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CIMA RESOURCES LTD.

MT. HUNDERE PROPERTY
FEASIBILITY STUDY REVIEW

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INTRODUCTION

This report reviews the feasibility study carried out by H.A. Simons (International) Ltd. and Canadian Geoscience Corporation on the Mt. Hundere property in August 1980. Increased ore reserves have been calculated from the results of the 1980 diamond drilling program, and the study has been reviewed to determine the effect of this increase on the economics of operation.

Certain sections of the feasibility study are not considered to be significantly affected by the increased ore reserves, and so are not reviewed in this report. These sections were:

- Review of Metallurgy
- Description of Proposed Plantsite Facilities and Operations
- Manpower Requirement, Facilities and Transportation
- Environmental Considerations
- Project Schedule
- Preproduction Costs

It is important that this report be read in conjunction with the feasibility report, as it is incomplete if considered alone.

SUMMARY

Ore Reserves

From interpretation of sections based on the results of 1979 and 1980 drilling, the following geological ore reserves have been calculated:

Proven Geological Ore Reserves	
- Main Zone	69,099 tonnes
- West Zone	54,910 tonnes
- East Zone	<u>122,462 tonnes</u>
Total	246,471 tonnes

Drilling of the West Zone was limited by topography. If it is assumed to extend to the outcrop, a further 18,290 tonnes of possible ore is indicated.

The East Zone is classed as "probable minable" ore because of poor core recovery and low grade. Broken ground in this area indicates extensive oxidation.

Mine Plans

The orebody is described in sections referred to as "pits" for convenience of identification.

Pit No. 1 containing 66,442 tonnes, is that described in the feasibility study, and is based on interpretation of 1979 drilling results. It is included in this report for reference purposes only.

Pit No. 2 containing 86,396 tonnes, is a modification of Pit No. 1, based on interpretation of the results of the 1980 drilling program. It includes high-grade ore on the lower benches which was formerly classified as "possible" ore, and extra dilution associated with extensive faulting which has become apparent.

Pit No. 3 containing 14,732 tonnes, is a lower grade portion of the orebody which would be mined after Pit No. 2 has been mined. It is a westerly extension of Pit No. 2, containing proven ore.

Pit No. 4 contains the remaining "proven geological" ore reserves in the West Zone and some possible ore. It is a further westerly extension of Pits No. 2 and 3 and, if proven, could be mined after Pit No. 3. It is a relatively low grade ore block (17.7% combined lead and zinc) which could be evaluated by diamond drilling while Pits No. 2 and 3 are in production. Topography makes drilling difficult in this area.

Pit No. 5 containing 99,834 tonnes of diluted ore, is the East Zone. This is a low grade block of ore relatively near the surface. Drilling to date has not been sufficient, and core recovery was poor, so that this pit can only be classed as "probable". Oxidation appears to be extensive.

SUMMARY (Cont'd.)

Mine Plans (Cont'd.)

<u>Pit</u>	<u>Diluted Ore, Tonnes</u>	<u>% Pb</u>	<u>% Zn</u>	<u>Ozs./t.Ag</u>	<u>Waste Tonnes</u>	<u>Stripping Ratio Waste:Ore</u>
Pit No. 2	86,396	14.6	16.9	2.22	195,875	2.26
Pit No. 3	<u>14,732</u>	<u>8.3</u>	<u>8.95</u>	<u>1.66</u>	<u>77,596</u>	<u>5.27</u>
Total Pits No. 2 & 3	101,128	13.7	15.7	2.14	273,471	2.70
Pit No. 4 *	41,130	7.4	10.3	1.55	245,365	5.96
Pit No. 5	<u>99,834</u>	<u>5.8</u>	<u>6.9</u>	<u>2.32</u>	<u>96,465</u>	<u>0.97</u>
Total Pits No. 4 & 5	140,964	6.3	8.0	2.12	341,830	2.42
Total of All Pits (Proven + indicated ore)	<u>242,092</u>	<u>9.4</u>	<u>11.2</u>	<u>2.13</u>	<u>615,301</u>	<u>2.54</u>

* Although Pit No. 4 contains part of the "proven geological ore reserves", it is classified here as indicated because it will not be mined unless the "possible" ore in this section also becomes "proven".

SUMMARY (Cont'd)

Duration of Operation

In the feasibility study it was estimated that the mill would operate at a design capacity of 250 tonnes/day with 95% availability, for 5 months/year (May through September). This is equivalent to an annual throughput of 36,100 tonnes.

It is anticipated that throughput may drop to 34,200 tonnes in the third year as availability drops to 90%, and then to 32,300 tonnes in the fourth year as availability drops to 85%.

It is therefore, estimated that the ore from Pits No. 2 and 3 could be processed in 3 years, and if the fourth season is extended to 6 months, it should be possible to process the ore from Pit No. 4.

If Pit No. 5 is "proven", the life of the mine will be extended by a further 3 years, provided 85% availability can be maintained.

<u>Source of Ore</u>	<u>Year 1</u>	<u>Year 2</u>	<u>Year 3</u>	<u>Year 4</u>	<u>Year 5</u>	<u>Year 6</u>	<u>Year 7</u>
Pit 2	36,100	36,100	14,196	-	-	-	-
Pit 3	-	-	14,732	-	-	-	-
Pit 4	-	-	5,272	35,858	-	-	-
Pit 5	-	-	-	-	32,300	32,300	35,234

SUMMARY (Cont'd)

Pre-Production and Operating Costs.

Pre-production costs are not expected to be affected by the change in ore reserves. Although the mobile equipment will now be used for up to seven years, the total operating hours and "tonnage moved" will still be relatively low.

Operating costs have been split into two sections in this report.

- Mine Operating Costs....

are expected to increase slightly due to the higher stripping ratio. An average figure of \$11.015/tonne of ore has been calculated for the 101,169 tonnes from Pits 2 and 3. This figure is expected to be the same for Pits 4 and 5.

- Plant site Operating Costs....

are expected to increase with time, as availability of equipment drops. The cost increases from \$29.89/tonne of ore in the first year, to \$33.39/tonne of ore in the sixth year.

SUMMARY (Cont'd.)

Summary of Income and Costs

	<u>Pit 2</u>	<u>Pit 3</u>	<u>Pits 2 & 3</u>	<u>Pit 4</u>	<u>Pit 5</u>	<u>Pits 4 & 5</u>	<u>Pits 2, 3, 4 & 5</u>
Diluted Ore (tonnes)	86,396	14,732	101,128	41,130	99,834	140,964	242,092
Lead Concentrate (tonnes)	14,170	1,373	15,543	3,418	4,207	7,625	23,168
Zinc Concentrate (tonnes)	19,544	1,764	21,308	5,668	6,110	11,778	33,086
<u>Income (\$ 1,000)</u>							
Revenue from Concentrate Sales	<u>16,367</u>	<u>1,649</u>	<u>18,016</u>	<u>4,542</u>	<u>6,262</u>	<u>10,804</u>	<u>28,820</u>
<u>Expenses (\$ 1,000)</u>							
Preproduction Costs	6,010	-	6,010	-	-	-	6,010
Minesite Operating Costs	952	162	1,114	453	1,100	1,553	2,667
Plantsite Operating Costs	2,604	462	3,066	1,345	3,320	4,665	7,731
Concentrate Transportation	2,478	231	2,709	667	759	1,426	4,135
Total Expenses	<u>12,044</u>	<u>855</u>	<u>12,899</u>	<u>2,465</u>	<u>5,179</u>	<u>7,644</u>	<u>20,543</u>
Pre-tax Profit	<u>4,323</u>	<u>794</u>	<u>5,117</u>	<u>2,077</u>	<u>1,083</u>	<u>3,160</u>	<u>8,277</u>

SUMMARY (Cont'd.)

Sensitivity to Metal Prices

Pit No. 2:

86,437 tonnes of ore; 14,170 tonnes lead concentrate, 19,544 tonnes of zinc concentrate.

A 1.0¢/lb. increase in the price of lead would increase revenue by \$207,740 (Can.).

A 1.0¢/lb. increase in the price of zinc would increase revenue by \$212,418 (Can.).

A \$1.00/oz. increase in the price of silver would increase revenue by \$105,549 (Can.).

Total Proven and Indicated Ore:

242,133 tonnes of ore; 23,168 tonnes of lead concentrate; 33,086 tonnes of zinc concentrate.

A 1.0¢/lb. increase in the price of lead would increase revenue by \$339,657 (Can.).

A 1.0¢/lb. increase in the price of zinc would increase revenue by \$359,601 (Can.).

A \$1.00/oz. increase in the price of silver would increase revenue by \$249,021 (Can.).

SUMMARY (Cont'd.)

Sensitivity to Metal Recovery

Pit No. 2:

86,396 tonnes of ore; 14,170 tonnes of lead concentrate; 19,544 tonnes of zinc concentrate.

If lead recovery was reduced by 1% (to 77.6%), concentrate tonnage would drop to 13,990 tonnes, representing a \$85,488 decrease in revenue.

If silver recovery was reduced by 1% (to 64.9%), concentrate grade would drop to 8.79 ozs./tonne, representing a \$44,210 decrease in revenue.

If zinc recovery was reduced by 1% (to 76.6%), concentrate tonnage would drop to 19,292 tonnes, representing a \$60,238 decrease in revenue.

Total Proven and Indicated Ore:

242,092 tonnes of ore; 23,168 tonnes of lead concentrate; 33,086 tonnes of zinc concentrate.

If lead recovery was reduced by 1%, concentrate tonnage would drop to 22,896 tonnes, representing a \$128,982 decrease in revenue.

If silver recovery was reduced by 1%, concentrate grade would drop to 11.80 ozs./tonne, representing a \$63,775 decrease in revenue.

If zinc recovery was reduced by 1%, concentrate tonnage would drop to 32,490 tonnes, representing a \$109,567 decrease in revenue.

GEOLOGICAL AND MINABLE RESERVES

The geological and minable reserves are summarized in Table 1 with tonnages rounded to the nearest 100 t.

MAIN ZONE : PROVEN RESERVE

The geological reserve of the main zone was calculated by W.S. Read (Ref: 1) for the portion drilled in 1979. The mineralized areas were shown on geological cross sections at each 10 m interval from 4+70W (on the grid system used) to 5+40W. Each section was assumed to represent a length of 10 m, so that volumes were calculated between 4+65W and 5+45W. The volumes were converted to tonnes using a specific gravity factor of 4.0. This gave a total of 71 236 t of mineralized material in place at grades of 15.6% lead, 18.9% zinc and 2.36 troy ounces silver per short t. Later test work provided an average value for the specific gravity of 3.88. With this figure the tonnage re-calculates to 69 099 t; the grades remain unchanged.

The 1980 work does not achieve as detailed a development pattern as the 1979 program. Areas of mineralization have been developed by the drilling carried out on geological cross sections at 5+50W, 5+70W, 6+00W and 6+20W. On the assumption that continuity extended halfway to the preceding section and 10 m past the last section a tonnage has been calculated as an additional geological reserve of 54 910 t at grades of 11.52% lead, 13.15% zinc and 2.11 ounces silver per st. A specific gravity factor of 3.88 was used in this calculation between sections 5+45W and 6+30W.

The section of the main zone which extends from 4+65W to 6+30W has been defined to a degree which permits its classification as a "proven geological reserve". The total of the 1979 and 1980 figures are shown in an accompanying table section by section. The total proven reserve is 124 009 t at average grades of 13.77% lead, 16.37% zinc and 2.25 ounces silver per st. The 1979 work indicated some values in cadmium to be present but assays for this metal were not carried out in 1980.

With consideration for the position of the mineralized material with respect to the surface, several mining plans have been developed to remove the main zone by open pit methods. An initial plan, based upon the 1979 work, would remove 66 442 t of the mineralized material for milling. This material was calculated to have grades of 14.8% lead, 18.1% zinc and 2.23 oz silver per st. In this case a 10% dilution factor was assumed and a stripping ratio of 1.37 t waste per t ore resulted. This proposal has been called Pit 1.

A second mining plan termed Pit 2 was prepared to take in the better grade portion of the 1980 drilling as well as all of the material contained within the limits of Pit 1. Pit 2 extends from 4+65W to section 5+74W in the mineralized zone. With indications of some disturbance of ore boundaries by longitudinal faulting it was decided to increase the dilution factor from 10% to 20% by volume. The Pit 2 calculation extends to the flat floor of Bench 8 at 1380 m elevation. The mineralized material to those limits was calculated as 86 396 t at grades of 14.6% lead, 16.9% zinc and 2.22 oz silver per st, with a stripping ratio of 2.26 t waste per t ore.

Pit 3 was then planned as a further expansion of Pit 2 to take in most of the proven geological reserve which lies above the floor of bench 8

at 1 380 m. The proposed pit extends to cross section 6+00W. The increment between Pit 2 and Pit 3 amounts to 14 732 t but the grade is lower and the stripping ratio is 5.27 t waste per t ore. The overall figure for Pit 3 is 101 128 t at grades of 13.7% lead, 15.7% zinc and 2.14 oz silver per st with a stripping ratio of 2.70 t waste per t of ore.

Because of effects of dilution factors and the establishment of mining limits the mining tonnages and grades cannot be directly related to the geological reserve but apparently about 80% of the proven geological reserve would be mined by Pit 3.

WEST ZONE: POSSIBLE RESERVE

The main zone extends from section 4+65W and is apparently structurally controlled by the intersection of several steeply dipping east-west (grid direction) faults with a favorable limestone horizon of the country rock. The northwesterly strike and gentle dip to the south-west (grid direction) of the country rock gives a pipe-like body which plunges to the west as shown on the longitudinal section (Fig. 13). The plunge is at about 20 degrees below the horizontal at the outcrop but it gradually flattens to about 10° below the horizontal. The 10° plunge of the pipe carries it from the drilled section to an outcrop area where the surface profile steepens to intersect the pipe. It is about 65 m to the outcrop from drill hole S-32 on section 6+20W. Although holes S-25 and S-26 attempted to intersect the zone, they appear to have been drilled too far south. The outcrop area has a sample across its southern end for 11.0 m which assays 9.9% lead, 29.5% zinc and 2.99 oz silver per st. It cannot be assumed that this represents the entire outcrop area since the sample direction closely

follows the trend of the mineralizing faults which could be higher grade. This block of the zone which is undefined in terms of quantity and grade remains to be proven by a few drill holes from the surface above, which is steep and difficult to access. In the meantime, this continuation should be classified as geologically inferred.

A mining plan has been prepared which would remove the portion of the mineralized zone which lies between Pit 3 and the outcrop area to the west. This plan would extract the balance of the proven geological reserve which lies below bench 8 at 1 380m elevation, and from the face of the pit at 6+00W to the limit of the proven geological reserve at 6+30W. This would be some 20 000 t of material. In addition it could extract the undefined inferred material between 6+30W and the outcrop. The mining plan is based upon the projection of data over large distances. Development extends to bench 10 at 1 366.7 m elevation. Access is planned from the south west with a revision of the present road to permit 10% grades.

The mining plan for the west zone would make available 41 130 t of material with grades of 7.4% lead, 10.3% zinc and 1.55 oz silver per st with a stripping ratio of 5.96 t waste per t ore. By virtue of the relatively low grade, the uncertainty factor of part of the block, and the high stripping ratio this material is classified as "Possible Minable Reserve".

EAST ZONE: PROBABLE RESERVE

About 50 m to the east from the Main Zone there is an area of lead zinc mineralization which was previously developed by three surface strips

and two drill holes, S-4 and S-5. During the past season an additional nine drill holes were put down which have proven a substantial zone of mineralization. The assays for lead, zinc and silver average out to lower figures than in the main zone. It may be more important that all of the drill holes show the lead and zinc minerals to be in an oxidized state which may seriously affect the mill recoveries.

