ADANAC FEASIBILITY REPORT

By:
Climax Molybdenum Corporation
of British Columbia, Limited

December 19, 1975

104N

April 12, 1977

Mr. Jack Pelletier
Adanac Mining & Exploration Ltd.
1111 West Hastings St&et
Vancouver, B.C.

Dear Jack,

Re: Adanac Cost Summary

With apologies for the delay, I enclose some comparative figures based partly on current and projected costs at Faro, and partly from C.M.C. data on their Thompson Creek molybdenum property. While we do not pretend that our numbers are particularly accurate, they do reflect a general and often considerable increase over Adanac's projected cost.

Areas in which we would be particularly concerned that Adanac's figures may require adjustment are:

1. Mill Labour - where a minimum discrepancy of 27¢ based on current labour rates would be apparent.

2. Mill Repair Supplies - although Adanac should be somewhat kinder on equipment, a 50¢ discrepancy would be apparent by comparison with Faro and reference to Thompson Creek estimates.

3. If reagent mix is similar to Thompson Creek, a 14¢ discrepancy would arise, although your cost should be good on this.

4. General and administrative costs as experienced at Faro are considerably higher than you would propose for Adanac. However, we certaanly can't brag that we are very efficient in this area so you would have to weigh this one fairly carefully.

5. We would also have some reservations at applying the relatively high product price of $3.82 although, provided your contracts were for payment in U.S. dollars, the present exchange rate would probably be sufficient cover.

In general, we would not feel happy about an involvement in the Adanac property at this stage of our development, particularly where such involvement might be tied to a production date. For what it is worth, I attach a cost comparison where we would estimate the cost per ton milled in the $5.50 to $7.00 per ton range, rather than the $4.00 range you envisage. Perhaps we can get together with our Mr. Biggs, who was responsible for the data collection and comparative numbers, so that you can get a better idea of just how strongly me might feel on certain items or how reliable any particular comparison might be.

Yours truly,

CYPRUS ANVIL MINING CORPORATION

J. G. Simpson
Exploration Manager

JGS/cb
Attach.

c.c. T. Biggs
J. Olk

J. Bruk (May 19th)

COMPARISON OF OPERATING COST ESTIMATES

ADANAC

	Adanac Per Ton Milled		C.A.M.C. Est. Per TonMilled
	Initial Pit	Onward Pit	
Total Mining	1.24	0.83	1.25
Milling			
Labour	0.30	0.30	0.57
Steel	0.67	0.67	0.67
Reagents	0.37	0.37	0.51
Power	0.63	0.63	0.92 *
Maintenance Supplies	0.10	0.10	0.60
Heating	0.04	0.04	0.46 *
G & A	0.80	0.80	2.06 *
	4.15	3.74	7.04

* Some probable discrepancy in C.A.M.C. figures due to unfair comparison with Faro. If these figures are substituted by Adanac numbers, total cost C.A.M.C. estimate = $5.03.

MEMO.

To
GLEN SIMPSON
FROM: T. H B

SUBJECT: COSTS v. ADANAC STUDY

ATTACHED ARE SOME ~~very~~ ROUGH ESTIMATES ~~COMPARISONS~~ OF ADANAC COSTS based on current FARO 77 PLAN cost ~~TO FARO 77 PLAN~~. PERHAPS THE MOST SIGNIFICANT BEING

for ADANAC FIGURES

1) · LABOUR IN THE MILL — $ 1,774,000 (27¢/DST.)

THIS IS MINIMUM DISCREPANCY.

2) · MILL REPAIR SUPPLIES — $ 3,286,000 (50¢/DST.)

· REAGENTS. MAY USE A DIFFERENT MIX TO THOMPSON CRK.
IF SAME MIX — 920,000 — 14¢

· G + A. - SAME AS FARO — 2,250

5,980,000

5 YRS — 30,000,000

· CURRENT Mo PRICE US$ 3.45/lb — 30,000,000

TOTAL 5 YR ADS PRE-TAX. — $ 60,000,000

BASED ON THE ABOVE I WOULD BE RELUCTANT TO COMMIT FUND TO THE PROJECT PARTICULARLY IF TIED TO A PRODUCTION DATE COMMITMENT.

COST COMPARISONS

MINING COSTS : AT $1.40 / Cu YD IN LINE WITH FARO AFE FOR NEW TRUCKS.

MILLING : REAGENTS : ABOUT 40% OF FARO $/TON MILLED

CMC THOMPSON CREEK MLO '75 32.16 ¢

ESCALATION TO 77 20% 6.44

EXTRA FREIGHT ON 3 lbs REAGENTS 12.00

50.60 ¢

ADANAC STUDY 37.0 ¢

STEEL : COULD BE REASONABLE

REPAIR SUPPLIES : FARO 77 PLAN 65 ¢ / TON

ADANAC STUDY 10 ¢ / TON

LABOUR :

C.A.M.C. Op + Repair $1.48 / DST - 77 Plan.

Based on 166 direct employees & 10000 TPD

ADANAC 115 MILL EMPLOYEES & 18000 TPD

EQUIVALENT AT FARO RATES $1.48 × 115 / 166 ÷ 6570

= 57 ¢

ADANAC STUDY USES 30 ¢

ALSO SHOULD QUESTION THE NUMBER OF EMPLOYEES USED BY ADANAC.

GENERAL & ADMIN. ANVIL MINESITE G&A PLAN '77 $2.06/DST $7530

ADANAC PROPOSAL $0.80/DST $5250

MOLYBDENUM PRICE HISTORY

$/lb. Mo

January 1, 1974	$1.72
March 1, 1974	$1.87
July 1, 1974	$2.05
September 1, 1974	$2.30
January 1, 1975	$2.43
October 1, 1975	$2.62
February 29, 1976	$2.90
August 29, 1976	3.20
December 31, 1976	3.45

Cyprus Mines Corporation

From: **R. B. Ellingsen** Date: 3/9/77

To:

MAR 1 1 1977

MR. TOM BIGGS

CYPRUS ANVIL MINING CORP.

330-355 BURRARD ST.

VANCOUVER, B.C., CANADA

V6C 2G8

CYPRUS

((T. BIGGS.

ADANAC MINING AND EXPLORATION LTD.

Ninth Floor, 1111 West Hastings Street, Vancouver, B.C. V6E 2J3 Telephone (604) 685-0351

January 19, 1977

MEMO TO: W.W. Bennett

FROM: J.D. Pelletier

SUBJECT: Update of Adanac Feasibility Study

The Equity Mining staff took a quick look at updating projected capital and operating costs for Adanac in the light of their recent costing of a similar plant for the Sam Goosly project. The original feasibility was made in 1971 under the auspices of Kerr Addison Mines with capital costs estimated on the basis of a production commitment in late 1971 and operating costs and metal prices projected to production start in 1974. Our updating reflects a production commitment at year-end 1976 and production startup in the latter half of 1978. Molybdenum prices and operating costs were estimated as of year-end 1976. In keeping with today's marketing requirements, it was assumed that oxide molybdenum would be produced. Summaries of capital and operating cost estimates are attached.

The results of the study indicate that there has been a net increase in the viability of the Adanac project during the five years that have elapsed since Kerr Addison's estimate. On an all equity basis, payback of capital costs is in year 6 and return on investment is approximately 10% (this compares with 8 years and 6% return estimated in 1971). The cash flow was rerun using an assumed 10% increase in molybdenum price to show the sensitivity of the project to small changes in the assumptions. This gave a payback in the 5th year and return on investment of approximately 14%. Another run using 100% bank financing and the higher assumed price of molybdenum showed project loan payback in 6 years. Cash flows are summarized on an attached sheet. Certain critical assumptions that could affect viability of the project are listed below for your consideration:

1. Kerr Addison's ore reserve estimate of 1971 was used. This reserve takes the large upgrading indicated by underground development work in tonnage rather than grade. Further work is needed to establish whether this approach or its alternative of higher grade but fewer tons is valid.

2. We believe our updating of capital costs is within 10% of a value that would be arrived at by a more definitive study. Contingencies were not taken as a blanket percentage allowance. Most of the costs can be directly interpolated from Sam Goosly project costs and are considered realistic. Certain mine equipment

costs were updated by contacting suppliers. Commonwealth Construction furnished valuable assistance in estimating construction costs applicable to the region.

3. Our update of the operating cost came out surprisingly high as compared with Kerr Addison's estimate, even allowing for inflationary increases in costs since their estimate was made. We believe that a definitive study of operating costs could reduce our estimate by anywhere up to 10%.

4. The (oxide) molybdenum price of U.S. $3.82 that became effective at year-end 1976 was used. Frequent significant increases in price have been customary over the past several years and it is difficult to accurately determine an appropriate price level for this type of study. Similarly, no exchange differential was estimated between the value of Canadian and U.S. dollars although many observers are projecting a differential favourable to Canadian producers.

5. As in the Kerr Addison study, no provision was made for the effects of producing and marketing a tungsten byproduct. Recent increases in the price of tungsten indicate that its recovery may benefit the project.

It is recommended that further work to enhance the economics of the project be directed at:

(1) More precisely determining the grade mined in the early years, and

(2) More precisely determining power and grinding steel costs and the optimum tradeoff between grinding costs and metal recoveries.

It appears to me that any improvements that could be made in the project are worth investigating now in view of the very strong outlook for molybdenum markets.

J.D. Pelletier

J.D. Pelletier

JDP:dc
Attachments

xc: T.E. Congdon

Summary of Adanac Capital Cost Estimate
Updated to Bid Tenders Year-End 1976
Production Startup Late 1978 @ 18,000 TPD

A. Preproduction Pit Preparation:			
Till Overburden Stripping	1,917,000 yds^3	$2,109,000	
Rock Waste	4,050,000 tons	3,038,000	
Subtotal			$ 5,147,000
B. Mining Equipment:			
2-15 Cu. Yd. Electric Mining Shovels		3,410,000	
2-45R Electric Rotary Drills		936,000	
1-15 Cu. Yd. End Loader		520,000	
9-100T Haul Trucks		4,500,000	
1-D8 Dozer		186,000	
1-824 Wheel Dozer		168,000	
1-M14 Grader		131,000	
Misc. Vehicles, Supplies, and Inventory		480,000	
Subtotal			10,331,000
C. Road Construction per Feasibility Study			500,000
D. Plant Construction Costs:			
General Site and Utilities		3,290,000	
Crushing and Stockpile Area		8,076,000	
Grinding		8,398,000	
Flotation, Drying and Packaging		2,061,000	
Leaching		600,000	
Plant Service Buildings and Equipment		2,755,000	
Power Plant		6,339,000	
Mobile Equipment - Plant		500,000	
Piping		2,580,000	
Electrical		2,423,000	
Water and Tailings		4,700,000	
Freight and Shipping		2,000,000	
Contractor's Overhead		12,768,000	
Conversion from 15,000 to 18,000 tpd		2,986,000	
Engineering and Construction Management		4,000,000	
Subtotal			63,476,000
E. Escalation at 8% of Items A to D			6,356,000
F. Allowance for Townsite			7,000,000
G. Options and Royalties (Preproduction)			420,000
H. Working Capital			8,000,000
Total Capital Cost (Excluding Interest and Contingency Allowance)			$101,230,000
Estimated Additional Cost for Roaster			$ 3,500,000

JDP:dc
January 19, 1977

ENDACO 828.74 Total mill

Summary of Adanac Operating Cost Estimate
Updated to Year-End 1976
Production Rate = 18,000 TPD

	Per Ton Mined		Per Ton Milled		
	Initial Pit	Onward Pit	Initial Pit	Onward Pit	
Mining:					
Drilling	0.074	0.074			
Blasting	0.119	0.109			
Loading	0.047	0.047			
Hauling	0.234	0.187			
Supervision and General	0.079	0.118			
Stockpile Reclaim	-	0.016			
Total Mining	0.553	0.551	1.24	0.83	1.24
Milling:					
Labour			0.30 1.48	0.30	.67
Steel			0.67 .84	0.67	.67
Flotation Reagents			0.37 1.02	0.37	.51
Power			0.63 .92	0.63	.92
Maintenance Supplies			0.10 .64	0.10	.60
Heating			0.04 .46	0.04	.46
Total Milling			2.11 5.36	2.11	2.59 3.73
General and Administrative Costs			0.80	0.80	1.50 2.06
Total Operating Costs*, Year-End 1976			4.15* #12.62	3.74* 5.03	7.03
Kerr Addison Feasibility Estimate (esc. to 1974)			2.586	2.280	

ANVIL 77 ... OK.

KAISER 74 × 2 × 150

*Excluding cost of conversion to oxide estimated at 10¢ per pound of oxide produced, and cost of sales estimated at 1% of selling price.

5000

1111 West Hastings
Burrard Bldg
9th Floor

Stripping Ratio? 1.00 : 1.00

2.7

62

128 CuYD

2.25 Tons CuYD

$1.40 CuYD

3.48

2.23 DST/CuYD

JDP:dc
January 19, 1977

Summary of Updated Adanac Cash Flows
Based on 18,000 TPD Production Rate

	I	II	III
Financing			
Project	100% Equity	100% Equity	100% Bank Loan
Working Capital	85% Prov. Payments	Equity	Bank Loan
Mo Price (Oxide)	3.82	4.20	4.20
5 Year Summary (Millions):			
Sales	250.5	275.4	275.4
Operating Costs	139.8	140.0	140.0
Interest Expense	6.9	-	38.5
Taxes	1.7	12.6	-
Settlement Lag	2.6	17.0	17.0
Net Cash Flow	99.5	105.8	79.9
Payback in Years	5¼	4 3/4	6
Total 17 Year Summary (Millions):			
Sales	704.1	773.4	773.4
Operating Costs	427.9	428.6	428.6
Interest Expense	19.1	-	49.5
Taxes	64.8	113.9	78.9
Ongoing Capital Cost	7.0	7.0	7.0
Net Cash Flow	185.3	223.9	209.4
Return on 104.7 M, capital cost	10%	14%	

JDP:dc
January 19, 1977

June 1975

VII. OPERATING COST ESTIMATE
FOR
THOMPSON CREEK MO. PROJECT
COSTER COUNTY, IDAHO

A. DEFINITIONS

The operating cost is limited to primary crushing, fine crushing, grinding, flotation, dewatering, concentrate loadout, and haulage to Mackay, tailing berm raising, process water reclaiming, and fresh water supply. The format includes only direct costs specifically identified as: maintenance and operating supplies delivered to site; power; analytical services; maintenance and operating labor; and direct mill administration, supervision, technical and clerical personnel.

