphate is a promising additive that may accelerate the passivation of reactive surface of those iron-containing sulfides.

For copper sulfides, phosphate may also improve the leaching of gold. Although the thermodynamic calculation does not support cupric phosphate as the most

stable species under typical thiosulfate leaching conditions, the adsorption of phosphate on the active site of chalcocite surface is still possible. And the chemical bond between  $\text{Cu}^{2+}$  and phosphate, therefore, may slow down the leaching of copper.

In the presence of galena, the function of phosphate is different. Without phosphate, dissolved lead may form an inert film on gold surface. In the presence of phosphate, however, the formation of inert lead precipitates may be hindered. The reactive surface of galena can also be partially passivated with this precipitate. The dissolved lead ion may precipitate as lead phosphate before it deposits on the surface of gold.

In general, phosphate acts as a surface passivation agent for pyrite, pyrrhotite, arsenopyrite, chalcocite and galena, and may enhance the gold extraction and reduce the consumption of thiosulfate in the leaching of gold ores containing these minerals.

#### **3.2.4.3. Lead ion**

As discussed in previous sections, the dissolvable lead ion can be very harmful to the leaching of gold. The formation of lead thiosulfate precipitate, lead hydroxide, lead thiosulfate and  $3\text{PbO}\cdot\text{PbSO}_4$  precipitates may passivate the gold surface and consume thiosulfate. However, in the presence of some sulfide minerals, when the lead precipitates are formed on the surface of sulfides preferentially, the detrimental effects of these minerals can be masked. Therefore

by adding trace amounts of lead minerals or salts, the reactive surface of the sulfide will be coated with lead species and result in a better gold extraction.

### 3.2.4.4. Calcium ion

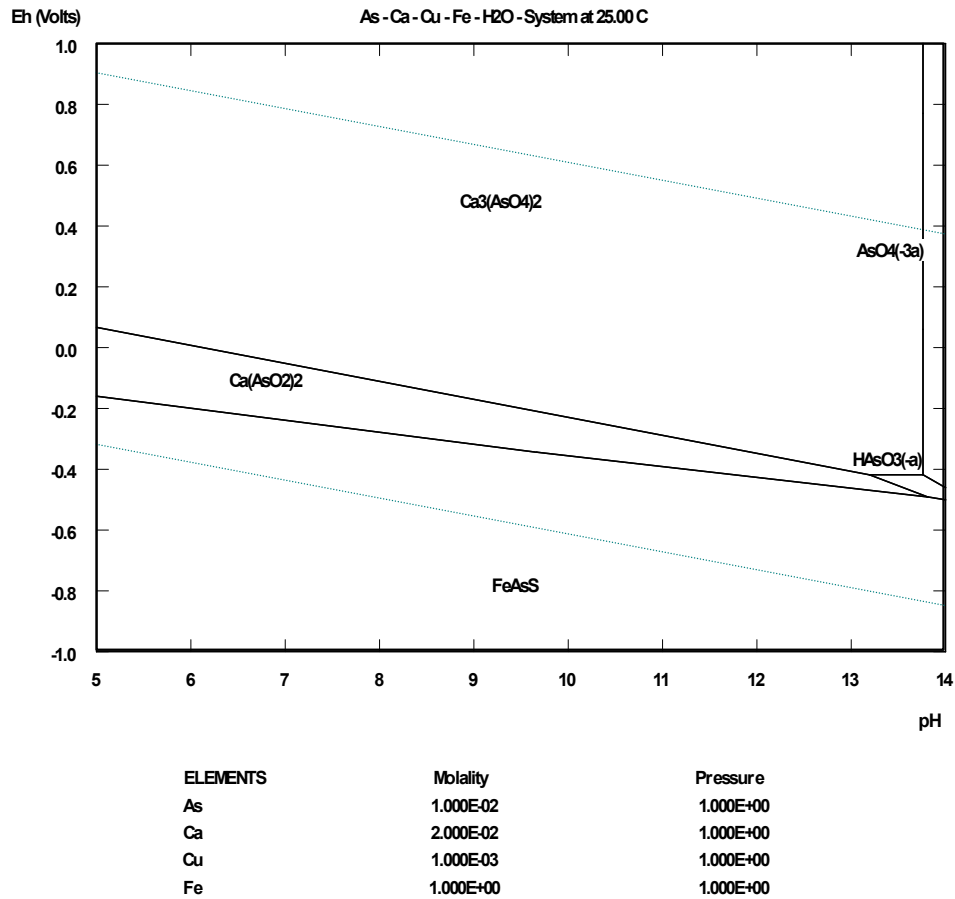
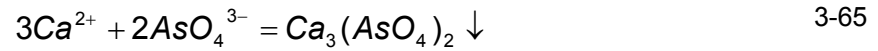
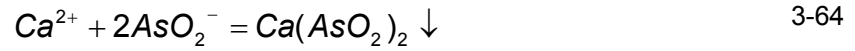


Figure 3-32: Eh-pH diagram of As-Ca-Cu-Fe-H<sub>2</sub>O system

In the leaching of arsenopyrite, it is proposed to use the calcium ion as a passivation agent. Figure 3-32 is an Eh-pH diagram of arsenopyrite in the presence of calcium ion. Ca<sub>3</sub>(AsO<sub>4</sub>)<sub>2</sub> predominates under typical leaching

conditions and this precipitate may passivate the reactive surface of arsenopyrite (see Equations 3-64 and 3-65).



There are a few other metal ions that may work in similar fashion to the passivation of arsenate, e.g.,  $\text{Zn}^{2+}$ ,  $\text{Ni}^{2+}$ ,  $\text{Mn}^{2+}$ ,  $\text{Mg}^{2+}$ ,  $\text{Fe}^{3+}$  and  $\text{Pb}^{2+}$ .

### 3.2.4.5. Carbonate anion

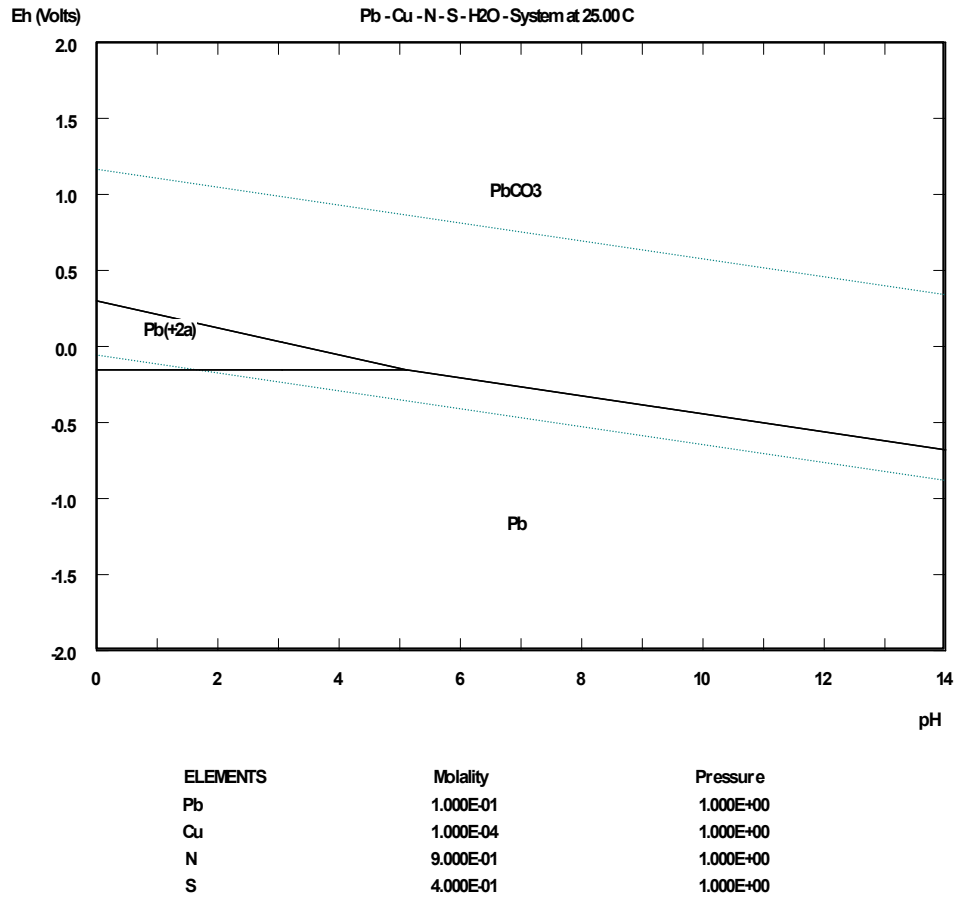


Figure 3-33: Eh-pH diagram of Pb-S-N-Cu-C-H<sub>2</sub>O system

Figure 3-33 indicates that the carbonate anion reacts with the lead ion and forms a precipitate, PbCO<sub>3</sub> (see Equation 3-66).



Once formed, lead carbonate will be difficult to dissolve in alkaline thiosulfate solution. Therefore, carbonate may be used for the passivation of galena before

further dissolution of lead. For the lead ion that is already dissolved, the formation of lead carbonate may also be beneficial for preventing the formation of a dense film of deposit on other minerals or the gold surface.

### **3.2.5. Summary**

In this section, various methods for improving the thiosulfate leaching of gold in presence of detrimental minerals were proposed. Among these methods, adjusting the reagent compositions is the most basic approach for most gold ores. Oxygen control and using a milder oxidation process both provide options for the leaching of those gold ores consuming large amount of thiosulfate. An oxidative pre-leach is also suggested in the treatment of sulfidic gold ores. For free-milling gold ores containing detrimental minerals, the leaching performance could be improved by enhancing the passivation of reactive surfaces or masking the dissolved metal ions with chemical additives, such as phosphate and lead ions.

In the following section, examples will be discussed on how to make use of these suggested methods to improve the leaching performance in presence of various detrimental minerals.

## Chapter 4. Experimental

### 4.1. Materials

#### 4.1.1. Ore samples

To understand the effect of various minerals in the thiosulfate leaching of gold, a pure silicate gold ore was used as a reference. The reference ore (Ore #1) sample contains mainly quartz which is known as an inert mineral in thiosulfate solution. The gold grade determined by fire assay of this ore was 16.26 g/t. Trace amounts of iron oxide (0.18%) and copper oxide (0.002%) were detected. The carbon and sulfur contents by the combustion volumetric technique using a LECO titrater SC-444DR were 0.19% and 0.0% respectively.

Another ore sample used in this research was a copper-bearing sulfidic gold ore that was a mildly refractory during cyanidation. This ore sample (Ore #2) contains 3.16 g/t gold (standard fire assay), 4.89% iron, 4.08% sulfur, 0.27% copper and 0.20% carbon. Mineralogical analysis of natural gold ore (Ore #2) showed that there was 7.25% pyrite, 0.58% chalcopyrite, 1.91% hematite, 2.40% Ankerite, 0.12% malachite, and 81.53% quartz and clay.

Two POX residue (pressure-oxidized) ore samples (Ore #3 and #4) were used in this research to confirm the established optimal conditions. The autoclave feed was a sulfidic ore containing a significant amount of organic carbon. It was found that most of the gold value was still locked inside of the sulfidic matrix, even after

fine grinding. The POX process was conducted at a Canadian commercial laboratory using a bench-scale autoclave. The sulfidic contents were fully oxidized in the POX process. However, the carbonaceous materials remained so that cyanidation was not feasible due to the preg-robbing phenomena. The other associated minerals in this POX residue sample were found to be iron oxides, quartz and some other minor impurities. The oxidation of Ore #3 was not fully completed, while Ore #4 was fully oxidized sample. Due to the limited quantity of the sample, the elemental and mineralogy analysis data was not available.

#### **4.1.2. Minerals**

Various minerals of high purities were purchased and analyzed for this research. The elemental and mineralogy compositions of these minerals are listed in Tables 4-1 and 4-2. The elemental compositions of each sample were obtained through ICP analysis. Based on the elemental compositions, the mineralogy contents of each sample were determined by semi-quantitative XRD study followed by a metal balance calculation.

**Table 4-1: Mineral compositions of the natural mineral samples used in this research**

Mineral contents	Natural mineral samples												
	Pyrite	Pyrrhotite	Galena	Litharge	Chalcocite	Chalcopyrite	Bornite	Arsenopyrite	Realgar	Orpiment	Goethite	Hematite	Magenetite
Pyrite	93.76%	3.18%	0.33%			1.03%		1.36%					
Pyrrhotite	5.78%	66.95%			0.52%		1.36%						
Arsenopyrite	0.03%	0.17%						95.02%					
Chalcopyrite	0.03%	2.08%			1.01%	95.09%	1.69%	0.67%					
Chalcocite					95.06%		0.67%						
Bornite						0.10%	90.11%						
Hematite	0.10%	1.05%	0.12%								2.89%	96.78%	1.15%
Magenetite	0.30%	2.37%					2.35%						93.07%
Quartz		20.13%			2.21%	3.23%	2.03%				5.92%	2.05%	3.08%
Pentlandite		0.12%											
Realgar								0.81%	93.01%	10.85%			
Sphalerite		1.82%											
Galena		0.15%	97.69%										
Litharge			0.11%	99.00%									
Calcite				0.78%		0.23%			1.98%	1.06%			
Dolomite			0.45%								0.89%		
Cerussite				0.22%	0.51%								
Siderite								1.08%			0.86%		
Orpiment									3.31%	83.93%			
Gypsum										2.86%			
Goethite											89.15%		
Talc													2.62%

**Table 4-2: Elemental compositions of the natural minerals samples used in this research**

Minerals	Fe	S	Cu	As	Pb	Si	Ni	Zn	Ca	Mg	C
Pyrite	47.24%	52.12%	0.01%	0.01%							
Pyrrhotite	46.44%	28.34%	0.72%	0.08%	0.13%	9.41%	0.04%	1.17%			
Galena	0.24%	13.27%			84.70%				0.10%	0.06%	0.06%
Litharge					92.07%				0.31%		0.10%
Sphalerite	5.29%	32.22%	0.03%			1.45%		58.76%			
Chalcoite	0.63%	19.70%	76.26%		0.40%	1.03%					
Chalcopyrite	29.43%	33.80%	32.99%			1.51%			0.09%		0.03%
Bornite	13.09%	24.27%	58.17%			0.95%					
Arsenopyrite	33.95%	19.91%	0.23%	44.29%							0.11%
Realgar		29.17%		67.15%					0.79%		0.24%
Orpiment		36.60%		58.71%					1.09%		0.13%
Goethite	58.47%					2.77%			0.19%	0.12%	0.21%
Hematite	67.69%					0.96%					
Magnetite	68.15%					2.22%				0.50%	

### 4.1.3. Reagents

All chemical reagents used in this research were technical grade. De-ionized water was used to prepare the solutions. Tap water was used in the wet-grinding step.

## 4.2. Experimental procedures

A natural Quartz–Au ore (Au 16.26 g/t) was ground with a ring grinder to produce a particle size of 90% -200 mesh. Natural minerals were ground to 85% -270 mesh immediately before the leaching process to avoid surface oxidation. The ground reference gold ore was mixed with quartz flour (99% -200 mesh) and a certain amount of ground natural minerals as specified in the experimental conditions. Chemicals were dissolved in a flask with distilled water at the concentrations specified in the experimental conditions. The slurry was pulped in a glass reactor at 33.3% solid. During “open-air” tests, the glass reactor was

loosely covered with a cap to allow the air to circulate. For the test of oxygen controlled tests, the reactor was capped and sealed with two tubes plugged into the cap to allow the gas mixture, with a controlled oxygen percentage, to flow through the reactor. To avoid significant loss of ammonia during the leaching process, the mixed gas was first guided into a container filled with ammonia solution at a designated concentration. The reactor was placed on an orbital shaker. The shaking speed was 210 rpm, unless otherwise specified. Leaching tests were conducted at ambient temperatures (20-24°C). Unless otherwise specified, the slurry pH was not adjusted during the leaching process. At specified time interval, the shaker was stopped and the clear solution samples were taken for analysis purpose. The concentration of thiosulfate was detected with an altered iodine titration method. Finally, the leached solid slurry was collected. The filter cake was washed with water twice. All solution samples were assayed for Au concentration using atomic adsorption spectroscopy. Most solid residues in this investigation were boiled and leached with aqua regia solution (50%) on a hot plate for 30 minutes. The resulting solutions were further analyzed for Au using an atomic absorption method. Some residues were subjected to standard fire assays to obtain more accurate readings of the gold.

Titration of thiosulfate with iodine, based on Equation 3-1, was used in this research for determining the concentration of thiosulfate anion.



The presence of Au(I), Cu(I) and Cu(II) ions in the solution may affect the accuracy of the thiosulfate determination [Chow, et al., 1963]. However, the concentrations of these ions were typically on the low side and therefore can be ignored within the expected accuracy level.

Ion chromatography is another method for the determination of the concentrations of sulfite, sulfate, thiosulfate, and polythionates. As an analytical technology that requires delicate equipment and longer duration of time, it is not suitable for a real-time determination of thiosulfate. A recent published work suggested that thiosulfate can be detected using a rotating electrochemical quartz crystal microbalance (REQCM) as a sensor for the reaction of silver and thiosulfate in a flow injection analysis system. Because the changes of solution chemistry may interfere with the oxidation of silver, an estimation of various sources of errors shall be based on the chemical natures of the target solution. The inventor of this technology also suggested that thiosulfate in the complex of Cu(I) is not detectable by this method [Breuer et al., 2002].

These advanced analytical technologies provide higher accuracy and less human error. However, iodine titration method was found to be sufficiently accurate for serving the purpose of this investigation.

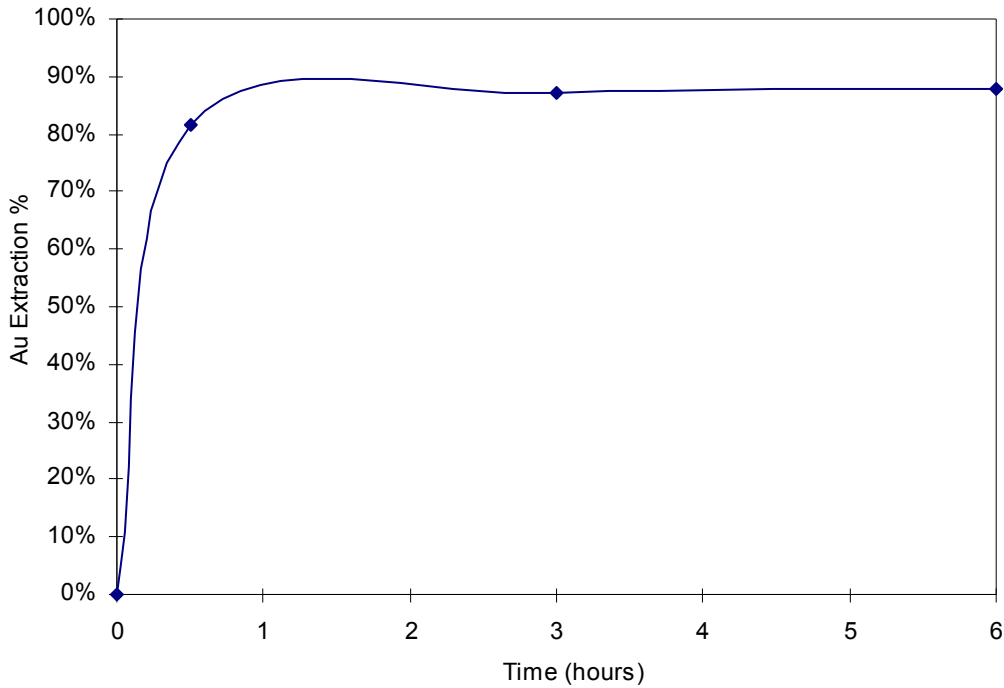
## **Chapter 5. Results and discussion**

In this chapter, the effects of some associated minerals will be introduced with experimental results. Theoretical analysis on the detrimental effects will be discussed. Some potential solutions for the problems caused by these minerals will be proposed and the leaching test results will be demonstrated for verifying these proposed solutions. Some practical examples of using these improved methods in the leaching of natural occurring gold ores will be introduced at the end of this chapter.

### **5.1. Effects of associated minerals: leaching results**

In this section, the leaching tests on synthetic ore samples will be presented. The reference ore contains gold grains distributed in a quartz substrate. Quartz is known for its inert properties. Therefore, when a sulfide mineral was blended into this ore, the possible effects of this mineral will be reflected by changes on the leaching performance. This can be accessed by comparing with the leaching result of the silicate gold ore (the reference ore) with the blended ore.

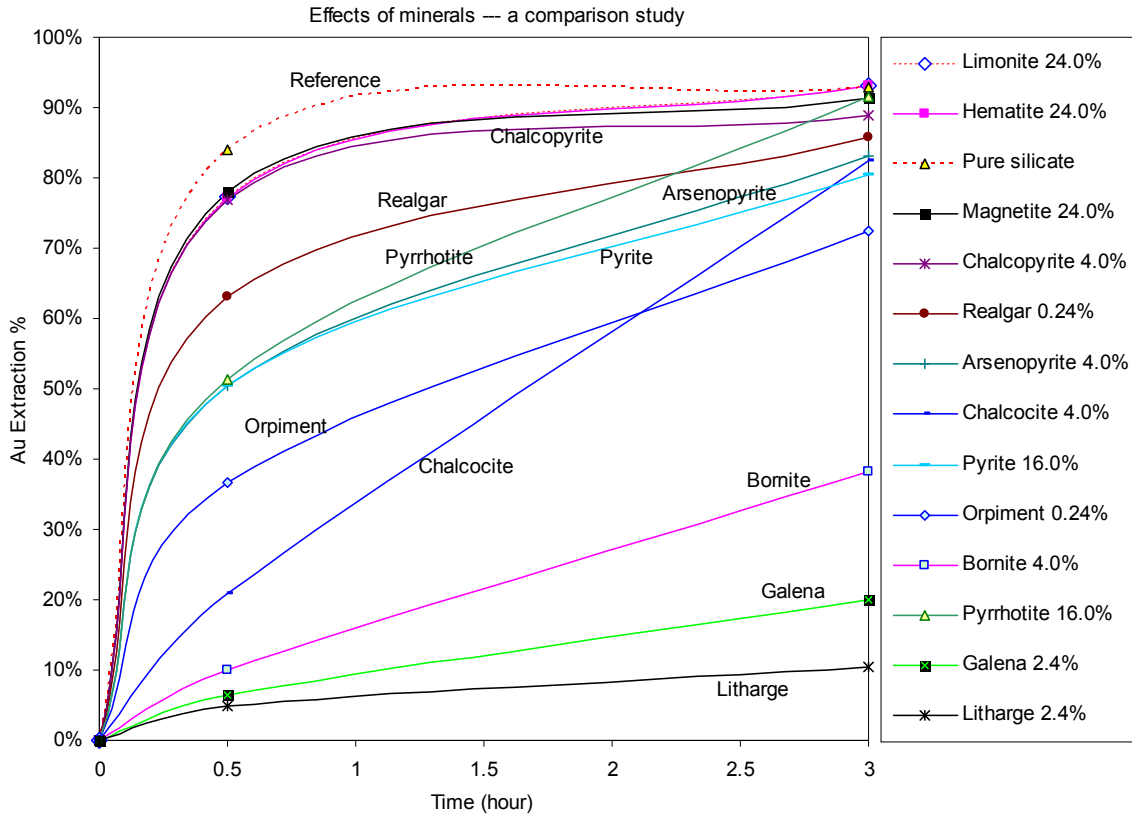
### 5.1.1. Effects of various minerals: a preliminary comparison



**Figure 5-1: Leaching kinetics of the reference gold ore**  
(ATS: 0.20 M, Cu: 1.2 mM, oxygen: 21.1%, AH: 0.3 M)

As shown in Figure 5-1, thiosulfate leaching of the reference gold ore in absence of sulfide mineral was fast under moderate leaching conditions: 0.20 M ATS, 1.2 mM Cu, 0.3 M ammonia. This test was used as the standard for other tests performed on synthetic ores. About 81.5% of gold was extracted in 30 minutes. At 3.0 hours, the gold extraction reached 87.3%. Extending the leaching time to 6.0 hours had insignificant effect on the overall gold extraction. Therefore, for most of the tests in this research, the duration of leaching test was set at 3.0 hours. The gold extractions were also measured at 0.5 and 3.0 hours. Thiosulfate concentration and consumption was measured at 3.0 hours.

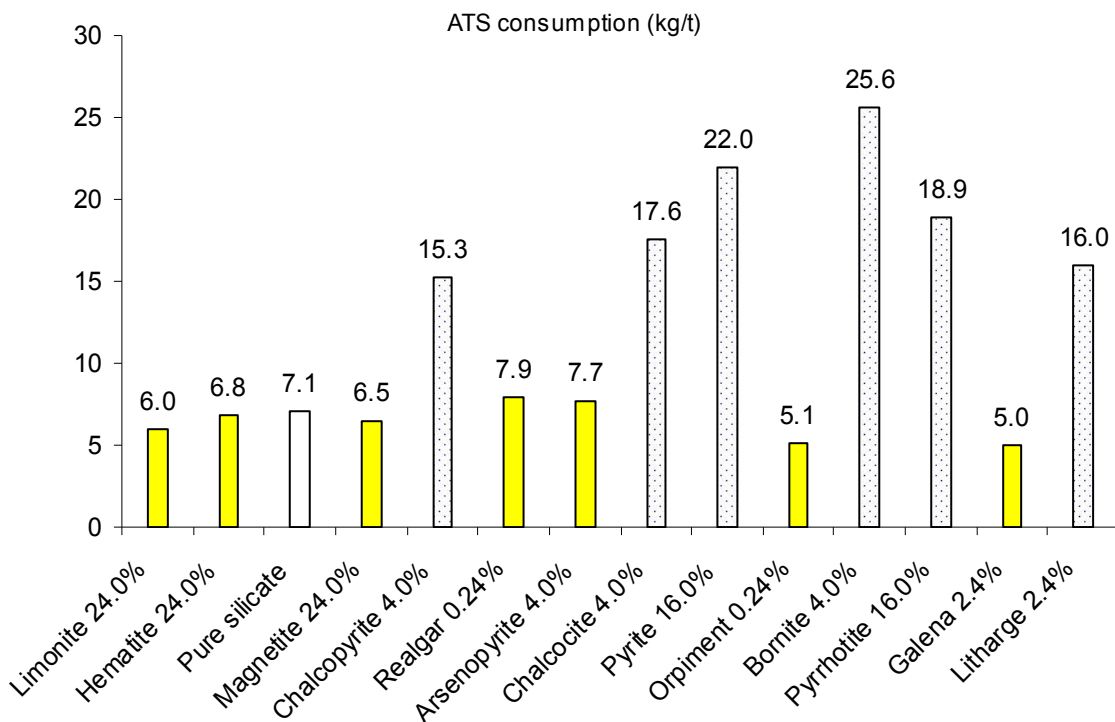
A preliminary comparison study was conducted using the reference ore mixed with selected minerals. The leaching performances were compared with each other. The purpose of this study was to recognize the general effects of the different associated minerals.



**Figure 5-2: Effect of associated minerals on gold extractions - a preliminary comparison study (I)**  
 (Cu: 1.2 mM, AH: 0.9 M, ATS: 0.20 M, oxygen: 21.1%)

As shown in Figure 5-2, various minerals were blended respectively into the silicate reference ore at different percentages. The dashed curves represent the reference ore in the absence of any mineral impurity. The leaching performance of this reference ore was the best among the group, 93.0% gold was leached in

3.0 hours. Iron oxides at 24%, has a minor detrimental effect on the leaching kinetics. Iron sulfides, chalcopyrite, arsenopyrite and realgar appeared to be moderately detrimental. The gold extractions at 3.0 hours were between 80 and 89%. Bornite, chalcocite, galena, and litharge, even at low concentrations, were the most detrimental species. The gold extractions at 3.0 hours were from 10% to 80%.



**Figure 5-3: Effect of associated minerals on thiosulfate consumptions - a preliminary comparison study (II)**

(AH: 0.9 M, Cu: 1.2 mM, ATS: 0.20 M, oxygen: 21.1%)

The effects of various minerals on the thiosulfate consumption were also studied and the result is shown in Figure 5-3. Without any additional mineral, the reference ore consumed 7.1 kg/t. In the presence of limonite, hematite,

magnetite, realgar, arsenopyrite, orpiment, and galena, the ATS consumptions (solid yellow bars) were close to 7.1 kg/t. The bars that are filled with dots represent the problematic minerals: chalcopyrite, chalcocite, pyrite, pyrrhotite, bornite, and litharge. The thiosulfate consumption in the presence of these minerals were doubled or even tripled.

The initial slurry electric potentials in the leaching of various minerals are listed in Table 5-1. In the absence of additional minerals, the slurry potential was -30 mV. The presence of limonite, hematite and magnetite did not make any significant impacts on slurry potential. The addition of litharge resulted in a significant increase of slurry potential (-8 mV). Such an increase of slurry potential may explain the increase of thiosulfate consumption in the presence of litharge.

**Table 5-1: Initial slurry potentials in the presence of various minerals (Standart Hydrogen Electrode)**

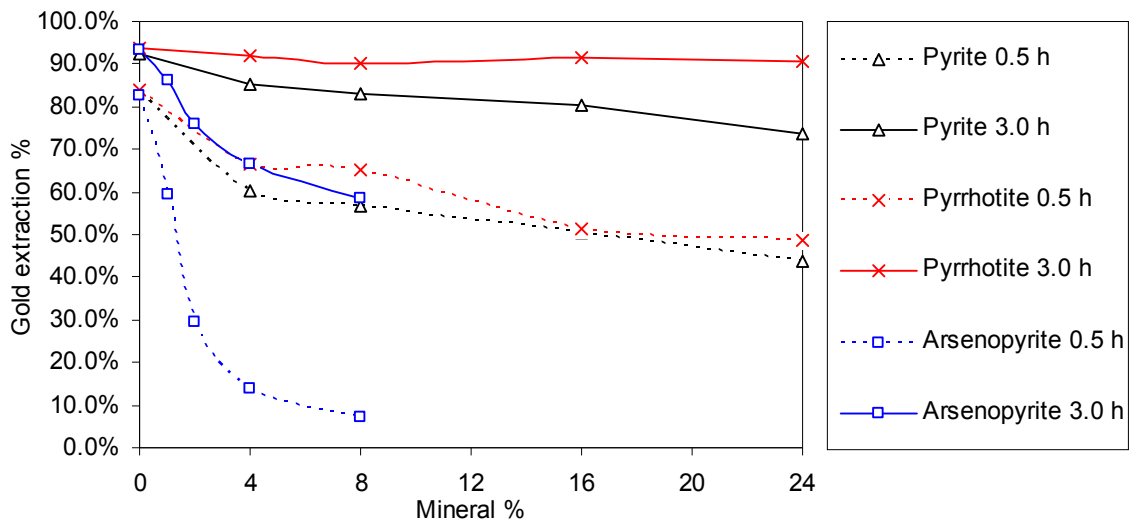
Additional minerals	Initial slurry potential (mV)
None	-30
Limonite 24.0%	-26
Hematite 24.0%	-25
Magnetite 24.0%	-27
Chalcopyrite 4.0%	-96
Realgar 0.24%	-168
Arsenopyrite 4.0%	-75
Chalcocite 4.0%	-179
Pyrite 16.0%	-97
Orpiment 0.24%	-63
Bornite 4.0%	-122
Pyrrhotite 16.0%	-101
Galena 2.4%	-93
Litharge 2.4%	-8

All sulfide minerals led to a change of slurry potential in the negative direction. Such a reduction of slurry potential is helpful to explain the moderate drop of gold

extractions observed in the presence of iron sulfides. Realgar, chalcocite, bornite and pyrrhotite were the minerals that had the most negative slurry potentials (< -100 mV). Other than pyrrhotite, all the other three minerals appeared to have significantly reduced gold extractions (from 85.9% to 38.3%). The high gold extraction (91.6%) in the presence of pyrrhotite indicated that the surface of this mineral was rapidly passivated and became less reductive. The extremely low gold extractions in the presence of lead minerals can obviously not be explained by the slurry potentials. Therefore, the inhibited gold leaching is possibly a result of surface passivation.

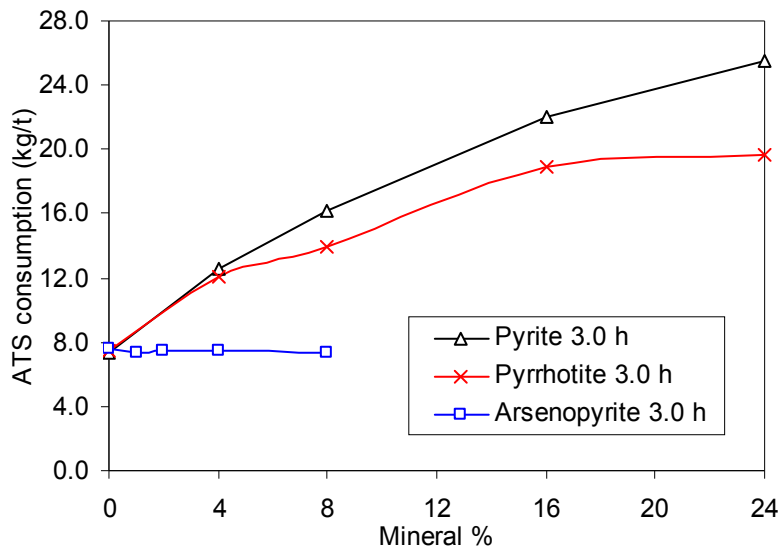
It can be concluded that thiosulfate leaching of the silicate reference ore containing free-milling metallic gold was fast and complete. The thiosulfate consumption was also on the low side, i.e., 7.1 kg/t. This result, therefore, indicated that the problematic gold extractions and thiosulfate instability problems observed in the leaching of many natural gold ores is related to various associated minerals. It is necessary to study the effects of these minerals, understand the problems and more importantly, and suggest some leaching strategies for minimizing them.

### 5.1.2. Effects of pyrite, pyrrhotite and arsenopyrite



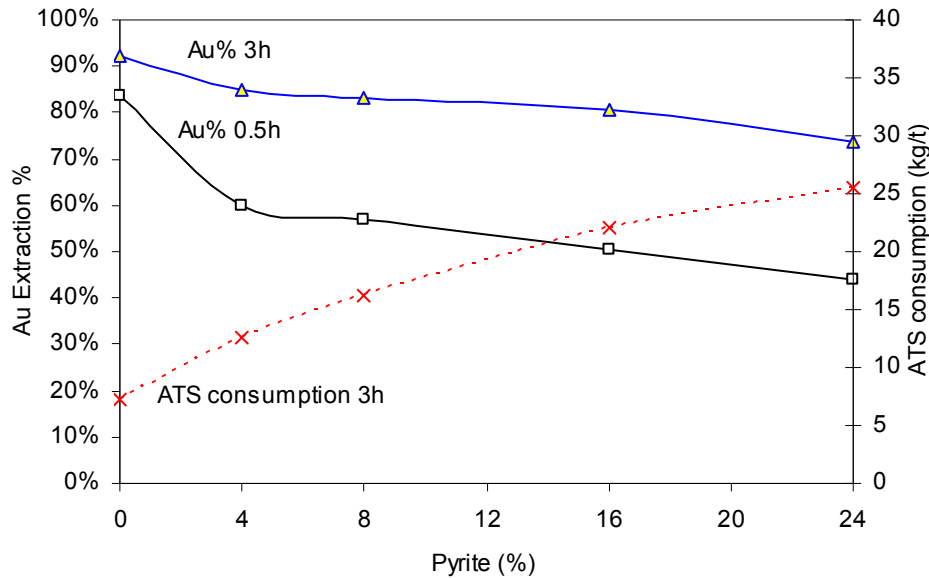
**Figure 5-4: Effects of pyrite, pyrrhotite and arsenopyrite (I) on gold extractions (ATS: 0.20 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

The effects of pyrite, pyrrhotite and arsenopyrite at various weight percentages in the quartz gold ore are compared in Figure 5-4. In the presence of pyrite, pyrrhotite or arsenopyrite, gold extractions were largely reduced within the initial 0.5 hour. The dashed curves in Figure 5-4 represent gold extraction at 0.5 hour. Among these three sulfides, the detrimental effects were in the order of arsenopyrite >> pyrite > pyrrhotite. At 3.0 hours, the negative effects of all three minerals became less significant than they were at 0.5 hour of leach. The passivation theory stated in section 3.1.3 can be used to explain this phenomenon.



**Figure 5-5: Effects of pyrite, pyrrhotite and arsenopyrite (II) on thiosulfate consumptions (ATS: 0.20 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

On the other hand, the effects of these sulfides on thiosulfate consumption are shown in Figure 5-5. The ATS consumptions in the presence of 24% pyrite or 24% pyrrhotite were doubled or tripled. Arsenopyrite, within the range of 0.0-8.0%, has almost no impact on the stability of thiosulfate. The increased consumption of thiosulfate in the presence of pyrite and pyrrhotite verified the surface catalysis theory introduced in section 3.1.3.6.



**Figure 5-6: Effect of pyrite on gold extraction and thiosulfate consumption (ATS: 0.20 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

After a comparison of these three minerals, the effects of each individual mineral will be discussed in details. As shown in Figure 5-6, the gold extractions decreased with the increase of pyrite percentage. At 0.5 hours, the addition of 4% pyrite led to a 20% reduction in gold extraction. At 3.0 hours, the reduction of gold extraction was 7.2% in the presence of 4% pyrite.

The consumption of ATS increased with the increase of pyrite percentage. By adding 24% pyrite, about 25.5 kg/t ATS was consumed within 3.0 hours, 18.2 kg/t more than that in the leaching test without pyrite.

In the leaching test with 4% pyrite, the copper concentration was found to be reduced from 1.2 mM to 0.6 mM, immediately after the contact of sulfide with the leaching solution. The decrease of copper concentration indicated the adsorption

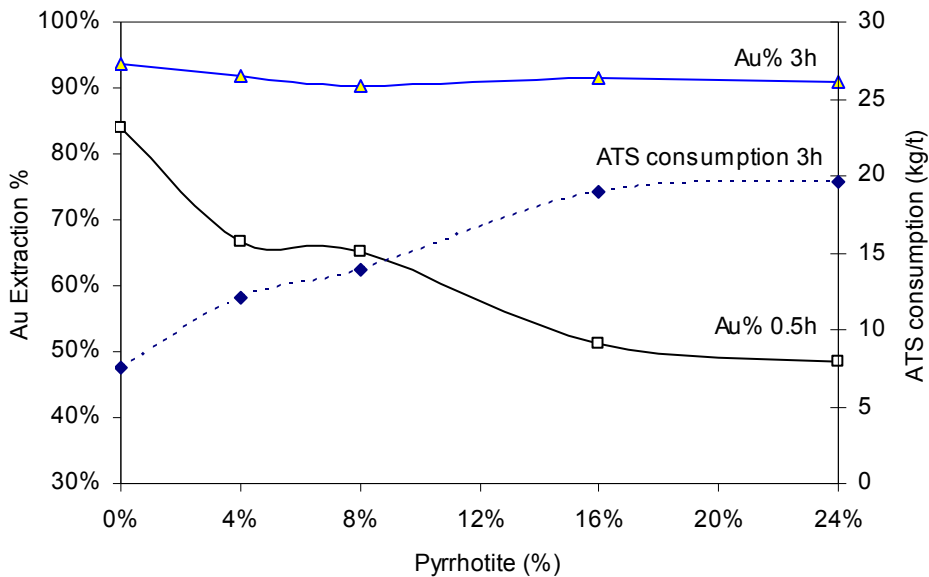
or precipitation of copper onto the surface of pyrite. This observation verified the co-precipitation theory stated in section 3.1.3. At 0.5 hours and 3.0 hours, the concentrations of dissolved copper were 0.83 and 0.90 mM respectively. The restoration of copper concentration at the end of leach was an indication of oxidation of pyrite surface and re-dissolution of precipitated copper.

The percentage of sulfide oxidation was analyzed for a few leaching tests. The oxidation percentage was estimated based on the changes of sulphur compositions before and after leaching. As shown in Table 5-2, the degree of oxidation of pyrite is significantly different under various solution conditions. In the presence of 12.0% pyrite, the leaching test with 3.6 mM copper oxidized 5.68% sulfide within 3.0 hours. The increase of copper concentration to 12 mM resulted in a 7.05% oxidation, which proved that higher copper concentration accelerated the oxidation of pyrite. Under limited oxygen supply (i.e., 0.7% oxygen), the oxidation of sulfide is reduced to only 4.90%. Such a oxidation level indicates that the surface oxidation of pyrite was not significantly inhibited at such a low oxygen percentage. The addition of potassium amyl xanthate (PAX) inhibited the oxidation of sulfides and lead to the lowest oxidation percentage, 2.97%. At higher ammonia concentration (i.e., 3.0 M), the oxidation of pyrite was increased to 6.95%.

**Table 5-2: Pyrite oxidation % (at 3.0 hours)**

Pyrite%	AMTS (mol/L)	Cu (mM)	AM (mol/L)	PAX (mg/L)	DO (ppm)	Sulfide oxidation %
12.0	0.2	3.6	3.0	0	21.1%	6.95
12.0	0.2	12.0	0.9	0	21.1%	7.05
12.0	0.2	3.6	0.9	0	21.1%	5.68
12.0	0.2	3.6	0.9	20	21.1%	2.97
12.0	0.2	3.6	0.9	0	0.7%	4.90

This result demonstrates that increased ammonia concentration and copper concentrations resulted in a higher level of oxidation. The presence of the surfactant (i.e., PAX) and limited oxygen supply resulted in a reduced oxidation of pyrite. As discussed in Section 3.1.3, the oxidation of the sulfide surface is beneficial to the leaching of gold for two reasons. Firstly, the oxidation of the sulfide surface increases the slurry potential and benefits the oxidation of gold. Secondly, the oxidation of sulfide surface promotes the passivation of the reactive surfaces of these minerals, which prevents further interference by these minerals on the dissolution of gold.

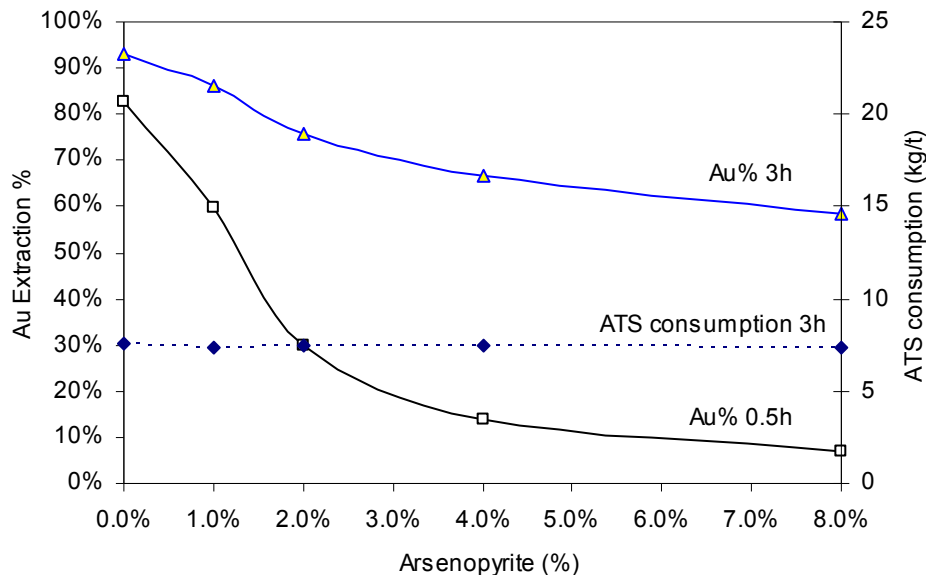


**Figure 5-7: Effect of pyrrhotite on gold extraction and thiosulfate consumption**

**(ATS: 0.20 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

Figure 5-7 demonstrates the leaching results at various percentage of pyrrhotite. As expected, the addition of pyrrhotite was also detrimental to both gold extractions and thiosulfate stability. By adding 4.0% pyrrhotite, 4.6 kg/t more ATS (12.1 vs. 7.5 kg/t) was consumed and the gold extraction was reduced by 17.2% (from 83.9% to 66.7%) at 0.5 hour and 1.8% (from 93.6% to 91.8%) at 3.0 hours. When the pyrrhotite percentage was increased to 24%, the consumption of ATS increased to 19.6 kg/t with 90.8% gold leached.

In the leaching test with 4.0% pyrrhotite, the initial copper concentration was only 0.4 mM. The decrease of copper concentration, again, indicated an adsorption or precipitation of copper onto the pyrrhotite surface. At 0.5 and 3.0 h, the concentrations of copper were restored to 1.1 mM and 1.2 mM respectively. This is believed to be an indication of surface oxidation of pyrrhotite. Such a change of copper concentration can be explained by the mechanism of copper co-precipitation on iron sulfide surface (see Equation 4-27).

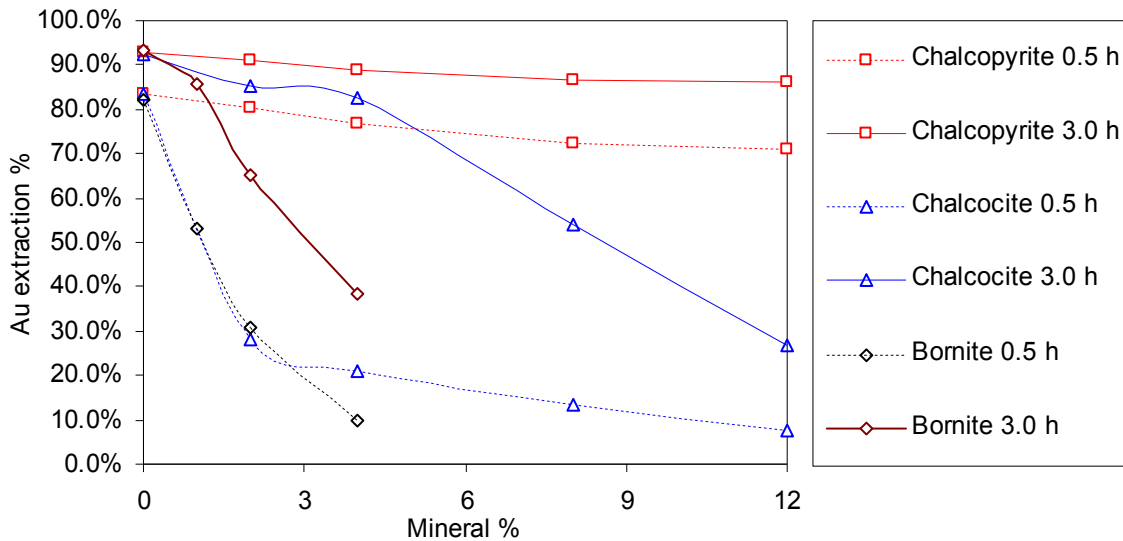


**Figure 5-8: Effect of arsenopyrite on gold extraction and thiosulfate consumption (ATS 0.2 M, Cu 1.2 mM, AH 0.9 M, oxygen: 21.1%)**

The detrimental effect of arsenopyrite was much more significant than that of either pyrite or pyrrhotite. As shown in Figure 5-8, by adding 1.0% arsenopyrite the gold extraction at 0.5 hour was reduced by 23.0% (from 86.2% to 59.6%). The overall gold extraction decreased from 93.2% to 86.2%. By adding more arsenopyrite, e.g., 8.0%, the gold extraction at 0.5 hour was reduced to 7.0% and the final gold extraction was reduced to 58.3%. However, the addition of this mineral had almost no impact on the ATS stability. For all tests, the thiosulfate consumptions remained at about 7.4-7.6 kg/t. This indicated that this arsenopyrite sample did not catalyze the oxidation of thiosulfate. The reason why arsenopyrite shows such a different trend in the catalysis of thiosulfate oxidation is still unknown.

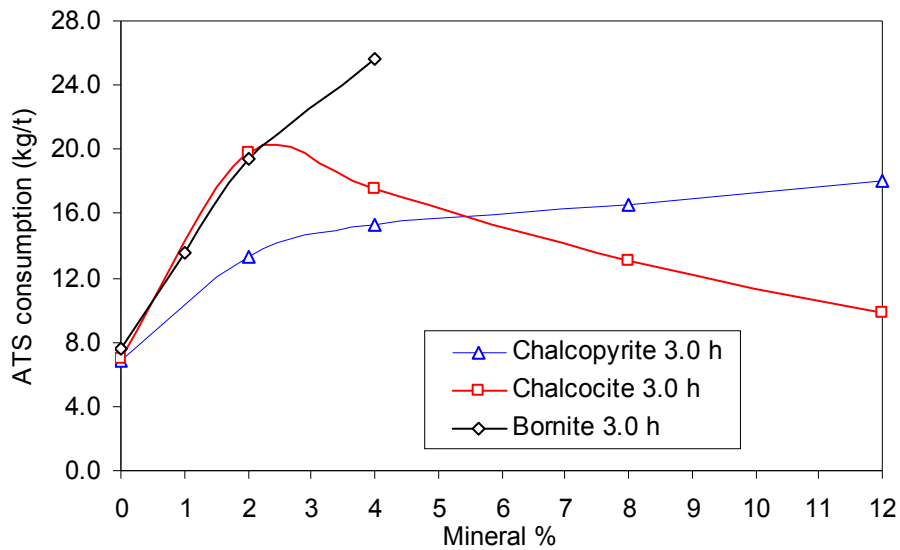
### 5.1.3. Effects of chalcopyrite, chalcocite and bornite

Three common copper sulfide minerals associated with gold were investigated in this study, i.e., chalcopyrite, chalcocite and bornite.



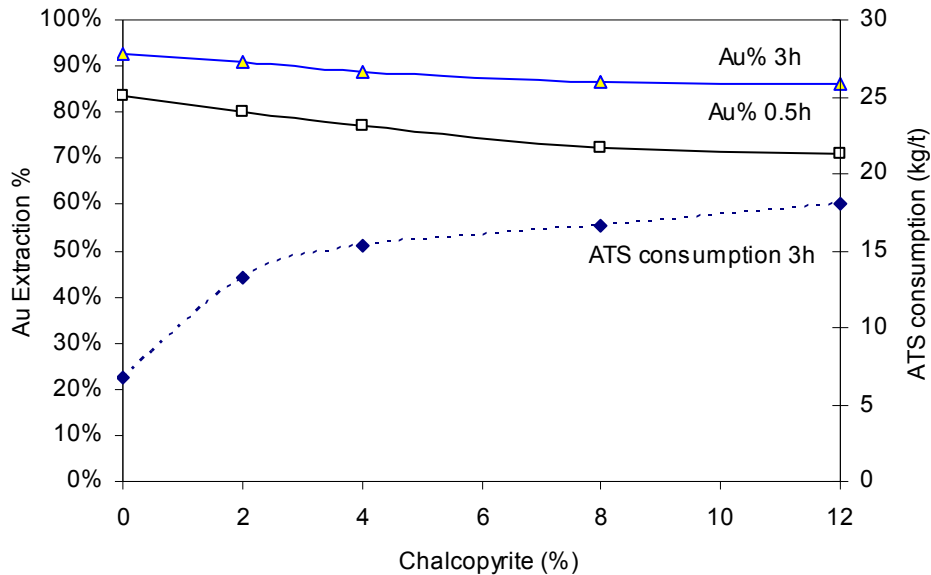
**Figure 5-9: Effects of chalcocite, chalcopyrite and bornite on gold extractions (ATS: 0.20 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

In the presence of chalcopyrite, chalcocite and bornite, the gold extractions are compared in Figure 5-9. The dashed curves represent the gold extractions at 0.5 hour. Among these three sulfides, the detrimental effects have the order of bornite > chalcocite >> chalcopyrite. At 3.0 hours of leach, the negative effects of chalcopyrite and chalcocite became less significant than they were at 0.5 hour of leach.



**Figure 5-10: Effects of chalcocite, chalcopyrite and bornite on thiosulfate consumption (ATS: 0.20 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

Figure 5-10 compares the effects of three copper minerals on thiosulfate consumptions. All three minerals increased the consumption of ATS. The highest thiosulfate consumption was achieved in the leaching of the bornite bearing ore. The consumption was tripled by adding 4.0% of bornite. By adding 8.0% of chalcopyrite, the ATS consumption was doubled. The impact of chalcocite on thiosulfate stability was significant within the range of 0.0 to 2.0%. By adding more chalcocite, ATS consumption started to decrease. Such a drop of thiosulfate consumption might indicate the conversion of sulfide from the leached surface to thiosulfate anion.

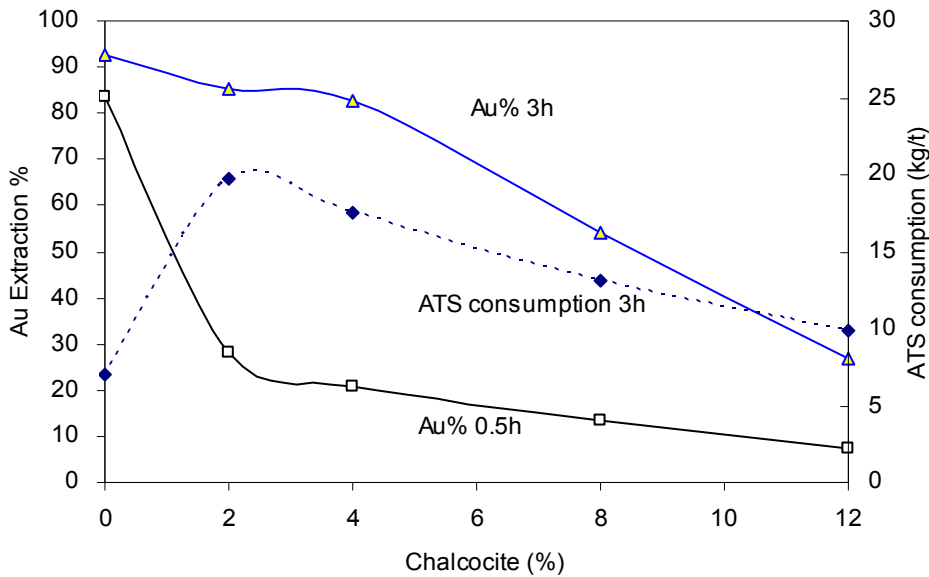


**Figure 5-11: Effect of chalcopyrite on gold extraction and thiosulfate consumption (ATS: 0.20 M, AH: 0.9 M, Cu: 1.2 mM, oxygen: 21.1%)**

After a comparison of these three minerals, the effects of each individual mineral will be discussed in detail. Figure 5-11 shows the effect of chalcopyrite at various percentages. By adding 12% of chalcopyrite, the gold extractions were reduced from 83.6% to 71.1% at 0.5 hour and from 92.9% to 86.0% at 3.0 hours of leach. The ATS consumption increased from 6.8 to 18.1 kg/t by adding 12% of chalcopyrite.

In the leaching tests with 4.0% chalcopyrite, the copper concentration in thiosulfate leaching solution was determined. Surprisingly, it was found that the copper concentration was reduced from 1.2 mM to 0.6 mM immediately after the contact of the chalcopyrite with the leaching solution. This phenomenon indicated that copper was adsorbed or precipitated onto the surface of chalcopyrite. At 0.5 and 3.0 hours, the amount of copper dissolved from chalcopyrite was 0.2 mM

and 0.4 mM respectively. This result demonstrated that the dissolution of copper from chalcopyrite was very limited. Again, the surface passivation theory discussed in section 3.1.3 can be used to explain the observations in this section. Due to the formation of FeOOH on the surface of chalcopyrite, the dissolution of copper and the interference on gold leaching were minimized

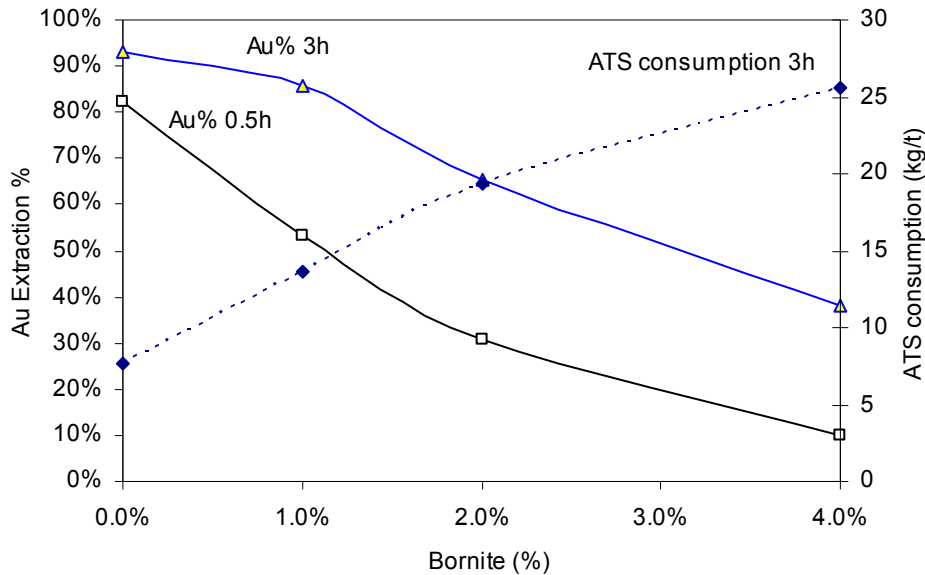


**Figure 5-12: Effect of chalcocite on gold extraction and thiosulfate consumption (ATS: 0.20 M, Cu 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

The effect of chalcocite at various concentrations is demonstrated in Figure 5-12. In the presence of 2% chalcocite, the gold extraction at 0.5 hour was largely reduced from 83.3% to 28.3%. The gold extraction at 3.0 hours was also reduced from 92.5% to 85.2%. The leaching performance became worse in presence of more chalcocite. As discussed in section 3.1.3, the highly reactive sulfide anion on the leached surface of chalcocite is responsible for the reduced gold extraction.