A geological reserve has been proven extending from cross section 3+45W to 4+20W which includes the better mineralization of the zone which amounts to 122 462 t of material at average grades of 4.61% lead, 6.97% zinc and 2.91 oz silver per st. The tonnage is based upon a specific gravity estimate of 3.8 which is about the same value as was used for the main zone. This zone with lower metal content and greater intermixture of country rock with the skarn may have a lower specific gravity than the figure used but no determinations have been made.

A relatively poor core recovery in the mineralized zone of the east block of less than 50% introduces considerable uncertainty as to the true nature of the material which was tested by the diamond drill holes. The material which was recovered from the mineral zone by the core drilling was to a large extent oxidized and badly broken up. Eventual bulk sampling would be desirable accompanied by milling tests.

A mining plan has been prepared by which 99 834 t of the mineralized material with average grades of 5.83% lead, 6.94% zinc and 2.32 oz silver per st with a stripping ratio of 0.97 t waste per t ore could be removed by the conventional equipment proposed for the main zone. This calculation is based on a 3.0 specific gravity. A 10% dilution factor was used.

Access to this mining area will be by a road on the north or footwall side of a proposed pit which will finally climb at 10% grade out of the pit to connect with the main haul road about 100 metres east of the main pit at an elevation of 1426.7 which corresponds with Bench 1 of the main pit proposal.

In view of the uncertainty as to mill recovery and the lower grade of the east zone it is classed as "Probable Movable Reserve".

MINE OPERATIONS AND EQUIPMENT

DRILLING

One Gardner-Denver airtrack ATD 3100/PR123 with a P750D portable compressor was selected to drill the argillite and limestone. The basic compressor for the ATD 3100 is a 600 cfm unit. However, the factor for operating at 4600 ft. elevation is 1.2 and thus the compressor required is $600 \times 1.2 = 720$ cfm. A 750 cfm unit was selected. The compressor requires a 2-inch air hose with clamps. A Bitco bit grinder is required to regrind drill bits.

Holes which are to be drilled 10 ft.(3.0m) or deeper should all be started with a 2.5"(63.5mm) diameter drill bit. Both the argillite and limestone will wear the body of the bit without too much wear on the carbilloy inserts. This will necessitate grinding down the shoulders of the insert to prevent chipping of the corners. The worn bits, with diameters down to slightly less than 2"(50mm), can be utilized in the short holes of less than 10 ft.(3.0m) when no couplings are added. The increased penetration rate on the small diameter holes will maintain the estimated rate per hour even though the intervals for spotting holes is increased.

DRILL LENGTH REQUIRED FOR PIT 3

In the calculation for Pit 1 average ratios of volume to length of drill hole were obtained for the two types of rock for all types of

benches for the pit. These were as follows:

Ore - 2.12 m³ per meter of drill hole

Waste - 3.65 m³ per meter of drill hole

Accepting these ratios, from the total volumes of ore and waste for Pit 3 the length of drilling required for Pit 3 would be 12 939 m in ore (27 431 m³/2.12) and 27 543 m in waste (100 532 m³/3.65). The total length drilled would then be 40 432 m at an average ratio of 3.16 m³/m. The productivity per 10 hour shift is based on 80% productive hours and 80% availability. Using a basic production rate of 15 m/h (49 ft/hr) this gives a shift average of 95.9 m (315 ft). A total of 422 shifts are scheduled for drilling. Drilling and blasting are scheduled for the 434 days of operation. If these productivities are not met then a second shift could be scheduled (i.e. 2 x 5 instead of 1 x 7 shifts per week).

BLASTING

The small diameter bore holes will provide good distribution of explosives, resulting in fine fragmentation. Powder factors for the waste rock should range around 0.75 to 0.85 lbs. per cu. yd. The heavy massive ore will require powder factors in the neighborhood of 1.25 lbs. per cu. yd. This figure for ore may sound high but the additional expense should be more than saved in the crushing costs.

Considerable thought should be given to drilling and blasting the contact zones between ore and waste. Regardless of the care taken, some dilution or loss of ore will take place if both are blasted together.

METHOD OF OPERATION

To take complete advantage of the cost savings by AN/F0 one must consider blending the product on site. Mixing AN/F0 for a small operation is made easy by the use of a Portable AN/F0 Mixer. The unit can be mounted on the rear of a one ton truck, or larger, and is run hydraulically by a power take-off from the truck's transmission. Two men can easily lift the unit off the truck onto a prepared bracket or stand so that the truck can be utilized for other purposes.

To operate the unit AN Prills in bags are dumped into the hopper. A pump built into the unit draws diesel fuel from a storage tank and properly blends the mixture at the rate of 94% AN Prill to 6% diesel fuel. Gravity feed of AN/F0 is directed into the borehole by a poly-sock. A counter gives the operator the number of pounds being poured into the borehole.

The major advantage of blending AN/F0 is the savings available in purchase price, shipping and storage.

LOADING

A small, hydraulic, front shovel unit was selected to load ore and waste. For example, a Poclain HC300 with a 2.3m^3 (3 cu.yd) bucket would be more than adequate to handle the materials.

The loader is not highly utilized in this operation (less than 20% of the time), thus it is considered a used unit should be satisfactory for production purposes.

The ore and waste loading studies are summarized in Table 17. Using a 2.3m^3 (3 cu.yd) bucket and 90% fill factor, the specific gravity of the materials with a 50% swell factor in each case; six shovel swings per truck for ore and eight shovel swings per truck for waste are required for each load. The trucks make a total of 3 243 ore trips and 9 110 waste trips. This indicates 19 456 shovel passes with ore and 72 877 shovel passes with waste, requiring a total loading of 769 hours of the 4 763 hours which the trucks will operate.

Since the shovel must be available while the truck is operating, modified service costs were applied as follows. The full charge for general service and fuel was applied plus 60% of the wear and 50% of the repairs to reflect the lighter utilization of the equipment. The loader is scheduled for the same 297 x 10-hour shifts as the trucks.

Alternatively, a Poclain 400, or equivalent, with a 3.4m^3 (4.5 cu.yd) or 4m^3 (5.25 cu.yd) bucket, or a Cat 245, or equivalent, front shovel with a 3.8m^3 (5 cu.yd) bucket would be adequate to load the trucks. Also an equivalent front-end loader alternative was not pursued due to operational considerations and the relatively high operating cost of tires of this unit. At the production rate considered, it is more economic to take the smallest unit available with a reasonable shovel match to the truck. If a used unit, similar to the Poclain 300 with a 2.3m^3 (3 cu.yd) bucket, could be purchased, this would be adequate to load the material.

HAULAGE

A 31.75 t (35 ton) pit truck was selected to haul ore and waste from the mine. Based on considerations of haul cycle time and the distances involved, two units are recommended.

The final truck selection requires further engineering analysis. A study of the haul profile shows that this may be the single critical item in the operation. With a long, down hill haul, it is likely that with an overload condition, the TMPH rating for the tires will be exceeded. When the initial haulage profile was considered, the TMPH rating for the trucks could be exceeded or the haul time would be adversely affected by the gradients encountered. A road relocation was considered and the haulage profile used in this report is based on the assumption that the road will be upgraded to this profile. The road improvement was discussed in the section on mill preproduction preparation and the cost included in that cost centre.

From the loading study (Table 17) each truck carries 31.2 t lead/zinc per trip and 30 t waste per trip. Thus, 3 243 return trips from the mine to the mill are required; 9 110 trips to dump waste are needed.

From the haul study and productivities (Table 18), hauling requires 4 763 hours operating and requires 5 954 hours of shifts to be scheduled.

A total of 595 shifts are necessary to move all the material, (297 shifts x 2).

Either two 10-hour shifts per day or two trucks are required. For operational considerations, it is suggested that two trucks should be purchased. If the haul speeds of study (a) are not met, then the same number of trips could take a total of 5 736 hours operating and require 7 170 shift hours scheduled, (717 10-hour shifts for one truck, or 359 10-hour shifts for two trucks). There are still some additional shifts available if the operating availabilities are not met due to local conditions.

The ore haul is 29 km (18 miles) return trip against 146 m (480 feet) for the waste haul. The cycle time of the truck on the ore haul is estimated to be 4.66 minutes plus travel time of 54 minutes. The waste haul is estimated to be 10.92 minutes in total. The time required to haul the waste is considerably less than that required to haul the ore. Due to the long ore haul, this will be the important factor in the analysis. If a 32 km/h (20 mph) speed takes 54 minutes, a 24 km/h (15 mph) haul increases the travel time to 72 minutes per round trip; and at 16 km/h (10 mph), the return trip is increased to 108 minutes. Further work is required to insure that this item is covered in sufficient detail.

Before final truck selection, a full study based on the actual profile and likely operating conditions and life of the property should be made. Caterpillar have a depot at Watson Lake and the Wabco truck is serviced from a depot at Whitehorse or Vancouver. Cassiar, (which uses Wabco), is served by Arrow Transfer from Stewart out of Vancouver.

A radio dispatch system is recommended for safety.

MISCELLANEOUS EQUIPMENT

A D7 dozer and 120G grader were selected as auxiliary units.

A 1-ton truck is included for general purpose use, as a fuel and lube truck and as a unit to carry the AN/F0 mix unit for blasting.

For staff and supply transportation on the property a pick-up truck and two small trucks were selected. A small bus is required to transport the crew.

MANPOWER FOR MINING

The staff and labour complements for the mining operation are outlined in Table 19. So far as possible, people will be hired at Watson Lake but key people may have to be hired from Vancouver.

CONCENTRATE PRODUCTION

It is estimated that the proportion of oxidized mineral in Pits No. 2, 3 and 4 is 16.1%, as stated in the feasibility study. New geological data does not suggest any change. It is, therefore, estimated that metal recoveries and concentrate grades will be unchanged, except for changes in silver grade in the lead concentrate. This grade is related to the ratio of lead to silver in the ore.

Metal recoveries are expected to be:

- 78.6% lead recovery
- 77.6% zinc recovery
- 65.9% silver recovery

Lead concentrate is expected to contain 70% lead, and zinc concentrate is expected to contain 58% zinc.

The following table shows the expected concentrate production from the four proposed pits (2, 3, 4 and 5) and production from Pit No. 1 is included for comparison. It must be stressed that Pits No. 4 and 5 contain some possible ore, the tonnage and grade estimates being based on insufficient data.

The lead:silver ratio in Pit No. 5 is considerably lower than in the samples submitted for metallurgical testwork and, consequently, it is very difficult to predict silver recovery, and concentrate grade. Oxidation also appears to be more extensive.

Using 65.9% silver recovery, lead concentrate grade becomes 23.32 ozs./tonne, so that the lead concentrate is worth approximately 50% more than that produced from the ore in Pits No. 2 and 3. If we assumed a lower recovery of silver, and 10 ozs./tonne silver in the lead concentrate, revenue would be reduced by approximately \$1 million. For the purposes of this report, we have assumed lower metal recovery in this pit (50% lead, 50% zinc and 40% silver recovery).

CONCENTRATE PRODUCTION (Cont'd.)

	<u>Pit 1</u>	<u>Pit 2</u>	<u>Pit 3</u>	<u>Pit 4</u>	<u>Pit 5</u>
Mill Feed (tonnes)	66,442	86,396	14,732	41,130	99,834
Grade - % Pb	14.8	14.6	8.3	7.4	5.8
- % Zn	18.1	16.9	8.95	10.3	6.9
- ozs./t. Ag	2.23	2.22	1.66	1.55	2.32
Cumulative Mill Feed (tonnes)	66,442	86,396	101,128	142,158	242,092
Grade - % Pb	14.8	14.6	13.7	11.9	9.4
- % Zn	18.1	16.9	15.7	14.1	11.2
- ozs./t. Ag	2.23	2.22	2.14	1.97	2.13
Lead Concentrate (tonnes)					
- Increment	11,041	14,170	1,373	3,418	4,207
- Cumulative	-	-	15,543	18,961	23,168
Zinc Concentrate (tonnes)					
- Increment	16,094	19,544	1,764	5,668	6,110
- Cumulative	-	-	21,308	26,976	33,086
Silver Grade of Lead Concentrate					
- Increment	8.85	8.92	11.74	12.29	22.40
- Cumulative	-	-	9.17	9.73	12.03

CONCENTRATE TRANSPORTATION

As per the feasibility study, it is assumed that sales would be to Japan and transportation would be \$70/tonne of concentrate.

	<u>Pit 1</u>	<u>Pit 2</u>	<u>Pit 3</u>	<u>Pit 4</u>	<u>Pit 5</u>
Cumulative Concentrate Tonnage:					
- Lead	11,041	14,170	15,543	18,961	23,168
- Zinc	16,094	19,544	21,308	26,976	33,086
- Lead and Zinc	27,135	33,714	36,851	45,937	56,255
5% Moisture (tonnes)	1,357	1,686	1,843	2,297	2,813
Total tonnage of Wet Concentrate	28,492	35,400	38,694	48,234	59,068
Transportation Cost (x \$1,000)					
- Cumulative	1,994	2,478	2,709	3,376	4,135
- Increment	1,994	484	231	667	759

CONCENTRATE SALES

It is assumed that sales will be to Japan. At the request of the Client, the metal prices used are the averages for September 1980, with the Canadian dollar set at U.S. \$0.8587.

Zinc: 42.0¢ Can. per pound (L.M.E.)
Lead: 46.6¢ Can. per pound (L.M.E.)
Silver: Can. \$23.51 per pound (H & H, N.Y. Price)

Lead Concentrate

Pay \$719.14/tonne for contained lead.
Pay \$ 22.22/tonne for contained zinc.
* Pay \$184.34/tonne for contained silver.
Deduct \$197.16/tonne for treatment, etc.
\$728.54/tonne net smelter return.

Zinc Concentrate

Pay \$456.48/tonne for contained zinc.
Pay \$ 3.15/tonne for contained cadmium.
Deduct \$150.40/tonne for treatment, etc.
\$309.23/tonne net smelter return.

* The net smelter return for lead concentrate varies with silver content. The above calculation is true for a silver grade of 8.92 ozs./tonne. For higher grade concentrates, the following figures apply:

<u>Pit</u>	<u>Ozs./Tonne Silver in Pb Conc.</u>	<u>\$/Tonne For Contained Silver</u>	<u>Net Smelter Return for Pb Conc. (\$/tonne)</u>
2	8.92	184.34	728.54
3	11.74	259.58	803.78
4	12.29	271.75	815.95
5	22.40	495.29	1,039.49

CONCENTRATE SALES (Cont'd.)

Total net smelter returns (N.S.R.) for the concentrates from each pit are summarized below:

<u>Pit</u>	<u>N.S.R. Lead Concentrate</u>	<u>N.S.R. Zinc Concentrate</u>	<u>Total N.S.R.</u>
2	10,323,411	6,043,591	16,367,002
3	1,103,590	545,482	1,649,072
Total for Pits 2 & 3	11,427,000	6,589,073	18,016,073
4	2,788,917	1,752,716	4,541,633
Total for Pits 2, 3 & 4	14,215,917	8,341,789	22,557,706
5	4,373,134	1,889,395	6,262,529
Total for Pits 2, 3, 4 & 5	18,589,051	10,231,184	28,820,235

MINE OPERATING COSTS

There is no change in capital costs which are given in Table 20. No salvage values are assigned for the mine equipment at the end of operation. The capital and operating costs are summarized in Table 21.

DRILLING

Current prices for new and used equipment are shown in Table 20. A rental schedule was obtained for the alternative of renting, rather than purchasing this equipment. Details of drilling supplies and operating costs are provided in Table 22.

BLASTING

Blasting waste separately from the ore will increase blasting costs because of the short holes and the cost of a detonator opposite the yield of these short holes. For the Pit 1 calculations charts were prepared to show these costs in increments of 1.5m (5 ft.) depths. The escalation of costs increase as the depth of bench is diminished.

