PROJECT TERRANE

INTERIM REPORT

Prepared for

ARGOR EXPLORATIONS LIMITED

CANADIAN BECHTEL LIMITED

JULY 27, 1967

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1.0 SUMMARY AND PROJECT ECONOMICS

1.1 Summary

This report comprises a preliminary mining appraisal together with an order of magnitude assessment of both capital and operating costs of the proposed mining and processing operations to be performed on Argor Explorations' Limited pyrochlore-columbite deposit located in the James Bay lowlands area of Northern Ontario.

The property is located approximately 31 miles south of Moosonee, Ontario. Access to the property is by helicopter from Moosonee in the summertime; in the winter by either winter trail from Moosonee or ski-plane from Moosonee or Timmins.

The mining study has been carried out on the basis of the following three cases:

Case I: Mining the entire orebody by open pit methods, excluding the wedge of low grade material in the north end of the deposit, but including all columbite ore.

Case II: Mining the orebody entirely by underground methods, excluding all columbite ore.

Case III: Mining the south end of the orebody by open pit followed by underground development of the northwest limb, excluding all columbite ore.

The assessment of the three cases is based on an annual tonnage sufficient to produce 7.5 million pounds per year of Cb_2O_5 for a period of 20 years at an overall mill recovery of 75%.

Recoverable ore reserves have been calculated on the basis of the preliminary mining layouts and are estimated to be as follows:

Tons in Thousands

	I	II	III Open Pit	III Underground
Indicated	23,039	8,732	12,714	6,365
$\%Cb_2O_5$	0.51	0.57	0.59	0.49
Possible	-	8,474	-	-
$\%Cb_2O_5$	-	0.59	-	-
Total	23,039	17,206	12,714	6,365
$\%Cb_2O_5$	0.51	0.58	0.59	0.49

Additional deep hole drilling is required to prove up the ore tonnages and grades for Case II between the 525 and 825 elevation.

Open pit mining will be carried out using wheel mounted front end loaders loading into 22-ton capacity trucks. Stripping of the overburden will be done with 5½ yard wheel mounted front end loaders loading into 35-ton capacity trucks.

Pre-production stripping will involve the removal of some 4.9 million cubic yards of overburden consisting of peat, till, unconsolidated silty sand and a siltstone-mudstone-sandstone sequence. Stripping quantities are based on an overall slope of 3 horizontal to 1 vertical. It has been assumed that stripping will be carried out on a contract basis.

Mining underground will be done using a sublevel-longhole method. Ore will be trammed from drawpoints to the ore pass system by low profile, rubber tired frontend loaders. The mining efficiency has been taken at 75% of the total available ore.

Total funds are estimated for each of the three cases as $24,212,000, $18,589,000 and $23,291,000 respectively. In addition a further expenditure will be involved in Case III on termination of the open pit in order to develop the underground mine and to purchase equipment suitable for underground extraction. This additional sum is estimated to be $2,621,000 at 1967 costs and has been taken into account in the preparation of the financial projections included in this report.

Average mining and processing costs over the 20-year period for the three cases considered are estimated to be $3.77, $5.05 and $3.56 per ton of ore respectively.

1.2 Project Economics

Cash Flows have been developed over the twenty-year life for each of the three cases. These cash flows have been developed on the following basis:

1. Value of concentrate has been taken at $1.00 Canadian per pound of contained Cb_2O_5.

2. Schedules of annual mining tonnage have been prepared for the open pit cases. A constant average annual tonnage has been used for the underground schemes.

3. These schedules have been combined with the operating costs to arrive at an annual mining and processing cost.

4. Total funds required are estimated as follows:

CASE	I	II	III
Plant Costs	8,650,000	9,922,000	8,650,000
Pre-production	12,271,000	6,652,000	12,271,000
Other Costs	3,291,000	2,015,000	2,370,000
Total	$24,212,000	$18,589,000	$23,291,000

5. A 24 month construction period has been assumed with 40% of the total funds spent in the first year.

6. All funds are assumed to have been raised internally and no interest has been charged against these funds.

7. Plant replacement costs have been estimated for all mining equipment on the basis of an economic life recommended by the equipment manufacturer.

8. Ontario provincial depreciation rates and tax on profit have been calculated in accordance with the Mining Tax Act, Chapter 242, Volume 3, the "Revised Statutes of Ontario, 1960".

9. Mining pre-production expenses have been deducted immediately following the 3-year tax free period.

10. Capital cost allowance has been taken according to the varying rates as set forth in the Federal Income Tax Act.

11. A depletion allowance of 33 1/3% of the profits has been taken and federal tax calculated at 50% of the remainder.

The following table summarizes the total cash flow for each of the three cases:

CASE	I	II	III
	(Dollars in Thousands)		
20 year value of product	150,000	150,000	150,000
20 year operating cost	73,817	86,560	67,971
Plant additions	4,006	2,297	6,237
Taxes	17,763	15,164	18,644
Subtotal Deductions	95,586	104,021	92,852
Cash Flow	54,414	45,979	57,148
Discounted Rate of Return	9%	12%	12%

The following three tables illustrate the full calculation of the 20 year cash flow.

ARGOR EXPLORATIONS LIMITED

PRELIMINARY CASH FLOW

CASE I OPEN PIT MINING

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	TOTAL
Revenue($1,000)	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	150,000
Expenses																					
Mining	1,618	1,618	1,713	1,798	1,794	1,710	1,580	781	693	671	708	840	770	737	737	688	688	638	638	638	21,058
Other	2,080	2,080	2,645	3,214	3,185	2,645	2,645	2,970	2,430	2,295	2,160	2,970	2,540	2,700	2,700	2,700	2,700	2,700	2,700	2,700	52,759
Total	3,698	3,698	4.358	5,012	4,979	4,355	4,225	3,751	3,123	2,966	2,868	3,810	3,310	3,437	3,437	3,388	3,388	3,338	3.338	3 338	73,817
Gross Oper.Prof.	3.802	3,802	3.142	2.488	2,521	3,145	3,275	3,749	4,377	4,534	4,632	3,690	4,190	4,063	4,063	4,112	4,112	4,162	4,162	4.162	76.183
Prov. Tax	191	217	267	106	122	198	226	288	367	394	412	302	364	357	357	354	370	372	378	375	6.017
Income	3,611	3.585	2,875	2,382	2,399	2,947	3,049	3,461	4,010	4,140	4,220	3,388	3,826	3,706	3,706	3,758	3,742	3,790	3,784	3,787	70,166
Cap. Cost Allow.	-	-	-	-	-	-	1,749	3,102	2,274	1,687	1,257	1,210	908	702	643	558	444	502	374	423	15,833
Pre-Prod.	-	-	-	2,382	2,399	2,947	1,300	-	-	-	-	-	-	-	-	-	-	-	-	-	9,028
Tax Profit	3.611	3 585	2,875	-	-	-	-	359	1,736	2,453	2,963	2,178	2,918	3,004	3,063	3,200	3,298	3,288	3,410	3,364	45,305
Fed. Tax	-	-	-	-	-	-	-	120	579	818	988	726	973	1,001	1,021	1,067	1,099	1,096	1,137	1,121	11,746
Prof.After Tax	3,611	3.585	2.875	-	-	-	-	239	1,157	1,635	1,975	1,452	1,945	2,003	2,042	2,133	2,199	2,192	2 273	2,243	33,559
Plant Addition	-	23	20	508	328	483	15	210	20	34	-	870	-	35	340	210	-	480	-	430	4,006
Net Profit	3.611	3,562	2,855	(508)	(328)	(483)	(15)	29	1,137	1,601	1,975	582	1,945	1,968	1,702	1,923	2,199	1,712	2,273	1,813	29,553
Cash Flow	3,611	3.562	2.855	1,874	2,071	2,464	3,034	3,131	3,411	3,288	3,232	1,792	2,853	2,670	2,345	2,481	2,643	2,214	2,647	2,236	54,414
Cum.Cash Flow	3.611	7,173	10.028	11,902	13.973	16,437	19,471	22,602	26,013	29,301	32,533	34,325	37,178	39,848	42,193	44,674	47,317	49,531	52,178	54,414	

