017855

CINOLA GOLD PROJECT

STAGE II ADDENDUM

REPORT



CITY RESOURCES (CANADA) LIMITED

CINOLA GOLD PROJECT

STAGE II ADDENDUM REPORT

Submitted by:

CITY RESOURCES (CANADA) LIMITED 2000 - 666 Burrard Street Vancouver, B.C. V6C 2X8

Coordinated by:

ROBINSON DAMES AND MOORE 45 - 1199 Lynn Valley Road North Vancouver, B.C. V7J 3H2

August 1988

TABLE OF Contents

,

,

. #

.

r

.

TABLE OF CONTENTS

Page

TABLE OF CONTENTS				
LIST OF TABLES				
LIST O	f figuf	RES		
LIST O	F APPEI	NDICES		
1.0	INTRO	DUCTION		
	1.1	Transmittal Statement		
	1.2	Testing Programs		
	1.3	Report Format		
	1.4	Responsibility		
2.0	EXECL	JTIVE SUMMARY		
	2.1	Introduction		
	2.2	Mill Effluent 2-1		
	2.3	Process Update		
	2.4	Tailings Properties 2-3 Wests Deck Liming Experiments 2-3		
	2.5	Waste Noter Treatment 2-4		
	2.0			
	2.7	Commitment 2-5		
3.0	MILL E	FFLUENT CHARACTERIZATION		
	3.1	Introduction		
	3.2	Summary of Metallurgical Process		
	3.3	Pilot Plant Program - Objectives and Responsibilities		
		3.3.1 Pilot Plant Operation		
		3.3.1.1 Ore selection		
		3.3.1.2 Ore preparation		
		3.3.1.3 Nitrate oxidation stage		
		3.3.1.4 Neutralization Stage		
		3.3.1.5 Cyanidation		
		3.3.1.6 Cyanide destruction		
		3.3.1.7 Mercury removal by sulphide addition		
		3.3.2 Sampling program		
	3.4	Characterization of Slurry		
		3.4.1 Liquid phase		
		3.4.1.1 Metals		
		3.4.1.2 Uyanide species		
		3.4.1.3 Nitrogen species		

iii

		3.4.1.4 Major ions	7
		3.4.2 Solids phase	7
		3.4.2.1 Acid generation potential	7
		3.4.2.2 Metals levels in pilot plant samples	3
		3.4.2.3 Short-term leach results	Э
		3.4.2.4 Long-term leach results	2
		3.4.2.5 Grain size distribution	6
	3.5	Comparison of Pilot Plant Results with Bench Scale Test Results	6
		3.5.1 Comparison of results - liquid phase	6
		3.5.1.1 Metals	6
		3.5.1.2 Cyanide species	4
		3.5.1.3 Nitrogen species	4
		3.5.1.4 Major ions	5
		3.5.2 Comparison of results - solid phase	5
		3.5.2.1 Acid generation	5
		3.5.2.2 Metals	5
		3.5.2.3 Short-term leach	5
		3.5.2.4 Long-term leach	6
		3.5.2.5 Grain size distribution	7
	3.6	Mercury Treatment	7
		3.6.1 Introduction	7
		3.6.2 Methods and results	7
	3.7	Comparison of Cyanide Destruction Methods	1
		3.7.1 Hydrogen peroxide	1
		3.7.2 SO ₂ /air process	3
	3.8	Summary	6
4.0	PROCE	ESS UPDATE	1
	4.1	Introduction	1
	4.2	Neutralization	1
	4.3	Cvanide Leaching and Adsorption	1
	4.4	Gold Elution and Recovery	1
	4.5	Cyanide Destruction	2
	4.6	Plant Water Balance	2
	4.7	Mercury Balance	2
	4.8	Lime Calcining	3
5.0	TAILIN	GS PROPERTIES	1
	51	Introduction 5-1	1
	5.2	Objective of laboratory Testing	1
	5.3	Tailings Samples	•
	5.4	Laboratory Test Program and Results	•
	U . 1	5.4.1 Grain size analyses	2
		5.4.2 Atterberg limit tests	- 2

		5.4.3 Specific gravity tests
		5.4.4 Large diameter column settled density testing
		5.4.4.1 Drained test
		5.4.4.2 Undrained test
		5.4.4.3 Hydraulic conductivity
		5.4.5 Consolidation test
		5.4.6 Gypsum content
	5.5	Discussion and Conclusions
6.0	EXPER	RIMENTS WITH LIMESTONE
	6.1	Background
	6.2	Laboratory Experiments
		6.2.1 Introduction
		6.2.2 Methods
		6.2.2.1 Sample selection
		6.2.2.2 Equipment, procedures, and sampling program
		6.2.3 Results
		6.2.3.1 pH
		6.2.3.2 Sulphate
		6.2.3.3 Alkalinity
		6.2.3.4 Acidity
		6.2.3.5 Iron
		6.2.3.6 Metal leaching
	6.3	On-site Barrel Experiments
		6.3.1 Introduction
		6.3.2 Methods
		6.3.2.1 Sample selection
		6.3.2.2 Equipment, procedures, and sampling program
		6.3.3 Results
	6.4	Discussion
	6.5	Conclusion
7.0	WATE	R TREATMENT PLANT
	7.1	Introduction
	7.2	Bench Scale Treatment Tests
		7.2.1 Initial lime treatment tests
		7.2.2 High density precipitate system tests
		7.2.3 Caustic neutralization
		7.2.4 Confirmatory tests with lime
	7.4	Summary and Conclusions
8.0		TE OF STAGE II REPORT IMPACT ASSESSMENT 8-1
0.0	0 1	
	0.1	Comparison of Impacts Based on Banch coale and Bilet Mill Test Desuite
	0.2	Compansion or impacts based on bench-scale and Pilot Mill Test Results 8-1
		0.2.1 Dauryiuuiu 8-1

v

		8.2.2 Assessment of effects on Florence Creek	8-3
		8.2.3 Assessment of effects on Barbie Creek	8-10
	8.3	Water Treatment Plant	8-11
	8.4	Summary	8-16
9.0	TEMP	ORARY MINE SHUTDOWN AND PREMATURE CLOSURE	9-1
	9.1	Introduction	9-1
	9.2	Temporary Shutdown	9-1
	9.3	Permanent Early Closure	9-2
		9.3.1 Permanent closure: year 3	9-2
		9.3.1.1 Project facilities	9-2
		9.3.1.2 Reclamation activities	9-4
		9.3.2 Permanent closure: year 7	9-5
		9.3.2.1 Project facilities	9-5
		9.3.2.2 Reclamation activities	9-6
10.0	EXPA		10-1
	10.1	Introduction	10- 1
	10.2	Premature Closure or Temporary Mine Shutdown	10-1
	10.3	Reclamation Funding	10-1
	10.4	On-going Environmental Monitoring and Sampling	10-2
	10.5	Training	10-2
11.0	ERRA	TA AND OMISSIONS	11-1
	11.1	Volume I	11-1
		11.1.1 Errata	11-1
	11.2	Volume II	11-2
		11.2.1 Errata	11-2
		11.2.2 Omissions	11-5
	11.3	Volume II Appendices	11-5
		11.3.1 Errata	11-5
	11.4		11-6
		11.4.1 Errata	11-6
		11.4.2 Omissions	11-7
	11.5	Volume III Appendices	11-10
		11.5.1 Errata	11-10
		11.5.2 Omissions	11-10
	11.6		11-11
	_	11.6.2 Omissions	11-15
	11.7		11-16
		11.7.1 Errata	11-16
	11.8		11-18
		11.8.1 Errata	11-18
			11-18
	11.9	Volume V Appendices	11-19

.

 11.9.1 Errata
 11-19

 11.9.2 Omissions
 11-19

 11.10 Volume VI
 11-20

 11.10.1 Errata
 11-20

 11.11 Volume VI Appendices
 11-22

 11.11.1 Errata
 11-22

 REFERENCES
 11-22

LIST OF TABLES

Table		Page
3.3.1-1	Ore Samples Received by Hazen Research from City Resources for Pilot Plant Program, February 1988, Cinola Gold Project	3-9
3.3.1-2	Type and Composition of Ore Used in Pilot Plant Program, February, 1988, Cinola Gold Project	3-10
3.3.2-1	Detection Limits Used for Analysis of Solids and Liquids from the Pilot Plant Program, Norecol Test Program, Cinola Gold Project	3-16
3.4-1	Summary of Tailings Liquid Characterization by Norecol from Pilot Plant Program for the Cinola Gold Project	3-20
3.4-2	Summary of Tailings Gold Characterization from Pilot Plant Program by Norecol for the	3.00
3.4.2-1	Leachate Water Quality Analysis from Short-term Leach Tests of Tailings Solids ^a by	0 22
3.4.2-2	Leachate Water Quality from Norecol's Long-term Leach Tests of Tailings after	3-30
3.4.2-3	Sodium Sulphide Addition, Cinola Gold Project	3-33
3.5-1	Program, Cinola Gold Project	3-37
3.5-2	Project	3-38
3.5-3	Program ^a to Bench Scale Testwork ^b for the Cinola Gold Project Ratios Calculated by Norecol to Compare Leachate Water Quality from Short-term	3-40
3.5-4	Cinola Gold Project	3-41
352-1	Gold Project	3-42
0.0.2-1	Bench Scale Testwork ^b , Cinola Gold Project	3-48
3.6.2-1	Slurry by Sodium Sulphide Addition, Cinola Gold Project	3-49
3.6.2-2	Results of Tests by Hazen Research on Mercury Removal from Final SO ₂ /Air-treated Slurry by Sodium Sulphide Addition, Cinola Gold Project	3-50
3.7.1-1	Summary of H2O2 Cyanide Destruction Tests by Hazen Research, Cinola Gold Project	3-52
3.7.2-1	Results of Optimization Studies for SO ₂ /Air Process for Tailings from the Pilot Plant Program, May 1988, Cinola Gold Project	3-54

viii

LIST OF TABLES (continued)

Table		Page
3.7.2-2	Characteristics of Tailings Liquid After Cyanide Destruction, Optimized SO ₂ /Air Process, Pilot Plant Test Program, May 1988, Cinola Gold Project	3-55
5.5-1	Physical and Chemical Properties of Tailings for the Cinola Gold Project: A	
	Comparison of Stage II Values and Laboratory Tests Results	5-4
6.2.2-1	Waste Rock Composition of Rock Sub-group for the Limestone Columns, Cinola Gold Project	6-3
6.2.2-2	Grain Size Distribution for the Rock Mixture and Limestone used in Limestone Columns, Cinola Gold Project	6-4
6.2.2-3	Results of Acid-base Accounting for the Rock Mixture and Limestone used in the Limestone Columns, Cinola Gold Project	6-5
622-4	Bock-I imestone Proportions in the Limestone Columns. Cinola Gold Project	6-8
0.2.2	Application Decemptors Managured in Effluent from Limestone Columns by Noracol for	
6.2.2-5	the Cinola Gold Project	6-10
6.2.3-1	Sulphate Loadings (g) Calculated by Norecol from Limestone Columns, Cinola Gold Project	6-14
6.2.3-2	Average Weekly Sulphate Production Rates from Limestone Columns, Cinola Gold Project	6-15
6.2.3-3	Figures Depicting Metal Concentrations vs pH in Stage II Volume V, by Norecol for the Cinola Gold Project	6-25
6.3.2-1	Acid-base Accounting of the Rock Fills for the Norecol Limestone Barrel Experiments, Cinola Gold Project	6-32
632-2	Rock: Limestone Proportions in Norecol Limestone Barrel Experiment, Cinola Gold	
	Project	6-34
7.2-1	Estimated Average Concentrations of Influent to the Mine Site Water Treatment Plant at the Cinola Gold Project	7-2
7.2-2	Estimated Maximum Concentration of Influent to the Mine Site Water Treatment Plant at the Cinola Gold Project	7-3
7.2-3	Analysis of Supernatant from Lime and Settling Tests on Acid Drainage Sample from Waste Rock at the Cinola Gold Project ^a	7-5
7 2-4	Analysis of Filtrate from Line and Settling Tests on Acid Drainage Sample from Waste	
705	Rock at the Cinola Gold Project ^a	7-6
1.2-0	Acid Drainage Sample from Waste Rock at the Cinola Gold Project	7-8
7.2-6	Analysis of Filtrate and Supernatant from Caustic Neutralization, Cinola Gold Project	7-10

LIST OF TABLES (concluded)

Table		Page	
7.2-7	Analysis of Filtrate and Supernatant from Lime and Settle Tests - Bucket #1, Cinola Gold Project	7-12	
7.2-8	Analysis of Filtrate and Supernatant from Lime and Settle Tests - Bucket #2, Cinola Gold Project	7-13	
7.2-9	Analysis of Supernatant from Lime and Settle Tests - Bucket #3, Cinola Gold Project	7-14	
7.2-10	Expected Levels of Contaminants in Effluent from the Mine Site Water Treatment Plant at the Cinola Gold Project	7-16	
8.2-1	Comparison of Mill Effluent Characterization Conducted by Norecol Based on May 1988 Pilot Mill and February 1988 Bench Scale Tests	8-4	
8.2-2	Year 5 Water Quality of Florence Creek Based on Discharge from Reclaimed Impoundment 1, Predicted by Norecol for the Cinola Gold project from Pilot Plant Test Data	8-6	
8.2-3	Year 9 Water Quality of Florence Creek Receiving Outflow from Reclaimed Impoundments 1 and 2, Predicted by Norecol for the Cinola Gold Project from Pilot		
8.2-4	Plant Test Data	8-7	
8.2-5	Water Quality of Florence Creek for Extreme Wet and Dry Flow Conditions, at Closure, predicted by Norecol for the Cinola Gold Project from Pilot Plant Program Data	8-9	
8.2-6	Nitrate Concentrations (mg N/L) in Outflow from the Reclaimed Pit and Resultant Concentrations in Barbie Creek, Predicted by Norecol, Cinola Gold Project	8-12	
8.3-1	Water Quality of Barbie Creek After Receiving Effluents from the Lime Treatment Facility and Waste Rock Stockpile Underdrain, Predicted by Norecol for the Cinola		
8.3-2	Gold project	8-13	
		0-15	

LIST OF FIGURES

Figure		Page
3.3-1	Cinola Pilot Plant Operation 3, Campaign 8, Block Flowsheet	3-4
3.3-2A	Cinola Pilot Plant Process Flowsheet, Operation 3, Campaign 8	3-5
3.3-2B	Cinola Pilot Plant Process Flowsheet, Operation 3, Campaign 8	3-7
3.3.3-1	Sample Preparation Flowsheet	3-12
6.2.2-1	Schematic Diagram of Limestone Columns	6-7
6.2.3-1	pH vs Time for Columns 1 to 5	6-11
6.2.3-2	SO4 vs Time for Columns 1 to 5	6-13
6.2.3-3	SO4 Production vs CaCO3 Production	6-18
6.2.3-4	Weekly Alkalinity Loading vs Time for Limestone Columns	6-19
6.2.3-5	Cumulative Carbonate Consumption vs Time	6-21
6.2.3-6	Acidity vs Time for Limestone Columns	6-22
6.2.3-7	Cadmium vs pH for Limestone Columns	6-26
6.2.3-8	Mercury Concentrations vs pH for Limestone Columns	6-27
6.2.3-9	Selenium Concentrations vs pH for Limestone Columns	6-29
6.2.3-10	Silver Concentrations vs pH for Limestone Columns	6-30
6.3.2-1	Schematic Diagram of Limestone Barrels	6-33
6.3.3-1	pH vs Time for Limestone Barrels	6-36
8.2-1	High West Area General Arrangement - Years 5 to 8	8-1

xi

LIST OF APPENDICES

Appendix

- 3.3-1 Description of Metallurgical Samples During Pilot Plant Program, May 1988, Cinola Gold Project
- 3.4-1 May 1988 Pilot Plant Program for the Cinola Gold Project
- 3.5-1 February 1988 Bench Scale Test Program for the Cinola Gold Project
- 3.7-1 Cyanide Destruction Program Using Samples from the May 1988 Pilot Plant Program for the Cinola Gold Project
- 5.1 Sample Log and Test Schedule for Laboratory Testing of May 1988 Pilot Mill Tailings for the Cinola Gold Project
- 5.2 Grain Size Analyses, Specific Gravity and Atterberg Limit Test Results from May 1988 Pilot Mill Tailings for the Cinola Gold Project
- 5.3 Column Settled Density and Hydraulic Conductivity Test Results from May 1988 Pilot Mill Tailings for the Cinola Gold Project
- 5.4 Consolidation Test Results from May 1988 Pilot Mill Tailings for the Cinola Gold Project
- 5.5 Gypsum Content Analytical Report from May 1988 Pilot Mill Tailings for the Cinola Gold Project
- 6.2.3-1 Limestone Column Laboratory Experiments for the Cinola Gold Project
- 6.2.3-2 Correction of Alkalinity Data to Reflect True Bicarbonate Concentration
- 6.3.3-1 Limestone Barrel Experiments at the Cinola Gold Project Site
- 7.2 Experimental Procedures and Analytical Results for Bench Scale Test for Mine Site Water Treatment Facility - Analysis Performed on Acidic Wastewater from the Cinola Gold Project
- 8.3-1 Receiving Water Quality Predictions for Extreme Flow Conditions Using May 1988 Pilot Mill and Bench Scale Mine Site Water Treatment Plant Results

SECTION 1.0

1.0 INTRODUCTION

1.1 Transmittal Statement

The Stage II Report was submitted in June 1988 in compliance with the requirements of the Mine Development Steering Committee. The Stage II Report was submitted in support of the request by City Resources (Canada) Limited to proceed with development of the Cinola Gold Project. At the time of submittal of the Stage II Report, a number of test programs relevant to the environmental assessment were still in progress.

The Stage II Addendum Report has been prepared to present the results of the test work that was not available for inclusion in the Stage II Report, as well as submitting a description of different scenarios for premature or early closure of the property.

1.2 Testing Programs

The testing programs reported in this Addendum Report cover four major areas, and include:

- a) pilot scale mill process tests and characterization of mill effluent;
- b) tests to characterize the pilot mill tailings;
- c) waste rock liming tests; and
- d) additional treatment tests of acidic water.

1.3 Report Format

The Stage II Addendum Report for the Cinola Gold Project is presented in a single volume. Each of the major areas of testing is discussed in a separate section and supporting data are included in the Appendices attached to the report.

The Stage II Addendum Report presents a description of the upgraded mill process and provides an assessment of the potential environmental impacts that could be anticipated based on the results of the new test data. A discussion of premature closure scenarios is also included to describe the appropriate sequence of reclamation procedures to be implemented in the event of a mine closure prior to the scheduled ultimate life of the mine.

A final section in the Stage II Addendum Report is included to provide errata for the June Stage II Report.

1.4 Responsibility

Corporate responsibility for the Stage II Addendum Report was assumed by Peter Cowdery, P. Eng. on behalf of City Resources (Canada) Limited.

Co-ordination of the activities of the contributors to the report was provided by Robinson Dames & Moore. Norecol Environmental Consultants Ltd. prepared the Stage II Addendum Report for publication.

Technical responsibilities for the preparation of the Stage II Addendum Report were as follows:

Hazen Research Inc.

• Pilot mill data

Minproc (U.S.A.) Inc.

• Mill process update.

Norecol Environmental Consultants Ltd.

Mill effluent characterization, liming experiments with waste rock, environmental impact assessment.

Steffen Robertson & Kirsten

• Tailings characterization, acidic water treatment studies, premature closure scenarios.

SECTION 2.0

.

2.0 EXECUTIVE SUMMARY

2.1 Introduction

At the time of submission of the Stage II Report, several tests were still in progress to obtain information relevant to the environmental assessment. This Stage II Addendum Report has been prepared to present the test results not included with the Stage II Report, as well as to document changes in the mill process and describe different scenarios for premature mine closure.

The results of tests presented in this Addendum Report cover four major areas, including:

- a) pilot scale mill process tests and characterization of mill effluent;
- b) laboratory tests to characterize the pilot mill tailings;
- c) laboratory column tests and in situ barrel tests to evaluate waste rock liming; and
- d) additional treatment tests of acidic water.

2.2 Mill Effluent

Results from bench scale testing were used in the Stage II Report to identify areas of concern related to the mill effluent and to assess the impact of the proposed Waste Management Plan on the environment. The differences in the operation of the May, 1988 pilot plant as compared to the earlier bench scale tests reported in Stage II include: addition of sulphuric acid to the circuit, continuous operation, and cyanide destruction using the SO₂/air process as well as hydrogen peroxide. During the earlier bench scale tests, all stages were conducted as batches.

The comparison of the results from the pilot mill and bench scale tests indicated that although there were some differences, the trends and characteristics of the effluents from the two stages of testing were very similar. Some differences in slurry characteristics resulted from the addition of sulphuric acid before nitrate oxidation, a step which caused oxidation of more minerals.

Acid-base accounting results for the final tailings for both test programs were similar, with a slight increase in sulphide content of the pilot plant tailings slurry (0.13% compared to 0.07%).

Leaching characteristics of the tailings solids after cyanide destruction using the SO₂/air process indicated very similar trends at pH 8.0 (short-term leach) and 8.5 (long-term leach) with greater variability in lower pH leaching solutions. The variability at the lower pH's were believed to be due to increased oxidation and differences in cyanide destruction processes.

Mercury treatment tests were performed by Hazen Research Inc. using sodium sulphide addition. These demonstrated that mercury was removed from the liquid phase to levels below 1 ug/L with the addition of 4.8 mg Na₂S/L of liquor in the SO₂/air process slurry.

2-1

The comparison of the two methods of cyanide destruction indicated that the SO₂/air process produced lower total cyanide and weak acid dissociable cyanide after cyanide destruction, and lower metal concentrations following sodium sulphide addition than does the hydrogen peroxide method. Therefore, the SO₂/air process produces a more environmentally favourable effluent using the Cinola ore.

The impact of tailings disposal was reviewed in light of the predicted tailings effluent quality as determined from the pilot mill tests. The pilot mill tests resulted in lower effluent concentrations of cyanide, copper, and mercury, and slightly higher concentrations of nitrate and aluminum than the bench scale tests. The lower cyanide and copper levels do not rely upon natural cyanide degradation in reclaimed Impoundments Nos. 1 and 2 to meet receiving water criteria in Florence Creek, as was indicated in the Stage II Report. The higher levels of nitrate in the mill effluent of the pilot plant will result in higher nitrate levels than was previously predicted in Barbie Creek (due to transfer of Impoundment No. 3 water to the reclaimed pit) as well as in Florence Creek. However, because of phosphorus limitation, the higher nitrate concentrations will not cause detrimental algal growth. The aluminum concentration in Florence Creek will increase slightly, but the increase will be impossible to distinguish from background variability and therefore is not considered an impact on water quality.

2.3 Process Update

The major changes to the mill processes subsequent to the submission of the Stage II Report involve neutralization, cyanide leaching and adsorption, gold elution and recovery, cyanide destruction and water recycling. The main reason for the changes is a more cost-effective design.

The neutralization process has been augmented with the addition of slurry coolers. The added stages are required to reduce the slurry temperature from 80°C to 60°C and to increase the slurry pH to in excess of 9.5 in preparation for cyanide leaching.

The total number of tanks in the cyanide leach circuit has been reduced from ten to nine. The circuit has also been modified to a hybrid Carbon-in-pulp/Carbon-in-leach system which results in a shortened total residence time in the circuit from 24 hours to 18 hours.

The gold elution and recovery system has been made larger, to handle fewer, but larger, batches of carbon. The batch capacity has been increased to 5.2 t with the same cycle time as the smaller circuit previously considered. The design offers improved economics over the previous method.

The SO₂/air system has been selected over hydrogen peroxide as the cyanide destruction process. The selection was based on both environmental and economic considerations. In both areas, the SO₂/air process was found to be superior.

The water usage in the plant has been modified to eliminate the need for treatment of water recycled from the tailings pond. An additional 3 m³/hour of fresh water for make-up is estimated based on a recalculation of the overall water balance. The low fresh water demand has been achieved by

minimizing the need for fresh water in the plant. With the proposed modifications, the water balance still conforms to the zero effluent discharge design of the tailings impoundments.

A further refinement involves the siting of a lime calcining plant at Ferguson Bay. Rather than barging the lime to the Queen Charlotte Islands, it is now proposed that an independent supplier process the material locally.

2.4 Tailings Properties

Tests were undertaken on tailings from the pilot mill program to compare the properties of the tailings produced from the pilot mill with properties used in the Stage II Report. This comparison showed that the tailings parameters used in the Stage II analyses were essentially correct and would not require adjustment of the water balance predictions. Pilot mill tailings characterization also showed that revision of the tailings impoundment storage capacity estimates is not required.

The average hydraulic conductivity was found to be 10^{-7} m/sec which is within the range used for the earlier water balance estimates and therefore no changes to the water balance calculations are indicated.

2.5 Waste Rock Liming Experiments

In order to evaluate the effectiveness of limestone in the abatement of acid drainage from the proposed waste rock stockpile, laboratory-based and on-site experiments were carried out. Ratios of limestone: waste rock varying from zero to 6.6% were tested based on estimates of limestone quantities theoretically required to neutralize all acidity generated in the waste rock.

Five laboratory-based column experiments provided information on the neutralization of pH and the regulation of acid generation in samples of the waste rock by the addition of finely ground limestone. Through the 24-week duration of the tests, the limestone was observed to maintain pH in the range of 6.8 to 8.2. The limestone thoroughly neutralized acidity from pyrite oxidation. No significant differences in neutralization were observed as a result of the different limestone proportions. Compared to the control column, to which no limestone was added, the limestone also decreased the rate of acid generation by a factor of almost 19 and minimized metal concentrations in the effluents through limitation of acid generation and pH control.

Based on the column experiments, the proposed 50 000 t limestone addition to the waste rock stockpile is capable of neutralizing acidity for about 25 years.

On-site barrel tests were undertaken to test the effectiveness of limestone neutralization of acid generation in waste rock under field conditions.

It was found that the design of the on-site tests using plastic barrels precluded the circulation of air and did not simulate the waste rock stockpile. The exclusion of air from the lower parts of the barrels, together with the low hydraulic conductivity of the waste rock samples utilized, contributed to reducing conditions in the barrels. This allowed ferrous iron to be flushed from the samples in the leachate. Subsequent oxidation and precipitation of ferric hydroxide in the tubing and collection buckets of the test apparatus generated low pH drainage water. These conditions will not exist in the free-draining stockpile where oxygen will be present and water infiltration will be limited.

The barrel tests were designed to evaluate the effects of layering of waste rock and limestone; low, uncontrolled flow rates; and potential encapsulation of the limestone fragments. Until the development of reducing conditions, the tests demonstrated the success of limestone addition in controlling acid generation. Further, it was also observed that the limestone lowered the rate of acid generation.

Examination of the limestone distributed throughout the barrels indicated that the limestone remained highly reactive, was able to neutralize acidity generated by the waste rock and had not become encapsulated during 3 months of testing.

2.6 Waste Water Treatment

The recent bench scale tests carried out to assess the water treatment process for the mine site confirmed the water quality results obtained in the tests reported in the Stage II submission. The effluents from the three sets of bench tests indicated excellent removal efficiencies for the parameters analyzed. Metals precipitation with caustic soda was found to be less efficient than precipitation with lime.

The new data confirm the validity of the previous water quality tests and are below the levels normally attainable by commercial treatment plants. The previously reported data represent limits that are attainable by commercial treatment processes.

The bench scale water treatment tests provided predictions of total metal levels in the treatment plant effluent. These values, which were slightly higher than the dissolved metal concentrations used in the Stage II Report, were used to reassess the potential impacts of treatment plant effluent on receiving waters. Because of removal of particulate metals as the effluent passes through Wetland MSW1, and a predicted increase in hardness of Barbie Creek, increased effluent metal concentrations will be counteracted such that all metals will meet receiving water criteria, except in those instances where background concentrations already exceed criteria. Dissolved aluminum concentrations are predicted to increase slightly, but the increase will be impossible to distinguish from background variability.

2.7 Closure Scenarios

The Reclamation Plan presented in the Stage II Report outlined the procedures and sequence of activities that would be taken in preparation for, and after, mine closure. The plan was predicated on closure occurring at the end of the scheduled life of the mine - at least 12 years after start of production. Under certain circumstances, temporary shutdown or premature, permanent closure could occur before the scheduled site closure. An assessment of possible premature mine closure

scenarios and the required responses for reclamation has been completed and is discussed in this Addendum Report. Premature or ultimate closure of the mine site can be accomplished in an environmentally acceptable manner.

The conditions considered included temporary shutdown; permanent closure between Years 1 and 4; and, permanent closure between Years 5 and 8. The conditions at closure after Year 8 would be similar to those at ultimate mine life and were not discussed as a separate scenario. For each scenario, the prevailing conditions are described for the High West and the mine area. Details for disposal of accumulated water, control of drainage, disposal of waste rock and reclamation of the impoundments and the open pit are discussed.

Reclamation for premature closure requires no activities or procedures that have not already been considered in the Reclamation Plan. However, the schedule and sequence of implementation of the procedures would be altered to suit the conditions at the time of curtailment of mining operations.

2.8 Commitment

City Resources (Canada) Limited will undertake the commitments described in the Stage II Report as part of the project development. The details outlined in the Reclamation Plan would be implemented as scheduled throughout the life of the project, or would be adjusted to accommodate premature closure of the mine. The Company will set aside the funds for completing the site reclamation, most likely in the form of a Trust Fund.

In discussion with local groups, an independent body will be set up to carry out the environmental monitoring and sampling required throughout the life of the project.

City Resources will adhere to the recruitment and training policies outlined in the Stage II Report to ensure that the opportunities presented by the Cinola Gold Project are fully available to the local residents.

SECTION 3.0

3.0 MILL EFFLUENT CHARACTERIZATION

3.1 Introduction

Stage II waste management plans for the High West tailings/waste rock impoundment were described in Section 5.0, Volume II and Section 2.1, Volume IV of the Stage II Report. The most significant waste that will be generated in the mill will be tailings slurry which is comprised of process water and ground rock.

As described in Stage II, tailings slurry will be deposited in the High West tailings/waste rock impoundment, which will also contain argillically altered and potentially acid generating waste rock from the pit, and precipitate from the mine site water treatment facility (Section 2.1.3, Volume IV). The proposed impoundment system will be subdivided into three distinct impoundments, each of which will be operated for approximately four years. The impoundment system will be operated sequentially from upstream to downstream. Process water will be recycled from the active impoundment to the mill such that no surface water will be discharged from the active impoundment to the next active impoundment is deactivated, the supernatant will be drained to the next active impoundment and the impoundment area will be reclaimed. Water from the last active impoundment (Impoundment No. 3) will be pumped to the mined out pit.

City Resources (Canada) Limited has chosen a nitrate oxidation/cyanidation process for extracting gold from the ore. The process was modelled at bench scale in February 1988 in a program which consisted of sequential batch preparation of each stage of the mill process. Sampling was conducted at each stage of the process from ore to final tailings slurry and the results of this sampling program were reported in the Stage II Report (Appendix 2.1.3-1, Volume IV).

The process was subsequently run at pilot plant scale in May 1988 under continuous operating conditions for the various stages. Sampling was conducted during each stage of the process. The May 1988 program also included an assessment of two different cyanide destruction methods in order to select the most appropriate method for the Cinola Gold Project. Successful mercury removal had been demonstrated at bench scale in preliminary mercury treatment tests using sodium sulphide addition during the February 1988 metallurgical test program. Additional bench scale mercury removal experiments were conducted in conjunction with the May 1988 pilot program to refine the proposed mercury removal parameters. Results of the May 1988 pilot plant tests form the basis of the updated tailings characterization provided in this Addendum Report.

The tailings characterization programs conducted in February 1988 and May 1988 included characterization of the chemistry of both solid and liquid phases of the tailings slurry at various stages, as well as an assessment of the stability of the tailings solids under various environmental conditions to define short- and long-term conditions in the tailings/waste rock impoundment. The objectives of the mill effluent characterization program have been to (1) indicate the potential water quality in the tailings/waste rock impoundment during mine operation and after closure, (2) identify potential metals of concern in the effluent, and (3) examine removal of mercury by the addition of sodium sulphide.

This section of the Addendum report presents an overview of the May 1988 pilot plant operation and the sampling program. In addition, it presents the results of the updated tailings characterization based on the May program, and compares these results with those reported in the Stage II Report.

3.2 Summary of Metallurgical Process

The metallurgical process proposed for the Cinola ore will involve various stages to recover gold and to treat the tailings to ensure environmental acceptability. The process is described in detail in Section 5.0, Volume II of the Stage II Report with subsequent process updates described in Section 4.0 of this Addendum Report. In summary, the proposed metallurgical process will involve crushing and grinding the ore; nitrate oxidation, followed by neutralization; and cyanidation in a carbon-in-leach (CIL) circuit. Gold will be stripped from the carbon, electrowinned and smelted. The tailings slurry will undergo cyanide destruction by the SO₂/air process with the precipitation of mercury from the tailings liquid to an insoluble form by sodium sulphide addition to the slurry prior to discharge to the impoundment. (A detailed mercury balance is presented in Section 4.0 of this Addendum Report.)

3.3 Pilot Plant Program - Objectives and Responsibilities

In January 1988, City Resources (Canada) Limited authorized Hazen Research, Inc. (Hazen) to proceed with a third phase (Operation 3) of pilot testing of the Nitrate Oxidation Process on Cinola ores at their Denver research facility. Preparatory work for the 1988 pilot plant runs began in January, with construction and operation of the 1350 kg/d plant from March to May 1988. The results of the pilot plant allow for refinement of the proposed metallurgical process for the Cinola ore.

The overall objectives of the pilot program were to demonstrate operability of the process, to generate sufficient metallurgical and engineering data to allow for detailed design and feasibility studies, and to produce process effluent for environmental evaluation. The specific objectives for the pilot operation were as follows:

- a) To demonstrate the technical feasibility of the continuous Nitrate Oxidation Process for pretreatment of Cinola ores so that the gold can be recovered by conventional cyanidation;
- b) To establish the recovery rates of gold and silver;
- c) To establish accurate material balances throughout the principal flowsheet circuit;
- d) To establish the effectiveness of pyrite concentrate additional to reduce residual nitrate in solution;
- e) To evaluate procedures for returning from upset conditions to normal operation;

3-2

- f) To obtain sufficient metallurgical, chemical, and engineering data to allow completion of the feasibility study, and to permit smooth transition to the detailed engineering phase; and
- g) To provide process effluent for characterization and environmental evaluation.

From April 11 to May 6 1988, the nitric acid oxidation pilot plant was operated in eight runs, or campaigns, to assess the application of this process to the Cinola ore. The final run (Campaign 8) provided a demonstration run for the process and incorporated design information obtained during the previous runs. Operating conditions were selected which had previously demonstrated the lowest concentrations of gold in the tailings after the CIL extraction, and had resulted in effluent nitrate concentrations of 1.5 g/L (340 mg N/L) or less. The Campaign 8 demonstration run was the longest and provided the feed for a continuous CIL circuit which operated during May 5 to 26, 1988.

Conditions and results of the Campaign 8 run serve as criteria for the feasibility study, and this run provided the samples for environmental evaluation. The block flowsheet for Campaign 8 is shown in Figure 3.3-1. A process flow sheet for the nitrate oxidation and neutralization stages is provided in Figure 3.3.2A and B.

Responsibilities for the pilot plant test program were as follows:

- Minproc (U.S.A.) Inc. acted as overall pilot program co-ordinator;
- Hazen carried out the pilot testing (except for the SO₂/air cyanide destruction process);
- Inco Ltd. conducted the cyanide destruction testwork for the SO₂/air process;
- Steffan Robertson Kirsten Ltd. (SRK) was responsible for the direction of the mercury removal test program;
- Norecol Environmental Consultants Ltd. designed the environmental test program, carried out sampling, and interpreted the environmental test program results. Norecol also prepared Section 3.0 of the Stage II Addendum Report, based on material provided by Hazen (Section 3.3, 3.6, and 3.7.1) and material provided by Inco (Section 3.7.2).

3.3.1 Pilot Plant Operation

3.3.1.1 Ore selection

An 18-t sample of Cinola ore was used for the pilot tests. The ore sample was a composite of ore types A, B, and C, identified by City Resources as silicified conglomerate, hydrothermal breccia, and fine-grained silicified sediments, respectively. The samples were received by Hazen in February 1988 in fifty-eight 55-gal drums. Particle size of the as-received samples ranged from about 1.3 to 35.6 cm. The number of drums and approximate weights of each type of ore (A, B, and C) are shown in Table 3.3.1-1. The composition of the feed ore is given in Table 3.3.1-2.







Ore Samples Received by Hazen Resea	Samples Received by Hazen Research from City Resources for Pilot Plant Program,		
February 19	February 1988, Cinola Gold Project		
ORE TYPE	No. OF	WEIGHT	WEIGHT
	DRUMS	(kg)	(%)
A (silicified conglomerate)	28	8900	48.3
B (hydrothermal breccia)	20	6360	34.5

10

58

3180

18440

17.2

100.0

TABLE 3.3.1-1

Source: Hazen Research Inc.

Total

C (fine-grained silicified sediments)

-

(in the second

February 1988, Cinola Gold Project			
PARAMETERS	UNITS	FEED ORE	
Ore type		A, B, C Composite	
Grind		80% passing 325 mesh	
Gold	g/t	4.21	
	oz/t	0.123	
Silver	g/t	3.4	
	oz/t	0.10	
Total sulphur	%S	1.42	
Sulphide	%S	1.15	
Sulphate	%S	0.26	
Iron	%	2.35	
Copper	ppm	50	
Arsenic	ppm	219	
Selenium	ppm	2.2	
Mercury	ppm	4.3	

0.02

%

TABLE 3.3.1-2Type and Composition of Ore Used in Pilot Plant Program for Environmental Run,
February 1988, Cinola Gold Project

Source: Hazen Research Inc.

Carbonate

3.3.1.2 Ore preparation

The ore sample preparation flowsheet is presented in Figure 3.3.1-1. The entire 18-t sample was crushed in a primary 10- by 20-in Denver jaw crusher and secondary 2-ft Symons short head cone crusher to a nominal 1 cm particle size, and blended by coning and quartering with a Bobcat on a concrete pad. Head samples were then split from the blended sample.

3.3.1.3 Nitrate oxidation stage

The nitrate oxidation stage used four tanks with an overall retention time of 102 min. Nitric acid was added to the slurried ore at a stoichiometric ratio of 0.92 (nitrate to sulphide), and 93% sulphuric acid was added at 21.5 kg/t ore. A nitrogen stripper was used after nitrate oxidation for removing entrained or dissolved nitric acid.

3.3.1.4 Neutralization stage

The continuous neutralization circuit allowed a total of 15 min for limestone addition during flow through the system, followed by 3 min for lime addition. The limestone tanks were kept at 80° C and the lime tanks at 60° C. At the end of the circuit, a tank equipped with a cooling water coil cooled the product slurry for storage.

Texada limestone was slurried at 25% solids and added to the first two of the four limestone tanks (4.1 L) at a constant rate. Texada lime was added at 10% solids to the three lime tanks (1.7 L). Initially, the lime slurry flow rate was controlled with pH controllers with the objective of achieving pH 6.0, 8.0, and 9.5 in the first, second, and third tanks, respectively. However, due to the low surge capacity of the lime tanks with a retention time of 1 min each, the pH in these tanks varied widely. The pH controllers were removed, and a constant flow into each tank was established, which gave approximately the required pH values.

3.3.1.5 Cyanidation

The continuous Carbon-In-Leach (CIL) circuit simulated a commercial plant and was operated to obtain the following: (1) carbon loading data, (2) samples of leach tailings for cyanide destruction tests, and (3) samples of leach tailings for detailed environmental analyses. The circuit was run using ten flow-through tanks in series and starting with fresh, pre-attritioned carbon; the scope of testing did not include provisions for moving carbon.

The scope of testing included at least 10 days of continuous operation to achieve equilibrium conditions for the carbon loading. The equilibrium was determined by assaying the gold in the barren tailings liquor from the last three stages of the cyanidation circuit and defining a value of 0.02 mg Au/L as the point of breakthrough.



Neutralized slurries obtained from the neutralization stage of the pilot plant operations on May 4 and 5, 1988, were used as feed to the continuous CIL circuit. The neutralized slurries were stored in plastic barrels at ambient temperatures prior to use in the continuous CIL circuit. Handling the slurries was facilitated by mechanically mixing the slurries and removing 5 gal portions, which were then transferred to a 15 gal plastic storage tank. The material in this tank was agitated, and a tubing pump was used to continuously transfer slurry to a 5-gal feed tank. The slurry in this tank was maintained at 40° C, mechanically agitated, and sparged with air at the rate of 1 L air/min.

Slurry from the feed tank was pumped to the first leach stage at the rate of 33 mL/min, which corresponded to a retention time of 1.8 h per stage. A milk-of-lime solution containing 20 g Ca(OH)₂/L was metered to the first stage to maintain pH 10.5. A cyanide solution containing 10 or 20 g NaCN/L was metered in equal amounts to flow-through Tanks 1 and 3. During later periods of operation a small amount of cyanide was added to Tank 5.

The circuit was monitored by measuring the pH, temperature, dissolved oxygen, and cyanide concentrations every 3 h. The slurry and reagent feed rates were checked at the same intervals. The gold concentrations in the leach liquors for various stages were monitored less frequently. The resulting leach tailings were collected in 6-h composites.

3.3.1.6 Cyanide destruction

Cyanide destruction tests were performed on tailings slurry from the continuous CIL using two methods, hydrogen peroxide and SO₂/air. The hydrogen peroxide tests were performed by Hazen personnel. The SO₂/air tests were done at Hazen by Inco Limited personnel.

Hydrogen peroxide

A short continuous run was made to demonstrate the hydrogen peroxide process and produce samples for environmental analyses. Three 1700-mL stirred reactors in series were used for this test. Slurry was fed to the system at 50 mL/min for a total retention time of about 100 min. A 50% solution of H₂O₂ was combined with the feed stream at 1.7 and 2.3 mL/kg of slurry. Mechanical constraints prevented lower peroxide additions.

SO₂/air

A 5.6-L flotation cell was used for the cyanide removal test for the environmental sample. The cells were equipped with pH and oxidation reduction potential probes.

Feed pulp was metered by a pump at the desired rate into the cells. Agitation in the cells was 1000 rpm with air introduced at 1 L/min/L. Sulphur dioxide (SO₂) was introduced as a solution of Na₂S₂O₅ in water, which was metered at a set rate. The pH was controlled in the reaction cells using H₂SO₄ addition to a CuSO₄ solution. This solution was also metered into the reaction cell.

3.3.1.7 Mercury removal by sulphide addition

Slurry was collected after cyanide destruction from both cyanide destruction methods, and the samples were separately treated with sodium sulphide. Based on the results of the May 1988 bench scale optimization program performed at Hazen (Section 3.6), each slurry was treated with 24 mL of 0.19 g Na₂S/L in 1 N NaOH per litre of slurry and then mixed slowly for 20 min.

3.3.2 Sampling program

The mill effluent characterization sampling program was carried out at Hazen from May 3 to 17, 1988, during the operation of the various stages of the pilot plant. There were two phases to the sampling program: Phase 1 included sampling during the pilot plant operation up to the end of the nitrate oxidation/neutralization stages; and Phase 2 included sampling during the cyanidation/cyanide destruction and mercury treatment circuits.

Phase 1 in the pilot plant program was during the piloting of the nitrate oxidation/neutralization. Samples of the ore, feed slurry, slurry after nitrate oxidation and after neutralization were taken during this time. Two ore samples were taken from the head ore splits before the grinding circuit. Each type of sample was composited over a period of time each day for three days that the pilot plant was running under optimum operating conditions. Samples of feed slurry from the feed tank were taken at two-hour intervals during the pilot plant run. Two samples of feed slurry, for each day, representing a four-hour time interval were composited. Samples after nitrate oxidation were taken at two-hour intervals from Tank 9 (Figure 3.3-2). Four samples, representing an eight-hour time interval, were composited each day. An additional grab sample was taken during the run. Samples after neutralization were taken at one-hour time intervals from Tank 17 (Figure 3.3-2). Four samples of neutralized slurry, representing a four-hour time interval grab sample was taken during the run.

Phase 2 involved sampling slurry before the cyanidation circuit, after the cyanidation circuit, before cyanide destruction, after cyanide destruction, and after sodium sulphide addition. The feed to the cyanidation circuit was selected by Hazen from barrels holding slurry produced under optimum conditions from the neutralization stage of the process. Only one barrel was used as feed during the several-day operation of the cyanidation stage, and three grab samples of the feed were taken on different days before cyanidation. The resulting slurry after cyanidation was collected in buckets where each bucket held six hours of slurry discharge from the cyanidation circuit. Samples for environmental analysis were taken from each bucket and were composited for each day (4 samples each day) during the time when the circuit was operating at optimum conditions. The slurry from all buckets was then composited by Hazen, mixed, and split into two batches (one for each cyanide destruction method). A sample of each composited batch was taken.

The two cyanide destruction methods were run on a continuous basis for several hours. Two grab samples were taken during the operation of each process. The slurry from each process was directed into containers and allowed to stand for over 30 h and a composite sample from each process was collected after the standing period. The standing period allowed time for the cyanide

3-14

destruction reaction to reach equilibrium. After an additional standing time of several days, the slurries from each process were treated by sodium sulphide addition. Samples were taken from each slurry before sodium sulphide addition and after sodium sulphide addition. The detailed list of samples taken is given in Appendix 3.3-1.

The set of parameters analyzed for each sample was dependent on the stage of the process. Liquids were analyzed at each stage, except for feed slurry and ore, to determine levels of metals, cyanide species (when relevant), nitrogen species, and major ions. Each liquid sample was split into three sub-samples: (1) a general sample stored at 4°C with no preservatives added; (2) a dissolved metals sample filtered through a 0.45 um filter and preserved with nitric acid; and (3) a mercury sample filtered through a 0.45 um filter and preserved with potassium dichromate and sulphuric acid. All sampling conducted following the cyanidation stage included a fourth unfiltered subsample for cyanide species which was preserved with sodium hydroxide.

Solid samples were characterized for metals content for each stage of the process, for acid generation potential in the feed and final tailings, and for grain size distribution of the final tailings. The solids samples were all washed with de-ionized water and dried at low temperature, with the exception of samples from cyanide destruction and sodium sulphide addition, which remained unwashed.

Liquid analyses were performed using standard methods (APHA 1985). Metals were analyzed using atomic absorption spectrophotometer, with graphite furnace for low detection limits. Solid component analyses were done using standard methods, including an inductive coupled argon plasma spectrophotometer and atomic absorption spectrophotometer. Acid-base accounting was performed using methods outlined by Sobek et al. (1978). Detection limits for each analysis are given in Table 3.3.2-1.

Short-term leach tests were carried out on unwashed slurry solids taken after the SO₂/air cyanide destruction stage and after the sodium sulphide addition stage. Representative subsamples were placed in bottles and leached for 24 h at pH 5 maintained with acetic acid, and pH 8 maintained with sodium hydroxide at a ratio of 42.5 g of solids to 850 mL of solution. Blanks of the initial solution at each pH level were analyzed. The pH was adjusted initially and after 1, 3, and 6 h by additions of analytical grade acetic acid, or sodium hydroxide. Throughout the 24-h leaching period, samples were agitated on a bottle-roll apparatus set at 110 rpm. Leachate was collected after the 24-h period by vacuum filtering the samples through a 0.45 um membrane filter. A portion of each sample was acidified with nitric acid and analyzed for metals by ICAP analysis. The remainder of the sample was analyzed for pH, Eh, conductivity, and sulphate.

Long-term leach tests at various pH levels were conducted on tailings solids from the SO₂/air process after sodium sulphide addition. A sample of the solids was rinsed with 1 L of de-ionized water before leaching. The equivalent of 500 g dry weight washed solids was placed in a plastic container with 10 L of solution. Two separate solutions at pH 5 and 7 were prepared using de-ionized water with analytical grade nitric acid and a pH 8.5 solution was prepared with analytical grade sodium hydroxide. The pH was adjusted initially, and after 1, 3 and 6 h. The pH was then
TABLE 3.3.2-1 Detection Limits Used for Analysis of Solids and Liquids from the Pilot Plant Program, Norecol Test Program, Cinola Gold Project

PARAMETER	SOLID/LIQUID	METHOD	DETECTION LIMIT
 рН	-	-	-
Conductance (umhos/cm)	L	Seibold Conductivity Meter	5
Alkalinity (mg CaCO ₃ /L)	L	Titration with 0.02 N H ₂ SO ₄	1
Sulphate (mg/L)	L	Turbimetric	0.005
Nitrate-N (mg/L)	L	Turbimetric	1
Nitrate-N (mg/L)	L	Colorimetric (Cadmium Reduction)	0.005
Ammonia-N (mg/L)	L	Indophenol method	0.005
Total cyanide (mg/L)	L	Colorimetric	0.001
Thiocyanate (mg/L)	L	Colorimetric	0.1
Weak Acid Dissociable Cyanide (mg/L)	L	Colorimetric	0.001
Cyanate (mg/L)	L	Colorimetric	0.02
Chloride (mg/L)	L	lon exchange	0.02
Eh (mV)	L	Ag/AgCI Reference Electrode	-
Paste pH	S	•	-
Sulphur (%)	S	Leco-IR Detector	0.001
Sulphate (% SO4)	S	Gravimetric-HCI leach	0.001
Sulphide (% S)	S	Gravimetric - HNO2 bromide digestion	0.01
Neutralization Potential (t CaCO ₃ /1000 t)	S	Titration	-
AL (%)	S	ICAP ^a	0.01
Ag (ppm) ^e	S	AA ^b	0.1
(mo/L)	Ĺ	Flameless AA ^c	0.001
As (ppm)	S	AA	0.1
(ma/L)	L	AA(2) ^d	2
(ma/L)	L	Flameless AA	0.001
Ba (ppm)	S	ICAP	10.
(mg/L)	L	Flameless AA	0.005

continued . . .

PARAMETER	SOLID/LIQUID	METHOD	DETECTION LIMIT
Ca (mg/L)	L	AA(2)	0.05
Cd (ppm)	S	AA	0.1
(mg/L)	L	Flameless AA	0.0002
Co (ppm)	S	ICAP	1
(mg/L)	L	Flameless AA	0.001
Cr (ppm)	S	ICAP	1
(mg/L)	L	Flameless AA	0.001
Cu (ppm)	S	AA	1
(mg/L)	L	AA(2)	0.03
Fe (%)	S	ICAP	0.01
(mg/L)	L	AA(2)	0.1
Hg (ppb)	S	AA	5
(ug/L)	L	Flameless AA	0.05
K (mg/L)	L	AA(2)	0.05
Mg (mg/L)	L	AA(2)	0.01
Mn (ppm)	S	ICAP	1
(mg/L)	L	AA(2)	0.02
Mo (ppm)	S	AA	1
(mg/L)	Ł	Flameless AA	0.005
Na (mg/L)	L	AA	0.05
Ni (ppm)	S	ICAP	1
(mg/L)	L	AA(2)	0.04
P (ppm)	S	ICAP	10
(mg/L)	L	Ascorbic Acid	0.003
Pb (ppm)	S	AA	1
(mg/L)	L	Flameless AA	0.001
Sb (ppm)	S	AA	0.2
(mg/L)	L	Flameless AA	0.002

TABLE 3.3.2-1 (continued) Detection Limits Used for Analysis of Solids and Liquids from the Pilot Plant Program, Norecol Test Program, Cinola Gold Project

continued . . .

TABLE 3.3.2-1 (concluded) Detection Limits Used for Analysis of Solids and Liquids from the Pilot Plant Program, Norecol Test Program, Cinola Gold Project

PARAMETER	SOLID/LIQUID	METHOD	DETECTION LIMIT
Se (ppm)	S	AA	0.2
(mg/L)	L	Flameless AA	0.001
Zn (ppm)	S	AA	1
(mg/L)	L	Flameless AA	0.0005

Jarrell Ash Inductive Coupled Argon Plasma Spectrograph. a

ь

Varian Atomic Absorption Spectrophotometer. Perkin-Elmer Model 5000 Spectrophotometer. MGA-500 Graphite Furnace. Perkin-Elmer Model 5000 Spectrophotometer. С

d

e Concentration for solids phase analysis are reported as ppm which can be considered as approximately equivalent to ug/g for the purposes of this assessment.

checked and adjusted, if necessary, at daily intervals. At weekly intervals, the tailings solids were stirred for 1 h and the suspension was allowed to settle before taking a 500 mL aliquot for analysis. Each aliquot removed was replaced by an equivalent amount of leaching solution.

The aliquot was split into three sub-samples: (1) a general sample collected and stored at 4° C with no preservative and analyzed for pH, Eh, conductivity, sulphate, alkalinity, acidity, and chloride; (2) a dissolved metals sample filtered through a 0.45 um filter and preserved with nitric acid; and (3) a mercury sample preserved with potassium dichromate and sulphuric acid.

The analyses for each parameter for both short-and long-term leaching tests were done using standard methods (APHA 1985). Metals were analyzed using the following instruments: a Jarrell Ash Model #975 inductively coupled plasma spectrograph (ICAP), a graphite furnace atomic absorption spectrophotometer, a hydride vapor generation atomic absorption spectrophotometer, and cold vapor generation atomic absorption spectrophotometer.

3.4 Characterization of Slurry

The key results of the mill effluent characterization program are summarized in Tables 3.4-1 and 3.4-2, with complete results given in Appendix 3.4-1. Average concentrations in the liquid phase component are presented in Table 3.4-1, and average concentrations in the solid tailings component are summarized in Table 3.4-2.

The following detailed description of the results follows the changes in the slurry chemistry through key stages of the proposed mill process with emphasis on interpretation of the chemistry in the final effluent. The results are discussed separately for the solid and liquid phases. The final mill effluent quality was used for environmental impact assessment purposes and was characterized based on the effluent quality after SO₂/air cyanide destruction and sodium sulphide addition stages. (The SO₂/air process is the preferred cyanide destruction method.) The final effluent is considered representative of the expected characteristics of the slurry at the time of deposition in the tailings/waste rock impoundment. Data from short and long term leach tests on the tailings solids are discussed under Section 3.4.2, solids component. These studies were conducted to provide data on the quality of potential leachate from tailings when exposed to water at pH values of 5.0, 7.0 and 8.0 to 8.5 over time.

3.4.1 Liquid phase

3.4.1.1 Metals

The acidification of the feed ore with sulphuric acid, and subsequent addition of nitric acid for oxidation decreased the slurry pH to 1.2. Consequently, the measured concentrations of metals in solution were, as expected, high at the end of this stage due to the dissolution of minerals at low pH. Aluminum and iron had the highest solution concentrations because of their corresponding higher content in the ore and their elevated solubilities at acidic pH. Lead was found at the lowest

TABLE 3.4-1

Summary of Tallings Liquid Characterization by Norecol from Pilot Plant Program for the Cinola Gold Project

PARAMETER	UNITS	AFTER NITRATE OXIDATION	AFTER NEUTRALIZATION	BEFORE CYANIDATION ⁶	AFTER CYANIDATION	AFTER CYANIDE DESTRUCTION SO2/AIR	AFTER SULPHIDE ADDITION SO2/AIR	AFTER CYANIDE DESTRUCTION H2O2	AFTER SULPHIDE ADDITION H2O2
	<u></u>	3	3	3	2	2	1	2	1
pH ·		1.2	7.3	6.1	9.7	7.7	8.0	8.0	8.0
Eh	٣V	653	500	533	105	203	170	205	190
Conductivity	umhos/cm	50400	3657	3900	7200	7535	10010	7845	8710
Sulphate	mg/L	44375	1448	1323	1657	2313	3250	1704	2438
Chloride	mg/L	22.5	13.6	18.2		-	-	•	-
Alkalinky	mg CaCO3A	•	-	-		150	170	178	175
Nitrate	mg N/L	•		•	283	274	295	294	265
Nitrite	mg N/L	-			-	2.86	2.80	1.90	3.29
Ammonia	mg N/L	•		-		0.19	0.115	0.183	0.205
Total Phosphorus	mg/L	119	0.412	0.228	0.35	0.40	0.50	0.38	0.85
Tolal Cyanide	mg/L	-	•	•	152	0.59	0.54	1.48	0.78
WAD Cyanide	mgA.	-	-	•	•	0.13	0.33	0.66	0.46
Thiocyanate	mg/L	•	•	•	1350	1525	1463	1347	1300
Cyanale	mg/L	•	•	•	1.59	2.04	3.26	2.64	4.67

5

1

continued . . .

ĩ

						AFTER	AFTER	AFTER	AFTER
PARAMETER	UNITS	AFTER NITRATE OXIDATION	AFTER NEUTRALIZATION	BEFORE CYANIDATION ⁶	AFTER CYANIDATION	CYANIDE DESTRUCTION SO ₂ /AIR	SULPHIDE ADDITION SO2/AIR	CYANIDE DESTRUCTION H ₂ O ₂	SULPHIDE ADDITION H2O2
Dissolved Metals									
Aluminum	mg/L	1640	0.73	0.82	0.73	1.1	1.4	0.85	1.9
Antimony	mg/L	0.21	<0.05	0.061	0.36	0.61	0.10	0.56	0.16
Arsenic	mg/L	82	0.033	0.036	0.15	0.19	0.13	0.15	0.055
Barium	mg/L	0.27	<0.05	<0.05	<0.05	<0.05	< 0.05	0.14	<0.05
Cadmium	mg/L	0.09	<0.001	<0.001	0.029	<0.001	< 0.001	0.001	<0.001
Calcium	mg/L	-	-	-		825	650	960	650
Chromium	mg/L	16.3	<0.004	<0.002	0.023	0.060	0.017	0.17	0.19
Cobalt	mg/L	5.3	<0.05	<0.05	0.43	0.52	0.52	0.52	0.50
Copper	mg/L	28.7	0.05	0.04	13.8	0.37	0.10	0.99	0.75
Iron	mg/L	8373	<0.05	<0.06	0.08	0.12	0.07	0.14	0.13
Lead	mg/L	<0.02	<0.02	<0.02	< 0.02	0.03	< 0.02	0.02	< 0.02
Magnesium	mg/L	•	-	•	•	17	0.90	1.48	0.15
Manganese	mg/L	113	0.15	0.03	<0.02	<0.02	< 0.02	<0.02	< 0.02
Mercury	ug/l.	45	0.15	0.15	645	4.67	0.24	0.39	0.34
Molybdenum	mg/L	0.65	0.180	0.26	0.30	0.23	0.13	0.36	0.19
Nickel	mg/L	17.1	0.06	0.06	1.47	0.05	0.08	0.35	0.31
Potassium	mg/L	•	•	•	-	47	54	39	35
Selenium	mg/L	0.48	0.017	0.008	0.007	0.002	0.006	0.060	0.035
Silver	mg/L	0.009	<0.001	<0.001	0.020	<0.001	0.0020	0.0020	0.0035
Sodium	mg/L		•	-	•	1550	2200	1000	1800
Zinc	mg/L	19.3	<0.02	<0.04	2.62	0.02	0.02	<0.02	0.05

TABLE 3.4-1 (concluded)

Summary of Tailings Liquid Characterization by Norecol from Pilot Plant Program for the Cinola Gold Project

a Number of samples used to calculate mean.

b Samples were similar to after neutralization samples but had been aged 5 days before beginning the cyanidation circuit.

PARAMETER	UNITS	ORE	FEED TO PILOT PLANT	AFTER NITRATE OXIDATION	AFTER NEUTRALIZATION	BEFORE CYANIDATION	AFTER CYANIDATION	AFTER CYANIDE DESTRUCTION SO2/AIR	AFTER SULPHIDE ADDITION SO2/AIR	AFTER CYANIDE DESTRUCTION H2O2	AFTER SULPHIDE ADDITION H2O2
n ^a		2	2	4	3	3	2	2	1	2	1
Acid-Base Accoun	ting										
Paste pH		4.2	3.9	-	-	-	-	8.7	8.6	8.7	8.8
Total Sulphur	%S	1.5	1.4	-	-	-	-	1.9	1.9	1.9	1.8
Sulphide	%S	1.1	1.1	-	-	-	-	0.15	0.16	0.13	0.12
Sulphate	%SO4	0.48	0.25	-	-	•		4.9	4.8	4.8	4.7
Maximum Potential Acidity	t CaCO3/1000 t	35	34	-		-	-	4.7	5.0	4.1	3.8
Neutralization Potential	t CaCO3/1000 t	-0.50	-1.0	-	-	-	-	19	23	20	23
Net Neutralization Potential	t CaCO3/1000 t	-36	-35	-	-	-		14	18	15	19
Metals Analysis											
Aluminum	%	0.35	0.33	0.13	0.28	0.29	0.36	0.34	0.41	0.32	0.37
Antimony	ppm ^b	24.5	25.0	24.8	26.0	27.3	25.0	26.5	26.0	26.0	26.0
Arsenic	ppm	195	195	145	190	193	195	195	190	190	190
Barium	ppm	35	40	40	30	30	40	40	50	35	40
Beryllium	ppm	0.5	0.5	<0.5	<0.5	<0.5	<0.5	0.5	<0.5	0.5	<0.5
Bismuth	ррт	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Calcium	%	0.09	0.05	0.02	2.3	2.3	2.6	2.38	2.37	2.27	2.29

.

I

1

.

 TABLE 3.4-2

 Summary of Tailings Solid Characterization from Pilot Plant Program by Norecol for the Cinola Gold Project

continued . . .

ł

t

PARAMETER	UNITS	ORE	FEED TO PILOT PLANT	AFTER NITRATE OXIDATION	AFTER NEUTRALIZATION	BEFORE CYANIDATION	AFTER CYANIDATION	AFTER CYANIDE DESTRUCTION SO ₂ /AIR	AFTER SULPHIDE ADDITION SO ₂ /AIR	AFTER CYANIDE DESTRUCTION H2O2	AFTER SULPHIDE ADDITION H ₂ O ₂
Metals Analysis (co	ontinued)										
Cadmium	ppm	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Cobalt	ppm	5	5	1	6	6	3	4	5	5	5
Chromium	ppm	152	60	34	50	67	63	78	184	76	162
Copper	ppm	15	35	6.5	29	24	17	42	41	31	31
Iron	%	1.74	2.48	1.7	2.1	2.2	2.5	2.15	2.22	2.10	2.18
Lanthanum	ppm	<10	<10	<10	<10	<10	<10	<10	<10	<10	<10
Lead	ppm	1	1	3	1	1	8	1	1	1	1
Magnesium	%	0.12	0.11	0.04	0.10	0.10	0.14	0.11	0.12	0.11	0.12
Manganese	ppm	88	71	29	126	128	149	125	130	122	130
Mercury	ррь	3550	4500	4675	4167	4200	2500	3600	3700	3650	3300
Molybdenum	ppm	1	5	5	5	5	6	5	5	5	4
Nickel	ррт	7	16	3.8	19	20	18	18	19	16	16
Phosphorus	ppm	185	200	115	197	200	200	180	180	170	170
Potassium	%	0.17	0.14	0.18	0.17	0.17	0.17	0.19	0.26	0.18	0.22
Selenium	ppm	3.0	2.4	3.3	3.1	2.6	3.3	3.0	3.0	3.3	3.2
Silver	ppm	3.0	3.3	3.2	2.7	2.0	1.4	1.4	1.4	1.5	1.5
Sodium	%	0.01	0.01	0.01	0.01	0.02	0.05	0.08	0.11	0.07	0.09
Strontium	ppm	9	8	6.3	51	53	57	54	56	52	54
Zinc	ppm	33	31	15	30	29	34	32	33	36	33

 TABLE 3.4-2 (concluded)

 Summary of Tailings Solid Characterization from Pilot Plant Program by Norecol for the Cinola Gold Project

Number of samples used to calculate mean.

ppm = ug/g

3-23

concentration (generally at or below 0.020 mg/L) after nitrate oxidation and at most other stages, a reflection of the fact that there is little lead in the ore (<1 ppm, Table 3.4-2). Mercury was found at an average concentration of 45 ug/L due to the oxidation of cinnabar (cinnabar was the only mercury mineral identified in the ore used in the pilot plant by scanning election microscopy).

Hot neutralization of the slurry using limestone and lime increased the pH, which resulted in a large decrease in concentrations of all dissolved metals due to chemical precipitation and co-precipitation reactions.

Mercury, for example, decreased from 45 ug/L to 0.15 ug/L due to the change in pH. It is believed that the mercury was co-precipitated with ferric hydroxide minerals. Arsenic concentrations decreased from 82 mg/L to 0.033 mg/L after neutralization due to the formation of ferric arsenate and calcium arsenate species. Ferric arsenate, a relatively insoluble form of arsenic, will predominate under oxidizing conditions with a Fe/As molar ratio of over 4. The Fe/As ratio in the slurry was 100.

Samples of neutralized slurry analyzed at the end of the neutralization step and after aging for 5 d (before cyanidation) suggested that selenium and manganese decreased with aging and nickel and zinc increased slightly during that period with no other changes in most metals.

During cyanidation, sodium cyanide was added to the circuit to form gold complexes. There were increases in solution concentrations of cadmium, copper, iron, mercury, nickel, zinc and silver during the continuous cyanidation run because of the increase in pH to alkaline values above 8 (see Section 4.0, Volume V Stage II Report) and the aqueous complexation with cyanide. After subsequent cyanide destruction metal cyanide complexes are destroyed except for the most stable cyanide complexes of iron and cobalt, and the pH decreased toward neutral values. Consequently, dissolved metal concentrations in both the SO₂/air and the hydrogen peroxide processes were dramatically lower for all of the aforementioned metals. The concentrations, for example, of cadmium, copper, mercury decreased from 0.029 mg/L, 13.8 mg/L and 645 ug/L to <0.001 mg/L, 0.37 mg/L and 4.67 ug/L, respectively, after the SO₂/air cyanide destruction process and 0.001 mg/L, 0.99 mg/L and 0.39 ug/L, respectively, after the hydrogen peroxide cyanide destruction process.

Aging of samples collected during the continuous SO₂/air cyanide destruction process had an effect of metal concentrations. After the standing period of over 30 h following the SO₂/air cyanide destruction stage, dissolved copper, selenium, and iron concentrations decreased from 0.84, 0.11, and 0.24 mg/L (Table 5, Appendix 3.4-1) to 0.37, 0.002, and 0.12 mg/L (Table 3.4-1), respectively. These decreases were due to equilibration of the slurry over time. Samples from the hydrogen peroxide process did not display this trend (Table 6, Appendix 3.4-1), indicating that this process rapidly equilibrates.

Although cyanide destruction resulted in lower concentrations of several metals, other metals increased in concentration. In particular, antimony and chromium increased after both processes and selenium increased after the peroxide process. This is attributed to the formation of oxide

3-24

complexes (eg., SbO₃, CrO₄²⁻, SeO₄²⁻) during the oxidation that leads to cyanide destruction. However, both these metals were at relatively low concentrations after cyanidation and the increase in their concentrations was not significant.

The addition of sodium sulphide to the slurry following cyanide destruction was intended to lower aqueous concentrations of mercury. The sulphide was prepared as a solution of 0.19 g/L Na₂S in 1 N NaOH. Twenty-four millilitres of this solution was added to each litre of slurry for a total addition of 4.6 mg S/L, 550 mg Na/L, and 0.024 M OH⁻. An increase in sodium concentration is seen after sulphide addition (Table 3.4-1), but the corresponding addition of hydroxide had little effect on pH indicating the tailings liquid is well buffered against pH change. In this case, the addition of hydroxide apparently caused the precipitation of carbonate minerals, primarily calcite (CaCO₃) with some magnesium, which is reflected in decreased concentrations of these elements after sulphide addition. This precipitation provided the buffering to maintain pH. Upon precipitation of CaCO₃, the tailings liquid became undersaturated with gypsum, causing gypsum in the tailings solid to dissolve and aqueous concentrations of sulphate to increase.

The addition of sodium sulphide was found to be effective in decreasing mercury concentrations. Mercury in the SO₂/air slurry was decreased from 4.67 ug/L to 0.24 ug/L which is approximately a 20 fold decrease. The peroxide treated slurry was at a relatively low level of mercury (0.39 ug/L) after cyanide destruction and little change in concentration was seen with sodium sulphide addition. Concentrations of several other metals decreased upon sulphide addition for both slurries, with notable decreases for antimony, arsenic, chromium (SO₂/air only), and copper.

A comparison of concentrations in the two slurries following cyanide destruction (Table 3.4-1) indicated that the SO₂/air process provided lower concentrations of barium, calcium, chromium, copper, iron, molybdenum, nickel, selenium, and silver, whereas the peroxide process provided lower concentrations of the other metals. Following sulphide addition, the same ranking was found, except that the SO₂/air slurry also had lower concentrations of aluminum, antimony, mercury, and zinc.

The metals concentrations in samples taken after the SO₂/air process and sulphide addition were compared to the Pollution Control Objectives for the Mining, Smelting, and Related Industries of British Columbia (Pollution Control Board 1979). The sample chosen is believed to most represent mill effluent quality at the point of discharge to the tailings impoundment. Generally, most metals concentrations were below the minimum objective set by the regulatory agencies except for aluminum, arsenic, cobalt, copper and mercury (minimum objective nil). No metals concentrations exceeded their respective maximum objective levels except for aluminum which just exceeded the upper limit.

3.4.1.2 Cyanide species

Total cyanide, thiocyanate, and cyanate were measured in the liquid phase after cyanidation, after cyanide destruction (both processes), and after sodium sulphide addition stages. Weak-acid dissociable (WAD) cyanide was measured at all points except after cyanidation.

As noted in Section 3.4, the complete results of the cyanidation stage are provided in Appendix 3.4-1. Total cyanide averaged 220 mg/L during the run (Table 4, Appendix 3.4-1) and decreased to 152 mg/L (Table 3.4-1) in the sample composited for cyanide destruction. At this point, the thiocyanate concentration was 1350 mg/L and cyanate concentration was 1.59 mg/L. After cyanide destruction by the SO₂/air process, total cyanide decreased to 0.59 mg/L of which 0.13 mg/L was WAD cyanide. Thiocyanate increased to 1525 mg/L which indicates that about half of the original total cyanide (152 mg/L) was converted to this species by the addition and subsequent reduction of SO₂. Cyanate increased slightly from 1.59 to 2.04 mg/L.

After cyanide destruction by the peroxide process, total cyanide decreased to 1.48 mg/L, of which 0.66 mg/L was WAD cyanide. Unlike the SO₂/air process, the peroxide process did not increase thiocyanate concentrations, presumably because there was no source of sulphur. Cyanate increased from 1.59 to 2.64 mg/L.

Following several days of aging and subsequent sulphide addition, total cyanide was generally steady in the SO₂/air slurry (Table 3.4-1) although the increase in WAD cyanide to 0.33 mg/L indicated metal-cyanide complexes were degrading to free cyanide. Thiocyanate decreased to 1463 mg/L and an increase in cyanate to 3.26 mg/L suggested some cyanide may have been oxidizing to cyanate. Total cyanide in the peroxide slurry showed an almost two-fold decrease in concentration after standing and sulphide addition. WAD cyanide also decreased, but proportionally less than the decrease of total cyanide, indicating an increasing proportion of total cyanide was becoming WAD. Like the SO₂/air slurry after sulphide addition, thiocyanate decreased and cyanate increased. Further data on cyanide destruction efficiency are given in Section 3.7.

3.4.1.3 Nitrogen species

Nitrate analyses were performed after cyanidation, after cyanide destruction, and after sodium sulphide addition. Nitrite and ammonia analyses were performed after cyanide destruction and after sodium sulphide addition. Nitrogen concentrations in the slurry were due to the addition of nitric acid in the nitrate oxidation stage which is used to oxidize the sulphide minerals and enhance gold recovery.