The Portable AN/FO Unit sells for \$7 750.00. Adding taxes, mechanical work, power take-off, hoses, etc., the entire unit should be in operation for a total of \$12 800.

AN/F0 Prills can be purchased in Calgary in bags, palletized, stretch wrapped and shipped. The costs are given in Table 23.

Handling and transportation costs around the property would be about equal for factory-made as an on-site prepared product. The saving in storage, i.e. magazine costs, property, travel to and from is difficult to assess but it is probably significant. Current prices of explosives and blasting caps delivered to the site are given in Table 23.

The cost of blasting labour is flexible as most of this work can be performed by the foreman or driller, provided that he has a Blasting Certificate. It is assumed that the labour cost is included in the drill operating cost centre.

PIT 3: BLASTING COST

From the calculation for Pit 1 costs of drill holes in different rock types were given. By segregating ore and waste, assuming holes are 50% wet and 50% dry we get average costs which when applied to the volumes for Pit 3 give the costs as shown in Table 24.

LOADING COST

Capital costs for the loader are shown on Table 20. One supplier quoted approximately \$125 000 for a small, used front shovel. The operating costs are shown in Table 25.

For this project the calculated total haul time is 4 763 actual and 5 954 hours scheduled for two trucks. The actual time required to load material is 769 hours. The proportion of time during which shovel is working on loading is $100 \times 769 / (4763/2) = 8\%$. The rest of the time is idling or light duty. For costing assume 50% for repairs, 60% for wear and 100% for service and fuel.

HAULING COST

Capital costs for the haulage units are shown in Table 20. The initial capital cost for the Haulpac 35D is approximately \$70 000 less than that for the Cat 769C. The estimated operating costs are shown in Table 26.

CAPITAL COSTS OF MISCELLANEOUS EQUIPMENT

The capital costs of the required equipment are provided in the equipment list (Table 20). Because of the short indicated life of the operation and the light duty, the dozer and grader are listed as being purchased in used condition rather than new. No salvage value is allowed on the miscellaneous equipment. Maintenance and service costs of the miscellaneous equipment is estimated as per Table 27.

MANPOWER COSTS

Rates for labour (Table 28) are those currently being paid to Cyprus Anvil employees under their labour agreement with the addition of 25%

burden for service and fringe benefits. Table 29 summarizes the operating labour cost for the mine by major cost centre as per the operating rationale given above.

The staff rates (Table 30) correspond with rates being paid in Yukon for positions with similar duties and responsibilities.

So far as possible, people will be hired at Watson Lake but key people may have to be hired from Vancouver.

PLANTSITE OPERATING COSTS

In the feasibility study, the plantsite operating costs were broken down into the following sections:

- Crushing Plant
- Concentrator
- Power Plant
- Camp
- Mobile Equipment
- Plant Maintenance Materials
- Watchman During Non-operating Periods
- Shutdown and Startup for Second Season
- Tailings Treatment and Reclamation

Most of these sections are directly related to the duration of operation and, consequently, to tonnes of ore being processed. The three sections which are not directly related to time and throughput are:

- Plant Maintenance Materials
- Shutdown and Startup for Second Season
- Watchman During Non-operating Periods.

Plant maintenance can be expected to increase after the second season. It is estimated that in the third season this cost will be \$1.50/tonne of ore, and will reach \$2.00/tonne of ore in the fourth season. A figure of \$2.50/tonne is used for the remaining years.

It has been stated earlier in this report that milling rate is expected to decrease as the equipment gets older. It is, therefore, reasonable to assume that that part of the operating cost associated with labor costs will increase in terms of \$/tonne. The following list breaks down the plantsite operating costs estimated for Pit No. 1 in the feasibility study.

Total Plantsite Operating Cost	\$ 29.89/tonne of ore
--------------------------------	-----------------------

a) Labor-related costs:

- Camp	\$ 7.55	
- Crusher labor	0.50	
- Concentrator labor	8.97	
		\$ 17.01/tonne of ore

b) "Tonnage"-related costs:

- Power	\$ 3.48	
- Concentrator materials	3.57	
- Mobile equipment	2.79	
- Tailings treatment	1.66	
		\$ 11.50/tonne of ore

PLANTSITE OPERATING COSTS (Cont'd.)

- c) Watchman \$ 0.27/tonne of ore
- d) Startup and Shutdown \$ 0.10/tonne of ore
- e) Plant Maintenance Materials \$ 1.00/tonne of ore

During the life of the operation, these costs will change (in terms of \$/tonne) as follows:

<u>Year</u> <u>No.</u>	<u>Milling</u> <u>Rate</u> <u>(t/hr.)</u>	<u>Cost A</u> <u>(\$/t)</u>	<u>Cost B</u> <u>(\$/t)</u>	<u>Cost C</u> <u>(\$/t)</u>	<u>Cost D</u> <u>(\$/t)</u>	<u>Cost E</u> <u>(\$/t)</u>	<u>Total Plantsite</u> <u>Operating Cost</u> <u>\$/Tonne</u>
1	237.5	17.02	11.50	0.27	0.10	1.00	29.89
2	237.5	17.02	11.50	0.27	0.10	1.00	29.89
3	225.0	17.97	11.50	0.27	0.10	1.50	31.34
4	212.5	19.02	11.50	0.27	0.10	2.00	32.89
5	212.5	19.02	11.50	0.27	0.10	2.50	33.39
6	212.5	19.02	11.50	0.27	0.10	2.50	33.39
7	212.5	19.02	11.50	-	-	2.50	33.02

The total plantsite operating costs for each year of operation, and each pit are shown below.

<u>Year</u> <u>No.</u>	<u>Tonnes</u> <u>Milled</u>	<u>Pit</u> <u>No.</u>	<u>Plantsite Oper.</u> <u>Cost (\$/Tonne)</u>	<u>Plantsite Oper.</u> <u>Cost (\$)</u>	<u>Total \$ For</u> <u>Each Pit</u>
1	36,100	2	29.89	1,079,029	
2	36,100	2	29.89	1,079,029	
3	14,196	2	31.34	446,188	2,604,246
	14,732	3	31.34	461,701	461,701
	5,272	4	31.34	163,940	
4	35,858	4	32.89	1,180,718	1,344,658
5	32,300	5	33.39	1,078,497	
6	32,300	5	33.39	1,078,497	
7	35,234	5	33.02	1,163,427	3,320,421

CASH FLOW AND RETURN ON INVESTMENT

Income from sale of concentrates will be received approximately one month after a shipment is made. (It is common for the buyer to pay 90% of the price within 30 days of the ship sailing, and the rest upon agreement of tonnage and assays.)

For the purposes of these calculations, it is assumed that all concentrates are shipped at the end of each mining "season" (October 1st), so that income is received once/year, at the end of the "season", and the year's expenses are accrued to that time.

Year No.	Date	Income	Expenses	Cash Flow	R.O.I *	R.O.I *
		(\$ x 1,000)	(\$ x 1,000)	(\$ x 1,000)	#1	#2
1	Sept. 31st	-	8,530	(8,530)	-	-
	Oct. 1st	6,835	-	(1,695)	-	-
2	Sept. 31st	-	2,520	(4,215)	-	-
	Oct. 1st	6,835	-	2,620	31%	31%
3	Sept. 31st	-	2,163	457	-	-
	Oct. 1st	4,924	-	5,381	63%	128%
4	Sept. 31st	-	2,151	3,230	-	-
	Oct. 1st	3,963	-	7,193	84%	84%
5	Sept. 31st	-	1,676	5,517	-	-
	Oct. 1st	2,026	-	7,543	88%	21%
6	Sept. 31st	-	1,676	5,867	-	-
	Oct. 1st	2,026	-	7,893	93%	21%
7	Sept. 31st	-	1,827	6,066	-	-
	Oct. 1st	2,211	-	8,277	97%	21%

* R.O.I #1 is the return on the original investment of \$8,530,000 (required to finance the venture until revenue is received from sale of concentrates.)

* R.O.I #2 is the profit in each particular year, calculated as a percentage of the investment necessary in that year.

They do not take into account any taxes payable, financing costs, or exploration and development expenses.

TABLES

NO.	TITLE	DIVISION
1	SUMMARY OF RESERVES	BOTH
2	MAIN ZONE - TONS	PROVEN GEOL. RES.
3.	MAIN ZONE - ASSAYS	PROVEN GEOL. RES.
4.	MAIN ZONE - METAL	PROVEN GEOL. RES.
5.	EAST ZONE - TONS & GRADE	PROVEN GEOL. RES.
6.	MAIN ZONE - PIT 2 TONS	POSITIVE MIN. RES.
7.	MAIN ZONE - INCR. B TONS	POSITIVE MIN. RES.
8.	MAIN ZONE - ASSAYS	POSITIVE MIN. RES.
9.	MAIN ZONE - PIT 2 GRADE	POSITIVE MIN. RES.
10.	MAIN ZONE - INCR. B TONS	POSITIVE MIN. RES.
11.	EAST ZONE - TONS	PROBABLE MIN. RES.
12.	EAST ZONE - ASSAYS	PROBABLE MIN. RES.
13.	EAST ZONE - GRADES	PROBABLE MIN. RES.
14.	WEST ZONE - TONS	POSSIBLE MIN. RES.
15.	WEST ZONE - TONS	POSSIBLE MIN. RES.
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TABLE 1
SUMMARY OF RESERVES

1. GEOLOGICALLY PROVEN RESERVES

BLOCK	TONNES	GRADES		
		LEAD	ZINC	SILVER
MAIN ZONE	124 000	13.8	16.4	2.25
EAST ZONE	122 500	4.6	7.0	2.91
TOTAL	246 500	9.2	11.7	2.58

2. MINABLE RESERVES

BLOCK	CATEGORY	TONNES	GRADES			S.R.
			LEAD	ZINC	SILVER	
PIT 3	POSITIVE	101 100	13.7	15.7	2.14	2.7
EAST ZONE	PROBABLE	99 800	5.8	6.9	2.32	1.0
WEST ZONE	POSSIBLE	41 100	7.4	10.3	1.55	6.0

TABLE 2
MAIN ZONE (1980)
TONNAGE CALCULATIONS BY SECTIONS

BLOCK	SECTION AREA M ²	WIDTH M	VOLUME M ³	SPEC.GRAV	TONNES
5 + 50	220.8	15	3 312	3.88	12 851
5 + 60					
5 + 70	212.0	25	5 300	3.88	20 564
5 + 80					
5 + 90					
6 + 00	117.6	25	2 940	3.88	11 407
6 + 10					
6 + 20	130.0	20	2 600	3.88	10 088
TOTAL			14 152	3.88	54 910

TABLE 3
PROVEN GEOLOGICAL RESERVE
MAIN ZONE (1980)
AVERAGE ASSAYS OF CROSS SECTIONS
BY DRILL HOLES

WORKING	WIDTH	LEAD		ZINC		SILVER	
		ASSAY %	W x ASSAY	ASSAY %	W x ASSAY	ASSAY OZ/T	W x ASSAY
SECTION 5 + 50							
DH S-29	13.95	19.78	275.931	16.31	227.524	2.89	40.3155
DH S-47	8.63	27.30	235.599	23.50	202.805	5.39	46.5157
TOTAL			511.530		480.329		86.8312
AVG	22.58	22.65		19.06		3.85	
SECTION 5 + 70							
DH S-30	10.76	8.98	96.6248	12.07	129.8732	1.34	14.4184
DH S-40	12.30	9.41	115.7430	8.90	109.4700	2.21	27.1830
DH S-44	2.90	10.40	30.1600	11.71	33.9590	1.65	4.7850
DH S-44	1.41	.72	1.0152	.32	.4512	.16	.2256
TOTAL	27.37		243.5430		273.7534		46.6120
AVG		8.90		10.00		1.70	
SECTION 6 +00							
DH S-31	7.37	7.41	54.6117	13.80	101.7060	1.57	11.5709
DH S-43	8.32	11.57	96.2624	15.38	127.9616	1.66	13.8112
TOTAL	15.69		150.8741		229.6676		25.3821
AVG		9.62		14.64		1.62	
SECTION 6 + 20							
DD S-32	10.01	3.12	31.2312	7.89	78.9789	1.14	11.4114
DD S-42	1.37	17.45	23.9065	28.30	38.7710	2.45	3.3565
TOTAL	11.38		55.1377		117.7499		14.7679
AVG		4.85		10.35		1.30	

TABLE 4
PROVEN GEOLOGICAL RESERVE
MAIN ZONE (1979 & 1980)
ORE GRADE CALCULATION

BLOCK	TONNES	LEAD		ZINC		SILVER	
		GRADE%	TONNES	GRADE%	TONNES	GRADE OZ/ST	TROY OUNCES
4+70	2 033	13.8	280.55	14.1	286.65	2.11	4 740
4+80	7 938	17.6	1 397.09	19.8	1 571.72	2.59	22 718
4+90	5 432	15.5	841.96	18.9	1 026.65	2.32	13 925
5+00	6 507	16.3	1 060.64	17.8	1 158.25	2.64	18 982
5+10	6 860	18.0	1 234.80	19.9	1 365.14	2.94	22 286
5+20	12 020	15.0	1 803.00	16.9	2 031.38	2.23	29 619
5+30	16 855	11.9	2 005.74	17.8	3 000.19	1.89	35 201
5+40	11 454	18.6	2 130.44	23.1	2 645.87	2.58	32 654
5+50	12 851	22.65	2 910.75	19.06	2 449.40	3.85	54 671
5+70	20 564	8.90	1 830.20	10.00	2 056.40	1.70	38 629
6+00	11 407	9.62	1 097.35	14.64	1 669.98	1.62	20 420
6+20	10 088	4.85	489.28	10.35	1 044.11	1.30	14 491
1979	69 099	15.56	10 754.22	18.94	13 085.85	2.36	180 125
1980	54 910	11.52	6 327.57	13.15	7 219.89	2.11	128 211
TOTAL	124 009	13.77	17 081.79	16.37	20 305.74	2.25	308 336

NOTE: THE 1979 FIGURES ARE TAKEN FROM READ'S REPORT EXCEPT TONNAGES ARE REDUCED TO 3.88 S.G.