By adding 2.0% chalcocite, the ATS consumption was increased from 7.1 to 19.8 kg/t. When the addition of chalcocite was more than 2%, the ATS consumption started to decrease. With 12% chalcocite, the consumption of ATS was reduced to 9.9 kg/t. The decrease of ATS consumption indicated two possible mechanisms (as stated in section 3.1.3): firstly, the very reductive surface of leached chalcocite consumed most of the oxidants and resulted in a reduced oxidation rate of ATS; secondly, the sulfide anion on the leached surface of chalcocite was oxidized to produce thiosulfate anion.

In the leaching test with 4% chalcocite, the concentration of copper and slurry potential was examined. The initial copper concentration was 1.6 mM, which indicated that copper was leached immediately after the start of the leaching process. At 0.5 hour, the copper concentration went up to 34.3 mM. At 3.0 hours, 49.1 mM of copper was found in solution. Obviously, the dissolution of chalcocite was easy and fast. The initial slurry potential was -129 mV. It became more negative, i.e., -231 mV at 0.5 hour and -239 mV at 3.0 hours. These observations verified that chalcocite is highly dissolvable and the dissolution of copper was not effectively inhibited by surface passivation.



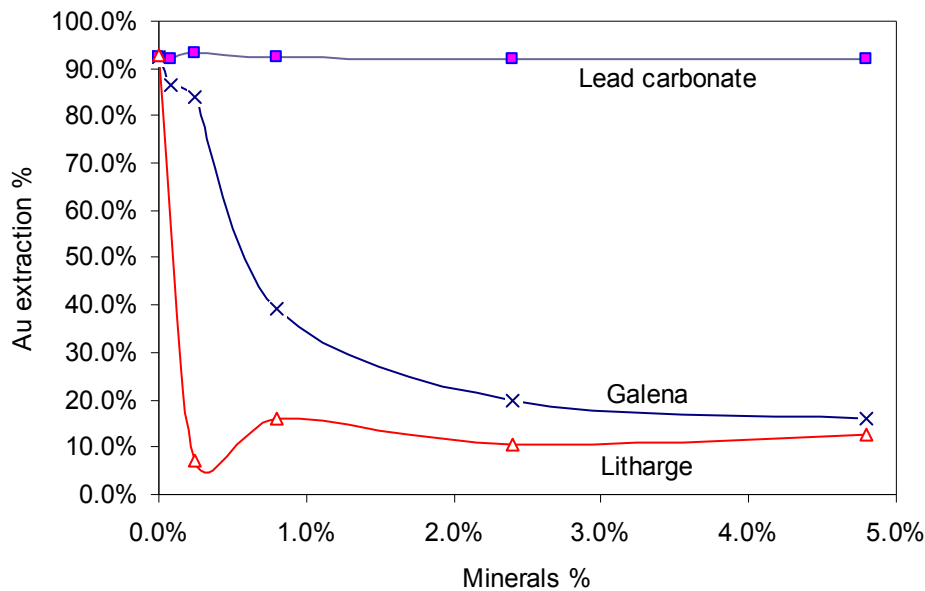
**Figure 5-13: Effect of bornite on gold extraction and thiosulfate consumption (ATS: 0.2 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

The effect of bornite is shown in Figure 5-13, by adding 4.0% bornite, the gold extraction at 0.5 hours was reduced from 82.3% to 10.0%. The overall gold extraction also reduced from 93.3% to 38.3%. The consumption of ATS increased from 7.6 kg/t to 25.6 kg/t. The copper concentration in the test with 2.0% bornite was monitored. The initial Cu concentration was 1.5 mM and went up to 34.6 mM at 0.5 hours and the final copper concentration reached 47.3 mM.

The detrimental effect of bornite was the worst among the three copper minerals. By comparing the gold extractions at 0.5 hour and 3.0 hours, it was found that the detrimental effect of bornite did not diminish with time.

#### 5.1.4. Effects of lead minerals

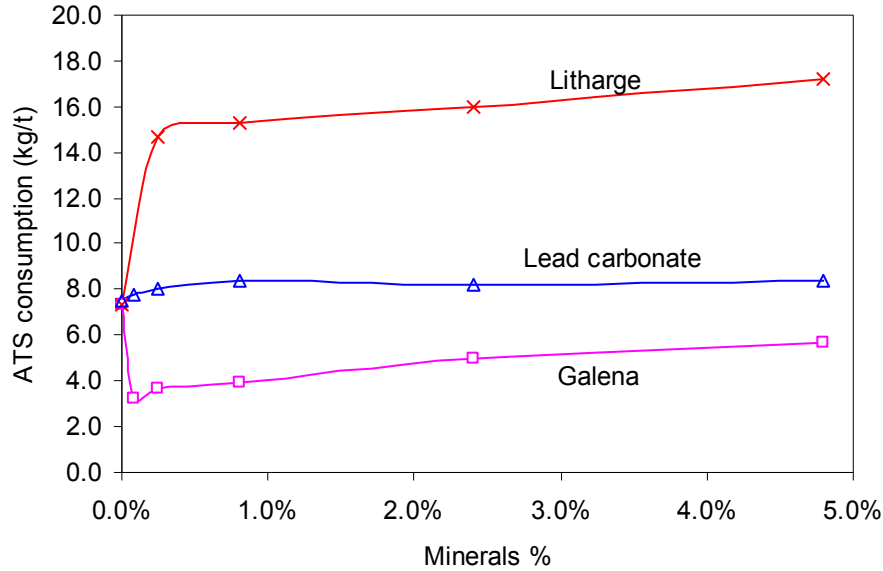
In this research, both lead minerals and salts were studied for their effects on gold leaching with thiosulfate. They were: lead nitrate, galena, litharge and lead carbonate. The Eh-pH study of lead species in thiosulfate leaching indicated that lead is dissolvable in thiosulfate solution. Lead may consume free thiosulfate anion in the dissolution procedure and affect the gold extraction. More importantly, lead can be precipitated again. The significance of the detrimental effect of lead, however, must be answered with leaching result shown as follows.



**Figure 5-14: Effects of litharge, galena and lead carbonate on gold extractions (ATS: 0.20 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

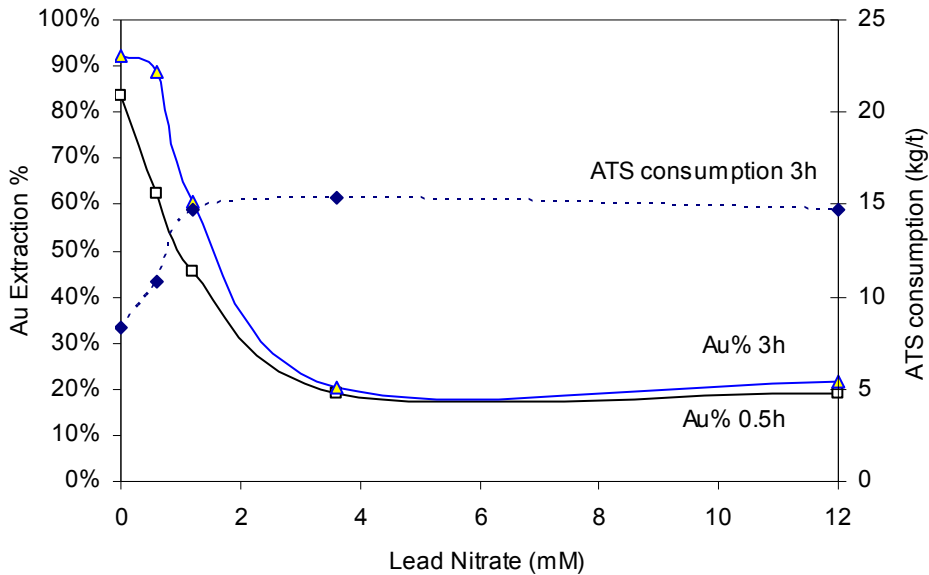
By comparing the impacts of the three lead minerals, i.e., galena, litharge and lead carbonate (Figure 5-14), it was found that the detrimental effects had the order of litharge > galena >> lead carbonate. The presence of lead carbonate

had almost no impact of gold extraction. While, the other two lead minerals were found to be extremely detrimental.



**Figure 5-15: Effects of litharge, galena and lead carbonate on thiosulfate consumptions (ATS: 0.20 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

The effects of the lead minerals on ATS consumption are compared in Figure 5-15. Lead carbonate had very limited influence on the stability of thiosulfate. The presence of galena reduced the consumption of ATS by about 3 kg/t. While the ATS consumption was largely increased to about 15 kg/t by adding litharge.

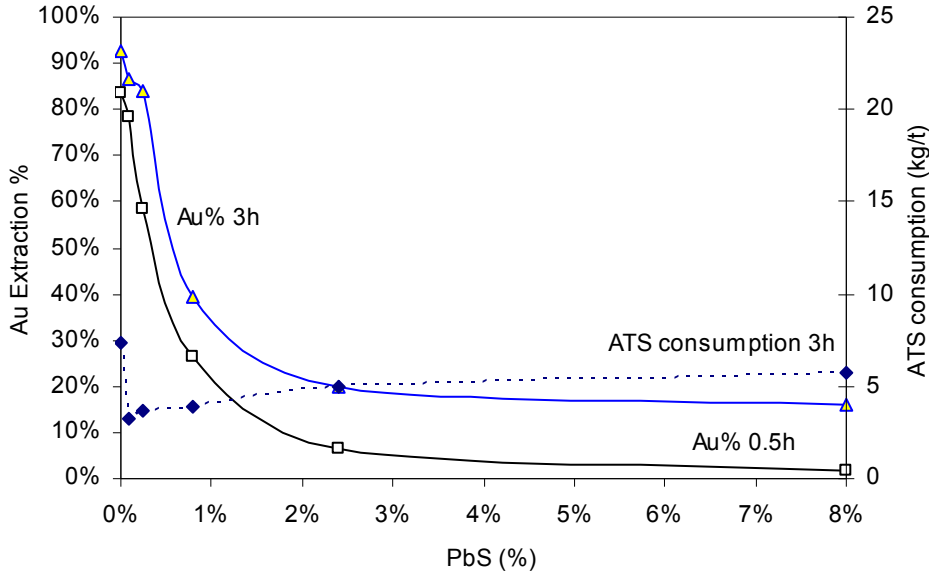


**Figure 5-16: Effect of lead nitrate on gold extraction and thiosulfate consumption (Cu: 1.2 mM, ATS: 0.20 M, AH: 0.9 M, oxygen: 21.1%)**

The effect of each individual lead mineral or salts are discussed in details as follows. The effect of lead nitrate in thiosulfate leaching solution is shown in Figure 5-16. Adding 0.6 mM lead nitrate reduced the gold extraction by 21.1% (at 0.5 hour) and by 3.3% (at 3.0 hours). As the amount of lead nitrate increased to 3.6 mM, the gold extractions at both 0.5 hour and 3.0 hours were reduced to about 20%. By adding 1.2 mM lead nitrate, the ATS consumption was increased from 8.3 kg/t to 14.7 kg/t. The consumption of ATS remained steady at lead nitrate >1.2 mM.

Figure 5-16 also shows that in the presence of more than 3.6 mM lead nitrate, the gold extraction was completely slowed down from 0.5 hour to 3.0 hours. This observation verified that lead precipitate occurred on the surface of gold so that

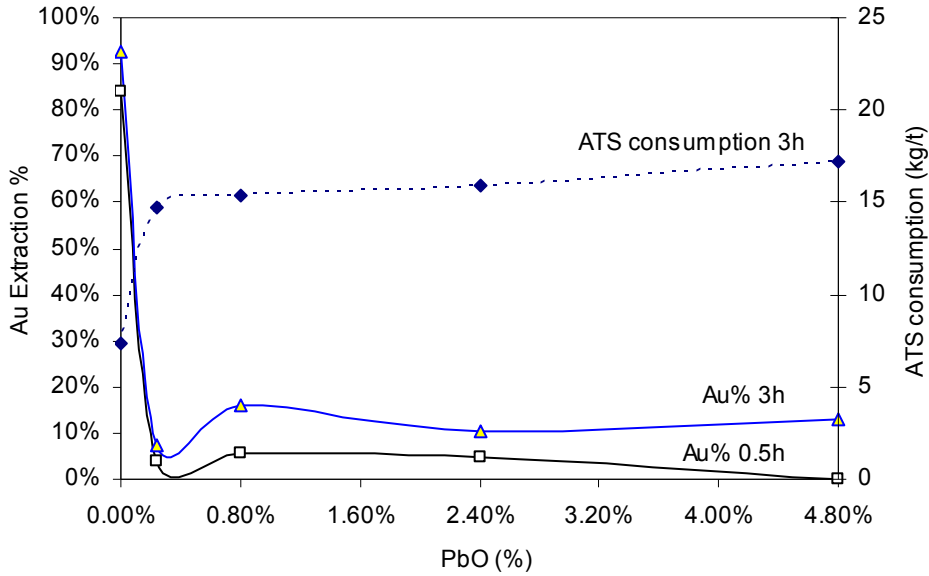
the contact of gold with the leaching solution was largely inhibited by this precipitate.



**Figure 5-17: Effect of galena on gold extraction and thiosulfate consumption (Cu: 1.2 mM, ATS: 0.20 M, AH: 0.9 M, oxygen: 21.1%)**

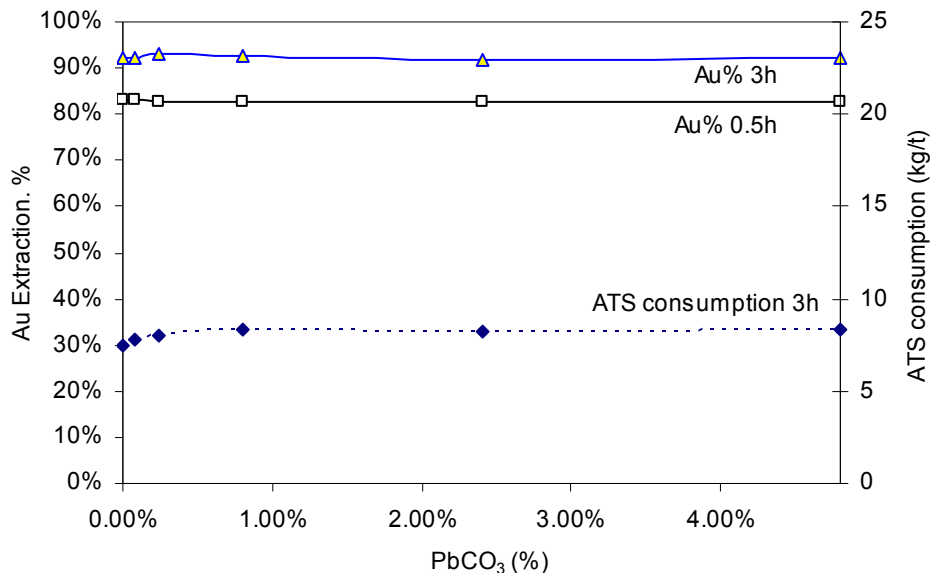
In natural complex sulfidic gold ores, the most common associated lead mineral is galena. The effect of galena at various percentages in thiosulfate leaching solution is shown in Figure 5-17. By adding 0.08% galena, the gold extraction (3.0 hours) was reduced from 92.5% to 86.7%. It was reduced by more than 50% when the galena composition was 0.80%. The consumption of ATS decreased from 7.3 to 3.2 kg/t by adding 0.08% galena. By further increases of galena addition to 8.00% slightly increased the consumption from 3.2 to 5.7 kg/t. The dissolution of lead was studied in the leaching test of galena. In the presence of 0.8% galena, the concentration of lead has increased to 0.60 mM in 30 minutes. By increasing the amount of galena or the extension of the leaching time, the

concentration of lead remained in the range of about 0.84 mM to 1.08 mM. Meanwhile, a white semitransparent film that formed on quartz particles was observed, which indicated the formation of lead depositions.



**Figure 5-18: Effect of litharge on gold extraction and thiosulfate consumption (Cu: 1.2 mM, ATS: 0.20 M, AH: 0.9 M, oxygen: 21.1%)**

Figure 5-18 displays the detrimental effects of litharge. By adding 0.24% to 4.80% litharge, gold extractions at 0.5 hour and 3.0 hours of leach were both reduced to about 10%. The consumption of ATS increased by about 7 and 10 kg/t respectively in the presence of 0.24% to 4.80% litharge.



**Figure 5-19: Effect of lead carbonate on gold extraction and thiosulfate consumption (Cu: 1.2 mM, ATS: 0.2 M, AH: 0.9 M, oxygen: 21.1%)**

Figure 5-19 demonstrates the gold extraction and thiosulfate consumption in the presence of various amounts of lead carbonate. Lead carbonate had no impact on the leaching performance over the range from 0.00% to 4.00%. The gold extractions were about 83% after 0.5 hour and about 92% at 3.0 hours. The consumption of ATS was between 7.5 to 8.4 kg/t. In the leach solution, only 0.05 mM lead was detected.

To understand the behaviour of the lead species in thiosulfate leaching, a few precipitation tests were conducted. In the first test, a solution containing 0.20 M ATS, 0.9 M AH and 1.2 mM copper was prepared, 0.3 mM lead nitrate was added into the thiosulfate solution as pre-dissolved solution. A white precipitate was observed immediately, which disappeared in about 10 seconds of agitation. With excessive addition of lead nitrate, i.e., 3.0 mM, the precipitate persisted in

the solution phase. According to the thermodynamic analysis conducted in previous sections, the temporary precipitate can be one of the followings: lead hydroxide or lead thiosulfate.

In the second test, 0.3 mM or 3.0 mM lead nitrate was added in a solution containing 0.9 M AH only. No precipitation was found in both cases. This observation indicated that the formation of lead precipitate required thiosulfate.

In the third test, an ATS solution at 0.20 M was prepared at pH 6.8. The addition of lead nitrate at 0.3 mM or 3.0 mM did not produce any precipitation, which indicated that the lead precipitate only forms at high pH and it is possibly  $\text{Pb}(\text{OH})_2$ .

In the fourth test, a sodium thiosulfate solution at 0.20 M was prepared and the pH was adjusted to 10.5 by adding NaOH. By adding 0.3 mM lead nitrate, precipitate was formed instantly, which was similar to that formed in the first test. This test indicated that the formation of both the temporary and the persisting precipitate has no relation to the ammonia or the ammonium anion. The relevant species that are responsible for the precipitations are thiosulfate and hydroxide anions.

In the fifth test, the ATS concentration was increased to 1.00 M, while the other conditions were kept the same as in the first test. No precipitation was found by adding 0.3 mM or 6 mM lead nitrate. Therefore, it can be concluded that the precipitate of lead can be readily dissolved by excessive thiosulfate. Through

these tests, it can be deduced that the precipitate of lead is possibly  $\text{Pb}(\text{OH})_2$ . However the formation of this precipitate requires thiosulfate anion. However, excessive thiosulfate anion may prevent the formation of this precipitate.

In order to confirm this observation, and identify the lead precipitate that is the most responsible for the passivation of gold, XPS (X-ray photoelectron spectroscopy) analysis was used to characterize the surface chemistry. A piece of gold foil was rinsed briefly in acetone, concentrated nitric acid and 2.0 M NaOH solutions respectively; and washed with deionized water. Then it was immersed in slurry containing quartz flour at 33.3% solid. The leaching conditions were: 0.20 M ATS, 0.9 M AH, 1.2 mM copper and 0.8% galena. After 1.0 hour, the gold foil was removed from the slurry, rinsed with water and analyzed by XPS (see appendix). It was found that the leached surface of gold had a coating of  $\text{Pb}(\text{OH})_2$  with a small amounts of S and Si.

#### **5.1.5. Summary**

In section 3.1.3, the thermodynamic analysis of thiosulfate leaching in the presence of sulfide minerals indicated that the reductive nature of sulfide surfaces is responsible for slowing down the gold extraction. In this section, the leaching results confirmed this conclusion. All sulfide minerals studied, i.e. pyrite, pyrrhotite, chalcocite, bornite and arsenopyrite significantly reduced the gold extractions. Among these minerals, chalcocite, bornite and arsenopyrite were the most detrimental minerals.

In the presence of pyrite, pyrrhotite, chalcopyrite and arsenopyrite, the detrimental effects were much milder at 3.0 hours of leach, indicating that with the oxidation of sulfide surface, passivation occurred and made the surface less reactive. Surface passivation also largely prevented the dissolution of copper from chalcopyrite.

The passivation of chalcocite and bornite was found ineffective based on the observation of continuous copper dissolutions. In the thermodynamic considerations, it is suggested that the lack of sufficient iron in the lattice of these minerals is the reason.

As discussed in the thermodynamic analysis, copper could be precipitated as copper sulfide or co-precipitated with iron hydroxide. Both precipitates are re-dissolvable under oxidative conditions. The results in this section support the thermodynamic analysis result. The copper concentrations were found reduced in the beginning of leach in the presence of pyrite, pyrrhotite, arsenopyrite and chalcopyrite. The copper concentrations were completely or partially restored in the later stage of leach.

As suggested by the thermodynamic analysis, lead may be dissolved and then precipitated on the gold surface and passivate the metal extraction. The leaching result confirmed the theories. The gold extractions in the presence of lead minerals or salts was significantly suppressed. It was also found that the inert film on the gold surface was lead hydroxide.

As for thiosulfate consumption, it was found that pyrite, pyrrhotite, and chalcopyrite significantly accelerated the decomposition of thiosulfate and increased the consumption. These results verified the surface catalysis theory cited in the previous section.

Chalcocite and bornite also resulted in higher consumption of ATS. However, for these minerals, the increased copper concentration due to their dissolution is a more rational explanation for the accelerated decomposition of thiosulfate.

It was suggested that the leached surface of sulfide may also produce thiosulfate or compete with thiosulfate in the consumption of oxidant. As a result, the thiosulfate consumption is reduced. In the leaching results, it was found that by adding galena, the consumption of thiosulfate was reduced. A possible explanation is that galena does not have the surface catalysts function as that of pyrite. Once lead was leached from the surface of galena; the reactive sulfide anion on the leached surface oxidizes and produces thiosulfate. Therefore, the overall consumption of thiosulfate is reduced. A similar reduction of thiosulfate consumption was also found in the leaching of chalcocite, when more than 2.0% chalcocite was added.

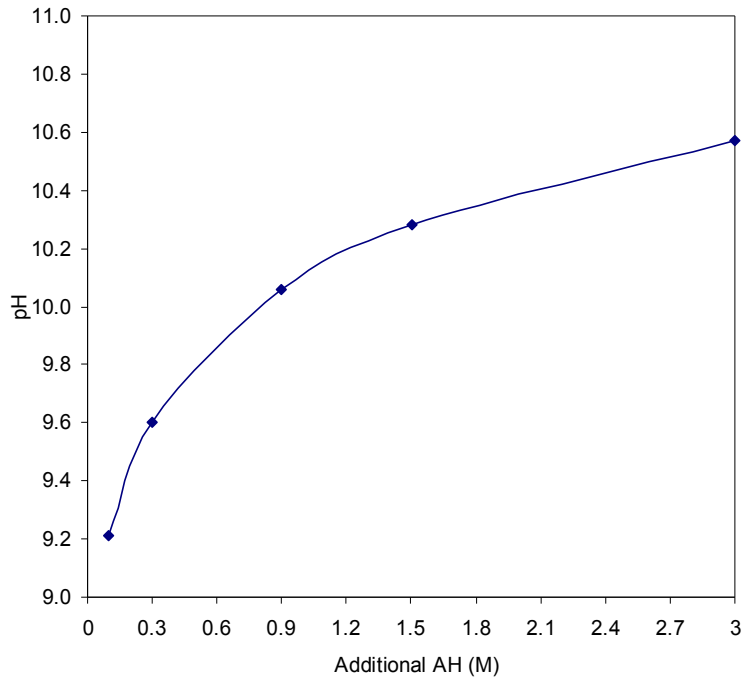
In the presence of arsenopyrite, thiosulfate was found intact, which either means that the surface of arsenopyrite has no catalysis property as pyrite does or the surface of arsenopyrite is rapidly passivated; the catalysis property is masked thereafter.

## **5.2. Optimization of leaching conditions**

As discussed in section 3.2.1, the concentrations of ammonia, copper and thiosulfate were the basic variables in thiosulfate leaching. Therefore, their effects on the leaching performance must be studied. And the result shall be used to verify the analysis made in that section.

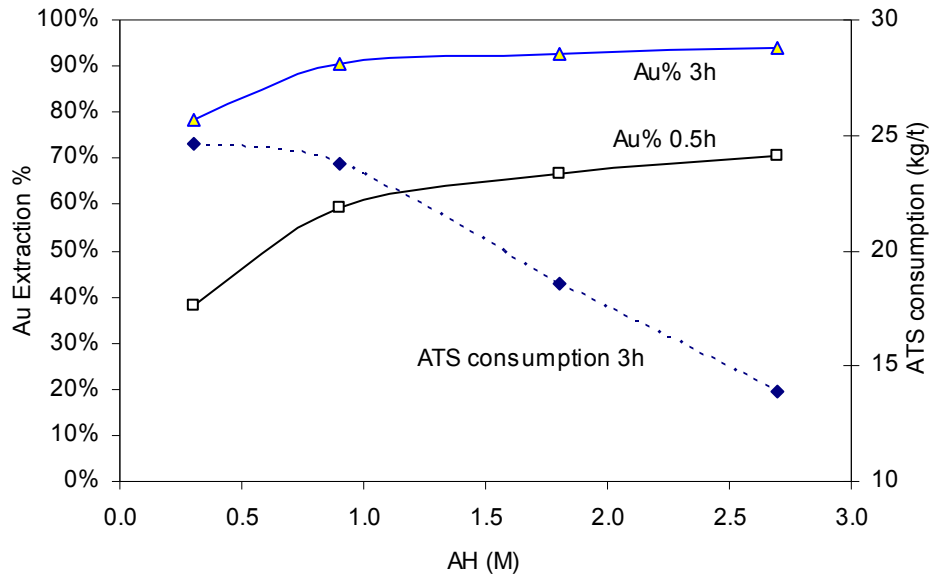
### **5.2.1. Ammonia concentration**

As discussed in section 3.1.1, ammonia concentration is one of the most important variables in thiosulfate leaching system. First of all, the concentration of ammonia affects the pH values, which will also change the thermodynamic status of the reaction. Secondly, the ammonia forms auroous and copper complexes, which play important roles in the dissolution of gold. In addition, the ammonia may have more functions on the surface of gold as a leaching catalyst.

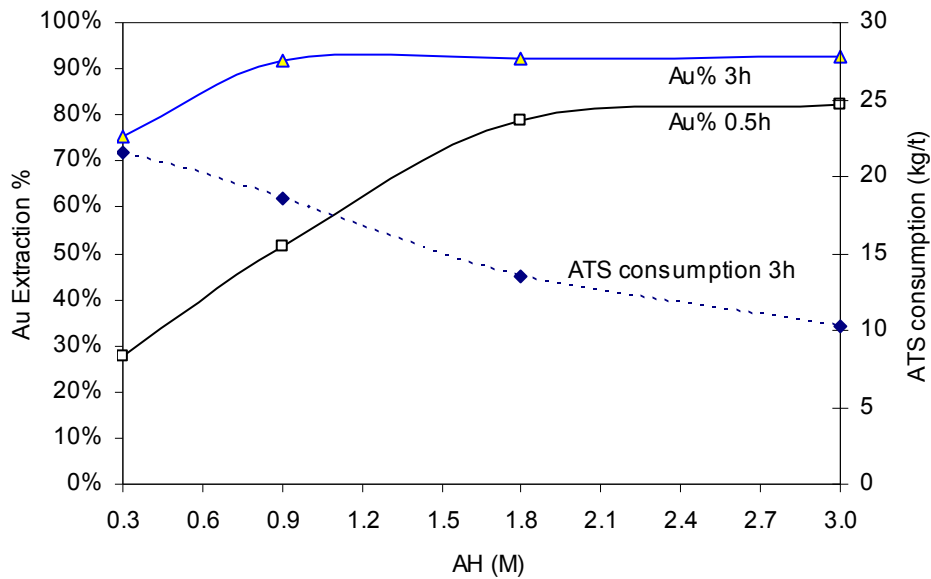


**Figure 5-20: Effects of the additional ammonium hydroxide on slurry pH (ATS: 0.2 M, Cu: 1.2 mM)**

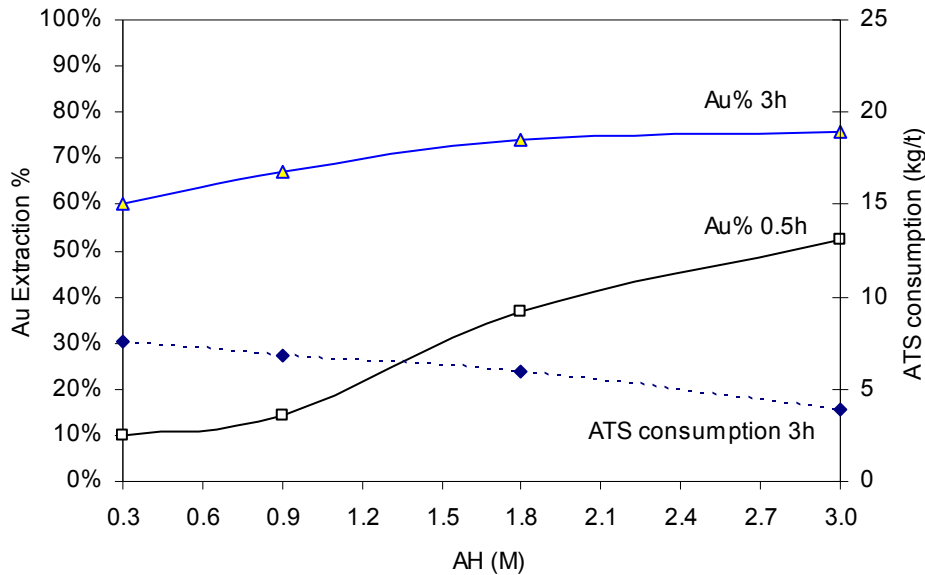
Figure 5-20 shows the relationship between the ammonia concentration and the slurry pH. The additional ammonium hydroxide means that the ammonium contents were added as ammonia other than ATS. Obviously, the pH value increased with the concentration of ammonia hydroxide. Increasing ammonia concentration shifts the oxidation potential of gold and thiosulfate in the negative direction and makes them easier to be oxidized (section 3.1.2).



**Figure 5-21: Effect of ammonia on gold extraction and thiosulfate consumption in the presence of 16.0% pyrite**  
**(Cu: 3.6 mM, ATS: 0.20 M, oxygen: 21.1%)**



**Figure 5-22: Effect of ammonia on gold extraction and thiosulfate consumption in the presence of 16.0% pyrrhotite**  
**(Cu: 1.2 mM, ATS: 0.20 M, oxygen: 21.1%)**

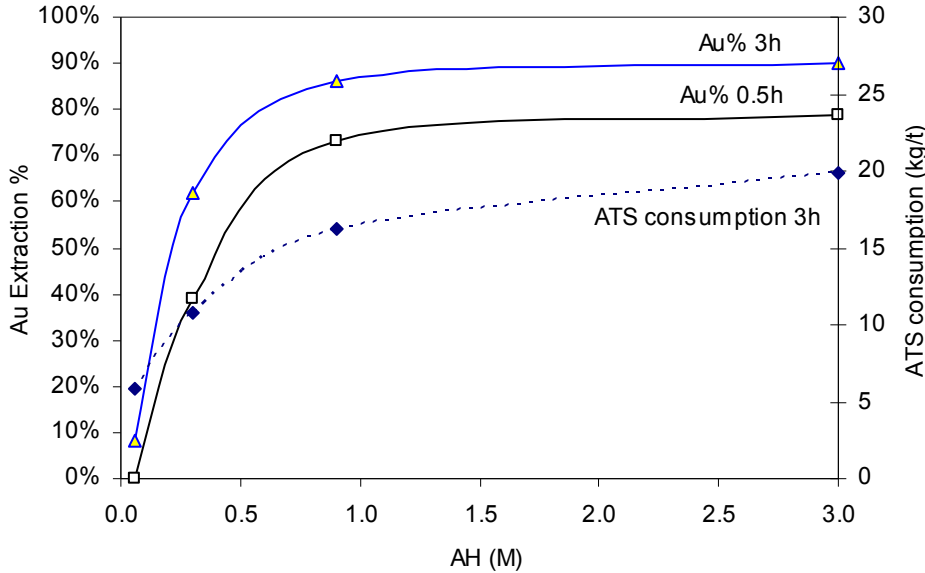


**Figure 5-23: Effect of ammonia on gold extraction and thiosulfate consumption in the presence of 4.0% arsenopyrite (ATS: 0.20 M, Cu: 1.2 mM, oxygen: 21.1%)**

Figures 5-21, 5-22 and 5-23 demonstrate the effect of ammonia concentration on gold extraction and thiosulfate consumption in the presence of 16% pyrite, 16% pyrrhotite, and 4% arsenopyrite respectively. Ammonia appeared to be beneficial to both gold leaching kinetics and thiosulfate stability. As shown in Figure 4-21, in the presence of 16% pyrite, the increase in the ammonia concentration from 0.3 to 2.7 M significantly reduced the consumption of ATS by 10.7 kg/t. Faster leaching kinetics were also obtained with higher ammonia concentrations. At 0.5 hour, the gold extraction of pyrite-bearing ore increased from 38.2% to 70.7% when the concentration of ammonia was increased from 0.3 to 3.0 M.

Similar trends were also observed in the leaching of the pyrrhotite and arsenopyrite bearing gold ores. As shown in Figures 5-22 and 5-23, higher

ammonia concentration improved both the gold extractions and the thiosulfate stabilities in these tests.

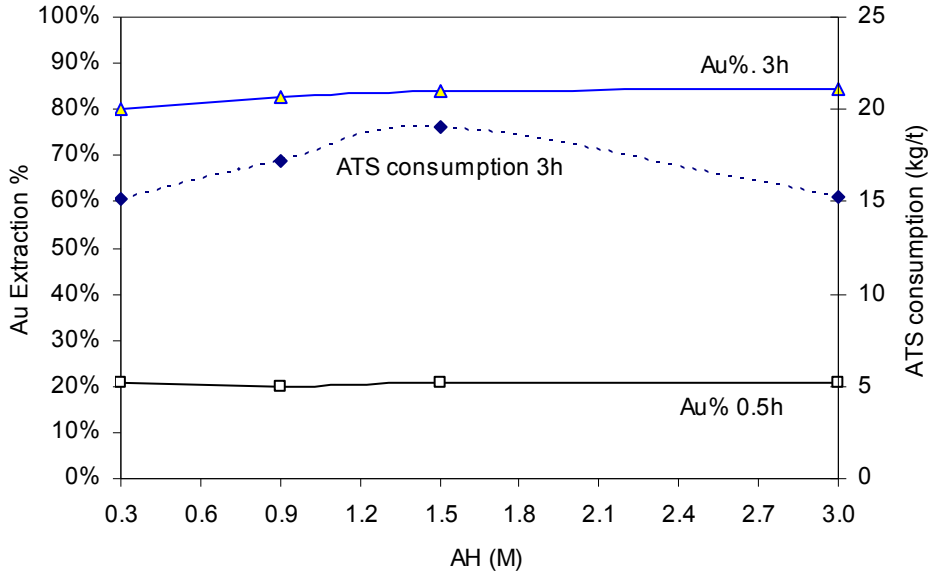


**Figure 5-24: Effect of ammonia on gold extraction and thiosulfate consumption in the presence of 8.0% chalcopyrite (ATS: 0.20 M, Cu: 1.2 mM, oxygen: 21.1%)**

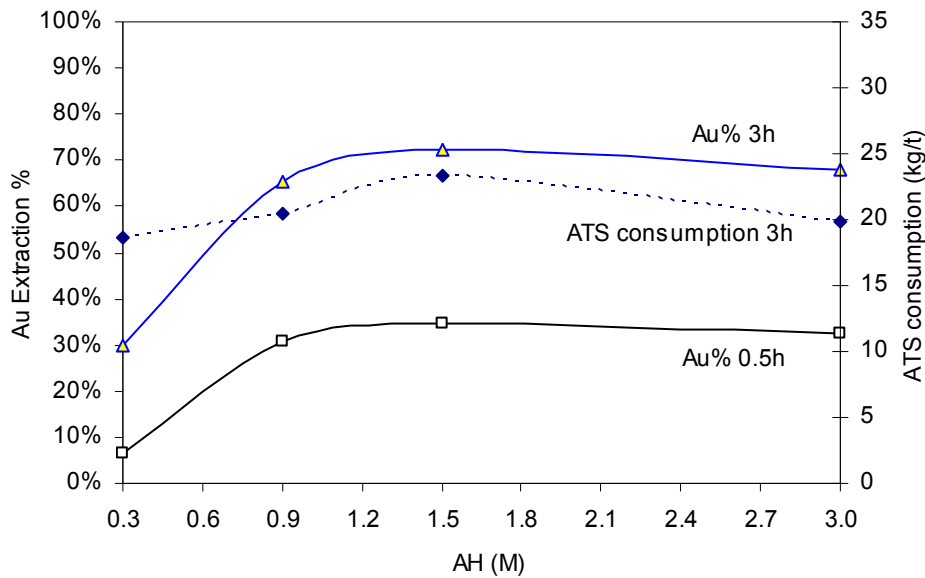
Figure 5-24 demonstrates the effect of ammonia concentration in the leaching of the chalcopyrite bearing ore. By increasing the concentration of ammonia from 0.06 M to 3.00 M, the gold extractions were increased from 0.0% up to 79.0% at 0.5 hour and from 8.1% up to 90.1% at 3.0 hours. Meanwhile, the consumption of thiosulfate was increased from 5.9 to 19.8 kg/t.

In the test with 3.0 M ammonia, it was found that about 1.8 mM copper was dissolved from the chalcopyrite. The dissolved copper was responsible for the accelerated consumption of thiosulfate. However, compared to chalcocite and

bornite, the dissolution of copper from chalcopyrite was minimal even at higher ammonia concentration.



**Figure 5-25: Effect of ammonia on gold extraction and thiosulfate consumption in the presence of 4.0% chalcocite (ATS: 0.20 M, Cu: 1.2 mM, oxygen 21.1%)**



**Figure 5-26: Effect of ammonia on gold extraction and thiosulfate consumption in the presence of 2.0% bornite**

**(Cu: 1.2 mM, ATS: 0.2 M, oxygen: 21.1%)**

Figures 5-25 and 5-26 show the effects of ammonia on gold extraction and thiosulfate consumption in the presence of chalcocite and bornite. As depicted in Figure 5-25 (chalcocite), within the range of 0.3 to 3.0M, the gold extractions were about 20% for all tests. At 3.0 hours, the gold extractions were increased from 80.2% to 84.2%. The higher the ammonia concentration, the higher the gold extractions. Increasing ammonia concentration from 0.3 to 1.5 M increased the ATS consumption by 4.0 kg/t. However, when the ammonia concentration was over 1.5 M, the consumption of ATS started to decrease.

For bornite, within the range of 0.3 to 3.0 M, the increase of ammonia concentration also improved the gold extraction (Figure 5-26). The gold extraction at 0.5 hour increased from 6.5% to 32.6%. The overall gold extraction was also improved from 30.0% to 68.2%. The ATS consumption at 0.3 M ammonia was 18.7 kg/t, which was increased to 23.3 kg/t when the ammonia concentration went up to 1.5 M. Further increasing the ammonia concentration resulted in a reduction of ATS consumption.

In the presence of galena, due to the passivation of the gold surface, the effects of ammonia will be less significant. As shown in Table 5-3, by increasing the ammonia concentration from 0.9 to 3.0, the gold extraction at 0.5 hour was slightly improved from 6.5% to 17.6%. But the overall gold extraction at 3.0 hours

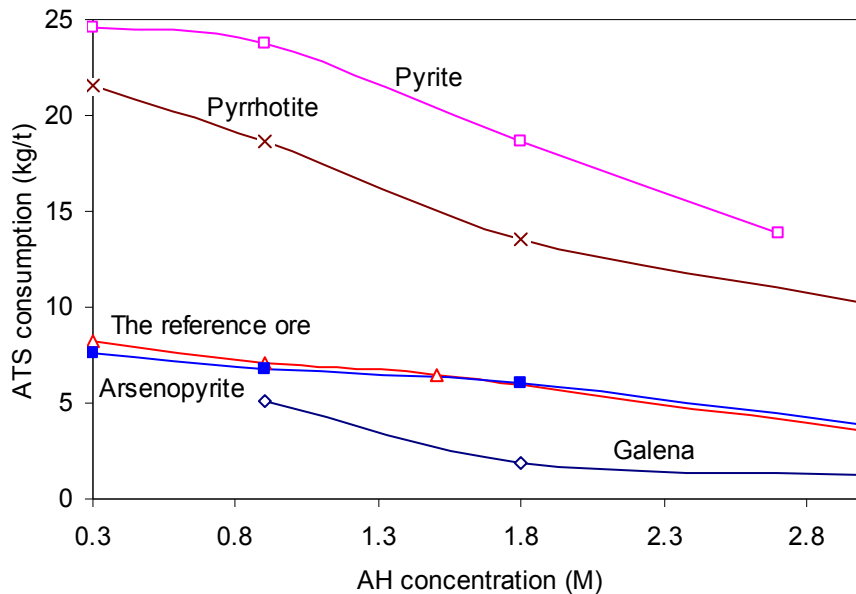
remained low (i.e., about 22%). The ATS consumption, on the other hand, was reduced from 5.1 to 1.3 kg/t by adding 3.0 M instead of 0.9 M ammonia.

**Table 5-3: Effect of ammonia on gold extraction and thiosulfate consumption in the presence of 2.4% galena**

Leaching conditions				ATS consumption (kg/t)	Gold extraction %	
ATS (M)	Cu (mM)	AH (M)	Oxygen %		3h	0.5h
0.20	1.2	0.9	21.1%	5.1	6.5%	21.9%
0.20	1.2	1.8	21.1%	1.9	14.4%	19.2%
0.20	1.2	3.0	21.1%	1.3	17.6%	22.4%

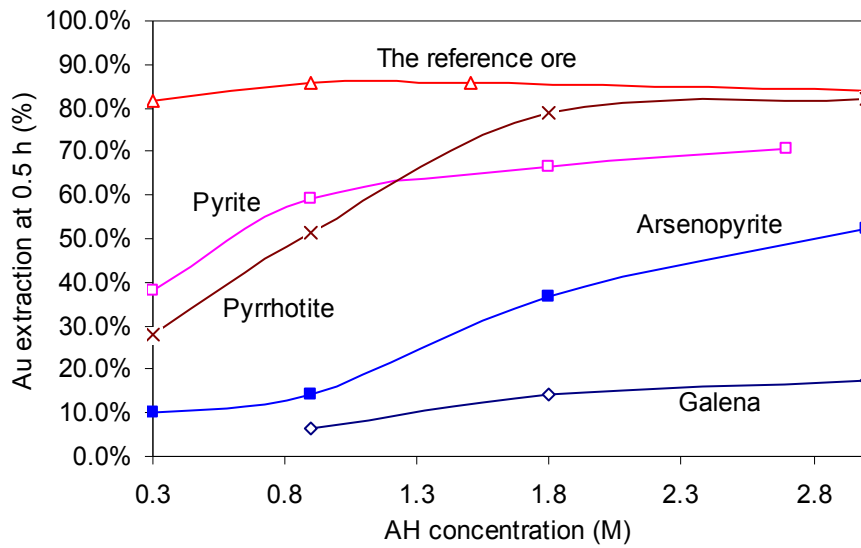
### 5.2.1.1. Discussion

To discuss the effect of ammonia in the presence of various minerals, some results were combined and are shown in the following figures.

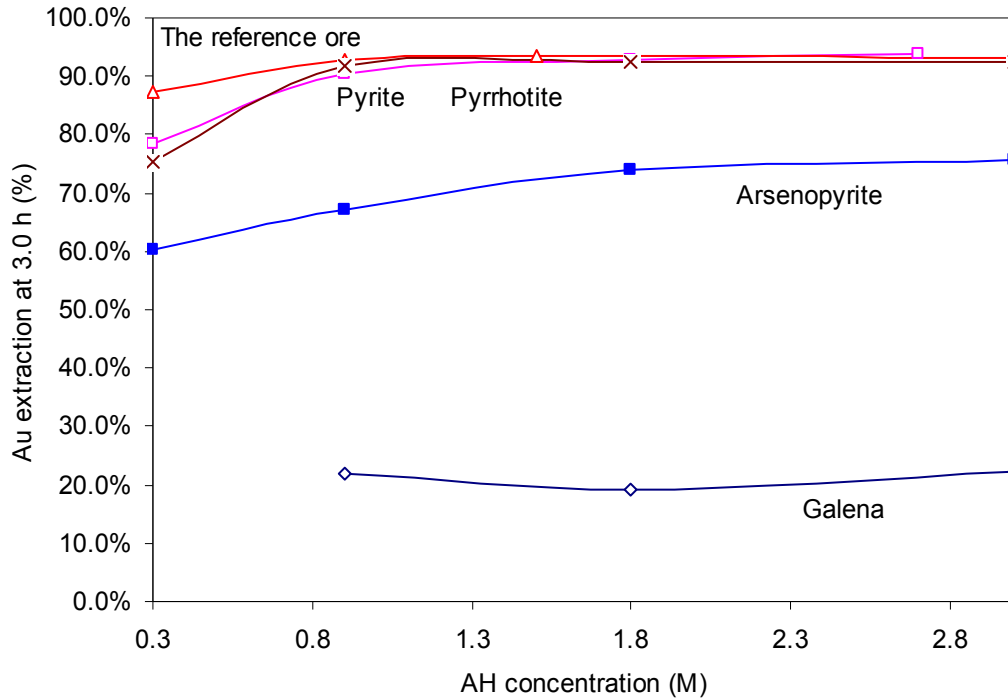


**Figure 5-27: Effect of ammonia concentration on thiosulfate consumptions (I)**  
(Leaching conditions: please refer to Figures 5-215 to 5-26 and Table 5-3)

Figure 5-27 compares the effects of ammonia concentration on thiosulfate consumptions in the presence of pyrite, pyrrhotite, arsenopyrite as well as galena. The arsenopyrite-bearing synthetic gold ore showed almost identical trends as the reference ore. The consumption of thiosulfate decreased with the increase of ammonia.



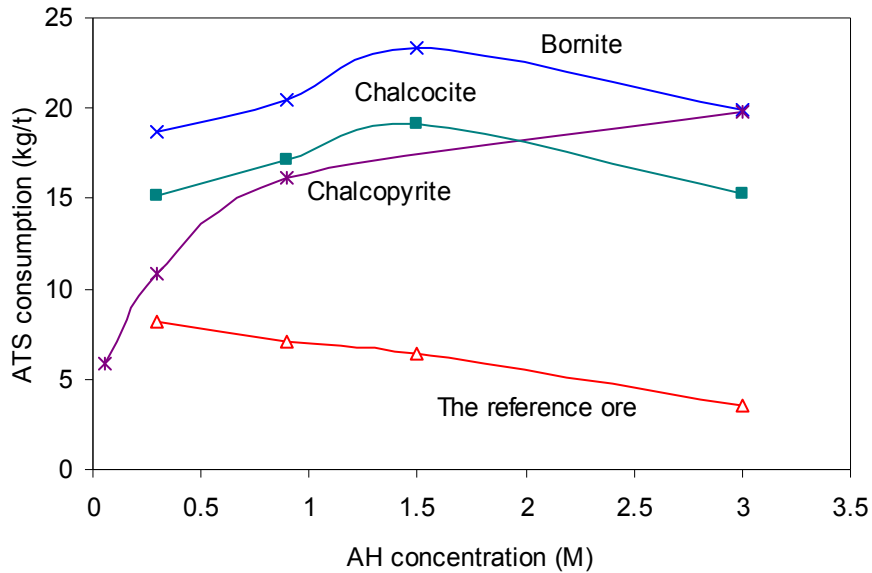
**Figure 5-28: Effect of ammonia on gold extraction at 0.5 hour (I)**  
 (Leaching conditions: please refer to Figures 5-215 to 5-26 and Table 5-3)



**Figure 5-29: Effect of ammonia on gold extractions at 3.0 hours (I)**  
**(Leaching conditions: please refer to Figures 5-215 to 5-26 and Table 5-3)**

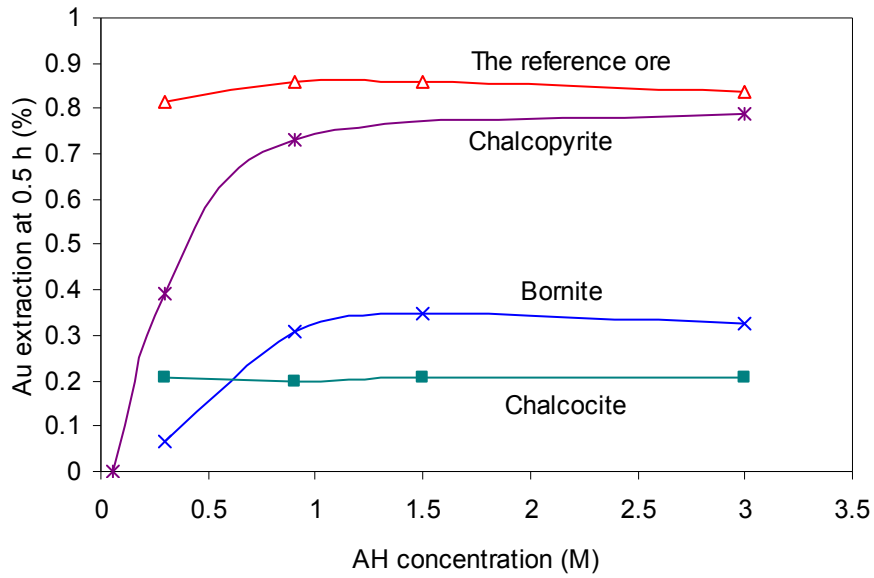
Figures 5-28 and 5-29 show the effects of ammonia concentration on gold extractions at 0.5 and 3.0 hours in the presence of pyrite, pyrrhotite, arsenopyrite and galena. For these minerals, the higher the ammonia concentration, the higher the gold than at 3.0 hours.

According to the theoretical analysis in Section 3.2.1, ammonia should improve both gold extractions and thiosulfate stability. The results in the current section have confirmed the conclusions with the exception of galena; this is because the gold surface was passivated by lead precipitates.

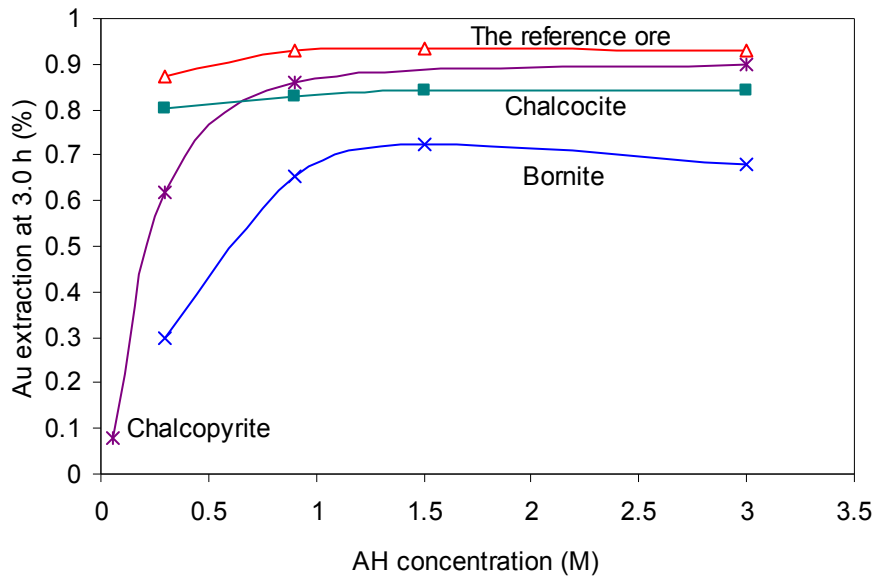


**Figure 5-30: Effect of ammonia concentration on thiosulfate consumptions (II)**  
**(Leaching conditions: please refer to Figures 5-215 to 5-26 and Table 5-3)**

Figure 5-30 demonstrates the effects of ammonia on thiosulfate consumption in the presence of copper sulphide minerals: chalcopyrite, chalcocite and bornite. The consumption of thiosulfate increased with the increase of ammonia concentration. For both bornite and chalcocite, the ATS consumption started to decrease when the ammonia concentration was higher than 1.5 M. This is possibly because the slurry potential became more negative and started inhibiting the oxidation of thiosulfate.



**Figure 5-31: Effect of ammonia on gold extraction at 0.5 hour (II)**  
 (Leaching conditions: please refer to Figures 5-215 to 5-26 and Table 5-3)



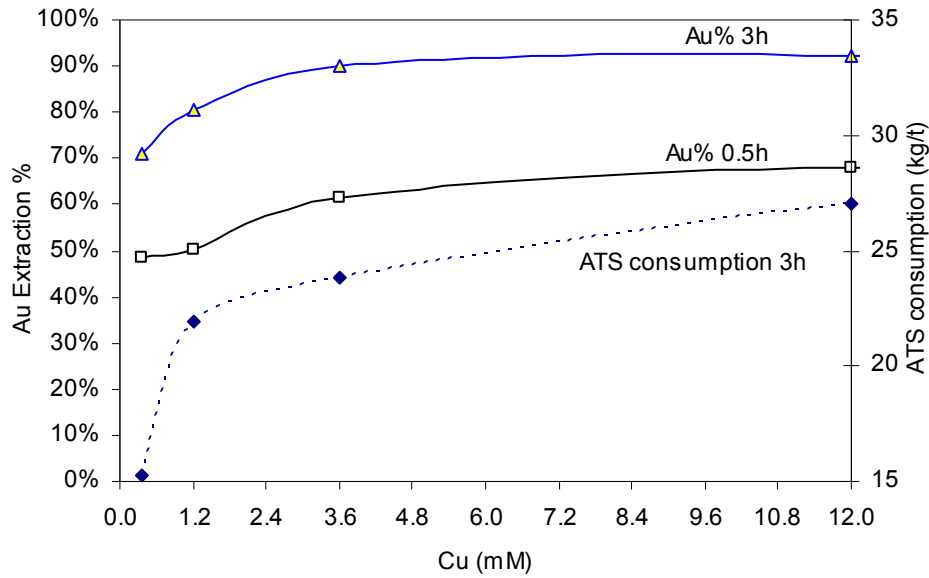
**Figure 5-32: Effect of ammonia on gold extraction at 3.0 hours (II)**  
 (Leaching conditions: please refer to Figures 5-215 to 5-26 and Table 5-3)

Figures 5-31 and 5-32 show the effects of ammonia on gold extraction and thiosulfate consumption in the presence of three copper sulfide minerals. For bornite and chalcopyrite, the gold extraction increased with the increase of ammonia concentration and reached a plateau at 1.5 M ammonia. However, in the leaching of chalcocite, the concentration of ammonia had almost no effect on the gold extraction.

The soluble copper ion released from the copper minerals further complicated the leaching reaction. On one hand, copper dissolution consumes free ammonia, and changes the oxidation state of the leaching system. On the other hand, the leached surface of the sulfide also has significant impacts on the leaching result. Therefore, for natural gold ores containing dissolvable copper minerals, the effect of ammonia must be carefully examined through leaching tests.

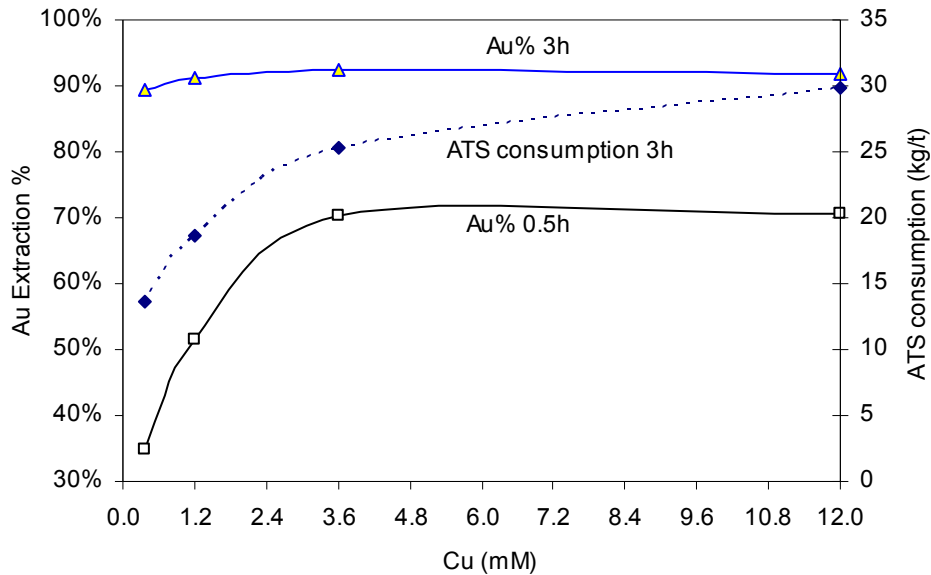
### **5.2.2. Copper concentration**

As discussed in section 3.2.1, copper is also an important factor in thiosulfate leaching, especially when the slurry contains significant amounts of reductive sulfide minerals.



