Appendix "C"

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THE CINOLA PROJECT,

PROCESS AND ENVIRONMENT

by

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City Resources (Canada) Limited

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January, 1939

SUMMARY

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A practical application of nitric acid oxidation of sulphides, based on the principles of the Arseno process, has been developed for the Cinola ore. Development of the process has been based on testwork conducted by Bacon Donaldson of Vancouver and Hazen Research of Denver, and on conceptual engineering performed by Minproc Engineers of Denver.

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A comprehensive Stage II report for the project has been submitted to the B.C. permitting agencies, containing provisions for a zero discharge tailings impoundment system and prevention of acid mine drainage. Approval in principle in expected in early 1989.

A feasibility study recently completed by Minproc Engineers and Davy McKee Corporation demonstrates that this project can be brought to production for a capital expenditure of C\$120 million and operated at a total direct operating cost of less than \$20/tonne.

INTRODUCTION

The Cinola orebody was discovered in 1970. It is situated approximately in the centre of Graham Island, the largest of the Queen Charlotte Islands. It represents the mid to upper levels of a hydrothermal precious metals system, characterized by intense brecciation and silicification. Gold is associated with finely disseminated sulphides in a breccia matrix and in free form in numerous quartz veins.

Ownership of the property has changed hands a number of times, and previous attempts to bring the property to production have failed. Two critical factors have frequently been blamed for difficulty in creating a successful project; namely process and environment factors, hence the title of this paper.

City Resources (Canada) Limited ("City") acquired the property in 1986, and commenced an aggressive campaign to verify ore reserves, to carry out metallurgical research and to address environmental issues.

PROJECT OVERVIEW

When the Cinola orebody was discovered, the significance of hydrothermal mineralization associated with crustal plate boundaries was not recognized, and for several years, the orebody was considered an uneconomic curiosity. If you were now to write a specification of the type and location of orebody sought by many companies engaged in gold exploration, you would be writing a description of the Cinola orebody - it is a large (greater than 1.5 million ounce), low grade (2.45 gm/tonne), refractory ore reserve associated with the Pacific rim crustal plate boundary. The industry appears to have recognized in the past few years that its future lies in the exploitation of such orebodies, and is prepared to come to grips with the technical problems associated with such ores. In this sense, the Cinola orebody might be considered to have come of age.

One aspect on which there has been little controversy over the years is the existence of the reserve. Numerous reserve estimates have been prepared by various methods, resulting in estimates of mineable reserves in the range of 22 to 23 million tonnes at a grade of 2.2 to 2.45 gm/tonne. The current estimate used by City in its economic analysis of the project was prepared by Davy McKee Corporation, who concluded, after detailed consideration of selectivity of mining, internal waste and dilution, that a mineable reserve of 23.8 million tonnes at a grade of 2.45 gm/tonne exists at a cut off grade of 1.10 gm/tonne. However, there is a considerable tonnage of material only slightly below cut off grade, and small changes in economics could change that number. The orebody is shaped approximately like an inverted wedge, lying in the hanging wall of the Specogna fault, and not only lends itself to relatively low cost open pit mining methods, but also to selective mining of higher grade material during the early years of production, while lower grade material can be stockpiled. The overall stripping ratio is 2.45:1, but is much lower in the early years. The orebody has only been drilled to 300 metres below surface, where it narrows and increases in grade substantially, leading to the possibility of future underground mining potential. A lateral extension was also drilled last year, but the results have not been incorporated in the reserve estimate.

The orebody is situated near the Yakoun River, a major salmon spawning area, which is the reason for many environmental concerns about the project. Part of City's environmental mitigation measures has been to select a mill and tailings impoundment site some 3 km. away in the separate watershed of the Florence Creek. Ore will be trucked to the mill site. The basis for the feasibility study's economic analysis is a daily mill throughput of 6,000 tonnesper day, though throughputs larger and smaller than this are currently being evaluated. While economies of scale clearly play a part in achieving a successful project, the staging of capital expenditure, minimization of capital at risk and maximizing the potential for self financing of deferred capital costs are an important part of the plant capacity selection process.

The climate of the area is typical B.C. coastal, with high rainfall but negligible snowfall. The terrain is not severe, and the project area is well served by existing logging roads which will require a minimum of upgrading. Infrastructure requirements include a power plant and a dock for the importation to the island of equipment and materials during construction and reagents and supplies during operation. A lime burning facility will also be required to obviate the high cost of sea transport of burnt lime.