Payback	8.5 years
Return on Investment	9%

ARGOR EXPLORATIONS LIMITED

PRELIMINARY CASH FLOW

CASE II UNDERGROUND MINING

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	
Revenue ($1,000)	7.500	7.500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	150,000
Expenses																					
Mining	1.800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	1,800	2,106	2,106	2,106	2,106	2,106	2,106	2,106	2,106	2,106	2,106	39.060
Other	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	2,375	47,500
Total	4,175	4,175	4,175	4,175	4,175	4,175	4,175	4,175	4,175	4,175	4,481	4,481	4,481	4,481	4,481	4,481	4,481	4,481	4,481	4,481	86,560
Gross Oper.Prof.	3.325	3,325	3,325	3,325	3,325	3,325	3,325	3,325	3,325	3,325	3,019	3,019	3,019	3,019	3,019	3,019	3,019	3,019	3,019	3,019	63,440
Prov.Tax	138	164	186	201	217	230	240	246	256	227	232	235	242	246	250	247	253	257	260	264	4,591
Income	3.187	3,161	3,139	3,124	3,108	3,095	3,085	3,079	3,069	3,098	2,787	2,784	2,777	2,773	2,769	2,772	2.766	2,762	2,759	2,755	58,849
Cap.Cost Allow.	-	-	-	307	3,108	2,330	1,762	1,404	1,060	880	730	632	495	405	364	393	319	253	211	173	14,826
Pre.Prod.	-	-	-	2,817	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	2,817
Tax Profit	3 187	3 161	3.139	-	-	765	1,323	1,675	2,009	2,218	2,057	2,152	2,282	2,368	2,405	2,379	2,447	2,509	2,548	2,582	41,206
Fed.Tax	-	-	-	-	-	255	441	558	670	739	686	717	761	789	802	793	816	836	849	861	10,573
Prof.After Tax	3,187	3.161	3,139	-	-	510	882	1,117	1,339	1,479	1,371	1,435	1,521	1,579	1,603	1,586	1,631	1,673	1,699	1,721	30,633
Plant Addition	13	22	63	234	57	109	147	313	34	257	194	234	23	59	144	343	37	-	14	-	2,297
Net Profit	3.174	3.139	3,076	(234)	(57)	401	735	804	1,305	1,222	1,177	1,201	1,498	1,520	1,459	1,243	1,594	1,673	1,685	1,721	28,336
Cash Flow	3,174	3.139	3,076	2.890	3,051	2,731	2,497	2,208	2,365	2,102	1,907	1,833	1,993	1,925	1,823	1,636	1,913	1,926	1,896	1.894	45,979
Cum.Cash Flow	3.174	6,313	9,389	12279	15330	18061	20558	22766	25131	27233	29140	30973	32966	34891	36714	38350	40263	42189	44085	45979	

Payback	6.3 years
Return on Investment	12%

ARGOR EXPLORATIONS LIMITED

PRELIMINARY CASH FLOW

CASE 111 PIT AND UNDERGROUND

	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	TOTAL
Revenue	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	7,500	150,000
Expenses																					
Mining	343	563	1,471	1,659	1,296	1,394	708	717	618	649	650	645	645	641	645	1,142	1,776	1,776	1,776	1,776	20,890
Other	2,105	2,105	2,105	2,105	2,160	2,215	2,215	2,270	2,240	2,240	2,240	2,215	2,215	2,190	2,215	2,592	2,592	2,592	2,592	2,592	45,795
Total	2,448	2,668	3,576	3,764	3,456	3,609	2,923	2,987	2,858	2,889	2,890	2,860	2,860	2,831	2,860	3,734	4,368	4,368	4,368	4,368	66,685
Gross Op.Prof.	5,052	4,832	3,924	3,736	4,044	3,891	4,577	4,513	4,642	4,611	4,610	4,640	4,640	4,669	4,640	3,766	3,132	3,132	3,132	3,132	83,315
Prov. Tax	342	336	254	348	269	284	373	375	397	399	407	406	415	408	364	281	219	226	236	244	6,583
Income	4,710	4,496	3,670	3,388	3,775	3,607	4,204	4,318	4,245	4,212	4,203	4,234	4,225	4,261	4,276	3,485	2,913	2,906	2,896	2,888	76,732
Cap.Cost All.	-	-	-	-	-	1,742	2,841	2,155	1,604	1,301	979	950	722	862	1,146	864	730	627	470	372	17,365
Pre.Prod.	-	-	-	3,388	3,775	1,865	-	-	-	-	-	-	-	-	1,286	-	-	-	-	-	10,314
Tax Profit	4,710	4,496	3,670	-	-	-	1,363	1,983	2,641	2,911	3,224	3,284	3,503	3,399	1,844	2,621	2,183	2,279	2,426	2,516	49,053
Fed. Tax	-	-	-	-	-	-	454	661	880	970	1,075	1,095	1,168	1,133	615	874	728	760	809	839	12,061
Prof.After Tax	4,710	4,496	3,670	-	-	-	909	1,322	1,761	1,941	2,149	2,189	2,335	2,266	1,229	1,747	1,455	1,519	1,617	1,677	36,992
Post Production	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1,286	-	-	-	-	-	1,286
Plant Add.	-	346	260	203	328	483	-	230	33	340	-	678	-	1,020	1,654	65	289	268	-	40	6,237
Net Profit	4,710	4,150	3,410	(203)	(328)	(483)	909	1,092	1,728	1,601	2,149	1,511	2,335	1,246	(1,711)	1,682	1,166	1,251	1,617	1,637	29,469
Cash Flow	4,710	4,150	3,410	3,185	3,447	3,124	3,750	3,247	3,332	2,902	3,128	2,461	3,057	2,108	721	2,546	1,896	1,878	2,087	2,009	57,148
Cum.Cash Flow	4,710	8,860	12,270	15,455	18,902	22,026	25,776	29,023	32,355	35,257	38,385	40,845	43,903	46,011	46,732	49,278	51,174	53,052	55,139	57,148	

Payback	6.5 years
Return on Investment	12.3 %

2.0 INTRODUCTION

2.1 Scope and Responsibilities

Canadian Bechtel Limited was engaged by Argor Explorations Limited on January 6th, 1967 to develop preliminary plans, flowsheets and mining methods together with capital and operating costs for their proposed mine and mill located south of Moosonee, Ontario. As part of the terms of reference Bechtel was required to make a comparison of open pit versus underground mining.

At a meeting held in Toronto on May 17th, 1967 it was agreed that the mining studies would be carried out for the following three cases:

1. All open pit
2. All underground
3. Open pit followed by underground

The responsibilities of Argor and Bechtel in the preparation of this interim report have been as follows:

Argor Explorations Limited

a) Supply of diamond drill hole logs, assay data, geological plan and geological cross-sections with interpreted ore-waste boundaries.

Canadian Bechtel Limited

a) Calculation of geological and recoverable ore reserves.

b) Preliminary design of open pit and underground mining systems.

c) Estimate of preproduction costs including mobilization, access road, camp construction and maintenance, stripping of the overburden, construction of airstrip and demobilization.

d) Development of a factored plant capital cost estimate.

e) Estimate of mining operation costs.

The rough estimate of concentrating, general overhead, transportation, sales and head office and contingency expenses were developed jointly by Argor and Bechtel.

3.0 CAPITAL AND OPERATING COST ESTIMATES

3.1 Basis of Capital Cost Estimate

No formal estimate has been prepared of the cost of the three plants. The capital costs have been developed by relating the cost of flotation plants of similar capacity to the situation at Moosonee. It is believed that the figures are sufficiently accurate for comparison purposes, but they should not be used for absolute costs. The plant capital costs are inclusive of engineering, supervision and applicable contractor's fees. The estimate excludes all exploration costs, land purchase and easement costs, legal expenses, the cost of the feasibility study and other expenses incurred to date by the owner.

The pre-production costs for the open pit case involving the removal of 4.9 million cubic yards have been estimated by C. A. Pitts General Contractor Limited and independently by Bechtel. The costs as estimated by Bechtel are used in the estimate of the total cash requirements. The Bechtel estimate includes an additional $46,000 for the rock work required to sink the ramp and a slot into the orebody. The costs of the site access road, the fuel facilities, the airfield and the mine pre-production for Case II have been estimated by Bechtel.

Mine equipment costs, both underground and open pit, shaft excavation and equipment costs, hoisting equipment and the headframe costs have been obtained either directly from the manufacturers involved, or from past quotes previously submitted to Bechtel.

The following table summarizes the order of magnitude total cash requirements for each of the three cases:

SUMMARY OF ORDER OF MAGNITUDE CAPITAL COST ESTIMATE

ARGOR EXPLORATIONS LIMITED

	I Open Pit Mining X $1000	II Underground Mining X $1000	III O.P. and U.G. Mining X $1000
PLANT COSTS			
General Yard Facilities	545	360	545
Water Services	350	350	350
Tailings Disposal	290	235	290
Power Plant	990	1,457	990
Offices and Laboratories	250	250	250
Headframe and Shops	-	1,939	-
Warehouse and Shops	445	-	445
Primary Crushing	870	254	870
Concentrator	3,790	3,785	3,790
Contingency at 15%	1,120	1,292	1,120
Sub-Total Plant	8,650	9,922	8,650
PRE-PRODUCTION COSTS			
Access Road to Site	1,854	2,178	1,854
Fuel, Oil & Grease Facilities	773	359	773
Stream Diversion	382	-	382
Mine Pre-Production	8,646	2,817	8,646
Airfield & Road	616	1,298	616
Sub-Total Pre-production	12,271	6,652	12,271
OTHER COSTS			
Mine Equipment	2,016	774	1,195
Townsite	300	300	300
Initial Inventory	325	291	225
Working Capital	650	650	650
Sub-Total Other	3,291	2,015	2,370
TOTAL PROJECT COST	$ 24,212	$ 18,589	$ 23,291

3.1.1 Plant Costs

Initial plant costs are estimated for the three cases to be $8,650,000, $9,922,000 and $8,650,000 respectively. Higher capital is required for the underground case due to increased electric power requirements and the cost of the hoisting equipment and shaft house.