The unit cost figures are derived from an operating capacity of 7,140,000 dry short tons per year of ROM ore, and 36,128,400 dry pounds per year (excluding an estimated 5% contained flotation oil) of 92.5% molybdenite concentrate containing 20,032,100 pounds of molybdenum.

Freight and delivery costs for materials and supplies are included in the estimate. Thus the cost factors herein given reflect manufacturers' and suppliers' prices plus all freight to the plantsite. Railroad freight is accounted to the railhead at Mackay, Idaho; at that point the items are transferred to appropriate trucks for the 90-mile haul to the site on Bruno Creek. A trucking figure of $0.06 per ton-mile was estimated by Tuscarora; however, considering the additional handling for the freight transferred to trucks at the railhead, an average total transfer cost of $8.00 per ton is used for delivery to the site and unloading to plant storage of materials fob Mackay railhead. The molybdenite concentrate is backhauled to Mackay for shipment to the conversion plant; the additional cost for this backhauling operation is assumed to be $2.00 per ton fob railcar Mackay.

The Tuscarora Mining Corporation operating cost factors not included in this estimate are:

1. Development and mining operations.

2. Subsequent refining or processing of molybdenite concentrates after leaving the Mackay railhead.

3. Onsite general services, specifically administration, personnel, purchasing, accounting, warehousing, safety, security, and shops extraneous to the direct milling operations.

4. Tuscarora Mining Corporation home office administration.

5. Taxes, fees, rents, or royalties.

The sources of information for this operating cost summary are Tuscarora Mining Corporation, Cyprus Mines Corporation, Cyprus Pima Mining Company, Kaiser Engineers staff, and equipment manufacturers and materials suppliers. The major factors used are itemized in this text, and the estimate summary is presented in Table VII-A, Concentrator Operating Cost Estimate, at the end of this section.

B. MAINTENANCE SUPPLY FACTORS

The crushing and grinding liner costs are projected from: the unit consumption of steel used, percent scrap, current foundry prices, and delivery to the site. These costs are itemized as follows:

Liner Description	Pounds Per Ton of Ore*	Cents Per Pound	Cents Per Ton
Primary Crusher	0.0083	36.0	0.299
Standard Crusher	0.0107	37.9	0.406
Shorthead Crusher	0.0178	36.1	0.643
Rod Mill	0.0671	38.2	2.563
Primary Ball Mills	0.0571	45.0	2.567
Regrind Mills	0.0037	40.0	0.148

*Includes metal wear and scrap.

The slurry pump maintenance parts are projected at 25% per year of the initial pump prices. The maintenance parts costs for the crushers and grinding mills (excluding liners), and the belt conveyors are projected at 1.5% per year of the initial equipment prices. The maintenance parts costs for the remainder of the mechanical equipment are projected at percentages of the initial equipment prices per year based on manufacturer and experience factors.

C. OPERATING SUPPLY FACTORS

The major operating supplies, grinding media and flotation reagents, are calculated on a per item price consumption basis. The grinding media costs are derived from the projected unit consumption of the iron, current material costs, and delivery to the site. These costs are itemized as follows:

Grinding Media	Pounds Per Ton of Ore	Cents Per Pound	Cents Per Ton
Rods	0.60	17.5	10.50
Balls - 2" Cast (Primary Mills)	1.17	20.05	23.46
Balls - 1-1/2" Cast (Regrind Mill)	0.05	20.05	1.00

The reagent costs are derived from projected unit consumption, current material costs, and delivery to the site as follows:

Reagents	Pounds Per Ton of Ore	Cents Per Pound	Cents Per Ton
Lime	1.5	2.5	3.75
Vapor Oil	0.56	10.8	6.05
Pine Oil	0.11	34.9	3.84
Dowfroth 250	0.03	44.4	1.33
Syntex	0.04	44.3	1.77
Sodium Cyanide	0.08	34.9	2.79
Sodium Hydroxide*	0.18	27.3	4.91
Phosphorous Pentasulfide*	0.12	39.9	4.79
Sodium Silicate	0.38	7.7	2.93

*Nokes reagent components.

Fuel costs for heating purposes are estimated separately and included in operating supplies for each facility. The fuel is No. 2 diesel at $0.36 per gallon delivered to the plant storage tank. Miscellaneous operating supplies inclusive of lubricants are projected at 1% of the major equipment purchase price for all items except belt conveyors, cranes, and major crushing and grinding equipment; these exceptions are projected at 0.5% of purchase price.

D. POWER COST

Power costs are calculated into the estimate on the basis of connected operating horsepower. The power cost was obtained from Tuscarrora at $0.025 per kilowatt hours. Lighting costs are included at 5% of total mechanical connected horsepower.

E. WATER COST

For this operating cost estimate, the water costs are treated as a separate plant operational entity; that is, the water costs are spread against labor, supplies, and power rather than distributed to the plant operational divisions. This format permits a better examination of the components, and the option remains to distribute the total water costs to grinding and flotation functions if required. The water costs are kept separate for well water and reclaim water. Also, this keeps water reclaim costs separate from the tailing dam lifting account, which is a major operating cost and will require separate analysis.

F. LABOR COST

Labor costs are calculated on an annual cost per shift basis from a pay schedule received from Tuscarora plus 23% for payroll burden. The work shifts* are classified as continuous and non-continuous, defined as follows:

1. Continuous (Cont.) - for tasks based on a 3 shift per day and a 7 day per week schedule. This four-crew concept results in one overtime shift per 21 shifts of work performed. The overtime cost is calculated into the annual shift costs for this work classification.

2. Noncontinuous (N. C.) - for tasks based on one shift per day and a 5 day per week schedule. Note that the crushing plants are operating 2 shifts per day and 7 days per week; though actually continuous, these tasks are classified as noncontinuous because overtime shifts do not occur with the three-crew manning arrangement.

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*Work shifts are not to be confused with individual workmen, as each man is nominally scheduled for an 8-hour day 5 days per week, whereas the estimate accounting is on a task concept, for better definition.

Based on advice from Tuscarora and expanded to suit all necessary job categories, the labor pay schedule is:

Shift Duty Description	Hourly Base Pay ($)	Job Classification	Gross Annual Cost ($)
Instrumentman	$6.58	N. C.	$16,510
Electrician	6.58	Cont.	18,020
Electrician	6.58	N. C.	16,510
Dozer Operator	6.58	N. C.	12,280 (2)
Repairman	6.46	Cont.	17,690
Repairman	6.46	N. C.	16,210
Carpenter/Painter	6.46	N. C.	16,210
Belt Repairman	6.46	N. C.	16,210
Pumpman	6.46	N. C.	16,210
Concentrator or Control Rm. Operator (1)	5.77	Cont.	15,830
Crusher Operator	5.54	N. C.	14,030
Ball Mill Operator	5.54	Cont.	15,210
Reagentman	5.54	N. C.	13,900
Sampler	5.54	Cont.	15,210
Filter Operator	5.54	Cont.	15,210
Tailing/Cycloneman	5.43	Cont.	14,920
Truck Driver	5.43	N. C.	13,620
Oiler	5.43	N. C.	13,620
Helpers - Mechanical & Operations	5.31	Cont.	14,590
Helpers - Crushing Plant	5.31	N. C.	13,450
Helpers - Mechanical & Operations	5.31	N. C.	13,320
Beltman Crushing Plant	5.31	N. C.	13,450
Fine Ore Bin Man	5.31	N. C.	13,450
Loadoutman	5.31	N. C.	13,320
Launder Movers	5.31	N. C.	9,990 (2)
Labor	5.00	N. C.	12,550

(1) Flotation operator
(2) 9 months only, for tailing berm lifting operation

The overall projected ratio of maintenance personnel to operations personnel is approximately four to five. Note that for this operating cost estimate, maintenance is treated as a separate plant operation entity; that is, the costs are spread against labor, supplies, and power rather than distributed to the plant production divisions. This format permits a better examination of the components of this plant function cost. The option remains to distribute the maintenance costs against direct operational functions if required.

G. MILL ADMINISTRATION COST

For this operating cost estimate, the supervisory, technical, and clerical staffs are limited to those involved directly in mill operations. Supporting services such as general administration, personnel, accounting, purchasing, warehousing, general plant shops, safety, and security are excluded from this operating cost estimate. The incurred mill analytical service is accounted on an average cost of $1.00 per analysis for 250 analyses per day; these are included in mill technical costs in the Miscellaneous column. If the total operation requires 400 analyses per day, the analytical laboratory will employ approximately 10 analysts and sample preparation men.

Based on advise from Tuscarora and expanded to suit position categories, the mill staff pay Schedule is:

Staff Duty Description		Annual Base Salary ($)	Gross Annual Cost ($)*
Superintentent	(one)	$ 25,000	$ 30,750
Assistant Superintendent	(one)	22,500	27,680
Metallurgist	(one)	20,000	24,600
Maintenance Foreman	(one)	16,000	19,680
Concentrator Shifters	(four)	14,000	68,880
Crushing Plant Shifters	(three)	14,000	51,660
Instrument Engineer	(one)	14,000	17,220
Ore Testing Engineer	(one)	13,000	15,990
Technicians	(two)	11,000	27,060
Clerks, Metallurgy	(two)	11,000	27,060
Clerk, Maintenance	(one)	11,000	13,530
Stenographer	(one)	9,000	11,070

* Includes 23% payroll burden.

H. TAILING DAM ROCK FILL COST

Since there is insufficient local borrows, the rock fill for the starter dam construction and the operations berm lifting is principally mine stripping and waste. The delivery is by the mine haul trucks. The cost is projected as the extra travel distance that would be incurred beyond the normal haul to the geometric center of the waste dump; there is no loading charge at the pit. The net cost for this proportional delivery is $0.1655 per cubic yard directly chargeable to the tailing disposal operation, and it is accounted as a direct mine backcharge in this estimate. This cost is comprised of $0.0675 per cubic yard per mile for 2.2 miles plus $0.017 per cubic yard for road maintenance.

TABLE VII-A

CONCENTRATOR OPERATING COST ESTIMATE

Tuscarora Mining Corporation: Thompson Creek Project
Annual Production: 7,140,000 Short Tons ROM Ore; 36,128,400 lb Molybdenite Concentrate; on a 357 day per year schedule.

Plant Functions	Man Shifts Per Week (1)	Labor	Supplies: Maintenance	Supplies: Operating	Power	Misc.	Total	Unit Costs Per Ton-Ore	Unit Costs Per lb-Conc.
Primary Crushing and Overland Conveying	45	$ 122,790	$ 70,720	$ 16,440	$ 294,090	$ -	$ 504,040	$0.0706	$0.0139
Fine Crushing and Storage	45	122,790	124,390	13,900	398,580	-	659,660	0.0924	0.0183
Primary Grinding	42	119,200	436,640	2,437,830	1,440,410	-	4,434,080	0.6210	0.1227
Flotation	83	235,420	127,590	2,474,050	872,250	-	3,709,310	0.5195	0.1026
Dewatering and Loadout	31	87,480	11,940	3,940	28,570	38,030 (7)	169,960	0.0238	0.0047
Tailings Disposal and Berm Lifting	21 / 25 (2)	163,930	91,720	174,950	30,660	546,150 (5)	1,007,410	0.1411	0.0279
Process Water Reclaim	5 (3)	16,210	4,300	1,180	20,970	-	42,660	0.0060	0.0012
Fresh Water (From Wells)	5 (3)	16,210	25,000	2,600	41,950		85,760	0.0120	0.0024
Maintenance	251	626,620	10,000	5,000	6,000	-	647,620	0.0907	0.0179
Supervision and Technical Services (4)	95	335,180	47,500	47,500	1,000	89,250 (6)	520,430	0.0729	0.0144
Total Annual Costs		$1,845,830	$949,800	$5,177,390	$3,134,480	$673,430	$11,780,930	$1.6500	$0.3260

NOTES

(1) All shifts are on the 357 day per year or 12 month per year basis except as noted.

(2) These shifts are on a 9 month per year basis due to berm lifting weather problems.

(3) Duties distributed between both plant functions to assure daily coverage and preventive maintenance.

(4) Includes metallurgical laboratory costs and plant instrumentation hardware.

(5) Mine back charge for delivering 3,300,000 cubic yards per year of stripping and waste to the tailing berm at $0.1655 per cubic yard for the additional haul of 2.2 miles over normal mine waste disposal.

(6) Analytical laboratory work is included at an estimated $1.00 per average analysis.

(7) Concentrate haul cost to Mackay fob rail car at $2.00 per ton, backhaul for reagents and supplies from Mackay to concentrator site.

MANNING SCHEDULE

	12 Month	9 Month (2)
Operating men	54	5
Maintenance men	47	-
Total	101	5

If replacement requirements are 10% for vacations and absenteeism, then total mill labor payroll is 117 employees.

Total staff employees, direct supervision, technical and clerical - 19.

MOLYBDENUM PRICE HISTORY

$/lb. Mo

January 1, 1974	$1.72
March 1, 1974	$1.87
July 1, 1974	$2.05
September 1, 1974	$2.30
January 1, 1975	$2.43
October 1, 1975	$2.62
February 29, 1976	$2.90
August 29, 1976	3.20
DECEMBER 31, 1976	3.45

MARCH 9, 1977

TO: TOM RIGGS,
ANVIL, VANCOUVER.

RE: MOLYBDENUM

TALKING TO CLIFF MARK, OUR CHIEF GEOLOGIST, IT APPEARS THAT CLIMAX IN US AND NORANDA IN CANADA CONTROL A SIZEABLE PORTION OF MOLYBDENUM MARKET. PRICE IN US IS \$ ~~3.40~~ 3-40 AND CANADA \$3.20/LB BECAUSE OF CANADIAN GOVT CONTROL BUT CANADA MAY NOT EXPORT MOLY TO THE U.S.

ATTACHED ARE TABLES AND A FEASIBILITY STUDY ON THOMPSON CREEK.

PLEASE DO NOT PUBLISH THIS INFORMATION.

REGARDS

LEONEL.

to the changes in [illegible] taken place since 1970 when we [illegible] of the attached Chart !!

MAJOR MOLYBDENUM DEPOSITS AND PROSPECTS IN 1970
THE UNITED STATES AND CANADA

Return CAMARK.