There is an adequate labour supply on the islands for operations labour, while temporary facilities will be required to house about three quarters of the construction labour force, which is expected to peak at about 250.

ORE CHARACTERIZATION AND PROCESS DEVELOPMENT

The Cinola ore may best be described as "deceptively" refractory; it is not very refractory in that 50 to 70% recovery may be obtained by direct cyanidation, albeit at high cyanide consumptions. This has lead researchers to believe that a simple upgrading by physical means such as flotation followed by a unit process to deal with the refractory nature of the ore in a small quantity of concentrate would be an appropriate process route. Unfortunately, this is not the case; it is possible to recover by flotation a small amount of free gold and some that can be physically liberated from its associated sulphide by grinding, but there is a variable but significant gold population that is locked in sulphides that are themselves too fine (down to one micron) to be physically liberated by grinding. Confirmatory testwork carried out by City showed locses to flotation tails of sulphide locked in gangue even after grinding to 90% minus 325 mesh. Of course, the effect on recovery and ratio of concentration of such a fine grind, even with the addition of desliming agents, was significant.

Confirmatory flotation tests formed the first part of City's research program. Ore samples were categorized by ore type identified by geological criteria. With the exception of silicified sediments, which form only a small part of the ore reserve, results were consistently disappointing, showing recoveries of both gold and sulphur to flotation concentrate generally less than 30% and concentration ratios of three or four to one.

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Following confirmation of the fact that an economic preconcentration step would be difficult or impossible to achieve, research focussed on process routes for the oxidation of sulphides in the ore without preconcentration. Sulphide sulphur levels in the ore average 1.7%, and sulphides are present largely as pyrite and marcasite in varying proportions. Minor copper, arsenic and mercury report in analysis, but have not been mineralogically identified.

The testwork program investigated oxidation methods of increasing severity. Early success was achieved with chlorination and caustic oxidation, but at prohibitively high reagent consumptions; in addition, caustic oxidation resulted in a slurry that was so viscous as to be unmanageable. Early testwork with pressure oxidation gave disappointing metallurgical results, and although good results were later achieved, preliminary financial analysis indicated that the inherent high capital and operating costs of this method would be prohibitive. Turning to less conventional oxidation routes, encouraging results were obtained in oxidation by nitric acid, and it was decided to explore this route in greater detail.

The basic principle of the method is that the oxidizing agent is regenerated from the gaseous products of the oxidation reaction; the nitric acid is therefore considered to be an intermediate by means of which oxygen is transferred to sulphides. This concept forms the basis of the Arseno Process, on which early testwork was based. It was recognized early in the test program that, in order for the method to be of practical use, very low residual nitrate levels in solution would have to be obtained to prevent excessive discharge of nitrate to the environment and minimize consumption of nitric acid.

Early attempts at continuous operation of the process in pilot plant resulted in a cumbersome circuit that was both difficult to operate and gave disappointing metallurgical results. Two pipe reactors were used, separated by a solids liquid separation step; unreacted nitrates in the liquid phase after the oxidation reactor were recycled to a catalyst recovery reactor from which nitric oxide gas was recovered and reinjected to the oxidation reactor. The reactors were operated at temperatures of 102 to 138 degrees C and feed pressures ranging from 90 to over 200 psi. Gaseous oxygen was injected into the oxidation reactor to oxidize the nitric oxide to nitrogen dioxide which was absorbed in the liquid phase of the slurry to reform nitric acid.

Considerable difficulty was experienced in maintaining a constant feed to the reactors due to high wear rates in feed pumps, and although sulphide exidation in excess of 95% was readily achieved, results of both batch CIL tests on slurry drawn from intermediate parts of the circuit and of a continuous CIL test on pilot plant product showed disappointing gold recoveries generally in the range of 75 to 85%.

Consideration of a means of simplifying the mechanical complexity of the pilot plant configuration resulted in an appreciation of the fact that the operating conditions of the pilot plant were dictated not by the conditions required for the exidation reaction but for the regeneration of nitric acid; the exidation reaction occurs at temperatures down to 75 degrees C, and is not affected by pressure. If the exidation and nitric acid regeneration functions were to be separated, the exidation reaction could be carried out in a series of stirred tanks at atmospheric pressure, while product gases could be drawn off to a separate regeneration plant, which would have the added advantage of producing nitric acid using ambient air instead of gaseous exygen.