TABLE 5
PROVEN GEOLOGICAL RESERVE
EAST ZONE
AVERAGE GRADE CALCULATIONS: TONNAGE BLOCKS

BLOCK	SECTIONAL AREA M ²	INFLUENCE WIDTH M	VOLUME M ³	SPECIFIC GRAVITY	TONNES	GRADE % PB	GRADE % ZN	GRADE OZ AG/ST
3 + 55 W	317.5	20	6 350	3.8	24 130	6.49	3.24	5.52
3 + 75 W	615.3	20	12 306	3.8	46 763	3.37	6.72	2.74
3 + 95 W	585.3	17.5	10 243	3.8	38 923	3.83	8.35	2.02
4 + 10 W	190.2	17.5	3 328	3.8	12 646	8.08	10.34	1.52
TOTAL					122 462	4.61	6.97	2.91

TABLE 6
POSITIVE MINABLE RESERVE
MAIN ZONE PIT 2
CALCULATION OF TONNAGES

BENCH	ELEV.	ENTIRE BENCH			ORE ON BENCHES				20% DILUTION		DILUTED ORE		WASTE		S.R.
		AREA M ²	HEIGHT M	VOLUME M ³	AREA M ²	HEIGHT M	VOLUME M ³	TONNES SG3.88	VOLUME M ³	TONNES SG2.72	VOLUME BY ADD.	TONNES BY ADD.	VOLUME BY SUB.	TONNES SG2.72	
1	1 426.7	780	2.50	1 950	220	2.5	550	2 134	110	299	660	2433	1 290	3 509	1.44
2	1 420.0	1 975	6.67	9 184	543	6.67	2 541	9 859	508	1 382	3 049	11 241	6 135	16 687	1.48
3	1 413.3	2 432	6.67	14 694	488	6.67	3 435	13 327	687	1 869	4 122	15 196	10 572	28 756	1.90
4	1 406.7	3 150	6.67	18 615	484	6.67	3 241	12 575	648	1 762	3 889	14 337	14 726	40 055	2.76
5	1 400.0	2 587	6.67	19 129	560	6.67	3 481	13 506	696	1 893	4 177	15 399	14 952	40 669	2.64
6	1 393.3	2 117	6.67	15 687	453	6.67	3 375	13 095	675	1 836	4 050	14 931	11 637	31 653	2.11
7	1 386.7	1 200	6.67	11 058	155	6.67	2 027	7 864	405	1 102	2 432	8 966	8 626	23 463	2.61
8	1 380.0	337	6.67	5 122	110	6.67	880	3 414	176	479	1 056	3 893	4 066	11 059	2.84
		95 439			19 530 75 774				3 905 10 622		23 435 86 396		72 004 195 851		2.26

TABLE 7
POSITIVE MINABLE RESERVE
MAIN ZONE INCREMENT "B" TO PIT 3
CALCULATION OF TONNAGES

BENCH	ELEV.	ENTIRE BENCH			ORE ON BENCHES				20% DILUTION		DILUTED ORE		WASTE		S.R.
		AREA M ²	HEIGHT M	VOLUME M ³	AREA M ²	HEIGHT M	VOLUME M ³	TONNES SG3.88	VOLUME M ³	TONNES SG2.72	VOLUME BY ADD.	TONNES BY ADD.	VOLUME BY SUB.	TONNES SG2.72	
1	1 426.7	NIL	2.50	-	NIL	-	-	-	-	-	-	-	NIL	-	-
2	1 420.0	66	6.67	147	NIL	-	-	-	-	-	-	-	147	400	+
3	1 413.3	405	6.67	1 571	NIL	-	-	-	-	-	-	-	1 571	4 273	+
4	1 406.7	1 125	6.67	5 103	NIL	-	-	-	-	-	-	-	5 103	13 880	+
5	1 400.0	1 158	6.67	7 614	NIL	-	-	-	-	-	-	-	7 614	20 710	+
6	1 393.3	985	6.67	7 174	50	6.67	111	431	22	60	133	491	7 014	19 078	38.80
7	1 386.7	812	6.67	5 993	295	6.67	1 151	4 466	230	626	1 381	5 092	4 612	12 545	2.46
8	1 380.0	672	6.67	4 949	325	6.67	2 068	8 024	414	1 125	2 482	9 149	2 467	6 710	0.73
		32 524			3 330 12 921				666	1 811	3 996	14 732	28 528	7 7596	5.27

TABLE 8
POSITIVE MINABLE RESERVE
MAIN ZONE
DRILLING ASSAYS BY BENCH

BENCH	WORKING	LENGTH M	AVERAGE ASSAYS				
			LEAD	ZINC	LEAD + ZINC	CADMIUM	SILVER
1	PREV CALC.	7.8	16.0	13.8	29.8	.050	2.38
2	PREV CALC.	76.4	16.8	19.8	34.8	.098	2.60
3	PREV CALC.	55.8	15.9	18.4	34.3	.089	2.41
4	PREV CALC.	37.8	15.4	19.4	34.8	.102	2.37
5	PREV CALC.	31.2	15.1	20.3	35.4	.101	2.27
	S-27	3.8	14.81	18.31	35.57		2.14
	NEW TOTAL	35.0	15.1	20.3	35.4		2.25
6	PREV CALC.	19.8	16.5	20.7	37.2	.123	2.37
	S-27	0.5	14.8	18.3	33.1		2.14
	S-27	1.9	14.2	18.4	32.6		1.76
	S-29	5.4	19.8	16.4	36.2		2.89
	NEW TOTAL	27.6	17.0	19.6	36.6		2.42
7	PREV CALC.	6.3	20.7	25.2	45.9	.167	2.85
	S-29	7.0	19.8	16.4	36.2		2.89
	S-47	5.7	27.6	23.7	51.3		5.39
	S-30	5.0	8.6	12.1	20.7		1.34
	NEW TOTAL	24.0	19.6	19.5	39.1		3.15
	TOTAL TO XY	19.0	22.4	21.5	43.9		3.63
8	S-29	1.5	19.8	16.4	36.2		2.89
	S-47	2.9	27.6	23.7	51.3		5.39
	S-44	2.9	10.7	11.6	22.3		1.65
	S-30	5.8	8.6	12.1	20.7		1.34
	S-40	5.0	9.4	8.9	18.3		2.54
	S-43	2.1	11.7	15.4	27.1		1.66
	NEW TOTAL	20.2	13.0	13.6	26.6		2.41
	TOTAL TO XY	13.1	14.6	15.0	29.6		2.48
9	S-43	6.7	11.7	15.4	27.1		1.66
	S-31	6.0	8.1	14.5	22.6		1.65
	S-40	7.0	9.4	8.9	18.3		2.54
	S-32	6.0	3.9	7.9			1.13
	NEW TOTAL	25.7	8.4	11.7	22.6		1.77
ALL		304.3	15.8	18.4	33.8		2.46

TABLE 9
POSITIVE MINABLE RESERVE
MAIN ZONE: PIT 2
GRADE DILUTIONS

BENCH	ORE IN PLACE MINABLE				DILUTED ORE			
	TONNES	LEAD %	ZINC %	SILVER OZ/T	TONNES	LEAD %	ZINC %	SILVER OZ/T
1	2 134	16.0	13.8	2.38	2 433	14.0	12.1	2.09
2	9 859	16.8	19.8	2.60	11 241	14.7	17.4	2.28
3	13 327	15.9	18.4	2.41	15 196	13.9	16.1	2.11
4	12 575	15.4	19.4	2.37	14 337	13.5	17.0	2.08
5	13 506	15.1	20.3	2.25	15 399	13.2	17.8	1.97
6	13 095	17.0	19.6	2.42	14 931	14.9	17.2	2.12
7	7 864	22.4	21.5	3.63	8 966	19.7	18.9	3.19
8	3 414	14.6	15.0	2.48	3 893	12.9	13.2	2.18
TOTAL	75 774	16.6	19.3	2.53	86 396	14.6	16.9	2.22

TABLE 10
POSITIVE MINABLE RESERVE
INCREMENT B TO PIT 3
GRADE DILUTIONS

BENCH	ORE IN PLACE				DILUTED ORE			
	TONNES	LEAD	ZINC	SILVER	TONNES AT 20% DIL	LEAD	ZINC	SILVER
1								
2								
3								
4								
5								
6	431	(9.5)	(10.2)	(1.89)	491			
7	4 466	8.6	12.1	1.34	5 092			
8	8 024	10.1	10.8	2.28	9 149			
		9.56	11.23	1.94				
	12 921	9.47	10.2	1.89	14 732	8.30	8.95	1.66

TABLE 11
PROBABLE MINABLE RESERVE
EAST ZONE
CALCULATION OF TONNAGES

		ENTIRE BENCH			ORE ON BENCHES				10% DILUTION		DILUTED ORE		WASTE		S.R.
BENCH	ELEV.	AREA M²	HEIGHT M	VOLUME M³	AREA M²	HEIGHT M	VOLUME M³	TONNES SG 3.0	VOLUME M³	TONNES SG2.72	VOLUME BY ADD.	TONNES BY ADD.	VOLUME BY SUB.	TONNES SG2.72	TW:TORE
00	1 440.0	740	2.50	617											
0	1 433.3	2 600	6.67	11 139	840	6.67	1 867	5 601	187	500	2 054	6 101	9 085	24 711	4.05
1	1 426.7	4 040	6.67	22 144	1 810	6.67	8 838	26 514	884	2 404	9 722	28 918	12 422	33 788	1.17
2	1 420.0	2 662	6.67	22 351	1 800	6.67	12 039	36 117	1 204	3 275	13 243	39 392	9 108	24 774	0.63
3	1 413.3	1 170	6.67	12 780	530	6.67	7 770	23 310	777	2 113	8 547	25 423	4 233	11 514	0.45
				69 031	4 980		30 514	91 542	3 052	8 292	33 566	99 834	35 465	96 465	0.97

TABLE 12
PROBABLE MINABLE RESERVE
EAST ZONE
ASSAY AVERAGES BY BENCHES

BENCH	WORKING	LENGTH	LEAD	ZINC	LEAD + ZINC	CADMIUM	SILVER
00	NONE						
0	TR NO 1	10.98	14.5	6.4			3.88
1	S-4	9.50	3.98	8.40			2.50
	S-5	3.00	4.70	12.58			1.91
	S-34	2.60	3.08	8.50			1.87
	TR NO 2	19.66	7.95	5.60			2.12
PART	TR NO 3	9.00	5.90	2.60			3.50
	S-37	5.50	8.08	10.34			1.52
	S-38	3.32	2.90	3.42			0.46
2	PART TR NO 3	12.95	5.90	2.60			3.50
	S-4	9.00	3.98	8.40			2.50
	S-5	7.60	4.70	12.58			1.91
	S-34	6.67	3.08	8.50			1.87
	S-37	6.70	8.08	10.34			1.52
3	S-37	0.82	8.08	10.34			1.52
	S-38	3.40	10.06	8.12			1.27
	S-5	4.65	4.70	12.58			1.91
	S-34	1.40	3.08	8.50			1.87
	S-35	7.74	4.65	8.59			2.73
	S-36	4.88	9.16	6.12			14.61
							(4.78) CUT
0		10.98	14.5	6.4			3.88
1		52.58	6.15	6.49			2.23
2		42.92	5.19	7.71			2.45
3		22.89	6.45	8.86			2.69

TABLE 13
PROBABLE MINABLE RESERVE
EAST ZONE
ORE GRADE AND DILUTION

BENCH	ORE IN PLACE				DILUTED ORE			
	TONNES	LEAD %	ZINC %	SILVER OZ/ST.	TONNES	LEAD %	ZINC %	SILVER OZ/ST.
00	-				-			
0	5 601	14.5	6.4	3.88	6 101	13.31	5.88	3.56
1	26 514	6.15	6.49	2.23	28 918	5.64	5.95	2.04
2	36 117	5.19	7.71	2.45	39 392	4.76	7.07	2.25
3	23 310	6.45	8.86	2.69	25 423	5.91	8.12	2.47
TOTAL	91 542	6.36	7.57	2.53	99 834	5.83	6.94	2.32

TABLE 14
POSSIBLE MINABLE RESERVE
WEST ZONE (INCREMENT C)
CALCULATION OF TONNAGES

BENCH	POSSIBLE ORE INCREMENT				ORE ON BENCHES				20% DILUTION		DILUTED	ORE	WASTE		S.R.
	ELEV	AREA M ²	HEIGHT M	VOLUME M ³	AREA M ²	HEIGHT M	VOLUME M ³	TONNES SG3.88	VOLUME M ³	TONNES SG2.72	VOLUME M ³	TONNES BY ADD.	VOLUME BY SUB.	TONNES SG2.72	TW/TO
1	1426.7	0	2.50												
2	1420.0	0	6.67												
3	1413.3	0	6.67												
4	1406.7	220	6.67	734									734	1996	
5	1400.0	950	6.67	3902									3902	10613	
6	1393.3	2000	6.67	9838									9838	26759	
7	1386.7	3000	6.67	16675									16675	45356	
8	1380.0	4375	6.67	24596	550								24596	66901	
9	1373.3	3625	6.67	26680	770	6.67	4402	17080	880	2394	5282	19474	21398	58203	2.99
10	1367.7	1325	6.67	16508	450	6.67	4069	15788	814	2214	4883	18002	11625	31620	1.76
10		0	3.67	24311	0	3.67	826	3205	165	449	991	3654	1440	3917	1.07
EXT															
				101364			9297	36073	1859	5057	11156	41130	90208	245365	5.96

TABLE 15
POSSIBLE MINABLE RESERVE
WEST ZONE (INCREMENT C)
GRADES AND DILUTION

BENCH	ORE IN PLACE				DILUTED ORE			
	TONNES	LEAD %	ZINC %	SILVER OZ/ST	TONNES	LEAD %	ZINC %	SILVER OZ/ST
9	17 080	8.4	11.7	1.77	19 474	7.37	10.26	1.55
10	15 788				18 002			
10X	3 205				3 654			
	36 073	8.4	11.7	1.77	41 130	7.37	10.26	1.55

TABLE 16
POSITIVE AND POSSIBLE MINABLE RESERVES
MAIN AND WEST ZONES
GRADES AND METAL

#	TONNES	Pb %	Zn %	Ag OZ/ST	CONTAINED METAL			WASTE TONNES	SR TW:TO	

					t Pb	t Zn	oz Ag			
PIT										
1	66 442	14.8	18.1	2.23	9 833	12 026	163 352	90 955	1.37	
2	86 396	14.6	16.9	2.22	12 614	14 601	211 458	195 851	2.26	
3	101 128	13.7	15.7	2.14	13 837	15 919	238 420	273 447	2.70	
INC.										
a	19 954	13.9	12.9	2.18	2 781	2 575	48 106	104 896	5.25	
b	14 732	8.3	8.95	1.66	1 223	1 318	26 962	77 596	5.27	
c	41 130	7.4	10.3	1.55	3 042	4 234	70 286	245 365	5.96	

NOTES:

1. Pit 1 ore was calculated to bench 6 from mineral boundaries defined by drilling to 1979 at 10% dilution factor.
2. The mineral boundaries were revised following the 1980 drilling.
3. Pit 2 was calculated to bench 8 based on the revised outlines at 20% dilution factor.
4. Pit 2 was enlarged to Pit 3 to include lower grade material.
5. Increment "a" was determined by subtraction as the difference between Pit 1 and Pit 2.
6. Increment "b" which expands Pit 2 to Pit 3 was calculated by areas at 20% dilution factor.
7. Increment "c" was calculated and extends to bench 10. It is now classified as a possible minable reserve and is referred to as the "West Zone".

TABLE 17
ORE AND WASTE LOADING FACTORS
2.3m³ (3yd³) Hydraulic Shovel, e.g.: Poclain HC300

	ORE	WASTE
1. Bucket Capacity	2.3m ³ (3 yd ³)	2.3m ³ (3 yd ³)
2.1 In Situ Density	3.77	2.72
.2	3 770 kg/bank m ³ (6,355 lb/yd ³)	2 720 kg/bank m ³ (4,585 lb/yd ³)
3. Swell %	150%	150%
4. Loose Mass (2.2)/(3)	2 513 kg/bank m ³ (4,235 lb/yd ³)	1 813 kg/bank m ³ (3,056 lb/yd ³)
5. Fill Factor	.9	.9
6. Volume Per Cycle (1)x(5)	2.07m ³ (2.7 yd ³)	2.07m ³ (2.7 yd ³)
7. Mass Per Cycle (4)x(6)	5 203 kg (11 438 lb)	3 754 kg (8 252 lb)
8. Average Cycle Time	30 sec.	30 sec.
9. Loads Per Truck	6	8
10. Mass Per Truck	31.2 t (34.3s.ton)	30.02 t (33.00 s.ton)
11. Total Mass°	101 169 t (111,519s.ton)	273 471 t (301,447s.ton)
12. No. of Truck Hauls (11)/(10)	3 243	9 110
13. No. of Shovel Passes (12)x(9)	19 456	72 877
14. Number of hours (13)x(8)/3600	162	607
Total Loading	769 hours	

°[These numbers have been revised to 101 128 t ore and 273 447 t waste: the effect is minimal]

TABLE 18
TIME FACTORS FOR HAULING

Loading Shovel: 2.3m³ (3yd³) Hydraulic Shovel, e.g. Poclain HC300
Hauling Unit 31.75 t (35 s ton) Truck, e.g. Haulpac 35D or Cat. 769C

1.1 Capacity of Truck	31.75 t (35 s.ton)
1.2 Struck	17.6m ³ (23yd ³)
2. Loose mass ore	2 513 kg/m ³

Note: Load capacity of truck (1.1)x1000/(2) 12.63m³ (16.5 yd³)

ORE HAUL (return trip):

AVERAGE WASTE HAUL (each way):

29 km (18 mi)	a) @ 20mph = 54 min	146 m (480 ft)
	b) @ 15mph = 72 min	
	c) @ 10mph = 108 min	
Waiting and spotting	40s	40s
Load	180s	240s
Turning & dumping	60s	300s

Sub Total	280s = 4.6 min	To dump 245s
		Return 215s

Total Time	a) 58.67m = .98 h	625s = 10.42 min = .174h
	b) 76.67m = 1.28 h	
	c) 112.67m = 1.88 h	

From loading analysis 3 243 return ore hauls and 9 110 waste hauls.