Property Name	Location	Ownership	Year Estimate Made	Reserves Millions of tons	GRADE % Mo Average	Pounds Mo Per Ton	Cutoff Grade % Mo	Gross $ Per Ton Average	Total Gross $ Value (Millions)	Mining Method	Stripping Ratio	Milling Tons Per Day	Pounds Mo Per Year (Millions)
Thompson Creek	Idaho	Cyprus	1970	135.0	0.124	2.48	0.050	4.27	576.5	Open Pit	2:1	25,700	20.0
			1970	18.0	0.231	4.62	0.150	7.95	14.3	Underground		10,000	14.9
Adanac[1]	British Columbia	~~Kerr Addison~~	1969	69.9	0.085	1.70	0.048	2.92	204.1	Open Pit		25,000	7.1
B.C. Moly (Alice Arm)	British Columbia	~~Kennecott~~	1968	40.2	0.138	2.76		4.75	109.9	Open Pit	2:1	6,000	5.0
Boss Mtn.	British Columbia	Noranda	1969	3.0	0.240	4.80		8.26	24.8	Underground		1,500	2.3
Climax	Colorado	Amax	1966	400.0	0.240	4.80		8.26	3,302.4	Block Caving		>43,000	57.0[3]
Endako	British Columbia	Placer	1964	66.5	0.126	2.52		4.33	288.2				
			1969	230.0	0.090	1.80	0.048	3.10	735.3	Open Pit	0.8:1	27,000	14.0
Henderson[2]	Colorado	Amax	1969	303.0	0.294	5.88		10.11	3,064.4	Block Caving		30,000	50.0
Questa	New Mexico	~~Moly Corp.~~	1964	20.5	0.178	3.56		6.12	125.5	Open Pit	3:1	8,000	
			1968	157.0	0.112	2.24		3.85	604.9	Open Pit	2.5:1	11,000	
			1969	232.6	0.097	1.94	0.048	3.34	642.9	Open Pit	6.4:1	16,500	10.0
Urad	Colorado	Amax	1967	12.0	0.240	4.80		8.26	99.1	Block Caving		5,000	[3]
TOTAL				1,454.7	0.181	3.61		6.22					

ed this up in March 1976 — Cyprus evaluated in 1970-72 & turned it down. Too low grade in this poor location.
Amax
Cyprus had the Boss Mtn. only stored in 1969-70 but dropped when Noranda [illegible] the mine in 1971. [illegible]
Now developing a large open pit - underground operation - 1976-77
native.

1/ To be placed in production in mid-1973
2/ To be placed in production in 1975
3/ Climax and Urad Mo production are combined

Grade tons
T. Creek 100 m T @ 0.17 - 17

Climax
BC Moly 40 M @ .138 5.5
Colo 400 @ .24 96
Henderson 300 @ .29 87 188.5

Noranda Adanac 70 @ .085 6.3
Boss 3 @ .24 .72
Endako 234 @ .09 21.5 28.5
KCC + Moly Co 232 @ .10 23 25

76
3.45 - 2.62
3.45

w/R Feb 18 ...
3.20 - 3.45

II. ECONOMICS

Cash flow calculations are based on the following controls and assumptions:

Production Start Up	- 3 years after go ahead decision
Annual Milling Rate	- 7,000,000 tons ore
	20,000 tons per day for 350 days per year
Stripping ~~Stripping~~ Ratio	5 tons waste per ton ore
Grade of Ore	0.152% Mo
Metallurgical Recovery	90% Mo
Metal Price	$1.87/lb. Mo (FOB on railcars Mackay)

Operating Costs

Mining	24 - 32¢/ton material depending on depth in mine pit
Milling	88¢/ton ore x2 = 1.76 x 150 = $2.64
G & A	$2.5 million per year (equivalent to 35.7¢ per ton ore)

Capital Costs

Production Facilities	$43,967,000
Preproduction Stripping	17,650,000
Mine Development & Testing	2,500,000
Ecological Reserve	2,000,000
Total	$66,117,000

- Additional capital required in:

Year + 1	$ 2,940,000
Year + 5	$ 550,000
Working capital requirements	$ 6,250,000

Tax Allowances

- Depreciation - Straight line over 15 years.
- Depletion - 22% of net sales or 50% of gross income, whichever is less.

Tax Levy

State

Property	1.3% of Market Value
Corporation	6.5% of State Taxable Income
Mine License	3 % of State Taxable Income

Federal

48% on Federal Taxable Income

Rate of Return

The cash flow calculations on the listed controls resulted in a 13.9% D.C.F. rate of return with a payout of $66.12 million in 5.3 years.

Breakeven Price

The breakeven molybdenum price ranged between $1.15 and $1.37 per pound:

	Years 1 - 4	Years 8 - 12
Direct Cost, $000	16,660	19,065
G & A	2,500	2,500
Property Tax	538	538
8 % Capital Cost	6,612	-
Total	26,310	22,103
Breakeven Price	1.37	1.15

Effect of Varying Price, Recovery and Stripping Ratio

Additional cash flow calculations have been made at variable prices, recoveries and stripping ratios. The results are shown in graph form.

- Price - Each 1¢ increase in price results in an increased ROI of 0.21 precentage points. Industry experts predict a 15¢/lb. price increase about mid year 1974. Such an increase would increase ROI to 17%.
- Recovery - Each 1% increase in recovery results in an increased ROI of approximately 0.38 percentage points.

- Stripping Ratio - Each 0.1 decrease in stripping ratio results in an increased ROI of approximately 0.25 percentage points.

- Comparison with September 1972 Technical Services Order of Magnitude Feasibility

Assumptions:

	April 1974	September 1972
Annual Milling Rate		
Tons ore	7,000,000	7,000,000
Tons per day	20,000	20,000
Cutoff Grade, % Mo	0.080	0.080
Mill Heads, % Mo	0.152	0.148
Stripping Ratio, tons waste per ton ore	5.0	4.5
Preproduction Stripping, 000 tons	75,000	75,000
Metallurgical Recovery, % Mo	90	90
Operating Costs Total direct and indirect, $ per ton ore	2.74	2.25
Molybdenum Price, $ per pound	1.87	1.70
Capital Investment, $000		
Production facilities	43,967	40,270
Preproduction stripping	17,650	14,650
Mine development & testing	2,500	3,500
Ecological reserve	2,000	2,000
Total	66,117	60,420
Working Capital	6,250	6,000
Equity Position Equity Investment, %	100	100

Assumptions: (Continued)

	April 1974	September 1972
Tax Allowances & Levies		
Depreciation	Straight-Line Over Mine Life	Straight-Line Over Mine Life
Depletion	22% of net sales or 50% of gross income	22% of net sales or 50% of gross income
State Taxes		
Income, %	6	6.5
Mine License, %	3	3
Property	1.3% of market value	1.3% of market value
Federal Taxes, %	48	48
Payout Period, years	5.30	4.96
Return on Investment (DCF)	13.9	14.2