3.1.2 Pre-Production Costs

Pre-Production costs are estimated for the three cases to be $12,271,000, $6,652,000 and $12,271,000 respectively.

The following table summarizes the costs of various items which are contained in the pre-production estimate:

	Case I & III		
	Pitts	Bechtel	Case II
Mobilization & Demobilization	167,000	160,800	102,700
Access Roads	933,000	865,900	865,900
Airstrip Construction	83,000	242,200	470,600
Ferry - Build Wharves & Operate	204,000	230,000	220,100
Service Vehicles - Supply & Operate	218,000	201,200	100,000
Fuel Storage Facilities	60,000	95,300	93,400
Air Freight & Personnel	129,000	41,500	24,500
Camp Construction & Operation	480,000	757,900	456,100
Utilities - Power, Water, Sewage, etc.	197,000	238,700	94,000
Clear Production Area	32,000	7,500	-
River Diversion & Dewatering	608,000	256,300	-
Excavation of Overburden	3,780,000	4,028,500	-
Indirect Costs	1,134,000	1,077,600	263,700
Plant Write-Off	2,690,000	2,421,600	544,000
Fee & Contingency	1,150,000	1,600,000	600,000
Sub-Total	11,865,000	12,225,000	3,835,000
Mine Rockwork	-	46,000	2,817,000
TOTAL	$11,865,000	$12,271,000	$6,652,000

It should be noted that variations in the individual cost centres between the Pitts and Bechtel estimates can generally be reconciled by different allocation of costs or by differences in some of the basic assumptions. In general the estimate comparisons are considered excellent for this stage of the project development.

3.1.3 Mine Equipment

Mine equipment costs for the three cases are estimated to be $2,016,000, $774,000 and $1,195,000 respectively. The higher equipment costs for Case I over Case III are due to the different stripping schedules developed. Case I requires stripping of the overburden to continue in the first year of operation, whereas in Case III overburden stripping commences during the second year. Capital for the equipment of the latter case has been taken from the annual operating profit.

3.1.4 Townsite

An allowance of $300,000 has been included in each case to cover the cost of housing for senior personnel only.

3.1.5 Initial Inventory

An allowance of $325,000, $291,000 and $225,000 has been included to cover the cost of major equipment spares.

3.1.6 Working Capital

An allowance of $650,000 has been included in each case to cover the Owner's cost during construction and approximately 30 days of accounts receivable.

3.2 OPERATING COST ESTIMATES

3.2.1 Open Pit

Operating cost estimates for the open pit operations are based on the removal of 3000 tons per day of ore on a two shift basis, approximately 1,530,000 cubic yards per year of overburden and approximately 360,000 cubic yards per year of waste rock.

The following table summarizes the unit mining costs:

	Overburden $/C.Y.	Ore $/Ton	Waste $/Ton
Drilling	.013	.035	.035
Blasting	.015	.075	.075
Loading	.046	.065	.065
Hauling	.241	.126	.126
Haul Road Maintenance	.046	.010	.010
Slope Maintenance	.227	.010	.010
Tractors	-	.020	.020
Drainage	-	.010	.010
General Pit	-	.020	.020
Supervision	.013	.035	.035
Engineering	-	.034	.034
Total	.601	.440	.440

3.2.2 Underground

Operating cost estimates for the underground operations are based on the removal of 3000 tons per day of ore. In Case III the first 15 years are by open pit mining, the costs of which are included in the previous section.

The following table summarizes the unit underground mining costs:

	CASE II		CASE III
	Year 1-10	Year 11-20	Year 16-20
Hoist	.22	.30	-
Pump	.03	.05	.02
Ventilation	.03	.05	.02
Crusher	.10	.12	.08
Fill	.10	.12	.10
Development	.41	.46	.35
Timber	.02	.03	.02
Drill and Blast	.42	.42	.42
Haul & Load	.20	.30	.44
Supervision & Eng'g	.12	.12	.08
Power, Air, Water	.10	.12	.06
Rock Bolt	.03	.04	.03
Training	.05	.05	.05
Miscellaneous	.22	.22	.18
Total	$ 2.05	$ 2.40	$ 1.85

3.2.3 Other Costs

Concentrating, overhead, transportation and miscellaneous expenses are estimated to total $2.70 per ton of ore milled, as follows:

Concentrating	1.40
General Overhead	.70
Transportation	.10
Sales & Head Office	.25
Contingency	.25
Total	$ 2.70

4.0 ORE RESERVES

4.1 Geological Reserves

Utilizing the drill hole data, assay data and geological sections showing the ore-waste divisions, geological ore reserves have been calculated for the pyrochlore and columbite ore zones between section 6+00S and 18+00N.

Reinterpretation of the extent of the columbite zone will change the pyrochlore-columbite tonnage ratio.

The following table summarizes the total geological ore reserves:

SUMMARY OF GEOLOGICAL RESERVES

SECTION 6+00S TO SECTION 18+00N

Section	Ore Type	Location	Tons x 10^{-6}	$\%Cb_2O_5$
6S-6N	Pyrochlore	Crown Pillar	3.176	0.64
		175 - 400	8.818	0.57
		400 - 475	2.825	0.57
		Sill Pillar	2.002	0.59
		525 - 825	11.298	0.59 *
	P + Columbite	Crown Pillar	0.981	0.41
		175 - 400	2.095	0.39
		400 - 475	0.406	0.48
6N-18N W.Limb	Pyrochlore	Crown Pillar	0.733	0.40
		175 - 400	2.744	0.55
		400 - 475	0.944	0.44
		Sill Pillar	0.682	0.46
		525 - 825	3.683	0.46 *
E.Limb	Pyrochlore	Crown Pillar	1.429	0.51
		175 - 400	3.411	0.46
		400 - 475	1.419	0.49
		Sill Pillar	1.120	0.48
		525 - 825	8.187	0.48 *
	P + Columbite	Crown Pillar	0.932	0.39
		175 - 400	2.724	0.42
		400 - 475	0.2855	0.48
		Sill Pillar	0.379	0.45
		525 - 825	0.717	0.45 *
	Total		61.560	0.52

* Estimated only. Very little drill information.

4.2 RECOVERABLE RESERVES

4.2.1 Case I

Recoverable ore reserves for Case I are based on a pit floor elevation 400 feet below the surface. These reserves are estimated to be 23,039,000 short tons with an average grade of 0.51% Cb_2O_5.

To recover this tonnage an estimated 14,720,000 cubic yards of overburden and 7,257,000 cubic yards of waste rock must be removed.

These reserves are inclusive of the columbite ore zone and are classified as drill indicated.

The following table summarizes the ore reserves and stripping requirements by 50 foot intervals for Case I.

CASE I

RECOVERABLE ORE RESERVES AND STRIPPING QUANTITIES

		Ore		Waste
Elevation		Tons x 10^{-6}	%Cb_2O_5	Cu.Yd. x 10^{-6}
+ 100		-	-	14.720
+ 150		4.249	0.54	2.063
+ 200		4.099	0.53	1.744
+ 250		3.848	0.50	1.388
+ 300		3.706	0.50	1.208
+ 350		3.648	0.50	.675
+ 400		3.489	0.50	.359
	Total	23.039	0.51	21.977

4.2.2 Case II

Potential recoverable reserves are estimated to be 17,206,000 short tons at a grade of 0.58% Cb_2O_5 for an underground mine planned in the area bounded by sections 6+00S to 6+00N and down to the 825 elevation at an extraction recovery of 75%. The following table summarizes these reserves:

CASE II

RECOVERABLE ORE RESERVES

	Tons x 10^{-6}	$\%Cb_2O_5$
Drill Indicated (-175'+ 475')	8.732	0.57
Possible (-525' + 825')	8.474	0.59
Total	17.206	0.58

These reserves are exclusive of the columbite ore zone.

Additional ore, in case of shortages in tonnage or grade, can be obtained from the following sources:

- access provided to east and west limbs
- extension beyond 6+00S to 8+00S at least.

4.2.3 Case III

Recoverable ore reserves are based on a pit floor elevation of 430 followed by a ramp in ore to 475 and an underground operation in the area above the 475 elevation bounded by sections 7+00N to 16+00N in the east and west limbs.

Open pit ore reserves are estimated to be 12,714,000 short tons at a grade of 0.59% Cb_2O_5. They are classified as drill indicated reserves.