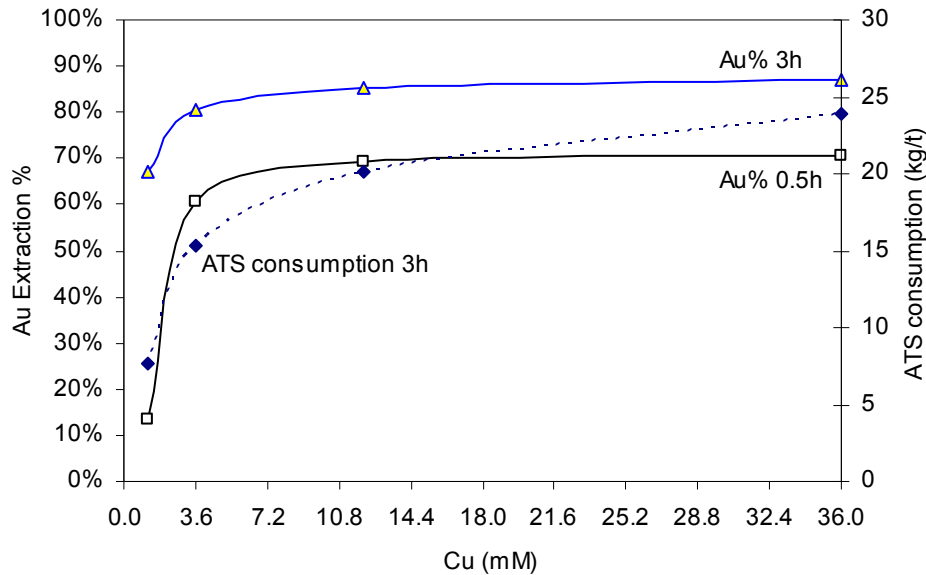
**Figure 5-33: Effect of copper on gold extraction and thiosulfate consumption in the presence of 16.0% pyrite (ATS: 0.20 M, AH: 0.9 M, oxygen: 21.1%)**

The effect of copper concentration on the leaching of the pyrite ore is demonstrated in Figure 5-33. By increasing copper concentration from 0.4 to 12.0 mM, the gold extraction increased from 48.7% to 65.1% at 0.5 hours and from 70.9% to 81.1% at 3.0 hours. However, the consumption of ATS at 3.0 hours also increased from 15.3 kg/t to 27.0 kg/t.



**Figure 5-34: Effect of copper on gold extraction and thiosulfate consumption in the presence of 16.0% pyrrhotite (ATS: 0.20 M, AH: 0.9 M, oxygen: 21.1%)**

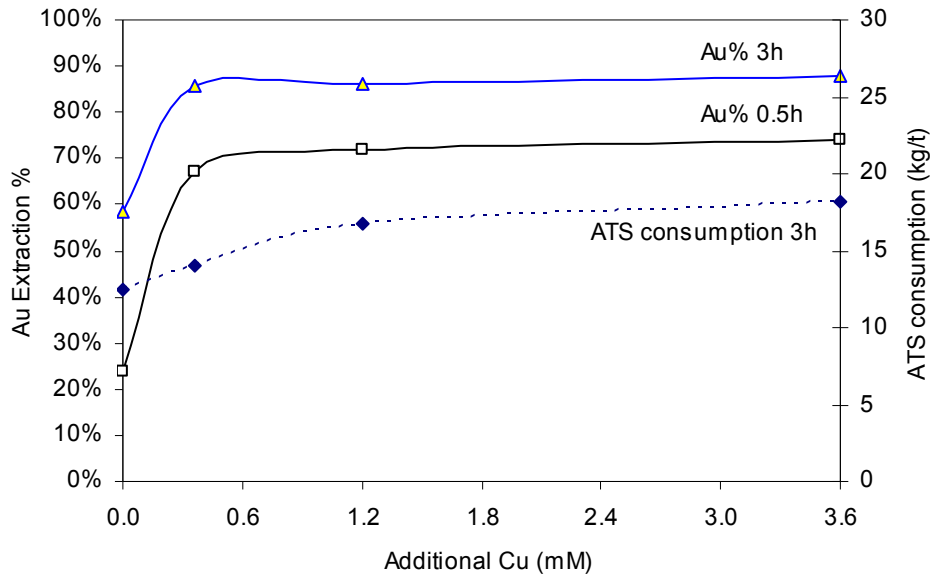
The pyrrhotite ore demonstrated similar trends as the pyrite ore. As shown in Figure 5-34, with the increase of copper concentration, increasing both gold extractions and thiosulfate consumptions. At 0.4 mM Cu, the gold extractions were 34.9% at 0.5 hour and 89.3% at 3.0 hours. By increasing copper concentration to 3.6 mM, the gold extractions at 0.5 hour and 3.0 hours were increased to 70.6% and 92.6% respectively. The consumption of ATS, however, was also increased by 11.6 kg/t (from 13.7 to 25.3 kg/t).



**Figure 5-35: Effect of copper on gold extraction and thiosulfate consumption in the presence of 4.0% arsenopyrite**

**(ATS: 0.20 M, AH: 0.9 M, oxygen: 21.1%)**

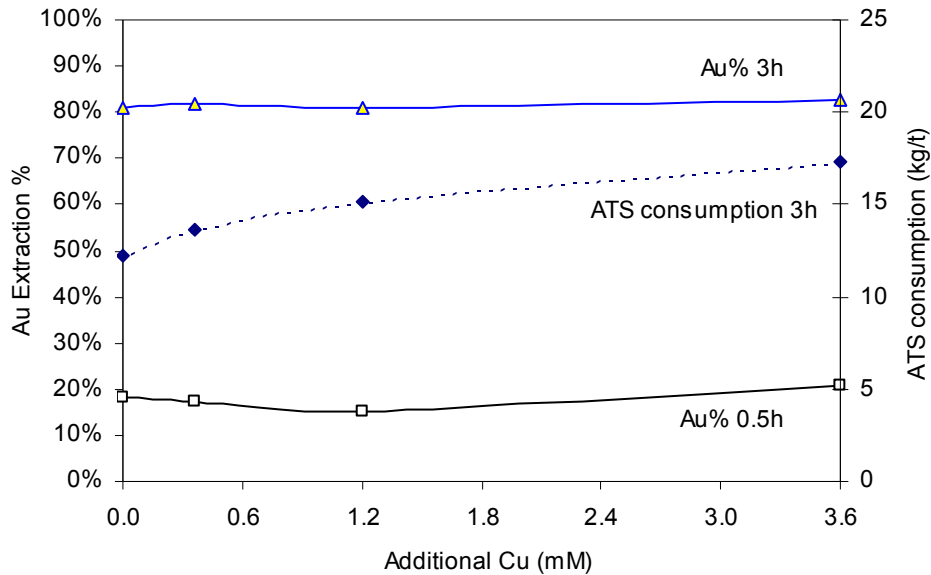
The effect of copper additions on thiosulfate leaching in the presence of arsenopyrite is shown in Figure 5-35. At low copper concentration, i.e., 1.2 mM, the gold extractions at 0.5 hour and at 3.0 hours were 13.2% and 67.1% respectively. By adding more copper, e.g., 12 mM, the gold extractions at 0.5 hour and 3.0 hours increased to 80.7% and 85.2% respectively. Again, the increase in copper concentrations resulted in a significant increase of the thiosulfate consumption. With 3.6 or 12 mM copper, the thiosulfate consumptions were 15.3 or 20.1 kg/t respectively.



**Figure 5-36: Effect of additional copper in presence of 8.0% chalcopyrite (ATS: 0.20 M, AH: 0.9 M, oxygen: 21.1%)**

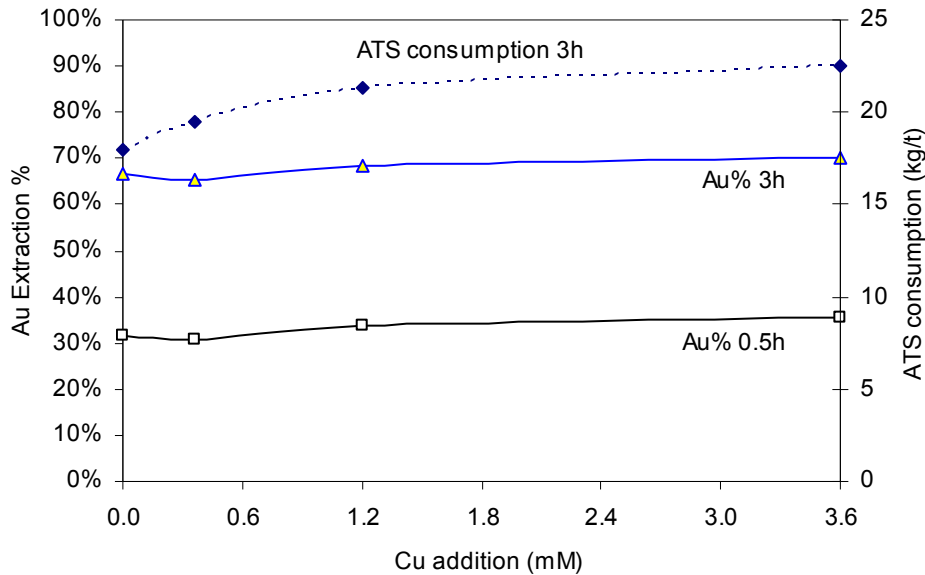
The leaching result in the presence of chalcopyrite is shown in Figure 5-36 with added copper concentration as the variable. With the increase of copper concentration from 0 to 3.6 mM, the gold extraction increased from 23.8% to 73.9% at 0.5 hour and 58.3% to 87.9% at 3.0 hours. Meanwhile, the consumption of ATS was also increased from 12.5 to 18.2 kg/t.

The gold extractions in the absence of copper sulfate were obviously very low. This result can be explained by the very limited dissolution of copper from chalcopyrite.



**Figure 5-37: Effect of copper concentration on gold extraction and thiosulfate consumption in the presence of 4.0% chalcocite (ATS: 0.20 M, AH: 0.9 M, oxygen: 21.1%)**

The addition of copper sulphate is essential for a fast leaching rate in the thiosulfate leaching of gold. In the presence of chalcocite, however, the additional copper sulphate was not required. As shown in Figure 5-37, the addition of copper was from 0 mM to 3.6 mM. The gold extractions were at about 19% at 0.5 hour and 81% at 3.0 hours for all tests. They were independent of the amount of copper sulphate addition. In terms of ATS consumption, increasing the copper additions increased the consumption from 12.2 kg/t to 17.3 kg/t.



**Figure 5-38: Effect of copper addition on gold extraction and thiosulfate consumption in the presence of 2.0% bornite (ATS: 0.20 M, AH: 0.9 M, oxygen: 21.1%)**

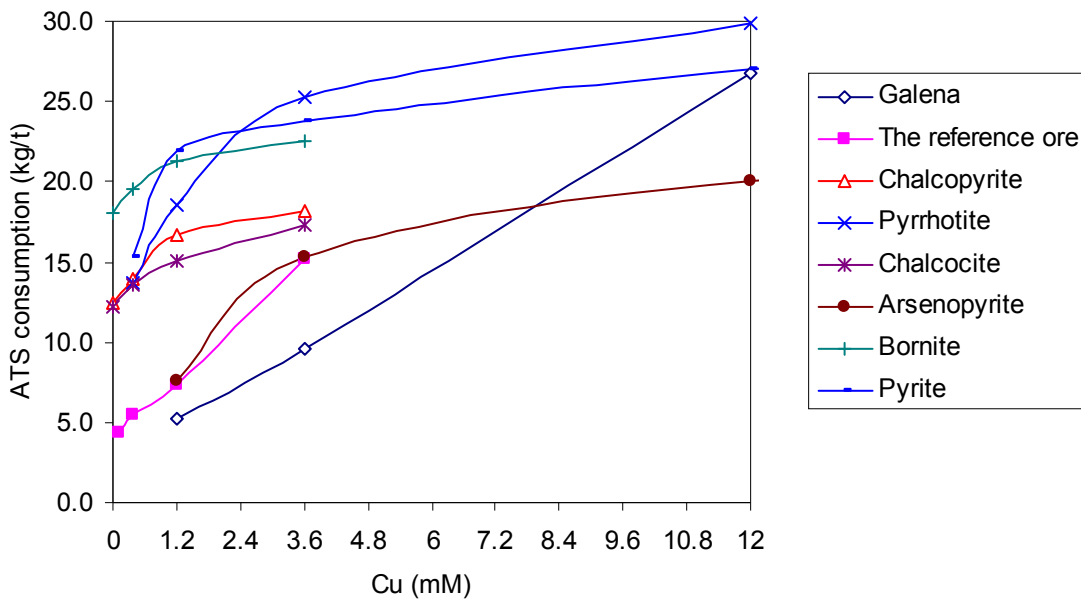
In the presence of bornite, the effects of the additional copper concentration on gold extraction and thiosulfate consumption are depicted in Figure 5-38. By increasing the copper concentration from 0.4 to 3.6 mM, the gold extractions increased from 18.0% to 22.5% at 0.5 hour and from 66.9% to 70.2% at 3.0 hours. Meanwhile, the ATS consumption increased from 18.0 to 22.5 kg/t.

In thiosulfate leaching of the galena bearing gold ore, the gold leaching was not improved by increasing the concentration of copper. As shown in Table 5-4, the gold extractions increased from 6.5% to 8.8% at 0.5 hour with 1.2 to 12.0 mM Cu. The gold extractions at 3.0 hours were between 18.7% and 21.9%. On the other hand, the ATS consumption increased from 5.2 to 26.8 kg/t with the increase of copper concentration from 1.2 to 12.0 mM.

**Table 5-4: Effect of copper on gold extraction and thiosulfate consumption in presence of 2.4% galena**

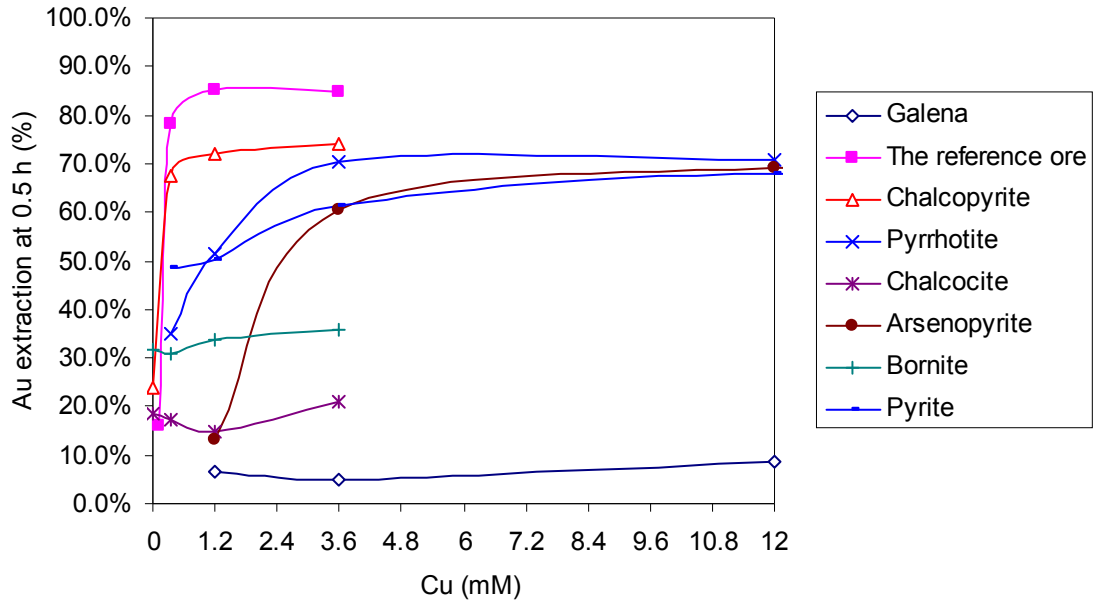
Leaching conditions				Au extractions (%)		ATS consumption (kg/t)
ATS (M)	Cu (mM)	AH (M)	Oxygen %	0.5 h	3.0 h	3.0 h
0.20	1.2	0.9	21.1%	6.5%	21.9%	5.2
0.20	3.6	0.9	21.1%	5.1%	18.7%	9.6
0.20	12	0.9	21.1%	8.8%	20.8%	26.8

**5.2.2.1. Discussion**

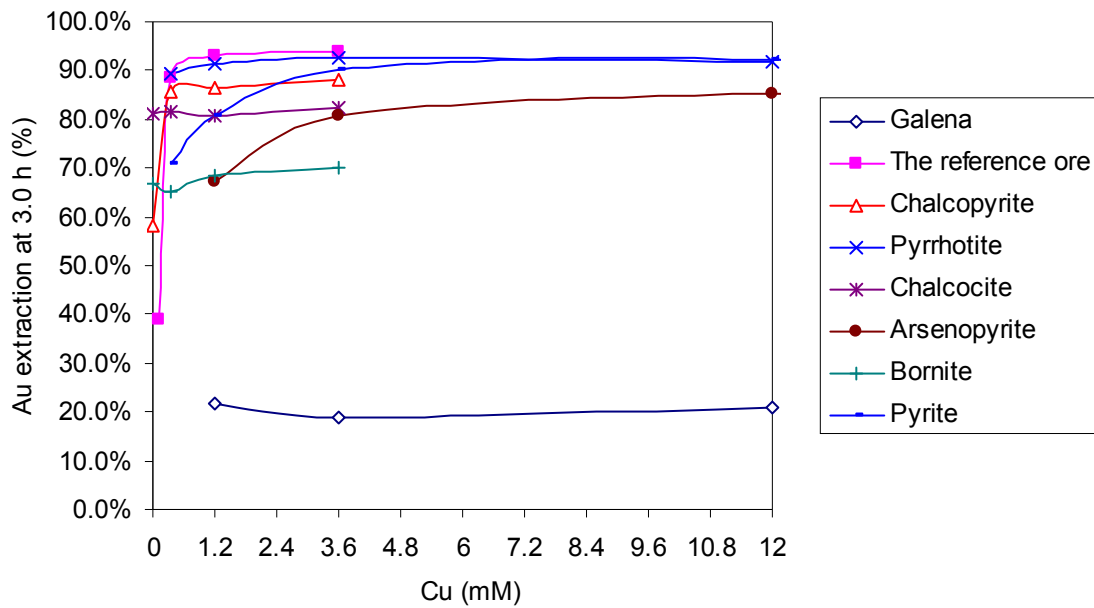


**Figure 5-39: Effects of copper on ATS consumption**  
 (Leaching conditions: please refer to Figures 5-33 to 5-38 and Table 5-4)

Figure 5-39 summarizes the effects of copper concentration on thiosulfate consumption in the presence of various minerals. For all minerals, increasing the amount of copper additions increased the consumption of ATS. As previously discussed, thiosulfate oxidation can be catalyzed by copper in the solution phase and enhanced in the presence of more copper.



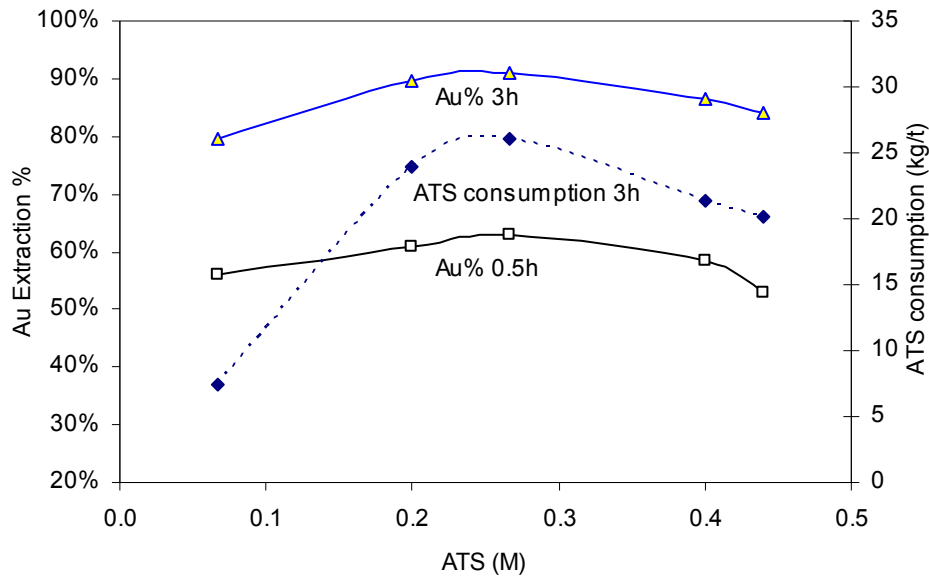
**Figure 5-40: Effects of copper on gold extraction at 0.5 hour leach**  
 (Leaching conditions: please refer to Figures 5-33 to 5-38 and Table 5-4)



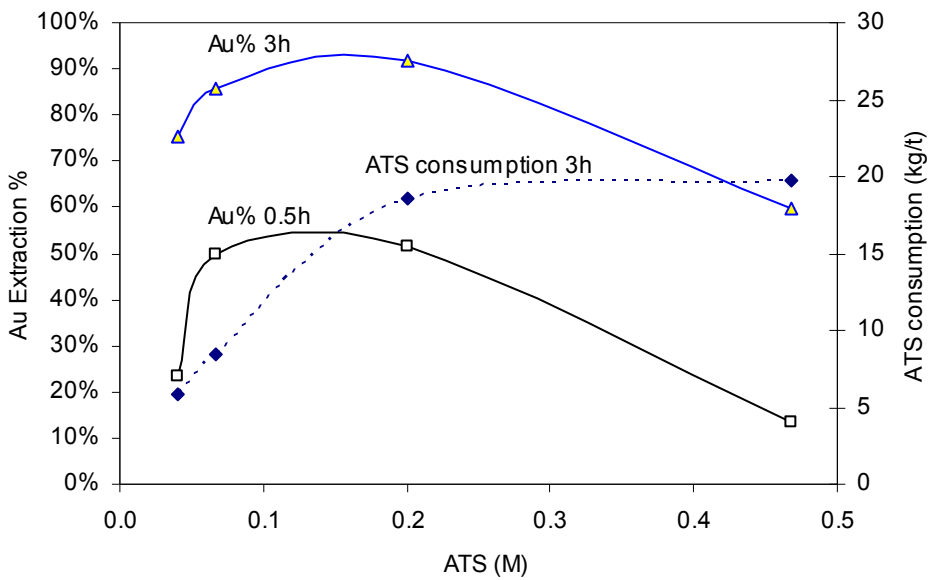
**Figure 5-41: Effects of copper concentration on gold extraction at 3.0 hours leach**  
 (Leaching conditions: please refer to Figures 5-33 to 5-38 and Table 5-4)

Figures 5-40 and 5-41 summarize the effect of copper concentration on the gold extractions at 0.5 hour and 3.0 hours in the presence of various minerals. Increasing the copper additions resulted in increasing the gold extractions in the presence of most minerals. The conclusions in Sections 3.1.2 and 3.2.1 suggest that copper is beneficial in that it provides more oxidative conditions for the oxidation of gold, the sulfide surface, as well as the thiosulfate. The results shown in this section have confirmed these conclusions. However, in the presence of chalcocite, bornite and galena, the copper concentration had no effect on the gold extractions. As for galena, the major problem is the dissolution of lead and subsequent passivation of the gold surfaces by lead precipitates. The increase of copper concentration can prevent neither the dissolution of lead nor the deposition of lead on gold. Therefore, copper concentration had no significant effect on thiosulfate leaching in the presence of galena.

### 5.2.3. Thiosulfate concentration



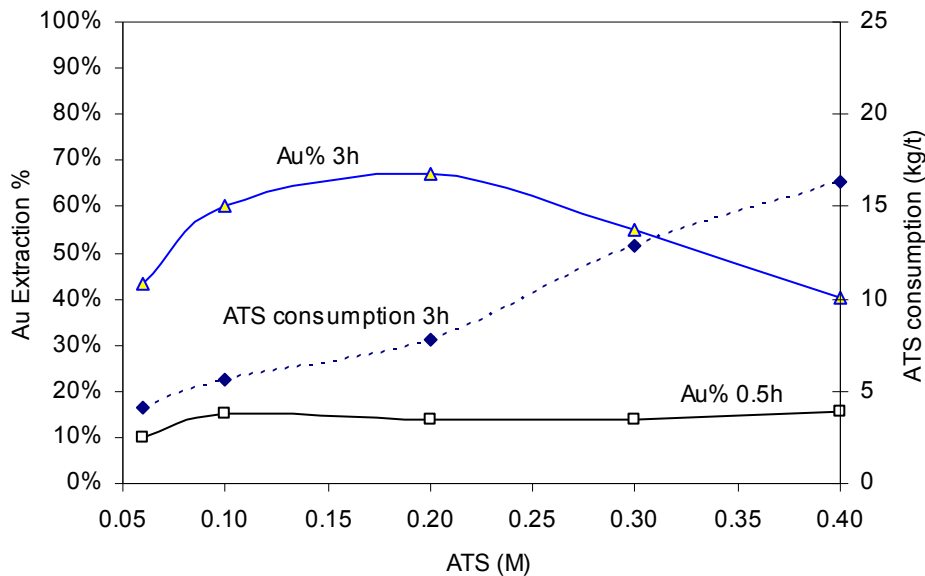
**Figure 5-42: Effect of ATS on gold extraction and thiosulfate consumption in the presence of 16.0% pyrite (Cu: 3.6 mM, AH: 0.9 M, oxygen: 21.1%)**



**Figure 5-43: Effect of ATS on gold extraction and thiosulfate consumption in the presence of 16.0% pyrrhotite**

(Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)

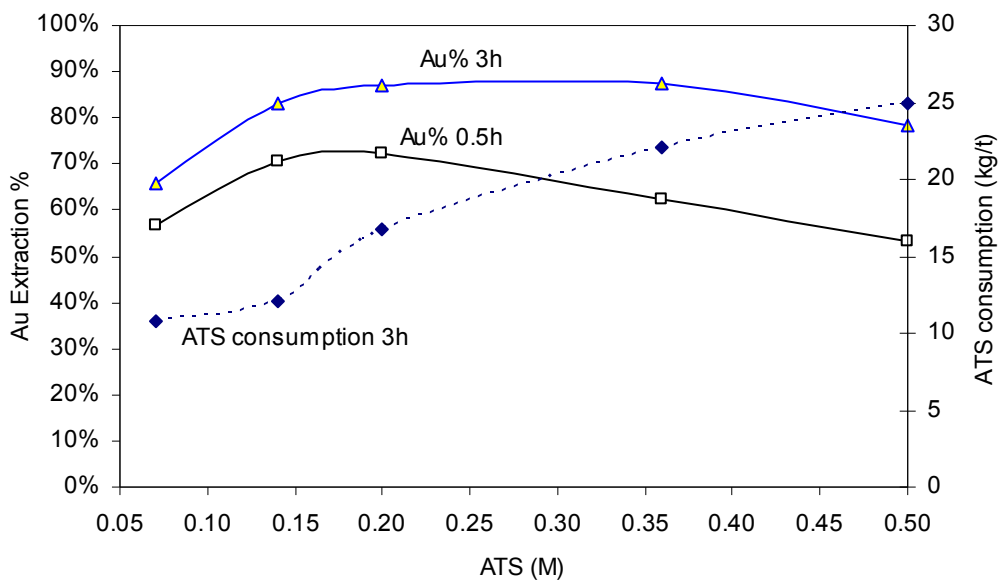
In the presence of 16% pyrite or 16% pyrrhotite, the effects of ATS concentration on the gold extraction and the thiosulfate consumption are shown in Figures 5-42 and 5-43. Results show that the insufficient and excessive additions of ATS reduced the gold extractions. The optimum ATS concentration was 0.20 M for pyrite ore and 0.27 M for pyrrhotite. The gold extractions were 90.9% and 91.6% for pyrite and pyrrhotite respectively under optimum conditions. The consumption of thiosulfate responded differently. When the ATS is lower than the optimum condition, the consumption increased with the increase of thiosulfate concentration. When the concentration of thiosulfate was higher than 0.20 M for pyrite or 0.27 M for pyrrhotite, the consumption decreased or remained the same.



**Figure 5-44: Effect of ATS on gold extraction and thiosulfate consumption in the presence of 4.0% arsenopyrite**

(Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)

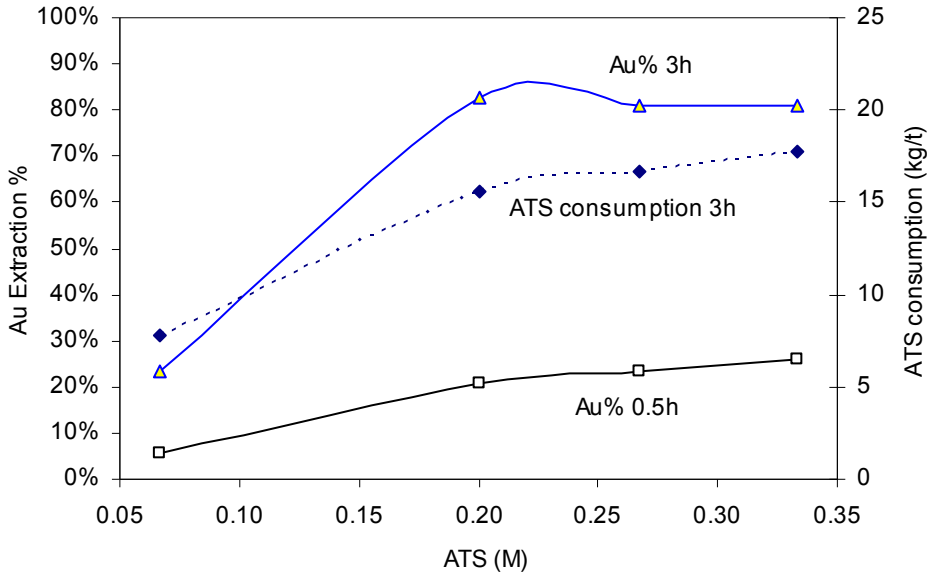
Figure 5-44 shows that gold extraction at 0.5 hour was about 14% with 0.10 to 0.40 M ATS. By increasing the ATS concentration from 0.06 M to 0.20 M, the gold extraction at 3.0 hours increased from 43.3% to 67.1%. Further increasing the ATS concentration from 0.2 to 0.4M, the gold extraction decreased from 67.1% to 40.3%. The ATS consumption at 3 hours increased from 7.6 to 17.3 kg/t.



**Figure 5-45: Effect of ATS on gold extraction and thiosulfate consumption in the presence of 8.0% chalcopyrite (Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

Figure 5-45 demonstrated the effects of ATS concentration on thiosulfate leaching of the chalcopyrite bearing gold ore. Insufficient ATS concentration, i.e., 0.07 M, resulted in reduced gold extraction, i.e., 56.6% at 0.5 hours and 65.8% at 3.0 hours. An excessive addition of ATS also led to low gold extractions. With 0.36 M ATS, the gold extraction at 0.5 hours was only 62.6%. The optimum ATS

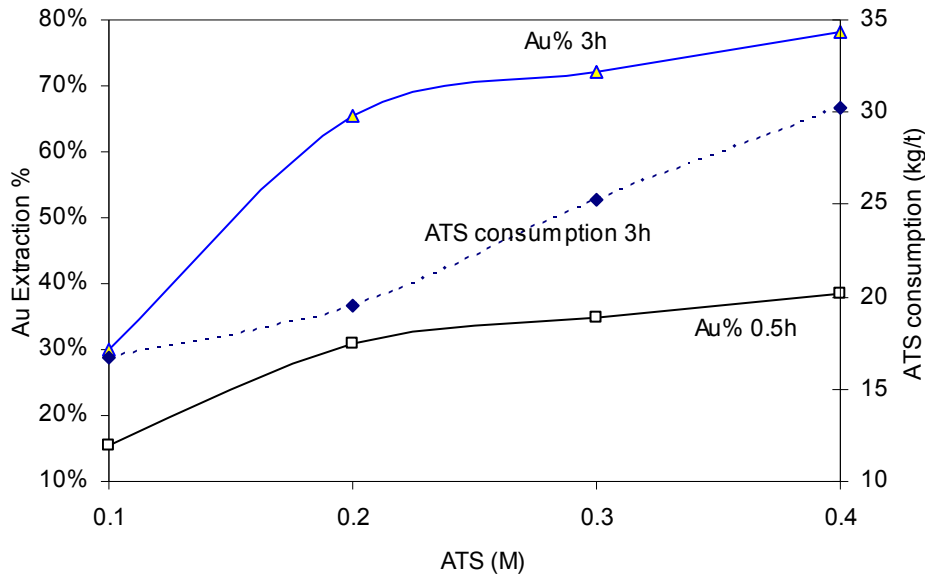
concentration was between 0.14 and 0.20 M. Within 0.5 hour, more than 70% of the gold was leached. At 0.20 M ATS, the gold extraction reached 86.9% at 3.0 hours.



**Figure 5-46: Effect of ATS on gold extraction and thiosulfate consumption in the presence of 4.0% chalcocite**

**(Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

Figure 5-46 shows the consumption of ATS increased from 7.8 to 17.8 kg/t. The gold extraction at 0.5 hour increased from 5.8% to 25.8% as the ATS concentration increased from 0.07 to 0.33 M. The overall gold extraction increased from 23.3% to 80.8% with the increase of ATS concentration in the same range. The increase of overall gold extractions reached a plateau after the thiosulfate concentration had reached 0.2 M.



**Figure 5-47: Effect of ATS on gold extraction and thiosulfate consumption in presence of 2.0% bornite**

**(Cu: 1.2 mM, AH: 0.9 M, oxygen 21.1%)**

Figure 5-47 displays the effect of thiosulfate concentration on thiosulfate gold leaching in the presence of bornite. It was found that the gold extractions could be improved significantly by adding more thiosulfate. By increasing the ATS concentration from 0.10 to 0.40 M, the gold extractions increased from 15.5% to 38.6% at 0.5 hour and the gold extraction at 3.0 hours increased from 30.0% to 78.2%. Meanwhile, the ATS consumption increased from 16.7 kg/t to 30.2 kg/t with the increase of ATS addition within the same concentration range.

In the presence of galena, thiosulfate concentration appeared to have substantial influence on the gold extraction. As shown in Table 5-5, by increasing thiosulfate concentration from 0.20 to 0.80 M, the gold extractions at 0.5 hour increased from 6.5% to 56.0%. The gold extraction at 3.0 hours improved from 21.9% to

88.4%. Meanwhile, the consumption of ATS increased by 14.3 kg/t when the concentration was increased from 0.20 to 0.80 M.

**Table 5-5: Effect of ATS concentration in the presence of 2.4% galena**

Leaching conditions				ATS consumption (kg/t)	Gold extraction %	
ATS (M)	Cu (mM)	AH (M)	Oxygen %	3h	0.5h	3h
0.20	1.2	0.9	21.1%	4.9	6.5%	21.9%
0.40	1.2	0.9	21.1%	8.9	58.4%	74.4%
0.80	1.2	0.9	21.1%	19.1	56.0%	88.4%

### 5.2.3.1. Discussion

In the presence of iron sulfide minerals, i.e., pyrite, pyrrhotite, arsenopyrite and chalcopyrite, the optimum thiosulfate concentrations for gold extraction were between 0.1 and 0.3 M. The concentration of thiosulfate below or beyond this range would result in reduced gold extractions.

For those highly dissolvable minerals, such as chalcocite, bornite and galena, the gold extractions increased with an increase of thiosulfate concentration. This is probably because all these three minerals have much higher concentrations of dissolved metals in the solutions. For chalcocite and bornite, large amounts of copper were dissolved during the leaching process, resulting in the amount of free thiosulfate anion being decreased due to the higher concentrations of cuprous thiosulfate complex. For galena, the dissolution of lead is competing with the gold to form complexes with thiosulfate. Therefore, it is necessary to retain a higher concentration of thiosulfate so that the extraction of gold is not affected. In addition, higher ATS concentrations may contribute to improve the stabilization of

more lead ions in solution and prevent the deposition of lead on the surface of the gold.

In the presence of most sulfide minerals (i.e., pyrrhotite, chalcopyrite, chalcocite, bornite, and galena), the higher the concentration of thiosulfate, the higher the consumption. For pyrite, however, the ATS consumption reached a maximum level at the optimum ATS consumption (i.e. about 0.20 to 0.27 M). Further additions of ATS resulted in a reduction of ATS consumption. This is possibly due to the excessive additions of thiosulfate causing a reductive condition and suppressing the oxidant of thiosulfate that is catalyzed by the pyrite surface.

In general, the typical range of the optimum thiosulfate concentration remained between 0.1 and 0.3 M. For any specific ore sample, the best way of determine the optimum concentrations of ATS is still through experimental methods. In the presence of some specific minerals, such as those dissolvable copper sulfides, and galena, more work is needed to optimize thiosulfate concentrations; typically these minerals will require higher additions of thiosulfate.

### **5.3. Examples of using chemical additives to enhance the leaching performance in the presence of detrimental minerals**

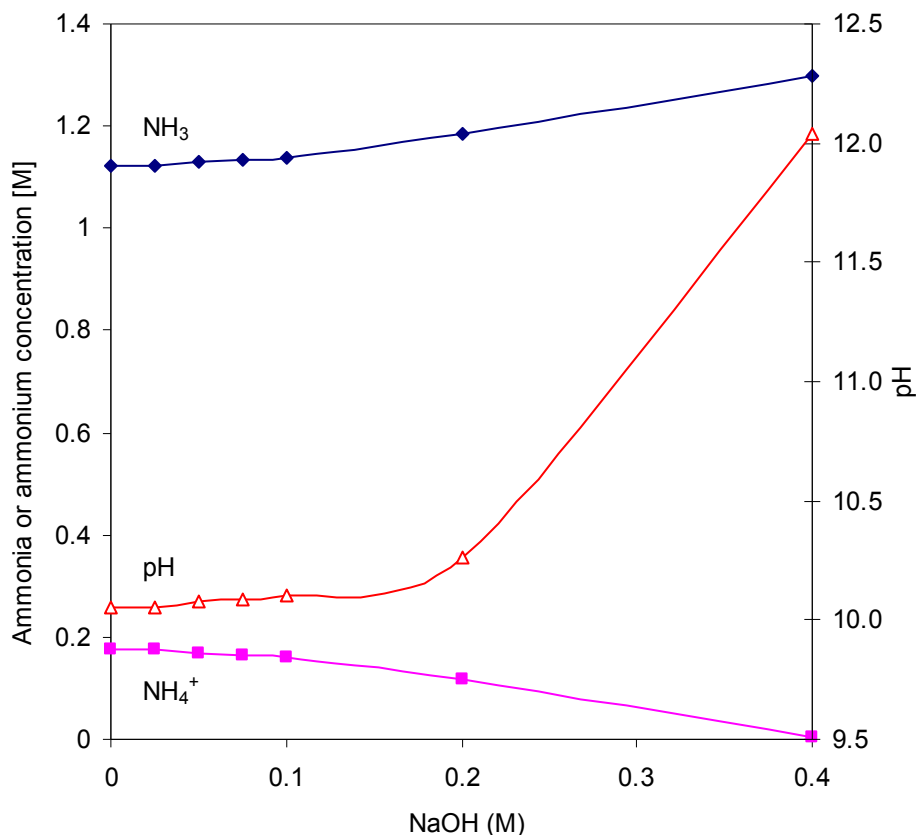
In this section, the effect of chemical additives, e.g., hydroxide, carbonate phosphate anion, calcium ion and galena on thiosulfate gold leaching will be discussed. The theoretical aspects have been proposed and discussed in

Section 4.3.4. The objective is to investigate the possibility of using additional reagents to improve gold extractions in the presence of detrimental minerals.

### **5.3.1. Additional hydroxide anion**

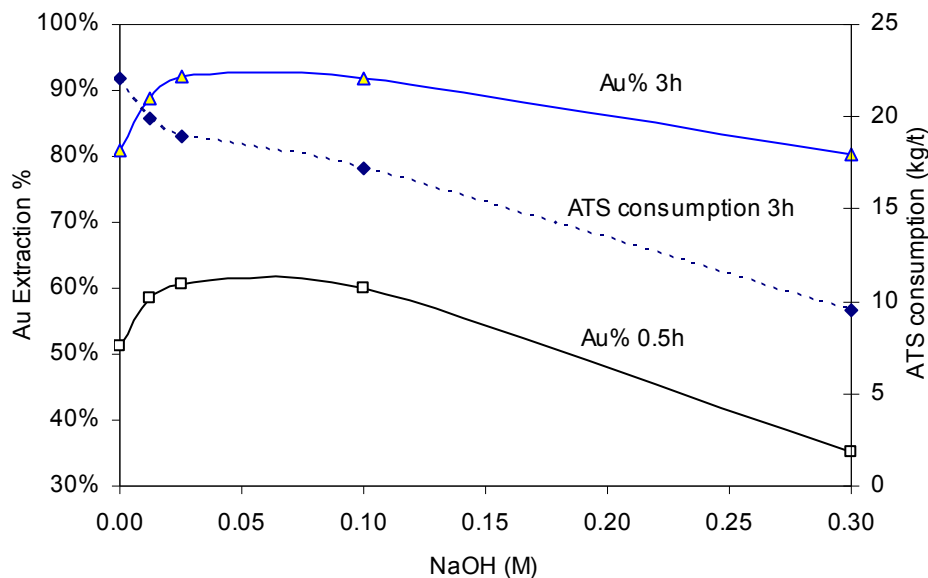
The effects of the additional NaOH are similar to those of the additional ammonia. By adding ammonia, both the hydroxide anion and the total ammonia-ammonium concentrations are increased. As discussed in the previous Section (3.2.1), adding ammonia is beneficial for both gold extraction and thiosulfate stability.

By adding NaOH, the ammonium ion will instantly be converted to free ammonia. However due to the total ammonia/ammonium concentration not being changed, the extra free ammonia will ultimately be converted back to the ammonium anion. The additional hydroxide ion will be consumed either by the oxidation reactions of thiosulfate or by the oxidation of sulfide minerals.



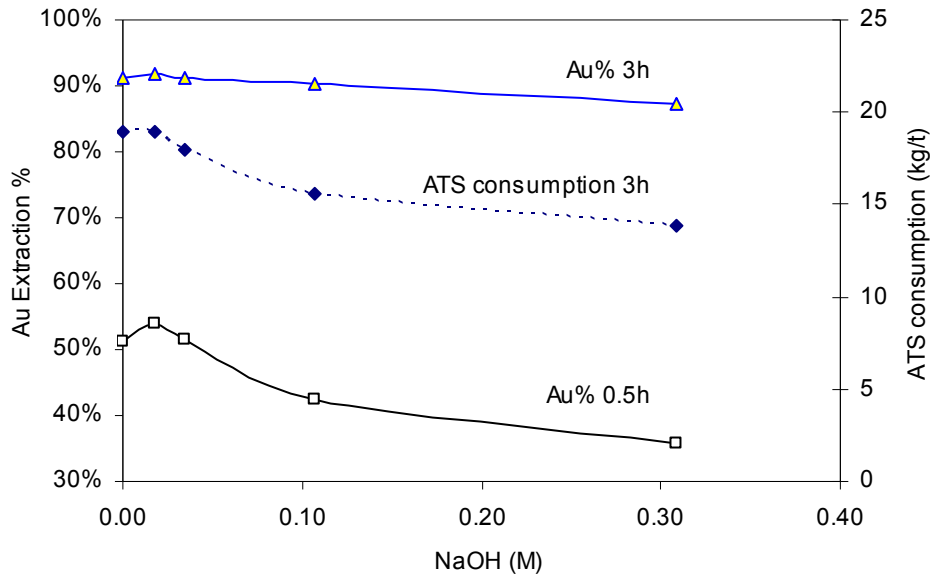
**Figure 5-48: Effects of NaOH on the concentration of NH<sub>3</sub>/NH<sub>4</sub><sup>+</sup> and slurry pH (ATS: 0.2 M, AH: 0.9 M, Cu: 1.2 mM)**

The effect of NaOH on the concentration of NH<sub>3</sub>/NH<sub>4</sub><sup>+</sup> and slurry pH is shown in Figure 5-48. The pH value was measured and the concentration of ammonia and ammonium were calculated from the pH values. The addition of NaOH increased the concentration of free ammonia (NH<sub>3</sub>) and decreased the concentration of ammonium ion (NH<sub>4</sub><sup>+</sup>). The change in pH appeared to be very slow within the range of 0.0 – 0.1 M NaOH because of the buffering effect of NH<sub>3</sub>/NH<sub>4</sub><sup>+</sup> solution. By adding 0.0 – 0.4 M NaOH, the increase of ammonia concentration was about 15.7% while the decrease of the ammonium ion was 98.8%.



**Figure 5-49: Effect of NaOH in the presence of 16.0% pyrite (ATS: 0.2 M, Cu: 3.6 mM, AH: 0.9 M, oxygen: 21.1%)**

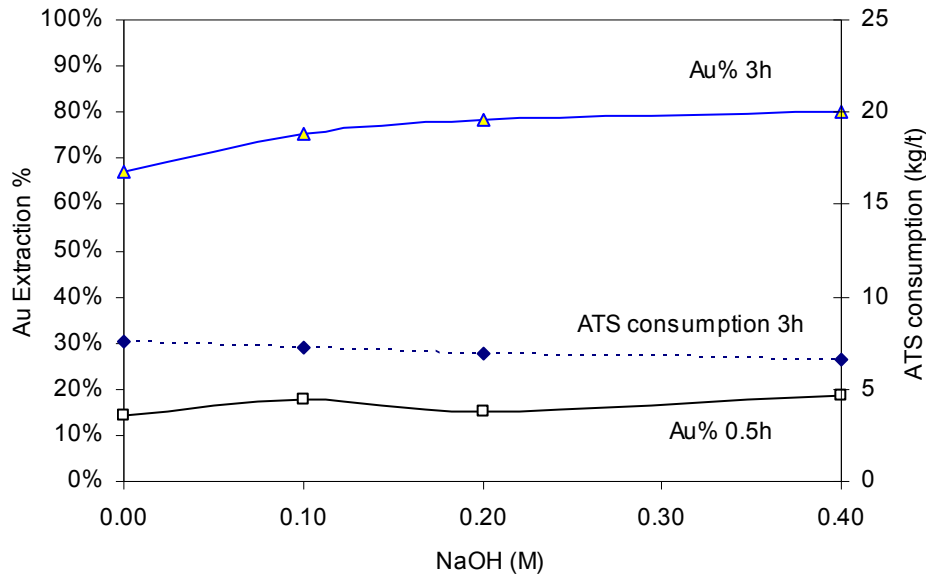
The effects of NaOH addition were tested in the ATS leaching of the pyrite-bearing ore. Figure 5-49 shows that the gold extraction was significantly improved by adding 25 or 100 mM NaOH. After both 0.5 and 3.0 hours, the gold extraction was increased by more than 10% respectively. However, excessive addition of NaOH (i.e., 300 mM) reduced the gold extractions. With 300 mM NaOH, the gold extractions at 0.5 and 3.0 hours were 35.2% and 80.3% only. It was also found that NaOH addition stabilized the thiosulfate anion. The reagent consumption was reduced from 22.1 to 9.5 kg/t by adding 300 mM NaOH.



**Figure 5-50: Effect of NaOH on gold extraction and thiosulfate consumption in the presence of 16.0% pyrrhotite**

**(ATS: 0.2 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

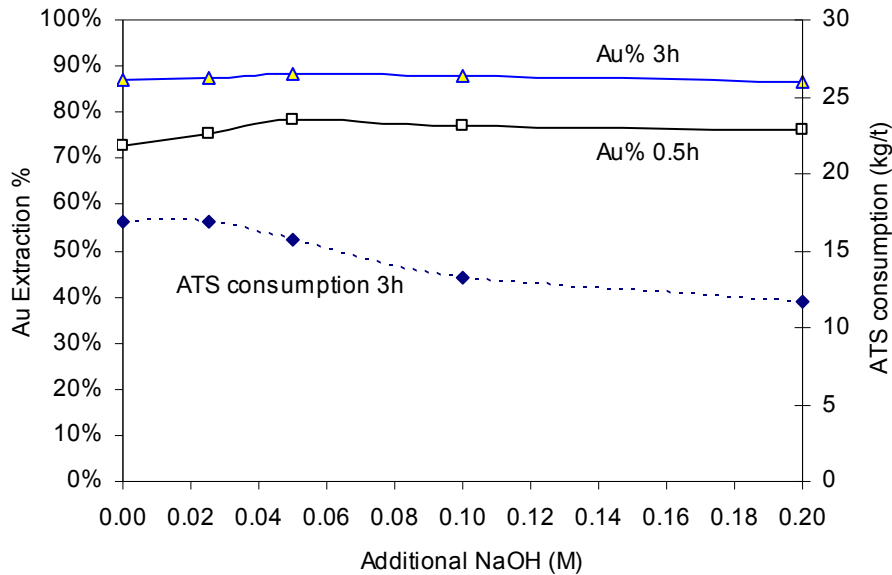
Figure 5-50 demonstrates the effects of NaOH on thiosulfate gold leaching in the presence of pyrrhotite. Compared to the pyrite bearing ore, the pyrrhotite gold ore had a different response to NaOH. An addition of 17.5 mM NaOH was beneficial to gold extractions. Addition of more NaOH appeared to be excessive and resulted in a reduced gold extraction. The gold extraction at 0.5 hours was reduced from 51.3% to 35.8% by adding 309 mM NaOH. The effect of NaOH on thiosulfate consumption of the pyrrhotite ore was similar to that of the pyrite ore. NaOH addition (309 mM) helped to reduce the reagent consumption by 5.1 kg/t (from 18.9 to 13.8 kg/t).



**Figure 5-51: Effect of NaOH on gold extraction and thiosulfate consumption in the presence of 4.0% arsenopyrite**

**(ATS: 0.2 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

As shown in Figure 5-51, the effect of NaOH in the thiosulfate leaching of the arsenopyrite bearing gold ore was shown. No noticeable improvement was found in the thiosulfate stability. The thiosulfate consumption remained in the range of 6.6 to 7.6 kg/t. By adding 0.4 M NaOH, the gold extraction at 3.0 hours increased from 67.2% to 78.4%.



**Figure 5-52: Effect of NaOH on gold extraction and thiosulfate consumption in the presence of 8.0% chalcopyrite**

**(ATS: 0.2 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

Figure 5-52 demonstrated the effect of NaOH on thiosulfate leaching in the presence of chalcopyrite. In the presence of chalcopyrite, the addition of NaOH has a slight improvement on the gold extraction. The ATS consumption was significantly reduced from 16.9 kg/t to 11.7 kg/t by adding 0.20 M NaOH.

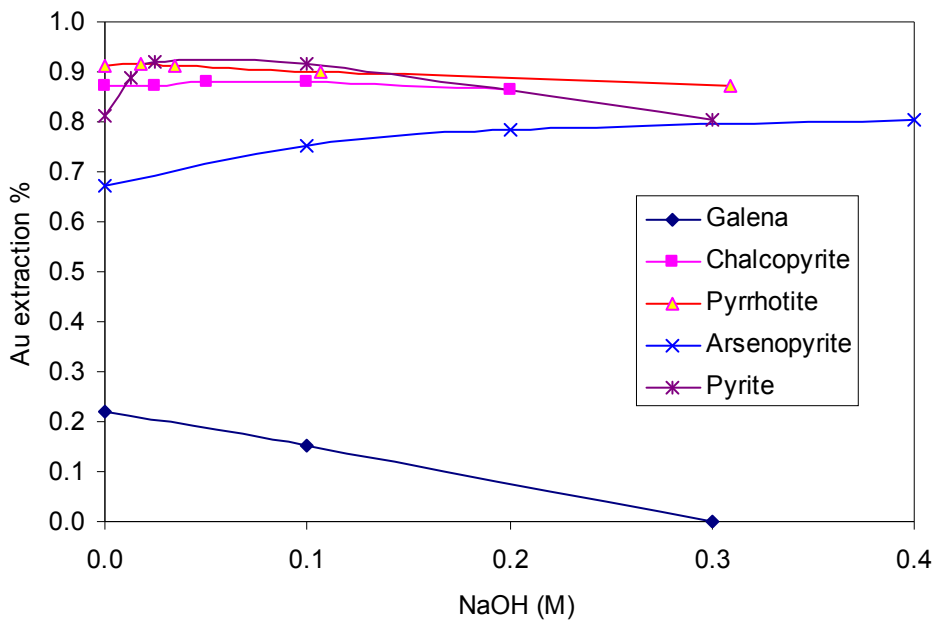
Table 5-6 listed the gold extractions and thiosulfate consumption in the presence of galena with various amount of NaOH. In the presence of galena, however, the presence of additional hydroxide anion made the passivation problem even worse. By adding 0.30 M NaOH, the gold extraction at 0.5 and 3.0 hours were reduced from 6.5% and 21.9% to 0.0%. This mean that the gold surface was completely passivated by the lead precipitate and hydroxide is the major species responsible for the passivation of gold.

**Table 5-6: Effects of NaOH on gold extraction and thiosulfate consumption in the presence of 2.4% galena**

Leaching conditions					ATS consumption (kg/t)	Gold extraction (%)	
ATS (M)	Cu (mM)	AH (M)	O2%	NaOH (M)	3	0.5	3
0.20	1.2	0.9	21.1%	0.00	5.1	6.5%	21.9%
0.20	1.2	0.9	21.1%	0.10	3.8	8.8%	15.2%
0.20	1.2	0.9	21.1%	0.30	1.9	0.0%	0.0%

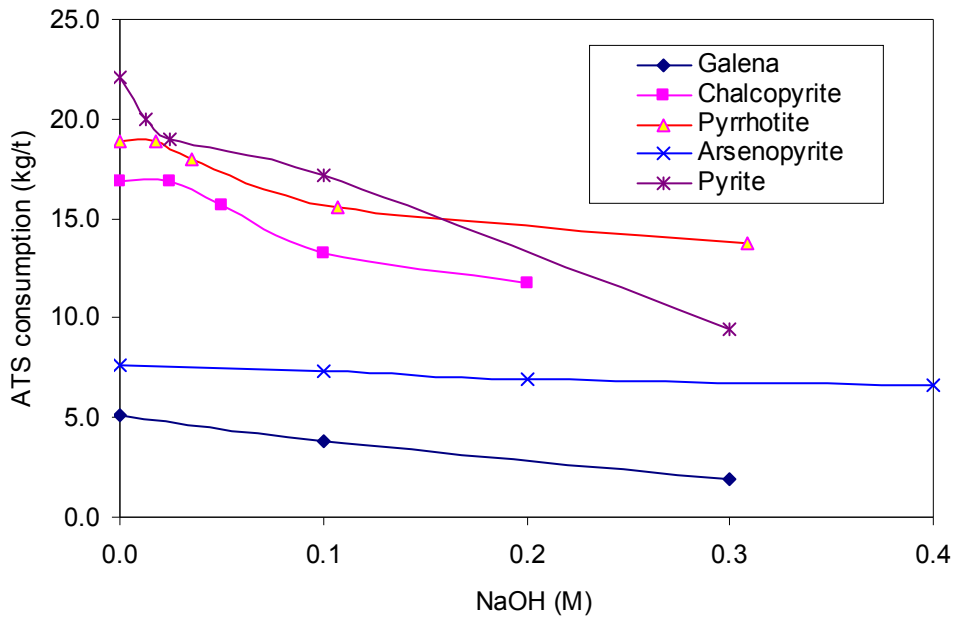
### 5.3.1.1. Discussion

The effects of additional hydroxide anion include: increasing slurry pH, increasing the ammonia concentration, and decreasing the ammonium concentration. In thiosulfate leaching, due to the increase of ammonia concentration, the gold extraction improved slightly. This is because the extra hydroxide anion enhanced the passivation of the sulfide surface due to the formation of FeOOH. Meanwhile, due to the higher alkalinity, the thiosulfate decomposition became slower.



**Figure 5-53: Effects of NaOH on gold extractions**

(Leaching conditions: please refer to Figures 5-48 to 5-52 and Table 5-6)



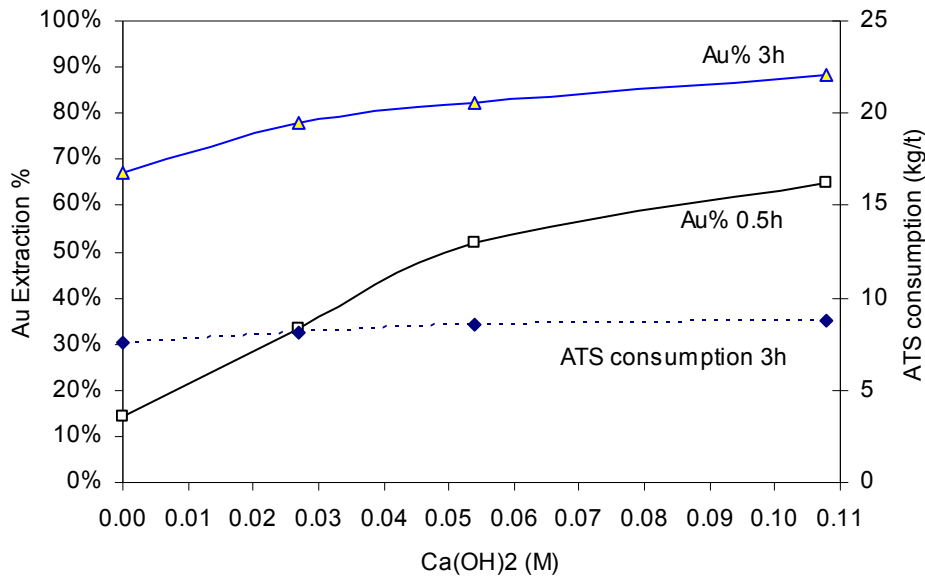
**Figure 5-54: Effects of NaOH on thiosulfate consumptions**

(Leaching conditions: please refer to Figures 5-48 to 5-52 and Table 5-6)

Figures 5-53 and 5-54 summarize the effects of NaOH on thiosulfate gold leaching. In the presences of pyrite, pyrrhotite, galena and chalcopyrite, the ATS consumption was significantly reduced by the addition of NaOH.