Nitrate concentrations were essentially unaffected by the process stage with concentrations ranging from 265 to 304 mg N/L. After cyanide destruction by the SO₂/air process, nitrite and ammonia concentrations were 2.86 and 0.190 mg N/L, respectively; ammonia decreased to 0.115 mg N/L after sodium sulphide addition. After the hydrogen peroxide cyanide destruction process, nitrite concentrations were slightly lower than for the SO₂/air process (1.90 mg N/L), with similar ammonia concentrations (0.183 mg N/L). Nitrite increased slightly to 3.29 mg N/L after sodium sulphide addition. Neither nitrate/nitrite nor ammonia concentrations in the final liquid effluents after sulphide addition exceeded the B.C. Pollution Control Objectives for mines.

3.4.1.4 Major ions

Sulphate, chloride, calcium, magnesium, sodium, potassium, and alkalinity were measured to characterize the major ion composition of the liquid phase.

Sulphate concentrations averaged 44 375 mg/L after nitrate oxidation. Sulphates were derived from sulphuric acid used to acidify the feed slurry and the oxidation of the sulphide minerals in the ore to sulphate. After neutralization, sulphate concentrations decreased to 1448 mg/L due to precipitation of calcium sulphate (gypsum) and metal hydroxides with the addition of lime and limestone and the subsequent increase in pH. Sulphate concentrations then increased somewhat to around 1650 mg/L after cyanidation. For the SO₂/air cyanide destruction process, sulphate concentrations increased after cyanide destruction to 2300 mg/L due to the SO₂ reagent used in the process. Concentrations of sulphate for both processes increased after sodium sulphide addition due to the complex geochemical scenario presented in Section 3.4.1.1. This scenario also accounted for increased sodium and decreased calcium and manganese. Potassium essentially remained unchanged because it was not involved in solid-liquid interactions.

3.4.2 Solids phase

The results of tests performed on the solids component of the tailings slurry are discussed below under four headings: (1) acid generation potential in pilot plant samples, (2) metal levels in pilot plant samples, (3) short term leach results of final effluent tailings, and (4) long term leach results of final effluent tailings.

The first two sections present the results of sampling conducted during the pilot plant run to define the tailings chemistry at key stages of the mill process. The last two sections describe the results of leaching tests conducted over time to provide information on potential chemical changes in the tailings with time.

3.4.2.1 Acid generation potential

Acid-base accounting was performed for the ore and feed solids to characterize the acid generation potential prior to processing, on samples of the slurry after cyanide destruction (both processes), and after sodium sulphide addition to characterize the acid generating potential of the tailings.

Acid-base accounting of the ore and of solids from the feed slurry indicated an acid producing potential with average net neutralization potentials of -36 and -35 t CaCO₃/1000 t of rock, respectively, and an average sulphur content of 1.5% and 1.4% (Table 3.4-2). Of the total sulphur content, approximately 79% was sulphide. The ore had an average negative neutralization potential of -0.50 t CaCO₃/1000 t of rock and an acid paste pH of 4.2. These results indicate that the ore and solids in the plant feed will be potentially acid generating, and the acid paste pH indicates that acid generation may have already been initiated.

The results of acid-base accounting for tailings samples after the cyanide destruction (both processes) and sulphide addition stages were similar and confirm the acid consuming potential of the tailings (net neutralization potentials between 14 and 18 t CaCO₃/1000 t of rock). The percent sulphur at 1.9% was slightly higher than in the feed, presumably due to addition of sulphur from sulphuric acid added during the mill process. Sulphur was present in the tailings mainly in the form of sulphate, with only approximately 0.14% sulphide present. Paste pH of the tailings was 8.7.

3.4.2.2 Metals levels

The solids phase of the slurry was analyzed for total metals content at each key stage of the mill process (Table 3.4-2). The results at each stage varied for different metals. The metals least affected by process stage were antimony, barium, bismuth, cadmium, and molybdenum.

After the nitrate oxidation stage, the metals content of the solids was significantly reduced for most metals due to the solubilization and oxidation of minerals in the low pH, oxidizing conditions resulting from nitric acid addition, and due to increased pressure (13 to 25 cm of H₂O) and temperature (85° C). For example, aluminum, arsenic, chromium, copper decreased from 0.33%, 195 ppm, 60 ppm, and 35 ppm to 0.13%, 145 ppm, 34 ppm, and 6.5 ppm, respectively. There was essentially no change in mercury content of solids after nitrate oxidation stage.

After the subsequent hot neutralization using limestone and lime, which precipitated metals from solution, the solids metal content was similar to that of the pre-oxidation feed slurry (Table 3.4-2). Total mercury concentrations were essentially unchanged through these stages of the process (Table 3.4-2) Although mercury concentrations in the liquid phase indicated a decrease in concentration after neutralization (Table 3.4-1), the relatively low concentrations of mercury in solution would not produce a notable change in the mercury solid content.

Arsenic returned to the pre-oxidation feed slurry concentration levels after neutralization. Arsenic that was dissolved during nitrate oxidation was probably removed from solution during neutralization as a ferric arsenate due to the high ferric iron content in the liquid phase (Table 3.4-1), oxidizing conditions and increase in pH. As discussed earlier, some arsenic may have precipitated as calcium arsenate due to lime addition.

Manganese, strontium, and calcium increased significantly in the solids content, from 29 to 126 ppm, 6.3 to 51 ppm and 0.02 to 2.3%, respectively, after neutralization due to the additional to the slurry of limestone and lime which contain these elements.

During the cyanidation stage, mercury, silver, and copper content of the solids were significantly reduced due to resolubilization and metal complexation with cyanide, and removal of a portion of these metals from the solution by adsorption onto the carbon used in the cyanidation circuit (Table 3.4-2). This step resulted in a decrease in the content of mercury in the solids from 4200 ppb to 2500 ppb, of silver from 2.0 to 1.4 ppm, and of copper from 24 to 17 ppm.

Following cyanide destruction, solid concentrations of mercury and copper increased because of the precipitation of these metals after breaking of the aqueous metal-cyanide complexes. Copper was used as a reagent during the SO_2 /air cyanide destruction process and as a result, the copper content in the samples from this method was higher than that found after peroxide destruction (Table 3.4-2).

After sodium sulphide addition, chromium, sodium, and mercury concentrations in the solids increased in the SO₂/air process sample (Table 3.4-2). Significant levels of aqueous mercury were removed as mercuric sulphide into the solids phase, although the effect of this removal was barely

detectable in the solid-phase analyses (Table 3.4-2) because of low solution concentrations. The chromium increase from around 77 ppm to over 160 ppm was probably due to an impurity in the sodium sulphide reagent used.

For the hydrogen peroxide treated slurry, mercury content in the solids remained essentially unchanged after sodium sulphide addition. Concentrations of mercury were very low in the liquid component before sodium sulphide addition, and significant precipitation did not occur after treatment (Table 3.4-2).

3.4.2.3 Short-term leach results

Short-term leach tests were performed on the tailings solids to characterize the solids stability and metal solubilities under the influence of different environmental conditions. Short-term leach tests were performed on the unwashed solids from the slurry after the cyanide destruction and the sodium sulphide stages for the SO₂/air process. The unwashed solids were considered characteristic of the solids expected in the mill effluent at the time of deposition to the impoundment in that they contained pore water of mill effluent quality. The method used leaching solutions at two pH levels (5.0 and 8.0) to simulate as closely as possible the expected leaching behaviour on exposure to precipitation (pH 5.0), and the expected leaching behaviour at the operating pH of tailings/waste rock impoundment (pH 8.0). The results of the short-term leach test are shown in Table 3.4.2-1.

Metal leaching from rock is typically pH dependent with highest metal concentrations often found at acidic pH (Section 4.0, Volume V). Leached metal concentrations are often relatively low at neutral and alkaline pH, although some metals are leached at somewhat elevated levels at alkaline pH. Overall, short-term leach concentrations from the tailings solids displayed a trend of this nature in that lower concentrations were leached at pH 8 than at pH 5. Specific anomalies to the trend, and selected parameters, are discussed below.

The metals having a higher solubility (i.e., increased solution concentrations) at pH 8 rather than at pH 5 were antimony, arsenic, and molybdenum (Table 3.4.2-1). These metals also leached at elevated concentrations at alkaline pH from waste rock and overburden as was discussed in Section 4.0, Volume V of the Stage II Report.

Copper concentrations of 0.10 and 0.09 mg/L in leachate at pH 5 were comparable to the highest values noted for waste rock and overburden at pH 5. This is attributed to a more soluble form of copper in the tailings, than in waste rock and overburden resulting from the addition of 10 mg Cu/L to the process slurry as a catalyst for cyanide destruction by the SO₂/air process.

No detectable mercury concentrations (0.05 ug/L) were leached from tailings at pH 8, but solution concentrations up to 16 ug/L were obtained at pH 5. The leach concentrations from the tailings at pH 8 were typical of concentrations from waste rock and overburden (Section 4.0, Volume V), but concentrations at pH 5 exceeded concentrations from waste rock and overburden. The elevated mercury levels in concentrations were obtained in the long-term leach test (Section 3.4.2.4).

			pH 5.0			pH 8.0					
PARAMETER	UNITS	BLANK	AFTER CYANIDE DESTRUCTION	AFTER SODIUM SULPHIDE ADDITION	BLANK	AFTER CYANIDE DESTRUCTION	AFTER SODIUM SULPHIDE ADDITION				
рН		5.5	5.0	5.1	7.2	7.8	9.1				
Conductivity	umhos/cm	41	3100	3200	43	2250	2290				
Sulphate	mg/L	< 1	1281	1281	< 1	1531	1500				
Nitrate	mg N/L	< 0.005	4.70	4.75	0.006	5.30	5.05				
Acidity (to pH 8.3)	mg CaCO ₃ /L	-	600	530	-	-	-				
Alkalinity (to pH 4.5)	mg CaCO ₃ /L	< 1	40	55	< 1	30	50				
Alkalinity (to pH 8.3)	mg CaCO ₃ /L	-	-	-	-	-	-				
Total Dissolved P	mg P/L	0.003	0.103	0.120	0.005	0.210	0.093				
Dissolved Metals											
Aluminum	mg/L	< 0.015	0.91	0.69	< 0.015	< 0.015	< 0.015				
Antimony	mg/L	< 0.001	0.061	0.065	< 0.001	0.075	0.095				
Arsenic	mg/L	< 0.001	0.019	0.022	< 0.001	0.050	0.056				
Barium	mg/L	0.0004	0.0048	0.0053	0.0002	0.0033	0.010				
Beryllium	mg/L	< 0.0003	< 0.0003	< 0.0003	< 0.0003	< 0.0003	< 0.0003				
Bismith	mg/L	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05				
Cadmium	mg/L	0.0002	0.0024	0.0022	< 0.0001	0.0002	0.0001				
Calcium	mg/L	0.028	830	860	0.051	567	578				
Chromium	mg/L	< 0.003	0.005	0.009	< 0.003	< 0.003	< 0.003				
Cobalt	mg/L	< 0.002	0.070	0.076	< 0.002	0.007	0.007				
Copper	mg/L	0.0050	0.10	0.090	< 0.0015	0.0035	0.013				

 Table 3.4.2-1

 Leachate Water Quality Analysis from Short-term Leach Tests of Tailings Solids^a by Norecol, Cinola Gold Project

continued . . .

Table 3.4.2-1 (concluded)	
Leachate Water Quality Analysis from Short-term Leach Tests of Tailings Solids ^a by Norecol, Cinola Gold Project	

			pH 5.0		рН 8.0				
PARAMETER	UNITS	BLANK	AFTER CYANIDE DESTRUCTION	AFTER SODIUM SULPHIDE ADDITION	BLANK	AFTER CYANIDE DESTRUCTION	AFTER SODIUM SULPHIDE ADDITION		
Dissolved Metals									
Iron	mg/L	0.0080	0.14	0.16	< 0.003	0.13	0.16		
Lead	mg/L	< 0.001	0.008	0.003	< 0.001	< 0.001	< 0.001		
Magnesium	mg/L	0.0069	24.9	25.4	0.0078	2.31	0.61		
Manganese	mg/L	< 0.0003	3.11	3.26	< 0.0003	0.0042	0.0007		
Mercury	ug/L	< 0.05	1.5	16	< 0.05	< 0.05	< 0.05		
Molybdenum	mg/L	< 0.004	< 0.004	< 0.004	< 0.004	0.008	0.006		
Nickel	mg/L	< 0.0025	0.31	0.30	< 0.0025	0.0029	< 0.0025		
Potassium	mg/L	0.030	1.09	1.25	0.051	0.72	0.88		
Selenium	mg/L	< 0.001	< 0.001	< 0.001	< 0.001	< 0.001	< 0.001		
Silicon	mg/L	0.99	65.0	78.4	0.31	27.2	37.6		
Silver	mg/L	< 0.003	< 0.003	< 0.003	< 0.003	< 0.003	< 0.003		
Sodium	mg/L	0.10	39.3	50.0	0.61	36.4	53.9		
Strontium	mg/L	0.0002	1.42	1.42	0.0002	0.70	0.74		
Tin	mg/L	< 0.003	< 0.003	< 0.003	< 0.003	< 0.003	< 0.003		
Titanium	mg/L	< 0.0006	< 0.0006	< 0.0006	< 0.0006	< 0.0006	< 0.0006		
Vanadium	mg/L	< 0.001	< 0.001	< 0.001	< 0.001	0.002	< 0.001		
Zinc	mg/L	0.0050	0.14	0.15	0.013	0.0037	< 0.0015		

• Unwashed tailings solids from SO₂/air cyanide destruction before and after sodium sulphide addition.

At pH 5, acid had to be added to the short-term leach sample to maintain the pH, whereas little base had to be added to the pH 8 sample because of the alkaline nature of the tailings. The addition of acid to the pH 5 sample is reflected in higher calcium concentrations than at pH 8 (Table 3.4.2-1) from CaCO₃ dissolution. In turn, sulphate concentrations are lower at pH 5 because gypsum solubility requires that sulphate decrease as calcium increases in order to maintain mineral saturation.

Relative to the B.C. Pollution Control Objectives for mines, all parameters were less than or within the range of objectives except the following. The short-term leach at pH 5, by design, was below the minimum recommended pH of 6.5 for final effluent discharge. At pH 5, the associated manganese concentration of 3.26 mg/L exceeded the upper objective of 1.0 mg/L. Additionally, the anomalous mercury concentration of 16 ug/L at pH 5 exceeded the upper objective of 5 ug/L; however, this exceedance of the objective is not significant in the long term as indicated by the long-term leach data (Section 3.4.2.4).

3.4.2.4 Long-term leach results

Long-term leach tests were performed on the pilot plant tailings solids after cyanide destruction by SO₂/air process and after sodium sulphide addition. The objective was to characterize the solids stability under the influence of different pH regimes over time in an aerobic environment. Tailings were leached under varying pH conditions to simulate the effect of: (1) groundwater flow through the tailings (pH 7.0); (2) exposure to precipitation (pH 5.0); and (3) conditions similar to that of the operating tailings/waste rock impoundment (pH 8.5). The tailings solids were washed with deionized water and were leached at a 1:20 ratio with three leaching solutions (pH 5.0, 7.0, and 8.5). Results are reported here for the first four weeks; tests are ongoing (up to Week 10, as of August 8). The results from the long-term leach test are given in Table 3.4.2-2.

The first step in the long-term leach test was the initial wash of the tailings solids to remove the tailings liquid from the pore space. A high pH of 8.7 in the wash water indicated a strong alkalinity in the tailings solids.

For the four-week leaching period, the temporal trends in iron concentrations at pH 5.0 and 7.0 indicate initially increasing concentrations followed by decreasing concentrations. This trend is possibly an aging effect of ferric-oxide minerals such as the experimentally observed transformations of Fe(OH)₃ to FeOOH to Fe₂O₃ through time. The aging effect produces temporal variations in the solubility and concentration of ferric iron. Other metals such as copper followed a similar trend to iron, and thus their concentrations are apparently regulated through adsorption to, or co-precipitation with, ferric-oxide minerals.

Generally, concentrations of metals from the long-term leach tests were higher at pH 5.0 than at pH 7.0 and 8.5 (Table 3.4.2-2). The exceptions were arsenic and antimony which had significantly lower concentrations at pH 5.0 (<0.001 to 0.010 mg/L and 0.030 to 0.051 mg/L, respectively compared to 0.050 to 0.085 and 0.090 to 0.15 mg/L, respectively in the pH 8.5 leaching solution).

				рH	5.0			рH	7.0			p⊦	18.5	
				WE	EK			W	EEK		ماند الله مي	w	EEK	
PARAMETER	UNITS	WASH WATER	1	2	3	4	1	2	3	4	1	2	3	4
pH		8.17	5.23	5.16	4.84	4.82	6.82	7.02	6.98	6.95	8.28	8.30	8.92	8.49
Eh	mV	310	310	280	280	290	300	240	240	280	300	210	210	275
Conductivity	uS/cm	4230	3830	3320	2630	3330	2930	2870	2860	2750	2590	2580	2590	2550
Acidity (to pH 8.3)	mg CaCO ₃ /L	5.15	18.0	15.9	16.0	20.6	3.58	3.18	6.44	6.50	1.58	1.86	-	-
Alkalinity (to pH 4.5)	mg HCO ₃ /L	67.7	3.38	3.08	3.30	1.69	20.7	27.1	30.5	31.5	45.7	45.6	44.0	44
Chloride	mg/L	190	12.3	11.8	5.14	4.11	9.25	1.54	1.54	1.54	7.2	2.57	2.06	2.06
Sulphate	mg/L	1550	1240	1190	832	1370	1370	1260	1250	1350	1390	1290	1290	1470
Total Acid Added [®]	ml	•	119	9	23	3	58	3	3	0	20	5	40	20
Dissolved Metals														
Aluminum	mg/L	<0.015	0.37	0.33	5.22	2.10	<0.015	<0.015	<0.015	<0.015	<0.015	<0.015	<0.015	<0.015
Antimony	mg/L	0.12	0.051	0.048	0.030	0.030	0.14	0.15	0.14	0.15	0.090	0.15	0.15	0.14
Arsenic	mg/L	0.11	0.010	0.006	0.003	<0.001	0.011	0.008	0.003	0.004	0.085	0.088	0.05	0.077
Barium	mg/L	0.0015	0.0015	0.0014	0.0027	0.0012	0.0006	0.0004	0.0006	0.0012	0.0008	0.0009	0.0011	0.0008
Beryllium	mg/L	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.00033	<0.0003
Bismuth	mg/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Cadmium	mg/L	<0.0001	0.0005	0.0009	0.0013	0.001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	0.0001	<0.0001	<0.0001
Calcium	mg/L	472	546	712	773	876	542	497	675	744	430	568	607	621
Chromium	mg/L	<0.003	<0.003	<0.003	0.005	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003

 TABLE 3.4.2-2

 Leachate Water Quality from Long-term Leach Tests by Norecol of Tailings Solids After Sodium Sulphide Addition for the Cinola Gold Project

continued . . .

3-33

				pH t	5.0	·····			pH 7	' .0			pH (8.5		
				WE	EK		-		WE	ΞK			WE	EK		
PARAMETER	UNITS	WASH WATER	1	2	3	4	-	1	2	3	4	1	2	3	4	
Dissolved Metals (co	ontinued)															
Cobalt	mg/L	0.067	0.057	0.069	0.083	0.080		0.010	0.011	0.009	0.013	0.005	0.006	0.006	0.005	
Copper	mg/L	0.027	0.027	0.030	0.12	0.078		0.0023	0.0037	<0.0015	<0.0015	<0.0015	<0.0015	<0.0015	<0.0015	
Iron	mg/L	0.076	0.065	0.098	0.59	0.27		0.092	0.31	0.005	0.072	0.058	0.035	0.014	0.065	
Lead	mg/L	<0.0001	<0.0001	<0.0001	0.0035	0.0003		c0.0001	0.0009	0.0014	0.0009	<0.0001	<0.0001	<0.0001	<0.0001	
Magnesium	mg/L	0.70	20.0	24.1	27.1	28.3		13.7	17.2	19.3	19.4	3.05	4.74	6.42	6.93	
Manganese	mg/L	<0.0003	2.64	3.20	3.53	3.7		0.42	0.46	0.63	0.52	0.011	0.014	0.011	0.015	
Mercury	ug/L	0.08	0.18	0.10	<0.05	<0.05		0.10	0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	
Molybdenum	mg/L	0.060	<0.004	<0.004	<0.004	<0.004		0.005	0.005	0.006	0.005	0.004	0.005	0.006	0.005	
Nickel	mg/L	<0.0025	0.24	0.34	0.37	0.32		0.021	0.019	0.030	0.020	<0.0025	<0.0025	<0.0025	<0.0025	
Potassium	mg/L	6.42	1.91	2.16	2.58	2.59		1.52	1.79	1.86	2.05	1.36	1.64	1.7	1.89	
Selenium	mg/L	0.010	<0.001	<0.001	-	-		0.001	0.002	-	•	0.005	0.011	-	-	
Silicon	mg/L	4.52	56.3	70.9	73.8	9.64	:	34.8	1.83	37.7	38.2	32.3	37.7	33.7	14.2	
Silver	mg/L	<0.003	<0.003	<0.003	<0.003	<0.003		c0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	
Sodium	mg/L	384	34.6	41.8	38.1	28.0	:	34.9	42.4	34.1	28.0	37.0	40.2	47.9	60.9	
Strontium	mg/L	0.59	0.92	1.10	1.2	1.29		0.67	0.63	0.78	0.80	0.65	0.75	0.73	0.76	
Tin	mg/L	<0.003	<0.003	<0.003	<0.003	<0.003		c0.003	<0.003	<0.003	<0.003	< 0.003	< 0.003	< 0.003	< 0.003	
Titanium	mg/L	<0.0006	<0.0006	<0.0006	<0.0006	<0.0006		0.0006	<0.0006	<0.0006	<0.0006	< 0.0006	<0.0006	<0.0006	<0.0006	
Vanadium	mg/L	0.001	<0.001	<0.001	<0.001	<0.001		c0.001	<0.001	<0.001	<0.001	⊲0.001	< 0.001	< 0.001	< 0.001	
Zinc	mg/L	<0.0015	0.15	0.16	0.21	0.22		0.0015	<0.0021	<0.0015	<0.0015	<0.0015	<0.0015	<0.0015	0.010 ⁶	

.

TABLE 3.4.2-2 (concluded)

Leachate Water Quality from Long-term Leach Tests by Norecol of Tailings Solids After Sodium Sulphide Addition for the Cinola Gold Project

Total amount of 1.0 N nitric acid added in corresponding week (in Week 1, 0.2 N nitric acid was used).
 Data point believed to be erroneous based on subsequent data.

ŧ

ł

It appears that the arsenic and antimony may have been removed from the pH 5.0 solution due to the reaction with other minerals present at the lower pH.

Aluminum concentrations in the pH 5.0 leach test increased from 0.33 mg/L in Week 2 to 5.22 mg/L in Week 3 which was related to the relatively high solubility of aluminum at this pH.

Mercury concentrations in the leach solutions displayed the same pH-dependence noted in the short-term leach tests (Section 3.4.2.3) and in leaching of waste rock and overburden (Section 4.0, Volume V). At pH 8, mercury concentrations were consistently below the detection limit of 0.05 ug/L, whereas there were detectable concentrations up to 0.10 ug/L at pH 7 and up to 0.18 ug/L at pH 5. Based on mass balance calculations, a concentration of 0.18 ug/L represents the leaching of less than 1% of total solid-base mercury in the tailings solids so that depletion of mercury from the solid phase is not a limiting factor on aqueous concentrations. Furthermore, because only a small proportion of the leach water was removed each week for analysis, the decreasing concentrations through time at pH 5 and 7 represent an initial small release of mercury to the water followed by a return of mercury to the solid phase through time. This behaviour is consistent with the initial partial dissolution of a poorly crystalline, rapidly formed mineral which would have a relatively high solubility as compared to a mineral aged to a higher degree of crystallinity, which would lower aqueous concentration through time. Such a scenario has been reported in the literature for ferric iron-hydroxide minerals and aluminum-hydroxide minerals. In general, the results of both leach tests indicate mercury concentrations that could leach from the tailings at alkaline pH are expected to be consistently below 0.05 ug/L through time. Leach concentrations at pH 5 could be expected to be initially greater than 0.1 ug/L (short-term leach), decreasing to less than 0.05 ug/L within 3 weeks.

Relative to the B.C. Pollution Control Objectives for mines, all parameters were less than or within the range of objectives except the following. The long-term leach test work was carried out at nominal pH 5 which is less than the minimal objective pH of 6.5 and thus provided a conservative estimate of metal leaching. At pH 5, aluminum concentrations of 5.22 and 2.10 mg/L at Week 3 and 4 exceeded the upper objective of 1.0 mg/L, whereas aluminum concentrations at pH 7.0 and 8.5 were less than the detection limit of 0.015 mg/L. Also at pH 5, manganese concentrations of 2.6 - 3.7 mg/L exceeded the upper objective limit of 0.015 mg/L. Also at pH 5 manganese concentrations of 0.4 - 0.6 mg/L at pH 7.0 fell within the objective range of 0.1 - 1.0 mg/L and concentrations of 0.01 mg/L at pH 8.5 were significantly less than the lower objective.

3.4.2.5 Grain size distribution

Grain size distribution results are shown in Table 3.4.2-3. Two thirds of the tailings solids passed through a 270 mesh (wet sieve) with approximately 5% of the weight retained in the other sieves.

3.5 Comparison of Pilot Plant Results with Bench Scale Test Results

A comparison was made of the February 1988 bench scale test results and the May 1988 pilot plant results to assess the representativeness of using the bench scale test in the impact assessment of the mill effluent presented in the Stage II Report. The February bench scale test program used only the hydrogen peroxide cyanide destruction process; results of the February cyanide destruction process are compared to both SO₂/air and hydrogen peroxide cyanide destruction work performed during the May pilot plant program.

The results from the May and February programs were assessed by calculating ratios for each parameter from each stage of the process. The ratios were calculated by dividing each parameter from the summary table of the pilot plant program with the corresponding parameter from the summary table of the bench scale program. Where parameters were not detected, the analytical detection limit was used for the assessment. Ratios above 1 indicate that levels were higher in the pilot plant program than in the bench scale testwork, and ratios less than 1 indicate that the levels were lower. The ratios from the liquid and solid component test programs are given in Tables 3.5-1 and 3.5-2, respectively. The ratios for the short-term and long-term leach tests are given in Tables 3.5-3 and 3.5-4, respectively.

3.5.1 Comparison of results - liquid phase

3.5.1.1 Metals

Mercury concentrations during the first three stages of the pilot plant program (nitrate oxidation, neutralization, before cyanidation) were lower than during the bench scale test by factors of 9 (Ratio = 0.11), 210 (Ratio = 0.0048) and 7 (Ratio = 0.14) respectively. The higher mercury concentration in the bench scale program after neutralization was due to one anomalously high mercury value. After cyanidation, mercury was higher by 4.1 times in the pilot plant program than the bench scale due to the nature of continuous cyanidation circuit in the pilot plant program compared to the batch cyanide leach in the bench scale test. During the continuous run in the pilot plant program, less mercury loaded onto the carbon, therefore, more mercury remained in solution. Despite the higher mercury concentrations after cyanidation in the pilot plant program, after both cyanide destruction processes during the pilot plant program the mercury concentrations were lower by 17 times (Ratio = 0.06) after the SO₂/air process and 100 times after the hydrogen peroxide process than reported in the bench scale tests. This is probably due to the optimization of each cyanide destruction process during the pilot plant program. Mercury concentrations after the sodium sulphide addition were similar in both the pilot plant program and the bench scale tests. This is due to the removal efficiency of sodium sulphide lowering mercury concentrations to a certain level.

Arsenic concentrations in the pilot plant program were similar to concentration in the bench scale test in all stages except after neutralization when arsenic was 5.8 times (Ratio = 0.17) lower during the pilot plant program compared to the bench scale test. Higher iron concentrations in solution after nitrate oxidation during the pilot plant program enhanced the arsenic precipitation during the

	• •	•
MESH NO.	WEIGHT RETAINED (%)	CUMULATIVE WEIGHT RETAINED (%)
100	6.6	6.6
140	5.3	11.9
200	5.5	17.4
250	5.7	23.1
270	1.7	24.8
325	8.2	33.0
Passing 325	67.0	100.0

TABLE 3.4.2-3Norecol's Size Analysis of Tailings After Sodium Sulphide Addition,
Pilot Plant Program, Cinola Gold Project

PARAMETER	AFTER NITRATE OXIDATION	AFTER NEUTRALIZATION	BEFORE CYANIDATION	AFTER CYANIDATION	CYANIDE DESTRUCTION SO2/AIR	AFTER SULPHIDE ADDITION SO ₂ /AIR	CYANIDE DESTRUCTION H2O2	AFTER SULPHIDE ADDITION H ₂ O ₂
ρH	0.69	0.79	-	0.95	0.79	0.82	0.82	0.82
Eh	1.0	1.7	-	1.2	1.2	0.93	1.2	1.0
Conductivity	2.3	0.93	-	1.1	3.6	4.2	3.1	3.6
Sulphate	2.9	1.0	-	0.91	1.2	1.8	0.90	1.4
Chloride	-	-	-	-	-	-	-	-
Alkalinity	-	-	-	-	1.2	-	1.4	-
Nitrate	-	-	-	1.3	1.2	1.4	1.3	1.3
Nitrite	-	-	-	-	1.0	-	0.68	-
Ammonia	-	-	-	-	0.85	-	0.82	-
Total Phosphorus	1.5	0.70	0.88	1.9	2.1	1.9	2.0	3.2
Total Cyanide	-	-	-	-	0.09	0.07	0.22	0.10
WAD Cyanate	-	-	-	-	0.63	-	3.2	-
Thiocyanate	-		-	-	2.1	-	1.8	-
Cyanate	-	-	-	-	0.71	-	0.92	-

TABLE 3.5-1

Ratios Calculated by Norecol to Compare Values of Parameters in Tailings Liquid Characterized from Pilot Plant Program^a to Bench Scale Testwork^b, Cinola Gold Project

continued

PARAMETER	AFTER NITRATE OXIDATION	AFTER NEUTRALIZATION	BEFORE CYANIDATION	AFTER CYANIDATION	CYANIDE DESTRUCTION SO2/AIR	AFTER SULPHIDE ADDITION SO2/AIR	CYANIDE DESTRUCTION H2O2	AFTER SULPHIDE ADDITION H2O2
Dissolved Metals								
Aluminum	2.3	1.8	1.9	1.9	5.5	4.7	4.3	6.3
Antimony	1.4	0.38	0.49	2.3	4.0	0.54	3.7	0.86
Arsenic	1.4	0.17	0.18	0.84	1.9	1.1	1.5	0.46
Barium	0.92	0.17	0.15	0.17	0.20	0.31	0.57	0.31
Cadmium	1.4	1.0	1.0	2.9	1.0	1.0	1.0	1.0
Calcium	-	-	-	-	0.92	0.62	1.1	0.62
Chromium	6.0	0.93	2.0	9.6	2.7	0.54	7.5	6.1
Cobalt	1.5	1.0	1.0	1.4	1.7	1.6	1.7	1.5
Copper	1.6	1.5	0.52	0.98	0.38	0.23	1.0	1.7
Iron	4.3	0.71	2.8	0.02	0.06	0.03	0.07	0.05
Lead	0.39	5.0	12	20 ^c	20 ^c	20 ^c	20 ^c	20 ^c
Magnesium	-	•	-	-	5.6	0.45	0.49	0.08
Manganese	2.4	7.3	1.5	1.0	1.0	1.0	1.0	1.0
Mercury	0.11	0.0047	0.11	4.1	0.06	0.72	0.01	1.0
Molybdenum	11	1.7	2.6	2.6	2.0	1.0	3.1	1.5
Nickel	3.5	4.3	3.4	51	5.0	8.0	35	31
Potassium	-	-	-	-	0.52	0.61	0.43	0.40
Selenium	0.94	0.62	0.43	0.11	0.19	0.50	5.8	2.9
Silver	5.9	1.4	0.52	1.9	0.06	2.0	0.11	3.5
Sodium	-	-	-	-	2.0	2.8	1.3	2.3
Zinc	1.2	0.67	2.2	0.83	1.0	1.0	1.0	2.5

TABLE 3.5-1 (concluded)Ratios Calculated by Norecol to Compare Values of Parameters In Tailings Liquid Characterized
from Pilot Plant Program^a to Bench Scale Testwork^b, Cinola Gold Project

a Pilot plant program data taken from Table 3.4-1.

b Bench scale testwork data taken from summary Table 1.4-1, Volume IV Appendices, Stage II Report. The data are also given in Appendix 3.5-1, Table 1.

c Ratio due to higher detection limit of 0.02 in the pilot plant program divided by lower detection limit of 0.001 in the bench scale testwork.

PARAMETER	FEED	AFTER NITRATE OXIDATION	AFTER NEUTRALIZATION	AFTER I CYANIDATION	AFTER CYANIDE DESTRUCTION SO2/AIR	AFTER SULPHIDE ADDITION SO ₂ /AIR	AFTER CYANIDE DESTRUCTION H ₂ O ₂	AFTER SULPHIDE ADDITION H ₂ O ₂
Acid-Base Accounti	ng							
Paste pH Total Sulphur Sulphide Sulphate Maximum Potential	0.85 0.99 0.78 1.1	- - -	- - - -	- - -	1.5 2.3 1.5	- - -	1.5 2.0 1.4	- - -
Acidity Neutralization Potential Net Neutralization	0.78 0.67 0.77	-	-	-	2.3 0.82	-	2.0 0.86 0.75	-
Metal Analysis	0.77	-	-	-	0.07	-	0.75	-
Aluminum Antimony Arsenic Barium Cobalt Chromium Copper Iron Lead Manganese Mercury Molybdenum Nickel Phosphorus Selenium Silver Zinc	$\begin{array}{c} 1.1\\ 1.2\\ 0.87\\ 1.3\\ 1.0\\ 1.0\\ 6.0\\ 1.4\\ 1.1\\ 0.67\\ 0.79\\ 1.3\\ 4.5\\ 2.0\\ 0.82\\ 0.05\\ 0.82\\ 0.05\\ 0.7\\ 0.09\end{array}$	0.66 1.1 0.89 1.2 1.0 1.0 8.5 1.2 0.87 0.89 0.67 1.3 5.0 3.8 0.70 0.05 1.0 0.21	$\begin{array}{c} 0.93 \\ 1.2 \\ 0.88 \\ 1.0 \\ 1.0 \\ 1.4 \\ 5.5 \\ 1.1 \\ 1.0 \\ 0.27 \\ 1.2 \\ 1.1 \\ 5.3 \\ 2.8 \\ 0.89 \\ 0.06 \\ 0.92 \\ 0.07 \end{array}$	$\begin{array}{c} 1.3\\ 1.1\\ 0.89\\ 1.3\\ 1.0\\ 0.90\\ 7.8\\ 1.2\\ 1.2\\ 2.5\\ 1.5\\ 1.6\\ 6.0\\ 3.4\\ 0.92\\ 0.05\\ 1.3\\ 0.04\end{array}$	$\begin{array}{c} 0.99\\ 1.2\\ 0.86\\ 1.1\\ 1.0\\ 1.1\\ 1.4\\ 0.20\\ 1.0\\ 0.38\\ 1.2\\ 2.6\\ 5.0\\ 2.9\\ 0.82\\ 0.05\\ 1.2\\ 0.04\end{array}$	$\begin{array}{c} 1.2\\ 1.1\\ 0.84\\ 1.5\\ 1.0\\ 1.5\\ 4.3\\ 0.19\\ 1.1\\ 0.30\\ 1.3\\ 2.4\\ 5.0\\ 3.4\\ 0.84\\ 0.07\\ 1.1\\ 0.04\end{array}$	0.94 1.1 0.84 0.95 1.0 1.2 1.3 0.15 0.99 0.38 1.2 2.7 5.0 2.6 0.77 0.05 1.3 0.04	$\begin{array}{c} 1.1\\ 1.1\\ 0.84\\ 1.2\\ 1.0\\ 1.5\\ 3.8\\ 0.15\\ 1.1\\ 0.30\\ 1.3\\ 2.2\\ 4.0\\ 2.8\\ 0.80\\ 0.06\\ 1.1\\ 0.04\end{array}$

TABLE 3.5-2Ratios Calculated by Norecol for Tailings Solid Characterization from Pilot Plant Programato Bench Scale Testwork^b for the Cinola Gold Project

a Pilot plant program data taken from Table 3.4-2.

b Bench scale testwork data taken from summary Table 1.4-5, Volume IV Appendices, Stage II Report.

The data are also given in Appendix 3.5-1, Table 2.

	pł	15.0	pH 8.0			
PARAMETER	AFTER CYANIDE DESTRUCTION	AFTER SODIUM SULPHIDE ADDITION	AFTER CYANIDE DESTRUCTION	AFTER SODIUM SULPHIDE ADDITION		
pH	0.94	0.96	0.85	1.0		
Conductivity	1.0	1.1	1.4	1.4		
Sulphate	1.4	1.4	1.8	1.7		
Acidity (to pH 4.5)	-	-	-	-		
Acidity (to pH 8.3)	2.0	1.8	-	-		
Alkalinity (to pH 4.5)	0.07	0.09	2.2	3.7		
Alkalinity (to pH 8.3)	-	-	-	-		
Nitrate	1.1	1.2	1.1	1.1		
Total Dissolved P	1.4	1.6	7.7	3.4		
Dissolved Metals						
Aluminum	0.58	0.44	0.45	0.45		
Antimony	0.74	0.79	1.8	2.3		
Arsenic	0.50	0.57	1.2	1.3		
Cadmium	1.4	1.3	0.20	0.10		
Chromium	0.06	0.11	0.64	0.64		
Cobalt	10	11	2.3	2.3		
Copper	0.06	0.06	0.45	1.7		
Iron	0.05	0.06	0.23	0.29		
Lead	1.0	3.0	1.0	3.0		
Manganese	1.7	1.8	2.1	0.36		
Mercury	0.18	19	1.0	1.0		
Molybdenum	1.0	1.0	2.0	1.5		
Nickel	3.1	3.0	1.2	1.0		
Selenium	1.0	1.0	1.0	1.0		
Silver	0.23	0.23	0.47	0.47		
Zinc	0.34	0.37	1.2	0.49		

Ratios Calculated by Norecol to Compare Leachate Water Quality from Short-term Leach Tests of Tailings Solids from Pilot Plant Program^a to Bench Scale Testwork^b, Cinola Gold Project

^a Pilot plant program data taken from Table 3.4.2-1; SO₂/air cyanide destruction process.

^b Bench scale testwork data are on hydrogen peroxide cyanide destruction process. Data are taken from Table 1.4.2-1, Appendix 2.1.3-3, Volume IV Appendices, Stage II. Report data are also given in Appendix 3.5-1, Table 3.

TABLE 3.5-3

		pН	5.0			рН	7.0			pH	8.5	
	WEEK				WEEK				WEEK			
PARAMETER	1	2	3	4	1	2	3	4	1	2	3	4
рН	0.80	0.99	0.95	0.91	0.85	0.92	0.90	0.91	0.93	0.97	1.0	1.0
Eh	0.67	0.60	0.60	0.67	0.64	0.52	0.51	0.65	0.66	0.46	0.45	0.64
Conductivity	1.7	0.99	0.90	1.17	1.9	1.3	1.3	1.3	1.8	1.7	1.9	2.1
Acidity (to pH 8.3)	3.2	0.11	0.36	-	0.65	0.57	1.2	1.6	0.28	0.64	-	-
Alkalinity (to pH 4.5)	0.05	1.8	2.1	11	0.39	0.89	1.2	0.83	3.1	1.3	1.5	1.1
Chloride	1.8	2.0	0.86	0.28	9.3	1.5	1.5	1.5	7.2	0.43	2.1	2.1
Sulphate	1.7	1.8	1.3	0.01	1.9	1.8	1.9	2.1	1.8	1.8	1.9	2.3
Dissolved Metals												
Aluminum	25	0,19	2.5	3.0	1.0	1.0	0.29	0.05	1.0	1.0	0.26	0.04
Antimony	1.1	0.92	0.94	1.25	4.2	3.8	2.4	2.4	3.8	4.8	3.66	3.59
Arsenic	0.09	0.04	0.03	0.33	0.17	0.31	0.21	0.04	1.8	1.6	1.09	1.12
Barium	0.65	1.4	2.7	2.0	0.38	0.67	0.30	0.92	8.0	0.90	0.61	0.53
Beryllium	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Bismuth	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Cadmium	0.42	0.82	0.93	0.67	0.25	0.08	0.50	0.14	0.25	0.17	1.0	0.25
Calcium	1.2	1.5	1.7	1.8	1.3	1.2	1.7	1.8	1.0	1.9	2.1	2.1
Chromium	1.0	1.0	1.7	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Cobalt	7.1	1.4	1.2	1.1	3.3	2.8	1.8	3.3	1.7	2.0	1.5	1.7
Copper	2.5	0.03	0.08	0.08	1.5	0.28	0.04	0.12	1.0	0.21	0.10	0.21

1

TABLE 3.5-4

Ratios Calculated by Norecol to Compare Leachate Water Quality from Long-term Leach Tests of Tailings Solids from Pilot Plant Program^a to Bench Scale Testwork^b, Cinola Gold Project

continued . . .

F

.

		рН	5.0		pH 7.0			рН 8.5					
		WE	EEK			WEEK				WEEK			
PARAMETER	1	2	3	4	1	2	3	4	1	2	3	4	
Dissolved Metals (co	ontinued)						· · · · · · · · · · · · · · · · · · ·						
Iron	0.68	1.6	2.6	3.5	0.58	2.4	0.01	0.60	0.31	0.21	0.02	0.65	
Lead	1.0	0.03	35	3.0	1.0	4.5	1.4	9.0	1.0	1.0	0.14	1.0	
Magnesium	0.29	0.15	0.17	0.17	0.61	0.24	0.19	0.17	0.37	0.37	0.33	0.32	
Manganese	18	2.3	2.0	1.9	35	4.9	4.5	6.0	14	7.0	2.0	3.3	
Mercury	0.09	0.01	0.01	0.18	0.56	0.06	0.09	0.42	1.0	1.0	0.33	1.0	
Molybdenum	0.31	1.0	1.0	1.0	1.3	1.3	1.5	1.3	1.0	1.3	1.5	1.3	
Nickel	17	5.0	3.9	3.3	8.4	7.0	6.7	5.6	1.0	1.0	1.0	1.0	
Potassium	1.6	0.82	0.97	0.91	0.66	0.67	0.68	0.81	0.53	0.70	0.73	0.80	
Selenium	0.14	1.0	-	-	1.0	2.0	-	-	5.0	11	-	-	
Silicon	3.4	1.8	2.4	0.21	3.0	0.13	2.4	1.9	2.9	3.4	15	0.44	
Silver	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Sodium	3.8	6.4	4.7	3.1	4.9	6.4	4.7	3.4	3.3	4.3	4.2	5.9	
Strontium	3.4	4.4	4.8	5.6	2.8	3.0	3.9	4.2	2.8	4.2	4.1	4.8	
Tin	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Titanium	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Vanadium	0.17	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	
Zinc	8.3	0.57	0.49	0.50	0.23	0.25	0.12	0.27	0.14	0.29	0.06	10 ^c	

TABLE 3.5-4 (concluded)

Ratios Calculated by Norecol to Compare Leachate Water Quality from Long-term Leach Tests of Tailings Solids from Pilot Plant Program^a to Bench Scale Testwork^b, Cinola Gold Project

Pilot plant program data taken from Table 3.4.2-2.

Bench scale testwork data taken from Tables 1.4.2-7, -8, and -9, Appendix 3.1.3,

Volume IV Appendices, Stage II Report.

The data are also given in Appendix 3.5-1, Table 3.

⁶ Higher concentrations observed in the pilot plant program long-term leach experiment believed to be analytical error based on subsequent data.

neutralization stage which lowered the arsenic concentrations in solution after neutralization below the concentrations measured during the bench scale test.

Aluminum was higher in all stages during the pilot plant program than in the bench scale test by a factor of approximately 2 to 6 times. Higher aluminum concentrations were probably due to sulphuric acid solubilizing more aluminum minerals. Soluble aluminum forms a stable complex with the sulphates that are present in the solution due to the addition of sulphuric acid and the oxidation of sulphide to sulphate in the nitrate oxidation process.

Chromium, molybdenum, and nickel concentrations were higher in the pilot plant program by a factor of more than 2 in most stages (Table 3.5-1). This is probably related to increased oxidation during the pilot plant program and higher content in the ore sample (Section 3.5.2) used in the pilot plant program. Iron concentrations were lower by a factor of 14 times (Ratio = 0.07) to 50 times (Ratio = 0.02) from the cyanidation stage to the end of the process in the pilot plant program. This was probably due to less formation of ferrocyanides during cyanidation and more efficient removal of ferrocyanides during the bench scale testwork.

Copper was lower in the pilot plant SO₂/air process than after the bench scale hydrogen peroxide process. The SO₂/air process for cyanide destruction was able to use the copper more efficiently, which reduced the concentration of copper remaining in solution. Copper results for the hydrogen peroxide process were similar in both the February and May test programs. Final copper concentrations after the May pilot plant SO₂/air process and sulphide addition were lower by a factor of 4 (Ratio = 0.23) compared to concentrations measure in the earlier bench scale test program.

3.5.1.2 Cyanide species

Total cyanide was significantly lower during both continuous pilot plant cyanide destruction processes compared to February bench scale testwork (Table 3.5-1). Both processes were able to demonstrate effective cyanide removal during the pilot plant program (Table 3.5-1). Thiocyanate concentration was doubled in the pilot plant program and may be related to the increase in sulphur content due to sulphuric acid addition not used in the bench scale studies. Cyanate concentration indicated no significant changes. In comparison with the bench scale program, WAD cyanide in the May pilot plant program indicated no significant differences in the case of the SO₂/air process but was three times higher for the hydrogen peroxide process.

3.5.1.3 Nitrogen species

Nitrogen species indicated no significant change between the two test programs.

3.5.1.4 Major ions

Most major ions were similar in the bench scale and pilot plant programs throughout the process. Sulphate concentrations were higher after the oxidation stage in the pilot plant program due to the addition of sulphuric acid. Potassium, calcium, and magnesium concentrations in the pilot plant

3-44

program were one half those in the bench scale studies, due to precipitation reactions. Sodium concentrations were doubled during the pilot plant program compared to the bench scale test due to the addition of reagents during the cyanide destruction and sulphide addition stages.

3.5.2 Comparison of results - solid phase

3.5.2.1 Acid generation

Ratios comparing the results of solid phase analysis for the bench scale and pilot scale test programs are presented in Table 3.5-2. Acid-base accounting results were very similar in both bench scale and pilot plant test programs. The only difference was the two-fold increase in sulphide content of the tailings solids at the pilot plant scale.

3.5.2.2 Metals

The major differences between the metals content of the ore used in the bench scale and pilot plant programs were the higher levels of chromium (6 times), molybdenum (5 times), and nickel (2.7 times) in the ore sample used for the pilot plant program. There was also a significant decrease in selenium content by 20 times (0.05) and in zinc by 11 times (0.09) in the feed for the pilot plant program.

There was a significant decrease in copper content of the pilot plant solids after both cyanide destruction processes by a factor of 5 times (0.2) compared to the bench scale program (which used the hydrogen peroxide process). This was probably related to a reduction in the copper reagent used in each process during the pilot plant program.

Mercury in the solids after cyanide destruction was only a factor of 2 greater in the pilot plant program than the bench scale program. This could be due to lower mercury loading onto the carbon during the continuous run of the cyanidation circuit during the pilot plant run.

3.5.2.3 Short-term leach

Short-term leach results of solids after cyanide destruction using the hydrogen peroxide process from the summary table of the bench scale testwork were compared to results obtained from leaching the solids after cyanide destruction (SO₂/air process) and subsequent sodium sulphide addition. Short-term leach results were very similar between both programs despite the different cyanide destruction processes. The ratios comparing the programs are given in Table 3.5-3.

At pH 5.0, the major differences in short-term leach results were in the leaching of copper, mercury, iron, nickel, cobalt, and silver. Copper, iron, and silver concentrations in the pilot plant program leachate were lower by a factor of 17 times (ratio = 0.06), 17 times (ratio = 0.06), and 4.3 times (ratio = 0.23), respectively, than in the bench scale program. The lower copper and silver leachate concentrations are likely related to lower content in the solids. The lower copper content reflects the lower quantity of CuSO₄ added as a catalyst during cyanide destruction in the pilot plant. Iron concentrations were lower in the pilot plant leaching which is likely the result of differences in aqueous complexing, ionic strength, redox conditions and pH. Higher mercury concentrations at

the lower pH levels (up to 16 ug/L in the pilot plant tailings compared to 0.84 ug/L in the bench scale tailings) may be due to the differences in cyanide destruction processes. Higher cobalt and nickel leaching may be due to greater oxidation and breakdown of sulphide minerals holding these metals during the process and, for nickel, to higher content of nickel in the ore.

At pH 8.0, the major differences between the process products were in leaching of sulphate, phosphorus, and aluminum. Sulphate concentrations were higher in the pilot plant program by a factor of 1.7 due to an increase of sulphur from the addition of SO₂ during cyanide destruction, sulphuric acid addition before nitrate oxidation, and sodium sulphide addition.

3.5.2.4 Long-term leach

In the pH 8.5 leach solutions, long-term leaching of the pilot plant sodium sulphide-treated tailings after SO₂/air cyanide destruction gave results very similar to the bench scale tailings after cyanide destruction. Some slight differences were noted at pH 5.0 and 7.0.

Arsenic, copper, mercury, and cadmium were leached at lower concentrations at pH 5.0 and 7.0 from the pilot plant program tailings solids than from the bench scale tailings (Table 3.5-4). Copper content was lower in the pilot plant tailings, because of the lower quantity of CuSO₄ used as a catalyst for cyanide destruction.

Lead concentrations peaked at different times during the two test programs in the pH 5.0 leach solution, as indicated by the differences in ratios in Weeks 2 (0.03) and 3 (35).

Nickel content was higher in the pilot plant solids than in the bench scale solids (Table 3.5-2) and leached 3.3 to 17 times higher in the pH 5.0 and 7.0 leaching solutions from the pilot plant solids than the bench scale solids.

Sodium and strontium leached at higher concentrations from the pilot plant program tailings than from the bench scale tailings in leach solutions at all pH levels (from 3.1 to 5.4 times higher for sodium and 2.8 to 5.6 times higher for strontium, Table 3.5-4). Sodium was added during the sodium sulphide addition stage both as sodium sulphide and sodium hydroxide, and was measured at higher concentrations in the pilot plant than in the bench scale. The strontium content increases in the pilot plant tailings were probably due to the higher content in the lime and limestone used or to the greater volume of lime and limestone used during the pilot plant program.

Other metals that leached at higher concentrations from the pilot plant tailings than the bench scale tests were manganese (all pH levels), aluminum (pH 5.0), barium (pH 5.0), iron (pH 5.0), and antimony (pH 7.0 and 8.5). This could be related to increased oxidation due to sulphuric acid addition before nitrate oxidation, which may enable greater solubilization of certain metals which precipitated in a more soluble mineral form.

3.5.2.5 Grain size distribution

The grain size distribution of the pilot plant tailings is shown in Table 3.5.2-1.

TABLE 3.5.2-1

Comparison by Norecol of Size Analysis of Tailings from the Pilot Plant Program^a to Bench Scale Testwork^b, Cinola Gold Project

	CUMULATIVE WEIGHT RETAINED (%)						
MESH No.	PILOT PLANT PROGRAM	BENCH SCALE TESTWORK					
100	6.6	1.0					
200	11.9	15.8					
250	17.4	35.7					
270	23.1	43.2					
325°	57.8	55.7					
Passing 325	100.0	99.7					

a Pilot Plant Program data from Table 3.4.2-3.

ь Bench scale testwork data from average of the three tailings grain size results. Detailed data are given in Appendix 3.5-1, Table 5. c Combined 270 and 325 mesh sizes from Table 3.4.2-3.

3.6 Mercury Treatment

3.6.1 Introduction

The concentration of soluble mercury in the slurry after cyanide destruction was evaluated as a potential environmental concern, target mercury concentration of 1 ug/L was established as an experimental guideline.

The mercury removal experiments were performed in batch tests, using slurries obtained from the cyanide destruction tests and from the demonstration run (Campaign 8). Both tests used sodium sulphide to precipitate mercury, removing soluble mercury from solution as solid mercuric sulphide.

3.6.2 Methods and results

The mercury removal tests with sodium sulphide used the procedure outlined by SRK (Cronin pers. comm.). A solution containing 0.19 g/L sodium sulphide in 1-N NaOH was added to the test slurry and mixed for 20 min. The slurry was filtered, and the solids washed, dried, and weighed. Only the filtrate from each test was analyzed, with the wash and solids saved. All equipment that came in contact with the slurry was washed with nitric acid to prevent mercury contamination.

Mercury removal tests were performed in the May 1988 pilot plant program using slurries produced from SO₂/air cyanide destruction process and hydrogen peroxide cyanide destruction process of the slurry produced from the pilot plant after cyanidation.

The hydrogen peroxide treated slurry contained low initial mercury and copper concentrations. The reduction of mercury and copper from varying additions of sodium sulphide to the slurry is given in Table 3.6.2-1. A high ratio of sodium sulphide to mercury and copper was due to the low initial concentrations. The mercury concentration decreased by half from 0.7 ug/L with addition of 2.5 mg Na₂S/L liquor. The copper concentration was not affected by the sodium sulphide addition.

The SO₂/air treated slurry contained a higher initial mercury concentration than the hydrogen peroxide treated slurry. The sodium sulphide addition rates and resulting mercury and copper concentrations for the SO₂/air treated slurry are given in Table 3.6.2-2.

Mercury concentrations were reduced from 13.9 ug/L to 0.2 ug/L with a sodium sulphide addition rate of 4.8 mg Na₂S/L liquor. The addition of sodium sulphide had no effect on the copper concentration because the feed slurry had a low initial concentration of 0.3 mg/L. It was noted that the addition of the sodium sulphide solution greatly increased the viscosity of the slurry.

TABLE 3.6.2-1

Sample Tested	Feed	517-3	517-4	517-5
Na ₂ S addition ^a (mg/L liquor)	0	2.5	5	9
Hitrate Analyses Hg (ug/L)	0.7	0.3	0.4	0.4
Cu (mg/L)	0.9	1.0	0.9	0.8

Results of Tests by Hazen Research on Mercury Removal from Peroxide-treated Slurry by Sodium Sulphide Addition, Cinola Gold Project

^a Based on an estimated solids content of 43%.

TABLE 3.6.2-2

So	Socium Sulphide Addition, Cinola Gold Project									
Sample Tested	Feed	601-1	601-2	601-3	601-4					
Na ₂ S addition ^a (mg/L liquor)	0	2.9	4.8	9.5	19					
Filtrate analyses:										
Hg (ug/L)	13.9	0.6	0.2	0.4	0.9					
Cu (mg/L)	0.3	0.2	0.3	0.2	0.2					
CN (mg/L)	0.97	-	-	-	-					

Results of Tests by Hazen Research on Mercury Removal from Final SO₂/Air-treated Slurry by Sodium Sulphide Addition, Cinola Gold Project

^a Based on an estimated solids content of 43%.

-

105

Ì

-

3.7 Comparison of Cyanide Destruction Methods

3.7.1 Hydrogen peroxide

Batch cyanide destruction tests using hydrogen peroxide were done on a composite sample of tailings taken from the continuous CIL between 1200 h on May 7, 1988, and 1800 h on May 8, 1988. This sample was 46.4% solids, and contained about 200 ppm CN (total), 0.3 ppm Fe, 15 ppm Cu, and 2.7 ppm Zn. The tests were performed by placing 1000 g of slurry in a stirred beaker and adding the appropriate amounts of hydrogen peroxide and copper. A 50% solution of H₂O₂ was used, and copper was added as a copper sulphate solution. Normally, hydrated lime would be added as necessary to control the pH; however, it was not necessary in most of these tests because pH was sufficiently alkaline. In those tests where retention time was investigated, sodium arsenite was added at the end of the test to consume any residual peroxide and stop the reaction. The cyanide in the test products was estimated using the picric acid colorimetric procedure. Only those samples that indicated low cyanide levels were analyzed by the more rigorous distillation methods.

Four tests were done to determine the required amount of hydrogen peroxide to achieve acceptable levels of cyanide destruction. These tests were done at pH 10.0 with 25 mg/L copper for 2 h. Results indicated that 1 to 1.5 mL of 50% H₂O₂/kg of slurry were required to reach minimum residual cyanide levels of less than 1 mg/L. This is 4 to 6 times the stoichiometric ratio of peroxide to cyanide. The large excess of peroxide is typical of cyanide destruction systems, especially when treating slurries.

The effect of time on the residual cyanide concentration showed that 50 to 60 min were required for maximum cyanide destruction. Residual cyanide levels (picric acid method) decreased from 23 to less than 1 mg/L as the reaction time was increased from 15 to 60 min. Subsequent cyanide determinations by distillation show CN^- levels of about 0.6 mg/L after 40 and 50 min. There was little benefit from reaction times longer than 60 min.

Copper is beneficial as a catalyst for the oxidation of cyanide. However, the results indicated similar residual cyanide concentrations of 1.1 to 1.9 mg/L (picric acid method) for copper additions from 0 to 46 ppm. This indicates that copper additions had little or no effect, and that the copper in the sample is sufficient for catalysis.

The filtrates from the cyanide destruction tests were analyzed for iron, copper, and zinc as shown in Table 3.7.1-1. These results show that typical residual concentrations of 0.5 mg iron/L and 0.1 to 0.15 mg zinc/L. Copper precipitation was less consistent, with residual levels of 0.4 to 1.4 mg/L.

The best results were obtained with 1.5 mL 50% H_2O_2/kg slurry and 50 to 60 min reaction time. This resulted in residual total cyanide levels of less than 1 mg/L.
TABLE 3.7.1-1

		co	CONDITIONS ^a			RESULTS (mg/L)	
SAMPLE TESTED	H ₂ O ₂ ^b (mL/kg)	Cu ^c (mg/L)	Time (min)	CN ^d (mg/L)	CN	Fe	Cu	Zn
Feed 5/10/87	-	8	a	300	196	0.31	15.7	2.68
Feed 5/12/87	-		-	257	210	0.33	15.1	2.80
4	0.5	23	120	100	-	0.62	89.1	0.11
5	1.0	23	120	2.9	0.69	1.67	1.25	0.10
6	1.5	23	120	0.78	0.35	1.08	0.86	0.11
7	2.5	23	120	0.71	0.61	0.38	0.62	0.12
8	1.5	0	60	1.1	0.58	-	-	-
9	1.5	6	60	1.1	-	0.46	0.52	0.12
10	1.5	12	60	1.9	0.26	0.69	0.62	0.11
11	1.5	46	60	1.4	-	0.92	0.62	0.13
12	1.5	0	15	23	-	0.50	1.35	0.15
13	1.5	0	30	3.9	0.78	0.50	1.30	0.15
14	1.5	0	60	0.85	1.32	0.50	1.35	0.15
15	1.5	0	90	1.1	•	0.44	1.35	0.13
16 ^e	1.9	0	60	6.3	0.27	0.23	3.44	0.54
17 ^e	2.3	0	60	11.5	-	0.31	5.34	1.26
18 ^e	2.8	0	60	24.5	0.21	0.15	4.60	1.04
19	1.5	0	60	2.1	-	•	-	-
20	1.5	0	40	1.6	0.58	0.62	0.43	0.11
21	1.5	0	50	1.1	0.61	2.00?	0.43	0.092

Summary of H₂O₂ Cyanide Destruction Tests by Hazen Research Cinola Gold Project

^a All tests were pH 10.0.

^b H₂O₂ addition in mL of 50% per kg of slurry.

^c Cu addition is mg Cu/L of solution.

^d Picric acid method used to determine total cyanide.

* The feed was filtered before these tests. The H_2O_2 addition is mL/L of filtrate.

-

٠

÷.

The products from continuous operation contained 1 to 3.5 mg/L CN⁻ as determined by the picric acid method. A copy of the log sheet from this run is given Appendix 3.7-1.

3.7.2 SO₂/air process

Several preliminary tests were carried out on a composite sample of tailings taken from the continuous CIL circuit between 1200 h May 7, 1988, and 1800 h on May 8, 1988, to look at the effects of retention time, SO₂ and copper dosage, and pH. Previous work by Inco (1987) on Cinola pulps had shown that a single stage with 3-h retention and dosing the pulp with 6 g SO₂/g total cyanide and 30 mg/L copper at pH 8.5 produced treated effluent with less than 1 mg/L each of total cyanide and copper. This was the starting point for the present work. Results of the pilot plant program are summarized in Table 3.7.2-1.

The results of tests A and B show that slightly better cyanide and copper removal were achieved with a 3-h than with a 2-h pulp residence time. Previous work on Cinola pulp generally indicated that higher residual copper was obtained with lower retention time. A retention time of 2 h was sufficient in the laboratory to remove cyanide.

The SO₂ dosage was varied between 3.5 and 6.0 g/g total cyanide (tests B, C, D and J, K). The laboratory results suggest that the required SO₂ dosage is between 4.5 and 5.0 g SO₂/g total cyanide to achieve about 1 mg/L total cyanide (picric acid method). The actual SO₂ demand will depend on the amount of thiocyanate in the feed; for example, about 10% of the thiocyanate is oxidized.

The test results indicated that no copper addition was required to achieve total cyanide (picric acid method) of <1 mg/L (test N). No effect of copper is observed above 10 mg/L, and a dosage of 5 to 10 mg/L copper is therefore sufficient (tests G and M).

Removal of cyanide to <1 mg/L can be achieved at any pH between 8.5 and 9.5. At pH 9.0, residual iron will remain in solution. Copper removal was improved at higher pH and a compromise pH of 9.0 is, therefore, preferred (tests G and K).

Based on the optimization tests, a sample representative of the proposed treatment method was prepared as follows: cyanide was removed in a 2 h single stage reactor dosing the pulp with 5.7 g SO₂/g total cyanide (target was 6 g/g) and 10 mg/L copper; and the pH was controlled at 9.0 with sulphuric acid. Excellent results were achieved, as shown in Table 3.7.2-2. The composite treated sample was analyzed and found to contain 0.81 mg/L total cyanide with 1 mg/L copper.

		RETENTIC	N					TREATED	DEFFLUEN	r		
	FEED	TIME	Cu ²⁺	SO ₂	pН	СNт ^b	CNp ^c	Cu	Cu ^d	Fe	Zn	Ni
TEST	No. ^a	(min)	(mg/L)	(g/g CN _T)				(mg/L)				
A	1 ^b	180	30	6.0	8.5	0.65	0.33	2.32	1.05	0.58	0.34	0.25
В	1	120	30	6.0	8.5	1.70	0.35	2.58	1.30	0.83	0.21	0.17
С	1	120	30	4.5	8.5	-	0.70	2.84	-	0.44	0.18	<0.10
D	1	120	30	3.5	8.5	-	2.25	6.82	-	0.30	0.15	<0.10
E	1	120	10	6.0	8.5	0.49	<0.10	1.86	-	0.40	0.20	0.20
F	1	120	30	6.0	9.0	0.40	0.35	0.91	-	0.30	0.14	0.13
G	1	120	10	6.0	9.0	-	0.24	1.00	0.49	0.27	0.13	0.21
н	1	120	20	6.0	9.0	0.17	0.29	0.70	0.52	0.27	0.13	0.21
J	2 ^c	120	10	4.5	9.3	-	1.30	5.08	-	0.53	-	0.46
К	2	120	10	6.0	9.0	-	0.11	1.43	-	0.21	0.12	0.31
М	2	120	10	6.0	8.8	-	0.28	1.23	-	0.25	0.07	0.21
Ν	2	120	0	6.0	9.6	-	0.79	2.09	0.60	0.87	0.25	0.14

TABLE 3.7.2-1

Results of Optimization Studies for SO₂/Air Process for Tailings from the Pilot Plant Program, May 1988, Cinola Gold Project

Source: Hazen Research, Inc.; work conducted by Inco Limited

^a Feed No. 1: CNT 196 ppm, Cu 6.75 ppm, Fe 0.50, Zn 2.44, Ni 1.00 Feed No. 2: CNT 210 ppm, Cu 15.1 ppm, Fe 0.33, Zn 2.80, Ni 1.83
^b CNT = Total cyanide (distillation method)
^c CNP = Total cyanide (picric acid method)
^d After 18 h standing

				mg/L			
	TOTAL CYANIDE	THIOCYANATE	Cu	Ni	Fe	Zn	
Feed (46% Solids)	210	-	15.1	1.83	0.33	2.8	
Treated	0.81	1420	1.0	<0.1	0.2	<0.1	

Characteristics of Tailings Liquid After Cyanide Destruction, Optimized SO₂/Air Process, Pilot Plant Test Program, May 1988, Cinola Gold Project

Source: Hazen Research, Inc.

TABLE 3.7.2-2

Results from the continuous run during the pilot plant program for both cyanide destruction methods are discussed in Sections 3.4.1 and 3.4.2. The SO₂/air process was able to reduce the total cyanide and weak-acid dissociable cyanide to levels lower than those achieved by the hydrogen peroxide method. The SO₂/air process slurry showed a reduction in metals over an 18 h period which was not observed in the slurry produced by the hydrogen peroxide method (Appendix 3.4-1, Table 5). After sodium sulphide addition, most metal concentrations were lower in the SO₂/air process slurry than in the hydrogen peroxide process slurry. Therefore, because the SO₂/air method had lower cyanide and metals concentrations, it is the most environmentally favourable process.

3.8 Summary

Results from the bench scale tests carried out in February 1988 were used in the Stage II Report to determine the quality of the mill effluent and to assess the impact of the mill effluent on the environment. The differences in the operation of the May 1988 pilot plant as compared to the bench scale tests reported in Stage II included the addition of sulphuric acid before the nitrate oxidation process, as well as continuous neutralization, cyanide leach, carbon-in-leach and cyanide destruction. Pilot plant cyanide destruction included both the SO₂/air process and the hydrogen peroxide process whereas bench scale destruction only used the peroxide process. During the earlier bench scale tests all process stages were conducted in individual batches rather than continuous circuits.

The comparison of the results from the bench scale and pilot plant test programs indicated that although there were some differences, the trends and characteristics of the effluents were similar. Some differences in slurry characteristics resulted from the addition of sulphuric acid before nitrate oxidation, a step that caused increased oxidation of minerals, and from different initial levels of metals in the ore feeds.

Dissolved aluminum concentrations were higher in the liquid component of the slurry at the pilot plant than in the bench scale slurry. The sulphate from sulphuric acid and from pyrite oxidation likely complexed with the aluminum, which can account for higher levels of total aluminum throughout the circuit.

Nickel also showed higher concentrations in both the solid and liquid (dissolved) phases during the pilot plant program as compared to the bench scale tests. Higher values were probably due to higher initial nickel content of the ore and greater oxidation of the ore during the pilot plant program than the bench scale program.

Copper concentrations in the solid phase after cyanide destruction and sodium sulphide addition were lower in the pilot plant program (for both cyanide destruction processes) than in the bench scale tests. Reduced copper content was related to the fact that less copper was added as a catalyst to the slurry during the cyanide destruction processes in the continuous runs. Reduced copper concentration in the liquid phase were measured in the pilot plant program after the SO₂/air cyanide destruction process and subsequent sodium sulphide addition relative to the peroxide process in both the pilot plant and bench scale testwork. This indicated the SO₂/air process was

more effective at lowering aqueous copper concentrations despite the initial addition of CuSO₄ as a catalyst.

Acid-base accounting results for both test programs were similar, indicating both ore feeds were similar. A slightly higher sulphide content of the pilot plant tailings slurry (0.13% compared to 0.07% in the bench scale) was measured, but was not considered significant.

The comparison of the two methods of cyanide destruction in the pilot plant indicated that the SO₂/air process produced lower total cyanide and weak-acid dissociable cyanide after cyanide destruction, and lower metal concentrations following sodium sulphide addition than did the hydrogen peroxide method. Therefore, the SO₂/air process appears to produce a more environmentally favourable slurry using the Cinola ore.

Further mercury treatment tests were performed by Hazen using sodium sulphide addition. These demonstrated that mercury was removed from the liquid phase to levels below 1 ug/L with the addition of 4.6 mg Na₂S to each litre of the tailing slurry after the SO₂/air process. Following this treatment, all aqueous concentrations met the B.C. Pollution Control Objectives for mines, except aluminum which slightly exceeded the objectives.

Both short-term and long-term leach tests were performed on the final tailings solids from the SO₂/air process after sodium sulphide addition. These leach tests demonstrated that concentrations were often relatively low at neutral to alkaline pH (7.0 to 8.5) relative to concentrations at more acidic pH. However, elevated concentrations of some parameters such as arsenic were observed at alkaline pH relative to concentrations at neutral pH. These trends were also noted in leaching tests of rock and overburden (Section 4.0, Volume V).

All leach concentrations at neutral and alkaline pH (7.0 to 8.5) were less than or within the range of Pollution Control Objectives for mines. On the other hand, at pH 5, the B.C. Pollution Control Objectives were exceeded for manganese and mercury in the short-term test and for manganese and aluminum in the long-term test.

Based on the short-term and long-term leach tests, the mercury mineral(s) formed by the mill process have solubilities which decrease through time and are pH-dependent. At acid and neutral pH, the resulting aqueous concentrations were initially relatively high (over 0.1 ug/L) then decreased within three weeks to less than 0.05 ug/L as the mercury returned to the solid phase through decreasing solubility. At alkaline pH, the solubility of the mercury mineral(s) was sufficiently depressed so that no concentrations above 0.05 ug/L were encountered at any time.

The comparison of leach tests on pilot plant tailings (SO₂/air process) and bench scale tailings (peroxide process) indicated values were relatively similar at alkaline pH (8.0 - 8.5) with greater variability of lower pH. This increasing variability with decreasing pH is likely related to the increasing solubility of many minerals with decreasing pH.

SECTION 4.0

4.0 PROCESS UPDATE

4.1 Introduction

Pilot testing of the oxidation, cyanide leaching and cyanide destruction processes was conducted by Hazen Research Inc. in April and May of 1988. The results of those tests (reported by Norecol in Section 3.0 in this Addendum) have led to some modifications and refinements in the proposed flow sheet for the Cinola Gold Project. The flow sheet sections most affected by these changes are: neutralization, cyanide leaching and adsorption, gold elution and recovery, cyanide destruction and water recycling. The changes in each of these sections are discussed below.

4.2 Neutralization

The circuit has had slurry cooling inserted to recover residual heat from the nitrate oxidation stage. A primary cooler, consisting of a barometric condenser, cools the slurry from 80°c to 60°C after the limestone addition stage. Gases from limestone neutralization and cooling are vented to the nitric acid regeneration plant because some residual NO gas is stripped in neutralization and cooling. Recovered steam is used to supplement live steam for preheating the slurry prior to nitrate oxidation. A slurry of slaked lime is added to the last tank of the nitrate oxidation stage to raise the pH from 4.0 to 10.5 prior to cyanidation.

4.3 Cyanide Leaching and Adsorption

The cyanide leaching circuit has been modified to a hybrid Carbon-in-pulp/Carbon-in-leach (CIP/CIL) system. The number of tanks in this part of the process has been reduced from ten to nine. The first two are for leaching only and the last seven are leach/adsorption tanks. Activated carbon is added to the last tank and transferred, counter-current to the slurry. Carbon is removed from the third tank and sent to the gold elution and recovery circuit.

The change to a hybrid CIP/CIL section with 9 stages in total was a result of interpretation of the recent pilot plant testwork. Some minor changes in flows and carbon loading have also been made.

4.4 Gold Elution and Recovery

The elution section has been simplified to a single column of 5.2 t capacity compared to the previous 2.9 t size. The larger column will be used for both acid washing and stripping. The cycle time will be reduced to 5 strips/week compared to the previous 7 strips/week. This design offers improved economics over the previous method.

Gold and silver will be recovered electrolytically in two cells as previously proposed. The carbon regeneration furnace has been changed from a gas-fired vertical type to an oil-fired horizontal design. The principal reason for this change is that the horizontal furnace offers better control of the mercury that is retained on the carbon.