A new pilot plant was planned based on this principle, and in the course of bench scale testwork which was almost at defining the operating parameters for this plant, a number of factors in the fundamental chemistry emerged that led to an increased understanding of the sulphur chemistry in the process and its effect on subsequent gold extraction. At the same time, an understanding was gained of the apparently complex interdependencies between some of the process variables. The effect of some process variables was also found to be exaggerated by the small scale of the testwork; for example, in analyzing the effect of particle size, the relatively large amount of elemental iron and iron oxides added from grinding media in laboratory scale mills was found to have the effect of consuming acid and, in some cases, of preventing the reaction from going to completion. It was also found that complete oxidation of sulphides does not necessarily give the best subsequent gold recovery, and that the effect of increased residence time to achieve a high level of sulphide oxidation is more than offset by the deleterious effect of that residence time on subsequent gold recovery.

The factors noted in this bench scale testwork were later confirmed in pilot plant operation. The pilot plant was operated during May and June of 1988 at a rate of 55 Kg/hr, and resulted in demonstration of the ability of the process to achieve adequate oxidation of sulphides to permit subsequent gold recovery of up to 92% while limiting nitrate levels in the discharge to acceptable levels. The configuration of the pilot plant will allow relatively simple scale up to full size operation. The simplicity of the resulting plant design will result in low capital and operating costs.

It appears in retrospect that the disappointing metallurgical results of the earlier series of pilot plant tests were due to a lack of appreciation of the effect of critical process variables that were later determined in the simpler pilot plant arrangement.

OXIDATION PROCESS DESIGN AND OPERATING PARAMETERS

The overall reaction by which pyrite is oxidized by nitric acid may be summarized as:

$2FeS_2 + 10HNO_3 - - - > Fe_2(SO_4)_3 + H_2SO_6 + 10NO + 4H_2O_6$

In practice, this represents the end result of a series of reactions involving the intermediate products of the reduction of nitric acid. Of these, the most important is nitrous acid, which is the oxidizing agent of primary importance. The presence of free acid is essential to the formation and stability of nitrous acid; in practice free acid is supplied as sulphuric acid. The reaction will not proceed at a pH in excess of 1.7. During testwork, the sulphuric acid requirement was considerably exaggerated by the presence of elemental iron from grinding media.

The objective of the process is to adequately oxidize sulphides, making the associated gold available for cyanidation, while maximizing the recovery of nitric acid and minimizing the level of nitrates in solution in the effluent. The proposed plant configuration is a series of closed, stirred tanks from which gaseous products are drawn off for external regeneration of nitric acid. The process design and control parameters must take into account the following process variables:

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- * Grind
- * Pulp density
- * Leach retention time
- * Leach temperature
- * Stoichiometric ratio of nitric acid:sulphide sulphur
- * Extent and rate of reaction
- * Sulphuric acid addition
- * Neutralization technique
- * Cyanidation conditions

The application of the process to achieve the objectives is probably best described by examining each of these variables and their interrelationships.

Grind

The effect of particle size is more than that of simple mechanical liberation; at finer grinds the effect of increased specific surface area of exposed sulphides appears to affect the rate of reaction, which in turn affects required leach retention time and achievable final nitrate concentrations. Laboratory scale tests are affected by the relatively large quantities of elemental iron and iron oxides introduced by grinding. In tests intended to remove as far as possible the consequences of these side effects, it appears that recovery is increased about three percentage points between P_{80} 's of 100 and 44 microns. Aqua regia digestion tests indicate that a small amount of the contained gold (2-4%) cannot be exposed to cyanide regardless of grind due to silica encapsulation.

A fully autogenous grinding circuit will be used, and a grind of 80% passing 60 microns has been used as the basis of evaluation.

Pulp Density

The rate of reaction increases with pulp density, while a higher concentration of nitrate per unit volume of effluent is permissible than at lower pulp densities to achieve the same final effluent level, due to subsequent dilution. The targets set for neutralization discharge are 1.5 gm/litre NO_3 at 50% solids or 1.0 gm/litre at 40% solids. A high pulp density also reduces the heat requirement to maintain leach temperature. Pilot plant work was carried out at 50% solids, and this has also been used as the design criterion for plant design. However, settling data on the ground ore indicate that in practice higher pulp densities may be achievable.