Waste truck time = 9 110 x .174 = 1 585 h/oper. avail. 1 585/.8 = 1 981h sched.

Ore haul time a) 3 243 x .98 = 3 178 oper./ .8 = 3 973h sched.
b) 3 243 x 1.28 = 4 151/.8 = 5 189h
c) 3 243 x 1.88 = 6 097/.8 = 7 621h

Ore	a) 3 973	b) 5 189	c) 7 621
Waste	1 981	1 981	1 981
	-----	-----	-----
Total	5 954h	7 170h	9 602h
	-----	-----	-----

/10h shifts	595 shifts	717 shifts	960 shifts
/ 2 shifts/day	297	359	480
(3 shifts/day	198	239	320 shifts)

Note: Scheduled operation 434 days (7 days/week)
310 days (5 days/week)
80% productive hours per 10 h shift.
& 80% availability of equipment.

TABLE 19
MANPOWER REQUIREMENTS

- 1 Mine Superintendent/Manager
- 1 Secretary/Clerk
- 1 Engineer
- 1 Engineer Assistant
- 1 Shift Supervisor, Mine
- 1 Shovel Loader Operator
- 1 Driller
- 1 Drill Helper
- 1 Auxiliary Equipment Operator
- 2 Truck Drivers
- 1 Heavy Duty Mechanic

TABLE 20
EQUIPMENT CAPITAL COST SCHEDULE

EQUIPMENT	NEW	ALTERNATIVES USED	RENT/MO	SELECTION CONDITION	COST
DRILLING					
1 G D AIRTRACK (ATD3100/PR123)	\$ 85 050	\$25-40 000	\$2 460	U	\$ 40 000
1 SP750D Portable Compressor	78 235	25-35 000	2 570	U	35 000
Air Hose/Clamps	299	-	118	N	299
Bitco Bit Grinder	1 405	-	140	N	1 405
HAULAGE					
2 @ 769C Truck	332 200	-	-	-	-
2 @ Haulpac 35D	256 532	-	-	N	513 064
LOADING					
1 Poclain 300	-	125 000	-	U	125 000
1 Poclain 400	475 000	-	-	-	-
1 Cat 245 Frnt.Shv.	512 000	-	-	-	-
1 Frnt-end Loader	389 640	-	-	-	-
MISC. EQUIPMENT					
1 D7 Dozer	208 765	70-160 000	-	U	160 000
1 120E Grader	122 050	60- 95 000	-	U	95 000
1 Ford 1-Ton	10-12 000	-	-	N	11 000
2 Datsun Pick-ups	6 000	-	-	N	12 000
1 Minibus	10 500	-	-	N	10 500
BLASTING					
Magazines, Blasting Machine, etc.	21 800	-	-	N	21 800
Sub-Total					\$1 025 068
+15% Contingency					153 760
TOTAL EQUIPMENT COST					1 178 828

TABLE 21
SUMMARY OF MINE COSTS

	CAPITAL	MAINT. & SUPPLIES	LABOUR	TOTAL	PER TONNE
Drilling/	\$ 76 704	\$ 85 198)	\$106 413	\$444 042	4.39
Blasting	21 800	153 927)			
Loading	125 000	96 633	45 248	266 881	2.63
Hauling	513 064	51 443	80 172	644 679	6.37
Auxiliary	288 500	40 306	30 710	359 516	3.55
Staff & Sup.	-	-	279 000	279 000	2.76
	1 025 068	427 507	541 543	1 994 118	19.71
Plus 15% Contingency	153 760	64 126	81 231	299 118	2.96
TOTAL COST	\$1 178 828	491 633	622 774	229 323	22.67
COST/TONNE	\$17.74	\$4.85	\$6.16	22.67	
Plus Mobilization and Travel				\$35 349	\$.35
Total including mobilization and travel				\$2 328 584	\$23.02
Cost per tonne including mobilization and travel				23.02	

NOTES:

1. Cost per tonne calculated on 101 169 t of ore.
2. No salvage values are included in this summary.

TABLE 22
DRILLING SUPPLIES

ITEMS	COST \$	# PER STRING	COST PER STRING	LIFE FT.	NO. REQUIRED	TOTAL COST \$
12 ft steel rod	154.10	-	-	-	-	-
10 ft steel rod	135.50	2	\$270.10	5 000	27	7 293
coupling	29.50	2	59.00	2 500	53	3 127
striker bar	77.95	1	77.95	300	443	34 532
2.5" carbide cross bit	109.00	1	118.00	4 000	33	3 894
			----- \$554.15			----- 48 846

COMPRESSOR FUEL

8 gal/hr x 4.535 l/gal x 8 hrs x 422 = 122 481 litres
at 205 litres/barrel = 597 barrels

Cost @ \$0.26/litre 122 481 x .26 = 31 845

ROCK DRILL OIL

1.5 gals. per 8 hr. shift x 422 shifts = 633 gals
x 4.535 l/gal = 2 871 l

Cost @ \$1.57/l = 2 871 x 1.57 = 4 507

TOTAL COST SUPPLIES
Cost/m \$2.10 (\$0.64/ft)

\$85 198

NOTES:

Total Drill Depth 40 482 m (132 815 ft)
Production rate 15 m/h (49 ft/hr)
Productivity per 10 h shift based on
80% productive hours per shift and
80% availability 95.9 m (315 ft)
Scheduled shifts 40 482/95.9 = 422

TABLE 23
COST OF EXPLOSIVES & CAPITAL REQUIREMENTS

Cilgel B 2" x 16	\$135.00/cwt	AN/FO Prills	\$210.00/ton
Amex Factory prepared	51.00/cwt	Palletized and wrapped	160.00/ton
AN/FO - site mixed	27.00/cwt		-----
Short Period Caps			\$370.00/ton
(average delay 5th to 8th)			-----
10 ft. lead wires	1.00 each	Truck Rate to Jobsite	140.00
16 ft. lead wires	1.19 each	Landed	510.00/ton
20 ft. lead wires	1.49 each	AN/FO Prills Landed	25.50/cwt
24 ft. lead wires	1.79 each	Diesel Fuel - 6#	1.50/cwt
30 ft. lead wires	2.30 each		-----
Capital expenditures:			\$ 27.00/cwt
Explosives Magazine			-----
8' x 12' Walk-in, F.O.B.		Pre-mixed Factory Product	51.00/cwt
Vancouver, taxes included	\$7 482.00	Difference	24.00/cwt
Cap Magazine, 12 cu. ft.			
F.O.B. Vancouver	936.00		
Blasting Machine	550.00		
Galvanometer	57.00		
Portable AN/FO Mixer	12 800.00		

TABLE 24

SUMMARY OF BLASTING COSTS

ROCK TYPE	VOLUME M ³	TOTAL COST	COST \$/M ³
-----	-----	-----	-----
PIT 1			
ORE	17 603	\$29 748.0	\$1.69
WASTE	33 453	35 678.3	\$1.07
PIT 3			
ORE	27 431	46 358	
WASTE	100 532	107 569	

TOTAL FOR PIT 3		\$153 927	
(based on pit 1 unit costs)			

TABLE 25
M & S COSTS FOR LOADING

BASIC COSTS

Service	\$US 1.43/h
Fuel	14.50/h
Repairs	20.42/h operating
Wear	3.06/h operating

ADJUSTED COSTS FOR OPERATING CONDITIONS

Service	100% X	\$ 1.43	=	\$US 1.43
Fuel	100% x	\$14.50	=	14.50
Repairs	50% x	\$20.40	=	10.21
Wear	60% x	\$ 3.16	=	1.84

				\$US27.98

At 16% exchange = \$CD32.46/h

Total cost of maintenance and service for loading:

$$\$32.46 \times 5\,954/2 = \$96\,633$$

TABLE 26
M & S COSTS FOR 35 TON TRUCK

OPERATING COSTS

	PER HOUR

FUEL	\$ 3.82
LUBE AND GREASE, ETC.	1.01
TIRES	1.06
REPAIRS	2.75
-----	-----
TOTAL	\$ 8.64
-----	-----

Total operating time : ore and waste haul : 5 954 h

Total M & S cost = \$8.64 x 5 954 = \$ 51 443

TABLE 27
M & S COSTS FOR MISCELLANEOUS EQUIPMENT

PER HOUR RATE	D7 DOZER	120-G GRADER
-----	-----	-----
Fuel	\$ 7.28	\$ 5.90
Lubricants, etc.	.54	.40
Tires	-	.91
Undercarriage	6.00	-
Repairs	2.40	1.65
Special Items	1.20	.60
-----	-----	-----
TOTALS	\$ 17.42	\$ 9.46

Note:

Operating time 297 shifts at 8 hours for both pieces of equipment or estimated 148.5 shifts for each.

1 - D7 Dozer	148.5 x 8 x 17.42 =	\$20 695
1 - 120G Grader	148.5 x 8 x 9.46 =	11 238

PER DAY RATE	1-TON TRUCK	8-PASS. MINIBUS	PICK-UP TRUCK
-----	-----	-----	-----
Fuel	\$ 4.38	\$ 2.61	\$ 1.58
Filters, Oil, etc.	.43	.43	.19
Tires	3.14	1.80	1.36
Repairs	2.00	2.00	2.00
-----	-----	-----	-----
TOTALS	\$ 9.95	\$ 6.84	\$ 5.11

Operating time 310 days (Schedule Operation 5 days/week)

1 - 1-ton Truck	310 x 9.95	=	\$3 085
1 - Minibus	310 x 6.84	=	2 120
2 - Pick-ups	2 x 310 x 5.11	=	3 168

TOTAL FOR OPERATING COST MISCELLANEOUS EQUIPMENT	----- \$40 306 -----
--	----------------------------

TABLE 28
LABOUR RATES - MINE OPERATION

CLASS(1) -----	POSITION -----	HOURLY RATE(2) -----
12	Haul Truck Operator	\$12.27
3	Labourer/Helper	9.91
10	Driller/Aux. Equip. Op.	9.40
14	Driller/Blaster	12.80
16	Heavy Duty Mechanic	13.33
18	Shovel Operator	13.85

Notes:

(1) Per Cyprus Agreement

(2) Hourly Rates Include: 1. Trade Base Rate: Cyprus Collective Agreement 1979 rate
2. Payroll burden 25% of Base Rate.

TABLE 29
COSTS FOR MINE OPERATING LABOUR

DRILLING & BLASTING

434 shifts (7 x 10 h)	
Driller 434 x 11 x \$12.89	\$61 537
Helper 434 x 11 x \$ 9.40	44 876

\$106 413

SHOVEL LOADING

297 shifts (5 x 10 h)	
1 Operator 297 x 11 x \$13.85	45 248

HAULING

2 Operators 297 x 11 x \$12.27	80 172
--------------------------------	--------

CLEAN-UP

Grader/Dozer 297 shifts	
Operator 297 x 11 x \$9.40	30 710

\$262 543

TABLE 30
LABOUR COST : MINE STAFF

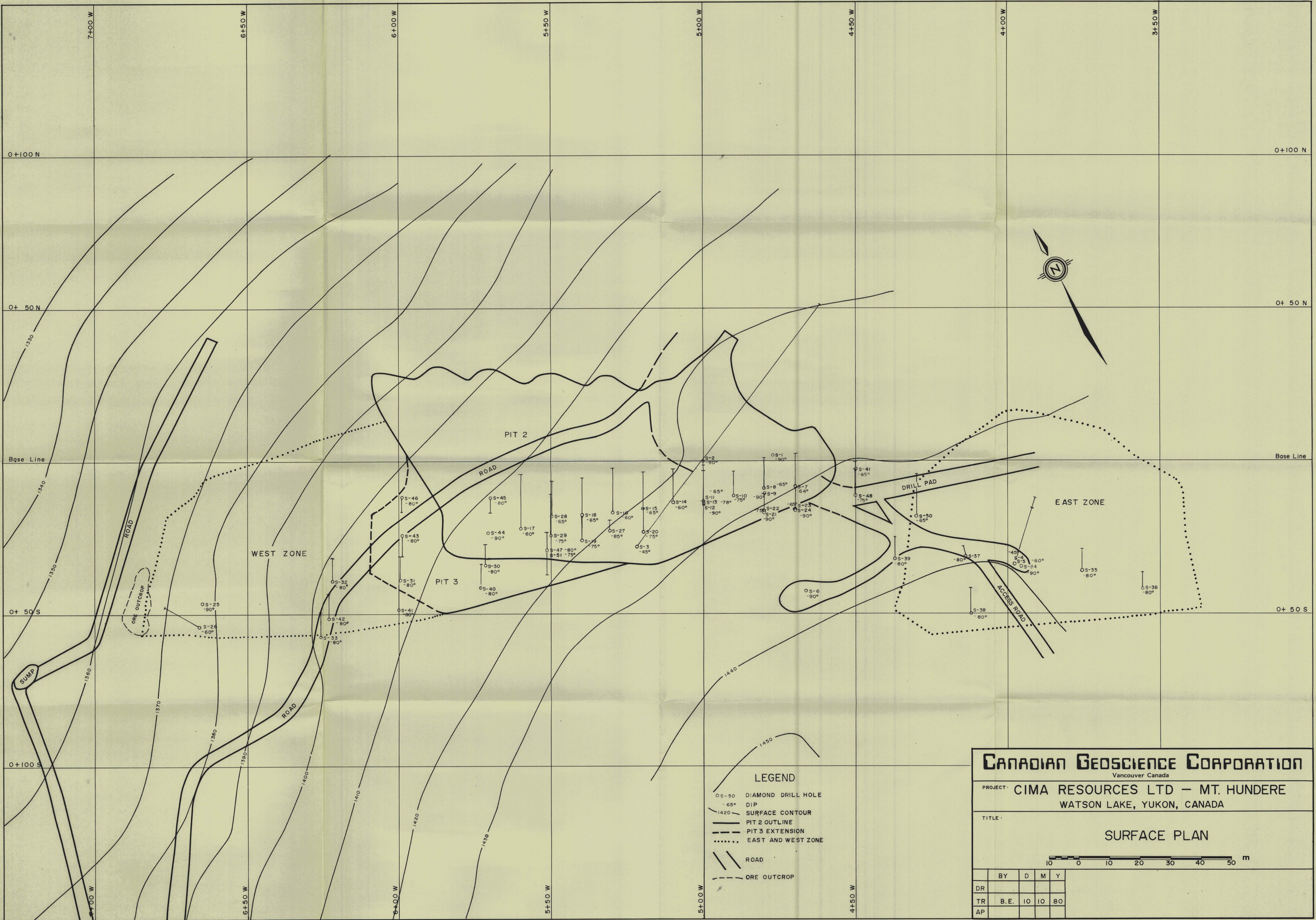
CLASSIFICATION	BASE RATE	H/DAY	DAYS/WK	WEEKLY WAGE
-----	-----	-----	-----	-----
Mine Superintendent			5	\$ 1 000.00
Secretary/Clerk	\$9.19	8	5	367.60
Engineer		8	5	634.62
Engineer Assistant	\$9.19	8	5	367.60
Shift Supervisor	11.29	10	5	620.95
H.D. Mechanic	11.08	10	5	609.40

