THE UNIVERSITY OF ZAMBIA

SCHOOL OF MINES

DEPARTMENT OF METALLURGY AND MINERAL PROCESSING

MM 590 PROJECT REPORT

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NKANA MINE SITE,
COBALT REFINING PLANT

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MM590 FINAL YEAR PROJECT

LEACHING OF COLBALT ANODE SLUDGE FOR THE PRODUCTION OF COBALT METAL USING SODIUM SULPHITE AS A REDUCTANT

THIS REPORT IS SUBMITTED IN PARTIAL FULFILMENT OF THE REQUIREMENT FOR THE AWARD OF THE DEGREE OF BACHELOR OF MINERAL SCIENCES (B.MIN.SC) IN METALLURGY AND MINERAL PROCESSING OF THE UNIVERSITY OF ZAMBIA

BY

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2008/9

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DEDICATIONS

I would like to dedicate this dissertation to my late dad – Mr. Maliji Mateu Arthur, for his seed of determination and self reliance he sowed in my life, Mum for her expression of love through sacrifice and hard – work, and my brothers and sisters for their continued support in my academic work.

Above all, to my great God who supply all my needs according to his riches in glory.

ACKNOWLEDGEMENT

I acknowledge with thanks the kind assistance rendered to me during my project period. I would first like to extend my sincere gratitude to my project supervisor, Mr. J. Manchisi (senior lecturer, UNZA), whose patience and guidance gave me confidence in my work. I cannot think of a word that would sufficiently express my indebtedness to him; thank you alone is not adequate.

Many thanks to Mr. Shankar A. Rao (senior metallurgist), John Kane (metallurgist) and the entire metallurgists at Mopani Cobalt Plant, for the help they extended to me during my project test works.

Lastly by no means the least, to all who stood by me in the time of need during my academic career at UNZA , many though they may be, there are only two words which come close to expressing my gratitude, inadequate though they may be – THANK YOU.

ABSTRACT

During electrowinning in the Nkana cobalt tank house, sludge is formed from the anodes and it is periodically collected from the cells and stored. The sludge contains appreciable amount of cobalt which is not recovered at the moment. It is for this reason that the project was proposed to investigate the possibility of recovering cobalt metal from the sludge.

Therefore, the objective of this study was to establish the potential of recovering cobalt metal from the anode sludge by determining the dissolution rate of cobalt metal as a function of pH, redox potential and temperature.

Leaching experiments were conducted using sulphuric acid as a leachant and Sodium Sulphite as a reductant to reduce the Co^{3+} and for redox potential control in the cobalt anode sludge pulp to a readily leacheable Co^{2+}.

From the results, highest leach efficiencies of 52.27% cobalt at the pH of 2.0 and 68.6% at the temperature of 60°C were obtained. It can be concluded that cobalt is potentially leachable from the anode sludge but there is need to conduct more tests in order to increase the leach efficiencies.

It is recommended that further tests should be done such as observing the effects of leachant concentration and use of other reductants on leaching efficiency as well as the mineralogical examinations to identify the other elements/compounds present in the cobalt anode sludge to optimise the leach efficiency.

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Chapter 1: Introduction

1.0 Project Background

1.1 Introduction

This project was done at Nkana Cobalt Leach Plant of Mopani Copper Mines (MCM) plc in Kitwe. It is based on experimental investigations on the leacheability of cobalt anode sludge for the production of cobalt metal.

The Nkana Cobalt Plant was commissioned in 1982. Current cobalt metal production capacity is 2940 tonnes per annum at 95 % plant availability. It is one of the two existing cobalt electrowinning refineries in Zambia. The design capacity for copper (which is a by-product) is about 14000 tonnes per annum. The plant is divided into two circuits, the copper circuit and the cobalt circuit. The copper side treats copper and cobalt sulphides via a fluidized bed reactor. This converts the acid insoluble sulphides to acid soluble sulphates at an average conversion of 90% for both copper and cobalt mineral values.

The copper is removed from a cobalt rich electrolyte by preferential electrowinning in the copper tank house. Further purification takes place by selective precipitation with quick lime and sulphuric acid in the cobalt circuit. Cobalt metal is produced in plate form in the cobalt tank house after which is reduced to chip form, burnished, degassed and exported in 68 litre drums, weighing 250 or 200 kg net. Nkana cobalt Plant MCM is BSI 9001 accredited

1.2 Plant description

The material treated at the Nkana Cobalt Leach Plant is the sulphide concentrate received from the concentrator after it has been finely ground (milled) and separated from the copper oxide by a floatation process with the aid of frothers, depressants, collectors and pH variation. The sulphide concentrate is directly taken to the roaster were it is further treated.

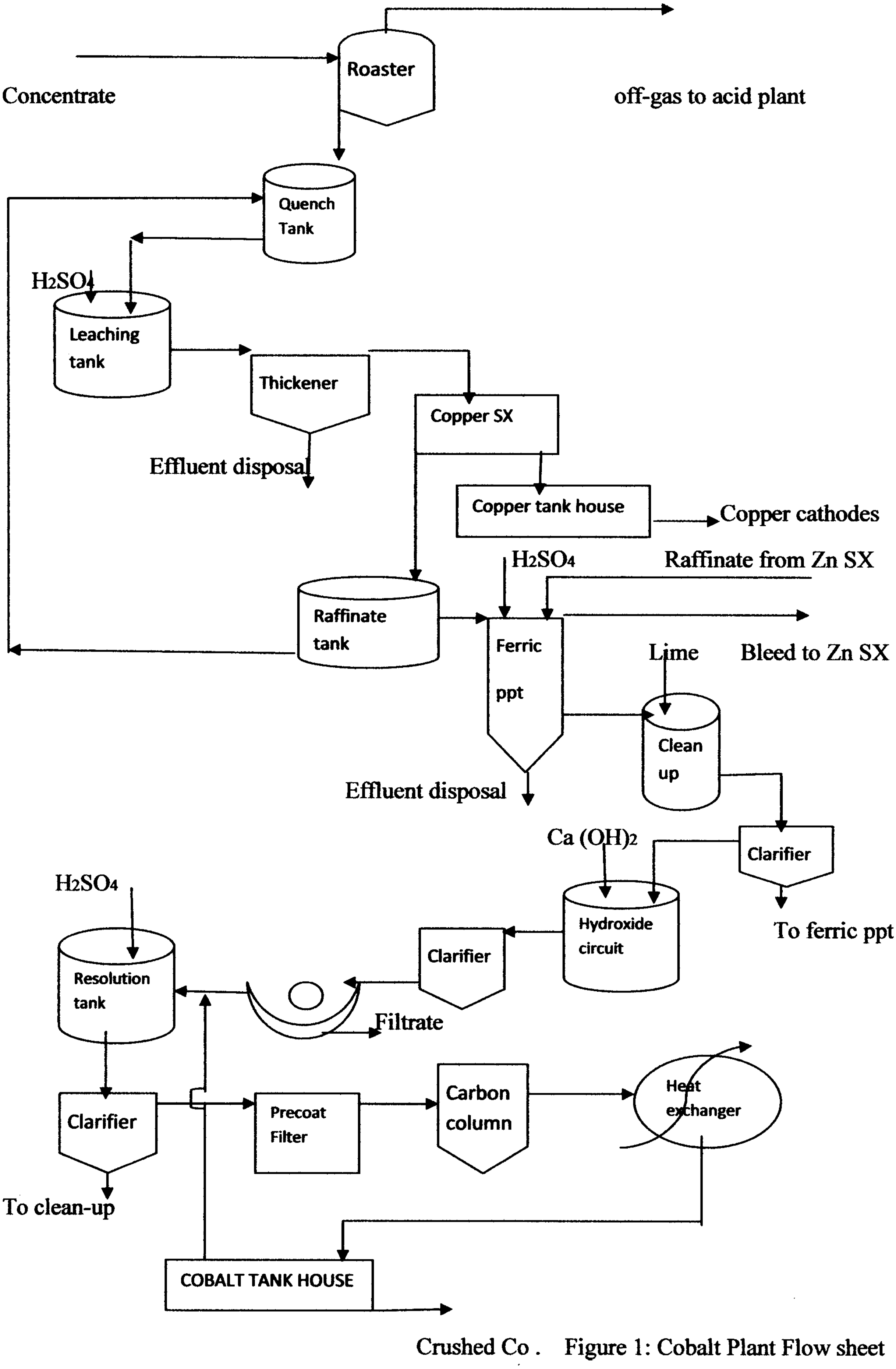

Figure 1: Cobalt Plant Flow sheet

1.2.1 Roasting

Sulphating roasting of cobalt sulphide concentrates in slurry form at solid specific gravity of 1.8 to 2.0 is done in a fluidised bed reactor (roaster) employing enriched air at 500 tonnes per day. The bed temperature of the roaster is maintained between 695 to 705°C to achieve good copper/cobalt conversions.

Currently the bulk of concentrates treated is from the Mopani Nkana sulphide concentrator. Previously Nkana cobalt concentrates and Chibuluma concentrates were treated through the plant. Pyrite is sometimes used to sustain the sulphur/heat balance for roasting when the sulphur grade in the roaster feed is low. Roaster concentrate throughput of about 465 tonnes per day is currently being achieved with oxygen enrichment of 30%, due to recent operation innovation, which have pushed this up from the previous capacity restriction of 420 tonnes per day.

The two realisable products of roasting are roaster-off-gas at about 4 to 6% sulphur dioxide and complex sulphate (calcine). The roaster –off-gas is cooled in the cooling and cleaning section prior to venting. To comply with environment regulations, a newly built acid plant consumes the flue from the roasting reactor. Calcine is leached with sulphuric acid for the eventual electrowinning of both copper and cobalt respectively.

1.2.2 Leaching, thickening and filtration

Roaster calcine is quenched from about 700 to 65 °C and subsequently leached with copper spent solution containing 15 to 25 gpl sulphuric acid (from the copper electro win section of the copper tank house) in the calcine leach cascades. For good leaching efficiencies, the pH of the leach cascades is maintained at levels below 1.50 by addition of fresh sulphuric acid if required.

The leached slurry is thickened and filtered. The filtrate is returned to the calcine thickeners, whereas the washed filter cake is repulped in water and routed to effluent disposal. The copper and cobalt losses from the repulped cake are maintained below 1.5 and 0.35% respectively. The calcine thickener overflow solution is pumped to the copper tank house as feed to the electro win section.

1.2.3 Copper tank house

The copper tank house has an electrowinning section and a copper stripping section ("A" and "B") with nominal current densities of 200 A/m², 250 A/m² and 110A/m² respectively. The nominal total copper production capacity of the tank house is about 40 tonnes per day.

The copper from the tank house at a combined average grade of about 97% Cu is dispatched to the smelter for further processing. The decopperised electro win spent bleed liquor from the electro strip 'B' section of less than 0.5 gpl copper is then fed to the cobalt circuit.

1.2.4 Cobalt circuit

The decopperised solution from the copper tank house electro stripping section is fed to the cobalt purification and concentration circuit. Impurity control at purification circuit is achieved by precipitation using lime rock or quick lime as precipitants under controlled pH conditions.

The first precipitation is mainly for iron and residual copper removal, termed the ferric precipitation circuit. Over flow from the ferric thickener, which is maintained at a terminal pH of 3.0 to 4.0, is fed to the second precipitation circuit, called the clean up stage, for zinc removal. A bleed (approximately 25 -30%) of this liquor is fed to the zinc solvent extraction stage, and the raffinate returned to the ferric cascade.

The underflow is filtered and washed prior to discard. The underflow is the only outlet for solids from the purification circuit. The underflow losses from the ferric thickener are maintained at below 0.3% cobalt. Overflow from the clean up circuit (at a pH of up to 6.70) is fed to the hydroxide circuit for total cobalt precipitation (at a pH of 8.7 -8.9), whilst the underflow is recycled back to the ferric stage. At the hydroxide circuit the thickener overflow is essentially water (with less than 50 ppm cobalt) and is disposed off to effluent. This is the main dewatering point in the cobalt circuit.