As for the gold extractions, the additions of NaOH significantly increased the gold extractions in the presence of pyrite and arsenopyrite. For other minerals, the effects of NaOH were insignificant.

### 5.3.2. Calcium ion



**Figure 5-55: Effect of  $\text{Ca}(\text{OH})_2$  on gold extraction and thiosulfate consumption in the presence of 4.0% arsenopyrite (ATS: 0.2 M, Cu: 1.2 mM, AH: 0.9 M, oxygen: 21.1%)**

Figure 5-55 shows the effect of calcium hydroxide on thiosulfate leaching of gold in the presence of arsenopyrite. It is obvious that the improvement with the calcium hydroxide is much more significant than that with the NaOH (Figure 5-51). Without adding quick lime, the gold extractions were 14.3% at 0.5 hour and 67.2% at 3.0 hours respectively. When the addition of quick lime was increased to 0.11 M, the gold extraction at 0.5 hour increased to 55.1%. The gold extractions at 3.0 hours were also improved to 88.3%. On the other hand, the addition of quick lime slightly increased the ATS consumption from 7.6 to 8.8 kg/t. The improvement of gold extraction by calcium hydroxide was explained in the

previous section (3.2.4.4). The formation of calcium arsenate accelerated the passivation of the reactive surface of arsenopyrite.

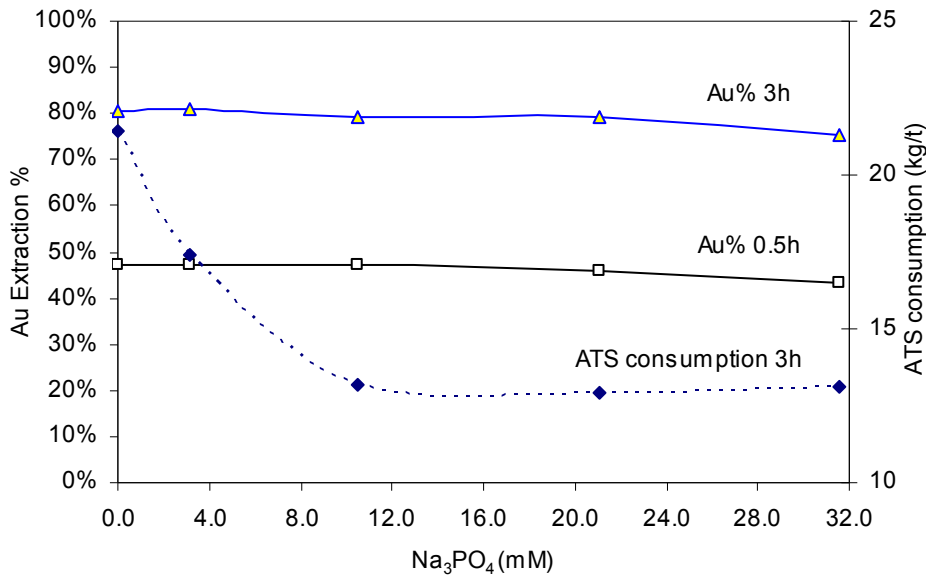
### 5.3.3. Phosphate anion

As shown in Table 5-7, in the thiosulfate leaching of the reference ore (pure silicate gold ore), the addition of the phosphate anion had no effect on the thiosulfate consumption. On the other hand, the gold extraction was slightly reduced by this anion. By adding 10.5 mM phosphate, the gold extraction at 3.0 hour was reduced from 92.0% to 87.6%.

**Table 5-7: Effect of sodium phosphate on gold extraction and thiosulfate consumption (the reference ore)**

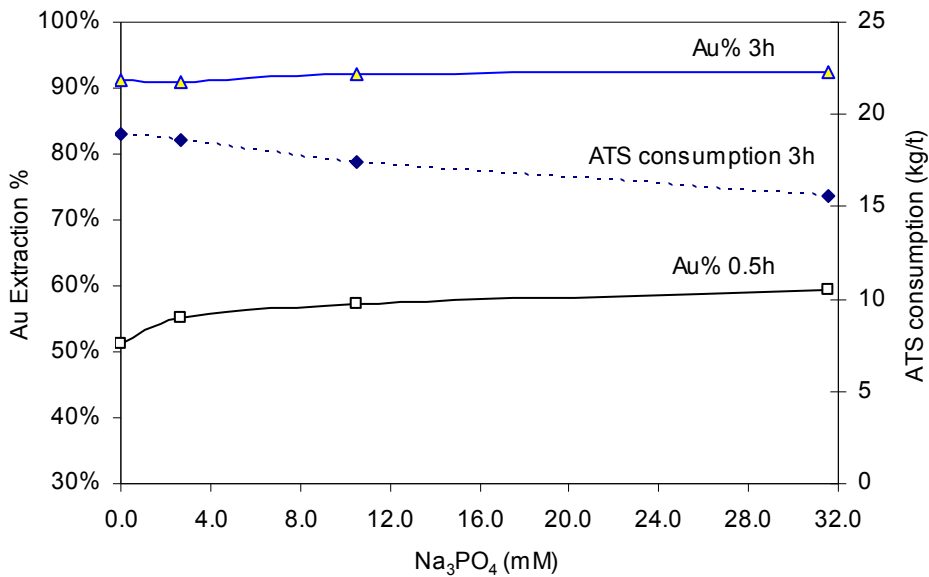
Na <sub>3</sub> PO <sub>4</sub> (mM)	ATS consumption	Gold extraction % 0.5h	Gold extraction % 3.0h
2.6	7.5	84.2%	92.0%
5.3	7.4	81.6%	90.6%
10.5	7.5	77.2%	87.6%

(ATS 0.2 M, AH 0.9 M, Cu 1.2 mM, oxygen 21.1%)



**Figure 5-56: Effect of trisodium phosphate on gold extraction and thiosulfate consumption in the presence of 16.0% pyrite**  
 (ATS: 0.2 M, AH: 0.9 M, Cu: 1.2 mM, oxygen: 21.1%)

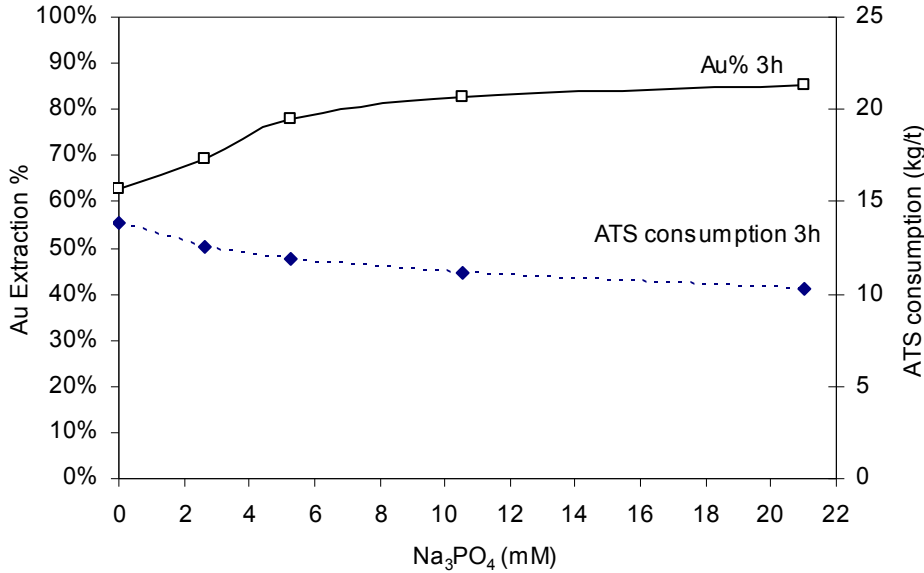
However, the leaching of sulfide bearing gold ores responded in a very different way to the phosphate anion. Figure 5-56 demonstrates the gold extraction and thiosulfate consumption in the presence of 16% pyrite. When 32 mM phosphate was added, the gold extraction decreased from 80.6% to 75.2%. By adding 10.5 mM phosphate, ATS consumption decreased by 8.2 kg/t (from 21.4 to 13.1 kg/t).



**Figure 5-57: Effect of trisodium phosphate on gold extraction and thiosulfate consumption in the presence of 16.0% pyrrhotite**  
 (ATS: 0.2 M, AH: 0.9 M, Cu: 1.2 mM, oxygen: 21.1%)

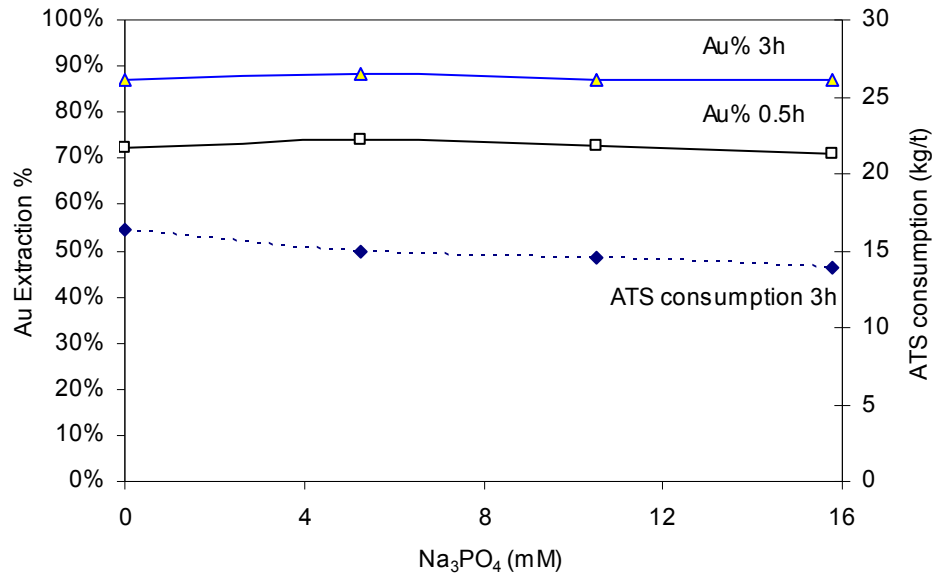
Figure 5-57 depicts the thiosulfate gold leaching results in the presence of pyrrhotite at various phosphate concentrations. The addition of phosphate was effective in reducing ATS consumption. By adding 32 mM phosphate, the gold extractions were improved from 51.3% to 59.5% at 0.5 hour and 91.1% to 92.3%

at 3.0 hours. Meanwhile, the consumption of ATS was reduced from 18.9 kg/t to 15.6 kg/t.



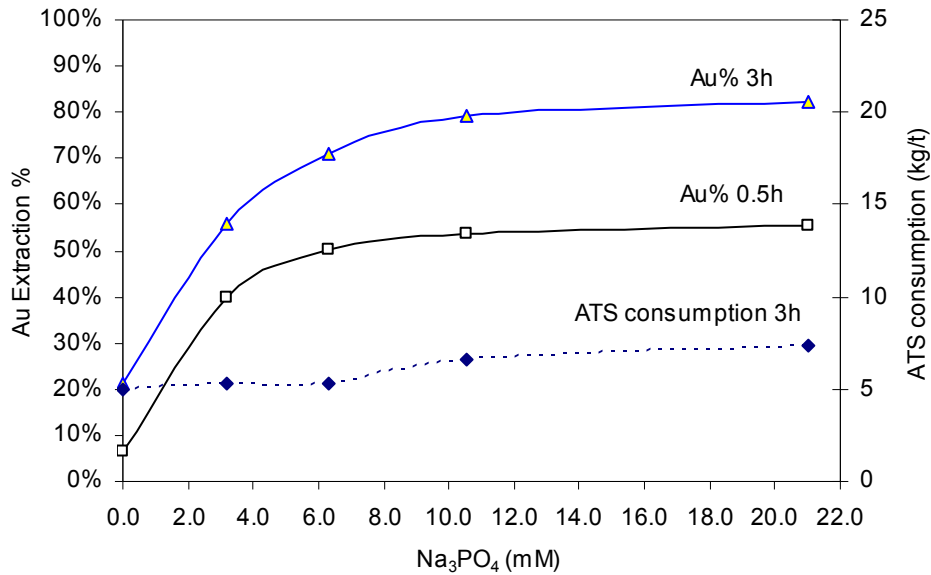
**Figure 5-58: Effect of sodium phosphate on gold extraction and thiosulfate consumption in presence of 8.0% chalcocite (ATS: 0.2 M, AH: 0.1 M, Cu: 1.2 mM, oxygen: 21.1%)**

Figure 5-58 demonstrates that the gold extraction in the presence of chalcocite was increased and the consumption of thiosulfate was decreased due to the addition of phosphate. The gold extraction at 3.0 hours increased from 62.8% to 85.2% with addition of phosphate from 0.0 to 21.0 mM. Meanwhile, the consumption of ATS was reduced from 13.8 kg/t to 10.3 kg/t.



**Figure 5-59: Effect of trisodium phosphate on gold extraction and thiosulfate consumption in the presence of 8.0% chalcopyrite (ATS: 0.2 M, AH: 0.9 M, Cu: 1.2 mM, oxygen: 21.1%)**

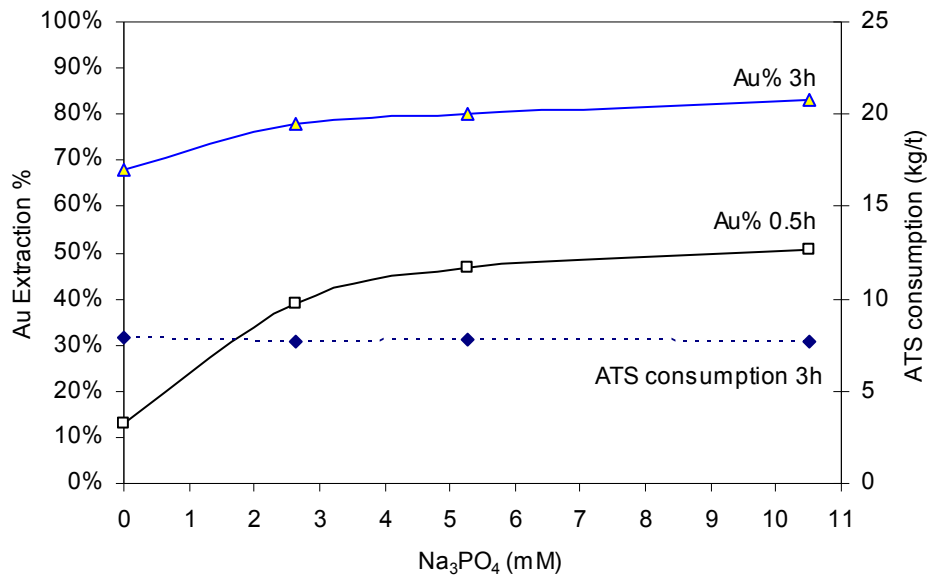
Figure 5-59 demonstrates that the phosphate has no effect on gold extractions in the presence of chalcopyrite. The ATS consumption was found to be slightly reduced from 16.3 kg/t to 13.9 kg/t by adding 15.8 mM phosphate.



**Figure 5-60: Effect of trisodium phosphate on gold extraction and thiosulfate consumption in the presence of 2.4% galena**

**(ATS: 0.2 M, AH: 0.9 M, Cu: 1.2 mM, oxygen: 21.1%)**

As previously discussed, galena could reduce gold extraction by thiosulfate. The addition of phosphate was beneficial to gold extraction. Figure 5-60 shows the effect of phosphate in the presence of 2.4% galena. Without adding phosphate, the gold extraction was about 20.0% at 3.0 hours. By adding 10.5 mM sodium phosphate, the gold extraction increased to 53.6% at 0.5 hour and 79.2% at 3.0 hours leaching. The consumption of ATS was slightly increased from 5.0 kg/t to 7.3 kg/t by adding 21 mM phosphate.



**Figure 5-61: Effect of trisodium phosphate on gold extraction and thiosulfate consumption in presence of 4.0% arsenopyrite (ATS: 0.2 M, AH: 0.9 M, Cu: 1.2 mM, oxygen: 21.1%)**

Figure 5-61 shows that phosphate has no influence on the consumption of ATS. However, by the addition of 10.5 mM phosphate, the gold extraction increased from 68.1% to 83.2% at 3.0 hours thiosulfate leaching.

### 5.3.3.1. Discussion

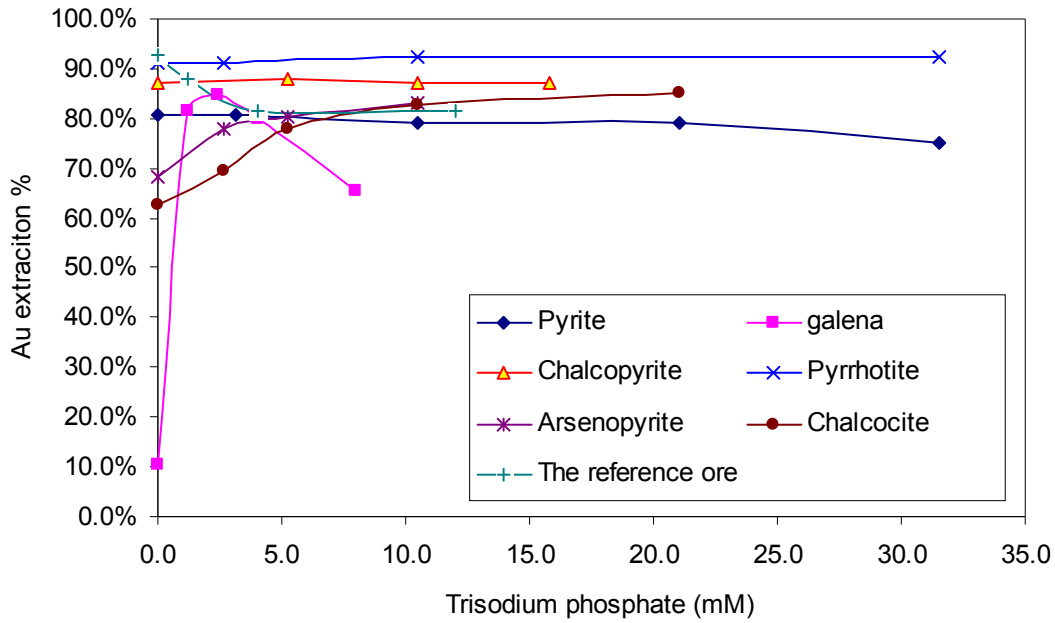


Figure 5-62: Effects of phosphate on gold extractions

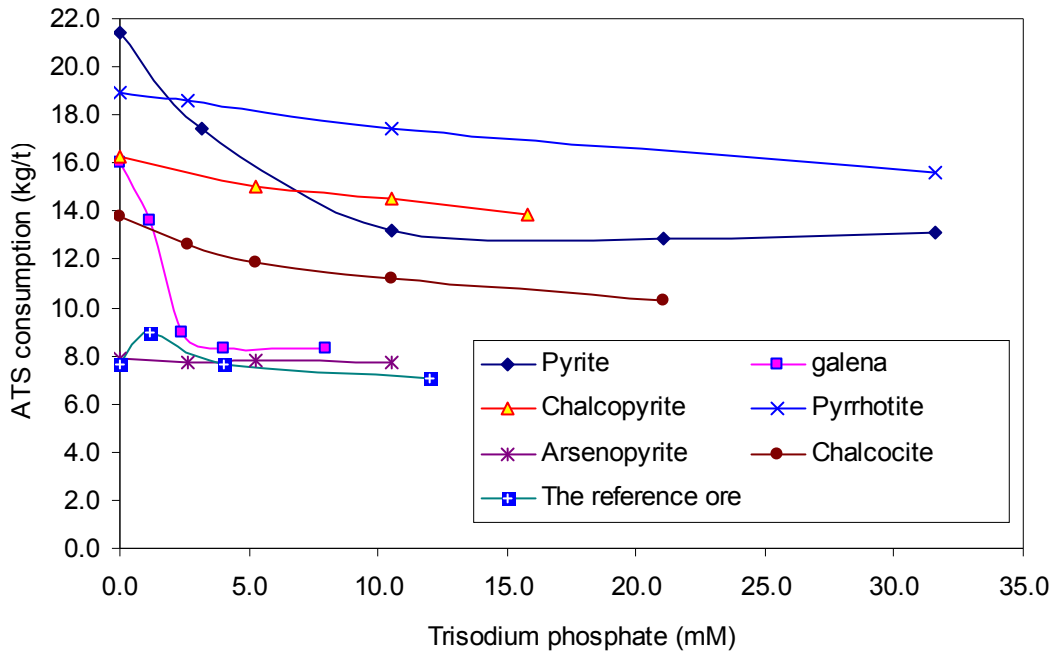


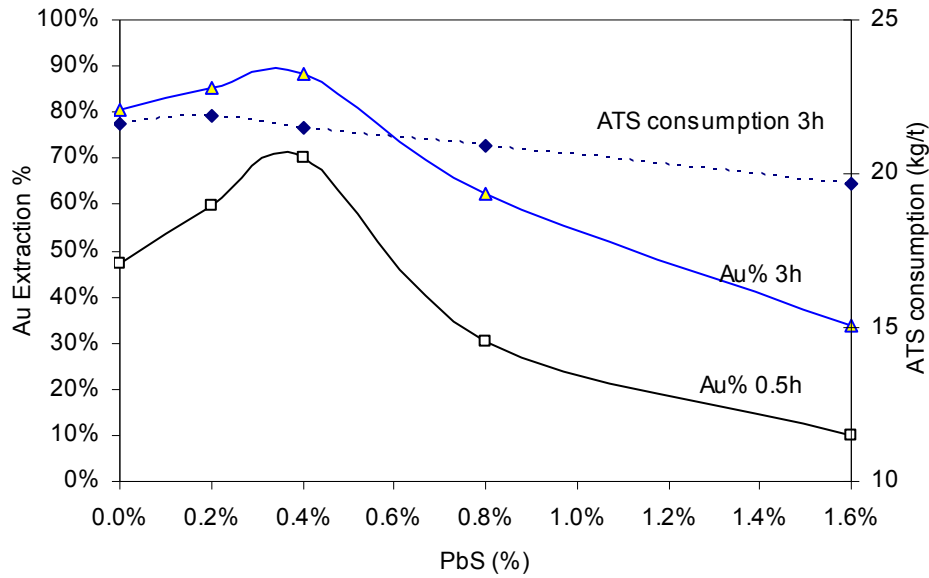
Figure 5-63: Effects of phosphate on ATS consumptions

Figures 5-62 and 5-63 summarize the effect of phosphate on the gold extractions and thiosulfate consumptions. By adding phosphate in the leaching slurry, the gold extractions were increased in the presence of pyrrhotite, chalcocite, galena, and arsenopyrite ores. Phosphate also reduced the ATS consumption in the presence of pyrite, pyrrhotite, chalcopyrite and chalcocite.

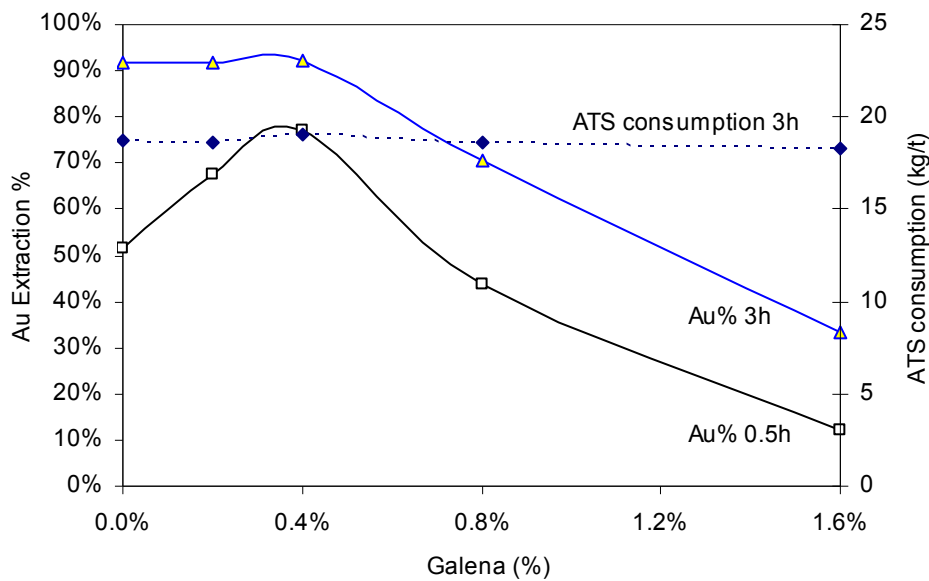
By adding 10.5 mM phosphate, the improvements on gold extractions in the presence of chalcocite, galena and arsenopyrite were 19.7%, 58.1% and 14.9% respectively. The reasons behind these improvements were discussed in Section 3.2.4.2. Phosphate is mainly playing a role as mineral surface passivation agent. By the fact that it works in the presence of some of the most detrimental minerals, i.e., chalcocite, arsenopyrite and galena, it can be concluded that phosphate is a promising additive for gold ores containing these minerals.

#### **5.3.4. Lead ion (Galena)**

As previously discussed, the presence of galena in thiosulfate leaching of gold may result in a largely reduced gold extraction. However, in the leaching of iron sulfide ores, the presence of lead species, i.e. galena, would enhance the gold extraction.



**Figure 5-64: Effect of galena on gold extraction and thiosulfate consumption in the presence of 16% pyrite**  
**(ATS: 0.2 M, AH: 0.9 M, oxygen: 21.1%)**



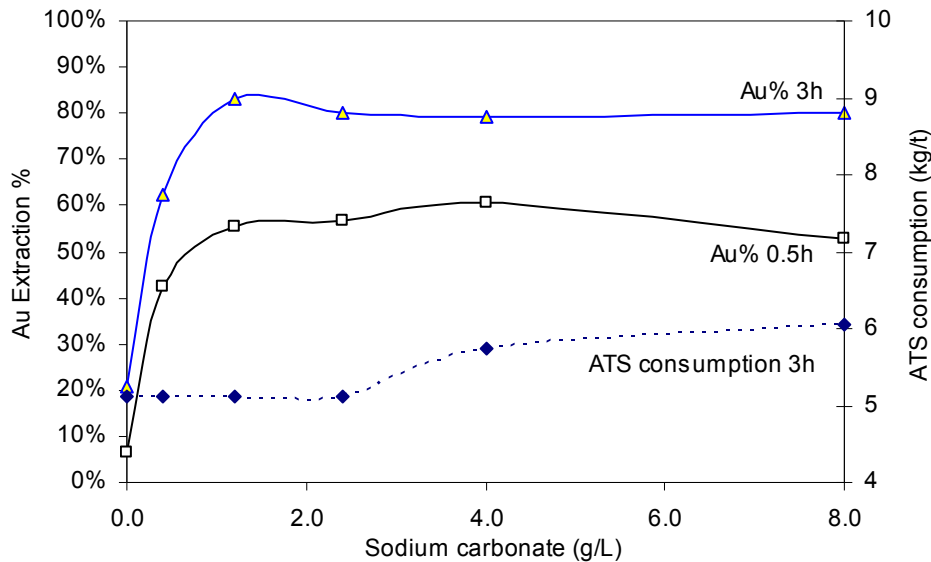
**Figure 5-65: Effect of galena on gold extraction and thiosulfate consumption in the presence of 16% pyrrhotite**  
**(ATS: 0.2 M, AH: 0.9 M, oxygen: 21.1%)**

Figures 5-64 and 5-65 show the result of thiosulfate gold leaching in the presence of pyrite and pyrrhotite. By adding 0.4% galena, the gold extractions at 0.5 hour were improved by 25.8% for pyrrhotite and 23.0% for pyrite. As for the gold extraction at 3.0 hours, the improvements were 7.5% and 1.0% for pyrite and pyrrhotite respectively. The presence of galena had no effect on the ATS consumption in the presence of pyrrhotite. However, the ATS consumption in the presence of pyrite was reduced slightly by about 1.9 kg/t.

In general, dissolvable lead could be a potential additive to thiosulfate leaching system in the presence of reactive surfaces of sulfides. By carefully controlling the addition amount and addition strategy, this ion may be used to passivate some detrimental problems. Other than passivation, this ion may also enhance the oxidation of sulfide as it does in the cyanidation of sulfidic gold ores. This aspect of the consideration will be further investigated in the future.

### **5.3.5. Carbonate anion**

In the thiosulfate leaching of galena bearing ore, due to the precipitation of lead species on the gold surface, the extraction of gold can be very difficult. As suggested in section 3.2.4, carbonate might improve the gold extraction by thiosulfate in the presence of lead minerals.



**Figure 5-66: Effect of  $\text{Na}_2\text{CO}_3$  on gold extraction and thiosulfate consumption in the presence of 2.4% galena (ATS: 0.2 M, Cu: 1.2 mM, AH: 0.9 M, open-air)**

Figure 5-66 depicts the significant improvement of the carbonate anion has on thiosulfate gold leaching in the presence of galena. The gold extraction in the presence of 2.4% galena was as low as 20.9%. With the addition of 11.3 mM sodium carbonate, the gold extraction was improved to 55.2% at 0.5 hour and 83.2% at 3.0 hours. The addition of 75.5 mM sodium carbonate slightly increased the consumption of ATS by 1.0 kg/t.

Sodium carbonate is an inexpensive chemical that can be applied in the leaching of gold ores containing excessive dissolvable lead. The mechanism has been discussed in Section 3.2.4, Carbonate was used as a passivation agent that prevents the dissolution of lead and the formation of inert film on gold surface.

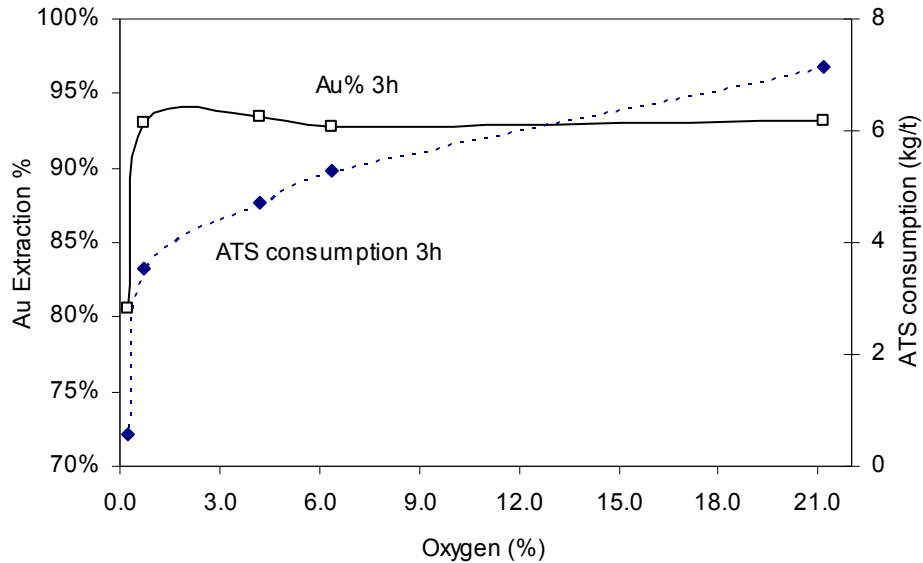
### **5.3.6. Summary**

In Section 5.3, a few examples were given for improving thiosulfate leaching performances with chemical additives, i.e.,  $\text{OH}^-$ ,  $\text{PO}_4^{3-}$ ,  $\text{Ca}^{2+}$ ,  $\text{Pb}^{2+}$ , and  $\text{CO}_3^{2-}$ . All these additives appeared to be beneficial to either the gold extraction or the thiosulfate stability of thiosulfate leaching in the presence of various detrimental minerals. Most of these additives are inexpensive and widely available in bulk chemical market. Therefore, the use of these chemicals in thiosulfate leaching is a promising approach for counteracting the problems of minerals. Most of these additives tend to enhance the passivation of the surfaces of minerals or tend to mask the harmful ions. Therefore, the previously suggested theory on how to improve thiosulfate leaching were verified through the examples in this section.

## **5.4. Thiosulfate leaching with limited oxygen supply**

The objective of this section is to improve the stability of thiosulfate leaching system in the presence of various minerals by oxygen control. As discussed in Section 3.2.2, one of the theoretical methods for improving the leaching performance is to limit the oxygen supply so that the oxidative strength is high enough for gold dissolution but not too high to consume thiosulfate reagents. Leaching tests under limited oxygen supply conditions will be discussed both on the silicate reference ore and synthetic ores containing various sulfide minerals.

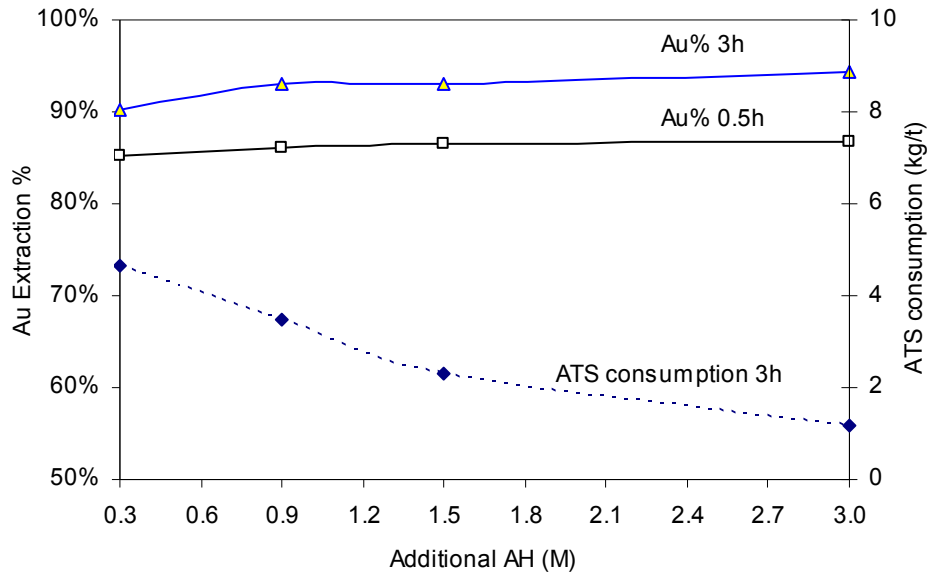
### 5.4.1. The reference ore



**Figure 5-67: Effect of oxygen percentage in the leaching of the reference ore (ATS: 0.20 M, Cu: 1.2 mM, AH: 0.9 M)**

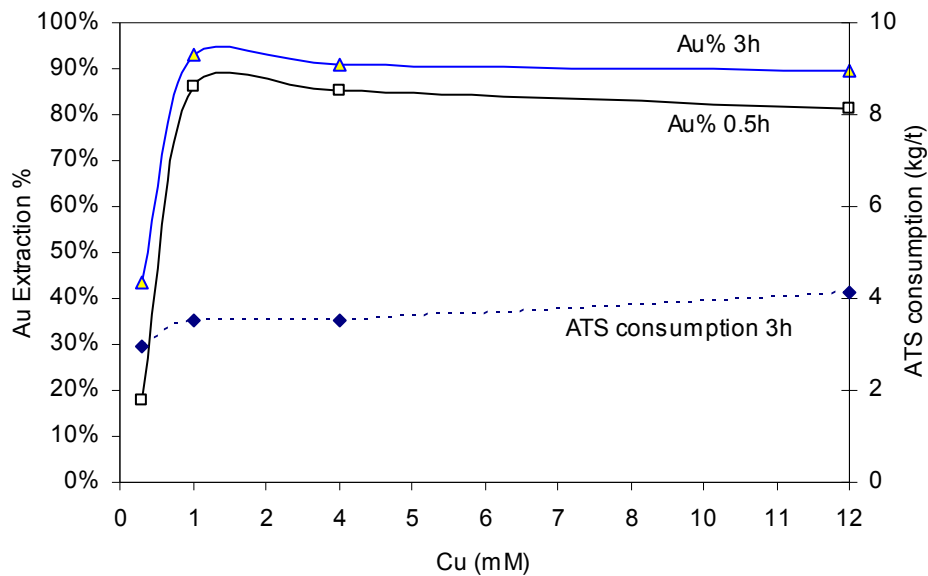
Figure 5-67 shows the effect of oxygen on the gold extraction with thiosulfate and thiosulfate consumption from the reference ore. The oxygen percentage refers to the oxygen content in the gas circulating into the leaching vessel.

In the leaching of the reference ore in the absence of minerals impurities, only 80.6% gold was leached at 0.2% oxygen. The gold extraction increased to 93.0% at 0.7% oxygen. A further increase of oxygen percentage beyond 0.7% resulted in slightly lower extra gold extraction. The consumption of ATS was 7.2 kg/t at 21.1% oxygen, which was reduced by half at 0.7% oxygen. Obviously, within the range from 0.7% to 21.1%, the increased oxygen supply accelerated thiosulfate decomposition and decreased the gold extraction. Therefore, the oxygen supply should remain at 0.7% through out the next steps of the rest works.



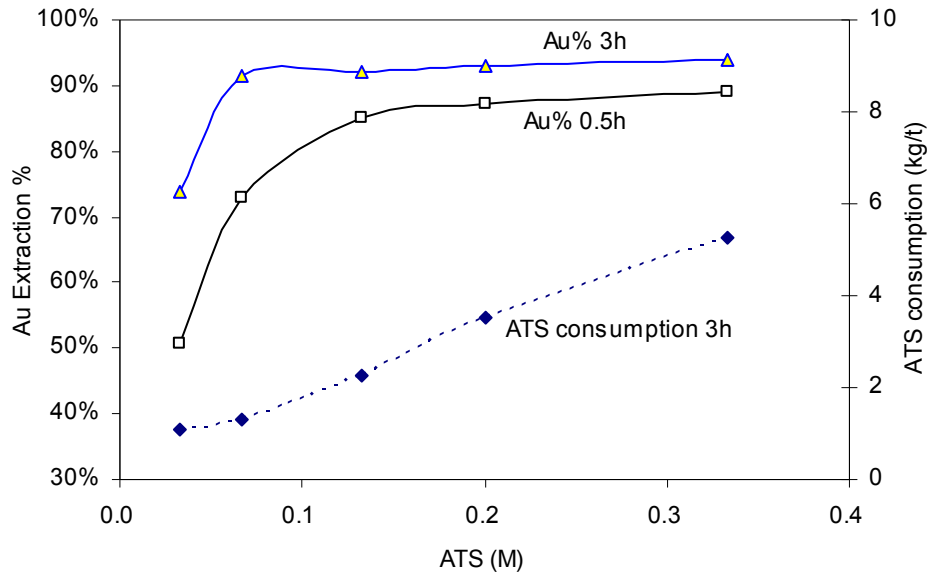
**Figure 5-68: Effect of ammonia concentration on gold extraction and thiosulfate consumption under limited oxygen content.**

(ATS: 0.20 M, Cu: 1.2 mM, O<sub>2</sub>: 0.7%)



**Figure 5-69: Effect of copper concentration on gold extraction and thiosulfate consumption under limited oxygen content.**

(ATS: 0.20 M, AH: 0.9 M, O<sub>2</sub>: 0.7%)



**Figure 5-70: Effect of ATS concentration on gold extraction and thiosulfate consumption under limited oxygen content.**

**(AH: 0.9 M, Cu: 1.2 mM, O<sub>2</sub>: 0.7%)**

Under limited oxygen content in the sparging gas, e.g., 0.7% O<sub>2</sub>, the effects of lixiviant compositions such as thiosulfate, copper and ammonia concentrations were studied and the results are shown in Figures 5-68, 5-69, and 5-70.

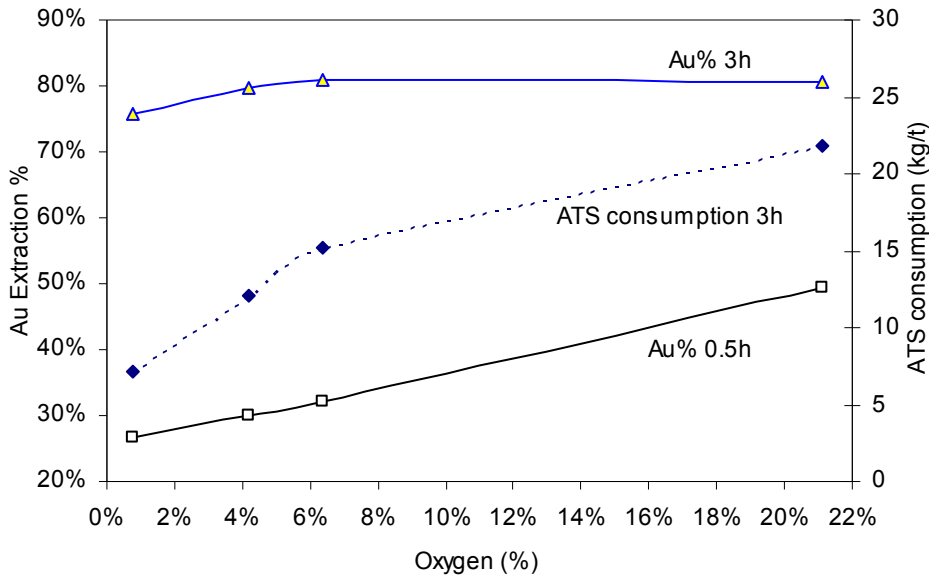
By increasing the ammonia concentration from 0.3 to 3.0 M, the gold extraction at 3.0 hours increased from 90.1% to 94.3% and the consumption of ATS was reduced from 4.6 to 1.2 kg/t (Figure 5-68). The optimum thiosulfate concentration ranged from 0.13 to 0.33 M produced 92.1% to 93.8% gold extractions. The consumption of ATS increased by 4.2 kg/t with an increase of ATS concentration from 0.03 to 0.333 M (Figure 5-70).

The increase of copper concentration did not result in a significant increase of thiosulfate consumption. The optimum copper concentration was found to be 1.2

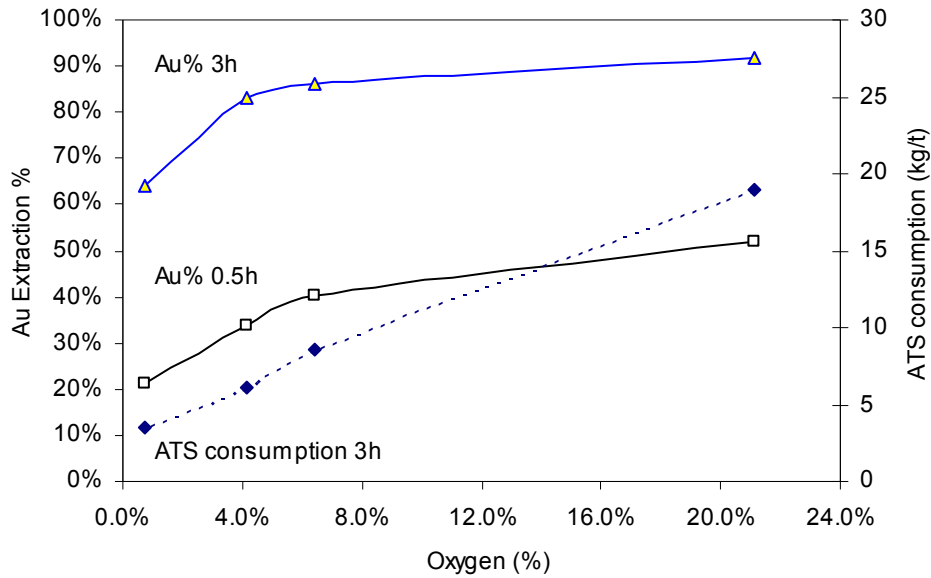
mM. With an insufficient addition of copper (i.e., 0.36 mM), the gold extraction was very low, i.e., 43.3% (Figure 5-69).

#### 5.4.2. Pyrite, pyrrhotite and arsenopyrite and chalcopyrite ores

In the presence of sulfides, however, the reduced oxygen content in the gas might slow down the oxidation rate of the reactive sulfide surfaces and the gold extraction might be also reduced.



**Figure 5-71: Effect of oxygen percentage on gold extraction and thiosulfate consumption in the presence of 16.0% pyrite (Cu: 1.2 mM, ATS: 0.20 M, AH: 0.9 M)**

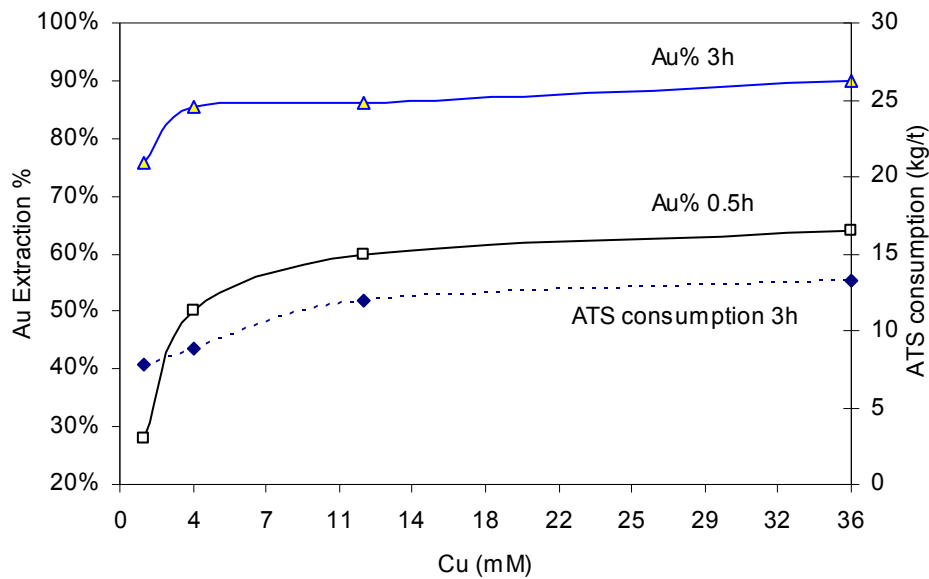


**Figure 5-72: Effect of oxygen percentage on gold extraction and thiosulfate consumption in presence of 16.0% pyrrhotite (Cu: 1.2 mM, ATS: 0.20 M, AH: 0.9 M)**

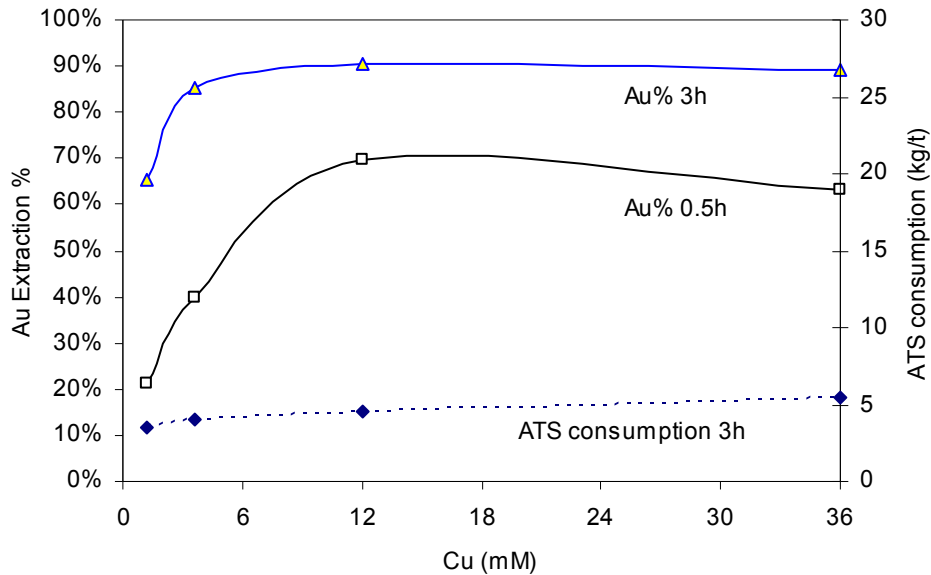
Figures 5-71 and 5-72 depict the effects of the oxygen percentage on gold extractions and thiosulfate consumptions in the presence of pyrite and pyrrhotite. It was previously concluded that the leaching of the reference ore in the absence of mineral impurities required only 0.7% oxygen in the gas content. However, in the presence of iron sulfides, the reduction of the oxygen percentage significantly reduced the gold extraction by thiosulfate. As shown in Figure 5-71, the gold extractions from the pyrite-bearing ore at 21.1% oxygen were 49.4% at 0.5 hour and 80.5% at 3.0 hours. When the oxygen percentage was reduced to 0.7%, the gold extraction decreased to 26.7% and 75.8% respectively. Similar reduction of gold extractions was also obtained in the leaching of pyrrhotite ore (Figure 5-22);

i.e. the gold extraction was reduced from 92.0% at 21.14% oxygen to 63.9% at 0.7% oxygen supply.

Reducing the oxygen percentage from 21.1% to 0.7% resulted in a reduced ATS consumption. The reductions were 14.6 kg/t and 15.5 kg/t respectively in the presence of pyrite and pyrrhotite.



**Figure 5-73: Effect of copper on gold extraction and thiosulfate consumption in the presence of 16.0% pyrite (ATS: 0.20 M, AH: 0.9 M, oxygen: 0.7%)**

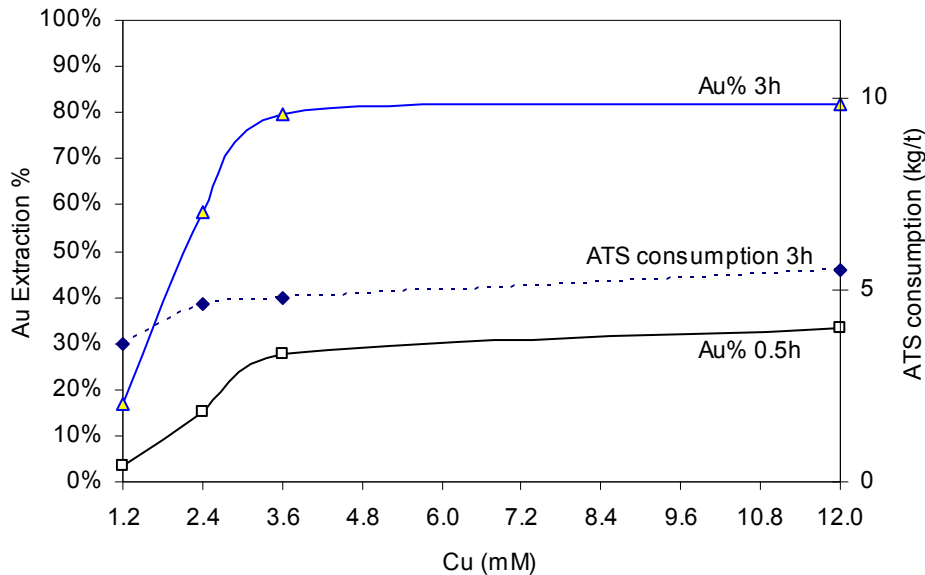


**Figure 5-74: Effect of Cu on gold extraction and thiosulfate consumption in presence of 16.0% pyrrhotite (ATS: 0.20 M, AH 0.9 M, oxygen 0.7%)**

The leaching performances of iron sulfides ore was largely reduced under limited oxygen supply conditions (i.e. 0.7% oxygen). However, by adjusting the lixiviant compositions such as copper concentration, the reduced gold extractions can be increased. Figures 5-73 and 5-74 demonstrate the effect of copper concentration on gold extraction by thiosulfate in the leaching of pyrite and pyrrhotite bearing gold ores with limited oxygen supply.

As shown in Figure 5-73 (pyrite ore), with 0.74% oxygen and 1.2 mM of copper, the gold extractions were 28.0% at 0.5 hour and 75.6% at 3.0 hours. With more copper (36 mM), the gold extractions increased to 63.9% and 89.9% respectively. More copper addition also resulted in an increase of ATS consumption. When the Cu concentration increased from 1.2 to 36 mM, the ATS consumption increased

from 7.8 to 13.2 kg/t. In the test on pyrrhotite ore (Figure 5-74), similar result was observed. By using 12 mM copper instead of 1.2 mM, the gold extractions were increased from 21.0% to 69.9% at 0.5 hour and 65.4% to 90.6% at 3.0 hours. The increase of ATS consumption, however, was very little (by 1.1 kg/t).



**Figure 5-75: Effect of Cu on gold extraction and thiosulfate consumption in the presence of 4.0% arsenopyrite (ATS: 0.2 M, AH: 0.9 M, oxygen: 0.7%)**

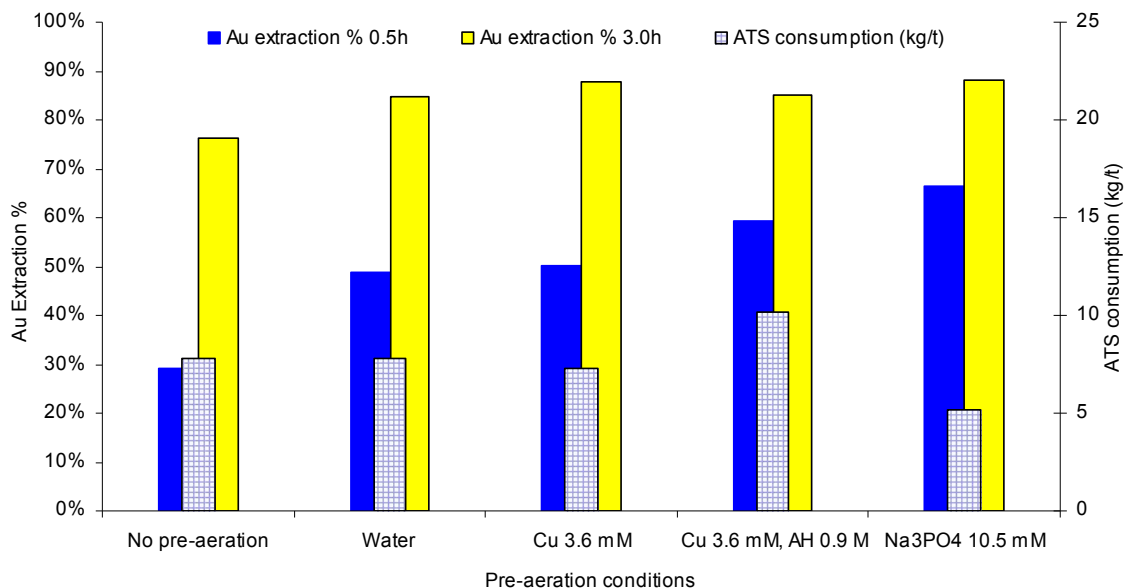
Figure 5-75 shows the effect of copper concentration on the gold extraction and the thiosulfate consumption in the leaching of the arsenopyrite bearing gold ore with limited oxygen supply (0.7% oxygen). With 1.2 mM copper, the overall gold extraction was only 17.0% at 3.0 hours. However, when the copper concentration was increased from 1.2 mM to 12 mM, the gold extraction increased to 81.7%. The thiosulfate consumption increased from 4 kg/t to 6 kg/t.

Table 5-8 lists the gold extraction and thiosulfate consumption in the presence of 8% chalcopyrite under reduced oxygen content (0.7% oxygen). Within the range of 0.4 to 12.0 mM of copper, the gold extraction increased with the increased amount of copper addition. The gold extraction at 3.0 hours reached 86.2% by adding 12 mM copper. The consumption of ATS was also increased with the increase of copper. The increase of copper concentration from 0.4 mM to 12.0 mM resulted a 4.3 kg/t increase of thiosulfate consumption from 4.7 to 9.0 kg/t.

**Table 5-8: Effect of Cu on gold extraction and thiosulfate consumption in the presence of 8% chalcopyrite**

Leaching conditions				Gold extraciton (%)		ATS consumption (kg/t)
ATS (M)	Cu (mM)	AH (M)	O2%	0.5h	3.0h	3.0h
0.20	0.4	0.9	0.7	19.6%	55.7%	4.7
0.20	1.2	0.9	0.7	31.1%	62.1%	6.3
0.20	3.6	0.9	0.7	52.2%	80.2%	8.2
0.20	12.0	0.9	0.7	64.7%	86.2%	9.0

### 5.4.3. Leaching of gold with limited oxygen after pre-aeration



**Figure 5-76: Thiosulfate leaching of gold in the presence of pyrite with reduced oxygen supply after pre-aeration**

**(Pre-treatment conditions: 1.0 hour, 21.1% oxygen, and 16.0% pyrite, Leaching conditions: 0.20 M ATS, 0.9 M AH, 0.7% oxygen, and 1.2 mM Cu)**

Another approach for enhancing the gold extraction in the thiosulfate leaching with limited oxygen supply is to pre-condition the surfaces of minerals with or without copper, ammonia and phosphate for the duration of one hour. The oxygen contents in the pre-conditioning were 21.1% and 0.7% in the thiosulfate gold leaching.