To recover this tonnage an estimated 8,859,000 cu. yds. of overburden and 4,489,000 cu. yds. of waste rock must be removed.

Underground ore reserves are estimated to be 6,365,000 tons at a grade of 0.49 % Cb_2O_5 and an extraction recovery of 75% for an underground mine planned in the area bounded by 6N to 16N down to the 475 horizon. The ore reserves are exclusive of the columbite ore zone and are classified as drill indicated.

The following table summarizes the ore reserves for Case III.

CASE III

RECOVERABLE ORE RESERVES
AND STRIPPING QUANTITIES

	Ore		Waste
Elevation	Tons x 10^{-6}	$\%Cb_2O_5$	Cu.Yd. x 10^{-6}
Open Pit			
+ 100	-	-	8.859
+ 150	2.160	0.64	1.172
+ 200	2.064	0.60	.932
+ 250	1.956	-	.670
+ 300	1.880	-	.464
+ 350	1.980	-	.246
+ 400	1.762	0.57	.098
+ 450	.912	0.57	.017
Sub Total	12.714	0.59	12.458
Columbite			.890
Underground			
+ 475	6.365	0.49	-
TOTAL	19.079	0.56	13.348

5.0 MINING METHODS

5.1 OPEN PIT CASE I & III

5.1.1 Pit Layout

The following parameters have been used in the design of the open pits:

Bench Height	37½ feet
Bench Slope	70 degrees
Overall Pit Slope	55 degrees
Berm Width	13 feet
Road Width	50 feet in rock
	60 feet in dirt
Road Grade	10 percent
	5 percent on curves
Overburden Slope	3H to 1V

The ultimate pit road exits from the pits at the south end and is located in the west wall of the pit for the upper 200 feet and in the east wall of the pit for the lower 200 feet. The location of the haul road has been selected for ease of access to the higher grade ore at the south end in the early years of operation. It avoids a potential fault zone along the eastern carbonatite contact, and permits a more logical sequence of overburden removal in the early operating years.

A 50 foot berm has been retained at the contact between the carbonatite and the siltstone-mudstone sequence. This will permit access at all times to clean up sloughed material and maintain drainage ditches.

Two 35 foot berms have been left in the overburden for road access to continue the stripping during operations. These berms are located at the base of the silty sand and midway in the siltstone-mudstone sequence at the approximate elevation of 70 feet. A twenty foot berm has been left at the base of the till.

The peat has been stripped back a distance of 50 feet from the edge of the pit and an interceptor drainage ditch has been established in the till around the perimeter of the pit.

5.1.2 Pre-Production

To conform with the proposed mining schedules and pit designs for Cases I & III an estimated 4,900,000 and 4,450,000 cubic yards of overburden respectively must be removed in the pre-production phase. Some of the till material will be used in the construction of tailing pond starting dykes and perimeter dykes around the perimeter of the plant site and mining area.

This material will be removed on a contract basis using 5½ cubic yard wheel mounted front end loaders in conjunction with 35 ton haulage trucks.

As soon as the overburden has been stripped to expose the western side of the orebody a ramp is sunk from the 100 elevation at the south end of the orebody to the 150 elevation. A sump is established at the 150 elevation and a slot is driven from the 137 elevation eastwards to the mid-point of the ore zone. This will permit at least three working faces to be established in ore on the 137 elevation.

5.1.3 Pit Development

A preliminary mining schedule showing the annual stripping requirements has been drawn up. One of the prime considerations of this schedule has been to keep stripping requirements to a minimum in the tax free period. However, because of the large volumes of overburden and the rapid rate at which ore must be exposed in the early years, this requirement has been very difficult to meet.

The basic approach used in the development of the mining schedule has been to mine from only 2 levels at any one time in order to keep equipment movements to a minimum. In the early years it has not been entirely possible to develop a full years ore requirement one year in advance. However, once the overburden removal is completed this requirement is more easily met.

The following tables show the annual ore and waste requirements for Case I and III. Additional studies will be required to optimize these schedules.

CASE I

ANNUAL ORE AND WASTE PRODUCTION

		Waste	
Period	Ore Tons x 10^{-6}	Overburden Cu.Yd. x 10^{-6}	Rock Cu.Yd. x 10^{-6}
Pre-production	.061	4.900	.023
1	.780	1.530	.360
2	.780	1.530	.360
3	.980	1.540	.360
4	1.190	1.530	.360
5	1.180	1.530	.360
6	.980	1.530	.360
7	.980	.630	.775
8	1.100	-	.300
9	.900	-	.300
10	.850	-	.300
11	.800	-	.360
12	1.100	-	.360
13	.940	-	.360
14	1.000	-	.300
15	1.000	-	.300
16	1.000	-	.251
17	1.000	-	.251
18	1.000	-	.200
19	1.000	-	.200
20	1.000	-	.200
TOTAL PIT	19.621	14.720	6.340

CASE III

ANNUAL ORE AND WASTE PRODUCTION

Period	Ore Tons x 10^{-6}	Waste: Overburden Cu.Yds. x 10^{-6}	Waste: Rock Cu.Yds. x 10^{-6}
Pre-production	.052	4.450	.037
1	.780	-	-
2	.780	.200	.100
3	.780	1.298	.350
4	.780	1.500	.420
5	.800	.711	.500
6	.820	.700	.620
7	.820	-	.350
8	.840	-	.350
9	.830	-	.255
10	.830	-	.287
11	.830	-	.287
12	.820	-	.287
13	.820	-	.287
14	.810	-	.259
15	.810	-	.100
16	.512	-	-
TOTAL PIT	12.714	8.859	4.489

5.1.4 Pit Equipment

Drilling of the overburden, waste rock and ore will be done with air track machines capable of drilling 4 inch diameter holes.

It is anticipated that light blasting only will be required in approximately 50% of the overburden. A powder factor of 0.2 pounds per cubic yard has been assumed for the purpose of costing.

Powder costs for ore and waste rock are based on loading the blast holes with 40% slurry in the toe and 60% ANFO in the column. A loading factor of 0.5 pounds per ton of rock.

Loading of ore and waste rock will be done with either the Cat 988 or the Eimco 916, both rubber tired front end loaders equipped with 5½ yard buckets. The Eimco 916 is four wheeled drive articulated vehicle which can also be used in an underground operation.

Loading of the overburden will be done by a Cat 988 tire mounted front end loader equipped with a 5½ cubic yard bucket, aided by a D-8 equipped with a ripper for dozing and ripping the till. Haulage of the overburden will be done with trucks of the Euclid R-30 series type.

A 3½ diesel shovel with backhoe and dragline attachments will be used as a utility shovel for ditching and dyke construction.

5.2 UNDERGROUND CASE II

5.2.1 Choice of Mining Method

The Moosonee area has no previous record of mining. Beyond information yielded by the soils examination and test drilling no practical knowledge is available concerning the behaviour of overburden when exposed nor the ground conditions and amenability to stoping of the underlying carbonatite mass.

Having regard to these uncertainties any proposed mining method must at this stage be considered as tentative and in urgent need of verification at the earliest possible opportunity by direct methods - exploratory shaft, experimental stoping etc.

The required extraction demands a minimum daily production of 2823 tons of ore and the mine falls into the "medium " range. The ore is apparently of reasonably regular grade throughout without any notably high grade areas.

A final mining scheme should be evolved to include at least two separate mining horizons in view of the established thicknesses and this method should be easily adaptable to the lens-like plan of the orebody.

Block caving is recognized as the cheapest form of extraction provided (a) that the ore will cave and break into reasonable sizes due to its own weight and (b) that the subsidence of the surface will not affect the operation adversely. In this case it seems probable that the unsupported spans in this carbonatite are likely to be very large before caving would occur and that the blocks so formed would be unmanageable.

In addition since this area has very poor natural drainage and a potentially heavy water bearing horizon in the overburden it is felt that disturbance of the surface would give rise to a pumping problem of very great magnitude since once the overlying strata deliver water into the mine these entries may never be sealed off. Caving has therefore not been further examined and it is considered that the overlying strata must be held intact by a substantial crown pillar over the whole extent of the orebody.

A tentative thickness of 75' has been assumed and the loss of ore amounts to 3.176 million tons at 0.64%. Based on the experiences of another producer having a carbonatite orebody it is possible that open stopes may be considered as an extraction method in all parts of the orebody and on successively lower horizons provided adequate pillars are left between working stopes and that working stopes are filled immediately on completion.

Some skeleton pillars must be left between filled working stopes to provide continuous support to the crown pillar although on a final salvage operation these could be removed by another method.

Since the supply of skilled labour in the area might be scarce it is believed that sublevel stopes be tried in the opening phase of the mine since this method gives relatively very high output per manshift and low cost of mining.