The underflow, which is filtered, is repulped with cobalt –spent electrolyte and pumped to the resolution cascade for redissolving the precipitated cobalt. Sulphuric acid is added as necessary to control the pH between 6.20 and 6.50.This pH range is necessary to maintaining zinc in the overflow below 1.0 ppm, which is important for the production of quality cobalt metal.

The resolution thickener overflow is routed to the cobalt electrolyte reactor clarifier for further solution clarification and also through the precoat filter to reduce further the suspended solids, after which it is passed through carbon columns to remove soluble sulphides and entrained organics which arise mainly from lime and flocculants used in the circuit. A bleed from the carbon column discharge is routed to the nickel plant for nickel removal before rejoining the carbon column discharge (now termed the cobalt advance electrolyte). Rejection of nickel is by an ion exchange process.

The cobalt advance electrolyte, nominally at 20 gpl cobalt is preheated before being fed to the cobalt tank house via direct electric heaters. The heated solution at about 55°C is fed to the cobalt tank house which comprises 92 cells, where the cobalt is electroplated at a current density of up to about 300A/m^2 on stainless steel blanks. Antimonial lead anodes are used. On harvesting, the electroplated cathodes are manually stripped and crushed into cobalt flakes using a roll.

The crushed metal is degassed under vacuum in a furnace at a temperature of 800°C to remove occluded hydrogen gas to a level of 1 ppm or less. After degassing the metal is burnished to remove the surface oxidation then assayed, graded and drummed for export.

1.3 Problem statement

During electrowinning in the cobalt tank house, sludge is formed at the anodes and it is periodically collected from the cells and safely stored. The sludge contains appreciable amount of cobalt which is not recovered at the moment. Cobalt metal would be beneficial to the company and the country at large. It is for this reason that the project was proposed to investigate the possibility of recovering cobalt metal from the sludge.

1.4 Objectives

The main objective of the study was to establish the potential of recovering cobalt metal from the cobalt anode sludge which is formed from the anodes during electrowinning of cobalt metal.

Specifically, the objectives were to:

- Determine the dissolution rate of cobalt metal from sludge as a function of pH, redox potential and temperature.
- Determine the leachability using sodium sulphite as a reductant

Chapter 2

Literature Review

2.0 Introduction

Leaching may be referred to as the selective dissolution of an ore or more components from a metal bearing solid material due to the action of a leachant. The leachant/lixiviant is a liquid reagent which contains the active cation or anion that causes dissolution of specified in a hydrometallurgical process (Jackson, 1987)

Obtaining of optimal leaching results requires consideration of several factors, e.g. ore/material characteristics, proper selection of the leachant to use, proper choice of leaching conditions and appropriate leaching method.

2.1 Thermodynamic aspect of leaching

During leaching, splitting species or participating species often exchange hydrogen cations (H^+) or hydroxyl (OH^-) anions and / or electrons (e^-).The hydrogen cation and hydroxyl anion creates changes in the pH of the solution while the electrons create changes in solution potential or reduction/oxidation changes.

Therefore, studying Eh – pH relationships of leaching systems is of importance to a metallurgist. Results of such studying are presented diagrammatically on Eh – pH diagrams also known as potential – pH diagrams or pourbaix diagrams.

An Eh – pH diagram shows for leaching systems at equilibrium the effect of solution potential and pH on the nature of the species in solution and on the stability of solid phase and constructed at a specified temperature, usually 25°C.Figure 33 shows an illustration of the Eh – pH diagram.

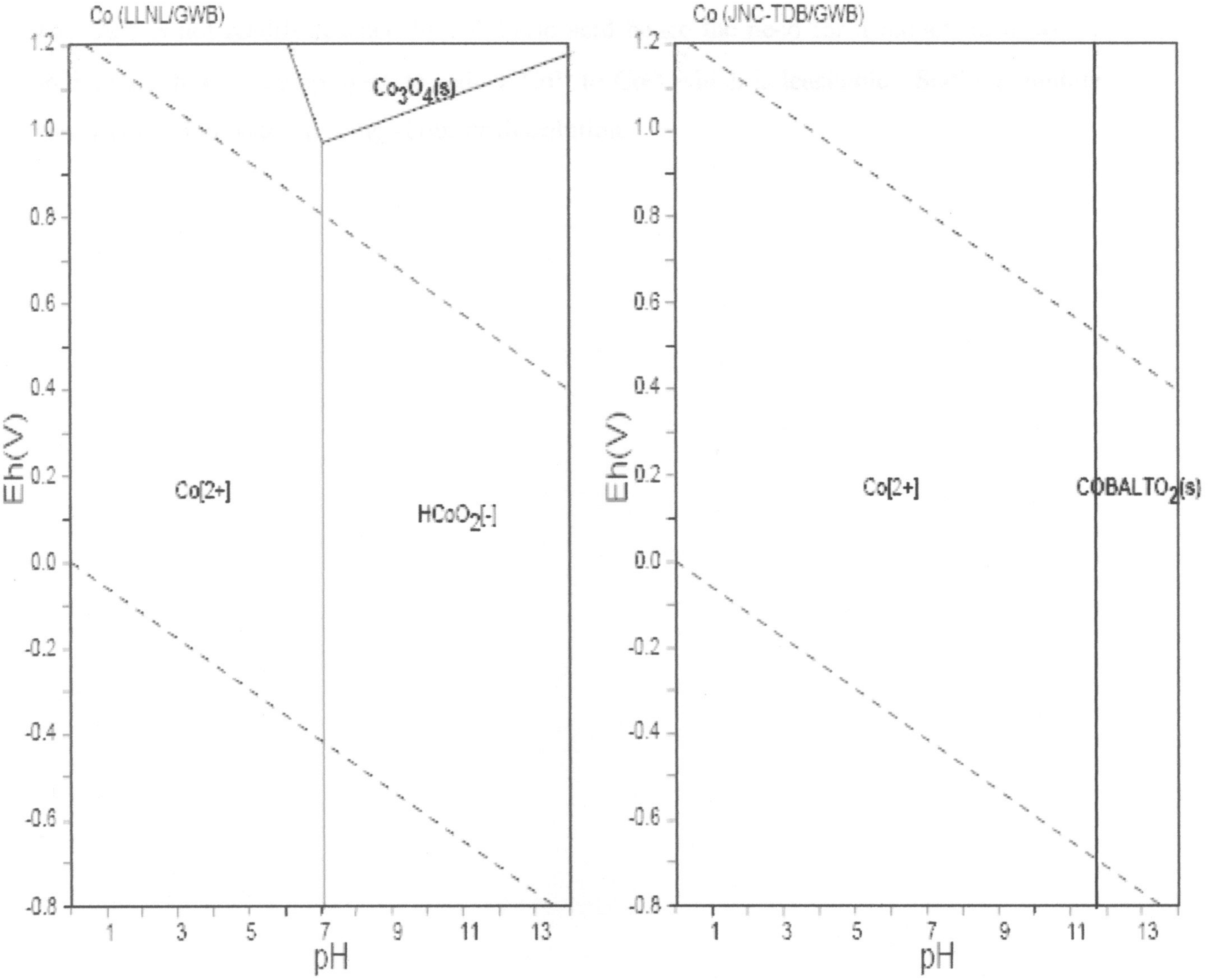

Figure 33: Eh-pH diagrams of the system Co-O-H (1). $\sum \mathrm{Co} = 10^{-10}$, 298.15K, 10^5 Pa.

2.2 Chemistry of dissolution

The cobalt anode sludge consists of Co^{3+} and Co^{2+}. Co^{3+} in the cells is formed when oxygen reacts with the Co^{2+} in the solution to form cobalt (iii) oxy – hydroxide. During the leaching process, Co^{2+} is readily leached by dilute sulphuric acid as expressed in the reaction below;

$$Co^{2+} + H_2SO_4 = CoSO_4 + 2H^+$$

The Co^{3+} is not readily leached by sulphuric acid hence the need for a reductant, however sodium sulphite was employed, to reduce Co^{3+} to Co^{2+} which is leachable. Sodium sulphite was dissolved in water to bring about its dissolution.

Chapter 3

Materials and Methods

3.1 Introduction

The leaching experiments were conducted on a laboratory scale at Mopani Copper Mine in the physical and analytical department.

3.2 Apparatus

- Burette and burette stand
- pH metre
- thermometer
- suction filtering machine
- potential metric titration meter
- measuring cylinder, wash bottle and funnel
- laboratory leach vessel
- stop watch
- heating water bath
- rock lab mill
- electronic balance
- drying oven
- agitating machine

3.3 Reagents

The reductant used in this study was sodium sulphite which the plant could readily provide and the leachant used was sulphuric acid.

Sodium sulphite solution was prepared by dissolving 200g of sodium sulphite powder in 1 litre of distilled water. Sulphuric acid was prepared by adding 500g of concentrated sulphuric acid (98%) to 1 litre of water.

3.4 Experimental Procedure

The sample was collected from the cobalt tank house were it is safely kept to form composites. The composites were washed and subsequently filtered thoroughly to remove the spent electrolyte. The sample was then taken to the ovens in order to remove the moisture. Grinding using the rock lab mill was then done. This was followed by sieving using 200 mesh of grind (75µm) and collecting 100% passing on shakers.

The undersize was then taken to the plant control section in the analytical department to determine the %TCo (percentage total cobalt). Weighing of 300g and forming four different composites was done to be leached under different conditions.

1. 300g of the formed composites was dissolved in 2.0 litres of water in a 3 litre leaching vessel
2. Prior to starting the test, the pH and Eh of the solution was measured and recorded. This was adjusted by addition of sulphuric acid and sodium sulphite
3. The stop watch was simultaneously started with agitation and measuring the pH throughout the experiment.
4. The pH and Eh was recorded after 15 minutes during the first hour and after 30 minutes in the last 3 hours.
5. During the first hour, 10ml of the sample was cut from the leach liquor and after 1hour in the last 3 hours. The cut sample was filtered and the filtrate was taken to the accounting section for the determination of gplCo.
6. After 4 hours (residence time), the leach liquor was filtered .The volume of the pregnant solution was recorded and 10ml of the leach liquor was taken for gplCo determination.
7. The residue was washed to obtain the entrained leach liquor with distilled water and the volume of the wash liquor was recorded and a 10ml was collected for gplCo determination.
8. The residue after washing was dried and its weight recorded.10g of the dried residue was taken for %TCo contained.