4.5 Cyanide Destruction

The SO₂/air cyanide oxidation process has been selected for cyanide destruction in preference to the hydrogen peroxide process. With either process, copper sulphate is added to the tailings slurry immediately ahead of an agitator tank. The sulphur dioxide will be purchased and stored in liquified form, but vaporized and added with the air to the reaction tank.

The choice of the SO₂/air system was based on environmental and economic considerations. SO_2 /air operating costs are less than 25% of the peroxide method. Operating experience also indicates that the SO₂/air method is simpler to use and control.

4.6 Plant Water Balance

The water usage in the plant has been modified to eliminate the need for treatment of water recycled from the tailings pond. Treated water required for drinking, washing and the power plant will be produced from fresh water taken from the water storage reservoir. The water usage is still designed to use the maximum amount of recycled water to reduce fresh water requirements.

The overall water balance from the plant has been recalculated based on the design changes made since the submission of the Stage II Report. The total volume of process water discharged into the tailings impoundment is 422.8 m³/h. The return water from the tailings impoundment consists of supernatant, precipitation and runoff water. The anticipated volume of return water from the impoundment is 419.8 m³/h, resulting in a 99.3% use of impoundment water. The modified water balance does not materially affect the "zero discharge" concept.

The net effect of the recalculated water balance is an increase of 3 m^3 /h of fresh water make-up compared to the fresh water make-up requirement estimated in the Stage II Report (Volume 2, Section 5.5.12).

4.7 Mercury Balance

A projected mercury balance for the process plant, based on testwork conducted during the pilot plant campaigns at Hazen Research Inc. is as follows:

	MER CONCEN	CURY ITRATION	DISTRIBUTION (%)		
	SOLID	LIQUID	SOLIDS (ppb)	LIQUID (mg/L)	
Ore	4450	•	100.0	*	
Nitrate Oxidation	4370	64.0	99.0	1.0	
Neutralization	4000	2.5	99.9	0.1	
Cyanidation/ Adsorption	2960	701.0	74.6	21.0	
Cvanide Destruction	3600	5.2	95.4	0.2	
Mercury Precipitation	3700	0.22	95.6	<0.008	
Carbon Desorption	200 g/t		4.4	-	

4-2

The balance shows that the mercury content of the solids phase remains essentially unchanged following the nitrate oxidation and neutralization stages. This corresponds with observations of relatively low concentrations of mercury measured in the liquids phase of the slurry (Section 3.4-1). About 21% remains solubilized in the cyanidation leach at the end of the adsorption circuit and it appears that although more than this amount of mercury may be extracted in the initial part of the CIL circuit, only 4.4% of the overall mercury input from the ore is adsorbed onto the carbon. The carbon stripping and gold recovery circuits recover about 63% of the absorbed mercury. The remaining 37% of the loaded mercury, retained on the carbon, is subsequently vapourized during carbon regeneration and collected in a scrubber by an impregnated carbon cartridge designed for mercury removal. Subsequently, the mercury may be recovered in the retort or the cartridge returned to the manufacturer for regeneration and proper disposal of mercury. The cyanide destruction and the mercury precipitation stages will fix the remaining mercury in an inert sulphide form. Overall, 95.6% of incoming mercury reports to tailings as an inert solid with only 0.008% in solution.

4.8 Lime Calcining

It is intended that a company other than City Resources will establish a limestone calcining facility at Ferguson Bay. City Resources will purchase the lime from this company for use in the Cinola Gold Project. Appropriate permits and approvals will be obtained by the limestone calcining company.

SECTION 5.0

5.0 TAILINGS PROPERTIES

5.1 Introduction

A program of laboratory testing was carried out on pilot mill tailings in order to predict the physical characteristics of the tailings to be produced from the mill at the Cinola Gold Project. The testing was carried out by Steffen Robertson and Kirsten in their laboratory in Denver, Colorado. Samples of tailings, generated by Hazen Research Inc. during May 1988, in the course of pilot scale mill process testing, were obtained for purposes of geotechnical testing. The testing was in progress at the time the Stage II report was submitted and hence the presentation of the results was deferred for inclusion in the Stage II Addendum report. This section describes the laboratory test program and presents the results obtained from the testing.

5.2 Objective of Laboratory Testing

The objective of the laboratory test program was to obtain data on the physical properties of tailings samples from the mill process testing in order to predict the behaviour of the tailings during deposition. The following specific characteristics were identified as test objectives:

- a) The tailings sedimentation rate;
- b) The density and moisture content of settled tailings;
- c) The expected rate of tailings consolidation under self-weight; and
- d) A laboratory value for the hydraulic conductivity of settled tailings.

5.3 Tailings Samples

Tailings samples were obtained by Steffen Robertson and Kirsten from Hazen Research Inc. Golden, Colorado. Two 3.5 gallon buckets containing tailings slurry and fifteen smaller bag samples of tailings were received on May 13, 1988. The preparation of the large bucket samples consisted of reconstituting the tailings to a slurry density of approximately 38% solids by weight, which is representative of the expected production slurry density. The tailings samples were split into separate quantities for placement as individual lifts in the column tests.

5.4 Laboratory Test Program And Results

The laboratory test program included grain size analyses, specific gravity tests, Atterberg limit tests, column settled density tests (drained and undrained conditions), hydraulic conductivity tests on column samples, a consolidation test and determination of the gypsum content of the tailings. A description of the tests and summary of the test results follows. Details of the test results are included in Appendices 5.1 to 5.5 and the significance of the test results are discussed in Section 5.5.

5.4.1 Grain size analyses

Three sieve and hydrometer analyses were carried out on the large tailings sample used for the column settled density tests and on two of the bag samples received. The test results indicate that 97% of the material passes the No. 100 sieve. The clay content of the tailings varied between 9 and 23%. The grain size distributions for the samples tested are shown in Appendix 5.2.

5.4.2 Atterberg limit tests

Atterberg limits were determined for the same samples used in the grain size analyses. The results of two tests indicated a non-plastic material and the third test resulted in a liquid limit of 27% and plasticity index of 3%.

5.4.3 Specific gravity tests

The specific gravity of solids was determined for two samples using the pycnometer method. The test results show specific gravities of 2.60 and 2.61 for the two samples tested.

5.4.4 Large diameter column settled density testing

Samples of tailings were slurried into 125 mm diameter columns to determine initial values of density and moisture content achieved under self weight consolidation. Deposition of the tailings slurry within the columns was accomplished by pumping a predetermined amount of tailings slurry using a peristaltic pump. Once settlement of the first lift was complete, the next lift was placed in each of the columns.

Three lifts were placed in each of the columns. The lifts ranged in height from 99 to 173 mm each. The quantity of tailings slurry in each of the lifts was weighed before being placed and the initial moisture content of each lift was calculated. All lifts were placed with the peristaltic pump set at a very slow rate of feed in order to minimize disturbance of the surface of the previous lift.

Two column settled density tests were carried out simultaneously to model two separate cases. One of the tests was carried out under drained conditions (bottom drainage), while the other test was carried out under undrained conditions (no bottom drainage permitted). The column settled density test results are included in Appendix 5.3.

5.4.4.1 Drained test

The drained test was carried out by placing the tailings in the column with a bottom filter in place and maintaining the drainage valve at the base of the column in an open position throughout the test. Therefore, immediately after deposition, the tailings slurry tended to drain in two directions: upward to the surface and downward through the tailings to the bottom drain. The water that collected on top of the tailings during initial sedimentation eventually drained back through the tailings and out the bottom drain. Readings of the height of solids showed continued consolidation during this drainage. When further drainage of the tailings water had ceased, the next lift was placed. Once drainage and settlement for the last lift had occurred, the columns were dismantled and moisture contents were determined for each of the lifts. The test results indicate final moisture contents of settled tailings of 50 to 52% for each of the lifts in the drained test.

5.4.4.2 Undrained test

The second column was tested with no provision for bottom drainage. Therefore, the tailings water could only displace in an upward direction during settlement. The tailings water was left in place during placement of the subsequent lifts. At the completion of the test, samples were recovered for moisture content determination. The undrained test results indicate final moisture contents of settled tailings of 65% and 71% for lifts 1 and 2 respectively. A moisture content of 113% was calculated for a sample obtained from lift 3, placed in the tailings water above lifts 1 and 2.

5.4.4.3 Hydraulic conductivity

Values for hydraulic conductivity of the tailings in the vertical direction were calculated in the drained column test, using the falling head method after sedimentation of the tailings had occurred. The test results indicate a mean value of hydraulic conductivity of approximately 10^{-7} m/sec, with a range of 7 x 10^{-8} to 3 x 10^{-7} m/sec at void ratios of 1.4 to 1.5.

5.4.5 Consolidation test

A conventional consolidation test was performed on a tailings sample. A specimen was prepared by placing the tailings material in a Buckner funnel and applying a low vacuum at the bottom of the funnel. This produced a sample stiff enough to place in an oedometer ring. The initial value of void ratio was 0.926. After a seating load had been applied, load was applied in six increments as follows: 0.08 kg/cm²; 0.16 kg/cm², 0.32 kg/cm², 0.64 kg/cm²; 1.28 kg/cm² and 2.56 kg/cm². The results of this test are presented in Appendix 5.4.

5.4.6 Gypsum content

A sample of tailings was submitted to an analytical laboratory for determination of hydrated gypsum content. The test results indicate a gypsum (CaSO₄.2H₂O) content of 2.8%. The test procedure and test result are included in Appendix 5.5.

5.5 Discussion and Conclusions

The results of laboratory tests carried out on pilot mill tailings samples generally confirm the physical properties reported in the Stage II report. Some parameters differ slightly from those previously adopted for preliminary design purposes, however, these differences are not sufficient to affect the water balance or to change any design criteria. A comparison of the properties assumed in the Stage II report and the laboratory test results are shown in Table 5.5-1.

The grain size distribution and Atterberg limits show that the pilot mill tailings are a low plasticity to non-plastic silt. The gypsum content of the tailings was calculated to be 2.8%. The specific gravity of solids was shown to be 2.6 compared to 2.7 used in the volume calculations in the Stage

5-3

TABLE 5.5.1Physical and Chemical Properties of Tailings for the Cinola Gold Project:
a Comparison of Stage II Values and Laboratory Test Results

PARAMETER	STAGE II REPOR	T LABORATORY TEST RESULTS
Grain size distribution; % passing #100 sieve	55 to 90	97
Specific gravity	2.7	2.6
Average void ratio	1.25	1.25
Dry density of tailings	1:2 t/m ³	1.16 t/m ³
Moisture content; upper tailings layers	60%	50% to 71%
Hydraulic conductivity	10 ⁻⁸ m/s	7 * 10 ⁻⁸ to 3 * 10 ⁻⁷ m/s @ e = 1.4 to 1.5
Gypsum content (CaSO ₄ .2H ₂ O)	10 to 15%	2.8%

Il report. Assuming an average void ratio of 1.25 as used in the Stage II Report, the average dry density of tailings has been recalculated to be 1.16 t/m³ compared to 1.2 t/m³ used in the Stage II report. This does not significantly change the calculated volume of storage capacity required.

The column settled density tests serve to indicate the tailings sedimentation rate and the likely range of moisture content at the surface of the tailings deposit. The solids in a column of tailings slurry and water approximately 0.3 m in height, settled out of suspension within a period of approximately 30 hours (lift No. 3, undrained test). It is reported that the water overlying the settled tailings solids was clear in appearance, indicating a very low suspended solid content after this period. The implication of this result is that sedimentation of the tailings slurry occurs fairly rapidly, allowing recovery of the bulk of the tailings water in a relatively short period.

The moisture content of the settled tailings in the column tests ranged from 50% in the drained test to 71% in the undrained test. These values are considered to be representative of the moisture content of the tailings at the surface of the deposit. The moisture content obtained from the drained test (50%) models beached tailings while the undrained test (71%) models underwater deposition. These values are in agreement with the assumed moisture content of 60% near the surface as presented in the Stage II report. In practice, it is likely that the tailings will be beached where possible while a certain proportion may be deposited directly into water. The moisture content of the tailings may therefore be controlled to a certain extent by the method of deposition used. A moisture content value of 113% was calculated for lift No. 3 in the undrained test. This high moisture content is considered to represent only the uppermost surficial layer of tailings, generally less than 300 mm thick.

The mean value of hydraulic conductivity of 10^{-7} m/sec, at tailings void ratios of 1.4 to 1.5 m/s, calculated from falling head measurements during the column tests agrees with previous assumptions. The Stage II report presents a best estimate of hydraulic conductivity of 10^{-8} m/s at a void ratio of approximately 1.0. However, the laboratory determination and the field value of hydraulic conductivity may vary by as much as an order of magnitude. The test results do nevertheless indicate a higher hydraulic conductivity than that obtained from a similar laboratory study carried out by Steffen Robertson and Kirsten (1983). The results of the 1983 study were used in arriving at design parameters for the Cinola Gold Project.

Consolidation testing was performed in an attempt to determine the amount of consolidation that would occur under different loading conditions and the rate that this consolidation would occur. Simulation of the time-dependent deformation characteristics by laboratory tests is difficult, mainly because of the difficulty in preparing and testing samples at low initial densities (i.e., high void ratios). The column settled density tests indicate a void ratio of approximately 1.4 to 1.5 at the surface of the deposit. It is desirable to test a sample in the oedometer with an initial void ratio within this range in order to model the behaviour of the tailings under loads imposed by successive layers of tailings.

A consolidation test was carried out on a tailings sample using a standard oedometer. However, it is considered that the sample preparation resulted in an over-consolidated sample. The

consolidation test results indicate an initial void ratio of 0.926 for the sample tested. Based on previous work (Steffen Robertson and Kirsten 1983; Blight and Steffen 1979) the actual void ratio under the maximum consolidation load expected in the tailings impoundment (800 kPa) is approximately 1.0. The consolidation curve is shown in Figure 5.4.1, Appendix 5.4. It is not possible, from the consolidation test results, to determine the over-consolidation pressure. Steffen Robertson and Kirsten in Denver, Colorado, are currently in the process of manufacturing equipment that will enable loads to be applied to tailings samples in the large diameter columns. This test would be a form of consolidation test carried out at the completion of the column settled density tests, on samples settled under self-weight. It is proposed that if necessary, these tests be carried out during Stage III design.

SECTION 6.0

•

6.0 EXPERIMENTS WITH LIMESTONE

6.1 Background

Detailed study results of acid generation tests conducted using waste rock from the Cinola Gold Project are presented in Section 3.0, Volume V of the Stage II Report and are summarized in Section 3.7 of that volume. The results of these studies demonstrated that much of the rock is capable of generating acid. However, some types of rock, particularly Haida mudstone and some Skonun sediments (those which have high neutralization potential as defined by high carbonate content), did not generate acidic drainage when tested in experimental weathering conditions. The maintenance of neutral pH conditions in the rock appeared to be the result of the in-site neutralization of any acidity that was generated, as well as a significantly lowered rate of acid generation at the neutral pH conditions.

As a result of these findings, the waste management plans presented in Stage II stressed selective handling of the waste rock and utilization of the high neutralization potential of the waste rock and/or limestone (CaCO₃) addition for control of acid generation, depending upon disposal location (Volumes II and IV). The management plans for the waste rock stockpile as presented in the Stage II Report call for limestone to be crushed and mixed with the potentially acid generating waste rock as this rock is being placed in the stockpile. Rhyolite porphyry (which is borderline in terms of acid generation potential and is not expected, on average, to generate acid drainage) is to be placed with the acid consuming Haida mudstone in the mudstone dump. Potential acid generation from argillically altered acid generating rock placed in the tailings/waste rock impoundment will be offset by the acid neutralizing tailings.

During Stage II, two sets of experiments (laboratory and on-site) were designed to demonstrate the effectiveness of the proposed mixing of limestone with potentially acid generating rock as a means to neutralize acid generation in the waste rock stockpile. (These studies were in their preliminary stages at the time of preparation of the Stage II Report.) A number of potential concerns regarding this treatment method were adressed in the design and/or interpretation of the limestone experiments. First, it was necessary to attain an optimum grain size for the limestone - small enough for high geochemical reactivity and adherence to the rock, yet large enough that the limestone would not be washed down to the base of the stockpile, thereby allowing acidic conditions to develop in the upper part of the pile. Second, encapsulation of limestone by mineral precipitation. which decreases the geochemical reactivity of the limestone was recognized as a potential concern. and was visually evaluated in the on-site experiments. Third, the effect of purity of the limestone on neutralization capacity was taken into account. (Calculations used in Volumes IV and V assumed pure CaCO₃.) In order to address these concerns and to demonstrate experimentally the potential success of limestone treatment on Cinola waste rock, laboratory-based and on-site experiments were initiated using limestone from Texada Island, which is the likely source for the Cinola Gold Project.

The purpose of this section is to report the results of the laboratory and on-site experiments on limestone addition, to compare results with respect to metal leaching with those of earlier test programs, and to present conclusions related to proposed waste management plans.

6.2 Laboratory Experiments

6.2.1 Introduction

The laboratory experiments on limestone addition consisted of a series of plastic columns containing various ratios of acid generating waste rock and limestone to simulate the conditions in the proposed waste rock stockpile. (These test columns are referred to in this report as the "limestone columns"; the term "limestone columns" corresponds to the complete experimental set including the four columns with limestone added as well as the control column.) All columns were subjected to controlled environmental conditions of humidity and water infiltration to test the effectiveness of the various lime to waste rock mixing ratios in neutralizing acid generation. The mechanisms of neutralization and inhibition of acid generation were explored, and observations were compared with findings of acid generation studies reported in Volume V.

6.2.2 Methods

6.2.2.1 Sample selection

Humidity cells test work reported in Volume V of the Stage II Report identified categories of waste rock that were potentially acid generating and sufficiently competent for storage in the waste rock stockpile. These included Skonun sediments and multiphase breccia. Samples of these rock types for the limestone experiments were collected from drill cuttings that had been stored under water in large containers (the same source of rock as used for many of the experiments discussed in Section 3.0, Volume V). The measurement of pH in the overlying waters indicated most of the samples did not generate acidity while stored under water.

The sub-groups of rock were chosen to approximate the composition of the proposed waste rock stockpile. Samples totaling about 150 kg were assembled and mixed in the proportions indicated in Table 6.2.2-1. The mixture was passed through a splitter box and coned three times to assure thorough mixing. According to core logs, the rock mixture contained an average of 0.45% clay as random inclusions of argillic alteration. The incorporation of this argillically altered component in the samples would simulate the imperfect separation of argillically and non-argillically altered rock that could occur during mining.

The grain size distribution of the rock mixture (Table 6.2.2-2) indicates that the waste was relatively fine grained with a maximum grain diameter of 1 cm. If the grains are assumed to be generally spherical in shape, the average surface area was calculated to be approximately 5.39 m²/kg. The moisture content of the rock prior to placement in the columns was 10.8%.

Acid-base accounting was conducted in triplicate in order to evaluate the natural variability of mineral distribution in the mixture (Table 6.2.2-3). The results were similar to the mean values

TABLE 6.2.2-1

ROCK SUB-GROUP ^a	PERCENTAGE OF MIXTURE
Skonun Sediments	
Subgroup 2b	24.0
Subgroup 2c	40.1
Subaroup 2d	2.3
Subgroup 2bd	1.2
Subgroup 4c	24.5
Multiphase Breccia	
Subgroup 4b	7.9
	100.0%

Waste Rock Composition by Rock Subgroup for the Limestone Column Experiments conducted by Norecol for the Cinola Gold Project

Note: 0.45% of clay, believed to be random inclusions of argillic alteration, is distributed through the rock.

^a A detailed geologic description of lithologic subgroups is presented in Section 2.3, Volume II of the Stage II Report.

TYPE OF	SCREE	EN SIZE	WEIGHT	PERCENTAGE	CUMULATIVE PERCENTAGE
MATERIAL	(mm)	Mesh	RETAINED (g)	RETAINED	RETAINED
Rock Mixture ^a	3.35	6	53	17.7	17.7
	2.00	9	48	16.0	33.7
	1.00	18	69	23.0	56.7
	0.600	28	43	14.3	71.0
	0.300	48	42	14.0	85.0
	0.212	65	14	4.7	89.7
	0.106	150	18	6.0	95.7
	0.053	270	6	2.0	97.7
Limestone ^b	0.600	28	30	30	30
	0.300	48	23	23	53
	0.212	65	5	5	58
	0.106	150	8	8	66
	0.053	270	5	5	71
	0.045	325	4	4	75

TABLE 6.2.2-2

Grain Size Distribution for the Rock Mixture and Limestone Used in Limestone Column Experiments Conducted by Norecol for the Cinola Gold Project

All grains larger than 10 mm were removed prior to sieving.
Limestone was crushed to a maximum grain size of 0.600 mm.

		by Norecorror the		••	
				t CaCO ₃ /1000 t	
SAMPLE	PASTE pH	TOTAL SULPHUR (%S) ^a	MAXIMUM POTENTIAL ACIDITY	NEUTRALIZATION POTENTIAL	NET NEUTRALIZATION POTENTIAL
Grab #1	6.8	2.07	64.7	7.76	-56.9
Grab #2	6.9	2.12	66.3	8.27	-58.0
Grab #3	6.9	2.09	65.3	7.96	-57.3
Limestone	7.8	0.22	6.88	932	+925

TABLE 6.2.2-3 Results of Acid-Base Accounting for the Rock Mixture and Limestone Used in the Limestone Columns, by Norecol for the Cinola Gold Project

* Leachable sulphate for the grab samples were measured at less than 0.0025%.

.

reported for Skonun sediments (Table 3.2.4-1, Volume V), and Skonun sediments dominated the mixture. The measurements of the net neutralization potential suggested the rock had the potential to generate acid, and the kinetic experiments reported in Section 3.0, Volume V, indicated that this rock did indeed generate acid.

Limestone with a reported purity greater than 90% as CaCO₃ was obtained from Texada Island, the proposed source of limestone for the Cinola Gold Project. The limestone was dried and crushed to 0.6 mm diameter and smaller. The measured grain size distribution of limestone (Table 6.2.2-2) indicated about one-half of the limestone by weight was 0.3 mm or larger in diameter. The surface area of the limestone was calculated to be approximately 30.9 m²/kg based on spherical particles. Acid-base accounting (Table 6.2.2-3) indicated that the sulphur content of the limestone was 0.22% S, which was likely in the form of non acid-generating sulphate or organic sulphur. Based on the neutralization potential, the limestone had a purity of 93% as CaCO₃. An alternative measurement of CaCO₃ content in which CO₂ was determined by absorption while Ca was determined by atomic absorption yielded a purity of 75%. The reason for the discrepancy in purity is not known.

6.2.2.2 Equipment, procedures, and sampling program

Five PVC plastic columns of 0.15 m diameter and approximately 1 m in length, identical to those used in column experiments reported in Section 3.5, Volume V, were fitted with bottom drains to direct water through a short length of tubing (approximately 0.3 to 0.5 m length) and into a collection bucket. Rock was placed in the columns and the columns covered with plastic plates to minimize evaporation (Figure 6.2.2-1). A hole in each plate allowed humid air to be pumped into each column to simulate the humid conditions expected in the proposed waste rock stockpile. Masterflex pumps were used to drip de-ionized water into each column via tubing passed through a hole in the top plate. Each top plate with the drip tubing was rotated 51° daily (one complete rotation weekly) so that water was introduced onto the rock in seven different locations each week at a rate of approximately 0.4 to 0.5 mL/min. This design allowed most of the rock to come into contact with flowing water while providing the potential for preferential channeling of water through specific pathways in the waste rock. Such pathways could arise in the proposed stockpile as a consequence of cracks and leaks in the proposed capping material (Section 3.6, Volume II). A potential concern with respect to development of this drainage pattern would be preferential dissolution of limestone and subsequent acid generation in the drainage pathways.

The waste rock mixture and limestone were mixed in various proportions (Table 6.2.2-4) and placed in the columns to a depth of approximately 0.5 m. The limestone was roughly mixed into the rock; no attempt was made to thoroughly mix the limestone and rock in order to simulate the imperfect mixing conditions that would occur during stockpile construction. Column 1, containing no limestone, was the control column to determine the rate of acid generation in the waste rock without limestone. Column 2 contained the highest proportion of limestone to rock, at 6.6% limestone. This quantity of limestone was theoretically sufficient to neutralize all acidity that could be generated by the 2.1% S in the rock. Column 3 contained 3.2% limestone, which was sufficient to neutralize acidity if 50% of the sulphur in the rock generated acid. Column 3 was capped with a 1 cm thick



TABLE 6.2.2-4

Rock:Limestone Proportions in the Limestone Columns, by Norecol for the Cinola Gold Project

COLUMN	ROCK MIXTURE DRY WT (kg)	LIMESTONE DRY WT (g)	CAPPINO ROCK (g)	G MATERIAL LIMESTONE (g)	TOTAL ROCK (kg)	TOTAL LIMESTONE (kg)	RATIO OF LIMESTONE: ROCK (%)	COMMENTS
1	13.72	None	None	None	13.72	None	0	Control column
2	13.72	905	None	None	13.72	0.905	6.6	High-carbonate column
3	13.72	430	280	17.7	14.00	0.448	3.2	Medium-carbonate column
4	13.72	100	280	17.7	14.00	0.118	0.84	Low-carbonate column
5	14.12	169	None	None	14.12	0.169	1.2	Layered carbonate column ^a

• Column contained five layers of Column 2 material 1 cm thick interspersed with four layers of Column 4 material 10 cm thick.

layer of rock and limestone mixed at 6.6% limestone. The cap was designed to simulate the proposed limestone-enriched top layer proposed for the waste rock stockpile. Column 4 contained 0.84% limestone, a quantity calculated to neutralize acidity for approximately 10 weeks given a full, unhindered rate of acid generation in the waste rock. Column 4 also included a 1 cm thick limestone-enriched cap. Column 5 contained five layers of Column 2 material of 1 cm thickness, with four intervening layers of Column 4 material of 10 cm thickness (Table 6.2.2-4).

On a weekly basis, drainage water from each column was collected directly from the drain tubing and measured for pH, Eh, temperature, dissolved oxygen, and specific conductance. Additional water was collected, filtered (0.45 um filter), and analyzed for the parameters listed in Table 6.2.2-5. Following Week 15 (May 11, 1988), metals were analyzed every second week. At the end of each week, water was also obtained from the collection buckets for analysis of pH and sulphate, and then the buckets were emptied. The pH and sulphate values from the collection buckets represented one-week composites from the columns.

The limestone column experiments are ongoing; analytical results are reported here up to Week 24, with the exception of dissolved metal concentrations, which were available only to Week 20.

6.2.3 Results

6.2.3.1 pH

The value of pH in drainage from the columns is an indicator of the net effect of acid generation and subsequent neutralization reaction in the columns under different limestone addition rates and methods. Neutral pH values in drainage from columns containing acid-generating rock indicate that in situ neutralization is successfully regulating drainage pH.

Column 1 (experimental control) contained no added limestone, and pH values decreased significantly in the initial weeks then decreased gradually to around pH 2.1 (Figure 6.2.3-1 and Appendix 6.2.3-1). This trend in pH was similar to those in many of the experiments described in Section 3.0, Volume V. On the other hand, pH values from Columns 2, 3, 4, and 5 were consistently between 6.8 and 8.2 and, in fact, were nearly identical through time (Figure 6.2.3-1). This indicates that there are no significant differences in pH among the columns for the reporting period (up to Week 24) as a result of the limestone:rock ratio or the limestone layering.

6.2.3.2 Sulphate

In Volume V, sulphate was used as the primary indicator of acid generation, because the oxidation of sulphide produces sulphate. Sulphate is a valid indicator, on the condition that (1) all sulphur in the effluent exists as sulphate, (2) all accumulated sulphate is removed by the drainage from week to week, and (3) gypsum solubility does not limit sulphate concentrations in the drainage. Measurements of both sulphate and totai sulphur in drainage (Appendix 6.2.3-1) show that essentially all sulphur in the effluent was sulphate. The other two assumptions are discussed in detail below and were found to be valid throughout most of the experiment.

TABLE 6.2.2-5

Analytical Parameters Measured in Effluent from Limestone Columns, by Norecol for the Cinola Gold Project

Unfiltered Water

pH, Eh, temperature, dissolved oxygen, specific conductance

Filtered Water

Non-metals: Alkalinity, acidity to pH 4.5 and 8.3, sulphate, total sulphur

Metals: Al, Na, Si, K, Ca, Mg, Mn, P, Sb, As, Zn, Cu, Pb, Fe, Hg, Ba, Be, B, Cd, Cr, Co, Li, Mo, Ni, Ag, Sr, Th, Ti, U, V, Zr



Sulphate concentrations from Column 1 (no added limestone) in water taken directly from the drain tubing (Figure 6.2.3-2) were initially relatively low, then increased and reached a peak value of nearly 6500 mg/L at Week 21. The initial lag and acceleration are typical of other experiments described in Section 3.0, Volume V; however, the time to peak value at 21 weeks is longer by about a factor of 2 than was observed for most humidity cell experiments, while within the range of values observed for acid generation experiments as a whole (Volume V, Stage II Report).

Sulphate concentrations from Columns 2 to 5 (limestone added) (Figure 6.2.3-2) were similar to Column 1 at Week 1. During this first week, sulphate concentrations in the collection buckets were higher (Appendix 6.2.3-1) than those collected directly from the drain tubing, which indicated that a large quantity of sulphate was flushed initially from the columns. Calculations by the geochemical computer program MINTEQ (Felmy et al. 1984) suggested that this sulphate was likely derived from gypsum dissolution.

After the first week, sulphate concentrations decreased sharply as residual gypsum was removed, and sulphate concentrations began to reflect primarily acid generation. The MINTEQ calculations indicated that gypsum solubility did not limit sulphate concentrations after Week 1. Throughout most of the experiment to Week 24, sulphate concentrations in Columns 2 to 5 were generally consistent in the four columns, and much lower than concentrations in Column 1. The significant difference in acid generation in Columns 2 to 5 as compared to Column 1 is attributed to the pH-neutral conditions caused by the limestone, which has been found to decrease acid generation rates (Section 3.7, Volume V).

Sulphate loadings in all columns (weekly concentrations multiplied by weekly effluent volume, Table 6.2.3-1) showed trends similar to those in Figure 6.2.3-2 as a consequence of similar effluent volumes among all columns through all weeks. For Columns 2 to 5, the differences in total loadings from the collection bucket versus the drain tubing were primarily the result of significant gypsum flushing early in Week 1, which produced relatively high sulphate concentrations in the buckets. Based on samples from the drain tubing, average weekly rates of sulphate production were 898, 43.6, 43.3, 53.5, and 51.6 mg SO₄/kg/wk for Columns 1, 2, 3, 4, and 5, respectively (Table 6.2.3-2). Because the waste rock surface area is calculated at 5.39 m²/kg, production rates as mg SO₄/m²/wk were 167, 8.09, 8.03, 9.93, and 9.57, respectively.

For Column 1, the rate of 898 mg SO₄/kg/wk is similar to, but somewhat greater than, the values of up to 860 mg SO₄/kg/wk that were measured in other experiments for full, unhindered acid generation (Table 5.7.1-2, Volume V). Volume V discusses the importance of surface area on the rate of acid generation. The corresponding surface area rate for Column 1 of 167 mg SO₄/m²/wk compares to rates of up to 150 mg SO₄/m²/wk reported in Volume V. The unit area rates of 8.03 to 9.93 mg SO₄/m²/wk for Columns 2 to 5 are comparable to rates of 11 and 4.2 mg SO₄/m²/wk reported for pH-neutral Skonun sediments and Haida mudstone, respectively, in Table 3.7.1-2, Volume V. Because the rates are similar to or greater than those reported in Volume V, it appears that all of the sulphate generated by the rock on a weekly basis was removed by the drainage from the columns. This indicates that preferential drainage pathways through the columns have not developed.



	COLU	MN 1	COLU	MN 2	COLU	IMN 3	COLU	MN 4	COLL	IMN 5
WEEK	BUCKET	SAMPLE								
1	5.57	1.80	5.99	2.44	6.02	1.88	6.26	1.75	5.67	1.68
2	2.15	1.18	1.57	1.10	1.78	1.22	1.61	1.02	1.34	1.20
3	2.40	2.09	1.18	0.98	1.30	1.02	1.34	1.08	0.97	1.28
4	2.66	3.32	0.88	1.03	0.94	1.11	0.92	0.83	0.95	0.65
5	3.44	3.20	0.73	0.63	0.68	0.70	0.82	0.88	0.76	0.76
6	3.02	2.44	0.68	0.62	0.67	0.61	0.73	0.68	0.65	0.57
7	3.00	3.47	0.46	0.51	0.49	0.60	0.70	0.72	0.55	0.53
8	3.99	4.07	0.55	0.55	0.58	0.57	0.64	0.63	0.55	0.56
9	3.66	3.77	0.45	0.45	0.51	0.47	0.50	0.35	0.61	0.42
10	4.18	4.12	0.43	0.42	0.45	0.44	0.51	0.51	0.25	0.52
11	5.35	6.30	0.42	0.42	0.47	0.48	0.54	0.55	0.66	0.56
12	7.18	8.35	0.46	0.46	0.50	0.47	0.63	0.65	0.58	0.52
13	9.84	10.2	0.48	0.44	0.48	0.44	0.59	0.56	0.53	0.53
14	12.0	13.3	0.40	0.40	0.41	0.40	0.52	0.50	0.50	0.44
15	14.6	16.0	0.39	0.35	0.40	0.38	0.57	0.60	0.46	0.40
16	17.8	19.5	0.39	0.39	0.40	0.40	0.58	0.60	0.53	0.50
17	22.3	23.9	0.37	0.39	0.38	0.39	0.67	0.63	0.53	0.53
18	23.7	21.3	0.34	0.30	0.36	0.35	0.58	0.54	0.53	0.48
19	22.8	22.2	0.36	0.32	0.38	0.36	0.51	0.60	0.52	0.52
20	25.2	26.3	0.36	0.41	0.39	0.45	0.68	0.66	0.68	0.63
21	29.8	28.7	0.48	0.48	0.50	0.47	0.70	0.71	1.00	0.81
22	27.4	26.4	0.42	0.46	0.48	0.42	0.93	0.97	1.07	0.99
23	24.4	22.3	0.37	0.43	0.46	0.46	1.05	0.88	0.93	1.10
24	19.8	21.6	0.42	0.38	0.45	0.45	1.02	1.09	1.29	1.30
TOTAL	296	296	18.6	14.4	19.5	14.5	23.6	18.0	22.1	17.5

TABLE 6.2.3-1

.

.

Sulphate Loadings (g) Calculated by Norecol from Limestone Columns, Cinola Gold Project

COLUMN	SAMPLE LOCATION	AVERAGE WEEKLY SULPHATE PRODUCTION BY WEIGHT (mg SO4/kg/wk)	AVERAGE WEEKLY SULPHATE PRODUCTION BY SURFACE AREA (mg SO4/m ² /wk)
1	Bucket	899	167
	Drain Sample	898	167
2	Bucket	56.4	10.5
	Drain Sample	43.6	8.09
3	Bucket	58.0	10.8
	Drain Sample	43.3	8.03
4	Bucket	70.2	13.0
	Drain Sample	53.5	9.93
5	Bucket	65.2	12.1
	Drain Sample	51.6	9.57

TABLE 6.2.3-2 Average Weekly Sulphate Production Rates Calculated by Norecol for Limestone Columns, Cinola Gold Project

For Columns 2 to 5, the rate of acid generation increased from 8.09 to 9.93 mg $SO_4/m^2/wk$ as the limestone:rock ratio decreased from 6.6% to 0.84%. This relationship is not considered significant however, because the rate increased by a factor of only 1.23 as the limestone content decreased by a factor of 7.86.

In summary, sulphate concentrations and loadings indicated that rates of acid generation in the control column (Column 1) were similar to those reported for earlier acid generation studies (Volume V). The results of limestone addition in Columns 2 to 5 demonstrate not only that limestone neutralizes acidity generated, but also that the pH-neutral conditions lower the rate of acid generation as indicated by sulphate production.

6.2.3.3 Alkalinity

Alkalinity is a measure of the neutralization potential of a water sample and was measured in the column drainage water by titrating with acid to pH 4.5. Alkalinity is a general parameter reflecting concentrations of such species as carbonate $(CO_3^{2^-})$, bicarbonate (HCO_3^-) , and aluminum hydroxyl complexes $(Al(OH)_4^-)$. Because limestone added to Columns 2 to 5 increased the pH in those columns relative to Column 1, alkalinity measurements for these columns likely reflected to a high degree the concentrations of bicarbonate (HCO_3^-) derived from CaCO₃ dissolution. Calculations using the geochemical computer program MINTEQ (Felmy et al. 1984) also indicated that alkalinity predominantly represented bicarbonate levels. Conversion of alkalinity data (Appendix 6.2.3-1) to bicarbonate concentrations requires a correction factor of 2 (Appendix 6.2.3-2).

Acid mine drainage is a consequence of the oxidation of sulphide minerals, particularly pyrite (FeS₂). The chemical reactions involved in acid generation and the opposing process of acid neutralization by CaCO₃ are discussed in detail in Section 2.2, Volume V.

Because the water passing through the columns was well oxygenated and at neutral pH, most of the ferrous iron released from the pyrite was oxidized and precipitated as Fe(OH)₃. This is confirmed by the low concentrations of iron relative to sulphate concentrations (Appendix 6.2.3-1). As a result, the appropriate oxidation reaction (Reaction 5, Section 2.2, Volume V) is as follows:

FeS₂ + 15/4 O₂ + 7/2 H₂O -> Fe(OH)₃ + 2SO₄²⁻ + 4H⁺

The reaction indicates that 2 moles of H⁺ are produced for each mole of $SO_4^{2^-}$. When neutralization by carbonate minerals occurs to a neutral pH (pH 6.4 to 10.3), the dominant neutralization reaction is as follows:

 H^+ + CaCO₃(solid) <-> Ca²⁺ + HCO₃⁻

This reaction indicates that 1 mole of CaCO₃ is consumed for the neutralization of each mole of H^+ (see also Appendix 6.2.3-2). Consequently, the molar ratio of carbonate (corrected alkalinity) to sulphate in the column water should be 2:1, or 2.08:1 on a gram basis.

The experimental data from Columns 2 and 3 (Figure 6.2.3-3) showed that the ratio of sulphate to corrected alkalinity as CaCO₃ (Appendix 6.2.3-1) in weekly samples from the columns was close to 2:1, except when sulphate production rates were greater than 1 g/wk. The high production rates occurred in the initial weeks of testing when sulphate release was associated with dissolution of gypsum (CaSO₄ \cdot 2H₂O) rather than acid generation. As gypsum was flushed out of the columns, the relationship of sulphate and CaCO₃ approached the 2:1 ratio. This was confirmed by calcium concentrations (Appendix 6.2.3-1), which were equivalent to sulphate on a molar basis (1 Ca: 1 SO₄) in initial weeks and were later equivalent to corrected alkalinity (1 Ca: 1 CO₃).

Although sulphate and carbonate production rates from Columns 2 and 3 (Figure 6.2.3-3) were frequently close to the theoretical 2:1 ratio, many data points lie to the right of the line, indicating that sulphate was being generated at a rate of less than twice that of H^+ . This may indicate that the sulphide minerals do not have iron and sulphide in a 1:2 ratio (FeS₂). Either the sulphide minerals may contain less sulphide (FeS_{2-x}), which has been reported in the mineralogic literature, or there are metals other than iron in the sulphide (e.g., Fe_{1-x}Cu_xS₂).

In Column 4, CaCO₃:SO₄ ratios approached 2:1 and were similar in value to those of Columns 2 and 3. However, the ratios did not approach the 2:1 line (Figure 6.2.3-3) as closely as ratios for Columns 2 and 3. This may be a result of the lower quantity of limestone in Column 4 and less contact of column water with limestone, but there is not a significant discrepancy with the results for Columns 2 and 3.

Column 5, containing layers of limestone and rock, produced sulphate and CaCO₃ at rates similar to those of other columns following the initial phase of gypsum dissolution. The values approach the 2:1 line after gypsum removal which occurred during the first few weeks.

Trends of weekly corrected alkalinity loadings through time (Figure 6.2.3-4) show that loadings from Columns 2, 3, and 4 were essentially identical within the standard error of estimate of 0.04 to 0.07 g CaCO₃/wk. This indicated that the production rate for alkalinity was essentially independent of the limestone:rock ratio within the range of ratios tested in this experiment, probably because the rate of acid generation was similar in the three columns.

Column 5, with layers of high-carbonate and low-carbonate rock mixtures, generated alkalinity in a distinctly different manner from Columns 2 to 4 (Figure 6.2.3-4). An initial acceleration in alkalinity production was followed by a peak value at Week 11 and a subsequent decreasing trend in values. After Week 15, the weekly alkalinity production from Column 5 was similar in value and trend to the other columns, and consequently the primary difference in Column 5 was the lack of high alkalinity production in initial weeks. The cause of the depressed initial production is not known, but there was no effect on the overall capacity of the column to neutralize pH to values similar to other columns.

For extrapolation of alkalinity production and carbonate consumption into the future, the best-fit curve for Column 4 (Figure 6.2.3-4) was used because the limestone:rock ratio proposed for the on-land waste rock stockpile is less than 1.0%, as in Column 4. The right-side scale of Figure









CALCIUM CA vs SULP FOR LIMI	ARBONATE PRODUCTION HATE PRODUCTION ESTONE COLUMNS			
Figure no. 6.2.3-3 CINOLA GOLD PROJECT STAGE II ADDENDUM				
Date Aug. 1988	Drawn by Norecol			


6.2.3-5 is carbonate consumption normalized to 1 m² of rock surface based on 14 kg of rock in a column with an average surface area of 5.39 m²/kg. The calculation of cumulative carbonate consumption from the best-fit curve (Figure 6.2.3-5) indicated that all carbonate would be consumed by Week 284 in Column 4, whereas carbonate would be exhausted by Week 4290 in Column 2 (high carbonate). (These depletion rates are directly related to the size distribution of the column rock.) On the other hand, if the limestone were not successful in slowing the rate of acid generation, and acid generation proceeded at the higher rate observed in Column 1, the carbonate in Column 4 would be consumed by Week 6 and in Column 2 by Week 2010, according to the extrapolation equation in Section 3.7, Volume V. This highlights the advantage of distributing the limestone throughout the rock mass and constantly maintaining neutral-pH conditions.

Because the acid generation extrapolation equation in Volume V assumed that acidity would be neutralized at a ratio of 1 CaCO₃:1 SO₄ (neutralized pH < 6.4) rather than 2 CaCO₃:1 SO₄ (neutralized pH > 6.4) as found in these column experiments; the CaCO₃ in the column may, therefore, be consumed in half the time period of those estimated by the equation. The on-going experiments will provide further information on this for Stage III.

After all carbonate available in the limestone is consumed (Figure 6.2.3-5), acidic conditions could develop, depending on the residual rate of acid generation at the time of carbonate depletion. Complexities such as the non-linear behaviour of aqueous pH buffering, the effects of interactive neutralization reactions, and the degree to which water moving through the rock may contact carbonate will also influence the point at which acidic conditions may develop in rock drainage. Further evaluation of these factors is an objective of on-going column experiments.

6.2.3.4 Acidity

Acidity is a generalized parameter reflecting concentrations of aqueous species which can consume OH⁻ or release H⁺ (including species such as free Fe³⁺, Al³⁺, HSO₄⁻, and H⁺). Because acidity to pH 4.5 is measured by titration from a pH<4.5 in a sample up to pH 4.5, it is, by definition, zero whenever sample pH is greater than pH 4.5. Acidity to pH 8.3 is similarly zero whenever sample pH is greater than pH 4.5. Acidity concentration represents a summation of concentrations of several aqueous species which have different chemical behaviour in water, acidity cannot always be used for quantitive calculations of acid generation.

Acidity in effluent from Column 1 (Figure 6.2.3-6) was initially lower than sulphate (Figure 6.2.3-2) because of initial neutralization by the inherent neutralization potential of the rock and the relatively low level of acid generation. By Week 11, acidity and sulphate concentrations in Column 1 were similar. The correspondence of acidity to sulphate after initial neutralization was also noted in experiments described in Section 3.0, Volume V.

Acidities from Columns 2 to 5 compared to Column 1 (Figure 6.2.3-6) demonstrated the success of limestone in suppressing acidity in the effluents. For these columns, acidity did not reflect sulphate concentrations (Figure 6.2.3-2) but were significantly less than sulphate, reflecting the





neutralization reactions. The increase in acidity in Columns 2 to 5 towards Week 24 indicated that one or more of the aqueous species detected by the measurement of acidity to pH 8.3 was increasing in concentration. No significant increases in concentrations of aqueous species in the column drainage which could explain the increase in acidity were, however, observed (Appendix 6.2.3-1). Furthermore, the concentrations and temporal trends of acidity in all four columns were similar, indicating the limestone:rock ratio had no effect. Because the acidity concentrations were relatively low and the acidity titration has inaccuracies and uncertainties at these low concentrations (Vos pers. comm.), the trend of increasing acidity may not be real. In any case, there was no significant changes in water quality associated with acidity in Columns 2 to 5 up to Week 24.

6.2.3.5 Iron

Because the oxidation of pyrite (FeS₂) releases 1 mole of Fe into the water for every 2 moles of S, concentrations of iron should be approximately 0.88 mg Fe/L for each 1 mg SO₄/L. Under anoxic (reducing) conditions, the iron will remain in the ferrous (2+) state and be washed from the rock by drainage waters. Alternatively, if well oxygenated (oxidizing) conditions are present, the ferrous iron will oxidize to ferric (3+) iron (Reaction 2, Section 2.1, Volume V). The oxidized iron will precipitate as Fe(OH)₃ (Reaction 3, Section 2.1, Volume V) if pH is greater than 3.0 to 3.5 and will generate 3 moles of H⁺ for each precipitated mole of iron.

For Column 1, iron concentrations increased from 0.045 mg/L at Week 1 to a peak of 2020 mg/L at Week 17 (Appendix 6.2.3-1). In particular, iron concentrations increased above 10 mg/L as pH decreased to 3.1 and lower. This was the result of ferric iron becoming stable in water as the pH decreased. Nevertheless, the ratio of iron:sulphate did not reach the theoretical ratio of 0.88:1.00, indicating that some iron consistently precipitated and was retained in Column 1. This precipitate likely contributed to the depressed pH by contributing 3 moles of H⁺ for each precipitated mole of Fe³⁺.

Iron concentrations in Columns 2 to 5 were particularly noteworthy because they rarely exceeded 1 mg/L despite sulphate concentrations of tens to hundreds of mg/L (Appendix 6.2.3-1). The highly oxygenated and pH-neutral conditions in the columns caused the ferrous iron to convert to ferric iron and precipitate as $Fe(OH)_3$. As a result, essentially all the iron was retained in the limed columns and the resulting H⁺ was apparently neutralized by the limestone. This reaction was discussed earlier in Section 6.2.5.2 and Figure 6.2.3-2, where the ratio of $CaCO_3:SO_4$ approached 2:1, which required the precipitation of $Fe(OH)_3$ and the neutralization of the resulting H⁺.

If strongly reducing conditions had existed in the columns, the iron would not have precipitated at neutral pH, the CaCO₃:SO₄ ratio would have been around 1:1, and the ferrous iron would have been flushed from the column. Once the drainage had been exposed to the atmosphere outside the column, the iron would have oxidized and precipitated, aqueous iron concentrations would have decreased, H^+ would have been generated, pH would have decreased, and the orange-brown Fe(OH)₃ precipitate would have settled out of the water. At this point, analytical measurements would have indicated that the water had an acidic pH, little iron, and no alkalinity, which would not

reflect the true nature of the drainage. This problem was not encountered in the columns because of oxidizing conditions, but was apparently encountered in the on-site barrels (Section 6.3).

6.2.3.6 Metal leaching

Section 4.0, Volume V, demonstrates the role of pH in regulating metal concentrations. In all of these previously-reported experiments, relatively high metal concentrations were obtained at acidic pH, relatively low concentrations were obtained at neutral pH, and low or somewhat elevated concentrations were obtained at alkaline pH. The metals that frequently occurred at higher concentrations at alkaline pH were aluminum, antimony, arsenic, and mercury. It is noted that these concentrations, while higher than those at neutral pH, were consistently below B.C. mine effluent guidelines as defined by the "Pollution Control Objectives for the Mining, Smelting and Related Industries of British Columbia" (Pollution Control Board 1979).

In drainage from Columns 1 to 5, concentrations of aluminum, antimony, arsenic, cobalt, chromium, copper, lead, nickel, and zinc were compared to concentrations from experiments in Volume V (Table 6.2.3-3). The comparison showed that metals concentrations vs pH were similar and, therefore, no further discussion is provided in this report. However, cadmium, mercury, selenium, and silver were targeted for further examination.

Cadmium concentrations (Figure 6.2.3-7) of up to nearly 0.1 mg/L at acidic pH and of less than 0.01 mg/L at neutral pH were similar to those from short-term leach column experiments and waste rock pads discussed in Volume V. However, cadmium concentrations from the limestone columns above pH 7 were sometimes elevated to 0.155 mg/L (Column 2) over those at pH 6 to 7. The occurrence of elevated concentrations of cadmium at alkaline pH was not noted in Volume V, but based on these studies, cadmium should now be added to the list of metals with slightly elevated concentrations at alkaline pH. Nevertheless, except in one instance in drainage from Column 2, the cadmium concentrations at alkaline pH did not exceed the B.C. Pollution Control Objectives for metal mine effluents (Pollution Control Board 1979) which set maximum concentrations of cadmium ranging between 0.01 to 0.1 mg/L.

Mercury concentrations (Figure 6.2.3-8) were noteworthy in the limestone columns because they were significantly lower at acidic and alkaline pH than in previously reported experiments (Volume V). Concentrations of mercury in samples of effluent from the limestone columns were below the B.C. Pollution Control Objectives for metal mine effluents (Pollution Control Board 1979) in all cases, and were below detection limits in all except one case. The lower concentrations of mercury from the limestone columns as compared to earlier studies are reflected in the vertical scales of concentration, which are in hundredths of ug/L in Figure 6.2.3-8 and in fifths of ug/L in Figures 4.3.9-1 and 4.3.9-2, Volume V.

TABLE 6.2.3-3

METAL	FIGURES IN VOLUME V		
Aluminum	4.3.1-1, 4.3.1-2		
Antimony	4.3.2-1		
Arsenic	4.3.3-1, 4.3.3-2, 4.3.3-3		
Cadmium ^a	4.3.4-1, 4.3.4-2		
Cobalt	4.3.5-1		
Chromium	4.3.6-1		
Copper	4.3.7-1, 4.3.7-2		
Lead	4.3.8-1		
Mercury ^a	4.3.9-1, 4.3.9-2, 4.3.9-3		
Nickel	4.3.10-1		
Selenium ^a	No figure, see Section 4.3.11 of Volume V		
Silver ^a	No figure, see Section 4.3.12 of Volume V		
Zinc	4 3 13-1 4 3 13-2 4 3 13-3		

Norecol Volume V, Stage II Figures Depicting Metal Concentrations vs pH for the Cinola Gold Project

^a Additional diagrams presented in this section of the Addendum.





Selenium concentrations (Figure 6.2.3-9) were examined in detail because of the relatively limited data presented in Volume V. Concentrations in column drainage were often close to or below detection limits. At acidic pH, concentrations approached 0.01 mg/L. Between pH 3 and 7, concentrations were below detection limits of 0.001 mg/L. Like some other metals, such as arsenic, the concentrations of selenium at pH >7 increased above detection to 0.003 mg/L. These results are in agreement with the data from Volume V. In all cases, concentrations were less than the B.C. Pollution Control Objectives for metal mines.

Silver concentrations (Figure 6.2.3-10) were examined in detail because values close to a detection limit of 0.002 mg/L were frequently encountered in earlier experiments as reported in Section 4.3.1.2, Volume V. The detection limit used for effluent analysis in this experiment was 0.0002 mg/L. Concentrations below pH 3.5 reached 0.0018 mg/L, whereas concentrations above pH 3.5, including alkaline pH, were close to or below detection. These concentrations were notably less than the limited data of Volume V; however, all concentrations were consistently less than the B.C. Pollution Control Objectives for metal mine effluents.

6.3 On-site Barrel Experiments

6.3.1 Introduction

In order to test the effectiveness of limestone in neutralizing acid generation in waste rock under on-site conditions, an experiment was designed using barrels containing limestone:rock mixtures. In this experiment, the barrels, located at the project site, were open to the atmosphere and water input was determined by natural precipitation and evaporation. This uncontrolled condition is more extreme than that anticipated in the waste rock stockpile, which will be capped to control infiltration and evaporation. This experiment was also designed to test the effect of alternating layers of rock and limestone, which allows the rock to generate acid at the full, unhindered rate and results in a higher rate of limestone consumption.

The experimental design was intended to mimic conditions in the limed waste rock stockpile, however the airtight nature of the plastic barrels themselves and the drainage system that was used, combined with the low hydraulic conductivity of the fine grained waste rock samples used, caused a water table and an anoxic zone to form in the barrels. As a result, this experiment unintentionally demonstrates the effect of allowing anoxic (reducing) conditions to develop in the rock mass with the consequent severe change in water quality.

The observation of this condition in the experimental work served to confirm the soundness of the waste rock stockpile design. The coarse rock in the stockpile and the stockpile basal drain will provide free draining conditions and the placement of a gas permeable cap on the stockpile (mudstone) while minimizing water infiltration will allow oxygen migration into the stockpile. The active face which will not be capped will provide an additional pathway for oxygen migration into the stockpile.





6.3.2 Methods

6.3.2.1 Sample selection

The only source of fresh rock on-site at the start of the experiment was the adit. Rock was collected from the underground workings where ponded water covered loose rock and isolated it from the atmosphere. Because the adit does not provide access to all rock groups, a rock mixture exactly similar to that of the limestone columns and proposed waste rock stockpile could not be obtained. Instead, an undefined mixture of Skonun sediments and multiphase breccia was prepared from the available loose rock as each barrel was filled.

Grain size of the loose rock was heterogeneous throughout each barrel and thus could not be accurately determined. Visual estimates suggested that on average the grain size distribution by weight was 15% greater than 2.0 cm diameter, 60% between 2.0 and 0.5 cm, and 25% less than 0.3 cm, including silt and perhaps some clay. The rock mixture was cohesive because of the fine-grained material and likely had low hydraulic conductivity. By comparison, rock in the waste rock stockpile is expected to range from 2.0 to .0001 m in diameter with approximately 20% of the rock greater than 32 cm in diameter, 60% between 3.2 and 0.5 cm and 20% finer than 0.5 cm which will result in much greater conductivities than occured in the barrel tests.

Acid-base accounting of the rock fill in each barrel (Table 6.3.2-1) demonstrated the noticeable variability in the geochemical nature of the rock. As a result, the rates of acid generation were expected to vary somewhat among the barrels. Furthermore, the paste pH values below 5.0 and the negative values of neutralization potential indicated that accumulated acid products were held within the rock. The initial release of this stored acidity dissolved some of the limestone and, in the case of Barrel 4 with 0.55 t CaCO₃/1000 t of rock, the majority of the limestone was consumed through neutralization of the initial acid release.

The limestone used in this experiment was identical to that used for the limestone columns. The description and characterization of this limestone are given in Section 6.2.2.

6.3.2.2 Equipment, procedures, and sampling program

Four plastic barrels and water collection systems were assembled on-site, as depicted schematically in Figure 6.3.2-1. The barrels were approximately 0.46 m in diameter and 0.71 m high. The tops of the barrels were open to the atmosphere.

A 5-cm layer of fine sand (residual from the supply used in waste rock pad construction) was placed in the bottom of each barrel, covering the entrance to the collection tubing. Approximately 100 kg of rock and limestone were placed in each barrel to a depth of about 0.4 m. No limestone was added to Barrel 1, which served as the experimental control. Limestone was added to Barrels 2, 3 and 4 in different ratios ranging from high carbonate content in Barrel 2 to low carbonate content in Barrel 4. The distribution of limestone and rock is presented in Table 6.3.2-2. The limestone:

TABLE 6.3.2-1

Acid-base Accounting of the Rock Used in the Norecol Limestone Barrel Experiments, Cinola Gold Project

					t CaCO ₃ /1000 t		
SAMPLE	PASTE pH	TOTAL SULPHUR (% S)	SULPHIDE (% S)	LEACHABLE SULPHATE (% S)	MAXIMUM POTENTIAL ACIDITY	NEUTRALIZATION POTENTIAL	NET NETRALIZATION POTENTIAL
Barrel 1	4.7	2.35	2.17	0.14	73.4	-2.00	-75.4
Barrel 2	4.7	2.12	1.99	0.11	66.3	-3.00	-69.3
Barrel 3	4.6	2.78	2.47	0.16	86.9	-2.00	-88.9
Barrel 4	4.8	2.44	2.21	0.13	76.3	-1.00	-77.3



TABLE 6.3.2-2
Rock:Limestone Proportions In Norecol Limestone Barrel Experiments, Cinola Gold Project

BARREL	APPROX. QUANTITY OF ROCK (kg)	RATIO OF LIMESTONE:ROCK (%)	COMMENTS
1	100	-	Experimental control
2	100	5.5	High-carbonate barrel (corresponds to Limestone Column 1)
3	100	0.55	Low-carbonate barrel (corresponds to Limestone Column 3)
4	100	0.055	Very low carbonate barrel

rock ratios for Barrels 2 and 3 were chosen to correspond to the ratios of limestone Columns 2 and 3 respectively. Barrel 4 with very low carbonate content was designed to monitor complete limestone consumption within a relatively short period of time.

The barrels were filled by placing several kilograms (several cm depth) of rocks into a barrel, sprinkling the required weight of limestone on top of the layer, then adding another layer of rock and limestone, etc. This layering of limestone allows the full, unhindered rate of acid generation to occur in the rock layers and then tests the capability of the limestone layers to neutralize the high acidity and acid pH. For Barrel 4, each limestone, "layer" consisted of a few grams of limestone which was insufficient to form a continuous layer and was, in fact, hardly noticeable. Nevertheless, the purpose of Barrel 4 was to test whether such a negligible addition of limestone would have a detectable effect on water quality. (Subsequent acid-base accounting analysis indicated that the initial acid release would likely consume all of the limestone.)

Based on an average annual precipitation of 1.6 m/a, the average daily rainfall on each barrel (0.46 m diameter) was estimated to be in the order of 0.74 L/d, or 0.52 mL/min. This flow rate is a factor of about 86 times less than the flow rate in the limestone columns. This flow rate was sufficiently high to cause a water table to develop in the barrels, however, and it is believed that anoxic (reducing) conditions evolved in the barrels. The quantity of water draining into the collection buckets was often negligible, particularly following days of little or no precipitation and high evaporation. Consequently, water quality data are sporadic. Measurements of pH are available for numerous days during the experiment; however, collection of the sample for pH measurement took almost a full day, and therefore some deviation from in-situ pH no doubt occurred. Sufficient water for metal analyses had to be collected over several days, and the resulting iron staining in the bucket indicated that some metal concentrations and pH had probably decreased from in-situ conditions.

6.3.3 Results

The pH values from each barrel (Figure 6.3.3-1 and Appendix 6.3.3-1) were relatively high initially, but decreased sharply within 25 days. The comparison of pH values from Barrel 4 (very low limestone content) and Barrel 1 (experimental control, no limestone) demonstrated that a limestone content of 0.055% had essentially no neutralizing effect on pH under the experimental conditions. This is attributed primarily to the neutralization of the initial acid release (accumulated acid generation products stored on the rock surface), which likely consumed all of the limestone.

The expected times for total limestone consumption at the full, unhindered rate of acid generation in Barrels 2 and 3 were calculated prior to the experiment to be approximately 120 and 2200 weeks, respectively. The comparison of pH values from these barrels with Barrel 1 suggested that limestone was being consumed in a significantly shorter period of time. While neutralization of the initial acid release had probably consumed some limestone, this was not considered sufficient to account for the observed acid pH after only several weeks. Because encapsulation of limestone by metal precipitates, particularly by Fe(OH)₃, was suspected, one side of each barrel was excavated to the bottom on days 147 and 148 of the experiment. Visual observations indicated



that orange-brown iron staining had developed in rock layers near the limestone layer, although the degree of iron staining reduced significantly with depth. The limestone, however, was not stained and still displayed the original gray colouring. Furthermore, tests with dilute HCI showed that the limestone was readily reactive (often within 1 s) throughout the depth of the barrels. Therefore there were still significant quantities of reactive limestone in Barrels 2 and 3 despite the acidic pH values in the effluent.

An unexpected observation during the excavation of one side of the barrels was the very moist condition of the rock and, in fact, the presence of a water table in Barrel 2. The implication of the moisture is that pore space was significantly filled with water, limiting the amount of oxygen reaching the rock at depth in the barrel. This probably accounted for the decrease in iron staining with depth.

There was obviously sufficient oxygen at the top of the barrel to oxidize both sulphide minerals and ferrous iron (Reactions 1 and 2, Section 2.2, Volume V), and ferric iron was precipitated at neutral pH (Reaction 3). Lower in the barrels there was probably only sufficient oxygen to oxidize ferrous iron. Consequently, iron was not precipitated and retained in the lower portions of the barrels, but was flushed from the barrels with the drainage water. Because the limestone was reactive throughout Barrels 2 and 3, the pH of the effluent was likely neutral. It is not unusual for low oxygen groundwater to carry up to thousands of mg/L of ferrous iron at neutral pH after carbonate has neutralized acid drainage (e.g., Morin et al. in press).

The discrepancy in the above scenario is that acidic pH values, not neutral values, were measured in the effluent of Barrels 2 and 3. Visual inspection of the effluent tubing and the collection buckets confirmed that significant quantities of iron consistently precipitated from the effluent. For this to occur, the ferrous iron must oxidize to ferric iron (upon exposure to the air in the tubing) and precipitate as Fe(OH)₃ (Reaction 3, Section 2.2, Volume V). For each 100 mg/L of precipitated iron, there is a production of 5.4 mmol/L of H⁺, which corresponds to a pH of 2.3. Aqueous buffering reactions would moderate this pH decrease to a value greater than 2.3. The experimental results demonstrated the potential to create acidic pH in water with neutral pH, low oxygen, and high ferrous iron as it moves from a reducing environment to an oxidizing environment. In the upper portions of the barrels, where iron oxidized and precipitated in place, the H⁺ was neutralized by the limestone.

Excavation of the barrels allowed oxygen to reach the bottom of the barrels, resulting in precipitation of some iron in the barrel and neutralization of acidity by the limestone. This was indicated by an increase in pH in Barrel 2 drainage water after Day 150 (Figure 6.3.3-1). However, pH values continued to be altered to some degree by iron precipitation in the tubing and collection buckets.

There are indirect data to determine the possible pH of the effluent from Barrels 2 and 3 prior to iron oxidation and precipitation. These data are concentrations of sulphate, iron, and other metals. Sulphate can be used as a general indicator of pH (e.g., Figure 3.7.2-4, Volume V) because the rate of acid generation in Cinola rock decreases with increasing pH. Sulphate can be used as an indicator providing that (1) all sulphur in the effluent exists as sulphate, (2) all accumulated sulphate is removed by the drainage water, and (3) gypsum solubility does not limit sulphate concentrations.

6-38

All sulphur is assumed to exist as sulphate in the effluent based on the experiments of Volume V and the limestone columns. However, because the water flushing rate of the barrels was approximately 86 times less than that of the limestone columns, it is expected that not all acid-generated sulphate was removed from the barrels. The results of the on-site waste rock pad experiments (Section 3.4, Volume V) indicated that incomplete flushing occurs during mid-year months in particular. Furthermore, geochemical calculations indicate that gypsum solubility was limiting sulphate concentrations in the effluent. Consequently, sulphate could not be used as an indicator of pH, nor as a reliable indicator of acid generation in this experiment.

Iron has been used in other experiments as an indicator of pH (Figure 3.7.2-6, Volume V), where pH decreases as iron increases. This trend is applicable to ferric iron under oxygenated conditions, however, and is not applicable to this experiment. Furthermore, precipitation of iron in the tubing and collection buckets precluded determination of the original effluent concentration.

The relationship between pH and concentrations of metals such as zinc is discussed in Section 4.0, Volume V. Relatively high concentrations are obtained at acidic pH, whereas minimal concentrations are obtained at neutral and/or alkaline pH.

Concentrations of dissolved metals in Barrel 1 effluent (Appendix 6.3.3-1) were relatively high at more than 1 mg/L for most metals and were comparable to those reported in Section 4.0, Volume V. Metals concentrations from Barrel 4 (very low limestone content) were similar to those of Barrel 1, but were often somewhat less in value by factors of 2 to 4. Nevertheless, the Barrel 4 metals concentrations suggest acidic pH values closest to those of Barrel 1. The concentrations of dissolved metals from Barrel 3 (low limestone content) are significantly lower than those of Barrel 1, suggesting pH values greater than those of Barrel 1. The pH values suggested by Barrel 3 metals concentrations cannot be accurately determined, but are probably no greater than 4.0 to 5.0 because of elevated aluminum concentrations. Finally, dissolved metal concentrations from Barrel 2 up to and including April 21 (Appendix 6.3.3-1) were relatively low and reflected the measured pH values of 5.9 to 7.3. On May 27, during the period of excavation activity, measured pH was 3.3 and the elevated metal concentrations reflected a lower, more acidic pH. However, on June 23, dissolved metal concentrations essentially returned to March 22 levels at pH 7.3.

In conclusion, the development of reducing conditions within the barrels greatly complicated the geochemical evaluation of the experiment, but emphasized the importance of maintaining oxidizing conditions in the proposed waste rock stockpile and confirmed the suitability of the proposed stockpile design. The stockpile basal drainage layer and provision of a gas permeable cap to minimize water infiltration will ensure oxidizing conditions in the stockpile such that iron will precipitate within the rock mass and that all acidity will be neutralized by the limestone.

A more detailed evaluation of data, as performed for the limestone columns and the experiments of Volume V, has been limited by a number of factors, including the low flow rates through the barrels (which limited sampling for water quality analysis), the limitation of gypsum solubility, and the alteration of pH and some metal concentrations in the barrel drainage water within the drainage collection systems.

6.4 Discussion

Five laboratory limestone column tests provide information on the potential neutralization of acidity and the control of pH in waste rock by limestone addition. Column 1, the experimental control with no added limestone, demonstrated that the waste rock mixture in the columns was capable of unhindered acid generation at rates similar to those reported in Section 3.0, Volume V. All data from Columns 2 through 5, containing 0.84% to 6.6% added limestone by weight, demonstrated the success of limestone addition in controlling acid generation. The limestone maintained pH in the drainage water at neutral values (6.8 to 8.2) and essentially neutralized all acidity generated by the waste rock. The limestone also lowered the rate of acid generation by a factor of almost 19 and minimized metal concentrations in the effluents through pH control. The lower rates of acid generation were consistent with those reported in Volume V for pH-neutral conditions.

Metal concentrations in the effluents were consistent with those expected based on studies reported in Volume V, except for minor differences with cadmium, mercury, selenium, and silver. Cadmium concentrations were found to be somewhat higher at alkaline pH than at neutral pH, which was not noted in Volume V; mercury concentrations were significantly lower at acidic and alkaline pH than reported in Volume V; concentrations of selenium at alkaline pH were generally higher than at neutral pH, which was not noted in Volume V because of limited data; and silver concentrations were found to be significantly less than those reported in Volume V with the lower detection limit used in this experiment.

Because the pH of the drainage water from the limestone columns was consistently greater than 6.4, the appropriate neutralization reaction indicated that 1 mole of CaCO₃ would consume only 1 mole of H⁺ (see Appendix 6.2.3-2). This compared to an assumed consumption rate in Volume V of 2 moles H₊ per mole of CaCO₃ assuming pure CaCO₃. As a result, alkalinity values were multiplied by a factor of 2 to calculate real carbonate consumption. Geochemical calculations and a mass balance confirmed that the corrected alkalinity values represented bicarbonate concentrations. Concentrations of corrected alkalinity and sulphate in drainage from the columns were close to the theoretical value of 2:1 after the initial removal of gypsum, indicating that both sulphide oxidation and ferric iron precipitation were generating acidity within the columns and that the limestone was actively neutralizing the acidity.

The temporal trends of carbonate consumption in Columns 2, 3, and 4 throughout the experiment and in Column 5 after Week 15 were essentially the same, indicating that carbonate consumption was independent of the limestone concentration through Week 24. Extrapolation of carbonate consumption into the future was made with the following equation:

Weekly Consumption (g CaCO₃/wk) = 1.298 WEEK^{-0.2517}

Using this equation and the limestone addition rates and rock grain size of Column 4 as an example, the projected cumulative consumption of carbonate in Column 4 (0.84% limestone by weight) would exhaust available carbonate within 284 weeks. This is significantly longer than the time period

previously projected for carbonate consumption if the rate of acid generation were not to be significantly slowed by the pH-neutral conditions.

The proposed waste rock stockpile will contain 25 700 000 t of rock with a total surface area of 6 550 000 000 m², based on a unit surface area of 0.255 m²/kg (Section 2.1.2.2, Volume V). The previously calculated requirement for limestone addition to the stockpile was 49 480 t of CaCO₃ (the quantity required if acid generation were to occur at the full rate, Table 2.1.2-3, Volume IV). The resulting average ratio of limestone to rock surface area is 7.55 g CaCO₃/m². The right-side scale of Figure 6.2.3-5, representing the rate of carbonate consumption at neutral pH's indicates that this carbonate will not be consumed until Week 2310 (44 years). In other words, because of the neutral pH, at the previously calculated rate of addition, the carbonate is expected to be effective for 44 years rather than only for life of mine, as predicted in Stage II. This approximation is conservative in that it assumes that all rock and limestone is placed immediately in the stockpile. Based on the staged waste rock distribution schedule of Table 2.1.2-3, Volume IV, the lifespan of the carbonate is more realistically calculated to be approximately 65 years.

The limestone columns suggest that drainage from the stockpile will have a pH greater than 6.4 and, thus, the molar ratio of CaCO₃:SO₄ will approach 2:1. Consequently, the limestone in the stockpile will be consumed twice as fast, but will still remain for at least 25 years, which exceeds the predicted 14-year lifespan of the stockpile before it is backfilled to the pit. Factors that could shorten the minimum 25-year lifespan of limestone in the stockpile include (1) increased grain surface area of the waste rock, (2) an increased rate of acid generation if pH-neutral conditions were not maintained in the stockpile, and (3) non-ideal conditions such as insufficient contact of water and limestone and various non-linear buffering reactions. Continuing and future testwork for Stage III will address these potential concerns.

The four on-site barrels provided information on the effects of (1) reducing conditions in the rock mass, (2) low, uncontrolled flow rates, (3) the encapsulation of limestone, and (4) layering of rock and limestone. Within several weeks after initiation, the pH values in drainage from all barrels dropped to acidic values despite limestone:rock ratios similar to the limestone columns. This result was traced to the reducing conditions in the lower portion of the barrels, which allowed ferrous iron to be flushed from the barrels. Subsequent oxidization and precipitation of ferric hydroxide in the tubing and collection bucket generated a low pH in the drainage water. This emphasizes the importance of maintaining oxidizing conditions in the stockpile. The proposed design of a coarse, free-draining stockpile with a coarse underdrain is expected to maintain oxygenated conditions. Nevertheless, the mine plan continues to include provisions for the collection of effluent from the underdrain and to allow for optional treatment if water quality problems do arise.

The effect of low, uncontrolled flow rates through the barrels was rendered more complex by the low hydraulic conductivity of the rock mass and resultant retardation of water movement. The primary effect of the flow rate was the limitation of sulphate concentrations by gypsum solubility. This did not present a water quality problem but precluded an estimation of the acid-generation rate based on sulphate production.

The potential existed for encapsulation of limestone within the barrels by precipitation of gypsum and iron hydroxide. Examination of limestone distributed throughout Barrels 2 and 3 indicated that the limestone remained highly reactive and was not encapsulated after approximately 3 months.