Leach Retention Time

Bench scale testwork indicated and pilot plant operations verified that subsequent gold recoveries deteriorated after an optimum leach residence time, even if that time is not sufficient to achieve complete sulphide oxidation. Confirmatory testwork using perchloroethylene as a solvent for sulphur indicated that elemental sulphur is formed in the latter stages of the leach, and that this adversely affects gold recovery. The presence of thiosulphates have also been noted, and subsequent cyanide consumption is adversely affected by the formation of thiocyanate. The optimum leach time appears to lie between 90 and 120 minutes at 85 degrees C, but given the number of factors affecting the rate of the reaction, a flexible leach time and plant design will be required. This is simply achieved by changing the number of leach tanks in use, with the capability of discharging to neutralization by pumping from any tank.

A critical factor is the achievement of acceptable nitrate levels in a comparable leach residence time as that which will give optimum subsequent gold recovery. The major controlling factors are the use of the correct stolchiometric ratio of nitric acid to sulphide sulphur and the use of sulphuric acid to modify reaction rate. Provision is made for the addition of a barren pyrite as a denitrating agent in the latter stages of the leach, but this is considered to be required under upset conditions only.

Leach Temperature

The exidation reaction occurs over a wide range of temperatures, and has been demonstrated at bench or pilot scales from 60-200 degrees Centigrade. Clearly plant configuration is simplified and heat input minimised by operating the leach at a lower temperature, particularly one that allows operation at ambient pressure. Another significant factor is the effect of temperature on the precipitation of jarosites: lower temperatures inhibit such precipitation in the leach vessels. The rate of reaction has been found to be

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adequate at 85 degrees C and may, in practice, be acceptable at lower temperatures. Operating at 85 degrees as opposed to temperatures closer to ambient boiling substantially reduces the heat loss to steam drawn off with the reactor gases.

Stoichiometric Ratio of Nitric AcidSulphide Sulphur

Use of the correct ratio of nitric acid to sulphide sulphur is essential to achieve adequate sulphide oxidation without excess nitrate remaining in solution in the leach effluent. Both bench scale and pilot plant tests show that this ratio should be maintained at between 92 and 94% of the theoretical stoichiometric requirement of nitric acid. Addition rates above this range result in excess residual nitrate, while addition rates below this result in both unreacted nitrates and sulphides, leading to poor gold recovery as well as high residual nitrates.

This ratio will be used as the primary process control. Filot plant operations showed that there was little difficulty in maintaining the correct ratios, even at the low flow rates employed. In the full scale plant, slurry will be stored in three tanks of two hours capacity each, allowing the sulphide sulphur content of any tank to be accurately known before it is fed to the leach circuit.

Rate and Extent of Reaction

The rate of reaction is essentially governed by the concentrations of nitrate and sulphide present in the slurry, although, as we have already seen, a number of factors may modify that rate. At stoichiometric ratios of nitric acid:sulphide sulphur in the range used in practice, the oxidation reaction proceeds rapidly for the first thirty minutes, after which the rate of decline in both nitrate and sulphide levels becomes asymptotic. If either becomes depleted before the other, the reaction tends to "die." This is not a sudden event, but is evidenced by a gradual drop in emf and gradual rise in residual nitrates and/or sulphides.

Sulphuric Acid Addition

In pilot plant operations, it was found that the rate of addition of sulphuric acid was the main means of controlling the rate of reaction to ensure that acceptable residual nitrate levels are achieved in the same leach residence time as that required to achieve adequate sulphide oxidation. Determining a specific addition rate for a full scale plant is difficult because of the acid consumers present at the pilot scale plant that would not be present in a full scale plant, particularly after autogenous grinding. It is clear that more acid will be required when treating low sulphur ores, because of the smaller amount of acid produced in the leach.

In plant design, provision has been made for the addition of a high rate thickener immediately after the leach to allow recycle of a small amount of free sulphuric acid and some residual nitrate. In general, sulphuric acid addition must be sufficient to maintain a pH below 1.7 throughout the leach.

Neutralization Technique

The technique used for neutralization technique proved to be an important factor in both bench scale tests and in pilot plant tests. Neutralization at ambient temperature was found to lead to low subsequent gold recovery. With recognition of the importance of elemental sulphur, it became clear that the reasons for proper neutralization technique centred on the sulphur chemistry that occurred in the neutralization step. Line reacts at high temperatures with elemental sulphur to form polysulphides which prevent the subsequent occlusion of gold by the elemental sulphur. Not all of the sulphur is taken up in JAN-10-1999 15:52 FROM CITY REDOURCES (COMADA) TO

neutralization, and improved recoveries can be achieved by taking the final pH to very high levels with lime and/or caustic. However, improvement in recovery is not sufficient to overcome the additional reagent cost.