				3 600.17
AVERAGE DAILY LABOUR COST				720.00
TOTAL FOR 310 DAYS				223 200.00
+ 25% BURDEN				55 800.00

				\$ 279 000.00

FIGURES

NO.	TITLE
---	-----
1	SURFACE PLAN
2	MINING BENCH PLAN
3-12	PLANS: BENCH 1-10
13	DIAGRAMATIC LONG. SECTION



LEGEND

- OS-50 DIAMOND DRILL HOLE
- 65° DIP
- 1420 SURFACE CONTOUR
- PIT 2 OUTLINE
- PIT 3 EXTENSION
- EAST AND WEST ZONE
- ROAD
- ORE OUTCROP

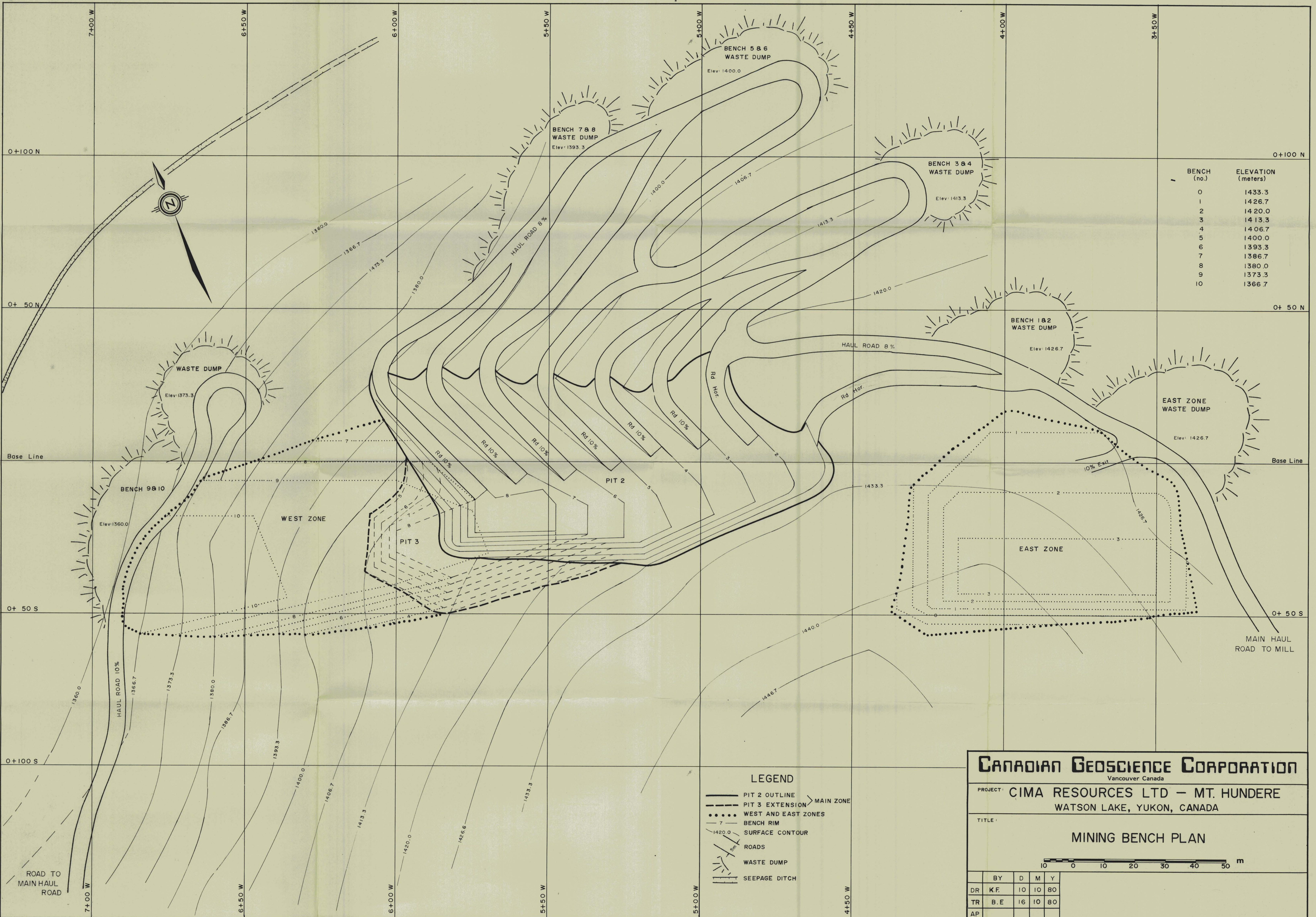
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Vancouver Canada

PROJECT: CIMA RESOURCES LTD - MT. HUNDERE
WATSON LAKE, YUKON, CANADA

TITLE: SURFACE PLAN

10 0 10 20 30 40 50 m

	BY	D	M	Y
DR				
TR	B.E.	10	10	80
AP				



BENCH (no.)	ELEVATION (meters)
0	1433.3
1	1426.7
2	1420.0
3	1413.3
4	1406.7
5	1400.0
6	1393.3
7	1386.7
8	1380.0
9	1373.3
10	1366.7

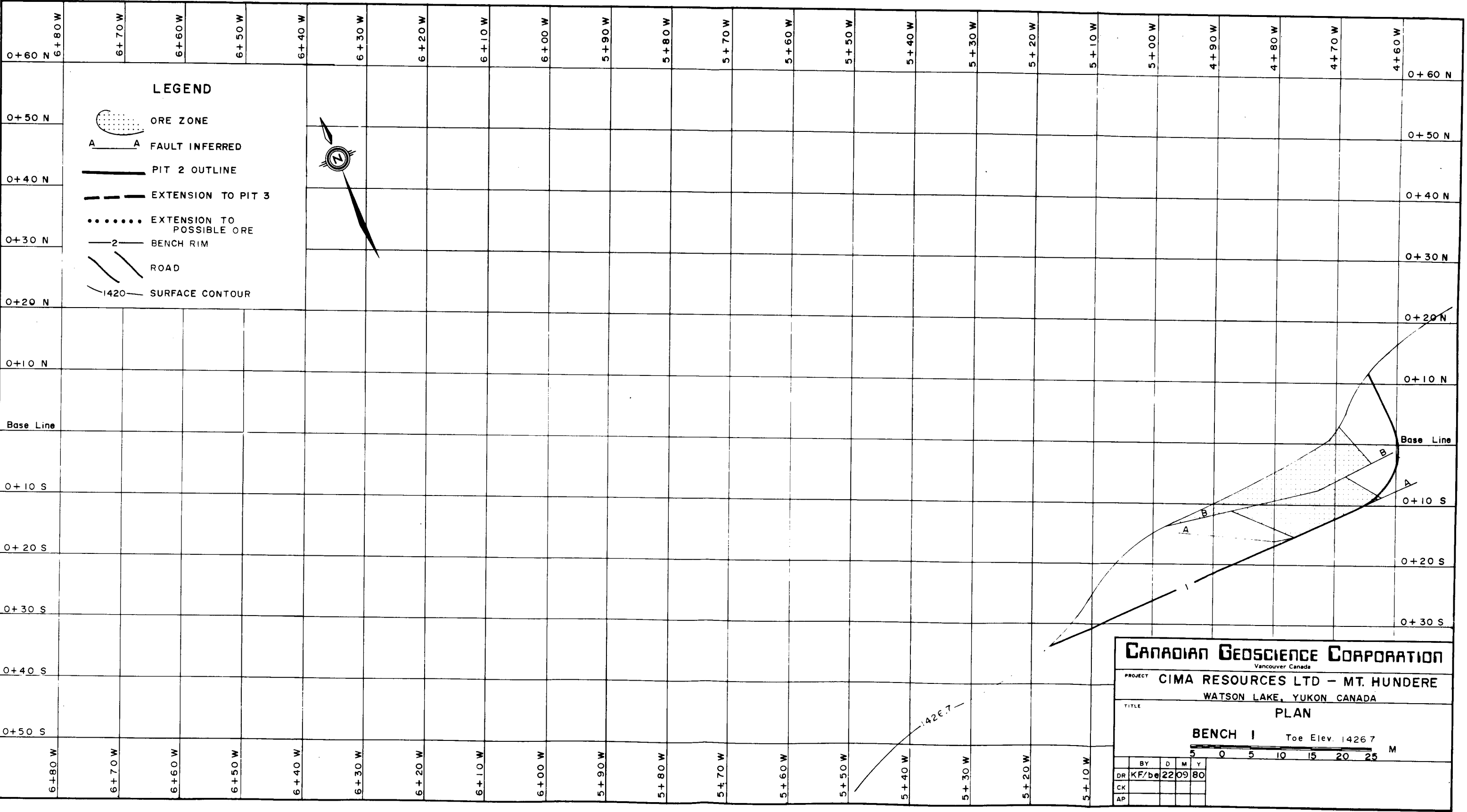
- LEGEND**
- PIT 2 OUTLINE
 - PIT 3 EXTENSION
 - WEST AND EAST ZONES
 - BENCH RIM
 - SURFACE CONTOUR
 - ROADS
 - WASTE DUMP
 - SEEPAGE DITCH

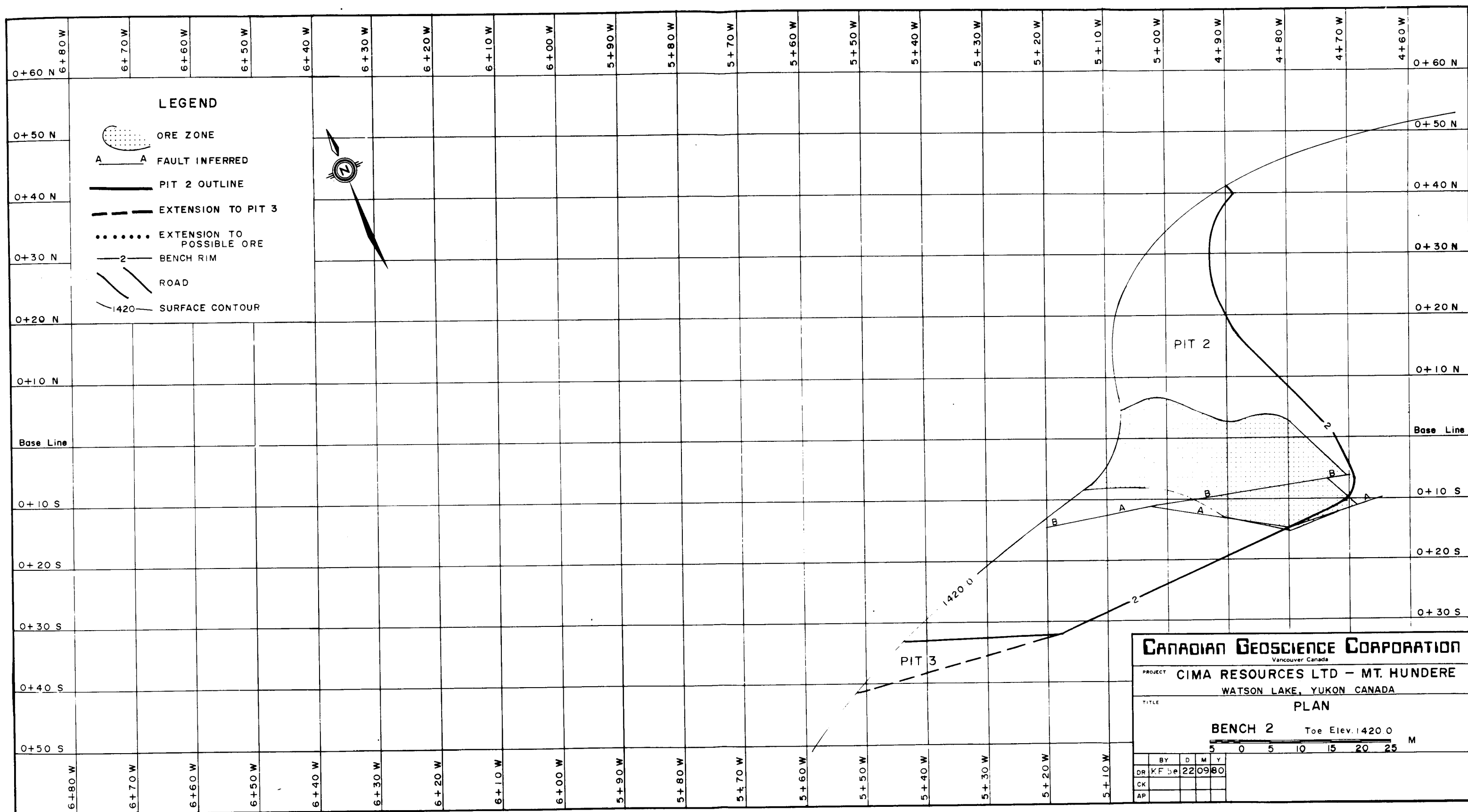
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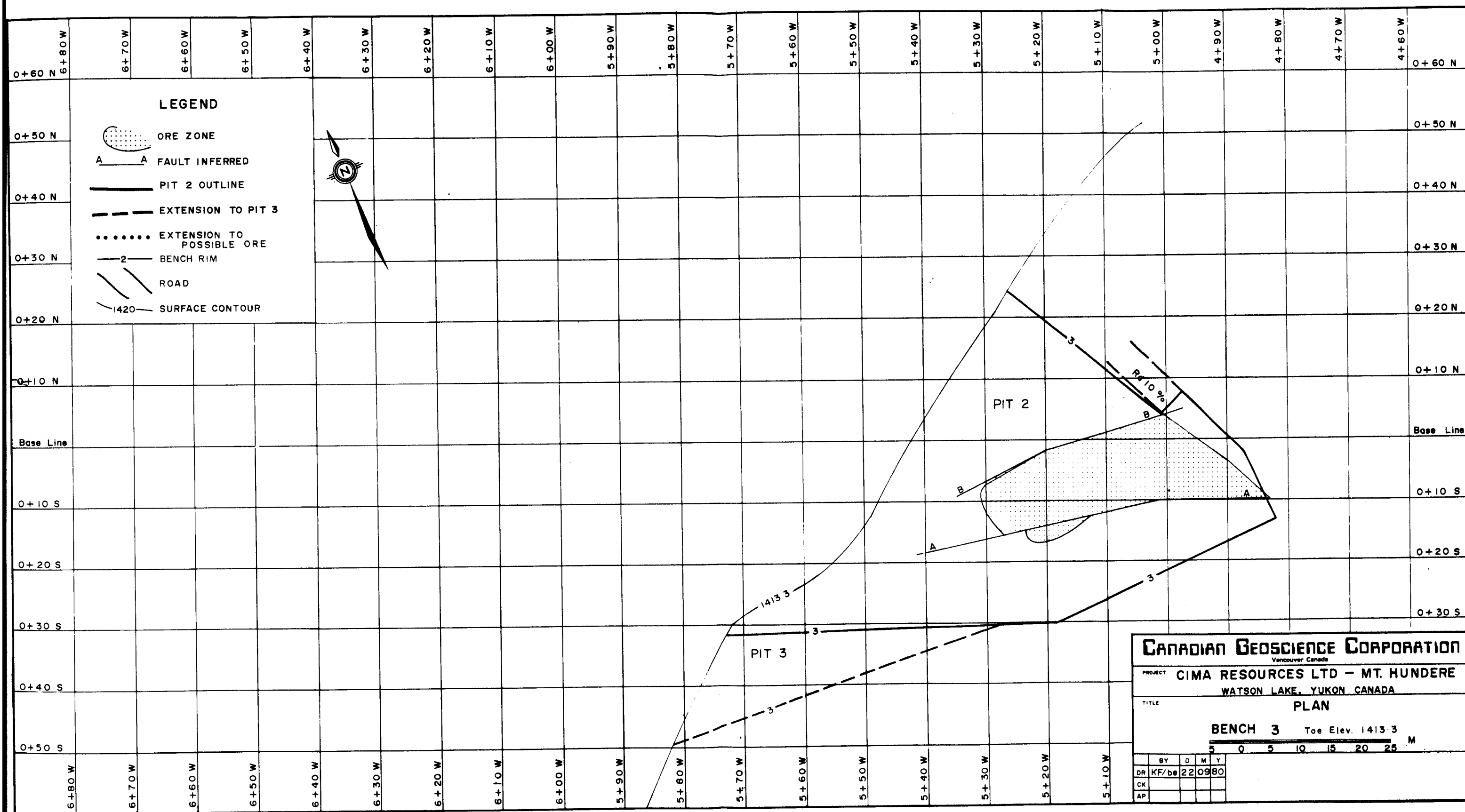
PROJECT: CIMA RESOURCES LTD — MT. HUNDERE
WATSON LAKE, YUKON, CANADA