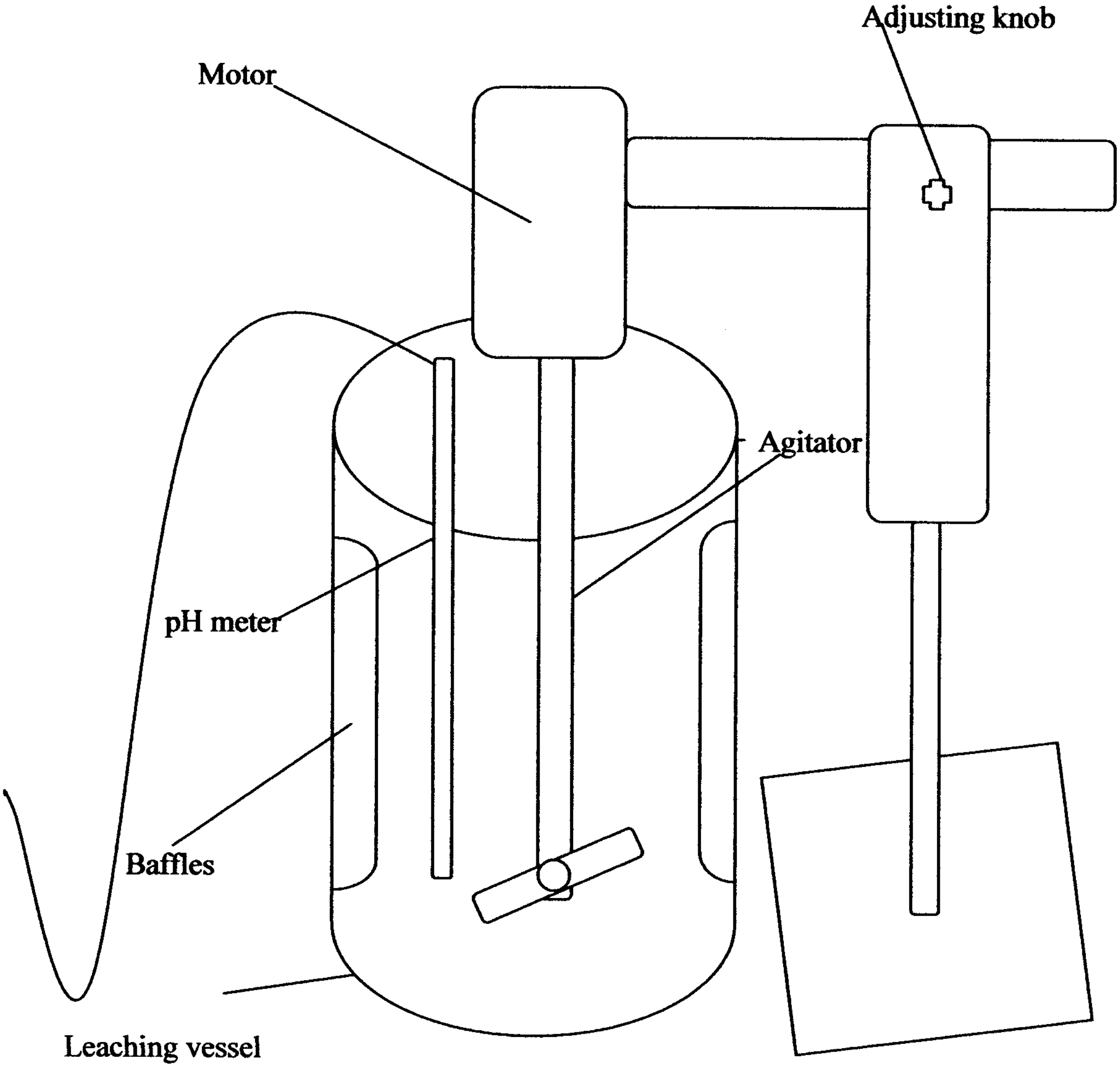

Figure 3: Experimental Setup

3.5 Effect of pH

- The above steps were followed (1 – 7). Leaching was conducted at the pH of 1.5 and at ambient temperature (~25°C).
- Repeating steps 1 – 7, the pH was raised to 1.8 and maintaining the ambient temperature (~25°C) .
- While maintaining ambient temperature, the pH was further raised to 2.0 and taking steps 1-7

3.6 Effect of temperature

- The procedure 1 – 7 was followed and the temperature was raised to 35 °C and the pH was fixed at 1.5.
- Repeating steps 1- 7, the temperature was further raised to 60°C while maintaining the pH at 1.5.

Chapter 4

Results and Discussion

4.1 Introduction

This chapter gives a discussion of the results that were obtained in this investigation under various experimental conditions. The variables that were investigated on the leaching of cobalt sludge include temperature and pH. The solution redox potential was fixed at 370mV.

4.2 Effect of pH

In order to determine the effect of pH on cobalt leach efficiency, leaching was conducted at the selected pH values of 1.5, 1.8 and 2.0. For each of these values, temperature was fixed (ambient temperature). For each pH value a pulp was prepared and leached for 4 hours (residence time) using a 3 litre leaching vessel.

Since sulphuric acid was used to control the pH while sodium sulphite was used to control the redox potential as well as being a reductant for Co^{3+} to Co^{2+}, the consumption of these reagents with time are shown in Figures 4, 8, and 12 as well as the variation of pH and Eh with time as illustrated in Figures 5 and 6.

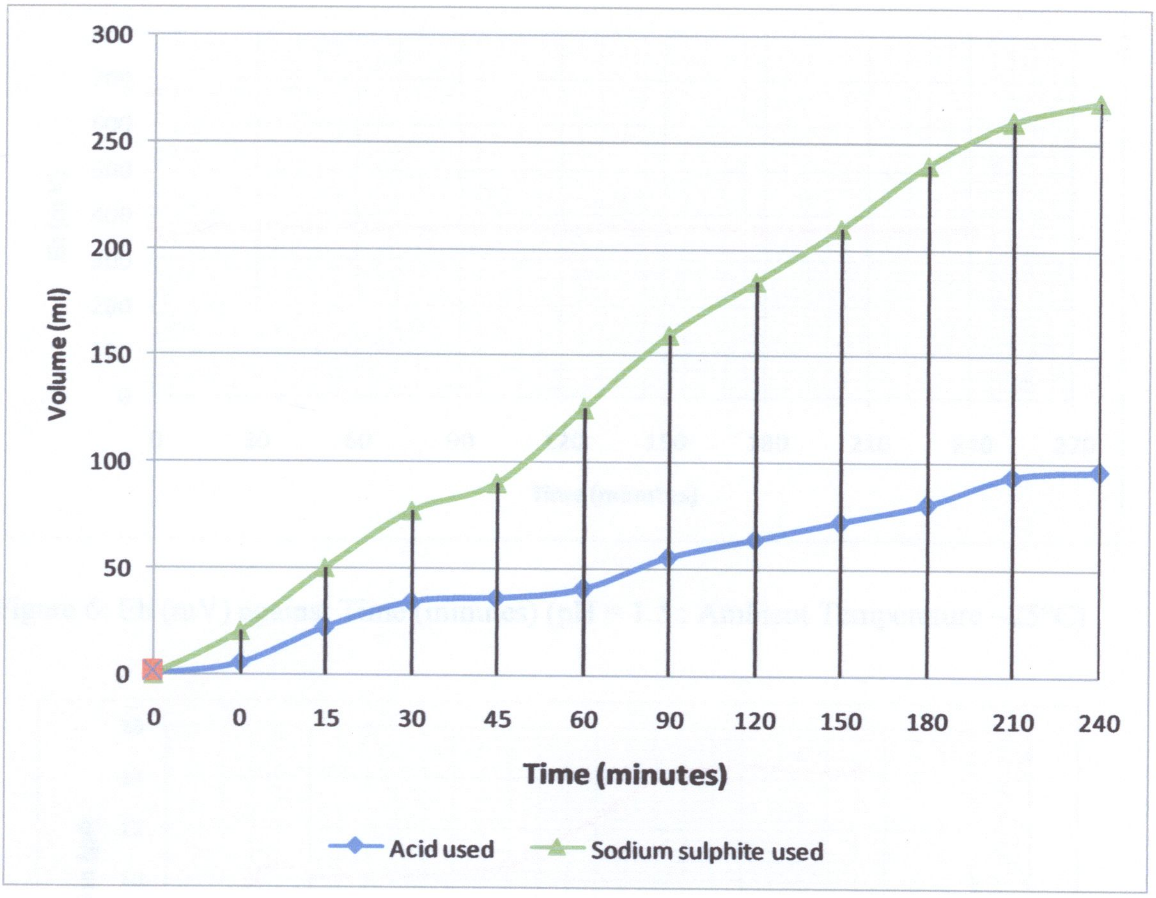

Figure 4: Volume (ml) of Sulphuric Acid and Sodium Sulphite consumed against time (pH =1.5; Ambient Temperature~ 25°C)

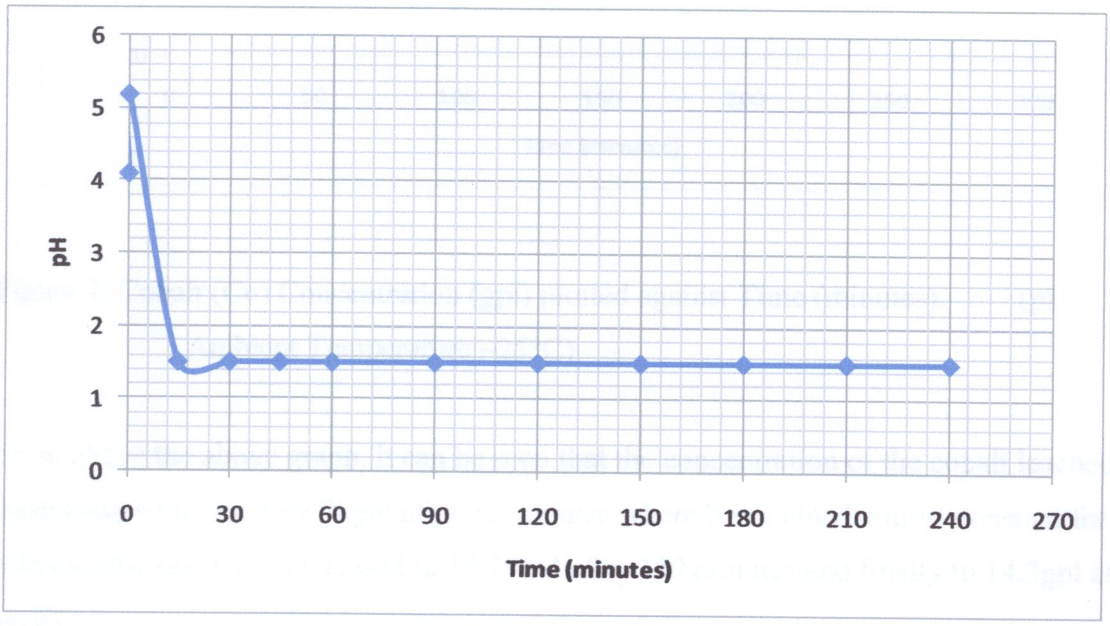

Figure 5: pH against Time (minutes) (ph = 1.5 ; Ambient Temperature ~25°C)

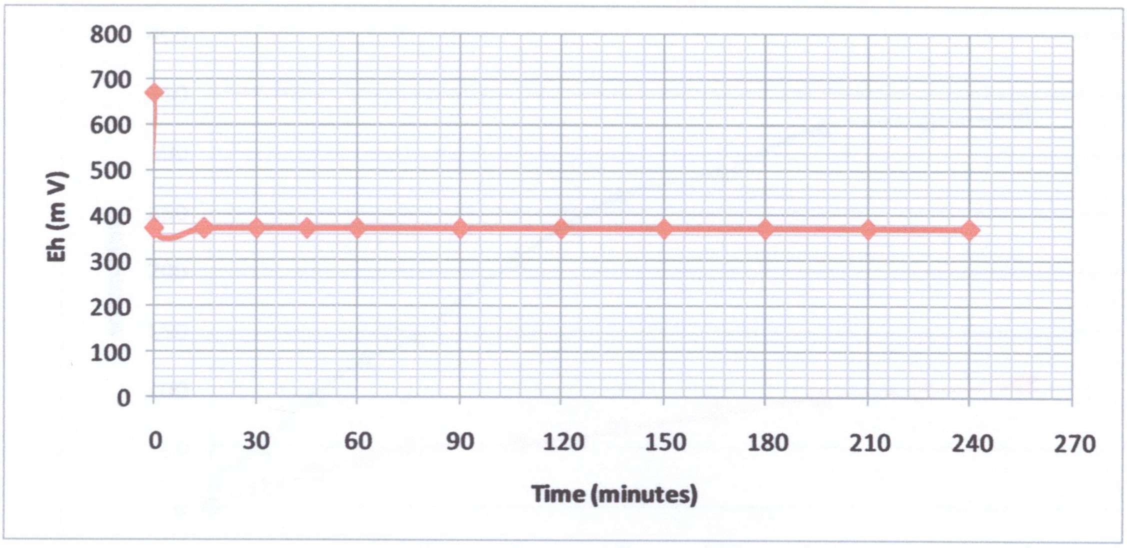

Figure 6: Eh (mV) against Time (minutes) (pH = 1.5 ; Ambient Temperature ~25°C)