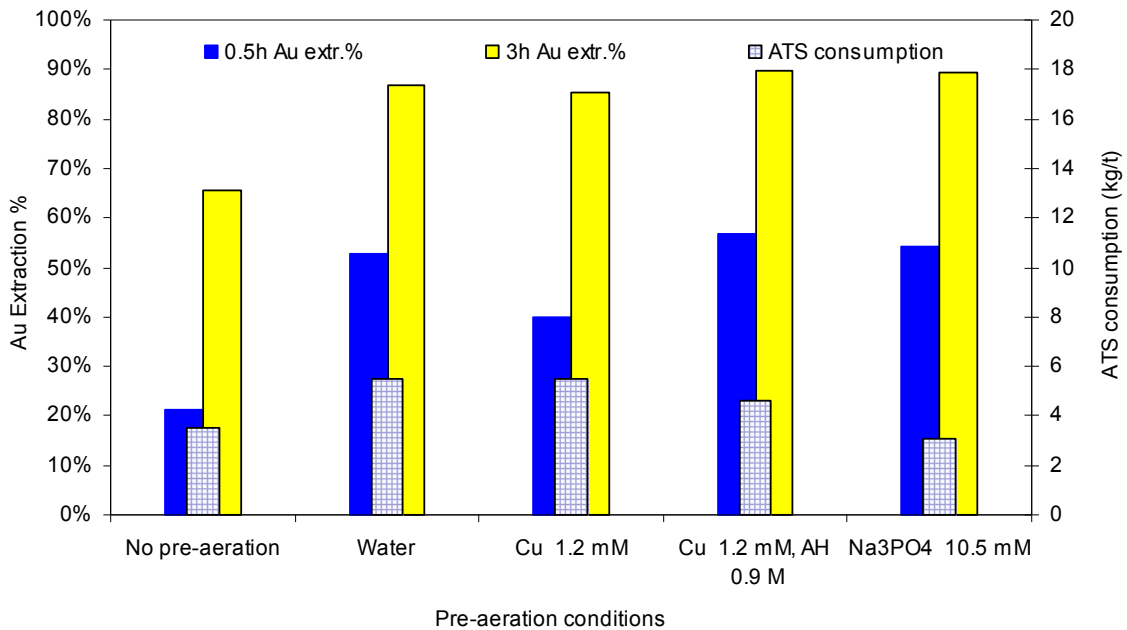
The dark and light bars in Figure 5-76 represent the gold extractions at 0.5 hour and 3.0 hours. The bars with the net pattern represent the consumption of ATS. The copper concentration in the leaching process was 1.2 mM, unless more copper was added in the pre-treatment process. The pulp density remained the same (i.e., 33.3%) in both the pre-aeration and the leaching process. The slurry was not filtered after pre-treatment and the lixivants were added directly to the pre-conditioned slurry.

Without pre-aeration, the gold extraction was 29.2% at 0.5 hour and 76.3% at 3.0 hours. By pre-conditioning with water, the gold extractions were improved to 48.9% and 84.8% respectively at 0.5 hour and 3.0 hours. The ATS consumption was 7.8 kg/t for both cases.

By adding 3.6 mM of copper in the pre-condition, the gold extraction was 50.1% at 0.5 hour and 88.0% at 3 hours. With 0.9 M ammonia in the pre-treatment

together with 3.6 mM copper, the gold extractions reached 59.3% at 0.5 hour and 85.1% at 3.0 hours.

The optimum pre-aeration conditions were adding 10.5 mM Na<sub>3</sub>PO<sub>4</sub>. The gold extractions under this condition were 66.4% (0.5 hour) and 88.0% (3.0 hours) respectively and the ATS consumption decreased to 5.2 kg/t.

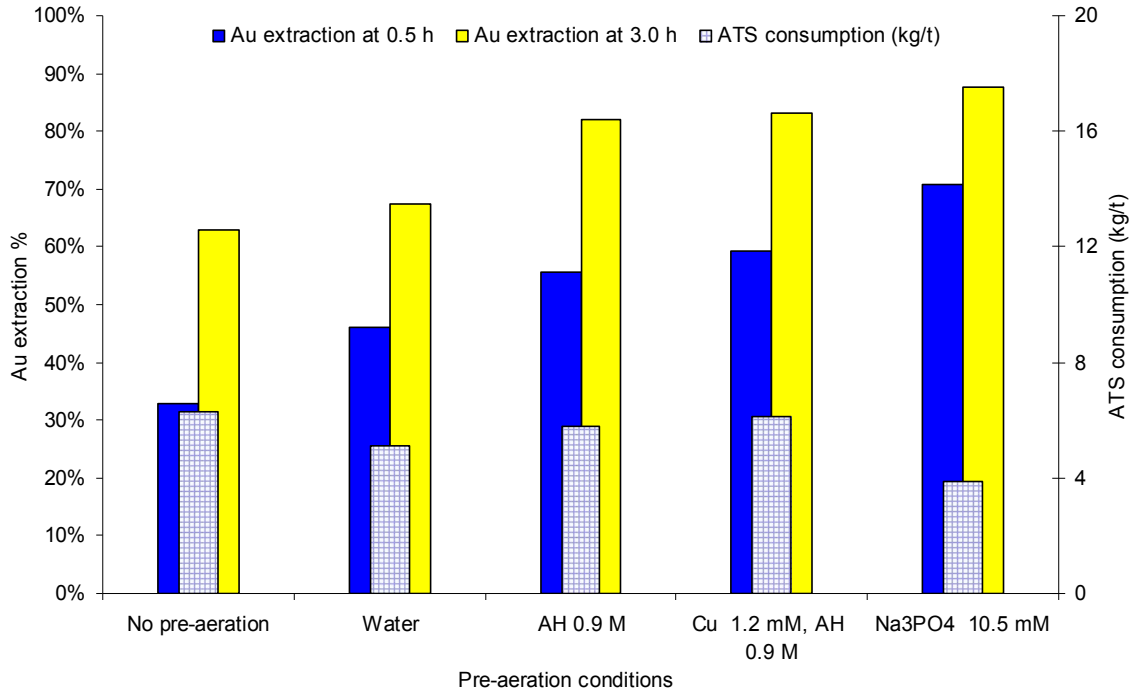


**Figure 5-77: Thiosulfate gold leaching in the presence of pyrrhotite with reduced oxygen supply after pre-aeration**

**(Pre-treatment conditions: 1.0 hours, 21.1% oxygen, and 16% pyrrhotite; Leaching conditions: 0.20 M ATS, 0.9 M AH, 0.7% oxygen, and 3.6 mM Cu)**

Similar pre-conditioning was also conducted on the pyrrhotite ore. The results are shown in Figure 5-77. Without pre-conditioning, thiosulfate leaching with 0.7% oxygen extracted 21.1% gold at 0.5 hour and 65.5% at 3.0 hours. By pre-conditioning the ore with water for 1.0 hour, the gold extraction increased to

52.8% and 86.6% respectively at 0.5 hour and 3.0 hours. Pre-conditioning with 3.6 mM copper or 3.6 mM copper and 0.9 M ammonia also enhanced the gold extraction. With 10.5 mM  $\text{Na}_3\text{PO}_4$ , the gold extraction reached 89.46% at 3.0 hours with 3.1 kg/t ATS consumed.



**Figure 5-78: Thiosulfate gold leaching in the presence of chalcopyrite with reduced oxygen supply after pre-aeration (Pre-treatment conditions: 3.0 hours, 21.1% oxygen, and 8% chalcopyrite; Leaching conditions: 0.20 M ATS, 0.9 M AH, 0.7% oxygen, and 1.2 mM Cu)**

Figure 5-78 displays the thiosulfate gold leaching result in the presence of chalcopyrite. A pre-aeration with 21.1% oxygen was applied in prior to gold leaching with reduced oxygen concentration. Without pre-treatment, thiosulfate leaching with 0.7% oxygen extracted 32.9% gold at 0.5 hour and 63.0% at 3.0 hours. By pre-conditioning the ore with water for 3.0 hours, the gold extraction increased to 46.2% and 67.5% respectively at 0.5 hour and 3.0 hours. Pre-

aeration with 0.9 M ammonia provided more significant improvement on the gold extractions. The gold extractions increased to 82.0% at 3.0 hours. The addition of 1.2 mM copper in the pre-aeration with ammonia further increased the gold extraction to 83.3%. The best gold extractions were received when 4 g/L trisodium phosphate in the pre-treatment stage. The gold extraction under these conditions reached 87.7% and the ATS consumption was reduced to 3.9 kg/t.

#### **5.4.4. Summary**

The mechanism of thiosulfate leaching under reduced oxygen supply has been discussed in Section 3.2.2. The results from this section verify the previously concluded theory. In the absence of sulfide minerals, the thiosulfate gold leaching can be conducted at very low oxygen content (0.7% oxygen in the gas supply) with satisfactory gold extraction. However, in the presence of sulfide minerals, the gold extractions were significantly lower at low oxygen content.

Pre-aeration can be used to condition the sulfide bearing gold ore. After such a pre-conditioning, a thiosulfate gold leaching under reduced oxygen supply was effective and the gold extractions were acceptable.

### **5.5. Thiosulfate leaching with thiosulfate-EDA-copper solution**

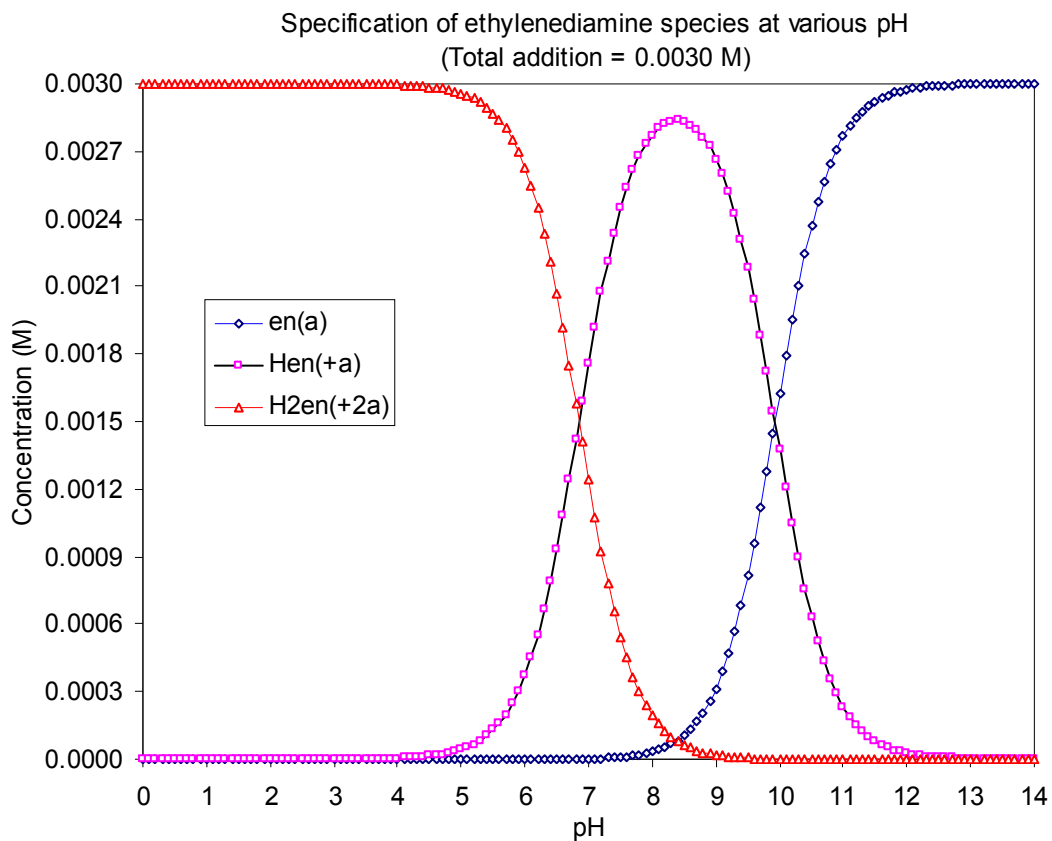
In this section, a leaching system using ethylenediamine (EDA or en) and Cu as the oxidation catalyst in absence or presence of ammonia will be discussed. The properties of ethylenediamine (EDA or en) will be introduced. These properties

predict that this chemical could be used to build a more stable thiosulfate leaching system and to minimize the detrimental effects of some of the associated minerals.

A thermodynamic analysis of this system will also be included in this section. The results from the leaching tests on both the reference quartz gold ore and the synthetic ores will be discussed to verify the variables of this leaching system and demonstrating improvement of gold extraction and reduction of thiosulfate consumption in the presence of some detrimental minerals.

#### **5.5.1. Properties of ethylenediamine**

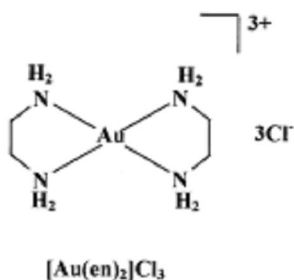
Ethylenediamine ( $\text{NH}_2(\text{CH}_2)_2\text{NH}_2$  or  $\text{C}_2\text{H}_8\text{N}_2$ ) is also known as “1,2-Ethanediamine” or “1,2-diaminoethane”. “EDA” or “en” was used as abbreviations of ethylenediamine. Pure EDA is corrosive and flammable. The evaporation of pure EDA liquid is easy and fast. However, once it is dissolved in water, the solution is very stable without any significant evaporation problem,



**Figure 5-79: Specification of en, Hen<sup>+</sup> or H<sub>2</sub>en<sup>2+</sup>**  
(Total EDA concentration: 3 mM)

After reacting with the hydrogen ion, EDA produces Hen<sup>+</sup> or H<sub>2</sub>en<sup>2+</sup> (Equations 5-1 and 5-2). The three forms of EDA in aqueous solution were studied for their relations (Figure 5-79). In neutral to base solutions, the two stable forms of EDA are en and Hen<sup>+</sup>.





**Figure 5-80: Molecular structure of  $\text{Au}(\text{en})_2^{3+}$**

EDA has been extensively used in the gold plating industry as a stabilizer of gold dissolved in the sulfite bath solutions. The reaction between  $\text{AuCl}_4^-$  and EDA to form  $\text{Au}(\text{en})_2^{3+}$  was investigated and reported. The structure of  $[\text{Au}(\text{en})_2]\text{Cl}_3$  is demonstrated in Figure 5-80 [Marcon, et al., 2003]. The fast formation of  $\text{Au}(\text{en})_2^{3+}$  indicates the strong chelating ability of EDA with Au(III) [Bekker et al., 1972] [Louw et al., 1969]. However, the complex stability of  $\text{Au}(\text{en})_2^{3+}$  and EDA-Au(I) complex has not been reported in the literatures.

The combination of EDA and sulfite was reported in some gold plating technologies, where gold is stabilized by EDA as  $\text{Au}(\text{en})_2^{3+}$ . The presence of  $\text{Au}(\text{en})\text{SO}_3^-$  was also suggested in this type of plating electrolytes [Zuntini et al., 1980, 1974, Losi et al., 1972]. An electrochemistry study on the polarization curves of the gold electrode in solutions containing EDA provided further evidence for the existence of the gold EDA complex [Doesburg et al., 2000]. In the presence of significant amount of reducing agents, such as thiosulfate,  $\text{Au}(\text{en})_2^{3+}$  is not a stable species and will be reduced to different aurous complexes (possibly  $\text{Au}(\text{en})\text{S}_2\text{O}_3^-$ ).

EDA forms various complexes with copper ions. For the Cu(I)-EDA complex ( $\text{Cu(en)}_2^+$ ), the log value of the formation constant (Log K) is 10.80, which is lower than that of Cu(I)-thiosulfate complex ( $\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$ ,  $\log K=13.84$ ) and similar to that of Cu(I)-ammonia complex ( $\text{Cu}(\text{NH}_3)_4^{2+}$ ,  $\log K=10.86$ ). Therefore, in a thiosulfate solution,  $\text{Cu(en)}_2^+$  is not a stable species. The Cu(II) EDA ( $\text{Cu(en)}_2^{2+}$ ) complex with Log K value at about 20.0 is much more stable

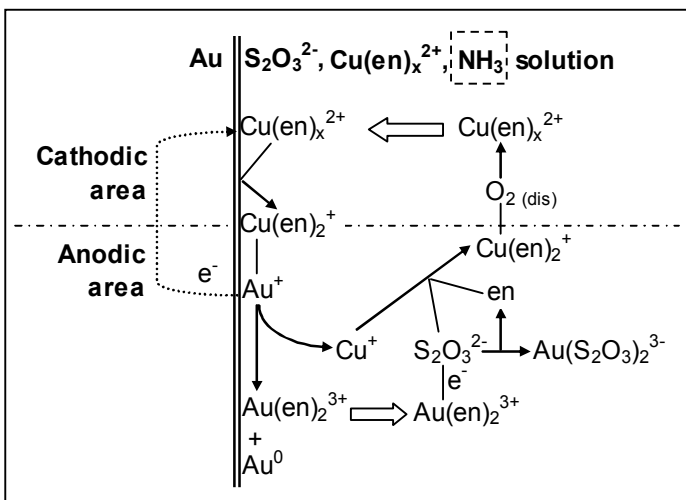
With a sufficient addition of EDA,  $\text{Cu(en)}_2^{2+}$  is the most common form of all cupric complexes. It is much more stable than  $\text{Cu}(\text{NH}_3)_4^{2+}$ . Therefore, similar to EDTA, EDA is a stronger stabilizer of copper in base solution than ammonia.

The association of the thiosulfate anion with Cu(II)-EDA complex was also reported, which indicates the possible existence of a thiosulfate-Cu-EDA mixed complex [Matheson et al., 1967] [Blokhin et al., 1971]. Another possible mixed complex,  $[\text{Cu}(\text{en})_2(\text{S}_2\text{O}_3)]$ , was also observed. The log value of the stability constant was reported to be 21.34 [Popel et al., 1975]. The high degree of polarization in the electro-deposition of copper from an electrolyte containing copper ethylenediamine was found to be affected by thiosulfate anion, which may displace the inhibiting film of ethylenediamine-Cu complexes from the cathode surface [Gerenrot et al., 1972].

In Au(I) solutions without a strong ligand, EDA may trigger a disproportionate reaction to generate Au(0) and Au(III). This reaction explains the functions of EDA in sulfite–Au(I) plating solutions [Kato et al., 2004].

Another important property of EDA is that elemental sulfur can be dissolved in this organic solvent [Srivastava et al., 1988; Matthew, 1966]. The capability of reacting with elemental sulfur would suggest that EDA could affect the formation of elemental sulfur on the surface of gold.

### 5.5.2. Thermodynamic consideration of thiosulfate-EDA-copper leaching system

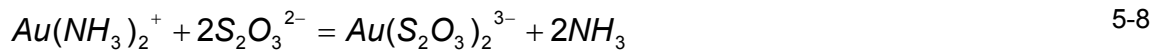
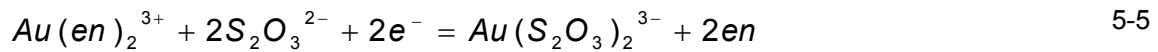
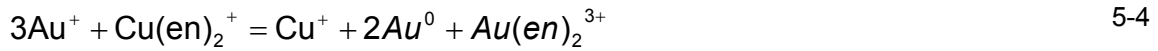


**Figure 5-81: A proposed half reaction model for gold dissolution in thiosulfate-EDA-copper solution (ammonia or ammonium anion is optional)**

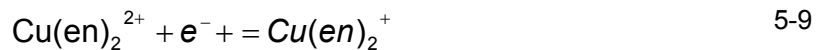
Figure 5-81 depicts a possible reaction model of the thiosulfate-EDA-copper leaching system. On the surface of the anodic area, the elemental gold loses one electron and is oxidized to aurous ion. A disproportionation reaction of aurous ion produces  $Au(en)_2^{3+}$  and  $Au^0$ . The  $Au(en)_2^{3+}$  diffuses to the bulk solution phase and is reduced and complexed by thiosulfate to  $Au(S_2O_3)_2^{3-}$ . The cuprous ion that is released from  $Cu(en)_x^+$  was complexed by EDA and oxidized to  $Cu(en)_x^{2+}$  by dissolved oxygen or other oxidant in the bulk solution phase. On the surface of

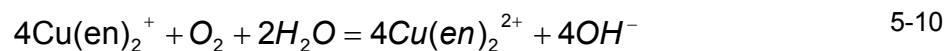
the cathodic area,  $\text{Cu(en)}_x^{2+}$  is preferentially adsorbed. As the direct oxidant of gold, it accepts the electron and produces  $\text{Cu(en)}_x^+$ . The reactions proposed by this model are listed as Equation 5-3 to 5-11.

Anodic area:

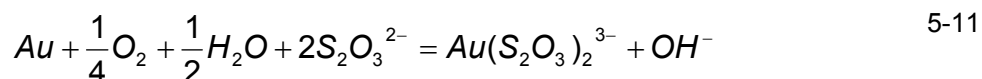


Cathodic area:



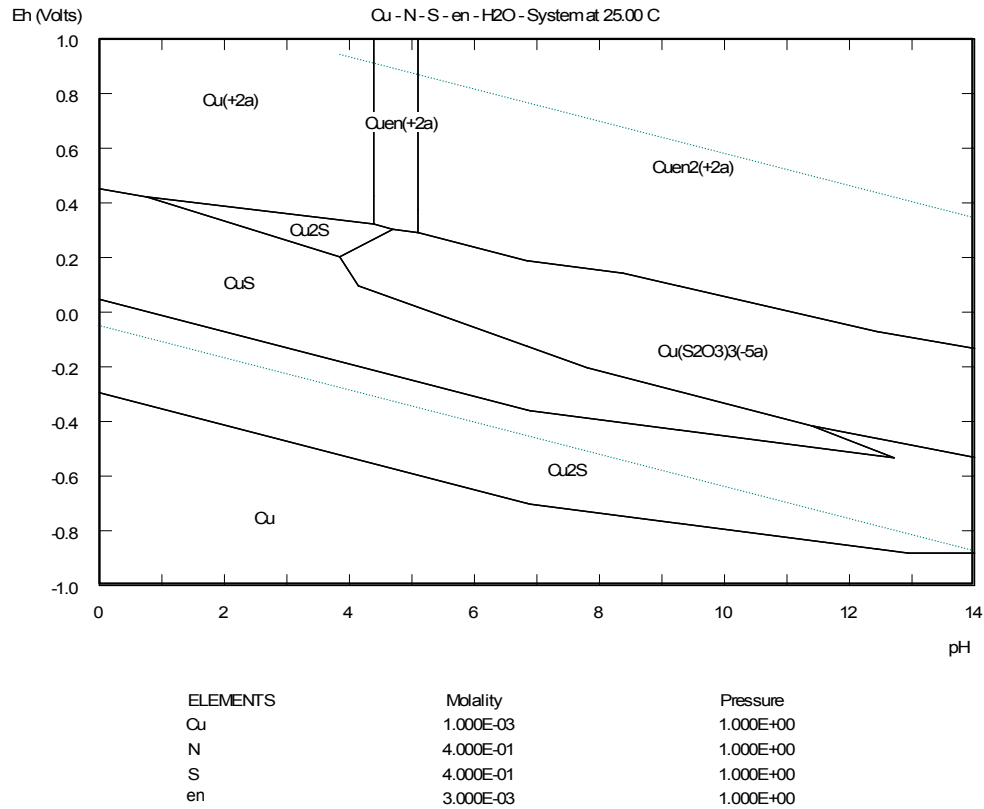


Overall:



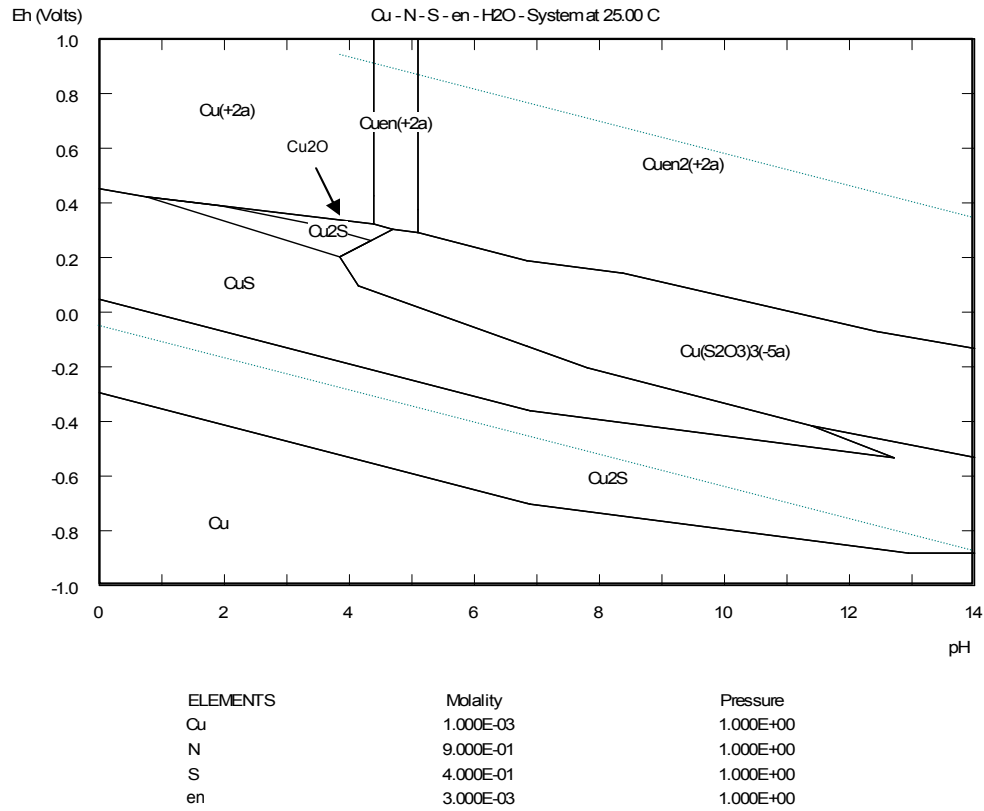
The whole process does not require ammonia. However, considering that ammonia is also easily adsorbed on gold surface, it could react with the aurous ion to produce  $\text{Au}(\text{NH}_3)_2^+$ . Once this complex leaves the surface of the gold, it is replaced by an aurous thiosulfate complex. Due to the lack of knowledge about the stability constant of  $\text{Au}(\text{en})_2^{3+}$  in comparison to that of  $\text{Au}(\text{NH}_3)_2^+$ , the actual reactions of thiosulfate-EDA-ammonia-copper system on gold surface are not known.

In the theoretical study of thiosulfate-EDA-copper system, the decomposition of EDA is not investigated due to the lack of thermodynamic data and the fact that EDA is stable under typical leaching conditions. This study focused on how this copper ligand changes the chemistry of oxidation catalyst in thiosulfate leaching system.



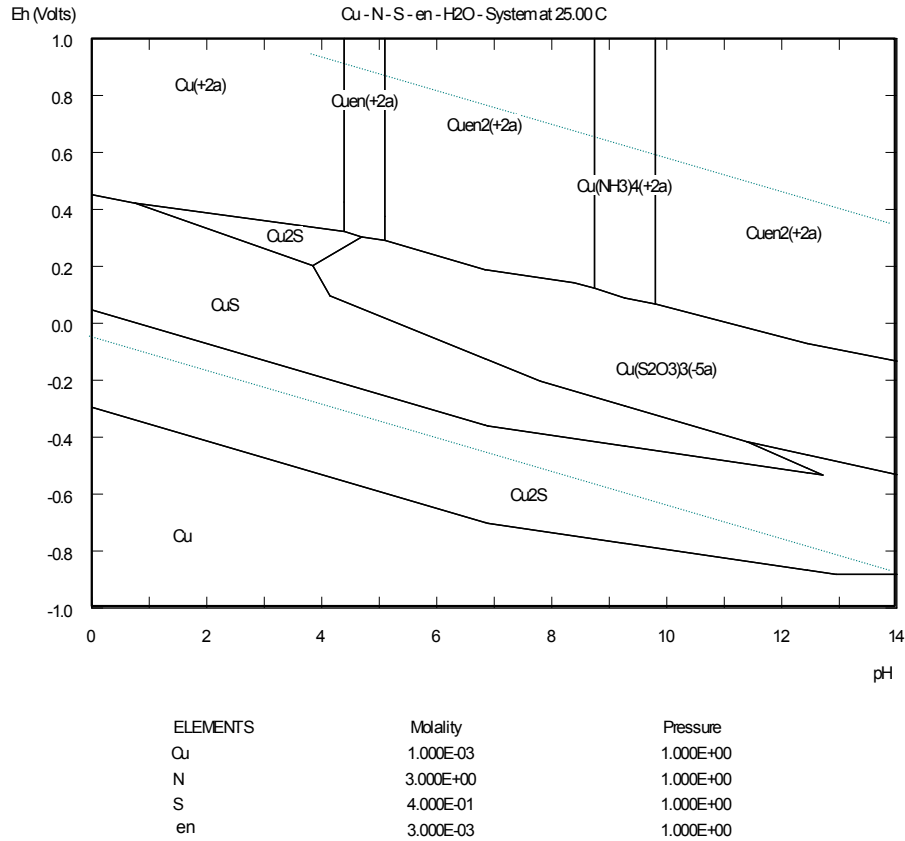
**Figure 5-82: Eh-pH diagram of thiosulfate-EDA-ammonia-copper system (I)**

Figure 5-82 demonstrates the chemical equilibrium of an EDA leaching system using ATS as the lixiviant.  $\text{Cu(en)}_2$  has a very wide stable area, i.e., from pH 5.1 to 14. For the full range of pH, there is no copper amine window in the presence of 3 mM EDA. The oxidation potential of  $\text{Cu(en)}_2$  drops with increasing pH. Between pH 9 and pH 10, the typical pH range for EDA thiosulfate leaching, the oxidation potential falls from 0.1 to 0.0 V.



**Figure 5-83: Eh-pH diagram of thiosulfate-EDA-copper system (II)**

The thermodynamic analysis of a thiosulfate-EDA-copper leaching system was shown in Figure 5-83. The reaction equilibrium is almost identical to that of the thiosulfate-EDA-ammonia-copper leaching system.



**Figure 5-84: Eh-pH diagram of thiosulfate-EDA-ammonia-copper system (III)**

However, at high ammonia concentration, i.e., 3.0 M, the thiosulfate-EDA-ammonia-copper leaching system has a different equilibrium state. As shown in Figure 5-84, a narrow cupric ammine window appears in the middle of the stable area of  $\text{Cu(en)}_2$ . It was also found that the ammonia window starts to show when the concentration of ammonia exceeds 1.65 M.

Through the analysis of the thiosulfate-EDA-copper system, it can be concluded that this system is thermodynamically capable of oxidizing gold. However, the reduction potential of the Cu-EDA complex is lower than that of the Cu-ammonia

complex. Therefore, the thiosulfate stability in thiosulfate-EDA-copper system should be higher than in the thiosulfate-ammonia-copper system.

Ammonia at a lower concentration, i.e. <1.65 M does not participate in the oxidation catalysis of gold. However, in the presence of ammonia or ammonium, gold dissolution may be enhanced due to the formation of  $\text{Au}(\text{NH}_3)_2^+$ , which will increase the gold leaching kinetics. The presence of ammonia in the thiosulfate-EDA-copper system may also produce trace amounts of cupric ammine. Even in trace level, cupric ammine can accelerate the oxidation of gold and thiosulfate.

### **5.5.3. Thiosulfate-ethylenediamine-copper leaching of gold**

#### **5.5.3.1. The reference ore**

The possibility of leaching gold with thiosulfate-ethylenediamine-copper system was studied and the result was shown in Table 5-9. The  $\text{CuEn}_3$  was prepared by mixing copper sulfate with ethylenediamine at a molar ratio of 1:3. Once prepared, the solution can be stored in a flask for later usage.

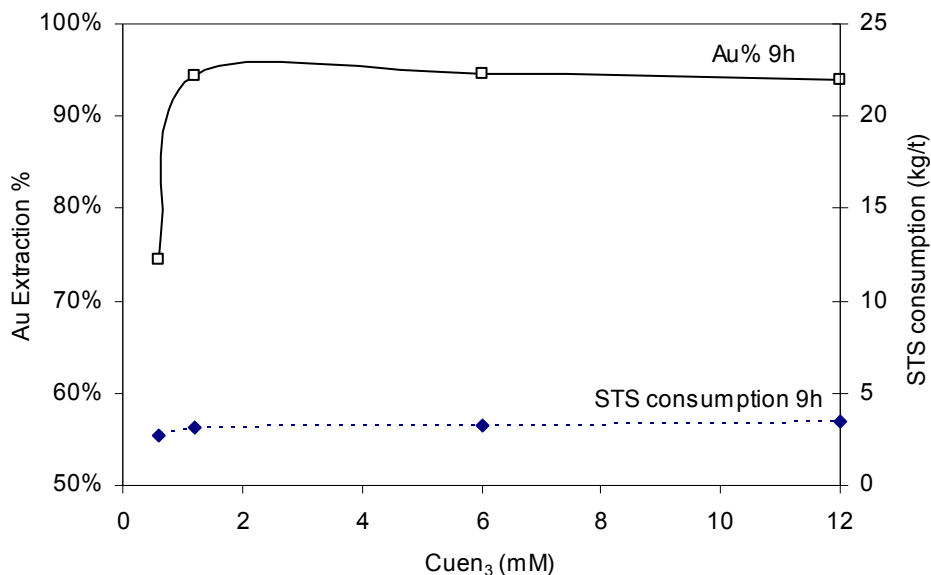
In Test #1, 92.8% gold was extracted within 3.0 hours using the conventional ATS-AH-Cu solution. By adding 1.2 mM  $\text{CuEn}_3$  and reducing the concentration of ammonia to 0.06 M (Test #2), 94.1% gold was extracted and the thiosulfate consumption was reduced from 7.8 to 3.5 kg/t. In Test #3, the leaching of gold was performed without ammonia and the gold extraction at 3.0 hours had decreased to 87.7%. In Test #4, STS was replaced by ATS, which provided 0.2 M ammonium anion to the leaching system. In this case, the gold extraction was

restored to 94.3% with 2.6 kg/t ATS consumption in 6 hours. In the final test (#5), the same leaching conditions as Test #3 was used but with extended leaching time (6 hours). 93.3% gold was extracted with STS consumption of 2.5 kg/t.

**Table 5-9: Thiosulfate-EDA-copper explorative leaching tests**

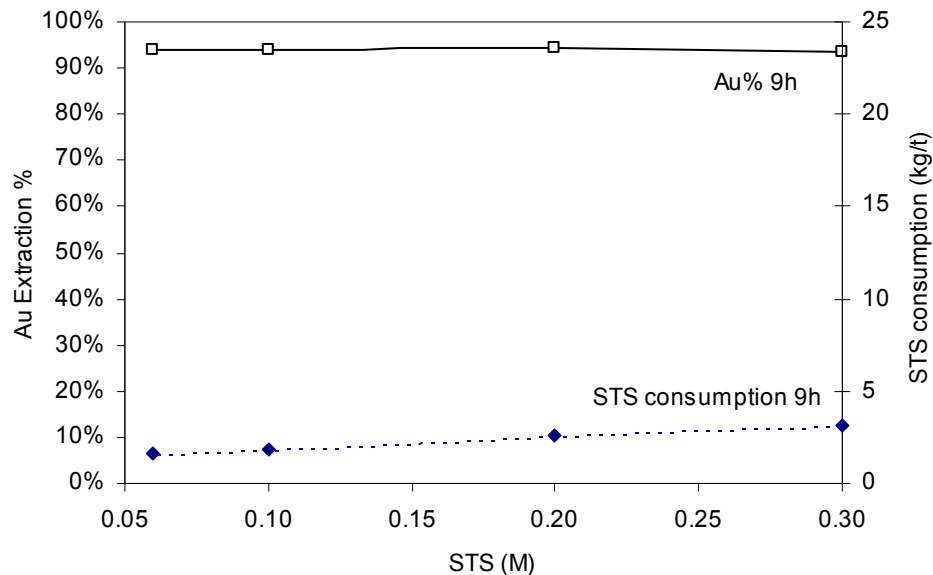
Test #	TS	Cu <sub>en3</sub>	CuSO <sub>4</sub>	AH	O <sub>2</sub> %	Time	Extraction	Consumption
Unit	M	mM	ppm	M		h	Au%	kg/t
1	ATS 0.20	0.0	76.0 (Cu)	0.90	21.1%	3.0	92.8%	7.8
2	STS 0.10	1.2	0	0.06	21.1%	3.0	94.1%	3.5
3	STS 0.10	1.2	0	0.00	21.1%	3.0	87.7%	2.3
4	ATS 0.10	1.2	0	0.00	21.1%	6.0	94.3%	2.6
5	STS 0.10	1.2	0	0.00	21.1%	6.0	93.3%	2.5

By comparing the tests shown in Table 4-11, it can be concluded that the STS-EDA leaching is slower than the conventional thiosulfate-ammonia-copper leaching. In the presence of free ammonia (as low as 0.06 M), however, the thiosulfate-EDA-copper leaching became as fast as the conventional system; and the thiosulfate consumption was much lower. For thiosulfate-EDA-copper leaching without added ammonia, an extended leaching time is required to achieve a similar gold extraction.



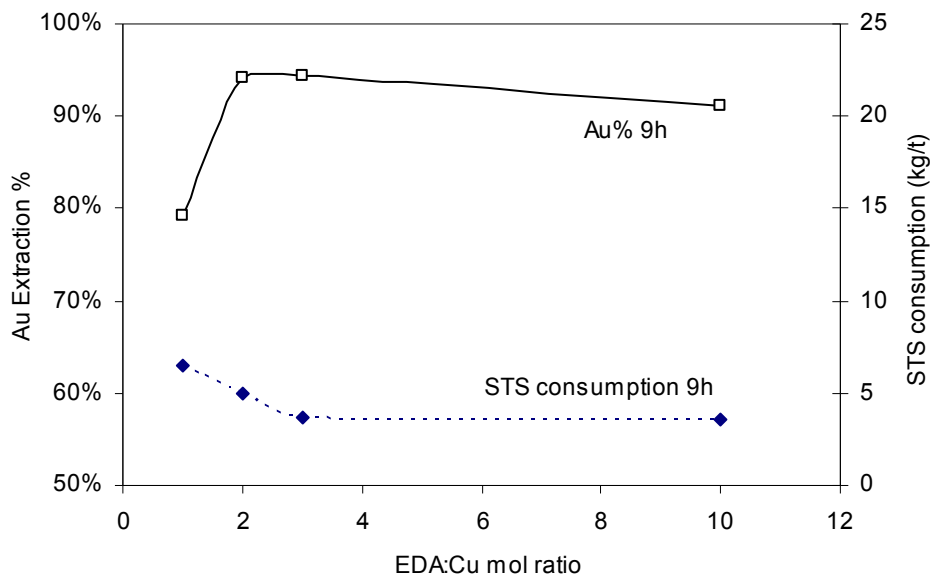
**Figure 5-85: Effect of Cuen<sub>3</sub> concentration on gold extraction and thiosulfate consumption (STS: 0.10 M, oxygen: 21.1%)**

To improve the kinetics of thiosulfate-EDA-copper leaching of gold, the effect of Cuen<sub>3</sub> concentration was studied and is shown in Figure 5-85. The leaching time was set at 9 hours. In the range of 0.6 to 12 mM Cuen<sub>3</sub>, the consumption of STS was increased slightly from 2.7 to 3.5 kg/t. With insufficient Cuen<sub>3</sub> complex, i.e., <0.6 mM, the gold extraction was only 74.4%. When the Cuen<sub>3</sub> concentration was above 1.2 mM, the gold extractions increased to about 94%.



**Figure 5-86: Effect of STS concentration on gold extraction and thiosulfate consumption (Cuen<sub>3</sub>: 1.2 mM, oxygen 21.1%)**

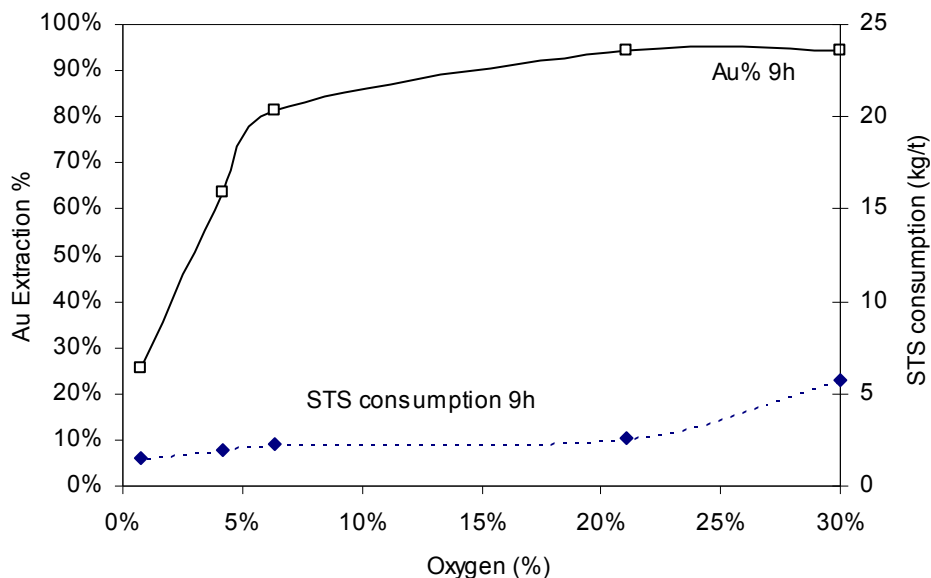
Another important variable in thiosulfate-EDA-copper leaching is the thiosulfate concentration. As shown in Figure 5-86, within the range of 0.06 to 0.30 M STS, the gold extraction remained at about 94%. STS consumption was slightly increased at a higher concentration, i.e. from 1.6 kg/t at 0.06 M STS to 3.1 kg/t at 0.30 M STS. This result indicated that the thiosulfate-EDA-copper system is more efficient than the thiosulfate-ammonia-copper system at lower thiosulfate concentration.



**Figure 5-87: Effect of EDA:Cu mol ratio on gold extraction and thiosulfate consumption (STS: 0.10M, Cu 1.2 mM, oxygen 21.1%)**

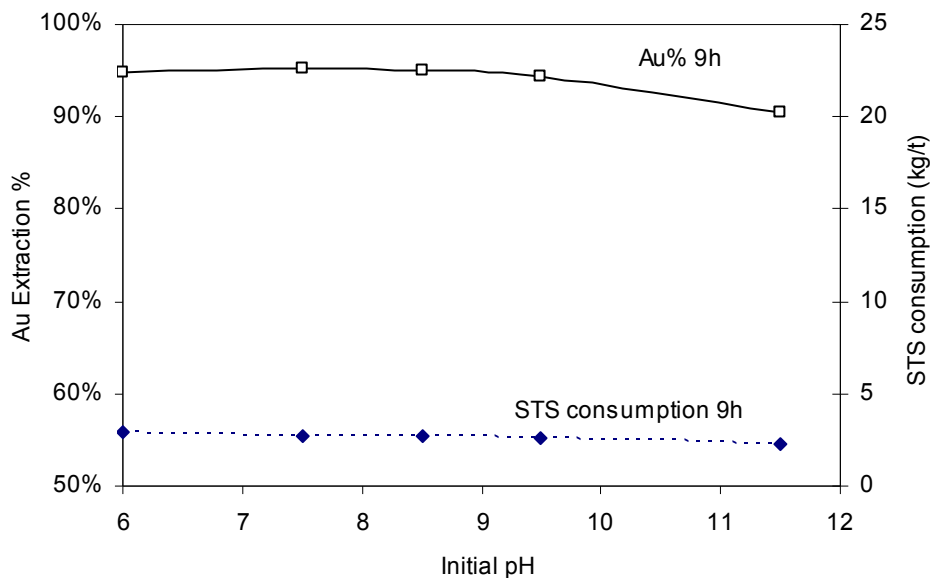
The molar ratio of EDA:Cu was also important for the stability of copper and the leaching kinetics. As shown in Figure 5-87, when the EDA:Cu ratio increased from 1 to 3, the consumption of STS was reduced from 6.5 to 3.7 kg/t. Further increasing the EDA:Cu ratio to 10 did not result in any further reduction of STS consumption. As for gold extraction, the maximum gold extractions appeared when the EDA:Cu ratio was between 2 and 3. Without sufficient EDA, the gold extraction was significantly lower (79.2%). Excessive additions of EDA also resulted in a slightly reduced gold extraction (91.2%).

This result indicated that the leaching system is most stable when  $\text{CuEn}_2$  is the predominant complex so that the lowest ratio should be 2:1. A little more EDA will help to stabilize  $\text{CuEn}_2$  as well as the thiosulfate.



**Figure 5-88: Effect of oxygen percentage on gold extraction and thiosulfate consumption (STS 0.1M, Cu 1.2 mM)**

Under thiosulfate-EDA-copper leaching conditions, the effect of oxygen supply was studied and shown in Figure 5-88. Within the range of 0.74% to 21.14% oxygen, the STS consumption appeared to be very low (1.5 to 2.6 kg/t). At 30% oxygen, the consumption was doubled (5.7 kg/t). The optimum gold extraction, i.e., >90%, required at least 21.14% to 30% oxygen in the air supply. The gold extraction reduced from 94.2% to 25.6% when the oxygen percentage was decreased from 21.14% to 0.74%. Compared to conventional thiosulfate-ammonia-copper leaching system, the minimum oxygen % required was much higher. This result indicates that the redox potential of  $\text{Cu(en)}_2^{2+}/\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$  is higher than  $\text{Cu}(\text{NH}_3)_4^{2+}/\text{Cu}(\text{S}_2\text{O}_3)_3^{5-}$ .



**Figure 5-89: Effect of pH on gold extraction and thiosulfate consumption (STS 0.1M, Cu 1.2 mM, oxygen 21.1%)**

Figure 5-89 displays the effect of pH in the thiosulfate-EDA-copper system. Within the pH range from 6 to 11.5 adjusted by either sulphuric acid or sodium hydroxide, the STS consumption was between 2.3-2.9 kg/t. The gold extraction remained stable at about 94% within the pH range from 6 to 9.5. However, the gold extraction was reduced to 90.4% at pH 11.5. The wide pH range that works in thiosulfate-EDA-copper system can be explain by the wide windows in the Eh-pH diagram of this system.

### 5.5.3.2. Leaching of synthetic sulfide ores with TS-EDA-Cu system

TS-EDA-Cu leaching of the reference ore containing 4% chalcocite was tested. The result (Table 5-10) indicated that 90.7% gold was extracted in 18 hours with

a STS consumption of 13.8 kg/t. Chalcocite is highly dissolvable in conventional thiosulfate leaching.

**Table 5-10: TS-EDA leach of chalcocite-bearing gold ore**

Test #	STS	EDA	Oxygne	Chalcocite	Extraction	Consumption	Duration
unit	M	mM	%	%	Au%	kg/t	h
1	0.3	24	21.14%	4%	90.70%	13.8	18.0

As shown in Table 5-11, the thiosulfate-EDA-copper leaching was applied to a chalcopyrite-bearing synthetic gold ore. In the presence of 8% chalcopyrite, the STS-EDA Test (#1) appeared to be slow in leaching kinetics. After 18 hours, the gold extraction reached 88.9%. When ATS was used instead of STS (Test #2), the leaching rate was much faster so that the overall gold extraction reached 90.9% within 3.0 hours. The addition of 0.1 M ammonia in the thiosulfate-EDA-ammonia-copper Test (#3) also resulted in 90.9% gold extraction. However, the ATS consumption was reduced from 13.5 to 6.8 kg/t.

**Table 5-11: Thiosulfate-EDA-copper leaching of chalcopyrite-bearing gold ore**

Test #	TS	EDA	AH	Oxygne	additional Cu	Chalcopyrite	Extraction	Consumption	Duration
unit	M	mM	M	%	mM	%	Au%	kg/t	h
1	STS 0.3 M	24	0	21.1%	1.2	8.0%	88.9%	13.9	18
2	ATS 0.2 M	24	0	21.1%	2.4	8.0%	90.9%	13.5	3
3	ATS 0.2 M	24	0.1	21.1%	2.4	8.0%	90.9%	6.8	3

As shown in Table 5-12, the EDA leaching system was tested on synthetic ores containing either pyrite or pyrrhotite. The conventional thiosulfate-ammonia-copper leaching produced 86.9% and 91.5% gold extraction within 3.0 hours in the presence of pyrite and pyrrhotite ore respectively (Test #1 and #2). The thiosulfate consumptions were high: 16.7 and 12.1 kg/t respectively. In the EDA

leaching system (Tests #3 and #4), the duration was extended to 24 hours. The consumption of STS was 2.5 kg/t for pyrite and 2.6 kg/t for pyrrhotite. At 24 hours, the gold extractions were 91.9% for pyrite ore and 93.7% for pyrrhotite ore. This result indicated that the catalyst oxidation of thiosulfate on the surface of pyrite and pyrrhotite was largely inhibited in the thiosulfate-EDA-copper system.

**Table 5-12: Thiosulfate-EDA-copper leaching of pyrite and pyrrhotite bearing gold ore**

Test #	TS	EDA	AH	Oxygne	additional Cu	Mineral	Extraction	Consumption	Duration
unit	M	mM	M	%	mM	%	Au%	kg/t	h
1	ATS 0.2 M		0.9	21.1%	1.2	Pyrite 8.0%	86.9%	16.7	3.0
2	ATS 0.2 M		0.9	21.1%	1.2	Pyrrhotite 8.0%	91.5%	12.1	3.0
3	STS 0.1M	60		21.1%	20.0	Pyrite 8.0%	91.9%	2.5	24.0
4	STS 0.1M	60		21.1%	20.0	Pyrrhotite 8.0%	93.7%	2.6	24.0

In the TS-EDA-Cu system, the effect of galena is shown in Table 5-13. The leaching of gold ore containing 2.4% galena with conventional thiosulfate-ammonia-copper system. In EDA leaching, 60.9% of gold was leached in 9 hours with 3.9 kg/t STS consumed. The reduced gold extraction was due to the formation of lead thiosulfate complex and precipitate as well as lead hydroxide and lead sulfate. Because EDA system works well within a wide range of pH. And the gold extraction was improved by 19% to 79.1% at pH 7.5. This was due to that the dissolution of lead and the formation of lead hydroxide precipitate becomes less significant at neutral pH. Carbonate anion was also tested in EDA system and was found to effectively improve the gold extraction. With addition of 3.8 mM sodium carbonate (0.4 g/L), the gold extraction was improved to 70.3%. The gold extraction increased to 90.2% by adding 22.6 mM sodium carbonate

(2.4 g/L). The formation of lead carbonate, which reduced the dissolution of lead and prevented the formation of lead precipitate on gold surface.

**Table 5-13: Thiosulfate-EDA-copper leaching of galena-bearing gold ore**

Test #	STS	Cu <sub>en3</sub>	Oxygen	Cu	Galena	Other variations	Extraction	Consumption	Duration
unit	M	mM	%	mM	%	-	Au%	kg/t	h
1	0.10	1.2	21.1%	1.2	2.4%	-	60.9%	3.9	9.0
2	0.10	1.2	21.1%	1.2	2.4%	pH 7.5	79.1%	4.3	9.0
3	0.10	1.2	21.1%	1.2	2.4%	Na <sub>2</sub> CO <sub>3</sub> 0.4 g/L	70.3%	3.7	9.0
4	0.10	1.2	21.1%	1.2	2.4%	Na <sub>2</sub> CO <sub>3</sub> 2.4 g/L	90.2%	3.7	9.0

This result demonstrates that the thiosulfate-EDA-copper leaching has the advantage of a wider pH range, which is essential for preventing the formation of Pb(OH)<sub>2</sub> or PbO•PbSO<sub>4</sub> that precipitate on the gold surface.

By testing the thiosulfate-EDA-copper leaching system on the gold ore containing various sulfide minerals, it was found that the thiosulfate consumption is much lower than that in the conventional leaching system; and the gold extraction was slower and required a longer retention time to achieve an acceptable gold extraction.

#### **5.5.4. Summary**

The mechanism and performance of the thiosulfate-EDA-copper leaching system have been discussed in this section. In general, this is a slower leaching approach and requires longer retention time. However, due to the highly stable chemistry in this system, many complicated problems that occur in the thiosulfate-ammonia-copper leaching were simplified. The gold extractions in the

thiosulfate-EDA-copper system were comparable to those in the thiosulfate-ammonia-copper leaching. Therefore, this method has potential.

Due to the higher stability of the Cu-EDA complex, the thiosulfate-EDA-copper leaching can be performed under open-air conditions without consuming too much thiosulfate. For the same reason, this system works well with or without ammonia and is feasible within a much wider range of pH. The presence of ammonium anion or free ammonia was found beneficial for thiosulfate-EDA-copper system and may increase the extraction of gold.

Due to the reduction of thiosulfate oxidation in thiosulfate-EDA-copper system, the oxidations of both gold and sulfide are more effective. Therefore, this system also works better for the leaching of sulfidic ores. For the same reasons, the thiosulfate concentration required in this system is lower than that in the thiosulfate-ammonia-copper system.

## **5.6. Validation of optimal leaching conditions on natural gold ores**

In this section, the optimal conditions developed in this study were applied in the leaching of several natural gold ores. However, because the purpose of this work is not to produce optimum leaching conditions for these ores, systematic investigations were not conducted on these ores.

### 5.6.1. Ore #1

Ore #1 was a pure silicate gold ore containing no sulfide. The mineralogical composition of this ore was introduced in Chapter 3.

In the absence of atmospheric control equipment, the optimum leaching conditions were: open air, 0.20 M ATS, 0.9 M AH, 1.2 mM Cu, and 3.0 hours duration. The gold extraction was about 93% with 7.5 kg/t of ATS consumed.

The dissolved oxygen concentration can be controlled by sparging nitrogen or another inert gas into the slurry, maintaining the slurry temperature at a higher level or limiting the air that is supplied to the leaching slurry. In this study, sparging of nitrogen was used as the method for controlling the dissolved oxygen concentration.

The optimum leaching conditions were 0.74% oxygen, 0.20 M ATS, 0.9 M AH, 1.2 mM Cu, and 3.0 hours. The gold extraction was about 93.0% and the ATS consumption was 3.5 kg/t.

The thiosulfate-EDA-copper leaching method could be used to further reduce the consumption of ATS. However, the leaching time was longer than in the conventional thiosulfate-ammonia-copper methods. The optimum leaching conditions were: 0.10 M STS or ATS, 1.2 mM  $\text{Cu}(\text{en})_3$ , for 6-9 hours duration. The gold extraction reached about 93% and the consumption was 2.7 kg/t.

### 5.6.2. Ore #2

This ore is a copper-bearing gold ore containing 0.58% chalcopyrite and 7.25% pyrite (see Section 4.1). The ore sample was ground to 93.5% passing 200 mesh with a ring grinder immediately before the leaching tests. A 200 gram portion of ground ore was placed in a reactor. The agitation was supplied by an orbital shaker at the speed of 220 rpm. After leaching, the slurry was filtered, the solid was washed and dried and the pregnant solution as well as the wash solution was collected. Both the solution and the solid residue were analyzed by fire assay method.

The conventional thiosulfate-ammonia-copper leaching conditions were optimized as follows: 0.30 M ATS, 3.0 M AH, 1.2 mM Cu, 0.7% oxygen, and 24 hours duration. The gold extraction was 91.8% and the consumption was 6.6 kg/t. Under the same reagent conditions, but with 21.1% oxygen, there was 92.3% gold extracted with 27.1 kg/t of ATS consumed.

In another trial test, 10.5 mM  $\text{Na}_3\text{PO}_4$  was added in the 6 hours pre-treatment process. The pulp density was remained the same, i.e., 33.3% in both the pre-treatment and the leaching process. The gold leaching was conducted at 0.20 M ATS, 0.9 M AH and 1.2 mM Cu, 0.74% oxygen and for 24 hours duration. The gold extraction was 93.5% and the thiosulfate consumption was 5.2 kg/t.

By applying the thiosulfate-EDA-copper leaching method, the following leaching conditions were tested: 0.1 M STS, 12 mM  $\text{Cu}(\text{en})_3$ , and 36 hours duration without

atmospheric control. The gold extraction was 93.3% and the STS consumption was 8.3 kg/t. As expected, the thiosulfate-EDA-copper leaching has slower kinetics than the conventional thiosulfate leaching. However, with 12 hours extra leaching time, a comparable gold extraction was reached.

This group of leaching tests demonstrated that by controlling the oxygen percentage in the air supply, pre-treating the ground sample with phosphate and using the STS-EDA leaching method, the consumptions of thiosulfate were largely reduced and the gold extractions remained above 91%.

### **5.6.3. Ore #3**

This ore is a pressure oxidation (POX) residue for the testing of the thiosulfate-EDA-copper leaching method. The autoclave oxidation was performed at the SGS Lakefield Laboratories. The POX sample contains mainly oxides and quartz with minor amount of sulfide residues. The gold grade was 3.30 g/t. All the leaching tests were performed at 25% pulp density without dissolved oxygen (DO) control. The slurry was agitated on a temperature controllable orbital shaker at 220 rpm.