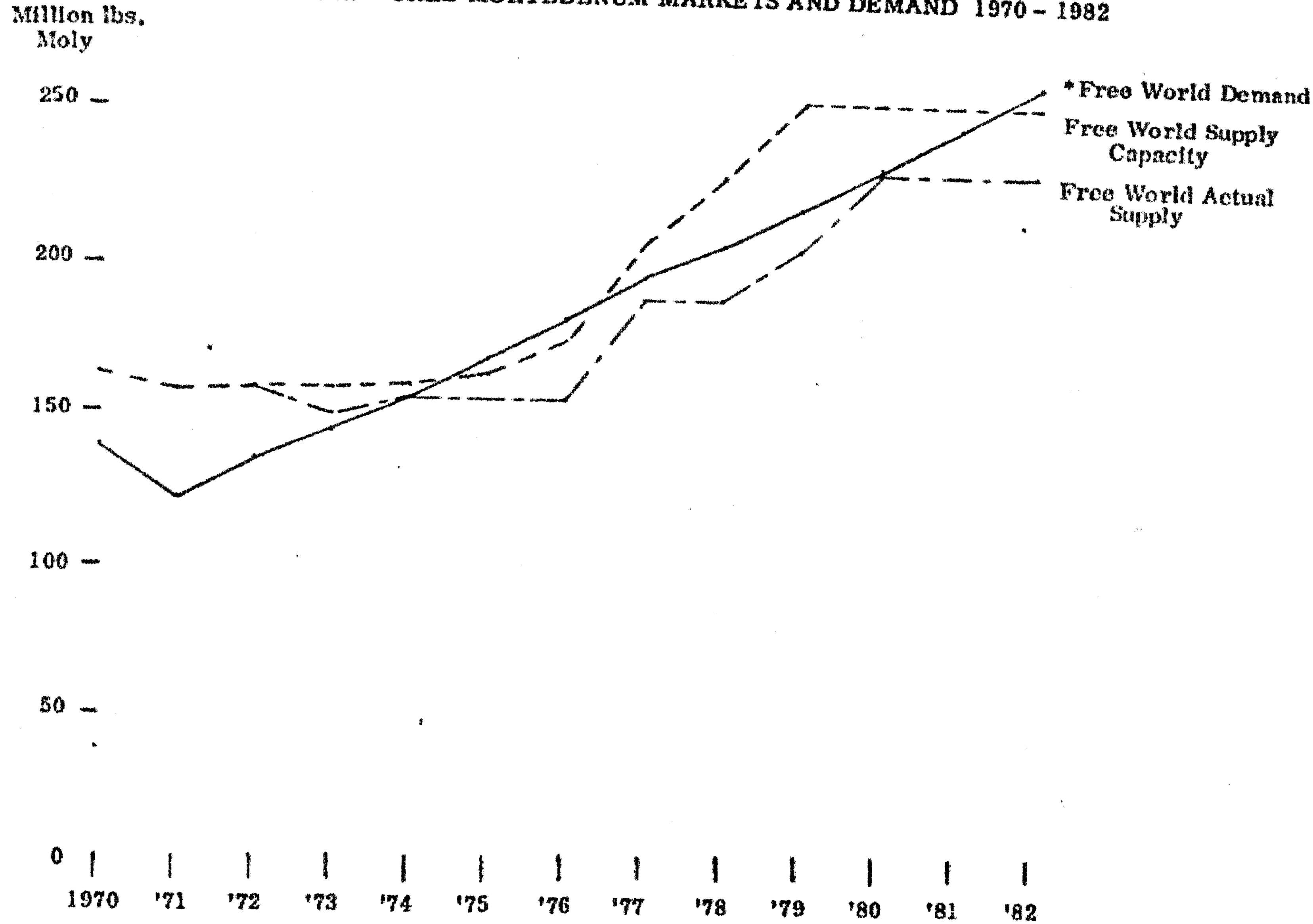
THOMPSON CREEK PROJECT

Order-of-Magnitude Feasibility

3.5 : 1 Stripping, 90% Mo Recovery, $1.87/lb. Molybdenum Price

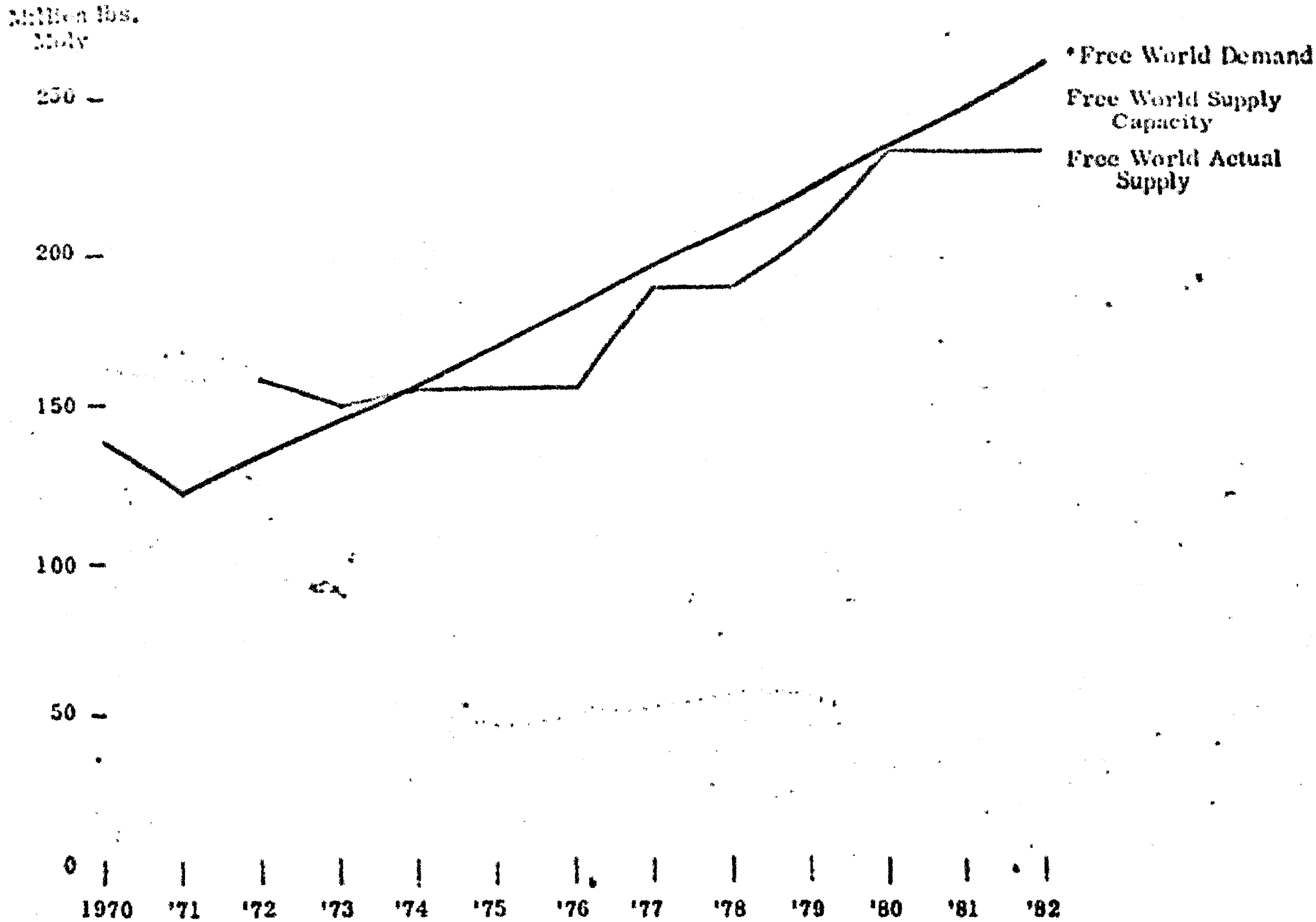
	Year-3	Year-2	Year-1	+1	+2	+3	+4	+5	+6	+7	+8	+9	+10	+11	+12	+13	+14	+15	Total
PRODUCTION DATA																			
Stripping, 000 tons				35,035	35,000	35,000	35,000	35,000	31,500	37,500	37,500	37,500	37,500	37,500	37,500	12,500	-	-	458,000
Ore, 000 tons				7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	7,000	105,000
Grade, % Mo				.152	.152	.152	.152	.152	.152	.152	.152	.152	.152	.152	.152	.152	.152	.152	
Pounds Mo Recovered, 000's				19,152	19,152	19,152	19,152	19,152	19,152	19,152	19,152	19,152	19,152	19,152	19,152	19,152	19,152	19,152	287,280
Tons Concentrate at 54%				17,733	17,733	17,733	17,733	17,733	17,733	17,733	17,733	17,733	17,733	17,733	17,733	17,733	17,733	17,733	265,995
INCOME, $000																			
Sales at $1.87/lb. Mo				35,814	35,814	35,814	35,814	35,814	35,814	35,814	35,814	35,814	35,814	35,814	35,814	35,814	35,814	35,814	537,210
Sales Expense (freight to rail)				96	96	96	96	96	96	96	96	96	96	96	96	96	96	96	1,440
Net Sales				35,718	35,718	35,718	35,718	35,718	35,718	35,718	35,718	35,718	35,718	35,718	35,718	35,718	35,718	35,718	535,770
EXPENSES, $000																			
Direct Costs																			
Stripping				8,750	8,750	8,750	8,750	9,450	10,125	10,125	10,125	10,875	10,875	10,875	10,875	3,875	-	-	132,200
Mining				1,750	1,750	1,750	1,750	1,890	1,890	1,890	1,890	2,030	2,030	2,030	2,030	2,170	2,170	2,170	29,190
Concentrating				6,160	6,160	6,160	6,160	6,160	6,160	6,160	6,160	6,160	6,160	6,160	6,160	6,160	6,160	6,160	92,400
Total Direct				16,660	16,660	16,660	16,660	17,500	18,175	18,175	18,175	19,065	19,065	19,065	19,065	12,205	8,330	8,330	243,750
Indirect Costs																			
G & A				2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	2,500	37,500
Total Direct & Indirect				19,160	19,160	19,160	19,160	20,000	20,675	20,675	20,675	21,565	21,565	21,565	21,565	14,705	10,830	10,830	281,250
Depreciation				4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	69,000
GROSS INCOME, $000																			
Income before Depletion				11,958	11,958	11,958	11,958	11,118	10,443	10,443	10,443	9,553	9,553	9,553	9,553	16,413	20,288	20,288	165,480
Depletion				5,979	5,979	5,979	5,979	5,559	5,221	5,221	5,221	4,776	4,776	4,776	4,776	7,858	7,858	7,858	87,806
Investment Credit				2,100															2,100
STATE TAXES, $000																			
Taxable Income (State)				5,879	5,979	5,979	5,979	5,559	5,222	5,222	5,222	4,777	4,777	4,777	4,777	8,555	12,430	12,430	95,661
State Income Tax at 6.5%				252	389	389	389	361	339	339	339	310	310	310	310	556	808	808	6,209
Mine License Tax at 3%				118	179	179	179	167	157	157	157	143	143	143	143	257	373	373	2,866
Property Tax at 1.2% market value				538	538	538	538	538	538	538	538	538	538	538	538	538	538	538	8,070
Total State Taxes				908	1,106	1,106	1,106	1,066	1,034	1,034	1,034	991	991	991	991	1,351	1,718	1,718	17,145
FEDERAL TAX, $000																			
Taxable Income (Federal)				2,973	4,873	4,873	4,873	4,493	4,188	4,188	4,188	3,786	3,786	3,786	3,786	7,204	10,711	10,712	78,513
Income Tax at 48%				1,427	2,339	2,339	2,339	2,157	2,010	2,010	2,010	1,817	1,817	1,817	1,817	3,458	5,141	5,141	37,679
INCOME AFTER TAX, $000				9,625	8,513	8,513	8,513	7,895	7,399	7,399	7,399	6,745	6,745	6,745	6,745	11,034	13,426	13,426	130,656
CAPITAL INVESTMENT, $000																			
Production Facilities	1,617	24,000	36,000	2,940				362											
Mine Development & Testing	2,650																		
Ecological Reserve	2,050																		
Working Capital		6,250																	
Total Capital Investment	6,317	30,250	36,000																
CASH FLOW, $000																			
Income After Tax				9,625	8,513	8,513	8,513	7,895	7,399	7,399	7,399	6,745	6,745	6,745	6,745	11,034	13,426	13,426	130,656
Depreciation				4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	4,600	69,000
Working Capital																		6,250	6,250
Total	(6,317)	(30,250)	(36,000)	11,285	13,113	13,113	13,113	11,743	11,999	11,999	11,999	11,345	11,345	11,345	11,345	15,204	18,026	24,276	232,456
CUMULATIVE CASH FLOW	(6,317)	(36,567)	(72,567)	(61,082)	(47,969)	(34,856)	(21,743)	(9,703)	2,301	14,300	26,199	37,544	48,889	60,234	71,579	86,783	104,811	129,089	

ESTIMATED FREE WORLD MOLYBDENUM MARKETS AND DEMAND 1970 - 1982



*Source: Arthur D. Little Inc.

ESTIMATED FREE WORLD MOLYBDENUM MARKETS AND DEMAND 1970 - 1982



*Source: Arthur D. Little Inc.

	1970	1971	1972	1973	1974	1975	1976	1977	1978	1979	1980	1981	1982
UNITED STATES													
Questa – Molycorp	10.1	10.1	11.0	11.0	10.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0	11.0
Thompson Creek – Cyprus	–	–	–	–	–	–	–	10.0	15.0	15.0	15.0	15.0	15.0
Climax – Amax	59.0	53.0	53.0	53.0	55.0	55.0	55.0	55.0	55.0	55.0	55.0	55.0	55.0
~~Urad – Amax~~ Moly Byprod.	7.0	7.0	7.0	6.0	–	–	–	–	–	–	–	–	–
~~Henderson – Amax~~	–	–	–	–	–	–	–	15.0	30.0	50.0	50.0	50.0	50.0
Utah et al – Kennecott	17.0	13.4	14.0	15.0	15.0	15.0	18.0	18.0	18.0	18.0	18.0	18.0	18.0
Sierrita – Duval	5.0	12.0	12.0	14.0	14.0	14.0	14.0	14.0	14.0	14.0	14.0	14.0	14.0
Mineral Park – Duval	3.5	3.5	3.2	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5
Esperanza – Duval By Product	2.0	2.0	–	1.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
San Manuel – Newmont	3.6	3.8	4.9	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0
Twin Buttes – Anaconda	.3	1.0	2.1	2.5	3.2	3.1	2.7	3.5	3.5	3.5	3.5	3.5	3.5
Pima – Cyprus	1.5	1.2	1.1	1.5	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Bagdad – Cyprus				.5	.5	.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Mission – Asarco	1.2	1.2	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Silver Bell – Asarco	.4	.4	.2	–	.2	.5	.5	.5	.5	.5	.5	.5	.5
Inspiration – Inspiration	.5	.3	–	.3	.5	.5	.5	.5	.5	.5	.5	.5	.5
Other				.2	.6	.6	.8	1.0	1.0	1.0	1.0	1.0	1.0
Total U.S.	111.1	108.9	110.0	115.0	113.0	114.2	118.0	144.0	164.0	184.0	184.0	184.0	184.0
CANADA X Amax bought in 1972													
Alice Arm – Kennecott	6.1	5.1	1.1	–	–	–	–	–	–	–	–	–	–
Endako – Noranda – Placer	18.2	14.5	11.0	12.0	15.0	15.0	15.0	18.0	18.0	18.0	18.0	18.0	18.0
Brenda – Noranda	8.1	9.5	10.4	10.1	9.3	9.0	9.0	9.0	9.0	9.0	9.0	9.0	9.0
Gibraltar – Noranda – Placer	–	–	0.5	1.0	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Lornex – Rio Algom	–	–	0.6	1.5	2.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0
Island Copper – Utah Int'l.	–	–	1.2	1.6	1.8	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Highmont	–	–	–	–	–	–	3.2	5.3	5.3	5.3	5.3	5.3	5.3
Valley Copper	–	–	–	–	–	–	2.0	4.0	4.0	4.0	4.0	4.0	4.0
~~Gaspe – Noranda~~	0.4	0.4	0.4	0.8	1.2	1.2	1.2	1.5	1.5	1.5	1.5	1.5	1.5
Other	5.8	3.1	0.5	–	.2	.3	1.1	1.7	1.7	1.7	1.7	1.7	1.7
Total Canada	38.6	32.6	25.7	27.0	31.0	32.0	38.0	46.0	46.0	46.0	46.0	46.0	46.0
OTHER													
Chile	10.8	13.1	13.4	13.0	13.0	14.0	15.0	15.0	15.0	18.0	18.0	18.0	18.0
Peru	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Other	1.3	1.3	1.3	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5
Total Other	13.6	15.9	16.2	16.0	16.0	17.0	18.0	18.0	18.0	21.0	21.0	21.0	21.0
TOTAL	163.3	157.4	151.9	158.0	160.0	163.2	174.0	208.0	228.0	251.0	251.0	251.0	251.0

Source: Molycorp

and NWC listed in Section 4 (a),
4 (b) (ii) were paid by the Company during Year 1 to its employees when they worked.

5. For Year 2 of the Collective Agreement compensation shall be paid and provided as follows:

(a) Wages - Subject to Section 3 of this Part II the wage rates provided for in Article 12 of Part III which incorporates Appendix "A" shall not apply and in their stead the Company shall pay the following rates:

Year 2 - Wage Rates

1.	Labourer	6.280
2.		6.404
3.	Coal crusher helper, blaster helper II, crusher helper, Labourer (Coal Mine), filter helper (sectors)	6.528
4.	Coal crusher operator, pump man, equipment operator under 75 h.p.	6.652
5.	Reagent operator II, load out helper, primary crusher operator, tradesman's helper, mill repair helper	6.776
6.	Tailings operator, service truck driver	6.900

7.	Shift assayer, bucker, dryer operator miner helper (Coal Mine)	7.024
8.	Grinding operator, secondary crusher operator, reagent operator I, rotary driller trainee, blaster helper I, secondary driller, lube serviceman II, heating plant operator - 4th class	7.148
9.	Filter Operator II, trammer (Coal Mine)	7.272
10.	Tireman I, filter operator I, HD truck driver - under 100 tons, equipment operator II, tool crib attendant (mill & shop)	7.396
11.	HD truck driver - over 100 tons, heating plan operator - 3rd class.	7.520
12.	Load out operator, mill repairman II, HD mechanic II, machinist II, electrician II, welder II, gas mechanic II, flotation operator II, mill & mechanical lube serviceman I, HD truck driver - 150 ton & over, equipment operator I, tradesman II, instrument mechanic II, equipment operator & maintenance man (Coal Mine), miner (Coal Mine)	7.644
13.	Rotary driller, blaster	7.768
14.	Heating plant operator (prov. 2nd class)	7.892
15.	HD mechanic I, instrument mechanic I, machinist I, welder I, flotation operator I, electrician I, mill repairman I, gas mechanic I, tireman, tradesman I.	8.016

16.		8.140
17.	Shovel operator, chief mill operator, journeyman certified in Canada, heating plant operator - 2nd class permanent, mobile crane operator - 25 tons and over, drag line operator	8.264
18.		8.388
19.	Lead hands	8.512

(b) <u>Non-Wage Compensation (NWC)</u>

Subject to Section 3 of this Part II, the Cost of Living Allowance provided for in Section 11.26 of Part III in so far as it incorporates Appendix "E" shall not apply, but all other NWC provided for in Part III shall apply.

(ii) Subject to Section 3 of this Part II, the NWC$ paid by the Company on September 30, 1975, pursuant to the Agreement executed on March , 1973 shall apply insofar as the matters listed in Section 4(b)(i) above are concerned.

(iii) For greater certainty the rates of pay and NWC listed in Section 4 (a) and 4 (b) (ii) were paid by the Company during Year 1 to its employees when they worked.

5. For Year 2 of the Collective Agreement compensation shall be paid and provided as follows:

(a) Year 2 - Wages - Subject to Section 3 of this Part II, the wage rates provided for in Article 12 of Part III, which incorporates Appendix "A", shall not apply and in their stead the Company shall pay the following rates:

Year 2 - Wage Rates

JOB CLASS		HOURLY RATE
1	Switchboard Operator/ Receptionist Clerk Typist II Accounting Clerk III Kardex Clerk II	$6.280

Year 2 - Wage Rates (continued)

JOB CLASS		HOURLY RATE
2	Clerk Typist I Key Punch/Machine Operator Accounting Clerk II Warehouseman III	$6.466
3	Engineering Assistant Accounting Clerk I Secretary	6.652
4	Kardex Clerk I Guard Operations Clerk Warehouseman II	6.838
5		7.024
6	Shipper/Receiver II	7.210
7	Shipper/Receiver I	7.396
8		7.582
9	Paymaster Technician II Assayer II Senior Warehouseman	7.768
10		7.954
11	Technician I Assayer I Accountant I Engineering Draftsman	8.140
12		8.326
13	Senior Assayer Senior Met. Technician	8.512

(b) Co-Operative Wage Study (CWS)

The parties agree to implement CWS effective to the date of the signing of this agreement. The parties agree to use the necessary portion of the allotted 77 cents referred to in Appendix "A" of Part III that is necessary from time to time to implment the CWS program.

The parties recognize the principle of parity as specified in Recitals 3 and 4, and Mr. Pepin's telex of August 13, 1976.

The Company will submit AIB-2 forms, or such other forms as

Milling	1974 18,000 TPD.		1976 18,000 TPD.	Increase	
Labour.	19.94	1.50	29.91 ✓	9.97	Could use 1.33 factor
Steel	41.26		67.15. ✓	25.89.	
Flot. Reagents	18.53.		40.2 ✓	21.37.	
Power.	35.04 × 45/1.83		85.23 ✓	50.19.	
Mill Supplies	6.00 × 1.7		10.20. ✓	4.20	
Heating	1.95 × 1.7		3.32. ✓	1.37	
	1.2302		236.01.	or 1.92 times	

Milling 1976 = $2.36 per ton.

G & A Costs

0.4546 $ per ton × 1.75 = $0.7956 per ton.

Total G & A + Milling = $3.16 per ton.

Mining 1.24

$4.40 per ton total property.

reduce by 22¢ per ton for purchase power.
" " 3¢ " " " Lime

Adanac   Assumed $100 per ton Frt.