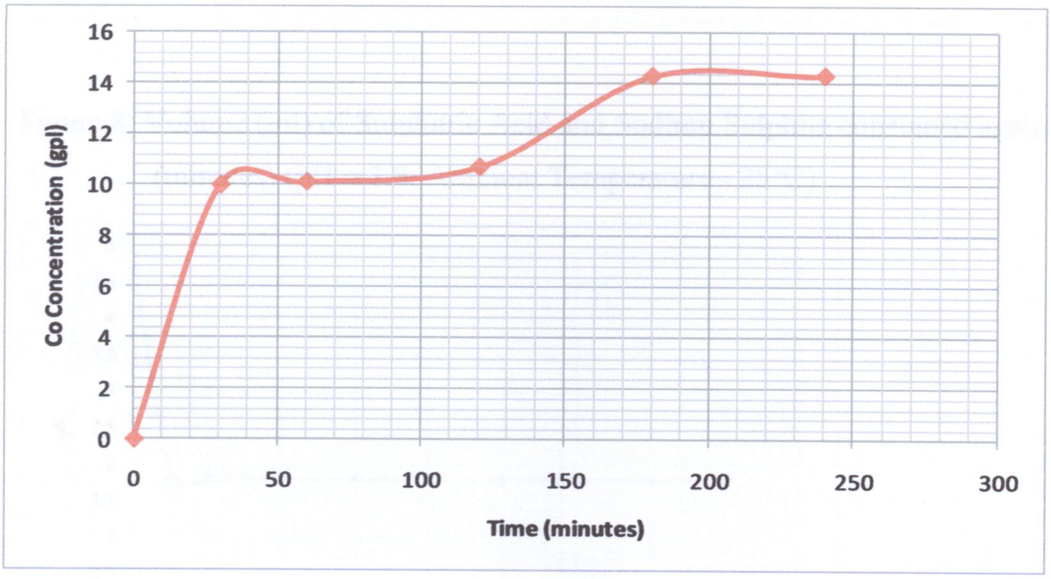

Figure 7: Cobalt (Co) Concentration (gpl) plotted against Time (minutes) (pH=1.5, Ambient Temperature ~25°C)

From above the above graph, it can be seen that the concentration of the cobalt leached was increasing with time to 9.98gpl after 30 minutes where it remained almost constant for 90 minutes and gradually increased to 14.29gpl after 180 minutes and finally to 14.3gpl after 4 hours

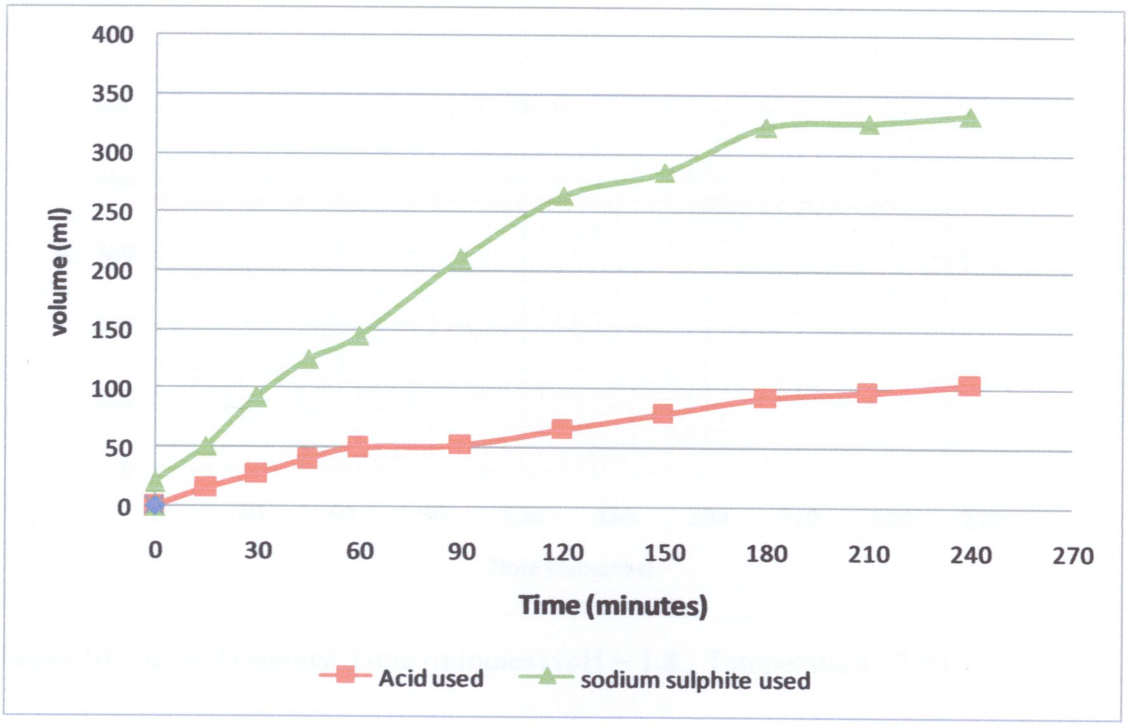

Figure 8: Volume (ml) of Sulphuric Acid and Sodium Sulphite consumed against Time (minutes) (pH = 1.8; Ambient Temperature~ 25°C)

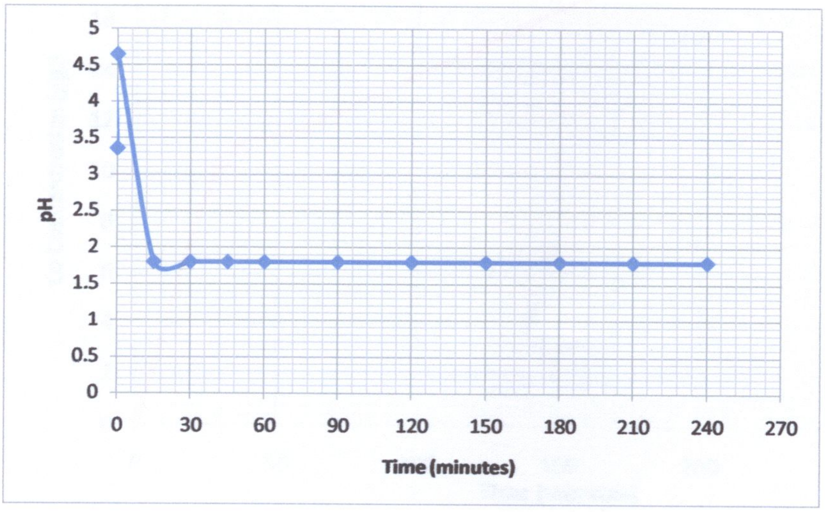

Figure 9: pH against Time (minutes) (pH = 1.8 ; Ambient Temperature ~ 25°C)

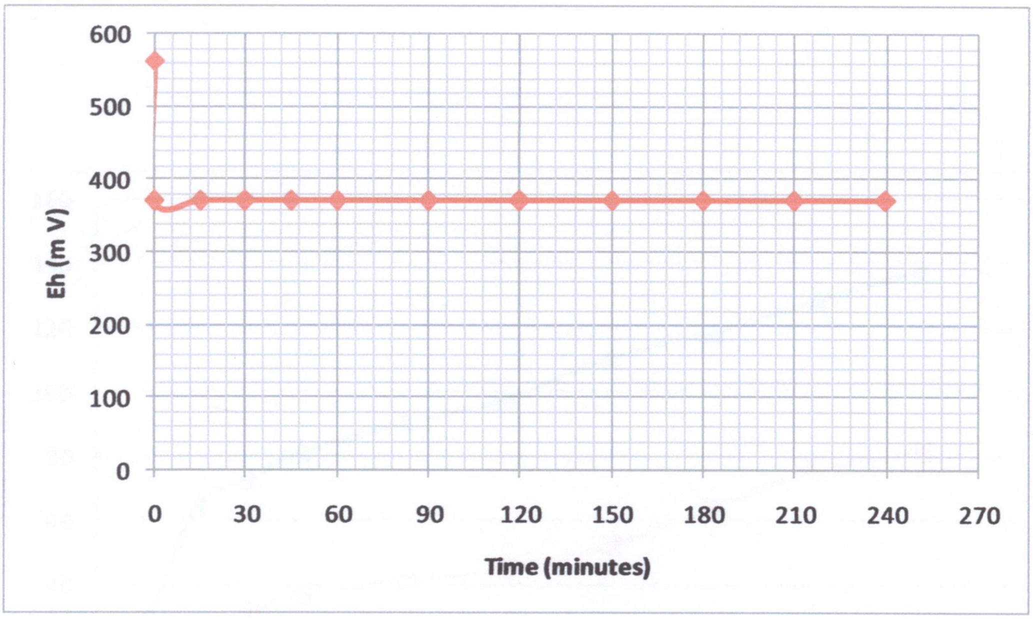

Figure 10: Eh (mV) against Time (minutes) (pH = 1.8 ; Temperature ~25°C)

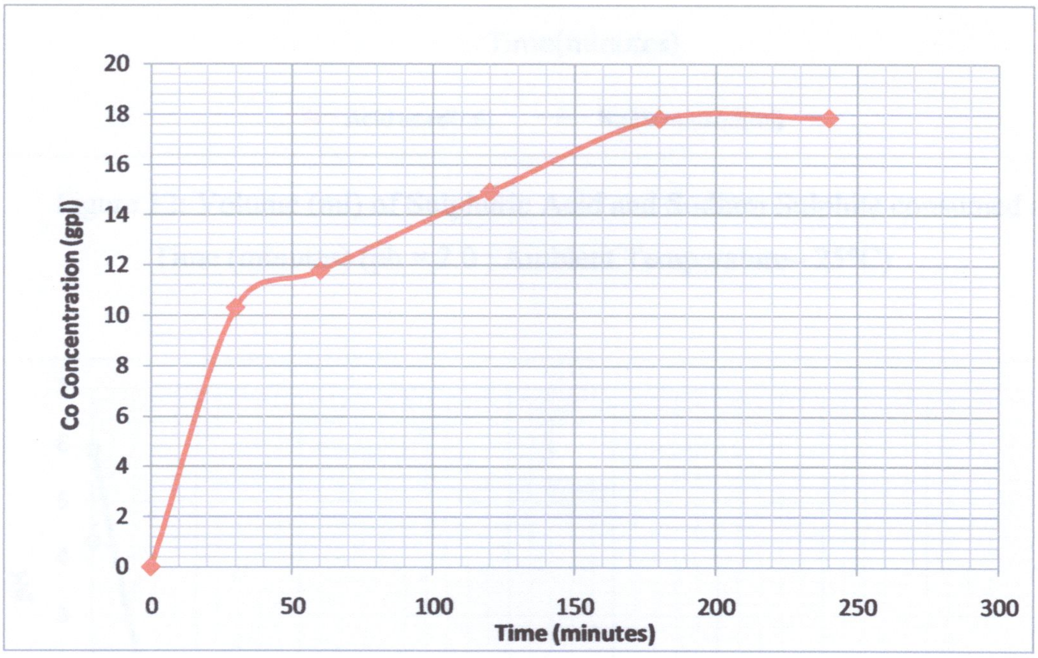

Figure 11: Cobalt (Co) Concentration (gpl) plotted against Time (minutes) (pH=1.8; Temperature=25°C)

At pH 1.8, it can be observed that Co concentration in the pregnant solution was increasing with time with 10.32gplCo after 30 minutes and gently increased to 11.78gpl, 14.93 gpl and 17.85gpl after 60, 120 and 180 minutes respectively after which 17.85gplCo after 180 minutes and finally 17.87gplCo after 240 minutes (see Figure 11).