Test #11 was a standard cyanidation test at 1 g/L NaCN, with the pH maintained between 10.5 and 11.0 by quick lime ( $\text{Ca}(\text{OH})_2$ ). The leaching time was 16 hours with the temperature controlled at 23°C. The overall gold extraction was found to be 11.7%. The low gold extraction can be explained by the preg-robbing property of this sample.

Table 5-14 lists the leaching results of the Test #2 to #6. Test #2 was a conventional thiosulfate-ammonia-copper leaching. The gold extraction was 88.1%. The ATS consumption was 15.8 kg/t. The high consumption of reagent indicated that the iron sulfides left in the POX residue were acting as the surface catalyst in thiosulfate oxidations. Tests #2 to #6 were conducted under thiosulfate-EDA-copper leaching conditions. Obviously, the thiosulfate-EDA-copper leaching consumes much less ATS (2.4 and 3.8 kg/t) or STS (3.8 and 4.0 kg/t). As for the gold extractions, those that use ATS as lixiviant produced about 76% gold extractions. While, for those using sodium thiosulfate was used instead of ammonium thiosulfate, the gold extractions were improved to 80.6% and 89.4%. Elevating the temperature from 23°C to 50°C significantly enhanced the leaching performance by an additional 8.8% of gold extraction.

**Table 5-14: Thiosulfate-EDA-copper tests on Ore #3**

#	ATS	STS	AH	Cu	Cu <sub>en3</sub>	Time	Temp.	ATS consu.	STS consu.	Au extr.
	g/L	g/L	M	mM	mM	h	°C	kg/t	kg/t	%
2	0.054	0	0.135	0.6		8	50	15.8		88.1%
3	0.054	0	0	1.6	3.2	16	23	2.4		76.0%
4	8	0	0	1.6	3.2	16	50	3.8		76.8%
5	0	0.054	0	1.6	3.2	16	23		3.8	80.6%
6	0	0.054	0	1.6	3.2	16	50		4	89.4%

It can be concluded from this group of tests that the advantages of thiosulfate-EDA-copper leaching are significant in the leaching of this POX residue. The thiosulfate consumption was reduced from 15.8 kg/t (ATS) to 4.0 kg/t (STS).

#### 5.6.4. Ore #4

This ore is also a POX residue for the testing of the thiosulfate-EDA-copper leaching method. The autoclave oxidation was also performed in SGS Lakefield Laboratories. This POX sample contains only oxides and quartz with minor amounts other non-sulfide minerals. The gold grade was 6.7 g/t. All the leaching tests were performed at 33.3% pulp density without DO control. The slurry agitation was achieved by shaking the reactor vessel on an orbital shaker at 220 rpm.

As shown in Table 5-15, the conventional thiosulfate-ammonia-copper leaching of this ore (Test #1) consumed 2.4 kg/t ATS and extracted 94.2% gold. The thiosulfate-EDA-ammonia-copper leaching Test #2 has the same ATS consumption but had a slightly improved gold extraction, i.e., 94.9%.

On the other hand, when sodium thiosulfate was used instead of ammonium thiosulfate, the gold extraction was 94.1%. The consumption of STS was 2.6 kg/t. This result indicates that without the presence of the ammonium anion and free ammonia, the thiosulfate-EDA-copper leaching system still worked efficiently.

**Table 5-15: Thiosulfate-EDA-copper tests on Ore #4**

#	ATS	STS	AH	Cu	EDA	Time	Temp.	ATS consu.	STS consu.	Au extr.
	M	M	mM	mM	mM	h	°C	kg/t	kg/t	%
1	0.040	0.000	40	0.6	0.0	8.0	50.0	2.4	0.0	94.2
2	0.040	0.000	0	0.6	2.4	8.0	50.0	2.4	0.0	94.9
3	0.000	0.040	0	0.6	2.4	8.0	50.0	0.0	2.6	94.1

It can be concluded that the thiosulfate-EDA-copper leaching system worked well on this ore sample. Considering that ethylenediamine does not evaporate significantly and the dosage of this ligand is much smaller than that of ammonia, thiosulfate-EDA-copper leaching method can be a better choice with respect to the environmental and occupational health and safety.

## Chapter 6. Conclusions and future work

### 6.1. Conclusions

The effects of some associated minerals on thiosulfate leaching of gold were studied using synthetic ore samples that had been prepared by blending specific high purity minerals into a free-milling quartz gold ore. Conclusions drawn from this work are listed as follows:

#### A. Thiosulfate leaching in the absence of associated minerals

- Thiosulfate leaching of the quartz gold ore was quick. About 93% of gold was leached within 3.0 hours with 7.1 kg/t ammonium thiosulfate (ATS) consumed.
- The optimum reagent conditions were: 0.2 M ATS, 0.9 M ammonia, and 1.2 mM copper.

#### B. Thiosulfate leaching in the presence of associated minerals

- Iron oxides, such as limonite, magnetite and hematite had no significant impact on thiosulfate consumption and overall gold extraction. The initial gold extraction at 0.5 hours were slightly reduced (i.e., by 5%) in the presence of these minerals compared to the pure silicate gold ore.

- The relative detrimental effects of pyrite, pyrrhotite and arsenopyrite on gold extractions had the order of arsenopyrite >> pyrite > pyrrhotite. The initial gold extractions were largely reduced in the presence of these minerals. Due to the oxidation and passivation of their reductive surfaces, the detrimental effects became less significant at 3.0 hours. Pyrite and pyrrhotite substantially increased the consumption of thiosulfate; while arsenopyrite had no impact on thiosulfate stability.
- Copper dissolution from chalcopyrite in ammoniacal thiosulfate solution was very limited. Chalcocite and bornite appeared to be highly dissolvable. The detrimental effects of these minerals on gold extractions had the order of bornite > chalcocite > chalcopyrite. All three minerals resulted in higher consumptions of thiosulfate.
- Lead nitrate, galena and litharge were dissolvable in thiosulfate solution. The gold extractions in the presence of these minerals were largely reduced due to the surface passivation of gold by lead precipitate (mostly lead hydroxide). The consumption of thiosulfate reduced in the leaching of galena but increased in the leaching of litharge.

C. Suggested approaches for improving the leaching performance:

- By adjusting the reagent conditions, e.g. thiosulfate, copper and ammonia concentrations, various improvements were achieved in the presence of the detrimental minerals. For pyrite, pyrrhotite, arsenopyrite, chalcocite,

chalcopyrite the optimum thiosulfate concentration was about 0.20 M. For bornite, 0.4 M thiosulfate was suggested. For galena, 0.8 M thiosulfate was suggested to produce the best gold extractions. By increasing the thiosulfate concentration, the reagent consumption increased. For pyrite, pyrrhotite, chalcopyrite, arsenopyrite, and bornite, the higher the ammonia concentrations, the higher the gold extractions. In the presence of chalcocite and galena, the gold extractions were not improved by adding more ammonia above 0.9 M. Higher copper concentration improved the gold extractions in the presence of pyrite, pyrrhotite, arsenopyrite, and chalcopyrite. Copper concentration has no impact on gold extraction in the presence of chalcocite, bornite and galena. In each case, a higher copper concentration resulted in increased consumption of thiosulfate.

- When appropriate amount of copper presented, thiosulfate leaching of gold under controlled oxygen supply conditions (i.e., reducing the oxygen percentage in the atmosphere to 0.7%) largely reduced the consumption of thiosulfate and had no significant impact on the gold extraction. This technique was verified on both the quartz gold ore and the synthetic gold ores containing pyrite, pyrrhotite, chalcopyrite, and arsenopyrite. A pre-aeration step on the sulfide ores proved to be beneficial to the thiosulfate leaching of gold with oxygen control.
- Thiosulfate leaching using ethylenediamine as the copper ligand and as the leaching catalyst was found to be successful for the silicate gold ore.

Compared to conventional thiosulfate leaching, ethylenediamine-thiosulfate (TS-EDA) leaching consumed less thiosulfate reagent. TS-EDA leaching had slower leaching kinetics than the conventional thiosulfate leaching. By extending the leach duration, the final gold extraction is satisfactory. TS-EDA leaching could be conducted with or without the presence of additional ammonia. However, ammonia was found to significantly accelerate the leaching rate. TS-EDA leaching could be conducted within a wide range of pH values (i.e. 6 – 11). The optimum conditions for the quartz gold ore were: 0.10 M STS, 1.2 mM  $\text{CuEN}_3$ . It was critical to maintain the ratio of EDA:Cu at above 2:1. The optimum ratio was 3:1. The dissolved oxygen concentration required by TS-EDA leaching was about 8.6 ppm (21.1% oxygen). Unlike conventional thiosulfate leaching, the gold extractions were significantly reduced at low dissolved oxygen (DO) conditions (i.e. 0.3 ppm DO at 0.74% oxygen). In the presence of sulfide minerals, such as chalcocite, chalcopyrite, galena, pyrite and pyrrhotite, TS-EDA leaching could successfully extract gold with significantly reduced reagent consumption. However, longer leaching time was necessary. TS-EDA leaching was also tested using a natural copper bearing sulfidic gold ore and a pressure oxidized carbonaceous ore, the resulting gold extractions were similar to those in the conventional thiosulfate leaching. The oxidation of thiosulfate was found to be minimized.

- Chemical additives were tested in thiosulfate leaching in the presence of various minerals. Appropriate amounts of these additives either improved the

gold extraction or reduced the consumption of thiosulfate. Additional hydroxide anion improved the leaching results in the presence of pyrite, pyrrhotite and arsenopyrite. Calcium ion improved the gold leaching in the presence of arsenopyrite. Phosphate anion significantly improved the leaching performance in the presence of chalcocite, galena, pyrrhotite and arsenopyrite. A small amount of galena improved the leaching of gold in the presence of pyrite, and pyrrhotite. Carbonate anion largely increased the gold extraction in the leaching of the galena bearing ore.

Generally, the associated minerals, especially the sulfides and the highly dissolvable minerals were detrimental to thiosulfate leaching. Measures that are capable of suppressing these detrimental effects were suggested and tested in this research. In the leaching of natural ores, by comprehensively applying these measures, the performance of thiosulfate leaching could be improved.

## **6.2. Future work**

- A. In the future, alternative thiosulfate technology, such as thiosulfate-ethylenediamine leaching, should be further studied and improved in order to build a stable and effective leaching method. Similar leaching approaches using other copper ligands instead of EDA should also be investigated.
- B. The mechanisms of thiosulfate decomposition catalyzed by sulfide surface should be further studied.

- C. The effects of hydroxide, phosphate, lead, and carbonate should be further investigated as economic additives to thiosulfate leaching. More work must be performed in the fields of mineralogical and surface chemistry in order to deepen the understanding of the behaviors of these additives.
  
- D. The effects of different combinations of various minerals should be studied and the problems should be analyzed; galvanic effects, surface activation and passivation being the most interesting topics. Methods provided in this work that have improved leaching should be tested for other gold ores containing complex minerals.
  
- E. Conclusions of this study should be tested on industrial gold ores. For the best results, the solutions for the problematic minerals should be comprehensively applied to naturally occurring gold ores.

## References

- Abbruzzese, C., Fornari, P., Massidda, R., Veglio, F. and Ubaldini, S., 1995, Thiosulfate leaching for gold hydrometallurgy. *Hydrometallurgy*, 39(1-3), 265-276.
- Acharya, S., Anand, S. and Das, R.P., 1987, Silver losses from ammoniacal solutions on complex sulfide and chalcopyrite ammonia leach residues. *Int. J. Miner. Process.* 20(1987) 229-239.
- Arima, H., Fujita, T., and Yen, W.T., 2003, Gold recovery from nickel catalyzed ammonium thiosulfate solution by strongly basic anion exchange resin. *Material Transactions* 44(10) pp. 2099-2107.
- Arima, H., Fujita, T., and Yen, W.T., 2004, Using nickel as a catalyst in ammonium thiosulfate leaching for gold extraction. *Materials Transactions*, Vol. 45, No.2 (2004), pp. 516-526.
- Aylmore, M.G., 2001, Treatment of a refractory gold-copper sulfide concentration by copper ammoniacal thiosulfate leaching. *Minerals Engineering*, Vol. 14, No. 6, pp. 615-637.
- Balaz, P., Ficeriova, J., Boldizarova, E., Haber, M., Jelen, S. and Kammel, R., 2000, Thiosulfate leaching of gold from a mechanochemically pretreated complex sulfide concentrate. Massaci, P., (Ed.). *XXI International Mineral processing Congress*, Rome, Italy, Elsevier pp. 74-81

Bean, S.L., 1997, Thiosulfates. In: Kroschwitz, J.I. (Ed.), Kirk-Othmer Encyclopaedia of Chemical Technology. John Wiley & Sons Inc. New York, pp. 51-58.

Bekker, V. Z., Louw W. J., and Robb, W., 1972, An investigation into the mechanisms of the substitution reactions by ethylenediamine and diethylenetriamine in tetrachlorogold(III). *Inorganica Chimica Acta*, 1972. Vol. 6, pp. 564-566.

Berezowsky, R. M. G. S., Sefton, V. B. and Gormely, L. S., 1978, Recovery of precious metals from metal sulfides. U.S. Patent 4,070,182.

Bhakta, P., 2003, Ammonium thiosulfate heap leaching. *Hydrometallurgy 2003: Processing of the 5th International Symposium Honoring Professor Ian M. Ritchie*, Minerals, Metals and Materials Society/TMS, Warrendale, PA, ISBN 0873395549, pp. 259-267.

Block-Bolten, A., Daita, M. S., Torma, A. E. and Steensma, R., 1985. New possibilities in the extraction of gold and silver from zinc and lead sulfide flotation wastes. *The Metallurgical Society/AIME*, Warrendale, PA, pp. 149-166.

Blokhin, V.V., et al., 1971, Thermodynamics and structure of anisotropic and outer-sphere complexes of copper (II) containing thiosulfate ions. *Zhurnal Fizicheskoi Khimii*, 1971. 45(7): p. 1860.

Breuer, P.L., Jeffrey, M.I., 2000, Thiosulfate leaching kinetics of gold in the presence of copper and ammonia. *Minerals Engineering*, Vol. 13, No. 10-11, pp. 1071-1081

Buisman, C.J.N. and Picavet, M.A., 2004, Process for regenerating thiosulphate from a spent thiosulphate gold leachant. European Patent EP 1433860.

Cai, M. F., Dang, Z., Chen, Y.W, Belzile, N., 2005, The passivation of pyrrhotite by surface coating. *Chemosphere* 61 (2005) 659-667.

Cao, C., Hu, J. and Gong, Q., 1992, Leaching gold by low concentration thiosulfate solution. *Randol Gold Forum '92 Vancouver*, Randol International, Golden, Co, pp. 293-298.

Chandra, I. and Jeffrey, M., 2003, Can a thiosulfate leaching process be developed which does not require copper and ammonia? *Hydrometallurgy 2003 – Fifth International Conference in Honor of Professor Ian Ritchie – Vol. 1: Leaching and Solution Purification. Conference Proceeding 1*, pp. 169-182.

Chandra, I., and Jeffrey, M., 2005, A fundamental study of ferric oxalate for dissolving gold in thiosulfate solutions. *Hydrometallurgy*. Volume 77, Issues 3-4, June 2005, Pages 191-201.

Choo, W.L., and Jeffery, M.I., 2004, An electrochemical study of copper cementation of gold(I) thiosulfate. *Hydrometallurgy* 71 (3-4), 351-362.

Chow, A., Beamish, F.E., 1963, Studies of titrimetric and spectrophotometric methods for the determination of gold. *Talanta*, 1963, Vol. 10, Issue 8, pp 883-890.

Dean, J. A., 1985, *Lange's handbook of chemistry*, 13th edition

Doesburg, J.C., The Structure and Composition of an Electroplated Gold-Tin Solder Alloy Co-deposited by Pulse Plating from Solutions Containing Ethylenediamine as a Stabilizer. Master Thesis, University OF Alberta, 2000.

EEC/FDA, 2001. (i) European MSDS: Directive 67/548/EEC. (ii) American MSDS: 29 CFR 1910-1200 (US FDA).

Elsetinow, A. R., Schoonen, M. A. A., and Strongin, D. R., 2001, Aqueous geochemical and surface science investigation of the effect of phosphate on pyrite oxidation. *Environ. Sci. Technol.* 2001, Vol. 35(II), pp. 2252-2257.

Fang, Zhaoheng and Han, Baoling, 2002, Leaching of gold using oxidation products of elemental sulfur in  $\text{Ca}(\text{OH})_2$  solution under oxygen pressure. *The Chinese Journal of Process Engineering*, Vol. 2, Issue 3, June, 2002, pp. 31-38

Feng, D., Van Denventer, J. S. J., 2001, Preg-robbing phenomena in the thiosulphate leaching of gold ores. *Minerals Engineering*, Vol. 14, Issue 11, pp 1387-1402

Feng, D., Van Denventer, J. S. J., 2002, Leaching behaviour of sulfides in ammoniacal thiosulfate systems. *Hydrometallurgy* Vol. 63, Issue 2, 2002, pp. 189-200.

Feng, D., Van Denventer, J. S. J., 2002, The role of heavy metal ions in gold dissolution in the ammoniacal thiosulphate system. *Hydrometallurgy* Vol. 64, Issue 3, June 2002, pp. 231-246.

Feng, D., Van Denventer, J. S. J., 2005, Thiosulfate decomposition in the presence of sulfides. First International Symposium on the Treatment of Gold Ores, 44<sup>th</sup> Annual Conference of Metallurgists, CIM, Calgary, Canada. pp.129-207.

Ficeriova, J., Balaz, P., Boldizarova, E. and Jelen, S., 2002, Thiosulfate leaching of gold from a mechanically activated CuPbZn concentrate. *Hydrometallurgy*, Vol. 67, Issue 1-3, pp.37-43.

Fleming, C. A., 2000, Recent advances in the development of an alternative to the cyanidation process - based on thiosulfate leaching and resin in pulp. *Ballarat Gold Forum*, Australia.

Fleming, C.A., McMullen, J., Thomas, K.G., Wells, J.A., 2003, Recent advances in the development of an alternative to cyanidation process: Thiosulfate leaching and resin in pulp. *Minerals & Metallurgical Processing*, Vol. 20, No.1. pp. 1-9.

Gerenrot, Y.E. and P.I. Dymarskaya, 1972, Possible causes of depolarization during the electrodeposition of copper from an ethylenediamine electrolyte. *Elektrokhimiya*, 1972. Vol. 8, Issue 4, pp. 509-13.

Ghosh, M. K., Das, R. P., Biswas, A. K., 2003, Oxidative ammonia leaching of sphalerite, part II: Cu(II)-catalyzed kinetics. *Int. J. Miner. Process.* Vol. 70, Issue 1-4, pp. 221-234.

Gong, Q., Hu, J. and Cao, C., 1993. Kinetics of gold leaching from sulfide gold concentrates with thiosulfate solution. *Transactions of the Nonferrous metals Society of China*, Vol. 3, Issue 4, pp. 30-36.

Groudev, S. N., Ivanov, I. M., Spasova, I. I. and Groudeva, V. I., 1995, Pilot scale microbial leaching of gold and silver from an oxide ore in Elshitza Mine, Bulgaria *Minerals Bioprocessing II*, Minerals, Metals, and Materials Society/AIME, Warrendale, PA, July 1995, pp. 135-144.

Groudev, S. N., Ivanov, I. M., Spasova, I. I., 1996, Two-stage microbial leaching of a refractory gold-bearing pyrite ore. *Minerals Engineering*, Vol. 9, Issue 7, 707-713.

Hamilton, I.C., Woods, R., 1983, An investigation of the decomposition and reactions of sulfur on gold electrodes. *Journal of Applied Electrochemistry*, Vol. 13, 1983. pp. 783-794.

Hemmati, M., Hedrix, J.L., Nelson J.H., and Milosavljevic, E. G., 1989, Study of the thiosulphate leaching of gold from carbonaceous ore and the quantitative determination of thiosulfate in the leached solution. Extraction Metallurgy '89 Symposium, the Institute of Mining and Metallurgy, London, UK, pp. 665-678.

Hu, J. and Gong, Q., 1991, Substitute sulfate for sulfite during extraction of gold by thiosulfate solution. Hydrometallurgy, Vol. 60, Issue 1, pp. 7-16.

Ji, C. and Yu, Q., 1991, Research and optimization of thiosulfate leaching technology of gold. Rare metals, 10(4), 257-280

Jorgensen, C. K., and Pouradier, J., 1970, Un Nouveau Type De Stabilisation Du "Champ Des Ligands" Dans Les Complexes Lineaires Du Cuivre (I) Et De L'or (I) Journal De Chemie Physique, Vol. 67, pp. 124-127.

Kato, M. and Okinaka, Y., 2004, Some recent developments in non-cyanide gold plating for electronics applications. Gold Bulletin (London, United Kingdom), 2004. Vol. 37, Issue 1-2, pp. 37-44.

Kerley, B.J.Jr, 1981, Recovery of precious metals from difficult ores. U.S. Patent 4269622.

Kerley, B.J.Jr., 1983, Recovery of precious metals from difficult ores. U.S.Patent 4369061.

La Brooy, S.R., Linge, H.G., and Walker, G.S., 1994, Review of gold extraction from ores. Minerals Engineering, Vol. 7, Issue 10, pp. 1213-1241.

Langhans Jr, J.W., Lei, K.P.V. and Carnahan, T.G., 1992. Copper-catalyzed thiosulfate leaching of low-grade gold ores. *Hydrometallurgy*, Vol. 29, Issue 1-3, pp. 191-203.

Liu, K.J., Yen, W.T., Shibayama, A., Miyazaki, T., and Fujita, T., 2004, Gold extraction from thiosulfate solution using trioctylmethylammonium chloride. *Hydrometallurgy* Vol. 73, Issue 1-2, April 2004, pp. 41-53.

Losi, S.A., Zuntini, F.L. and Meyer, A.R., Electrodeposition of gold-copper alloys from a cyanide free acid solution. *Electrodeposition and Surface Treatment*, 1972. Vol. 1, Issue 1, pp. 3-20.

Louw, W.J. and Robb, W., 1969, Kinetics and mechanisms of reactions of gold(III) complexes III. Substitution reaction of tetrachlorogold(III) with ethylenediamine. *Inorganica Chimica Acta*, 1969. Vol. 3: pp. 29-32.

Marchbank, A. R., Thomas, K. G., Dreisinger, D. and Fleming, C., 1996. Gold recovery from refractory carbonaceous ores by pressure oxidation and thiosulfate leaching. U.S.patent 5536297.

Marcon, G., et al., 2003, Reactions of gold(III) complexes with serum albumin. *European Journal of Biochemistry*, 2003. Vol. 270, Issue 23, pp. 4655-4661.

Matheson, R.A., 1967, The association of thiosulfate ions with bis(ethylenediamine) copper(II) ions in aqueous solution. *Journal of Physical Chemistry*, 1967. Vol. 71, Issue 5, pp. 1302-1308

Matthew, W.R., Removal of sulfur from petroleum hydrocarbons. 1966, (Petrolite Corp.). Application: US. p. 4

McNulty, T., 2001, Cyanide substitutes. Mining Magazine, 2001, Vol 184, Issue 5, pp. 256-261.

Molleman, E. and Dreisinger, D., 2002, The treatment of copper-gold ores by ammonium thiosulfate leaching. Hydrometallurgy, Vol. 66, Issue 1-3, October 2002, pp. 1-21.

Moses, C. O., Nordstorm, D. K., Herman, J. S., and Mills, A. L., Aqueous pyrite oxidation by dissolved oxygen and by ferric iron. Geochimica et Cosmochimica Acta, Vol. 51, June 1987, pp. 1561-1571

MSDS, <http://physchem.ox.ac.uk/MSDS>.

Murai, R., Kurakane, K. and Sekine, T., 1976, The stability constants of nickel(II) complexes with chloride, thiocyanate, sulfate, thiosulfate and oxalate ions, as determined by a solvent extraction method. Bulletin of the Chemical Society of Japan, Vol. 43, pp. 1365-1372.

Murthy, D. S. R., 1991, Some Studies on the extraction of gold and silver from lead-zinc sulfide flotation tailings through non-cyanide leachants. Transactions of the Indian Institute of Metals, 44(5), 349-354.

Navarro, P., Alvarez, R., Vargas, C., and Alguacil, F.J., 2004, On the use of zinc for gold cementation from ammoniacal-thiosulfate solutions. *Minerals Engineering*, Vol. 17, Issue 16, January 2004, pp. 825-831.

Navarro, P., Vargas, C., Villarroel, A., and Alguacil, F.J., 2004, On the use of ammoniacal/ammonium thiosulfate for gold extraction from a concentrate. *Hydrometallurgy*, Vol 65, Issue 1, July 2002, pp. 37-42.

NOHSC, 1999, Approved criteria for classifying hazardous substances, National Occupational Health and Safety Commission, Sydney, Australia.

Osseo-Asare, K., and Fuerstenau, D. W., 1979, Adsorption phenomena in hydrometallurgy, 1 The uptake of copper, nickel and cobalt by oxide adsorbents in aqueous ammoniacal solutions. *Int. J. Miner. Process.* Vol 6, October 1979 pp. 85-104.

Perez, A. E., 1987, Method for recovery of precious metals from difficult ores with copper-ammonium thiosulfate. U.S.Patent 4,654,078

Popel, A.A., Boos, G.A. and Bychkova, T.I., 1975, Complexing in the copper (II)-ethylenediamine-thiosulfate(2-)-water system. *Izvestiya Vysshikh Uchebnykh Zavedenii, Khimiya i Khimicheskaya Tekhnologiya*, 1975. 18(8): p. 1336.

Pourbaix, M., 1966, *Atlas of Electrochemical Equilibria in Aqueous Solution*, Oxford University Press, U.K, ISBN 0915567889.

Rainforest Information Centre, Cyanide ban worldwide. Last updated on October 2007. <http://www.rainforestinfo.org.au/gold/Bans.html>

Rao, K.S., and Ray, H.S., 1998, A new look at characterisation and oxidative ammonia leaching behaviour of multiple sulfides. *Minerals Engineering* Vol. 11, Issue 11, pp. 1011-1024

Rolia, E. and Chakrabarti, C.L., 1982, Kinetics of decomposition of tetrathionate, trithionate and thiosulfate in alkaline media. *Environmental Science and Technology*, Vol 16, Issue 12, pp. 852-857.

Rosenhek, R., Reported incidences of cyanide spills from May 2002 to August 2004, [http://www.mpi.org.au/campaigns/cyanide/cyanide\\_spills](http://www.mpi.org.au/campaigns/cyanide/cyanide_spills).

Rosso, K. M., 2006, Reactivity of sulfide minerals surfaces. *Reviews in Mineralogy and Geochemistry*. 2006, Vol. 61, pp. 557-607

Schmitz, P.A., Duyvesteyn, S., Johnson, W.P., Enloem L. and McMullen, J., 2001, Ammoniacal thiosulfate and sodium cyanide leaching of preg-robbing goldstrike ore carbonaceous matter. *Hydrometallurgy*, Vol 60, Issue 1, pp. 25-40.

Sillen, L.G. and Martell, A.E., 1971, *Stability Constants of Metal-Ion Complexes*, Special Publications N17 and 25, The Chemical Society, London, Vol. 1, 1964, and Vol. 2 1971.

Skibsted, L.H., and Bjerrum, J., 1974a, Studies on gold complexes. I. robustness, stability and acid dissolution of the tetraamminegold (III) ion. *Acta Chemica Scandinavica*, A28, 1974, pp. 740-746.

Skibsted, L.H., and Bjerrum, J., 1974b, Studies on gold complexes. II. The equilibrium between gold (I) and gold (III) in the ammonia system and the standard potentials of the couples involving gold, diamminegold(I) and tetraaminegold(III). *Acta Chemica Scandinavica*, A28, 1974, pp. 764-770.

Smith, R.M., Martell, A.E., and Motekaitis, R.J., 1998, Critically selected constants of metal complexes database, V 5.0, National Institute of Standards and Technology.

Srivastava, S.K. and D. Singh, Polarographic studies of sulfur solutions in amines. *Transactions of the SAEST*, 1988. Vol 23 Issue 4, pp. 325-8.

Thomas, K. G., Fleming, C., Marchbank, A.R. and Dreisinger, D., 1998, Gold recovery from refractory carbonaceous ores by pressure oxidation, thiosulfate leaching and resin-in-pulp adsorption. U.S.Patent 5785736.

Tremblay, L., Deschenes, G., Ghali, E., MacMullen, J. and Lanouette, M., 1996, Gold recovery from a sulfide bearing gold ore by percolation leaching with thiourea, *Intl. J. Min. Proc.*, Vol 48, pp. 225-244.

Wagman, D.D., Evans, W.H. Parker, V. B., Schumm, R.H., Halow, I., Bailey, S. M., Churney, K. L. and Nuttall, R. L., 1982, J. Phys. Chem. Ref. Data Suppl., 11, Supplement No. 2.

Wan, R.Y., Le Vier, M., 1993, and Milles J.D., Chapter 27, Research and development activities for the recovery of gold from non-cyanide solutions. Hydrometallurgy (1993), Fundamentals, Technology and Innovatives.

Wan, R.Y., and Le Vier, M., and Clayton, R. B., 1994, Hydrometallurgical process for the recovery of precious metal values from precious metal ores with thiosulfate lixiviant. U.S. Patent 5,354,359.

Wan, R.Y., 1997, Importance of solution chemistry for thiosulfate leaching of gold. World Gold '97, The Australasian Institute of Mining and Metallurgy, Carlton, Vic, pp 159-162.

Wan, R.Y., and LeVier, K.M., 2003, Solution chemistry factors for gold thiosulfate heap leaching, int. J. Miner. Process. 72 (2003), Issue 1-4, 311-322.

West-Sells, P.G. and Hackl, R.P., 2005, A novel thiosulfate leach process for the treatment of carbonaceous gold ores. Treatment of Gold Ores, CIM, Alberta, pp. 209-223.

Xia, C., Yen, W.T., Deschenes, G., 2002, An investigation on minimizing reagent consumption in ammonium thiosulfate gold leaching. 34th Annual Meeting of the Canadian Mineral Processors. Jan, 2002, pp. 353-368.

Xu, Y. and Schoonen M. A.A., 1995, The stability of thiosulfate in the presence of pyrite in low-temperature aqueous solutions. *Geochimica et Cosmochimica Acta*, Volume 59, Issue 22, November 1995, 4605-4622

Yazawa, A. 1980, *Non-ferrous Metallurgy*. Japan Institute of Metals, pp. 322-324

Yen, W.T., Aghamirian, M., Deschense, G. and Theben, S., 1998. Gold extraction from mild refractory ore using ammonium thiosulfate. *International Symposium on Gold Recovery*, The Canadian Institute of Mining, Metallurgy and Petroleum, May 1998, Montreal, Canada.

Yen, W.T., Stogram, K. and Fujita, T., 1996. Gold extraction from a copper bearing ore by thiosulfate leaching. *Resources Treatment Technology*, 43(2), pp. 83-87.

Zhang, H., and Dreisinger, D., 2003, The kinetics of the decomposition of tetrathionate in alkaline solutions. *Hydrometallurgy* 66 (1-3), pp. 59-65.

Zhang H. and Dreisinger D.B., 2004, The recovery of gold from ammoniacal thiosulfate solutions containing copper using ion exchange resin columns. *Hydrometallurgy* 72 (2004), pp. 225-234.

Zhang, H., Nicol, M.J., Staunton, W.P., 2005, An electrochemical study of an alternative process for the leaching of gold in thiosulfate solutions. *Treatment of Gold Ore*, 44<sup>th</sup> Annual Conference of Metallurgists, Met. Soc, CIM, pp. 243-257.

Zhao, J., Wu, Z., Chen, J., 1997, Extraction of gold from thiosulfate solutions with alkyl phosphorous esters. *Hydrometallurgy*, Vol. 46, Issue 3, October 1997, pp.363-372.

Zipperian, D., Raghavan, S. and Wilson, J. P., 1988, Gold and silver extraction by ammoniacal thiosulfate leaching from a rhyolite ore. *Hydrometallurgy*, 19(3), January 1988. pp. 361-375

Zuntini, F., 1980, Special compositions and particular additives for gold electroplating baths. 1980, (Systemes de Traitements de Surfaces S. A., Fr.).  
Application: CH. p. 7

Zuntini, F., et al., 1974, Amine gold complex useful for the electrodeposition of gold and its alloys. US patent. US 3,787,463

## Appendix – 1: Thermodynamic data

### 1) Gibbs free energy of formation of selected species

Species	(Kcal)	Species	(Kcal)	Species	(Kcal)	Species	(Kcal)
Au	0.000	Fe	0	As <sub>2</sub> O <sub>5</sub>	-186.945	Pb(OH) <sub>2</sub>	-100.714
Au(+3a)	103.563	Fe(+2a)	-21.875	As <sub>4</sub> O <sub>6</sub>	-275.353	PbO*PbSO <sub>4</sub>	-244.487
Au(+3a)	103.316	Fe(+3a)	-4.107	As <sub>2</sub> O <sub>5</sub> *4H <sub>2</sub> O	-410.863	*2PbO*PbSO <sub>4</sub>	-289.147
Au(+a)	38.988	Fe(OH)O(a)	-105.7	AsS	-8.196	*3PbO*PbSO <sub>4</sub>	-341.154
Au(+a)	39.175	Fe <sub>0.877</sub> S	-25.544	As <sub>2</sub> S <sub>2</sub>	-16.396	*4PbO*PbSO <sub>4</sub>	-383.543
Au(HS) <sub>2</sub> (-a)	2.487	Fe <sub>2</sub> (OH) <sub>2</sub> (+4a)	-111.743	As <sub>2</sub> S <sub>3</sub>	-18.836	Pb <sub>3</sub> (PO <sub>4</sub> ) <sub>2</sub>	-567.284
Au(NH <sub>3</sub> ) <sub>2</sub> (+a)	-9.819	FeO(+a)	-53.093	As <sub>4</sub> S <sub>4</sub>	-31.029	PbS	-23.182
Au(NH <sub>3</sub> ) <sub>4</sub> (+3a)	15.385	FeO(a)	-50.715	As <sub>4</sub> S <sub>6</sub>	-8.021	PbSO <sub>4</sub>	-195.078
Au(OH) <sub>3</sub>	-75.729	FeO*OH	-116.484	Ca(AsO <sub>2</sub> ) <sub>2</sub>	-336.081	PbS <sub>2</sub> O <sub>3</sub>	-137.441
Au(OH) <sub>3</sub> (P)	-75.729	FeO <sub>2</sub> (-a)	-92.642	Ca <sub>3</sub> (AsO <sub>4</sub> ) <sub>2</sub>	-732.117	AsO <sub>2</sub> (-a)	-83.623
Au(S <sub>2</sub> O <sub>3</sub> ) <sub>2</sub> (-3a)	-250.893	FeOH(+2a)	-57.83	Cu <sub>3</sub> AsO <sub>4</sub>	-149.16	AsO <sub>4</sub> (-3a)	-154.932
Au <sub>2</sub> O <sub>3</sub>	18.61	FeOH(+a)	-65.845	Cu <sub>3</sub> (AsO <sub>4</sub> ) <sub>2</sub>	-314.547	As(OH) <sub>3</sub> (a)	-152.855
Cu	0	FeS	-24.37	CuHPO <sub>4</sub>	-236.886	As(OH) <sub>4</sub> (-a)	-196.975
Cu(+2a)	15.668	FeS <sub>2</sub>	-38.259	Cu(H <sub>2</sub> PO <sub>4</sub> ) <sub>2</sub>	-511.044	CaHPO <sub>4</sub> (a)	-412.349
Cu(+a)	11.947	H <sub>2</sub> S(a)	-6.516	CuHPO <sub>4</sub> *H <sub>2</sub> O	-301.61	Ca(H <sub>2</sub> PO <sub>4</sub> ) <sub>2</sub> (a)	-672.66
Cu(NH <sub>3</sub> )(+2a)	3.685	HCuO <sub>2</sub> (-a)	-60.096	CuO*Fe <sub>2</sub> O <sub>3</sub>	-206.32	Ca <sub>3</sub> (PO <sub>4</sub> ) <sub>2</sub> (a)	-883.844
Cu(NH <sub>3</sub> ) <sub>2</sub> (+2a)	-7.299	HFeO <sub>2</sub> (a)	-105.7	Cu <sub>2</sub> O*Fe <sub>2</sub> O <sub>3</sub>	-219.815	Ca <sub>5</sub> (PO <sub>4</sub> ) <sub>3</sub> OH(a)	-1429.752
Cu(NH <sub>3</sub> ) <sub>3</sub> (+2a)	-17.498	HFeO <sub>2</sub> (-a)	-95.353	Cu <sub>3</sub> (PO <sub>4</sub> ) <sub>2</sub>	-451.198	Cu <sub>3</sub> (AsO <sub>4</sub> ) <sub>2</sub> (a)	-263.025
Cu(NH <sub>3</sub> ) <sub>4</sub> (+2a)	-26.567	HS(-a)	2.887	Cu <sub>3</sub> (PO <sub>4</sub> ) <sub>2</sub> *3H <sub>2</sub> O	-652.407	HAsO <sub>2</sub> (a)	-96.275
Cu(OH) <sub>2</sub>	-85.808	HS <sub>6</sub> (-a)	8.653	FeAsO <sub>4</sub>	-184.616	H <sub>3</sub> AsO <sub>3</sub> (a)	-152.876
Cu(S <sub>2</sub> O <sub>3</sub> )(-a)	-129.243	NH <sub>3</sub> (a)	-6.326	Fe <sub>3</sub> (AsO <sub>4</sub> ) <sub>2</sub>	-422.097	H <sub>3</sub> AsO <sub>4</sub> (a)	-183.079
Cu(S <sub>2</sub> O <sub>3</sub> ) <sub>2</sub> (-3a)	-258.984	NH <sub>4</sub> (+a)	-18.977	FeAsS	-25.772	HAsO <sub>3</sub> (-a)	-144.937
Cu(S <sub>2</sub> O <sub>3</sub> ) <sub>3</sub> (-5a)	-388.129	S	0	FePO <sub>4</sub> *2H <sub>2</sub> O	-396.144	HAsO <sub>4</sub> (-2a)	-170.761
Cu <sub>2</sub> O	-35.345	S(-2a)	20.547	FePO <sub>4</sub> *4H <sub>2</sub> O	-356.944	H <sub>2</sub> AsO <sub>3</sub> (-a)	-140.281
Cu <sub>2</sub> O*Fe <sub>2</sub> O <sub>3</sub>	-219.815	S <sub>2</sub> O <sub>3</sub> (-2a)	-123.962	H <sub>3</sub> PO <sub>4</sub>	-267.43	H <sub>2</sub> AsO <sub>4</sub> (-a)	-179.999
Cu <sub>2</sub> S	-20.666	S <sub>3</sub> O <sub>6</sub> (-2a)	-228.818	H <sub>3</sub> PO <sub>4</sub> *0.5H <sub>2</sub> O	-296.85	HCuO <sub>2</sub> (-a)	-60.096
Cu <sub>5</sub> FeS <sub>4</sub>	-93.823	S <sub>4</sub> O <sub>6</sub> (-2a)	-248.631	Pb	0	H <sub>3</sub> PO <sub>4</sub> (a)	-243.385
CuFeO <sub>2</sub>	-110.007	S <sub>6</sub> (-2a)	15.822	Pb <sub>3</sub> (AsO <sub>4</sub> ) <sub>2</sub>	-371.218	HPO <sub>4</sub> (-2a)	-260.298
CuFeS <sub>2</sub>	-45.53	SO <sub>3</sub> (-2a)	-116.288	PbO	-45.088	H <sub>2</sub> PO <sub>4</sub> (-a)	-270.128
CuO	-30.663	As	0	PbO <sub>2</sub>	-51.481	Pb(+2a)	-5.728
CuO*Fe <sub>2</sub> O <sub>3</sub>	-206.32	As <sub>2</sub> O <sub>3</sub>	-137.841	Pb <sub>2</sub> O <sub>3</sub>	-97.182	Pb(HS) <sub>2</sub> (a)	-20.024
CuS	-12.78	As <sub>2</sub> O <sub>4</sub>	-167.528	Pb <sub>3</sub> O <sub>4</sub>	-143.787	Pb(HS) <sub>3</sub> (-a)	-19.908
Pb <sub>3</sub> (OH) <sub>4</sub> (+2a)	-212.297	PbOH(+a)	-53.993	Pb(OH)O(-a)	-81.019		
Pb <sub>4</sub> (OH) <sub>4</sub> (+4a)	-223.802	Pb <sub>6</sub> (OH) <sub>8</sub> (+4a)	-430.254	PbO(a)	-39.333		

[Smith et al., 1998; Jorgensen et al., 1970; Skibsted et al., 1974a, 1974b; Murai et al., 1976; Yazawa. 1980; Pourbaix, 1966]

## 2) Cumulative formation constants for metal complexes

Ligand	Metal ion	Log K1	Log K2	Log K3	Log K4	Log K5	Log K6
Ammonia	Cu(+2a)				13.3		
Ammonia	Cu(+a)		10.9				
Ammonia	Zn(+2a)				9.5		
Hydroxide	Cu(+2a)			17.0	18.5		
Hydroxide	Fe(+2a)		9.8				
Hydroxide	Fe(+3a)		21.2	29.7			
Hydroxide	Pb(+2a)			14.6			61.0
Thiosulfate	Cu(+a)		12.2	13.8			
Thiosulfate	Pb(+2a)			6.4			
EDA	Cu(+a)		10.8				
EDA	Cu(+2a)		20.0	21.0			

[Dean, 1985]

## 3) Logarithm of the stability constant of some reactions

Reaction	Log(K)
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$\text{en}_2 + \text{H}^+ = \text{Hen}^+$	9.93
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$\text{Hen}^+ + \text{H}^+ = \text{H}_2\text{en}^{2+}$	6.85
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$\text{Cu}^{2+} + \text{en} = \text{Cuen}^{2+}$	10.5
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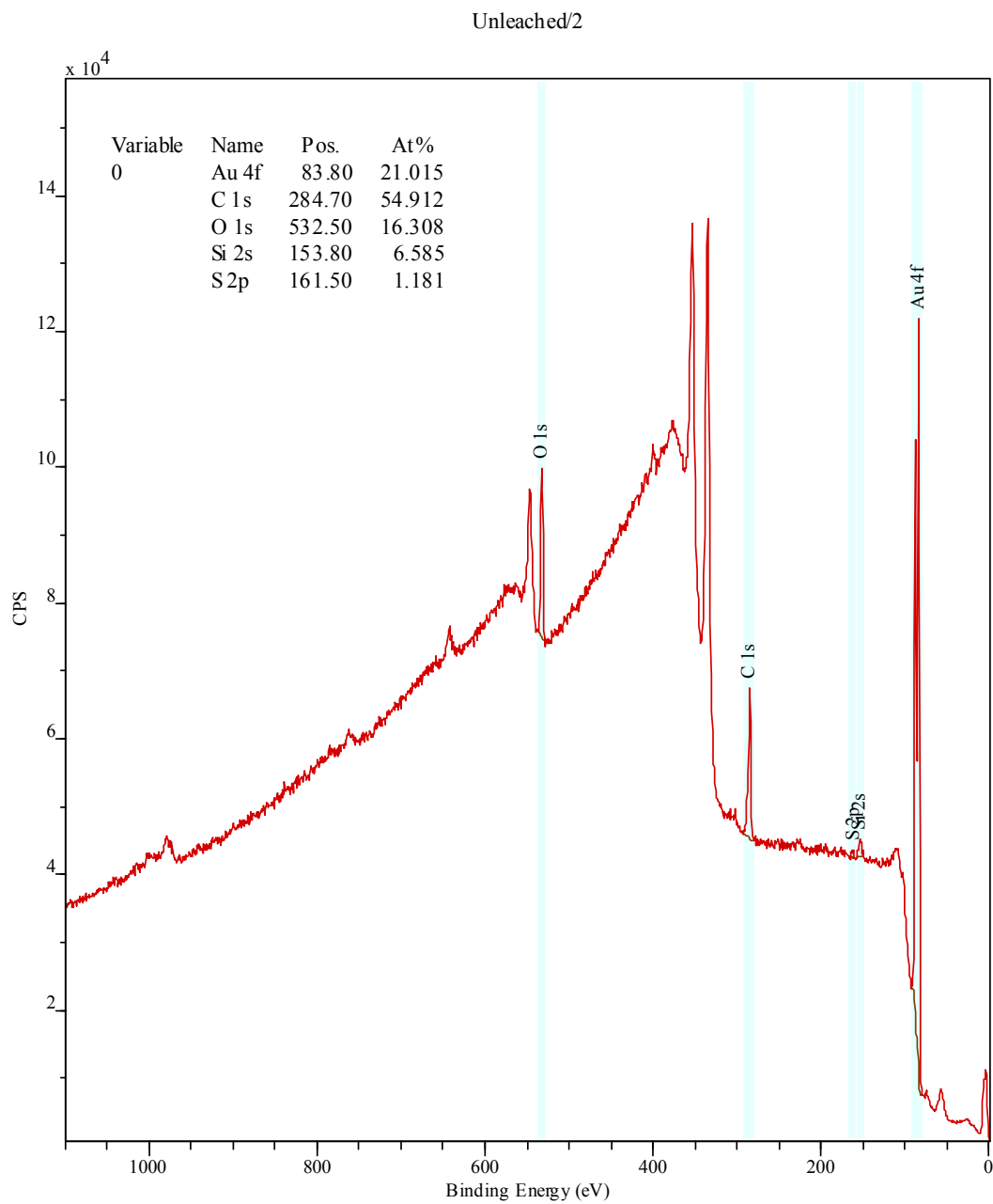
$\text{Cu}^{2+} + \text{en} = \text{Cuen}_2^{2+}$	19.6
---	------

$\text{Cu}^+ + 2\text{en} = \text{Cuen}_2^+$	11.2
--	------

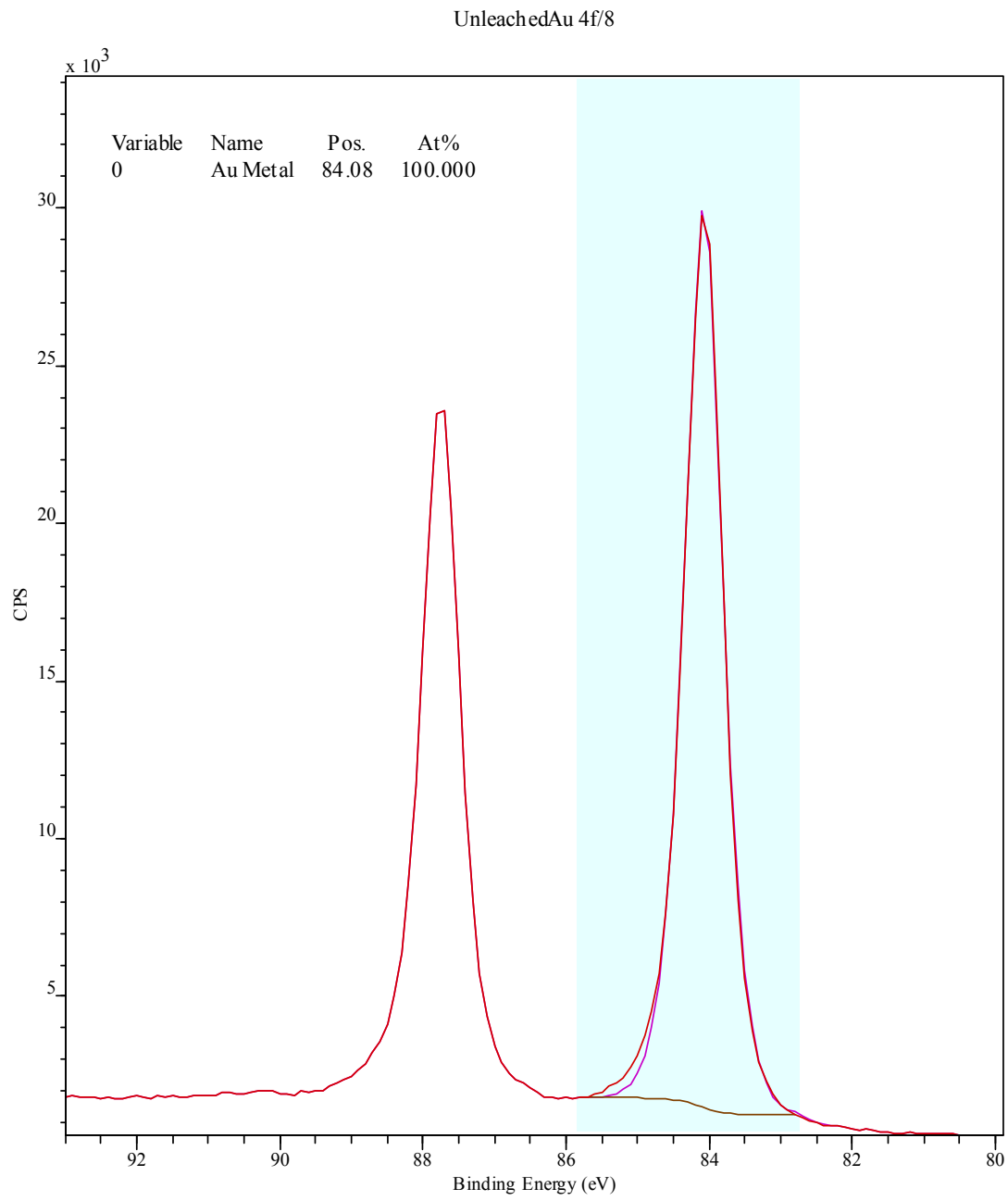
[Sillen et al, 1971 and Wagman et al, 1982]

## Appendix – 2: XPS analysis of gold surface in the presence of galena

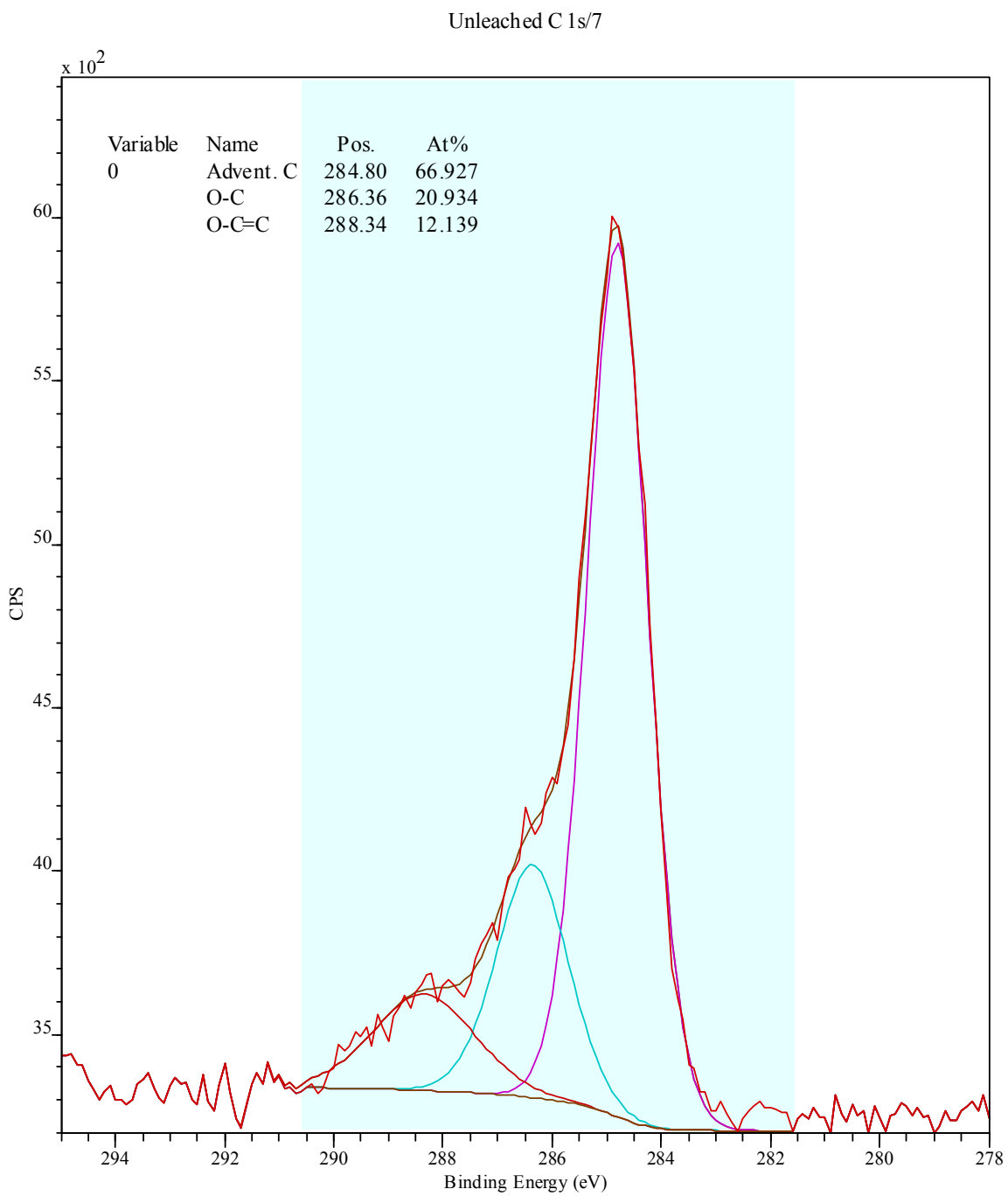
### 1) Surface scan of unleached gold foil: survey



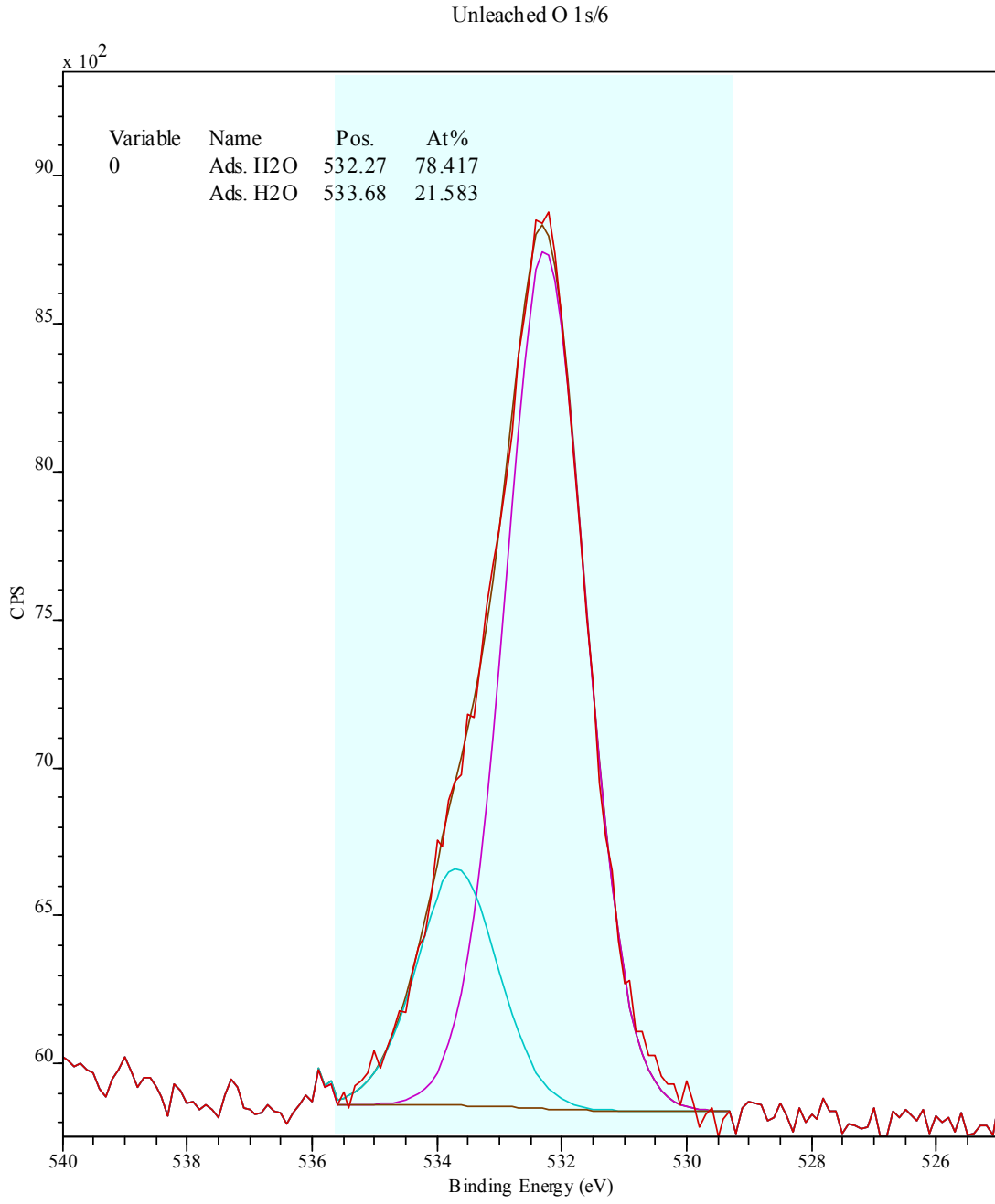
## 2) Surface scan of unleached gold foil: Au 4f