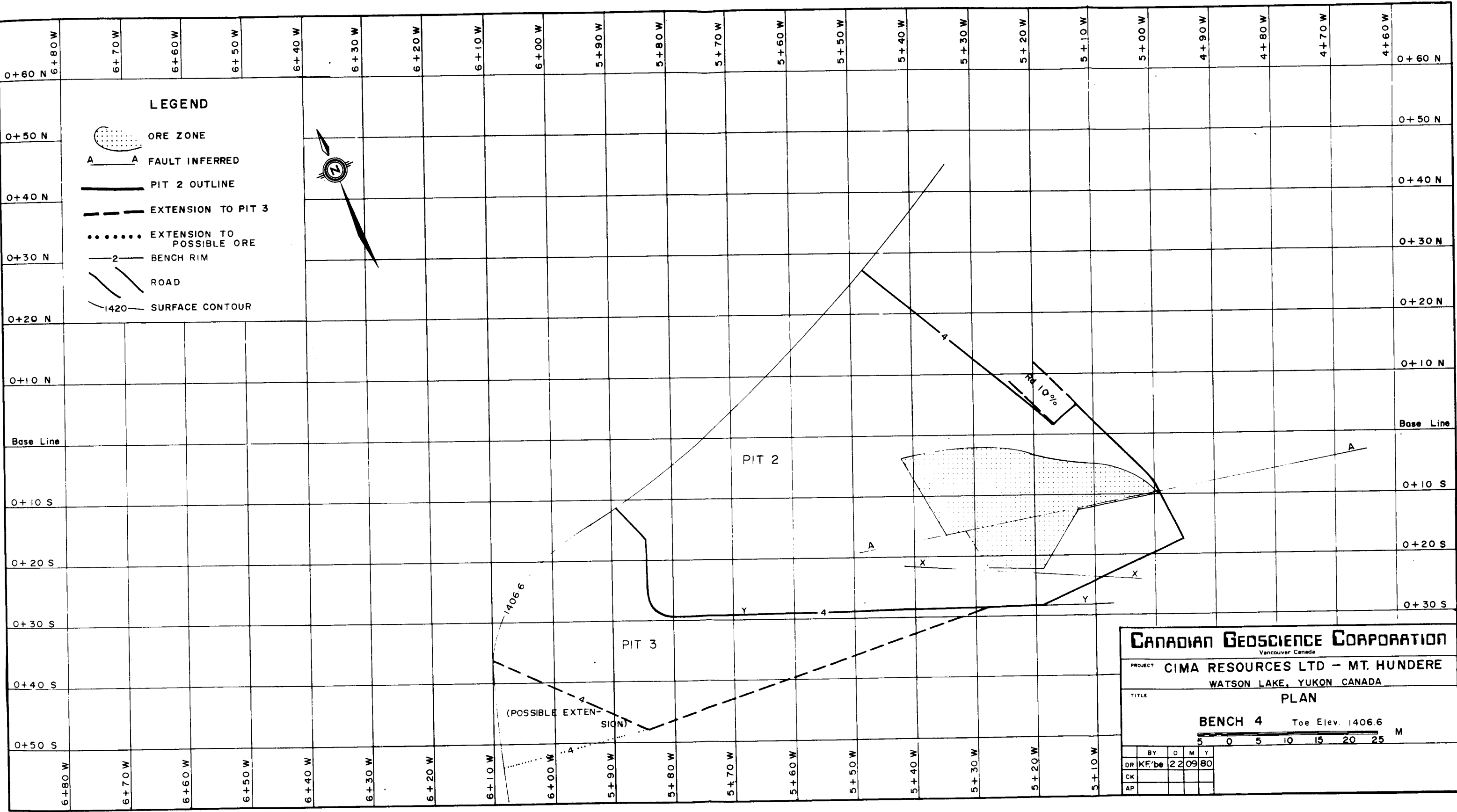
TITLE: MINING BENCH PLAN

	BY	D	M	Y
DR	K.F.	10	10	80
TR	B.E.	16	10	80
AP				









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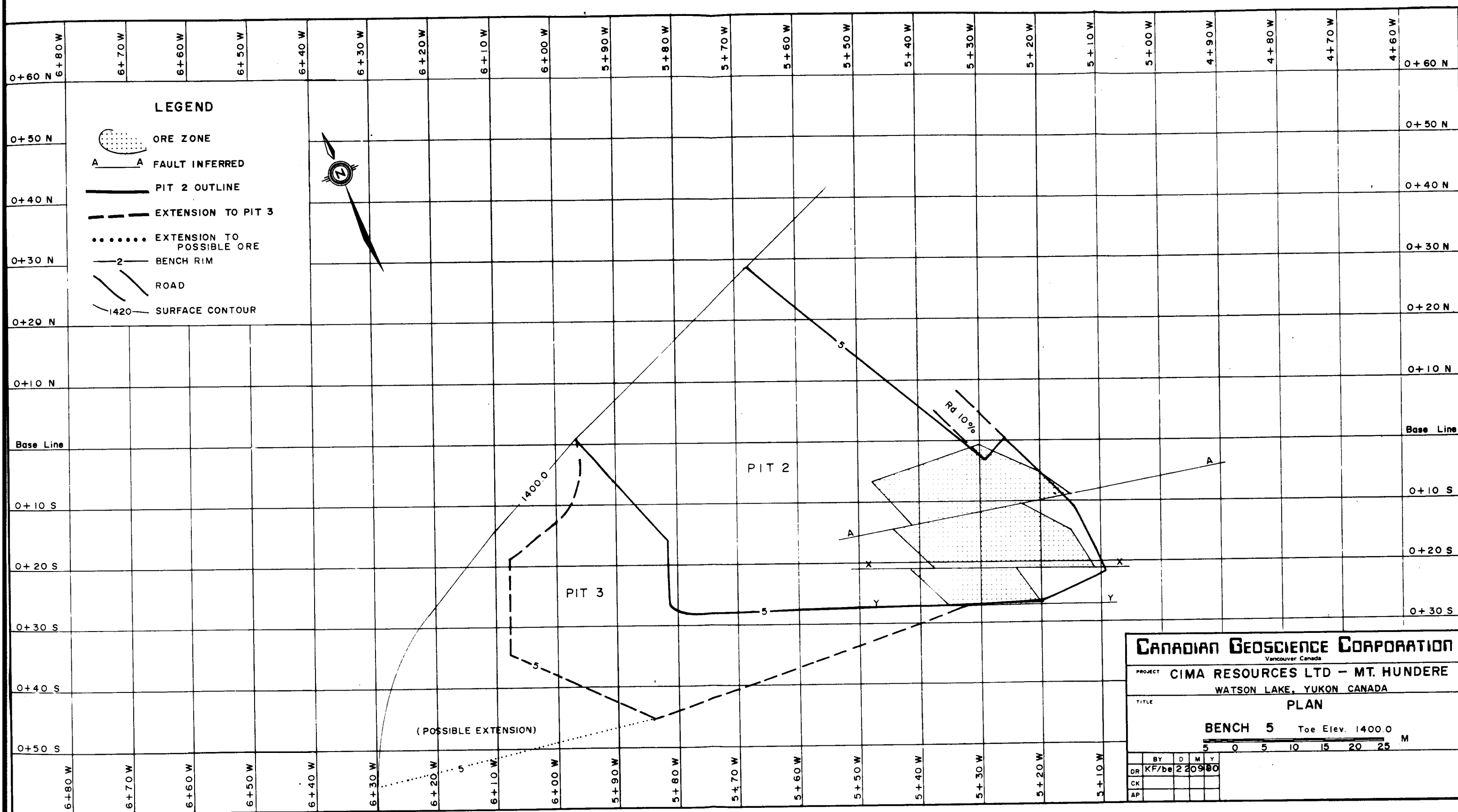
PROJECT CIMA RESOURCES LTD - MT. HUNDERE
WATSON LAKE, YUKON CANADA