A tentative scheme has been laid out using a basic stope having 6 sublevels at 50' intervals, a span of 80' with a final pillar between worked stopes of 20'. Length has been limited to a maximum of 200' for single stopes and 350' for double or back-to-back working. Drilling will be performed by long hole machine from the sublevels and ore will be withdrawn, after breaking, from haulage drives placed in the dividing pillars using front end loaders tramming to a central ore-pass system.

5.2.2 Extraction Plan

Using the basic stope detailed above all ore beneath the crown pillar down to the undercut levels of these stopes, less pillars, will be removed during the first ten years.

Stopes will be arranged transverse i.e. from wall to wall across the width of the mass and stoping will start from the east wall which dips west at about 75°. Mining will be progressive from the north and south extremities retreating towards the shaft situated on 2N at 7W about 250' outside the orebody.

Fill distribution drifts will be cut in the residual pillars just below the crown pillar to enable filling with classified mill tailings to start immediately on completion of mining. Arrangements will be made to admit a small proportion of cement in the opening and closing stages to give additional strength to floor and backs.

5.2.3 Mining Second Horizon

Provided the stoping method chosen proves satisfactory and further exploration below shows no radical change in ground conditions the same method is proposed for the second lift leaving a sill or floor pillar to provide a safe roof for the second phase. A distance of 50' is suggested.

It should be stressed that reserve tonnages and grades in the second horizon have a lower order of reliability than the first mining horizon since less diamond drill information is available. Further development and drilling from the upper level will be necessary to prove this latter 10 year reserve adequately.

5.2.4 Details of Mining Sequence

5.2.4.1 Development

i) Shaft

A vertical shaft is proposed to be sunk by contractors from a position on 2N at 7W approximately 250' from the orebody and in a non-mineralized calcite bearing carbonatite. This position is approximate at this stage until further detailed knowledge of precise ground conditions has been obtained by closely spaced diamond drilling.

The location has been chosen for the following reasons.

a) It is situated on the approximate axis of the centre of gravity of the orebody between 6S and 6N from which the extraction for the first 20 years will take place.

b) The orebody at the west side of the ore zone, adjacent to the shaft position, has nearly vertical walls.

c) Between the carbonatite in which the shaft will be sunk and the orebody is a band of uralitized pyroxenite, a harder rock and apparently not extensively faulted. In the corresponding position on the east side of the ore zone the wall rock has been identified as a gneiss and probably contains a fault or shear zone. The orebody here has a westerly dipping east wall at about 70°and even if ground problems are not experienced, all crosscuts to the orebody will increase substantially with depth.

For twenty years life, ore must be recovered down to 825 below ground level and the initial decision should therefore be taken to provide for the shaft to be sunk to its final depth of approximately 1050'.

All ore from the first mining horizon at 475 will be hauled from the stopes to a central ore pass system and conveyed by raises to the crushing station on 825 and thence by a 1000 ton shaft pocket and loading pockets, to a shaft loading station at 1025 for skip hoisting. During the second 10-year period with all ore coming from the 875 haulage level the same crushing, storage and hoisting system will be used.

The shaft will require four compartments two for 5-ton skips in balance driven by a Koepe hoist, two for a large sized cage capable of transporting major pieces of mobile equipment below ground and its counterweight and two more for a manway and service cables, pipes etc. An approximate dimension will be 22' x 12' outside timbers. Shaft stations will be cut at 150' intervals down to 475 then at 525, 675 and 825.

Below this, stations are needed as access to the ore pass system at the crusher discharge level, shaft pocket and loading pocket levels and for the spill pocket.

From the surface to approximately 80' a concrete collar with water seal at the rock interface is allowed for while in rock the further 970' will be fitted with normal timber sets at 6'0" to 6'6" intervals depending on actual conditions.

The shaft will be designed for 3000 tons per day hoisting in 12 effective hoisting hours or 250 t.p.h. maximum. A winding speed of 1500 feet per minute with the 5 ton skips is indicated to satisfy this condition.

ii) Stope Preparation

From the shaft stations at 175, 325 and 475 crosscuts will be put out to the western ore boundaries and the west wall drives established to serve initially as diamond drilling drifts.

When precise contacts and ore grades have been established cross-cuts will be put through the orebody to establish the east wall drives on these levels from which further diamond drilling will take place to facilitate stope layout. Assuming this close-spaced drilling supports the conclusions drawn from the preparatory drilling from the surface the orebody will be divided transversely into stopes of varying lengths having a height of 300', a width of 80' and a separating pillar between stopes of 20'.

Each stope will be prepared by providing at the 475 level two haulage drives sufficiently sized to permit front end loader operation.

These haulage drives are placed in the separating pillars and are connected to the stopes on either side by boxholes leading to draw slots between 475 and the undercut level at 425.

Access to the stopes is made by raise from 475 to 325 and 325 to 175 with connections to the west wall drives at these levels. At intermediate levels of 425, 375, 275 and 225 one raise serves a group of 3 stopes by short longitudinal connecting drives. Sublevels are put through from end to end and a slot raise will be developed at a convenient position depending on stope length. Some stopes - those on the east wall - have the slot raise on the east wall and others, the larger, having a central slot raise to permit working in both directions.

5.3 UNDERGROUND CASE III

5.3.1 Choice of Mining Method

The same mining method has been adopted in this scheme as in the full underground scheme and costs of mining have been taken directly from the earlier underground analysis.

5.3.2 Extraction Plan

Stope layout however has been freely adapted to meet the requirements dictated by the shape of the orebody and the bifurcation into two limbs between sections 6 and 16N. It has been decided to mine by sub-level stopes, therefore, in a transverse W-E direction and to develop these stopes by means of wall drives, extraction to commence in the northern extremities and to continue by retreating to the open-pit entries.

Two different stope heights are proposed. The majority have six sublevels at 50' intervals as in the full scheme but in the boundary zone between the north face of the open-pit and the underground section stopes having only 150' total height i.e. 3 x 50' sublevels have been laid out. A minimum safety crown pillar of 75' has been left above these stopes for their protection, this distance being measured between the final opencast face and the top of the 150' stopes and vertically above the tops of the 300' stopes.

5.3.4 Development

Due to the lack of definitive information concerning the distribution of columbite all stopes have been laid out and the development plan relates to systematic preparation regardless of columbite values. Life of the underground section is based on non-columbite ore only however and should any proposed stopes subsequently prove to have no columbite free ore then development of these need not, in this event, take place. Stopes showing minor proportions of columbite will be worked until columbite appears and is defined and may then be abandoned and filled or, if metallurgical research has by that time solved the problem of treating a pyrochlore-columbite mixture, completely extracted and filled.

5.3.5 Access and Haulage

Access will be independently to both west and east limbs of the underground section with a central access for services and supplies only. From the established open-pit bottom proposed to go to 430' a ramp will be put down to 485' below surface and new sumps established. From 485' the main entries will be driven, after portals are secured and concreted, at +2% to cause the underground section to be self-draining. It is anticipated that the northern extremities will reach about 465' and the mining scheme has been based on the average horizon of 475' to coincide with calculated reserve as regards tonnage and estimated grades.