The primary effect of layering the limestone and the rock in the barrels was expected to be increased rates of acid generation in the rock. However, because sulphate concentrations were limited by gypsum solubility, the rates could not be evaluated.

6.5 Conclusion

The laboratory limestone column experiments demonstrated that limestone effectively neutralized acidity from pyrite oxidation and also decreased the rate at which the oxidation occurred. The rate of oxidation was decreased by a factor of approximately 19. A mixture of limestone and rock at 0.84% limestone by weight was predicted to neutralize acidity for 284 weeks, for rock of the size distribution used in the column experiments. This rate of limestone consumption in the columns establishes the suitability of this proposed treatment technique for the waste rock stockpile. Previous calculations of limestone required to neutralize acidity in the waste rock stockpile (50 000 t) for the 14 year life of the mine assumed the full unhindered rate of acid generation in the stockpile. Based on the available data on reduced acid generation rates at neutral pH the 50 000 t of limestone is predicted to neutralize acidity for 25 to 44 years based on the expected surface area: weight ratio for rock in the stockpile, which is lower than the surface area: weight ratio in the limestone columns. (The range of 25 to 44 years is related to potential variation in the actual ratio of CaCO₃ consumed: SO₄ produced.)

Effluent concentrations of metals from the limestone columns were similar to those from experiments in Volume V. The exceptions were mercury and silver which were lower in the limestone column effluents than in other studies, and cadmium which shows slightly elevated concentrations at alkaline pH. Because of the similar concentrations to Volume V, the impact predictions for metal concentrations in drainage from the proposed stockpile (Section 2.2.2.3, Volume IV) remain valid.

The on-site barrel tests highlighted the water quality problems that can arise if reducing conditions develop within the rock mass. Reducing conditions allow the iron to be carried out of the rock in the drainage water and subsequently to oxidize, precipitate, and generate acid (H^+). Because this acid is generated outside the rock and limestone mass, it is not neutralized by the limestone and an acidic effluent can result although metal levels would be generally much lower than in typical acid mine drainage. The design of the proposed waste rock stockpile will allow free-draining conditions so that a water table will not develop. Furthermore, while the proposed cap for the stockpile is designed to minimize water infiltration and will reduce air infiltration, it is not expected to exclude oxygen to such a degree as to create reducing conditions. As a result, iron oxidation and precipitation is expected to occur within the stockpile and the drainage is expected to remain pH neutral.

Examination of rock in the barrels after several months demonstrated that the limestone was unaltered and remained highly reactive, so that encapsulation of limestone was not a problem. Consequently limestone encapsulation is not expected to be a problem in the proposed stockpile.

For the proposed waste rock stockpile, it is predicted that the Stage II calculated requirement of nearly 50 000 t of CaCO₃ would neutralize acidity for at least 25 to 44 years, assuming pH-neutral conditions are maintained in the rock mass. If acidic conditions were to develop resulting in an acceleration of the rate of acid generation, the proposed CaCO₃ quantity would still be sufficient to neutralize acidity for up to 14 years at a 1:1 ratio of CaCO₃:SO₄. Ongoing tests will enable better definition of the ratio of CaCO₃ consumption: SO₄ production over time.

SECTION 7.0

1

7.0 WATER TREATMENT PLANT

7.1 Introduction

The Cinola Gold Project deposit contains rock which can generate acidic water. The mine design includes several measures for short- and long-term control of acid generation. These include management of waste rock, surface water management, a water treatment plant and constructed wetlands.

Portions of the pit are expected to produce acid drainage due to contact of freshly exposed rock to air and water. To minimize the acidic water from the pit, plans have been made to divert surface and groundwater away from areas of active acid generation resulting in the discharge of this uncontaminated water to the environment after sediment removal. The remainder of the pit water and any poor quality seepage from the waste rock stockpile will be routed through a water treatment system which is designed to neutralize any acid, reduce phosphate, nitrate, sulphate, total dissolved solids (TDS), and metals to acceptable environmental levels.

The water treatment system will take in water with a low pH, high sulphate, phosphate, nitrate and metals and discharge treated water of sufficient quality to maintain aquatic life in a small salmonid spawning and rearing stream. Barbie Creek will receive discharge from the treatment system and during minimum flow periods up to 50 percent of the water entering this sensitive habitat will originate in the treatment plant. Water quality from the treatment plant is expected to be higher than background surface water concentrations in sulphates and calcium, with resulting high conductivity. TDS and hardness are also expected to be higher than background levels. The levels of dissolved solids and hardness found in this test work reduce metal toxicity to aquatic organisms significantly.

The water treatment plant, which is designed to neutralize the acidic water, precipitate metals as hydroxides, and phosphates and sulphates as slightly soluble calcium salts, has been described in the Stage II Report. Precipitate from this plant will be placed in the high pH environment of the active tailings impoundment. An equalization pond will be provided after the treatment plant to provide for mixing of effluent to yield a consistent discharge over time. It also acts as a holding pond for effluent which is out of specification so that it can be recycled through the treatment plant. As the final stage of the treatment system, a wetland will be provided to remove nitrates and to further reduce levels of any metals in the system. It is also expected to reduce sulphate and phosphate levels prior to discharge.

7.2 Bench Scale Treatment Tests

Bench scale tests have been performed to determine a number of important parameters for the design of a water treatment plant receiving acidic influent from the mine pit. Tables 7.2-1 and 7.2-2 (Tables 7.3.11-2 and 7.3.11-3 from Volume II of the Stage II Report) represent the expected quality

ie.

-

TABLE 7.2-1

Estimated Average Concentrations of Influent to the Mine Site Water Treatment Plant at the Cinola Gold Project

COMPONENT ^a	ACIDIC WATER	GROUND WATER	SURFACE WATER	INFLUENT COMPOSITION
Elow (m ³ /d) ^b	1220	500	50	1020
	26	500	50	1030
Acidity to pH 8.3(mg CaCOaL)	1867	7.0	5.0	542 7
Alkalinity to pH 8.5(mg CaCO ₃ /L)		56	3	
Conductance (umho/cm)	5767	300	62	3938
Sulphate (mg/L)	7821	10	11	5217
Total Phosphate(mg P/L)	2.977 ^c	1.9713	0.0438	2.589
Ortho Phosphate (mg P/L)	2.733 ^c	0.0560	0.0253	0.1512
As (mg/L)	15.1	0.031	0.003	10.01
AI (mg/L)	315	1.50	0.40	210
Cd (mg/L)	0.06	0.001	0.002	0.040
Co (mg/L)	7.1		0.002	1.134
Cr (mg/L)	0.19	0.005	0.001	0.128
Cu (mg/L)	2.5	0.004	0.0009	1.67
Fe (mg/L)	2233	1.98	0.90	1489
Hg (mg/L)	0.20	0.19		0.20
Mn (mg/L)	16.0		0.37	10.79
Ni (mg/L)	1.24	0.006	0.002	0.83
Pb (mg/l)	0.005	0.001	0.001	0.004
Zn (mg/L)	12.5	0.21	0.0060	8.40

а Concentrations reported as total metals.

ь

Based on average precipitation: Year 7. Value determined from SRK, Lime Treatment Studies for the Cinola Gold Project, May to June 1988. c

TABLE 7.2-2

Estimated Maximum Concentration of Influent to the Mine Site Water Treatment Plant at the Cinola Gold Project

COMPONENT ^a	ACIDIC WATER	GROUND- WATER	SURFACE WATER	INFLUENT COMPOSITION
Flow (m ³ /d) ^b	1220	560	50	1830
pH	1.9	7.0	5.8	2.0
Acidity to pH 8.3(mg CaCO ₃ /L)	24000			15983
Alkalinity to pH 8.5 (mg CaCO ₃ /L) ^c		56	3	
Conductance (umho/cm)	11100	300	62	7419
Sulphate (mg/L)	23331	10	11	15557
Total Phosphate (mg P/L)	2.977	1.9713	0.0438	2.589
Ortho Phosphate (mg P/L)	2.733 ^c	0.0560	0.0253	0.1512
As (mg/L)	77.0	0.031	0.003	51.30
AI (mg/L)	1420	1.50	0.40	947.1
Cd (mg/L)	3.37	0.001	0.002	0.25
Co (mg/L)	7.1		0.002	4.73
Cr (mg/L)	0.82	0.005	0.001	0.55
Cu (mg/L)	11.1	0.004	0.0009	7.40
Fe (mg/L)	7660	1.98	0.90	5180
Hg (mg/L)	0.54	0.19		0.42
Mn (mg/L)	95.0		0.37	63.46
Ni (mg/L)	5.8	0.006	0.002	3.87
Pb (mg/L)	0.030	0.001	0.001	0.019
Zn (mg/L)	60.0	0.21	0.0060	40.1

а Concentrations reported as total metals.

ь

Based on average precipitation: Year 7. Value determined from SRK, Lime Treatment Studies for the Cinola Gold Project, May to June, 1988 с

of water to be received as influent into the treatment plant. Estimated average influent characteristics are:

pН	<u>></u> to 2.0
Acidity	> 5000 mg /L.
Sulphate	>5000 mg /L
Arsenic	>10 mg /L
Aluminum	>200 mg /L
Copper	>1 mg /L
Iron	>1500 mg /1

The treatment process will be required to consistently improve this quality to that required by federal and provincial guidelines. In some cases this requires removal efficiencies of greater than 99.97%, (e.g. copper).

Bench scale tests, as described in Appendix 7.2, were designed to obtain information about removal efficiencies versus pH, total versus dissolved metals concentrations, in the effluent settlement characteristics, precipitate characteristics and metals concentrations, recycle of precipitate, reagent consumption, alternative reagent utilization and effects of influent variability.

7.2.1 Initial lime treatment tests

These tests, which were described in the Stage II Report, evaluated the process parameters such as removal efficiencies versus pH using lime as a precipitant, settling rates, dissolved versus total metal concentrations in the effluent, solids concentrations of the precipitate and reagent usage based on an acid/base titration of the solution.

The first test utilized drainage water from the Cinola site with acidity of 2545 mg/L, sulphate of 991 mg/L and iron of 238 mg/L, and copper 0.301 mg/L, all well below the average values expected from the pit during operation. Eight samples, were neutralized with lime, two samples to each pH value of 7.5, 8.5, 9.5 and 10.5. The precipitate was allowed to settle and one supernatant and one filtrate sample was taken at each pH and analyzed. The results showing total metal concentrations and dissolved metal concentrations of these tests are presented in Tables 7.2-3 and 7.2-4 respectively. These tables indicate that treatment efficiency is best for aluminum at a pH 8.5 which is to be expected from the solubility curves for aluminum. The solubility of aluminum increases above and below this pH. Lead has a minimum solubility at pH 9.5 which is reflected in the tables. The remainder of the metals show levels either below detection limits or decreasing effluent concentrations with increasing pH.

•

TABLE 7.2-3
Analysis of Supernatant from Lime and Settling Tests on Acid Drainage Sample from
Waste Rock at the Cinola Gold Project ^a

	TOTAL METAL CONCENTRATIONS ^b						
COMPONENT	INITIAL PH	FINAL PH					
	2.46	7.58	8.46	9.49	10.48		
Acidity to pH 8.3 (mg CaCO ₃ /L)	2545						
Alkalinity to pH 8.5 (mg CaCO ₃ /L)		50	50	44	37		
Sulphate (mg/L)	991		942	925	942		
Total Phosphate (mg P/L)	2.977	0.080	0.186	0.087	0.0835		
Ortho Phosphate (mg P/L)	2.733		0.021	0.005	<0.001		
Ag (mg/L)	<0.005	<0.005	<0.005	<0.005	<0.005		
As (mg/L)	1.23	0.021	0.026	0.027	0.029		
AI (mg/L)	17.3	0.18	<0.125	0.41	1.09		
Cd (mg/L)	0.008	<0.001	<0.001	<0.001	<0.001		
Co (mg/L)	0.25	<0.2	<0.2	<0.2	<0.2		
Cr (mg/L)	0.020	0.132	0.0030	0.0014	<0.001		
Cu (mg/L)	0.301	0.034	0.015	0.014	0.015		
Fe (mg/L)	238	4.41	4.88	5.70	5.70		
Hg (mg/L)	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003		
Mn (mg/L)	0.45	<0.2	<0.2	<0.2	<0.2		
Ni (mg/L)	0.13	<0.05	<0.05	<0.05	<0.05		
Pb (mg/L)	0.002	0.013	0.002	<0.001	0.002		
Zn (mg/L)	1.38	0.20	0.34	<0.05	<0.05		

Lime Treatment Studies for the Cinola Gold Project, May to June 1988
Unfiltered Supernatant

	DISSOLVED METAL CONCENTRATION ^b						
COMPONENT	INITIAL pH	FINAL pH					
	2.46	7.46	8.47	9.57	10.59		
Acidity to pH 8.3 (mg CaCO ₃ /L)	2545						
Alkalinity to pH 8.5 (mg CaCO ₃ /L)	***	50	42	17	80		
Sulphate (mg/L)	991	1057	1140	1016	1011		
Total Phosphate (mg P/L)	2.977	<0.005	<0.005	0.042	0.006		
Ortho Phosphate (mg P/L)	2.733	0.005	<0.001	0.010	<0.002		
Ag (mg/L)	<0.005	<0.005	<0.005	<0.005	<0.005		
As (mg/L)	1.23	<0.0001	0.0002	0.0006	0.0017		
AI (mg/L)	17.3	<0.125	<0.125	0.18	0.86		
Cd (mg/L)	0.008	<0.001	<0.001	<0.001	<0.001		
Co (mg/L)	0.25	<0.2	<0.2	<0.2	<0.2		
Cr (mg/L)	0.020	<0.0001	0.0004	0.0026	<0.0001		
Cu (mg/L)	0.301	0.010	0.005	0.009	0.005		
Fe (mg/L)	238	<0.2	<0.2	<0.2	<0.2		
Hg (mg/L)	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003		
Mn (mg/L)	0.45	<0.2	<0.2	<0.2	<0.2		
Ni (mg/L)	0.13	<0.05	<0.05	<0.05	<0.05		
Pb (mg/L)	0.002	0.002	<0.001	<0.001	0.002		
Zn (mg/L)	1.38	0.11	<0.05	<0.05	<0.05		

TABLE 7.2-4

Analysis of Filtrate from Lime And Settling Tests on Acid Drainage Sample from Waste Rock at the Cinola Gold Project^a

Lime Treatment Studies for the Cinola Gold Project, May to June 1988. Filtered through 0.45 um filter a

ь

These results probably reflect metal absorption onto the ferric hydroxide precipitate formed during neutralization of the solution. A comparison of Tables 7.2-3 and 7.2-4 illustrate the differences between total and soluble metal concentrations in solution. For instance, in the pH range of 8.5 to 10.5 copper concentrations between 0.014 and 0.015 mg/L are found in total concentrations while soluble concentrations range between 0.005 and 0.009 mg/L. The differences between the two are attributed to suspended particulates not removed in the settling process. If it is assumed that the iron in Table 7.2-3 is all particulate with the same copper composition as the precipitate, then the total copper concentration can be calculated from a total suspended solids concentration of approximately 12 mg/L plus the soluble copper in Table 7.2-4. Therefore, all the copper can be accounted for as either soluble copper or particulate copper contained in suspended precipitate particles.

This illustrates the fact that if the very low levels of metals required by the guidelines are to be achieved in the receiving waters every effort must be given to separation of the solids from the liquid effluent.

Other information pertaining to the settling characteristics of the precipitate, its chemical composition, its solids content and reagent consumption were presented in the Stage II Report (Volume II, Section 7.3).

7.2.2 High density precipitate system tests

A high density precipitate (HDP) system must be designed into the water treatment process to minimize the water that would be placed in the tailings impoundment with the disposal of the precipitate.

Neutralized iron-rich solutions commonly produce low density precipitates which are difficult to handle. Typically, solids densities of 10 percent are common but in some cases densities as low as 2 percent are encountered. The use of the high density system can produce precipitates in the 25 - 50 percent solids range.

The simplest type of HDP system requires a recycle of precipitate from the clarifier/thickener underflow pumpback into the center well of the clarifier where it contacts additional flocculent and precipitate from the front end of the process. This simple HDP system allows for additional time in the thickener for compaction and dewatering of the precipitate. In a more sophisticated version of the HDP system, the recycle of underflow is back to the lime mix tank where conditioned precipitate contacts the lime slurry prior to being reintroduced into the process.

The tests run for the Cinola Gold Project were designed to test the effect of precipitate recycle into the lime slurry prior to adding lime to the influent stream. The initial tests using solution from the column tests were run to the same specification as the tests previously completed by B.C. Research. The supernatant was decanted and the precipitate added to the lime slurry to be added to the subsequent batch. This was repeated 9 times. The results of the effluent analysis are shown in Table 7.2-5. The sulphate levels reduced from the initial 3760 mg/L to below 2000 mg/L with

COMPONENT	INITIAL	CYCLE NUMBER				
	SOLUTION	1	2	3	4	
Acidity to pH 8.3 (mg CaCO ₃ /L)		10	10	5	5	
Alkalinity to pH 8.5 (mg CaCO ₃ /L)		30	12	24	22	
Sulphate (mg/L)	3760	2040	1885	1863	1460	
Total Phosphate (mg P/L)	17.19	<0.03	0.06	<0.03	<0.03	
Hardness (mg CaCO ₃ /L)	120	1820	1970	2140	2320	
Ag (mg/L)	0.017	<0.005	<0.005	0.019	<0.005	
As (mg/L)	2.057	0.016	0.02	0.007	0.002	
AI (mg/L)	62.4	<0.35	<0.35	3.57	2.63	
Cd (mg/L)	0.008	<0.001	<0.001	<0.001	<0.001	
Cr (mg/L)	0.292	0.008	0.005	0.008	0.005	
Cu (mg/L)	1.92	0.011	0.006	0.013	0.014	
Fe (mg/L)	1430	0.81	2.26	1.98	1.08	
Hg (mg/L)	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	
Mn (mg/L)	3.04	<0.05	<0.05	<0.05	<0.05	
Ni (mg/L)	0.89	<0.05	<0.05	<0.05	<0.05	
Pb (mg/L)	0.012	0.002	<0.002	0.002	0.006	
Zn (mg/L)	2.90	<0.04	0.040	<0.04	<0.04	

Analysis Of Supernatant from Lime and Settling Test #2 with Precipitate Recycle on Acid Drainage Sample from Waste Rock at the Cinola Gold Project

TABLE 7.2-5

^a Concentrations reported as total metals.

the last cycle at 1460 mg/L. The reduction of sulphate in this system results from the precipitation of gypsum, CaSO₄, which has a nominal solubility of about 1800 mg/L, given the chemistry of the treatment plant.

Generally, gypsum supersaturates in these systems forming large masses of scale on process equipment and instrument probes. The recycle of the underflow with the HDP system tested provides for gypsum seed crystals being introduced at the point of lime addition resulting in crystal growth on the seeds rather than process equipment thus relieving the supersaturated state of the solution. The results presented indicate good control of the supersaturation.

Hardness in the effluent from these tests is in the range of 1800-2300 mg/L, well above any receiving water in the area. This results from the addition of lime, a calcium source, in order to neutralize the solution and where insufficient carbonate and sulphate are present to precipitate the excess calcium. In solutions that are originally high in sulphate, excess sulphate will persist yielding much lower hardness values. A high sulphate solution is predicted for the influent and therefore lower hardness values than those presented are expected.

The data also indicate high aluminum levels in latter runs with levels reaching 3.57 mg/L at cycle #8. This is probably due to carry-over of suspended aluminum hydroxide, which tends to be light and separate from the iron precipitate. Levels of the other metals analyzed in the supernatant are much the same as reported in Table 7.2-5.

7.2.3 Caustic neutralization

Three tests were run using caustic soda, NaOH, for neutralization of the acid in solution to determine treatability with a backup reagent and to observe the coagulation, settling and removal efficiencies without lime present in the solution. The results are summarized in Table 7.2-6.

Each solution was generated from argillically altered rock taken from the adit and neutralized with 0.5M sodium hydroxide. The mixture was allowed to settle in a graduated cylinder with supernatant siphoned off and filtrate from a 0.45um filter. Both filtrate and supernatant were analyzed from the pH 9.5 solution.

At pH 9.5, metals were removed with some levels higher than found with lime treatment. For instance, copper and zinc levels were both higher than previously found with lime treatment. At pH 7.5 metal levels remaining in the supernatant were, in general, higher than found with lime treatment. Although aluminum was well below previous results with lime treatment, a low residual level would be expected at pH 7.5. Metals such as cadmium, iron, manganese, nickel. and zinc are considerably higher than values found from lime treatment at this pH value (Table 7.2-3). Chromium and copper were somewhat lower with caustic treatment at pH 7.5. The original levels of sulphate were found in the effluent because no calcium was present in the solution or the reagent, therefore no gypsum was precipitated.

PARAMETER	INITIAL SOLUTION BUCKET #1	pH 9.5 SUPERNATANT BUCKET #1	pH 9.5 FILTRATE BUCKET #1	pH 7.5 SUPERNATANT BUCKET #1	DETECTION LIMIT			
pH	2.00	N/A	N/A	N/A	None			
Conductance (umhos/cm)	11000	11000	11300	11000	None			
Total Phosphorus (mg P/L)	99.30	Detected	0.382	0.393	0.1000			
Sulphate (mg/L)	9100	7500	7740	7580	None			
Alkalinity to pH 4.5 (mg CaCO ₃ /L)	0.0	20.0	25.0	0.0	None			
Alkalinity to pH 8.5 (mg CaCO ₃ /L)	0.0	0.0	0.0	0.0	None			
Acidity to pH 8.3 (mg CaCO ₃ /L)	8350.0	10.0	7.5	700	None			
Ag (mg/L)	<0.004	<0.004	<0.004	<0.004	0.0040			
AI (mg/L)	147.000	0.135	0.093	Detected	0.020			
As (mg/L)	9.6800	<0.040	<0.040	<0.020	0.0040			
Cd(mg/L)	0.3330	<0.001	<0.001	0.055	0.0010			
Cr (mg/L)	0.017	<0.004	<0.004	<0.004	0.004			
Cr (VI) (mg/L)	N.A.	N.A.	<0.010	N.A.	0.0100			
Cu (mg/L)	2.070	0.37	0.131	0.028	0.002			
Fe (mg/L)	2860.00	4.59	1.02	383.00	0.010			
Hg (mg/L)	<0.034	<0.010	<0.010	<0.010	0.0100			
Mn (mg/L)	30.800	1.090	0.672	20.10	0.002			
Ni (mg/L)	2.99	Detected	<0.010	0.328	0.0100			
Pb (mg/L)	0.145	<0.010	Detected	Detected	0.010			
Zn (mg/L)	41.000	0.072	0.042	0.897	0.010			

2

TABLE 7.2-6 Analysis of Filtrate and Supernatant from Caustic Neutralization, Cinola Gold Project

Detected = Detected but not quantitated.

Quantication Limit = 3.3 times detection limit.

N.A. = Not Analyzed.

Concentrations reported as total metals.

7.2.4 Confirmatory tests with lime

Effluent variability from the water treatment plant was determined by testing a range of influent samples and the treatability of each solution. Three samples of argillically altered rock were obtained. Each of the three water samples for the tests was generated by immersing argillically altered rock in a five-gallon bucket, and decanting the resulting solution as the test solution. The rock used to produce for each solution was:

- Bucket #1 from the adit
- Bucket #2 from 1987 drill cores
- Bucket #3 from 1987 drill cores

Each solution had unique characteristics with Bucket #1 having high conductivity, phosphorous, sulphate, acidity, iron, arsenic, cadmium, manganese, and lead. Bucket #2 had "average" sulphate and conductivity with mid-range levels of other metals. Bucket #3 had moderately high copper, nickel, and zinc in relation to previous samples tested. Overall, the solutions to be tested had a relatively good range of parameters.

Each solution was treated with a lime slurry with pH adjusted to 9.5. The solutions were placed in graduated cylinders for settling of the precipitates, the supernatant decanted and filtrates obtained by filtering the supernatant solutions through 0.45um filters to obtain dissolved metal values.

The results from the experiment using the Bucket #1 solution are shown in Table 7.2-7. Due to high initial sulphate values insufficient calcium was available from the neutralization to bring sulphate below 2400 mg/L. Aluminum was adequately removed with 0.167 mg/L dissolved aluminum, and 0.089 mg/L total. All other metals were removed adequately except copper which has a relatively high total concentration of 0.047 mg/L, while the dissolved concentration remains below 0.01 mg/L.

Table 7.2-8 illustrates the effluent concentrations from the tests from Bucket #2 which had much lower initial concentrations compared to Bucket #1, yet higher concentrations than solutions previously studied. Sulphates were reduced to 2060 to 2140 mg/L consistent with lower initial sulphates, while total aluminum was adequately removed at 0.0093 mg/L. Copper was removed quite efficiently with effluent total concentration reported as 0.009 mg/L and soluble below 0.0066 mg/L. Both soluble iron and manganese and other metals tested were at lower levels than found in Bucket #1 test results.

Analytical results from the tests on Bucket #3 are presented in Table 7.2-9 for the supernatant only. The sulphates were reduced only to 2920 mg/L probably due to the lower lime requirements reducing the input of lime to the solution. In this test, all metals were within limits acceptable for discharge.
TABLE 7.2-7

Analysis of Filtrate and Supernatant
from Lime and Settle Tests - Bucket #1,
Cinola Gold Project

PARAMETER	INITIAL SOLUTION BUCKET #1	PH 9.5 SUPERNATANT BUCKET #1	PH 9.5 FILTRATE BUCKET #1	DETECTION
 рН	2.00	N/A	N/A	None
Conductance (umhos/cm)	11000	3550	3500	None
Total Phosphorus (mg P/L)	99.30	<0.010	<0.010	0.1000
Sulphate (mg/L)	9100	2470	2660	None
Alkalinity to pH 4.5 (mg CaCO ₃ /L)	0.0	30.0	30.0	None
Alkalinity to pH 8.5 (mg CaCO ₃ /L	0.0	2.5	2.5	None
Acidity to pH 8.3 (mg CaCO ₃ /L)	8350.0	<2.0	2.5	None
Ag (mg/L)	<0.004	<0.004	<0.004	0.0040
AI (mg/L)	147.000	0.089	0.167	0.020
As (mg/L)	9.6800	<0.040	<0.04	0.0040
Cd(mg/L)	0.3330	<0.001	Detected	0.0010
Cr (mg/L)	0.017	<0.004	<0.004	0.004
Cr (VI) (mg/L)	N.A.	N.A.	N.A.	0.100
Cu (mg/L)	2.070	0.047	Detected	0.002
Fe (mg/L)	2860.00	0.57	<0.01	0.010
Hg (mg/L)	<0.034	<0.010	<0.01	0.0100
Mn (mg/L)	30.800	0.048	0.109	0.002
Ni (mg/L)	2.99	<0.010	<0.01	0.0100
Pb (mg/L)	0.145	0.010	<0.01	0.010
Zn (mg/L)	41.000	Detected	Detected	0.010

Detected = Detected but not quantitated.

Quantitation Limit = 3.3 times detection limit. N.A. = Not Analyzed.

Concentrations reported as total metals.

ŧ

PARAMETER	INITIAL SOLUTION BUCKET #2	PH 9.5 SUPERNATANT BUCKET #2	PH 9.5 FILTRATE BUCKET #2	
	2.54	N//A	N1/A	
pri Conductance (umbos/cm)	6300	N/A 2200	IN/A 2250	None
Total Phoenhorus (mg P/L)	0500	3300	3350	None
Subboto (mg/l)	0.70	<0.10	<0.10	0.1000
	4150	2140	2060	None
Alkalinity to pH 4.5 (mg CaCO ₃ /L)	0.0	47.5	47.5	None
Alkalinity to pH 8.5 (mg CaCO ₃ /L)	0.0	5.0	5.0	None
Acidity to pH 8.3 (mg CaCO ₃ /L)	2925.0	0.0	0.0	None
Ag (mg/L)	<0.05	<0.004	<0.004	0.0040
AI (mg/L)	111.000	0.248	0.093	0.020
As (mg/L)	2.8140	<0.040	<0.040	0.0040
Cd(mg/L)	0.0437	Detected	<0.001	0.0010
Cr (mg/L)	0.170	<0.004	< 0.004	0.004
Cr (VI)(mg/L)	N.A.	N.A.	N.A.	0.0100
Cu (mg/L)	3.900	0.009	Detected	0.002
Fe (mg/L)	1090.00	0.796	<0.015	0.010
Hg (mg/L)	0.0016	<0.010	<0.010	0.0100
Mn (ma/L)	12.700	0.265	0.233	0.002
Ni (ma/L)	2.37	Detected	<0.010	0.0100
Pb (ma/l)	<0.05		<0.010	0.010
Zn (mg/L)	33.400	0.068	Detected	0.0100

TABLE 7.2-8 Analysis of Filtrate and Supernatant from Lime and Settle Tests - Bucket #2, Cinola Gold Project

Detected = Detected but not quantitated

Quantitation Limit = 3.3 times detection limit.

N.A. = Not Analyzed. Concentrations reported as total metals.

		•	
PARAMETER	INITIAL SOLUTION BUCKET #3	pH 9.5 SUPERNATANT BUCKET #3	DETECTION LIMIT
рН	2.22	N/A	None
Conductance (umhos/cm)	5500	3900	None
Total Phosphorus (mg P/L)	3.90	<0.53	0.1000
Sulphate (mg/L)	3780	2920	None
Alkalinity to pH 4.5 (mg CaCO ₃ /L)	0.0	50.0	None
Alkalinity to pH 8.5 (mg CaCO ₃ /L)	0.0	10.0	None
Acidity to pH 8.3 (mg CaCO ₃ /L)	2375.0	0.0	None
Ag (mg/L)	<0.05	<0.05	0.0040
AI (mg/L)	152.000	<0.25	0.020
As (mg/L)	0.7250	0.0070	0.0040
Cd (mg/L)	0.0898	<0.0002	0.0010
Cr (mg/L)	0.090	<0.025	0.004
Cr(VI) (mg/L)	N/A	N/A	0.0100
Cu (mg/L)	8.680	0.010	0.002
Fe (mg/L)	1330.00	<0.05	0.010
Hg (mg/L)	<0.0001	<0.05	0.0100
Mn (mg/L)	11.200	0.920	0.002
Ni (mg/L)	7.35	<0.025	0.0100
Pb (mg/L)	<0.05	<0.05	0.010
Zn (mg/L)	84.100	<0.05	0.010

TABLE 7.2-9Analysis of Supernatant From Lime and Settle Tests - Bucket #3,
Cinola Gold Project

Detected = Detected but not quantitated.

Quantitation Limit = 3.3 times detection limit.

N.A. = Not Analyzed.

Concentrations reported as total metals.

7.4 Summary and Conclusions

Bench scale tests were run on simulated acid water from the Cinola Gold Project to test the effectiveness of contaminant removal by lime treatment. Several different solutions were used for these tests covering much of the expected range of contaminants in the influent. In all cases, effluent water quality resulting from the treatment process was improved significantly.

Basic parameters such as pH, conductivity, sulphate, phosphate and acidity were controlled in the process with pH being adjusted from 2.0 to 2.5 up to 9.5. High sulphate levels in the influent were reduced to the 1500 to 3000 mg/L range dependent on the sulphate concentration and acidity of the solution. In conjunction with sulphate reduction, conductivity and TDS levels were reduced as a result of lime treatment. Phosphorous is also reduced by precipitation as calcium phosphate and other inorganic precipitates.

The general level of metal reduction in the solutions is significant in all cases. Removal efficiencies above 99 percent were obtained in all cases where high metal concentrations were present in the influent. Of the metals studied, all precipitated out to an exceptional degree in these bench scale tests compared with normal treatment plant operating levels. This is probably due to the precipitation matrix of lime and iron hydroxides plus the relatively high TDS of the solutions creating high surface charge on the hydroxides as they precipitate. This high surface charge on the metal hydroxides results in a high level of metal adsorption on the surface of the precipitate and thus provides for removal efficiencies well above that predicted from solubility considerations.

There is evidence from the HDP tests that total aluminum levels may increase with repeated cycling of the precipitate as fine particulate aluminum hydroxide is recycled back through the system. Precipitation of the solutions at pH's less than 9.5 may result in less aluminum recycle but could decrease the removal efficiencies of metals with minimum solubilities in the pH 9 to 11 range.

The results of the caustic neutralization tests indicate lower removal efficiency than the lime treatment process and therefore the caustic neutralization process will not to be used in the water treatment plant.

Copper, a metal of some concern due to its toxicity, even at low concentrations, was controlled to an extraordinary degree in these bench scale tests. Soluble levels below 0.01 mg/L were reached more than 90 percent of the time with total levels below 0.02 mg/L.

The results of the tests presented here verify the work previously presented in the Stage II Report. Table 7.2-10 summarizes the estimate of effluent quality from the water treatment plant based on the current bench scale tests for both total and dissolved characteristics. Further bench scale and pilot scale tests are required for final design of the process parameters. These tests should also further define effluent quality parameters in large scale equipment.

In conclusion, the lime addition and precipitation technology results in excellent contaminant removal efficiencies for all of the bench scale tests.

TABLE 7.2-10 Expected Levels of Contaminants in Effluent from the Mine Site Water Treatment Plant at the Cinola Gold Project

	EFFLUENT TO	O WETLANDS
PARAMETER	DISSOLVED ^a	TOTAL
Alkalinity (mg CaCO ₃ /L)		14-126
Conductance (umho/cm)		2600-2760
Sulphate (mg/L)	1565-1785	1565-1785
Total Phosphate (mg P/L)		0.001-0.006
Ortho Phosphate (mg P/L)	0.001-0.005	
Hardness (mg CaCO ₃ /L)	2000	2500
As (mg/L)	<0.02	<0.02
AI (mg/L)	<0.45	<2.00
Cd (mg/L)	<0.0002	<0.0004
Co (mg/L)	0.02	0.03
Cr (mg/L)	<0.002	<0.003
Cu (mg/L)	<0.01	<0.02
Fe (mg/L)	<0.1	<3.00
Hg (mg/L)	0.0001	0.0002
Mn (mg/L)	0.129	0.30
Ni (mg/L)	0.019	0.03
Pb (mg/L)	0.0011	0.002
Zn (mg/L)	<0.01	0.03

a As presented in Table 7.3.11-6, Volume II, Stage II Report.

SECTION 8.0

8.0 UPDATE OF STAGE II REPORT IMPACT ASSESSMENT

8.1 Introduction

Since the submission of the Stage II Report, additional test programs have been completed to provide more detailed information for final project design. The programs include the 1988 pilot plant metallurgical test programs, waste rock liming experiments, and water treatment test programs described in Sections 3.0, 6.0 and 7.0 respectively of this Stage II Addendum Report. The purpose of this section is to update the impact assessment presented in Volume IV of the Stage II Report, based on the results of the detailed testwork.

This section specifically addresses:

- a) potential water quality impacts associated with the tailings/waste rock impoundments during both operation and at closure (following reclamation);
- b) potential water quality impacts from the discharge of effluent from the minesite water treatment plant on Barbie Creek during operation; and
- c) potential water quality impacts of outflow from the reclaimed pit on Barbie Creek at closure.

In all cases, impacts were assessed by using mass balance calculations to predict quality of receiving water after mixing with discharges. Calculations were identical to those described in the Stage II Report, with the new data used to characterize the various discharges.

8.2 Comparison of Impacts Based on Bench-scale and Pilot Mill Test Data

8.2.1 Background

As described in detail in the Stage II report (Section 5, Volume II), the High West impoundment is a three impoundment system (Impoundment Nos. 1, 2, and 3) contained by four embankments. The three principal embankments are located in the Florence Creek valley and a small Saddle Embankment is located on the Florence Creek/Boucher Creek drainage divide (Figure 8.2-1). The three impoundments will be activated sequentially for the disposal of mill tailings. Each impoundment will be active for approximately four years and will subsequently be reclaimed. Florence Creek will be diverted around the impoundment system during operation. The expected water quality impacts from the system were described in Section 3.6, Volume IV of the Stage II Report. In summary, the impoundment. Consequently, the only changes to water quality in Florence Creek will be associated with outflow from reclaimed impoundments and seepage under the embankments. As discussed in Section 3.5.4, Volume IV, seepage under the Saddle Embankment to Boucher Creek will be attenuated and will not affect water quality. Seepage under the embankments to Florence Creek will be slow and is not expected to influence surface waters until closure.



The Stage II report concluded that water quality in the reclaimed impoundments during operation will be largely determined by groundwater movement up into the impoundments (70 m^3/d in Impoundment No. 1; 50 m^3/d in Impoundment No. 2). For the purposes of the impact assessment, it was assumed that tailings consolidation and groundwater discharge to the impoundment area will result in displacement of mill effluent quality pore water from the tailings mass into the surface water of the ponds. During operation, this porewater seepage will be diluted by rainfall in reclaimed Impoundment Nos. 1 and 2. After dilution it will discharge to the Florence Creek diversion channel through wetlands established in the respective impoundments.

At closure, Florence Creek will be redirected through the three reclaimed impoundments. Water quality changes will be associated with continued upward groundwater flow and displacement of tailings pore water into reclaimed Impoundment Nos. 1, 2 and 3 and seepage of water under Embankment No. 3 and into Florence Creek below the impoundment area.

8.2.2 Assessment of effects on Florence Creek

.

In the Stage II Report, the water quality impacts on Florence Creek were assessed using data from the bench scale test program to define the characteristics of the tailings effluent, and hence, the tailings pore water quality. For this Addendum Report, impacts on the water quality of Florence Creek are assessed using data from the May 1988 pilot mill test program to characterize the expected quality of tailings pore water. As in the Stage II Report, impact predictions are made for discharge from reclaimed Impoundment No. 1 to Florence Creek (Year 5 or 6), for discharge from reclaimed Impoundment Nos. 1 and 2 to Florence Creek (Year 9 or 10), and for closure, when Florence Creek will flow through the entire impoundment area and will be affected by pore water seepage from all the impoundments. The Stage II assessment considered both average flow conditions and extreme flow conditions. Extreme low flows were defined as June low flows with a 75% probability of exceedance; extreme high flows were defined as October high flows with a 25% probability of exceedance (Section 3.4.3, Volume IV).

Since there will be no outflow from the reclaimed impoundments in dry periods during the operations phase, extreme flow conditions will have no effect on dilution rates and predicted impacts during operation. Therefore, detailed water quality predictions for extreme flow conditions during operation are not presented in this report. However, at closure pore water seepage directly to Florence Creek will be constant; therefore, precipitation rates and resultant stream flows will determine available dilution and potential water quality impacts. Accordingly, the effects of extreme flow conditions, including a mean annual one day low flow, on water quality in Florence Creek are assessed.

No significant water quality impacts {defined in terms of Ministry of Environment (MOE) receiving water quality criteria} were predicted in the Stage II Report. This conclusion was based on using bench scale data to characterize mill effluent and tailings pore water quality. The pilot plant tests, which are more representative of operating conditions, confirmed this conclusion. Levels of cyanide, mercury and most other metals were in fact lower in the pilot mill tests than in the bench scale tests (Table 8.2-1).

PARAMETER	PILOT MILL ^a	BENCH SCALE ^b
рН	8.0	9.7
Alkalinity (mg CaCO3/L)	170	120
Conductance (umhos/cm)	10010	6433
Sulphate (mg/L)	3250	1787
Nitrate (mg N/L)	295	206
Total Phosphorus (mg P/L)	0.50	0.27
Total Cyanide (mg/L)	0.54	1.0
WAD Cyanide (mg/L)	0.33	0.5
Calcium (mg/L)	650	1050
Magnesium (mg/L)	0.90	2.0
Hardness (mg CaCO ₃ /h)	1629	2633
Dissolved Metals (mg/L)		
Ag	0.0020	0.001
Al	1.4	0.30
As	0.13	0.12
Ba	0.05	0.16
Cd	0.001	0.001
Co	0.52	0.32
Cr	0.017	0.031
Cu	0.10	0.44
Fe	0.07	2.7
Hg (ug/L)	0.24	0.33
Mn	0.02	0.02
Мо	0.13	0.13
Ni	0.08	0.01
Рb	0.02	0.001
Sb	0.10	0.19
Se	0.006	0.012
Zn	0.02	0.02

TABLE 8.2-1Comparison of Mill Effluent Characterization Conducted by Norecol Based on
May 1988 Pilot Mill and February 1988 Bench Scale Tests

^a Single value; taken after sulphide addition; based on SO₂/air cyanide destruction process.

^b Mean of three tests; cyanide levels based on typical values obtained for operating mills.

8-4

Copper and cyanide

Total cyanide and copper concentrations (Table 8.2-1) were considerably lower in the May 1988 pilot mill test than levels obtained in the bench scale tests. Tables 8.2-2 and 8.2-3 indicate that, when mill effluent quality pore water is displaced into the reclaimed impoundments, the resultant concentrations in the overlying water will meet MOE criteria such that the water is acceptable for immediate discharge. Thus, based on the pilot mill numbers, it would not be necessary to wait for natural cyanide degradation and copper precipitation to occur in the tailings pore water prior to discharge. In the Stage II Report, a 12-month waiting period was suggested as a management strategy to allow for sufficient cyanide degradation and copper precipitation to meet target levels. This recommendation which was based on bench scale data would not be required, according to findings of the pilot scale work. However, water quality in the reclaimed impoundments will be monitored during operation to insure that it is acceptable prior to discharge.

Based on the 1988 pilot plant data and predicted rates of seepage of pore water to Florence Creek at closure, some degradation of cyanide complexes and precipitation of copper in the tailings mass would be necessary to produce acceptable water quality. Based on mass balance calculations, the average weak acid dissociable (WAD) cyanide concentration in the pore water of all three reclaimed impoundments would need to be 0.25 mg/L and the copper would need to be 0.08 mg/L in order to meet receiving water quality criteria under average flow conditions. Cyanide and copper concentrations of 0.20 mg/L and 0.06 mg/L, respectively, would be required to meet criteria in extreme low flow conditions.

As described in the Stage II Report, cyanide is expected to degrade exponentially with a half-life of six months (Caldwell et al. 1984; Englehardt 1984; Worsley 1986). Copper is expected to precipitate in proportion to the rate of cyanide degradation until it reaches an equilibrium concentration of approximately 0.05 mg/L (Smith 1988; see also Volume IV, Section 3.7.2). Accordingly, it is predicted that within six months (one half-life) following the end of operation, WAD cyanide and copper concentrations in the tailings pore water of Impoundment No. 3 should be 0.17 mg/L and 0.05 mg/L respectively. By that time, WAD cyanide levels in reclaimed Impoundments Nos. 1 and 2 should be less than 0.001 mg/L, while copper concentrations should be at the equilibrium value of 0.05 mg/L. Therefore, the discharge will meet receiving water quality criteria in Florence Creek as soon as reclamation has been completed (Tables 8.2-4 and 8.2-5).

No discharge will be made from any reclaimed impoundment until monitoring confirms that the water is of suitable quality to meet receiving water criteria. Contingency plans have been developed to control discharges from reclaimed impoundments such that receiving water criteria will be met. Contingency measures include the provision of sufficient capacity in the reclaimed impoundments to store water for up to two years following deactivation and the option to divert discharges to the active impoundment during operation. Several other contingencies could also be developed due to the flexibility of the impoundment system. Ongoing monitoring and refinement of mitigative design during operation will result in effective and proven design for closure.

Other Parameters

Levels of some parameters were slightly higher in the pilot mill effluent than in the bench scale tests, but the differences did not change the conclusions of the impact assessment. Silver, cobalt,

	FLORENCE	MOE RECEIVING			<u></u>		CONC	ENTRATION	IS IN RECE	VING WAT	ER	····			
PARAMETER	CREEK BACKGROUND	WATER CRITERIA®	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP ^b	ост	NOV	DEC	MEAN ANNUAL
Florence Creek Flow Rate (m ³ /s)			0.48	0.43	0.47	0.36	0.21	0.10	0.15	0.15	0.17	0.74	0.65	0.49	0.37
Reclaimed Impoundment No. 1 Discharge Rate ^c (m ³ /s)			0.033	0.036	0.020	0.023	0.009	0.001	0.000	0.000	0.014	0.041	0.040	0.040	0.021
Hardness (mg CaCOyL) Sulphate (mg/L) Nitrate (mg NL) Total Phosphorus (mg P/L) Total Cyanide (mg/L) WAD Cyanide (mg/L)	10 1 0.014 0.029 0.0005 0.0005	_d 1000 ⁹ _d _d _d 0.005	14 9 0.799 0.028 0.002 0.001	14 11 0.971 0.028 0.002 0.002	12 6 0.491 0.029 0.001 0.001	13 8 0.733 0.028 0.002 0.001	12 6 0.488 0.029 0.001 0.001	11 2 0.160 0.029 0.001 0.001	10 1 0.014 0.029 0.001 0.001	10 1 0.014 0.029 0.001 0.001	24 31 2.78 0.031 0.006 0.004	13 8 0.654 0.028 0.002 0.001	13 8 0.720 0.028 0.002 0.601	14 11 0.951 0.028 0.002 0.002	13 8 0.670 0.028 0.002 0.001
Totai Metais (mg/L)															
Ag Al ⁹ As Ba Cd Co Cr Cr Cu Fe Hg (ug/L) Mn Mo Ni Pb Sb Se Zn	0.0001 0.27 0.001 0.005 0.0001 0.0005 0.0005 0.0006 0.45 0.025 0.023 0.0025 0.001 0.0005 0.001 0.0005 0.001 0.0005 0.0005 0.0008	0.0001 0.05 0.05 1 0.002 0.05 0.002 0.3 0.1 0.1 1 0.25 0.003 0.05 0.001 0.03	0.0001 0.25 0.001 0.005 0.001 0.002 0.001 0.008 0.42 0.02 0.022 0.022 0.022 0.003 0.001 0.001 0.001 0.0005 0.0008	0.0001 0.25 0.001 0.005 0.0001 0.003 0.001 0.009 0.41 0.02 0.021 0.003 0.001 0.001 0.001 0.001 0.005 0.0008	0.0001 0.26 0.001 0.005 0.0001 0.002 0.001 0.0008 0.43 0.02 0.002 0.003 0.001 0.001 0.0001 0.0005 0.0008	0.0001 0.25 0.001 0.005 0.0001 0.002 0.001 0.0008 0.42 0.02 0.022 0.003 0.001 0.001 0.001 0.001 0.0005 0.0008	0.0001 0.26 0.001 0.005 0.0001 0.002 0.001 0.0008 0.43 0.02 0.002 0.003 0.001 0.001 0.001 0.001 0.0005 0.0008	0.0001 0.26 0.001 0.005 0.0001 0.001 0.001 0.001 0.023 0.003 0.003 0.001 0.001 0.001 0.0005 0.0008	0.0001 0.27 0.001 0.005 0.0001 0.001 0.001 0.0008 0.45 0.03 0.023 0.003 0.001 0.0001 0.0001 0.0001 0.0005 0.0008	0.0001 0.27 0.001 0.005 0.0001 0.001 0.001 0.0006 0.45 0.03 0.023 0.003 0.001 0.001 0.001 0.001 0.0005 0.0008	0.0001 0.26 0.002 0.005 0.0001 0.001 0.0015 0.41 0.03 0.022 0.004 0.002 0.001 0.002 0.001 0.002	0.0001 0.25 0.001 0.005 0.0001 0.002 0.001 0.0008 0.42 0.02 0.022 0.003 0.001 0.001 0.001 0.001 0.0005 0.0008	0.6301 0.25 0.001 0.005 0.0001 0.002 0.001 0.0008 0.42 0.02 0.022 0.022 0.022 0.022 0.003 0.001 0.001 0.001 0.001 0.0005 0.0008	0.0001 0.25 0.001 0.005 0.0001 0.003 0.001 0.0009 0.41 0.02 0.021 0.003 0.001 0.001 0.001 0.001 0.0005 0.0008	0.0001 0.25 0.001 0.005 0.0001 0.002 0.001 0.0008 0.42 0.02 0.002 0.002 0.003 0.001 0.001 0.001 0.001 0.0005 0.0008

TABLE 8.2-2 Year 5 Water Quality of Florence Creek Based on Discharge from Reclaimed Impoundment No. 1, Predicted by Norecol for the Cinola Gold Project from May 1988 Pilot Plant Test Data

* For protection of aquatic life, unless otherwise specified.

•

September pond concentrations reflect storage over July and August. Water quality of this effuent is pilot mill concentration (Table 8.2-1) divided by a dilution factor of 25.7. Indicates no applicable criterion. .

.

I

I

1

4

Criterion for livestock watering.

. Drinking water criterion.

Dissolved Al concentration.

E

	FLORENCE	MOE RECEIVING					CONCI	ENTRATION	IS IN RECEI	IVING WAT	ER				
PARAMETER	CREEK BACKGROUND	WATER CRITERIA [®]	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	ост	NOV	DEC	MEAN ANNUAL
Florence Creek Flow Rate (m ³ /s)			0.48	0.43	0.47	0.36	0.21	0.10	0.15	0.15	0.17	0.74	0.65	0.49	0.37
Reclaimed Impoundment No. 1 Discharge Rate ^b (m ³ /s)			0.033	0.036	0.020	0.023	0.009	0.001	0.000	0.000	0.014	0.041	0.040	0.040	0.021
Reclaimed Impoundment No. 2 Discharge Rate ^c (m ³ /s)			0.036	0.039	0.022	0.025	0.011	0.003	0.000	0.001	0.020	0.045	0.044	0.043	0.024
Hardness (mg CaCO3/L) Sulphate (mg/L) Nitrate (mg N/L) Total Phosphorus (mg P/L) Total Cyanide (mg/L) WAD Cyanide (mg/L)	10 1 0.014 0.029 0.0005 0.0005	_d 1000 ⁹ 10 ¹ _d _d 0.005	16 15 1.32 0.027 0.003 0.002	17 18 1.61 0.027 0.003 0.002	14 9 0.817 0.028 0.002 0.001	15 14 1.222 0.027 0.003 0.002	14 10 0.854 0.028 0.002 0.001	12 5 0.387 0.028 0.001 0.001	10 1 0.025 0.029 0.001 0.001	10 1 0.046 0.029 0.001 0.001	28 41 3.7 0.029 0.007 0.004	15 12 1.084 0.027 0.002 0.002	15 13 1.190 0.027 0.003 0.002	17 18 1.57 0.027 0.003 0.002	15 13 1.128 0.027 0.002 0.002
Totai Metais (mg/L)															
Ag Al ⁹ As Ba Cd Cd Co Cr Cu Fe Hg (ug/L) Mn Mo N Pb Sb Sc	0.0001 0.27 0.001 0.005 0.0001 0.001 0.0005 0.0006 0.45 0.025 0.023 0.0025 0.001 0.0005 0.001	0.0001 0.05 1 0.0002 0.05 0.002 0.3 0.1 0.1 1 0.025 0.003 0.05 0.003 0.05	0.0001 0.23 0.002 0.005 0.0001 0.003 0.001 0.38 0.02 0.02 0.02 0.003 0.001 0.001 0.001	0.0001 0.23 0.002 0.005 0.0001 0.004 0.001 0.0011 0.02 0.02 0.02 0.003 0.001 0.001 0.001	0.0001 0.25 0.001 0.005 0.0001 0.002 0.001 0.0008 0.41 0.02 0.02 0.02 0.003 0.001 0.001 0.001	0.0001 0.24 0.001 0.005 0.0001 0.003 0.001 0.009 0.39 0.02 0.02 0.02 0.003 0.001 0.001 0.001	0.0001 0.25 0.001 0.005 0.0001 0.002 0.001 0.000 0.41 0.02 0.02 0.02 0.003 0.001 0.001 0.001	0.0001 0.26 0.001 0.005 0.0001 0.002 0.001 0.43 0.02 0.02 0.02 0.003 0.001 0.001 0.001	0.0001 0.27 0.001 0.005 0.0001 0.001 0.001 0.0006 0.45 0.02 0.02 0.02 0.003 0.001 0.001 0.001	0.0001 0.26 0.001 0.005 0.0001 0.001 0.001 0.006 0.45 0.02 0.02 0.02 0.003 0.001 0.001 0.001	0.0001 0.23 0.002 0.005 0.0001 0.007 0.001 0.0017 0.36 0.02 0.02 0.02 0.02 0.004 0.002 0.001 0.002	0.0001 0.24 0.001 0.005 0.0001 0.003 0.001 0.0009 0.40 0.02 0.02 0.02 0.003 0.001 0.001 0.001 0.001	0.0001 0.24 0.001 0.005 0.0001 0.003 0.001 0.009 0.39 0.02 0.02 0.02 0.003 0.001 0.001 0.001	0.0001 0.23 0.002 0.005 0.001 0.004 0.001 0.37 0.02 0.02 0.02 0.003 0.001 0.001 0.001	0.0001 0.24 0.001 0.005 0.0001 0.003 0.001 0.009 0.39 0.02 0.02 0.02 0.003 0.001 0.001 0.001
Se Zn	0.0005	0.001 0.03	0.0005	0.0004	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0005	0.0004	0.0005

TABLE 8.2-3 Year 9 Water Quality of Florence Creek Receiving Outflow from Reclaimed Impoundment Nos. 1 and 2, Predicted by Norecol for the Cinola Gold Project from May 1988 Pilot Plant Test Data

8

* For protection of aquatic life, unless otherwise specified.

For protection of aquatic tire, unless otherwise specified.
 Water quality of this effluent is pilot mill concentration (Table 8.2-1) divided by a dilution factor of 25.7.
 Water quality of this effluent is pilot mill concentration (Table 8.2-1) divided by a dilution factor of 42.
 Indicates no applicable criterion.
 Criterion for livestock watering.
 Diriking water criterion
 Dissolved Al concentrations.

8.7

.

N FLORENCE REC CREEK WA	MOE CONCENTRATIONS IN RECEIVING WATER RECEIVING													
CREEK BACKGROUND	WATER CRITERIA [®]	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	ост	NOV	DEC	MEAN ANNUAL
		0.48	0.43	0.47	0.36	0.21	0.1	0.15	0.15	0.17	0.74	0.65	0.49	0.37
		0.0013	0.0013	0.0013	0.0013	0.0013	0.0013	0.0013	0.0013	0.0013	0.0013	0.0013	0.0013	0.0013
		0.00022	0.00022	0.00022	0.00022	0.00022	0.00022	0.00022	0.00022	0.00022	0.00022	0.00022	0.00022	0.00022
10 1 0.014 0.029 0.0005 0.0005	_d 1000• 10 ^f _d _d 0.005	16 12 1.07 0.031 0.001 0.001	17 14 1.19 0.031 0.001 0.001	16 12 1.09 0.031 0.001 0.001	18 16 1.42 0.031 0.001 0.001	23 27 2.43 0.033 0.001 0.001	38 56 5.09 0.038 0.001 0.001	29 38 3.40 0.035 0.001 0.001	29 38 3.40 0.035 0.001 0.001	26 33 3.00 0.034 0.001 0.001	14 8 0.700 0.030 0.001 0.001	14 9 0.795 0.030 0.001 0.001	16 12 1.05 0.031 0.001 0.001	18 16 1.39 0.031 0.001 0.001
0.0001 0.27 0.001 0.005 0.0001 0.001 0.0005 0.0006 0.45 0.025 0.025 0.023 0.0025 0.001 0.0005 0.001 0.0005	0.0001 0.05 0.05 1 0.002 0.05 0.002 0.3 0.1 0.1 1 0.025 0.003 0.05 0.001	0.0001 0.27 0.002 0.006 0.0001 0.003 0.001 0.009 0.45 0.03 0.023 0.003 0.003 0.001 0.001 0.001 0.0001	0.0001 0.27 0.002 0.006 0.0001 0.003 0.001 0.0009 0.45 0.03 0.023 0.003 0.003 0.001 0.001 0.001 0.001	0.0001 0.27 0.002 0.006 0.0001 0.003 0.001 0.009 0.45 0.03 0.023 0.003 0.001 0.001 0.001 0.001	0.0001 0.27 0.002 0.006 0.0001 0.003 0.001 0.001 0.45 0.03 0.023 0.003 0.003 0.001 0.001 0.001 0.001	0.0001 0.28 0.002 0.006 0.0001 0.005 0.001 0.0013 0.45 0.03 0.023 0.004 0.002 0.001 0.002 0.0015	0.0001 0.29 0.003 0.006 0.0001 0.010 0.001 0.0020 0.45 0.03 0.024 0.005 0.002 0.002 0.001 0.003 0.0006	0.0001 0.28 0.003 0.006 0.0001 0.007 0.001 0.0015 0.45 0.03 0.023 0.004 0.002 0.001 0.002 0.0001	0.0001 0.28 0.003 0.006 0.0001 0.007 0.001 0.0015 0.45 0.03 0.023 0.004 0.002 0.001 0.002 0.001	0.0001 0.28 0.002 0.006 0.0001 0.008 0.001 0.0014 0.45 0.03 0.023 0.004 0.002 0.001 0.002 0.001	0.0001 0.27 0.001 0.006 0.0001 0.002 0.001 0.0008 0.45 0.03 0.023 0.003 0.003 0.001 0.001 0.001 0.001	0.0001 0.27 0.001 0.006 0.0001 0.002 0.001 0.0008 0.45 0.03 0.023 0.003 0.003 0.001 0.001 0.001	0.0001 0.27 0.002 0.006 0.0001 0.003 0.001 0.0009 0.45 0.03 0.023 0.003 0.003 0.001 0.001 0.0005	0.0001 0.27 0.002 0.006 0.0001 0.003 0.001 0.001 0.45 0.03 0.023 0.003 0.003 0.001 0.001 0.001 0.001 0.001
	FLORENCE CREEK BACKGROUND 10 1 0.014 0.029 0.0005 0.0005 0.0005 0.0005 0.0001 0.27 0.001 0.001 0.001 0.001 0.001 0.001 0.0005 0.0008 0.45 0.025 0.001 0.0005 0.001 0.0005 0.001 0.0005 0.001 0.0005 0.0005 0.0005 0.0005 0.0005	MOE RECEIVING WATER BACKGROUND MOE RECEIVING WATER CRITERIA ⁴ 10 -d 1 1000 ⁶ 0.014 10 ^f 0.229 -d 0.0005 -d 0.0005 -d 0.0005 -d 0.0005 1 0.0005 0.005 0.0005 1 0.0005 1 0.0005 0.002 0.001 0.002 0.001 0.002 0.002 0.11 0.0025 1 0.0025 1 0.0025 0.1 0.0025 1 0.0025 0.01 0.0025 0.002 0.001 0.025 0.001 0.025 0.001 0.025 0.001 0.025 0.001 0.025 0.001 0.025 0.001 0.025 0.001 0.03	MOE CREEK BACKGROUND MOE RECEIVING WATER CRITERIA ⁴ JAN 0.48 0.0013 0.00022 0.00022 10 -d 16 1 1000 ⁶ 12 0.014 10 ^f 1.07 0.229 -d 0.001 0.0005 -d 0.001 0.0005 -d 0.001 0.0005 0.005 0.001 0.001 0.0001 0.001 0.0005 0.005 0.001 0.001 0.002 0.0001 0.0021 0.002 0.001 0.0025 1 0.006 0.0031 0.002 0.001 0.005 1 0.006 0.005 0.002 0.001 0.005 0.002 0.001 0.005 0.002 0.001 0.005 0.002 0.001 0.005 0.002 0.001 0.005 0.002 0.001 0.0025 0	MOE CREEK RECEIVING WATER CRITERIA ⁴ JAN FEB 0.48 0.43 0.0013 0.0013 0.00022 0.00022 0.00022 10 -d 16 17 1 1000 ⁶ 12 14 0.014 10 ^f 1.07 1.19 0.029 -d 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.001 0.001 0.005 0.005 0.001 0.001 0.001 0.002 0.001 0.001 0.005 1 0.002 0.001 0.005 1 0.006 0.002 0.005 0.002 0.001 0.001 0.005 0.002 0.001 0.001 0.005 0.002 0.003 0.003 0.005 0.002 0.001 0.001 0.005 0.002 0.001 0.001 0.005 0.002 0.003 <td>MOE CREEK BACKGROUND RECEIVING WATER CRITERIA⁴ JAN FEB MAR 0.48 0.43 0.47 0.0013 0.0013 0.0013 0.0013 0.00022 0.00022 0.00022 0.00022 10 -d 16 17 16 1 1000⁶ 12 14 12 0.014 10⁷ 1.07 1.19 1.09 0.029 -d 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.0005 0.005 0.001 0.001 0.001 0.001 0.001 0.002 0.002 0.002 0.002 0.002 0.005 1 0.006 0.006 0.006 0.006 0.0005 0.002 0.001 0.001 0.001 0.001 0.005 1 0.003 0.003</td> <td>MOE CREEK BACKGROUND MOE RECEIVING WATER CRITERIA* JAN FEB MAR APR 0.48 0.43 0.47 0.36 0.0013 0.0013 0.0013 0.0013 0.0013 0.00022 0.00022 0.00022 0.00022 0.00022 10 -d 16 17 16 18 1 1000⁶ 12 14 12 16 0.029 -d 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.001 0.0001 0.0001 0.0001 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.001 0.0005 0.005 0.002 0.002 0.002 <t< td=""><td>MOE CREEK MACE WATER CONCE BACKGROUND CRITERIA⁴ JAN FEB MAR APR MAY 0.48 0.43 0.47 0.36 0.21 0.0013 0.0013 0.0013 0.0013 0.0013 0.0012 0.00022 0.00011 0.0011</td><td>MOE CREEK RECEIVING WATER CRITERIA* JAN FEB MAR APR MAY JUN 0.48 0.49 0.43 0.47 0.36 0.21 0.1 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.00022 0.0001 0.0011</td><td>MOE CREEK BACKGROUND MOE RECEIVING CRITERIA⁴ JAN FEB MAR APR MAY JUN JUL 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.0013 0.0011 0.0011 0.0011</td><td>MOE CREEK CAREK BACKGROUND MOE RECEIVING WATER CRITERIA* JAN FEB MAR APR MAY JUN JUL AUG 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.0013 0.0012 0.00022 0.00022 0.00022 0.00022 0.00022 0.00022 0.00022 0.0002 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001</td><td>MOE CREEK BACKGROUND MOE WATER CRITERIA* JAN FEB MAR APR MAY JUN JUL AUG SEP 0.48 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.17 0.0013 0.0011 0.0011 0.0011 0.0011 0.0011 0.0011 0.0011</td><td>MOE CREEK BACKGROUND RECEIVING WATER CATERIA* JAN FEB MAR APR MAY JUN JUL AUG SEP OCT 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.17 0.74 0.0013 0.0014 0.0011 0.0011 0.001 0.0011 0.001 0.001 0.001 0.001 0.001</td><td>MOC CREEK BACKGROUND RECEIVING WATER CRITERIA* JAN FEB MAR APR MAY JUN JUL AUG SEP OCT NOV 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.17 0.74 0.85 0.0013 0.0014 0.0001</td><td>INCE CREEK CREEK DACKGROUND INCE RECEVENTING WATER CRITERIA* INA FEB MAR APR MAY JUN JUL AUG SEP OCT NOV DEC 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.17 0.74 0.65 0.49 0.0013 0.001</td></t<></td>	MOE CREEK BACKGROUND RECEIVING WATER CRITERIA ⁴ JAN FEB MAR 0.48 0.43 0.47 0.0013 0.0013 0.0013 0.0013 0.00022 0.00022 0.00022 0.00022 10 -d 16 17 16 1 1000 ⁶ 12 14 12 0.014 10 ⁷ 1.07 1.19 1.09 0.029 -d 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.0005 0.005 0.001 0.001 0.001 0.001 0.001 0.002 0.002 0.002 0.002 0.002 0.005 1 0.006 0.006 0.006 0.006 0.0005 0.002 0.001 0.001 0.001 0.001 0.005 1 0.003 0.003	MOE CREEK BACKGROUND MOE RECEIVING WATER CRITERIA* JAN FEB MAR APR 0.48 0.43 0.47 0.36 0.0013 0.0013 0.0013 0.0013 0.0013 0.00022 0.00022 0.00022 0.00022 0.00022 10 -d 16 17 16 18 1 1000 ⁶ 12 14 12 16 0.029 -d 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.001 0.0001 0.0001 0.0001 0.001 0.001 0.001 0.001 0.0005 -d 0.001 0.001 0.001 0.001 0.001 0.0005 0.005 0.002 0.002 0.002 <t< td=""><td>MOE CREEK MACE WATER CONCE BACKGROUND CRITERIA⁴ JAN FEB MAR APR MAY 0.48 0.43 0.47 0.36 0.21 0.0013 0.0013 0.0013 0.0013 0.0013 0.0012 0.00022 0.00011 0.0011</td><td>MOE CREEK RECEIVING WATER CRITERIA* JAN FEB MAR APR MAY JUN 0.48 0.49 0.43 0.47 0.36 0.21 0.1 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.00022 0.0001 0.0011</td><td>MOE CREEK BACKGROUND MOE RECEIVING CRITERIA⁴ JAN FEB MAR APR MAY JUN JUL 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.0013 0.0011 0.0011 0.0011</td><td>MOE CREEK CAREK BACKGROUND MOE RECEIVING WATER CRITERIA* JAN FEB MAR APR MAY JUN JUL AUG 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.0013 0.0012 0.00022 0.00022 0.00022 0.00022 0.00022 0.00022 0.00022 0.0002 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001</td><td>MOE CREEK BACKGROUND MOE WATER CRITERIA* JAN FEB MAR APR MAY JUN JUL AUG SEP 0.48 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.17 0.0013 0.0011 0.0011 0.0011 0.0011 0.0011 0.0011 0.0011</td><td>MOE CREEK BACKGROUND RECEIVING WATER CATERIA* JAN FEB MAR APR MAY JUN JUL AUG SEP OCT 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.17 0.74 0.0013 0.0014 0.0011 0.0011 0.001 0.0011 0.001 0.001 0.001 0.001 0.001</td><td>MOC CREEK BACKGROUND RECEIVING WATER CRITERIA* JAN FEB MAR APR MAY JUN JUL AUG SEP OCT NOV 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.17 0.74 0.85 0.0013 0.0014 0.0001</td><td>INCE CREEK CREEK DACKGROUND INCE RECEVENTING WATER CRITERIA* INA FEB MAR APR MAY JUN JUL AUG SEP OCT NOV DEC 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.17 0.74 0.65 0.49 0.0013 0.001</td></t<>	MOE CREEK MACE WATER CONCE BACKGROUND CRITERIA ⁴ JAN FEB MAR APR MAY 0.48 0.43 0.47 0.36 0.21 0.0013 0.0013 0.0013 0.0013 0.0013 0.0012 0.00022 0.00011 0.0011	MOE CREEK RECEIVING WATER CRITERIA* JAN FEB MAR APR MAY JUN 0.48 0.49 0.43 0.47 0.36 0.21 0.1 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.0013 0.00022 0.0001 0.0011	MOE CREEK BACKGROUND MOE RECEIVING CRITERIA ⁴ JAN FEB MAR APR MAY JUN JUL 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.0013 0.0011 0.0011 0.0011	MOE CREEK CAREK BACKGROUND MOE RECEIVING WATER CRITERIA* JAN FEB MAR APR MAY JUN JUL AUG 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.0013 0.0012 0.00022 0.00022 0.00022 0.00022 0.00022 0.00022 0.00022 0.0002 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001	MOE CREEK BACKGROUND MOE WATER CRITERIA* JAN FEB MAR APR MAY JUN JUL AUG SEP 0.48 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.17 0.0013 0.0011 0.0011 0.0011 0.0011 0.0011 0.0011 0.0011	MOE CREEK BACKGROUND RECEIVING WATER CATERIA* JAN FEB MAR APR MAY JUN JUL AUG SEP OCT 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.17 0.74 0.0013 0.0014 0.0011 0.0011 0.001 0.0011 0.001 0.001 0.001 0.001 0.001	MOC CREEK BACKGROUND RECEIVING WATER CRITERIA* JAN FEB MAR APR MAY JUN JUL AUG SEP OCT NOV 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.15 0.17 0.74 0.85 0.0013 0.0014 0.0001	INCE CREEK CREEK DACKGROUND INCE RECEVENTING WATER CRITERIA* INA FEB MAR APR MAY JUN JUL AUG SEP OCT NOV DEC 0.48 0.43 0.47 0.36 0.21 0.1 0.15 0.17 0.74 0.65 0.49 0.0013 0.001

.

E

TABLE 8.2-4 Water Quality of Florence Creek at Closure (Year 13) Based on May 1988 Pilot Plant Test Data, Predicted by Norecol for the Cinola Gold Project

. For protection of equatic life.

.

Quality of this pilot mill effluent (Table 8.2-1) except total cyanide = 0.002 mg/L, WAD cyanide = 0.001 mg/L, and total Cu = 0.05 mg/L. Quality of this effluent is pilot mil effluent (Table 8.2-1) except total cyanide = 0.14 mg/L, WAD cyanide = 0.08 mg/L, and total Cu = 0.05 mg/L. .

4 Indicates no applicable criterion. Criterion for livestock watering.

٠

f Drinking water criterion. B Dissolved Al concentration.

Table 8.2-5

Water Quality of Florence Creek for Extreme Wet and Dry Flow Conditions, at Closure, Predicted by Norecol for the Cinola Gold Project from May 1988 Pilot Plant Test Data

PARAMETER	FLORENCE CREEK BACKGROUND	MOE RECEIVING WATER CRITERIA ^a	DRY JUN	1 DAY LOW FLOW ^b	WET OCT
Florence Creek Flow Rate (m ³ /s)			0.08	0.074	0.82
Groundwater Discharge Into Impoundment ^c (m ³ /	s)		0.0015	0.0015	0.0015
Groundwater Discharge Below Impoundment ^d (n	n ³ /s)		0.00022	0.00022	0.00022
Hardness (mg CaCO ₃ /L) Sulphate (mg/L) Nitrate (mg N/L) Total Phosphorus (mg P/L) Total Cyanide (mg/L) WAD Cyanide (mg/L) Total Metals Ag (mg/L) Ai ⁿ (mg/L) As (mg/L) Cd (mg/L) Co (mg/L) Co (mg/L) Cr (mg/L) Fe (mg/L) Hg (ug/L) Mn (mg/L) Mo (mg/L)	10 1 0.014 0.029 0.0005 0.0005 0.0001 0.27 0.001 0.005 0.0001 0.001 0.005 0.0005 0.0006 0.45 0.025 0.025 0.025	- e 1000 ^f 10 ^g - e 0.005 0.0001 0.05 0.05 1 0.0002 0.05 0.002 0.002 0.002 0.002 0.002 0.002 0.002 0.002 0.002 0.002 0.002 0.002 0.002 0.005 0.005	45 70 6.36 0.040 0.001 0.001 0.001 0.001 0.004 0.006 0.0001 0.012 0.001 0.0017 0.45 0.03 0.024 0.005	48 76 6.87 0.041 0.001 0.001 0.001 0.030 0.004 0.007 0.0001 0.013 0.001 0.018 0.045 0.03 0.024 0.006	13 7 0.633 0.030 0.001 0.001 0.001 0.001 0.001 0.006 0.0001 0.002 0.001 0.002 0.001 0.002 0.001 0.0007 0.45 0.03 0.023 0.003 0.003
Pb (mg/L) Sb (mg/L) Se (mg/L) Zn (mg/L)	0.0005 0.001 0.0005 0.0005	0.003 0.05 0.001 0.03	0.0009 0.003 0.0006 0.0012	0.0001 0.003 0.001 0.0012	0.0005 0.001 0.0005 0.0008

a For protection of aquatic life; 30-day average, if applicable..

b

Mean annual. Mean annual. Water quality of this effluent is pilot mill effluent (Table 8.2-1) except total cyanide = 0.002 mg/L, WAD cyanide = 0.001 mg/L, and total Cu = 0.05 mg/L. Water quality of this effluent is pilot mill effluent (Table 8.2-1) except total cyanide = 0.14 mg/L, WAD cyanide = 0.08 mg/L, and total Cu = 0.05 mg/L. Indicates no applicable criterion. Criterion for livesteek watering c d

f Criterion for livestock watering.