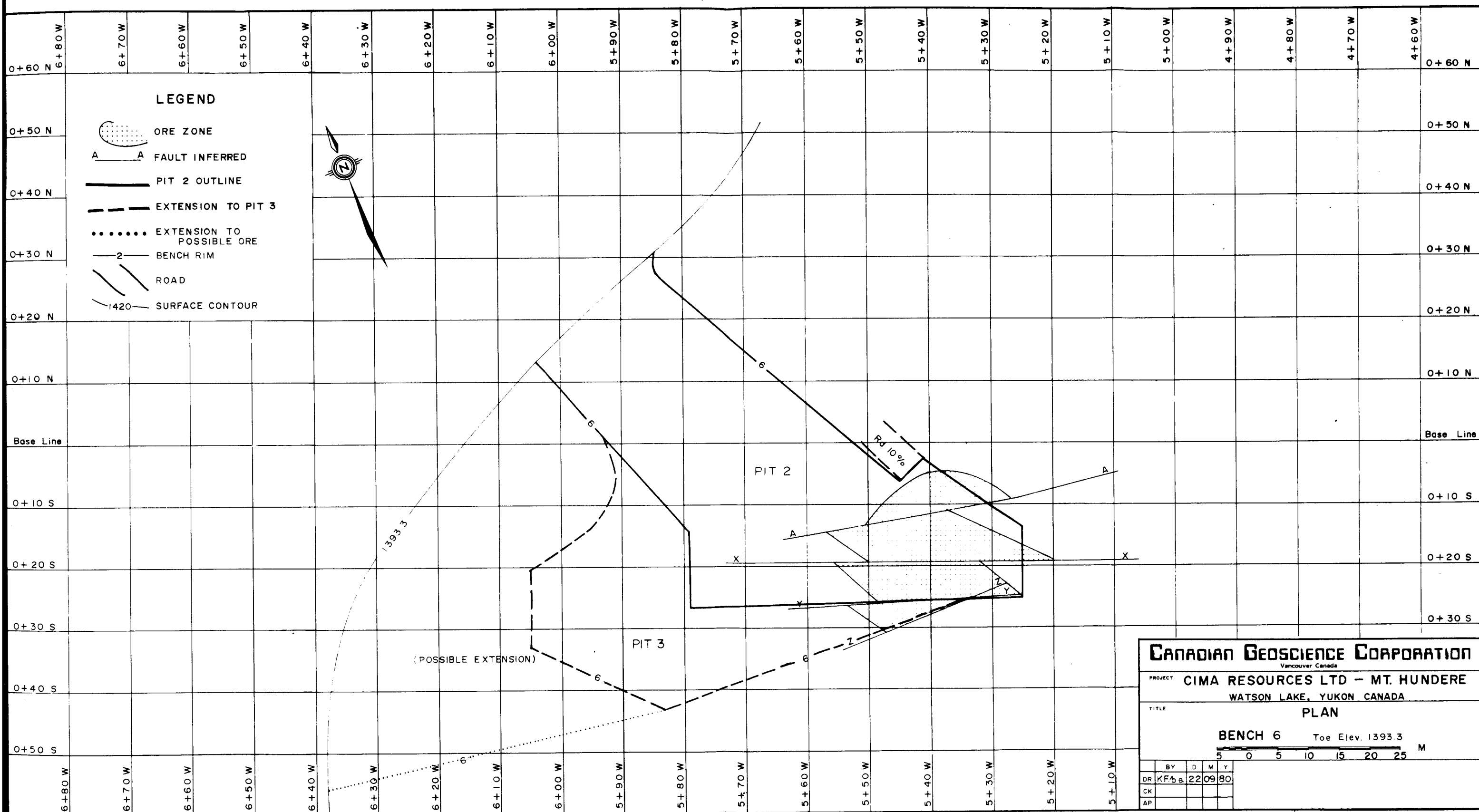
TITLE PLAN

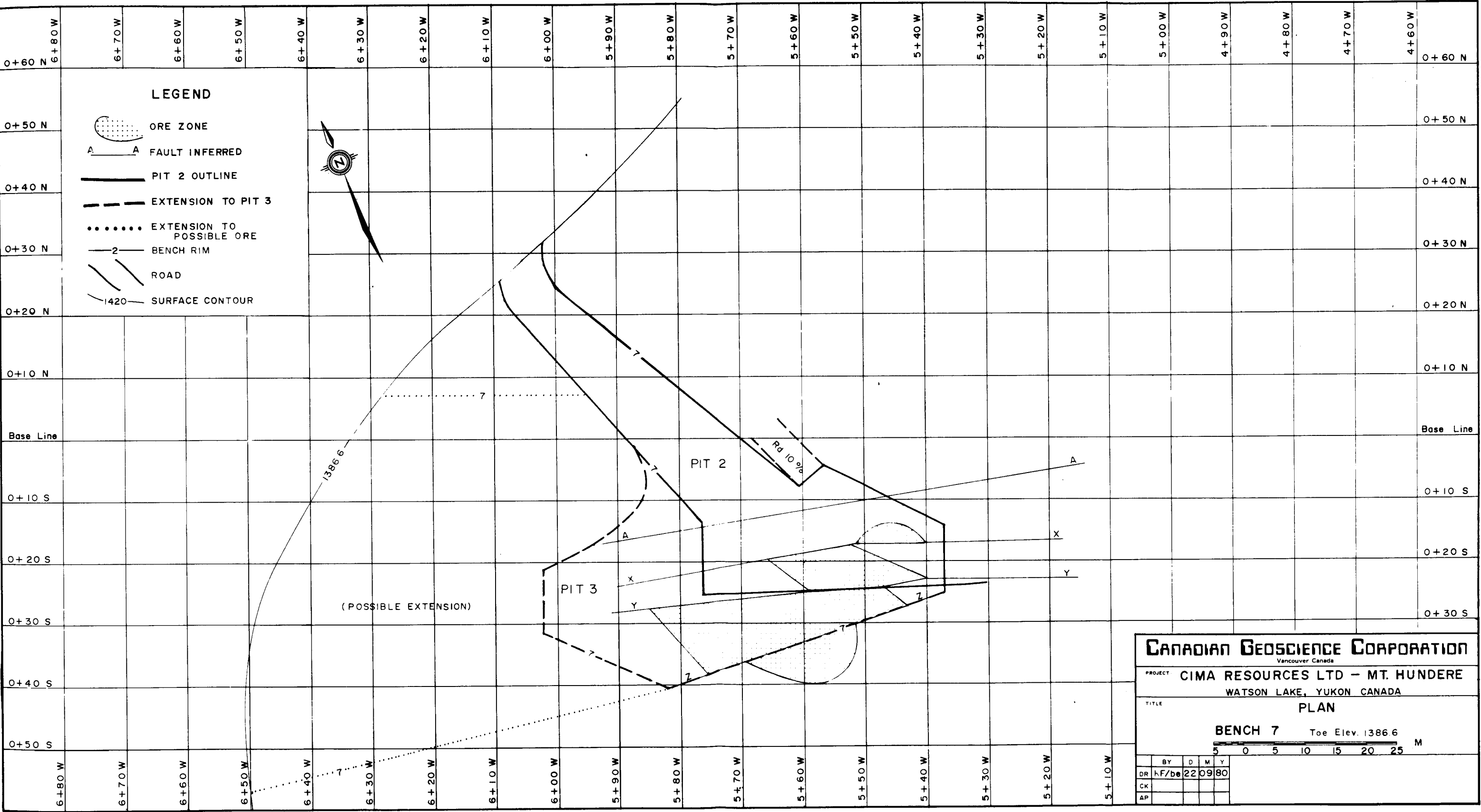
BENCH 4 Toe Elev. 1406.6

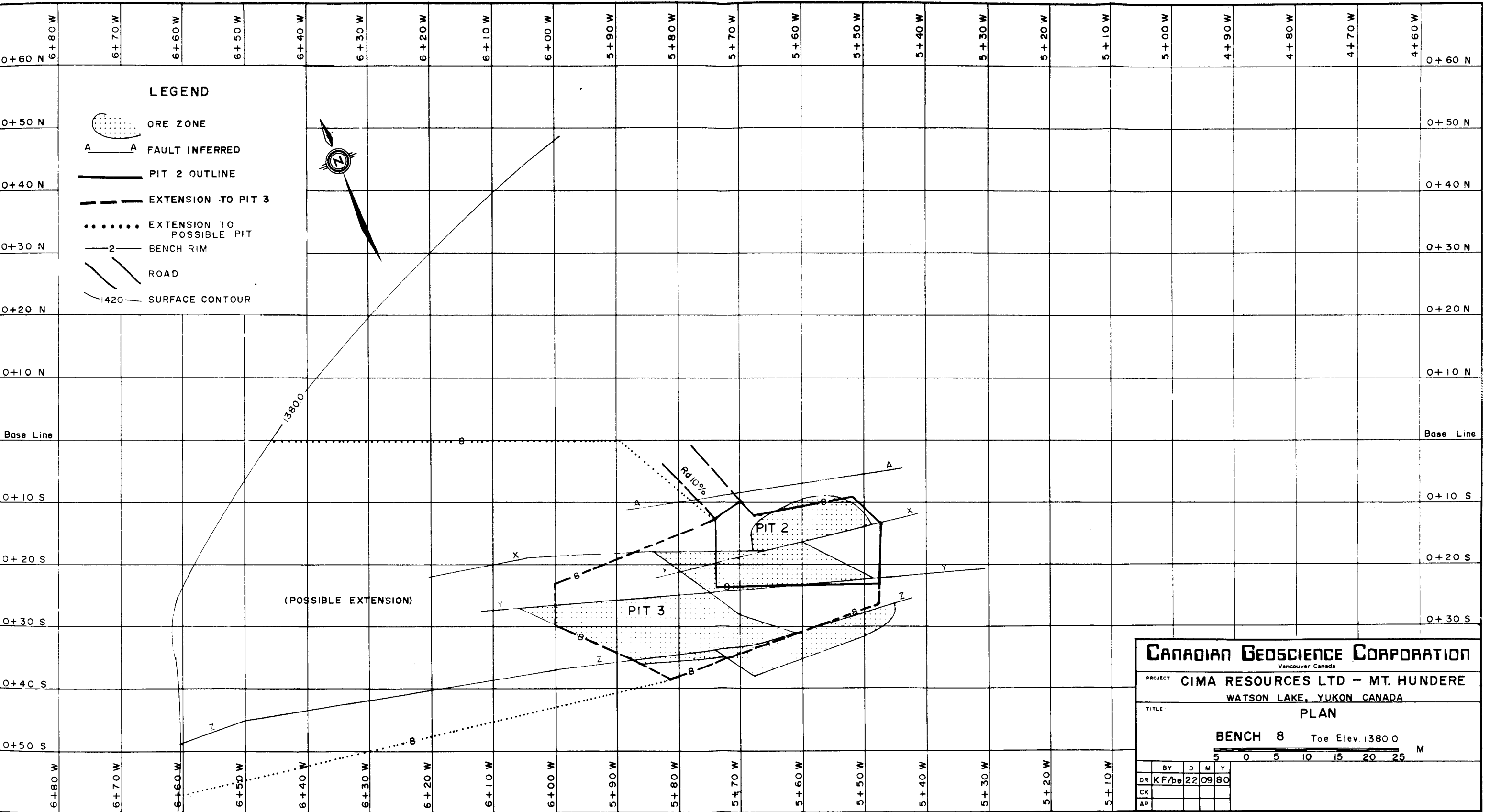
5 0 5 10 15 20 25 M

BY	D	M	Y
DR	KF/be	22	09/80
CK			
AP			









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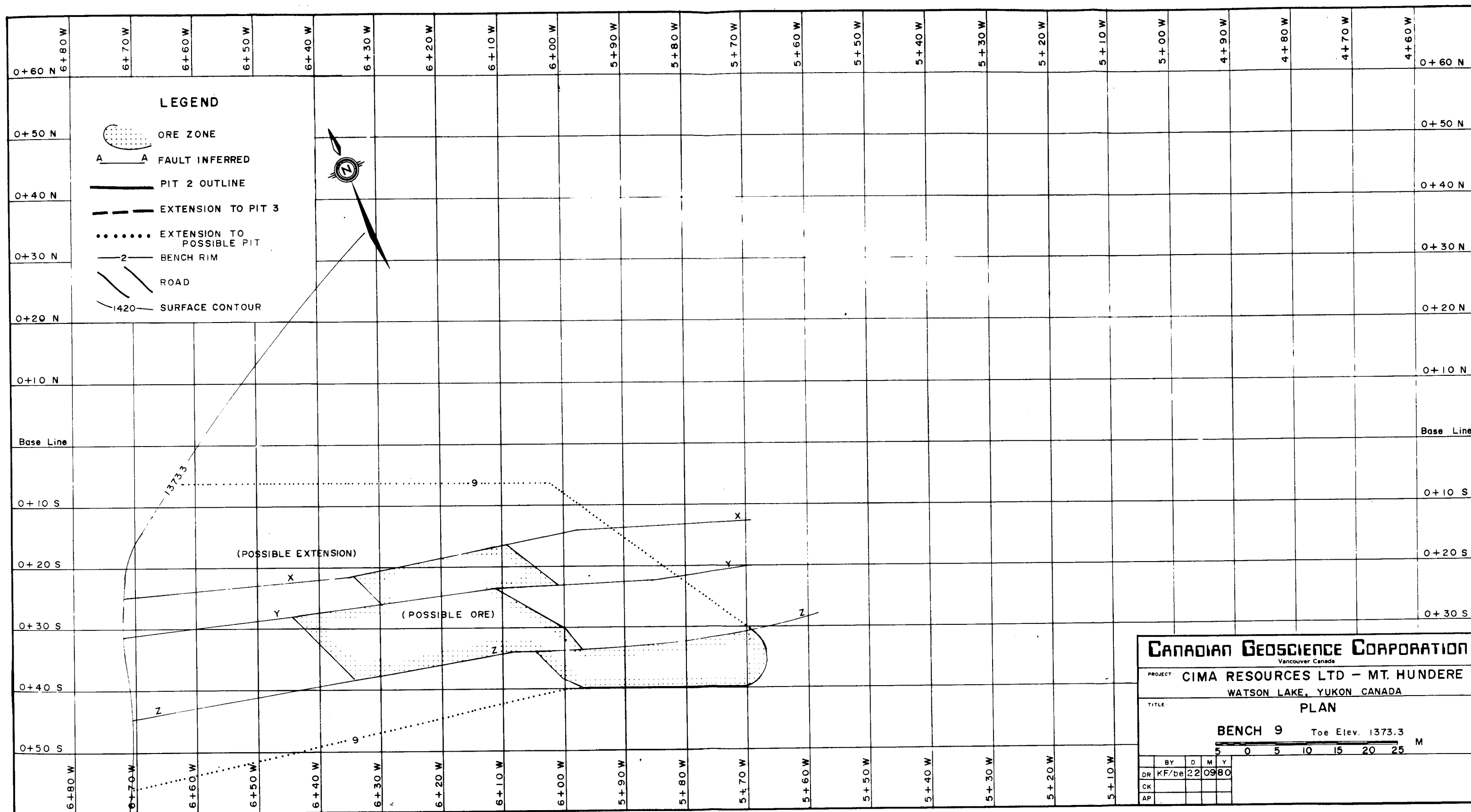
PROJECT: CIMA RESOURCES LTD - MT. HUNDERE
WATSON LAKE, YUKON CANADA

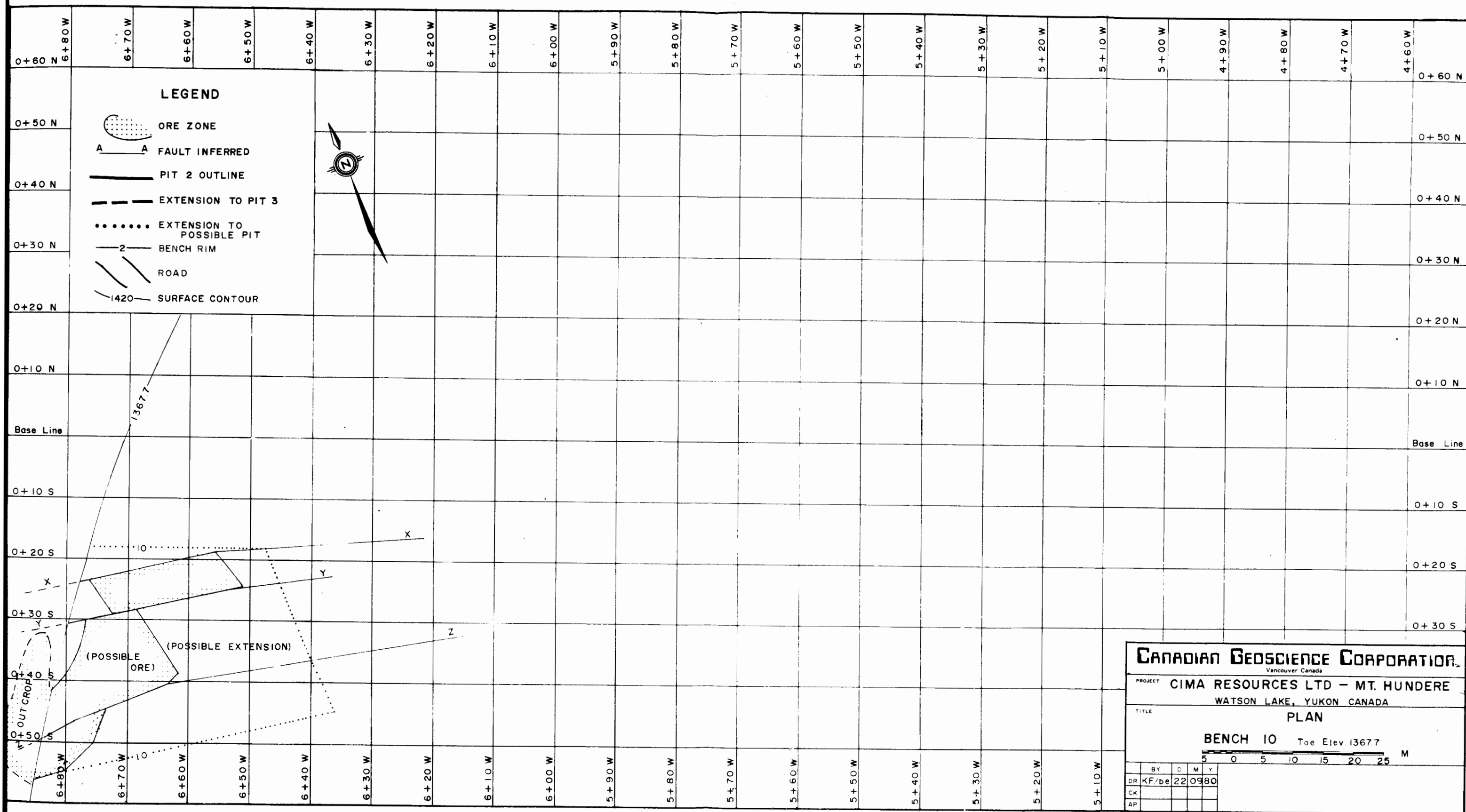
TITLE: PLAN

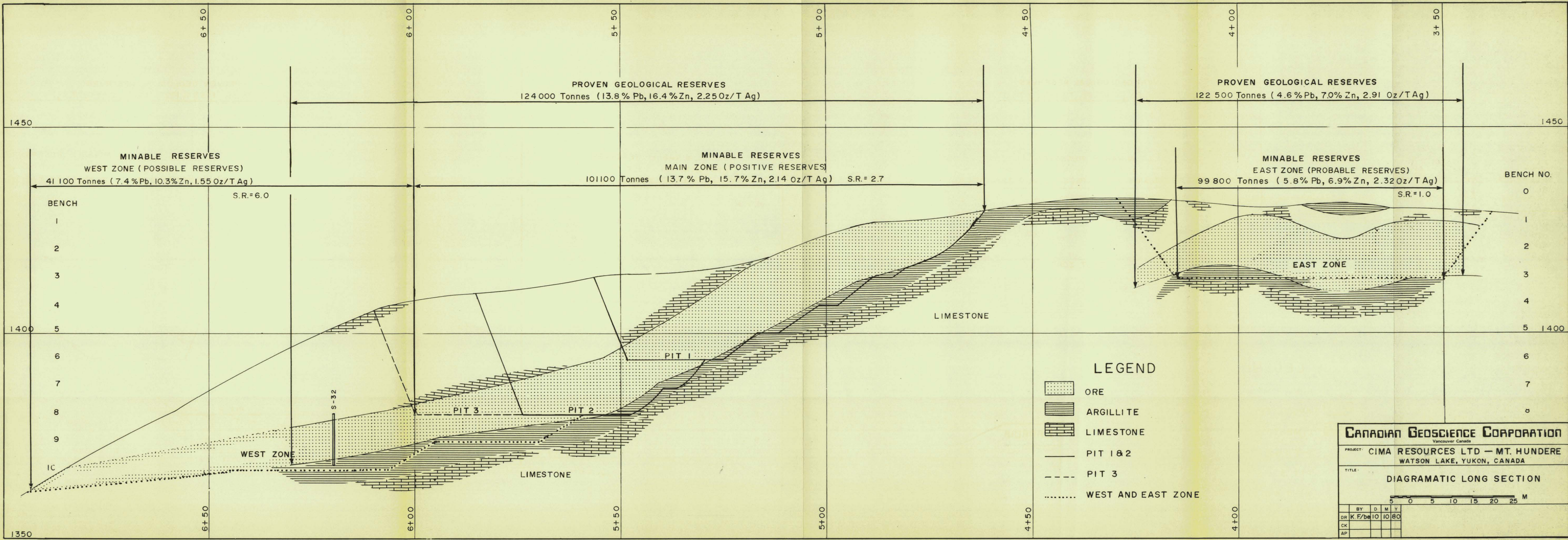
BENCH 8 Toe Elev. 1380.0

5 0 5 10 15 20 25 M

BY	D	M	Y
DR	KF	22	09/80
CK			
AP			







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PROJECT: CIMA RESOURCES LTD - MT. HUNDERE
WATSON LAKE, YUKON, CANADA

TITLE: DIAGRAMATIC LONG SECTION

BY: K.F./be 10 10 80
DR: K.F./be 10 10 80
CK:
AP:
5 0 5 10 15 20 25 M