Reagents.	FOB Van. 1976 Unit Prices.	New. c/w Frt 5¢ per lb.	Consumption lbs/ton	CAMC.	¢ per ton		
Lime	$44/ton	7.2	1.1	5.25¢/lb	7.92	5.78	* Cyprus Anvil cost. 4.5¢ per ton.
Syntex	55¢ per lb.	60	.01		0.60		
Sod Sulfide.	22¢ per lb.	45	.4	18.04	18.00	7.20	41.87¢ per lb Houston.
Sod Silicate	8.2 "	13.2	.05		0.66		
Shell Cornea 21	32	37	.22		8.14		
Dowfroth 250	70	75	.065		4.88		
Super Floc	255.	260.					
were 15.80.					40.20.		

USE 40.2¢ per ton

Steel.	Units Consumption	Cost ¢ w Frt. 5¢ per lb.	¢ per ton.
Primary Crusher.	0.01	75	0.75
Secondary Crushers.	0.05	75	3.75
Rod Mill	0.052	65	3.38
Ball Mill	0.055	45	2.48
Rods	1.02	20.5	20.91
Balls	1.30	20.5	26.65
Regrind	0.45	20.5	9.23

Total Steel use   67.15¢ per ton.

Power Costs.

Study = $0.7832 per ton × $\frac{4.5}{1.85}$ = 190.5 ¢ per ton.

Lighting

Study $0.0630 per ton × $\frac{4.5}{1.85}$ 15.3 ¢ per ton.

TABLE XVI -1

PERSONNEL

SUMMARY

Rates escalated to 1974	Number	Base Month	Cost Year $	Fringe Cost/Yr $	Total Cost/Yr $	Cost/T Milled $
A. 15,000 TPD Mill						
Administration	19	18,850	226,200	17,800	244,000	0.0465
General Engineering	8	7,450	89,400	6,600	96,000	0.0183
Mining	107	100,541	1,206,492	169,569	1,376,061	0.2621
Milling	103	91,670	1,100,039	144,040	1,244,079	0.2370
Plant Services	46	42,626	511,512	55,047	566,559	0.1079
Townsite	10	9,016	108,192	14,589	122,781	0.0234
	293	270,153	3,241,835	407,645	3,649,480	0.6951
B. 18,000 TPD MILL						
Administration	19	18,850	226,200	17,800	244,000	0.0387
General Engineering	8	7,450	89,400	6,600	96,000	0.0152
Mining	115	107,919	1,295,028	182,849	1,477,877	0.2345
Milling	104	92,533	1,110,395	145,593	1,255,988	0.1994
Plant Services	46	42,626	511,512	55,047	566,559	0.0899
Townsite	10	9,016	108,192	14,589	122,781	0.0195
	302	278,394	3,340,727	422,478	3,763,205	0.5973

TABLE XVI -1 Cont'd

SUMMARY cont'd

	Number	Base Month	Cost Year $	Fringe Cost/Yr $	Total Cost/Yr $	Cost/T Milled $
C. 20,000 TPD Mill						
Administration	19	18,850	226,200	17,800	244,000	0.0348
General Engineering	8	7,450	89,400	6,600	96,000	0.0137
Mining	123	115,297	1,383,564	196,129	1,579,693	0.2257
Milling	104	92,533	1,110,395	145,593	1,255,988	0.1794
Plant Services	46	42,626	511,512	55,047	566,559	0.0809
Townsite	10	9,016	108,192	14,589	122,781	0.0175
	310	285,772	3,429,263	435,758	3,865,021	0.5521

PROPOSED WORK SCHEDULE FOR

3 SHIFT X 7 DAY OPERATION

13 X 29 DAY PERIODS

FOUR CREWS

Each man works 21 days out of 28 (42 hr week)

	1 Su	2 M	3 T	4 W	5 Th	6 F	7 S	8 Su	9 M	10 T	11 W	12 Th	13 F	14 S	15 Su	16 M	17 T	18 W	19 Th	20 F	21 S	22 Su	23 M	24 T	25 W	26 Th	27 F	28 S
NITE SHIFT 12 M - 8 A	-	-	-	-	-	-	-	S	S	S	-	-	-	-	-	-	-	S	S	-	-	-	-	-	-	-	S	S
DAY SHIFT 8A - 4 P	-	-	-	-	S	S	-	-	-	-	-	-	-	S	S	S	-	-	-	-	-	-	-	S	S	-	-	-
AFTERNOON SH. 4P - 12 M	-	S	S	-	-	-	-	-	-	-	S	S	-	-	-	-	-	-	-	S	S	S	-	-	-	-	-	-

TABLE XVI -2

PERSONNEL

DETAIL

Classification	Number	Unit Rate	Base Cost Month	Base Cost Year	Fringe Cost/Yr	Total Cost/Yr	Cost/T Milled
1. ADMINISTRATION							
Head Office			$3,000	$36,000	-	$ 36,000	
Mine Office							
Manager	1	2,300	2,300	27,600	7,400	35,000	
Assistant Manager	1	1,800	1,800	21,600	3,100	24,700	
Chief Accountant	1	1,200	1,200	14,400	500	14,900	
Personnel-PR Director	1	1,100	1,100	13,200	500	13,700	
Purchasing Agent	1	1,000	1,000	12,000	500	12,500	
Payroll Accountant	1	800	800	9,600	500	10,100	
Stores Accountant	1	800	800	9,600	500	10,100	
Stores & Shipping Clerk	2	700	1,400	16,800	800	17,600	
Warehouseman	4	650	2,600	31,200	1,600	32,800	
Timekeeper	1	650	650	7,800	400	8,200	
Steno	4	450	1,800	21,600	1,600	23,200	
Receptionist	1	400	400	4,800	400	5,200	
	19		$18,850	$226,200	$17,800	$244,000	$0.0465

PERSONNEL cont'd

Classification	Number	Unit Rate	Base Cost Month	Base Cost Year	Fringe Cost/Yr	Total Cost/Yr	Cost/T Milled
2. GENERAL ENGINEERING							
Chief Engineer	1	1,500	1,500	18,000	2,500	20,500	
Geologist	1	1,200	1,200	14,400	1,500	15,900	
Pit Engineer	1	1,100	1,100	13,200	500	13,700	
Surveyor	1	950	950	11,400	500	11,900	
Surveyor Helper	1	750	750	9,000	400	9,400	
Mine Clerk	1	700	700	8,400	400	8,800	
Draftsman	1	650	650	7,800	400	8,200	
Technician	1	600	600	7,200	400	7,600	
	8		$7,450	$89,400	$6,600	$96,000	$0.0183

PERSONNEL cont'd

Classification		Number	Unit Rate	Base Cost Month	Base Cost Year	Fringe Cost/Year	Total Cost/Year
3. MINING							
Mine Superintendent		1	1,600	1,600	19,200	2,600	21,800
Pit Gen. Foreman		1	1,300	1,300	15,600	500	16,100
Pit Shift Boss		4	1,200	4,800	57,600	2,000	59,600
Equipt. Operations Trainer		1	1,100	1,100	13,200	500	13,700
Blasting Foreman		1	925	925	11,100	500	11,600
Shovel Operation		4	6.36	4,274	51,288	7,693	58,981
Maint. Lead Hand		4	6.34	4,260	51,120	7,668	58,788
HD Mechanic	grd. 1	4	6.11	4,106	49,272	7,391	56,663
Electrician	grd. 1	5	6.11	5,132	61,584	9,238	70,822
Welder	grd. 1	6	5.98	6,028	72,336	10,850	83,186
HD Mechanic	grd. 2	4	5.74	3,857	46,284	6,943	53,227
Tireman		1	5.74	964	11,568	1,735	13,303
Rotary Driller		8	5.62	7,553	90,636	13,595	104,231
Shovel Helper		4	5.49	3,689	44,268	6,640	50,908
Pit Truck Operator		20	5.49	18,446	221,352	33,203	254,555
* Dozer Operator		4	5.49	3,689	44,268	6,640	50,908
* Grader Operator		4	5.49	3,689	44,268	6,640	50,908
* Loader Operator		4	5.49	3,689	44,268	6,640	50,908
Welder	grd. 2	2	5.49	1,845	22,140	3,321	25,461
Lube Serviceman		1	5.25	882	10,584	1,588	12,172
Secondary Driller		1	5.14	863	10,356	1,553	11,909
* Sand Truck Operator		4	5.14	3,454	41,448	6,218	47,666
Drill Helper		8	4.51	6,061	72,732	10,910	83,642

* Operators interchange seasonably as required.

PERSONNEL cont'd

Classification	Number	Unit Rate	Base Cost Month	Base Cost Year	Cost/Year	Cost/Year	Milled
3. MINING cont'd							
Blaster Helper	3	4.51	2,273	27,276	4,091	31,367	
Maintenance Helper	4	4.51	3,031	36,372	5,456	41,828	
Dry Attendant	4	4.51	3,031	36,372	5,456	41,828	
TOTAL MINING	107		$100,541	$1,206,492	$169,569	$1,376,061	$0.2621

PERSONNEL cont'd

Classification	Number	Unit Rate	Base Cost Month	Base Cost Year	Fringe Cost/Year	Total Cost/Year	Cost/T Milled
4. MILLING							
Supervision							
Mill Superintendent	1	1,600	1,600	19,200	2,600	21,800	
Chief Metallurgist	1	1,250	1,250	15,000	1,500	16,500	
Mill Foreman	1	1,200	1,200	14,400	500	14,900	
Shift Foreman	4	1,000	4,000	48,000	2,000	50,000	
Crusher Foreman	1	1,000	1,000	12,000	500	12,500	
Repair Foreman	2	1,000	2,000	24,000	1,000	25,000	
Metallurgist	1	950	950	11,400	500	11,900	
	11		$12,000	$144,000	$8,600	$152,600	$0.029
Assay Office							
Chief Chemist	1	1,100	1,100	13,200	500	13,700	
Shift Chemist	4	900	3,600	43,200	2,000	45,200	
Technician	4	5.14	3,454	41,448	6,217	47,665	
Sampler	1	4.77	801	9,612	400	10,012	
Helper	1	4.51	758	9,096	400	9,496	
	11		$ 9,713	$116,556	$9,517	$126,073	$0.0240

PERSONNEL cont'd

Classification	Number	Unit Rate	Base Cost Month	Base Cost Year	Fringe Cost/Year	Total Cost/Year	Cost/T Milled
4. MILLING cont'd							
Operating Labour							
Chief Mill Operator	4	5.74	3,857	46,284	6,943	53,227	
Grinding Operator	4	5.38	3,615	43,380	6,507	49,887	
Flot. Operator	4	5.38	3,615	43,380	6,507	49,887	
Filter-Dryer-Leach Op.	4	5.14	3,454	41,448	6,217	47,665	
Conc. Loader Operator	1	5.14	863	10,356	1,553	11,909	
Primary Crusher Op.	2	5.14	1,726	20,712	3,107	23,819	
Secondary Crusher Op.	4	5.14	3,454	41,448	6,217	47,665	
Tailings Operator	8	4.77	6,410	76,920	11,538	88,458	
Reagent Man	1	4.77	801	9,612	1,442	11,054	
Packer	4	4.51	3,030	36,360	5,454	41,814	
Crusher Helper	4	4.51	3,030	36,360	5,454	41,814	
Labourer	8	4.41	5,927	71,124	10,669	81,793	
	48		$39,782	$477,384	$71,608	$548,992	$0.1046

PERSONNEL cont'd

Classification		Number	Unit Rate	Month	Year	Fringe Cost/Year	Cost/Year	Cost/T Milled
4. MILLING cont'd								
Maintenance Labour								
Instrument Technician		1	6.11	1,026	12,312	1,847	14,159	
Electrician	grd.1	4	6.11	4,106	49,271	7,391	56,662	
Welder	grd.1	4	5.98	4,018	48,216	7,232	55,448	
Mill Repairman	#1	4	5.74	3,857	46,284	6,943	53,227	
Electrician	grd.2	4	5.74	3,857	46,284	6,943	53,227	
Welder	grd.2	4	5.49	3,689	44,268	6,640	50,908	
Mill Repairman	#2	4	5.14	3,454	41,448	6,217	47,665	
Mill Repairman	#3	4	4.77	3,205	38,460	5,769	44,229	
Labourer		4	4.41	2,963	35,556	5,333	40,889	
		33		30,175	362,099	54,315	416,414	$0.0793
TOTAL MILLING		103		$91,670	$1,100,039	$144,040	$1,244,079	$0.2370

PERSONNEL cont'd

Classification	Number	Unit Rate	Base Cost Month	Base Cost Year	Fringe Cost/Year	Total Cost/Year	Cost/T Milled
5. PLANT SERVICES							
Plant Engineer	1	1,600	1,600	19,200	1,000	20,200	
Electrical Supt.	1	1,400	1,400	16,800	500	17,300	
Shop Foreman	1	1,100	1,100	13,200	500	13,700	
Electrical Foreman	1	1,100	1,100	13,200	500	13,700	
Surface Foreman	1	1,000	1,000	12,000	500	12,500	
Safety Engr. &Security Chf.	1	1,000	1,000	12,000	500	12,500	
Medical Officer	1	1,000	1,000	12,000	1,000	13,000	
Power & Heat Plant Fore.	1	1,000	1,000	12,000	500	12,500	
Maintenance Sched. Cont.	2	850	1,700	20,400	1,000	21,400	
First Aid Attendant	4	800	3,200	38,400	2,000	40,400	
Maintenance Clerk	1	700	700	8,400	400	8,800	
Security Guard	4	700	2,800	33,600	1,600	35,200	
Electrician Grd. 1	1	6.11	1,026	12,312	1,847	14,159	
HD Mechanic Grd. 1	1	6.11	1,026	12,312	1,847	14,159	
Machinist Grd. 1	1	6.11	1,026	12,312	1,847	14,159	
Carpenter	2	6.11	2,053	24,636	3,695	28,331	
Pipefitter	1	5.98	1,005	12,060	1,809	13,869	
Rigger	2	5.98	2,010	24,120	3,618	27,738	
Welder Grd. 1	1	5.98	1,005	12,060	1,809	13,869	
Machinist Grd. 2	1	5.74	964	11,568	1,735	13,303	
Power Plant Operator	4	5.49	3,689	44,268	6,640	50,908	
Heating Plant Operator	4	5.49	3,689	44,268	6,640	50,908	
Dozer Operator	1	5.49	922	11,064	1,660	12,724	
* Grader Operator	2	5.49	1,844	22,128	3,319	25,447	

* 1 @ 6 mos.; 3 @ 6 mos.

CHAPMAN WOOD & GRISWOLD LTD.