Drinking water criterion. Dissolved Al concentration. g h

and nickel concentrations were higher, but the concentrations of these metals predicted in receiving waters (Tables 8.2-2 to 8.2-5) were far below MOE criteria. The higher nitrate concentration indicated by the pilot mill studies, (295 mg N/L) resulted in slightly higher expected nitrate concentrations in Florence Creek. Thus, at closure maximum nitrate concentrations are predicted to be 5.1 mg N/L during an average June, and 6.4 mg N/L during June extreme low flow conditions and 6.9 mg N/L during a one-day low flow event. These levels are well below the drinking water criterion of 10 mg N/L. Due to phosphorus limitation, the nitrate concentrations will not cause increased algal growth beyond the slight enhancement predicted in the Stage II Report.

The greatest difference between the pilot mill and bench scale tests was in the concentration of dissolved aluminum in the liquid portion of the slurry, which was 1.4 mg/L from the pilot mill as compared to 0.27 mg/L from the bench scale tests. In an average June at closure, this concentration of aluminum in pore water displaced from the tailings is predicted to cause the dissolved aluminum level in Florence Creek to increase to 0.29 mg/L. Although this level exceeds the MOE criterion for dissolved aluminum (0.05 mg/L), it is less than 10% greater than the 0.27 mg/L average background concentration in Florence Creek. The maximum aluminum concentration predicted for both a one-day low flow and for an extremely dry June (0.30 mg/L), is 11% greater than the average background level and well within the range of variability observed in the baseline monitoring program. Thus, the predicted increases would be impossible to distinguish from natural variability.

It is likely that actual aluminum concentrations in discharges from the reclaimed impoundments would be lower than the predicted levels due to removal in the wetland/pond system which will be established in the reclaimed impoundments. Published sources give conflicting reports regarding preferential adsorption of aluminum by humic material in wetlands (Cronan et al. 1986, Brown et al. 1987). There is, however, unpublished evidence generated by Conservation and Protection, Environment Canada, that wetlands are capable of removing aluminum from mine drainage, particularly during the summer months (Ferguson 1986). At the Mt. Washington Mine on Vancouver Island, the average summer aluminum concentration (June-September 1986) in mine drainage entering a natural wetland was reduced from 7.9 mg/L to 4.7 mg/L, representing a 40.5% removal efficiency. In this case, water flow through a wetland had been modified by damming to increase the amount of plant/water contact.

The evidence cited for aluminum removal at a high altitude site suggests that the proposed wetlands in the impoundments at Cinola Gold Project would also contribute significantly to aluminum removal.

8.2.3 Assessment of effects on Barbie Creek

ŧ

At the end of operation, tailings water from Impoundment No. 3 will be pumped to the pit. Impacts of outflow from the reclaimed pit to the Barbie Creek drainage were reviewed in the light of the pilot mill findings. Predictions of pit effluent metal concentrations given in the Stage II Report remain valid. Metal levels will be controlled by pH in the pit and by the leaching characteristics of the waste rock in the backfill; which will not change. Therefore, the metals concentrations in the outflow from the pit will not be affected by the change in predicted quality of water pumped from the impoundment. Potential changes in cyanide and nitrate levels in outflow from the pit based on current estimates of concentrations in the impoundment water are discussed below.

In the Stage II Report, cyanide concentrations in outflow from the reclaimed pit were predicted to be below detection levels. Since the pilot mill tests suggest that cyanide levels in the impoundment water would be lower than levels predicted by bench scale work, the cyanide concentrations in the final pit outflow will be lower and undetectable.

The new data suggest that the nitrate concentration in Impoundment No. 3 water will be higher than concentrations predicted by the bench scale tests. As a result, somewhat higher nitrate levels are expected in both outflow from the reclaimed pit and in Barbie Creek (Table 8.2-6). The mean annual nitrate concentration in lower Barbie Creek at closure is now projected to be 0.19 mg N/L, while the maximum (June) nitrate concentrations expected under average and 10 year low flow conditions are 0.71 and 2.4 mg N/L, respectively. These levels are well below the drinking water criterion for nitrate of 10 mg N/L and, due to phosphorus limitations will not cause any increase in algal growth beyond the slight enhancement predicted in the Stage II Report.

8.3 Water Treatment Plant

The expected concentrations of metals and other parameters in the effluent from the water treatment plant have been refined subsequent to the Stage II Report submission, based on recent water treatment plant design tests (Section 7.0). Predicted effluent concentrations are provided in Section 7.3, Table 7.3-10. Impacts of this discharge on Barbie Creek were assessed using predicted effluent values for total metals, except in the case of aluminum, for which the dissolved value was used, consistent with MOE receiving water criteria. An average effluent hardness of 2050 mg CaCO₃/L (based on an average of reported hardness of the supernatant in Settling Test #2, Section 7.0) was used for assessment purposes.

The resulting water quality predictions for lower Barbie Creek, assuming mixing of the discharge from the water treatment facility with flows in lower Barbie Creek, are presented for Years 7 and 12 in Table 8.3-1. Predictions for cumulative impacts of all mine site discharges based in the current data set, are given in Table 8.3-2. (This table includes impacts of all settling ponds discharges.) The concentrations of metals presented in these tables represent worst case values in that they are based on total metal concentrations (except as noted) while in reality, most of the particulate fraction of the various metals is likely to remain in the constructed wetland, with mainly dissolved metals entering Barbie Creek. If only the dissolved metals fraction enters Barbie Creek, the only parameter which will differ in concentration from levels predicted in the Stage II Report is aluminum.

Dissolved aluminum concentrations are expected to be slightly higher than those predicted at Stage II, and to increase slightly over background levels, to a maximum of 0.39 mg/L in year 12 during a dry June (Appendix 8.3-1). This level exceeds the MOE receiving water criterion of 0.01 mg/L for dissolved aluminum at a median pH in Barbie Creek of 5.7, but it is only 9% higher than the average background dissolved aluminum concentration in Barbie Creek of 0.36 mg/L. An increase of this magnitude would be impossible to distinguish from natural variability.

Mercury concentrations will not exceed the MOE criterion of 0.1 ug/L even during extreme low flow (Appendix 8.3-1).

8-11

TABLE 8.2-6

PIT OUTFLOW^a BARBIE CREEK^b January 1.80 0.129 February 1.80 0.134 March 1.80 0.138 April 2.52 0.232 May 3.14 0.350 June 3.59 0.705 July 3.14 0.621 August 4.19 0.489 September 2.80 0.344 October 1.80 0.152 November 1.27 0.110 December 1.27 0.125 Mean annual 2.10 0.190 10 year dry June 12.5 2.35 10 year wet October 1.02 0.097

Nitrate Concentrations (mg N/L) In Outflow from the Reclaimed Pit and Resultant Concentrations In Barbie Creek, Predicted by Norecol, Cinola Gold Project

^a Estimated using flows for Adit Creek at Branch 45 Road

b Barbie Creek downstream of Barbie Wetland.

عزوان

	BARBIE CREEK	MOE RECEIVING	CONCENTRATIONS IN RECEIVING WATER, YEAR 7												
PARAMETERS	BACKGROUND CONCENTRATION (mg/L)	WATER CRITERIA ^a (mg/L)	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	ост	NOV	DEC	MEAN ANNUAL
Lower Barble at Branch 40 Flow Rate (m ³ /s)			0.39	0.34	0.30	0.20	0.11	0.07	0.10	0.10	0.17	0.37	0.54	0.46	0.26
Lime Treatment Discharge Rate ^b (m ³ /s)			0.03	0.03	0.02	0.02	0.02	0.01	0.02	0.01	0.02	0.03	0.03	0.03	0.02
WRS Underdrain Discharge Rate ^c (m ³ /s)			0.0075	0.0074	0.0051	0.0057	0.0041	0.0039	0.0042	0.0037	0.0058	0.0082	0.0089	0.0073	0.0060
Hardness (mg CaCO ₃ /L) Sulphate (mg/L) Total Phosphorus (mg P/L) Orthophosphate (mg P/L)	12 3 0.036 0.0005	_d 1000 ⁰ _d _d	151 124 0.034 0.0009	167 138 0.034 0.0009	137 111 0.034 0.0008	208 174 0.034 0.0010	253 212 0.033 0.0011	372 315 0.032 0.0014	290 244 0.033 0.0012	267 225 0.033 0.0011	236 198 0.033 0.0011	172 142 0.034 0.0009	133 108 0.035 0.0008	128 104 0.035 0.0008	174 144 0.034 0.0009
Total Metals (mg/L)															
AI ¹ As Cd Co Cr Cr Cu Fe Hg (ug/L) Mn Ni Pb Zn	0.36 0.003 0.001 0.001 0.0009 1.65 0.025 0.21 0.001 0.0005 0.0031	0.01 0.05 0.0002 0.05 0.002 0.002 0.30 0.1 0.05 0.025 0.003 0.03	0.36 0.004 0.003 0.001 0.0023 1.72 0.04 0.21 0.003 0.001 0.0052	0.36 0.004 0.003 0.001 0.0025 1.72 0.04 0.21 0.003 0.001 0.0054	0.36 0.004 0.002 0.001 0.0022 1.71 0.04 0.21 0.003 0.001 0.0050	0.37 0.005 0.0001 0.003 0.001 0.0030 1.74 0.04 0.21 0.004 0.001 0.0060	0.37 0.005 0.0001 0.004 0.001 0.0034 1.76 0.05 0.22 0.005 0.001 0.0067	0.37 0.006 0.002 0.006 0.001 0.0047 1.81 0.06 0.22 0.006 0.001 0.0084	0.37 0.005 0.001 0.005 0.001 0.0038 1.78 0.05 0.22 0.005 0.001 0.0072	0.37 0.005 0.0001 0.004 0.001 0.0036 1.77 0.05 0.22 0.005 0.001 0.0069	0.37 0.005 0.0001 0.004 0.001 0.0033 1.75 0.04 0.22 0.004 0.001 0.0064	0.37 0.004 0.003 0.001 0.0026 1.72 0.04 0.21 0.003 0.001 0.0055	0.36 0.004 0.002 0.001 0.002 1.71 0.04 0.21 0.003 0.001 0.0049	0.37 0.004 0.0001 0.002 0.001 1.70 0.04 0.21 0.003 0.001 0.0048	0.37 0.004 0.0001 0.003 0.001 0.0026 1.72 0.04 0.21 0.003 0.001 0.0055

TABLE 8.3-1A Water Quality of Barble Creek After Receiving Effluents from the Lime Treatment Facility and Waste Rock Stockpile Underdrain, Predicted by Norecol for the Cinola Gold Project

For protection of aquatic life.
 Water quality of this effluent is given in Table 7.3-10.
 Water quality of this effluent is given in Vol IV, Table 2.2.2-9.

đ

.

Indicates no applicable criterion. Criterion for livestock watering. Dissolved Al concentration, criterion for median pH 5.7. 1

	BARBIE CREEK	MOE RECEIVING WATER	VIOE CONCENTRATIONS IN RECEIVING WATER, YEAR 12 CONCENTRATIONS IN RECEIVING WATER, YEAR 12 VATER												
PARAMETERS	CONCENTRATION (mg/L)	CRITERIA (mg/L)	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEP	ост	NOV	DEC	MEAN ANNUAL
Lower Barbie at Branch 40A Flow Rate (m ³ /s)			0.39	0.34	0.30	0.20	0.11	0.07	0.10	0.10	0.17	0.37	0.54	0.46	0.26
Llme Treatment Discharge Rate ^b (m ³ /s)			0.03	0.03	0.03	0.03	0.02	0.02	0.02	0.02	0.03	0.04	0.04	0.03	0.03
WRS Underdrain Discharge Rate ^c (m ³ /s)			0.0076	0.0076	0.0052	0.0057	0.0041	0.0039	0.0042	0.0037	0.0058	0.0084	0.0090	0.0074	0.0060
Hardness (mg CaCO ₃ /L) Sulphate (mg/L) Total Phosphorus (mg P/L) Orthophosphate (mg P/L)	12 3 0.036 0.0005	_d 1000 ⁰ _d _d	173 143 0.034 0.0009	191 159 0.034 0.0009	169 140 0.034 0.0009	252 211 0.033 0.0011	332 281 0.032 0.0013	486 413 0.030 0.0016	376 320 0.031 0.0014	357 303 0.032 0.0013	284 239 0.033 0.0012	192 160 0.034 0.0009	147 120 0.034 0.0008	147 120 0.034 0.0008	208 173 0.034 0.0010
Total Metals (mg/L)															
Ai ^f As Cd Co Cr Cu Fe Hg (ug/L) Mn NI Pb Zn	0.36 0.003 0.001 0.001 0.0009 1.65 0.025 0.21 0.001 0.001 0.001	0.01 0.05 0.002 0.002 0.002 0.3 0.1 0.05 0.025 0.003 0.03	0.37 0.004 0.0001 0.003 0.001 0.0025 1.73 0.04 0.21 0.003 0.001 0.05	0.37 0.005 0.0001 0.003 0.001 0.0027 1.74 0.04 0.21 0.004 0.001 0.006	0.37 0.004 0.003 0.001 0.0025 1.73 0.04 0.21 0.003 0.001 0.005	0.37 0.005 0.0001 0.0004 0.001 0.0034 1.77 0.05 0.22 0.004 0.001 0.007	0.37 0.006 0.002 0.005 0.001 0.0042 1.81 0.05 0.22 0.006 0.001 0.008	0.38 0.007 0.0002 0.007 0.001 0.0057 1.89 0.07 0.23 0.008 0.001 0.001	0.37 0.006 0.002 0.006 0.001 0.0046 1.84 0.06 0.22 0.006 0.001 0.008	0.37 0.006 0.002 0.006 0.001 0.0044 1.83 0.05 0.22 0.006 0.001 0.008	0.37 0.005 0.0001 0.005 0.001 0.0037 1.78 0.05 0.22 0.005 0.001 0.007	0.37 0.005 0.0001 0.003 0.001 0.0028 1.74 0.04 0.21 0.004 0.001 0.006	0.36 0.004 0.003 0.001 0.0023 1.72 0.94 0.21 0.003 0.001 0.005	0.36 0.004 0.003 0.001 0.0023 1.72 0.04 0.21 0.003 0.001 0.005	0.37 0.005 0.0001 0.003 0.001 0.0029 1.75 0.04 0.21 0.004 0.001 0.006

Ê

ŧ

ŧ

1

For protection of aquatic life.
 Water quality of this effluent is given in Table 7.3-10.
 Water quality of this effluent is given in Vol IV, Table 2.2.2-9
 Indicates no applicable criterion.
 Criterion for Ivestock watering.
 Dissolved Al concentration, criterion for median pH 5.7.

TABLE 8.3-1B

Water Quality of Barbie Creek After Receiving Effluents from the Lime Treatment Facility and Waste Rock Stockpile Underdrain, Predicted by Norecol for the Cinola Gold Project

8-14

I

ł

	BARBIE CREEK BACKGROUND CONCENTRTION	MOEP RECEIVING	CONCENTRATIONS IN RECEIVING WATER											
PARAMETER		WATER CRITERIA [®]	JAN	FEB	MAR	APR	MAY	JUN	JUL	AUG	SEPT	ост	NOV	DEC
YEAR 7														
Barbie Creek at Branch 40A Flow Rate (m ³ /s) Hardness (mg CaCO ₃ /L) Sulphate (mg/L) Nitrate (mg/L)	- 12 3 0.020	_6 1000 ^c 10 ^d	0.36 133 107 1.64	0.31 145 117 1.56	0.28 124 99 1.87	0.18 174 142 1.87	0.10 208 172 0.702	0.06 278 232 0.516	0.09 230 191 0.405	0.16 219 182 0.496	0.34 191 157 0.421	0.49 147 119 0.740	0.42 119 94 0.522	116 92 1.83
Total Metals (mg/L)														
As Cu Fe Hg(ug/L) Zn	0.003 0.0009 1.65 0.025 0.0031	0.05 0.002 0.3 0.1 0.03	0.005 0.0026 1.63 0.04 0.0083	0.005 0.0027 1.62 0.05 0.0088	0.005 0.0024 1.63 0.04 0.0079	0.005 0.0031 1.62 0.05 0.010	0.006 0.0034 1.62 0.05 0.011	0.006 0.0044 1.61 0.06 0.014	0.006 0.0038 1.62 0.06 0.012	0.006 0.0035 1.62 0.05 0.012	0.005 0.0033 1.62 0.05 0.011	0.005 0.0028 1.62 0.05 0.0089	0.005 0.0024 1.63 0.04 0.0077	0.004 0.0023 1.63 0.04 0.0076
YEAR 12														
Barble Creek at Branch 40A Flow Rate (m3/s) Hardness (mg CaCO3/L) Sulphate (mg/L) Nitrate (mg N/L)	12 3 0.020	یة 1000° 10 ^d	0.36 149 120 1.33	0.31 162 131 1.27	0.27 150 122 1.52	0.18 204 168 149	0.10 263 219 0.556	0.06 346 291 0.402	0.09 288 241 0.323	0.09 281 236 0.395	0.16 222 184 0.344	0.33 161 131 0.608	0.49 130 104 0.434	0.42 131 105 1.50
Total Metals (mg/L)														
As Cu Fe Hg(ug/L) Zn	0.003 0.0009 1.65 0.025 0.0031	0.05 0.002 0.3 0.1 0.03	0.005 0.0028 1.62 0.05 0.0097	0.005 0.0030 1.62 0.05 0.010	0.005 0.0027 1.63 0.05 0.0095	0.006 0.0035 1.62 0.05 0.012	0.006 0.0040 1.63 0.06 0.014	0.007 0.0050 1.62 0.07 0.018	0.007 0.0044 1.62 0.06 0.015	0.006 0.0042 1.63 0.06 0.015	0.006 0.0037 1.61 0.06 0.013	0.005 0.0031 1.61 0.05 0.010	0.005 0.0026 1.62 0.04 0.0089	0.005 0.0026 1.62 0.04 0.0089

TABLE 8.3-2 Water Quality of Barbie Creek After Mixing With All Mine Site Discharges, Predicted by Norecol for the Cinola Gold Project

a For protection of aqautic life.
 b No applicable criterion.
 c Criterion for livestock watering.
 d Drinking water criterion.

Mercury discharged from the water treatment facility will be predominantly in the form of cinnabar or adsorbed to iron hydroxide. This particulate mercury will be removed in the constructed wetland. The detectable mercury predicted during summer low flows (Table 8.3-2) do not reflect removal in the wetland. Mercury will therefore not be transported or be available for methylation and mercury levels in fish will not increase.

Based on new total copper concentrations expected in the water treatment effluent higher copper concentrations are predicted for the receiving waters. The maximum potential copper concentrations in Barbie Creek based on the data from the water treatment plant test program are 0.0050 mg/L during an average June (Table 8.3-2) and 0.0073 mg/L during a dry June (Appendix 8.3-1, Table 2). These values will be lowered by approximately 50% through removal of copper in the constructed wetlands. The higher copper concentrations will be further mitigated by an increase in water hardness. In addition, irrigation of treated flows during the summer low flow period remains as a viable contingency option.

Since copper concentrations will vary directly with hardness, the receiving water criterion will be met at all times. Based on a simple dilution calculation, the average hardness in Barbie Creek (hardness of the effluent 2050 mg CaCO₃/L the effluent) is predicted to increase to between 150 and 200 mg CaCO₃/L (Tables 8.3-1 and 8.3-2). During operation, hardness in the receiving water is expected to exceed 100 mg CaCO₃/L at all times, and in low flow periods, it could exceed 300 mg CaCO₃/L. Thus, the MOE receiving water criterion for copper (0.00004 x receiving water hardness) will be between 0.004 and 0.012 mg/L.

The predicted hardness in Barbie Creek is generally within the upper range (mid to upper 200 mg CaCO₃/L) reported for streams of Canada's Pacific and central regions (CCREM 1987). Maximum hardness is within the range found in groundwaters, including groundwaters which have been used in salmonid hatcheries (McLeay, pers. comm.).

The predicted increases in level of hardness in Barbie Creek are not considered harmful to aquatic life. There are no established criteria or guidelines for aquatic life which limit hardness; rather, increased hardness is viewed as beneficial since it reduces the toxicity of numerous heavy metals.

Sulphate concentrations in Barbie Creek are predicted to increase to approximately 100 to 200 mg/L and to exceed 500 mg/L during extremely low flows. These levels are well below 1 000 mg/L, the MOE criterion for livestock watering (no criteria have been established for the protection of aquatic life). Thus, the availability of Barbie Creek as a drinking water supply for wildlife, the only water use potentially affected by sulphate concentration, will be protected.

8.4 Summary

The impact assessment presented in Volume IV of the Stage II Report has been updated, based on the results of pilot scale metallurgical testing and additional water treatment testwork. The numerical predictions have been refined, but the new data available have not changed the original conclusion of no significant impacts on water quality in either the High West area or the mine area.

The pilot mill testwork, which is more representative of operating conditions than the bench scale testing, resulted in lower effluent concentrations of cyanide, copper, and mercury and slightly higher

8-17

concentrations of nitrate and aluminum. At the lower cyanide and copper levels, natural cyanide degradation will not be required for outflow from reclaimed Impoundment Nos. 1 and 2 to meet receiving water criteria in Florence Creek. Nitrate in the mill effluent will result in higher nitrate levels in Barbie Creek (due to transfer of Impoundment No. 3 water to the reclaimed pit) as well as in Florence Creek. However, because of phosphorus limitation, the higher nitrate concentrations will not cause any additional algal growth. The aluminum concentration in Florence Creek will increase slightly, but the increase will be impossible to distinguish from background variability and therefore is not considered an impact on water quality.

Slightly higher levels of all metals are predicted in Barbie Creek, based on use of total rather than dissolved metals in the mass balance calculations for discharge from the water treatment plant. The conservative approach assumes that some particulate metals pass through Wetland MSW1 and enter the creek. However, a predicted increase in hardness of Barbie Creek will counteract increased metals concentrations such that all metals will meet receiving water criteria, except in those instances where background concentrations already exceed criteria. Dissolved aluminum concentrations will also increase slightly, but the increase will be impossible to distinguish from background variability and therefore does not represent an impact on water quality.

SECTION 9.0

. .

9.0 TEMPORARY MINE SHUTDOWN AND PREMATURE CLOSURE

9.1 Introduction

The conceptual plans for mine site reclamation have been presented in Volume IV, Section 2.5 of the Stage II Report. These plans involve a program of progressive, sequential reclamation leading to complete reclamation of the mine site on final closure.

In addition to planned mine closure following depletion of the ore reserves, temporary shutdown or premature closure of the mine might occur at any stage in the mine life due to a wide variety of causes. In the event of such an unforeseen occurrence, City Resources would complete the required reclamation to suit the conditions at the time of curtailment of operations and for the duration of the curtailment, if that curtailment is to be temporary.

The following material outlines three typical scenarios for suspension of operations and describes the actions that City Resources would take to complete reclamation in each case.

9.2 Temporary Shutdown

A temporary shutdown could occur at any time during the life of the mine. The duration of this type of shutdown would be dependent upon the perceived situation and economics at that time. Operations would resume as soon as the reason for the shutdown is no longer applicable. The reclamation measures would therefore be temporary, and devised to maintain environmental integrity without alienating any part of the ore deposit or mine site from subsequent reactivation. The measures required to respond to a temporary shutdown would be based on procedures developed for the Water Management Plan. The mine operation would be expected to resume and therefore no effort would be made to backfill, regrade or revegetate any of the components of the mine which would later be reactivated.

In the High West area, poor quality water would be stored and monitored in the tailings impoundment that was in use at the time of the shutdown. Several methods for disposal of poor quality water at the end of mine operation were evaluated and presented in Volume IV, Section 2.2.3.3 of the Stage II Report. These methods included the following:

- a) Direct discharge of the impoundment water to Florence Creek;
- b) Land irrigation in the Florence Creek watershed;
- c) Discharge via Wetland MSW1 to Barbie Creek; and
- d) Discharge to the open pit.

9-1

The preferred option at the end of the planned mine operation would be to pump the water from Impoundment No. 3 into the pit during backfilling operations. This option is not applicable in the event of a temporary mine shutdown. In the case of a long shut down and/or heavy rainfall, disposal of some or all of the stored water would be considered based on the available storage capacity at that time. Discharge of poor quality water using a combination of the first three methods listed above, with modifications, would be evaluated. Modifications to these methods might include land irrigation in the mine area and other watersheds, and discharge to Barbie Creek via Wetland MSW2.

The volume of water disposed of by each method would be based on the quality of the impoundment water, total volume to be disposed, time of year at shutdown, result of wetland monitoring and other pertinent site conditions.

Additional measures that would be considered, again depending on conditions at the time of shutdown and the expected duration, would involve establishing a wetland system in or adjacent to the active tailings impoundment and constructing embankment spillways. These measures would result in facilities similar to those proposed in the final reclamation plan.

In the mine area the water management systems would be maintained in operational condition throughout the shutdown. The waste rock stockpile would be covered with additional mudstone to reduce precipitation infiltration. The water treatment plant would operate throughout the shutdown.

Regular inspections of the open pit slopes would be carried out. Temporary slope stabilization measures would be developed and implemented as necessary.

9.3 Permanent Early Closure

Two scenarios have been developed to establish the course of action City Resources would undertake to complete reclamation should permanent closure occur prior to the planned final closure at the end of 12 years. Premature closure events at year 3 and year 7 have been considered. These years represent the end of mining operation phases one and two respectively, as described in Volume II, Section 3.4 of the Stage II Report.

Year 3 was chosen since there will be a minimum number of embankments constructed at that time, with no reclamation of the impoundment started. The final reclamation plan includes backfilling the open pit with potentially acid generating waste rock to a level below the final groundwater elevation (125 m). The object of this part of the plan is to store the waste below water in the long-term, but prior to year 7 the open pit has no storage capacity for backfill below elevation 125 m. Consequently year 7 represents a worst case in terms of tonnage of potentially acid generating rock that would require disposal in the event of permanent early closure, as discussed in Section 9.3.2.

9.3.1 Permanent closure: year 3

9.3.1.1 Project facilities

At the end of year 3 the project facilities in the High West area (Stage II Report, Volume II, Figure 6.5.1-1) will be developed to the following extent:

- a) Water storage reservoir. Embankment construction will be complete and the facility will be in full operation.
- b) Impoundment No. 1. The crest elevation of Embankment No. 1 will be 306 m and Impoundment No. 1 will contain 7.2 · 10⁶ of tailings and waste rock.
- c) Impoundment No. 3. The crest elevation of Embankment No. 3 will be 230 m. The impoundment will be serving as a sediment control structure.
- d) Low grade ore stockpile. The stockpile will contain 2.6 ⁻ 10⁶ of low grade ore and the embankment collecting drainage from the stockpile will be in place.
- e) Water management diversion ditches. Clean runoff diversion ditches associated with the water storage reservoir, Impoundment No. 1, low grade ore stockpile and mill will be in place and operating.
- f) Mill site. All mill facilities will be operational, including the runoff collection pond.
- g) Other facilities. The distribution road from the mill site to Impoundment No. 1, fresh water distribution pipeline and tailings pipelines will be in operation. The soil stockpile to the west of the water storage reservoir will be in place with natural revegetation being allowed to establish.

In the mine area the project facilities (Stage II Report, Volume II, Figure 3.4.1-2) will be developed to the following extent:

- h) Open pit. The bottom of the open pit will be at elevation 130 m.
- i) Waste rock stockpile. The stockpile will contain a total of 4.1 · 10⁶ of potentially acid generating waste and construction materials.
- j) Mudstone dump. The dump will contain 4.2 · 10⁶ of mudstone.
- k) Overburden stockpile. The stockpile will contain 3.0 · 10⁶ of overburden soils.
- I) Water management structures. All water management ditches and settling ponds will be in place and operational.

- m) Water treatment facility and wetlands. The treatment plant, equalization pond and wetlands will all be in operation.
- n) Other facilities. The relocated Branch 42 Road, mine office and explosives magazine will all be operational.

9.3.1.2 Reclamation Activities

At the end of year 3 the planned progressive reclamation of the High West area will not have been initiated. In the event of permanent closure at year 3, reclamation would consist of the following activities:

- a) Embankment No. 1 would be constructed to final design elevation (310 m).
- b Potentially acid generating waste rock would be transported from the mine area stockpile to the High West area and placed below water in Impoundment No. 1 and in the water storage reservoir. The volume of water that would be required to provide a water cover to this material lies between $0.5 \cdot 10^6$ m³ and $0.8 \cdot 10^6$ m³. This water would be obtained from a combination of storage in Impoundment No. 1 (minimum $0.1 \cdot 10^6$ m³), inflow due to precipitation during reclamation (average $0.6 \cdot 10^6$ m³), and the water storage reservoir ($1.7 \cdot 10^6$ m³). Additional limestone could also be added to the waste rock during disposal to the reservoir or Impoundment No. 1 as required.
- c) The low grade ore would either be processed in the mill before disposal in Impoundment No. 1 or placed directly underwater in Impoundment No. 1 and the water storage reservoir, depending on the results of environmental monitoring and economic criteria that would be evaluated in year 3.
- d) Impoundment No. 1 would be reclaimed as outlined in the detailed reclamation plan (Stage II Report, Volume IV, Section 2.5), including construction of a wetland and permanent spillway. It is not expected that there would be excess water in Impoundment No. 1 that would require disposal. However, should there be poor quality water remaining in the impoundment after placing of the waste, this would be disposed of using the methods described for a temporary shutdown (Section 9.2).
- e) Embankment No. 3 and the sediments contained in this impoundment would be removed and placed in Impoundment No. 1.
- f) The diversion ditches, all mill site and low grade ore stockpile facilities and roads would be reclaimed as outlined in the final reclamation plan.

In the event of permanent closure at year 3, reclamation of the mine area would consist of the following activities:

- g) A mudstone/overburden cover combined with a surface layer of sufficient ground limestone to abate the remaining acid generating potential of the exposed surfaces would be placed over slopes in the open pit in which potentially acid generating material is exposed. The volume of mudstone/overburden/limestone incorporated in the cover is relatively small and cannot be considered an economic impediment to subsequent potential extraction of the remainder of the reserves.
- h) All material contained in the waste rock stockpile would be recovered and disposed of in the High West area as described above.
- i) The material remaining in the mudstone dump and overburden stockpile after reclamation activities would be left with stable side slopes and revegetated as outlined in the reclamation plan.
- j) The water treatment facility, wetlands and appropriate water management structures would be maintained and operated consistent with the commitments in the final reclamation plan.
- k) Reclamation of the remainder of the mine area, including demolition of the mine plant, would be carried out as outlined in the final reclamation plan.

9.3.2 Permanent closure: year 7

The open pit is assumed to have negligible storage capacity below the final groundwater elevation at year 7. Under this assumption, the tonnage of potentially acid generating waste rock that would be required to be stored underwater in the High West tailings/waste rock impoundment in the event of premature closure is a maximum at the end of year 7.

9.3.2.1 Project facilities

At the end of year 7 the project facilities in the High West area (Stage II Report, Volume II, Figure 6.5.1-2) will be developed to the following extent:

- a) Water storage reservoir. This facility will be in full operation as described for the end of year
 3.
- b) Impoundment No. 1. The operating period of this impoundment will have ceased at the end of year 4. At the end of year 7 the embankment will be at final design elevation (310 m), the impoundment will contain 10.2 · 10⁶ of tailings and waste rock, and reclamation of the impoundment will be complete.
- c) Impoundment No. 2. The four-year operating period of this impoundment will have begun at the beginning of year 5. At the end of year 7 the crest elevation of both the saddle and Embankment No. 2 will be 282 m and the impoundment will contain 9.3 · 10⁶ of tailings and waste rock.

- d) Impoundment No. 3. This impoundment will be operating for purposes of sediment control as described for the end of year 3.
- e) Low grade ore stockpile. The stockpile will have been processed and the area completely reclaimed.
- f) Water management diversion ditches. Clean runoff diversion ditches associated with the water storage reservoir, Impoundment No. 1, Impoundment No. 2 and the mill will be operational.
- g) Mill site and other facilities. The mill and other High West facilities will be in operation as described for the end of year 3.

At the end of year 7 the project facilities in the mine area (Stage II, Volume II, Figure 3.4.1-3) will be developed to the following extent:

- h) Open pit. The bottom of the open pit will be at elevation 90 m.
- i) Waste rock stockpile. The stockpile will contain a total of 15.6 10⁶ of potentially acid generating waste and construction material.
- j) Mudstone dump. The dump will contain 8.5 · 10⁶ of mudstone and rhyolite.
- k) Overburden stockpile. The stockpile will contain 2.9 · 10⁶ of overburden soils.
- I) Water management and treatment facilities, wetlands and other mine area facilities. These facilities as described in Section 9.3.1 and shown on Figure 3.4.1-3 in Volume II of the Stage II Report will be operational.

9.3.2.2 Reclamation activities

Prior to the end of year 7 the planned staged reclamation of the High West area will have been initiated. By the end of year 7 Impoundment No. 1 will be reclaimed. In the event of permanent closure at the end of year 7, reclamation would consist of the following activities:

- a) Embankment No. 2 and the saddle embankment would be constructed to final design elevation (290 m).
- b) Embankment No. 3 would be constructed to elevation 263 m.
- c) The material contained in the waste rock stockpile in the mine area would be transported to the High West area and placed below water in the water storage reservoir, Impoundment No. 2 and Impoundment No. 3. Additional limestone could be added to the waste rock as required for disposal. The total volume of water that would be required to provide a water cover to this material lies between 2.0 · 10⁶ and 3.0 · 10⁶ m³. This water would be obtained from the water storage reservoir (1.7 · 10⁶ m³), Impoundment No. 2 (minimum 0.1 · 10⁶ m³),

Impoundment No. 3 (minimum $0.1 \cdot 10^6 \text{ m}^3$) and precipitation on the impoundments during reclamation (average $1.8 \cdot 10^6 \text{ m}^3$ per year).

- d) Impoundment Nos. 2 and 3 would be reclaimed as outlined in the detailed reclamation plan, including construction of wetlands and permanent spillways.
- e) The diversion ditches, mill site, roads and other facilities would be reclaimed as outlined in the reclamation plan.

The reclamation activities in the mine area, in the event of permanent closure at the end of year 7, would be as described for year 3 in Section 9.3.1.1.

SECTION 10.0

.

10.0 EXPANDED STATEMENT OF COMMITMENT

10.1 Introduction

The major commitments to be undertaken by City Resources (Canada) Limited were outlined in the appropriate sections of the Stage II Report. These commitments related to technical, environmental and socio-economic issues.

Subsequent to the submission of the Stage II Report, a number of issues were raised that either were not specifically addressed by the commitments initially undertaken by City Resources, or required additional emphasis. These are discussed below and include:

- a) reclamation in the event of mine shutdown or closure prior to planned ultimate mine life;
- b) reclamation funding;
- c) post Stage II environmental monitoring and sampling; and
- d) training.

10.2 Premature Closure or Temporary Mine Shutdown

The conceptual plans for mine site reclamation have been presented in Volume IV, Section 2.5 of the Stage II Report. The basic concept involves a program of progressive, sequential reclamation leading to complete reclamation on final closure of the mine site. City Resources has accepted the commitment to complete the reclamation as part of the mine site abandonment procedures.

In addition to planned mine closure following conventional depletion of the ore reserves, temporary shutdown or premature closure of the mine could occur at any stage in the mine life due to a wide variety of causes. In the event of such an unforeseen occurrence, City Resources will complete the required reclamation to suit the conditions at the time of curtailment of operations and for the anticipated duration of the curtailment. In addition to moving up the schedule for the reclamation activities, this may also include altering the planned sequence of the activities. The planned reclamation procedures will result in an environmentally sound closure of the mine site.

Three potential scenarios for temporary suspension or premature closure of operations and the actions City Resources will take toward reclamation in each case are described in Section 9.0 of this Addendum Report.

10.3 Reclamation Funding

City Resources will undertake to set up a vehicle for providing funds to cover the costs of reclamation. Of the alternatives available, a trust fund presently appears to be the most appropriate means of providing for the costs of reclamation. This will be evaluated as part of the Stage III work preparatory to permitting.

The fund would accumulate year by year over the life of the mine and so would be available for early closure as well as for final abandonment.

10.4 On-going Environmental Monitoring and Sampling

Throughout the operational life of the mine and for a period after closure, it will be necessary to perform environmental monitoring and sampling. These activities are required to ensure compliance with permit criteria, as well as to refine reclamation procedures.

City Resources intends to fund an independent organization to undertake the environmental monitoring and sampling. The Company plans to discuss the terms of reference and the composition of the monitoring body with local, Island groups. The scope and extent of the monitoring program will be a matter for discussion in the Stage III process prior to permitting.

10.5 Training

City Resources has long recognized that the success of the project largely depends on their involving and utilizating locally available manpower in developing and operating the mine.

In Volume VI, Section 3.0 of the Stage II Report, the recruitment and training policies to be adopted by City Resources are listed in detail. The Company will adhere to these policies and will provide the training for the pre-production and operational phases necessary to ensure that the opportunities presented by the Cinola Gold Project are fully available to the local residents.

11.0 ERRATA AND OMISSIONS

The following sections provide errata in, and omissions from, the Cinola Gold Project Stage II Report, Volumes I to VI plus Appendices. The few remaining typographical errors which do not affect clarity or meaning have not been included.

11.1 Volume I

11.1.1 Errata

Page	For	Read
1-3,line 12	\$14 million	\$5.9 million
1-3,line 13	for goods	for labour, goods
	another	[delete]
1-3,line 14	Approximately	Up to
6-4,line 31	\$110 million	\$71 million
6-4,line 32	\$43.8 million per annum	\$38.9 million
	during the operational period and \$8.4 million	per annum during the operational period and \$4.8 million.

11-1
11.2 Volume II

11.2.1 Errata

Page	For	Read
title page	Prepared for:	Submitted by:
v,line 18 v,line 19	3.5 Pit dewatering 3.5.5 Pit	3.5.5 Pit dewatering 3.5.5.1 Pit
v,line 20 1-4.line 28	3.5.5.3 Selected 43.5 10 ⁶ /t	3.5.5.2 Selected 43.5 [.] 10 ⁶ t
1-6,line 22	may	any [dalata]
1-6,line 32	(Figure 1.4.2-1) minesite.	minesite (Figure 1.4.2-1).
1-6,line 31 3-48, table	MW power [Delete data and replace with th	MW. Power e following:]

TABLE 3.5.1-2ANNUAL STRIPPING RATIOS WASTE TO ORE CINOLA GOLD PROJECT

YEAR	STRIPPING RATIO	COMMENTS
 Preproduction	N/A	
1	0.52:1	Ore stockpiled
2	0.92:1	Ore stockpiled
3 ^a	3.71:1	Stockpile reclaimed
4	4.54:1	Stockpile reclaimed
5	2.64:1	Stockpile reclaimed
6	6.60:1	Stockpile reclaimed
7	2.05:1	
8	2.05:1	
9	2.05:1	
10	2.05:1	
11	1.94:1	
12	1.38:1	
OVERALL	2.07:1	

^a The ore stockpiled in years 1 to 3 was not included in the waste tonnage.

Page

For

Read

 3-24,line 9
 16 000 t/day

 3-49,line 17
 3.5 Pit dewatering

 3-49,line 18
 3.5.5 Pit water

 3-49,line 25
 west wall

18 600 t/day 3.5.5. Pit dewatering 3.5.5.1 Pit water east wall -

Page	For	Read
3-52,line 14	3.5.5.3	3.5.5.2
3-52,line 18	at 5	at 5 ⁰
3-52,line 19	300 000 m	60 000 m
3-52,line 29 to 38	As far ditches	[move to follow first
		paragraph on page 3-56]
3-63,table [row 4]	20 20 50 20 50	50 50 50 50 50
3-65,line 1	% argillically	<% argillically
3-65,line 2	60% argillically	60% argillically
3-119, Table 3.8.3-1	Total 426 500	Total 394 500
5-5, line 7	-1.8°C	-1.3°C
5-22, table	Fe ₂ O ₃	Fe ₂ O ³ 3.8 2.5
5-24	[Delete applicable data and replace with following:]	

VALUE	UNITS	
2 100 000	t/a	
1 226 000	t/a (maximum)	
52	wks/a	
6	d/wk	
20	h/d	
80	% (daily basis)	
7350	t/h	
459	t/h	
1000	mm	
230	mm	
200	m	
1.5	t/m ³	
12 500	t	
2 000 000	t maximum	

Page		For	Read
5-27, line 1	2	[Delete formula and replace with	following:]
2	$FeS_2 + 10HNO_3 = Fe^2(S)$	O4)s + 10NO + H2SO4 + 4H2	0
5-32,line 2	5	1% NaOH and 2%	2% NaOH and 0.2%
5-35,line 2	6	Feeport's	Freeport's
5-35,line 2	7	Gerrit	Jerrit

Page	For	Read
6-56,line 11	Lindsey 1975	Linsley, Kohler and Paulhus 1975
7-2,line 3	Section 3.5.7	Section 3.5.6
7-3,line 7	Figure 6.5.1-2	Figure 6.5.1-1
7-3,line 20	mudfill	rockfill
7-4,line 2	Figures 6.5.1-2 and 3.5.7-1	Figures 6.5.1-1 and 3.5.6-1
7-16,line 3	Figure 3.5.6-1	Figure 3.5.5-1
7-43,line 4	Section 3.3	Section 3.4
7-43,lines	Pumping of the	Trucking of the
17,19,21		
7-44,table	[delete applicable data and repla	ace with following:]

DRINKING WATER	FEDERAL AQUATIC	PROVINCIAL AQUATIC
500	-	-
500	-	-
-	0.1	0.05 ^e
0.5	0.05	0.05
0.005	0.0002	0.0002
-	-	0.05
-	0.002 ^f	0.002 ^f
1.0	0.002	0.002
0.3	0.3	0.3
0.001	0.0001	0.0001
0.05	-	0.1
-	0.025	0.025
0.05	0.001	0.003
5.0	0.03	0.03

Page	For	Read
-		
7-52,table	TIME REQUIREMENT	LIME REQUIREMENT
7-52,table	(\$)	(\$/a)
7-56,line 24	17 000 t	130 000 t
7-58,line 32	the cofferdam for	the first lift of
7-65,line 21	Figures 6.5.4-2 and 6.5.4-4	Figures 6.5.1-1 to 6.5.1-3
7-65,line 23	Section 3.2	Section 6.3
7-66,line 12	1 in 2000-year	1 in 200-year
7-66,line 37	Figure 6.5.1-4	Figure 6.5.1-3
7-71,line 17	Section 6.8.8	Section 6.8
8-1,line 19	Figure 4.1-1	Figure 4.2-1

Page	For	Read
8-1,line 23	Proposed Diesel	Proposed Power
8-1,line 31	No. 32 diesel	No. 2 diesel
Figure 9.2.2-1	[Roads near pit are incorrect as the Relocated Branch 42 and the main Access/Haul Road are not connected]	
10-8, line 19	5 to 60%	50 to 60%
10-15,line 25	in the general vicinity of the primary crusher	[delete]

11.2.2 Omissions

Figure 9.1.4-1 Scale 1:1000

References

- B.C. Ministry of Environment. 1979. Pollution control objectives for the mining, smelting and related industries. Pollution Control Board.
- B.C. Ministry of Environment. 1987. Approval and working criteria for water quality.
- Canada Council of Resource and Environment Ministers. 1987. Canada water quality guidelines. Environment Canada.
- Ministry of Health and Welfare. 1978. Guidelines for Canadian drinking water quality. Supply and Services Canada.

11.3 Volume II Appendices

11.3.1 Errata

Page	For	Read
title page	Prepared for:	Submitted by:

11.4 Volume III

11.4.1 Errata

Page	For	Read
title page	Prepared for:	Submitted by:
1-4,line 28	43.5 ⁻ 10 ⁶ /t	43.5 [.] 10 ⁶ t
1-5,line 1	Charlotte Main	Port Clements
1-6,line 29	(Figure 1.4.2-1)	[delete]
1-6,line 22	may	any
1-6,line 31	MW power	MW. Power
1-6,line 32	minesite.	minesite (Figure 1.4.2-1).
2-16,line 14	A re-glacial	A pre-glacial
5-28, table	Average Annual Runoff	Average Annual Runoff
	(mm/d) 44.86	(mm/d) 5.86
5-28,table	[Column 2]	[Column 2]
	44.86	5.86
5-115,line 14	andhigher	and higher
5-141,line 15	intrate	nitrate
5-352, table	[source]	Source: Thomas and
		Goyette (in prep.)
5-353,line 11	(1985)	(1982)
R-2,line 1	Phase I. Victoria	Phase I. Water quality
		in Region 8, the lower
		Columbia River Basin.
		Victoria.
R-3,line 23	race	trace
R-4,line 10	Canada Land Inventory	Canada Land Inventory.
	1972. Reprint. Silt	1965. Soil
R-7,line 27	1985	1985a
R-7,line 29	1985	1985b
R-7,line 33	Evenson, W.E. 1981	Evenson, W.E.,
		S.R. Rushforth,
		J.D. Brotherson and
		N. Fungladda. 1981.
R-14,line 8	1978	1978a
R-14,line 11	1987	1978b
R-16,line 7	Salanki, J. 1982	Salanki, J.,K.V. Balogh
		and E. Berta. 1982
R-16,line 18	Screenivasa	Sreenivasa
R-17,line 3	Smith, A.L. et al.	Smith, A.L., G.L. Ennis,
		S.W. Sheehan and
		T.M. Tuominen
R-19,line 10	Wong, P.T.S. et al.	Wong, P.T.S., Y.K.
		Chau, L. Luxon and
		G.A. Bengert.

-

-

١.

-

ŰD:

1

×.

11.4.2 Omissions

References

- American Public Health Association. 1975. Standard methods for the examination of water and waste water. 14th Edition. Washington, D.C. : American Public Health Association, American Water Works Association, and Water Pollution Control Federation. 1268 pp.
- American Public Health Association. 1985. Standard methods for the examination of water and waste water. 16th Edition. Washington, D.C.: American Public Health Association, American Water Works Association, and Water Pollution Control Federation.

Archibald, R. Personal Communication. 1988. B.C. Ministry of Environment. Victoria, B.C.

- B.C. Ministry of Environment. 1976. A laboratory manual for the chemical analysis of waters, wastewaters, sediments and biological materials. 2nd Edition. Victoria, B.C.: Environmental Laboratory.
- B.C. Ministry of Environment. 1978. Kootenay air and water quality study, Stage II; water quality in the Elk and Flathead River basins. Victoria, B.C.: Ministry of Environment, Water Investigations Branch.
- B.C. Ministry of Environment. 1981. Kootenay air and water quality study Phase II, water quality in the Kootenay River basin. Victoria, B.C.: Ministry of Environment, Aquatic Studies Branch.
- Beak Consultants Limited. 1980. Fertilization of Yakoun Bay: a preliminary study under the salmonid enhancement program. Prep. for Supply and Services Canada. Richmond, B.C.: Beak Consultants Limited.
- Beijer K., and A. Jernelow. 1979. Methylation of mercury in aquatic environments. In: J.O. Nriagu (Ed). The Biogeochemistry of Mercury in the Environment. New York: Elsevier/North-Holland Biomedical Press. pp. 203-210.
- Blackman, M. 1979. Northern Haida land and resource utilization. In: Tales from the Queen Charlotte Islands. pp. 43-55.
- Breitkreutz, A. Personal Communication. 1980. Conservation Officer. B.C. Fish and Wildlife Branch. Queen Charlotte City.
- British Columbia Ministry of the Environment. 1976. A laboratory manual for the chemical analysis of waters, wastewaters, sediments and biological materials. 2nd Edition. Environmental Laboratory, Victoria, B.C.
- Brooks, R.R. and M.G. Rumsby. 1965. The biogeochemistry of trace element uptake by some New Zealand bivalves. Limnology and Oceanography 10:521-527.

- Brown, D.H. 1982. Mineral nutrition. In: A.J.E. Smith. (Ed). Bryophyte Ecology. London and N.Y.: Chapman and Hall. pp. 383-444.
- Bruce, P.G. 1979. Mamin River fisheries reconnaissance. Photocopy. Nanaimo, B.C.: MacMillan Bloedel Limited, Queen Charlotte Division.
- Bustard, D.R. and D.W. Narver. 1975. Aspects of the winter ecology of juvenile coho salmon (<u>Oncorhyncus leisoteh</u>) and steelhead trout (<u>Salmo gairdneri</u>). Journal of the Fisheries Board of Canada, 32: 667-680
- Canada Food and Drug Directorate. 1979. Food and drug regulations. Ottawa: Health Protection Branch.
- Canada Land Inventory. 1965. Soil capability classification for agriculture. The Canada Land Inventory Report No. 2. Ottawa, Ontario: Lands Directorate, Department of the Environment.
- Cinola Operating Company. 1980. Map, unpublished M.S. Vancouver: Cinola Operating Company.
- City Resources (Canada) Limited. 1987. Cinola Gold Project Stage 1 update report. Prep. by Norecol Environmental Consultants Ltd. Various paging.
- Clifford, W. Personal Communication. 1987. B.C. Ministry of Forests and Lands. Victoria, B.C.
- Coatta, E. Personal Communication. 1987. Atmospheric Environment Service, Environment Canada. Vancouver, B.C.
- Collison, F. Personal Communication. 1987. Council of the Haida Nation. Massett, B.C.
- Cowan, I. McT. 1956. What and where are the mule and blacktailed deer. In: W.P. Taylor. (Ed). The Deer of North America. Harrisburg, Penn: The Stackpole Company. 668 pp.
- DFO and MOEP: see Department of Fisheries and Oceans and B.C. Ministry of Environment and Parks.
- de Leeuw, A.D. Personal Communication. 1987. B.C. Ministry of Environment and Parks. Fish and Wildlife Branch. Queen Charlotte City, B.C.
- Dalzell, K.E. 1968. The Queen Charlotte Islands Volume 1. 1774-1966. Queen Charlotte City: Bill Ellis. 340 pp.
- Davies, D. Personal Communication. 1987. Yakoun River Hatchery.
- Describing Ecosystems in the Field. 1980. RAB Technical Paper 2, Resource Analysis Branch, B.C. Ministry of Environment, Victoria, 224 pp.

- Dinman, B.D. and L.H. Hecker. 1972. The dose-response relationship resulting from exposure to alkyl mercury compounds, In: R. Hartung (Ed.). Environmental Mercury Contamination. Ann Arbor, Mich: Ann Arbor Science Publishers Inc. pp. 290-301.
- D'Itri, F.M. 1973. Mercury in the aquatic ecosystem. In: G.E. Glass, (Ed). Bioassay Techniques and Environmental Chemistry. Ann Arbor, Mich.: Ann Arbor Science Publishers pp. 1-70.
- Dreher, W. Personal Communication. 1987. B.C. Ministry of Environment. Smithers, B.C.
- Dunbar, D. Personal Communication. 1987. Department of Fisheries and Oceans. Masset, B.C.
- Duncan, J. Personal Communication. 1987. MacMillan Bloedel Ltd. Juskatla, B.C.
- Environment Canada. 1980. Canadian climate normals, temperature and precipitation, 1951-1980, British Columbia. 268 pp.
- Environment Canada. 1987. Surface water data reference index, 1986. Ottawa, Ontario: Inland Waters Directorate, Water Resources Branch, Water Survey of Canada. 401 pp.
- Environment Canada. 1987. Unpublished Data. Provided by Oliver Nagy, Water Survey of Canada. Vancouver.
- Environment Canada 1980, 1981, 1982. Streamflow summaries, British Columbia. Ottawa: Water Survey of Canada. Ottawa.
- Fladmark, K. 1970. Preliminary report on the archaeology of the Queen Charlotte Islands In: R.L. Carlsons (Ed.). Archaeology in British Columbia, New Discoveries. B.C. Studies No. 6-7. pp. 18-45.
- Furukawa, K., and K. Tonomura. 1973. Cytochrome C involved in the reductive decomposition of organic mercurials. Purification of cytochrome C-1 from mercury-resistant <u>Pseudomonas</u> and reactivity of cytochrome c from various kinds of bacteria. Biochem. Biophys. Acta. 352:413-423.
- Goldwater, L.J. 1972. Detection and appraisal of subclinical intoxication. In: R. Hartung (Ed.). Environmental Mercury Contamination, Ann Arbor, Mich: Ann Arbor Science Publishers inc. pp. 319-325.
- Gunter, S. Personal Communication. 1987. B.C. Ministry of Environment and Parks. Victoria, B.C.
- Ham, L. 1988. An archaeological impact assessment of the Cohoe Creek site (FjUb10), Port Clements, Queen Charlotte islands, B.C. (Permit 1988-4). Report on file with Resource Information Centre, Resource Management Branch, Victoria, B.C.

- Hustedt, F. 1930. Bacillariophyta (diatomeae). In: A. Pascher (Ed.) Die Susswasser-flora Mitteleuropas. 10. Gustav Fischer, Jena.
- IEC Beak. 1982. Cinola Operating Company, Queen Charlotte Gold Project, draft Stage II report. Prep. for Cinola Operating Company, Vancouver, B.C.
- McKeague, J.A. and B. Kloosterman. 1974. Mercury in horizons of some soil profiles in Canada. Can. J. Soil Sc; 54:503-509.
- National Building Code of Canada. 1985. Associate Committee on the national building code. Ottawa: National Research Council of Canada. 454 pp.
- National Research Council of Canada, Associate Committee on Scientific Criteria for Environmental Quality. 1979. Effects of mercury in the Canadian environment. Environmental Secretariat Publication No. 16743. Ottawa, Ontario.
- Tripp, D. 1986. Using large organic debris to restore fish habitat in debris-torrented streams. Land Management Report No. 47. Victoria, B.C. : B.C. Ministry of Forests and Lands.
- WMB. 1987. Unpublished data from Brian Wilkes, Waste Management Branch, Smithers.
- Windom, H.L. and D.R. Kendall. 1979. Accumulation and biotransformation of mercury in coastal and marine biota. In: J.O. Nriagu, (Ed.) The Biogeochemistry of Mercury in the Environment. New York: Elsevier/North-Holland Biomedical Press. pp. 303-323.
- Wong, P.T.S., Y.K. Chau, L. Luxon and G.A. Bengert. 1977. Methylation of arsenic in the aquatic environment. Trace Subst. Environment Health X1: 100-106
- Wood, E.D., F.A.J. Armstrong and F.A. Richards. 1967. Determination of nitrate in seawater by cadmium-copper reduction to nitrite. J. Mar. Biol. 47: 23-31.

11.5 Volume III Appendices

11.5.1 Errata

	Page	For	Read
	title page	Prepared for:	Submitted by:
11.5.2	Omissions		
	References		
	[Please see Omissions for Volume III	above]	

11.6 Volume IV

Page

11.6.1 Errata

20

-

1000

For

title page	Prepared for:	Submitted by:
1-4,line 28	43.5 10 ⁶ /t	43.5 [·] 10 ⁶ t
1-6,line 29	(Figure 1.4.2-1)	[delete]
1-6,line 31	MW power	MW. Power
1-6,line 32	minesite.	minesite (Figure 1.4.2-1).
1-12,line 27	corresopnd	correspond
1-12,line 35	coordinated the	·
	production of Volume IV	[delete]
2-5,line 18	followed by a gradual	
	deceleration in rate.	then gradually declines.
2-11,line 23	flushed	unflushed
2-12,line 2	180 ma/L	60 to 180 mg/L
2-12,line 3	0.55 X 10Pk surface	6.55 10 ⁹ m ² of
	dur 0 ⁹ m ² /2100 wk	surface during each time
		period of 2100 weeks
		(180 mg/L 4.4
		10 ⁹ L/6.55 10 ⁹
		m ² /2100 wk
2-19,line 30	repre <the< td=""><td>represents the</td></the<>	represents the
2-44,line 1	is	in
2-51,line 9	879 m ³	879 000 m ³
2-52,line 13	expect	except
2-52,line 25	because of the small	than at the Branch 45
	addition at the pit outflow	road
2-54,line 18	dearly	early
2-57,table	Ci	Cu
2-66,line 13	was reasonable assumed	was assumed
2-66,line 24	flow augmentation	flow of 0.008 m ³ /s.
	-	While the low flow
2-80,line 23	Ingles (1987)	Ingles (1981)
2-91,line 2	Table 2.2.3-6	Table 2.2.3-5
2-92,table	Table 2.2.3-6	[delete, repeat of
		Table 2.2.3-5]
2-123,line 7	et al	Fleming and McMahon
2-133,table	mmhos/cm	umhos/cm
line 9		
2-137,line 9	Sanders	Sanders and Wilford
2-144,line 9	this could will	this will
2-147,line 27	spread any portions of the	spread over any portions
	substrate amendments	of the substrate

Read

of the substrate requiring amendments Page

For

Read

2-77, Table 2.2.3-1

[Delete data and replace with the following]

PA	RAMETER	UNITS	IMPOUNDMENT No. 1	IMPOUNDMENT No. 2 ^b
Su	spended Solids	mg/L	75	75
Nit	rate	mg N/L	25	25
Tot	tal Cyanide	mg/L	0.05	0.05
WA	AD Cyanide	mg/L	0.05	0.05
To	tal Metals			
Ag		mg/L	0.0006	0.0006
Al		mg/L	0.32	0.29
As		mg/L	0.59	0.54
Ва		mg/L	12	11
Cd		mg/L	0.001	0.001
. Co		mg/L	0.59	0.54
Cr		mg/L	0.02	0.02
Cu		mg/L	0.017	0.015
Fe		mg/L	0.54	0.49
Hg		ug/L	0.90	0.83
Мп	1	mg/L	0.92	0.85
Мо)	mg/L	12	11
Ni		mg/L	0.29	0.26
Pb		mg/L	0.04	0.03
Sb		mg/L	0.59	0.54
Se		mg/L	0.006	0.006
Zn		mg/L	0.35	0.32

^aMaximum permissible concentrations based on groundwater seepage with moderate tailings permeability. (Groundwater - 70 m³/d into Impoundment No. 1 and 50 m³/d into Impoundment No. 2).

^bAllowable concentrations for Impoundment No. 2 surface discharge take into account the additive effect of loadings from both Impoundments No. 1 and 2.

^cAluminum concentrations based on dissolved concentration; all other metals total.

Page	For	Read
2-81,line 14	such metals as since they are	metals such as copper are
2-151,line 21	and section	[delete]
2-152,line 27	impoundment surface.	impoundment.
2-155,line 19	305 m	305 mm
2-156,line 3	regarded	regraded
3-18,line 6	SRK using for	SRK for
3-18,line 29	рН	pit

Page	For	Read
3-22,line 35	actual flow increases	actual increases in
3-28,line 2	exceeded during the life of the mine operation is about 25 and 75%, respectively.	exceeded in any single year of the mine operation is about 25 and 75%, respectively, due to the much shorter period of concern (one year versus twelve years).
3-28,line 28	6 300 t/d	6000 t/d
3-28,line 35	0.006 mg/L.C)	0.006 mg/L.
3-36,line 36	Creek at upstream	Creek upstream
3-74,line 21	not obvious	not intuitively obvious
3-74,line 22	common impoundment	common Q = KiA formulation, where Q is the flow, K is the hydraulic conductivity, i is the hydraulic gradient, and A is the cross-sectional area across which flow occurs. In the case where an impoundment
3-74,line 23	arezen	area. When

For

Ű.

۳Ľ

Ň

Read



Page	For	Read
3-76,line 22 3-85,figure	infinitey	infinitely
3-98,lines 6,7	Concentrations of molybdenum, nickel, and antimony may be detectable during the lowest flows, but will be well below their respective criteria.	[delete]
3-103,line 4	sensitive costal environments such as that of in	is within the anticipated guidelines being developed by B.C. Water Management Branch for sensitive coastal environments such as that in
3-103,lines	Streams will	[move to follow first
35 to 38	not be crossed.	paragraph on page 3-105]
3-120,table	Source:SRK	[delete]
3-132,line 10	sediments will occur in	sediments in
3-144,IINE 27		Kelley
3-145,line 32	nowever, the periods of elevated	however, elevated
3-146,line 10	1985	1986
3-146,line 18	1985	1986
3-146,line 29	obstruction.	obstruction. The assessment of potential obstructions is also provided in the following sections.
1-147,line 5,6	During mine construction the personnel associated with the project will increase to a peak of over 300 individuals.	During mine construction most people will not have vehicles, making access to fishing areas difficult. Also, rigorous work schedules during construction will result in many workers leaving the project site when off-duty.
3-156,line 6	be more than compensated	be compensated
3-157,line 4	not maintain actual creek flows above a specified level.	not necessarily maintain creek flows above a specified level under conditions.
3-157,line 5	example in a schedule based on mean flow during June, baseline	example if the augmentation program attempted to maintain

÷.

Ċ.

()

Page

For

Read

flows at the mean June level, baseline While the mean 3-159,line 25 While the phase Coreshack Creek mean Sections 4.2.1 and 4.2.2. Sections 3.4.2.1 and 3-168,line 3 3.4.2.2 3-170,line 1 compensating compensation 3-179,line 3 Pearson (Pearson ppb or almost 3-182,line 27 ppb almost 3-183,line 26 Wand land 3-186,line 15 TABLE 3.12.1-1 [delete] establishment of 3-186,line 19 of establishment 3-203,line 20 setting settling Total life of mine 3-219,line 27 Total land 3-220,line 12 reasoning [delete] R-2, line 19 Denison, P.R. Fleming Denison, P.J., R. Fleming and T.A. McMahon and P.A. McMahon R-6, line 9 Tripp, D.B. 1986 Tripp, D.B. and V. Poulin 1986

11.6.2 Omissions

xv

2.2.3-1	Maximum Permissible Concentrations in Tailings Impoundment Surface Waters for Discharge to Florence Creek after Deactivation, Determined by Norecol, Cinola Gold Project	2-77
2.2.3-2	High West Impoundments No. 1 and No. 2 Predicted Surface Water Quality One Year After Deactivation, Predicted by Norecol, Cinola Gold Project	2-79
2.2.3-3	High West Impoundments No. 1 and No. 2 Predicted Surface Water Quality One Year After Deactivation, Predicted by Norecol, Cinola Gold Project	2-84
2.2.3-4	Predicted Water Quality from the Low Grade Ore Stockpile by Norecol, Cinola Gold Project	2-86
2.2.3-5	Estimated Flows (m3/s) in Florence Creek at the Middle Florence Creek Gauging Station for Various Conditions With and Without Low Flow Augmentation	2-90
2.3.2-1	Fuel and Explosives Monthly Supply Requirements for the Mine Area, Cinola Gold Project	2-94
2.3.3-1	Typical Reagent Consumption Rates and Storage Capacity, Cinola Gold Project	2-97

2.3.4-1	Estimated Shipments on Road Corridors, Cinola Gold Project	2-104
2.4.2-1	Performance Data for One Diesel Engine Proposed for the Power Plant, Cinola Gold Project	2-117
2.4.2-2	Comparison of Diesel Engine Emissions, Cinola Gold Project, with B.C. Objectives	2-118
2.4.3-1	Predicted Mill Emissions, Cinola Gold Project	2-119
2.4.3-2	Comparison of Controlled Mill Emissions with B.C. Objectives	2-122
2.4.4-1	Results of the Wet Deposition Screening Model for Mill Site Emissions, Cinola Gold Project	2-125
2.4.4-2	Sulphate and Nitrate Wet Deposition from Combined Emissions of the Cinola Gold Project Power Plant and Mill Expressed as Wet Sulphate Deposition	2-126
xviii		
2.3.4-1	Main Access Route	2-107
2.3.5-1	Preliminary Dock Site Layout	2-111
R-5	RRCC:see Resource Recovery and Conservation Consultants	
R-6	William Hill Mining. Personal Communication. 1988. Bill Hill, Toronto, Canada.	

11.7 Volume IV Appendices

11.7.1 Errata

Page	For	Read
title page	Prepared for:	Submitted by:
6,line 9	limits <0.006	limits of <0.006
6,line 18	top water	tap water
7,line 10	zone confined	zone was confined
12,line 14	an early flat slope	a nearly flat slope
21,lines 33-35	After the solubility	[Move to follow line
		16, page 21]
21,lines 17-18	chloride(Table 1.4-1)	[Move to follow line
		31, page 23]
23,lines 1-11	For cyanideaddition.	[Move to follow line 16, page 21]

Page	For	Read
23,lines 32-35	A high peroxide.	[Move to follow line 32, page 21]
24,lines 1-3	The week (Table 1.4.1-1)	[Move to follow line 32, page 21]

11.8 Volume V

11.8.1 Errata

Page	For	Read
title page	Prepared for:	Submitted by:
1-4,line 28	43.5 ⁻ 10 ⁶ /t	43.5 [.] 10 ⁶ t
1-6,line 22	may	any
1-6,line 29	(Figure 1.4.2-1)	[delete]
1-6,line 31	MW power	MW. Power
1-6,line 32	minesite.	minesite (Figure 1.4.2-1).
2-4,line 7	H ₂ CO ₃ ⁻	HCO3 ⁻
2-10,line 2	lake tailings.	lake.
2-13,line 4	to 83 ha Benson	to Benson
3-5,line 17	Hill Mining	William Hill Mining
3-55,footnote	3.3.5-5	3.3.4-5
3-56,table	42	4.2
3-91,line	pads and was after low no	pads after
3-104,line 17	buffering affected as pH,	low buffering affected
3-107,line 25	acidity and iron by other geochemical reactions.	by other geochemical reactions as pH, acidity, and iron.
3-158,line 5	an	can
5-4,line 36	stockpile	pads
R-1,line 21	Caruccio, F.T.,et.al.	Caruccio, F.T, J.C Ferm, J. Horne, G. Geidel, and B. Baganz.
R-1,line 36	A Felmy MINTEQ computer	A computer

١÷

١Ň

÷.

100

÷.

4

11.8.2 Omissions

4-2,table

EXPERIMENT	TEST MATERIAL	SECTION IN WHICH EXPERIMENT
DESCRIBED		
Short-term Leach Experiments	Haida mudstones; Skonun sediments; argillically altered sediments; rhyolite; multiphase breccia; overburden	4.0
		······································

11.9.1 Errata

Page For Read

title page

Prepared by:

Submitted by:

11.9.2 Omissions

References

William Hill Mining. Personal Communication. 1988. Bill Hill, Toronto, Canada.

11.10 Volume VI

11.10.1 Errata

Page	For	Read		
title page	Prepared for	Submitted by		
1-4,line 28	43.5 [.] 10 ⁶ /t	43.5 [.] 10 ⁶ t		
1-5,line 1	Clements Road, Queen	Clements Road, and		
	Charlotte Main and			
1-6,line 22	may	any		
1-6,line 31	MW power	MW. Power		
3-17,line 1	provide	province		
5-6, Table 5.2.4-1	[delete applicable data and	[delete applicable data and replace with following:]		

		\$ MILLION (CONSTANT 1988 DOLLARS)				
YEAR	DIRECT	INDIRECT	INDUCED	TOTAL		
2001	5.9	14.36	5.07	25.33		
2002	2.7	0.49	0.80	3.99		
2003	2.6	0.48	0.77	3.85		
TOTAL	76.1	113.96	47.54	237.60		

[Note: the row representing the year 2002 was deleted]

[delete applicable data and replace with following:]

	\$ MILLION (CONSTANT 1988 DOLLARS)				
YEAR	TOTAL INCOME ^a	INCOME LOST ^b	NET INCOME ^c		
2001	25.33	1.03	24.30		
2002	3.99	1.03	2.96		
2003	3.85	1.03	2.82		
TOTAL	237.60	14.42	223.18		

[Note: the row representing the year 2002 was deleted]

^{5-7,} Table 5.2.4-2

Page

For

Read

6-8,tableSandspit 0.3Sandspit 0.16-11, table[delete applicable data and replace with following:]

For

Read

COMMUNITY	COMMUNITY		
Port Clements	Port Cleme	ents	
1989	1989		
1990	1990		
1991-2002	1991-20	001	
2003	2002		
2004	2003		
Masset/Haida	Masset/Ha	ida	
1989	1989		
1990	1990		
1991-2002	1991-20)01	
2003	2002		
2004	2003		
Queen Charlotte City/Skidegate	Queen Cha	arlotte City/Skidegate	
1989	1989		
1990	1990		
1991-2002	1991-20	001	
2003	2002		
2004	2003		
Sandspit	Sandspit		
1989	1989		
1990	1990		
1991-2002	1991-20	001	
2003	2002		
2004	2003		
Page	For	Read	
7 40 10 4 00			
7-10,IINE 30	2003 when mine operations	2003 as employees	
	cease.	begin to relocate	
		in response to mine	
7 10 line 17	2005 22		
/-13,1110 1/	2005 as	2005 in response to the	
7 12 line 19	activition and		
7-13,000 10	activities end.	activities.	

Page	For	Read
7-13,line 32	2005 when reclamation activities at the minesite are completed.	2005 in response to the end of mine reclamation activities.
7-16,line 10	As in the case of Haida, three	Three
10-2,Table 10.1.1-1	[Column 4] :49	[Column 4] 8-10:49
11-9,line 7 R-1,line 5	clarify on 3Vancouver	clarify or Vancouver

11.11 Volume VI Appendices

11.11.1 Errata

Page	For	Read
title page	Prepared for:	Submitted by:
8,line 6 9 Table 4	879 Idelete applicable data and repl	893 ace with following:
	Lacioto applicable data and repr	ace manieming.j

1976	No.	%	1976	No.	%
155	-57	-63.3	155	-43	-47.8
564	+102	+21.7	565	+116	+24.7

Page	For	Read
22,line 13	190%	290%
22,line 13	18.9%	28.9%

REFERENCES

REFERENCES

- American Public Health Association. 1965. Standard methods for the examination of water and wastewater. 16th Edition. Washington, D.C.: American Public Health Association, American Water Works Association, and Water Pollution Control Federation. 1268 pp.
- B.C. Ministry of Environment. 1979. Pollution control objectives for the mining, smelting, and related industries of British Columbia. Victoria, B.C.: Pollution Control Board. 16 pp.
- Blight G.E. and O. K. H. Steffen. 1979. Geotechnics of Gold Mining Waste Disposal. Current Geotechnical Practice in Mine Waste Disposal, ASCE.
- Brown, A., D.J. Kushner and S.P. Mathur. 1987. Feasibility of assessing metal pollution in Canadian Shield mining areas through analysis of peat soils. pp. 299-305 in: Proc. Symposium '87 Wetlands/Peatlands compiled by C.D.A. Ruber and R.P. Overend. Edmonton, Alberta, Aug. 23-27, 1987.
- Caldwell, J.A., D. Moore, and A.C.S. Smith. 1984. Impoundment design for cyanided tailings case history of the Cannon Mine project: Conference on Cyanide and the Environment, Tucson, AZ: 249-263.
- Canadian Council of Resource and Environment Ministers (CCREM). 1987. Canadian water quality guidelines. Ottawa, Ontario: Water Quality Branch, Inland Waters Directorate.
- Cronan, C.S., W.J. Walker and P.R. Bloom. 1986. Predicting aqueous aluminum concentrations in natural waters. Nature 324:140-143.
- Cronin, J. 1988. Personal Communication. Steffen, Robertson and Kirsten, San Fransisco CA.
- Englehardt, P.R. 1984. Long-term degradation of cyanide in an inactive leach heap. Proceedings: Conference of Cyanide and the Environment, Tucson, AZ.
- Felmy, A.R., D.C. Girvin and E.A. Jenne. 1984. MINTEQ A computer program for calculating aqueous geochemical equilibria. EPA 800/3-84/032. Environmental Protection Agency Battelle North West Laboratories, Richland, Washington.
- Ferguson, K.D. 1986. Preliminary evaluation of wetland treatment at Mt. Washington. Letter report to L. Erickson, Waste Management Branch, Vancouver Island Region, Ministry of Environment, Nanaimo, B.C.
- Inco Limited. 1987. Unpublished data. Inco SO₂/Air Cyanide Removal Process for City Resources.
- McLeay, D. J. 1988. Personal Communication. D. McLeay and Associates Ltd. West Vancouver, B.C.
- Morin, K.A., J.A. Cherry, N.K. Davé, T.P. Lim and A.J. Vivyurka. 1988. Migration of acidic groundwater seepage from uranium tailings impoundments. No. 1 Field Study and Hydrogeochemical Conceptual Model. J. of Contam. Hydrol. Fall 1988 Issue.
- Smith, Adrien. 1988. Testimony to Department of Health and Environmental Control, South Carolina Permit #SC 0041378. Appeal Hearing, Columbia, SC., December 1987 - Ridgeway Mining Company, South Carolina.
- Sobek, A.A., W.A. Schuller, J.R. Freeman and R.M. Smith. 1978. Field and laboratory methods applicable to overburden and minesoils. Report EPA-600/Z-78-054 Cincinnati, Ohio: U.S. Environmental Protection Agency.
- Steffen Robertson and Kirsten. 1983. Program to control cyanide seepage. Appendix N: Physical properties of tailings. Report No. 04201.
- Vos, R. 1988. Personal Communication. B.C. Research.