Prior to lime neutralization, the slurry pH is taken to approximately 3.5 with limestone, thus making a significant saving in lime. An additional benefit of this stage of neutralization is that the intense gassing that takes place on addition of limestone helps strip residual NO from the slurry; product gases from the limestone neutralization step are therefore added to the nitric acid plant feed stream.

Cyanidation Conditions

The complexity of the sulphur chemistry continues to affect the process through to the cyanidation step. High cyanide consumption has been observed due to reaction with elemental sulphur to form thiceyanate, and possibly due to HCN evolution at elevated temperatures. Cyanide consumption is predicted to be 1.75 kg/ tonne, but there appears to be scope for considerable improvement in this.

Upset Conditions

The two main anticipated upset conditions are a dying reaction or excess nitrates in leach effluent, both of which are functions of the rate and extent of reaction. Considerable test work has been carried out to ensure that these conditions can be dealt with in practice. Excess nitrates can be reduced by the addition of barren sulphide in the form of pyrite concentrates which should be added stagewise in substoichiometric quantities; excess pyrite addition or addition of pyrite concentrates that result in excess ferrous iron in solution will lead to the reaction dying. The addition of sulphuric acid also assists in the denitration reaction.

The rate of reaction may slow to a point that it may be considered to be "dying" as a result of a deficiency of either nitrate or sulphide or after addition of a stoichiometric excess of pyrite. The key to restarting the reaction is to increase the level of nitrous acid in the slurry, which may be achieved by adding additional nitric and sulphuric acid by diverting vigorously reacting slurry and/or NO gas from the first reaction vessel to the latter stages of the leach.

Summary

It is clear that the chemistry of the system is complex, in terms of both nitrogen and sulphur species. It might appear at first sight that this would lead to a complex process plant. In fact the reverse is true: although the testwork program that has allowed us to understand the interdependencies between the process variables has been very complex, the plant itself will be extraordinarily simple. Our understanding of the chemistry is sufficient to indicate process conditions that should be nvoided to prevent deleterious effects on subsequent gold recovery, and to indicate the process variables that may be used for process control. Our understanding at this stage enables us to predict gold recoveries in excess of 90% at very low operating costs; the potential exists to improve on both recovery and cost by means of testwork currently planned to explore in more detail the interdependencies between process variables over a broader range of conditions.

PROCESS PLANT FLOWSHEET AND FACILITIES

A simplified flowsheet of the whole process plant is shown in figure 1. With the exception of the oxidation circuit, it is essentially conventional. Ore will be crushed by a gyratory crusher and stored in an open stockpile. From the stockpile reclaim, it will be fed

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FIGURE 1



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to an autogenous primary mill followed by two pebble secondary mills. The ore is both extremely hard and abrasive, with a work index approaching 20 kwh/tonne and an abrasion index of 0.85, characteristics which lend themselves to this type of grinding circuit. After thickening to 50% solids, slurry will be held in three holding tanks for sulphide sulphur determination before being fed to the oxidation circuit, which will consist of four stainless steel agitated tanks. Fifty per cent nitric acid will be metered from the nitric acid plant.

Leach discharge will be diluted and rethickened to 40% solids, with the thickener overflow reporting to leach feed. Thickener underflow will report to a neutralization circuit where limestone and lime slurry will be added to bring the pH to 10.5 prior to a hybrid leach/CIL circuit consisting of two leach tanks and seven CIL tanks. Carbon loadings in excess of 6 kg/tonne have been demonstrated. Carbon will be stripped in a conventional Zadra system, and gold values recovered by electrowinning. Cathodes will be retorted for mercury removal prior to melting.

Leach discharge will be treated by either sulphur dioxide or hydrogen peroxide for removal of remaining cyanide, and any mercury remaining in solution will be precipitated by addition of sodium sulphide.