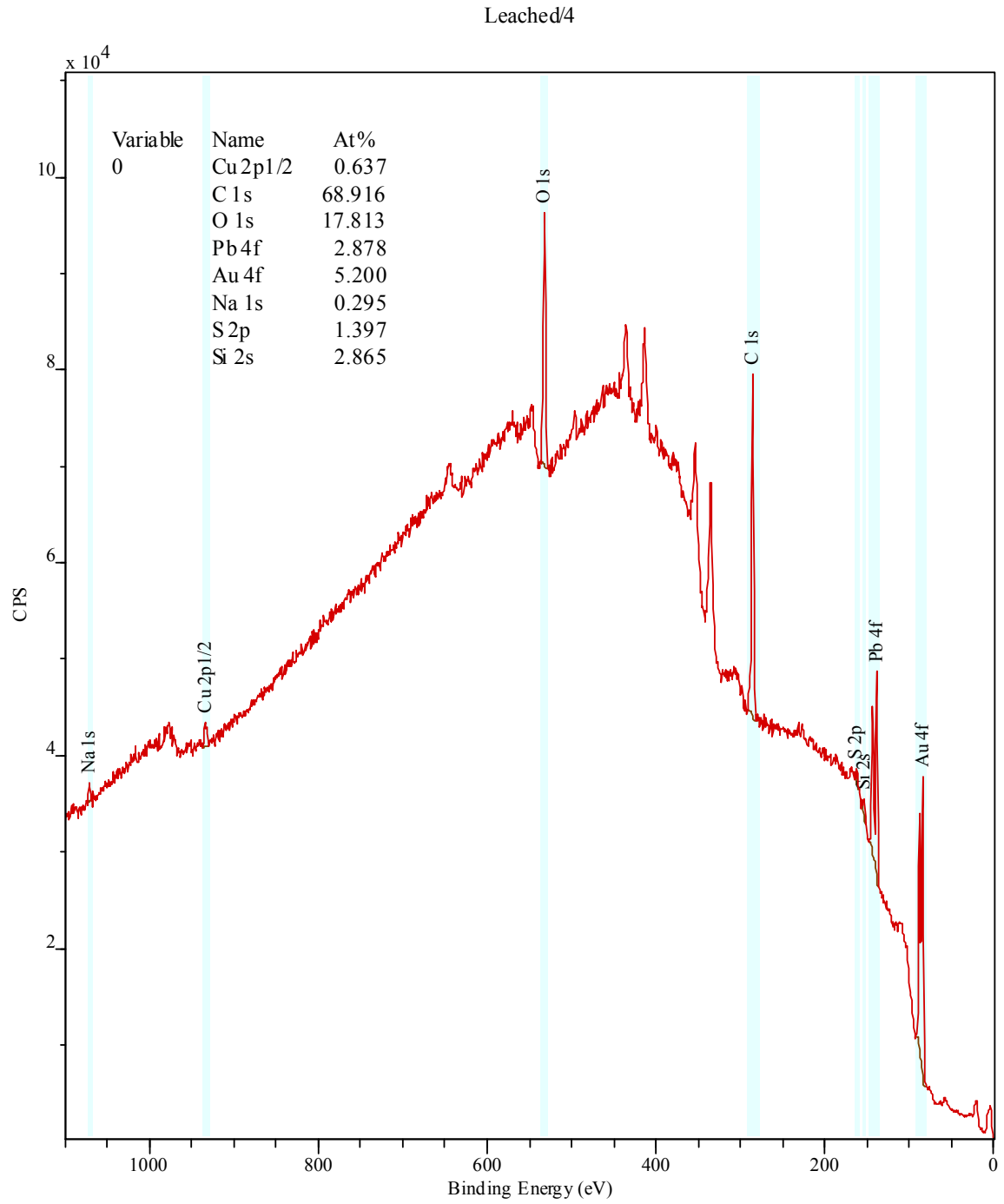
### 3) Surface scan of unleached gold foil: C 1s



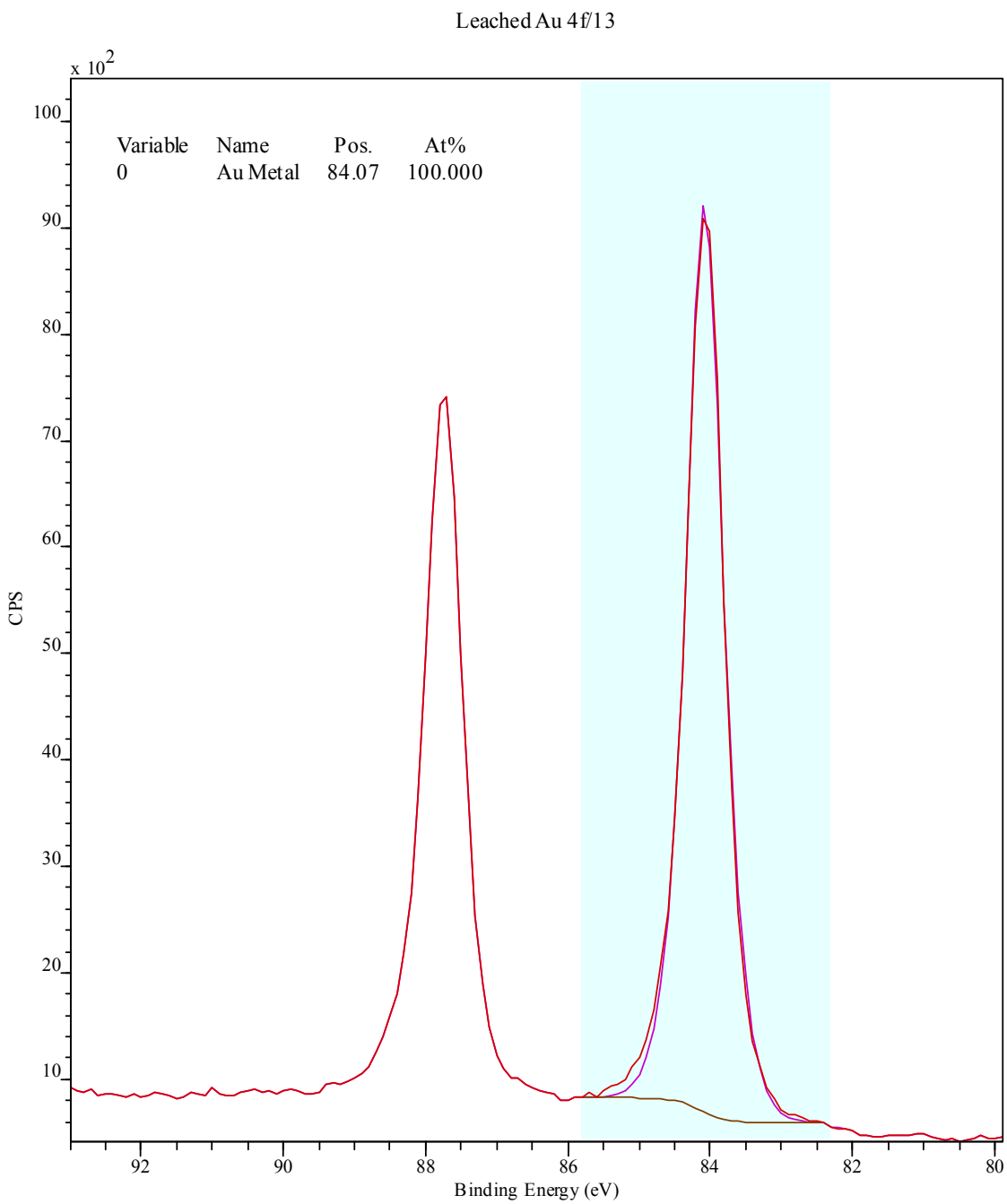
#### 4) Surface scan of unleached gold foil: O 1s



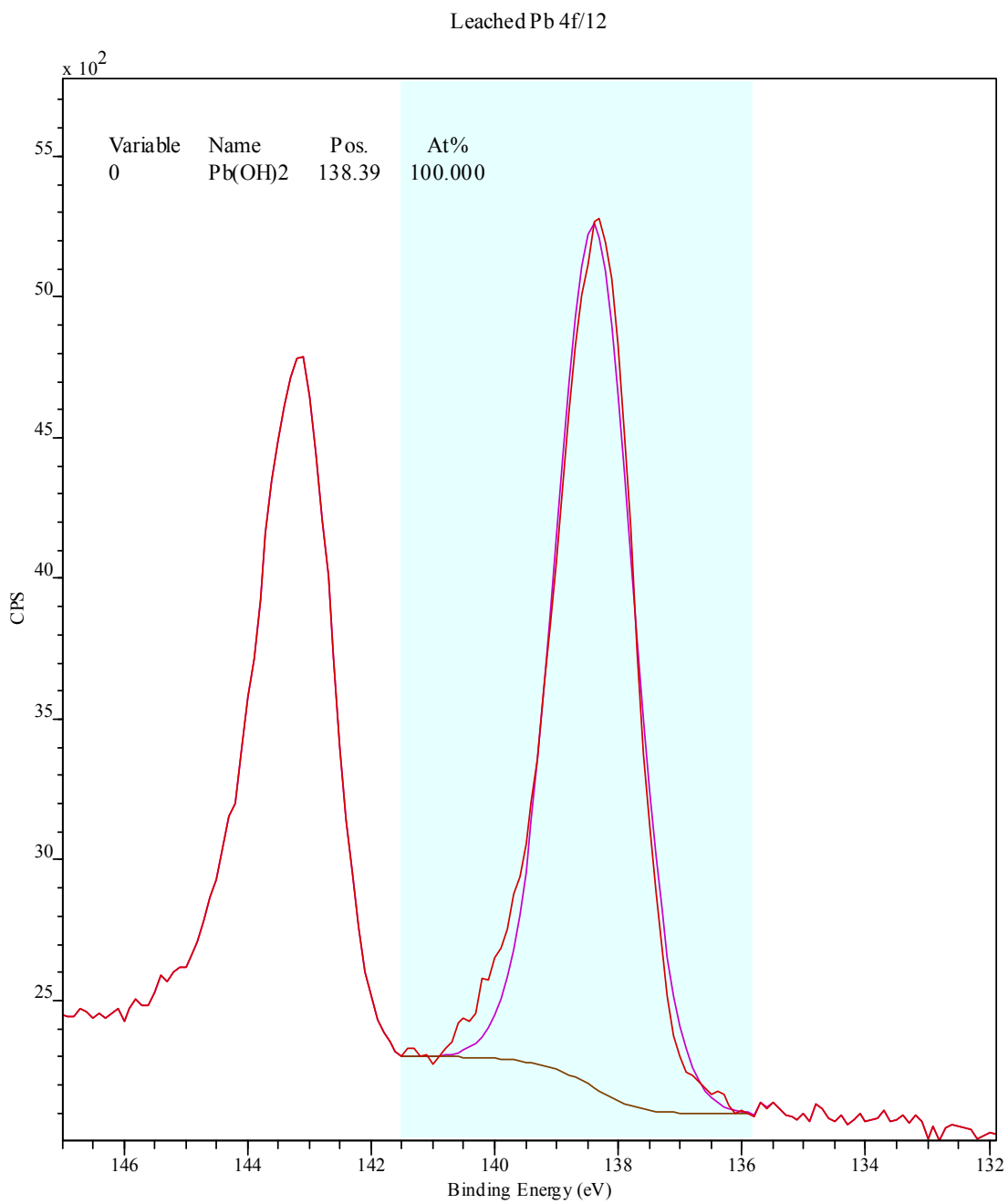
5) Scan of leached gold surface: survey



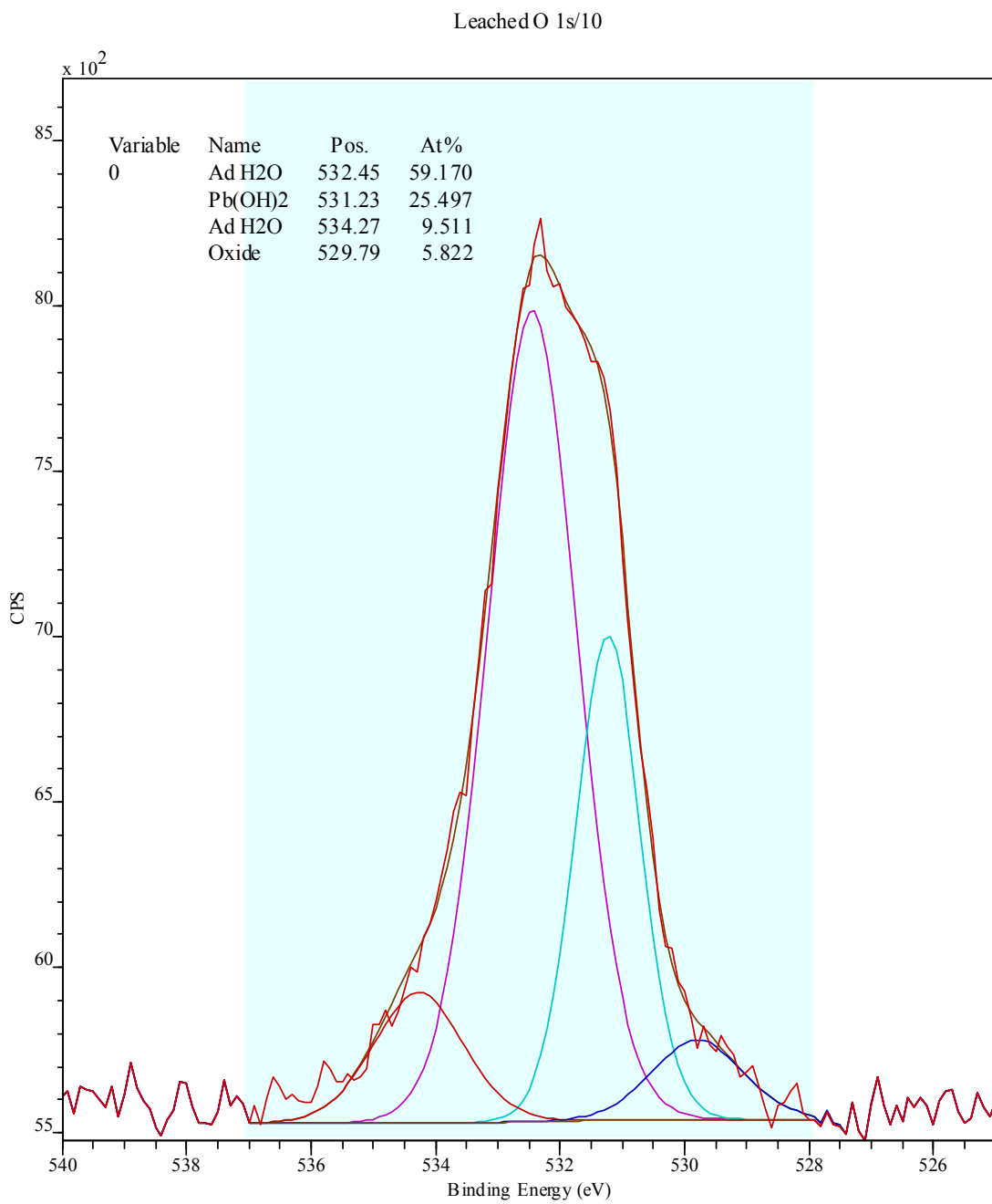
6) Scan of leached gold surface: Au 4f



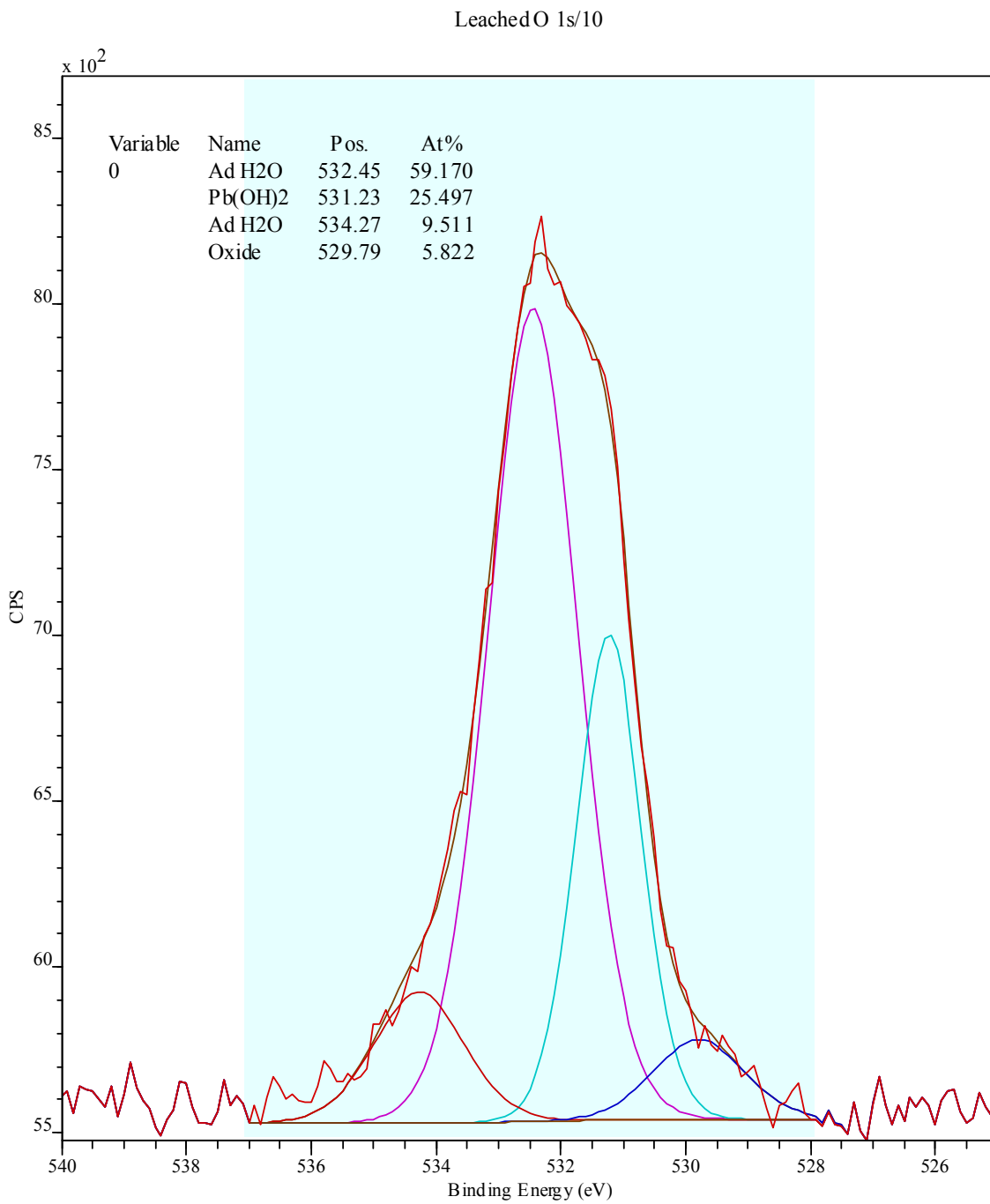
### 7) Scan of leached gold surface: Pb 4f



### 8) Scan of leached gold surface: O 1s



### 9) Scan of leached gold surface: C 1s



## **Appendix – 3: Leaching tests**

**TS leaching test**

<b>Group No.:</b>	S-A1	<b>Date:</b>	Aug-01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	#1 (see p)
<b>Purpose:</b>	Thiosulfate leaching of reference synthetic gold ore						
<b>Procedure:</b>	#1 (see p)						

**Synthetic ore and pulp properties**

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	No	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

**Leaching conditions**

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> supply	pH				
1	0.20	76	0.0	21.1%	9.5				
2	0.20	76	0.0	21.1%	10.2				
3	0.20	76	0.0	21.1%	10.9				
4	0.20	76	0.3	21.1%	9.6				
5	0.20	76	0.3	21.1%	10.0				
6	0.20	76	0.3	21.1%	10.6				
7	0.20	76	0.3	21.1%	11.1				
8	0.20	76	0.3	21.1%	12.7				
9	0.20	76	0.9	21.1%	10.1				
10	0.20	76	0.9	21.1%	10.6				
11	0.20	76	0.9	21.1%	11.2				
12	0.20	76	0.9	21.1%	12.6				
13	0.20	76	1.5	21.1%	10.2				
14	0.20	76	1.5	21.1%	10.8				
15	0.20	76	1.5	21.1%	11.7				
16	0.20	76	1.5	21.1%	12.2				
17	0.20	76	3.0	21.1%	10.6				
18	0.20	76	3.0	21.1%	11.3				
19	0.20	76	3.0	21.1%	12.7				

**Result: Thiosulfate**

**(as ATS (M))**

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	6	0	0.5	3	6
1	0.20		0.17		0		8.2	
2	0.20		0.17		0		7.6	
3	0.20		0.17		0		7.6	
4	0.20		0.17		0		8.2	
5	0.20		0.18		0		7.1	
6	0.20		0.18		0		5.9	
7	0.20		0.18		0		6.5	
8	0.20		0.19		0		3.5	
9	0.20		0.18		0		7.1	
10	0.20		0.18		0		7.1	
11	0.20		0.18		0		6.5	
12	0.20		0.19		0		3.5	
13	0.20		0.18		0		6.5	
14	0.20		0.18		0		5.9	
15	0.20		0.18		0		4.7	
16	0.20		0.19		0		3.5	
17	0.20		0.19		0		3.5	
18	0.20		0.19		0		3.5	
19	0.20		0.19		0		2.4	



**TS leaching test**

<b>Group No.:</b>	S-B1	<b>Date:</b>	Aug-01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Primilinary comparison study						

**Ore properties and pulp conditions**

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

**Leaching conditions**

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral content
1	0.20	76	0.9	21.1%	Limonite 24.0%
2	0.20	76	0.9	21.1%	Hematite 24.0%
3	0.20	76	0.9	21.1%	Pure silicate
4	0.20	76	0.9	21.1%	Magnetite 24.0%
5	0.20	76	0.9	21.1%	Chalcopyrite 4.0%
6	0.20	76	0.9	21.1%	Realgar 0.24%
7	0.20	76	0.9	21.1%	Arsenopyrite 4.0%
8	0.20	76	0.9	21.1%	Chalcocite 4.0%
9	0.20	76	0.9	21.1%	Pyrite 16.0%
10	0.20	76	0.9	21.1%	Orpiment 0.24%
11	0.20	76	0.9	21.1%	Bornite 4.0%
12	0.20	76	0.9	21.1%	Pyrrhotite 16.0%
13	0.20	76	0.9	21.1%	Galena 2.4%
14	0.20	76	0.9	21.1%	Litharge 2.4%

**Result: Thiosulfate**

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		6.0	
2	0.20		0.18		0		6.8	
3	0.20		0.18		0		7.1	
4	0.20		0.18		0		6.5	
5	0.20		0.15		0		15.3	
6	0.20		0.17		0		7.9	
7	0.20		0.17		0		7.7	
8	0.20		0.14		0		17.6	
9	0.20		0.13		0		22.0	
10	0.20		0.18		0		5.1	
11	0.20		0.11		0		25.6	
12	0.20		0.14		0		18.9	
13	0.20		0.18		0		5.0	
14	0.20		0.15		0		16.0	

**Result: Metal balance**

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.44	0.22	0.14	0.01	93.03%	6.97%	3.10
2	100	50	1.47	0.22	0.15	0.01	93.02%	6.98%	3.17
3	100	50	1.45	0.22	0.15	0.01	92.96%	7.04%	3.12
4	100	50	1.46	0.28	0.15	0.01	91.36%	8.64%	3.20
5	100	50	1.38	0.35	0.14	0.02	88.87%	11.13%	3.11
6	100	50	1.38	0.45	0.14	0.02	85.86%	14.14%	3.21
7	100	50	1.31	0.53	0.13	0.03	83.02%	16.98%	3.15
8	100	50	1.29	0.55	0.13	0.03	82.50%	17.50%	3.13
9	100	50	1.29	0.62	0.13	0.03	80.55%	19.45%	3.20
10	100	50	1.17	0.89	0.12	0.04	72.38%	27.62%	3.22
11	100	50	0.60	1.94	0.06	0.10	38.26%	61.74%	3.15
12	100	50	1.43	0.26	0.14	0.01	91.61%	8.39%	3.12
13	100	50	0.32	2.53	0.03	0.13	20.00%	80.00%	3.16
14	100	50	0.16	2.81	0.02	0.14	10.40%	89.60%	3.14

**Result: Leaching kinetics**

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	1.20	1.44		0	77.33%	93.03%	
2	0	1.22	1.47		0	77.15%	93.02%	
3	0	1.31	1.45		0	83.90%	92.96%	
4	0	1.25	1.46		0	78.05%	91.36%	
5	0	1.20	1.38		0	76.98%	88.87%	
6	0	1.01	1.38		0	63.19%	85.86%	
7	0	0.80	1.31		0	50.50%	83.02%	
8	0	0.33	1.29		0	20.83%	82.50%	
9	0	0.81	1.29		0	50.45%	80.55%	
10	0	0.59	1.17		0	36.76%	72.38%	
11	0	0.16	0.60		0	10.01%	38.26%	
12	0	0.80	1.43		0	51.30%	91.61%	
13	0	0.10	0.32		0	6.40%	20.00%	
14	0	0.08	0.16		0	4.80%	10.40%	

### TS leaching test

<b>Group No.:</b>	S-A4	<b>Date:</b>	Aug/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Oxygen %						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	No	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %				
1	0.20	76	0.9	21.14				
2	0.20	76	0.9	6.39				
3	0.20	76	0.9	4.18				
4	0.20	76	0.9	0.74				
5	0.20	76	0.9	0.25				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		7.2	
2	0.20		0.18		0		5.3	
3	0.20		0.18		0		4.7	
4	0.20		0.19		0		3.5	
5	0.20		0.20		0		0.6	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.49	0.22	0.15	0.01	93.21%	6.79%	3.20
2	100	50	1.47	0.23	0.15	0.01	92.77%	7.23%	3.16
3	100	50	1.50	0.21	0.15	0.01	93.40%	6.60%	3.22
4	100	50	1.48	0.22	0.15	0.01	93.00%	7.00%	3.19
5	100	50	1.27	0.61	0.13	0.03	80.58%	19.42%	3.15

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0		1.49		0		93.21%	
2	0		1.47		0		92.77%	
3	0		1.50		0		93.40%	
4	0		1.48		0		93.00%	
5	0		1.27		0		80.58%	

### TS leaching test

<b>Group No.:</b>	S-A5	<b>Date:</b>	Aug/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz -Au Ore
<b>Purpose:</b>	Effect of ammonia at 0.7% oxygen						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	No	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %				
1	0.20	76	0.3	0.7				
2	0.20	76	0.9	0.7				
3	0.20	76	1.5	0.7				
4	0.20	76	3.0	0.7				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		4.6	
2	0.20		0.19		0		3.5	
3	0.20		0.19		0		2.3	
4	0.20		0.20		0		1.2	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.40	0.31	0.14	0.02	90.14%	9.86%	3.11
2	100	50	1.48	0.22	0.15	0.01	92.96%	7.04%	3.18
3	100	50	1.47	0.22	0.15	0.01	93.01%	6.99%	3.17
4	100	50	1.48	0.18	0.15	0.01	94.30%	5.70%	3.13

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	1.32	1.40		0	85.18%	90.14%	
2	0	1.37	1.48		0	86.18%	92.96%	
3	0	1.37	1.47		0	86.58%	93.01%	
4	0	1.36	1.48		0	86.85%	94.30%	

### TS leaching test

<b>Group No.:</b>	S-A6	<b>Date:</b>	Sep/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz -Au Ore
<b>Purpose:</b>	Effect of Cu at 0.7% oxygen						

#### Synthetic ore and pulp properties

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	No	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %				
1	0.20	23	0.9	0.7				
2	0.20	76	0.9	0.7				
3	0.20	228	0.9	0.7				
4	0.20	760	0.9	0.7				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.20		0.19		0		2.9	
2	0.20		0.19		0		3.5	
3	0.20		0.19		0		3.5	
4	0.20		0.19		0		4.1	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.68	1.78	0.07	0.09	43.35%	56.65%	3.15
2	100	50	1.44	0.22	0.14	0.01	92.93%	7.07%	3.11
3	100	50	1.44	0.29	0.14	0.01	90.97%	9.03%	3.17
4	100	50	1.44	0.33	0.14	0.02	89.70%	10.30%	3.22

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	0.2793103	0.6828		0	17.73%	43.35%	
2	0	1.3356224	1.445		0	85.89%	92.93%	
3	0	1.3507635	1.4418		0	85.22%	90.97%	
4	0	1.3086207	1.4442		0	81.28%	89.70%	

### TS leaching test

<b>Group No.:</b>	S-A7	<b>Date:</b>	Sep/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz -Au Ore
<b>Purpose:</b>	Effect of ATS at 0.7% oxygen						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	No	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %				
1	0.033	76	0.9	0.7				
2	0.067	76	0.9	0.7				
3	0.133	76	0.9	0.7				
4	0.200	76	0.9	0.7				
5	0.333	76	0.9	0.7				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.03		0.03		0		1.1	
2	0.07		0.06		0		1.3	
3	0.13		0.13		0		2.2	
4	0.20		0.19		0		3.5	
5	0.33		0.32		0		5.3	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.17	0.83	0.12	0.04	73.89%	26.11%	3.18
2	100	50	1.42	0.26	0.14	0.01	91.63%	8.37%	3.11
3	100	50	1.48	0.25	0.15	0.01	92.12%	7.88%	3.22
4	100	50	1.47	0.22	0.15	0.01	92.93%	7.07%	3.16
5	100	50	1.47	0.19	0.15	0.01	93.79%	6.21%	3.13

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	0.81	1.17		0	50.74%	73.89%	
2	0	1.13	1.42		0	72.91%	91.63%	
3	0	1.37	1.48		0	85.22%	92.12%	
4	0	1.38	1.47		0	87.19%	92.93%	
5	0	1.40	1.47		0	89.16%	93.79%	

### TS leaching test

<b>Group No.:</b>	S-C1	<b>Date:</b>	Oct/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au ore
<b>Purpose:</b>	Effect of pyrite						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Mineral %		
1	0.20	76	0.9	21.1%	Pyrite	0%		
2	0.20	76	0.9	21.1%		4%		
3	0.20	76	0.9	21.1%		8%		
4	0.20	76	0.9	21.1%		16%		
5	0.20	76	0.9	21.1%		28%		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	3	10	-	0	3	10	-
1	0.20	0.18	0.15		0	7.3	15.8	
2	0.20	0.16	0.13		0	12.6	21.9	
3	0.20	0.15	0.10		0	16.2	28.8	
4	0.20	0.13	0.08		0	22.0	36.6	
5	0.20	0.11	0.07		0	25.5	39.2	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.48	0.25	0.15	0.01	92.32%	7.68%	3.20
2	100	50	1.33	0.47	0.13	0.02	85.07%	14.93%	3.12
3	100	50	1.31	0.53	0.13	0.03	83.20%	16.80%	3.16
4	100	50	1.27	0.61	0.13	0.03	80.55%	19.45%	3.15
5	100	50	1.15	0.82	0.11	0.04	73.79%	26.21%	3.11

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	1.34	1.48		0	83.56%	92.32%	
2	0	0.94	1.33		0	60.05%	85.07%	
3	0	0.90	1.31		0	56.90%	83.20%	
4	0	0.79	1.27		0	50.45%	80.55%	
5	0	0.68	1.15		0	43.82%	73.79%	

### TS leaching test

<b>Group No.:</b>	S-C2	<b>Date:</b>	Sep/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of additional hydroxide ion						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	NaOH (g/L)		
1	0.20	76	0.9	21.1%	16% Pyrite	0		
2	0.20	76	0.9	21.1%		0.5		
3	0.20	76	0.9	21.1%		1		
4	0.20	76	0.9	21.1%		4		
5	0.20	76	0.9	21.1%		12		

#### Result: Thiosulfate

(as ATS (M))

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20				0		22.1	
2	0.20				0		19.9	
3	0.20				0		18.9	
4	0.20				0		17.2	
5	0.20				0		9.5	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.27	0.59	0.13	0.03	81.01%	18.99%	3.13
2	100	50	1.40	0.35	0.14	0.02	88.92%	11.08%	3.16
3	100	50	1.43	0.25	0.14	0.01	92.02%	7.98%	3.11
4	100	50	1.46	0.26	0.15	0.01	91.80%	8.20%	3.18
5	100	50	1.29	0.63	0.13	0.03	80.30%	19.70%	3.21

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.8014256	1.27		0	51.21%	81.01%	
2	0	0.9264434	1.40		0	58.64%	88.92%	
3	0	0.9409558	1.43		0	60.51%	92.02%	
4	0	0.954986	1.46		0	60.06%	91.80%	
5	0	0.56496	1.29		0	35.20%	80.30%	

### TS leaching test

<b>Group No.:</b>	S-C3	<b>Date:</b>	Sep/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu						

#### Synthetic ore and pulp properties

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	23	0.9	21.1%	16% Pyrite				
2	0.20	76	0.9	21.1%					
3	0.20	228	0.9	21.1%					
4	0.20	760	0.9	21.1%					
5	0.20	2280	0.9	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.20		0.15		0		15.3	
2	0.20		0.13		0		21.9	
3	0.20		0.12		0		23.8	
4	0.20		0.11		0		27.0	
5	0.20		0.09		0		33.6	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.12	0.92	0.11	0.05	70.92%	29.08%	3.17
2	100	50	1.25	0.60	0.13	0.03	80.70%	19.30%	3.11
3	100	50	1.42	0.31	0.14	0.02	90.18%	9.82%	3.16
4	100	50	1.44	0.25	0.14	0.01	92.09%	7.91%	3.13
5	100	50	1.28	0.60	0.13	0.03	81.14%	18.86%	3.16

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	0.77	1.12		0	48.65%	70.92%	
2	0	0.78	1.25		0	50.29%	80.70%	
3	0	0.97	1.42		0	61.50%	90.18%	
4	0	1.06	1.44		0	67.87%	92.09%	
5	0	1.03	1.28		0	65.07%	81.14%	

### TS leaching test

<b>Group No.:</b>	S-C4	<b>Date:</b>	Oct/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ATS						

#### Synthetic ore and pulp properties

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.07	228	0.9	21.1%	16% Pyrite			
2	0.20	228	0.9	21.1%				
3	0.27	228	0.9	21.1%				
4	0.40	228	0.9	21.1%				
5	0.44	228	0.9	21.1%				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.07		0.04		0		7.4	
2	0.20		0.12		0		23.9	
3	0.27		0.18		0		26.0	
4	0.40		0.33		0		21.3	
5	0.44		0.37		0		20.2	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50		3.17	0.00	0.16	0.00%	100.00%	3.17
2	100	50		3.11	0.00	0.16	0.00%	100.00%	3.11
3	100	50		3.18	0.00	0.16	0.00%	100.00%	3.18
4	100	50		3.23	0.00	0.16	0.00%	100.00%	3.23
5	100	50		3.16	0.00	0.16	0.00%	100.00%	3.16

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	0.89	1.26		0	55.90%	79.52%	
2	0	0.95	1.39		0	60.81%	89.50%	
3	0	1.00	1.45		0	63.10%	90.92%	
4	0	0.95	1.40		0	58.60%	86.40%	
5	0	0.83	1.33		0	52.82%	83.93%	

### TS leaching test

<b>Group No.:</b>	S-C5	<b>Date:</b>	Oct/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ammonia						

#### Synthetic ore and pulp properties

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.20	228	0.3	21.1%	16% Pyrite			
2	0.20	228	0.9	21.1%				
3	0.20	228	1.8	21.1%				
4	0.20	228	2.7	21.1%				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.12		0		24.6	
2	0.20		0.12		0		23.8	
3	0.20		0.14		0		18.6	
4	0.20		0.15		0		13.9	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.24	0.69	0.12	0.03	78.30%	21.70%	3.18
2	100	50	1.44	0.31	0.14	0.02	90.28%	9.72%	3.20
3	100	50	1.46	0.23	0.15	0.01	92.69%	7.31%	3.15
4	100	50	1.47	0.19	0.15	0.01	93.86%	6.14%	3.13

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.61	1.24		0	38.19%	78.30%	
2	0	0.95	1.44		0	59.20%	90.28%	
3	0	1.05	1.46		0	66.68%	92.69%	
4	0	1.11	1.47		0	70.71%	93.86%	

### TS leaching test

<b>Group No.:</b>	S-C6	<b>Date:</b>	Nov/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of oxygen %						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.20	76	0.9	21.1%	16% Pyrite			
2	0.20	76	0.9	6.4%				
3	0.20	76	0.9	4.2%				
4	0.20	76	0.9	0.7%				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.13		0		21.8	
2	0.20		0.15		0		15.2	
3	0.20		0.16		0		12.1	
4	0.20		0.18		0		7.2	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.26	0.61	0.13	0.03	80.51%	19.49%	3.12
2	100	50	1.27	0.60	0.13	0.03	80.83%	19.17%	3.15
3	100	50	1.28	0.65	0.13	0.03	79.60%	20.40%	3.21
4	100	50	1.20	0.77	0.12	0.04	75.77%	24.23%	3.17

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.770016	1.26		0	49.36%	80.51%	
2	0	0.50715	1.27		0	32.20%	80.83%	
3	0	0.4800555	1.28		0	29.91%	79.60%	
4	0	0.4230365	1.20		0	26.69%	75.77%	

### TS leaching test

<b>Group No.:</b>	S-C7	<b>Date:</b>	Nov-01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu at 0.7% oxygen						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.20	76	0.9	0.7	16% Pyrite			
2	0.20	228	0.9	0.7				
3	0.20	760	0.9	0.7				
4	0.20	2280	0.9	0.7				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.8	
2	0.20		0.17		0		8.8	
3	0.20		0.16		0		11.9	
4	0.20		0.16		0		13.2	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.19	0.77	0.12	0.04	75.60%	24.40%	3.16
2	100	50	1.33	0.45	0.13	0.02	85.50%	14.50%	3.12
3	100	50	1.40	0.45	0.14	0.02	86.30%	13.70%	3.25
4	100	50	1.43	0.32	0.14	0.02	89.80%	10.20%	3.18

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.4424	1.19		0	28.00%	75.60%	
2	0	0.78312	1.33		0	50.20%	85.50%	
3	0	0.97175	1.40		0	59.80%	86.30%	
4	0	1.01601	1.43		0	63.90%	89.80%	

### TS leaching test

<b>Group No.:</b>	S-C9	<b>Date:</b>	Nov/01	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of phosphate ion						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Na <sub>3</sub> PO <sub>4</sub> (g/L)		
1	0.20	76	0.9	0.7	No	0		
2	0.20	76	0.9	0.7		1		
3	0.20	76	0.9	0.7		4		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.5	
2	0.20		0.18		0		7.4	
3	0.20		0.17		0		7.5	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.46	0.25	0.15	0.01	92.01%	7.99%	3.18
2	100	50	1.41	0.29	0.14	0.01	90.62%	9.38%	3.12
3	100	50	1.43	0.41	0.14	0.02	87.56%	12.44%	3.27

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	1.34	1.46		0	84.22%	92.01%	
2	0	1.27	1.41		0	81.63%	90.62%	
3	0	1.26	1.43		0	77.23%	87.56%	

### TS leaching test

<b>Group No.:</b>	S-C11	<b>Date:</b>	Jan-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of pre-treatment in presence of pyrrhotite						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Pretreatment (1h, 21.1% oxygen)
1	0.20	76	0.9	0.7%	16% Pyrite	No
2	0.20	76	0.9	0.7%		Water
3	0.20	76	0.9	0.7%		Cu 3.6 mM
4	0.20	76	0.9	0.7%		Cu 3.6 mM, AH 0.9 M
5	0.20	76	0.9	0.7%		Na <sub>3</sub> PO <sub>4</sub> 10.5 mM

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)			Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3
1	0.20		0.17		0		7.8
2	0.20		0.17		0		7.8
3	0.20		0.18		0		7.3
4	0.20		0.17		0		10.2
5	0.20		0.18		0		5.2

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.22	0.76	0.12	0.04	76.28%	23.72%	3.20
2	100	50	1.37	0.49	0.14	0.02	84.81%	15.19%	3.23
3	100	50	1.45	0.40	0.14	0.02	87.93%	12.07%	3.29
4	100	50	1.37	0.48	0.14	0.02	85.08%	14.92%	3.22
5	100	50	1.40	0.38	0.14	0.02	88.02%	11.98%	3.18

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.47	1.22		0	29.16%	76.28%	
2	0	0.79	1.37		0	48.93%	84.81%	
3	0	0.82	1.45		0	50.09%	87.93%	
4	0	0.95	1.37		0	59.25%	85.08%	
5	0	1.06	1.40		0	66.37%	88.02%	

### TS leaching test

<b>Group No.:</b>	S-C12	<b>Date:</b>	Feb-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of galena						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	PbS (%)		
1	0.20	76	0.9	21.1%	16% Pyrite	0.00%		
2	0.20	76	0.9	21.1%		0.20%		
3	0.20	76	0.9	21.1%		0.40%		
4	0.20	76	0.9	21.1%		0.80%		
5	0.20	76	0.9	21.1%		1.60%		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.13		0		21.6	
2	0.20		0.13		0		21.9	
3	0.20		0.13		0		21.5	
4	0.20		0.13		0		20.9	
5	0.20		0.13		0		19.7	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.29	0.62	0.13	0.03	80.67%	19.33%	3.21
2	100	50	1.34	0.46	0.13	0.02	85.29%	14.71%	3.15
3	100	50	1.37	0.37	0.14	0.02	88.12%	11.88%	3.10
4	100	50	1.00	1.21	0.10	0.06	62.33%	37.67%	3.20
5	100	50	0.53	2.07	0.05	0.10	33.76%	66.24%	3.12

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.76	1.29		0	47.23%	80.67%	
2	0	0.94	1.34		0	59.81%	85.29%	
3	0	1.09	1.37		0	70.17%	88.12%	
4	0	0.48	1.00		0	30.12%	62.33%	
5	0	0.16	0.53		0	10.12%	33.76%	

### TS leaching test

<b>Group No.:</b>	S-D1	<b>Date:</b>	Feb-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of pyrrhotite						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Mineral %		
1	0.20	76	0.9	21.1%	Pyrrhotite	0%		
2	0.20	76	0.9	21.1%		4%		
3	0.20	76	0.9	21.1%		8%		
4	0.20	76	0.9	21.1%		16%		
5	0.20	76	0.9	21.1%		24%		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.5	
2	0.20		0.16		0		12.1	
3	0.20		0.15		0		13.9	
4	0.20		0.14		0		18.9	
5	0.20		0.13		0		19.6	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.46	0.20	0.15	0.01	93.62%	6.38%	3.12
2	100	50	1.45	0.26	0.14	0.01	91.78%	8.22%	3.15
3	100	50	1.45	0.31	0.15	0.02	90.26%	9.74%	3.22
4	100	50	1.46	0.27	0.15	0.01	91.61%	8.39%	3.19
5	100	50	1.42	0.29	0.14	0.01	90.82%	9.18%	3.13

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	1.31	1.46		0	83.86%	93.62%	
2	0	1.05	1.45		0	66.70%	91.78%	
3	0	1.05	1.45		0	65.20%	90.26%	
4	0	0.82	1.46		0	51.30%	91.61%	
5	0	0.76	1.42		0	48.60%	90.82%	

### TS leaching test

<b>Group No.:</b>	S-D2	<b>Date:</b>	Mar-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of NaOH						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	NaOH		
1	0.20	76	0.9	21.1%	16% Pyrrhotite	0.00		
2	0.20	76	0.9	21.1%		0.70		
3	0.20	76	0.9	21.1%		1.40		
4	0.20	76	0.9	21.1%		4.28		
5	0.20	76	0.9	21.1%		12.36		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		18.9	
2	0.20		0.14		0		18.9	
3	0.20		0.14		0		18.0	
4	0.20		0.15		0		15.6	
5	0.20		0.15		0		13.8	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.45	0.28	0.15	0.01	91.19%	8.81%	3.19
2	100	50	1.47	0.27	0.15	0.01	91.67%	8.33%	3.20
3	100	50	1.44	0.28	0.14	0.01	91.10%	8.90%	3.16
4	100	50	1.45	0.32	0.14	0.02	90.18%	9.82%	3.21
5	100	50	1.38	0.40	0.14	0.02	87.32%	12.68%	3.15

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.82	1.45		0	51.33%	91.19%	
2	0	0.86	1.47		0	53.92%	91.67%	
3	0	0.81	1.44		0	51.39%	91.10%	
4	0	0.68	1.45		0	42.56%	90.18%	
5	0	0.56	1.38		0	35.87%	87.32%	

### TS leaching test

<b>Group No.:</b>	S-D3	<b>Date:</b>	Mar-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of phosphate						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Na <sub>3</sub> PO <sub>4</sub>			
1	0.20	76	0.9	21.1%	16% Pyrrhotite	0			
2	0.20	76	0.9	21.1%		1.36			
3	0.20	76	0.9	21.1%		4.28			
4	0.20	76	0.9	21.1%		12.36			

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		18.9	
2	0.20		0.14		0		18.6	
3	0.20		0.14		0		17.4	
4	0.20		0.15		0		15.6	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.41	0.28	0.14	0.01	91.10%	8.90%	3.10
2	100	50	1.45	0.29	0.14	0.01	91.00%	9.00%	3.18
3	100	50	1.44	0.24	0.14	0.01	92.20%	7.80%	3.12
4	100	50	1.45	0.24	0.15	0.01	92.30%	7.70%	3.15

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.80	1.41		0	51.30%	91.10%	
2	0	0.88	1.45		0	55.20%	91.00%	
3	0	0.89	1.44		0	57.20%	92.20%	
4	0	0.94	1.45		0	59.50%	92.30%	

### TS leaching test

<b>Group No.:</b>	S-D4	<b>Date:</b>	Mar-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Oxygen%						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	76	0.9	21.1%	16% Pyrrhotite				
2	0.20	76	0.9	6.4%					
3	0.20	76	0.9	4.2%					
4	0.20	76	0.9	0.7%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		19.0	
2	0.20		0.17		0		8.6	
3	0.20		0.18		0		6.1	
4	0.20		0.19		0		3.5	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.44	0.25	0.14	0.01	91.98%	8.02%	3.13
2	100	50	1.35	0.43	0.13	0.02	86.22%	13.78%	3.12
3	100	50	1.33	0.54	0.13	0.03	83.20%	16.80%	3.20
4	100	50	1.04	1.18	0.10	0.06	63.89%	36.11%	3.26

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.81	1.44		0	51.95%	91.98%	
2	0	0.63	1.35		0	40.26%	86.22%	
3	0	0.54	1.33		0	33.89%	83.20%	
4	0	0.35	1.04		0	21.25%	63.89%	

### TS leaching test

<b>Group No.:</b>	S-D5	<b>Date:</b>	Apr-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect Cu at 0.7% oxygen						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	76	0.9	0.7%	16% Pyrrhotite				
2	0.20	228	0.9	0.7%					
3	0.20	760	0.9	0.7%					
4	0.20	2280	0.9	0.7%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.19		0		3.5	
2	0.20		0.19		0		4.0	
3	0.20		0.18		0		4.6	
4	0.20		0.18		0		5.5	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.02	1.08	0.10	0.05	65.40%	34.60%	3.13
2	100	50	1.33	0.46	0.13	0.02	85.20%	14.80%	3.12
3	100	50	1.49	0.31	0.15	0.02	90.60%	9.40%	3.28
4	100	50	1.41	0.34	0.14	0.02	89.10%	10.90%	3.16

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.33	1.02		0	21.00%	65.40%	
2	0	0.62	1.33		0	39.80%	85.20%	
3	0	1.15	1.49		0	69.90%	90.60%	
4	0	1.00	1.41		0	63.20%	89.10%	

### TS leaching test

<b>Group No.:</b>	S-D7	<b>Date:</b>	Apr-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of pre-treatment in presence of pyrrhotite						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Pretreatment (1h, 21.1% oxygen)
1	0.20	76	0.9	0.7%	16% Pyrrhotite	No
2	0.20	76	0.9	0.7%		Water
3	0.20	76	0.9	0.7%		Cu 1.2 mM
4	0.20	76	0.9	0.7%		Cu 1.2 mM, AH 0.9 M
5	0.20	76	0.9	0.7%		Na3PO4 10.5 mM

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.19		0		3.5	
2	0.20		0.18		0		5.5	
3	0.20		0.18		0		5.5	
5	0.20		0.18		0		4.6	
0	0.20		0.19		0		3.1	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.02	1.08	0.10	0.05	65.48%	34.52%	3.12
2	100	50	1.37	0.42	0.14	0.02	86.64%	13.36%	3.17
3	100	50	1.38	0.48	0.14	0.02	85.23%	14.77%	3.23
5	100	50	1.39	0.32	0.14	0.02	89.67%	10.33%	3.09
0	100	50	1.41	0.33	0.14	0.02	89.46%	10.54%	3.16

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.33	1.02		0	21.12%	65.48%	
2	1	0.84	1.37		0	52.80%	86.64%	
3	2	0.64	1.38		0	39.88%	85.23%	
5	3	0.88	1.39		0	56.87%	89.67%	
0	4	0.86	1.41		0	54.33%	89.46%	

### TS leaching test

<b>Group No.:</b>	S-D8	<b>Date:</b>	Apr-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ATS						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.040	76	0.9	21.1%	16% Pyrrhotite				
2	0.067	76	0.9	21.1%					
3	0.200	76	0.9	21.1%					
4	0.334	76	0.9	21.1%					
5	0.468	76	0.9	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.04		0.02		0		5.9	
2	0.07		0.04		0		8.4	
3	0.20		0.14		0		18.6	
4	0.33		0.26		0		20.6	
5	0.47		0.40		0		19.8	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.20	0.78	0.12	0.04	75.35%	24.65%	3.18
2	100	50	1.34	0.45	0.13	0.02	85.62%	14.38%	3.13
3	100	50	1.47	0.27	0.15	0.01	91.58%	8.42%	3.20
4	100	50	1.25	0.65	0.12	0.03	79.22%	20.78%	3.15
5	100	50	0.95	1.28	0.09	0.06	59.68%	40.32%	3.18

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.37	1.20		0	23.21%	75.35%	
2	0	0.78	1.34		0	49.80%	85.62%	
3	0	0.82	1.47		0	51.33%	91.58%	
4	0	0.53	1.25		0	33.68%	79.22%	
5	0	0.21	0.95		0	13.25%	59.68%	

### TS leaching test

<b>Group No.:</b>	S-D9	<b>Date:</b>	May-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ammonia						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.200	76	0.3	21.1%	16% Pyrrhotite				
2	0.200	76	0.9	21.1%					
3	0.200	76	1.8	21.1%					
4	0.200	76	3.0	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.13		0		21.6	
2	0.20		0.14		0		18.6	
3	0.20		0.15		0		13.5	
4	0.20		0.17		0		10.2	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.20	0.79	0.12	0.04	75.29%	24.71%	3.19
2	100	50	1.45	0.26	0.14	0.01	91.81%	8.19%	3.15
3	100	50	1.47	0.24	0.15	0.01	92.37%	7.63%	3.19
4	100	50	1.44	0.23	0.14	0.01	92.61%	7.39%	3.12

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.44	1.20		0	27.81%	75.29%	
2	0	0.81	1.45		0	51.30%	91.81%	
3	0	1.26	1.47		0	78.98%	92.37%	
4	0	1.28	1.44		0	82.13%	92.61%	

### TS leaching test

<b>Group No.:</b>	S-D10	<b>Date:</b>	May-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.200	23	0.9	21.1%	16% Pyrrhotite				
2	0.200	76	0.9	21.1%					
3	0.200	228	0.9	21.1%					
4	0.200	760	0.9	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.15		0		13.7	
2	0.20		0.14		0		18.6	
3	0.20		0.11		0		25.3	
4	0.20		0.10		0		29.9	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.42	0.34	0.14	0.02	89.25%	10.75%	3.19
2	100	50	1.44	0.27	0.14	0.01	91.33%	8.67%	3.15
3	100	50	1.46	0.23	0.15	0.01	92.57%	7.43%	3.16
4	100	50	1.49	0.27	0.15	0.01	91.70%	8.30%	3.25

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.56	1.42		0	34.87%	89.25%	
2	0	0.81	1.44		0	51.56%	91.33%	
3	0	1.11	1.46		0	70.25%	92.57%	
4	0	1.15	1.49		0	70.58%	91.70%	

### TS leaching test

<b>Group No.:</b>	S-D11	<b>Date:</b>	May-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of galena						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	PbS		
1	0.20	76	0.9	21.1%	16% Pyrrhotite	0.00%		
2	0.20	76	0.9	21.1%		0.20%		
3	0.20	76	0.9	21.1%		0.40%		
4	0.20	76	0.9	21.1%		0.80%		
5	0.20	76	0.9	21.1%		1.60%		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		18.7	
2	0.20		0.14		0		18.6	
3	0.20		0.14		0		19	
4	0.20		0.14		0		18.6	
5	0.20		0.14		0		18.3	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.43	0.26	0.14	0.01	91.61%	8.39%	3.12
2	100	50	1.47	0.26	0.15	0.01	91.99%	8.01%	3.20
3	100	50	1.46	0.24	0.15	0.01	92.28%	7.72%	3.16
4	100	50	1.12	0.94	0.11	0.05	70.47%	29.53%	3.18
5	100	50	0.51	2.07	0.05	0.10	33.17%	66.83%	3.09

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.80	1.43		0	51.30%	91.61%	
2	0	1.08	1.47		0	67.61%	91.99%	
3	0	1.22	1.46		0	77.09%	92.28%	
4	0	0.70	1.12		0	43.81%	70.47%	
5	0	0.19	0.51		0	11.98%	33.17%	

### TS leaching test

<b>Group No.:</b>	S-H1	<b>Date:</b>	Nov-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of chalcopyrite						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	(%)			
1	0.20	76	0.9	21.1%	Chalcopyrite	0			
2	0.20	76	0.9	21.1%		2			
3	0.20	76	0.9	21.1%		4			
4	0.20	76	0.9	21.1%		8			
5	0.20	76	0.9	21.1%		12			

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		6.8	
2	0.20		0.16		0		13.3	
3	0.20		0.15		0		15.3	
4	0.20		0.14		0		16.6	
5	0.20		0.14		0		18.1	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.46	0.23	0.15	0.01	92.85%	7.15%	3.15
2	100	50	1.46	0.29	0.15	0.01	90.99%	9.01%	3.22
3	100	50	1.42	0.36	0.14	0.02	88.87%	11.13%	3.19
4	100	50	1.35	0.41	0.14	0.02	86.76%	13.24%	3.12
5	100	50	1.32	0.43	0.13	0.02	86.02%	13.98%	3.08

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	1.32	1.46		0	83.58%	92.85%	
2	0	1.29	1.46		0	80.19%	90.99%	
3	0	1.23	1.42		0	76.98%	88.87%	
4	0	1.13	1.35		0	72.12%	86.76%	
5	0	1.09	1.32		0	71.06%	86.02%	

### TS leaching test

<b>Group No.:</b>	S-H2	<b>Date:</b>	Dec-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ammonia						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Result: Thiosulfate

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	76	0.06	21.1%	8% Chalcopyrite				
2	0.20	76	0.3	21.1%					
3	0.20	76	0.9	21.1%					
4	0.20	76	3.0	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		5.9	
2	0.20		0.16		0		10.8	
3	0.20		0.15		0		16.2	
4	0.20		0.13		0		19.8	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.13	2.86	0.01	0.14	8.11%	91.89%	3.11
2	100	50	0.98	1.20	0.10	0.06	62.01%	37.99%	3.15
3	100	50	1.37	0.44	0.14	0.02	86.16%	13.84%	3.17
4	100	50	1.45	0.32	0.15	0.02	90.07%	9.93%	3.23

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.00	0.13		0	0.00%	8.11%	
2	0	0.62	0.98		0	39.12%	62.01%	
3	0	1.16	1.37		0	73.31%	86.16%	
4	0	1.28	1.45		0	78.96%	90.07%	

### TS leaching test

<b>Group No.:</b>	S-H3	<b>Date:</b>	Dec-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of phosphate						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Result: Thiosulfate

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Na <sub>3</sub> PO <sub>4</sub> (g/L)		
1	0.20	76	0.90	21.1%	8% Chalcopyrite	0		
2	0.20	76	0.90	21.1%		2		
3	0.20	76	0.90	21.1%		4		
4	0.20	76	0.90	21.1%		6		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		16.3	
2	0.20		0.15		0		15.0	
3	0.20		0.15		0		14.5	
4	0.20		0.15		0		13.9	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.40	0.42	0.14	0.02	86.98%	13.02%	3.21
2	100	50	1.39	0.37	0.14	0.02	88.12%	11.88%	3.15
3	100	50	1.38	0.41	0.14	0.02	87.09%	12.91%	3.17
4	100	50	1.36	0.40	0.14	0.02	87.21%	12.79%	3.13

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0	1.16	1.40		0	72.33%	86.98%	
2	0	1.17	1.39		0	73.97%	88.12%	
3	0	1.16	1.38		0	72.91%	87.09%	
4	0	1.11	1.36		0	71.08%	87.21%	