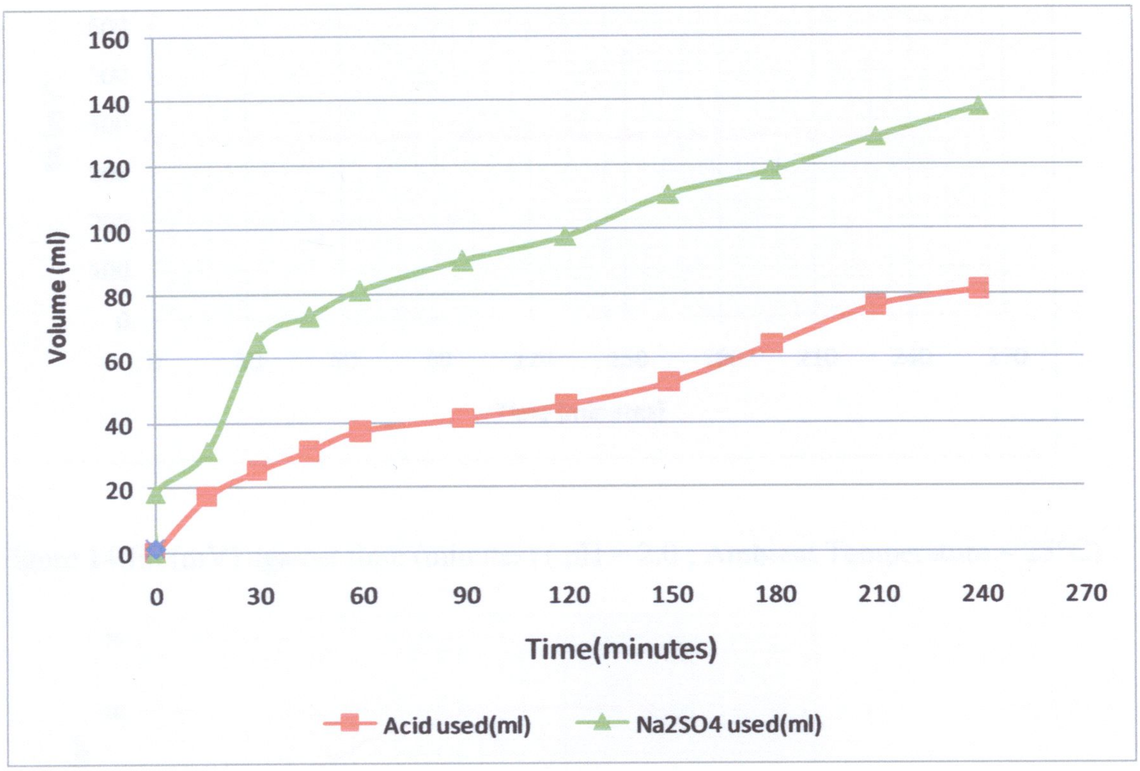

Figure 12: Volume (ml) of Sulphuric Acid and Sodium Sulphite consumed against Time (minutes) (ph = 2.0 ; Ambient Temperature~ 25°C)

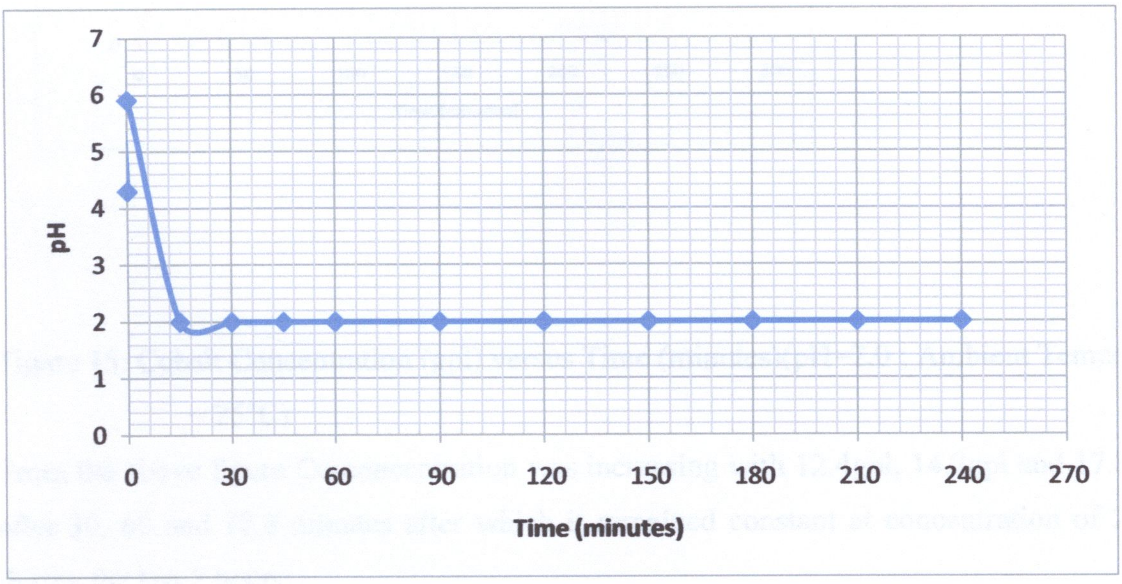

Figure 13: pH against Time (minutes) (pH = 2.0 ;Ambient Temperature)

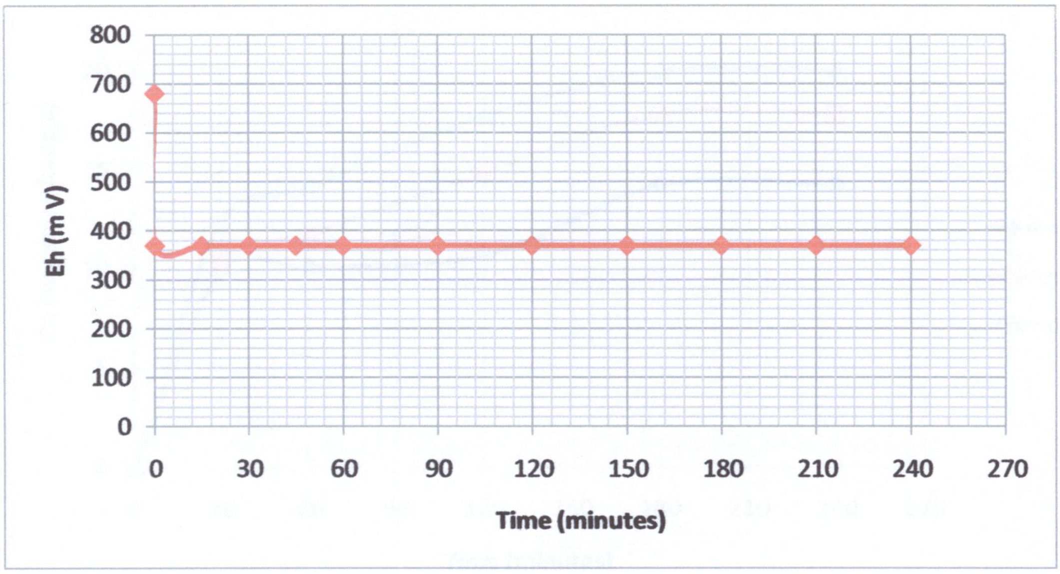

Figure 14:Eh (mV) against time (minutes) (pH = 2.0 ; Ambient Temperature ~ 25°C)

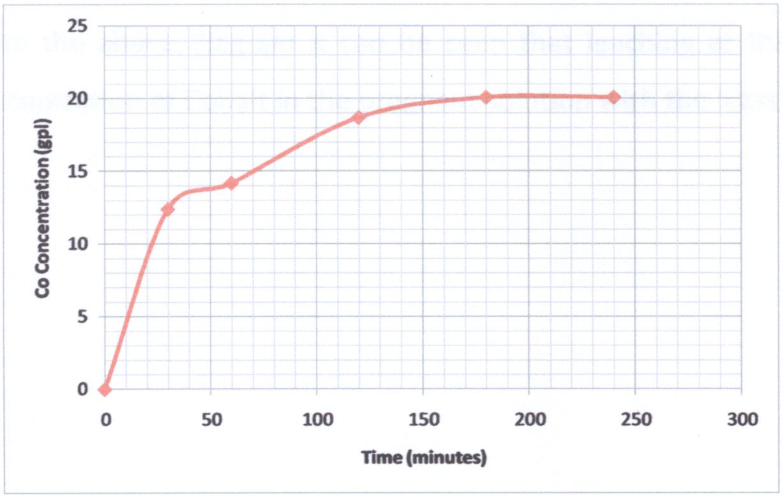

Figure 15: Cobalt Concentration (gpl) versus Time (minutes)(pH=2.0 ; Ambient Temperature ~ 25°C)

From the above figure Co concentration was increasing with 12.4gpl, 14.2gpl and 17.8gplCo after 30, 60 and 17.8 minutes after which it remained constant at concentration of 20.1gpl during the last 2 hours.

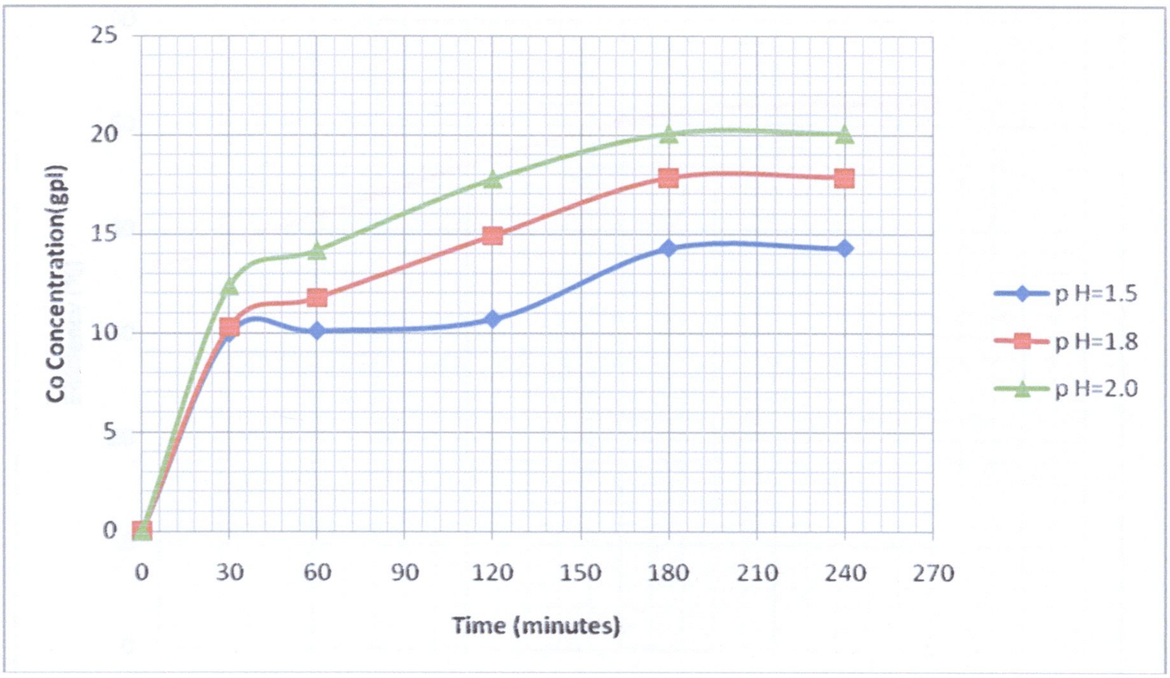

Figure 16: A plot of Co concentration (gpl) against Time (minutes) (Ambient Temperature)

From the above diagram it can be seen that leaching at the pH of 2.0 gave the highest concentration of Cobalt in the pregnant solution with the least recoveries at the pH of 1.5.