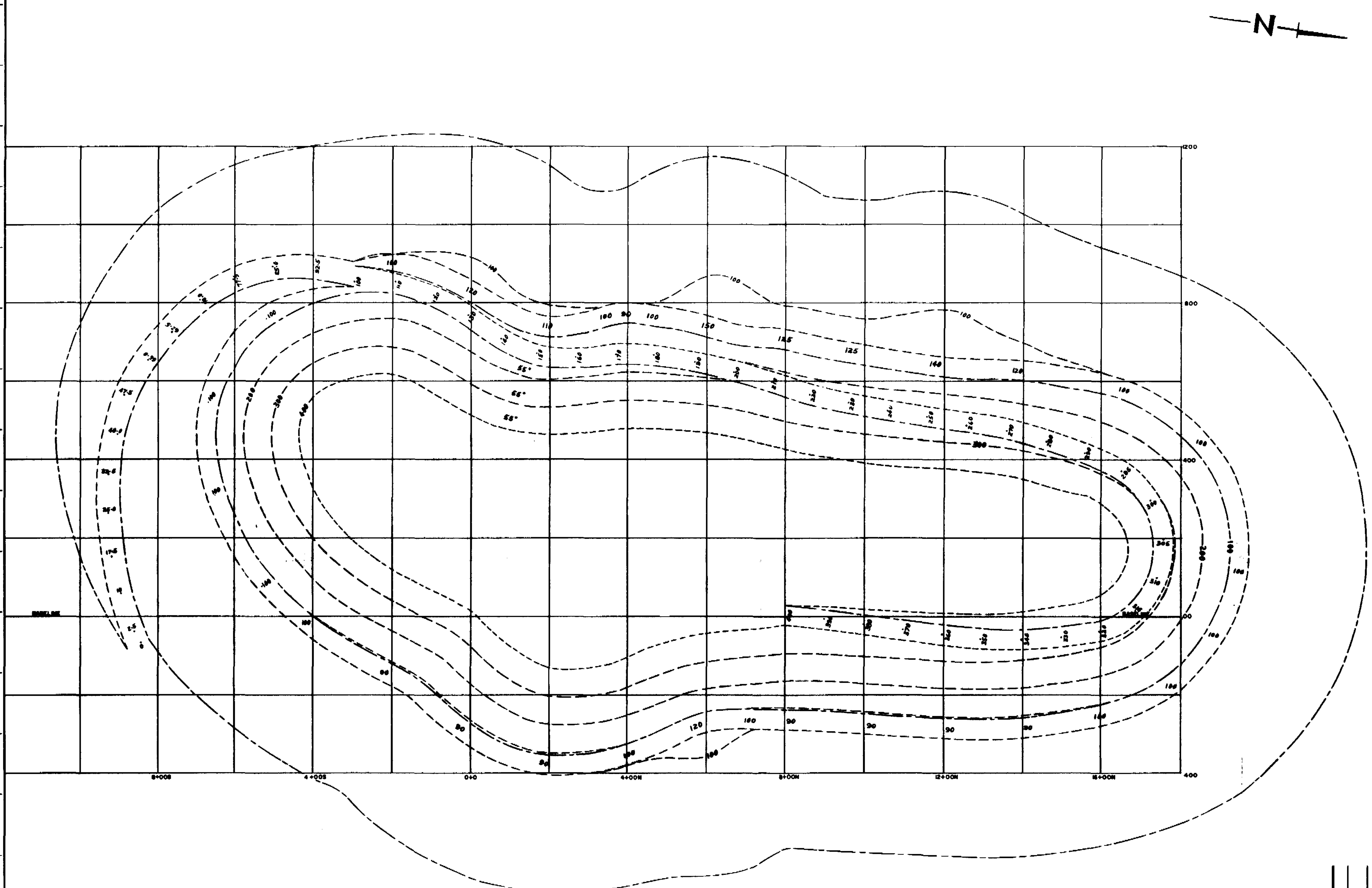
Due to the distances of the stopes from the open-pit front end loaders, Eimco 916, will need to be supplemented by a low profile transporter of

10 ton capacity, such as the Wagner vehicle designed for this purpose.

The main wall haulages are driven at 20' wide by 9' high to permit passing of these vehicles. Front end loaders in this scheme will transport only to stope boundaries, before transferring their loads to these vehicles. Stope drives on "475" level remain as before at 15' x 9'.

5.3.6 Loading of Open-Pit Trucks

From a point on the main underground haulage roads ramps will be driven to connect with the open-pit at about level 440'. These ramps can be driven up to 15% and are designed to connect with dumping bins of reinforced concrete built into the walls of the open-pit above the main pit haulage road. These bins will be of 1000 ton capacity and are controlled by suitable chute gates to permit loading into the regular open-pit haul trucks for transport via the open-pit haul road to the surface crushing installation.



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No.	DATE	REVISIONS	BY	CH'K	APP'D	ENG'R	CONST. ENG'R	CLIENT

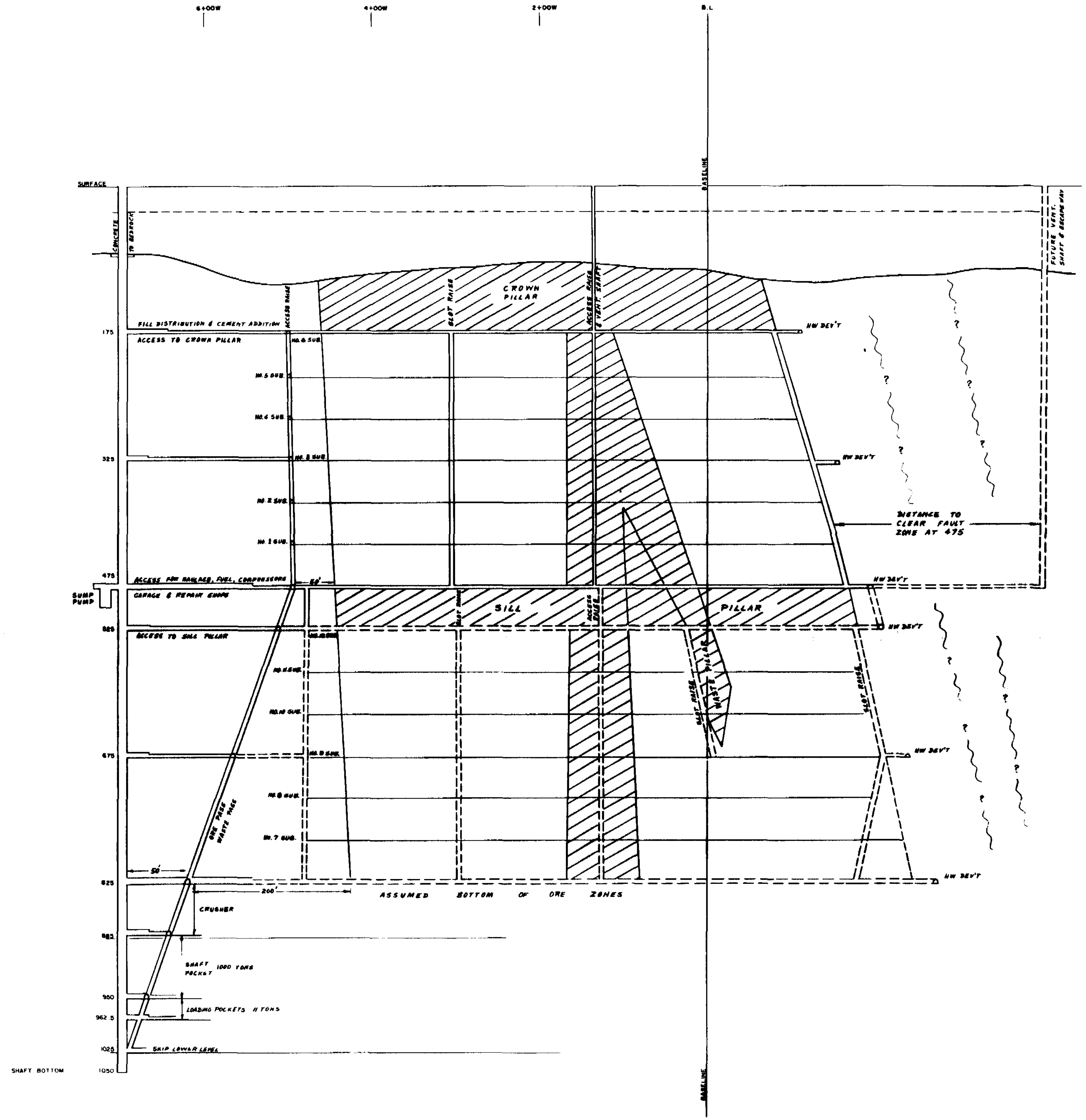
SCALE 1" = 100' | DESIGNED K.A.G. | DRAWN S. D'LIMA

BECHTEL AND COMPANY
MONTREAL, P.Q.

ARGOR EXPLORATIONS LTD.
PROJECT TERRANE

ULTIMATE PIT
CASE I

BECHTEL | JOB No 6396 | DRAWING No | REV.



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PROJECT TERRANE

TYPICAL TRANSVERSE STOPING ARRANGEMENT CASE II

JOB No 6396 | DRAWING No | REV.