PERSONNEL cont'd

Classification	Number	Unit Rate	Base Cost Month	Base Cost Year	Fringe Cost/Year	Total Cost/Year	Cost/T Milled
5. PLANT SERVICES cont'd							
Tradesman Helper	2	4.89	1,643	19,716	2,957	22,673	
Power Plant Helper	1	4.89	821	9,852	1,478	11,330	
Truck Operator	1	4.89	821	9,852	1,478	11,330	
Labourer	2	4.41	1,482	17,784	2,668	20,452	
	46		$42,626	$511,512	$55,047	$566,559	$0.1079

PERSONNEL cont'd

Classification	Number	Unit Rate	Base Cost Month	Base Cost Year	Fringe Cost/Year	Total Cost/Year	Cost/T Milled
6. TOWNSITE							
Supervisor	1	1,100	1,100	13,200	1,500	14,700	
Recreation Director	1	1,000	1,000	12,000	1,500	13,500	
Clerk	1	700	700	8,400	400	8,800	
Carpenter	2	6.11	2,053	24,636	3,695	28,331	
Plumber	1	5.98	1,005	12,060	1,809	13,869	
Tradesman Helper	2	4.89	1,643	19,716	2,957	22,673	
Janitor	2	4.51	1,515	18,180	2,727	20,907	
	10		$9,016	$108,192	$14,589	$122,781	$0.0234

TABLE XVIII - 2

FLOTATION REAGENT COSTS

Thompson

Reagent		Lb/Ton	Cost/Lb. Delivered (Cents)	Cost/Ton (Cents)
Lime	1.5	1.10	6.0	6.600
Arctic Syntex L		0.01	54.5	0.545
Sodium Sulphide		0.40	15.6	6.240
Sodium Silicate		0.05	7.2	0.360
Shell Carnea 21		0.22	9.5	2.090
Dowfroth 250	.03	0.065	46.0	2.990
Superfloc 127		-	247.0	0.004
Total		1.845		18.829

Annual Cost at 5.25 million T.P.Y. = $ 988,500

Annual Cost at 6.30 million T.P.Y. = 1,186,200

Annual Cost at 7.00 million T.P.Y. = 1,318,000

Thompson Carrak 32.16¢ 75.

Extra

15.

47.16

TABLE XVIII - 3

CRUSHING AND GRINDING STEEL COSTS

Item	Lb/Ton	Cost/Lb (Cents)	Cost/Ton (Cents)
Gyratory Crusher	0.010	52.0	0.52
Cone Crushers	0.050	60.0	3.00
Rod Mill Liners	0.052	41.0	2.13
Ball Mill Liners	0.055	41.0	2.26
Rods	1.020	12.6	12.85
Balls (forged steel)	1.300	15.0	19.50
Regrind mills-allow	-	-	1.00
TOTAL			41.26

Annual Cost at 5.25 million T.P.Y.	=	$2,166,150
Annual Cost at 6.30 million T.P.Y.	=	2,599,380
Annual Cost at 7.00 million T.P.Y.	=	2,888,200

GENERAL AND ADMINISTRATIVE COSTS AT VARIOUS MILLING RATES

		A 15,000 TPD			B 18,000 TPD		C 20,000 TPD	
		$ Year	$ T Milled		$ Year	$ T Milled	$ Year	$ T Milled
Administration - salaries		$ 244,000	.0465		$ 244,000	.0387	$ 244,000	.0348
General Engrg. - salaries		96,000	.0183		96,000	.0152	96,000	.0137
Plant Services								
Salaries and Wages (excl. power labour)	$491,825							
Fuel, parts, supplies for roads, yard, office	100,000	591,825	.1127		591,825	.0939	591,825	.0845
Insurance, taxes, permits, est.		100,000	.0190	800	100,000	.0159	100,000	.0143
Outside services		50,000	.0095		50,000	.0079	50,000	.0071
Townsite, wages and salaries		122,800	.0234	182	122,800	.0195	122,800	.0175
Communications		33,000	.0063	100	33,000	.0052	33,000	.0047
Catering subsidy, 150 men @ $9/day		492,750	.0939	1442	492,750	.0782	492,750	.0704
Ancillary heating, KE Table VIII-7		155,790	.0297		155,790	.0247	155,790	.0222
Miscellaneous power and lighting		208,900	.0398		234,100	.0371	251,100	.0359
Concentrate haul equalization, mine to North Vancouver docks		621,900	.1185		734,900	.1181	815,915	.1165
TOTAL G & A		$2,716,965	.5175		$2,864,165	.4546	$2,953,180	.4219

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THE ADANAC MINERAL DEPOSIT

ATLIN, B. C.

A paper presented to the

Fourth Northern Resources Conference

Whitehorse, Yukon Territory

April 7, 1972

James E. Wallis, P. Eng.

Adanac Mining & Exploration Ltd. (N.P.L.)

ACKNOWLEDGEMENT

I would like to thank the principals of Adanac Mining & Exploration Ltd. (N.P.L.) for permission to publish the data included in this paper.

THE ADANAC MINERAL DEPOSIT

The Adanac mineral deposit is located approximately 20 miles northeast of Atlin, British Columbia. Access to the property is by well maintained gravel road from Atlin to Surprise Lake and by rough access road from Surprise Lake to the headwaters of Ruby Creek.

Early History

Molybdenum mineralization at the 5000 foot elevation along Ruby Creek was located by early day prospectors searching for the lode source of the placer gold deposits on the surrounding creeks. The prospect was examined and noted by geologists of the Canadian Geological Survey in 1905. Although the property was staked and held almost continuously over the following years, exploration of the mineralized zone was not seriously attempted until late 1968.

Regional Geologic Setting

The Adanac deposit occurs within a regional tectonic unit known as the Atlin Horst, which in turn lies within the Whitehorse Trough. The Coast Range Mountains are located 40 miles to the west. The Atlin Horst is composed mainly of Paleozoic rocks which have been intruded by large granitic batholiths during the Cretaceous and Jurassic periods.

The area was glaciated during the Pleistocene, effects of the glaciation being readily apparent as widespread and locally thick morainic deposits.

Regional geology of the Atlin area was mapped by Dr. J. D. Aitken during the period 1951 to 1955, and is described in Geological Survey of Canada Memoir 307.

The molybdenum deposit occurs near the periphery of a small pluton called the Mount Leonard Boss, which probably is connected at shallow depth to the Surprise Lake Batholith. The rocks of both the boss and the batholith are classed as alaskite, a leucocratic granite usually containing less than 2 per cent mafics by volume.

The boundary relationships between the Alaskites and older intruded rocks are obscured by extensive roof pendants of the Cache Creek Group and contained ultrabasic Atlin intrusives. The contact between the Alaskites and the older granodiorites which lie immediately northwest of the deposit appears to dip steeply away from the younger intrusive. The boundary contact with amphibolitized volcanics to the northeast of the deposit is not exposed, while glacial debris and fluvial deposits effectively mask relationships at the head of Ruby Creek valley. Olivine basalts and scoria cover the lower valley. At lower elevations, Ruby Creek has cut through the basalt cover forming an impressive canyon and exposing underlying auriferous gravels. The principal source of the basalts and scoria was f[illegible] the volcano forming Ruby Mountain which is located approximately 2 miles southeast of the deposit. These volcanic rocks are dated as being late Tertiary in age.

In addition to molybdenum and minor tungsten at the Adanac mineral deposit, economically significant amounts of tungsten and placer gold occur in the same general area.

Early-day placer mining flourished on Boulder Creek and that portion of Ruby Creek which was not covered by basalt flows. Portions of the auriferous gravels underlying the Basalts on Ruby Creek were drift mined until the late 1930's.

Transcontinental Resources Ltd. in 1951 - 2 attempted to develop the Black Diamond Tungsten prospect which lies approximately 1 mile south of the Adanac deposit. This prospect consists of a quartz vein up to 4 feet in width, carrying variable amounts of wolframite and trace amounts of sulphides.

GEOLOGY OF THE ADANAC MINERAL DEPOSIT

GENERAL

The Adanac mineral deposit is classified as an epigenetic, bulk-type, low-grade molybdenum property. It occurs entirely within a late-Cretaceous Alaskite pluton which lies in contact with an older dioritic batholith to the north.

Molybdenum occurs primarily in quartz filled fractures within an alaskite host. Molybdenum content appears to be almost directly proportional to fracture density, but is erratically distributed within the individual fractures.

A notable characteristic of the deposit is the lack of hypogene minerals other than quartz and molybdenum.

Known molybdenite mineralization of economic significance is confined to an area along Ruby Creek between Ruby Creek and Molly Lake. The deposit is roughly oval in shape, with dimensions of approximately 2000 feet in an east-west direction and 1,200 feet from north to south. Reserve grade mineralization extends from surface to approximately 500 feet below surface. Concentration of mineralization appears to diminish gradationally in an outward direction from a central area or core.

Lithology

Within the immediate area of the deposit the Mount Leonard Boss is composed of four distinct lithological units. These are:

a) Coarse Alaskite
b) Alaskite Porphyry
c) Fine Alaskite
d) Granodiorite

The alaskite porphyry is essentially a transitional phase between the coarse and fine alaskites. Contacts between them are generally gradational and irregular.

Granodiorite is actually a misnomer for this rock unit as it is in fact a slightly more mafic alaskite. In addition, the plagioclase feldspar makes up a higher percentage of the total feldspars.

This classification is based almost entirely on textural variations of chemically similar rock units. Each of the four phases is effectively an alaskite characterized by abundant smoky quartz (approximately 35 %), low mafic content (1 - 5%) and lack of colour contrast between the two feldspars. In surface exposures it has a characteristic light brown colour while fresh specimens are a mottled light-grey.

Average modal compositions are shown in the following table:

Average Composition, %

Minerals	Coarse Alaskite	Alaskite Porphyry	Fine Alaskite	Granodiorite
Quartz	36.6	33.1	34.9	31.3
K-Feldspars	41.1	39.1	37.8	27.8
Plagioclase	20.5	26.0	24.9	36.2
Mafics	1.7	1.8	2.4	4.8
Opaques	0.5	0.3	0.2	1.4

Analyses indicate that the principal minerals in the deposit other than molybdenite are pyrite and magnetite with trace amounts of scheelite, chalcopyrite, arsenopyrite, galena, monazite, zircon, rutile, gold and silver. Of these scheelite probably is the only one of economic significance.

Mineralization

Molybdenum is found as unusually coarse platey rosettes or scales in otherwise barren quartz fractures. The fractures are for the most part randomly oriented although the stronger fractures exhibit a low angle, northerly dip. These fractures are quite continuous and range in thickness from a normal 1 cm. up to 5 or 7 cm. Molybdenite is also found in some cases as an erratically distributed filling in dry fractures.

Molybdenum content within the deposit appears to be closely related to fracture density and magnitude. No fixed relationship between molybdenite concentration and the various phases of the alaskite has been determined. Locally the fine alaskites appear to be more closely fractured than the other rock units, perhaps because of their more brittle nature. Sampling within a number of shear zones indicates that the molybdenum content in these zones is comparable to that of the wall rock. It thus appears that shearing does not significantly affect the overall grade.

Alteration

Hydrothermal alteration within the deposit is noticeably lacking. Alteration of shear zones is well defined and does not extend into the wall rock.

Beyond the boundaries of the deposit there is a halo of greater alteration, which although still relatively weak, does exhibit more intense chloritization and sericitization along with an increase in pyrite content.

Supergene alteration is evidenced by a shallow oxidization cap over the deposit. The degree of oxidization is slight and rarely extends more than 50 feet into bedrock except within the confines of some large shear zones. Minor decomposition of the feldspars and some oxidization of the molybdenite is noted within the capping.

Structure

Some post mineralization shear zones have been located although movement along them does not appear to be extensive.

One major fault zone involving substantial movement cuts the deposit near the known northern limit. The magnitude and direction of movement along this fault zone has not been effectively determined. Diamond drilling to the north of the zone indicates that the ore zone was faulted off, probably by a down thrust fault.

Structural relationships between the various phases of the alaskite pluton have not yet been determined due to its complex nature.

EXPLORATION TECHNIQUES

Geochemistry

During the 1968 summer field season, soil samples were collected on 200 foot centres over a 400 foot grid covering the 12 original Adera claims. These samples were analysed for Mo, Ag, and Cu. Significant Mo anomolous areas covering portions of the upper Ruby Creek valley were indicated by the study.

Diamond Drilling

A series of 500 foot holes were spotted on the north side of Ruby Creek parallel to the surface exposure on the basis of structural information gained from surface mapping. Initially 2 BQW holes were drilled from each drill site; one at -90° and the other at -50°. The angle holes were drilled to investigate the possibility of sampling bias based on fracture intersection angles.Later studies showed the amount of bias to be insignificant. All future holes were drilled at -90°. Approximately 3000 feet of drilling was completed before inclement weather forced closure of the program in 1968.

Assay results indicated that the program was worthy of pursuit the following year with only minor changes.

Core recoveries in the 70 per cent range experienced with the BQW drilling prompted changing the hole size to NQW during 1969. This change over raised recoveries to approximately 80 per cent. Several HQW holes were drilled within a few feet of existing NQW holes to investigate the relationship between the larger hole size and both recovery and grade. Little variation was noted.

Logging and Sampling

The drill core from each hole was carefully logged both descriptively and visually and all molybdenite bearing fractures carefully noted. The molybdenite bearing fractures were marked on the visual log in red; a practice which proved to be a help in later fracture density - mineral intensity studies. All holes were then sampled in 10 foot sections and the core split and bagged. Carefull examination and subsequent testing indicated that bias was being introduced to the results depending on which portion of the split core was being assayed. Coarse rosettes of molybdenite irregularly distributed through the core resulted in variations of assay value up to 100 per cent, depending on the split. As a result of these findings, the entire 10 foot sections were assayed.

Sludge Sampling

Sludge samples were collected throughout the diamond drilling program in the hopes of finding a means of determining the true grades of the sample sections. Initially the return water from the holes was run through a vane type sludge splitter and the 1/16th split run through settling tubs. Separan was added to the tubs to help depress any floating molybdenite. This system was later changed slightly whereby the split portion was filtered through a porous sample bag. This method proved to be as effective and eliminated the human error encountered when adding separan and decanting the sample.

Intensive correlation studies were undertaken in hopes of finding a logical correlation between core assays and sludge assays. Unfortunately no correlation existed.

Diamond Drill Summary

Since 1968, 106 NQW drill holes on a pattern grid varying from 200 to 400 feet have been completed. Total diamond drilling footage completed to date approaches 60,000 feet.

UNDERGROUND DEVELOPMENT AND BULK SAMPLING

General

During the early spring of 1970 an agreement was signed with Kerr Addison Mines Limited to undertake a feasibility study of the Ruby Creek prospect. Kerr Addison was to retain a 40 per cent interest in the property for continued development. Based on the results of the feasibility study, Kerr Addison deemed it financially unwarranted for them to continue development. Their option was relinquished early in 1971.