### TS leaching test

<b>Group No.:</b>	S-H4	<b>Date:</b>	Dec-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	0	0.90	21.1%	8% Chalcopyrite				
2	0.20	22.8	0.90	21.1%					
3	0.20	76	0.90	21.1%					
4	0.20	228	0.90	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0.20		0.16		0		12.5	
2	0.20		0.15		0		14.0	
3	0.20		0.14		0		16.7	
4	0.20		0.14		0		18.2	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.91	1.30	0.09	0.06	58.30%	41.70%	3.11
2	100	50	1.34	0.45	0.13	0.02	85.67%	14.33%	3.13
3	100	50	1.33	0.42	0.13	0.02	86.29%	13.71%	3.09
4	100	50	1.40	0.38	0.14	0.02	87.91%	12.09%	3.18

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0	0.37	0.91		0	23.83%	58.30%	
2	0	1.05	1.34		0	67.29%	85.67%	
3	0	1.11	1.33		0	71.82%	86.29%	
4	0	1.17	1.40		0	73.88%	87.91%	

### TS leaching test

<b>Group No.:</b>	S-H5	<b>Date:</b>	Dec-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ATS						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.07	76	0.90	21.1%	8% Chalcopyrite				
2	0.14	76	0.90	21.1%					
3	0.20	76	0.90	21.1%					
4	0.36	76	0.90	21.1%					
5	0.50	76	0.90	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.07		0.03		0		10.8	
2	0.14		0.10		0		12.1	
3	0.20		0.14		0		16.7	
4	0.36		0.29		0		22.1	
5	0.50		0.42		0		25.0	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.06	1.10	0.11	0.05	65.82%	34.18%	3.21
2	100	50	1.28	0.53	0.13	0.03	82.98%	17.02%	3.09
3	100	50	1.38	0.42	0.14	0.02	86.92%	13.08%	3.18
4	100	50	1.37	0.40	0.14	0.02	87.27%	12.73%	3.13
5	100	50	1.24	0.69	0.12	0.03	78.27%	21.73%	3.16

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.91	1.06		0	56.62%	65.82%	
2	0	1.09	1.28		0	70.36%	82.98%	
3	0	1.15	1.38		0	72.17%	86.92%	
4	0	0.98	1.37		0	62.55%	87.27%	
5	0	0.84	1.24		0	53.19%	78.27%	

### TS leaching test

<b>Group No.:</b>	S-H6	<b>Date:</b>	Dec-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of NaOH						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	NaOH			
1	0.20	76	0.90	21.1%	8% Chalcopyrite	0			
2	0.20	76	0.90	21.1%		1			
3	0.20	76	0.90	21.1%		2			
4	0.20	76	0.90	21.1%		4			
5	0.20	76	0.90	21.1%		8			

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		16.9	
2	0.20		0.14		0		16.9	
3	0.20		0.15		0		15.7	
4	0.20		0.16		0		13.3	
5	0.20		0.16		0		11.7	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.40	0.41	0.14	0.02	87.09%	12.91%	3.21
2	100	50	1.35	0.39	0.13	0.02	87.32%	12.68%	3.09
3	100	50	1.40	0.38	0.14	0.02	88.17%	11.83%	3.18
4	100	50	1.37	0.38	0.14	0.02	87.83%	12.17%	3.13
5	100	50	1.37	0.42	0.14	0.02	86.56%	13.44%	3.16

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	1.17	1.40		0	72.75%	87.09%	
2	0	1.16	1.35		0	75.22%	87.32%	
3	0	1.25	1.40		0	78.38%	88.17%	
4	0	1.21	1.37		0	77.16%	87.83%	
5	0	1.20	1.37		0	76.16%	86.56%	

### TS leaching test

<b>Group No.:</b>	S-H4	<b>Date:</b>	Dec-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	0	0.90	21.1%	8% Chalcopyrite				
2	0.20	22.8	0.90	21.1%					
3	0.20	76	0.90	21.1%					
4	0.20	228	0.90	21.1%					
5	0.20	760	0.90	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.19		0		1.9	
2	0.20		0.18		0		4.7	
3	0.20		0.18		0		6.3	
4	0.20		0.17		0		8.2	
5	0.20		0.17		0		9.0	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.58	2.01	0.06	0.10	36.62%	63.38%	3.17
2	100	50	0.90	1.43	0.09	0.07	55.69%	44.31%	3.22
3	100	50	0.97	1.18	0.10	0.06	62.11%	37.89%	3.12
4	100	50	1.27	0.63	0.13	0.03	80.17%	19.83%	3.16
5	100	50	1.32	0.46	0.13	0.02	85.22%	14.78%	3.10

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.05	0.58		0	3.02%	36.62%	
2	0	0.32	0.90		0	19.62%	55.69%	
3	0	0.49	0.97		0	31.13%	62.11%	
4	0	0.83	1.27		0	52.23%	80.17%	
5	0	0.99	1.32		0	63.71%	85.22%	

### TS leaching test

<b>Group No.:</b>	S-D7	<b>Date:</b>	Apr-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of pre-treatment in presence of pyrrhotite						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Pretreatment (3h, 21.1% oxygen)
1	0.20	76	0.9	0.7%	8% Chalcopyrite	No
2	0.20	76	0.9	0.7%		Water
3	0.20	76	0.9	0.7%		AH 0.9 M
4	0.20	76	0.9	0.7%		Cu 1.2 mM, AH 0.9 M
5	0.20	76	0.9	0.7%		Na <sub>3</sub> PO <sub>4</sub> 10.5 mM

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		6.3	
2	0.20		0.18		0		5.1	
3	0.20		0.18		0		5.8	
5	0.20		0.18		0		6.1	
0	0.20		0.19		0		3.9	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.97	1.14	0.10	0.06	63.02%	36.98%	3.09
2	100	50	1.05	1.01	0.10	0.05	67.52%	32.48%	3.11
3	100	50	1.32	0.58	0.13	0.03	81.97%	18.03%	3.22
5	100	50	1.31	0.53	0.13	0.03	83.23%	16.77%	3.15
0	100	50	1.39	0.39	0.14	0.02	87.71%	12.29%	3.18

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	0.51	0.97		0	32.86%	63.02%	
2	1	0.72	1.05		0	46.18%	67.52%	
3	2	0.89	1.32		0	55.53%	81.97%	
5	3	0.93	1.31		0	59.35%	83.23%	
0	4	1.13	1.39		0	70.79%	87.71%	

### TS leaching test

<b>Group No.:</b>	S-E1	<b>Date:</b>	Jun-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of arsenopyrite						

#### Synthetic ore and pulp properties

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Mineral %		
1	0.20	76	0.9	21.1%	Arsenopyrite	0%		
2	0.20	76	0.9	21.1%		1%		
3	0.20	76	0.9	21.1%		2%		
4	0.20	76	0.9	21.1%		4%		
5	0.20	76	0.9	21.1%		8%		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.6	
2	0.20		0.18		0		7.4	
3	0.20		0.17		0		7.5	
4	0.20		0.17		0		7.4	
5	0.20		0.18		0		7.4	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.47	0.22	0.15	0.01	93.15%	6.85%	3.15
2	100	50	1.38	0.44	0.14	0.02	86.23%	13.77%	3.20
3	100	50	1.18	0.76	0.12	0.04	75.67%	24.33%	3.12
4	100	50	1.04	1.04	0.10	0.05	66.52%	33.48%	3.12
5	100	50	0.90	1.29	0.09	0.06	58.26%	41.74%	3.09

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	1.30	1.47		0	82.61%	93.15%	
2	0	0.95	1.38		0	59.57%	86.23%	
3	0	0.46	1.18		0	29.68%	75.67%	
4	0	0.22	1.04		0	13.87%	66.52%	
5	0	0.11	0.90		0	6.99%	58.26%	

### TS leaching test

<b>Group No.:</b>	S-D2	<b>Date:</b>	Jul-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ATS						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.06	76	0.9	21.1%	4% Arsenopyrite			
2	0.10	76	0.9	21.1%				
3	0.20	76	0.9	21.1%				
4	0.30	76	0.9	21.1%				
5	0.40	76	0.9	21.1%				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.06		0.05		0		4.1	
2	0.10		0.08		0		5.6	
3	0.20		0.17		0		7.8	
4	0.30		0.26		0		12.9	
5	0.40		0.34		0		16.3	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.68	1.77	0.07	0.09	43.32%	56.68%	3.12
2	100	50	0.96	1.27	0.10	0.06	60.09%	39.91%	3.18
3	100	50	1.05	1.03	0.10	0.05	67.12%	32.88%	3.12
4	100	50	0.87	1.42	0.09	0.07	55.07%	44.93%	3.15
5	100	50	0.62	1.85	0.06	0.09	40.28%	59.72%	3.09

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	0.15	0.68		0	9.86%	43.32%	
2	0	0.24	0.96		0	15.06%	60.09%	
3	0	0.22	1.05		0	13.82%	67.12%	
4	0	0.21	0.87		0	13.65%	55.07%	
5	0	0.24	0.62		0	15.69%	40.28%	

### TS leaching test

<b>Group No.:</b>	S-E3	<b>Date:</b>	Jul-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu						

#### Synthetic ore and pulp properties

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.20	76	0.9	21.1%	4% Arsenopyrite			
2	0.20	228	0.9	21.1%				
3	0.20	760	0.9	21.1%				
4	0.20	2280	0.9	21.1%				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.6	
2	0.20		0.15		0		15.3	
3	0.20		0.13		0		20.1	
4	0.20		0.12		0		23.9	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.06	1.04	0.11	0.05	67.12%	32.88%	3.15
2	100	50	1.28	0.62	0.13	0.03	80.65%	19.35%	3.18
3	100	50	1.37	0.48	0.14	0.02	85.22%	14.78%	3.22
4	100	50	1.35	0.40	0.13	0.02	87.02%	12.98%	3.10

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.21	1.06		0	13.22%	67.12%	
2	0	0.96	1.28		0	60.69%	80.65%	
3	0	1.11	1.37		0	69.13%	85.22%	
4	0	1.09	1.35		0	70.38%	87.02%	

### TS leaching test

<b>Group No.:</b>	S-E4	<b>Date:</b>	Jul-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ammonia						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.20	76	0.3	21.1%	4% Arsenopyrite			
2	0.20	76	0.9	21.1%				
3	0.20	76	1.8	21.1%				
4	0.20	76	3.0	21.1%				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.6	
2	0.20		0.18		0		6.8	
3	0.20		0.18		0		6.0	
4	0.20		0.19		0		3.9	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.94	1.24	0.09	0.06	60.17%	39.83%	3.11
2	100	50	1.07	1.04	0.11	0.05	67.18%	32.82%	3.18
3	100	50	1.18	0.84	0.12	0.04	73.87%	26.13%	3.20
4	100	50	1.19	0.77	0.12	0.04	75.56%	24.44%	3.16

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.15	0.94		0	9.92%	60.17%	
2	0	0.23	1.07		0	14.27%	67.18%	
3	0	0.59	1.18		0	36.72%	73.87%	
4	0	0.83	1.19		0	52.22%	75.56%	

### TS leaching test

<b>Group No.:</b>	S-E5	<b>Date:</b>	Jul-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of NaOH						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	NaOH (g/L)		
1	0.20	76	0.9	21.1%	4% Arsenopyrite	0		
2	0.20	76	0.9	21.1%		4		
3	0.20	76	0.9	21.1%		8		
4	0.20	76	0.9	21.1%		16		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.6	
2	0.20		0.18		0		7.3	
3	0.20		0.18		0		6.9	
4	0.20		0.18		0		6.6	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.04	1.02	0.10	0.05	67.18%	32.82%	3.11
2	100	50	1.20	0.79	0.12	0.04	75.27%	24.73%	3.18
3	100	50	1.22	0.68	0.12	0.03	78.35%	21.65%	3.12
4	100	50	1.28	0.63	0.13	0.03	80.21%	19.79%	3.20

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0	0.22	1.04		0	14.27%	67.18%	
2	0	0.28	1.20		0	17.67%	75.27%	
3	0	0.24	1.22		0	15.24%	78.35%	
4	0	0.30	1.28		0	18.65%	80.21%	

### TS leaching test

<b>Group No.:</b>	S-E6	<b>Date:</b>	Aug-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu at 0.7% oxygen						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.20	76	0.9	0.7%	4% Arsenopyrite			
2	0.20	152	0.9	0.7%				
3	0.20	228	0.9	0.7%				
4	0.20	760	0.9	0.7%				

#### Result: Thiosulfate (as ATS (M))

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.19		0		3.6	
2	0.20		0.18		0		4.6	
3	0.20		0.18		0		4.8	
4	0.20		0.18		0		5.5	

#### Result: Metal balance (as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.26	2.57	0.03	0.13	16.98%	83.02%	3.09
2	100	50	0.93	1.32	0.09	0.07	58.36%	41.64%	3.18
3	100	50	1.25	0.64	0.13	0.03	79.62%	20.38%	3.14
4	100	50	1.27	0.57	0.13	0.03	81.67%	18.33%	3.10

#### Result: Leaching kinetics (as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.05	0.26		0	3.25%	16.98%	
2	0	0.24	0.93		0	15.28%	58.36%	
3	0	0.44	1.25		0	27.82%	79.62%	
4	0	0.52	1.27		0	33.23%	81.67%	

### TS leaching test

<b>Group No.:</b>	S-E7	<b>Date:</b>	Aug-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of phosphate						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Phosphte		
1	0.20	76	0.9	21.1%	4% Arsenopyrite	0		
2	0.20	76	0.9	21.1%		1		
3	0.20	76	0.9	21.1%		2		
4	0.20	76	0.9	21.1%		4		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.9	
2	0.20		0.17		0		7.7	
3	0.20		0.17		0		7.8	
4	0.20		0.17		0		7.7	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.06	0.99	0.11	0.05	68.12%	31.88%	3.10
2	100	50	1.22	0.70	0.12	0.03	77.72%	22.28%	3.13
3	100	50	1.29	0.64	0.13	0.03	80.22%	19.78%	3.22
4	100	50	1.33	0.54	0.13	0.03	83.02%	16.98%	3.20

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0	0.20	1.06		0	13.00%	68.12%	
2	0	0.61	1.22		0	39.11%	77.72%	
3	0	0.75	1.29		0	46.89%	80.22%	
4	0	0.81	1.33		0	50.50%	83.02%	

### TS leaching test

<b>Group No.:</b>	S-E7	<b>Date:</b>	Aug-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of phosphate						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Ca(OH) <sub>2</sub>		
1	0.20	76	0.9	21.1%	4% Arsenopyrite	0		
2	0.20	76	0.9	21.1%		2		
3	0.20	76	0.9	21.1%		4		
4	0.20	76	0.9	21.1%		8		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.6	
2	0.20		0.17		0		8.1	
3	0.20		0.17		0		8.5	
4	0.20		0.17		0		8.8	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.07	1.04	0.11	0.05	67.23%	32.77%	3.17
2	100	50	1.22	0.70	0.12	0.03	77.72%	22.28%	3.13
3	100	50	1.30	0.56	0.13	0.03	82.26%	17.74%	3.16
4	100	50	1.42	0.38	0.14	0.02	88.33%	11.67%	3.22

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.23	1.07		0	14.31%	67.23%	
2	0	0.52	1.22		0	33.26%	77.72%	
3	0	0.82	1.30		0	52.12%	82.26%	
4	0	1.05	1.42		0	65.14%	88.33%	

### TS leaching test

<b>Group No.:</b>	S-G2	<b>Date:</b>	Oct-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	228.0	0.9	21.1%	4% Chalcocite				
2	0.20	76.0	0.9	21.1%					
3	0.20	22.8	0.9	21.1%					
4	0.20	0.0	0.9	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		17.3	
2	0.20		0.15		0		15.1	
3	0.20		0.15		0		13.6	
4	0.20		0.16		0		12.2	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.29	0.55	0.13	0.03	82.50%	17.50%	3.12
2	100	50	1.29	0.61	0.13	0.03	80.83%	19.17%	3.18
3	100	50	1.26	0.57	0.13	0.03	81.67%	18.33%	3.09
4	100	50	1.30	0.61	0.13	0.03	81.03%	18.97%	3.20

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.33	1.29		0	20.83%	82.50%	
2	1	0.24	1.29		0	15.00%	80.83%	
3	2	0.27	1.26		0	17.33%	81.67%	
4	3	0.29	1.30		0	18.33%	81.03%	

### TS leaching test

<b>Group No.:</b>	S-G3	<b>Date:</b>	Oct-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ATS						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.07	76.0	0.9	21.1%	4% Chalcocite				
2	0.20	76.0	0.9	21.1%					
3	0.27	76.0	0.9	21.1%					
4	0.33	76.0	0.9	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0.07		0.04		0		7.8	
2	0.20		0.15		0		15.6	
3	0.27		0.21		0		16.7	
4	0.33		0.27		0		17.8	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.36	2.39	0.04	0.12	23.33%	76.67%	3.12
2	100	50	1.31	0.56	0.13	0.03	82.50%	17.50%	3.18
3	100	50	1.25	0.59	0.12	0.03	80.83%	19.17%	3.09
4	100	50	1.28	0.61	0.13	0.03	80.83%	19.17%	3.16

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0	0.09	0.36		0	5.83%	23.33%	
2	1	0.33	1.31		0	20.83%	82.50%	
3	2	0.36	1.25		0	23.33%	80.83%	
4	3	0.41	1.28		0	25.83%	80.83%	

### TS leaching test

<b>Group No.:</b>	S-G4	<b>Date:</b>	Nov-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect ammonia						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	76.0	0.3	21.1%	4% Chalcocite				
2	0.20	76.0	0.9	21.1%					
3	0.20	76.0	1.5	21.1%					
4	0.20	76.0	3.0	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.15		0		15.1	
2	0.20		0.14		0		17.2	
3	0.20		0.14		0		19.1	
4	0.20		0.15		0		15.3	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.24	0.61	0.12	0.03	80.22%	19.78%	3.10
2	100	50	1.32	0.55	0.13	0.03	82.80%	17.20%	3.18
3	100	50	1.33	0.50	0.13	0.03	84.16%	15.84%	3.16
4	100	50	1.36	0.51	0.14	0.03	84.22%	15.78%	3.22

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.32	1.24		0	20.83%	80.22%	
2	1	0.32	1.32		0	20.02%	82.80%	
3	2	0.33	1.33		0	20.89%	84.16%	
4	3	0.33	1.36		0	20.68%	84.22%	

### TS leaching test

<b>Group No.:</b>	S-G5	<b>Date:</b>	Nov-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of oxygen %						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.20	76.0	0.9	21.1%	4% Chalcocite			
2	0.20	76.0	0.9	6.4%				
3	0.20	76.0	0.9	4.2%				
4	0.20	76.0	0.9	0.7%				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		16.9	
2	0.20		0.16		0		12.8	
3	0.20		0.17		0		8.5	
4	0.20		0.19		0		1.9	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.34	0.45	0.13	0.02	85.65%	14.35%	3.13
2	100	50	0.79	1.58	0.08	0.08	50.02%	49.98%	3.16
3	100	50	0.68	1.73	0.07	0.09	43.98%	56.02%	3.09
4	100	50	0.55	2.00	0.06	0.10	35.67%	64.33%	3.11

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.40	1.34		0	25.33%	85.65%	
2	0	0.34	0.79		0	21.37%	50.02%	
3	0	0.30	0.68		0	19.62%	43.98%	
4	0	0.29	0.55		0	18.39%	35.67%	

### TS leaching test

<b>Group No.:</b>	S-G6	<b>Date:</b>	Nov-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of phosphate						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Result: Thiosulfate

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Na <sub>3</sub> PO <sub>4</sub> (g/L)		
1	0.20	76.0	0.10	21.1%	8% Chalcocite	0		
2	0.20	76.0	0.10	21.1%		1		
3	0.20	76.0	0.10	21.1%		2		
4	0.20	76.0	0.10	21.1%		4		
5	0.20	76.0	0.10	21.1%		8		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.15		0		13.8	
2	0.20		0.16		0		12.6	
3	0.20		0.16		0		11.9	
4	0.20		0.16		0		11.2	
5	0.20		0.17		0		10.3	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.98	1.16	0.10	0.06	62.83%	37.17%	3.11
2	100	50	1.11	0.98	0.11	0.05	69.34%	30.66%	3.20
3	100	50	1.24	0.70	0.12	0.03	77.99%	22.01%	3.18
4	100	50	1.29	0.54	0.13	0.03	82.56%	17.44%	3.12
5	100	50	1.31	0.45	0.13	0.02	85.23%	14.77%	3.08

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.00	0.98		0		62.83%	
2	0	0.00	1.11		0		69.34%	
3	1	0.00	1.24		0		77.99%	
4	2	0.00	1.29		0		82.56%	
5	3	0.00	1.31		0		85.23%	

### TS leaching test

<b>Group No.:</b>	S-G7	<b>Date:</b>	Dec-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of pretreatment						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Result: Thiosulfate

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Pre-leach 3 hour, 21.1% oxygen
1	0.20	76.0	0.90	21.1%	4% Chalcocite	No
2	0.20	76.0	0.90	21.1%		Water
3	0.20	76.0	0.90	21.1%		0.9 M NH <sub>3</sub>
4	0.20	76.0	0.10	21.1%		0.1 M NH <sub>3</sub> and 4g/L Na <sub>3</sub> PO <sub>4</sub>

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		16.3	
2	0.20		0.14		0		19.1	
3	0.20		0.13		0		21.6	
4	0.20		0.14		0		17.1	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.31	0.59	0.13	0.03	81.66%	18.34%	3.21
2	100	50	1.27	0.56	0.13	0.03	81.80%	18.20%	3.10
3	100	50	1.35	0.45	0.14	0.02	85.61%	14.39%	3.16
4	100	50	1.42	0.32	0.14	0.02	90.02%	9.98%	3.16

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.00	1.31		0		81.66%	
2	0	0.00	1.27		0		81.80%	
3	0	0.00	1.35		0		85.61%	
4	0	0.00	1.42		0		90.02%	

### TS leaching test

<b>Group No.:</b>	S-J1	<b>Date:</b>	May-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of lead nitrate						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	(mg/L)			
1	0.20	76	0.9	21.1%	Pb(NO <sub>3</sub> ) <sub>2</sub>	0			
2	0.20	76	0.9	21.1%		200			
3	0.20	76	0.9	21.1%		400			
4	0.20	76	0.9	21.1%		1200			
5	0.20	76	0.9	21.1%		4000			

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		8.3	
2	0.20		0.16		0		10.9	
3	0.20		0.15		0		14.7	
4	0.20		0.15		0		15.3	
5	0.20		0.15		0		14.7	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.45	0.25	0.15	0.01	92.15%	7.85%	3.15
2	100	50	1.40	0.35	0.14	0.02	88.83%	11.17%	3.16
3	100	50	0.97	1.26	0.10	0.06	60.65%	39.35%	3.19
4	100	50	0.31	2.49	0.03	0.12	20.13%	79.87%	3.12
5	100	50	0.35	2.51	0.03	0.13	21.67%	78.33%	3.21

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	1.32	1.45		0	83.58%	92.15%	
2	1	0.99	1.40		0	62.46%	88.83%	
3	2	0.73	0.97		0	45.66%	60.65%	
4	3	0.30	0.31		0	19.22%	20.13%	
5	4	0.30	0.35		0	18.86%	21.67%	

### TS leaching test

<b>Group No.:</b>	S-J2	<b>Date:</b>	May-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of galena						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	(%)			
1	0.20	76	0.9	21.1%	Galena	0.00%			
2	0.20	76	0.9	21.1%		0.08%			
3	0.20	76	0.9	21.1%		0.24%			
4	0.20	76	0.9	21.1%		0.80%			
5	0.20	76	0.9	21.1%		2.40%			
6	0.20	76	0.9	21.1%		4.80%			

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		7.3	
2	0.20		0.19		0		3.2	
3	0.20		0.19		0		3.7	
4	0.20		0.19		0		3.9	
5	0.20		0.18		0		5.0	
6	0.20		0.18		0		5.7	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L.g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.48	0.24	0.15	0.01	92.53%	7.47%	3.20
2	100	50	1.37	0.42	0.14	0.02	86.67%	13.33%	3.15
3	100	50	1.33	0.51	0.13	0.03	84.00%	16.00%	3.16
4	100	50	0.63	1.96	0.06	0.10	39.20%	60.80%	3.22
5	100	50	0.31	2.46	0.03	0.12	20.00%	80.00%	3.08
6	100	50	0.25	2.60	0.02	0.13	16.00%	84.00%	3.10

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	1.34	1.48		0	83.50%	92.53%	
2	1	1.23	1.37		0	78.33%	86.67%	
3	2	0.92	1.33		0	58.40%	84.00%	
4	3	0.43	0.63		0	26.40%	39.20%	
5	4	0.10	0.31		0	6.40%	20.00%	
6	5	0.02	0.25		0	1.60%	16.00%	

### TS leaching test

<b>Group No.:</b>	S-J3	<b>Date:</b>	May-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of litharge						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	(%)			
1	0.20	76	0.9	21.1%	Litharge	0.00%			
2	0.20	76	0.9	21.1%		0.03%			
3	0.20	76	0.9	21.1%		0.24%			
4	0.20	76	0.9	21.1%		0.80%			
5	0.20	76	0.9	21.1%		2.40%			
6	0.20	76	0.9	21.1%		4.80%			

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		7.4	
2	0.20		0.16		0		12.1	
3	0.20		0.15		0		14.7	
4	0.20		0.15		0		15.3	
5	0.20		0.15		0		16.0	
6	0.20		0.14		0		17.2	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.44	0.23	0.14	0.01	92.66%	7.34%	3.11
2	100	50	1.01	1.13	0.10	0.06	64.00%	36.00%	3.15
3	100	50	0.12	2.98	0.01	0.15	7.20%	92.80%	3.21
4	100	50	0.25	2.63	0.03	0.13	16.00%	84.00%	3.13
5	100	50	0.17	2.85	0.02	0.14	10.40%	89.60%	3.18
6	100	50	0.20	2.79	0.02	0.14	12.80%	87.20%	3.20

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	1.30	1.44		0	83.83%	92.66%	
2	0	0.83	1.01		0	52.80%	64.00%	
3	0	0.06	0.12		0	4.00%	7.20%	
4	0	0.09	0.25		0	5.60%	16.00%	
5	0	0.08	0.17		0	4.80%	10.40%	
6	0	0.00	0.20		0	0.00%	12.80%	

### TS leaching test

<b>Group No.:</b>	S-G1	<b>Date:</b>	Oct-02	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of chalcocite						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	(%)			
1	0.20	76	0.9	21.1%	Chalcocite	0			
2	0.20	76	0.9	21.1%		2			
3	0.20	76	0.9	21.1%		4			
4	0.20	76	0.9	21.1%		8			
5	0.20	76	0.9	21.1%		12			

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		7.0	
2	0.20		0.13		0		19.8	
3	0.20		0.14		0		17.6	
4	0.20		0.16		0		13.1	
5	0.20		0.17		0		9.9	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.46	0.24	0.15	0.01	92.50%	7.50%	3.15
2	100	50	1.32	0.46	0.13	0.02	85.22%	14.78%	3.10
3	100	50	1.32	0.56	0.13	0.03	82.50%	17.50%	3.19
4	100	50	0.87	1.48	0.09	0.07	54.17%	45.83%	3.22
5	100	50	0.41	2.26	0.04	0.11	26.67%	73.33%	3.08

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	1.31	1.46		0	83.33%	92.50%	
2	1	0.44	1.32		0	28.33%	85.22%	
3	2	0.33	1.32		0	20.83%	82.50%	
4	3	0.21	0.87		0	13.33%	54.17%	
5	4	0.12	0.41		0	7.50%	26.67%	

### TS leaching test

<b>Group No.:</b>	S-J4	<b>Date:</b>	May-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of phosphate						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Phosphate (g/L)			
1	0.20	76	0.9	21.1%	0.0			
2	0.20	76	0.9	21.1%	1.2			
3	0.20	76	0.9	21.1%	4.0			
4	0.20	76	0.9	21.1%	12.0			

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.7	
2	0.20		0.17		0		8.9	
3	0.20		0.17		0		7.7	
4	0.20		0.18		0		7.0	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.43	0.23	0.14	0.01	92.69%	7.31%	3.09
2	100	50	1.41	0.39	0.14	0.02	87.82%	12.18%	3.21
3	100	50	1.27	0.57	0.13	0.03	81.66%	18.34%	3.11
4	100	50	1.28	0.57	0.13	0.03	81.66%	18.34%	3.13

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	1.29	1.43		0	83.60%	92.69%	
2	0	1.29	1.41		0	80.22%	87.82%	
3	0	1.07	1.27		0	68.58%	81.66%	
4	0	1.02	1.28		0	65.20%	81.66%	

### TS leaching test

<b>Group No.:</b>	S-J5	<b>Date:</b>	Jun-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of phosphate in presence of galena						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Na <sub>3</sub> PO <sub>4</sub> (g/L)		
1	0.20	76	0.9	21.1%	2.4% Galena	0.0		
2	0.20	76	0.9	21.1%		1.2		
3	0.20	76	0.9	21.1%		2.4		
4	0.20	76	0.9	21.1%		4.0		
5	0.20	76	0.9	21.1%		8.0		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		5.0	
2	0.20		0.18		0		5.3	
3	0.20		0.18		0		5.3	
4	0.20		0.18		0		6.6	
5	0.20		0.18		0		7.3	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.33	2.45	0.03	0.12	21.12%	78.88%	3.10
2	100	50	0.88	1.39	0.09	0.07	56.00%	44.00%	3.15
3	100	50	1.12	0.91	0.11	0.05	71.20%	28.80%	3.16
4	100	50	1.28	0.67	0.13	0.03	79.20%	20.80%	3.22
5	100	50	1.27	0.54	0.13	0.03	82.40%	17.60%	3.08

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.10	0.33		0	6.63%	21.12%	
2	1	0.63	0.88		0	40.00%	56.00%	
3	2	0.80	1.12		0	50.40%	71.20%	
4	3	0.86	1.28		0	53.60%	79.20%	
5	4	0.85	1.27		0	55.20%	82.40%	

### TS leaching test

<b>Group No.:</b>	S-J6	<b>Date:</b>	Jun-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of phosphate in presence of litharge						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Na <sub>3</sub> PO <sub>4</sub> (g/L)		
1	0.20	76	0.9	21.1%	2.4% Litharge	0.0		
2	0.20	76	0.9	21.1%		1.2		
3	0.20	76	0.9	21.1%		2.4		
4	0.20	76	0.9	21.1%		4.0		
5	0.20	76	0.9	21.1%		8.0		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.15		0		16.0	
2	0.20		0.15		0		13.6	
3	0.20		0.17		0		8.9	
4	0.20		0.17		0		8.3	
5	0.20		0.17		0		8.3	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.17	2.84	0.02	0.14	10.58%	89.42%	3.18
2	100	50	1.29	0.58	0.13	0.03	81.60%	18.40%	3.15
3	100	50	1.36	0.49	0.14	0.02	84.80%	15.20%	3.20
4	100	50	1.25	0.62	0.12	0.03	80.00%	20.00%	3.12
5	100	50	1.02	1.07	0.10	0.05	65.60%	34.40%	3.10

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.08	0.17		0	5.03%	10.58%	
2	1	0.71	1.29		0	44.80%	81.60%	
3	2	1.02	1.36		0	64.00%	84.80%	
4	3	1.06	1.25		0	68.00%	80.00%	
5	4	0.91	1.02		0	58.40%	65.60%	

### TS leaching test

<b>Group No.:</b>	S-J7	<b>Date:</b>	Jun-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of lead carbonate						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	(%)			
1	0.20	76	0.9	21.1%	PbCO <sub>3</sub>	0.00%			
2	0.20	76	0.9	21.1%		0.08%			
3	0.20	76	0.9	21.1%		0.24%			
4	0.20	76	0.9	21.1%		0.80%			
5	0.20	76	0.9	21.1%		2.40%			
6	0.20	76	0.9	21.1%		4.80%			

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.5	
2	0.20		0.17		0		7.8	
3	0.20		0.17		0		8.0	
4	0.20		0.17		0		8.4	
5	0.20		0.17		0		8.2	
6	0.20		0.17		0		8.4	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.48	0.25	0.15	0.01	92.33%	7.67%	3.20
2	100	50	1.45	0.25	0.15	0.01	92.18%	7.82%	3.15
3	100	50	1.49	0.22	0.15	0.01	93.13%	6.87%	3.19
4	100	50	1.44	0.23	0.14	0.01	92.56%	7.44%	3.12
5	100	50	1.46	0.26	0.15	0.01	91.85%	8.15%	3.18
6	100	50	1.44	0.24	0.14	0.01	92.15%	7.85%	3.12

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	1.33	1.48		0	83.20%	92.33%	
2	1	1.31	1.45		0	83.01%	92.18%	
3	2	1.32	1.49		0	82.87%	93.13%	
4	3	1.29	1.44		0	82.51%	92.56%	
5	4	1.31	1.46		0	82.51%	91.85%	
6	5	1.29	1.44		0	82.83%	92.15%	

### TS leaching test

<b>Group No.:</b>	S-J8	<b>Date:</b>	Jun-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ATS						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	76	0.9	21.1%	2.4% Galena				
2	0.40	76	0.9	21.1%					
3	0.80	76	0.9	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		4.9	
2	0.40		0.37		0		8.9	
3	0.80		0.74		0		19.1	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.34	2.44	0.03	0.12	21.89%	78.11%	3.12
2	100	50	1.20	0.82	0.12	0.04	74.40%	25.60%	3.22
3	100	50	1.41	0.37	0.14	0.02	88.40%	11.60%	3.20

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.10	0.34		0	6.49%	21.89%	
2	1	0.94	1.20		0	58.40%	74.40%	
3	2	0.90	1.41		0	56.00%	88.40%	

### TS leaching test

<b>Group No.:</b>	S-J9	<b>Date:</b>	Jul-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	76	0.9	21.1%	2.4% Galena				
2	0.20	228	0.9	21.1%					
3	0.20	760	0.9	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		5.2	
2	0.20		0.17		0		9.6	
3	0.20		0.11		0		26.8	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.35	2.48	0.03	0.12	21.89%	78.11%	3.17
2	100	50	0.30	2.60	0.03	0.13	18.69%	81.31%	3.20
3	100	50	0.33	2.50	0.03	0.13	20.80%	79.20%	3.16

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.10	0.35		0	6.49%	21.89%	
2	1	0.08	0.30		0	5.12%	18.69%	
3	2	0.14	0.33		0	8.82%	20.80%	

### TS leaching test

<b>Group No.:</b>	S-J10	<b>Date:</b>	Jul-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ammonia						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral				
1	0.20	76	0.9	21.1%	2.4% Galena				
2	0.20	76	1.8	21.1%					
3	0.20	76	3.0	21.1%					

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		5.1	
2	0.20		0.19		0		1.9	
3	0.20		0.20		0		1.3	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.34	2.46	0.03	0.12	21.89%	78.11%	3.15
2	100	50	0.30	2.55	0.03	0.13	19.20%	80.80%	3.15
3	100	50	0.35	2.41	0.03	0.12	22.40%	77.60%	3.11

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.10	0.34		0	6.49%	21.89%	
2	1	0.23	0.30		0	14.40%	19.20%	
3	2	0.27	0.35		0	17.60%	22.40%	

### TS leaching test

<b>Group No.:</b>	S-J11	<b>Date:</b>	Jul-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of NaOH						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	NaOH (g/L)		
1	0.20	76	0.9	21.1%	2.4% Galena	0.0		
2	0.20	76	0.9	21.1%		4.0		
3	0.20	76	0.9	21.1%		12.0		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		5.1	
2	0.20		0.19		0		3.8	
3	0.20		0.19		0		1.9	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.35	2.48	0.03	0.12	21.89%	78.11%	3.17
2	100	50	0.24	2.67	0.02	0.13	15.20%	84.80%	3.15
3	100	50	0.00	3.08	0.00	0.15	0.00%	100.00%	3.08

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.10	0.35		0	6.49%	21.89%	
2	1	0.14	0.24		0	8.80%	15.20%	
3	2	0.00	0.00		0	0.00%	0.00%	

### TS leaching test

<b>Group No.:</b>	S-J13	<b>Date:</b>	Aug-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of carbonate						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Na <sub>2</sub> CO <sub>3</sub> (g/L)		
1	0.20	76	0.9	21.1%	2.4% Galena	0.0		
2	0.20	76	0.9	21.1%		0.4		
3	0.20	76	0.9	21.1%		1.2		
4	0.20	76	0.9	21.1%		2.4		
5	0.20	76	0.9	21.1%		4.0		
6	0.20	76	0.9	21.1%		8.0		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	0.5	3	-	0	0.5	3	-
1	0.20		0.18		0		5.1	
2	0.20		0.18		0		5.1	
3	0.20		0.18		0		5.1	
4	0.20		0.18		0		5.1	
5	0.20		0.18		0		5.7	
6	0.20		0.18		0		6.1	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.32	2.44	0.03	0.12	20.90%	79.10%	3.09
2	100	50	0.97	1.17	0.10	0.06	62.40%	37.60%	3.12
3	100	50	1.33	0.54	0.13	0.03	83.20%	16.80%	3.20
4	100	50	1.29	0.64	0.13	0.03	80.00%	20.00%	3.22
5	100	50	1.26	0.66	0.13	0.03	79.20%	20.80%	3.18
6	100	50	1.27	0.64	0.13	0.03	80.00%	20.00%	3.18

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	0.5	3	-	0	0.5	3	-
1	0	0.10	0.32		0	6.29%	20.90%	
2	0	0.66	0.97		0	42.40%	62.40%	
3	0	0.88	1.33		0	55.20%	83.20%	
4	0	0.91	1.29		0	56.80%	80.00%	
5	0	0.97	1.26		0	60.80%	79.20%	
6	0	0.84	1.27		0	52.80%	80.00%	

**TS leaching test**

<b>Group No.:</b>	S-Z1	<b>Date:</b>	Jan-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of oxygen %						

**Ore properties and pulp conditions**

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

**Leaching conditions**

Test #	ATS (M)	Cuen2 (mM)	AH (M)	O <sub>2</sub> %	Mineral	STS (M)	Total EDA	(mM)
1	0.00	1.2	0.0	30.0%		0.1	3.6	
2	0.00	1.2	0.0	21.1%		0.1	3.6	
3	0.00	1.2	0.0	6.39%		0.1	3.6	
4	0.00	1.2	0.0	4.18%		0.1	3.6	
5	0.00	1.2	0.0	0.74%		0.1	3.6	

**Result: Thiosulfate**

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	3	9	-	0	3	9	-
1	0.10		0.08		0		5.7	
2	0.10		0.09		0		2.6	
3	0.10		0.09		0		2.3	
4	0.10		0.09		0		2.0	
5	0.10		0.09		0		1.5	

**Result: Metal balance**

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.48	0.17	0.15	0.01	94.56%	5.44%	3.13
2	100	50	1.49	0.18	0.15	0.01	94.22%	5.78%	3.16
3	100	50	1.30	0.60	0.13	0.03	81.23%	18.77%	3.21
4	100	50	1.01	1.16	0.10	0.06	63.58%	36.42%	3.19
5	100	50	0.40	2.31	0.04	0.12	25.63%	74.37%	3.10

**Result: Leaching kinetics**

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	3	9	-	0	3	9	-
1	0		1.48		0		94.56%	
2	0		1.49		0		94.22%	
3	0		1.30		0		81.23%	
4	1		1.01		0		63.58%	
5	2		0.40		0		25.63%	

**TS leaching test**

<b>Group No.:</b>	S-Z2	<b>Date:</b>	Jan-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of pH						

**Ore properties and pulp conditions**

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

**Leaching conditions**

Test #	ATS (M)	Cuen2 (mM)	AH (M)	O <sub>2</sub> %	Initial pH	STS (M)	Total EDA (mM)	
1	0.00	1.2	0.0	21.1%	6	0.1	3.6	
2	0.00	1.2	0.0	21.1%	7.5	0.1	3.6	
3	0.00	1.2	0.0	21.1%	8.5	0.1	3.6	
4	0.00	1.2	0.0	21.1%	9.5	0.1	3.6	
5	0.00	1.2	0.0	21.1%	10.5	0.1	3.6	
6	0.00	1.2	0.0	21.1%	11.5	0.1	3.6	

**Result: Thiosulfate**

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	3	9	-	0	3	9	-
1	0.10		0.09		0		2.9	
2	0.10		0.09		1		2.7	
3	0.10		0.09		2		2.7	
4	0.10		0.09		3		2.6	
5	0.10		0.09		4		2.4	
6	0.10		0.09		5		2.3	

**Result: Metal balance**

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	(g/t)
1	100	50	1.47	0.16	0.15	0.01	94.70%	5.30%	3.11
2	100	50	1.52	0.15	0.15	0.01	95.30%	4.70%	3.20
3	100	50	1.50	0.16	0.15	0.01	95.00%	5.00%	3.15
4	100	50	1.46	0.18	0.15	0.01	94.30%	5.70%	3.09
5	100	50	1.47	0.26	0.15	0.01	91.90%	8.10%	3.19
6	100	50	1.44	0.31	0.14	0.02	90.40%	9.60%	3.18

**Result: Leaching kinetics**

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	3	9	-	0	3	9	-
1	0		1.47		0		94.70%	
2	0		1.52		0		95.30%	
3	0		1.50		0		95.00%	
4	0		1.46		0		94.30%	
5	0		1.47		0		91.90%	
6	0		1.44		0		90.40%	

### TS leaching test

<b>Group No.:</b>	S-Z3	<b>Date:</b>	Jan-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ammonia						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cuen2 (mM)	AH (M)	O <sub>2</sub> %	STS (M)	Total EDA (mM)	Cu (mM)
1	0.20		0.9	21.1%			1.2
2	0.00	1.2	0.0	21.1%	0.1	3.6	
3	0.00	1.2	0.0	21.1%	0.1	3.6	
4	0.10	1.2	0.0	21.1%		3.6	
5		1.2	0.06	21.1%	0.1	3.6	

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	3	6	-	0	3	6	-
1	0.20	0.17			0	7.8		
2	0.10	0.09			0	2.3		
3	0.10		0.09		0		2.5	
4	0.10		0.09		0		2.6	
5	0.10	0.09			0	3.5		

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.48	0.23	0.15	0.01	92.81%	7.19%	3.18
2	100	50	1.38	0.39	0.14	0.02	87.72%	12.28%	3.15
3	100	50	1.49	0.22	0.15	0.01	93.28%	6.72%	3.20
4	100	50	1.50	0.18	0.15	0.01	94.33%	5.67%	3.19
5	100	50	1.47	0.18	0.15	0.01	94.12%	5.88%	3.12

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	3	6	-	0	3	6	-
1	0	1.48			0	92.81%		
2	0	1.38			0	87.72%		
3	0		1.49		0		93.28%	
4	0		1.50		0		94.33%	
5	0	1.47			0	94.12%		

### TS leaching test

<b>Group No.:</b>	S-Z4	<b>Date:</b>	Feb-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of PR and PRTT						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cuen2 (mM)	AH (M)	O <sub>2</sub> %	Mineral	STS (M)	Total EDA	Cu (mM)
1	0.20		0.9	21.1%	8% Pyrite			1.2
2	0.20		0.9	21.1%	8% Pyrrhotite			1.2
3		20.0			8% Pyrite	0.1	60.0	
4		20.0			8% Pyrrhotite	0.1	60.0	

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	3	24	-	0	3	24	-
1	0.20	0.14			0	16.7		
2	0.20	0.16			0	12.1		
3	0.10		0.09		0		2.5	
4	0.10		0.09		0		2.6	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.36	0.41	0.14	0.02	86.89%	13.11%	3.12
2	100	50	1.46	0.27	0.15	0.01	91.51%	8.49%	3.19
3	100	50	1.45	0.25	0.14	0.01	91.94%	8.06%	3.15
4	100	50	1.50	0.20	0.15	0.01	93.72%	6.28%	3.20

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	3	24	-	0	3	24	-
1	0	1.36			0	86.89%		
2	0	1.46			0	91.51%		
3	0		1.45		0		91.94%	
4	0		1.50		0		93.72%	

**TS leaching test**

<b>Group No.:</b>	S-X1	<b>Date:</b>	Oct-05	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	TS-EDA leaching of a POX sample						

**Ore properties and pulp conditions**

<b>POX ore #1 (g)</b>	25	<b>Solution (mL)</b>	75
<b>Mineral (g)</b>		<b>Au grade (g/t)</b>	3.30
<b>Mixture (g)</b>	25	<b>Pulp Density (%)</b>	25.0%

**Leaching conditions**

Test #	ATS (g/L)	Cuen2 (mM)	AH (M)	O <sub>2</sub> %	T (°C)	STS (g/L)	Total EDA (mM)	Cu (mM)	pH (by NaOH)
1	8.00		0.135	21.1%	50			1	
2	8.00	1.6		21.1%	23		3.2		6.76
3	8.00	1.6		21.1%	50		3.2		6.76
4		1.6		21.1%	23	13.3	3.2		6.76
5		1.6		21.1%	50	13.3	3.2		6.76

**Result: Thiosulfate**

Time (h)	Concentration (g/L)				Consumption (kg/t of ore)			
Test #	0	8	16	-	0	8	16	-
1	8.00	2.73			0	15.8		
2	8.00		7.20		0		2.4	
3	8.00		6.73		0		3.8	
4	13.3		12.1		0		3.8	
5	13.3		12.0		0		4.0	

**Result: Metal balance**

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	75	25	1.23	0.89	0.09	0.02	80.53%	19.47%	3.35
2	75	25	1.09	1.07	0.08	0.03	75.26%	24.74%	3.25
3	75	25	1.18	0.98	0.09	0.02	78.30%	21.70%	3.33
4	75	25	0.96	1.25	0.07	0.03	69.72%	30.28%	3.16
5	75	25	1.21	0.88	0.09	0.02	80.55%	19.45%	3.3

**Result: Leaching kinetics**

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	8	16	-	0	8	16	-
1	0	1.23			0	73.39%		
2	0		1.09		0		66.97%	
3	0		1.18		0		70.64%	
4			0.96				60.55%	
5	0		1.21		0		73.41%	

**TS leaching test**

<b>Group No.:</b>	S-X2	<b>Date:</b>	Nov-05	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	TS-EDA leaching of a POX sample						

**Ore properties and pulp conditions**

<b>POX ore #2 (g)</b>	50	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>		<b>Au grade (g/t)</b>	6.70
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

**Leaching conditions**

Test #	ATS (g/L)	Cuen2 (mM)	AH (M)	O <sub>2</sub> %	T (°C)	STS (g/L)	Total EDA (mM)	Cu (ppm)	pH (by NaOH)
1	6.00		0.04	21.1%	50			0.5	
2	6.00	0.6		21.1%	50		2.4		7.4
3		0.6		21.1%	50	10.0	2.4		7.8
4		0.6		21.1%	50	10.0	2.4		7.8

**Result: Thiosulfate**

Time (h)	Concentration (g/L)			Consumption (kg/t of ore)		
Test #	0	8	-	0	8	-
1	6.00	4.80		0	2.4	
2	6.00	4.80		0	2.4	
3	10.00	8.70		0	2.6	
4	10.00	8.70		0	2.6	

**Result: Metal balance**

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	49.6	3.11	0.38	0.31	0.02	94.24%	5.76%	6.65
2	100	48.9	3.14	0.35	0.31	0.02	94.86%	5.14%	6.77
3	100	50.1	3.17	0.40	0.32	0.02	94.07%	5.93%	6.73
4	100	49.7	3.09	0.44	0.31	0.02	93.35%	6.65%	6.66

**Result: Leaching kinetics**

(as Au)

Time (h)	Solution assay (mg/L)			Extraction (%)		
Test #	0	8	-	0	8	-
1	0	3.11		0	94.24%	
2	0	3.14		0	94.86%	
3	0	3.17		0	94.07%	
4	0	3.09		0	93.35%	

**TS leaching test**

<b>Group No.:</b>	S-Z5	<b>Date:</b>	Feb-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu-EDA						

**Ore properties and pulp conditions**

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cuen2 (mM)	AH (M)	O <sub>2</sub> %	Mineral	STS (M)	Total EDA (mM)	
1	0.00	0.6	0.0	21.1%		0.1	1.8	
2	0.00	1.2	0.0	21.1%		0.1	3.6	
3	0.00	6.0	0.0	21.1%		0.1	18.0	
4	0.00	12.0	0.0	21.1%		0.1	36.0	

**Result: Thiosulfate**

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	3	9	-	0	3	9	-
1	0.10		0.09		0		2.7	
2	0.10		0.09		0		3.1	
3	0.10		0.09		0		3.3	
4	0.10		0.09		0		3.5	

**Result: Metal balance**

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.16	0.80	0.12	0.04	74.36%	25.64%	3.13
2	100	50	1.49	0.18	0.15	0.01	94.29%	5.71%	3.16
3	100	50	1.50	0.17	0.15	0.01	94.66%	5.34%	3.17
4	100	50	1.47	0.19	0.15	0.01	93.95%	6.05%	3.12

**Result: Leaching kinetics**

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	3	9	-	0	3	9	-
1	0		1.16		0		74.36%	
2	0		1.49		0		94.29%	
3	0		1.50		0		94.66%	
4	1		1.47		0		93.95%	

**TS leaching test**

<b>Group No.:</b>	S-Z6	<b>Date:</b>	Feb-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of EDA:Cu ratio						

**Ore properties and pulp conditions**

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

Test #	ATS (M)	Cu (mM)	AH (M)	O <sub>2</sub> %	Mineral	STS (M)	EDA (mM)	Ratio	(EDA:Cu)
1	0.00	1.2	0.0	21.1%		0.1	1.2	1	
2	0.00	1.2	0.0	21.1%		0.1	2.4	2	
3	0.00	1.2	0.0	21.1%		0.1	3.6	3	
4	0.00	1.2	0.0	21.1%		0.1	12.0	10	

**Result: Thiosulfate (as ATS (M))**

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
	0	3	9	-	0	3	9	-
1	0.10		0.08		0		6.5	
2	0.10		0.08		0		5.0	
3	0.10		0.09		0		3.7	
4	0.10		0.09		0		3.6	

**Result: Metal balance (as Au)**

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.24	0.65	0.12	0.03	79.24%	20.76%	3.13
2	100	50	1.49	0.19	0.15	0.01	94.08%	5.92%	3.16
3	100	50	1.52	0.18	0.15	0.01	94.29%	5.71%	3.23
4	100	50	1.45	0.28	0.15	0.01	91.16%	8.84%	3.19

**Result: Leaching kinetics (as Au)**

Time (h)	Solution assay (mg/L)				Extraction (%)			
	0	3	9	-	0	3	9	-
1	0		1.24		0		79.24%	
2	0		1.49		0		94.08%	
3	0		1.52		0		94.29%	
4	1		1.45		0		91.16%	

### TS leaching test

<b>Group No.:</b>	S-Z7	<b>Date:</b>	Mar-03	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	TS-EDA leaching in presence of galena						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (mM)	AH (M)	O <sub>2</sub> %	Mineral	STS (M)	EDA (mM)	pH	Na <sub>2</sub> CO <sub>3</sub> (g/L)
1	0.00	1.2	0.0	21.1%	2.4% galena	0.1	3.6		
2	0.00	1.2	0.0	21.1%		0.1	3.6	7.5 by H <sub>2</sub> SO <sub>4</sub>	
3	0.00	1.2	0.0	21.1%		0.1	3.6		0.4
4	0.00	1.2	0.0	21.1%		0.1	3.6		2.4

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	3	9	-	0	3	9	-
1	0.10		0.09		0		3.9	
2	0.10		0.09		0		4.3	
3	0.10		0.09		0		3.7	
4	0.10		0.09		0		3.7	

#### Result: Metal balance

(as Au)

Test #	Products (ml,g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.98	1.26	0.10	0.06	60.89%	39.11%	3.22
2	100	50	1.24	0.66	0.12	0.03	79.12%	20.88%	3.14
3	100	50	1.09	0.92	0.11	0.05	70.29%	29.71%	3.10
4	100	50	1.43	0.31	0.14	0.02	90.16%	9.84%	3.18

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	3	9	-	0	3	9	-
1	0		0.98		0		60.89%	
2	0		1.24		0		79.12%	
3	0		1.09		0		70.29%	
4	0		1.43		0		90.16%	

### TS leaching test

<b>Group No.:</b>	S-F1	<b>Date:</b>	Aug-05	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of arsenopyrite						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral	Mineral %		
1	0.20	76	0.9	21.1%	Bornite	0.0%		
2	0.20	76	0.9	21.1%		1.0%		
3	0.20	76	0.9	21.1%		2.0%		
4	0.20	76	0.9	21.1%		4.0%		

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.17		0		7.6	
2	0.20		0.15		0		13.6	
3	0.20		0.13		0		19.4	
4	0.20		0.11		0		25.6	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.48	0.21	0.15	0.01	93.26%	6.74%	3.18
2	100	50	1.34	0.44	0.13	0.02	85.86%	14.14%	3.13
3	100	50	1.01	1.07	0.10	0.05	65.22%	34.78%	3.09
4	100	50	0.62	1.99	0.06	0.10	38.26%	61.74%	3.22

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	1.31	1.48		0	82.31%	93.26%	
2	0	0.83	1.34		0	53.19%	85.86%	
3	0	0.48	1.01		0	30.82%	65.22%	
4	0	0.16	0.62		0	10.01%	38.26%	

### TS leaching test

<b>Group No.:</b>	S-F2	<b>Date:</b>	Sep-05	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of Cu						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.20	0	0.9	21.1%	2% Bornite			
2	0.20	23	0.9	21.1%				
3	0.20	76	0.9	21.1%				
4	0.20	228	0.9	21.1%				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		18.0	
2	0.20		0.13		0		19.5	
3	0.20		0.13		0		21.3	
4	0.20		0.12		0		22.5	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	1.07	1.06	0.11	0.05	66.87%	33.13%	3.19
2	100	50	1.02	1.08	0.10	0.05	65.34%	34.66%	3.12
3	100	50	1.09	1.01	0.11	0.05	68.27%	31.73%	3.19
4	100	50	1.13	0.96	0.11	0.05	70.18%	29.82%	3.22

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
<b>Test #</b>	0	0.5	3	-	0	0.5	3	-
1	0	0.51	1.07		0	31.67%	66.87%	
2	0	0.48	1.02		0	30.80%	65.34%	
3	0	0.54	1.09		0	33.75%	68.27%	
4	0	0.57	1.13		0	35.62%	70.18%	

### TS leaching test

<b>Group No.:</b>	S-F3	<b>Date:</b>	Sep-05	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ATS						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.10	76	0.9	21.1%	2% Bornite			
2	0.20	76	0.9	21.1%				
3	0.30	76	0.9	21.1%				
4	0.40	76	0.9	21.1%				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.10		0.04		0		16.7	
2	0.20		0.13		0		19.5	
3	0.30		0.21		0		25.3	
4	0.40		0.30		0		30.2	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.48	2.23	0.05	0.11	30.02%	69.98%	3.19
2	100	50	1.05	1.12	0.11	0.06	65.34%	34.66%	3.22
3	100	50	1.12	0.86	0.11	0.04	72.27%	27.73%	3.10
4	100	50	1.23	0.69	0.12	0.03	78.18%	21.82%	3.15

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.25	0.48		0	15.52%	30.02%	
2	0	0.50	1.05		0	30.80%	65.34%	
3	0	0.54	1.12		0	34.75%	72.27%	
4	0	0.61	1.23		0	38.62%	78.18%	

### TS leaching test

<b>Group No.:</b>	S-F4	<b>Date:</b>	Sep-05	<b>Operator:</b>	Chen Xia	<b>Ore:</b>	Quartz-Au Ore
<b>Purpose:</b>	Effect of ammonia						

#### Ore properties and pulp conditions

<b>Quartz-Au ore (g)</b>	10	<b>Solution (mL)</b>	100
<b>Mineral (g)</b>	Yes, see leaching conditions	<b>Au grade (g/t)</b>	3.25
<b>Mixture (g)</b>	50	<b>Pulp Density (%)</b>	33.3%

#### Leaching conditions

Test #	ATS (M)	Cu (ppm)	AH (M)	O <sub>2</sub> %	Mineral			
1	0.20	76	0.3	21.1%	2% Bornite			
2	0.20	76	0.9	21.1%				
3	0.20	76	1.5	21.1%				
4	0.20	76	3.0	21.1%				

#### Result: Thiosulfate

Time (h)	Concentration (mol/L)				Consumption (kg/t of ore)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0.20		0.14		0		18.7	
2	0.20		0.13		0		20.5	
3	0.20		0.12		0		23.3	
4	0.20		0.13		0		19.9	

#### Result: Metal balance

(as Au)

Test #	Products (ml.g)		Assay (mg/L,g/t)		Au unit (mg)		Au distribution (%)		Calc head (g/t)
	Preg.	Residue	Preg.	Residue	Preg.	Residue	Preg.	Residue	
1	100	50	0.48	2.23	0.05	0.11	30.02%	69.98%	3.18
2	100	50	1.01	1.07	0.10	0.05	65.34%	34.66%	3.09
3	100	50	1.15	0.88	0.11	0.04	72.27%	27.73%	3.17
4	100	50	1.09	1.02	0.11	0.05	68.18%	31.82%	3.20

#### Result: Leaching kinetics

(as Au)

Time (h)	Solution assay (mg/L)				Extraction (%)			
Test #	0	0.5	3	-	0	0.5	3	-
1	0	0.10	0.48		0	6.52%	30.02%	
2	0	0.48	1.01		0	30.80%	65.34%	
3	0	0.55	1.15		0	34.75%	72.27%	
4	0	0.52	1.09		0	32.62%	68.18%	