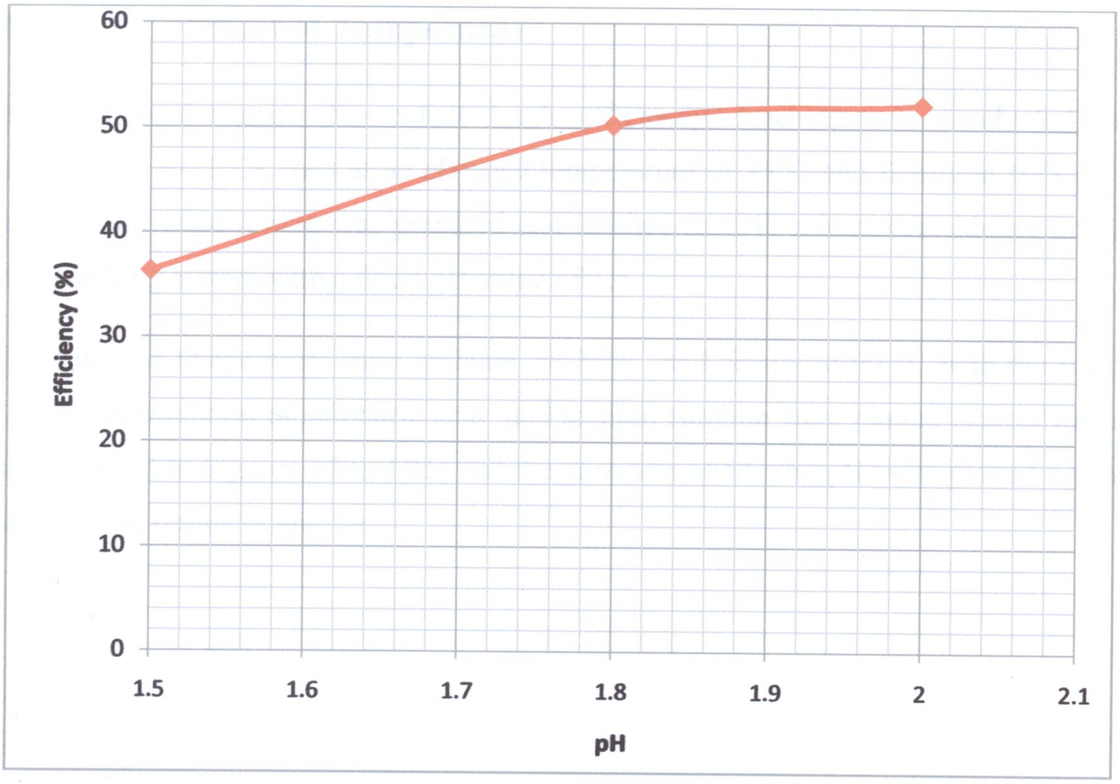

Figure 17: Cobalt Leach Efficiency plotted against pH

The pregnant solutions were analysed and the leaching efficiencies calculated were plotted against the pH as illustrated the above figure. From the graph it can be seen that leaching efficiency was increasing with pH with 36.38% and 50.3% at the pH of 1.5 and 1.8 respectively and with a maximum leaching efficiency of 52.27% at the pH of 2.0.

4.3 Effect of Temperature

In order to determine the cobalt leach efficiency, leaching was conducted at room temperature (~25°C), then at an elevated temperature of 35°C and finally at 60°C. For each temperature value a pulp was prepared and leached for 240 minutes (residence time) in a 3 litre leaching system with a fixed pH value of 1.5.

Elevating the temperature from ambient to 35°C and finally to 60 °C with the fixed pH at 1.5,the consumption of sulphuric acid and sodium sulphite were plotted against temperature as illustrated in Figures 17 and 21 and the variation of Eh and pH were plotted against time as shown in figure 18 and 19.

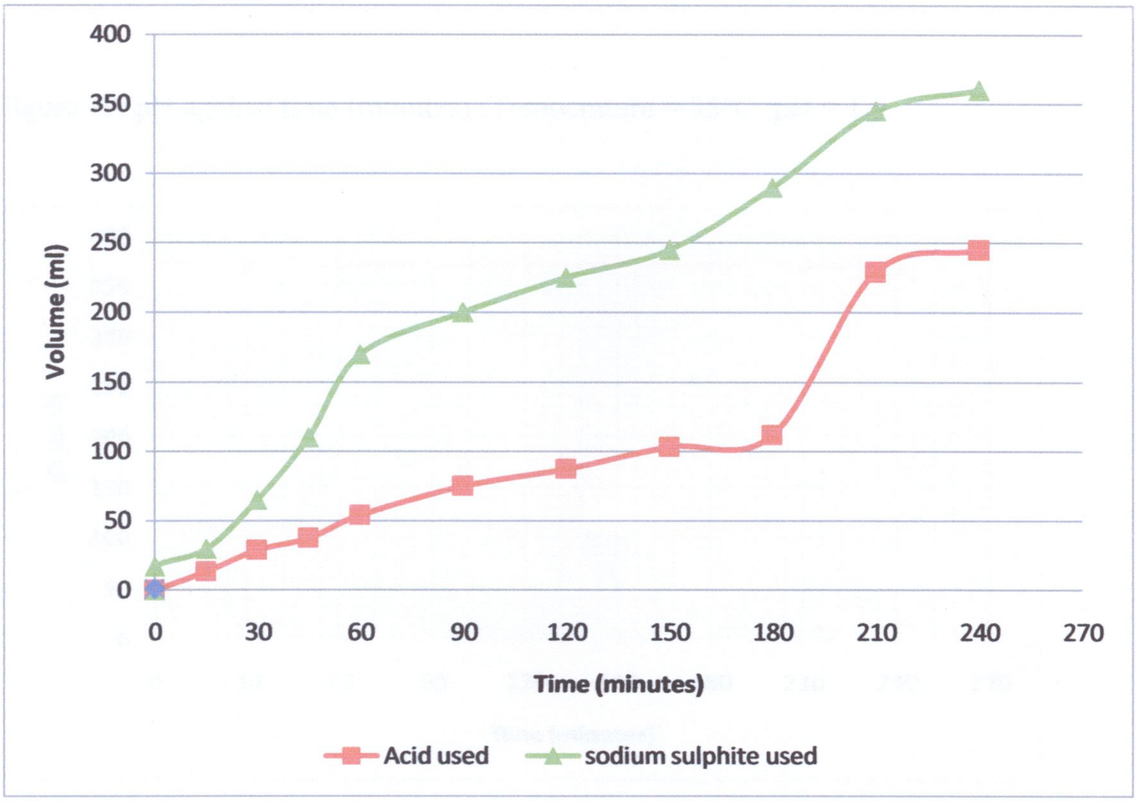

Figure 18: Volume (ml) of Sulphuric Acid and Sodium Sulphite consumed against Time (minutes) (Temperature = 35°C ; pH = 1.5)

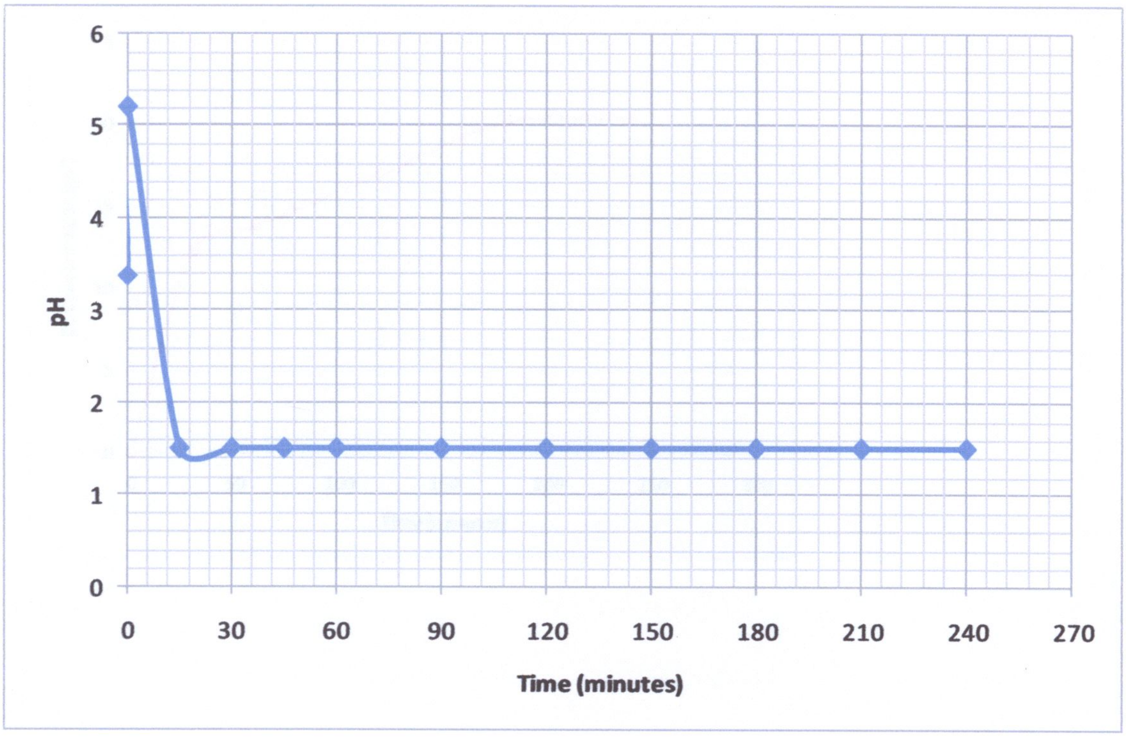

Figure 19: pH against time (minutes) (Temperature = 35°C ;pH = 1.5

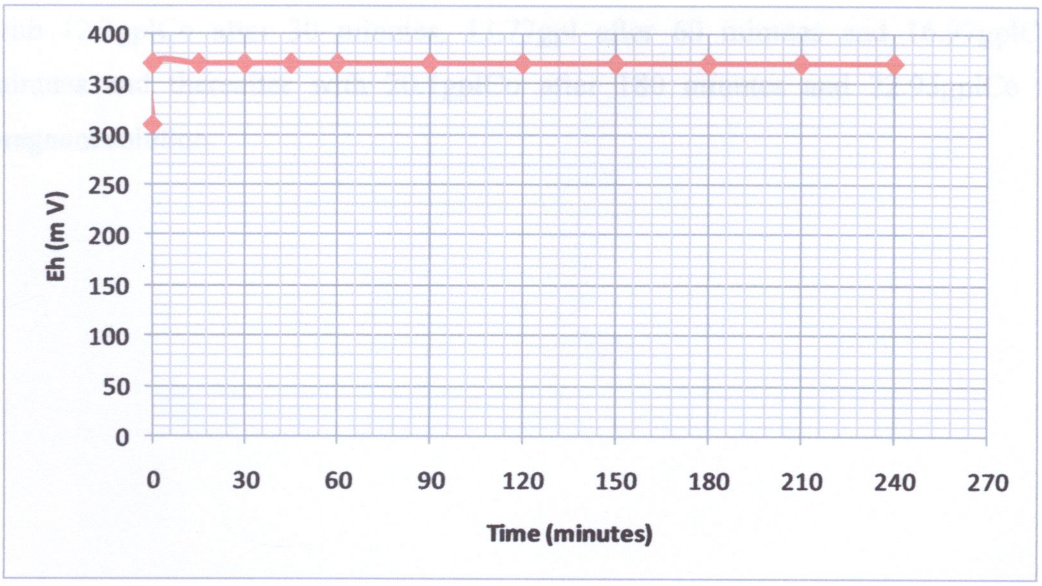

Figure 20: Eh (mV) against Time (minutes) (Temperature = 35°C ; pH = 1.5)