Underground Development

A drift was collared on Ruby Creek at the 4690 level in May 1970 with drifting to intersect a line of diamond drill holes on an approximate east west grid line. Both north and south cross cuts were developed from this drift to intersect diamond drill holes on a north-south grid line. Raises were driven on seven of the drill holes that were intersected to provide grade correlation with the drill hole assays.

Total underground exploration consisted of 2, 711 feet of lateral development and 873 feet of raising.

Underground Mapping

Staff geologists mapped the underground geology using conventional methods and closely following daily advance. The only deviation from normal mapping practice was that the ribs rather than the back were mapped to enable accurate recording of both horizontal and vertical fracture systems.

The surveying staff maintained up to date plans of all workings showing daily advance and limits of each round. This made it possible to plot the average grade of each round on the plans of the underground workings and establish average grades for both the horizontal and vertical planes of that portion of the orebody tested.

Bulk Sampling Methods

Seven large open concrete bins were constructed in a line approximately 100 feet downstream from the portal. The mine tracks continued from the portal to dump points on the upper edges of the individual bins. Muck from each round was dumped in separate bins and transfered to the crusher-loading-hopper by rubber tired,966B loaders.

Muck from the individual rounds travelled from the crusher-loading-hopper by conveyor to a roll crusher where it was crushed to -3/4 inch and sent by conveyor to the sampling tower. The crushed product flowed by gravity through a timed sample cutter with the reject going to a product bin. The cut sample then passed through a small cone crusher where it was reduced in size to - 10 mesh and fed to a Denver Vezin splitter. The average round sample of approximately 35 pounds was then bagged and sent to the assay office.

A yard stockpile of lateral development crushed product was maintained for use as feed during tune up of a 100 ton per day pilot mill.

The raise crushed product was stored in large heated bins, one bin for each raise, and partially dryed for final mill feed.

PILOT MILL OPERATION

A 100 ton per day pilot mill was purchased from Brenda Mines Limited, with reconstruction on the Adanac property begining in mid-May. Equipment within the mill was set as determined during the bench scale milling tests. Basically the circuit consisted of primary grinding, bulk flotation, primary cleaning, secondary grinding, secondary cleaner and concentrate dewatering through a vacuum pan filter. A bank of new Humphrey Spirals on the head of the tailings line produced a rough tungsten concentrate.

The mill tune up proceeded on schedule and encountered fewer problems than anticipated. When final tune up was completed the mill was shut down for several days to permit a thorough flushing of all circuits in preparation for processing of the raise product. The crushed raise product was then transfered to the mill bin and processed raise by raise with temporary shut downs and cleaning of all circuits between raises. Both head assays and total recoveries were then checked against the calculated grade of each raise, providing an excellent check on the bulk sampling method used. These checked within tolerable limits.

Mill recovery during processing of the raise product averaged 97 per cent and produced a concentrate averaging 97 per cent MoS_2. Individual runs showed recoveries as high as 98 per cent with concentrate grades as high as 98 per cent MoS_2.

ROTARY DRILLING

A rotary drilling contract was let at this stage with the view of finding another method of sampling which would give results comparable to bulk sampling. Two holes were drilled within a few feet of two of the completed raises to enable comparisons to be made with the diamond drill and bulk sampling results. Comparison of these results showed the rotary assays to compare almost exactly with the bulk sample assays and to be slightly higher than the diamond drill assays. Extreme winter conditions forced closure of the program at this point.

On the strength of these results, further rotary drilling was undertaken during the 1971 season. After considerable testing, it became apparent that the rotary results were comparing with the bulk sampling results only in dry ground. Those holes which encountered appreciable water gave results identical to the diamond drill results. As a result, rotary drilling was suspended pending clarification of this problem.

ORE RESERVES

The Adanac deposit lends itself to removal of higher than average grade material during the early operating years, with stockpiling of marginal ore for subsequent reclamation.

Preliminary pit designs based on variable adjustment reserves indicate that the following production schedules could be anticipated:

Pit Source	Mill Feed % MoS_2	Mill Feed Tons	Tons to Stockpile/Waste	Cumulative Stripping Ratio
Preproduction			10,036,000	
Stage 1	0.210	6,293,000	2,534,000	0.40 : 1
Stage 2	0.185	7,373,000	13,680,000	1.18 : 1
Stage 3	0.184	5,613,000	933,000	0.89 : 1
Stage 4	0.184	7,027,000	15,827,000	1.25 : 1
Stage 5	0.183	7,306,000	1,799,000	1.04 : 1
Onward Pit	0.146	70,622,000	37,433,000	0.53 : 1
TOTALS	0.160	104,234,000	65,414,000	0.63 : 1

The pattern of metal distribution as observed in the underground workings is such that grade control in mining should not be a difficult problem. Large tonnages of low grade material lie within the normal pit path and hence could be stockpiled and eventually reclaimed for mill feed. These factors combine to indicate a low waste to mill feed ratio of 0.63: 1 for a proposed pit.

CONCLUSIONS

Metallurgically the Adanac ore is relatively simple and is amenable to inexpensive beneficiation techniques. The high recoveries and premium grade of concentrate produced during the pilot mill studies greatly enhance the overall grade of the deposit.These factors together with ore reserves virtually assures that the deposit will make a profitable producing mine with the advent of stronger mineral markets and demand.

ADANAC FEASIBILITY REPORT

By

Climax Molybdenum Corporation

of British Columbia, Limited

December 19, 1975

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LIST OF ATTACHMENTS

This report and the enclosed attachments are in fulfillment of the obligation to provide a feasibility report as described in the Agreement between Adanac Mining and Exploration, Limited, and Climax Molybdenum Corporation of British Columbia, Limited, dated December 6, 1972.

Grateful acknowledgment is extended to the following people for their advice and participation in preparing this feasibility report:

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AMAX Inc.	Greenwich, Connecticut
Mr. A. Fred Banfield, Jr.	President
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Thomas E. Cherrier

Thomas E. Cherrier
Mine Evaluation Engineer

I. BACKGROUND

The property was staked by Adanac Mining and Exploration, Limited, in 1967. During 1968, an access road was extended to the claims, a camp established, a geochemical survey carried out, and a diamond drilling program initiated. Twelve diamond drill holes totaling 4,928 feet were completed prior to winter shutdown.

During 1969, a major grid pattern diamond drilling program consisting of 36,985 feet in 68 holes was implemented. Bench scale metallurgical test work was begun, and preliminary economic studies were undertaken.

Early in 1970 an agreement between Kerr Addison Mines, Limited, and Adanac was finalized, and a full-scale feasibility program commenced. It included a continued grid pattern diamond drilling program, a large underground bulk sampling program, a pilot milling program, together with related engineering studies. In April, 1970, Chapman, Wood and Griswold, Limited, was retained by Kerr Addison to act as coordinating consultant with prime responsibility for producing a final feasibility report covering all aspects of the property.

The conclusion of this report, completed in February, 1971, indicated a potentially mineable open pit reserve of 104,234,000 tons at an average overall grade of 0.16% MoS_2, based on a 0.10% MoS_2 cutoff grade, with an average strip ratio of 0.63 to 1. Economic studies indicated that this might not be a viable mining situation, and Kerr Addison terminated their agreement with Adanac. The report also enumerated substantial difficulties in the areas of sampling, assaying, and grade interpretation for the mineralized zone.

Climax signed an agreement with Adanac on December 6, 1972, for the primary purpose of expanding the geologic knowledge of the mineralized area and specifically for investigating the potential, both laterally and at depth, of expanding geologic reserves. Diamond drilling was carried out in both 1973 and 1974, and the discussion and results of the program are contained in Attachment 1.

In an effort to better understand the grade interpretation problems of the deposit, Climax retained a geostatistical consultant, Dr. Michel David of IREM-MERI located in Montreal, Quebec, to analyze the drill hole data used in the Chapman, Wood and Griswold study. Dr. David's report is included as Attachment 2.

An additional study of the same data was done by Mr. John Thornton of AMAX, Inc., in cooperation with Mr. Fred Banfield of Mintec, Inc., located in Tucson, Arizona, to provide a third opinion on grade interpretation and to provide open pit mine design techniques for the mineralized area. Mr. Thornton's report is included as Attachment 3.

II. CONCLUSIONS

A. The geologic investigations based on the two-summer diamond drilling program indicate that the possibilities of finding significant additional quantities of potentially mineable ore reserves are very remote.

B. The grade interpretation techniques applied by Mess'rs. David and Thornton both indicate that the quantity of potentially mineable reserves appears to be substantially less than previously indicated, while the grade appears to be slightly less.

C. At the present time, this molybdenum deposit is not economically mineable, but perhaps it could be in the future with a higher product price and less restrictive British Columbia royalty and tax legislation. In fact, it will be shown later in the study that the deposit is uneconomic even assuming all factors except costs were correct as stated in the Chapman, Wood and Griswold report.

In view of these circumstances, it is not felt necessary to reassess Chapman, Wood and Griswold's descriptions of preproduction work, production equipment, mill and support facilities, etc., for this study. Their philosophy of developing a mining plan is correct (i.e., obtain the best grade first) and remains applicable. No additional assessments regarding metallurgical treatment of the ore have been undertaken for the same reason.

III. RECOMMENDATIONS

In an effort to resolve the differences that are apparent in grade interpretation for the lower grade areas of the mineralized zone, it is recommended that additional drilling be done to confirm the estimated grades in these areas.

IV. GEOLOGIC INVESTIGATION

During 1973 and 1974, Climax completed nine diamond drill holes and deepened two others for a total of 9,581 feet. The drilling was undertaken to further delimit the mineralized zone and to enhance the geologic knowledge of the area. The zone is associated with the western contact of the Ruby Creek stock and is cut off to the north by the Adera fault. It was felt that the stock contacts were not well defined in the other directions or at depth.

Drilling on the east, south, and extreme western contacts of the stock indicated no additional mineralization of consequence. Drilling through and on the north side of the Adera fault showed that the displaced section of the mineralized zone is probably included within the slip surfaces of the fault or is very deep. Drilling at the 1W - 1N location in the central portion of the deposit revealed a 1,700 foot intersection of post-mineral intrusive that effectively eliminates mineralization potential at depth. Complete details and analysis of the geologic investigation program are included in Attachment 1.

V. GRADE INTERPRETATION

The techniques used by Mess'rs. David and Thornton to determine the best grade estimate to assign to each 100 x 100 x 40 foot block in the mineralized zone vary slightly but arrive at essentially the same result. David uses a method called Kriging, described in Attachment 2, and Thornton uses a mathematical technique contained in the MEDS computer program package developed by Banfield, which is described in Attachment 3.

Table 1 lists the geologic reserves and grade at various cut-off grades for both methods. It can be seen that the tonnages and grades compare favorably. The higher grade ranges show less tonnage in David's method because the Kriging technique apparently does more smoothing of higher grades than does the MEDS procedure.

Selecting a 0.10% MoS_2 cutoff grade for comparison purposes, the table shows 86 million tons of geologic reserves from David's mineral model and 82 million tons from the MEDS model, all grading 0.10% MoS_2 or better. Analyzing the result of the "variable adjustment reserve calculation" used by Chapman, Wood and Griswold to estimate grade for 200 x 200 x 40 foot blocks, one arrives at a total of 2,222 blocks grading 0.10% MoS_2 or greater. At 12.5 cubic feet per ton, this is 284 million tons of geologic reserves.

TABLE 1

COMPARISON OF GEOLOGIC RESERVE ESTIMATES FOR ADANAC

Cut-off Grade % MoS_2	M. David Model Tons (000)	M. David Model Grade % MoS_2	MEDS Model Tons (000)	MEDS Model Grade % MoS_2
0.02	437,024	.070	415,936	.072
0.04	341,920	.082	332,064	.083
0.06	232,928	.097	222,592	.099
0.08	148,480	.112	139,200	.118
0.10	85,984	.129	81,568	.138
0.12	46,016	.147	47,232	.159
0.14	22,272	.167	28,736	.178
0.16	10,176	.189	16,288	.201
0.18	5,184	.211	10,080	.220
0.20	2,432	.236	5,760	.244

This rather large difference appears to be caused by the variable adjustment methodology of Chapman, Wood and Griswold wherein large upward adjustments were made to low grade drill hole assay values, and smaller downward adjustments were made to high grade drill hole assay values. The adjustment equation was developed based on apparent differences between raise assays versus coincident drill hole assay values.

The statistical analyses done by David and those of Thornton indicate that there is no significant difference between assays when several assays are available for a given sample. The best estimate of the grade of a sample is the average of all assays available for that sample and not the arbitrary selection of one or some of them. This point, however, may not have much overall significance since the average of all available assays is nearly identical to the average of all "select" values.

The statistics also point out that there does not appear to be a significant difference between the assays of the raises when compared to their coincident drill hole assays. In fact, David points out that in the area bounded by sections 2W, 2E, 0W, and 4N and between 4700 level and 4800 level, the average grade of raise samples is 0.191% MoS_2 and of coincident drill hole samples is 0.190% MoS_2.

David intimates that the variable adjustment technique is not wrong; however, it was erroneously applied in the low-grade area. Since the adjustment equation was developed from information only in the higher grade zone, it then is the only area in which the adjustment can be applied. This is evidenced by the fact that the point of zero adjustment was at a grade of about 0.20%, which is nearly the average of the area containing the

agreed

Seem logical to bulk sample low grade areas to solve the problem.

raises <u>but</u> which is much higher than the average grade of the deposit. The potential hazards of applying the variable adjustment on a deposit-wide basis were mentioned several times in the Chapman, Wood and Griswold study.

Because this is a low-grade deposit having an essentially lognormal distribution of grade, there is an enormous amount of lower grade material relative to higher grade material. This is demonstrated quite vividly in Table 1. Making an upward grade adjustment to the lower grade material will affect a major portion of the mineralized zone, which apparently occurred.

VI. MINE DESIGN

Applying the open-pit mine design technique contained in the MEDS computer programs to the two current mineral models, the quantity of mineable reserves comes out quite close. The parameters estimated by Climax for design purposes were as follows:

Strip Cost	$0.50/ton strip
Mining Cost	$0.50/ton ore
Milling Cost	$1.25/ton ore
General and Administration	$0.75/ton ore
Depreciation	$0.50/ton ore
Pit Slope	45 Degrees
Mill Recovery	90%
Selling Price	U.S. $2.62/pound molybdenum contained

The results of the mine design on both mineral models are contained in Table 2. It is seen that the total mineable reserves compare amazingly closely, and the average grade is comparable. The grade for the MEDS model is slightly higher, again because of the smoothing technique. The smoothing along with a slightly different pit shape is responsible for the slight difference in stripping ratio.