Product gases from the nitrate oxidation leach and limestone neutralization tanks will be ducted to the nitric acid plant. The technology used in nitric acid regeneration has been developed by Minproc Engineers, and is substantially simpler than conventional nitric acid technology. Nitric oxide is oxidized to nitrogen dioxide by ambient air and absorbed in three consecutive absorption columns at ambient pressure. Absorption in water of nitrogen dioxide produces one mole of nitric oxide for three moles of nitrogen dioxide absorbed. In conventional nitric acid plant practice, this leads to a requirement for multistage oxidation and absorption of gases that are produced as a result of absorption. In this plant, residual gases will be absorbed in sulphuric acid to form nitrosyl sulphuric acid. Since both sulphuric acid and nitric acid are required in the process, the nitrosyl sulphuric acid will be only produce nitric acid of sufficient strength operating at ambient pressure and requiring relatively simple materials of construction. Make up nitric acid will also be produced in the plant by oxidizing ammonia.

Power will be generated on site, with process heat requirements satisfied by waste heat boilers. The projected average electrical demand is 17 MW. A 25 MW power station is planned, with 20 MW on line and one standby unit.

PROJECTED CAPITAL AND OPERATING COSTS

The capital cost of a 6,000 tonne/day plant is estimated at C\$51 million. Of this, some C\$10 million is attributed to the oxidation circuit (leach fircuit plus nitric acid plant). Total project preproduction capital costs are estimated at C\$120 million, including contingency, with an additional C\$52 million in capital costs over the twelve year life of the property, mostly in ongoing tailings impoundment construction.

Operating costs are summarized in figure 2. Total operating costs, including mining, are a little under CS20/tonne with an owner operated power plant. Purchase of power from a contractor operated power plant would increase costs by approximately CS1.50/tonne. Of the total costs, CS12.29/tonne are direct operating costs of the process plant, of which CS2.75/tonne are attributable to the oxidation circuit.

Direct operating costs per ounce will average USS230/oz over the life of the property, but with the enhanced feed grades in the first two years, direct operating costs will be less than USS200/oz in these years, leading to a pay back period of just over two years.

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PROCESS PLANT										
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Stat	1,350,000	0.64	1,950,000	0.84	1,350,000	0.64	1,350,000	0.64	1,350,000	0.64
Operating	2,227,801	1.08	2,227,601	1.08	2,227,801	1.08	2,227,801	1.03	2,227,601	1.06
Maintenance	1.093,374	0.52	1,095,374	0.82	1,005,374	0.52	1,005,374	0.52	1,095,374	0.52
Constratios										
Operating Materials	14,574,811	8.04	14,574,811	6.04	14,674,811	6.94	14,574,811	8.94	14,574,811	6.94
Maintenance Materials	1,379,700	0.69	1,379,700	0.88	1,379,700	0.88	1,379,200	0.68	1,379,700	0.68
Power (including minesite)	4,085,947	2.32	4,865,947	2,32	4,865,947	2.32	4,665,947	2.32	4,865,947	2.32
Laboratory Consumables	920,300	0.13	320,800	0.15	320,800	0.15	320,800	0.15	320,500	0.15
Subtotal	26,014,233	12.20	25,814,233	12.29	25,814,233	12.29	25,814,233	12.29	25,814,203	12.20
TRANSPORTATION			1.							
Transport & Frainht	2.408.000	1.19	2.495.000	1.19	2.495.000	1.19	2.498.000	1.19	2 495 000	1.19
Iknasiona	975,150	0.46	875.150	0.48	975 150	0.45	975 150	0.46	975 150	0.48
Generation	from and a series of the series			and the second second second		warmen als and an agene	a commence de la interior			
Bubiolal	3,473,150	1.65	3,473,150	1.65	3,473,150	1.65	3,473,150	1.65	3,473,150	1.65
ADMINISTRATION							•			
Labor	999 818	0.48	998,618	0.48	899.618	0.48	999.618	0.48	999.615	0.48
Caparal	1 855,000	0.79	1.055.000	0.79	1,655,000	0.79	1.655.000	0.79	1.655.000	0.73
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Subtotal	2,854,818	1.28	2,854,618	1,26	2,654,613	1.20	2,654,618	1.28	2,654,618	1.26
TOTAL	\$40,610,001	\$19.23	\$40,782,001	\$19.41	\$41,000,001	\$19.57	\$41,035,001	\$10.84	\$42,064,001	\$20.03

NOTE: Average mining costs used for years 4-7 and 3-12.

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ENVIRONMENTAL MITIGATION

The development of the project has raised numerous environmental concerns in the local community, and City has adopted an "open door" philosophy to encourage discussion and resolution of the issues.