Worsley Alumino Pty Ltd. 1986. Report on studies into cyanide decay process residues.

APPENDICES

DESCRIPTION OF METALLURGICAL SAMPLES DURING PILOT PLANT PROGRAM, MAY 1988, CINOLA GOLD PROJECT

SAMPLE No.	SAMPLING POINT	DATE/TIME	COMPOSITE	COMMENTS
H3-OR-1	ORE A	880516	-	-
H3-OR-2	ORE B	880516	-	-
H3-FD-1	Feed Tank	880504	1500, 1700	-
H3-FD-2	Feed Tank	880505	1100, 1300	-
H3-FD-3	Feed Tank	880506	0600, 0900	-
H3-AS-1	Tank 9	880504	1400, 1600, 1800, 2000	After nitrate oxidation
H3-AS-2	Tank 9	880505	0800, 1000, 1400, 1600	After nitrate oxidation
H3-AS-3	Tank 9	880506	0800, 1000, 1100, 1300	After nitrate oxidation
H3-AS-4	Tank 9	880506/1800	-	After nitrate oxidation
H3-NL-1	Tank 17	880504	1700, 1800, 1900, 2000	After neutralization
H3-NL-2	Tank 17	880505	0800, 1000, 1400, 1600	After neutralization
H3-NL-3	Tank 17	880506	0900, 1000, 1100, 1200	After neutralization
H3-BC-1	Feed to CIL	880510/1400	-	Feed source from Tank 17 taken during pilot
H3-BC-2	Feed to CIL	880511/1200	-	plant run
H3-BC-3	Feed to CIL	880514/1200	-	880505
H3-CN-C1	After CIL	880509	1200, 2400	-
H3-CN-C2	After CIL	880510	1200, 1800, 2400	
H3-CN-C3	After CIL	880511	0600, 1200	
H3-CN-8	After CIL	880513	880509 - 880511	Composite of entire CIL run to SO ₂ /Air CD ^a
H3-CN-9	After CIL	880513	880509 - 880511	Composite of entire CIL run to H ₂ O ₂ CD
H3-CD-I-1	After SO ₂ /Air CD	880514/1430	-	Grab sample during run
H3-CD-I-2	After SO ₂ /Air CD	880514/1900	-	Grab sample during run
H3-CD-I-3	After SO ₂ /Air CD	880516/1200	-	Composite of entire run
H3-CD-I-4	After SO ₂ /Air CD	880516/2100	-	Composite of entire CD run before Na ₂ S addition
H3-CD-D-1	After H ₂ 0 ₂ CD	880513/1730	-	Grab sample during run
H3-CD-D-2	After H ₂ 0 ₂ CD	880513/2230	-	Grab sample during run
H3-CD-D-3	After H ₂ 0 ₂ CD	880515/0830		Composite of entire CD run
H3-CD-D-4	After H ₂ 0 ₂ CD	880516/2100	-	Composite of entire CD run before Na ₂ S addition
H3-S2-I-1	After Na ₂ S	880516/2030	-	Treated sample from SO ₂ /Air CD
H3-S2-D-1	After Na ₂ S	880516/1930	-	Treated sample from H ₂ O ₂ CD

APPENDIX 3.3-1

Description of Metallurgical Samples During Pilot Plant Program, May 1988, Cinola Gold Project

^a CD cyanide destruction CIL carbon-in-leach

MAY 1988 PILOT PLANT PROGRAM FOR THE CINOLA GOLD PROJECT

(Pages 1 to 16)

APPENDIX 3.4-1 CONTENTS

Table

1

2

Water Quality of Liquid Phase of Slurry after Nitrate Oxidation during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project Water Quality of Liquid Phase of Slurry after Neutralization during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project Water Quality of Liquid Phase of Slurry after Neutralization during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

3	Water Quality of Liquid Phase of Slurry before Cyanidation during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	2
4	Water Quality of Liquid Phase of Slurry after Cyanidation during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	4
5	Water Quality of Liquid Phase of Slurry after Cyanide Destruction Using SO ₂ /Air during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	5
6	Water Quality of Liquid Phase of Slurry after Cyanide Destruction Using Hydrogen Peroxide during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	6
7	Water Quality of Liquid Phase of Slurry after Sodium Sulphide Addition during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	7
8	Acid-base Accounting and Metal Content of Ore Used During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	8
9	Acid-base Accounting and Metal Content of Feed Slurry During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	9
10	Metal Content of Solid Component of Slurry after Nitrate Oxidation During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	10
11	Metal Content of Solid Component of Slurry after Neutralization During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	11
12	Metal Content of Solid Component of Slurry Before Cyanidation During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	12
13	Metal Content of Solid Component of Slurry After Cyanidation During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	13
14	Acid-base Accounting and Metal Content of Solid Component of Slurry After Cyanide Destruction Using SO ₂ /Air During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	14
15	Acid-base Accounting and Metal Content of Solid Component of Slurry After Cyanide Destruction Using Hydrogen Peroxide During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project	15
16	Acid-base Accounting and Metal Content of Solid Component of Slurry After Sodium Sulphide Addition During the May 1988 Pilot Plant Program by Norecol for the Cinola	10
	Gold Project	16

1

2

TABLE 1

PARAMETER	UNITS	H3-AS-1 ^a	H3-AS-2	H3-AS-3	H3-AS-4
рН		1.2	1.2	1.2	1.5
Eh	mV	650	650	660	655
Conductivity	umhos/cm	50000	49200	52000	37800
Sulphate	mg/L	43750	44375	45000	44375
Chloride	mg/L	22.5	22.5	22.5	23.8
Total Phosphorus	mg P/L	117	119	122	153
Total Mercury	ug/L	43	55	38	125
Dissolved Metals					
Aluminum	mg/L	1620	1640	1660	1940
Antimony	mg/L	0.35	0.20	0.07	0.07
Arsenic	mg/L	81	82	83	71
Barium	mg/L	0.24	0.23	0.33	0.41
Cadmium	mg/L	0.08	0.12	0.07	0.08
Chromium	mg/L	16.5	16.5	16.0	17.5
Cobalt	mg/L	5.4	5.6	5.0	5.8
Copper	mg/L	28.2	29.0	28.8	29.2
Iron	mg/L	8380	8420	8320	9200
Lead ^b	mg/L	<0.02	<0.02	<0.02	<0.02
Manganese	mg/L	112	116	112	122
Molybdenum	mg/L	0.65	0.65	0.65	0.47
Nickel	mg/L	17.4	17.4	16.6	17.8
Selenium	mg/L	0.48	0.48	0.49	0.17
Silver	mg/L	0.008	0.008	0.010	0.001
Zinc	mg/L	19.0	19.4	19.4	20.6

Water Quality of Liquid Phase of Slurry after Nitrate Oxidation during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

^a Description of samples is given in Appendix 3.3-1.

^b Lower detection limits were not obtained due to interferences.

Note: H3-AS-1, H3-AS-2, and H3-AS-3 were averaged for summary Table 3.4-1.

TABLE 2

PARAMETER	UNITS	H3-NL-1 ^a	H3-NL-2	H3-NL-3
 рН		7.0	7.5	7.4
Eh	mV	490	500	510
Conductivity	umhos/cm	3820	3750	3400
Sulphate	mg/L	1469	1531	1344
Chloride	mg/L	15.0	13.3	12.5
Total Phosphorus	mg P/L	0.067	0.103	0.257
Total Mercury	ug/L	0.22	0.15	0.08
Dissolved Metals				
Aluminum	mg/L	0.80	0.60	0.80
Antimony	mg/L	0.01	<0.005	0.13
Arsenic	mg/L	0.029	0.016	0.055
Barium	mg/L	<0.05	<0.05	<0.05
Cadmium	mg/L	<0.001	0.001	<0.001
Chromium	mg/L	0.001	<0.001	0.011
Cobalt	mg/L	<0.05	<0.05	<0.05
Copper	mg/L	0.05	0.10	0.010
Iron	mg/L	<0.05	<0.05	<0.05
Lead ^b	mg/L	<0.02	<0.02	<0.02
Manganese	mg/L	0.06	0.35	0.03
Molybdenum	mg/L	0.22	0.10	0.22
Nickel	mg/L	0.08	0.07	0.03
Selenium	mg/L	0.022	0.014	0.014
Silver	mg/L	<0.001	<0.001	<0.001
Zinc	mg/L	<0.02	<0.02	<0.02

Water Quality of Liquid Phase of Slurry after Neutralization during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

^a Description of samples is given in Appendix 2.3-1.

^b Lower detection limits were not obtained due to interferences.

TABLE 3

PARAMETER	UNITS	H3-BC-1 ^a	H3-BC-2	H3-BC-3
 рН		4.2	6.5	7.6
Eh	mV	560	530	510
Conductivity	umhos/cm	3950	3850	3900
Sulphate	mg/L	1328	1313	1328
Chloride	mg/L	16.3	21.0	17.2
Total Phosphorus	mg P/L	0.193	0.240	0.250
Total Mercury	ug/L	0.10	0.21	0.14
Dissolved Metals				
Aluminum	mg/L	0.65	1.0	0.80
Antimony	mg/L	0.069	0.11	0.004
Arsenic	mg/L	0.032	0.031	0.046
Barium	mg/L	<0.05	<0.05	<0.05
Cadmium	mg/L	<0.001	<0.001	<0.001
Chromium	mg/L	0.004	0.001	<0.001
Cobalt	mg/L	<0.05	<0.05	<0.05
Copper	mg/L	0.04	0.04	0.03
Iron	mg/L	0.07	<0.05	<0.05
Lead ^D	mg/L	<0.02	<0.02	<0.02
Manganese	mg/L	0.02	0.04	0.03
Molybdenum	mg/L	0.24	0.25	0.30
Nickel	mg/L	0.12	0.020	0.035
Selenium	mg/L	0.012	0.008	0.005
Silver	mg/L	<0.001	<0.001	<0.001
Zinc	mg/L	<0.02	0.09	<0.02

Water Quality of Liquid Phase of Slurry before Cyanidation during the May 1988 Pilot Plant Program by Noreco! for the Cinola Gold Project

^a Description of samples is given in Appendix 3.3-1.

^b Lower detection limits were not obtained due to interferences.

TABLE 4

PARAMETER	UNITS	H3-CN-C1 ^a	H3-CN-C2	H3-CN-C3	H3-CN-8	H3-CN-9
рН		9.8	10.1	9.9	9.8	9.6
Eh	mV	130	210	160	110	100
Conductivity	umhos/cm	7500	7200	7420	7200	7200
Sulphate	mg/L	1719	1625	1719	1688	1625
Nitrate	mg N/L	280	285	285	280	285
Total Phosphorus	mg P/L	0.40	0.44	0.32	0.43	0.27
Total Mercury	ug/L	750	583	883	650	640
Total Cyanide	mg/L	180	232	248	154	149
Thiocyanate	mg/L	1290	1150	1130	1340	1360
Cyanate	mg/L	1.32	1.73	1.89	1.38	1.79
Dissolved Metals						
Aluminum	mg/L	1.3	1.3	1.6	0.70	0.75
Antimony	mg/L	0.44	0.34	0.42	0.36	0.36
Arsenic	mg/L	0.14	0.11	0.10	0.13	0.16
Barium	mg/L	<0.05	<0.05	<0.05	<0.05	<0.05
Cadmium	mg/L	0.020	0.010	0.015	0.025	0.033
Chromium	mg/L	0.035	0.026	0.020	0.024	0.021
Cobalt	mg/L	0.42	0.43	0.42	0.42	0.44
Copper	mg/L	13.7	14.2	13.8	13.7	13.9
Iron	mg/L	0.08	0.07	0.09	0.08	0.08
Lead ^b	mg/L	0.02	<0.02	0.03	<0.02	<0.02
Manganese	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02
Molybdenum	mg/L	0.30	0.30	0.29	0.29	0.31
Nickel	mg/L	1.32	1.42	1.49	1.47	1.47
Selenium	mg/L	0.008	0.004	0.007	0.007	0.007
Silver	mg/L	0.013	0.011	0.016	0.019	0.021
Zinc	mg/L	2.60	2.46	2.42	2.65	2.59

Water Quality of Liquid Phase of Slurry After Cyanidation during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

ŵ

^a Description of samples is given in Appendix 3.3-1.

^b Lower detection limits were not obtained due to interferences.

Note: H3-CN-8, H3-CN-9 were averaged for after cyanidation run immediately before cyanide destruction.

TABLE 5

Water Quality of Liquid Phase of Slurry after Cyanide Destruction using SO_2 /air during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

					10 00-1-4
pH	/ / / / / /	8.1	7.3	7.4	8.0
Eh	mV	250	250	215	190
Conductivity	umhos/cm	8510	7940	8170	6900
Sulphate	mg/L	2344	2313	2375	2250
Chloride ^b	mg/L	-	-	-	-
Alkalinity	mg CaCO ₃ /L	125	95	155	145
Nitrate	mg N/L	305	303	259	288
Nitrite	mg N/L	2.72	2.25	2.92	2.80
Ammonia	mg N/L	0.120	0.245	0.165	0.215
Total Phosphorus	mg P/L	0.35	0.39	0.41	0.38
Total Mercury	ug/L	1.29	1.51	4.95	4.38
Total Cyanide	mg/L	0.61	1.22	0.58	0.60
WAD Cyanide	mg/L	0.23	0.19	0.12	0.14
Thiocyanate	mg/L	1380	1360	1575	1475
Cyanate	mg/L	3.17	2.75	1.95	2.13
Dissolved Metals	-				
Aluminum	mg/L	1.8	1.4	1.1	1.1
Antimony	mg/L	0.73	0.75	0.69	0.53
Arsenic	mg/L	0.20	0.23	0.20	0.18
Barium	mg/L	<0.05	<0.05	< 0.05	<0.05
Cadmium	mg/L	<0.001	0.001	<0.001	<0.001
Calcium	mg/L	830	850	860	790
Chromium	mg/L	0.022	0.019	0.11	0.010
Cobalt	mg/L	0.56	0.52	0.54	0.50
Copper	mg/L	0.91	0.76	0.45	0.29
Iron	mg/L	0.24	0.24	0.10	0.13
Lead ^c	mg/L	<0.02	<0.02	<0.02	0.03
Magnesium	mg/L	21	17	18	16
Manganese	mg/L	<0.02	0.02	<0.02	<0.02
Molybdenum	mg/L	0.28	0.30	0.27	0.19
Nickel	mg/L	0.02	0.11	<0.01	0.09
Potassium	mg/L	35	34	46	47
Selenium	mg/L	0.09	0.13	0.003	<0.001
Silver	ma/L	0.0040	0.0025	<0.001	<0.001
Sodium	ma/L	1600	1500	1600	1500
Zinc	mg/L	<0.02	0.03	<0.02	0.02

^a Description of samples is given in Appendix 3.3-1.

^b Chloride analyses were not obtained due to thiocyanate interferences.

^c Lower detection limits were not obtained due to interferences.

Note: H3-CD-I-3 and H3-CD-I-4 were averaged for after the cyanide destruction run.

TABLE 6

-

1ú

Water Quality of Liquid Phase of Slurry after (Cyanide Destruction using Hydrogen Peroxide
during the May 1988 Pilot Plant Progra	m by Norecol for the Cinola Gold Project

PARAMETER	UNITS	H3-CD-D-1 ^a	H3-CD-D-2	H3-CD-D-3	H3-CD-D-4
pН	-	-	7.6	7.9	8.0
Eh	mV	-	230	210	200
Conductivity	umhos/cm	-	-	7410	8280
Sulphate	mg/L	-	1719	1719	1688
Chloride	mg/L	-	-	-	-
Alkalinity	mg CaCO ₃ /L	-	190	155	200
Nitrate	mg N/L	3	298	315	273
Nitrite	mg N/L	-	3.04	1.86	1.94
Ammonia	mg N/L	-	0.480	0.135	0.230
Total Phosphorus	mg P/L	•	0.31	0.32	0.43
Total Mercury	ug/L	0.39	0.30	0.43	0.34
Total Cyanide	mg/L	0.64	0.56	1.08	1.87
WAD Cyanide	mg/L	0.45	0.34	0.066	1.25
Thiocyanate	mg/L	-	1338	1363	1331
Cyanate	mg/L	-	0.80	2.57	2.70
Dissolved Metals					
Aluminum	mg/L	0.25	1.7	0.8	0.9
Antimony	mg/L	0.47	0.40	0.55	0.57
Arsenic	mg/L	0.14	0.14	0.15	0.14
Barium	mg/L	0.09	<0.05	0.23	<0.05
Cadmium	mg/L	<0.001	0.001	0.001	<0.001
Calcium	mg/L	860	940	980	940
Chromium	mg/L	0.17	0.15	0.15	0.18
Cobalt	mg/L	0.53	0.51	0.53	0.51
Copper	mg/L	1.01	0.66	1.09	0.88
Iron	mg/L	0.14	0.14	0.14	0.14
Lead ^c	mg/L	<0.02	<0.02	<0.02	0.02
Magnesium	mg/L	2.45	1.40	1.50	1.45
Manganese	mg/L	<0.02	<0.02	<0.02	<0.02
Molybdenum	mg/L	0.36	0.35	0.36	0.35
Nickel	mg/L	0.33	0.27	0.38	0.32
Potassium	mg/L	12	49	30	47
Selenium	mg/L	0.050	0.060	0.061	0.058
Silver	mg/L	0.0020	0.0025	0.0025	0.0015
Sodium	mg/L	1060	1100	900	1100
Zinc	mg/L	<0.02	<0.02	<0.02	<0.02

^a Description of samples is given in Appendix 3.3-1.

^b Chloride analyses were not obtained due to thiocyanate interferences.
 ^c Lower detection limits were not obtained due to interferences.
 Note: H3-CD-D-3 and H3-CD-D-4 were averaged for after the after cyanide destruction run.

TABLE 7

PARAMETER	UNITS	H3-S2-I-1 ^a	H3-S2-D-1
pH	<u></u>	8.0	8.0
Eh	mV	170	190
Conductivity	umhos/cm	10010	8710
Sulphate	mg/L	3250	2438
Chlorideb	mg/L	-	-
Alkalinity	mg CaCO ₃ /L	170	175
Nitrate	mg N/L	295	265
Nitrite	mg N/L	2.80	3.29
Ammonia	mg N/L	0.115	0.205
Total Phosphorus	mg P/L	0.50	0.85
Total Mercury	ug/L	0.24	0.34
Total Cyanide	mg/L	0.54	0.78
WAD Cyanide	mg/L	0.33	0.46
Thiocyanate	mg/L	1463	1300
Cyanate	mg/L	3.26	4.67
Dissolved Metals			
Aluminum	mg/L	1.4	1.9
Antimony	mg/L	0.10	0.16
Arsenic	mg/L	0.13	0.055
Barium	mg/L	<0.05	<0.05
Cadmium	mg/L	<0.001	<0.001
Calcium	mg/L	650	650
Chromium	mg/L	0.017	0.19
Cobalt	mg/L	0.52	0.50
Copper	mg/L	0.10	0.75
Iron	mg/L	0.07	0.13
Lead ^c	mg/L	<0.02	<0.02
Magnesium	mg/L	0.90	0.15
Manganese	mg/L	<0.02	<0.02
Molybdenum	mg/L	0.13	0.19
Nickel	mg/L	0.08	0.31
Potassium	mg/L	54	35
Selenium	mg/L	0.006	0.035
Silver	mg/L	0.0020	0.0035
Sodium	mg/L	2200	1800
Zinc	mg/L	0.02	0.05

Water Quality of Liquid Phase of Slurry after Sodium Sulphide Addition during the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

^a Description of samples is given in Appendix 3.3-1.

^b Chloride analyses were not obtained due to thiocyanate interferences.

^c Lower detection limits were not obtained due to interferences.
TABLE 8

Acid-base Accounting and Metal Content of Ore Used During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

PARAMETER	UNITS	H3-OR-1	H3-OR-2
Acid-Base Accounting			
Paste pH	-	3.9	4.4
Total Sulphur	%S	1.5	1.5
Sulphide	%S	1.1	1.2
Sulphate	%SO4	0.63	0.33
Maximum Potential Acidity	t CaCO3/1000 t	33	38
Neutralization Potential	t CaCO3/1000 t	-2.0	1.0
Net Neutralization Potential	t CaCO ₃ /1000 t	-35	-37
Metal Analysis			
Aluminum	%	0.35	0.35
Antimony	ppm	26.0	23.0
Arsenic	ppm	200	190
Barium	ppm	30	40
Beryllium	ppm	<0.5	0.5
Bismuth	ppm	0.1	0.1
Calcium	%	0.09	0.09
Cadmium	ppm	0.1	0.1
Cobalt	ppm	5	4
Chromium	ppm	139	164
Copper	ppm	15	14
Iron	%	1.76	1.71
Lanthanum	ppm	<10	<10
Lead	ppm	1	1
Magnesium	%	0.12	0.12
Manganese	ppm	100	76
Mercury	ppb	3500	3600
Molybdenum	ppm	1	1
Nickel	ppm	6	8
Phosphorus	ppm	200	170
Potassium	%	0.15	0.19
Selenium	ppm	3.0	3.0
Silver	ppm	3.4	2.6
Sodium	%	0.01	0.01
Strontium	ppm	8	10
Zinc	ppm	33	32

ينتقا

1

.

-

TABLE 9

Acid-base Accounting and Metal Content of Solid Component of Feed Slurry During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

Ш.

PARAMETER	UNITS	H3-FD-1	H3-FD-2	H3-FD-3
Acid-Base Accou	unting			
Paste pH		3.9	3.9	3.9
Total Sulphur	%S	1.3	1.4	1.4
Sulphide	%S	1.1	1.1	1.1
Sulphate	%SO4	0.24	0.26	0.22
Maximum Potentia	al			
Acidity	t CaCO ₃ /1000 t	35	33	34
Neutralization				
Potential	t CaCO ₃ /1000 t	-1.0	-1.0	-1.0
Net Neutralization				
Potential	t CaCO ₃ /1000 t	-36	-34	-35
Metal Analysis				
Aluminum	%	0.33	0.33	0.32
Antimony	mqq	25.0	25.0	24.0
Arsenic	maa	190	200	190
Barium	mag	40	40	40
Beryllium	ppm	0.5	0.5	0.5
Bismuth	ppm	0.1	0.1	0.1
Calcium	%	0.05	0.05	0.05
Cadmium	ppm	0.1	0.1	0.1
Cobalt	ppm	5	4	4
Chromium	ppm	60	60	55
Copper	ppm	35	34	34
Iron	%	2.44	2.52	2.41
Lanthanum	ppm	< 10	< 10	< 10
Lead	mag	1	1	1
Magnesium	%	0.10	0.11	0.10
Manganese	ppm	70	72	68
Mercury	dqq	4500	4500	4400
Molvbdenum	maa	4	5	5
Nickel	ppm	17	15	16
Phosphorus	maa	200	200	180
Potassium	%	0.14	0.14	0.14
Selenium	ppm	2.6	2.2	2.4
Silver	ppm	3.8	2.7	2.8
Sodium	%	0.01	0.01	0.01
Strontium	ppm	8	8	7
Zinc	mqq	31	31	32

دنتان

TABLE 10

PARAMETER	UNITS	H3-AS-1	H3-AS-2	H3-AS-3	H3-AS-4
Aluminum	%	0.14	0.14	0.13	0.12
Antimony	ppm	25.0	25.0	25.0	24.0
Arsenic	ppm	50	140	140	150
Barium	ppm	40	40	40	40
Beryllium	ppm	< 0.5	< 0.5	< 0.5	< 0.5
Bismuth	ppm	0.1	0.1	0.1	0.1
Calcium	%	0.02	0.02	0.01	0.01
Cadmium	ppm	0.1	0.1	0.1	0.1
Cobalt	ppm	1	1	1	1
Chromium	ppm	30	33	31	42
Copper	ppm	9	6	6	5
Iron	%	1.78	1.81	1.66	1.74
Lanthanum	ppm	< 10	< 10	<10	<10
Lead	ppm	4	4	3	2
Magnesium	%	0.05	0.05	0.04	0.03
Manganese	ppm	30	34	28	24
Mercury	ppb	4200	4900	4900	4700
Molybdenum	ppm	5	5	5	5
Nickel	ppm	4	5	5	1
Phosphorus	ppm	130	130	110	90
Potassium	%	0.18	0.18	0.17	0.19
Selenium	ppm	3.4	3.2	3.4	3.2
Silver	ppm	3.2	3.2	3.2	3.0
Sodium	%	0.01	0.01	0.01	0.01
Strontium	ppm	6	6	6	7
Zinc	ppm	16	15	15	13

Metal Content of Solid Component of Slurry after Nitrate Oxidation During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

Note: H3-AS-1, H3-AS-2, and H3-AS-3 were averaged for summary table for after nitrate oxidation.

TABLE 11

PARAMETER	UNITS	H3-NL-1	H3-NL-2	H3-NL-3
Aluminum	%	0.28	0.28	0.27
Antimony	ppm	26.0	25.0	27.0
Arsenic	ppm	200	190	180
Barium	ppm	30	30	30
Beryllium	ppm	< 0.5	<0.5	<0.5
Bismuth	ppm	0.1	0.1	0.1
Calcium	%	2.12	2.20	2.43
Cadmium	ppm	0.1	0.1	0.1
Cobalt	ppm	6	6	5
Chromium	ppm	49	51	49
Copper	ppm	31	26	31
Iron	%	2.22	2.18	2.02
Lanthanum	ppm	10	10	10
Lead	ppm	1	1	1
Magnesium	%	0.11	0.10	0.10
Manganese	ppm	130	128	120
Mercury	ppb	3800	4200	4500
Molybdenum	ppm	5	5	6
Nickel	ppm	21	18	19
Phosphorus	ppm	200	200	190
Potassium	%	0.17	0.16	0.17
Selenium	ppm	2.6	3.2	3.4
Silver	ppm	2.7	2.6	2.8
Sodium	%	0.01	0.01	0.01
Strontium	ppm	49	50	55
Zinc	ppm	32	28	31

Metal Content of Solid Component of Slurry After Neutralization During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

TABLE 12

PARAMETER	UNITS	H3-BC-1	H3-BC-2	H3-BC-3
Aluminum	%	0.29	0.30	0.29
Antimony	ppm	26	29	27
Arsenic	ppm	190	200	190
Barium	ppm	30	30	30
Beryllium	ppm	< 0.5	< 0.5	<0.5
Bismuth	ppm	0.1	0.1	0.1
Calcium	%	2.26	2.35	2.34
Cadmium	ppm	0.1	0.1	0.1
Cobalt	ppm	5	6	6
Chromium	ppm	79	63	60
Copper	ppm	29	16	28
Iron	%	2.12	2.22	2.19
Lanthanum	ppm	10	10	10
Lead	ppm	1	1	1
Magnesium	%	0.10	0.11	0.10
Manganese	ppm	125	131	129
Mercury	ppb	4700	3300	4600
Molybdenum	ppm	4	5	5
Nickel	ppm	20	20	20
Phosphorus	ppm	190	210	200
Potassium	%	0.18	0.17	0.17
Selenium	ppm	2.6	2.8	2.4
Silver	ppm	2.4	1.2	2.4
Sodium	%	0.01	0.04	0.01
Strontium	ppm	53	52	54
Zinc	ppm	30	28	30

Metal Content of Solid Component of Slurry Before Cyanidation During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

TABLE 13

PARAMETER	UNITS	H3-CN-C1	H3-CN-C2	H3-CN-C3	H3-CN-C8	H3-CN-C9
Aluminum	%	0.28	0.30	0.27	0.37	0.34
Antimony	ppm	27.0	28.0	27.0	24.0	26.0
Arsenic	ppm	190	200	190	190	200
Barium	ppm	30	40	30	40	40
Beryllium	ppm	< 0.5	< 0.5	< 0.5	< 0.5	< 0.5
Bismuth	ppm	0.1	0.1	0.1	0.1	0.1
Calcium	%	2.38	2.52	2.31	2.75	2.52
Cadmium	ppm	0.1	0.1	0.1	0.1	0.1
Cobalt	ppm	5	5	4	2	4
Chromium	ppm	51	60	58	64	61
Copper	ppm	15	22	18	15	18
Iron	%	2.14	2.28	2.11	2.60	2.41
Lanthanum	ppm	10	10	10	10	10
Lead	ppm	3	2	2	9	6
Magnesium	%	0.11	0.11	0.10	0.14	0.13
Manganese	ppm	128	135	126	156	141
Mercury	ppb	3000	3700	3100	2400	2600
Molybdenum	ppm	4	5	5	6	6
Nickel	ppm	18	18	17	18	18
Phosphorus	ppm	200	210	190	210	190
Potassium	%	0.18	0.18	0.18	0.17	0.16
Selenium	ppm	3.0	3.2	3.2	3.2	3.4
Silver	ppm	1.2	1.7	1.3	1.4	1.3
Sodium	%	0.03	0.03	0.04	0.04	0.05
Strontium	ppm	53	57	51	60	54
Zinc	ppm	28	31	29	35	33

Metal Content of Solid Component of Slurry After Cyanidation During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

Note: H3-CN-8 and H3-CN-9 were averaged for summary table for after cyanidation.

TABLE 14

Acid-base Accounting and Metal Content of Solid Component of Slurry After Cyanide Destruction Using SO₂/Air During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

PARAMETER	UNITS	H3-CD-I-3	H3-CD-I-4
Acid-Base Account	ting	<u></u>	
Paste pH		8.7	8.7
Total Sulphur	%S	1.9	1.9
Sulphide	%S	0.17	0.13
Sulphate	%SO4	4.87	5.02
Maximum Potential			
Acidity	t CaCO ₃ /1000 t	5.3	4.1
Neutralization	-		
Potential	t CaCO3/1000 t	18	19
Net Neutralization		-	
Potential	t CaCO3/1000 t	13	15
Metal Analysis			
Aluminum	%	0.33	0.34
Antimony	maa	27.0	26.0
Arsenic	mqq	200	190
Barium	mqq	40	40
Beryllium	ppm	< 0.5	< 0.5
Bismuth	ppm	0.1	0.1
Calcium	%	2.33	2.42
Cadmium	mqq	0.1	0.1
Cobalt	ppm	4	4
Chromium	ppm	81	74
Copper	mqq	43	40
Iron	%	2.11	2.19
Lanthanum	ppm	< 10	<10
Lead	ppm	1	1
Magnesium	%	0.11	0.11
Manganese	ppm	123	127
Mercury	pb	3700	3500
Molybdenum	ppm	5	5
Nickel	ppm	17	18
Phosphorus	ppm	180	180
Potassium	%	0.18	0.19
Selenium	ppm	3.0	3.0
Silver	pm	1.4	1.4
Sodium	%	0.08	0.08
Strontium	ppm	53	55
Zinc	pm	33	31

The second

in the second

TABLE 15

Acid-base Accounting and Metal Content of Solid Component of Slurry After Cyanide Destruction Using Hydrogen Peroxide During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

PARAMETER	UNITS	H3-CD-D-3	H3-CD-D-4	
Acid-Base Accour	nting			
Paste pH		8.7	8.7	
Total Sulphur	%S	1.9	1.8	
Sulphide	%S	0.12	0.14	
Sulphate	%SO4	4.8	4.7	
Maximum Potential				
Acidity	t CaCO ₃ /1000 t	3.8	4.4	
Neutralization				
Potential	t CaCO ₃ /1000 t	20	19	
Net Neutralization				
Potential	t CaCO ₃ /1000 t	16	15	
Metal Analysis				
Aluminum	%	0.30	0.34	
Antimony	ppm	26.0	26.0	
Arsenic	ppm	190	190	
Barium	ppm	30	40	
Beryllium	ppm	< 0.5	< 0.5	
Bismuth	ppm	0.1	0.1	
Calcium	%	2.18	2.36	
Cadmium	ppm	0.1	0.1	
Cobalt	ppm	4	5	
Chromium	ppm	70	81	
Copper	ppm	31	30	
Iron	%	2.02	2.18	
Lanthanum	ppm	< 10	< 10	
Lead	ppm	1	1	
Magnesium	%	0.10	0.11	
Manganese	ppm	117	127	
Mercury	ppb	3700	3600	
Molybdenum	ppm	5	5	
Nickel	ppm	15	16	
Phosphorus	ppm	160	180	
Potassium	%	0.17	0.19	
Selenium	ppm	3.2	3.4	
Silver	ppm	1.5	1.5	
Sodium	%	0.06	0.07	
Strontium	ppm	50	54	
Zinc	ppm	36	35	

TABLE 16

Acid-base Accounting and Metal Content of Solid Component of Slurry After Sodium Sulphide Addition During the May 1988 Pilot Plant Program by Norecol for the Cinola Gold Project

1000

نصع

التناز

١.

	UNITS	H3-S2-D-1	H3-S2-I-1
Acid-Base Accounti	ng		
Paste pH		8.8	8.6
Total Sulphur	%S	1.8	1.9
Sulphur	%S	0.12	0.16
Sulphate	%SO4	4.7	4.8
Maximum Potential			
Acidity	t CacO ₃ /1000 t	3.8	5.0
Neutralization			
Potential	t CaCO3/1000 t	23	23
Net Neutralization			
Potential	t CaCO3/1000 t	19	18
Metal Analysis			
Aluminum	%	0.37	0.41
Antimony	ppm	26.0	26.0
Arsenic	ppm	190	190
Barium	ppm	40	50
Beryllium	ppm	< 0.5	<0.5
Bismuth	ppm	0.1	0.1
Calcium	%	2.29	2.37
Cadmium	ppm	0.1	0.1
Cobalt	ppm	5	5
Chromium	ppm	162	184
Copper	ppm	31	41
Iron	%	2.18	2.22
Lanthanum	ppm	< 10	< 10
Lead	ppm	1	1
Magnesium	%	0.12	0.12
Manganese	ppm	130	130
Mercury	ppb	3300	3700
Molybdenum	ppm	4	5
Nickel	ppm	16	19
Phosphorus	ppm	170	180
Potassium	%	0.22	0.26
Selenium	ppm	3.2	3.0
Silver	ppm	1.5	1.4
Sodium	%	0.09	0.11
Strontium	ppm	54	56
Zinc	ppm	33	33

FEBRUARY 1988 BENCH SCALE TEST PROGRAM FOR THE CINOLA GOLD PROJECT

(Pages 1 to 7)

CONTENTS

Table Page 1 Summary of Tailings Liquid Component Characterization from the February 1988 Bench Scale Testwork by Norecol, Cinola Gold Project 1 Summary of Tailings Solid Component Characterization from the February 1988 Bench 2 Scale Testwork by Norecol, Cinola Gold Project 3 Summary of Short Term Leach Results of Tailings Solids from the February 1988 Bench 3 Scale Testwork by Norecol, Cinola Gold Project 4 Long Term Leach Results of Tailings Solids from the February 1988 Bench Scale 4 Testwork by Norecol for the Cinola Gold Project 5 5 Size Analysis of Tailings from the February 1988 Bench Scale Testwork, After Sodium Sulphide Addition by Norecol, Cinola Gold Project 7

TABLE 1

Summary of Tailings Liquid Component^a Characterization From the February 1988 Bench Scale Testwork by Norecol, Cinola Gold Project

PARAMETER	UNITS	AFTER NITRATE OXIDATION	AFTER NEUTRALIZATION	BEFORE CYANIDATION	AFTER CYANIDATION	AFTER CYANIDE DESTRUCTION	AFTER SODIUM SULPHIDE ADDITION
						100	100
EMF	mv	652	288	•	87	108	183
рН		1.7	9.2	-	10.2	9.7	9.7
Conductivity	umhos/cm	21667	3917	-	6567	6167	6433
Sulphate	mg/L	15300	1427	-	1823	1903	1787
Ammonia - N	mg/L	-	-	-	-	0.222	-
Nitrate - N	ma/L	-	-	-	218	231	206
Nitrite - N	mg/L	-	-	-	-	2.78	-
Alkalinity	mg CaCO ₃ /L		-	-	-	128	-
Chloride	mg/L	-	-	-	-	502	513
Total Cyanide	mg/L	-	-		377	6.83	7.99
Total Cyanide ^b	mg/L	-	-	-	-	0.27	-
WAD Cyanide	mg/L	-	-	-	-	0.21	-
WAD Cyanide ^b	mg/L	-	-	-	-	0.13	-
Thiocyanate	mg/L	-	-	-	-	730	-
Cyanate	mg/L	-	-	-		2.9	-

Р

continued ...

TABLE 1 (concluded)

Summary of Tailings Liquid Component^a Characterization From the February 1988 Bench Scale Testwork by Norecol, Cinola Gold Project

PARAMETER	UNITS	AFTER NITRATE OXIDATION	AFTER NEUTRALIZATION	BEFORE CYANIDATION	AFTER CYANIDATION	AFTER CYANIDE DESTRUCTION	AFTER SODIUM SULPHIDE ADDITION
Dissolved Metals							
Aluminum	mg/L	705	0.40	0.44	0.39	0.20	0.30
Antimony	mg/L	0.15	0.13	0.12	0.16	0.15	0.19
Arsenic	mg/L	60	0.20	0.20	0.17	0.10	0.12
Barium	mg/L	0.29	0.29	0.34	0.29	0.25	0.16
Cadmium	mg/L	0.07	0.001	0.001	0.010	<0.001	<0.001
Cobalt	mg/L	3.6	<0.05	0.05	0.30	0.31	0.32
Chromium	mg/L	2.7	0.005	0.001	0.002	0.022	0.031
Copper	mg/L	17.5	0.04	0.07	14.1	0.98	0.44
Iron	mg/L	1945	0.07	0.02	3.3	1.9	2.7
Lead	mg/L	0.051	<0.004	0.002	<0.001	<0.001	<0.001
Manganese	mg/L	46	<0.02	0.02	<0.02	<0.02	<0.02
Mercury	ug/L	426	32	1.1	156	75	0.33
Molybdenum	mg/L	0.06	0.11	0.10	0.11	0.11	0.13
Nickel	mg/L	4.8	<0.014	0.017	0.029	<0.01	<0.01
Phosphorus	mg/L	77	0.203	0.26	0.184	0.19	0.27
Silver	mg/L	0.0015	0.0007	0.0019	0.011	0.018	<0.001
Selenium	mg/L	0.51	0.027	0.019	0.065	0.010	0.012
Zinc	mg/L	16.0	<0.03	0.02	3.2	<0.02	<0.02
Calcium	mg/L	-	-	-		894	1050
Magnesium	mg/L	-	-	-		3.02	2.0
Potassium	mg/L	-	-	-		90	88
Sodium	mg/L	-	-	-		758	783

^a Average Value

^b Concentrations achieved by Degussa during bench scale work on subsamples of batch (R. Norcross, pers. com. March, 1988).

TABLE 2

Summary of Tailings Solid^a Characterization from the February 1988 Bench Scale Testwork by Norecol Cinola Gold Project

PARAMETER	UNITS	FEED	AFTER NITRIC ACID ADDITION	AFTER NITRATE OXIDATION	AFTER NEUTRALIZATIO	AFTER N CYANIDATION	AFTER CYANIDE DESTRUCTION	AFTER SODIUM SULPHIDE ADDITION
Acid-Base Accounting								
Paste pH		4.6	3.2	-	-	-	9.0	-
Sulphur	%S	1.4	1.4	-	-	-	1.3	-
Sulphate	%SO4	0.23	0.19	-	-	-	3.3	-
Sulphide	%S	1.4	1.2	-	-	-	0.07	-
Maximum Potential Acidity	t CaCO ₃ /1000 t	44	39	-	-	-	2.0	-
Neutralization Potential	t CaCO ₃ /1000 t	-1.5	-2.3	-	-	-	23	-
Net Neutralization Potential	t CaCO ₃ /1000 t	-45	-41	-	-	-	21	-
Metal Analysis								
Aluminum	%	0.30	0.25	0.20	0.30	0.28	0.34	0.34
Antimony	ppm	21.5	22.0	22.7	22.3	22.3	23.0	22.7
Arsenic	ррт	225	210	163	217	220	227	227
Barium	ppm	30	27	33	30	30	37	33
Cadmium	ppm	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Cobalt	ррт	5	4	< 1	4	3	4	3
Chromium	ppm	10	3	< 4	9	8	57	43
Copper	ррт	26	15	5	27	14	205	213
Iron	%	2.26	1.87	2.00	2.11	2.08	2.11	2.06
Lead	ppm	2	2	4	4	3	3	3
Manganese	ppm	90	65	43	101	99	103	100
Mercury	ррр	3500	3267	3733	3867	1533	1367	1533
Molybdenum	ppm	1	1	1	1	1	1	1
Nickel	ppm	8	5	< 1	7	5	6	6
Phosphorus	ppm	245	80	163	220	217	220	213
Silver	ppm	3.4	3.2	3.3	3.3	2.6	2.6	2.8
Selenium	ppm	2.8	2.7	3.5	3.0	3.1	3.7	3.7
Zinc	ppm	35	26	15	38	32	35	35

^a Average value from three tests.

TABLE 3

Summary of Short Term Leach Results of Tailings Solids from the February 1988 Bench Scale Testwork, Cinola Gold Project

		p	н
PARAMETER	UNITS	5.0	8.0
рН		5.3	9.1
Conductivity	umhos/cm	2983	1597
Sulphate	mg/L	918	867
Alkalinity to pH 8.3	mg CaCO ₃ /L	-	-
Nitrate	mg N/L	4.11	4.81
Total P	mg/L	0.073	0.027
Dissolved Metals			
Aluminum	mg/L	1.56	0.03
Antimony	mg/L	0.08	0.04
Arsenic	mg/L	0.038	0.042
Cadmium	mg/L	0.002	< 0.001
Cobalt	mg/L	0.079	0.005
Chromium	mg/L	0.007	< 0.003
Copper	mg/L	1.61	0.008
Iron	mg/L	2.83	0.55
Lead	mg/L	< 0.001	< 0.001
Manganese	mg/L	1.8	0.002
Mercury	ug/L	0.84	< 0.05
Molybdenum	mg/L	< 0.004	< 0.004
Nickel	mg/L	0.099	< 0.0025
Silver	mg/L	0.013	0.006
Selenium	mg/L	< 0.001	< 0.001
Zinc	mg/L	0.407	0.003

	Long Term Leach R	esults of Tailings	Solids	from the	e February 19	88 Bench Scal	e Testwo	ork by No	precol for	the Cinola Gol	d Projec	t	
			pН	5.0			pН	7.0			pН	8.5	
			WE	EK			WE	EK			W	EEK	
PARAMETER	UNITS	1	2	3	4	1	2	3	4	1	2	3	4
рН		>6.5	5.2	5.1	5.3	>8.0	7.6	7.8	7.6	8.9	8.6	8.5	8.3
Eh	mV	460	470	470	430	470	460	470	430	455	455	470	430
Conductivity	uS/cm	2220	3360	2910	2840	1580	2210	2140	2190	1440	1520	1340	1230
Acidity (to pH 8.3)	mg CaCO ₃ /L	5.55	139	44.4	38.9	5.55	5.55	5.25	4.16	5.55	2.90	2.78	2.70
Alkalinity (to pH 4.5)	mg HCO₃/L	71.4	1.71	1.55	1.80	53.6	30.3	25.9	38.0	14.9	33.9	29.0	41.7
Chloride	mg/L	7.00	6.00	6.00	6.00	<1	<1	<1	<1	<1	6.00	1.00	1.00
Sulphate	mg/L	725	665	640	593	726	686	662	640	755	705	686	647
Total Acid ^a	ml	475	135	18	1	129	70	32	12	5	15	3	0
Aluminum	mg/L	<0.015	1.76	2.10	0.71	<0.015	<0.015	0.051	0.31	<0.015	<0.015	0.057	0.36
Antimony	mg/L	0.045	0.052	0.032	0.024	0.033	0.039	0.059	0.063	0.024	0.031	0.041	0.039
Arsenic	mg/L	0.11	0.14	0.096	0.003	0.066	0.026	0.014	0.095	0.048	0.055	0.046	0.069
Barium	mg/L	0.0023	0.0010	0.0010	0.0006	0.0016	0.0006	0.0020	0.0013	<0.0001	0.0010	0.0018	0.0015
Beryllium	mg/L	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003
Bismuth	mg/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	< 0.05	< 0.05	< 0.05	< 0.05	<0.05
Cadmium	mg/L	0.0012	0.0011	0.0014	0.0015	0.0004	0.0013	0.0002	0.0007	0.0004	0.0006	<0.0001	0.0004
Calcium	mg/L	463	473	443	474	425	412	389	423	413	326	293	294
Chromium	mg/L	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	< 0.003	< 0.003	< 0.003	< 0.003	< 0.003
Cobalt	mg/L	0.008	0.051	0.072	0.074	0.003	0.004	0.005	0.004	0.003	0.003	0.004	0.003

APPENDIX 3.5-1 TABLE 4 ong Term Leach Results of Tailings Solids from the February 1988 Bench Scale Testwork by Norecol for the Cinola Gold Project.

continued . . .

ഗ

.

	Long Term Leach Re	esults of Tailings	Solids 1	rom the	February	1988 Bench Scale	Testwo	rk by No	precol for	the Cinola Gold	i Project		
·····			pH :	5.0			рH	7.0			pН	8.5	
			WE	EK			WE	EK			WE	EK	
PARAMETER	UNITS	1	2	3	4	1	2	3	4	1	2	3	4
Copper	mg/L	0.011	0.86	1.51	0.94	<0.0015	0.013	0.034	0.013	< 0.0015	0.0072	0.015	0.0071
Iron	mg/L	0.096	0.062	0.23	0.078	0.16	0.13	0.55	0.12	0.19	0.17	0.57	0.10
Lead	mg/L	<0.0001	0.0033	<0.0001	<0.0001	<0.0001	0.0002	0.0010	<0.0001	<0.0001	<0.0001	0.0007	<0.0001
Magnesium	mg/L	69.2	162	160	164	22.6	72.5	102	117	8.16	12.8	19.2	21.9
Manganese	mg/L	0.15	1.38	1.78	1.94	0.012	0.093	0.14	0.086	0.0008	0.0020	0.0056	0.0046
Mercury	ug/L	1.9	20	5.0	0.28	0.18	0.85	0.58	0.12	< 0.05	< 0.05	0.15	< 0.05
Molybdenum	mg/L	0.013	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004	< 0.004
Nickel	mg/L	0.014	0.068	0.095	0.097	<0.0025	0.0027	0.0045	0.0036	<0.0025	<0.0025	<0.0025	<0.0025
Potassium	mg/L	1.18	2.62	2.65	2.85	2.30	2.69	2.72	2.53	2.59	2.35	2.33	2.35
Selenium	mg/L	0.007	0.001	0.001	0.001	<0.001	<0.001	0.006	0.006	<0.001	<0.001	<0.001	<0.001
Silicon	mg/L	16.5	39.0	30.9	44.9	11.7	14.5	15.5	20.4	11.1	11.0	2.25	32.1
Silver	mg/L	0.003	0.003	0.003	0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	< 0.003	< 0.003
Sodium	mg/L	9.02	6.55	8.18	8.93	7.06	6.63	7.31	8.19	11.1	9.28	11.4	10.4
Strontium	mg/L	0.27	0.25	0.25	0.23	0.24	0.21	0.20	0.19	0.23	0.18	0.18	0.16
Tin	mg/L	< 0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003	<0.003
Titanium	mg/L	<0.0006	<0.0006	<0.0006	<0.0006	<0.0006	<0.0006	<0.0006	<0.0006	<0.0006	<0.0006	<0.0006	<0.0006
Vanadium	mg/L	0.006	<0.001	< 0.001	< 0.001	< 0.001	< 0.001	< 0.001	< 0.001	< 0.001	< 0.001	< 0.001	< 0.001
Zinc	mg/L	0.018	0.28	0.43	0.44	0.0064	0.0085	0.013	0.0056	0.011	0.0052	0.026	0.0010

APPENDIX 3.5-1 TABLE 4 (concluded)

Total amount of 1.0 N nitric acid added in corresponding week (in week 10.2 N nitric acid was used.)

.

σ

ł

.

.

I

TABLE 5

MESH NO.	WEIGHT RETAINED (%)	CUMULATIVE WEIGHT RETAINED (%)
Test B1		
100	0.97	0.97
140	14.5	15.4
200	19.5	34.9
250	5.8	39.7
325	15.4	55.1
Passing 325	44.0	99.1
Test B2		
100	1.06	1.0
140	13.59	14.7
200	22.0	36.8
250	5.47	42.3
325	11.56	53.9
Passing 325	46.25	100.2
Test B3		
100	0.94	0.94
140	16.31	17.2
200	18.14	35.3
250	12.27	47.6
325	10.44	58.0
Passing 325	41.90	99.9

Size Analysis of Tailings from the February 1988 Bench Scale Testwork, After Sodium Sulphide Addition by Norecol, Cinola Gold Project

CYANIDE DESTRUCTION PROGRAM USING SAMPLES FROM THE MAY 1988 PILOT PLANT PROGRAM FOR THE CINOLA GOLD PROJECT

Table 1

SO₂ Cyanide Destruction Process Continuous Circuit Log Sheet for Chemical Assays by Hazen Research for Cinola Gold Project

				ASSAYS (mg/L)			
STREAM	CNT	CNp	SCN	Cu	Ni	Fe	Zn
Feed	210	-	-	15.1	1.83	0.33	2.8
12:30 Collection	-	0.38	-	0.95	0.21	1.00	0.35
15:10 Collection	-	0.37	-	1.1	0.10	0.87	0.27
17:45 Collection	-	0.32	-	1.1	0.14	0.87	0.11
19:00 Collection	-	0.21	-	1.0	0.21	0.87	0.11
Composite ^a (18h)	0.81	-	1420	1.0	<0.1	0.20	<0.1

^a Analysed at Inco's laboratories.

b Total cyanide by distillation method.

c Total cyanide by picric acid method

.Conditions: 230c, Single stage, 120 minutes retention, pH 9, 5.7 g SO2/g CNT, 10 mg/L Cu^{2+.}

APPENDIX 5.0

APPENDIX 5.1

SAMPLE LOG AND TEST SCHEDULE FOR LABORATORY TESTING OF MAY 1988 PILOT MILL TAILINGS FOR THE CINOLA GOLD PROJECT

(Pages 1 to 3)

APPENDIX 5.1 CONTENTS

Table Page 5.1-1 Lab Sample Log 1 5.1-2 Lab Test Request (Physical Properties) 3

aller were were were were were were and and aller were aller and aller and aller aller and aller

	Consulting	Engineers		DATE_3	13/	7N	L0G0	GED BY	Rock	1.5002	-
			5	LAB_5	Rx- :	* 1)			PA	GE	OF
			LA	TABLE B SAMF	5.1-1 PLE	LOG					
		LOCA	TION	DE	РТН	[EIEL D	DATES	
SAMPLE NO.	DATE OBTAINED	A (GENERAL)	B (LOCAL)	UNITS	REF.	TYPE OF SAMPLE	MATERIAL TYPE	CONTRACTOR OR DRILLER	DENSITY TEST Y or N	TIME RECEIVED 5/13/8%	SUBMITTED By
1.0	5/10/28	TOTAL TALLASS				Buchers	PER CIL	-	N	NOON	HAZEN \$
											ARNALDO MINH
2.0	5/11/88	FEED RESIDUE	1787-06			SA5	TRAIFU	A1402 3	//	"	1
3 U		1866-4-2	6787-06		 	11	4	/	"		~
4.D		1560-5-2	6787-06			11	1	л	11		<i>n</i>
5.0		1866 - 6-2	6787-06		ļ	1	71	/:	//	11	2)
1.0		1866-7-2	6787-05			1,	1	"	//	· ·	11
1.0		1866-8-2	6781-05			"	~	^		~	"
3.0		1866-4-2	6787-00			۸	1	- 1	"	11	1
7.1		186670-2	6787-06			//	/1	//	"	1	1
0.0		1816-11-2	6777-26			/1	/1	"	1:	1	1
1.0		1866-12-2	6787-05			11	"	1	11		1
2.1		1866-13-2	6787-00			1	-1	1	-1	^	1
3.0		1816 - 14-2	1787-01			-1	1	11	^	<i>2</i> 1	4
4.5		1866-15-2	6727-06			21	11	11	•)	15	r
EMARKS					I	l	5				l

					13/	18		ED BY	T.K.D	JIGUE 2	30 1-10
V	Consulting	Engineers		LAB_5	RKee				PA	GE 2	OF
	à			5.1-1 SAMP	(co LE	NT'D) LOG	_				
	T	LOC	CATION	DEI	РТН	*****				Date f	1
SAMPLE NO.	DATE OBTAINED	A (GENERAL)	B (LOCAL)	UNITS	REF.	TYPE OF SAMPLE	MATERIAL TYPE	CONTRACTOR OR DRILLER	DENSITY TEST Y or N	TIME RECEIVED	SUBMITTED BY
15.0		1866-20-2	6787-06			Buch	TREATED	THANS	N	Non	ANZEN C
16.0		1866-21-2	5787-85			21	, '	^	~	1.	PRIMILOD 5.
REMARKS											
11/84											SRK FORM

N

			N .				0 N 0	VIDA					PR	OJE	ст	C	1.1	1) 4	M	60	10					P	SOI	EC1	NO). <u>/</u> ,	2.61	:2.	1
		15	NI	(0)	SFR	(IS	UN &	KIRS	TE	N			FE/	ATU	RE_		5.4.1	Vá	7							1	113	/	V_{i}	X	36	ι,	
N/	Cons	sult	ing	En	gin	eer	<u> </u>					7	DA	TE_	5%	1.5	18	:		REC	UE	STE	ED I	BY_	Tr.	XI		2	1.7	c.r.	<u>۷</u>	Peris	65
V							12.4 .						LA	В	52	<u> </u>	• نه `)	(e.								_P	AGI	E	1_	_ c)F	/	
												TĄ	BL	E !	5.1	- 2													5				
						L	-AB	TES	Т	RE	Q	UE	S	Г	(PI	HY	SI	C4	۱L	P	PR(OP	EF	RTI	ES)								
	GRA	DAT	ION	AT	TER		MOIST./	DENSITY	SP.G	. 8 A	BSO	RPT	ON	≥́		M.D.	REL	ATI	ON	•			PE	RME	ABILII	Y			OTHE	R (S	PECI	FY)	٦
AMPLE	SE		Q	ند	PT.	LURE	W C	1.	+ S		e.	4	RP.	. EQU		Q			Q	RAN	TIVE	M	OLI	D	LOAD	HE	AD		Π	Τ	Τ	Τ	1
NU.	COAR	FINE	HYDF	FLOW	- INO	MOIS	WOLLI	DISPI	- NO.	-NO.	ABSO	+ NO.	ABSO	SAND	698	METH	RAPII	1557	METH	CALT	RELA	8"	6"	4"		F	с						
1.15	1_	Z	7						Z																	-						_	
									-																			-	$\left - \right $		-	-	-
?. ()		-	1		F				K		_						_													-	-+	+	-
1.0		1	7		7				-				-									-				-	-	-	┼╌┤	-	-	+	-
		ŕ	ŕ																														
																																\rightarrow	
	-								 																		-		$\left - \right $		-	+	_
	+	-																								-					-	+	-
				-																												+	
																													$\left \right $			_	
																_													$\left - \right $	_	\rightarrow	+	_
	+	-	-	-	-				-													-		-			-	-	+		-	+	_
FASE			F TI	EST		RF				NG	WIT	H A		AGO)NA				ΔΡ	PR			E P				L	I					
		5-11						0.011					. <u>91</u>																				

and are also and and also also are and are also and also are also and and are also are also

APPENDIX 5.2

GRAIN SIZE ANALYSES, SPECIFIC GRAVITY AND ATTERBERG LIMIT TEST RESULTS FROM MAY 1988 PILOT MILL TAILINGS FOR THE CINOLA GOLD PROJECT

(Pages 1 to 3)

APPENDIX 5.2 CONTENTS

Figure Page 5.2-1 Grain Size Analysis Sample No.1 1 5.2-2 Grain Size Analysis Sample No.2 2

J.2-2	Grain Size Analysis Sample NO.2	•	•	•	•	•	• •	• •	• •	•	•	•	٠	·	•	•	•	•	•	•	•	•	•	•	•	•	•	•	•	2
5.2-3	Grain Size Analysis Sample No.11		•	•	•	•				•	•	•	•	•	•		•	•	•	•		•	•	•	•	•	•	•	•	3







APPENDIX 5.3

COLUMN SETTLED DENSITY AND HYDRAULIC CONDUCTIVITY TEST RESULTS FROM MAY 1988 PILOT MILL TAILINGS FOR THE CINOLA GOLD PROJECT

(Pages 1 to 6)

APPENDIX 5.3 CONTENTS

Table

Page

1

2

3

4

5

6

TABLE 5.3-I

COLUMN SETTLED DENSITY / PERMEABILITY

 PROJECT	CINOLA	SAMPLE NO.	=	1	COLUMN NO.	50
JOB #	62602-16	SPECIMEN N	=	1	BOTTOM DRAINAGE:	YES
 LAB #	L3640	M/C % (initial)	=	161.80 %	DRY DENSITY(initial)	31.2 pcf
LAB	SRK-COLO.	VOID RATIO	=	4.226	POROSITY	80.9 %
FEATURE	TAILINGS	M/C % (final) =		51.50 %		
DATE	05/31/88					

LIFT NO. 1

...

0	Date	Time	Void Ratio	Dry Density	Saturated Moisture	Solids Content	Constant	Permeabi Falling	lity Test Ft./Yr.	cm/sec	Gradient	Sample Height - Lift	Remarks
	NDY	(Hrs.)	(e)	(PCF)	\$	*					(i)	(cm)	
	05/31/88	16.40									1.0	10.03	
	05/31/88	16.41	4.226	31.2	161.9	38.2		X	0.0	0.0E+00	1.0	10.03	
	05/31/88	16.42	6 226	31.2	161.9	38.2		X	0.0	0.0E+00	1.0	10.03	
	05/31/88	16.43	4,200	31.3	160.9	38.3		X	0.0	0.0E+00	1.0	9.98	
	05/31/88	16.43	4,200	31.3	160.9	38.3		X	0.0	0.0E+00	1.0	9.98	
	05/31/88	16.47	4,190	31.4	160.5	38.4		¥	0.0	0.0E+00	1.0	9,96	
	05/31/88	16 53	4 138	31 7	158 5	38 7		¥	0 0	0.0E+00	1.0	9.86	
	05/31/88	16 67	3 908	33 2	149 7	40 0		Y	0.0	0.0E+00	1.1	9.42	
	05/31/88	16 93	3 460	36 5	132 6	43 0		¥	0.0	0 0E+00	1.2	8.56	
	05/31/88	17 40	2 772	4.3 2	106.2	48.5		¥	0.0	0.0E+00	1.4	7.24	
	06/01/88	8 67	1 715	60 0	65.7	60 4		X	40.9	4.0E-05	1.3	5.21	
	06/01/88	14 83	1.715	60 0	65.7	60.4		X	29.6	2.9E-05	1.1	5.21	
	06/02/88	14.46	1.517	64.7	58.1	63.2		X	11.1	1.1E-05	1.0	4.83	
	06/03/88	9.43	1,501	65.1	57.5	63.5		X	0.5	4.4E-07	1.0	4.80	
	06/03/88	11.14	1.501	65.1	57.5	63.5						4.80	2ND LIFT
	06/03/88	13.14	1.501	65.1	57.5	63.5		X	23.9	2.3E-05	3.1	4.80	
	06/03/88	15.33	1.501	65.1	57.5	63.5		X	17.4	1.7E-05	3.0	4.80	
	06/03/88	18.30	1.491	65.4	57.1	63.6		X	14.5	1.4E-05	2.9	4.78	
	06/06/88	9.75	1.491	65.4	57.1	63.6		X	8.1	7.9E-06	2.0	4.78	
	06/06/88	10.25	1.491	65.4	57.1	63.6						4.78	3RD LIFT
	06/06/88	11.25	1.491	65.4	57.1	63.6		X	8.6	8.3E-06	4.3	4.78	
	06/06/88	12.25	1.491	65.4	57.1	63.6		X	12.0	1.2E-05	4.3	4.78	
	06/06/88	14.25	1.491	65.4	57.1	63.6		X	9.4	9.1E-06	4.2	4.78	
	06/06/88	17.75	1.491	65.4	57.1	63.6		X	7.8	7.6E-06	4.2	4.78	
	06/07/88	8.80	1.491	65.4	57.1	63.6		X	8.0	7.8E-06	3.8	4.78	
	06/07/88	16.62	1.491	65.4	57.1	63.6		X	8.8	8.5E-06	3.6	4.78	
	06/08/88	8.38	1.491	65.4	57.1	63.6		X	7.9	7.7E-06	3.3	4.78	
	06/08/88	17.15	1.491	65.4	57.1	63.6		X	0.0	0.0E+00	3.3	4.78	
	06/09/88	8 78	1 480	65 7	56.7	63.8		X				4.76	

 TABLE
 5.3-2

 COLUMN SETTLED DENSITY
 / PERMEABILITY

PROJECT	CINOLA	SAMPLE NO.	1	COLUMN NO.	50
J08 #	62602-16	SPECIMEN NO.	1	BOTTOM DRAINAGE:	YES
LAB #	L3640	M/C % (initial) =	165.50 \$	DRY DENSITY(initial)	31.0 pcf
LAB	SRK-COLO.	VOID RATIO	4.254	POROSITY	81.0 %
FEATURE	TAILINGS	M/C % (final) =	51.30 \$		
DATE	06/03/88				

LIFT NO. 2

LIFT NU. 2		Values for Lift No. 2			Total Values of Lifts 182												
Date	Time	Void Ratio	Dry Density	Saturated	Solids Content	Void Ratio	Dry Density	Saturated	Solids Content	Constant	Permeabi Falling	lity Test Ft./Yr.	cm/sec	Gradient	Sample Total	Height - Lift 2	Remarks
MDY	(Hrs)	(e)	(PCF)	\$	\$	(e)	(PCF)	\$	2				,	(i)	(cm)	(cm)	
06/03/88	11.14	4.254	31.0	163.0	38.0	2,918	41.6	111.8	47.2					1.0	15.49	10.69	
06/03/88	11.15	4.254	31.0	163.0	38.0	2.918	41.6	111.8	47.2		X	0.0	0.0E+00	1.0	15.49	10.69	
06/03/88	11.15	4.229	31.2	162.0	38.2	2.905	41.7	111.3	47.3		X	0.0	0.0E+00	1.0	15.44	10.64	
06/03/88	11.17	4.204	31.3	161.1	38.3	2.887	41.9	110.6	47.5		X	0.0	0.0E+00	1.0	15.37	10.59	
06/03/88	11.20	4.170	31.5	159.8	38.5	2.875	42.1	110.1	47.6		X	0.0	0.0E+00	1.0	15.32	10.52	
06/03/88	11.27	4.145	31.7	158.8	38.6	2.862	42.2	109.7	47.7		X	0.0	0.0E+00	1.0	15.27	10.47	
06/03/88	11.39	3.693	34.7	141.5	41.4	2.791	43.0	106.9	48.3		X	0.0	0.0E+00	1.0	14.99	9.55	
06/03/88	11.64	3.143	39.3	120.4	45.4	2.629	44.9	100.7	49.8		X	0.0	0.0E+00	1.1	14.35	8.43	0
06/03/88	12.14	3.143	39.3	120.4	45.4	2.346	48.7	89.9	52.7		X	163.2	1.6E-04	1.1	13.23	8.43	
06/03/88	13.14	2.632	44.9	100.8	49.8	2.083	52.9	79.8	55.6		X	46.4	4.5E-05	1.2	12.19	7.39	
06/03/88	15.33	2.258	50.0	86.5	53.6	1.891	56.4	72.4	58.0		X	41.4	4.0E-05	1.3	11.43	6.63	
06/03/88	18.30	1.895	56.3	72.6	57.9	1.699	60.4	65.1	60.6		X	52.4	3.1E-05	1.3	10.6/	5.89	
06/06/88	9.75	1.408	6/./	53.9	65.0	1.448	00.0	22.2	64.3		X	10.4	1.62-05	1.0	9.08	4.90	700 1 151
06/06/88	10.25	1.408	0/./	53.9	00.0	1.448	00.0	22.2	04.0		v	17 /	1 75 05	2.1	9.00	4.90	SKU LIFI
06/06/88	11.20	1.408	67.7	53.9	60.0	1.440	00.0	55.5	64.3		Ş	24.2	2 35-05	2.1	9.00	4 90	
06/06/00	14.25	1.400	67.7	57 0	65.0	1.440	60.0	55.5	64.3		÷.	10 0	1 85-05	2.1	9.68	4.90	280
06/06/00	19.20	1,400	67.7	53.9	65.0	1.440	66.0	55.5	64.3		Ŷ	15.8	1.55-05	2.1	9.68	4 90	
06/00/00	17.75	1 303	69 1	53.7	45.0	1.990	66.6	55 2	64.5		Ŷ	16.2	1.6E-05	1 9	9.65	4 87	
06/07/00	16 62	1 303	60.1	57 4	45 2	1.441	66.0	55 2	64.4		Ç	17.8	1 75-05	1.7	9.65	4.07	
06/09/88	8 38	1 303	68 1	53.4	65 2	1 441	66.8	55 2	64 4		Ŷ	15 9	1 55-05	1.6	9.65	4 87	
06/08/88	17 15	1 393	68 1	53 4	65 2	1 661	66.8	55 2	66 6		Ŷ	0.0	0.05+00	1.6	9.65	4 87	
00/00/00	17.19	1.070	00.1	00.4	00.2	1.441	00.0	00.4	0.4.4		~	0.0	0.01100	1. V	1.00	4.07	

TABLE 5.3-3 COLUMN SETTLED DENSITY / PERMEABILITY

PROJECT	CINOLA	SAMPLE NO.	1	COLUMN NO.	50
108 #	62602-16	SPECIMEN NO.	1	BOTTOM DRAINAGE:	YES
LAB #	L3640	H/C % (initial) =	161.3 \$	DRY DENSITY(initial)	32.2 pcf
LAB	SRK-COLO.	VOID RATIO	4.056	POROSITY	80.2 %
FEATURE	TAILINGS	M/C % (final) =	50.10 \$		
DATE	06/06/88				

LIFT NO. 3					•												
Values for Lift No. 3			101	lotal Values of Lifts 1,283													
Date	Time	Void Ratio	Dry Density	Saturated Moisture	Solids Content	Void Ratio	Dry Density	Saturated Moisture	Solids Content	Constant	Permeab Falling	ility Test Ft./Yr.	cm/sec	Gradient (i)	Sample Total (cm)	Height - Lift 3 (cm)	Remarks
MDY	(Hrs.)	(e)	(PCF)	*	\$	(e)	(PCF)	*	\$								
06/06/88	10.25	4.056	32.2	155.4	39.2	2.382	48.2	91.3	52.3					No. 1007/	20.83	11.15	
06/06/88	10.25	4.056	32.2	155.4	39.2	2.382	48.2	91.3	52.3		X	0.0	0.0E+00	1.0	20.83	11.15	
06/06/88	10.25	4.056	32.2	155.4	39.2	2.382	48.2	91.3	52.3		X	0.0	0.0E+00	1.0	20.83	11.15	
06/06/88	10.26	4.056	32.2	155.4	39.2	2.382	48.2	91.3	52.3		X	0.0	0.0E+00	1.0	20.83	11.15	
06/06/88	10.27	4.052	32.3	155.2	39.2	2.380	48.2	91.2	52.3		X	0.0	0.0E+00	1.0	20.82	11.14	
06/06/88	10.28	4.043	32.3	154.9	39.2	2.377	48.2	91.1	52.3		X	0.0	0.0E+00	1.0	20.80	11.12	
06/06/88	10.32	4.034	32.4	154.5	39.3	2.374	48.3	91.0	52.4		X	0.0	0.0E+00	1.0	20.78	11.10	
06/06/88	10.38	4.020	32.5	154.0	39.4	2.369	48.4	90.8	52.4		X	0.0	0.0E+00	1.0	20.75	11.07	
06/06/88	10.50	3.893	33.3	149.2	40.1	2.323	49.0	89.0	52.9		X	0.0	0.0E+00	1.0	20.47	10.79	
06/06/88	10.75	3.331	37.6	127.6	43.9	2.122	52.2	81.3	55.2		X	0.0	0.0E+00	1.1	19.23	9.55	
06/06/88	11.25	2.868	42.1	109.9	47.6	1.957	55.1	75.0	57.2		X	65.3	6.3E-05	1.1	18.21	8.53	
06/06/88	12.25	2.868	42.1	109.9	47.6	1.957	55.1	75.0	57.2		X	43.3	4.2E-05	1.1	17.29	8.53	
06/06/88	14.25	2.451	47.2	93.9	51.6	1.807	58.0	69.2	59.1		X	32.8	3.2E-05	1.2	16.69	7.61	
06/06/88	17.75	2.179	51.3	83.5	54.5	1.710	60.1	65.5	60.4		X	25.9	2.5E-05	1.2	15.88	7.01	
06/07/88	8.80	1.825	57.7	69.9	58.8	1.578	63.2	60.5	62.3		X	26.5	2.6E-05	1.1	15.75	6.23	
06/07/88	16.62	1.766	58.9	67.7	59.6	1.557	63.7	59.7	62.6		X	29.0	2.8E-05	1.1	15.75	6.10	
06/08/88	8.38	1.766	58.9	67.7	59.6	1.557	63.7	59.7	62.6		X	25.9	2.5E-05	1.0	15.72	6.10	
06/08/88	17.15	1.753	59.2	67.1	59.8	1.552	63.8	59.5	62.7		X	0.0	0.0E+00	1.0	15.04	0.07	
06/09/88	8.78	1.453	66.4	55.7	64.2	1.442	66.7	55.2	64.4		X	0.0	0.0E+00	1.0	14.68	5.41	

ω
TABLE 5.3-4

COLUMN SETTLED DENSITY / PERNEABILITY

88

1

PROJECT	CINOLA	SAMPLE NO.	Ξ	1	COLUMN NO.	51
J08 #	62602-16	SPECIMEN	=	2	BOTTOM DRAINAGE:	NONE
LAB #	L3640	M/C * (initial)	=	161.80	DRY DENSITY (initial)	31.1 ocf
LAB	SRK-COLO.	VOID RATIO	:	4.233	POROSITY	30.9
FEATURE	TAILINGS	M/C % (final) =		65.30 \$		
DATE	05/31/88					

LIFT NO. 1

Date	Time	Void	Drv	Saturated	Solids		Permeabi	lity Test		Gradient	Sample Height -	Remarks
		Ratio	Density	Moisture	Content	Constant	Falling	Ft./Yr.	cm/sec		Lift	
MDY	(Hrs)	[4]	(PCF)	\$	\$					(i)	(cm)	
05/31/88	16.72		NG 80 1		50.47					1.0	9.86	
05/31/88	16 73	4 233	31.1	162 2	38.1		X	0.0	0.0E+00	1.0	9.86	
05/31/88	16 73	4 233	31.1	162 2	38.1		¥	0.0	0.0E+00	1.0	9.86	
05/31/88	16 75	4 217	31 2	161.6	38.2		¥	0.0	0.0E+00	1.0	9,83	
05/31/88	16 78	6.201	31.3	161.0	38.3		Y	0.0	0.0E+00	1.0	9.80	
05/31/88	16 85	4 137	31.7	158.5	38.7		X	0.0	0.0E+00	1.0	9.68	
05/31/88	16 98	3 999	32.6	153.2	39.5		X	0.0	0.0E+00	1.0	9.42	
05/31/88	17 25	3 559	35 7	136 4	42 3		X	0.0	0.0E+00	1.1	8.59	
05/31/88	17 72	3 447	36 6	132 1	43.1		¥	0.0	0.0E+00	1.2	8.38	
06/01/88	8 65	2.099	52.6	80.4	55.4		X	0.0	0.0E+00	1.7	5.84	
06/01/88	14.83	2.099	52.6	80.4	55.4		¥	0.0	0.0E+00	1.7	5.84	
06/02/88	14.46	2.099	52.6	80.4	55.4		¥	0.0	0.0E+00	1.7	5.84	
06/03/88	9.43	2.099	52.6	80.4	55.4						5.84	2NO LIFT
06/03/88	9.91	2.099	52.6	80.4	55.4						5.34	
06/03/88	12.86	1.967	54 9	75.4	57.0		X	0.0	0.0E+00	3.6	5.59	
06/03/88	14.83	1.951	55.2	74.7	57.2		¥	0.0	0.0E+00	3.7	5.56	
06/03/88	18.28	1.951	55.2	74.7	57.2		Y	0.0	0.0E+00	3.7	5.56	
06/06/88	9.78	1.829	57.6	70.1	58.8		X	0.0	0.0E+00	3.8	5.33	
06/06/88	10.62	1.829	57.6	70.1	58.8						5.33	3RD LIFT
06/06/88	14.63	1.829	57.6	70.1	58.8						5.33	
06/06/88	17.75	1.829	57.6	70.1	58.8		X	0.0	0.0E+00	5.5	5.33	
06/07/88	8.82	1.829	57.6	70.1	58.3		X	0.0	0.0E+00	5.5	5.33	
06/07/88	16.67	1.829	57.6	70.1	58.8		X	0.0	0.0E+00	5.5	5.33	
06/08/38	8.43	1.829	57.6	70.1	58.8		x	0.0	0.0E+00	5.5	5.33	
06/08/88	17.20	1.829	57.6	70.1	58.8		¥	0.0	0.0E+00	5.5	5.33	
06/09/88	8 93	1 829	57 6	70 1	58 8		X	0.0	0.0E+00	5.5	5.33	

4

TABLE 5.3-5

.