The significant factor is that the average of the two totals of mineable tons is now directly comparable to the mineable reserves listed in the Chapman, Wood and Griswold study. We can compare 39 million tons at .145% MoS_2, based on a .10% MoS_2 cutoff, having an average strip ratio of 0.95 to 1, against 104 million tons at .160% MoS_2, based on a .10% MoS_2 cutoff, having an average strip ratio of 0.63 to 1.

This is a very dramatic difference and can be attributed to the methods of grade interpretation used to develop the respective mineral model inventories.

TABLE 2

COMPARATIVE MINE DESIGN TONNAGE AND GRADE AT

VARIOUS CUT-OFF GRADES FOR AN ULTIMATE PIT

Cut-off	M. David Model			MEDS Model		
Grade $\% MoS_2$	Tons (000)	Grade $\% MoS_2$	Strip Ratio	Tons (000)	Grade $\% MoS_2$	Strip Ratio
0.14	15,067	.173	3.9	18,400	.184	3.3
0.13	20,767	.163	2.5	22,433	.175	2.5
0.12	26,267	.155	1.8	27,967	.165	1.8
0.11	32,567	.147	1.3	33,267	.157	1.4
0.10*	38,633	.140	0.9	38,533	.150	1.0

*0.10% MoS_2 is assumed to be the lowest possible grade that can be mined and put in the mill with no stripping.

VII. ECONOMICS

Figure 1 relates grade of ore in the ground to cost per pound of molybdenum produced at selected costs per ton of ore based on 90% mill recovery. It can be seen that at a \$3.00 cost per ton of ore and a \$2.62 per pound of molybdenum selling price the break-even mineable grade is between 0.10% and 0.11% MoS_2. The \$3.00 cost assumption includes mining, milling, G and A, and depreciation but no stripping, royalty or profit.

Don't understand

6K- #4.70

It was decided that a cut-off grade of 0.10% MoS_2 would be used for mine design and minimum mill feed grade because of the general nature of the cost estimates and differences in philosophy and opinion of what costs should be included when determining a minimum mineable grade. It is unlikely that anything less than 0.10% material could be economically mined with no stripping considered.

An evaluation of the potentially mineable reserves defined in this study was undertaken by Thornton with the fairly obvious result that it is uneconomic. In an effort to shed additional economic light on the deposit, an evaluation was made assuming that grade and tonnage as defined in the Chapman, Wood and Griswold report were correct. Table 3 lists their annual tons of ore along with grade and pounds of molybdenum recovered.

Their capital and operating cost estimates were made current by making an adjustment based on the change in the CANADIAN GENERAL WHOLESALE PRICE INDEX. These index figures are shown in Table 4 and were obtained from the PRICES AND PRICE INDEXES catalogue of Statistics Canada.

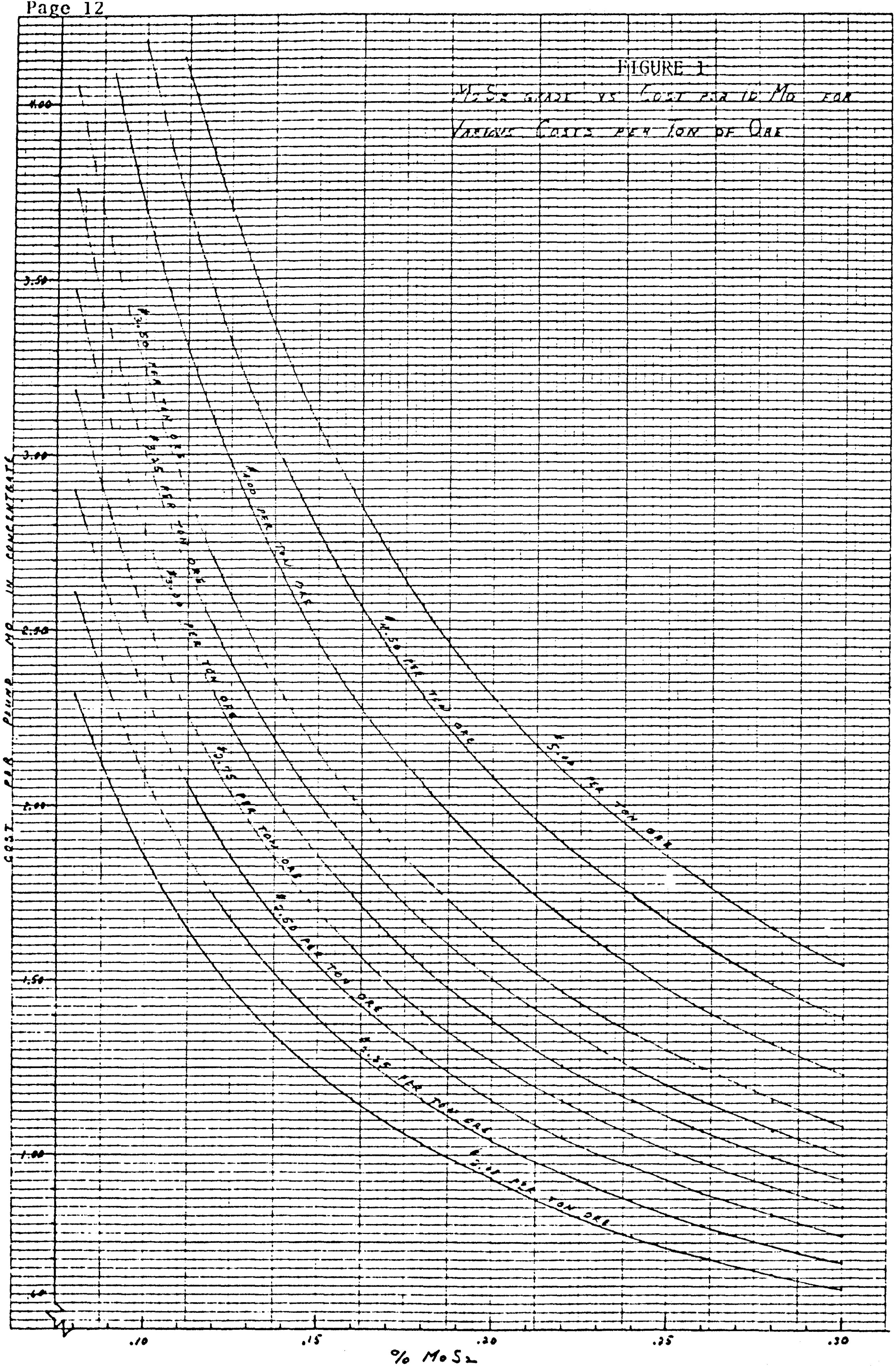
FIGURE 1
MoS2 GRADE VS COST PER lb Mo FOR
VARIOUS COSTS PER TON OF ORE
COST PER POUND MO IN CONCENTRATE
4.00
3.50
3.00
2.50
2.00
1.50
1.00
.60
.10
.15
.20
.25
.30
% MoS2
$3.50 PER TON ORE
$3.25 PER TON ORE
$3.00 PER TON ORE
$2.75 PER TON ORE
$2.50 PER TON ORE
$2.25 PER TON ORE
$2.00 PER TON ORE
$4.00 PER TON ORE
$4.50 PER TON ORE
$5.00 PER TON ORE

TABLE 3

PRODUCTION SCHEDULE

YR.		STOCKPILE & WASTE	MILL FEED 350 DAYS	GRADE	LBS. MO RECOVERED @ 90% R.
1	Preproduction				
2	Preproduction				
3	Preproduction	10,036			
4	Production	7,875	6,300	.210	14,288
5		7,875	6,300	.185	12,587
6		7,875	6,300	.184	12,519
7		7,875	6,300	.183	12,451
8		7,875	6,300	.183	12,451
9		3,150	6,300	.161	10,954
10		3,150	6,300	.148	10,069
11		3,150	6,300	.148	10,069
12		3,150	6,300	.148	10,069
13		3,150	6,300	.148	10,069
14		3,150	6,300	.148	10,069
15		3,150	6,300	.148	10,069
16		3,150	6,300	.148	10,069
17		3,150	6,300	.148	10,069
18		3,150	6,300	.148	10,069
19		1,077	6,300	.142	9,662
20		-	3,434	.110	4,080
		81,988	104,234	.160	180,000

NOTE: Waste total includes 16,574,000 tons of Stockpile material that is also included in Mill Feed total. This material is double handled. True Waste total is 65,414,000 tons.

TABLE 4

CANADIAN GENERAL WHOLESALE PRICE INDEX VALUES

1931-39 = 100

Date	Value	Annual Percent Change	1974-76 Percent Change	1972-76 Percent Change
Jan. '71	285.0			
		5.2		
Jan. '72	299.7			
		12.6		
Jan. '73	337.4			
		27.2		
Jan. 74	429.2			81
		12.8		
Jan. 75	484.3		26	
		12.0*		
Jan. 76	542.4*			

*Estimated value.

Capital costs contained in the report were based on an assumed 1971 bid situation; therefore, they were escalated from January, 1972, to January, 1976, which means they were increased 81%. Likewise their operating costs were expressed in 1974 dollars and needed escalating from January, 1974, to January, 1976, which meant an increase of 26%.

The total previous capital was $73,763,000 which, when increased 81%, becomes $133,511,000. The previous figure did not contain any money for replacement equipment which is now taken as $4 million in the ninth and in the fifteenth years. The preproduction period has been increased to 2½ years, and development costs were added which reflect moving 10 million tons of material at $1.00 per ton and allowing $1.00 per ton to cover general, administration, and start-up costs. This totals $20 million in the third year. $2.16/cu yd?

Original unit operating costs and the current estimates are as follows:

Activity	Original	x 1.26 = January, 1976
Strip/Ton Strip	$0.40	$0.50
Mine/Ton Ore	0.40	0.50
Milling/Ton Ore	1.23	1.55
G and A/Ton Ore	0.45	0.57

Table 5 lists these total capital and operating costs by year. The amount of capital shown in each of the first three years is a rather arbitrary estimate and is only done to facilitate a reasonable cash-flow schedule over time.

TABLE 5

COST SCHEDULE

YR.		CAPITAL	DEVELOPMENT	OPERATING Stripping 50¢/T	All Other $2.62/T	TOTAL OPERATING
1	Preproduction	$ 33,000	$	$	$	$
2	Preproduction	50,000				
3	Preproduction	50,000	20,000			
4	Production			3,938	16,506	20,444
5				3,938	16,506	20,444
6				3,938	16,506	20,444
7				3,938	16,506	20,444
8				3,938	16,506	20,444
9		4,000		1,575	16,506	18,081
10				1,575	16,506	18,081
11				1,575	16,506	18,081
12				1,575	16,506	18,081
13				1,575	16,506	18,081
14				1,575	16,506	18,081
15		4,000		1,575	16,506	18,081
16				1,575	16,506	18,081
17				1,575	16,506	18,081
18				1,575	16,506	18,081
19				539	16,506	17,045
20				–	8,997	8,997
		$141,000	$20,000			$309,072

The production and expenditure data were put into a computer program which calculates the Discounted Cash Flow Return on Investment based on the net cash flows generated. The following situations were examined:

CASE 1. Project undertaken by a U.S. corporation paying Canadian branch tax, U.S. income tax, and 27.5 percent of net proceeds to the optionor.

CASE 2. Project undertaken by a U.S. corporation paying Canadian branch tax and U.S. income tax but no payment to the optionor.

CASE 3. Project undertaken by a Canadian corporation paying Canadian taxes and 27.5 percent of net proceeds to the optionor.

CASE 4. Project undertaken by a U.S. corporation paying Canadian branch tax, U.S. income tax, and 27.5 percent of net proceeds to the optionor but with a 20% reduction in capital expenditures.

The super-royalty was not used in any of the runs under the assumption that it will be removed in the near future in order to stimulate the mining industry. The provincial royalty was used and is assumed deductible for provincial taxes but not for dominion taxes. All capital is assumed to be available on an equity basis. Runs were made at prices of $2.62, $3.00, and $3.50 per pound of molybdenum contained for all cases.

The results of the computer runs are found in Table 6.

TABLE 6

ADANAC FINANCIAL DATA

January 1976 Dollars

Mo Price	Capital Cost	Tax Resident	Optionor's % of Net Proceeds	DCF	Payback	($000,000) Avg Inc/Yr	NPV ($000,000) @10%	@12%	@15%
$2.62	Std	U. S.	27.5%	--%	Never	$(1.11)	$(64.9)	$(69.3)	$(73.5)
3.00	Std	U. S.	27.5	1.7%	13.7 yr	0.20	(53.3)	(59.1)	(65.0)
3.50	Std	U. S.	27.5	3.9	9.7	1.84	(39.6)	(47.0)	(55.0)
2.62	Std	U. S.	0	--	Never	(1.11)	(64.9)	(69.3)	(73.5)
3.00	Std	U. S.	0	2.1	12.7	0.54	(50.4)	(56.5)	(62.9)
3.50	Std	U. S.	0	5.1	8.3	2.77	(32.1)	(40.4)	(49.4)
2.62	Std	Canadian	27.5	--	Never	(1.88)	(74.0)	(77.9)	(81.2)
3.00	Std	Canadian	27.5	1.5	14.3	0.17	(57.4)	(63.3)	(69.3)
3.50	Std	Canadian	27.5	4.2	9.7	2.36	(40.0)	(48.1)	(56.7)
2.62	-20%	U. S.	27.5	1.0	15.5	(0.24)	(45.6)	(49.8)	(54.1)
3.00	-20	U. S.	27.5	3.2	10.5	1.04	(34.3)	(39.8)	(45.6)
3.50	-20	U. S.	27.5	5.7	7.4	2.49	(21.8)	(28.8)	(36.4)

Canadian Tax Assumptions: Bill 31 without super-royalty, Bill 31 deductible for Provincial taxes but not for Dominion taxes. Dominion income tax effective tax rate of 27% with Branch Tax of 25% on U. S. resident corporation.

U. S. Tax Assumptions: WHTC, income tax rate @34% with 14% gross percentage depletion allowance.

Feed back latest Adanac capital cost & revised operating cost into this.

As shown in Table 6, the Discounted Cash Flow Return on Investment does not get up to 6 percent even under the most optimistic conditions. The number of years required for payback does not include the preproduction period. The rather large negative Net Percent Value figures also demonstrate the lack of economic viability of the Adanac deposit.

Thus, even when assuming that the original tonnage and grade exists, the deposit cannot support today's capital and operating costs at today's price nor at escalated prices. Quite obviously a small deposit has even less chance with the higher unit costs involved in all areas.