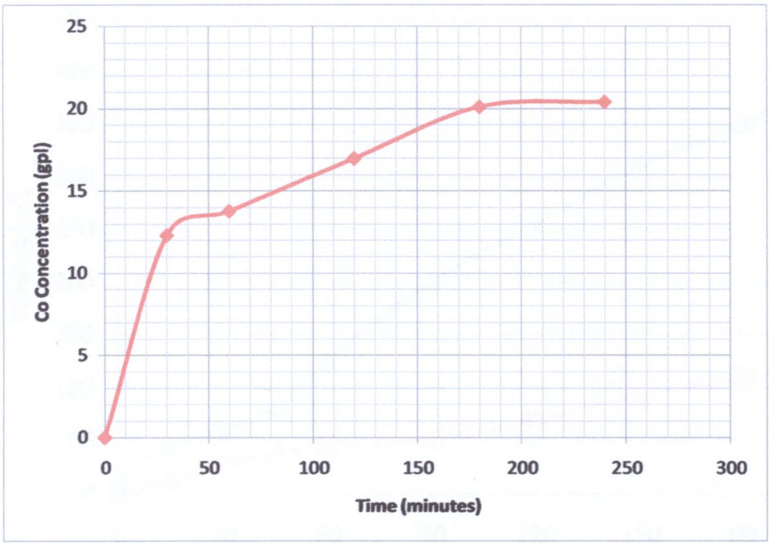

Figure 21: Cobalt (Co) Concentration plotted against Time (minutes)
(Temperature=35°C; pH=1.5)

It can be seen from the Figure above that Co concentration in the leach liquor was increasing with 12.3gplCo after 30 minutes, 13.77gpl after 60 minutes and 16.97gplCo after 120 minutes and thereafter with 20.1gplCo after 180 minutes and 22.95gplCo in the final pregnant solution.

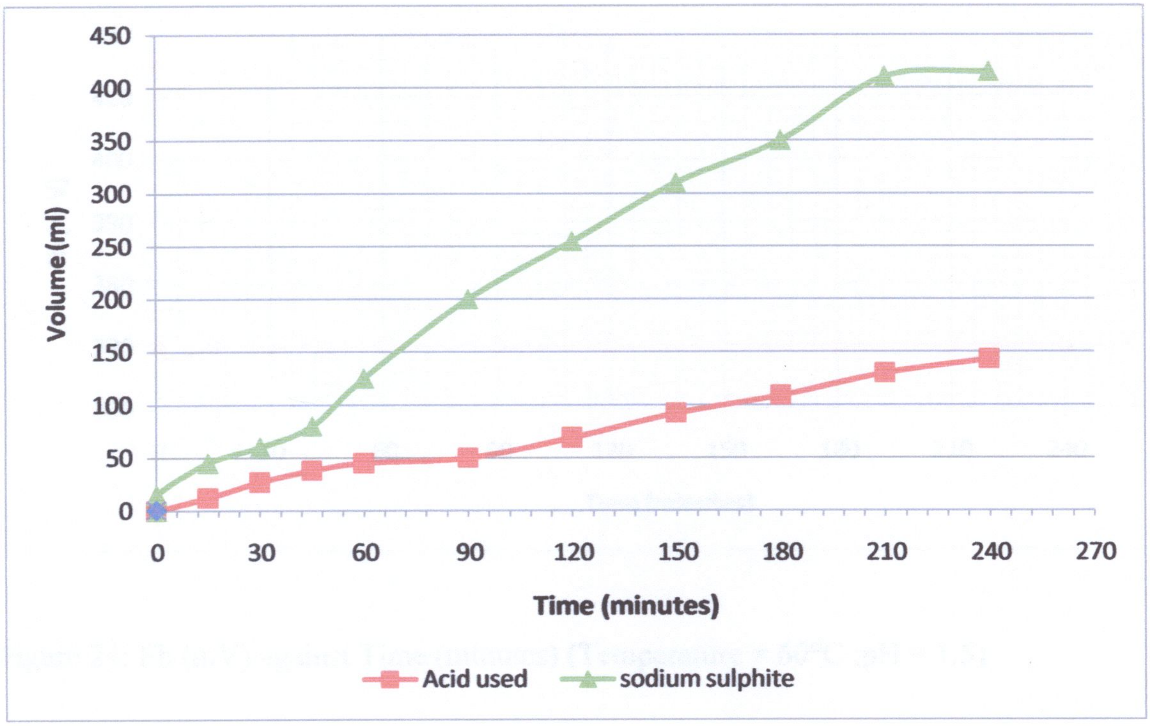

Figure 22: Volume (ml) of Sulphuric Acid and Sodium Sulphite against Time (minutes) (Temperature = 60°C ; pH = 1.5)

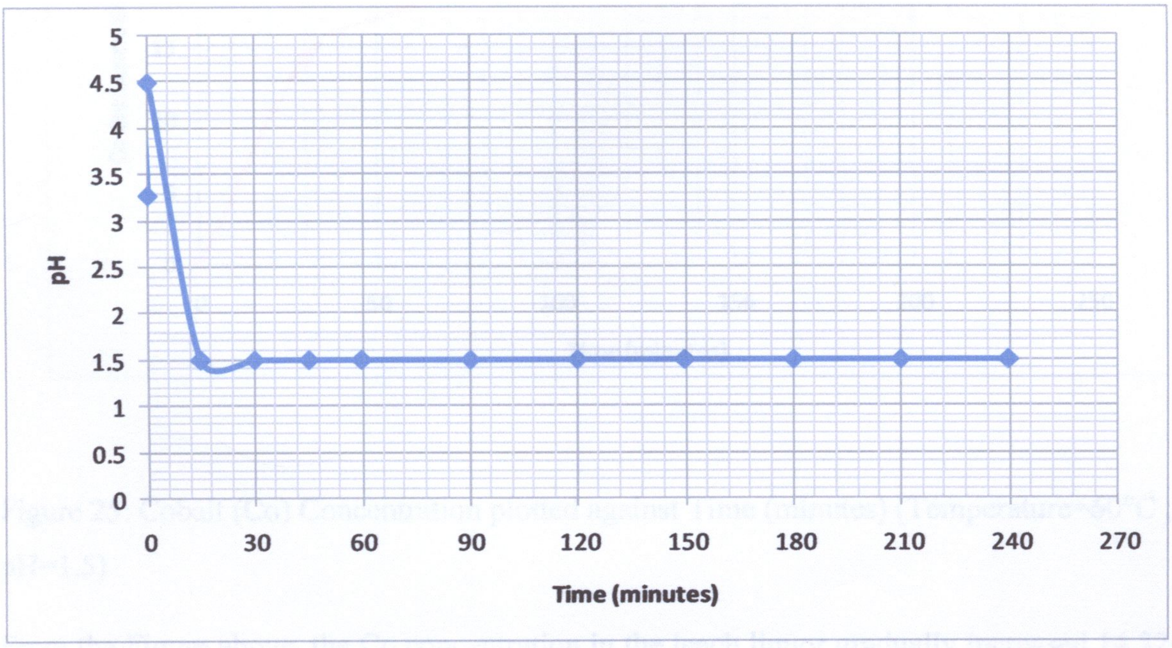

Figure 23: pH against Time (minutes) (Temperature = 60°C ; pH = 1.5)

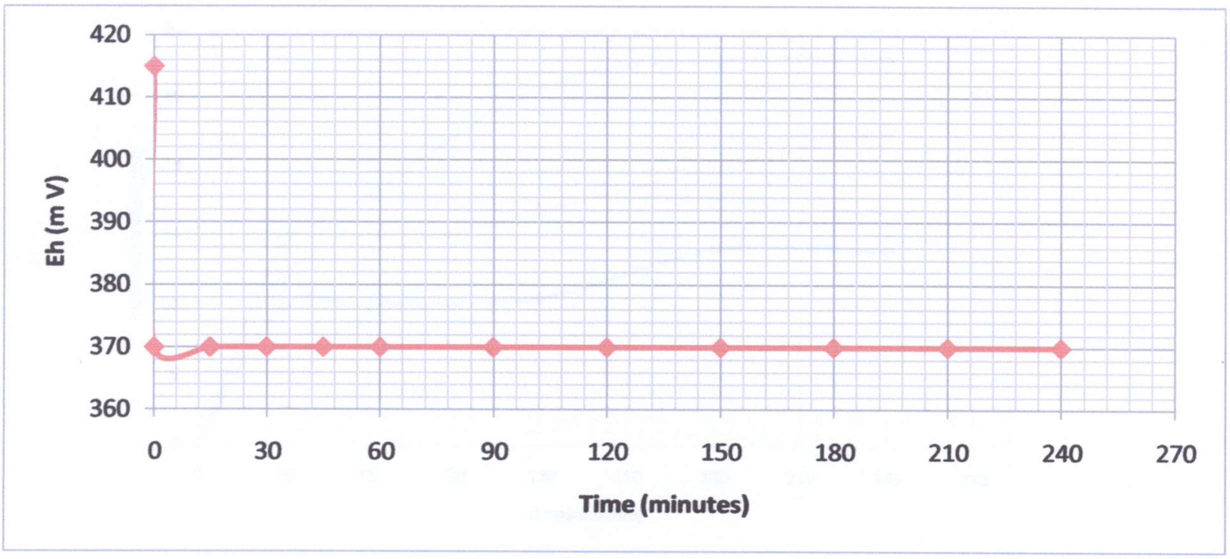

Figure 24: Eh (mV) against Time (minutes) (Temperature = 60°C ;pH = 1.5)

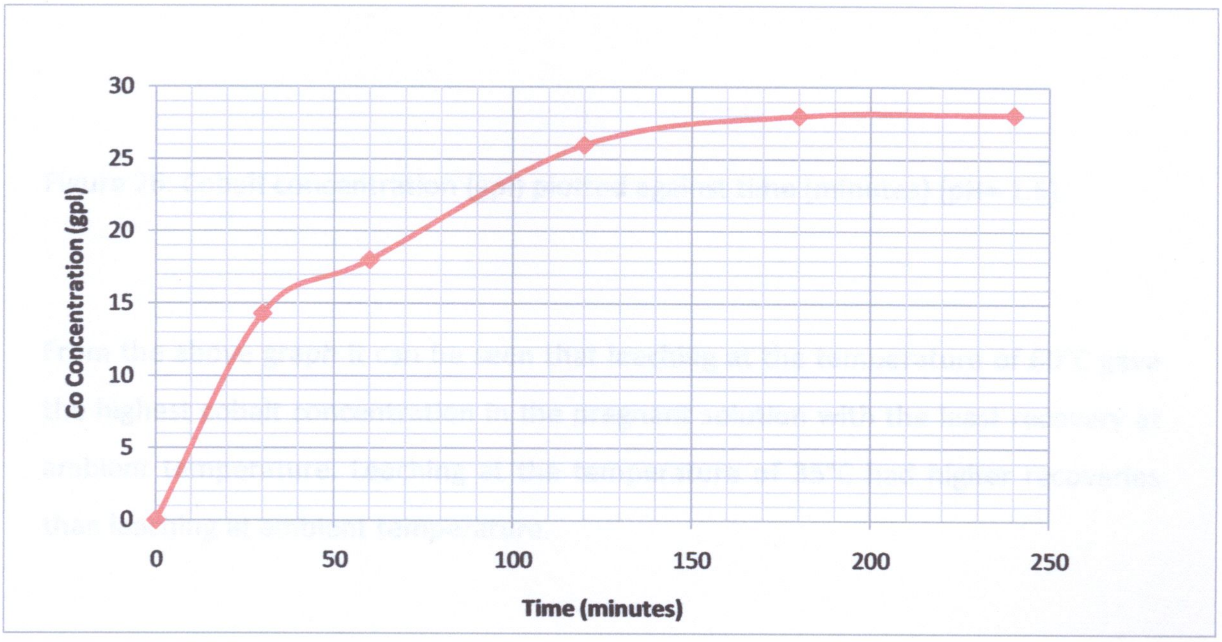

Figure 25: Cobalt (Co) Concentration plotted against Time (minutes) (Temperature=60°C ; pH=1.5)