COLUMN SETTLED DENSITY / PERMEABILITY

PROJECT	CINOLA	SAMPLE NO.	Ξ	1	COLUMN NO.	51
JOB #	62602-16	SPECIMEN	=	2	BOTTOM DRAINAGE:	NONE
LAB #	L3640	M/C % (initial)	=	161.80	DRY DENSITY(initial)	32.4 ncf
LAB	SRK-COLO.	VOID RATIO	= .	4.028	POROSITY	80.1
FEATURE	TAILINGS	M/C % (final) =		71.30 %		
DATE	06/03/88					

			Values fo	or Lift No.	2	Total Values of Lifts 182											
LIFT NU. 2		Void Ratio	Dry Density	Saturated Moisture Content	Solids Content	Void Ratio	Dry Density	Saturated Moisture Content	Solids Content	Constant	Permeab Falling	ility Test Ft./Yr.	cm/sec	Gradient	Sample Total	e Height - Lift 2	Remarks
M D Y 06/03/88	(Hrs.) 9 91	(e)	(PCF)	\$	3	(e)	(PCF)	*	2					(í)	(cm) 20 32	(cm) 14 48	
06/03/88	9 92	5 884	23 7	225 4	30 7	6 092	32 0	156 8	38 9		X	0 0	0 0F+00	1 0	20 27	14 43	
06/03/88	9 93	5 846	23.8	224.0	30.9	4.072	32 1	156.0	39 1		X	0.0	0.0E+00	1.0	20.19	14.35	
06/03/88	9.94	5,822	23.9	223.1	31.0	4,060	32.2	155.5	39.1		X	0.0	0.0E+00	1.0	20.14	14.30	
06/03/88	9.98	5.789	24.0	221.8	31.1	4.042	32.3	154 9	39.2		X	0.0	0.0E+00	1.0	20.07	14.23	
06/03/88	10.06	5.664	24.4	217.0	31.5	3.977	32.7	152.4	39.6		X	0.0	0.0E+00	1.0	19.81	13.97	
06/03/88	10.18	5.483	25.1	210.1	32.2	3.881	33.4	148.7	40.2		X	0.0	0.0E+00	1.0	19.43	13.59	
06/03/88	10.43	5.001	27.2	191.6	34.3	3.628	35.2	139.0	41.8		X	0.0	0.0E+00	1.1	18.42	12.58	
06/03/88	11.02	3.704	34.6	141.9	41.3	2.944	41.3	112.8	47.0		X	0.0	0.0E+00	1.3	15.70	9.86	
06/03/88	11.83	3.146	39.3	120.5	45.3	2.650	44.6	101.5	49.6		X	0.0	0.0E+00	1.4	14.53	8.69	ហ
06/03/88	12.86	2.974	41.0	113.9	46.7	2.497	46.6	95.7	51.1		X	0.0	0.0E+00	1.5	13.92	8.33	
06/03/88	14.83	2.649	44.6	101.5	49.6	2.319	49.1	82.8	53.0		X	0.0	0. 0E+00	1.5	13.21	7.65	
06/03/88	18.28	2.344	48.7	89.8	52.7	2.158	51.6	82.7	54.7		X	0.0	0.0E+00	1.6	12.57	7.01	
06/06/88	9.78	2.249	50.2	86.2	53.7	2.050	53.4	78.5	56.0		X	0.0	0.0E+00	1.7	12.14	6.81	
06/06/88	10.62	2.225	50.5	85.2	54.0										12.09	6.76	3RD LIFT
06/06/88	10.62	2.215	50.7	84.9	54.1	2.032	53.7	77.9	56.2		X	0.0	0.0E+00	2.4	12.07	6.74	
06/06/88	10.63	2.206	50.8	84.5	54.2	2.027	53.8	77.7	56.3		X	0.0	0.0E+00	2.4	12.05	6.72	
06/06/88	10.63	2.201	50.9	84.3	54.3	2.025	53.9	77.6	56.3		X	0.0	0.0E+00	2.5	12.04	6.71	
06/06/88	10.65	2.201	50.9	84.3	54.3	2.025	53.9	77.6	56.3		X	0.0	0.0E+00	2.5	12.04	6.71	
06/06/88	10.68	2.187	51.1	83.8	54.4	2.017	54.0	77.3	56.4		X	0.0	0.0E+00	2.5	12.01	6.68	
06/06/88	10.75	2.163	51.5	82.9	54.7	2.005	54.2	76.8	56.6		X	0.0	0.0E+00	2.5	11.96	6.63	
06/06/88	10.87	2.163	51.5	82.9	54.7	2.005	54.2	76.8	56.6		X	0.0	0.0E+00	2.5	11.96	6.63	
06/06/88	11.14	2.129	52.1	81.6	55.1	1.987	54.5	76.1	56.8		X	0.0	0.0E+00	2.5	11.89	6.20	
06/06/88	11.62	2.115	52.5	81.0	55.2	1.980	54./	/5.8	26.9		X	0.0	U.UE+UU	2.5	11.86	0.00	
06/06/88	12.62	2.067	53.1	79.2	55.8	1.954	55.1	74.9	57.2		v	0.0	0.05.00	2.5	11.76	0.43	
06/06/88	14.65	1.910	56.U	75.2	5/./	1.8/2	20./	/1./	58.2		X	0.0	0.00+00	2.0	11.43	6.IU (05	
06/06/88	17.75	1.000	20.2	72.5	58.0	1.859	57.0	/1.2	28.4		\$	0.0	0.00+00	2.0	11.38	0.00	
06/07/88	8.82	1.000	50.5	72.5	58.0	1.009	0/.U	/1.2	20.4		Č.	0.0	0.00+00	2.0	11.08	6.U0 5.07	
06/0//88	10.0/	1.048	57.2	70.0	28.2	1.009	57 4	70.5	20./		Ŷ	0.0	0.00+00	2.0	11.00	5.97	
06/08/88	17 20	1.040	57.2	70.0	50.5	1.039	57.6	70.3	58 7		Ŷ	0.0	0.00+00	2.0	11.00	5 95	
06/09/88	8.93	1.838	57.4	70.4	58.7	1.834	57.5	70.3	58.7		Ŷ	0.0	0.0E+00	2.6	11.28	5.95	

TABLE 5.3-6

COLUMN SETTLED DENSITY / PERMEABILITY

PROJECT	CINOLA	SAMPLE NO.	Ξ	1	COLUMN NO.	51
JOB #	62602-16	SPECIMEN	=	2	BOTTOM DRAINAGE:	NONE
LAB #	L3640	M/C ¥ (initial)	z	161.30 \$	DRY DENSITY(initial)	31.3 ocf
LAB	SRK-COLO.	VOID RATIO	=	4.199	POROSITY	30.8 *
FEATURE	TAILINGS	M/C (final) =		113.30 *		
DATE	06/03/88					

	Values for Lift No. 3	Total Values of Lifts 1.283
LIFT NO. 3		

Date	Time	Void	Dry	Saturated	Solids	Void	Dry	Saturated	Solids		Permeab	ility Test		Gradient	Sample	Height -	Remarks
		Ratio	Density	Moisture	Content	Ratio	Density	Contant	Content	Constant	Falling	rt./1r.	CRISEC		10[4]	LITES	
MDY	(Hrs.)	(.)	(PCF)	tontent	2	(*)	(PCF)	*	1					(i)	(cm)	(cm)	
06/06/88	10.62	(.)	0.017												34		
06/06/88	10.62	3.826	16.6	338.2	22.8	4.134	31.7	153.4	38.7						29.51	17.37	
06/06/88	10.62	8.855	16.5	339.3	22.8	4.134	31.7	158.4	38.7		X	0.0	0.0E+00	1.0	29.51	17.42	
06/06/88	10.63	8.866	16.5	339.7	22.7	4.134	31.7	158.4	38.7		¥	0.0	0.0E+00	1.0	29.51	17.44	
06/06/88	10.63	2.877	16.5	340.1	22.7	4.134	31.7	158.4	32.7		¥	0.0	0.0E+00	1.0	29.51	17.46	
06/06/88	10.65	8.883	16.5	340.3	22.7	4.134	31.7	158.4	38.7		X	0.0	0.0E+00	1.0	29.51	17.47	
06/06/88	10.68	2.883	16.5	340.3	22.7	4.134	31.7	158.4	35.7		¥	0.0	0.0E+00	1.0	29.51	17.47	
06/06/88	10.75	8.244	17.6	315.9	24.0	3.932	33.0	150.7	39.9						28.35	16.34	6
06/06/88	10.87	8.040	18.0	308.1	24.5	3,861	33.5	147.9	40.3		x	0.0	0.0E+00	1.1	27.94	15.98	
06/06/88	11.14	7.723	18.7	295.9	25.3	3.763	34.2	144.2	41.0		X	0.0	0.0E+00	1.1	27.58	15.42	
06/06/88	11.62	6.971	20.4	267.1	27.2	3.520	36.1	134.9	42.6		X	0.0	U. 0E+UU	1.1	25.96	14.04	
06/06/88	12.62	5.856	23.8	224.4	30.8	3.172	39.1	121.5	45.1		X	0.0	0.06+00	1.2	23.98	12.12	
06/06/88	14.63	3.944	33.0	151.1	39.5	2.566	45.7	98.3	50.4				2.05.20	1 4	20.50	5.74	
06/06/88	17.75	3.452	36.6	132.3	43.1	2.358	43.5	90.3	52.5		X	0.0	0.00+00	1.7	19.50	(07	
06/07/88	8.82	2.920	41.6	111.9	47.2	2.194	51.0	84.1	54.3		X	0.0	0.02+00	1.9	12.00	0.90	
06/07/88	16.67	2.479	46.8	95.0	51.3	2.050	55.4	78.5	56.0		Š	0.0	0.02+00	1./	17.55	0.10	
06/08/88	8.43	2.507	46.5	96.1	51.0	2.044	53.5	75.3	56.1		ð	0.0	0.00+00	1.08	17.50	0.20	
06/08/88	17.20	2.496	46.6	95.6	51.1	2.041	53.6	78.2	56.1		Š	0.0	0.02+00	1.09	17.45	0.10	
06/09/88	8.93	2.462	67.1	94.3	51.5	2.027	53.2	11.1	20.3		X	0.0	0.00+00	1.09	17.40	0.12	
		2.462	47.1	94.3	51.5	2.027	53.8	11.1	56.5		X	0.0	0.02+00	1.04	17.40	D.12	



APPENDIX 5.4 CONTENTS

Table Page 5.4-1 Consolidation Test, Seating Load 1 Consolidation Test, 0.08 kg/cm² 2 5.4-2 Consolidation Test, 0.16 kg/cm² 5.4-3 3 Consolidation Test, 0.32 kg/cm² 5.4-4 4 . . Consolidation Test, 0.64 kg/cm² 5.4-5 5 Consolidation Test, 1.28 kg/cm² 5.4-6 6 Consolidation Test, 2.56 kg/cm² 5.4-7 7 Figure Page 5.4-1 Consolidation Curve (Voice Ratio vs Effective Stress) 8

TABLE 5.4-I

CONSOLIDATION TEST

PROJECT:	CINOLA	SAMPLE NO.:	1		
PROJECT NO.:	62692-16	SPECIMEN:	1		
LAB NO.:	L364Ø	M/C % (initial)	36.6	M/C 🗣 (final)	28.7
FEATURE:	TAILINGS	VOID RATIO (initial)	0.926	VOID RATIO (final)	0.703
DATE:	96/97/88	DRY DENSITY (initial)	84.6	DRY DENSITY (final)	95.7
		SATURATION % (initial)	103.2	SATURATION & (final)	196.6

VERTICAL EFFECTIVE STRESS APPLIED: Seating Load

Date	Time	Load	Square Rt	Log	Corrected	Height of	Void	Dry
		Elapsed	Elapsed	Elapsed	Ld. Dial	Specimen	Ratio	Density
		Time	Time	Time	Reading			
	(Hrs.)	(min.)	(min.)	(min.)	(mm)	(c m)	(c)	(P c f)
96/97/88	15.55							
96/97/88	15.57	1.0	1.99	a . a a	9.9599	2.38	9.922	84.8
06/07/88	15.62	4.0	2.00	0.60	0.3300	2.35	0.899	85.8
96/97/88	15.68	8.0	2.83	Ø.9Ø	0.4500	2.34	0.889	86.2
96/97/88	15.80	15.0	3.87	1.18	0.6550	2.32	0.873	87.0
96/97/88	16.95	39.9	5.48	1.48	9.7859	2.31	Ø.862	87.5
96/97/83	16.06	30.3	5.50	1.48	8.8888	. 2.31	0.861	87.5
Ø6/Ø8/88	8.37	1,009.0	31.76	3.00	0.8400	2.30	0.858	87.7

F

TABLE 5.4-2

CONSOLIDATION TEST

PROJECT:	CINOLA	SAMPLE NO.:	1		
PROJECT NO. :	62682-16	SPECIMEN:	1		
LAB NO.:	L3640	M/C % (initial)	36.6	M/C % (final)	28.7
FEATURE:	TAILINGS	VOID RATIO (initial)	9.926	VOID RATIO (final)	a . 7 a 3
DATE:	06/07/88	DRY DENSITY (initial)	84.6	DRY DENSITY (final)	95.7
		SATURATION % (initial)	103.2	SATURATION % (final)	106.6

VERTICAL EFFECTIVE STRESS APPLIED: 0.08 Kg/cm2

Date	Time	Load	Square Rt	Log	Corrected	Height of	Vold	Dry
		Elapscd	Elapscd	Elapsed	Ld. Dial	Specimen	Ratio	Density
		Time	Time	Time	Reading			
	(Hrs.)	(min.)	(min.)	(min.)	(mm)	(cm)	(e)	(P c f)
06/08/88	8.57							
06/08/88	8.57	Ø.1	0.31		1.0250	2.29	0.843	88.4
06/08/88	8.57	0.2	0.50		1.0650	2.28	0.840	88.6
96/98/88	8.58	Ø.5	9.71		1.1599	2.27	0.833	88.9
06/08/89	8.58	1.0	1.00	Ø. ØØ	1.2450	2.26	0.825	89.3
06/08/88	8.60	2.0	1.41	0.30	1.3300	2.25	0.818	89.6
96/98/88	8.53	4.0	2.99	9.69	1.4025	2.25	0.813	89.9
96/98/88	8.79	8.0	2.83	a . 9a	1.4525	2.24	Ø. 8Ø9	90.1
06/08/88	8.82	15.0	3.87	1.19	1.4800	2.24	0.806	90.2
06/08/88	9.07	30.0	5.48	1.48	1.4975	2.24	0.805	90.3
06/08/88	9.78	73.0	8.54	1.86	1.5200	2.24	9.893	90.4
96/98/88	19.57	129.9	10.95	2.08	1.5300	2.23	9.892	90.4
06/08/88	12.57	240.0	15.49	2.38	1.5375	2.23	0.802	90.4
06/08/88	16.57	480.0	21.91	2.68	1.5475	2.23	0.801	90.5
96/99/88	8.57	1,440.0	37.95	3.16	1.5575	2.23	Ø.899	90.5

N

TABLE 5.4-3

CONSOLIDATION TEST

Note and then have then have then have

PROJECT:	CINOLA	SAMPLE NO.:	1		
PROJECT NO. :	62692-16	SPECIMEN:	1		
LAB NO.:	L364Ø	M/C % (initial)	36.6	M/C 🔨 (final)	28.7
FEATURE:	TAILINGS	VOID RATIO (initial)	0.926	VOID RATIO (fir.al)	2.703
DATE:	06/07/88	DRY DENSITY (initial)	84.6	DRY DENSITY (final)	95.7
		SATURATION % (initial)	193.2	SATURATION & (final)	106.6

VERTICAL EFFECTIVE STRESS APPLIED: 0.10 Kg/cm2

Date	Time	Load Elapsed Time	Square Rt Elapsed Time	Log Elapsed Time	Corrected Ld. Dial Reading	Height of Specimen	Void Ratio	Dry Density
	(Hrs.)	(min.)	(min.)	(min.)	(mm)	(c m)	(c)	(P c f)
06/09/88	8.63							
96/99/88	8.64	9.1	9.32		1.6025	2.23	9.899	94.5
06/09/89	8.64	Ø.3	0.50		1.6025	2.23	0.800	90.5
06/09/88	8.64	Ø.5	0.71		1.6025	2.23	0.800	96.5
06/09/88	8.65	1.0	1.00	0.00	1.6925	2.23	9.899	90.5
96/99/88	8.67	2.9	1.42	9.39	1.6050	2.23	a . 8 a a	90.5
06/09/88	8.70	4.0	2.00	0.60	1.6200	2.23	0.799	90.6
06/09/88	8.77	8.0	2.83	0.90	1.6375	2.23	0.797	90.7
06/09/88	8.88	15.0	3.87	1.18	1.6425	2.23	0.797	90.7
96/99/88	9.13	39.9	5.48	1.48	1.6450	2.23	9.797	99.7
06/09/88	9.63	60.0	7.75	1.78	1.6475	2.23	0.797	99.7
06/09/88	10.63	120.0	10.95	2.08				
06/09/88	12.63	240.0	15.49	2.38	1.7475	2.22	0.788	91.1
96/99/88	16.93	498.0	22.32	2.79	1.7575	2.22	9.788	91.1
@6/10/88	8.83	1,452.0	38.11	3.16	1.7750	2.21	0.786	91.2

ω

TABLE 5.4-4 CONSOLIDATION TEST

PROJECT:	CINOLA	SAMPLE NO.:	1		
PROJECT NO.:	62692-16	SPECIMEN:	1		
LAB NO.:	L3640	M/C % (initial)	36.6	M/C % (final)	28
FEATURE :	TAILINGS	VOID RATIO (initial)	0.926	VOID RATIO (final)	8.783
DATE:	06/07/88	DRY DENSITY (initial)	84.6	DRY DENSITY (final)	95.7
		SATURATION % (initial)	193.2	SATURATION & (final	146. 6

VERTICAL EFFECTIVE STRESS APPLIED: 0.32 Kg/cm2

Date	Time	Load Elapsed Time	Square Rt Elapsed Time	Log Elapsed Time	Corrected Ld. Dial Reading	Height of Specimen	Void Ratio	Dry Density
	(Hrs.)	(min.)	(min.)	(min.)	(mm)	(cm)	(c)	(Pcf)
@6/1@/88	8.97							
96/19/88	8.97	Ø.1	9.31		1.8325	2.21	9.786	91.2
@6/1@/88	8.97	0.2	0.50		1.8325	2.21	0.786	91.2
@6/1@/88	8.98	0.5	0.71		1.8325	2.21	0.786	91.2
06/10/88	8.98	1.0	1.00	0.00	1.8325	2.21	0.786	91.2
96/19/88	9.99	2.9	1.41	9.39	1.8525	2.21	0.785	91.3
06/10/88	9.03	4.2	2.90	0.60	1.9900	2.21	0.782	91.4
06/10/88	9.10	8.0	2.83	0.90	1.9050	2.21	0.780	91.5
96/19/88	9.22	15.0	3.87	1.18	1.9325	2.20	0.778	91.6
96/19/88	9,47	39.9	5.48	1.48	1.9550	2.20	9.776	91.7
06/10/88	10.00	62.0	7.87	1.79	1.9900	2.20	0.774	91.9
06/10/88	10.97	120.0	10.95	2.08	2.0025	2.20	0.773	91.9
06/10/88	12.97	240.0	15.49	2.38	2.0200	2.20	0.771	92.0
96/19/88	17.52	513.9	22.65	2.71	2.0375	2.19	9.779	92.1
@6/13/88	8.58	4,297.0	65.55	3.63	2.0625	2.19	0.768	92.2

100

4

B

TABLE 5.4-5 CONSOLIDATION TEST

PROJECT:	CINOLA	SAMPLE NO.:	1		
PROJECT NO.:	62602-16	SPECIMEN:	1		
LAB NO.:	L3640	M/C % (initial)	36.6	M/C % (final)	28.7
FEATURE:	TALLINGS	VOID RATIO (initial)	0.926	VOID RATIO (final)	9.793
DATE:	06/07/88	DRY DENSITY (initial)	84.6	DRY DENSITY (final)	95.7
		SATURATION % (initial)	103.2	SATURATION % (final)	106 6

VERTICAL EFFECTIVE STRESS APPLIED: Ø.64 Kg/cm2

.

Date	Time	Load	Square Rt	Log	Corrected	Height of	Void	Dry
		Elapsed	Elapsed	Elapsed	Ld. Dial	Specimen	Ratio	Density
		Time	Time	Time	Reading			
	(Hrs.)	(min.)	(min.)	(min.)	(mm)	(c m)	(e)	(Pcf)
06/13/88	8.63							
@6/13/88	8.64	Ø.1	0.32		2.1375	2.19	0.768	92.2
06/13/88	8.64	0.3	0.50		2.1375	2.19	0.768	92.2
96/13/88	8.64	9.5	9.71		2.1375	2.19	9.768	92.2
Ø6/13/88	8.65	1.0	1.00	0.00	2.1375	2.19	0.768	92.2
@6/13/88	8.67	2.0	1.42	0.30	2.1375	2.19	0.768	92.2
@6/13/88	8.70	4.9	2.00	Ø.6Ø	2.1375	2.19	0.768	92.2
96/13/88	8.77	8.9	2.83	a. 9a	2.1675	2.19	9.765	92.3
@6/13/88	8.88	15.0	3.87	1.18	2.2025	2.19	0.762	92.5
96/13/88	9.13	30.0	5.48	1.48	2.2250	2.18	0.761	92.5
96/13/88	9.63	69.9	7.75	1.78				
96/13/88	11.63	189.9	13.42	2.26	2.2959	2.18	0.755	92.8
96/13/88	12.63	249.0	15.49	2.38	2.3075	2.17	0.754	92.9
96/13/88	16.63	489.9	21.91	2.68	2.3300	2.17	0.752	93.0
96/14/88	6.72	1,325.0	36.40	3.12	2.3600	2.17	9.759	93.1

ហ

TABLE 5.4-6 CONSOLIDATION TEST

PROJECT:	CINOLA	SAMPLE NO.:	1		
PROJECT NO.:	62692-16	SPECIMEN:	1		
LAB NO.:	L364Ø	M/C % (initial)	36.6	M/C % (final)	29.7
FEATURE:	TAILINGS	VOID RATIO (initial)	0.926	VOID RATIO (final)	0.703
DATE:	96/97/88	DRY DENSITY (initial)	84.6	DRY DENSITY (final)	95.7
		SATURATION % (initial)	193.2	SATURATION & (final)	196.6

VERTICAL EFFECTIVE STRESS APPLIED: 1.28 Kg/cm2

Date	Time	Load Elapsed Time	Square Rt Elapsed Time	Log Elapsed Time	Corrected Ld. Dial Reading	Height of Specimen	Void Ratio	Dry Density
	(Hrs.)	(min.)	(min.)	(min.)	(mm)	(cm)	(c)	(P c f)
06/14/88	6.75							
96/14/88	6.75	9.1	9.32		2.4559	2.17	9.759	93.1
06/14/88	6.75	0.3	0.50		2.4550	2.17	0.750	93.1
06/14/88	6.76	0.5	0.71		2.4550	2.17	0.750	93.1
96/14/88	6.77	1.0	1.00	Ø. ØØ	2.4725	2,17	0.748	93.2
@6/14/88	6.78	2.9	1.41	9.39	2.4950	2.17	9.747	93.3
06/14/88	6.82	4.0	2.00	0.60	2.5200	2.16	0.744	93.4
06/14/88	6.88	8.0	2.83	0.90	2.5475	2.16	0.742	93.5
06/14/88	7.00	15.0	3.87	1.18	2.5700	2.16	0.740	93.6
96/14/88	7.25	39.9	5.48	1.48	2.5950	2.16	9.738	93.7
96/14/88	7.75	60.0	7.75	1.78	2.6250	2.15	0.736	93.9
\$6/14/88	8.75	120.0	10.95	2.08	2.6550	2.15	0.734	94.0
06/14/88	10.75	240.0	15.49	2.38	2.6775	2.15	9.732	94.1
\$6/14/88	15.45	522.0	22.85	2.72	2.7199	2.14	9.729	94.2
Ø6/15/88	7.05	1.458.0	39.18	3.16	2.7400	2.14	0.727	94.4

10 B.F.

and the second

80

6

8

TABLE 5.4-7

CONSOLIDATION TEST

PROJECT:	CINOLA	SAMPLE NO.:	1		
PROJECT NO.:	62692-16	SPECIMEN:	1		
LAB NO.:	L364Ø	M/C % (initial)	36.6	M/C % (final)	28
FEATURE:	TAILINGS	VOID RATIO (initial)	0.926	VOID RATIO (fir.al)	9.723
DATE:	06/07/88	DRY DENSITY (initial)	84.6	DRY DENSITY (final)	95 ~
		SATURATION % (initial)	143.2	SATURATION & (final	196 -

VERTICAL EFFECTIVE STRESS APPLIED: 2.56 Kg/cm2

.

.

Date	Time	Load	Square Rt	Log	Corrected	Height of	Void	Dry
		Elapsed Time	Elapsed Time	Elapsed Time	Ld. Dial Reading	Specimen	Ratio	Density
	(Hrs.)	(min.)	(min.)	(min.)	(mm)	(cm)	(c 1	(Pcf)
06/15/88	7.15							
@6/15/88	7.15	9.1	9.32			2.14	9.727	94.4
06/15/88	7.15	Ø.3	0.50		2.8500	2.14	0.727	94.4
@6/15/88	7.16	Ø.5	0.71		2.8500	2.14	0.727	94.4
06/15/88	7.17	1.0	1.00	0.00	2.8500	2.14	0.726	94.4
96/15/88	7.18	2.0	1.41	9.39	2.8550	2.14	9.725	94.5
06/15/88	7.22	4.0	2.00	Ø.6Ø	2.8725	2.14	0.723	94.6
06/15/88	7.28	8.0	2.83	Ø.9Ø	2.8950	2.13	0.721	94.7
86/15/88	7.40	15.0	3.87	1.18	2.9225	2.13	0.719	94.8
96/15/88	7.65	39.9	5.48	1.48	2.9599	2.13	9.716	94.9
06/15/88	8.15	60.0	7.75	1.78	2.9800	2.13	0.714	95.0
06/15/88	9.15	120.0	10.95	2.08	3.0050	2.12	0.712	95.2
86/15/88	11.15	240.0	15.49	2.38	3.0375	2.12	0.799	95.3
96/15/88	15.32	499.9	22.14	2.69	3.9675	2.12	9.797	95.5
06/16/88	13.70	1.833.0	42.81	3.26	3.1000	2.11	0.703	95.7



APPENDIX 5.5

GYPSUM CONTENT ANALYTICAL REPORT FROM MAY 1988 PILOT MILL TAILINGS FOR THE CINOLA GOLD PROJECT

(Pages 1 to 6)



CORE LABORATORIES

RECEIVED JUL 05 1988 Ansid.....

Analytical Report

1

880498

STEFFEN ROBERTSON AND KIRSTEN

The analysis, opinions or interpretations contained in this report are based upon observations and material supplied by the client for whose exclusive and conlidential use this report has been made. The interpretations or opinions expressed represent the best judgment of Core Laboratories. Core Laboratories assumes no responsibility and makes no warrantly or representations, express or implied, as to the productivity, proper operations or profitableness nowever of any oil, gas, coal or other mineral property. Well or sund in connection with which such report is used or relevations upon for any reason whatsoever.



CORE LABORATORIES

ANALYTICAL REPORT

July 1, 1988

Client Sample I.D. (CINOLA)	TAILINGS SAMPLE (L364.0-2.0)	
Lab Sample I.D. (880	498)	#1	
PARAMETER	UNITS		
% GYPSUM CONTENT	\$	2.8	

The analysis, opinions or interpretations contained in this report are based upon observations and material supplied by the client for whose exclusive and confident all use this report has been made. The interpretations or opinions expressed represent the best judgment of Core Laboratories. Core Laboratories assumes no responsibility and makes no watrants in represents trons, express or implied, as to the productivity, proper operations or prohitableness nowever of any oil, gas, coal or other mineral, property, well or sand in connection with which output report is used or relied upon for any reason whatsoever.

SOGYP

GYPSUM CONTENT BY PRECIPITATION WITH ACETONE

1.0 Abstract

This soils procedure is adequate for analyzing gypsum content $(CaSO_4 \cdot 2H_2O)$ in a variety of samples. Other forms of calcium sulfate will be analyzed by this same method, which could be a weakness $(CaSO_4.1/2H_2O)$ is also detected). But many clients do not care what hydrated form that Calcium Sulfate is in and use the term gypsum as a broad definition. This procedure is designed to be as specific for the "true form" of gypsum as possible by not oven drying the sample prior to analysis. In soils, the major portion of calcium sulfate will exist as gypsum.

2.0 General Methodology.

2.1 Summary of the Method.

A 20 gram sample 60-mesh air dried soil is transferred to an 8 oz. plastic bottle. Then 200 ml of DI water is added and the bottle is sealed and agitated for 2 hours on a mechanical shaker at room temperature. Then an appropriate aliquot of the filtered solution is added to an equal aliquot of acetone in a centrifuge tube and thoroughly mixed. The sample is allowed to precipitate for 10 to 15 minutes. The sample is centrifuged, supernate decanted, then washed with successive amounts of acetone. Then an aliquot of water is added to dissolve the gypsum, and calcium and sulfate are analyzed by AA and gravimetry respectively.

2.2 Precautions.

The major precaution is to air dry samples at 35 degrees or less because gypsum will dehydrolyze to the $.5H_2O$ form readily which is supposedly more water soluble, therefore causing a positive interference according to the literature. In testing this procedure we have not always found this to be the case. The 1/2 hydrate many times shows a poorer recovery than the 2 hydrate. It is important to pulverize the samples to 60-mesh prior to analysis in order to increase surface area.

A second important constraint is to reanalyze any sample that is found to contain greater than 5.0% GYPSUM in the sample on the 10 g (first run) aliquot. If this occurs, it is advisable to reanalyze by using 2 to 5 gram samples because the solubility of the $CaSO_4 \cdot 2H_2O$ may be exceeded by using the prior sized aliquot.

3.0 Equipment

- 3.1 8 oz. plastic bottles.
- 3.2 Appropriate glassware including 100 ml vol. flasks, long stem funnels, etc.

- 3.3 Shaking apparatus appropriate for the 8 oz. bottles.
- 3.4 Whatman 42 analytical grade, 12 cm dia. filter paper.
- 3.5 Centrifuge, with 50 ml plastic centrifuge tubes.
- 3.6 Wash bottles for the reagent grade acetone and DI water.

4.0 Reagents

- 4.1 DI water, reagent grade.
- 4.2 Acetone, reagent grade.

5.0 Procedure

- 5.1 Air dry, pulverize, and blend according to the sample prep. procedure found in the soils manual.
- 5.2 Weigh a 20.00 gram sample of 60-mesh sample into an 8 oz. bottle, add ______ 200.0 ml of DI water and cap the bottle tightly.
- 5.3 Agitate on a shaker for 2 hours.
- 5.4 Remove and filter through Whatman 42 filter (or other comparable filter) in Buchner funnel.
- 5.5 Retain most of the filtrate in a plastic bottle and label with the original sample weight and DI water aliquot.
- 5.6 Make a 20 ml (or other as appropriate) aliquot of solution into a 50 ml centrifuge tube.
- 5.7 Add an equal volume of reagent grade acetone and mix thoroughly and allow to sit for 10-15 minutes.
- 5.8 Once the precipitation of the "gypsum" is complete, centrifuge and ecant the solution.
- 5.9 Add 10 ml of acetone to the c-tube, cap and disperse the ppt by tapping the side of the tube with a finger.
- 5.10 Wash down the sides of the centrifuge tube with 2-3 extra ml of acetone if necessary, centrifuge, and again decant the solution.
- 5.11 Repeat steps 5.9 and 5.10.
- 5.12 Invert the tube with the ppt and allow to drain to remove most of the acetone within 10 minutes.
- 5.13 Add 40.00 ml of DI water to the sample tube, cap tightly and agitate to completely dissolve.

- 5.14 Transfer to a 100 ml volumetric flask and dilute to volume with DI water and mix well.
- 5.15 Analyze for calcium by atomic absorption and sulfate by gravimetry or an appropriate automated technique. If another analyst is to do these, specify that they be reported in mg/l as analyzed.

6.0 Calculations

6.1 Convert the mg/l of calcium and sulfate to Meg/l.

The two values should be within 5% of each other, if not try to determine the nature of the interference of the characteristic of the sample that is causing the problem. If none can be found utilize the smaller of the 2 values as the maximum value. Otherwise use the average of the meg's to calculate the gypsum content.

 $difference = 200 \times (Meq Ca - Meq SO_4)/(Meq Ca + Meq SO_4)$ Avg. Meq = (Meq Ca + MeqSO_4)/2

6.2 Total Mg of gypsum extracted from original soil aliquot.

 $A = ((X \times B)/C) \times D \times 86.032$

A = Total mg of gypsum (CaSO₄·2H₂O) extracted from soil.

X = Meq's from 6.1.

B = Final solution volume in step 5.14 in liters.

C = Aliquot of solution to be precipitated in step 5.6 in liters. D = The original aliquot as found in step 5.2 in liters (200 ml). 86.032 = the meg wt. in mg of gypsum.

6.3 % Gypsum.

(Mg gypsum/G) \times 0.10 G = wt in grams of original soil aliquot.

7.0 Quality Control and Data Reporting

- 7.1 Analyze duplicates 10% of the time. They should repeat within 0.3% and 15% relative of each other to be acceptable.
- 7.2 Spike 10% of the samples with gypsum at a level of 200 mg to 10 g of soil. The acceptable % recovery should be between 90 and 110%.
- 7.3 If a sample is encountered that contains greater than or equal to 5% gypsum in the soil, THEN THIS SAMPLE SHOULD BE REPEATED BY TAKING A SMALLER SOIL ALIQUOT SUCH AS 2 TO 5 G OF SAMPLE AND REDOING THE PROCEDURE. This is because higher concentrations in the soil will exceed the solubility of gypsum in water.

7.4 Report as % gypsum as CaSO₄·2H₂O to the nearest 0.01.

7.5 Report a detection limit of 0.01%.

8.0 Reference

8.1 Handbook 60, Saline and Alkali Soils, USDA, p. 103.

.

APPENDIX 6.0

.

LIMESTONE COLUMN LABORATORY EXPERIMENTS FOR THE CINOLA GOLD PROJECT

(Pages 1 to 15)

6

M

CONTENTS

Table		Page
1a	General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 1 by Norecol for the Cinola Gold Project	1
1b	Dissolved Metal Concentrations for Tubing Samples from Limestone Column 1 by Norecol for the Cinola Gold Project	3
2a	General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 2 by Norecol for the Cinola Gold Project	4
2b	Dissolved Metal Concentrations for Tubing Samples from Limestone Column 2 by Norecol for the Cinola Gold Project	6
3a	General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 3 by Norecol for the Cinola Gold Project	7
3b	Dissolved Metal Concentrations for Tubing Samples from Limestone Column 3 by Norecol for the Cinola Gold Project	9
4a	General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 4 by Norecol for the Cinola Gold Project	10
4b	Dissolved Metal Concentrations for Tubing Samples from Limestone Column 4 by Norecol for the Cinola Gold Project	12
5a	General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 5 by Norecol for the Cinola Gold Project	13
5b	Dissolved Metal Concentrations for Tubing Samples from Limestone Column 5 by Norecol for the Cinola Gold Project	15

TABLE 1a

General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 1 by Norecol for the Cinola Gold Project

				WEEK										
COLUMN DRAINAGE WATER	PARAMETER	UNITS	1	2	3	4	5	6	7	8	9	10	11	12
BUCKET	pH Sulphoto		5.3	4.8	4.1	3.5	3.3	3.1	3.1	3.1	2.8	3.1	2.9	2.9
	Volume	ing/L	3 40	400	400	3 70	370	742	3.61	3 70	3 70	3 4 1	3 96	3 01
TUBING	oH	L	5.5	3.6	4.4	3.5	3.1	3.2	3.1	3.3	2.8	3.1	3.0	2.9
100110	Eh	mV	475	480	455	455	480	470	480	480	480	480	480	495
	DC	mg/L	10.6	9.2	9.3	9.8	10.0	10.1	10.2	10.2	10.4	10.2	9.0	8.8
	Conductivity	umhos/cm	930	690	890	1120	1400	1220	1360	1360	1500	1830	2120	2520
	Temperature	С	13	12	11.5	15	16.5	15	15	15	12.5	15	17.5	17.0
	Acidity to pH 4.5	mg CaCO ₃ /L	. 0	<1	<1	18	6	58	83	112	248	280	700	1040
	Acidity to pH 8.3	mg CaCO ₃ /L	. 4	20	18	56	84	106	180	280	364	566	960	1432
	Alkalinity to pH 4.	5 mg CaCO ₃ /L	. 10	0	0	0	0	0	0	0	0	0	0	0
	Sulphate	mg/L	470	220	425	750	717	600	780	900	867	992	1312	1781
	Total S (as SO ₄)	mg/L	477	220	425	750	735	766	805	900	883	1040	1312	1781
	As (dissolved)	mg/L	0.006	0.013	0.006	0.051	0.110	0.090	0.034	0.049	0.002	0.070	0.170	0.74
	Hg (total)	ug/L	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	< 0.05	<0.05	<0.05	<0.05	<0.05	< 0.05	<0.05
	H ₂ O Added	L	4.6	5.43	4.95	4.5	4.48	4.07	4.54	4.67	4.37	4.20	4.80	4.70
	H ₂ O Removed	L	3.83	5.38	4.91	4.4	4.47	4.07	4.4	4.52	4.35	4.15	4.00	4.69

continued . . .

щ

TABLE 1a (concluded)

General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 1 by Norecol for the Cinola Gold Project

								w	EEK		·			
COLUMN DRAINAGE WATER	PARAMETER	UNITS	13	14	15	16	17	18	19	20	21	22	23	24
BUCKET	рН		2.5	2.7	2.4	2.2	2.2	2.2	2.1	2.0	2.2	2.2	2.6	2.2
	Sulphate	mg/L	2000	2813	3313	4438	5313	5714	4643	5261	6495	6000	4792	4400
	Volume	L	4.2	3.6	3.6	3.25	3.32	3.65	4.10	3.87	3.61	3.78	4.25	3.77
TUBING	рН		2.35	2.3	2.3	2.2	2.2	2.2	2.2	2.0	2.0	2.1	2.6	2.1
	Eh	mV	505	520	580	555	640	570	580	620	600	640	640	650
	DO	mg/L	8.4	9.3	8.7	7.8	8.7	8.9	9.0	7.8	7.3	8.0	8.9	8.5
	Conductivity	umhos/cm	3100	3780	4700	5820	6150	6000	5750	6550	7020	7450	6900	6200
	Temperature	С	15.0	17.0	19.0	16.5	18	18	19	23	23	23	21.5	21.5
	Acidity to pH 4.5	mg CaCO ₃ /	L 1400	2320	3116	4520	5052	3910	3690	4800	5030	5050	4000	4430
	Acidity to pH 8.3	mg CaCO ₃ /	L 2000	3120	4096	5620	6368	4970	4510	5720	6033	5870	4984	5408
	Alkalinity to pH 4.	5 mg CacO ₃ /L	_ 0	0	0	0	0	0	0	0	0	0	0	0
	Sulphate	mg/L	2063	3125	3625	4875	5688	5143	4524	5480	6254	5780	4375	4800
	Total S (as SO4)	ma/L	2063	3125	3625	4875	5688	5143	4762	5490	6254	5780	4385	4800
	As (dissolved)	mg/L	2.0	6.7	15	-	34	-	23	-	28	-	16	-
	Hg (total)	ug/L	< 0.05	< 0.05	< 0.05	-	< 0.05	-	0.065	-	0.15	-	0.18	-
	H ₂ O Added	Ľ	5.00	4.30	4.41	4.00	4.22	4.43	5.02	4.82	4.62	4.60	5.16	4.58
	H ₂ O Removed	L	4.92	4.25	4.40	4.00	4.20	4.15	4.91	4.79	4.59	4.56	5.10	4.49

ł

		D133014				13 101 10	ibing be	inpies i		WEEI	K					na rioje			
PARAMETER	UNITS	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15 ^ª	17	19	21
Aluminum	mg/L	0.43	0.28	1.13	2.73	4.90	5.98	7.45	10.8	12.9	18.3	25.8	41.6	46.8	59.0	47.6	47.1	32.5	24.8
Antimony	mg/L	<0.005	0.055	0.008	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.05	0.14
Arsenic	mg/L	0.006	0.013	0.006	0.051	0.110	0.090	0.034	0.049	0.002	0.070	0.170	0.74	2.1	6.7	15	34	23	-
Barium	mg/L	0.027	0.040	0.040	0.042	0.045	0.036	0.039	0.038	0.032	0.034	0.023	0.015	0.012	0.006	0.001	0.003	0.001	0.001
Beryllium	mg/L	<0.0001	<0.0001	0.0008	0.002	0.0039	0.0047	0.0066	0.0091	0.0094	0.0100	0.0092	0.0087	0.0083	0.0078	0.0048	0.0037	0.002	0.003
Boron	mg/L	<0.001	<0.01	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	< 0.001	<0.001	<0.01	<0.01	<0.001	<0.01	<0.01
Cadmium	mg/L	0.0048	0.0006	0.0059	0.0059	0.0065	0.0050	0.0064	0.0058	0.0064	0.072	0.0034	0.004	0.0057	0.0032	<0.0002	<0.0002	<0.002	<0.002
Calcium	mg/L	159	102	153	210	234	223	205	197	177	171	172	164	141	139	99.6	88.0	59.3	55.7
Chromium	mg/L	0.0010	0.0009	0.0008	0.0005	0.0012	0.0019	0.0044	0.0078	0.0141	0.0248	0.0472	0.0975	0.0187	0.175	0.183	0.104	0.093	0.098
Cobalt	mg/L	0.307	0.0174	0.613	0.919	1.13	1.09	1.05	0.868	0.852	0.841	0.811	1.24	1.38	1.91	1.19	1.40	0.384	0.386
Copper	mg/L	0.0200	0.0058	0.0538	0.132	0.0684	0.0865	0.174	0.216	0.309	0.424	0.487	1.03	1.23	1.54	2.21	0.917	0.894	0.947
Iron	mg/L	0.0453	0.0656	0.387	3.55	13.3	22.2	39.8	77.5	101	196	305	469	612	966	1140	2020	1590	1440
Lead	mg/L	0.002	0.001	0.001	0.002	0.001	0.001	0.003	0.002	0.002	0.006	0.003	0.003	0.003	0.004	0.002	0.003	0.002	<0.02
Lithium	mg/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Magnesium	mg/L	17.0	7.35	12.8	14.7	13.1	10.2	7.81	6.08	4.80	4.38	4.05	4.23	3.78	4.11	3.34	4.07	2.86	3.69
Manganese	mg/L	5.17	1.09	5.41	8.03	8.21	7.65	6.43	5.56	4.95	4.58	4.56	4.39	3.95	4.67	3.21	3.14	2.02	1.95
Mercury (total)	ug/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	0.065	0.15
Molybdenum	mg/L	< 0.001	<0.01	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	0.03	<0.001	<0.001	<0.001	<0.01	0.04
Nickel	mg/L	0.229	0.0149	0.458	0.762	0.921	0.831	0.776	0.568	0.509	0.474	0.426	0.466	0.711	0.784	0.412	0.296	0.412	0.449
Phosphorus	mg/L	0.05	0.08	0.06	<0.05	0.09	<0.05	0.12	0.09	<0.05	0.08	0.10	0.35	0.94	3.21	5.01	14.3	9.77	14.0
Potassium	mg/L	47.5	29.9	40.8	39.4	35.9	28.0	24.4	22.2	18.5	18.0	18.6	17.2	14.9	12.4	10.9	8.9	6.9	4.8
Selenium	mg/L	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	0.002	0.001	0.002	0.005	0.006	0.004	0.005	0.22
Silicon	mg/L	6.43	9.24	12.9	16.5	17.2	17.2	16.5	16.2	14.2	13.3	14.1	16.2	11.3	33.5	41.5	49.7	42.1	44.8
Silver	mg/L	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	0.0013	<0.0002	<0.0002	0.0018	<0.0002	<0.0002	<0.0002	0.0009	0.006
Sodium	mg/L	5.28	0.69	0.68	0.53	0.56	0.35	0.46	0.39	0.22	0.55	0.05	0.32	0.30	0.21	0.31	0.42	0.22	0.34
Strontium	mg/L	0.244	0.157	0223	0283	0293	0261	0.209	0.181	0.145	0.126	0.120	0.114	0.100	0.109	0.091	0.092	0.062	0.069
Thorium	mg/L	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.05	<0.05
Titanium	mg/L	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.002	<0.001	0.002	<0.001	0.001	0.003	<0.001	<0.001	0.002	0.001	1.1
Uranium	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.2	<0.2
Vanadium	mg/L	0.0004	0.0002	0.0002	0.0002	0.0002	0.0002	0.0002	0.0002	0.0002	0.002	0.0002	0.0002	0.018	0.0311	0.0230	0.0002	0.011	0.029
Zinc	mg/L	0.525	0.0211	1.63	3.09	4.04	4.27	4.64	5.54	5.26	5.66	5.56	5.57	5.05	5.22	4.22	1.98	1.88	1.98
Zirconium	mg/L	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	0.005	<0.005	0.007	0.007	<0.005	0.043	0.015	0.054

TABLE 1b

Bi-weekly sampling initiated at Week 15.

ω

TABLE 2a

General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 2 by Norecol for the Cinola Gold Project

······				<u>,</u>				WE	EEK					
COLUMN DRAINAGE WATER	PARAMETER	UNITS	1	2	3	4	5	6	7	8	9	10	11	12
BUCKET	рН		7.7	7.7	7.5	7.6	7.6	7.6	7.7	7.7	7.7	7.8	7.3	7.8
	Sulphate	mg/L	1636	314	256	204	164	142	91	107	98	98	90	100
TUDING	volume	L	3.20	4.36	3.75	3.53	3.70	3.90	4.20	4.20	3.82	3.58	4.04	3.92
TUDING	Pri Fh	mV	465	470	460	455	460	440	455	465	460	470	460	475
	DO	mo/L	10.6	10.1	8.7	10.0	10.2	10.2	10.2	10.2	10.4	10.2	8.9	8.8
	Conductivity	umhos/cm	1510	630	630	605	575	400	390	345	360	360	355	352
	Temperature	С	13	12	11.5	15	16.5	15	15	15	12.5	15	17.5	17.0
	Acidity to pH 4.5	mg CaCO ₃ /L	0	0	0	0	0	0	0	0	0	0	0	0
	Acidity to pH 8.3	mg CaCO ₃ /L	0	< 1	< 1	< 1	< 1	<1	1	< 1	<1	< 1	1	7
	Alkalinity to pH 4.5	i mg CaCO ₃ /L	106	118	115	112	105	98	80	90	91	88	73	84
	Sulphate	mg/L	667	220	213	240	140	130	100	108	98	95	90	100
	Total S (as SO ₄)	mg/L	667	220	223	240	140	130	104	108	98	98	93	100
	As (dissolved)	mg/L	0.003	0.004	0.004	0.003	0.007	0.005	0.006	0.008	0.007	0.007	0.008	0.008
	Hg (total)	ug/L	< 0.05	< 0.05	< 0.05	< 0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	< 0.05	0.05
	H ₂ O Added	L	4.40	5.02	4.60	4.25	4.47	4.77	5.16	5.19	4.64	4.41	4.71	4.66
	H ₂ O Removed	L	3.66	5.00	4.59	4.29	4.47	4.76	5.10	5.10	4.61	4.40	4.70	4.64

B

.

8

.

.

I

continued ...

TABLE 2a (concluded)

General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 2 by Norecol for the Cinola Gold Project

								W	EEK						
COLUMN DRAINAGE WATER	PARAMETER	UNITS	13	14	15	16	17	18	19	20	21	22	23	24	_
BUCKET	рН		7.6	8.1	7.7	8.0	8.0	7.7	7.5	7.2	7.5	7.2	7.6	7.6	-
	Sulphate	mg/L	100	97	88	94	84	83	76	78	106	92	90	91	
	Volume	L	3.88	3.38	3.7	3.47	3.76	3.48	3.96	3.88	3.62	3.88	3.58	3.95	
TUBING	ρН		7.6	7. 3	8.2	7.8	7.3	7.3	7.3	7.3	7.1	6.9	7.4	7.4	
	Eh	mV	470	460	455	450	480	330	350	340	465	455	480	480	
	DO	mg/L	8.3	9.2	9.1	8.9	9.0	9.4	9.2	8.7	8.3	8.3	9.0	8.8	
	Conductivity	umhos/cm	370	325	355	355	340	330	286	355	326	365	335	345	σ
	Temperature	С	15.0	17.0	19.0	16.5	18	18	19	23	23	23	21.5	21.5	-
	Acidity to pH 4.5	mg CaCO3/L	0	0	0	0	0	0	0	0	0	0	0	0	
	Acidity to pH 8.3	mg CaCO ₃ /L	6	<1	2	2	3	13	13	9	12	17	13	8	
	Alkalinity to pH 4.5	mg CaCO3/L	80	78	80	78	78	70	74	70	71	65	69	56	
	Sulphate	mg/L	91	97	78	94	88	75	69	89	106	100	104	82	
	Total S (as SO4)	mg/L	91	97	78	94	88	82	71	92	106	100	104	97	
	As (dissolved)	mg/L	0.007	0.020	0.011	-	0.011	-	0.014		0.051	-	0.041	-	
	Hg (total)	ug/L	< 0.05	< 0.05	<0.05	-	< 0.05	-	< 0.05		< 0.05	-	< 0.05	-	
	H ₂ O Added	L	4.90	4.17	4.48	4.12	4.49	4.07	4.72	4.65	4.57	4.64	4.15	4.68	
	H ₂ O Removed	L	4.80	4.15	4.46	4.1	4.46	4.05	4.69	4.61	4.51	4.61	4.12	4.62	

TABLE 2b

Dissolved Metal Concentrations for Tubing Samples from Limestone Column 2 by Norecol for the Cinola Gold Project

										WEEK									
PARAMETER	UNITS	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15 ⁴	17	19	21
Aluminum	mg/L	0.42	0.29	0.33	0.32	0.34	0.23	•	0.28	0.26	0.25	0.31	0.15	0.26	0.19	0.04	0.05	<0.02	0.18
Antimony	mg/L	0.015	0.062	0.007	0.038	0.050	0.041	-	0.038	0.036	0.040	0.053	0.052	0.032	0.030	0.042	0.028	0.024	0.033
Arsenic	mg/L	0.003	0.004	<0.004	0.003	0.007	0.005	0.006	0.008	0.007	0.007	0.008	0.008	0.007	0.020	0.011	0.011	0.014	-
Barium	mg/L	0.028	0.040	0.037	0.037	0.040	0.027	•	0.023	0.018	0.016	0.014	0.015	0.012	0.014	0.014	0.018	0.011	0.015
Beryllium	mg/L	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	-	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	0.0001
Boron	mg/L	0.068	0.04	<0.001	<0.001	<0.001	<0.001	•	<0.001	<0.001	<0.001	<0.001	<0.001	0.009	0.005	0.003	0.016	<0.001	0.007
Cadmium	mg/L	0.0011	0.0016	0.0002	<0.0002	<0.0002	<0.0002	•	<0.0002	0.0003	0.165	0.0019	<0.0002	0.0017	0.0028	0.0269	<0.0002	<0.0002	<0.0002
Calcium	mg/L	314	110	116	111	102	77.5		66.9	64.0	63.8	68.3	66.2	64.4	56.2	56.1	60.3	62.7	67
Chromium	mg/L	0.0023	0.0007	<0.0002	<0.0002	0.0005	0.0004	-	0.0004	0.0003	0.0007	<0.0002	<0.0002	<0.0002	0.0003	0.0005	<0.0002	<0.0002	<0.0002
Cobalt	mg/L	0.0455	0.0157	0.0183	0.0086	0.0040	<0.005	•	0.0086	0.0019	0.0058	0.0015	<0.0005	0.0071	0.0009	<0.0005	<0.0005	<0.0005	<0.0005
Copper	mg/L	0.034	0.0066	0.0114	0.0135	0.0377	0.0119	•	0.0053	0.0122	0.0057	0.0005	0.0239	0.0433	0.0088	0.0496	0.0224	<0.0005	0.0032
Iron	mg/L	0.0176	0.0831	0.0046	0.0198	0.0662	0.0409	-	0.549	0.0750	0.811	0.015	0.155	1.66	1.22	0.0041	0.0729	0.0149	0.0668
Lead	mg/L	0.001	<0.001	<0.001	0.001	<.001	0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	0.001	<0.001	<0.001	<0.002
Lithium	mg/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	-	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Magnesium	mg/L	26.8	7.42	7.09	6.08	5.20	3.61	-	2.71	2.53	2.35	2.25	2.03	1.90	1.66	1.42	1.34	1.16	1.09
Manganese	mg/L	3.31	0.804	0.659	0.165	0.041	0.003	-	0.056	0.005	0.024	0.005	<0.001	0.013	0.007	0.004	0.006	0.002	0.004
Mercury (total)	ug/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Molybdenum	mg/L	0.020	0.048	0.042	0.030	0.036	0.031	•	0.025	0.026	0.024	0.024	0.025	0.020	0.024	0.023	0.015	0.011	0.015
Nickel	mg/L	0.0360	0.0119	0.0138	0.0092	0.0073	0.0025	-	0.064	0.0024	0.0047	0.0023	0.0017	0.0086	0.0054	0.0019	<0.0005	<0.0005	<0.0005
Phosphorus	mg/L	0.07	0.06	<0.05	<0.05	<0.05	<0.05		0.07	0.06	<0.05	0.09	<0.05	0.09	0.11	0.05	Ü.09	<0.05	0.11
Potassium	mg/L	51.8	30.9	26.9	22.1	19.9	13.3	-	9.1	7.7	7.2	7.2	6.8	6.3	5.8	5.2	4.9	4.3	4.8
Selenium	mg/L	<0.001	<0.001	<0.001	<0.001	0.002	<0.001	<0.001	0.001	<0.001	0.001	0.001	0.002	0.002	0.002	0.002	0.002	0.002	<0.005
Silicon	mg/L	2.01	2.30	2.42	1.43	1.39	2.58	•	2.63	2.40	2.70	2.95	2.96	2.70	2.83	3.35	3.28	3.53	3.93
Silver	mg/L	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	0.0004	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	0.005
Sodium	mg/L	7.49	0.77	0.42	0.26	0.21	0.15	-	0.08	0.10	0.11	<0.05	0.10	0.07	0.51	0.13	0.12	<0.05	0.06
Strontium	mg/L	0.497	0.177	0.188	0.179	0.175	0.130	•	0.105	0.100	0.097	0.105	0.099	0.091	0.084	0.085	0.091	0.088	0.096
Thorium	mg/L	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Titanium	mg/L	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	< 0.001	<0.001	0.003	<0.001	<0.001	0.007	0.002	<0.001	<0.001	0.005
Uranium	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Vanadium	mg/L	<0.0002	<0.0002	<0.0002	<0.0002	0.0003	<0.0002		<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002
Zinc	mg/L	0.312	0.0140	0.0306	0.0225	0.0175	0.0083		0.0427	0.0093	0.0298	0.0099	0.0071	0.015	0.0128	0.0077	0.0110	0.0008	0.0072
Zirconium	mg/L	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	-	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	0.012	<0.005	<0.005	<0.005	0.008

^a Bi-weekly sampling initiated at Week 15

TABLE 3a

General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 3 by Norecol for the Cinola Gold Project

								WE	EEK					
COLUMN DRAINAGE WATER BUCKET TUBING	E PARAMETER	UNITS	1	2	3	4	5	6	7	8	9	10	11	12
BUCKET	pH Sulphate Volume	mg/L	7.7 1697 3.06	7.4 322 4.60	7.6 256 4.20	7.6 196	7.6 143 3.92	7.6 143 3.83	7.7 104 3.87	7.8 122 3.90	7.6 105 3.02	7.9 98 3.72	7.3 95	7.9 100
TUBING	pH Fh	mV	7.6 430	6.9 465	7.2	7.3	7.7 460	7.4	7.7	7.7	7.7 460	7.6 470	7.8	7.9 475
	DO Conductivity	mg/L	10.6	9.3	8.6	9.8	10.2	10.2	10.2	10.2	10.4	10.2	8.9	8.8
	Temperature	C	13	12	12	15	16.5	15	15	15	12.5	15	17.5	17.0
	Acidity to pH 4.5 Acidity to pH 8.3	mg CaCO ₃ /l mg CaCO ₃ /l	_ 0	0 < 1	0 < 1	0 < 1	0 < 1	0 <1	0 <1	0 <1	0 <1	0 <1	0 < 1	0 12
	Alkalinity to pH 4. Sulphate Total S (as SO ₄)	5 mg CaCO ₃ /I mg/L	- 111 530	114 220	104 200	104 230	103 147	94 130	94 126	88 120	87 98	85 97	94 98	78 94
	As (dissolved) Hg (total)	mg/L. ug/L	0.003 < 0.05	0.005 < 0.05	0.005 < 0.05	0.004 < 0.05	0.006 < 0.05	0.006 < 0.05	0.005 < 0.05	0.009 < 0.05	0.010 < 0.05	0.008 <0.05	0.009 < 0.05	0.008 <0.05
	H ₂ O Added H ₂ O Removed	L L	4.3 3.55	5.86 5.54	5.25 5.09	4.82 4.82	4.76 4.73	4.68 4.68	4.74 4.74	4.80 4.78	4.81 4.81	4.57 4.55	4.93 4.92	5.10 5.02

continued . . .

7

TABLE 3a (concluded)

General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 3 by Norecol for the Cinola Gold Project

								W	EEK				<u> </u>	
COLUMN DRAINAGE WATER BUCKET TUBING	PARAMETER	UNITS	13	14	15	16	17	18	19	20	21	22	23	24
BUCKET	pH Sulphate	mg/L	7.5	8.1 97 2.50	8.0 84 3.0	7.9 91 2.65	8.1 84 3.83	7.8 76	7.6 83 3.85	7.5 82 4.02	7.7 106 3.87	7.5 105 3.80	7.7 102 3.74	7.6 100 3.89
TUBING	pH Fh	r V	7.4 470	7.3 465	8.1 460	7.6 455	8.0 480	7.7	7.7	7.3	7.2 445	7.4	7.5 480	7.6 470
	DO	mg/L	8.5	9.2	9.15	9.0	9.0	9.2	9.2	8.7	8.1	8.3	8.95	8.8
	Conductivity Temperature	umhos/cm C	350 15.0	325 17.0	345 19.0	340 16.5	325 18	305 18	286 19	330 23	350 23	355 23	330 21.5	350 21.5
	Acidity to pH 4.5	mg CaCO ₃ /L	0	0	0	0	0	0	0	0	0	0	0	0
	Acially to pH 8.3 Alkalinity to pH 4.5	mg CaCO3/L 5 mg CaCO3/L	8 78	<1 80	1 80	< 1 77	2 76	9 75	9 72	8 70	69	66	72	68
	Sulphate Total S (as SO ₄)	mg/L mg/L	91 91	94 94	78 78	91 91	86 88	74 74	79 79	94 94	101 103	92 92	102 102	100 100
	As (dissolved)	mg/L	0.007	0.010	0.012	-	0.012	-	0.013		0.037	-	0.034	-
	Hg (total)	ug/L	< 0.05	< 0.05	< 0.05	-	< 0.05	-	< 0.05	-	< 0.05	-	< 0.05	-
	H2O Removed	L	4.90 4.83	4.30 4.26	4.82 4.81	4.43 4.41	4.62 4.59	4.80 4.70	4.60 4.57	4.95	4.74 4.68	4.64 4.61	4.54 4.51	4.58 4.52

I

		DISSOIV	ed Meta	Conce	ntration	s for Tu	bing Sa	mples t		WEEK	Column	3 DY NO	recol to	r the Cir	iola Gol	a Proje			- Caranadara - Cara
DADAMETED	UNITS		 2	 2			6	7	Ω	0	10		10	12	14	158	17		
	01113				4	0.00	0.01		0.05	0.06	0.12		0.11	0.04	-0.00	0.04	0.04		0.00
Aluminum	mg/L	0.41	0.29	0.33	0.32	0.29	0.21	0.27	0.25	0.20	0.13	0.33	0.11	0.04	<0.02	0.04	0.04	<0.02	0.22
Antimony	mg/L	0.015	0.056	0.048	0.041	0.046	0.024	0.017	0.042	0.047	0.033	0.050	0.033	0.082	0.035	0.039	0.022	0.020	0.031
Arsenic	mg/L	0.003	0.005	0.005	0.004	0.006	0.000	0.005	0.009	0.010	0.008	0.009	0.008	0.007	0.010	0.012	0.012	0.013	-
Barium	mg/L	0.032	0.046	0.039	0.038	0.038	0.027	0.026	0.022	0.017	0.016	0.014	0.013	0.011	0.013	0.007	0.012	0.012	0.015
Beryllium	mg/L	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001
Boron	mg/L	0.037	0.05	0.004	<0.001	<0.001	<0.001	<0.001	<0.001	<0.01	<0.001	<0.001	<0.001	0.008	0.010	0.003	0.005	<0.001	0.007
Cadmium	mg/L	<0.0002	<0.002	<0.0002	<0.0002	<0.0002	<0.0002	0.0003	0.0005	0.0007	0.057	0.0035	<0.0002	0.0012	0.0201	0.0066	<0.0002	<0.0002	<0.0002
Calcium	mg/∟	254	106	102	95.4	99.7	81.0	/6./	/1.5	61.9	63.1	66.6	63.7	62.9	59.9	52.2	60.7	61.2	63.4
Chromium	mg/L	0.0010	0.0008	0.0002	0.0002	0.0002	0.0004	0.0006	0.0003	<0.002	0.0002	<0.0002	<0.0002	0.0013	<0.0002	0.0004	0.0012	<0.0002	<0.0002
Cobalt	mg/L	0.0599	0.0213	0.0239	0.0114	0.0060	0.0044	0.01	0.0026	0.0050	0.0035	0.0020	0.0015	0.0049	0.0017	<0.0005	<0.0005	<0.0005	<0.0005
Copper	mg/L	0.0250	0.0090	0.0130	0.0066	0.0059	0.0032	0.0075	0.0459	0.0086	0.0022	<0.0005	0.0033	0.0048	0.0234	0.0083	0.0258	<0.0005	0.0177
Iron	mg/L	0.0225	0.124	0.0029	0.0041	0.0141	0.0207	0.215	0.0646	0.331	0.0325	0.0206	0.0080	0.0383	0.156	0.0054	0.0434	0.0122	0.0454
Lead	mg/L	0.002	0.002	<0.001	0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	0.007
Lithium	mg/L	<0.05	<0.05	<0.05	0.08	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.08	<0.05	<0.05	<0.05
Magnesium	mg/L	21.1	7.75	6.45	5.32	4.95	3.75	3.43	2.87	2.39	2.31	2.13	1.93	1.87	1.58	1.31	1.30	0.98	0.98
Manganese	mg/L	3.17	0.977	0.731	0.288	0.038	0.005	0.051	0.003	0.015	0.003	0.003	<0.001	0.003	0.003	<0.001	0.002	0.005	0.006
Mercury (total)	ug/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Molybdenum	mg/L	0.022	0.042	0.046	0.034	0.035	0.029	0.026	0.026	0.025	0.026	0.024	0.023	0.02	0.042	0.020	0.017	0.010	0.014
Nickel	mg/L	0.0402	0.0140	0.0126	0.0075	0.0059	0.0028	0.0073	0.0111	0.0038	0.0017	0.0017	0.0009	0.0049	0.0202	0.0012	<0.0005	<0.0005	<0.0005
Phosphorus	mg/L	0.07	<0.05	<0.05	<0.05	0.05	<0.05	<0.05	<0.05	0.06	0.07	0.07	0.05	0.08	<0.05	<0.05	<0.05	<0.05	0.10
Potassium	mg/L	50.3	32.1	25.3	20.5	18.7	13.0	11.2	9.2	7.3	7.1	7.0	6.5	6.2	4.8	4.9	4.8	4.3	4.6
Selenium	mg/L	<0.001	<0.001	<0.001	<0.001	0.002	0.001	0.002	0.002	0.001	0.001	0.002	0.002	0.002	0.002	0.002	0.002	0.003	<0.005
Silicon	mg/L	2.16	2.27	2.19	1.11	0.80	2.44	2.53	2.48	2.32	2.54	2.80	2.76	2.59	2.74	3.13	3.16	3.22	3.39
Silver	mg/L	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	0.005
Sodium	mg/L	6.19	0.77	0.40	0.33	0.19	0.10	0.18	0.07	0.05	0.27	0.05	0.05	0.05	0.13	0.12	0.08	<0.05	0.08
Strontium	mg/L	0.376	0.170	0.164	0.151	0.161	0.133	0.118	0.112	0.094	0.095	0.099	0.094	0.087	0.089	0.075	0.088	0.083	0.090
Thorium	mg/L	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Titanium	mg/L	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	0.002	<0.001	0.002	0.002	<0.001	0.001	<0.001	0.003
Uranium	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Vanadium	mg/L	<0.0002	<0.0002	<0.0002	<0.0002	0.0003	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	0.0022	0.0185	<0.0002	<0.0002	<0.0002	<0.0002
Zinc	mg/L	0.312	0.0140	0.0306	0.0225	0.0175	0.0083	0.0334	0.0427	0.0093	0.0298	0.0060	0.0060	0.0020	0.0464	0.0036	0.0071	0.0003	0.0122
Zirconium	mg/L	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005

TABLE 3b

. .

* Bi-weekly sampling initiated at Week 15.

TABLE 4a

General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 4 by Norecol for the Cinola Gold Project

<u></u>							<u></u>	WE	EK						-
COLUMN DRAINAGI WATER	E PARAMETER	UNITS	1	2	3	4	5	6	7	8	9	10	11	12	_
BUCKET	pH	4	7.5	7.5	7.6	7.6	7.6	7.6	7.7	7.8	7.6	7.9	7.3	7.9	_
	Sulphate	mg/L	1546	288	264	188	170	155	135	135	120	120	113	119	
TUBING	nH	L	7.3	4.78	4.00	4.06	3.90	3.87 7.4	77	3.90 7.6	3.00 7.7	7.6	7.8	3.73 7 9	
TODING	Eh	mV	420	465	450	450	460	455	465	465	450	475	470	475	
	DO	mg/L	10.6	10.1	8.8	9.9	10.2	10.2	10.2	10.2	10.4	10.2	8.8	8.7	
	Conductivity	umhos/cm	1020	595	575	540	535	395	435	380	350	375	375	380	
	Temperature	С	13	12	12	15	16.5	15	15	15	12.5	15	17.5	17.0	Ľ
	Acidity to pH 4.5	mg CaCO ₃ /l	. 0	0	0	0	0	0	0	0	0	0	0	0	0
	Acidity to pH 8.3	mg CaCO ₃ /I	_ 2	<1	<1	< 1	<1	< 1	2	<1	<1	<1	<1	8	
	Alkalinity to pH 4.	5 mg CaCO₃/l	_ 110	98	90	94	91	84	84	71	78	76	84	74	
	Sulphate	mg/L	432	202	213	170	182	143	139	133	85	120	115	122	
	Total S (as SO ₄)	mg/L	432	220	213	178	182	147	139	133	85	120	115	122	
	As (dissolved)	mg/L	0.004	0.004	0.004	0.004	0.006	0.006	0.006	0.008	0.007	0.007	0.008	0.007	
	Hg (total)	ug/L	< 0.05	< 0.05	< 0.05	<0.05	<0.05	<0.05	<0.05	<0.05	< 0.05	< 0.05	<0.05	<0.05	
	H ₂ O Added	L	4.91	5.92	5.27	4.98	4.84	4.74	5.26	4.85	5.20	4.29	4.80	5.30	
	H ₂ O Removed	L	4.05	5.60	5.07	4.91	4.83	4.73	5.20	4.75	4.15	4.28	4.75	5.29	

6

continued ...