The major environmental impacts that are perceived as critical are:

- * The effect of acid mine drainage
- * The effect of mill effluent
- * The effect of elevated levels of mercury and other heavy metals due to both mining and milling
- * Protection of facilities from seismic activity

Other concerns include protection of the environment from spiils of reagents in transport, reduction in stream flow due to fresh water and tailings impoundments and air pollution due to particulate and gaseous effluents.

The main emphasis in City's approach has been to assure both local communities and regulatory agencies that the environment can be protected both during operations and after closure by measures carefully planned at an early stage of development. The two major measures proposed are a zero discharge tailings impoundment system, designed to minimize impact of mill effluent on the environment, and the treatment of acid generating mine waste during the life of the mine by addition of limestone, and the eventual backfilling of the pit with waste material to prevent generation of acid both in it and in the pit walls. During the life of the pit, mine drainage will be treated for acidity and precipitation of heavy metals. A small amount of argillically altered waste material, which has been found to generate acid unusually quickly, will be trucked to the tailings dam and stored under water.

The tailings impoundment system will take the form of a series of impoundments that will be activated sequentially, with each active impoundment protected from inflow of precipitation from its catchment area by diversion ditches. The water balance will be maintained by careful minimization of fresh water usage in the mill and maximum use of return water. Discharge of mill effluent to the environment can be avoided by this method even under extremely wet conditions. After closure, there will be a small but continuous discharge of tailings pore water due to rising ground water flow. For this reason, effluent will be treated to ensure that eventual discharge after closure will meet receiving water standards. Cyanide will be destroyed by either the sulphur dioxide or hydrogen peroxide methods and heavy metals precipitated by sodium sulphide. Pilot plant testwork included the testing of these processes and demonstration of their effectiveness.

These measures have been presented to the B.C. permitting agencies in July 1988 in a Stage II report. The review process is nearing completion, and the first formal comments have been received. While these comments request a considerable amount of clarification of data, and in some cases additional data, there is no comment that might constitute a fatal flaw in the proposed environmental mitigation measures. Although the costs of many of the proposed measures may seem high, they are not prohibitive in the context of the project cash flow.

Approval in principle for the project to proceed is expected in the first half of 1989.

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CONCLUSION AND ACKNOWLEDGEMENTS

This paper has presented some of the more critical aspects of the Cinola project. It should be stressed that the solutions presented in this paper to metallurgical and environmental problems are project and site specific. It is felt that the nitrate oxidation process has potential in treating other ores and sulphide concentrates; nevertheless, the particular application described here is an answer to the specific problems of the Cinola ore. Other plant configurations and methods of applying the same basic chemistry could well be more appropriate to other ores. The environmental mitigation measures adopted by City are also specific to the particular problems of the site and to particular local concerns; the same measures may be totally inappropriate at other sites.

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The conclusions presented in this paper are drawn from a feasibility study prepared by Minproc Engineers and Davy McKee Corporation, based on testwork carried out by Bacon Donaldson of Vancouver and Hazen Research of Denver. The Stage II report was prepared by Norecol Environmental Consultants and Steffan, Robertson and Kirsten of Vancouver. City believes that the two reports together demonstrate not only the viability but the profitability of this challenging project, and it is anticipated that project engineering and construction will commence in the summer of 1989. JUNE 26/96

PINE GROVE -

- Tunget Hill - Silie'd Rhy - QP flow & bx's - N-5 elongation 400-600 x 200-300 m - Veius trending 070-0800 - 1 Minnova vertical from top - 2 East - 50-60° Hectas at M&S ends

TOKE

180 km E

280 Kan SE

Regard and

fo and

Feell

West Ridge - intense silic's, voggy in places, any aboute? - stockworks and silic's - grey silica vn + episodic bandis/bladed caleste at 020° / Im. in dip steep East.

P.G.-12 - at 60'

NW. - Silic'd thy flows up to poss vent - some may be obsidian glass froms. limited namow gtz valets - botryoidal si + chale sulet. bluich - late - Chalcedony blocks ??? blow-outs bx? alon 0600 - no comb text, no banding, no collotoria

- Structure + Detail May + Soholes = 2 yrs 1:50 \$ 1-10,000 - Contouring grade /Thickness

Look at Rawhide, Round Mtr. Samples

Consultant? HiBonham?