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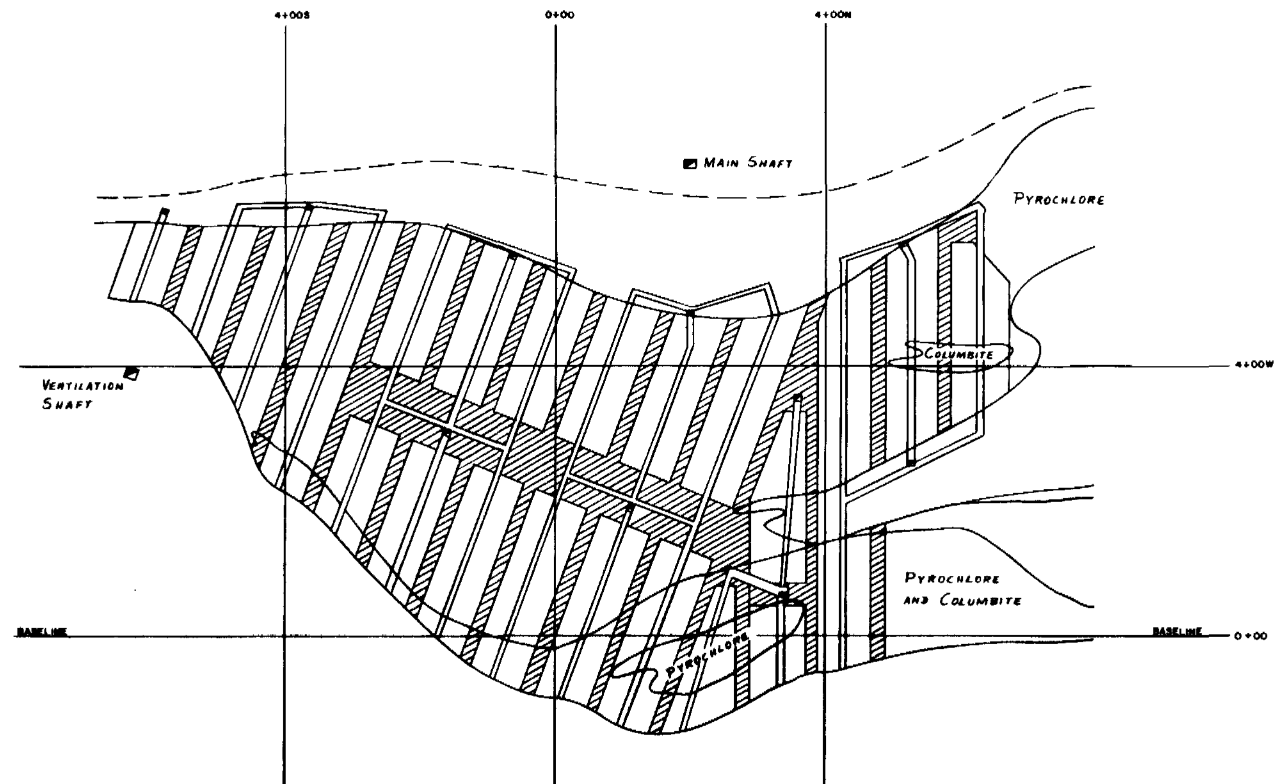

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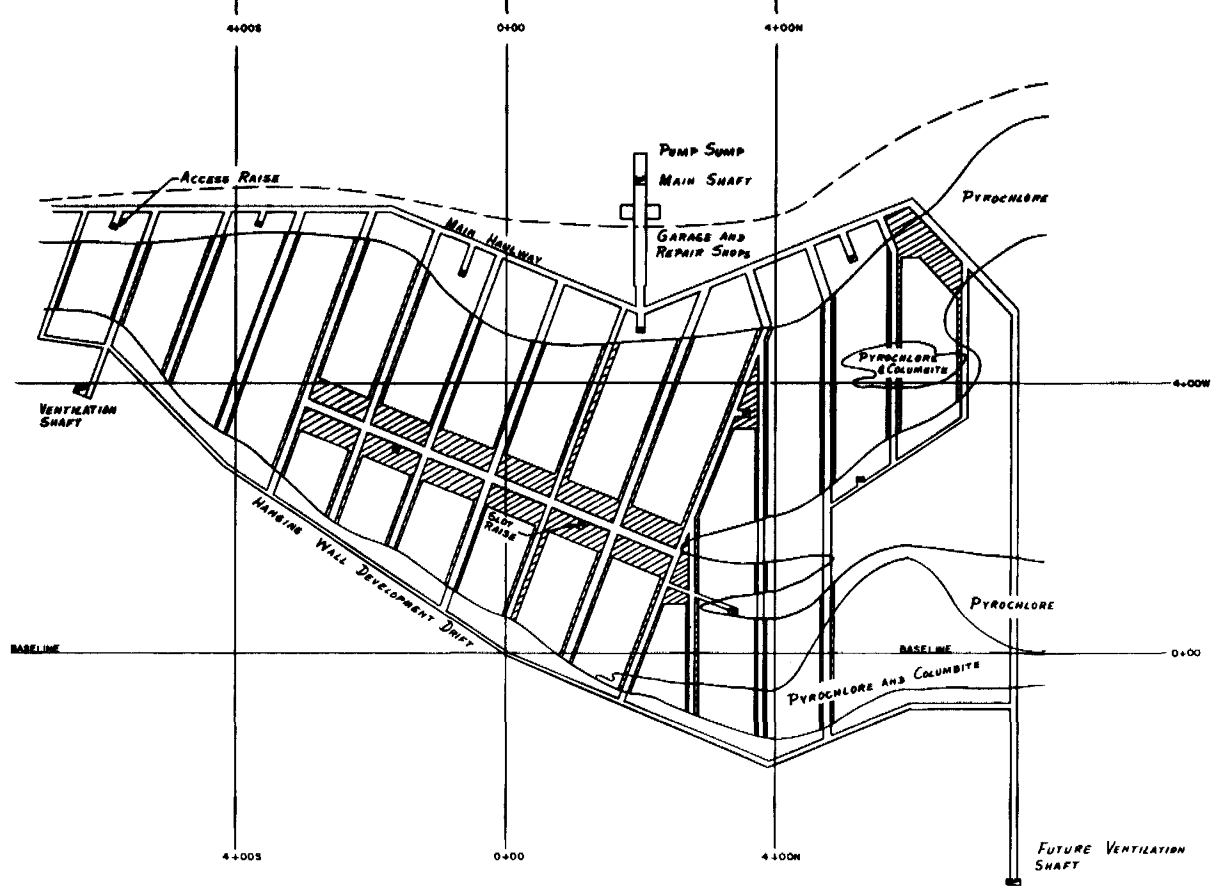

PLAN LEVEL 475

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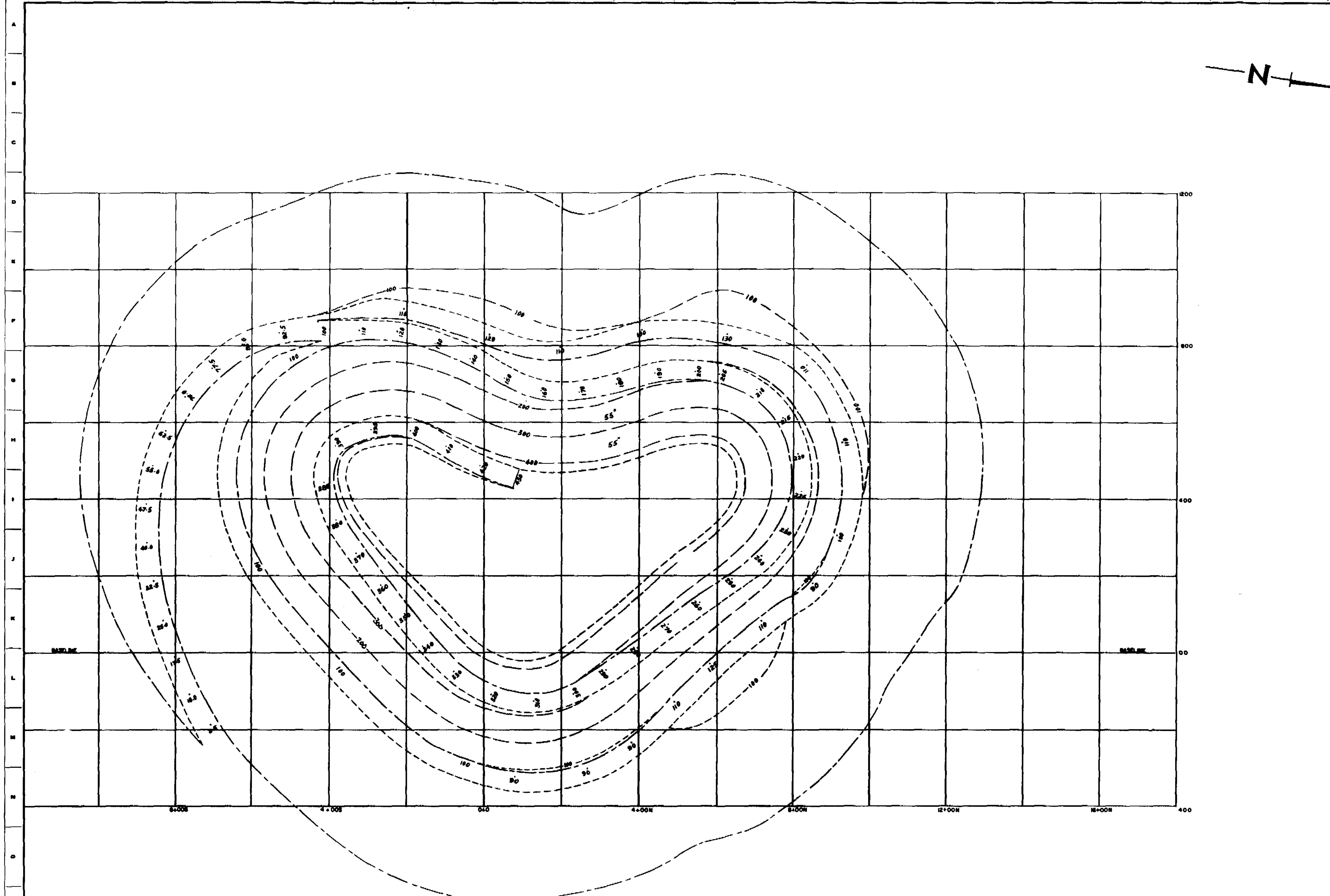
PRELIMIN. MINING PLANS
ELEV. 225 & 475

BECHTEL | JOB No. 6396 | DRAWING No | REV.