ł

ł

ł

TABLE 4a (concluded)

General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 4 by Norecol For The Cinola Gold Project

							······································	W	EEK					
COLUMN DRAINAGE WATER	PARAMETER	UNITS	13	14	15	16	17	18	19	20	21	22	23	24
BUCKET	pH Sulphate	mg/L	7.3 125	7.4 128 2.26	7.9 109	7.8 113 2.70	8.0 134	7.7 105	7.6 100	7.6 130	7.6 164	7.4 183	7.6 200	7.6 186
TUBING	pH	L m\/	7.6	7.4	8.1	7.7	3.52 8.0	4.00 7.7	3.67 7.7	3.75 7.5	3.55 7.1	4.10 7.3	3.44 7.5	3.92 7.5
	DO Cooductivity	mg/L	480 8.5	485 9.3	460 9.1	450 8.9	9.0 9.0	9.4 9.5	9.2	8.6	430 8.0	440 8.3	480 8.95	470 8.7
	Temperature	umnos/cm C	390 15.0	365 17.0	410 19.0	370 16.5	385 18	350 18	340 19	405 23	455 23	510 23	455 21.5	480 21.5
	Acidity to pH 4.5 Acidity to pH 8.3	mg CaCO₃/L mg CaCO₃/L	0 6	0 < 1	0 2	0 3	0 3	0 9	0 10	0 11	0 12	0 13	0 23	0 8
	Alkalinity to pH 4.8 Sulphate	5 mg Ca CO ₃ /L mg/L	_ 72 119	71 122	69 116	67 116	66 125	61 98	60 117	61 126	54 166	48 190	64 167	57 200
	Total S (as SO ₄)	mg/L	119	122	116	116	125	98	117	130	166	190	167	200
	Hg (total)	ug/L	< 0.005	<0.007	< 0.010	-	0.009 < 0.05	-	0.019 <0.05	-	0.043 < 0.05	-	0.047 < 0.05	-
	H ₂ O Added H ₂ O Removed	L L	4.80 4.73	4.12 4.09	5.26 5.21	5.16 5.14	5.06 5.02	5.60 5.50	5.14 5.12	5.27 5.20	4.34 4.29	5.16 5.09	5.28 5.24	5.50 5.47

TABLE 4b

Dissolved Metal Concentrations for	Tubing Samples from Limestone Colu	mn 4 by Norecol for the Cinola Gold Project

										WEEK								·····	
PARAMETER	UNITS	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15 ^a	17	19	21
Aluminum	mg/L	0.39	0.43	0.27	0.30	0.32	0.29	0.19	0.22	0.22	0.16	0.21	0.15	0.14	0.04	<0.02	<0.02	<0.02	0.41
Antimony	mg/L	0.023	0.015	0.019	0.043	0.052	0.038	0.022	0.037	0.039	0.043	0.046	0.046	0.077	0.030	0.035	0.016	<0.005	0.006
Arsenic	mg/L	0.004	0.004	0.004	0.004	0.006	0.006	0.006	0.008	0.007	0.007	0.008	0.007	0.006	0.007	0.010	0.009	0.019	-
Barium	mg/L	0.030	0.032	0.037	0.039	0.039	0.028	0.024	0.022	0.018	0.018	0.017	0.015	0.015	0.013	0.008	0.016	0.011	0.023
Beryllium	mg/L	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001
Boron	mg/L	0.012	0.02	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.01	<0.001	<0.001	<0.001	0.002	0.008	0.009	0.007	<0.001	0.005
Cadmium	mg/L	0.0026	0.0089	<0.0001	<0.0002	<0.0002	<0.0002	0.0004	<0.0002	0.0004	0.0078	0.0282	0.0012	0.0012	<0.0002	0.0012	<0.0002	0.0002	<0.0002
Calcium	mg/L	199	97.7	101	99.4	100	81.8	78.2	73.1	65.0	68.5	71.9	71.6	68.2	67.1	60.7	72.0	71.1	90.1
Chromium	mg/L	0.0014	0.0010	0.0004	0.0003	<0.0002	0.0005	0.0005	<0.0002	0.0011	0.0003	<0.0002	0.0002	<0.0002	<0.0002	<0.0002	0.0005	<0.0002	<0.0002
Cobalt	mg/L	0.0375	0.016	0.0058	0.0079	0.0098	0.0013	0.0017	0.0054	0.0013	0.0010	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	0.0130
Copper	mg/L	0.0304	0.0187	0.0065	0.0042	0.0063	0.065	0.0003	0.0030	<0.0005	0.0018	0.0021	0.0027	0.0046	0.0047	0.0054	0.0117	0.0061	0.0143
Iron	mg/L	0.0617	0.279	<0.0003	0.0053	0.0906	0.0092	0.0346	0.349	0.0625	0.0395	0.0274	0.0115	0.0073	0.0812	0.0016	0.0417	0.293	5.1
Lead	mg/L	0.002	0.002	<0.001	0.001	<0.001	0.003	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.002
Lithium	mg/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.06	<0.05	<0.05	<0.05
Magnesium	mg/L	17.4	9.37	6.48	5.62	4.93	3.69	3.36	2.83	2.46	2.43	2.20	2.17	2.00	1.71	1.56	1.64	1.32	1.77
Manganese	mg/L	2.93	3.30	0.799	0.319	0.100	0.005	0.005	0.029	0.003	0.004	0.003	<0.001	<0.001	0.003	<0.001	0.002	0.005	0.182
Mercury (total)	ug/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Molybdenum	mg/L	0.023	0.003	0.022	0.031	0.034	0.028	0.026	0.027	0.026	0.027	0.025	0.025	0.01	0.037	0.022	0.014	0.006	0.003
Nickel	mg/L	0.0360	0.211	0.0074	0.0084	0.0118	0.0042	0.0032	0.0140	0.0028	0.0027	0.0020	0.0016	0.013	0.0023	0.0024	<0.0005	<0.0005	0.0218
Phosphorus	mg/L	<0.05	0.06	<0.05	<0.05	0.05	<0.05	0.07	<0.05	<0.05	0.05	0.06	<0.05	0.10	<0.05	<0.05	<0.05	<0.05	0.10
Potassium	mg/L	45.6	38.4	24.6	20.5	18.5	12.7	10.6	9.1	7.3	7.3	7.4	6.9	6.4	5.2	5.2	5.2	4.6	5.7
Selenium	mg/L	<0.001	<0.001	<0.001	<0.001	0.002	<0.001	0.002	0.002	<0.001	0.001	0.001	0.002	0.002	0.002	0.002	0.002	0.003	<0.005
Silicon	mg/L	1.79	9.43	2.08	1.19	0.88	2.37	2.37	2.54	2.31	2.61	2.83	2.94	2.55	2.84	3.45	3.55	3.50	4.27
Silver	mg/L	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.002
Sodium	mg/L	4.82	0.95	0.35	0.24	0.22	0.05	0.11	<0.05	0.08	0.12	<0.05	<0.05	<0.05	0.07	0.10	0.10	<0.05	0.08
Strontium	mg/L	0.294	0.148	0.160	0.154	0.156	0.131	0.117	0.109	0.097	0.099	0.102	0.103	0.090	0.093	0.085	0.100	0.092	0.124
Thorium	mg/L	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005 T	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Titanium	mg/L	< 0.001	<0.001	<0.001	0.003	<0.001	<0.001	< 0.001	<0.001	<0.001	<0.001	0.002	< 0.001	0.002	<0.001	<0.001	<0.001	<0.001	0.004
Uranium	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Vanadium	mg/L	<0.0002	<0.0002	<0.0002	<0.0002	0.0002	<0.0008	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002
Zinc	mg/L	0.296	0.019	0.0099	0.0123	0.0165	0.0068	0.0284	0.0074	0.0071	0.0079	0.0079	0.0058	0.0018	0.0054	0.0064	0.0092	0.0052	0.136
Zirconium	ma/L	< 0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005

* Bi-weekly sampling initiated at Week 15

TABLE 5a

.

General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 5 by Norecol for the Cinola Gold Project

		UNITS	WEEK													
COLUMN DRAINAGE WATER	E PARAMETER		1	2	3	4	5	6	7	8	9	10	11	12		
BUCKET	рH		7.4	7.0	6.9	7.6	7.6	7.6	7.7	7.8	7.5	7.9	7.4	8.0	-	
	Sulphate	mg/L	1303	254	178	196	140	145	113	115	120	103	116	116	Ц	
	Volume	L	3.94	4.39	4.36	4.00	4.52	3.7	3.92	4.00	4.03	1.88	4.77	3.87		
TUBING	pН		7.5	6.9	7.1	7.4	7.7	7.4	7.7	7.7	7.7	7.6	7.8	7.9		
	Eh	mV	420	460	460	460	465	455	465	465	460	470	470	475		
	DO	mg/L	10.6	10.4	8.8	10.0	10.2	10.2	10.2	10.2	10.4	10.2	9.0	8.7		
	Conductivity	umhos/cm	900	560	610	455	415	3 70	390	340	310	495	335	345		
	Temperature	С	13	12	12	15	16.5	15	15	15	12.5	15	17.5	17.0	ယ်	
	Acidity to pH 4.5	mg CaCO ₃ /L	0	0	0	0	0	0	0	0	0	0	0	0		
	Acidity to pH 8.3	mg CaCO ₃ /L	8.3	<1	< 1	2	<1	<1	2	<1	<1	<1	<1	<7		
	Alkalinity to pH 4.	5 mg CaCO ₃ /L	70	68	62	66	74	80	82	76	74	65	80	70		
	Sulphate	mg/L	386	228	235	134	140	127	109	117	83	213	98	103		
	Total S (as SO₄)	ma/L	386	220	235	134	140	137	109	117	85	213	100	103		
	As (dissolved)	ma/L	0.004	0.005	0.005	0.005	0.009	0.007	0.008	0.009	0.010	0.008	0.010	0.011		
	Ho (total)	ua/L	< 0.05	<0.05	< 0.05	< 0.05	<0.05	<0.05	<0.05	<0.05	< 0.05	<0.05	< 0.05	<0.05		
	H ₂ O Added	L	4.9	5.38	5.39	4.92	5.55	4.60	4.96	4.88	5.10	2.50a	5.70	5.05		
	H ₂ O Removed	L	4.35	5.28	5.44	4.85	5.44	4.49	4.85	4.78	5.06	2.45	5.68	5.02		

continued . . .
TABLE 5a (concluded)

General Analytical Parameters for Bucket Samples and Tubing Samples from Limestone Column 5 by Norecol For The Cinola Gold Project

								W	EEK					
COLUMN DRAINAGE WATER	PARAMETER	UNITS	13	14	15	16	17	18	19	20	21	22	23	24
BUCKET	pH Sulphata	mall	7.1	7.8	7.8	7.8	8.1	7.8	7.7	7.5	7.5	7.5	7.6	7.5
	Volume	ing/L	46	4 1	3 47	44	4 12	4 28	3.88	3.76	4.80	4 70	3 28	43
TUBING	pH	L	7.5	7.4	7.9	7.8	8.0	7.8	7.8	7.5	7.2	7.4	7.4	7.5
	Eh	mV	470	465	460	455	480	390	330	330	430	440	490	460
	DO	mg/L	8.6	8.7	9.1	8.9	9.0	9.5	9.2	8.7	8.1	8.4	8.95	8.7
	Conductivity	umhos/cm	340	310	355	355	340	330	340	395	345	455	500	565
	Temperature	С	15.0	17.0	19.0	16.5	18	18	19	23	23	23	21.5	21.5
	Acidity to pH 4.5	mg CaCO ₃ /L	0	0	0	0	0	0	0	0	0	0	3	0
	Acidity to pH 8.3	mg CaCO ₃ /L	6	<1	1	2	3	9	10	8	9	13	19	12
	Alkalinity to pH 4.5	img CaCO ₃ /L	72	70	68	62	65	63	60	44	48	48	55	59
	Sulphate	mg/L	94	88	88	97	113	91	110	137	137	170	257	246
	Total S (as SO ₄)	mg/L	94	88	88	97	113	93	110	137	144	170	257	246
	As (dissolved)	mg/L	0.011	0.012	0.012	-	0.013	-	0.017	•	0.017	-	0.037	-
	Hg (total)	ug/L	< 0.05	<0.05	0.05	-	< 0.05	-	< 0.05	-	< 0.05	-	< 0.05	-
	H ₂ O Added	L	5.75	5.00	4.52	5.12	5.12	5.29	4.80	4.68	6.0	5.90	4.30	5.35
	H ₂ O Removed	L	5.69	4.95	4.51	5.11	5.10	5.25	4.70	4.60	5.91	5.85	4.29	5.28

.

ŧ

PARAMETER UNITS 1 2 3 4 5 6 7 6 9 10 11 12 13 14 15 ⁴ 17 19 21 Aluminum ngL 0.34 0.29 0.32 0.24 0.33 0.04 0.14 0.14 0.01 0.012 0.032 0.025 6.01 0.011 0.011 0.012 <td< th=""><th></th><th></th><th>Dissolv</th><th>ed Meta</th><th>I Conce</th><th>ntration</th><th>s for Tu</th><th>bing Sa</th><th>mples f</th><th>rom Lin</th><th>nestone</th><th>Column</th><th>5 by No</th><th>recol fo</th><th>r the Ci</th><th>nola Go</th><th>ld Proje</th><th>ct</th><th></th><th></th></td<>			Dissolv	ed Meta	I Conce	ntration	s for Tu	bing Sa	mples f	rom Lin	nestone	Column	5 by No	recol fo	r the Ci	nola Go	ld Proje	ct		
PARAMETER UNITS 1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 ⁴ 17 19 21 Auminum mgL 0.34 0.28 0.28 0.24 0.33 0.04 0.044 0.04 0.01 0.11 0.02 0.02 0.02 0.01 0.011 0.012 0.012 0.013 0.011 0.017 0.013 0.017 0.013 0.011 0.011 0.012 0.000 0.001 0.0							· · · · · · · · · · · · · · · · · · ·				WEEI	κ								
Aluminum mpL 0.34 0.22 0.24 0.23 0.21 0.18 0.24 0.20 0.14 0.10 0.11 0.03 0.05 0.022 0.18 Animony mpL 0.019 0.044 0.055 0.005 0.065 0.065 0.065 0.069 0.011 0.011 0.011 0.011 0.011 0.011 0.012 0.001	PARAMETER	UNITS	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15 ⁸	17	19	21
Antimory mp4 0.019 0.047 0.033 0.046 0.036 0.036 0.036 0.036 0.016 0.011 0.012 0.032 0.035 0.015 0.011 0.012 0.012 0.012 0.011 0.011 0.012 0.012 0.011 0.011 0.012 0.010 0.011 0.012 0.010 0.011 0.011 0.012 0.000 <t< td=""><td>Aluminum</td><td>mg/L</td><td>0.34</td><td>0.29</td><td>0.32</td><td>0.29</td><td>0.22</td><td>0.24</td><td>0.23</td><td>0.21</td><td>0.18</td><td>0.24</td><td>0.30</td><td>0.14</td><td>0.10</td><td>0.11</td><td>0.03</td><td>0.05</td><td><0.02</td><td>0.18</td></t<>	Aluminum	mg/L	0.34	0.29	0.32	0.29	0.22	0.24	0.23	0.21	0.18	0.24	0.30	0.14	0.10	0.11	0.03	0.05	<0.02	0.18
Arsenic mpl 0.044 0.055 0.005 0.009 0.007 0.026 0.010 0.011 0.011 0.012 0.012 0.012 0.012 0.011 0.010 0.0001 <	Antimony	mg/L	0.019	0.047	0.033	0.046	0.051	0.036	0.031	0.040	0.043	0.038	0.054	0.043	0.01	0.032	0.036	0.025	0.011	0.021
Barly mpL 0.030 0.040 0.031 0.021 0.013 0.021 0.011 0.011 0.001 0	Arsenic	mg/L	0.004	0.005	0.005	0.005	0.009	0.007	0.008	0.009	0.010	0.008	0.010	0.011	0.011	0.012	0.012	0.013	0.017	
Berylin mpl c-0.001 c-0.002 c-0.002 <thc-0.002< th=""> <thc-0.001< th=""> <thc-0.0< td=""><td>Barium</td><td>mg/L</td><td>0.030</td><td>0.040</td><td>0.052</td><td>0.034</td><td>0.030</td><td>0.023</td><td>0.020</td><td>0.018</td><td>0.013</td><td>0.024</td><td>0.012</td><td>0.013</td><td>0.009</td><td>0.012</td><td>0.006</td><td>0.012</td><td>0.011</td><td>0.011</td></thc-0.0<></thc-0.001<></thc-0.002<>	Barium	mg/L	0.030	0.040	0.052	0.034	0.030	0.023	0.020	0.018	0.013	0.024	0.012	0.013	0.009	0.012	0.006	0.012	0.011	0.011
Bron mgL 0.04 0.05 0.011 -0.001 -0.001 -0.001 -0.001 -0.001 -0.001 -0.001 -0.001 -0.002 -0.003	Beryllium	mg/L	<0.0001	<0.001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	< 0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001
Cadeum mgL 0.015 0.0002 c.0.002 c.0.002 c.0.002 c.0.003 c.0.005 c.0.005 c.0.007 c.0.002 c.0.002 Calcum mgL 0.0.013 0.0.013 0.0.001 c.0.002 c.0.0.002 c.0.002 c.0.003 c.0.003 c.0.001 c.0.001 <thc.0.001< th=""> c.0.001 c.0.001</thc.0.001<>	Boron	mg/L	0.084	0.05	0.014	<0.001	<0.001	<0.001	<0.001	<0.001	<0.01	<0.001	<0.001	<0.001	0.006	0.008	0.009	0.010	0.007	0.004
Calcim mg/L 100 62.3 62.7 58.6 55.0 53.9 64.7 72.2 61.0 Chromium mg/L 0.0013 0.0019 0.0002 0.0005 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0001 0.001 <	Cadmium	mg/L	0.0136	0.0005	<0.0002	<0.0002	<0.0002	<0.0002	0.0004	0.0002	<0.0002	0.0084	0.0230	<0.0002	0.0005	<0.0002	0.0017	<0.0002	<0.0002	<0.0002
Chronimim mg/L 0.0013 0.0010 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0002 0.0003 0.0013 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001 0.001	Calcium	mg/L	190	93.9	103	82.3	76.5	73.4	68.9	66.1	54.4	100	62.3	63.7	58.6	55.0	53.9	64.7	72.2	61.0
cbalit mg/L 0.0218 0.0075 0.0007 0.0007 0.0007 0.0007 0.0005 0.0001 0.0001 0.0011 0.0011 0.001 0.0011 <td>Chromium</td> <td>mg/L</td> <td>0.0013</td> <td>0.0010</td> <td><0.0002</td> <td><0.0002</td> <td><0.0002</td> <td>0.0005</td> <td>0.0006</td> <td>0.0005</td> <td>0.0011</td> <td>0.0003</td> <td><0.0002</td> <td><0.0002</td> <td><0.0002</td> <td><0.0002</td> <td>0.0004</td> <td>0.0014</td> <td><0.0002</td> <td><0.0002</td>	Chromium	mg/L	0.0013	0.0010	<0.0002	<0.0002	<0.0002	0.0005	0.0006	0.0005	0.0011	0.0003	<0.0002	<0.0002	<0.0002	<0.0002	0.0004	0.0014	<0.0002	<0.0002
Copper mg/L 0.4462 0.0992 0.0051 0.0041 0.025 0.017 0.0139 0.0047 0.023 0.0047 0.0054 0.0054 0.0014 0.0054 0.0047 0.0054 0.0054 0.0144 0.0017 0.0147 0.0141 0.0027 0.0037 0.0034 0.0007 0.0014 0.001 0.001 0.0011 0.001	Cobalt	mg/L	0.0218	0.0105	0.0079	0.0030	0.0007	0.0006	0.0009	0.0006	0.0007	0.0007	<0.0005	<0.0005	0.0055	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
iron mg/L 0.126 0.0982 0.0032 0.0176 0.0178 0.0161 0.0171 0.0448 0.0447 0.0047 0.0031 0.0008 0.0068 0.0261 0.0011 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001	Copper	mg/L	0.0462	0.0092	0.0069	0.0051	0.0041	0.025	0.0012	0.0139	0.0009	0.0017	0.023	0.0044	0.0040	0.0047	0.0054	0.0124	<0.0005	0.0023
Lead mg/L 0.003 0.002 0.001 0	Iron	mg/L	0.126	0.0992	0.0032	0.0100	0.0076	0.0178	0.0301	0.0163	0.0517	0.0447	0.0448	0.0467	0.0027	0.0334	0.0008	0.0460	0.0281	0.0341
Lithium mg/L <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <t< td=""><td>Lead</td><td>mg/L</td><td>0.003</td><td>0.002</td><td><0.001</td><td>0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.001</td><td><0.002</td></t<>	Lead	mg/L	0.003	0.002	<0.001	0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.002
Magnesium mg/L 12.9 5.85 6.00 4.27 3.58 3.34 3.00 2.68 2.09 3.71 1.99 2.02 1.59 1.42 1.28 1.30 1.23 0.89 Manganese mg/L 1.74 0.628 0.407 0.056 0.001 0.002 0.001 0.001 0.005 0.001 0.001 0.002 0.001 0.001 0.002 0.001 0.001 0.002 0.001 0.002 0.001 0.001 0.002 0.001 0.002 0.001 0.002 0.005 0.005 0.005 0.005 0.005 0.005 0.005 0.001 0.002 0.001 0.001 0.005	Lithium	mg/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	0.10	<0.05	<0.05	<0.05
Manganese mg/L 1.74 0.628 0.407 0.056 0.008 <0.001 0.002 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001	Magnesium	mg/L	12.9	5.85	6.00	4.27	3.58	3.34	3.00	2.68	2.09	3.71	1.99	2.02	1.59	1.42	1.28	1.30	1.23	0.89
Mercury (total) ug/L <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05	Manganese	mg/L	1.74	0.628	0.407	0.056	0.008	<0.001	0.002	0.002	<0.001	<0.001	0.005	< 0.001	<0.001	0.002	<0.001	<0.001	<0.001	0.005
Molybdenum mg/L 0.021 0.041 0.029 0.030 0.030 0.021 0.021 0.022 0.019 0.019 0.01 0.024 0.018 0.011 0.008 0.007 Nickel mg/L 0.024 0.0108 0.005 0.005 0.0027 0.0026 0.0130 0.021 0.001 0.001 0.001 0.005	Mercury (total)	ug/L	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Nickel mg/L 0.0204 0.0108 0.0066 0.0045 0.0037 0.0026 0.0037 0.0037 0.0016 0.0006 0.0019 0.0017 <0.0005 <0.005 0.0048 Phosphorus mg/L <0.05 0.09 <0.05 <0.05 <0.05 0.066 0.07 0.055 0.056 0.066 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <td>Molybdenum</td> <td>mg/L</td> <td>0.021</td> <td>0.041</td> <td>0.029</td> <td>0.030</td> <td>0.030</td> <td>0.021</td> <td>0.021</td> <td>0.023</td> <td>0.023</td> <td>0.022</td> <td>0.022</td> <td>0.019</td> <td>0.01</td> <td>0.024</td> <td>0.018</td> <td>0.011</td> <td>0.008</td> <td>0.007</td>	Molybdenum	mg/L	0.021	0.041	0.029	0.030	0.030	0.021	0.021	0.023	0.023	0.022	0.022	0.019	0.01	0.024	0.018	0.011	0.008	0.007
Phosphorus mg/L <0.05 0.09 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05 <0.05	Nickel	mg/L	0.0204	0.0108	0.0066	0.0045	0.0035	0.0027	0.0026	0.0130	0.0022	0.0037	0.0031	0.0016	0.0006	0.0019	0.0017	<0.0005	<0.0005	0.0048
Potassium mg/L 37.8 26.3 23.1 17.3 14.6 10.8 9.1 8.0 6.3 8.3 6.6 6.1 5.0 4.8 4.7 4.4 4.4 4.1 Selenium mg/L <0.001	Phosphorus	mg/L	<0.05	0.09	<0.05	<0.05	<0.05	<0.05	0.07	<0.05	0.06	0.07	0.05	0.05	0.06	<0.05	<0.05	<0.05	≺0.05	0.12
Selenium mg/L <0.001 <0.001 <0.001 <0.001 <0.001 0.002 <0.001 0.002 <0.001 0.003	Potassium	mg/L	37.8	26.3	23.1	17.3	14.6	10.8	9.1	8.0	6.3	8.3	6.6	6.1	5.0	4.8	4.7	4.4	4.4	4.1
Silicon mg/L 1.20 1.79 1.94 0.86 0.33 2.51 2.36 2.56 2.24 2.59 2.89 2.85 2.57 2.89 3.31 3.36 3.33 3.33 Silver mg/L -0.0002 -0.001 -0.01	Selenium	mg/L	<0.001	<0.001	<0.001	<0.001	0.002	0.001	0.002	0.002	<0.001	0.002	0.001	0.003	0.003	0.003	0.003	0.003	0.003	<0.005
Silver mg/L <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.0002 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <0.001 <	Silicon	mg/L	1.20	1.79	1.94	0.86	0.33	2.51	2.36	2.56	2.24	2.59	2.89	2.85	2.57	2.89	3.31	3.36	3.53	3.33
Sodium mg/L 3.41 0.69 0.42 0.24 0.11 0.13 0.13 0.10 <0.05 0.06 <0.05 0.08 <0.05 0.24 0.12 <0.05 <0.05 0.18 Strontium mg/L 0.260 0.142 0.164 0.126 0.117 0.115 0.099 0.096 0.080 0.145 0.090 0.080 0.075 0.075 0.074 0.090 0.094 0.083 Thorium mg/L <0.005	Silver	mg/L	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	<0.0002	0.004
Strontium mg/L 0.260 0.142 0.164 0.126 0.117 0.115 0.099 0.096 0.087 0.090 0.080 0.075 0.074 0.090 0.094 0.083 Thorium mg/L <0.005	Sodium	mg/L	3.41	0.69	0.42	0.24	0.11	0.13	0.13	0.10	<0.05	0.06	<0.05	0.08	<0.05	0.24	0.12	<0.05	<0.05	0.18
Thorium mg/L <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 </td <td>Strontium</td> <td>mg/L</td> <td>0.260</td> <td>0.142</td> <td>0.164</td> <td>0.126</td> <td>0.117</td> <td>0.115</td> <td>0.099</td> <td>0.096</td> <td>0.080</td> <td>0.145</td> <td>0.087</td> <td>0.090</td> <td>0.080</td> <td>0.075</td> <td>0.074</td> <td>0.090</td> <td>0.094</td> <td>0.083</td>	Strontium	mg/L	0.260	0.142	0.164	0.126	0.117	0.115	0.099	0.096	0.080	0.145	0.087	0.090	0.080	0.075	0.074	0.090	0.094	0.083
Titanium mg/L < 0.001 0.002 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 < 0.001 <	Thorium	mg/L	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	< 0.005	<0.005	<0.005	<0.005	<0.005	< 0.005	<0.005	<0.005	< 0.005	<0.005	<0.005
Uranium mg/L < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.02 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002 < 0.002	Titanium	mg/L	<0.001	0.002	<0.001	0.003	<0.001	< 0.001	< 0.001	< 0.001	<0.001	<0.001	<0.001	<0.001	< 0.001	0.003	< 0.001	< 0.001	<0.001	0.002
Vanadium mg/L < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 < 0.0002 <th< td=""><td>Uranium</td><td>mg/L</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td>< 0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td><td><0.02</td></th<>	Uranium	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	< 0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Zinc mg/L 0.0209 0.0177 0.0068 0.0101 0.0028 0.0069 0.0059 0.0059 0.0058 0.0088 0.0083 0.0067 0.0024 0.0048 0.0031 0.0077 0.0027 0.0062 Zirconium mg/L <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.00	Vanadium	mg/L	<0.0002	< 0.0002	<0.0002	<0.0002	0.0002	<0.0008	<0.0002	< 0.0002	0.0002	<0.0002	0.0003	<0.0002	< 0.0002	<0.0002	<0.0002	< 0.0002	< 0.0002	<0.0002
Zirconium mg/L <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <0.005 <	Zinc	ma/L	0.0209	0.0177	0.0068	0.0101	0.0028	0.0069	0.0059	0.0070	0.0059	0.0088	0.0083	0.0067	0.0024	0.0048	0.0031	0.0077	0.0027	0.0062
	Zirconium	mg/L	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	< 0.005	< 0.005	<0.005	<0.005	<0.005	<0.005	< 0.005	<0.005	< 0.005	< 0.005	<.005	<.005

TABLE 5b

* Bi-weekly sampling initiated at Week 15

CORRECTION OF ALKALINITY DATA TO REFLECT TRUE BICARBONATE CONCENTRATION

Correction of Alkalinity Data to Reflect True Bicarbonate Concentration

Alkalinity titrations on waters from the limed columns were performed by personnel at B.C. Research. The titration involved the gradual addition of $0.02 \text{ N} \text{ H}_2\text{SO}_4$ until sample pH was lowered to 4.5. At this point, each millilitre of H₂SO₄ added to the sample corresponded to 1 mg of alkalinity as CaCO₃ (Vos, pers. comm.). If 1 L is added, there is 1000 mg of alkalinity as CaCO₃.

From a chemical viewpoint, 1 L of $0.02 \text{ N H}_2\text{SO}_4$ is equivalent to 1 L of 0.02 N H^+ or 0.02 M H^+ . The dominant neutralization reaction by carbonate between a pH of 6.4 and 10.3 is:

$$H^{+} + CaCO_3 = Ca^{2+} + HCO_3^{-}$$
 (1)

This reaction demonstrates that 0.02 moles of H^+ requires 0.02 moles of CaCO₃ (2000 mg CaCO₃) for neutralization. Consequently, if 1 L of 0.02 N H₂SO₄ is added to the sample, alkalinity as CaCO₃ is 2000 mg, which contradicts the value of 1000 mg from the previous paragraph.

The discrepancy is resolved upon the realization that carbonate (CO_3^{2-}) has a valence of 2 and can neutralize a maximum of 2 H⁺ for each CO_3^{2-} . A standard laboratory titration assumes each carbonate ion in the sample has neutralized 2 H⁺ and, thus, the reported alkalinity is divided by 2 to reflect CaCO₃. In reality, each carbonate ion between pH 6.4 to 10.3 has neutralized only one H⁺ so that alkalinity should not be divided by 2 if geochemical and mass balance calculations are involved. This discrepancy in titration interpretation is recognized in the scientific literature and, for the purpose of geochemical calculations in this Addendum, alkalinity values from B.C. Research (Appendix 6.2.3-1) were multiplied by 2.

LIMESTONE BARREL EXPERIMENTS AT THE CINOLA GOLD PROJECT SITE

(Pages 1 to 5)

CONTENTS

Table		Page
1	Measurements of pH in Water Taken from the Collection Buckets of the Limestone Barrels by Norecol (1988) for the Cinola Gold Project	1
2	Water Quality Results from On-site Limestone Barrels for Barrel 1 by Norecol for the Cinola Gold Project	2
3	Water Quality Results from On-site Limestone Barrels for Barrel 2 by Norecol for the Cinola Gold Project	3
4	Water Quality Results from On-site Limestone Barrels for Barrel 3 by Norecol for the Cinola Gold Project	4
5	Water Quality Results from On-site Limestone Barrels for Barrel 4 by Norecol for the Cinola Gold Project	5

TABLE 1

-

DATE CALENDAR DAY 1 2 3 4 APRIL 6 97 - 6.3 3.6 2.6 10 101 2.5 5.9 - 2.6 12 103 2.4 5.9 2.5 3.2 13 104 2.4 5.9 - - 14 105 2.6 5.8 3.5 2.4 19 110 2.3 5.5 3.0 2.4 20 111 2.3 5.5 3.0 2.4 20 111 2.3 5.5 2.8 2.3 21 112 2.4 5.7 2.9 2.3 22 113 2.5 4.6 3.0 2.5 23 114 2.2 5.6 3.3 2.3 24 115 2.2 2.6 2.3 2.3 41 122 2.2 4.5 2.8 2.4					BARF	REL	
APRIL 6 97 - 6.3 3.6 2.6 10 101 2.5 5.9 - 2.6 12 103 2.4 5.9 2.5 3.2 13 104 2.4 5.9 - - 14 105 2.6 5.8 - - 18 109 2.3 5.5 3.0 2.4 20 111 2.3 5.7 2.9 2.3 21 112 2.4 5.7 3.0 2.5 22 113 2.5 4.6 3.0 2.5 23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 29 120 2.1 4.7 2.8 2.3 3 124 2.3 4.5 2.8 2.3 3 124 2.3 3.3 2.4 2.4	DATE		CALENDAR DAY	1	2	3	4
10 101 25 59 - 26 12 103 2.4 5.9 2.5 3.2 13 104 2.4 5.9 - - 14 105 2.6 5.8 - - 18 109 2.3 5.8 3.0 2.4 20 111 2.3 5.5 3.0 2.4 20 111 2.3 5.5 3.0 2.5 21 115 2.2 5.6 3.3 2.3 23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 27 118 2.2 2.6 6 3.3 2.3 21 122 2.2 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.3 3 125 2.4 4.6 6 127 2.2 4.5	APRIL	6	97	-	6.3	3.6	2.6
12 103 2.4 5.9 2.5 3.2 13 104 2.4 5.9 - - 14 105 2.6 5.8 - - 18 109 2.3 5.8 3.5 2.4 20 111 2.3 5.7 2.9 2.3 21 112 2.4 5.7 3.0 2.5 23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 27 118 2.2 5.1 2.8 2.2 29 120 2.1 4.7 2.8 2.3 3 124 2.3 4.5 2.8 2.3 3 124 2.3 3.7 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.2 2.8 2.4 11 <t< td=""><td></td><td>10</td><td>101</td><td>2.5</td><td>5.9</td><td>-</td><td>2.6</td></t<>		10	101	2.5	5.9	-	2.6
13 104 2.4 5.9 - - 14 105 2.6 5.8 - - 18 109 2.3 5.5 3.0 2.4 20 111 2.3 5.7 2.9 2.3 21 112 2.4 5.7 3.0 2.5 22 113 2.5 4.6 3.0 2.5 23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 24 115 2.2 2.6 3.3 2.3 27 118 2.2 2.6 3.3 2.3 23 124 2.3 4.5 2.8 2.3 4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.7 2.7 2.4 13 <		12	103	2.4	5.9	2.5	3.2
14 105 2.6 5.8 - - 18 109 2.3 5.8 3.5 2.4 20 111 2.3 5.7 2.9 2.3 21 112 2.4 5.7 3.0 2.5 22 113 2.5 4.6 3.0 2.5 23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 27 118 2.2 5.6 3.3 2.3 27 118 2.2 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.3 3 124 2.3 3.7 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.4 2.2 2.4 11 132 2.3 3.2 2.8 2.5 10		13	104	2.4	5.9	-	-
18 109 2.3 5.8 3.5 2.4 19 110 2.3 5.5 3.0 2.4 20 111 2.3 5.7 2.9 2.3 21 112 2.4 5.7 3.0 2.5 22 113 2.5 4.6 3.0 2.5 23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 29 120 2.1 4.7 2.8 2.3 3 124 2.3 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.3 10 131 2.3 4.4 2.2 2.4 4.6 11 132 2.3 3.7 2.7 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.7 2.4		14	105	2.6	5.8	-	-
19 110 2.3 5.5 3.0 2.4 20 111 2.3 5.7 2.9 2.3 21 112 2.4 5.7 3.0 2.5 22 113 2.5 4.6 3.0 2.5 23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 24 115 2.2 5.6 3.3 2.3 27 118 2.2 5.1 2.8 2.3 3 124 2.3 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.3 3 124 2.3 3.7 2.7 2.4 4 125 2.4 4.6 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.2 2.4 2.5 2.4		18	109	2.3	5.8	3.5	2.4
20 111 2.3 5.7 2.9 2.3 21 112 2.4 5.7 3.0 2.5 22 113 2.5 4.6 3.0 2.5 23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 27 118 2.2 5.1 2.8 2.3 29 120 2.1 4.7 2.8 2.3 3 124 2.3 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.4 4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.1 2.5 2.4 11 132 2.3 3.3 2.7 2.4 12 133 2.1 3.3 2.7 2.4 15		19	110	2.3	5.5	3.0	2.4
21 112 2.4 5.7 3.0 2.5 22 113 2.5 4.6 3.0 2.5 23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 29 120 2.1 4.7 2.8 2.3 29 120 2.1 4.5 2.7 2.1 2 123 2.2 4.5 2.8 2.3 3 142 2.3 4.5 2.8 2.3 4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 11 132 2.3 3.7 2.7 2.4 12 133 2.3 3.1 2.5 2.4 13 134 2.3 3.2 2.7 2.4 15 136 2.1 3.3 2.7 2.4 15		20	111	2.3	5.7	2.9	2.3
22 113 2.5 4.6 3.0 2.5 23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 27 118 2.2 5.1 2.8 2.2 29 120 2.1 4.7 2.8 2.3 3 124 2.3 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.3 4 125 2.4 2.6 2.3 2.4 4 122 2.3 3.7 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.1 2.5 2.4 12 133 2.3 3.1 2.5 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.6 2.7 2.3 15 136		21	112	2.4	5.7	3.0	2.5
23 114 2.3 5.5 2.8 2.3 24 115 2.2 5.6 3.3 2.3 27 118 2.2 5.1 2.8 2.3 29 120 2.1 4.7 2.8 2.3 MAY 1 122 2.2 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.4 4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.7 2.7 2.4 11 132 2.3 3.7 2.7 2.4 12 133 2.3 3.3 2.8 2.4 15 136 2.1 3.3 2.7 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4		22	113	2.5	4.6	3.0	2.5
24 115 2.2 5.6 3.3 2.3 27 118 2.2 5.1 2.8 2.3 29 120 2.1 4.7 2.8 2.3 3 122 2.2 4.5 2.7 2.1 2 123 2.2 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.3 4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.7 2.7 2.4 11 132 2.3 3.7 2.7 2.4 12 133 2.3 3.3 2.8 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.7 2.4 15 136 2.1 3.3 2.7 2.4 15		23	114	2.3	5.5	2.8	2.3
27 118 2.2 5.1 2.8 2.2 MAY 1 122 2.2 4.5 2.7 2.1 3 124 2.3 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.3 4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 4.4 2.2 2.4 11 132 2.3 3.7 2.7 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.7 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 18 139 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2		24	115	2.2	5.6	3.3	2.3
29 120 2.1 4.7 2.8 2.3 MAY 1 122 2.2 4.5 2.7 2.1 2 123 2.2 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.4 4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.7 2.7 2.4 11 132 2.3 3.7 2.7 2.4 12 133 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.7 2.4 15 136 2.1 3.3 2.7 2.4 15 136 2.1 3.3 2.7 2.4 15 136 2.1 3.3 2.7 2.4 15 138 2.1 3.3 2.7 2.4		27	118	2.2	5.1	2.8	2.2
MAY 1 122 2.2 4.5 2.7 2.1 2 123 2.2 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.4 4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.7 2.7 2.4 12 133 2.3 3.7 2.7 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.8 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 18 139 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2		29	120	2.1	4.7	2.8	2.3
2 123 2.2 4.5 2.8 2.3 3 124 2.3 4.5 2.8 2.4 4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 4.4 2.2 2.4 11 132 2.3 3.7 2.7 2.4 12 133 2.3 4.1 2.5 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.7 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 18 139 2.3 3.2 2.7 2.4 18 139 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.	MAY	1	122	2.2	4.5	2.7	2.1
3 124 2.3 4.5 2.8 2.4 4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 3.7 2.7 2.4 11 132 2.3 3.7 2.7 2.4 12 133 2.3 4.1 2.5 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.7 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 17 138 2.1 3.3 2.5 2.5 19 140 2.3 3.3 2.5 2.5 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 23 144 2.5 3.3 3.0 2.7 24 145 2		2	123	2.2	4.5	2.8	2.3
4 125 2.4 2.5 2.4 4.6 6 127 2.2 4.2 2.8 2.3 10 131 2.3 4.4 2.2 2.4 11 132 2.3 3.7 2.7 2.4 12 133 2.3 4.1 2.5 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.8 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 17 138 2.1 3.3 2.7 2.4 18 139 2.3 3.2 2.7 2.4 18 139 2.3 3.2 2.7 2.4 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144		3	124	2.3	4.5	2.8	2.4
6 127 2.2 4.2 2.8 2.3 10 131 2.3 4.4 2.2 2.4 11 132 2.3 3.7 2.7 2.4 12 133 2.3 4.1 2.5 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.7 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 17 138 2.1 3.3 2.7 2.4 18 139 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 23 144 2.5 3.3 3.0 2.7 23		4	125	2.4	.2.5	2.4	4.6
10 131 2.3 4.4 2.2 2.4 11 132 2.3 3.7 2.7 2.4 12 133 2.3 4.1 2.5 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.8 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 17 138 2.1 3.3 2.7 2.4 18 139 2.3 3.3 2.5 2.5 19 140 2.3 3.2 2.7 2.4 18 139 2.3 3.2 2.7 2.4 19 140 2.3 3.2 2.7 2.4 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 <t< td=""><td></td><td>6</td><td>127</td><td>2.2</td><td>4.2</td><td>2.8</td><td>2.3</td></t<>		6	127	2.2	4.2	2.8	2.3
11 132 2.3 3.7 2.7 2.4 12 133 2.3 4.1 2.5 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.8 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 17 138 2.1 3.3 2.7 2.4 18 139 2.3 3.3 2.5 2.5 19 140 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.7 23 144 2.5 3.3 3.0 2.7 24 145 <t< td=""><td></td><td>10</td><td>131</td><td>2.3</td><td>4.4</td><td>2.2</td><td>2.4</td></t<>		10	131	2.3	4.4	2.2	2.4
12 133 2.3 4.1 2.5 2.4 13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.8 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 17 138 2.1 3.3 2.7 2.4 18 139 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 24 145 2.2 2.9 2.6 2.2 25 146 2.3 3.1 2.8 2.4 JUNE 30 <		11	132	2.3	3.7	2.7	2.4
13 134 2.3 3.2 2.8 2.5 14 135 2.3 3.3 2.8 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 17 138 2.1 3.3 2.7 2.4 18 139 2.3 3.3 2.5 2.5 19 140 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 24 145 2.2 2.9 2.6 2.2 25 146 2.3 3.1 2.8 2.4 JUNE 30 <		12	133	2.3	4.1	2.5	2.4
14 135 2.3 3.3 2.8 2.4 15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 17 138 2.1 3.3 2.7 2.4 18 139 2.3 3.3 2.5 2.5 19 140 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 24 145 2.2 2.9 2.6 2.2 25 146 2.3 3.1 2.8 2.4 JUNE <td></td> <td>13</td> <td>134</td> <td>2.3</td> <td>3.2</td> <td>2.8</td> <td>2.5</td>		13	134	2.3	3.2	2.8	2.5
15 136 2.1 3.3 2.7 2.4 16 137 2.0 3.3 2.7 2.4 17 138 2.1 3.3 2.7 2.4 18 139 2.3 3.3 2.5 2.5 19 140 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 24 145 2.2 2.9 2.6 2.2 25 146 2.3 3.1 2.8 2.4 JUNE 30 182 2.6 - 2.7 2.8		14	135	2.3	3.3	2.8	2.4
16 137 2.0 3.3 2.7 2.4 17 138 2.1 3.3 2.7 2.4 18 139 2.3 3.3 2.5 2.5 19 140 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 24 145 2.2 2.9 2.6 2.2 25 146 2.3 3.1 2.8 2.4 JUNE 30 182 2.6 - 2.9 2.8 JULY 4 186 2.4 3.3 2.8 2.7 6 188 2.5 4.5 2.7 2.8 <td< td=""><td></td><td>15</td><td>136</td><td>2.1</td><td>3.3</td><td>2.7</td><td>2.4</td></td<>		15	136	2.1	3.3	2.7	2.4
17 138 2.1 3.3 2.7 2.4 18 139 2.3 3.3 2.5 2.5 19 140 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 24 145 2.2 2.9 2.6 2.2 25 146 2.3 3.1 2.8 2.4 JUNE 30 182 2.6 - 2.9 2.8 JULY 4 186 2.4 3.3 2.8 2.7 6 188 2.5 4.5 2.7 2.8 7 189 2.6 4.4 2.8 2.7 8 190 2.6 4.3 2.8 2.8 9		16	137	2.0	3.3	2.7	2.4
18 139 2.3 3.3 2.5 2.5 19 140 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 24 145 2.2 2.9 2.6 2.2 25 146 2.3 3.1 2.8 2.4 JUNE 30 182 2.6 - 2.9 2.8 JULY 4 186 2.4 3.3 2.8 2.7 6 188 2.5 4.5 2.7 2.8 7 189 2.6 4.4 2.8 2.7 8 190 2.6 4.3 2.8 2.8 9 191 2.7 4.6 2.8 2.8 10		17	138	2.1	3.3	2.7	2.4
19 140 2.3 3.2 2.7 2.3 20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 24 145 2.2 2.9 2.6 2.2 25 146 2.3 3.1 2.8 2.4 JUNE 30 182 2.6 - 2.9 2.8 JULY 4 186 2.4 3.3 2.8 2.7 6 188 2.5 4.5 2.7 2.8 7 189 2.6 4.4 2.8 2.7 8 190 2.6 4.3 2.8 2.8 9 191 2.7 4.6 2.8 2.8		18	139	2.3	3.3	2.5	2.5
20 141 2.1 3.0 2.6 2.2 21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 24 145 2.2 2.9 2.6 2.2 25 146 2.3 3.1 2.8 2.4 JUNE 30 182 2.6 - 2.9 2.8 JULY 4 186 2.4 3.3 2.8 2.7 6 188 2.5 4.5 2.7 2.8 7 189 2.6 4.4 2.8 2.7 8 190 2.6 4.3 2.8 2.8 9 191 2.7 4.6 2.8 2.8 10 192 2.6 - - - 11 193 2.6 4.6 2.8 2.8 <td></td> <td>19</td> <td>140</td> <td>2.3</td> <td>3.2</td> <td>2.7</td> <td>2.3</td>		19	140	2.3	3.2	2.7	2.3
21 142 2.3 3.1 2.6 2.2 22 143 2.5 3.3 3.0 2.7 23 144 2.5 3.3 3.0 2.7 24 145 2.2 2.9 2.6 2.2 25 146 2.3 3.1 2.8 2.4 JUNE 30 182 2.6 - 2.9 2.8 JULY 4 186 2.4 3.3 2.8 2.7 6 188 2.5 4.5 2.7 2.8 7 189 2.6 4.4 2.8 2.7 8 190 2.6 4.3 2.8 2.7 8 190 2.6 4.3 2.8 2.8 9 191 2.7 4.6 2.8 2.8 9 191 2.7 4.6 2.8 2.8 10 192 2.6 - - - 11 193 2.6 4.6 2.8 2.8		20	141	2.1	3.0	2.6	2.2
221432.53.33.02.7231442.53.33.02.7241452.22.92.62.2251462.33.12.82.4JUNE301822.6-2.92.8JULY41862.43.32.82.761882.54.52.72.871892.64.42.82.781902.64.32.82.891912.74.62.82.8101922.6111932.64.62.82.8		21	142	2.3	3.1	2.6	2.2
231442.53.33.02.7241452.22.92.62.2251462.33.12.82.4JUNE301822.6-2.92.8JULY41862.43.32.82.761882.54.52.72.871892.64.42.82.781902.64.32.82.891912.74.62.82.8101922.6111932.64.62.82.8		22	143	2.5	3.3	3.0	2.7
241452.22.92.62.2251462.33.12.82.4JUNE301822.6-2.92.8JULY41862.43.32.82.761882.54.52.72.871892.64.42.82.781902.64.32.82.891912.74.62.82.8101922.6111932.64.62.82.8		23	144	2.5	3.3	3.0	2.7
25 146 2.3 3.1 2.8 2.4 JUNE 30 182 2.6 - 2.9 2.8 JULY 4 186 2.4 3.3 2.8 2.7 6 188 2.5 4.5 2.7 2.8 7 189 2.6 4.4 2.8 2.7 8 190 2.6 4.3 2.8 2.8 9 191 2.7 4.6 2.8 2.8 10 192 2.6 - - - 11 193 2.6 4.6 2.8 2.8		24	145	2.2	2.9	2.6	2.2
JUNE 30 182 2.6 - 2.9 2.8 JULY 4 186 2.4 3.3 2.8 2.7 6 188 2.5 4.5 2.7 2.8 7 189 2.6 4.4 2.8 2.7 8 190 2.6 4.3 2.8 2.8 9 191 2.7 4.6 2.8 2.8 10 192 2.6 - - - 11 193 2.6 4.6 2.8 2.8		25	146	2.3	3.1	2.8	2.4
JULY 4 186 2.4 3.3 2.8 2.7 6 188 2.5 4.5 2.7 2.8 7 189 2.6 4.4 2.8 2.7 8 190 2.6 4.3 2.8 2.8 9 191 2.7 4.6 2.8 2.8 10 192 2.6 - - - 11 193 2.6 4.6 2.8 2.8	JUNE	30	182	2.6	-	2.9	2.8
61882.54.52.72.871892.64.42.82.781902.64.32.82.891912.74.62.82.8101922.6111932.64.62.82.8	JULY	4	186	2.4	3.3	2.8	2.7
71892.64.42.82.781902.64.32.82.891912.74.62.82.8101922.6111932.64.62.82.8		6	188	2.5	4.5	2.7	2.8
81902.64.32.82.891912.74.62.82.8101922.6111932.64.62.82.8		7	189	2.6	4.4	2.8	2.7
91912.74.62.82.8101922.6111932.64.62.82.8		8	190	2.6	4.3	2.8	2.8
101922.6111932.64.62.82.8		9	191	2.7	4.6	2.8	2.8
11 193 2.6 4.6 2.8 2.8		10	192	2.6	-	-	-
		11	193	2.6	4.6	2.8	2.8
12 194 2.7 4.6 2.8 2.7		12	194	2.7	4.6	2.8	2.7

Measurements of pH in Water Taken from the Collection Buckets of the Limestone Barrels by Norecol (1988) for the Cinola Gold Project

TABLE 2

Water Quality Results from On-site Limestone Barrels for Barrel 1 by Norecol For The Cinola Gold Project

Week 5 6 11 pH 2.5 2.2 2.2 Alkalinity mg CaCO ₃ /L - - Turbidity NTU - 1.3 1.3 Conductance umhos/cm - 4530 8570 63 Total Solids mg/L - 8753 31140 5400 5400 17800 100 Suspended Solids mg/L - 25 16 16 16 16 17 10 12 10 </th <th>15 2.1 0.5 310 - 2 740 0.7 300</th>	15 2.1 0.5 310 - 2 740 0.7 300
pH 2.5 2.2 2.2 Alkalinity mg CaCO ₃ /L - - - Turbidity NTU - 1.3 1.3 Conductance umhos/cm - 4530 8570 63 Total Solids mg/L - 8753 31140 Suspended Solids mg/L - 25 16 EDTA-Hardness mg CaCO ₃ /L - - - Color APHA - - 11 Fluoride mg/L - 0.1 0.12	2.1 0.5 310 - 2 740 0.7 300
Alkalinity mg CaCO ₃ /L - - - Turbidity NTU - 1.3 1.3 Conductance umhos/cm - 4530 8570 63 Total Solids mg/L - 8753 31140 Suspended Solids mg/L - 25 16 EDTA-Hardness mg CaCO ₃ /L - - - Color APHA - - 17 Fluoride mg/L - 0.1 0.12 Sulfate mg/L - 5400 17800 100	0.5 310 2 740 0.7 300
Turbidity NTU - 1.3 1.3 Conductance umhos/cm - 4530 8570 63 Total Solids mg/L - 8753 31140 31140 Suspended Solids mg/L - 25 16 16 EDTA-Hardness mg CaCO ₃ /L - - - 17 Color APHA - - 17 10 12 Sulfate mg/L - 5400 17800 100	0.5 310 - 2 740 0.7 300
Conductance umhos/cm - 4530 8570 63 Total Solids mg/L - 8753 31140 31	310 2 740 0.7 300
Total Solids mg/L - 8753 31140 Suspended Solids mg/L - 25 16 EDTA-Hardness mg CaCO ₃ /L - - - Color APHA - - 1 Fluoride mg/L - 0.1 0.12 Sulfate mg/L - 5400 17800 100	2 740 0.7 300
Suspended Solids mg/L - 25 16 EDTA-Hardness mg CaCO ₃ /L - - - - Color APHA - - - 1 Fluoride mg/L - 0.1 0.12 Sulfate mg/L - 5400 17800 100	2 740 0.7 300
EDTA-Hardness mg CaCO ₃ /L - - - - - - 1 Color APHA - - - 1 <t< td=""><td>- 740 0.7 300</td></t<>	- 740 0.7 300
Color APHA - - 1 Fluoride mg/L - 0.1 0.12 Sulfate mg/L - 5400 17800 100	740 0.7 300
Fluoride mg/L - 0.1 0.12 Sulfate mg/L - 5400 17800 100	0.7 300 -
Sulfate mg/l - 5400 17800 100	500
	-
Ammonia mg N/L 0.28	
Nitrate mg N/L 0.050	-
Nitrite mg N/L - 0.003	< 0.002
Total Phosphorus mg P/L - 28.5 180	140
Total Dissolved Phosphorus mg P/L - 28.0 178	123
Dissolved Metals	
Ag mg/L - 0.0002 -	-
Al mg/L 60 111 326	128
As mg/L 9.0 29 210	31
Ba mg/L - 0.060 -	-
Ca mg/L 113 168 375	225
Cd mg/L - 0.015 0.06	-
Co mg/L - 2.36 6.9	2.50
Cr mg/L - 0.23 0.87	-
Cu mg/L 2.46 3.88 8.4	4.48
Fe mg/L 878 1650 5540 2-	425
K mg/L 20	-
Mg mg/L 14.8	-
Mn mg/L 5.0 9.2 20	7.0
Mo mg/L - 0.012 -	-
Na mg/L 5.9	-
Ni mg/L - 1.59 4.38	1.66
Pb mg/L ~ 0.017 0.006	-
Sb mg/L 0.038 0.075 0.21	-
Se mg/L - 0.002 0.036	-
Si mg/L 38	-
Zn mg/L 14.2 23 72	26

TABLE 3

Water Quality Results From On-site Limestone Barrels For Barrel 2 By Norecol For The Cinola Gold Project

PARAMETER	UNITS	MAR 22/88	APR 10/88	APR 21/88	MAY 27/88	JUNE 23/88
Week		1	5	6	11	15
pН		7.3	5. 9	6.6	3.3	-
Alkalinity	mg CaCO ₃ /	L 76	-	16	-	-
Turbidity	NTU	96	-	50	5.3	-
Conductance	umhos/cm	1430	-	1550	2420	-
Total Solids	mg/L	1612	-	1559	2560	-
Suspended Solids	mg/L	167	-	73	7	-
EDTA-Hardness	mg CaCO ₃ /	L 8	-	-	-	-
Color	APHA	5	-	-	-	-
Fluoride	mg/L	0.25	-	0.12	0.35	-
Sulfate	mg/L	858	-	925	1719	-
Ammonia	mg N/L	1.13	-	0.408	0.14	-
Nitrate	mg N/L	0.243	-	1.34	0.108	-
Nitrite	mg N/L	0.009	-	0.33	< 0.002	-
Total Phosphorus	mg P/L	0.395	-	0.312	0.086	-
Total Dissolved Phosphorus	mg P/L	0.020	-	0.150	0.065	-
Dissolved Metals					,	
Ag	mg/L	0.0002	-	0.0002	-	-
AI	mg/L	0.22	1.4	0.020	38	-
As	mg/L	0.001	< 0.001	0.028	3.0	-
Ва	mg/L	0.050	-	0.074	-	-
Са	mg/L	-	238	370	525	450
Cd	mg/L	0.0006	-	0.0010	0.013	-
Со	mg/L	0.09	-	0.20	0.42	0.14
Cr	mg/L	0.001	-	0.003	0.040	-
Си	mg/L	0.0034	0.04	0.010	0.38	-
Fe	mg/L	0.15	1.09	0.04	47	0.59
к	mg/L	-	6.8	-	-	-
Mg	mg/L	-	15.0	-	-	-
Mn	mg/L	1.49	1.13	2.51	4.3	1.00
Мо	mg/L	0.005	-	0.005	-	-
Na	mg/L	-	6.3	-	-	-
Ni	ma/L	0.11	•	0.13	0.26	0.10
Pb	mg/L	0.001	-	0.001	0.001	-
Sb	mg/L	0.008	0.005	0.009	0.010	-
Se	mg/L	0.001	•	0.001	< 0.001	-
Si	mg/L	•	4	-		-
Zn	mg/L	0.31	0.37	0.68	3.1	0.33

.

TABLE 4

Water Quality Results from On-site Limestone Barrels for Barrel 3 by Norecol for the Cinola Gold Project

PARAMETER	UNITS	APR 21/88	MAY 27/88	JUNE 23/88
WEEK		6	11	15
На		2.9	2.9	2.7
Alkalinity	mg CaCO ₃ /L	-	•	-
Turbidity	NTU	6.1	2.4	1.0
Conductance	umhos/cm	3700	3820	2910
Total Solids	mg/L	4168	5260	-
Suspended Solids	mg/L	49	4	3
EDTA-Hardness	mg CaCO ₃ /L	-	-	-
Color	APHA	•	-	180
Fluoride	mg/L	0.1	0.14	1.15
Sulfate	mg/L	2680	3188	2920
Ammonia	mg N/L	0.192	< 0.1	-
Nitrate	mg N/L	•	0.024	-
Nitrite	mg N/L	-	< 0.002	< 0.002
Total Phosphorus	mg P/L	0.238	5.0	3.7
Total Dissolved Phosphorus	mg P/L	0.219	3.8	3.4
Dissolved Metals				
Ag	ma/L	0.0002	-	-
AI	ma/L	34	93	49
As	ma/L	0.14	7.0	-
Ba	ma/L	0.011	-	-
Ca	ma/L	825	625	425
Cd	ma/L	0.011	0.03	-
Co	ma/L	2.59	4.3	1.65
Cr	ma/L	0.055	0.19	-
Cu	ma/L	0.89	2.88	1.50
Fe	ma/L	101	390	225
Mn	ma/L	13.8	17.0	6.8
Mo	ma/L	0.005	-	-
Ni	ma/L	2.01	3.16	1.19
Ph	ma/L	0.004	0.003	-
Sh	ma/L	0.017	0.028	-
Se	ma/L	0.001	0.042	-
Zn	mg/L	16.5	38	16

-

Ű.

TABLE 5

Water Quality Results from On-site Limestone Barrels for Barrel 4 by Norecol for the Cinola Gold Project

PARAMETER	UNITS	APR 10/88	APR 21/88	MAY 27/88	JUNE 23/88
WEEK		5	6	11	15
рH		2.6	2.4	2.4	2.3
Alkalinity	mg CaCO ₃ /	L -	-	-	-
Turbidity	NŤU	-	1.2	0.6	0.8
Conductance	umhos/cm	-	4050	5380	4420
Total Solids	mg/L	•	6443	12990	-
Suspended Solids	mg/L	-	20	6	5
EDTA-Hardness	mg CaCO ₃ /	L -	-	-	-
Color	APHA	-	-	-	965
Fluoride	mg/L	-	0.1	0.11	2.20
Sulfate	mg/L	-	3960	7500	6000
Ammonia	mg N/L	-	-	< 0.1	-
Nitrate	mg N/L	-	-	0.022	-
Nitrite	mg N/L	-	-	< 0.002	< 0.002
Total Phosphorus	mg P/L	-	13.6	56	47
Total Dissolved Phosphorus	mg P/L	-	13.6	55	47
Dissolved Metals					
Ag	mg/L	-	0.0002	-	- ·
AI	mg/L	64	117	177	55
As	mg/L	2.7	8.0	29	8
Ва	mg/L	-	0.048	-	-
Са	mg/L	190	235	300	125
Cd	mg/L	-	0.009	0.03	-
Со	mg/L	-	1.99	3.2	0.97
Cr	mg/L	-	0.24	0.62	-
Cu	mg/L	1.92	2.82	4.4	1.24
Fe	mg/L	568	1025	2060	800
к	mg/L	18	-	-	-
Mg	mg/L	17.5	-	-	-
Mn	mg/L	5.2	10.0	11.2	3.1
Мо	mg/L	-	0.011	-	-
Na	mg/L	7.2	-	-	-
Ni	mg/L	-	1.04	2.06	0.61
Pb	mg/L	-	0.011	0.004	-
Sb	mg/L	0.021	0.021	0.07	-
Se	mg/L	0.002	< 0.001	-	
Si	mg/L	40	-	-	-
Zn	mg/L	12.5	18.3	33	9.5

APPENDIX 7.0

APPENDIX 7.2

EXPERIMENTAL PROCEDURES AND ANALYTICAL RESULTS FOR BENCH SCALE TEST FOR MINE SITE WATER TREATMENT FACILITY -ANALYSIS PERFORMED ON ACIDIC WASTEWATER FROM THE CINOLA GOLD PROJECT

DESCRIPTION OF WATER SAMPLES:

Acid water generated by Cinola Gold Project waste rock was obtained by three methods. The first set of samples were obtained on site from an underdrain splitter, the second set were produced from leachate samples, and the third set were gathered after direct rinsing of argillically altered waste rock from the mine site.

Waste Rock Drainage Sample

a. Experimental Procedures

The initial analyses utilized the acidic drainage collected from the on site waste rock test pad. A supersaturated solution of lime (10% w/w Ca(OH)2) was well mixed before addition to a one litre aliquot of the waste drainage sample. After initial pH measurements, the solution was well mixed and the lime slurry was added until the desired pH's values of 7.5, 8.5, 9.5, and 10.5 were obtained, and the volume of 10% lime slurry required was recorded. The formation of golden, orange flocs occurred immediately upon addition of the 10% lime slurry. The flocs were allowed to settle for approximately one hour and the volume of precipitate was recorded before decanting the supernatant by siphoning. A portion of the supernatant was filtered through a 0.45um millipore filter and this filtrate was analyzed for metals and inorganics along with the supernatant. The filtrate/dissolved concentrations were assessed to compare to the concentrations remaining in the supernatant (suspended) phase. A measured volume of the precipitate was filtered through dried, pre-weighed Whatman GF/C filter paper placed on top of a 0.45 um millipore filter. The precipitate and filter were weighed wet and then dried in a 106° C oven and reweighed. The percent moisture was then calculated. The dried precipitates from a pH 7.5 sample and a pH 9.5 sample were analyzed for metals concentrations.

b. Analytical Results

Tables 7.2-3 and 7.2-4 of the Stage II Addendum Report summarize the analytical results of the supernatant (total) and the filtrate (dissolved) components from the lime treatment and settling tests on the waste rock drainage sample. Obtaining a constant pH reading while the solution was being stirred presented a difficulty. There was also inconsistency in the volume of 10% lime slurry required to raise the pH to a desired level. After settling, the supernatants appeared cloudy and yellowish with some suspended flocs visible. Hard white granules were observed in the bottom of the precipitate. These are thought to have been calcium sulphate.

At a final pH of 9.5, the majority of the metals of concern (chromium, copper mercury, zinc) was effectively removed to approximately 95% efficiency. An exception to this removal effectiveness at a pH of 9.5 was aluminum, which tends to resolubilize at the higher pH's due to its amphoteric nature.

The lime dosages required to achieve the reported pH's, the volume of the precipitate measured after settling, and the percent moisture of two precipitate samples are tabulated below:

pН	g Ca(OH) ₂ /Liter H ₂ 0	Volume of Precipitate	% Moisture
7.5	1.6 g/L	140 ml	86
8.5	1.7 g/L	120 ml	
9.5	1.8 g/L	115 ml	83
10.5	1.9 g/L	120 ml	

The concentrations of metals reported for the two precipitate samples are reported in Table 7.3.11-11 of Vol. II of the Stage II Report.

Leachate Samples

a. Experimental Procedure

Ten limestone column leachate samples were obtained from Norecol Environmental Consultants Ltd., Vancouver, B.C. on May 28, 1988. The columns were established in January 1988 and consisted of 92% Skonun sediments and 8% Breccia. A 20% (w/w) lime slurry was used instead of the 10% lime slurry used for the underdrain splitter samples test. The procedure used for the leachate sample was similar to that used for the previous tests, however, the pH was increased to 9.5 in all of the tests run on this water. The lime slurry was added in two ways. For one test group, the 20% lime slurry was added in the same fashion as used in the previous tests. In the second test group, an aliquot of the 20% lime slurry was added directly to the neutralized precipitate of the preceding run which was then well mixed with an un-neutralized one litre sample. The flocs were allowed to settle for approximately two hours, and the volume of the precipitate in the graduated cylinder recorded. The supernatant was then decanted by siphoning and retained for analysis. No filtrate samples were analyzed using these water samples.

The precipitates from two consecutively neutralized samples were consolidated. This procedure was then repeated with three consecutively neutralized samples. The neutralized samples were allowed to set between two and twenty hours before compositing. The volume of precipitate was recorded after allowing time for settling and compaction. An intermediate supernatant and a final supernatant were analyzed for concentrations of selected metals and inorganics. The precipitate densities and percent moistures were also calculated.

b. Analytical Results

The calculated concentrations of metals and inorganics for the intermediate and final supernatant samples are reported in Table 7.2-5. The following is a description of the treatment for each of the cycles:

Cycle 1: precipitate from two consecutively neutralized samples allowed to settle for two hours.

Cycle 2: precipitate from four consecutively neutralized samples allowed to settle for two hours.

Cycle 3: 20% lime slurry was added to the precipitate of a neutralized sample which was then mixed with an un-neutralized sample and allowed to settle for two hours.

Cycle 4: 20% lime slurry was added to the neutralized consolidated precipitates of two previously neutralized samples which were then added to an un-neutralized sample and allowed to settle 17 hours.

The leachate samples required greater dosages of lime compared to the waste rock drainage sample due to their higher acidity. A golden, orange floc formed immediately upon addition of the 20% lime slurry. It was noted that less lime was required to neutralize the sample when using a fresh batch of 20% lime slurry. Larger volumes of precipitate were formed from these samples than were formed in the waste rock drainage sample. The presence of white granules in the bottom of the precipitate layer, probably calcium sulphate, was observed. A fluffy tan layer, probably ferric hydroxide, on the very top of the precipitate layer was noted for some of the samples. The supernatants appeared yellowish with some visible suspended flocs. The precipitate percent moisture was slightly lower for these samples than was calculated for the waste rock drainage samples. The mass of lime required to reach pH 9.5 was estimated based on four experimental runs. The precipitate volume, percent moisture, and density are summarized below:

рН 	g Ca(OH) ₂ /L H ₂ 0	Precipitate Volume	Moisture (%)	Density
9.5	4.2	178 mL	78	0.08g/cm ³

Argillically Altered Waste Rock Rinse Samples

a) Experimental Procedure

Three buckets of argillically altered waste rock from the mine site were provided to SRK (Pacific). The material was broken up to fragments less than two inches in diameter and homogenized. Deionized water was poured onto the waste rock, mixed, and left to set for

approximately twenty-four hours. The acidic water was collected and the preceding process repeated. After initial pH measurements, a one litre aliquot of the solution was neutralized to pH 9.5. A 0.5 M sodium hydroxide solution was initially used as the neutralizing agent, while subsequent runs utilized the 20% lime slurry. The sample was well mixed and the precipitate allowed to settle for a minimum of two hours. The supernatant was then siphoned off and a portion of this filtered through a 0.45um millipore filter. The initial solution, the supernatant, and the filtrate for each argillically altered waste rock sample were analyzed for metals and inorganics.

b) Analytical Results

Table 7.2-6 shows the concentration of metals and inorganics after treatment with the 0.5 M sodium hydroxide. The reaction of the rinse samples upon addition of neutralizing agent was visibly different from the observed reactions of the other two sample sets. A dark, blue-green globular precipitate formed as compared to the golden-orange flocs that were observed in the previous analyses. Once the samples were mixed with the magnetic stirrer, the globules broke up into smaller flocs giving the solution an opaque greenish black appearance. It was also observed that the precipitate formed using the sodium hydroxide took longer to settle than the samples neutralized with the 20% lime slurry. Sulphate remained in solution when the sample was neutralized with sodium hydroxide (see Table 7.2-6) maintaining a high conductivity in the sample solution. Sulphate was removed when using the 20% lime slurry as the neutralizer (see Tables 7.2-7 through 7.2-9). The sulphate was precipitated out as calcium sulphate, which appeared as hard, white granules at the bottom of the precipitate.

APPENDIX 8.3-1

RECEIVING WATER QUALITY PREDICTIONS FOR EXTREME FLOW CONDITIONS USING MAY 1988 PILOT MILL AND BENCH SCALE MINE SITE WATER TREATMENT PLANT RESULTS

(Pages 1 to 2)

APPENDIX 8.3-1 CONTENTS

Table Page 1 Water Quality of Barbie Creek After Receiving Effluents from the Lime Treatment Facility and Waste Rock Stockpile Underdrain, Extreme Conditions, Predicted by Norecol for the Cinola Gold Project 1 2 Water Quality of Barbie Creek after Mixing with All Mine Site Discharge, Extreme Conditions, Predicted by Norecol for the Cinola Gold Project 1

APPENDIX 8.3-1

TABLE 1

Water Quality of Barbie Creek After Receiving Effluents from the Lime Treatment Facility and Waste Rock Stockpile Underdrain, Extreme Conditions, Predicted by Norecol for the Cinola Gold Project

				CON	ICENTRATIONS I	N RECEIVING WA	TER
				YEA	AR 7	YEA	R 12
PARAMETER	UNITS	BARBIE CREEK BACKGROUND CONCENTRATION	MOE RECEIVING WATER CRITERIA ^a	10 YEAR DRY JUN	10 YEAR WET OCT	10 YEAR DRY JUN	10 YEAR WET OCT
Lower Barbie at Branch 40A Flow Rate Clean Water Flow Rate Lime Treatment Discharge Rate ^b WRS Underdrain Discharge Rate ^c	m ³ /s m ³ /s m ³ /s m ³ /s			0.020 0.000 0.0094 0.0024	0.63 0.0008 0.048 0.013	0.020 0.000 0.016 0.0024	0.63 0.0008 0.051 0.013
Hardness Sulphate Total Phosphorus Orthophosphate	mg CaCO ₃ /L mg/L mg P/L mg P/L	12 3 0.036 0.0005	_d 1000 ^e _d _d	626 536 0.028 0.0020	157 129 0.034 0.0009	868 749 0.025 0.0025	166 136 0.034 0.0009
Total Metals	-						
Al ^f As Cd Co Cr Cu Fe Hg Mn Ni Pb Zn	mg/L mg/L mg/L mg/L mg/L ug/L mg/L mg/L mg/L mg/L	0.36 0.003 0.0001 0.0005 0.001 0.0009 1.65 0.03 0.21 0.001 0.001 0.001	0.01 0.05 0.0002 0.05 0.002 0.3 0.1 0.05 0.025 0.003 0.03	0.38 0.008 0.0002 0.010 0.0012 0.0074 1.93 0.08 0.23 0.010 0.001 0.001	0.36 0.004 0.0001 0.003 0.0007 0.0024 1.72 0.04 0.21 0.003 0.001 0.0053	0.39 0.010 0.0002 0.013 0.0015 0.0095 2.11 0.10 0.24 0.013 0.001 0.015	0.36 0.004 0.0001 0.003 0.0007 0.0025 1.72 0.04 0.21 0.003 0.001 0.0054

а

ь

For protection of aquatic life. Water quality of this effluent is given in Table 7.3-10. Water quality of this effluent is given in Vol IV, Table 2.2.2-9. No applicable criterion. Criterion for livestock watering. Dissolved Al concentration, criterion for median pH 5.7. с

d

е f

APPENDIX 8.3-1

TABLE 2

Water Quality of Barbie Creek After Mixing With All Mine Site Discharges, Extreme Conditions, Predicted by Norecol for the Cinola Gold Project

					CONCENTRA	TIONS IN RECEI	VING WATER
				YEA	AR 7	YEA	R 12
PARAMETER	UNITS	BARBIE CREEK BACKGROUND CONCENTRATION	MOE RECEIVING WATER CRITERION ^a	10 YEAR DRY JUN	10 YEAR WET OCT	10 YEAR DRY JUN	10 YEAR WET OCT
Lower Barbie at Branch 40A Flow Rate	m ³ /s			0.019	0.57	0.019	0.57
Hardness Sulphate	mg CaCO ₃ /L mg/L	12 3	_b 1000 ^c	533 454	137 110	601 515	141 113
Total Metals							
As Cu Fe Hg Zn	mg/L mg/L mg/L ug/L mg/L	0.003 0.0009 1.65 0.025 0.0031	0.05 0.002 0.3 0.1 0.03	0.008 0.0068 1.75 0.08 0.012	0.005 0.0027 1.62 0.05 0.0072	0.010 0.0073 1.69 0.09 0.020	0.005 0.0029 1.61 0.05 0.0085

For protection of aquatic life. No applicable criterion. а

ь

^c Criterion for livestock watering.