From the Figure above, the Co concentration in the leach liquor gradually increased 14.32gpl Co after 30 minutes, 18.02 after 60 minutes to 26gplCo after 120 minutes, 28.01gplCo after 180 minutes and finally to a maximum of 28.1gplCo in the final pregnant solution after 4 hours (residence time)

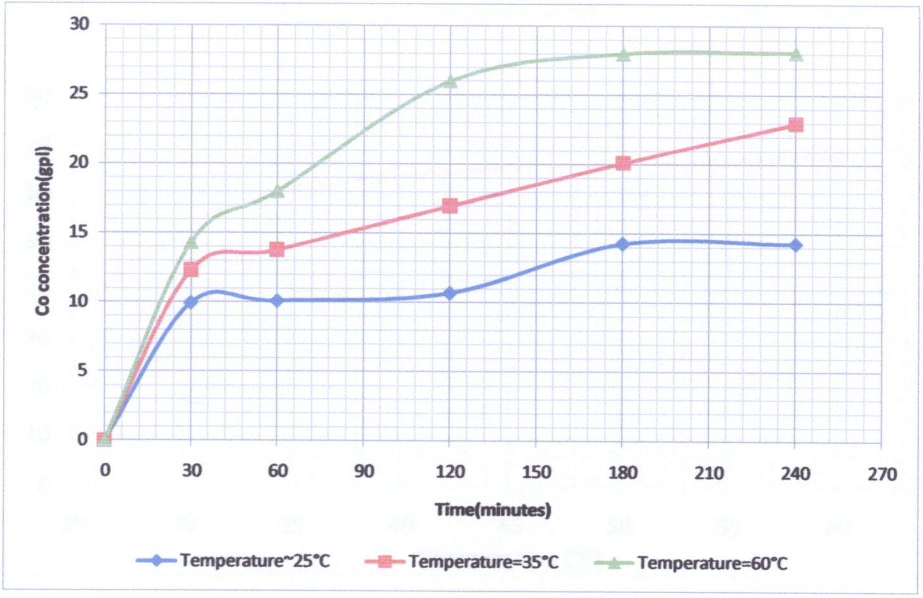

Figure 26: Cobalt concentration (gpl) plotted against time (minutes) (pH= 1.5)

From the above graph it can be seen that leaching at the temperature of 60°C gave the highest cobalt concentration in the pregnant solution with the least recovery at ambient temperature. Leaching at the temperature of 35°C had higher recoveries than leaching at ambient temperature.

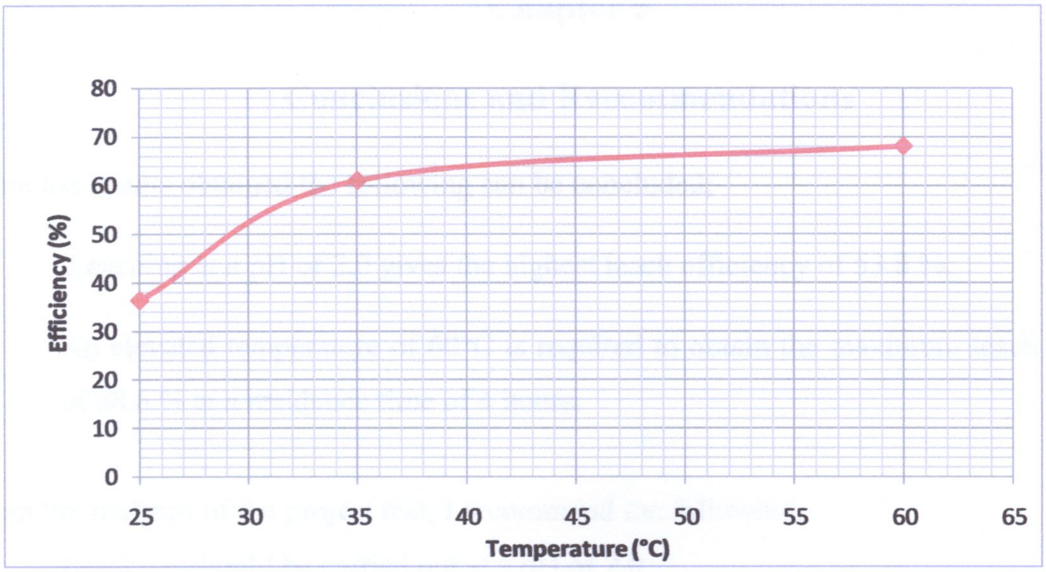

Figure 27: Efficiency against Temperature

The pregnant solutions were analysed and the leaching efficiencies calculated were plotted against temperature as shown in the above figure. From the graph it can be seen that the leaching efficiency was increasing with temperature with leaching efficiency of 36.38% and 61% at the ambient temperature (~25°C) and 35°C respectively and with the maximum leaching efficiency of 68.6% at 60°C.

Chapter 5

Conclusions and Recommendations

From the results obtained the following can be concluded;

(i) Leaching at a pH of 2.0 gives the highest leach efficiency of 53.8 %.

(ii) An elevated temperature of 60°C is required to obtain the maximum leach efficiency of 68.6 % at a residence time of 4 hours.

From the findings of the project test, I recommend the following:

(i) Leaching should be carried out at a pH of 2.0.

(ii) Elevated temperature up to 60°C should be used.

(iii) A residence time of 4 hours should be employed.

(iv) Mineralogical examinations should be carried out on the cobalt anode sludge before tests are conducted for identification of other elements/compounds to optimise the leach efficiency.

(v) A reductant is required in the leaching of the cobalt anode sludge.

(vi) Further work should be undertaken to confirm these findings.

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APPENDIX A: EFFECT OF pH

TABLE 1: Leaching at the pH of 1.5 and Room Temperature

Time (minutes)	Acid used (ml)	pH	Na_2SO_3 used (ml)	Eh (mV)
0	0	4.1	0	670
0	0	5.2	20	370
15	21.5	1.5	50	370
30	34	1.5	77	370
45	36	1.5	90	370
60	40.2	1.5	125	370
90	55	1.5	160	370
120	63.8	1.5	185	370
150	71.8	1.5	210	370
180	80.3	1.5	240	370
210	93.7	1.5	260	370
240	96.2	1.5	270	390

Table 2: Results for Leaching at the pH of 1.5 and Room Temperature

	Concentration(gpl Co) of cobalt
Pregnant solution after 30 minutes	9.98
Pregnant solution after 60 minutes	10.1
Pregnant solution after 120 minutes	10.7
Pregnant solution after 180 minutes	14.29
Pregnant solution after 240minutes	14.3
Wash liquor	10.3
Weight of sample	300g
%TCo in sample	55.2
%TCo in residue	39.7
Volume of pregnant liquor	1.98 litre
Volume of wash liquor	3.1 litre
Efficiency	36.38%

Table 3: Leaching at the pH of 1.8 and Room Temperature

Time (minutes)	pH	Acid used (ml)	Na_2SO_3 used (ml)	Eh (ml)
0	3.36	0	0	563
0	4.65	0	20	370
15	1.8	16	51	370
30	1.8	27.5	92	370
45	1.8	40	125	370
60	1.8	50.3	145	370
90	1.8	52.3	210	370
120	1.8	65.8	265	370
150	1.8	79.5	285	370
180	1.8	93.3	323	370
210	1.8	98.3	327	370
240	1.8	104.3	333	370

TABLE 4: Results for Leaching at the pH of 1.8 and Room Temperature

	Concentration of cobalt (gpl)
Pregnant solution after 30 minutes	10.32
Pregnant solution after 60 minutes	11.78
Pregnant solution after 120 minutes	14 .93
Pregnant solution after 180 minutes	17.85
Pregnant solution after 240 minutes	17.87
Wash liquor	13.9
Mass of sample	300g
Mass of residue	235.3g
%TCo in sample	55.2
%TCo in residue	39.7
Volume of leach liquor	2.3 litre
Volume of wash liquor	3.04 litre
Efficiency	50.3%

Table 5: Leaching at the pH of 2.0 and Room Temperature

Time (minutes)	pH	Acid used (ml)	Na_2SO_3 used (ml)	Eh (mV)
0	4.3	0	0	681
0	5.9	0	18.3	370
15	2	17.3	31.2	370
30	2	25	65	370
45	2	31	73	370
60	2	37.2	81	370
90	2	41	90.2	370
120	2	45.3	98	370
150	2	52.1	111	370
180	2	64	118.3	370
210	2	76	129	370
240	2	81	138	370

Table 6: Results for Leaching at the pH of 2.0 and Room Temperature

	Concentration of cobalt (gpl)
Pregnant solution after 30 minutes	12.4
Pregnant solution after 60 minutes	14.2
Pregnant solution after 120 minutes	17.8
Pregnant solution after 180 minutes	20.1
Pregnant solution after 240 minutes	20.1
Wash liquor	12.7
Mass of residue	215g
Mass of sample	300g
%TCo in sample	55.2
%TCo in residue	36
Volume of leach liquor	2.19 litre
Volume of wash liquor	3.35 litre
Efficiency	52.27%

APPENDIX B: EFFECT OF TEMPERATURE

Table 7: Leaching at the Temperature of 35°C and the pH =1.5

Time (minutes)	pH	Acid used(ml)	Na_2SO_3 used(ml)	Eh(mV)
0	3.38	0	0	310
0	5.2	0	17.2	370
15	1.5	13.5	30	370
30	1.5	29	65	370
45	1.5	37.5	110	370
60	1.5	54	170	370
90	1.5	75	200	370
120	1.5	87.5	225	370
150	1.5	103.5	245	370
180	1.5	112	290	370
210	1.5	229	345	370
240	1.5	246	360	370

Table 8: Results for Leaching at the Temperature of 35°C and the pH of 1.5

	Concentration of cobalt (gpl)
Pregnant solution after 30 minutes	12.3
Pregnant solution after 60 minutes	13.77
Pregnant solution after 120 minutes	16.97
Pregnant solution after 180 minutes	20.1
Pregnant solution after 240 minutes	22.95
Wash liquor	16.95
Mass of sample	300g
Mass of residue	215g
%TCo in sample	55.2
%TCo in residue	33.9
Volume of leach liquor	2.1 litre
Volume of wash liquor	3.12 litre
Efficiency	61%

Table 9: Leaching at the Temperature of 60°C and the pH of 1.5

Time (minutes)	pH	Acid used (ml)	Na_2SO_4 (ml)	Eh (mV)
0	3.28	0	0	415
0	4.5	0	15	370
15	1.5	12.5	45	370
30	1.5	7	60	370
45	1.5	38	80	370
60	1.5	45	125	370
90	1.5	50	200	370
120	1.5	69	255	370
150	1.5	92	310	370
180	1.5	109	350	370
210	1.5	130	410	370
240	1.5	143	415	370

Table 10: Results for Leaching at the Temperature of 60°C and the pH of 1.5

	Concentration of cobalt (gpl)
Pregnant solution after 30 minutes	14.32
Pregnant solution after 60 minutes	18.02
Pregnant solution after 120 minutes	26
Pregnant solution after 180 minutes	28.01
Pregnant solution after 240 minutes	28.1
Wash liquor	16.3
Mass of sample	300g
Mass of residue	199g
%TCo in sample	55.2
%TCo in residue	27.2
Volume of leach liquor	2.06 litre
Volume of wash liquor	3.42 litre
Efficiency	68.6%