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MONTREAL, P.Q.

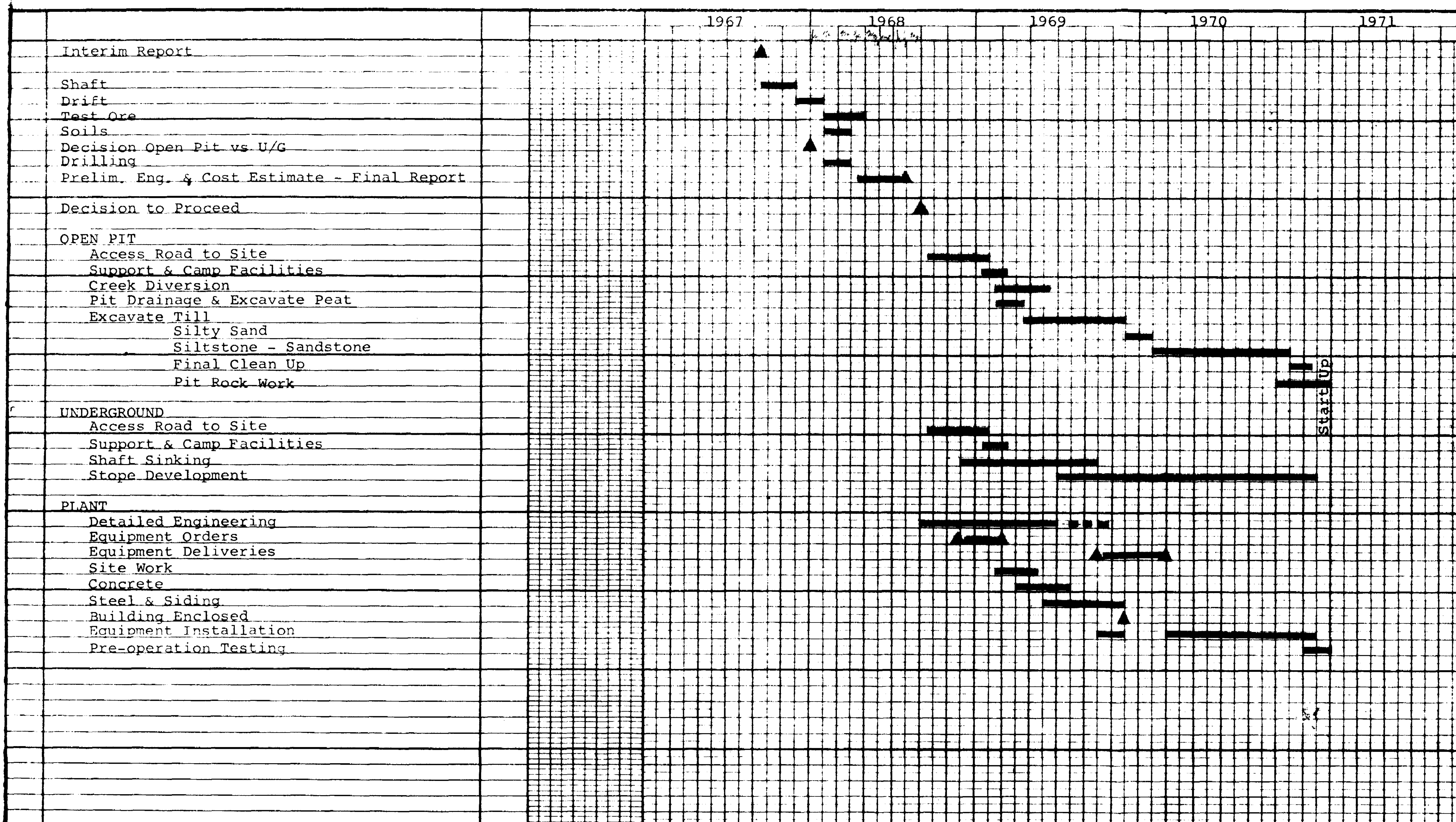
ARGOR EXPLORATIONS LTD.
PROJECT TERRANE

ULTIMATE PIT
CASE III

JOB No	DRAWING No	REV.
6396		

ARGOR EXPLORATIONS LIMITED

SCHEDULE



CANADIAN BECHTEL LIMITED

5640 PARE STREET
MONTREAL 9, P.Q.
TELEPHONE · 342-0160
MAILING ADDRESS:
P.O. BOX 2340 ST. LAURENT P.Q.

July 27, 1967

Mr. R.M. Smith
Consolidated Morrison Explorations Ltd.
Suite 1700
11 King Street West
Toronto, Ontario

Dear Mr. Smith:

We are pleased to present this interim report covering our very preliminary assessment of your potential columbium property near Moosonee, Ontario.

It should be stressed that the data presented in this report are of a very preliminary nature and are not intended to be used for appropriation of the capital required to develop the property into production.

The purpose of this interim report is to assist your company in making the decisions necessary to continue with the feasibility study and to proceed with further exploration of the property. To aid you in your decisions, we have presented you with three possible mining schemes together with order of magnitude capital, operating costs and projected earning potentials, assuming a minimum of 20 years reserves.

No consideration has been given to the effect on the cash flows should the recommendations of the Carter Report be implemented, or to savings in capital requirements which might be made in the following areas :

a) Government subsidy in building the access road and operation of the ferry at Moosonee.

b) Elimination of the airfield.

c) Reduced pit stripping quantities when additional soils data becomes available.

d) Takeover of the contractor's equipment for continuation of the stripping programme.

The preliminary economic assessments given in this report indicate that the discounted rate of return on investment for each of the three schemes, although low, are sufficient to warrant further expenditures to more fully define the total project, to allow a detailed capital cost estimate to be made, and to allow the preparation of more definitive cash flow calculations. Of prime importance is the work required to obtain a bulk sample of the ore for metallurgical testing prior to final plant design. This testing will provide concentrates which will be of value in market studies and will assist in establishing firm sales committments for the product. A further very important benefit from the bulk sample programme is that in sinking an exploration shaft through the overburden, valuable additional information of the overburden strata will be obtained.

A schedule of work, together with key decision points, are presented in the appendix of this report. This schedule will allow additional work to progress in an orderly manner with a minimum of investment at each stage. This work would be divided into three phases, as follows :

Phase I

a) Initiate shaft sinking and underground drifting to produce a bulk sample.

b) Examine, log and record all soil information developed in sinking the shaft through the overburden.

c) Evaluate the rock encountered during drifting through a preliminary rock mechanics programme.

d) Prepare final comparison of open-pit versus underground mining methods. Decision required by December 1967.

Phase II

a) The bulk sample will be tested by gravity and flotation methods to produce firm criteria for plant design.

b) If a decision is reached to mine by underground methods, deep drilling will be required to prove up ore reserves between the 525 and 825 horizons. This should be initiated by February 1, 1968.

c) A preliminary design should be made of the plant and a comprehensive report assembled giving full definition of the total project, with a detailed capital cost estimate.

d) Cash flow calculations prepared at this stage will form the basis of the decision to proceed with the project. Such decision should be made by August 1968, so that some long delivery equipment can be ordered, and concrete and steel design can be initiated.

Phase III

a) A contract should be let by September 1968 to prepare the access road, so that the mine can be fully developed by February 1971.

b) Plant construction should commence early 1969 so as to complete all civil and structural work before the 1969/1970 winter. The buildings should be enclosed by November 1969.

c) Mechanical and electrical installations will be carried out during 1970.

During our meeting with you on July 17, we presented verbally the contents of this interim report, and reached agreement on the principle concerning the approach to be followed over the next several months. We are in the process of obtaining bids for the shaft and cross-cut work necessary to obtain the bulk sample, and at the same time, we are assembling an estimate of the costs involved in the additional work required to complete the feasibility study.

The effect of these events on our present contract with you is, of course, to extend the schedule so that the feasibility report will be completed approximately nine to ten months from now. Some additional costs will probably be incurred on our part which will increase our contract limit. We will assess the probable extent of these costs.

It is anticipated that during the period of shaft sinking, cross-cutting, diamond drilling and bulk sample testing, a minimum of activity will be required on our part. We shall continue to use Mr. K. Culver as Project Engineer on a part time basis, so that he may co-ordinate the activities required during this period and arrange for the necessary application of technical skills for soils evaluation, mining assessments, metallurgical work and other services as required.

Yours very truly

CANADIAN BECHTEL LIMITED

D Rhodes

D. Rhodes
Manager of Engineering

DR:cjm